Santo Domingo Project
Region III, Chile
NI 43-101 Technical Report on Feasibility Study

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Capstone Mining Corporation

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Effective Date:
22 May 2014

Project Number:
M40206
CERTIFICATE OF QUALIFIED PERSON

I, Joyce Maycock, P.Eng., am employed as a Project Manager with AMEC International Ingeniería y Construcción Limitada (AMEC).

This certificate applies to the technical report titled “Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study” that has an effective date of 22 May 2014 (the “technical report”).

I am a Professional Engineer in British Columbia (13331). I graduated from the Royal School of Mines, Imperial College, University of London, with a Bachelor of Science (Engineering) degree in Metallurgy in 1969.

Since 1969 I have continually been involved in mineral processing operations and projects for precious and base metals in Argentina, Canada, Chile, Peru, and Zambia. From 2009 to date I have worked as report co-ordinator for many prefeasibility and feasibility reports including a feasibility study for El Espino and Santo Domingo, a prefeasibility study for Lobo Marte, Kinross, a feasibility study for Maricunga, Kinross; a feasibility study for Angostura, GreyStar, a prefeasibility and feasibility study respectively for Cerro Casale, Barrick/Kinross; a scoping study for Zaldivar Sulphides, Barrick, and a feasibility study for Guanaco for Compañía Minera Guanaco.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I have not visited the Santo Domingo Project.

I am responsible for Sections 1.1, 1.2, 1.3, 1.20, 1.21, and 1.24; Section 2; Section 4; Section 5; Sections 21.3, 21.4; Section 23; Section 24; Sections 25.1 and 25.17 and Section 26.1 of the technical report. I am co-responsible for Sections 1.25 and 1.26, Section 3; Section 25.18; Section 26.3 and Section 27.

I am independent of Capstone Mining Corporation (Capstone) as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Santo Domingo Project during the preparation of this technical report and the supporting feasibility study.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 8 July, 2014

“Signed and sealed”

Joyce Maycock, P.Eng.
CERTIFICATE OF QUALIFIED PERSON

I, Hans Gopfert, CMC, am employed as a Project Manager with AMEC International Ingeniería y Construcción Limitada (AMEC).

This certificate applies to the technical report titled “Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study” that has an effective date of 22 May 2014 (the “technical report”).

I am a Member of the Instituto de Ingenieros de Minas de Chile and am registered with the Comision Calificadora de Competencias en Recursos y Reservas Mineras (CMC; Membership Number 20235). I graduated from the Universidad de Chile as a mining engineer in 1970.

I have practiced my profession for over 40 years. I have been directly involved in underground and mining design, mine planning, mine construction and operations, and project appraisals in Chile, Brazil, Peru and Columbia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I have not visited the Santo Domingo Project.

I am responsible for Sections 1.15.1, 1.15.2, 1.15.7; Sections 18.1, 18.2.1, 18.2.2, 18.2.3, 18.6.1, 18.7.3, 18.7.4, 18.7.6, 18.7.7, 18.10, 18.11 and 18.13; Sections 21.1.1, 21.1.5, 21.1.8, 21.1.9, 21.1.10, 21.1.11, 21.2.4, 21.2.6 and 21.2.7 of the technical report. I am co-responsible for Sections 1.18, 1.19, 1.25 and 1.26; Section 3; Sections 21.1.12, 21.1.13, 21.2.3, 21.2.8, 21.2.9 and 21.5; Sections 25.11 and 25.18; Section 26.3 and Section 27.

I am independent of Capstone Mining Corporation (Capstone) as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Santo Domingo Project during the preparation of this technical report.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 8 July 2014

“Signed”

Hans Gopfert, CMC.
CERTIFICATE OF QUALIFIED PERSON

I, David Frost, FAusIMM, am employed as Technical Director, Process, with AMEC International Ingeniería y Construcción Lida (AMEC).

This certificate applies to the technical report titled “Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study” that has an effective date of 22 May 2014 (the “technical report”).

I am a Fellow of the Australasian Institute of Mining and Metallurgy FAusIMM; #11089. I graduated with a Bachelor of Metallurgical Engineering (B. Met Eng.) from the Royal Melbourne Institute of Technology in 1991.

I have worked as a metallurgist and process engineer for over 22 years since my graduation from university. I have been involved in process operations and process plant design in various commodities and in various capacities during that time. This experience has included lead process and testwork management and design roles for conventional large scale conventional copper flotation and magnetite projects in South America.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I have not visited the Santo Domingo Project.

I am responsible for Sections 1.8, 1.14, and 1.16; Section 13; Section 17; Section 19; Sections 21.1.3, 21.2.2., 21.2.5; Sections 25.6, 25.10, and 25.12 of the technical report. I am co-responsible for Sections 1.18, 1.19, 1.25 and 1.26; Sections 21.1.12, 21.1.13, 21.2.3, 21.2.8, 21.2.9, and 21.5; Sections 25.14, 25.15 and 25.18;, and Section 26.3 that pertain to metallurgical testwork and the proposed process design. I am also co-responsible for Sections 3 and 27.

I am independent of Capstone Mining Corporation (Capstone) as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Santo Domingo Project during the preparation of this technical report.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 8 July 2014

“Signed”

David Frost, F.AusIMM
I, Anna Klimek, P.Eng., am employed as the Manager, Ports and Marine Group, with AMEC Americas Ltd (AMEC).

This certificate applies to the technical report titled “Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study” that has an effective date of 22 May 2014 (the “technical report”).

I hold a Master of Science degree in Structural Engineering from the University of Manitoba, graduating in 1993. I am a registered member of the Association of Professional Engineers and Geoscientists of British Columbia (Member Number 112553).

I have worked as a structural engineer for 18 years. My work experience includes planning and design of bulk handling infrastructure and marine facilities which are part of the infrastructure required to develop mining properties. I am a qualified person for port design in support of mining projects.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I have not visited the Santo Domingo Project.

I am responsible for Section 1.15.6; Section 18.12 and Section 21.1.7; of the technical report. I am co-responsible for those portions of Sections 1.18, 1.19, 1.25, and 1.26; Sections 21.1.12 and 21.1.13; Section 25.11; Section 26.3 that pertain to the port. I am also co-responsible for Sections 3 and 27.

I am independent of Capstone Mining Corporation (Capstone) as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Santo Domingo Project during the preparation of this technical report.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 8 July, 2014

“Signed and sealed”

Anna Klimek, P.Eng.
CERTIFICATE OF QUALIFIED PERSON

I, Vikram Khera, P.Eng., am employed as a Financial Analyst with AMEC Americas Ltd (AMEC).

This certificate applies to the technical report titled “Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study” that has an effective date of 22 May 2014 (the “technical report”).

I am a member of Professional Engineers Ontario. I graduated from the University of British Columbia in 2002 with a Bachelor of Applied Science degree in Chemical Engineering.

I have practiced my profession for over 10 years.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I have not visited the Santo Domingo Property.

I am responsible for Sections 1.22 and 1.23; Section 22 and Section 25.16 of the technical report. I am co-responsible for Sections 3 and 27.

I am independent of Capstone Mining Corporation (Capstone) as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Santo Domingo Project during the preparation of this technical report.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 8 July 2014

“Signed and sealed”

Vikram Khera, P.Eng.
CERTIFICATE OF QUALIFIED PERSON

I, David Rennie, P.Eng, am employed as a Principal Geologist with Roscoe Postle Associates Inc.

This certificate applies to the technical report titled “Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study” that has an effective date of 22 May 2014 (the “technical report”).

I am a Professional Engineer in the Province of British Columbia (Reg.#13572). I graduated from the University of British Columbia, Vancouver, BC, Canada, in 1979 with a Bachelor of Applied Science degree in Geological Engineering.

I have practiced my profession for 35 years since graduation. I have extensive experience in a number of geological environments and have carried out numerous Mineral Resource estimates, audits, and reviews for a wide variety of commodities, including copper, gold, silver, nickel laterite, tungsten, iron, uranium, PGEs, and industrial minerals. My relevant experience for the purpose of the technical report includes:

- Preparation of Mineral Resource estimates and audits on numerous copper-gold exploration projects and mining operations in North and South America.
- Pre-Feasibility and Feasibility Study work on several projects.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101)


I am responsible for Sections 1.4 to 1.7, Section 1.9 and 1.10, Section 6 to 12, Section 14, Sections 25.2 to 25.5, 25.7, and Section 26.2 of the technical report. I am co-responsible for Sections 1.26 and 27.

I am independent of Capstone Mining Corporation (Capstone) as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Santo Domingo Project during the preparation of the 2014 feasibility study and this technical report, and I was a previous co-author on the following technical reports for the Project:

Associates for Far West Mining Ltd, re-addressed to Capstone Mining Corp., effective date August 26, 2010.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 8 July 2014

“Signed and sealed”

_________________________________________

David Rennie, P.Eng.
CERTIFICATE OF QUALIFIED PERSON

I, Carlos Guzman, CMC, am employed as the Principal / Project Director with NCL Ltda.

This certificate applies to the technical report titled “Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study” that has an effective date of 22 May 2014 (the “technical report”).

I am a Member of the Instituto de Ingenieros de Minas de Chile and am registered with the Comision Calificadora de Competencias en Recursos y Reservas Mineras (CMC; N° 0119). I am also a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM, N°229036). I graduated from the Universidad of Chile as a mining engineer in 1995.

I have practiced my profession for 19 years since graduation. My relevant experience for the purpose of the Technical Report is:

- Review and report as a consultant on numerous exploration, mining operation and projects around the world for due diligence and regulatory requirements.
- I have extensive experience in mining engineering. I have worked on mining engineering assignments.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I visited the Santo Domingo Project on 15 October 2013.

I am responsible for Sections 1.11, 1.12, 1.13, 1.15.3, 1.15.4; Section 15; Section 16; Section 18.3 and 18.4; Sections 21.1.2, 21.2.1; Sections 25.8 and 25.9 of the technical report. I am co-responsible for Sections 1.18, 1.19 and 1.25; Sections 21.1.12, 21.1.13, 21.2.3, 21.2.8, 21.2.9 and 21.5; Sections 25.14, 25.15, and 25.18 where information that pertains to the mineral reserves and mine plan is included in those sections. I am also co-responsible for Sections 3 and 27.

I am independent of Capstone Mining Corporation (Capstone) as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Santo Domingo Project during the preparation of this technical report and the supporting feasibility study.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 8 July, 2014

“Signed”

Carlos Guzman, CMC, FAusIMM.

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CERTIFICATE OF QUALIFIED PERSON

I, Thomas F. Kerr, P.E., am employed as the President of Knight Piésold and Co. (USA).

This certificate applies to the technical report titled “Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study” that has an effective date of 22 May 2014 (the “technical report”).

I am a Registered Professional Engineer in the State of Colorado (Registration No. 44505). In addition, I am a member in good standing of the following professional associations:

a. Registered Professional Engineer in the State of Michigan (Registration No. 6201057916)
b. Registered Professional Engineer in the State of Alaska (Registration No. 10969)
c. Registered Professional Engineer in the State California (Registration No. C49260)
d. Registered Professional Engineer – Ontario, Canada (No. 90407230)
e. Registered Professional Engineer – British Columbia, Canada (No. 14906)

I graduated from the University of Saskatchewan, Canada with a B.Sc., Eng. (Civil) in 1982 and from Imperial College, United Kingdom with an M.Sc.,D.I.C. Eng (Soil Mechanics) in 1986.

I have worked as a Civil Geotechnical Engineer for a total of 32 years since my graduation from the University of Saskatchewan, Canada in 1982. I have extensive experience in design, construction, operation, and closure of geotechnical, water, and environmental projects for the mining industry. My principal areas of expertise are in tailings and heap leach management facilities and I have been responsible for the planning, design, and operational assistance on many such facilities around the world. In tailings management, my work has included sub-aerial, cycloned, thickened, and paste projects.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I visited the Santo Domingo Project on 24 October 2013.

I am responsible for Sections 1.15.5, 1.17; Sections 18.5 and 18.6.1; Section 20; Section 21.1.4; Section 25.13 and Section 26.5 of the technical report. I am co-responsible for Sections 1.18, 1.19 and 1.26; Section 21.1.12 and Section 21.1.13; and Sections 25.11, 25.14, 25.15; where information that pertains to the tailings storage facility, environmental, social and permitting considerations is included in those sections. I am co-responsible for Sections 3 and 27.

I am independent of Capstone Mining Corporation (Capstone) as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Santo Domingo Project during the preparation of this technical report and the supporting feasibility study.
I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated July 8, 2014

"Signed and sealed"

__________________________________________

Thomas F. Kerr, P.E.
CERTIFICATE OF QUALIFIED PERSON

I, Roy Betinol, P.Eng, am employed as the General Manager of BRASS Chile SA.

This certificate applies to the technical report titled “Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study” that has an effective date of 22 May 2014 (the “technical report”).

I am a Registered Professional Engineer from the State of California, Registration number M30166. I am an affiliate member of the American Society of Mechanical Engineers, USA. I graduated from Silliman University in 1976 with the degree of Bachelor in Science of Mechanical Engineering.

I have practiced my profession for 37 years. I have been directly involved in the design of slurry concentrate pipelines which are currently operating. Some of the most recent projects include the Cerro Negro Norte Iron Concentrate Pipeline, 2013, Chile; Escondida Expansion Copper Concentrate Pipeline, 2013, Chile; Hierro Atacama Iron Concentrate Pipeline, 2008, Chile; and, Paragominas Bauxite Pipeline, 2006, Brazil.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I have not visited the Santo Domingo Project.

I am responsible for Sections 18.2.4, 18.7.1, 18.7.2, 18.7.5, 18.8, 18.9; Section 21.1.6; and Section 26.4 of the technical report. I am co-responsible for Sections 1.18 and 1.19; Section 1.26; Sections 21.1.12, and 21.1.13; and Sections 25.11, 25.14 that pertain to the pipelines. I am also co-responsible for Sections 3 and 27.

I am independent of Capstone Mining Corporation (Capstone) as described in Section 1.5 of NI 43–101.

I have been involved with the Santo Domingo Project during the preparation of this technical report and the supporting feasibility study.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 8 July 2014

“Signed and sealed”

__________________________________________
Roy Betinol, P.Eng. M30166

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IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Technical Report for Minera Santo Domingo SCM (Minera Santo Domingo) by AMEC International Ingeniería y Construcción Limitada (AMEC), BRASS Chile SA (BRASS), Knight Piésold S.A. (KP), NCL Ltda (NCL), and Roscoe Postle Associates Inc (RPA), collectively the “Report Authors”. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors’ services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Minera Santo Domingo subject to terms and conditions of the individual contracts with the Report Authors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other use of, or reliance on, this report by any third party is at that party’s sole risk.
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1.0 SUMMARY

1.1 Introduction and Terms of Reference

AMEC International Ingeniería y Construcción Limitada (AMEC) was commissioned by Capstone Mining Corporation (Capstone) to prepare an independent Qualified Person’s Review and NI 43-101 Technical Report (the Report) for the Santo Domingo Project (the Project), located in the Third Region of Chile.

The Report is based on the results of a feasibility study (the 2014 Feasibility Study) on the Project completed in 2014. The 2014 Feasibility Study included, in addition to input from Capstone, contributions from a number of engineering and consulting firms, including AMEC; BRASS Chile SA (BRASS; sea water and magnetite concentrate pipelines); Cliveden Trading AG (CTAG; metals marketing); CRU Group (CRU; metal prices); Ghisolfo y Cia Ingeniería de Consulta Ltda (Ghisolfo; road by-pass design and copper concentrate transport study); Knight Piésold S.A. (KP; tailings storage facility (TSF) and environmental studies); NCL Ltda (NCL; Mineral Reserves, geotechnical design, open pit designs, and waste rock facilities); PRDW Chile (PRDW; port materials handling, concentrate storage and shiploading facilities); and Roscoe Postle Associates Inc (RPA; geological interpretation and Mineral Resource estimates).

The firms and consultants who contributed Qualified Persons (QPs) for the content of this Report, which is based on the 2014 Feasibility Study, are, in alphabetical order, AMEC, BRASS, KP, NCL, and RPA.

The Report will be used in support of Capstone’s press release dated 4 June, 2014, entitled “Capstone Mining Reports Positive Feasibility Study Results for Santo Domingo Project in Chile”.

The Project is held 70% by Capstone and 30% by Korea Resources Corporation (Kores). The companies use an operating entity, Minera Santo Domingo SCM (Minera Santo Domingo), as the Chilean holding company for the Project. For the purposes of this Report, as Capstone is the Project operator, Minera Santo Domingo and Capstone are used interchangeably.

Unless otherwise noted, all dollar figures used in this summary are US$. Although calendar years are mentioned in this summary, these are illustrative only, as no decision has been made on mine construction by Capstone, and the relevant permits for Project development remain to be secured.
1.2 Project Description and Location

The Santo Domingo project is located approximately 5 km southeast of the town of Diego de Almagro in the Atacama Region of northern Chile. The Santo Domingo property was originally part of the BHP's Candelaria area which consisted of eight non-contiguous concessions along a north–south corridor that stretched between the towns of Taltal in the north to a point about 75 km south of the city of Copiapó.

AMEC was provided with information that supports that Minera Santo Domingo is a mining company (sociedad contractual minera) that is legally organized under the laws of the Republic of Chile. Capstone has advised AMEC that under the terms of the shareholder agreement signed between Capstone and Kores on June 17, 2011, Capstone is Project operator.

Capstone has four groups of concessions with a total of 178 claims (82 exploitation concessions and 96 exploration concessions) which cover a total of 36,375 hectares and includes the areas of the planned mine site, plant area and auxiliary facilities including proposed port facilities and the planned sea water and concentrate pipelines from the port to the mine. Concessions are held in the name of Minera Santo Domingo.

The total concession area is divided as follows:

- 19,375 ha of exploitation concessions that encompass the area where the mine, plant, construction camp and ancillary facilities are planned
- 17,000 ha of exploration concessions that encompass the sea water and concentrate pipeline route from the port to the mine site, and the port.

Concessions are surveyed as part of the grant process, and are protected under Chilean law by payment of the annual mining license fees. Capstone advised AMEC that all concession fees were current as of 22 May 2014, and will continue to be paid on a regular basis as due, using a formal status tracking system.

No surface rights are currently held by Capstone in the Project area; however the process to acquire surface rights is well understood. AMEC was provided with information that supports that the surface land where the Project is located (in the Community of Diego de Almagro) is part of a larger lot that it is owned by the state, and is managed and represented by the Ministerio de Bienes Nacionales. Capstone proposes to consolidate Capstone’s property in the areas covering the deposit and the process facilities by purchasing these lands through the Ministerio de Bienes Nacionales. Capstone notes that it will be necessary to either acquire a total of 3,901.3 ha or complete the creation of mining easements for the installation and use of
various facilities. Capstone also proposes to apply for one or more mining rights of way in the areas of interest of the Project such as the pipeline route, access roads and off-site ancillary facilities to safeguard these areas. The Project has received government guarantees for the rights of way required by the Project for the areas currently identified. There is sufficient suitable land available within the exploitation concessions for the planned tailings disposal, mine waste disposal, and mining-related infrastructure such as the open pit, process plant, workshops and offices.

The Project as currently envisaged will not require an application for water rights. The water for the operation of the Project will consist solely of sea water. A maritime concession has been requested which will allow the extraction of sea water. Water for construction will be obtained from an authorized third-party provider, Aguas Chañar S.A.

From 1 January 2006 mine operators whose annual sales exceed the equivalent of 12,000 tonnes of fine copper a year must pay a mining tax. The mining tax is a tax on operational mining income, and levied on a sliding-scale rate basis of between 5% and 14% depending on operating margins.

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

Current access to the Project area is via the paved Pan-American Highway (Route 5 North) and a network of generally well maintained gravel roads. The Santo Domingo property is about five hours’ travel time by road south of Antofagasta, and two hours by road north of Copiapó.

The climate is generally warm, dry and clear in all seasons. The area of the proposed mine site is classified as interior desert, whereas the proposed port location is considered to be in a coastal desert regime. Rainfall is low and concentrated in the winter months. Mining activities are expected to be possible on a year-round basis.

Elevations in the Project area range from approximately 900 masl to 1,500 masl. Vegetation is very sparse.

The Project area is likely to have high seismicity and the site is considered to be Zone 3 according to the Chilean standard NCh 2.369, with a peak ground acceleration value of 0.4 g.

The Region has well established infrastructure (energy, water, transportation and labour) to serve the mining industry. However, there is currently no infrastructure on the Santo Domingo property except gravel roads for access to the concessions and drill sites. The nearby town of Diego de Almagro is connected to the regional power
grid and the main railway line. Details of the proposed infrastructure to be constructed in support of planned mining activities are discussed in Section 1.13.

1.4 History

Artisanal mining activities commenced in the general project area during the early 19th century. The major commodities targeted were gold and iron. As a result, there are a significant number of small workings and pits throughout the Project area. However, most of the surface workings are typical of artisanal activities, being less than a few tens of metres in length.

Modern exploration commenced in 2002. Between 2002 and June 2011 work by Far West Mining included a regional airborne geophysical survey and interpretation of results, geological mapping, surface and drainage sampling, an induced polarization (IP) survey, core and reverse circulation (RC) drilling, and resource estimation. A preliminary assessment was conducted in 2008.

Capstone acquired the Project from Far West Mining and completed a pre-feasibility study. A feasibility study was commissioned in 2012.

1.5 Geology and Mineralization

The Project is located within the Cretaceous Iron Belt (CIB) of the Atacama fault zone, a ductile/brittle sinistral strike-slip and dip-slip crustal scale structure that parallels the coast of Chile for over 1,200 km. The CIB is a segment of the Atacama fault zone approximately 630 km by 40 km in dimension that hosts numerous iron oxide–copper–gold (IOCG) type deposits. The Santo Domingo Project lies on the east side of the Atacama fault complex, which, in this area, consists of numerous clusters of generally north–south structural breaks in a belt approximately 30 km wide.

The base of the stratigraphic sequence in the Santo Domingo Project area is interpreted to be Punta del Cobre Formation sedimentary rocks. These rocks grade upwards into an interdigitated contemporaneous sequence of limestone and marine sediments of the Chañarcillo Group and andesitic flows and volcaniclastic rocks of the Bandurrias Group. The upper Punta del Cobre Formation near its contact with the overlying Bandurrias–Chañarcillo Group sequences is the stratigraphic host location of the Candelaria deposit (operated by third parties) approximately 120 km to the south. The Project area is divided into a number of structural blocks with different lithological characteristics suggesting that the blocks are part of different stratigraphic levels.
Mineralization within the Project consists of:

- Stratiform replacement mantos and breccias within tuffaceous sedimentary rocks (e.g. Santo Domingo Sur deposit)
- Structurally controlled mineralization along the east-west Santo Domingo fault zone (e.g. Estrellita deposit)
- Small, closely spaced (100 m to 200 m) northwest-trending and moderately to steeply northeast-dipping veins which range in width from a few centimetres to several metres
- Minor copper oxide minerals disseminated in amygdules in volcanic flows and encountered as small chalcocite nodules in limestone

Drilling at 100 m centres or less at the Santo Domingo Sur deposit has outlined a 150 m to 500 m thick copper bearing, specularite-magnetite manto sequence covering an area of approximately 1,300 m by 800 m. The mantos are zoned from an outer rim of specular-hematite toward a magnetite rich core. The mantos consist of semi-massive to massive specularite and magnetite layers with clots and stringers of chalcopyrite, that range in thickness from approximately 4 m to 20 m. The upper parts of the manto sequence are frequently oxidized and contain various amounts of copper oxides and chalcocite. The deposit strikes approximately northeast and dips at low angles to the northwest. The southern and eastern margins of the deposit appear to be structural and are defined by drill holes into adjacent structural blocks with different geology. The western margin appears to be a transitional boundary from the tuff sequence to a sedimentary sequence in the west with gradually weakening manto development. The mineralization has been traced towards the north where the iron oxide mantos continue to dip under cover but contain weaker and less continuous copper mineralization. The spatial extent of the deposit is reasonably well defined. Drilling below a depth of 350 m is sparse and mineralization below that depth is not well defined at this time.

The Iris deposit is a narrow zone (100 m to 250 m wide) of copper-bearing iron mantos and breccias extending over 1,900 m that are hosted by andesitic tuffs and andesitic breccias. The deposit is close to surface at the southern end and plunges towards the north. The Iris deposit appears to have formed in a north–northwest striking fault zone that is bounded by a west-dipping fault that can be traced along most of the deposit’s western side. The eastern side of the deposit is bordered by a steeply dipping fault that divides andesitic tuffs on the western side from calcareous sedimentary rocks and limestone to the east. The dominating iron oxide at Iris is hematite and the main copper mineral is chalcopyrite. There are some old mine workings at the southern end of the deposit where copper oxides such as brochantite and chrysocolla were mined at surface.
Mineralization at Iris Norte is very similar to the Iris deposit; however, part of the mineralization appears to be hosted by andesitic flows. The deposit is approximately 500 m wide and has been tested over a strike length of 1,600 m. The Iris Norte deposit has been intruded by significant amounts of diorite dykes and sills that separate the deposit into two lenses. The main sulphides are pyrite and chalcopyrite, with the latter providing the copper content of the deposit.

Drilling at the Estrellita deposit has a tabular body of copper mineralization hosted by breccias and mantos along a fault zone around the Estrellita artisanal mine workings. The east–west extent of the Estrellita deposit along the Santo Domingo fault adds up to more than 1,000 m and the deposit remains open in both directions. The mineralization is faulted down by approximately east–west striking faults to the south and north of the main zone around the old workings. Vertical displacement along the faults varies from roughly 60 m to about 100 m. The Estrellita deposit has an as yet unquantified oxide component, consisting of chrysocolla, brochantite and various amorphous copper oxides such as pitch limonite, tenorite and copper wad.

1.6 Drilling, Sampling and Analysis

Prior to Capstone’s Project interest, 348 RC drill holes (90,611 m) and 50 core holes (16,275 m) were completed. Since Project acquisition, Capstone has undertaken an additional 66 core holes (13,282 m).

Most holes are vertical because mineralization at Santo Domingo Sur and Estrellita is horizontal or gently dipping. Inclined holes, particularly diamond holes, were drilled in order to establish the limits of mineralization at the edges of the deposits as well as to establish the structural framework at Estrellita, Iris and Iris Norte.

Drill cuttings and core were logged using a set of pre-set codes. All geological data were entered digitally into summary logs. Geotechnical data were also recorded. Drill collars were located using a differential GPS. Downhole surveying was conducted using a combination of gyroscope and accelerometer, with measurements taken every 10 m.

Reverse circulation drill cuttings were collected at 2 m intervals. Core was nominally sampled at 2 m intervals. Samples for assay were marked at 1 m and 2 m intervals by technicians and subsequently adjusted by the geologist to correspond to major lithological contacts. For programs conducted prior to 2011, sample lengths were not less than 0.5 m and most did not exceed 2 m. The shortest and longest sample lengths in 2011–2012 were 0.7 m and 2.7 m, respectively, and most samples were 2 m long.
The primary analytical laboratory was ALS Chemex, and the facilities in La Serena, Chile and Antofagasta, Chile were used. Both of these facilities have ISO 9001:2008 accreditation and La Serena has ISO 17025 accreditation. Sample preparation consisted of drying, crushing to minus #10 Tyler >70%, homogenizing and then pulverizing to minus #200 Tyler >85%. Samples were analysed for 27 elements via ALS Chemex procedure ME-ICP61, using inductively coupled plasma (ICP). Gold assays were determined using fire assay with an AAS finish. Copper values over 10,000 ppm were re-assayed. Due to the ME-ICP61 method understating the iron content, 7,401 samples from the 2010 drill program were resubmitted for assay using a method with a more aggressive digestion; including all samples over 15% Fe inside the existing block model for which sample material was still available. Soluble copper analysis was conducted on 1,035 samples from 2011–2012 drilling.

A total of 19,302 magnetic susceptibility measurements have been recorded. There are 2,229 density measurements, performed by Far West Mining personnel on core samples using the water displacement method. RPA developed regression formulae based on the specific gravity values reported by Far West Mining to convert volumes to weights, using Fe concentration as the independent variable.

The QA/QC protocols have remained largely consistent throughout all programs conducted by Far West Mining and Capstone. Minor changes have been implemented by Capstone to accommodate issues and recommendations from past programs and to include magnetic susceptibility measurements. Certified reference materials (CRM), or standards, are inserted every 25th sample, constituting 4% of the total number of samples submitted. Blanks, consisting of common Portland cement, were inserted every 50th sample. Field duplicates are taken every 25th sample.

RPA considers that the drilling has been conducted in a manner consistent with standard industry practices. The spacing and orientation of the holes are appropriate for the deposit geometry and mineralization style. Sampling methods are acceptable, meet industry-standard practice, are appropriate for the mineralization style, and are acceptable for Mineral Resource estimation. The quality of the analytical data is reliable, and analysis and security are generally performed in accordance with exploration best practices and industry standards.

1.7 Data Verification

Regular data verification programs have been undertaken by third-party consultants from 2005 to date on the data collected in support of technical reports on the Project.
RPA considers that as a result of this work, the data verification findings acceptably support the geological interpretations and the database quality, and therefore support the use of the data in Mineral Resource estimation.

1.8 Metallurgical Testwork

Metallurgical testwork has been undertaken from 2006 to 2014.

Two separate physical characterisation testwork programs, including semi-autogenous grind (SAG) mill competency (SMC) testwork campaigns, were conducted in order to confirm the throughput rate of the comminution circuit. The complete data set tested was spatially and lithologically representative of the first three years of mining.

As a result of variability testing of Hematite and Magnetite ore types and the dominant proportion of magnetic iron (magnetite), it was decided to modify the comminution flowsheet from a semi-autogenous, ball mill, crushing (SABC) circuit that was used in the pre-feasibility study (PFS) to a direct semi-autogenous, ball mill (DSAG) for the 2014 Feasibility Study. The decision to remove pebble crushing from the design was due to the operational and maintenance complexity of managing the detection and removal of tramp metal from the SAG mill discharge pebble stream. An estimated nominal throughput rate of 65,000 t/d (first five years) and 60,000 t/d for the remaining life of mine (LOM) was determined.

In the testing program completed in June 2011, SGS Lakefield carried out 51 open circuit flotation tests using synthetic sea water and sodium cyanide depression to understand the variability in flotation response associated with the process plant feed. Sample results which were low in total sulphur content, had a significant proportion of soluble copper present, or were classified as being in a waste zone of the pit (23 of the 51 samples) were not considered in the final process evaluation.

A subsequent flotation testing program by SGS Santiago was conducted in 2014. The program objectives were to understand the impact on ultimate copper recovery and copper flotation kinetics using sea water and sodium metabisulphite (SMBS) as a pyrite depressant (replacing sodium cyanide). Testing was completed on composite samples in order to confirm the optimal process flowsheet and conditions. The composite sample testwork was followed by open (OCT) and closed circuit (LCT) variability testing.

The variability testwork results from the 2011 and 2014 programs were used to develop a copper recovery algorithm based on the feed copper grade. The feed head grade of the copper was used to predict the resulting copper recovery using the copper recovery algorithm:
Global Cu Recovery = 1.00 x 0.993 x (2.844 x Ln (Cu%) + 92.15)

The factors used in the model are:

- 1.0 = copper recovery factor for bench scale to full scale
- 0.993 = scaling factor between Cu flotation OCT and LCT tests for all variability and composite sample tests (Average LCT/OCT = 0.993).

Outotec conducted settling testwork on a third (final) cleaner stage copper concentrate generated from the composite known as 8 Years or 8Y which represents the first eight years of mine production. Based on the results of the 8Y tests, the process design for the copper concentrate uses a standard high rate thickener with a settling rate of 0.25 t/h/m² and a P80 of 45 µm concentrate product.

Outotec also completed copper concentrate filtration testwork in 2011 using sea water to determine filtering requirements. Based on the Outotec testwork report, pressure concentrate filters were selected for the process design with a filtration rate of 495 kg/h/m²; a concentrate washing rate of 0.1 m³/t using desalinated water; final concentrate moisture content between 8% and 10%, and a chloride concentration below 300 ppm. Based on the tests, it was determined that when a copper rougher concentrate feed with a regrind size P80 of 45 µm is introduced into the cleaner flotation circuit a final copper concentrate product size P80 of 52 µm is achieved.

Testwork was conducted in 2009, 2010 and 2011 using Davis Tube tests (DT) and low intensity magnetic separation (LIMS) tests by SGA, ALS and CMP to determine the recovery of magnetite from the primary copper flotation tailings stream. The results obtained from LIMS were used as the basis for the design of the recovery of magnetic iron in both the primary magnetic separation step and the subsequent magnetic separator cleaning stages.

Confirmatory DT programs were completed at ALS and CMP using variability samples in 2011 and 2012. Additional DT and LIMS testing was completed by ALS in 2014. This latest testwork was used to confirm the feed regrind size (P80 of 40 µm) for the first, second and third magnetic separator cleaner stages; and to augment the PFS data for the magnetic iron mass recovery to final concentrate relationship. The testwork also provided support for the iron concentrate grades and associated elements.

Using the composite and variability sample testwork results, an algorithm was developed relating magnetic susceptibility values to iron mass recovery. Under the magnetic susceptibility assumptions, Hematite was classified as having magnetic
susceptibility values of between 2,000 and 8,000 and Magnetite was classified as having magnetic susceptibility values of greater than 8,000).

If the magnetic susceptibility is $\geq 2,000$, the algorithm is:

$$\text{Mass Recovery of Fe} = 0.0011 \times (\text{MagSus}) - 3E^{-09} \times (\text{MagSus})^2$$

If the magnetic susceptibility is $< 2,000$, then the mass recovery is measured as zero.

Concentrate quality variability LIMS testwork completed during the PFS and DFS programs indicated an average magnetic iron content exceeding 65% Fe in the magnetite concentrate. This average value of 65% Fe was used with the mass recovery algorithm to determine the total tonnes of magnetite concentrate.

Magnetite settling testwork was performed by Outotec and Delkor in 2012 using bench-scale dynamic thickening equipment. The results of the testwork indicated a magnetite concentrate process design settling rate of 0.678 t/h/m$^2$.

Magnetite concentrate filtration testwork was completed by Outotec in two separate programs in 2011 and 2012. The objective of these tests was to determine the filtration design parameters using Larox pressure filters (PF) and cake washing with desalinated water. This testwork resulted in a final filter cake moisture content of 8% w/w; a maximum filter cake chloride content of less than 300 ppm; and a desalinated wash water consumption of 0.4 m$^3$/t solids. Based on the test results, a filtration rate of 730 kg/m$^2$hr was used for the process design.

Final tailings samples were tested by Outotec and Delkor in 2012 and 2013 to evaluate the settling behaviour of the final tailings and provide thickener design parameters. Based on the results of these tests and associated rheological characterization, a trade-off study was conducted to evaluate capital and operating costs for different thickener configurations.

It was determined that two stages of thickening in series will be the most effective in achieving the desired tailings density. The first stage of thickening will be completed at the process plant with a high rate thickener designed for an underflow density of 55% solids w/w at a settling rate of 0.65 t/h/m$^2$. The second thickening stage will be located at the TSF using two high-density thickeners. The second stage thickening design settling rate will be 0.5 t/h/m$^2$ at an underflow density of 67% solids w/w.

1.9 Mineral Resource Estimation

The Mineral Resource estimates for Santo Domingo Sur, Iris and Iris Norte were competed in 2012. The estimate for Estrellita was conducted in 2007.
RPA constructed 3D wireframe or solid models and gridded surfaces of the mineralized zones, fault structures and topography for use in constraining the block grade interpolations. The principal controls were lithology and structure; however, in some places a nominal grade shell boundary was used. Most zones required construction of wireframes for post-mineral dikes that transect the mineralized mantos. There are also some sequences of barren tuffs that were modelled. A wireframe model was also created to enclose oxidized material which has been demonstrated to yield much lower metallurgical recoveries than the un-oxidized mineralization. A modest amount of underground and open pit mining has been carried out at Estrellita. Far West Mining personnel provided raw cavity monitoring device (CMD) data from which RPA was able to construct approximate wireframe models of the void spaces.

A grade capping strategy was utilized. In total, 24 Cu and 27 Au assay intervals were capped at Santo Domingo Sur, Iris and Iris Norte. These intervals represent approximately 0.2% of the total number of assays. Grades at Estrellita were capped at 3% Cu and 0.3 g/t Au.

Samples from Santo Domingo Sur, Iris and Iris Norte were composited in down-hole intervals of 4 m starting at the contact for each zone and continuing until the hole exited the zone. Drill samples at Estrellita were composited to 2 m lengths, weighted by both length and density.

Grades for Cu, Au, Fe and magnetic susceptibility were interpolated into each block using ordinary kriging (OK) for the Santo Domingo Sur, Iris and Iris Norte deposits. The interpolation was configured to use an ellipsoidal search with a minimum of three and a maximum of 18 composites and a maximum of three composites allowed from any one drill hole. For Estrellita, OK was utilized to interpolate Cu and Au grades into each block. Iron was not estimated. The search was constrained to a minimum of three and maximum of 12 composites, with a maximum of three composites from any one drill hole. Grade interpolations were validated, and no significant errors or biases were noted.

Blocks receiving an estimate for Cu were assigned to at least the Inferred category at Santo Domingo Sur, Iris and Iris Norte. All blocks with an average distance to composites of 200 m or less and for which the nearest composite was within 100 m were classified as Indicated. Within the area of infill drilling completed in 2011–2012, a boundary was drawn around the 50 m drilling pattern and Indicated blocks encompassed by it were nominally assigned to the Measured classification. The final step in the classification was to use the oxide wireframe to tag oxidized blocks and remove these from the Mineral Resources. The classification of Indicated at Estrellita was applied to all blocks estimated by at least two drill holes with the closest composite less than 65 m away. Remaining blocks were classified as Inferred.
RPA ran a pit optimization during 2009 using a Lerchs–Grossmann (LG) algorithm for Santo Domingo Sur, Iris and Iris Norte. Copper equivalent (CuEq) grades were calculated using estimates for recovery, toll treatment/refinement charges (TC/RC) and transport costs for each metal and based on the operating cost estimates contained in the 2008 Preliminary Assessment. At the 0.25% CuEq cut-off, all but 5% of the Mineral Resources were captured by the pit shell. On the basis of this result, it was concluded that there was little merit in restricting the Mineral Resources to those blocks contained only within the pit shell. In RPA’s opinion, the shape and depth of the Mineral Resources have not changed since the previous estimate and it is still valid to consider them as having reasonable prospects of economic extraction by open pit mining.

The Estrellita resource estimate is not constrained within a LG shell. RPA’s opinion was that a 0.3% Cu cut-off would be appropriate for the reporting of the estimate. At the time of the estimate in 2007, RPA considered that the 0.3% Cu cut-off was similar to that used in other operations of similar size and grade.

1.10 Mineral Resource Statement

The Mineral Resource estimates and geological models were prepared by Mr David Rennie, P.Eng., an employee of RPA. Mr. Rennie is the Qualified Person as defined under NI 43-101 for the estimate. Mineral Resources for Santo Domingo Sur, Iris and Iris Norte have an effective date of August 31, 2012. Mineral Resources estimated for Estrellita have an effective date of 30 October 2007. Mineral Resources in Table 1-1 are reported inclusive of Mineral Reserves.

Risk factors that could potentially affect the Mineral Resources estimates include the following: long-term commodity price and exchange rate assumptions, changes in the assumptions used in the LG shell constraining Mineral Resources at Santo Domingo Sur, Iris, and Iris Norte, the assumed mining methods and cost assumptions for the Santo Domingo Sur, Iris, and Iris Norte deposits being those from the 2008 Preliminary Analysis not those arising from the Feasibility Study, no LG shell being employed to support reasonable prospects at Estrellita, delays or other issues in reaching agreements with local communities, changes in permitting, surface rights and environmental assumptions.
Table 1-1: Mineral Resource Estimates

<table>
<thead>
<tr>
<th>Mineral Resource Estimates</th>
<th>Mt</th>
<th>%CuEq</th>
<th>%Cu</th>
<th>g/t Au</th>
<th>%Fe</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Measured</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Santo Domingo Sur (1–4)</td>
<td>63.3</td>
<td>0.95</td>
<td>0.62</td>
<td>0.083</td>
<td>31.3</td>
</tr>
<tr>
<td>Iris (5–6)</td>
<td>1.54</td>
<td>0.46</td>
<td>0.43</td>
<td>0.052</td>
<td>25.3</td>
</tr>
<tr>
<td><strong>Total Measured</strong></td>
<td>64.8</td>
<td>0.94</td>
<td>0.62</td>
<td>0.082</td>
<td>31.2</td>
</tr>
<tr>
<td><strong>Indicated</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Santo Domingo Sur (1–4)</td>
<td>214</td>
<td>0.72</td>
<td>0.33</td>
<td>0.045</td>
<td>27.4</td>
</tr>
<tr>
<td>Iris (5–6)</td>
<td>111</td>
<td>0.63</td>
<td>0.19</td>
<td>0.028</td>
<td>26.0</td>
</tr>
<tr>
<td>Iris Norte (7–8)</td>
<td>92.3</td>
<td>0.67</td>
<td>0.12</td>
<td>0.015</td>
<td>26.7</td>
</tr>
<tr>
<td><strong>Subtotal Indicated (Santo Domingo Sur /Iris)</strong></td>
<td>417</td>
<td>0.68</td>
<td>0.25</td>
<td>0.033</td>
<td>26.9</td>
</tr>
<tr>
<td>Estrellita</td>
<td>31.7</td>
<td>n/a</td>
<td>0.53</td>
<td>0.050</td>
<td>n/a</td>
</tr>
<tr>
<td><strong>Total Indicated</strong></td>
<td>449</td>
<td>—</td>
<td>0.27</td>
<td>0.034</td>
<td>25.0</td>
</tr>
<tr>
<td><strong>Total Measured and Indicated</strong></td>
<td>514</td>
<td>—</td>
<td>0.31</td>
<td>0.040</td>
<td>25.8</td>
</tr>
<tr>
<td><strong>Inferred</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Santo Domingo Sur (1–4)</td>
<td>29.8</td>
<td>0.55</td>
<td>0.26</td>
<td>0.037</td>
<td>23.6</td>
</tr>
<tr>
<td>Iris (5–6)</td>
<td>5.05</td>
<td>0.60</td>
<td>0.18</td>
<td>0.024</td>
<td>26.7</td>
</tr>
<tr>
<td>Iris Norte (7–8)</td>
<td>20.5</td>
<td>0.70</td>
<td>0.08</td>
<td>0.009</td>
<td>28.0</td>
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<tr>
<td><strong>Subtotal Inferred (Santo Domingo Sur /Iris)</strong></td>
<td>55.4</td>
<td>0.61</td>
<td>0.19</td>
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<td>25.5</td>
</tr>
<tr>
<td>Estrellita</td>
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<td>0.050</td>
<td>n/a</td>
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<td><strong>Total Inferred</strong></td>
<td>58.1</td>
<td>—</td>
<td>0.20</td>
<td>0.026</td>
<td>24.3</td>
</tr>
</tbody>
</table>

Notes to Accompany Mineral Resource Table:

1. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. The Qualified Person for the estimates is Mr David Rennie, P.Eng., an employee of Roscoe Postle Associates Inc.
4. Mineral Resources for the Santo Domingo Sur, Iris, and Iris Norte deposits are reported using a cut-off grade of 0.25% copper equivalent (CuEq). CuEq grades are calculated using average long term prices of US$3.50/lb Cu, US$1,500/oz Au and US$1.94/dmt Fe (US$120/dmt conc. at 62% Fe). The CuEq equation is: Metal Value = Grade*Cm*R%/100*(Price-TCRC-Freight)*(100-Royalty)/100, where Cm is a constant to convert grade of metal m to metal price units; R is metallurgical recovery and %Cu Equivalent = (Cu Value + Au Value + Fe Value)/(Cu Value per 1%Cu)
5. An assessment of Mineral Resources for the Santo Domingo Sur, Iris, and Iris Norte deposits was performed using a Lerchs–Grossman pit shell that has the following assumptions: pit slopes averaging 45°; mining cost of US$1.19/t, processing cost of US$4.49/t; processing recovery of 85%; selling price of US$2.25/lb, and a selling cost of US$0.247/lb. At the 0.25% CuEq cut-off, all but 5% of the Mineral Resources were captured by the pit shell. On the basis of this result, it was concluded that there was little merit in restricting the Mineral Resources to those blocks contained only within the pit shell. Accordingly, the Mineral Resource inventory was reported in its entirety.
6. Mineral Resources for the Estrellita deposit are reported using a cut-off grade of 0.3% Cu.
7. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.
8. Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.
1.11 Mineral Reserve Estimates

Pit optimization, mine design and mine planning were carried out by NCL using the 2012 block model prepared by RPA and did not include consideration of material classified as Inferred. Inferred Mineral Resources were treated as waste. A block size of 12.5 m E x 12.5 m N x 12 m RL was selected for the block model. The selected block size was based on the geometry of the domain interpretation and the data configuration.

The mining cost estimate for the pit optimization process is based on studies developed by NCL during 2012. The estimated average project mining cost was separated into various components such as fuel, explosives, tires, parts, salaries and wages, benchmarked against similar current operations in Chile. Each component was updated for first-quarter 2013 prices and the exchange rate from Chilean Pesos to US dollars. This resulted in an estimated mining cost of approximately $1.53/t. The metal prices, processing costs, refining costs, and processing recoveries were provided to NCL by Capstone.

A number of calculations were performed in the model in order to determine the net smelter return (NSR) of each individual block. The internal (or mill) cut-off of $7.84/t milled incorporates all operating costs except mining. This internal cut-off is applied to material contained within an economic pit shell where the decision to mine a given block was determined by the pit optimization, and was applied to all of the Mineral Reserve estimates. Marginal ore was calculated for the same $7.84/t cut-off, but for a NSR determined at higher metal prices.

Final slope angles used for the pit optimization process were a result of multiple iterations and analysis carried out by the NCL mining team and geotechnical specialists Derk Ingeniería y Geología Ltda (Derk).

The original block model was based on an ore percentage with dimensions of 12.5 m x 12.5 m x 12 m, resulting in a 1,875 m³ block volume; this means that every block has a defined “ore” portion with an ore density, and a corresponding “waste” portion with a waste density. To accommodate selective mining methods, any resource block with an ore percentage that was <10% was treated as waste. Blocks with an ore percentage that was higher than 90% were diluted with waste such that all high-ore blocks were considered to contain only 90% ore. Selective mining therefore will be performed on those blocks that have an ore percentage of between 10% and 90%.
1.12 Mineral Reserves Statement

Mineral Reserves are summarized in Table 1-2 and have an effective date of 2 May, 2014. The Qualified Person for the estimate is Mr Carlos Guzman, CMC, an NCL employee.

In the opinion of the NCL QP, the main factors that may affect the Mineral Reserves estimate are metallurgical recoveries and operating costs (fuel, energy and labour). NCL notes that the base price, as well as changes in the price of metals, even though this is the most important factor for revenue calculation, does not affect the Mineral Reserves estimate to any significant degree.

A revenue factor of 0.86 was used for the LG shell that was employed as the guide for the practical design for both the Santo Domingo and Iris Norte pits. This selected revenue factor is conservative and as such allows for a broad swing in metals pricing before any salient effect on the Mineral Reserves estimate will occur.

1.13 Mine Plan

A mine plan was developed for the Santo Domingo Project to process 60,000 t/d to 65,000 t/d of feed (21.9 to 23.7 Mt/a) with a peak total mining rate of 107.5 Mt/a in Years 1 to 4. Because of the softer characteristics of the initial feed (higher copper content and lower magnetite), an initial period of five years was scheduled for a plant feed of 65 kt/d. From Year 6 the plant throughput is scheduled for 60 kt/d. Year 1 feed to the plant is made up of material mined during pre-production and Year 1. Oxide material has been identified and will be stockpiled separately.

Mill throughput was also restricted to a magnetite concentrate production capacity of a maximum 4.5 Mt/a up to Year 10; and 5.4 Mt/a from Year 11.

The mine is scheduled to work seven days per week, 365 days per year. Each day will consist of two 12 hour shifts. Four mining crews will cover the operation.

The final pit design was based on a Lerchs–Grossmann (LG) shell that used a copper price of $2.75 per pound and US$80 per tonne for magnetite concentrate. Two pits, the Santo Domingo pit and the Iris Norte pit, were designed. The Santo Domingo pit will have four phases; three mining phases are planned for the Iris Norte pit.
Table 1-2: Mineral Reserve Statement

<table>
<thead>
<tr>
<th>Reserve Category</th>
<th>Ore Grade</th>
<th>Contained Metal</th>
<th>Stage</th>
<th>Stage</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td>Ore (Mt)</td>
<td>(%</td>
</tr>
<tr>
<td>Proven Mineral Reserves</td>
<td></td>
<td></td>
<td>Santo Domingo</td>
<td>65.3</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>Iris Norte</td>
<td>—</td>
</tr>
<tr>
<td>Total Proven Mineral Reserves</td>
<td></td>
<td></td>
<td>65.3</td>
<td>0.61</td>
</tr>
<tr>
<td>Probable Mineral Reserves</td>
<td></td>
<td></td>
<td>Santo Domingo</td>
<td>251.6</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>Iris Norte</td>
<td>74.8</td>
</tr>
<tr>
<td>Total Probable Mineral Reserves</td>
<td></td>
<td></td>
<td>326.4</td>
<td>0.24</td>
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<tr>
<td>Total Mineral Reserves (Proven and Probable)</td>
<td></td>
<td></td>
<td>Santo Domingo</td>
<td>316.9</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td>Iris Norte</td>
<td>74.8</td>
</tr>
<tr>
<td>Total Mineral Reserves (Proven and Probable)</td>
<td></td>
<td></td>
<td>391.7</td>
<td>0.30</td>
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</tbody>
</table>

Notes to Accompany Mineral Reserves Table

1. Mineral Reserves have an effective date of 2 May, 2014 and were prepared by Mr Carlos Guzman, CMC, an employee of NCL.
2. Mineral Reserves are reported as constrained within Measured and Indicated pit designs, and supported by a mine plan featuring variable throughput rates and cut-off optimization. The pit designs and mine plan were optimized using the following economic and technical parameters: metal prices of US$2.75/lb Cu, US$1,275/oz Au and US$80/dmt of Fe concentrate; recovery to concentrate assumptions of a maximum of 93.6% for Cu and 75% for Au, with magnetite concentrate recovery varying on a block-by-block basis; copper concentrate treatment charges of US$70/dmt, US$0.07/lb of Cu refining charges, US$5.0/oz of Au refining charges, US$48/wmt and US$3/wmt for shipping Cu and Fe concentrates respectively; waste mining cost of $1.53/t, mining cost of US$1.53/t ore, and process and G+A costs of US$7.84/t processed; average pit slope angles that range from 37.6º to 43.6º; a 2% royalty rate assumption, and an assumption of 100% mining recovery.
3. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content
4. Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.
In the Santo Domingo pit, the Phase 1 targets the material with the highest grade and lowest strip ratio in the central area, down to 892 m elevation. Phases 2 and 3 are successive expansions to the north, down to 772 m and 736 m elevation, respectively. Phase 4 in Santo Domingo is in the area called Iris which is at the north of the Santo Domingo pit, but has a separate access to the east side and will go down to 676 m elevation.

Three mining phases were designed in Iris Norte, as successive expansions from south to north, going down to 736 m, 724 m and 664 m elevation, respectively. Each phase has access from the east and west sides.

The Santo Domingo pit will have two exits on the west side providing access to the ROM pad area and the primary crusher. On the east side there will be another exit to access the main waste storage area. The final pit will be 2,200 m long in the north–south direction and 1,500 m wide in the east–west direction. The pit bottom will be at the 676 m elevation and the highest wall will be about 552 m on the southeastern side. The total area to be disturbed by the pit is approximately 229 ha.

The Iris Norte pit will have one exit on the west side providing access to the run-of-mine (ROM) pad area and the primary crusher. On the east side there will be an exit to access the waste storage area. The final pit will be 1,600 m long in the north–south direction and 900 m wide in the east–west direction. The pit bottom will be at the 664 m elevation, and the highest wall will be about 315 m on the north side. The total area to be disturbed by the pit is about 124 ha.

Mine equipment requirements were calculated based on the annual mine production schedule, the mine work schedule and equipment hourly production estimates. The study is based on operating the mine with 42 m³ capacity hydraulic excavators (shovels) and trucks with a capacity of 290 t. The fleet will be complemented with drilling rigs for ore and waste. Auxiliary equipment will include tracked dozers, wheel dozers, motor graders and a water truck. A small drill rig was also included for presplitting purposes.

1.14 Recovery Plan

1.14.1 Crushing and Grinding

The primary crushing plant will receive run-of-mine (ROM) feed directly from the open pits. The crusher is designed to allow two 290 t trucks to discharge directly into the crusher dump pocket which has a rated capacity of 450 t. The crushed product will be conveyed to the coarse ore stockpile which has a live capacity of 13,121 t (equivalent to six to eight hours of operation). The SAG mill will operate in a DSAG/SAB mode.
Oversize pebbles from the SAG mill discharge screen will be recycled to the SAG mill. The SAG mill product will discharge onto a conventional vibrating double deck screen. The SAG mill discharge screen undersize will flow into a common SAG and ball mill discharge pump box and is pumped in parallel using two separate feed pumps to two separate batteries of hydrocyclones. The hydrocyclone oversize (or underflow) fraction will return by gravity to the balls mills for further size reduction.

1.14.2 Copper Flotation

The hydrocyclone overflow streams (the copper flotation circuit feed stream) with a P80 of 180 µm will be combined and fed to a single bank of conventional forced air tank type flotation cells. Combined flotation rougher concentrate produced from all of the rougher cells will flow by gravity to the rougher concentrate regrind stage. The rougher flotation stage tailings will be pumped to magnetic separation. Combined rougher/scavenger concentrate flows by gravity to the concentrate regrinding stage will consist of a single vertical mill and hydrocyclone battery, operating in closed circuit. The concentrate stream will first be classified by the hydrocyclones with the coarse underflow fraction reporting back to the vertical mill. Ground product from the vertical mill will be returned to the feed distribution box where it is combined with fresh concentrate feed.

The overflow from the regrind hydrocyclones will feed a single conditioning tank before it feeds the first cleaner flotation which will be carried out in a single bank of conventional forced air tank type flotation cells. The concentrate produced will be pumped to the second cleaning flotation stage. The first cleaning flotation scavenger stage will be carried out in a single bank of conventional forced air tank type cells. The scavenger concentrate slurry will flow by gravity back to the original concentrate regrinding stage where it will be combined with fresh rougher circuit flotation concentrate. Primary cleaner circuit scavenger tailings will report to the final tailings stream where it will be combined with magnetite recovery circuit tailings. The second stage of flotation cleaning will be performed in a single bank of conventional forced air tank cells with the concentrate slurry produced flowing by gravity to feed the third bank of flotation cleaning cells; the tailings produced will be pumped back to the first cleaning flotation stage. The third cleaning flotation stage will be conducted in a single bank of conventional forced air tank cells. The concentrate slurry from the third stage of cleaning which is the final copper concentrate product will be pumped to the copper concentrate thickening stage. The tailings from the third flotation stage will be recirculated by pump to the second cleaning stage feed. The copper concentrate thickener underflow will discharge at 60% solids w/w and will be pumped to the copper concentrate filter section.
1.14.3 Magnetic Separation

Tailings from the primary rougher flotation stage (the magnetite circuit feed) will be pumped to a central distribution box with pneumatic dart valves feeding two lines each with five individual primary LIMS (1,000 gauss low intensity magnetic separators) magnetic drum separators operating in parallel. It is planned that initially 10 LIMS drum separators will be installed with space for additional LIMS to be installed later if required to accommodate higher grade magnetite feed.

The rougher magnetic drums will operate in parallel to maximize the rougher concentrate iron grade. The rougher magnetic concentrate from each magnetic drum line will be sent to grinding and classification; the rougher magnetic concentration tailings will report to the main plant tailings stream. Hydrocyclone overflow from the magnetite concentrate grinding and classification circuit at a P80 of 40 μm will be sent to cleaner magnetic separation.

The cleaning circuit magnetic LIMS concentrator will consist of two parallel lines each with three LIMS drum separators operating in a counter-current configuration to facilitate high selectivity. The final magnetite concentrate produced will be pumped to the magnetite concentrate thickener and the tailings from the cleaner magnetic stage will be combined with rougher LIMS tailings and will be sent to the final tailings stream. Overflow water from the concentrate thickener will report to the main process water pond and thickener underflow from the magnetite concentrate thickener at 65% solids w/w will be pumped to the concentrate transport system.

1.14.4 Tailings Thickening

The first stage of tailings thickening (pre-thickening) will be conducted at the process plant and the second stage (final thickening) is conducted at the TSF area. Pre-thickening of tailings will be done in high rate thickeners which deliver tailing thickener underflow at 55% solids w/w. Recovered water from the thickeners will be pumped back to the process water pond. Thickened tailings will be pumped to the TSF area.

1.14.5 Copper Concentrate Filtration

Copper concentrate will be filtered in one filter press fed by a dedicated pump. During the concentrate filtration washing stage, desalinated water will replace the sea water contained in the copper concentrate cake to reduce the chloride content to less than 300 ppm. Copper concentrate filter cake will discharge by gravity to the copper concentrate stockpile.
1.14.6 High Density Tailings and Tailings Transport

Final tailings thickening will be carried out in two high density thickeners which produce a 67% solids w/w discharge. The water recovered from the thickeners will be stored in a tank; some water will be pumped to filtration and used as dilution water for flocculant preparation and some water will flow by gravity to the process water pond. The thickened tailings discharge will be pumped to a tank at the TSF and will flow by gravity into the TSF.

1.14.7 Plant Infrastructure

A reverse osmosis (RO) plant will be installed at the process plant to desalinate sea water. A portion of the brine product produced will be used for road watering and the remainder will be sent to the process pond. The potable water supply for internal consumption and to supplement the potable water supply for the town of Diego de Almagro will also be produced from the desalinated water by chlorination.

Compressed air will be supplied from the compressed air plant consisting of four 200 kW compressors, one accumulator and one dryer, and one accumulator for the instrument air.

1.14.8 Port Infrastructure

There will be a filter plant at the port for magnetite concentrate. Magnetite concentrate will be received at the port in an agitated storage tank and then pumped directly to the filter plant to obtain a magnetite concentrate with a moisture content of 8% w/w. There will be four horizontal filter presses with two pumps feeding these filter presses. Desalinated water will be used in the washing stage to replace the sea water contained in the filtered cake to reduce the chloride content to less than 300 ppm. The magnetite concentrate filter cake product will discharge onto a conveyor feeding the concentrate transfer tower and then the magnetite concentrate stockpile.

An RO plant in the port area will prepare water for magnetite concentrate washing. The brine produced will be returned to the sea. Potable water will be produced from the desalinated water by chlorination. Instrument air and plant air will be provided via distribution ring main systems.

1.14.9 Production Plan

The production plan obtained from the Mine Plan and the metallurgical models for Cu and Fe recovery assumes yearly average treatment rates of 65,000 t/d and 60,000 t/d, with an annual production limit of 494,000 t of copper concentrate and an annual
production limit for magnetite concentrate of 4,050,000 t for the first six years of production and 5,400,000 t for the remaining mine life.

In Years 0 and 1 the Hematite reaches the maximum treatment rate within the plan (about 33.6% of the total processed in the year). The maximum treatment rate of Magnetite is close to 90% in Year 18. In some periods the plant could process more tonnage than projected.

The head grade varies between 0.68% Cu and 0.42% Cu during the first five full years of production. After the fifth full year the head grade varies between 0.37% Cu and 0.07% Cu. At the end of the mine life the head grade is only 0.06% Cu. For the first five full years the head grade is about 30% Fe, with an average of 27% Fe with little variation over the LOM.

1.15 Infrastructure

1.15.1 Planned Facilities

The principal Project facilities are planned to be located at the following sites:

- Santo Domingo plant site: Located at approximately 26°28’00”S and 70°00’30”W
- Permanent Camp: Located adjacent to the town of Diego de Almagro
- Concentrate pipeline: 111.6 km long from the Santo Domingo plant site to the port site at Punta Roca Blanca
- Santo Domingo Port facilities: Located about 43.5 km north of Caldera at Punta Roca Blanca (Puerto Santo Domingo)
- Sea water pipeline: 111.6 km long from Puerto Santo Domingo pump station to the Santo Domingo plant site.

1.15.2 Access Considerations

The planned route for transporting cargo, staff and equipment to the Santo Domingo site is from the south by Route C-17, and from the north by Route C13.

The closest commercial airport to the Santo Domingo site is the El Salvador Airport, 44 km from the site. The next closest commercial airport is the Desierto Atacama Airport, 113 km south from Chañaral.

The planned port for transport and shipment of heavy machinery, equipment and materials is Punta Angamos in Mejillones, Antofagasta Region, 520 km from the plant.
site. This port operates throughout the year and is accessed directly from Route 5 North.

Ghisolfo was engaged to prepare a study for the overland transport of copper concentrate from the mine to the proposed Santo Domingo port. The approximate distance of the haulage is 118.5 km using the preferred route from the completed haulage study.

The proposed route for the concentrate and sea water pipelines (pipeline route) was defined during earlier studies and modified for the 2014 Feasibility Study to allow for a change in the port location. In addition, the pipeline routing was revised to by-pass a proposed tailings storage facility at the Mantoverde copper operation, being developed by third-parties. The pipeline route was optimized using a single right-of-way (RoW) and a common trench for both pipelines.

1.15.3 Waste Rock Storage Facilities

Three waste rock storage facilities (WRFs) were designed at the west and south of the open pits. The WRFs were designed in 50 m lifts. Each lift is constructed at the approximate angle of repose of 37°. A 75 m set-back between every lift maintains the overall angle at 22° to facilitate reclamation and long term stability. A constant 2.0 t/m³ loose density was assumed.

Based on a waste characterisation study undertaken internally by Capstone, Capstone has concluded that the WRFs show a moderate to low potential for generation of acid rock drainage. As a result, no significant acid generation is expected by Capstone from the mined waste, and the dry climate conditions are also not expected by Capstone to produce sufficient water to generate drainage through the waste rock facilities to mobilize any acid solutions.

1.15.4 Stockpile Facilities

During the pre-production period, the ROM pad area will be constructed close to the initial pit for later re-handling to the primary crusher. The total ore to be stockpiled during this period amounts to 0.5 Mt.

The marginal ore stockpile and the oxide stockpile will be located in areas between the Santo Domingo and Iris Norte pits. The stockpiles are designed with 20 m lifts and 30 m set-backs in order to facilitate later re-handling.
1.15.5 Tailings Disposal Facility

The Tailings Storage Facility (TSF) embankment will be constructed from compacted, non-acid generating mine waste rock. A 1.5 mm thick HDPE geomembrane liner will be installed on the upstream face of the dam wall, placed over a geotextile and a 3 m thick bedding layer. The geomembrane will be extended beneath the area of the supernatant pond and 100 m further to reduce seepage from the pond. The pond will remain in direct contact with the upstream face of the embankment throughout the operating life of the facility. During future detailed design and water balance calculations, it may be necessary to lower density slurry parameters which may require expansion of the lined area. The embankment will be constructed in stages using the downstream method. A 26.7 m high saddle dam will be constructed at the southwest limits of the TSF to provide containment at that location. A perimeter channel will also be constructed to reduce the ingress of stormwater runoff into the TSF during the operational stage. The feasibility study contemplates that the TSF will store 314 Mt of copper and iron tailings, equivalent to an estimated total volume of 196 Mm³. These tailings will be deposited over 18 years.

Monitoring of the facility will utilize a fiber optic monitoring system for the embankment and will include piezometers, clinometers, inclinometers, accelerometers, and three existing monitoring wells and a pumping well which were drilled during field investigations. If any impacted water is detected in the monitoring wells, it is estimated that up to three additional pumping wells will be installed downstream of the main embankment. The water eventually recovered will be either treated for release or returned to the TSF for recycling in the process. After Year 10, all ground water flows from the TSF, if occurs, should be intercepted by the Iris Norte well system that will be installed to dewater the pit.

The tailings will be pumped from the plant as conventional slurry (55% by mass solids content) to thickeners located at the southern end of the TSF. After thickening to approximately 67% solids content, the tailings will be deposited using the sub-aerial deposition method. Liquid solids separation will occur on the beach and the liberated water will flow downslope to the supernatant pond from where it will be recycled to the process plant.

Closure of the TSF will include installing an emergency spillway and treatment of the final tailings beach to reduce the generation of dust and prevent run-on of surface water onto the tailings.
1.15.6  Port

The port of Puerto Santo Domingo will be located in the Punta Roca Blanca area which is located between Caleta Hornos and Punta Choros, in the Atacama Region. The maximum required annual port capacity is 5.5 Mt/a of magnetite concentrate and 0.52 Mt/a of copper concentrate. Ship sizes envisaged range from Handymax to Cape size. The nominal loading rate will be 4,000 t/h for magnetite concentrate and 2,000 t/h for copper concentrate.

Copper concentrate will be delivered to the port by concentrate haul trucks. Magnetite concentrate will be delivered by pipeline.

Offshore facilities included in the port design comprise an access trestle, take up tower, berthing dolphins, mooring dolphins, a quadrant beam (the structure that allows the ship loader radial movement), ship loader, and the ship loader support platform.

Onshore facilities include the terminal station of the concentrate pipeline, storage tanks and filter plant for magnetite concentrate; a copper concentrate storage building, a magnetite concentrate stockpile, sea water intake, integrated building (offices, laboratories, change house and lunch room), guard checkpoint, workshop and warehouse, and ancillary facilities to support the operation.

1.15.7  Power and Electrical Supply

The Project facilities requiring power will be located at the following sites:

- Mine and plant site located near Diego de Almagro. The mine and plant site area includes the mine, process plant, infrastructure and tailings facility electrical loads

- Santo Domingo Port at Punta Roca Blanca. The port facilities include the sea water pump station, desalination plant, magnetite concentrate filtration plant, concentrate storage and handling, and associated infrastructure

The maximum (peak) demand during operations estimated to be 111.7 MW. The estimated average demand during operations will be approximately 85.7 MW. The average demand and power consumption for the mine site and the port are calculated on the basis of treating between 60,000 t/d and 65,000 t/d over a mine life of approximately 18 years and an annual average availability of 93%.

The mine site and port site will be connected to the Central Interconnected System (Sistema Interconectado Central, SIC) which covers the central part of Chile. The closest connection point between the SIC and the mine site is via a direct connection to the Diego de Almagro substation, located about 5 km from the proposed mine area.
Based on independent market analysis research provided to Capstone, Chile is likely to have a difficult situation regarding power availability for the next five years. However, recent studies support that measures being currently undertaken to provide grid interconnections and to expand power generation, are likely to ameliorate this situation after the projected five-year bottleneck. In the Norte Chico area of the SIC system where Santo Domingo is located, there is currently a lack of generation and there are power line restrictions that do not allow power from the south-central area of the SIC to be transmitted to the Norte Chico area. Consequently, the marginal costs in this part of the SIC system are currently the highest in Chile and this is affecting the development of mining projects in the area.

Capstone has been looking for a power contract at competitive prices over the last several years with little success. The Project currently does not have a power purchase agreement (PPA) in place. The present strategy is to continue to investigate and follow up on reputable developments in the power sector in Chile. This monitoring and investigation will allow Capstone to take advantage of any opportunity that may arise for the supply of power for the Project.

Current solar project developments in the area may provide an opportunity for Capstone to obtain power for the project at affordable prices. Discussions with some of these project owners are providing some promising opportunities for the Project and will continue to be monitored.

1.16 Marketing

Capstone requested Cliveden Trading AG (CTAG) and CRU to prepare papers on price projections, sales potential and shipping costs for the copper and iron ore concentrates to be produced by Minera Santo Domingo over the life of mine.

Kores is required to purchase 50% of the annual production of copper and iron ore concentrates produced by the Project. Capstone will market and sell the remaining 50%. The Kores terms and conditions will reflect the Capstone terms negotiated independently in the market.

The Capstone copper concentrate would generally be considered clean. For trading companies specializing in blending various complex copper concentrates outside of China a clean concentrate such as that from Capstone would be in high demand in the opinion of CTAG. The timing to secure sales contracts would be entirely dependent on the progress of financial arrangements for Project financing. It is likely that banks or financial institutions would want to have signed Letters of Intent (LOIs) or Memorandums of Understanding (MOUs) from smelters followed by full long term contracts as a condition for the completion of financing.
Capstone will produce a high magnetite ultra-fine (UF) iron ore concentrate and will need to shortlist a number of potential pellet plants that can process their iron ore concentrate as a starting point of a campaign to contract suitable long-term off-takers. There is likely to be strong competition to supply these pellet plants in the years ahead from other mines producing UF iron ore concentrate. CTAG recommended that Capstone makes advances at an early stage with a number of potential buyers in order to have meaningful MoUs in place that would eventually be developed into long-term off-take contracts.

Each steel mill complex has its own level of tolerance in terms of impurities. The main levels of impurities as far as the magnetite concentrate is concerned are silica and copper. Copper is below the threshold but may in some circumstances represent a non-preferred feed; silica is only likely to be a cost factor or penalty element rather than a rejectable quality issue.

Using the CRU predictions, on average from 2016 through to, and including, 2025 (i.e. 10 years) copper prices are expected to average $3.13 per pound. CTAG agrees that this is a realistic price to be used as an average over this period. However, for the purposes of the economic analysis in this Report, a copper price of $2.85/lb was used.

CRU estimates that prices for 62% Fe content sinter fines (IODEX) CFR Qingdao delivery (deemed the standard product for CFR China delivery) can be expected to decline on average over the next 10 years, reaching a long-term price of approximately $114 per wmt by 2024. CTAG is of the same opinion as CRU that prices will decline over the next four to five years to long term trend rates, likely to be around the $114 per wmt for 62% Fe, CFR Qingdao, China basis, or equivalent to $83 to $88 FOB North Chile (when adjusted for freight). A base case price of $85 per wmt was used in the economic analysis.

### 1.17 Environmental, Permitting and Social Considerations

#### 1.17.1 Baseline Studies

Baseline studies were carried out for communities in the area of influence of the Project: Diego de Almagro, Inca de Oro, El Salado, Chañaral, Flamenco, Torres del Inca, Obispito and Caldera.

Physical environment baseline studies included characterization of climate, meteorology (wind, rain, humidity, solar radiation, temperature, atmospheric pressure and evaporation), air quality (inhalingable particulate material (PM10), inhalingable fine particulate material (PM2.5)), sedimentable particulate material (SPM), gases, noise
and vibration, geology, geomorphology, natural hazards, soils, hydrology and hydrogeology.

The marine environment baseline studies included characterization of the physical environment (currents, tides and dispersion studies), chemical (sub-tidal sediment and water quality), and biological (zooplankton, phytoplankton, and benthic fauna).

The biotic environment baseline studies addressed the fauna and flora components of the Project in both the port and planned mine site areas.

The anthropological environment baseline studies for the port and proposed mine site included the description of human component, constructed environment, cultural heritage and palaeontology and landscape issues. For the description of the human component five aspects were studied; geographic, demographic, socio-economic, anthropological and basic social well-being.

Baseline studies were also completed to address current water resources. Based upon the study results, no impacts to local water resources are anticipated as the Project will use sea water for the mining process. In addition, no infiltration is expected from the WRF or thickened tailings deposit to the groundwater resources in the area.

Baseline studies encompassing the marine environment were completed. The Project includes a seawater intake and brine outfall to the sea in the port area. In the engineering design of the sea water system, environmental criteria have been used that minimize the environmental impact from the seawater intake or brine outfall. To support the design and operation of the sea water system, the Project will implement an Environmental Monitoring Plan during operations to monitor the performance of the sea water system, compare against any anticipated environmental impacts, and confirm the environmental management mitigation measures.

Four key areas of risk were identified from the completed baseline studies, as follows:

- **Water:**
  - Alteration of the surface water flow and drainage patterns
  - Alteration in the underground water flow and/or water quality
- **Air quality:**
  - Increases in the levels of breathable particulate material (PM10), breathable fine particulate material (PM2.5) and gases (primarily as a result of wind activity on stockpiles, dust generation from construction and mining activity and material transport)
  - Increases in levels of sedimentable particulate matter
• Marine environment:
  – Potential disruption to benthic communities due to the operation of the sea water intake and brine discharge systems and port construction activities
• Human environment:
  – Effects of the Project on the current lifestyles of local communities

Studies were completed to identify potential mitigation measures to address the recognized risks. Mitigations proposed include, but are not limited to, community liaison and development programs, construction of settlement by-pass roads, implementation of zero-discharge facilities, and reviews of and modifications to infrastructure designs to accommodate community and environmental concerns.

1.17.2 Closure

A provisional closure plan has been developed for the Project and will be updated following EIA approval. The following is a summary of the key items of the provisional closure plan:

• All materials that can be recycled will be identified and sent to the appropriate recycling facilities
• Process plant, mine and port, reagents and supplies will be removed and will be returned, sold, disposed of in approved facilities or transported for certified disposal
• Electrical equipment, cables and other above-ground facilities will be dismantled or demolished
• All foundations will be demolished and covered
• Where berms and walls were constructed, these will be re-graded to approximate pre-construction land contours
• If soil or other contamination is detected, remediation will be completed
• Access to areas such as the open pits, waste rock facilities and the tailings facility will be restricted and warning signs posted
• Final pit and WRF slopes will be re-profiled to be structurally stable for the long term
• Re-profiling of the TSF will be carried out and the embankment spillway will be constructed.
For the closure plan, the main identified risks are:

- Not updating the closure plan to include required modifications, updates and revised closing actions during the life of mine
- Not complying with the commitments in the closure plan.

Closure costs are treated in the financial analysis as operating costs, and total $92.1 M. The closure costs are accrued on an annual basis and treated as expenses in the year in which they are spent.

1.17.3 Social Considerations

A stakeholder identification study has been completed, and has identified a number of parties will be either directly or indirectly affected by Project influence. A number of communication sessions were undertaken during 2012 and 2013, and included open houses and meetings, sessions to address specialist interests (such as fishermen); meetings with regional authorities, community support service authorities, and professional organizations.

Community issues identified during these meetings include:

- Job opportunities for local residents during the construction and operation phases of the Project
- Decreased quality of life due to increased demand for goods and services, housing, and health services, due to the arrival of workers linked to the Project
- Environmental effects related to mining activities
- Changes to road usages due to by-pass construction and concentrate transport
- Effect of the proposed port facilities on seafood extraction activities
- Effects of seawater intake and brine discharge from the desalination plant.

During the EIA the environmental citizen participation (PAC) process as required by the evaluation process will continue. The citizen participation process with indigenous communities takes into account the special rules that govern the consultation and participation processes of such peoples. Although the lands of the Colla Community of Diego de Almagro are not within the direct area of Project influence, Capstone will keep lines of communication open for possible approaches or inquiries from this community.

The communications strategy for the Santo Domingo Project will continue to focus on the building of a positive reputation and supportive environment for project
development in the Atacama Region. Specific development strategies are focusing on the communities of Diego de Almagro and Caldera. A communications plan, communications committee and crisis response management plan are being developed.

1.17.4 Permitting

The Project requires the construction of works and installations for operations facilities for mining, waste rock and tailings disposal, processing in the concentrator plant, concentrate transport by pipeline to the shipping port, and port facilities. The Project also includes the construction of support works and facilities such as the camp; service support (guard houses, lunchrooms, first aid rooms, waste storage); sea water intake, desalination plant and pumping of sea water; power lines; and roads (internal and access). This work requires the identification, preparation, submission and approval of environmental and sectorial permits. These permits are essential for the construction and operation of the project works and facilities.

To date, 752 permits have been identified that will be required to support operations. Fifteen of these permits are considered to be on the critical path for timely construction and start-up of the Project.

1.18 Capital Cost Estimates

All capital costs are in third-quarter 2013 US$. An exchange rate of 480 CLP to US$1 was used as the foreign exchange rate for the detailed estimate (excluding mine equipment and services). A foreign exchange rate of 500 CLP to US$1 was used to estimate the mine equipment and services costs.

Capital cost estimates were prepared by the various consultants working on the 2014 Feasibility Study and were based on battery limits established by Capstone. Owner costs were provided by Capstone. Estimates were based on a combination of direct quotes and benchmarking. The estimate is a Type 3 estimate according to AMEC standards (and the Association for the Advancement of Cost Engineering International, AACE), with an accuracy of -10 to +15% at the 85% confidence level.

The initial capital cost was estimated at $1,751 M. The estimated sustaining capital cost totals $376.3 M. The combined initial and sustaining capital costs for the life of mine were estimated to be $2,127 M in total (Table 1-3).
Table 1-3: Initial Capital Cost Estimate

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<td>2,127.0</td>
</tr>
</tbody>
</table>

Note: Costs in this table are summarized by major area, and may include costs from consultants, AMEC, Capstone, or all relevant parties.

1.19 Operating Cost Estimates

The costs are presented in Q3 2013 US dollars. Costs are based on an exchange rate of CLP 500 to US$1.00 for mining and CLP 480 to US$1.00 for processing. For the copper equivalent estimate, prices of $2.85/lb copper and $85.00/t magnetite concentrate were used.

The operating cost estimate is considered to be at a feasibility-study level, with an accuracy of -10% to +15%.

Operating costs are summarized in Table 1-4. The total operating cost over the projected life-of-mine is $5,760.9 M.
Table 1-4: Operating Cost Estimate

<table>
<thead>
<tr>
<th>Cost Centre</th>
<th>LOM Total (MUS$)</th>
<th>LOM Average (US$/t)</th>
<th>LOM Average (US$/lb CuEq)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Process</td>
<td>2,753.4</td>
<td>7.03</td>
<td>0.607</td>
</tr>
<tr>
<td>Copper Concentrate Transport</td>
<td>54.5</td>
<td>0.14</td>
<td>0.012</td>
</tr>
<tr>
<td>G &amp; A</td>
<td>439.6</td>
<td>1.12</td>
<td>0.097</td>
</tr>
<tr>
<td>Mining</td>
<td>2,513.4</td>
<td>6.42</td>
<td>0.555</td>
</tr>
<tr>
<td>Total</td>
<td>5,760.9</td>
<td>14.71</td>
<td>1.271</td>
</tr>
</tbody>
</table>

1.20 Foreign Exchange Rates

For purposes of the Project capital and operating cost estimates, a fixed foreign exchange rate between Chilean Pesos (CLP) and US dollars (US$) was initially used. However, during the estimate development, the foreign exchange rate between the CLP and US$ changed appreciably. To accommodate this change Capstone completed an update to the foreign exchange rate for the operating and capital cost estimates.

- For an updated foreign exchange rate for the development period from 2014 through 2017, Capstone used the mean value of the projected CLP to US$ foreign exchange rate from a total of 29 analyst firms compiled by Bloomberg (as of 6 May 2014).

- For an updated foreign exchange rate for the operating period from 2018 through 2035, Capstone used an algorithm that was developed using the CLP/US$ exchange rate value versus the market sales price of copper. This information was gathered over the last 10 years on a daily basis and resulted in the following algorithm:
  - CLP/US$ Exchange Rate = -0.0204 (price of Cu in US$/t) + 660.41

  For the 2014 Feasibility Study copper price of US$2.85 (US$6,281/t), this equates to a CLP/US$ rate of 532.

The exchange rate assumptions are detailed in Table 1-5.
Table 1-5: Foreign Exchange Rate Assumptions

<table>
<thead>
<tr>
<th>Cost Estimate Item</th>
<th>Initial Foreign Exchange Rate (CLP/US$)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Initial Capital Cost Estimate (excluding mine equipment)</td>
<td>480</td>
</tr>
<tr>
<td>Sustaining Capital Cost Estimate (excluding mine equipment)</td>
<td>480</td>
</tr>
<tr>
<td>Process Operating Cost Estimate</td>
<td>480</td>
</tr>
<tr>
<td>G&amp;A and Copper Hauling Operating Cost Estimate</td>
<td>480</td>
</tr>
<tr>
<td>Initial Mine Equipment Capital Cost Estimate</td>
<td>500</td>
</tr>
<tr>
<td>Sustaining Mine Equipment Capital Cost Estimate</td>
<td>500</td>
</tr>
<tr>
<td>Mine Operating Cost Estimate</td>
<td>500</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Cost Estimate Item (Revised by Year/Period)</th>
<th>Revised Foreign Exchange Rate (CLP/US$)</th>
</tr>
</thead>
<tbody>
<tr>
<td>2014</td>
<td>553</td>
</tr>
<tr>
<td>2015</td>
<td>557</td>
</tr>
<tr>
<td>2016</td>
<td>517</td>
</tr>
<tr>
<td>2017</td>
<td>519</td>
</tr>
<tr>
<td>2018 through 2035 (Operating Period)</td>
<td>532</td>
</tr>
</tbody>
</table>

1.21 Restatement of Operating and Capital Cost Estimates

In order to reconcile the estimates that were modified as a result of the updated foreign exchange rate assumptions (and which were used in the financial analysis) with the values in the detailed estimate back up that were completed using the previous 480 CLP/US$ and 500 CLP/US$ foreign exchange rate, a comparison table is included as Table 1-6.

For the copper hauling operations and G&A there were no impacts as these values were originally estimated in US$. 
### 1.22 Economic Analysis

The results of the economic analysis to support Mineral Reserves represent forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

Forward-looking statements include, but are not limited to, statements with respect to future metal prices and concentrate sales contracts, the estimation of Mineral Reserves and Mineral Resources, the realisation of Mineral Reserve estimates including the achievement of the dilution and recovery assumptions, the timing and amount of estimated future production, costs of production, capital expenditures, costs and timing of the development of ore zones, permitting time lines, requirements for additional capital, government regulation of mining operations, environmental risks, unanticipated reclamation expenses, and title disputes.

Additional risk can come from actual results of reclamation activities; conclusions of economic evaluations; changes in Project parameters as mine and process plans continue to be refined, possible variations in ore reserves, grade or recovery rates; geotechnical considerations during mining; failure of plant, equipment or processes to operate as anticipated; shipping delays and regulations; accidents, labour disputes and other risks of the mining industry; and delays in obtaining governmental approvals.
The Project was evaluated using an 8% discounted cash flow (DCF) analysis on an after tax basis. Metal prices used were $2.85/lb for copper, $1,275/oz for Au, and $85/t for iron (assuming 65% Fe content).

On an after tax basis, the cumulative net cash flow for the base case is US$3,226.7 M, the IRR is 17.9% and the payback period is 4.2 years. At an 8% discount rate, the after tax net present value (NPV) of the project is $797.4 M. A cashflow summary table is included as Table 1-7. The life-of-mine cashflow is shown in Figure 1-1.

Cash costs are summarized in Table 1-8. The Au and Fe credits fully offset the operating costs, resulting in a negative C1 cash cost.

1.23 Sensitivity Analysis

A sensitivity analysis was performed on the financial model taking into account variations in:

- Metal price
- Operating costs (including power)
- Power supply costs alone
- Capital costs.

The analysis shows that the Santo Domingo Project NPV8% is most sensitive to changes in metal price. The sensitivity analysis showed that the project is less sensitive to changes in operating costs; less sensitive still to capital expenditure changes; and least sensitive to changes in power costs. Figure 1-2 shows the sensitivity of the IRR and Figure 1-3 shows the sensitivity of the NPV8% to the variations imposed in the parameters listed above.
Table 1-7: Summary of Cash Flow

<table>
<thead>
<tr>
<th>Cost Item</th>
<th>Unit</th>
<th>LOM</th>
<th>Per tonne milled</th>
<th>Per lb Cu payable</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Revenue (after losses and before deductions)</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cu</td>
<td>kUS$</td>
<td>6,521,912</td>
<td>16.65</td>
<td>2.95</td>
</tr>
<tr>
<td>Au</td>
<td>kUS$</td>
<td>363,312</td>
<td>0.93</td>
<td>0.16</td>
</tr>
<tr>
<td>Fe</td>
<td>kUS$</td>
<td>6,382,181</td>
<td>16.29</td>
<td>2.89</td>
</tr>
<tr>
<td><strong>Sub-Total</strong></td>
<td>kUS$</td>
<td>13,267,404</td>
<td>33.87</td>
<td>6.01</td>
</tr>
<tr>
<td><strong>Smelting costs</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Treatment</td>
<td>kUS$</td>
<td>(268,447)</td>
<td>(0.69)</td>
<td>(0.12)</td>
</tr>
<tr>
<td>Cu deduction</td>
<td>kUS$</td>
<td>(228,267)</td>
<td>(0.58)</td>
<td>(0.10)</td>
</tr>
<tr>
<td>Au deduction</td>
<td>kUS$</td>
<td>(36,331)</td>
<td>(0.09)</td>
<td>(0.02)</td>
</tr>
<tr>
<td>Refining – Cu</td>
<td>kUS$</td>
<td>(165,622)</td>
<td>(0.42)</td>
<td>(0.08)</td>
</tr>
<tr>
<td>Refining – Au</td>
<td>kUS$</td>
<td>(1,282)</td>
<td>(0.00)</td>
<td>(0.00)</td>
</tr>
<tr>
<td>Transport</td>
<td>kUS$</td>
<td>(272,230)</td>
<td>(0.69)</td>
<td>(0.12)</td>
</tr>
<tr>
<td><strong>Sub-Total</strong></td>
<td>kUS$</td>
<td>(972,180)</td>
<td>(2.48)</td>
<td>(0.44)</td>
</tr>
<tr>
<td><strong>Operating costs</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Mining</td>
<td>kUS$</td>
<td>(2,471,905)</td>
<td>(6.31)</td>
<td>(1.12)</td>
</tr>
<tr>
<td>Process</td>
<td>kUS$</td>
<td>(2,725,682)</td>
<td>(6.96)</td>
<td>(1.23)</td>
</tr>
<tr>
<td>G&amp;A</td>
<td>kUS$</td>
<td>(439,567)</td>
<td>(1.12)</td>
<td>(0.20)</td>
</tr>
<tr>
<td><strong>Sub-Total</strong></td>
<td>kUS$</td>
<td>(5,637,154)</td>
<td>(14.39)</td>
<td>(2.55)</td>
</tr>
<tr>
<td><strong>Other</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Royalty</td>
<td>kUS$</td>
<td>(245,904)</td>
<td>(0.63)</td>
<td>(0.11)</td>
</tr>
<tr>
<td>Closure</td>
<td>kUS$</td>
<td>(92,077)</td>
<td>(0.24)</td>
<td>(0.04)</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td>kUS$</td>
<td>(337,982)</td>
<td>(0.86)</td>
<td>(0.15)</td>
</tr>
<tr>
<td><strong>Earnings before interest, taxes, depreciation, and amortization (EBITDA)</strong></td>
<td>kUS$</td>
<td>6,320,089</td>
<td>16.13</td>
<td>2.86</td>
</tr>
<tr>
<td><strong>Construction capital</strong></td>
<td>kUS$</td>
<td>(1,699,773)</td>
<td>(4.34)</td>
<td>(0.77)</td>
</tr>
<tr>
<td><strong>Sustaining capital</strong></td>
<td>kUS$</td>
<td>(368,419)</td>
<td>(0.94)</td>
<td>(0.17)</td>
</tr>
<tr>
<td><strong>Undiscounted margin (cumulative net cash flow)</strong></td>
<td>kUS$</td>
<td>4,251,897</td>
<td>10.85</td>
<td>1.93</td>
</tr>
</tbody>
</table>

Note: * = Capital & operating costs are stated using 2014 exchange rate updates (see Section 1.20 and Section 1.21)
Figure 1-1: Cashflow Summary

Table 1-8: Cash Cost Summary

<table>
<thead>
<tr>
<th>Cash Costs</th>
<th>Years 1–5 (Excludes 2017) (kUS$)</th>
<th>Cost per tonne milled (US$/t)</th>
<th>Cost per pound Cu payable (US$/lb)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cash Costs Years 1–5</td>
<td>2,200,955</td>
<td>18.61</td>
<td>1.84</td>
</tr>
<tr>
<td>Au and Fe Credits</td>
<td>(1,609,809)</td>
<td>(13.61)</td>
<td>(1.35)</td>
</tr>
<tr>
<td>Adjusted Cash Costs Total</td>
<td>591,145</td>
<td>5.00</td>
<td>0.49</td>
</tr>
<tr>
<td>Cash Costs LOM</td>
<td>6,609,334</td>
<td>16.87</td>
<td>2.99</td>
</tr>
<tr>
<td>Au and Fe Credits</td>
<td>(6,745,492)</td>
<td>(17.22)</td>
<td>(3.05)</td>
</tr>
<tr>
<td>Adjusted Cash Costs Total</td>
<td>(136,159)</td>
<td>(0.35)</td>
<td>(0.06)</td>
</tr>
</tbody>
</table>
Figure 1-2: Sensitivity of IRR (Spider Chart)

Note: Figure prepared by AMEC, 2014

Figure 1-3: Sensitivity of NPV8% for Base Case (Spider Graph)

Note: Figure prepared by AMEC, 2014
1.24 Risk and Opportunity Analysis

Following a risk and opportunity analysis conducted as part of the 2014 Feasibility Study, the most significant risks facing the Project include:

- Imposition of new taxes or royalties
- Increases in capital costs
- Lack of appropriate human resources
- The infrastructure and utilities required to support the Project are not secured
- Community opposition
- Electrical supply infrastructure and capacity not sufficient to meet Project demand
- Lack of water during construction.

Project opportunities are planned to be reviewed during more detailed Project design studies.

1.25 Conclusions

Under the assumptions used in the 2014 Feasibility Study, the Project shows positive economics. The QPs consider that the scientific and technical information available on the Project can support proceeding with detailed mine design phase studies. However, the choice to proceed to a mining decision on the Project is at Capstone’s discretion.

1.26 Recommendations

A single work phase is proposed for the Project and is broken down by QP recommendation. Work can be performed concurrently, and no program is dependent on the results of any of the others. Overall the program estimate has a total range from approximately $1.6 M to $2.6 M.

RPA recommends $75,000–$80,000 of work, comprising an update to the Mineral Resource estimate for the Estrellita deposit, and better characterization of the ore types in the model for metallurgical purposes.

AMEC has recommended $1.1 M to 2 M of work, which includes supporting studies for more detailed geotechnical designs for the planned open pits, collection of site-specific sea condition data for the Santo Domingo port area to provide metrics for the actual port availability and support for detailed dock design and concentrate loading and
storage arrangements, and bench-scale and pilot-scale metallurgical testwork to optimize plant operation and potentially reduce plant-related operating costs.

BRASS has recommended $335,000 of work, consisting of soil characterization testing, hydrological surveys of water crossing areas, and additional slurry testing to better simulate the final slurry product.

Knight Piésold has recommended between $85,000 and $175,000 of work, including additional tailings testwork, testing of the construction material that is to be included in the TSF design, and integration of these results into the TSF design. Knight Piésold has also recommended further analysis to update geomembrane liner limits within the TSF impoundment. Depending on the results of the liner limit analysis, there could be as much as an additional $2 M increase on the construction costs to account for any liner size increases.
2.0 INTRODUCTION

AMEC International Ingeniería y Construcción Limitada (AMEC) was commissioned by Capstone Mining Corporation (Capstone) to prepare an independent Qualified Person's Review and NI 43-101 Technical Report (the Report) for the Santo Domingo Project (the Project), located in the Third Region of Chile. The Project location is shown in Figure 2-1.

The Report is based on the results of a feasibility study (the 2014 Feasibility Study) on the Project completed in 2014. The 2014 Feasibility Study included, in addition to input from Capstone, contributions from a number of engineering and technical firms, including AMEC; BRASS Chile SA (BRASS; sea water and magnetite concentrate pipelines); Cliveden Trading AG (CTAG; metals marketing); CRU Group (CRU; metal price forecasts); Ghisolfo y Cia Ingenieria de Consulta Ltda (Ghisolfo; road by-pass design and copper concentrate transport study); Knight Piésold S.A. (KP; tailings storage facility (TSF) and environmental studies); NCL Ltda (NCL; Mineral Reserves, geotechnical design, open pit designs, and waste rock facilities); PRDW Chile (PRDW; port materials handling, concentrate storage and ship-loading facilities); and Roscoe Postle Associates Inc (RPA; geological interpretation and Mineral Resource estimates).

The firms and consultants who are responsible for the content of this Report, which is based on the 2014 Feasibility Study, are, in alphabetical order, AMEC, BRASS, KP, NCL, and RPA.

2.1 Terms of Reference

The Report will be used in support of Capstone's press release dated 4 June, 2014, entitled “Capstone Mining Reports Positive Feasibility Study Results for Santo Domingo Project in Chile”.

The Project is held 70% by Capstone and 30% by Korea Resources Corporation (Kores). The companies use an operating entity, Minera Santo Domingo SCM (Minera Santo Domingo), as the Chilean holding company for the Project. For the purposes of this Report, as Capstone is the Project operator and the major shareholder, Minera Santo Domingo and Capstone are used interchangeably.
Figure 2-1: Project Location Plan

Note: Figure courtesy Capstone, 2013. Map north is to the top of the plan. As an indicator of the map scale, the distance between Diego de Almagro and El Salado is approximately 30 km.

All measurement units used in this Report are metric, and currency is expressed in US dollars unless stated otherwise. The Report uses Canadian English.

Years discussed in the mine and production plan, and in the financial analysis are presented for illustrative purposes only, as no decision has been made on mine construction by Capstone, and the relevant permits for the Project remain to be secured.
2.2 Qualified Persons

The following serve as the qualified persons for this Technical Report as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects, and in compliance with Form 43-101F1:

- Ms Joyce Maycock, P.Eng., Project Manager, AMEC Santiago
- Mr Hans Gopfert, CMC., Principal Engineer, AMEC Santiago
- Mr David Rennie, P.Eng., Principal Geologist, RPA Vancouver
- Mr Carlos Guzman, CMC., Principal and Project Director, NCL Santiago
- Mr David Frost, F.AusIMM, Technical Director, Process, AMEC Santiago
- Mr Tom Kerr, P.Eng., President, Knight Piésold and Co. (USA)
- Mr Roy Betinol, P.Eng., Manager, BRASS Chile
- Ms Anna Klimek, P.Eng., Manager, Port and Marine Group, AMEC Vancouver

2.3 Site Visits and Scope of Personal Inspection

The following QPs have visited site.

Mr Rennie visited the Project from June 14–16, 2010 and again from June 14–15 2012. During the visits, Mr Rennie reviewed the geological setting of the deposits and viewed drilling operations and drill core. Mr Rennie’s site visit is still considered current as there has been no drilling activity on the Project since 2012.

Mr Guzman visited the Project site on 15 October 2013. During this visit he inspected the area planned for the mine and process infrastructure to assess topography, and reviewed the layout and general site overview with respect to mine planning and execution. He also viewed Project drill core.

Mr Kerr visited the Project site on 24 October 2013. During this visit he viewed the area proposed for the tailings storage facility.

2.4 Effective Dates

The Report has a number of effective dates as follows:

- Date of supply of last assay data used in resource estimation: 30 June 2012
2.5 Information Sources and References

The key information source for the Report was the 2014 Feasibility Study, entitled:


Information used to support this Report was also derived from previous technical reports on the Project, and from the reports and documents listed in the References section. Additional information was sought from Capstone personnel where required.

Mr Tom Kerr, the KP QP, has relied upon information supplied by Solange Gantenbein, Environmental Manager of the Knight Piésold S.A., Santiago, Chile, office on environmental, permitting and social issues that was used in support of the information presented in Section 20 of the Report. Information from Mrs. Gantenbein was further reviewed on behalf of Mr Kerr by Chris Brodie, R.P.Bio., Environmental Manager of the Knight Piésold office in Vancouver, Canada. Mr. Luis Rebolledo, Project Director Engineering for Knight Piésold S.A., Santiago, Chile, was relied upon by Mr Kerr for aspects of the tailings facility engineering design and cost estimates pertaining to the tailings facilities and the estimated closure and reclamation costs.

Mr Roy Betinol, the BRASS QP, has relied upon information provided by Mr Jose Escarate, Civil Engineer with BRASS; this information is used in Section 18.2.4 and Section 18.9. Mr Escarate visited the site from May 14 to May 17, 2013 to select and define the proposed route for the water and concentrate pipelines. A second visit was made by Mr Escarate on June 4, 2013 to review the planned routing as modifications had been made to avoid critical areas. This specialist information on the pipeline route was provided to Mr Betinol for use in the Report.
Mr Hans Gopfert of AMEC has relied upon information provided by Ghisolfo staff on the proposed transport route for copper concentrate; this information is used in Section 18.2.3. Three Ghisolfo staff, Mr Franscisco Ghisolfo, Mr Alvaro Fernandez, and Mr Daniel Pizarro, visited the Project area from 27 to 30 June 2013, and reviewed the routes planned for transport of copper concentrate through the towns of Diego de Almagro, El Salado and Chañaral. This specialist information on the concentrate transport route was provided to Mr Gopfert for use in the Report.

### 2.6 Previous Technical Reports

Capstone has filed the following technical reports on the Project:


Far West Mining Ltd, a predecessor company to Capstone, filed the following reports:


- **Allen, G.J., and Höy, T., 2005: Exploration Activities in the 4a (Santo Domingo) Area of the Candelaria Project, Region III, Chile: unpublished report prepared by Far West Mining Ltd., effective date 10 December 2005**

- **Penner, R., Lacombe, P., Maycock, T., and Henry, E., 2008: Review of the Santo Domingo Sur and Iris Project, Region III, Chile: unpublished report prepared by AMEC International (Chile) S.A. or Far West Mining Ltd., effective date April 30, 2008**


3.0 RELIANCE ON OTHER EXPERTS

The QPs have relied upon the following other expert reports, which provided information regarding mineral rights, surface rights, property agreements, and marketing sections of this Report as noted below.

3.1 Project Ownership

The QPs have not reviewed the Project ownership, nor independently verified the ownership legal status. The QPs have fully relied upon, and disclaim responsibility for, information derived from Capstone experts through the following document:

- Capstone, 2013: Legal Opinion, NI 43-101: Email from Capstone to AMEC, 28 November 2013

This information is used in Section 4.2 of the Report. It is also used in support of the Mineral Resource statement in Section 14.11, the Mineral Reserve statement in Section 15.3, and the financial analysis result in Section 22.3.

3.2 Mineral Tenure, Rights of Way and Easements

The QPs have not reviewed the mineral tenure, nor independently verified the legal status, ownership of the Project area, underlying property agreements or permits. The QPs have fully relied upon, and disclaim responsibility for, information derived from Capstone experts and experts retained by Capstone for this information through the following documents:

- Maria Elizabeth Orrego Espinosa (Abogados): Propiedad Minera - Minera Santo Domingo SCM: legal opinion prepared for Minera Santo Domingo, 15 October 2013
- Maria Elizabeth Orrego Espinosa (Abogados): Informe De Titulos Concesiones Mineras MLO hoy MSD: legal opinion prepared for Minera Santo Domingo, 15 October 2013
- Maria Elizabeth Orrego Espinosa (Abogados): Terrenos Superficiales - Servidumbres Mineras - Minera Santo Domingo SCM: legal opinion prepared for Minera Santo Domingo, 15 October 2013
3.3 Taxation

The QPs have fully relied upon, and disclaim responsibility for, information supplied by Capstone staff and experts retained by Capstone for information related to taxation as applied to the financial model as follows:

- Ernst and Young, 2014: Minera Santo Domingo Model Review – Final Report Tax Aspects: confidential powerpoint presentation prepared by Ernst and Young for Minera Santo Domingo, dated 16 June 2014

This information is used in the financial model in Section 22 of the Report.

3.4 Markets

The QPs have fully relied upon, and disclaim responsibility for, information supplied by Capstone staff and experts retained by Capstone for information related to price projections, sales potential and shipping costs for copper and iron ore concentrates as follows:


This information is used in Sections 19 and 22 of the Report, and in support of the Mineral Reserve statement in Section 15.3 and the process marketing assumptions in Section 13.

Metals marketing is a specialized business requiring knowledge of supply and demand, economic activity and other factors that are highly specialized and requires an extensive database that is outside of the purview of a QP.

The QPs consider it reasonable to rely upon Cliveden Trading AG (CTAG), as CTAG is a professional marketing and trading company that has been operating for more...
than 50 years. The company offers the mining sector a full suite of advisory and logistical implementation services for all non-ferrous mine products, including operations, shipping, hedging and marketing.

3.5 Metal Pricing

The QPs have relied upon information supplied by Capstone staff and experts retained by Capstone for information related to the metal pricing for copper, treatment and refining charges, and iron ore concentrate (62% Fe content sinter fines) as follows:


This information is used in Sections 19 and 22 of the Report, and in support of the Mineral Reserve statement in Section 15.3 and the process marketing assumptions in Section 13.

Metals price forecasting is a specialized business requiring knowledge of supply and demand, economic activity and other factors that are highly specialized and requires an extensive global database that is outside of the purview of a QP.

The QPs consider it reasonable to rely upon CRU, as the company is an independent company specializing in market analysis of major commodities for the mining, metals and fertilizer sectors.

The AMEC QPs were able to view CTAG’s review of the CRU report. Overall the CTAG review supports the CRU conclusions as to metal price forecasts.
4.0 PROPERTY DESCRIPTION AND LOCATION

The Santo Domingo Project is located approximately 5 km southeast of the town of Diego de Almagro in the Atacama Region of northern Chile.

The Santo Domingo area was originally part of the BHP’s Candelaria area which consisted of eight non-contiguous concessions along a north–south corridor that stretched between the towns of Taltal in the north to a point about 75 km south of the city of Copiapó.

The centre of the Santo Domingo property is approximately 398,000E and 7,074,000N (datum: PSAD 56, Zone 19S).

4.1 Property and Title in Chile

Information in this subsection is based on data in the public domain (Ortuzar, 2013; Vergara and Mackenna, 2011; Barriga, 2011, Fraser Institute, 2014), and has not been independently verified by the QPs.

4.1.1 Regulations

The mining industry is regulated by the following laws:

- Constitution of the Republic of Chile
- Constitutional Organic Law of Mining
- Code and Regulations governing Mining
- Code and Regulations governing Water Rights
- Laws and Regulations governing Environmental Protection as related to mining.

Chile’s mining policy is based on legal provisions that were enacted as part of the 1980 constitution. These were established to stimulate the development of mining and to guarantee the property rights of both local and foreign investors. According to the law, the state owns all mineral resources, but exploration and exploitation of these resources by private parties is permitted through mining concessions, which are granted by the courts.

The concessions grant both rights and obligations, as defined by the Constitutional Organic Law on Mining Concessions (JGRCh, 1982) and the Mining Code (JGRCh, 1983). Many of the steps involved in the constitution of the mining concession are published weekly in Chile’s official mining bulletin for the relevant region as are court processes due to conflicting claims.
4.1.2 Mineral Tenure

The concessions have both rights and obligations as defined by a Constitutional Organic Law (enacted in 1982). Concessions can be mortgaged or transferred and the holder has full ownership rights and is entitled to obtain the rights of way for exploration (pedimentos) and exploitation (mensuras). In addition, the concession holder has the right to defend ownership of the concession against state and third parties. A concession is obtained by a claims filing and includes all minerals that may exist within its area.

Mining rights in Chile are acquired in the following stages.

**Pedimento**

A pedimento is an initial exploration claim whose position is well defined by UTM coordinates which define north-south and east-west boundaries. The minimum size of a pedimento is 100 ha and the maximum is 5,000 ha with a maximum length-to-width ratio of 5:1.

The duration of validity is for a maximum period of two years; however, at the end of this period, and provided that no overlying claim has been staked, the claim may be reduced in size by at least 50% and renewed for an additional two years. If the yearly claim taxes are not paid on a pedimento, the claim can be restored to good standing by paying double the annual claim tax the following year.

New pedimentos are allowed to overlap with pre-existing ones; however, the underlying (previously-staked) claim always takes precedent, providing the claim holder avoids letting the claim lapse due to a lack of required payments, corrects any minor filing errors, and converts the pedimento to a manifestacion within the initial two-year period.

**Manifestacion**

Before a pedimento expires, or at any stage during its two-year life, it may be converted to a manifestacion or exploration concession.

Within 220 days of filing a manifestacion, the applicant must file a “Request for Survey” (Solicitud de Mensura) with the court of jurisdiction, including official publication to advise the surrounding claim holders, who may raise objections if they believe their pre-established rights are being encroached upon.

A manifestation may also be filed on any open ground without going through the pedimento filing process.
The owner is entitled to explore and to remove materials for study only (i.e. sale of the extracted material is forbidden). If an owner sells material from a manifestation or exploration concession, the concession will be terminated.

**Mensura**

Within nine months of the approval of the “Request for Survey” by the court, the claim must be surveyed by a government licensed surveyor. Surrounding claim owners may be present during the survey.

Once surveyed, presented to the court, and reviewed by the National Mining Service (Sernageomin), the application is adjudicated by the court as a permanent property right (a mensura), which is equivalent to a “patented claim” or exploitation right.

Exploitation concessions are valid indefinitely, and are subject to the payment of annual fees. Once an exploitation concession has been granted, the owner can remove materials for sale.

**Claim Processes**

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

Many of the steps involved in establishing the claim are published in Chile’s official mining bulletin for the appropriate region (published weekly). At the manifestacion and mensura stages, a process for resolution of conflicting claims is allowed.

Most companies in Chile retain a mining claim specialist to review the weekly mining bulletins and ensure that their land position is kept secure.

Legislation is being considered that seeks to further streamline the process for better management of natural resources. Under the new proposed law, mining and exploration companies will have to declare their reserves and resources and report drilling results. The legislation also aims to facilitate funds for mining projects across the country. In addition to the mining law, the Organic Constitutional Law on Mining Concessions (1982) and the Mining Code of 1983 are the two key mechanisms governing mining activities in Chile.
4.1.3 Surface Rights

Ownership rights to the subsoil are governed separately from surface ownership. Articles 120 to 125 of the Chilean Mining Code regulate mining easements. The Mining Code grants to the owner of any mining exploitation or exploration concessions full rights to use the surface land, provided that reasonable compensation is paid to the owner of the surface land.

4.1.4 Rights of Way

The Mining Code also grants the holder of the mining concession general rights to establish a right of way, subject to payment of reasonable compensation to the owner of the surface land. Rights of way are granted through a private agreement or legal decision which indemnifies the owner of the surface land. A right of way must be established for a particular purpose and will expire after cessation of activities for which the right of way was obtained. The owners of mining easements are also obliged to allow owners of other mining properties the benefit of the right of way, as long as this does not affect their own exploitation activities.

There is a mining right-of-way tax that provides license protection and is calculated as a percentage of the Unidad Tributaria Mensual (UTM or monthly tax unit) and applies to each hectare of land included in the mining right of way and for mining exploitation concessions. This tax is paid annually in a single payment before 31 March of each year.

For mining rights-of-way the tax rate is currently 10% of a UTM per hectare; for mining exploration concessions the tax rate is currently 2% of a UTM per hectare. The value of the UTM is adjusted monthly according to the consumer price index (IPC) in Chile.

4.1.5 Water Rights

Article 110 of the Chilean Mining Code establishes that the owner of record of a mining concession is entitled, by operation of law, to use waters found in the works within the limits of the concession, as required for exploratory work, exploitation and processing, according to the type of concession the owner might engage in. These rights are inseparable from the mining concession.

Water is considered part of the public domain and is considered to be independent of the land ownership. Individuals can obtain the rights to use public water in accordance with the Water Code. In accordance to the Code (updated in 1981) water rights are expressed in litres per second (L/s) and usage rights are granted on the basis of total water reserves.
4.1.6 Environmental Regulations

Environmental impact statements are required for projects such as dams, thermoelectric and hydroelectric plants, nuclear power plants, mining, oil and gas, roads and highways, ports, development of real estate in congested areas, water pipelines, manufacturing plants, forestry projects, sanitary projects, production, storage and recycling of toxic, flammable and hazardous substances. Developments not covered by these categories must submit a sworn statement of environmental impact indicating that the project or activity does not affect the environment and does not violate environmental laws. All projects must be approved by the national or regional environmental commission.

On 12 August 2013 the Official Gazette published the new Regulations for the System of Environmental Impact Assessment (Reglamento del Sistema de Evaluación de Impacto Ambiental, RSEIA). In general terms the new regulation updates the assessment procedure in accordance with the legal and regulatory changes enacted in Chile from 2001 to date. It redefines the information that must be submitted when entering an Environmental Impact Statement (EIA) or an Environmental Impact Declaration (DIA), seeking to give greater certainty to those regulated and to the citizens. The RSEIA seeks to make assessments early, to raise the standard of information and evaluation, and to reduce time to complete the process. The changes are consolidated in Law 19.300, especially with regard to public participation in EIAs. Indigenous consultation is included for projects entering the system, complying with ILO Agreement 169 in force in Chile since 2009.

4.1.7 Land Use

Chile’s zoning and urban planning are governed by the General Law of Urban Planning and Construction (Ley General de Urbanismo y Construcción). This law contains several administrative provisions that are applicable to different geographical and hierarchical levels and sets specific standards for both urban and inter-urban areas.

In addition to complying with the Environmental Law (Ley Ambiental) and other legal environmental requirements, projects must also comply with urban legislation governing the different types of land use. Land use regulations are considered part of the Chilean environmental legal framework.

Land use regulatory requirements are diverse and operate at different levels, the main instruments are the inter-community regulatory plans (Planes Reguladores Intercomunales, PRI) and the community regulatory plans (Planes Reguladores...
Comunales, PRC). The IRPs regulate territories of more than one municipality, including urban and rural territory.

4.1.8 Closure Considerations

Law 20.551, Law of Mine Closure, enacted in October 2011, took effect in November 2012 and imposes on the mining industry the obligation to execute closure of its operations, incorporating closure as part of the life cycle of a mining project.

To comply with these regulations, the owner of the project must, prior to starting a mining operation, submit a closure plan to Sernageomin, with an approval procedure that depends on the mine capacity. The first procedure is applicable to mining projects with a mine capacity greater than 10,000 tonnes per month. A simplified procedure is allowed for projects with a mine capacity equal to or less than 10,000 tonnes per month or which are exploration projects.

The differences between these procedures are in the type of information required to be submitted for evaluation of the closure plan. The closure plans for the larger operations must provide more detailed information and must also provide a monetary guarantee to ensure the full and timely compliance with the closure plans. The guarantee must cover the costs of the measures associated with closure and post-closure. Each five years, to comply with the plan, the execution of any closure activities and an update of the closure plan must be audited as a complementary instrument of control by Sernageomin. For smaller mining projects or exploration projects that are subject to the simplified procedure, no financial guarantee is required and no audit of the closure plan is required.

The following are the requirements for the guarantee:

- The amount of the guarantee must cover the total value of the cost for the closure plan including post-closure, and is determined by an estimate of the current costs of the plan. The guarantee is periodically updated.

- The guarantee must be paid in full within the first two-thirds of the estimated life of the project if less than 20 years, or within a period of 15 years if the estimated life of the project is more than 20 years.

- The payment of the guarantee must begin within the first year of the start of operations, and the value must be equal to 20% of the total closure cost. From the second year on, the payment must be proportional to the period which remains for the complete amount. The guarantee increases until the total value of the closure costs is deposited. The instruments of guarantee must be liquid and easy to execute.
The financial guarantee can be gradually released as the closure plan is executed. Once the closure is complete and a certificate of final closure is issued by Sernageomin all guarantees will be released.

Mining companies that are obliged to provide a guarantee have a period of two years to estimate the cost of the closure plan. The closure plan must be approved under the regulation of Mining Safety Regulations and Environmental Qualification Resolution (RCA). After this period the company must submit the cost of executing the closure plan as well as the guarantee to Sernageomin. Sernageomin will then confirm that the company is in compliance.

4.1.9 Foreign Investment

In Chile, foreign investors may own 100% of a company based in Chile with no limit of duration for property rights. Within the limits of the Chilean law, investors can undertake any type of economic activity.

Potential foreign investors have two legal mechanisms for bringing their capital into Chile. These are alternative and parallel mechanisms which, in both cases, serve to register the entry of foreign capital. Foreign investors can freely choose one of these alternatives when materializing an investment: Chapter XIV of the Chilean Central Bank’s Compendium of Foreign Exchange Regulations or the Foreign Investment Statute established in Decree Law 600 (D.L. 600).

- Chapter XIV of the Chilean Central Bank: Under this administrative system, the entry of foreign capital is registered by commercial banks which, in turn, coordinate with the Central Bank of Chile. A minimum of $10,000 can be brought in through this mechanism in the form of currency or loans. This mechanism does not imply signing a contract of any type. Capital entering Chile under Chapter XIV is subject to the general regulation and cannot apply to be subject to the provisions of DL 600.

- Foreign Investment Statute or Decree Law 600 (D.L. 600): This voluntary mechanism is administered by the Executive Vice-Presidency of the Foreign Investment Committee. DL 600 is a mechanism for the entry of capital into Chile. Under this regime, whose use is optional, foreign investors bringing capital, physical goods or other forms of investment into Chile may ask to sign a foreign investment contract with the State of Chile. This contract establishes the rights and obligations of both parties and cannot be modified or rescinded unilaterally by either party. Foreign investors can choose to use one of the following invariability tax regimes:
  - Article 7: which states an effective overall income tax rate of 42%, in relation to those taxes established in the Income Tax Law in force at the time the contract
is executed. The Mining Tax is not considered for the determination of the effective tax rate

- Article 11 bis: For investment of no less than USD 50 million, grants invariability for the respective investor, as from the date of execution of such contract, of the legal provisions with respect to the assets depreciation rights, carrying forward losses and startup and organization expenses

- Article 11 ter: For investment of no less than USD 50 million to develop mining projects. It grants the following rights:
  - Invariability of the legal provisions applied at the date of signature of the respective contract regarding the specific tax on mining activities
  - Therefore, they will not be affected by an increase in the rate, the extension on the calculation base or any other amendment that may be introduced afterwards
  - They will not be affected by any new tribute, including royalties, canons or similar tax burden, specifically levied on mining activities, established after the signature of the respective foreign investment contract, that is based on or considers when calculating its base or amount, the incomes on mining activities or the investments, assets or rights used in mining activities
  - They will not be affected by modification introduced to the amount or form of calculation of the development and explorations license
  - The term of 15 years will be counted in calendar years, starting from the year in which the respective company commences operation.

In order to request those rights, the foreign investor must commit the respective companies (Chilean entity subject of the investment, that performs the mining activity) to submit their annual financial statements to external audit and to submit before the Securities and Insurance Supervisor their individual and consolidated financial statements in a quarterly and annual basis, as well as an annual report containing information on the property of the entity.

4.1.10 Policy Perception Index

AMEC has used the Policy Perception Index from the 2013 Fraser Institute Annual Survey of Mining Companies report (the Fraser Institute survey) as a credible source for the assessment of the overall political risk facing an exploration or mining project in Chile. Each year, the Fraser Institute sends a questionnaire to selected mining and exploration companies globally. The Fraser Institute survey is an attempt to assess how mineral endowments and public policy factors such as taxation and regulatory uncertainty affect exploration investment.
In 2013, 4,100 companies were approached, and 690 companies responded. The companies participating in the survey reported exploration spending of US$4.6 billion in 2012 and US$3.4 billion in 2013.

AMEC has relied on the Fraser Institute survey because it is globally regarded as an independent report-card style assessment to governments on how attractive their policies are from the point of view of an exploration manager or mining company and forms a proxy for the assessment by industry of political risk in Chile from the mining perspective.

In the Fraser Institute survey, the Policy Perception Index is quantified as providing:

“Is a composite index, measuring the overall policy attractiveness of the 112 jurisdictions in the survey. The index is composed of survey responses to policy factors that affect investment decisions. Policy factors examined include uncertainty concerning the administration of current regulations, environmental regulations, regulatory duplication, the legal system and taxation regime, uncertainty concerning protected areas and disputed land claims, infrastructure, socioeconomic and community development conditions, trade barriers, political stability, labour regulations, quality of the geological database, security, and labor and skills availability. The Policy Perception Index is normalized to a maximum score of 100”.

Chile has a Policy Perception Index number of 70.9 in the 2013 edition of the Fraser Institute survey, which, as shown in Table 4-1, is one of the highest rankings of the South American and Caribbean Basin countries. Chile is overall ranked 30 out of the 112 jurisdictions in the Fraser Institute survey.

4.1.11 Taxation

Income Taxes

The effective tax rate is 35% applicable on income distributed to shareholders. There are two types of tax on income. First category tax paid by the companies is 20% of accumulated annual taxable dividends. The second tax is an additional tax of 35% of the gross dividend remitted to shareholders who are not domiciled or resident in Chile, less a tax credit for any first category tax paid by the company. The additional tax payment may be postponed if the funds are reinvested in Chile within a period of 20 days in an individual company, a limited partnership or in shares in a corporation. The additional tax is paid when dividends are collected or are distributed to foreign shareholders.
Table 4-1: 2013 Fraser Institute Policy Perception Index Rankings, South America

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<td>61/96</td>
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<td>*</td>
<td>*</td>
<td>97/112</td>
<td>60/96</td>
<td>67/93</td>
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<td>82/96</td>
<td>86/93</td>
<td>65/79</td>
<td>71/72</td>
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<tr>
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<td>64.6</td>
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<td>36/112</td>
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<td>Uruguay</td>
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Note: Table sourced from the 2013 Fraser Institute Annual Survey of Mining Companies report. In the Fraser Institute report, the *** against French Guiana denotes that the country is a French overseas department.

As an alternative, an investment contract can be signed between a foreign investor and the government, allowing the investor to choose a tax rate that is effectively fixed at 42% for a period of 10 years. This period can be extended up to 20 years if the Foreign Investment Committee grants to the foreign investor the rights in Article 11 bis of D.L. 600. An investor can choose to return to the normal tax regime only once.

A First category tax rate of 20% was used in Section 22 of this Report and for the 2014 Feasibility Study.

Customs Duties

A flat rate of 6% is applicable on the cost, insurance and freight (C.I.F.) value for goods imported to Chile. Goods which are considered as an original product from countries with which Chile has special free trade agreements are subject to lower import taxes or may be tax exempt.

Value-Added Tax

D.L. 600 establishes that foreign investments in tangible assets shall be subject to the general tax regulations and valued-added tax (IVA). However, foreign investors may freeze the IVA (which is currently set at 19%) as well as the taxes for imported capital.
equipment at the prevailing rates as of the date of investment. This special regime applies throughout the period authorized for the investment. Some capital goods (such as machinery or equipment) are exempt from IVA when these are not produced in Chile and are considered to be in the interests of the country. These capital goods are as published in a list issued by the Ministry of Economy from time to time.

**Mining Tax**

On 1 January 2006 a specific tax was imposed for mining activities (mining tax). The 2006 legislation also amended D.L. 600 and added a new Article (11 ter) which establishes a regime of invariable mining tax (15 calendar years) for investors who sign a new contract for foreign investment for a minimum amount of $50 million.

From 1 January 2006 mine operators whose annual sales exceed the equivalent of 12,000 tonnes of fine copper a year must pay the mining tax. The tax rate varies depending on the annual sales as follows:

- A Mining Tax Rate of 5% will be applied to all mining operators whose annual sales are greater than the equivalent of 50,000 t of fine copper
- All mining operators with annual sales of equivalent fine copper between 12,000 t and 50,000 t will be subject to a Mining Tax between 0.5% and 4.5%.

The mining tax is a tax on operational mining income, and levied on a sliding-scale rate basis of between 5% and 14% depending on operating margins. Foreign investors who signed an investment contract with the government after 1 December 2004 may elect to use an invariable tax regime. This tax regime guarantees that the company will not be subject to any new tax, including royalties, duties or similar taxes that apply specifically to the mining sector. The company will not be affected by changes in the amount or manner in which taxes are calculated for the mining operation for a maximum period of 15 years.

The choice of the tax regime described above cannot be combined with the choice of the flat rate of income tax of 42%.

**Other Taxation Considerations**

Withholding taxes apply to payments made to persons or entities not domiciled or resident in Chile.

Municipal licenses are payable for any profession, business, industry, trade, art, or secondary or tertiary economic activity, regardless of the type. The licences also apply to certain primary or extractive activities. The license is valid for a period of 12 months...
from 1 July to 30 June. The cost of municipal licenses is 0.25% to 0.5% of the capital of the company. Companies can deduct that part of the capital that is invested in other businesses or companies that pay the municipal license.

4.2 Project Ownership

Information provided to AMEC supports that Minera Santo Domingo SCM (Minera Santo Domingo) is a mining company (sociedad contractual minera) legally organized under the laws of the Republic of Chile. Minera Santo Domingo has various mining concessions in the area of Diego de Almagro; collectively, these properties constitute the Santo Domingo Project.

The capital of Minera Santo Domingo is indirectly 70% owned by Capstone and 30% by Korean Resource Corporation (Kores) as follows:

- Capstone owns 70% of the issued and outstanding common shares of 0908113 BC Ltd
- A subsidiary of Kores, Korea Chile Mining Corporation, owns the remaining 30% of 0908113 BC Ltd
- 0908113 BC Ltd owns 100% of the issued and outstanding common shares of Far West Mining Ltd
- Far West Mining Ltd owns 100% of the issued and outstanding common shares of Minera Santo Domingo SCM.
- Minera Santo Domingo SCM owns 100% of the Santo Domingo Project.

Capstone has advised AMEC that under the terms of the shareholder agreement signed between Capstone and Kores on June 17, 2011, Capstone is Project operator.

4.3 Mineral Tenure

Minera Santo Domingo holds four groups of concessions, totalling 178 claims, which cover a total of 36,375 hectares and include the proposed mine site, plant area and auxiliary facilities including port facilities and the planned sea water and concentrate pipelines from the port to the mine. The tenure includes 82 exploitation concessions and 96 exploration concessions. Concessions are held in the name of Minera Santo Domingo.
The total concession area is divided as follows:

- 19,375 ha of exploitation concessions that encompass the area where the mine, plant, construction camp and ancillary facilities are planned
- 17,000 ha of exploration concessions that encompass the sea water and concentrate pipeline route from the port to the mine site, and the port.

As part of the grant process, the concessions have been surveyed by a government-licensed surveyor.

Concessions are protected under Chilean law by payment of the annual mining license fees. Capstone advised AMEC that all concession fees were current as of 22 May 2014, and will continue to be paid on a regular basis as due, using a formal status tracking system.

A simplified location plan for the infrastructure contemplated in the 2014 Feasibility Study is included as Figure 4-1. A summary of the mineral tenure is provided in Table 4-1 for the exploitation concessions. Figure 4-2 and Figure 4-3 show the layout of the concessions in Table 4-1. Table 4-2 summarizes the exploration concessions. Figure 4-4 shows the locations of the concessions along the proposed pipeline route in relation to the proposed port location and the exploitation concessions.

4.4 Surface Rights

Based on the current state of development of the Project, the existing legislation in Chile and the legal assurances necessary and required to safeguard the areas impacted by the project, a strategy has been developed for acquisition of surface lands sufficient to support the Project operation.

It is proposed to consolidate Minera Santa Domingo’s property holdings in the areas that cover the deposit and the proposed process facilities by purchasing these lands through the Ministerio de Bienes Nacionales (Ministry of National Property). It is also proposed to apply for one or more mining rights of way in the areas of interest of the project (this will safeguard these areas) such as the pipeline route, access roads and off-site ancillary facilities.

The surface land in the Community of Diego de Almagro where the Project is located is part of a larger lot that is owned by the State, and is managed and represented by the Ministerio de Bienes Nacionales. The planned acquisition of these areas will remove any legal impediments to the granting of surface land rights and other rights of way for access to the property and for the construction of the planned sea water and concentrate pipelines.
Figure 4-1: Proposed Project Facility Locations

Note: Figure courtesy Minera Santo Domingo, 2014. The grid on the figure illustrates the map scale.
Table 4-2: Exploitation Concessions

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Figure 4-2: Exploitation Concessions in Relation to Proposed Infrastructure Locations

Note: Figure courtesy Capstone, 2013. Area in white is the town of Diego de Almagro; dark blue area is the proposed tailings storage facility, open purple outlines are the proposed Santo Domingo Sur open pit and waste rock facilities; striped purple area is the planned Iris Norte open pit; red line is the proposed C-17 by-pass road. Map north is to top of plan. It is approximately 7 km from the planned concentrator site (purple rectangle located between the Iris Norte pit and the tailings storage facility) to the town of Diego de Almagro as an indicator of scale.
Figure 4-3: Location Plan, Exploitation Concessions

Note: Figure from Brimage et al., 2011. The status of the third-party claims that are not held by Capstone has not been verified. MLO refers to Minera Lejano Oeste S.A., a predecessor company to Minera Santo Domingo. Grid coordinates indicate map scale. Map north is to the top of the plan.
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Figure 4-4: Location Plan, Exploration Claims

Note: Figure courtesy, 2013. Backdrop is based on Google Earth image. Both the port location and the mine site location as indicated on this plan are proposed locations and are not operational from the Project standpoint at the effective date of this Report.
Capstone notes that it will be necessary to either acquire a total of 3,901.3 ha or complete the creation of mining easements for the installation and use of various facilities. This total area covers:

- 3,041.3 ha required for the plant and infrastructure (process plant, tailings disposal and waste disposal)
- 780 ha for the pipelines
- 10 ha for the temporary construction camp
- 70 ha for the planned port area.

The Project has received government guarantees for the rights of way required for the areas currently identified.

Capstone also proposes to apply for one or more mining rights of way in the areas of interest of the Project such as the pipeline route, access roads and off-site ancillary facilities to safeguard these areas.

4.5 Water Rights

The Project will not require an application for water rights. The water for operations will consist solely of sea water. A maritime concession has been requested which will allow the extraction of sea water.

Water for construction will be obtained from Aguas Chañar S.A., an authorized third party provider.

4.6 Royalties and Encumbrances

Government royalties are levied in the form of a mining tax.

A 2% net smelter return (NSR) royalty is payable to BHP on the original Candelaria Project land package concessions; this includes the concessions that host the proposed open pit.

4.7 Permits

The Project permitting status is discussed in Section 20.

4.8 Environment and Environmental Liabilities

The Project environmental and closure plan status is discussed in Section 20.
4.9 Social License

The Project social licence status is discussed in Section 20.

4.10 Comments on Section 4

The AMEC QPs were provided with legal opinion and information from Capstone staff and experts retained by Capstone that supports:

- Capstone holds 70% of the Santo Domingo Project; the remaining 30% is held by Kores
- Capstone is the Project operator
- The mining tenure held is valid and is sufficient to support declaration of Mineral Resources and Mineral Reserves
- No surface rights are currently held; however, the process that is required to obtain surface rights is well understood
- Royalties in the form of the Chilean mining tax will be payable
- No water rights are currently envisaged as the water will be sourced from the ocean and piped to the site for use
- Capstone advised AMEC that the company is not aware of any significant environmental, social or permitting issues that would prevent future exploitation of the Project deposits.
5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility

Access to the Project area is via the paved Pan-American Highway (Route 5 North) and a network of generally well maintained gravel roads. The Santo Domingo property is roughly five hours' travel time by road south of Antofagasta, and two hours by road north of Copiapó. Access is via Route 5 North, heading east from the town of Chañaral for 12 km to the El Salvador turn-off, and then 50 km east to the town of Diego de Almagro. A southbound, paved highway connects Diego de Almagro with Copiapó. At 3.3 km southeast from Diego de Almagro along this highway, a secondary gravel road (the Santo Domingo access road) leads south into the property.

The total distance by road from Chañaral to the Santo Domingo property is approximately 68 km, with a travel time of approximately 50 minutes.

5.2 Climate

The Project is located in an area that is as one of the driest places in the country and in the world with high solar radiation, evaporation rates and salt concentration in the soil. Rainfall is occasional and irregular, and in some years only received during the winter period.

Because of this there are only temporary surface run-offs, except for the El Salado River which is the only permanent water course in the Project area of influence. The El Salado River has a predominantly pluvial regime and is located about 7 km downstream of the planned mine and plant area. The river is highly altered and of low flow; it was used in the past to transport tailings from the El Salvador mine to the coast.

Meteorological data were collected at three different areas using monitoring stations, and define two climate zones:

- Normal desert: This extends from the south of the Copiapó Valley to the southern boundary of the Region and is characterized by low annual rainfall, increasing towards the south. The average annual temperature is 15°C. The main feature in the valleys of the Region is the frost-free condition for 11 months (from August to June). Minimum temperatures occur in July and reach 5°C; maximum temperatures occur in the summer and reach 28°C. There is strong seasonal precipitation in the area concentrated in the period from May to August, when more than 80% of the total annual precipitation falls.
Coastal desert: This is present in all the coastal sectors of the Region and to the north close to Chañaral. The relief does not present barriers to the marine influence; the amount of cloud depends on the presence of the Pacific Anticyclone, a high pressure that generates dry air masses. This type of climate is characterized by abundant and dense cloud cover that appears during the night and is dissipated during the morning; it is sometimes accompanied by heavy fog and drizzle. The ocean influence produces a moderate thermal regime with a small temperature range, both daily and annual. Precipitation is mostly associated with fronts and increases from north to south, occurring almost exclusively in winter. Chañaral receives an average of 12 mm per year.

Mining and port activities are expected to be able to be conducted on a year-round basis.

5.3 Local Resources and Infrastructure

There are several towns and villages near the Project. Diego de Almagro, located adjacent to the mine-plant area, has a population of several thousand people. Chañaral is a deep sea port less than one hour's drive to the west of the property. Chañaral has a population of about 10,000 people, hotel accommodation, food, fuel and minor services. The most important logistics centre in the Region is Copiapó, about two hours south of the Santo Domingo property. Copiapó has a population of approximately 150,000 people, an airport with daily scheduled flights to Santiago and Antofagasta and companies that offer abundant services for mining and exploration.

The Region has well established infrastructure (energy, water, transportation and labour) to serve the mining industry. However, there is currently no infrastructure on the Santo Domingo property except gravel roads for access to the concessions and drill sites. The nearby town of Diego de Almagro is connected to the regional power grid and the main railway line.

Proposed Project infrastructure is described in Section 18 of this Report.

5.4 Physiography

Elevations in the deposit area range from approximately 900 masl to 1,500 masl. Hills of gentle to moderate relief have been cut by deep gullies and are flanked with gravel-filled valleys and alluvial fans. The vegetation is very sparse. In the valleys, plant life consists of small widely-spaced bushes a few centimetres high. Hillsides and peaks are generally devoid of vegetation.
The coastline in the port area is aligned along a west–southwest–east–northeast direction. The soil type is a rocky soil, and a lens of sand and gravel on top of the bedrock is easily recognizable.

5.5 **Seismicity**

Seismic maps of South America indicate that the Project area is likely to have high seismicity and the site is considered to be within Zone 3 according to the Chilean standard NCh 2.369, with a peak ground acceleration value of 0.4 g. A seismic hazard assessment was performed by Rodolfo Saragoni, a recognized Chilean seismic reviewer, for the Santo Domingo Project site as part of the 2014 Feasibility Study, and his recommendations are included in the Project design.

5.6 **Comments on Section 5**

In the opinion of the AMEC QPs,

- There is sufficient suitable land available within the exploitation concessions for the planned tailings disposal, mine waste disposal, and mining-related infrastructure such as the open pit, process plant, workshops and offices.

- The planned pipeline route is held under exploration concessions.

- No surface rights are currently held; however the process for obtaining such rights is well understood. AMEC was provided with information that supports that the Project has received government guarantees for the rights-of-way required by the Project for the areas currently identified.

- Mining and port activities can be conducted year-round.

- The Project area is likely to have high seismicity. A seismic hazard assessment was performed by a third-party on behalf of Capstone, and recommendations arising from the study are included in the Project design.
6.0 HISTORY

Artisanal mining activities commenced in the general project area during the early 19th century. The major commodities targeted were gold and iron. As a result, there are a significant number of small workings and pits throughout the Project area. However, most of the surface workings are typical of artisanal activities, being less than a few tens of metres in length. There is limited information as to the extent of underground mining activities. No production records from this activity have been located.

Modern exploration commenced in 2002 when Far West and BHP Billiton formed a strategic alliance to explore the Chilean Iron Oxide Copper Gold Belt (IOCGB) in the Coastal Cordillera. In 2002 BHP Billiton flew a 10,700 line km Falcon™ airborne gravity gradiometer survey covering 5,145 km² in eight blocks along a 300 km strike length of the IOCGB between Taltal and south of Copiapó in northern Chile. The survey outlined more than 76 target areas containing one or more distinct gravity anomalies.

Far West commenced exploration activity in July 2003 and activity continued until 2011, when the company was acquired by Capstone. BHP’s interest in the project was terminated in May 2005. BHP transferred concession titles to Far West in exchange for a retained 2% NSR royalty.

Work completed comprised initial geological mapping, surface and drainage sampling, interpretation of existing airborne geophysical data, an induced polarization (IP) survey, and core and reverse circulation (RC) drilling (Table 6-1). This work resulted in the outlining of the Santo Domingo Sur, Estrellita and Iris deposits.

An initial copper–gold resource estimate was performed in 2006 for the Santo Domingo Sur deposit, and updated in 2007. The 2007 update also included first-time copper–gold resource estimates for Estrellita and Iris.

In 2008, a preliminary assessment (PA) was undertaken. This envisaged two open pit mining options, one being mining the Santo Domingo Sur deposit for the recovery of copper, gold and iron from magnetite; the second being mining the Santo Domingo Sur and Iris deposits for the recovery of copper, gold and iron from magnetite and hematite.

The resource estimate supporting the PA was updated to include iron as an element of interest. Results indicated that the options were revenue negative under the assumptions in the study; however, changes to the base case metal price assumptions did result in positive economics, and additional work was recommended.
## Table 6-1: Exploration Summary Table

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</tr>
<tr>
<td>July 2003 to November 2005</td>
<td>Far West/BHP Billiton*</td>
<td>Geological mapping</td>
<td>Approximately 50 km² of geological mapping at 1:25000</td>
<td>Höy and Allen, 2005</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Surface rock samples</td>
<td>50 samples submitted for analysis for Au and a 27-element ICP suite. Samples were generally taken where copper oxides were apparent, and hence most samples contained anomalous levels of copper.</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Drainage sediment samples</td>
<td>47 sieved (106 µm) samples, submitted for analysis for Au and a 27-element ICP suite.</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>IP survey</td>
<td>17.6 line km</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>RC drilling</td>
<td>67 holes (20,592 m) analysed for Au and a 27-element ICP suite</td>
<td></td>
</tr>
<tr>
<td>November 2005 to May 2006</td>
<td>Far West</td>
<td>Geophysical data interpretation</td>
<td>Falcon™ gravity and magnetic susceptibility plots were produced for data from Quantec Geofisica Limitada. The gravity anomalies define a north–south-oriented cluster of northwest-trending features up to 5 km long within the Project area.</td>
<td>Lacroix, 2006</td>
</tr>
<tr>
<td></td>
<td></td>
<td>RC drilling</td>
<td>15 holes (5,176 m) analysed for Au and a 27-element ICP suite</td>
<td></td>
</tr>
<tr>
<td></td>
<td></td>
<td>Core drilling</td>
<td>4,057 m in eight holes; analysed for Au and a 27-element ICP suite</td>
<td></td>
</tr>
<tr>
<td>May 2006 to September 2007</td>
<td>Far West</td>
<td>RC drilling</td>
<td>215 holes (51,909.5 m); analysed for Au and a 27-element ICP suite</td>
<td>Lacroix and Rennie, 2007</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Core drilling</td>
<td>15 holes (2,649.75 m); analysed for Au and a 27-element ICP suite</td>
<td></td>
</tr>
<tr>
<td>September 2007 to December 2008</td>
<td>Far West</td>
<td>RC drilling</td>
<td>37 holes (10376.5 m); analysed for Au and a 27-element ICP suite</td>
<td>Lacroix, 2009</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Core drilling</td>
<td>One hole (495.25 m); analysed for Au and a 27-element ICP suite</td>
<td></td>
</tr>
<tr>
<td>December 2008 to May 2010</td>
<td>Far West</td>
<td>RC drilling</td>
<td>Nine holes (2,557 m); analysed for Au and a 27-element ICP suite</td>
<td>Rennie, 2010</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Core drilling</td>
<td>26 holes (9,073 m); analysed for Au and a 27-element ICP suite</td>
<td></td>
</tr>
<tr>
<td>2011–2012</td>
<td>Capstone</td>
<td></td>
<td>66 holes (some were abandoned due to operational difficulties) totalling 13,282 m</td>
<td></td>
</tr>
</tbody>
</table>

Note: The BHP Billiton interest was terminated on 4 May 2005.
During 2009, the copper–gold resource estimates for Santo Domingo Sur and Iris were updated, and the Iris Norte deposit added to the estimate. Iron was included in the updated resource estimates. A further copper–gold–iron resource update was performed in 2010, covering Santo Domingo Sur, Iris, and Iris Norte.

Following acquisition of Far West by Capstone on 17 June, 2011, Capstone completed a pre-feasibility study (PFS). The study envisaged conventional open pit mining of the Santo Domingo Sur, Iris, and Iris Norte deposits, a semi-autogenous grind, ball mill and pebble crushing comminution circuit (SABC), conventional copper flotation, magnetic separation, tailings disposal and storage, water and concentrates pipelines and port facilities, and associated site infrastructure requirements. Using the assumptions in the study, the project showed positive project economics.

In 2012, a feasibility study was commissioned. The remainder of this Report discusses the results of that study.
7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Chilean Iron Belt (CIB) is a narrow, north–south trending belt stretching for over 2,000 km parallel to the Chilean coast, from approximately 25°S to 31°S. The dominant feature of the belt is a complex sinistral strike-slip and dip-slip fault system known as the Atacama fault zone. Faulting is interpreted to be related to an oblique subduction of a Jurassic to early Cretaceous magmatic arc. Initial faulting took the form of strike-slip, causing mylonite development and ductile deformation. This gave way to dip-slip fault movement and brittle deformation during later extensional tectonism.

Between approximately 132–106 Ma, a number of tabular-shaped mafic to felsic plutonic complexes were emplaced along the Atacama fault zone. Emplacement occurred during both strike-slip (ductile) and dip-slip (brittle) deformation regimes.

A number of volcanic- or intrusive-hosted breccia zones were developed in association with the strike-slip and dip-slip faulting, which became sites for the formation of a number of metasomatic iron oxide and iron-oxide–copper–gold (IOCG) deposits.

IOCG deposits in the CIB are divided into more iron-rich and more copper-rich end members:

- The iron-rich end members are classified as Kiruna-type magnetite–apatite deposits with associated actinolite–albite–quartz–tourmaline alteration. Host rocks are typically brecciated volcanic materials, or brecciated intrusions thought to be genetically related to the formation of the deposits. The majority of these iron deposits are spatially related to pyroxene diorites (Ménard, 1995). Some examples of the larger Kiruna-type deposits in the CIB include Romeral, Los Colorados, Boquerón Chañar, Algarrobo, Cerro Iman, and Rodados Negros
- Copper-bearing end members include La Candelaria and Manto Verde.

7.2 Project Geology

7.2.1 Lithologies

The Santo Domingo Project lies on the east side of the Atacama fault complex, which, in this area, consists of numerous clusters of generally north–south structural breaks in a belt approximately 30 km wide (Figure 7-1).
Figure 7-1: Regional Geology and Fault Structures in the Greater Project Area

Note: Figure courtesy Capstone, 2013. Deposits shown as red stars outside the project area are held by third parties. Grid co-ordinates indicate map north and figure scale.
The base of the stratigraphic sequence in the Project area is interpreted to be sedimentary rocks of the Punta del Cobre Formation. The only known surface expression of the Punta del Cobre Formation is a poorly-exposed sequence of sedimentary and volcanic rocks outcropping in the extreme southeast part of the Project area. Geology in this area consists of intercalated calcareous sedimentary rocks, crystal tuff, lapilli tuff, hornfels and andesite porphyry. One exposure of thinly laminated, moderately west-dipping, red hematitic siltstone may be correlative with the hematitic terrigenous basal conglomerate of the Algarrobo Member of the Punta del Cobre Formation in the Copiapó area (Marschik and Fontboté, 2001). Capstone geologists note that if this is in fact the lower part of the Algarrobo Member, the lithology in this area is in the same stratigraphic position as the host rocks of the Candelaria deposit that is approximately 120 km to the south of the Project.

The Punta del Cobre Formation units grade upwards into a contemporaneous, interdigitated sequence of limestone and marine sediments of the Chañarcillo Group and andesitic flows and volcaniclastic rocks of the Bandurrias Group. The upper Punta del Cobre Formation near its contact with the overlying Bandurrias–Chañarcillo Group sequences is the stratigraphic host location of the Candelaria deposit.

Limestone units vary in thickness from a few metres to over 100 m, but can be the predominant rock type across several hundred metres of stratigraphy. They are generally massive to thickly bedded, fine grained, and dark to light grey, predominantly forming the top parts of many prominent hills in the area.

True sediments are not abundant, with most clastic rocks classified as tuffaceous sediments or crystal tuffs. These rocks are generally massive to poorly bedded, fine to medium grained, and commonly difficult to differentiate from fine-grained, massive flows. Individual units reach thicknesses of up to 50 m but can comprise the bulk of the stratigraphy, reaching over 300 m in thickness, with minor intervals of limestone and andesite lavas.

In some places the andesitic volcaniclastic rocks are interlayered with significant volumes of light grey to cream coloured aphanitic and, rarely, thinly-laminated material. In drill holes this material was logged as possible felsic tuff horizons, but subsequent petrographic work suggests that they are carbonate–potassic feldspar-altered andesitic tuffaceous sediments (Ross, 2005).

Several relatively narrow hematite and magnetite (±copper oxide or sulphide) mantos up to 12 m thick occur sporadically within the tuffaceous sequence across a 200 m stratigraphic interval, with associated weak to strong actinolite-potassic feldspar alteration. This stratigraphy and related iron oxide-copper mantos have been
tentatively identified throughout the Project, and probably underlie most or all of the area.

Andesite flows range from near aphanitic to coarse-grained feldspar-phyric, but are generally medium grained, with 20% to 30% euhedral, white, prismatic plagioclase (±minor hornblende) phenocrysts in a grey to brownish aphanitic groundmass. Some flows are massive, whereas others contain abundant amygdules up to 1 cm in diameter (average 1 mm to 2 mm) filled with varying proportions of quartz, calcite, epidote, chlorite, potassic feldspar, limonite (pyrite) and “almagre” (minute grains of distinct copper minerals admixed with the red hematite) or other copper oxides).

The Bandurrias Group is defined as a predominantly volcanic sequence of andesite flows and volcaniclastic rocks. Chañarcillo Group rocks consist primarily of limestone and calcareous marine sediments. These two groups are thought to be contemporaneous, deposited at the same time in different parts of the same back-arc basin. Capstone geologists observe that the overall geological descriptions of the two groups match observed Project geology. However, the andesite-tuff succession that hosts the mantos may be part of the Punta del Cobre sequence. This would suggest the presence of faulted contacts between this sequence and the structurally-adjacent limestone that is more clearly correlated with the Chañarcillo Group.

Based on the Sernageomin regional geology map, Capstone geologists have interpreted that at least nine intrusive events have affected the Project area. Intrusions are generally younger eastward and range in age from 145 Ma to 90 Ma (Figure 7-2).

7.2.2 Structure

Faults

The Project area is divided into a number of structural blocks with different lithological characteristics suggesting that the blocks are part of different stratigraphic levels.

Faults trend variably north, northwest, northeast and east–west. These faults are complex and seem to have been active repeatedly through time. Many mark the boundary of pronounced lithological changes.
Figure 7-2: Local Geology Plan Showing Major Intrusive Events

Note: Figure courtesy Capstone, 2013.
Most faults recognized in the area, either through mapping, drill intersections, or magnetic lows, appear to be high angle faults with both dip-slip and strike-slip movement. As well, some low angle faults with probable reverse displacement have been noted in several outcrops, suggesting the presence of thrust faulting, most notably the fault that bounds Santo Domingo Sur in the south (where mineralization has been intersected below the limestone unit in the south). The extent of these interpreted thrust faults is not known.

The most obvious structure, referred to as the Santo Domingo fault, crosses the Estrellita and Estefánia areas. It is a large east-west trending, steeply north dipping, north-side-down block fault, with a probable right lateral strike-slip component. Most of the historic copper production in the area comes from or near this structure.

The most prominent fault set, as interpreted from magnetic lows, trends northwest and has fault separations of approximately 1 km. Several northwest-trending faults are also recognized in the Santo Domingo Sur area.

High-angle block faulting played an important role in localizing manto- and fault-related iron oxide–copper mineralization in the Santo Domingo area. These faults have uplifted the central part of the Santo Domingo Sur area, bringing the manto succession close to surface. To the east and south there is potential that this prospective horizon is present at depth, beneath limestone cover rocks.

**Fault Blocks**

In the Santo Domingo Sur block, a thick package of andesitic flows is underlain by a sequence of tuffaceous rocks of similar composition. The tuffs have been intruded by fine-grained diorite sills. The entire package consisting of andesitic flows, andesitic tuffs and diorite intrusions have been cut by later feldspar–hornblende porphyry dykes that cut all other rock types and do not host any mineralization.

The structural block to the west of the Santo Domingo Sur deposit consists of a gently to moderately north–northwest dipping, bedded sequence of limestone and intercalated tuffaceous andesitic rocks grading into less calcareous tuffs and volcanic sediments towards the south.

The geology to the south of the Santo Domingo Sur deposit is somewhat distinct from the rest of the Project area, as the Bandurrias–Chañarcillo Group rocks have been intruded by a series of small diorite plugs and sills assigned to the Sierra Santo Domingo plutonic suite.
The area to the northeast of the Santo Domingo Sur deposit that hosts the Iris deposit is structurally complex and is not well understood at this point as the drill spacing of 100 m does not in many cases allow correlations from one drill hole to the next. Some smaller structural blocks may only be represented by a single drill hole.

Between the two structural blocks that host the Santo Domingo Sur and Iris deposits respectively, there is another fault block that consists of andesitic flows hosting massive magnetite mantos that are barren of copper mineralization.

The structural block to the east of the Iris deposit is characterized by thick sequences of limestone that can be observed at surface. This structural block has not been tested by drill holes and it is unknown what lithological units are positioned below the limestone sequence.

The northern part of the Santo Domingo area where the Iris Norte deposit is situated, is characterized by andesitic flows and andesite porphyries at surface. The highest ridges in the area are typically made up of a thick sequence of limestone that overlies the volcanic sequence. A large part of the northernmost structural block is covered by younger gravel that displays a thickness of up to 150 m that appears to increase towards the north.

Drilling at Estrellita has shown that the package of andesitic porphyries and flows has a thickness of up to 200 m. In the Estrellita area this package is underlain by a sequence of volcanioclastics with minor intercalations and interbeds of andesite porphyry, limestone and altered tuff.

Folds

Limited mapping and recognition of outcrop-scale, open folds indicate that the rocks have been gently folded along north–northeast-trending axes.

7.2.3 Alteration

Hydrothermal alteration and mineralization in the Santo Domingo area affects all rocks and exhibits numerous styles and events with multiple overprinting components. At the deposit and district scales four styles of alteration are recognized: sodic (-calcic), potassic, carbonate and calc-silicate skarn. A clear hydrothermal zoning occurs from proximal to distal assemblages at deposit scale (Santo Domingo Sur) and apparently at district scale at depth and towards the diorite intrusive complex.

Main sodic (-calcic) alteration minerals are albite, actinolite, chlorite, epidote and titanite that replace mainly volcanic and intrusive rocks. Scapolite–actinolite–pyroxene
veins can be found at the southern portions of the area close by and within the diorite stocks and dikes. At surface, actinolite, chlorite and carbonate typically occur as infilling amygdules and open spaces. Pink albite replaces plagioclase in the more porphyritic rocks.

Potassium silicate alteration is less common but is found as K-feldspar–chlorite–carbonate–quartz mineral assemblages. Patchy K-feldspar mainly replaces plagioclase (albite) and is also found in veins with carbonate and quartz. This alteration is mainly located within the copper-iron mantos.

Carbonate rich assemblages are widespread and overprint the previous mineral associations. In addition, carbonate (calcite, ankerite, siderite)–chlorite–quartz veins and stockwork are commonly found cutting all rock types of the area.

Calc-silicate skarn minerals are found south of Santo Domingo where carbonate rich rocks and lesser volcanic rocks are in contact with diorite intrusives. Main alteration minerals are garnet (andradite), epidote, pyroxene, actinolite and carbonate.

7.2.4 Weathering and Supergene Development

Supergene processes are weakly developed in the Santo Domingo area. Oxidation is shallow (70 m to 90 m below surface) and enrichment is minimal, consistent with the low total sulphide contents and the calcareous and feldspathic nature of the host rock.

The Fe–Cu–Au mineralization in Santo Domingo is almost entirely hypogene; the proportions of sulphides to oxides are approximately 13:1 (Rennie, 2010).

At shallow levels, typical Cu–Fe mineralization comprises small veins, hydrothermal breccias and mantos. Specular hematite ± magnetite is commonly altered to earthy hematite and goethite, and usually is found with mixtures of copper oxides (chrysocolla, brochantite, malachite and copper wad). In addition, almagre has been previously described filling amygdules within the andesitic volcanic flows and in the Estrellita area (Rennie, 2010).

At Santo Domingo Sur, gypsum is locally found filling fractures and open spaces, presumably as a hydration product of anhydrite. Relatively more scarce digenite ± chalcocite ± covellite are locally present, partially replacing fractures and rims of bornite crystals. Various amounts of native copper are found interstitial to the matrix of hydrothermal breccias and veins, especially the shallower portions of the central part of the Santo Domingo Sur deposit.
7.3 **Deposits**

To date, four deposits, three of which support the 2014 Feasibility Study, and a number of prospects have been identified in the Project area (Figure 7-3).

7.3.1 **Santo Domingo Sur**

The andesitic flows and tuffs are the primary host to mineralization at Santo Domingo Sur. Mineralization consists of thick semi-massive to massive iron oxide mantos that have replaced the tuffaceous rocks. The stratigraphic sequence of andesitic flows and tuffs dips gently (at an angle of approximately 15°) towards the north–northwest under gravel cover.

The tuff sequence has been intruded by fine-grained diorite sills that are present in almost all drill holes at Santo Domingo Sur, varying in drilled thickness from a few metres to more than 60 m. Similar diorites have been intersected in the Iris deposit and have been observed in outcrop to the south of Santo Domingo Sur. The diorites are typically altered and in rare cases contain copper mineralization. These observations suggest that the diorite intrusion is more or less contemporaneous with the mineralizing event and may in fact have been the heat engine for the formation of the deposit. The last intrusive events in the area are feldspar–hornblende porphyry dykes.

Mineralization occurs in the form of copper-bearing semi-massive to massive iron oxide mantos with minor veins and breccias. The mantos are zoned from an outer rim of specular hematite toward a magnetite-rich core.

Drilling has identified a 150 m to 500 m thick, copper-bearing, specularite–magnetite sequence covering an area of approximately 1,300 m by 800 m, and traced to a depth of approximately 525 m below surface. Mineralization consists of stacked chalcopyrite bearing, specularite-magnetite mantos within tuff and tuffaceous sediments overlain by andesitic flows.

The mantos consist of semi-massive to massive specularite and magnetite layers with clots and stringers of chalcopyrite, that range in thickness from approximately 4 m to 20 m. The upper parts of the manto sequence directly below the overlying andesite flows are frequently oxidized and contain various amounts of copper oxides and chalcocite.
Figure 7-3: Deposit and Prospect Layout Plan

Figure courtesy Capstone, 2013. SDS = Santo Domingo Sur deposit.
Mineralization in the deposit is strongest in the southern part and in the upper levels. Copper grade and intensity of the mineralization weaken towards the northern part of the deposit as well as with depth. The high-grade core of the deposit is located along the southern margin and close to surface. It appears likely that the bounding fault in the south of the deposit has been the main conduit for mineralizing fluids as mineralization and alteration is strongest along that fault.

Recent drilling has outlined a zone of hydrothermal brecciation in the centre of the deposit. The breccia consists of andesite and andesitic tuff fragments in a fine-grained matrix of iron oxides. The upper part of the breccia is oxidized with both limonite, which is the dominant iron oxide, and copper oxides. Native copper has also been observed. The lower part of the breccia contains regular sulphide mineralization and differs from the surrounding rock only in texture. The breccia has been intercepted by multiple drill holes, establishing a complex geometry that forms a narrow body at depth, but which widens toward the surface.

Figure 7-4 presents a geology and structural plan of the Santo Domingo Sur deposit. Figure 7-5 is an example cross-section through the deposit showing the location and orientation of the mineralization.

7.3.2 Iris

The Iris deposit is essentially blind, covered by a sequence of Quaternary gravel. The elongated shape of the deposit and textures observed in diamond drill holes indicate that the Iris deposit has formed in a north-northwest striking fault zone that is bounded by a west-dipping fault that can be traced along most of the deposit’s western side. The eastern side of the deposit is bordered by a steeply dipping fault that divides andesitic tuffs on the western side from calcareous sedimentary rocks and limestone to the east.

The Iris deposit footprint, when projected to surface, is approximately 500 m wide, has a strike length of 1,800 m, and has been traced from surface to a depth of approximately 670 m below surface. When the dip and plunge of the zones is considered, the real width of the deposit is of the order of 200 m.

The deposit consists of iron oxide mantos and breccias developed along a north–northwest-striking fault zone. The dominant iron oxide at Iris is hematite and the main copper mineral is chalcopyrite.
Figure 7-4: Geology and Structure Plan, Santo Domingo Sur

Note: Figure courtesy Capstone, 2013.
Figure 7-5: Santo Domingo Sur Simplified Geology (Cross Section 99600)

Note: Figure courtesy Capstone, 2013. Blue lines indicate faults and interpreted structures
Mineralization occurs close to surface at the southern end and plunges gently towards the north. The distribution of copper mineralization in the Iris deposit is more erratic and irregular when compared to the Santo Domingo Sur deposit. This is attributed to structural controls playing a greater role in the formation of the Iris deposit as contrasted with the more continuous stratiform replacement style mineralization at Santo Domingo Sur.

There are some old mine workings at the southern end of the deposit where copper oxides such as brochantite and chrysocolla were mined at surface. The oxide mineralization is hosted by a specularite manto that is cut by steeply-dipping structures. The extent of oxide mineralization at surface is approximately 100 m by 60 m.

The Iris Mag zone is located between the Iris deposit and the SDS deposit in a separate structural block. Mineralization in the zone consists of magnetite and chalcopyrite with a very high magnetite content (40% and more) and typically low copper content (approximately 0.1% Cu on average). The host rocks are andesitic flows and andesite breccias with a much smaller tuff component than the other zones. It appears that this part of the deposit has been subject to the initial high temperature magnetite event, but shows little evidence of a later oxidizing overprint that has introduced high copper and gold elsewhere.

Figure 7-6 is a geology and structure plan of the Iris deposit. Figure 7-7 is a cross-section through the deposit.

### 7.3.3 Iris Norte

The Iris Norte deposit is located about 600 m to the north of the Iris deposit and is also blind, being entirely covered by a gravel sequence. The deposit is very similar in character to Iris, and occurs on the eastern edge of a pronounced gravity anomaly. The deposit is approximately 500 m wide and has been tested over a strike length of 1,600 m and to a depth of 320 m below surface.

Mineralization is primarily hosted in andesitic flows, which differs to the tuff host at Santo Domingo Sur and Iris. The Iris Norte deposit is also elongated in shape and seems to have formed in a structural zone. The deposit displays a northeasterly strike which is a rotation of approximately 55° clockwise versus the strike of the Iris deposit.

The Iris Norte deposit has been intruded by significant numbers of diorite dykes and sills, which have separated the deposit into two lenses.
Figure 7-6: Geology and Structure Plan, Iris

Note: Figure courtesy Capstone, 2013.
Figure 7-7: Iris Simplified Geology (Cross Section 100500)

Note: Figure courtesy Capstone, 2013. Blue lines indicate faults and interpreted structures.
Mineralization consists of mixed magnetite and hematite mantos. The main sulphides in Iris Norte are pyrite and chalcopyrite, with the latter providing the copper content of the deposit. Iris Norte contains a higher proportion of magnetite than the Iris deposit and there are a higher proportion of intrusive rocks.

Figure 7-8 is a geology plan of the Iris Norte deposit. Figure 7-9 is a cross-section through the deposit.

7.3.4 Estrellita

Estrellita comprises an east-west striking, flat lying to shallow north-dipping tabular body lying approximately 3.5 km northwest of Santo Domingo Sur. Mineralization is interpreted by Capstone geologists to occur at a higher stratigraphic level than Santo Domingo Sur, Iris and Iris Norte, which are hosted in tuff sequences below the level of mineralization at Estrellita. Drilling at Estrellita has shown that the host package of andesitic porphyries and flows has a thickness of up to 200 m. In the Estrellita area this package is underlain by a sequence of volcanioclastics with minor intercalations and interbeds of andesite porphyry, limestone and altered tuff.

Estrellita has been faulted into a series of four blocks which step downwards to the north, with displacement across the faults ranging up to approximately 75 m. The overall footprint of the zone measures 900 m long by 450 m wide and is up to 100 m thick. The deepest drill intersections are in the order of 250 m below surface. The zone is thickest in the middle and narrows somewhat towards the periphery. There are narrower zones of limited lateral extent in the footwall of the main zone but it is open ended to the east and west.

The character of mineralization at the Estrellita deposit is a mixture of manto style, iron oxide and structurally-controlled, vein-style mineralization. The central part of the Estrellita deposit consists of a more or less horizontal tabular body of iron oxide manto that appears to have formed at the intersection of a flat-lying and a steeply-dipping set of specularite structures.

Copper mineralization typically consists of copper oxides such as brochantite, chrysocolla, almagre, cuprite, and chalcocite. The oxidized mineralization at surface becomes gradually less oxidized with depth and transitions through a mixed zone of oxides and sulphides into a sulphide zone where the main copper mineral is chalcopyrite.

Figure 7-10 is a cross-section through the Estrellita deposit. Figure 7-11 is a cross-section showing a simplified geological interpretation.
Figure 7-8: Geology Plan, Iris Norte

Note: Figure courtesy Capstone, 2013.
Figure 7-9: Iris Norte Simplified Geology

Note: Figure courtesy Capstone, 2013. Drill traces are colour-coded to lithologies. Green: andesite; Red: Fe manto; Brown: andesitic tuff; Pink: diorite; Turquoise: limestone; Yellow: overburden.
Figure 7-10: Geology and Structure Plan, Estrellita

Note: Figure from Lacroix and Rennie, 2007.
Figure 7-11: Estrellita Geological Cross-Section (Section 369300E)

Note: Figure prepared by RPA, 2013. Drill traces show individual assay results, larger bars on left-hand-side of drill trace show composite grades. Composite grades are based on a lower cut-off grade of 0.1% Cu.
7.4 Prospects/Exploration Targets

7.4.1 Estrellita and Estefánia Areas

In the Estrellita and Estefánia areas several gently north dipping, stratabound, iron oxide (specular hematite near surface grading to magnetite at depth) ± copper horizons up to 12 m thick occur in roughly the same 200 m stratigraphic interval, and have been tentatively traced with drilling or extrapolated across 3 km of strike length. Mineralization typically occurs within a simple, single phase breccia of fine grained, calcareous tuffaceous sediment. The breccia matrix typically consists of fine grained specular hematite with disseminated, stringer and fracture coating copper oxides and rare clots of chalcopyrite. Breccia horizons appear to be largely stratabound, but to the south are discordant, following the steeply-dipping Santo Domingo fault, and suggesting that this fault may have been a fluid conduit.

7.4.2 Santo Domingo Fault

Andesite units both north and south of the Santo Domingo fault have been cut by a closely spaced (100 m to 200 m) set of northwest striking and steeply northeast dipping carbonate veins mineralized with specular hematite and copper oxides. Although these veins historically have supported very small-scale mining operations for some time, even collectively they do not appear to have had significant production.

Copper mineralization also occurs disseminated in the andesite and limestone peripheral to the Santo Domingo fault. Andesite flows north of the fault host copper minerals including chrysocolla, malachite, almagre (a cupriferous limonite) and chalcopyrite, sporadically in amygdules with quartz, calcite, epidote and chlorite.

7.4.3 Limestones

In the limestone sequence copper occurs rarely as small chalcocite nodules with associated malachite. It is currently uncertain how these disseminated copper occurrences are genetically related to the vein and manto mineralization.
8.0 DEPOSIT TYPES

Information discussed in this section is from public domain sources as noted in the text. RPA has not verified this information, and cautions that the discussions of lengths, widths, grades and other indications of mineralization and mining activity on these properties may not necessarily be indicative of the mineralization in the Project area, or of any future mining activity that may be able to be conducted in the Project area.

Mineralization at Santo Domingo occurs primarily as iron oxide–copper–gold (IOCG) deposits with related vein and skarn bodies.

The largest and most extensively mined IOCG type deposits in Chile occur within a structurally complex zone extending between La Serena and Taltal over an area of 500 km by 50 km. Deposits within the CIB have two general end members; a magnetite–apatite–actinolite mineral assemblage similar to the Kiruna deposit in Sweden, and a copper-rich type similar to the Olympic Dam deposit in Australia.

The magnetite-rich deposits in Chile have been mined for iron since the early 1800s and Los Colorados mine south of Copiapó is still in production. Examples of copper-rich IOCG deposits in the belt include Candelaria and Manto Verde.

The descriptions of the Candelaria and Manto Verde deposits that follow provide more detail on the IOCG-type deposits in the CIB.

8.1.1 Candelaria Deposit

The Candelaria deposit is hosted in altered volcanic and volcaniclastic rocks of the Punta del Cobre Formation which were deposited in an Early Cretaceous continental volcanic arc and marine back-arc basin terrane. Punta del Cobre Formation rocks have been divided into the lower Geraldo Negro Member and the upper Algarrobo Member. The Geraldo Negro Member consists of massive andesite and minor dacite. The overlying Algarrobo Member is a coarsely bedded sequence of andesitic volcaniclastic and flow rocks with an upper tuffaceous sediment horizon. Rocks of the Algarrobo Member are overlain by calcareous sediments and limestone of the Chañarcillo Group. These marine environment sediments grade laterally into coeval terrestrial volcanic and volcaniclastic rocks of the Bandurrias Group.

The shallow east-dipping stratigraphic sequence above has been gently folded into an open anticline in the deposit area. It has also been cut by closely spaced sets of faults with three dominant orientations: north-northwest to northwest-trending steeply-dipping sinistral strike-slip faults; northeast-trending steeply to moderately northwest-dipping faults; and east–northeast striking high-angle left-lateral offset strike-slip faults. These
faults are probably responsible for the channelling of metal bearing fluids and appear to be important controls for metal deposition. An early Cretaceous granitoid pluton in the Chilean Coastal Batholith, which intrudes into the volcano-sedimentary sequence approximately 5 km to the west, is generally believed to be the heat engine responsible for fluid movement and subsequent metal deposition.

Mineralization at the Candelaria deposit is typically an assemblage of magnetite-chalcopyrite-pyrite with lesser amounts of specular hematite and/or pyrrhotite. Mineralization is predominantly restricted to the upper part of the Geraldo Negro andesite and the overlying volcano-sedimentary rocks of the Algarrobo Member. Mineralization appears to be roughly stratabound with upward fluid movement restricted by an impermeable scapolite-rich skarn located at the base of the Chañarcillo Group.

Host rocks are strongly altered and zoned into distinct mineral assemblages. In the deeper parts of the deposit area and close to the batholith, rocks are intensely altered to a biotite-quartz-magnetite assemblage. Fracture related calcic amphibole (actinolite) cuts this hydrothermal mineral assemblage. Higher up in the system alteration mineralogy consists of an assemblage of potassium feldspar with chlorite and/or biotite, plus quartz and magnetite and/or hematite. The upper part of the system is typified by a broad zone of sodic alteration with an albite-chlorite-calcite-hematite assemblage. Sulphide stringers (predominantly chalcopyrite and pyrite) post-date all alteration events.

Iron oxide mineralization at Candelaria has been dated at 116 Ma to 114 Ma and subsequent copper mineralization at 112 Ma to 110 Ma (Marschik et al., 2000). Ca-amphibole has been dated at 111.7 ±0.8 Ma (Ullrich and Clark, 1998) and hence is closely associated with the copper-mineralizing event. These ages are broadly coincident with the age of the adjacent granitoid pluton which is therefore thought to be genetically related to mineralization.

### 8.1.2 Manto Verde Deposit

Anglo American’s (Mantos Blancos) Manto Verde mine is located approximately 104 km north of the Candelaria deposit and 25 km southwest of the Santo Domingo area.

The oldest lithologies in the Manto Verde area are variably altered (hornfelsed and mylonitized) andesitic volcanic rocks. According to Vila et al. (1996), these are part of a 2,000(+) m thick, east-dipping sequence of predominantly sub-aerial andesite flows and volcanic breccias with minor intercalated sandstone and limestone. Segerstrom (1960) and Brown et al. (1993) have placed the volcanic rocks around Manto Verde into the Early Cretaceous Bandurrias Formation. According to Zamora and Castillo
(2000?) and the Quebrada Salitrosa geological map by Godoy and Lara (1998), these volcanic rocks have at least in part been assigned to the Mid to Upper Jurassic La Negra Formation.

The main part of the Atacama fault zone passes through the Manto Verde mine area. In this region it is interpreted as a 10 km wide zone of structural deformation with three main branches: the eastern, central and western faults. There are many prominent north-south structures apparent on both sides of this complex Atacama fault zone; however, it is clear that the actual zone of deformation is much wider. Volcanic rocks have been cut by numerous phases of north-south elongated granitic to dioritic intrusions. These are interpreted to be syntectonic emplacements along the Atacama fault complex.

Geology in the area, therefore, is typified by generally north-south elongated, fault and intrusion bounded blocks of volcanic rocks within a multiphase intrusive complex. Plutonic rocks occur as dykes, plugs, stocks and batholiths, ranging in size from a few metres to a few tens of kilometres.

The Manto Verde deposit is located along the Manto Verde fault, a north-northwest trending, 40° to 50° east dipping, riedel shear connecting the east and central branches of this western part of the Atacama fault zone. Host andesitic volcanic rocks, and possibly coeval dioritic intrusions (sills?) of the Mid to Upper Jurassic La Negra Formation as well as the Lower Cretaceous Bandurrias Formation, have undergone brittle deformation along the Manto Verde fault during a regime of extensional tectonism.

Tabular breccia bodies up to 100 m wide developed along the Manto Verde fault contain fragments of altered host rock within a matrix composed largely of iron oxide and a variety of copper oxide minerals. In the main pit the iron oxide is predominantly specularite, whereas in the south pit magnetite is more abundant. Copper minerals appear to both pre-date and post-date iron oxide mineralization. In some cases copper oxides occur as angular breccia fragments in a specularite matrix. In other cases copper minerals are clearly late, occurring as disseminations, open space fillings or stringers, cutting massive hematite or magnetite as well as the host rock.

Oxidation occurs to depths of over 200 m within the Manto Verde fault. Copper minerals in the oxide zone consist of:

- Copper sulphates; brochantite, antlerite
- Copper carbonate; malachite
- Copper silicate; chrysocolla
- Copper chloride; atacamite
- Pitchy copper ore; cupriferous limonite (almagre).

A narrow (generally less than 5 m), discontinuous zone of supergene enrichment is developed at the oxide-sulphide transition. Copper mineralogy in this zone consists of chalcocite and cuprite. Sulphides below the oxide zone consist of disseminated and stringer related pyrite and chalcopyrite within an iron oxide breccia matrix. Magnetite appears to become the more dominant iron oxide at depth.

The host andesite-diorite sequence has undergone widespread chloritization and potassic metasomatism (microcline), probably as a result of intrusion by adjacent granitic to dioritic plutons. Intense hydrothermal alteration peripheral to the mineralized structures masks the ubiquitous contact metamorphism. This hydrothermal alteration consists of a sequence of overprinting mineral assemblages. From earliest to latest they are (Zamora and Castillo, 2000):

- Chlorite–quartz
- Calcite–sericite–hematite–magnetite
- K-feldspar–quartz–specularite.

Earlier formed microcline is altered to sericite, and plagioclase breaks down to sericite and carbonate. Silica and possibly potassium may be the only significant non-metallic additions during the hydrothermal alteration associated with iron and copper mineralization. Hydrothermal sericite associated with the copper mineralization has been dated at 121 ±3 Ma and 117 ±3 Ma (Vila et al., 1996). The nearby La Tazas pluton has been dated at 130 Ma to 126 Ma, and the Sierra Dieciocho pluton at 126 Ma to 115 Ma (Godoy and Lara, 1998). The age of mineralization at Manto Verde is coincident with the age of the Sierra Dieciocho pluton which outcrops some 4 km to the east of the pit. Late north-trending mafic dykes cut all rock types, alteration assemblages and mineralization.
9.0 EXPLORATION

9.1 Grids and Surveys

The coordinate system in use on the property is UTM Zone 19S, PSAD-56 datum.

The topography used for the 2014 Feasibility Study was from a detailed aerial survey of the plant site area using a scale of 1:1,000 and 1 m contour spacing (prepared by Fugro Interra S.A. for Minera Santo Domingo, April 2012). Topography at 1:2,000 scale was used for other Project areas. The topography covers an area of approximately 16,000 ha for the plant site, port facilities and pipeline routes. The supporting grid for the Project and the pipeline system consists of six main points and a secondary grid of 53 points.

Fugro Interra S.A. provided a coordinate transformation program that allows coordinate conversion in various systems, WGS84 <> PSAD56 and WGS84 <> LTM.

A new global positioning system (GPS) network for the plant site was prepared including 20 survey monuments to be used for the next stage of engineering design. A topographic coordinate conversion program was provided to correlate data from one datum base to the other. All the survey restitution work was performed by GEOCEN Aerofotogrametría Digital.

9.2 Geological Mapping

Approximately 50 km² of geological mapping at 1:25,000 scale was completed during 2003–2005, and used for exploration targeting.

9.3 Geochemical Sampling

Far West Mining collected a total of 50 rock chip samples and 47 stream sediment samples (sieved to 100% passing 106 µm in the field) and generated copper and gold plots to assist exploration efforts.

Most rock chip samples were collected near copper showings and hence contain anomalous copper values. The gold plot shows that the mineralization within a 2 km radius of the Estrellita mine is commonly weakly gold-anomalous. These samples are generally from the narrow northwest-trending specularite and copper oxide-bearing veins cutting andesite flows.

Drainages in the areas underlain by andesite flows, especially in the north and northwest part of the Project area, are generally copper anomalous. These values
form a broad anomaly corresponding to northwest-trending specularite–copper oxide mineralized veins that cut the andesite rocks. Gold values in sediments are generally low.

9.4 Geophysics

9.4.1 Airborne

In 2002 BHP Billiton flew a Falcon™ gravity and magnetic survey over a portion of the Northern Chilean CIB, including the current Project area. Falcon™ gravity and magnetic susceptibility plots were produced based on information from Quantec Geofisica Limitada.

Gravity anomalies defined a cluster of northwest trending features up to 5 km long. Most of the significant mineralization in the Santo Domingo area is coincident with the gravity anomalies, and these areas were considered to be high priority exploration targets.

The magnetic susceptibility images show a widely-spaced set of northwest-trending faults, and less abundant northeast- and north–south-trending faults. The Santo Domingo fault cuts through the Estrellita deposit and the Estefánia mine workings and shows up as a series of coincident magnetic lows and truncated magnetic features that give this structure a tentative strike length of about 17 km. Many of the more significant mineralized zones in the Santo Domingo area appear to be related to this fault, and its entire surface trace was considered prospective.

The Santo Domingo gravity anomaly is a west–northwest trending feature approximately 4 km long by 1.5 km wide for much of its length. The eastern part of the target area may actually be a separate anomaly. It is a north–south-trending, sinuous, linear feature approximately 4 km long by 500 m wide. Most of the Santo Domingo gravity anomaly has coincident high magnetic susceptibility except where cut by faults which show up as linear magnetic lows. Andesite porphyry flows are the dominant lithology underlying most of the Santo Domingo gravity anomaly area. The eastern anomaly boundary is roughly coincident with an andesite–limestone contact.

The northwestern part of the Santo Domingo anomaly (Estrellita area) is generally parallel to a series of west–northwest striking faults as defined on the magnetic images, and to a closely spaced series of specularite and copper oxide bearing veins, stockwork and shear zones cutting the andesite. These mineralized veins occur both within the anomaly and outside the anomaly to the south.
Copper-bearing manto mineralization at the artisanal Estrellita mine workings underlies the westernmost part of the Santo Domingo gravity anomaly area. The direct association of mineralization with a gravity feature is unusual in the Candelaria area and hence the Santo Domingo target received a good deal of exploration work.

Magnetic susceptibility in the Santo Domingo gravity anomaly area is generally high except where cut by faults, most notably the east–west Santo Domingo fault and a prominent northwest trending fault along the southwest side of the anomaly. In drill holes into the northern part of the anomaly well away from the Santo Domingo fault the volcanic flows in the oxide zone (60 m to 120 m below surface) contained an average of 1% to 2% magnetite. Below this level the magnetite content, both disseminated and in magnetite mantos, was estimated to be 5% to 10%. These amounts of magnetite appear to explain the magnetic anomaly, and may also be responsible for the gravity anomaly. Magnetite in the oxide zone (near surface) and within the Santo Domingo fault (to depth) has been largely altered to specularite, and may explain the magnetic low along the fault. The gravity signature does not show a similar lineament, possibly because the alteration of magnetite to specularite does not change the bulk density.

The Santo Domingo Sur area is located in the extreme southeast part of the Santo Domingo anomaly. It was selected as an initial drill target because there were specularite–copper oxide mantos exposed on the flank of a 500 m wide gravity anomaly. The southern part of this anomaly has a coincident magnetic low which may in part be related to magnetite destruction (formation of specularite) along a northwest-trending fault. It has similar geological and geophysical signatures to the mineralized mantos at the Estrellita mine. Drilling within this gravity feature has outlined the Santo Domingo Sur deposit. The deposit has a gravity and magnetic signature that reflects the high magnetite and hematite content.

The Iris deposit is located along the eastern flank of the Santo Domingo gravity anomaly where mineralization formed in a fault zone that is more or less coincident with the eastern edge of the gravity feature. The deposit has an associated magnetic anomaly that is much wider than the deposit itself as the extent of magnetic iron oxide is greater than the extent of copper sulphide mineralization.

The Iris Norte deposit follows the eastern side of the same gravity anomaly that hosts the Iris deposit. The strike of the gravity anomaly is rotated by approximately 55° clockwise compared to the southern area that hosts the Iris deposit. Iris Norte has a magnetic expression that is less pronounced than that seen over Iris.
9.4.2 Ground

Far West Mining completed 17.6 line kilometres of IP survey through contractor Quantec Geofisica Ltda. of Antofagasta from April to August 2004. The survey was designed to test for chargeable zones within known gravity and magnetic geophysical anomalies. The time-domain IP survey used a pole-dipole array with a 100 m station separation on lines oriented perpendicular to the general trend of airborne gravity anomalies. Stations were located using a differential GPS.

The IP survey generated chargeability anomalies in various parts of the Santo Domingo area. Subsequent drilling of the anomalies demonstrated that IP is not a suitable method to distinguish between massive barren iron oxides and iron oxides that host copper sulphides. This is due to the fact that magnetite itself is chargeable and generates many anomalies in areas where barren iron oxide bodies are present. The application of IP as an exploration tool in the area was therefore discontinued.

9.5 Petrology, Mineralogy, and Research Studies

Detailed petrography and mineralogy studies have been completed on selected areas within the Santo Domingo Project. These studies have been completed to identify and quantify ore and gangue minerals and for the descriptions of textural variations in several rocks. Modal analysis studies (QEMSCAN) were performed on various mineralization types at Santo Domingo Sur to determine mineral species and their compositions for recovery tests and determining grinding parameters.

Two theses have been completed on the Santo Domingo deposit:


9.6 Exploration Potential

The Project has been explored for its large tonnage potential as a primary consideration. There has been no exploration targeting small lenses of mineralization in the 1–5 Mt range. Oxide mineralization has also not been targeted specifically.
The San Domingo Sur, Iris, and Iris Norte deposits and the Iris Mag prospect have been explored to a depth of approximately 350 m. Drilling below this level is very sparse, but deep drill holes at San Domingo Sur have intersected mineralization as deep as 650 m. The character and extent of deep mineralization has not been established and potential for additional mineralization exists.

Additional potential exists for iron mineralization without copper, which so far has been deemed uneconomic by itself but has potential once an operation is built in the Project area. The main iron potential is located around Iris Norte and to the south of San Domingo Sur where magnetite occurs in skarn zones of unknown size.
10.0 DRILLING

Between July 2003 and August 2012, a total of 589 core and RC holes (146,738.32 m) were drilled over the project as a whole. This drilling is summarized in Table 10-1 by purpose, and in Table 10-2 by hole type. Figure 10-2 provides a regional-scale collar location plan for this drilling.

Drilling that supports the resource estimate, or was used in support of the construction of the geological models comprises 464 holes (120,168 m). This drilling included 348 RC drill holes in the target area for a total of 90,611 m and 50 diamond drill holes for a total of 16,275 m that was completed by Far West Mining between May 2004 until June 2011. Subsequently, Capstone completed an infill campaign of an additional 12,140 m diamond drilling in 62 holes at Santo Domingo Sur, and 1,142 m in four holes at Iris Norte. This drilling is summarized in Table 10-3 and shown in Figure 10-2.

10.1 Drill Methods

Over the Project life, Chilean-based drill companies Harris y Cia., Major Drilling, Geo Operaciones and Captagua have undertaken drilling operations.

The majority of the RC drilling was conducted using a truck mounted Schramm Rotadrill and a centre return hammer and a 5.5 in. (13.97 cm) carbide button bit.

Core drilling used various drill rig types. HQ core (63.5 mm diameter) was typically drilled to a depth of approximately 300 m below which NQ core (47.6 mm diameter) was drilled.

The drill programs were originally designed to target gravity and magnetic anomalies for mineralization of the Candelaria or Manto Verde IOCG-style. Later programs consisted of core drill holes that were designed provide information on geology, mineralization and structure, and provide material for metallurgical testwork. Later RC drill holes were designed to tighten the drill spacing within the initial proposed mining areas and to provide sample material for metallurgical testwork.
### Table 10-1: Drill Summary Table by Hole Purpose

<table>
<thead>
<tr>
<th>Purpose</th>
<th>Number</th>
<th>Length (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Condemnation</td>
<td>18</td>
<td>4,818.00</td>
</tr>
<tr>
<td>Exploration</td>
<td>191</td>
<td>43,344.35</td>
</tr>
<tr>
<td>Exploration / Resource Definition</td>
<td>227</td>
<td>66,781.10</td>
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<tr>
<td>Geotechnical</td>
<td>21</td>
<td>3,531.70</td>
</tr>
<tr>
<td>Hydrologic</td>
<td>19</td>
<td>1,890.90</td>
</tr>
<tr>
<td>Metallurgy / Resource definition</td>
<td>103</td>
<td>24,808.27</td>
</tr>
<tr>
<td>Water Exploration</td>
<td>10</td>
<td>1,564.00</td>
</tr>
<tr>
<td><strong>Grand Total</strong></td>
<td>589</td>
<td><strong>146,738.32</strong></td>
</tr>
</tbody>
</table>

### Table 10-2: Drill Summary Table by Drill Hole Type

<table>
<thead>
<tr>
<th>Type</th>
<th>Number</th>
<th>Length (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>DD</td>
<td>51</td>
<td>9,823.40</td>
</tr>
<tr>
<td>RC</td>
<td>446</td>
<td>112,841.90</td>
</tr>
<tr>
<td>RC/DD</td>
<td>92</td>
<td>24,073.02</td>
</tr>
<tr>
<td><strong>Grand Total</strong></td>
<td>589</td>
<td><strong>146,738.32</strong></td>
</tr>
</tbody>
</table>

Figure 10-1: Project Drill Location Plan

Note: Figure prepared by RPA, 2013.
Table 10-3: Drill Summary Table – Drill Holes Supporting Resource Estimate

<table>
<thead>
<tr>
<th>Area</th>
<th>No. of Holes</th>
<th>Type</th>
<th>Length (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>SDS</td>
<td>103</td>
<td>RC</td>
<td>31,810</td>
</tr>
<tr>
<td>SDS</td>
<td>88</td>
<td>DD</td>
<td>22,837</td>
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<td>SDS</td>
<td>143</td>
<td>RC</td>
<td>30,528</td>
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<td>Estrellita</td>
<td>13</td>
<td>DD</td>
<td>2,366</td>
</tr>
<tr>
<td>Iris/Iris Norte</td>
<td>102</td>
<td>RC</td>
<td>28,273</td>
</tr>
<tr>
<td>Iris/Iris Norte</td>
<td>11</td>
<td>DD</td>
<td>3,212</td>
</tr>
<tr>
<td>Totals</td>
<td>464</td>
<td></td>
<td>120,168</td>
</tr>
</tbody>
</table>
Figure 10-2: Deposit Area Drill Location Plan

Note: Figure prepared by RPA, 2013.
10.2 Geological Logging

Drill cuttings and core were logged using a set of codes similar to those used for surface mapping. All geological data were entered digitally into summary logs. All digital data (analyses and geological logs) were subsequently entered into an MS Access project database. Data are exported as required to GEMS for presentation and interpretation purposes.

Core was placed into wooden core boxes by the drilling contractor at the drill site. The depth of each interval of core pulled was marked on a wooden block and placed in the core box. The core was then transported to a logging facility by personnel of the company at the time of drilling.

At the logging facility the core was photographed and a geotechnical log completed. Geotechnical data recorded included recovery, rock quality designation (RQD), fracture frequency, rock alteration and weathering, structure type, angle and roughness, joint compressive strength (JCS) and bulk density. Cut core samples with a length of 15 cm or 20 cm were also collected and stored for subsequent triaxial and point load tests.

The core was then geologically logged noting lithology, mineralogy, and other characteristics using the same codes employed for logging of the RC cuttings. Structural information was also noted during core logging, something that was not possible for RC cuttings.

10.3 Recovery

Overall sample recoveries tended to be quite good throughout all drilling programs. RPA reviewed the recovery data for 37 holes and found that the recovery was well within acceptable limits. Recovery was calculated as a ratio of the actual core length in the box to the drilled length as indicated on the metre blocks. It was noted that some intervals had recoveries greater than 100%, which is not realistic. These intervals tended to be just over 100%, and in RPA’s opinion, are probably due to slight gaps between pieces of core that caused inaccurate measurements. After normalizing all of these spurious values to 100%, the length-weighted average recovery was 91.1%.

RPA also noted the following:

- Minimum recorded recovery was 0% in one interval across a 2.2 m downhole interval
- 65 intervals had values greater than 100% recovery
• 24 out of the 4,199 measured intervals (0.6%) were below 50% recovery. All but four of these low recovery intervals were less than a metre in length.

10.4 Collar Surveys

Drill collars were located using a differential GPS. Coordinates are considered by RPA to be accurate to within 1 m or less.

Relative elevations between holes in close proximity (such as at Santo Domingo Sur) were determined using a tight chain and clinometer.

10.5 Downhole Surveys

Downhole surveying was conducted by Comprobe Ltda. (Comprobe) using a combination of gyroscope and accelerometer, with measurements taken every 10 m.

RPA notes that the downhole survey instruments are not affected by magnetic interference.

10.6 Sample Length/True Thickness

Most holes are vertical or near-vertical because the mineralization in the Project area tends to be horizontal or gently dipping. Approximately 25% of the holes included in the resource estimate were drilled at angles shallower than -80°.

Inclined holes, particularly core holes, were drilled in order to establish the limits of mineralization at the edges of the deposits as well as to establish the structural framework at Estrellita, Iris and Iris Norte.

Drill sections in Section 7 show the orientations of selected drill holes in relation to the mineralization at each deposit.

10.7 Summary of Drill Programs

10.7.1 Santo Domingo Sur

The Santo Domingo Sur deposit is defined by 191 drill holes (103 RC and 88 core holes), completed on an approximate 100 m spacing and reducing to 50 m spacing in the centre of the deposit. Drilling data indicate that the deposit strikes approximately northeast and dips at low angles to the northwest. A northwest-trending fault, only recognized in drill intersections, appears to have displaced the northeastern portion of the deposit down by approximately 45 m to 65 m.
The southern and eastern margins of the deposit are interpreted to be structurally-controlled and are defined by drill holes into adjacent structural blocks that have a different geology. The western margin appears to be a transitional boundary from the tuff sequence to a sedimentary sequence in the west with gradually weakening manto development.

Drilling below a depth of 350 m is sparse and mineralization below that depth is not well defined.

The deposit remains open at depth.

10.7.2 Iris and Iris Norte

The Iris and Iris Norte deposits are defined by 102 RC holes and 11 core drill holes, drilled on approximately 100 m spacings.

The Iris deposit forms part of the eastern flank of a gravity anomaly that strikes north–northwest. The deposit is truncated by a west dipping fault on the western side and by a steeply east dipping fault on the eastern side that divides volcanic tuffs and flows in the west from limestone and calcareous sediments in the east.

Iris Norte occupies part of the same gravity anomaly that hosts Iris, but in the Iris Norte area, the strike of the anomaly is north–northeast.

Both Iris and Iris Norte also appear to be open at depth.

10.7.3 Estrellita

A total of 156 holes (143 RC and 13 core holes) have been drilled in the Estrellita area.

Mineralization is faulted down by approximately east–west-striking faults to the south and north of the main zone around the old workings where drill holes intersected the mineralized zone at deeper levels. Vertical displacement along the faults varies from about 60 m to as much as 100 m.

The deposit remains open to the east and to the west. The zones are interpreted to be flat-lying, so down-dip extensions are unlikely; however, there is potential for additional mantos to occur below the presently-drilled area.
10.7.4 Exploration

Additional holes have been drilled to test other gravity and magnetic features in the Santo Domingo area and intersected widespread but discontinuous copper and iron mineralization around the four outlined deposits.

10.8 Geotechnical and Condemnation Drilling

10.8.1 Geotechnical Drilling

Geotechnical drilling was conducted by Far West Mining between 2006 and 2010 and comprised a total of 28 core oriented diamond drill holes (26 with geotechnical core logging), representing more than 7,000 m of core. The 2010 geotechnical campaign (four holes totalling 1,155 m) was supervised by AMEC. During 2011–2012 additional drilling was conducted by MSD to gather geotechnical data to complete slope calculations for the SDS/Iris pit and the Iris Norte pit. The 2011–2012 geotechnical/hydrological drilling campaign was designed and supervised by AMEC and consisted of 16 bore holes, for a total of 2,841 m for Iris Norte, SDS/Iris and the tailings area.

10.8.2 Condemnation Drilling

Condemnation drilling was conducted by Far West Mining during early 2011 and by Minera Santo Domingo during early 2012. A total of 3,576 m in 13 RC holes were drilled in the proposed waste dumps, process plant and tailings areas. The condemnation drilling was in addition to 5,627 m in 20 historic exploration drill holes that were drilled within the boundaries of the proposed mine site installations (waste dumps and process plant area).

10.9 Comments on Section 10

In RPA’s opinion, the drilling has been conducted in a manner consistent with standard industry practices. The spacing and orientation of the holes are appropriate for the deposit geometry and mineralization style.

RPA has not reviewed the geotechnical drilling or the condemnation drilling in detail.
11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sampling Methods

11.1.1 Geochemical Sampling

A total of 47 sediment samples were collected from drainages within and immediately peripheral to the Santo Domingo area. The samples were analyzed by ALS Chemex for gold and a 27-element ICP package. Most drainage channels in the area were sampled. Approximately 200 g of -106 μm material was collected from each sample site using an Endecott No. 140 sieve (or equivalent) and simple bubble plots of copper and gold in sediments were produced.

11.1.2 RC Drilling

Reverse circulation drill cuttings were blown into a cyclone and collected every 2 m from top to bottom of each hole, regardless of lithology changes. This material was dumped directly into a riffle splitter with a bar separation of approximately 1 cm. Both parts of the initial split were reintroduced to the splitter and divided a second time to ensure adequate mixing of the entire sample. Half of this initial split was re-split and then split again. These three consecutive splits resulted in a final sample one-eighth the size of the initial complete sample.

Apart from most overburden material and a few obviously barren bedrock intervals, all samples were sent for analyses.

A 2 kg to 3 kg portion of the final split was bagged and ticketed with a unique assay number, ready to be sent to the laboratory for analyses. A second sample of 3 kg to 4 kg was collected from the other half of the final split and stored (buried) at or near the drill site.

11.1.3 Core Drilling

Samples for assay were marked at 1 m and 2 m intervals by technicians and subsequently adjusted by the geologist to correspond to major lithological contacts. For programs conducted prior to 2011, sample lengths were not less than 0.5 m and most did not exceed 2 m. The shortest and longest sample lengths in 2011–2012 were 0.7 m and 2.7 m, respectively, and most samples were 2 m long. Sampled intervals were cut in half along the drill axis using a diamond saw. Half of the sample was returned to the core box and stored at the core facility. The other half was bagged and shipped (via ALS Chemex truck) to the ALS Chemex laboratory at La Serena or Antofagasta, Chile, for analyses.
11.2 Metallurgical Sampling

Metallurgical sampling is discussed in Section 13.

11.3 Density Determinations

Specific gravity (SG) determinations were performed by Far West Mining personnel on 1,990 core samples from 11 drill holes for different lithologies in the Estrellita deposit. Far West Mining made direct measurements on core samples using the water displacement method and calculated specific gravity via:

\[ \text{SG} = \frac{M_{\text{air}}}{M_{\text{air}} - M_{\text{w}}} \]

Where

- \( M_{\text{air}} \) = weight of the dry sample in air
- \( M_{\text{w}} \) = weight of the sample in water

In addition, 295 determinations were completed on RC samples from two drill holes at ACME Analytical Laboratories (ACME) in Chile. ACME used the pycnometer method in pulps prepared from RC samples. The specific gravity was determined with the following formula:

\[ \text{SG} = \frac{W_{s}}{W_{d} \times SG_{s}} \]

Where \( W_{s} \) is the weight of the sample; \( W_{d} \) is the weight of the displaced solvent; and \( SG_{s} \) is the specific gravity of the solvent. The common solvent is acetone but methanol can also be used.

RPA developed regression formulae based on the specific gravity values reported by Far West Mining to convert volumes to weights, using Fe concentration as the independent variable. The regression curve relationship was as follows:

\[ \text{SG} = 2.53 + 0.0276 \times \text{Fe}. \]

A summary of the specific gravity data is included by major lithological unit in Table 11-1.
### Table 11-1: Summary Table, Specific Gravity

<table>
<thead>
<tr>
<th>Rock Type</th>
<th>Num.</th>
<th>SG (t/m³)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Andesite</td>
<td>488</td>
<td>2.90</td>
</tr>
<tr>
<td>Andesite Tuff</td>
<td>685</td>
<td>3.05</td>
</tr>
<tr>
<td>Basement</td>
<td>9</td>
<td>2.85</td>
</tr>
<tr>
<td>Diorite</td>
<td>48</td>
<td>2.81</td>
</tr>
<tr>
<td>Dyke</td>
<td>70</td>
<td>2.69</td>
</tr>
<tr>
<td>Fault</td>
<td>34</td>
<td>3.06</td>
</tr>
<tr>
<td>Limestone</td>
<td>3</td>
<td>2.72</td>
</tr>
<tr>
<td>Manto (High Fe)</td>
<td>883</td>
<td>3.55</td>
</tr>
<tr>
<td>Sedimentary</td>
<td>7</td>
<td>2.75</td>
</tr>
<tr>
<td><strong>Total Number SG Samples</strong></td>
<td><strong>2,227</strong></td>
<td><strong>Average SG</strong></td>
</tr>
</tbody>
</table>

### 11.4 Magnetic Susceptibility

A total of 19,302 magnetic susceptibility determinations have been made to date. Plastic bags of sample reject material from the laboratory are shaken to homogenize the material then laid flat on a table. The magnetic susceptibility instrument is pressed against the plastic bag and the magnetic susceptibility reading is taken. Measurements are taken at four locations in the sample and averaged. If a significant deviation between readings occurs, the measurements are repeated until consistency is achieved between all four points.

Of the readings, 2,093 were conducted on pulps owing to the lack of remaining reject material. Measurements taken on pulps routinely yield lower readings than do those taken on rejects.

For quality assurance, 191 pulp reject samples from the 2011–2012 drilling campaign were submitted to ALS Chemex in Perth, Australia, for percent magnetite analysis. Capstone reported that a correlation factor of 0.943 between the average of four magnetic susceptibility readings and percent magnetite was achieved.

### 11.5 Analytical and Test Laboratories

The primary analytical laboratory was ALS Chemex, and the facilities in La Serena, Chile and Antofagasta, Chile were used. Both of these facilities have ISO 9001:2008 accreditation and La Serena has ISO 17025 accreditation.
The check laboratory was Andes Analytical Assay Ltda. in Santiago, which also holds ISO 9001:2008 accreditations.

11.6 Sample Preparation and Analysis

Upon arrival at the laboratory the RC and core samples were organized, recorded and prepared for analyses using ALS Chemex’s Prep-31 process.

This process consists of:

- Drying at 60°C
- Crushing (jaw crusher) to minus #10 Tyler >70%
- Homogenizing and splitting to 500 g with a Jones splitter
- Storage of reject material (over 500 g)
- Pulverizing 500 g sample with a ring pulverizer to minus #200 Tyler >85%
- Storage in 250 g envelopes.

All samples were analyzed for 27 elements using inductively-coupled plasma (ICP) methods. Samples were initially analyzed using ALS Chemex procedure ME-ICP61, which is ICP following four acid, total digestion (HF-HNO₃–HClO₄ acid digestion, HCl leach) and more recently by ME-ICP81.

Copper values over 10,000 ppm were assayed using ALS Chemex method Cu-AA62, which involved total digestion and an Atomic Absorption Spectroscopy (AAS) finish. Gold content was determined using method Au-AA24 (30 g sample, fire assay with an AAS finish). These analytical procedures conform to industry standards.

The ME-ICP61 protocol was recognized as understating the iron content, particularly for high grades. The upper limit for ME-ICP61 is 50% Fe; this resulted in a significant downward bias in the block model iron grades in previous resource estimates. For the 2010 program onwards, the ALS Chemex ME-ICP81 protocol was implemented. This incorporated a more aggressive digestion (peroxide fusion) and has no upper limit to the iron assays. A total of 7,401 samples were submitted for re-analysis using ICP81, including all samples over 15% Fe inside the existing block model for which sample material was still available.

Soluble copper analysis was conducted on 1,035 samples from 2011–2012 drilling at ALS Chemex in La Serena, Chile. Assay protocol was the ALS Chemex Cu-AA05 method for non-sulphide copper by dilute sulphuric acid leach and AAS finish on a 1 g sample.
11.7 Quality Assurance and Quality Control

An independent quality control/quality assurance (QA/QC) program was implemented by Far West Mining to monitor the analytical results. Three types of quality control sample inserts were utilized during the drilling programs:

- Standards
- Blanks
- Duplicates.

The QA/QC protocols have remained largely consistent throughout all programs conducted by Far West Mining and Capstone. Minor changes have been implemented by Capstone to accommodate issues and recommendations from past programs and to include magnetic susceptibility measurements.

Certified reference materials (CRM), or standards, are inserted every 25th sample, constituting 4% of the total number of samples submitted. Blanks, consisting of common Portland cement, were inserted every 50th sample. Field duplicates are taken every 25th sample.

11.8 Databases

Drill cuttings and core were logged and data collected entered into an MS Excel computer database. Each geologist was responsible for entering his/her own logs. Data from these individual “unproofed” logs were printed out and then checked line by line against the original handwritten log by a team of two geologists. Corrections were made and a “proofed” version of the individual log was saved. Each individual “proofed” geology log was then added to a “master geology” log. This master file was then available for further analysis and/or display by exporting the data in the required format.

A separate assay ledger is also kept for each hole. Initially sample intervals and numbers are entered manually into the ledger and then transcribed into an MS Excel spreadsheet. The initial ledgers or logs are completed by the samplers at the drill site for RC cuttings and at the core logging facility for core. Inserted blanks, standards and duplicates are also recorded in this ledger. Assay results, when available from the laboratory, are cut and pasted into the digital ledger from an MS Excel file provided by the laboratory. Once complete, data from the ledger are imported to a master MS Access database containing all the Project drill assays.
One person is responsible for management of the database, posting of final results and controlling user access. A copy of the database is imported to Gemcom (Gems) for interpretation and presentation purposes.

Data for density, magnetic susceptibility, and surveys are also captured in spreadsheets and then imported to the master Access database.

Capstone has a corporate policy on data backup, and the database is subject to regular backup procedures.

11.9 Sample Security

The logging facility is fenced, locked when not occupied, and is secure. Samples are handled only by company employees or their designates (i.e. ALS Chemex personnel).

Once leaving the drill camp on the property, sample security could not be confirmed. However, Capstone advises that, in virtually all cases, copper estimates in logged chips correlate well with analytical results.

11.10 Comments on Section 11

In RPA’s opinion:

The sampling methodologies employed by Far West Mining and Capstone are consistent with industry best practice and appropriate for the mineralization style. The sampling is configured such that it will be representative of the deposit as a whole.

The database is reasonably free from error and suitable for use in Mineral Resource estimation.

The standards assays were carried out at an acceptable insertion rate, were reviewed in a timely fashion, and the results triggered reasonable and appropriate responses. The standards results indicate that the assaying was generally of good quality and acceptable for use in Mineral Resource estimation.
12.0 DATA VERIFICATION

Data verification has been undertaken by third-parties in support of technical reports on the Project.

12.1 Höy and Allen (2005)

No details of data verification steps were reported in Höy and Allen (2005), but the authors noted:

“The project manager, G. Allen (P.Geo.) designed and participated in both the geological drilling data verification process and the QAQC procedures for the analytical work. In his opinion all reasonable steps were taken to ensure that data presented in this technical report are accurate and a true representation of the geology and mineralization encountered during the program. The author (T. Höy; P.Eng.) has visited the site, traversed in the area, examined and observed the collection and drill sampling procedures, and compared visual estimates of mineralized samples with returned assays. He also believes that the data presented in this report is a true representation of both the geology and mineralization of the camp”.

12.2 Lacroix, (2006)

Data provided to RPA for verification purposes were based on exports from the Gemcom database in MS Access format.

RPA independently verified a portion of the 2006 database by randomly selecting a hole on each drill section and comparing the copper, gold, and silver values in the provided data with the assay certificates from the laboratory and Far West Mining’s master Excel database. In total, assay results for the mineralized portions of six of the 34 holes that intersected the mineralized portion of the manto were verified.

No errors were found in the data provided to RPA, but a number of inconsistencies in the treatment of gold and silver assays were noted. These inconsistencies were not present in the ‘master’ database, but appeared to be the result of data in the Gemcom database being added incrementally as assay results were returned from the laboratory.

RPA collected several independent samples from the diamond drilling program underway at the time of the 2006 site visit. RPA noted that while not generally indicative of the average grade or meant to serve as duplicates, the results do confirm the presence of significant copper mineralization.
12.3 Lacroix and Rennie (2007)

In 2007, Excel spreadsheets from the laboratory were consolidated into a database for comparison with the assays provided by Far West Mining. The ICP data from the laboratory contained within 86 individual spreadsheets were combined into a table comprising 5,161 assay records. This table was then compared to the assay table in the GEMS database received from Far West Mining. Of these, 4,677 records could be matched via the sample identifier. There were no errors or discrepancies found in either the copper or gold entries.

Independent witness samples were collected from drilling programs at Iris (hole 156) and Estrellita (hole 188) during RPA’s August 2007 site visit. RPA concluded that while the witness samples were not generally indicative of the average grade or meant to serve as duplicates, the sampling results do confirm the presence of significant copper mineralization.

12.4 AMEC (2008)

AMEC checked the integrity of the Santo Domingo geological and assay database for the period of 2005 to 2007 and verified the completeness of the documentation supporting the geological and assay database.

Sixteen files corresponding to 8.06% of the drill holes included in the database for the Santo Domingo Sur deposit were reviewed. Files for the Estrellita deposit were not audited. AMEC compared drill hole logs and geological sections (in paper format) with cores and cuttings, with special attention to lithology and mineralized units and contacts. Lithology records in the paper logs were also compared with the database records. In AMEC’s opinion, the database was reliable and agreed with the geological interpretations, log records, and lithological observations.

AMEC randomly selected nine drill holes representing 7.25% of the total drill holes and compared the original certificate records against 25 assay records for Cu, Au and Fe included in the database. No discrepancies were found.

AMEC checked the collar locations for 12 drill holes in the field, corresponding to 7.1% of the drill hole database in the Santo Domingo Sur deposit. The locations where checked using a portable E-Trex GPS device. No significant differences were found in the collar locations.

The survey database reviewed by AMEC included 168 holes, of which only 18 holes had the collar orientation surveyed. Most of the down hole surveys were performed
using the gyroscope method. The average deviations were 0.53°/100 m in azimuth and 0.73°/100 m in dip, which were considered acceptable.

AMEC investigated the possibility of RC down hole contamination at Santo Domingo Sur for all RC drill holes. The decay and cyclicity analysis using AMEC in-house software showed no detectable down hole contamination.

12.5 Rennie (2009)

RPA compared Cu, Au, and Fe values in the database provided with individual certificates for 11 of the 52 holes drilled subsequent to the 2007 Mineral Resource estimate. No errors, inconsistencies, or discrepancies were noted.

12.6 RPA (2012)

In addition to the verification work undertaken by RPA and discussed above under Lacroix (2006), Lacroix and Rennie (2007), and Rennie (2009), RPA undertook the additional data verification steps in support of the 2012 Mineral Resource estimate in Section 14 of this Report.

The major findings of all four RPA verification programs are summarized in the following subsections.

12.6.1 Analytical QAQC

Standards

Certified reference materials (CRM), or standards, are inserted every 25th sample, constituting 4% of the total number of samples submitted. Standard samples are inserted into the sample sequence and analyzed by ALS Chemex in a normal way.

Eleven standard samples were purchased from CDN Resource Laboratories Ltd. (CDN); however, from 2004 to 2007 the majority of the inserted standards were from seven of the 11 standards. RPA reviewed the standards results for this time period, and noted that although most averaged close to the nominated best values, the assays for the two lowest-grade CRMs were marginally higher than the accepted range. It was further noted that several of the standards appeared to have been misidentified or mislabelled resulting in apparent assay failures that were, in fact, spurious.

Standards used in 2008–2009 included CDN-CGS-7, -8 and -11. Results for CDN-CGS-11 showed a large number outside of the acceptable limits. This was attributed to a problem with the CRM and samples in question were re-assayed using a different assay protocol. RPA also noted that the mislabelling issues with the standards had
largely been addressed. It was recommended that the CRM suite include an iron standard owing to the increased importance of iron in the resource estimate.

A new set of three standards (high, medium and low grade variants) were prepared in 2010, which included iron in the suite of elements. The standards results were compiled on a spreadsheet and plotted against the best value as well as the averages for the program. Assayed values were compared to the best value plus or minus 5% of the nominated confidence limit to check for accuracy. Results were also plotted against the mean of the CRM results ± two and three standard deviations to check for precision. RPA reviewed the plots prepared by project personnel. There were isolated instances of assays that were outside the control limits, but no evidence of significant concerns or systematic errors in the assaying.

During the 2011–2012 programs, standards were inserted at a rate of at least one every 25th sample. Standards assays comprised 7% of the total sample count in 2011-2012 drilling. The same CRMs were used as for the 2010 drilling.

Capstone personnel define failures as results outside three standard deviations from the best value. Re-analysis of part of the work order, typically 20 samples, was requested when one standard in a work order failed. If more than two standards failed, the entire work order was re-analyzed. The re-analysis results replaced original rejected results in the assay database and rejected results were stored in a separate table for auditing purposes. Results between two and three standard deviations from the best value were considered “performance warnings”. Two consecutive performance warnings triggered re-analysis of partial or whole work orders.

RPA reviewed the CRM results for copper, iron and gold and found that they were generally within an acceptable tolerance limit.

Blanks

Blanks, consisting of common Portland cement, were inserted every 50th sample during the 2004 to 2010 programs, and analyzed for copper as well as for gold if the copper results exceeded 0.1% Cu. More recently analyses for iron were included. For the most part the blanks results were within a reasonable tolerance, although some of the copper results suggested that either there was some contamination or the blank material contained a high background concentration of copper. Blank results for copper in 2007 averaged 60 ppm Cu, even after three of the highest assays were removed owing to apparent misidentification of the packets. RPA noted that this appeared to have been addressed in more recent drill programs.
For the 2011–2012 drill programs, the same one in fifty insertion rate for the blanks was used as in previous programs. A total of 5.5% of the total sample count comprised blanks. Results were reviewed as received, with failures deemed to occur at ten times the detection limit. The blanks material used alternated between a coarse landscaping stone and a fine Portland cement. There were two blank failures for the 2011–2012 drilling campaign. In RPA's opinion, this was an acceptable failure rate and there are no concerns regarding the blanks assay results for the 2011–2012 definition drilling.

**Duplicates**

The Project standard is for field duplicates to be taken every 25th sample. Core field duplicates consist of quarter-core splits. Prior to December 2005, RC duplicates were collected from the cuttings remaining after the primary sample had been taken. This protocol has since been modified and the duplicate now comprises a split from the primary sample.

From 2004–2007, duplicates were analyzed for copper and gold. The mean grades of the duplicates were observed to average lower than the originals. RPA conducted t-test analyses on the results and determined that the differences in paired values were not significant. Scatter diagrams and relative difference plots comparing original and duplicate results indicated also that there were no apparent biases.

RPA conducted t-test analyses on the duplicates for copper and gold from the 2008–2009 program, and found a bias in the gold results. The duplicate gold assays averaged 22% lower than the originals, with an 11% probability that the difference was not statistically significant. Re-runs of the duplicate analyses did not show the same bias.

Duplicated data from the 2010 drill program were collected and plotted on scatter and precision diagrams configured to show each duplicate pair relative to an error limit. RPA reviewed the duplicates results and confirmed that they were within an acceptable tolerance.

Field duplicate data from the 2011–2012 drill programs were plotted on scatter diagrams comparing the duplicate result to the original. RPA reviewed the duplicates results and noted that there were apparent biases in both the copper and gold results. However, when results from one outlying pair were removed the bias was eliminated. In RPA's opinion, this indicated that the field duplicate results were reasonably unbiased.
It was also noted that the degree of scatter was somewhat high, even for field duplicates. Capstone’s failure criterion for field duplicates is that 10% or less of pairs should differ by more than 30%. In RPA’s opinion, higher scatter between duplicate pairs can be indicative of higher natural variability in the metal content of the deposit, improper sampling and/or an issue in the laboratory with reproducibility. Increased variability of assays, regardless of the cause, can result in a reduction in the accuracy of local kriged block grades. In RPA’s opinion, the impact of this on the grade interpolation is not likely to be severe. In addition, the lack of any bias in the results indicated that there would not be any material effect on the global grades. RPA considered this to be an issue for follow-up with the laboratory, but it was not considered to be a serious concern with respect to the Mineral Resource estimate.

**Pulp Duplicates**

A suite of 77 pulp duplicates comprising 2.1% of the total was collected and sent to an alternate laboratory, Andes Analytical Assay Ltda. in Santiago. RPA reviewed the results of the pulp duplicates assays and noted that there were significant biases for gold, copper and iron. The copper showed a positive bias for the primary laboratory relative to the secondary (i.e., primary is higher) for lower-grade samples below about 0.25% Cu.

Biases were also apparent for iron and gold, but in the opposite sense; that is, the secondary laboratory results were higher than the primary. As with copper the bias occurs for only some samples.

For iron, there is a cluster of points showing between a -40% to -70% relative difference; however, the balance of the pairs display a reasonably tight scatter and no bias. Gold displayed a more subtle negative bias in the grade range of 0.05 g/t Au to 0.23 g/t Au. In RPA’s opinion, this could be due to some random effect and may not necessarily be indicative of a persistent bias.

In RPA’s opinion, the pulp duplicates for copper and iron displayed fairly obvious biases that require follow-up. The reasons for the biases are unknown, and the pulp duplicates results pose the potential of a moderately serious issue with respect to the laboratory. The other assay QA/QC results obtained to date have not raised any issues, hence it is unlikely that there is a systematic problem with the database. However, to have this issue outstanding is not consistent with good practice and RPA recommended that an effort be made to determine the cause, or causes, of the biases.
Magnetic Susceptibility

Magnetic susceptibility measurements were conducted to provide a basis for estimation of the proportion of the iron that could be recovered by magnetic means. As stated previously, most of the measurements were carried out on reject material. However, in 2,093 cases, due to a lack of reject material, it was necessary to use pulps, which tended to yield higher readings than rejects. In RPA’s opinion, there is a significant bias between magnetic susceptibility measurements taken on pulps versus rejects.

In order to ameliorate the apparent bias, a regression line equation was derived from a scatter plot of rejects versus pulps. The equation was then used to adjust the pulp measurements downwards to an estimated reject value. The regression line used to derive the equation was deliberately chosen to be conservative. In RPA’s opinion, the use of factors on analytical data is generally undesirable; however, in this case a conservative approach has been applied that is well supported with testwork, and hence is considered to be acceptable.

Checks have been routinely carried out on standard reference materials to confirm that the magnetic susceptibility instrument was reporting consistently. The reference materials comprised 13 different samples prepared from reject material. The results from these calibration tests were plotted in chronological order to monitor for instrument drift over time. Following the 2010 program it was noted that for some of the reference samples there were significant variations from earlier measurements.

For the most recent drill program it was determined that the instrument drift was sufficiently severe that a replacement was warranted. A new instrument was acquired and its readings were calibrated to the old ones to ensure consistency in the data. Measurements were taken with both old and new instruments on a suite of 551 samples from 10 holes and plotted on a scatter diagram (Figure 12-1). The regression line derived from this diagram was then used to adjust the new instrument readings relative to the old one. In RPA’s opinion, the scatter diagram displays a relatively tight clustering of points and a clear regression trend, suggesting that the correction process should yield results consistent with previous measurements.
Readings were collected from the 13 magnetic susceptibility reference standards over a period of one month with the new instrument and averaged. These average measurements were plotted against the average measurements obtained with the old instrument over the entire period for which it was used (shown as red squares in Figure 12-1).

In RPA’s opinion, the results from the calibration standards show good agreement with the measurements obtained from the drill samples which further supports the regression line correction curve.

**12.6.2 Twin Holes**

Several holes have been twinned over the course of the exploration work conducted on the project. Most of these twins were drilled in the 2010 campaign in order to acquire magnetic susceptibility data in areas for which sample material was no longer available for testing.

RPA matched intervals of 4 m composites for each of the pairs and plotted the grades for gold, copper and iron to compare the results. In RPA’s opinion, for most of the pairs, the assay results compared reasonably well. The data were observed to be
quite noisy at the 4 m resolution, however, it was generally noted that high and low grade zones matched, and that the grades tended to cluster in the same ranges. Only one pair of twinned holes (4a3-06-099/4a3-10-099-B) displayed significant differences that could not be attributed to things such as hole deviation. In RPA’s opinion, the twinning has provided a reasonably consistent verification of the earlier drill results particularly considering the differences in assay protocols and possible survey errors.

12.6.3 Mass Recovery/Magnetic Susceptibility

Magnetic susceptibility readings were used to estimate the proportion of the mass of each block that could be recovered by low intensity magnetic separation (LIMS) methods. Far West Mining conducted Davis Tube testwork in order to first determine if a saleable Fe concentrate could be produced and, secondly to calibrate the expected mass recovery to magnetic susceptibility.

In 2008, a bulk sample was collected by blending drill cuttings from a number of holes in Santo Domingo Sur and Iris. This sample was subject to bulk flotation to remove the sulphide components and the tailings from this process were subject to iron recovery testing. The results of the testwork indicated that LIMS would produce a good quality magnetite iron concentrate.

The iron mineralization at Santo Domingo Sur is dominantly magnetite which can be recovered by LIMS, and hematite which generally cannot be recovered by LIMS. Consequently, the assays for total iron collected to date do not provide a basis for estimation of the recoverable iron component. In Figure 12-2, more than 10,000 iron values are plotted against corresponding magnetic susceptibility measurements taken from laboratory rejects. The magnetic susceptibility value provides an indication of the proportion of magnetite within each sample; it can clearly be seen that there is no definitive relationship with total iron content. The magnetic susceptibility is bound by an upper limit representing the case where virtually all the iron present in the sample is magnetite.

Davis Tube, Satmagan and magnetic susceptibility tests were conducted on a set of 22 sub-samples from the bulk composite. A very strong linear relationship was found to exist between the magnetic susceptibility and both Satmagan and Davis Tube mass recovery readings. Capstone subsequently embarked on an expanded testing program in order to confirm the observed relationship and develop a reliable regression line equation for relating magnetic susceptibility to mass recovery (Figure 12-3).
Figure 12-2: Total Iron versus Magnetic Susceptibility

Note: Figure prepared by RPA, 2012. Magnetic susceptibility values are divided by 1,000.

Figure 12-3: Magnetic Susceptibility versus Mass Recovery

Note: Figure prepared by RPA, 2012.
There is a degree of scatter in the data points plotted in Figure 12-3. In order to model the mass recovery with magnetic susceptibility, Capstone chose an equation which skirts along the bottom of the point distribution. The relationship thus derived is:

\[ MR\% = (1.1063 \times MS) + (-0.003 \times MS^2) \]

Where: \( MS = \text{Magnetic Susceptibility Reading}/1,000 \)

This line represents the minimum mass recovery observed for the corresponding magnetic susceptibility values; in RPA’s opinion, this will result in conservative estimates of mass recovery.

### 12.7 Comments on Section 12

In RPA’s opinion:

- The standards results indicate that the assaying was generally of good quality and acceptable for use in Mineral Resource estimation.
- For the 2004–2010 drill programs, for the most part the blanks results were within a reasonable tolerance, although some of the copper results suggested that either there was some contamination or the blank material contained a high background concentration of copper. There are no concerns regarding the blanks assay results for the 2011–2012 definition drilling.
- Field duplicate results are reasonably unbiased.
- The pulp duplicates for copper and iron display fairly obvious biases that require follow-up.
- The approach used for monitoring and calibration of the magnetic susceptibility instrument is reasonably rigorous and indicates that the magnetic susceptibility data should be valid.
- Twin hole drilling has provided a reasonably consistent verification of the earlier drill results particularly considering the differences in assay protocols and possible survey errors.
13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Metallurgical Testwork

A summary of the metallurgical testwork performed to date is included in Table 13-1.

13.1.1 Physical Characterisation

During the pre-feasibility phase, a total of 128 core samples were selected, with the following tests completed:

- Bond Ball Mill Work Index (BWi)
- Rod Mill Work Index (RWi)
- Semi-autogenous grind (SAG) Mill Competency (SMC)
- Abrasion Index (Ai)
- 110 samples defined as Derrick Barratt & Associates (DJB) or Main Drill Core Samples; results from this testing were used as a preliminary estimate for comminution requirements
- JKSimmet SMC testing was completed on an additional 19 drill samples. These results were used to finalise the PFS comminution requirements for the grinding circuit.

As part of the feasibility study phase, the following was undertaken:

- Two comminution circuit sizing exercises were completed
- 91 of the previous 110 PFS Main Drill Core samples were re-evaluated using the SMC test methodology to verify the comminution circuit throughput capacities
- A second SMC testing campaign using 58 Infill Samples was completed to increase the data set and confidence level of the evaluation of mill power requirements. The 58 Infill Samples were taken within the area of the proposed Santo Domingo open pit and were obtained from within the area planned to be the source of the material processed in the first three years of mine production
## Table 13-1: Metallurgical Testwork Summary Table

<table>
<thead>
<tr>
<th>Date</th>
<th>Testwork Type</th>
<th>Laboratory/Testwork Facility</th>
<th>Work Performed</th>
</tr>
</thead>
<tbody>
<tr>
<td>2006</td>
<td>Comminution</td>
<td>SGS Santiago</td>
<td>Grindability response testwork on two drill core samples</td>
</tr>
<tr>
<td>2008</td>
<td>Comminution</td>
<td>SGS Mineral Services</td>
<td>Grindability response testwork on five composite drill core samples. Included Bond Ball tests (BWi); Bond Ball Modified tests; Bond Rod tests (RWi); and SPI tests</td>
</tr>
<tr>
<td>2008</td>
<td>Cu flotation</td>
<td>SGS Santiago</td>
<td>Two master composites (MC-A and MC-B). Copper rougher kinetic, copper and pyrite rougher kinetic and copper cleaner and pyrite rougher flotation tests; magnetic separation tests on pyrite rougher flotation tailings; pyrite rougher flotation on copper rougher tailings to maximise recovery of sulphur from the flotation rougher tailings</td>
</tr>
<tr>
<td>2009</td>
<td>Magnetic concentrate</td>
<td>SGS Lakefield</td>
<td>Response of composite samples to magnetic separation using Davis Tube laboratory tests</td>
</tr>
<tr>
<td>2009</td>
<td>Magnetic concentrate</td>
<td>SGA Germany</td>
<td>Low intensity magnetic separation (LIMS) testing to develop a marketable magnetic concentrate</td>
</tr>
<tr>
<td>2010</td>
<td>Comminution</td>
<td>SGS Santiago</td>
<td>Grindability program on 128 samples; ball mill calibration program on four samples</td>
</tr>
<tr>
<td>2010</td>
<td>Cu Flotation</td>
<td>SGS Santiago</td>
<td>Copper mineralogy</td>
</tr>
<tr>
<td>2010</td>
<td>Cu Flotation</td>
<td>Aminpro</td>
<td>Chemical and mineralogical analysis (Qemscan) and rougher kinetic flotation tests on five samples; all rougher flotation tests were conducted using sea water</td>
</tr>
<tr>
<td>2010</td>
<td>Magnetite Concentrate</td>
<td>SGS Lakefield</td>
<td>Copper flotation performance and recoveries on five samples of the same composite; copper performance and recovery testwork on three different composite samples from Santo Domingo Sur; testing used three different water types: Lakefield water, Capstone-supplied saline water, and synthetic seawater; five flotation tests to determine the effect of primary grind on copper recovery</td>
</tr>
<tr>
<td>2011</td>
<td>Comminution</td>
<td>Ammtec (now ALS-Ammtec)</td>
<td>SAG mill competency; confirmatory ball mill tests; 19 samples tested</td>
</tr>
<tr>
<td>2011</td>
<td>Cu Flotation</td>
<td>SGS Santiago</td>
<td>Copper flotation performance testing on four composite samples (8 Year, Hematite, Magnetite and Oxide) and a set of 38 variability samples; investigated optimized use of sea water</td>
</tr>
<tr>
<td>2011</td>
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<td>Magnetite Concentrate</td>
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<td>Magnetite recovery program using three composite samples selected by AMEC and Capstone; optimum magnetic rougher and cleaning grind sizes and conditions including washing and magnetic strengths was confirmed for design purposes</td>
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The Main Drill Core samples returned the following results:

- For the Iris and Iris Norte ores there is no significant difference in competency between Magnetite and Hematite ores. The Iris ores are the softest materials, with an average $A \times b$ value of about 80.
- The BWi showed no significant differences between the Hematite and Magnetite zones for the Iris and Iris Norte areas.
- The Santo Domingo material shows significant differences between the Hematite and Magnetite feed types, with a 25% variance in the $A \times b$ values of the ore types. The Hematite zone is the most competent with the lower $A \times b$ average value of 39.3.
- In the Santo Domingo deposit, the Hematite zone is 15% harder than the Magnetite zone with respect to BWi, with BWi indices of 14.1 kWh/t and 12.1 kWh/t respectively.

The Infill Samples had the following characteristics:

- The Hematite ores have an $A \times b$ value of 42.2 and are the most competent ores in the first three years of operation. The Magnetite ores are 10% less competent with an average $A \times b$ of 46.5.
- There is a similar trend with respect to the BWi with the Hematite ores being 15% harder than the Magnetite ores with BWi values of 13 kWh/t and 11.2 kWh/t respectively.
- The RWi showed differences of around 10%, with averages of 14.4 kWh/t and 13.4 kWh/t for the Hematite and Magnetite ores respectively.

**Comminution Circuit Energy Calculations**

The JK Simmet methodology was used to determine the throughput capacity for SAG milling using the $A \times b$ parameter. Based on operating experience in Chile on various feed types, a correlation has been established between the $A \times b$ parameter and specific power consumption (CEE) when operating the SAG mill in an semi-autogenous-ball milling-crushing (SABC-A) type circuit with the discharge pebbles crushed and returned to the feed of the SAG mill. This allowed a correlation equation to be developed from the laboratory data to predict the SAG mill throughput capacity.

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1 A “SABC-A” circuit is where the pebbles discharging the SAG mill are screened and crushed in a pebble crusher, then the crushed pebbles are returned to the SAG mill feed.
The Bond methodology was used to estimate the power required to reduce the material from the feed size (F80) to the product size (P80) in the secondary (ball mill) grinding circuit.

For the samples selected, tested and averaged by deposit area and feed type, the throughput rates determined by calculation exceeded the design production rate of 65,000 t/d for the direct SAG (DSAG) circuit, except for the Santo Domingo-Hematite material type where a throughput rate of only 58,538 t/d was calculated. Grouping the results by specific feed type only, the treatment capacity for a DSAG circuit for Hematite and Magnetite feeds was determined to be 63,900 t/d and 77,313 t/d, respectively.

Information provided by Capstone in the life-of mine (LOM) Production Plan V8.1 shows that the distribution in the first five years of the mine operation will be an average blend containing 29% hematite and 71% magnetite ores. The results indicate that the throughput rate for a DSAG circuit will be approximately 65,000 t/d for the first five years. In subsequent years, when feeding the DSAG circuit with predominantly Hematite material, the throughput rate of the comminution circuit will reduce to a nominal throughput rate of 60,000 t/d.

13.1.2 Copper Flotation Testwork

SGS Lakefield conducted 38 variability sample locked cycle test (LCT) flotation tests on the 8 Year, Hematite and Magnetite composite samples as part of the PFS. The 8 Year sample gave the best copper recovery result.

Flotation testwork was conducted during feasibility-level studies to define the residence times for the rougher and cleaner circuits and the correct dosage locations, amounts and conditioning times for sodium metabisulphite (SMBS). Two separate testwork programs, the Short Term and Long Term programs, were completed and the results were used to finalize the process design.

The Short Term program consisted of Hematite and Magnetite composite samples of approximately 40 kg each. Both samples had copper grades, soluble copper, sulphur and iron contents similar to the life-of-mine feed types. Open circuit testwork (OCT) conditions were:

- Grind stage: ball mill (top size 1”)
- Regrind stage: ball mill (top size ½”)
- Grind stage: 65% w/w
- Regrind stage: 50% w/w
• Flotation paddle: every 10 seconds
• Lime used: Ca(OH)$_2$ powder and 10% milk of lime.

Based upon the OCT results the following set of conditions were used for the Short Term LCTs for both the Hematite and Magnetite sample composites:

• No addition of SMBS to rougher flotation section
• No addition of SMBS to cleaner flotation section
• No addition of NaCN to the cleaning flotation section
• Natural pH levels for roughing stage and 150 g/t addition rate of lime to cleaning stage
• The same reagent scheme, frother can be reduced due to recirculation of water during the LCTs.

The flowsheet used for the locked cycle tests comprised:

• The flowsheet is a conventional rougher flotation circuit with flotation tailings reporting to the magnetite recovery circuit
• Rougher circuit concentrate is reground and fed to the first cleaner flotation stage
• The first cleaner flotation stage concentrate is fed to the second cleaner stage and the first cleaner stage tailings are fed to a scavenger circuit
• Scavenger concentrate is fed to the regrind circuit and then to the first cleaner bank; the scavenger concentrate reports to the final flotation tailings by-passing the magnetite recovery circuit
• The second cleaner circuit concentrate reports to the third cleaner flotation circuit; third cleaner concentrate, final concentrate and third cleaner tailings report back to the second cleaner circuit.

It was evident from the Short Term program LCT results that the process residence time and reagent conditions would need to be modified in order to achieve the copper concentrate grades and recoveries reached during the PFS testwork program. The global recoveries achieved were just over 81% to final concentrate. From the rougher circuit recoveries (which achieved 95% copper recovery) it was probable that copper losses in the first cleaner and scavenging circuit were the result of reduced flotation kinetics due to the use of sea water. The use of sea water resulted in pyrite activation and competition with the copper mineral species present.
The procedure developed for the Long Term program provided an optimized flotation solution with final copper recoveries and grades closer to those achieved during the PFS testwork program.

Three separate composites (5 Year Average, Hematite and Magnetite) and 15 variability samples were selected for the Long Term program. The samples were characterized as:

- Hematite material is defined as material with a magnetic susceptibility of between 2,000 and 8,000. The average copper grade for the Hematite material is 0.59%
- Magnetite material has a magnetic susceptibility level above 8,000. The average copper grade for the Magnetite material is 0.31%
- The 5 Yr Average composite was selected to be representative of a combination of Hematite and Magnetite material during the first 5 years of operation. The average copper grade for the 5 Yr Average composite was 0.56%.

For the Long Term program, the samples were blended in proportion to the lithologies to represent the areas of the deposit that would be processed. The samples were also blended to ensure that they did not contain soluble copper levels exceeding 10% of the Cu feed grade. High oxidation (soluble copper) levels indicate waste material located within the deposit.

Core samples were cut at the Capstone storage area at Diego de Almagro and shipped to SGS in Santiago where they were weighed, split and composited for testing. Sample splits were taken for head characterisation and Qemscan analysis to have a clear understanding of the materials to be treated and to assist in the development of the reagent schemes to be used for each of the composites.

**Mineralogy**

Each of the composites was found to be different with respect to the ratios of the contained copper minerals and liberation of the copper species present. The copper minerals present were also noted to have different associations. The copper species present within each of the three composites in various percentages include chalcopyrite, covellite, chalcocite and bornite (Figure 13-1). Chalcopyrite is the dominant species within the entire deposit.
The conditions used in the kinetics tests were based on the PFS program and the optimized conditions from the Short Term program. Pyrite depression was not used in the rougher stage. Tests were based on all pyrite rejection taking place in the first cleaner and cleaner scavenger stages and an extended conditioning period for the SMBS. These conditions were achieved using pH control and adjusting the quantities and rates of the SMBS added.

The rougher kinetics test results were reviewed before the cleaner test program was started. Program observations included:

- The highest Cu recovery was reached in the test using an SMBS addition rate of 50 g/t (92.8% recovery); elevated pH levels did not improve Cu recovery. The use of SMBS in the cleaning circuit increased the Cu grade significantly in the first
cleaner stream. This was determined using results from tests completed using no SMBS and compared to tests with SMBS. Significantly higher Cu grades were noted when SMBS was used

- There is a significant difference in the Cu recovery achieved at the beginning of the first cleaner flotation stage (approximately 10% difference when comparing the highest Cu recovery of 72.2% at two minutes versus the lowest Cu recovery achieved of 53.8%) due to the use of SMBS

- From two minutes to six minutes there is a notable difference in Cu recovery (between 15% and 25% for the different tests). After six minutes, the Cu recovery rate reduces significantly and only increases slightly until the end of 25 minutes. It appears that Cu recoveries are still increasing after 25 minutes but at a very slow rate

- The cleaner kinetic test results show that global recoveries in excess of 90% can be achieved for each of the composites tested with concentrate head grades of between 28% Cu and 33% Cu. The results show that with sufficient conditioning time using SMBS, and using extended roughing and cleaning residence times, higher copper recoveries and grades in concentrate can be achieved.

Long Term program cleaner kinetic test results were used to define the conditions for the locked cycle tests. The LCT tests were conducted with seven cycles.

Prior to the LCT, OCTs were conducted to determine the amount of material re-circulating in intermediate streams and the potential copper grades obtainable for each of the composites. The OCT results showed that global recoveries in excess of 90% can be achieved for each of the composites tested with grades between 28% Cu and 33% Cu. The results demonstrated that with sufficient conditioning time using SMBS, and using extended roughing and cleaning residence times, higher copper recoveries and grades could be achieved for the samples tested.

The LCT results confirmed the OCT results for the 5 Year Average and Hematite composites with similar Cu concentrate grades and recoveries being maintained. The LCT tests for the Magnetite composite showed a lower Cu concentrate grade than the OCT partially due to the lower Cu head grade maintaining standard LCT test conditions which were determined from the 5 Year Average sample.

Based upon the 5 Year Average and Hematite composites, it was determined that a conversion of 0.993 should be used when converting OCT recovery results to LCT recovery results. For the Magnetite composite, it was determined that for a copper concentrate grade of 28.6% Cu, a copper recovery of 92.3% would be achieved.
Variability Testing

The objective of the variability sample testing was to confirm the OCT and LCT results using a standard flowsheet and test conditions derived from the 5 Yr Average composite testing. Variability samples were selected to represent the Santo Domingo deposit and included consideration of:

- Cu and Fe head grades
- Spatial representation
- Lithology
- Within the limits of mineralization as defined by Production Plan V8.1.

These samples were selected to obtain variable copper and magnetic susceptibility (iron content) values and representing the major lithologies within the deposit. Samples were also selected for a low soluble copper content, and were sourced from below the oxide zone cap. The predominant lithology groups were ANDS (ANDT and ANDE) and MANTO. Combined, these two lithologies represent more than 90% of the deposit.

Variability test results include:

- The two samples (Var 7 and Var 8) were tested for both OCT and LCT
- Var 7 had an OCT Cu recovery of 87.9% at a concentrate grade of 40.1% Cu, and an LCT Cu recovery of 89.7% at a concentrate grade of 37.4% Cu. Both of these Var 7 concentrate grades are high and an increase in Cu recovery could be expected at a reduced copper concentrate grade. A Cu recovery well in excess of 90% could be expected at a concentrate grade of over 30% Cu.
- Var 8 had an OCT Cu recovery of 93.3%; at a concentrate grade of 32.3% Cu and an LCT Cu recovery of 93.6% at a concentrate grade of 27.7% Cu. For Var 8, Cu recovery has been maintained at the expense of concentrate grade.

Due to variability sample availability it was decided to combine the SGS Lakefield and SGS Santiago variability results with those of the SGS Santiago composite samples to derive the final Cu recovery algorithm.

During the PFS, a copper grade vs Cu recovery algorithm was developed considering only the final (third) cleaner Cu concentrate value. The methodology used in the PFS was not repeated for the 2014 Feasibility Study, due to the use of sea water and SMBS (for pyrite depression, replacing sodium cyanide due to environmental considerations). The resulting reduction in flotation kinetics required that the
intermediate stream (IS) values be incorporated into the final Cu recovery value for the OCT tests.

The average copper recovery for all the samples (PFS samples, feasibility study variability samples, and feasibility study composite samples) was calculated to be 89%, with a maximum of 95% and a minimum of 81.6%. The 89% recovery was recorded at an average head grade of 0.38% Cu. The algorithm developed generates a calculated plant result of 88.4% recovery for the same 0.38% Cu head grade and takes into account the scale-up from bench scale to plant scale.

Copper Concentrate Thickening Tests

Outotec conducted thickener settling testwork on a third and final cleaner concentrate generated from a composite known as 8 Year Average or 8Y as part of the pre-feasibility phase. The concentrate sample was prepared using synthetic sea water, which had been prepared by adding sea salt crystals to fresh water.

A comparison was carried out between anionic flocculants MF10 and MF155 using static settling testwork. A higher settling velocity and improved overflow clarity were obtained using MF10.

Two dynamic thickening tests were carried out to analyze the effect of the feed rate (t/h/m²) on the discharge (Cp%) and the clarity of the overflow (ppm). A rheology test of the thickener discharge was also completed. Feed rates used were 0.25 t/h/m² and 0.40 t/h/m². The behaviour showed a reduction in the Cp of the discharge with an increase in the feed rate, although for both tests a Cp over 28% in the discharge and clarity less than 50 ppm in the overflow were obtained. In spite of the clarity of the overflow, the concentrate formed foam due to the reagents used in flotation. Therefore, a de-aerator was included in the design prior to the thickener.

Outotec stated that testwork should be conducted with feed rates higher than 0.25 t/h/m², and recommended that the testwork be undertaken using sea water without flocculant addition, to reach the design Cp of 60% w/w solids in the underflow discharge. The use of a low flocculant dose (around 3 g/t) was suggested to obtain the required 60% solids w/w in the discharge. Based on the results of the sample tested, the operating design parameters used for the copper concentrate thickener are as follows:

- Type of thickener: high rate (HRT)
- Feed to thickener: final cleaner concentrate
- Particle size (P80): 45 µm
- Settling rate: 0.25 t/h/m²
- Flocculant dose: 3 g/t
- Percentage of solids in discharge: 60% w/w.

**Copper Concentrate Filtration Testwork**

Outotec also conducted filtration testing on copper concentrate samples during the pre-feasibility study phase to evaluate the filtering characteristics using ceramic (CC) and pressure (PF) Larox filters and to evaluate the effect of washing the concentrate. Recommendations arising from the work included:

- The use of filter presses; these were considered to have an advantage compared to ceramic filters because they operate with a higher filtration rate and a lower consumption of wash water
- A design filtration rate of 495 kg/h/m² which is conservative based on the testwork results obtained
- Use desalinated water with a chloride concentration of 250 ppm for the concentrate washing at a water consumption rate of 0.1 m³/t.

The recommended design parameters are:

- Type of filter: press
- Feed size p80: 52 µm
- Rate of filtration: 495 kg/h/m²
- Moisture of filtered concentrate: 8% to 10% w/w
- Rate of concentrate washing: 0.1 m³/t.

The maximum chloride content is:

- Liquid phase of filtered concentrate: 300 ppm
- Water from concentrate washing: 250 ppm.

**13.1.3 Magnetic Separation Testwork**

During the PFS, several tests were carried out to set the recovery of magnetite using the tailings from the primary copper flotation stage of the 8Y composite sample.

Tests were carried out using a Davis Tube (DT) bench-scale machine on tailings from the primary copper flotation step. The results indicated that for a grind size of 100% passing 0.063 mm, the highest iron grade obtainable in the final concentrate was
66.5% Fe w/w. With a grind size of 100% passing 0.1 mm, the maximum Fe grade obtained in final concentrate was 64.0%. Grinding to less than 100% passing 0.04 mm showed no improvement in the Fe grade with a maximum grade obtainable of 66.1%.

Tests were also performed for the primary (routhier) magnetic separation stage using both fresh water and sea water. The results show that the weight recovery percentage for the primary stage using fresh water was 26.5% of the initial weight, slightly lower than the recovery obtained for the tests conducted with sea water. The percentage of recovered magnetite in the primary magnetic separation stage with fresh water was 94.2% of magnetite, which is slightly higher than the result obtained with sea water at 93.4%. The Fe grade in the roughtier concentrate grade is slightly lower using sea water at 53.9% Fe w/w versus fresh water at 56.2% Fe w/w.

The roughtier magnetic separation product was re-ground and then a second (cleaning) stage of magnetic separation was carried out using fresh water. A total of 10 regrind cleaner tests were analysed. The Fe grades achieved were acceptable varying between 64% and 68% and reflected the fineness of grind employed. The SiO$_2$ grades varied inversely with the Fe grade and grind with the finer grind producing lower final SiO$_2$ grades. The Blaine index measured indicated that the filtration and washing would be more effective at finer grind sizes and this would reduce the deleterious element levels.

At the PFS stage, three different composite samples were tested to verify the mineralogy of the deposit:

- 8Y composite
- Hematite sample
- Magnetite sample.

The samples were ground to the established primary flotation tailings grind size. The results show that the mineralogy of the feed influences the concentrate product produced. The final iron grades achieved were 64.1% for Hematite, 66.6% for Magnetite and 66.1% for the 8Y composite.

For the feasibility phase, testwork was completed by ALS Ammtec on three composite samples of copper flotation tailings produced in early 2014 to confirm the anticipated magnetite concentrate quality achieved during the magnetic separation process. A comparison of these testwork results with prior test programs was also completed. Three copper flotation tailings samples were tested:

- 5 Year Average Composite #2 (sample derived from the Long Term testwork program at SGS Santiago)
• Hematite (sample derived from the Short Term testwork program at SGS Santiago)
• Magnetite (sample derived from the Short Term testwork program at SGS Santiago).

Based on testing a range of grind sizes for the three samples and subjecting each size distribution to a DT test, the optimal cleaner low intensity magnetic separator (LIMS) feed grind size was determined to be a P80 of 40 µm. This was compared to the original PFS design grind size P80 of 45 µm.

The results of a DT test for a given grind size indicate the expected performance of the LIMS circuit for the material ground to a similar size. The DT testwork indicated that all three concentrates (5 Year Average, Hematite and Magnetite) could potentially achieve the concentrate specification of less than 4.1% SiO₂ content if the feed to the cleaner LIMS is ground to a P80 of 40 µm. The final concentrate particle sizes measured varied in the range of P80 of 37 µm to 41 µm.

Grinding the material finer than 40 µm was not recommended due to the following reasons:

• Based on the testwork completed to date, the concentrate quality can be only marginally improved by grinding the LIMS feed to a size less than P80 of 40 µm
• Grinding to a size of less than P80 of 40 µm could lead to iron losses from ultrafines, due to the fine particle size distribution of this material
• Commercial LIMS machines have been observed to have difficulties collecting ultrafine iron.

The three ALS Ammtec final concentrate particle size distributions from the 2014 testwork were similar to those obtained from the 8 Year Average sample evaluated by SGA in 2011. The 8 Year Average concentrate was used to develop the target product specification. The three ALS Ammtec concentrates had a P50 size of between 20 µm and 25 µm, further supporting the recommendation of maintaining a P80 of 40 µm. The performance of the three ALS Ammtec samples was in line with the previous program results, and also supported the DT concentrate mass recovery versus magnetic susceptibility relationship developed for the magnetite concentrate recovery predictions.

Tests using coarser grind sizes for the three samples were completed as part of the ALS Ammtec testwork. The results demonstrate the concentrate quality expected using coarser regrind sizes.
Magnetic Separation Variability Testwork Review

The PFS database consisted of a total of 211 samples from ALS Chemex, SGA and CMP, and included test samples located both within and outside of the proposed Santo Domingo and Iris Norte pit limits. If the results from samples located outside the proposed pit limits are excluded, there are 164 test samples for the planned Santo Domingo pit area and 15 test samples for the planned Iris Norte pit area.

The Santo Domingo samples were classified based on major lithologies. Overall, the target iron grade was achieved with the variability samples tested. However, the samples with test results that were outside of the target specification were found to be located adjacent to fault zones and had a high degree of alteration. It is anticipated that during production planning these alteration zones will be reclassified as waste with respect to recoverable magnetite.

All of the major lithologies are represented in the testwork data set; however, the ANDE lithology has a lower proportional representation in the data set. AMEC conducted an analysis of the likely block model outputs and confirmed that the ANDE lithology represents between 13% and 17% of the life-of-mine feed.

Testwork indicated that for some of the ANDE samples, the contained silica does not separate to the same degree as it does for the MANTO samples. The results do not distinguish between magnetite core samples and hematite rim samples. Of the 164 Santo Domingo samples tested, there were only a few samples that contained low iron levels. The test with low iron produced magnetic concentrates with silica content that exceeded the target specification, indicating that there is a potential complex association between silica and iron. Additional optimization and analysis of these results will need to be conducted to generate additional magnetite product from the low magnetic susceptibility feed samples.

The major feed types in the Iris Norte deposit are MANTO and ANDT with approximately equal proportions. The highest silica content measured in the 15 samples tested was 7.75% and the lowest was 2.08%. The weighted average SiO₂ and Fe contents in the magnetic concentrates from these samples were 3.4% and 67.7% respectively. Both of these results are within the target specification.

Magnetite Concentrate Thickening

Outotec undertook testing to determine the main characteristics of dynamic settling and thickening of the magnetite concentrate slurry using a bench-scale dynamic thickener. An Outotec 99 mm diameter laboratory thickener was used. Ten thickening tests using dilutions of 20%, 25%, 30%, 35% and 55% w/w solids were performed. An
additional test with the thickener operating continuously was carried out, simulating an industrial operation. The results indicate that the yield stress in the thickened solids increases from 30 Pa to 210 Pa as the solids in the discharge increases from 73% to 79% solids w/w. The thickener will operate with a nominal discharge density of 65% solids w/w and a design maximum of 70% solids w/w. Outotec concluded that for the magnetite concentrates the optimal feed density is 35% solids w/w (design values of 55% solids w/w can be used). Outotec also stated that the optimal feed rate was between 0.5 t/h/m² and 0.6 t/h/m², achieving a clarity of about 200 NTU and 74% solids w/w in the discharge using laboratory bench scale equipment.

The objective of testwork undertaken by Delkor was to determine settling rates, evaluate the fluidity of the slurry, and determine the capacity of the production thickening equipment using bench-scale laboratory equipment. For the thickening tests using magnetite concentrate, 19 settling tests and three compaction tests were performed. Delkor carried out free settling tests with anionic flocculant AP2020. Slurry dilution densities between 17% and 20% solids were used with flocculant addition rates from 0 g/t to 7 g/t. The results for the 17% dilution density test indicated that settling rates varied between 18 m/h and 9 m/h as the flocculant dose was increased from 0 g/t to 7 g/t. Delkor carried out forced compaction tests (with rakes) and free compaction tests (without rakes) and observed the highest densities over 24 hours. Flocculant was not used in these tests. The results indicate that for both cases, the solids density after 24 hours is more than 75% w/w.

Delkor carried out rheology tests to assess the fluidity of the slurry at different solids concentrations using the material from the compaction tests. The results indicated that for a concentrate slurry with 65% solids, the unsheared yield stress is approximately 12.4 Pa and for concentrate slurries at 68% and 70% solids, the yield stresses were 27.1 Pa and 34.4 Pa, respectively.

AMEC recommended that a unit value of 0.68 t/h/m² is used for the design of the magnetite concentrate thickener which results in a 36 m diameter high rate thickener without a clarifier. This is in the mid-range of the recommendations made by Outotec and Delkor.

13.1.4 Concentrate Filtration Testwork

During 2011 Outotec performed a series of filtration tests on magnetite and copper concentrates. The objective of the tests was to determine the main filtration characteristics of the copper and magnetite concentrates.

Outotec recommended horizontal plate pressure filters (PF) be used based on the results obtained with the copper concentrate in the ceramic filter. For the magnetite
concentrate the results of the filtration tests also showed that PF filters would operate better than the ceramic plate (CC) type.

During 2012 additional filtration tests were carried out by Outotec. The objective of the tests was to determine the main filtration parameters for the magnetite concentrate using Larox type pressure filters to obtain a cake with a final moisture content of 8% w/w and a maximum of 300 ppm of chloride.

The test results without cake washing indicated that filtration rates between 950 kg/m²-hr and 807 kg/m²-hr were achieved for slurry feed densities between 65.5% and 67.3% solids w/w. Cake moisture contents of between 8.7% and 10.2% were obtained. The tests with cake washing showed that the rate of filtration was between 692 kg/m²-hr and 730 kg/m²-hr for slurry feed densities between 63.7% and 65.6% solids w/w. This gave a range in cake moisture contents of between 8.5% and 9.3%. Water consumptions of 0.2 m³/t solids, 0.4 m³/t solids and 0.6 m³/t solids resulted in chloride levels in the residual water in the filtered cake of 610 ppm, 184 ppm and 148 ppm, respectively.

The results obtained by Outotec for cake washing indicate a filtration rate of 680 kg/m²-hr, with a cake moisture content of 8.5%. Outotec was prepared to guarantee these results in an industrial application. Based on the testwork outcomes, Outotec recommended the following design parameters:

- Filtration technology: Pressure filters
- Filtration rate: 730 kg/m²-hr
- Concentrate feed: 65% solids w/w
- Cake moisture: 8.5%
- Cake wash rate: 0.5 m³/t solids

13.1.5 Tailings Thickening Testwork

Outotec and Delkor conducted bench scale thickening testwork on tailings samples from the Santo Domingo deposit using a 99 mm diameter thickener. The tailings testwork results were used to evaluate the behaviour of the tailings under conditions of dynamic settling and to determine design values for the two-stage tailings thickening system. Two stages of thickening in series were selected due to the properties of thickened tailings and to avoid capital and operating costs associated with operating using non-thickened tails.

In the first stage of testing, flocculant screening tests were conducted by Outotec, to determine the type of flocculant and the slurry dilution. The resulting recommendation
was the use of anionic flocculant MG-1011 at a slurry dilution of 14% solids w/w. Later in the program dynamic testing was performed for both high-rate thickening and high-capacity thickening.

Using free sedimentation tests, Delkor determined the type of flocculant and the dose. Delkor recommended flocculant AP-2020 at a dilution between 9% and 13% solids w/w. Dynamic tests were also carried out.

Based on the testwork results, the recommended tailings thickening conditions are:

- **First Stage**
  - Type of thickener: High rate
  - Thickening rate: 0.65 t/h/m²
  - Solid percent of thickened tailings: 55%
  - Flocculant dose: 10 g/t tailings.

- **Second Stage**
  - Type of thickener: High density
  - Thickening rate: 0.5 t/h/m²
  - Solid percent of thickened tailings: 67%
  - Flocculant dose: 10 g/t tailings.

### 13.1.6 Flow Properties of Feed and Concentrate Material

During 2012 Jenike and Johanson Chile S.A (JJC) carried out laboratory tests on representative samples of plant ore feed and wet magnetite concentrate to determine the material flow characteristics and fluid properties. JJC performed a series of flow tests on the fine fraction less than ¼” (100% <6.3 mm) of the feed ore sample and on the concentrate samples.

Tests were carried out at two moisture levels to determine how an increase in moisture content affected the flow properties of the materials. The moisture content of the feed was adjusted to 3% and 5%, and the magnetite concentrate moisture was adjusted to 6% and 12%. Timed tests at 24 hours and 72 hours were performed to simulate the maximum time that these materials might experience stored in a silo, chute or stockpile under pressure. For the magnetite concentrate a one-hour timed test was also carried out to simulate the maximum time that this material may experience under pressure in chutes.

The fluidity tests performed on both samples included:

- Cohesive resistance
13.2 Recovery Estimates

13.2.1 Copper

For copper recovery the 2014 Feasibility Study used the Cu feed grade to predict the recovery in the deposit using the results from SGS Santiago metallurgical testwork. The equation derived is:

\[ \text{Global Cu Recovery} = 1.00 \times 0.993 \times (2.844 \times \ln (\text{Cu\%}) + 92.15) \]

The factors included in the model represent the following;

- 1.00 = copper recovery factor from bench scale to full scale
- 0.993 = scaling factor between Cu flotation OCT and LCT tests for bench batch variability and composite samples (average LCT/OCT = 89/89.6 = 0.993)

The copper model is shown in Figure 13-2. Figure 13-3 shows the flotation testwork results compared to the model results (this includes both PFS and 2014 Feasibility Study results). The average variability test copper recovery was 89% versus the predicted average from the algorithm derived of 88.4%.

It should be noted that the scaling factor of 0.993 is due to the residence times being increased from the OCT to the LCT, and therefore the ratio of 0.993 is not a perfect conversion.

13.2.2 Iron

For magnetite recovery, magnetic susceptibility was used to predict the mass recovery to the final magnetite concentrate. The relationship derived is presented below.

\[
\text{Rec. Mas Fe} = \begin{cases} 
0.0011 \times (\text{MagSus}) - 3 \times 10^{-9} \times (\text{MagSus})^2; & \text{if MagSus} \geq 2,000 \\
0; & \text{if MagSus} < 2,000 
\end{cases}
\]

Note: MagSus = magnetic susceptibility; Rec. Mas Fe = mass recovery to final magnetite concentrate
Figure 13-2: Copper Model Recovery

Figure 13-3: Model for Cu Recovery vs Variability Test Number
Figure 13-4 compares the values from the model versus testwork results; it can be seen that the model predicts the results of the magnetite testwork conducted. The values of magnetite recovery from the rougher tailings from DT testwork were corrected to reflect the mass recovery equivalent to industrial scale at 90% of the test results. The model was compared with this information and it was observed that there is a good correlation between the model and the test results.

13.3 Metallurgical Variability

Metallurgical variability testwork is discussed in Section 13.1. Variability samples were selected during the PFS program by Capstone personnel to spatially represent the deposit in terms of grade and major lithologies.

13.4 Deleterious Elements

Strongly oxidized material has the potential to impact on the metallurgical recoveries from the process plant. While the current mine plan does not envisage mining of the oxide cap, there may be sections of the proposed pits where increased clay contents are encountered, such as in the vicinity of faults. Capstone plans to manage this by appropriate grade control measures and in-pit mapping.

Soluble copper content levels are variable within the deposit. This indicates the presence of different copper mineral species within the ores. Test sample results with high soluble copper contents (>10% soluble Cu) were excluded from the algorithm development analysis. Blocks with soluble copper contents above 10% are classified as waste within the mine plan and will not be sent to the plant.

A review of the analyses of the concentrate generated from the 5 Year Average copper composite (refer to Table 13-2) indicated that arsenic values were low, the silica level is acceptable, and heavy minerals such as Bi, Sb and Cd are low. In the QP’s opinion, the levels of deleterious elements in the copper concentrate are such that no penalties are likely to be levied.

As no final market specification has been concluded with an end-user for purchase of the magnetite concentrates produced, Capstone derived a target specification from testwork results of concentrates from the PFS study, with the values indicated in Table 13-2. The majority of concentrate samples produced from the Davis tube tests returned elemental grade values within target specification and indicate that a marketable concentrate within Capstone’s specification can be produced. Additional information on marketability is included in Section 19.
Figure 13-4: Mass Recovery vs Results

Mass Recovery to Magnetite Conc. %

Variability Test

Test Work  Model
### Table 13-2: Comparison of Final Concentrate Properties and Capstone Target Specification

<table>
<thead>
<tr>
<th></th>
<th>5 Yr Average Sample Final LIMS Concentrate</th>
<th>Magnetite Sample Final LIMS Concentrate</th>
<th>Hematite Sample Final LIMS Concentrate</th>
<th>Current Capstone Target</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mass Yield</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>%</td>
<td>23.4</td>
<td>23.4</td>
<td>14.5</td>
<td></td>
</tr>
<tr>
<td>Fe Grade (%)</td>
<td>66.0</td>
<td>66.3</td>
<td>64.8</td>
<td>&gt; 65</td>
</tr>
<tr>
<td>SiO₂ Grade (%)</td>
<td>4.56</td>
<td>3.66</td>
<td>3.92</td>
<td>4.10</td>
</tr>
<tr>
<td>Al₂O₃ Grade (%)</td>
<td>1.00</td>
<td>0.77</td>
<td>0.85</td>
<td>1.00</td>
</tr>
<tr>
<td>TiO₂ Grade (%)</td>
<td>0.17</td>
<td>0.10</td>
<td>0.09</td>
<td></td>
</tr>
<tr>
<td>Mn Grade (%)</td>
<td>0.07</td>
<td>0.08</td>
<td>0.08</td>
<td>0.07</td>
</tr>
<tr>
<td>CaO Grade (%)</td>
<td>0.69</td>
<td>0.80</td>
<td>0.75</td>
<td>0.57</td>
</tr>
<tr>
<td>P Grade (%)</td>
<td>0.01</td>
<td>0.01</td>
<td>0.00</td>
<td>0.01</td>
</tr>
<tr>
<td>S Grade (%)</td>
<td>0.01</td>
<td>0.01</td>
<td>0.05</td>
<td>0.02</td>
</tr>
<tr>
<td>MgO Grade (%)</td>
<td>0.48</td>
<td>0.39</td>
<td>0.55</td>
<td>0.46</td>
</tr>
<tr>
<td>K₂O Grade (%)</td>
<td>0.12</td>
<td>0.10</td>
<td>0.09</td>
<td>0.11</td>
</tr>
<tr>
<td>Na₂O Grade (%)</td>
<td>0.14</td>
<td>0.08</td>
<td>0.08</td>
<td>0.15</td>
</tr>
<tr>
<td>Zn Grade (%)</td>
<td>0.004</td>
<td>0.002</td>
<td>0.004</td>
<td></td>
</tr>
<tr>
<td>LOI (1,000) Grade (%)</td>
<td>-1.1600</td>
<td>-1.43</td>
<td>0.47</td>
<td></td>
</tr>
<tr>
<td>As Grade (%)</td>
<td>0.0010</td>
<td>0.0010</td>
<td>0.0030</td>
<td></td>
</tr>
<tr>
<td>Ba Grade (%)</td>
<td>0.0010</td>
<td>0.0020</td>
<td>0.0050</td>
<td></td>
</tr>
<tr>
<td>Cl Grade (%)</td>
<td>0.0060</td>
<td>0.0140</td>
<td>0.0150</td>
<td>0.0060</td>
</tr>
<tr>
<td>Co Grade (%)</td>
<td>0.0060</td>
<td>0.0020</td>
<td>0.0030</td>
<td></td>
</tr>
<tr>
<td>Cr₂O₃ Grade (%)</td>
<td>0.0900</td>
<td>0.0814</td>
<td>0.1020</td>
<td></td>
</tr>
<tr>
<td>Cu Grade (%)</td>
<td>0.0090</td>
<td>0.0050</td>
<td>0.0120</td>
<td>0.0081</td>
</tr>
<tr>
<td>Ni Grade (%)</td>
<td>0.0340</td>
<td>0.0350</td>
<td>0.0400</td>
<td></td>
</tr>
<tr>
<td>Pb Grade (%)</td>
<td>0.0050</td>
<td>0.0080</td>
<td>0.0050</td>
<td></td>
</tr>
<tr>
<td>Sn Grade (%)</td>
<td>0.0005</td>
<td>0.0060</td>
<td>0.0010</td>
<td></td>
</tr>
<tr>
<td>Sr Grade (%)</td>
<td>0.0010</td>
<td>0.0010</td>
<td>0.0010</td>
<td></td>
</tr>
<tr>
<td>V Grade (%)</td>
<td>0.0080</td>
<td>0.0060</td>
<td>0.0050</td>
<td></td>
</tr>
<tr>
<td>Zr Grade (%)</td>
<td>0.0040</td>
<td>0.0040</td>
<td>0.0005</td>
<td></td>
</tr>
<tr>
<td>FeO Grade (%)</td>
<td>27.1</td>
<td>28.3</td>
<td>24.6</td>
<td>23.1</td>
</tr>
</tbody>
</table>
From the Davis tube testwork results, 12 of the 164 feed samples were lithologically defined as ANDE and returned values that were outside the target range estimated for Capstone for contained silica in magnetite concentrate. ANDE material represents between 13% and 17% of the LOM plant feed. Treatment of the ANDE ore types in high mill feed proportions could potentially lead to the production of concentrates containing low iron and high silica contents, due to the inability to separate magnetite from silica.

Alternatively, in order to meet the target magnetite concentrate market specifications, there is a risk that a portion of the ANDE material formerly designated as plant feed will be designated as waste. In order to quantify this potential impact on concentrate marketability, additional targeted variability testwork is recommended to better understand magnetite concentrate variations in specific ANDE lithology zones (e.g. near barren dyke alterations).

13.5 Comments on Section 13

Metallurgical testwork completed during the PFS and the 2014 Feasibility Study includes physical characterization; conventional sulphide flotation using fresh and sea water; settling and filtration tests on the copper concentrate; magnetic separation of magnetite; and settling and filtration tests on the magnetite concentrate. Settling testwork was also completed on final flotation tailings.

The average for the variability data set gave a copper head grade of 0.38% Cu, with a recovery of 89% and a concentrate grade of 30.5% Cu. It is expected that there will be no penalty elements in the copper concentrates.

The magnetite grades achieved varied between 64% and 68% and reflected the fineness of grind employed.

No final market specification has been concluded with an end-user for purchase of the magnetite concentrates produced. Target specifications were developed by Capstone for use in the 2014 Feasibility Study. The majority of concentrate samples returned elemental grade values within the target specification and indicate that a marketable concentrate within Capstone’s specification can be produced. Additional information on marketability is included in Section 19. To quantify any potential impact on the magnetite concentrate marketability from inclusion of ANDE material in high mill feed proportions, additional targeted variability testwork is recommended to understand magnetite concentrate variations in specific zones of ANDE lithology (e.g. near barren dyke alterations) with respect to iron and silica relationships.
14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction and Background

The cut-off for assay data was 30 June 2012 for the Iris, Iris Norte and Santo Domingo Sur deposits. The assay database contained 35,817 assay intervals. Of these, 30,938 assays had non-zero values for Cu, Au, or Fe. Most sampled intervals were 2 m in length for RC holes and 1 m or 2 m for diamond holes. A total of 16,224 Cu, Au and Fe intervals were located within the interpreted mineralized zones. There are 13,192 samples with non-zero magnetic susceptibility values within the mineralized domains. The database provided to RPA contained collar records for 283 holes. Of these, 101 are core drill holes or holes collared as RC and then finished as core holes. Sixteen holes were drilled as twins. Most of the holes are vertical or near vertical, with 76 holes collared at a dip shallower than -80°. Hole lengths vary widely, but are typically in the range between 200 m and 400 m.

The Estrellita resource estimate was performed in 2007 and is based on 114 RC holes (22,594 m) and 13 core holes (2,366 m). Assays for 18 of the holes located within the interpreted zone boundaries of Estrellita were not available at the time of the estimate and were not included. There were 4,702 Cu assays and 3,595 Au assays with non-zero values.

14.2 Geological Models

14.2.1 Wireframes (Santo Domingo Sur, Iris, Iris Norte)

RPA constructed 3D wireframe or solid models and gridded surfaces of the mineralized zones, fault structures and topography for use in constraining the block grade interpolations. All zones required construction of wireframes for post-mineral dikes that transect the mineralized mantos. There are also some sequences of barren tuffs that were modelled. The wireframe outlines were copied from the 2010 models and modified to honour the 2011–2012 drilling. The principal controls were lithology and structure; however, in some places a nominal grade shell boundary was used. There was no rigorous grade cut-off for this boundary, as it was rarely needed, but as a general rule the cut-off was either a Cu grade of 0.15% or a magnetic susceptibility value of 15,000.

Eight domains were modelled. Three of these (Zones 1, 2 and 3) were further subdivided into magnetite-rich and magnetite-poor variants.
14.2.2 Wireframes (Estrellita)

Three-dimensional wireframe models of the mineralized zones, topography, and mined volumes were constructed for use in constraining the block grade interpolations. The grade shell boundaries were constructed using a lower limit of 0.1% Cu. The applied limits are considered nominal because it was often necessary to include lower-grade intervals for continuity, using lithology as a guide.

A modest amount of underground and open pit mining has been carried out at Estrellita. Far West Mining personnel provided raw cavity monitoring device (CMD) data from which RPA was able to construct approximate wireframe models of the void spaces.

14.2.3 Oxide Model

A wireframe model was also been created to enclose oxidized material which has been demonstrated to yield much lower metallurgical recoveries than the un-oxidized mineralization. This was a very preliminary model owing to the lack of a complete data set for leachable copper. The primary criteria for defining the base of the oxidized zone was presence of significant quantities of leachable copper, or strong oxidation noted in the logs. The oxide model was not used to constrain the grade interpolation. However, it was used to tag material within the resource volume such that this material could be excluded from the Mineral Resource estimate.

14.2.4 Santo Domingo Sur

For the purpose of the resource estimates, the Santo Domingo Sur deposit was modelled as four primary structures (Zones 1 to 4). Several interpreted faults serve to constrain the mineralization on the eastern, western and southern extents as well as divide the deposit into three distinct fault blocks.

Capstone geologists have defined a magnetite rich zone (termed the Mag Zone) which occupies the core of Zones 1, 2 and 3 at Santo Domingo Sur. Surrounding the Mag Zone is relatively more hematite-rich Fe mineralization (Hematite Rim). The magnetic susceptibility values tend to be markedly higher for the Mag Zone than for the Hematite Rim. Consequently, a separate wireframe model for the Mag Zone was constructed and then used to constrain the interpolation values for magnetic susceptibility.
14.2.5 Iris

Iris is subdivided by an internal fault that separates a magnetite-rich zone to the west from the main deposit. The deposit is separated from the Santo Domingo Sur deposit by a north-trending, west-dipping fault and constrained on the eastern boundary by a series of east-dipping faults that separate it from limestone sequences to the east.

14.2.6 Iris Norte

Iris Norte is also bounded on the east by an interpreted east-dipping fault structure with limestone sequences located on the east side. The north-trending, west-dipping fault that divides Iris from Santo Domingo Sur appears to extend along the western flank of Iris Norte, limiting its western extent as well as intercepting the east-dipping faults between the limestone and Iris Norte to effectively cut the deposit off to the north.

14.2.7 Estrellita

The zone has been faulted into a series of four blocks which step downwards to the north. The zone is thickest in the middle and narrows somewhat towards the periphery. There are narrower zones of limited lateral extent in the footwall of the main zone.

14.3 Grade Capping/Outlier Restrictions

Samples were capped prior to compositing.

14.3.1 Santo Domingo Sur, Iris, and Iris Norte

The sample grade distributions for copper and gold are positively skewed, in some cases resembling log-normal distributions. For the 2009 estimate RPA produced a series of log-normal probability curves for copper and gold within the interpreted zones to examine the distribution of the assay data (Lacroix, 2009). The distribution curves for Santo Domingo Sur and Iris exhibited breaks or inflection points at about 3.5% Cu and 0.52 g/t Au, indicating distinct populations for each metal. The data review with the latest drilling indicated the inflection points were still valid.

In total, 24 Cu and 27 Au assay intervals were capped for the 2012 estimate. These intervals represent approximately 0.2% of the total number of assays.

In RPA’s opinion, the net impact of the capping was to reduce the average Cu and Au assay grades by a negligible amount (Table 14-1 and Table 14-2 respectively).
Table 14-1: Assay Capping Levels – Cu

<table>
<thead>
<tr>
<th>Zone</th>
<th>Cap Grade (% Cu)</th>
<th># Std. Dev. from Mean</th>
<th>Population Maximum Grade (% Cu)</th>
<th># Samples Capped</th>
<th>Avg. Cu Grade (% Cu) Before Capping</th>
<th>Avg. Cu Grade (% Cu) After Capping</th>
</tr>
</thead>
<tbody>
<tr>
<td>SDS (1–4)</td>
<td>3.5</td>
<td>5.6</td>
<td>6.38</td>
<td>32</td>
<td>0.486</td>
<td>0.485</td>
</tr>
<tr>
<td>Iris (5–6)</td>
<td>3.5</td>
<td>10.2</td>
<td>3.34</td>
<td>0</td>
<td>0.192</td>
<td>0.192</td>
</tr>
<tr>
<td>Iris Norte (7–8)</td>
<td>3.5</td>
<td>10.5</td>
<td>3.10</td>
<td>0</td>
<td>0.173</td>
<td>0.173</td>
</tr>
<tr>
<td>Totals (1–8)</td>
<td>3.5</td>
<td>6.3</td>
<td>6.38</td>
<td>32</td>
<td>0.380</td>
<td>0.378</td>
</tr>
<tr>
<td>Estrellita</td>
<td>3.0</td>
<td>4.8</td>
<td>8.79</td>
<td>34</td>
<td>0.375</td>
<td>0.366</td>
</tr>
</tbody>
</table>

Note: Includes “below detection” as 0.0

Table 14-2: Assay Capping Levels – Au

<table>
<thead>
<tr>
<th>Zone</th>
<th>Cap Grade (g/t Au)</th>
<th># Std. Dev. from Mean</th>
<th>Population Maximum Grade (g/t Au)</th>
<th># Samples Capped</th>
<th>Avg. Au Grade (g/t Au) Before Capping</th>
<th>Avg. Au Grade (g/t Au) After Capping</th>
</tr>
</thead>
<tbody>
<tr>
<td>SDS (1–4)</td>
<td>0.52</td>
<td>5.9</td>
<td>2.38</td>
<td>23</td>
<td>0.066</td>
<td>0.065</td>
</tr>
<tr>
<td>Iris (5–6)</td>
<td>0.52</td>
<td>5.4</td>
<td>4.71</td>
<td>7</td>
<td>0.029</td>
<td>0.027</td>
</tr>
<tr>
<td>Iris Norte (7–8)</td>
<td>0.52</td>
<td>10.4</td>
<td>0.98</td>
<td>1</td>
<td>0.022</td>
<td>0.022</td>
</tr>
<tr>
<td>Totals (1–8)</td>
<td>0.52</td>
<td>5.8</td>
<td>4.71</td>
<td>31</td>
<td>0.052</td>
<td>0.051</td>
</tr>
<tr>
<td>Estrellita</td>
<td>0.30</td>
<td>4.3</td>
<td>0.979</td>
<td>26</td>
<td>0.049</td>
<td>0.048</td>
</tr>
</tbody>
</table>

Note: Includes “below detection” as 0.0.

14.3.2 Estrellita

The Estrellita data do not display distinct inflections at the high end of the distributions, and so the caps were established at the 99th percentile. These values were at 3% Cu and 0.3 g/t Au. Grade cap data are included in Table 14-1 (Cu) and Table 14-2 (Au).

14.4 Composites

14.4.1 Santo Domingo Sur, Iris, and Iris Norte

Assay intervals have been composited on the basis of hanging wall and foot wall contacts determined by the application of the geological constraints. Samples were composited in down-hole intervals of 4 m starting at the contact for each zone and continuing until the hole exited the zone. Inevitably, the final composite in each zone will be shorter than the fixed composite length unless the zone intercept is an exact multiple of the selected length. These short composites, known as “orphans”, numbered 440 out of a total of 7,783 composites.
The mean grades of orphan composites were compared to those of the full length composites. The orphans averaged 23% to 29% lower in Au, Cu, Fe and magnetic susceptibility. These composites were left in the database and treated as full 4 m composites. In RPA’s opinion, this may impart a slight negative bias to the overall grade interpolations; however, the impact is expected to be negligible.

The 4 m composite length was deemed most suitable, because it was an exact multiple of the most common assay sample interval of 2 m, as well as being an appropriate length for modelling grade in the 12 m high blocks. The former provided relatively discrete composite values that did not straddle the assay intervals; for modelling the number of composites per drill hole could be limited to three or four and still provide sufficient sample coverage for each interpolated block.

Composites for each zone or lithological feature were assigned unique numeric codes to differentiate them from the surrounding material.

14.4.2 Estrellita

The individual mineralized zones are smaller than at SDS/Iris, and so a smaller block size and composite interval was used. Drill samples were composited to 2 m lengths, weighted by both length and density. The modeling software used, GEMS, does not have the capability to manipulate “orphan” composites, so they were left in the database as is.

RPA notes that instances of orphaned composites were relatively rare, owing to the fact that the composite interval was generally equivalent to the sample length, and the wireframe construction was done on sample boundaries. As a result, only a few composites straddled a wireframe boundary.

14.5 Variography

14.5.1 Santo Domingo Sur, Iris, and Iris Norte

RPA carried out a geostatistical analysis to see if revisions to the variogram models developed in 2010 were warranted following the addition of the 2011–2012 definition drill holes. The analysis was conducted using Sage 2001 and GEMS software. There were some significant differences in the updated experimental variogram models from those used in previous grade interpolations; hence the models were revised accordingly (Table 14-3).
### Table 14-3: Variogram Models

<table>
<thead>
<tr>
<th>Metal/Zone</th>
<th>Model Type</th>
<th>Nugget</th>
<th>C</th>
<th>Total Sill</th>
<th>Orientation (Az/Plunge)</th>
<th>Range (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Copper</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Major</td>
<td>Semi</td>
</tr>
<tr>
<td>SDS (1–4)</td>
<td>Exp</td>
<td>0.114</td>
<td>0.866</td>
<td>1.000</td>
<td>118/13</td>
<td>030/-08</td>
</tr>
<tr>
<td>Iris/Iris Mag Sph</td>
<td>0.064</td>
<td>0.936</td>
<td>1.000</td>
<td>168/-02</td>
<td>078/00</td>
<td>168/88</td>
</tr>
<tr>
<td>Iris Norte Sph</td>
<td>0.228</td>
<td>0.772</td>
<td>1.000</td>
<td>079/00</td>
<td>169/00</td>
<td>079/90</td>
</tr>
<tr>
<td><strong>Iron</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Major</td>
<td>Semi</td>
</tr>
<tr>
<td>SDS (1–4)</td>
<td>Exp</td>
<td>0.021</td>
<td>0.979</td>
<td>1.000</td>
<td>130/15</td>
<td>042/-07</td>
</tr>
<tr>
<td>Iris/Iris Mag Sph</td>
<td>0.228</td>
<td>0.772</td>
<td>1.000</td>
<td>191/00</td>
<td>101/00</td>
<td>191/90</td>
</tr>
<tr>
<td>Iris Norte Sph</td>
<td>0.364</td>
<td>0.636</td>
<td>1.000</td>
<td>136/-04</td>
<td>048/34</td>
<td>219/56</td>
</tr>
<tr>
<td><strong>Gold</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Major</td>
<td>Semi</td>
</tr>
<tr>
<td>SDS (1–4)</td>
<td>Sph</td>
<td>0.188</td>
<td>0.812</td>
<td>1.000</td>
<td>073/-08</td>
<td>342/-07</td>
</tr>
<tr>
<td>Iris/Iris Mag Sph</td>
<td>0.031</td>
<td>0.969</td>
<td>1.000</td>
<td>146/-11</td>
<td>056/00</td>
<td>146/79</td>
</tr>
<tr>
<td>Iris Norte Sph</td>
<td>0.400</td>
<td>0.600</td>
<td>1.000</td>
<td>119/06</td>
<td>210/00</td>
<td>299/84</td>
</tr>
<tr>
<td><strong>Mag Sus</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>Major</td>
<td>Semi</td>
</tr>
<tr>
<td>SDS Mag</td>
<td>Exp</td>
<td>0.125</td>
<td>0.875</td>
<td>1.000</td>
<td>010/-10</td>
<td>096/19</td>
</tr>
<tr>
<td>SDS Non-Mag Exp</td>
<td>0.029</td>
<td>0.971</td>
<td>1.000</td>
<td>016/-15</td>
<td>104/07</td>
<td>349/73</td>
</tr>
<tr>
<td>Iris/Iris Mag Sph</td>
<td>0.131</td>
<td>0.869</td>
<td>1.000</td>
<td>190/00</td>
<td>101/34</td>
<td>280/56</td>
</tr>
<tr>
<td>Iris Norte Sph</td>
<td>0.029</td>
<td>0.971</td>
<td>1.000</td>
<td>049/33</td>
<td>139/00</td>
<td>048/-57</td>
</tr>
</tbody>
</table>

RPA notes that for some domains, notably Iris Norte, the lack of data made it difficult to obtain coherent variograms. At times it was necessary to force the variogram model to match the interpreted geology.

#### 14.5.2 Estrellita

Variogram ranges were somewhat shorter at Estrellita than at the other deposits; this was interpreted to be probably due to the different style of mineralization and structural discontinuities in the zones. The variogram ranges for Cu were 106 m x 93 m x 35 m, and this turned out to be too short to completely fill the majority of blocks within the models. Consequently, the search ellipsoid was increased to 150 m x 150 m x 50 m.
14.6 Estimation/Interpolation Methods

14.6.1 Model Dimensions

Santo Domingo Sur, Iris, and Iris Norte

The block size for Santo Domingo Sur, Iris and Iris Norte is 12.5 m east-west, 12.5 m north-south and 12 m high. This is a significant reduction in block size from the previous model in 2010, which used block sizes of 25 m x 25 m x 12 m. Each block that was located at least partially within an interpreted zone was assigned a zone code and, potentially, an interpolated grade. Where a block straddled more than one zone (i.e. across a fault) the block received the code of the zone with the largest portion within the block.

Grades were estimated for only those blocks falling at least partially within one of the eight interpreted domains (envelopes). Integer codes were assigned to the blocks according to the zone with the highest proportion of material contained within the block.

Estrellita

The block size for the Estrellita model was 10 m x 10 m x 5 m. Each block located at least partially within an interpreted zone was assigned a zone code, percent within the zone, and, potentially, an interpolated grade.

14.6.2 Interpolation

Santo Domingo Sur, Iris, and Iris Norte

Grades for Cu, Au, Fe and magnetic susceptibility were interpolated into each block using OK. Block estimates for each zone were constrained to use only composites from that zone. For magnetic susceptibility the interpolation was also configured to discriminate between composites and blocks inside and outside of the Mag Zone. The Mag Zone constraint was not applied for the Au, Cu or Fe estimates.

The interpolation was configured to use an ellipsoidal search with a minimum of three and a maximum of 18 composites and a maximum of three composites allowed from any one drill hole. Grades were estimated in two passes: the first at twice the copper variogram semi-major axis range (300 m x 300 m), and the second using distances equal to the variogram semi-major axis range (150 m x 150 m).
The minor axis searches were 60 m for Pass 1 and 30 m for Pass 2. These search radii were selected to approximately match the anisotropy ratio of the copper variogram model.

Although unique variograms were developed for each component, the search was made consistent for all. This was done to ensure that if a block received an estimate for one component, it was estimated for the other components. The ellipsoids were oriented parallel to the copper variogram models.

**Estrellita**

Separate block models were constructed for each individual zone. Following grade interpolation, the individual block grades were combined into one global block model. Grades for blocks straddling zone boundaries were calculated by averaging the individual block estimates and weighting them by the proportion of the block contained within each zone.

OK was utilized to interpolate Cu and Au grades into each block. Iron was not estimated. Only composites with zone codes that matched the block codes were used in grade estimates. The search was constrained to a minimum of three and maximum of 12 composites, with a maximum of three composites from any one drill hole.

**14.6.3 Specific Gravity**

**Santo Domingo Sur, Iris, and Iris Norte**

The specific gravity values were calculated for each block based on the interpolated Fe grade and the formula developed by RPA:

\[ SG = 2.53 + 0.0276*Fe \]

**Estrellita**

Specific gravity was based on a regression formula developed by RPA as follows:

\[ SG = 2.72 + 0.0018*Fe + 0.0006*Fe^2 \]
14.6.4 Magnetic Susceptibility/Mass Recovery

Santo Domingo Sur, Iris, and Iris Norte

The formula:

\[ MR\% = (1.1063 \times MS) + (-0.003 \times MS^2) \]

Where: \(?MS = \text{Magnetic Susceptibility Reading/1,000}\)

was applied to the interpolated block magnetic susceptibility values to estimate block mass recovery in percent. The percentage of the block that can be recovered (as magnetite) by means of a LIMS process can then be used to estimate the metal value contributed by iron. The mass recovery was assumed to be zero for all blocks grading less than 15\% Fe and the calculated block mass recovery was capped at 95%.

14.7 Block Model Validation

Santo Domingo Sur, Iris, and Iris Norte

RPA validated the grade interpolations using the following methods:

- Visual inspection of the estimated block grades and comparison with the drill composite grades
- Comparison of global composite and block grades
- Swath plots comparing OK and nearest-neighbour (NN) block estimates to composite grades
- Cross-validation (i.e. estimating individual composite grades using the surrounding composites)
- Comparison with the previous model.

No significant errors or biases were noted from the validations performed.

Estrellita

Validation was performed on the grade estimates using utilizing inverse distance weighting to the third power (ID3) interpolation. RPA concluded that while the results for kriging were very close to the actual composite means, the lower standard deviations indicate that kriging was unable to model the extremes as well.
14.8 Classification of Mineral Resources

14.8.1 Santo Domingo Sur, Iris, and Iris Norte

Blocks receiving an estimate for Cu were assigned to at least the Inferred category. All blocks with an average distance to composites of 200 m or less and for which the nearest composite was within 100 m were classified as Indicated.

Definition drilling was carried out during 2011 and 2012 to support potential upgrade of Mineral Resource confidence categories within the area planned for the first three years of production to Measured. Within the area drilled in this drill program, the drill spacing is nominally 50 m. A boundary was drawn around the 50 m drilling pattern and Indicated blocks encompassed by it were nominally assigned to the Measured classification. Blocks below the lowermost extent of the definition holes were excluded from the Measured classification, except in isolated areas where tightly-spaced groups of holes extended to depth.

The upper portions of the deposit are oxidized to some extent, as evidenced by limited assaying for leachable copper and the presence of copper and iron oxides in the core and cuttings. The oxide layer typically extends to a depth of about 80 m from surface, with deeper penetration along faults in localized areas. For the purposes of engineering studies, metallurgical recoveries in the oxide mineralization are expected to be poor and hence this material has not been included as Mineral Resources or Mineral Reserves. RPA and Capstone personnel constructed a wireframe model of the oxidized zone based on the presence or absence of oxide copper mineralization or intense weathering. The final step in the classification was to use the oxide wireframe to tag oxidized blocks and remove these from the Mineral Resources.

14.8.2 Estrellita

The major and semi-major variogram ranges modelled for Cu at Estrellita were 106 m and 93 m, respectively. The drill spacing is approximately 50 m at Estrellita, and well within the two-thirds range limit of the Cu variogram (at approximately 65 m). Consequently, the classification of Indicated was applied to all blocks estimated by at least two drill holes with the closest composite less than 65 m away.

The Inferred portion at Estrellita occupies the fringes of the deposit, and also occurs at depth where a number of holes failed to penetrate the deeper areas.
14.9 Reasonable Prospects of Economic Extraction

14.9.1 Santo Domingo Sur, Iris, and Iris Norte

RPA ran a pit optimization using a Lerchs–Grossmann (LG) algorithm in 2009 and the assumptions listed in Table 14-4.

The optimization was run based on the combined Indicated and Inferred Mineral Resources in the 2009 estimate. At the 0.25% CuEq cut-off, all but 5% of the Mineral Resources were captured by the pit shell. On the basis of this result, it was concluded that there was little merit in restricting the Mineral Resources to those blocks contained only within the pit shell. Accordingly, the entire block model was reported as Mineral Resources. In RPA’s opinion, the shape and depth of the Mineral Resources have not changed since the previous estimate and it is still valid to consider them as having reasonable prospects of economic extraction by open pit mining.

14.9.2 Estrellita

The Estrellita resource estimate is not constrained within a LG shell. RPA notes that this deposit is similar in geometry to the other mineralized bodies in the Project area and is located very close to surface.

14.10 Cut-off Grades

14.10.1 Santo Domingo Sur, Iris, and Iris Norte

The deposits are polymetallic in nature, with elevated copper, iron and gold, and copper being the primary contributor. For this reason a copper equivalent (CuEq) grade was derived which recognizes the potential contributions of all economic components.

The methodology for calculation of CuEq was derived by RPA for the 2009 estimate and remains unchanged for this estimate. CuEq grades were calculated using estimates for recovery, treatment/refinement charges (TC/RC) and transport costs for each metal and based on the operating cost estimates contained in the 2008 Preliminary Assessment (AMEC, 2008).

Parameters used in the calculations are listed in Table 14-5.
Table 14-4: LG Parameters

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Wall Slope</td>
<td>45°</td>
</tr>
<tr>
<td>Mining Cost</td>
<td>US$1.19/t</td>
</tr>
<tr>
<td>Processing Cost</td>
<td>US$4.49/t</td>
</tr>
<tr>
<td>Processing Recovery</td>
<td>85%</td>
</tr>
<tr>
<td>Selling Price</td>
<td>US$2.25/lb</td>
</tr>
<tr>
<td>Selling Cost</td>
<td>US$0.247/lb</td>
</tr>
</tbody>
</table>

Table 14-5: Parameters for CuEq

<table>
<thead>
<tr>
<th>Metal</th>
<th>Price (US$)</th>
<th>Recovery (%)</th>
<th>TC/RC (US$)</th>
<th>Freight (US$4.00/wmt)</th>
<th>Royalty NSR</th>
</tr>
</thead>
<tbody>
<tr>
<td>Copper</td>
<td>3.5/lb Cu</td>
<td>85</td>
<td>0.24/lb Cu</td>
<td>0.007/lb Cu</td>
<td>2%</td>
</tr>
<tr>
<td>Gold</td>
<td>1,500/oz Au</td>
<td>65</td>
<td>6.0/oz Au</td>
<td>-</td>
<td>2%</td>
</tr>
<tr>
<td>Iron</td>
<td>1.94/dmtu Fe</td>
<td>0 to 35</td>
<td>-</td>
<td>6.40/dmt</td>
<td>2%</td>
</tr>
</tbody>
</table>

Notes:
Iron recovery is based on magnetic susceptibility, as explained above.
Freight is based on cost per wmt of concentrate at 8% moisture.
US$1.94/dmtu Fe metal is approximately equivalent to $120/dmt concentrate at 62% Fe.

The metal prices used in Table 14-5 were current at the effective date of the estimate which was August 2012. Typically, the metal prices for RPA’s resource estimations are based on consensus (average) price forecasts by banks and financial institutions. For iron, the pricing basis is for Australian fines of 62% Fe, fob. RPA considers long-term average price forecasts to be appropriate for use in estimating Mineral Reserves, and higher prices (up to +20%) to be appropriate for estimating Mineral Resources.

Since iron ore pricing is quite variable from project to project (due to transport costs, contaminants, etc.), forecast pricing based on a marketing study specific to the project is preferred; however, this was not available at the time the resources were estimated. Once such a source is available, RPA recommends that it is used in future updates to the estimates.

The $1,500/oz gold price and $3.50/lb copper price assumptions were based on consensus inputs and the use of similar prices by major companies at the time the estimate was undertaken.
The formula for the CuEq calculation is provided below (Lacroix, 2009)

\[ Metal\ Value = Grade \times C_m \times R\% / 100 \times (Price - TCRC - Freight) \times (100 - Royalty) / 100 \]

Where: \( C_m \) is a constant to convert grade of metal \( m \) to metal price units; \( R \) is metallurgical recovery in percent; and TCRC is toll treatment and refining charges

and

\[ \%Cu\ Equivalent = (Cu\ Value + Au\ Value + Fe\ Value) / (Cu\ Value\ per\ 1\%Cu) \]

At the request of Capstone, RPA used a cut-off of 0.25% CuEq for reporting the Mineral Resources. Based on the costs developed in the preliminary assessment coupled with the parameters for copper detailed in Table 14-5, the break-even, cut-off grade is calculated as follows:

\[ \text{Cut-off } \%CuEq = (\text{Unit Mineralization Cost} - \text{Unit Waste Cost}) / \text{Value per 1\% Cu} \]

Where

- Mineralization Cost = US$5.68/t
- Waste Cost = US$1.19/t
- Value/1% Cu = 22.05\( \times 85/100 \times (3.50 - 0.24 - 0.007) \times (100 - 2)/100 \)

And therefore:

\[ \text{Cut-off} = (5.68 - 1.19) / 59.74 = 0.075\% \text{ Cu Equivalent} \]

Consequently, in RPA’s opinion, the cut-off grade used is conservative.

14.10.2 Estrellita

RPA’s opinion is that a 0.3% Cu cut-off would be appropriate for the reporting of the estimate. No work has been done on the area since 2007, and the geological interpretation and grade interpolations are unchanged. The style, geometry, and proximity to surface of Estrellita are similar to the Santo Domingo Sur/Iris deposits.

At the time of the estimate in 2007, RPA considered that the 0.3% Cu cut-off was similar to that used in other operations of similar size and grade. Metal prices for both copper and gold have increased significantly since 2007. Using the same cut-off criteria as for Santo Domingo Sur/Iris (i.e. 0.25% copper equivalence with the inclusion of iron content) would be a significantly less rigorous cut-off constraint than that presently in place.

However, for consistency, RPA recommends that the Estrellita resource estimate be updated and have the same constraints applied as for the rest of the Project.
14.11 Mineral Resource Statement

The Mineral Resource estimates and geological models were prepared by David Rennie, P.Eng., an RPA employee. Mr Rennie is the Qualified Person as defined under NI 43-101 for the estimate. Mineral Resources for the Santo Domingo Sur, Iris, and Iris Norte deposits have an effective date of 31 August, 2012. Mineral Resources for the Estrellita deposit were estimated as of 30 October 2007. Mineral Resources in Table 14-6 are reported inclusive of Mineral Reserves. RPA cautions that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

14.12 Factors That May Affect the Mineral Resource Estimate

In RPA’s opinion, there are certain risk factors which could materially impact estimates of Mineral Resources. Such risk factors include:

- Assumptions used to generate the conceptual data for consideration of reasonable prospects of economic extraction including:
  - Commodity price assumptions
  - Exchange rate assumptions
  - Density assumptions
  - Geotechnical and hydrogeological assumptions
  - Operating and capital cost assumptions
  - Metal recovery assumptions
  - Concentrate grade and smelting/refining terms.

- Delays or other issues in reaching agreements with local communities
- Changes in land tenure requirements or in the permitting requirements
- Changes in interpretations of mineralization geometry and continuity of mineralization zones
## Table 14-6: Mineral Resource Estimates

<table>
<thead>
<tr>
<th>Deposit (Zone)</th>
<th>Mt</th>
<th>%CuEq</th>
<th>%Cu</th>
<th>g/t Au</th>
<th>%Fe</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Measured</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Santo Domingo Sur (1–4)</td>
<td>63.3</td>
<td>0.95</td>
<td>0.62</td>
<td>0.083</td>
<td>31.3</td>
</tr>
<tr>
<td>Iris (5–6)</td>
<td>1.54</td>
<td>0.46</td>
<td>0.43</td>
<td>0.052</td>
<td>25.3</td>
</tr>
<tr>
<td><strong>Total Measured</strong></td>
<td>64.8</td>
<td>0.94</td>
<td>0.62</td>
<td>0.082</td>
<td>31.2</td>
</tr>
<tr>
<td><strong>Indicated</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Santo Domingo Sur (1–4)</td>
<td>214</td>
<td>0.72</td>
<td>0.33</td>
<td>0.045</td>
<td>27.4</td>
</tr>
<tr>
<td>Iris (5–6)</td>
<td>111</td>
<td>0.63</td>
<td>0.19</td>
<td>0.028</td>
<td>26.0</td>
</tr>
<tr>
<td>Iris Norte (7–8)</td>
<td>92.3</td>
<td>0.67</td>
<td>0.12</td>
<td>0.015</td>
<td>26.7</td>
</tr>
<tr>
<td><strong>Subtotal Indicated (Santo Domingo Sur /Iris)</strong></td>
<td>417</td>
<td>0.68</td>
<td>0.25</td>
<td>0.033</td>
<td>26.9</td>
</tr>
<tr>
<td>Estrellita</td>
<td>31.7</td>
<td>n/a</td>
<td>0.53</td>
<td>0.050</td>
<td>n/a</td>
</tr>
<tr>
<td><strong>Total Indicated</strong></td>
<td>449</td>
<td>—</td>
<td>0.27</td>
<td>0.034</td>
<td>25.0</td>
</tr>
<tr>
<td><strong>Total Measured and Indicated</strong></td>
<td>514</td>
<td>—</td>
<td>0.31</td>
<td>0.040</td>
<td>25.8</td>
</tr>
<tr>
<td><strong>Inferred</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Santo Domingo Sur (1–4)</td>
<td>29.8</td>
<td>0.55</td>
<td>0.26</td>
<td>0.037</td>
<td>23.6</td>
</tr>
<tr>
<td>Iris (5–6)</td>
<td>5.05</td>
<td>0.60</td>
<td>0.18</td>
<td>0.024</td>
<td>26.7</td>
</tr>
<tr>
<td>Iris Norte (7–8)</td>
<td>20.5</td>
<td>0.70</td>
<td>0.08</td>
<td>0.009</td>
<td>28.0</td>
</tr>
<tr>
<td><strong>Subtotal Inferred (Santo Domingo Sur /Iris)</strong></td>
<td>55.4</td>
<td>0.61</td>
<td>0.19</td>
<td>0.025</td>
<td>25.5</td>
</tr>
<tr>
<td>Estrellita</td>
<td>2.7</td>
<td>n/a</td>
<td>0.48</td>
<td>0.050</td>
<td>n/a</td>
</tr>
<tr>
<td><strong>Total Inferred</strong></td>
<td>58.1</td>
<td>—</td>
<td>0.20</td>
<td>0.026</td>
<td>24.3</td>
</tr>
</tbody>
</table>

### Notes to Accompany Mineral Resource Table:

1. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

2. The Qualified Person for the estimates is Mr. David Rennie, P.Eng., an employee of Roscoe Postle Associates Inc.


4. Mineral Resources for the Santo Domingo Sur, Iris, and Iris Norte deposits are reported using a cut-off grade of 0.25% copper equivalent (CuEq). CuEq grades are calculated using average long term prices of US$3.50/lb Cu, US$1,500/oz Au and US$1.94/dmtu Fe (US$120/dmt conc. at 62% Fe). The CuEq equation is: Metal Value = Grade*CM*R%/100*(Price-TCRC-Freight)*(100-Royalty)/100, where CM is a constant to convert grade of metal m to metal price units; R is metallurgical recovery and %Cu Equivalent = (Cu Value + Au Value + Fe Value)/(Cu Value per 1%Cu).

5. An assessment of Mineral Resources for the Santo Domingo Sur, Iris, and Iris Norte deposits was performed using a Lerchs–Grossman pit shell that has the following assumptions: pit slopes averaging 45°; mining cost of US$1.19/t, processing cost of US$4.49/t; processing recovery of 85%; selling price of US$2.25/lb, and a selling cost of US$0.247/lb. At the 0.25% CuEq cut-off, all but 5% of the Mineral Resources were captured by the pit shell. On the basis of this result, it was concluded that there was little merit in restricting the Mineral Resources to those blocks contained only within the pit shell. Accordingly, the Mineral Resource inventory was reported in its entirety.

6. Mineral Resources for the Estrellita deposit are reported using a cut-off grade of 0.3% Cu.

7. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.

8. Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.
15.0 MINERAL RESERVE ESTIMATES

15.1 Block Model

NCL was provided with the September 2012 updated resource block model that was developed by RPA and which included 2012 drilling campaign results. The September 2012 block model included Mineral Resources that were classified as Measured, Indicated or Inferred. Pit optimization, mine design and mine planning were carried out using this block model and did not include consideration of material classified as Inferred. Inferred Mineral Resources were treated as waste.

15.1.1 SMU Sizing

A block size of 12.5 m E x 12.5 m N x 12 m RL was selected for the block model. The selected block size was based on the geometry of the domain interpretation and the data configuration.

15.2 Throughput Rate and Supporting Assumptions

The mining cost estimate for the pit optimization process is based on studies developed by NCL during 2012. The estimated average project mining cost was separated into various components such as fuel, explosives, tires, parts, salaries and wages, benchmarked against similar current operations in Chile. Each component was updated for first-quarter 2013 prices and the exchange rate from Chilean Pesos to US dollars. This resulted in an estimated mining cost of approximately $1.53/t. The metal prices, processing costs, refining costs, and processing recoveries were provided to NCL by Capstone.

A summary of the initial input parameters used in the constraining LG pit shell is included in Table 15-1; however, it should be noted that these input parameters were modified during the later stages of the study.
Table 15-1: Lerchs–Grossmann Optimization Parameters

<table>
<thead>
<tr>
<th>Item</th>
<th>Units</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Metal Price</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Copper</td>
<td>US$/lb</td>
<td>2.75</td>
</tr>
<tr>
<td>Gold</td>
<td>US$/oz</td>
<td>1,275</td>
</tr>
<tr>
<td>Iron</td>
<td>US$/dmt conc</td>
<td>80</td>
</tr>
<tr>
<td><strong>Recovery to Concentrate</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Copper</td>
<td>%</td>
<td>1*(0.993*(2.8437*LN(Cu%)+92.151))</td>
</tr>
<tr>
<td>Gold</td>
<td>%</td>
<td>(1.13*(0.98*(30.14*LN(Au g/t)+138))/100, Max 75%</td>
</tr>
<tr>
<td>Mass recovery for magnetite concentrate</td>
<td>%</td>
<td>Variable on a block by block basis</td>
</tr>
<tr>
<td><strong>Cu Concentrate Grade</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Copper</td>
<td>%</td>
<td>29%</td>
</tr>
<tr>
<td>Gold</td>
<td>g/t</td>
<td>Calculated</td>
</tr>
<tr>
<td>Moisture content</td>
<td>%</td>
<td>8%</td>
</tr>
<tr>
<td><strong>Magnetite Concentrate Grade</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Iron</td>
<td>%Fe</td>
<td>65%</td>
</tr>
<tr>
<td>Moisture content</td>
<td>%</td>
<td>8%</td>
</tr>
<tr>
<td><strong>Smelter Payables</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Copper in Cu Conc.</td>
<td>%</td>
<td>100%</td>
</tr>
<tr>
<td>Payable Copper</td>
<td>%</td>
<td>96.50%</td>
</tr>
<tr>
<td>Gold in all conc.</td>
<td>%</td>
<td>97%</td>
</tr>
<tr>
<td>Gold deduction in all concentrate</td>
<td>g/t in conc.</td>
<td>0</td>
</tr>
<tr>
<td><strong>Off-Site Costs</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cu conc. treatment</td>
<td>US$/dmt conc.</td>
<td>70</td>
</tr>
<tr>
<td>Cu refining charge</td>
<td>US$/lb pay Cu</td>
<td>0.07</td>
</tr>
<tr>
<td>Au refining charge</td>
<td>US$/oz pay Au</td>
<td>5.0</td>
</tr>
<tr>
<td>Shipping copper concentrate Cu</td>
<td>US$/wmt concentrate</td>
<td>48</td>
</tr>
<tr>
<td>Shipping magnetite concentrate Fe</td>
<td>US$/wmt concentrate</td>
<td>3</td>
</tr>
<tr>
<td><strong>Operating Cost</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Waste mining cost</td>
<td>US$/waste tonne</td>
<td>1.53</td>
</tr>
<tr>
<td>Ore mining cost</td>
<td>US$/ore tonne</td>
<td>1.53</td>
</tr>
<tr>
<td>Processing + G&amp;A</td>
<td>US$/t proc</td>
<td>7.84</td>
</tr>
<tr>
<td><strong>Average Overall Pit Slope Angle</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Overburden</td>
<td>37.6˚</td>
<td></td>
</tr>
<tr>
<td>Sector 1 South</td>
<td>43.6˚</td>
<td></td>
</tr>
<tr>
<td>SDS/Iris &amp; Iris Norte</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Sector 2 West</td>
<td>43.6˚</td>
<td></td>
</tr>
<tr>
<td>Sector 3 North</td>
<td>43.6˚</td>
<td></td>
</tr>
<tr>
<td>Sector 4 East</td>
<td>40.2˚</td>
<td></td>
</tr>
<tr>
<td><strong>Other</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Grade factor (1-Dilution)</td>
<td>%</td>
<td>100</td>
</tr>
</tbody>
</table>
A number of calculations were performed in the model in order to determine the net smelter return (NSR) of each individual block. The internal (or mill) cut-off of $7.84/t milled incorporates all operating costs except mining. This internal cut-off is applied to material contained within an economic pit shell where the decision to mine a given block was determined by the pit optimization, and was applied to all of the Mineral Reserve estimates.

### 15.2.1 Geotechnical Considerations

Final slope angles used for the pit optimization process were a result of multiple iterations and analysis carried out by the NCL mining team and geotechnical specialists Derk Ingeniería y Geología Ltda (Derk) as follows:

- A pit optimization was carried out with an initial set of overall slope angles for selected geotechnical domains
- A pit shell was selected for detail mine design, adding haul roads, safety and geotechnical berms and applying detail bench configuration (batter height, batter angle, berm widths)
- The obtained overall angles per slope domain were measured and compared with the initial assumptions
- The detail pit design was again analyzed by Derk and an updated configuration was generated
- A new optimization was carried out with the updated configuration and the final mine design was developed.

Figure 15-1 shows the geotechnical domains used for the pit optimization for the Santo Domingo and Iris Norte pits. Table 15-2 summarizes the pit slope angles and parameters.
Figure 15-1: Geotechnical Slope Domains

Table 15-2: Slope Domain Data

<table>
<thead>
<tr>
<th>Sector</th>
<th>IRA Face Angle</th>
<th>Inter-Ramp Slope</th>
<th>Overall Slope</th>
<th>Slope Angle</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>β (º)</td>
<td>γ (º)</td>
<td>H (m)</td>
<td>a (m)</td>
</tr>
<tr>
<td>Overburden</td>
<td>44</td>
<td>55</td>
<td>12</td>
<td>8.4</td>
</tr>
<tr>
<td>West</td>
<td>52</td>
<td>75</td>
<td>24</td>
<td>6.4</td>
</tr>
<tr>
<td>East</td>
<td>52</td>
<td>70</td>
<td>24</td>
<td>8.7</td>
</tr>
<tr>
<td>North and South</td>
<td>52</td>
<td>75</td>
<td>24</td>
<td>6.4</td>
</tr>
</tbody>
</table>

15.2.2 Dilution and Mine Losses

The original block model was based on an ore percentage with dimensions of 12.5 m x 12.5 m x 12 m, resulting in a 1,875 m³ block volume; this means that every block has a defined “ore” portion with an ore density, and a corresponding “waste” portion with a waste density.
To accommodate selective mining methods, any resource block with an ore percentage that was <10% was treated as waste. Blocks with an ore percentage that was higher than 90% were diluted with waste such that all high-ore blocks were considered to contain only 90% ore. Selective mining therefore will be performed on those blocks that have an ore percentage of between 10% and 90%. A diluted model was developed to take into account simultaneously the ore loses and dilution, representing 0.1% and 0.3% respectively:

- All blocks with a value lower than 10% in the ore percent item field were considered as pure waste (ore percent = 0); grades were also set to zero in those blocks. A new density was calculated using a weighted average that was based upon the original percentage assignment (SGdil).

- All blocks with a value greater than 90% in the ore percent item field were considered as pure ore (ore percent = 100), and a new density was calculated using a weighted average upon the original percentage assignment (SGdil). Diluted grades were calculated as follows:
  - Diluted Grade = (Original ore percent / 100 * Original SG_ore * Original grade) / SG_dil

- All blocks with values equal to or between 10% and 90% in the ore percent item field were kept as the original percentage assignments; the original specific gravity and grade assignment were also retained.

NCL notes that careful grade control will need to be practiced during mining operations to avoid sending sub-grade material to the plant, because of the important effect of head grade on recovery. These efforts should include the following standard procedures:

- Implement an intense and systematic program of sampling, mapping, laboratory analyses and reporting
- Utilize specialized in-pit, bench sampling drills for sampling well ahead of production drilling and blasting
- Use of shovels to selectively mine ore zones
- Maintain high quality laboratory staff, equipment and procedures to provide accurate and timely assay reporting
- Utilize trained geologists and technicians to work with shovel operators in identifying, marking and selectively mining and dispatching ore and waste.
15.2.3 Cut-off Grades

For mine production schedule purposes a NSR in US$/t was calculated to take into account the value of copper, gold and iron; and the off-site costs (transport, smelting, refining).

The internal (or mill) cut-off of $7.84/t milled incorporates all operating costs except mining. Mining is treated as a sunk cost for the purposes of the cut-off determination. This internal cut-off is applied to material contained within the mining phases, defining the difference between ore and waste.

Marginal ore was calculated for the same $7.84/t cut-off, but for a NSR determined at higher metal prices than shown in Table 15-1, consisting of $3.50/lb Cu, $96.50/t magnetite concentrate and $1,500/oz Au.

15.3 Mineral Reserves Statement

Mineral Reserves are summarized in Table 15-3 and have an effective date of 2 May, 2014. The Qualified Person for the estimate is Mr Carlos Guzman, CMC, an NCL employee.

15.4 Factors That May Affect the Mineral Reserve Estimate

In the opinion of the NCL QP, the main factors that may affect the Mineral Reserves estimate are metallurgical recoveries and operating costs (fuel, energy and labour). NCL notes that the base price, as well as changes in the price of metals, even though this is the most important factor for revenue calculation, does not affect the Mineral Reserves estimate to any significant degree.

A revenue factor of 0.86 was used for the LG shell that was employed as the guide for the practical design for both the Santo Domingo and Iris Norte pits (refer to discussion in Section 16). This selected revenue factor is conservative and as such allows for a broad swing in metals pricing before any salient effect on the Mineral Reserves estimate will occur.
### Table 15-3: Mineral Reserve Statement

<table>
<thead>
<tr>
<th>Reserve Category</th>
<th>Stage</th>
<th>Ore Grade</th>
<th>Contained Metal</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Ore (Mt)</td>
<td>Cu (%)</td>
</tr>
<tr>
<td>Proven Mineral Reserves</td>
<td>Santo Domingo</td>
<td>65.3</td>
<td>0.61</td>
</tr>
<tr>
<td></td>
<td>Iris Norte</td>
<td>—</td>
<td>—</td>
</tr>
<tr>
<td><strong>Total Proven Mineral Reserves</strong></td>
<td></td>
<td>65.3</td>
<td>0.61</td>
</tr>
<tr>
<td>Probable Mineral Reserves</td>
<td>Santo Domingo</td>
<td>251.6</td>
<td>0.27</td>
</tr>
<tr>
<td></td>
<td>Iris Norte</td>
<td>74.8</td>
<td>0.13</td>
</tr>
<tr>
<td><strong>Total Probable Mineral Reserves</strong></td>
<td></td>
<td>326.4</td>
<td>0.24</td>
</tr>
<tr>
<td>Total Mineral Reserves (Proven and Probable)</td>
<td>Santo Domingo</td>
<td>316.9</td>
<td>0.34</td>
</tr>
<tr>
<td></td>
<td>Iris Norte</td>
<td>74.8</td>
<td>0.13</td>
</tr>
<tr>
<td><strong>Total Mineral Reserves (Proven and Probable)</strong></td>
<td></td>
<td>391.7</td>
<td>0.30</td>
</tr>
</tbody>
</table>

**Notes to Accompany Mineral Reserves Table**

1. The Qualified Person for the estimate is Mr Carlos Guzman, CMC, an NCL employee. Mineral Reserves have an effective date of 2 May, 2014.
2. Mineral Reserves are reported as constrained within Measured and Indicated pit designs, and supported by a mine plan featuring variable throughput rates and cut-off optimization. The pit designs and mine plan were optimized using the following economic and technical parameters: metal prices of US$2.75/lb Cu, US$1,275/oz Au and US$80/dmt of Fe concentrate; recovery to concentrate assumptions of a maximum of 93.6% for Cu and 75% for Au, with magnetite concentrate recovery varying on a block-by-block basis; copper concentrate treatment charges of US$70/dmt, US$0.07/lb of Cu refining charges, US$5.0/oz of Au refining charges, US$48/wmt and US$3/wmt for shipping Cu and Fe concentrates respectively; waste mining cost of $1.53/t, mining cost of US$1.53/t ore, and process and G+A costs of US$7.84/t processed; average pit slope angles that range from 37.6º to 43.6º; a 2% royalty rate assumption, and an assumption of 100% mining recovery.
3. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.
4. Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces.
16.0 MINING METHODS

16.1 Pit Designs

Initial pit design considerations are included in Section 15.

Nested pit shells were generated for several revenue factors applied to the base case values. Whittle shell #37 is the revenue factor 1 shell for Santo Domingo. However, after analyzing the results (and reviewing total, as well as incremental values), the Santo Domingo optimized pit shell #30 (revenue factor 0.86) was chosen as the basis for the detailed ultimate pit design. Whittle shell #23 is the revenue factor 1 shell for Iris Norte, and as for the Santo Domingo, pit shell #16 (revenue factor 0.86) was chosen as the basis for the detailed ultimate pit design. The final pit design was based on the economic shells obtained at revenue factor 0.86 for Santo Domingo and Iris Norte, with variable overall slope angles according to geotechnical domains ranging from 38° to 44°. The mine design parameters are summarized in Table 16-1.

A road width of 40 m was selected to accommodate 290 t trucks. NCL used a 10% road gradient which is common in the industry for this type of truck. The current mine plan is designed with 12 m benches stacked to 24 m (i.e. double benching) for the fresh rock material. Mining costs are based on blasting 12 m benches for the waste zones and for the ore.

Additional 40 m wide safety berms were included in the design when the slope height exceeds 100 m at the east wall and 150 m elsewhere, and are in accordance with geotechnical recommendations.

The Santo Domingo pit will have two exits on the west side to provide access to the ROM pad area and primary crusher. On the east side there will be another exit to access the main waste storage area. The final pit will be 2,200 m long in the north–south direction and 1,500 m wide in the east–west direction. The pit bottom will be at the 676 m elevation. The highest wall will be about 552 m, and is situated on the southeast side of the pit. The total area disturbed by the pit will be approximately 229 ha. Figure 16-1 shows the final Santo Domingo pit layout.

The Iris Norte pit will have one exit on the west side to provide access to the ROM pad area and primary crusher. On the east side there will be an exit to access the waste storage area. The final pit will be 1,600 m long in the north–south direction and 900 m wide in the east–west direction. The pit bottom will be at the 664 m elevation. The highest wall will be about 315 m, and is located on the north side of the pit. The total area disturbed by the pit is about 124 ha. Figure 16-2 shows the final Iris Norte pit layout.
<table>
<thead>
<tr>
<th>Parameter</th>
<th>Unit</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Haul road width</td>
<td>m</td>
<td>40</td>
</tr>
<tr>
<td>Haul road grade</td>
<td>%</td>
<td>10</td>
</tr>
<tr>
<td>Bench height</td>
<td>m</td>
<td>12</td>
</tr>
<tr>
<td>Stacked bench height with 2 benches stacked (fresh rock)</td>
<td>#</td>
<td>24</td>
</tr>
<tr>
<td>Nominal minimum mining phase width</td>
<td>m</td>
<td>100</td>
</tr>
<tr>
<td>Batter angle</td>
<td>°</td>
<td>As per geotechnical domains</td>
</tr>
<tr>
<td>Berm width</td>
<td>m</td>
<td></td>
</tr>
<tr>
<td>Security berm width every 100 m/150 m of pit wall</td>
<td>m</td>
<td>40</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Geotechnical Domains</th>
<th>Batter Height (m)</th>
<th>Batter Angle (°)</th>
<th>Berm Width (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Overburden (all pit walls)</td>
<td>12</td>
<td>55</td>
<td>4.0</td>
</tr>
<tr>
<td>West wall</td>
<td>24</td>
<td>75</td>
<td>12.4</td>
</tr>
<tr>
<td>East wall</td>
<td>24</td>
<td>70</td>
<td>10.1</td>
</tr>
<tr>
<td>North and south wall</td>
<td>24</td>
<td>75</td>
<td>12.4</td>
</tr>
</tbody>
</table>
Figure 16-2: Iris Norte Pit Layout Plan

Note: Figure prepared by NCL, 2013. Map north is to top of plan. Grid indicates scale. Grid squares are 1 km x 1 km.
16.2 Pit Phases

Seven pit phases are planned; four for Santo Domingo and three for Iris Norte (refer to Figure 16-3).

In Santo Domingo, Phase 1 (SD01) targets the ore with the highest grade and lowest strip ratio in the central area, down to 892 masl elevation. Phases 2 and 3 (SD02 and SD03) are successive expansions to the north, down to 772 masl and 736 masl elevation, respectively. Phase 4 in Santo Domingo (SD04) is the final expansion to the north, deepening the central portion down to 676 masl. This expansion includes the sector named Iris, which is mined together with Santo Domingo in the northern portion of the pit. This sector has a separate access on the east side and goes down to the 820 masl elevation.

Three mining phases were designed in Iris Norte (IN01, IN02, IN03), which are successive expansions from south to north, going down to 736 masl, 724 masl and 664 masl elevation, respectively. Each phase has accesses from the east and west sides.

16.3 Production Schedule

A mine production schedule was developed to show the ore tonnes, metal grades, waste material and total material by year, throughout the life of the mine (Table 16-2). The distribution of ore and waste contained in each of the mining phases was used to develop the schedule, ensuring that criteria such as continuous ore exposure, mining accessibility, and consistent material movements were met.

NCL used an in-house developed system to evaluate several potential production mine schedules. The required annual ore tonnes and user-specified annual total material movements are provided to the algorithm, which then calculates the mine schedule. Several runs at various proposed total material movement rates were done to determine a good production schedule strategy. This program is not a simulation package, but a tool for calculation of the mine schedule and haulage profiles for a given set of phases and constraints that must be set by the user.

The schedule is based on process plant throughput of 65,000 t/d for the first five years and 60,000 t/d from Year 6 (23.7 Mt/a and 21.9 Mt/a). The mined material movement peaks at 107.5 Mt/a during Years 1 to 4. The production is limited by the number of benches that it is possible to mine in a single phase in a year, or the amount of vertical development per phase.
Figure 16-3: Mining Phases

Note: Figure prepared by NCL, 2013. The Santo Domingo pit is shown at the 1036 masl mining bench, whereas the Iris Norte pit is shown at the 892 masl bench.
<table>
<thead>
<tr>
<th>Period</th>
<th>Year</th>
<th>Ore ('000 t)</th>
<th>Cu (%)</th>
<th>Fe (%)</th>
<th>Wi (kWh/t)</th>
<th>Marginal Ore ('000 t)</th>
<th>Cu (%)</th>
<th>Fe (%)</th>
<th>Wi (kWh/t)</th>
<th>Waste ('000 t)</th>
<th>Total ('000 t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>A0</td>
<td>2016</td>
<td>0</td>
<td>0.000</td>
<td>0.000</td>
<td>-</td>
<td>0</td>
<td>0.000</td>
<td>0.000</td>
<td>-</td>
<td>19,931</td>
<td>20,000</td>
</tr>
<tr>
<td>A01</td>
<td>2017_1-2</td>
<td>474</td>
<td>0.725</td>
<td>29.17</td>
<td>13.0</td>
<td>2</td>
<td>0.147</td>
<td>0.66</td>
<td>23.9</td>
<td>237</td>
<td>25,000</td>
</tr>
<tr>
<td>A02</td>
<td>2017_3-4</td>
<td>2,505</td>
<td>0.684</td>
<td>32.88</td>
<td>12.1</td>
<td>2</td>
<td>0.148</td>
<td>3.80</td>
<td>22.1</td>
<td>3,928</td>
<td>55,000</td>
</tr>
<tr>
<td>A1</td>
<td>2018</td>
<td>22,934</td>
<td>0.677</td>
<td>31.64</td>
<td>12.1</td>
<td>30</td>
<td>0.155</td>
<td>11.37</td>
<td>18.4</td>
<td>2,515</td>
<td>107,500</td>
</tr>
<tr>
<td>A2</td>
<td>2019</td>
<td>23,776</td>
<td>0.602</td>
<td>29.80</td>
<td>12.4</td>
<td>170</td>
<td>0.152</td>
<td>14.60</td>
<td>17.3</td>
<td>1,273</td>
<td>107,500</td>
</tr>
<tr>
<td>A3</td>
<td>2020</td>
<td>23,805</td>
<td>0.488</td>
<td>30.69</td>
<td>12.1</td>
<td>137</td>
<td>0.123</td>
<td>16.61</td>
<td>16.4</td>
<td>345</td>
<td>107,500</td>
</tr>
<tr>
<td>A4</td>
<td>2021</td>
<td>23,816</td>
<td>0.459</td>
<td>30.63</td>
<td>12.1</td>
<td>138</td>
<td>0.120</td>
<td>18.86</td>
<td>15.7</td>
<td>979</td>
<td>107,500</td>
</tr>
<tr>
<td>A5</td>
<td>2022</td>
<td>23,833</td>
<td>0.415</td>
<td>26.94</td>
<td>13.3</td>
<td>599</td>
<td>0.107</td>
<td>17.88</td>
<td>15.9</td>
<td>45</td>
<td>96,200</td>
</tr>
<tr>
<td>A6</td>
<td>2023</td>
<td>22,007</td>
<td>0.373</td>
<td>27.42</td>
<td>13.0</td>
<td>686</td>
<td>0.112</td>
<td>16.37</td>
<td>16.6</td>
<td>478</td>
<td>96,200</td>
</tr>
<tr>
<td>A7</td>
<td>2024</td>
<td>22,011</td>
<td>0.304</td>
<td>28.01</td>
<td>12.6</td>
<td>535</td>
<td>0.104</td>
<td>21.17</td>
<td>14.8</td>
<td>2,576</td>
<td>96,200</td>
</tr>
<tr>
<td>A8</td>
<td>2025</td>
<td>21,875</td>
<td>0.232</td>
<td>27.88</td>
<td>12.6</td>
<td>894</td>
<td>0.082</td>
<td>21.20</td>
<td>14.5</td>
<td>4,372</td>
<td>96,200</td>
</tr>
<tr>
<td>A9</td>
<td>2026</td>
<td>21,954</td>
<td>0.229</td>
<td>26.33</td>
<td>13.1</td>
<td>578</td>
<td>0.061</td>
<td>19.26</td>
<td>15.2</td>
<td>947</td>
<td>96,200</td>
</tr>
<tr>
<td>A10</td>
<td>2027</td>
<td>21,820</td>
<td>0.194</td>
<td>25.71</td>
<td>13.4</td>
<td>224</td>
<td>0.071</td>
<td>17.81</td>
<td>15.7</td>
<td>0</td>
<td>96,200</td>
</tr>
<tr>
<td>A11</td>
<td>2028</td>
<td>22,017</td>
<td>0.185</td>
<td>26.36</td>
<td>13.2</td>
<td>127</td>
<td>0.099</td>
<td>16.66</td>
<td>16.3</td>
<td>1,963</td>
<td>96,200</td>
</tr>
<tr>
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<td>23,379</td>
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Note: *The total of 5.4 M tonnes of marginal ore mined and stockpiled for later re-handle for a total LOM mill throughput of 391.7 Mt.
The total mined waste considers two main destinations for the material as the main waste storage areas and the tailings storage facility for the embankment construction:

- Waste requirements for the tailings storage facility (TSF) construction were provided to NCL by Knight Piésold, and are based on the schedule for the dam embankment raises.
- The material to be sent to the mine waste storage areas corresponds to the difference between the total mined waste from the mine production schedule and the requirement for the TSF.

Three waste rock storage areas at the west and south of the pits were designed for the Santo Domingo Project (refer to Section 18).

The mined ore will be hauled to the primary crusher for direct tipping. Marginal ore will be mined and hauled to a stockpile located between the Santo Domingo and Iris Norte pits until Year 13. This material will be re-handled and will become part of the plant feed in the later years. From Year 14 on, the marginal ore will be sent directly to the plant. The total marginal ore amounts to 5.3 Mt and the maximum size of the marginal stockpile is 4.5 Mt.

The oxide material is treated as waste in the mine plan. No economic process has been defined to treat this material; however, a stockpile area for the oxide material with copper content greater than 0.2% was set aside so this material could be stockpiled for possible future processing.

The work completed by NCL, using the in-house NCL software, assessed the pre-stripping on a quarterly basis; the first 15 months of commercial production on a monthly basis; and quarterly production from the second to fifth years.

The pre-production period requires the mining of 45 Mt of total material to expose sufficient ore to start commercial production in Year 1. The pre-production period will be approximately 15 months. The ore mined during pre-production will be stockpiled in the ROM pad area and will make up part of Year 1 ore production. The total stockpiled ore amounts to 0.5 Mt. Mill throughput will be restricted to the maximum magnetite concentrate production of 4.5 Mt/a up to Year 10; and to 5.4 Mt/a from Year 11 onward.

The production plan showing material sent to mill and to stockpile is included as Table 16-3.
## Table 16-3: Plant Feed Production Schedule

<table>
<thead>
<tr>
<th>Period</th>
<th>Year</th>
<th>Plant Feed</th>
<th>High Grade Stockpile</th>
<th>Marginal Stockpile</th>
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<td>In ('000 t)</td>
<td>Out ('000 t)</td>
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<tr>
<td></td>
<td></td>
<td></td>
<td>Cu (%)</td>
<td>ConCu ('000 t)</td>
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<td>Total</td>
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<td>391,734</td>
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<td>89.1</td>
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</table>
16.4 Blasting and Explosives

The drilling equipment will consist of diesel units capable of drilling 9½” diameter holes for ore and 12¼” diameter holes for waste. Additionally, support units capable of drilling 6½” diameter holes for pre-splitting are included. Two units will be required for the pre-production period. During commercial production from Year 1 through Year 14 six units will be required. Support unit requirements are one during pre-production and two during the life of mine.

A general design for the drilling and blasting patterns has been carried out, using the assistance of Orica to design the patterns. According to the drill pattern specified, a blasting powder factor between 181 g/t and 450 g/t were estimated, as a function of the rock type. Both estimated values are common for fresh rock material.

16.5 Mining Equipment

Mine equipment requirements were calculated based on the annual mine production schedule, the mine work schedule, and equipment annual production capacity estimates. This represents the equipment necessary to perform the following duties:

- Construct roads to the initial mining areas as well as to the crusher, waste storage areas and stockpiles. Construct additional roads as needed to support mining activity.
- The pre-production development required to expose ore for initial production.
- Mine and transport ore to the primary crusher.
- Mine and transport waste from the pit to the waste storage areas.
- Maintain all the mine work areas, in-pit haul roads and external haul roads; and maintain the waste storage areas.
- Re-handle the ore and marginal ore (load, transport and auxiliary equipment) from the stockpiles to feed the primary crusher.

The mine major equipment was selected based on the mine production schedule, 15 months of pre-production and approximately 18 years of commercial mining operations. The pre-production period will include preparing roads, preparing bench openings and pre-production stripping. The total material mined during pre-production is 45 M tonnes. Re-handling of ore will be required in Year 1 for material mined during pre-production to complete the plant feed requirement.

An average dry bank density of 3.16 t/m³ was used for ore and 2.80 t/m³ for waste. The density values are based on the resource block model values for the various
materials tabulated from the mine production schedule. The material handling swell was estimated at 30%. NCL assumed a moisture content of 2%, which represents the weight percent of the wet weight of the material. The density of wet, loose material was used to calculate truck allowable payload limits.

A job efficiency factor (operational losses) of 83.3%, to allow for operational losses, was used to estimate all major units of equipment and productivities; this corresponds to 50 minutes per operating hour. A job efficiency of 85% was used for the haul trucks.

The 2014 Feasibility Study assumes that the mining operation will use 42 m$^3$ hydraulic excavators and trucks with a capacity of 290 t. This type of equipment is able to achieve the required productivity for an annual total material movement of 107.5 t, and will provide sufficient mining selectivity with the excavators as required for good grade control. The fleet will be complemented with drill rigs for ore and waste delineation. Auxiliary equipment will include track dozers, wheel dozers, motor graders and a water truck. The mine fleet will also include the necessary equipment to re-handle the ore from the stockpiles to the primary crusher. This operation will be carried out using a front-end loader and the same 290 t trucks used in the open pit.

The peak equipment requirements for the pre-production and mine life are included as Table 16-4. Fleet requirements by year are included in Table 16-5.

During pre-production one shovel will be required. Four operating shovels will be required for the commercial production period from Year 1 through Year 12, the number will then drop to the end of mine life as less material is mined.

The number of front-end loaders required is less than one for all of the mine life. The front-end loader will also be used as back-up for production loading activities.

The number of truck units required was obtained by dividing the annual capacity of transport of a truck for each combination and period by the corresponding tonnage according to the defined assignment per loading unit. Truck operating hours were calculated per period, type of material and loading unit dividing the tonnage that has to be transported by the hourly productivity of each combination.
Table 16-4: Peak Fleet Requirements for Pre-Production and Commercial Production

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<th>Type of Equipment</th>
<th>Peak Pre-Production</th>
<th>Peak Requirement</th>
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<td>Hydraulic Shovel PC 8000</td>
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<tr>
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<td>29</td>
</tr>
<tr>
<td>Diesel Drill DR 460</td>
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</tr>
<tr>
<td>Support Drill</td>
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</tr>
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</tr>
<tr>
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<td>Wheel dozer 2 WD 900-3A</td>
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<td>Water Truck HD 785-7</td>
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<td>Mobile Crane 200t</td>
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### Table 16-5: Fleet Requirements by Year

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<tr>
<td><strong>Tire Handler WD 600-3</strong></td>
<td>1</td>
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<tr>
<td><strong>Lighting Plant MOTOR LDW 1003 GE</strong></td>
<td>7</td>
<td>8</td>
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<td>13</td>
<td>13</td>
<td>13</td>
<td>10</td>
<td>8</td>
</tr>
</tbody>
</table>
The total haulage distance varies from a minimum of 1.5 km to a maximum of 6.9 km. Truck speeds were determined using typical values obtained from supplier information and similar operations. The truck cycle assignments include fixed times for loading, dumping and queuing. Two and a half minutes have been added to every cycle for dumping and queuing.

Operational indices considered for the trucks were:

- Availability (MA): Variable profile according to vendor and fleet life
- Use of availability (UA): 86%
- Operational losses: 85% (accounting for operator factor, inspection, training).

The number of trucks required during pre-production is six. The requirement gradually increases from 19 units in Year 1 to a maximum of 29 units in Years 9 to 11, then decreases to the end of mine life as less material is mined.

The primary duties that will be assigned to the auxiliary equipment are as follows:

- Mine development including access roads, drop cuts, temporary service ramps, safety berms
- Waste rock storage area control; this includes maintaining access to the dumping areas and maintaining the travel surfaces
- Ore stockpile storage area control; this includes maintaining access to the stockpile areas and maintaining the travel surfaces
- Maintenance and clean-up in the mine and waste storage areas
- Drilling for pre-splitting.

Equipment types included in the auxiliary mine fleet are:

- Komatsu D375A-6R Track Dozer (525 HP)
- Komatsu D475A-5E0 Track Dozer (860 HP)
- Komatsu WD600-3 Wheel Dozer (485 HP)
- Komatsu WD900-3A Wheel Dozer (853 HP)
- Komatsu GD825A-2 Grader (280 HP)
- Komatsu Water Truck HD 785-7 (85 m³)
- Sandvik DR560 Support Drill (6½”).

In general, six track dozers, four wheel dozers, three motor-graders and two water trucks will be required.
16.6 **Mine Rotation Schedule**

The mine is scheduled to work seven days per week, 365 days per year. Each day will consist of two 12 hour shifts. Four mining crews will rotate to cover the operation (two working and two on time off).
17.0 RECOVERY METHODS

17.1 Process Flow Sheet

A simplified schematic of the proposed process flow sheet is included as Figure 17-1, and a more detailed design drawing of the flowsheet is included as Figure 17-2.

17.1.1 Coarse Ore Handling and Crushing

The primary crushing plant will process run-of-mine feed in open circuit. The feed area for the primary crushing will be designed to position two 290 t trucks simultaneously. These trucks will be able to unload material simultaneously to the 450 t capacity feed hopper which has a fixed rock breaker. Primary crushing will be carried out in a 600 kW, 60” x 89” gyratory crusher. The crushed product will fall into a 450 t hopper which will unload onto a plate feeder that feeds the conveyor transporting material to the coarse ore stockpile. The stockpile will have a live capacity equivalent to six to eight hours of operation. The stockpile will discharge onto three feeders located within the reclaim tunnel, which then will feed the primary SAG mill.

17.1.2 Grinding and Classification

The grinding circuit will receive fresh feed from the coarse ore stockpile. The 24 MW (at the shell), 40’ x 26’ EGL SAG mill will operate in DSAG mode. The SAG mill will discharge onto a screen. The oversize pebbles from the discharge screen will be recycled to the SAG mill without being crushed. The screen will be a conventional, double deck, 12’ x 24’, vibratory screen with screen openings of 1” on the top deck and ½” on the bottom deck. The oversize pebbles will be transported by a conveyor system to the SAG mill feed conveyor. The undersize from the discharge screen will be fed to secondary grinding circuit pump box.

The undersize from the discharge screen will be fed to secondary grinding circuit which will consist of two 24’ x 36’ ball mills each with a 12 MW twin pinion drive system. The ball mills will operate in closed circuit with two batteries of 10 x 33” hydrocyclones with 20% spare cyclone flow capacity. The coarse (underflow) fraction from the hydrocyclones will be returned the ball mill feed and the fine (overflow) fraction will be the final comminution circuit product with a P80 of 180 µm.

Each battery of hydrocyclones will be fed by a single centrifugal pump from a common pump box. There will be no spare pumps installed. The pump box will receive the undersize from the SAG mill screen and the discharge from the ball mills. The coarse discharge from each hydrocyclone battery will feed the dedicated ball mill and the fine discharge will be sent to the primary flotation.
Figure 17-1: Proposed Feasibility Study Flowsheet

Note: Figure prepared by AMEC, 2014.
Figure 17-2: Detailed Design Flowsheet

Note: Figure prepared by AMEC, 2014. SIAM = seawater pipeline.
17.1.1 Copper Flotation

Copper rougher flotation will be carried out in a single bank of six 600 m$^3$ conventional, forced air tank cells arranged in a 1-1-1-1-1-1 configuration. Flotation rougher concentrate will be produced from the rougher cells flows by gravity and will be combined with first cleaner scavenger concentrate at the concentrate regrinding stage which will consist of a single vertical mill and hydrocyclone battery operating in closed circuit.

The overflow from the hydrocyclones will feed a single conditioning tank preceding the first cleaner and cleaner scavenger flotation circuit. First cleaner stage flotation will be carried out in a bank of four conventional, forced air 250 m$^3$ tank cells; the concentrate will be pumped to the second cleaner stage and the tailings flow to the scavenger flotation cells. Cleaner scavenger concentrate will be recycled to the regrind cyclone pump box and the tailings will report directly to final plant tailings.

The second cleaner flotation stage will be performed in a bank of three conventional 70 m$^3$ tank cells with the concentrate flowing by gravity to feed the third stage of cleaner flotation. Second cleaner stage tailings produced will be pumped back to the feed of the first cleaning flotation stage.

The third (final) cleaning stage will be performed in a single conventional 70 m$^3$ tank cell.

17.1.2 Copper Thickening

The final copper concentrate will be thickened in an 18 m diameter high rate type thickener. The copper concentrate thickener underflow will discharge at 60% solids w/w and will be pumped to the copper concentrate filter section.

17.1.3 Copper Filtration and Load Out

Copper concentrate will be filtered in a single filter press. Filter area design considerations include an area of 136 m$^2$ and a filter cycle of 12 minutes. Desalinated water will be used for the wash water for the concentrate and the filter cloth, and recycled water from the filtration and concentrate washing will be used for washing the manifold. Concentrate filtration washing with desalinated water will reduce the chloride content to less than 300 ppm in the final filter cake. The recovered water from filtration and concentrate washing will be sent to the manifold wash water tank and then to the clarifier. The clarifier will also receive the recovered water from the filter operation. It is estimated that the underflow from the clarifier will be approximately 25% solids by
weight and this will be returned to the filtration header tank. The clarifier overflow will be pumped back to the plant and will be used as process water.

The filtered concentrate cake at approximately 8.5% moisture content will be discharged to a feeder which feeds a conveyor system. This system will unload into the copper concentrate stockpile. The concentrate will be loaded using front-end loaders (FEL) into trucks to be transported to the port.

17.1.4 Magnetic Separation

The primary flotation tailings from the copper section will be fed to the primary magnetic separation step. The magnetic separation area will include the primary magnetic separation step, regrinding, classification by hydrocyclones, magnetic separation cleaning and magnetite concentrate thickening. Tailings from the primary rougher flotation stage will be pumped to a central distribution box which will feed two parallel lines of five magnetic drums in parallel (1,000 Gauss), 48” diameter x 144” long. It is planned to add one additional magnetic drum to each line after Year 5 due to the scheduled increase in magnetite concentrate production.

Rougher Magnetic Separation and Regrinding

Rougher magnetic concentrate will be sent to regrinding and classification. Rougher magnetic concentration tailings will report to the final plant tailings stream. Hydrocyclone overflow from the magnetite concentrate grinding and classification circuit (P80 of 40 μm) will be sent to cleaner magnetic separation. The primary magnetic concentrate will be reground in a 6.5 MW ball mill in closed circuit with a battery of 12 x 15” hydrocyclones (10 operating, two stand-by). It is planned to add a second regrind circuit after Year 5.

Cleaner Magnetic Separation

The cleaning circuit magnetic LIMS concentrator will consist of two parallel lines each with three LIMS drum separators operating in a counter-current configuration to facilitate high selectivity. The final magnetite concentrate produced will be pumped to the magnetite concentrate thickener and the tailings from the cleaner magnetic stage will be combined with rougher LIMS tailings and sent to the final tailings stream.

Each line will operate counter-currently in series using LIMS drums (750 Gauss, 700 Gauss and 650 Gauss, respectively). Each of the drums will be 48” diameter x 144” long. It is planned to add a third line of three magnetic drums in series after Year 5. Dilution water will be added to the feed to each magnetic drum to agitate the slurry. Wash water will be sprayed onto the magnetic drum to remove the entrained silica
within the magnetite concentrate. Concentrate from each line will flow by gravity to the magnetite regrind circuit. The tailings will report to the general final tailings launder.

17.1.5 Magnetite Thickening

The final magnetite concentrate from each line will be collected in a central launder, feeding the magnetite concentrate thickener via gravity. Magnetite concentrate will be thickened in a 36 m diameter high rate thickener. Flocculant will be added to the thickener feed producing a thickened concentrate of 65% solids w/w with overflow water from the concentrate thickener reporting to the main process water pond. A deflocculator will be added to the discharge from the thickener. Thickened concentrate will be fed to two 200 m³ tanks. A pumping and pipeline system will transfer the magnetite slurry to the receiving tanks at the port.

17.1.6 Lime and Reagent Preparation Plants

The lime and reagent preparation plants (including storage and distribution systems) will be located near the flotation area; the flocculant preparation plants will be located near the tailings thickeners.

The flotation reagent plant will include the primary collector, secondary collector, frother and SMBS systems, each of which will have reception, storage and distribution facilities. The metering pump systems for lime distribution, primary collector, secondary collector, frother, sodium metabisulphite and flocculant will supply the reagents to each of the required points in the process. The storage tanks will be designed for seven days capacity. Reagents will be programmed to be received on a regular basis.

There will be two flocculant plants. One will be situated at the process plant and will provide flocculant for the copper concentrate thickener, first stage tailings thickening and magnetite concentrate thickener. The second plant will be located at the tailings area and supplies flocculant for the final thickening stage.

17.1.7 Grinding Media

There will be several grinding media handling systems to serve the mills, including provision of balls for the SAG mill, ball mill, copper concentrate regrind mill, and the magnetite concentrate regrinding mills.

17.1.8 Tailings Thickening

The tailings from magnetic separation will be combined with copper scavenger flotation tailings. Final plant tailings will be about 20% solids by weight. The first stage of
tailings thickening will be conducted at the process plant and the second stage will be undertaken at the tailings storage facility (TSF) area. First stage thickener tailings will be flocculated and thickened in a 70 m diameter high rate thickener to achieve a discharge of 55% w/w solids. The tailings recirculation pump is designed to recirculate 100% of the tailings if required.

Discharge from the first stage thickener will be transferred, via centrifugal pumps, to a second thickening stage of two parallel 60 m diameter high density thickeners. The flocculant dose will be 10 g/t of tailings feed and the thickened tailings will be 67% solids by weight. To lift the first stage thickener product to the second stage thickeners, seven 600 kW transfer pumps will be installed in series (six operating, one stand-by). The water recovered from the thickeners will be stored in a tank; some water will be filtered to be used as dilution water for flocculant preparation at the TSF. Surplus water will flow by gravity overflow to the process water pond. Final thickened tailings will be pumped to a tank at the TSF.

### 17.1.9 Plant Sea Water Distribution

The sea water received from the port area will be discharged into a distribution box at the plant site. This system will have three separate discharge lines; two of which will discharge by gravity to the fresh sea water tanks that will feed the desalination plant, and the third discharge line will feed the sea water ponds when sea water mixed with port clarified water is received.

Water from the two 2,900 m³ desalination feed tanks will be treated in a reverse osmosis (RO) plant. The desalinated water will feed two systems, one for plant services and the other for consumption at the plant site.

Brine produced by the desalination plant (45% of treated flow) will be transferred to a 120 m³ tank. This will service the site dust suppression requirements; surplus brine will overflow the transfer tank and flow by gravity to the two process water ponds (40,000 m³ each).

During the period that port process discharge water is transferred with sea water, the distribution box will feed two sea water ponds (20,000 m³ each designed for an availability of 24 hours) which will have a pump system to distribute to the appropriate consumption points.

The sea water pumping system will operate at a nominal flow calculated based on the nominal consumption. The system will have a variable flow design capacity enabling operation with fluctuations of ±20% of the nominal feed rates.
17.1.10 Plant Auxiliary Facilities

The air distribution in the plant will provide compressed air for consumption as plant air and as instrument air. The compressed air plant will consist of four 200 kW compressors (three operating and one stand-by), one accumulator, and one dryer with an accumulator for the instrument air. The distribution networks will consist of carbon steel piping and valves, oil filters, moisture traps, cut-off valves, quick connections and controls. These networks will supply grinding, flotation, regrinding, magnetic separation and magnetite concentrate thickening with lines to the stockpile areas, lime plant, reagents plant, copper concentrate thickener and tailings thickeners.

Compressed air for primary crushing and the blower air for flotation will be provided by dedicated equipment. The plant and instrument air for primary crushing will be provided by a 160 kW compressor, one accumulator and a dryer/accumulator for the instrument air. The air for flotation will be provided by three 650 kW blowers (two operating, one stand-by) with a distribution network to each flotation bank.

17.1.11 Port

Copper Concentrate

The copper concentrate will be delivered to the port by trucks which will discharge the material within the copper concentrate storage building adjacent to the concentrate stockpiles. The copper concentrate will be handled inside the stockpile building by FELs to form the stockpiles. The enclosed concentrate storage building will have a negative air pressure system and a dust collection system to minimize environmental impacts from the copper concentrate. The total copper concentrate storage capacity will be approximately 50,000 t in two piles within the building.

The copper concentrate is reclaimed using FELs which transfer the copper concentrate from the stockpiles to the belt feeder which in feeds the copper concentrate onto the shiploader conveyor belt. The conveyor will be fully enclosed to minimize dust emission. The conveyor will have auxiliary equipment such as metal detector, magnet, sampler and belt scale.

Based on the current mining plan for the first five years of operation the following are the expected peak production rate and stockpile capacity requirements:

- Copper concentrate (peak): 494 kt/a
- Stockpile capacity: 50,000 t.
From Year 5 on, the following are the expected peak production rate and stockpile capacity requirements:

- Copper concentrate (peak): 251 kt/a
- Stockpile capacity: 50,000 t.

**Magnetite Concentrate**

Magnetite concentrate will be received at the port in an agitated receiving tank from the magnetite slurry pipeline. The magnetite concentrate will then be pumped directly to the filter plant holding tank. The filter plant will contain four filters each with a filter area of 156 m². Two additional filters will be added as the magnetite concentrate production increases.

The concentrate at 65% solids w/w will be pumped to the filters in a batch sequence with each batch cycle lasting 12.5 minutes. Desalinated water will be used in the washing stage to reduce the chloride content to less than 300 ppm in the filter cake. The recovered water from filtration, concentrate washing and filter cloth washing will be pumped to the clarifier. The underflow from the clarifier will be recirculated to the magnetite concentrate storage tank. There will be a common belt feeder which receives discharge from each pair of filters. Both belt feeders will then discharge onto a conveyor which will transfer the filtered concentrate to the mobile stacker at the 320,000 t magnetite stockpile (stockpile capacity will be increased in the fifth year of operation to 500,000 t). The mobile stacker will run along the north side of the magnetite concentrate stockpile area. The filtered concentrate will have a moisture content of about 8%.

Due to environmental regulations, protection around the perimeter of the magnetite concentrate stockpile area is required to reduce the wind speed at the face of the stockpile which reduces the generation of dust. A steel structure is planned to provide this protection.

Based on the current mining plan for the first five years of operation the following are the expected peak production rate and stockpile capacity requirements:

- Magnetite concentrate (peak): 4.05 Mtpa
- Stockpile capacity: 320,000 t.

From Year 5 on the following are the expected peak production rate and stockpile capacity requirements:

- Magnetite concentrate (peak): 5.40 Mtpa
- Stockpile capacity: 500,000 t.

17.1.12 Water Supply Facilities

The water for all of the facilities (mine and plant site and the port) will be pumped from the ocean, at a point adjacent to the port facilities. The water will first be treated in an electro-chlorination system to eliminate organic build-up in the suction line. A filtration system will then remove suspended materials prior to the water being sent to a holding tank. From the holding tank, water will be pumped to two separate systems, the first system supplying the port water requirements, and the second feeding the pumping system to the mine and plant site.

17.1.13 Port Auxiliary Facilities

Plant Air

The port will require both plant and instrument air. The compressor plant will consist of two 110 kW compressors, one accumulator, and one dryer with an accumulator for instrument air. The distribution networks will consist of carbon steel piping and valves, oil filters, moisture traps, cut-off valves, quick connections and controls. The air for the filter plant operation will include two 450 kW compressors with one accumulator for the blower air, plus a dryer and one accumulator for the instrumentation air. There will be two 75 kW compressors with one accumulator for the filter area. The plant and instrument air will be provided via distribution ring main systems.

Dust Control

Dust suppression systems at transfer points will use specialized nozzles to produce extremely small water droplets in a dispersed mist. These nozzles will operate by atomizing water with compressed air. This type of dust control system will consume water at a rate of about 0.1 L/t to 0.5 L/t of copper concentrate.

The dust suppression system in stockpiles will use large volume water nozzles. For copper concentrate loading and conveyor transfers points, the dust will be collected by dry bag filter systems.

Dust suppression with pressurized air-water systems will be considered for the magnetite concentrate transfer points.
17.2 Plant Design

17.2.1 Design Criteria

The main process design criteria include:

- Nominal capacity (first five years): 65,000 t/d
- Nominal capacity (after five years): 60,000 t/d
- Operating period: 365 days per year

Table 17-1 provides the projected utilization rate for the various plant components. Table 17-2 provides a summary of the planned crushing and grinding designs. The copper concentrate circuit design summary is included in Table 17-3 and the magnetite circuit in Table 17-4.

17.2.2 Mineral Classification

It was determined that the magnetic susceptibility parameter could be used to classify the feed in order to define the different types of plant feed as follows:

- Magnetite = magnetic susceptibility $\geq 8,000$
- Hematite = magnetic susceptibility $2,000 \leq 8,000$

In the feasibility stage, 52 samples representative of the three first years of operation were classified using magnetic susceptibility parameter. These samples were tested using the abbreviated JKSimmet method (SMC). From the results, the treatment rates were set for Magnetite and Hematite materials for the first five years of operation. AMEC notes that because designations of material are based only on the magnetic susceptibility readings, rather than the typical physical characteristics of magnetite and hematite, there may be significant variations in hardness within each of the Hematite or Magnetite classifications. This resulted in the following average treatment capacities for the two feed types:

- Magnetite: 66,629 t/d
- Hematite: 61,844 t/d.

The proportion of the two feed types generated an average five year plant feed blend of 28.7% Hematite and 71.3% Magnetite, which produces a weighted average treatment capacity of 65,422 t/d. These treatment rates were used to define a new version of the Mine Plan (V8.1) with a maximum yearly average production of 65,000 t/d for the first five years of operation and 60,000 t/d after the first five years (equivalent to 23.72 Mt/a and 21.9 Mt/a respectively).
### Table 17-1: Utilization Rates

<table>
<thead>
<tr>
<th>Area</th>
<th>Utilization</th>
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<tbody>
<tr>
<td>Primary Crusher</td>
<td>65%</td>
</tr>
<tr>
<td>Grinding</td>
<td>93%</td>
</tr>
<tr>
<td>Flotation</td>
<td>93%</td>
</tr>
<tr>
<td>Tailings Thickener</td>
<td>93%</td>
</tr>
<tr>
<td>Copper Concentrate Thickener</td>
<td>93%</td>
</tr>
<tr>
<td>Reagent (Lime – Flocculant)</td>
<td>93%</td>
</tr>
<tr>
<td>Magnetic Separation</td>
<td>93%</td>
</tr>
<tr>
<td>Magnetite Concentrate Thickener</td>
<td>93%</td>
</tr>
<tr>
<td>Concentrate Pipeline</td>
<td>98.5%</td>
</tr>
<tr>
<td>Filter and Copper Conc. Handling</td>
<td>90%</td>
</tr>
<tr>
<td>Filter and Magnetite Conc. Handling</td>
<td>90%</td>
</tr>
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</table>

### Table 17-2: Crushing and Grinding

<table>
<thead>
<tr>
<th>Area</th>
<th>Specification</th>
</tr>
</thead>
<tbody>
<tr>
<td>Crushing</td>
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<td>Crushing Work Index:</td>
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</tr>
<tr>
<td>Design</td>
<td>8.4 kWh/t</td>
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<tr>
<td>Open Size Setting (O.S.S.)</td>
<td>180 mm</td>
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<tr>
<td>Grinding</td>
<td></td>
</tr>
<tr>
<td>SAG Mill</td>
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<tr>
<td>Transfer Size (K80) design</td>
<td>2,500 µm</td>
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<td>Specific Energy Consumption DSAG</td>
<td>7.1 kWh/t</td>
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<td>Ball Mill</td>
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<tr>
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<tr>
<td>Product Size (P80)</td>
<td>180 µm</td>
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<tr>
<td>Ball Work Index (BWi)</td>
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<tr>
<td>Average</td>
<td>11.7 kWh/t</td>
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<tr>
<td>Design</td>
<td>12.5 kWh/t</td>
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### Table 17-3: Copper Circuit

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<thead>
<tr>
<th>Area</th>
<th>Flotation Time (mins)</th>
<th>pH</th>
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</thead>
<tbody>
<tr>
<td>Copper Flotation</td>
<td>40</td>
<td>7.5 – 8.2</td>
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<tr>
<td>Conditioning Time</td>
<td>12</td>
<td>8.8 – 9.2</td>
</tr>
<tr>
<td>First Cleaner</td>
<td>25</td>
<td>8.8 – 9.2</td>
</tr>
<tr>
<td>Cleaner Scavenger</td>
<td>55</td>
<td>8.8 – 9.2</td>
</tr>
<tr>
<td>Second Cleaner</td>
<td>18</td>
<td>8.8 – 9.2</td>
</tr>
<tr>
<td>Third Cleaner</td>
<td>10</td>
<td>8.8 – 9.2</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Area</th>
<th>Specification</th>
</tr>
</thead>
<tbody>
<tr>
<td>Copper Regrind Mill</td>
<td></td>
</tr>
<tr>
<td>Specific Energy Consumption</td>
<td>4.5 kWh/t</td>
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<td>Product Size</td>
<td>P80 of 34 µm</td>
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<tr>
<td>Copper Concentrate Thickener</td>
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</tr>
<tr>
<td>Type of Thickener</td>
<td>High Rate</td>
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<tr>
<td>Settling Rate</td>
<td>0.25 t/h/m²</td>
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<tr>
<td>Solid Percentage underflow</td>
<td>60% w/w</td>
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<tr>
<td>Copper Concentrate Filter</td>
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</tr>
<tr>
<td>Capacity per Cycle</td>
<td>13.9 t/cycle/filter</td>
</tr>
<tr>
<td>Unit Rate</td>
<td>495 kg/h/m²</td>
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### Table 17-4: Magnetite Circuit

<table>
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<tr>
<th>Area</th>
<th>Specification</th>
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<tbody>
<tr>
<td><strong>Magnetic Separation</strong></td>
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</tr>
<tr>
<td>Rougher</td>
<td></td>
</tr>
<tr>
<td>Type of Drum</td>
<td>LIMS (Low Intensity Magnetic Separators)</td>
</tr>
<tr>
<td>Intensity of Magnetic Field</td>
<td>1,000 Gauss</td>
</tr>
<tr>
<td>Unit Capacity</td>
<td>80 t/h/m</td>
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<tr>
<td>Regrind Ball Mill</td>
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</tr>
<tr>
<td>Type of circuit</td>
<td>Closed</td>
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<tr>
<td>Product Size</td>
<td>P80 of 40 µm</td>
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<tr>
<td>Ball Work Index (BWi)</td>
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</tr>
<tr>
<td>Average</td>
<td>11.7 kWh/t</td>
</tr>
<tr>
<td>Design</td>
<td>13.6 kWh/t</td>
</tr>
<tr>
<td>CLEANERS</td>
<td></td>
</tr>
<tr>
<td>Type of Drum</td>
<td>LIMS</td>
</tr>
<tr>
<td>Stages of Cleaning</td>
<td>3</td>
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<tr>
<td>Intensity of Magnetic Field</td>
<td>650–750 Gauss</td>
</tr>
<tr>
<td>Unit Capacity</td>
<td>80 t/h/m</td>
</tr>
<tr>
<td><strong>Magnetite Concentrate Thickener</strong></td>
<td></td>
</tr>
<tr>
<td>Type of Thickener</td>
<td>High Rate</td>
</tr>
<tr>
<td>Unit Rate</td>
<td>0.68 t/h/m²</td>
</tr>
<tr>
<td>Solid % underflow</td>
<td>65% w/w</td>
</tr>
<tr>
<td><strong>Magnetite Concentrate Filter</strong></td>
<td></td>
</tr>
<tr>
<td>Type of Filter</td>
<td>Horizontal Plate</td>
</tr>
<tr>
<td>Capacity per Cycle</td>
<td>23.7 t/cycle/filter</td>
</tr>
<tr>
<td>Unit Rate</td>
<td>730 kg/h/m²</td>
</tr>
<tr>
<td><strong>Tailings Thickener</strong></td>
<td></td>
</tr>
<tr>
<td>First Stage</td>
<td></td>
</tr>
<tr>
<td>Type of Thickener</td>
<td>High Rate</td>
</tr>
<tr>
<td>Unit Rate</td>
<td>0.65 t/h/m²</td>
</tr>
<tr>
<td>Solid Percentage underflow</td>
<td>55% w/w</td>
</tr>
<tr>
<td>Second Stage</td>
<td></td>
</tr>
<tr>
<td>Type of Thickener</td>
<td>High Density</td>
</tr>
<tr>
<td>Unit Rate</td>
<td>0.5 t/h/m²</td>
</tr>
<tr>
<td>Solid Percentage underflow</td>
<td>67% w/w</td>
</tr>
</tbody>
</table>

### 17.2.3 Production Plan

The production schedule for copper concentrate and magnetite concentrate is based on the production plan and the recovery models for copper and iron. Table 17-5 presents the production plan obtained from the mine plan and the metallurgical models for Cu and Fe recovery at yearly average treatment rates of 65,000 t/d and 60,000 t/d, with an annual production limit of 494,000 t of copper concentrate and an annual production limit for magnetite concentrate of 4,050,000 t for the first six years of production, and 5,400,000 t for the remaining mine life.
Table 17-5: Production Plan

<table>
<thead>
<tr>
<th>Period</th>
<th>Year</th>
<th>Plant Feed ('000 t)</th>
<th>Cu (%)</th>
<th>Rec. (%)</th>
<th>Cu in Conc ('000 t)</th>
<th>ConCu ('000 t)</th>
<th>Fe (%)</th>
<th>MagSus MassRec (Mt)</th>
<th>ConFe (Mt)</th>
<th>Au (g/t)</th>
<th>Hem (%)</th>
</tr>
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<tbody>
<tr>
<td>A0</td>
<td>2016</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>A01</td>
<td>2017</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
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<td>90.59</td>
<td>54.76</td>
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<td>32.92</td>
<td>14,574</td>
<td>15.13</td>
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<td>31.90</td>
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<td>0.68</td>
<td>90.61</td>
<td>494.07</td>
<td>315.88</td>
<td>31.65</td>
<td>11,279</td>
<td>11.61</td>
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<tr>
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<td>0.60</td>
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<td>445.58</td>
<td>284.88</td>
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<td>13,712</td>
<td>14.01</td>
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<tr>
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<td>0.49</td>
<td>89.76</td>
<td>359.41</td>
<td>229.79</td>
<td>30.70</td>
<td>15,608</td>
<td>16.08</td>
<td>3.82</td>
<td>27.79</td>
</tr>
<tr>
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<td>0.46</td>
<td>89.58</td>
<td>337.02</td>
<td>215.47</td>
<td>30.67</td>
<td>16,832</td>
<td>17.08</td>
<td>4.05</td>
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<tr>
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<td>12.42</td>
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</tr>
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<td>97.39</td>
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<td>16.38</td>
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<td>18.75</td>
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<td>15.78</td>
</tr>
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<td>31.13</td>
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<tr>
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<td>27.72</td>
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<td>10.95</td>
</tr>
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<td>Total</td>
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<td>391,734</td>
<td>0.30</td>
<td>89.12</td>
<td>3,582.88</td>
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<td>28.18</td>
<td>19,137</td>
<td>19.17</td>
<td>7.08</td>
<td>21.09</td>
</tr>
</tbody>
</table>

The distribution of Hematite and Magnetite over the life of mine is indicated in Figure 17-2. In Years 0 and 1 the Hematite will reach the maximum treatment rate within the plan (about 33.6% of the total processed in the year). The maximum rate treatment of Magnetite is close to 90% in Year 18. In some periods the plant could process more tonnage than projected; however, the plan is restricted by the maximum treatment rate and the maximum concentrate production rates, and this potential extra capacity is not used.

Figure 17-3 shows the grades of Cu and Fe in the plant feed. The head grade will vary between 0.42% and Cu 0.68% Cu during the first five years of production. After the fifth year, the head grade is projected to drop to between 0.14% Cu and 0.37% Cu. At the end of the mine life the head grade will be about 0.06% Cu. For the first five years, the head grade will be about 30% Fe, averaging of 27% Fe, with little variation over the LOM. AMEC notes that copper production is economically viable (refer to Section 22), even at the lower grades at the end of the LOM, such that there is no specific copper cut-off grade where the copper circuit closes down.

Figure 17-4 shows the annual tonnes of copper and magnetite concentrate planned to be produced.
17.3 Energy, Water, and Process Materials Requirements

The power requirements for the Project are discussed in Section 18.13. Water provision for the plant is outlined in Sections 17.1.9, 17.1.12, and 18.7–18.8.

Reagents required for the plant operation include lime, primary collector (3418A) and secondary collector (3926), flocculant, frother (MIBC), and sodium metabisulphite. Balls are also required for the grinding circuit, ranging from 5” diameter for the SAG mill to 1–1.5” diameters for the concentrate regrinding mills.

17.4 Comments on Section 17

For the first five years of operation, Santo Domingo will have an annual average production of approximately 248 Mlb of copper contained in 388,000 dmt of concentrate (at an average copper content of 29%). The LOM average is 128 Mlb of copper in approximately 200,000 t of concentrate per year over a period of approximately 18 years. The total life of mine production is estimated to be 2.29 Blb of copper contained in 3.58 Mt of concentrate.

For the same period, the average magnetite concentrate production is estimated to average 3.26 Mdmt per year. The magnetite concentrate production will average 4.19 Mdmt per year with a total estimated production of approximately 75.0 Mdmt for the life of mine. The first five years of production do not include the Year 0 ramp up.
18.0 PROJECT INFRASTRUCTURE

18.1 Introduction

The principal Project facilities are planned to be located at the following sites:

- Santo Domingo plant site: Located at approximately 26°28'00"S and 70°00'30"W
- Operations camp: Located near Diego de Almagro, 7 km from the Santo Domingo site just off Route C-7
- Port facilities: Located about 43.5 km north of Caldera at Punta Roca Blanca
- Sea water pipeline: 111.6 km long from Punta Roca Blanca pump station to the Santo Domingo plant site location
- Concentrate pipeline: 111.6 km long from the Santo Domingo plant site location to the Santo Domingo port site at Punta Roca Blanca
- High voltage transmission line: from Diego de Almagro to the proposed mine and plant site
- High voltage transmission line: from Totoralillo to the port.

Figure 18-1 shows the overall Project layout from the mine site to the proposed port location. Figure 18-2 shows the details of the proposed mine site and plant layout plan.

18.2 Road and Logistics

18.2.1 Access

The planned route for transporting cargo, staff and equipment to the Santo Domingo site is from the south by Route C-17 (Figure 18-3), and from the north by Route C13 (Figure 18-4).
Figure 18-1: Project Location Plan Showing Proposed Pipeline Infrastructure

Note: Figure uses Esri Digital Globe as a base, modified by AMEC, 2013. As an indicator of map scale, it is approximately 117 km from the proposed process plant location to the proposed port site.
Figure 18-2: Proposed Mine Site and Plant Layout Plan

Note: Figure uses Esri Digital Globe as a base, modified by AMEC, 2013. It is approximately 7 km from the planned concentrator site (orange outline located between the Iris Norte pit and the tailings storage facility) to the town of Diego de Almagro as an indicator of scale.
Figure 18-3: Access Routes from the South

Note: Figure uses Google Earth backdrop, modified by AMEC, 2013. As an indicator of map scale, it is approximately 117 km from the proposed process plant location to the proposed port site.
Figure 18-4: Access Routes from the North

Note: Figure uses Google Earth backdrop, modified by AMEC, 2013. As an indicator of map scale, it is approximately 118 km from the proposed process plant location to the proposed port site.

The closest commercial airport to the Santo Domingo site is the El Salvador Airport, 44 km from the site. The next closest commercial airport is the Desierto Atacama Airport, 113 km south from Chañaral, which has regular scheduled flights to Antofagasta and Santiago.

The planned port for transport and shipment of heavy machinery, equipment and materials is Punta Angamos in Mejillones, Antofagasta Region, 520 km from the plant site. This port is in year-round operation and is accessed directly from Route 5 North (Figure 18-5).
18.2.2 On Site Access

Approximately 13 km of roads will be built on the Santo Domingo site in order to connect the plant areas. Roads will be between 6 m and 40 m wide depending on the purpose and will be used for service, operations, and mine truck access to the mine infrastructure. All roads will have a controlled backfill surface.

18.2.3 Copper Concentrate Haulage Study

Ghisolfo was engaged to prepare a study for the overland transport of copper concentrate from the mine to the proposed Santo Domingo port. The approximate distance of the haulage is 118.5 km using the preferred route from the completed haulage study. A comparison was undertaken of operating costs for two contractor-operated haulage operations and two Capstone-operated options. The cost of the 10 hr/day option was consistently higher than the cost of the 20 hr/day option, reflecting the higher fixed costs (primarily additional tractors/trailers) associated with a lower number of trips per truck over the same period of time. The overall finding was
that the owner-operated and contractor-operated haulage estimated costs are relatively close; either operating execution approach can be used.

As part of the transportation study, Capstone requested that a conceptual design for a by-pass around the village of El Salado be completed. The bypass is required to minimize issues with the haulage of loaded copper transport trucks through the centre of El Salado. Ghisolfo recommended a by-pass option that is located further from the village to allow for future village growth.

Ghisolfo also made recommendations for the preparation of contingency plans for spill management and accidents and for training of an incident management team. These recommendations are generally in accordance with standard transport operating requirements and legal requirements.

18.2.4 Pipeline Route Studies

The proposed route for the concentrate and sea water pipelines (collectively the pipeline route) was defined during earlier studies and modified for the 2014 Feasibility Study to allow for a change in the port location. In addition, the pipeline routing was revised to bypass a proposed tailings storage facility at the third-party owned Manto Verde mine.

The pipeline route was optimized using a single right-of-way (RoW) and a common trench for both pipelines. The route is designed to run parallel to the existing roads, and uses existing RoW access to avoid the construction of new roads. The pipeline route was defined to comply with the maximum pipeline grade requirements. The selected 15 m RoW width allows for the safe execution of different construction activities such as pipe trenching, stringing, laying, bending, welding, inspection and testing.

Ongoing access to the pipeline route during operation will be along the platform and construction road.

18.3 Waste Storage Facilities

Three waste rock storage areas, to be located to the west and south of the pits, were designed for the Project. The final configuration is shown in Figure 18-6.
Figure 18-6: Final Pit and Waste Rock Facility Configuration

Note: Figure prepared by NCL, 2013. Waste dump = waste rock facility. Figure north is to top of plan.
The pre-stripping activities will generate approximately 44.2 Mt of waste rock that will be transported by trucks to the WRFs and 4.06 Mt will be used in the TSF starter dam.

The facilities were designed in 50 m lifts. Each lift will be constructed at an approximate angle of repose of 37°. A 75 m set-back between each lift will maintain the overall angle at 22° to facilitate reclamation and long term stability. A constant 2.0 t/m³ loose density was assumed in the design.

The construction sequence of the mine waste storage areas is from bottom to top. The piles were divided into modules, with the horizontal extension of the full areas and the capacity of each section calculated every 50 m lift. The general strategy applied was to reduce long horizontal and uphill hauling distances within the waste storage areas when mining occurs at greater depths in the pit. The destination assignment to the different mine waste areas was assigned based on the minimum cycle time.

Based on a waste characterisation study undertaken internally by Capstone, Capstone has concluded that the WRFs show a moderate to low potential for generation of acid rock drainage. As a result, no significant acid generation is expected by Capstone from the mined waste, and the dry climate conditions are also not expected by Capstone to produce sufficient water to generate drainage through the waste rock facilities to mobilize any acid solutions.

18.4 Stockpiles

During the pre-production period, the ROM pad area will be constructed close to the initial pit for later re-handling to the primary crusher. The total ore to be stockpiled during this period amounts to 0.5 Mt.

The marginal ore stockpile and the oxide stockpile will be located in areas between the Santo Doming and Iris Norte pits. The stockpiles are designed with 20 m lifts and 30 m set-backs in order to facilitate later re-handling.

18.5 Proposed Tailings Storage Facilities

18.5.1 Introduction

The TSF will be located approximately 2 km southeast of the process plant (Figure 18-7), within a basin area having gentle to moderate slopes with an average slope of approximately 6% to the northwest. The area is underlain by the Atacama Gravels and surrounded by ridges of lesser limestone lithologies.
Figure 18-7: Location Plan, Proposed Tailings Storage Facility

Note: Figure courtesy Capstone, 2013. Grid indicates scale. Grid squares are approximately 2.5 x 2.5 km. Map north is to top of plan.
The TSF has been designed for a total tailings storage capacity of approximately 196 Mm³ or 314 Mt at an estimated dry density of 1.6 t/m³. The tailings will be deposited using the sub-aerial method from spigots around the basin perimeter with the active spigots regularly alternated and at a slurry solids concentration of 67%. The overall beach slope is expected to be in the order of 1.5% towards the embankment. Tailings deposition will take place over an 18 year period.

18.5.2 Main Embankment and Saddle Dam

Description

The main embankment will comprise of an initial starter embankment that will be stage raised by the downstream method. Each stage have a crest width of 20 m, a 2.5H:1V slope on the upstream face, and a 2H:1V slope on the downstream face. Unsuitable foundation material will be removed to an average depth of 2 m below the embankment footprint.

The embankment will be constructed with compacted, non-acid generating, mine waste rock. A 1.5 mm thick HDPE geomembrane liner will be installed on the upstream face over a geotextile and a 3 m thick layer of fine-grained bedding material that will act as kind bedding layer for the liner. A total of 7.7 Mm³ of fill will be placed into the embankment in four planned stages to reach a maximum height of 55.5 m. Care will be taken during operations to keep any equipment or other items that could damage the geomembrane in the pond area away from the geomembrane. During detail engineering this requirement will be addressed. A 26.7 m high saddle dam will be constructed at the southwest corner of the TSF prior to Year 5 of operations. This dam will not be in contact with the supernatant pond and therefore will be constructed entirely from compacted, non-acid generating, mine waste rock. No geomembrane liner will be incorporated. The dam will be constructed in a single stage with a crest width of 10 m and slopes at 2H:1V on both the upstream and downstream sides. The design includes for the removal of unsuitable material to an average depth of 1 m below the foundation.

Seismic Setting

Seismicity studies were completed by S y S Ingenieros Consultores Ltda. These studies provided the following design earthquake characteristics and parameters.

a) Maximum credible earthquake (MCE):
   - Intra-plate medium depth earthquake
     - Magnitude Mw = 8.0
Free field (hard ground) acceleration $a_{\text{max}} = 1.02 \, g$
- $KH = 0.22$

b) Operational earthquake
- Intra-plate earthquake of medium depth
  - Magnitude $M_w = 7.5$
  - Free field (hard ground) acceleration $a_{\text{max}} = 0.54 \, g$
  - $KH = 0.16$

**Stability Analyses**

Stability analyses carried out for the main embankment and saddle dam indicate that under static load conditions the safety factors are above 1.5 during operation and after closure. Under pseudo-static conditions, horizontal seismic coefficients of 0.16 and 0.22 were used for the operational and closure stages and the stability analyses gave safety factors higher than the 1.2 minimum safety factor required for Chilean regulations for said conditions. Under post-earthquake static loading conditions, the analyses also gave safety factors above the required minimum of 1.2. These results confirm the dams are predicted to remain stable under the loading conditions modeled.

**18.5.3 Water Management**

**Water Balance**

Knight Piésold prepared a monthly water balance for the TSF that estimates the quantities of water entering, exiting and being stored in the TSF. The inflows included for water in the tailings stream and precipitation while the outflows included for recycle water to the process, evaporation from the supernatant pond and active beach as well as seepage from the tailings and pond. Storage included for water losses to the pores in the tailings and the varying volumes of surface water in the supernatant pond.

Water from the supernatant water pond will be recovered throughout the operating life of the TSF. This is necessary for water use efficiency and to control the size of the supernatant pond in order to limit seepage losses. The general operating principle will be to keep the supernatant pond as small as possible such that under a range of normal operating conditions it will remain over a geomembrane lined area just upstream of the dam. The water balance described above has been used to estimate the limits of this pond to establish the limits of the liner. Follow up work in the next stage of design will involve refining the water balance, particularly for the first year or so of operations, when a lower density slurry is possible and expansion of the lined area may be necessary.
The rate of recycling water from the TSF to the process is predicted to be restricted by the size of the supernatant pond. During the first year, the mean recycle flow to the process is planned to be 131 L/s. Between Years 2 and 5 this is predicted to fall to 63 L/s and then from Year 6 to Year 15 it is predicted to decrease to 50 L/s. After that, during the final years of operation the rate is predicted to be to 40 L/s. The reducing size of the pond is largely due to an increased rate of evaporation from an expanding beach area.

**Probable Maximum Precipitation**

The 24 hour probable maximum precipitation (PMP) was calculated to determine the required storage capacity to manage this event within the TSF. The synthetic unitary hydrograph method (Hershfield, 1961) was used and two design scenarios were analyzed with different frequency factors (K) and catchment areas:

- A small basin immediately upstream of the TSF and K=19.8
- A larger basin including an additional catchment area further upstream and K=11

The first scenario produced the largest volume reporting to the TSF at 1.33 Mm$^3$ and was used to size the storm runoff capacity in the TSF. The stage by stage raise levels of the TSF include for temporary storage of this volume above the tailings and the normal operating pond at all times with an additional 2 m minimum of freeboard. If a major precipitation event occurs, it will be important to evacuate the excess water as quickly as possible. With the exception of Year 1, the additional pumping capacity from the supernatant pond is planned to be capable of evacuating the excess water produced by a 1,000 year storm (0.60 Mm$^3$) in three months, and from the PMP (K=19.8) (1.33 Mm$^3$) in six months. During the first year there will be sufficient temporary storage capacity available to allow for a slower rate of extraction. As indicated, further detail design and water balance calculations should be completed to determine if further expansion of the lined area may be necessary.

**Seepage**

Provided that the supernatant pond is kept above the geomembrane liner as intended by the design, the seepage flows from the supernatant pond are expected to remain low and vary between 0.17 L/s and 0.33 L/s during the operational life of the TSF. They are also predicted to fall below this after closure. Seepage monitoring wells will be installed downstream of the embankment to detect changes that may occur in the quality of underground water. If necessary, the water can be pumped back into the TSF for return to the process and/or to treatment.
The maximum total flow seepage from the TSF beach plus the supernatant pond is predicted to be 7.5 L/s during the first year of operation and then reduce to 6.2 L/s in Year 2 and 5.0 L/s from Year 6 to the end of the operation.

**Surface Run-off Considerations**

A large portion of the catchment area above the TSF will be diverted around the TSF by a 3.7 km long channel. The channel will be above the eastern side of the TSF and the diverted run-off will be discharged downstream of the TSF embankment. The diversion design incorporates:

- Hydraulic capacity to pass the maximum instantaneous flow from the 24-hour storm with a return period of 100 years, which is a flow of 3.50 m³/s
- A minimum longitudinal slope of 0.4%
- A Manning roughness coefficient of 0.02, corresponding to an unlined, earth surface
- A trapezoidal cross section with a base width of 1.0 m, minimum depth of 0.9 m, and side slopes of 2H:1V.

**Spillway**

A spillway will be constructed during the final raise of the TSF. The storage available behind the ultimate embankment will be sufficient to contain the storm associated with a return period of 1,000 years and the spillway has been sized to safely pass flows above this up to the peak discharge from the PMP. The spillway will be located behind the right abutment of the embankment and is planned to be lined with grouted riprap in order to reduce erosion.

**18.5.4 Thickened Tailings Distribution**

The tailings will be pumped from the plant as conventional slurry (55% by mass solids content) to thickeners located at the southern end of the TSF. After thickening to approximately 67% solids content, the tailings will be discharged into a tailings distribution box and then directed in a pipe to points around the perimeter of the TSF for deposition using the sub-aerial method.

The tailings distribution box has been designed to provide flexibility to the system for passing a range of flows and tailings characteristics including slurry solids concentrations. Occasional variations in the quantity and quality of the tailings will require action or checking to ensure that the system operates within appropriate ranges. The box will have five discharge lines as well as drainage and overflow pipes.
It will have interior dimensions of 3.6 m and 3.0 m, and a functional height to the overflow outlet of 2.8 m.

During the first year of operation the thickened tailings will be deposited gravitationally at an average solids content of approximately 58% through two parallel pipes to location P-2 (Figure 18-8).

In the second year of operation the tailings will be thickened to an average solids content of approximately 65% and discharged through two pipes operating simultaneously to points P-1 and P-2. From Year 3 through the end of the project (approximately Year 18), the tailings will be deposited at a concentration of between approximately 65% and 67% via five pipelines depositing simultaneously at points P-1, P-2, P-3, P-4 and P-5. Gravity flows are expected, with the exception of the two last years in which the southwest area of the TSF will require a pumping system.

18.5.5 Water Recovery and Transport System

The water recovery and transport system from the supernatant pond includes pumping and piping to the process plant. The pond will form against the main embankment at the northern end of the TSF. The water recovery system consists of:

- A barge-mounted pump station with two Class #150 vertical pumps (one operating, one stand-by), 100 HP (75 kW) each, located at the supernatant pond. This system must be kept away from the lined surface to avoid damaging the geomembrane.
- Pumping the recovered water from the supernatant pond to a transfer tank, via pipes.
- A transfer tank located above the left abutment of the main embankment.
- Gravity flow from the transfer tank to the process water pond at the processing plant via pipes.

18.5.6 Monitoring

The monitoring system for the TSF includes three existing monitoring wells and a pumping well drilled during the geotechnical and hydrogeological studies. If seepage from the TSF is detected above acceptable levels, up to three additional pumping wells would be installed.
Piezometers will be installed in and under the main embankment and saddle dam and if appropriate in the tailings mass to monitor pore pressures. These will be at strategic locations defined in a follow up stage of design. Instruments possibly including inclinometers, extensometers and survey prisms will be installed in and on the embankments to monitor for deformations under both static loading and earthquake conditions. In addition, seismic accelerometers will be installed to monitor ground movement accelerations during an earthquake. These will be at strategic locations defined in a follow up stage of design.

18.5.7 Closure Considerations

Designs for the closure phase of the TSF include the emergency spillway and treatment of the final tailings beach to reduce the generation of dust and the ingress of surface runoff into the tailings mass. A 0.3 m thick granular cover will be considered to meet this objective.

18.5.8 Risk Evaluation

The design of the TSF provides for a freeboard of at least 2 m above the maximum flood level resulting from the PMP and approximately 4 m above the supernatant pond level under normal operating conditions. Thus the risk of overtopping is expected to be small. The embankment will be constructed from hard, durable, high quality, waste
rock which will have high strength and erosion resistance when placed and compacted in horizontal lifts. Thus the risk of it experiencing a piping or structural failure is considered to be small. Seepage will be controlled by a geomembrane liner on the upstream face of the main embankment and under the supernatant pond. Thus the risk of significant seepage losses being experienced is also considered to be small unless the pond exceeds the limits of the liner or the liner is damaged. Care will be required to reduce the likelihood of either.

18.6 Water Management

18.6.1 Hydrology

A feasibility level analysis of meteorological and hydrological data was carried out by Knight Piésold based on regional stations with historical data and site studies for water resources carried out by third-parties on the Santo Domingo property since 2009. The purpose of this analysis was to characterize the water resources in the area and to support the development of the environmental impact assessment of the project. This study included a general description of the climate, regional and site precipitation data analysis, a review of the meteorological data, and analysis of storm events precipitations and runoff.

Climate data were obtained from one meteorological station at the project site and several regional stations. The meteorological characterization is based on the meteorological datalogger from a monitoring network installed in the study area and owned by Minera Santo Domingo. The meteorological datalogger has been operating since 2010 and consists of five stations.

18.6.2 Water Requirements

The mineral processing will use sea water without desalination. There will be desalination plants at the mine site and at the port to provide water where sea water cannot be used, primarily for washing concentrate and for potable water for consumption in the mine and port areas and to supplement water resources in Diego de Almagro. Brine and recycled water from various sources will be used for dust control. Additional information is provided in Section 18.6.

The water requirement during the construction phase will be provided by Aguas Chañar S.A. (Aguas Chañar), the local water supplier. As part of its supply contract with Aguas Chañar, Capstone will assist with improvements to the system to ensure that the water supply to Diego de Almagro is not impacted.
For water supply requirements during the operations phase of the Project, the following water sources will be used:

- Sea water pumped directly from the ocean (fresh sea water) and used to supply the port site and mine site for process make-up and non-potable water needs such as dust control. The fresh sea water will be provided from a single water intake system located at the port.

- Desalinated water (using fresh sea water) for magnetite concentrate washing at the port; for copper concentrate rinsing at the mine site and as non-potable water for wash-down and general process use at the mine-plant site and the port. The desalinated water will be used where chlorides in water are an issue but potable quality water is not required. Desalinated water will be produced by reverse osmosis; there will be a desalination water plant at both the port and the mine-plant site.

- Potable water will be produced by adding chlorine to a portion of the desalinated water at the port and the mine-plant site. The potable water will be used at the port, the mine-plant site and to supply Diego de Almagro.

The fresh sea water system at the port includes a sea water intake and pump system. The sea water intake system will transfer the water to the port desalination plant and to the port sea water supply tank. The port sea water supply tank will feed the main transfer pump station for the sea water transportation system to the mine site. The fresh sea water will be pumped to the mine site.

Excess, low quality wash water from the magnetite concentrate washing will be retained in a storage pond at the port. Periodically, this wash water will be pumped into the sea water supply line to the mine site. Hence, there will be no discharge of this water; it will be recycled in the process plant. The effluent transfer will be managed to minimize the impact of this water on the fresh sea water quality at the mine–plant site.

**Process Plant**

The nominal requirement for sea water is 334 L/s. The average water content contained in the tailings slurry at 68% solids by weight of tailings will be 304.5 L/s. The maximum water content in the tailings slurry at 68% solids by weight of tailings will be 329.8 L/s. With other water requirements for operations, the estimated water use is a nominal total of 39.4 L/s.
Other Water Requirements

Potable water, water for construction, and dust control (provided by Aguas Chañar S.A.) are variable, with a maximum of 15 L/s.

During operations, additional water supplies will be needed for:

- Dust control on roads: 17.5 L/s
- Potable water in the process plant: 2.5 L/s
- Potable water at the port: 0.2 L/s
- Potable water for Diego de Almagro: 20.0 L/s.

Capstone has committed to provide 10 L/s of potable water to Diego de Almagro. The 20 L/s figure used in the water balance model includes an allowance for possible future water needs.

Sea Water Intake

The sea water supply requirements will be provided via the intake pumping system at the port. The sea water will be distributed to the port reverse osmosis desalination plant and the sea water pipeline pump station. The total sea water required from the intake is generally distributed as shown in Table 18-1.

Sea Water Pipeline

It is estimated that the total flow in the sea water pipeline will be 349 L/s and will consist of the flow rate breakdowns illustrated in Table 18-2.

Process

The mineral processing system at the process plant will use fresh and recycled sea water. The forecast water requirement is 349 L/s and is generally distributed as shown in Table 18-3.
Table 18-1: Sea Water Intake Requirements

<table>
<thead>
<tr>
<th>Sea Water Intake</th>
<th>Flow (L/s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Port reverse osmosis treatment plant</td>
<td>120</td>
</tr>
<tr>
<td>Sea water supply pipeline system</td>
<td>239</td>
</tr>
<tr>
<td>Total flow – sea water intake system</td>
<td>359</td>
</tr>
</tbody>
</table>

Table 18-2: Sea Water Pipeline Flow Rates

<table>
<thead>
<tr>
<th>Sea Water Pipeline</th>
<th>Flow (L/s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sea water intake supply</td>
<td>239</td>
</tr>
<tr>
<td>Port magnetite filter plant discharge</td>
<td>110</td>
</tr>
<tr>
<td>Total flow – sea water pipeline system</td>
<td>349</td>
</tr>
</tbody>
</table>

Table 18-3: Process Water Requirements

<table>
<thead>
<tr>
<th>Process</th>
<th>Flow (L/s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Feed water for the desalination plant</td>
<td>59</td>
</tr>
<tr>
<td>Water for the process plant</td>
<td>290</td>
</tr>
<tr>
<td>Total flow - process</td>
<td>349</td>
</tr>
</tbody>
</table>

Desalinated Water

Port

The desalinated water requirements at the port will be provided by a 54 L/s capacity reverse osmosis plant with distribution as indicated in Table 18-4.

Brine from the reverse osmosis treatment plant will be returned to the sea at an estimated rate of 66 L/s.

Process

The desalinated water requirements at the process area will be provided by a 27 L/s capacity reverse osmosis plant that will supply water for the areas listed in Table 18-5.

The brine from the reverse osmosis plant (17.5 L/s) will be used for dust control on roads and other areas.
Table 18-4: Desalinated Water Requirements at the Port

<table>
<thead>
<tr>
<th>Port reverse osmosis plant</th>
<th>Flow (L/s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Desalinated water to Fe filter plant</td>
<td>53</td>
</tr>
<tr>
<td>Potable water for port facilities</td>
<td>1</td>
</tr>
<tr>
<td>Total desalinated water - port</td>
<td>54</td>
</tr>
</tbody>
</table>

Table 18-5: Desalinated Water Requirements – Process Area

<table>
<thead>
<tr>
<th>Process reverse osmosis plant</th>
<th>Flow (L/s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Water for mill cooling system</td>
<td>2</td>
</tr>
<tr>
<td>Water for copper concentrate filter plant</td>
<td>2</td>
</tr>
<tr>
<td>Potable water for process &amp; infrastructure</td>
<td>3</td>
</tr>
<tr>
<td>Potable water for Diego de Almagro</td>
<td>20</td>
</tr>
<tr>
<td>Total desalinated water - Process</td>
<td>27</td>
</tr>
</tbody>
</table>

18.7 Sea Water Pipeline, Desalination and Water Usage

18.7.1 Introduction

The Project requires sea water for the process plant operation and for potable water (using reverse osmosis technology). The sea water supply is designed to deliver between 375 L/s (nominal) and 408 L/s (design). When designing the pipeline, BRASS used an early version of the water balance model that assumed that there would be no water reclaim from the TSF.

By comparison, AMEC’s Revision Q Flow Diagram, Water Balance reflects a total make-up water requirement projection of between 334 L/s (nominal) and 356 L/s (maximum). The water is primarily supplied from the sea water supply system, and assumes that 45 L/s will be reclaimed from the TSF. The TSF reclaim water volume is from the Knight Piésold TSF water balance. Therefore, the BRASS sea water supply system is a more conservative design.

The fresh sea water will be provided from a single water intake system located at the port and pumped to the port and mine sites for process make up, non-potable water and desalination needs.

Desalinated water will be produced using reverse osmosis, desalination water plants located at both the port and the mine. Desalinated water will be used for magnetite (Fe) concentrate rinsing at the port; for copper concentrate rinsing at the mine site and as non-potable water for wash down and general process use.
Potable water will be produced by adding chlorine to a portion of the desalinized water at the port and the mine. The chlorinated water will be used for potable water at the port, the mine and to provide water to the town of Diego de Almagro.

18.7.2 Sea Water Intake and Supply System

The fresh sea water supply at the port includes a sea water intake and pump system to provide water to the port desalination plant and the main transfer pump station for the mine sea water transportation system.

The sea water will be extracted from the sea by a carbon steel siphon pipe. The sea water intake pipe will discharge directly into a submerged concrete tank. From the tank the water will flow through two concrete channels to the sea water transfer pump suction. The submerged sea water tank will have two pumping systems, one system with three pumps (two operating, one stand-by) that will pump sea water to the port desalination plant. The second system with four pumps (three operating, one stand-by) will provide water to the mine site.

From the buried tank sea water will be pumped to the sea water pump station storage tank; from the sea water pump station water will be pumped through a buried pipeline to the distribution tank at the plant site. From this tank the sea water will be distributed to the fresh sea water tanks or the sea water storage ponds.

At the mine site, the sea water will be discharged into a distribution box that will feed the sea water storage ponds with a capacity of 20,000 m$^3$ each and lined with a high density polyethylene liner and fresh sea water tanks. The fresh sea water tanks will provide water to the desalination plant. From these storage ponds the sea water will be pumped by three pumps (two operating and one stand-by), to the sea water filtered distribution system and to the flocculant plant. From the storage pond the sea water will also flow by gravity through a pipeline to the process water ponds for make-up.

18.7.3 Process Water Storage

Two HDPE-lined open ponds for process water will be located near the process plant and will each have a capacity of 40,000 m$^3$. These ponds will store water reclaimed from the copper and magnetite concentrate thickeners and tailings thickener overflows, and make-up from the plant sea water storage ponds.

Process water will be pumped from the open ponds by five 1,100 kW each pumps (four operating/one stand-by), through a carbon steel pipe that will supply water to the distribution networks.
18.7.4 Desalinated Water

The water from the port desalination plant for washing the magnetite concentrate at the filter plant will be stored and distributed as follows:

- Stored in a 30 m$^3$ carbon steel tank
- Distributed by two 30 KW pumps (one operating/one stand-by) to the filter plant tanks and to the fire protection system.

The water from the process desalination plant at the mine site will be stored in a 50 m$^3$ carbon steel tank from where it will be pumped by two 3.6 kW pumps (one operating/one stand-by) to feed:

- The mill cooling water tank (50 m$^3$)
- The copper concentrate filtration system (200 m$^3$)
- Fire water tank.

The desalinated water tank for the mill cooling systems will have two 1 kW pumps (one operating, one stand-by) which will provide water to the mill cooling system via a carbon steel pipe network. The desalinated water system for the copper concentrate filtration system will also have two pumps (one operating, one stand-by) to supply wash water for filter cloth washing and two pumps (one operating, one stand-by) for filter cake washing.

18.7.5 Port Outfall System

The brine effluent from the port desalination plant will be pumped to an outfall chamber located onshore. From the outfall chamber, the brine will flow by gravity through an outfall pipeline to a vertical discharge chimney. The chimney outlet will be located below sea level and will have a one-port diffuser installed on its lower end.

Results of the numerical modeling of the brine plume showed an estimated influence area of approximately 100 m$^2$. Using the diffuser results, with a 10 m by 10 m dilution zone there is minimal risk of brine recirculation into the sea water intake system. The distance between the sea water intake and the brine diffuser is approximately 100 m; this distance minimizes the potential for cross flow between the intake and diffuser systems. The modeling shows that the proposed outfall system meets established environmental standards.
18.7.6 Potable Water

Construction

During construction, potable water required for the construction camp will be provided by Aguas Chañar S.A.

Operations

Chlorinated desalinated water from the process reverse osmosis (RO) plant will be used as potable water. The potable water will feed into a 2,300 m³ capacity potable water tank for gravity flow to Diego de Almagro. The water will be pumped for infrastructure and process plant use via two 12.5 kW each pumps (one operating, one stand-by) and a carbon steel pipe distribution network.

Chlorinated water at the port from the port RO plant will be stored in a 30 m³ carbon steel tank and distributed by two 0.55 kW pump (one operating/one stand-by) for the various port water consumptions.

18.7.7 Water Treatment

The Project will include the following infrastructure for the treatment of water and waste water:

- Sea water at port: one desalination (reverse osmosis) water plant for sea water treatment with a maximum capacity of 123 L/s to supply water at the port
- Sea water at mine site: one desalination and potable (reverse osmosis) water plant for sea water treatment with a maximum capacity of 60.5 L/s to supply desalinated water for operations; and potable water for infrastructure use and Diego de Almagro
- Water from filtration (at the port): the washing system for the magnetite filters will generate washed water for return to the process plant via mixing with the fresh sea water in the sea water supply pipeline
- At the port: brine will be produced as desalination plant by-product. The brine will be stored in a tank and pumped by two pumps to a submerged, outfall structure
- At the process plant: brine will also be produced as desalination plant by-product. The brine will be stored in a brine tank and used for dust control; any overflow from the brine tank will return to the process water pond
Residual water and solids from heavy vehicle work shop operations will be stored in a pond. As required, the stored effluent will be withdrawn and transported by an authorized company for final disposal at an off-site facility.

The Project will have three sewage treatment plants (STP) at the following locations:

- Construction camp (temporary): 600 m³/d capacity
- Plant site: 100 m³/d capacity
- Port: 20 m³/d capacity

The permanent camp will be connected to the Diego de Almagro sewage system.

Treated water from these plants will meet Chilean irrigation water quality standards. The plants will use the aerobic digestion system. Sludge generated by the operation of the sewage treatment plants will be disposed of in landfill(s). Treated water from waste water treatment at the mine site will be primarily used for dust control.

18.8 Seawater Pipeline

The pipeline system, including pipe and stations, is designed in accordance with ASME Code B31.4-2012, Pipeline Transportation Systems for Liquids and Slurries. The following general process criteria were used for design of the sea water pipeline:

- The water pipeline is designed for a maximum flow of 408 L/s. The average flow condition requires three pumps operating (with one installed stand-by)
- The pipeline nominal diameter is 24 inch, API 5L Gr X65, with variable wall thickness (depending on the operating pressure)
- The design includes corrosion protection to support the anticipated life of about 18 years as follows:
  - Internal: Fusion Bonded Epoxy (FBE)
  - External: Tri-layer of PLE plus Cathode Protection System.
- The system is designed to avoid excess internal negative pressure that may cause the pipeline to collapse
- The pipe will be buried for protection.

The fresh sea water will be supplied by an intake pipe (siphon) which will discharge into a buried tank. The buried tank will have two vertical turbine pumps that will pump water to the port desalination plant and the port sea water supply tank. The sea water and the water recovered from the filter plant will be stored in a 1,319 m³ tank. Sea water (and concentrate wash water effluent) will be pumped by a single pump station.
(located at the port) via the sea water pipeline to the sea water distribution box located at the process plant. This single pump station will include four multi-stage, horizontal pumps (three pumps operating, one stand-by).

The lowest point of the pipeline route is at km 75.35 where a drain station will be installed. An emergency containment pond (with a capacity of 11,750 m³) will be required to collect water drained from the pipe section associated with this low point.

The electrical power supply for the intake and main pump stations will be provided from the electrical system at the port area. Process control (PC) for the sea water transportation system will be managed using a dedicated and independent control system, connected to the main process control system (PCS). Communications for both the sea water and concentrate pipeline systems will be managed by means of a fibre optic cable buried beside the pipelines in the common trench.

### 18.9 Magnetite Concentrate Pipeline

The concentrate transportation pipeline has been designed to be installed with the sea water transportation line. Both pipelines will run parallel and will be buried in a common trench for the majority of the pipeline route. At the port and plant locations, each line will be routed separately to their respective facilities. The majority of the route is the same for both pipelines.

The concentrate transportation system is designed to transport magnetite concentrate slurry from the process plant to the concentrate filter plant at the port (Table 18-6).

The concentrate pipeline system, including pipe and stations, is designed in accordance with the ASME Code B31.4-2012, Pipeline Transportation Systems for Liquids and Slurries. For the concentrate pipeline, longitudinal grades are to be typically less than +/-12% with specific areas up to +/-15%. The concentrate pipe is to be constructed of steel pipe with an HDPE internal liner. Inspection of the concentrate pipeline will be done on a regular basis.

The main pump station will be located in the process plant area. The pump station will have a single, pressurized distribution box, two pre-charged centrifugal pumps (one operating + one standby) and three positive displacement pumps (two operating + one standby) as main equipment.
Table 18-6: Projected Magnetite Concentrate Transport Volumes

<table>
<thead>
<tr>
<th>Condition</th>
<th>Flow (m³/h)</th>
<th>Cp</th>
<th>Capacity</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
<td>Dry Tonnage per Year (dt/a)</td>
</tr>
<tr>
<td>Maximum Tonnage</td>
<td>415</td>
<td>68%</td>
<td>5,400,080</td>
</tr>
<tr>
<td>Average Tonnage</td>
<td>335</td>
<td>66%</td>
<td>4,305,263</td>
</tr>
<tr>
<td>Minimum Tonnage</td>
<td>289</td>
<td>68%</td>
<td>3,756,342</td>
</tr>
</tbody>
</table>

The system includes a single, intermediate choke station located at km 65 at an elevation of 448 masl. A terminal choke station will be located at the concentrate pipeline termination at the port. The terminal choke station will dissipate the high pressure from the concentrate pipe and will allow a smooth discharge of the slurry into the port slurry storage tank.

At the highest point along the pipeline profile, a monitoring station is to be installed to manage the internal pipe pressure and to maintain optimal flow conditions in the concentrate pipeline (e.g. avoid slack flow). A drain station at the lowest point along the pipeline profile (at km 36.10 and 418 masl) will be installed to allow the drainage of the concentrate pipeline.

The drain, choke, and terminal stations will each have emergency ponds.

The electrical power supply for the concentrate pump station will be provided from the electrical system at the plant site. The electrical power supply for the terminal station at the port will be provided by the port electrical system. The electrical power supply for the choke and monitoring stations will be provided by photo-voltaic systems installed at each station. Process control for the concentrate transportation system will be by a dedicated, independent control system, connected to the main PCS. This dedicated control system will use a PLC network or PCs. Communication for the seawater transportation and concentrate systems will be done by a fibre optic cable buried alongside the pipeline in the common trench.

18.10 Building Infrastructure

18.10.1 Mine and Plant Site

The buildings required at the mine and plant site include:

- Administration: The administration area will consist of four separate buildings and one parking area that will provide parking for 30 light vehicles:
Operations: The operations area will consist of three separate buildings and two parking areas that will provide parking for 50 light vehicles and eight buses:
- Process plant control room and dispatch office
- Change house and training building
- Dining room for process plant and mine

Contractors: The contractors’ area is designed to provide space for the installation of:
- Lubricants shop
- Scheduled maintenance workshop
- Workshop 1 (electric, instrumentation, structures and minor project)
- Workshop 2 (belts and rubbers)
- Change house

First aid and emergency: This building is designed to provide work space for emergency and first aid staff, including areas for offices, training room, examination rooms, a bathroom and a roofed area for the ambulance.

Maintenance: This building is designed to provide a work space for mechanical maintenance, welding, warehouse and offices for the plant maintenance personnel.

Assay laboratory: This facility will provide laboratory services for the process plant and mine. The building will include an enclosed area for offices, sampling preparation, wet laboratory and service facilities, and an open sided, roofed area for sample storage.

Primary crusher: This building is designed to provide a work space, local control room, bathrooms and services for the management and supervision of the primary crusher area operations.

Mine truck operators: This building is designed to provide lunchroom space to serve the mine staff, and will be located on the mine access road.

Maintenance

The plant mobile equipment will be maintained at the mine heavy vehicle work shop by contractors. The cost of this maintenance is allocated to the plant general services.
Capstone will provide building and maintenance areas to be used by the service contractors for plant and mobile equipment maintenance. The mine equipment maintenance will be done under a MARC contract for the first five years; after that Capstone will carry out its own maintenance. The workshop area is designed to provide maintenance areas for:

- Mine trucks and equipment
- Light vehicles
- Mine truck wash bay
- Mine truck tire shop
- Welding shop
- Spare parts storage area
- Offices
- Maintenance dining room
- Change house.

The heavy vehicle work shop building will have service bays for mobile mine equipment, light trucks, mine trucks, and tracked vehicles for maintenance and routine servicing. This area includes the following buildings:

- Heavy vehicle work shop: 15,510 m²
- Mine maintenance office: 120 m²
- Change house: 39 m²
- Lunchroom: 59 m²
- Warehouse: 440 m²
- Light vehicle maintenance shop: 97 m²

Vehicles required by maintenance personnel and for process plant operations will be rented. The parking area is designed for eight mine trucks.

**Warehouse and Storage**

The warehouse area will consist of two buildings and four open areas for storage. The four open areas include:

- General yard storage area: 9,509 m²
- Lubricant storage: 400 m²
18.10.2 Port Site

The buildings required at the port site include:

- Port office building: This building is designed to provide work space for external services at the port such as police, customs, SAG and marine services
- Port operations offices, control room and laboratory building: This building is designed to provide work space for offices for the staff at the port area, the control room and the laboratory
- Port change house: This building consists of a change house to serve the staff at the port area
- Port lunchroom: This building is designed to provide space for a dining room at the port area
- Port workshop and warehouse: This building is designed to provide work space for mechanical, electrical and instrumentation maintenance, warehouse and offices for the maintenance personnel at the port
- Port access control: This building is designed to provide access control at the entrance to the port area
- Magnetite filter building: This building is designed to provide an enclosed area for magnetite concentrate filtration
- Copper concentrate storage shed.

18.10.3 Camps and Accommodation

Accommodation for construction and operations personnel will be in two camps as follows:

- Temporary construction camp: this camp will be used to house and support construction and operations personnel during the construction and commissioning period of the project. The planned location of the temporary camp is 2.5 km from the mine and process area. The construction camp will have capacity for up to 3,100 beds (including 307 beds for operations staff)
- Permanent operations camp: this camp will be used to house and support operations personnel after the completion of the Project. The proposed permanent
camp will be located 6.8 km from the plant near Diego de Almagro and will accommodate approximately 500 people.

There is no plan to continue with a construction camp once operations start. For ongoing construction and maintenance activities, it is planned to accommodate personnel in off-site accommodations in Diego de Almagro or other nearby locations. As capacity permits in the permanent operations camp, temporary personnel may also be housed there.

18.11 Ancillary Infrastructure

18.11.1 Fire Protection

The fire detection system will protect process facilities, buildings and electrical and control rooms. Fire water will be desalinated water and will be stored in a fire water tank. The estimated fire water volume required is 500 m³; this will be the tank minimum reserve volume.

At the port, the fire system will use desalinated water that is stored in the fire water tank. This tank is sized for 500 m³ to feed the fire water network via an electrical pump with a diesel back-up pump.

18.11.2 Compressed Air Systems

The compressed air supply has been designed as a central compressor station to provide plant air and instrument air distribution. The distribution network will supply plant air and instrument air for:

- Grinding
- Flotation
- Copper regrind and thickening
- Magnetite grinding, hydrocyclones and thickening
- Copper and iron concentrate tanks
- Tailings thickener
- Lime plant and reagents.

18.11.3 Dust Control

The following principles have been applied to control dust emissions:

- Wetting of active access roads and roads in operations areas
- Vehicle speeds will be limited on all internal and access roads
- Conveyors belts will be covered or dust control systems installed in tunnels
- Dust mitigation at conveyor chutes with dust collection installed in transfer chutes
- A wet fog system will be installed at the primary crusher.

### 18.11.4 Solid Waste Management

The Project includes the construction of facilities for management of solid waste (recycling, storage and disposal) generated during the construction, operation and project closure stages.

At the mine and plant site and at the port, facilities for domestic solid waste, recyclable waste management, construction waste (only during the construction stage), non-hazardous industrial solid waste and industrial hazardous waste have been included.

The facilities included in the mine and plant site design are:

- Landfill
- Recycle yard (non-hazardous industrial solid wastes)
- Storage yards for hazardous industrial wastes
- Storage yards for non-hazardous industrial wastes.

For the port, the design facilities are:

- Hazardous waste storage yard
- Non-hazardous waste storage yard.

Domestic waste and similar that is generated at the mine site will be identified prior to final disposal in the landfill. Waste will be removed for final disposal in accordance with current regulations. Materials to be disposed will be covered as required to prevent dust generation by wind during transport. Sludge will be removed from the sewage treatment plant and transported to the landfill in accordance with current regulations. Prior to the completion of the landfill, wastes will be disposed of in authorized facilities in the Atacama Region.

Domestic waste generated at the port site will be stored in waste containers in all operating areas, and then collected and sent to authorized landfills in the Atacama Region.
18.12 Port

The port design for the 2014 Feasibility Study was prepared by PRDW Aldunate Vasquez (PRDW) and reviewed by AMEC. In this sub-section, AMEC initially presents the PRDW design and design considerations, and then provides AMEC’s comments and observations.

18.12.1 Introduction

The proposed port, Puerto Santo Domingo, will be located in the Punta Roca Blanca area which is located between Caleta Hornos and Punta Choros, in the Atacama Region.

Based upon current Capstone concentrate production requirements, the maximum required annual port capacity is 5.5 Mt/a of magnetite concentrate and 0.52 Mt/a of copper concentrate. Magnetite concentrate is planned to be shipped using a mixture of Panamax and Cape size vessels. Copper concentrate would be shipped using Panamax and Handymax size vessels.

18.12.2 Oceanic Conditions

Bathymetry

A bathymetric survey was performed by DESMAR for Capstone, along with official cartography by the Chilean Naval Hydrographic and Oceanographic Service. The coastline in the Project area is aligned along a west–southwest–east–northeast direction. The sea bottom near the shore (based upon the current geophysical and bathymetric information) appears to be irregular in contouring and has a slope of approximately 5% (away from shore). The seabed slope is less irregular further offshore, with an approximate slope of 3% (away from shore).

Tides

It is estimated that tides in the Project area are diurnal with a semi-diurnal component. There are two uneven high tides and two uneven low tides per day. Tide data used in the port design are based on readings from Chañaral, located 45 km north of the proposed port site. PRDW anticipated that there will be little difference between the tide characteristics at Chañaral and at the Project site.

Currents

No field data on ocean currents is available for the Project area. It has been estimated that current magnitudes are low (less than 0.5 m/s in extreme conditions). The
estimated values were for Chañaral. PRDW anticipated that there will be little difference between the currents at Chañaral and at the port site. For the port design, a mean current magnitude of 0.2 m/s and an extreme current magnitude of 0.5 m/s were used.

**Waves**

Spectral wave modelling was undertaken in order to characterize the operational and extreme local wave climate at the proposed port area. The model considered the propagation of offshore wave climate into shallow water at the port site. The model used a 20-year database of offshore wave climate, with sea states provided in three hourly intervals. Model results were validated using directional wave data measured during a field survey at the port site.

Significant wave heights\(^2\) at the berthing site are between \(H_{m0}^3\) = 0.2 m and 2.9 m, with peak periods in the range \(T_p^4\) = 6 s to 20s, predominantly at intervals of \(T_p\) = 12 s to 14 s, as expected for the Pacific Ocean.

Figure 18-8 provides a wave rose diagram for \(H_0\) and \(T_p\) for modelled waves at the proposed berthing site.

The incident wave climate has a narrow directional dispersion due to the bathymetric configuration of the coast at the port site, with a predominant incident wave direction of 290°N.

The results of the wave climate study have been used to define the berthing site orientation. The berthing orientation has been aligned to the predominant incident wave direction of 290°N.

**Winds**

Based on information from the maritime authority for operational wind magnitude, the following values were used by PRDW:

- Maximum wind speed – manoeuvring: 15 knots (~7.7 m/s)
- Maximum wind speed – moored vessels: 20 knots (~10.3 m/s).

For extreme wind values, wind gusts of five seconds in duration were used for the port structural design.

---

\(^2\) Significant wave height: average height of the third-highest waves in sea state period.

\(^3\) Significant wave height calculated with spectral data as zero\(^\text{th}\) moment wave height.

\(^4\) Peak period: corresponds to the period associated with the energy peak in the wave spectrum.
18.12.3 Geotechnical Considerations

Based on a visual inspection of the soil within the limits of the proposed onshore facilities, the area has a rocky soil. The soil in the seabed near the port and berthing area is also quite rocky. The depth of the rock at the berthing line is 25 m with no sand or gravel lenses. In deeper areas, where wave induced currents are milder, an overlying lens of sand and gravel was found during the preliminary geotechnical survey.

Using the results of the geophysical survey, the piles that will be required for the offshore facilities will be constructed as driven piles. These driven piles will include an anchoring system at the bottom to fix the pile into the seabed.

18.12.4 Port Layout

The proposed port layout plan is presented in Figure 18-9.
Figure 18-9: Proposed Port Layout Plan

Note: Figure prepared by PRDW, 2013. Grid squares on the figure are approximately 200 m.

**Offshore Facilities**

*Berth Orientation*

The recommended berth orientation of 290°N was determined using engineering models, a vessel motion study, berth availability estimates (operational downtime), and mooring and fender analyses.

*Draught Depth Requirements*

The required depth at the berth was defined as being 23 m, assuming the maximum draught of the largest vessel of the fleet is 19.6 m. Geophysical profiling indicated that the seabed was rocky, rather than sandy, in the port vicinity. Considering the high cost
of a possible dredge in a rocky seabed, the berth layout was prepared so as to avoid the need for dredging. The depth in the approaches to the berth is 24.0 m.

A manoeuvring circle of 664 m diameter was assumed.

Structure Heights

Studies indicate that the maximum wave crest elevation is +8.23 m natural reflection site (NRS) in the wave breaking zone, for a 100-year return period. As a result, elevation +9.0 m NRS was defined as the elevation of the top of steel beam of the access trestle. The elevation of the top of the concrete for mooring and breasting dolphins was established as +6.5 m NRS.

Access Trestle

The access trestle has been designed with a steel gallery to support the shiploading conveyor and medium duty vehicle access. The vehicle access will consist of a concrete slab roadway supported by structural steel beams along the trestle length. The trestle structure is approximately 316 m long and is supported by inclined piles driven into the seabed. Due to environmental requirements related to the loading of copper concentrate, all conveyor sections and galleries (onshore and offshore) will be enclosed in accordance with governmental regulations. The loading conveyor gallery within the access gantry will include a walkway for operator and maintenance access on both sides of the loading conveyor which will run the full length of the access trestle.

Take-up Tower

A take-up tower will be installed immediately prior to the ship loader which will contain the tensioning system for the loading conveyor. The take-up tower is required to eliminate the effects of live loads on the ship loader. An electrical room will also be installed on the first floor of the take-up tower. The other floors of the tower will be used for mechanical equipment for the conveyor system.

Platforms, Mooring Dolphins and Berthing Dolphins Structures

A total of five berthing dolphins, six mooring dolphins, a quadrant beam (the structure that allows the ship loader radial movement) and the ship loader support platform will be installed along the berthing line. The support platform is sized to allow vehicles to turn.

The berthing dolphins and the shiploader support platform will consist of concrete slabs over a steel beam structural system. The structural support system will be
attached to piles installed into the seabed. The mooring dolphins will be constructed using a concrete slab connected directly to the piles. The quadrant beam structure is integral with the steel beam support system and piles. For all structures the piles will be inclined in both transverse and longitudinal directions. These inclined piles are designed to provide the capacity to resist transverse and longitudinal seismic, wind, mooring, berthing and wave loads.

On the front side (facing the ship berth) of the berthing dolphins elastomeric fenders will be installed. These fenders will be designed and installed to dissipate the berthing energy of the ships due to wave and sea action.

Mooring hooks will be installed on all the berthing and mooring dolphins. These devices are used to receive the mooring lines for the ships. The mooring hooks located on top of the mooring dolphins will have a triple configuration. The hooks located on the top of the berthing dolphins will have a double configuration.

**Catwalks**

All the structures along the berthing line will be connected by steel catwalks. These catwalks will allow access to the berthing dolphins, mooring dolphins and quadrant beam.

**Onshore Facilities**

The onshore facilities for concentrate handling at the port are:

- Magnetite concentrate stockpile area; mechanical stacker and reclaim conveyor
- Copper concentrate storage building and reclaim conveyor
- Transfer towers
- Conveyor galleries
- Reclalm hoppers.

All structures have been designed using structural steel members. The structural steel and preliminary earthworks design have been completed for the magnetite concentrate stockpile area and the copper concentrate storage building.

**Magnetite Concentrate**

Key metrics for the magnetite concentrate, magnetite stockpile and ship transport requirements are included in Table 18-7.
<table>
<thead>
<tr>
<th>Concentrate</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Moisture</td>
<td>7.5–9.5%</td>
</tr>
<tr>
<td>Density - stock pile</td>
<td>2.8 t/m³</td>
</tr>
<tr>
<td>Density - conveyors</td>
<td>3.0 t/m³</td>
</tr>
<tr>
<td>Repose angle</td>
<td>34 to 36º</td>
</tr>
<tr>
<td>Surcharge angle</td>
<td>15º</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Stockpile</th>
<th>Value Phase 1</th>
<th>Value Phase 2</th>
</tr>
</thead>
<tbody>
<tr>
<td>Number of piles</td>
<td>1</td>
<td>1</td>
</tr>
<tr>
<td>Length</td>
<td>298 m</td>
<td>450 m</td>
</tr>
<tr>
<td>Width</td>
<td>50 m</td>
<td>50 m</td>
</tr>
<tr>
<td>Height, m</td>
<td>16.9 m</td>
<td>16.9 m</td>
</tr>
<tr>
<td>Capacity</td>
<td>320,000 t</td>
<td>500,000 t</td>
</tr>
<tr>
<td>Stockpile Area</td>
<td>14,900 m²</td>
<td>25,000 m²</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Shipping</th>
<th>Ship Capacity (DWT*)</th>
<th>Load Distribution (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cape size</td>
<td>250,000</td>
<td>20</td>
</tr>
<tr>
<td>Cape size</td>
<td>180,000</td>
<td>70</td>
</tr>
<tr>
<td>Panamax</td>
<td>80,000</td>
<td>10</td>
</tr>
<tr>
<td>Total</td>
<td>—</td>
<td>100</td>
</tr>
</tbody>
</table>

Notes: * = deadweight tonnage; # = Percentage of the total annual load distribution of magnetite concentrate

The reclaim hopper system for the magnetite concentrate will be located adjacent to the magnetite concentrate stockpile. The reclaim hopper system for the copper concentrate will be situated inside the copper concentrate storage building. The reclaim hoppers will discharge concentrate directly onto the transfer conveyor belt. All of the conveyors will be supported on steel structures with reinforced concrete foundations.

The magnetite concentrate will be filtered in a filter plant and transported by conveyor belt from the filter plant to a mobile stacker. The mobile stacker will run along the north side of the magnetite concentrate stockpile area. To minimize dust from the magnetite concentrate, the stockpile area will have wind breaks on all four sides as well as a water fog cannon system. The magnetite concentrate reclaim system will use front-end loaders that will load the magnetite concentrate from the stockpile to the
conveyor feed hoppers. From the transfer conveyor, the magnetite concentrate will be discharged onto a belt conveyor which will deliver the magnetite concentrate to the ship loader.

\textit{Copper Concentrate}

Key metrics for the copper concentrate, concentrate stockpile and shipping requirements are included in Table 18-8.

The copper concentrate will be delivered to the port by concentrate haul trucks which will discharge inside the copper concentrate storage building. The copper concentrate will be stored into two stockpiles within the enclosed concentrate storage building. Front-end loaders will distribute the concentrate to the stockpiles inside the building. The enclosed concentrate storage building will have a negative air pressure system and a dust collection system to minimize environmental impacts from loss of the copper concentrate. The total concentrate storage capacity will be approximately 50,000 dmt.

The copper concentrate reclaim system will use front-end loaders to transfer the concentrate from the stockpiles to conveyor hoppers installed over belt feeders that will in turn feed a transfer conveyor and a belt conveyor which will deliver the copper concentrate to the ship loader.

The ship loading system will consist of conveyors and a ship loader. The ship loader will be a radial design of the type currently used in several Chilean ports for the shipment of copper concentrate. The nominal capacity of the shiploader will be 4,000 t/h for iron concentrate and 2,000 t/h for copper concentrate.

\textit{Stockpile Area}

The first phase stockpile area considers a platform of approximately 66,500 m$^2$, of which 34,700 m$^2$ is for the iron concentrate stockpile. The other 31,800 m$^2$ will be to be used for the copper concentrate stockpile. The second phase needs an expansion of the iron concentrate stockpile area of about 14,800 m$^2$.

Environmental regulations require some protection around the perimeter of the iron concentrate stockpile area to reduce the wind speed in the vicinity of the stockpile so as to reduce iron dust generation.
Table 18-8: Copper Concentrate, Stockpile, and Shipping Parameter Assumptions

<table>
<thead>
<tr>
<th>Concentrate</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Moisture</td>
<td>8–9 %</td>
</tr>
<tr>
<td>Density - stock pile</td>
<td>2.0 t/m$^3$</td>
</tr>
<tr>
<td>Density - conveyors</td>
<td>2.5 t/m$^3$</td>
</tr>
<tr>
<td>Repose angle</td>
<td>34 to 36º</td>
</tr>
<tr>
<td>Surcharge angle</td>
<td>15º</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Stockpile</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Number of piles</td>
<td>2</td>
</tr>
<tr>
<td>Length, m</td>
<td>100</td>
</tr>
<tr>
<td>Width, m</td>
<td>30</td>
</tr>
<tr>
<td>Height, m</td>
<td>5</td>
</tr>
<tr>
<td>Capacity, t</td>
<td>25,000</td>
</tr>
<tr>
<td>Surface, m²</td>
<td>3,000</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Shipping</th>
<th>Ship Capacity (DWT)</th>
<th>Load Distribution$^5$ (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Panamax</td>
<td>60,000</td>
<td>15</td>
</tr>
<tr>
<td>Handymax</td>
<td>45,000</td>
<td>70</td>
</tr>
<tr>
<td>Handy size</td>
<td>20,000</td>
<td>15</td>
</tr>
<tr>
<td>Total</td>
<td>—</td>
<td>100</td>
</tr>
</tbody>
</table>

18.12.5 Port Planning

Ship Loading Rates

A port operations simulation was performed by PRDW using Arena software to simulate the operation of the concentrate handling and loading; to determine the occupancy level of the handling and loading system; to determine the size of the copper and magnetite concentrate stockpiles, verify ship loader loading rates, and estimate the scheduling and rotation of concentrate transport ships.

$^5$ Percentage of the total annual load distribution of copper concentrate.
The mining plan that was developed in mid-2013 was used as the assessment of the basis for the production of copper and magnetite concentrates over the life of the Project as follows:

- **Phase 1** is for the first three years of concentrate production, with the production of magnetite concentrate peaking at 3.3 Mt/a and copper concentrate reaching a maximum of 0.52 Mt/a. The volume of copper concentrate during Phase 1 determined the final sizing of the copper storage building and the reclaim rate.

- **Phase 2** is Year 4 to the end of life of mine (approximately Year 18). During Phase 2, the magnetite concentrate production peaks at 5.4 Mt/a and copper at 0.38 Mt/a. The volume of the magnetite concentrate during Phase 2 determined the size of the magnetite concentrate stockpile and the required reclaim rate.

For Phase 1 and 2, three nominal (or average) ship loading rates were tested: 3,000 t/h, 4,000 t/h and 5,000 t/h. The maximum occupancy level is assumed to be 40% in accordance with the recommendations of the United Nations Conference on Trade and Development (UNCTAD). Using a 3,000 t/h loading rate for magnetite concentrate, the estimated occupancy level exceeds the 40% maximum recommendation. Using a loading rate of 4,000 t/h for magnetite concentrate, the estimated occupancy level is below the maximum recommended level of 40%.

Based upon the study results, the selected nominal loading rate will be 4,000 t/h for magnetite concentrate and 2,000 t/h for copper concentrate. For Phase 1, the maximum level of the magnetite concentrate stockpile will be 320,000 t. For Phase 2, the maximum level of the magnetite concentrate stockpile will be 500,000 t. For copper concentrate, the maximum stockpile level for Phase 1 will be 50,000 t and 30,000 t for Phase 2.

**Loading Times**

The average loading time per type of vessel is summarized in Table 18-9.
### Table 18-9: Estimated Loading Time Averages

<table>
<thead>
<tr>
<th>Material</th>
<th>Ship Capacity (t)</th>
<th>Total Loaded (t)</th>
<th>Average Loading Time (h)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Iron Concentrate</td>
<td>80,000</td>
<td>80,000</td>
<td>38</td>
</tr>
<tr>
<td></td>
<td>180,000</td>
<td>180,000</td>
<td>64</td>
</tr>
<tr>
<td></td>
<td>250,000</td>
<td>250,000</td>
<td>83</td>
</tr>
<tr>
<td>Copper Concentrate</td>
<td>20,000</td>
<td>20,000</td>
<td>13</td>
</tr>
<tr>
<td></td>
<td>45,000</td>
<td>20,000</td>
<td>11</td>
</tr>
<tr>
<td></td>
<td>60,000</td>
<td>20,000</td>
<td>10</td>
</tr>
</tbody>
</table>

### Manoeuvring Studies

A manoeuvring feasibility study was performed to analyze the berthing and unberthing manoeuvres of the ships at the terminal. PRDW's conclusions included:

- The berthing and unberthing manoeuvres are relatively simple with enough space in front of the terminal.
- The manoeuvring area is partially sheltered from offshore waves. Therefore, the limiting wave height for manoeuvres is defined as $H = 1.5$ m in comparison with the limiting wave height of $H = 1.0$ m for the nearby existing terminals. This condition imposes a higher bollard pull requirement, but ensures a lower downtime due to manoeuvres.
- The manoeuvring circle was slightly displaced to avoid a shallow-water area near the terminal. It is recommended that a further bathymetric study be done of the access channel in the future stages of the project. The pilot station would be located approximately one nautical mile from the terminal. The anchorage area would be located at Caldera Bay, in line with the regulations for existing terminals.
- The recommended manoeuvre analyzed at this stage of the Project is considered to be the most simple and safe. However, other manoeuvres can be considered in future stages of the Project under requirement of the Navy Authority or as a result of a more detailed manoeuvring study.
- The berthing and unberthing maneuvers will require support of tugs; the number of tugs needed will depend on the ship size and loading conditions.

#### 18.12.6 Port Availability

In order to determine the expected operational port availability, the dynamic response of the moored vessels to the environmental conditions was modeled. For the copper
concentrate vessel, just one mooring system was considered in the model. For the iron concentrate vessel, forward and backward shifting positions, were considered due to vessel’s dimensions. Both vessels were tested at fully-laden conditions.

The modeled scenarios cover the range of local environmental conditions, considering winds, waves and currents. Waves are the main exciting agent for moored vessels.

To assess the port availability for both types of vessels the following criteria were followed:

- Safety limits: 50% of MBL\textsuperscript{6} and safety factor of 2.0 for mooring lines, and 72% of maximum deflection for fenders.
- Operational limits: PIANC\textsuperscript{7} recommendations for amplitude of motions and turns.

The availability was determined over a 20-year time series of wave conditions. The expected average non-availability for copper concentrate loading over a 20-year period is 10.4%. For iron concentrate loading, the expected downtime over a 20-year period is 1.1% and 2.1% for backward and forward shifting positions respectively. Motion failures were found to be associated with excessive amplitude of yaw motion for copper concentrate vessel, whilst excessive loads on mooring elements, caused by environmental forces, are the main reason of failure in the iron concentrate vessel case.

From the results of the moored vessel and operational study and coastal conditions, PRDW developed an assessment of the operational non-availability for copper and iron ore.

Based on PRDW’s analysis, the sea conditions could have an impact on copper concentrate shiploading primarily from November till May. However, loading of iron ore ships will not be greatly impacted by weather conditions, based on the same analysis. In summary, PRDW concluded that the terminal would be able to load the annual throughput envisaged in the 2014 Feasibility Study.

The non-availability for copper concentrate shiploading is summarized in Table 18-10. The annual availability for iron concentrate vessels, stern shifting is presented in Table 18-11. Table 18-12 shows the annual availability for iron concentrate vessels, bow shifting. From these tables, it can be seen that there is significant year to year variation in the non-availability and the peak downtime in any given month during winter season can be considerably higher than the average. The non-availability affects copper vessels more than the iron vessels.

\textsuperscript{6} MBL= Minimum Breaking Load
\textsuperscript{7} PIANC= Permanent International Association of Navigation Congresses.
18.12.7 AMEC Comments on Port

PRDW provided AMEC with the information in Table 18-10 to Table 18-12 on monthly port availability over the period 1985 to 2006 for copper and magnetite concentrate. Data in these tables indicate that in some years, there are six months (November to May) where the port availability to load copper concentrate can be impacted by sea conditions.

During these months, sea conditions can be such that ship movements at the dock will exceed safe ship loading operational parameters. Such sea conditions are a common issue for harbours along the Peru–Chile coastline, and are typically mitigated through selection of a sheltered harbour location and/or construction of breakwaters. Another option is to utilize a spread mooring system and a travelling shiploader in order to eliminate the need for ship repositioning during ship loading operation.

These likely monthly port availability impacts will have flow-on effects for copper storage requirements during the years that have more extreme sea conditions. They may also impact the concentrate supply scheduling, with short-term impacts on concentrate delivery contracts.
## Table 18-10: Annual Non-Availability for Copper Concentrate Vessel

<table>
<thead>
<tr>
<th>Year</th>
<th>Ene</th>
<th>Feb</th>
<th>Mar</th>
<th>Abr</th>
<th>May</th>
<th>Jun</th>
<th>Jul</th>
<th>Ago</th>
<th>Sep</th>
<th>Oct</th>
<th>Nov</th>
<th>Dic</th>
<th>% Anual</th>
</tr>
</thead>
<tbody>
<tr>
<td>1985</td>
<td>53.6</td>
<td>9.4</td>
<td>19.0</td>
<td>6.9</td>
<td>4.6</td>
<td>3.2</td>
<td>0.8</td>
<td>56.5</td>
<td>12.8</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1986</td>
<td>54.4</td>
<td>85.3</td>
<td>67.7</td>
<td>0.8</td>
<td>12.9</td>
<td>12.1</td>
<td>16.7</td>
<td>58.9</td>
<td>25.7</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1987</td>
<td>69.5</td>
<td>62.1</td>
<td>38.3</td>
<td>8.8</td>
<td>1.7</td>
<td>6.9</td>
<td>17.5</td>
<td>74.2</td>
<td>28.6</td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>1988</td>
<td>15.4</td>
<td>9.3</td>
<td>9.9</td>
<td>8.8</td>
<td>17.9</td>
<td>9.3</td>
<td>13.9</td>
<td></td>
<td></td>
<td></td>
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<td>1989</td>
<td>22.8</td>
<td>8.5</td>
<td>11.7</td>
<td>6.5</td>
<td>12.9</td>
<td>3.2</td>
<td>12.9</td>
<td>8.1</td>
<td>7.7</td>
<td></td>
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<td></td>
</tr>
<tr>
<td>1990</td>
<td>54.8</td>
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<td></td>
<td></td>
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</tr>
<tr>
<td>2006</td>
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<table>
<thead>
<tr>
<th>Year</th>
<th>Min</th>
<th>Mean</th>
<th>Max</th>
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<tbody>
<tr>
<td>1985</td>
<td>0.0</td>
<td>7.2</td>
<td>50.4</td>
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<td>1986</td>
<td>0.0</td>
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<td>50.4</td>
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<td>1988</td>
<td>0.0</td>
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<td>1989</td>
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<td>1990</td>
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</tr>
<tr>
<td>1991</td>
<td>0.0</td>
<td>0.0</td>
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<td>1992</td>
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<tr>
<td>1993</td>
<td>0.0</td>
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<td>1995</td>
<td>0.0</td>
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<td>2005</td>
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<td>0.0</td>
</tr>
<tr>
<td>2006</td>
<td>0.0</td>
<td>0.0</td>
<td>0.0</td>
</tr>
</tbody>
</table>
AMEC recommends that Capstone commissions a field data collection program to establish the site-specific sea conditions for the Santo Domingo port. Further berth alignment analysis is required to estimate port availability for ship loading operation and its impact on the upstream operation. Based on this information, dock design and concentrate loading and storage arrangements should be further optimized. Consideration should also be given to reviewing the potential impact of the port non-availability on the shipping concentrate scheduling during the winter season.

18.13 Power and Electrical

Project facilities requiring power are located at the following sites:

- Mine and plant site located near Diego de Almagro
- Santo Domingo Port at Punta Roca Blanca.

The estimated overall project maximum power loads are presented in Table 18-13. The total average demand is included as Table 18-14.

The mine and plant site area includes the mine, process plant, infrastructure and tailings facility electrical loads. The port facilities include the sea water pump station, desalination plant, magnetite concentrate filtration plant, concentrate storage and handling, and associated infrastructure.

18.13.1 Mine and Plant Site

The mine and plant main substation will have two power transformers of 55 MVA (ONAN), 220/23 kV to support the maximum demand of the system of 96.4 MVA.

The loads at the plant site include the following:

- The maximum demand for the process plant and mine is 95.1 MVA.
- The maximum demand for the TSF including thickening, distribution and water reclaim is 1.3 MVA. The TSF consumption is included in the main substation loads.

The maximum demand for the permanent camp is 0.4 MVA. This demand is included in the loads at the main plant substation but will be supplied directly from the Diego de Almagro distribution system.
Table 18-13: Total Estimated Loads and Maximum Demand

<table>
<thead>
<tr>
<th>Area Description</th>
<th>( P_{\text{nom}} ) (kW)</th>
<th>( P_{\text{connect}} ) (kW)</th>
<th>( D_{\text{max}} ) (kVA)</th>
<th>( D_{\text{max}} ) (kW)</th>
<th>Power Transformers</th>
</tr>
</thead>
<tbody>
<tr>
<td>Main plant substation</td>
<td>105,139</td>
<td>103,641</td>
<td>105,163</td>
<td>95,553</td>
<td>Main plant substation</td>
</tr>
<tr>
<td>Subtotal On-Site</td>
<td>105,139</td>
<td>103,641</td>
<td>105,163</td>
<td>95,553</td>
<td>Subtotal On-Site</td>
</tr>
<tr>
<td>Port substation facilities</td>
<td>20,873</td>
<td>17,969</td>
<td>18,551</td>
<td>16,145</td>
<td>Port substation facilities</td>
</tr>
<tr>
<td>Subtotal Off-Site</td>
<td>20,873</td>
<td>17,969</td>
<td>18,551</td>
<td>16,145</td>
<td>Subtotal Off-Site</td>
</tr>
<tr>
<td>Total Project</td>
<td>126,012</td>
<td>121,610</td>
<td>123,714</td>
<td>111,698</td>
<td>Total Project</td>
</tr>
</tbody>
</table>

- \( P_{\text{nom}} \): Rated power (includes stand-by load) (kW)
- \( P_{\text{connect}} \): Connected power (does not include stand by load) (kW)/(kVA)
- \( D_{\text{max}} \): Maximum system demand (kW)/(kVA)

Table 18-14: Total Average Demand and Annual Power Consumption

<table>
<thead>
<tr>
<th>Site</th>
<th>Average Demand ( (P_{\text{AVD}}) ) (kW)</th>
<th>Annual Power Consumption ( (E_y) ) (MWh)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mine and Plant Site</td>
<td>72,788</td>
<td>592,991</td>
</tr>
<tr>
<td>Port</td>
<td>12,891</td>
<td>105,017</td>
</tr>
</tbody>
</table>

18.13.2 Port Site

The port main substation will have one power transformer (25 MVA (ONAN), 220/23 kV) to support the maximum demand of 16.5 MVA. The loads at the port include the following:

- The maximum demand at the pump station is 8.6 MVA (provided by BRASS). This is for a (nominal throughput) design flow of approximately 375 L/s nominal (maximum 408 L/s)
- The maximum demand for the filter plant and desalination plant at the port site is 4.9 MVA.
- The maximum demand for the sea water intake at the port is 3.0 MVA (provided by BRASS).
The average demand and power consumption for the mine site and the port are calculated in Table 18-16 on the basis of treating between 60,000 t/d and 65,000 t/d over the approximate 18-year mine life and an assumption of an annual average availability of 93%.

18.13.3 Power Supply

Capstone’s mine site and port site will be connected to the Central Interconnected System (SIC in the Spanish acronym) which covers the central part of Chile and has coal and diesel thermoelectric plants in the Project area. The closest connection point between the SIC and the mine site is via a direct connection to the Diego de Almagro substation, located about 5 km from the mine area.

18.13.4 Power Availability

Based on independent market analysis research provided to Capstone, Chile is likely to have a difficult situation regarding power availability for the next five years. However, recent studies support that measures being currently undertaken to provide grid interconnections and to expand power generation, are likely to ameliorate this situation after the projected five-year bottleneck. In the Norte Chico area of the SIC system, where Santo Domingo is located, there is currently a lack of generation and there are power line restrictions that do not allow power from the south-central area of the SIC to be transmitted to the Norte Chico area. Consequently, the marginal costs in this northern part of the SIC system are currently the highest in Chile.

AMEC notes that there are numerous mining projects being evaluated in the Norte Chico with potentially large increases in power demand. Power generation projects were planned to meet the demand (Barrancones, Castilla, Punta Alcalde). However, all of these have been rejected or suspended because of environmental conditions or community opposition. Currently, only the Punta Alcalde project may be able to be approved.

There is excess generating capacity available in the south–central area. However, this surplus cannot be transmitted to the north due to limitations on transfer capacity into the Norte Chico area (particularly between Los Vilos and Maitencillo). This limited transfer capacity results in differences in marginal costs between the south and the north. Until the SIC interconnection or the SIC–SING interconnection transmission system is extended to the north, the increase in the power demand must be met by the development of new generation projects in the north.

The regulatory authority is trying to increase the capacity of transmission between the central zone and the extreme north of the SIC (lines between the Diego de Almagro
and Polpaico substations) (Resolution Ex. 885, December 2010, Expansion Plan for Trunk Line Transmission System). By 2018, it is possible that power can be supplied to the north from the excess capacity in the central zone. Currently, there is discussion of a law that will facilitate the procedures for developing power transmission systems. There is no firm timing on the implementation of this law.

There is also a project to interconnect the SING between Mejillones (or Crucero) to the Cardones substation in the SIC. This interconnection would also increase the power supply in the northern part of the SIC. There are two initiatives for this project, a private one by the Suez company, and a public project. This project would have a transmission capacity of 1,500 MW. A tentative date for starting operations of the completed interconnection is 2020. The government wishes to make a legislative change to define the form of payment for this interconnection between the SING and the SIC.

The mining operation at Santo Domingo is expected to start up prior to 2020. To enhance viability and provide incentive for further project advancement, it is preferred that a power supply source and cost for the initial years of operations be confirmed. This confirmation of supply and pricing could be done using a short-term bridging contract. However, a short-term bridging contract may leave Capstone exposed to future market conditions. Current bridging contract discussions indicate that prices would be at marginal costs and there is no guarantee that these costs would reduce at the end of the contract.

**Power Production Options**

In Chile, power production is private (there is no public generation capacity) and power is sold by contracts that the providers execute directly with either non-regulated customers (large industrial and mining companies such as Capstone) or with large regulated customers (distribution companies that supply domestic customers, service companies and small industry). The only participation that the state has is oversight of contracts for regulated customers.

Capstone will be a non-regulated customer because its connection to the system will exceed 2 MW. This means that Capstone must arrange its own power supply through private generation or third party supply.

**Assessment of Private Generation**

Private power generation for the mine and plant and/or the port is an option. However, this is not economic as it would require a power plant (conventional or renewable) with sufficient capacity for both the maximum power and the maximum demand.
Current solar project developments in the area may provide an opportunity for Capstone to obtain power for the Project at affordable prices. Discussions with some of these project owners are providing promising opportunities for the Project. Capstone intends to continue to develop these opportunities.

To complete a private power generation project and interconnect to the national grid, it would be necessary to set up a power company. This approach would give Capstone the right to access power from the SIC. However, depending upon when the SIC power is required (time of day, week), marginal costs may be in effect. This supply cost risk can be minimized by increasing the installed private generation capacity and installing back-up generation. The increased capacity would further increase the private power generation costs.

For generation projects larger than 3 MW, an environmental impact study must be prepared and submitted.

The most common supply option in Chile is to agree to a Power Purchase Agreement (PPA) with a generating company. Regulations in Chile allow any connected generating company to take electricity at any transmission point to supply their customers.

**Power Supply Strategy**

Capstone’s power supply strategy is consistent with the Chilean market. Capstone plans to sign a PPA with either an independent provider currently operating in the SIC, or one who will be connected with the SIC when Capstone needs power. Two market surveys have been carried out to investigate potential power suppliers for both options.

A further supply opportunity for Capstone is to develop a mixed supply using conventional technology and renewable energy (energia renovable no convencional or ERNC). The power supply options being advanced by Capstone are:

- Contract a power supply block to a single, conventional technology energy supply company
- Combine a conventional energy technology offer with an ERNC energy offer to produce a mixed power supply package
- Contract with several ERNC suppliers for the total power required by the Project.
19.0 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

Capstone commissioned Cliveden Trading AG (CTAG) to prepare an independent marketing study on the Project product. The study was prepared on the understanding that Kores has agreed and is required to purchase 50% of the annual production of copper concentrate and iron concentrate, leaving Capstone to market and sell the remaining 50%. The Kores terms and conditions will be based on the Capstone terms obtained independently in the market.

Capstone contracted the Copper Research Unit (CRU) to supply a short paper on price projections for copper and their best estimate for treatment charges (TC) and refining charges (RC) over the life of the Project. These estimations are based on current assumptions of global supply and demand which may change, and depend on a number of factors including, the world financial climate, permitting issues, personnel availability, political issues and directives, natural disasters and other unforeseen events that may have an impact on supply or demand.

TC/RCs and marketability of the concentrate are based on CTAG’s in-depth knowledge of the copper concentrates market, various documents in the public domain and statistics and papers from CRU.

19.1.1 Copper Concentrate

Santo Domingo Project Likely Product Specifications

For the purposes of assessing the marketability of the copper concentrates, Minera Santo Domingo supplied the analysis of the copper concentrate specifications in Table 19-1. The specifications are from the PFS-level metallurgical testwork and are not smelter specifications.

Deleterious Elements and Penalties in Copper Concentrates

China has strictly controlled the importation of copper concentrates with specified limits on certain deleterious elements, imposing a ban on the importation of materials containing more than (any one element) 0.5% arsenic (by weight) per dmt (of copper concentrate), 6% lead per dmt, 1,000 ppm fluorine per dmt, 500 ppm cadmium per dmt, and 100 ppm mercury per dmt.
Table 19-1: Copper Ore Concentrate Specification

<table>
<thead>
<tr>
<th>Chemical Element</th>
<th>Unit</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cu</td>
<td>%</td>
<td>30.3</td>
</tr>
<tr>
<td>Fe</td>
<td>%</td>
<td>29.4</td>
</tr>
<tr>
<td>S</td>
<td>%</td>
<td>30.9</td>
</tr>
<tr>
<td>Au</td>
<td>g/t</td>
<td>3.4</td>
</tr>
<tr>
<td>Ag</td>
<td>g/t</td>
<td>26.6</td>
</tr>
<tr>
<td>Hg</td>
<td>g/t</td>
<td>4.6</td>
</tr>
<tr>
<td>Cl</td>
<td>g/t</td>
<td>292</td>
</tr>
<tr>
<td>F</td>
<td>%</td>
<td>0.013</td>
</tr>
<tr>
<td>SiO₂</td>
<td>%</td>
<td>1.89</td>
</tr>
<tr>
<td>As</td>
<td>g/t</td>
<td>67</td>
</tr>
<tr>
<td>Bi</td>
<td>g/t</td>
<td>&lt;40</td>
</tr>
<tr>
<td>Cd</td>
<td>g/t</td>
<td>&lt;10</td>
</tr>
<tr>
<td>Co</td>
<td>g/t</td>
<td>416</td>
</tr>
<tr>
<td>Cr</td>
<td>g/t</td>
<td>102</td>
</tr>
<tr>
<td>Pb</td>
<td>g/t</td>
<td>198</td>
</tr>
<tr>
<td>Mn</td>
<td>g/t</td>
<td>312</td>
</tr>
<tr>
<td>Ni</td>
<td>g/t</td>
<td>79</td>
</tr>
<tr>
<td>Sb</td>
<td>g/t</td>
<td>195</td>
</tr>
<tr>
<td>Se</td>
<td>g/t</td>
<td>195</td>
</tr>
<tr>
<td>Sn</td>
<td>g/t</td>
<td>36</td>
</tr>
<tr>
<td>Zn</td>
<td>g/t</td>
<td>&lt;20</td>
</tr>
<tr>
<td>P80 µm</td>
<td>—</td>
<td>44.4</td>
</tr>
</tbody>
</table>

Note: The specifications are from the PFS-level metallurgical testwork and are not smelter-derived specifications.

China is the world’s single largest (by tonnage) consumer of seaborne copper concentrates, hence any copper concentrate containing levels in excess of the above limits will be placed at a significant disadvantage in terms of its marketability, and therefore is likely to be discounted heavily compared to ‘clean’ copper concentrate market terms.

Due to the restriction on arsenic-containing copper concentrate worldwide for most smelters, concentrate is heavily penalized if arsenic levels exceed 0.3% per dmt of copper concentrate. There are only a limited number of smelters worldwide that can treat high levels of arsenic in copper concentrate; most such concentrates are sold at steeply-discounted terms to traders who dilute it with ‘clean’ copper concentrates and sell the blended product.
In addition to the arsenic, mercury, cadmium, fluorine and lead that the Chinese authorities limit, other elements are also regarded by copper smelters as deleterious to the smelting process or harmful to the environment. Such elements include antimony, bismuth, selenium, tellurium, cadmium and zinc.

Silica, although a normal constituent of most copper concentrates, is not strictly a deleterious element because it is needed in minor amounts to create slag; however, it is likely to be penalized if it is over the 10% threshold. In contrast there are some smelters that prefer higher silica content; hence this is subject to individual smelter requirements.

Due to more exacting environmental standards smelters in general are rarely allowed to vent gases directly to the atmosphere and are forced to capture a number of elements at a high cost. As a result most smelters will usually penalize particular elements to take account of the extra charges incurred in processing them.

Each smelter has its own tolerances on deleterious and environmentally unacceptable elements, but typically penalties for copper concentrate would be as shown in Table 19-2.

**Assessment of Project Concentrates**

Based on specifications of the copper concentrate provided by Capstone to CTAG, CTAG concluded that the concentrate produced from the Project would generally be considered clean. Chlorine and fluorine are under the limits at which penalties are normally applied, and if they are occasionally over the limit it is likely that only a nominal penalty would apply.

Because of the clean elemental composition of the concentrate, CTAG considers that Capstone’s concentrate from the Santo Domingo Project can expected to be in high demand from trading companies specialising in blending complex materials with clean materials.
<table>
<thead>
<tr>
<th>Element</th>
<th>Terms</th>
<th>Note: values are in % or ppm of contained element per dmt in the concentrate</th>
</tr>
</thead>
<tbody>
<tr>
<td>Arsenic (As)</td>
<td>Below 0.2% no penalty 0.2% to 0.5% of As content then US$2 per 0.1%(As) per dmt Additionally from 0.5% to 1.5% of As content then US$5 per 0.1% (As) per dmt Additionally above 1.5% of As content then $8 per 0.1% (As) per dmt</td>
<td></td>
</tr>
<tr>
<td>Antimony (Sb)</td>
<td>A penalty of US$4.00 for each 0.1% (Sb) over 0.1% (contained Sb) per dmt</td>
<td></td>
</tr>
<tr>
<td>Zinc (Zn)</td>
<td>A penalty of US$1.50 for each 1.0% (Zn) over 3% (contained Zn) per dmt</td>
<td></td>
</tr>
<tr>
<td>Lead (Pb)</td>
<td>A penalty of US$0.20 for each 0.1% (Pb) over 0.6% (contained Pb) per dmt</td>
<td></td>
</tr>
<tr>
<td>Mercury (Hg)</td>
<td>A penalty of US$2.00 for each 10 ppm (Hg) over 10 ppm (contained Hg) per dmt</td>
<td></td>
</tr>
<tr>
<td>Bismuth (Bi)</td>
<td>A penalty of US$3.00 for each 0.01% (Bi) above 0.01% (contained Bi) per dmt</td>
<td></td>
</tr>
<tr>
<td>Nickel plus Cobalt (Ni + Co)</td>
<td>A penalty of US$1.00 for each 0.1% (Ni and Co combined) above 0.5% (contained Ni and Co combined) per dmt</td>
<td></td>
</tr>
<tr>
<td>Chlorine (Cl)</td>
<td>A penalty of US$0.50 for each 0.01% (Cl) above 0.05% (contained Cl) per dmt</td>
<td></td>
</tr>
<tr>
<td>Fluorine (F)</td>
<td>A penalty of US$0.10 for each 0.001% (F) over 0.03% (contained F) per dmt</td>
<td></td>
</tr>
</tbody>
</table>

**Copper Concentrate Global Market Assessment**

Copper concentrates can be sold under a number of different agreements:

- Long term off-take agreements or frame contracts
- Mid-term agreements or mid-terms
- Evergreen contracts
- Spot contracts
- Trader off-take agreements

Copper concentrates when delivered to end users are sold based on a payment which is the sum of the addition of all the component ‘payable’ metals (copper, gold, silver and sometimes platinum and palladium) less the sum of the TCs, less the sum of the RCs for copper, silver and gold, less the sum of any penalties and discounts. The amount of payable metal and TC/RCs vary from contract to contract.

Copper content is paid for at 96.5% of the full and final (after final assays are agreed) assayed quantity, but this would typically be subject to a minimum deduction of one
unit of copper. For all practical purposes if the copper content drops below 28.59% in the concentrate, the payable copper will be the copper content less one unit (e.g. if the copper content is 28% the amount payable would be 27% (i.e. 28 - 1)). The price paid for the copper content is usually an averaged price based on a quotation period of the London Metal Exchange (LME) quoted cash copper settlement price (i.e. the seller’s price of copper at the midday close on the LME) on each day during that average period.

For precious metal payments there are two different methodologies usually used to determine the payable quantities. The first and most widely used, due to its dominance of the smelting market is referred to as Asian-style pricing. The lesser used, but more traditional, is termed European-style pricing.

- **Asian-style pricing:** silver is paid for at 90% of the full and final assayed quantity of the silver, provided that the silver content is above a minimum of 30 g per dmt. Below this threshold silver would not normally be payable. Gold is payable on a percentage based on a sliding scale, of the full and final assayed quantity provided that there is a minimum of 1 g per dmt of gold contained. Below this threshold gold is not payable.

- **European-style pricing:** silver is payable on the full and final assayed quantity of silver less a deduction of 30 g. Any content below 30 g would not be payable. In higher silver content concentrates there is often a deduction of 50 g made rather than the 30 g. Gold is payable on the full and final assayed quantity of gold less a deduction of 1 g. In concentrate containing less than 1 g per dmt there would be no payment.

Another variation on the actual price paid for each metal (copper, silver and gold) occurs when copper concentrates are exported to the United States. Prices paid for the payable metals are based on the Comex (the New York Mercantile Exchange’s Commodity Exchange division) traded first position (essentially the spot month). This is only used on concentrate delivered to, or internally within, the United States.

Copper concentrate long term frame contracts are typically highly sought after by smelters. Smelters, especially in China, have been operating at well below capacity. Over the last decade spot TC/RCs for concentrate supply have been running at a discount of $30 to $40 to the long-term contract rates. There is a trend in worldwide concentrates to a higher average arsenic content. The trend is partly a result of general trends in large orebodies currently being mined, but is also due to higher commodity prices for contained copper, gold and silver in concentrates. This results in many high arsenic mines in, for example, Peru, Mexico, South America, the Philippines and Bulgaria continuing production, despite very high penalties for the arsenic content of the concentrates produced relative to the clean concentrate market.
Project Concentrate Marketing Assessment

The Project copper concentrate has a low gold content (around 3 g) and a low silver content (around 28 g). CTAG notes that it is likely that there will be considerable value to pricing the material on an Asian-style basis as opposed to European-style pricing. This will be accentuated when, occasionally, the final silver content rises above 30 g per dmt. If this percentage is payable using European terms the payment is very low, but with Asian terms +89% would be payable.

A number of factors must be taken into consideration when assessing the best contract partners for Capstone on a long term basis. Freight, assay bias, geographic location and contractual party reliability will all be factors.

The normal contract split for mines of the proposed size of the Project are:

- 60% to 70% on long term frame contracts with four or five major smelters
- 10% to 20% to traders on three- to five-year fixed TC/RC or TC/RC to be negotiated annually
- 20% to 30% spot contracts for up to one year with traders at fixed terms.

Long-term contracts should be adjusted to incorporate the initial high concentrate production in the first year of operation, and the gradual annual decline in copper production thereafter.

The timing to secure sales contracts is entirely dependent on the progress of financial arrangements for the project financing. It is likely that banks or financial institutions will want to have signed letters of intent (LOIs) or memorandums of understanding (MOUs) from smelters followed by a graduation to full long-term contracts as a condition of the completion of future Project financing.

It is planned to ship the copper concentrate from a Northern Chilean port. However, due to the low monthly shipment tonnages (approximately 40,000 t to 50,000 t per month) and storage facilities (40,000 wmt storage) most of the shipments will be made using Handymax vessels (under 27,000 wmt carrying capacity) or Supramax vessels (50,000 to 55,000 wmt). CTAG’s expectations of freight rates for the next few years is that for the next two years there is likely to be no change, thereafter a 6% to 7% increase per year is expected.

19.1.2 Iron Concentrate

Capstone contracted CRU to supply a report on price projections for iron ore concentrate (62% Fe content sinter fines) and, using their best judgement, provide a
marketability report and estimate of expected sales revenue that the iron ore concentrate produced by Capstone might yield if sold on the market. These estimations by CRU are based on current assumptions of forward global supply and demand and therefore may change depending on a number of factors, including, the world financial climate, permitting issues, project specific problems and a number of other contributing events.

CTAG was contracted by Capstone to comment on the general aspects of the iron ore business, background information, pricing structure, revenue expectations based on a CRU report, weighing and sampling, and freight.

**Santo Domingo Project Likely Product Specifications**

For purposes of assessing the marketability of the iron ore concentrate Minera Santo Domingo supplied the expected analysis of the iron ore concentrate shown in Table 19-3. The specifications given in the table are from the PFS-level metallurgical testwork and are not smelter specifications.

**Deleterious Elements and Penalties in Iron Concentrates**

Steel is made in a two step process; first the iron ore is smelted in a blast furnace to make pig iron and the pig iron is further refined in a different furnace to reduce impurities to produce the final steel product.

Each steel mill has different impurity allowances and tolerances. Each is unique in its requirement for feed and will try very carefully to blend the constituent elemental requirements in the iron ore concentrate. However, as a generality, it can be said that most mills prefer SiO₂ <3.5% though in pellet plants this may be as high as 5.5%; Al₂O₃ <1%; Mn <0.5%; P <0.1%; S <0.1% Cu <0.01%; and a combined Na₂O and K₂O <0.5%. The Al₂O₃ is a cost factor due to its endothermic reaction (and consequent heat absorption cost). Other rarely-found elements in iron ore such as copper are also problematic beyond a certain concentration, but the reality is that Cu can often be blended out in the charge feed mix since Cu is usually only found in trace amounts in most iron ore types. If impurities are higher than the levels discussed, then it becomes more difficult, but not impossible, to place material with mills.

The iron ore used in the charge mix for blast furnaces usually consists of the following materials: 15% to 20% pellet, 70% to 75% sinter, and 10% to 15% lump.
Table 19-3: Iron Ore Concentrate Specification

<table>
<thead>
<tr>
<th>Chemical Element</th>
<th>Unit</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fe_{tot}</td>
<td>%</td>
<td>66.06</td>
</tr>
<tr>
<td>FeO</td>
<td>%</td>
<td>23.08</td>
</tr>
<tr>
<td>SiO$_2$</td>
<td>%</td>
<td>4.10</td>
</tr>
<tr>
<td>Al$_2$O$_3$</td>
<td>%</td>
<td>1.00</td>
</tr>
<tr>
<td>CaO</td>
<td>%</td>
<td>0.57</td>
</tr>
<tr>
<td>MgO</td>
<td>%</td>
<td>0.455</td>
</tr>
<tr>
<td>P</td>
<td>%</td>
<td>0.011</td>
</tr>
<tr>
<td>S</td>
<td>%</td>
<td>0.020</td>
</tr>
<tr>
<td>Cl</td>
<td>ppm</td>
<td>60</td>
</tr>
<tr>
<td>Na$_2$O</td>
<td>%</td>
<td>0.145</td>
</tr>
<tr>
<td>K$_2$O</td>
<td>%</td>
<td>0.105</td>
</tr>
<tr>
<td>Mn</td>
<td>%</td>
<td>0.069</td>
</tr>
<tr>
<td>Cu</td>
<td>%</td>
<td>0.0081</td>
</tr>
<tr>
<td>L.O.I</td>
<td>%</td>
<td>1.34</td>
</tr>
<tr>
<td>&gt;40 µm</td>
<td>%</td>
<td>21.3</td>
</tr>
<tr>
<td>Blaine</td>
<td>cm$^2$/g</td>
<td>1,896</td>
</tr>
</tbody>
</table>

Note: The specifications are from the PFS-level metallurgical testwork and are not smelter-derived specifications.

Haematite in its natural form from the Pilbara and Carajas (which makes up the majority of the feedstock for steel mills around the world) is too fine untreated to add directly to the blast furnace; it would tend to clog the charge mix and render the furnace inefficient. Therefore, the ultra-fine (UF) iron ore is pre-treated in a process known as sintering. Sintering involves mixing the UF material with coal breeze dust or other materials and heating the mixture. The resultant brittle material is then broken into smaller pieces of sufficient size and strength that when added to the blast furnace will allow air to flow. Pellets and lump iron ore (naturally occurring lump iron ore is 10 mm to 60 mm in size) are also added with the sinter feed.

Pellets can also be made from UF iron ore. Pellets are made by agglomerating the UF material into small pellets with a binding agent and then baking them at a high temperature in a grated oven. The resultant pellets can then be stored. The advantage of pellets compared to sinter is that pellets do not rapidly degrade and can be readily transported.

Pellets are best when made from UF material, as it is much easier to bind the particles together to form the initial ‘green’ pellets to be baked. To measure the grain size suitability for this process a measurement called the Blaine Index is used. The Blaine
Index is a measure of the surface area per gram of weight, and is a measurement of the general grind size of the material. Pellet plants normally require a Blaine Index higher than 1,200 cm$^2$/g.

Magnetite is the preferred feed for pellet plants, because when heated in the pelletizing process heat is exothermically generated, thus saving on fuel bills. Western firms have tended, in recent years, to use a 25% to 30% pellet feed overall, to lessen fuel costs and improve the quality of the steel product. China has on average a lower percentage of pellet feed in charge feed mix, because pellet feed is more expensive and energy savings are less relevant for them. As China moves towards higher quality standards it is likely that the percentage of pellets in the feed mix will increase, thus increasing the demand for pellet feed.

**Assessment of Project Concentrates**

The main levels of impurities as far as the Project is concerned are silica and copper. Copper is below the threshold but may in some circumstances represent a non-preferred feed. Silica is only likely to be a cost factor or penalty element rather than a rejectable quality issue. Silica penalties are variable but would be of the order of $1.5 to $2 per each 1% above 3.5%.

China is a silica-impaired destination, as their local ores often have very high levels. Therefore, blending down in China is not readily possible. Some mills will reject material above 6% silica, others will reject at higher percentages. There is a very large variance in silica tolerance. Capstone will need to be selective in finding a steel mill with the right fit for the Project material in terms of quality.

Iron ore in China is normally sold on the basis of 62% iron content. For material under 62% there are penalties that are based on double the Fe content unit cost. For example, if a merchant delivers 60% Fe content and the original price committed to the end consumer was $124 per dmt of iron ore, this would equate to a penalty of approximately $8/t. If, on the other hand, the merchant delivers 66% Fe content, the additional fee is not equivalent to the two-times penalty multiple but closer to 1.75 times the Fe unit cost (i.e. approximately $14/t premium payable by the mill to the miner). The exact premium or penalty will usually be defined in the contract.

**Marketing**

The UF iron ore concentrate that will be produced by Capstone is not suitable for conventional sinter plants because it is too fine. However, because of the high Blaine Index, CRU reports that the material should be acceptable for a conventional pelletizing plant. The quality of Capstone material (high silica and copper content)
would not be suitable for any of the merchant pellet plants outside of China, nor the new direct reduction plants planned for the Middle East, leaving China as the most likely alternative.

Since pellet plants all have different and variable quality requirements and impurity flexibility, Capstone will need to short-list a number of potential plants that can comfortably process the iron ore concentrate that will be produced by the Project, prior to beginning a campaign to contract a suitable long-term offtaker. CRU notes that there is likely to be strong competition for space at these pellet plants from other mines producing UF iron ore concentrate.

CRU recommended that Capstone engages in dialogue at an early stage of Project development with a number of suitable parties in order to finalize contracts or have meaningful MOUs in place. Capstone will need to focus their marketing efforts towards the new or planned pellet plants in China, or build their own pellet plant in Chile near the port from which the material will be shipped.

Although more difficult, because it is a speciality product, Capstone should also explore the option of contracting long term with one or more traders who may be able to sell the material to a specific end user on a long-term contract.

19.2 Commodity Price Projections

19.2.1 Copper

Using the CRU predictions, on average from 2016 through to, and including, 2025 (i.e. 10 years) prices are expected to average $3.13 per pound of copper. CTAG agreed with CRU that this is a realistic price to be used as an average over this period. However, for the purposes of the economic analysis in Section 22, a base case price of $2.85/lb Cu was used.

It is CTAG’s view that TC/RCs of $75 and $0.075 will be the norm over the next 10 years (averaged over the period). This forecast translates to a combined TC/RC for Capstone of approximately $0.19/lb. This figure was used in the economic analysis.

19.2.2 Iron

Lower price trends can be expected in the coming years as new projects are brought on line. CRU’s latest report estimates that prices for 62% Fe content sinter fines (IODEX) CFR Qingdao delivery (deemed the standard product for CFR China delivery) can be expected to decline on average over the next 10 years, reaching a long term price of approximately $114 per wmt by 2024.
The iron ore concentrate to be produced by Capstone is not suitable for normal sinter fines delivery due to its fine size, but as China puts a value on the actual Fe content in the ore, there should be a relationship with the benchmark index projections. CRU’s methodology on this is to assume that a pellet plant would economically value the UF iron ore concentrate by calculating the cost of fuel (to operate the pellet plant) along with the cost of removing impurities (this is implied in the discount made to steel mills if this is a custom pellet plant), the grade of the Fe in the concentrate, and any handling costs or weight losses associated with drying and dusting from the UF material.

CRU’s Value In Use method is in CTAG’s opinion, although subjective, a fair and reasonable method of measuring the value of the material in a market which is very specific (pellet plant feed tends to be direct with individual mines only) and therefore lacks a benchmark index.

CTAG is of the same opinion as CRU that prices will decline over the next four to five years to long-term trend rates, likely to be around the $114 per wmt for 62% Fe, CFR Qingdao, China basis, or equivalent to $83 to $88 FOB North Chile (when adjusted for freight). A base case price of $85 per wmt was used in the financial analysis.

19.3 Contracts

Kores is required to purchase 50% of the annual production of copper concentrate and iron ore concentrate, leaving Capstone to market and sell the remaining 50%. The Kores terms and conditions will reflect the Capstone terms negotiated independently in the market.

No contracts are currently in place for Capstone’s 50% of the production for either the copper or iron ore concentrates.

19.4 Comments on Section 19

In the opinion of the AMEC QPs, the marketing studies support that there is potential for the sale of the copper and iron concentrates from the Project as follows.

- Specifications provided to CTAG for the copper and magnetite concentrates are from the PFS-level metallurgical testwork and are not smelter specifications
- The Santo Domingo copper concentrate would generally be considered clean. Cl and F are safely under the limits, and if they are occasionally over the limit it is likely that only a nominal penalty would apply. CTAG notes that for trading companies specialising in blending various complex copper concentrates outside of China a clean concentrate such as that from Santo Domingo would be in high demand
• CTAG and CRU note that it is important that Capstone is ready to enter the market at an early stage to begin the process of finding a partner to buy its ultra-fine iron ore concentrate. CTAG and CRU note that a long term, larger volume guarantee of pellet feed should be an attractive proposal for the big Chinese mills

• Kores is required to purchase 50% of the annual production of copper concentrate and iron ore concentrate under terms and conditions that will reflect terms negotiated independently in the market by Capstone

• No contracts are currently in place for Capstone’s 50% of the production for either the copper or iron ore concentrates.
20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Baseline Studies

20.1.1 Air Quality

Air quality measurements were taken from six monitoring stations; in the proposed mine site area, along the pipeline and at the proposed port. The air quality was characterized on the basis of inhalable particulate material (PM10), inhalable fine particulate material (PM2.5) and gases (CO, SO₂, NO, NOₓ and NO₂). Measurements of sedimentable particulate material (SPM) were conducted and the chemical composition of the particulate matter (PM10, PM2.5 and sedimentable particulate material (SPM)) was analyzed.

The Diego de Almagro area had higher particulate readings than did the proposed mine site. This was attributed to the urban activity at Diego de Almagro, the presence of vacant properties with active erosion, the presence of environmental polluters, mining activity around the city, and its geographic position in a desert area, as opposed to the mine site location which is in a rural area where the only major emission source is associated with the transit of vehicles on unpaved roads.

The area planned for the construction of the pipeline runs through rural areas (with the exception of El Salado) and the main source of manmade emissions are from the movement of vehicles and some from nearby mines.

Elevated particulates in the proposed port area was attributed to the transit of vehicles on nearby roads, wind erosion from vacant lots and dumps with material being removed due to winds in this area, and marine spray.

Dust control is planned and will consist of a combination of covering or enclosing conveyors and other machinery, speed limiting traffic on unpaved roads, and wetting of traffic areas.

20.1.2 Noise

For noise and vibration characterization, three field campaigns were carried out; the first campaign from 20 to 23 February 2012, the second campaign between 26 and 28 June 2012, and the third campaign from 15 to 18 April 2013.

The main sources of noise are from the interaction between the wind and the foliage in the area, vehicular traffic on nearby roads, wild birds, dogs and the breaking of the
waves for points located near the shoreline. Most of the points showed a decrease in values at night due to less human activity such as vehicular traffic and community noise. Near the coast, variations were not observed because the main source of noise is from wave activity.

Noise abatement will be practiced and will include:

- Diversion of trucks outside the urban area of Diego de Almagro
- Installation of acoustic panels at sensitive points during the concentrate pipeline and sea water pipeline construction
- Restriction of noise-generating activities at night during construction of the concentrate pipeline and sea water pipeline.

20.1.3 Natural Hazards

An evaluation of natural hazards was undertaken, based on the geology and geomorphology of the planned mine site area, pipeline route and port location.

The geological and structural conditions and the historical background of the natural phenomena indicate that a seismic event may result in rock fall or detachment of blocks, increasing in volume depending on the magnitude of the earthquake. During the detailed engineering for the Project, critical facilities at the mine and the port will be designed to ensure human safety and continued operations after a design earthquake event. However, some distortion of, and disruption to, non-critical facilities can be expected from seismic events.

Tsunami mitigation measures for the port include offshore facilities designed to minimize wave damage and allow for continued operation after a design event. Critical onshore facilities have been located outside the tsunami wave and water zones. Disruption from a tsunami should be considered as mainly site clean-up and potential ship arrival disruptions.

In the study area, seismic activity and road cuts in rock with a high degree of fracturing or alteration are the main factors that could trigger rock falls or landslides that might damage infrastructure or cause occasional blockages of roads. The sites where this danger was identified are in an area of the Sierra Santo Domingo, the Quebrada Guamanga, the Quebrada Flamenco and along the coast within the study area.

Aerial photographs show morphological evidence of sporadic phenomena of “avenues” (mass flows caused by sporadic episodes of major precipitation) in some of the quebradas that drain to the El Salado and Flamenco. These cut the older land forms
and show dragging of sediment by episodes of intense rainfall. There are small quebradas that drain the water from sporadic storms that fall on the Coastal Mountains which in abnormally rainy years cause floods that have damaged roads.

20.1.4 Soils

Soils were analyzed to give a site description, general characteristics of the soils and a morphological description of the sections analyzed. Fourteen soil mapping units were identified which differed depending on the geomorphological units, vegetation units and land description. In general, soils in the study area are moderately deep to deep and generally porous.

In the study area there is only soil usage capacity classification; this is a Class VIII soil which is classed as “usable for construction purposes”. Class VIII soils are soils which are considered to have no value for agricultural, livestock or forestry purposes.

20.1.5 Hydrology/Hydrogeology

The analysis of the water level records for the mine and plant area indicated that there is no seasonal variation in the ground water levels. There was also no evidence of inter-annual declines or rises in the level of groundwater in the period from August 2007 to April 2013. Water levels were found to vary from 1,041 masl to 772 masl approximately at each well. The isodepth range resulting from the study varied between a few metres and as much as 120 m.

Two main drainage directions have been identified:

- From southeast to northwest in the plains area northeast of the Santo Domingo Sierra
- From east to west in the area of the El Salado River valley.

The hydraulic gradient average in the plain to the northeast of the Santo Domingo Sierra is approximately 0.021. Along the El Salado River valley, between the northwestern limit of the mine–plant area and Diego de Almagro, the average hydraulic gradient is approximately 0.025.

20.1.6 Fauna

The presence of 68 species of fauna were found in the study area, 29 of which were found in the area planned for the mine and plant, 23 in the pipeline corridor, 37 in the port area and 20 in the electrical transmission line area. Of the total species recorded, 13 are in a conservation category, 10 are protected by the Chilean Hunting Law, two by the classification of species list, and one is fully protected.
The Project must deal with the possibility of affecting species under conservation categories by implementing the following measures:

- Rescue and re-location plan for low-mobility species prior to starting works
- Staff training on protection of wildlife
- Hunting prohibition within construction areas
- Prohibition on owning and/or caring for domestic animals on all sites.

20.1.7 Flora

The area of study for flora and terrestrial vegetation also covered the four major areas planned for construction activities.

In general, most of the mine–plant area is classified as a denuded zone lacking in vegetation and flora. Vegetation, if there is any, appears in the bottom of creeks, gullies, depressions or alluvial cones. A total of 38 species of vascular plants were identified; of these, 19 are grasses, 15 are shrubs and four are trees. No endemic species confined only to the Atacama Region were detected in the mine–plant area and only one species classified as vulnerable was noted.

In general terms, most of the Route 17 by-pass area is also classified as a denuded zone. A total of 25 species of vascular plants were observed, of which 11 were shrubs, four were tree and one was a succulent. Two species are classified as vulnerable.

The Chañarcito area has two clearly distinguishable areas, the first covers almost the entire studied surface and is mainly classified as denuded, the second shows effects of human habitation. A total of 22 species of vascular plants were detected, and included 22 species, of which 11 were trees, eight were grasses and three were shrubs. No restricted endemic species were found, but two vulnerable species were observed.

Between the mine–plant area where the pipeline starts and the intersection of Route C-205 with Route C-225 the area is mainly classified as a denuded zone. From the intersection of Route C-205 with Route C-225 to the end of the pipeline route in Obispito the flora is composed of homogeneous units of vegetation, corresponding mainly to thickets less than 1 m high and sparse coverage (50% to 75%), meadow formed mainly by annual herbaceous species can be locally developed. A total of 107 species of vascular plants were noted, of which two were trees, 48 are shrubs, 45 are grasses and 12 are succulents. Fifteen species belong to conservation categories with classifications that range from critically endangered (_Eriosyce sociabilis_) to minor concern.
In general terms, most of the El Salado by-pass area is a denuded zone, with 25 species of vascular plants detected. None of the native species are cited in conservation categories in the official lists.

Vegetation in the area planned for the electrical transmission line occurs as thickets less than 1 m high with coverage that can be up to 50%. Meadows are not very common. A total of 46 species of vascular plants were recorded, of which 25 are shrubs, 16 are grasses and five are succulents. Five species are in conservation categories according to official lists, and range from near-threatened to minor concern.

For species under conservation categories the Project will implement the following mitigation measures:

- Micro-routing and demarcation of flora in conservation categories
- Rescue and transplanting of species of cacti in conservation categories
- Reproduction of shrubs and grass species under conservation categories
- Staff training on protection of flora and vegetation
- Installation of informative signs in relation to protection of the flora and vegetation.

20.1.8 Port Area

Setting

The summer months are very dry, and precipitation is concentrated in the winter months with the highest values in July. The atmospheric circulation pattern is mainly from the west, with a clear predominance of winds from the west–southwest (17.9%), west (8.8%) and west–northwest (15.2%). The wind intensity is generally low (6.5 m/s with 89% of incidence).

Tides

Tides in Caldera and surroundings have a semi-diurnal mixed regime, with two high tides and two low waters in a lunar day with diurnal inequality. This inequality affects the high tide more than the low tide, with a monthly average of 0.29 m. The average tidal amplitude is 1.41 m.

The eulerian results show a modulated movement pattern of the tide and the winds, highly rotational, with flow and backflow, moving into the bay to the east and out of the bay to the west.
The main centres of upwelling (rising of deep water towards the upper strata of the ocean and the removal of the surface water to the west; associated with higher nutrient contents) in the Atacama Region are located south of Caldera. The thermal structure shows a decreasing vertical pattern with depth, with no indication of a pronounced thermocline.

**Chemical Environment**

The concentrations of chemical in the water obtained in the study area do not show any evidence of contamination for pH, transparency and fecal count. No contamination of the water column was noted. The organic matter content indicates the water is of regular quality for this parameter; the oxygen demand levels could be low due to the bacterial decomposition of organic matter. Temperature and salinity data were within normal limits.

The total content of sulphates, Kjedahl total nitrogen and phosphorus reported in Obispito Bay are within the ranges reported for other locations in the north of Chile. However, the concentration of ammonia nitrogen in several samples was higher than the guidelines for the protection of aquatic communities; this may have toxic effects on the biota. Suspended solids, fluorides and detergents detected in Obispito Bay were in low concentrations and below the levels of toxic effects on biota. The concentrations of chlorides and boron are consistent with other values reported for sea water. Traces metals (Zn, Pb and Cu) recorded in some areas showed levels that exceed international guidelines for the protection of the biota. However, overall the concentration of all elements means that the body of water can be classified as suitable for aquaculture activities.

The study area showed relatively low trace metal concentrations that were below the threshold value for internationally-accepted environmental quality criteria for the observation of toxic effects on the biota; these data indicate good quality sediments.

**Marine Environment**

Around 50 species of seaweed, fish, molluscs and crustaceans are harvested in the area. Harvests consists of brown seaweed (55%); fish (39%), molluscs (5.2%), crustaceans (0.2%) and other species (0.6%).

A total of 19 fish species were recorded in the area of study; two species with pelagic habits and 17 with benthic habits. The low abundance of commercial species of benthic habits may be due to the effects of fishermen and divers in the area.
The zooplankton community consists mainly of chitinous zooplankton; herbivorous copepods of wide distribution and abundance in the Humboldt Current system, representing 75% of the community. The spatial distribution of zooplankton showed an area of greater abundance in the south, and lower abundance in the north.

The phytoplankton community was dominated by the diatom group. There were no silicoflagellates.

The intertidal communities are dominated by shellfish such as *Excirolana hirsuticauda* and *Emerita analoga* (sea flea), both recorded in high abundance in the areas south and north of the beach in Obispito Bay. The beach condition means that this is a not very diverse community. Both species are common in beach systems.

The hard bottom intertidal community of the study area was represented by 29 species of algae and invertebrates and two vertebrate species recorded in intertidal pools.

The vegetation in the port area corresponds to thickets less than 1 m high with coverage that can reach 75%. There were 35 species of vascular plants registered during surveys; of which 22 are shrubs, 10 are grasses, and three are succulents. Three species are included in conservation categories and are classified either as vulnerable or minor concern.

### 20.1.9 Human Environment

#### Setting

The Communities of Diego de Almagro and Chañaral have seen declines in various sources of mining work due to several related phenomena; depletion of veins and ore deposits, falls in the price of some metals; decrease of the grades of the ore mined by small scale miners, local and global economic crises which affected investment in mining projects slowing exploration and new development.

Factors such as dryness and lack of water resources prevent agricultural activities and there is also a lack of industrial and tertiary development. There is very limited availability of technical and professional skills which would support the development of non-mining activities.

#### Surveys

A total of 262 archaeological sites and 18 animitas (small memorials to accident victims) were recognized, and were mainly assigned as pre-Hispanic. A number of sites were classified as indeterminate sites, meaning that the site was not ascribable to any specific period.
Twelve key site types were categorized:

- Rock accumulations
- Eaves
- Structures
- Stone concentrations
- Carving event
- Ceramic concentrations
- Isolated findings
- Shell material
- Cave painting
- Tracks and trails
- Mining remains, pits
- Areas (concentrations or structures) with current or undetermined occupation such as La Aguada and Chañarcito.

The information indicates that most of the sites found such as trails, cart tracks and footprints are evidence of the activities outlined in the zone between the puna and the Atacama Desert. The sites were determined by the landscape and it was possible to find large amount of structures, rectangular and circular, as well as windbreaks in areas of plains, to provide refuge during passage through this area.

The eaves and the shell material were predominantly pre-Hispanic sites. One site was also found with rock concentrations, another with signs of carving, one with ceramic concentrations and a fourth with a cave painting. A large percentage of the sites are in a good state of conservation, because they are far from main or secondary roads and the inhabitants of the area take care of them (as can be seen for the animitas (shrines)).

20.1.10 Palaeontology

There are four geological formations in the study area hosting various fossils.

Based on the distribution of the geological-paleontological potential in these lithologies, and results obtained from a study of remote images, a number of sites were selected as prospective areas of paleontological interest: outcrops of the Chañarcillo Group in
the proposed mine–plant area and the outcrops of Caldera strata along the planned pipeline route.

The surveys confirmed that outcrops with Cretaceous fossils exist in the proposed mine–plant area, with fossils of sponges, gastropods, bivalves, cephalopods, echinoderms, serpulids and traces of invertebrates indentified, ranging from 1 to 16 taxa for valanginous invertebrates in the Sierra Santo Domingo.

Outcrops with Pleistocene-Holocene fossils were identified along the planned pipeline route near the coast, with numerous taxa of marine invertebrates, including gastropods, bivalves and barnacles of Quaternary age reported.

Capstone plans to initiate a management plan to monitor the known outcrops and to liaise with the National Monuments Council if sites must be disturbed during Project construction and operation.

20.1.11 Visual Landscape

Surveys were undertaken to define visual landscape units within the planned mine-plant area, along the pipeline route, the port facility and the proposed transmission line route. The units associated with the coast have greater visual quality and visual fragility; however, moving to the interior valley results in decreased visual values.

20.2 Environmental Issues

20.2.1 Water

The only active superficial riverbed is the El Salado River which has been altered and impacted through anthropogenic activities related to historical mining. There are no exploitable groundwater courses.

The potential impacts on water resources are:

- Alteration of the flow and drainage patterns of surface water
- Underground water flow alteration
- Underground water quality alteration.

No impacts to water resources are anticipated because the project will use desalinated sea water for the mining process and, in turn, no infiltration is expected from waste rock deposits or thickened tailings deposit to the scarce groundwater resources in the zone.
During the operation phase it is expected that impacts on water resources will be caused by the following activities identified as impact sources:

- Exploitation of the Santo Domingo and Iris Norte pits
- Storage of material in the WRFs
- Disposal of tailings
- Management of contact and non-contact waters.

During the closing phase it is expected that impacts on water resources will be caused by the following activities identified as impact sources:

- Closure and abandonment of the Santo Domingo and Iris Norte pits
- Closure and abandonment of the waste rock facilities
- Closure and abandonment of the TSF.

Planned mitigation measures for the key water impacts comprise:

- Construction of facilities to divert non-contact water
- Construction of concrete around water course crossings
- Construction of the TSF wall with HDPE waterproof liner and leak management system
- Use of thickened tailings technology
- Process that maximizes the recovery and reuse of water (minimal discharge of effluents from the concentrator except for tailings to the TSF)
- Reuse of effluent from the sewage treatment plant and the mine desalination plant.

20.2.2 Air Quality

The Diego de Almagro weather station indicates high levels of PM10 and PM2.5 Particulate Material (higher than the values established as the annual average in D.S. 59/1998 Primary Air Quality Standard for PM10 and D.S. 12/2011 Primary Standard of Air Quality for PM2.5). These high readings can be explained by the location of the station close to a major urban area and its geographical position. For the project activities during construction and operation, it was determined that the indicators of the impact of this component are: breathable particulate material (PM10), breathable fine particulate (PM2.5), gases, and sedimentable particulate material (SPM).
During the construction phase the main particulate material and gas emissions will be generated as a result of the construction and transportation activities in the mine-plant area, pipeline area and port area.

During the operations phase the main particulate material and gas emissions in the mine-plant area will be associated with the exploitation of the Santo Domingo and Iris Norte pits, the transport of material from the pits to the crushing plant and from the pits to the waste dumps. For the port area the main emissions of particulate matter and gas are associated with wind erosion of the magnetite concentrate stockpile and the activities of transfer of copper and magnetite concentrates.

Based on the emission sources identified above for each project area during construction and operations, the impacts will be increases in the levels of PM10, PM2.5 and gases in areas with the presence of receptors and also an increase in SPM.

Air quality measured during three years of monitoring, averaged 47 μg/m3N for PM10 in Diego de Almagro; the limit in the Chilean regulations is 50 μg/m3N. It is forecast that the PM10 could reach 52 μg/m3N the tenth year of operation. The Project has prepared a plan to mitigate the impact of dust in Diego de Almagro. The mitigation plan is to pave some streets and to sweep the streets in Diego de Almagro to remove dust. This will ensure that the project will have a minimal impact on the PM10 concentration, thus not affecting the current quality of air in Diego de Almagro. This mitigation plan will also mitigate the levels of PM2.5 particles in Diego de Almagro.

Planned mitigation measures for the key air quality impacts include:

- Speed control on roads in active use
- Wetting of roads during construction
- Application of brine for stabilization of mine roads and internal roads in the mine-plant area
- Enclosed conveyor from crushing to the stockpile and material transfer points
- Sprinkler installation in primary crusher.
- Installation of wind breaks around the magnetite concentrate stockpile area
- Installation of sprinklers at the magnetite concentrate stockpile area
- Enclosed building with negative pressure for the copper concentrate stockpile
- Maintenance of vehicles and generators.
20.2.3 Marine Environment

During the construction phase, the main impacts generated by Project activities will be due to construction of the berthing structures and the installation of the sea water intake piping and brine discharge. This work will result in a temporary disruption of intertidal and subtidal benthic communities.

The operation of the port will cause two impacts. One related to discharge of brine and the area of dilution in the sea which may result in a deterioration of the quality of the sea water column. The other is intake of sea water which may result in an alteration of the water column and a local alteration of small zooplankton and phytoplankton of small size in the water intake area.

Regarding the marine environment, although the project includes seawater intake and brine discharge to the sea in the port area, the Project has considered in its engineering design, environmental criteria oriented to minimize the impacts on the environment. Additionally, the Project includes an environmental monitoring plan, to be executed during the operations phase, in order to validate the magnitude of the forecasted environmental impacts and verify the efficiency of environmental management measures proposed.

Planned mitigation measures for the marine environment will consist of:

- Installation of a diffuser at the brine discharge point
- Sea water intake situated 20 m deep
- Brine discharge located 16 m above the sea bed
- Design of a low speed suction hood
- Training of staff on protection of the marine environment
- Prohibition on hunting of marine fauna

20.2.4 Human Environment

The major impacts on the Human Environment component that are expected to be generated by the Project include:

- Alteration of lifestyle due to by-pass on Route C-17
- Temporary effect on the identity and cultural customs
- Decrease of goods and service supplies
- Increase in the demand for health services
• Increase in social problems due to the arrival of workers.
• Temporary effect on residents’ life system in the coastal towns dedicated to the seaweed harvesting.

Planned mitigation measures for the human environment will comprise:

• Information and coordination of project actions that might interfere with the normal community activities
• Implementation of an office and information system for management of complaints and claims
• Installation of a clinic in the permanent camp and/or mine area
• Installation of potable water plant at site
• Installation of sewage treatment plant at site
• Training of workers to respect neighbouring communities
• Diversion of trucks around the urban area of Diego de Almagro
• Transfer of workers from outside of the area between the Project and transfer points (bus terminal, airport)
• Speed control on roads
• Safe driver training for drivers and contractors
• Fostering respect for cultural diversity
• Grant funds for local projects entrepreneurship
• Infrastructure support productive activity of seaweed independent Sector Punta Roca Blanca
• Contribution of 10 L/s drinking water for the community of Diego de Almagro.

In addition, the following are planned:

• Public roads outside the main population centres near the mine site will be used
• A by-pass of Route C-13 will be built around Diego de Almagro to reduce congestion in the town.
• Near the village of El Salado the Project will use a planned by-pass to be built by the Roads Department (MOP).
20.3 Closure Plan

20.3.1 Regulatory Considerations

Five main areas of legislation cover the closure of the proposed operations:

- D.S. 132/2004 - Mining Safety Regulation N° 72
- Law 20.551/2011 - Regulates the Closure of Mine Sites and Mining Facilities
- D.S. 41/2012 - Regulation for the Closure Law for Mine Sites and Mining Facilities
- D.S. 186/2008 - Regulation about Sanitary Conditions and Basic Safety in Sanitary Landfills.

Works and facilities that must be included in the closure plan are identified based on D.S.41/2012 and include the proposed mining operations, plant site, port, pipelines and transmission lines.

20.3.2 Risk Evaluation

The regulations require a risk evaluation to establish and analyze the risks in order to estimate the post-closure conditions. The closure criteria are defined based on the evaluation of risks and technical requirements set in the regulations of Law 20.511 and international leading practices. Evaluation of the risks as confined to health, safety, community/culture and environment and did not consider operational or financial risks.

To facilitate the risk analysis, the facilities were classified as industrial, ancillary and other. The industrial facilities include the process plant and process tanks, pipelines and port facilities. The ancillary facilities comprise buildings such as mine offices and warehouses. The TSF, waste rock dumps and open pits will remain after closure and form part of the “other” facilities.

The risk evaluation was semi-quantitative in nature and based on a failure modes and effects analysis (FMEA) methodology. The evaluation carried out was limited to key facilities and events with high probability of occurrence during operation. The risk level was determined by determining the severity of the consequence and the probability of occurrence.

In general, the highest risks are related to the stability of the remaining works, the presence of industrial facilities and the presence of hazardous substances and
materials. No acid generation is expected from the materials remaining from the mine
operation because they show a low potential for acid generation and the dry climate
conditions will not produce enough water to generate drainage through the waste
dumps to mobilize any acid solutions. The low phreatic level means that this is not
intercepted by the Santo Domingo pit and for the Iris Norte pit, the low phreatic level,
and the poor hydraulic conditions of the local aquitard, means that the risk of acid
drainage is low.

20.3.3 Closure Action Summary

The provisional closure plan is based on nominal closure dates which are in turn
based on provisional assumptions of permitting and construction schedules. Until a
decision is made by Capstone as to project construction, these dates should be
considered as illustrative only. The closure phase would begin in 2036 and according
to the planning of closure measures and actions this phase will extend for three years
to 2038. The post-closure phase will start in 2039; this phase comprises monitoring
and inspection and will extend for three years to 2041. An exception is the monitoring
of the landfill which could be up to 20 years to comply with the Sanitary Landfill

Capstone will evaluate during future detailed studies if there are opportunities to
selectively undertake progressive closure activities prior to 2036. At this level of
closure planning the main risks are considered to be:

- Modifications and updates of the approved closure plan are not made as aspects
  of the Project change during the Project life cycle
- Closure actions and measures committed to in the closure plan are not executed
  or are badly executed
- Not meeting the commitments made in the closure plan may result in the following:
  - The sectorial authority may take charge of the necessary actions to guarantee
    the complete application of the closure plan. Non-fulfillment of the technical and
    legal requirements for the closure of a mine site is regulated by Law 20.551, 28
    October 2011, Ministry of Mines. If not in compliance, the Ministry of Mines can
    take over the closure plan implementation.
  - Sanctions will be applied to a mining company in accordance with Title X of
    Law 20.551/2011 if not in compliance
  - Not complying with the general planning of the works and activities for the
    Project, because of non-compliance with the schedule of actions and measures
    implementation of the project Closure Plan.
It is expected that the closure plan will be revised when the environmental approval (RCA) has been issued, so as to include the closure measures and actions that will be agreed during the process of environmental impact evaluation. During such future updates, a review of what infrastructure can be kept for future use rather than dismantled will also be undertaken.

Closure actions and measures for each of the sites and facilities for the mine-plant, pipeline, port and electrical transmission line areas are summarized in Table 20-1 to Table 20-4.

20.4 Permitting

The Project will include the following process related works facilities:

- Mine–plant area: open pits, waste dumps and tailings facility, crushing, copper concentrate production, magnetite separation and copper concentrate filtration; camp, services support (guard houses, lunchrooms, first aid facilities, temporary and permanent waste storage), power distribution, roads (internal, mining and access) and modifications of public roads and power lines
- Pipeline area: a magnetite concentrate pipeline and sea water pipeline
- Port area: magnetite concentrate filtration, storage of copper and magnetite concentrates, conveying and ship loading of concentrates
- Power transmission line area: transmission lines to supply power to the port.

To date about 140 works and installations have been identified, distributed between the four operations areas.

The number of permits required for all facilities is estimated to be about 750 in total (Table 20-5). Permits that have been classed as critical to ensure the timely construction and start-up of the Project are summarized in Table 20-6. For operations, 15 sectorial environmental permits (Permisos Ambientales Sectoriales or PAS) were identified, which are required by the Regulations of the Environmental Impact Evaluation System (SEIA). These permits must be submitted during the Project environmental evaluation process.

The General Permitting Schedule is based on the date of submission of the EIA to the SEIA for approval and is 30 October 2013. It is assumed that the EIA approval and issue of the Resolución de Calificación Ambiental (Environmental Qualification Resolution, RCA) will take 16 months. This sets the timing for the application and granting of critical permits and sectorial permits required for the Project.
Table 20-1: Closure Aspects and Measures, Mine and Plant Area

<table>
<thead>
<tr>
<th>Work Site and Main Facilities</th>
<th>Measure</th>
</tr>
</thead>
<tbody>
<tr>
<td>Open Pits</td>
<td>Removal of mining equipment and reusable materials; dismantling of facilities; demolition and covering of foundations; closure of access roads; slope stabilization; signage installation</td>
</tr>
<tr>
<td>Waste Rock Facilities</td>
<td>Closure of access roads; slope stabilization; signage installation</td>
</tr>
<tr>
<td>Primary Crushing and Coarse Ore Stockpile</td>
<td>Inventory of the area and removal of reusable materials; dismantling/demolition of structures/foundations and dismantling and washing of equipment; signage installation</td>
</tr>
<tr>
<td>Concentrator</td>
<td>Removal or sale of remaining reagents; inventory of the area and removal of reusable materials; cleaning of equipment and dismantling facilities; cleaning of floors and sumps before demolition; demolition and covering of foundations, re-profiling of the area; sampling of soil in surrounding areas</td>
</tr>
<tr>
<td>Tailings Storage Facility</td>
<td>Inventory of the area and removal of reusable materials; dismantling/demolition of structures and dismantling and cleaning of equipment; demolition and covering of foundations; commission the rain water evacuation system into operation; covering of ponds and slopes; stabilization of slopes and crown; reclaim water system; construction of emergency spillway; closure of access roads to the TSF; signage installation</td>
</tr>
<tr>
<td>Tailings Pumping</td>
<td>Removal of electrical supply and related infrastructure; dismantling and removal of facilities and equipment; signage installation</td>
</tr>
<tr>
<td>Mining Equipment Maintenance Area</td>
<td>Removal of electrical supply and related infrastructure; cleaning and washing of equipment; dismantling of facilities; hydrocarbon and contaminated soils management</td>
</tr>
<tr>
<td>Truck Wash Area</td>
<td>Removal of electrical supply and related infrastructure; cleaning and washing of equipment; dismantling of facilities; liquid industrial waste and contaminated soil management</td>
</tr>
<tr>
<td>Fuel Supply</td>
<td>Removal of electrical supply and related infrastructure; removal of fuel tanks; dismantling of facilities; hydrocarbon management</td>
</tr>
<tr>
<td>Laboratories</td>
<td>Removal of electrical supply and related infrastructure; demolition of concrete structures and dismantling of facilities; equipment removal; hazardous materials management</td>
</tr>
<tr>
<td>Services Building (change room, laboratories, offices)</td>
<td>Removal of electrical supply and related infrastructure; dismantling of facilities; equipment removal; hazardous materials management; signage installation</td>
</tr>
<tr>
<td>Camp</td>
<td>Removal of electrical supply and related infrastructure; demolition of concrete structures and dismantling of facilities; signage installation</td>
</tr>
<tr>
<td>Electrical Transmission Lines and Electrical Substation</td>
<td>Removal of electrical supply and related infrastructure; demolition of concrete structures and dismantling of facilities; contaminated soils management</td>
</tr>
<tr>
<td>Mining Roads and Internal Roads</td>
<td>Recovery of original drainage; construction of berms</td>
</tr>
</tbody>
</table>
### Work Site and Main Facilities

<table>
<thead>
<tr>
<th>Water Supply System</th>
<th>Measure</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ponds</td>
<td>Inventory of the area and removal of reusable materials; dismantling/demolition of structures, foundations, and equipment; cleaning of ponds and surrounding area; liner removal; re-profiling of land</td>
</tr>
<tr>
<td>Water Treatment Plant</td>
<td>Removal of electrical supply and related infrastructure; removal or sale of supplies or reagents; cleaning of plant structure and equipment; structure and equipment inventory and dismantling; demolition and covering of the foundations</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Waste Management Facilities</th>
<th>Measure</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sanitary Landfill</td>
<td>Removal of electrical supply and related infrastructure; dismantling of facilities; signage installation</td>
</tr>
<tr>
<td>Waste Storage Yard</td>
<td>Dismantling of facilities; demolition of concrete structures; hazardous waste management, signage installation</td>
</tr>
<tr>
<td>Sewage Treatment Plants</td>
<td>Removal of electrical supply and related infrastructure; removal of equipment and dismantling of facilities; demolition of concrete structures; signage installation</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Explosives Magazine</th>
<th>Measure</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Closure of explosives storage; dismantling of facilities; removal of concrete; contaminated soils management</td>
</tr>
</tbody>
</table>

### Table 20-2: Closure Aspects and Measures, Pipelines

<table>
<thead>
<tr>
<th>Worksite and/or Facility</th>
<th>Measure</th>
</tr>
</thead>
<tbody>
<tr>
<td>Concentrate Transport System (STC)</td>
<td>Washing of pipes; installation of plugs and seals; dismantling of pump and valve stations; removal of foundations to ground level; filling of excavated areas and elimination of berms and walls; re-profiling of land</td>
</tr>
<tr>
<td>Sea Water Pumping System (SIAM)</td>
<td>Inventory of the area and removal of reusable materials; dismantling/demolition of structures and dismantling of equipment; cleaning of ponds and surrounding area; removal of magnetite concentrate or other materials from the emergency ponds; removal of the impermeable liner and other installations; demolition and covering of foundations; re-profiling of the area</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Emergency Ponds</th>
<th>Measure</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Inventory of the area and removal of reusable materials; dismantling/demolition of structures and dismantling of equipment; cleaning of ponds and surrounding area; removal of magnetite concentrate or other materials from the emergency ponds; removal of the impermeable liner and other installations; demolition and covering of foundations; re-profiling of the area</td>
</tr>
</tbody>
</table>
### Table 20-3: Closure Aspects and Measures, Port

<table>
<thead>
<tr>
<th>Worksite and/or Facility</th>
<th>Measure</th>
</tr>
</thead>
<tbody>
<tr>
<td>Magnetite Concentrate Filter Plant – Storage.</td>
<td>Inventory of the area and removal of reusable materials; removal or sale of remaining reagents; removal of electrical supply and related infrastructure; cleaning of equipment and facilities; sampling of soil in surrounding areas; cleaning of floors and sumps before demolition; demolition and covering of foundations; dismantling of facilities; re-profiling of the area</td>
</tr>
<tr>
<td>Conveyors</td>
<td>Removal of electrical supply and related infrastructure; cleaning of belts; dismantling of facilities</td>
</tr>
<tr>
<td>Shipping Dock</td>
<td>Removal of electrical supply and related infrastructure; cleaning of structures and equipment and the shipping facilities; dismantling of facilities; removal of foundations and concrete</td>
</tr>
<tr>
<td>Desalination Plant</td>
<td>Removal or sale of supplies or reagents; cleaning of the structures and equipment; dismantling and demolition; demolition and covering of foundations</td>
</tr>
<tr>
<td>Ponds</td>
<td>Inventory of the area and removal of materials; dismantling/demolition of structures; cleaning of ponds and surrounding area; demolition and covering of foundations</td>
</tr>
</tbody>
</table>

### Table 20-4: Closure Aspects and Measures, Transmission Lines

<table>
<thead>
<tr>
<th>Worksite and/or Facility</th>
<th>Measure</th>
</tr>
</thead>
<tbody>
<tr>
<td>Transmission Lines</td>
<td>Removal of electrical supply and related infrastructure; dismantling of facilities; contaminated soils management</td>
</tr>
</tbody>
</table>
Table 20-5: Total Permits Required

<table>
<thead>
<tr>
<th>Operations Area</th>
<th>Number of Works and Facilities</th>
<th>Estimated Number of Permits Required</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mine-Plant</td>
<td>76</td>
<td>462</td>
</tr>
<tr>
<td>Pipelines</td>
<td>14</td>
<td>119</td>
</tr>
<tr>
<td>Port</td>
<td>34</td>
<td>144</td>
</tr>
<tr>
<td>Power Transmission Lines</td>
<td>9</td>
<td>27</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>133</strong></td>
<td><strong>752</strong></td>
</tr>
</tbody>
</table>

Note: Total number of permits required includes general permits and temporary construction facility permits

Table 20-6: Critical Permits

- EIA
- Santo Domingo Port Maritime Concession
- DOP Permit
- Public Road Route C-17 By-Pass
- Permit for Construction of Walls over 5 m height or more than 50,000 m$^3$ of fill
- Tailings Facility Approval
- Authorization for Works in a Water Course (DGA Art.294 letter c))
- Authorization for Works Modifying a Water Course
- Sectorial Permit to Discharge into National Waters
- Health Authority Approval for Brine Discharge
- Exploitation Method Authorization (Open Pit)
- Authorization for a Stockpile or Waste Dump
- Construction Permit
- Final Works Reception
- Process Plant Operating Permit

A number of areas where the Project design will impact existing infrastructure were noted. These include:

- Tailings Facility: Mainstream’s solar (photovoltaic) plant, Entel’s fibre optic line, and Route C-17
- Mine–Plant Area - GDF Suez’s Mejillones-Cardones transmission line, the Chañaral–Diego de Almagro transmission line and the Cardones–Diego de Almagro transmission line

Electrical and fibre optic lines will be relocated on a strip of land provided by Capstone. The planned Diego de Almagro Solar Park would overlap with an area of the tailings facility of approximately 18.9 ha, which are legally reserved for Capstone’s use.
electrical transmission lines and the ditch surrounding the solar park are also in part on land reserved for the Santo Domingo Project. Capstone is currently working with Mainstream on a formal agreement to eliminate interferences between the two projects.

20.4.1 Permitting Risks

At the level of general permits the main strategic risk is not having a plan for permitting for the project that is known and integrated into the master schedule for execution of the works. Not having a permitting plan or failure to follow the permitting plan could have undesirable effects, such as:

- Delays in obtaining the environmental approval (RCA) for the project
- Not having the technical and legal information to prepare each of the files to be submitted for processing
- Delays in obtaining any of the permits identified as critical to the project
- Delays in the submission of permit files to the authorities.

Other risks for general permits include:

- Changes or modifications of the project configuration which were not considered in the 2014 Feasibility Study for which new permits may be required
- Modifications to the existing legal framework which could lead to new authorizations and/or permits not considered in the 2014 Feasibility Study.

The strategic risk of not having a plan for preparation and submission of permits prior to the start of the execution of works would affect the following permits:

- Request for the maritime concession (new or modified)
- Authorization for building the by-passes for public roads Route C-17 and Route C-17
- Interferences of overhead lines with the Iris Norte pit development.

20.5 Considerations of Social and Community Impacts

20.5.1 Area of Influence

The area of influence of the Project includes the Provinces of Chañaral and Copiapó in the Atacama Region, and particularly the communities and towns of Diego de Almagro, Chañaral and Caldera:
- In the mine site and plant site direct area of influence are the town of Diego de Almagro and the village of Inca del Oro in the Community of Diego de Almagro; because of their proximity to the mine-plant area

- In the Community of Chañaral the village of El Salado, settlements close to Route 225 and the areas of Flamenco and Torres del Inca are included; because of their proximity to the proposed Project pipelines and the roads which will be used by the Project

- In the Community of Caldera the coastal settlements close to Route 5 North extending from Las Lisas to Caleta Obispito are included; because of their proximity to the proposed pipelines and the port facilities.

### 20.5.2 Indigenous Groups

There are two major ethnic groups recognized by the Indigenous Law (19,253) in the region: the Colla and Diaguitas communities. The Diaguitas are the largest ethnic group in the Region (representing 3.5%), followed by the Colla (1.5%), and then the Aymaras (1.0%) and Mapuche (0.9%) communities.

In the Province of Chañaral the largest ethnic group is the Colla community (1.1%) which is an Atacama ethnic group, followed by the Mapuche (1.0%), a population that has migrated from the south. Members of these ethnic groups are scattered in the cities of El Salvador and Diego de Almagro, and other towns and rural areas of the province.

There are two organizations recognized by the Indigenous Law (N°19,253): the Indigenous Colla Community of Diego de Almagro (created in July 1995) with 37 families, most of whom live in Diego de Almagro and Copiapó. The other group is the Colla Geoxcultuxial Indigenous Community (created in November 2001) with 11 families, most of who live in Portal del Inca, El Salvador and Diego de Almagro. The communities do not own lands, but feel linked to and claim traditional territories in the areas of Inés Chica and La Encantada in the mountains near the Montandón border post.

A third Colla Community is planned to be recognized in Diego de Almagro; the members belong to the Colla lineage of the Jerónimo family and the group will be called the Quillaga Community.

There are no indigenous lands or territories of any kind being claimed in the Project area.
20.5.3 Stakeholder and Issue Identification

Semi-structured interviews with people in all the communities within the area of Project influence were conducted, and supplemented by background information provided by social sources in each community. Subsequently, a list of the major stakeholders was established for each community.

The local communities in Diego de Almagro and Chañaral have a degree of knowledge of mining and related activities as mining is part of the local identity and the main economic activity for both communities. The main concern of local communities is protection of water resources and air quality. The potential increase in pollution because of the development of the Project, especially from the tailings storage facility (TSF), for the long term safety of the dam and its location, and also how it may affect the town of Diego de Almagro were also expressed as concerns. These concerns were mitigated by subsequently moving the planned location of the TSF.

Another issue expressed by other stakeholders in the meetings conducted to date has been job opportunities for local residents during the construction and operation phases of the Project. There was also a desire expressed to set up a round table where the community and Capstone can prepare a defined program of social benefits (support to youth education, student scholarships, quotas for students in work practice, lost income, and training). The Colla Community of Diego de Almagro lands are not within the direct area of Project influence; however, Capstone will keep lines of communication open for possible approaches or inquiries from this community.

The small-scale fishing fleet based out of Chañaral is considered to be a key stakeholder in the planned port area due to community perceptions about potential loss of income, and potential impacts on seafood extraction activities. An additional coastal community issue was related to the environmental assessment for the sea water intake and brine return into the sea; with the communities requesting that alternatives be examined for brine discharge.

Community issues identified during these stakeholder and community meetings include:

- Job opportunities for local residents during the construction and operation phases of the Project
- Decreased quality of life of the inhabitants of Diego de Almagro, due to increased demand for housing by the arrival of workers linked to the Project.
- Potential increases in environmental effects such as pollution because of the development of the Project, especially from the TSF; the long-term safety of the
TSF and its location; effects on the town of Diego de Almagro from the tailings (in response to this concern the TSF was relocated)

- Effect of the proposed port facilities on seafood extraction activities
- Effects of brine discharges from the desalination plant.

In the Environmental Impact Assessment (EIA) of the Project, the impacts associated with the major concerns of the community are:

- Increased Particulate Material PM10 and PM2.5 in the Mine Site of the project. The rating of these impacts was very relevant for Operation Phase. It is for this reason that in order to offset the entire effect on air quality in Diego de Almagro, for both PM10 and PM2.5 a Mitigation Plan will be implemented consisting of paving streets and sweeping throughout the life of the project (18.5 years).

- The increased demand for homes in the town of Diego de Almagro during the operation phase of the Project was rated a significant impact on the EIA of Santo Domingo, because in this town housing supply is less than the expected demand generated by the Project. Mitigation for this includes contribution of 10 L/s of potable water for the town. This would allow increased availability of the vital resource for the inhabitants of Diego de Almagro, facilitating increased residency in town.

- The location of Puerto Santo Domingo is near Caleta Obispito, a small settlement dependent on the resources of the sea, and some independent seaweed pickers located near the Port. The Port interrupts access to some coastal areas due to the construction and subsequent operation of the Project, in addition to the allocation of marine resources due to the uptake of seawater and brine discharge during operation of the Project. These impacts were rated in the EIA as relevant and moderately relevant respectively. The Action Plan developed by SDM, includes increased support of infrastructure to develop this activity. As an example: the provision of basic equipment, and the implementation of an algae drying area. Finally, competitive grants to local entrepreneurship projects for independent seaweed pickers located near the area of Puerto Santo Domingo will be offered.

As part of the EIA review, it is expected that there will be active involvement by the Project in the environmental citizen participation process as required by the authority as part of the evaluation process. The citizen participation process with indigenous communities takes into account the special rules that govern the consultation and participation processes of indigenous peoples. A contact process was designed for these local stakeholders in the context of the general early participation process.
The communications strategy for the Santo Domingo project will be focused on building a positive reputation and supportive environment for the project development in the Atacama Region. Specific development strategies are focussing on the communities of Diego de Almagro and Caldera. A communications plan, communications committee and crisis response management plan are being developed.

A health and safety management system has been developed to meet local legal requirements and industry best practices. Capstone will implement policies, standards, plans and security procedures and will use facilities, equipment and personnel required to provide adequate security levels for its staff and facilities.

20.5.4 Consultation

The stakeholder plan is to develop an early citizen participation program (PACA) that allows the local communities involved in the Project to understand the major Project components, and to gather community opinions, comments and feedback.

During March, 2012, the first round of the PACA was held in Diego de Almagro, Chañaral and Copiapó. During August and September 2012, the second round of PACA was held in Diego de Almagro and Chañaral. A consultation process was organized with the Community of Caldera in June 2013. Finally, during August and September 2013, the third round of PACA was held in Diego de Almagro, Chañaral and Caldera. Consultations included open houses, open meetings, meetings for specialist interests such as fishermen, and meetings with authorities, regional and community services as well as with professional organizations.

As a result of the Early Citizen Participation process, changes in the design of the project were made to minimize impacts to the environment and surrounding communities as follows:

- New location and technology for the tailings storage facility. The TSF was relocated 8.5 km southeast of the town of Diego de Almagro and thickened tailings technology will be used
- The building of a bypass road for the town of Diego de Almagro, which will reduce traffic congestion and will avoid the transit of heavy equipment vehicles through the township
- Plan for hiring local workers. It entails retraining programs and strategic partnerships with technical schools in Chañaral and Diego de Almagro
• Defining guidelines for a community relations plan to contribute to sustainable
development in Diego de Almagro, Chañaral and Caldera, according to the real
needs of the community in the area of influence of the Project.

As the Project has entered the environmental impact assessment system (SEIA), it is
expected that there will be active involvement by the Project in the environmental
citizen participation (PAC) process as part of the evaluation process. The citizen
participation process with indigenous communities takes into account the specific rules
that govern the consultation and participation processes of indigenous peoples. A
contact process was designed for indigenous stakeholders, and is based on the
general early participation process.

20.5.5 Community Relations Plan

Capstone’s Community Relations Plan will contribute to the development of local
economic, cultural, educational, sport, health and entrepreneur activities. This is
expected to integrate the project with the local communities due to the support for local
community organization initiatives.

The Community Relations Plan will be implemented based on the following guidelines:

• Assess the local community to define their needs and priorities for support
• Maximize the benefit of the resources that the community and the local
government obtain from support for their projects
• Promote the development of local community capacities through education and
  specific skills training.

This plan will be developed mainly in the communities of Diego de Almagro, Chañaral
and Caldera, particularly in the towns of Diego de Almagro, Inca de Oro, Chañaral, El
Salado, Flamenco, Obispito and Caldera.

20.5.6 Communications Strategy

The communications strategy for the Santo Domingo Project is focused on building a
positive reputation and supportive environment in the Atacama Region. Specific
development strategies are focussed on the Communities of Diego de Almagro and
Caldera. The goal is to have these communities become familiar with the project in
advance and to promote a mutually beneficial environment.
20.5.7 Health and Safety

Capstone has developed a health and safety management system to meet legal requirements and mining industry best practices. A risk identification study was completed where risks and potential consequences for health and safety were identified for project development and operations. Procedures and prevention measures were identified that will be considered during the construction and operating phases of the Project. These procedures and measures include:

- Health and Safety induction for new employees
- Medical examinations for all workers
- Risk identification based on standard industry protocols
- Development of occupational health and hygiene plans
- Development of emergency plans and management systems
- Requirements for safe operation of mobile equipment.

Management and procedures will be supported by a health, safety, environmental and community management (HSEC) policy that shows commitment to people and establishes high standards for the development of Capstone’s operation.

Data collected during early planning will provide information to define the resources, activities and technology required for the establishment, implementation and improvement of the HSEC management system. The competence of workers will be enhanced through regular internal and external training programs to provide the skills necessary for safe and effective execution of work activities.

Capstone plans to provide facilities on site for monitoring equipment and work standards in order to provide adequate occupational health and hygiene protection to all Capstone and contractor employees. The planned procedure for incident investigation includes registration, investigation, and preparation of action plans. Capstone has established a plan to respond to emergencies and crisis situations caused by its construction and operation activities directly or by third parties that could affect people’s health and/or Capstone’s continued operations.

As part of the 2014 Feasibility Study, a review of the legal requirements in relation to health and safety was performed, and the key legislations were identified.

20.5.8 Communications Policy

The Project has three development stages that require tailored communication plans:
Prior to entering into the Environmental Impact Assessment System (SEIA, Sistema de Evaluación del Impacto Ambiental):

- First Stage: Visits to regional authorities (October 2011 to January 2012).
- Second Stage: Early consultation process with local communities.
  - Open House consultations were held with the communities of Diego de Almagro, Chañaral, Inca de Oro and El Salado (August and September 2012).
  - In 2013 Caldera was included to cover the revised Santo Domingo port location and an Open House was held in Caldera in June 2013.
  - Open House consultations were held with communities of Diego de Almagro, Chañaral, El Salado, Flamenco and Caldera (August and September 2013).

During EIA review and approval processing:

- First Stage: Formal community and citizen participation
- Second Stage: Community agreements and social license for the project

After EIA approval:

- Start up of the project and implementation of agreements.

The following objectives and communications strategy were followed prior to entering the project EIA into the SEIA:

- Build a positive reputation for Capstone and support for its project in the region, particularly in Diego de Almagro, Caldera and with regional authorities
- Ensure that the consultations are perceived favourably by stakeholders, so that submission to the SEIA is seen as legal and valid
- When the Project enters the SEIA it should have a defined public identity so that it is prepared for the higher public profile that it will have at this time.

Capstone has contacted authorities from government, municipality, business and trade associations, and other non-government organizations (NGOs) in the Region. Identification of those communities that may be impacted by the Project has been advanced. The project completed the first two stages of public contact during the early participation program (PACA) in 2012 and 2013 and included meetings with stakeholders, local and regional authorities including Intendente, municipalities, trade associations and special interest groups (e.g. fishermen).

A communications plan has been developed including the formation of a communications committee. The communications committee will receive monthly analyses from the community relations team. A strategy has been developed to
evaluate communications actions and activate specific initiatives as necessary. The communications plan will include public agenda follow up and project issues, and requires a crisis management plan to be developed.

Key issues that the communications policy will address are:

- Water availability: Although the Project plans to use sea water for plant operation, it will require water during the construction stage. For construction water will be provided using a supply contract with Aguas Chañar.
- Manpower: competition for workers from other industries
- Local infrastructure: Limited infrastructure is currently available in Diego de Almagro. This may affect the recruitment and relocation of personnel required for the Project and increase potential social impacts during construction and operations
- Port: Communities of fishermen and seaweed harvesters living in the vicinity of the port will be opposed to any construction on the coast that may impact fishing and commercial maritime activities. A specific management communications strategy has been developed for this issue.
- Power: Chile currently has a limited supply of electrical power generation and transmission capacity, and the use of power by the project will be of significance to local communities.
- Political issues.

20.5.9 Security

Capstone will implement policies, standards, plans and security procedures and will use facilities, equipment and personnel required to provide adequate security levels for its staff and facilities.

Chile is a country with low levels of crime and corruption. The Project is located in the Atacama Region where mining activity is generally accepted by stakeholders as a benefit to the community.

20.6 Discussion on Risks

An environmental risk assessment was completed. Delays in obtaining the necessary permits or authorizations identified as critical for the Project were recognized as a key risk to the proposed Project schedule.
21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimates

21.1.1 Basis of Estimate

The capital cost estimate for the Santo Domingo Project was developed by the following companies:

- AMEC: design and estimating of the mine located process plant and ancillaries; the port located filter plant and ancillaries, power distribution system; and on-site and off-site infrastructure
- PRDW: design and estimating for the port-located materials handling, concentrate storage and ship loading facilities
- BRASS: design and estimating for the sea water and magnetite concentrate pipelines
- Knight Piésold: design and estimating for the tailings storage facility
- NCL: design and estimating for the mine equipment and mine development
- Ghisolfo: design and estimating for the Route C-17 by-pass road
- Capstone: Owner costs.

The capital costs were consolidated by AMEC. The estimate is classed as a Type 3 estimate according to AMEC standards (and the Association for the Advancement of Cost Engineering International, AACE), with an accuracy of -10 to +15% at the 85% confidence level.

The study cost estimate was divided into the following areas:

- Initial Capital
  - Direct costs
  - Indirect costs
  - Owner costs
  - Contingency
- Sustaining Capital
  - Direct costs
  - Indirect costs
  - Owner costs
  - Contingency
All construction activity was assumed to be completed by construction contractors. No provision was included for Owner or engineering, procurement and contract management (EPCM) managed direct hire work in the estimate.

Direct costs included direct construction labour, equipment, materials, consumables and miscellaneous items that form the permanent facilities. Indirect costs included items required to support the construction of the permanent facilities. Owner costs comprised Capstone’s costs prior to the start of operations. Sustaining capital costs include ongoing costs for facilities and equipment required to maintain or increase production. Contingency reflects the potential growth in capital costs excluding changes in the scope of work.

The initial capital cost estimate is included as Table 21-1.

21.1.2 Mine Capital Costs

The total estimated mining capital costs are summarized in Table 21-2. The estimate includes:

- The initial project period includes all funds spent prior to the start-up of processing and metals production
- The initial capital period mine pre-production development of $53.6 M is for pre-stripping and ore stockpiling
- Mine equipment for pre-production (pre-stripping and ore stockpiling) is $105.8 M
- Sustaining capital is from the start of metals production to the end of LOM
- Sustaining capital expenditures reflect the increase of the equipment fleet to achieve the required material movement
- Sustaining capital totals $281.5 million from Year 1 through Year 16.
Table 21-1: Initial Capital Cost Estimate

<table>
<thead>
<tr>
<th>Area</th>
<th>Cost (US$M)</th>
<th>% of Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mine</td>
<td>174.4</td>
<td>10%</td>
</tr>
<tr>
<td>Process Plant</td>
<td>341.8</td>
<td>20%</td>
</tr>
<tr>
<td>Tailings and Water Reclaim</td>
<td>49.9</td>
<td>3%</td>
</tr>
<tr>
<td>Plant Infrastructure (On Site)</td>
<td>97.1</td>
<td>6%</td>
</tr>
<tr>
<td>Port</td>
<td>157.5</td>
<td>9%</td>
</tr>
<tr>
<td>Port Infrastructure (On Site)</td>
<td>27.5</td>
<td>2%</td>
</tr>
<tr>
<td>External Infrastructure (Off Site)</td>
<td>235.9</td>
<td>13%</td>
</tr>
<tr>
<td>Indirect Costs</td>
<td>437.3</td>
<td>25%</td>
</tr>
<tr>
<td>Contingency</td>
<td>229.3</td>
<td>13%</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td>1,750.7</td>
<td>100%</td>
</tr>
</tbody>
</table>

Note: Costs in this table are summarized by major area, and may include costs from consultants, AMEC, Capstone, or all relevant parties.

Table 21-2: Mine Capital Costs

<table>
<thead>
<tr>
<th>Cost Area</th>
<th>Initial Capital 2016</th>
<th>2017_H1</th>
<th>Total</th>
<th>Sustaining Capital</th>
<th>Total Capital</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mine development</td>
<td>26,207</td>
<td>27,737</td>
<td>53,944</td>
<td>-</td>
<td>53,944</td>
</tr>
<tr>
<td>Equipment purchase</td>
<td>69,078</td>
<td>11,267</td>
<td>80,346</td>
<td>199,672</td>
<td>280,017</td>
</tr>
<tr>
<td>Equipment rebuild</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>77,811</td>
<td>77,811</td>
</tr>
<tr>
<td>Other investments</td>
<td>22,527</td>
<td>79</td>
<td>22,606</td>
<td>2,329</td>
<td>24,935</td>
</tr>
<tr>
<td>Dispatch</td>
<td>2,862</td>
<td>49</td>
<td>2,910</td>
<td>1,738</td>
<td>4,648</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>120,674</strong></td>
<td><strong>39,132</strong></td>
<td><strong>159,806</strong></td>
<td><strong>281,549</strong></td>
<td><strong>441,356</strong></td>
</tr>
</tbody>
</table>

Total capital for mine equipment is $280 M over the mine life. The costs for the major equipment are based on quotes obtained by AMEC during third-quarter 2013. The costs for minor equipment are based on quotations obtained by AMEC during third quarter 2012. Estimated freight costs for shipping the mine equipment to site are included. NCL has shown the major mining equipment being purchased in the first year that it is needed to maintain production. The replacement schedule for the equipment is based on the estimated life of the equipment in operating hours and the number of operating hours that the equipment is scheduled for each production year during the mine life. New pieces of equipment were considered up to the maximum requirement. If replacement is needed, rebuild was assumed at a cost of 60% of a new unit.

The initial capital for other costs during the pre-production period amounts to $22.6 M and the sustaining capital for light trucks replacements amounts to $2.3 M. Other
costs are related to spare parts not included with the main equipment and minor equipment and are based on NCL’s database for similar projects in Chile. The initial and sustaining capital costs are $2.9 M and $1.7 M respectively.

21.1.3 Process Capital Costs

The process area estimate was prepared by AMEC on a detailed commodity basis.

This was completed by developing detailed material take-offs for most major equipment, materials and commodity items such as earthworks, concrete, piping, structural, electrical, instrumentation. The equipment, materials and commodities were organized per major work areas (e.g. crushing, grinding, flotation, thickening). This methodology provides a first order, detailed capital cost estimate per major work area.

The capital cost estimate is based on the purchase of new equipment with quantities provided by AMEC engineering. Bulk material take-offs were taken from the 3D model and general arrangement drawings.

All equipment, piping and valves over 3” in diameter, cables and cable trays, instruments, steel structures and concrete shown on the drawings were included. Pipes and valves less than 3” in diameter were included as an allowance in the estimate.

Material quantity take-offs (MTO) were used for the estimate. The MTOs were provided by AMEC engineering and the third party consultants (BRASS, Ghisolfo, Knight Piésold, NCL and PRDW).

Labour rates include both the direct and indirect cost of labour. Rates were determined using typical crew mix from projects similar in nature. The different labour classes and area productivity rates were used to develop the total estimated manhours.

AMEC’s productivity rates are based on North American projects (US and Canada). AMEC’s procedure is to develop factors that reflect the country-specific productivities plus impacts from local project conditions. Based on this information, project productivity factors are then developed that reflect the country and project specifics.

AMEC used construction equipment rates from a periodically updated AMEC database for work in South America.

The civil works costs include manhours, equipment and supplies. The completed costs have been compared to pricing from similar projects in Chile using AMEC’s
internal database. Quotes were sourced for structural steel and concrete pricing. Quotes were also obtained for a unit price per square metre of modular buildings.

Mechanical equipment and platework prices were provided by AMEC’s procurement department using bid package response pricing. Firm quotes were received for critical items and budget quotes were received for other major items. Firm or budget pricing received and used in the estimate accounted for 84% of the equipment and platework costs. The remaining 16% was based on recent projects from AMEC in-house database. Piping unit rates were obtained from formal quotations. The costs for the main electrical equipment and materials were obtained from the formal quotations received. Instrumentation prices were also based on quotations received.

21.1.4 Tailings Storage Facility

The TSF design and capital cost estimate was prepared by Knight Piésold. The TSF design is for a final capacity of 314 Mt of tailings, equivalent to a total volume of 196 Mm³, which will be deposited over approximately 18 years. The costs include all earthworks, and supply and installation of materials, for the underdrains, liner system, tailings distribution system and tailings water reclaim system. Costs are also included for contractor mobilization and demobilization, overhead and profit, indirect costs, and design and construction management.

It was assumed that waste rock will be supplied from the mine open pits by the Capstone mine fleet. The waste rock will be delivered to the embankment with the additional haul cost included in the cost estimate.

Table 21-3 summarizes the cost estimate for the TSF. It should be noted that during future detailed design water balance calculations may include lower density slurry parameters which may require expansion of the lined area.

21.1.5 Infrastructure Capital Costs

Camp

The unit price per square metre based on modular buildings was obtained from a budget quotation from Tecno Fast Atco. The total capital cost of the camp was estimated by Correa 3 and Capstone, and was provided to AMEC for inclusion in the capital estimate.
Table 21-3: TSF Capital Cost Estimate

<table>
<thead>
<tr>
<th>TFS Stage</th>
<th>Type of Cost and When Applied</th>
<th>Cost (US$M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Stage 1: Starter Dam</td>
<td>Initial Capital Cost</td>
<td>24.2</td>
</tr>
<tr>
<td>Stage 2</td>
<td>Sustaining Capital Year 2</td>
<td>9.3</td>
</tr>
<tr>
<td>Stage 3</td>
<td>Sustaining Capital Year 8</td>
<td>7.7</td>
</tr>
<tr>
<td>Stage 4</td>
<td>Sustaining Capital Year 13</td>
<td>7.4</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td></td>
<td><strong>48.6</strong></td>
</tr>
</tbody>
</table>

Built Infrastructure

Budget quotes were obtained for the following built infrastructure, based on building designs that were prepared by the AMEC architectural group:

- **Mine site**
  - Security gatehouse
  - Process plant offices, control room and lunchroom
  - Process plant maintenance workshop
  - General plant warehouse
  - Chemical, metallurgical and sampling laboratory
  - Heavy vehicle work shop*
  - Light vehicle work shop
  - Mine administration offices, change house and lunchroom.
  - Change house
  - Clinic.

- **Port site**
  - Security gatehouse
  - Maintenance workshop and warehouse
  - Port administration building and lunchroom
  - Metallurgical laboratory
  - Change house.

*Note: the heavy vehicle work shop (HVWS) was designed by the AMEC engineering group. For the pricing of the HVWS, material take-offs were prepared and unit rate pricing from the AMEC database was used to determine the overall cost of the facility.
Roads

To facilitate the construction of the project, the existing Route C-17 will be re-routed around the mine site. Other road improvements include a by-pass for Route C-13 around Diego de Almagro to minimize Project traffic through the town. The road design and capital cost estimate were prepared by Ghisolfo, with the final estimate as summarized in Table 21-4.

Power and Electrical

A 220 kV high voltage transmission line from the Diego de Almagro substation to the mine and process plant site will be required. The line will be approximately 8.4 km long. A 220 kV high voltage transmission line from the Totoralillo substation to the Santo Domingo port is also required. This line will be approximately 14.4 km long. Transmission lines and substation design and cost estimates were prepared as part of the electrical design by AMEC.

21.1.6 Sea Water and Concentrate Pipelines

The sea water and magnetite concentrate transport pipeline systems were designed and estimated by BRASS for a total estimated cost of $177.7 M. Costs are broken down by pipeline in Table 21-5 and Table 21-6.

21.1.7 Port Facility

The PRDW scope included the magnetite concentrate stockpile; copper concentrate storage building; concentrate handling; ship loading system; and ancillary installations. The PRDW cost estimate is provided in Table 21-7 and totals $124.3 M.

21.1.8 Indirect Costs

AMEC estimated the indirect costs for the project execution phase as summarized in Table 21-8. Indirect costs total $319.7 M.

21.1.9 Owner Costs

The Owner Costs were estimated by Capstone and were provided to AMEC to incorporate into the project capital cost estimate. The Owner Costs total an estimated $117 M (Table 21-9).
### Table 21-4: Road Capital Costs

<table>
<thead>
<tr>
<th>Description</th>
<th>Cost (US$M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Earth moving</td>
<td>3.4</td>
</tr>
<tr>
<td>Granular layers</td>
<td>2.2</td>
</tr>
<tr>
<td>Paving</td>
<td>7.5</td>
</tr>
<tr>
<td>Drainage and structures</td>
<td>0.7</td>
</tr>
<tr>
<td>Safety, survey and inspection</td>
<td>1.3</td>
</tr>
<tr>
<td>Overhead and profit @ 41.9%</td>
<td>6.3</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>21.4</strong></td>
</tr>
</tbody>
</table>

### Table 21-5: Sea Water Pipeline Costs

<table>
<thead>
<tr>
<th>Description</th>
<th>Cost (US$M)</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Direct Costs</strong></td>
<td></td>
</tr>
<tr>
<td>Construction and Assembly</td>
<td>30.7</td>
</tr>
<tr>
<td>Overhead Expenses and Profit (40%)</td>
<td>12.3</td>
</tr>
<tr>
<td>Subcontractors</td>
<td>7.4</td>
</tr>
<tr>
<td>Sea Water Pipeline</td>
<td>18.5</td>
</tr>
<tr>
<td>Pump Station #1 – pumps, valves, misc. , valves, misc.</td>
<td>9.3</td>
</tr>
<tr>
<td>Pressure Monitoring Stations - valves, fittings, misc.</td>
<td>0.2</td>
</tr>
<tr>
<td>Terminal Station - valves , fittings, misc.</td>
<td>0.7</td>
</tr>
<tr>
<td>Bilge Station – pumps, valves, misc.</td>
<td>3.2</td>
</tr>
<tr>
<td><strong>Total Direct Costs</strong></td>
<td><strong>82.3</strong></td>
</tr>
<tr>
<td><strong>Total Indirect Costs</strong></td>
<td><strong>4.6</strong></td>
</tr>
<tr>
<td><strong>Total Sea Water Pipeline Cost</strong></td>
<td><strong>86.9</strong></td>
</tr>
</tbody>
</table>

### Table 21-6: Magnetite Concentrate Pipeline Costs

<table>
<thead>
<tr>
<th>Description</th>
<th>Cost (US$M)</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Direct Costs</strong></td>
<td></td>
</tr>
<tr>
<td>Construction and Assembly</td>
<td>33.9</td>
</tr>
<tr>
<td>Overhead Expenses and Profit (40%)</td>
<td>13.7</td>
</tr>
<tr>
<td>Subcontractors</td>
<td>15.3</td>
</tr>
<tr>
<td>Pipe Material, spools</td>
<td>9.4</td>
</tr>
<tr>
<td>Positive Displacement Pumps</td>
<td>7.6</td>
</tr>
<tr>
<td>Charge Pumps, valves, fittings</td>
<td>5.8</td>
</tr>
<tr>
<td>Electrical and instrumentation equipment</td>
<td>1.2</td>
</tr>
</tbody>
</table>
### Table 21-7: Port Costs

<table>
<thead>
<tr>
<th>Facility</th>
<th>Supplies Costs (US$M)</th>
<th>Subcontracts Cost (US$M)</th>
<th>Installation Cost (US$M)</th>
<th>Total Cost (US$M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Copper concentrate building</td>
<td>1.8</td>
<td>—</td>
<td>3.3</td>
<td>5.1</td>
</tr>
<tr>
<td>Copper concentrate materials handling equipment</td>
<td>6.6</td>
<td>—</td>
<td>1.6</td>
<td>8.2</td>
</tr>
<tr>
<td>Magnetite concentrate transport</td>
<td>2.1</td>
<td>—</td>
<td>0.3</td>
<td>2.4</td>
</tr>
<tr>
<td>Magnetite concentrate stockpile</td>
<td>1.4</td>
<td>0.3</td>
<td>1.5</td>
<td>3.2</td>
</tr>
<tr>
<td>Magnetite concentrate materials handling equipment</td>
<td>9.0</td>
<td>0.9</td>
<td>2.7</td>
<td>12.6</td>
</tr>
<tr>
<td>Access bridge for conveyor system – offshore</td>
<td>6.5</td>
<td>0.0</td>
<td>5.6</td>
<td>12.1</td>
</tr>
<tr>
<td>Shiploader marine structure</td>
<td>1.6</td>
<td>0.0</td>
<td>3.2</td>
<td>4.8</td>
</tr>
<tr>
<td>Mooring and berthing infrastructure</td>
<td>7.3</td>
<td>0.0</td>
<td>9.3</td>
<td>16.6</td>
</tr>
<tr>
<td>Copper and magnetite loading system</td>
<td>—</td>
<td>0.3</td>
<td>14.8</td>
<td>15.1</td>
</tr>
<tr>
<td>General marine works</td>
<td>0.1</td>
<td>0.0</td>
<td>1.4</td>
<td>1.5</td>
</tr>
<tr>
<td>Site preparation &amp; internal roads</td>
<td>0.1</td>
<td>0.0</td>
<td>9.5</td>
<td>9.6</td>
</tr>
<tr>
<td>Fire control system</td>
<td>—</td>
<td>1.0</td>
<td>—</td>
<td>1.0</td>
</tr>
<tr>
<td>Main electrical distribution system</td>
<td>—</td>
<td>3.5</td>
<td>—</td>
<td>3.5</td>
</tr>
<tr>
<td>CCTV, communications, fibre optic, DCS</td>
<td>0.1</td>
<td>1.3</td>
<td>—</td>
<td>1.4</td>
</tr>
<tr>
<td>Indirect costs</td>
<td>—</td>
<td>2.7</td>
<td>2.3</td>
<td>5.0</td>
</tr>
<tr>
<td>Overhead expenses @ 25%</td>
<td>—</td>
<td>—</td>
<td>—</td>
<td>13.9</td>
</tr>
<tr>
<td>Profit @ 15%</td>
<td>—</td>
<td>—</td>
<td>—</td>
<td>8.3</td>
</tr>
<tr>
<td><strong>Total Port Cost</strong></td>
<td><strong>36.6</strong></td>
<td><strong>10.0</strong></td>
<td><strong>55.5</strong></td>
<td><strong>124.3</strong></td>
</tr>
</tbody>
</table>

### Table 21-8: Indirect Costs

<table>
<thead>
<tr>
<th>Cost Item</th>
<th>Cost (US$M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Engineering and Procurement services (EP)</td>
<td>54.1</td>
</tr>
<tr>
<td>Construction Management (CM)</td>
<td>70.7</td>
</tr>
<tr>
<td>Home &amp; field office materials (EPCM)</td>
<td>6.9</td>
</tr>
<tr>
<td>Support engineering</td>
<td>0.6</td>
</tr>
<tr>
<td>Construction camp</td>
<td>43.4</td>
</tr>
<tr>
<td>Catering</td>
<td>41.2</td>
</tr>
<tr>
<td>Temporary installations</td>
<td>12.1</td>
</tr>
<tr>
<td>Cost Item</td>
<td>Cost (US$M)</td>
</tr>
<tr>
<td>--------------------------------------------------------------------------</td>
<td>-------------</td>
</tr>
<tr>
<td>Water supply (industrial and potable water)</td>
<td>3.4</td>
</tr>
<tr>
<td>Power</td>
<td>3.5</td>
</tr>
<tr>
<td>Firefighting, cleaning, maintenance, waste management</td>
<td>1.3</td>
</tr>
<tr>
<td>Safety, communications</td>
<td>0.7</td>
</tr>
<tr>
<td>Consulting, specialists, third party services</td>
<td>8.9</td>
</tr>
<tr>
<td>Warehouse</td>
<td>2.8</td>
</tr>
<tr>
<td>Crane</td>
<td>1.2</td>
</tr>
<tr>
<td>Off-site and on-site transport</td>
<td>1.1</td>
</tr>
<tr>
<td>Pre-commissioning, commissioning</td>
<td>4.6</td>
</tr>
<tr>
<td>Freight and customs</td>
<td>30.2</td>
</tr>
<tr>
<td>Vendor reps.</td>
<td>6.0</td>
</tr>
<tr>
<td>Start-up and first year spares</td>
<td>14.5</td>
</tr>
<tr>
<td>First fill</td>
<td>2.8</td>
</tr>
<tr>
<td>Transport for local staff</td>
<td>0.3</td>
</tr>
<tr>
<td>Indirect additional/allowance</td>
<td>9.4</td>
</tr>
<tr>
<td><strong>Total Indirect Costs</strong></td>
<td><strong>319.7</strong></td>
</tr>
</tbody>
</table>

**Table 21-9: Owner Costs**

<table>
<thead>
<tr>
<th>Description</th>
<th>2014</th>
<th>2015</th>
<th>2016</th>
<th>Jan–Jul 2017</th>
<th>Total Cost (US$M)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Labour cost - project</td>
<td>1.6</td>
<td>3.2</td>
<td>0.8</td>
<td>0.3</td>
<td>5.9</td>
</tr>
<tr>
<td>General management</td>
<td>0.4</td>
<td>0.4</td>
<td>0.3</td>
<td>0.2</td>
<td>1.3</td>
</tr>
<tr>
<td>Administration and finance</td>
<td>1.2</td>
<td>3.7</td>
<td>1.9</td>
<td>1.8</td>
<td>8.7</td>
</tr>
<tr>
<td>Legal, mining property and permits</td>
<td>2.1</td>
<td>9.7</td>
<td>1.9</td>
<td>0.9</td>
<td>14.7</td>
</tr>
<tr>
<td>Health, safety, environmental and community relations</td>
<td>1.9</td>
<td>11.4</td>
<td>9.4</td>
<td>5.2</td>
<td>27.9</td>
</tr>
<tr>
<td>Labour cost (project support)</td>
<td>3.1</td>
<td>4.9</td>
<td>5.6</td>
<td>3.4</td>
<td>17.0</td>
</tr>
<tr>
<td>Labour cost (services)</td>
<td>2.7</td>
<td>5.3</td>
<td>7.4</td>
<td>5.2</td>
<td>20.6</td>
</tr>
<tr>
<td>Recruitment and selection process</td>
<td>0.6</td>
<td>1.0</td>
<td>0.4</td>
<td>0.7</td>
<td>2.7</td>
</tr>
<tr>
<td>Mine, process and other training</td>
<td>0.1</td>
<td>0.6</td>
<td>2.1</td>
<td>5.3</td>
<td>8.0</td>
</tr>
<tr>
<td>Personnel transport</td>
<td>-</td>
<td>1.4</td>
<td>1.9</td>
<td>1.2</td>
<td>4.5</td>
</tr>
<tr>
<td>Catering</td>
<td>-</td>
<td>0.4</td>
<td>0.8</td>
<td>0.7</td>
<td>1.9</td>
</tr>
<tr>
<td>Labour accreditation, consultants</td>
<td>0.1</td>
<td>0.3</td>
<td>0.5</td>
<td>0.24</td>
<td>1.1</td>
</tr>
<tr>
<td>Vehicles</td>
<td>-</td>
<td>0.6</td>
<td>0.6</td>
<td>0.7</td>
<td>2.0</td>
</tr>
<tr>
<td>PPE</td>
<td>-</td>
<td>0.1</td>
<td>0.1</td>
<td>0.1</td>
<td>0.3</td>
</tr>
<tr>
<td>Others (payroll, newsletter, travel, HR team)</td>
<td>0.1</td>
<td>0.2</td>
<td>0.4</td>
<td>0.2</td>
<td>0.8</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td>13.9</td>
<td>43.3</td>
<td>34.1</td>
<td>26.3</td>
<td>117.6</td>
</tr>
</tbody>
</table>
21.1.10 Contingency

A range analysis was developed, based on AMEC’s probabilistic model with Capstone’s participation. AMEC and Capstone’s sub-consultants’ (BRASS, Ghisolfo, Knight Piésold, NCL and PRDW) estimate values were used for the contingency model. For the range analysis, the major factors that affect the estimated cost of an item or area were identified. The maximum and minimum anticipated percentage variation for each of these factors was then assigned.

AMEC’s contingency model is based on the Monte Carlo method and simulates the probability distribution curve of the overall estimated cost. The confidence interval of 85% was used as the basis for calculating the contingency. The total amount of contingency at this level was $229.3 M.

21.1.11 Taxation Considerations

Local taxes on contractor-supplied materials and installation labour are included in the direct cost estimate. Value-added tax (IVA in the Spanish acronym) on process equipment, contractor-supplied material and contractors’ profit are not included in the estimates of indirect and direct costs. No escalation has been applied.

21.1.12 Initial Capital Cost Estimate

The initial capital cost estimate by area is presented in Table 21-10.

21.1.13 Exclusions

Exclusions to the capital cost estimates include:

- Changes in design criteria
- Scope changes or accelerated schedule
- Closure costs (closure is considered in the operating costs)
- Escalation
- Currency variations
- Force majeure
- Changes in the law
- Schedule delays
- Any rights and licenses not included in Owner costs
Table 21-10: Initial Capital Cost Estimate (by area)

<table>
<thead>
<tr>
<th>Area</th>
<th>Cost (US$M)</th>
<th>% of Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mine</td>
<td>174.4</td>
<td>10%</td>
</tr>
<tr>
<td>Process Plant</td>
<td>341.8</td>
<td>20%</td>
</tr>
<tr>
<td>Tailings and Water Reclalm</td>
<td>49.9</td>
<td>3%</td>
</tr>
<tr>
<td>Plant Infrastructure (On Site)</td>
<td>97.1</td>
<td>6%</td>
</tr>
<tr>
<td>Port</td>
<td>157.5</td>
<td>9%</td>
</tr>
<tr>
<td>Port Infrastructure (On Site)</td>
<td>27.5</td>
<td>2%</td>
</tr>
<tr>
<td>External Infrastructure (Off Site)</td>
<td>235.9</td>
<td>13%</td>
</tr>
<tr>
<td>Indirect Costs</td>
<td>437.3</td>
<td>25%</td>
</tr>
<tr>
<td>Contingency</td>
<td>229.3</td>
<td>13%</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>1,750.7</strong></td>
<td><strong>100%</strong></td>
</tr>
</tbody>
</table>

- Financing costs
- Sunk costs
- Working capital
- Plant mobile equipment (this is included in operating costs).
- Potential impacts from any strikes, riots or looting.

### 21.2 Operating Costs

The estimate is considered to be feasibility-study level with an accuracy of -10% to +15%. The overall assumptions for operating costs that apply to all areas (including mining) include:

- Costs are presented in Q3 2013 US dollars, unless stated otherwise
- Costs are based on an exchange rate of CLP500 to $US1.00 for mining and CLP 480 to $US1.00 for processing
- The costs per tonne of material treated (US$/t) provided in this Report are the average costs over the life of the mine
- Personnel salaries were estimated by Capstone, hourly rates and overheads are based on information from similar Chilean operations
- An average burden rate of 25% has been applied to the salaried and hourly labour to account for social insurance, medical and insurance costs, pensions and vacation costs
• Staffing levels for process were estimated by Capstone, mining personnel levels were estimated by NCL
• The average concentrate grade is 29% for copper and 65% for magnetite
• For the copper equivalent estimate, average life-of-mine prices of $2.85/lb copper and $85.00/t magnetite concentrate were used.
• Operating costs for the Project were based on the Mine Plan V8.1 (developed 3 May 2014) using a maximum throughput of 65,000 t/d for the first five years and nominal 60,000 t/d for subsequent years.

21.2.1 Mining Costs

The basis of the estimate for the 2014 Feasibility Study was that the Santo Domingo open pit operation will be an owner-operated mine and the following assumptions were made:
• Open pit mining costs are the sum of operating and maintenance labour, supervisory labour, parts and consumables, fuel and miscellaneous operating supplies
• Personnel are divided into salaried and hourly personnel
• Parts, non-energy consumables, fuel and miscellaneous operating costs were based on the projected mining fleet requirements
• Quotes from explosives suppliers and equipment suppliers were used
• A diesel fuel cost of $0.90/L delivered to site was used in the operating cost estimate
• The mine fleet replacement considered to be part of the sustaining capital estimate.

Mine operating cost forecasts are included in Table 21-11.

21.2.2 Process Costs

Estimation of the process costs included the following:
• Power consumption is based on the average power demand of the installed equipment in the mechanical equipment list in the 2014 Feasibility Study
• Power costs
• Reagent quotes and equipment supplier quotations were used as applicable
• Logistics and transport costs from supplier quotations were used as applicable
Table 21-11: Mine Operating Costs

<table>
<thead>
<tr>
<th>Item</th>
<th>LOM Total (MUS$)</th>
<th>LOM Average (US$/t Material Mined)</th>
<th>LOM Average (US$/t Treated)</th>
<th>LOM Average (US$/lb CuEq)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Drilling</td>
<td>184.12</td>
<td>0.11</td>
<td>0.47</td>
<td>0.041</td>
</tr>
<tr>
<td>Blasting</td>
<td>340.49</td>
<td>0.21</td>
<td>0.87</td>
<td>0.075</td>
</tr>
<tr>
<td>Loading</td>
<td>473.31</td>
<td>0.29</td>
<td>1.21</td>
<td>0.104</td>
</tr>
<tr>
<td>Hauling</td>
<td>1,112.67</td>
<td>0.68</td>
<td>2.84</td>
<td>0.245</td>
</tr>
<tr>
<td>Ancillary</td>
<td>227.45</td>
<td>0.14</td>
<td>0.58</td>
<td>0.050</td>
</tr>
<tr>
<td>Support Equipment</td>
<td>50.79</td>
<td>0.03</td>
<td>0.13</td>
<td>0.011</td>
</tr>
<tr>
<td>Engineering and Administration</td>
<td>124.60</td>
<td>0.08</td>
<td>0.32</td>
<td>0.0.27</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>2,513.42</strong></td>
<td><strong>1.55</strong></td>
<td><strong>6.42</strong></td>
<td><strong>0.555</strong></td>
</tr>
</tbody>
</table>

- Tailings storage facility operating costs were estimated by Knight Piésold.
- Sea water and concentrate pipeline pumping and operating costs were estimated by BRASS.
- PRDW estimated the port facilities materials handling costs.

Process operating cost forecasts are included in Table 21-12.

### 21.2.3 Labour

The following assumptions were used in estimating the labour costs:

- Labour includes Capstone’s direct staff associated with operations and maintenance.
- The number of personnel was estimated by Capstone and includes coverage for absenteeism, vacation, and staff turnover.
- Shifts with 7 x 7 and 4 x 3 rotations are used for the operations and administration areas respectively, for process, mining and overhead.
- Labour rosters were based on similar operations in Chile from benchmarking by Capstone.
- Labour costs include allowances and other payroll taxes, training, accommodation, insurance and medical costs.

Labour costs are summarized in Table 21-13 and total $213.8 M over the LOM.
### Table 21-12: Process Operating Costs – Commodity Summary

<table>
<thead>
<tr>
<th>Area</th>
<th>LOM Total (MUS$)</th>
<th>LOM Average (US$/t)</th>
<th>LOM Average (US$/lb CuEq)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Process Operating/Plant</td>
<td>2,254.2</td>
<td>5.75</td>
<td>0.497</td>
</tr>
<tr>
<td>Labour</td>
<td>114.3</td>
<td>0.29</td>
<td>0.025</td>
</tr>
<tr>
<td>Power</td>
<td>1,157.3</td>
<td>2.95</td>
<td>0.255</td>
</tr>
<tr>
<td>Reagents</td>
<td>190.1</td>
<td>0.49</td>
<td>0.042</td>
</tr>
<tr>
<td>Steel</td>
<td>546.4</td>
<td>1.39</td>
<td>0.121</td>
</tr>
<tr>
<td>Operating Supplies</td>
<td>18.9</td>
<td>0.05</td>
<td>0.004</td>
</tr>
<tr>
<td>Maintenance Materials</td>
<td>80.0</td>
<td>0.20</td>
<td>0.018</td>
</tr>
<tr>
<td>Other Costs</td>
<td>147.1</td>
<td>0.38</td>
<td>0.032</td>
</tr>
<tr>
<td>TSF and Tailings Water Reclaim</td>
<td>43.4</td>
<td>0.11</td>
<td>0.010</td>
</tr>
<tr>
<td>Labour</td>
<td>13.4</td>
<td>0.03</td>
<td>0.003</td>
</tr>
<tr>
<td>Power</td>
<td>15.7</td>
<td>0.04</td>
<td>0.003</td>
</tr>
<tr>
<td>Reagents</td>
<td>11.3</td>
<td>0.03</td>
<td>0.002</td>
</tr>
<tr>
<td>Steel</td>
<td>0.0</td>
<td>0.00</td>
<td>0.000</td>
</tr>
<tr>
<td>Operating Supplies</td>
<td>0.2</td>
<td>0.00</td>
<td>0.000</td>
</tr>
<tr>
<td>Maintenance Materials</td>
<td>0.7</td>
<td>0.00</td>
<td>0.000</td>
</tr>
<tr>
<td>Other Costs</td>
<td>2.2</td>
<td>0.01</td>
<td>0.000</td>
</tr>
<tr>
<td>Magnetite Concentrate Transport System</td>
<td>68.3</td>
<td>0.17</td>
<td>0.015</td>
</tr>
<tr>
<td>Labour</td>
<td>3.7</td>
<td>0.01</td>
<td>0.001</td>
</tr>
<tr>
<td>Power</td>
<td>16.9</td>
<td>0.04</td>
<td>0.004</td>
</tr>
<tr>
<td>Reagents</td>
<td>0.0</td>
<td>0.00</td>
<td>0.000</td>
</tr>
<tr>
<td>Steel</td>
<td>0.0</td>
<td>0.00</td>
<td>0.000</td>
</tr>
<tr>
<td>Operating Supplies</td>
<td>0.0</td>
<td>0.00</td>
<td>0.000</td>
</tr>
<tr>
<td>Maintenance Materials</td>
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<td>0.006</td>
</tr>
<tr>
<td>Other Costs</td>
<td>4.2</td>
<td>0.01</td>
<td>0.001</td>
</tr>
<tr>
<td>Management</td>
<td>16.3</td>
<td>0.04</td>
<td>0.004</td>
</tr>
<tr>
<td>Sea Water Transfer System</td>
<td>170.1</td>
<td>0.43</td>
<td>0.038</td>
</tr>
<tr>
<td>Labour</td>
<td>3.7</td>
<td>0.01</td>
<td>0.001</td>
</tr>
<tr>
<td>Power</td>
<td>128.2</td>
<td>0.33</td>
<td>0.028</td>
</tr>
<tr>
<td>Reagents</td>
<td>0.0</td>
<td>0.00</td>
<td>0.000</td>
</tr>
<tr>
<td>Steel</td>
<td>0.0</td>
<td>0.00</td>
<td>0.000</td>
</tr>
<tr>
<td>Operating Supplies</td>
<td>0.0</td>
<td>0.00</td>
<td>0.000</td>
</tr>
<tr>
<td>Maintenance Materials</td>
<td>12.5</td>
<td>0.03</td>
<td>0.003</td>
</tr>
<tr>
<td>Other Costs</td>
<td>9.3</td>
<td>0.02</td>
<td>0.002</td>
</tr>
<tr>
<td>Management</td>
<td>16.3</td>
<td>0.04</td>
<td>0.004</td>
</tr>
<tr>
<td>Magnetite Filtering – Port</td>
<td>110.6</td>
<td>0.28</td>
<td>0.024</td>
</tr>
<tr>
<td>Labour</td>
<td>32.9</td>
<td>0.08</td>
<td>0.007</td>
</tr>
<tr>
<td>Power</td>
<td>28.1</td>
<td>0.07</td>
<td>0.006</td>
</tr>
<tr>
<td>Area</td>
<td>LOM Total (MUS$)</td>
<td>LOM Average (US$/t)</td>
<td>LOM Average (US$/lb CuEq)</td>
</tr>
<tr>
<td>-----------------------------------</td>
<td>------------------</td>
<td>---------------------</td>
<td>----------------------------</td>
</tr>
<tr>
<td>Reagents</td>
<td>0.0</td>
<td>0.00</td>
<td>0.00</td>
</tr>
<tr>
<td>Steel</td>
<td>0.0</td>
<td>0.00</td>
<td>0.00</td>
</tr>
<tr>
<td>Operating Supplies</td>
<td>7.8</td>
<td>0.02</td>
<td>0.002</td>
</tr>
<tr>
<td>Maintenance Materials</td>
<td>25.5</td>
<td>0.07</td>
<td>0.006</td>
</tr>
<tr>
<td>Other Costs</td>
<td>0.0</td>
<td>0.00</td>
<td>0.000</td>
</tr>
<tr>
<td>Management</td>
<td>16.3</td>
<td>0.04</td>
<td>0.004</td>
</tr>
<tr>
<td><strong>Fe and Cu Handling, Storage and Loading</strong></td>
<td>106.8</td>
<td>0.27</td>
<td>0.024</td>
</tr>
<tr>
<td>Labour</td>
<td>15.5</td>
<td>0.04</td>
<td>0.003</td>
</tr>
<tr>
<td>Power</td>
<td>30.9</td>
<td>0.08</td>
<td>0.007</td>
</tr>
<tr>
<td>Reagents</td>
<td>0.0</td>
<td>0.00</td>
<td>0.000</td>
</tr>
<tr>
<td>Steel</td>
<td>0.0</td>
<td>0.00</td>
<td>0.000</td>
</tr>
<tr>
<td>Operating Supplies</td>
<td>14.5</td>
<td>0.04</td>
<td>0.003</td>
</tr>
<tr>
<td>Maintenance Materials</td>
<td>0.0</td>
<td>0.00</td>
<td>0.000</td>
</tr>
<tr>
<td>Other Costs</td>
<td>29.7</td>
<td>0.08</td>
<td>0.007</td>
</tr>
<tr>
<td>Management</td>
<td>16.3</td>
<td>0.04</td>
<td>0.004</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td>2,753.4</td>
<td>7.03</td>
<td>0.607</td>
</tr>
<tr>
<td>Area</td>
<td>LOM Total (kUS$)</td>
<td>LOM Average (US$/t)</td>
<td>LOM Average (US$/lb Cu Eq)</td>
</tr>
<tr>
<td>-----------------------------------------</td>
<td>------------------</td>
<td>---------------------</td>
<td>----------------------------</td>
</tr>
<tr>
<td>Process operating/plant</td>
<td>114,324</td>
<td>0.29</td>
<td>0.025</td>
</tr>
<tr>
<td>Labour</td>
<td>114,324</td>
<td>0.29</td>
<td>0.025</td>
</tr>
<tr>
<td>TSF and tailings water reclaim</td>
<td>13,408</td>
<td>0.03</td>
<td>0.003</td>
</tr>
<tr>
<td>Labour</td>
<td>13,408</td>
<td>0.03</td>
<td>0.003</td>
</tr>
<tr>
<td>Fe concentrate transport system</td>
<td>11,266</td>
<td>0.03</td>
<td>0.002</td>
</tr>
<tr>
<td>Labour</td>
<td>3,713</td>
<td>0.01</td>
<td>0.001</td>
</tr>
<tr>
<td>Labour in Management</td>
<td>7,553</td>
<td>0.02</td>
<td>0.002</td>
</tr>
<tr>
<td>Sea water transfer system</td>
<td>11,266</td>
<td>0.03</td>
<td>0.002</td>
</tr>
<tr>
<td>Labour</td>
<td>3,713</td>
<td>0.01</td>
<td>0.001</td>
</tr>
<tr>
<td>Labour in Management</td>
<td>7,553</td>
<td>0.02</td>
<td>0.002</td>
</tr>
<tr>
<td>Fe filtering - port</td>
<td>40,482</td>
<td>0.1</td>
<td>0.009</td>
</tr>
<tr>
<td>Labour</td>
<td>32,929</td>
<td>0.08</td>
<td>0.007</td>
</tr>
<tr>
<td>Labour in Management</td>
<td>7,553</td>
<td>0.02</td>
<td>0.002</td>
</tr>
<tr>
<td>Fe &amp; Cu handling, storage and loading</td>
<td>23,049</td>
<td>0.06</td>
<td>0.005</td>
</tr>
<tr>
<td>Labour</td>
<td>15,496</td>
<td>0.04</td>
<td>0.003</td>
</tr>
<tr>
<td>Labour in Management</td>
<td>7,553</td>
<td>0.02</td>
<td>0.002</td>
</tr>
<tr>
<td>Total</td>
<td>213,796</td>
<td>0.55</td>
<td>0.047</td>
</tr>
</tbody>
</table>
21.2.4 Power

Power consumption is based on the annual consumption of each piece of equipment and allows for the power factor and operating hours for each piece of equipment.

The power cost per year was provided by Capstone. Power costs are summarized in Table 21-14 and for the LOM total $1,377 M. This equates to a power cost of $3.52/t treated. Power costs range from a maximum of $146.9/MWh in 2017 to a minimum of $122.2/MWh in 2023.

21.2.5 Reagents and Consumables

Reagents will include lime, flotation reagents (primary collector, secondary collector, frother, NaMBS) and flocculants. Consumption rates for each reagent were calculated based on throughput, feed grade, recovery, metallurgical testwork and benchmarking. Unit costs were taken from budget quotations and benchmarking. Suppliers included freight costs where required. Reagents are estimated to total $201.4 M over the LOM. This equates to a LOM average per US$/t ore of 0.51 and a LOM average per US$/lb CuEq of 0.044.

Steel

Steel includes liners and ball requirements for crushers and mills. Steel requirements are estimated to total $546.4 M over the LOM. This equates to a LOM average per US$/t ore of 1.39 and a LOM average per US$/lb CuEq of 0.121.

Operating Supplies

Operating supplies include considerations of wear items costs (hydrocyclones and screens), fuel costs for the process plant, filter cloth costs, and operating supplies costs for the tailings and water reclaim. The estimates are summarized in Table 21-15.
Table 21-14: Power Consumption and Costs

<table>
<thead>
<tr>
<th>Description</th>
<th>Total LOM</th>
<th>LOM Average</th>
<th>Total LOM Cost</th>
<th>LOM Average</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>(MWh)</td>
<td>(kWh/t ore)</td>
<td>(kUS$)</td>
<td>(US$/t Treated)</td>
</tr>
<tr>
<td>2100 Process Plant</td>
<td>8,910,482</td>
<td>22.75</td>
<td>1,096,272</td>
<td>2.80</td>
</tr>
<tr>
<td>2110 Materials Handling</td>
<td>129,388</td>
<td>0.33</td>
<td>15,917</td>
<td>0.04</td>
</tr>
<tr>
<td>2120 Grinding</td>
<td>6,097,815</td>
<td>15.57</td>
<td>750,130</td>
<td>1.91</td>
</tr>
<tr>
<td>2130 Flotation and Copper Regrinding</td>
<td>1,292,608</td>
<td>3.30</td>
<td>158,985</td>
<td>0.41</td>
</tr>
<tr>
<td>2140 Magnetic Separation and Regrinding</td>
<td>912,303</td>
<td>2.33</td>
<td>112,375</td>
<td>0.29</td>
</tr>
<tr>
<td>2150 Thickening and Tailings pumping system</td>
<td>433,865</td>
<td>1.11</td>
<td>53,381</td>
<td>0.14</td>
</tr>
<tr>
<td>2160 Reagents Plant</td>
<td>15,443</td>
<td>0.04</td>
<td>1,899</td>
<td>0.00</td>
</tr>
<tr>
<td>2170 Copper Concentrate Filtering</td>
<td>29,060</td>
<td>0.07</td>
<td>3,584</td>
<td>0.01</td>
</tr>
<tr>
<td>3100 Tailings And Water Reclaim</td>
<td>127,531</td>
<td>0.33</td>
<td>15,694</td>
<td>0.04</td>
</tr>
<tr>
<td>3110 High Density Thickening and Tailings Pumping System</td>
<td>98,870</td>
<td>0.25</td>
<td>12,163</td>
<td>0.03</td>
</tr>
<tr>
<td>3120 Tailings Distribution System</td>
<td>15,889</td>
<td>0.04</td>
<td>1,954</td>
<td>0.00</td>
</tr>
<tr>
<td>3160 Water Reclaim and Pumping System</td>
<td>12,772</td>
<td>0.03</td>
<td>1,577</td>
<td>0.00</td>
</tr>
<tr>
<td>4100 Plant Infrastructure (On-Site)</td>
<td>496,622</td>
<td>1.27</td>
<td>61,072</td>
<td>0.16</td>
</tr>
<tr>
<td>4120 Water Distribution and Desalination Plant</td>
<td>344,380</td>
<td>0.88</td>
<td>42,350</td>
<td>0.11</td>
</tr>
<tr>
<td>4130 Plant Administration Buildings</td>
<td>103,561</td>
<td>0.26</td>
<td>12,736</td>
<td>0.03</td>
</tr>
<tr>
<td>4150 Plant Power Supply and Distribution</td>
<td>7,991</td>
<td>0.02</td>
<td>983</td>
<td>0.00</td>
</tr>
<tr>
<td>4160 Plant Ancillary Facilities</td>
<td>40,690</td>
<td>0.10</td>
<td>5,004</td>
<td>0.01</td>
</tr>
<tr>
<td>5100 Port</td>
<td>335,392</td>
<td>0.86</td>
<td>41,236</td>
<td>0.11</td>
</tr>
<tr>
<td>5110 Concentrate Reception and Storage</td>
<td>27,203</td>
<td>0.07</td>
<td>3,345</td>
<td>0.01</td>
</tr>
<tr>
<td>5120 Magnetite Concentrate Filtering</td>
<td>201,515</td>
<td>0.51</td>
<td>24,772</td>
<td>0.06</td>
</tr>
<tr>
<td>5130 Copper Concentrate Transportation and Storage</td>
<td>12,266</td>
<td>0.03</td>
<td>1,508</td>
<td>0.00</td>
</tr>
<tr>
<td>5140 Magnetite Concentrate Pumping System</td>
<td>48,305</td>
<td>0.12</td>
<td>5,940</td>
<td>0.02</td>
</tr>
<tr>
<td>5150 Maritime Works</td>
<td>46,104</td>
<td>0.12</td>
<td>5,670</td>
<td>0.01</td>
</tr>
<tr>
<td>5200 Port Infrastructure (On-Site)</td>
<td>144,301</td>
<td>0.37</td>
<td>17,746</td>
<td>0.05</td>
</tr>
<tr>
<td>5220 Concentrate Storage and Water Reclaim</td>
<td>120,754</td>
<td>0.31</td>
<td>14,850</td>
<td>0.04</td>
</tr>
<tr>
<td>5250 Plant Power Supply and Distribution</td>
<td>15,981</td>
<td>0.04</td>
<td>1,965</td>
<td>0.01</td>
</tr>
<tr>
<td>5260 Port Ancillary Facilities</td>
<td>7,566</td>
<td>0.02</td>
<td>930</td>
<td>0.00</td>
</tr>
<tr>
<td>6100 External Infrastructure (Off-Site)</td>
<td>1,177,502</td>
<td>3.01</td>
<td>145,136</td>
<td>0.37</td>
</tr>
<tr>
<td>6120 Concentrate Transportation System</td>
<td>137,390</td>
<td>0.35</td>
<td>16,886</td>
<td>0.04</td>
</tr>
<tr>
<td>6150 Sea Water Supply</td>
<td>1,040,112</td>
<td>2.66</td>
<td>128,250</td>
<td>0.33</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>11,191,831</strong></td>
<td><strong>28.57</strong></td>
<td><strong>1,377,154</strong></td>
<td><strong>3.52</strong></td>
</tr>
</tbody>
</table>
Table 21-15: Operating Supplies Estimates

<table>
<thead>
<tr>
<th>Description</th>
<th>LOM Total (MUS$)</th>
<th>LOM Average (US$/t ore)</th>
<th>LOM Average (US$/lb CuEq)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Wear items</td>
<td>3.51</td>
<td>0.01</td>
<td>0.001</td>
</tr>
<tr>
<td>Fuel (process plant)</td>
<td>28.67</td>
<td>0.07</td>
<td>0.006</td>
</tr>
<tr>
<td>Filter cloth</td>
<td>9.03</td>
<td>0.02</td>
<td>0.002</td>
</tr>
<tr>
<td>Operating supplies tailings and water reclaim</td>
<td>0.16</td>
<td>0.00</td>
<td>0.000</td>
</tr>
</tbody>
</table>

21.2.6 Maintenance

The estimated cost for the process plant maintenance spares and consumables was estimated as an annual percentage of the estimated direct capital equipment costs. The estimated maintenance spares and materials cost is $0.39/t treated and includes:

- Spares and materials used for preventive and corrective maintenance of equipment and facilities
- Major maintenance contracts and third party services costs.

Assumptions in the estimate include:

- Consumption and costs were estimated by La Cumbre based on equipment quotations and benchmarking
- Information was supplemented with information from Knight Piésold, BRASS and PRDW
- Exclusion of start-up and first year spares which are included in capital costs
- Exclusion of mining equipment fleet spares.

Over the life of mine, the maintenance costs are estimated to total $154.1 M, which equates to a LOM average cost of $0.034/lb CuEq.

21.2.7 Other Costs

Other costs include third party contracts for leasing, minor maintenance, operations and/or support contracts, and total $222.7 M. This equates to a LOM average per $/t ore of 0.57 and a LOM average per $/lb CuEq of 0.049.

21.2.8 Operating Cost Estimate

The initial operating cost estimate by area is included as Table 21-16.
Table 21-16: Initial Operating Cost Estimate (by area)

<table>
<thead>
<tr>
<th>Cost Centre</th>
<th>LOM Total (MUS$)</th>
<th>LOM Average (US$/t)</th>
<th>LOM Average (US$/lb CuEq)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Process Operations/Plant</td>
<td>2,254.2</td>
<td>5.75</td>
<td>0.497</td>
</tr>
<tr>
<td>Tailings storage facility and tailings water reclaim</td>
<td>43.4</td>
<td>0.11</td>
<td>0.010</td>
</tr>
<tr>
<td>Concentrate transport system</td>
<td>68.3</td>
<td>0.17</td>
<td>0.015</td>
</tr>
<tr>
<td>Sea water transfer system</td>
<td>170.1</td>
<td>0.43</td>
<td>0.038</td>
</tr>
<tr>
<td>Magnetite filtering – port</td>
<td>110.6</td>
<td>0.28</td>
<td>0.024</td>
</tr>
<tr>
<td>Magnetite and copper handling, storage and loading</td>
<td>106.8</td>
<td>0.27</td>
<td>0.024</td>
</tr>
<tr>
<td><strong>Total - Process</strong></td>
<td><strong>2,753.4</strong></td>
<td><strong>7.03</strong></td>
<td><strong>0.607</strong></td>
</tr>
<tr>
<td>Copper Concentrate Transport</td>
<td>54.5</td>
<td>0.14</td>
<td>0.012</td>
</tr>
<tr>
<td>G&amp;A</td>
<td>439.6</td>
<td>1.12</td>
<td>0.097</td>
</tr>
<tr>
<td><strong>Mining</strong></td>
<td><strong>2,513.4</strong></td>
<td><strong>6.42</strong></td>
<td><strong>0.555</strong></td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>5,760.9</strong></td>
<td><strong>14.71</strong></td>
<td><strong>1.271</strong></td>
</tr>
</tbody>
</table>

21.2.9 Exclusions

The following items are excluded from the overall operating cost estimate:

- Offshore transportation costs for concentrate transport and treatment (copper and iron)
- Escalation and exchange rate fluctuations
- Exploration
- Permits
- Import duty
- Taxes
- Interest and financing charges
- Corporate overheads
- Operating cost contingency.

21.3 Exchange Rates

For purposes of the Project capital and operating cost estimates, a fixed foreign exchange rate between Chilean pesos (CLP) and US dollars (US$) was initially used as shown in Table 21-17.
Table 21-17: Initial Foreign Exchange Value Assumptions

<table>
<thead>
<tr>
<th>Cost Estimate Item</th>
<th>Initial Foreign Exchange Rate (CLP/US$)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Initial capital cost estimate (excluding mine equipment)</td>
<td>480</td>
</tr>
<tr>
<td>Sustaining capital cost estimate (excluding mine equipment)</td>
<td>480</td>
</tr>
<tr>
<td>Process operating cost estimate</td>
<td>480</td>
</tr>
<tr>
<td>G&amp;A and copper hauling operating cost estimate</td>
<td>480</td>
</tr>
<tr>
<td>Initial mine equipment capital cost estimate</td>
<td>500</td>
</tr>
<tr>
<td>Sustaining mine equipment capital cost estimate</td>
<td>500</td>
</tr>
<tr>
<td>Mine operating cost estimate</td>
<td>500</td>
</tr>
</tbody>
</table>

However, during the estimate development, the foreign exchange rate between the CLP and US$ changed appreciably and the following modifications were undertaken:

- For an updated foreign exchange rate for the development period from 2014 through 2017, Capstone used the mean value of the projected CLP to US$ foreign exchange rate from a total of 29 analyst firms compiled by Bloomberg (as of 6 May 2014).
- For an updated foreign exchange rate for the operating period from 2018 through 2035, Capstone used an algorithm that was developed using the CLP/US$ exchange rate value versus the market sales price of copper. This information was gathered over the last 10 years on a daily basis and resulted in the following algorithm:

  \[
  \text{CLP/US$ Exchange Rate} = -0.0204 \times \text{(price of Cu in US$/t)} + 660.41 
  \]

For the 2014 Feasibility Study copper price of $2.85 ($6,281/t), use of this algorithm equates to a CLP/US$ rate of 532.

The updated foreign exchange values used for the Project financial calculations are summarized in Table 21-18.

21.4 Final Capital and Operating Cost Estimate

Table 21-19 summarizes the capital and operating costs for the Project using the previous foreign exchange rate. The table also provides the updated foreign exchange rates; and summarizes the cost impact of the change in exchange rate assumptions on the capital and operating costs.
Table 21-18: Current Foreign Exchange Values

<table>
<thead>
<tr>
<th>Period (Year)</th>
<th>Updated Foreign Exchange Rate (CLP/US)</th>
</tr>
</thead>
<tbody>
<tr>
<td>2014</td>
<td>553</td>
</tr>
<tr>
<td>2015</td>
<td>557</td>
</tr>
<tr>
<td>2016</td>
<td>517</td>
</tr>
<tr>
<td>2017</td>
<td>519</td>
</tr>
<tr>
<td>2018 through 2035</td>
<td>532</td>
</tr>
</tbody>
</table>

Table 21-19: Changes in Capital and Operating Costs - Updated Foreign Exchange Rates

<table>
<thead>
<tr>
<th>Cost Estimate Item</th>
<th>Total Cost (US$M)</th>
<th>Total Cost (US$M)</th>
<th>Delta (US$M)</th>
<th>Current less Previous Forex</th>
</tr>
</thead>
<tbody>
<tr>
<td>Initial Capital Cost Estimate (excluding mine equipment)</td>
<td>1,576.3</td>
<td>1,527.3</td>
<td>-48.9</td>
<td></td>
</tr>
<tr>
<td>Initial Mine Equipment and Development Capital Cost Estimate</td>
<td>174.4</td>
<td>172.4</td>
<td>-2.0</td>
<td></td>
</tr>
<tr>
<td><strong>Subtotal Initial Capital Cost</strong></td>
<td><strong>1,750.7</strong></td>
<td><strong>1,699.7</strong></td>
<td><strong>-50.9</strong></td>
<td></td>
</tr>
<tr>
<td>Sustaining Capital Cost Estimate (excluding mine equipment)</td>
<td>94.7</td>
<td>89.2</td>
<td>-5.5</td>
<td></td>
</tr>
<tr>
<td>Process Operating Cost Estimate</td>
<td>2,753.4</td>
<td>2,725.7</td>
<td>-27.7</td>
<td></td>
</tr>
<tr>
<td>Copper Hauling Operating Cost Estimate*</td>
<td>54.5</td>
<td>54.5</td>
<td>0.0</td>
<td></td>
</tr>
<tr>
<td>G&amp;A Operating Cost Estimate</td>
<td>439.6</td>
<td>439.6</td>
<td>0.0</td>
<td></td>
</tr>
<tr>
<td>Sustaining Mine Equipment Capital Cost Estimate</td>
<td>281.5</td>
<td>279.2</td>
<td>-2.4</td>
<td></td>
</tr>
<tr>
<td>Mine Operating Cost Estimate</td>
<td>2,513.4</td>
<td>2,471.9</td>
<td>-41.5</td>
<td></td>
</tr>
<tr>
<td><strong>Total Life of Mine Capital Costs</strong></td>
<td><strong>2,127.0</strong></td>
<td><strong>2,068.2</strong></td>
<td><strong>-58.8</strong></td>
<td></td>
</tr>
<tr>
<td><strong>Total Life of Mine Operating Costs</strong></td>
<td><strong>5,760.9</strong></td>
<td><strong>5,691.7</strong></td>
<td><strong>-69.2</strong></td>
<td></td>
</tr>
<tr>
<td><strong>Total Life of Mine Capital &amp; Operating Costs</strong></td>
<td><strong>7,887.8</strong></td>
<td><strong>7,759.9</strong></td>
<td><strong>-128.0</strong></td>
<td></td>
</tr>
</tbody>
</table>

Note: "* = Land freight only, excludes ocean freight, insurance, marketing and umpiring

For the copper hauling operations and G&A estimates, there were no impacts as these values were originally estimated in US$.

The updated foreign exchange rates and associated capital and operating costs were used in all the financial calculations completed for the Project and detailed in this Report.

21.5 Comments on Section 21

The estimated total LOM capital cost is $2,068.2 M.

The corresponding LOM operating cost estimate is $5,691.7 M.
22.0 ECONOMIC ANALYSIS

The results of the economic analysis to support Mineral Reserves represent forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

Forward-looking statements in this Report include, but are not limited to, statements with respect to future metal prices and concentrate sales contracts, the estimation of Mineral Reserves and Mineral Resources, the realisation of Mineral Reserve estimates including the achievement of the dilution and recovery assumptions, the timing and amount of estimated future production, costs of production, capital expenditures, costs and timing of the development of ore zones, permitting time lines, requirements for additional capital, government regulation of mining operations, environmental risks, unanticipated reclamation expenses, and title disputes.

Additional risk can come from actual results of reclamation activities; conclusions of economic evaluations; changes in Project parameters as mine and process plans continue to be refined, possible variations in ore reserves, grade or recovery rates; geotechnical considerations during mining; failure of plant, equipment or processes to operate as anticipated; shipping delays and regulations; accidents, labour disputes and other risks of the mining industry; and delays in obtaining governmental approvals.

Years discussed in this sub-section are presented for illustrative purposes only, as no decision has been made on mine construction by Capstone, and the relevant permits for the Project remain to be secured.

22.1 Methodology Used

The Project has been evaluated using an 8% discounted cash flow (DCF) analysis on an after tax basis. Cash inflows consist of annual revenue projections for the mine and approximately four years of pre-production. Cash outflows such as capital costs, operating costs and royalties are subtracted from the inflows to arrive at the annual cash flow projections.

To reflect the time value of money, annual net cash flow (NCF) projections are discounted back to the 2014 Feasibility Study valuation date of third quarter 2013 using an 8% discount rate. The discount rate appropriate for the Project has been determined using several factors, including the type of commodity and the level of project risks (market risk, technical risk and political risk). The discounted present values of the cash flows are summed to arrive at the Project net present value (NPV).
An NPV sensitivity analysis was completed using discount rates of 5%, 8% (selected rate), 10%, 12% and 15%. In addition to the NPV, the internal rate of return (IRR) and payback period were also calculated. The IRR is defined as the discount rate that results in an NPV equal to zero. Generally speaking, the higher the IRR, the more desirable it is to undertake a Project. In the calculation of IRR it is assumed that any intermediate cash flows can be reinvested at the same rate of return. Cash flows are assumed to occur at the end of each period.

22.2 Financial Model Parameters

The financial model is based on the Mineral Reserves outlined in Section 15, and the mining rates and assumptions discussed in Section 16 and the recovery and processing rates and assumptions discussed in Section 13 and Section 17 respectively.

Exchange rate assumptions are outlined in Section 21.3. As noted in that sub-section, the capital and operating costs using the updated foreign exchange rate have been used for all financial evaluations and reported results.

Operating costs are summarized in Table 22-1. Initial capital costs for the base case are estimated to be $1,700 M for the construction phase. Over the LOM, sustaining capital is estimated to be $368.4 M. Closure and reclamation costs have been estimated to be $92.1 M.

Smelting and refining terms considered in the evaluation are listed in Table 22-2. Transport and insurance charges for copper concentrate are provided in Table 22-3. Life of mine (LOM) copper transport costs are estimated to be $54.5 M for land freight, $203.9 M for ocean freight, $3.1 M for insurance and $10.7 M for marketing.

For the transport and insurance charges for the magnetite concentrate, the following are included in the operating costs and the financial model:

- The magnetite concentrate is transferred to the port via pipeline with the costs included in operating costs
- The magnetite concentrate sales price is adjusted to FOB (Free On Board) Santo Domingo port shipping basis.

As such, no additional transport or insurance charges are required to be included in the financial model.
### Table 22-1: Total Project Operating Costs

<table>
<thead>
<tr>
<th>Cash Costs</th>
<th>LOM Total (kUS$)</th>
<th>LOM Average (US$/t)</th>
<th>LOM Cost (US$/lb Cu payable)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mining</td>
<td>2,471,905</td>
<td>6.31</td>
<td>1.12</td>
</tr>
<tr>
<td>Process</td>
<td>2,725,682</td>
<td>6.96</td>
<td>1.23</td>
</tr>
<tr>
<td>G&amp;A</td>
<td>439,567</td>
<td>1.12</td>
<td>0.20</td>
</tr>
<tr>
<td>Cu Concentrate transport (onshore &amp; offshore) insurance and sales</td>
<td>272,230</td>
<td>0.69</td>
<td>0.12</td>
</tr>
<tr>
<td>Sub-Total</td>
<td>5,909,384</td>
<td>15.09</td>
<td>2.68</td>
</tr>
<tr>
<td>By-product metal credits</td>
<td></td>
<td>(-3.05)</td>
<td></td>
</tr>
<tr>
<td>TC/RC Costs</td>
<td></td>
<td></td>
<td>0.32</td>
</tr>
<tr>
<td>TOTAL - Cu Cash Cost (net of Fe &amp; Au by-product credits)</td>
<td></td>
<td>(-0.06)</td>
<td></td>
</tr>
</tbody>
</table>

### Table 22-2: Smelter Terms

<table>
<thead>
<tr>
<th>Item</th>
<th>Unit</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Concentrate Cu grade</td>
<td>%</td>
<td>29.0</td>
</tr>
<tr>
<td>Concentrate moisture</td>
<td>%</td>
<td>8.0</td>
</tr>
<tr>
<td>Concentrate losses</td>
<td>%</td>
<td>0.10</td>
</tr>
<tr>
<td>Land freight</td>
<td>US$/wmt</td>
<td>14.00</td>
</tr>
<tr>
<td>Ocean freight</td>
<td>US$/wmt</td>
<td>52.36</td>
</tr>
<tr>
<td>Marketing and umpiring</td>
<td>US$/wmt</td>
<td>3.00</td>
</tr>
<tr>
<td>Insurance premium</td>
<td>%</td>
<td>0.05</td>
</tr>
<tr>
<td>Treatment charge</td>
<td>US$/dmt</td>
<td>75.00</td>
</tr>
<tr>
<td>Cu pay factor</td>
<td>%</td>
<td>96.5</td>
</tr>
<tr>
<td>Cu unit deduction</td>
<td>%</td>
<td>1.0</td>
</tr>
<tr>
<td>Cu refining charge</td>
<td>US$/lb Cu</td>
<td>0.075</td>
</tr>
<tr>
<td>Magnetite concentrate grade</td>
<td>%</td>
<td>65.0</td>
</tr>
<tr>
<td>Magnetite concentrate moisture</td>
<td>%</td>
<td>8.0</td>
</tr>
<tr>
<td>Magnetite concentrate price (FOB port)</td>
<td>US$/dmt</td>
<td>85.00</td>
</tr>
<tr>
<td>Au pay factor</td>
<td>%</td>
<td>90.0</td>
</tr>
<tr>
<td>Au unit deduction</td>
<td>g/t</td>
<td>0.0</td>
</tr>
<tr>
<td>Au refining charge</td>
<td>US$/oz</td>
<td>5.00</td>
</tr>
</tbody>
</table>
Table 22-3: Cu and Magnetite Concentrate Transport and Insurance Charges

<table>
<thead>
<tr>
<th>Item</th>
<th>Unit</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Cu concentrate land freight</td>
<td>US$/wmt</td>
<td>14.00</td>
</tr>
<tr>
<td>Cu concentrate ocean freight</td>
<td>US$/wmt</td>
<td>52.36</td>
</tr>
<tr>
<td>Cu concentrate insurance</td>
<td>%</td>
<td>0.05</td>
</tr>
<tr>
<td>Cu concentrate marketing and umpiring</td>
<td>US$/wmt</td>
<td>3.00</td>
</tr>
<tr>
<td>Magnetite concentrate land freight</td>
<td>N/A</td>
<td>In operating costs</td>
</tr>
<tr>
<td>Magnetite concentrate ocean freight</td>
<td>N/A</td>
<td>FOB Santo Domingo port</td>
</tr>
<tr>
<td>Magnetite concentrate insurance</td>
<td>N/A</td>
<td>0</td>
</tr>
<tr>
<td>Magnetite concentrate marketing and umpiring</td>
<td>N/A</td>
<td>0</td>
</tr>
</tbody>
</table>

The Project was evaluated using a range of six different sets of metal prices. The ranges are shown in Table 22-4. The base case price is Case 3, reflecting averages of analysts’ medium term projections.

Royalties of 2% of net smelter return (NSR) are payable on 100% of the production. The NSR of 2% is charged on all of the metals (Cu, Fe and Au) forecast to be recovered. The LOM royalty payment is estimated to be $245.9 M.

Working capital is considered to be a temporary use of funds, incurred at the start of operations, intended to fund mining and production operations until the full receipt of revenues. As revenues and costs typically vary from year to year, the working capital will also change each year. However, all working capital is theoretically recovered at the end of the project. The formal definition of working capital is the value of current assets minus current liabilities. To estimate working capital for the Project, three months of operating costs were assumed. On this basis, working capital is US$28.5 M in the first year of operation (Year 2017) and a life of mine maximum of $92.8 M (on a previous years cumulative basis) three years later (Year 2020). All costs are based on the updated foreign exchange rates.

The economic analysis assumes that no inflationary adjustments are made. Capital and operating costs are based on third-quarter 2013 US dollars.

Possible salvage values for the mine and crusher were not considered, due to the approximately 18 year mine life. At closure, sale of assets may present an opportunity to offset a portion of the closure and reclamation costs.
### Table 22-4: Metal Prices

<table>
<thead>
<tr>
<th>Pay Metals</th>
<th>Case 1</th>
<th>2</th>
<th>3</th>
<th>4</th>
<th>5</th>
<th>6</th>
</tr>
</thead>
<tbody>
<tr>
<td>Copper US$/lb</td>
<td>2.25</td>
<td>2.50</td>
<td>2.85</td>
<td>3.00</td>
<td>3.25</td>
<td>3.50</td>
</tr>
<tr>
<td>Gold US$/oz</td>
<td>1,100</td>
<td>1,200</td>
<td>1,275</td>
<td>1,400</td>
<td>1,500</td>
<td>1,600</td>
</tr>
<tr>
<td>Iron (Fe 65%) US$/t</td>
<td>75</td>
<td>80</td>
<td>85</td>
<td>90</td>
<td>95</td>
<td>100</td>
</tr>
</tbody>
</table>

Note: Base case is highlighted

#### 22.3 Taxation Considerations

The project was evaluated on an after tax basis with taxes payable in three forms:

- Government mining royalty
- First Category corporate income tax
- IVA.

The government mining royalty is a tax on operating mine income levied on a sliding scale between 5% and 14%, depending on operating margins. The royalty is estimated to be $241.0 M over the LOM and is deductible as an expense against corporate tax.

Corporate income tax consists of the First Category Tax at 20%. Total First Category Tax payments over the LOM are estimated to be $784.2 M. The 15% Second Category “Additional” Tax was not evaluated for the Project. This Second Category Tax is levied on dividend distributions to foreign shareholders.

An IVA of 19% is applicable to a number of goods and services purchased but this tax is refundable once the mine is in operation. Other than the delay in the recovery of IVA during construction and the impact of the time value of money, the LOM net effect of IVA is zero.

The Project evaluation is primarily on an equity funded basis. Where opportunities to utilize debt capital to fund the Project are considered, interest shields may reduce the income tax burden of the Project, but will require proper planning to consider Chilean thin capitalization requirements and withholding taxes on interest.

Capstone advised that Project investors will need to consider the merits of utilizing a D.L. 600 Foreign Investment Contract to contribute the capital investment into the Project. A D.L. 600 Contract provides the ability to elect tax invariability treatment for the Project. Article 11 ter of D.L. 600 provides foreign investors the right to an invariable mining royalty rate for a period of 15 years from the Project start date. Article 11 bis of D.L. 600 allows an investor to elect an invariable income tax rate of
22.4 Financing Considerations

AMEC's economic analysis of the Project is based on 100% owner equity financing. The reason for this is that a project with a return that exceeds the cost of borrowing tends to show increasingly improved results as more leverage is applied and more of the risk is transferred to a third party such as a bank.

A second scenario which included debt financing was considered. Capstone was provided with independent advice that the project could have debt capacity in excess of 65% of initial capital; however, a more conservative figure of 50% capacity figure was used for evaluation purposes. Under this scenario, 50% of the construction capital is borrowed in two equal tranches of $425 M in 2016 and 2017. An assumed 3% raising fee is deducted from the funds raised. The debt is repaid over a period of 10 years, beginning in 2018, at an interest rate of 7% per annum. The effect of this, on an after tax basis at a standard 8% discount rate, is:

- NPV (8% discount rate) of $860.0 M
- IRR of 25.7%.

22.5 Financial Results

On an after-tax basis, the cumulative net cash flow for the base case is $3,226.7 million, the IRR is 17.9% and the payback period is 4.2 years. Based on the assumptions discussed in this Report, the cash flow analysis shows that the Project will generate positive cash flows from the first full year of production onwards. At an 8% discount rate, the after-tax net NPV of the project is $797.4 million. The cash flow analysis for the base case is provided in Table 22-5. The after-tax annual and cumulative cash flows are shown in Figure 22-1.
Table 22-5: Results of Financial Analysis

<table>
<thead>
<tr>
<th>Summary of Cash Flow</th>
<th>Unit</th>
<th>Pre-tax</th>
<th>After-tax</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Cumulative net cash flow</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Undiscounted US$ M</td>
<td>4,251.9</td>
<td>3,226.7</td>
<td></td>
</tr>
<tr>
<td><strong>Net present value</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Discounted at 5% US$ M</td>
<td>1,889.8</td>
<td>1,374.7</td>
<td></td>
</tr>
<tr>
<td>Discounted at 8% US$ M</td>
<td>1,154.1</td>
<td>797.4</td>
<td></td>
</tr>
<tr>
<td>Discounted at 10% US$ M</td>
<td>818.8</td>
<td>534.7</td>
<td></td>
</tr>
<tr>
<td>Discounted at 12% US$ M</td>
<td>568.0</td>
<td>338.8</td>
<td></td>
</tr>
<tr>
<td>Discounted at 15% US$ M</td>
<td>302.5</td>
<td>132.5</td>
<td></td>
</tr>
<tr>
<td><strong>Internal rate of return</strong></td>
<td>%</td>
<td>21.3</td>
<td>17.9</td>
</tr>
<tr>
<td><strong>Payback period</strong></td>
<td>Years</td>
<td>4.0</td>
<td>4.2</td>
</tr>
</tbody>
</table>

Note: Base case is highlighted

Figure 22-1: Project After-Tax Cash Flow

The C1 cash cost is as defined by Brook Hunt and is as stated below:

“C1 Cash Costs are the costs of mining, milling and concentrating, on-site administration and general expenses, metal concentrate treatment charges, and freight and marketing costs less the net value of the by-product credits.”

Cash costs are summarized in Table 22-6 for the first five years of operation and in Table 22-7 for the life of mine. The Au and Fe credits fully offset the operating costs, resulting in a negative C1 cash cost.
Table 22-6: Summary of Cash Costs First 5 Years

<table>
<thead>
<tr>
<th>Cash Costs</th>
<th>Years 1 – 5 (Excludes 2017) (kUS$)</th>
<th>Cost per tonne milled (US$/t)</th>
<th>Cost per pound Cu payable (US$/lb)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Costs</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Mining</td>
<td>755,807</td>
<td>6.39</td>
<td>0.63</td>
</tr>
<tr>
<td>Process</td>
<td>789,532</td>
<td>6.68</td>
<td>0.66</td>
</tr>
<tr>
<td>G&amp;A</td>
<td>126,274</td>
<td>1.07</td>
<td>0.11</td>
</tr>
<tr>
<td>Smelter deductions</td>
<td>146,044</td>
<td>1.23</td>
<td>0.12</td>
</tr>
<tr>
<td>Treatment charges</td>
<td>145,376</td>
<td>1.23</td>
<td>0.12</td>
</tr>
<tr>
<td>Refining charges</td>
<td>90,483</td>
<td>0.77</td>
<td>0.08</td>
</tr>
<tr>
<td>Concentrate transport</td>
<td>147,437</td>
<td>1.25</td>
<td>0.12</td>
</tr>
<tr>
<td><strong>Sub-total</strong></td>
<td><strong>2,200,955</strong></td>
<td><strong>18.61</strong></td>
<td><strong>1.84</strong></td>
</tr>
<tr>
<td>Credits</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Au</td>
<td>(224,274)</td>
<td>(1.90)</td>
<td>(0.19)</td>
</tr>
<tr>
<td>Fe</td>
<td>(1,385,536)</td>
<td>(11.72)</td>
<td>(1.16)</td>
</tr>
<tr>
<td><strong>Sub-total</strong></td>
<td><strong>(1,609,809)</strong></td>
<td><strong>(13.61)</strong></td>
<td><strong>(1.35)</strong></td>
</tr>
<tr>
<td><strong>Adjusted Cash Costs Total</strong></td>
<td><strong>591,145</strong></td>
<td><strong>5.00</strong></td>
<td><strong>0.49</strong></td>
</tr>
</tbody>
</table>

Table 22-7: Summary of Cash Costs - LOM

<table>
<thead>
<tr>
<th>Cash Costs</th>
<th>LOM Total (kUS$)</th>
<th>Cost per tonne milled (US$/t)</th>
<th>Cost per pound Cu payable (US$/lb)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Costs</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Mining</td>
<td>2,471,905</td>
<td>6.31</td>
<td>1.12</td>
</tr>
<tr>
<td>Process</td>
<td>2,725,682</td>
<td>6.96</td>
<td>1.23</td>
</tr>
<tr>
<td>G&amp;A</td>
<td>439,567</td>
<td>1.12</td>
<td>0.20</td>
</tr>
<tr>
<td>Smelter deductions</td>
<td>264,598</td>
<td>0.68</td>
<td>0.12</td>
</tr>
<tr>
<td>Treatment charges</td>
<td>288,447</td>
<td>0.69</td>
<td>0.12</td>
</tr>
<tr>
<td>Refining charges</td>
<td>166,905</td>
<td>0.43</td>
<td>0.08</td>
</tr>
<tr>
<td>Concentrate transport</td>
<td>272,230</td>
<td>0.69</td>
<td>0.12</td>
</tr>
<tr>
<td><strong>Sub-Total</strong></td>
<td><strong>6,609,334</strong></td>
<td><strong>16.87</strong></td>
<td><strong>2.99</strong></td>
</tr>
<tr>
<td>Credits</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Au</td>
<td>(363,312)</td>
<td>(0.93)</td>
<td>(0.16)</td>
</tr>
<tr>
<td>Fe</td>
<td>(6,382,181)</td>
<td>(16.29)</td>
<td>(2.89)</td>
</tr>
<tr>
<td><strong>Sub-Total</strong></td>
<td><strong>(6,745,492)</strong></td>
<td><strong>(17.22)</strong></td>
<td><strong>(3.05)</strong></td>
</tr>
<tr>
<td><strong>Adjusted Cash Cost Total</strong></td>
<td><strong>(136,159)</strong></td>
<td><strong>(0.35)</strong></td>
<td><strong>(0.06)</strong></td>
</tr>
</tbody>
</table>
The cash flow summary in Table 22-8 provides a breakdown of the Project life of mine cash flow that results in an undiscounted margin of $1.93/lb payable copper after application of all costs other than taxes. Table 22-9 provides the cashflow on an annualized basis.

22.6 Sensitivity Analysis

A sensitivity analysis was performed on the financial model taking into account variations in:

- Metal price
- Operating costs (including power)
- Power supply costs alone
- Capital costs.

The analysis shows that the Santo Domingo Project NPV8% is most sensitive to changes in metal price. The sensitivity analysis showed that the project is less sensitive to changes in operating costs; less sensitive still to capital expenditure changes; and least sensitive to changes in power costs.

Figure 22-2 shows the sensitivity of the IRR and Figure 22-3 shows the sensitivity of the NPV8% to the variations imposed in the parameters listed above.

Considering that the Project is priced in US dollars, the effects of exchange rate variation do not apply in the current model, although in reality some equipment, supplies and services may be priced in other currencies.

In addition to the base case metal prices five other sets of metal prices were considered. The sensitivity of the Project to metal price fluctuations is summarized in Table 22-10.

22.7 Comments on Section 22

Using the assumptions outlined in this Report, the Project has a positive cashflow.
Table 22-8: Summary of Cash Flow

<table>
<thead>
<tr>
<th>Cost Item</th>
<th>Unit</th>
<th>LOM</th>
<th>Per tonne milled</th>
<th>Per lb Cu payable</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Revenue (after losses and before deductions)</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Cu</td>
<td>kUS$</td>
<td>6,521,912</td>
<td>16.65</td>
<td>2.95</td>
</tr>
<tr>
<td>Au</td>
<td>kUS$</td>
<td>363,312</td>
<td>0.93</td>
<td>0.16</td>
</tr>
<tr>
<td>Fe</td>
<td>kUS$</td>
<td>6,382,181</td>
<td>16.29</td>
<td>2.89</td>
</tr>
<tr>
<td><strong>Sub-Total</strong></td>
<td>kUS$</td>
<td>13,267,404</td>
<td>33.87</td>
<td>6.01</td>
</tr>
<tr>
<td><strong>Smelting costs</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Treatment</td>
<td>kUS$</td>
<td>(268,447)</td>
<td>(0.69)</td>
<td>(0.12)</td>
</tr>
<tr>
<td>Cu deduction</td>
<td>kUS$</td>
<td>(228,267)</td>
<td>(0.58)</td>
<td>(0.10)</td>
</tr>
<tr>
<td>Au deduction</td>
<td>kUS$</td>
<td>(36,331)</td>
<td>(0.09)</td>
<td>(0.02)</td>
</tr>
<tr>
<td>Refining – Cu</td>
<td>kUS$</td>
<td>(165,622)</td>
<td>(0.42)</td>
<td>(0.08)</td>
</tr>
<tr>
<td>Refining – Au</td>
<td>kUS$</td>
<td>(1,282)</td>
<td>(0.00)</td>
<td>(0.00)</td>
</tr>
<tr>
<td>Transport</td>
<td>kUS$</td>
<td>(272,230)</td>
<td>(0.69)</td>
<td>(0.12)</td>
</tr>
<tr>
<td><strong>Sub-Total</strong></td>
<td>kUS$</td>
<td>(972,180)</td>
<td>(2.48)</td>
<td>(0.44)</td>
</tr>
<tr>
<td><strong>Operating costs</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Mining</td>
<td>kUS$</td>
<td>(2,471,905)</td>
<td>(6.31)</td>
<td>(1.12)</td>
</tr>
<tr>
<td>Process</td>
<td>kUS$</td>
<td>(2,725,682)</td>
<td>(6.96)</td>
<td>(1.23)</td>
</tr>
<tr>
<td>G&amp;A</td>
<td>kUS$</td>
<td>(439,567)</td>
<td>(1.12)</td>
<td>(0.20)</td>
</tr>
<tr>
<td><strong>Sub-Total</strong></td>
<td>kUS$</td>
<td>(5,637,154)</td>
<td>(14.39)</td>
<td>(2.55)</td>
</tr>
<tr>
<td><strong>Other</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Royalty</td>
<td>kUS$</td>
<td>(245,904)</td>
<td>(0.63)</td>
<td>(0.11)</td>
</tr>
<tr>
<td>Closure</td>
<td>kUS$</td>
<td>(92,077)</td>
<td>(0.24)</td>
<td>(0.04)</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td>kUS$</td>
<td>(337,982)</td>
<td>(0.86)</td>
<td>(0.15)</td>
</tr>
<tr>
<td>Earnings before interest, taxes, depreciation, and amortization (EBITDA)</td>
<td>kUS$</td>
<td>6,320,089</td>
<td>16.13</td>
<td>2.86</td>
</tr>
<tr>
<td>Construction capital *</td>
<td>kUS$</td>
<td>(1,699,773)</td>
<td>(4.34)</td>
<td>(0.77)</td>
</tr>
<tr>
<td>Sustaining capital *</td>
<td>kUS$</td>
<td>(368,419)</td>
<td>(0.94)</td>
<td>(0.17)</td>
</tr>
<tr>
<td>Undiscounted margin (cumulative net cash flow)</td>
<td>kUS$</td>
<td>4,251,897</td>
<td>10.85</td>
<td>1.93</td>
</tr>
</tbody>
</table>

Note: * = Capital & operating costs are stated using 2014 exchange rate updates discussed in Section 21.3 and Section 21.4
Santo Domingo Project
Region III, Chile
NI 43-101 Technical Report on Feasibility Study
Table 22-9: Project Cashflow on an Annualized Basis
Calendar year

2013

2014

2015

2016

2017

2018

2019

2020

2021

2022

2023

2024

2025

2026

2027

2028

2029

2030

2031

2032

2033

2034

2035

Production year

-4

-3

-2

-1

0

1

2

3

4

5

6

7

8

9

10

11

12

13

14

15

16

17

18

Metal prices
Cu

US$/lb

2.85

2.85

2.85

2.85

2.85

2.85

2.85

2.85

2.85

2.85

2.85

2.85

2.85

2.85

2.85

2.85

2.85

2.85

2.85

Ag

US$/oz

28.00

28.00

28.00

28.00

28.00

28.00

28.00

28.00

28.00

28.00

28.00

28.00

28.00

28.00

28.00

28.00

28.00

28.00

28.00

Au

US$/oz

1,275

1,275

1,275

1,275

1,275

1,275

1,275

1,275

1,275

1,275

1,275

1,275

1,275

1,275

1,275

1,275

1,275

1,275

1,275

99,679

899,357

811,086

654,241

613,481

553,743

457,056

372,385

280,873

277,282

235,374

224,195

250,668

210,451

172,354

134,688

128,822

89,299

56,877

Life of Mine
Extracted metal value
(after losses & before deductions)
Cu

US$000

6,521,912

Ag

US$000

-

Au

US$000

363,312

6,981

61,123

51,119

41,427

36,890

33,715

26,040

19,351

14,087

14,485

11,159

10,597

11,362

9,071

6,311

4,628

3,382

908

675

Fe Concentrate

US$000

6,382,181

32,802

229,947

282,443

325,072

344,349

203,725

231,258

331,944

344,447

304,919

334,722

408,276

393,688

426,893

458,008

459,056

459,107

458,959

352,567

Total

US$000

13,267,404

139,462

1,190,427

1,144,648

1,020,740

994,719

791,183

714,354

723,680

639,408

596,686

581,255

643,067

655,719

646,415

636,673

598,372

591,310

549,166

410,120

Cu

US$000

(228,267)

(3,489)

(31,477)

(28,388)

(22,898)

(21,472)

(19,381)

(15,997)

(13,033)

(9,831)

(9,705)

(8,238)

(7,847)

(8,773)

(7,366)

(6,032)

(4,714)

(4,509)

(3,125)

(1,991)

Ag

US$000

Au

US$000

(36,331)

(698)

(6,112)

(5,112)

(4,143)

(3,689)

(3,371)

(2,604)

(1,935)

(1,409)

(1,449)

(1,116)

(1,060)

(1,136)

(907)

(631)

(463)

(338)

(91)

(68)

Total

US$000

(264,598)

(4,187)

(37,590)

(33,500)

(27,041)

(25,161)

(22,752)

(18,601)

(14,969)

(11,239)

(11,153)

(9,354)

(8,906)

(9,910)

(8,273)

(6,663)

(5,177)

(4,847)

(3,216)

(2,058)

US$000

(268,447)

(4,103)

(37,018)

(33,385)

(26,929)

(25,251)

(22,793)

(18,813)

(15,328)

(11,561)

(11,413)

(9,688)

(9,228)

(10,318)

(8,662)

(7,094)

(5,544)

(5,302)

(3,676)

(2,341)

Cu

US$000

(165,622)

(2,531)

(22,839)

(20,597)

(16,614)

(15,579)

(14,062)

(11,607)

(9,457)

(7,133)

(7,042)

(5,977)

(5,693)

(6,366)

(5,344)

(4,377)

(3,420)

(3,271)

(2,268)

(1,444)

Ag

US$000

Au

US$000

(1,282)

(25)

(216)

(180)

(146)

(130)

(119)

(92)

(68)

(50)

(51)

(39)

(37)

(40)

(32)

(22)

(16)

(12)

(3)

(2)

Total

US$000

(166,905)

(2,556)

(23,055)

(20,778)

(16,761)

(15,709)

(14,181)

(11,699)

(9,525)

(7,182)

(7,093)

(6,017)

(5,731)

(6,406)

(5,376)

(4,399)

(3,437)

(3,283)

(2,271)

(1,447)

Land freight

US$000

(54,507)

(833)

(7,516)

(6,779)

(5,468)

(5,127)

(4,628)

(3,820)

(3,112)

(2,347)

(2,317)

(1,967)

(1,874)

(2,095)

(1,759)

(1,440)

(1,126)

(1,077)

(746)

(475)

Port storage & handling

US$000

Smelter deductions/premiums

Treatment charge
Cu concentrate
Refining charges

Cu Concentrate transport

Ocean freight

US$000

(203,913)

(3,117)

(28,119)

(25,359)

(20,455)

(19,181)

(17,313)

(14,290)

(11,643)

(8,782)

(8,669)

(7,359)

(7,010)

(7,837)

(6,580)

(5,389)

(4,211)

(4,028)

(2,792)

(1,778)

Marketing & other

US$000

(10,749)

(164)

(1,482)

(1,337)

(1,078)

(1,011)

(913)

(753)

(614)

(463)

(457)

(388)

(369)

(413)

(347)

(284)

(222)

(212)

(147)

(94)

Insurance charges

US$000

(3,062)

(47)

(427)

(383)

(309)

(289)

(261)

(215)

(174)

(131)

(130)

(110)

(104)

(117)

(98)

(79)

(62)

(59)

(40)

(26)

Total

US$000

(272,230)

(4,161)

(37,545)

(33,858)

(27,311)

(25,608)

(23,115)

(19,078)

(15,543)

(11,723)

(11,574)

(9,824)

(9,357)

(10,462)

(8,783)

(7,193)

(5,621)

(5,375)

(3,726)

(2,373)

US$000

12,295,225

124,455

1,055,220

1,023,128

922,698

902,990

708,342

646,163

668,315

597,702

555,453

546,372

609,845

618,624

615,320

611,324

578,593

572,502

536,278

401,901

Net Smelter Return

US$000

12,295,225

124,455

1,055,220

1,023,128

922,698

902,990

708,342

646,163

668,315

597,702

555,453

546,372

609,845

618,624

615,320

611,324

578,593

572,502

536,278

401,901

Portion of production

%

100%

100%

100%

100%

100%

100%

100%

100%

100%

100%

100%

100%

100%

100%

100%

100%

100%

100%

100%

100%

Net Smelter Return
Royalty payment

Royalty percentage of NSR

%

2.0%

2.00%

2.00%

2.00%

2.00%

2.00%

2.00%

2.00%

2.00%

2.00%

2.00%

2.00%

2.00%

2.00%

2.00%

2.00%

2.00%

2.00%

2.00%

2.00%

Applied royalty

US$000

(245,904)

(2,489)

(21,104)

(20,463)

(18,454)

(18,060)

(14,167)

(12,923)

(13,366)

(11,954)

(11,109)

(10,927)

(12,197)

(12,372)

(12,306)

(12,226)

(11,572)

(11,450)

(10,726)

(8,038)

Mining

US$000

(2,471,905)

(57,780)

(138,545)

(145,787)

(160,996)

(158,074)

(152,405)

(132,102)

(134,505)

(142,541)

(144,352)

(143,629)

(150,691)

(146,447)

(132,065)

(140,969)

(129,472)

(125,899)

(86,672)

(48,974)

Process

US$000

(2,725,682)

(37,489)

(153,459)

(161,693)

(158,040)

(157,617)

(158,725)

(150,292)

(150,638)

(153,450)

(150,798)

(150,862)

(154,468)

(151,452)

(151,391)

(154,537)

(144,280)

(132,547)

(151,674)

(102,272)

Tailings

US$000

Production costs

G&A

US$000

(439,567)

(14,397)

(27,055)

(24,959)

(24,890)

(24,976)

(24,395)

(24,303)

(24,335)

(24,447)

(24,374)

(23,666)

(23,566)

(23,572)

(23,478)

(23,443)

(23,432)

(22,990)

(22,710)

(14,579)

Total

US$000

(5,637,154)

(109,666)

(319,058)

(332,439)

(343,925)

(340,667)

(335,525)

(306,697)

(309,477)

(320,439)

(319,524)

(318,158)

(328,724)

(321,471)

(306,934)

(318,948)

(297,184)

(281,436)

(261,057)

(165,824)

Project No.: M40206
July 2014

Page 22-11


## Santo Domingo Project

**Location:** Region III, Chile  
**Calendar year:** 2013-2025  
**Relevant data in US$000:**

### Closure & Savelage Costs
<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>(92,077)</td>
</tr>
</tbody>
</table>

### Salvage value - mine

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>US$000</td>
</tr>
</tbody>
</table>

### Salvage value - crusher

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>US$000</td>
</tr>
</tbody>
</table>

### Total Closure & Savelage Costs

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>(92,077)</td>
</tr>
</tbody>
</table>

### Finishing Charges

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>US$000</td>
</tr>
</tbody>
</table>

### Interest on principal

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>US$000</td>
</tr>
</tbody>
</table>

### Repayment of capitalized interest

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>US$000</td>
</tr>
</tbody>
</table>

### Debt restructuring charges

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>US$000</td>
</tr>
</tbody>
</table>

### Total Finishing Charges

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>US$000</td>
</tr>
</tbody>
</table>

### Earnings

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>(6,320,088)</td>
</tr>
</tbody>
</table>

### Earnings before taxes, depreciation & amortization

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>12,306,715</td>
</tr>
</tbody>
</table>

### Taxation

#### Operation incomes and expenses

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>6,930,071</td>
</tr>
</tbody>
</table>

#### Royalty payment

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>(2,488)</td>
</tr>
</tbody>
</table>

#### Start-up expenses (Corporate Tax)

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>(90,000)</td>
</tr>
</tbody>
</table>

#### Tax Loss

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>(1,506,613)</td>
</tr>
</tbody>
</table>

#### Tax Depreciation for the First Category Income Tax

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>(672,985)</td>
</tr>
</tbody>
</table>

#### Mines Closure

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>92,077</td>
</tr>
</tbody>
</table>

#### Taxable Base for the Mining Royalty

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>4,280,057</td>
</tr>
</tbody>
</table>

#### Tax rate for the Mining Royalty

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>5.63%</td>
</tr>
</tbody>
</table>

#### Mining Royalty

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>241,010</td>
</tr>
</tbody>
</table>

#### Tax amount

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>(1,025,188)</td>
</tr>
</tbody>
</table>

### Interim payment of absorbed earnings

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>US$000</td>
</tr>
</tbody>
</table>

### Value Added Tax (VAT)

#### Payment Value Added Tax costs

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>(1,102,014)</td>
</tr>
</tbody>
</table>

#### Payment Value Added Tax

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>(1,315,837)</td>
</tr>
</tbody>
</table>

### Capital expenditures

#### Construction

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>(1,494,773)</td>
</tr>
</tbody>
</table>

#### Sustaining

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>(384,419)</td>
</tr>
</tbody>
</table>

#### Debt drawdown

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>US$000</td>
</tr>
</tbody>
</table>

### Non-current liabilities

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>US$000</td>
</tr>
</tbody>
</table>

### Net project cash flow

<table>
<thead>
<tr>
<th>Year</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>2013</td>
<td>US$000</td>
</tr>
<tr>
<td>---------------</td>
<td>------</td>
</tr>
<tr>
<td>Production year</td>
<td>4</td>
</tr>
<tr>
<td>Pre-tax</td>
<td>US$000</td>
</tr>
<tr>
<td>Payback</td>
<td>Pre-tax cumulative net cash flow US$000</td>
</tr>
<tr>
<td>Pre-tax payback Years</td>
<td>4.0</td>
</tr>
<tr>
<td>After tax cumulative net cash flow US$000</td>
<td>(22,270)</td>
</tr>
<tr>
<td>After tax payback Years</td>
<td>4.2</td>
</tr>
</tbody>
</table>
Figure 22-2: Sensitivity of IRR (Spider Chart)

![Spider Chart for IRR Sensitivity]

Note: Figure prepared by AMEC, 2014

Figure 22-3: Sensitivity of NPV8% for Base Case (Spider Graph)

![Spider Graph for NPV8% Sensitivity]

Note: Figure prepared by AMEC, 2014
Table 22-10: Sensitivity to Metal Price

<table>
<thead>
<tr>
<th>Item</th>
<th>Unit</th>
<th>Case 1</th>
<th>Case 2</th>
<th>Case 3</th>
<th>Case 4</th>
<th>Case 5</th>
<th>Case 6</th>
</tr>
</thead>
<tbody>
<tr>
<td>Copper price</td>
<td>US$/lb</td>
<td>2.25</td>
<td>2.50</td>
<td>2.85</td>
<td>3.00</td>
<td>3.25</td>
<td>3.50</td>
</tr>
<tr>
<td>Gold price</td>
<td>US$/oz</td>
<td>1,100</td>
<td>1,200</td>
<td>1,275</td>
<td>1,400</td>
<td>1,500</td>
<td>1,600</td>
</tr>
<tr>
<td>Iron price</td>
<td>US$/t</td>
<td>75</td>
<td>80</td>
<td>85</td>
<td>90</td>
<td>95</td>
<td>100</td>
</tr>
<tr>
<td>Pre-tax CNCF</td>
<td>US$million</td>
<td>2,174</td>
<td>3,108</td>
<td>4,252</td>
<td>4,976</td>
<td>5,909</td>
<td>6,843</td>
</tr>
<tr>
<td>Pre-tax NPV 8%</td>
<td>US$million</td>
<td>303</td>
<td>681</td>
<td>1,154</td>
<td>1,438</td>
<td>1,816</td>
<td>2,194</td>
</tr>
<tr>
<td>Pre-tax IRR</td>
<td>%</td>
<td>11.7</td>
<td>16.1</td>
<td>21.3</td>
<td>24.2</td>
<td>28.0</td>
<td>31.6</td>
</tr>
<tr>
<td>Pre-tax Payback</td>
<td>Years</td>
<td>6.2</td>
<td>4.8</td>
<td>4.0</td>
<td>3.7</td>
<td>3.3</td>
<td>3.0</td>
</tr>
<tr>
<td>After tax CNCF</td>
<td>US$million</td>
<td>1,659</td>
<td>2,368</td>
<td>3,227</td>
<td>3,768</td>
<td>4,462</td>
<td>5,154</td>
</tr>
<tr>
<td>After Tax NPV 8%</td>
<td>US$million</td>
<td>141</td>
<td>436</td>
<td>797</td>
<td>1,011</td>
<td>1,295</td>
<td>1,577</td>
</tr>
<tr>
<td>After Tax IRR</td>
<td>%</td>
<td>9.8</td>
<td>13.6</td>
<td>17.9</td>
<td>20.3</td>
<td>23.4</td>
<td>26.4</td>
</tr>
<tr>
<td>After Tax Payback</td>
<td>Years</td>
<td>6.5</td>
<td>4.9</td>
<td>4.2</td>
<td>3.9</td>
<td>3.5</td>
<td>3.2</td>
</tr>
<tr>
<td>C1 cash cost before credits</td>
<td>US$/lb</td>
<td>2.97</td>
<td>2.98</td>
<td>2.99</td>
<td>3.00</td>
<td>3.01</td>
<td>3.02</td>
</tr>
<tr>
<td>C1 cash cost after credits</td>
<td>US$/lb</td>
<td>0.28</td>
<td>0.10</td>
<td>(0.06)</td>
<td>(0.24)</td>
<td>(0.41)</td>
<td>(0.59)</td>
</tr>
</tbody>
</table>

Note: Base case is highlighted
23.0 ADJACENT PROPERTIES

This section is not relevant to this Report.
24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Plan

It is currently intended to complete the project using Capstone-managed, engineering, procurement and contract management (EPCM) and engineering, procurement and construction (EPC) contractor(s) execution. All commitments for contracts and purchase orders will be made in the name of Minera Santo Domingo. Signing authority agreements will be put in place with Minera Santo Domingo’s prime EPCM contractor providing contract and procurement services. These agreements will allow specified representatives to sign documents on Minera Santo Domingo’s behalf once the appropriate project documentation has been approved by Minera Santo Domingo.

24.2 Risk and Opportunity Analysis

A risk and opportunity analysis was performed as part of the 2014 Feasibility Study. The overall project risks were divided into major work areas.

The risk matrix recorded the level of risk which was determined by the relationship between the probability of occurrence (likelihood) of an incident occurring from the hazard, and the consequence level caused by the hazard. The numerical result from the probability of occurrence (likelihood) and consequence determined how significant the hazard was. The level of risk is referred to as a risk priority rating. This priority rating then allows evaluators to prioritize the hazards identified to ensure that the hazards with high potential of creating an incident are eliminated or controlled. Controlling a hazard may involve inspection, investigation and/or monitoring control activities with managers, supervisors and work teams involved as appropriate.

The risk identification process used a list of approximately 300 typical risks from similar projects as well as specific identified Project risks. Using risk review sessions, the risks were reduced to approximately 100 items applicable to the Project and scope of work. Mitigation strategies were subsequently defined and assigned to the risks.

The first risk workshop was held on 23 August 2013, and the first task was to eliminate from the list the risks which were not applicable to the project, remove repetitions and add any new risks identified. Risks were separated into technical or engineering risks and business risks. Capstone’s corporate group then separately carried out the business risks analysis.

The second risk workshop was held on 30 August 2013. This workshop was divided into three parts to review and identify technical risks relevant to each consultant’s area of input.
During the workshop sessions, risks were identified related to market conditions (cost of materials, supplies, skilled local labour, copper prices, exchange rate) which may represent opportunities or threats. It was decided at that stage that only threatening risks would be evaluated and opportunities would be identified during the detailed engineering phase.

In early November 2013, a meeting was held to validate the Risk Management Plan.

The seven highest level risks at a Project-wide level (across all work areas) are listed in Table 24-1. These all have an exposure rating of 10 or higher after mitigation.
Table 24-1: Project Wide Highest Risks

<table>
<thead>
<tr>
<th>Risk</th>
<th>Causes</th>
<th>Impacts</th>
</tr>
</thead>
<tbody>
<tr>
<td>New taxes or royalties</td>
<td>The recently elected centre left government has proposed a bill to increase in business taxes from 20% to 25%</td>
<td>Project economics are impacted</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Project cost increases and potential schedule delays</td>
</tr>
<tr>
<td>Rising capex</td>
<td>Project capital and other costs increase.</td>
<td>Project not viable</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Capstone strategic objectives not achieved and shareholder value eroded</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Development impacted</td>
</tr>
<tr>
<td>Lack of appropriate human resources</td>
<td>Competitive labour market, with other operating mines and projects in development. However the site is at a low altitude and relatively close to towns and cities</td>
<td>Cost increases and schedule delays</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Company strategy not achieved</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Lost revenue</td>
</tr>
<tr>
<td>The infrastructure and utilities required to support the project are not secured</td>
<td>Unavailability of power or port</td>
<td>Cost and schedule overruns</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Project becomes unviable</td>
</tr>
<tr>
<td>Community opposition</td>
<td>Negative community perception of the development, e.g. tailings facility, desalination process and its waste products, pipeline location</td>
<td>Schedule delays and increased costs</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Development possibly stopped and Capstone reputation impacted</td>
</tr>
<tr>
<td>Electrical supply infrastructure and capacity not sufficient to meet project demand</td>
<td>Restricted power availability in the Region</td>
<td>The long term impact on production and lost revenues could have a severe impact on the achievement of strategic and operational goals</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Project schedule delays and increased costs</td>
</tr>
<tr>
<td>Lack of water during construction</td>
<td>The project has no water rights and scarcity of water is a major problem</td>
<td>Project halted</td>
</tr>
<tr>
<td></td>
<td></td>
<td>Company reputation impacted</td>
</tr>
</tbody>
</table>
25.0 INTERPRETATION AND CONCLUSIONS

The QPs, as authors of this Report, have reviewed the data for the Project and note the following conclusions, interpretations and opinions:

25.1 Mineral Tenure, Royalties and Surface Rights

- Information provided to AMEC supports that the capital of Minera Santo Domingo is indirectly 70% owned by Capstone and 30% by Korean Resource Corporation (Kores)
- Capstone has advised AMEC that under the terms of the shareholder agreement signed between Capstone and Kores on June 17, 2011, Capstone is Project operator
- Mineral tenure documentation provided to AMEC supports that Capstone has four groups of concessions with a total of 178 claims (82 exploitation concessions and 96 exploration concessions) which cover a total of 36,375 ha and includes the areas of the planned mine site, plant area and auxiliary facilities including proposed port facilities and the planned sea water and concentrate pipelines from the port to the mine. Concessions are held in the name of Minera Santo Domingo. The information provided supports that the mining tenure held is valid and is sufficient to support declaration of Mineral Resources and Mineral Reserves
- Concessions are surveyed as part of grant requirements
- Concessions are protected under Chilean law by payment of the annual mining license fees. Capstone advised AMEC that all concession fees were current as of 1 October 2013
- No surface rights are currently held by Capstone in the Project area
- Capstone proposes to consolidate Capstone's property in the areas covering the deposit and the area proposed for the mining and process facilities by purchasing these lands through the Ministerio de Bienes Nacionales
- Capstone notes that it will be necessary to either acquire a total of 3,901.3 ha or complete the creation of mining easements for the installation and use of various facilities. Capstone also proposes to apply for one or more mining rights of way in the areas of interest of the Project such as the pipeline route, access roads and off-site ancillary facilities to safeguard these areas. The Project has received government guarantees for the rights of way required by the Project for the areas currently identified
There is sufficient suitable land available within the exploitation concessions for the planned tailings disposal, mine waste disposal, and mining-related infrastructure such as the open pit, process plant, workshops and offices.

The Project as currently envisaged will not require an application for water rights. The water for the operation of the Project will consist solely of sea water. A maritime concession has been requested which will allow the extraction of sea water.

A mining tax will be payable once operations commence, and is a sliding-scale tax.

Capstone advised AMEC that to the extent known, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the property.

25.2 Geology and Mineralization

Mineralization at Santo Domingo occurs primarily in the form of IOCG deposits with related vein and skarn bodies.

Drilling at 100 m centres or less at the Santo Domingo Sur deposit has outlined a 150 m to 500 m thick copper bearing, specularite-magnetite manto sequence hosted within tuffaceous sedimentary rocks, and covering an area of approximately 1,300 m by 800 m. Mineralization occurs in the form of copper-bearing semi-massive to massive specularite and magnetite layers with clots and stringers of chalcopyrite. The upper parts of the manto sequence are frequently oxidized and contain various amounts of copper oxides and chalcocite.

The Iris deposit is a narrow zone (100 m to 250 m wide) of copper-bearing iron mantos and breccias extending over 1,900 m that are hosted by andesitic tuffs and andesitic breccias. The dominant iron oxide at Iris is hematite and the main copper mineral is chalcopyrite.

Mineralization at Iris Norte is very similar to the Iris deposit; however, part of the mineralization appears to be hosted by andesitic flows. The deposit area has been intruded by significant amounts of diorite dykes and sills. The deposit is approximately 500 m wide and has been tested over a strike length of 1,600 m. Mineralization consists of mixed magnetite and hematite mantos. The main sulphides in Iris Norte are pyrite and chalcopyrite.

Drilling at the Estrellita deposit has delineated a tabular body of copper mineralization hosted by breccias and mantos along a fault zone around the Estrellita artisanal mine workings. The east–west extent of the Estrellita deposit along the Santo Domingo fault totals more than 1,000 m and the deposit remains open in both directions. Mineralization is a mixture of manto style, iron oxide and
structurally-controlled, vein-style mineralization. Copper mineralization typically consists of copper oxides such as brochantite, chrysocolla, almagre, cuprite, and chalcocite, and transitions through a mixed zone of oxides and sulphides into a sulphide zone where the main copper mineral is chalcopyrite.

25.3 Exploration and Drilling

- Modern exploration commenced in 2002. Between 2002 and June 2011 work by Far West Mining included a regional airborne geophysical survey and interpretation of results, geological mapping, surface and drainage sampling, an induced polarization (IP) survey, core and reverse circulation (RC) drilling, and resource estimation. A preliminary assessment was conducted in 2008. Capstone acquired Far West Mining and completed a pre-feasibility study. A feasibility study was commissioned in 2012
- Prior to Capstone’s Project interest, 348 RC drill holes (90,611 m) and 50 core holes (16,275 m) were completed. Since Project acquisition, Capstone has undertaken an additional 66 core holes (13,282 m)
- Most holes are vertical because mineralization is typically horizontal or gently dipping
- Drill cuttings and core were logged using a set of rock type codes. Drill collars were located using a differential GPS. Downhole surveying was conducted using accepted down-hole survey tools
- RPA considers that the drilling has been conducted in a manner consistent with standard industry practices. The spacing and orientation of the holes are appropriate for the deposit geometry and mineralization style.

25.4 Sample Preparation and Analysis

- Reverse circulation drill cuttings were collected at 2 m intervals. Core was nominally sampled at 2 m intervals
- Samples were shipped to an independent analytical and preparation laboratory, ALS Chemex, in Chile
- Samples were analysed for 27 elements using ICP. Gold assays were determined using fire assay with an AAS finish. Copper values over 10,000 ppm were re-assayed. All samples over 15% Fe inside the existing block model for which sample material was still available were re-assayed as a check on the Fe analyses as the initial analytical method was considered to have understated the iron content. Soluble copper analysis was conducted on some samples from the 2012 drill campaign
• RPA developed regression formulae based on the specific gravity values reported by Far West Mining to convert volumes to weights, using Fe concentration as the independent variable

• The QA/QC protocols have remained largely consistent throughout all programs conducted by Far West Mining and Capstone

• RPA considers that sampling methods are acceptable, meet industry-standard practice, are appropriate for the mineralization style, and are acceptable for Mineral Resource estimation. The quality of the analytical data is reliable and analysis, and security are generally performed in accordance with exploration best practices and industry standards.

25.5 Data Verification

• Regular data verification programs have been undertaken by third-party consultants from 2005 to date on the data collected in support of technical reports on the Project

• RPA considers that as a result of this work, the data verification findings acceptably support the geological interpretations and the database quality, and therefore support the use of the data in Mineral Resource estimation.

25.6 Metallurgical Testwork

• Metallurgical testwork completed includes physical characterization; conventional sulphide flotation using fresh and sea water; settling and filtration tests on the copper concentrate; magnetic separation of magnetite; and settling and filtration tests on the magnetite concentrate. Settling testwork was also completed on final flotation tailings

• The variability testwork results from 2014 were used to develop a copper recovery algorithm based on the feed copper grade. The copper recovery algorithm is Global Cu Recovery = 1.00 x 0.993 x (2.844 x Ln (Cu%) + 92.15). The average for the variability data set gave a copper head grade of 0.38% Cu, with a recovery of 89% and a concentrate grade of 30.5% Cu

• Using the composite and variability sample testwork results, an algorithm was developed relating the magnetic susceptibility to iron mass recovery. If MagSus ≥2,000 the algorithm is Mass Recovery of Fe = 0.0011 x (MagSus) – 3E-09 x (MagSus)². If MagSus <2,000, then the mass recovery is measured as zero. An average value of 65% Fe was used with the mass recovery algorithm to determine the total tonnes of magnetite concentrate
- The copper concentrate specifications for other elements are expected to be below penalty thresholds
- No final market specification has been concluded with an end-user for purchase of the magnetite concentrates produced. Target specifications were developed by Capstone for use in the 2014 Feasibility Study. The majority of concentrate samples returned elemental grade values within target specification and indicate that a marketable concentrate within specification can be produced
- To quantify any potential impact on the magnetite concentrate marketability from inclusion of ANDE material in high mill feed proportions, additional targeted variability testwork is recommended to understand magnetite concentrate variations in specific zones of ANDE lithology (e.g. near barren dyke alterations) with respect to iron and silica relationships.

25.7 Mineral Resource Estimates

- The Mineral Resource estimates for Santo Domingo Sur, Iris and Iris Norte were completed in 2012. The estimate for Estrellita was conducted in 2007
- Modelling included construction of 3D wireframes that incorporated consideration of mineralized zones, fault structures and topography. In some places a nominal grade shell boundary was used. CMD data were used to model void spaces arising from artisanal mining activity
- A grade capping strategy was utilized. For Santo Domingo Sur, Iris, and Iris Norte, copper was capped at 3.5 % Cu, and gold at 0.52 g/t Au. In total, 24 Cu and 27 Au assay intervals were capped at Santo Domingo Sur, Iris and Iris Norte. These intervals represent approximately 0.2% of the total number of assays. Grades at Estrellita were capped at 3% Cu and 0.3 g/t Au
- Samples from Santo Domingo Sur, Iris and Iris Norte were composited in down-hole intervals of 4 m; those from Estrellita were composited to 2 m intervals
- Grades for Cu, Au, Fe and magnetic susceptibility were interpolated into each block using OK for the Santo Domingo Sur, Iris and Iris Norte deposits. For Estrellita, OK was utilized to interpolate Cu and Au grades into each block; iron was not estimated. An ellipsoidal search strategy with limits on the number of composites and the number of composites from any one drill hole was employed
- Resource confidence categories are based on drill hole spacing
- Reasonable prospects for eventual economic extraction were assessed for the Santo Domingo Sur, Iris and Iris Norte deposits using the parameters from the 2008 PEA. At the 0.25% CuEq cut-off, all but 5% of the Mineral Resources were captured by the pit shell. On the basis of this result, it was concluded that there
was little merit in restricting the Mineral Resources to those blocks contained only within the pit shell. In RPA’s opinion, the shape and depth of the Mineral Resources have not changed since the previous estimate and it is still valid to consider them as having reasonable prospects of economic extraction by open pit mining.

- The Estrellita resource estimate is not constrained within a LG shell. RPA’s opinion was that a 0.3% Cu cut-off would be appropriate for the reporting of the estimate. At the time of the estimate in 2007, RPA considered that the 0.3% Cu cut-off was similar to that used in other operations of similar size and grade.

- Measured and Indicated Mineral Resources total 514 Mt grading 0.31% Cu, 0.040 g/t Au and 25.8% Fe. Inferred Mineral Resources total an additional 58.1 Mt grading 0.20% Cu, 0.026 g/t Au and 24.3% Fe.

- Risk factors that could potentially affect the Mineral Resources estimates include the following: long-term commodity price and exchange rate assumptions, changes in the assumptions used in the LG shell constraining mineral resources at Santo Domingo Sur, Iris, and Iris Norte, the assumed mining methods and cost assumptions for the Santo Domingo Sur, Iris, and Iris Norte deposits being those of the 2008 PA and not the 2014 Feasibility Study, no LG shell being employed to support reasonable prospects at Estrellita, delays or other issues in reaching agreements with local communities, changes in permitting, surface rights and environmental assumptions.

25.8 Mineral Reserve Estimates

- Open pit Mineral Reserves were constrained within an LG shell. An internal cut-off of US$7.84/t milled was applied to all of the Mineral Reserve estimates. This internal cut-off was applied to material contained within an economic pit shell where the decision to mine a given block was determined by the pit optimization. Marginal ore was calculated for the same US$7.84/t cut-off, but for a NSR determined at higher metal prices.

- To accommodate selective mining methods, any resource block with an ore percentage that was <10% was treated as waste. Blocks with an ore percentage that was higher than 90% were diluted with waste such that all high-ore blocks were considered to contain only 90% ore. Selective mining will be performed on those blocks that have an ore percentage of between 10% and 90%.

- A mine plan was developed for the Project to process 65/60 kt/d (21.9 to 23.7 Mt/a) with a peak total mining rate of 107.5 Mt/a. Inferred Mineral Resources were treated as waste in the mine plan.
NCL performed pit optimization and mine planning without introducing any additional factors to account for dilution, as the block model was considered to be a fully diluted resource model. NCL considered a 100% mining recovery to be appropriate due to the disseminated characteristics of the ore.

Proven and Probable Mineral Reserves total 391.7 Mt grading 0.30% Cu, 0.04 g/t Au and 28.2% Fe.

The main factors that may affect the Mineral Reserve estimates are metallurgical recoveries and operating costs (fuel, energy and labour). The base price of the metals, even though the most important factor for revenue calculation, does not affect the Mineral Reserves estimate. The low metal price effect is due to the fact that the selected LG shell used as the guide for the practical design was based upon a 0.86 revenue factor for both the Santo Domingo and Iris Norte pits. This selected revenue factor was conservative and as such allows for a broad swing in metals pricing before any effect on the Mineral Reserves estimate occurs.

25.9 Mine Plan

Pit designs for the Project were based on optimized LG shells at a revenue factor of 0.86 with variable overall slope angles according to geotechnical domains ranging from 38° to 44°.

Seven pit phases are planned; four for Santo Domingo and three for Iris Norte.

The mine plan will process 60,000 t/d to 65,000 t/d of feed (21.9 to 23.7 Mt/a) with a peak total mining rate of 107.5 Mt/a in Years 1 to 4.

Mill throughput will be restricted to the maximum magnetite concentrate production of 4.5 Mt/a up to Year 10; and to 5.4 Mt/a from Year 11 onward.

The 15 month pre-production period requires the mining of 45 Mt of total material to expose sufficient ore to start commercial production in Year 1.

The total mined waste considers two main destinations for the material (the main waste storage areas and the tailings storage facility for the embankment construction) and top soil stripping and disposal to a specific stockpile location.

The mined ore will be hauled to the primary crusher for direct tipping. Marginal ore will be mined and hauled to a stockpile located between the Santo Domingo and Iris Norte pits until Year 13. This material will be re-handled and will become part of the plant feed in the later years. From Year 14 on, the marginal ore will be sent directly to the plant.
The mine is scheduled to operate seven days per week or 365 days per year. Each day will consist of two 12-hour shifts. Four mining crews will rotate (two working and two on time off) to cover the operation.

The 2014 Feasibility Study assumes that the mining operation will use 42 m³ hydraulic excavators and trucks with a capacity of 290 t. The fleet will be complemented with drill rigs for ore and waste delineation. Auxiliary equipment will include track dozers, wheel dozers, motor graders and a water truck.

Mining and port activities are expected to be able to be conducted on a year-round basis.

25.10 Process

- The process route envisages a DSAG/SAB comminution circuit. Copper will be recovered in a flotation circuit. Magnetite will be recovered through a magnetic LIMS concentrator.
- A two-stage tailings thickener process will be utilized prior to the tailings materials being deposited in the TSF.
- Copper concentrate will be trucked to the shipment port; magnetite concentrate will be transferred by pipeline.
- The process plant water requirements will be provided from sea water, pumped from the port to the plant site.
- The nominal capacity of the process plant for the first five years is 65,000 t/d or 23.7 Mt/a excluding the ramp up period. The nominal capacity after the first five years is 60,000 t/d or 21.9 Mt/a.
- The highest production rate of copper concentrate occurs during the first full year of production and is 315.9 Mlb of contained copper. The highest production of magnetite concentrate is 5.4 Mt which occurs in Years 15, 16 and 17.

25.11 Infrastructure

- The planned route for transporting cargo, staff and equipment to the Santo Domingo site is from the south by Route C-17, and from the north by Route C13. Approximately 13 km of roads will be built on the Santo Domingo site in order to connect the plant areas.
- The principal infrastructure will include:
  - Santo Domingo plant site: Located at approximately 26°28'00"S and 70°00'30"W.
Operations camp: Located near Diego de Almagro, 7 km from the Santo Domingo site just off Route C-7
Port facilities: Located about 43.5 km north of Caldera at Punta Roca Blanca
Sea water pipeline: 111.6 km long from Punta Roca Blanca pump station to the Santo Domingo plant site location
Concentrate pipeline: 111.6 km long from the Santo Domingo plant site location to the Santo Domingo port site at Punta Roca Blanca
High voltage transmission line: from Diego de Almagro to the proposed mine and plant site
High voltage transmission line: from Totoralillo to the port.

- Three waste rock storage areas, to be located to the west and south of the pits, were designed for the Project. The WRFs show a moderate to low potential for generation of acid rock drainage
- The proposed locations of the marginal ore and oxides stockpiles are between the Santo Domingo and Iris Norte pits
- The TSF will be constructed from compacted, non-acid generating mine waste rock with a geo-membrane liner installed on the upstream dam wall face. Tailings will be thickened to 67% solids prior to deposition. A final capacity of approximately 314 Mt of copper and iron tailings, equivalent to a total volume of about 196 Mm³, is proposed to be deposited over the approximate 18 years of operations
- Mineral processing will use sea water without desalination; there will be desalination plants at the mine site and at the port. The nominal requirement for sea water is 334 L/s. The water requirement during the construction phase will be provided by Aguas Chañar with a maximum of 15 L/s.
- Supernatant water will be recoverable throughout the operational life of the facility, and will be pumped to the process facility
- Capstone has committed to provide 10 L/s of potable water to Diego de Almagro. The 20 L/s figure used in the water balance model includes an allowance for possible future water needs
- Copper concentrate will be delivered to the port by concentrate haul trucks. Magnetite concentrate will be delivered by pipeline
- The approximately 111 km long concentrate and sea water pipeline route was optimized using a single right of way, is designed to run parallel to the existing roads, and uses existing ROW access to avoid the construction of new roads
• The sea water supply is designed to deliver between 375 L/s (nominal) and 408 L/s (design). The water is primarily supplied from the sea water supply system, and assumes that 45 L/s will be reclaimed from the TSF.

• The magnetite concentrate pipeline design maximum capacity is 415 m³/h at 68% concentration.

• Based upon current Capstone concentrate production requirements, the maximum required annual port capacity is 5.5 Mt/a of magnetite concentrate and 0.52 Mt/a of copper concentrate. Magnetite concentrate is planned to be shipped using a mixture of Panamax and Cape size vessels. Copper concentrate would be shipped using Panamax and Handymax size vessels. The selected nominal loading rate will be 4,000 t/h for magnetite concentrate and 2,000 t/h for copper concentrate.

• Capstone’s mine site and port site will be connected to the SIC. The closest connection point between the SIC and the mine site is via a direct connection to the Diego de Almagro substation, located about 5 km from the mine area.

• Capstone’s power supply strategy is consistent with the Chilean market. Capstone plans to sign a PPA with either an independent provider currently operating in the SIC, or one who will be connected with the SIC when Capstone needs power. No PPA is currently in place.

• Current solar project developments in the area may provide an opportunity for Capstone to obtain power for the Project at affordable prices.

25.12 Marketing

• Kores is required to purchase 50% of the annual production of copper concentrate and iron ore concentrate, leaving Capstone to market and sell the remaining 50%. The Kores terms and conditions will reflect the Capstone terms negotiated independently in the market.

• The Project will produce a high magnetite ultra-fine (UF) iron ore concentrate, suitable for pellet feed. In the opinion of CTAG, Capstone will need to shortlist a number of potential pelletizing plants that can process their iron ore concentrate as a starting point of a campaign to contract a suitable long term offtaker. A number of potential buyers should be contacted in order to negotiate meaningful MoUs and eventually contracts.

• Capstone will need to focus their marketing efforts towards the new or planned pellet plants in China, or build their own pellet plant in Chile near the port from which the material will be shipped.
• Although more difficult, because it is a speciality product, Capstone should also explore the option of contracting long term with one or more traders who may be able to sell the material to a specific end user on a long-term contract

• The Project copper concentrate would generally be considered clean, and the concentrate will not have any penalty elements that will be consistently above the normal thresholds. Chlorine and fluorine are under the limits at which penalties are normally applied, and if they are occasionally over the limit it is likely that only a nominal penalty would apply

• Each steel mill complex has its own level of tolerance in terms of impurities in magnetite concentrates. The main levels of impurities as far as the Santo Domingo magnetite concentrate is concerned are silica and copper. Copper is below the threshold but may in some circumstances represent a non-preferred feed; silica is only likely to be a cost factor or penalty element rather than a rejectable quality issue

• CTAG considers that Capstone’s concentrate from the Santo Domingo Project can be expected to be in high demand from trading companies specialising in blending complex materials with clean materials

• No contracts are currently in place for Capstone’s 50% of the production for either the copper or iron ore concentrates.

25.13 Environment, Social and Permits

• Baseline studies were carried out for communities in the area of influence of the Project and included Diego de Almagro, Inca de Oro, El Salado, Chañaral, Flamenco, Torres del Inca, Obispito and Caldera

• Studies included collection of physical, marine, biotic, and anthropological environment baseline data. Areas which may become important during the environmental evaluation process and in the processing of some sectorial permits include the impact of the Project on water resources, air quality, and the marine and human environments

• A provisional closure plan has been developed. As the Project will produce >10,000 t/month, Capstone will be required to post a guarantee. The guarantee must be paid within the first two-thirds of the estimated life of the Project if the mine life is less than 20 years

• A total of 752 permits will be required in support of the Project, including 462 for the mine/plant area, 119 for the pipeline, 144 for the port, and 27 for the power transmission lines. Some permits have been classed as critical because of the strategic importance to ensure the construction and start-up of the Project
The submission of the EIA to the SEIA for approval was completed on 30 October 2013. It is assumed that the EIA approval (Environmental Qualification Resolution, Resolución de Calificación Ambiental, RCA) will take 15 months. This sets the timing for the application for and granting of critical permits and all sectorial permits required for the Project.

The main strategic risk to the permitting assumptions is not having a permit plan for the Project that is integrated into the master schedule.

A stakeholder identification study has been completed, and has identified a number of parties that are in the direct and indirect area of Project influence. A series of communications have been undertaken, and include open houses, open meetings, themed meetings for specialist interests, and meetings with authorities, regional and community.

25.14 Capital Cost Estimates

- The estimate is a Type 3 estimate according to AMEC and ACCE standards, with an accuracy of -10 to +15% at the 85% confidence level.
- All capital cost estimates are in third-quarter 2013 US dollars.
- During the estimation period, the exchange rate assumptions were revised.
- The estimates are based on a combination of direct quotes, benchmarking and Capstone-supplied data.
- The initial capital cost estimate is $1,750.7 M. The estimated sustaining capital cost is $376.3 M. The combined initial and sustaining capital costs for the life of mine are estimated to be $2,127 million.
- When updated for the change in foreign exchange rate assumptions, the final capital cost estimate is $1,699.8 M. The final estimated sustaining capital cost is $368.4 M. The combined final initial and sustaining capital costs for the life of mine are estimated to be $2068.2 M.

25.15 Operating Cost Estimates

- The estimate is considered to be feasibility study level with an accuracy of -10% to +15%.
- The estimates are based on a combination of direct quotes, benchmarking and Capstone-supplied data.
- Initial total process operating costs over the LOM are estimated at $2,753.4 M; copper concentrate transport costs at $54.5 M; G & A costs at $439.6 M, and total...
mining costs at $2,513.4 M. The initial total operating cost over the LOM is an estimated $5,760.9 M.

- Final total process operating costs over the LOM are estimated at $2,725.7 M; copper concentrate transport costs at $54.5 M; G & A costs at $439.6 M, and total mining costs at $2,471.9 M. The initial total operating cost over the LOM is an estimated $5,691.7 M.

- The life of mine operating cost is a negative $0.06 per pound of copper produced

25.16 Economic Analysis

- Based on the assumptions in this Report, the Project has a positive cashflow.
- On an after-tax basis, the cumulative net cash flow for the base case is $3,226.7 M, the IRR is 17.9% and the payback period is 4.2 years.
- At an 8% discount rate, the after-tax NPV is $797.4 M.

25.17 Risks and Opportunities

- The most significant risks facing the Project include:
  - New taxes or royalties
  - Rising capex
  - Lack of appropriate human resources
  - The infrastructure and utilities required to support the Project are not secured
  - Community opposition
  - Electrical supply infrastructure and capacity not sufficient to meet Project demand
  - Lack of water during construction
- Opportunities will be evaluated during the detailed design period

25.18 Conclusions

Under the assumptions used in the 2014 Feasibility Study, the Project shows positive economics.

The QPs consider that the scientific and technical information available on the Project can support proceeding with detailed mine design phase studies. However, the choice to proceed to a mining decision on the Project is at Capstone’s discretion.
26.0 RECOMMENDATIONS

26.1 Introduction

The work program recommendations are based on a one-phase work program. The work indicated in the program can be conducted concurrently, and the findings/results from the recommended work can be directly used in more detailed engineering studies. Overall the program estimate has a total range from $1,595,000 to $2,595,000.

26.2 RPA Recommendations

RPA recommends that the Mineral Resource estimate for the Estrellita deposit be updated. This should use all available information, including consideration of the parameters and assumptions in the 2014 Feasibility Study in supporting assessments of reasonable prospects of eventual economic extraction and in the development of the copper equivalency formula.

With the updated metallurgical testwork available, a study should be undertaken to better characterize the ore types in the model for metallurgical purposes, including more definition of the oxide material types.

This work is budgeted at about $75,000 to $85,000, assuming a third-party consultant performs the work.

26.3 AMEC Recommendations

AMEC recommends that bench-scale kinematic analyses and bench reliability studies be undertaken in support of more detailed geotechnical designs for the planned open pits. Details of the final factor of safety (FoS) and probability of failure have not been established for the west wall of the Santo Domingo Sur pit, and should also be undertaken in support of detailed geotechnical design. No geotechnical domains have been established for the Iris Norte pit design, due to a lack of data. Additional data is required to adequately characterize structures required for bench scale kinematic analyses. These programs will include a combination of site investigations, data and stability analyses and design recommendations, and are estimated to cost approximately $180,000 (excluding any additional drilling and logging).

AMEC notes that the geotechnical consultant, Derk, has identified some individual lifts on the WRF that may be seismically unstable. A review of these lifts should be undertaken and a mitigation plan developed. This work is estimated at $20,000.
AMEC recommends that Capstone commissions a field data collection program to establish the site-specific sea conditions for the Santo Domingo port. Further berth alignment analysis is required to estimate port availability for ship loading operation and its impact on the upstream operation. Based on this information, dock design and concentrate loading and storage arrangements should be further optimized. Consideration should also be given to reviewing the potential impact of the port non-availability on the shipping concentrate scheduling during the winter season. The required sufficient length of the stockpile for reclaiming activity should be verified during operations and will be primarily affected by ship arrival frequency; concentrate production; and stockpile stacking height and length. This work is estimated at $300,000.

Bench-scale metallurgical testwork and a pilot plant testwork program are recommended to optimize plant operation and potentially reduce operating costs. The following bench test programs are proposed to be conducted prior to the commencement of operation of a pilot plant:

- Conduct chemical optimization tests using various flotation frothers and collectors with the aim of reducing operating costs
- Develop a geometallurgical recovery model addressing lithologies and copper solubility levels in plant feed to optimize mine and process plant production planning
- Conduct copper concentrate grade versus copper recovery for the three composites tested during the 2014 Feasibility Study program at SGS Santiago. This will assist with optimizing copper flotation cleaner circuit residence times, reagent addition rates and targeted primary and regrind size distributions
- To further optimize the mine production plan, additional targeted variability testwork is suggested to understand magnetite concentrate variations in specific zones of ANDE lithology (e.g. near barren dyke alterations) with respect to iron and silica relationships.

The recommended pilot plant is designed to be conducted prior to the completion of the basic engineering stage to:

- Optimize the metallurgical parameters such as chemical addition types and amounts, residence time requirements and grind and regrind sizes
- Produce sufficient copper and magnetite concentrates for market evaluation
- Provide a training opportunity for process plant staff and hence optimize the process plant ramp up
Further define the optimal marketing conditions for the magnetite concentrate with the goal of reducing operating costs. The bench-scale testwork is estimated at $300,000–$500,000. Depending on the number of physical tests completed as part of the plant optimization runs, the pilot plant costs could range from $300,000 to $1 M.

26.4 BRASS Recommendations

BRASS has made the following recommendations in support of detailed Project design:

- Improve soil characterization by increasing geotechnical survey and soil mechanics testing along the pipeline route. This will provide better estimates for excavation and earthmoving works

This work is estimated to require $200,000 to complete.

- Improve hydrologic surveys to define river scour depths and improve special crossing design

The budget estimate for completion of this work is $75,000.

- Schedule additional slurry testing on representative samples with the target concentrate size grind and with simulated amount of flocculants (if any is used) on the final slurry product. This can improve pump sizing and pipe wall thickness distribution along the pipeline

This work is estimated at $60,000.

26.5 Knight Piésold Recommendations

The following recommendations are made by Knight Piésold in support of Project detailed design.

- Revise water balance model and analysis with updated slurry solids contents and density to determine final limits of geomembrane liner within the TSF impoundment

Depending on any future liner size increase, this could result in as much as an additional $2 M cost to the projected construction costs.

- Perform tailings tests on new samples including:
  - Rheology
  - Atterberg limits
  - Specific gravity
- Particle size distribution (with hydrometer)
- Undrained settling
- Drained settling
- Air drying
- Coefficient of permeability
- Slurry consolidation – low stress
- Slurry consolidation – high stress
- Triaxial (multistage)

This testwork is estimated to require a budget of $25,000.

- Perform a large triaxial test in a representative sample of construction materials for the dam since currently the Chilean authorities are asking for backup tests for parameters used in design.

A budget of $50,000 will be required to support the large triaxial testwork planned.

- Integrate the results of the tailings and large triaxial tests into the overall TSF design.

A range of $10,000–$100,000 is provided for the integration of the results of the testwork into the overall TSF design. The range provides scope for some redesign work on the TSF if the testwork indicates it is necessary.
27.0 REFERENCES

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