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

Prepared by:
Ms Joyce Maycock, P. Eng., Wood
Mr. Antonio Luraschi, CMC, Wood
Mr. Marcial Mendoza, CMC, Wood
Dr. Mario Bianchin, P. Geo., Wood
Mr. David Rennie, P. Eng., RPA
Mr. Carlos Guzman, CMC, NCL
Mr. Roger Amelunxen, P. Eng., Aminpro
Mr. Michael Gingles, QP MMSA, Sunrise Americas
Mr. Tom Kerr, P. Eng., Knight Piésold
Mr. Roy Betinol, P. Eng., BRASS.

Prepared for:
Capstone Mining Corp.

Effective Date:
26 November, 2018

Project Number:
M40387

CERTIFICATE OF QUALIFIED PERSON

Av. Apoquindo 3846, Piso 15
Las Condes, Santiago
7550123, Chile

I, Joyce Maycock, P.Eng., am employed as a Project Manager with Amec Foster Wheeler Ingeniería y Construcción Ltda (doing business as Wood), located at Av. Apoquindo 3846, Piso 15, Las Condes, Santiago, 7550123, Chile.

This certificate applies to the technical report titled "Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study Update" that has an effective date of 26 November, 2018 (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 Santo Domingo; for feasibility studies for Mina Justa, Minsur and Zafranal, CMZ in Peru; for a feasibility study for El Espino, Pucobre; for a prefeasibility study for Lobo Marte, Kinross; a feasibility study for Maricunga, Kinross; a feasibility study for Angostura, Greystar; a prefeasibility and feasibility study 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 to 1.3, 1.15.1, 1.22 to 1.24; Section 2.1 to 2.3 and 2.6 to 2.8; Section 3.1 and 3.2; Section 4; Section 5; Section 13.1.7; Sections 18.1, 18.2.3, 18.6, 18.9, 18.10; Section 23; Section 24; Sections 25.1, 25.11, 25.17; Section 26.1; and Section 27 of the technical report.

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

I have previously co-authored a technical report on the project, entitled:

- Maycock, J., Gopfert, H., Rennie D., Guzman, C., Frost, D., Kerr, T., Betinol, R., Klimek, A., and Khera V., 2014: Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study: technical report prepared by AMEC International Ingeniería y Construcción Limitada, NCL Ltda, Roscoe Postle Associates Inc., Knight Piésold and Co., and BRASS Chile SA, effective date 22 May, 2014

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 the technical report not misleading.

Dated: 3 January, 2019

“Signed and sealed”

Joyce Maycock, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Av. Apoquindo 3846, Piso 15
Las Condes, Santiago
7550123, Chile

I, Dr. Antonio Luraschi, CMC, am employed as a Manager of Metallurgical Development with Amec Foster Wheeler Ingeniería y Construcción Ltda. (doing business as Wood), located at Av. Apoquindo 3846, Piso 15, Las Condes, Santiago 7550123, Chile.

This certificate applies to the technical report titled "Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study Update" that has an effective date of 26 November, 2018 (the "technical report").

I am a Registered Member of the Chilean Institute of Mining Engineers and a Qualified Person (persona competente) with the Chilean Mining Commission, registration number #0188. I graduated from University of Concepcion, Chile, as a Chemical Professional Engineer and Metallurgical Professional Engineer in 1971, and obtained M.Sc. (1973) and Ph.D. (1976) degrees from the Massachusetts Institute of Technology in the United States.

I have practiced my profession for 42 years. I have been directly involved in a number of mining and metallurgical projects, specifically in their process and project development and financial analysis, including financial evaluation of several similar mining development projects in recent 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 Project site.

I am responsible for Sections 1.1, 1.15.5, 1.16, 1.18 to 1.21, 1.23 to 1.24; Sections 2.1 to 2.3, 2.5; Section 3; Section 18.11; Section 19; Section 21; Section 22; Section 25.12, 25.14 to 25.16, 25.18; and Section 27 of the technical report.

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

I have been involved with the Santo Domingo report during preparation of an updated 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 the technical report not misleading.

Dated: 3 January, 2019

"Signed"

Dr Antonio Luraschi, CMC.

wood.

CERTIFICATE OF QUALIFIED PERSON

Av. Apoquindo 3846, Piso 15
Las Condes, Santiago
7550123, Chile

I, Marcial Mendoza, CMC., am employed as a Supervising Engineer Process and Technology with Amec Foster Wheeler Ingeniería y Construcción Ltda (doing business as Wood), located at Av. Apoquindo 3846, Piso 15, Las Condes, Santiago, 7550123, Chile.

This certificate applies to the technical report titled "Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study Update" that has an effective date of 26 November, 2018 (the "technical report").

I am a Qualified Person (persona competente) with the Chilean Mining Commission, registration number #0175. I graduated from University of Concepcion, Chile, as a Metallurgical Professional Engineer in 1984.

I have practiced my profession for 34 years. During this time I have been directly involved in, and Supervised, the design of metallurgical testwork programs and pilot plant testing, in designing process Flowsheets and selection of mineral processing equipment. I have been directly involved in process Engineering design and construction for copper projects in Chile.

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, 1.8, 1.14.1, 1.14.2 to 1.14.9, Sections 2.1 to 2.3; Sections 3.1, 3.2; Sections 13.1.1 to 13.1.6; 13.2 to 13.5, Section 17; Section 18.5.2, Sections 25.6, 25.10, Section 26.4 and Section 27 of the technical report.

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

I have been involved with the Santo Domingo report during preparation of an updated 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 the technical report not misleading.

Dated: 3 January, 2019

"signed"

Marcial Mendoza, CMC.

CERTIFICATE OF QUALIFIED PERSON

#600 – 4445 Lougheed Hwy
Burnaby, BC, V5C 0E4
Canada

I, Dr Mario Bianchin, P.Geo., am employed as a Senior Associate Hydrogeologist with Wood Canada Limited (doing business as Wood), located at #600 – 4445 Lougheed Hwy, Burnaby, BC, V5C 0E4, Canada

This certificate applies to the technical report titled “Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study Update” that has an effective date of 26 November, 2018 (the “technical report”).

I am a Professional Geoscientist with Engineers and Geoscientists of British Columbia, membership #39051. In addition, I am a Professional Geoscientist with the Association of Professional Engineers and Geoscientists of Alberta, membership #201901. I graduated with a PhD in Hydrology from the University of British Columbia in 2010.

I have practiced my profession for 18 years. I have been directly involved in environmental baseline studies, water and water management scopes for mining projects since 2010.

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 1.1; 1.17.1, 1.17.2, 1.17.3, 1.17.5, 1.23, 1.24; Sections 2.1 to 2.3; Sections 3.1, 3.2; Section 18.5; Sections 20.1. to 20.4; 20.6, 20.7; Section 25.13; and Section 27 of the technical report.

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

I have been involved with the Santo Domingo report during preparation of an updated 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 the technical report not misleading.

Dated: 3 January, 2019

“Signed and sealed”

Dr Mario Bianchin, P.Geo..

CERTIFICATE OF QUALIFIED PERSON

Suite 501, 55 University Ave
Toronto, ON, M5J 2H7
Canada

I, David Rennie, P.Eng., am employed as an Associate Principal Geologist with Roscoe Postle Associates Inc., located at Suite 501, 55 University Ave, Toronto, ON, M5J 2H7.

This certificate applies to the technical report titled "Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study Update" that has an effective date of 26 November, 2018 (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 39 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 visited the Santo Domingo Project site on 14 to 16 June 2010 and again from 14 to 15 June 2012.

I am responsible for Sections 1.1, 1.4 to 1.7, 1.9, 1.10, 1.24; Sections 2.1 to 2.5; Sections 3.1, 3.2; Section 6; Section 7; Section 8; Section 9; Section 10; Section 11; Section 12; Section 14; Sections 25.2 to 25.5, 25.7; Section 26.2 and Section 27 of the technical report.

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

I have previously co-authored a technical report on the project, entitled:

- Maycock, J., Gopfert, H., Rennie D., Guzman, C., Frost, D., Kerr, T., Betinol, R., Klimek, A., and Khera V., 2014: Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study: technical report prepared by AMEC International Ingeniería y Construcción Limitada, NCL Ltda, Roscoe Postle Associates Inc., Knight Piésold and Co., and BRASS Chile SA, effective date 22 May, 2014

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 the technical report not misleading.

Dated: 3 January, 2019

"Signed and sealed"

David Rennie, P.Eng.

NCL Ingeniería y Construcción SpA.
General del Canto 235
Providencia, Santiago, Chile
Tel: +56 2 2651-0800
Fax: +56 2 2651-0890
ncl@ncl.cl

CERTIFICATE OF QUALIFIED PERSON

I, Carlos Guzman, CMC, FAusIMM, am employed as the Principal/Project Director with NCL SpA, located at General del Canto 235, Providencia, Santiago, Chile.

This certificate applies to the technical report titled "Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study Update" that has an effective date of 26 November, 2018 (the "technical report").

I am a 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 23 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 site on 15 October 2013 and again on 29 October 2018.

I am responsible for Sections 1.1, 1.11, 1.12, 1.13, 1.15.3, 1.15.4; Sections 2.1 to 2.4; Section 3; Section 15; Section 16; Sections 18.3, 18.4; Sections 21.1.2, 21.2.1; Sections 25.8, 25.9; and Section 27 of the technical report.

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

I have previously co-authored a technical report on the project, entitled:

- Maycock, J., Gopfert, H., Rennie D., Guzman, C., Frost, D., Kerr, T., Betinol, R., Klimek, A., and Khera V., 2014: Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study: technical report prepared by AMEC International Ingeniería y Construcción Limitada, NCL Ltda, Roscoe Postle Associates Inc., Knight Piésold and Co., and BRASS Chile SA, effective date 22 May, 2014.

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 the technical report not misleading.

Dated: 3 January, 2019

“Signed”

Carlos Guzman, CMC, FAusIMM.

CERTIFICATE OF QUALIFIED PERSON

Amelunxen Mineral Processing Ltd
41961 Ross Road (Box 296)
Garibaldi Highlands, BC, V0N 1T0

I, Roger Amelunxen, P.Eng.., am employed as a Principal with Amelunxen Mineral Processing Ltd, with an office at 41961 Ross Road Garibaldi Highlands, BC, V0N 1T0.

This certificate applies to the technical report titled "Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study Update" that has an effective date of 26 November, 2018 (the "technical report").

I am a P.Eng of the Association of Professional Engineers and Geoscientists of British Columbia I graduated from McGill University, Montreal Quebec, in 1974

I have practiced my profession for 44 years. I have been directly involved in:

- Placer Development: Dump Leaching Process Design – Gibraltar Mines-1984
- SPCC: Toquepala Dump Leaching SX-EW Design-1992
- Freeport McMoRan: 103 K Expansion Design Grasberg-1993
- BHP-Escondida : Design Phase 3.5 , Design Phase IV, Design Hamburgo Tailings
- Newmont – Minas Conga – Process plant design -2004-2008
- Freeport McMoRan: Design Grasberg Tailings Pyrite process to 2042; 2013
- Codelco – Andina – Moly plant Design - 1992
- Freeport McMoRan – Design of Bagdad Moly plant - 2018
- Quadra - Sierra Gorda – Bulk Plant Design - 2008
- Newcrest – Cadia – Moly Plant design - 2018

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, 1.8 (2018 flotation testwork), 1.24; Sections 2.1 to 2.3; Sections 3.1 and 3.2; Sections 13.2.1, 13.2.2; Section 25.6; Section 26.3; and Section 27 of the technical report.

I am independent of Capstone Mining Corp. 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 feasibility study update.

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 the technical report not misleading.

Dated: 3 January, 2019

“Signed and sealed”

Roger Amelunxen, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

7902 Glen Ridge Drive, Castle Pines, Colorado 80108, USA

I, Michael J. Gingles, am employed as the Principal of Sunrise Americas LLC with a business address at 7902 Glen Ridge Drive, Castle Pines, Colorado 80108, USA.

This certificate applies to the technical report titled "Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study Update" that has an effective date of November 26, 2018 (the "Technical Report").

I am a Qualified Professional (QP) member of the Metallurgical & Mining Society of America (MMSA) with special expertise in Mineral Valuation; membership number 01393QP. I graduated from Kings College, University of London, with a BSc. in Geology in 1984, I obtained an MSc. from the Royal School of Mines, University of London, in Mineral Exploration in 1985, and I obtained an MBA from Imperial College Business School, University of London, in 1991.

I have practiced my profession for 30 years since graduation. I have been directly involved in strategic planning, valuations, corporate development, business development and commercial contracts in mining, energy, water and infrastructure throughout my career. I have worked on four continents, both for major mining companies (Outokumpu; Placer Dome) and in the junior sector. Through my consultancy firm, I have advised a number of national and international firms on mining, energy, water, infrastructure and EPC contracts.

I have visited and participated in development projects and mines in Chile since 1992, including living in Chile between 1995-2004. I was responsible for management of key supply and infrastructure projects as Strategic Planning Manager for Compañia Minera Zaldívar between 1995-1998; I was EVP Corporate Development for Placer Dome Americas between 2001-2006, and was involved in major acquisitions and project development of the company's major copper and gold assets in Chile and the Americas; and I was President and CEO of Fortune Valley Resources between 2006-2010, a TSX Venture Exchange-listed junior mining company with a portfolio of mining assets in Chile. I have participated in the planning and negotiation of several major electricity contracts in Chile and the Americas.

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 site on 11 October 2018.

I am responsible for Sections 1.1, 1.15.2, 1.15.6; Sections 2.1 to 2.4; Sections 3.1, 3.2, Sections 18.2.1, 18.2.2, 18.12; Section 21.2.4; and Section 27 of the Technical Report.

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

I have been involved with the Santo Domingo Project since 2018, supporting strategic planning for infrastructure and key supplies required for the project, and supporting preparation of the feasibility study update.

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 the technical report not misleading.

Dated: 3 January, 2019

“signed”

Michael Gingles, QP MMSA.

Knight Piésold CONSULTING

CERTIFICATE OF QUALIFIED PERSON

1999 Broadway, Suite 600
Denver, Colorado 80202-5706
USA

I, Thomas F. Kerr, P.E., am employed as the President of Knight Piésold and Co. (USA)., located at 1999 Broadway, Suite 600, Denver, Colorado 80202-5706, USA.

This certificate applies to the technical report titled “Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study Update” that has an effective date of 26 November, 2018 (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:

Registered Professional Engineer in the State of Michigan (Registration No. 6201057916)

Registered Professional Engineer in the State of Alaska (Registration No. 10969)

Registered Professional Engineer in the State California (Registration No. C49260)

Registered Professional Engineer – Ontario, Canada (No. 90407230)

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 36 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 site on 24 October 2013.

I am responsible for Sections 1.1, 1.17.4; 1.24; Sections 2.1 to 2.4; Sections 3.1, 3.2; Section 20.5; Section 21.1.4; Section 25.11; Section 26.6, and Section 27 of the technical report.

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

I have previously co-authored a technical report on the project, entitled:

- Maycock, J., Gopfert, H., Rennie D., Guzman, C., Frost, D., Kerr, T., Betinol, R., Klimek, A., and Khera V., 2014: Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study: technical report prepared by AMEC International Ingeniería y Construcción

Knight Piésold CONSULTING

Limitada, NCL Ltda, Roscoe Postle Associates Inc., Knight Piésold and Co., and BRASS Chile SA, effective date 22 May, 2014

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 the technical report not misleading.

Dated: 3 January, 2019

"Signed and sealed"

Thomas F. Kerr, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Cerro el Plomo 5420
Las Condes
Santiago, Chile

I, Roy Betinol, P. Eng., am employed as the General Manager of BRASS Chile SA, located at [Cerro el Plomo 5420, Las Condes, Santiago, Chile.

This certificate applies to the technical report titled "Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study Update" that has an effective date of 26 November, 2018 (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 41 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 1.1, 1.24; Sections 2.1, 2.2, 2.3; Sections 3.1, 3.2; Sections 18.2.4, 18.7, 18.8; Section 21.1.6; Section 25.11; Section 26.5; and Section 27 of the technical report.

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

I have previously co-authored a technical report on the project, entitled:

- Maycock, J., Gopfert, H., Rennie D., Guzman, C., Frost, D., Kerr, T., Betinol, R., Klimek, A., and Khera V., 2014: Santo Domingo Project, Region III, Chile, NI 43-101 Technical Report on Feasibility Study: technical report prepared by AMEC International Ingeniería y Construcción Limitada, NCL Ltda, Roscoe Postle Associates Inc., Knight Piésold and Co., and BRASS Chile SA, effective date 22 May, 2014.

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 the technical report not misleading.

Dated: 3 January, 2019

“signed and sealed”

Roy Betinol, P.Eng.

IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Technical Report for Minera Santo Domingo SCM (Minera Santo Domingo) by Amec Foster Wheeler Ingeniería y Construcción Limitada, a Wood company, BRASS Chile SA, Knight Piésold S.A., NCL Ltda, Roscoe Postle Associates Inc, Aminpro Chile SPA, and Sunrise Americas LLC, 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 Foster Wheeler Ingeniería y Construcción Limitada, a Wood company (Wood) (formerly AMEC International Ingeniería y Construcción Limitada or AMEC), was commissioned by Capstone Mining Corp. (Capstone) to prepare an update (the Report) to the independent Qualified Person's (QP) Review and National Instrument 43-101 (NI 43-101) Technical Report for the Santo Domingo Project (the Project), located in the Atacama Region (Region III) of the Republic of Chile.

The Report is partly 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 Wood (then AMEC); BRASS Chile SA (BRASS; magnetite concentrate pipeline); 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. (Knight Piésold; tailings storage facility 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 2014 Feasibility Study was revised to include pilot testing conducted by Wood in 2015, updates carried out in 2018 to reflect changes in the concepts for the Project, current pricing, and the 2018 inclusion of cobalt in the Mineral Resource estimate. These updates were completed by Wood, NCL, RPA, BRASS, Knight Piésold, PRDW, David J. Trotter (iron fines marketing and pricing), Braemar ACM Shipbroking (Braemar; vessel chartering, market research and ship brokering), Aminpro Chile SPA (Aminpro; metallurgical testwork), and Sunrise Americas LLC (Sunrise Americas; electrical systems and electricity marketing). In addition, Capstone prepared an internal strategic guidance document setting out the parameter estimation basis.

The firms and consultants who are responsible for the content of this Report are, in alphabetical order, Aminpro, BRASS, Knight Piésold, NCL, RPA, Sunrise Americas, and Wood.

The Report will be used in support of Capstone's press release dated 26 November 2018, entitled "Capstone Mining Releases Positive Technical Report and Launches a Strategic Partnership Process for Santo Domingo".

The Project is held 70% by Capstone and 30% by Korea Resources Corporation (Kores). The companies use an operating entity, SCM (Minera Santo Domingo), as the Chilean holding company for the Project.

Unless otherwise noted, all dollar figures used are United States of America (US) dollars ($). The Chilean currency is the Chilean peso (CLP). Mineral Resources and Mineral Reserves are reported using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves (the 2014 CIM Definition Standards). 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.

AMEC and Amec Foster Wheeler Ingeniería y Construcción Limitada (Amec Foster Wheeler) are predecessor companies to Wood. Where work was specifically undertaken by AMEC or Amec Foster Wheeler, that name is used in the Report. For all other purposes in this Report, the name Wood is used to refer to both Wood and predecessor AMEC/Amec Foster Wheeler companies.

1.2 Project Description and Location

The portion of the Santo Domingo Project area that hosts the Santo Domingo deposits and will host the mine and plant site areas is located approximately 5 km southeast of the town of Diego de Almagro in the province of Diego de Almagro in the Atacama Region of northern Chile. The deposit area was originally part of the BHP's Candelaria project area which consisted of eight non-contiguous concessions along a north–south corridor that stretched between the town of Taltal in the north to a point about 75 km south of the city of Copiapó.

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

Capstone has two groups of concessions with a total of 116 claims which cover a total of 28,897 ha and includes the areas of the planned mine site, plant area and auxiliary facilities including the port facilities. The tenure consists of 96 exploitation concessions and 20 exploration concessions. All the concessions are held in the name of Minera Santo Domingo.

The total concession area is divided as follows:

- 27,597 ha of exploitation concessions that encompass the area where the mine, plant, construction, and operations camp, and ancillary facilities are planned
- 1,300 ha of exploration concessions that encompass the port area.

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 Wood that all concession fees were in good standing and were fully paid as of 31 October 2018, and will continue to be paid on a regular basis as due, using a formal status tracking system.

Wood was provided with information that supports that the surface land where the Project is located (in the Communities of Diego de Almagro, Caldera and Chañaral) is owned by the state, and managed and represented by the Ministerio de Bienes Nacionales.

Capstone has developed a legal strategy to obtain the necessary surface rights to cover the planned mine, plant, camps, tailings storage facility, mine waste disposal, pipelines, port and transmission lines.

Capstone currently possesses 17 registered provisional surface rights (covering 1,966 ha) and eight definitive surface rights (covering 1,732 ha). Capstone has 17 applications in progress for definitive surface rights (covering 2,574 ha). Together these easements cover 100% of facilities and infrastructure area.

The Project as currently envisaged will not require an application for water rights. The original plan in 2014 was that the water for the operation of the Project would consist primarily of sea water. However, the current plan is to use desalinated water from a desalination plant to be constructed at the port. A maritime concession has been approved which will allow the extraction of sea water for processing in the desalination plant. Water for construction will be obtained from an authorized third-party provider, Aguas Chañar S.A. (Aguas Chañar).

1.3 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

Current access to the planned mine and plant site area where the deposits are located is via the paved Pan-American Highway (Route 5 North) and a network of generally well-maintained paved roads. The deposits are about five hours travel time by road south from Antofagasta, and two hours by road north from 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 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 mine–plant site 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 Atacama Region has well established infrastructure (roads and electrical transmission systems) and capacity (labour, support services) 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. Highway C-17 highway connecting Diego de Almagro and Copiapó is paved, and passes immediately east of the proposed mine–plant site area.

The nearby town of Diego de Almagro (population around 15,000) is connected to the regional power grid and can provide some support services for the Project. Details of the proposed infrastructure to be constructed in support of planned mining activities are discussed in Section 1.15.

1.4 History

Artisanal mining activities commenced in the general mine and plant site 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 planned mine–plant site 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 Ltd. (Far West) 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 in 2011 and completed a pre-feasibility study (2011 Pre-feasibility Study) in the same year. The 2014 Feasibility Study was commissioned in 2012 and completed in 2014. In July 2015, the Project Environmental Impact Assessment (EIA), including the mine, infrastructure, process facilities, development of a greenfields port, and iron concentrate and water supply pipelines (as outlined in the 2014 Feasibility Study), was approved by the Chilean authorities.

In late 2015, Capstone announced that it would discontinue work on the Project in response to low copper and iron prices. Capstone continued to maintain the Project holdings and community relations activities and commenced updates to the 2014 Feasibility Study in 2018 as outlined in this Report.

1.5 Geology and Mineralization

The deposits are 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 deposits lie 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 deposit 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 deposit area 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 outlined 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 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 ownership of the Project, 348 RC drill holes (90,611 m) and 50 core holes (16,275 m) were completed. Since acquiring the Project, Capstone has undertaken an additional 80 core holes (16,488 m).

Most drill 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 table 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 global positioning system (GPS) instrument. 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 also 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 atomic absorption spectroscopy (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 personnel on core samples using the water displacement method. RPA developed regression formulae based on the specific gravity (SG) values reported by Far West to convert volumes to weights, using iron concentration as the independent variable.

The quality assurance and quality control (QA/QC) protocols have remained largely consistent throughout all programs conducted by Far West 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 (CRMs), 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. No CRMs were inserted for cobalt.

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, including RPA, 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 2018.

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 composite ore types and the dominant proportion of magnetic iron (magnetite), it was decided to modify the comminution flow sheet from a semi-autogenous, ball mill, crushing (SABC) circuit that was used in the 2011 Pre-feasibility Study 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 flow sheet and conditions. The composite sample testwork was followed by open cycle tests (OCT) and locked cycle tests (LCT).

A pilot plant was operated in 2015 to produce concentrate for testwork and also to verify design criteria. Composites were prepared from drill core from the 2014–2015 drill program to represent each of the first five years of operation and a combined composite. The pilot plant used sea water and the flowsheet for copper and iron was the flowsheet current at the time.

Based on the decision to use desalinated water in the process, copper flotation testwork was carried out in 2018 to develop recovery algorithms for copper and gold based on the feed copper grade. The copper head grade was used to predict the copper and gold recoveries using the following recovery algorithms:

$$\textit{Global Cu Recovery} = 0.98 * 96.9018*(\textit{Feed, \% Cu})^{0.0199}$$

The factors included in the model represent the following:

- 0.98 = copper cleaning recovery factor from Aminpro testwork
- 96.9018 and 0.0199 = optimized constants of the potential equation.

$$\textit{Global Au Recovery} = 0.85 * 82.646*(\textit{Feed, \% Cu})^{0.1611}$$

The factors included in the model represent the following:

- 0.85 = gold cleaning recovery factor from Aminpro testwork
- 82.646 and 0.1611 = optimized constants of the potential equation.

As part of the pilot testing program, samples of copper concentrate, iron concentrate and tailings resulting from the pilot plant operation were sent to equipment manufacturers for testing. The aim of the testing was to determine the filtration, settling and rheology parameters for the slurries.

Testwork was conducted by three laboratories, ALS Chemex, Studien-Gesellschaft für Eisenerz-Aufbereitung (SGA), and Compañía Minera del Pacifico (CMP) in 2009. In 2010 and 2011, Davis Tube tests (DT) and low intensity magnetic separation (LIMS) tests were used to determine the recovery of magnetite from the primary copper flotation tailings stream. The results obtained from LIMS testing 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 2011 Pre-feasibility Study 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, the Hematite composite 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 ≥2,000, the algorithm is:

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

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

Concentrate quality variability LIMS testwork completed during the 2011 Pre-feasibility Study and 2014 Feasibility Study 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 $t/hr/m^2$.

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 ceramic rotary filters. This testwork resulted in a final filter cake moisture content of 8% w/w. Based on the test results, a filtration rate of 1,300 $kg/m^2/hr$ 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 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 to achieve 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/hr/m^2. The second thickening stage will be located at the tailings storage facility (TSF) using two high-density thickeners. The second stage thickening design settling rate will be 0.5 t/hr/m^2 at an underflow density of 67% solids w/w.

A pilot plant was operated in 2015 using a composite designed to represent the first five years of operation. The plant was operated using sea water and the flowsheet current at the time. Concentrate from the pilot plant operation was tested by FLSmidth and Outotec to determine filtration and thickening characteristics.

Adjustments were made to the LIMS circuit in the pilot plant operation in order to maintain a high Fe grade and low silica content. The testwork indicated that further improvement may be possible without reducing the P80 grind size. A clear relationship was demonstrated between managing iron grade and the level of silica contamination. Further testwork is recommended.

Initial cobalt testwork was performed in 2018 by various laboratories under the guidance of Blue Coast Metallurgy Ltd. (Blue Coast). This limited testwork indicated that, under the present flowsheet, it may be possible to recover about 80% of the cobalt as a secondary concentrate. Capstone has authorized further studies to define the optimum flow sheet for cobalt processing. Additional testwork is also planned to determine the quality of the cobalt sulfate and assess the impact of any impurities that might affect marketing.

1.9 Mineral Resource Estimation

The Mineral Resource estimates for Santo Domingo Sur, Iris, and Iris Norte were completed in 2012. The estimate for Estrellita was conducted in 2007. In 2018, RPA revised the block models to include an estimate for cobalt. The economic parameters were also changed, with updates to the metal prices, revisions to the mass recovery calculation, updates to the copper equivalent (CuEq) calculation, and the addition of pit shell constraints.

RPA constructed three-dimensional (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 personnel provided raw cavity monitoring device (CMD) data from which RPA was able to construct approximate wireframe models of the void spaces.

The current wireframe models for copper/iron mineralization were used for the cobalt (Co) estimates. In RPA's opinion the interpolations could be improved by constructing separate wireframe models for the cobalt mineralization.

At Santo Domingo Sur, Iris, and Iris Norte copper, gold, and cobalt assays were capped at 3.5% Cu, 0.52 g/t Au, and 1,750 ppm Co, respectively. Grades at Estrellita were capped at 3.0% Cu, 0.3 g/t Au, and 1,000 ppm Co.

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 copper, gold, iron, and cobalt 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, with a maximum of three composites allowed from any one drill hole. For Estrellita, OK was utilized to interpolate copper, gold, iron, and cobalt grades into each block. 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 copper 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 on the block models using a Lerchs–Grossmann (LG) algorithm. The resulting pit shells were used to constrain the resource reporting. The analysis suggested that a cut-off grade of 0.125% CuEq would be appropriate for the Mineral Resource estimate.

1.10 Mineral Resource Statement

The Mineral Resource estimates and geological models were prepared by Mr. David Rennie, P. Eng., an associate of RPA. Mr. Rennie is the Qualified Person as defined under NI 43-101 for the estimate. Mineral Resources for the Project have an effective date of 31 October 2018. Mineral Resources in Table 1-1 are reported inclusive of Mineral Reserves.

Table 1-1: Mineral Resource Estimates (31 October 2018)

Deposit (Zone)	Tonnes (Mt)	CuEq (%)	Cu (%)	Au (g/t)	Fe (%)	Co (ppm)
Measured						
Santo Domingo Sur (1–4)	64	0.82	0.62	0.082	31.1	254
Iris (5–6)	2	0.42	0.39	0.047	23.6	250
Total Measured	***66***	***0.81***	***0.61***	***0.081***	***30.9***	***254***
Indicated						
Santo Domingo Sur (1–4)	224	0.54	0.31	0.043	26.6	275
Iris (5–6)	103	0.44	0.19	0.027	25.9	166
Iris Norte (7–8)	89	0.44	0.12	0.014	26.7	230
Subtotal Indicated (Santo Domingo Sur /Iris)	*416*	*0.49*	*0.24*	*0.033*	*26.4*	*238*
Estrellita	55	0.40	0.38	0.039	13.7	125
Total Indicated	***471***	***0.48***	***0.26***	***0.034***	***25.0***	***225***
Total Measured and Indicated	***537***	***0.52***	***0.30***	***0.039***	***25.7***	***229***
Inferred						
Santo Domingo Sur (1–4)	24	0.40	0.22	0.033	22.8	195
Iris (5–6)	4	0.42	0.18	0.024	26.6	125
Iris Norte (7–8)	14	045	0.09	0.009	28.1	256
Subtotal Inferred (Santo Domingo Sur /Iris)	*43*	*0.42*	*0.17*	*0.024*	*25.0*	*208*
Estrellita	5	0.32	0.31	0.030	12.3	108
Total Inferred	***48***	***0.41***	***0.19***	***0.025***	***23.6***	***197***

Notes to Accompany Mineral Resource Estimates:

1. Mineral Resources are classified according to CIM (2014) guidelines.
2. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. The Qualified Person for the estimates is Mr. David Rennie, P. Eng., an associate of Roscoe Postle Associates Inc.
4. Mineral Resources for the Santo Domingo Sur, Iris, Iris Norte, and Estrellita deposits have an effective date of 31 October 2018.
5. Mineral Resources for the Santo Domingo Sur, Iris, Iris Norte, and Estrellita deposits are reported using a cut-off grade of 0.125% copper equivalent (CuEq). CuEq grades are calculated using average long-term prices of US$3.50/lb) Cu, US$1,300/oz Au and US$99/(dmt) Fe conc. The CuEq values were calculated as noted in the text in this Report.
6. Only copper, gold, and iron were recognized in the CuEq calculation; cobalt was excluded until further studies are completed to confirm reasonable prospects for eventual economic extraction.
7. Mineral Resources are constrained by preliminary pit shells derived using a Lerchs–Grossmann algorithm and the following assumptions: pit slopes averaging 45º; mining cost of US$1.90/t, processing cost of US$7.27/t (including G&A cost); processing recovery of 89% copper and 79% gold; and metal prices of US$3.50/lb Cu, US$1,300/oz Au, and US$99/dmt Fe concentrate.
8. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade, and contained metal content.
9. Tonnage measurements are in metric units. Copper and iron are reported as percentages, gold as grams per tonne.

Risk factors that could potentially affect the Mineral Resources estimates include the following:

- Assumptions used to generate the conceptual data for consideration of reasonable prospects of eventual 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
- Changes in interpretations of mineralization geometry and continuity of mineralization zones.

There are no known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors other than as discussed in this Report that could affect the Mineral Resource estimates.

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 2018. 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 third quarter 2018 prices and the exchange rate from Chilean Pesos to US dollars. This resulted in an estimated mining cost of approximately $1.75/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 $7.53/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 $7.53/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 m^3 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 14 November 2018. The QP 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.

Table 1-2: Mineral Reserves Statement (14 November 2018)

Reserve Category	Stage	Ore (Mt)	Ore Grade Cu (%)	Ore Grade Au (g/t)	Ore Grade Fe (%)	Contained Metal Au (koz)	Contained Metal Cu (Mlbs)	Contained Metal Magnetite Conc. (Mt)
Proven Mineral Reserves	Santo Domingo	65.4	0.61	0.08	30.9	169.9	878.5	8.2
	Iris Norte	—	—	—	—	—	—	—
Total Proven Mineral Reserves		***65.4***	***0.61***	***0.08***	***30.9***	***169.9***	***878.5***	***8.2***
Probable Mineral Reserves	Santo Domingo	252.1	0.27	0.04	27.8	300.8	1,486.1	48.2
	Iris Norte	74.8	0.13	0.01	26.9	36.0	208.1	18.7
Total Probable Mineral Reserves		***326.9***	***0.24***	***0.03***	***27.6***	***336.8***	***1,694.2***	***66.9***
Total Mineral Reserves (Proven and Probable)	Santo Domingo	317.5	0.34	0.05	28.5	470.7	2,364.6	56.4
	Iris Norte	74.8	0.13	0.01	26.9	36.0	208.1	18.7
Total Mineral Reserves (Proven and Probable)		***392.3***	***0.30***	***0.04***	***28.2***	***506.7***	***2,572.7***	***75.1***

Notes to Accompany Mineral Reserves Estimate:

1. Mineral Reserves have an effective date of 14 November 2018 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$3.00/lb Cu, US$1,290/oz Au, and US$100/dmt of Fe concentrate; average recovery to concentrate is 93.4% for Cu and 60.1% for Au, with magnetite concentrate recovery varying on a block-by-block basis; copper concentrate treatment charges of US$80/dmt, U$0.08/lb of copper refining charges, US$5.0/oz of gold refining charges, US$33/wmt and US$20/wmt for shipping copper and iron concentrates respectively; waste mining cost of $1.75/t, mining cost of US$1.75/t ore, and process and G&A costs of US$7.53/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 measurements are in metric units. Copper and iron grades are reported as percentages, gold as grams per tonne. Contained gold ounces are reported as troy ounces, contained copper as million pounds and contained iron as metric million tonnes.

A revenue factor of 0.84 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 changes 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 LG shell that used a copper price of $3.00/lb and $100/t 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.

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 of the Santo Domingo pit is in the Iris area which is at the northern end of the Santo Domingo pit, but will have a separate access from the east side and will go down to 676 m elevation.

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

The Santo Domingo pit will have two exits 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 another exit to access the main waste rock 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 ROM pad area and the primary crusher. On the east side there will be an exit to access the waste rock 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 m^3 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 pre-splitting purposes.

1.14 Recovery Plan

1.14.1 Crushing and Grinding

The primary crushing plant will receive 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 (rated capacity of 450 t). The crushed product will be conveyed to the coarse ore stockpile (live capacity equivalent to six to eight hours of operation). The SAG mill will operate in a DSAG 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 hydrocyclone clusters. The hydrocyclone oversize (or underflow) fraction will return by gravity to the ball mills for further size reduction.

1.14.2 Copper Flotation

The hydrocyclone overflow streams (the copper flotation circuit feed stream) with a P80 of 150 µm will be combined and fed to a single bank of mechanical 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, which consists of a single vertical mill and hydrocyclone cluster, 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 mechanical 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 mechanical 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 mechanical 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 mechanical 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 filters.

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) 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 as required to treat increased 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) will be conducted at the TSF area. Pre-thickening of tailings will be done in high rate thickeners which deliver tailings 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 at the mine site in three ceramic disc filters. 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 surplus 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

The plant will use desalinated water from the desalination plant at the port for the process. The current plan is to negotiate for the supply of desalinated water at the mine site on a build, own, operate, transfer (BOOT) or build, own, operate (BOO) contract. The plant will be built at the port site and the desalinated water will be pumped via a pipeline to the process water ponds on site. Capstone will operate potable water treatment plants at the mine site (using desalinated water) for the potable water supply on site, and for the potable water supply to the town of Diego de Almagro.

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. These will be installed in the plant area.

1.14.8 Port Infrastructure

There will be a filter plant at the port for magnetite concentrate. Magnetite concentrate will be transported by pipeline from the mine site and 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. Initially there will be two ceramic disc filters (increasing to four by Year 5) and the magnetite concentrate filter cake product will discharge onto a conveyor feeding the concentrate transfer tower and then the magnetite concentrate stockpile.

Process water (desalinated water) required at the port will be provided from the desalination plant. Potable water at the port 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 copper and iron recovery assumes yearly average treatment rates of 65,000 t/d and 60,000 t/d, with an annual peak production of 514.1 kt of copper concentrate in the first full year of production and an annual peak production for magnetite concentrate of 4.04 Mt for the first 6 years of production and 5.40 Mt for the remaining mine life.

In Years 0 and 1 Hematite ore reaches the maximum treatment rate within the plan (about 32% of the total processed in the year). The maximum treatment rate of Magnetite ore 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 0.14% 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 around 28% 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 and temporary construction camp: Located on the mine site; part will be rented and removed after construction and the rest will be permanent
- Iron concentrate and water pipelines: 111.6 km long between the Santo Domingo plant site and 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).

1.15.2 Access Considerations

The planned route for transporting cargo, staff, and equipment to the Santo Domingo site is from the south of the mine site by Route C-17, and from the north by Route C-13.

The closest airport to the Santo Domingo site is the El Salvador Airport, a private airport, 44 km from the site. The 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 planned Santo Domingo port. The approximate distance of the haulage is approximately 117 km using the preferred route from the completed haulage study.

The proposed route for the magnetite concentrate pipeline 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 common trench with the desalinated water pipeline.

1.15.3 Waste Rock Storage Facilities

Three waste rock storage facilities (WRF) 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/m^3 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 0.47 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 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. 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 for the copper concentrate to Cape size for the magnetite concentrate. The nominal loading rate will be 4,000 t/hr for magnetite concentrate and 2,000 t/hr 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, a special berthing dolphin to support the gangway, mooring dolphins, quadrant beam (the structure that allows the ship loader radial movement), ship loader, gangway and the ship loader support platform.

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

Capstone has held discussions with major water supply companies currently operating in Chile to confirm interest to supply desalination water to the Project, from a facility at the port or from another facility, to verify commercial assumptions for the economic analyses. The current plan is that the BOOT or BOO contractor will construct and operate the sea water intake, reverse osmosis desalination plant and brine return system at the port, and the sea water pipeline as part of the BOOT contract. The contractor will purchase power from Capstone at the port.

1.15.6 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 magnetite concentrate filtration plant, magnetite and copper concentrate storage and handling and associated infrastructure, and the desalination system operation.

The peak power demand during operations is estimated to be approximately 111 MW (excluding the desalinated water system).

The mine site and port site will be connected to the national grid system (Sistema Eléctrico Nacional (SEN)). The closest electrical substations to the Project are the Diego de Almagro substation, located about 9 km from the proposed mine area, and the Totoralillo substation, located about 14 km from the port site.

Capstone is considered a non-regulated customer in the Chilean electrical sector and is not eligible to access the national electrical grid directly. Capstone must arrange its power supply through self-generation or from a generation company operating in the SEN.

There has been a significant reduction in the price of electricity in the national grid system since the 2014 Feasibility Study. The downward trend in the electricity price has been driven by lower coal and natural gas prices delivered into Chile, increased supply from new conventional generating capacity and from non-conventional renewable energy (NCRE) generating capacity, lower NCRE prices (as a result of new technology that has lowered investment costs and capacity charges), and lower than expected demand from the mining industry (due to delay in the development of many projects due to depressed copper prices).

Capstone plans to sign a power purchase agreement for a firm and continuous, long-term supply of electricity with an experienced power producer having existing power generating capacity in the SEN. Capstone has held discussions with major power companies currently operating in Chile to confirm interest and capacity to supply electricity to the Project, and to verify commercial assumptions for the economic analyses.

1.16 Marketing

Capstone requested David J. Trotter and Braemar to prepare papers on price projections, sales potential and shipping costs for the iron ore concentrates to be produced by Capstone over the LOM. These documents updated information provided by CRU and CTAG for the 2014 Feasibility Study.

Capstone has been shipping copper concentrate from the Pinto Valley Mine since 2013 and has established a reputation as a reliable supplier. Capstone's Marketing Group is experienced and will establish and implement the marketing strategy for the Santo Domingo Project's copper concentrate offtake agreements.

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

The Capstone copper concentrate would generally be considered clean; low in impurities (deleterious or penalty elements). For trading companies specializing in blending various complex copper concentrates, a clean concentrate such as that from Capstone would be in high demand. The timing to secure sales contracts would be dependent on the progress of 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 and/or sintering plants that can process the iron ore concentrate as a starting point of a campaign to contract suitable long-term offtakers. Capstone has made contact with Chinese iron ore processors in Hebei province at an early stage in order to start the process to have meaningful MOUs in place that can be developed into long-term offtake 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.

In the 2014 Feasibility Study, the CRU prediction was that on average from 2016 through to and including 2025, copper prices were expected to average $3.13/lb. CTAG agreed that this was a realistic price to be used as an average over this period. For the purposes of the economic analysis in this Report, a copper price of $3.00/lb was used based on Capstone corporate guidelines.

A forward pricing report prepared for Capstone by David J. Trotter in 2018 estimated that prices for 62% Fe content sinter fines (Platts Iron Ore Index or IODEX) cost-and-freight (CFR) Qingdao delivery (deemed the standard product for CFR China delivery) can be expected to be in the range of $62/dmt to $72/dmt over the next 10 years. This study is based on a long-term price of $69/dmt for 62% Fe concentrate. Premiums for 65% Fe concentrate ($24/dmt), value-in-use (VIU) for 66% Fe ($1.50/dmt), magnetite content ($2.50/dmt) and low alumina ($7/dmt for each 1% below 2.5%) are expected to remain relatively stable because of the direct impact on furnace productivity and decrease in emissions. This study discounted the current premiums to approximately 80%.

Braemar conducted a long-term estimate of shipping costs to include new construction and new environmental regulations on sulfide emissions. Long-term contracted prices are expected to drop from the current spot market price of $20/dmt to below $15/dmt. This study has assumed a long-term shipping cost of $20/dmt. The net result is a base case price of $80/dmt free-on-board (FOB) Chile.

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 (inhalable particulate material (PM10), inhalable 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 desalinated sea water for the mining process. In addition, no infiltration is expected from the WRFs or thickened tailings deposit to the ground water resources in the area.

Baseline studies encompassing the marine environment were completed.

In 2014, the Project included a sea water intake and brine outfall to the sea in the port area. Capstone will submit an application for an update to the EIA to cover the 2018 plan to produce all the desalinated water at the port site and for the BOOT operation of the desalination facility.

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. These mitigation measures were approved when the Project EIA was approved in July 2015.

1.17.2 Closure

A Closure Plan has been developed for the Project. The plan will be submitted following receipt of all permits for operations. The following is a summary of the key items of the 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, and disposed of in approved facilities or transported for certified disposal
- Electrical equipment, cables, and other above-ground facilities will be dismantled or demolished
- Above ground concrete structures 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, WRFs and the TSF will be restricted and warning signs posted
- Final pit and WRF slopes will be re-profiled to be structurally stable for the long term
- The TSF will be covered and an 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 LOM
- Not complying with the commitments in the Closure Plan.

Closure costs are treated in the financial analysis as operating costs, and total $102.1 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 that 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 with regional authorities, community support service authorities, and professional organizations, as well as sessions to address specialist interests (such as fishermen). These communications were continued at a lower level through 2014 and 2015 to keep the community informed about the Project.

Community issues identified during these communication sessions 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 marine food extraction activities
- Effects of sea water intake and brine discharge from the desalination plant.

During the EIA approval period the environmental citizen participation process continued as required. 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 Santo Domingo will continue to focus on building a positive reputation and supportive environment for 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 were developed and will be updated for the next phase.

1.17.4 Tailings Storage Facility

The TSF embankment will be constructed from compacted, non-acid generating mine waste rock. A 1.5 mm thick high-density polyethylene (HDPE) geomembrane liner will be installed on the upstream face and will be placed over a geotextile and a 3 m thick soil bedding layer. In the TSF basin the geomembrane will be installed beneath the area of the supernatant pond and extended 100 m beyond the expected pond limits to reduce vertical 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. The liner may be extended further than 100 m in the future detailed design stage to provide coverage for a potentially larger pond. This could be required if the tailings thickening and deposition is not always as effective as planned. The anchor trench along the upstream limits of the liner may be deepened and widened to further reduce seepage.

The embankment will be constructed in stages using the downstream method. In the final configuration, the crest elevation will be between 1,058.5 masl and 1,069.7 masl, 55.5 m maximum height. The final dam will require a total of 7.7 Mm^3 of fill. The initial dam will have a crest elevation of 1,044.9 masl, maximum height of 41 m and 2,418 m long. It will require 2.2 Mm^3 of fill. A 26.7 m high saddle dam will be constructed at the southwest limits of the TSF to provide containment at that location. Foundation preparation will involve removing loose and deleterious material and material with a high salt content that could develop solution cavities due to seepage.

A storm water diversion channel will be constructed around the perimeter of the TSF to reduce the ingress of storm water run-off into the TSF.

It is anticipated that the TSF will store 314 Mt of copper and iron tailings, equivalent to an estimated total volume of 196 Mm^3 at an overall average dry density of 1.6 t/m^3. These tailings will be deposited over 18 years.

Monitoring of the facility will include piezometers, clinometers, inclinometers, accelerometers, and three existing monitoring wells and a pumping well that were drilled during field investigations. A fibre optic data communication system will be incorporated for near-continuous and real time data reception and assessment. If any impacted water is detected in the monitoring wells, additional pumping wells may be installed as needed to intercept the source.

The water recovered will be either treated for release or returned to the TSF for re-cycling in the process. After Year 10 any ground water flows from the TSF are expected to be intercepted by the Iris Norte well system that will be installed to dewater the pit.

The tailings will be pumped from the plant after first stage thickening at 55% by mass solids content, to the second stage thickeners located at the southern end of the TSF. The second stage thickeners will produce tailings with approximately 67% solids content. These tailings will be pumped to and deposited in the TSF 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 against the embankment from where it will be recirculated to the process plant. It is possible that during start up and periodically during operations, the tailings thickening may produce a less dense slurry than the target 67%; this will result in a larger volume of water introduced into the TSF than planned. During detail design the impact of this will be assessed and accounted for.

Closure of the TSF will include removal of any supernatant water, covering the surface of the tailings with non-acid generating (NAG) granular material and installing a spillway to remove surface run-off from the low point (this will be in the north area of the TSF). The spillway will have sufficient capacity to convey the probable maximum precipitation (PMP) event throughout the post-closure phase of the facility.

1.17.5 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, and waste storage); power lines; and roads (internal and access). The construction of the BOOT sea water intake, desalination plant and pumping of sea water will be covered in the update to the EIA. 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, approximately 750 permits have been identified that will be required to support operations. Sixteen of these are considered to be critical for timely construction and start-up of the Project. In October 2018, permits had been received for the WRF, exploitation method and mining process, and the process plant.

1.18 Capital Cost Estimates

All capital costs are in Q3 2018 US$. A foreign exchange rate of 600 CLP to US$1 was used for the detailed estimate.

Capital cost estimates were prepared by the various consultants working on the 2018 update to 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 Wood standards (and AACE International), with an accuracy of -10 to +15% at the 85% confidence level.

The initial capital cost was estimated to be $1,512 M. The estimated sustaining capital cost total approximately $379 M. The combined initial and sustaining capital costs for the LOM were estimated to be about $1,891 M (Table 1-3).

1.19 Operating Cost Estimates

All operating cost estaimtes are in Q3 2018 US$. Costs are based on a foreign exchange rate of CLP600 to US$1.00. For the CuEq estimate, prices of $3.00/lb Cu, $1,290/oz Au, and $80.00/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 $5,770.0 M (excluding copper concentrate land transport).

Table 1-3: Initial Capital Cost Estimate

Area		Cost ($ M)
Initial Capital	Mine	177.5
	Process Plant	313.3
	Tailings and Water Reclaim	48.2
	Plant Infrastructure (On Site)	81.9
	Port	147.4
	Port Infrastructure (On Site)	21.9
	External Infrastructure (Off Site)	143.2
	Indirect Costs	381.0
	Contingency	197.8
Total Initial Capital		**1,512.3**
Total Project Sustaining Capital		***378.6***
Project Total Cost		***1,890.9***

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

Table 1-4: Operating Cost Estimate

Cost Centre	LOM Total ($ M)	LOM Average ($/t)	LOM Average ($/lb CuEq)
Process	*2,547.6*	*6.49*	*0.610*
General & Administrative (G&A)	*402.8*	*1.03*	*0.097*
Mining	*2,619.6*	*6.68*	*0.631*
Total	***5,570.0***	***14.20***	***1.34***

1.20 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 government approvals.

The Project was evaluated using non-inflated cash flows on an after-tax basis. Metal prices used were $3.00/lb Cu, $1,290/oz Au, and $80/t Fe (assuming 66% Fe content).

On a pre-tax basis with no discount, the cumulative cash flow for the Project is $4,666 M. On an after-tax basis the cumulative undiscounted cash flow is $3,250 M, the internal rate of return (IRR) is 21.8% and the payback period is 2.8 years. At an 8% discounted cash flow (DCF) rate, the after-tax net present value (NPV) of the Project is $1,032 M. A pre-tax summary table is included as Table 1-5. The LOM cash flow is shown in Figure 1-1.

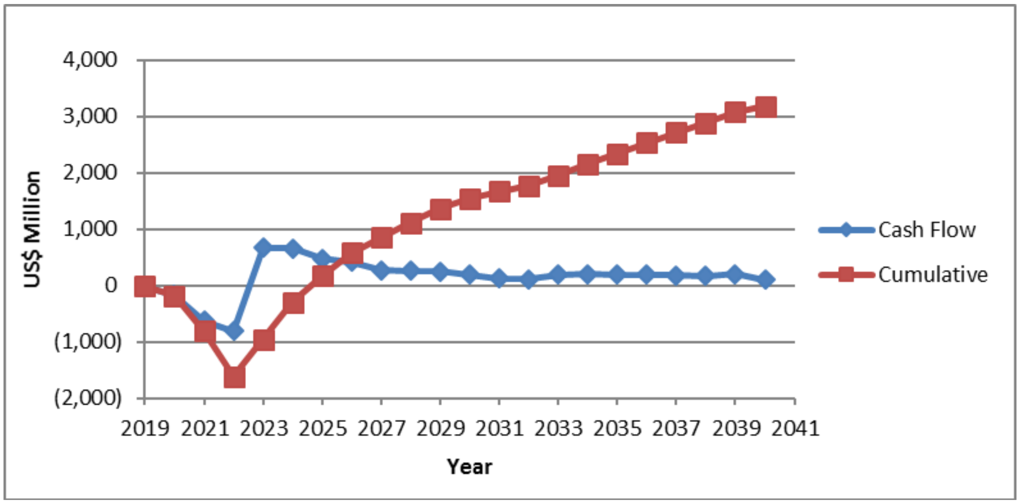
Cash costs are summarized in Table 1-6. The gold and iron credits offset the operating costs over the LOM, resulting in an almost zero C1 cash cost ($0.02/lb).

Table 1-5: Summary of Pre-Tax Cash Flow

Cost Item	LOM ($ M)	$/t milled	$/lb Cu payable
Revenue (after losses and before deductions)			
Cu	7,200.4	18.35	3.11
Au	392.6	1.00	0.17
Fe	6,005.1	15.31	2.59
Sub-Total	13,598.1	34.66	5.87
Smelting costs			
Treatment	(300.3)	(0.77)	(0.13)
Cu deduction	(252.0)	(0.64)	(0.11)
Au deduction	(155.7)	(0.40)	(0.07)
Refining – Cu	(185.3)	(0.47)	(0.08)
Refining – Au	(0.918)	(0.00)	(0.00)
Concentrate Transport	(225.3)	(0.57)	(0.10)
Sub-Total	(1,119.5)	(2.85)	(0.48)
Operating cost			
Mining	(2,619.6)	(6.68)	(1.13)
Process	(2,547.6)	(6.49)	(1.10)
G&A	(402.8)	(1.03)	(0.17)
Sub-Total	*(5,570.0)*	*(14.20)*	*(2.40)*
Other			
Royalties	(249.6)	(0.64)	(0.11)
Closure	(102.1)	(0.26)	(0.04)
Total	*(351.6)*	*(0.90)*	*(0.15)*
Earnings before interest, taxes, depreciation, and amortization (EBITDA)	6,557.0	16.71	2.83
Construction capital	(1,512.3)	(3.85)	(0.65)
Sustaining capital	(378.6)	(0.97)	(0.16)
Undiscounted margin (cumulative net cash flow)	4,666.1	11.89	2.01

Note: Totals may not sum due to rounding

Figure 1-1: Cash Flow Summary



Note: Figure prepared by Wood, 2018

Table 1-6: Cash Cost Summary LOM

Cash Costs	LOM Total ($ M)	Cost per tonne milled ($/t)	Cost per pound Cu payable ($/lb)
Costs			
Mining	2,619.6	6.68	1.13
Process	2,547.6	6.49	1.10
G&A	402.8	1.03	0.17
Treatment charges	300.3	0.77	0.13
Refining charges	186.2	0.47	0.08
Concentrate transport	225.3	0.57	0.10
Sub-Total	*6,281.9*	*16.01*	*2.72*
Credits			
Au	(392.6)	(1.00)	(0.17)
Fe	(6,005.1)	(15.31)	(2.59)
Sub-Total	*(6,397.7)*	*(16.31)*	*(2.76)*
Adjusted Cash Cost Total	***39.8***	***0.10***	***0.02***

1.21 Sensitivity Analysis

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

- Metal price (copper and iron)
- Operating costs (including electricity)
- Electricity costs alone
- Foreign exchange rates
- Capital costs.

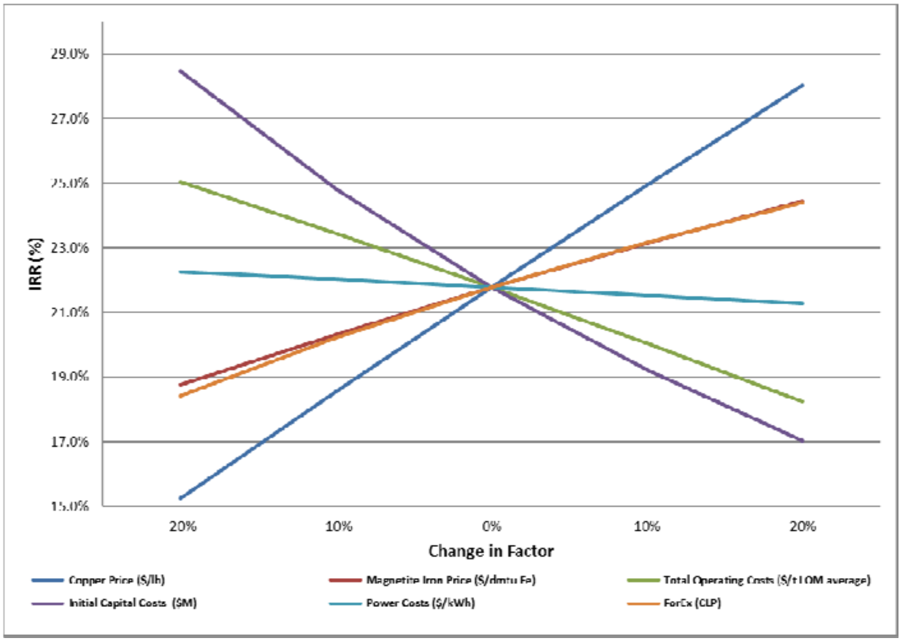
Figure 1-2 shows the sensitivity of the IRR and Figure 1-3 shows the sensitivity of the NPV8% to the variations used in the parameters listed in the bullet points above. Sensitivities to copper and iron grades are not shown, since changes in copper and iron grades are mirrored by the sensitivities to changes in the copper and iron prices, respectively. The analysis shows that the Project NPV8% is most sensitive to changes in the copper price (copper grade) and in the total capital and operating costs. The sensitivity analysis showed that the Project is less sensitive to changes in the iron price (iron price) and the dollar/peso exchange rate and least sensitive to changes in the electricity costs.

1.22 Risk and Opportunity Analysis

In 2018, Capstone reviewed the opportunities and risks associated with the current Project status. The main opportunities and risks include:

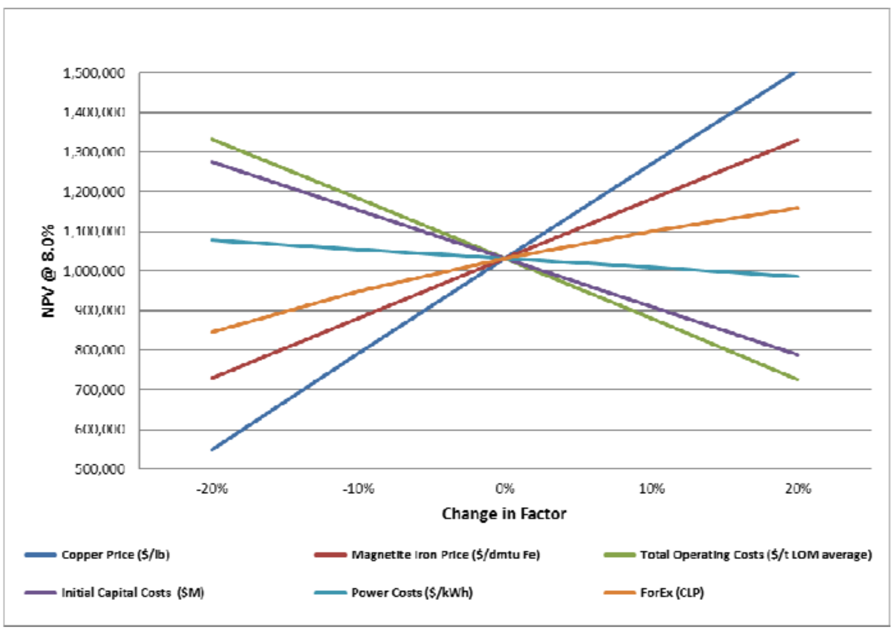
- Opportunities
  - Estimate cobalt in the Mineral Reserve
  - Enhance gold recovery through further metallurgical testing
  - Incorporate autonomous haulage
  - Share infrastructure with other local companies
  - Consider autogenous grinding.

Figure 1-2: Sensitivity of IRR



Note: Figure prepared by Wood, 2018

Figure 1-3: Sensitivity of NPV8% ($ x 1,000) for Base Case



Note: Figure prepared by Wood, 2018

- Risks
  - Delay in financing
  - Schedule delays
  - Contractor engagement and price uncertainty
  - Increased equipment and labor costs.

Risks and opportunities will be continuously assessed and reviewed throughout the various phases of the Project, in accordance with Capstone's Risk Management Framework. A risk register hosts the risk and opportunity information and is further recorded on a risk matrix according to the level of risk which is determined by the probability or occurrence (likelihood) and the resulting consequence of its impact. The level of risk assists in prioritizing actions to take advantage of opportunities and to mitigate risks.

1.23 Conclusions

Under the assumptions used in this Report, the Project shows positive economics with an after-tax IRR of 21.8%. The QPs consider that the scientific and technical information available on the Project can support proceeding with the detailed design phase. However, the choice to proceed to a mining decision on the Project is at Capstone's discretion.

1.24 Recommendations

A single work phase is proposed to improve the confidence in 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. The recommended program budget is between is between $4.6 M and $5.6 M.

RPA recommends $300,000 to $350,000 of work, including additional assaying for cobalt and a review of the mineralogy and mode of occurrence of cobalt, followed by an update of the Mineral Resource models, if required. If new mineralization wireframes are required, an additional $100,000 to $125,000 should be budgeted.

Aminpro recommends that a drilling campaign be conducted in order to generate sufficient sample material for the operation of a pilot plant to confirm the final design criteria, metallurgical results, and process alternatives for cobalt. It is estimated that the cost will be approximately $2.5 M.

Wood recommends $1.1 M to $2.0 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 recommends $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.

For detailed design of the TSF, Knight Piésold recommends additional tailings testwork, testing for soluble salts content at the dam foundation, testing of the construction material that will be used for the TSF construction, review of the hydrology to include the latest significant hydrological events in the area, and integration of these results into the TSF design. Knight Piésold also recommends further analysis to update the geomembrane liner limits within the TSF impoundment, as well as to assess the need for a deeper cut-off trench to anchor the geomembrane liner. The engineering and testing associated with this work would be approximately $595,000 to $785,000.

2.0 INTRODUCTION

2.1 Introduction

Amec Foster Wheeler Ingeniería y Construcción Limitada, a Wood company (Wood) (formerly AMEC International Ingeniería y Construcción Limitada or AMEC), was commissioned by Capstone Mining Corp. (Capstone) to prepare an update (the Report) to the independent Qualified Person's (QP) Review and National Instrument 43-101 (NI 43-101) Technical Report for the Santo Domingo Project (the Project), located in the Atacama Region (Region III) of the Republic of Chile. The Project location is shown in Figure 2-1.

The Report is partly 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 Wood (then AMEC); BRASS Chile SA (BRASS; magnetite concentrate pipeline); 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. (Knight Piésold; tailings storage facility 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 2014 Feasibility Study was revised to include pilot testing conducted by Wood in 2015, updates carried out in 2018 to reflect changes in the concepts for the Project, current pricing, and the 2018 inclusion of cobalt in the Mineral Resource estimate. These updates were completed by Wood, NCL, RPA, BRASS, Knight Piésold, PRDW, David J. Trotter (iron fines marketing and pricing), Braemar ACM Shipbroking (Braemar; vessel chartering, market research and ship brokering), Aminpro Chile SPA (Aminpro; metallurgical testwork), and Sunrise Americas LLC (Sunrise Americas; electrical systems and electricity marketing). In addition, Capstone prepared an internal strategic guidance document setting out the parameter estimation basis.

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.

The firms and consultants who are responsible for the content of this Report are, in alphabetical order, Aminpro, BRASS, Knight Piésold, NCL, RPA, Sunrise Americas, and Wood.

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.

2.2 Terms of Reference

The Report will be used in support of Capstone's press release dated 26 November 2018, entitled "Capstone Mining Releases Positive Technical Report and Launches a Strategic Partnership Process for Santo Domingo".

All measurement units used in this Report are metric, and currency is expressed in US dollars unless stated otherwise. The Report uses Canadian English. Mineral Resources and Mineral Reserves are reported using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves (the 2014 CIM Definition Standards).

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.

AMEC and Amec Foster Wheeler Ingeniería y Construcción Limitada (Amec Foster Wheeler) are predecessor companies to Wood. Where work was specifically undertaken by AMEC or Amec Foster Wheeler, that name is used in the Report. For all other purposes in this Report, the name Wood is used to refer to both Wood and predecessor AMEC/Amec Foster Wheeler companies.

2.3 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, Wood Santiago
- Mr. Antonio Luraschi, CMC, Manager of Metallurgical Development, Wood Santiago
- Mr. Marcial Mendoza, CMC, Supervising Engineer Process and Technology, Wood Santiago
- Dr. Mario Bianchin, P. Geo., Senior Associate Hydrogeologist, Wood Vancouver
- Mr. David Rennie, P. Eng., Associate Principal Geologist, RPA Vancouver

- Mr. Carlos Guzman, CMC, FAusIMM, Principal and Project Director, NCL Santiago
- Mr. Roger Amelunxen, P.Eng., Business Director, Aminpro
- Mr. Michael Gingles, QP MMSA, Principal, Sunrise Americas
- Mr. Thomas F. Kerr, P. Eng., President, Knight Piésold (USA)
- Mr. Roy Betinol, P. Eng., Regional Director, BRASS.

2.4 Site Visits and Scope of Personal Inspection

Mr. Rennie visited the Project from 14 to 16 June 2010 and again from 14 to 15 June 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 little drilling activity on the Project since 2012.

Mr. Guzman visited the Project site on 15 October 2013 and again on 29 October 2018. During the visits 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. Gingles visited the site on 11 October 2018. During the visit he reviewed the area planned for the mine site, including the open pit, process plant, and tailings storage facility (TSF) locations, reviewed the Project storage facilities in Diego de Almagro (storage of diamond core, reverse circulation (RC) drilling samples, and metallurgical samples), reviewed the area planned for the port site, and made a general review of the regional infrastructure that would be available to support the Project.

Mr. Kerr visited the Project site on 24 October 2013. During this visit he viewed the area proposed for the TSF and assessed the topography and general ground conditions.

2.5 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
- Date of Mineral Resource estimates: 31 October 2018

- Date of Mineral Reserve estimate: 14 November 2018
- Date of financial analysis: 26 November 2014.

The effective date of the Report is taken to be the date of the financial analysis, and is 26 November 2018.

2.6 References

The key information sources for the Report were the updates provided in 2018 and the 2014 Feasibility Study, entitled:

- AMEC, 2014: Definitive Feasibility Study: unpublished draft report prepared by AMEC, Knight Piésold, PRDW, BRASS, RPA, NCL, Ghisolfo, CRU and CTAG, dated June 2014, 30 vols.

The 2018 updates include:

- Wood technical memorandum, Capex Review Update for NI-43-101, 23 November 2018
- BRASS document numbers BPI13004-6150-EST-CS-001_Rev. 4, BPI13004-6120-EST-CS-001_Rev 4, BPI13004-6120-EST-CS-002_Rev 3, BPI13004-6150-EST-CS-002_Rev 3, Actualización de CAPEX/OPEX – Proyecto Santo Domingo
- Aminpro Memo_Algorithm_20181105_MSD_Aminpro (003), New Algorithm with Desalinated Water – Rougher Recovery (Copper & Gold), 5 November 2018
- Knight Piésold document number SA202-0015609.10010_Rev_4.1, Estimación de Costos
- PRDW document number C2151-2-BG-CS-001-RF y C2151-2-BG-CS-002-RE, Actualización Capex y Opex Puerto Santo Domingo
- Ghisolfo document, Estudio Definitivo De Factibilidad Estudio De Transporte De Concentrado De Cobre Proyecto Santo Domingo, 31 October 2018
- Ghisolfo document: 20180927 CAPEX REV 09, Actualización de Capex, Caminos y Accesos Santo Domingo.

Information used to support this Report was also derived from previous technical reports on the Project (Section 2.8), from the sources listed in Section 2.7, from expert documents cited in Section 3, and from the reports and documents listed in Section 27. Additional information was sought from Capstone personnel where required.

2.7 Information Sources

Mr. Tom Kerr, the Knight Piésold 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 TSF 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 14 to 17 May 2013 to select and define the proposed route for the water and concentrate pipelines. A second visit was made by Mr. Escarate on 4 June 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. Antonio Luraschi of Wood 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. Luraschi for use in the Report.

2.8 Previous Technical Reports

Capstone has filed the following technical reports on the Project:

- Maycock, J., Gopfert, H., Rennie D., Guzman, C., Frost, D., Kerr, T., Betinol, R., Klimek, A., and Khera V., 2014: Santo Domingo Project, Region III, Chile, NI 43-

101 Technical Report on Feasibility Study: technical report prepared by AMEC International Ingeniería y Construcción Limitada, NCL Ltda, Roscoe Postle Associates Inc., Knight Piésold and Co., and BRASS Chile SA, effective date 22 May 2014

- Brimage, D., Rennie, D., Nilsson, J., Winkers, A., and Davies, M., 2011: Technical Report on the Santo Domingo Project, Chile: unpublished report prepared by Ausenco Minerals and Metals, Roscoe Postle Associates Inc., Nilsson Mine Services Ltd., Arthur H. Winckers & Associates Mineral Processing Consulting Inc., and AMEC Environment & Infrastructure for Capstone Mining Corp., effective date 28 September 2011
- Rennie, D., 2010: Technical Report on The Santo Domingo Property, Region III, Atacama Province, Chile, NI 43-101 Report: unpublished report prepared by Scott Wilson Roscoe Postle Associates for Far West Mining Ltd, re-addressed to Capstone Mining Corp., effective date 26 August 26 2010.

Far West Mining Ltd. (Far West), a predecessor company to Capstone, filed the following technical reports:

- Allen, G.J., 2004: Report on the 2003 Exploration Activities in the 4c Santo Domingo Area of the Candelaria Project, Region III, Chile: unpublished report prepared by Far West Mining Ltd
- Allen, G.J., 2005: Report on the 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 31 July 2005
- 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. for Far West Mining Ltd., effective date 30 April 2008
- Lacroix, P.A., 2006: Technical Report on the 4A (Santo Domingo) Area of the Candelaria Project, Region III, Chile: unpublished technical report prepared by

Roscoe Postle Associates Inc. for Far West Mining Ltd., effective date 13 June 2006

- Lacroix, P.A., and Rennie, D.W., 2007: Technical Report on the 4A (Santo Domingo) Area of the Candelaria Project, Region III, Atacama Province, Chile: unpublished technical report prepared by Roscoe Postle Associates Inc. for Far West Mining Ltd., effective date 19 October 2007
- Lacroix, P.A., 2009: Technical Report on the Santo Domingo Property, Region III, Chile: unpublished technical report prepared by Roscoe Postle Associates Inc. for Far West Mining Ltd., effective date 4 June 2009
- Rennie, D., 2010: Technical Report on the Santo Domingo Property, Region III, Atacama Province, Chile, NI 43-101 Report: unpublished report prepared by Scott Wilson Roscoe Postle Associates for Far West Mining Ltd effective date 26 August 2010.

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, 2018: MSD Ownership Structure: Confirmation e-mail from Capstone to Wood, 19 December, 2018.

This information is used in Section 4 of the Report. It is also used in support of the Mineral Resource statement in Section 14, the Mineral Reserve statement in Section 15, and the financial analysis result in Section 22.

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, 10 October 2018
- Maria Elizabeth Orrego Espinosa (Abogados): Informe de Titulos Concesiones Mineras MLQ hoy MSD: legal opinion prepared for Minera Santo Domingo, 2018
- Quinzio & Anriquez Novoa (Abogados): Terrenos Superficiales y Servidumbres MSD: legal opinion prepared for Minera Santo Domingo, 8 October 2018

- Quinzio & Anriquez Novoa (Abogados): Terrenos Superficiales y Servidumbres MSD – Informe Anexo: legal opinion prepared for Minera Santo Domingo, 8 October 2018

This information is used in Section 4 of the Report. It is also used in support of the Mineral Resource statement in Section 14, the Mineral Reserve statement in Section 15, and the financial analysis result in Section 22.

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, 2018: Final Report Review of Tax Aspects of the Financial Model, dated 12 November 2018.

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

3.4 Markets

The QPs have relied upon information supplied by experts retained by Capstone to the iron fines market in Asia, as follows:

- Trotter, D.J., 2018: Pellet Feed Market Characterization and Forward Pricing Outlook for Capstone Mining Corporation: report prepared for Capstone, September 2018.

This information is used in Sections 19 and 22 of the Report, and in support of the Mineral Reserve statement in Section 15.

The QPs consider it reasonable to rely upon the information provided by Mr Trotter for iron ore concentrate. Mr Trotter is a global iron ore and pellet feed consultant, with significant experience in sales and marketing for major iron-producing companies, including BHP Billiton, Fortescue Metals Group, and Anglo American. Mr Trotter has been involved with sales and technology for marketing, trading and technical development in iron ore, metals and concentrate organizations across European, Chinese, Indian, American, and globally-developing markets. The QPs were able to review Mr. Trotter's report.

3.5 Metal Pricing

The QPs have relied on the following document for metal price support for long-term iron fines pricing and the premium for 66% Fe magnetite with low levels of alumina:

- Trotter, D.J., 2018: Pellet Feed Market Characterization and Forward Pricing Outlook for Capstone Mining Corporation: report prepared for Capstone, September 2018.

This information is used in Sections 19 and 22 of the Report, and in support of the Mineral Reserve statement in Section 15.

The QPs consider it reasonable to rely upon the information provided by Mr Trotter for iron ore concentrate. Mr Trotter is a global iron ore and pellet feed consultant, with significant experience in sales and marketing for major iron-producing companies, including BHP Billiton, Fortescue Metals Group, and Anglo American. Mr Trotter has been involved with sales and technology for marketing, trading and technical development in iron ore, metals and concentrate organizations across European, Chinese, Indian, American, and globally-developing markets. The QPs were able to review Mr. Trotter's report.

4.0 PROPERTY DESCRIPTION AND LOCATION

The portion of the Santo Domingo Project area that hosts the Santo Domingo deposits and will host the mine and plant site areas is located approximately 5 km southeast of the town of Diego de Almagro in the province of Diego de Almagro in the Atacama Region of northern Chile.

This area was originally part of the BHP's Candelaria project 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 and Chilean law (Chilean Civil Code, Chilean Mining Code, Chilean Tax Law, Fraser Institute, 2017), 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 Universal Transverse Mercator (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 manifestación within the initial two-year period.

Manifestación

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

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

A 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.

There is a mining tax that provides protection of rights; it 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 exploration and/or mining exploitation concessions. This tax is paid annually in a single payment before 31 March of each year.

For mining exploitation concessions, 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.

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 manifestación 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 (RoW), 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 RoW 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 RoW, as long as this does not affect their own exploitation activities.

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 with 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, thermo-electric 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, and 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 Environmental Commission (Comisión Nacional del Medio Ambiente, CONAMA) or regional Environmental Commission (Comisión Regional del Medio Ambiente, COREMA).

Decree No. 40/2012, 30 October 2012 Regulations for the System of Environmental Impact Assessment (Reglamento del Sistema de Evaluación de Impacto Ambiental, RSEIA) was approved and published in the Official Gazette on 12 August 2013. 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 PRIs 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 submit a Closure Plan to Sernageomin, prior to starting construction of a mining project, with an approval procedure that depends on the mine capacity. The main 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 the type of information required to be submitted for evaluation of the Closure Plan. Closure Plans for 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 Closure 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 must comply with the administrative system described in Chapter XIV of the Chilean Central Bank's Compendium of Foreign Exchange Regulations in order to register the entry of foreign capital into Chile. Under the administrative system of Chapter XIV of the Chilean Central Bank, the entry of foreign capital must be registered by commercial banks which, in turn, must 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 require a contract of any type. Capital entering Chile under Chapter XIV is not subject to any tax benefit and foreign investors using this regime are subject to the general taxation established by the Chilean Income Tax Law and the VA (VAT) Law.

Foreign investors complying with the above may freely choose to apply for the Foreign Investment Legal Framework established in Law No. 20.848 of 2015, which entered into force on 1 January 2016. The Foreign Investment Legal Framework regulates investments made by an individual or legal entity incorporated overseas, not residing or domiciled in Chile, whose investment is equal to or greater than $5 million, or the equivalent in other currencies.

Foreign investments authorized under this legal framework are entitled to:

- Remit abroad the equity invested and the net profits generated by the investment in Chile, when all tax obligations have been fulfilled according to the local regulations
- Access the formal exchange market to liquidate the currency constituting the investment.
- Access the formal exchange market in order to obtain the foreign currency required to remit the equity invested or the net profits generated by the investment in Chile
- A VAT exemption on the import of capital goods in projects worth over $5 million, as long as certain requirements are met
- No arbitrary discrimination, the foreign investor is subject to the same legal regime as local investors.

In order to qualify as a foreign investor and access the rights available under the Foreign Investment Legal Framework of Law 20.848 described above, the investor must request a certificate from the Agency for the Promotion of Foreign Investments demonstrating the investor's foreign status. The request submitted to the Agency must provide evidence (in a form determined by the Agency) that the investment will be materialized in the country; a detailed description of the investment; and the amount, purpose and nature of the investment.

Law 20.848 states that, for a period of four years from 1 January 2016, a foreign investor may request authorization to sign a tax invariability contract according to the terms, time frames and conditions established in Articles 7 and 11 ter of Decree Law No. 600 (DL 600 was replaced by Law 20.780 from 1 January 2016).

- Article 7 of Decree Law 600 establishes a tax invariability system that grants, for a period of 10 years, a total effective tax rate of 44.45% for investments of no less than $5 million for any investment purposes in Chile
- Article 11 ter of Decree Law 600 establishes a tax invariability system that grants, for a period of 15 years, specific rights for investments of no less than $50 million for mining projects.

On 7 February 2014, Capstone signed a foreign investment contract with the state of Chile, which fell under the provisions of DL 600. According to the DL 600 Contract, Capstone, acting as the foreign investor entity, is entitled to make an investment in the Chilean company Minera Santo Domingo SCM, the developer of the Santo Domingo mining project. According to the DL 600 Contract provisions, this investment can be carried out in Chile via capital contribution and debt for an authorized amount of up to $2,100 million. This amount must be fully contributed within 8 years from the date of the signature of the contract (the contract will expire on 7 February 2022).

Under the provisions of the DL 600 Contract, Capstone has the following rights:

- The right to transfer its capital and net profits abroad, in accordance with the provisions of Articles 4, 5 and 6 of DL 600 and the provisions of the Income Tax Law
- Access to the foreign exchange market, both to liquidate the foreign currency constituting the contribution, and for capital and profit remittances, at the most favorable exchange rate, according to Articles 2° letter a) and 4 of DL 600
- Tax invariability in accordance with the provisions of Article 11 ter of DL 600
- Non-discrimination, in accordance with Articles 9 and 10 of DL 600
- Exemption from any levy, tax or encumbrance on the net proceeds obtained by the alienation of the shares or rights representative of the foreign investment, or by the sale or liquidation of the receiving company, up to the amount actually invested under this contract, in accordance with the provisions of Article 6 of DL 600, without prejudice to the provisions of the Income Tax Law.

4.1.10 Fraser Institute Survey

Capstone has used the 2017 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 2017, 2,700 companies were approached and 360 companies responded.

Wood has used 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 or mining company and forms a proxy for the assessment by industry of political risk in Chile from the mining perspective.

Chile has a Policy Perception Index rank of 25 out of the 91 jurisdictions in the Fraser Institute survey. Chile's Investment Attractiveness Index rating is eighth out of the 91 jurisdictions, and it is ranked seventh on the Best Practices Mineral Potential Index.

4.2 Project Ownership

Information provided to Wood 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 Wood that under the terms of the shareholder agreement signed between Capstone and Kores on 17 June 2011, Capstone is the Project operator.

4.3 Mineral Tenure

Capstone holds two groups of concessions, totalling 116 claims, which cover a total of 28,897 ha and include the proposed mine site, plant area and auxiliary facilities including port facilities. The tenure includes 96 exploitation concessions and 20 exploration concessions. Concessions are held in the name of Minera Santo Domingo.

The total concession area is divided as follows:

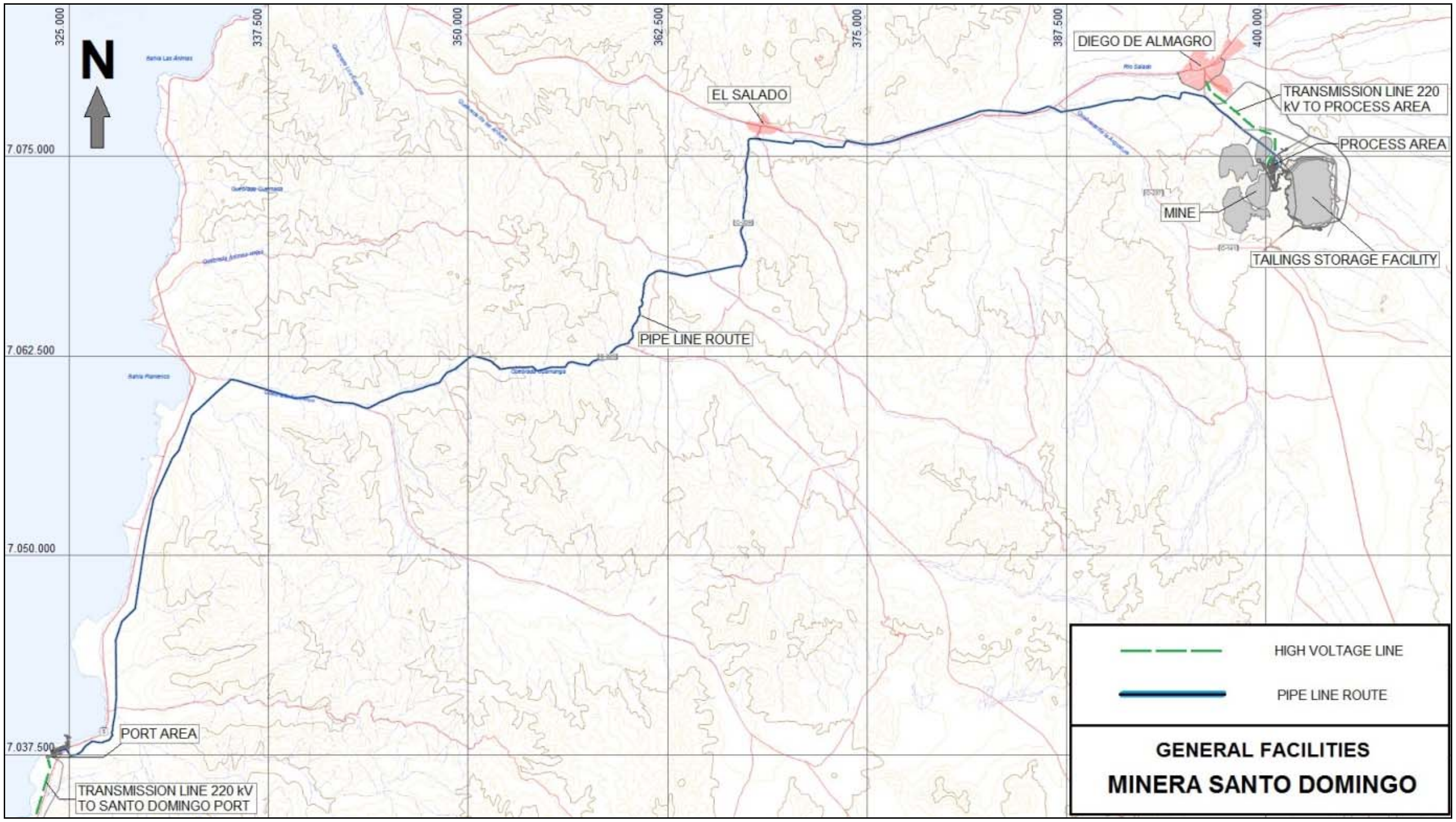
- 27,597 ha of exploitation concessions that encompass the area where the mine, plant, construction and operations camp and ancillary facilities are planned
- 1,300 ha of exploration concessions that encompass the port area.

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 Wood that all concession fees were current as of 31 October 2018, and will continue to be paid on a regular basis as due, using a formal status tracking system.

A simplified location plan for the contemplated infrastructure 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 and the surface rights for the Project facilities.

Figure 4-1: Proposed Project Facility Locations

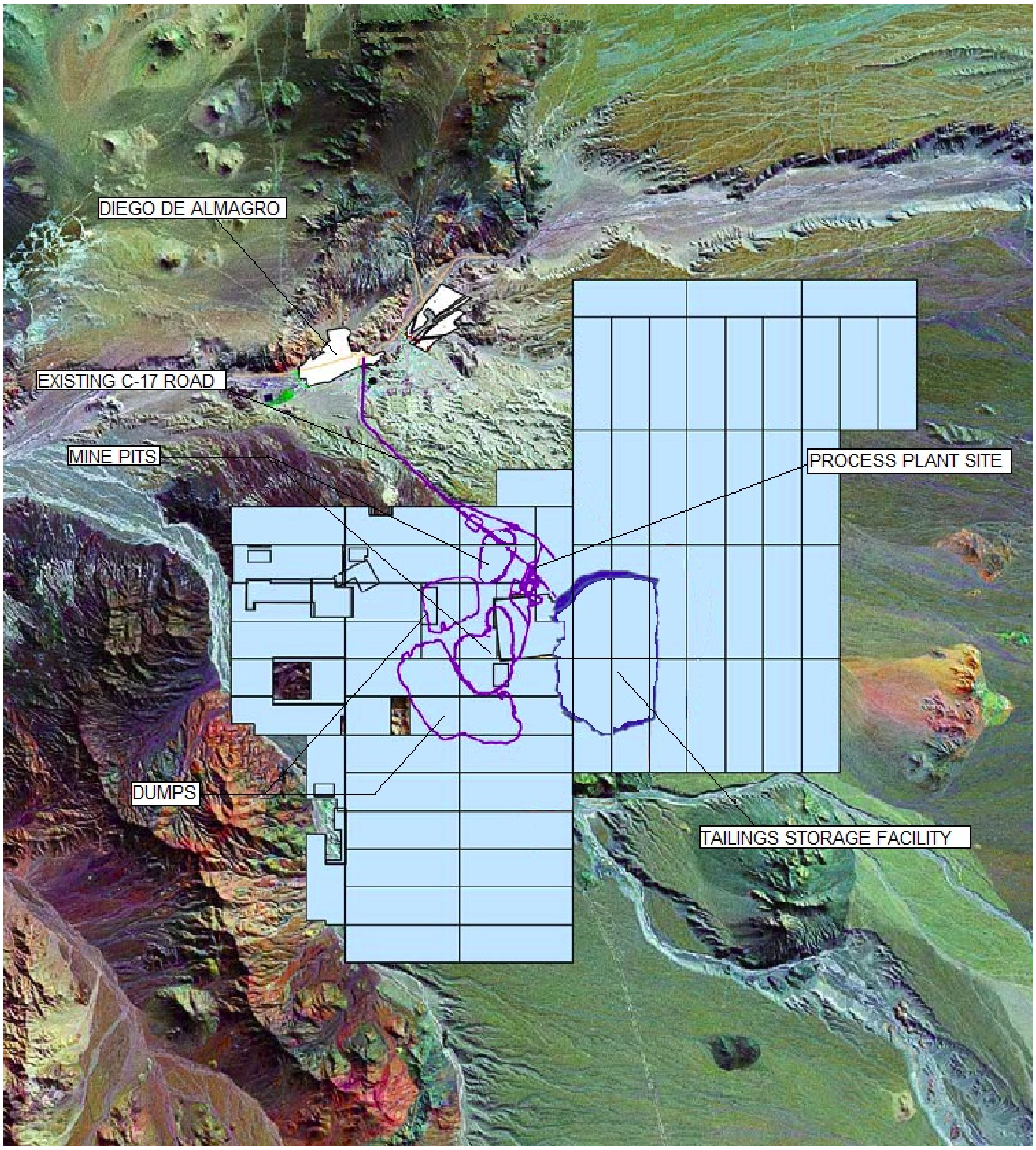


Note: Figure courtesy Capstone, 2014. The grid on the figure illustrates the map scale.

Table 4-1: Exploitation Concessions

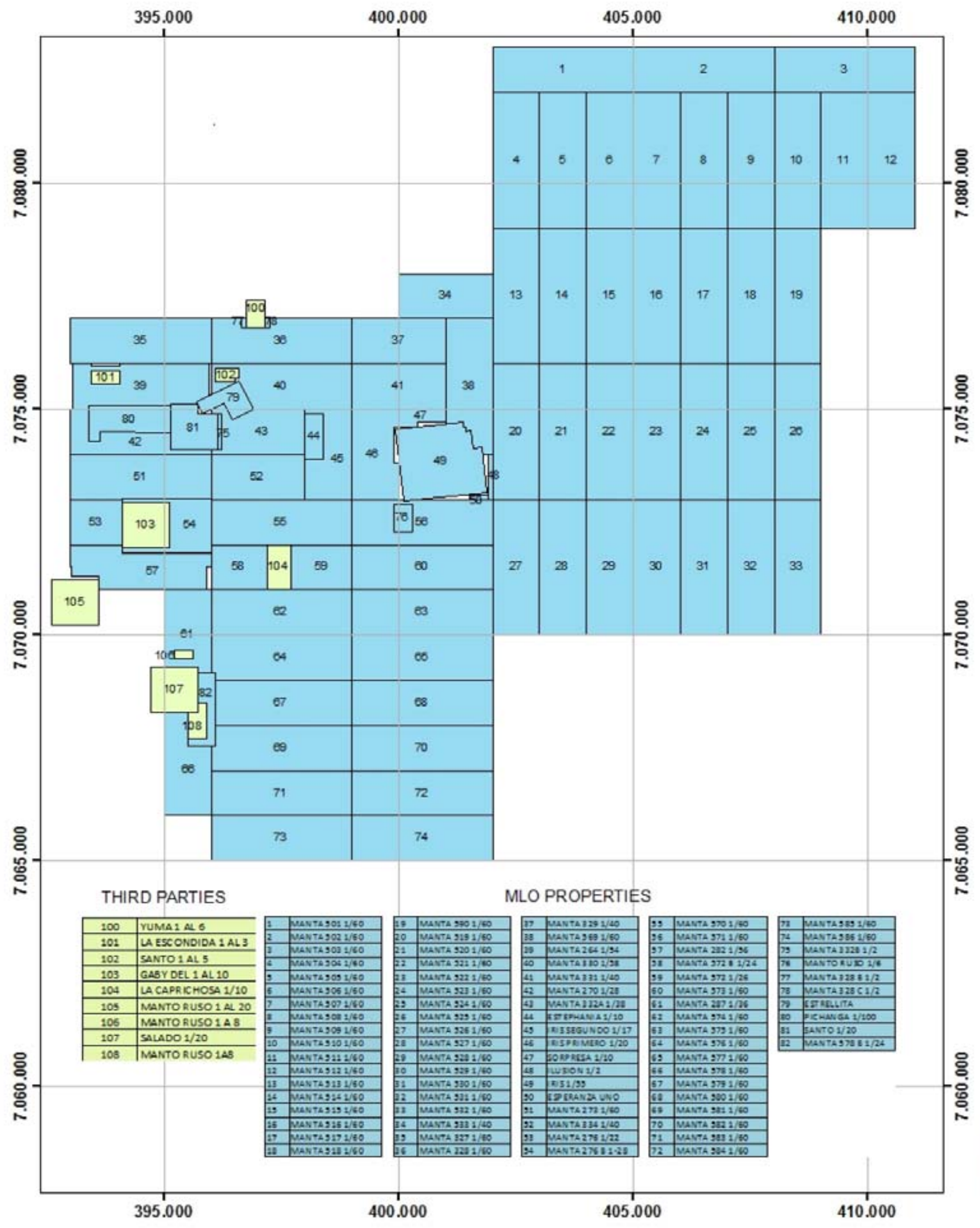
Name	Area (ha)	Name	Area (ha)	Name	Area (ha)
SORPRESA 1/10	34	MANTA 513 1/60	300	MANTA 575 1/60	300
ESPERANZA UNO	4	MANTA 514 1/60	300	MANTA 576 1/60	300
ILUSION 1/2	10	MANTA 515 1/60	300	MANTA 577 1/60	300
MANTA 264 1/54	266	MANTA 516 1/60	300	MANTA 328 B 1/2	2
MANTA 270 1/28	134	MANTA 518 1/60	300	MANTA 328 C 1/2	2
MANTA 273 1/60	300	MANTA 519 1/60	300	MANTA 505 1/60	300
MANTA 276 1/22	110	MANTA 520 1/60	300	MANTA 509 1/60	300
MANTA 282 1/56	239	MANTA 521 1/60	300	MANTA 501 1/60	300
MANTA 330 1/58	286	MANTA 522 1/60	300	MANTA 502 1/60	300
MANTA 332A 1/38	184	MANTA 523 1/60	300	MANTA 503 1/60	300
MANTA 332B 1/2	8	MANTA 524 1/60	300	MANTA 526 1/60	300
MANTA 334 1/40	200	MANTA 525 1/60	300	MANTA 528 1/60	300
MANTA 327 1/60	300	MANTA 517 1/60	300	MANTA 529 1/60	300
MANTA 328 1/60	290	MANTA 527 1/60	300	MANTA 530 1/60	300
MANTA 329 1/40	200	MANTA 531 1/60	300	MANTA 578 1/39	195
MANTA 331 1/40	200	MANTA 532 1/60	300	MANTA 583 1/60	300
IRIS 1/55	273	MANTA 579 1/60	300	MANTA 590 1/60	300
IRIS PRIMERO 1/20	192	MANTA 580 1/60	300	MANTA 578 B 1/9	31
IRIS SEGUNDO 1/17	160	MANTA 581 1/60	300	DOMINGO 03 1/60	300
ESTRELLITA 1/10	50	MANTA 582 1/60	300	DOMINGO 04 1/60	272
PICHANGA 1/100	110	MANTA 584 1/60	300	DOMINGO 07 1/60	300
SANTO 1/20	92	MANTA 585 1/60	300	DOMINGO 08 1/60	300
MANTO RUSO 1/6	24	MANTA 586 1/60	300	DOMINGO 09 1/60	300
ESTEPHANIA 1/10	40	MANTA 276 B 1/28	128	DOMINGO 11 1/60	300
MANTA 572 B 1/24	120	MANTA 287 1/36	161	DOMINGO 12 1/60	300
MANTA 504 1/60	300	MANTA 533 1/40	200	ALTO 2 1/60	300
MANTA 506 1/60	300	MANTA 569 1/60	300	ALTO 4 1/60	300
MANTA 507 1/60	300	MANTA 570 1/60	300	ALTO 6 1/60	300
MANTA 508 1/60	300	MANTA 571 1/60	300	ALTO 13 1/40	200
MANTA 510 1/60	300	MANTA 572 1/26	130	ALTO 14 1/60	250
MANTA 511 1/60	300	MANTA 573 1/60	300	SALADO 20 160	300
MANTA 512 1/60	300	MANTA 574 1/60	300	SALADO 27 160	300

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



Note: Figure courtesy Capstone, 2018. Area in white is the town of Diego de Almagro; dark blue outline is the proposed tailings storage facility, purple outlines are the proposed Santo Domingo open pit and waste rock facilities and the 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

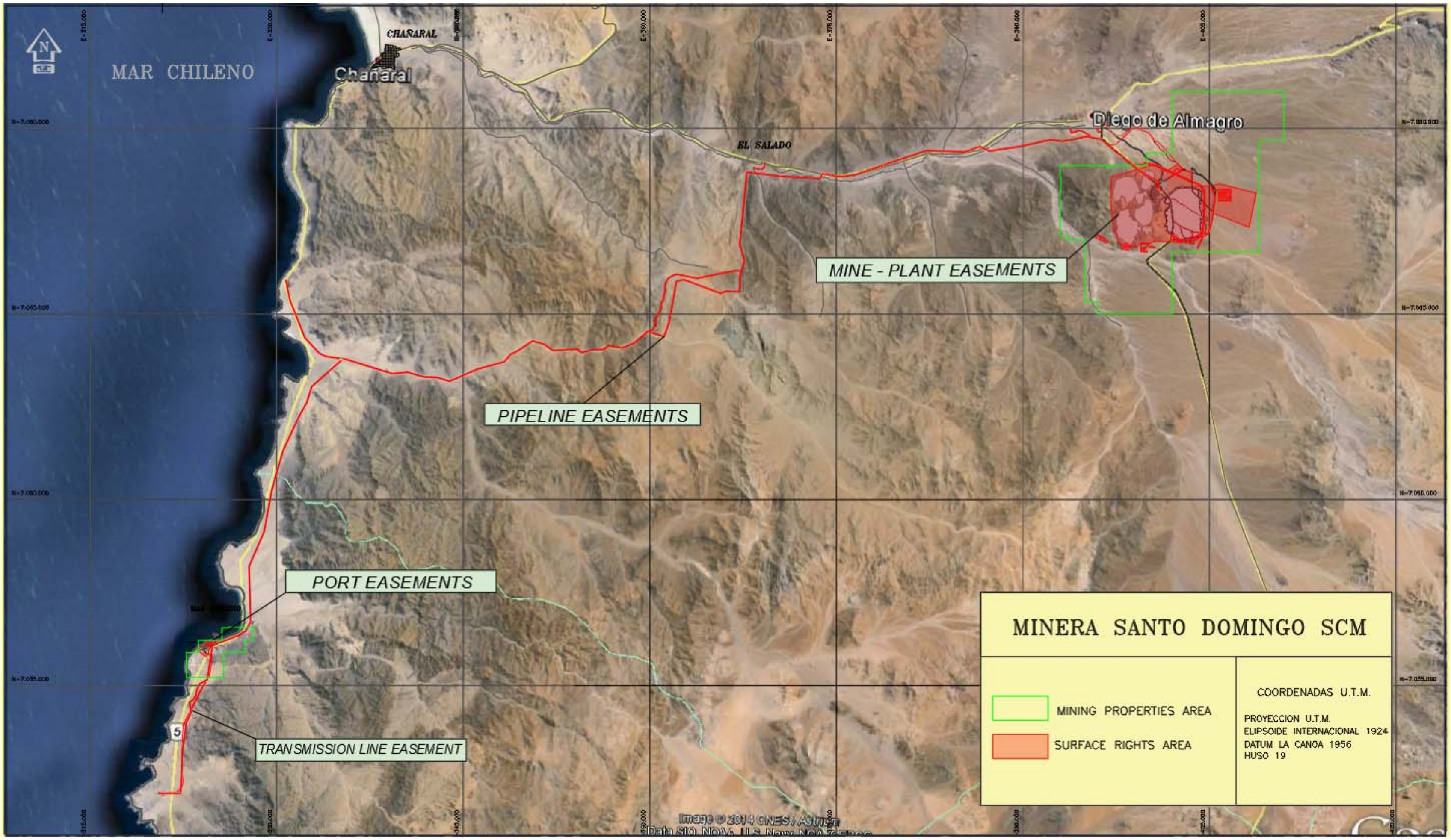


Note: Figure from Brimage et al., 2011. The status of the third-party claims that are not held by Capstone has not been verified. MLO refers to Minera Lejano Oeste S.A., a predecessor company to Capstone. Grid coordinates indicate map scale. Map north is to the top of the plan.

Table 4-2: Exploration Concessions

Name	Area (ha)
OESTE 1A	100
OESTE 2A	300
OESTE 3A	300
OESTE 4A	300
BLANCA 18	200
BLANCA 19	100
BLOCK 1	300
BLOCK 2	300
BLOCK 3	300
BLOCK 4	300
BLOCK 5	300
BLOCK 6	300
BLOCK 7	300
BLOCK 8	300
BLOCK 9	300
BLOCK 10	300
BLOCK 11	300
BLOCK 12	300
BLOCK 13	300
DOMINGO 16	300

Figure 4-4: Location Plan, Exploration Claims



Note: Figure courtesy Capstone 2018. Backdrop is based on Google Earth image.

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.

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 State is also the owner and the Ministerio de Bienes Nacionales also manages and represents the land in the districts of Caldera and Chañaral where the planned Project facilities will be located.

Capstone has developed a legal strategy to obtain the necessary surface rights to cover mine, plant, camps, tailings storage facilities, pipelines, port, and transmission lines.

Capstone currently possesses 17 registered provisional surface rights (listed in Table 4-3) and eight definitive surface rights (seven of the provisional rights have changed from provisional to definitive) as shown in Table 4-4). Capstone has 17 applications in progress for definitive surface rights as indicated in Table 4-5. These cover surface rights for facilities and infrastructure. All these surface rights are contracts between Capstone and the Chilean Treasury.

Capstone notes that to date, 1,966 ha have been granted definitive easements, 1,732 ha have been granted provisional easements, and 2,574 ha are in the process of the creation of definitive easements. This total area covers:

- The plant and infrastructure (process plant, tailings disposal, open pit and waste disposal)
- Pipelines
- Temporary construction and permanent operations camp
- The port area.

Table 4-3: Provisional Surface Rights Granted to Capstone

Nº	Court	ROL (ID)	Purpose	Area (ha)	Provisional Easement Register
1	1º	8-2012	Mine area	39.7068	Folio 103, Nº 27, Mortgages, Liens and Prohibitions Registry, 2012.
2	1º	9-2012	Mine area	10.1204	Folio 112, Nº 28, Mortgages, Liens and Prohibitions Registry, 2012
3	1º	10-2012	Mine area	390.2654	Folio 82, Nº 13, Mortgages, Liens and Prohibitions Registry, 2013
4	1º	205-2012	Mine area	298.2306	Folio 120 vta., Nº 29, Mortgages, Liens and Prohibitions Registry, 2012
5	1º	211-2012	Mine area	250.0000	Folio 91, Nº 14, Mortgages, Liens and Prohibitions Registry, 2013
6	1º	212-2012	Mine area	341.8750	Folio 129, Nº 30, Mortgages, Liens and Prohibitions Registry, 2012
7	1º	213-2012	Mine area	228.8750	Folio 13 vta., Nº 31, Mortgages, Liens and Prohibitions Registry, 2012
8	1º	214-2012	Mine area	187.7500	Folio 146, Nº 32, Mortgages, Liens and Prohibitions Registry, 2012
9	1º	215-2012	Mine area	127.5000	Folio 99 vta., Nº 15, Mortgages, Liens and Prohibitions Registry, 2013
10	1º	2674-2012	Pipeline	113.7320	Folio 108 vta., Nº 16, Mortgages, Liens and Prohibitions Registry, 2013
11	1º	2675-2012	Pipeline	193.1600	Folio 117 vta., Nº 17, Mortgages, Liens and Prohibitions Registry, 2013
12	1º	2677-2012	Pipeline	65.7600	Folio 34 vta., Nº 07, Mortgages, Liens and Prohibitions Registry, 2013
13	1º	3195-2012	Tailings storage facility	1,167.1511	Folio 61 vta., Nº 10, Mortgages, Liens and Prohibitions Registry,

Nº	Court	ROL (ID)	Purpose	Area (ha)	Provisional Easement Register
					2013
14	1º	1923-2013	Port	75.9336	Folio 1, Nº 1, Mortgages, Liens and Prohibitions Registry, 2014
15	1º	3130-2013	Camp	3.3278	Folio 123 vta., Nº 19, Mortgages, Liens and Prohibitions Registry, 2014
16	1º	915-2014	C-17 by-pass	132.7713	Folio 13, Nº 3, Mortgages, Liens and Prohibitions Registry, 2015
17	2º	1861-2014	Pipeline	8.3900	Folio 34 vta., Nº 4, Mortgages, Liens and Prohibitions Registry, 2015

Note: All Provisional Surface Rights granted were registered with the Diego de Almagro Registrar of Real Estate and Mines.

Table 4-4: Definitive Surface Rights Granted to Capstone

Nº	Court	ROL (ID)	Purpose	Area (ha)	Date of Notification
1	1º	10-2012	Mine (pit, plant)	390.2654	21 September 2018
2	1º	2674-2012	Pipeline	113.7320	21 September 2018
3	1º	2675-2012	Pipeline	193.1600	21 September 2018
4	1º	620-2013	Tailings storage facility	372.2046	21 September 2018
5	1º	1923-2013	Port	75.9336	21 September 2018
6	1º	211-2012	Mine area	250.0000	Not notified
7	1º	212-2012	Mine area	341.8750	Not notified
8	1º	213-2012	Mine area	228.8750	Not notified

Table 4-5: Capstone Surface Rights in the Process of Approval

Nº	Court	ROL (ID)	Purpose	Area (ha)
1	1º	8-2012	Mine area	39.7068
2	1º	214-2012	Mine area	187.7500
3	1º	215-2012	Mine area	127.5000
4	1º	2677-2012	Pipeline	65.7600
5	1º	3195-2012	Tailings storage facility	1,167.1511
6	1º	415-2013	Pipeline	166.3100
7	1º	1002-2013	Mainstream	842.6400
8	1º	3130-2013	Camp	3.3278
9	1º	915-2014	C-17 by-pass	132.7713
10	1º	2852-2014	El Salado by-pass	3.8170
11	1º	3260-2014	Borrow areas	51.8360
12	1º	2257-2016	Mine area	764.7944
13	2º	1861-2014	Pipeline	8.3900
14	2º	768-2015	Transmission line; C-17 By-pass	116.9712
15	2º	2303-2016	Mine area	307.9881
16	3º	1593-2016	Pipeline	56.2930
17	4º	2282-2016	Mine area	263.6770

4.5 Water Rights

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

Water for construction will be obtained from Aguas Chañar S.A. (Aguas Chañar), 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 Enami for minerals mined from certain concessions subject to the royalty agreement in force between Enami and Capstone.

A 2% NSR royalty is payable to BHP for minerals mined from certain other concessions subject to the royalty agreement in force between BHP and Capstone.

The majority of the proposed open pits are located on concessions subject to one or other of these two royalty agreements.

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 QP was 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
- Capston has 17 provisional surface rights and eight definitive surface rights; 17 definitive surface rights are currently in the process of approval. Together these easements cover 100% of the area needed for construction of facilities and infrastructure

- Royalties in the form of the Chilean mining tax will be payable
- Royalties to Enami and BHP are also payable in the form of NSRs
- No water rights are currently envisaged as the water will be sourced from the ocean, desalinated and piped to the site for use
- Capstone advised Wood that the Capstone is not aware of any issues that may affect access, title, or the right or ability to perform work on the Project.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

Access to the planned and plant site area where the deposits are located is via the paved Pan-American Highway (Route 5 North) and a network of generally well-maintained gravel roads. The C-17 paved highway connecting Copiapó and Diego de Almagro passes immediately east of the proposed and plant site area. The Project deposits are roughly five hours' travel time by road south of Antofagasta, and two hours by road north of Copiapó.

Access to the proposed mine site 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. C-17 southbound, a 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 planned Santo Domingo mine site is approximately 68 km, with a travel time of approximately 50 minutes.

5.2 Climate

The proposed mine site is located in an area that is 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 January 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 system 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 annually. 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.

It is expected that it will be possible to conduct mining, processing, desalination, and port activities on a year-round basis.

5.3 Local Resources and Infrastructure

There are several towns and villages near the proposed mine site. Diego de Almagro, located adjacent to the mine and 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 are available. 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 Atacama 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 by 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, as part of the 2014 Feasibility Study, and his recommendations are included in the current Project design.

5.6 Comments on Section 5

In the QP's opinion:

- 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
- Mining, processing, desalination, 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 current 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 km^2 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.

In January 2006, Far West announced an agreement with Empresa Nacional de Mineria (ENAMI), a Chilean government corporation, to acquire a 100% interest in its 673 ha Iris property. ENAMI transferred concession titles to Far West in exchange for staged payments and 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 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.

Table 6-1: Exploration Summary Table

Year	Company	Work Program	Program Details	Reported
2002	BHP Billiton	Airborne geophysical survey	10,700 line km Falcon™ airborne gravity gradiometer survey covering 5,145 km^2 in eight blocks along a 300 km strike length of the IOCGB between Taltal and south of Copiapó.	
July 2003 to November 2005	Far West/ BHP Billiton*	Geological mapping	Approximately 50 km^2 of geological mapping at 1:25000	Höy and Allen, 2005
		Surface rock samples	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.	
		Sediment samples	47 sieved (106 µm) samples, submitted for analysis for Au and a 27-element ICP suite.	
		IP survey	17.6 line km	
		RC drilling	67 holes (20,592 m) analysed for Au and a 27-element ICP suite	
November 2005 to May 2006	Far West	Geophysical data interpretation	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.	Lacroix, 2006
		RC drilling	15 holes (5,176 m) analysed for Au and a 27-element ICP suite	
		Core drilling	4,057 m in eight holes; analysed for Au and a 27-element ICP suite	
May 2006 to September 2007	Far West	RC drilling	215 holes (51,909.5 m); analysed for Au and a 27-element ICP suite	Lacroix and Rennie, 2007
		Core drilling	15 holes (2,649.75 m); analysed for Au and a 27-element ICP suite	
September 2007 to December 2008	Far West	RC drilling	37 holes (10376.5 m); analysed for Au and a 27-element ICP suite	Lacroix, 2009
		Core drilling	One hole (495.25 m); analysed for Au and a 27-element ICP suite	
December 2008 to May 2010	Far West	RC drilling	Nine holes (2,557 m); analysed for Au and a 27-element ICP suite	Rennie, 2010

Year	Company	Work Program	Program Details	Reported
		Core drilling	26 holes (9,073 m); analysed for Au and a 27-element ICP suite	
2011–2012	Capstone		66 holes (some were abandoned due to operational difficulties) totalling 13,282 m	Maycock et al., 2014
2015	Capstone	RC drilling Diamond drilling	14 drill holes, approximately 3,200 m in total (1,660 m RC, 1,540 m diamond drill). These were twin holes for selected existing drill holes to produce material for metallurgical testing	

Note: The BHP Billiton interest was terminated on 4 May 2005.

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 PA; however, changes to the base case metal price assumptions did result in potentially positive economics, and additional work was recommended.

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 the 2011 Pre-feasibility Study. 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.

The 2014 Feasibility Study commenced in 2012 and was completed in 2014. During 2014–2015, 14 drill holes were executed, approximately 3,200 m in total (1,660 m RC; 1,540 m diamond drill). These were twin holes for selected existing drill holes to produce composite material approximating the first five years of operation in the 2014 Feasibility Study for metallurgical testing. The work was carried out by MCA Spa and supervised by Amec Foster Wheeler. The drill core was used in pilot plant operation, which was conducted using the flowsheet assumptions envisaged in the 2014 Feasibility Study.

During 2018, a review of assumptions in the 2014 Feasibility Study was undertaken. This Report includes changes to the Project concepts and incorporates current pricing. Mineral Resources were updated, and the estimates in Section 14 now include cobalt as having reasonable prospects of eventual economic extraction. Mineral Reserves and the financial analysis presented in this Report include the findings of the 2015 and 2018 concept changes and currently-available information.

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 25ºS to 31ºS. 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 Ma to 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

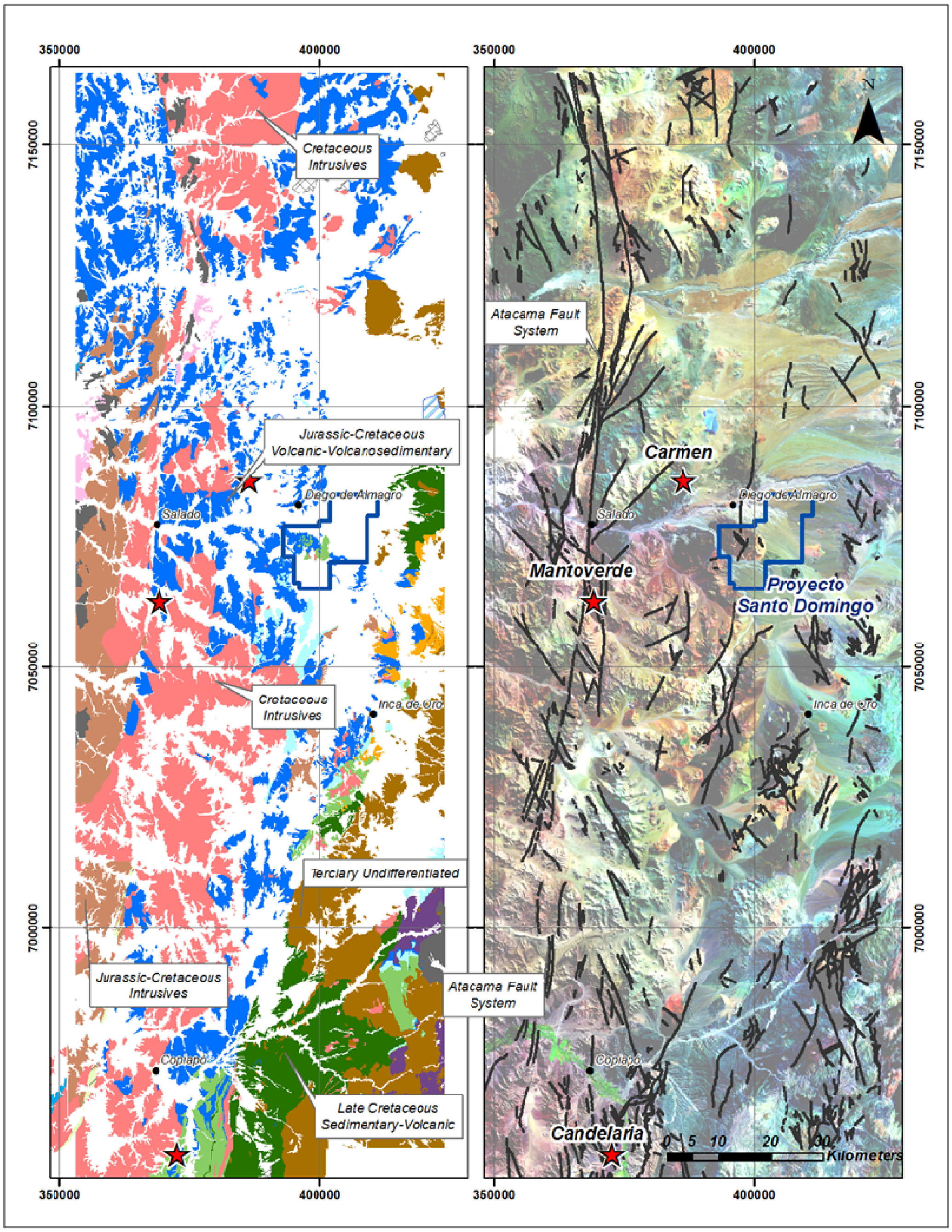
The Project lies on the east side of the Atacama fault complex, which, in this area, consists of numerous clusters of generally north–south structural breaks in a belt approximately 30 km wide (Figure 7-1).

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.

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 coordinates indicate map north and figure scale.

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.

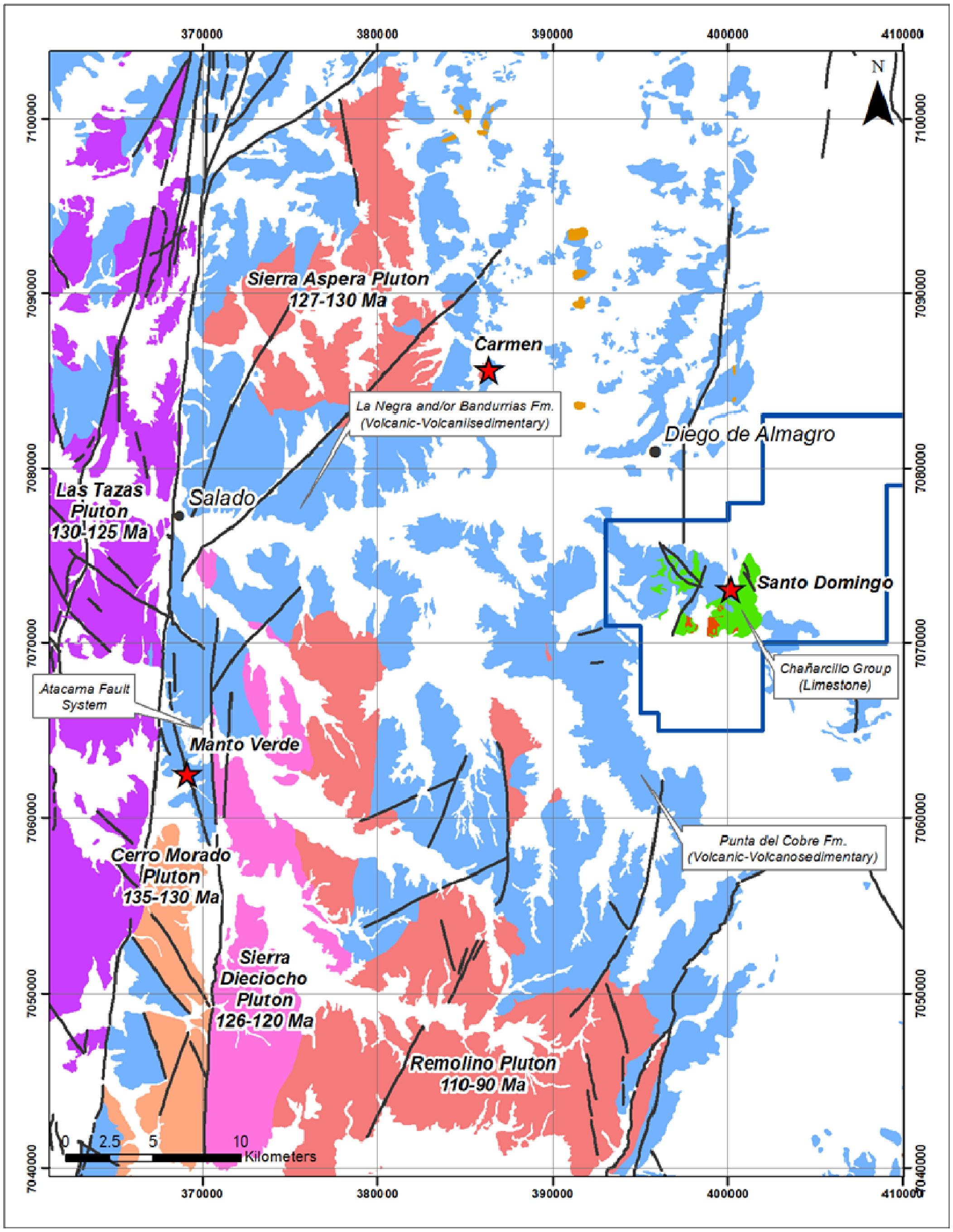
Faults trend variably north, northwest, northeast, and east–west. These faults are complex and seem to have been active repeatedly through time. Many of the faults appear to mark the boundary of pronounced lithological changes.

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.

Figure 7-2: Local Geology Plan Showing Major Intrusive Events



Note: Figure courtesy Capstone, 2013.

High-angle block faulting played an important role in localizing manto- and fault-related iron oxide–copper mineralization in the Project 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 volcaniclastics 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 iron–copper–gold mineralization in Santo Domingo is almost entirely hypogene; the proportions of sulphides to oxides are approximately 13:1 (Rennie, 2010).

At shallow levels, typical copper–iron 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 Mineral Reserves, and a number of prospects have been identified in the Project area (Figure 7-3).

Figure 7-3: Deposit and Prospect Layout Plan

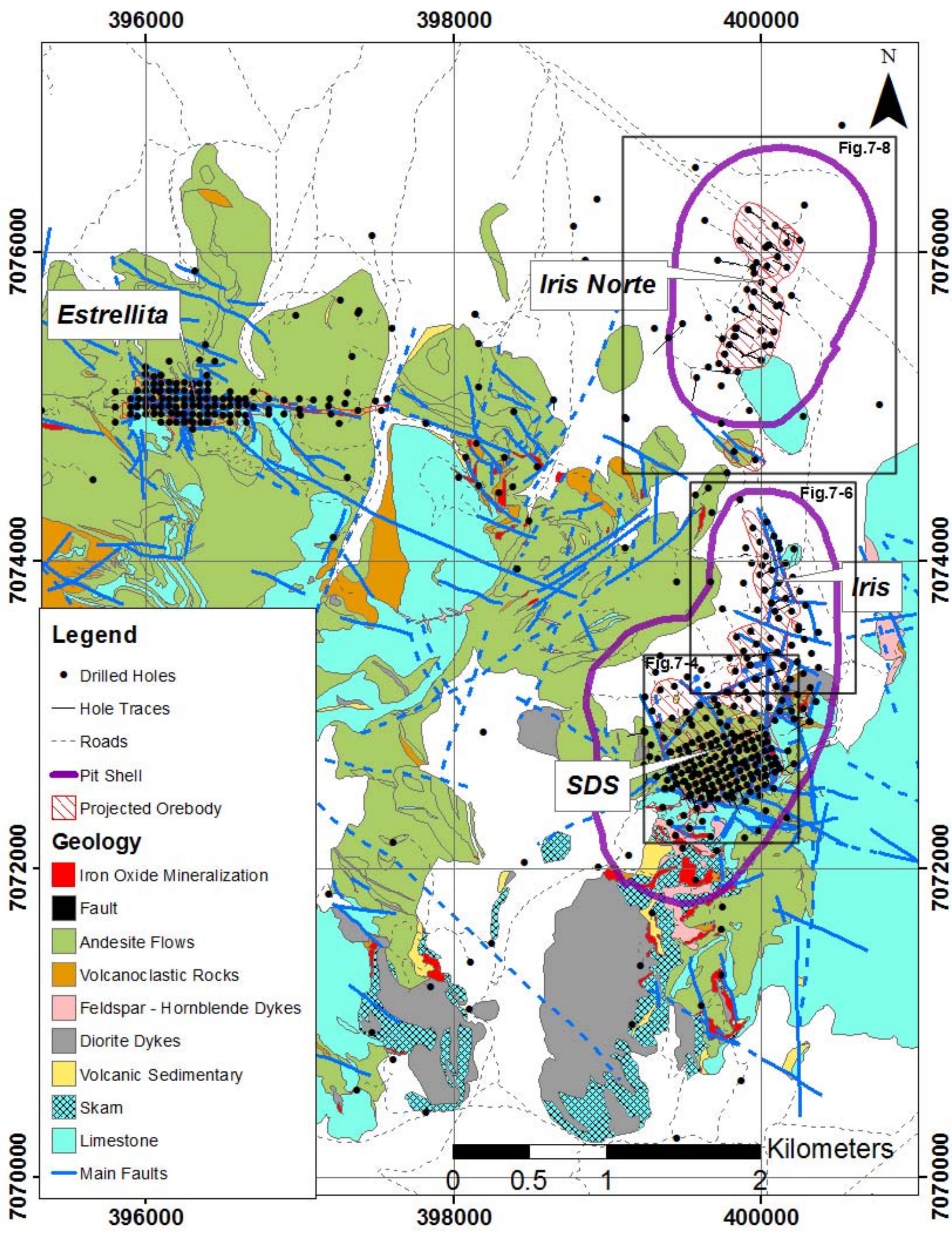


Figure courtesy Capstone, 2013. SDS = Santo Domingo Sur deposit.

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.

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.

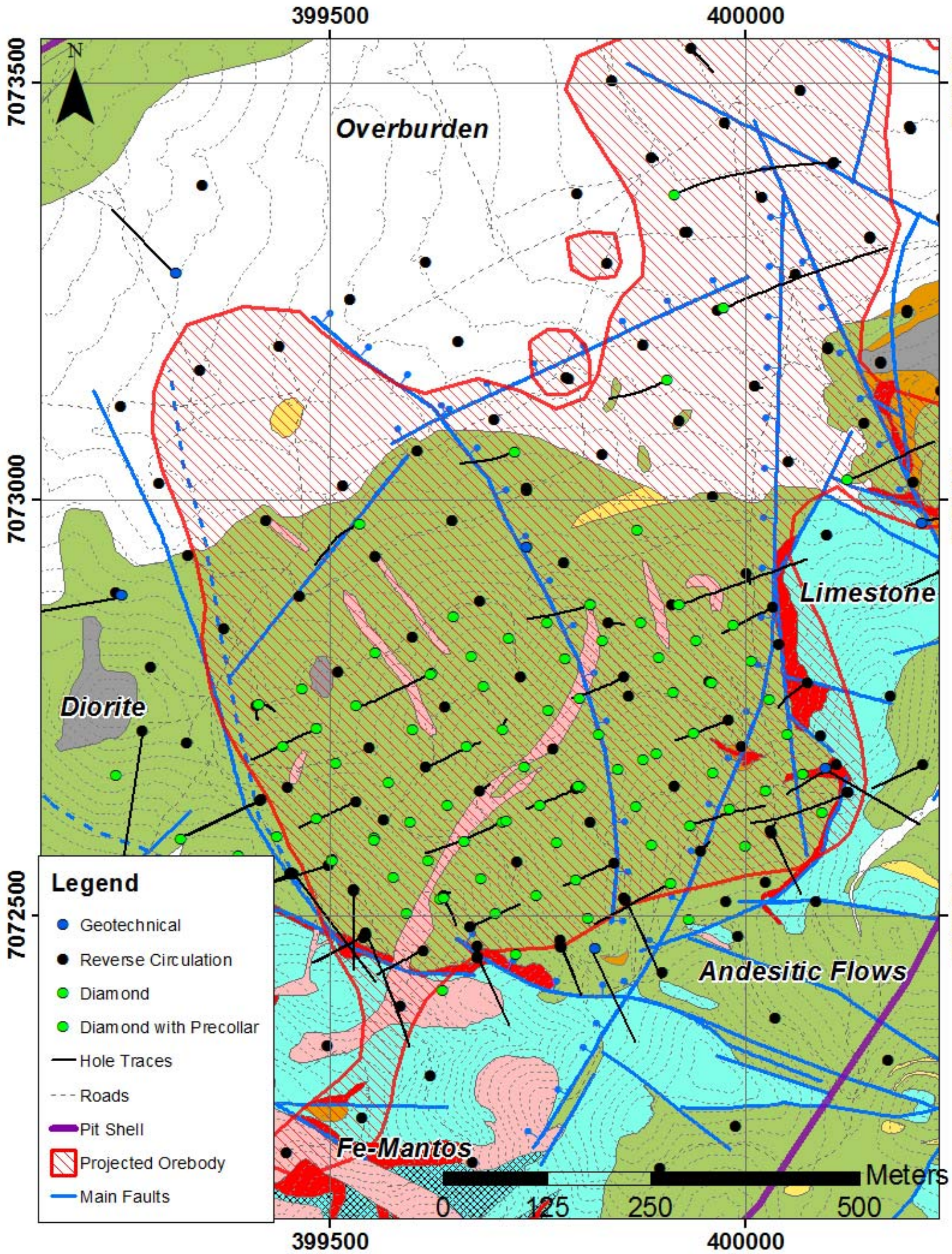
7.3.2 Iris

The Iris deposit is essentially blind, covered by a sequence of Quaternary gravel. The elongated shape of the deposit and textures observed in core 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.

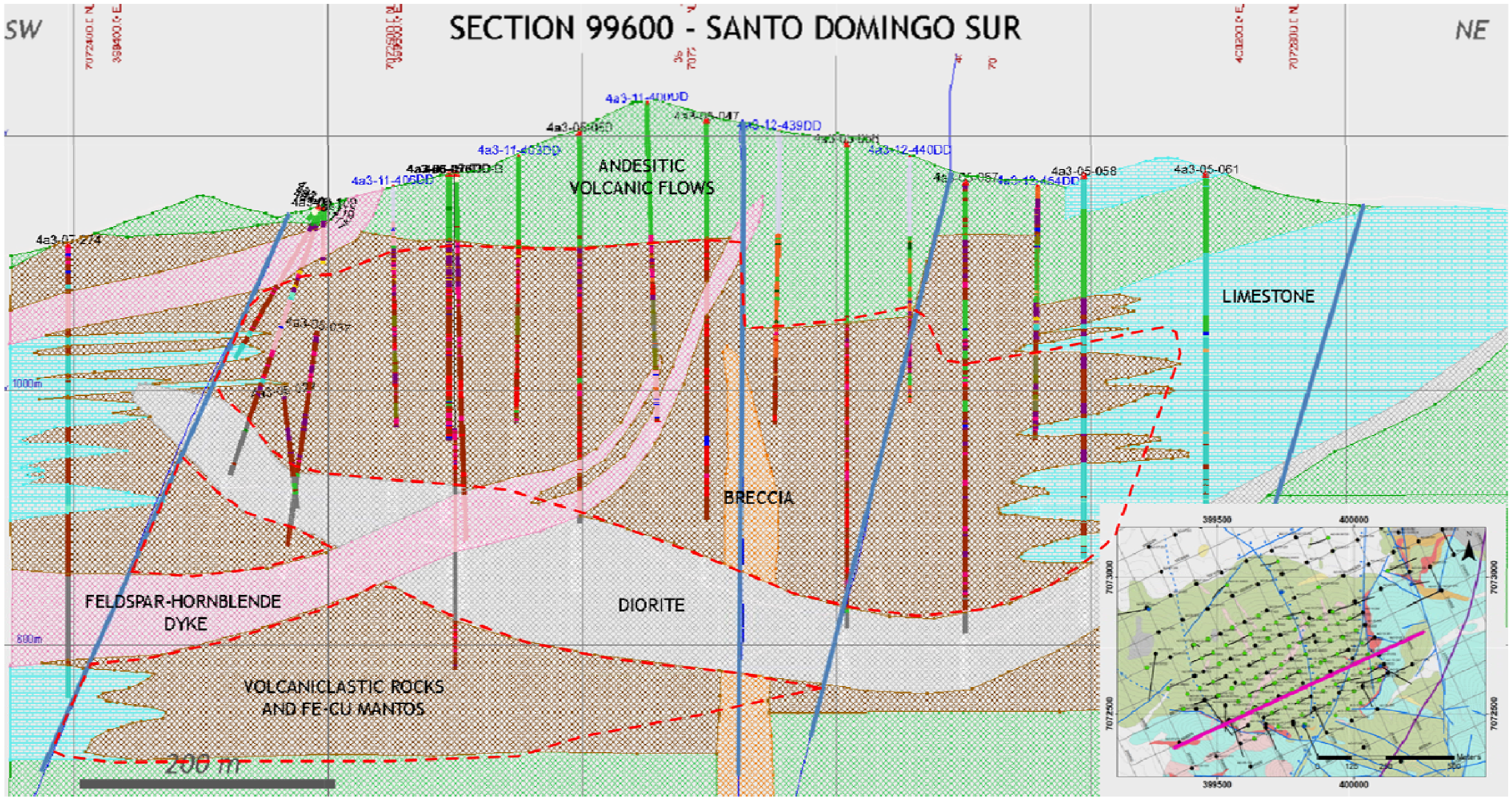
The deposit consists of iron oxide mantos and breccias developed along a north–northwest-striking fault zone. The dominant iron oxide at Iris is hematite and the main copper mineral is chalcopyrite.

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.

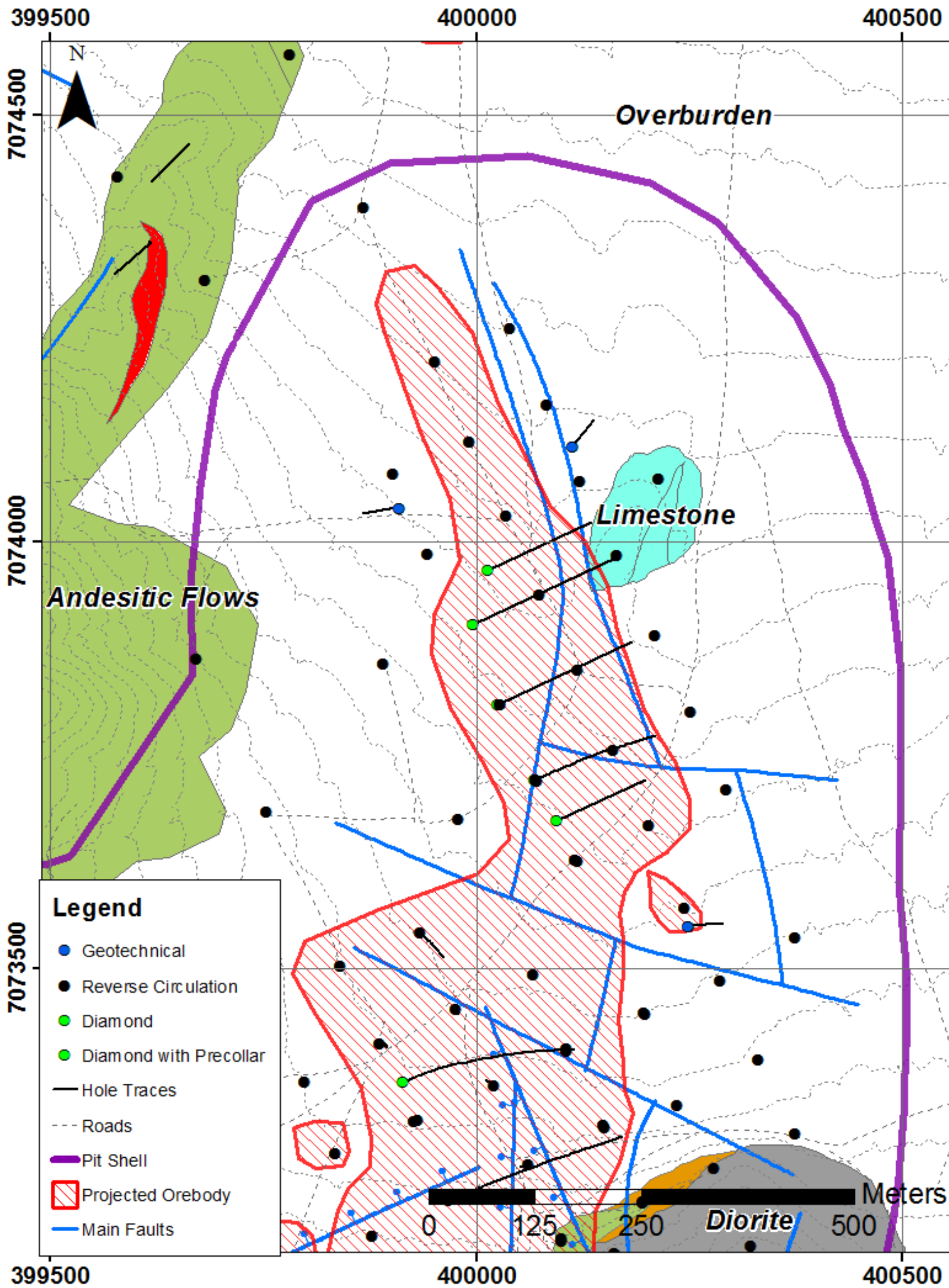
The Iris Mag zone is located between the Iris deposit and the Santo Domingo Sur deposit in a separate structural block. Mineralization in the zone consists of magnetite and chalcopyrite with a very high magnetite content (40% and more) and typically low copper content (approximately 0.1% Cu on average). 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-grade copper and gold values 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

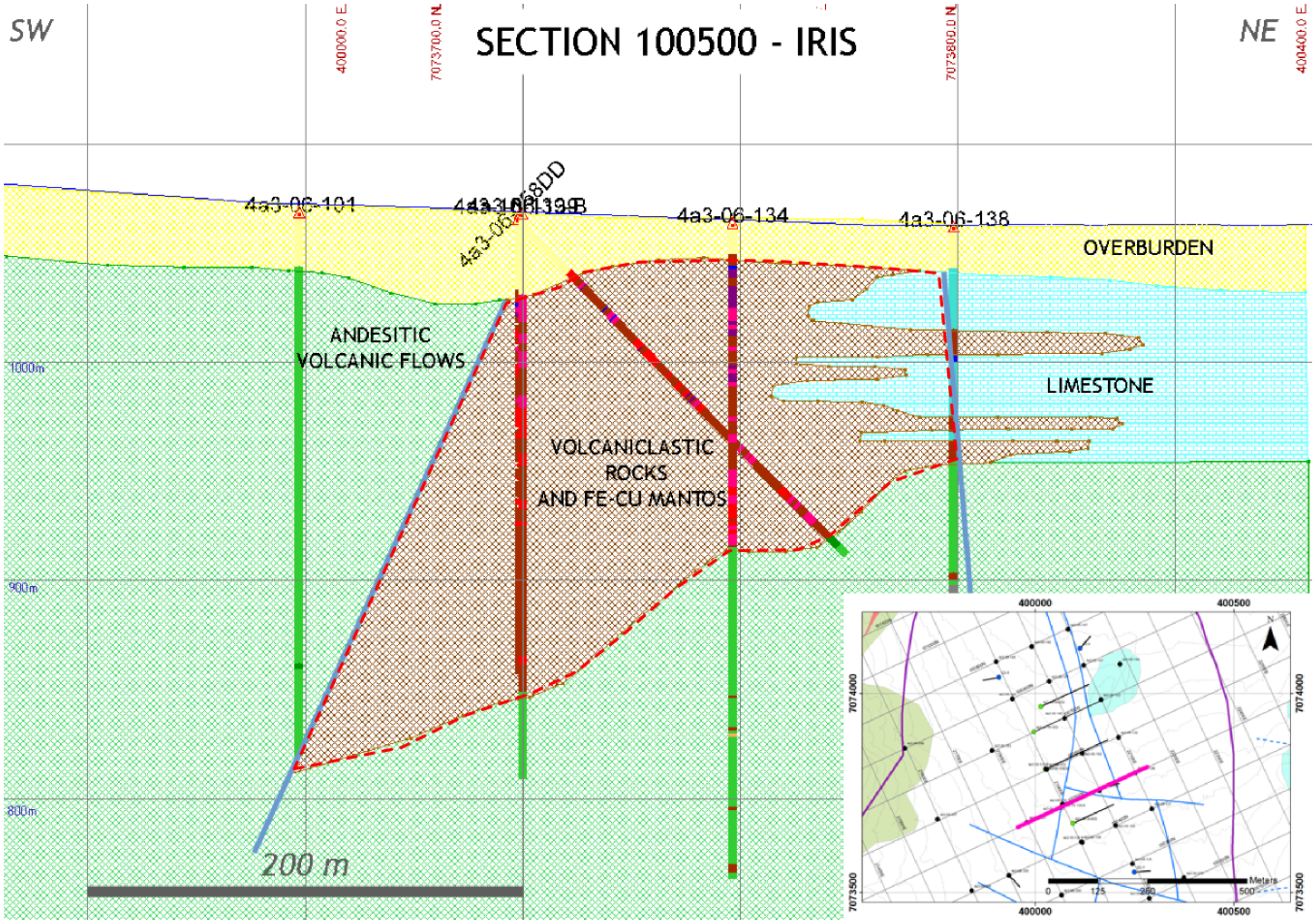
The Iris Norte deposit is located about 600 m to the north of the Iris deposit and is also blind, being entirely covered by a gravel sequence. The deposit is very similar in character to Iris and occurs on the eastern edge of a pronounced gravity anomaly. The deposit is approximately 500 m wide and has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

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 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 north-easterly 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.

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.

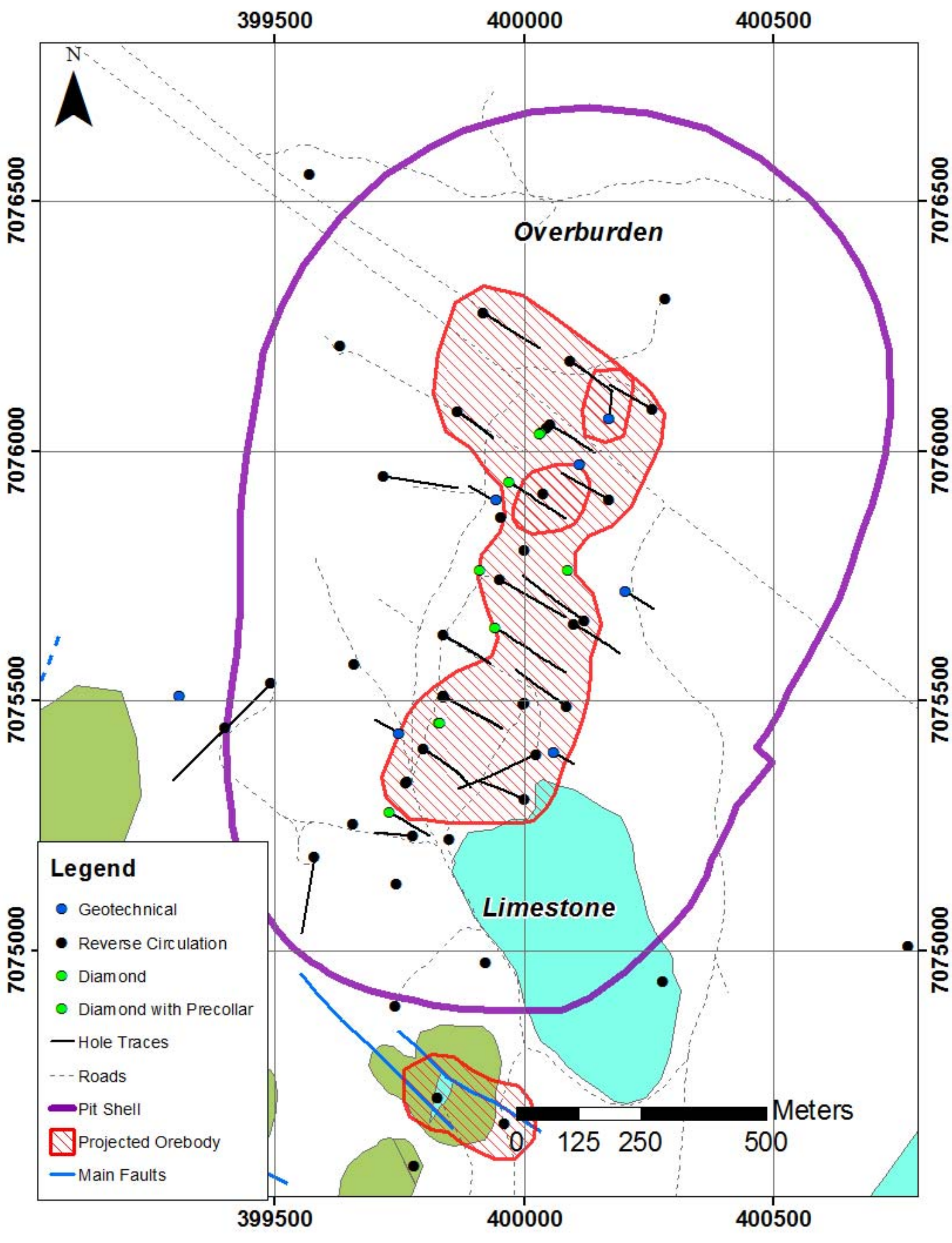
Figure 7-8 is a geology plan of the Iris Norte deposit. Figure 7-9 is a cross-section through the deposit.

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 volcaniclastics with minor intercalations and interbeds of andesite porphyry, limestone, and altered tuff.

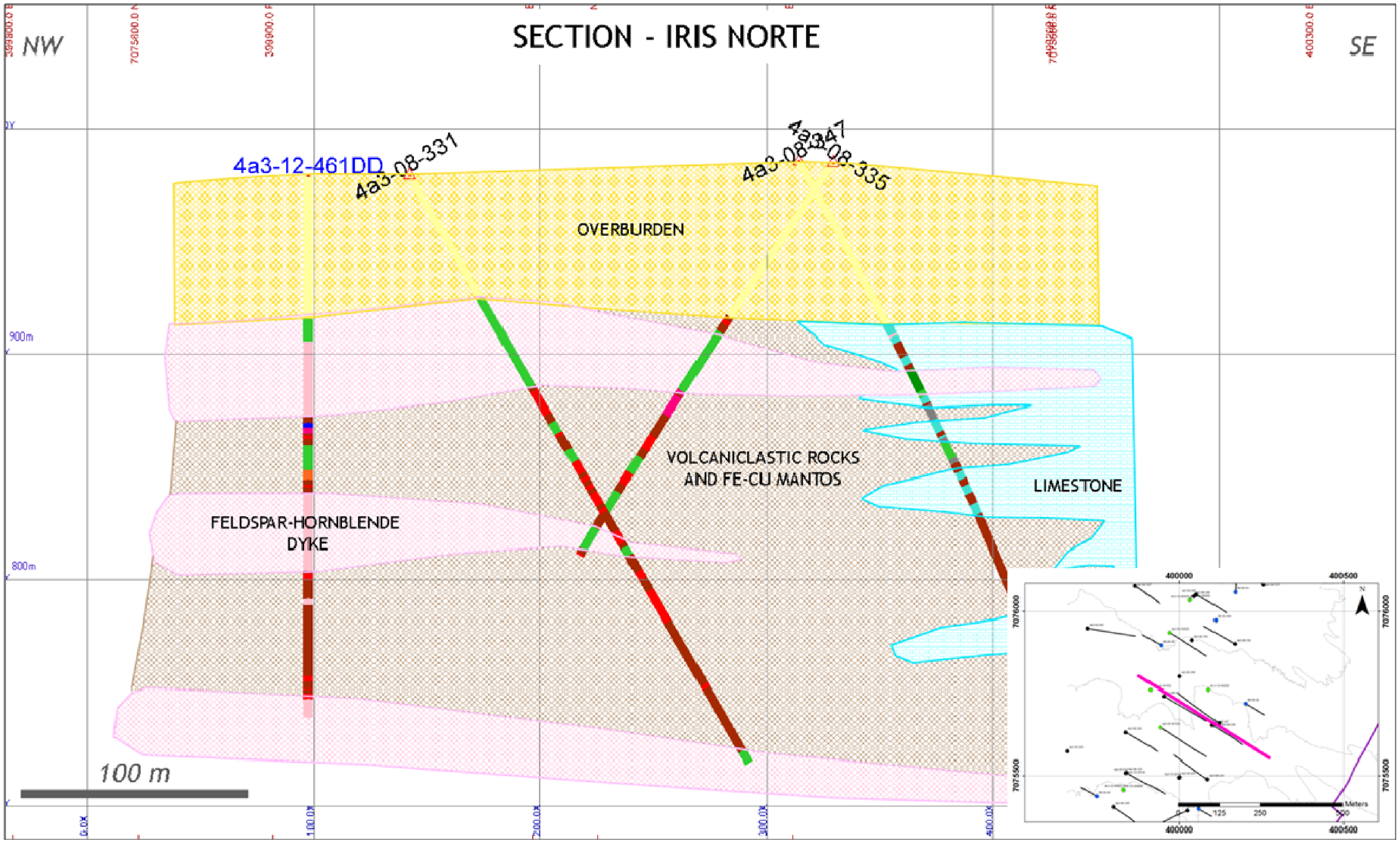
Estrellita has been faulted into a series of four blocks which step downwards to the north, with displacement across the faults ranging up to approximately 75 m. The overall footprint of the zone measures 900 m long by 450 m wide and is up to 100 m thick. The deepest drill intersections are in the order 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.

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.

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.

7.4 Prospects/Exploration Targets

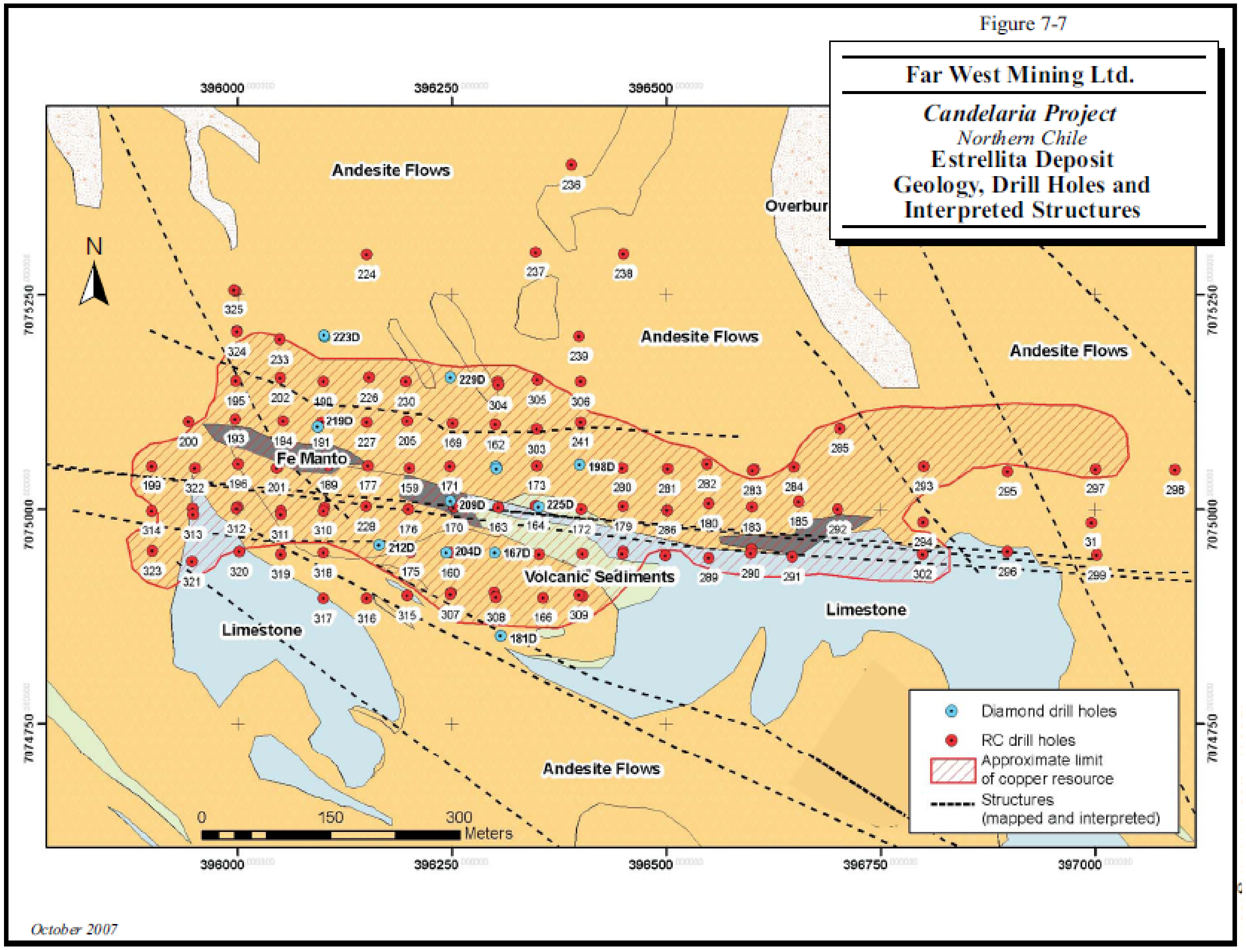
7.4.1 Estrellita and Estefánia Areas

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

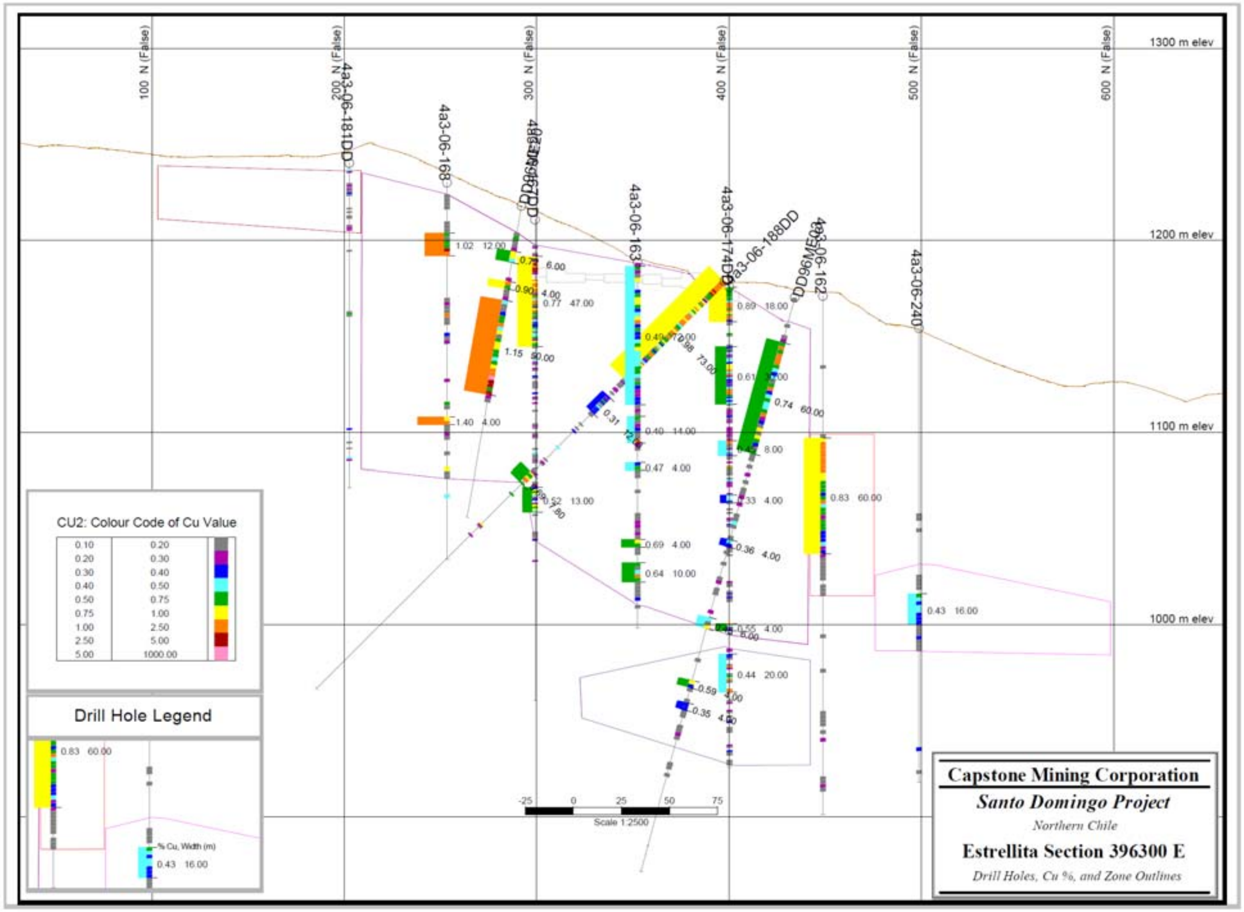
Andesite units both north and south of the Santo Domingo fault have been cut by a closely spaced (100 m to 200 m) set of northwest striking and steeply northeast dipping carbonate veins mineralized with specular hematite and copper oxides. Although these veins historically have supported very small-scale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

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.

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

8.1 Introduction

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 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 the 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.

8.2 Candelaria Deposit

Lundin Mining's Candelaria mine is located 20 km south of Copaipó. 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 strata-bound 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.

8.3 Manto Verde Deposit

Audley Capital's 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 multi-phase 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.

8.4 Comment on Section 8

The QP considers that exploration programs that use copper-rich IOCG deposit models are appropriate to the Project area.

9.0 EXPLORATION

9.1 Grids and Surveys

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

The topography used was from a detailed aerial survey of the planned plant site area using a scale of 1:1,000 and 1 m contour spacing, prepared by Fugro Interra S.A. (Fugro) for Capstone in 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 route. The supporting grid for the Project and the pipeline system consists of six main points and a secondary grid of 53 points.

Fugro 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 km^2 of geological mapping at 1:25,000 scale was completed during 2003–2005, and used for exploration targeting.

9.3 Geochemical Sampling

Far West 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 gravity 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-trending 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 the Iris deposit.

9.4.2 Ground

Far West 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 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:

- Daroch, G., 2011: Hydrothermal Alteration and Mineralization of the Iron Oxide (-Cu–Au) Santo Domingo Sur Deposit, Atacama Region, Northern Chile: unpublished M.Sc. thesis, University of Arizona, Tucson, Arizona, United States, 90 p
- Duran, M., 2008: Paragenesis of the Santo Domingo Sur Iron Oxide–Copper–Gold Deposit, Northern Chile: Unpublished M.Sc. thesis, Queen's University, Kingston, Ontario, Canada, 100 p.

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 Mt to 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 January 2015, a total of 603 core and RC holes (149,944 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 10-1 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 between May 2004 and 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, a centre return hammer and a 5.5 in. (13.97 cm) carbide button bit.

Core drilling used various drill rig types. HQ-size core (63.5 mm diameter) was typically drilled to a depth of approximately 300 m below which NQ-size 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 to provide information on geology, mineralization and structure, and material for metallurgical testwork. Reverse circulation drill holes were designed to tighten the drill spacing within the initial proposed mining areas and to provide sample material for metallurgical testwork.

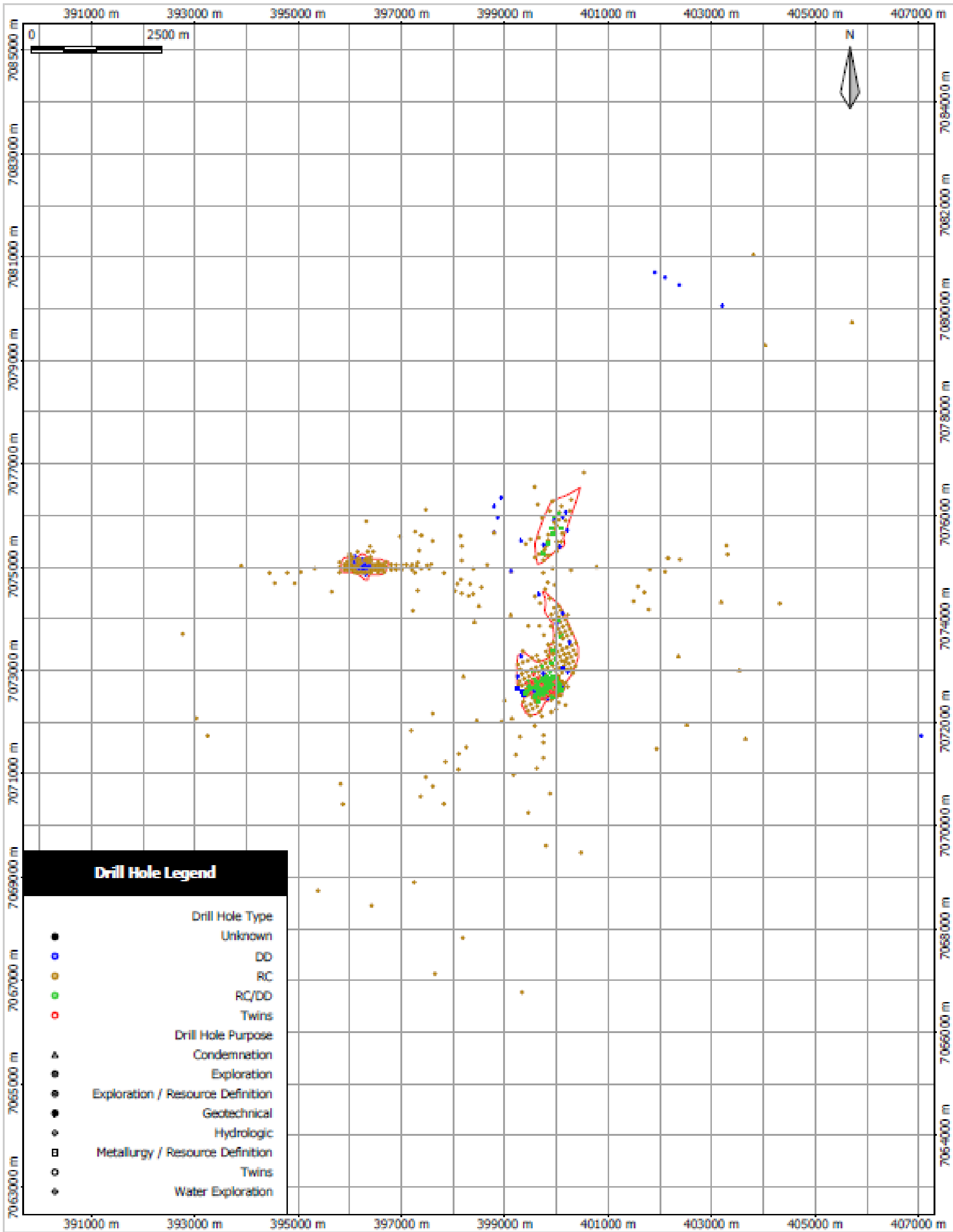
Table 10-1: Drill Summary Table by Hole Purpose

Purpose	Number	Length (m)
Condemnation	18	4,818.00
Exploration	191	43,344.35
Exploration/resource definition	227	66,781.1
Geotechnical	21	3,531.70
Hydrological	19	1,890.90
Metallurgy/resource definition	117	28,014.32
Water exploration	10	1,564.00
Grand Total	***603***	***149,944.37***

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

Type	Number	Length (m)
Core	51	9,823.40
RC	446	112,841.90
RC/Core	106	27,279.07
Grand Total	***603***	***149,944.37***

Figure 10-1: Project Drill Location Plan



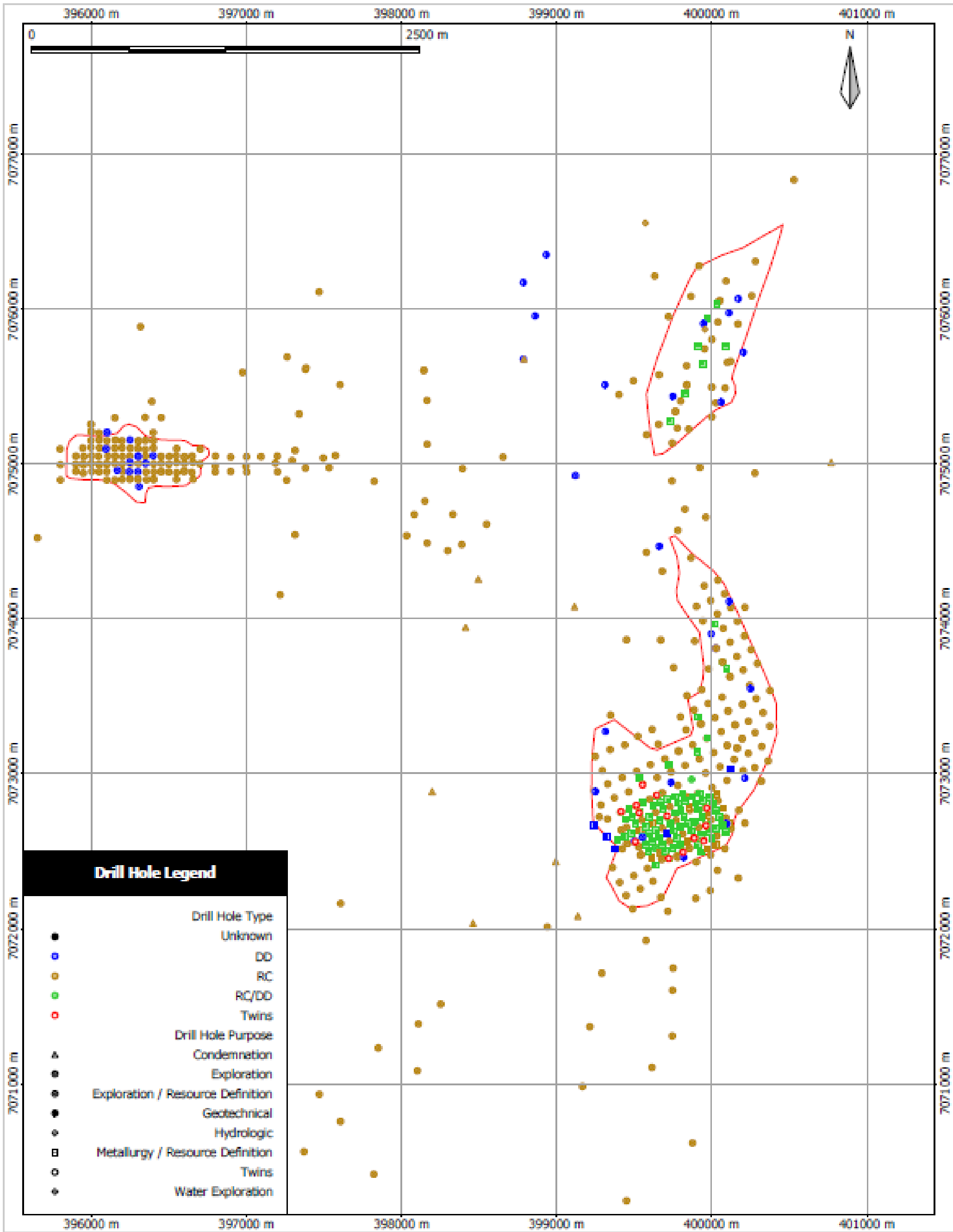
Note: Figure prepared by RPA, 2018

Table 10-3: Drill Summary Table – Drill Holes Supporting Resource Estimate

Area	No. of Holes	Type	Length (m)
SDS	103	RC	31,810
SDS	88	DD	22,837
SDS	143	RC	30,528
Estrellita	13	DD	2,366
Iris/Iris Norte	102	RC	28,273
Iris/Iris Norte	11	DD	3,212
Totals	***464***	**-**	***120,168***

Note: SDS = Santo Domingo Sur. DD = core drilling.

Figure 10-2: Deposit Area Drill Location Plan



Note: Figure prepared by RPA, 2018

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 a Microsoft Access (Access) database for the Project. Data were exported as required to Gemcom (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 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 were 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 113 holes (102 RC holes and 11 core 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, hence 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 between 2006 and 2010 and comprised a total of 28 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 Capstone to gather geotechnical data to complete slope calculations for the Santo Domingo Sur/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, Santo Domingo Sur/Iris, and the proposed TSF area.

10.8.2 Condemnation Drilling

Condemnation drilling was conducted by Far West during early 2011 and by Capstone during early 2012. A total of 3,576 m in 13 RC holes were drilled in the proposed waste rock facility (WRF), process plant and tailings areas. The condemnation drilling was in addition to 5,627 m in 20 old exploration drill holes that had been drilled within the boundaries of the proposed mine site installations (waste rock facility 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 inductively coupled plasma (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 Reverse Circulation 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 using the formula:

$$SG = M_{air}/(M_{air}-M_{w})$$

Where

M_{air} = weight of the dry sample in air

M_{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 pyncnometer method in pulps prepared from RC samples. The specific gravity was determined using the following formula:

$$SG = Ws/Wds \times SGs$$

Where Ws is the weight of the sample; Wds is the weight of the displaced solvent; and SGs is the specific gravity of the solvent. The most 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:

$SG = 2.53 + 0.0276*Fe$.

A summary of the specific gravity data is included by major lithological unit in Table 11-1.

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 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 also has ISO 17025 accreditation.

The check laboratory was Andes Analytical Assay Ltda. In Santiago, which also holds ISO 9001:2008 accreditations.

Table 11-1: Summary Table, Specific Gravity

Rock Type	No. of Samples	SG (t/m^3)
Andesite	488	2.90
Andesite Tuff	685	3.05
Basement	9	2.85
Diorite	48	2.81
Dyke	70	2.69
Fault	34	3.06
Limestone	3	2.72
Manto (High Fe)	883	3.55
Sedimentary	7	2.75
Total Number SG Samples	***2,227***	
Average SG		***3.20***

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 ICP methods. Samples were initially analyzed using ALS Chemex procedure ME-ICP61, which is ICP following four acid, total digestion (HF-HNO_3–$HclO_4$ 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 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 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. No CRMs were inserted for cobalt.

11.8 Databases

Drill cuttings and core were logged and data collected entered into a Microsoft Excel (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 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 Excel file provided by the laboratory. Once complete, data from the ledger are imported to a master 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 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, 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, including RPA, 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 QA/QC 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 area."

12.2 Lacroix (2006)

Data provided to RPA for verification purposes were based on exports from the Gemcom database in 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. 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. 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 copper, gold, and iron 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. This work consisted of importing drill data collected since 2009, inspection and validation. A few errors were captured during the import and validation; these were corrected. RPA also reviewed the validation work conducted by Capstone personnel.

12.7 RPA (2018)

The cobalt database was provided to RPA in Excel format. No inconsistencies were noted. RPA verified cobalt entries in the Excel sheet with the assay certificates available. No errors, inconsistencies or discrepancies were noted.

12.8 RPA Verification Results

12.8.1 Analytical QA/QC

Standards

Certified reference materials, 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 25 samples. Standards assays comprised 7% of the total sample count in 2011–2012 drilling. The same standards 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 50 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 10 times the detection limit. The blank 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 displays 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

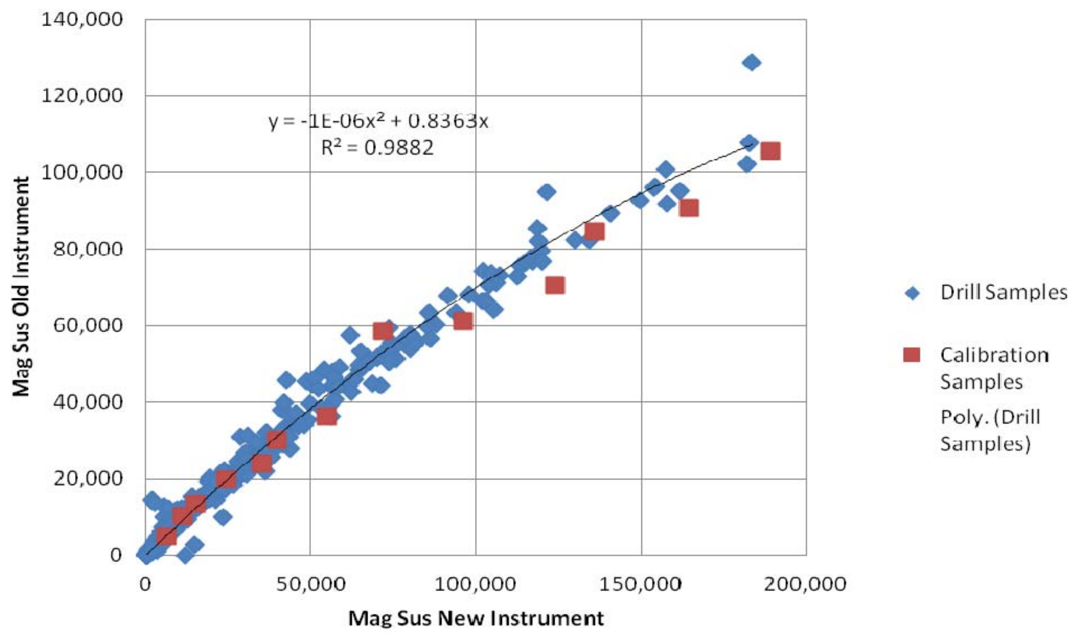
Magnetic susceptibility measurements were conducted to provide a basis for estimation of the proportion of the iron that could be recovered by magnetic means. As stated previously, most of the measurements were carried out on reject material. However, in 2,093 cases, due to a lack of reject material, 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 2011 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.

Figure 12-1: Comparison of Readings from the New and Old Magnetic Susceptibility Instruments



Note: Figure prepared by RPA, 2012.

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.

12.8.2 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. Additional twin drilling was carried out in 2014 and 2015.

For the 2010 twin drilling, 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 hole deviation.

In 2018, RPA compared copper, gold, iron, and cobalt assay values of the drilling done in 2014 and 2015 with the corresponding twin hole in the database (Figure 12-2). In RPA's opinion, for most of the pairs, the assay results compared reasonably well.

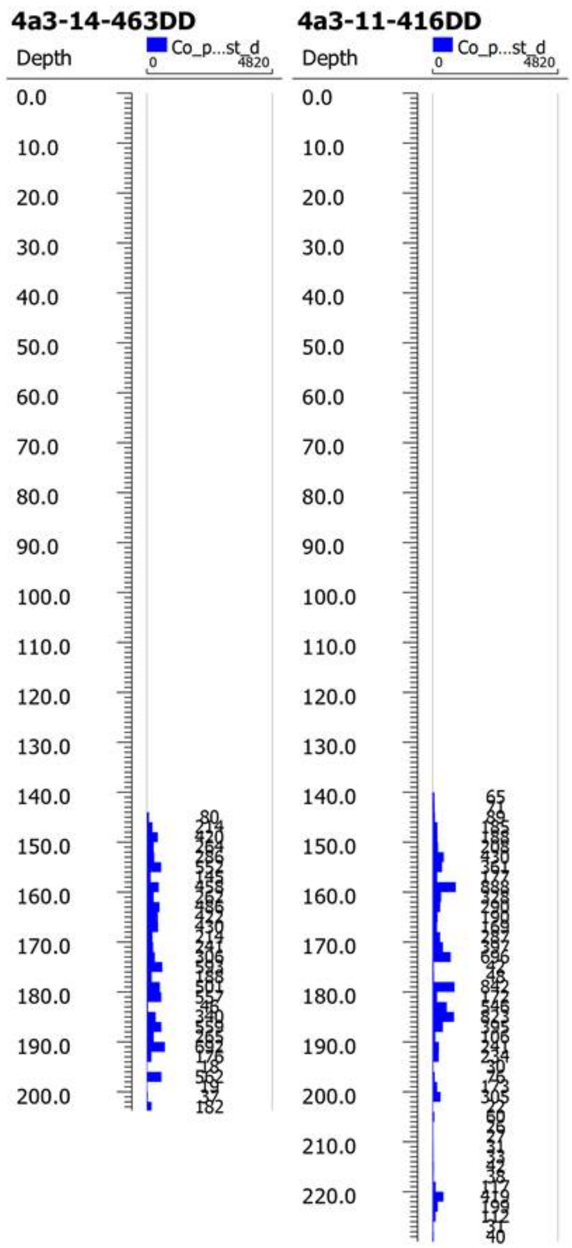
In RPA's opinion, the twinning has provided a reasonably consistent verification of the earlier drill results particularly considering the differences in assay protocols and possible survey errors.

12.8.3 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 (DT) testwork in order to first determine if a saleable iron 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 Santo Domingo Sur and Iris drill holes. 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.

Figure 12-2: Comparison of Twin Hole Results for Cobalt



Note: Figure prepared by RPA, 2012.

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 12-3, 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 DT 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 12-4).

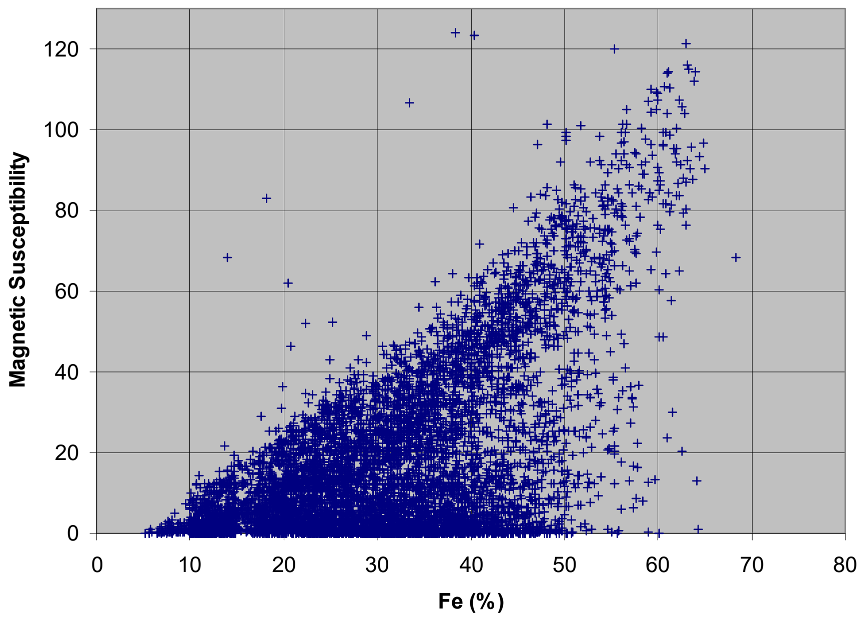
There is a degree of scatter in the data points plotted in Figure 12-4. In order to model the mass recovery with magnetic susceptibility, Capstone chose an equation which skirts along the bottom of the point distribution. The relationship thus derived is:

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

Where: MS = magnetic susceptibility reading/1,000

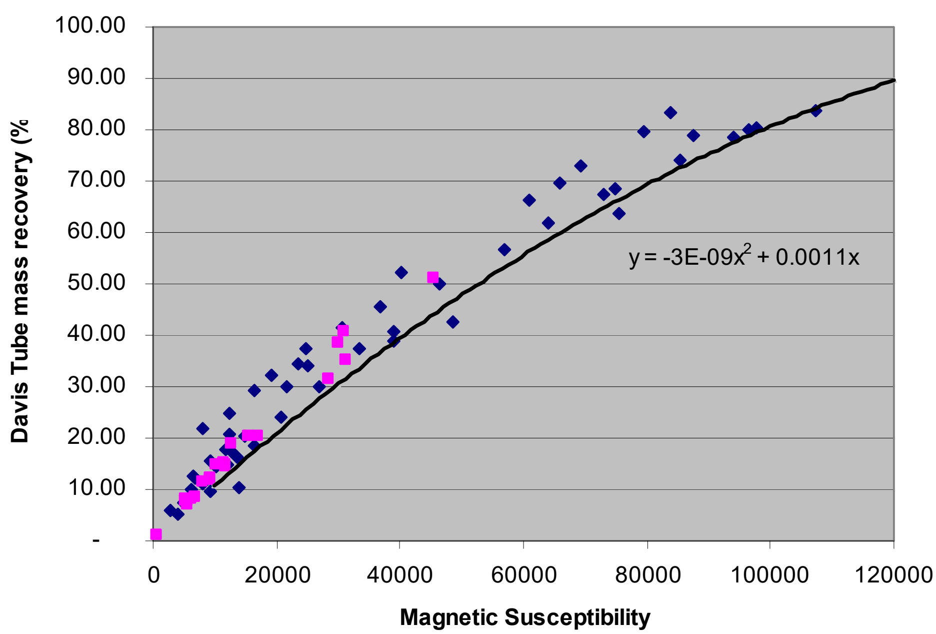
This line represents the minimum mass recovery observed for the corresponding magnetic susceptibility values; in RPA's opinion, this will result in conservative estimates of mass recovery.

Figure 12-3: Total Iron versus Magnetic Susceptibility



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

Figure 12-4: Magnetic Susceptibility versus Mass Recovery



Note: Figure prepared by RPA, 2012.

12.9 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 that the blank material contained a high background concentration of copper. There are no concerns regarding the blank 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 2011 Pre-feasibility Study phase, a total of 128 core samples were selected, with the following tests completed:

- Bond ball mill work index (BWi)
- Bond 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 2011 Pre-feasibility Study comminution requirements for the grinding circuit.

As part of the 2014 Feasibility Study phase, the following was undertaken:

- Two comminution circuit sizing exercises were completed
- A total of 91 of the 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 samples termed the “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

Date	Testwork Type	Laboratory/Testwork Facility	Work Performed
2006	Comminution	SGS Santiago	Grindability response testwork on two drill core samples
2008	Comminution	SGS Mineral Services	Grindability response testwork on five composite drill core samples. Included BWi; Bond Ball Modified tests; RWi; and SAG power index (SPI) tests
	Cu flotation	SGS Santiago	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
2009	Magnetic concentrate	SGS Lakefield	Response of composite samples to magnetic separation using DT laboratory tests
		SGA	LIMS testing to develop a marketable magnetic concentrate
2010	Comminution	SGS Santiago	Grindability program on 128 samples; ball mill calibration program on four samples
		Phillips Enterprises (Advanced Terra Testing Inc.)	Crushing and geotechnical program on 128 samples
	Cu flotation	SGS Santiago	Copper mineralogy
		Aminpro	Chemical and mineralogical analysis (QEMSCAN) and rougher kinetic flotation tests on five samples; all rougher flotation tests were conducted using sea water
		SGS Lakefield	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: SGS Lakefield water, Capstone-supplied saline water, and synthetic sea water; five flotation tests to determine the effect of primary grind on copper recovery
	Magnetite concentrate	SGA	Magnetite recovery tests on five whole ore samples of the same composite at a final grind of less than 63 µm
2011	Comminution	Ammtec	SAG mill competency; confirmatory ball mill tests; 19 samples tested

Date	Testwork Type	Laboratory/Testwork Facility	Work Performed
		(now ALS-Ammtec)	
		SGS Santiago	RWI, BWI, Ai and SMC tests on 58 samples
	Cu flotation	SGS Lakefield	Copper flotation performance testing on four composite samples (Eight-Year, Hematite, Magnetite and Oxide) and a set of 38 variability samples; investigated optimized use of sea water
		SGS Lakefield	Bulk flotation tests at optimum conditions to produce copper rougher concentrate for regrind power testing; copper rougher flotation tailings of the Hematite and Magnetite composites for LIMS recovery studies; and copper cleaner concentrate from the Eight-Year composite for concentrate filtration and washing tests
	Magnetite concentrate	SGA	Optimization testwork for LIMS iron recovery; fresh water and sea water used; variability sample testing to determine the correlation between DT test results and LIMS cleaner tests; and correlation between Satmagan/magnetic susceptibility head grade and DT test recovery; iron recovery variability tests on 35 samples that represented five defined ore zones
		Metso	Copper and LIMS iron rougher concentrates to determine the specific power required for regrinding
		Ausenco PSI	Rheology tests on a LIMS magnetite concentrate sample
		SGS Santiago	DT test and Satmagan tests using 52 rougher tailings generated by from rougher kinetics
	Filtration	Outotec	Concentrate filtration tests to determine filtration equipment (Larox PF and Ceramec)
2012	Cu flotation	SGS Santiago	Flotation testwork program using 51 variability samples for rougher kinetic test and 15 variability samples for open cycle test (OCT)
	Physical characterization	Jenike and Johanson (Chile)	Size characterization, flow properties, cohesive strength and bulk density for modeling the operation of the stockpile and material handling including the filter hopper
	Settling (concentrate and tailings)	FLSmidth	Tailings thickener tests and concentrate thickener tests on tailing samples and magnetite concentrate samples respectively; different flocculant and thickener technologies tested to achieve the highest settling rate; tailings thickener tests and concentrate thickener tests on tailings samples and magnetite concentrate samples respectively; different flocculants and doses tested; rheological tests

Date	Testwork Type	Laboratory/Testwork Facility	Work Performed
		Outotec	Tailings thickener tests and concentrate thickener tests on tailing samples and magnetite concentrate samples respectively; 10 tests completed for magnetite concentrate and 11 tests completed for tailings; two additional tests completed to compare two different flocculants
	Filtration	Outotec (Chile)	Filtration tests for a magnetite concentrate at 65% w/w solids; eight tests completed to obtain a chloride concentration lower than 300 ppm Cl
2013	Comminution	Moly-Corp Chile	Grinding test program using rougher magnetic concentrate samples provided by Capstone to determinate conditions for the design and operation of the magnetite regrind mill
	Cu flotation	SGS Lakefield	Rougher kinetics, cleaner flotation, LIMS tests and rougher magnetic circuit tests were completed on three composites (SD1, Hematite and Magnetite); tailings produced used for settling and rheological testing
	Physical characterization	Patterson & Cooke (Chile)	Rheology test program using two tailings samples to determinate the physical properties of the tailings
	Settling	Outotec (Chile)	Tailings thickener test using Hematite and Magnetite tailings provided by SGS Lakefield; two stages of thickening tested using different flocculants and doses; rheological tests
		FLSmidth (USA)	Sedimentation and rheology testing programs for Hematite and Magnetite tailings provided by SGS Lakefield to determine the sizing and operational parameters for a tailings thickeners considering both composites
		Tenova Delkor (Canada)	Settling and thickening tests on two composites (Hematite and Magnetite) provided by SGS Lakefield; sizing for the tailings thickeners completed considering two separate stages of thickening
2014	Cu flotation	SGS Santiago	Rougher, cleaner kinetics and locked cycle tests (LCT) completed on three AMEC-selected composites; variability samples (open circuit and locked cycle) tested to validate the algorithm developed during the 2011 Pre-feasibility Study
	Magnetite Concentrate	ALS Ammtec	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

Date	Testwork Type	Laboratory/Testwork Facility	Work Performed
2015	Pilot plant	ALS Santiago	Pilot plant testwork included: hardness testing of eight composites for BWi, RWi, Ai, SMC, JK dropweight and low-impact crusher The pilot plant generated concentrate for further testing including: thickening and filtration testing for copper and magnetite concentrates
2018	Cu flotation	Aminpro (Chile)	Rougher and cleaner kinetics with desalinated water for modelling the flotation circuit Selection of samples for the development of a recovery algorithm for copper and gold with desalinated water. Rougher kinetics on samples with different head assays LCT with desalinated water to determine a recovery factor for the cleaning stage and validate the algorithm developed by Aminpro

- During the 2015 pilot plant testwork, eight composites were included for hardness analysis (these composites represented each of the first five years of operation and a combined composite). BWi, RWi, Ai, SMC, JK-dropweight and low-impact crusher were evaluated
- In 2018, testwork was carried out using desalinated water to determine the recovery algorithms for copper and gold.

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 Axb 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 Axb values of the ore types. The Hematite zone is the most competent with the lower Axb 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 Axb 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 Axb 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.

13.1.2 Comminution Circuit Testwork

The comminution characterization pilot plant program started with an initial assessment of drill core from the 2015 drilling program. The comminution characterization program was focused on generating data to evaluate comminution response to obtain process design parameters.

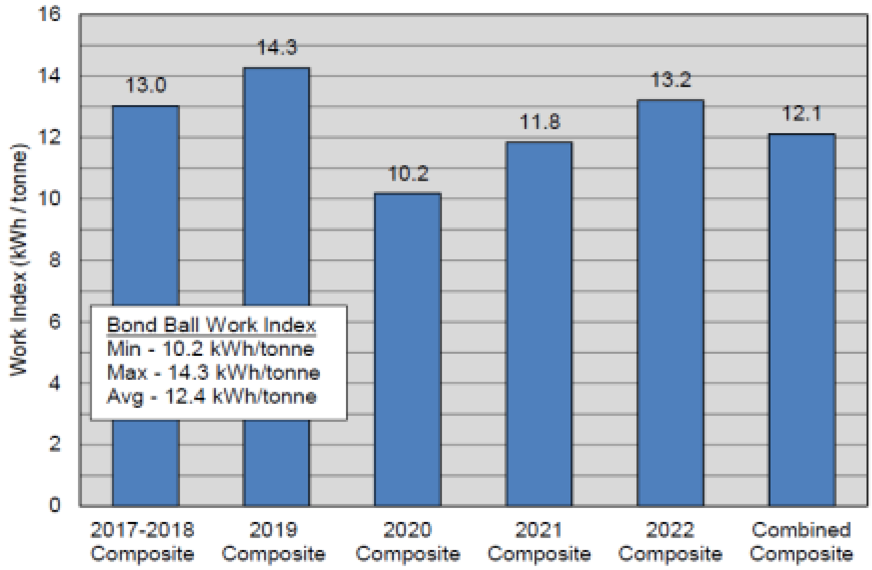
ALS executed the following comminution tests: Crushing work index, SMC, Ai, Bwi and Rwi tests.

- The SMC test generated a relationship between specific input energy (kWh/t) and the percent of broken product passing a specifc sieve size. This is an abbreviated form of the JK dropweight test but uses only one particle size that is impacted at five different energy level. The particle size used was 35 mm. The results were used to determine the dropweight index (DWi)
- The Bond all mill grindability test is used to calculate the net power requirement for ball milling. The test is performed according to the original Bond procedure using standard equipment. The test requires about 8 kg, top size 6 mm material prepared at the testing facility
- The Bond rod mill grindability test is used to calculate the net power requirement of the mill circuit. The test is performed according to the original Bond procedure using standard equipment. The test requires about 18 kg, top size 20 mm material prepared at the testing facility
- The Bond abrasion test measures the Ai, which can be used to estimate steel media and liner consumption for crushers, rod mills and ball mills.

Bond Ball Mill Work Index Results

Bond rod mill work index tests were carried out on six composites. The results indicated (Figure 13-1) that the minimum and maximum values were in the range of 10.2 kWh/t and 14.3 kWh/t. The average value was 12.4 kWh/t. These results indicate that the composites are considered to be moderately soft.

Figure 13-1: Pilot Plant Bwi Test Results



Note: Figure from Informe Final Pruebas Planta Piloto Minera Santo Domingo (2015)

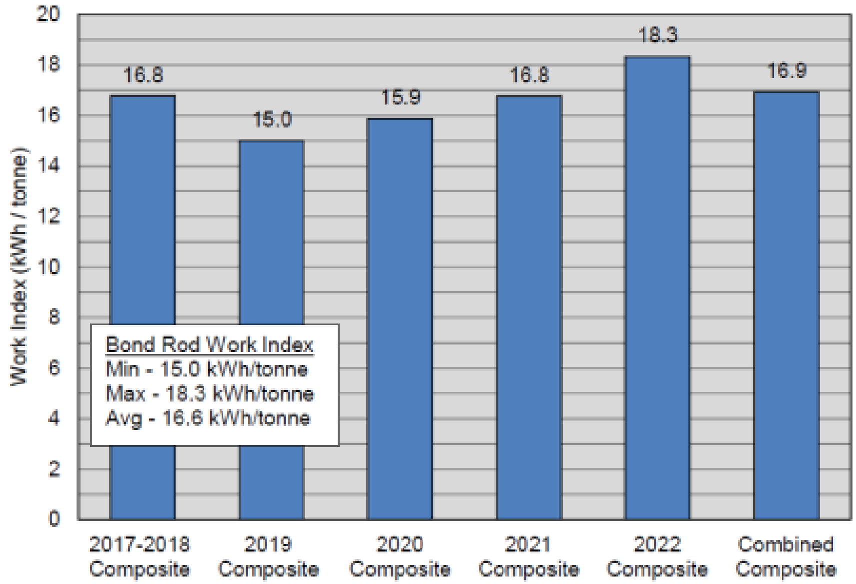
Bond Rod Mill Work Index Results

Bond rod mill work index tests were carried out on six composites. The results indicated (Figure 13-2) that the minimum and maximum values were in range 15 kWh/t and 18.3 kWh/t. The average value was 16.6 kWh/t. The results indicate that the composites tested are considered to be moderately hard.

Bond Abrasion Index

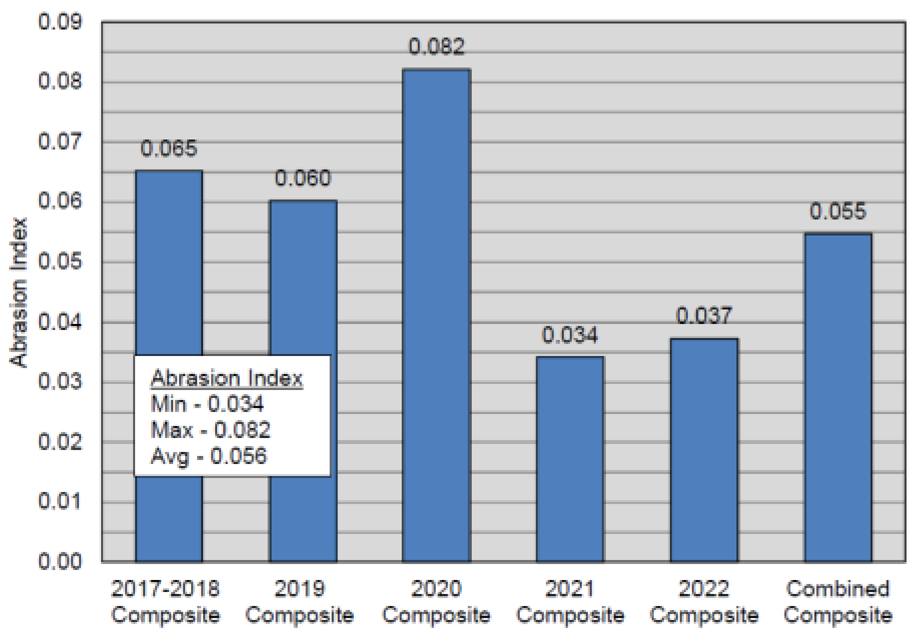
Bond abrasion index tests were carried out on six composites. The results indicated (Figure 13-3) that the minimum and maximum Ai values were in the range of 0.03–0.08. The average value was 0.06. The results indicate that the composites tested had low abrasion.

Figure 13-2: Pilot Plant BRWi Test Results



Note: Figure from Informe Final Pruebas Planta Piloto Minera Santo Domingo (2015)

Figure 13-3: Pilot Plant Ai Test Results



Note: Figure from Informe Final Pruebas Planta Piloto Minera Santo Domingo (2015)

SMC

The SMC tests were performed using a fraction between -22 mm +19 mm. The minimum and maximum values derived from the Axb test were 46.1 and 56.8, respectively (Figure 13-4). High Axb values indicate soft mineral and low Axb values indicate hard mineral. For the material tested the average Axb was 50.8 which JKTech categorizes as medium hardness.

JK Dropweight Test

These tests were performed on samples representing two different lithologies, manto, and andesite (ANDT). Densities were measured for 30 samples and the average densities of Manto and ANDT composites were 4.21 and 3.40, respectively. The ANDT density presented a bimodal distribution, with a lower value of 3.2 and a higher value of 4.2. The Manto density presented similar values.

The Axb results for Manto and ANDT were 86.3 and 38.8, respectively.

The results indicate that the two lithologies have different properties measured by the JK dropweight tests.

Low-Impact Crusher Test

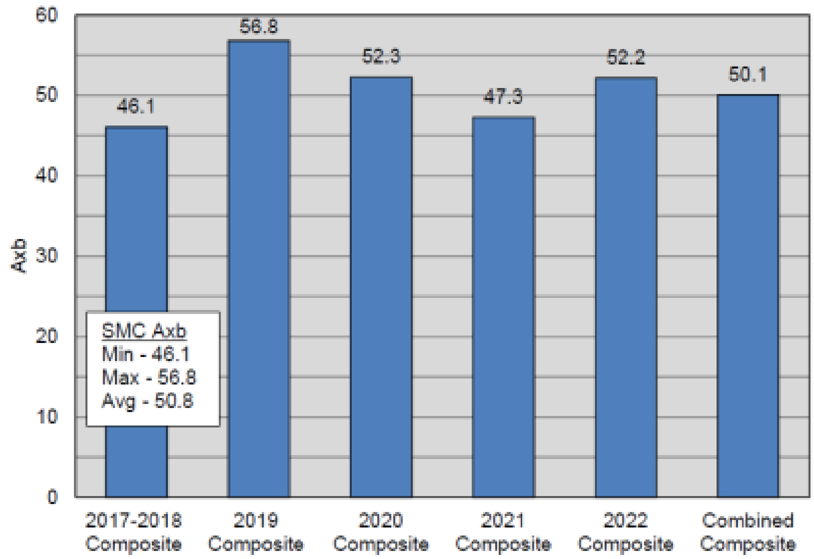
A low-impact crushing test was also performed on the two lithologies, Manto and ANDT. The apparent density and crusher work index (CWi) were tested. The CWi results for ANDT and Manto were (CWi (average)) 8.8 kWh/t and 7.3 kWh/t, respectively (see Figure 13-5). The JKTech testwork identified some increases in the hardness of the mineral with increasing particle size. The CWi test did not corroborate this observation.

Bulk density results are shown in Figure 13-6.

Metso Regrind Testwork

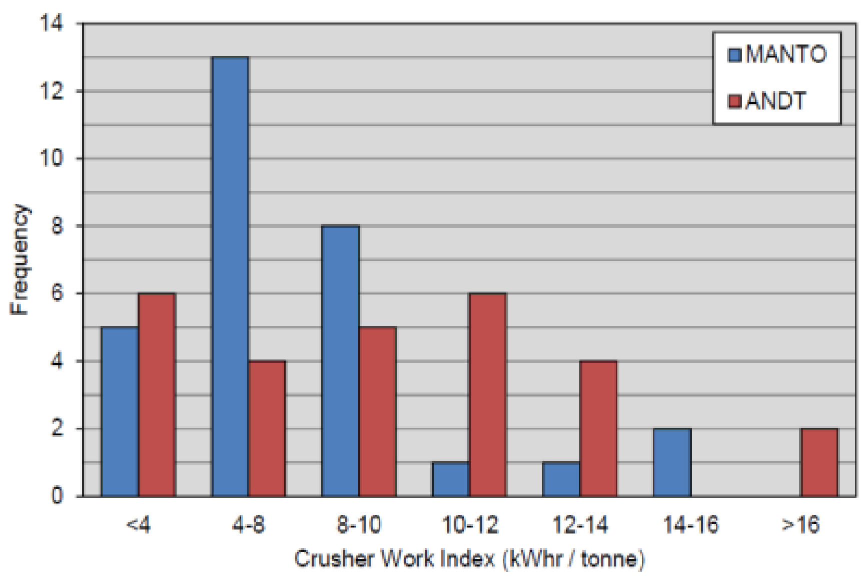
Magnetite ore regrind mill tests were carried by Metso. The purpose of the testing was to obtain the specific energy consumption of representative samples targeting a P80 of 44 µm. The grain size distribution of the feed sample is shown in Table 13-2.

Figure 13-4: Pilot Plant SMC Test Results



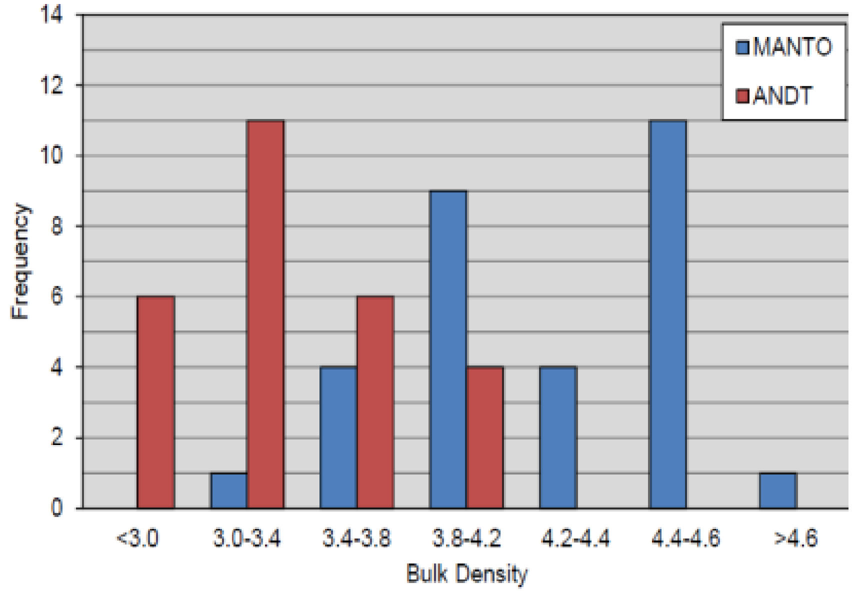
Note: Figure from Informe Final Pruebas Planta Piloto Minera Santo Domingo (2015)

Figure 13-5: Pilot Plant Cwi Test Results



Note: Figure from Informe Final Pruebas Planta Piloto Minera Santo Domingo (2015)

Figure 13-6: Pilot Plant Bulk Density Test Results



Note: Figure from Informe Final Pruebas Planta Piloto Minera Santo Domingo (2015)

Table 13-2: Feed Grain Size Distribution

Microns	% Passing
600	
425	99.8
300	95.6
212	83.0
150	65.2
106	48.6
75	30.0
53	17.0
45	13.6
38	11.9
25	9.9
d80 (µm)	201.2

The predicted specific energy consumption (VTM) for the sample with F80 of 201.2 µm and P80 of 44.0 µm using 19.1 mm media was 7.95 kWh/t. The particle size versus specific energy consumption is shown in Figure 13-7 and the feed and product particle size distribution are shown in Figure 13-8.

13.1.3 Copper Flotation Testwork

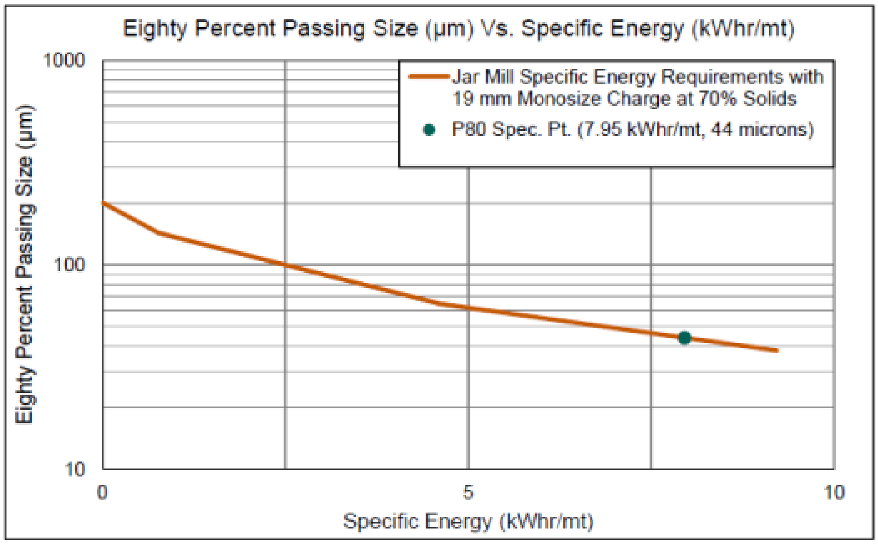
SGS Lakefield conducted 38 variability sample LCT flotation tests on the Eight-Year, Hematite and Magnetite type composite samples as part of the 2011 Pre-feasibility Study. The Eight-Year sample gave the best copper recovery result.

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 type 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 cycle testwork (OCT) conditions were:

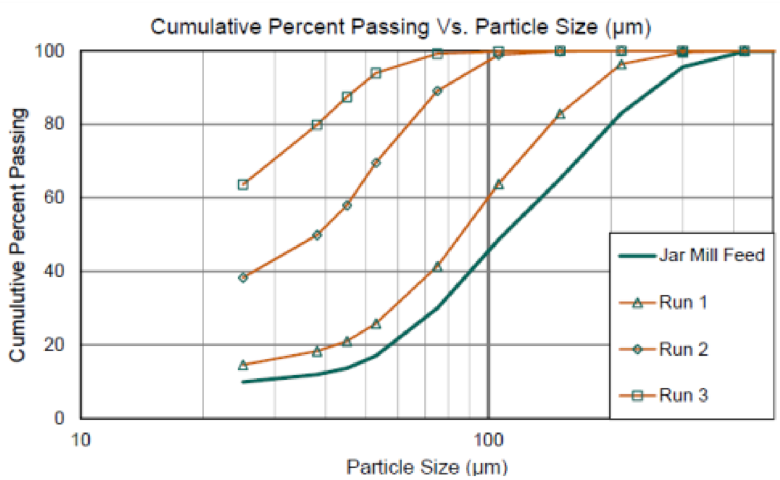
- Grind stage: ball mill (top size 1″)
- Regrind stage: ball mill (top size ½″)
- Grind stage: 65% w/w
- Regrind stage: 50% w/w
- Flotation paddle: every 10 seconds
- Lime used: $Ca(OH)_2$ powder and 10% milk of lime.

Figure 13-7: P80 vs Specific Energy Consumption



Note: Figure from Informe Final Pruebas Planta Piloto Minera Santo Domingo (2015)

Figure 13-8: Feed and Product Particle Size Distribution



Note: Figure from Informe Final Pruebas Planta Piloto Minera Santo Domingo (2015)

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

- No addition of SMBS to rougher flotation section
- No addition of SMBS to cleaner flotation section
- No addition of sodium cyanide (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 flow sheet used for the LCTs consisted of:

- The flow sheet 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 2011 Pre-feasibility Study sea water 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. It is expected that the use of desalinated water will improve these results.

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 2011 Pre-feasibility Study testwork program.

Three separate composites (Five-Year Average, Hematite sample and Magnetite sample) 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 type material is 0.59%
- Magnetite material has a magnetic susceptibility level above 8,000. The average copper grade for the Magnetite type material is 0.31%
- The Five-Year Average composite was selected to be representative of a combination of Hematite and Magnetite type material during the first five years of operation. The average copper grade for the Five-Year 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 that they did not contain soluble copper levels exceeding 10% of the copper 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 13-9). Chalcopyrite is the dominant species within the entire deposit.

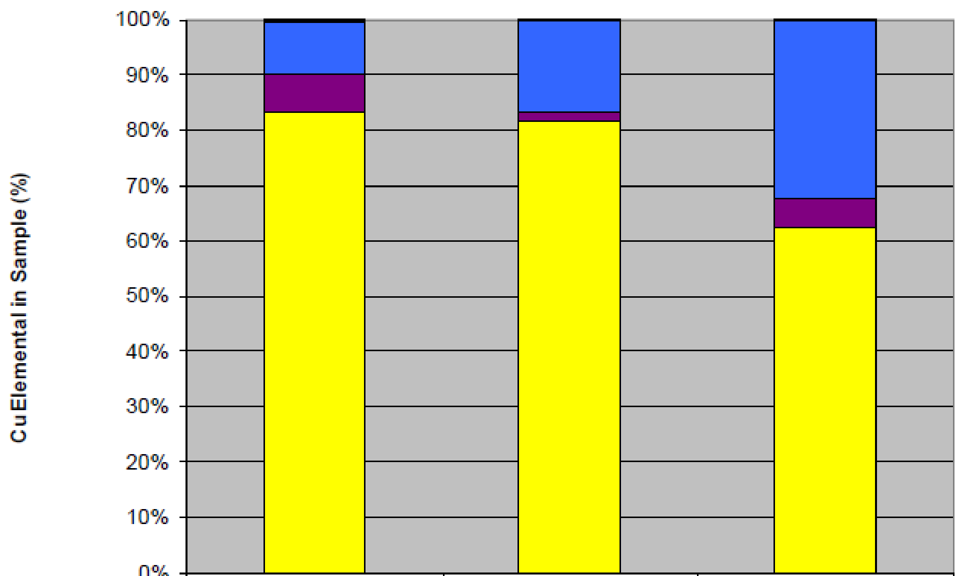
Kinetic Testwork

The conditions used in the kinetics tests were based on the 2011 Pre-feasibility Study 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 copper recovery was reached in the test using an SMBS addition rate of 50 g/t (92.8% recovery); elevated pH levels did not improve copper recovery. The use of SMBS in the cleaning circuit increased the copper 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 copper grades were noted when SMBS was used
- There is a significant difference in the copper recovery achieved at the beginning of the first cleaner flotation stage (approximately 10% difference when comparing the highest copper recovery of 72.2% at two minutes versus the lowest copper recovery achieved of 53.8%) due to the use of SMBS
- From two minutes to six minutes there is a notable difference in copper recovery (between 15% and 25% for the different tests). After six minutes, the copper recovery rate reduces significantly and only increases slightly until the end of 25 minutes. It appears that copper recoveries are still increasing after 25 minutes but at a very slow rate

Figure 13-9: Copper Mineralogy



	5 Year Average	Magnetite	Hematite
Cu Oxides	0,00	0,00	0,02
Cu-As-Sb Suphosalts	0,12	0,01	0,04
Covellite	0,05		
Chalcocite	9,87	16,88	32,17
Bornite	6,69	1,42	5,34
Chalcopyrite	83,27	81,32	62,08

Note: Figure from SGS (2010)

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

Based upon the Five-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 flow sheet and test conditions derived from the Five-Year Average composite testing. Variability samples were selected to represent the Santo Domingo deposit and included consideration of:

- Copper and iron head grades
- Spatial representation
- Lithology
- Within the limits of mineralization as defined by Production Plan V8.1, developed as part of the 2014 Feasibility Study.

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 types of andesite, termed ANDS (comprising subunits ANDT and ANDE) and MANTO. Combined, these two lithologies represent more than 90% of the deposit.

Variability test results include:

- Two samples (Var 7 and Var 8) were tested for both OCT and LCT
- Var 7 had an OCT copper recovery of 87.9% at a concentrate grade of 40.1% Cu, and an LCT copper 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 copper recovery could be expected at a reduced copper concentrate grade. A copper recovery well in excess of 90% could be expected at a concentrate grade of over 30% Cu.
- Var 8 had an OCT copper recovery of 93.3%; at a concentrate grade of 32.3% Cu and an LCT copper recovery of 93.6% at a concentrate grade of 27.7% Cu. For Var 8, copper 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 copper recovery algorithm.

During the 2011 Pre-feasibility Study, a copper grade versus copper recovery algorithm was developed considering only the final (third) cleaner copper concentrate value. The methodology used in the 2011 Pre-feasibility Study 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 copper recovery value for the OCT tests. The use of desalinated water makes these results no longer valid.

The average copper recovery for all the samples (2011 Pre-feasibility Study samples, 2014 Feasibility Study variability samples, and 2014 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

In 2014, Outotec conducted thickener settling testwork on a third and final cleaner concentrate generated from a composite known as the Eight-Year Average as part of the 2011 Pre-feasibility Study 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.

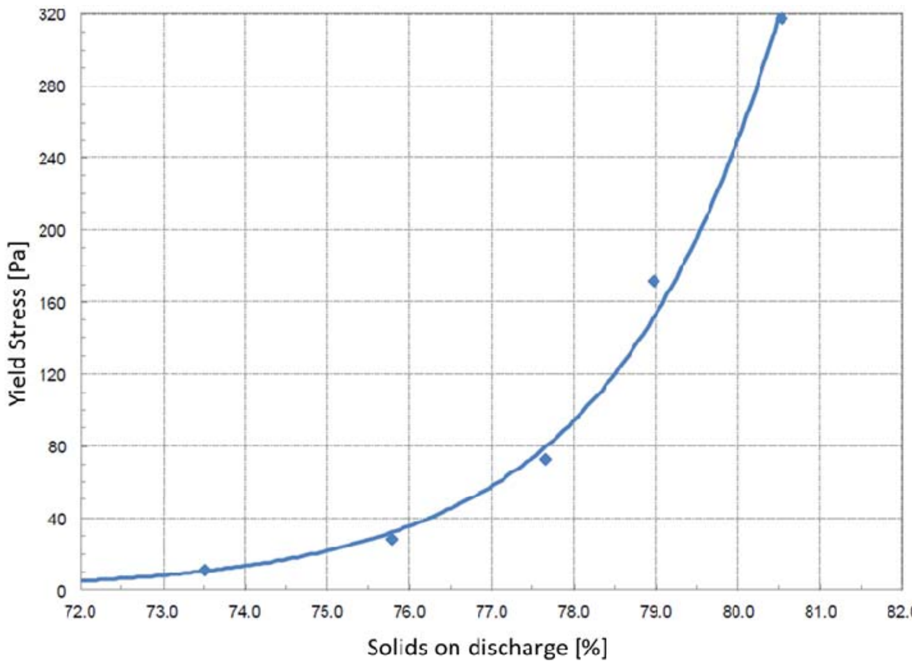
Two dynamic thickening tests were carried out to analyze the effect of the feed rate (t/hr/m^2) on the discharge (Cp%) and the clarity of the overflow (ppm). A rheology test of the thickener discharge was also completed. Feed rates used were 0.25 t/hr/m^2 and 0.40 t/hr/m^2. The behaviour showed a reduction 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/hr/m^2, 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 pilot plant testwork in 2015 used sea water and the flowsheet current at that time. Concentrate samples generated from the pilot plant were sent to FLSmidth and Outotec for testwork to determine sizing and operating parameters for the concentrate thickeners. Although the pilot plant used sea water, because the material has a low clay content, the thickening tests should provide valid results. These tests should be repeated following the next pilot plant testwork currently planned for 2019 to verify settling and rheology characteristics.

The samples received by FLSmidth and Outotec were the final copper concentrate, with a P80 target of approximately 43 µm, slurry pH in the range of 8.8 to 9.2 and solids SG in the range of 4.1 to 4.22. The testwork determined yield stress values as a function of solids concentration with the primary objective being to select a design value for torque for the thickeners and to predict the limits of underflow densities. Figure 13-10 shows the thickened underflow rheology results.

Figure 13-10: Discharge Rheology Curve



Note: Figure from Informe Final Pruebas Planta Piloto Minera Santo Domingo (2015)

The FLSmidth results showed that the thickeners can accommodate a solids loading of 0.166 m^2/t/d minimum unit area. At this unit area, the concentrate thickeners can produce underflow solids concentration of approximately 60% w/w with less than 0.9 hours of retention time depending on the composition of the material and a yield stress close to 10 Pa. These tests should be repeated using desalinated water to verify the final design criteria.

Copper Concentrate Filtration Testwork

In 2014, 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.

The pilot plant testwork in 2015 used sea water and the flowsheet current at that time. Concentrate samples generated from the pilot plant were sent to FLSmidth and Outotec for evaluation of the filtration characteristics using a plate and frame tower press. The objective of the testwork was to determine sizing and operating parameters such as filter cloth selection, cake thickness, filtration rate, filter cake moisture, cake handling for the copper concentrate filter. Although the pilot plant used sea water, because of the low clay content of the material the filtration tests should provide valid results and indicate the moisture content that can be achieved.

The samples received by FLSmidth and Outotec were final copper concentrate, with a P80 target of approximately 43 µm, slurry pH in the range of 8.8 to 9.2 and solids SG in the range from 4.1 to 4.22.

FLSmidth filtration tests reached the desired 8% to 10% w/w moisture target. Filtration rates were relatively high for a copper concentrate, in the range of 584 kg/hr/m^2 to 862 kg/hr/m^2. The cake thickness range tested was between 23 mm and 65 mm. In order to achieve a final moisture content below 9%, the optimum cake thickness was 30 mm and 45 mm, with a residence time of approximately 140 seconds and 180 seconds, for FLSmidth and Outotec, respectively (Figure 13-11).

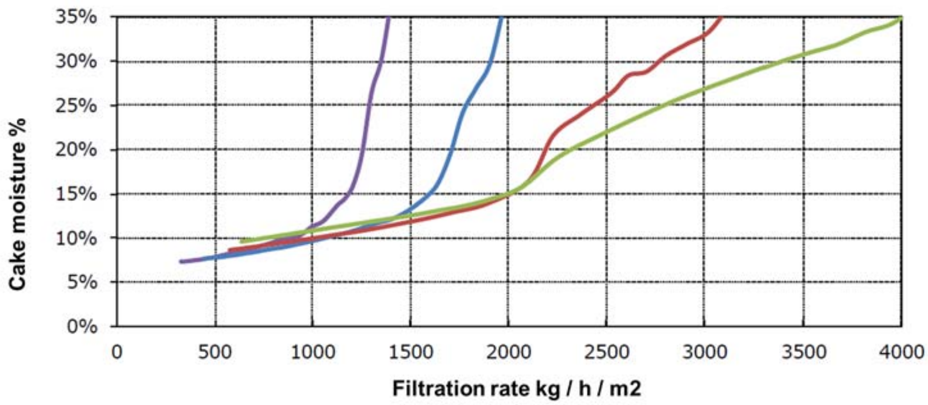
Copper concentrate filtering testwork was not carried out using desalinated water for the 2018 update. Using desalinated water in the process will eliminate the need for washing to lower the chloride content in the final concentrate and the water balance was adjusted. Pilot plant testing in the next stage will be used to optimize the type and size of the filtration equipment.

13.1.4 Magnetic Separation Testwork

During the 2011 Pre-feasibility Study, several tests were carried out to set the recovery of magnetite using the tailings from the primary copper flotation stage of the Eight-Year Average composite sample.

Tests were carried out using a 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 iron 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%.

Figure 13-11:Cake Moisture vs Fitration Rate



Note: Figure from Informe Final Pruebas Planta Piloto Minera Santo Domingo (2015)

Tests were also performed for the primary (rougher) 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 iron grade in the rougher concentrate grade is slightly lower using sea water at 53.9% Fe w/w versus fresh water at 56.2% Fe w/w.

The rougher 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 iron grades achieved were acceptable varying between 64% and 68% and reflected the fineness of grind employed. The silica grades varied inversely with the iron 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 2011 Pre-feasibility Study stage, three different composite samples were tested to verify the mineralogy of the deposit:

- Eight-Year Average 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 66.1% for the Eight-Year Average composite, 64.1% for Hematite sample and 66.6% for Magnetite sample.

For the 2014 Feasibility Study 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:

- Five-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 LIMS feed grind size was determined to be a P80 of 40 µm. This was compared to the original 2011 Pre-feasibility Study 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 (Five-Year Average, Hematite sample and Magnetite sample) could potentially achieve the concentrate specification of less than 4.1% SiO_2 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 to lead to iron losses from ultra-fines, due to the fine particle size distribution of this material
- Commercial LIMS machines have been observed to have difficulties collecting ultra-fine iron.

The three ALS Ammtec final concentrate particle size distributions from the 2014 testwork were similar to those obtained from the Eight-Year Average sample evaluated by SGA in 2011. The Eight-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 2011 Pre-feasibility Study database consisted of a total of 211 samples from ALS Chemex, Studien-Gesellschaft für Eisenerz-Aufbereitung (SGA) and Compañía Minera del Pacifico (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. Wood conducted an analysis of the likely block model outputs and confirmed that the ANDE lithology represents between 13% and 17% of the LOM 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 silica and iron contents in the magnetic concentrates from these samples were 3.4% and 67.7% respectively. Both of these results are within the target specification.

Magnetic Separation Pilot Testing

In 2015, a pilot testing program was conducted using a Year 1–5 mill feed composite. This testing confirmed the process design criteria for magnetic separation and also confirmed the mass recovery and concentrate projections used in previous studies. Testing using desalinated water was not carried out because it is expected that there would be no variation in the performance of the LIMS circuit using this type of water. In the next phase, pilot plant testing will be conducted to confirm the iron recovery circuit final design criteria.

In the pilot plant, very little variation of results was observed during testwork of the magnetite circuit and the previous bench work. The fundamental basis of the process is physical, and the process proved to be very stable. Prior to the pilot plant testwork the biggest concern was iron liberation and grind size relationships. The process design criteria P80 grind size of 42 µm was shown to be adequate to produce the desired recovery, grade and contaminant levels for concentrate sales.

Magnetic rougher concentrate was produced during the copper rougher flotation testwork by treating the copper rougher tailings in a LIMS operating at 1,000 gauss, producing a rougher magnetic concentrate at 54% Fe and 21% mass recovery. Laboratory analysis confirmed complete magnetics recovery to the concentrate. Figure 13-12 shows the preparation method and flowsheet for the magnetic recovery circuit.

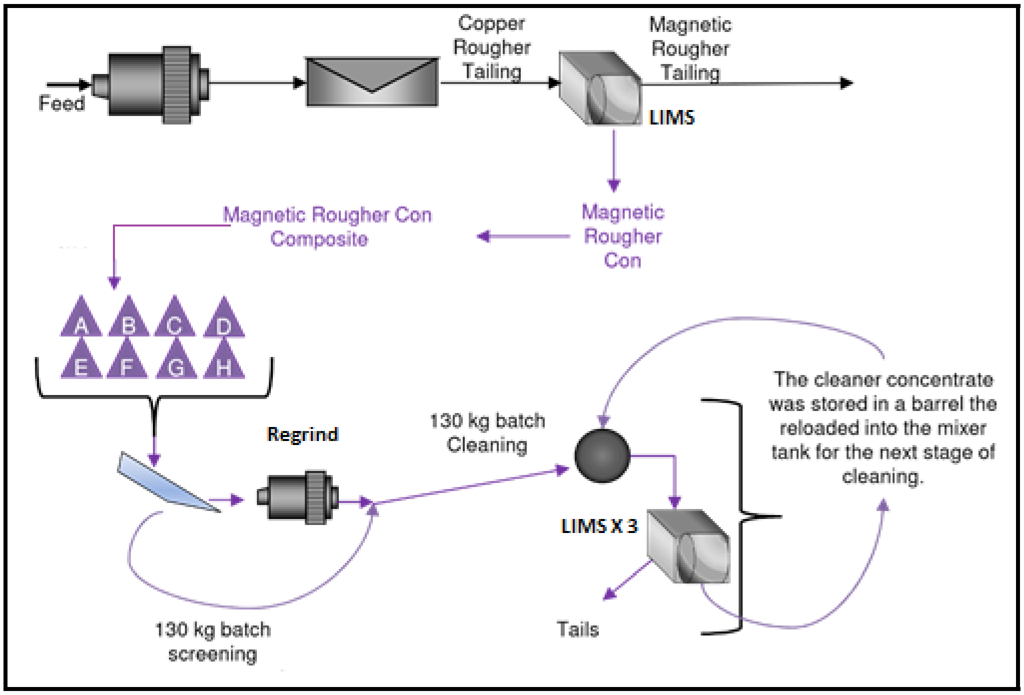
The pilot plant testwork was organized in two groups of four tests (A to D and E to H) with each test passing through three stages of cleaning. Table 13-3 summarizes the process conditions of the batch runs and LIMS stages for the tests A to D. The densities of these runs did not achieve the desired concentrate quality with respect to final grade or silica content. The target quality for the concentrate is grade greater than 65% Fe and less than 4.1% SiO_2.

The LIMS conditions were reviewed and an optimization plan was designed to improve the cleaner performance. A series of DT test enhancements were applied to the concentrates from tests A to D. The DT testwork indicated that silica contents of 3.5% and grades higher than 66% Fe were achievable without finer grinding (see Table 13-5).

The tests on samples E to H were conducted using the revised process conditions determined in the DT testwork (Table 13-6).

The results of the pilot plant runs using these condition for samples E to H are shown in Table 13-7.

Figure 13-12:Magnetic Separation Testwork



Note: Figure from Informe Final Pruebas Planta Piloto Minera Santo Domingo (2015)

Table 13-3: Initial LIMS Process Conditions

Sample		A	B	C	D
1st Cleaner Feed	L/min	8	8	8	8
	% Solids	14.8	16.3	12.9	11.1
2nd Cleaner Feed	L/min	8	5	8	8
	% Solids	21.3	22.1	28.6	19.2
3rd Cleaner Feed	L/min	8	5	8	8
	% Solids	22.3	17.3	26.7	22.8
Gap (mm)		6	6	6	6

Note: All trials performed at 750 gauss. Data prepared by Capstone.

Table 13-4: Result of Individual Batch Runs (Samples A to D)

	Sample	A	B	C	D
Assay (%)	Fe	64.26	64.54	63.75	64.33
	Si	4.8	4.6	5.1	4.7

Table 13-5: Davis Test Tube Test of Concentrate from Tests A to D

Sample	A	B	C	D
Fe %	66,22	66,56	66,73	66,46
Si %	3,48	3,21	3,14	3,33

Note: Data prepared by Capstone

Table 13-6: LIMS Process Conditions after Davis Tube Improvements

Sample		E	F	G	H
1st Cleaner Feed	L/min	8	8	6	6
	% Solids	27.8	32.8	26.1	30.4
2nd Cleaner Feed	L/min	2	2	4	4
	% Solids	17.9	11.3	17.4	14.5
3rd Cleaner Feed	L/min	2	2	4	4
	% Solids	20.6	13.5	20.9	23.8
Gap (mm)		7.5	7.5	7.5	7.5

Note: All trials performed at 750 gauss

Table 13-7: Optimized Pilot Plant Test Results (Samples E to H)

Sample		E	F	G	H
Assay (%)	Fe	65.1	66.4	68.5	68.7
	Si	4.6	3.9	4.1	4.1

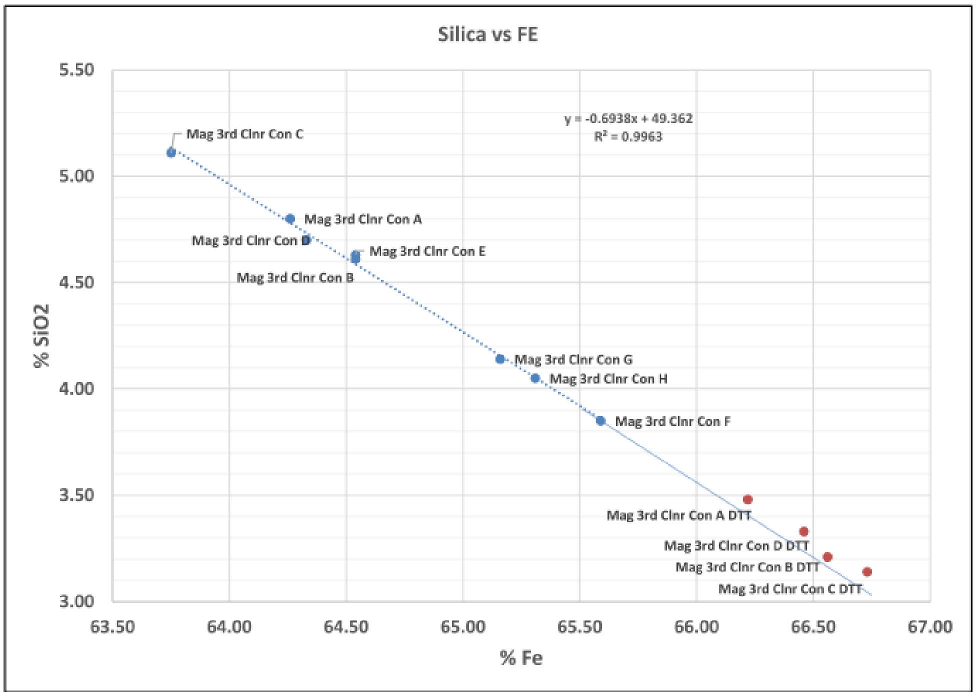
A clear relationship was demonstrated between managing iron grade and the level of silica contamination (Figure 13-13). All data was replotted to illustrate this relationship. Several conditions resulted in achieving the desired results. The DT results indicate that further optimization is possible during full scale operations.

The mass recovery relationship established during the pilot plant testwork confirms the criteria in the process design criteria. In the final stages of cleaning, the mass recovery should hold at approximately 15% in order to achieve an iron grade over 65%. The limits of the maximum grade recovery chart in Figure 13-14 also indicate that at iron grades in excess of 68%, recoveries will begin to significantly decline. All tests were conducted at a single value of magnetic susceptibility using the Five-Year Average Composite and variations experienced during operations could require changes in mass recovery and in LIMS setting to achieve optimum results.

Magnetite Concentrate Thickening

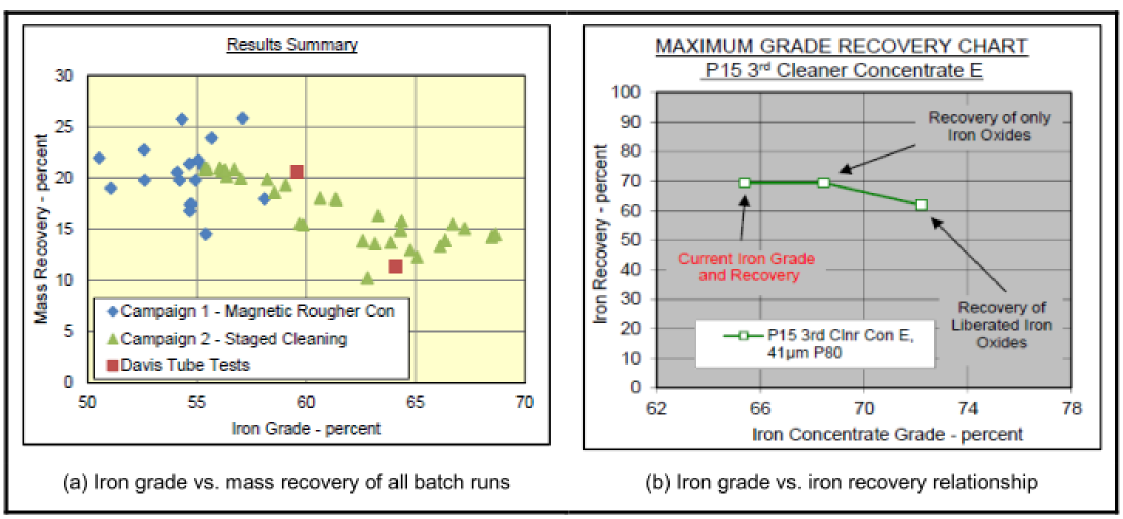
In 2014, Outotec undertook testing to determine the main characteristics of dynamic settling and thickening of the magnetite concentrate slurry using a bench-scale dynamic thickener. An Outotec 99 mm diameter laboratory thickener was used. Ten thickening tests using dilutions of 20%, 25%, 30%, 35% and 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.

Figure 13-13:Silica versus Iron Relationship from Magnetic Recovery Tests



Note: Figure from Informe Final Pruebas Planta Piloto Minera Santo Domingo (2015)

Figure 13-14:Iron Pilot Plant Grade Recovery Relationship



(a) Iron grade vs. mass recovery of all batch runs

(b) Iron grade vs. iron recovery relationship

Note: Figure from Informe Final Pruebas Planta Piloto Minera Santo Domingo (2015)

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 $t/hr/m^2$ and 0.6 $t/hr/m^2$, achieving a clarity of about 200 NTU and 74% solids w/w in the discharge using laboratory bench scale equipment.

Delkor also carried out testing in 2014. 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/hr and 9 m/hr 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.

Wood recommended that a unit value of 0.68 $t/hr/m^2$ be 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.

It is assumed that the process design criteria for concentrate thickening will not change when using desalinated water.

13.1.5 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 Cl.

The test results without cake washing indicated that filtration rates between 950 kg/m^2-hr and 807 kg/m^2-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/m^2-hr and 730 kg/m^2-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 m^3/t solids, 0.4 m^3/t solids and 0.6 m^3/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/m^2-hr, with a cake moisture content of 8.5%. Outotec was prepared to guarantee these results in an industrial application.

The 2015 pilot plant testwork used sea water and the current flowsheet at the time. Concentrate samples generated from the pilot plant and sent to FLSmidth and Outotec for iron concentrate filter evaluation using a plate and frame tower press. The objective of the testwork was to determine the size and operating parameters such as cake thickness, filtration rate, filter cake moisture and cake handling for the magnetite concentrate filters. Although the pilot tests were done with sea water, because of the low clay content of the material, the filtration tests should provide valid results to indicate the moisture content acheivable.

The samples sent to FLSmidth and Outotec (Figure 13-15) were final magnetite concentrate, with a P80 target of approximately 48 µm, slurry pH in the range of 8.8 to 9.2 and solids SG of 4.7: the desired final moisture content range was 8% to 9%. FLSmidth's filtration tests did reach the desired moisture targets. For Outotec, the cake thickness range tested was 25 mm to 65 mm and the residence times to achieve 9% moisture, varied from 30 seconds to 210 seconds.

Based on the projected use of desalinated water in the copper and iron recovery circuits the need for concentrate washing will be eliminated; the water balance was updated to reflect this. Following future pilot plant testing using desalinated water, the process design criteria for the filtration circuits will be updated to reflect the elimination of concentrate washing.

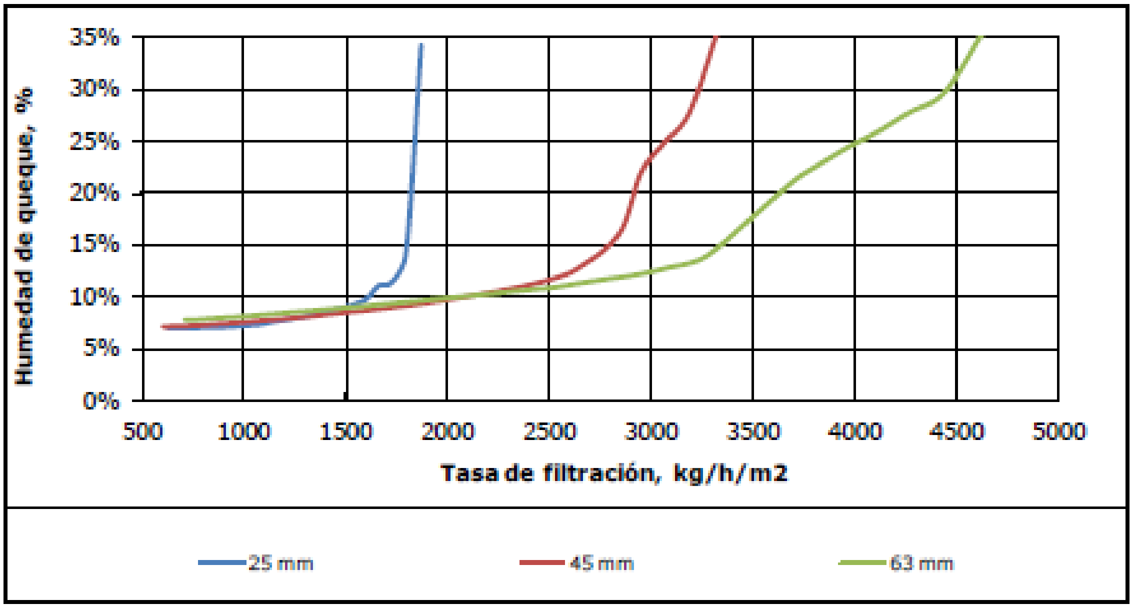
13.1.6 Tailings Thickening Testwork

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 tailings.

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.

Figure 13-15:Moisture Content vs Filtration Rate



Note: Figure from Informe Final Pruebas Planta Piloto Minera Santo Domingo (2015)

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

- First stage
  - Type of thickener: High rate
  - Thickening rate: 0.65 t/hr/m^2
  - Solid percent of thickened tailings: 55%
  - Flocculant dose: 10 g/t tailings.
- Second stage
  - Type of thickener: High density
  - Thickening rate: 0.5 t/hr/m^2
  - Solid percent of thickened tailings: 67%
  - Flocculant dose: 10 g/t tailings.

It is assumed that the use of desalinated water in the process plant will not materially change the process design criteria for the tailing handling circuit given the absence of swelling clays in the ore. This will be confirmed in the pilot plant testing to be carried out in the next phase.

13.1.7 Cobalt

A limited metallurgical testwork program was performed in 2018 on cobalt recovery, under the supervision of Blue Coast Metallurgy Ltd. (Blue Coast).

This work included:

- Production of a pyrite concentrate by batch scale flotation from the copper tails
- Mineralogical work to identify, locate and characterise the cobalt mineralisation using field emission automated scanning electron microscopy
- Preliminary (unoptimised) bench-scale total pressure oxidation, bioleaching, and selective pressure oxidation/leaching testwork.

The flotation work was done by Blue Coast in Parksville, British Columbia. The mineralogy work was executed by Surface Science Western in London, Ontario. The bioleaching testing was done by SGS in Lakefield, Ontario, and autoclave testwork by SGS Malaga, Western Australia.

The preliminary testwork, on a single bulk sample, indicated that conventional pyrite flotation from the copper circuit tails could recover about 80% of the cobalt as a secondary concentrate grading 0.4% Co, 0.6% Cu, and 29% S. Three potential processes were tested for extraction of these metals from the concentrate. Cobalt extraction to solution varied from 33–98% depending on methods and conditions. These initial results indicate that the extracted cobalt is amenable to leaching under acidic conditions. All three candidate processes would generate a similar pregnant leach solution containing dissolved cobalt and copper which, although not yet tested, are potentially amenable for recovery to the final end products in a purification process, namely crystallized cobalt sulfate heptahydrate and copper cathode.

13.2 Recovery Estimates

13.2.1 Copper

The 2018 Feasibility Study update used the copper feed grade to predict the recovery in the deposit using the results from Aminpro metallurgical testwork. The equation derived is:

*Global Cu Recovery = 0.98 * 96.9018*(Feed, % Cu)* $^{0.0199}$

The factors included in the model represent the following:

- 0.98 = copper cleaning recovery factor (it is planned to carry out studies with more samples to improve the confidence in this value)
- 96.9018 and 0.0199 = optimized constants of the potential equation.

The copper model is shown in Figure 13-16.

13.2.2 Gold

The 2018 Feasibility Study update used the copper feed grade to predict the gold recovery in the deposit using the results from Aminpro metallurgical testwork. The equation derived is:

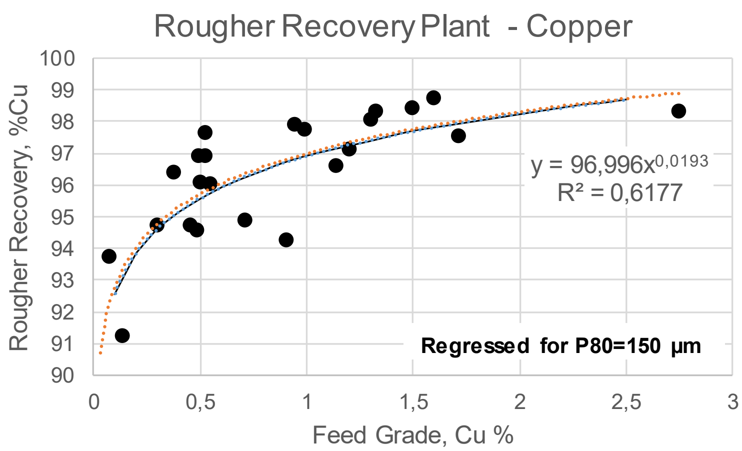
*Global Au Recovery = 0.85 * 82.646*(Feed, % Cu)* $^{0.1611}$

The factors included in the model represent the following:

- 085 = gold cleaning recovery factor it is planned to carry out studies with more samples to improve the confidence in this value)
- 82.646 and 0.1611 = optimized constants of the potential equation.

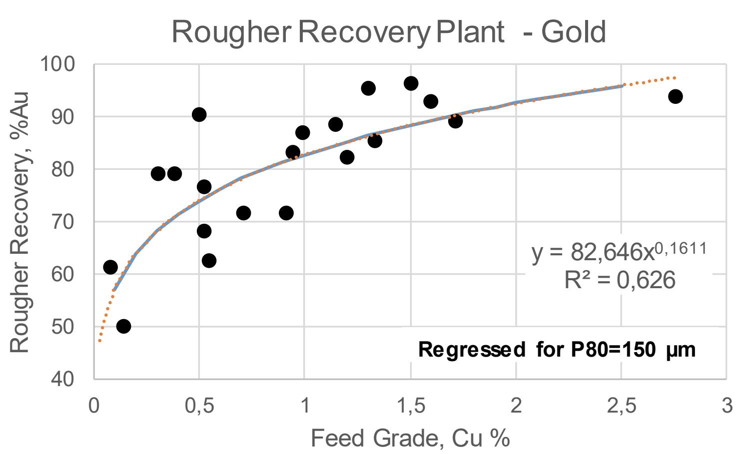
The gold model is shown in Figure 13-17.

Figure 13-16:Copper Model Recovery



Source: Aminpro, 2018

Figure 13-17:Gold Model Recovery



Source: Aminpro, 2018

13.2.3 Iron

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

$$\textit{Rec. Mas Fe} = \begin{cases} 0.0011 \times (MagSus) - 3E^{-09} \times (MagSus)^2; \text{ if } MagSus \geq 2{,}000 \\ 0; \text{ if } MagSus < 2{,}000 \end{cases}$$

Note: MagSus = magnetic susceptibility; Rec. Mas Fe = mass recovery to final magnetite concentrate

Figure 13-18 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.

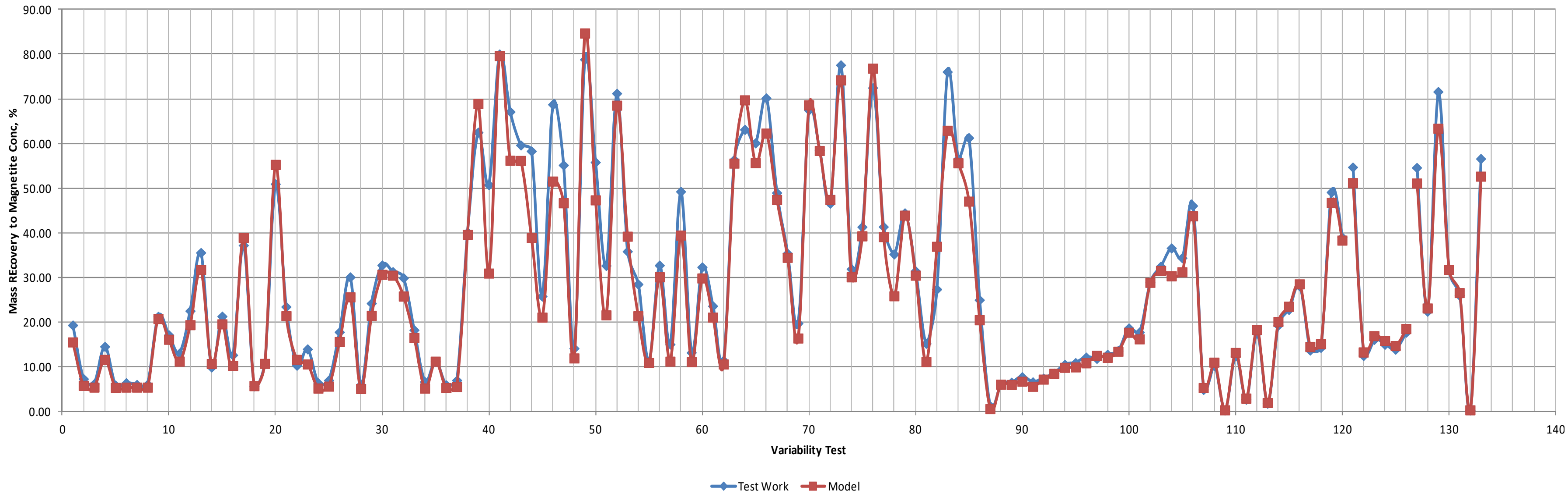
13.3 Metallurgical Variability

Metallurgical variability testwork is discussed in Section 13.1. Variability samples were selected during the 2011 Pre-feasibility Study 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.

Figure 13-18:Mass Recovery versus Results



Source: Aminpro, 2018

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 copper) 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 Five-Year Average copper composite indicated that arsenic values were low, the silica level is acceptable, and heavy minerals such as bismuth, antimony, and cadmium 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 2011 Pre-feasibility Study, with the values indicated in Table 13-8. The majority of concentrate samples produced from the DT 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.

From the DT 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.

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).

Table 13-8: Comparison of Final Concentrate Properties and Capstone Target Specification

			Five-Year Average Sample Final LIMS Concentrate	Current Capstone Target
	Mass Yield	%	23.4	—
Fe	Grade	(%)	66.0	> 65
SiO_2	Grade	(%)	4.56	4.10
Al_2O_3	Grade	(%)	1.00	1.00
TiO_2	Grade	(%)	0.17	-
Mn	Grade	(%)	0.07	0.07
CaO	Grade	(%)	0.69	0.57
P	Grade	(%)	0.01	0.01
S	Grade	(%)	0.01	0.02
MgO	Grade	(%)	0.48	0.46
K_2O	Grade	(%)	0.12	0.11
Na_2O	Grade	(%)	0.14	0.15
Zn	Grade	(%)	0.004	—
LOI (1,000)	Grade	(%)	-1.1600	—
As	Grade	(%)	0.0010	—
Ba	Grade	(%)	0.0010	—
Cl	Grade	(%)	0.0060	0.0060
Co	Grade	(%)	0.0060	—
Cr_2O_3	Grade	(%)	0.0900	—
Cu	Grade	(%)	0.0090	0.0081
Ni	Grade	(%)	0.0340	—
Pb	Grade	(%)	0.0050	—
Sn	Grade	(%)	0.0005	—
Sr	Grade	(%)	0.0010	—
V	Grade	(%)	0.0080	—
Zr	Grade	(%)	0.0040	—
FeO	Grade	(%)	27.1	23.1

13.5 Comments on Section 13

Metallurgical testwork completed during the 2011 Pre-feasibility Study, the 2014 Feasibility Study and during 2018 for the 2018 Feasibility Study update includes physical characterization; conventional sulphide flotation using fresh water, sea water, and desalinated 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 in 2014 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.

The testwork completed during 2018 by Aminpro consisted of rougher and cleaner flotation tests with the composite for the first five years of operation using desalinated water in order to generate process modelling parameters and to examine the grade of concentrate and the recoveries of copper and gold that could be produced. With head grade variability samples, feed grades were tested between 0.09% Cu and 2.76% Cu, and recovery algorithms were determined for copper and gold. The cleaner recoveries for copper and gold were later confirmed using LCT conducted on a composite of the variability samples using desalinated water. The LCT gave a copper global recovery of 92.4% with a concentrate grade of 32.8% Cu.

Tests were performed with desalinated water to determine the new recovery algorithm. It is recommended that tests be carried out with fresh samples to confirm the current design criteria for the plant.

Conceptual capital and operating cost estimates were prepared for cobalt recovery using a BIOX process, a pressure oxidation process, and a dead roasting process by AGP Mining Consultants in Barrie, Ontario. Each of these conceptual estimates indicated that there is a reasonable prospect of eventual economic production of cobalt as a by-product from the proposed Project process flowsheet. Capstone has authorized further studies to define the optimum flow sheet for cobalt processing. Additional testwork is also planned to determine the quality of the cobalt sulfate and assess the impact of any impurities that might affect marketing. There is upside potential for the Project if this work supports cobalt estimation as part of an updated Mineral Reserve estimate, such that cobalt as a by-product can be incorporated into a future financial analysis.

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 copper, gold, or iron; and 30,918 had non-zero values for cobalt. 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 copper, gold, and iron intervals and 16,140 cobalt 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 298 holes. Of these, 115 are core drill holes or holes collared as RC and then finished as core holes. Thirty 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. In 2018, RPA populated the existing block model with cobalt grades. The Mineral Resource estimate is based 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 copper assays, 3,595 gold assays, and 4,702 cobalt assays with non-zero values.

14.2 Geological Models

14.2.1 Wireframes (Santo Domingo Sur, Iris, and Iris Norte)

RPA constructed three-dimensional (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 copper 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 sub-divided into magnetite-rich and magnetite-poor variants.

RPA examined the distribution of cobalt and copper mineralization in the entire Santo Domingo area to determine if there is a good correlation between the two metals. Although, cobalt is associated with pyrite mineralization, the overall spatial distribution is similar to copper. RPA considers the current copper domains to be suitable for interpolation of cobalt grades. RPA is of the opinion that future models might benefit from a separate set of cobalt domains to better define higher grades.

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 personnel provided raw cavity monitoring device (CMD) data from which RPA was able to construct approximate wireframe models of the void spaces.

As for Santo Domingo, no separate wireframes were generated to outline the cobalt mineralization at Estrellita.

14.2.3 Oxide Model

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. 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 to 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 iron 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, gold, and cobalt 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, 0.52 g/t Au, and 1,750 ppm Co, indicating distinct populations for each metal. The data review with the latest drilling indicated the inflection points were still valid.

In total, 24 copper and 27 gold assay intervals were capped for the 2012 estimate. These intervals represent approximately 0.2% of the total number of assays. For the 2018 cobalt estimate, 27 cobalt assay intervals were capped representing approximately 0.4% of the total number of assays.

In RPA's opinion, the net impact of the capping was to reduce the average copper, gold and cobalt assay grades by a negligible amount (see Table 14-1 to Table 14-3).

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, 0.3 g/t Au, and 1,000 ppm Co. Grade cap data are included in Table 14-1 to Table 14-3.

Table 14-1: Assay Capping Levels – Copper

Zone	Cap Grade (% Cu)	# Std. Dev. from Mean	Population Maximum Grade (% Cu)	# Samples Capped	Avg. Cu Grade (% Cu) Before Capping	Avg. Cu Grade (% Cu) After Capping
SDS (1–4)	3.5	5.6	6.38	32	0.486	0.485
Iris (5–6)	3.5	10.2	3.34	0	0.192	0.192
Iris Norte (7–8)	3.5	10.5	3.10	0	0.173	0.173
Totals (1-8)	***3.5***	***6.3***	***6.38***	***32***	***0.380***	***0.378***
Estrellita	3.0	4.8	8.79	34	0.375	0.366

Note: Includes "below detection" as 0.0. SDS = Santo Domingo Sur.

Table 14-2: Assay Capping Levels – Gold

Zone	Cap Grade (g/t Au)	# Std. Dev. from Mean	Population Maximum Grade (g/t Au)	# Samples Capped	Avg. Au Grade (g/t Au) Before Capping	Avg. Au Grade (g/t Au) After Capping
SDS (1–4)	0.52	5.9	2.38	23	0.066	0.065
Iris (5–6)	0.52	5.4	4.71	7	0.029	0.027
Iris Norte (7–8)	0.52	10.4	0.98	1	0.022	0.022
Totals (1–8)	***0.52***	***5.8***	***4.71***	***31***	***0.052***	***0.051***
Estrellita	0.30	4.3	0.979	26	0.049	0.048

Note: Includes "below detection" as 0.0. SDS = Santo Domingo Sur.

Table 14-3: Assay Capping Levels – Cobalt

Zone	Cap Grade (ppm Co)	# Std. Dev. from Mean	Population Maximum Grade (ppm Co)	# Samples Capped	Avg. Co Grade (ppm Co) Before Capping	Avg. Co Grade (ppm Co) After Capping
SDS (1–4)	1,750	255	4,820	23	269	268
Iris (5–6)	1,750	203	1,990	1	189	189
Iris Norte (7–8)	1,750	222	2170	3	212	212
Totals (1–8)	***1,750***	***242***	***4,820***	***27***	***243***	***242***
Estrellita	1,000	157	3,290	19	133	132

Note: Includes "below detection" as 0.0. SDS = Santo Domingo Sur.

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 gold, copper, cobalt, 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 Santo Domingo Sur/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 14-4).

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.

Cobalt is a new metal to be incorporated into the block model, hence, RPA completed a variography analysis with Supervisor 8.8.0.1 software using 4 m composites. The variograms were modelled using two spherical structures and a nugget effect of between 15% and 20% of the sill. A summary of the variogram parameters is shown in Table 14-5. Lack of sufficient data for the Iris Norte area did not allow valid variogram calculation using composites for this domain. Thus, variogram parameters established using entire cobalt data were applied in the estimation run at Iris Norte.

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 copper 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. No variography was carried out for cobalt composites.

Table 14-4: Variogram Models

Metal/Zone	Model Type	Nugget	C	Total Sill	Orientation (Az/Plunge)			Range (m)		
					Major	Semi	Minor	Major	Semi	Minor
Copper										
SDS (1–4)	Exp	0.114	0.886	1.000	118/13	030/-08	329/74	276.3	152.6	42.1
Iris/Iris Mag	Sph	0.064	0.936	1.000	168/-02	078/00	168/88	151.7	48.7	34.7
Iris Norte	Sph	0.228	0.772	1.000	079/00	169/00	079/90	72.3	72.3	25.6
Iron										
SDS (1–4)	Exp	0.021	0.979	1.000	130/15	042/-07	334/73	119.2	65.2	35.9
Iris/Iris Mag	Sph	0.228	0.772	1.000	191/00	101/00	191/90	146.0	75.6	42.5
Iris Norte	Sph	0.364	0.636	1.000	136/-04	048/34	219/56	100.6	100.6	52.7
Gold										
SDS (1–4)	Sph	0.188	0.812	1.000	073/-08	342/-07	031/80	185.8	101.2	39.4
Iris/Iris Mag	Sph	0.031	0.969	1.000	146/-11	056/00	146/79	192.0	105.8	26.2
Iris Norte	Sph	0.400	0.600	1.000	119/06	210/00	299/84	135.0	135.0	20.0
Mag Sus										
SDS Mag	Exp	0.125	0.875	1.000	010/-10	096/19	307/68	128.6	69.1	34.3
SDS Non-Mag	Exp	0.029	0.971	1.000	016/-15	104/07	349/73	119.3	86.4	56.9
Iris/Iris Mag	Sph	0.131	0.869	1.000	190/00	101/34	280/56	171.7	171.7	55.0
Iris Norte	Sph	0.029	0.971	1.000	049/33	139/00	048/-57	102.4	102.4	16.7

Note: SDS = Santo Domingo Sur.

Table 14-5: Variogram Models

Zone	Nugget	Model Type	C1	Range 1			C2	Range 2			Total Sill	Orientation (Gems)		
				Major	Semi	Minor		Major	Semi	Minor		Z	X	Z
SDS (1–4)	0.20	Sph	0.50	92	40	12	0.30	375	300	145	1.000	-140	25	25
Iris/Iris Mag	0.15	Sph	0.25	450	70	11	0.60	455	190	75	1.000	85	-40	30
Iris Norte	0.20	Sph	0.50	40	90	12	0.30	300	375	145	1.000	119	6	30

Note: SDS = Santo Domingo Sur.

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. 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 copper, gold, iron, and cobalt and magnetic susceptibility were interpolated into each block using ordinary kriging (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 copper, gold, iron and cobalt estimates.

In 2012, the interpolation for copper, gold, iron, and magnetic susceptibility 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.

Following sensitivity analysis of the interpolation parameters, RPA chose to estimate cobalt grades in 2018 using a different set of search criteria than used previously for other metals. The primary grade interpolation used the OK estimator but was based on a minimum of four and a maximum of 12 composites in the first pass and a minimum of three and a maximum of 12 composites in the second pass. The first estimation run required composites from at least two drill holes to interpolate block grades. The second estimation pass limited the number of composites from a drill hole to a maximum of three composites. Search ellipse dimensions were chosen following a review of variography and interpolation efficiency. The search ranges for the first pass were set to 100 m, 70 m, and 20 m for the major, semi-major, and minor axes respectively. The second pass search ranges for the major and semi-major axes were adjusted to approximately 95% of the variogram sill at 240 m and 180 m respectively. The minor axis search in the second pass remained the same (20 m). The influence of higher-grade cobalt composites was spatially restricted to the search ranges of the first estimation run at 900 ppm Co.

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 copper, gold, and iron grades into each block. Cobalt was estimated using inverse distance weighting to the third power (ID^3) interpolation. Only composites with zone codes that matched the block codes were used in grade estimates. The search was constrained to a minimum of two and maximum of 12 composites, with a maximum of three composites from any one drill hole. The influence of higher-grade cobalt composites was spatially restricted to the half of the search ranges of the first estimation run at 750 ppm Co.

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:

$$SG = 2.72 + 0.0018*Fe + 0.0006*Fe^2$$

14.7 Block Model Validation

14.7.1 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.

14.7.2 Estrellita

Validation was performed on the grade estimates using ID^3. RPA concluded that while the OK results were very close to the actual composite means, the lower standard deviations indicated that OK was unable to model the extremes as well. To validate the cobalt grade interpolation, RPA generated swath plots comparing cobalt estimated grades with informing data.

14.8 Classification of Mineral Resources

14.8.1 Santo Domingo Sur, Iris, and Iris Norte

Blocks receiving an estimate for copper 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 copper at Estrellita were 106 m and 93 m, respectively. The drill spacing is approximately 50 m, and well within the two-thirds range limit of the copper 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

In RPA’s opinion the Mineral Resources for the Santo Domingo and Estrellita deposits are considered to have reasonable prospects of eventual economic extraction by open pit mining.

To fulfill the CIM requirement of “reasonable prospects for eventual economic extraction”, RPA prepared a preliminary open pit shell for each deposit to constrain the block model for resource reporting purposes.

Table 14-6 and Table 14-7 list the parameters used for cut-off grade calculation and pit optimization for Santo Domingo and Estrellita, respectively.

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; 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 the CuEq calculation was developed by RPA. The 2018 CuEq grades were calculated using estimates for recovery and treatment/refinement charges (TC/RCs) presented in Table 14-6.

Table 14-6: LG Optimization Parameters – Santo Domingo

Parameter	Value
Metal price	
Copper	3.50/lb
Gold	$1,300/oz
Iron	$99.00/dmt concentrate
Recovery to concentrate	
Copper	89%
Gold	79%
Iron recovery	Calculated (see text)
Mass recovery for magnetite	Rec. Mass Fe (see Section 13.2.3)
Copper concentrate grade	
Copper	29%
Gold	Calculated
Moisture content	8%
Magnetite concentrate grade	
Iron	67%
Moisture content	8%
Smelter payables	
Payable copper	96.5%
Payable gold	97.0%
Off-site costs	
Copper con. transport	$33.00/wmt
Magnetite con. transport	$20.00/wmt
Copper treatment charge	$80.00/dmt
Copper refining charge	0.08/lb
Gold refining charge	5.00/oz
Royalty	5% NSR
On-site operating costs	
Mining cost	$1.90/t mined
Processing and G&A cost	$7.27/t ore processed

Parameter	Value
Pit wall slope	
Overburden	38˚
Rock (south, west, north wall)	44˚
Rock (east wall)	40˚

Table 14-7: LG Optimization Parameters – Estrellita

Parameter	Value
Metal price	
Copper	$3.50/lb
Gold	$1,300/oz
Recovery to concentrate	
Copper	89%
Gold	79%
Copper concentrate grade	
Copper	29%
Gold	Calculated
Moisture content	8%
Smelter payables	
Payable copper	96.5%
Payable gold	97.0%
Off-site costs	
Copper con. transport	$33.00/wmt
Copper treatment charge	$80.00/dmt
Copper refining charge	$0.08/lb
Gold refining charge	$5.00/oz
Royalty	5% NSR
On-site operating costs	
Mining cost	$1.90/t mined
Processing and G&A cost	$7.27/t ore processed
Pit wall slope	45˚

The metal prices used in Table 14-6 and Table 14-7were current at the effective date of the estimate, which was 31 October 2018. Typically, the metal prices for RPA's resource estimations are based on consensus (average) price forecasts by banks and financial institutions. RPA considers long-term average price forecasts to be appropriate for use in estimating Mineral Reserves, and slightly higher prices to be appropriate for estimating Mineral Resources.

Iron ore pricing is quite variable from project to project due to quality and transport costs; thus, forecast pricing based on a marketing study specific to the project is preferred. RPA used an iron ore price of $99/t concentrate (67% Fe content) based on market studies conducted by Capstone.

The $1,300/oz Au price and $3.50/lb Cu price assumptions were based on consensus price forecasts and prices used by major companies at the time the estimate was undertaken.

Only copper, gold and iron were recognized in the CuEq calculation; cobalt was excluded until further studies are completed to confirm reasonable prospects for eventual economic extraction.

In RPA's opinion, based on the costs developed using the parameters detailed in Table 14-6, the cut-off grade to report Mineral Resources for Santo Domingo Sur, Iris, and Iris Norte is 0.125% CuEq.

14.10.2 Estrellita

In RPA's opinion a 0.125% CuEq cut-off grade is also appropriate for reporting of the Estrellita Mineral Resource estimate. The style, geometry and proximity to surface of Estrellita are similar to the Santo Domingo Sur/Iris deposits. The iron content is significantly lower at Estrellita; hence, iron has been excluded from the copper equivalence calculation.

14.11 Mineral Resource Statement

The Mineral Resource estimates and geological models were prepared under the supervision of David Rennie, P. Eng., Associate Principal Geologist for RPA. Mr. Rennie is the Qualified Person as defined under NI 43-101 for the estimate. Mineral Resources for both the Santo Domingo and Estrellita deposits have an effective date of 31 October 2018. Mineral Resources in Table 14-8 are reported inclusive of Mineral Reserves. RPA cautions that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

14.12 Changes from the Previous Mineral Resource Estimate

Table 14-9 compares the current estimate with an effective date of October 31, 2018 and to the previous one with an effective date of August 31, 2012 for the Santo Domingo Sur, Iris, and Iris Norte deposits and October 30, 2007 for the Estrellita deposit. Both the current and previous estimates in Table 14-9 were completed by RPA Qualified Person David Rennie and the resource estimation procedures and the resource block model values have not changed, except for the addition of the cobalt models, which do not generate any by-product revenue. RPA's previous resource estimates were disclosed in a Technical Report dated May 22, 2014 (AMEC, 2014).

The tonnage for the Measured category is similar, but the tonnages for Indicated categories have increased modestly, and Inferred tonnes have decreased. Current grades are similar to slightly lower; however, the CuEq grade has decreased significantly due to a change in the input assumptions and the calculation method, which include more information than the previous estimate.

In RPA's opinion, the principal reasons for the changes are the addition of a pit shell constraint to the estimates and the change in the CuEq calculation. In the previous estimate, no pit shell constraints were applied to the resource estimate. Application of the pit shells tends to reduce tonnes, especially in the Inferred category, because the Inferred resources tend to be located in the lower portions of the deposit.

Table 14-8: Mineral Resource Estimate

Deposit (Zone)	Tonnes (Mt)	CuEq (%)	Cu (%)	Au (g/t)	Fe (%)	Co (ppm)
Measured						
Santo Domingo Sur (1–4)	64	0.82	0.62	0.082	31.1	254
Iris (5–6)	2	0.42	0.39	0.047	23.6	250
Total Measured	***66***	***0.81***	***0.61***	***0.081***	***30.9***	***254***
Indicated						
Santo Domingo Sur (1–4)	224	0.54	0.31	0.043	26.6	275
Iris (5–6)	103	0.44	0.19	0.027	25.9	166
Iris Norte (7–8)	89	0.44	0.12	0.014	26.7	230
Subtotal Indicated (Santo Domingo Sur /Iris)	*416*	*0.49*	*0.24*	*0.033*	*26.4*	*238*
Estrellita	55	0.40	0.38	0.039	13.7	125
Total Indicated	***471***	***0.48***	***0.26***	***0.034***	***25.0***	***225***
Total Measured and Indicated	***537***	***0.52***	***0.30***	***0.039***	***25.7***	***229***
Inferred						
Santo Domingo Sur (1–4)	24	0.40	0.22	0.033	22.8	195
Iris (5–6)	4	0.42	0.18	0.024	26.6	125
Iris Norte (7–8)	14	045	0.09	0.009	28.1	256
Subtotal Inferred (Santo Domingo Sur /Iris)	*43*	*0.42*	*0.17*	*0.024*	*25.0*	*208*
Estrellita	5	0.32	0.31	0.030	12.3	108
Total Inferred	***48***	***0.41***	***0.19***	***0.025***	***23.6***	***197***

Notes to Accompany Mineral Resource Estimate:

1. Mineral Resources are classified according to CIM (2014) guidelines.
2. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. The Qualified Person for the estimates is Mr. David Rennie, P. Eng., an associate of Roscoe Postle Associates Inc.
4. Mineral Resources for the Santo Domingo Sur, Iris, Iris Norte, and Estrellita deposits have an effective date of 31 October 2018.
5. Mineral Resources for the Santo Domingo Sur, Iris, Iris Norte, and Estrellita deposits are reported using a cut-off grade of 0.125% CuEq. CuEq grades are calculated using average long-term prices of US$3.50/lb Cu, US$1,300/oz Au, and US$99/dmt Fe conc. The CuEq values were calculated as noted in the text in this report.
6. Only copper, gold, and iron were recognized in the CuEq calculation; cobalt was excluded until further studies are completed to confirm reasonable prospects for eventual economic extraction.
7. Mineral Resources are constrained by preliminary pit shells derived using a Lerchs–Grossmann algorithm and the following assumptions: pit slopes averaging 45º; mining cost of US$1.90/t, processing cost of US$7.27/t (including G&A cost); processing recovery of 89% copper and 79% gold; and metal prices of US$3.50/lb Cu, US$1,300/oz Au and US$99/dmt iron concentrate.
8. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade, and contained metal content.
9. Tonnage measurements are in metric units. Copper and iron are reported as percentages, gold as grams per tonne.

Table 14-9: Changes in the Mineral Resource Estimates

Category	Mt	%CuEq	%Cu	g/t Au	%Fe	ppm Co
Previous Resource Estimate (Santo Domingo Sur, Iris, and Iris Norte deposits as of August 31, 2012 and the Estrellita deposit as of October 30, 2007)						
Measured	65	0.94	0.62	0.082	31.2	n/a
Indicated	449	n/a	0.27	0.034	25.0	n/a
Inferred	58	n/a	0.20	0.026	24.3	n/a
Current Resource Estimate (October 31, 2018)						
Measured	66	0.81	0.61	0.081	30.9	254
Indicated	471	0.48	0.26	0.034	25.0	225
Inferred	48	0.41	0.19	0.025	23.6	197
Percent Difference						
Measured	1.9%	-13.8%	-1.6%	-1.2%	-1.0%	n/a
Indicated	4.9%	n/a	-3.7%	0.0%	0.0%	n/a
Inferred	-17.6%	n/a	-5.0%	-3.8%	-2.9%	n/a

In 2014 the metal prices used were \$3.50/lb Cu, \$1,500/oz Au, and \$120/dmt Fe concentrate. For this 2018 Mineral Resource estimate, the copper price was kept the same but the gold price was reduced to \$1,300/oz Au and the iron price was reduced to \$99/dmt Fe concentrate. This would reduce the block CuEq values and result in fewer blocks reporting above the cut-off grade, thereby reducing the tonnage. The cut-off grade was reduced from 0.250% CuEq used in 2014 to the calculated value of 0.125% CuEq for 2018, and this would increase the tonnage, while reducing average grades. This has resulted in an increase in both Measured and Indicated Mineral Resources.

14.13 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. These 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.

There are no known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors other than as discussed in this Report that could affect the Mineral Resource estimates.

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 2018. 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 third quarter 2018 prices and the exchange rate from Chilean Pesos to US dollars. This resulted in an estimated mining cost of approximately $1.75/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 Lerchs–Grossmann (LG) pit shell is included in Table 15-1.

Table 15-1: LG Optimization Parameters

Item	Units	Value
Metal Price		
Copper	$/lb	3.00
Gold	$/oz	1,290
Iron ($100/dmt CFR China less $20/dmt shipping = $80/dmt FOB Chile)	$/dmt	100
Recovery to concentrate		
Copper	%	$0.98*96.9018*Cu^{0.0199}$
Gold	%	$0.85*82.646*Cu^{0.1611}$
Mass recovery for magnetite concentrate	%	Variable on a block by block basis
Cu Concentrate Grade		
Copper	%	29%
Gold	g/t	Calculated
Moisture content	%	8%
Magnetite Concentrate Grade		
Iron	%Fe	66%
Moisture content	%	8%
Smelter Payables		
Copper in Cu conc.	%	100%
Payable copper	%	96.50%
Gold in all conc.	%	97%
Gold deduction in all concentrate	g/t in concentrate	0
Off-Site Costs		
Cu conc. treatment	$/dmt conc.	80
Cu refining charge	$/lb pay Cu	0.08
Au refining charge	$/oz pay Au	5.0
Shipping copper concentrate	$/wmt concentrate Cu	33
Shipping magnetite concentrate	$/wmt concentrate Fe	20
Operating Cost		
Waste mining cost	$/waste tonne	1.75
Ore mining cost	$/ore tonne	1.75

Item	Units	Value
Processing + G&A	$/t proc	7.53
Average Overall Pit Slope Angle		
	Overburden	37.6˚
	Sector 1 South	43.6˚
SDS/Iris & Iris Norte	Sector 2 West	43.6˚
	Sector 3 North	43.6˚
	Sector 4 East	40.2˚
Other		
Grade factor (1-Dilution)	%	100
Mining recovery	%	100
Royalties	%	2
Discount rate	%	8

Note: FOB = free on board. SDS = Santo Domingo Sur.

A number of calculations were performed in the model in order to determine the NSR of each individual block. The internal (or mill) cut-off of $7.53/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.

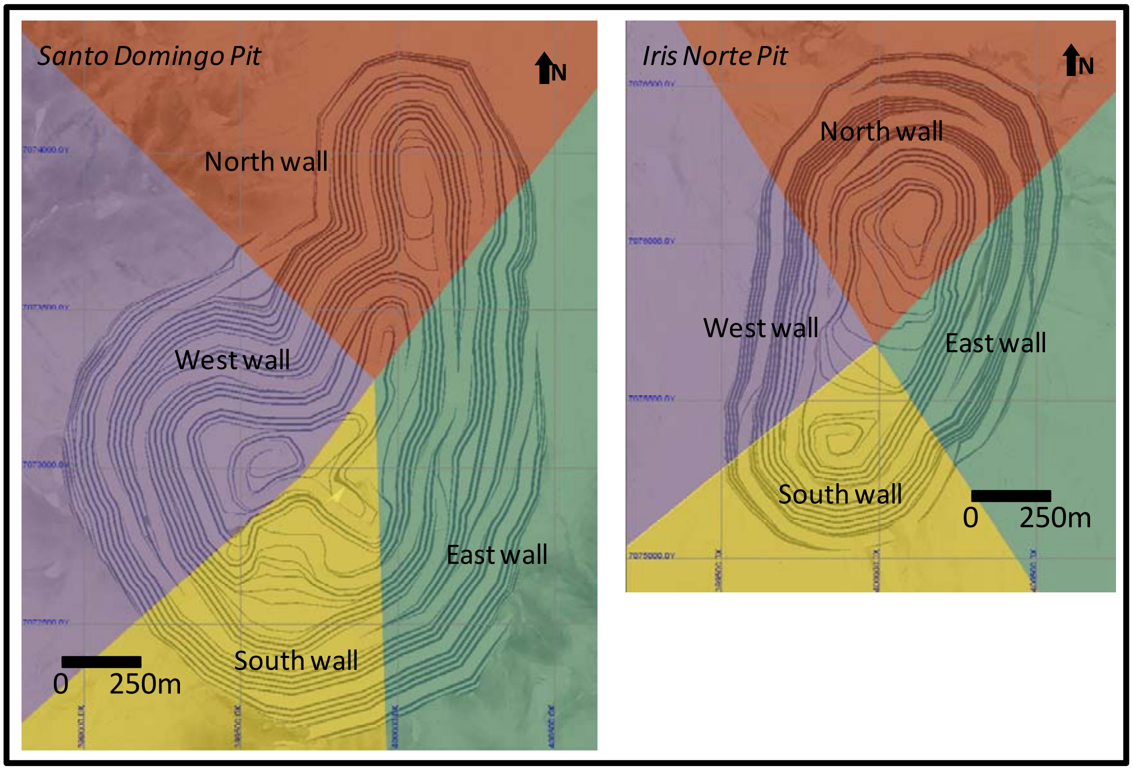
15.2.2 Dilution and Mine Losses

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 m^3 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).

Figure 15-1: Geotechnical Slope Domains



Note: Figure prepared by NCL, 2014.

Table 15-2: Slope Domain Data

Sector	Inter-Ramp Slope						Overall Slope	
	IRA (º)	Face Angle (º)	Height H (m)	Backbreak a (m)	Berm b (m)	Catch Berm c (m)	Slope Height L (m)	Slope Angle (º)
Overburden	44	55	12	8.4	4.0	40	150	38
West	52	75	24	6.4	12.4	40	150	44
East	52	70	24	8.7	10.1	40	100	40
North and South	52	75	24	6.4	12.4	40	150	44

- 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)/SGdil*

- 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 an NSR in $/t was calculated to take into account the value of copper, gold, and iron; and the off-site costs (transport, smelting, and refining).

The internal (or mill) cut-off of $7.53/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 $7.53/t cut-off, but for an NSR determined at higher metal prices than shown in Table 15-1 ($3.50/lb Cu, $102.63/t magnetite concentrate, and $1,500/oz Au were used).

15.3 Mineral Reserves Statement

Mineral Reserves are summarized in Table 15-3 and have an effective date of 14 November 2018. 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 0.84 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

Reserve Category	Stage	Ore (Mt)	Ore Grade Cu (%)	Ore Grade Au (g/t)	Ore Grade Fe (%)	Contained Metal Au (kOz)	Contained Metal Cu (Mlbs)	Magnetite Conc. (Mt)
Proven Mineral Reserves								
	Santo Domingo	65.4	0.61	0.08	30.9	169.9	878.5	8.2
	Iris Norte	—	—	—	—	—	—	—
Total Proven Mineral Reserves		*65.4*	*0.61*	*0.08*	*30.9*	*169.9*	*878.3*	*8.2*
Probable Mineral Reserves								
	Santo Domingo	252.1	0.27	0.04	27.8	300.8	1,486.1	48.2
	Iris Norte	74.8	0.13	0.01	26.9	36.0	208.1	18.7
Total Probable Mineral Reserves		*326.9*	*0.24*	*0.03*	*27.6*	*336.8*	*1,694.2*	*66.9*
Total Mineral Reserves (Proven and Probable)								
	Santo Domingo	317.5	0.34	0.05	28.5	470.7	2,364.6	56.4
	Iris Norte	74.8	0.13	0.01	26.9	36.0	208.1	18.7
Total Mineral Reserves (Proven and Probable)		***392.3***	***0.30***	***0.04***	***28.2***	***506.7***	***2,572.7***	***75.1***

Notes to Accompany Mineral Reserves Estimate:

5. Mineral Reserves have an effective date of 14 November 2018 and were prepared by Mr. Carlos Guzman, CMC, an employee of NCL.
6. 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$3.00/lb Cu, US$1,290/oz Au, and US$100/dmt of Fe concentrate; average recovery to concentrate is 93.4% for Cu and 60.1% for Au, with magnetite concentrate recovery varying on a block-by-block basis; copper concentrate treatment charges of US$80/dmt, U$0.08/lb of copper refining charges, US$5.0/oz of gold refining charges, US$33/wmt and US$20/wmt for shipping copper and iron concentrates respectively; waste mining cost of $1.75/t, mining cost of US$1.75/t ore, and process and G&A costs of US$7.53/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.
7. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade, and contained metal content.
8. Tonnage measurements are in metric units. Copper and iron grades are reported as percentages, gold as grams per tonne. Contained gold ounces are reported as troy ounces, contained copper as million pounds and contained iron as metric million tonnes.

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 #36 is the revenue factor 1 shell for Santo Domingo and Iris Norte. However, after analyzing the results (and reviewing total, as well as incremental values), the Santo Domingo and Iris Norte optimized pit shell #28 (revenue factor 0.84) was chosen as the basis for the detailed ultimate pit design. The difference between pit shell #28 and #36 is an expansion to the south of the Santo Domingo pit, including 70 Mt of low-grade material with a strip ratio of 6.7; considered to be high-risk material. This expansion would also compromise the waste rock storage facilities to the south.

The final pit design was based on the economic shells obtained at revenue factor 0.84 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.

Table 16-1: Mine Design Parameters

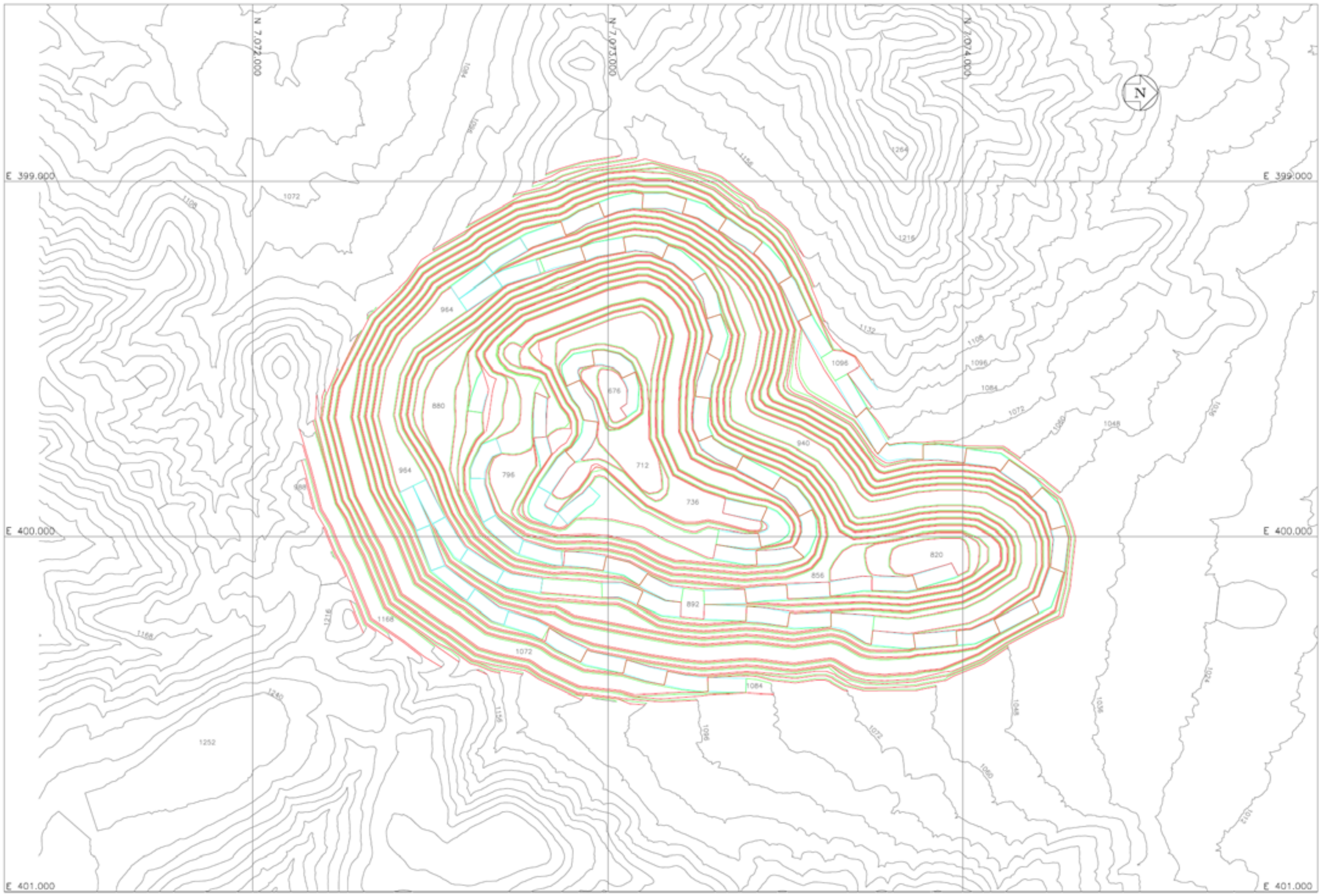
Parameter	Unit	Value
Haul road width	m	40
Haul road grade	%	10
Bench height	m	12
Stacked bench height with 2 benches stacked (fresh rock)	#	24
Nominal minimum mining phase width	m	100
Batter angle	°	As per geotechnical domains
Berm width	m	
Security berm width every 100 m/150 m of pit wall	m	40

Geotechnical Domains	Batter Height (m)	Batter Angle (°)	Berm Width (m)
Overburden (all pit walls)	12	55	4.0
West wall	24	75	12.4
East wall	24	70	10.1
North and south wall	24	75	12.4

The Santo Domingo pit will have two exits on the west side to provide access to the run-of-mine (ROM) pad area and the primary crusher. On the east side there will be another exit to access the main waste rock 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.

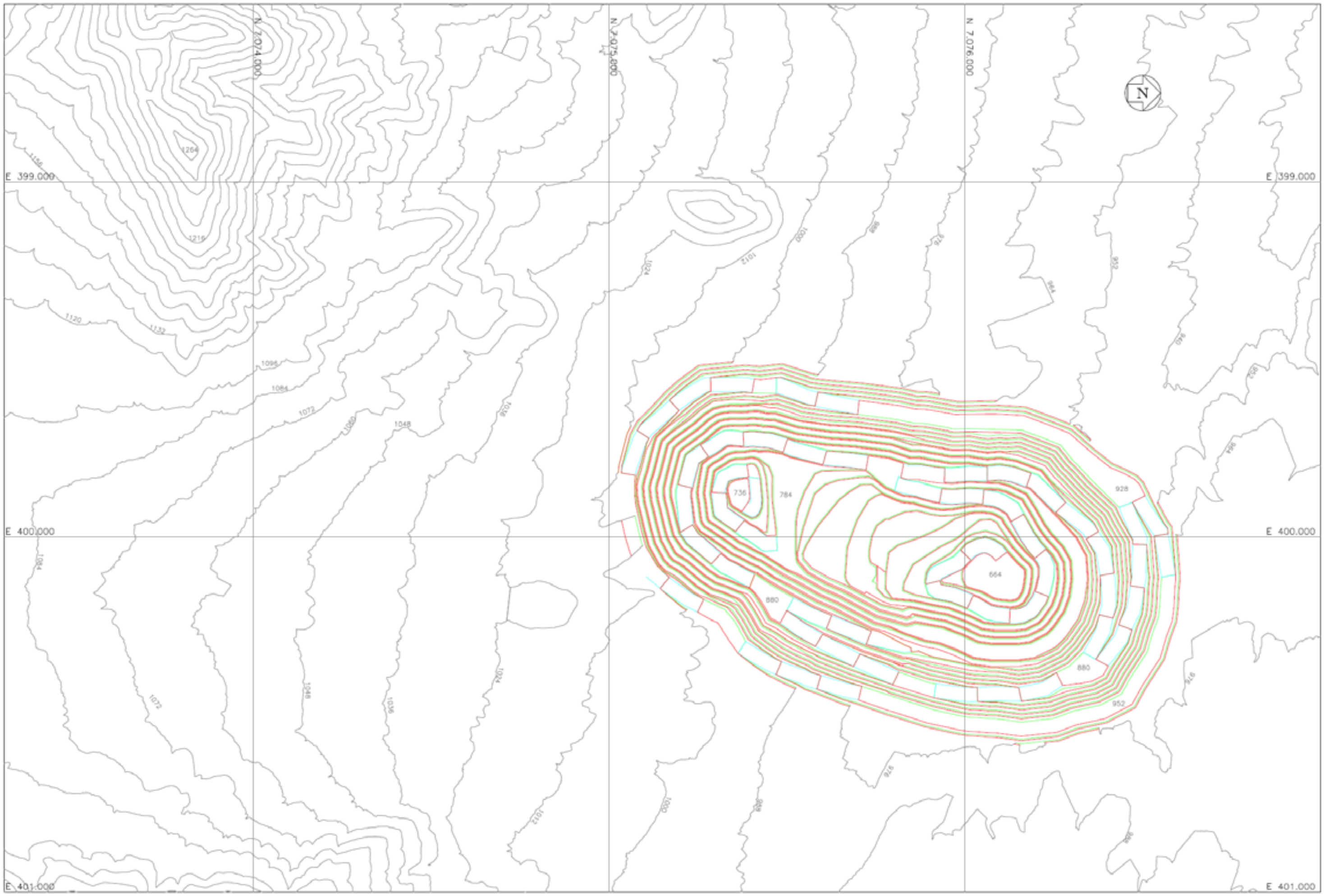
The Iris Norte pit will have one exit on the west side to provide access to the ROM pad area and primary crusher. On the east side there will be an exit to access the waste rock storage area. The final pit will be 1,600 m long in the north–south direction and 900 m wide in the east–west direction. The 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 16-2shows the final Iris Norte pit layout.

Figure 16-1: Santo Domingo Pit Layout Plan



Note: Figure prepared by NCL, 2018. Map north is to right of plan. Grid indicates scale. Grid squares are 1 km x 1 km.

Figure 16-2: Iris Norte Pit Layout Plan



Note: Figure prepared by NCL, 2018. Map north is to right 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 Sur and three for Iris Norte (refer to Figure 16-3).

In Santo Domingo Sur, 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 Sur (SD04) is the final expansion to the north, deepening the central portion down to 676 masl. This expansion includes the Iris sector, which is mined together with Santo Domingo Sur 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.

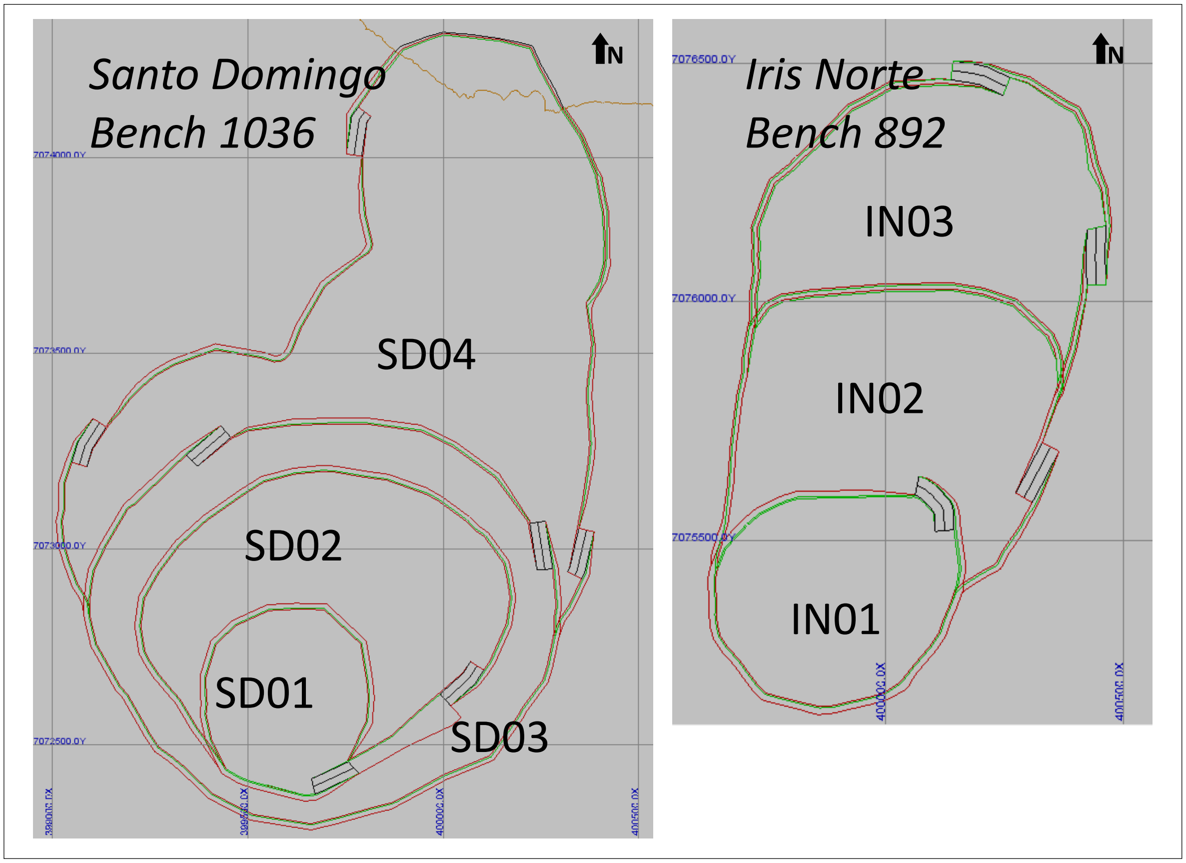
Three mining phases were designed in Iris Norte (IN01, IN02, IN03), which are successive expansions from south to north, going down to 736 masl, 724 masl, and 664 masl elevation, respectively. Each phase has accesses from the east and west sides.

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 LOM (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.

Figure 16-3: Mining Phases



Note: Figure prepared by NCL, 2018. The Santo Domingo Sur pit is shown at the 1,036 masl mining bench, the Iris Norte pit is shown at the 892 masl bench

Table 16-2: Mine Production Schedule Summary

Period	Ore				Marginal Ore				Oxide (Cu>0.2%)	Waste	Total
	('000 t)	Cu (%)	Fe (%)	Wi (kWh/t)	('000 t)	Cu (%)	Fe (%)	Wi (kWh/t)	('000 t)	('000 t)	('000 t)
Y-1	0	0.000	0.00	-	0	0.000	0.00	-	69	19,931	20,000
Y0H1	476	0.723	29.08	13.0	0	0.000	0.00	-	237	24,287	25,000
Y0H2	2,507	0.684	32.86	12.1	0	0.000	0.00	-	3,928	48,564	55,000
Y1	22,963	0.676	31.62	12.1	32	0.126	10.57	18.7	2,514	81,992	107,500
Y2	23,932	0.599	29.69	12.5	28	0.127	16.02	17.6	1,274	82,266	107,500
Y3	23,916	0.486	30.62	12.1	143	0.123	17.29	16.0	345	83,095	107,500
Y4	23,925	0.457	30.56	12.2	86	0.117	17.45	16.0	979	82,510	107,500
Y5	24,266	0.409	26.78	13.4	301	0.105	18.18	15.8	47	71,586	96,200
Y6	22,509	0.367	27.17	13.1	260	0.094	17.47	16.1	502	72,929	96,200
Y7	22,384	0.301	27.90	12.6	193	0.094	20.87	14.8	2,582	71,040	96,200
Y8	22,433	0.228	27.72	12.7	423	0.081	20.57	14.6	4,373	68,970	96,200
Y9	22,277	0.227	26.23	13.1	190	0.061	19.17	15.2	945	72,788	96,200
Y10	21,955	0.193	25.65	13.4	87	0.085	16.46	16.3	0	74,159	96,200
Y11	22,115	0.184	26.31	13.3	12	0.122	13.61	17.4	1,997	72,077	96,200
Y12	22,498	0.206	25.78	13.1	53	0.063	15.94	16.2	858	72,791	96,200
Y13	22,045	0.175	26.54	12.9	55	0.092	16.10	16.2	164	57,236	79,500
Y14	19,583	0.147	28.52	12.4	139	0.107	20.08	15.0	1,729	58,049	79,500
Y15	16,332	0.125	29.35	12.2	129	0.104	24.02	13.9	0	63,039	79,500
Y16	17,277	0.144	31.42	11.6	70	0.087	30.54	12.1	964	53,189	71,500
Y17	20,695	0.080	28.46	12.2	105	0.087	35.30	11.3	0	15,201	36,000
Y18	15,915	0.065	27.24	12.5	19	0.043	26.02	12.9	0	9,064	24,998
Total	***390,001***	***0.299***	***28.21***	***12.6***	***2,325***	***0.093***	***20.08***	***15.1***	***23,509***	***1,254,763***	***1,670,598***

Note: *The total of 2.3 Mt of marginal ore mined and stockpiled for later re-handle for a total LOM mill throughput of 392.3 Mt.

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.

The total mined waste considers two main destinations for the material; the main waste rock storage areas and the tailings storage facility (TSF) for the embankment construction:

- Waste requirements for the 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 ricj 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 (WRF) areas at the west and south of the pits were designed for the 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 2.3 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 that this material can 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 from the second to fifth years on a quarterly production basis.

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 the 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 provided in Table 16-3.

16.4 Blasting and Explosives

The drilling equipment will consist of diesel units capable of drilling 9⅞″ 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 LOM.

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 rock 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 rock storage areas

Table 16-3: Plant Feed Schedule

Period	Plant Feed									High Grade Stockpile			Marginal Stockpile		
										In	Out	Level	In	Out	Level
	('000 t)	Cu (%)	Rec. (%)	ConCu ('000 t)	Fe (%)	MassRec (%)	ConFe (Mt)	Au (g/t)	Recovered Au (koz)	('000 t)	('000 t)	('000 t)	('000 t)	('000 t)	('000 t)
Y-1	0	-	-	-	-	-	-	-	-	0	0	0	0	0	0
Y0H1	0	-	-	-	-	-	-	-	-	474	0	474	2	0	2
Y0H2	2,551	0.685	94.4	56.9	32.83	15.1	0.39	0.09	5.2	0	47	426	3	0	5
Y1	23,292	0.678	94.4	514.1	31.63	11.6	2.70	0.09	45.6	0	388	38	91	0	96
Y2	23,725	0.604	94.2	465.3	29.83	14.0	3.33	0.08	39.1	0	38	0	273	0	369
Y3	23,790	0.488	93.8	375.4	30.70	16.1	3.82	0.07	32.3	0	0	0	269	0	639
Y4	23,725	0.460	93.7	352.5	30.67	17.0	4.04	0.06	29.6	0	0	0	286	0	925
Y5	23,725	0.416	93.6	318.5	27.00	10.1	2.39	0.06	27.3	0	0	0	842	0	1,767
Y6	21,900	0.367	93.4	259.1	27.17	12.2	2.67	0.05	21.5	609	0	609	260	0	2,027
Y7	21,960	0.301	93.2	212.2	27.90	17.5	3.85	0.04	16.9	424	0	1,034	193	0	2,220
Y8	21,900	0.228	92.8	159.9	27.72	18.2	3.98	0.03	13.0	533	0	1,567	423	0	2,644
Y9	21,900	0.227	92.8	158.8	26.23	16.2	3.56	0.03	13.1	377	0	1,943	190	0	2,834
Y10	21,900	0.193	92.7	135.3	25.65	17.9	3.92	0.03	10.3	55	0	1,998	87	0	2,920
Y11	21,960	0.184	92.7	129.4	26.31	21.8	4.78	0.03	9.9	155	0	2,153	12	0	2,932
Y12	21,900	0.206	92.6	144.4	25.78	21.0	4.60	0.03	10.9	598	0	2,751	53	0	2,985
Y13	21,900	0.175	92.3	121.8	26.54	22.9	5.00	0.03	9.3	145	0	2,896	55	0	3,040
Y14	21,900	0.149	92.0	103.8	27.65	24.7	5.40	0.02	7.3	0	790	2,106	139	1,527	1,652
Y15	19,913	0.134	92.1	84.9	28.20	27.1	5.40	0.02	5.6	0	1,800	306	0	1,652	0
Y16	17,653	0.145	92.1	81.5	31.34	30.6	5.39	0.02	4.9	0	306	0	0	0	0
Y17	20,799	0.080	90.9	51.8	28.50	25.9	5.39	0.01	1.7	0	0	0	0	0	0
Y18	15,934	0.065	90.4	32.2	27.24	27.9	4.44	0.01	1.1	0	0	0	0	0	0
Total	***392,326***	***0.297***	***93.4***	***3757.8***	***28.16***	***19.1***	***75.06***	***0.04***	***304.6***	***2,407***	***3,369***	***0***	***3,179***	***3,179***	***0***

- Maintain all the mine work areas, in-pit haul roads and external haul roads; and maintain the waste rock 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 Mt. 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/m^3 was used for ore and 2.80 t/m^3 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 2018 Feasibility Study update 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 Mt, 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.

Table 16-4: Peak Fleet Requirements for Pre-Production and Commercial Production

Type of Equipment	Peak Pre-Production	Peak Requirement
FEL L-2350	1	1
Hydraulic shovel PC 8000	1	4
Haul truck 930E-4SE	6	29
Diesel drill DR 460	3	6
Support drill	1	2
Bulldozer 1 D 375A-6R	2	5
Bulldozer 2 D475A-5E0	1	1
Wheel dozer 1 WD 600-3	1	3
Wheel dozer 2 WD 900-3A	1	1
Motor grader 1 GD 825A-2	1	3
Water truck HD 785-7	1	2
Backhoe	1	1
Fuel truck 85 m^3	1	1
Mobile crane 200 t	1	1
Lowboy truck CXU 613/100 t	1	1
Tire handler WD 600-3	1	1
Lighting plant MOTOR LDW 1003 GE	8	15

Note: FEL= front-end loader

Table 16-5: Fleet Requirements by Year

	Y-1	Y0 H1	Y0 H2	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18
FEL L-2350	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Hydraulic shovel PC 8000	1	1	4	4	4	4	4	4	4	4	4	4	4	4	4	3	3	3	3	1	1
Haul truck 930E-4SE	4	6	13	19	18	22	22	23	22	23	26	29	29	29	28	24	27	26	26	15	16
Diesel drill DR 460	2	3	5	6	6	6	6	6	6	6	6	6	5	6	6	5	5	5	5	3	3
Support drill	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Bulldozer 1 D 375A-6R	2	2	5	5	5	5	5	5	5	5	5	5	5	5	5	4	4	4	3	2	2
Bulldozer 2 D475A-5E0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Wheel dozer 1 WD 600-3	1	1	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	2	1	1
Wheel dozer 2 WD 900-3A	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Motor grader 1 GD 825A-2	1	1	2	2	2	3	2	3	3	3	3	3	3	3	3	2	2	2	2	2	2
Water truck HD 785-7	1	1	1	1	1	1	1	2	1	2	2	2	2	2	2	2	2	2	2	1	1
Backhoe	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel truck 85 m^3	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mobile crane 200t	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lowboy truck CXU 613 / 100 t	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tire handler WD 600-3	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lighting plant MOTOR LDW 1003 GE	7	8	14	15	15	15	15	15	15	15	15	15	14	15	15	13	13	13	13	8	8

Note: FEL= front-end loader

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.

The total haulage distance varies from a minimum of 1.5 km to a maximum of 7.0 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, and 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, and 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 WRF 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 m^3)
- 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 design drawing of the flow sheet is included as Figure 17-1.

17.1.1 Coarse Ore Handling and Crushing

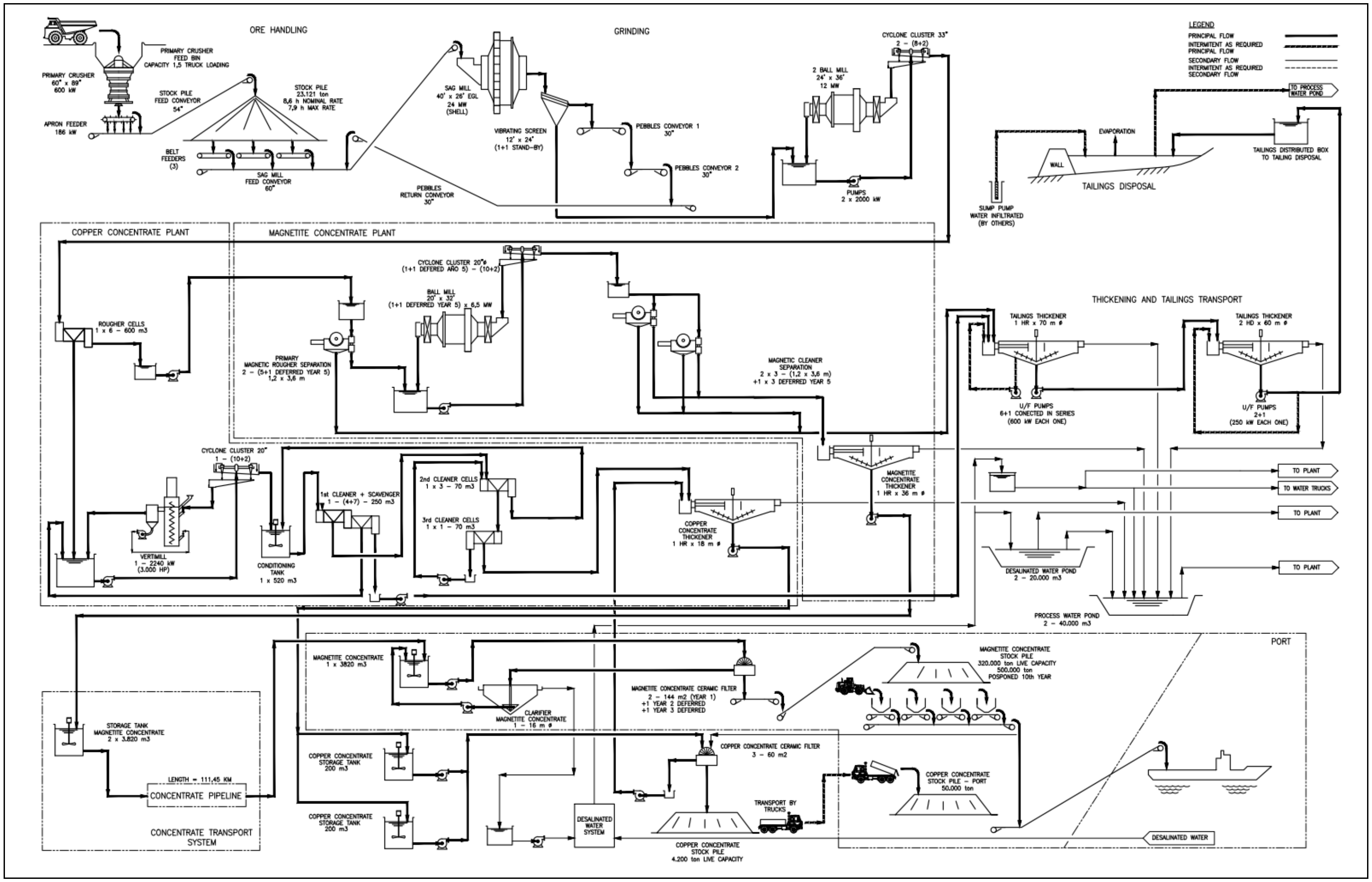
The primary crushing plant will process ROM feed in open circuit. The feed area for the primary crushing will be designed to position two 290 t trucks simultaneously. These trucks will be able to unload material simultaneously to the 450 t capacity feed hopper which has a fixed rock breaker. Primary 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 and desalinated water from the water storage pond. The 24 MW (at the shell), 40′ x 26′ EGL SAG mill will operate in a direct (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 secondary grinding circuit 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 cyclone clusters of ten 33″ hydrocyclones with 20% spare cyclone flow capacity. The coarse (underflow) fraction from the hydrocyclones will be returned to the ball mill feed and the fine material (overflow) fraction will be the final comminution circuit product with a P_{80} of 150 µm.

Figure 17-1: Design Flow Sheet



Note: Figure prepared by Wood, 2018

Each cyclone cluster 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 cyclone cluster will feed the dedicated ball mill and the combined fine discharge will be sent to the primary flotation.

17.1.3 Copper Flotation

Copper rougher flotation will be carried out in a single bank of six 600 m^3 mechanical, forced air tank cells arranged in a 1-1-1-1-1-1 configuration. Flotation rougher concentrate produced from the rougher cells will flow by gravity to the Vertimill feed box where it will be combined with first cleaner scavenger concentrate, and then fed to the regrinding stage which will consist of a single vertical mill and cyclone cluster 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 mechanical, forced air 250 m^3 tank cells; the concentrate will be pumped to the second cleaner stage and the tailings will 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 mechanical 70 m^3 tank cells with the concentrate flowing by gravity to feed the third stage of cleaner flotation. Second cleaner stage tailings will be pumped back to the feed of the first cleaning flotation stage.

The third (final) cleaning stage will be performed in a single mechanical 70 m^3 tank cell.

17.1.4 Copper Thickening

The final copper concentrate will be thickened in an 18 m diameter thickener. The copper concentrate thickener underflow will discharge at 60% solids w/w and will be pumped to the copper concentrate filtration section.

17.1.5 Copper Filtration and Load Out

Copper concentrate will be filtered in three ceramic disc filters. Filter area design considerations include an area of 60 m^2 and a unit filtration rate of 450 kg/hr/m^2. The recovered water from filtration 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. The conveyor system will unload into the copper concentrate stockpile. The concentrate will be loaded into trucks from the stockpile using front-end loaders (FEL), for transport to the port.

17.1.6 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 twelve 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.

17.1.7 Magnetite Thickening

The final magnetite concentrate from each line will be collected in a central launder, feeding the magnetite concentrate thickener via gravity flow. 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. Overflow water from the concentrate thickener will report 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 m^3 tanks. A pumping and pipeline system will transfer the magnetite slurry to the receiving tanks at the port.

17.1.8 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, 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 will supply flocculant for the final thickening stage.

17.1.9 Grinding Media

There will be several grinding media handling systems to serve the mills, to provide balls for the SAG mill, ball mills, copper concentrate regrind mill, and the magnetite concentrate regrinding mills.

17.1.10 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 conducted at the 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 re-circulate 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 to the process water pond. Final thickened tailings will be pumped to a tank at the TSF.

17.1.11 Plant Desalinated Water Distribution

The desalinated water received from the port area will be discharged into a distribution box at the plant site. This distribution box will have two separate discharge lines; one will discharge by gravity to the fresh desalinated water tank (2,900 m^3) that will feed the potable water plants, and the other will feed the desalinated water ponds (two, each 20,000 m^3, each designed to provide 24 hours of supply). Water from the tank will feed two potable water systems, one for plant services and the other for general plant site consumption.

From the two desalinated water ponds water will be pumped to the process water ponds or directly to consumption points.

From time to time, port process discharge water will be pumped to site with the desalinated water and this will supply only the two desalinated water ponds.

17.1.12 Plant Auxiliary Facilities

The air distribution in the plant will provide compressed air for consumption as plant air and 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 four 500 kW blowers (two operating, two stand-by) with a distribution network to each flotation bank.

17.1.13 Port

Copper Concentrate

The copper concentrate will be delivered to the port by trucks which will discharge the concentrate within the copper concentrate storage building. 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 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): 514.1 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): 259.1 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 initially contain two ceramic disc filters and four filters after Year 5, each with a filter area of 144 m^2. The requirement is for two filters for the first year of operation, three filters for the second and third years of operation and four filters from Year 5 onwards as magnetite concentrate production increases.

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. It is planned to install a steel structure 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): 4.04 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.

17.1.14 Water Supply Facilities

Desalinated water for all of the facilities (mine and plant site and the port) will be provided by a build, own, operate, transfer (BOOT) contractor and delivered to Capstone at the port and the mine site at an agreed price per cubic meter.

The BOOT water desalination facilities will consist of a sea water intake, filtration, treatment, and a reverse osmosis desalination plant located at the port area and operated by the BOOT water supplier. The BOOT contractor will also own and operate the desalinated water pipeline system to deliver water at the plant site.

Potable water treatment facilities will be operated by Capstone and will be located at the port and at the plant site, to supply potable water to the port and mine, plant and camp facilities.

17.1.15 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. Air is not required for the filter plant operation other than instrument air which will be provided from the instrument air accumulator. 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.

17.2.3 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 ≥ 8,000
- Hematite = magnetic susceptibility 2,000 ≤ 8,000.

During the 2014 Feasibility Study, 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. This resulted in the following average treatment capacities for the two feed types:

- Magnetite: 66,629 t/d
- Hematite: 61,844 t/d.

Wood 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 feed types.

Table 17-1: Utilization Rates

Area	Utilization
Primary crusher	65%
Grinding	93%
Flotation	93%
Tailings thickener	93%
Copper concentrate thickener	93%
Reagent (lime – flocculant)	93%
Magnetic separation	93%
Magnetite concentrate thickener	93%
Concentrate pipeline	98.5%
Filters and copper conc. handling	90%
Filters and magnetite conc. handling	90%

Table 17-2: Crushing and Grinding

Area	Specification
Crushing	
Crushing work index: design	8.4 kWh/t
Open size setting (O.S.S.)	180 mm
Grinding	
SAG mill	
Transfer size (K80) design	2,500 µm
Specific energy consumption DSAG	7.1 kWh/t
Ball mill	
Type of circuit	Closed
Product size (P80)	150 µm
Bond Ball work index (BWi)	
Average	12.4 kWh/t
Design	12.5 kWh/t

Table 17-3: Copper Circuit

Area	Flotation Time (mins)	pH
Copper flotation		
Rougher	40	7.5 – 8.2
Conditioning time	12	8.8 – 9.2
First cleaner	25	8.8 – 9.2
Cleaner scavenger	55	8.8 – 9.2
Second cleaner	18	8.8 – 9.2
Third cleaner	10	8.8 – 9.2
Area		**Specification**
Copper regrind mill		
Specific energy consumption		4.5 kWh/t
Product size		P80 of 34 µm
Copper concentrate thickener		
Settling rate		0.25 t/hr/m^2
Solid percentage underflow		60% w/w
Copper concentrate filter		
Unit filtration rate		60 m^2/filter
Unit rate		450 kg/hr/m^2

Table 17-4: Magnetite Circuit

Area	Specification
Magnetic separation	
Rougher	
Type of drum	LIMS
Intensity of magnetic field	1,000 gauss
Unit capacity	80 t/hr/m
Regrind ball mill	
Type of circuit	Closed
Product size	P80 of 40 µm
Ball work index (BWi)	

Area	Specification
Average	11.7 kWh/t
Design	13.6 kWh/t
Cleaners	
Type of drum	LIMS
Stages of cleaning	3
Intensity of magnetic field	650–750 gauss
Unit capacity	80 t/hr/m
Magnetite concentrate thickener	
Type of thickener	High rate
Unit rate	0.68 t/hr/m^2
Solid % underflow	65% w/w
Magnetite concentrate filter	
Type of filter	Ceramic disc
Unit filtration rate	144 m^2/filter
Unit rate	730 kg/hr/m^2
Tailings thickener	
First stage	
Type of thickener	High Rate
Unit rate	0.65 t/hr/m^2
Solid percentage underflow	55% w/w
Second stage	
Type of thickener	High density
Unit rate	0.5 t/hr/m^2
Solid percentage underflow	67% w/w

17.3 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 copper and iron recovery at yearly average treatment rates of 65,000 t/d and 60,000 t/d, with an annual peak production of 514.1 kt of copper concentrate in Year 1 and an annual peak for magnetite concentrate of 4.04 Mt in the first six years of production, and 5.40 Mt for the remaining mine life.

The distribution of Hematite and Magnetite type feeds over the LOM is indicated in Figure 17-2. The maximum treatment rate of Hematite in the plan is in Year 0 H1 and Year 1 (about 32% of the total processed in the year). The maximum rate treatment of Magnetite type 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.

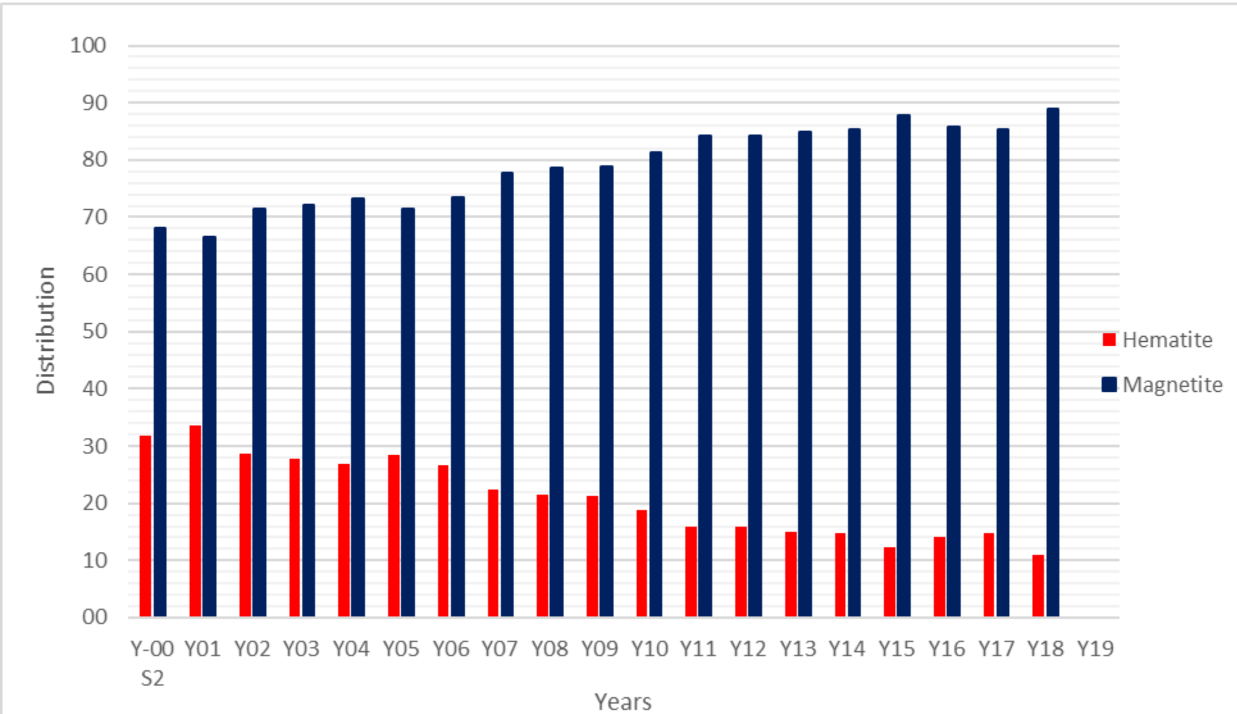
Figure 17-3 shows the grades of copper and iron in the plant feed. The head grade will vary between 0.42% Cu and 0.68% Cu during the first five years of production. After the fifth year up to Year 16, the head grade is projected to drop to between 0.37% Cu and 0.14% Cu. At the end of the mine life the head grade will be about 0.06% Cu. For the first five years, the head grade will be about 30% Fe, averaging 28% Fe with little variation over the LOM. Wood notes that copper production is economically viable (refer to Section 22), even at the lower grades at the end of the LOM, so that there is no specific copper cut-off grade when the copper circuit closes down.

Figure 17-4 shows the annual tonnes of copper and magnetite concentrate planned to be produced.

Table 17-5: Production Plan

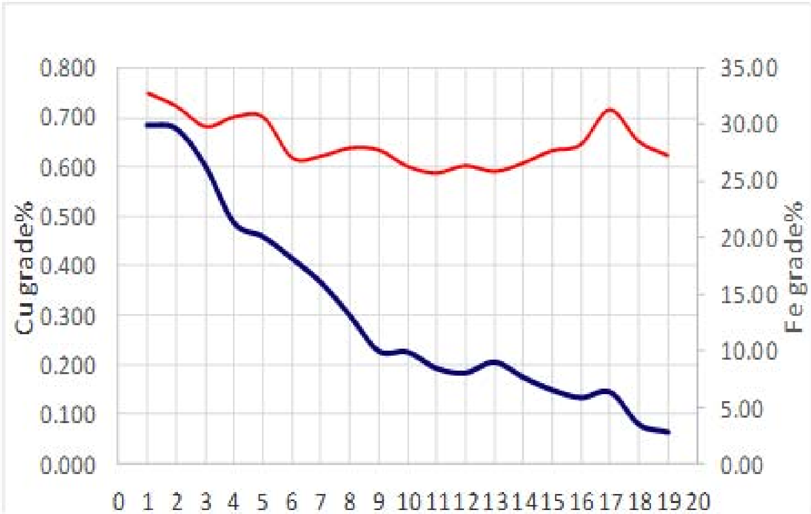
Period (year)	Tonnage ('000 t)	Cu (%)	Rec. (%)	Cu in Con ('000 t)	Fe (%)	MagSus	MassRec (%)	Fe in Con (Mt)	Au (g/t)	Rec Au (%)	Hem (%)
Y-1	-	-	-	-	-	-	-	-	-	-	-
Y0 H1	-	-	-	-	-	-	-	-	-	-	-
Y0 H2	2,551	0.685	94.354	56.9	32.83	14,537	15.1	0.38	0.09	66.8	31.8
Y1	23,292	0.678	94.367	514.1	31.63	11,267	11.6	2.70	0.09	66.8	33.6
Y2	23,725	0.604	94.169	465.3	29.83	13,726	14.0	3.33	0.08	65.6	28.6
Y3	23,790	0.488	93.807	375.4	30.70	15,594	16.1	3.82	0.07	63.6	27.8
Y4	23,725	0.460	93.685	352.5	30.67	16,806	17.0	4.04	0.06	62.9	26.8
Y5	23,725	0.416	93.583	318.5	27.00	10,019	10.1	2.39	0.06	62.3	28.5
Y6	21,900	0.367	93.448	259.1	27.17	12,061	12.2	2.67	0.05	61.2	26.6
Y7	21,960	0.301	93.246	212.2	27.90	17,259	17.5	3.85	0.04	59.6	22.3
Y8	21,900	0.228	92.756	159.9	27.72	18,023	18.2	3.98	0.03	56.6	21.4
Y9	21,900	0.227	92.787	158.8	26.23	16,071	16.2	3.56	0.03	57.0	21.2
Y10	21,900	0.193	92.693	135.3	25.65	18,089	17.9	3.92	0.03	55.0	18.7
Y11	21,960	0.184	92.686	129.4	26.31	22,124	21.8	4.78	0.03	54.1	15.8
Y12	21,900	0.206	92.637	144.4	25.78	20,855	21.0	4.60	0.03	54.8	15.8
Y13	21,900	0.175	92.292	121.8	26.54	22,931	22.9	5.00	0.03	52.5	15.1
Y14	21,900	0.149	92.026	103.8	27.65	25,372	24.7	5.40	0.02	50.4	14.7
Y15	19,913	0.134	92.067	84.9	28.20	27,879	27.1	5.40	0.02	49.2	12.2
Y16	17,653	0.145	92.106	81.5	31.34	31,239	30.6	5.39	0.02	48.8	14.1
Y17	20,799	0.080	90.851	51.8	28.50	25,590	25.9	5.39	0.01	31.8	14.7
Y18	15,934	0.065	90.419	32.2	27.24	27,500	27.9	4.44	0.01	28.2	11.0
Total	**392,326**	**0.297**	**93.387**	**3,757.8**	**28.16**	**19,107**	**19.1**	**75.06**	**0.04**	**60.1**	**21.0**

Figure 17-2: Mine Plan Hematite and Magnetite Distribution



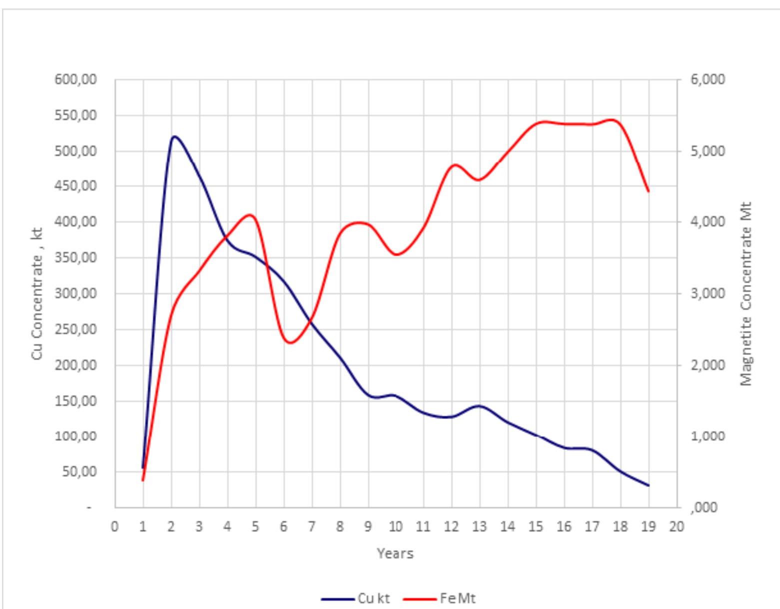
Note: Figure prepared by Wood, 2018.

Figure 17-3: Mine Plan Feed Grade Distribution



Note: Figure prepared by Wood, 2018. X-axis shows years. Blue line is copper grade, red line is iron grade.

Figure 17-4: Mine Plan Distribution of Concentrate Tonnes



Note: Figure prepared by Wood, 2018.

17.4 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, 18.7 and 18.8.

Reagents required for the plant operation include lime, primary collector (3418A) and secondary collector (3926), flocculant and frother (MIBC). Balls are required for the grinding circuit, ranging from 5″ diameter for the SAG mill to 1″ to 1.5″ diameter for the concentrate regrinding mills.

17.5 Comments on Section 17

For the first five years of operation, the Santo Domingo mine will have an annual average production of approximately 249 Mlb of copper contained in 390,000 dmt of concentrate (at an average copper content of 29%). The LOM average is 131 Mlb of copper in approximately 196,000 t of concentrate per year over a period of approximately 18 years. The total LOM production is estimated to be 2.40 Blb of copper contained in 3.758 Mt of concentrate.

For the same period, the average magnetite concentrate production is estimated to average 3.16 Mdmt per year. The magnetite concentrate production will average 4.15 Mdmt per year with a total estimated production of approximately 75.0 Mdmt for the LOM. The first five years of production do not include the Year 0 ramp up.

The effect of grinding to a P80 of 150 µm should be reviewed in the next stage of engineering, this would impact the energy consumption and hence also require an update of the operating costs. The updated copper and gold recovery models were obtained from flotation tests performed using material that was ground to P80 of 150 µm. These should be reviewed in the next engineering phase.

The use of desalinated water could require a modification in the flotation pH. This should be reviewed in the next stage of engineering and the mass recovery and/or consumption of lime should be confirmed through metallurgical testwork.

Based on experience in similar projects the use of desalinated water could result in an increase in copper recovery. It is recommended that in the next engineering stage the impact on equipment sizing be reviewed, especially flotation, regrind, filtration and copper concentrate transport system equipment.

18.0 PROJECT INFRASTRUCTURE

18.1 Introduction

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

- Santo Domingo mine and plant site: located at approximately 26°28′00″S and 70°00′30″W
- Operations camp: located on site
- Port facilities: located about 43.5 km north of Caldera at Punta Roca Blanca
- Concentrate and water pipelines: 111.6 km long between the Santo Domingo plant site location and 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 the Totoralillo substation to the port.

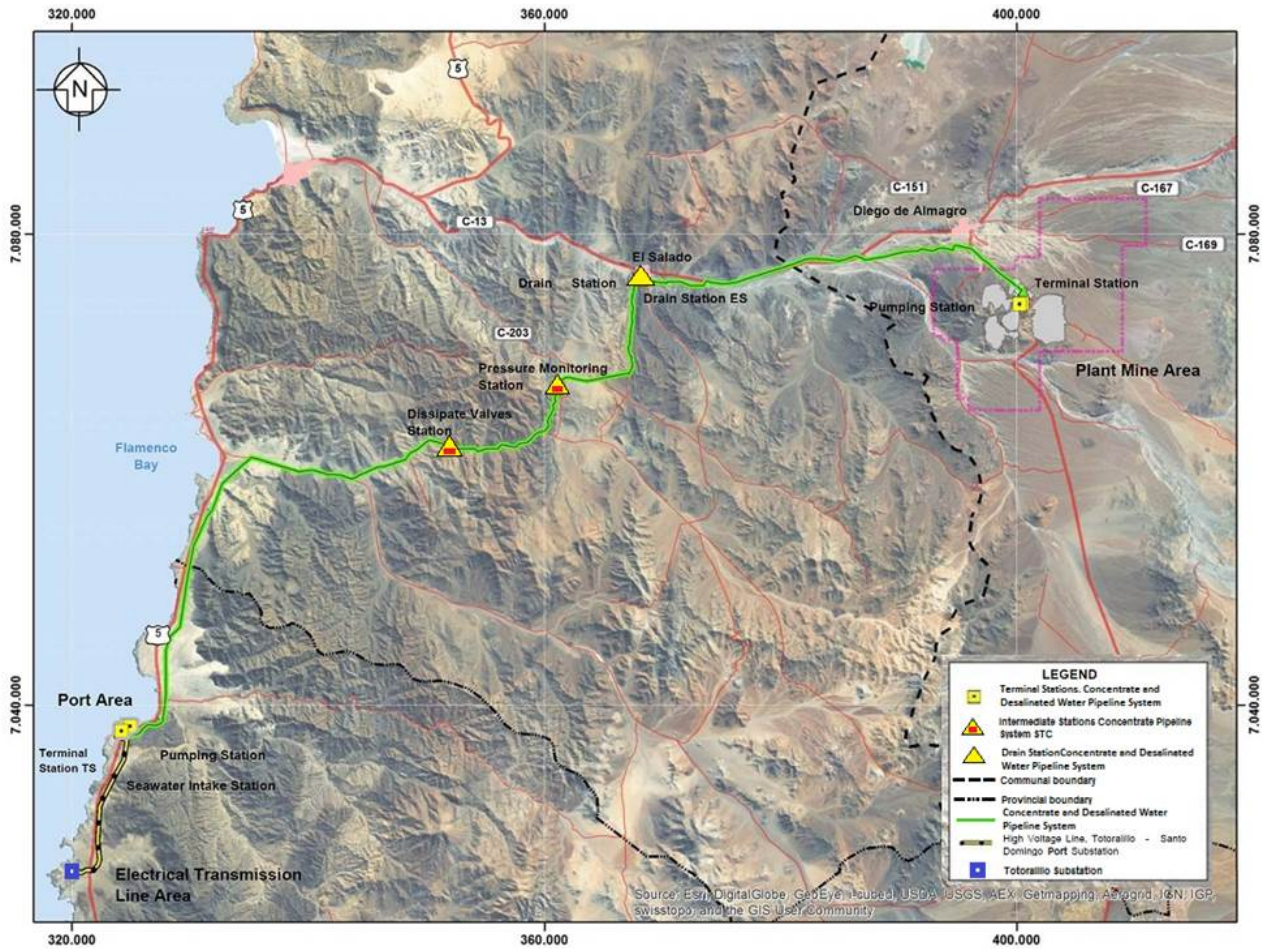
Figure 18-1 shows the overall Project layout from the mine site to the proposed port location. Figure 18-2 shows the details of the proposed mine site and plant layout.

18.2 Road and Logistics

18.2 1 Access

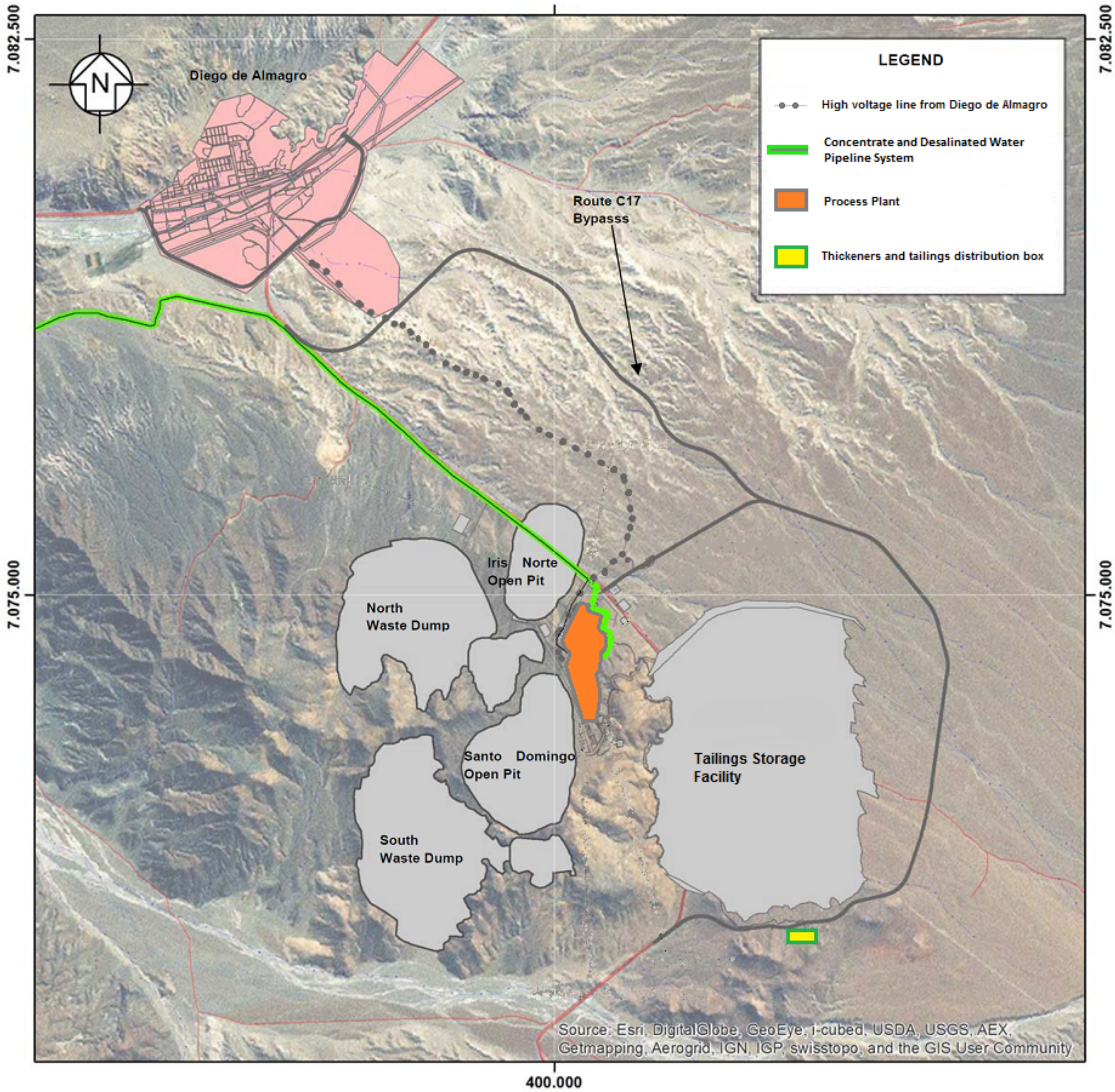
The planned route for transporting cargo, staff and equipment to the Santo Domingo site is from the south of the mine site by Route C-17 (Figure 18-3), and from the north by Route C13 (Figure 18-4).

Figure 18-1: Project Location Plan Showing Proposed Pipeline



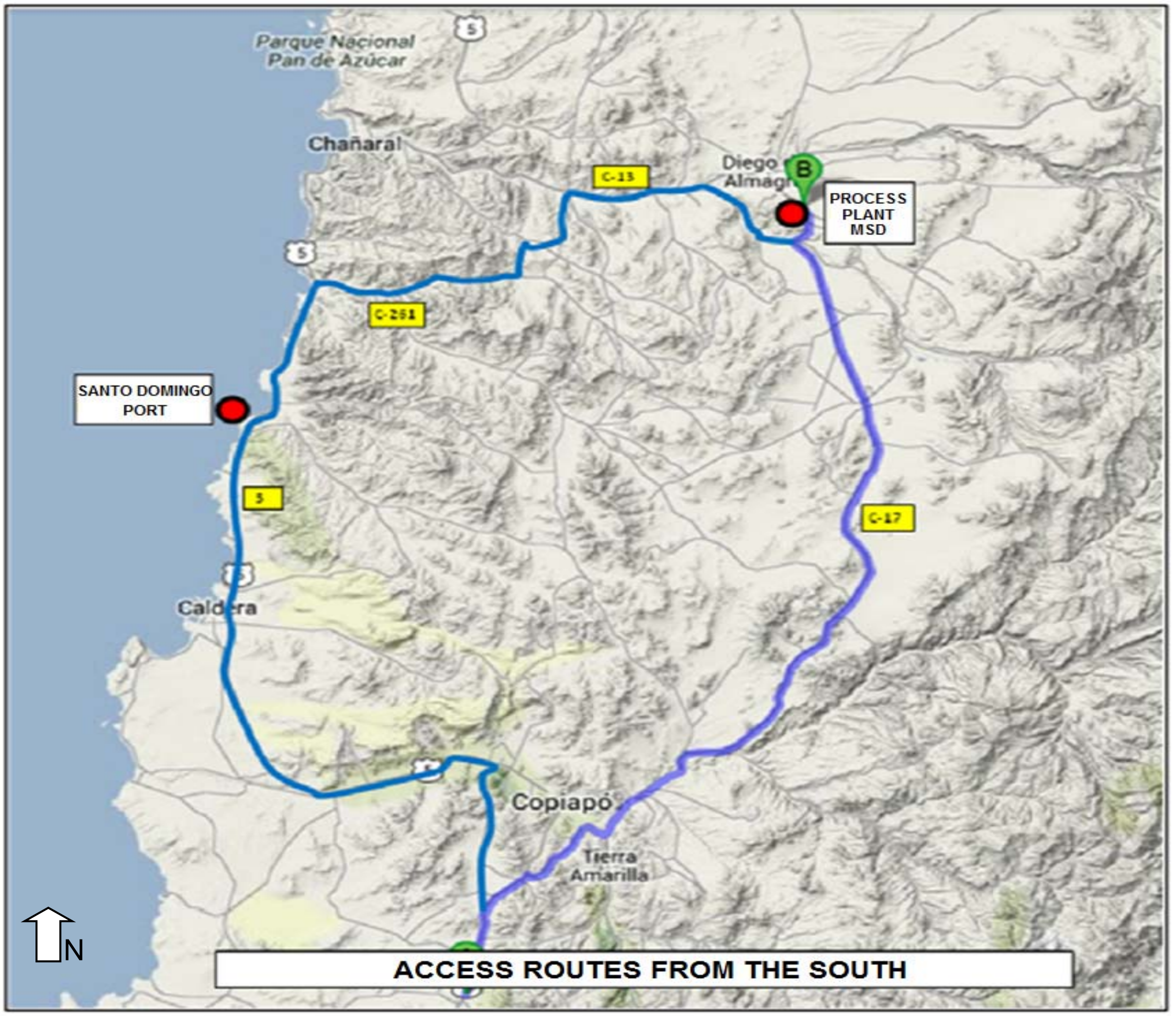
Note: Figure uses Esri Digital Globe as a base, modified by Wood, 2013. As an indicator of map scale, it is approximately 117 km from the proposed process plant location to the proposed port site

Figure 18-2: Proposed Mine Site and Plant Layout



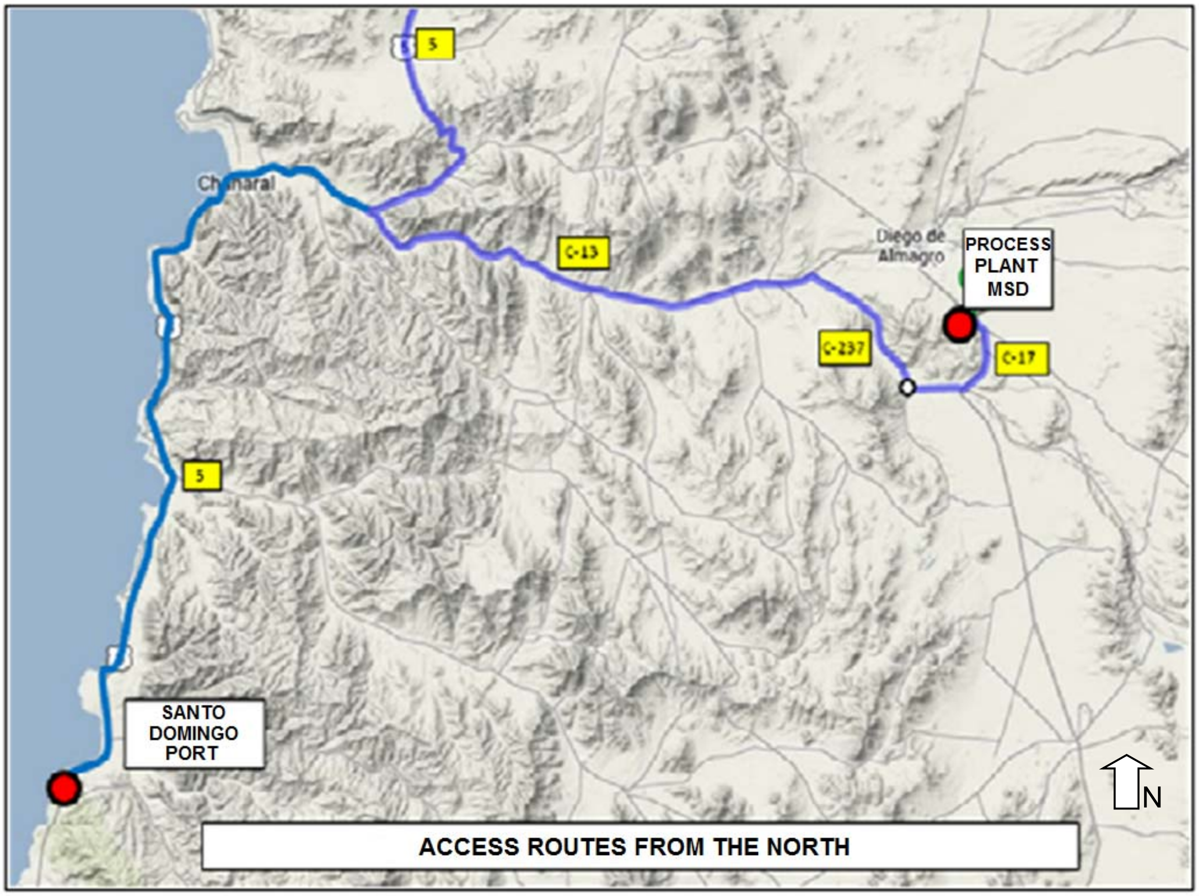
Note: Figure uses Esri Digital Globe as a base, modified by Wood, 2013. It is approximately 7 km from the planned concentrator site (orange outline located between the Iris Norte pit and the tailings storage facility) to the town of Diego de Almagro as an indicator of scale.

Figure 18-3: Access Routes from the South



Note: Figure uses Google Earth backdrop, modified by Wood, 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

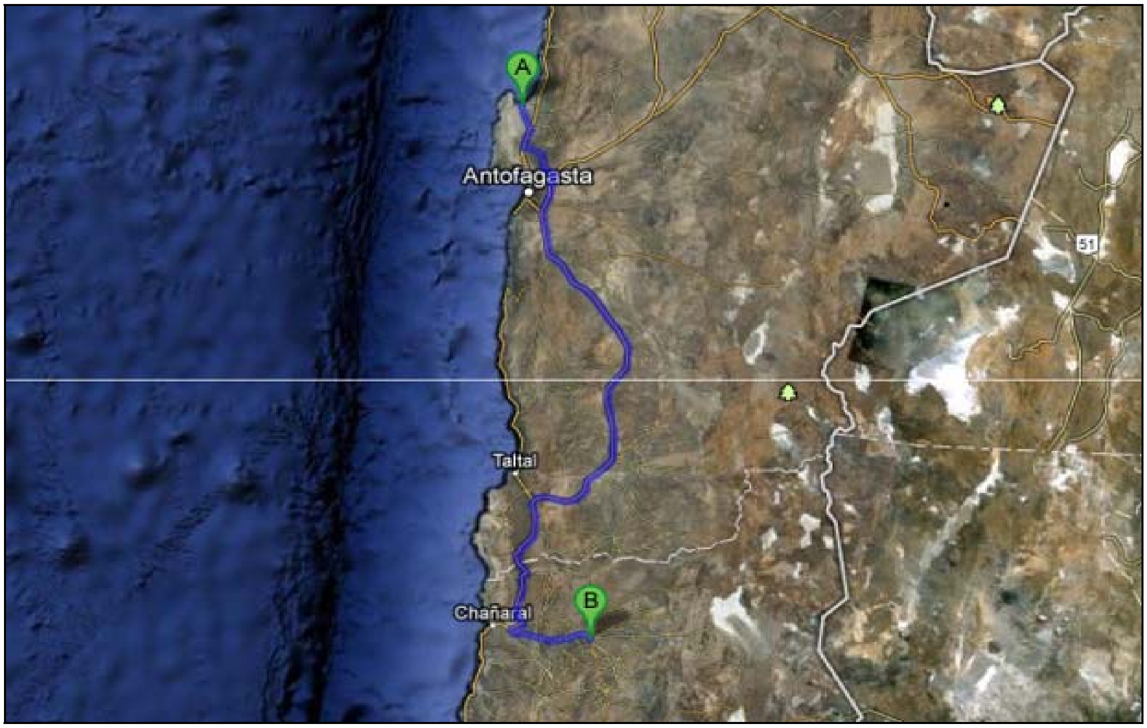


Note: Figure uses Google Earth backdrop, modified by Wood, 2013. As an indicator of map scale, it is approximately 117 km from the proposed process plant location to the proposed port site.

The closest airport to the Project site is the El Salvador Airport, a private airport, 44 km from the site. The 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 for construction is Punta Angamos in Mejillones, Antofagasta Region, 520 km from the plant site. This port is a year-round operation and is accessed directly from Route 5 North (Figure 18-5).

Figure 18-5: Punta Angamos Port, Mejillones



Note: Figure uses Google Earth backdrop, modified by Wood, 2013. Point A shown on the plan = Mejillones; point B = Santo Domingo site. Map north is to top of plan. The distance from Chañaral to Taltal is approximately 143 km as an indicative scale for the plan.

18.2.2 On-Site Access

Approximately 13 km of roads will be built on the Project site in order to connect the mine, plant, and infrastructure 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 site to the proposed Santo Domingo port. The approximate distance of the haulage is 117 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 by-pass 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 magnetite concentrate pipeline 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 third-party-owned Manto Verde mine.

The pipeline route was optimized using a single right-of-way (RoW) and a common trench for the concentrate pipeline and the desalinated water pipeline. 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 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 Rock Storage Facilities

Three WRF 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.

The pre-stripping activities will generate approximately 44.2 Mt of waste rock that will be transported by trucks to the WRFs and 4.73 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/m^3 loose density was assumed in the design.

The construction sequence of the WRF areas is from bottom to top. The WRFs 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 WRFs when mining occurs at greater depths in the pit. The destination assignment to the different WRF areas was based on the minimum cycle time.

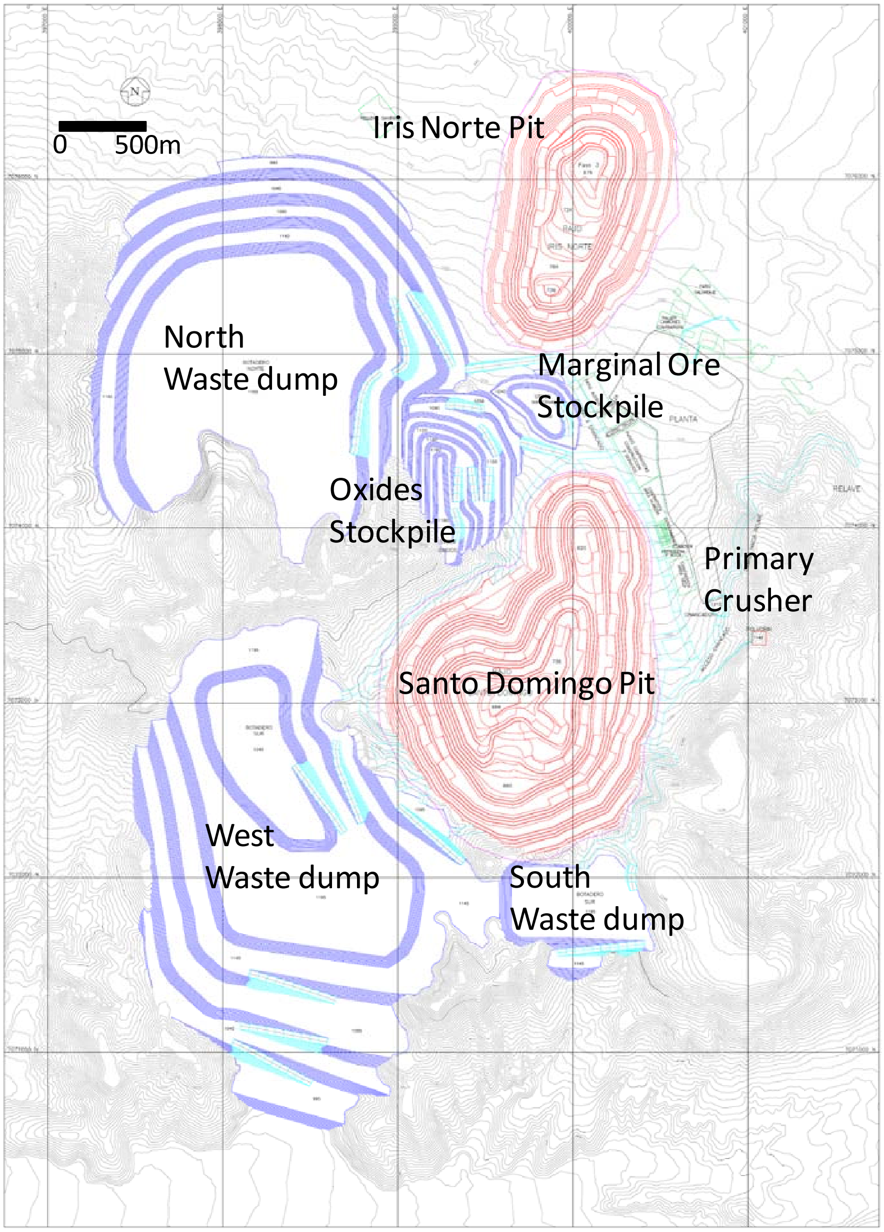
Based on a waste characterisation study undertaken internally by Capstone, Capstone has concluded that the WRFs show a moderate to low potential for generation of acid rock drainage. As a result, no significant acid generation is expected by Capstone from the mined waste, and the dry climate conditions are also not expected by Capstone to produce sufficient water to generate drainage through the WRFs 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 0.47 Mt.

The marginal ore stockpile and the oxide stockpile will be located 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.

Figure 18-6: Final Pit and Waste Rock Facility Configuration



Note: Figure prepared by NCL, 2018. Waste dump = waste rock facility or WRF. Figure north is to top of plan.

18.5 Water Management

18.5.1 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 proposed Santo Domingo mine site area 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 run-off.

Climate data were obtained from one meteorological station at the planned mine 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 Capstone. The meteorological datalogger has been operating since 2010 and consists of five stations.

18.5.2 Water Requirements

The mineral processing facility will use desalinated sea water. The current plan is to produce desalinated water under a BOOT or build, own, operate (BOO) contract with a third party. Desalinated water will be used for the process and Capstone will operate potable water treatment plants for consumption in the mine and port areas and to supplement water resources in Diego de Almagro.

The water requirement during the construction phase will be provided by Aguas Chañar, the water utility for the Atacama Region. 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:

- Desalinated water will be used to supply the port site and mine site for process make-up and other non-potable water needs

- Potable water will be produced by Capstone by adding chlorine to a portion of the fresh 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
- Water from the magnetite concentrate filtration will be retained in a storage pond at the port. Periodically, this wash water will be pumped into the desalinated water supply line to the mine site. The effluent transfer will be managed to minimize the impact of this water on the fresh desalinated water quality at the mine and plant site. Recovered water from the TSF will be recycled to the process. Hence, there will be no discharge of this water.

The nominal requirement for desalinated water is 348.6 L/s. The average water content contained in the tailings slurry at 67% solids by weight will be 266 L/s.

The mineral processing system at the process plant will use fresh desalinated and recycled water. The forecast water requirement is 348.6 L/s and is generally distributed as shown in Table 18-1.

Potable water, water for construction and dust control (provided by Aguas Chañar) during construction are variable, with a maximum of 15 L/s.

During operations water 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: 10.0 L/s (Capstone has committed to provide potable water to Diego de Almagro).

The desalinated water demand at the port is shown in Table 18-2.

Table 18-1: Desalinated Water Requirements – Plant Area

Area	Flow (L/s)
Potable water	2.5
Water for process and infrastructure	318.6
Potable water for Diego de Almagro	10
Water for dust control on roads	17.5
Total desalinated water – process plant	348.6

Table 18-2: Desalinated Water Requirements at the Port

Area	Flow (L/s)
Desalinated water for the filter plant	0.5
Potable and process water for port facilities	1.0
Total desalinated water - port	1.5

18.6 Desalinated Water Pipeline and Water Usage

The current plan is to provide desalinated water for the mine site and port operations through a BOOT or BOO contract. Potable water will be produced by adding chlorine to a portion of the desalinated water at the port and the mine. The chlorinated water will be used for potable water at the port and the mine and to provide water to the town of Diego de Almagro.

Desalinated water will be pumped from the port to the mine site. At the mine site, the desalinated water will be discharged into a distribution box that will feed the fresh desalinated water tanks and the desalinated water storage ponds.

18.6.1 Process Water Storage

Two high-density polyethylene (HDPE)-lined open ponds for process water will be located near the process plant and will have a total 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 desalinated water storage ponds.

Recycled water from the magnetite concentrate filtration at the port will also be pumped to site

Process water will be pumped from the open ponds by five 1,100 kW pumps (four operating/one stand-by), through a carbon steel pipe that will supply water to the distribution networks.

18.6.2 Desalinated Water

The plant and port will operate using desalinated water. The water from the port desalination plant will be distributed to the port and the mine site where it will be used as fresh make-up water and for other uses.

18.6.3 Potable Water

Construction

During construction, potable water required for the construction camp will be provided by Aguas Chañar.

Operations

Chlorinated desalinated water will be used as potable water. The potable water will feed into a 2,300 m^3 capacity potable water tank for gravity flow to Diego de Almagro. The water will be pumped for infrastructure and process plant use by two 12.5 kW pumps (one operating, one stand-by) and a carbon steel pipe distribution network.

Chlorinated water at the port will be stored in a 30 m^3 carbon steel tank and distributed by two 0.55 kW pump (one operating/one stand-by) for the various port water consumptions.

18.6.4 Water Treatment

The Project will have the following potable water treatment plants:

- At the port: one potable water plant will be required with a nominal capacity of 0.2 L/s
- At the plant site: one potable water plant at plant site will be required for Capstone use and to supply water to Diego de Almagro.

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 and operations camp: 600 m^3/d maximum capacity
- Plant site: 100 m^3/d capacity
- Port: 20 m^3/d capacity.

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.

18.7 Desalinated Water Pipeline

The current plan is that the pipeline system will be constructed and operated by the BOOT/BOO contractor. The electrical power supply for the intake, desalination plant and main pump stations will be provided by Capstone from the electrical system at the port area. Communications for both the desalinated water and concentrate pipeline systems will be managed by means of a fibre optic cable buried beside the pipelines in the common trench. The construction of the desalinated water line will be coordinated with the construction of the concentrate pipeline to use the same trench and minimize construction costs.

18.8 Magnetite Concentrate Pipeline

The magnetite concentrate transportation pipeline and the desalinated water transportation pipeline 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 concentrate transportation system is designed to transport magnetite concentrate slurry from the process plant to the concentrate filter plant at the port (Table 18-3). 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 magnetite concentrate pipeline, longitudinal grades will be typically less than ±12% with specific areas up to ±15%. The concentrate pipe will be constructed of steel pipe with an HDPE internal liner. The concentrate pipeline will be inspected 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).

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 magnetite 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 will 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 allow drainage of the concentrate pipeline.

The drain, choke, and terminal stations will each have emergency ponds.

Table 18-3: Projected Magnetite Concentrate Transport Volumes

Condition	Flow (m^3/hr)	Cp	Capacity		
			Dry Tonnes per Year (dt/a)	Dry Tonnes per Day (dt/d)	Dry Tonnes per Hour (dt/hr)
Maximum Tonnage	415	68%	5,400,080	14,795	626
Average Tonnage	335	66%	4,305,263	11,277	475
Minimum Tonnage	289	68%	3,756,342	10,291	435

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 process control system (PCS). This dedicated control system will use a programmable logic controller (PLC) network or personal computers (PCs).

18.9 Building Infrastructure

18.9.1 Mine and Plant Site

Buildings

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:
  - Office
  - Lunch room
  - Control gate
  - Access guard house.

- 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 projects)
  - 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 m^2
- Mine maintenance office: 120 m^2
- Change house: 39 m^2
- Lunchroom: 59 m^2
- Warehouse: 440 m^2
- Light vehicle maintenance shop: 97 m^2

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 m^2
- Lubricant storage: 400 m^2
- Reagent storage: 525 m^2
- Gas bottle storage: 75 m^2.

18.9.2 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
- Buildings associated with the desalination plant.

18.9.3 Camp and Accommodation

Accommodation for construction and operations personnel will be in one camp at the mine site using temporary units to increase the capacity during construction. These units will be removed when construction is complete. The planned location of the camp is 2.5 km from the mine and process area.

The camp will house and support construction and operations personnel during the construction and operation of the Project. The construction camp will be rented and will be removed from site when construction is complete leaving the permanent operations camp. During construction the camp will have capacity for up to 3,100 beds (including 307 beds for operations staff).

The permanent operations camp will house and support operations personnel after the completion of the Project. The proposed permanent camp will accommodate approximately 500 people. There is no plan to retain the 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.

18.10 Ancillary Infrastructure

18.10.1 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 m^3; this will be the tank minimum reserve volume. Extinguishers or other systems will be used in electrical rooms and other locations where water cannot be used.

At the port, the fire system will use desalinated water that is stored in the fire water tank. This tank is sized for 500 m^3 to feed the fire water network via an electrical pump with a diesel back-up pump.

18.10.2 Compressed Air Systems

The compressed air supply has been designed as a central compressor station to provide plant air and instrument air distribution. The system will include dryers for instrument air. The distribution network will supply plant air and instrument air for:

- Grinding
- Flotation
- Copper concentrate regrinding and thickening
- Magnetite concentrate LIMS area, regrinding, hydrocyclones and thickening
- Copper and iron concentrate storage tanks
- Tailings thickener
- Lime plant and reagents.

18.10.3 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.

18.10.4 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 and plant 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 solid waste generated at the port site will be stored in waste containers in the operating areas, and then collected and sent to authorized landfills in the Atacama Region.

18.11 Port

The port design for the 2014 Feasibility Study was prepared by PRDW and reviewed by Wood. PRDW reviewed and updated the port design in September 2018 to take into account changes in the Project assumptions since 2014. In this sub-section, Wood initially presents the PRDW design and design considerations, and then provides Wood's comments and observations.

18.11.1 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. It is planned to ship magnetite concentrate using a mixture of Panamax- and Cape-size vessels. Copper concentrate will be shipped using Panamax- and Handymax-size vessels.

18.11.2 Ocean Conditions

Bathymetry

A bathymetric survey was performed by the Chilean firm Desarrollo Maritimo, Servicios y Equipamiento (DESMAR) for Capstone, along with official cartography by the Chilean Naval Hydrographic and Oceanographic Service. The coastline in the proposed port 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 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 planned port site.

Currents

No field data on ocean currents is available for the port 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[1] at the berthing site are between Hm0[2] = 0.2 m and 2.9 m, with peak periods in the range Tp[3] = 6 s to 20 s, predominantly at intervals of Tp= 12 s to 14 s, as expected for the Pacific Ocean.

Figure 18-7 provides a wave rose diagram for Hm0 and Tp 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.

[1] Significant wave height: average height of the third-highest waves in sea state period.
[2] Significant wave height calculated with spectral data as zeroth moment wave height.
[3] Peak period: corresponds to the period associated with the energy peak in the wave spectrum.

Figure 18-7: Wave Rose Diagrams for Hm0 and Tp for Modelled Waves at the Berthing Site

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Notes: Direction is shown in five degree increments

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.

18.11.3 Geotechnical Considerations

Based on geotechnical investigations within the limits of the proposed onshore facilities, the area has a rocky soil. Seismic refraction analysis showed that there is rock at a depth of 10 m. No phreatic water (or other water) was found during the geotechnical investigations.

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.

18.11.4 Port Layout

The proposed port layout plan is presented in Figure 18-8.

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 maneuvering circle of 664 m diameter was assumed.

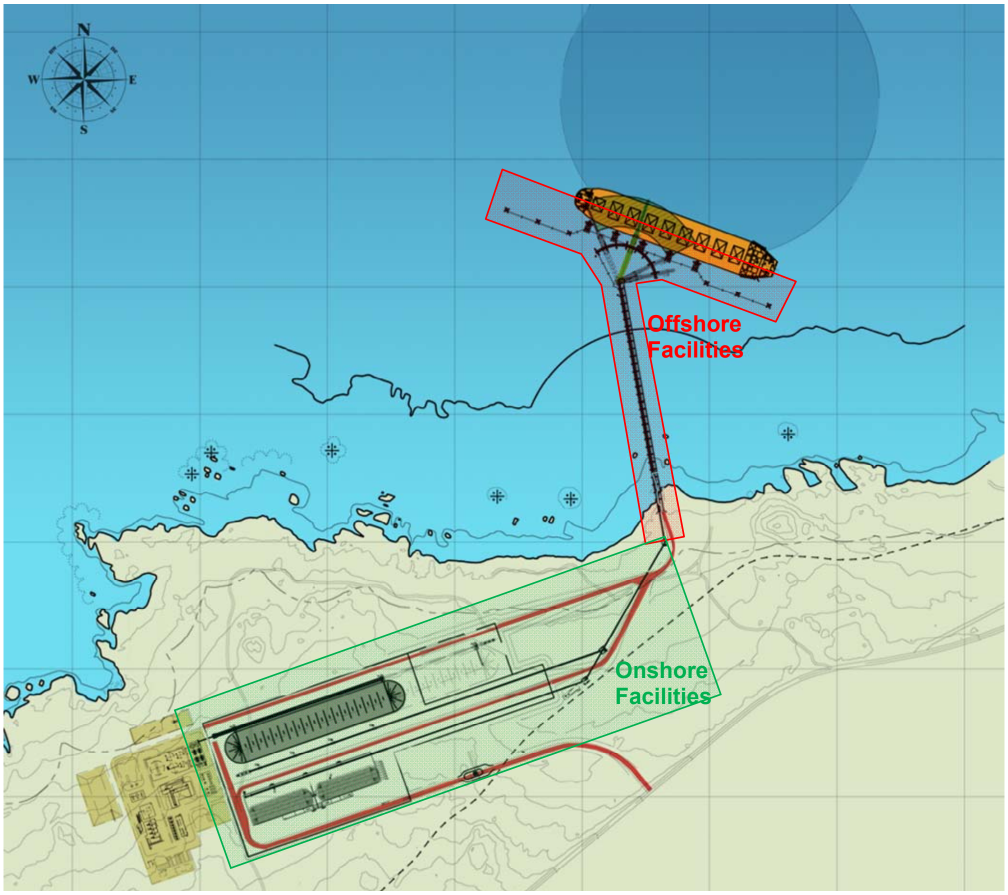
Structure Heights

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 ship-loading 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.

Figure 18-8: Proposed Port Layout Plan



Note: Figure prepared by PRDW, 2013. Grid squares on the figure are approximately 200 m.

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. One of the berthing dolphins is sized to support the gangway.

The berthing dolphins and the ship loader 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.

Elastomeric fenders will be installed on the front side (facing the ship berth) of the berthing dolphins. 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 will include:

- Magnetite concentrate stockpile area; mechanical stacker and reclaim conveyor
- Copper concentrate storage building and reclaim conveyor
- Transfer towers

- Conveyor galleries
- Reclaim 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 18-4.

The reclaim hopper system for the magnetite concentrate will be located adjacent to the magnetite concentrate stockpile at the port site. The reclaim hopper system for the copper concentrate will be situated inside the copper concentrate storage building at the port site. 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.

Copper Concentrate

Key metrics for the copper concentrate, concentrate stockpile and shipping requirements are included in Table 18-5.

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 in two stockpiles within the enclosed concentrate storage building. Front-end loaders will distribute the concentrate to the stockpiles inside the building.

Table 18-4: Magnetite Concentrate, Stockpile and Shipping Parameter Assumptions

Concentrate		
Characteristic	**Value**	
Moisture	7.5–9.5%	
Density - stock pile	2.8 t/m^3	
Density - conveyors	3.0 t/m^3	
Repose angle	34˚ to 36º	
Surcharge angle	15º	
Stockpile		
Characteristic	**Value Phase 1**	**Value Phase 2**
Number of piles	1	1
Length	298 m	450 m
Width	50 m	50 m
Height, m	16.9 m	16.9 m
Capacity	320,000 t	500,000 t
Stockpile Area	14,900 m^2	25,000 m^2
Shipping		
Ship	**Ship Capacity (DWT*)**	**Load Distribution# (%)**
Cape size	250,000	20
Cape size	180,000	70
Panamax	80,000	10
Total	—	100

Notes: * = dead weight tonnage; # = Percentage of the total annual load distribution of magnetite concentrate

Table 18-5: Copper Concentrate, Stockpile and Shipping Parameter Assumptions

Concentrate		
Characteristic	**Value**	
Moisture	8–9 %	
Density - stockpile	2.0 t/m³	
Density - conveyors	2.5 t/m³	
Repose angle	34˚ to 36º	
Surcharge angle	15º	
Stockpile		
Characteristic	**Value**	
Number of piles	2	
Length, m	100	
Width, m	30	
Height, m	5	
Capacity, t	25,000	
Surface area, m²	3,000	
Shipping		
Ship	**Ship Capacity (DWT)**	**Load Distribution[4] (%)**
Panamax	60,000	15
Handymax	45,000	70
Handy size	20,000	15
Total	—	100

The enclosed concentrate storage building will have a negative air pressure system and a dust collection system to minimize environmental impacts from loss of 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

[4] Percentage of the total annual load distribution of copper concentrate.

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 ship loader will be 4,000 t/hr for iron concentrate and 2,000 t/hr for copper concentrate.

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 magnetite concentrate stockpile. The other 31,800 m^2 will be used for the copper concentrate stockpile building. The second phase will require an expansion of the magnetite concentrate stockpile area by about 14,800 m^2. Environmental regulations require some protection around the perimeter of the magnetite concentrate stockpile area to reduce the wind speed in the vicinity of the stockpile to reduce dust generation.

18.11.5 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; to verify ship-loader loading rates; and to estimate the scheduling of concentrate transport ships.

The mining plan that was developed in mid-2013 was used as 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 magnetite and copper concentrate production, with the production of magnetite concentrate peaking at 3.3 Mt/a and copper concentrate reaching a maximum of 0.49 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 LOM (approximately Year 18). During Phase 2, the magnetite concentrate production peaks at 5.4 Mt/a and copper at 0.36 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 Phases 1 and 2, three nominal (or average) ship-loading rates were tested: 3,000 t/hr, 4,000 t/hr and 5,000 t/hr. The maximum occupancy level is assumed to be 40% in accordance with the recommendations of the United Nations Conference on Trade and Development. Using a 3,000 t/hr loading rate for magnetite concentrate, the estimated occupancy level exceeds the 40% maximum recommendation. Using a loading rate of 4,000 t/hr 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/hr for magnetite concentrate, and 2,000 t/hr 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 18-6.

Manoeuvring Studies

A manoeuvring feasibility study was performed to analyze the berthing and unberthing manoeuvres of the ships at the terminal. PRDW concluded that:

- 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

Table 18-6: Estimated Loading Time Averages

Material	Ship Capacity (t)	Total Loaded (t)	Average Loading Time (hr)
Iron concentrate	80,000	80,000	38
	180,000	180,000	64
	250,000	250,000	83
Copper concentrate	20,000	20,000	13
	45,000	20,000	11
	60,000	20,000	10

- 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 in 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 either as a requirement by 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.

18.11.6 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 dimensions. Both vessels were tested at fully-laden conditions.

The modelled 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 minimum breaking load (MBL) and safety factor of 2.0 for mooring lines, and 72% of maximum deflection for fenders
- Operational limits: Permanent International Association of Navigation Congresses 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; 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 ship loading primarily between November and 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 ship loading is summarized in Table 18-7. The annual availability for iron concentrate vessels, stern shifting, is presented in Table 18-8. Table 18-9 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 the winter season can be considerably higher than the average. The non-availability affects copper concentrate vessels more than the iron concentrate vessels.

18.11.7 Wood Comments on Port

PRDW provided Wood with the information in Table 18-7 to Table 18-9 on monthly port availability over the period 1985 to 2006 for copper and magnetite concentrate. The data in these tables indicate that in some years, there are six months (November to May) where the port availability to load copper concentrate may 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 by the selection of a sheltered harbour location and/or construction of breakwaters. Another option is to utilize a spread mooring system and a travelling ship loader 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.

Wood 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.

18.12 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.

Table 18-7: Annual Non- Availability for Copper Concentrate Vessel

	Ene	Feb	Mar	Abr	May	Jun	Jul	Ago	Sep	Oct	Nov	Dic	% Anual
1985	53.6	9.4	19.0		6.9				4.6	3.2	0.8	56.5	**12.8**
1986	54.4	85.3	67.7	0.8	12.9					12.1	16.7	58.9	**25.7**
1987	89.5	62.1	38.3	38.8	4.8		6.9			10.9	17.5	74.2	**28.6**
1988	61.7	15.1	39.9	8.8		1.7		6.0	1.7		22.9	9.3	**13.9**
1989		22.8	8.5	11.7	6.5	12.9	3.2	1.2		0.4	17.1	8.1	**7.7**
1990	54.8	34.4	9.7				0.8	2.8		5.6		3.2	**9.3**
1991	12.9	32.6	44.0	22.1									**22.3**
1992													
1993									6.7	0.4		45.2	**7.5**
1994	43.5	5.8		10.8		8.8	11.7	4.8			5.8	15.3	**8.9**
1995	56.5	10.7	4.8	7.1	1.6			2.8	5.4	8.9	2.5	13.7	**9.5**
1996	27.0	25.0	0.8	5.4	10.5		6.5	5.2	5.8	4.0	13.8	4.8	**9.1**
1997	16.9	23.2	2.4	7.5	4.4	15.0	4.4	3.2	1.3	21.0	34.6	29.8	**13.7**
1998	37.1	91.5	34.3	10.4	8.5	11.7	4.8	5.2	3.8	3.6		23.8	**19.6**
1999		12.5	12.5	5.0	2.8	5.8	2.4		4.6	1.6	5.8	14.9	**5.7**
2000		10.3	9.3	9.2	4.4	1.3	1.2		5.0		2.9	8.9	**4.4**
2001	38.3	14.3	13.3	1.3	0.4	5.0	1.6	0.8	2.1	0.8	7.9	3.2	**7.4**
2002	22.6		5.2	0.8				2.0	1.7	0.4	9.6	13.7	**4.7**
2003	12.1			1.3		4.2		3.2		6.9		2.4	**2.5**
2004	6.5	3.4	6.5	5.8		5.0	3.6		1.3			5.6	**3.1**
2005	2.8		9.3		5.6		2.8	4.0		0.4		6.5	**2.6**
2006	32.7	12.9	0.4		1.2	11.7	1.2	0.8		6.5	3.3	9.7	**6.7**

	Ene	Feb	Mar	Abr	May	Jun	Jul	Ago	Sep	Oct	Nov	Dic
Min	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	2.4
Mean	31.1	23.6	16.3	7.3	3.5	4.1	2.6	2.1	2.2	4.3	8.1	20.4
Max	89.5	91.5	67.7	38.8	12.9	15.0	11.7	6.0	6.7	21.0	34.6	74.2

Table 18-8: Annual Availability for Iron Concentrate Vessel, Stern Shifting

	Ene	Feb	Mar	Abr	May	Jun	Jul	Ago	Sep	Oct	Nov	Dic	% Anual
1985	4.8											11.3	**1.3**
1986	4.8	14.3	10.5									10.5	**3.3**
1987	27.4	8.5	10.5	6.7								10.5	**5.3**
1988	9.7	0.4											**0.8**
1989		4.0								1.6	3.3		**0.7**
1990	10.5												**0.9**
1991				8.3									**1.7**
1992													
1993												7.3	**1.0**
1994													**0.0**
1995	2.8											4.0	**0.6**
1996	1.6												**0.1**
1997										2.0			**0.2**
1998	7.3	22.3	2.8		4.0	2.1				1.6		1.6	**3.5**
1999			1.2		0.4	1.7			3.3				**0.6**
2000		4.7	1.6										**0.5**
2001	11.7												**1.0**
2002	4.8											3.2	**0.7**
2003						0.8							**0.1**
2004													**0.0**
2005			0.8										**0.1**
2006	10.1	2.7				2.1				0.4	0.8		**1.3**

	Ene	Feb	Mar	Abr	May	Jun	Jul	Ago	Sep	Oct	Nov	Dic
Min	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Mean	4.8	2.8	1.4	0.8	0.2	0.3	0.0	0.0	0.2	0.3	0.2	2.4
Max	27.4	22.3	10.5	8.3	4.0	2.1	0.0	0.0	3.3	2.0	3.3	11.3

Table 18-9: Annual Availability for Iron Concentrate Vessel, Bow Shifting

	Ene	Feb	Mar	Abr	May	Jun	Jul	Ago	Sep	Oct	Nov	Dic	% Anual
1985	14.5											27.0	**3.5**
1986	2.0	46.0	30.2									9.7	**7.3**
1987	50.4	3.1	12.1	11.7			0.4					24.2	**8.5**
1988	2.0		5.6										**0.6**
1989		8.9								1.6	4.6		**1.3**
1990	13.7	7.6											**1.8**
1991			10.5	11.3									**4.3**
1992													
1993												26.2	**3.7**
1994												9.3	**0.8**
1995	19.0											3.2	**1.8**
1996	1.2												**0.1**
1997										0.4	6.7	4.4	**1.0**
1998	6.9	50.4	10.5		6.5	4.2				1.6			**6.7**
1999			0.8			2.1			3.3			2.8	**0.8**
2000													**0.0**
2001	14.1										0.4		**1.2**
2002	4.4											6.0	**0.9**
2003						2.5							**0.2**
2004													**0.0**
2005			4.0										**0.3**
2006	14.9					1.7				0.4	0.8		**1.5**

	Ene	Feb	Mar	Abr	May	Jun	Jul	Ago	Sep	Oct	Nov	Dic
Min	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Mean	7.2	5.8	3.7	1.1	0.3	0.5	0.0	0.0	0.2	0.2	0.6	5.6
Max	50.4	50.4	30.2	11.7	6.5	4.2	0.4	0.0	3.3	1.6	6.7	27.0

The mine and plant site area includes the mine, process plant, infrastructure and tailings facility electrical loads. The port facilities include the desalinated water pump station and desalination plant (these are excluded from the electrical load estimate as they are included in the unit cost of the desalinated water), magnetite concentrate filtration plant, concentrate storage and handling, and associated infrastructure.

The total maximum (peak) demand during operations estimated to be approximately 111 MW (excluding the desalinated water system). The estimated average demand during operations will be approximately 85 MW.

The average demand and power consumption for the mine site and the port were calculated 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%.

18.12.1 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 approximately 96 MVA.

The loads at the plant site will include the following;

- The process plant and mine
- The TSF including thickening, distribution and water reclaim
- The permanent camp.

18.12.2 Port Site

The port main substation will have one power transformer (25 MVA (ONAN), 220/23 kV) to support the maximum demand of approximately 14 MVA (excluding the desalination system).

18.12.3 Power Supply

Capstone's mine site and port site will be connected to the Chilean national grid (Sistema Eléctrico Nacional or SEN). The closest connection point to the mine site is at the Diego de Almagro substation located about 9 km from the mine area; the closest connection point to the port is at the Totoralillo substation about 14 km from the port area.

Chilean Electrical System

The Chilean electrical sector is a deregulated market with open, competitive conditions for the generation and sale of electricity, and open access to transmission and distribution. The sector comprises three main activities: generation, transmission and distribution of power. These activities are conducted by private companies; the Chilean state only exercises functions of regulation, supervision and planning for generation and transmission.

The principal electrical grid system in Chile is the SEN, which contributes over 99% of the installed capacity in the country. The SEN covers the territory between Arica and Chiloé, and is the result of a recently completed interconnection between the former Sistema Interconectado del Norte Grande (SING) and the former Sistema Interconectado Central (SIC).

The state agency that participates in the regulation of the electricity sector in Chile is the Comisión Nacional de Energía (CNE) which is responsible for advising the Ministry of Energy on electricity policy, setting regulated distribution charges, developing and coordinating the plans, setting policies and rules necessary for the proper functioning and development of the national energy sector, ensuring compliance, and advising government agencies on all matters related to energy. The Superintendencia de Electricidad y Combustibles (SEC) works with the Ministry of Energy and is responsible for supervising compliance with laws, regulations and technical standards for generation, production, storage, transportation and distribution of liquid fuels, gas and electricity. The interconnected SEN system is administered by the Coordinador Eléctrico Nacional (CEN).

The General Electricity Services Law (DFL-1) established in 1982, and updated and modified in 1999, 2004 and 2005, regulates all aspects of the electricity sector in Chile. Law 20.257, known as the Non-Conventional Renewable Energy (NCRE) Law, established in 2008 targeted 10% of total electricity generation from non-conventional renewable sources by 2024. Law 20.698, known as the 20/25 law, established in 2013 mandated that power producers with more than 200 MW generating capacity should generate 20% of electricity from renewable sources by 2025.

Since 2014, there have been several important changes in the electrical sector that will improve the availability and reliability of electricity supply for the Project, and impact the commercial price of electricity in Chile. These include:

- The interconnection of the SING and SIC grids with the commissioning of a double circuit, 500 kV, HVAC, 1,500 MW, 600 km transmission line. The interconnection became operable in late 2017
- Improvements to the trunk transmission lines in the grid system have been commissioned and these will eliminate transmission bottlenecks in the area where the Project is located. These bottlenecks previously limited supply from capacity available in the south-central SIC system. The system improvements will become fully operable in early 2019
- A new transmission law (Law 20.936) published in 2016 that, among other changes, will require power producers to pay transmission tariffs according to their injections and withdrawals of power. The charges will be paid by customers on a pass-through basis; previously, they were paid 80% by the generators and 20% by the consumers
- A new carbon tax (Green Tax) became effective in 2018, equivalent to \$5.00/t of CO_2 generated by power generators. The charges will be paid by customers on a pass-through basis.

18.12.4 Power Availability

The SEN has installed generating capacity of about 24 GW, composed mainly of thermo-electric and hydro-electric power. The SING and SIC systems generated about 78 TWh electricity in 2016, composed of about 41% coal generation, 25% hydro generation, 16% natural gas, 15% NCRE, and 3% from other sources. Chile is almost entirely dependent on the import of the carbon-based fuels consumed in the thermo-electric power plants (CEN, 2018).

In 2013 Chile generated only 5% of its electricity from renewable energy sources but this increased to 18% in 2018; this trend is expected to continue due to the availability of substantial renewable resources, better grid connections, more efficient technology, successful public policies, increasing strategic focus by conventional power producers towards NCRE and, possibly, improved battery storage capabilities. The government has already approved a number of large-scale solar and wind projects, with about

2,759 MW of new installed NCRE capacity under construction or in an advanced development stage (Moray Energy Consulting, 2018).

Despite the expected increase in renewable sources, carbon-based fuels and hydroelectric sources are expected to continue to provide important base loads and support to the SEN. The carbon-based fuels and hydroelectric sources are forecast to supply 25% to 33% and 21% to 29% of electricity generation, respectively, to the SEN in 2030 (Moray Energy Consulting, 2018).

About 66% of the current installed generating capacity in Chile is owned by four major power companies who are experienced and established locally, and whose capacity is based principally on conventional generating sources. The increase in NCRE requirements has resulted in experienced renewable energy companies entering the market to provide new NCRE capacity.

Chile currently has one cross-border connection, which connects into the SEN from Argentina. However, the Andean countries are working to create an inter-connected regional electricity market. This could include an interconnection to Peru with the potential to increase capacity and supply of competitive electricity into the SEN.

In the past the SIC system has experienced rationing and supply bottlenecks, especially in the northern sector, and the SING has been subject to periodic outages triggered by generator failures or occasionally by extreme weather conditions (typically these last several hours). The new SING–SIC interconnection and improvements to the trunk transmission lines are expected to improve supply reliability.

The Project will require 109 MW peak power with an expected initial supply date in 2022. There are no concerns that the Chilean national grid system can provide a firm and continuous supply for the Project's electricity requirements.

In Chile, only generation and distribution companies may commercialize electricity. Wholesale competition in Chile occurs in the 'contract market', in which generators sell electricity freely to large (non-regulated) customers, and through tenders to distributors that supply small (regulated) customers. Capstone is considered a non-regulated customer and is not eligible to access the national electrical grid directly. Capstone must arrange its power supply through self-generation or from a generation company. The Project will receive electricity at points of delivery connected to the national grid system; alternatives for the points of supply are under review.

18.12.5 Electricity Prices

Since 2014, there has been a significant reduction in the price of electricity in the national grid system. The downward trend in electricity price has been driven by lower coal and natural gas prices delivered into Chile, increased supply from new conventional generating capacity and from new NCRE generating capacity, lower NCRE prices (which have resulted from technology improvements that have lowered investment costs and reduced capacity charges), and lower than expected demand from the mining industry (due to delay in the development of projects due to depressed copper prices).

The extent of the price reduction is illustrated by the electricity charged to distribution companies by generators. In the SIC prices fell from $146.60/MWh in 2014 to $59.20/MWh in 2017; in the SING prices fell from $86.70/MWh in 2014 to $51.40/MWh in 2017. Energy auctions which have features that allow independent tenders of separate hourly blocks, have resulted in 75% lower prices in 2017 than in 2013, with an average price of $32.50/MWh (Energy Transition, 2018).

In the regional electrical grid systems, the instantaneous marginal cost of electricity in the SIC system has fallen from $131/MWh in 2014 to $52/MWh in 2017; in the SING system, the instantaneous marginal cost has fallen from $76/MWh in 2014 to $55/MWh in 2017 (CEN, 2018).

The major energy companies have responded to the changing market for electricity in Chile and are developing new strategies with a strong focus on providing combined conventional-renewable or all-renewable contractual solutions for non-regulated customers. As a result of recent transmission and carbon legislation, future power purchase agreements (PPAs) will include the power producer's electricity charges, and traditional and new system charges set by the regulators and passed through to the customers.

18.12.6 Power Supply Strategy

Capstone plans to sign a PPA with an experienced, independent power producer with existing power generating capacity in the national electrical grid system. This is consistent with the strategy that major mining companies have used to procure long-term supply of electricity in Chile. Capstone has held discussions with major energy companies currently operating in Chile to confirm interest and capacity to supply electricity to the Project, and to verify commercial assumptions for the economic analyses.

Capstone will contract for a firm and continuous supply of electricity on a long-term basis. The company will have the option to contract electricity supply from conventional generating sources, renewable generating sources, or a combination of both. Electricity will be delivered through the national grid system to points of delivery at or near the planned mine and port sites.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Market Capabilities

Capstone currently markets copper concentrate from its mining operations, and has established a reputation as a reliable supplier. Capstone has existing commercial sales contracts and operates under current commercial terms. In addition to the current Asian markets used by Capstone, smelters in Chile are available for sales contracts.

Capstone staff and experts retained by Capstone provided information related to the metal pricing for copper, treatment and refining charges, and iron ore concentrate (62% Fe content sinter fines).

In accordance with the CIM 2015 Guidance for Mineral Reserve and Resource Pricing, Capstone maintains and publishes a Strategic Resource Planning Guidance; this was provided to the QPs

Capstone's Marketing Group is experienced and capable of establishing and implementing a marketing strategy that will provide offtake agreements with favourable NSRs and benefit all stakeholders.

19.2 Copper Concentrate Market

Beginning in 2000. and continuing through to 2017, China became an increasingly important factor in the demand for raw material feed in the form of copper concentrate and as a consumer of the refined metal. Copper concentrate imports to China have gone from practically zero in 2000 to 17.35 Mt in 2017. China now consumes 48% of the world's refined copper. Chinese smelting/refining capacity continues to increase with 1.1 Mt of refined metal capacity to be commissioned over the next one to two years. This alone will increase the demand for copper concentrate by 4 Mt annually. Overall, global market forecasts are for a compound annual growth rate (CAGR) of 2% or higher.

Refined copper is a key determinant in the growth of developed and developing nations. Copper smelters are located in various regions; however, the majority are in Asia. All the smelters are accessible from Chile, but logistic costs are a significant factor in determining which smelter location will result in the best long-term offtake arrangements, hence providing the best returns. Secondary considerations are the possibility of linking offtake agreements to financing arrangements and strategic diversification.

Potential smelter counter parties for direct sales contracts include:

- Domestic:
  - Chilean smelters (Codelco, Potrerillos; Enami Fundición Hernán Videla Lira (formerly Paipote))
- Export:

  South America

  - Brazil (Caraiba)

  Europe

  - Germany, Sweden, Bulgaria, Spain (Aurubis, Hamburg; Aurubis, Pirdop; Boliden, Ronnskar; Freeport, Huelva)

  Asia

  - China Various (Jiangxi, Tongling, Jinlong, Daye, XGC, Jinchuan, Yunnan)
  - Japan (Saganoseki, Naoshima, Onahama, Niihama, Hibi)
  - Korea (LS Nikko)
  - Philippines (PASAR)
  - India (Hindalco, Birla; Vedanta, Sterlite; Adani, Gujarat)

Geographic diversification provides some risk reduction in a marketing strategy, although other factors such as credit risk and performance risk must also be considered. It is expected that the emphasis for direct sales agreements will be placed on Asian smelters unless finance-linked contracts are available.

19.2.1 Supply

Current mine supply is considered inadequate to meet demand over the foreseeable future due to falling ore grades at maturing and depleting mines and not enough capital being expended to fill the gap. Capital expenditures for new copper mines and expansions to existing mines has dropped 60% globally over the last five years. The lack of spending relative to projected future demand has occurred primarily because copper prices have been too low relative to project hurdle rates in an environment of rising risks (water supply, more stringent environmental regulations, energy costs, political stability and resource nationalism). Current mine supply is set to decline by almost 5 Mt through 2030 even when including projects under construction.

It is estimated there are 250 projects globally with nearly 17 Mt of potential output. Of these, it is estimated that only about 6 Mt of the potential supply is viable; only just offsetting the decline in baseline copper supply over the period to 2030 with additional production of about 1 Mt.

Copper prices need to rise to $7,500/t ($3.40/lb) in today's dollars over the long-term ($8,280/t or $3.75/lb in 2023 dollars) in order to incentivize enough copper supply.

Global mine production is set to increase beyond 2018, but not sufficiently to meet the expected demand. New projects (Cobre de Panama, Michiquillay, Quellaveco, Spence, Quebrada Blanca and Kamoa–Kakula) will address some of the future demand. However, the copper price has not reached a level needed ($3.40/lb) to provide companies with the confidence to invest in additional mega-projects. The economics for smaller mines with lower capital costs can be quite different from the economics for the large mines. In the meantime, established and maturing mines are facing declining production due to falling head grades. In major producing areas, potential hurdles for new projects or brownfield expansions are water supply, more stringent environmental regulations, political stability, and resource nationalism.

19.2.2 Demand

China remains the key contributor to the forecast growth in copper consumption through 2030. Global industrial production (IP) growth is forecast to be 2.5% per year through 2030. This suggests a 1.5% per year growth in copper demand through the same period (China accounts for approximately 40% of this growth). At a 1.5% copper growth rate, copper demand is expected to grow by 4.5 Mt through 2030.

Renewable energy infrastructure is the single biggest driver of global copper demand growth over the coming years. The need to connect significant numbers of small-scale electricity generation units into the grid provides a major boost to copper (solar generation capacity is set to triple and wind generation capacity is set to double by 2025). The shift to areas such as offshore generation will further increase the demand for copper.

Growth from the electric vehicle segment (cars and charging infrastructure) is small, but not insignificant. Growth in electric vehicle production has exceeded previous expectations with China accounting for much of the growth but the western world manufacturers are gearing up for significant production post-2020. The growth in electric vehicle production is expected to increase copper demand by 1.3 Mt/a by 2025.

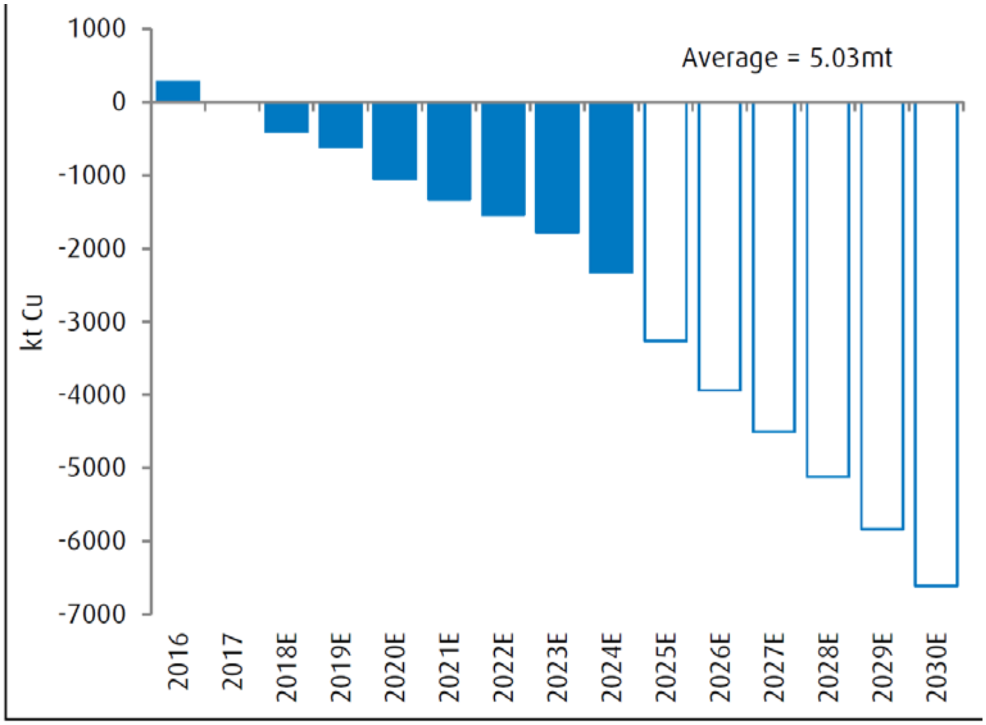
19.2.3 Supply/Demand Gap

A shortage of copper is inevitable over the period to 2030. Potential mine expansions will only cover the loss in production from maturing mines.

Growth in demand could subside for a period of time but it is considered unlikely even with the current tariff/trade disputes initiated by the current USA government. Substitution could occur to some degree, e.g. aluminum for medium-voltage wiring.

The supply/demand gap will naturally rise over time. Over the period to 2030, an average annual supply gap of just over 5 Mt is predicted. Given that deficits of this level do not (and cannot) occur in commodity markets, prices will rise to incentivize projects to fill the gap. Figure 19-1 shows the estimated copper market balance to the year 2030.

Figure 19-1: Copper Market Balance Estimate to 2030



Source: ICSG, BMO Capital Markets

19.2.4 Price Projections

It is difficult to project prices out to 2030. A 14% drop in the copper price in the June-July 2018 period was not anticipated by the analysts. Long-term price forecasts must consider the following:

- The need for miners to make a return in order to invest to replace their base production; this requires price growth to outpace the marginal cost growth
- A draw down in inventories
- The need for prices to be high enough over the long term to fill the 2025 to 2030 gap.

The global copper supply/demand forecast and price forecast from 2018 to 2030 is provided in Table 19-1.

Mine concentrate production will continue to increase based on analysis of mines currently under construction (e.g. Cobre de Panama), projects recently announced (Quebrada Blanca 2) and projects in the planning stages. Combined with announced production forecasts of existing mines, total copper concentrate supply is calculated. Solvent extraction/electrowin (SX/EW) copper production is added to determine total copper units available for the market. Historically, production forecasts have an error of approximately 5% due to unexpected and unplanned events such as labour strikes, natural disasters, pit wall failures or milling equipment failures.

On the consumption side, smelter production is forecast based on historical operating rates. The difference between the supply and the demand provides a view of the market and whether it is in surplus or deficit. The forecasts in Table 19-1 indicate a relatively balanced market through 2022 with small annual deficits. However, the impact of the small deficits over this period will be a reduction in global stocks. Historically, as global stocks approach two weeks consumption levels, prices tend to rise. It is forecast that the incentive price of $3.40/lb will materialize 2021, after which additional projects will be announced but these will take some years to permit and build.

19.2.5 Treatment and Refining Charges

Treatment and refining charges reflect the general status of the supply/demand balance of copper concentrate. Traditionally, the annual benchmark terms have been the guideline for individual producers to establish terms with their smelting counter parties. This process is breaking down and more producers are aligning with counter parties to establish distinct terms on the basis of agreed value propositions.

For smelters there is generally a preference for terms fixed for a period of a year to provide stability in their income stream. Many miners are willing to establish some or all of their terms on this basis. However, this practice does not provide flexibility to pursue terms where more value can be added through value propositions. Thus, it is difficult to predict TC/RCs on a longer-term basis.

Table 19-1: Global Copper Supply/Demand and Price Forecast to 2030

kt	2017	2018f	2019f	2020f	2021f	2022f	2023f	2024f	2025f	2026f	2027f	2028f	2029f	2030f
Concentrate Mine Production	16,289	16,501	16,839	17,354	17,817	18,106	18,381	19,197	19,600	19,997	20,281	20,362	20,134	19,155
SxEw Mine Production	3,738	3,667	3,700	3,641	3,513	3,467	3,396	3,204	2,874	2,567	2,421	2,366	2,337	2,309
Mine Production	**20,027**	**20,167**	**20,540**	**20,996**	**21,329**	**21,572**	**21,778**	**22,401**	**22,474**	**22,564**	**22,702**	**22,728**	**22,471**	**21,464**
% Change	-0.7%	0.7%	1.8%	2.2%	1.6%	1.1%	1.0%	2.9%	0.3%	0.4%	0.6%	0.1%	-1.1%	-4.5%
Including Total Disr. Allowance (t)		840	1,199	1,595	1,651	1,735	1,821	2,007	2,142	2,277	2,416	2,479	2,512	2,446
Including Total Disr. Allowance (%)		4.0%	5.5%	7.1%	7.2%	7.4%	7.7%	8.2%	8.7%	9.2%	9.6%	9.8%	10.1%	10.2%
Refined Production	**23,061**	**23,487**	**23,851**	**24,237**	**24,664**	**25,005**	**25,298**	**25,994**	**26,151**	**26,448**	**26,821**	**27,016**	**26,671**	**25,775**
% Change	1.4%	1.8%	1.6%	1.6%	1.8%	1.4%	1.2%	2.8%	0.6%	1.1%	1.4%	0.7%	-1.3%	-3.4%
Refined Consumption	**23,027**	**23,508**	**23,920**	**24,302**	**24,690**	**25,061**	**25,434**	**25,806**	**26,178**	**26,553**	**26,925**	**27,290**	**27,651**	**28,015**
% Change	2.2%	2.1%	1.8%	1.6%	1.6%	1.5%	1.5%	1.5%	1.4%	1.4%	1.4%	1.4%	1.3%	1.3%
Surplus/Deficit	**34**	**-21**	**-68**	**-64**	**-27**	**-57**	**-135**	**189**	**-27**	**-105**	**-104**	**-273**	**-981**	**-2240**
Trackable Stocks (incl bonded)	1,093	1,072	1,004	940	913	856	721	909	883	778	675	401	--	--
Stock: Consumption Ratio (wks)	2.5	2.4	2.2	2.0	1.9	1.8	1.5	1.8	1.8	1.5	1.3	0.8	--	--
Price (US$/lb)*	2.8	3.1	3.1	3.2	3.4	3.6	3.8	*3.8*	*3.9*	*4.0*	*4.1*	*4.1*	*4.2*	*4.3*
Price (US$/t)*	**6,172**	**6,785**	**6,800**	**7,000**	**7,500**	**8,000**	**8,281**	***8,446***	***8,615***	***8,787***	***8,963***	***9,142***	***9,325***	***9,512***

Source: Citi Research, *Nominal prices post 2022 are our LT price of $7,500/t, at 2%pa inflation.

Note: f = forecast

Currently, the average Chinese copper smelter requires a TC/RC of $65/6.5 cents to break even. By-product credits and currency exchange rates also affect the profitability.

Given that the analysis of concentrate supply/demand shows a shortage of concentrate in the future, it is reasonable to forecast lower TC/RCs. However, moving from a relatively balanced market in 2017 and 2018, where benchmark TC/RCs were agreed at $82.25/8.225 cents, 2019 benchmark terms have been agreed at $80.80/8.08 cents, recognizing that the concentrate is expected to move to a deficit. Potentially, a deficit of only 2 Mt could push TC/RCs well below the smelter breakeven point. If the deficit continues for an extended period of time, the result could be the closure of some higher cost smelters which would lead to even higher copper prices and also higher TC/RCs as the market seeks an equilibrium state.

Longer-term TC/RCs of $75/7.5 cents are considered reasonable given that this provides smelters with an operating margin and still reflects a market that will be in deficit for a period of time.

19.3 Copper Concentrate

19.3.1 Santo Domingo Project Likely Product Specifications

For the purposes of assessing the marketability of the copper concentrates, Capstone supplied the analysis of the copper concentrate specifications in Table 19-2. The specifications are from the 2011 Pre-feasibility Study metallurgical testwork and are not smelter specifications.

19.3.2 Deleterious Elements and Penalties in Copper Concentrates

China has strictly controlled the import 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.

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.

Given the expected qualitative analysis for the concentrate to be produced, the Santo Domingo concentrate would be considered a "premium" concentrate in the international smelter market. The concentrate is expected to have a higher than average copper content (30%) with no appreciable deleterious elements. 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.

The copper concentrate market is a global market. A large secondary market exists in the form of commodity traders (in excess of 17 Mt copper concentrate annually). Commodity traders can provide better net terms than smelters. Additionally, commodity traders provide more flexibility as they can deliver to numerous locations.

Table 19-2: Copper Ore Concentrate Specification

Chemical Element	Unit	Value
Cu	%	30.3
Fe	%	29.4
S	%	30.9
Au	g/t	3.4
Ag	g/t	26.6
Hg	g/t	4.6
Cl	g/t	292
F	%	0.013
SiO_2	%	1.89
As	g/t	67
Bi	g/t	<40
Cd	g/t	<10
Co	g/t	416
Cr	g/t	102
Pb	g/t	198
Mn	g/t	312
Ni	g/t	79
Sb	g/t	195
Se	g/t	195
Sn	g/t	36
Zn	g/t	<20
P80 µm	—	44.4

Note: The specifications are from the 2011 Pre-feasibility Study metallurgical testwork and are not smelter-derived specifications

It is anticipated the marketing strategy for Santo Domingo would include sales to both markets. Over the last four years Capstone's Marketing group has engaged with numerous parties from both markets and received expressions of interest if the Project proceed.

Because of the expected 'clean' composition of the Santo Domingo concentrate, Capstone considers that concentrate from the Project will be in high demand from trading companies specializing in blending complex materials with clean materials. This high-quality concentrate is highly sought after by smelters and traders.

19.3.3 Marketing Strategy

Copper concentrates can be sold under a number of different agreements:

- Long-term offtake agreements or frame contracts
- Mid-term agreements or mid-terms
- Evergreen contracts
- Spot contracts
- Trader offtake 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 assayed quantity (after final assays are agreed), 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 the average period.

For precious metal payments there are two different methods commonly 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 per dmt would not be payable. In higher silver content concentrates there is often a deduction of 50 g per dmt instead of 30 g per dmt. Gold is payable on the full and final assayed quantity of gold less a deduction of 1 g per dmt. 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 for concentrate delivered from overseas 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 $15 to $20 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 (e.g. in Peru, Mexico, the Philippines and Bulgaria) continuing production, despite very high penalties for the arsenic content of the concentrates produced relative to the clean concentrate market.

19.3.4 Project Concentrate Marketing Assessment

The Project copper concentrate will have a low gold content (around 3 g per dmt) and a low silver content (around 27 g per dmt). As a result, there will be considerable value to pricing the material on an Asian-style basis as opposed to a European-style pricing.

This will be accentuated when, occasionally, the silver content rises above 30 g per dmt. If this percentage is payable using European terms the payment will be very low, but with Asian terms over 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. Factors such as freight, assay bias, geographic location and contractual party reliability must be considered. The normal contract split for mines of the proposed size of Santo Domingo are:

- 60% to 70% on long-term frame contracts with four or five major smelters
- 10% to 20% to traders on three year to five-year fixed TC/RCs or TC/RCs 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 high copper concentrate production in the first years of operation and the gradual decline in copper concentrate production thereafter. Consideration must also be given to the terms and timing of the contract renewals so that renewals do not all occur at the same time.

The timing to secure sales contracts is dependent on the progress of 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 initially, followed by full long-term contracts as a condition of the completion of the Project financing.

19.3.5 Logistics

It is planned to ship the copper concentrate from a northern Chilean port. However, due to the low monthly shipment tonnages (approximately 30,000 t to 50,000 t per month) and storage facilities (40,000 wmt storage) most of the shipments will be made using Handymax vessels (30,000 wmt to 45,000 wmt carrying capacity) or Supramax vessels (50,000 wmt to 55,000 wmt).

Ocean freight rates are primarily driven by two factors; energy (fuel) costs and the supply/demand of vessels. It is almost certain that fuel rates for vessels will rise post-

2019 due to the International Maritime Organization (IMO) requirement that vessels burn low sulphur fuel or install scrubbers to process the engine exhaust.

For the period January 2016 to January 2018 time charter rates for Supramax vessels (the primary type of vessel used in transporting copper concentrate) have risen almost 60% from $7,000 per day to more than $11,000 per day.

Currently there is an adequate supply of vessels and new builds are effectively replacing scrapped vessels. The dry bulk market is expanding at about 3% per year. However, currently order books are thin as current freight rates do not provide an incentive for owners to expand their fleets beyond replacement. If this scenario continues, an annual increase of 5% in freight rates is expected. An escalated price of $40/dmt was used in the financial analysis in this Report, anticipating price escalation and the uncertainty of the new sulfur regulations.

19.4 Iron Concentrate

The Project will produce magnetite concentrate fines with high iron content (Fe >65%) for shipment overseas in Cape-sized vessels to iron and steel makers.

Capstone contracted CRU in 2014 to supply a report on marketability and price projections for iron ore concentrate (62% Fe content sinter fines). Capstone also contracted CTAG in 2014 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.

In order to supplement the previous information on the international market for the Santo Domingo iron ore concentrate, Capstone contracted the following:

- Mr. David Trotter, global iron ore and commodity marketing consultant, who prepared a study entitled “Pellet Feed Market Characterization and Forward Pricing Outlook” for Capstone, dated September 2018.
- Braemar, who prepared a study entitled “Very Large Ore Carrier (VLOC) Review”, for Capstone, dated 24 October 2018.

The Trotter (2018) forward pricing report prepared for Capstone estimated that prices for 62% Fe content sinter fines (Platts Iron Ore Index or IODEX) cost-and-freight (CFR)

Qingdao delivery (deemed the standard product for CFR China delivery) can be expected to be in the range of $62/dmt to $72/dmt over the next 10 years. This study is based on a long-term price of $69/dmt for 62% Fe concentrate. Premiums for 65% Fe concentrate ($24/dmt), value-in-use (VIU) for 66% Fe ($1.50/dmt), magnetite content ($2.50/dmt) and low alumina ($7/dmt for each 1% below 2.5%) are expected to remain relatively stable because of the direct impact on furnace productivity and decrease in emissions. This study discounted the current premiums to approximately 80%.

Braemar conducted a long-term estimate of shipping costs to include new construction and new environmental regulations on sulfide emissions. Long-term contracted prices are expected to drop from the current spot market price of $20/dmt to below $15/dmt. This study has assumed a long-term shipping cost of $20/dmt. The net result is a base case price of $80/dmt free-on-board (FOB) Chile.

19.4.1 Iron Ore Market

Iron ore is globally traded with hematite (Fe_2O_3) and magnetite (Fe_3O_4) ores making up the vast majority of the world seaborne trade with most of the supply coming from South America and Australia.

Iron ore is used either in a blast furnace or in a direct reduction furnace to make metallic iron for use in steelmaking. Globally, and in the key import markets, over 75% of steelmaking plants use hot iron from a blast furnace. The input of iron into the blast furnaces can be in the form of pellets (agglomerated fine pellet feed from a pellet plant), lump (naturally occurring agglomeration) and sinter fines (agglomerated coarser iron ore fines in a sinter plant).

Steel production dictates the demand for iron ore. Global steel production in 2017 was 1.69 billion tonnes (Trotter, 2018). In 2018 global steel production is forecast to be 1.75 Bt. Seaborne trade of iron ore in 2017 was 1.47 Bt. The iron ore seaborne trade is forecast to grow in excess of 2% annually. Globally, basic oxygen furnace (BOF) steel production is growing at a rate of just under 1% per year and electric arc furnace (EAF) production is growing at a rate of 4.4% per year (Trotter, 2018).

China dominates the global production of steel and thus the consumption of iron ore. China produced 831 Mt of steel in 2017. Other top producers in 2017 were Japan at 105 Mt, India at 101 Mt, and South Korea at 71 Mt.

Asian steel producers face increasingly stringent environmental regulations to limit emissions. This is most evident in China. Despite the limited growth in steel demand in China, the demand for pellet feed is expected to be strong and the dominant growth segment in the iron ore market. This is being driven by the changes in requirements in China which include:

- Environmental restrictions on new blast furnace construction
- Environmental restrictions on new sinter plant capacity
- Improving efficiencies in blast furnaces to 90%
- Lack of high-grade domestic pellet feed as average grades decline
- New large-scale pellet plants in China
- Higher margins for high quality steel
- Increasing use of pellet feed in sintering
- Changing individual customer demands and technical limits.

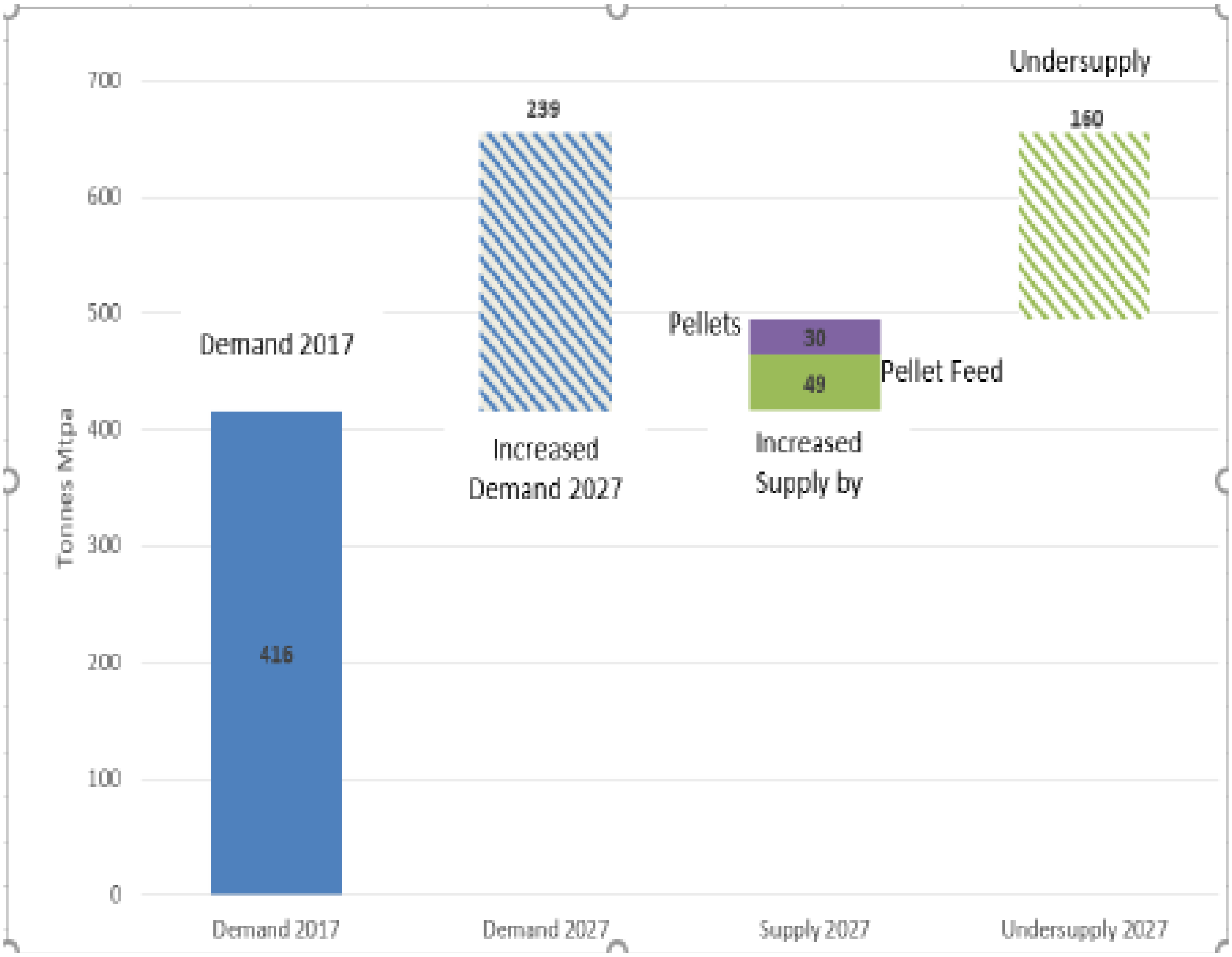
These factors are inter-related and are expected to raise steel making efficiencies from 82% in 2017 to finally around 90% by 2026.

Pellet consumption in blast furnaces has risen from 250 kg/t hot metal in 2014 to current level of 285 kg/t of hot metal and is forecast to be in excess of 300 kg/t of hot metal by 2022. Global demand for iron ore pellet feed is expected to increase from 416 Mtpa to 655 Mtpa in 2027 (Trotter, 2018).

19.4.2 Pellet and Pellet Feed Supply and Demand Summary

The pellet and pellet feed supply and demand summary is shown in Figure 19-2.

Figure 19-2: Pellet Supply/Demand Summary (2017–2027)



Note: Figure prepared by D. Trotter, 2018

19.4.3 Pricing of High-Grade Iron Ore Fines (>65% Fe)

The price of iron ore is widely accepted as the 62% Iron Ore Index as reported by price reporting companies in $ per dmt. In 2016 the price range variability was 112% and in 2017 the price range variability was 78%. To July 2018 the price range variability has been only 26%; this represents a decrease in volatility (Trotter, 2018).

In 2018 at the same time as reduced variability, the price of higher-grade iron ores (as represented by the Platts and Metal Bulletin 65 Iron Ore Index) has increased relative to the 62 Iron Ore index with a large price differential of $28 per dmt. This premium is expected to persist for higher-grade material. This premium is large relative to the base price of iron ore and higher than the incremental correction for iron content. Currently, the differential is more than 30% above the simple iron value correction. The premiums and discounts for 58–62% Fe and for 65–62% Fe are shown in Figure 19-3.

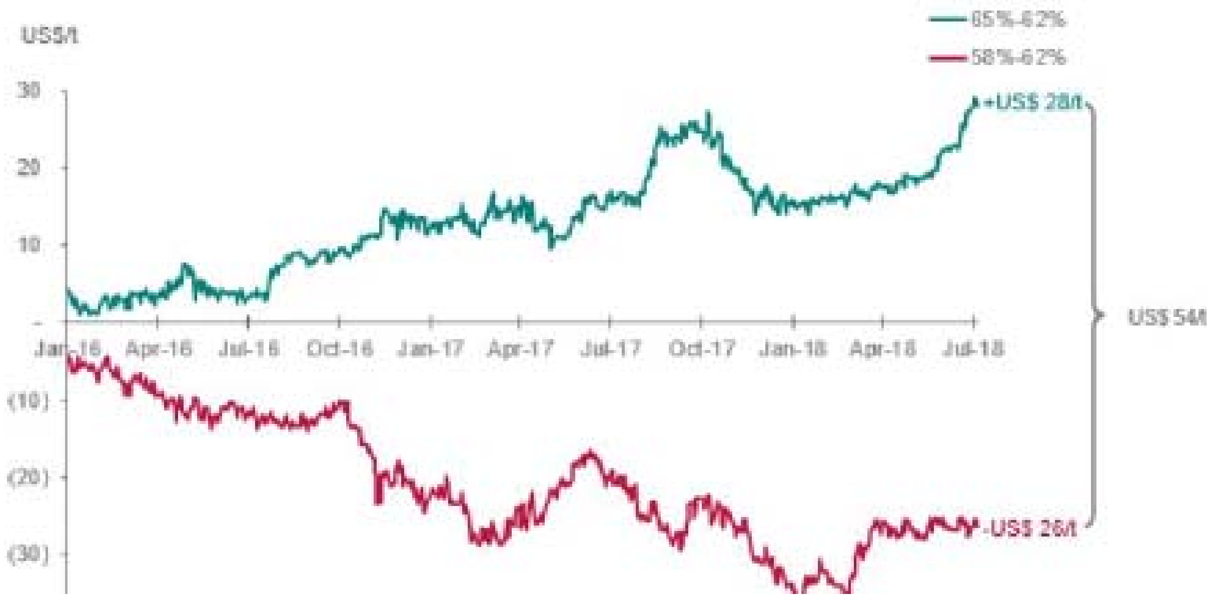
The premium should be considered a major part of any price forecast for iron ore products including pellet feeds. This has been mirrored to an extent by the low-grade ore index; the low-grade ores are under reduced demand and are increasingly being blended with high-grade ores.

In order to forecast the value of this premium over the next 10 years, assuming that the major factors remain, the other drivers will be coking coal cost (the other major steelmaking cost) and steel margins.

Using iron-making models it is possible to predict the scenarios which would affect the premium. Assuming a steel margin in the range $30/t to $100/t and a derived coke price between $170/t and $260/t the scenarios show a long-term premium between $19 and $33. This corresponds to a midpoint of $26/t which can be used as a base for forecasting for the remaining 2018 and 2019. This is a large percentage of the 65% Iron Ore Index price component which is expected to remain disconnected from 62% Iron Ore Index for the period 2020 to 2027. Table 19-3 shows the price forecast for Capstone pellet feed CFR China for 2018 to 2017.

The limits on alumina also help to drive up the premium for ores above 65% Fe because by mass balance the total oxides should be less than 6.0%. Three of the top four iron ore suppliers (BHP, Rio Tinto, FMG) have increased alumina from an average below 2% to a combined average of 2.35% in 2018 whilst maintaining an average silica content of 4.5%. These suppliers account for 650 Mt/a of the iron ore supply and 68% of the supply to China. Alumina levels are expected to increase with the depletion of the Yandi Mines (dropping from 150 Mt/a to zero over the next five years). The market is expected to be alumina-constrained for several years and, given the small impact that low alumina pellet feed will have on a year on year basis, this may continue.

Figure 19-3: Premiums and Discounts for Fe Content



Note: Figure from Metal Bulletin, 2018

Capstone pellet feed would attract the high iron grade premium and a further premium if the alumina content is less than 1% (currently at $7 per 1% of alumina below 2.5%). Further, if Capstone pellet feed maintains a silica content below 5.0% then it would not attract silica penalties in an alumina-constrained market.

19.4.4 Logistics

Because iron ore is a relatively low-value commodity, it is imperative that logistics costs are minimized to protect margins. Typically, this means utilizing very large ore carriers (VLOC) vessels. Based on the analysis undertaken by Braemar, the new class of VLOC (Guaibamax) (325,000 dwt), of which 38 are on order in South Korea shipyards, would be the most suitable and cost-effective vessel for shipping to Eastern Asia. The 400,000 dwt Valemax vessels would also be effective. This does not preclude the use of smaller vessels in the 250,000 dwt to 300,000 dwt range.

Table 19-3: Price Forecast for Capstone Pellet Feed

	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027
	(actual)										
IODEX 62% Fe, CFR Qingdao China ★	71.31	67.00	66.00	65.00	62.00	64.00	65.00	66.00	68.00	70.00	72.00
+Differential 65% Fe CFR North China	15.20	24.50	26.00	25.61	24.42	25.21	25.61	26.00	26.79	27.58	28.36
+Fe VIU for 66.06 vrs 65%@$1.35/1% Fe	1.43	1.43	1.41	1.41	1.36	1.48	1.45	1.45	1.47	1.47	1.52
-1% Yield Adjustment for 80%<0.45um	-0.88	-0.93	-0.93	-0.92	-0.88	-0.91	-0.92	-0.93	-0.96	-0.99	-1.02
+Magnetite Premium	1.52	2.45	2.60	2.56	2.44	2.52	2.56	2.60	2.68	2.76	2.84
Pricing Capstone Pellet Feed CFR China	**88.58**	**94.45**	**95.08**	**93.66**	**89.35**	**92.30**	**93.70**	**95.12**	**97.98**	**100.82**	**103.70**

Note: First row of table, marked with a blue star, sourced from Wood Mackenzie, June 2018. Remaining table information is from Trotter (2018). The table has a typographic error in the row entitled "-1% Yield Adjustment for 80%<0.45 µm", where the figure 0.45 µm should read 45 µm.

In determining a freight strategy, the following factors need to be considered:

- Scheduling of shipments and certainty of commodity production
- Shipment size and discharge port flexibility
- Freight pricing mechanisms, including fixed and index linked
- Charterer's risk appetite
- Outlook for freight rates
- Owners hedging positions
- Counter party risk.

18.4.5 Iron Ore Freight Rates

Over the last 10 years the market has seen extremes for freight rates for Cape size vessels. Brazil to China rates peaked in May 2008 with charter rates equivalent to $109/wmt and in January 2016 the rate for the same route was $5.33/wmt.

Given current bunker rates, port charges, vessel values and 60 kt/d load rate with 50 kt/d discharge rate, a projected freight rate of $12.76/wmt was determined for a Chile–China routing. Allowing for inflation and other possible changes in operating costs a rate of $20/dmt has been considered conservative and appropriate for the feasibility evaluation.

19.4.6 Santo Domingo Iron Ore Concentrate - Specifications

For the purposes of assessing the marketability of the iron ore concentrate, Capstone expects to produce an iron ore concentrate with the specifications shown in Table 19-4. The specifications given in the table are from the 2011 Pre-feasibility Study metallurgical testwork and are not smelter specifications.

The iron ore concentrate that will be produced by the Project is a typical pellet feed currently in use in pellet plants. Magnetite is the predominant mineral. The iron grade is high (Fe >65%) and the low alumina (Al_2O_3) and low phosphorus (P) make the concentrate suitable for most pellet plants. The suitability and demand for this pellet feed should be considered in the context of increasing use of pellets in iron making, the increased use of higher-grade ores generally, and as a premium additive to sinter plants by blending.

19.4.7 Deleterious Elements and Penalties in Iron Concentrates

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 SiO_2 <3.5%, though in pellet plants this may be as high as 5.5%; Al_2O_3 <1%; Mn <0.5%; P <0.1%; S <0.1%; Cu <0.01%; and a combined Na_2O and K_2O <0.5%. The Al_2O_3 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 copper can often be blended out in the charge feed mix since it 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.

Table 19-4: Iron Ore Concentrate Specification

Chemical Element	Unit	Value
Fe_{tot}	%	66.06
FeO	%	23.08
SiO_2	%	4.10
Al_2O_3	%	1.00
CaO	%	0.57
MgO	%	0.455
P	%	0.011
S	%	0.020
Cl	ppm	60
Na_2O	%	0.145
K_2O	%	0.105
Mn	%	0.069
Cu	%	0.0081
L.O.I	%	1,34
>40 µm	%	21,3
Blaine	cm^2/g	1,896

Note: The specifications are from the 2011 Pre-feasibility Study metallurgical testwork and are not smelter-derived specifications.

The iron ore concentrate specifications and the list of impurities are used by steelmakers to calculate a value in use for each blast furnace, and in turn the value of the pellet feed. The main levels of impurities expected in the Project iron ore concentrate are silica and copper. Copper is expected to be 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 $1.50/t to $2.00/t per each 1% above 3.5%.

China is a silica-impaired destination, as the local ores often have very high levels of silica. Therefore, blending down in China is not readily possible. Some mills will reject material above 6% SiO_2, others will reject at higher percentages; there is a very large variance in silica tolerance. Capstone will need to be selective in finding steel mills with the right fit for the iron ore concentrate from the Project. Iron ore in China is normally traded and priced on the basis of 62% Fe content, with premiums paid for higher iron content and discounts for lower iron content.

19.5 Contracts

Kores has the right to purchase up to 50% of the annual production of copper concentrate and iron ore concentrate, leaving Capstone to market and sell the remaining concentrate. The Kores terms and conditions will reflect the Capstone terms negotiated independently in the market.

No contracts are currently in place for Capstone's production for either the copper or iron ore concentrates.

No other contracts are in place for the Project.

19.6 Comments on Section 19

In the opinion of the QP, the marketing studies support that there is potential for the sale of the copper and iron concentrates from the Project as follows:

- The Santo Domingo copper concentrate would generally be considered clean. Chlorine and fluorine are safely under the limits, and if they are occasionally over the limit it is likely that only a nominal penalty would apply. For trading companies specialising in blending various complex copper concentrates a clean concentrate such as that from Santo Domingo would be in high demand
- 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
- Kores has the right to purchase up to 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 part of the production for either the copper or iron ore concentrates
- No other contracts are in place for the Project.

The QP is of the opinion that the marketing studies and metal price forecasts are acceptable for use in the financial analysis in Section 22.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Baseline Studies

Baseline studies were conducted between 2007 and 2013 for the environmental characterization to support the Environmental Impact Assessment (EIA). The following summarizes the main results registered for the environmental components.

20.1.1 Air Quality

Air quality measurements were taken from six monitoring stations; in the proposed mine site area, along the pipeline route, 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, SO_2, NO, NO_x and NO_2). 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 the proposed mine site. This was attributed to the urban activity around Diego de Almagro, the presence of vacant properties with active erosion, the presence of environmental polluters, mining activity, and its geographic position in a desert area. In contrast, the mine site 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 pipelines runs through rural areas (with the exception of El Salado), and the main source of man-made 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 limits for 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 pipeline construction
- Restriction of noise-generating activities at night during construction of the pipelines.

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 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 one 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

Six hydrogeological units were identified and grouped into three macro units. Aquifer systems are mainly associated with hydrogeological units linked to alluvial deposits (UH-1), which are primarily constrained within existing and paleo stream channels. The Atacama gravels (UH-2), which are dominated by cemented sandy-gravels or cemented sandy-clays, are inferred to act as an aquitard. The presence of neotectonic faults may serve as conduits of ground water flow from the shallow alluvial deposits (UH-1) to the basement rocks (UH-3) underlying the cemented units of UH-2. There is some evidence to suggest that UH-3 may contain ground water within a fractured bedrock system; however, the yield from such systems is estimated to be low.

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 ground water in the period from August 2007 to April 2013. Water levels were found to vary from approximately 1,041 masl to 772 masl at each well. Depth to ground water across the study area ranged from a few metres at lower elevations to as much as 120 m at higher elevations.

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 (2.1%). 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 (2.5%).

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 list of protected species is reviewed annually by the authorities and the categorization can change.

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 trees, 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 tides 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. Trace metals (zinc, lead, and copper) recorded in some areas showed levels that exceed international guidelines for the protection of the biota. However, overall the concentration of all elements indicates 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 consist 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, and 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. A number of sites were classified as indeterminate, 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, one site had evidence of carving, one site had ceramic remains, and a fourth site had a cave painting. A large percentage of the sites are in a good state of conservation, because they are far from main and secondary roads, and the inhabitants of the area take care of them (as can be seen from the animitas).

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 identified, ranging from 1 to 16 taxa in the Sierra Santo Domingo.

Outcrops with Quaternary-aged fossils were identified along the planned pipeline route near the coast, with numerous taxa of marine invertebrates, including gastropods, bivalves, and barnacles 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 ground water systems in the area.

The potential impacts on water resources are:

- Alteration of the flow and drainage patterns of surface water
- Ground water flow alteration
- Ground water quality alteration.

No impacts to water resources are anticipated, because the Project will use desalinated sea water for the mining process. Infiltration to ground water is expected to be minimal or zero from WRFs and from the thickened TSF. Contact water generation will be mitigated through engineering design.

During the operations phase it is expected that impacts on water resources will be caused by the following activities:

- Exploitation of the Santo Domingo Sur and Iris Norte pits
- Storage of material in the WRFs
- Disposal of tailings
- Management of contact and non-contact waters.

During the closure phase it is expected that impacts on water resources will be caused by the following activities:

- Closure and abandonment of the Santo Domingo Sur and Iris Norte pits

- Closure and abandonment of the WRFs
- Closure and abandonment of the TSF.

Environmental measures approved in the RCA for the key water impacts comprise:

- Construction of facilities to divert non-contact water
- Construction of concrete protection around water course crossings
- Construction of the TSF wall with a 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.

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: PM10, PM2.5, gases, and 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 Sur and Iris Norte pits, the transport of material from the pits to the crushing plant, and from the pits to the WRFs. For the port area, the main emissions of particulate matter and gas will be associated with wind erosion of the magnetite concentrate stockpile and the activities involved in 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/m^{3}N for PM10 in Diego de Almagro; the limit in the Chilean regulations is 50 µg/m^{3}N. It is forecast that the PM10 could reach 52 µg/m^{3}N 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.

Environmental measures approved in the RCA for the key air quality impacts include:

- Speed control on roads in active use
- Wetting of roads during construction
- Enclosed primary crusher
- Sprinkler installation in the 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 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.

Environmental measures approved in the RCA for the human environment are:

- 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 entrepreneurship projects
- Infrastructure support for the productive activity of the independent seaweed harvesters in the Punta Roca Blanca area
- Contribution of 10 L/s drinking water for the community of Diego de Almagro.

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.

A summary of the environmental measures approved in the RCA for significant impacts identified in the EIA is provided in Table 20-1.

Table 20-1: Measures Approved in the RCA

Measure	Component	Description
Compensation	Air quality	Street asphalting (3 km)
Compensation	Air quality	Street sweeping (2 km)
Mitigation	Air quality	Wetting of roads
Mitigation	Air quality	Covered stockpile to collect particulate matter
Mitigation	Air quality	A dust suppression system will be installed In the primary crusher
Compensation	Flora and vegetation	Seed collection, nursery and planting of species in the conservation category
Compensation	Flora and vegetation	Relocation of Cactaceaes and Bromilaceae to a similar habitat
Compensation	Flora and vegetation	Seed collection, nursery and plantation of endemic species
Mitigation	Flora and vegetation	Training for personnel on protection of flora and vegetation
Mitigation	Flora and vegetation	Installation of information signs for the protection of flora and vegetation
Mitigation	Flora and vegetation	2,049 individual *Copiapoa cinarescens* (either for 100% success in relocation or for purchase and planting)
Mitigation	Fauna	Rescue and relocation of low mobility fauna (9 species)
Mitigation	Fauna	Rescue and relocation of *Spalacopus cyanus* (where pipeline route crosses burrow areas)
Mitigation	Archaeology	Protection of 27 sites located outside construction areas
Mitigation	Archaeology	Training of personnel on protection of cultural heritage
Mitigation	Archaeology	Relocation of 5 animitas and memorials
Mitigation	Archaeology	Micro-routing to determine the presence of paleontological fossils in polygones with high potential
Mitigation	Archaeology	Archaeology monitoring

Measures for non-significant impacts are also stated in the RCA. The construction of two camps, one for construction with a capacity of 3,100 beds, and one for operation with a capacity of 450 beds, is approved. Another measure for non-significant impacts is the provision of 10 L/s of potable water for Diego de Almagro distributed by the local water company, Aguas Chañar.

If significant modifications to the project approved in the EIA are required, a Relevance Consultation (in Spanish, Consulta de Pertinencia) must be conducted, and sufficient background information should be provided to the authority so that it can evaluate and define the steps to be taken by Capstone (submission of a DIA or an EIA).

20.2.4 Overlaps and Stakeholder Concerns

In the public participation process, one topic covered about 30% of the observations made by citizens and stakeholders; this was the overlap of the works and activities for the port area with a Maritime Concession for the Development of Aquaculture. The concern was presented by Compañía Pesquera Camanchaca S.A., based on the potential environmental impact from Project activities (during construction and operation) on environment components which could affect the development of the economic activity.

Compañía Minera Sierra Norte S. A. presented an overlap between the area considered for its Diego de Almagro Project and Capstone's pipeline route.

In both cases, the environmental authority stated that the overlap of the areas does not correspond to the scope of the environmental assessment, and that this issue should be analyzed at the sectoral level for each interested party.

20.3 Closure Plan

20.3.1 Regulatory Considerations

Six main areas of legislation cover the closure of the proposed operations:

- Supreme Decree (D.S.) 132/2004 - Mining Safety Regulation N° 72
- D.S. 248/2007 - Regulations for the Approval of Design, Construction, Operation and Closure of Tailings Deposits.

- 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
- Law 19.300 - General Bases for the Environment, amended by law 20.417.

Criteria and contents of the mine Closure Plan under Chilean legislation are provided in Law 20.551 and D.S.41/2012. The Closure Plan must cover the mining facilities included on the EIA; for the Project this will include the mine site, port, pipeline route and transmission lines.

20.3.2 Closure Measures

In addition to the EIA closure obligations, Law 20.511/2011 requires the completion of a risk assessment of the physical and chemical stability of the main mine facilities. This risk assessment supports the establishment of the closure measures, and the post-closure mitigation and monitoring plan. This risk assessment is confined to health, safety, community/culture, and environment, and does not consider operational or financial risks.

The methodology of the risk assessment can be defined by a mining company; however, the Chilean government (through the Ministry of Mining, Sernageomin) has developed a risk assessment guideline. The guideline classifies mine facilities as follows: main facilities, complementary, and ancillary. Mine facilities that remain after closure (TSF, WRFs and open pits) are classified as main facilities. Complementary facilities include the process plant, pipelines, and port facilities. The ancillary facilities include domestic and industrial waste management facilities.

The Chilean government risk assessment methodology is semi-quantitative and based on a failure modes and effects anHDEPalysis (FMEA) methodology. The scope is limited to the main facilities. The risk level is determined by the severity of the consequences and the probability of occurrence in the long term.

In order to assess the risks related to physical and chemical stability for potential receptors, a baseline of the main environmental components potentially affected, and a physical and chemical characterization of the remaining facilities should be completed.

For the Santo Domingo Project, preliminary results indicate that the processed rock material has a low acid generation potential. However, a detailed material geochemistry characterization will be required during the next stages of the Project.

20.3.3 Financial Assurance

Law 20.551 requires the mining company to provide a bond which is calculated based on the closure costs. The state can execute this bond if the mining company does not comply with the closure commitments. The bond is submitted gradually (currently over a period of 15 years) with the amount to cover 20% of the total closure cost submitted during the first year of operation, discounted at a rate defined by the Central Bank of Chile (Sernageomin, 2018) .

20.3.4 Closure Action Summary

Capstone must apply for a closure permit to the government, the Closure Plan must be submitted and must specify closure measures defined through the risk assessment process and must include an estimate of the closure costs. The Closure Plan must be approved before construction starts and a bond must be delivered to the government of Chile during the first year of operation.

The closure phase for the Santo Domingo mine will occur between 2040 and 2042, followed by the post-closure phase. The post-closure phase consists of monitoring and inspection and is scheduled to end by 2047. Landfill monitoring may continue for another 20 years to comply with the Landfill Regulation (D.S. 186/2008 of the Ministry of Health).

20.3.5 Risks and Opportunities

The permit review and approval process presents a manageable risk to the Project schedule. Permitting planning and submission of permit applications has been a priority for the Project in 2017 and 2018. Closure Plan approval is preceded by obtaining all mining permits including those for the TSF, process plant, open pits (mine exploitation) and WRF. To date, Capstone has obtained permits for mine exploitation, the process plant and the WRF. Capstone has developed a Closure Plan following Sernageomin's Methodology Guide and has received permission to submit this Closure Plan after it has responded to technical questions about the TSF.

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 20-2 and Table 20-3.

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 and magnetite concentrate production, copper concentrate filtration; camp, services support (guard house, 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.
- Port area: magnetite concentrate filtration, storage of copper and magnetite concentrates, conveying and ship loading of concentrates, desalination system.
- Power transmission line area: transmission lines to supply power to the mine site and port.

About 140 works and installations were identified, distributed between the four areas. The number of permits required for all facilities was estimated to be about 750 in total, most of them related to the mine and plant area (about 60%). Permits that have been classed as critical to ensure that timely construction and start-up of the Project are summarized in Table 20-4.

The EIA was presented to the authorities in October 2013. The environmental assessment process took 22 months and the RCA was approved in July 2015.

Table 20-2: Closure Aspects and Measures, Mine and Plant Area

<table>
<tr><th colspan="2">Work Site and Main Facilities</th><th>Measure</th></tr>
<tr><td colspan="2">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 colspan="2">Waste Rock Facilities</td><td>Closure of access roads; slope stabilization; signage installation</td></tr>
<tr><td colspan="2">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 colspan="2">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 rowspan="2">Tailings Storage Facility</td><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 and put 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 rowspan="4">Services Area</td><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>
</table>

Work Site and Main Facilities		Measure
	Services Building (change room, laboratories, offices)	Removal of electrical supply and related infrastructure; dismantling of facilities; equipment removal; hazardous materials management; signage installation
Camp	Camp	Removal of electrical supply and related infrastructure; demolition of concrete structures and dismantling of facilities; signage installation
Electrical Transmission Lines and Electrical Substation	220 kV Transmission Line	Removal of electrical supply and related infrastructure; demolition of concrete structures and dismantling of facilities; contaminated soils management
	Electrical Substation	
Mining Roads and Internal Roads		Recovery of original drainage; construction of berms
Water Supply System	Ponds	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
	Water Treatment Plant	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
Waste Management Facilities	Sanitary Landfill	Removal of electrical supply and related infrastructure; dismantling of facilities; signage installation
	Waste Storage Yard	Dismantling of facilities; demolition of concrete structures; hazardous waste management, signage installation
	Sewage Treatment Plants	Removal of electrical supply and related infrastructure; removal of equipment and dismantling of facilities; demolition of concrete structures; signage installation
Explosives Magazine		Closure of explosives storage; dismantling of facilities; removal of concrete; contaminated soils management

Table 20-3: Closure Aspects and Measures, Pipelines, Port, and Transmission Lines

Worksite and/or Facility	Measure
Concentrate Transport System (STC)	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
Emergency Ponds	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; covering of foundations; re-profiling of the area
Magnetite Concentrate Filter Plant – Storage.	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; covering of foundations; dismantling of facilities; re-profiling of the area
Conveyors	Removal of electrical supply and related infrastructure; cleaning of belts; dismantling of facilities
Shipping Dock	Removal of electrical supply and related infrastructure; cleaning of structures and equipment and the shipping facilities; dismantling of facilities; removal of concrete
Desalination Plant	Removal or sale of supplies or reagents; cleaning of the structures and equipment; dismantling and demolition; covering of foundations
Ponds	Inventory of the area and removal of materials; dismantling/demolition of structures; cleaning of ponds and surrounding area; covering of foundations
Transmission Lines	Removal of electrical supply and related infrastructure; dismantling of facilities; contaminated soils management

Table 20-4: Critical Permits

Critical Permits
EIA Update (Consulta de Pertenencia recommended to determine if DIA or EIA required)
Closure Plan
Santo Domingo Port Maritime Concession*
DOP Permit*
Public Road Route 13 and Route C-17 By-Pass
Permit for Construction of Walls over 5 m height or more than 50,0000 m^3 of fill
Tailings Facility Approval (in process of approval in November 2018)
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*

Note: * These permits have been approved

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 any of the permits identified as critical to the Project. The Closure Plan permit has not yet been submitted. At the time of writing this report three of the four permits required prior to submitting the Closure Plan have been approved.
- Delays in the submission of permit files to the authorities.

A project owner has up to five years after the RCA is awarded to initiate the construction of the approved works or activities. Capstone's RCA was obtained in

2015, and will expire in July 2020. Hence, Capstone must consider this deadline when starting construction activities.

Other risks for general permits include:

- Changes or modifications of the Project configuration which were not included in the original EIA/RCA 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 2018 study update.

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 permit:

- Authorization for building the by-passes for public roads Route C-13 and Route C-17

The approved EIA (RCA obtained in 2015) considered the use of sea water for the process and the construction and operation of two small desalination plants, one in the mine and plant area and other in the port area. Later modifications were made to the Project to use desalinated water for the process and the construction of one large desalination plant in the port area, as part of a BOOT contract (the desalination plant and the desalinated water pipeline will be constructed and operated by a third party, who will deliver the water to the port and the mine and plant area).

This change will require a review of the environmental assessment. A Relevance Consultation must be conducted and sufficient background information must be provided to the authorities to allow them to evaluate and define the steps to be taken by Capstone. There are two main options to formalize modifications such as this: an Environmental Impact Statement (in Spanish, Declaración de Impacto Ambiental - DIA) or, an Environmental Impact Assessment (EIA). This may require a re-evaluation of the Project schedule in order to identify potential impacts.

20.5 Proposed Tailings Storage Facility

20.5.1 Introduction

The TSF will be located approximately 2 km southeast of the process plant (Figure 20-1), 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.

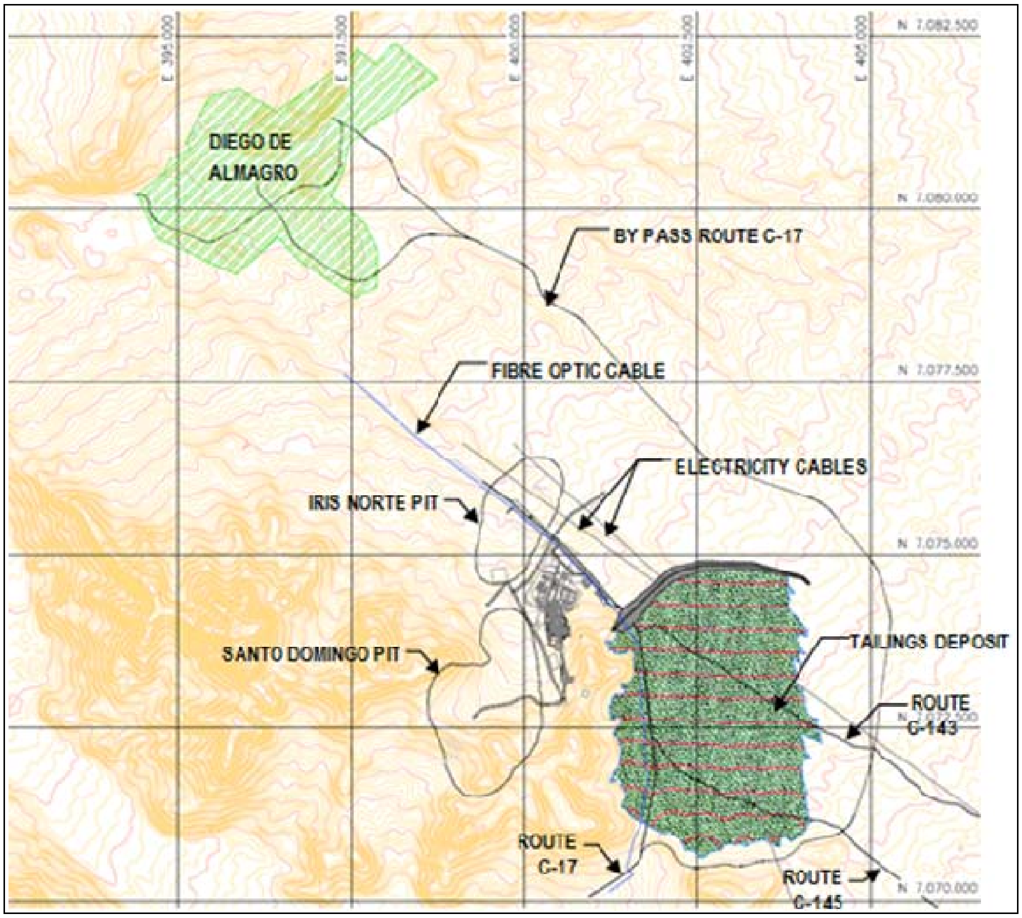
The TSF has been designed for a total tailings storage capacity of approximately 196 Mm^3 or 314 Mt at an estimated final overall average dry density of 1.6 t/m^3. The tailings will be deposited using the sub-aerial method from discharge points positioned in the basin of the TSF. The fresh tailings will be deposited at a slurry solids concentration of 67%. The overall beach slope is expected to be in the order of 1.5% towards the embankment on the north side. Tailings deposition will take place over an 18-year period.

20.5.2 Main Embankment and Saddle Dam

Description

The main embankment will comprise 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. It is recommended that future site investigations assess in more detail the soluble salts content within the foundation material to confirm that the recommended 2 m foundation excavations are deep enough to remove any material with excess soluble salts content. The results may require excavation and backfilling to greater depths.

Figure 20-1: 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 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 Mm^3 of fill will be placed into the embankment in four planned stages to reach a maximum height of 55.5 m. Additional fill may be required if the depth of the foundations increases. 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_{max} = 1.02 *g*
  - KH = 0.22

b) Operational earthquake

- Intra-plate earthquake of medium depth
  - Magnitude Mw = 7.5
  - Free field (hard ground) acceleration a_{max} = 0.54 *g*
  - KH = 0.16.

20.5.3 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 that the dams are predicted to remain stable under the loading conditions modelled. During detail design, the potential for locally saturated zones to develop in and under the dam will be analyzed.

20.5.4 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 and recycled to the process throughout the operating life of the TSF. This is necessary for efficient water use and to control the size of the supernatant pond in order to limit evaporation and potential 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 initial years of operations, when a lower density slurry is possible that may lead to larger volumes of water being stored in the TSF. The result may be that a- 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

Hershfield hydrograph method was used and two design scenarios were analyzed with different frequency factors (K) and catchment areas:

- A 27.3 km^2 basin immediately upstream of the TSF and K=19.8
- A larger (57.0 km^2) 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 run-off 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 the first year, additional pumping capacity will be in place from the supernatant pond, capable of evacuating the excess water produced by a 1,000-year storm (0.60 Mm^3) in three months, and from the 24-hour 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. During detail design the potential to temporarily store the 72-hour PMP within the TSF will be considered.

Seepage

Provided that the supernatant pond is kept above the geomembrane liner as intended by the design, and the geomembrane installation is of good quality, 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. During detail design the seepage rate through a lesser quality installation will be analyzed, including possible defects from installation and loading of the geomembrane liner and a larger than planned supernatant pond area particularly in the early period of operations. Seepage monitoring wells will be installed downstream of the embankment to monitor any potential changes in the quality of the ground water; if necessary, this water will be intercepted and pumped back into the TSF for return to the process and/or to treatment.

The maximum total flow seepage from the TSF including the tailings beach plus the supernatant pond based on the current liner limits is estimated to be 7.5 L/s during the first year of operation and is then predicted to 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 m^3/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 for use as a run-off discharge structure after closure. It will be positioned so that the storage available below it and above the tailings, for the short period of time to the end of deposition will be sufficient to contain the storm associated with a return period of 1,000 years. The spillway will be sized to safely pass flows up to the peak discharge associated with the 24-hour 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.

20.5.5 Thickened Tailings Distribution

The tailings will be pumped from the plant as thickened slurry (55% by mass solids content) to thickeners located at the southern end of the TSF where further thickening will be carried out to approximately 67% solids content. The tailings will be discharged into a tailings distribution box and directed in pipes to discharge points located in the TSF basin 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 distribution 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 by gravity at an average solids content of approximately 58% through two parallel pipes to location P-2 (Figure 20-2).

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 two points (P-1 and P-2) within the dam. 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 five points (P-1, P-2, P-3, P-4 and P-5). Gravity flows are expected, with the exception of the two last years when the southwest area of the TSF will require a pumping system.

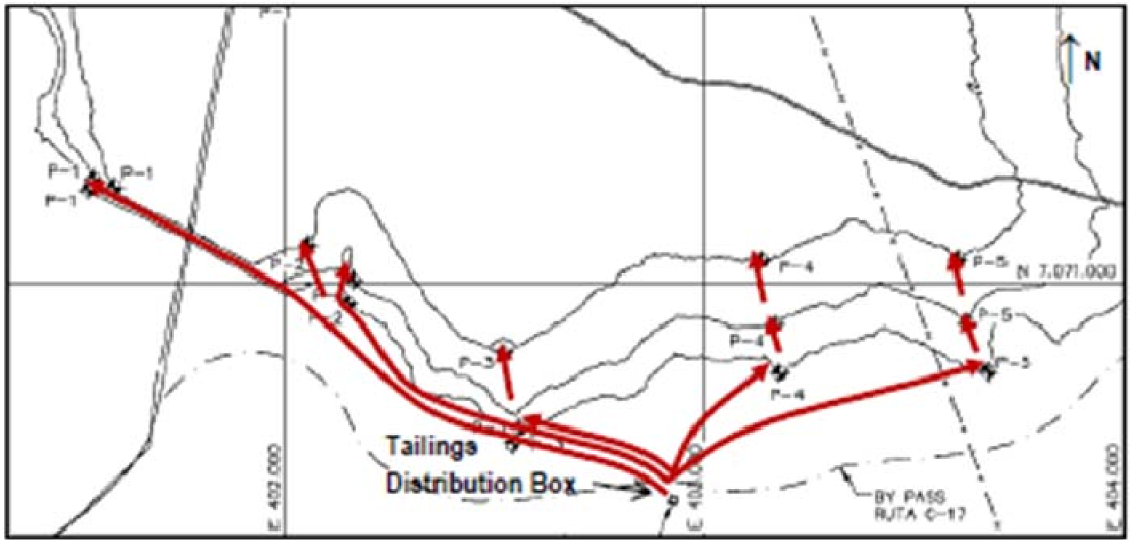
The final design of the TSF will take into account the potentially lower solids contents if the thickeners are less effective than planned.

20.5.6 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 will consist 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 separated from the geomembrane lined surface to avoid damaging the geomembrane
- Pumping the recovered water from the supernatant pond to a transfer tank, via pipes

Figure 20-2: Tailings Deposition Layout Plan (operations years 3–18)



Note: Prepared by Knight Piésold, November 2013.

- 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.

20.5.7 Monitoring

The monitoring system for the TSF will include the 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, additional pumping wells will be installed as needed.

Piezometers will be installed in and under the main embankment and saddle dam and in and under the tailings mass (including the geomembrane liner) 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.

20.5.8 Closure Considerations

Designs for the closure phase of the TSF include the emergency spillway and treatment or covering of the final tailings beach to reduce the generation of dust and the ingress of surface run-off into the tailings mass. A 0.3 m thick granular cover has currently been considered to meet this objective.

20.5.9 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 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. The lined area may be expanded in a future stage of design to further reduce this risk, and piezometers will be installed under the liner to detect any seepage

A number of areas where the Project design will impact existing infrastructure were noted. These include:

- TSF: 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 adjoin an area of the tailings facility which is legally reserved for Capstone's use. Capstone has a formal agreement with Mainstream to eliminate interferences between the two projects.

20.6 Considerations of Social and Community Impacts

20.6.1 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 the town of Diego de Almagro and the village of Inca del Oro in the Community of Diego de Almagro are included 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 pipeline route 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 pipeline route and the port facilities.

20.6.2 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 three 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 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
- The Indigenous Colla Community of Quillaga, located in the Quebrada El Jardin area of Diego de Almagro, the members belong to the Colla lineage of the Jerónimo family.

There are no indigenous lands or territories of any kind being claimed in the Project area.

20.6.3 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 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 influence of the Project.

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 EIA for the Project, the impacts associated with the major concerns of the community are:

- Increased particulate material PM10 and MP2.5 at the Project mine site. The rating of these impacts was very relevant for the operation phase. A Mitigation Plan will be implemented consisting of paving streets and sweeping throughout the life of the Project in order to offset the effect on air quality in Diego de Almagro, for both PM10 and PM2.5
- The increased demand for accommodation and housing in the town of Diego de Almagro during the operation phase of the Project was rated a significant impact in the EIA, because the 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 provide additional water for the inhabitants of Diego de Almagro, facilitating increased residency in town. This issue was very important for the local communities that participated in the public participation process.

- The location of Puerto Santo Domingo is near Caleta Obispito, a small settlement dependent on the resources of the sea, and some independent seaweed harvesters located near the planned port site. The port construction and operation will interrupt access to some coastal areas, in addition to the potential impact on marine resources due to the sea water intake and the brine discharge during operation. These impacts were rated in the EIA as relevant and moderately relevant respectively. The Action Plan developed by Capstone includes increased support of infrastructure to develop this activity. For example: the provision of basic equipment, and the implementation of an algae drying area. Also, competitive grants will be available for local entrepreneurship projects for independent seaweed harvesters located near the port.

The communications strategy for the Santo Domingo Project will focus on building a positive reputation and supportive environment for the Project development in the Atacama Region. Specific development strategies are directed to 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.

20.6.4 Consultation

The stakeholder plan was developed to implement an early citizen participation program (PACA) that allowed the local communities involved in the Project to understand the major Project components, and for the Project 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. 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 special interest groups 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 TSF. 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 by-pass road for the town of Diego de Almagro, which will reduce traffic congestion and will avoid the transit of heavy equipment vehicles through the town
- Plan for hiring local workers. This entails re-training 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.

20.6.5 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.

20.6.6 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.

20.6.7 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 legislation was identified.

20.6.8 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 the Santo Domingo 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 PACA in 2012 and 2013 and included meetings with stakeholders, local and regional authorities including the 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 that a crisis management plan be developed.

Key issues that the communications policy will address are:

- Water availability: Although the Project plans to use desalinated sea water for plant operation, it will require water during the construction stage. This construction water will be provided using a supply contract with Aguas Chañar.
- Manpower: Competition for workers from other industries and mining operations.
- 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 other commercial maritime activities. A specific management communications strategy has been developed for this issue.
- Political issues.

20.6.9 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.

20.7 Discussion on Risks

An environmental risk assessment was completed. Delays in obtaining the critical permits or authorizations for the Project were recognized as a key risk to the proposed Project schedule. However, Capstone has submitted four of the five permits that are required prior to starting construction, and three of these have been approved.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

21.1.1 Basis of Estimate

The capital cost estimate for the Santo Domingo Project was developed by the following companies:

- Wood: 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 magnetite concentrate pipeline
- Knight Piésold: design and estimating for the TSF
- NCL: design and estimating for the mine equipment and mine development
- Ghisolfo: design and estimating for the Route C-17 by-pass road, mine access road, port access road and Route C-13 detour
- Capstone: Owner costs.

The capital costs were consolidated by Wood. The estimate is classed as a Type 3 estimate according to Wood standards (and AACE International), 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 construction 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.

21.1.2 Mine Capital Costs

The total estimated mining capital costs are summarized in Table 21-1. The estimate includes:

- The initial Project period which includes all funds spent prior to the start-up of processing and metals production
- The initial capital period mine pre-production development cost of $57.1 M is for pre-stripping and ore stockpiling
- The cost for mine equipment for pre-production (pre-stripping and ore stockpiling) is $106.8 M
- Sustaining capital expenditures reflect the increase of the equipment fleet to achieve the required material movement
- Sustaining capital totals $285.3 M from Year 1 through Year 16.

Table 21-1: Mine Capital Cost Estimate Summary ($ x 1,000)

Cost Area	Initial Capital			Sustaining Capital	Total Capital
	Y-1	Y0 H1	Total		
Mine development	28,091	29,025	57,117	—	57,117
Equipment purchase	67,684	13,287	80,971	186,779	267,750
Equipment rebuild	—	—	—	93,102	93,102
Other investments	22,664	118	22,783	3,790	26,573
Dispatch	2,861	154	3,015	1,633	4,648
Total	***121,301***	***42,585***	***163,886****	***285,303***	***449,189***

* Does not include mine infrastructure ($13.6 M) which is included in the mine total in the summary tables

The total capital for mine equipment is $267.8 M over the mine life. The costs for the major and minor equipment are based on quotations obtained by Wood during third-quarter 2018. 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 $22.7 M, and the sustaining capital for light truck replacements amounts to $3.8 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 for dispatch are $3.0 M and $1.6 M respectively.

21.1.3 Process Capital Costs

The process area estimate was prepared by Wood 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, and instrumentation. The equipment, materials, and commodities were organized per major work areas (e.g. crushing, grinding, flotation, and 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 Wood 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 Wood 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 a typical crew mix from projects similar in nature. The different labour classes and area productivity rates were used to develop the total estimated manhours.

Wood's productivity rates are based on North American projects (US and Canada). Wood'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.

Wood used construction equipment rates from a periodically-updated Wood 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 Wood'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 Wood'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 Wood's 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.

The capital costs for the process plant are summarized in Table 21-2.

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 Mm^3, 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 Capstone's 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, and a deeper cut-off trench for anchoring the liner. A more detailed study of the soluble salts content in the dam foundation soils may result in additional excavation at the dam footprint. A review of the hydrology study to include the latest significant hydrological events in the area may change the volume for the PMP, and as a result have some impact on the size of the dam to accommodate this volume.

Table 21-2: Process Plant Capital Cost Estimate Summary

Description	Cost ($ M)
Process Plant General	7.1
Ore Handling	43.4
Grinding	115.0
Copper Flotation and Regrind	58.0
Magnetic Separation and Regrind	40.1
Tailings Thickening and Transport	28.4
Reagent Plant	9.4
Copper Concentrate Filtration	12.0
Total	***313.3***

Table 21-3: TSF Capital Cost Estimate Summary

TSF Stage	Type of Cost and When Applied	Cost ($ M)
Stage 1: Starter	Initial Capital Cost	21.8
Stage 2	Sustaining Capital Year 2	8.9
Stage 3	Sustaining Capital Year 8	7.5
Stage 4	Sustaining Capital Year 12	9.2
Total		***47.4***

21.1.5 Infrastructure Capital Costs

Camp

The unit price per square metre based on modular buildings was obtained from a budget quotation from TecnoFastAtco. The total capital cost of the camp was estimated by Correa 3 and Capstone, and was provided to Wood for inclusion in the capital estimate.

Built Infrastructure

Budget quotes were obtained for the following built infrastructure, based on building designs that were prepared by the Wood 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.

*Note: The heavy vehicle work shop (HVWS) was designed by the Wood engineering group. For the pricing of the HVWS, material take-offs were prepared and unit rate pricing from the Wood database was used to determine the overall cost of the facility.

- Port site:
  - Security gatehouse
  - Maintenance workshop and warehouse
  - Port administration building and lunchroom
  - Metallurgical laboratory
  - Change house.

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 9 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 km long. Transmission lines and substations design and cost estimates were prepared as part of the electrical design by Wood.

Table 21-4: Road Capital Cost Estimate Summary

Description	Cost ($ M)
Main access road	2.0
C-17 re-routing	14.7
C-13 by-pass	4.3
Port access road	0.7
Total	***21.7***

21.1.6 Concentrate Pipeline

The magnetite concentrate transport was designed and estimated by BRASS for a total estimated cost of $86.8 M. Costs are shown in Table 21-5.

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 21-6 and totals $121.4 M.

21.1.8 Indirect Costs

Wood estimated the indirect costs for the Project execution phase as summarized in Table 21-7. Indirect costs total $269.2 M.

21.1.9 Owner Costs

The Owner costs were estimated by Capstone and were provided to Wood to incorporate into the Project capital cost estimate. The Owner costs total an estimated $111.8 M (Table 21-8), and the indirect costs including Owner costs total $381.0 M.

Table 21-5: Magnetite Concentrate Pipeline Capital Cost Estimate Summary

Description	Cost ($ M)
Direct Costs	
Construction and assembly	27.5
Overhead expenses and profit (40%)	11.0
Subcontractors	14.3
Pipe material, spools	15.2
Positive displacement pumps	6.6
Charge pumps, valves, fittings	9.7
Electrical and instrumentation equipment	1.3
Total Direct Costs	*85.5*
Total Indirect Costs	*1.3*
Total Concentrate Pipeline Cost	***86.8***

Table 21-6: Port Capital Cost Estimate Summary

Facility	Supplies Costs ($ M)	Subcontracts Cost ($ M)	Installation Cost ($ M)	Total Cost ($ M)
Copper concentrate transport and loading	3.7	—	4.0	7.7
Copper concentrate loading and transport to loading platform	10.9	—	1.8	12.7
Magnetite concentrate handling and storage	0.8	—	0.4	1.2
Magnetite concentrate storage	1.4	0.3	1.8	3.5
Magnetite concentrate transport to loading platform	9.5	0.9	3.0	13.4
Access trestle – conveyor belt	14.4	—	5.0	19.4
Ship loader supports	1.6	—	2.0	3.6
Mechanical equipment – copper and magnetite concentrate	11.0	0.1	2.1	13.2
Marine works services	1.0	0.6	0.2	1.8
Mooring and berthing infrastructure	5.2	—	4.6	9.8
Port infrastructure (on site)	0.1	4.9	10.2	15.2
Control, communications and safety system	—	0.7	0.1	0.8
Sea water supply	1.1	—	0.6	1.8
Indirect costs	—	1.8	—	1.8
Profit @ 15%	—	15.4	—	15.4
Total port cost	***60.8***	***24.6***	***36.6***	***121.4***

Table 21-7: Indirect Costs Estimate Summary

Cost Item	Cost ($ M)
Engineering and procurement services (EP)	43.5
Construction management (CM)	62.9
Home & field office materials (EPCM)	6.0
Support engineering	0.6
Construction camp	34.5
Catering and camp services	24.6
Temporary installations	12.0
Water supply (industrial and potable water)	3.4
Power	3.5
Firefighting, cleaning, maintenance, waste management	1.3
Safety, communications	0.7
Consulting, specialists, third party services	8.1
Warehouse	2.6
Crane	1.2
Off-site and on-site transport	1.3
Pre-commissioning, commissioning	4.4
Freight and customs	28.1
Vendor reps.	6.0
Start-up and first year spares	12.3
First fill	2.5
Transport for local staff	0.3
Indirect additional/allowance	9.4
Total Indirect Costs	***269.2***

Table 21-8: Owner Costs Estimate Summary

Description	Cost ($ M)				Total Cost ($ M)
	Y-3	Y-2	Y-1	Y0 H1	
Labour cost - Project	1.6	3.1	0.7	0.3	5.7
General management	0.4	0.4	0.3	0.2	1.3
Administration and finance	1.2	3.5	1.8	1.7	8.2
Legal, mining property and permits	2.0	9.3	1.8	0.9	14.0
Health, safety, environmental and community relations	1.8	10.8	8.9	5.0	26.5
Labour cost (Project support)	2.8	4.7	5.3	3.2	16.0
Labour cost (services)	2.6	5.0	7.1	4.9	19.6
Recruitment and selection process	0.6	1.0	0.4	0.6	2.6
Mine, process and other training	0.1	0.6	1.9	5.0	7.6
Personnel transport	-	1.3	1.8	1.2	4.3
Catering	-	0.4	0.8	0.7	1.9
Labour accreditation, consultants	0.1	0.4	0.7	0.4	1.6
Vehicles	-	0.6	0.6	0.7	1.9
Personal protective equipment (PPE)	-	0.1	0.1	0.1	0.3
Others (payroll, newsletter, travel, human resources (HR) team)	0.0	0.1	0.1	0.1	0.3
Total	***13.1***	***41.2***	***32.5***	***25.0***	***111.8***

21.1.10 Contingency

A range analysis was developed, based on Wood's probabilistic model with Capstone's participation. Wood 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.

Wood'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 $197.8 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 Summary of Initial Capital Cost Estimate

The initial capital cost estimate by area is presented in Table 21-9.

21.1.13 Summary of Sustaining Capital

Over the LOM, the sustaining capital cost is $378.6 M. Table 21-10 summarizes the sustaining capital by year.

21.1.14 Exclusions

Exclusions to the capital cost estimates include:

- Changes in design criteria
- Desalinated water plant and pipeline system
- Scope changes or accelerated schedule
- Closure costs (closure is included in the operating costs)
- Escalation
- Currency variations
- Force majeure
- Changes in the law
- Schedule delays
- Any rights and licenses not included in Owner costs
- Financing costs
- Sunk costs
- Working capital

Table 21-9: Initial Capital Cost Estimate (by Area)

Area	Cost ($ M)	% of Total
Mine	177.5	12
Process plant	313.3	21
Tailings and water reclaim	48.2	3
Plant infrastructure (on site)	81.9	5
Port	147.4	10
Port infrastructure (on site)	21.9	1
External infrastructure (off site)	143.2	9
Indirect costs	381.0	25
Contingency	197.8	13
Total	***1,512.3***	***100%***

Notes: Costs in this table are distributed and summarized by major area and include costs from consultants, Wood, Capstone, or other parties.

Table 21-10:Summary of Sustaining Capital by Year

Description	Amount ($ x 1,000)
Year 0 – H2	98,614
Year 1	36,254
Year 2	18,204
Year 3	17,933
Year 4	9,007
Year 5	39,282
Year 6	2,277
Year 7	111
Year 8	37,856
Year 9	43,229
Year 10	17,925
Year 11	467
Year 12	20,412
Year 13	1,333
Year 14	5,441
Year 15	30,010
Year 16	249
Total Sustaining Capital	***378,602***

- Plant mobile equipment (this is included in the operating costs)
- Potential impacts from any strikes, riots or looting.

21.2 Operating Costs

The estimate is considered to be at 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 2018 US dollars, unless stated otherwise
- Costs are based on an exchange rate of CLP600 to $US1.00

- The costs per tonne of material treated ($/t) provided in the 2018 Feasibility Study update 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 salaried and hourly labour 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 66% for magnetite
- For the copper equivalent estimate, average life-of-mine prices of $3.00/lb copper and $80.00/t magnetite concentrate were used.
- Operating costs for the Project were based on the Mine Plan Nov2018 (issued 13 November 2018) 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 2018 Feasibility Study update was that the 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 $0.77/L delivered to site was used in the operating cost estimate (provided by Capstone)
- The mine fleet replacement was considered to be part of the sustaining capital estimate.

Mine operating cost forecasts are included in Table 21-11.

Table 21-11:Mine Operating Costs

Item	LOM Total ($ M)	LOM Average ($/t Material Mined)	LOM Average ($/t Treated)	LOM Average ($/lb CuEq)
Drilling	145.80	0.09	0.37	0.04
Blasting	351.13	0.21	0.89	0.08
Loading	400.34	0.24	1.02	0.10
Hauling	1,150.87	0.69	2.93	0.28
Ancillary	170.01	0.10	0.43	0.04
Support equipment	39.82	0.02	0.10	0.01
Engineering and administration	54.84	0.03	0.14	0.01
Labour	306.77	0.18	0.78	0.07
Total	***2,619.57***	***1.57***	***6.68***	***0.63***

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
- Power costs
- Reagent quotes and equipment supplier quotations were used as applicable
- Logistics and transport costs from supplier quotations were used as applicable
- Tailings storage facility operating costs were estimated by Knight Piésold
- Concentrate pipeline pumping and operating costs were estimated by BRASS
- PRDW estimated the port facilities materials handling costs.

The process operating cost forecast is provided in Table 21-12.

Table 21-12:Process Operating Costs – Commodity Summary

Area	LOM Total ($ M)	LOM Average ($/t)	LOM Average ($/lb CuEq)
Process Operating/Plant	*1,811.1*	*4.62*	*0.437*
Labour	110.8	0.28	0.027
Power	746.0	1.90	0.180
Reagents	199.2	0.51	0.048
Steel	508.1	1.30	0.122
Operating supplies	19.2	0.05	0.005
Maintenance materials	82.2	0.21	0.020
Other costs	145.7	0.37	0.035
TSF and Tailings Water Reclaim	*39.0*	*0.10*	*0.009*
Labour	18.25	0.05	0.004
Power	9.7	0.03	0.002
Reagents	7.8	0.02	0.002
Operating supplies	0.2	0.00	0.000
Maintenance materials	0.8	0.00	0.000
Other costs	2.2	0.01	0.001
Magnetite Concentrate Transport System	*74.3*	*0.190*	*0.018*
Labour	3.60	0.009	0.001
Power	10.0	0.026	0.002
Maintenance materials	39.6	0.101	0.009
Other costs	4.2	0.011	0.001
Management	*16.9*	*0.043*	*0.004*
*Desalinated Water Transfer System**	*437.1*	*1.11*	*0.105*
Other costs	420.2	1.07	0.101
Management	16.9	0.04	0.004
Magnetite Filtration – Port	*94.2*	*0.24*	*0.023*
Labour	31.9	0.08	0.008
Power	9.6	0.02	0.002
Operating supplies	9.1	0.02	0.002
Maintenance materials	26.6	0.07	0.006

Area	LOM Total ($ M)	LOM Average ($/t)	LOM Average ($/lb CuEq)
Other costs	0.0	0.00	0.000
Management	16.9	0.04	0.004
Fe and Cu Handling, Storage and Loading	*91.8*	*0.23*	*0.022*
Labour	15.0	0.04	0.004
Power	17.1	0.04	0.004
Operating supplies	13.0	0.03	0.003
Maintenance materials	0.0	0.00	0.000
Other costs	29.8	0.08	0.007
Management	16.9	0.04	0.004
Total	*2,547.6*	*6.49*	*0.61*

Note: * Based on potential BOOT operator quote, plus Capstone management

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 $205.4 M over the LOM.

Table 21-13:Labour Cost Breakdown

Area	LOM Total ($ x 1,000)	LOM Average ($/t)	LOM Average ($/lb CuEq)	Hours per Year (MH/a)	Total LOM (MH/LOM)	No. of Staff	Cost* ($/MH)
Process operating/plant	*110.8*	*0.28*	*0.027*	*261*	*4,588*	*116*	*24.1*
Labour	110.8	0.28	0.027	261	4,588	116	24.1
TSF and tailings water reclaim	*18.25*	*0.05*	*0.004*	*42*	*706*	*15*	*25.9*
Labour	18.25	0.05	0.004	42	706	15	25.9
Magnetite concentrate transport system	*10.06*	*0.026*	*0.002*	*22*	*372*	*10*	*27.05*
Labour	3.60	0.009	0.001	11	186	5	19.3
Labour in management	6.46	0.017	0.002	11	186	5	34.8
Desalinated water transfer system	*6.46*	*0.016*	*0.002*	*11*	*186*	*5*	*34.8*
Labour	0.00	0.00	0.000	0	0	0	0.0
Labour in management	6.46	0.017	0.002	11	186	5	34.8
Magnetite Filtration Port	*38.37*	*0.098*	*0.009*	*113*	*1,925*	*51*	*19.93*
Labour	31.90	0.081	0.008	102	1,739	46	18.3
Labour in management	6.46	0.016	0.002	11	186	5	34.8
Fe & Cu handling, storage and loading	*21.48*	*0.055*	*0.005*	*59*	*1,000*	*26*	*21.48*
Labour	15.01	0.038	0.004	48	814	22	18.4
Labour in management	6.46	0.016	0.002	11	186	5	34.8
Total	***205.4***	***0.524***	***0.049***	***508***	***8,777***	***228***	***22.91***

Note: * The total costs in this column are weighted totals from the unit costs for labour and management. MH = man hours.

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 unit electricity cost (in $/MWh), delivered to the nearest electrical substations to the Project at the Diego del Almagro substation for the mine site and the Totoralillo substation for the port site, is estimated at $72.00/MWh including all system-related charges. The estimate was provided by Capstone, who evaluated the estimated cost to deliver electricity to the mine, and held discussions with major power companies currently operating in Chile to verify the commercial assumptions used in the economic analysis. Power costs are summarized in Table 21-14, and for the LOM total is $792.4 M. This equates to a power cost of $2.02/t of ore processed.

21.2.5 Reagents and Consumables

Reagents will include lime, flotation reagents (primary collector, secondary collector, and frother), 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 $207 M over the LOM. This equates to a LOM average of $0.53/t ore, and a LOM average of $0.049/lb CuEq.

Steel

Steel includes liners and ball requirements for crushers and mills. Steel requirements are estimated to total $508.1 M over the LOM. This equates to a LOM average of $1.30/t ore and a LOM average of $0.122/lb CuEq.

Table 21-14:Power Consumption and Costs

	Description	Total LOM (MWh)	LOM Average (kWh/t ore)	Total LOM Cost ($ x 1,000)	LOM Average ($/t Treated)
2100	Process plant	9,888,920	25.21	713,964	1.82
2110	Materials handling	194,038	0.49	13,971	0.04
2120	Grinding	6,869,019	17.58	496,531	1.27
2130	Flotation and copper regrinding	1,384,253	3.53	99,666	0.25
2140	Magnetic separation and regrinding	874,070	2.23	62,933	0.16
2150	Thickening and tailings pumping system	493,488	1.26	35,531	0.09
2160	Reagents plant	22,127	0.06	1,593	0.00
2170	Copper concentrate filtration	51,925	0.13	3,739	0.01
3100	Tailings and water reclaim	135,281	0.34	9,741	0.02
3110	High density thickening and tailings pumping system	105,736	0.27	7,613	0.02
3120	Tailings distribution system	16,938	0.04	1,220	0.00
3160	Water reclaim and pumping system	12,607	0.03	908	0.00
4100	Plant infrastructure (on-site)	444,773	1.13	32,024	0.08
4120	Water distribution and desalination plant	279,372	0.71	20,115	0.05
4130	Plant administration buildings	113,474	0.29	8,170	0.02
4150	Plant power supply and distribution	8,069	0.02	581	0.00
4160	Plant ancillary facilities	43,858	0.11	3,158	0.01
5100	Port	333,355	0.85	24,001	0.06
5110	Concentrate reception and storage	25,937	0.07	1,867	0.00
5120	Magnetite concentrate filtration	107,514	0.27	7,741	0.02
5130	Copper concentrate transportation and storage	57,599	0.15	4,147	0.01
5140	Magnetite concentrate pumping system	70,139	0.18	5,050	0.01
5150	Maritime works	72,166	0.18	5,196	0.01
5200	Port infrastructure (on-site)	37,508	0.10	2,701	0.01
5220	Concentrate storage and water reclaim	13,216	0.03	952	0.00
5250	Plant power supply and distribution	16,139	0.04	1,162	0.00

	Description	Total LOM (MWh)	LOM Average (kWh/t ore)	Total LOM Cost ($ x 1,000)	LOM Average ($/t Treated)
5260	Port ancillary facilities	8,153	0.02	587	0.00
6100	External Infrastructure (off-site)	139,117	0.35	10,016	0.03
6120	Concentrate transportation system	139,117	0.35	10,016	0.03
6150	Desalinated water supply	—	—	—	—
Total		***10,978,954***	***27.98***	***792,447***	***2.02***

Desalinated Water

The capital and operating costs estimates supply of desalinated water to the mine and plant site are based on a third-party company building, owning and operating the desalination plant and water pipeline from the port to the mine (under a BOOT or BOO arrangement). It was assumed that Capstone will purchase the water at a price of $2.50/m^3, including reimbursement to the third-party water supplier for the capital investment and operating costs. The estimate was provided by Capstone, who evaluated the estimated cost to produce and deliver desalinated water to the mine, and held discussions with water supply companies to verify the commercial assumption used in the economic analysis.

Operating Supplies

Operating supplies include considerations of wear items costs (hydrocyclones and screens), fuel costs for the process plant, filter plate costs, and operating supplies costs for the tailings and water reclaim. The estimates are summarized in Table 21-15.

Table 21-15:Operating Supplies Estimates

Description	LOM Total ($ M)	LOM Average ($/t ore)	LOM Average ($/lb CuEq)
Wear items	3.55	0.01	0.001
Fuel (process plant)	27.17	0.07	0.007
Filter plates	10.61	0.03	0.003
Operating supplies tailings and water reclaim	0.19	0.00	0.000

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 $0.41/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
- Knight Piésold, BRASS and PRDW provided supplementary information
- Start-up and first year spares are excluded; these are included in capital costs
- Mining equipment fleet spares are excluded.

Over the LOM, the maintenance costs are estimated to total $159.9 M, which equates to a LOM average cost of $0.038/lb CuEq.

21.2.7 Other Costs

Other costs include third party contracts for the desalinated water supply, leasing, minor maintenance, operations and/or support contracts, and total $632 M. This equates to a LOM average of $1.62/t ore of and a LOM average of $0.610/lb CuEq.

21.2.8 Summary of Operating Cost Estimate

The operating cost estimate by area is shown in Table 21-16.

The cost of copper concentrate land transport is included in the financial analysis.

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) (included in the financial model)
- Onshore costs for copper concentrate transport (included in the financial model)
- Escalation and exchange rate fluctuations
- Exploration
- Permits
- Import duties (included in the financial model)
- Taxes (included in the financial model)
- Interest and financing charges
- Corporate overheads
- Operating cost contingency.

21.3 Comments on Section 21

The estimated total LOM capital cost is $1,512.3 M.

The estimated LOM operating cost estimate is $5,770.0 M.

Table 21-16:Operating Cost Estimate (by Area)

Cost Centre	LOM Total ($ M)	LOM Average ($/t)	LOM Average ($/lb CuEq)
Process operations/plant	1,811.1	4.62	0.437
Tailings storage facility and tailings water reclaim	39.0	0.10	0.009
Iron concentrate transport system	74.3	0.19	0.018
Desalinated water system	437.1	1.11	0.105
Magnetite filtration – port	94.2	0.24	0.023
Magnetite and copper handling, storage and loading	91.8	0.23	0.022
Total - Process	*2,547.6*	*6.49*	*0.614*
G&A	*402.8*	*1.03*	*0.097*
Mining	*2,619.6*	*6.68*	*0.631*
Total	***5,570.0***	***14.20***	***1.34***

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 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 a non-inflated, after tax basis. The Project cash flows consist of approximately three years of pre-production costs and 18 years of operations. Cash inflows consist of annual revenue projections for the mine. Cash outflows include capital costs, operating costs, royalties and taxes, which 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 present study valuation date of Q3 2018 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 to discount rates was completed using discount rates of 4%, 6%, 8% (selected rate), 10% and 12%. 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 on an average mid-year basis of each annual period.

22.2 Financial Model Parameters

The financial model is based on the Mineral Reserves outlined in Section 15, 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.

The capital and operating costs, financial evaluations and reported results were estimated using the foreign exchange rates noted in Table 22-1.

Capital costs and sustaining capital costs are summarized in Table 21-9 and Table 21-10, respectively. Initial capital costs for the Base Case are estimated to be $1,512.3 M. Over the LOM sustaining capital is estimated to be $378.6 M.

Operating costs are summarized in Table 21-16. LOM operating costs for the Base Case are estimated to be $5,770.0 M. Total and net LOM operating costs, as well as unit costs per tonne of ore treated and per pound of payable copper are summarized in Table 22-2.

Closure and reclamation costs have been estimated to be $102.1 M.

Smelting and refining terms considered in the evaluation are listed in Table 22-3.

Table 22-1: Exchange Rates Used

Currency	CLP per Unit
Dollar	600
Euro	732

Table 22-2: Total Project Operating Costs

Cash Costs	LOM Total ($ M)	LOM Average ($/t)	LOM Cost ($/lb Cu payable)
Mining	2,619.6	6.68	1.13
Process	2,547.6	6.49	1.10
G&A	402.8	1.03	0.18
Cu concentrate transport (onshore & offshore) insurance and sales	225.3	0.57	0.10
Sub-total	*5,795.2*	*14.77*	*2.51*
By-product metal credits	6,242.0	(15.91)	(2.70)
TC/RC costs	486.5	1.24	0.21
Total - Cu Cash Cost (net of Fe & Au by-product credits)	***39.8***	***0.10***	***0.02***

Note: Totals may not sum due to rounding

Table 22-3: Smelter Terms

Item	Unit	Value
Concentrate Cu grade	%	29.0
Cu concentrate moisture	%	8.0
Cu concentrate losses	%	0.10
Cu land freight	$/wmt	11.58
Cu ocean freight	$/wmt	40.00
Cu marketing and umpiring	$/wmt	3.00
Cu insurance premium	%	0.05
Cu treatment charge	$/dmt	80.00
Cu pay factor	%	96.5
Cu unit deduction	%	0.0
Cu refining charge	$/lb Cu	0.08
Magnetite concentrate grade	%	66.0
Magnetite concentrate moisture	%	8.0
Magnetite concentrate price	$/dmt	80.00
Au pay factor	%	97.0
Au unit deduction	g/t	1.0
Au refining charge	$/oz	5.00

Transport and insurance charges for copper concentrate are provided in Table 22-4. Life of mine copper transport costs are estimated to be $47.3 M for land freight, $163.4 M for ocean freight, $3.3 M for insurance and $11.3 M for marketing (total $225.3 M).

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 free on board (FOB) Santo Domingo port shipping basis.

Table 22-4: Copper and Magnetite Concentrate Transport and Insurance Charges

Item	Unit	Value
Cu concentrate land freight	$/wmt	11.58
Cu concentrate ocean freight	$/wmt	40.00
Cu concentrate insurance	%	0.05
Cu concentrate marketing and umpiring	$/wmt	3.00
Magnetite concentrate land freight	N/A	In operating costs
Magnetite concentrate ocean freight	N/A	FOB Santo Domingo port
Magnetite concentrate insurance	N/A	FOB Santo Domingo port
Magnetite concentrate marketing and umpiring	N/A	0

Note: N/A = not applicable

As such, no additional transport or insurance charges are required to be included in the financial model for iron concentrate.

The Project was evaluated using a range of six different sets of metal prices. The ranges are shown in Table 22-5. The base case price is Case 4, reflecting averages of analysts' medium-term projections.

Royalties of 2% NSR are payable to third parties on 100% of the production. The NSR is charged on all of the metals (copper, iron, and gold) recovered. The LOM royalty payments are estimated to be $249.6 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 $25.6 M in the first year of operation and a LOM maximum of $61.0 M (on a previous year's cumulative basis) in Year 2 and in Year 18.

The economic analysis assumes that no inflationary adjustments are made. Capital and operating costs are based on Q3 2018 US dollars.

Table 22-5: Metal Prices

Pay Metals	Case	1	2	3	4	5	6
Copper	$/lb	2.25	2.50	2.75	3.00	3.25	3.50
Gold	$/oz	1,000	1,100	1,200	1,290	1,400	1,500
Iron (Fe 65%)	$/t	65	70	75	80	85	90

Note: Base case is highlighted

Possible salvage values for the mine, plant and port 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.

22.3 Taxation Considerations

The Project was evaluated on an after-tax basis with taxes payable in three forms:

- Government royalty or specific mining tax
- Corporate income tax
- VAT (referred to as IVA in Chile).

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 $262.7 M over the LOM and is deductible as an expense against corporate income tax.

The corporate income tax consists of the First Category Tax (FCT) at 27%. Total FCT payments over the LOM are estimated to be $1,152.9 M. The Second Category or "Additional" Tax was not evaluated for the Project. This Second Category Tax is levied on dividend distributions to foreign shareholders.

Value Added Tax (IVA) of 19% is applicable to a number of goods and services purchased; however, this tax is refundable once the mine is in operation. Other than the delay in the recovery of IVA charged during construction and the impact of the time value of money, the LOM net effect of IVA is zero.

The Project evaluation was prepared on an all equity funded basis. There may be opportunities to utilize debt capital to fund the Project and improve the equity return, but this will require planning to consider Chilean thin capitalization requirements, stamp duties and withholding taxes on interest.

Capstone has signed a D.L. 600 Foreign Investment Contract to contribute the capital investment into the Project. The D.L. 600 Contract provides the ability to elect tax invariability treatment for the Project.

22.4 Financing Considerations

Wood'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.

22.5 Financial Results

The economic analysis is based on a real basis (no inflation). On an after-tax basis, the cumulative net cash flow for the base case is $3,250.5 M, the IRR is 21.8% and the payback period is 2.8 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.

The cash flow analysis for the base case is provided in Table 22-6. The after-tax annual and cumulative cash flows are shown in Figure 22-1.

The C1 cash cost as defined by Wood McKenzie is 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."

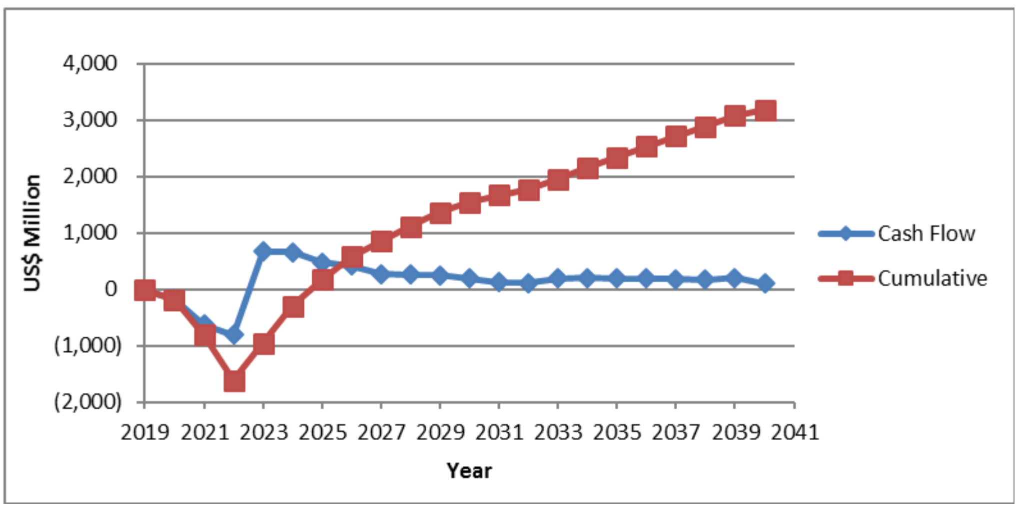
Cash costs are summarized in Table 22-7 for the first five years of operation and in Table 22-8for the LOM. The gold and iron credits fully offset the operating costs over the LOM, resulting in a C1 cash cost of almost zero ($0.02/lb).

Table 22-6: Results of Financial Analysis

Summary of Cash Flow	Unit	Pre-tax	After Tax
Cumulative net cash flow			
Undiscounted	$ M	4,666.1	3,250.5
Net present value			
Discounted at 4%	$ M	2,704.3	1,837.4
Discounted at 6%	$ M	2,073.4	1,381.0
Discounted at 8%	$ M	1,591.6	1,031.9
Discounted at 10%	$ M	1,219.4	761.8
Discounted at 12%	$ M	928.6	550.7
Internal rate of return	***%***	***26.6***	***21.8***
Payback period	***Years***	***2.6***	***2.8***

Note: Base case is highlighted

Figure 22-1: Project After Tax Cash Flow



Note: Figure prepared by Wood, 2018

Table 22-7: Summary of Cash Costs First Five Years

Cash Costs	**Years 1 – 5 (Excludes 2022) ($ M)**	**Cost Per Tonne Milled ($/t)**	**Cost Per Pound Cu Payable ($/lb)**
Costs			
Mining	781.1	6.60	0.63
Process	750.0	6.34	0.60
G&A	114.3	0.97	0.09
Smelter deductions	83.9	0.71	0.07
Treatment charges	161.9	1.37	0.13
Refining charges	100.4	0.85	0.08
Concentrate transport	121.4	1.03	0.10
Sub-total	2,113.1	17.87	1.69
Credits			
Au	(224.0)	(1.89)	(0.18)
Fe	(1,303.1)	(11.02)	(1.04)
Sub-total	(1,527.2)	(12.91)	(1.22)
Adjusted Cash Costs Total	***586.0***	***4.96***	***0.47***

Note: Totals may not sum due to rounding

Table 22-8: Summary of Cash Costs - LOM

Cash Costs	LOM Total (U M)	Cost Per Tonne Milled ($/t)	Cost Per Pound Cu Payable ($/lb)
Costs			
Mining	2,619.6	6.68	1.13
Process	2,547.6	6.49	1.10
G&A	402.8	1.03	0.17
Cu smelter deductions	155.7	0.40	0.07
Cu treatment charges	300.3	0.77	0.13
Cu refining charges	186.2	0.47	0.08
Cu concentrate transport	225.3	0.57	0.10
Sub-total	*6,437.5*	*16.41*	*2.78*
Credits			
Au	(392.6)	(1.00)	(0.17)
Fe	(6,005.1)	(15.31)	(2.59)
Sub-total	*(6,397.7)*	*(16.31)*	*(2.76)*
Adjusted cash cost total	***39.8***	***0.10***	***0.02***

Note: Totals may not sum due to rounding

The cash flow summary in Table 22-9 provides a breakdown of the LOM cash flow that results in an undiscounted margin of $2.01/lb payable copper after application of all costs other than taxes. Table 22-10 provides the cash flow on an annualized basis.

22.6 Sensitivity Analysis

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

- Metal price (copper and iron)
- Operating costs (including power)
- Power supply costs alone
- Capital costs
- Exchange rates.

Table 22-9: Summary of Cash Flow

Cost Item	LOM ($ M)	Milled ($/t)	Cu Payable ($/lb)
Revenue (after losses and before deductions)			
Cu	7,200.4	18.35	3.11
Au	392.6	1.00	0.17
Fe	6,005.1	15.31	2.59
Sub-Total	13,598.1	34.66	5.87
Smelting costs			
Treatment	(300.3)	(0.77)	(0.13)
Cu deduction	(252.0)	(0.64)	(0.11)
Au deduction	(155.7)	(0.40)	(0.07)
Refining – Cu	(185.3)	(0.47)	(0.08)
Refining – Au	(0.9)	(0.00)	(0.00)
Transport	(225.73)	(0.57)	(0.10)
Sub-Total	(1,119.5)	(2.85)	(0.48)
Operating cost			
Mining	(2,619.6)	(6.68)	(1.13)
Process	(2,547.6)	(6.49)	(1.10)
G&A	(402.8)	(1.03)	(0.17)
Sub-Total	*(5,570.0)*	*(14.20)*	*(2.40)*
Other			
Royalty	(249.6)	(0.64)	(0.11)
Closure	(102.1)	(0.26)	(0.04)
Total	*(351.6)*	*(0.90)*	*(0.15)*
Earnings before interest, taxes, depreciation, and amortization (EBITDA)	6,557.0	16.71	2.83
Construction capital	(1,512.3)	(3.85)	(0.65)
Sustaining capital	(378.6)	(0.97)	(0.16)
Undiscounted margin (cumulative net cash flow)	4,666.1	11.89	2.01

Note: Totals may not sum due to rounding

Table 22-10:Project Cash Flow on an Annualized Basis

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$/lb						3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00
Au	US$/oz						1,290	1,290	1,290	1,290	1,290	1,290	1,290	1,290	1,290	1,290	1,290	1,290	1,290	1,290	1,290	1,290	1,290	1,290	1,290
Iron ore	US$/t						80	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80	80
		Life of Mine																							
Extracted metal value (after losses & before deductions)																									
Cu	US$000	7,200,397					108,979	984,990	891,599	719,351	675,458	610,279	496,519	406,594	306,388	304,301	259,173	247,894	276,708	233,403	198,931	162,701	156,126	99,337	61,668
Au	US$000	392,563					6,651	58,712	50,327	41,678	38,123	35,199	27,659	21,799	16,770	16,822	13,324	12,706	14,101	12,010	9,413	7,270	6,342	2,244	1,414
Fe concentrate	US$000	6,005,123					30,781	216,135	266,116	305,745	323,594	191,531	213,719	307,955	318,218	284,411	313,823	382,697	368,136	400,354	431,917	431,935	431,461	431,249	355,347
Total	US$000	13,598,083					146,411	1,259,837	1,208,042	1,066,774	1,037,175	837,008	737,897	736,347	641,376	605,534	586,320	643,297	658,945	645,767	640,262	601,906	593,929	532,829	418,429
Smelter deductions/premiums																									
Cu	US$000	(252,014)					(3,814)	(34,475)	(31,206)	(25,177)	(23,641)	(21,360)	(17,378)	(14,231)	(10,724)	(10,651)	(9,071)	(8,676)	(9,685)	(8,169)	(6,963)	(5,695)	(5,464)	(3,477)	(2,158)
Au	US$000	(155,698)					(2,357)	(21,299)	(19,280)	(15,555)	(14,606)	(13,196)	(10,736)	(8,792)	(6,625)	(6,580)	(5,604)	(5,360)	(5,983)	(5,047)	(4,302)	(3,518)	(3,376)	(2,148)	(1,333)
Total	US$000	(407,712)					(6,171)	(55,774)	(50,485)	(40,732)	(38,247)	(34,556)	(28,115)	(23,023)	(17,349)	(17,231)	(14,675)	(14,037)	(15,668)	(13,216)	(11,264)	(9,213)	(8,840)	(5,625)	(3,492)
Treatment charge																									
Cu concentrate	US$000	(300,326)					(4,545)	(41,084)	(37,188)	(30,004)	(28,173)	(25,455)	(20,710)	(16,959)	(12,779)	(12,692)	(10,810)	(10,340)	(11,541)	(9,735)	(8,297)	(6,786)	(6,512)	(4,143)	(2,572)
Refining charges																									
Cu	US$000	(185,290)					(2,804)	(25,347)	(22,944)	(18,511)	(17,382)	(15,705)	(12,777)	(10,463)	(7,884)	(7,831)	(6,669)	(6,379)	(7,121)	(6,006)	(5,119)	(4,187)	(4,018)	(2,556)	(1,587)
Au	US$000	(918)					(17)	(145)	(120)	(101)	(91)	(85)	(66)	(50)	(39)	(40)	(30)	(28)	(31)	(27)	(20)	(15)	(11)	(0)	(0)
Total	US$000	(186,208)					(2,821)	(25,492)	(23,064)	(18,613)	(17,473)	(15,790)	(12,843)	(10,513)	(7,924)	(7,870)	(6,699)	(6,408)	(7,152)	(6,033)	(5,139)	(4,201)	(4,029)	(2,557)	(1,587)
Cu concentrate transport																									
Land freight	US$000	(47,300)					(716)	(6,470)	(5,857)	(4,725)	(4,437)	(4,009)	(3,262)	(2,671)	(2,013)	(1,999)	(1,703)	(1,628)	(1,818)	(1,533)	(1,307)	(1,069)	(1,026)	(653)	(405)
Port storage & handling	US$000																								
Ocean freight	US$000	(163,384)					(2,473)	(22,350)	(20,231)	(16,323)	(15,327)	(13,848)	(11,266)	(9,226)	(6,952)	(6,905)	(5,881)	(5,625)	(6,279)	(5,296)	(4,514)	(3,692)	(3,543)	(2,254)	(1,399)
Marketing & other	US$000	(11,273)					(171)	(1,542)	(1,396)	(1,126)	(1,058)	(956)	(777)	(637)	(480)	(476)	(406)	(388)	(433)	(365)	(311)	(255)	(244)	(156)	(97)
Insurance charges	US$000	(3,316)					(51)	(456)	(411)	(332)	(312)	(282)	(229)	(187)	(141)	(140)	(119)	(114)	(127)	(107)	(91)	(74)	(71)	(44)	(27)
Total	US$000	(225,273)					(3,410)	(30,819)	(27,896)	(22,507)	(21,133)	(19,094)	(15,534)	(12,721)	(9,586)	(9,521)	(8,108)	(7,755)	(8,657)	(7,302)	(6,223)	(5,090)	(4,884)	(3,106)	(1,928)
Net smelter return	US$000	12,478,564					129,463	1,106,668	1,069,408	954,918	932,149	742,113	660,695	673,132	593,738	558,220	546,028	604,758	615,927	609,481	609,338	576,616	569,664	517,398	408,849
Third party royalty payment																									
Net smelter return	US$000	12,478,564					129,463	1,106,668	1,069,408	954,918	932,149	742,113	660,695	673,132	593,738	558,220	546,028	604,758	615,927	609,481	609,338	576,616	569,664	517,398	408,849
Portion of production (non-cash)	%	100%					100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%
Royalty percentage of NSR (non-cash)	%	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%	2.00%
Applied royalty	US$000	(249,571)					(2,589)	(22,133)	(21,388)	(19,098)	(18,643)	(14,842)	(13,214)	(13,463)	(11,875)	(11,164)	(10,921)	(12,095)	(12,319)	(12,190)	(12,187)	(11,532)	(11,393)	(10,348)	(8,177)
Production costs																									
Mining	US$000	(2,619,572)					(59,757)	(144,730)	(149,316)	(165,633)	(164,585)	(156,794)	(139,048)	(142,214)	(155,784)	(164,834)	(159,090)	(160,671)	(160,012)	(138,459)	(145,888)	(139,466)	(130,143)	(79,701)	(63,448)
Process	US$000	(2,547,558)					(25,285)	(147,762)	(151,009)	(150,210)	(150,095)	(150,957)	(141,311)	(141,777)	(142,744)	(141,603)	(141,678)	(143,606)	(142,074)	(142,025)	(143,451)	(132,925)	(122,650)	(138,529)	(97,867)
G&A	US$000	(402,844)					(14,042)	(24,461)	(22,470)	(22,401)	(22,487)	(22,506)	(22,414)	(22,446)	(22,558)	(22,485)	(21,782)	(21,681)	(21,688)	(21,594)	(21,559)	(21,548)	(21,106)	(20,826)	(12,787)
Total	US$000	(5,569,973)					(99,084)	(316,953)	(322,795)	(338,244)	(337,167)	(330,257)	(302,773)	(306,438)	(321,086)	(328,922)	(322,550)	(325,959)	(323,774)	(302,078)	(310,898)	(293,940)	(273,899)	(239,056)	(174,102)
Cash Cost	US$/lb	0.02					2.13	0.51	0.40	0.33	0.26	0.90	0.76	0.20	0.23	0.66	0.32	(0.50)	(0.28)	(1.09)	(1.66)	(2.40)	(2.89)	(5.71)	(8.83)
Closure & salvage																									
Closure costs	US$000	(102,052)																							(102,052)
Total	US$000	(102,052)																							(102,052)
Earnings																									
Earnings before taxes. depreciation & amortization	US$000	6,556,967					27,790	767,582	725,225	597,576	576,339	397,014	344,709	353,231	260,777	218,133	212,557	266,704	279,834	295,213	286,253	271,144	284,372	267,994	124,518
Taxation																									
Operation incomes and expenses	US$000	6,908,590					30,380	789,715	746,613	616,674	594,982	411,857	357,922	366,694	272,652	229,298	223,478	278,799	292,153	307,403	298,440	282,676	295,765	278,342	234,747
Royalty payment	US$000	(249,571)					(2,589)	(22,133)	(21,388)	(19,098)	(18,643)	(14,842)	(13,214)	(13,463)	(11,875)	(11,164)	(10,921)	(12,095)	(12,319)	(12,190)	(12,187)	(11,532)	(11,393)	(10,348)	(8,177)
Start-up expenses (corporate tax) (non-cash)	US$000	(133,395)					(133,395)																		
Tax loss (non-cash)	US$000	(1,446,894)						(701,897)	(501,690)	(243,306)															
Tax depreciation for the first category income tax (non-cash)	US$000	(1,890,879)					(596,292)	(535,294)	(438,103)	(56,992)	(59,316)	(17,538)	(18,291)	(11,791)	(17,409)	(38,591)	(33,560)	(9,736)	(8,785)	(8,933)	(7,205)	(17,465)	(14,975)	(599)	(4)
Mine closure	US$000	(102,052)																	(17,009)	(17,009)	(17,009)	(17,009)	(17,009)	(17,009)	

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**
Mining royalty (specific tax for mining)	US$000	(262,739)						(32,081)	(28,738)	(19,692)	(18,313)	(9,305)	(8,407)	(10,229)	(5,748)	(8,511)	(8,957)	(11,860)	(12,674)	(14,047)	(13,607)	(14,064)	(16,033)	(16,043)	(14,429)
Taxable income (non-cash)	US$000	2,823,060					(701,897)	(501,690)	(243,306)	277,586	498,711	370,171	318,010	331,212	237,621	171,031	170,040	245,107	241,367	255,224	248,432	222,606	236,355	234,343	212,137
First category income tax A		1,152,887								74,948	134,652	99,946	85,863	89,427	64,158	46,178	45,911	66,179	65,169	68,910	67,077	60,104	63,816	63,273	57,277
Tax depreciation for the first category income tax (non-cash)	US$000	1,890,879					596,292	535,294	438,103	56,992	59,316	17,538	18,291	11,791	17,409	38,591	33,560	9,736	8,785	8,933	7,205	17,465	14,975	599	4
Regular tax depreciation (non-cash)	US$000	(1,872,553)					(179,010)	(184,182)	(186,105)	(188,640)	(189,644)	(194,209)	(181,367)	(151,880)	(151,942)	(50,561)	(35,376)	(34,385)	(34,814)	(33,705)	(30,962)	(17,477)	(11,622)	(8,856)	(7,818)
Start-up expenses (corporate tax) (non-cash)	US$000	133,395					133,395																		
Start-up expenses (specific tax for mining) (non-cash)	US$000	(133,395)					(22,233)	(22,233)	(22,233)	(22,233)	(22,233)	(22,233)													
Royalty payment (2% NSR)	US$000	249,571					2,589	22,133	21,388	19,098	18,643	14,842	13,214	13,463	11,875	11,164	10,921	12,095	12,319	12,190	12,187	11,532	11,393	10,348	8,177
Tax loss (non-cash)	US$000	1,446,894						701,897	501,690	243,306															
Mines closure	US$000	102,052																	17,009	17,009	17,009	17,009	17,009	17,009	
Taxable base for the mining royalty (specific tax for mining) (non-cash)	US$000	4,639,904					(170,863)	551,220	509,538	386,111	364,793	186,109	168,148	204,585	114,962	170,225	179,145	232,554	244,666	259,651	253,871	251,135	268,110	253,443	212,500
Tax rate for the mining royalty (specific tax for mining) (non-cash)	%	5.66					0.00%	5.82%	5.64%	5.10%	5.02%	5.00%	5.00%	5.00%	5.00%	5.00%	5.00%	5.10%	5.18%	5.41%	5.36%	5.60%	5.98%	6.33%	6.79%
Mining royalty (specific tax for mining) B	US$000	262,739						32,081	28,738	19,692	18,313	9,305	8,407	10,229	5,748	8,511	8,957	11,860	12,674	14,047	13,607	14,064	16,033	16,043	14,429
Tax amount (A+B)	US$000	**(1,415,626)**						**(32,081)**	**(28,738)**	**(94,640)**	**(152,964)**	**(109,252)**	**(94,270)**	**(99,656)**	**(69,906)**	**(54,690)**	**(54,868)**	**(78,039)**	**(77,843)**	**(82,958)**	**(80,684)**	**(74,167)**	**(79,849)**	**(79,316)**	**(71,706)**
Interim payment of absorbed earnings																									
Value-added tax (VAT)																									
Payment value added tax purchase (VAT) (Capex)	US$000	(195,980)		(79)	(13,688)	(79,762)	(82,562)	(18,919)	(971)																
Payment value added tax costs (VAT)	US$000	(1,125,604)					(19,300)	(75,500)	(74,477)	(73,395)	(72,261)	(69,671)	(62,510)	(61,192)	(61,531)	(62,825)	(60,759)	(61,098)	(61,348)	(56,702)	(57,464)	(53,780)	(50,211)	(43,024)	(48,554)
Outstanding VAT CF	US$000				(79)																				
Recuperation value added tax (net zero)	US$000	**1,321,584**			**13,767**	**79,762**	**101,862**	**94,420**	**75,448**	**73,395**	**72,261**	**69,671**	**62,510**	**61,192**	**61,531**	**62,825**	**60,759**	**61,098**	**61,348**	**56,702**	**57,464**	**53,780**	**50,211**	**43,024**	**48,554**
Payment value added tax	US$000	**(1,321,584)**		**(79)**	**(13,688)**	**(79,762)**	**(101,862)**	**(94,420)**	**(75,448)**	**(73,395)**	**(72,261)**	**(69,671)**	**(62,510)**	**(61,192)**	**(61,531)**	**(62,825)**	**(60,759)**	**(61,098)**	**(61,348)**	**(56,702)**	**(57,464)**	**(53,780)**	**(50,211)**	**(43,024)**	**(48,554)**
Capital expenditure																									
Construction	US$000	(1,512,277)		(15,094)	(171,064)	(625,198)	(700,439)	(482)																	
Sustaining	US$000	(378,602)					(98,614)	(36,254)	(18,204)	(17,933)	(9,007)	(39,282)	(2,277)	(111)	(37,856)	(43,229)	(17,925)	(468)	(20,412)	(1,333)	(5,441)	(30,010)	(249)		
Working capital	US$000						(25,623)	(61,320)	(730)	(2,515)	613	2,237	7,761	(213)	(2,878)	(1,943)	1,946	(764)	321	5,763	(1,935)	4,523	5,062	9,155	60,541
Construction. sustaining & working capital	US$000	(1,890,879)		(15,094)	(171,064)	(625,198)	(824,677)	(98,056)	(18,933)	(20,448)	(8,394)	(37,045)	5,484	(323)	(40,734)	(45,172)	(15,978)	(1,232)	(20,091)	4,430	(7,376)	(25,487)	4,813	9,155	60,541
Net Project cash flow																									
Pre-tax	US$000	4,666,088		(15,094)	(171,064)	(625,198)	(796,887)	669,525	706,292	577,128	567,945	359,970	350,193	352,908	220,043	172,962	196,579	265,472	259,743	299,643	278,877	245,657	289,185	277,149	185,058
After tax	US$000	3,250,461		(15,173)	(170,985)	(625,198)	(796,887)	637,444	677,554	482,488	414,981	250,718	255,923	253,252	150,137	118,272	141,711	187,433	181,900	216,685	198,193	171,490	209,336	197,833	113,353
Payback																									
Pre-tax cumulative net cash flow	US$000			(15,094)	(186,158)	(811,356)	(1,608,242)	(938,717)	(232,425)	344,703	912,648	1,272,618	1,622,811	1,975,719	2,195,762	2,368,723	2,565,302	2,830,775	3,090,518	3,390,161	3,669,038	3,914,695	4,203,881	4,481,029	4,666,088
After tax cumulative net cash flow	US$000			(15,173)	(186,158)	(811,356)	(1,608,242)	(970,798)	(293,244)	189,244	604,225	854,943	1,110,866	1,364,117	1,514,255	1,632,527	1,774,238	1,961,671	2,143,571	2,360,257	2,558,450	2,729,940	2,939,276	3,137,109	3,250,461

Note: Totals may not sum due to rounding

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 in the bullet points above. Sensitivities to copper and iron grades are not shown, because changes in copper and iron grades are mirrored by the sensitivities to changes in the copper and iron prices, respectively.

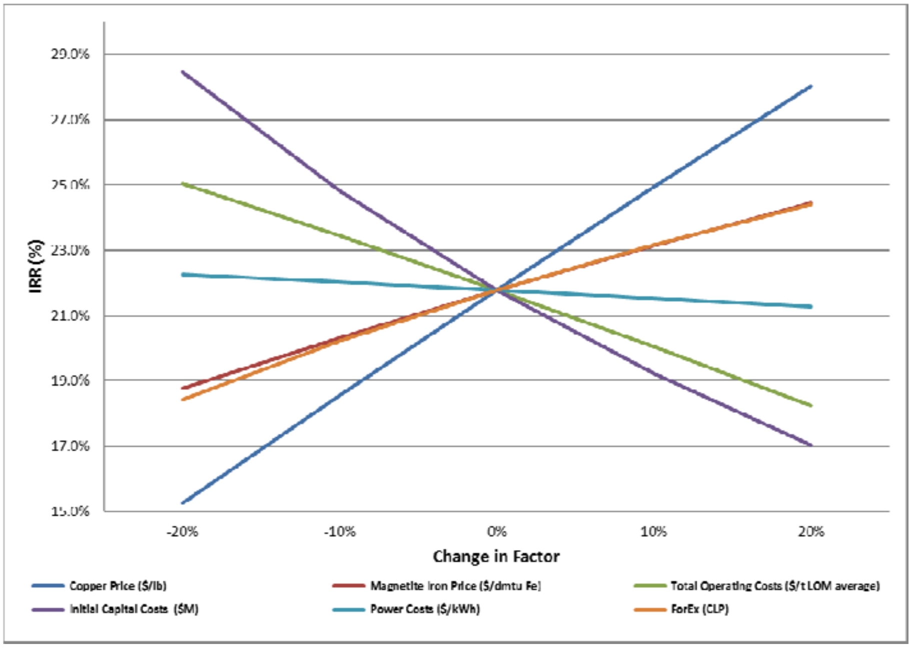
The analysis shows that the Santo Domingo Project NPV8% is most sensitive to changes in the copper price (copper grade) and in total operating and capital expenditures. The sensitivity analysis showed that the Project is less sensitive to changes in the iron price (iron grade) and the dollar/peso exchange rate and least sensitive to changes in power costs. Because the Project is priced in US dollars, the effects of exchange rate variation other than the CLP do not apply in the current model, although in reality some equipment, supplies and services may be priced in other currencies such as Euro.

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 22-11.

22.7 Comments on Section 22

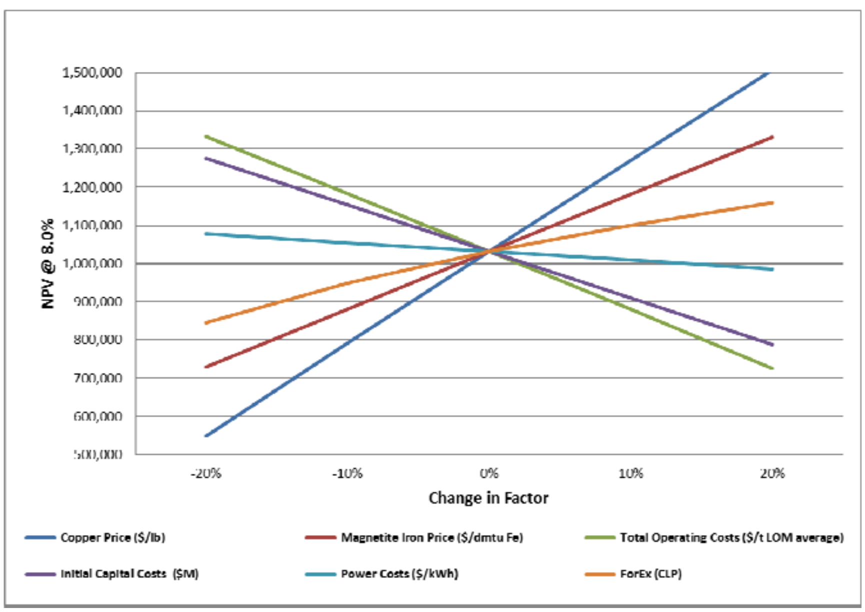
Using the assumptions outlined in this Report, the Project has a positive cash flow.

Figure 22-2: Sensitivity of IRR



Note: Figure prepared by Wood, 2018

Figure 22-3: Sensitivity of NPV8% ($ x 1,000) for Base Case



Note: Figure prepared by Wood, 2018

Table 22-11:Sensitivity to Metal Price

Item	Unit	Case					
		1	2	3	4	5	6
Copper price	$/lb	2.25	2.50	2.75	3.00	3.25	3.50
Gold price	$/oz	1,000	1,100	1,200	1,290	1,400	1,500
Iron price	$/t	65	70	75	80	85	90
Pre-tax CNCF	$ M	1,809	2,762	3,715	4,666	5,621	6,574
Pre-tax NPV 8%	$ M	295	727	1,160	1,592	2,025	2,458
Pre-tax IRR	%	11.9	17.1	22.0	26.6	31.0	35.2
Pre-tax payback	Years	4.9	3.7	3.1	2.6	2.3	2.0
After-tax CNCF	$ M	1,279	1,941	2,598	3,250	3,901	4,546
After-tax NPV 8%	$ M	116	426	732	1,032	1,331	1,627
After-tax IRR	%	9.7	14.0	18.0	21.8	25.3	28.7
After-tax payback	Years	5.1	3.9	3.2	2.8	2.4	2.2
C1 cash cost before credits	$/lb	2.76	2.77	2.77	2.78	2.79	2.79
C1 cash cost after credits	$/lb	0.53	0.36	0.19	0.02	(0.15)	(0.32)

Note: Base case is highlighted. CNCF = cumulative net cash flow.

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 Capstone's name. Signing authority agreements will be put in place with Capstone's prime EPC contractor providing contract and procurement services. These agreements will allow specified representatives to sign documents on Capstone's behalf once the appropriate Project documentation has been approved by Capstone.

24.2 Risk and Opportunity Analysis

A risk and opportunity analysis was performed as part of the 2014 Feasibility Study. In 2018 Capstone reviewed the opportunities and risks associated with the current Project status. A summary of the main opportunities and risks includes:

- Opportunities
  - Include cobalt inclusion in the Mineral Reserve
  - Enhance gold recovery through further metallurgical testing
  - Incorporate autonomous haulage
  - Share infrastructure with other local companies
  - Consider autogenous grinding.
- Risks
  - Delay in financing
  - Schedule delays
  - Contractor engagement and price uncertainty
  - Increased equipment and labor costs.

Risks and opportunities will be continuously assessed and reviewed throughout the various phases of the project, in accordance with Capstone's Risk Management Framework. A risk register hosts the risk and opportunity information, and is further recorded on a risk matrix according to the level of risk which is determined by the probability or occurrence (likelihood) and the resulting consequence of its impact. The level of risk assists in prioritizing actions to take advantage of opportunities and to mitigate risks

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 Wood supports that the capital of Minera Santo Domingo is indirectly 70% owned by Capstone and 30% by Korean Resource Corporation (Kores)
- Capstone has advised Wood that under the terms of the shareholder agreement signed between Capstone and Kores on 17 June 2011, Capstone is the Project operator
- Mineral tenure documentation provided to Wood supports that Capstone has two groups of concessions with a total of 116 claims which cover a total of 28,897 ha and includes the areas of the planned mine site, plant area and auxiliary facilities including the port facilities. 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 Wood that all concession fees were current as of 31 October 2018
- Capstone possess 17 provisional surface rights and eight definitive surface rights; 17 surface rights are currently in the process of approval. These easements cover 100% of the facilities and infrastructure area
- The surface land in the districts of Diego de Almagro, Caldera and Chañaral where the Project is located are owned by the State, and are managed and represented by the Ministerio de Bienes Nacionales
- Capstone has developed a legal strategy to obtain all necessary surface rights to cover mine, plant, camps, tailings storage facilities, pipelines, port and transmission line

- 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 desalinated sea water. A maritime concession has been obtained 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 Wood 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 included a regional airborne geophysical survey and interpretation of results, geological mapping, surface and drainage sampling, an IP survey, core and RC drilling, and resource estimation. A preliminary assessment was conducted in 2008. Capstone acquired Far West and completed a pre-feasibility study. A feasibility study was commissioned in 2012, completed in 2014, and aspects of the study were updated in 2018
- 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 80 core holes (16,488 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
- Assay QA/QC protocols have not been applied for cobalt. For this reason, the block cobalt grades are not appropriate for use in the economic evaluation of the Project
- 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, including RPA, 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 water, sea water and desalinated 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 2018 testwork results using desalinated water were used to develop algorithms for copper and gold recovery based on the feed copper grade. The copper recovery algorithm is $\mathit{Global\ Cu\ Recovery} = 0.98 * (96.9018*(\mathit{Feed}, \%\ \mathit{Cu})^{0.0199}$ and the gold recovery algorithm is $\mathit{Global\ Au\ Recovery} = 0.85*(82.646*(\mathit{Feed}, \%\ \mathit{Cu})^{0.1611}$. The locked cycle test gave a copper global recovery of 92.4% with a concentrate grade of 32.8% Cu
- Further testwork with desalinated water is recommended to confirm the design criteria and the recovery algorithms
- 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;
- A limited metallurgical testwork program was performed in 2018 on cobalt recovery, under the supervision of Blue Coast. Initial results indicate that the extracted cobalt is amenable to leaching under acidic conditions. Capstone has authorized further studies to define the optimum flow sheet for cobalt processing. Additional testwork is also planned to determine the quality of the cobalt sulfate and assess the impact of any impurities that might affect marketing There is upside potential for the Project if this work supports cobalt estimation as part of an updated Mineral Reserve estimate, such that cobalt as a by-product can be incorporated into a future financial analysis.

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. An estimate for cobalt was completed in 2018
- The Mineral Resource estimates were updated in 2018 using current metal prices, revised mass recovery and CuEq calculations, updated cut-off grades and the application of constraining pit shells
- 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, gold at 0.52 g/t Au and cobalt was capped at 1,750 ppm Co. 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. For the 2018 estimate 27 cobalt assay intervals were capped representing approximately 0.4% of the total number of assays. Grades at Estrellita were capped at 3% Cu, 0.3 g/t Au and 1,000 ppm Co

- 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 copper, gold, iron, cobalt, 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 copper, gold, iron and cobalt grades into each block. 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, Iris Norte and Estrellita deposits using parameters derived from 2018 prices. Measured and Indicated Mineral Resources total 537 Mt grading 0.30% Cu, 0.039 g/t Au, 25.7% Fe and 229 ppm Co. Inferred Mineral Resources total an additional 48 Mt grading 0.19% Cu, 0.025 g/t Au, 23.6% Fe and 197 ppm Co.
- Risk factors that could potentially affect the Mineral Resources estimates include changes to the assumptions used to determine reasonable prospects of eventual economic extraction, 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.

25.8 Mineral Reserve Estimates

- Open pit Mineral Reserves were constrained within an LG shell. An internal cut-off of $7.53/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 $7.53/t cut-off, but for an 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 392.3 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 0.84 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 0.84 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 2018 study update assumes that the mining operation will use 42 m^3 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 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 desalinated water, pumped from the port to the plant site, by a BOOT contractor
- The nominal capacity of the process plant for the first five years will be 65,000 t/d or 23.7 Mt/a excluding the ramp up period. The nominal capacity after the first five years will be 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 514.1 kt of contained copper. The highest production of magnetite concentrate is around 5.4 Mt which occurs in Years 14 and 15.

25.11 Infrastructure

- The planned route for transporting cargo, staff and equipment to the Santo Domingo site is from the south of the mine site 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 on the mine site
  - Port facilities: Located about 43.5 km north of Caldera at Punta Roca Blanca
  - 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 the Diego de Almagro substation to the proposed mine and plant site
  - High voltage transmission line: from the Totoralillo substation to the port
- Three WRF 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 behind a main embankment and a saddle dam from compacted, non-acid generating mine waste rock with a geo-membrane liner installed on the upstream face of the main embankment, extending into the impoundment area where it will be anchored in a cut-off trench. Tailings will be thickened to a target 67% solids prior to deposition. It is expected that the target solids content will not be reached in the early years of operation. A final capacity of approximately 314 Mt of copper and iron tailings, equivalent to a total volume

of about 196 Mm^3, is proposed to be deposited over the approximate 18 years of operations

- Mineral processing will use desalinated water provided by a third party; there will be a desalination plant at the port operated by the third party. The nominal total requirement for water is 348.6 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 recovered from the TSF throughout the operational life of the facility, and will be pumped back to the process facility
- Capstone has committed to provide 10 L/s of potable water to Diego de Almagro
- 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 pipeline route is designed to run parallel to the existing roads, and uses existing RoW access to avoid the construction of new roads
- Water will be primarily supplied by the BOOT contractor operating the desalinated water system, and assumes that 49 L/s will be reclaimed from the TSF
- The magnetite concentrate pipeline design maximum capacity is 415 m^3/hr at 68% concentration
- Based upon current Capstone concentrate production requirements, the maximum required annual port capacity is 5.4 Mt/a of magnetite concentrate and 0.51 Mt/a of copper concentrate. Magnetite concentrate is planned to be shipped using a mixture of Panamax and Cape size vessels, larger vessels may be used. Copper concentrate would be shipped using Panamax and Handymax size vessels. The selected nominal loading rate will be 4,000 t/hr for magnetite concentrate and 2,000 t/hr for copper concentrate
- Capstone's mine site and port site will be connected to the Sistema Eléctrico Nacional (SEN, National Electrical System). The closest connection point between the SEN and the mine site is via a direct connection to the Diego de Almagro substation, located about 9 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 SEN, or one who will be connected with the SEN 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 has an agreement to purchase up to 50% of the annual production of copper concentrate and iron ore concentrate, leaving Capstone to market and sell the remaining concentrate. The Kores terms and conditions will reflect the Capstone terms negotiated independently in the market
- The Project will produce a high magnetite UF iron ore concentrate, suitable for pellet feed. Capstone has contacted 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 will be contacted in order to negotiate meaningful MoUs and eventually contracts
- Although more difficult, because it is a speciality product, Capstone may 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
- 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
- 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
- It is expected that Capstone's concentrate from the Santo Domingo Project will be in high demand from trading companies specialising in blending complex materials with clean materials

- No contracts are currently in place for Capstone's portion of the production for either the copper or iron ore concentrates
- No other contracts are in place for the Project.

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 were important during the environmental evaluation process included the impact of the Project on water resources, air quality and the marine and human environments
- A Closure Plan has been developed but has not yet been submitted. As the Project will produce >10,000 t/month, Capstone will be required to post a guarantee. A bond based on the estimated cost of closure must be submitted to the state
- Around 750 permits will be required in support of the Project. Some permits have been classed as critical because of the strategic importance to ensure the construction and start-up of the Project
- The EIA was submitted to the SEIA for approval on 30 October 2013. The assessment process took 22 months and the SEIA approved the environmental licence in July 2015 in RCA No. 119/2015, this is valid for 5 years until July 2020
- 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 Wood and ACCE International standards, with an accuracy of -10% to +15% at the 85% confidence level
- All capital cost estimates are in third-quarter 2018 US dollars
- The estimates are based on a combination of direct quotes, benchmarking and Capstone-supplied data
- The initial capital cost estimate is $1,512.3 M. The estimated sustaining capital cost is $378.6 M. The combined initial and sustaining capital costs for the LOM are estimated to be $1,890.9 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
- Total process operating costs over the LOM are estimated at $2,547.6 M; G & A costs at $402.8 M, and total mining costs at $2,619.6 M. The total operating cost over the LOM is estimated to be $5,570.0 M
- The LOM average operating cost is $1.34/lb CuEqproduced
- The LOM copper concentrate transport costs are estimated to be $47.3 M and are not included in the total above.

25.16 Economic Analysis

- Based on the assumptions in this Report, the Project has a positive cash flow
- On an after-tax basis, the cumulative net cash flow for the base case is $3,250.5 M, the IRR is 21.8% and the payback period is 2.8 years
- At an 8% discount rate, the after-tax NPV is $ 1,031.9M
- The Santo Domingo Project NPV8% is most sensitive to changes in the copper price (copper grade) and in total operating and capital expenditures. The sensitivity analysis showed that the Project is less sensitive to changes in the iron

price (iron grade) and the dollar/peso exchange rate, and least sensitive to changes in power costs.

25.17 Risks and Opportunities

- The most significant risks facing the Project were evaluated in a risk review in 2018 and include:
  - Delay in financing
  - Schedule delays
  - Contractor engagement and price uncertainty
  - Increased equipment and labor costs.
- Opportunities identified in 2018 include:
  - Include cobalt inclusion in the Mineral Reserve
  - Enhance gold recovery through further metallurgical testing
  - Incorporate autonomous haulage
  - Share infrastructure with other local companies
  - Consider autogenous grinding.

Risks and opportunities will be continuously assessed and reviewed throughout the various phases of the project, in accordance with Capstone's Risk Management Framework.

25.18 Conclusions

Under the assumptions outlined in this Report, 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 for work required prior to starting the EPCM phase 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. The estimated budget required to completre this work is between $4.6 M and $5.6.

26.2 RPA Recommendations

RPA recommends that additional assaying be conducted to verify the cobalt data in order to include cobalt in the Project economics. If possible, pulps or rejects from the original analytical work should be re-run, along with CRMs and blanks in order to provide the basis for this verification.

A review of the mineralogy and mode of occurrence of cobalt should be undertaken to better define the controls to mineralization. On completion of this work, the resource models should be reviewed on the basis of this information and revised where appropriate.

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 $300,000 to $350,000, assuming a third-party consultant performs the block model work, the cobalt mineralogy study and the review of the cobalt analytical QA/QC. If, on the basis of the cobalt studies, it is determined that new mineralization wireframes are required, an additional $100,000 to $125,000 should be budgeted.

26.3 Aminpro Recommendations

Aminpro recommends that a drilling campaign be conducted in order to generate sufficient sample material to operate a pilot plant to confirm final design criteria, metallurgical results and process alternatives for cobalt. It is estimated that this will cost approximately $2.5M.

26.4 Wood Recommendations

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 testwork with desalinated water to confirm reagent additions and recoveries.
- Conduct chemical optimization tests using various flotation frothers and collectors with the aim of reducing operating costs.
- Review the impact of grinding to a P80 of 150 µm.
- 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 testwork to 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.
- Confirm equipment sizing for flotation, regrind, filtration and copper concentrate transport.
- 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.

26.5 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 hydrological surveys to define river scour depths and improve special crossing design.

The 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 will improve pump sizing and pipe wall thickness distribution along the pipeline.

This work is estimated at $60,000.

26.6 Knight Piésold Recommendations

The following recommendations are made by Knight Piésold in support of Project detailed design.

- Review the water balance model and analysis with updated slurry solids contents and, together with any updated in-place tailings density values from the additional tailings testing and the estimated beach configuration, determine the final limits of the geomembrane liner within the TSF impoundment and the need for a deeper cut-off trench for anchoring the geomembrane liner.

The engineering associated with this work is estimated to be $300,000.

- The final design should include a review of the hydrology study to include the latest significant hydrological events in the area. This review may change the volume for the PMP and this may have an impact on the size of the dam.

The engineering associated with this work is estimated to be approximately $50,000 to $100,000.

- Consider the impacts of a flatter beach slope (1.5% is currently planned) which could result if the tailings slurry densities are lower than expected or the discharge system is less effective than contemplated. The freeboard at the main embankment must then be checked. The final (initial and sustaining) cost estimate for the embankment is not expected to change as a result of this, but the initial construction stage may require a higher dam which could increase the initial Capex cost.

The engineering associated with this work is estimated to be approximately $50,000 to $100,000.

- Include additional vibrating wire piezometers underneath the lined area of the TSF to monitor for potential seepage and assess the effectiveness of the lining system.
- Perform additional 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
  - ICU triaxial.

This testwork is estimated to require a budget of $35,000.

- Perform a large triaxial test on a representative sample of construction materials for the dam (Chilean authorities are asking for back-up 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 estimated 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.

- Perform additional site investigations to check the soluble salts content in the foundation area of the main dam. This program should include at least 10 test pits with a minimum depth of 6 m. Samples should be taken each meter for every test pit. The soluble salts content testing should extend over enough time to simulate the long-term exposure to flow that these soils may undergo.

The cost of these tests is estimated to be $100,000. Depending on the results of these tests, additional foundation excavations and backfill may be required for the construction of the main embankment.

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