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Technical Report on the Santo Domingo Project, Chile

28 September 2011

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Date and Signature Page

Project Name:	Santo Domingo Project
Title of Report:	Santo Domingo Pre-Feasibility Study Technical Report
Location:	Region III, Chile
Effective Date of Report:	28 September 2011
Completion Date of Report:	28 September 2011

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Date Signed: 28 September 2011

"John Nilsson"
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CERTIFICATE OF AUTHOR

David Brimage

To accompany the report entitled, "Technical Report on the Santo Domingo Project, Chile" prepared for Capstone Mining Corp., and dated September 28, 2011 ("Technical Report").

I, David Brimage, MAusIMM CP, do hereby certify that:

1. I am Manager Process for Ausenco Solutions Canada Inc. 855 Homer Street, Vancouver, BC V6B 2W2, Canada.
2. I graduated with a degree in Metallurgical Engineering (Metallurgy) from the University of South Australia in 1993.
3. I am a Chartered Professional with the AusIMM.
4. I have worked as a Metallurgist continuously since my graduation from University. For the past 15 years I have been employed with Ausenco Minerals and Metals. During this period I have fulfilled roles as senior process engineer, principal process engineer, engineering manager, and am currently employed as Manager Process.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101) and certify that by reason of education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purpose of this NI 43-101.
6. I have participated in the preparation of Sections 1, 2, 3, 4, 5, 17, 18, 19, 20, 21, 22, 23, 24 (except 24.1), 25, 26, 27 of this technical report.
7. I have had no prior involvement with the property or project. I visited the property during February 8 to10, 2011.
8. Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of and Capstone Mining Corp., or any associated or affiliated entities.
9. Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of Capstone Mining Corp., or any associated or affiliated companies.
10. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from Capstone Mining Corp., or any associated or affiliated companies.
11. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
12. I have read National Instrument 43-101 and Form 43-101F1, and confirm that the Technical Report has been prepared in compliance with that instrument and form.
13. To the best of my knowledge, information and belief, this technical report contains all the scientific and technical information that is required to be disclosed to make this technical report not misleading.

Dated this 28th day of September, 2011

– Original Signed –

"D.J. Brimage"
David John Brimage, MAusIMM CP.

CERTIFICATE OF AUTHOR

David W Rennie

I, David W. Rennie, P.Eng., as an author of this report entitled "Technical Report on the Santo Domingo Property, Chile", prepared for Capstone Mining Corp., and dated September 28, 2011, do hereby certify that:

1. I am a Principal Geologist with Roscoe Postle Associates Inc. of Suite 388, 1130 West Pender St., Vancouver, BC, V6E 4A4.
2. I am a graduate of the University of British Columbia, Vancouver, BC, Canada, in 1979 with a Bachelor of Applied Science degree in Geological Engineering.
3. I am registered as a Professional Engineer in the Province of British Columbia (Reg.# 13572). I have worked as a Geological Engineer for a total of 32 years since my graduation. My relevant experience for the purpose of the Technical Report is:

   - Review and report as a consultant on numerous mining operations and projects around the world for due diligence and regulatory requirements.
   - Pre-Feasibility and Feasibility Study work on several projects.
   - Worked as a Geological Engineer at several mines and exploration projects in a number of countries

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Santo Domingo Property on June 14–16, 2010.
6. I am responsible for Sections 6 to 12, inclusive and Section 14 of the Technical Report.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I prepared previous Mineral Resource estimates and Technical Reports on the Project in 2007 and 2010.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. To the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28th day of September, 2011

– Original Signed –

"David W. Rennie"
David W. Rennie, P.Eng.

CERTIFICATE OF AUTHOR

John Nilsson

To accompany the report entitled, “Technical Report on the Santo Domingo Project, Chile” prepared for Capstone Mining Corp., and dated September 28, 2011 (“Technical Report”).

I, John Nilsson P.Eng. do hereby certify that:

1. I am President of Nilsson Mine Services Ltd. 20263 Mountain Place, Pitt Meadows, B.C., V3Y 2T9, Canada
2. This certificate applies to the Technical Report entitled "Technical Report on the Santo Domingo Project, Chile”, dated September 28, 2011
3. I graduated with a Bachelors degree in Geology from the Queen’s University in 1977. In addition, I obtained a Masters degree in Mining Engineering from the Queen’s University in 1990.
4. I am a member of the Association of Profession Engineers of British Columbia. I have worked as a geologist and mining engineer for a total of 33 years since my graduation from university. I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, past relevant work experience and affiliation with a professional association to be a “qualified person” for the purposes of NI 43-101.
5. I have visited the property on June 28 and June 29, 2011.
6. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101
7. I have had no prior involvement with the property that is the subject of the Technical Report.
8. I have participated in the preparation of Sections 15, 16.3 and 16.4.
9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
10. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific information that is required to be disclosed to make the report not misleading
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 28th Day of September, 2011.

– Original Signed –

“John Nilsson”

John Nilsson, P.Eng.

CERTIFICATE OF AUTHOR

Art Winckers

To accompany the report entitled, "Technical Report on the Santo Domingo Project, Chile" prepared for Capstone Mining Corp., and dated September 28, 2011 ("Technical Report").

I, Arthur H. Winckers, P. Eng. BC. FEC do hereby certify that:

1. I am the Principal of Arthur H. Winckers & Associates Mineral Processing Consulting Inc. located at 4345 Raeburn Street North Vancouver BC V7G 1K1 Canada.
2. I graduated with a degree in Mining Engineering from the Technical University of Delft in the Netherlands in 1965.
3. I am a Professional Engineer registered in the Province of British Columbia (Reg. # 8693)
4. I have worked as a Metallurgist for a total of 46 years since graduation from University.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101) and certify that by reason of education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purpose of NI 43-101.
6. I am responsible for preparation of Section13.
7. I have not visited the Santo Domingo property.
8. I have not had prior involvement with the property that is the subject of the report.
9. To the best of my knowledge, information and belief, this technical report contains all the scientific and technical information that is required to be disclosed to make this technical report not misleading. I am independent of the issuer applying all of the tests in section 1.5 of the National instrument 43-101.
10. I have read National instrument 43-101 and Form 43-101F, and Section 13 of the Technical Report has been prepared in compliance with that instrument and form.
11. I consent to the filing of the Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public compant files on their websites accessible by the public, of the Technical Report.
12. The Technical Report contains information relating to mineral titles, permitting, environmental issues, regulatory matters and legal agreements. I am not qualified to offer a professional opinion regarding these issues.

Dated this day 28th of September 2011.

– Original Signed –

"Art Winckers"
Arthur Winckers, P.Eng. BC

CERTIFICATE OF AUTHOR

Michael Davies

To accompany the report entitled, "Technical Report on the Santo Domingo Project, Chile" prepared for Capstone Mining Corp., and dated September 28, 2011 ("Technical Report").

I, Michael Davies, P.Eng./P.Geo., do hereby certify that:

1. I am Vice-President, Mining, and a Principal Geotechnical Engineer of AMEC Environment & Infrastructure, Suite 600, 4445 Lougheed Highway, Burnaby, British Columbia, Canada, V5C 0E4
2. I graduated with a degree in Geological Engineering from the University of British Columbia in 1985. In addition, I have obtained a Masters of Applied Science in Civil Engineering (Geotechnical) from the University of British Columbia in 1988 and a Ph.D. in Civil Engineering (Geotechnical) from the University of British Columbia in 1999.
3. I am a member of the Canadian Geotechnical Society, the American Society of Civil Engineers and the Canadian Institute of Mining and Metallurgy. I am a registered Professional Engineer (P.Eng.) and registered Professional Geoscientist (P.Geo.) in the Province of British Columbia.
4. I have worked as an engineer for a total of 26 years since my graduation from university.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for the preparation of the geotechnical and hydrology/hydrogeology portions of Sections 16.1, 16.2 and 24.1 of the technical report titled Technical Report on the Santo Domingo Project, Chile and dated September 2011 (the "Technical Report") relating to the Santo Domingo property.
7. I have not had prior involvement with the property that is the subject of the Technical Report.
8. I have not visited the Santo Domingo property.
9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
10. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 28th of September, 2011.

– Original Signed –

"Michael Paul Davies"

Michael Paul Davies, P.Eng., P.Geo.

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Important Notice

This Santo Domingo Project Technical Report (Technical Report or Report) was prepared for Far West Mining Ltd. (FWM), which was 100% aquired by Capstone Mining Corp. (Capstone) by Ausenco Minerals Canada Inc (Ausenco) to provide Capstone with sufficient information to determine the economic feasibility of developing the Santo Domingo deposits. The Santo Domingo deposits are 70% owned by Capstone and 30% by Korea Resources Corporation ("KORES").

This report describes the assumptions made in preparing the prefeasibility study and is based on information provided to Ausenco by FWM and other organizations nominated by FWM.

The results and opinions expressed in this report are based on the observations and the technical data listed in the report. Whilst Ausenco has reviewed all of the information provided by others, and believes the information to be reliable, Ausenco has not conducted an in-depth independent investigation to verify its accuracy and completeness.

The results and opinions expressed in this report are conditional upon the aforementioned information being current, accurate, and complete as of the date of this report, and the understanding that no information has been withheld that would affect the conclusions made herein.

Capstone is permitted to file this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to provincial securities legislation. Except for the purposes legislated under provincial securities laws, any other use of this Report by any third party is at that party's sole risk.

The Technical Report is to be read as a whole, and sections or parts of it should not be read or relied upon out of context. This notice, which is an integral part of the Report, must accompany every copy of the Report.

1 SUMMARY

1.1 Introduction

Far West Mining (FWM) was an international mineral exploration company headquartered in Vancouver, Canada. The Company was primarily engaged in the evaluation, acquisition, and exploration of mineral properties in Chile and Australia. On June 17th of 2011, Far West Mining was 100% aquired by Capstone Mining Corp. (TSX:CS). The Santo Domingo deposit is 70% owned by Capstone and 30% by Korea Resources Corporation ("KORES").

The Santo Domingo deposit is located at approximately 26°28'00"S and 70°00'30"W, 50 km west of the El Salvador mega porphyry copper deposit, and 130 km north northeast of Copiapó in Region III, Chile, see Figure 1-1. Elevation at the site ranges from 1,000 m to 1,280 masl, with relatively gentle topographic relief. Access to the project is 1 kilometre off the paved highway C-17 from Diego de Almagro to Copiapó. Regional infrastructure is good and highways connect the site to the main regional towns and cities. Regularly scheduled air services are available between Santiago (capital of Chile) and El Salvador and the Atacama airport located near the Caldera port, northwest of Copiapó.

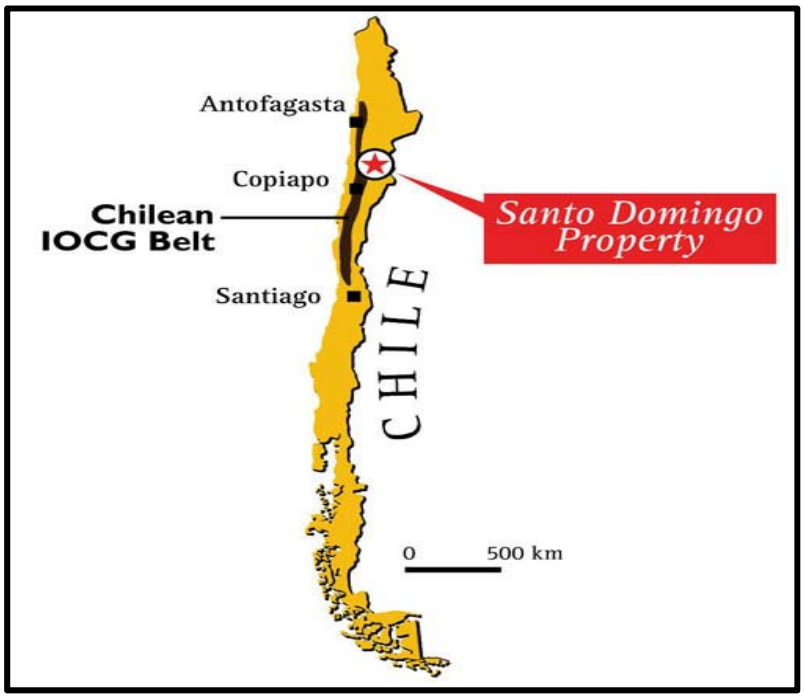


Figure 1-1: Location of the Santo Domingo Project

1.2 Study Objectives

Ausenco was contracted by FWM to complete a prefeasibility study (PFS) to provide an independent economic analysis and schedule for the project. The scope of work includes an open pit mine, conventional copper flotation and magnetite magnetic separation processing plant, seawater and concentrate pipelines, a port facility consisting of concentrate dewatering, storage and ship load out, associated services and utilities, supporting infrastructure, tailings storage facility (TSF), and waste stockpiles.

Several participants address individual areas of the study. Ausenco is responsible for the overall study coordination, long-distance pipelines, port facilities, tailings storage facility (TSF), plant geotechnical, and processing plant design. AMEC is responsible for the mine geotechnical design. Arthur H. Winckers & Associates Inc. is responsible for the metallurgical testwork and predictions, and Nilsson Mine Services Limited (NMS) is responsible for the reserve estimate and mine design with support from SRK Consulting (SRK). Roscoe Postle Associates Inc. (RPA), formerly Scott Wilson RPA, is responsible for the description of deposit history, geological setting, deposit types and mineralization and resource estimation.

This report is based on a nominal plant throughput rate of 63.5 kt/d and has developed preliminary pit shell designs, PFS-level designs for the mine, processing plant, pipelines, and port facilities, and PFS-level TSF design, sizing and scheduling as well as operating costs for each of these areas.

1.3 Study Basis

The basis of the PFS was for a target design mill feed rate of 63.5 kt/d, based on the use of conventional SAG, ball mill and pebble crushing comminution circuit (SABC), flotation technology to produce copper concentrates and magnetic separation to produce magnetite concentrates for sale.

Specifically, the study estimates the capital and operating costs of an operation consisting of conventional open pit mining, an SABC comminution circuit, conventional copper flotation, magnetic separation, tailings disposal and storage, water and concentrates pipelines and port facilities, and associated site infrastructure requirements. The capital and operating cost estimates are defined by the study scope and battery limits.

1.4 Resource and Reserves

The largest and most extensively mined iron oxide-copper-gold (IOCG-type) deposits in Chile occur within a structurally complex zone 630 km by 25 km extending between La Serena and Taltal. The Santo Domingo project lies within this zone approximately 74 km east of the Chañaral sea port.

Copper-bearing mineralization is widespread in the Santo Domingo area. Specular hematite, magnetite and copper oxides (including chrysocolla, brochantite, and malachite) are the

typical near-surface mineral assemblages. Copper oxides typically persist to 70 to 90 m below surface, with chalcopyrite being the dominant copper mineral at greater depths.

The Santo Domingo project currently comprises of the following deposits: Santo Domingo Sur (SDS), Iris, (currently grouped together as “SDS/IRIS”), and Iris Norte.

In the SDS deposit, copper mineralization occurs in a sequence of iron oxide mantos within a tuffaceous package between andesitic flows. Drilling has identified a 150 to 500 m thick, copper-bearing, specularite-magnetite sequence covering an area of approximately 1,300 m by 800 m.

The Iris deposit is approximately 500 m wide, with a strike length of 1,600 m. The deposit consists of iron oxide mantos and breccias along a north-northwest-striking fault zone. Mineralization occurs close to surface at the southern end and plunges gently towards the north.

In August 2010, RPA updated the Mineral Resource estimates for the SDS and Iris Zones[1]. The estimates include data from recent measurements of magnetic susceptibility, as well as 35 additional drill holes completed since the last estimate, which was carried out by RPA in 2009 (Lacroix, 2009). The updated estimate is summarized in Table 1.1.

Based on the analysis of a Whittle™ pit optimization evaluation for varying revenue factors the chosen Whittle™ shell was used as the basis for the detailed pit designs created for each of the Santo Domingo pits. These detailed pit designs take into consideration, minimum mining widths, access ramps, and detailed bench configurations.

The mineral reserves estimate for the detailed open pit designs are summarized in Table 1.2 for the probable reserve classification. The Santo Domingo Sur and Iris deposits formed the SDS/Iris open pit with Iris Norte forming its own pit. These open pits were then further divided into various stages for mine planning purposes. SDS/Iris pit is divided into four stages, while Iris Norte has been divided into three stages.

[1] *Santo Domingo Property Technical Report NI 43-101 – August 26, 2010*

Table 1.1: Indicated and Inferred Mineral Resources (15 May 2010)

Zone	Mt	% CuEq	% Cu	g/t Au	%Fe
Indicated					
SDS	275	0.64	0.41	0.056	27.8
Iris	111	0.50	0.23	0.033	26.3
Iris Norte	99.5	0.47	0.16	0.019	26.4
Total Indicated	**486**	**0.57**	**0.32**	**0.043**	**27.2**
Inferred					
SDS	30.5	0.46	0.26	0.037	23.7
Iris	5.52	0.47	0.19	0.026	26.0
Iris Norte	25.3	0.47	0.10	0.011	27.9
Total Inferred	**61.3**	**0.46**	**0.19**	**0.025**	**25.7**

Notes: CIM definitions were followed for Mineral Resources. Mineral Resources for SDS/Iris are estimated at a cut-off grade of 0.25% CuEq. CuEq grades are calculated using average long-term prices of US$2.25/lb Cu, US$950/oz Au, and US$0.74/dmtu Fe ($50/dmt conc. @ 67.5% Fe). CuEq calculations and metallurgical recovery factors are as stated in the RPA Technical report. Due to rounding, some figures may not add up to the totals shown. This study does not include the Estrellita zone (see Table 14.1, Section 14). Mineral Resources are inclusive of Mineral Reserves.

Table 1.2: Santo Domingo Open Pit Mineable Probable Reserves

		Ore Grade		Contained Metal		
Stage	**Ore (Mt)**	**Au (g/t)**	**Cu (%)**	**Au (kOz)**	**Cu (Mlbs)**	**Magnetite Conc. (Mt)**
SDS/Iris						
SDS Stage1	71.8	0.08	0.61	193	958	11
SDS Stage2	63.7	0.06	0.41	113	574	10
SDS Stage3	170.5	0.03	0.23	173	848	32
SDS Stage4	38.8	0.05	0.36	60	304	3
Subtotal SDS/Iris	**344.8**	**0.05**	**0.35**	**539**	**2,684**	**57**
Iris Norte						
IRN Stage 1	21.4	0.03	0.23	20	108	4
IRN Stage 2	28.0	0.01	0.13	12	78	7
IRN Stage 3	23.7	0.01	0.11	8	60	5
Subtotal Iris Norte	**73.1**	**0.02**	**0.15**	**41**	**246**	**17**
Grand Total	**418.0**	**0.04**	**0.32**	**580**	**2,930**	**73**

Notes: NSR cut-off of $5.79/t (incremental operating cost; does not include mining costs). Reserves are based on Indicated Resources only. Magnetite concentrate tonnage is based on average 65% iron grade. Due to rounding, some figures may not add up to the totals shown.

Within the pit designs there is a total of 8 Mt of inferred mineral resources. These inferred tonnes were not included in the life-of-mine production plan. There is no certainty that these inferred mineral resources will be converted to the measured or indicated categories through

further drilling, or into mineral reserves, once economic considerations are applied. There is also 31 Mt of oxide material that has not been included in the life-of-mine plan. This oxide material will be selectively placed on the waste rock fill to allow for potential processing in the future.

1.5 Mineral Processing

The deposits consist of the following zones of mineralization and are based on the predominant iron oxide mineral:

- Magnetite Core
- Hematite Rim, surrounding Magnetite Core
- Iris Mag, magnetite–rich, low-copper zone north east of Hematite Rim
- Iris, a hematite-rich northern extension, higher in copper
- Iris Norte, a magnetite-rich zone north of Iris.

The Magnetite Core and the Hematite Rim zones contain the highest grade copper mineralization.

The PFS flotation testwork program at SGS Lakefield was performed on the following composites and samples:

- 8-year composite
- Hematite composite
- Magnetite composite
- Oxide composite (oxide material is considered waste in the PFS mine plan)
- Variability samples.

Flotation optimization studies were performed on the 8-year composite. The optimized conditions were applied to the hematite and magnetite composites and the variability samples. In addition, copper flotation tailings were produced for magnetic iron recovery testwork. The testwork was undertaken in Germany by the iron ore research centre Studiengesellschaft für Eisenerz-Aufbereitung (SGA). Bulk copper rougher and cleaner concentrates were also produced for testwork by process equipment suppliers to determine regrind power and concentrate filter size.

Earlier phases of flotation test work had been conducted in fresh water. However, following on from the decision to use sea water for flotation in the plant, all the flotation and magnetite recovery tests in this test work phase were conducted in synthetic sea water. The projections of metallurgical performance, including recovery, presented in this study are based entirely on testwork using seawater.

Since locked cycle tests were not conducted on the variability samples scale-up to full-scale operation has been based on the results of the locked cycle tests on the 8-year pit, hematite and magnetite composites. The average copper results of the batch and locked cycle tests are shown in the Table 1.3.

Table 1.3: Average of Batch and Locked Cycle Test Results - Cu

	Head %Cu	Rougher Recovery		Cleaner Recovery	Total Recovery
		Wt %	Cu %	Cu %	Cu %
Variability Samples-Batch Average	0.36	9.4	93.1	89.1	83.0
Composite Samples-Batch Average	0.34	11.6	93.1	95.0	88.4
Composite Samples-LCT Average.	0.34	12.2	93.4	96.5	90.7

Magnetite recovery testwork at SGA was conducted on copper rougher flotation tailings samples produced at SGS Lakefield. Conditions were optimized for the 8-year composite and applied to the hematite and magnetite composites. A series of tests were performed on variability samples to determine the correlation between Davis Tube test results and LIMS cleaner tests and to determine the correlation between Satmagan/magnetic susceptibility head grade and Davis Tube test recovery.

Table 1.4 shows a summary of each composite recovery and grade.

Table 1.4: LIMS Cleaner Concentrate

Composite	Mag Fd. Grade %	Wt. %	Fe Grade %	Fe Rec. %	SiO_2 Grade %	Na_2O Grade %	K_2O Grade %	Mag. Grade %	Mag. Rec. %
8-Year Pit Comp.	15.2	16.8	66.1	38.2	4.1	0.145	0.105	81.6	90.1
Hematite Comp.	9.4	9.8	64.1	24.9	5.7	0.33	0.153	71.2	74.3
Magnetite Comp.	27.5	29.8	66.5	66.8	4.4	0.225	0.09	84.8	92.1

High grade LIMS concentrates at high magnetite recoveries were produced from the 8-year pit and magnetite composites. Further work is required to improve the concentrate grade from the lower grade hematite composite.

The copper and magnetite recovery plant and associated service facilities will process ROM ore delivered to the primary crusher. The proposed process encompasses crushing and grinding of the ROM ore, copper flotation (in seawater), and magnetite recovery on copper rougher tailings. Copper and magnetite concentrates will be thickened on site prior to being pumped via concentrate pipeline to the port. At the port, the concentrates will be washed, dewatered and loaded onto ships for transportation to third-party smelters. The process flow diagram is described in Figure 1-2.

The key criteria selected for the plant design are:

- nominal plant treatment rate of 60 000 t/d (60 kt/d), with the ability to handle increased throughputs of up to 70 kt/d for softer ores. (The average life-of-mine throughput is 63.5 kt/d.)
- design availability of 93% (after ramp-up), being 8,147 operating hours per year, with standby equipment in critical areas

- sufficient plant design flexibility for treatment of all ore types at design throughput
- production rates of 4.0 Mt/a of magnetite concentrate and 250,000 t/a of copper concentrate. (The average life-of-mine production rates are predicted to be 4.1 Mt/a of magnetite concentrate and 225,000 t/a of copper concentrate.

The tailings storage system consists of a TSF located north of the proposed mine. The TSF is designed to store approximately 353 Mt of conventional thickened tailings, enough for approximately 18 years of the project life. Storage of both fresh and seawater is proposed to be in lined ponds near the plant site. No other water storage reservoir is proposed. Water make-up is proposed to be untreated seawater. Based on the conventional thickened tailings disposal method, the estimated water make-up will be approximately 1,450 m^3/h (~400 L/s).

The TSF includes a starter dam for storing at least two years of thickened tailings. The starter dam crest will be raised in stages by the downstream method to contain the waste tailings within the current permitted boundary limits up to Year 18 of operations.

Basic layouts have been prepared based on an open-air concentrator design, with mobile crane maintenance access and minimal overhead cranage. This layout has taken account of the site topography and limits imposed by the preliminary locations of the pit, stockpiles, and waste dumps.

1.6 Infrastructure

Access to the mine site is 6 km to the south of Diego de Almagro on C-17. This section is paved and in good condition. Due to the location of the Iris Norte pit and process facility, approximately 2 km of the existing road will require diversion and an overpass. The overpass will allow vehicle access to the TSF without crossing Highway C-17.

High-voltage transmission will be achieved using a 220 kV double-circuit overhead line from Diego de Almagro. The high voltage power line will feed a 220/13.8 kV transformer yard at the project main substation. Correspondence with the Chilean government organisation for the coordination of electrical installations, SDEC-SIC, suggests there is sufficient 220 KV power available to meet the demands of the project. The SDEC-SIC has recommended further study work is required to confirm this.

The PFS is based on the port facility being located approximately 74 km west of the Santo Domingo project. Both the concentrate and seawater pipelines will run together in a pipeline corridor along the north side of the C-57 road and Route 5. This section includes the El Salado River crossing and El Salado town bypass. The ground corresponds to gravel, clay and sand with no high points and a continuous slope.

The port terminal will be required to serve vessels with the following geometry to a maximum LOA of 250 m and maximum draught of 18.28 m.

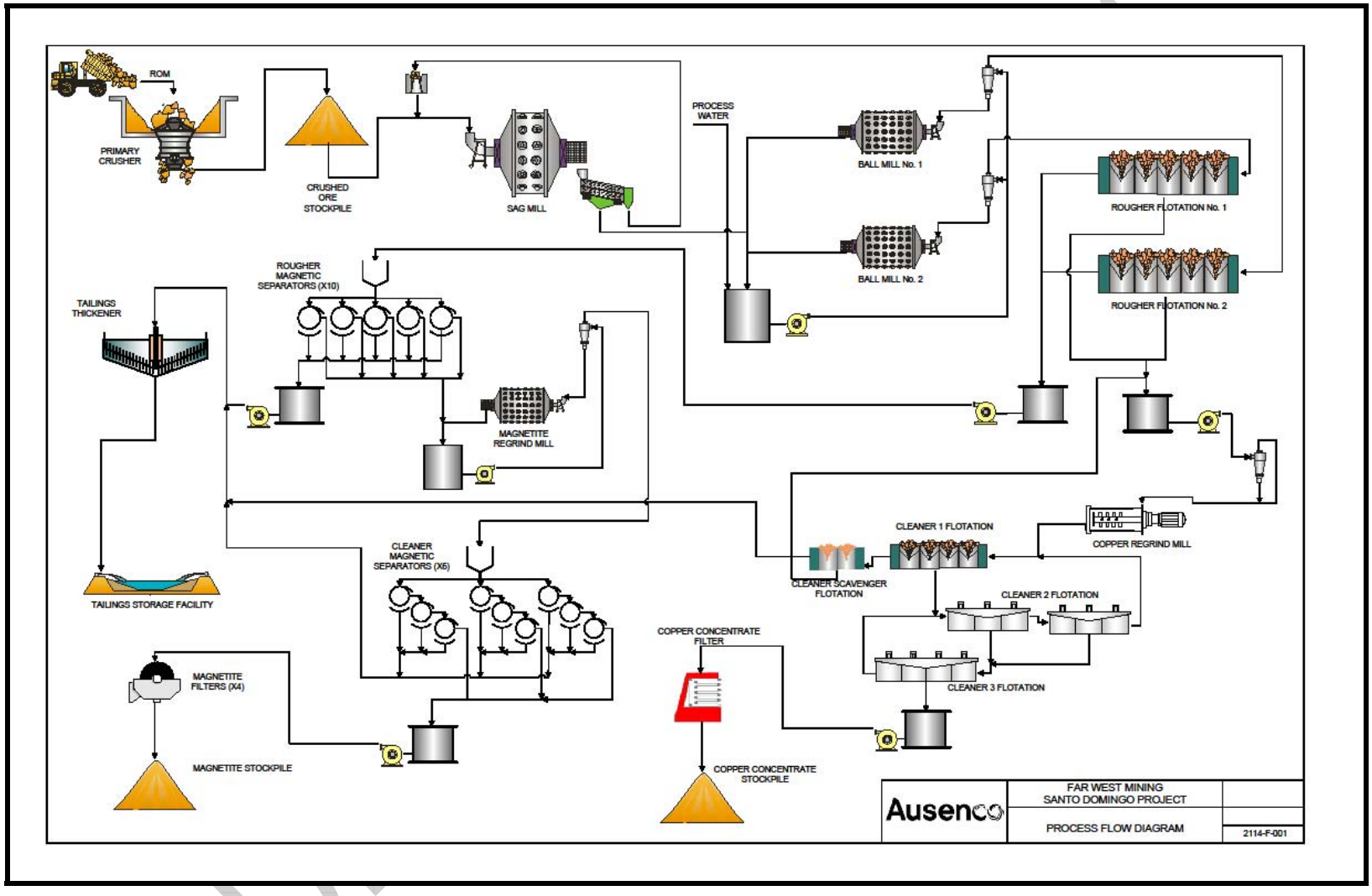


Figure 1-2: Santo Domingo Process Plant Flow Diagram

The port facility will receive magnetite and copper concentrate by pipeline and will operate 24 hours per day, seven days per week. The magnetite and copper concentrate will be handled at an open stockyard and an enclosed storage building, respectively.

The port facilities include:

- concentrate pipeline choke station
- a filter plant for concentrate dewatering and associated settling ponds
- maintenance facilities to support the port operations
- stockyards' inloading conveying systems
- magnetite and copper concentrate stockyards with storage capacity of 250,000 tonnes and 40,000 tonnes, respectively
- out-loading conveying systems from the stockyards to the new ship loading system
- off-shore infrastructure including trestle, ship loader, and mooring systems.

1.7 Environmental

The land and territory investigations regarding the project's current footprint, indicate there would be no impact on natural parks, biodiversity conservation priority sites, or indigenous development land in the Atacama Region. A series of baseline studies are still required for the project in order to achieve a proper characterization of the environmental components that should be included in the future Environmental Impact Study (EIS).

1.8 Financial

The total project capital cost estimate is summarized in Table 1.5 and have ±25% accuracy as of July 2011. The estimate is based on a foreign exchange rate of 1 US$ = 466 Chilean Pesos (CLP) and must be assessed against the study battery limits, exclusions and scope as detailed in the relevant sections of this report.

Table 1.5: Summary of Capital Costs

Area		$M
Mining		172
Pre-strip		54
Process plant		283
Tailings		29
On-Site Infrastructure		27
Site Power	6	
Concentrate Pipeline	49	
Seawater Pipeline	76	
Concentrate Dewatering, Storage and Load Out	121	
Off-Site Infrastructure (Total)		253
Total Direct Costs		**818**
Indirect Costs		186
Owners Cost		89
Total Indirect Costs		**275**
Contingency		**149**
Total Project Cost		**1,242**

The total project operating costs, excluding costs associated with concentrate sales, are summarized in Table 1.6. The costs are presented as life-of-mine (LOM) averages per tonne of ore processed.

Table 1.6: Summary of Average LOM Operating Costs

Cost Centre	US$M/a	US$/t ore
Mining	107	4.62
Process plant	101	4.37
Concentrate pipeline	2	0.09
Seawater pipeline	10	0.43
G&A	13	0.55
Port Facility	11	0.46
Total	**244**	**10.52**

The operating costs estimate was prepared with a base date of July 2011 to an accuracy level of ±25%.

Life of mine sustaining capital costs, estimated at $495 million over the 18 year mine life (including mine closure estimates) are not included in either the initial capital or operating cost figures above. The sustaining capital expenditure requirements have been included as part of the financial model.

The overall economic performance of the project (as measured by the IRR, NPV and payback period) is summarized in Table 1.7. Base case and spot price economic models

were developed. These models were based on the commodity prices, and operating and capital costs listed in Table 1.7.

Table 1.7: Summary of Inputs into Economic Model

Parameter	Base	Spot
NPV discount rate, %	8	
Base Copper price, US$/lb	2.50	4.00
Base Magnetite price, US$/dmtu Fe[2]	1.00	2.00
Base gold price, US$/oz	1,000	1,400
Base capital cost, US$M	1,242	
Site Operating Cost, US$M	4,403	
Sustaining capital cost, US$M	495	
Realisation Costs, US$M	1,091	

The total cash production costs for copper over the life of the project are estimated at $0.11 per pound of payable copper, when including gold and iron production as credits and selling costs. The co-product total cash production costs are estimated at $1.12 per pound of payable copper and $30.46 per tonne of magnetite concentrate.

A summary of the financial model is presented in Table 1.8. The base case economic performance of the project (as measured by the IRR, NPV and payback period) is summarized in Table 1.9. The base case analysis considers copper and gold prices significantly below the current spot price, providing significant upside to the project. Spot price economics are summarised in Table 1.10.

[2] $1.00/dmtu Fe is the equivalent of $65/dmt of concentrate at 65.0% Fe and $2.00/dmtu Fe is the equivalent of $130/dmt of concentrate

Table 1.8: Summary of Project Cash Flow, before and after tax

ITEM	UNIT	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Years 11 to 15	Years 16 to 20		TOTAL
PRODUCTION																		
ANNUAL PRODUCTION SCHEDULE	'000 tonnes				15,330	25,550	25,550	25,550	25,550	25,462	24,424	23,910	24,355	23,977	113,637	64,661		417,956
METAL PRODUCED																		
- Copper in Concentrate	tonnes				88,411	147,172	133,714	114,619	93,767	76,453	59,668	56,458	48,865	45,727	230,525	83,294		1,178,672
- Magnetite Concentrate	'000 tonnes				1,584	2,849	3,165	4,412	4,228	4,210	3,255	3,410	4,383	4,431	23,093	14,092		73,112
- Gold Production	oz				28,095	44,375	38,409	29,791	22,890	17,204	11,469	10,230	8,563	7,683	41,343	7,534		267,585
REVENUE																		
MAGNETITE CONCENTRATE BASE PRICE	$US/dmtu Fe				1.00	1.00	1.00	1.00	1.00		1.00	1.00	1.00	1.00	1.00	1.00		1.00
COPPER BASE PRICE	$US/lb				2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50		2.50
GOLD BASE PRICE	$US/oz				1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000		1,000
COPPER CONCENTRATE																		
Copper Gross Metal Value	$USM				470	783	711	610	499	407	317	300	260	243	1,226	443		6,271
Gold Gross Metal Value	$USM				27	43	37	29	22	17	11	10	8	7	40	7		260
MAGNETITE CONCENTRATE	$USM				103	185	206	287	275	274	212	222	285	288	1,501	916		4,752
TOTAL GROSS METAL VALUE	$USM				601	1,011	954	925	796	697	540	532	553	539	2,768	1,366		11,282
OPERATING COSTS																		
TOTAL SITE OPERATING COSTS	**$US/tonne ore**				**13.68**	**9.73**	**10.02**	**10.05**	**10.19**	**10.20**	**10.71**	**10.55**	**10.32**	**10.86**	**10.99**	**9.85**		**10.53**
REALISATION COSTS	**$US/tonne ore**				**4.20**	**4.23**	**3.94**	**3.68**	**3.12**	**2.68**	**2.18**	**2.16**	**2.11**	**2.06**	**2.23**	**1.81**		**2.61**
TOTAL UNIT OPERATING COSTS	**$US/tonne ore**				**17.88**	**13.96**	**13.96**	**13.73**	**13.31**	**12.89**	**12.88**	**12.72**	**12.43**	**12.92**	**12.97**	**14.78**		**13.14**
TOTAL OPERATING COSTS																		
TOTAL SITE OPERATING COSTS	$USM				209.8	248.6	256.0	256.8	260.4	259.8	261.5	252.3	251.4	260.4	1,249.2	636.6		4,402.8
TOTAL REALISATION COSTS	$USM				64.3	108.0	100.7	93.9	79.6	68.3	53.1	51.8	51.3	49.5	252.9	117.2		1,090.7
TOTAL OPERATING COSTS	**$USM**				**274.1**	**356.6**	**356.7**	**350.7**	**340.0**	**328.1**	**314.7**	**304.1**	**302.7**	**309.9**	**1,502.1**	**753.8**		**5,493.5**
CAPITAL COSTS																		
TOTAL INITIAL CAPITAL	**$USM**	28.6	501.1	712.6														
TOTAL DEFERRED/SUSTAINING CAP.	**$USM**				**19.1**	**34.9**	**9.4**	**17.1**	**9.4**	**21.0**	**47.8**	**63.2**	**46.0**	**29.6**	**75.8**	**121.6**		**495**
TOTAL CAPITAL	**$USM**	**28.6**	**501.1**	**712.6**	**19.1**	**34.9**	**9.4**	**17.1**	**9.4**	**21.0**	**47.8**	**63.2**	**46.0**	**29.6**	**75.8**	**121.6**		**1,737**
TOTAL PROJECT CASHFLOWS																		
PROJECT PRETAX CASHFLOW	$USM	-29	-501	-713	307	620	588	558	446	348	178	165	204	199	1,190	491		4,051
CORPORATE TAXATION	$USM	0	0	0	35	86	77	73	54	42	20	19	22	31	179	87		726
PROJECT AFTER TAX CASHFLOW	$USM	-35	-611	-861	525	495	480	454	372	291	150	136	173	157	948	374		3,048
PROJECT AFTER TAX CASHFLOW @ 8% DR RATE	$USM	-35	-565	-738	416	364	326	286	217	157	75	63	74	62	298	91		1,093
PRODUCTION STATISTICS																		
NET REVENUE /TONNE ORE TREATED	$US/tonne ore				35.0	35.3	33.4	32.5	28.0	24.7	19.9	20.1	20.6	20.4	22.1	19.3		24.4
TOTAL CASH COST / lb OF PAYABLE COPPER[3]	$US/lb Cu				0.71	0.35	0.34	0.07	0.14	0.15	0.65	0.52	-0.01	0.05	-0.15	-1.10		0.11

[3] Inclusive of by-product credits.

Table 1.9: Base Case Economic Analysis

Economic Parameters	EBITD&A	After Tax
NPV (US$M @ 8%)	1,620	1,092
IRR%	29.4	22.0
Simple Payback Period (years)	2.5	3.0
Total Cash Cost (US$ per lb of Cu)[4]	0.11	

Table 1.10: Spot Price Case Economic Analysis

Economic Parameters	EBITD&A	After Tax
NPV (US$M @ 8%)	5,472	3,985
IRR%	60.2	45.4
Simple Payback Period (years)	1.5	1.7
Total Cash Cost (US$ per lb of Cu)[1]	-1.75	

Variability analyses were conducted using different metal prices and varying capital and operating costs to determine the effect on the project economics. Figure 1-3 and Figure 1-4 and Table 1.11, summarise the findings of the sensitivity analysis.

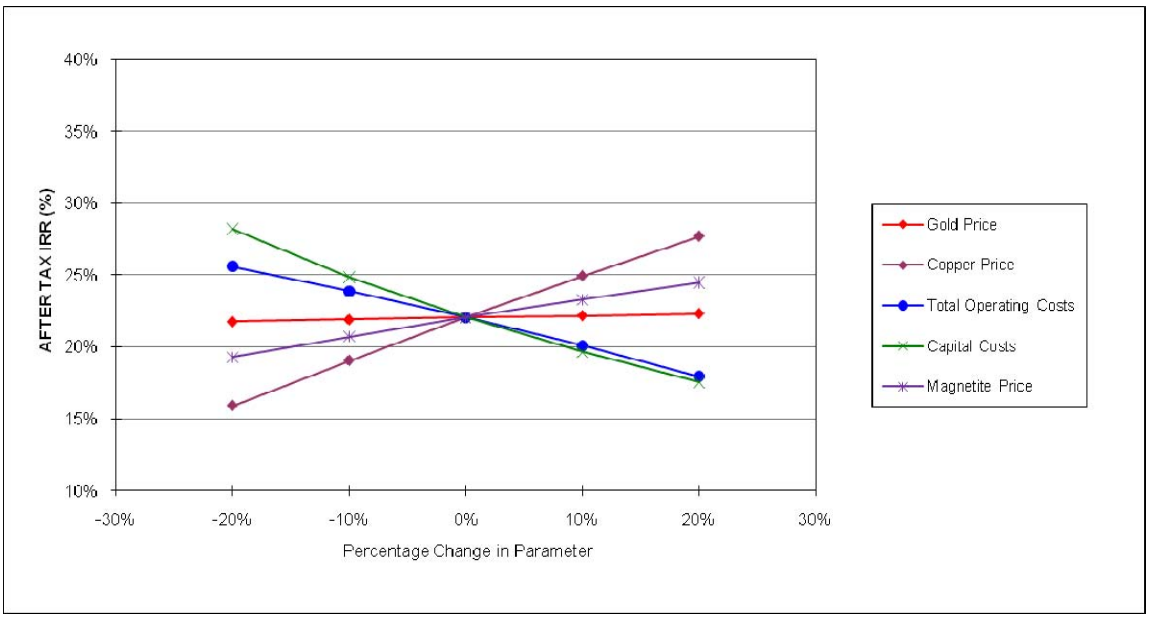


Figure 1-3: IRR Sensitivity

[4] Total Cash Production Costs (per lb of payable Cu) are inclusive of by-product credits and selling costs..

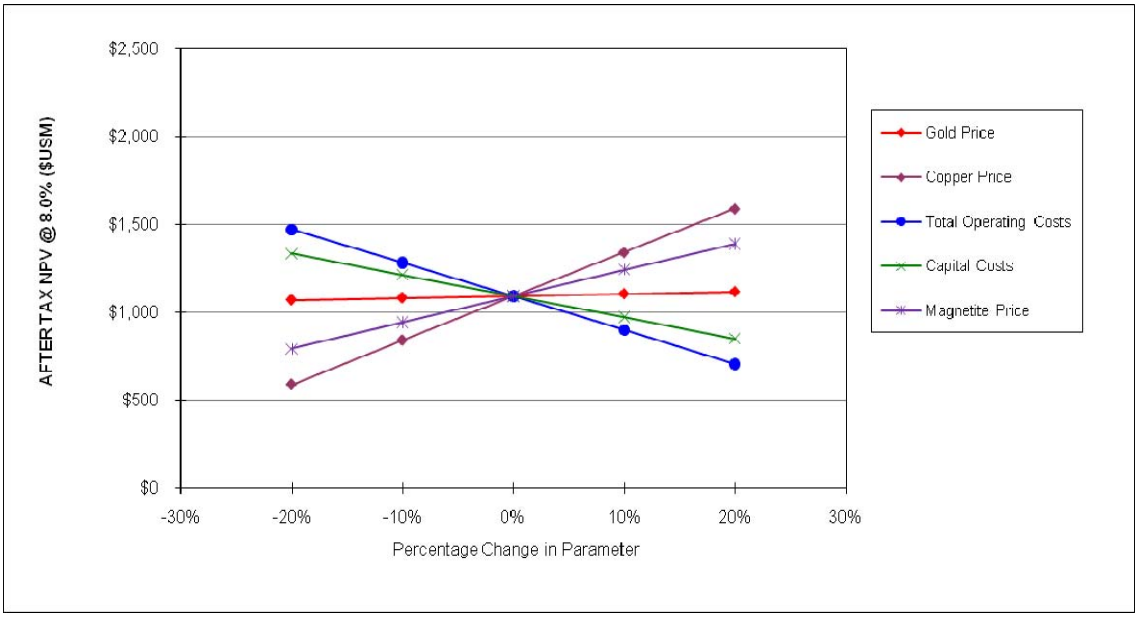


Figure 1-4: NPV Sensitivity

Table 1.11: Summary of Sensitivity Analysis

PARAMETER OR VARIATION	VALUE	EBITD&A IRR (%)	EBITD&A NPV @ 8.0% ($M)	AFTER TAX IRR (%)	AFTER TAX NPV @ 8.0% ($M)
Copper Price ($/lb)					
-20%	$2.00	21.3	967	15.9	589
-10%	$2.25	25.4	1,294	19.0	841
Base Case	$2.50	29.4	1,620	22.0	1,092
10%	$2.75	33.3	1,946	24.9	1,341
20%	$3.00	37.1	2,272	27.7	1,589
Total Operating Costs ($/t LOM average)					
-20%	$8.42	34.1	2,125	25.6	1,469
-10%	$9.47	31.8	1,872	23.9	1,282
Base Case	$10.52	29.4	1,620	22.0	1,092
10%	$11.57	26.9	1,367	20.1	899
20%	$12.63	24.3	1,115	17.9	704
Initial Capital Costs ($M)					
-20%	$994	37.4	1,882	28.2	1,334
-10%	$1,118	33.1	1,751	24.8	1,213
Base Case	$1,242	29.4	1,620	22.0	1,092
10%	$1,366	26.3	1,489	19.6	971
20%	$1,491	23.7	1,357	17.6	850
Magnetite Iron Price ($/dmtu Fe)					
-20%	$0.80	26.2	1,232	19.3	791
-10%	$0.90	27.9	1,426	20.7	942
Base Case	$1.00	29.4	1,620	22.0	1,092
10%	$1.10	31.0	1,814	23.3	1,241
20%	$1.20	32.4	2,008	24.5	1,390

Varying the gold price has little impact on the project IRR. The magnetite concentrate price has a moderate impact whereas the copper price and capital and operating costs have the largest impacts. The NPV shows less sensitivity to the capital cost, but the copper price and operating cost still have large impacts.

1.9 Project Schedule

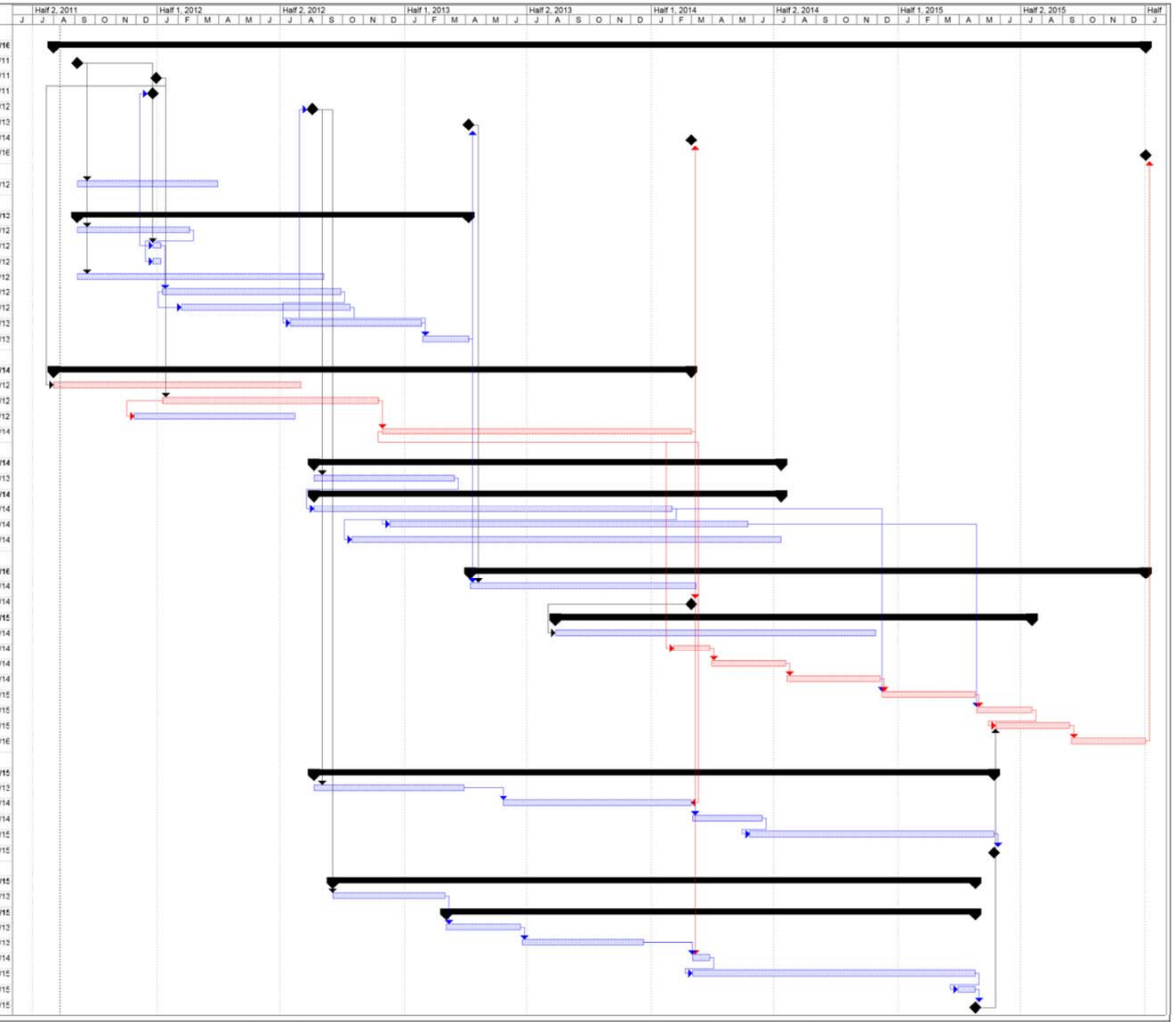
A project implementation schedule has been developed for a feasibility study (FS) and testwork phase followed by engineering, procurement and construction management (EPCM) of the process plant, related facilities, and prescribed infrastructure. The plan includes environmental baseline studies and the preparation of the EIS and permitting process. The plan is based on the successful completion of an integrated testwork program and FS. Due to the advanced nature of the Santo Domingo testwork, the FS can commence in parallel or slightly ahead of the testwork program and still allow the results to be incorporated in the study. Refer to Figure 1-5 for the summarised project schedule.

The critical path on the PFS schedule is the completion of the EIS to allow permitting to be completed to obtain access to site for construction of the mills and the concentrate and sea water pipelines. The proposed duration from the development of the EIS and award of the mining permit is approximately 112 weeks. This process is scheduled to commence in January 2012.

The critical, long-lead items for development of the plant are the grinding mills. SAG and ball mills delivery is currently forecast to be 85 weeks from manufacture to delivery at port of export. The commencement of plant engineering activities currently allows six months float time due to the duration EIS process to obtain access to site.

The schedule indicates an overall duration of approximately 230 weeks from the commencement of environmental monitoring (begun August 2011) through to the completion of commissioning in December 2015. This schedule does not incorporate any contingency. However, several opportunities have been identified to potentially shorten the schedule by undertaking parallel works or pre-ordering equipment.

ID	Task Name	Duration	Start	Finish
1				
2	**Santo Domingo Project**	**231 wks**	**Mon 01/08/11**	**Fri 01/01/16**
3	Interim Engineering for EIS & Testwork Program	0 wks	Mon 05/09/11	Mon 05/09/11
4	Port Location Frozen for EIS Commencement	0 wks	Fri 30/12/11	Fri 30/12/11
5	Commence Engineering Feasibility Study	0 wks	Mon 26/12/11	Mon 26/12/11
6	Long Lead Items Purchase (Board Approval)	0 wks	Fri 17/08/12	Fri 17/08/12
7	EPCM Phase Board Approval	0 wks	Fri 05/04/13	Fri 05/04/13
8	Site Access & Commence Construction	0 wks	Fri 28/02/14	Fri 28/02/14
9	Complete Commissioning	0 wks	Fri 01/01/16	Fri 01/01/16
10				
11	Interim Engineering for EIS & Testwork Program	30 wks	Mon 05/09/11	Fri 30/03/12
12				
13	**Feasibility Study**	**83 wks**	**Mon 05/09/11**	**Fri 05/04/13**
14	Interim Met testwork, programme design, sample ship & conduct	24 wks	Mon 05/09/11	Fri 17/02/12
15	Confirm Base Process Flowsheet and Design Criteria	2 wks	Mon 26/12/11	Fri 06/01/12
16	FS WBS and Project Controls	2 wks	Mon 26/12/11	Fri 06/01/12
17	Mine Design	52.2 wks	Mon 05/09/11	Mon 03/09/12
18	Process Engineering	38 wks	Mon 09/01/12	Fri 28/09/12
19	Design Engineering and Layouts	36 wks	Mon 06/02/12	Fri 12/10/12
20	Estimating	28 wks	Mon 16/07/12	Fri 25/01/13
21	Reporting	10 wks	Mon 28/01/13	Fri 05/04/13
22				
23	**Environmental Impact Study & Permitting**	**135 wks**	**Mon 01/08/11**	**Fri 28/02/14**
24	BASELINE STUDIES (currently ongoing)	52.4 wks	Mon 01/08/11	Tue 31/07/12
25	ENVIRONMENTAL IMPACT STUDY PREPARATION	46 wks	Mon 09/01/12	Fri 23/11/12
26	COMMUNITY AMD STAKEHOLDERS ENGAGEMENT	34 wks	Mon 28/11/11	Mon 23/07/12
27	EIS FOLLOW UP	327 days	Thu 29/11/12	Fri 28/02/14
28				
29	**Plant Project Basic Engineering**	**99 wks**	**Mon 20/08/12**	**Fri 11/07/14**
30	Engineering and Procurement	30 wks	Mon 20/08/12	Fri 15/03/13
31	**Order & Supply Long Lead Items**	**99 wks**	**Mon 20/08/12**	**Fri 11/07/14**
32	SAG Mill	76 wks	Mon 20/08/12	Fri 31/01/14
33	SAG Mill Motor	76 wks	Mon 10/12/12	Fri 23/05/14
34	Primary Ball & Regrind Mills	91 wks	Mon 15/10/12	Fri 11/07/14
35				
36	**Plant Project EPCM**	**143 wks**	**Mon 08/04/13**	**Fri 01/01/16**
37	EPCM Award & Detailed Engineering	48 wks	Mon 08/04/13	Fri 07/03/14
38	Mobilise to Site	0 wks	Fri 28/02/14	Fri 28/02/14
39	**EPCM Packages**	**101 wks**	**Mon 12/08/13**	**Fri 17/07/15**
40	EPCM Site Works Packages	68 wks	Mon 12/08/13	Fri 28/11/14
41	Bulk earthworks mobilise and batch plant operational	8 wks	Mon 03/02/14	Fri 28/03/14
42	Mills Bulk Earthworks	16 wks	Mon 31/03/14	Fri 18/07/14
43	Mills Civils Mobilise, Batch Plant & Pour	20 wks	Mon 21/07/14	Fri 05/12/14
44	Mills Mechanical Installation	20 wks	Mon 08/12/14	Fri 24/04/15
45	Mills Electrical Installation	12 wks	Mon 27/04/15	Fri 17/07/15
46	Pre commissioning	16 wks	Mon 25/05/15	Fri 11/09/15
47	Commissioning	16 wks	Mon 14/09/15	Fri 01/01/16
48				
49	**Mining Equipment and Pre Strip**	**144 wks**	**Mon 20/08/12**	**Fri 22/05/15**
50	Equipment Tender and Award Shovels & trucks	32 wks	Mon 20/08/12	Fri 29/03/13
51	Equipment Supply First Unit	40 wks	Mon 27/05/13	Fri 28/02/14
52	Supply Other Units and Commission	15 wks	Mon 03/03/14	Fri 13/06/14
53	Prestrip	52 wks	Mon 26/05/14	Fri 22/05/15
54	Ore Production	0 wks	Fri 22/05/15	Fri 22/05/15
55				
56	**Seawater & Concentrate Pipelines**	**136 wks**	**Mon 17/09/12**	**Fri 24/04/15**
57	Pipeline EPCM Contract Award & Engineering	24 wks	Mon 17/09/12	Fri 01/03/13
58	**Pipeline Construction**	**112 wks**	**Mon 04/03/13**	**Fri 24/04/15**
59	Tender and Award	16 wks	Mon 04/03/13	Fri 21/06/13
60	Pipe purchase	26 wks	Mon 24/06/13	Fri 20/12/13
61	Mobilisation	4 wks	Mon 03/03/14	Fri 28/03/14
62	Pipeline Construction	60 wks	Mon 03/03/14	Fri 24/04/15
63	Pipeline commissioning	4 wks	Mon 30/03/15	Fri 24/04/15
64	Seawater at site	0 wks	Fri 24/04/15	Fri 24/04/15



Project: Preliminary Draft Santo Doming
Date: Wed 10/08/11

Critical | Critical Progress | Split | Baseline | Baseline Milestone | Summary Progress | Project Summary | External Milestone
Critical Split | Task | Task Progress | Baseline Split | Milestone | Summary | External Tasks | Deadline

Page 1

Figure 1-5: Summarised Project Schedule

1.10 Recommendations for Future Work

Recommendations for future work are listed below.

- Complete a Feasibility Study that considers the following points:
  - In-fill drilling is recommended to bring the first 3 years of the mine reserves into the proven category.
  - Conduct mine optimization studies to smooth out the mill-feed grade profile and the mining schedule.
  - Evaluate alternative ramp locations in the pit stages taking advantage of changes in wall slopes.
  - Conduct further waste rock facility geotechnical engineering studies to test all assumptions made in this report.
  - Update the optimum primary grind selection based on prevailing economic parameters for a 3-year pit composite.
  - Complete a mine geotechnical drilling program to take the geological model, structural model (major features and fabric) and hydrogeological model for the SDS/Iris and Iris Norte pits to a project level status.
  - Plant and TSF geotechnical investigations including borehole drilling and test pit excavations to determine the foundation, borrow, and fill placement conditions for design.
  - Investigate paste fill tailings deposition vs conventional wet tailings dam.
  - In the tailings area, a more detailed investigation program to improve the characterization of soils and develop an approximate profile of these clayey/silty soils.
  - Review mill selection in light of the latest comminution data. Deferral of the pebble crushing circuit, twin trains of twin pinion SAG mills or optimising the current SAG mill size should be considered
  - Include a metallurgy testwork program to define the following:
    - further investigation to confirm the optimum regrind levels for the copper rougher and LIMS rougher concentrates
    - locked cycle tests to determine the effect of water recycle on the copper metallurgy
    - determine if the 3rd copper cleaning stage is warranted
    - batch copper flotation and LIMS tests on a larger suite of variability samples to improve the statistical confidence level of head grade recovery regression equations
    - hematite recovery studies
    - determining the copper and LIMS metallurgy for the first 12 quarters of mine operation
    - slurry pipeline transportation testing of copper, LIMS and hematite concentrates
    - tailings thickening testwork and rheology
    - concentrate transportable moisture limit testing

  - additional reagent optimization for copper flotation
  - additional comminution testing of the outstanding variability samples from this phase using SMC test
  - additional comminution testing on a larger suite of variability samples, focusing on ores scheduled for processing in early mine years
  - heap leach testwork on copper oxide material
- Prepare a Chilean labour rate report
- Confirm port facility location
- Continue environmental base line studies
- Prepare an EIS and commence environmental permitting process
- Identify if there is an opportunity to share port facilities with other developing projects in the region.
- Specific high voltage power studies, recommended by the SDEC-SIC are required for confirmation of High Voltage Supply.

2 INTRODUCTION

2.1 Background

Far West Mining (FWM) was an international mineral exploration company headquartered in Vancouver, Canada. The Company was primarily engaged in the evaluation, acquisition, and exploration of mineral properties in Chile and Australia. On June 17th of 2011, Far West Mining was sold and is now 70% owned by Capstone Mining Corp. (TSX:CS) and 30% by Korea Resources Corporation ("KORES"). The Santo Domingo deposit is also 70% owned by Capstone and 30% by KORES.

This technical report, "Technical Report on the Santo Domingo Project, Chile" prepared for Capstone Mining Corp.., and dated September 28, 2011 ("Technical Report") and the resource estimate have been prepared in compliance with the disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101), Companion Policy 43-101CP, and Form 43-101F1.

2.2 Project Scope and Terms of Reference

This Santo Domingo Project Technical Report (Technical Report or Report) was prepared for Far West Mining Ltd. (FWM), which is 70% owned by Capstone Mining Corp. (Capstone) by Ausenco Minerals Canada Inc (Ausenco) to provide Capstone with sufficient information to determine the economic feasibility of developing the Santo Domingo ore bodies, and also to decide whether and on what basis to proceed with a feasibility study.

The pre-feasibility study has, at its focus, two copper and iron ore deposits: SDS/IRIS and Iris Norte. Although FWM has explored extensively throughout its tenement, this report does not present any new information in relation to exploration, data, or detailed geology outside of these two zones.

The Santo Domingo project consists of an open pit mine and an associated processing facility along with on-site and off-site infrastructure to support the operation. The mine, process plant, and associated infrastructure are designed to process 60 kt/d of ore, with consideration given to achieving 70 kt/d on softer and lower head grade ores.

2.3 Qualified Persons

The following Qualified Persons have contributed to the Technical Report in the following areas:

- David Brimage of Ausenco for mineral processing. David visited the property during February 8 to 10, 2011.

- David W Rennie of Roscoe Postle Associates Inc. (RPA) for mineral resource estimation. David visited the property during June 14 to16, 2010.
- John Nilsson of Nilsson Mine Services Ltd. for reserve estimation and mining. John visited the property during June 28 to 29, 2011.
- Art Winckers of Arthur H. Winckers & Associates Mineral Processing Consulting Inc for metallurgy. Art has not visited the property. Art was present at SGS Canada Inc. during periods of the metallurgy testwork.
- Michael Davies of AMEC for mine geotechnical. Michael has not visited the property. AMEC did carry out the geotechnical field drilling program supplying the data required for Michael's evaluation.

The responsibilities of each author are provided in Table 2.1.

Table 2.1: Responsibilities of Each Qualified Person

Author	*Responsibility (Report Sections)*
Mr David Brimage	*1, 2, 3, 4, 5, 17, 18, 19, 20, 21, 22, 23, 24 (except 24.1), 25, 26, 27*
Mr David W Rennie	*6, 7, 8, 9, 10, 11, 12, 14,*
Mr John Nilsson	*15, 16.3, 16.4*
Mr Art Winckers	*13*
Mr Michael Davies	*16.1, 16.2, 24.1*

2.4 Frequently Used Acronyms, Abreviations, Definitions, Units of Measure

Unless otherwise indicated, all references to currency in this report refer to united States dollars (US$). Frequently used acronyms and abbreviations are listed below.

Above mean sea level	amsl
Annum (year)	a
Centimeter	cm
Cubic centimeter	cm^3
Cubic meter	m^3
Concentration by weight	Cw
Day	d
Days per year (annum)	d/a
Degree	°
Degrees Celsius	°C
Dry metric ton	dmt
Foot	ft
Gram	g
Grams per litre	g/L

Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Horsepower	hp
Hour	h
Hours per day	h/d
Inch	"
Inverse distance	ID
Kilogram	kg
Kilometer	km
Kilovolts	kV
Kilowatt hour	kWh
Kilowatt	kW
Less than	<
Litre	L
Litres per second	L/sec
Measure of the acidity or basicity of a solution	pH
Meter	m
Meters above sea level	masl
Meters per minute	m/min
Meters per second	m/sec
Micrometer (micron)	µm
Millimeter	mm
Million pounds	Mlb
Million tonnes	Mt
Million	M
Minute (plane angle)	'
Minute	min
Ounce	oz
Parts per billion	ppb
Parts per million	ppm
Percent	%
Pound(s)	lb
Second (plane angle)	"
Second (time)	sec
Square kilometer	km2
Square meter	m2
Tonne (1,000 kg)	t
Tonne Force	tonf
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Year (annum)	a

3 RELIANCE ON OTHER EXPERTS

In preparing this report, Ausenco has relied on input from FWM and a number of well qualified independent consulting groups.

3.1 Other Experts

Ausenco is not an expert in legal, land tenure, or environmental matters and is not an expert in Chilean law. Ausenco has relied on data and information provided by FWM and on previously completed technical reports, refer to Section 27 for report details. Although Ausenco has reviewed the available data and visited the site, these activities serve to validate only a portion of the entire data set. Therefore, Ausenco has made judgments about the general reliability of the underlying data; where deemed either inadequate or unreliable, the data were either not used or procedures were modified to account for the lack of confidence in that specific information.

Ausenco has relied upon Oscar León, Legal Representative and Manager of Minera Lejano Oeste S.A. for land and legal issues in Section 4.2. Ausenco has relied upon Humberto Rivas, Environmental Project Engineer with Knight Piesold., for Section 20. Knight Piesold. is responsible for conducting environmental studies on behalf of FWM for the Santo Domingo project. Ausenco has relied upon Monte Christie, Senior Geotechnical Engineer with Ausenco Vector, for Section 18.12. Monte is responsible for the design of the Tailings Storage Facility. Finally, Ausenco has relied upon advice from Deloitte for the taxation modeling discussed in Section 22.2.1.

4 PROPERTY DESCRIPTION AND LOCATION

The following section is an extract from the RPA mineral resource estimate dated 2010[5].

4.1 Location

The Santo Domingo project is based on a large open pit copper/magnetite resource located approximately two hours north of Copiapó by paved road and 5 km southeast of the town of Diego de Almagro in Region III of northern Chile.

The Santo Domingo property was originally part of the BHP Candelaria project area, which consisted of eight non-contiguous concessions in a north-south corridor extending between the towns of Taltal to the north and to a point about 75 km south of the city of Copiapó.

The Project area encompasses a number of non-contiguous concessions within a roughly north-south belt approximately 300 km long by up to 40 km wide. It extends from near the town of Taltal in the north to 75 km south of the city of Copiapó in the south, in Regions II and III of Northern Chile (Figure 4-1). The approximate geographical limits of this project area are: 324000E–410000E, 6900000N–7200000N (datum: PSAD ’56, Zone 19S). The centre of the Santo Domingo area is at approximately 398000E and 7074000N (datum: PSAD ’56, Zone 19S), in Region III of Northern Chile (Figure 4-2).

4.2 Land Tenure

The initial Candelaria Project land package assembled by BHP in 2002 consisted of 3,434.5 km^2 of exploration concessions. In 2002 and 2003, FWM and BHP entered into Project Area Agreements that allowed FWM to earn an interest in the concessions within the project area. Effective August 5, 2003, FWM assigned interests in the Project Area Agreements to its wholly owned Chilean subsidiary, Minera Lejano Oeste S.A. (MLO). On May 4, 2005, BHP terminated any interest in the concessions within the project area and commenced transfer of title of all these concessions to MLO in exchange for a retained 2% NSR royalty. As of the date of this report, all concessions in the Candelaria Project area are 100% owned by MLO.

All of the registered exploration concessions have been upgraded to exploitation status mining leases. Sixty-five of the exploitation concessions in progress were surveyed during 2010 and are waiting for technical approval by the government. FWM controls 100% of 82 exploitation concessions (‘constituidas’) in the Santo Domingo area, consisting of 15 established concessions totalling 1,780 ha including the exploitation concessions acquired through option (Estrellita 1/10, Iris I 1/200, Iris II 1/160, Iris 1/55, Estefánia, Manto Ruso 1/8, Pichanga 1/100, and Santo 1/20), and 67 exploitation concessions (17,978 ha) in progress. In all, the 82 exploitation concessions cover a total area of 19,578 ha. Concessions in the

[5] Far West Mining Ltd. Technical Report on the Santo Domingo Property, Region III, Atacama Province Chile, NI 43-101 Report, Author: David W. Rennie, P.Eng. August 26, 2010 Scott Wilson Roscoe Postle Associates Inc.

Santo Domingo area are listed in Figure 4-3. Note that 4a3 is FWM's denotation for the Santo Domingo area including the SDS, Iris, Iris Norte, and Estrellita deposits.

The entire concession package in the Santo Domingo area can be maintained over the next year for a cost of Ch$82,534,000, or approximately US$156,000, up to the end of March 2012. All exploitation costs (registration, etc.), and optioned annual fees are included, except for costs of the survey mentioned in the previous section. None of the concessions includes surface rights.

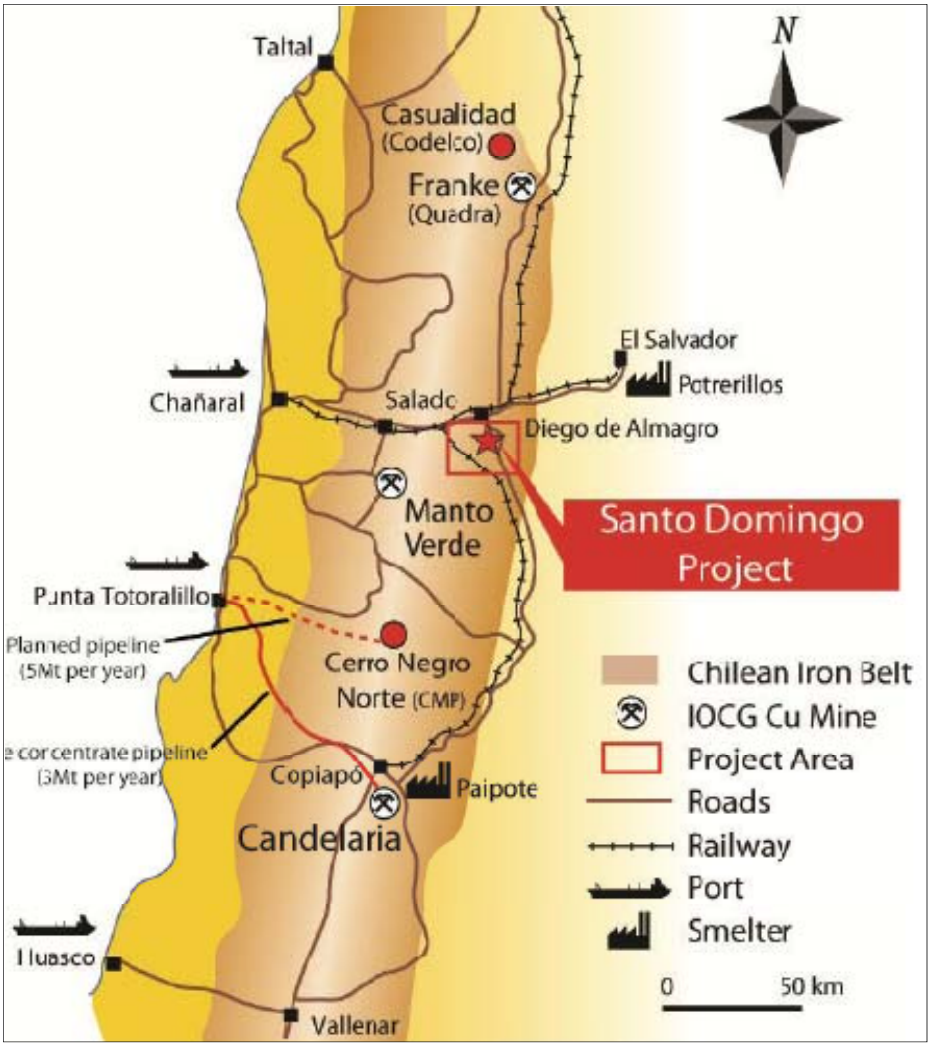


Figure 4-1: Mine and Port Locations

Figure 4-2: Property Location Map

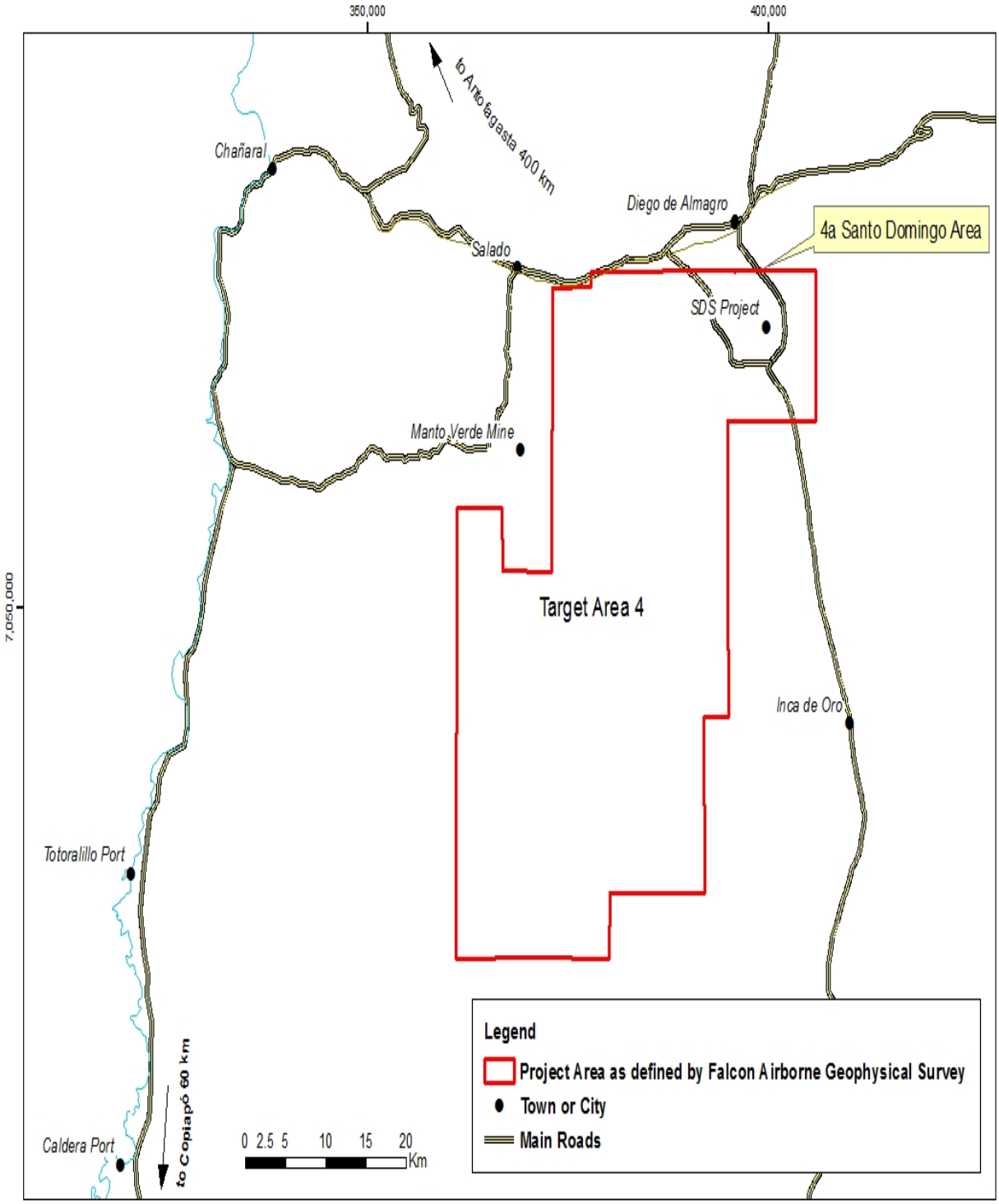
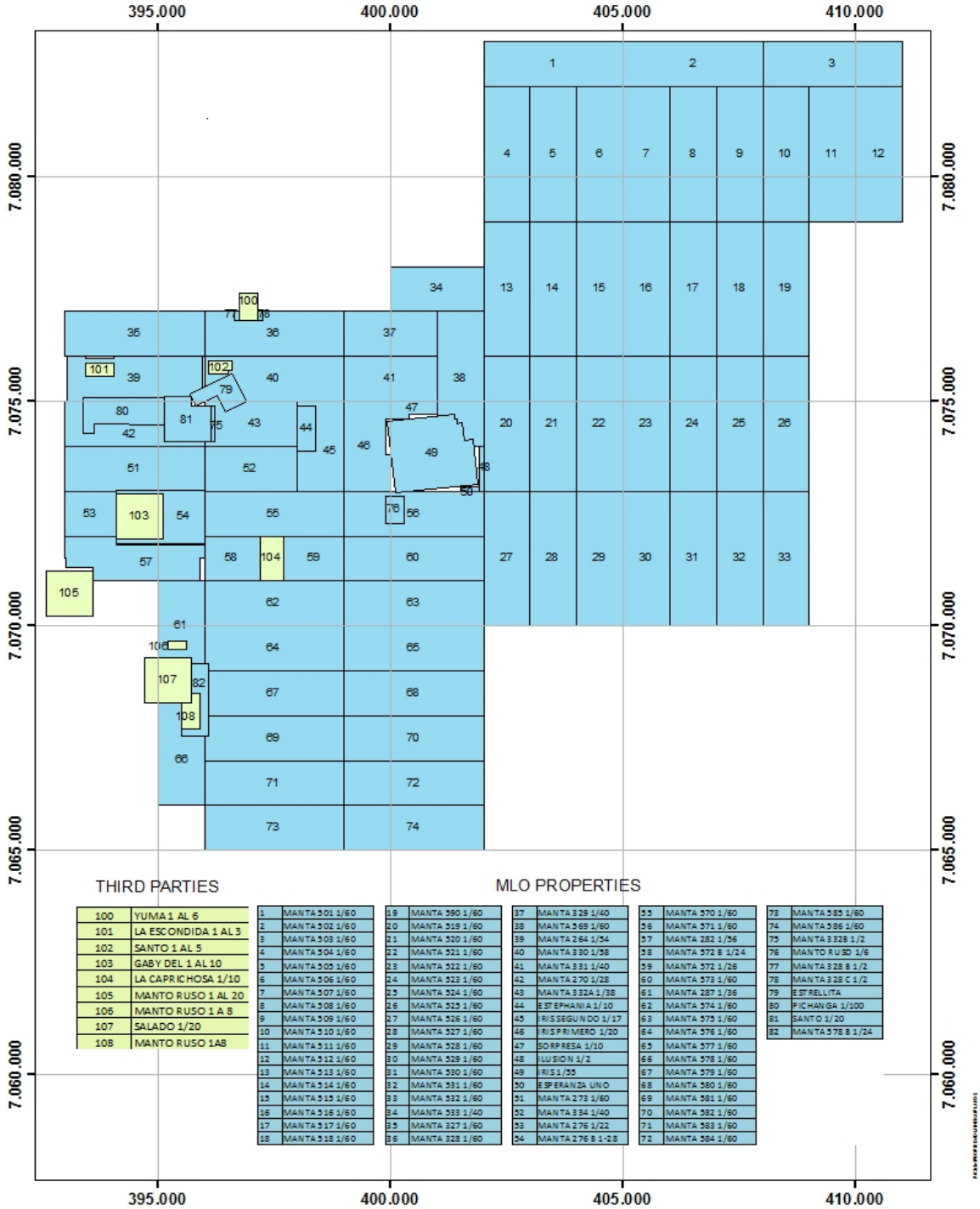


Figure 4-3: Property Concessions Map



5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The following section is an extract from the RPA mineral resource estimate dated 2010[6],

5.1 Accessibility

Access to the Santo Domingo property area is via the paved Pan-American Highway (Ruta 5) and a network of generally well maintained gravel roads. Most concessions in the project area are within a one-hour drive from the Pan American Highway. The Santo Domingo Property is roughly five hours' travel time by road south of Antofagasta, and two hours by road north of Copiapó. Access is via the Pan-American Highway, heading east from the town of Chañaral for 12 km to El Salvador turn-off, and then an additional 50 km east to the town of Diego de Almagro. A southbound paved highway connects Diego de Almagro with Copiapó. At 3.3 km southeast from Diego de Almagro along this highway, a secondary gravel road (Santo Domingo road) leads south into the property. The total distance by road from Chañaral to the Santo Domingo Property is approximately 68 km, with a travel time of roughly 50 minutes.

5.2 Climate and Topography

The Santo Domingo property is located in the Atacama Desert, one of the driest regions on earth. The climate is arid and the weather is generally clear and warm in all seasons and poses no limitations on field activities.

The closest weather station where temperature and precipitation measurements have been recorded for some time is the city of El Salvador. The daytime high and low temperatures there are 26°C and 0.8°C for July, and 30°C and 9.8°C for January, respectively. The highest average recorded precipitation is in May at 14.8 mm and the lowest is in December at 0 mm. El Salvador is over 2,000 masl, while the project is at 1,100 to 1,200 masl. In all probability, the climate at the project site is drier and the mean temperature slightly higher. In the interest of establishing environmental baselines, FWM installed a weather station at Santo Domingo in 2009.

5.3 Local Resources and Regional Infrastructure

The technical report on the Santo Domingo property by RPA, states that the region has available, well-established infrastructure (power, water, transportation, work force, etc.) to service the mining community. There is no infrastructure at Santo Domingo property other

[66] Far West Mining Ltd. Technical Report on the Santo Domingo Property, Region III, Atacama Province Chile, NI 43-101 Report, Author: David W. Rennie, P.Eng. August 26,2010 Scott Wilson Roscoe Postle Associates Inc.

than gravel roads for access to the property and drill sites. The nearby town of Diego de Almagro is serviced by the regional power grid and a major rail.

Several cities or towns are near the Santo Domingo property. Diego de Almagro, located adjacent to the property, 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. It has a population of approximately 10,000 people, hotel accommodations, food, fuel, and minor services. The most important logistical centre in the region is Copiapó, approximately two hours' drive to the south of the Santo Domingo property. It has a population of approximately 150,000 people, an airport with daily scheduled flights to Santiago and Antofagasta, and abundant businesses offering services specific to mining and exploration.

5.4 Physiography

Vegetation is very sparse. In the valley bottoms, plant life consists of small, widely-spaced bushes a few tens of centimeters in height. Hillsides and peaks are generally devoid of any vegetation. In spite of the dry conditions, hills of gentle to moderate relief have been cut by deep gullies and flanked with gravel-filled valleys and alluvial fans; evidence of water movement preserved since conditions were less arid. Elevations range from approximately 900 to 1,500 masl.

5.5 Seismicity

Seismic zone maps of South America indicate that the project area is likely to have high seismicity and the site is considered part of Zone 3 (shores) according to the Chilean National Design Code Nch2369, with a peak ground acceleration of 0.4 g. A seismic hazard assessment should be performed for the Santo Domingo project site as part of subsequent study work and the results used in the feasibility-level design of the facilities.

6 HISTORY

The following section is an extract from the RPA mineral resource estimate dated 2010[7].

Mining for copper, gold, and iron has been ongoing in this area since early in the 19th century. Small mines in the region supplied copper ore to smelters in both Chañaral and Pan de Azúcar. Independent copper mines have been in operation on what is now Anglo American's Manto Verde deposit (located 25 km southwest of the Santo Domingo property) since the late 1800s, but significant production in this area started in 1906. Between 1906 and 1935, a reported total of 400,000 tonnes grading in excess of 3% Cu was mined from the Manto Verde fault zone (Vila et al., 1996).

Previous ownership of concessions in the Santo Domingo property is unknown. The area appears to have had a relatively long history of small-scale mining and prospecting. Mining activities on the nearby Manto Verde deposit date back to the late 1800s and it is probable that workings in the Santo Domingo property have a similar age.

Many small inactive mines and a myriad of pits occur throughout the property area. The mines typically exploited copper mineralization hosted in narrow (one meter to five meters) steeply-dipping veins and, in some cases adjacent strata to these veins. The largest mines are located along approximately 700 m of the Santo Domingo structure. These mines include the active La Estrella, La Estrellita, El Iris, and others. Judging by the size of the dumps and number of adits, it is possible that this specific area produced upwards of 500,000 tonnes. A second area of minor production is a small open pit with peripheral underground workings on the Caprichosa concession in Target Area 4a2 (FWM nomenclature) that may have produced in the order of 20,000 tonnes of copper oxide-bearing coming from a specularite stratum.

However, surface workings at the majority of the mines in the Santo Domingo property (other than those noted above) are generally less than a few tens of meters in length and the extent of underground development is unknown. Judging by the quantity of dump material adjacent to most of these mines, it is probable that production was no more than a few thousand tonnes at any one site.

The initial Candelaria Project land package was assembled by BHP in 2002. In 2002 and 2003, FWM and BHP entered into Project Area Agreements that allowed FWM to earn an interest in the concessions within the project area. Effective August 5, 2003, FWM assigned interests in the Project Area Agreements to its wholly owned Chilean subsidiary, Minera Lejano Oeste S.A. (MLO). On May 4, 2005, BHP terminated any interest in the concessions within the project area and commenced transfer of title of all these concessions to MLO in exchange for a retained 2% NSR royalty.

No historic resource estimates or production records for workings in the Santo Domingo property have been located.

[7] Far West Mining Ltd. Technical Report on the Santo Domingo Property, Region III, Atacama Province Chile, NI 43-101 Report, Author: David W. Rennie, P.Eng. August 26, 2010 Scott Wilson Roscoe Postle Associates Inc.

7 GEOLOGICAL SETTING AND MINERALIZATION

The following section is an extract from the RPA mineral resource estimate dated 2010[8].

7.1 Regional Geology

The dominant geological feature of the Chilean Iron Belt (CIB) is the north-south Atacama fault zone, a complex sinistral strike-slip and dip-slip fault system that runs sub-parallel to the coast of northern Chile for over 1,200 km. It has been active since the Jurassic age, and is related to an oblique subduction of a Jurassic to early Cretaceous magmatic arc (e.g., Brown et al., 1993; Scheuber et al., 1995 in Allen and Höy, 2005). The Atacama fault system was most active between ca.132 Ma and ca.106 Ma, during which time tabular-shaped mafic to felsic plutonic complexes were emplaced along the deep-seated structures. The Atacama fault controlled mineralization of many CIB iron deposits (e.g., Bookstorm, 1977 in Allen and Höy, 2005; Ménard, 1995). Movement along the Atacama fault has been documented from the Jurassic to the late Miocene, a period of over 120 million years.

Initial movement along the Atacama fault zone was strike-slip, causing ductile deformation and the formation of mylonite. Subsequent extensional tectonism allowed dip-slip fault movement and brittle deformation. Intrusions were emplaced during both ductile (strike-slip) and brittle (dip-slip) deformation regimes. Volcanic- or intrusive-hosted breccia zones that formed during these ductile-brittle transitions were sites for the formation of many metasomatic iron oxide and iron-oxide-copper-gold (IOCG) deposits that define the metallogenic province known as the Cretaceous, or Chilean, Iron Belt, the CIB. The CIB is an area roughly 630 km by 40 km that extends from La Serena to Taltal and contains the Santo Domingo property. Many of the abundant iron ± copper deposits hosted within the CIB have been mined extensively.

The iron-rich end members of the IOCG mineral occurrences in the CIB are 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 of this deposit class within the CIB include La Candelaria and Manto Verde.

The Santo Domingo property lies some 10 km east of the main part of the Atacama fault, and roughly 25 km east of the part of the Atacama fault system that hosts the Manto Verde

[8] Far West Mining Ltd. Technical Report on the Santo Domingo Property, Region III, Atacama Province Chile, NI 43-101 Report, Author: David W. Rennie, P.Eng. August 26, 2010 Scott Wilson Roscoe Postle Associates Inc.

deposit (Figure 7-1 and Figure 7-2). The property is in the same structural setting relative to the Atacama fault as is the Candelaria deposit 120 km to the south. Both are located on the east flank of a 20 km to 30 km wide Lower Cretaceous plutonic complex that was emplaced along the deep-seated Atacama structures.

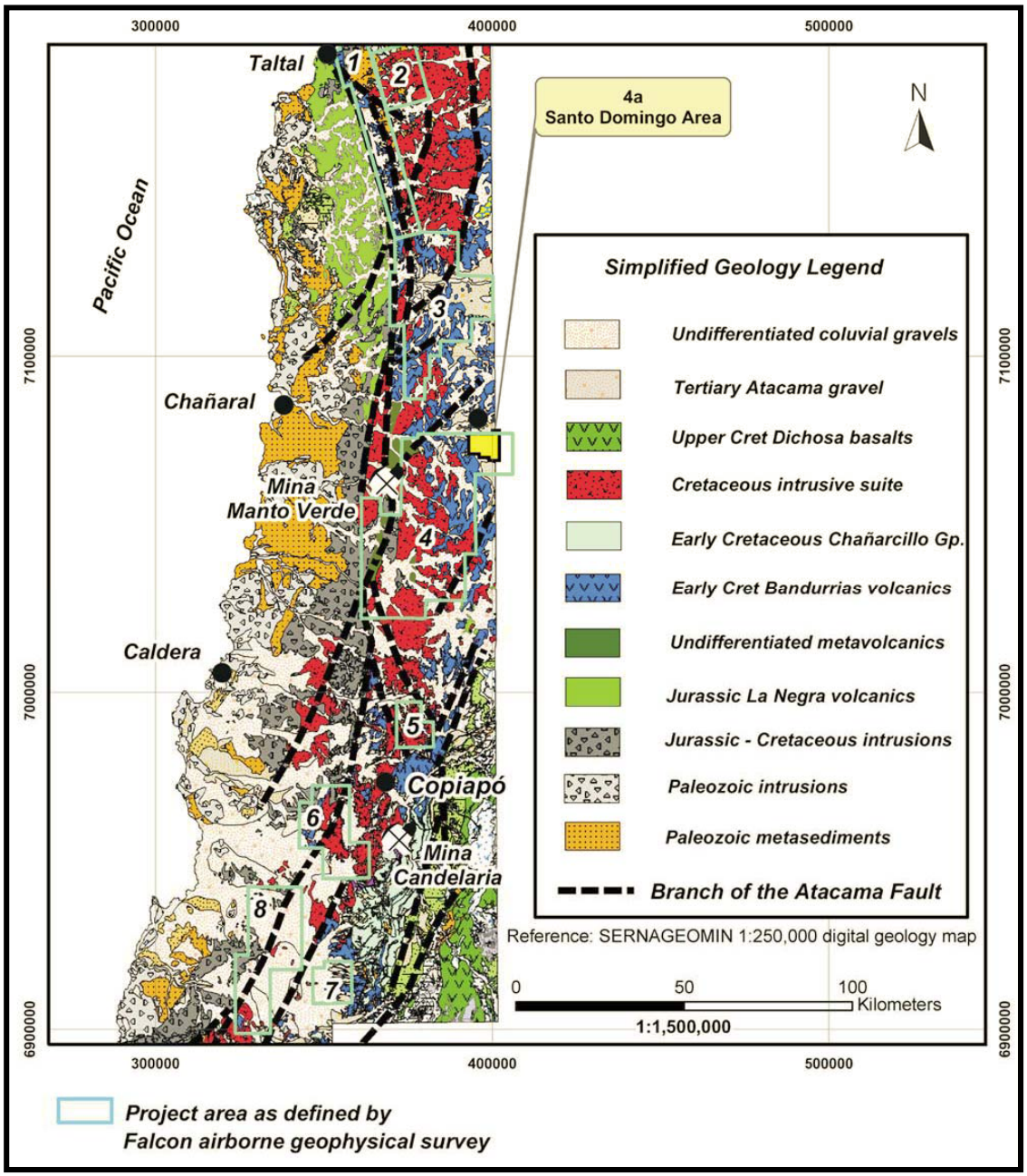


Figure 7-1: Regional Geology in the Candelaria Project Area

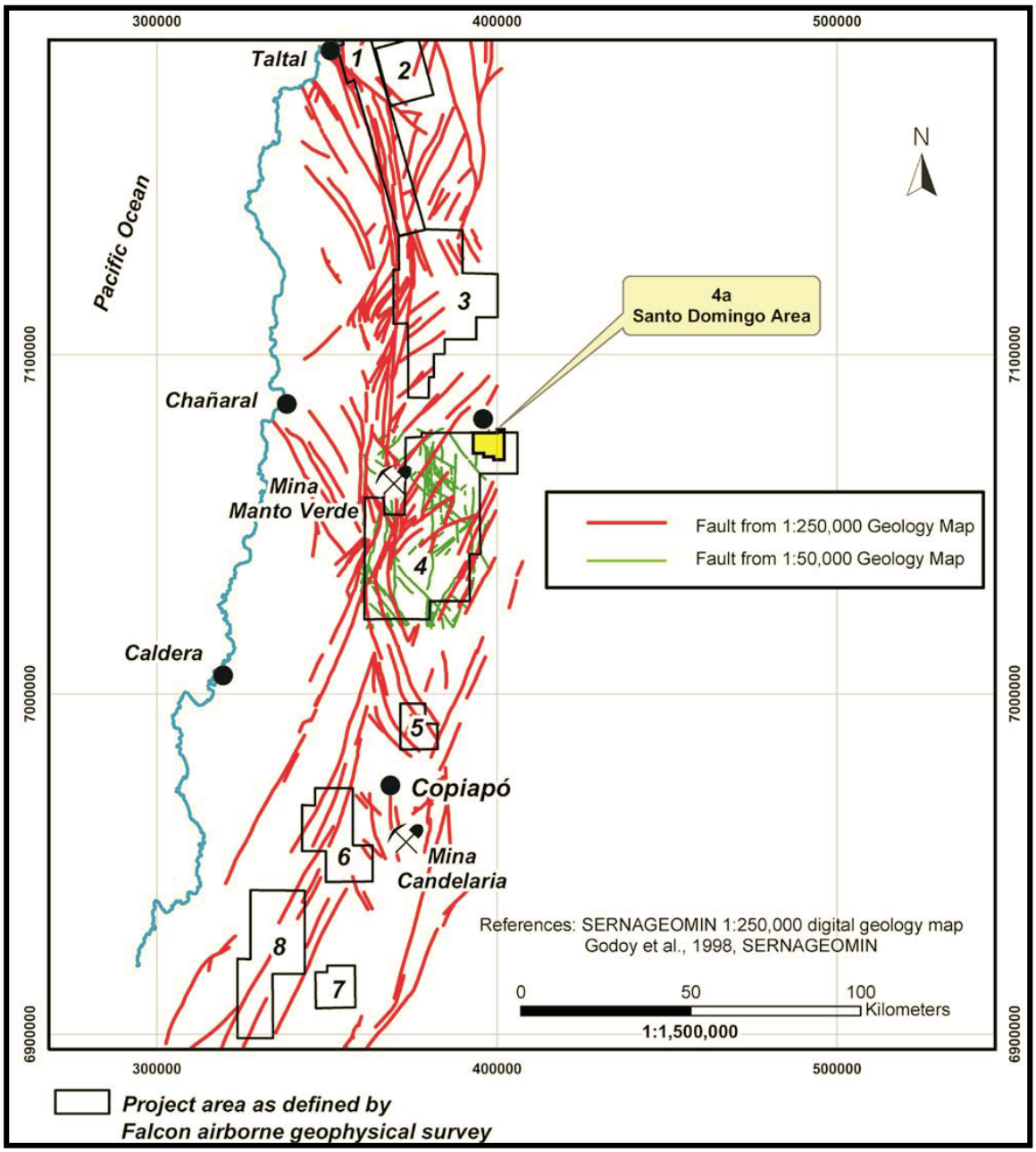


Figure 7-2: Fault Structures in the Candelaria Project Area

Marschik and Fontboté (2001) describe the stratigraphy of the Punta del Cobre district in the Candelaria deposit area. The oldest rocks in that area belong to the Punta del Cobre Formation, which is divided into a lower massive andesitic volcanic suite (probably flows; Geraldo-Negro Member) and an overlying volcaniclastic sequence (Algarrobos Member). The Punta del Cobre Formation is overlain by both the Chañarcillo and Bandurrias Groups. The Chañarcillo Group is composed predominantly of limestone and marine sedimentary rocks. Bandurrias Group rocks consist largely of andesitic flows and volcaniclastic rocks. Chañarcillo and Bandurrias Group rocks are interfingered sequences deposited in different

environments within the same marine back-arc basin. Mineralization in the Punta del Cobre district is largely stratigraphically controlled, and occurs at or near the Algarrobos–Geraldo Negro contact within the Punta del Cobre Formation.

The Santo Domingo property appears to be underlain by the same Lower Cretaceous volcano-sedimentary stratigraphy that hosts mineralization at Candelaria.

7.2 Local Geology

The Santo Domingo property lies on the east side of the Atacama fault complex (Figure 7-3) which, in this area, consists of numerous clusters of generally north-south structural breaks in a belt approximately 30 km wide. It appears that the 10 km wide westernmost cluster, which hosts the Manto Verde copper deposit, is the main part of the fault system. To the east, a second cluster of north-south faults cuts across the 4c Joint Venture area (Allen, 2004) where they host the Santa Teresita, 3M, and many other small shear-hosted IOCG deposits.

These north-south trending Atacama-related structures form clearly visible lineaments on both the Landsat images and the airborne magnetic susceptibility plots. They cut across both the volcanic units and the younger intrusions. On the ground, these structures are commonly difficult to identify. In the older volcanic units, the lineaments tend to follow topographic depressions. In the plutonic rocks, they more commonly occur along ridges, suggesting that the intrusions followed and obliterated the structures.

A Lower Cretaceous intrusive complex dominates geology within the Atacama fault zone, where tabular plutons have followed the deep-seated structures. According to the Servicio Nacional de Geologia y Minería (SERNAGEOMIN) regional geology map, at least nine intrusive events have affected the Manto Verde – 4a3 (Santo Domingo property) area. Intrusions are generally younger eastward, and range in age from 145 Ma to 90 Ma (Figure 7-3). The mineralization at Manto Verde is thought to date between 126 Ma to 114 Ma, and is probably related to one or more of the intrusions emplaced during this time interval, probably the Sierra Dieciocho plutons (126 Ma to 120 Ma) (Godoy and Lara, 1998).

Upper Jurassic to Lower Cretaceous La Negra Formation (or possibly Bandurrias Group) andesitic volcanic rocks occur as isolated irregular hornfelsed pendants within the Atacama fault-related intrusive complex. These rocks host the Manto Verde deposit. To the east, they grade into a relatively fresh and laterally continuous stratigraphic sequence of andesitic volcanic flows intercalated with volcaniclastics, calcareous tuffaceous sediments, and limestone. Classification of these rocks is not consistent in the various publications that document this area. On the 1:50,000 geology map (Godoy and Lara, 1998), the volcanic rocks in the 4a3 area are mapped as Upper Jurassic La Negra Formation. Clastic sedimentary rocks are classified as Upper Jurassic to Lower Cretaceous Punta del Cobre Group, and limestone was placed in the Lower Cretaceous (Berriasian–Barremian; 144 Ma to 115 Ma) Chañarcillo Group. On the SERNAGEOMIN 1:250,000 geology map, the entire sequence in the Santo Domingo property area is classified as Lower Cretaceous (Valanginian–Barremian; 138 Ma to 119 Ma) Bandurrias Formation.

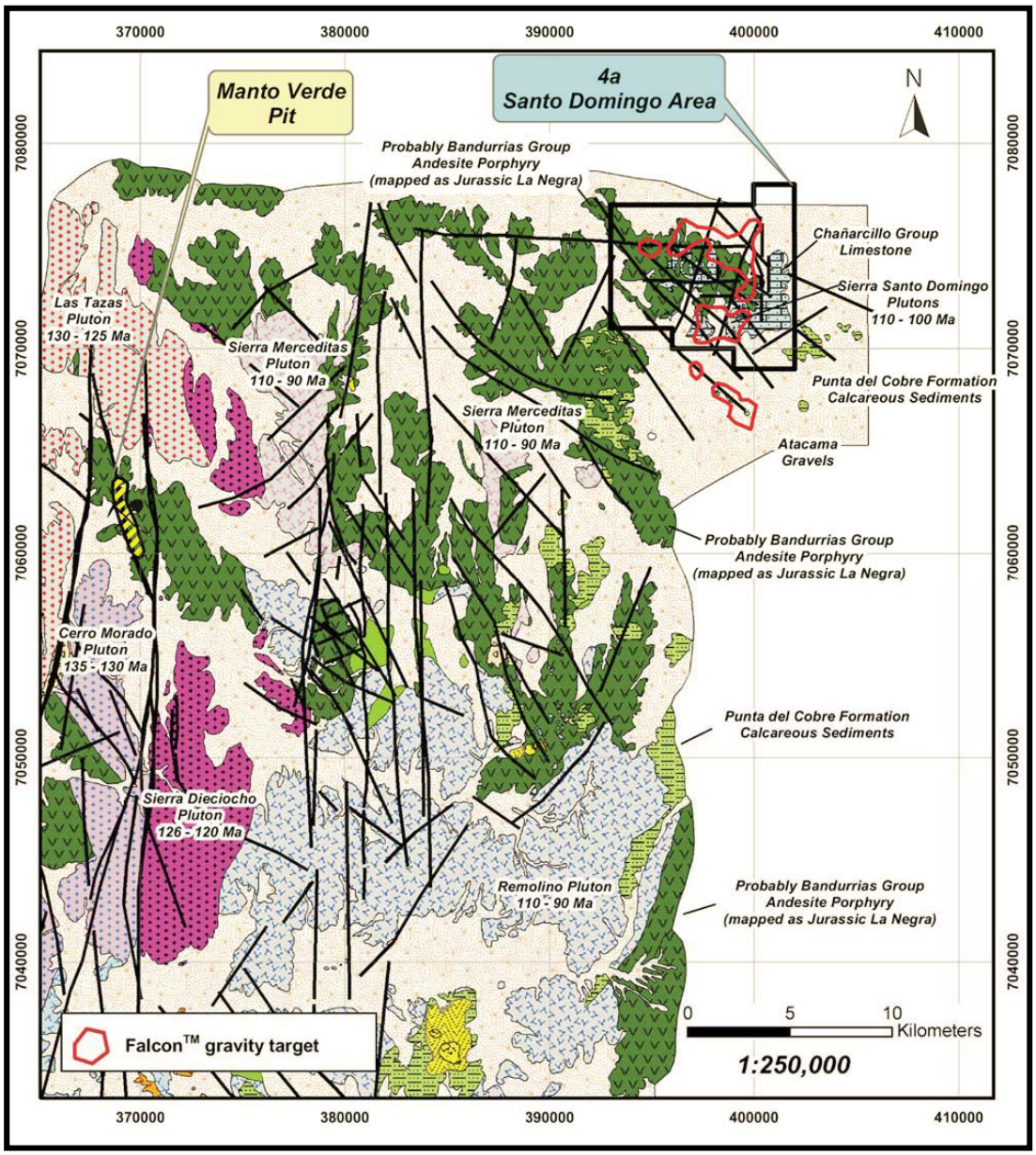


Figure 7-3: Local Geology

From field observations (both mapping and drilling) in the Santo Domingo property, the volcanic flows, calcareous tuffaceous, clastic sedimentary rocks, and limestone units appear to be stratigraphically intercalated (Allen, 2005 (2)). It is unlikely, therefore, that the volcanic flows, which underlie most of the project area, are part of La Negra Formation. The base of the stratigraphic sequence in the Santo Domingo property is presumed 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 approximately 120 km to the south.

7.3 Property Geology

7.3.1 Major Units

The oldest rocks in the Santo Domingo property are tentatively correlated with the Punta del Cobre Formation, a poorly exposed sequence of sedimentary and volcanic rocks outcropping in the extreme southeast part of the property. Small exposures of this material were mapped a few kilometers to the south of the Santo Domingo Sur (SDS) deposit. 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 Algarrobos Member of the Punta del Cobre Formation in the Copiapó area (Marschik and Fontboté, 2001). 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.

The bulk of the rock exposed in the Santo Domingo property appears to overlie the Punta del Cobre volcano-sedimentary sequence. It is an intercalated and interfingered sequence of volcaniclastics, andesite flows, limestone, and calcareous sedimentary rocks, probably of the Lower Cretaceous (Berriasian – Barremian; 144-119 Ma) Bandurrias and Chañarcillo Groups. The Bandurrias Group is defined as a predominantly volcanic sequence of andesite flows and volcaniclastic rocks. Chañarcillo Group rocks consist largely of limestone and calcareous marine sediments. Both definitions match observed geology on the Santo Domingo property. These two groups are thought to be contemporaneous, deposited at the same time in different parts of the same back-arc basin. Alternatively, the andesite-tuff succession that hosts the mantos may be part of the Punta del Cobre sequence. This would imply faulted contacts between this sequence and structurally adjacent limestone that is more clearly correlated with the Chañarcillo Group.

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

True sediments are not abundant, with most clastic rocks classified as tuffaceous sediments or crystal tuffs. They 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 over 300 m, with minor intervals of limestone and andesite lavas. In some places, the andesitic volcaniclastics 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 Santo Domingo property, 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 parts are massive and others contain abundant amygdules up to one centimeter in diameter (average one millimeter to two millimeters), filled with varying proportions of quartz, calcite, epidote, chlorite, potassic feldspar, limonite (pyrite), and almagre (or other copper oxides).

7.3.2 Structural Blocks

The geology of the Santo Domingo property is shown in Figure 7-4. The area is divided into a number of structural blocks with different lithological characteristics suggesting that the blocks are part of different stratigraphic levels. The southern part of the property hosts the SDS deposit (Figure 7-5). At surface, this structural block is almost entirely covered by volcanic flows of andesitic composition. Drill holes 258 and 263 have shown that this package of andesitic flows is up to 200 m thick and underlain by a sequence of tuffaceous rocks of similar composition.

This tuffaceous package is the host of the SDS deposit, which consists of thick semi-massive to massive iron oxide mantos that have replaced the tuffaceous rocks. The mantos consist of both hematite and magnetite, and various amounts of chalcopyrite and pyrite. The manto sequences have been drilled to a depth of approximately 550 m (hole 77D). They are underlain by a sequence of andesitic flows and andesite breccias that are mineralized by nearly vertical stringers of chalcopyrite and pyrite. The stratigraphic sequence of andesitic flows and tuffs dips gently (at an angle of approximately 15°) to the north-northwest under gravel cover.

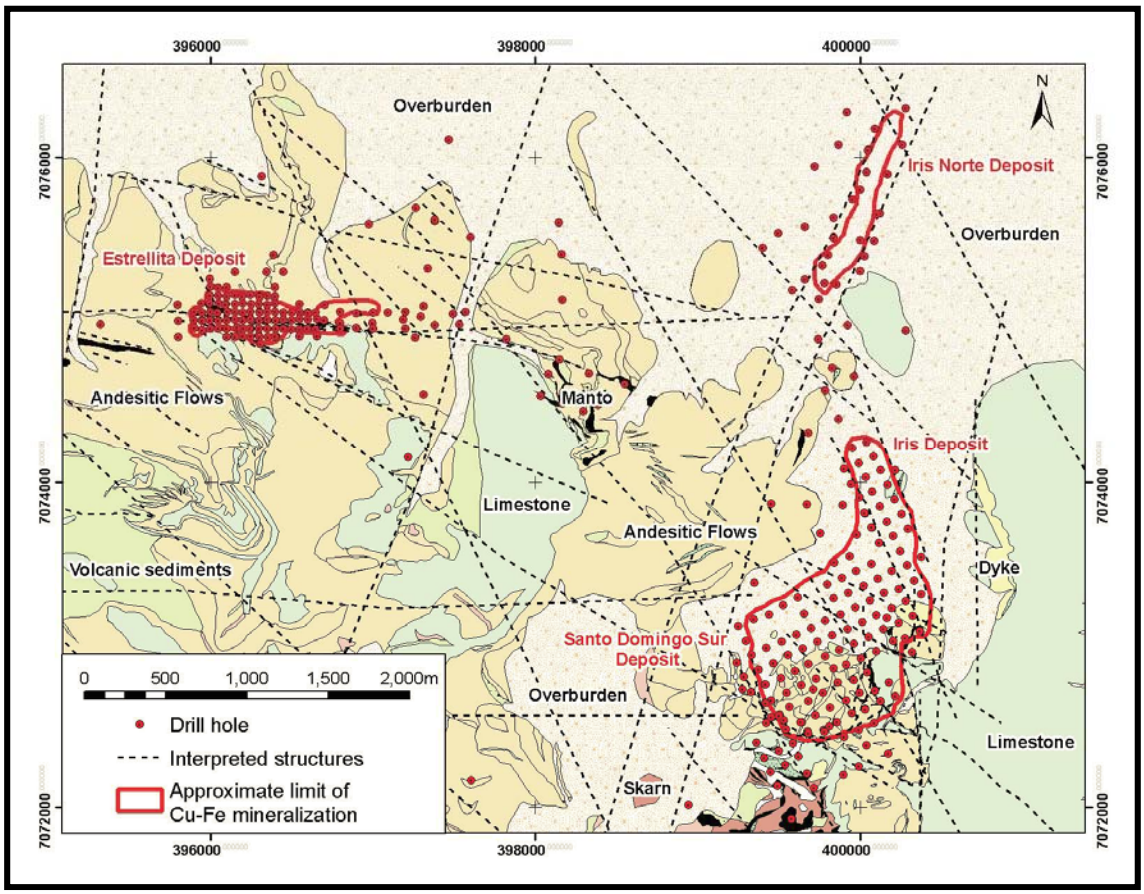


Figure 7-4: 4A3 Target Area – Geology, Drill Holes, and Interpreted Structures

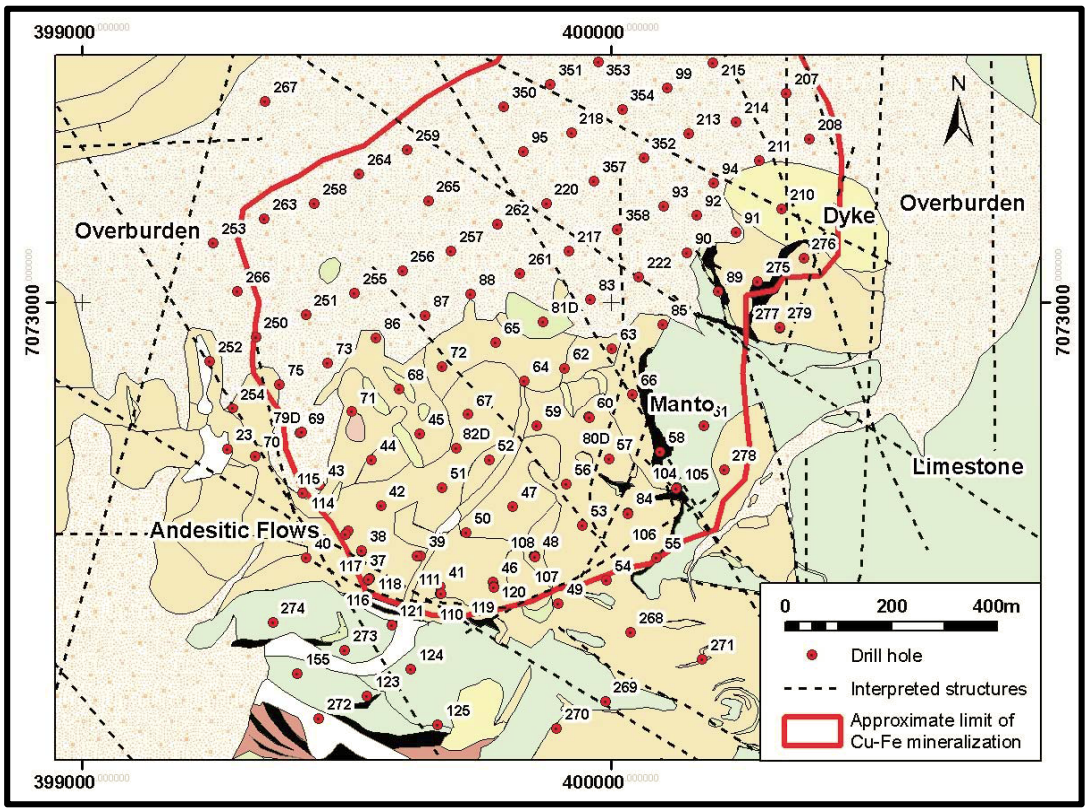


Figure 7-5: Santo Domingo Sur Deposit – Geology, Drill Holes and Interpreted Structures

The tuff sequence that hosts most of the mineralization at SDS has been intruded by fine-grained diorite sills that are present in almost all SDS drill holes varying in thickness from a few meters to more than 60 m. Similar diorites have been intersected in the Iris deposit and observed in outcrop to the south of SDS. 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 entire package, consisting of andesitic flows, andesitic tuffs, and diorite intrusions, has been cut by later feldspar-hornblende porphyry dykes that cut all other rock types and do not host any mineralization. The northern edge of this structural block is under cover and has not been defined by drilling at this point.

The structural block to the west of the SDS deposit consists of a gently to moderately (7° to 30°) north-northwest-dipping bedded sequence of limestone and intercalated tuffaceous andesitic rocks grading into less calcareous tuffs and volcanic sediments towards the south. Drill hole 23 intersected amygdaloidal andesite porphyry overlying non-amygdaloidal andesite porphyry and relatively continuous limestone at depth. Diorite dykes up to 25 m thick intrude the limestone. The stratigraphy contained no iron oxide mantos or significant copper grades. Drill hole 40 intersected a 160 m long sequence of intercalated andesite tuff, limestone, and calcareous sediment, overlying a more massive andesite tuff. No iron oxide or copper mantos were intersected in the hole; however, a narrow interval of chalcopyrite and pyrite mineralization occurs at the base of the limestone. The significant differences in the lithologies of these structural blocks have previously been interpreted as caused by a fault between the blocks with significant vertical and/or strike-slip offset. Recent diamond core drilling in 2010 targeted the fault that was thought to be the limit of mineralization in order to

better define its position. Several drill holes failed to intersect a fault zone, but intersected a gradual intercalation between the two lithologies described above. It appears that mineralization weakens towards the west as a function of lower host rock reactivity or diminished strength of fluid flow caused by larger distance to the conduit.

The geology to the south of the SDS deposit is somewhat distinct from the rest of the property. Here, the Bandurrias-Chañarcillo Group rocks have been intruded by a series of small diorite plugs and sills (Sierra Santo Domingo plutons, 110–100 Ma). These intrusions are relatively fresh, fine- to medium-grained equigranular plutons, with 15% to 20% hornblende, partly altered to chlorite. The intrusions have a halo of calc-silicate skarn, presumably altered limestone or calcareous sedimentary rock of the Chañarcillo Group. The skarn is primarily composed of calcite, epidote, and garnet, commonly with 15% to 20% clots up to one centimeter of a mixture of specular hematite, magnetite, and rarely copper oxides. In some places along the intrusive contacts, the iron oxides (largely magnetite) can be semi-massive, occurring in two- to three-meter thick mantos. The package described above has been intersected by drill holes 123 through 126 (Figure 7-6).

The area to the northeast of the SDS 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. Figure 7-6 shows that the Iris deposit is essentially blind, covered by a sequence of Quaternary gravel. Drill holes suggest that the Iris deposit consists of iron oxide mantos hosted by andesitic tuffs and andesitic breccias. The elongated shape of the deposit, and textures observed in diamond drill holes, indicate that the Iris deposit has formed in a north-northwest-striking fault zone that is bounded by a west-dipping fault that can be traced along most of the deposit's western side. The eastern side of the deposit is bordered by a steeply dipping fault that divides andesitic tuffs on the western side from calcareous sedimentary rocks and limestone on the east. While these rocks host some copper mineralization, they seem to have been less susceptible to replacement by iron oxides than the andesitic tuffs.

Between the two structural blocks that host the SDS and Iris deposits respectively, there is another fault block that consists of andesitic flows hosting massive magnetite mantos that are barren of copper mineralization (drill holes 351 through 359 and 96). The iron oxide mantos in this fault block are hosted by andesitic flows, which is quite unusual for the area as most iron oxide mantos are hosted by tuffaceous rocks. It appears that the mineralization of the Iris Norte deposit is at least partly hosted in andesitic flows as well.

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 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 and appears to increase towards the north. The Iris Norte deposit is entirely covered by this gravel sequence. Mineralization at Iris Norte is very similar to the Iris deposit. However, part of the mineralization appears to be hosted by andesitic flows, rather than the tuffs that are typical for SDS and Iris. The Iris Norte deposit is also elongated in shape and seems to have formed in a structural zone (Figure 7-7). The deposit displays a northeasterly strike, which is a rotation of approximately 55° clockwise from the strike of the Iris deposit. The Iris Norte

deposit has been intruded by significant amounts of diorite dykes and sills that separate the deposit into two lenses. The amount of diorite in Iris Norte is considerably higher than in Iris.

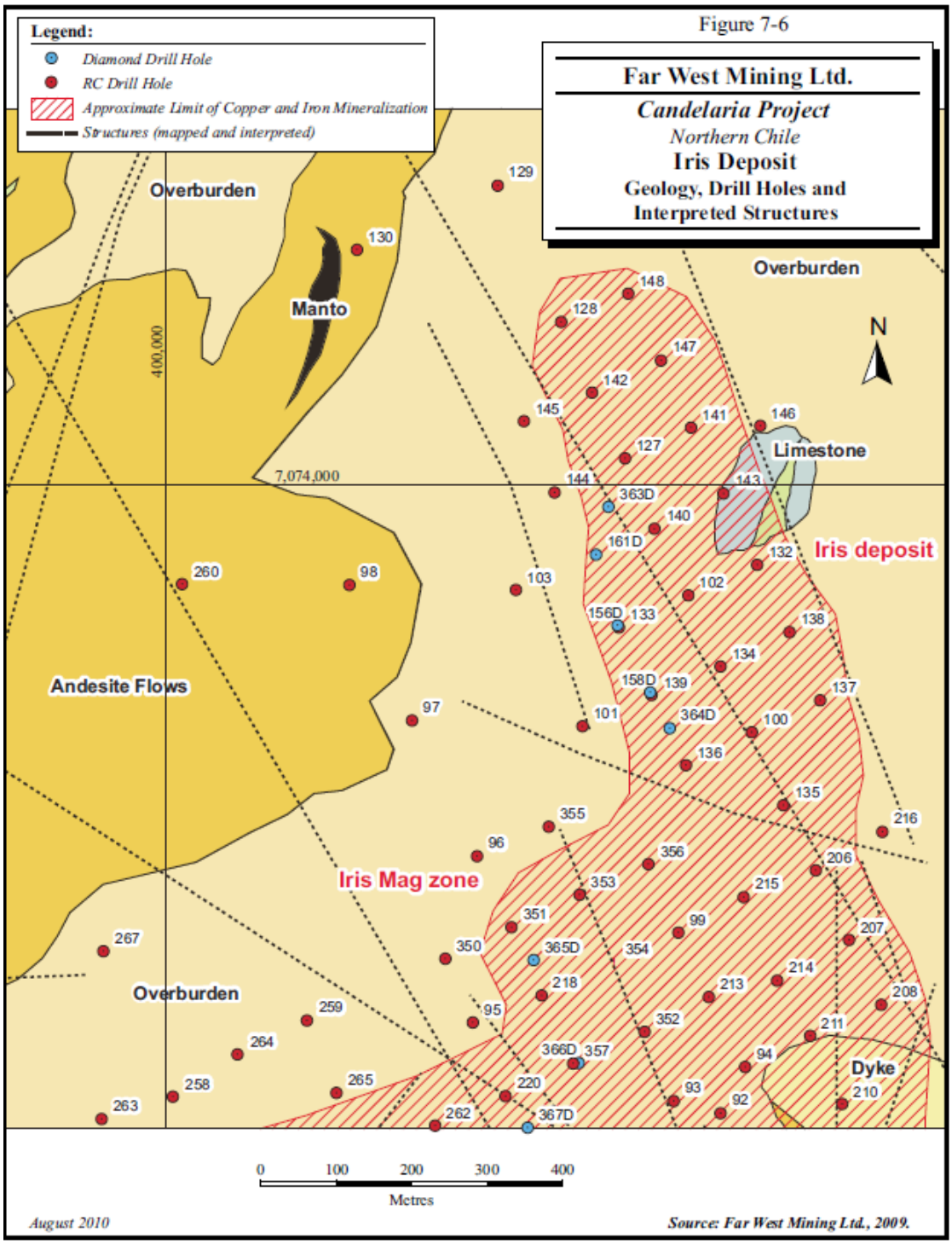


Figure 7-6: Iris Deposit – Geology, Drill Holes and Interpreted Structures

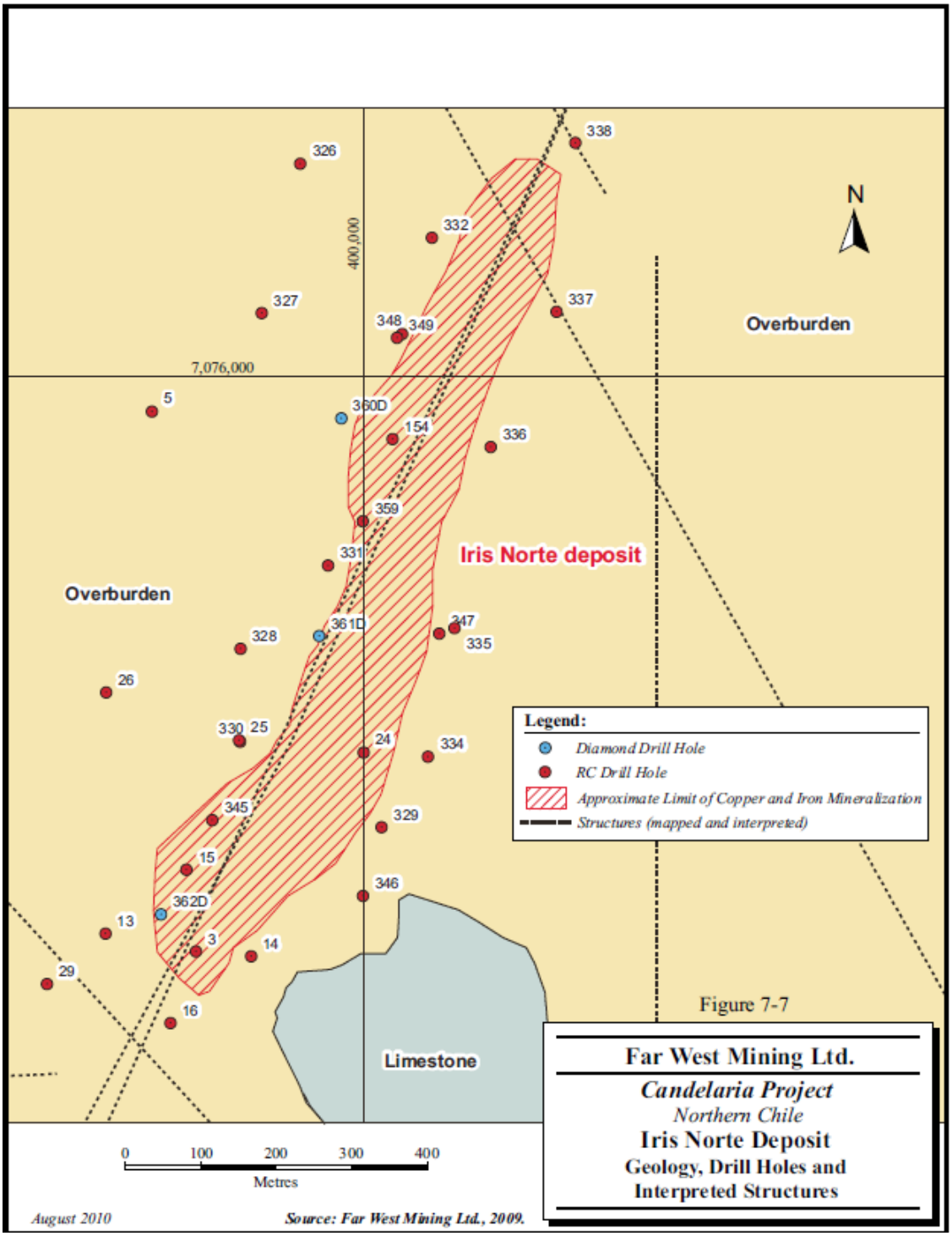


Figure 7-7: Iris Norte Deposit – Geology, Drill Holes and Interpreted Structures

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.

Mineralization at Estrellita is hosted by andesitic porphyries, typically just above an andesite breccia that has not been mapped at surface. The mineralization consists of both flat-lying and steeply-dipping mineralized structures, as well as iron oxide mantos that seem to have formed where both types of major structures intersect. The mineralization probably occurs at a higher stratigraphic level than at SDS, Iris, and Iris Norte, which are hosted in tuff sequences below the level of mineralization at Estrellita. The tuff sequences below Estrellita are intercalated with volcanic sediments and limestone, while limestone is practically absent from SDS. The structural break between the southern and northern blocks at Santo Domingo is under cover and has not been explored by drill holes at this point.

7.3.3 Structure

The Santo Domingo area is structurally complex, cut by an array of faults that trend variably north, northwest, northeast, and east-west. These faults are complex and seem to have been active repeatedly through time. Many mark the boundary of pronounced lithological changes, namely the faults that bound the SDS deposit to the south and east and separate the deposit from adjacent, non-mineralized blocks.

Limited mapping and recognition of outcrop-scale open folds indicate that the sequence described above has been gently folded along north-northeast-trending axes. Rocks in the area to the south (Target Area 4a2) have been more intensely deformed, possibly due to emplacement of several small diorite plugs that caused the developments of skarns around their margins.

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 in the fault that bounds SDS to 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 Estefanía areas. It is a large, east-west trending, steeply north-dipping, north-side-down block fault, with a probable right lateral strike-slip component. Bedding in sedimentary rocks adjacent to the fault in the south block has obviously been dragged down steeply to the north. This structure is clearly apparent on the FALCON™ magnetic susceptibility plots as a series of prominent magnetic lows, which collectively form a 7 km lineament crossing the entire area. The fault can be tentatively traced to the west for an additional 10 km. Based on offset magnetic features, horizontal movement along the fault appears to be dextral and in the order of 200 m to 300 m. Dip-slip movement is tentatively placed in excess of 200 m north side down, based on the offset of crudely defined stratigraphic packages observed in drilling. Most of the historic copper production in the area comes from or near this structure, which appears to be one of the more important controls to mineralization in the district.

The most prominent fault set, as interpreted from magnetic lows, trends northwest and has fault separations of roughly one kilometer. No corresponding topographic lineaments were noted. A set of narrow (generally less than 2 m), closely spaced (100 m to 200 m), carbonate-specularite-copper oxide veins cut the andesites in the north part of the 4a3 (Santo Domingo property) area. They dip moderately to steeply to the northeast, appear to parallel the inferred faults, and may be genetically related to these larger structures. Several northwest-trending faults are recognized in the SDS area. One of these, with northeast-side-down movement, is determined from drill intersections of the manto sequences, where holes to the northeast of the fault intersected the manto sequence approximately 35 m to 45 m deeper than projected. A possible parallel fault, just southwest of drill hole 63, may have dropped the manto sequence a further 35 m.

In summary, high-angle block faulting played an important role in localizing manto- and fault-related iron oxide-copper mineralization in the Santo Domingo area. These faults have uplifted the central part of the SDS area, bringing the manto succession close to surface. To the east and south, it is possible that this prospective horizon is present at depth, beneath a cover of limestone.

An illustration of the simplified geology of the Santo Domingo Sur, Iris and Iris Norte sections is shown in Figure 7-8, Figure 7-9 and Figure 7-10, respectively.

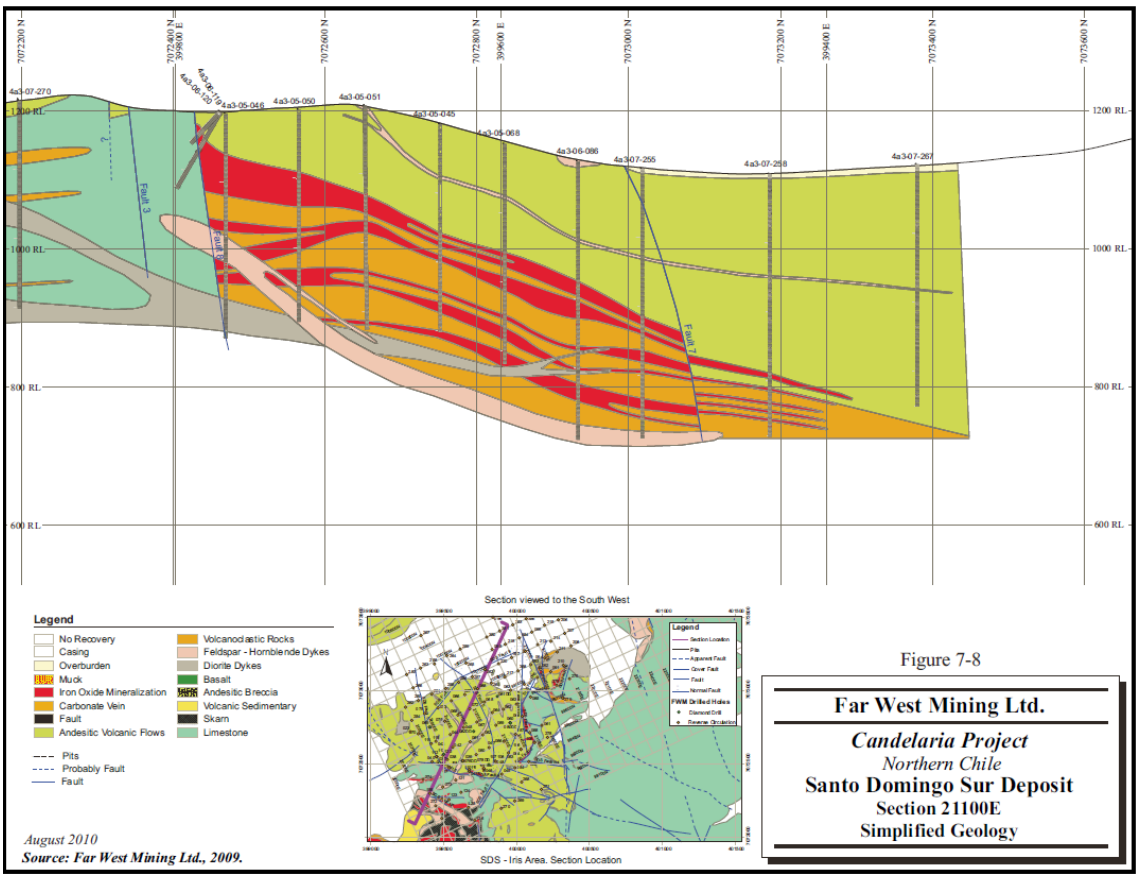


Figure 7-8: Santo Domingo Sur Section 21100E – Simplified Geology

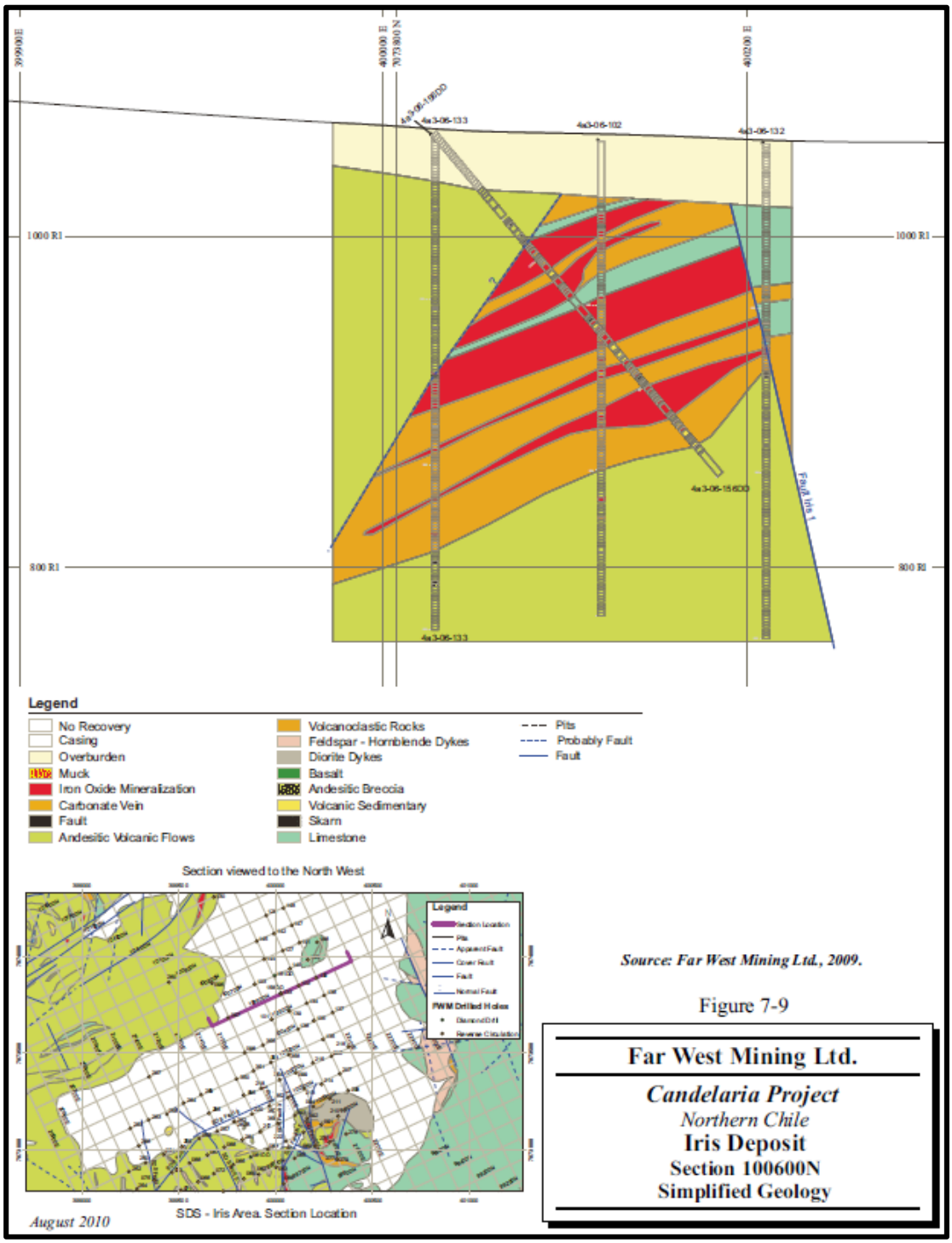


Figure 7-9: Iris Section 100600N – Simplified Geology

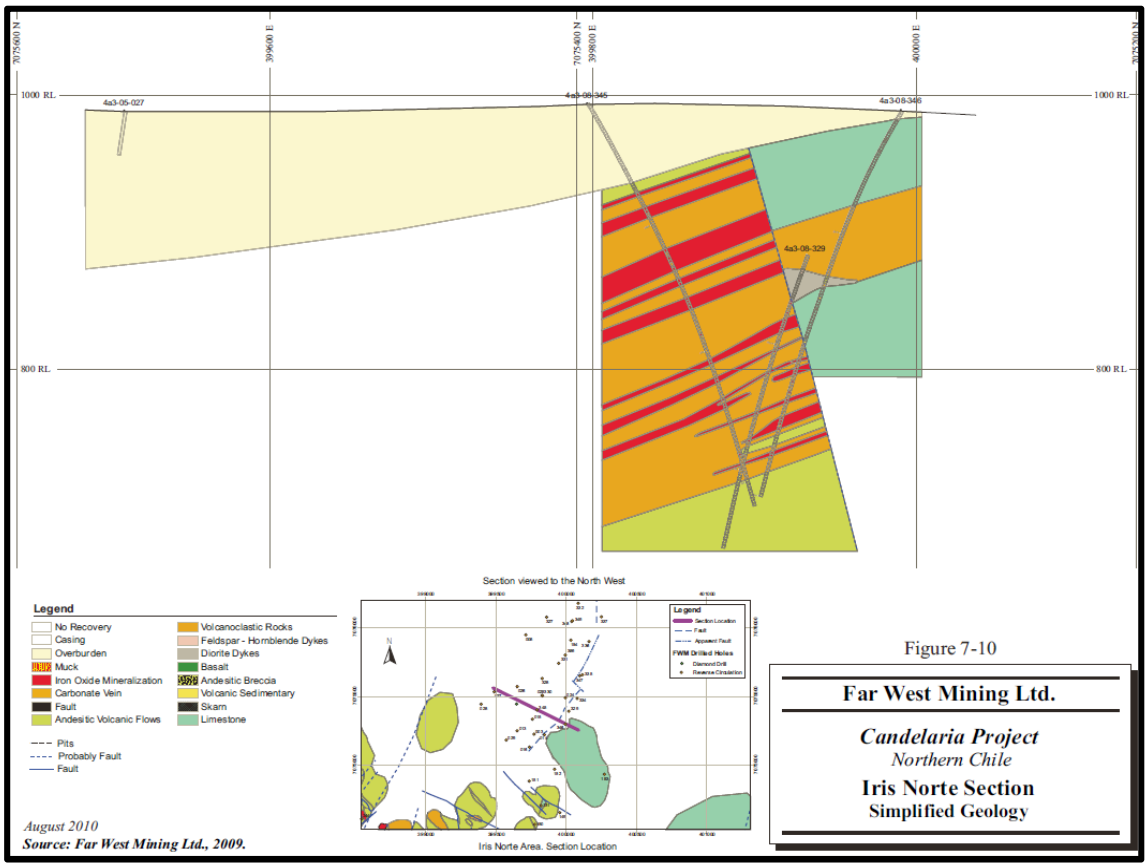


Figure 7-10: Iris Norte Section – Simplified Geology

7.4 Mineralization

The following section is an extract from the RPA mineral resource estimate dated 2010.[9]

Copper-bearing IOCG-type mineralization is widespread in the Santo Domingo area. Specular hematite and copper oxides (including chrysocolla, brochantite, and malachite) are the typical near-surface mineral assemblages. Copper oxides typically persist to 70 m to 90 m below surface, with chalcopyrite being the dominant copper mineral at greater depths. Modes of occurrence are:

- stratiform replacement mantos and breccias within tuffaceous sediments (e.g., SDS 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, ranging from a few centimeters to several meters in width
- minor copper oxide minerals disseminated in amygdules in volcanic flows and encountered as small chalcocite nodules in limestone.

[9] Far West Mining Ltd. Technical Report on the Santo Domingo Property, Region III, Atacama Province Chile, NI 43-101 Report, Author: David W. Rennie, P.Eng. August 26, 2010 Scott Wilson Roscoe Postle Associates Inc.

Manto mineralization in tuffaceous or calcareous sediments is widespread on the property. In the Estrellita and Estefania areas, several gently north-dipping, strata-bound 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 strata-bound, but to the south are discordant, following the steeply-dipping Santo Domingo fault, suggesting that this fault may have been a fluid conduit.

The mantos appear to have provided the bulk of material mined in the Santo Domingo district. There is evidence that they are thicker and more copper-rich close to the east-west Santo Domingo fault.

In the SDS deposit, copper mineralization occurs in a sequence of iron oxide mantos within a tuffaceous package between andesitic flows. 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. 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 SDS deposit is most strongly developed in the southern part and in the upper levels of the deposit. Copper grade and intensity of the mineralization weakens towards the northern part of the deposit and with depth. The high-grade core of the deposit is located along the southern margin and close to surface. It appears likely that the bordering fault in the south of the deposit has been the main conduit for mineralizing fluids, as mineralization and alteration is strongest along that fault.

The Iris deposit is approximately 500 m wide, with a strike length of 1,600 m. The deposit consists of iron oxide mantos and breccias along a north-northwest-striking fault zone. 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 than in the SDS deposit, owing to the fact that structural control seems to have played a greater role in the Iris deposit than in the more continuous stratiform replacement style mineralization at SDS. The dominating iron oxide at Iris is hematite, while 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. The mineralization is hosted by a specularite manto that is cut by steeply-dipping structures. The extent of mineralization at surface is approximately 100 m by 60 m.

The Iris Mag zone is located between the Iris deposit and the SDS deposit in a separate structural block. Mineralization in the zone consists of magnetite and chalcopyrite, with 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 in the other zones. It appears that this part of the deposit has been subject to the initial high temperature magnetite event, but shows little evidence of a later oxidizing overprint that has introduced high copper and gold content elsewhere.

The Iris Norte deposit is located about 600 m to the north of the Iris deposit. The deposit is very similar in character, 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. The strike direction of the deposit displays a rotation of approximately 55° clockwise from the strike of the Iris deposit. Mineralization in the Iris Norte deposit consists of mixed magnetite and hematite mantos hosted by a sequence of andesitic tuffs and flows. The fact that some of the host rocks are flows is unusual for the general area, where most mineralization is hosted by tuffs. The main sulphides in Iris Norte are pyrite and chalcopyrite, with the latter providing the copper content of the deposit. The deposit has been intruded by diorite sills and dykes that separate the mineralization into two lenses. The deposit contains a higher proportion of magnetite than the Iris deposit, and there is a higher proportion of intrusive rocks. Both deposits are very similar in character and were possibly continuous before tectonic activity caused a disruption and rotation of the two deposits.

The Estrellita deposit is an east-west-striking, flat-lying to shallowly north-dipping tabular body lying approximately 3.5 km northwest of SDS. The zone 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 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.

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 horizontal and a steeply dipping set of specularite structures.

Copper mineralization at Estrellita mainly consists of copper oxides such as brochantite, chrysocolla, pitch limonite (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.

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 have been extensively pitted and have historically supported very small-scale mining operations, even collectively they do not appear to have had significant production.

A large, northwest-trending structure is interpreted from magnetics to trend between the Estrellita mine and the SDS deposit. It is possible that the intersection of this structure with the Santo Domingo fault in the north, and with an east-west structure in the SDS area may have provided the structural focus for hydrothermal fluids and mineralization in both areas.

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.

In limestone, copper occurs rarely as small chalcocite nodules with associated malachite. Grab samples of this material graded up to 0.595% Cu. It is uncertain how these disseminated copper occurrences are genetically related to the vein and manto mineralization.

8 DEPOSIT TYPES

The following section is an extract from the RPA mineral resource estimate dated 2010[10].

The largest and most extensively mined iron oxide-copper-gold ore (IOCG-type) deposits in Chile occur within a structurally complex zone 630 km by 25 km extending between La Serena and Taltal. Deposits within the Chilean iron belt (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 as an example, Los Colorados mine south of Copiapó is still in production. Examples of copper-rich IOCG deposits in the belt include Candelaria (470 Mt at 0.95% Cu) and Manto Verde (250 Mt at 0.75% Cu). A list of IOCG-type deposits is presented in Table 8.1.

Table 8.1: Selected IOCG – Type Deposits – Santo Domingo Property

			Grades			
Deposit	**Location**	**Mt**	**%Cu**	**g/t Au**	**g/t Ag**	**Source**
Olympic Dam	Australia	2,000	1.60	0.60	3.5	Mark et al., 2001
Kiruna	Sweden	450	0.39	0.21	4.5	"
Candelaria	Chile	470	0.95	0.22	3.1	Marschik et al., 2000
Mantos Blancos	Chile	400	1.00	-	-	Hopper and Correa, 2000
Manto Verde	Chile	250	0.75	-	-	"
El Soldado	Chile	200	1.50	-	-	"

The description of the Candelaria and Manto Verde deposits that follows provides more detail on the IOCG-type deposits in the CIB.

8.1 Candelaria Deposit

The Candelaria deposit has published reserves of 470 Mt grading at 0.95% Cu, 0.22 g/t Au, and 3.1 g/t Ag (Marschik et al., 2000). It 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.

[10] Far West Mining Ltd. Technical Report on the Santo Domingo Property, Region III, Atacama Province Chile, NI 43-101 Report, Author: David W. Rennie, P.Eng. August 26, 2010 Scott Wilson Roscoe Postle Associates Inc.

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

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

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

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

8.2 Manto Verde Deposit

Anglo American's (Mantos Blancos) Manto Verde mine is located approximately 104 km north of the Candelaria deposit and 25 km southwest of the Santo Domingo area. It is a 250 Mt shear-hosted IOCG-type copper oxide deposit with an average grade of 0.75% Cu (Hopper and Correa, 2000).

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 subaerial 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, however, many prominent north-south structures apparent on both sides of this complex Atacama fault zone, and it is clear that the actual zone of deformation is much wider. Volcanic rocks have been cut by numerous phases of north-south elongated granitic to dioritic intrusions. These are interpreted to be syntectonic emplacements along the Atacama fault complex.

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

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 and 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. 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 Las Tazas pluton has been dated at 130 – 126 Ma, and the Sierra Dieciocho pluton at 126 Ma – 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 approximately 4 km to the east of the pit. Late north-trending mafic dykes cut all rock types, alteration assemblages, and mineralization.

8.3 Candelaria Project Exploration Strategy

In Chile, the massive magnetite deposits of this class are typically copper deficient. Many of the copper-rich IOCG deposits in the CIB region (such as Manto Verde) consist of predominantly specularite veins and breccia fillings along large fault structures. It was believed that the FALCON™ airborne gravity survey could identify large masses of unexposed iron oxide and, in conjunction with the magnetic data, differentiate between massive magnetite and specularite bodies. Hence, the gravity anomalies without coincident magnetic susceptibility were the main focus of the exploration program. Additional criteria, such as copper mineralization and large structural breaks in proximity to these anomalies, were used to prioritize drill targets.

9 EXPLORATION

The following section is an extract from the RPA mineral resource estimate dated 2010[11].

9.1 Exploration Program Summary

Exploration work in the Santo Domingo area was conducted by FWM from July 2003 to May 2010. It consisted of:

- 50 km^2 of geological mapping at 1:25,000
- 50 surface rock samples for analysis for Au and a 27-element Inductively Coupled Plasma (ICP) suite
- 47 sieved (106 micron) drainage sediment samples for analyses as above
- 17.6 km of Induced Polarization (IP) survey
- A total of 106,886 m of drilling in 398 holes, including 90,611 m of reverse circulation (RC) drilling in 348 holes and
- 16,275 m diamond (core) drilling in 50 holes
- Analysis for gold and 27-element ICP on two-metre intervals for RC and one-metre intervals for core

Details of the previous exploration can be found in Allen and Höy (2005). The initial Mineral Resource estimate for SDS can be found in Lacroix (2006). Updated estimates for SDS and an initial estimate for Iris and Estrellita can be found in Lacroix and Rennie (2007), and Lacroix (2009).

9.2 Surface Rock Chip Sampling Program

A total of 50 rock chip samples were collected from the Santo Domingo area and sent to ALS Chemex Laboratories in La Serena (ALS Chemex) for gold and 27-element ICP analyses. Samples with over 10 g Au and over 10,000 ppm Cu were assayed and bubble plots of copper and gold values produced. Samples were generally taken where copper oxides were apparent, and hence most samples contained anomalous levels of copper.

9.3 Drainage Geochemistry

A total of 47 sediment samples were collected from drainages within and immediately peripheral to the Santo Domingo area. The samples were analyzed by ALS Chemex for gold and a 27-element ICP package. Most drainage channels in the area were sampled.

[11] Far West Mining Ltd. Technical Report on the Santo Domingo Property, Region III, Atacama Province Chile, NI 43-101 Report, Author: David W. Rennie, P.Eng. August 26, 2010 Scott Wilson Roscoe Postle Associates Inc.

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. Drainages in the areas underlain by andesite flows, especially in the north and northwest part of the target area, are generally anomalous, with copper values typically in excess of 400 ppm. This broad anomaly is roughly coincident with the widespread distribution of northwest trending specularite-copper oxide mineralized veins that cut the andesites. The highest copper value in drainage sediment (sample 7954) was 1,865 ppm from within the Santo Domingo area, approximately two kilometres east-southeast of the Estrellita mine. No associated bedrock mineralization is known.

9.4 Airborne Gravity and Magnetic Susceptibility Surveys

FALCON™ gravity and magnetic susceptibility images were produced for data from Quantec Geofisica Limitada. Gravity anomalies define a cluster of northwest trending features up to five kilometres long. Most of the significant mineralization in the Santo Domingo area is coincident with the gravity anomalies, making these areas high-priority exploration targets.

Magnetic susceptibility, especially the first derivative plot, is an excellent tool to define fault structures. Since IOCG mineralization is often structurally controlled, structural interpretations are important for helping to pick out drill 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 17 km. Many of the important mineralized zones in the Santo Domingo area appear to be related to this fault, and its entire surface trace is therefore prospective.

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

The northwestern part of the Santo Domingo anomaly (Estrellita area) is generally parallel to a series of west-northwest striking faults as defined on the magnetic images, and to a closely spaced series of specularite and copper oxide-bearing veins, stockwork and shear zones cutting the andesite. These mineralized veins occur both within the anomaly and outside the anomaly to the south. Copper-bearing manto mineralization at the Estrellita mine 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 Project area and hence the Santo Domingo target has received a good deal of exploration work, which subsequently led to the discovery of the SDS, Iris, Iris Norte, and Estrellita deposits.

Magnetic susceptibility in the Santo Domingo gravity anomaly area is generally high except where cut by faults, most notably the east-west Santo Domingo fault and a prominent

northwest-trending fault along the southwest side of the anomaly. In drillholes 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, magnetite content, both disseminated and in magnetite mantos, was estimated to be 5% to 10%. These amounts of magnetite would easily explain the magnetic anomaly, and probably the gravity anomaly as well. It appears that magnetite in the oxide zone (near surface) and within the Santo Domingo fault (to depth) has been largely altered to specularite. This would 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 SDS area is located in the extreme southeast part of the Santo Domingo anomaly. It was chosen as a drill target because there were specularite-copper oxide mantos exposed on the flank of a 500 m wide gravity anomaly. The south 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 SDS deposit. The deposit has a gravity and magnetic signature that reflects the high content of magnetite and hematite in the deposit.

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 Iris. The Iris Norte has a magnetic expression that is less pronounced than the Iris anomaly.

9.5 Resistivity Survey

A total of 17.6 km of IP survey were completed in the Santo Domingo area to test for chargeable zones within the gravity and magnetic features. Work was conducted in two phases from April to August 2004 by Quantec Geofísica Ltda. of Antofagasta, Chile. The grid baseline (100000N) was established with an azimuth of 279º in order to parallel the general trend of the Santo Domingo gravity anomaly. Cross lines were orthogonal to the anomaly trend. Stations were located using a differential GPS. The time-domain IP survey used a pole-dipole array with a 100 m station separation.

The IP survey generated chargeability anomalies in various parts of the Santo Domingo area. Subsequent drilling of the resulting 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.

10 DRILLING

The following section is an extract from the RPA mineral resource estimate dated 2010[12].

10.1 Introduction

Drilling has been conducted in the Santo Domingo area since May 2004. FWM has completed 348 RC drillholes in the target area for a total of 90,611 m and 50 diamond drillholes for a total of 16,275 m. As of May 31, 2010, drilling in the Santo Domingo area totalled 106,886 m in 398 holes. The drilling statistics are summarized in Table 10.1.

Table 10.1: Drilling Statistics

Far West Mining Ltd. – Santo Domingo Property			
Area	**No. of Holes**	**Type**	**m**
Santo Domingo Sur	103	RC	31,810
Santo Domingo Sur	26	DD	10,697
Estrellita	143	RC	30,528
Estrellita	13	DD	2,366
Iris/Iris Norte	102	RC	28,273
Iris/Iris Norte	11	DD	3,212
Totals	**398**		**106,886**

Note. The numbers may not add due to rounding.

Drilling was contracted to Harris y Cia., Major Drilling, Geo Operaciones and Captagua, all based in Chile. Most of the RC drilling was conducted by a truck-mounted Schramm Rotadrill, using a centre return hammer and a 5.5 in. (13.97 cm) carbide button bit. The diamond drilling was conducted by various types of equipment. HQ core (63.5 mm diameter) was typically drilled to a depth of approximately 300 m, below which NQ core (47.6 mm diameter) was drilled. Drilling was conducted in two 12-hour shifts per day. Samples, taken in two-metre intervals for RC, were collected by drilling personnel, and tagged and organized by FWM personnel. A FWM geologist was generally on site during most of the day shift for RC drilling.

[12] Far West Mining Ltd. Technical Report on the Santo Domingo Property, Region III, Atacama Province Chile, NI 43-101 Report, Author: David W. Rennie, P.Eng. August 26, 2010 Scott Wilson Roscoe Postle Associates Inc.

Diamond drill core was sampled in one-metre (all DD holes before 2010) or two-metre (DD holes 2010) intervals that were marked by FWM geologists in order to adjust the samples to geological units.

Most holes are vertical as mineralization at SDS 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 collars were located using a differential GPS. Coordinates are accurate to within one metre or less. Relative elevations between holes in close proximity (such as at SDS) were determined using a tight chain and clinometer.

Drill cuttings and core were logged using a set of codes similar to those used for surface mapping. All geological data were entered digitally into summary logs. All digital data (analyses and geological logs) were subsequently entered into an MS Access project database for presentation and section generation.

10.2 Santo Domingo Area

To date, 398 holes have been completed in the Santo Domingo area. Drilling was originally designed to target gravity and magnetic anomalies for IOCG mineralization of Candelaria or Manto Verde style (see Section 8). In April 2005, drillhole 22 intersected iron oxide mantos with copper mineralization of grade and width that have the potential to be economic. Further drilling in the area outlined the SDS deposit. The deposit shows characteristics that are very similar to the Candelaria deposit.

Subsequent drilling to the northwest of SDS following a north-northwest trending gravity anomaly discovered and outlined the Iris deposit with mineralization of similar style to SDS. Additional drilling in the northwestern part of the Santo Domingo area, around the small-scale Estrellita mine workings, outlined the Estrellita deposit which is more similar to Manto Verde as it represents copper oxide mineralization along a fault zone. In 2008, FWM followed up on mineralized intercepts from earlier drilling (4a3-003, 4a3-154) in an effort to identify the northern extension of the Iris deposit. The 2008 drilling outlined a new zone of mineralization known as Iris Norte. Drilling in 2010 was conducted within the limits of previously established mineralization in order to collect sample material for a metallurgical test program in the context of a pre-feasibility study. Diamond drillholes were designed to test all zones of the deposit and provide sufficient sample material in proportion of the tonnages for each zone. RC drillholes were designed to tighten the drill spacing within the initial proposed mining area and to provide sample material for metallurgical test work.

Additional holes have been drilled to test other gravity and magnetic features in the Santo Domingo area and intersected widespread but discontinuous copper mineralization around the four outlined deposits. Additional targets in the area remain untested which FWM reports will be explored in the future.

10.3 Santo Domingo Sur Deposit

The SDS deposit is defined by 129 drillholes (103 RC and 26 diamond holes). Most of the drillholes define the extent of copper mineralization on a 100 m spaced grid that is aligned according to the dip and strike of the deposit. Another set of predominantly inclined holes located around the periphery of the deposit establishes the limits of the deposit or intersects different fault blocks that are not mineralized. Drilling at 100 m centres or less 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 southern and eastern margins of the deposit appear to be structural and are defined by drillholes 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 SDS deposit is bounded by faults in the south and east. The southern boundary of the deposit is defined by an east-west fault immediately south of hole 41 (Figure 7-5). Iron oxide-copper mineralization, exposed at surface in a gully, clearly marks this fault with thick limestone to the south in the south block. This fault appears to be a thrust with the fault plane dipping towards the south.

The west boundary is defined by a number of drillholes that define the transition between a tuffaceous sequence that hosts the mineralization towards a largely barren sedimentary sequence in the west. Inclined diamond drillholes failed to intersect a structural boundary but rather defined a gradual transition between the two sequences with mineralization weakening towards the west.

The eastern boundary is defined between drillholes that intersected a thick limestone unit (200 m to 300 m) with occasional intervening andesite flows or tuffs in the northeastern block (49, 54, 55 and 61) and those immediately to the west that all intersected the manto succession. An irregular, north-trending zone of dominantly massive and veined specularite, with occasional copper mineralization, marks the faulted boundary between the two blocks. Layering in the limestone is truncated by this fault.

The bounding faults are interpreted to be normal faults, with the structural block that hosts the deposit forming a horst structure, surrounded by down-dropped blocks either dominated by or containing considerable limestone.

Drilling indicates that the SDS deposit strikes approximately northeast and dips at low angles to the northwest. A northwest-trending fault, only recognized by drill intersections, appears displace the northeastern portion of the deposit down by approximately 45 m to 65 m.

Massive to semi-massive iron oxides are the main component of the manto horizons. They are specularite dominant near surface and along the western, southern, and eastern margins of the deposit forming a halo around the magnetite rich core of the deposit. Chalcopyrite is the dominant copper mineral. It is associated with both specular hematite and magnetite, but

is commonly more abundant with magnetite. Chalcocite and bornite were observed in traces hosted by specularite in the upper parts of the deposit, particularly in the northeast and at the western margin of the deposit. The mantos routinely also contain between 2%-10% pyrite in addition to the copper sulphides. Copper oxides have been observed at the southern end of the deposit where mineralization is closest to surface.

The host rocks show weak to moderate chlorite and sericite alteration, with sporadic potassic feldspar and actinolite. In a very general sense, potassic feldspar is associated with specularite in the upper parts of the deposit. Actinolite is typically associated with magnetite, and is more abundant deeper in the sequence. Although not noticed during logging, probably due to very small grain sizes, metallurgical work has shown that there is a significant amount (up to 15% in the mantos) of iron carbonate alteration. Main minerals are siderite and ankerite with lesser amounts of magnesite and dolomite. Both siderite and ankerite do not react to field tests with diluted hydrochloric acid, which explains why these alteration minerals went undiscovered.

10.4 IRIS, Iris Norte and Iris Mag Deposits

Following the discovery of the SDS deposit, additional drilling was conducted along a distinct north-northwest trending gravity anomaly that originates at the northeast corner of SDS. The Iris and Iris Norte deposits are defined by 102 RC holes and 11 diamond drillholes that intersected barren iron rich mantos and copper-bearing mantos and breccias in various locations along the gravity anomaly. The Iris deposit was discovered along the eastern flank of the gravity anomaly while exploring the gravity anomaly at 100 m to 200 m spacing. Further north, the Iris Norte deposit occupies the eastern edge of the same gravity anomaly, however, the strike has rotated clockwise from north-northwest to north-northeast. The deposit is completely covered by Quaternary gravel and was discovered during follow-up of earlier drillholes.

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 along the eastern boundary of the gravity anomaly. The deposit is close to surface at the southern end and plunges towards the north. The RC holes intersected copper mineralization of similar character as the SDS deposit. Three inclined diamond holes (156D, 158D, and 161D) were designed to resolve the internal structure of the Iris deposit that had been poorly defined by the 100 m spaced RC holes.

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. Copper oxides have been observed at the southern end of Iris where the deposit is closest to surface.

The Iris Norte deposit is very similar in character but is completely covered by younger gravel. Part of the deposit is hosted by andesitic flows as opposed to andesitic tuffs that host mineralization elsewhere on the property. Mineralization is magnetite rich, with chalcopyrite as the main copper mineral. Copper oxides have been observed in almost every drillhole to a depth of up to 100 m. The deposit contains more dioritic intrusions than Iris. The southern end of Iris Norte almost reaches the surface with mineralization plunging towards the north where it is covered by in excess of 150 m of gravel. The eastern and western edges of the

deposit are defined by faults that are very similar in orientation to the faults that bound Iris, i.e., gently westerly dip of the western fault and steep easterly dip of the eastern fault. The deposit is open towards the north but may be cut off by the intersection of the afore-mentioned east-dipping and west-dipping faults. The known extent of Iris Norte is about 1,600 m (north-south) by 500 m (east-west). The manto sequence is up to 300 m thick and dips to the west and plunges to the north at shallow angles.

The Iris Mag zone is a small pocket of magnetite rich mantos in a fault block between Iris and SDS. Mineralization is hosted by andesite flows and andesite breccias and doesn't have a high sulphide content. Copper grades are generally low in this zone.

10.5 Estrellita Deposit

Subsequent to the discovery of SDS and Iris, FWM explored the area around the small Estrellita mine workings where high grade copper oxide ore has been mined at surface by local miners. A total of 156 holes (143 RC and 13 diamond holes) have been drilled in the Estrellita area. Drilling outlines a tabular body of copper mineralization hosted by breccias and mantos along a fault zone around the Estrellita mine workings.

The mineralization is faulted down by roughly east-west striking faults to the south and north of the main zone around the old workings where drillholes intersected the mineralized zone at deeper levels. Vertical displacement along the faults varies from roughly 60 m to about 100 m. 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. Additional drilling will be conducted in the future to determine the full extent of mineralization at Estrellita.

The Estrellita deposit has an as yet unquantified oxide component that makes the deposit comparable to Manto Verde, which is located approximately 20 km to the southwest. The main copper minerals at surface are chrysocolla, brochantite, and various amorphous copper oxides such as pitch limonite, tenorite, and copper wad. The oxide component generally diminishes with depth where copper mineralization consists of chalcopyrite and hypogene chalcocite.

Metallurgical studies will have to be conducted in the future to determine the oxide-sulphide ratio at Estrellita in order to determine the economic viability of a possible oxide leach operation.

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.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The following section is an extract from the RPA mineral resource estimate dated 2010[13].

11.1 Sampling Method and Approach

The sampling protocols have remained unchanged from the last Technical Report described above.

Reverse circulation drill cuttings were blown into a cyclone and collected every two metres 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 one centimetre. 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 resplit and then split again. These three consecutive splits resulted in a final sample one-eighth the size of the initial complete sample. A 2 kg to 3 kg portion of this 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. This complete second set of samples can be used for confirmation assays, petrographic work, etc.

Observed sample recovery was excellent and no intervals with poor recovery were reported. Apart from most overburden material and a few obviously barren bedrock intervals, all samples were sent for analyses. Pre-laboratory sample preparation by Harris (drilling contractor) and FWM personnel was conducted under the supervision of FWM geologists. Samples were sealed in plastic bags using zip strips, subsequently sealed in woven polypropylene sacs, and stored in the drilling camp until collected by ALS Chemex personnel. Once leaving the drill camp on the property, sample security could not be confirmed. However, FWM advises that, in virtually all cases, copper estimates in logged chips correlate well with analytical results.

In the case of diamond drilling, core was placed into wooden core boxes by the drilling contractor at the drill. 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 FWM personnel. 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 in preparation for subsequent triaxial and point load tests.

[13] Far West Mining Ltd. Technical Report on the Santo Domingo Property, Region III, Atacama Province Chile, NI 43-101 Report, Author: David W. Rennie, P.Eng. August 26, 2010 Scott Wilson Roscoe Postle Associates Inc.

The core was then geologically logged noting lithology, mineralogy, etc., 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.

Samples for assay were marked at one-metre intervals by technicians, and subsequently adjusted by the geologist to correspond to major lithologic contacts. Sample lengths were not less than 0.5 m, nor did they exceed two metres. 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, Chile, for analyses.

In RPA's opinion, the sampling methodologies employed by FWM 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.

11.2 Security

Samples were collected at the drill in the case of RC and for the diamond drillholes, at the FWM logging facility in Diego de Almagro. The logging facility is fenced, locked when not occupied, and is secure. Samples are handled only by FWM employees or their designates (i.e., ALS-Chemex personnel).

11.3 Assay Methods

Samples were shipped to ALS Chemex in La Serena, Chile, an independent commercial ISO 9001-certified laboratory. Upon arrival at the laboratory, 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. 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 (see below). 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 has been recognized as understating the Fe content, particularly for high grades. The upper limit for ME-ICP61 is 50% Fe, which resulted in a significant downward bias in the block model grades in previous estimates. FWM has more recently implemented the ALS Chemex ME-ICP81 protocol, which incorporates a more aggressive digestion (peroxide fusion) and has no upper limit to the Fe assays. FWM submitted 7,401 samples for reanalysis using ICP81. This included all samples over 15% Fe inside the block model for which sample material was still available.

Magnetic Susceptibility (MS) measurements are made on site by FWM personnel using a handheld Fugro GMS-2 instrument. Plastic bags of sample reject material from the laboratory are shaken to homogenize the material, then laid flat on a table. The instrument is pressed against the plastic bag and the MS reading 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.

A total of 10,834 MS determinations have been made to date. Of these, 2,093 were conducted on pulps owing to the lack of remaining reject material. Measurements taken on pulps routinely yield lower readings that those taken on rejects. FWM conducted a comparison of 788 samples and concluded that there was a distinct and measureable bias between pulp and reject determinations. A scatter diagram comparing these measurements is shown in Figure 11-1.

In RPA's opinion, Figure 11-1 demonstrates that there is a significant bias between measurements taken on pulps versus rejects. FWM derived an equation, shown in magenta in Figure 11-1 and used it to adjust the pulp measurements to an estimated reject value. This equation was deliberately configured to roughly conform to the bottom of the indicated range of MS values for the rejects, which is a more conservative case. The black line in Figure 11-1 depicts a best-fit second order linear regression line for the data, and the FWM line plots significantly below it. In RPA's opinion, the use of factors on analytical data is generally undesirable, however, in this case FWM has used a conservative approach that is well supported with test work, and so is considered to be acceptable.

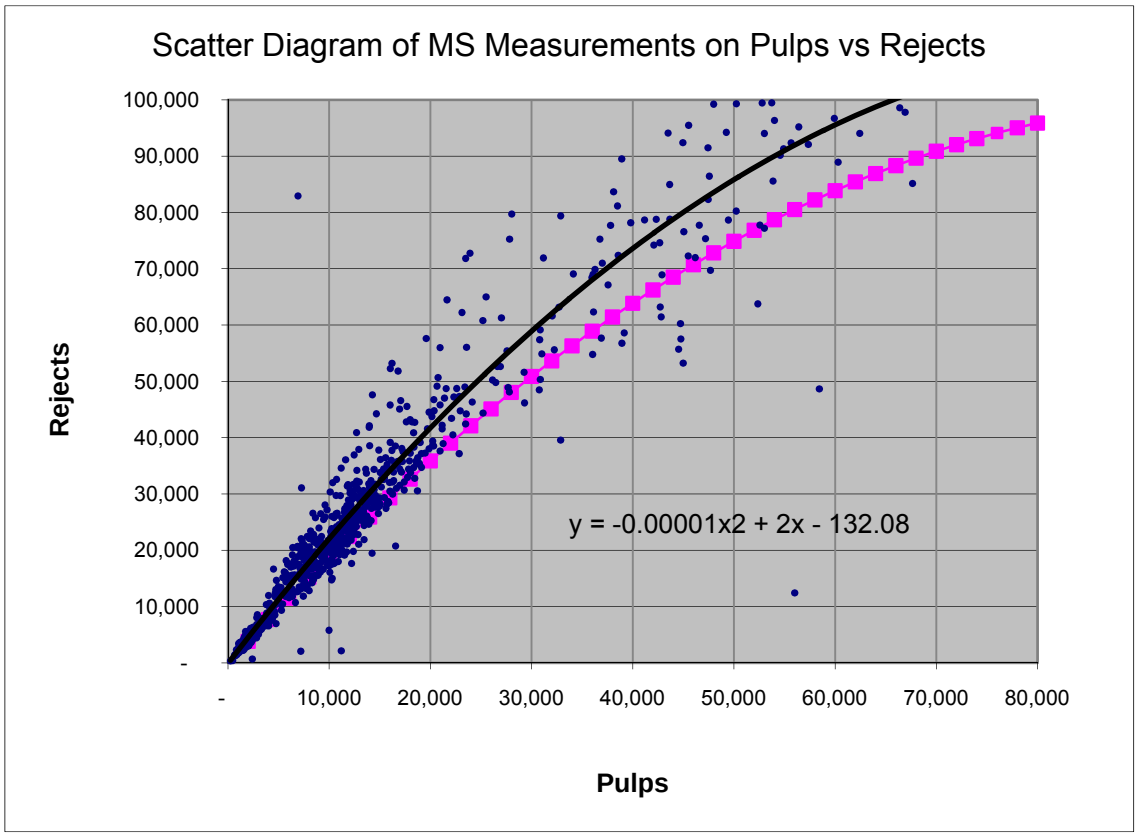


Figure 11-1: Pulp vs. Reject MS Measurement

11.4 Assay Quality Control/Quality Assurance

An independent Quality Control/Quality Assurance (QA/QC) program was implemented by FWM 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 of the programs conducted by FWM. Minor changes have been implemented to accommodate issues and recommendations from past programs, and to include the MS measurements, which is a relatively recent addition to the assay procedures.

RPA reviewed the results of the QA/QC program for the 2006, 2007, and 2009 Technical Reports, and did not find any concerns that would preclude the use of the assay data in resource estimation. Detailed discussion of the QA/QC results are available in those Technical Reports (Lacroix, 2006; Lacroix and Rennie, 2007; Lacroix, 2009). Summaries of

the earlier reviews are provided in this report, along with more detailed discussion of the 2010 results.

11.4.1 Standards

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

2004 to 2007

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 eleven (Table 11.1). RPA reviewed the standards results and noted that while 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.

Table 11.1: Standards 2004 - 2007

Far West Mining Ltd. – Santo Domingo Property		
Standard Number	**%Cu**	**g/t Au**
CDN-CGS-1	0.596 ± 0.029	0.53 ± 0.068
CDN-CGS-2	1.177 ± 0.046	0.97 ± 0.092
CDN-CGS-3	0.646 ± 0.031	0.53 ± 0.048
CDN-CGS-5	0.155 ± 0.006	0.13 ± 0.02
CDN-CGS-7	1.010 ± 0.070	0.95 ± 0.08
CDN-CGS-8	0.105 ± 0.008	0.08 ± 0.012
CDN-CGS-11	0.683 ± 0.026	0.73 ± 0.068

Note: Range is based on 95% confidence interval or 1.96 Std.Dev.

2008 To 2009

Standards used in 2008/09 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 reassayed using a different assay protocol. RPA also noted that the mislabeling 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.

2010

A new set of three standards were prepared which included iron in the suite of elements. The three standards consisted of high, medium, and low grade variants (Table 11.2).

Table 11.2: Standards 2009 - 2010

Far West Mining Ltd. – Santo Domingo Property			
Standard Number	**%Cu**	**g/t Au**	**%Fe**
ALTO (High)	1.631 ± 0.082	0.299 ± 0.051	49.90 ± 1.649
MEDIO (Med)	0.472 ± 0.020	0.098 ± 0.032	30.90 ± 1.244
BAJO (Low)	0.072 ± 0.006	0.011 ± n/a	17.05 ± 0.688

Note: Confidence limits for the low grade CRM for Au have not been established and so it was not used.

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 105% of the nominated confidence limit to check for accuracy. Results were also plotted against the mean of the CRM results plus or minus two and three standard deviations to check for precision. RPA reviewed the plots prepared by FWM personnel. There were isolated instances of assays that were outside of the control limits, but no evidence of significant concerns or systematic errors in the assaying. RPA notes that the standards results are being compiled in a timely fashion, using appropriate evaluation techniques that are commonly used in the industry.

11.4.2 Magnetic susceptibility Instrument Calibration

Checks were routinely carried out on standard reference material to confirm that the MS instrument was reporting consistently. The reference materials comprised thirteen different samples taken from reject material. FWM personnel plot the results from these calibration tests in chronological order to check that the instrument readings do not drift over time. It has been noted that for some of the reference samples, there are significant variations over time. Figure 11-2 shows the calibration measurements plotted for the period October 1, 2009 to June 10, 2010. The results for most of the calibration standards are observed to be reasonably consistent. The readings for the 70,000 standard are notably variable and appear more recently to be trending markedly downwards. The 40,000 standard is also trending downwards with time. RPA notes that most of the measurements in the database are in the range of 2,000 to 35,000 and that the calibration measurements for this range appear to be stable. In RPA's opinion, FWM's approach to calibration of the MS instrument is reasonably rigorous and indicates that the MS data is valid. However, the most recent calibration measurements suggest that a conservative bias may be developing, particularly for the higher ranges. This suggests that the instrument may be due for adjustment or even replacement.

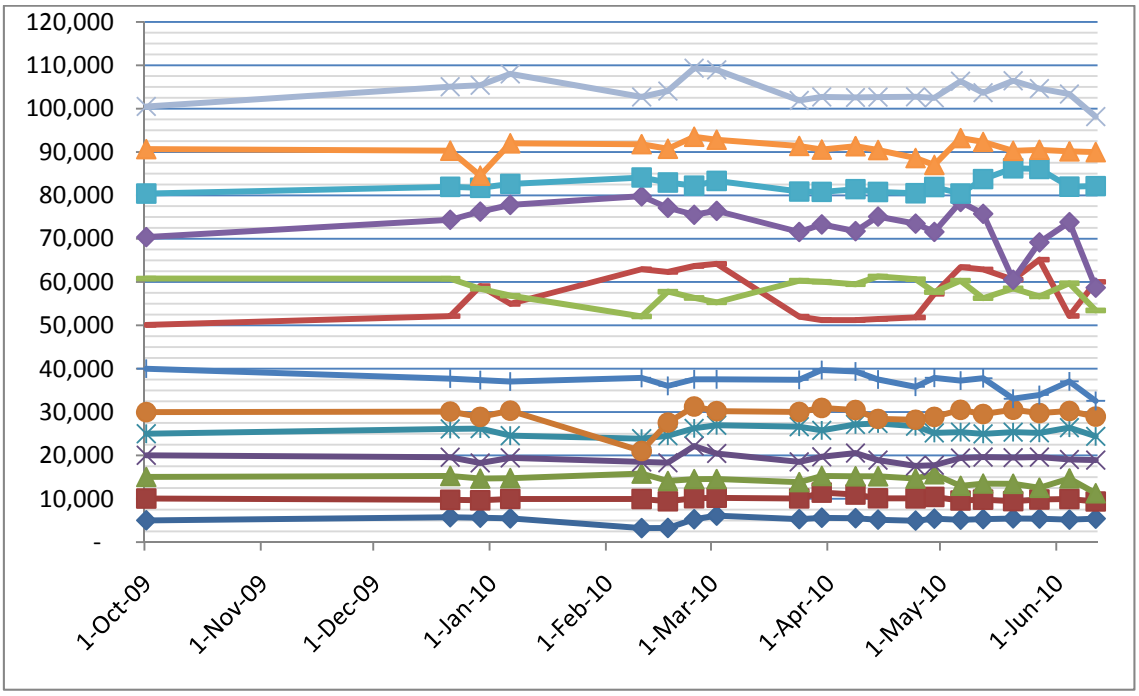


Figure 11-2: Magnetic Susceptibility Calibration Measurements

11.4.3 Blanks

Blanks, consisting of common Portland Cement, are inserted every 50th sample and analyzed for copper as well as for gold if the copper is greater than 0.1% Cu. More recently, analyses for iron have been included. For the most part, the blanks results are within a reasonable tolerance, although some of the copper results suggest that either there is some contamination or the blank material contains 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. This appears to have been addressed in more recent programs.

11.4.4 Duplicates

Duplicates are taken every 25th sample. Core 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 such that the duplicate now comprises a split off of the primary sample.

2004 To 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.

2008 To 2009

RPA conducted *t*-test analyses on the duplicates for copper and gold 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. Reruns of the duplicate analyses did not show the same bias.

2010

Duplicates were plotted on scatter diagrams configured to show each duplicate pair relative to an error limit. An example of this type of diagram is shown in Figure 11-3. The pairs are sorted such that the maximum and minimum for each pair will plot on the Y and X axes, respectively. The pair difference appears as the vertical distance above the X=Y line. The threshold above which the pair difference is considered to be over-limit is defined by the hyperbolic curve depicted as a red dashed line in Figure 11-3. This line is derived from the following equation:

$$Y = ((DL^2 * X^2) + C^2)^{1/2}$$

Where:

DL = practical detection limit

C = a constant dependent on the type of duplicate (e.g., 1.35 for field duplicates)

The practical detection limit is typically determined experimentally from a suite of paired pulp duplicates. The constant, C, is an empirical value which is highest for field duplicates, lower for reject duplicates, and lowest for pulp duplicates. For the case depicted in Figure 11-3, the value of C was 1.35, which corresponds to an approximate 35% relative difference between pairs. Varying this constant has the effect of changing the angle between the error limit and the X=Y line.

Points which plot above the hyperbolic line are considered to be failures. The number of acceptable failures varies according to the type of duplicate. For field duplicates, this limit is typically 10% of pairs. RPA reviewed the hyperbolic precision diagrams prepared by FWM and notes that for all duplicate data sets, the error rate was well within the 10% limit.

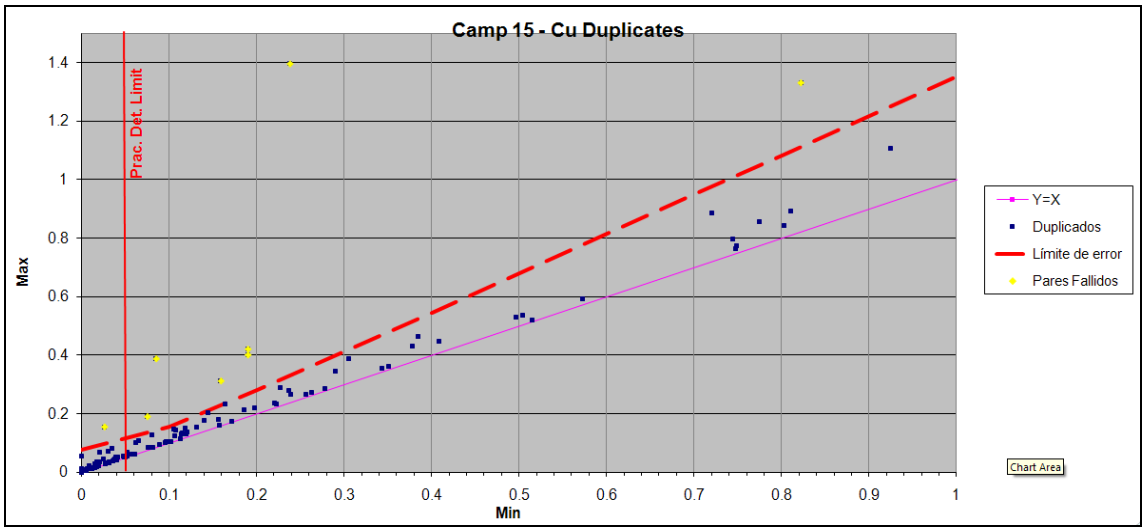


Figure 11-3: Example Duplicate Plot – Copper

In RPA's opinion, the duplicate QA/QC data are being compiled and evaluated in a timely and reasonable manner, consistent with industry best-practice.

11.4.5 Twinning of Drillholes

FWM have twinned several holes 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 MS data in areas for which sample material was no longer available for testing. Since most of the early holes were RC, this provided an opportunity to compare RC results with diamond coring. A list of twinned holes is provided in Table 11.3.

Table 11.3: Summary of Twinned Holes (Far West Mining Ltd. – Santo Domingo Property)

Section	Original	Twin
20000NE	4a3-05-069	4a3-06-079DD
	4a3-06-077DD	4a3-10-077DD-B
20100NE	4a3-06-076DD-B	4a3-10-039-B
20400NE	4a3-06-088	4a3-10-088-B
20500NE	4a3-07-262	4a3-07-262B
	4a3-05-060	4a3-06-088DD
20700NE	4a3-06-218	4a3-10-218-B
20800NE	4a3-06-093	4a3-10-093-B
20850NE	4a3-06-092	4a3-10-092-B
20900NE	4a3-06-099	4a3-10-099-B
	4a3-06-213	4a3-10-213-B
	4a3-06-094	4a3-10-094-B
21000NE	4a3-06-133	4a3-10-133-B
	4a3-06-139	4a3-10-139-B
	4a3-06-136	4a3-10-136-B
	4a3-06-215	4a3-10-215-B

RPA matched intervals of four-metre composites for each of the pairs and plotted the grades for Au, Cu, and Fe to compare the results. An example of one of these plots is shown below in Figure 11-4.

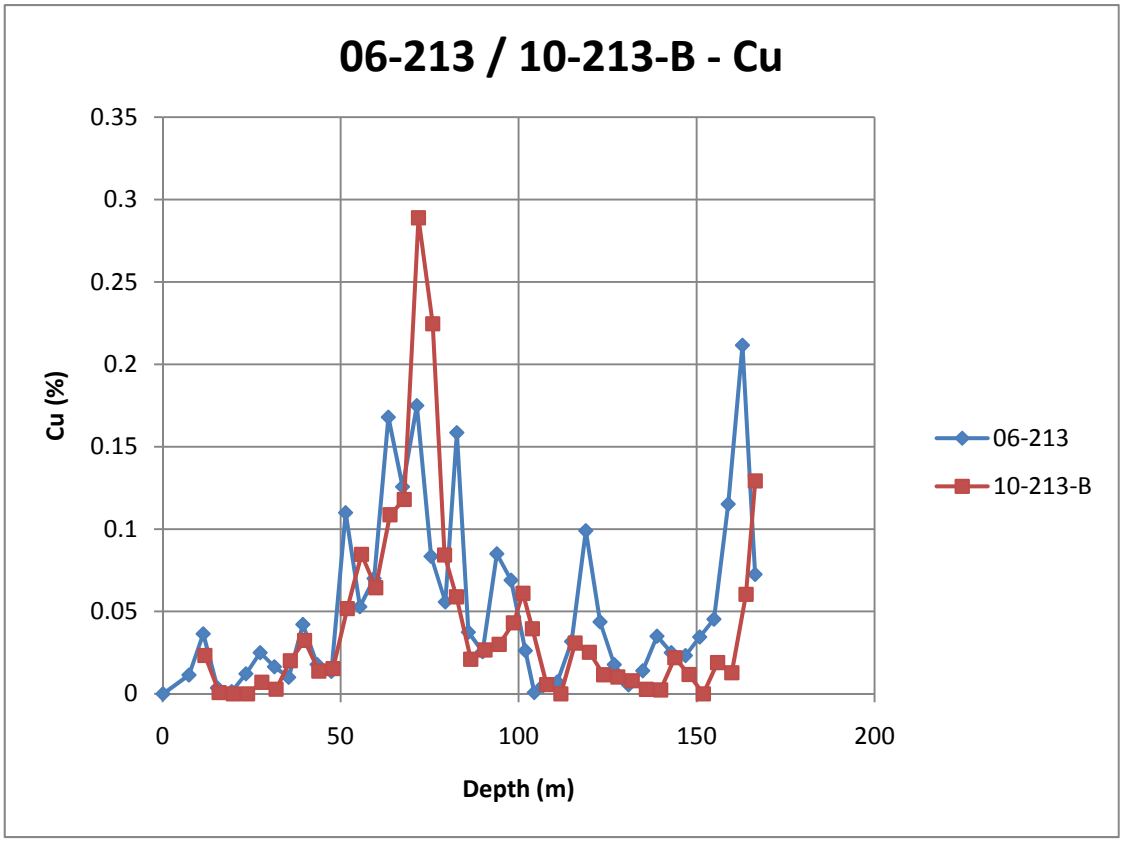


Figure 11-4: Example Plot of Twinned Drillhole Results

Figure 11-4 shows the assay results for copper in holes 4a3-06-213 and 4a3-10-213-B plotted by downhole depth. 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 four-metre resolution, however, it was generally noted that high and low grade zones matched, and that the grades tended to cluster in the same ranges. Only one pair of twinned holes, 4a3-06-099/4a3-10-099-B, displayed significant differences that could not be attributed to things such as hole deviation. In RPA's opinion, the twinning has provided a reasonably consistent verification of the earlier drill results particularly considering the differences in assay protocols and possible survey errors.

12 DATA VERIFICATION

The following section is an extract from the RPA mineral resource estimate dated 2010[14].

12.1 Database

The descriptions in this section are copied from Lacroix (2009). RPA reviewed the data capture and storage protocols during the 2010 site visit and noted that they are largely unchanged. However, FWM personnel were in the process of installing and configuring a new off-the-shelf drill database system that is expected to be in operation shortly.

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

A separate assay ledger is also kept for each hole. Initially, sample intervals and numbers are entered manually into the ledger and then transcribed into an MS Excel spreadsheet. The initial ledgers or logs are completed by the samplers at the drill for RC cuttings and at the core-logging facility for core. Inserted blanks, standards, and duplicates are also recorded in this ledger. Assay results, when available from the laboratory, are cut and pasted into the digital ledger from an MS Excel file provided by the lab. Once complete, data from the ledger are imported to a master MS Access database containing all the Candelaria Project drill assays.

One person is responsible for management of the database, posting of final results, and controlling user access.

Data provided to RPA were based on exports from the MS Access database. In 2006, RPA independently verified a portion of the 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 lab and FWM’s master database. In total, assay results for the mineralized portions of five (046, 050, 063, 066, 069) of the 34 holes that intersected the mineralized portion of the manto were verified. In 2007, MS Excel spreadsheets from the laboratory were consolidated into a database for comparison with the assays provided by FWM. 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 FWM. Of these, 4,677 records could be

[14] Far West Mining Ltd. Technical Report on the Santo Domingo Property, Region III, Atacama Province Chile, NI 43-101 Report, Author: David W. Rennie, P.Eng. August 26, 2010 Scott Wilson Roscoe Postle Associates Inc.

matched via the sample ID. There were no errors or discrepancies found in either the Cu or Au entries. In 2009, RPA compared Cu, Au, and Fe values in the database provided with individual certificates for 11 of the 52 holes drilled subsequent to the 2007 Mineral Resource estimate. For this report, RPA compared the certificates to the database entries for Au, Cu, and Fe for 5,271 samples. No errors, inconsistencies, or discrepancies were noted.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Summary

The objective of the pre-feasibility metallurgy study (PFS) was to determine the metallurgical response of the Santo Domingo copper and iron mineralization. The program was designed to develop parameters for process design criteria for grinding, and copper and magnetite recovery circuits of the plant, and to confirm the suitability of the use of seawater in the processing plant.

The PFS flotation program at SGS Lakefield was performed on the following composites and samples:

- 8-year composite[15]
- hematite composite
- magnetite composite
- oxide composite
- variability samples.

The samples were selected to represent the spatial distribution, ore grade, and tonnage of the mineralization in the ore zones.

Flotation optimization studies were performed on the 8-year composite. The optimized conditions were applied to the hematite and magnetite composites and the variability samples. In addition, copper flotation tailings were produced for magnetic iron recovery studies by SGA in Germany. Bulk copper rougher and cleaner concentrates were also produced for testwork by process equipment suppliers to determine regrind power and concentrate filter size.

As locked cycle tests were not conducted on the variability samples scale-up to full-scale operation has been based on the results of the locked cycle tests on the 8-year pit, hematite and magnetite composites. The average copper results of the batch and locked cycle tests are shown in the Table 13.1. Average of Batch and Locked Cycle Test Results – Cu

Table 13.1: Average of Batch and Locked Cycle Test Results – Cu

		Rghr. Rec.		Clnr. Rec	Global. Rec.
	Head %Cu	Wt %	Cu %	Cu %	Cu %
Var. Batch Avg.	0.36	9.4	93.1	89.1	83.0
Comp Batch Avg.	0.34	11.6	93.1	95.0	88.4
Comp LCT Avg.	0.34	12.2	93.4	96.5	90.7

[15] The 8-year composite is representative of the sulfide portion of a preliminary starter pit that contains the first eight years of production at a 70,000/60,000 t/day mining rate (70,000 t/day in the first six years of production, 60,000 t/day thereafter).

In order to project continuous full-scale operation recoveries, a series of correction factors were developed. The final algorithm projects a full-scale copper recovery of 90.2% for the average head grade of 0.34% Cu. This is close to the average locked cycle result of 90.7%, from the three composites.

Elemental analysis of the flotation concentrates indicate they are of high quality, containing attractive gold and silver concentrations, and have very low penalty element levels. The concentrate should therefore be easy to market.

A small number of tests were conducted to assess the flotation response of the oxide composite. Sequential sulphide-oxide flotation with sulphidization agents was explored. In the best of six scoping tests 25% of the copper was recovered in a rougher concentrate grading 3.2% Cu. Mineralogical examination of the oxide composite indicated that the bulk of the copper is present at non-floatable minerals.

The iron ore recovery testwork at SGA was conducted on copper rougher flotation tailings samples produced at SGS Lakefield. Conditions were optimized for the 8-year composite and applied to the hematite and magnetite composites. A series of tests were performed on variability samples to determine the correlation between Davis Tube test results and LIMS cleaner tests and to determine the correlation between Satmagan/magnetic susceptibility head grade and Davis Tube test recovery.

Table 13.2 shows a summary of each composite recovery and grade.

Table 13.2: LIMS Cleaner Concentrate

Composite	Mag Fd. Grade %	Wt. %	Fe Grade %	Fe Rec. %	SiO_2 Grade %	Na_2O Grade %	K_2O Grade %	Mag. Grade %	Mag. Rec. %
8-Year Comp.	15.2	16.8	66.1	38.2	4.1	0.145	0.105	81.6	90.1
Hematite Comp.	9.4	9.8	64.1	24.9	5.7	0.33	0.153	71.2	74.3
Magnetite Comp.	27.5	29.8	66.5	66.8	4.4	0.225	0.09	84.8	92.1

High grade LIMS concentrates at high magnetite recoveries were produced from the 8-year pit and magnetite composites. Further work is required to improve the concentrate grade from the lower grade hematite composite.

The magnetite recovery in the mine block model was calculated from the Magnetic Susceptibility (MS) values assigned to each block using an MS head grade-to-weight recovery algorithm. The algorithm was derived from MS measurements and Davis Tube tests on a large suite of whole ore samples. The LIMS mass recovery calculated with the algorithm was in good agreement with the LIMS test results.

13.2 Mineralogical Studies

Mineralogical testwork undertaken on samples from the Santo Domingo Deposit was undertaken by SGS and is reported in “Project 12066-001 An Investigation into Mineralogical

Analyses Of Samples From The Santo Domingo Project", October 22, 2010. The report is attached in Appendix 3-1.

13.2.1 Mineralogy of Ore Zones

The Santo Domingo IOCG project consists of the following zones of mineralization, as identified by the geologists, and based on the predominant iron oxide mineral:

- Magnetite Core
- Hematite Rim, surrounding Magnetite Core
- Iris Mag, magnetite–rich, low-copper zone north east of Hematite Rim
- Iris, a hematite-rich northern extension, higher in copper
- Iris Norte, a magnetite-rich zone north of Iris.

The Magnetite Core and the Hematite Rim zones contain the highest grade copper mineralization.

QEMSCAN® Mineralogy Studies

From the copper mineralogy summarized in Table 13.3, it can be seen that chalcopyrite is the only copper mineral in the Magnetite Core, while the Hematite Rim and Iris deposits contain a significant amount of secondary copper minerals, which will result in the production of higher-grade concentrates from these zones.

Table 13.3: Percent Cu and Fe Mineral Composition of Ore Type Composites

Mineral Phase	Iris	Iris Mag	Iris Norte	Hem Rim	Mag Core
Chalcopyrite	52.5	84.7	67.9	85.9	98.8
Bornite	8.0	2.5	11.0	2.7	1.0
Chalcocite-Digenite	36.0	11.4	14.3	10.5	0.1
Covellite	3.4	1.2	0.5	0.8	0.0
Enargite-Tennantite	0.0	0.1	0.0	0.0	0.0
Others	0.1	0.1	6.3	0.0	0.2
Total	100.0	100.0	100.0	100.0	100.0
Head % Cu	0.32	0.11	0.19	0.49	0.35
Fe Oxides	22.3	39.9	28.5	28.9	35.3
Head % Fe	22.6	30.7	24.9	26.5	30.8

Table 13.4 shows the summary of liberation of the copper and iron minerals in the composites. These data indicate that the degree of liberation varies between the different ore zones. The copper minerals are highly liberated in the Hematite Rim composite, while the Magnetite Core has the highest level of iron oxide liberation.

Table 13.4: Liberation of Copper Minerals in the Ore Type Characterization Composites

Description	Mineral % Area	Composites				
		Iris	Iris Mag	Iris Norte	Hem Rim	Mag Core
Free Cu Sulphide	>95	57.1	67.2	58.4	81.9	55.2
Lib Cu Sulphide	95-80	8.2	11.3	6.1	5.6	15.2
Cu Sulphide. Midds	80-50	15.7	7.3	18.2	6.6	21.3
Cu Sulphide Sub-Midds	50-20	4.0	4.1	7.9	2.7	1.2
Locked Cu Sulphide	<20	15.0	10.2	9.4	3.2	7.2
Total		100.0	100.0	100.0	100.0	100.0
Total Lib Cu Minerals	>80	65.3	78.4	64.5	87.5	70.4
Total Lib Fe-ox Minerals	>80	68.0	62.0	76.8	72.5	78.6
Particle Size % -26 µm		46.4	41.0	40.8	43.2	39.1

QEMSCAN® analysis of three screen size fractions of the 8-year composite shows the copper mineral composition and distribution of these fractions. Chalcopyrite contains 86% of the copper, while secondary copper minerals account for 10% of the copper. The sample was ground to a P80 of 200 µm.

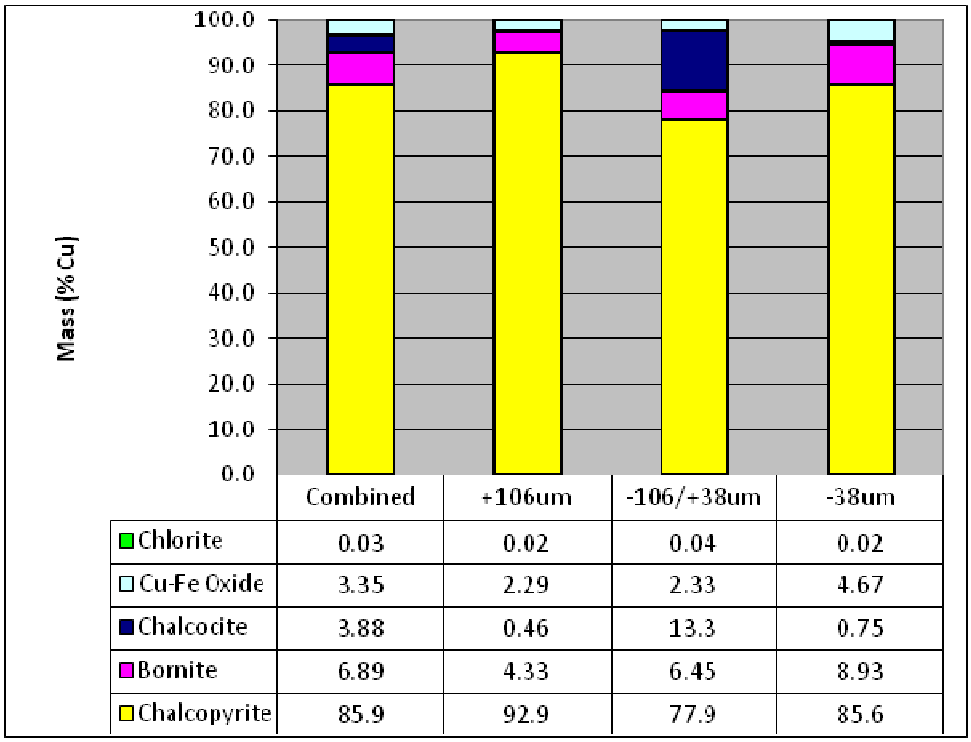


Figure 13-1: Elemental Deportment (Mass % Cu) 8-Year Composite

Table 13.5 shows the liberation of copper minerals by size fraction.

Table 13.5: 8-Year Composite Liberation of Copper Sulphides by Size Fraction

	Combined	+106 µm	-106/+38 µm	-38 µm
Size Fraction Weight %	100	49	20	31
Mineral Associations				
Free Cu-Sulphides	68.95	45.99	69.78	85.18
Lib Cu-Sulphides	10.29	10.49	14.12	8.04
Cu-Sulphides: Pyrite	0.69	0.21	2.40	0.11
Cu-Sulphides: Fe-Oxides	2.99	6.24	2.24	1.04
Cu-Sulphides: Carb	0.08	0.00	0.03	0.17
Cu-Sulphides: Fe-Ox:Carb	1.05	2.56	0.87	0.05
Cu-Sulphides: Hard Sil	0.83	1.53	0.82	0.33
Cu-Sulphides: Soft Sil	0.79	0.00	1.10	1.20
Cu-Sulphides: All Sil	2.09	5.46	0.49	0.52
Complex	12.23	27.52	8.15	3.34
Total	**100.0**	**100.0**	**100.0**	**100.0**
Liberated Cu Sulphides	79.2	56.5	83.9	93.2

The data suggest that P_{80} primary grind level of 200 µm and a regrind level of about 40 µm are required to achieve satisfactory copper recovery and concentrate grade.

13.3 Flotation and Magnetic Separation Testwork Program

13.3.1 Previous Testwork

Several rounds of metallurgical testing have been undertaken, at various laboratories, prior to the current PFS testwork phase. These diagnostic phases were conducted using various composites and test Composites from the respective zones. The composites were prepared to reflect average head grades, and subjected to QEMSCAN® mineralogy studies, kinetic and standard flotation tests, and Davis Tube tests to determine the magnetite recovery response. The results of these investigations are presented in the following reports, which can be found in Appendix-3-2:

- SGS Santiago Project OL-4029 (3584) “Flotation and grinding testwork on samples from Santo Domingo Sur”, August 2008
- SGS Lakefield Project 11594-001 “An Investigation into The Recovery Of Iron From Two Flotation Tailings Composites From The Santo Domingo Sur Deposit” 17 March 2009
- SGA report “Remarks about the Recovery of Magnetite Originating from Santo Domingo Sur Oxide Iron, Copper, Gold Deposit (IOCG) Chile”, 22 April 2009
- SGS
- A Minpro Project No. 2010-003 “Flotation Testwork with Five (5) Samples from the Santo Domingo project”, 12 July 2010
- SGS Lakefield Project 12066-001 “An Investigation into Copper And Gold Flotation Testwork On Samples From The Santo Domingo IOCG Deposit“ 28 July 2010
- SGS Lakefield Project 12066-002 “An Investigation into Copper And Gold Flotation Testwork On Samples From The Santo Domingo IOCG Deposit“ 15 October 2010

- SGA Report No. 7118 “Remarks about Testing 5 Crude Ore Samples from the Santo Domingo Sur Deposit, Chile” June 2010.

SGS Santiago Project OL-4029 (3584)

This report summarises the results obtained in the experimental testwork program carried out on two master composites (MC-A and MC-B). This testwork consisted of copper rougher kinetic, copper and pyrite rougher kinetic and copper cleaner and pyrite rougher flotation tests.

Several magnetic separation tests were conducted on pyrite rougher floatation tailings.

The rougher kinetic tests indicated that good copper recoveries could be achieved from both samples, with recoveries typically ranging from 90-95% with rougher concentrate grades of around 10-15%.

Several tests included pyrite rougher flotation on the copper rougher tailings to maximise recovery of sulphur from the flotation rougher tailings. These samples were submitted to SGS Lakefield for scoping magnetic separation testwork (SGS Lakefield Project 11594-001).

Following the rougher tests a suite of open circuit rougher cleaner tests were performed Table 13.6 summarises the copper cleaner performance from these tests.

Table 13.6: Copper Cleaner Flotation Performance

Sample	Test	Cleaner Concentrate	
		Concentrate Grade (%Cu)	Cu Recovery (%)
MC-A	17	25.7	92.4
	18	19.8	92.8
	19	31.0	88.1
	20	30.3	87.1
	22	33.3	79.0
	24	29.8	80.7
	25	24.5	90.4
	27	33.3	82.2
Average			
MC-B	21	28.0	89.8
	23	27.3	87.4
	26	22.9	92.2
Average			

The results show that typically a good copper concentrate grade can be achieved with an acceptable recovery of about 90%.

SGS Lakefield Project 11594-001

The study was conducted to evaluate response of the individual composites to magnetic separation by Davis Tube and also reverse silica flotation.

Magnetic separation testwork explored primary grind P80 sizes between 131 and 26 μm on the MC-A composite, and 146 to 46 μm on the MC-B composite. At a grind P80 of ~40 μm, Davis Tube magnetic concentrates were produced that recovered 91-96% of the magnetite. The iron content of these concentrates was approximately 65% Fe, with SiO2 grades ranging from 3.9- 4.8% SiO2. The SiO2 grade in the magnetic concentrate decreased to 3.1% SiO2 at a P80 of 26 μm.

Exploratory flotation testwork on the +38 μm fraction of the composites was unsuccessful at separation of iron and siliceous minerals. A large percentage of both reported to the slimes (-38 μm) fraction. It was generally found that iron recovery tracked weight recovery in all products.

It is estimated that magnetite and hematite are roughly equally present (~45% versus 55%, respectively), based on the examination with an optical microscope. Mineralogical analysis suggests that with finer grinding, liberation of silicates from Fe-oxides is possible. The overall liberation of the silicates and Fe-oxides is ~73% and ~72% of the calculated head, respectively, though only 52% of the Fe-oxides minerals are liberated in the +38 μm fraction.

SGA report April 2009

A sample of 120 kg of copper flotation tailings was provided to SGA for testing iron recovery into a marketable magnetite concentrate. Low Intensity Magnetic Separation (LIMS) testing was conducted on the samples which will target only iron present as magnetite and partly martite,

The sample tested in these tests was collected from copper rougher tailings. This stream represents about 90% of the mass of mill feed, with the remaining 10% in final copper concentrate and pyrite concentrate.

In preconcentrate LIMS stage conducted on the sulphide flotation tailings, a rougher concentrate with near to 60% Fe was achieved. The iron recovery was 43.7% and mass recovery was 22.4% with respect to the flotation feed stream. The Blaine value of this concentrate was measured at 853, which is in the normal range for a rougher concentrate.

Further liberation of the mineral phases was undertaken on the preconcentrate. Samples were reground in a laboratory rod mill for 20, 30 and 40 minutes. The ground samples were then subjected to 3-stage cleaner magnetic separation. Results from these tests are summarised below in

Table 13.7.

Table 13.7: Copper Cleaner Flotation Performance

Regrind Time (mins)	Mass Recovery (%)*	%Fe	%SiO_2	Blaine
20	16.2	67.1	3.22	1428
30	16.0	67.5	2.84	1667
40	15.8	68.3	2.59	1922

* with respect to flotation feed

The figures show that the SiO_2 content of the magnetite concentrate is approximately 3.0% for a Blaine number near 1600. For higher Blaine numbers near to 2000, the SiO_2 content will be closer to 2.5%.

These concentrates are very low in phosphorous content as well as low in sulphur and other non-ferrous metals.

13.3.2 Ore Type Characterization Studies

In preparation for the pre-feasibility study metallurgy program it was decided to study mineralogy and the metallurgical response of the main five ore types of the deposit as identified by the geologists for the purpose of determining the number of ore types that should studied during the pre-feasibility study.

For the purpose of the investigations five ore type composites were prepared from samples selected across the mineralization to represent the following ore types:

- Hematite Rim
- Magnetite core
- Iris Mag
- Iris Norte
- Iris

The program consisted of the following activities:

- a QemScan mineralogy study conducted by SGS Santiago (Q196-July 2010)
- a flotation kinetic response study conducted by AMinpro
- a flotation response study conducted at SGS Lakefield
- a LIMS response study conducted by SGA in Germany

SGS Santiago Project Q196 –July 2010

The copper mineralogy indicated that chalcopyrite is the only copper mineral in the Magnetite Core, while the Hematite Rim and Iris deposits contain a significant amount of secondary copper minerals, which will result in the production of higher-grade concentrates from these zones. The mineral phase liberation level varies between ore zones. The copper minerals are highly liberated in the Hematite Rim composite, while the Magnetite Core has the highest level of iron oxide liberation

AMinpro Project No. 2010-003

Five ore composites were tested for their chemical and metallurgical characteristics to determine whether any ot the ore types could be combined into single types for future testwork.The tests included chemical and mineralogical analysis (QEMSCAN) and rougher kinetic flotation tests. All rougher flotation tests were conducted in seawater.

The five ores were seen to have similar flotation characteristics, potentially allowing them to be grouped into one single ore type for predicting recovery and for mine planning. The five ores were compared on the basis of the variances of the main variables, which showed that two of the ores, Iris Mag and Mag Core, were consistently more variable than any of the other ores. This would suggest that the ores should be considered as three main oretypes as follows:

- Iris-Iris Norte-Hematite Rim
- Iris Mag
- Mag Core

The expected head grade and rougher recovery from these composites are summarised below in Table 13.8.

Table 13.8: Expected Grade and Recovery with Combined Samples

Sample	Cu Grade (%)	Au Grade (g/t)	R_{MAX} $CuFeS_2$	R_{MAX} Au
Iris-Iris Norte-Hematite Rim	0.34	0.06	89.1%	67.7%
Iris Mag	0.11	0.02	89.4%	30.1%
Mag Core	0.34	0.05	95.7%	88.3%

On the basis of these results, the recovery that should be obtained at industrial scale at 25 minutes of rougher residence time was estimated using the AMINPRO Flotation Model. This model is an Excel based, dynamic model that used the kinetic data obtained at laboratory scale, to predict plant scale recoveries at the same grind size. Estimated plant recoveries are listed in Table 13.9.

Table 13.9: Plant recovery estimated for 25 minutes, for different species

Sample	R Au	R $CuFeS_2$	R Cu_{OX}	R FeS_2	R Gangue
Iris Mag	27.1	75.2	72.4	42.3	0.8
Iris Norte	59.9	84.0	49.5	18.3	0.2
Mag Core	72.3	88.2	62.6	74.7	6.1
Hematite Rim	83.3	93.8	79.4	66.0	0.3
Iris	42.3	76.1	71.2	0.0	3.0

SGS Lakefield Project 12066-002 October 15 2010

The copper flotation response study on the five ore types is reported in the above SGS report.

The results of kinetic rougher test on the five composites conducted at SGS Lakefield (shown in Figure 13-2) illustrate that the copper recoveries were consistent with the head grades despite differences in mineralogy between the composites. The Iris Norte response was somewhat below average, perhaps due to the presence of the "other" copper minerals. However, the zone contains only 10% of the copper in the resource, and therefore its effect will not be significant.

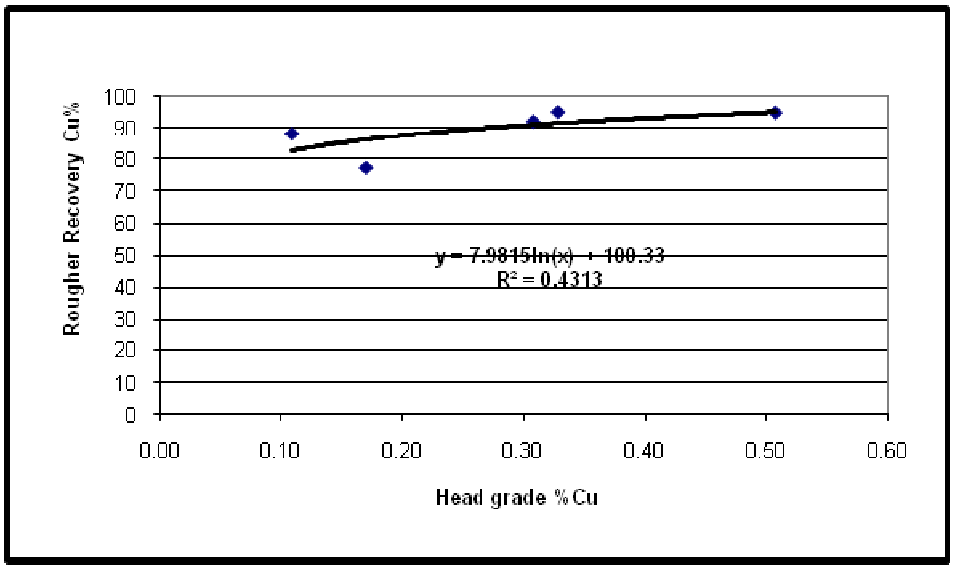


Figure 13-2: Ore Type Characterization Flotation Tests

SGA Report No. 7118 June 2010

5 samples of ore, consisting of composite samples from the 5 main ore types, were tested for magnetite recovery by grinding to less than 63 microns. Assay analysis showed that the iron content ranged from 23.8% to 32.55%. Magnetite, as measured by Satmagan, was in the range of 10.4% - 39.6%. The Hem Rim sample had the lowest proportion of iron present as magnetite at 27.3%, whereas Iris Mag, at 87.6%, had the highest.

Davis Tube testing produced magnetite concentrates that ranged from 64.0% to 67.1% Fe, with between 3.58% and 6.12% SiO_2. The Sulphur content for all of the concentrates was generally low, typically below 0.1%.

Given that these tests were conducted on whole ore, rather than rougher flotation tailings, (after the majority of the sulphur has been removed in the rougher flotation concentrate), the sulphur content of the magnetite concentrate produced from treating rougher flotation tailings has the scope to further lower the sulphur level, These figures indicate that there is very little or no pyrrhotite present in the tested ores and that flotation for removal of sulphur in the magnetite concentrate is unlikely to be required.

From the results of the ore type characterization studies it was concluded that the mineralization was sufficiently similar to avoid the need for differentiation however the Hematite Rim contains the highest grade copper mineralization and the lowest magnetite content compared to the predominantly magnetite mineralization with lower copper content in the other parts of the deposit it was decided to test two ore types, hematite and magnetite for the pre-feasibility study.

SGS Lakefield Project 12066-001 July 2010

Please note that this work was done before the Ore type Characterization Studies.

Testwork was conducted on three different composite samples from Santo Domingo Sur. The composites were designed to represent the mineralization from the entire orebody. The three composites varied due to the inclusion of RC drill cuttings (SDS-OA1) or exclusion of material from depleted sub-composites (SDS-OA3).

Testwork on SDS-OA1 used Lakefield water, while that on SDS-OA2 and SDS-OA3 examined three water types: Lakefield water, a client supplied saline water, and synthetic sea water. The client-supplied water was considered at the time to be a potential source of process water.

A series of initial tests on SDS-OA1 and SDS-OA2 using Lakefield water were conducted to optimise the flotation conditions. To predict the metallurgical performance with recycle streams, one locked cycle test (LCT-1) was completed on SDS-OA2. Metallurgical results exceeded those of the comparable batch test with production of a 2nd cleaner concentrate grading 30.7% Cu at a Cu recovery of 87.5%. It was projected that 70% of the Au in the feed would be recovered into the 2nd cleaner concentrate.

A series of seven rougher kinetics and sixteen cleaner kinetics tests were conducted on SDS-OA2 and SDS-OA3 evaluating the impact of synthetic sea water. The main parameters varied included collector type, pH level, depressant types, and minor flowsheet modifications. Ultimate Cu recoveries fell within the range of 88% to 97%. The rougher Cu kinetics responses were generally in line with that using the El Salvador water. However, selectivity was inferior to that using both Lakefield and El Salvador waters. Cleaner kinetics results were also inferior to the results obtained with Lakefield water.

Further flowsheet optimization on SDS-OA3 showed promising results compared to the previous fresh water tests. A concentrate grading 25% Cu was produced at a Cu recovery of 88% compared to a concentrate grade of 26% Cu at 89% Cu recovery in the fresh water testing. Metallurgical projections exceeded the results of the SDS-OA2 locked cycle test using fresh water (LCT-1), possibly due in part to a finer primary grind in LCT-2. (P_{80}~100µm in LCT- 2 vs. P_{80}~120µm in LCT-1) Gold recovery to the 2nd cleaner concentrate in LCT-2 was 69%, which was similar to LCT-1. Results from LCT-2 are summarised below in Table 13.10.

Table 13.10: Locked Cycle Test Metallurgical Projects – Synthetic Sea Water

Product	Wt	Assays (%, g/t)				Distribution (%)			
	%	Cu	Fe	S	Au	Cu	Fe	S	Au
2nd Cleaner Conc	1.5	32.6	30.6	33.2	3.62	88.5	1.4	19.3	68.6
1st Cleaner Scav Tail	5.3	0.71	35.3	20.3	0.12	6.8	5.7	41.2	8.0
Rougher Tail	93.2	0.028	32.8	1.10	0.02	4.8	92.9	39.4	23.4
Head	100.0	0.56	32.9	2.59	0.08	100.0	100.0	100.0	100.0

SGS Lakefield Project 12066-002 October 2010

Five flotation tests were conducted on sample SDS-OA3 in order to determine the effect of primary grind fineness on copper recovery. Copper recovery decreased from 94% at a P80 of 104 µm to 91% at a P80 of 153 µm. Gold recovery showed a slight upward trend in recovery as the primary grind coarsened, increasing from 76% at a P80 of 104 µm to 79% at a P80 of 153 µm. This may be attributable to an increase in pyrite recovery to the rougher concentrate at the coarser grind.

Rougher flotation variability tests were conducted on thirteen samples. Nine of the thirteen samples exhibited a copper over 90%and gold recovery over 70%. While eleven of the samples exhibited a similar response of head grade to copper recovery, flotation of samples Var-156A and Var-158A resulted in lower than expected recovery. Copper oxide specific assaying indicated that up to 25% of the loss may be attributable to copper oxide mineralization in the tailings.

Gold recovery showed a consistent relationship with gold head grade for all samples.

Conditions for all tests in this report were based on the optimised conditions developed in the previous test work as detailed in project number 12066-001, this included NaMBS addition to

the roughers, A3926 and A-3418A collector and promoter additions and flotation at natural pH. A synthetic sea water mixture was used in all flotation tests.

13.3.3 Prefeasibility Metallurgical Study

Sample Selection

The PFS flotation program at SGS Lakefield was performed on the following composites and samples (refer to SGS report 12066-003 in Appendix 3-3):

- 8-year composite
- hematite composite
- magnetite composite
- oxide composite
- variability samples.

The selection of the samples for the four composites and the variability testwork is described in the report titled “Metallurgical Sample Selection Santo Domingo PFS ,” which can be found in Appendix 3-3. A number of the variability samples did not fit the copper grade recovery curve; these samples were identified as coming from a fault zone, and not representative of the typical mineralogy. The samples identified as having come from the fault zone are described in the memo titled “Variability Samples from Fault Zone” in Appendix 3-3. Figure 13-3 (illustrates the spatial distribution of the samples selected for the metallurgical test work.

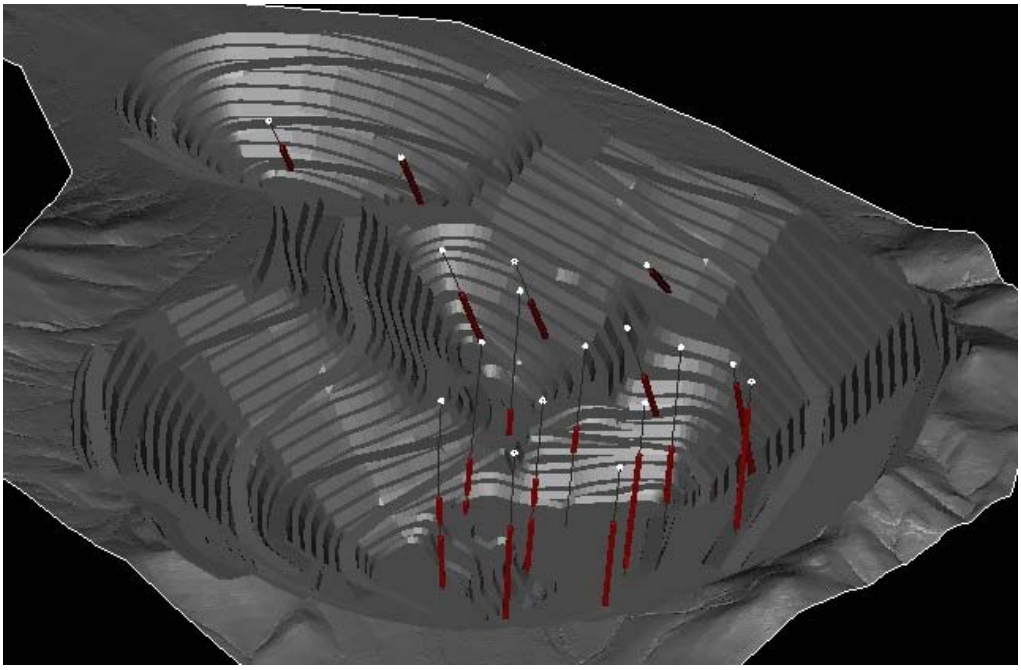


Figure 13-3: Three dimensional illustration of intervals that make up the metallurgical composites superimposed on a preliminary pit shell.

Metallurgy Composites

The samples for the metallurgy composites and variability tests were carefully selected to spatially represent the distribution, ore grade, and tonnage of the mineralization in the ore zones.

The 8-year composite was prepared from 38 intervals of 12 drill holes within the pit shell, totalling 964 m of mineralization.

The hematite composite was prepared from 46 intervals of 9 drill holes, totalling 1,420 m of mineralization.

The magnetite composite was prepared from 31 intervals of 11 drill holes, totalling 9,832 m of mineralization.

The oxide composite was prepared from 4 intervals of 3 drill holes, totalling 94 m of mineralization.

Table 13.11 shows the head assays of the composites.

Table 13.11: Composite Head Assays

Element	Comp 8 YR	Hem 1	Mag 1	Oxide
Cu %	0.45	0.38	0.24	0.67
Cu H_2SO_4 %	0.03	0.04	0.02	0.43
Au g/t	0.08	0.07	0.05	0.08
Ag g/t	0.60	0.80	0.80	1.10
Fe %	29.5	27.5	29.8	36.1
Magn Fe %	10.1	5.9	19.0	6.0

Variability Samples

A suite of 38 samples was selected from across the zones of mineralization to be representative of the copper and magnetic iron grade range. This selection was made to test the metallurgical response of these samples, in order to develop head grade recovery algorithms for the mine block model.

13.3.4 Copper Flotation Metallurgical Test Program

Flotation optimization studies were performed on the 8-year composite. The optimized conditions were applied to the hematite and magnetite composites and the variability samples. In addition, copper flotation tailings were produced for magnetic iron recovery studies by SGA in Germany. Bulk copper rougher and cleaner concentrates were also produced for testwork by process equipment suppliers to determine regrind power and concentrate filter size.

The results of the work done by SGS are presented in the report "A Laboratory Investigation into the Recovery of Copper and Gold from Santo Domingo IOCG samples prepared for Far West Mining Ltd. SGS Lakefield Project 12066-003" dated May 2011.

The scope and objectives of this program included:

- development of optimal process conditions in seawater on the 8-year composite for the design criteria
- flotation tests on the 8-year composite to compare flotation response in freshwater and seawater. All subsequent testing was conducted in seawater to match the design flotation condition
- production of bulk flotation tailings samples for the iron recovery program at SGA in Germany
- the production of rougher and cleaner concentrates for tests by equipment suppliers to determine equipment sizes for regrind mills, thickeners, and filters
- tests on variability samples of the ore types/zones to obtain metallurgical response information by ore type and to develop head grade recovery algorithms for the mine block model.

Primary Grind Size Evaluations

Rougher kinetics testing was conducted on the 8-year composite at three primary grind P_{80} targets: 105 µm, 150 µm, and 210 µm. Saltwater was used throughout. Tests were performed in duplicate. The conditions for these tests were based on those developed for seawater flotation during SGS Project 12066-001 and applied during the LCT2 test. In order to achieve above 90% rougher recovery at the 200 µm grind level, a fifth roughing stage had to be added, with 5 g/t of 3418A added for a total of 15 minutes of flotation time. The conditions are shown in Table 13.12. The flotation time versus rougher recovery response shows that the kinetics decrease as the grind gets coarser.

Table 13.12: Flotation Conditions for Grinding P_{80} of 105, 150 and 210 µm

	Reagents g/t				Time, Minutes		
Stage	**NaMBS**	**A-3926**	**3418A**	**MIBC**	**Grind**	**Cond.**	**Froth**
Primary Grind	50	10	-	-	14		
Rougher 1	-	-	-	5		1	2
Rougher 2	-	2.5	-	-		1	2
Rougher 3	-	2.5	-	-		1	4
Rougher 4	-	-	5	-		1	3
Rougher 5	-	-	5	-		1	4
Total	**50**	**15**	**10**	**5**	**14**	**5**	**15**

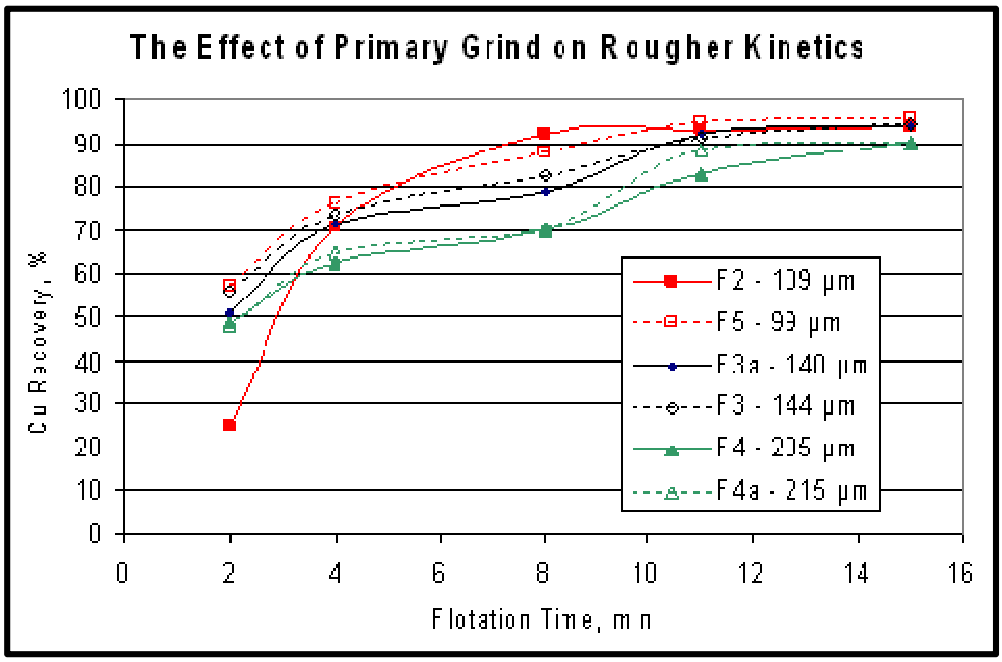


Figure 13-4: The Effect of Primary Grind on Rougher Kinetics

Regrind Size and Cleaner Conditions

The above rougher flotation conditions and primary grind size of 210 µm were used in subsequent tests to determine the optimum regrind level. A regrind size of 35 µm P_{80} gave the best results. However, the target >90% rougher mass recoveries were not consistently achieved in these tests. This was attributed to a lack of collector strength and led to the evaluation of stronger collectors and higher collector levels. With a stronger collector, such as PAX, higher copper rougher recoveries could be achieved, but upgrading the rougher concentrate proved to be difficult, as shown in Figure 13-5.

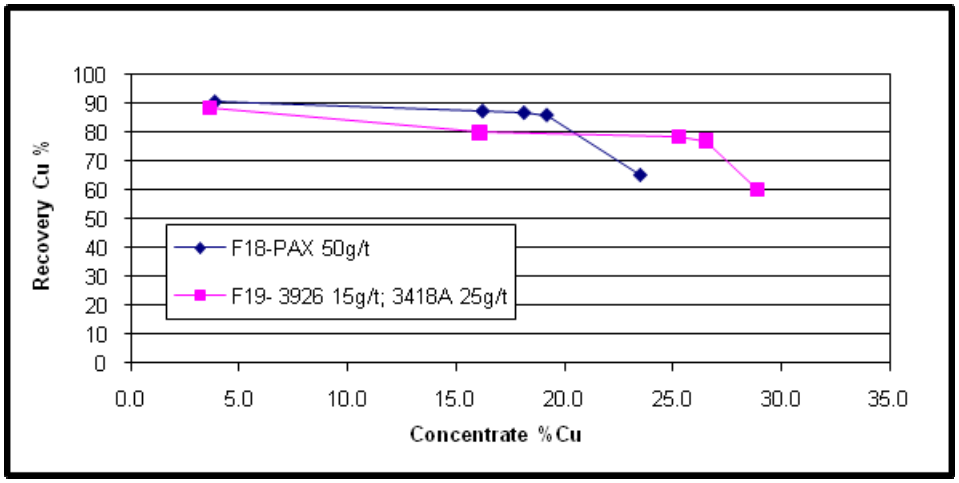


Figure 13-5: Effect of Collector on Cleaning Efficiency

It was therefore decided to revert to the more selective collector combination and evaluate the effect of increasing sodium MBS addition levels from 20 g/t to 100 g/t to the regrind mill. At constant regrind times of 15 minutes the addition of 60 g/t was found to be optimal.

The effect of a longer regrind time was tested but found not to be beneficial. The 15-minute regrind time produced a final concentrate size of 45 µm P_{80}. The 20-minute regrind reduced the particle size to 39 µm P_{80}, without an improvement in grade or recovery. Figure 13-6 demonstrates this.

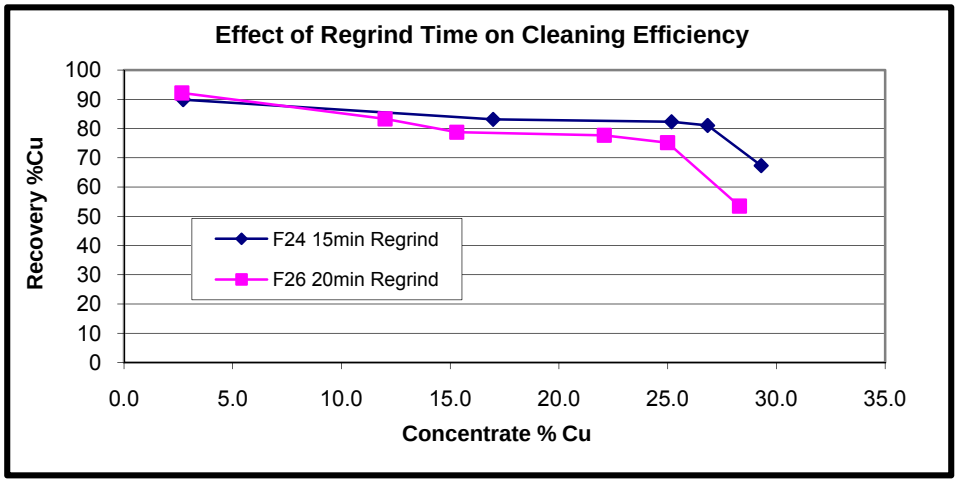


Figure 13-6: Effect of Regrind Time on Cleaning Efficiency

The evaluation of the effect of various parameter changes was complicated by significant variability in rougher recoveries (89% to 92%) for identical test conditions. The other concern was the large discrepancy between calculated and direct head grades, which ranged between 0.37% Cu calculated and 0.45% Cu assayed. The cause of these discrepancies was subsequently investigated and found to be the result of small concentrate weight losses incurred during sample preparation.

Primary Grind Re-evaluation

Rougher recoveries of over 92% had been achieved at a primary grind level of about 200 µm P_{80} by using higher amounts of the collector 3418A or by using PAX. However, the resulting rougher concentrate had proven difficult to upgrade. It was decided to examine primary grind levels and reagent conditions once more before proceeding with locked cycle tests.

As higher levels of sodium MBS had been shown to have little effect on pyrite depression in the cleaners above a dose of 60 g/t, the use of sodium cyanide was therefore evaluated for its effect on upgrading.

A series of four tests examined the effect of changing:

- the primary grind P_{80} from 200 µm to 170 µm
- reducing collector levels at the finer grind

- replacing sodium MBS with NaCN in the regrind mill
- the addition of cyanide to the regrind at rates of 50 g/t and 25 g/t.

The conditions and results are summarized in Table 13.13 and Figure 13-7. Regrind fineness varied between 23 and 30 µm.

Results indicate that cleaning efficiency was decreased when total collector addition rates were increased from 25 to 40 g/t. NaMBS was a less efficient pyrite depressant than NaCN. An increase in primary grind size from 170 µm to 200 µm resulted in a 1.5% loss in rougher recovery, despite using higher collector additions. The F39 test demonstrated maximum copper grade (32.2%) and recovery (89.4%).

Table 13.13: Optimization of Cleaner Conditions

Test No.	Grind P_{80}	Collector, g/t		Ro Float Min.	Ro Rec %Cu	Depressant		Cleaner Conc.	
		3926	3418A			Type	g/t	%Cu	Rec %
F36	210	15	25	17	92.3	NaCN	50	31.8	87.0
F37	170	15	25	17	93.3	NaCN	50	30.0	88.9
F38	170	15	10	13	93.1	MBS	60	29.2	88.0
F39	170	15	10	13	93.8	NaCN	25	32.2	89.4

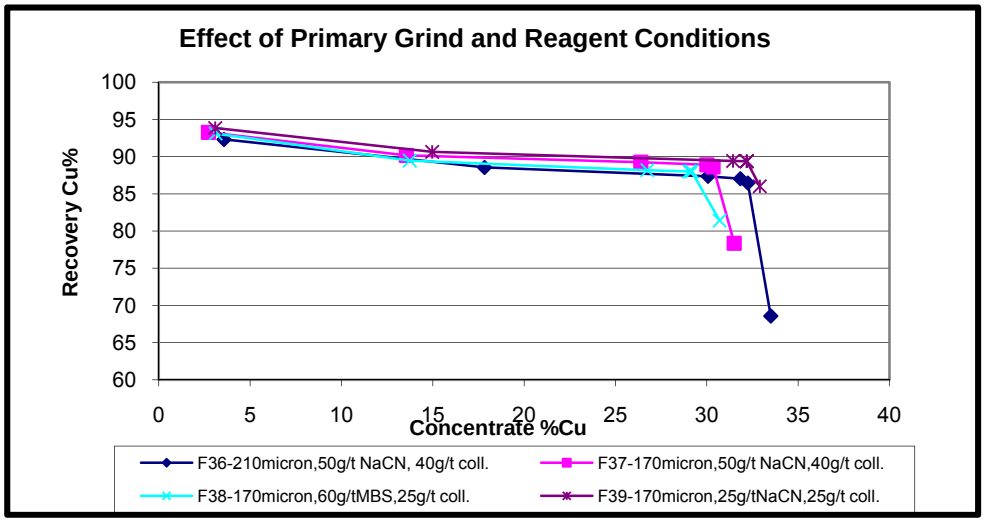


Figure 13-7: Effect of Primary Grind and Reagent Conditions

Locked Cycle Tests

An initial locked cycle test LCT1 was performed on the 8-year composite sample under the optimised conditions from flotation test F39:

- Primary grinding to a P_{80} of 170 µm and rougher flotation at the natural pH of 7.5; 50 g/t NaMBS was added to the grind
- Cytec Aero 3926 promoter was added to the grind at 10 g/t, and an additional 10 g/t was added to the roughers and cleaners
- Cytec Aerophine 3418A promoter was added to the 3rd and 4th rougher stages at 5 g/t per stage
- lime at 150 g/t and NaCN at 25 g/t were added to the regrind mill
- the rougher concentrate, after regrinding to a P_{80} of 27 µm, was cleaned in three stages.

Table 13.14 shows the metallurgy projection based on the last three of the 6-cycle locked cycle tests.

Table 13.14: Metallurgy Projection from LCT1 on 8-Year Composite

		Assays ,		% Distribution	
Product	**Wt %**	**Cu%**	**Au** g/t	**Cu**	**Au**
3rd Cleaner Concentrate	1.2	30.3	3.38	91.0	58.4
1st Cleaner Scavenger Tail	11.4	0.11	0.10	3.1	16.5
Rougher Tail	87.4	0.027	<0.02	5.9	25.2
Head Calculation	100.0	0.40	0.07	100.0	100.0

The regrind mill discharge particle size, after a 15-minute grind, was P_{80} of 26.5 µm. The cleaner concentrate grind size, however, was considerably coarser at P_{80} of 45 µm. It is thought that the regrind mill discharge has a bi-modal population consisting of hard, coarse sulphide minerals and soft iron oxide minerals.

The locked cycle test was repeated with a shorter regrind time of 10 minutes to confirm the regrind size. As a result of the somewhat coarser grind, the concentrate grade decreased to 28.8% Cu and the recovery to 90.2%.

The benefit of sodium MBS addition to the primary grind and lime addition to regrind were examined subsequent to the completion of the locked cycle tests and compared to the results obtained in tests F39.

The results of test F42 indicated that eliminating sodium MBS has no negative effect on the metallurgy. Test F43 showed that without the addition of lime the concentrate grade decreased from 32.2% to 29.4% Cu.

Figure 13-8 shows the locked cycle test flowsheet.

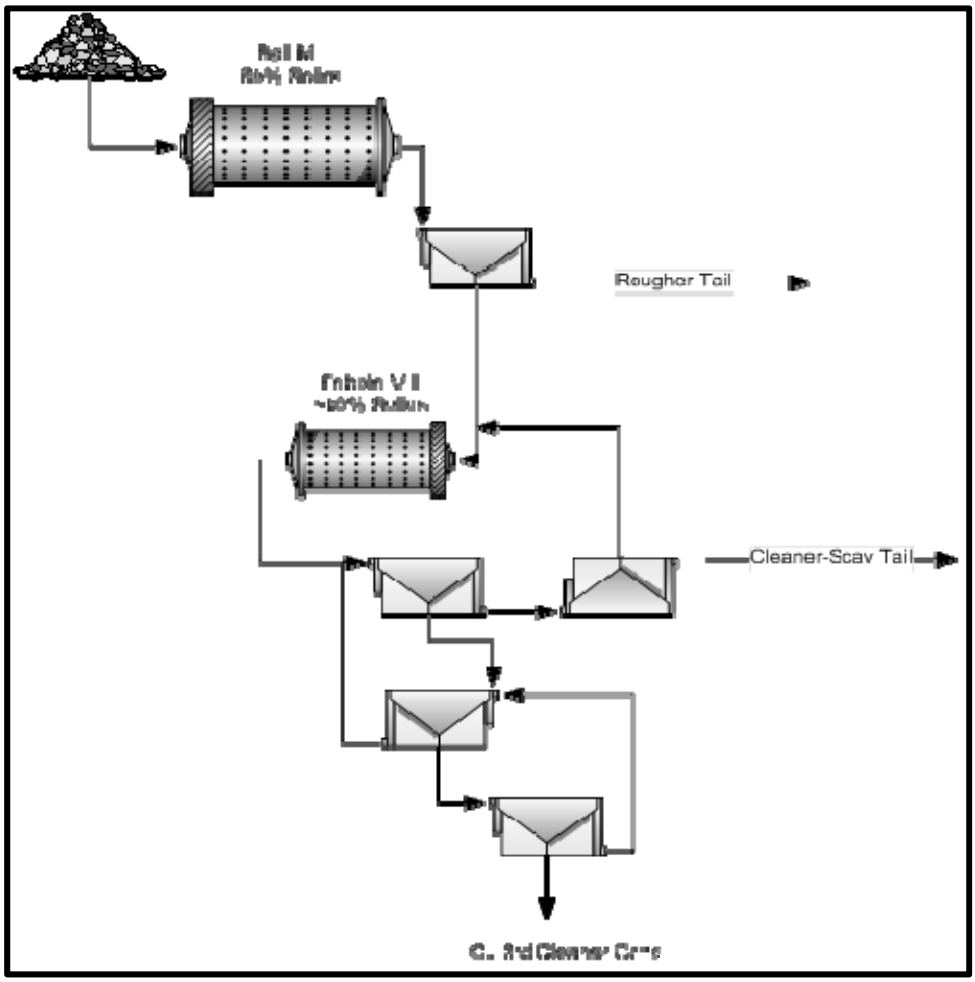


Figure 13-8: Metallurgy Projection from LCT1 on 8-Year Composite

The conditions of LCT1 were considered to be optimal and used in the locked cycle tests on the magnetite and hematite composites. These selected flotation conditions are summarised below in Table 13.15.

Table 13.15: Flotation Conditions used in Locked Cycle Tests

	Reagents, g/t						Time, minutes				
Stage	**LIME**	**NaMBS**	**3926**	**3418A**	**MIBC**		**Grind**	**Cond.**	**Froth**	**pH**	**Eh**
Primary Grind	-	50	10	-	-	-	17			7.4	-40
Rougher 1	-	-	-	-	15	-		1	2	7.4	-40
Rougher 2	-	-	2.5	-	5	-		1	4		

	Reagents, g/t						Time, minutes				
Stage	LIME	NaMBS	3926	3418A	MIBC		Grind	Cond.	Froth	pH	Eh
Rougher 3	-	-	2.5	5		-		1	4	7.6	60
Rougher 4	-	-	-	5	5	-		1	3		
		NaCN									
Regrind	150	25	2.5	-			15			8.2	100
(Pebble Mill)											
Cleaner 1					10				6		
Cleaner-Scav			2.5		10	-		1	2	8.2	100
Cleaner 2									5	8.0	90
Cleaner 3									4	7.9	80

In test H2, the 3rd cleaner concentrate grade of the hematite composite was 20% at a recovery of 91%. In test H3, additions of 3418A to the rougher were reduced from 10 g/t to 5 g/t, and the addition of 3926 to the regrind was eliminated. Concentrate grade increased to 29.1% Cu at 89.1% recovery.

In test M2, the 3rd cleaner concentrate grade of the magnetite composite was 27.6% at a recovery of 82.5%. In test M3, as hematite composite, 3418A additions to the rougher were reduced from 10 g/t to 5 g/t, and the addition of 3926 to the regrind was eliminated. Concentrate grade increased to 29.4% Cu at 88.8% recovery.

These conditions formed the basis for subsequent locked cycle testing on the two composites; the locked cycle test results are shown in Table 13.16 and Table 13.17.

Table 13.16: Metallurgy Projection from Locked Cycle Test on Magnetite Composite

		Assays, %, g/t		% Distribution	
Product	Wt %	Cu	Au	Cu	Au
3rd Cleaner Conc.	0.7	26.5	4.48	90.7	55.1
Head Calc.	100.0	0.20	0.06	100.0	100.0

Table 13.17: Metallurgy Projection from Locked Cycle Test on Hematite Composite

		Assays, %, g/t		% Distribution	
Product	%	Cu	Au	Cu	Au
3rd Cleaner Conc.	1.3	29.1	4.64	89.6	68.3
Head Calc.	100.0	0.42	0.09	100.0	100.0

Analysis of the copper concentrates produced in these locked cycle tests is shown in Table 13.18. Elemental analysis of the flotation concentrates indicate they are of high quality, containing attractive gold and silver concentrations, and have very low penalty element levels including chloride which was below 300ppm after fresh water washing. The concentrates should therefore be easy to market.

Table 13.18: Locked-Cycle Test Cu Concentrates

Element	8-Year Pit	Hematite	Magnetite
Cu %	30.3	29.1	26.5
Fe %	29.4	27.7	27.3
S%	30.9	28.4	28.5
Au g/t	3.4	4.6	4.5
Ag g/t	26.6	32.4	47.4
Hg g/t	4.6	4.0	2.4
Cl g/t	292	170	225
F %	0.013	0.007	0.012
SiO_2 %	1.89	5.32	6.7
As g/t	67	90	124
Bi g/t	<40	<60	<50
Cd g/t	< 10	< 20	107
Co g/t	416	624	433
Cr g/t	102	312	167
Pb g/t	198	317	1810
Mn g/t	312	422	556
Ni g/t	79	229	129
Sb g/t	195	224	25
Se g/t	36	< 40	46
Sn g/t	< 20	< 20	37
Zn g/t	804	1380	8140
size			
P_{80} µm	44.4	41.4	42.7

Oxide Composite

A small number of tests were conducted to assess the flotation response of the oxide composite. Sequential sulphide-oxide flotation with sulphidization agents was explored. In the best of six scoping tests 25% of the copper was recovered in a rougher concentrate grading 3.2% Cu. The poor recovery was explained by the results of the mineralogy study, which indicated that the bulk of the copper is present as non-floatable minerals, such as chlorite, delafossite and copper sulphates. Copper deportment of the oxide composite is shown in Figure 13-9.

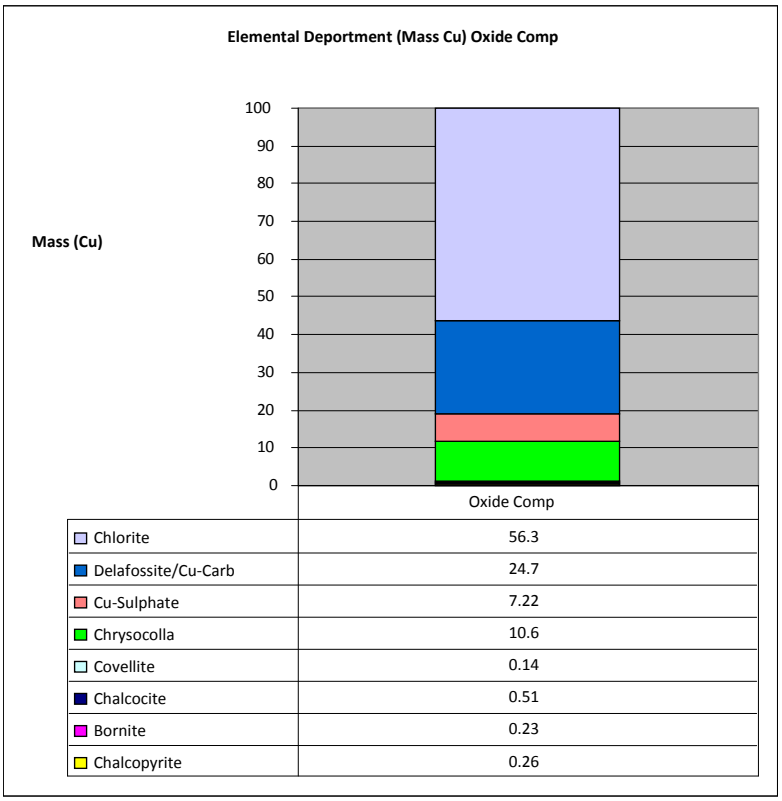


Figure 13-9: Elemental Deportment (Mass–Cu) Oxide Composite

Variability Tests

Thirty-eight variability samples were submitted to batch cleaner flotation tests, with conditions based on LCT1, to determine the copper head grade recovery response. About ten of the samples were removed from the database, as they contained above normal oxide and metallic copper concentrations.

The rougher tailings from the variability tests were shipped to SGA in Germany for Davis Tube and LIMS testing.

The copper head grade versus recovery data were plotted (see Figure 13-10) to determine a copper recovery algorithm for use in the mine block model. There is some scatter in the data, which may be caused in part by the fact that the test conditions were not optimized. The average concentrate grade achieved in these test was about 30% Cu, which is consistent with the results of the locked cycle tests.

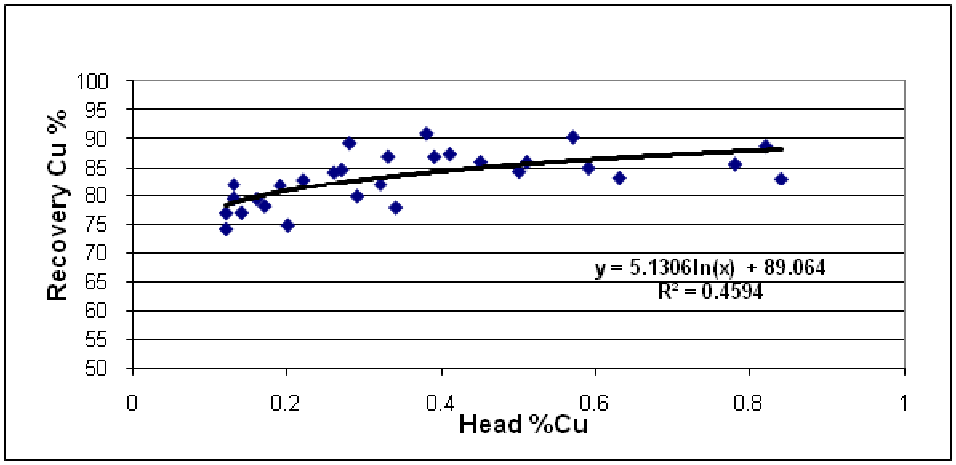


Figure 13-10: Head % Cu vs. Batch Cleaner Test Global Recovery

While the gold recovery to the rougher concentrate showed a lot of scatter, the gold recovery to the final copper concentrate showed a fairly strong correlation with the gold head grade, as can be seen in Figure 13-11. The scatter at the rougher stage is believed to be caused by the variability in mass recovery.

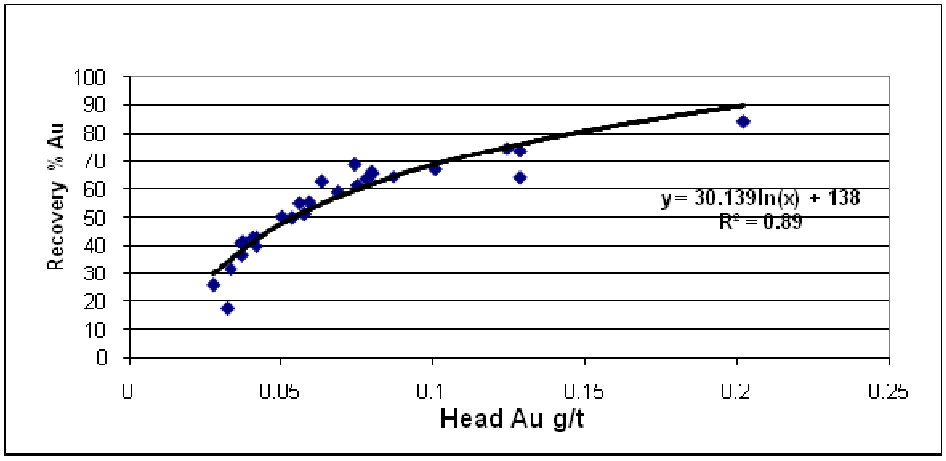


Figure 13-11: Gold Head vs. Gold Batch Test Global Recovery

The head grade-recovery trend line regression equations have acceptable correlation coefficients for this level of study, albeit a low correlation index for copper. The variability batch cleaner test conditions were not optimized and potential improvements can therefore be anticipated in the next level of study.

As locked cycle tests were not conducted on the variability samples scale-up to closed circuit operation has been based on the results of the locked cycle tests on the three composites. The average copper results of the batch and locked cycle tests are shown in the Table 13.9.

Table 13.19: Average of Batch and Locked Cycle Test Results - Cu

		Rghr. Rec.		Clnr. Rec	Global. Rec.
	Head %Cu	Wt %	Cu %	Cu %	Cu %
Var. Batch Avg.	0.36	9.4	93.1	89.1	83.0
Comp Batch Avg.	0.34	11.6	93.1	95.0	88.4
Comp LCT Avg.	0.34	12.2	93.4	96.5	90.7

Bulk Flotation Tests to Produce Samples for Additional Testwork

SGS performed bulk flotation tests at optimum conditions to produce the following products:

- Copper rougher concentrate from the 8-year composite for regrind power tests by Metso. The rougher tailings produced in these tests were shipped to SGA for LIMS testing (Tests C3-C22).
- Copper rougher flotation tailings of the hematite and the magnetite composites for LIMS recovery studies at SGA.
- Copper cleaner concentrate from the 8-year composite for concentrate filtration and washing tests by Outotec (Bulk1–Bulk 38 tests). The rougher tailings from these tests were also shipped to SGA.

Table 13.20 shows the mass balances of these tests.

Table 13.20: SGS Bulk Flotation to Produce LIMS Feed Samples for SGA

		Assays %		Distn. %		
	Weight %	Cu	Magn. Satm.	Cu	Magn.	Assay Magn. by
8-Year Composite						
Cleaner 3 Conc.	1.5	25.6	0	91.3	0.0	
Cleaner-Scav. Tail	8.1	0.15	5.5	2.8	3.1	SGS
Rougher Tails	90.4	0.028	15.2	5.9	96.9	SGA
Feed Calc.	100	0.43	14.2	100.0	100.0	
Feed Direct			14.0			SGS
Hematite Composite						
Rougher Conc.	9.2	4.26	8.2	93.5	8.1	SGS
Rougher Tail	90.8	0.03	9.4	6.5	91.9	SGA
Feed Calc.	100.0	0.42	9.3	100.0	100.0	
Feed Direct			8.2			SGS
Magnetite Composite						
Rougher Conc.	10.7	1.83	26.3	93.6	10.3	SGS
Rougher Tail	89.3	0.015	27.5	6.4	89.7	SGA

13.3.5 Iron Recovery Testwork Program

The optimization testwork of LIMS iron recovery for the pre-feasibility study was continued at SGA.

Most of the testwork was conducted in synthetic seawater, but some tests with fresh water were included so a response comparison could be made. During the course of the program it was decided not to pursue hematite recovery in the scope of work. It was also decided to exclude the copper cleaner scavenger tailings in the LIMS feed because of the relatively high sulphur the copper cleaner tailings from the magnetite recovery feed due to the relatively high levels of sulphur and copper content of this stream. The copper rougher flotation tailings represent about 90% of flotation feed mass.

The work at SGA, conducted on copper rougher flotation tailings samples produced at SGS Lakefield, consisted of the following activities:

- Optimization of the LIMS process conditions for the 8-year composite, with particular emphasis on optimizing the rougher concentrate regrind level
- Applying the optimized conditions to the hematite and magnetite composites
- Conducting a series of tests on variability samples to determine the correlation between Davis Tube test results and LIMS cleaner tests and to determine the correlation between Satmagan/magnetic susceptibility head grade and Davis Tube test recovery.

Table 13.21 shows the head assays of the samples received by SGA.

Table 13.21: SGA Cu-Rougher Tailings Composite Head Grades

		8-Year Pit Nov. 2010	8-Year Pit Mar. 2011	Hematite Mar. 2011	Magnetite Mar. 2011
Fe_{tot}	(%)	30.3	29.1	25.5	30.4
SiO_2	(%)	30.3	30.4	33.4	31.5
S	(%)	0.44	0.16	NA	NA
Na_2O	(%)	1.03	0.98	NA	NA
K_2O	(%)	1.55	1.55	NA	NA
Cu	(%)	N/A	0.025	0.034	0.015
Magnetite*	(%)	16.1	15.2	9.4	28.7
P_{80} size	µm	180	170	150	150

Note: Magnetite* measured by Satmagan.

Two shipments of flotation tailings of the 8-year composite were made; the November shipment was somewhat higher in grade, as more copper rougher concentrate was floated than for the March 2011 shipment.

LIMS Optimization Studies on the 8-Year Composite

The program began with a series of tests to determine the relationship between regrind level and SiO_2 content of the final LIMS concentrate.

Copper rougher tailings from the 8-year composite produced in November 2010 were used for this work. A LIMS cobber or rougher concentrate was produced for cleaning tests in fresh water at regrind times ranging between 15 and 70 minutes.

Table 13.22: LIMS Rougher from 8-Year Composite November 2010

Product	Weight %	Fe %	Fe Rec. %	Mag %	Mag Rec. %	S%
Cobber-Concentrate	26.5	56.4	49.3	56.2	94.2	0.22
Tailings	73.5	20.9	50.7	1.2	5.8	0.54
Feed	100.0	30.3	100.0	15.8	100.0	0.46

Table 13.23 shows that a 20-minute regrind time in a lab rod mill produced a high concentrate grade with a SiO_2 level of less than 5%. The initial 20-minute regrind test was repeated in fresh water and in seawater.

Table 13.23: The Effect of Regrind Size on LIMS Concentrate Grade and Impurity Levels

Grinding Min	Weight Rec %	Fe %	Fe Rec. %	SiO_2 %	Na_2O + K2O %	Conc. > 0,04 mm %	Blaine
Fresh Water							
15	78.4	64.0	89.0	5.76	0.390	43.7	1.246
20	75.6	65.2	87.4	4.65	0.285	35.3	1.409
20r	75.8	65.5	88.0	4.4	0.292	28.2	1.371
25	76.1	65.8	88.8	4.32	0.250	19.4	1.507
30	74.8	67.4	89.4	3.66	0.229	10.3	1.556
30	75.1	66.7	87.8	3.84	0.237	10.7	1.573
35	74.9	67.3	89.6	3.61	0.235	5.6	1.739
40	72.5	67.6	86.9	3.35	0.202	2.9	1.897
50	70.9	67.8	85.2	2.92	0.182	1.8	1.938
70	71.5	68.3	84.8	2.83	0.175	0.3	2.164
Seawater							
20	72.1	64.3	86.0	5.13	0.329	35.6	1.257

The results of the seawater test were somewhat lower than that achieved in fresh water; this was likely caused by the coarser grind of this test.

Regrind mill discharge sizes of 25% +0.4 mm and 30% +0.4 mm were calculated (from the concentrate and cleaner tailings sizes) for the 20-minute regrind tests in both fresh water and in seawater. The cleaner feed size is finer than the concentrate size, as fine specularite is rejected in the cleaner tailings.

From this work it was concluded that the coarsest regrind level of the LIMS rougher concentrate at which a final concentrate could be produced with a SiO_2 level below 5%, was P_{80} 40 µm.

The testwork was continued with the 8-year composite flotation tailings received in March 2011. These tailings were somewhat finer (at P_{80} 170 µm) than those of the previous

shipment, as they were produced from a finer flotation feed size resulting from later copper flotation feed size optimization investigations at SGS Lakefield.

Table 13.24 shows the mass balance of the bulk flotation tests from which these tailings were produced.

Table 13.24: Bulk Flotation of 8-Year Composite to produce LIMS Feed for SGA

		Assays %		Distn. %		Assay
	Weight %	Cu	Magn.	Cu	Magn.	Magn.
3rd Clnr. Conc.	1.5	25.6	0	91.3	0.0	
Cleaner-Scav. Tail	8.1	0.15	5.5	2.8	3.1	SGS
Rougher Tails	90.4	0.028	15.2	5.9	96.9	SGA
Feed Calc.	100	0.43	14.2	100.0	100.0	
Feed Direct			14.0			SGS

A three-stage LIMS cleaning test in seawater of the flotation tailings produced a high-grade concentrate low in silica and alkali content. The pre-concentrate was ground in a ball mill to 7.8% > 63 µm (Table 13.25).

Table 13.25: 3-Stage LIMS Cleaning of 8-Year Composite Rougher Flotation Tailings

Product	Wt. %	Fe Grade %	Fe Rec. %	SiO_2 Grade %	Na_2O Grade %	K_2O Grade %	Mag. Grade %	Mag. Rec. %
3rd Clnr. Conc.	16.8	66.1	38.2	4.1	0.145	0.105	81.6	90.1
Pre-Conc.	24.9	57.8	48.8	12.7	0.29	0.45	56.5	92.6
Tailings	75.1	20.1	51.2	-	-	-	1.5	7.4
LIMS feed	100.0	29.5	29.1	-	-	-	15.2	100.0

Three-Stage LIMS Cleaner Tests on Hematite and Magnetite Composites

Table 13.26 and Table 13.27 show the results of three-stage cleaning tests on the hematite and magnetite composites.

Table 13.26: 3-Stage LIMS Cleaning of Magnetite Composite Rougher Flotation Tailings

Product	Wt. %	Fe Grade %	Fe Rec. %	SiO_2 Grade %	Na_2O Grade %	K_2O Grade %	Mag. Grade %	Mag. Rec. %
Clnr. Conc.	29.8	66.5	66.8	4.4	0.225	0.09	84.8	92.1
Pre-Conc.	40.5	52.8	72.0	13.6	1.05	0.3	65.1	95.9
Tailings	59.5	13.9	28.0	-	-	-	1.9	4.1
LIMS feed	100.0	29.7	100.0	-	-	-	27.5	100.0

Table 13.27: 3-Stage LIMS Cleaning of Hematite Composite Rougher Flotation Tailings

Product	Wt %	Fe Grade %	Fe Rec. %	SiO_2 Grade %	Na_2O Grade %	K_2O Grade %	Mag. Grade %	Mag. Rec. %
Clnr. Conc.	9.8	64.1	24.9	5.7	0.33	0.153	71.2	74.3
Pre-Conc.	14.8	51.1	30.0	13.9	1.35	0.52	50.4	79.4
Tailings	85.2	21.8	70.0	-	-	-	1.7	20.6
LIMS feed	100.0	26.1	100.0	-	-	-	9.4	100.0

A high percentage of the magnetite in the 8-year pit and magnetite composites was recovered to LIMS cleaner concentrates. The resulting concentrates demonstrated iron grades greater than 66% while silica remained below 5%. The response of the hematite composite was not quite as good, as it had a lower magnetite recovery and slightly below-target concentrate specifications. The concentrate grind was coarser on this sample; finer grinding will likely improve the silica rejection.

Table 13.28 shows a summary of each composite recovery and grade.

Table 13.28: LIMS Cleaner Concentrate

Composite	Mag Fd. Grade %	Wt. %	Fe Grade %	Fe Rec. %	SiO_2 Grade %	Na_2O Grade %	K_2O Grade %	Mag. Grade %	Mag. Rec. %
8-Year Comp.	15.2	16.8	66.1	38.2	4.1	0.145	0.105	81.6	90.1
Hematite Comp.	9.4	9.8	64.1	24.9	5.7	0.33	0.153	71.2	74.3
Magnetite Comp.	27.5	29.8	66.5	66.8	4.4	0.225	0.09	84.8	92.1

Typically these concentrates compares very favourably with the major market supplier for fines (Vale) and should be well received in the market by end users looking for low impurity ore sources. The low phosphorous content, in particular, should be attractive to potential buyers.

High grade LIMS concentrates at high magnetite recoveries were produced from the 8-year pit and magnetite composites. The chloride levels of the concentrates were below 100ppm after fresh water washing. Further work is required to improve the concentrate grade from the lower grade hematite composite. Detailed concentrate analyses are presented in Table 13.29.

Table 13.29: Analysis of LIMS Concentrates

		8-Year Pit	Magnetite	Hematite
Fe_{tot}	(%)	66.06	66.50	64.10
FeO	(%)	23.08	26.60	22.60
SiO_2	(%)	4.10	4.35	5.68
Al_2O_3	(%)	1.00	0.94	1.36
CaO	(%)	0.57	0.65	0.67
MgO	(%)	0.455	0.40	0.51
P	(%)	0.011	0.012	0.017
S	(%)	0.02	0.02	0.043

		8-Year Pit	Magnetite	Hematite
Cl	ppm	60	80	90
Na_2O	(%)	0.145	0.225	0.33
K_2O	(%)	0.105	0.09	0.153
Mn	(%)	0.069	0.065	0.07
Cu	(%)	0.0081		
L.O.I.	(%)	1.34	0.76	1.59
>40 µm	%	21.3	18.2	25.3
Blaine		1,896	1,373	1,319

Variability Tests at SGA

Thirty-five variability samples, representing the five ore zones, were selected for the following test program:

- Davis Tube tests at the target regrind level of 20% plus 40 µm
- three-stage LIMS cleaner tests on ten of the samples.

The objective of the LIMS cleaner tests was to determine the correlation between Davis Tube and LIMS batch cleaner test results, as LIMS cleaner test results are considered to be indicative of full-scale plant metallurgy. It should be noted that the tests were-open circuit batch tests, as it had been decided not to recycle the cleaner tailings in order to insure concentrate quality. All tests were performed with synthetic seawater.

The standard method to measure the magnetic iron content of a sample is the Satmagan procedure. Far West Mining has used magnetic susceptibility (MS) measurements (a very inexpensive procedure) as a proxy for Satmagan and/or Davis Tube tests, based on the results of a study on a large number of samples performed in 2008 during the PEA. The excellent correlation between MS and Satamagan was confirmed by the tests performed on the variability samples, as shown in Figure 13-12.

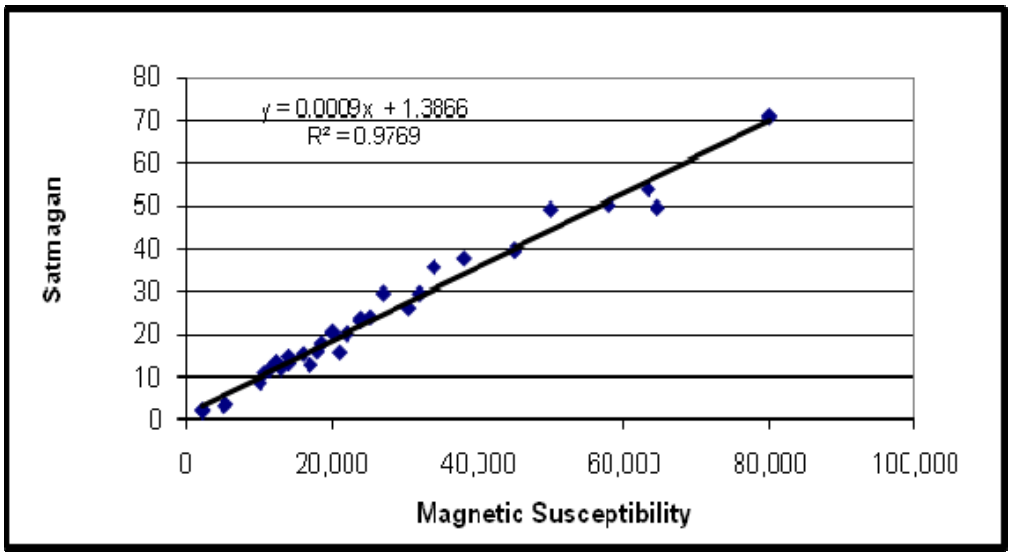


Figure 13-12: Var. Tests – Magnetic Susceptibility vs. Satmagan

It should be noted that these results are from single tests, not always performed at optimal target conditions. Nevertheless, a concentrate containing an average 65% Fe with low silica and alkali values was produced.

A number of the variability samples originating from the various ore zones/types were selected for LIMS tests in order to confirm the correlation between Davis Tube and LIMS tests. The results of these tests, summarized in Table 13.30 and Figure 13-13, indicate that Davis Tube tests can be used with a high degree of confidence to predict LIMS recovery.

Table 13.30: SGA LIMS Tests on Cu Rougher Tailings: Comparison of Davis Tube and LIMS Tests

Sample ID	Fltn Feed MS	Feed Mass %	Feed Mag %	Davis Tube Tests			LIMS 3rd Clnr. Conc.		
				Weight %	Fe %	SiO_2 %	Weight %	Fe %	SiO_2 %
HI 21	22,000	90.8	22.8	24.9	66.4	4.84	24.4	68.6	3.45
MIN 33	24,000	91.0	26.0	26.6	65.9	5.96	26.8	68.8	2.65
H 5	10,600	89.8	12.4	14.9	64.1	6.87	13.5	67.5	4.2
MIM 26	80,000	97.2	71.6	75.9	67.8	3.60	76.0	68.1	3.46
M 3	12,400	93.4	14.3	14.9	62.6	7.24	15.0	65.4	4.97
M 10	45,000	91.7	42.0	53.3	66.0	4.11	52.5	68.4	3.21
H 1	5,400	87.5	4.9	3.7	60.6	2.83	8.1	60.0	2.51
HI 27	13,000	90.8	13.7	16.0	68.2	3.10	15.9	66.2	4.67
M 9	32,000	92.3	31.0	34.9	69.6	1.98	32.3	69.2	1.91
M 13	17,000	88.0	15.0	17.1	68.2	2.47	15.4	68.2	2.36
Avg.		91.3	25.4	28.2	65.9	4.30	28.0	67.0	3.34

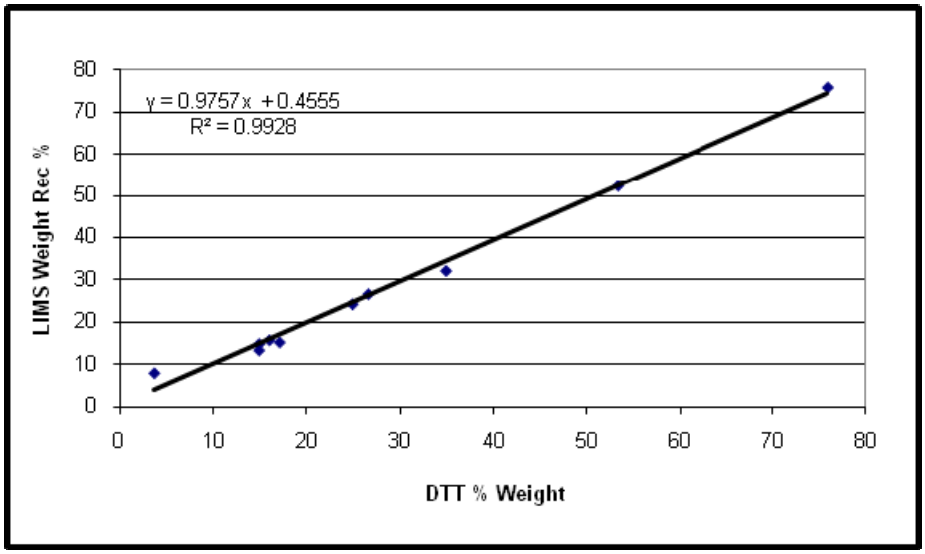


Figure 13-13: LIMS vs. DTT Results

13.3.6 Tests by Equipment Suppliers

Equipment suppliers performed re-grind power tests, concentrate thickening, and filtration and rheology tests for equipment sizing; the results are discussed in the body of this report. Appendix-8 includes the equipment suppliers' test reports.

Concentrate Regrind Power Tests

Copper and LIMS iron rougher concentrates were tested by Metso to determine the specific power required for regrinding; the results of the Jar mill tests are summarized in Table 13.31.

The specific power shown in the table applies to overflow Ball mills.

Table 13.31: Metso Regrind Power Tests

	F_{80} µm	P_{80} µm	Spec. Power kWh.mt
Cu Rougher Conc.	187	30	7.97
LIMS Rougher Conc.	184	40	11.60

The Metso Report can be found in Appendix-8.

Concentrate Filtration and Washing Tests

Concentrate filtration tests were conducted by Outotec Filtration using their Larox PF and Ceramec filters; the results are summarized below.

Ceramec Testwork

Copper Concentrate

Filtration rates using "blue" plates (approximate plate permeability of 1,00 to 1,500 L/m^2-h) were generally poor, with 333 kg/m²-h as the highest reported rate. Cakes were thin at 4 mm or less. Filtration rates would likely improve if feed solids concentration was increased to 65%w/w or higher. It is likely this would result in higher cake moisture.

LIMS Iron Concentrate

Filtration rates were generally good when using slurry at 66.8% w/w solids. At 60.6% w/w or 61.3% w/w, the rates were significantly lower. In all tests cake moistures were good, generally below 9.0%.

Wash results were generally below the 500 mg/L level when thinner cakes were used, but exceeded 1,100 mg/L where thicker cakes were reported.

Ceramec technology is unlikely to be a suitable alternative for Santo Domingo copper concentrate, but is an acceptable option for the LIMS iron concentrate.

Pressure Filtration

Copper Concentrate

Using Outotec Larox PF pressure filtration technology the copper concentrate filtered well, producing filtration rates of more than 500 kg/m²-h, even after washing.

Washing results were excellent, with chlorides concentration of less than 179 mg/L using as little as 0.1 m³/dry tonne of wash water.

LIMS Iron Concentrate

MS iron concentrate filtered as well or better than the copper concentrate. Filtration rates were 663 kg/m²-h or higher, at cake moistures not exceeding 6.3%.

Only one wash test was performed on the LIMS iron concentrate. The result showed that an acceptable chloride level of 398 mg/L can be achieved using a 0.3 m³/dry tonne of wash water.

Copper Concentrate Settling Testwork

A test campaign was conducted on one copper concentrate sample. The tests used the bench scale 100mm diameter thickener test unit to investigate the operating parameters required to obtain underflow density and overflow clarity and subsequent sizing of the concentrate thickener.

The sample had a measured P_{80} 58 microns.

The selected flocculant type and dose rate was MF 10 at 10 g/t. These conditions were used for the dynamic settling tests.

The test samples were diluted with salt water and kept at their natural pH of approximately 8.0. The optimal thickener feed density was determined at 17.4% w/w solids and was targeted throughout the dynamic tests.

The material was found to settle very well and underflow and overflow targets were met. From these results the following operating parameters are expected:

Table 13.32: Copper Concentrate Settling Tests

	flocculant Dosage g/t	Solids loading Rate t/m²-h	Rise Rate m/h	Underflow density % w/w solids	Overflow Density mg/L
Copper Concentrate.	17	0.40	10	74+	<50

LIMS Iron Concentrate Pipeline Slurry Testing

A LIMS iron concentrate sample was shipped to the Ausenco PSI laboratory in Concord California, for rheology tests. This work was not completed at the time of report writing. The Ausenco-PSI report will therefore be issued as a standalone document.

13.4 Comminution Testwork Program

Four comminution testwork phases have been undertaken to date on samples from the Santo Domingo Deposit to date:

- SGS Mineral Services, Santiago, 2006
- SGS Mineral Services, Santiago, 2008
- DJB Program, June 2010
- AMMTEC Program, April 2011.

13.4.1 SGS Mineral Services, Santiago, 2006

Two samples of drill core material, selected by FWM, were tested by SGS Santiago to determine their grindability response. The results from this program are summarized in Table 13.33.

Table 13.33: SGS Mineral Services Santiago 2006 Testwork

Sample	Test Metric					
	Ai	SG	CWi	SPI	Rod Mill Wi	Ball Mill Wi
Units	g	-	kWh/t	min	kWh/t	kWh/t
MCA (Zone A – upper portion)	0.1552	3.70	13.0	80.3	12.5	10.7
MCB (Zone A – lower portion)	0.1003	3.63	11.0	109.9	13.2	11.3

The results indicated low to moderate competency and hardness for the two samples. The high SG of the samples is typical for an IOCG deposit with hemetite and magnetite.

More details of the testwork program, sample selection procedure and results are included in the SGS report "OL-4029 Final Report Flotation & Grinding (2006-2007 Santo Domingo Samples)".

13.4.2 SGS Mineral Services, Santiago, 2008

Five composite samples of drill core material, selected by FWM from a total of 23 samples dispatched to SGS Santiago, were tested to determine their grindability response. The testwork program consisted of the following tests:

- Bond Ball Standard Grindability Tests
- Bond Ball Modified Grindability Tests
- Bond Rod Grindability Tests
- SPI test.

The results from this program are summarized in Table 13.34.

Table 13.34: SGS Mineral Services Santiago 2008 Testwork

Sample	Test Metric			
	Rod Mill Wi	Ball Mill Wi	Ball Mill Wi (mod)	SPI
Units	kWh/t	kWh/t	kWh/t	min
SB-MET08 81	12.0	11.8	13.9	78
SB-MET08 82-A	12.8	10.8	13.9	93
SB-MET08 156-B	14.2	12.3	14.7	117
SB-MET08 156-B	14.3	12.1	14.2	96
SB-MET08 161-B	11.7	10.6	13.5	74.6

The results obtained from this testwork program were similar to those obtained from the 2006 testwork and confirmed the moderate hardness and competency of the Santo Domingo ores.

More details of the testwork program, sample selection procedure and results are included in the SGS report “OL-4230 Final Report Grinding (2008 Santo Domingo samples)”.

13.4.3 DJB Program, June 2010

The 2010 grinding program managed by DJB consisted of three packages of laboratory work:

- SGS (Santiago) grindability program on 128 samples
- Phillips Enterprises (Colorado) crushing and geotechnical program on 128 samples
- SGS (Santiago) ball mill calibration program on four samples.

The geotechnical tests conducted in the Phillips Enterprises package were subcontracted to Advanced Terra Testing Inc. (ATT), also in Colorado.

The results from this program are summarized in Table 13.35.

Table 13.35: DJB Program, June 2010 Averages by Domain

Domain	Average Test Metric					
	SG	Crushing Wi	Rod Mill Wi	Crushing Index	SPI	Ball Mill Wi
Units	-	kWh/t	kWh/t		min	kWh/t
Iris	3.0	9.6	15.4	22.8	115.8	13.8
Iris Mag	3.1	6.6	15.6	24.6	95.5	13.5
Iris Norte	3.0	8.3	15.7	22.1	122.9	13.7
SDS Hem	3.2	9.2	15.7	20.0	137.6	13.8
SDS Mag	3.3	8.4	13.9	20.8	114.1	12.1

The general interpretation of these results is the five ore zones are very similar, with the SDS Magnetite and Iris Magnetite zones displaying slightly softer results than the other three zones. The Rod Mill and SPI data obtained from this round of testing seem to indicate a higher competency than previous tests.

13.4.4 AMMTEC Program, April 2011

At the commencement of this study, it was decided to undertake a limited additional testwork phase of SMC and confirmatory ball mill tests. This testwork was undertaken in Perth by AMMTEC (now ALS AMMTEC).

The samples were selected from the dataset previously selected by DJB to allow a direct comparison of the results to be made. Due to time and budgetary constraints, only a sub-selection of samples was tested.

Analysis on the location and assays of the samples selected by DJB also highlighted that many of samples were not ore, but waste; either outside the identified pit shells or internal dilution. The sub-sample selection criteria were therefore that these samples should have the following:

- be present in ore
- cover a good range of SPI values
- be biased towards the 5 year pit
- cover a good range of Fe values
- cover a good range of BWI & RWI
- cover the majority of the ore types (weighted approximately correctly to the proportions within the orebody).

On the basis of these criteria, 19 samples were selected and submitted for testing. The results of these tests are summarized in Table 13.36.

Table 13.36: AMMTEC April 2011 Testwork

Sample	Test Metric					
	SG	A	b	Axb	DWi	Ball Mill Wi
Units	-	-	-	-	min	kWh/t
DJB #02	2.91	42.2	1.84	77.6	3.75	11.5
DJB #13	3.15	48.3	1.68	81.1	3.88	11.6
DJB #14	3.23	60.9	0.92	56.0	5.78	11.1
DJB #35	2.97	54.1	1.58	85.5	3.47	8.2
DJB #37	2.98	59.3	0.78	46.3	6.48	14.5
DJB #64	3.40	60.9	0.80	48.7	6.97	12.3
DJB #67	2.92	51.9	0.83	43.1	6.79	16.6
DJB #68	3.28	68.1	0.68	46.3	7.11	11.7
DJB #78	3.02	55.8	0.64	35.7	8.5	18.2
DJB #89	3.10	55.0	2.91	160.1	1.93	13.0
DJB #90	3.06	51.0	2.02	103.0	2.97	9.2
DJB #91	3.81	64.4	1.08	69.6	5.48	10.6
DJB #94	2.81	44.8	0.76	34.0	8.22	20.7
DJB #95	3.60	59.7	1.27	75.8	4.75	11.5
DJB #103	2.87	58.4	0.87	50.8	5.63	14.0
DJB #105	2.91	58.5	0.99	57.9	5.04	11.3
DJB #114	3.35	87.0	0.45	39.2	8.62	12.2
DJB #120	2.89	87.4	0.43	37.6	7.77	15.3
DJB #124	3.33	59.9	1.31	78.5	4.24	9.8

For some orebodies, there is a strong negative correlation between ore competency (as measured by the DWi or Axb) and the iron assay. This results in higher mill throughputs for

increasing iron assays in the ore. This is quite often seen in IOCG orebodies like Santo Domingo, where increasing levels of hematite and magnetite result in both lower competency and hardness and hence higher throughputs.

The relationship between the modelled plant throughput and the %Fe for the comminution samples, shown below in Figure 13-14, confirms this relationship for the Santo Domingo orebody There is a real trend for increasing throughput with increasing iron assay, albeit with a wide spread in the data. Although the R2 value for this fit is 0.605, an iron assay of 30% is indicative of a throughput that can range from 60 to 80 kt/d.

Additional work should be considered to improve this correlation, either by subdividing the orebody into zones and mineralisation types (hematite, magnetite, andesite etc.) or by adding parameters to the model like magnetic susceptibility to attempt to improve its predictive power.

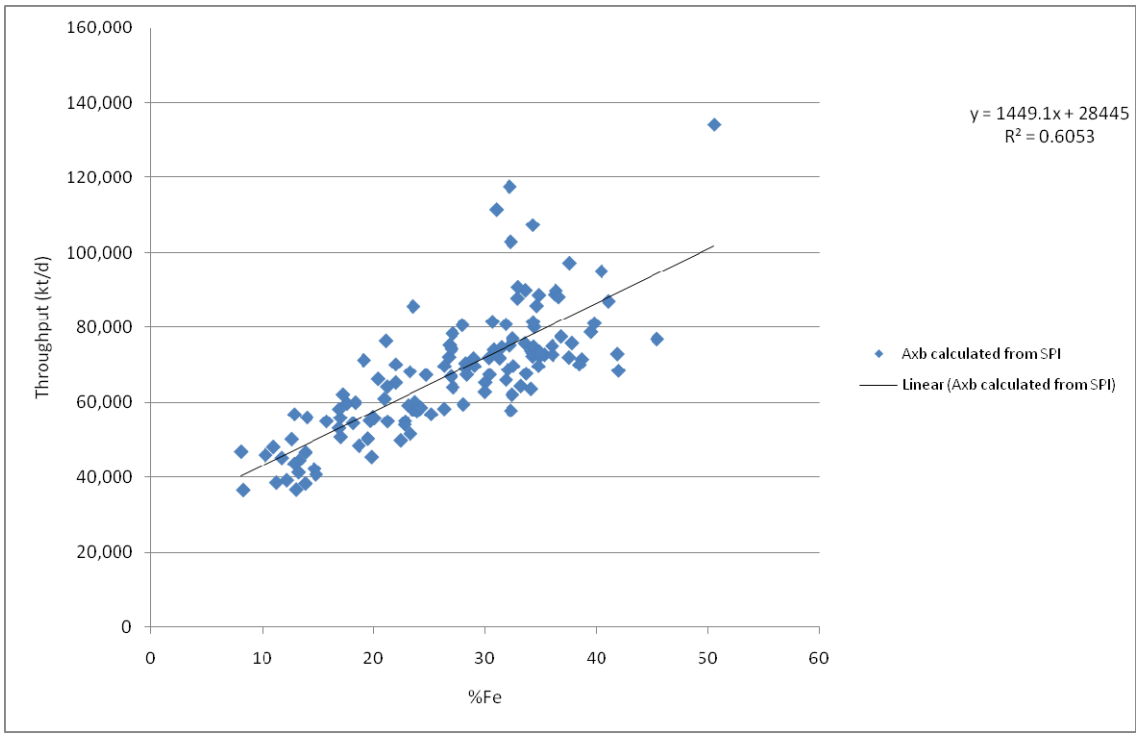


Figure 13-14: Relationship between Predicted Plant Throughput and Sample Iron Assay

This relationship has been incorporated into the SRK block model to allow a throughput prediction to be calculated for each block and hence a prediction of annual throughput to be made for each year of the mine life.

13.5 Recovery Models

13.5.1 Whittle Design Parameters

To facilitate the mine design work indicative grade and recovery estimates were developed and used in the Whittle optimisation model for the pit designs. The PFS testwork results were not available when work started and thus the following parameters were assumed:

- concentrate grade: 29% Cu
- tailings assay: 0.04% Cu constant, independent of head grade
- gold recovery: 70% of copper recovery.

Comparison of the copper recoveries from this approach and the results from testwork are presented in Figure 13-15. The comparison shows an acceptable correlation and resulted in a slightly conservative estimate in the key range of 0.2 to 0.5% Cu where essentially all of the ore blocks occur.

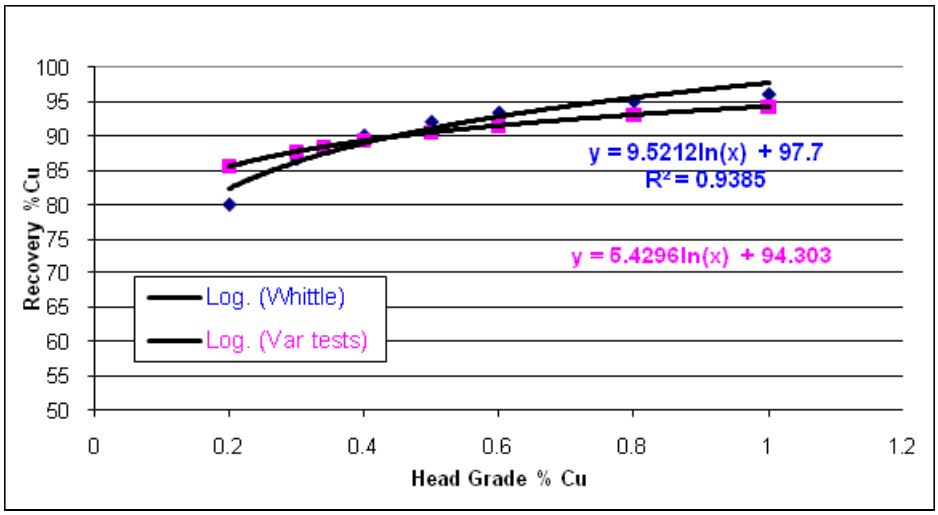


Figure 13-15: Cu Recovery Algorithm Comparison

Subsequent testwork has shown that the gold recovery used was optimistic but as gold contributes approximately 5% of the revenue it is not considered material in influencing the optimum Whittle shell selected.

The financial model utilizes the recovery algorithms developed from the testwork results.

13.5.2 Flotation Recovery Models

Locked cycle test results of the composites indicated that 96.5 % of the copper recovered in the rougher stage reports to the cleaner concentrate. In the variability batch cleaner tests the average stage copper recovery in the cleaners was 89.1%. In order to project continuous full-scale operation recoveries a series of correction factors were developed. The copper

recovery from batch to locked cycle was scaled-up according to the above averages. A 2% correction was also introduced to allow for a loss of recovery when scaling up from laboratory to full-scale. The correction factors for copper recovery were:

- copper recovery factor for batch to locked cycle: 96.5/89.1= 1.08
- copper recovery factor for bench scale to full scale: 98/100= 0.98
- projected full-scale copper recovery: y = 0.98*1.08*[5.13*Ln(x) +89.1].

The final algorithm projects a locked cycle recovery of 90.2% for the average head grade of 0.34% Cu. This is close to the average locked cycle result of 90.7%, from the three composites. As samples with head grades above 0.4% Cu were not tested, it is recommended to cap the copper recovery at 92.5% until testwork data at higher head grades become available.

The full-scale gold recovery projections are based on a similar approach as that used for copper. Gold head grades within the composites were low and reproducibility is limited by the accuracy of gold assay techniques. The average results of the batch and LCTs are compared in Table 13.37.

Table 13.37 Average of Batch and Locked Cycle Test Results - Au

		Rghr. Rec.		Clnr. Rec.	Global Rec.
	Head Au g/t	Wt %	Au %	Au %	%Au
Var. Batch Avg.	0.07	9.4	66.0	81.1	53.5
Comp. Batch Avg.	0.06	11.6	93.1	NA	NA
Comp. LCT Avg.	0.07	12.2	74.2	81.5	60.6

For copper recovery, the following scale-up factors were utilized to determine full-scale gold recovery:

- gold recovery factor, batch to locked cycle: 1.13
- gold recovery factor for bench scale to full scale: 98/100= 0.98
- projected full-scale gold recovery: y=1.13*0.98*[30.14*Ln(x) +138].

The algorithm predicts a locked cycle gold recovery of 57.8%; this compares to an average locked cycle gold recovery of 60.6%. The batch test recovery of highest grade samples (containing 0.2 g/t Au) tested was 88.7% and the algorithm predicts a batch test recovery of 89.5% and a full-scale plant recovery of 87.7%.

13.5.3 Magnetite Recovery Models

The magnetite recovery in the mine block model is calculated from the MS values assigned to each block using an MS head grade-to-weight recovery algorithm. The algorithm was derived from MS measurements and Davis Tube tests on a large suite of whole ore samples. The LIMS feed in the SGA variability tests were the copper rougher flotation tailings, not whole ore; the weight recovery algorithm derived from the test results therefore overestimates the LIMS recovery by about 10%, as the rougher tailings represent 90% of the original sample mass.

The LIMS weight recovery was reduced by 10% to indicate the LIMS weight recovery from whole ore, so that it can be compared to the mass recovery predicted by the algorithm used in the mine block model.

The mass recovery algorithm used in the block model is shown below.

MR% (Mass Recovery) = IF(MS>2,1.1063*MS-0.003*(MS)^2,0)

MS=magnetic susceptibility/1,000; MR = mass recovery

In Table 13.38 the LIMS mass recovery calculated with the algorithm is compared to the results of the LIMS tests after adjusting the weight recovery by 10%.

Table 13.38 LIMS Weight Recovery Test vs. Algorithm

Sample ID	Process Feed MS	LIMS Feed Mass %	DTT Weight %	LIMS Weight Recovery		
				Test %	Adj. 10%	Algor.
HI 21	22,000	90.8	24.9	24.4	22.4	22.9
MIN 33	24,000	91.0	26.6	26.8	23.9	24.8
H 5	10,600	89.8	14.9	13.5	13.4	11.4
MIM 26	80,000	97.2	75.9	76	68.3	69.3
M 3	12,400	93.4	14.9	15	13.4	13.3
M 10	45,000	91.7	53.3	52.5	48.0	43.7
H 1	5,400	87.5	3.7	8.1	3.3	5.9
HI 27	13,000	90.8	16.0	15.9	14.4	13.9
M 9	32,000	92.3	34.9	32.3	31.4	32.3
M 13	17,000	88.0	17.1	15.4	15.4	17.9
Avg.		91.3	28.2	27.99	25.4	25.5

The mass recovery calculated with the algorithm is in good agreement with the LIMS test results adjusted for the weight loss to the copper rougher concentrate.

14 MINERAL RESOURCE ESTIMATE

RPA has updated the Mineral Resource estimates for the SDS and Iris Zones.

The following sections are exerts from the report[16].

The resource estimates include data from recent measurements of magnetic susceptibility, as well as 35 drill holes completed since the last estimate, which was carried out by RPA in 2009 (Lacroix, 2009). Five additional holes were used in the geological interpretation, but not the grade interpolation, as the assay results had not been received. The cut-off for the assay data was 15 May 2010, and the estimate is considered to be current to that date. The updated estimate is summarized in Table 14.1.

Table 14.1: Indicated and Inferred Mineral Resources (15 May 2010)

Zone	Mt	% CuEq	% Cu	g/t Au	%Fe
Indicated					
SDS (1-4)	275	0.64	0.41	0.056	27.8
Iris (5-6)	111	0.50	0.23	0.033	26.3
Iris Norte (7-8)	99.5	0.47	0.16	0.019	26.4
Indicated (SDS/Iris)	486	0.57	0.32	0.043	27.2
Estrellita	31.7	n/a	0.53	0.050	n/a
Total Indicated	**517**		**0.33**	**0.044**	
Inferred					
SDS (1-4)	30.5	0.46	0.26	0.037	23.7
Iris (5-6)	5.52	0.47	0.19	0.026	26.0
Iris Norte (7-8)	25.3	0.47	0.10	0.011	27.9
Inferred (SDS/Iris)	61.3	0.46	0.19	0.025	25.7
Estrellita	2.7	n/a	0.48	0.050	n/a
Total Inferred	**64.0**		**0.20**	**0.026**	

Notes: CIM definitions were followed for Mineral Resources. Mineral Resources for SDS/Iris are estimated at a cut-off grade of 0.25% CuEq. The cut-off for Estrellita was 0.3% Cu. CuEq grades are calculated using average long-term prices of US$2.25/lb Cu, US$950/oz Au, and US$0.74/dmtu Fe ($50/dmt conc. @ 67.5% Fe). CuEq calculations are as stated in the text of this document. Metallurgical recovery factors were applied as described in this document. Mineral Resources are inclusive of Mineral Reserves.

The Estrellita Zone, which was estimated in 2007, was not included in this update as there has been no change to the database for this deposit.

The estimate was carried out using a block model constrained by three dimensional (3D) wireframe envelopes. The wireframes were constructed primarily from lithological boundaries. The principal rock types used for these models were the manto-hosting volcanic

[16] Far West Mining Ltd. Technical Report on the Santo Domingo Property, Region III, Atacama Province Chile, NI 43-101 Report, Author: David W. Rennie, P.Eng. August 26, 2010 Scott Wilson Roscoe Postle Associates Inc.

and sedimentary units, which were clipped against fault boundaries and wireframe models of post-mineral dykes or sills. Eight domains were created within the deposit and three of these (Zones 1, 2, and 3) were further subdivided into magnetite-rich and magnetite-poor variants. Both FWM and RPA had done much of the geology interpretation for the 2009 estimate. For this more recent estimate, the wireframe modelling consisted of updating the earlier work with the new drill results. RPA notes that only minor modifications to the interpretations were required.

Grades for Cu, Au, total Fe, and magnetic susceptibility (MS) were estimated into the blocks using ordinary kriging (OK). Estimates of recoverable Fe (Fe_rec) and bulk density were carried out from the estimated Fe and MS grades using linear regression relationships. Copper equivalent (CuEq) grades were calculated from the estimated Cu, Au, and Fe_rec, using recoveries estimated from recent metallurgical testing. These calculations are described in more detail in separate sections of this report.

The mineral resources for SDS/Iris were reported at a cut-off of 0.25% CuEq, which is consistent with the 2009 estimate.

The database used for the estimate contained records for 217 RC and diamond drill holes, with a total of 31,628 sampled intervals.

14.1 Previous Estimates

NI 43-101-compliant mineral resource estimates and technical reports for the SDS area were produced by RPA in June 2006 (Lacroix, 2006), October 2007 (Lacroix and Rennie, 2007), and June 2009 (Lacroix, 2009). An estimate for Estrellita was also completed by RPA in 2007, but has not been changed since and is still considered current. A summary of the Mineral Resource estimate as of June 2009 is provided in Table 14.2.

Table 14.2: 2009 Mineral Resource Estimate

Zone	Mt	% Cu	g/t Au	%Fe
Indicated				
SDS/Iris	383	0.39	0.05	27.0
Estrellita	31.7	0.53	0.05	n/a
Total	**415**	**0.40**	**0.05**	**25.0**
Inferred				
SDS/Iris	68.6	0.26	0.04	24.6
Estrellita	2.7	0.48	0.05	n/a
Total	**71.3**	**0.27**	**0.04**	**25.0**

Notes: CIM definitions were followed for Mineral Resources. Mineral Resources for SDS/Iris were estimated at a cut-off grade of 0.25% CuEq (the same used for the 2010 update). The cut-off grade for Estrellita was 0.3% Cu. CuEq grades were calculated using average long-term prices of US$2.25/lb Cu, US$950/oz Au, and US$0.74/dmtu Fe ($50/dmt conc. @ 67.5% Fe). CuEq calculations were as stated in the text of this document. Metallurgical recovery factors were applied as described in this document.

The updated 2010 estimate is compared to the 2009 estimate in Table 14.3. The tonnage is observed to have increased for the Indicated category, an increase accompanied by a drop in overall Cu and Au grades. RPA notes that the wireframe models for the domains did not change appreciably in overall extent from the last estimate. The tonnage difference would not

then have resulted from an increase in the volume of the deposit. In RPA's opinion, the tonnage difference is partially due to an increase in the estimated recoverable Fe, which resulted from the improved and expanded database for MS. This would have the effect of increasing the CuEq value in some blocks that are low in Cu and Au, but high in recoverable Fe. These blocks would have been excluded from the 2009 estimate on the basis of the CuEq grade, but would be captured in the 2010 estimate. There would also have been some Inferred blocks upgraded to Indicated due to the additional drill holes. This would increase the tonnage of the Indicated at the expense of the Inferred class.

The Inferred category is largely unchanged in terms of tonnage, but shows a decrease in grade. In RPA's opinion, the decrease in grade is again due to the increase in the estimated recoverable Fe. Any tonnage increase associated with this was offset by upgrading Inferred resources to Indicated resources.

Table 14.3: Comparison of 2010 and 2009 Estimates

Zone	Mt		% Cu	g/t Au	%Fe
Indicated					
2009	415		0.40	0.05	25.0
2010	517		0.33	0.04	25.0
% Diff	**24.6%**		**-16.9%**	**-13.1%**	**0.0%**
Inferred					
2009	68.6		0.26	0.04	24.6
2010	64.0		0.20	0.03	25.0
% Diff	-6.7%		-22.2%	-34.9%	1.6%

14.2 Database – General Description

The Mineral Resource estimates for the SDS, Iris, and Iris Norte deposits are primarily based on information from surface drilling, supplemented in part by surface mapping and geophysics to assist in the interpretations. As stated above, the database provided to RPA contained collar records for 217 holes. Of these, 37 are diamond drill holes or holes collared as RC and then finished as diamond holes. Sixteen holes were drilled as twins. Most of the holes are vertical or near vertical, with 76 collared at a dip shallower than -80°. Hole lengths vary widely, but are typically range between 200 m and 400 m.

Drilling on SDS, Iris, and Iris Norte covers an approximate area of 4,200 m (north-south) by 1,200 m (east-west), with the density of drilling decreasing toward the north and northwest. All but one of the holes are within the block model boundaries, and 208 actually pierce the interpreted mineralized zones. Further detail on drilling can be found in the drilling section.

14.3 Assays

The assay database provided to RPA contains 31,628 assay intervals, of which 26,749 have non-zero values for Cu, Au, or Fe. Most sampled intervals are two meters in length for RC holes and one meter or two meters for diamond holes. A total of 13,772 Cu, Au, and Fe intervals are located within the interpreted mineralized zones. In addition to the conventional

assays for Cu, Au, and total Fe, FWM personnel have collected MS readings on sample reject material. There are now 10,705 samples with MS values within the mineralized domains. Brief statistical summaries of the SDS, Iris, and Iris Norte zone assays are provided in Table 14.4, Table 14.5, Table 14.6, and Table 14.7.

Table 14.4: Sample Assay Statistics – %Cu

Zone	Count	Min	Max	Mean	Median	Std. Dev.	CV
1	4,937	0.000	5.34	0.452	0.311	0.555	1.143
2	784	0.000	4.29	0.284	0.150	0.414	1.455
3	1,384	0.000	5.38	0.397	0.250	0.485	1.222
4	917	0.000	3.15	0.515	0.394	0.464	0.902
5	2,889	0.000	3.34	0.226	0.050	0.362	1.599
6	955	0.000	1.00	0.089	0.060	0.099	1.116
7	526	0.000	2.93	0.274	0.130	0.383	1.398
8	1,382	0.000	3.10	0.132	0.030	0.275	2.085
All	**13,772**	**0.000**	**5.38**	**0.342**	**0.153**	**0.470**	**1.376**

Note: Includes “below detection” as 0.0.

Table 14.5: Sample Assay Statistics – g/t Au

Zone	Count	Min	Max	Mean	Median	Std. Dev.	CV
1	4,937	0.000	0.65	0.067	0.043	0.074	1.108
2	784	0.000	2.38	0.043	0.020	0.102	2.391
3	1,384	0.000	1.07	0.055	0.035	0.072	1.305
4	917	0.000	1.18	0.070	0.050	0.078	1.124
5	2,889	0.000	4.71	0.034	0.010	0.105	3.089
6	955	0.000	0.26	0.012	0.010	0.020	1.582
7	526	0.000	0.40	0.033	0.012	0.050	1.535
8	1,382	0.000	0.98	0.018	0.000	0.046	2.578
All	**13,772**	**0.000**	**4.71**	**0.048**	**0.020**	**0.081**	**1.699**

Note: Includes “below detection” as 0.0.

Table 14.6: Sample Assay Statistics – %Fe

Zone	Count	Min	Max	Mean	Median	Std. Dev.	CV
1	4,937	0.000	68.30	28.484	29.700	14.005	0.492
2	784	0.000	64.90	20.410	18.525	10.657	0.522
3	1,384	0.000	64.30	29.746	29.700	12.921	0.434
4	917	0.000	56.20	27.994	28.600	12.714	0.454
5	2,889	0.000	64.30	24.495	23.900	12.219	0.499
6	955	0.000	65.00	28.096	24.300	13.181	0.469
7	526	0.000	62.50	25.683	24.200	15.474	0.603
8	1,382	0.000	56.90	22.897	21.800	13.661	0.597
All	**13,772**	**0.000**	**68.30**	**26.594**	**26.300**	**13.513**	**0.508**

Note: Includes "below detection" as 0.0.

Table 14.7: Sample Assay Statistics – MS

Zone	Count	Min	Max	Mean	Median	Std. Dev.	CV
1	4,937	0.0	110,845	12,470.3	2,603.3	18,893.7	1.515
2	784	0.0	96,588	8,428.5	4,347.0	11,561.5	1.372
3	1,384	0.0	121,255	15,913.0	4,932.8	21,005.2	1.399
4	917	0.0	81,295	4,126.1	623.0	9,091.9	2.204
5	2,889	0.0	129,847	7,211.7	1,919.9	12,156.0	1.686
6	955	0.0	132,846	25,966.9	21,140.0	27,612.6	1.063
7	526	0.0	75,110	8,974.7	1,476.5	14,165.4	1.578
8	1,382	0.0	119,943	17,468.3	9,316.9	20,992.0	1.202
All	**13,772**	**0.0**	**132,846**	**12,141.0**	**3,128.7**	**18,671.9**	**1.538**

14.4 Geological and Structural Models

The mineral occurrences at SDS are IOCG deposits hosted in a package of Upper Jurassic to Lower Cretaceous andesite flows, volcaniclastic rocks, limestones, and clastic sedimentary rocks. Intrusion of diorite stocks and plugs led to the development of Cu-Fe oxide mantos and skarn bodies. The mantos are generally stratabound, and constrained to horizons of andesite tuff and reworked volcaniclastic sediments. The host rocks dip moderately to shallowly, and have been disrupted by numerous faults. The faults are believed to have channelled flow of mineralizing solutions, and later displaced blocks of the deposits. Late dykes also disrupt the mineralization.

Mineralization consists of disseminated, banded, and massive magnetite and specular hematite mantos with chalcopyrite and pyrite, as well as minor amounts of pyrrhotite, bornite, and chalcocite. Supergene processes have resulted in the development of oxide facies in some areas of the property.

For the purpose of the resource estimates, the SDS deposit has been modelled as four primary structures (Zones 1 to 4) which represent a 150 to 500 m thick copper-bearing specularite and magnetite manto sequence in an area approximately 1,300 m (north-south)

by 800 m (east-west). Several interpreted faults serve to constrain the mineralization on the eastern, western, and southern extents, as well as dividing SDS into three distinct fault blocks. The Iris deposit is located immediately to the east of the SDS deposit, separated from SDS 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. Iris is further subdivided by an internal fault that separates a magnetite-rich zone to the west from the main deposit. The known extent of the Iris deposit is approximately 1,900 m (north-south) by 500 m (east-west). The manto sequence in this area is up to 250 m thick, dips about 25° towards the west, and plunges towards the north.

Iris Norte, which lies approximately 600 m north of Iris, appears to be a continuation of Iris, rotated and offset by a number of poorly-defined fault structures. 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 SDS 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, effectively cutting the deposit off to the north. Further drilling is required to fully delineate the fault structures in this area and to determine the full extent of the deposit to the west. The known extent of Iris Norte is about 1,600 m (north-south) by 500 m (east-west). The manto sequence is up to 300 m thick, and dips to the west and plunges to the north at shallow angles.

RPA constructed 3D wireframe or solid models and gridded surfaces of the mineralized zones, fault structures, and topography for use in constraining the block grade interpolations. The SDS and Iris zones required construction of wireframes for post-mineral dykes that cut through the mineralized zones. There are also some sequences of barren tuffs that were modelled. The wireframe outlines were copied from the 2009 models, and modified to honour the latest drilling. The principal controls were lithology and structure; however, in some places a nominal grade shell boundary was used. There was no rigorous grade cut-off for this boundary, as it was rarely needed, but as a general rule the cut-off was either a Cu grade of 0.15% Cu or an MS value of 15,000.

FWM geologists have defined a magnetite-rich zone (termed the Mag Zone) which occupies the core of Zones 1, 2, and 3 in SDS. Surrounding the Mag Zone is relatively more hematite-rich Fe mineralization (the Hematite Rim). The MS values within the Mag Zone tend to be markedly higher than for the Hematite Rim. Consequently, a separate wireframe model for the Mag Zone was constructed and then used to constrain the grade interpolation for MS.

A wireframe model was also created to enclose oxidized material. This was a very preliminary model owing to the lack of a complete data set for leachable Cu. The primary criteria for defining the base of the oxidized zone was the presence of significant quantities of leachable Cu, or strong oxidation noted in the logs. The oxide model was not used to constrain the grade interpolation. It was used to make some estimates of the volume within the deposit that might be considered to be oxide.

14.5 Assay Capping (Cutting)

RPA notes that the sample grade distributions for Cu and Au are positively skewed, in some cases resembling lognormal distributions. For skewed distributions of this type, the highest-

grade samples can have an inordinately large effect on the average grades, which can result in biased grade interpolations. In order to reduce the influence of these high samples, a cap is applied prior to compositing. For the 2009 estimate, RPA produced a series of lognormal probability curves for copper and gold within the interpreted zones to examine the distribution of the assay data (Lacroix, 2009). The distribution curves for SDS and Iris exhibited breaks or inflection points at about 3.5% Cu and 0.52 g/t Au, indicating several distinct populations for each metal.

In total, 24 Cu and 27 Au assay intervals were capped. These intervals represent approximately 0.2% of the total number of assays. The net impact of the capping was to reduce the average Cu and Au assay grades by a negligible amount. Table 14.8 and Table 14.9 provide summaries of capping levels applied to the SDS, Iris, and Iris Norte data. Samples were capped prior to compositing.

Table 14.8: Assay Capping Levels – %Cu

Zone	Cap	# SDs from Mean	Population Maximum	# Capped	Avg. Before	Avg. After
SDS (1-4)	3.50	5.8	5.38	24	0.454	0.452
Iris (5-6)	3.50	10.2	3.34	0	0.192	0.192
Iris Norte (7-8)	3.50	10.6	3.10	0	0.171	0.171
Totals (1-8)	**3.50**	**6.7**	**5.38**	**24**	**0.342**	**0.341**

Note: Includes “below detection” as 0.0.

Table 14.9: Assay Capping Levels – g/t Au

Zone	Cap	# SDs from Mean	Population Maximum	# Capped	Avg. Before	Avg. After
SDS (1-4)	0.52	5.9	2.38	19	0.063	0.062
Iris (5-6)	0.52	5.3	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**	**5.9**	**4.71**	**27**	**0.047**	**0.047**

Note: Includes “below detection” as 0.0.

14.6 Composites

For SDS, Iris, and Iris Norte, assay intervals have been composited on the basis of hanging wall and footwall contacts determined by the application of the above-mentioned external cut-off grades. Samples were composited in downhole intervals of four meters, starting at the contact for each zone, and continuing until the hole leaves 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 only 77 out of a total of 6,642 composites. Since their impact was considered to be insignificant, they were left in the database. The 4 m composite length was deemed most suitable, as it was an exact multiple of the most common assay sample interval of two meters, 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, while 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 have been assigned unique numeric codes to differentiate them from the surrounding material. A summary of composite statistics is provided in Table 14.10, Table 14.11, Table 14.12, and Table 14.13.

Table 14.10: Composite Statistics – %Cu

Zone	Count	Min	Max	Mean	Median	Std. Dev.	CV
1-4	3,784	0	3.22	0.435	0.335	0.416	0.957
5-6	1,870	0	2.07	0.176	0.062	0.262	1.486
7-8	987	0	2.01	0.169	0.059	0.268	1.581
All	**6,642**	**0**	**3.22**	**0.323**	**0.185**	**0.381**	**1.182**

Table 14.11: Composite Statistics – g/t Au

Zone	Count	Min	Max	Mean	Median	Std. Dev.	CV
1-4	3,784	0	0.490	0.060	0.045	0.058	0.967
5-6	1,870	0	0.487	0.025	0.010	0.042	1.675
7-8	987	0	0.330	0.021	0.010	0.036	1.685
All	**6,642**	**0**	**0.490**	**0.044**	**0.025**	**0.054**	**1.219**

Table 14.12: Composite Statistics – %Fe

Zone	Count	Min	Max	Mean	Median	Std. Dev.	CV
1-4	3,784	0	63.188	26.885	27.710	12.399	0.461
5-6	1,820	0	62.813	24.764	23.706	11.174	0.451
7-8	987	0	60.341	23.254	22.303	12.726	0.547
All	**6,642**	**0**	**63.188**	**25.749**	**25.654**	**12.196**	**0.474**

Table 14.13: Composite Statistics – MS

Zone	Count	Min	Max	Mean	Median	Std. Dev.	CV
1-4	3,784	0	117,341.7	11,525.8	4,110.7	16,239.1	1.409
5-6	1,870	0	127,708.5	11,933.9	4,673.8	18,189.2	1.524
7-8	987	0	101,551.3	14,759.1	8,302.6	17,603.5	1.193
All	**6,642**	**0**	**127,708.5**	**12,119.6**	**4,716.7**	**17,049.8**	**1.417**

14.7 Block Model and Grade Estimation Procedures

The block size for SDS, Iris, and Iris Norte is 25 m east-west, 25 m north-south and 12 m high. Each block located at least partially within an interpreted zone was assigned a zone code, percent within the zone, 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, as well as the combined percentage of all zones contained within the block. Grades were estimated only for those blocks falling at least partially within one of the eight interpreted envelopes.

Integer codes were assigned to the blocks according to the zone with the highest proportion of material contained within the block. The rock codes are listed in Table 14.14.

Table 14.14: Domain Rock Codes

Code	Zone	Sub-Zone
1	SDS	Hematite Rim
2	SDS	Hematite Rim
3	SDS	Hematite Rim
4	SDS	-
5	Iris	-
6	Iris Mag	-
7	Iris Norte	-
8	Iris Norte	-
1001	SDS	Mag (Zone 1)
1002	SDS	Mag (Zone 2)
1003	SDS	Mag (Zone 3)

14.8 Variogram Models

For the 2009 estimate, SW RPA conducted a geostatistical analysis in order to define kriging models and search parameters. This analysis was carried out using MEDS software. For the update, the variogram analyses were rerun, using Sage software. Both MEDS and Sage are commercially available off-the-shelf packages.

Other than for the “vertical” orientation, which was basically downhole, the variograms contained very few pairs for lag distances less than 100 m. This is primarily due to the regular drill spacing within the deposit, which rarely falls below 100 m in plan. This posed some difficulty in developing models that were consistent with the interpreted geology. In some cases, the variography appeared to be driven more by drill hole orientation than by geology. When this occurred, it was necessary to force the variogram model to conform to the strike and dip of the host geologic units. More closely-spaced drilling within the core of the deposit would aid in refining these models and help in the design of future drilling programs. The variogram models developed and used by RPA are summarized in Table 14.15.

Table 14.15: Variogram Models

Metal/Zone	Model Type	Nugget	C	Total Sill	Rotation (MEDS)			Range (m)		
					Z	X'	Y'	Major	Semi	Minor
Copper										
SDS (1-4)	Sph	0.200	0.800	1.000	62	9	20	150.0	150.0	30.0
Iris/Iris Mag	Sph	0.220	0.780	1.000	56	23	22	160.0	100.0	30.0
Iris Norte	Sph	0.350	0.650	1.000	119	6	6	136.0	136.0	24.0
Iron										
SDS (1-2)	Exp	0.250	0.750	1.000	62	9	20	162.0	162.0	27.0
SDS (3-4)	Exp	0.300	0.700	1.000	62	9	20	132.0	132.0	30.0
Iris/Iris Mag	Exp	0.200	0.800	1.000	56	23	23	132.0	132.0	27.0
Iris Norte	Exp	0.250	0.750	1.000	119	6	5	180.0	180.0	30.0
Gold										
SDS (1-2)	Sph	0.320	0.680	1.000	62	9	20	180.0	180.0	26.0
SDS (3-4)	Sph	0.450	0.550	1.000	62	9	20	165.0	165.0	28.0
Iris/Iris Mag	Sph	0.240	0.760	1.000	56	23	-2	180.0	100.0	34.0
Iris Norte	Sph	0.400	0.600	1.000	119	6	5	135.0	135.0	20.0
Mag Sus										
SDS Mag	Sph	0.140	0.434	0.574	-31	1	64	63.7	63.4	15.3
	Sph		0.171	0.745	-63	63	-27	472.5	267.3	137.3
SDS Non-Mag	Sph	0.150	0.850	1.000	62	9	20	180.0	180.0	34.0
Iris/Iris Mag	Sph	0.180	0.820	1.000	56	23	22	132.0	132.0	40.0
Iris Norte	Exp	0.320	0.680	1.000	119	6	6	124.0	124.0	22.0

Notes: MEDS Rotations: ZXY LRL. 1st rotation is about Z axis (+ is clockwise). 2nd rotation is about new X axis (- is down). 3rd rotation is about new Y axis (- is down).

14.9 Grade Interpolation

Grades for Cu, Au, Fe, and MS were interpolated into each block using OK. Block estimates for each zone were constrained to use only composites from that zone. For MS, the grade interpolation was also configured to discriminate between composites and blocks inside and outside of the Mag Zone. The Mag Zone constraint was not applied for the Au, Cu, or Fe estimates.

The interpolation was configured to use an ellipsoidal search, with a minimum of three and a maximum of 18 composites, and a maximum of three composites allowed from any one drill hole. Grades were estimated in two passes, the first at twice the copper variogram range, and the second using distances equal to the variogram ranges. Although unique variograms were developed for each component, the search was made consistent for all to ensure that if a block received an estimate for one component, it was estimated for all. The ellipsoids were oriented parallel to the copper variogram models.

14.10 Bulk Density

Bulk density, used to convert volumes to weights, was based on regression formulas developed for the 2009 estimate by RPA using Fe as the independent variable. The regression formula used is as follows:

SDS/Iris/Iris Norte: Density = 2.5 + 0.025*Fe

This equation was derived from wet and dry measurements of core from over 1,900 samples. In this method, density is calculated as the weight of the dry sample in air divided by the difference between the weight of the dry sample in air and the weight of the sample suspended in water. It should be noted that core was only air-dried prior to weighing and not sealed for immersion, potentially introducing a positive bias proportional to the surface porosity (the amount of water absorbed by the specimen). RPA observed that some of the more magnetite-rich sections of core appeared to have a higher porosity than the surrounding rock mass. Under extreme circumstances, this might bias the density measurements, although at this time it is not viewed as a significant concern. It is recommended that rock specimens that exhibit excessive porosity be excluded from the density tests unless they are first properly sealed.

Since that time, additional density measurements have been taken, bringing the total to 2,229. Also, some of the original Fe assays have been rerun using the ME-ICP81 protocol, which will likely have resulted in higher overall Fe values. RPA reran the regression analysis and obtained the following relationship:

Density = 2.49 + 0.0264*Fe

This formula generates estimated densities that are very similar to those produced by the 2009 formula. Consequently, in order to provide a simpler basis for comparison of the two estimates, the density equation was not changed.

The bulk density values were calculated for each block based on the interpolated Fe grade and the above-stated formula. The average density for SDS, Iris, and Iris Norte was estimated at 3.12 t/m^3 for Indicated resources and 3.05 t/m^3 for Inferred resources. While it is recommended that samples be dry when weighed and sealed before immersion, in RPA's opinion, the overall difference is not considered material to the Mineral Resource estimates contained in this report.

14.11 Magnetic Susceptibility and Mass Recovery

The MS readings were used to estimate the proportion of the mass of each block that could be recovered by LIMS. FWM has conducted Davis Tube test work in order to, first, determine if a saleable Fe concentrate can be produced and, secondly, to calibrate the expected Mass Recovery (MR) to MS. In 2008, a bulk sample was collected by blending drill cuttings from a number of holes in SDS and Iris. This sample was subject to bulk flotation to remove the sulphide components, and the tails from this process were subject to iron recovery testing. The results of the test work indicated that LIMS would produce a good-quality magnetite iron concentrate.

The iron mineralization at SDS is dominantly magnetite, which can be recovered by LIMS, and hematite, which generally cannot. Consequently, the assays for total iron collected to date do not provide a basis for estimation of the recoverable iron component. This is demonstrated in Figure 18-1. Over 10,000 iron values are plotted against corresponding MS measurements taken from laboratory rejects. The MS value provides an indication of the proportion of magnetite within each sample, and it can clearly be seen that there is no relationship with total iron content. The MS is bound by an upper limit representing the case where virtually all the iron present in the sample is magnetite.

Davis Tube, Satmagan, and MS 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 MS and both Satmagan and Davis Tube mass recovery readings. FWM subsequently embarked on an expanded testing program in order to confirm the observed relationship and develop a reliable regression line equation for relating MS to MR. The results of this test work are shown in Figure 14-1.

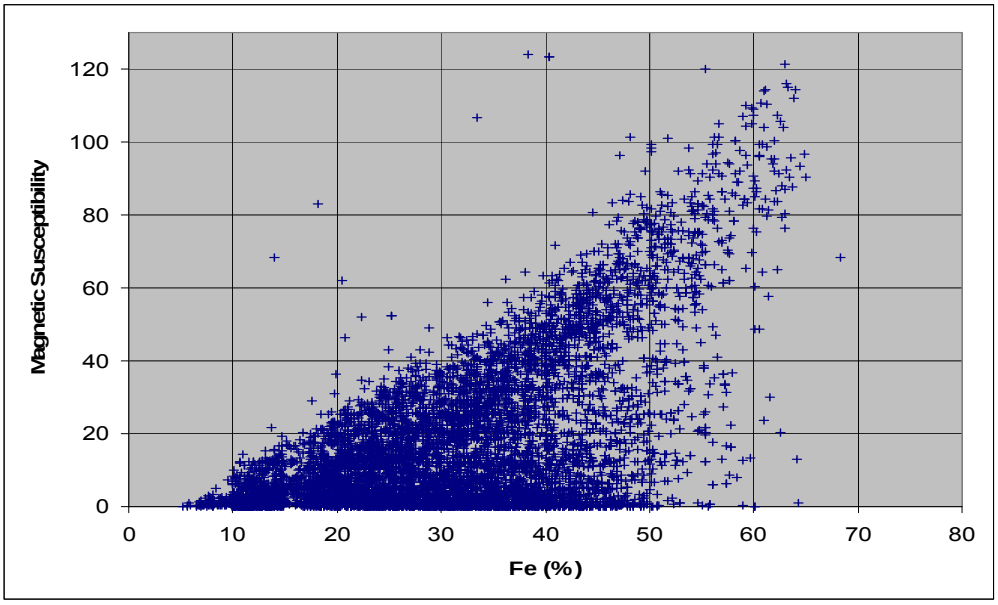


Figure 14-1: Total Iron vs. Magnetic Susceptibility

There is a degree of scatter in the data points plotted in Figure 18-2. In order to model the MR with MS, FWM chose an equation which skirts along the bottom of the point distribution (see magenta line in Figure 14-2). The relationship thus derived is:

MR% = (1.1063 x MS) + (-0.003 x MS2)

This line represents the minimum MR observed for the corresponding MS values, and in RPA's opinion, will result in conservative estimates of MR. The equation shown above was applied to the interpolated block MS values to estimate block MR in percent. The percentage of the block that can be recovered (as magnetite) by means of a LIMS process can then be

used to estimate the metal value contributed by iron. The MR was assumed to be zero for all blocks grading less than 15% Fe.

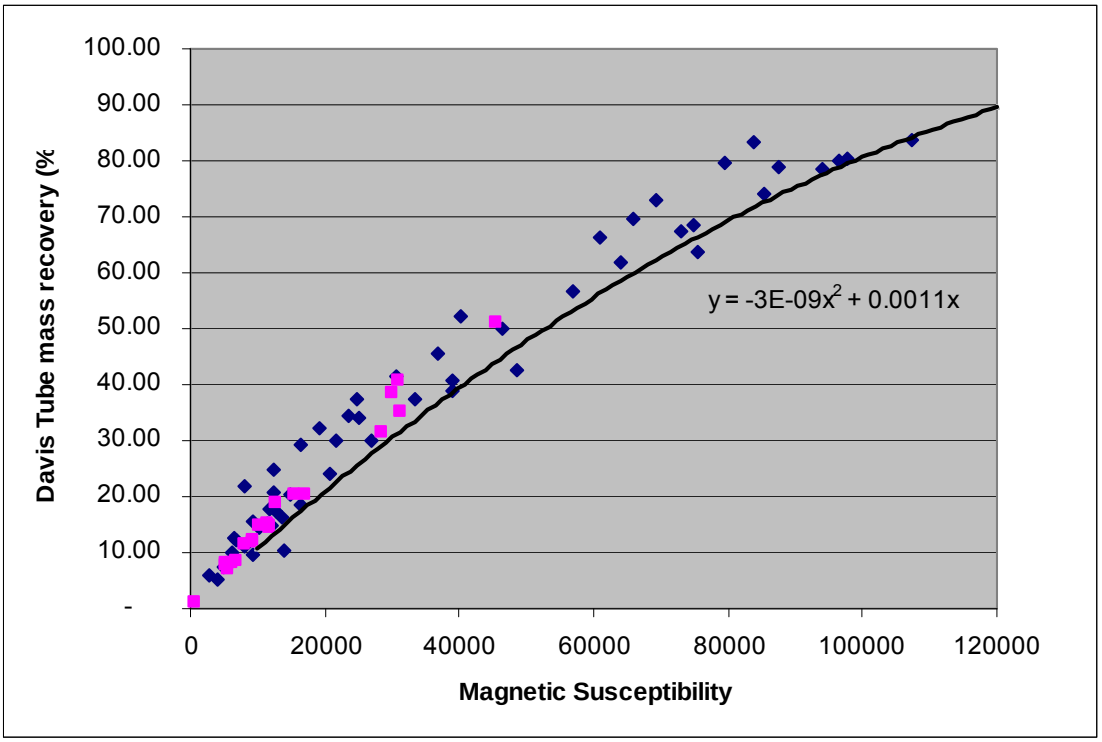


Figure 14-2: Magnetic Susceptibility vs. Mass Recovery

14.12 Cut-off Grade

The deposit is polymetallic, with significant values in copper, iron, and gold. As indicated previously, results from a preliminary assessment of SDS and Iris (AMEC, 2008), coupled with recent metallurgical test work for copper, gold and iron, indicate that all three of these metals will have measureable economic contributions to the project. For this reason, a copper equivalent grade (CuEq) has been derived which recognizes the potential contributions of all economic components.

The methodology for calculation of CuEq was derived by RPA for the 2009 estimate and remains unchanged for this estimate. CuEq grades were calculated using estimates for recovery, treatment/refinement charges (TC/RC), and transport costs for each metal and based on the operating cost estimates contained in the 2008 preliminary assessment (AMEC, 2008). Parameters used in the calculations are listed in Table 14.16. The formula for the CuEq calculation is provided below (Lacroix, 2009):

Metal Value = Grade*C_m*R%/100*(Price-TCRC-Freight)*(100-Royalty)/100

Where: C_m is a constant to convert grade of metal m to metal price units

R is metallurgical recovery

%Cu Equivalent = (Cu Value + Au Value + Fe Value)/(Cu Value per 1%Cu)

Table 14.16: Parameters for Cu Equivalent

Far West Mining Ltd. – Santo Domingo Property					
Metal	**Price (US$)**	**Recovery %**	**TC/RC (US$)**	**Freight (US$4.00/wmt)**	**Royalty NSR**
Copper	2.25/lb Cu	85	0.24/lb Cu	0.007/lb Cu	2%
Gold	950/oz Au	65	6/oz Au	-	2%
Iron	0.74/dmtu Fe	0 to 35	-	6.40/dmt	2%

Notes:

1. Iron recovery is based on MS, as explained in the text of this report.
2. Freight is based on cost per wmt of concentrate @ 8% moisture.
3. US$0.74/dmtu Fe metal is approximately equivalent to $50/dmt concentrate @ 67.5% Fe.

At the request of FWM, RPA used a cut-off of 0.25% CuEq for reporting the Mineral Resources. Based on the costs developed in the preliminary assessment coupled with the parameters for copper detailed in Table 17-16, the break-even cut-off grade is calculated as follows:

Cut-off %CuEq = (Unit Ore Cost – Unit Waste Cost)/Value/1% Cu

Where Ore Cost = US$5.68/t

Waste Cost = US$1.19/t

Value/1% Cu = 22.05/%*85/100*(2.25-0.24-0.007)*(100-2)/100

Cut-off = (5.68-1.19)/36.79=**0.122% Cu equivalent**

Consequently, in RPA's opinion, the cut-off used is conservative.

14.13 Pit Optimizaton

In order to satisfy the requirements of NI 43-101 that a resource has reasonable prospects of economic extraction, RPA evaluated the resource using pit optimization software (Lerchs-Grossmann algorithm). The pit shell was generated using the following parameters:

Wall Slope:	45 degrees
Mining Cost:	$1.19/t
Processing Cost:	$4.49/t
Processing Recovery:	85%
Selling Price:	$2.25/lb
Selling Cost:	$0.247/lb

RPA ran an optimization based on the combined Indicated and Inferred Mineral Resources. At the 0.25% CuEq cut-off, all but 5% of the Mineral Resources were captured by the pit shell. On the basis of these findings, it is RPA's opinion that there is little merit in restricting the Mineral Resources to those contained only within the pit shell. Accordingly, the Mineral Resource inventory has been reported in its entirety.

14.14 Model Validation

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 Inverse Distance Cubed (ID^3) blocks estimates to composite grades
- Estimation using an alternative method

Inspection of the block grades in section views indicates that there is reasonably good agreement with the drillhole grades. See Appendix 3 for example section views of the block model.

Table 14.17 compares the mean block and composite grades for all components in each of the interpolation domains. The blocks included in this comparison are those estimated in Pass 2, which used the smallest search, and should therefore represent the best informed blocks. For the most part, there is good agreement between block and composite means. Some notable exceptions are the Au in Zones 1001, 1002, and 8; Cu in Zones 1001, 1002, and 8; and MS in Zone 3. The global mean data suggests that the estimates could be conservative for Au and Cu and overstated for MS in the affected domains.

Table 14.17: Comparison of Block vs. Composite Global Means

Far West Mining Ltd. – Santo Domingo Property									
Au					Cu				
Zone	Blocks	Comps	Diff.	Pct. Diff.	Zone	Blocks	Comps	Diff.	Pct. Diff.
1	0.065	0.063	0.002	2.7%	1	0.475	0.459	0.016	3.6%
1001	0.041	0.063	-0.022	-34.6%	1001	0.297	0.459	-0.162	-35.3%
2	0.039	0.040	-0.001	-2.1%	2	0.293	0.287	0.006	2.1%
1002	0.031	0.040	-0.009	-21.9%	1002	0.161	0.287	-0.126	-43.9%
3	0.052	0.051	0.001	2.1%	3	0.390	0.368	0.023	6.1%
1003	0.045	0.051	-0.006	-11.5%	1003	0.315	0.368	-0.053	-14.3%
4	0.068	0.068	0.001	0.8%	4	0.515	0.504	0.011	2.3%
5	0.031	0.030	0.001	4.3%	5	0.219	0.208	0.011	5.2%
6	0.013	0.012	0.001	6.9%	6	0.087	0.087	0.001	0.6%
7	0.028	0.032	-0.004	-11.4%	7	0.257	0.270	-0.013	-5.0%
8	0.013	0.017	-0.004	-22.4%	8	0.102	0.130	-0.028	-21.6%
Fe					MS				
Zone	Blocks	Comps	Diff.	Pct. Diff.	Zone	Blocks	Comps	Diff.	Pct. Diff.
1	25.35	27.08	-1.732	-6.4%	1	6,142.4	5,838.3	304.1	5.2%
1001	28.44	27.08	1.360	5.0%	1001	22,842.1	22,458.8	383.3	1.7%
2	18.97	19.61	-0.641	-3.3%	2	7,455.9	7,429.6	26.3	0.4%
1002	22.76	19.61	3.143	16.0%	1002	16,563.8	18,082.9	-1,519.1	-8.4%
3	26.56	29.27	-2.713	-9.3%	3	7,000.9	5,480.2	1,520.7	27.7%
1003	29.03	29.27	-0.242	-0.8%	1003	19,212.7	17,815.4	1,397.3	7.8%
4	27.33	27.47	-0.143	-0.5%	4	4,159.8	4,031.1	128.8	3.2%
5	22.18	23.82	-1.640	-6.9%	5	8,222.7	8,673.9	-451.2	-5.2%
6	25.93	27.40	-1.473	-5.4%	6	27,701.7	30,125.5	-2,423.8	-8.0%
7	26.28	24.89	1.397	5.6%	7	11,620.2	10,086.4	1,533.8	15.2%
8	23.20	22.61	0.584	2.6%	8	19,502.8	19,022.6	480.2	2.5%

In RPA's opinion, the swath plots showed good agreement between the composite grades and the interpolated block grades.

Block grades were interpolated using ID^3 weighting in order to provide a comparison with the OK estimates. The unclassified block model results from the two estimation methods are shown in Table 14.18. There is a negligible difference in both tonnage and grade between the two block models in the range of cut-offs around 0.25% CuEq.

Table 14.18: Comparison of OK and ID3 Estimates

Far West Mining Ltd. – Santo Domingo Property					
% CuEq	Ordinary Kriging				
Cut-off	Kt	% Cu	g/t Au	% Fe	MS
1.500	3,014.86	1.142	0.143	35.60	27,653.1
1.000	46,273.75	0.712	0.095	34.16	26,577.6
0.750	135,316.93	0.532	0.071	32.62	24,643.8
0.500	334,062.90	0.382	0.052	29.69	20,837.8
0.450	384,873.57	0.358	0.048	29.01	20,038.2
0.400	438,004.00	0.337	0.046	28.30	19,164.3
0.350	490,570.49	0.318	0.043	27.61	18,342.3
0.300	536,995.77	0.305	0.041	27.05	17,503.7
0.250	578,487.88	0.292	0.040	26.51	16,805.4
0.200	612,479.42	0.281	0.038	26.06	16,265.6
0.150	639,802.99	0.272	0.037	25.67	15,778.2
0.100	662,157.92	0.265	0.036	25.37	15,369.4
0.050	678,726.14	0.259	0.035	25.12	15,040.8
0.001	695,321.64	0.254	0.035	24.92	14,688.9
% CuEq	Inverse Distance Cubed				
Cut-off	Kt	% Cu	g/t Au	% Fe	MS
1.500	6,116.48	1.195	0.157	37.48	29,528.9
1.000	51,611.60	0.737	0.100	35.60	28,624.2
0.750	137,970.89	0.550	0.075	33.67	25,535.1
0.500	334,298.98	0.394	0.053	30.25	20,983.6
0.450	382,299.02	0.369	0.050	29.56	20,176.1
0.400	432,309.12	0.347	0.047	28.85	19,369.3
0.350	478,919.59	0.329	0.045	28.18	18,588.8

% CuEq	Inverse Distance Cubed				
Cut-off	Kt	% Cu	g/t Au	% Fe	MS
0.300	524,561.77	0.313	0.043	27.56	17,757.7
0.250	565,711.20	0.299	0.041	26.97	17,047.8
0.200	600,258.95	0.287	0.039	26.46	16,457.3
0.150	630,130.73	0.277	0.038	26.02	15,901.3
0.100	655,796.25	0.268	0.037	25.64	15,409.0
0.050	676,737.14	0.261	0.036	25.29	14,987.8
0.001	695,200.17	0.254	0.035	25.05	14,598.1
% CuEq	Percent Difference				
Cut-off	Kt	% Cu	g/t Au	% Fe	MS
1.500	102.9%	4.7%	9.6%	5.3%	6.8%
1.000	11.5%	3.4%	5.4%	4.2%	7.7%
0.750	2.0%	3.5%	4.6%	3.2%	3.6%
0.500	0.1%	3.1%	3.6%	1.9%	0.7%
0.450	-0.7%	3.2%	3.6%	1.9%	0.7%
0.400	-1.3%	2.9%	3.2%	2.0%	1.1%
0.350	-2.4%	3.3%	3.4%	2.1%	1.3%
0.300	-2.3%	2.8%	3.0%	1.9%	1.5%
0.250	-2.2%	2.4%	2.6%	1.7%	1.4%
0.200	-2.0%	2.2%	2.4%	1.6%	1.2%
0.150	-1.5%	1.7%	1.8%	1.3%	0.8%
0.100	-1.0%	1.2%	1.3%	1.0%	0.3%
0.050	-0.3%	0.6%	0.7%	0.7%	-0.4%
0.001	0.0%	0.3%	0.5%	0.5%	-0.6%

Notes:

1. The block model summaries are unclassified and do not represent Mineral Resources estimates for the deposit. They are provided here for comparison purposes only.

14.15 Classification

CIM definitions (December 11, 2005) were followed for the classification of the Mineral Resources. Blocks receiving an estimate for Cu were assigned to at least the Inferred category. All blocks with an average distance to composites of 200 m or less and for which the nearest composite was within 100 m were updated to Indicated. A total of 84.8% of the Mineral Resources for SDS, Iris, and Iris Norte are in the Indicated category. The Inferred Mineral Resources are generally located at the margins of the deposit, where the drill density is lowest. For example, much of the Inferred material is located on the east and west flanks of Iris Norte, and in the deepest portions of SDS. Additional drilling would be needed to upgrade the Inferred to Indicated.

There were no Measured Mineral Resources.

14.16 Oxide Material

The upper portions of the deposit are oxidized to some extent, as evidenced by limited assaying for leachable copper and the presence in the drillholes of copper oxides. The oxide layer typically extends to a depth of about 80 m from surface, with deeper penetration along faults in places. RPA constructed a preliminary wireframe model of this zone to provide a rough means for determining the volume of oxidized material. It is a very preliminary interpretation based on incomplete data. The wireframe was not used to constrain or modify the grade interpolation in any way, and it was only used to tabulate volumes. The tonnage and grade of the Mineral Resources contained within this wireframe are summarized in Table 14.19.

Table 14.19: Minerals Resources in the Oxide Zone

Far West Mining Ltd. – Santo Domingo Property				
Category	**Mt**	**% Cu**	**g/t Au**	**% Fe**
Indicated	33.7	0.39	0.05	27.4
Inferred	2.61	0.25	0.03	25.0

In RPA's opinion, the copper recovery in the oxide zone may not be as good as for the rest of the deposit. The overall proportion of the mineral resources in this zone is fairly small (i.e., about 7% of the Indicated category) and so, if the copper recovery is not as high as the rest of the deposit, the effect on the overall project economics is not expected to be overly severe. However, the oxide layer is at surface and, as such, will be among the first material mined. This may have an impact on the early production results for the operation. RPA recommends that additional test work be undertaken to determine the metallurgical characteristics of the material in the oxide zone. It is also recommended that additional geological interpretation be carried out to improve and confirm the geometry of the oxide layer. This may have to include additional assaying and/or relogging of some of the holes. Once the metallurgical characteristics of this material have been determined, it will likely be necessary to include a factor in the block model to account for the oxides.

15 MINERAL RESERVE ESTIMATE

15.1 Mineral Reserves - Summary

Table 15.1 shows a summary of the Santo Domino reserve estimation.

Table 15.1: Santo Domingo Open Pit Mineable Probable Reserves (August 15, 2011)

		Ore Grade		Contained Metal		
Stage	**Ore (Mt)**	**Au (g/t)**	**Cu (%)**	**Au (Koz)**	**Cu (Mlbs)**	**Magnetite Conc. (Mt)**
SDS/Iris						
SDS Stage1	71.8	0.08	0.61	193	958	11
SDS Stage2	63.7	0.06	0.41	113	574	10
SDS Stage3	170.5	0.03	0.23	173	848	32
SDS Stage4	38.8	0.05	0.36	60	304	3
Subtotal SDS/Iris	**344.8**	**0.05**	**0.35**	**539**	**2,684**	**57**
Iris Norte						
IRN Stage 1	21.4	0.03	0.23	20	108	4
IRN Stage 2	28.0	0.01	0.13	12	78	7
IRN Stage 3	23.7	0.01	0.11	8	60	5
Subtotal Iris Norte	**73.1**	**0.02**	**0.15**	**41**	**246**	**17**
Grand Total	**418.0**	**0.04**	**0.32**	**580**	**2,930**	**73**

Note: NSR cut-off of $5.79/t . Note: Ore includes Indicated Resources only. Total Waste Mined includes Inferred Material, Ovb, Rock, and Material below cut-off. Magnetite concentrate tonnage based on average 65% iron grade.

15.2 Mineral Reserves – Open Pit

15.2.1 Net Smelter Model

The 3-D resource models, developed by RPA, were used as the basis for deriving the economic pit limits for the Santo Domingo pits. The models encompassed Santo Domingo Sur, Iris and Iris Norte deposits. A number of calculations were performed on the models in order to determine the net smelter return ("NSR") of each individual block. The parameters used in the calculations are summarized in Table 15.2 below.

Table 15.2: Open Pit NSR Parameters

Item	Unit	Value
Metal Prices		
Copper	US$/lb	2.50
Gold	US$/oz	1000.00
Iron	US$/dmtu	1.00
Recovery to Cu Concentrate		
Copper	%	Head grade - 0.04%
Copper fixed tail grade	%	0.04
Gold	%	Cu recov * 0.7
Magnetite Iron	%	variable with MS and MR
Cu Concentrate Grade		
Copper	%	29.0
Gold	g/t	calculated
Moisture content	%	8.0%
Fe Concentrate Grade		
Iron	% Fe	65.0
Moisture content	%	8.0%
Smelter Payables		
Copper in Cu concentrate	%	100.0
Copper deduction in concentrate	units	1.0
Payable Copper	%	96.6
Gold in all concentrate	%	97.0
Gold deduction in all concentrate	g/t in concentrate	0
Off-Site Costs		
Cu concentrate treatment	US$/dmt conc	60.00
Cu refining charge	US$/lb pay Cu	0.060
Au refining charge	US$/oz pay Au	6.50
Shipping	US$/wmt conc Cu	50.60
Shipping	US$/wmt conc Fe	5.52
Operating Costs		
Waste mining Cost	US$/waste tonne	1.21
Ore Mining Cost	US$/ore tonne	1.21
Processing Cost	US$/milled tonne	5.02
G&A Cost	US$/milled tonne	0.77
Cost	US$/milled tonne	5.79

Item	Unit	Value
Average Overall Pit Slope Angles	Overburden	38
SDS/Iris	Sector1 South	55
	Sector 2 West	50
	Sector 3 North & East	42
Iris Norte	Overburden	38
	All sectors	42
Grade Factor (1-Dilution)	%	100%
Mining recovery	%	100%
Royalties	%	2%
Discount Rate	%	8.0

The NSR calculations allow for the accounting of:

- ore grades (Cu, Au, and Fe) thus taking into account the variability in the metal content of the deposit
- ore mill recoveries
- contained metal in concentrates
- deductions and payable metal value
- metal prices
- freight costs (both shipping and slurry transport)
- smelting and refining charges
- royalty charges.

15.2.2 Economic Pit Limit

The ultimate economic open pit limits are based on Whittle™ pit optimization evaluations of the resources in the NSR models. This evaluation included the aforementioned NSR calculations as well as geotechnical parameters (as provided by AMEC), mining dilution and recoveries, and mining/milling/G&A costs. The economic pit limits have been constrained to only consider indicated resource class material (no measured resources have been classified and inferred resources have been excluded and treated as waste).

Optimization Parameters and Results

The geotechnical parameters, dilution/recovery, mining, milling and G&A costs (based on an assumed maximum mill throughput of 70 ktpd) are summarized in Table 15.2 above.

The open pit mining activities for the Santo Domingo pits were assumed to be undertaken by an owner-operator mining fleet for this pre-feasibility study. The owner-operator mining cost unit rate used in the Whittle™ optimization was US$1.21 per tonne of material for pit and dump operations, road maintenance, mine supervision, technical services and senior management costs. The mining unit rate was calculated based on equipment required to

achieve a processing rate of 70 ktpd. Mining costs were developed from first principles for similar sized operations, along with estimates for local labour, fuel and power costs.

The estimated existing topography was used as the starting surface for the pit optimization. The internal (or mill) cut-off grade of $5.79/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 Whittle™ optimization. The various mill cut-offs were applied to all of the mineral resource estimates that follow.

A series of Whittle™ pit shells were generated based on varying revenue factors and the results analyzed with pit shells chosen as the basis for further design work and preliminary phase designs for each of the deposits of Santo Domingo, Iris and Iris Norte.

The reserves within the various pit shells were generated from the following 3-D block model items:

- block centroid coordinates
- copper grade
- gold grade
- iron grade
- class (indicated only)
- topography percentage
- ore percentage
- oxide interface
- specific gravity.

The results of the Whittle™ pit optimization evaluation for varying revenue factors are summarized in Table 15.3 and

Table 15.4, as well as Figure 15-1 and Figure 15-2 for indicated resources only. The selected Whittle™ shell (based on an evaluation of the results) used as the basis for the detailed pit designs is highlighted in each of the tables.

Table 15.3: Santo Domingo Sur/Iris Put Optimization Results

Final	Revenue	Mine	Ore Insitu	Insitu Grades				Contained Metal			Waste	Strip	Total	Total CF	NPV Best	NPV Worst
Pit	Factor	Life	(ktonnes)	Au (g/t)	Cu (%)	Fe Mag(%)	NSR (US$/t)	Au (koz)	Cu (Mlbs)	Fe Mag (Mlbs)	(ktonnes)	Ratio	(ktonnes)	(C$)	$ disc	$ disc
1	0.30	3.0	76,507,564	0.08	0.61	9.54	36.87	206	1,026	16,099	158,923,952	2.08	235,431,517	2,092,924,147	1,782,775,739	1,782,775,739
2	0.32	3.8	97,208,689	0.08	0.58	9.82	35.90	250	1,252	21,046	214,352,426	2.21	311,561,115	2,549,879,072	2,122,668,884	2,112,413,546
3	0.34	4.2	107,356,054	0.08	0.57	10.03	35.20	267	1,341	23,738	235,823,165	2.20	343,179,219	2,741,803,020	2,257,142,900	2,240,711,382
4	0.36	4.6	117,926,094	0.08	0.55	10.26	34.68	286	1,435	26,663	266,691,217	2.26	384,617,310	2,942,018,569	2,395,230,901	2,367,476,915
5	0.38	5.0	129,185,709	0.07	0.53	10.28	33.82	305	1,522	29,286	288,620,981	2.23	417,806,691	3,116,154,353	2,507,313,674	2,464,672,485
6	0.40	5.8	148,118,388	0.07	0.51	9.86	32.37	335	1,674	32,193	317,940,160	2.15	466,058,547	3,372,370,453	2,670,589,445	2,607,618,243
7	0.42	6.4	165,071,846	0.07	0.49	9.89	31.30	358	1,786	35,981	348,206,926	2.11	513,278,771	3,589,508,749	2,801,337,399	2,722,757,979
8	0.44	6.9	176,740,266	0.07	0.48	9.96	30.66	372	1,857	38,813	371,185,774	2.10	547,926,040	3,731,988,783	2,881,984,546	2,790,153,515
9	0.46	7.3	185,880,430	0.06	0.47	9.77	30.18	386	1,927	40,052	390,279,058	2.10	576,159,487	3,835,632,868	2,940,136,069	2,834,426,518
10	0.48	8.8	224,550,305	0.06	0.42	10.58	28.49	416	2,089	52,368	482,944,326	2.15	707,494,630	4,241,041,768	3,146,145,927	2,956,552,888
11	0.50	9.1	233,979,697	0.06	0.41	10.65	28.13	425	2,133	54,930	504,904,347	2.16	738,884,044	4,334,127,259	3,189,533,428	2,981,780,062
12	0.52	10.1	259,063,853	0.05	0.39	11.24	27.26	440	2,200	64,195	564,461,854	2.18	823,525,706	4,566,514,653	3,295,179,926	3,032,524,022
13	0.54	10.5	267,961,703	0.05	0.38	11.38	27.01	447	2,232	67,220	591,013,014	2.21	858,974,717	4,647,597,597	3,330,722,487	3,056,479,542
14	0.56	10.7	274,634,951	0.05	0.38	11.34	26.84	455	2,271	68,637	614,080,445	2.24	888,715,396	4,705,817,580	3,354,996,767	3,066,548,967
15	0.58	11.1	283,059,368	0.05	0.37	11.28	26.54	464	2,310	70,366	632,490,173	2.23	915,549,540	4,765,951,301	3,378,982,653	3,071,519,148
16	0.60	11.3	290,517,235	0.05	0.37	11.35	26.46	472	2,351	72,709	674,473,127	2.32	964,990,362	4,836,304,208	3,408,232,254	3,082,385,360
17	0.62	11.5	294,992,399	0.05	0.36	11.35	26.35	477	2,373	73,787	692,237,390	2.35	987,229,789	4,869,538,201	3,421,456,798	3,085,105,889
18	0.64	11.6	297,733,908	0.05	0.36	11.31	26.24	479	2,386	74,211	697,547,455	2.34	995,281,363	4,883,538,293	3,426,781,858	3,083,622,816
19	0.66	12.4	318,503,986	0.05	0.36	11.00	25.74	506	2,522	77,229	786,643,838	2.47	1,105,147,824	5,018,429,549	3,476,840,931	3,088,608,043
20	0.68	12.8	328,035,854	0.05	0.36	10.96	25.60	518	2,581	79,281	841,819,666	2.57	1,169,855,520	5,083,292,767	3,499,393,837	3,088,930,390
21	0.70	12.9	329,521,979	0.05	0.36	10.94	25.56	519	2,589	79,476	846,646,424	2.57	1,176,168,403	5,090,359,472	3,501,658,459	3,086,452,572
22	**0.72**	**13.6**	**348,104,286**	**0.05**	**0.36**	**10.63**	**25.27**	**548**	**2,732**	**81,607**	**958,829,289**	**2.75**	**1,306,933,575**	**5,198,768,110**	**3,537,253,333**	**3,066,405,216**
23	0.74	13.7	350,405,863	0.05	0.35	10.62	25.20	550	2,742	82,056	967,132,260	2.76	1,317,538,123	5,208,255,377	3,540,056,164	3,064,300,278
24	0.76	13.7	351,014,472	0.05	0.35	10.61	25.18	551	2,745	82,133	968,897,759	2.76	1,319,912,231	5,210,317,153	3,540,634,671	3,063,456,145
25	0.78	13.8	352,210,418	0.05	0.35	10.60	25.15	552	2,751	82,345	974,210,731	2.77	1,326,421,149	5,214,631,382	3,541,855,699	3,061,819,045
26	0.80	13.9	354,574,345	0.05	0.35	10.60	25.10	554	2,761	82,887	987,166,229	2.78	1,341,740,574	5,223,089,405	3,544,216,612	3,056,289,249
27	0.82	14.0	359,311,889	0.05	0.35	10.57	25.00	560	2,787	83,735	1,016,010,840	2.83	1,375,322,729	5,238,726,029	3,548,615,319	3,047,293,909
28	0.84	14.1	361,766,086	0.05	0.35	10.54	24.95	563	2,802	84,048	1,031,140,096	2.85	1,392,906,182	5,245,995,505	3,551,047,840	3,041,773,458
29	0.86	14.3	365,725,559	0.05	0.35	10.51	24.88	567	2,826	84,719	1,058,865,597	2.90	1,424,591,156	5,256,697,309	3,554,555,017	3,033,582,943
30	0.88	14.5	371,942,215	0.05	0.35	10.51	24.66	572	2,838	86,153	1,076,563,972	2.89	1,448,506,187	5,266,697,063	3,557,669,583	3,031,018,233
31	0.90	14.6	373,860,034	0.05	0.35	10.49	24.63	574	2,850	86,493	1,092,130,015	2.92	1,465,990,050	5,270,819,734	3,558,943,684	3,023,810,822
32	0.92	14.6	374,982,287	0.05	0.35	10.48	24.60	575	2,854	86,674	1,095,879,854	2.92	1,470,862,141	5,272,040,063	3,559,296,134	3,022,288,700
33	0.94	14.7	376,300,861	0.05	0.34	10.48	24.55	576	2,857	86,952	1,100,555,019	2.92	1,476,855,880	5,273,223,842	3,559,626,357	3,021,429,554
34	0.96	14.9	380,350,387	0.05	0.34	10.44	24.52	582	2,888	87,554	1,145,178,808	3.01	1,525,529,195	5,277,655,507	3,560,876,717	3,002,562,862
35	0.98	14.9	381,795,772	0.05	0.34	10.43	24.51	585	2,899	87,807	1,162,012,878	3.04	1,543,808,650	5,278,606,679	3,561,115,055	2,995,392,818
36	1.00	14.9	382,574,192	0.05	0.34	10.43	24.48	585	2,901	87,949	1,165,633,970	3.05	1,548,208,162	5,278,708,926	3,561,112,634	2,993,445,682
37	1.02	15.0	384,291,282	0.05	0.34	10.42	24.46	587	2,911	88,291	1,181,972,207	3.08	1,566,263,489	5,278,431,983	3,560,960,862	2,986,057,648
38	1.04	15.0	384,908,840	0.05	0.34	10.42	24.44	587	2,913	88,414	1,186,878,860	3.08	1,571,787,700	5,278,126,443	3,560,863,086	2,983,800,002

Table 15.4: Iris Norte Pit Optimization Results

Final	Revenue	Mine	Ore Insitu	Insitu Grades					Contained Metal			Waste	Strip	Total	Total CF	NPV Best	NPV Worst
Pit	Factor	Life	(ktonnes)	Au (g/t)	Cu (%)	Fe (%)	Fe Mag(%)	NSR (US$/t)	Au (koz)	Cu (Mlbs)	Fe Mag(Mlbs)	(ktonnes)	Ratio	(ktonnes)	(C$)	$ disc	$ disc
1	0.58	1.7	44,419,633	0.02	0.18	27.16	14.96	20.33	29	176	14,647	175,596,417	3.95	220,016,050	379,552,031	339,427,011	339,427,011
2	0.60	1.9	47,854,387	0.02	0.18	27.07	14.95	20.10	31	185	15,772	184,386,122	3.85	232,240,509	403,743,564	358,257,449	357,544,871
3	0.62	2.0	51,067,666	0.02	0.17	27.05	14.98	19.90	32	192	16,860	193,346,690	3.79	244,414,355	424,820,085	374,169,784	372,599,167
4	0.64	2.1	52,540,047	0.02	0.17	26.96	14.90	19.87	33	199	17,255	199,435,345	3.80	251,975,392	435,076,512	382,818,518	380,819,040
5	0.66	2.1	53,703,348	0.02	0.17	26.93	14.86	19.78	34	202	17,597	202,307,642	3.77	256,010,989	441,443,603	388,194,304	385,918,221
6	0.68	2.2	55,966,240	0.02	0.17	26.84	14.86	19.63	34	207	18,330	209,316,078	3.74	265,282,318	453,376,220	398,160,968	394,967,474
7	0.70	2.3	58,155,563	0.02	0.17	26.73	14.79	19.49	35	213	18,960	216,590,374	3.72	274,745,937	464,227,720	407,092,214	402,858,592
8	0.72	2.4	62,065,913	0.02	0.16	26.67	14.80	19.33	37	222	20,255	233,002,053	3.75	295,067,966	483,346,751	422,532,490	416,172,074
9	0.74	2.5	64,059,353	0.02	0.16	26.51	14.66	19.24	38	230	20,708	241,261,315	3.77	305,320,668	491,990,102	429,352,523	421,561,839
10	0.76	2.6	65,554,647	0.02	0.16	26.51	14.67	19.16	39	233	21,198	247,329,053	3.77	312,883,700	497,916,944	433,956,493	425,140,359
11	0.78	2.6	67,717,580	0.02	0.16	26.48	14.67	19.06	39	238	21,902	256,653,401	3.79	324,370,981	505,904,229	440,063,077	429,693,806
12	0.80	2.7	68,382,665	0.02	0.16	26.48	14.68	19.02	40	239	22,134	259,606,825	3.80	327,989,490	508,165,475	441,765,734	430,864,658
13	0.82	2.7	69,650,789	0.02	0.16	26.40	14.64	18.95	40	242	22,477	264,808,248	3.80	334,459,036	511,690,356	444,354,568	432,280,136
14	0.84	2.9	73,160,014	0.02	0.15	26.38	14.66	18.79	41	248	23,648	282,162,363	3.86	355,322,376	520,856,834	450,939,126	435,312,143
15	0.86	2.9	73,458,402	0.02	0.15	26.38	14.67	18.79	41	248	23,765	284,334,356	3.87	357,792,758	521,727,420	451,565,326	435,534,120
16	0.88	2.9	73,970,104	0.02	0.15	26.38	14.68	18.75	41	249	23,940	286,563,359	3.87	360,533,463	522,612,345	452,149,625	435,602,930
17	0.90	2.9	74,556,675	0.02	0.15	26.34	14.64	18.74	42	252	24,064	290,829,416	3.90	365,386,090	523,714,607	452,887,191	435,434,738
18	**0.92**	**3.0**	**76,188,939**	**0.02**	**0.15**	**26.32**	**14.65**	**18.68**	**42**	**255**	**24,600**	**300,893,379**	**3.95**	**377,082,319**	**525,882,777**	**454,209,582**	**434,742,593**
19	0.94	3.2	81,317,130	0.02	0.15	26.33	14.78	18.53	43	261	26,487	335,621,436	4.13	416,938,566	531,508,519	458,107,850	433,852,977
20	0.96	3.2	81,903,201	0.02	0.15	26.28	14.73	18.50	44	264	26,593	338,893,959	4.14	420,797,160	531,949,896	458,399,527	433,481,694
21	0.98	3.2	82,516,285	0.02	0.15	26.27	14.73	18.48	44	264	26,799	342,828,927	4.15	425,345,212	532,220,881	458,554,617	432,972,760
22	1.00	3.2	82,885,529	0.02	0.14	26.27	14.73	18.46	44	265	26,920	345,013,617	4.16	427,899,146	532,284,289	458,569,909	432,640,384
23	1.02	3.2	83,014,171	0.02	0.14	26.26	14.73	18.45	44	265	26,961	345,814,305	4.17	428,828,476	532,272,921	458,549,151	432,492,567
24	1.04	3.3	83,587,647	0.02	0.15	26.23	14.69	18.43	44	267	27,065	349,763,083	4.18	433,350,730	531,972,790	458,262,729	431,452,296
25	1.06	3.3	83,800,500	0.02	0.14	26.22	14.68	18.41	44	268	27,123	350,993,146	4.19	434,793,646	531,830,483	458,132,621	431,045,273
26	1.08	3.3	83,868,274	0.02	0.14	26.22	14.68	18.41	44	268	27,149	351,658,307	4.19	435,526,581	531,741,392	458,057,203	430,866,532
27	1.10	3.3	84,686,528	0.02	0.14	26.15	14.61	18.36	45	271	27,283	356,319,253	4.21	441,005,781	530,788,337	457,244,004	428,942,940
28	1.12	3.3	84,781,091	0.02	0.14	26.14	14.61	18.35	45	271	27,301	356,959,466	4.21	441,740,556	530,645,406	457,124,860	428,691,435
29	1.14	3.4	88,048,098	0.02	0.14	26.28	14.86	18.45	46	276	28,852	401,746,540	4.56	489,794,638	521,954,090	450,168,127	414,566,454
30	1.16	3.5	88,386,246	0.02	0.14	26.29	14.88	18.45	46	276	29,000	405,728,295	4.59	494,114,540	520,991,949	449,407,757	413,107,625
31	1.18	3.5	88,805,019	0.02	0.14	26.25	14.85	18.43	46	278	29,064	409,039,677	4.61	497,844,695	519,952,988	448,585,035	411,489,057
32	1.20	3.5	89,031,261	0.02	0.14	26.24	14.84	18.42	46	278	29,132	410,926,065	4.62	499,957,326	519,338,606	448,100,790	410,596,892
33	1.22	3.5	89,306,866	0.02	0.14	26.23	14.84	18.40	46	279	29,213	413,027,993	4.62	502,334,859	518,554,843	447,484,752	409,561,411
34	1.24	3.5	89,353,256	0.02	0.14	26.23	14.84	18.41	46	279	29,224	413,924,074	4.63	503,277,330	518,296,282	447,284,353	409,216,483
35	1.26	3.5	89,529,516	0.02	0.14	26.21	14.82	18.40	47	280	29,254	415,763,257	4.64	505,292,773	517,603,600	446,744,799	408,325,778
36	1.28	3.5	89,631,689	0.02	0.14	26.21	14.82	18.39	47	280	29,280	416,602,531	4.65	506,234,221	517,219,184	446,445,363	407,871,953
37	1.32	3.5	89,947,373	0.02	0.14	26.21	14.83	18.39	47	280	29,401	420,141,675	4.67	510,089,048	515,702,633	445,270,437	406,041,804
38	1.34	3.5	90,388,693	0.02	0.14	26.25	14.88	18.41	47	280	29,643	428,504,415	4.74	518,893,108	512,442,198	442,762,627	402,046,316

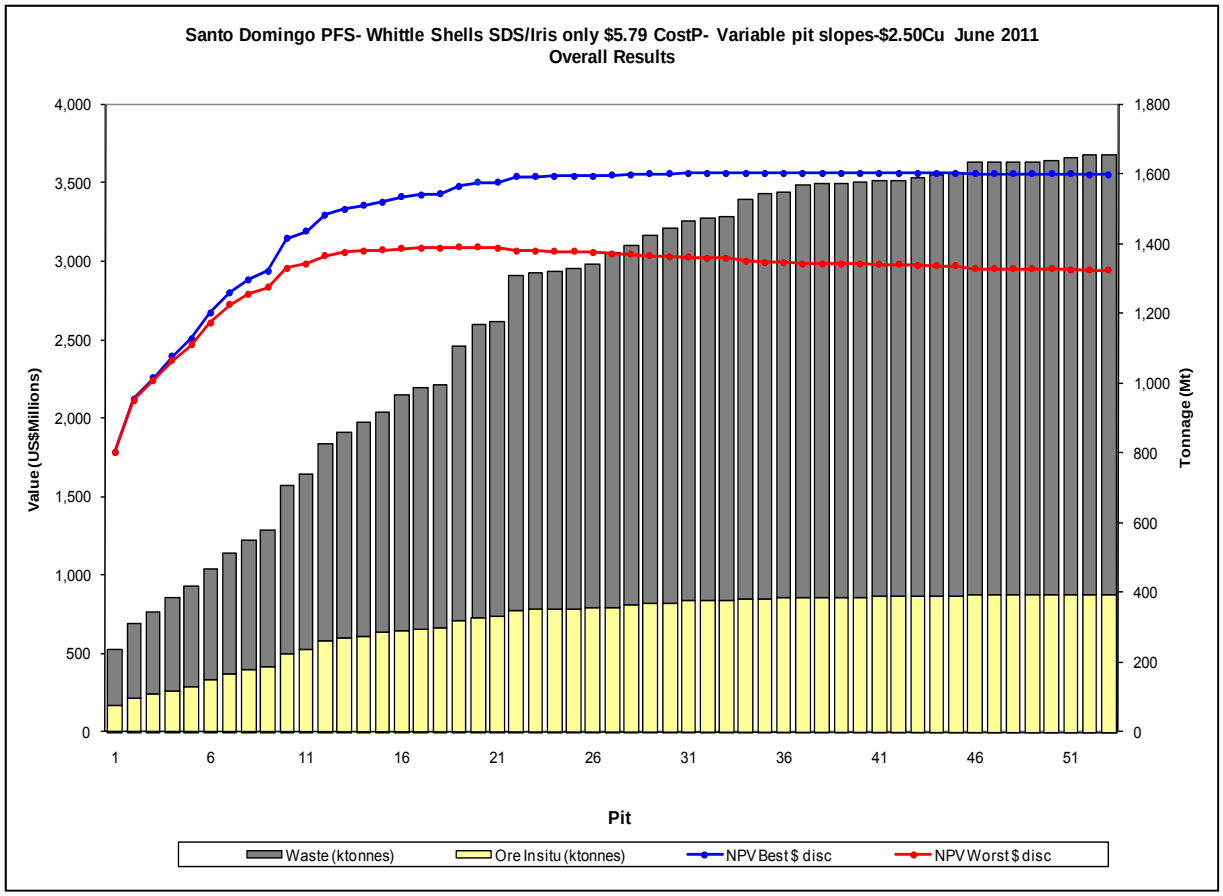


Figure 15-1: Cu Recovery Algorithm Comparison

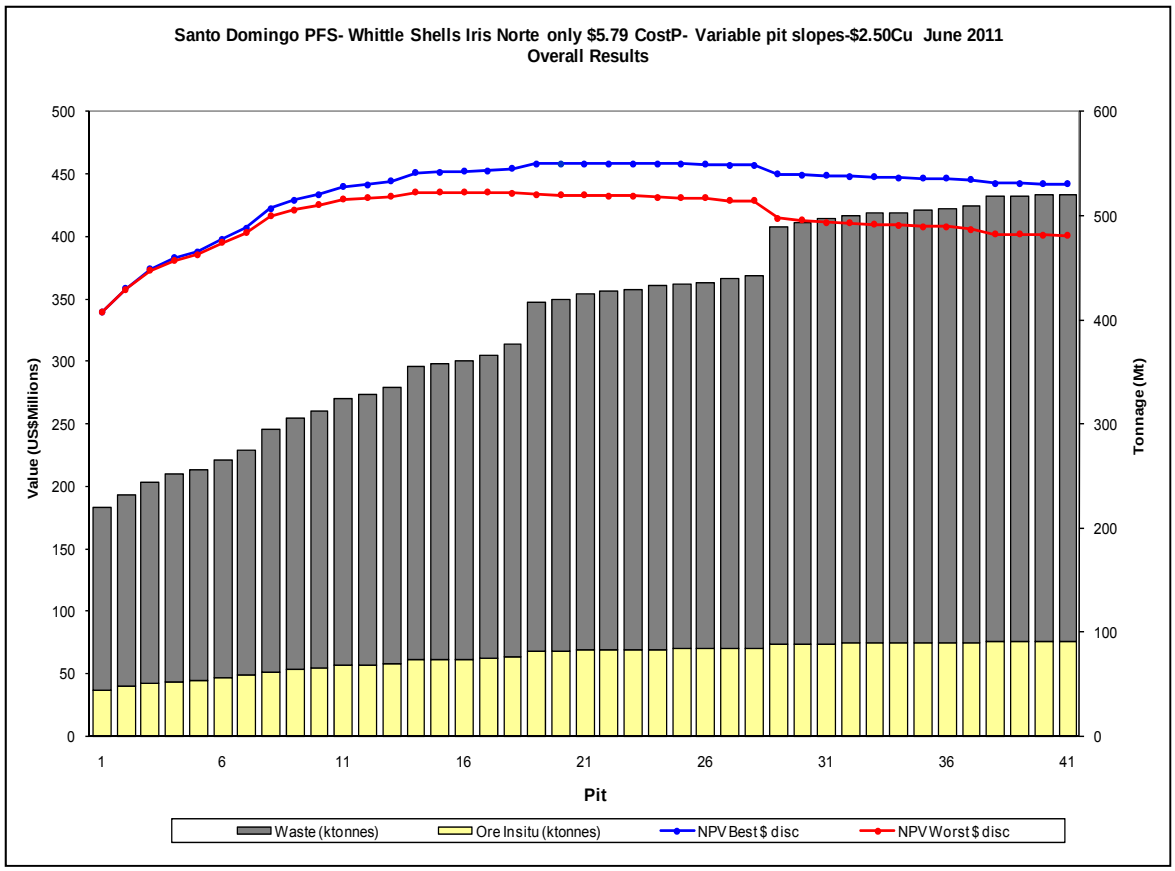


Figure 15-2: Iris Norte Optimization Results

Based on the thorough analysis of the above results the chosen Whittle™ shell was used as the basis for the detailed pit designs created for each of the Santo Domingo pits. These detailed pit designs take into consideration, minimum mining widths, access ramps, and detailed bench configurations as summarized in Table 15.5 below (as provided by AMEC).

Table 15.5: Detailed Pit Parameters

Design Parameter	SANTO DOMINGO/ IRIS				IRIS NORTE	
	All Sectors	Sector 1 (South)	Sector 2 (West)	Sector 3 (East)	Entire Pit one Sector	
Rock Type	Overburden	Bedrock	Bedrock	Bedrock	Overburden	Bedrock
Bench Face Angle (deg)	55	75	75	70	55	75
Bench Height (m)	12	12	12	12	12	12
Bench Configuration	Single (12 m)	Double (24 m)	Double (24 m)	Double (24 m)	Single (12 m)	Double (24 m)
Interramp Angle (deg)	Take from wall geometry meeting overall wall angle	60	56	53	Take from wall geometry meeting overall wall angle	Take from wall geometry meeting overall wall angle
Maximum Interramp Stack Height (m)	N/A	200	200	100	N/A	100
Geotechnical Stability Berm Width (m)	N/A	35	50	50	N/A	50
Blasting requirements	Cushion but to be trialed prior to commitment	Pre-split	Pre-split	Pre-split	Cushion but to be trialed prior to commitment	Pre-split
Ramp width (m)	28	28	28	28	28	28
Overall Wall angel (deg) for use in Whittle optimization	38	55	50	42	38	42

15.2.3 Open Pit Mineral Reserves

The mineral reserve estimate for the detailed open pit designs are summarized in Table 15.6 below for the probable reserve classification.

Table 15.6: Open Pit Waste Rock Summary

Stage	Ore (Mt)	Waste Material					
		Inferred Material (Mt)	Overburden (Mt)	Rock Waste (Mt)	Low grade Material (Mt)	Total Waste (Mt)	Total Material (Mt
SDS/Iris							
SDS Stage1	71.8	0.0	1.0	144.6	**8.6**	154.2	226.0
SDS Stage2	63.7	0.3	3.3	143.9	4.9	152.4	216.1
SDS Stage3	170.5	3.6	32.8	407.7	49.8	493.9	664.4
SDS Stage4	38.8	1.9	0.4	177.8	9.6	189.7	228.5
Subtotal SDS/Iris	**344.8**	**5.8**	**37.5**	**874.0**	**72.9**	**990.2**	**1,335.0**
Iris Norte							
IRN Stage 1	21.4	0.4	38.9	45.8	12.1	97.2	118.7
IRN Stage 2	28.0	1.5	53.9	28.2	13.1	96.7	124.7
IRN Stage 3	23.7	0.4	62.8	27.5	2.5	93.3	117.0
Subtotal Iris Norte	**73.1**	**2.3**	**155.6**	**101.6**	**27.7**	**287.2**	**360.3**
Grand Total	**418.0**	**8.2**	**193.0**	**975.6**	**100.6**	**1,277 .4**	**1,695.4**

Note: NSR cut-off of $5.79/t. Ore includes Indicated Resources only. Total Waste Mined includes Inferred Material, Overburden, Rock, and Material below cut-off.

Within these detailed pit designs there a total of 8 Mt of inferred mineral resources. These inferred tonnes were not included in the LOM production plan. There is no certainty that these inferred mineral resources will be converted to the measured or indicated categories through further drilling, or into mineral reserves, once economic considerations are applied. There is also 31 Mt of oxide material that has not been included in the LOM plan. This oxide material will be selectively placed on the WRF to allow for the potential processing in the future.

The mineable reserve estimate has been based upon economic parameters, geotechnical design criteria and metallurgical recovery assumptions detailed in the sections above. Changes in these assumptions will impact the reserve estimate. In general, increases in operating costs, reductions in revenue assumptions or reductions in metallurgical recovery may result in increased cutoff grades, reductions in reserves and increasing strip ratios. The converse is also true. Reductions in operating costs, increases in revenue assumptions or increases in metallurgical recovery may result in reduced cut-off grades and increases in reserves.

There is currently a highway crossing over the area of the Iris Norte open pit design. This infrastructure element will require re-location in order to mine the reserves in this area.

Regulatory and permitting factors affecting the project are discussed in Section 20. Reserves have been estimated assuming that project permitting is achievable.

15.2.4 Cut-off Grade Calculation

Table 15.7 summarizes the cut-off grade calculations for the various deposits at Santo Domingo. These copper equivalent cut-off grades are estimates only, since the actual modelling and optimization work was conducted with the NSR model previously described in the report.

Table 15.7: Copper Equivalent Cut – off Grade Estimate

Parameter	Unit	PFS FINAL Santo Domingo	
Revenue, smelting & refining		**Resource COG[1] (includes mining cost)**	**Incremental COG[2] (excludes mining cost)**
Cu price	US$/lb Cu	2.50	2.50
Payable metal	% Cu	100.00%	100.00%
TC/RC/Transport	C$/lb Cu payable	0.24	0.24
NSR (Cu only)	C$/lb Cu payable	2.26	2.26
Opex estimates			
Mining cost	US$/t mined	1.21	
Strip Ratio	t:t	3.3	
Mining Cost	US$/t milled	5.20	
Processing and G&A cost	US$/t milled	5.79	5.79
Site Cost	US$/tonne milled	10.99	5.79
Recovery and Dilution			
Recovered Cu grade	%Cu	0.22	0.12
Process Recovery	average %	87.5%	87.5%
Plant feed Cu grade	diluted %Cu	0.25	0.13
Dilution	%	0.0%	0.0%
Cut-off Grade			
In-situ cut-off Cu grade (Cu only)	%Cu	0.25	0.13
Au and Fe contribution (estimated)	% of Cu value	42%	42%
In-situ cut-off Cu grade (inc. Au, Fe value)	%Cu	0.18	0.09

1. For material that need not be mined (mine or external cut-off NSR)

2. For material that must be mined but can be either wasted or processed (mill or internal cut-off NSR)

16 MINING METHODS

16.1 Mine Geotechnical

The Santo Domingo project currently comprises four deposits: Santo Domingo Sur (SDS), Iris, (currently grouped together as “SDS/IRIS”), Estrellita, and Iris Norte. This PFS assessment included two of the deposits, SDS/Iris and Iris Norte (Figure 16-1).

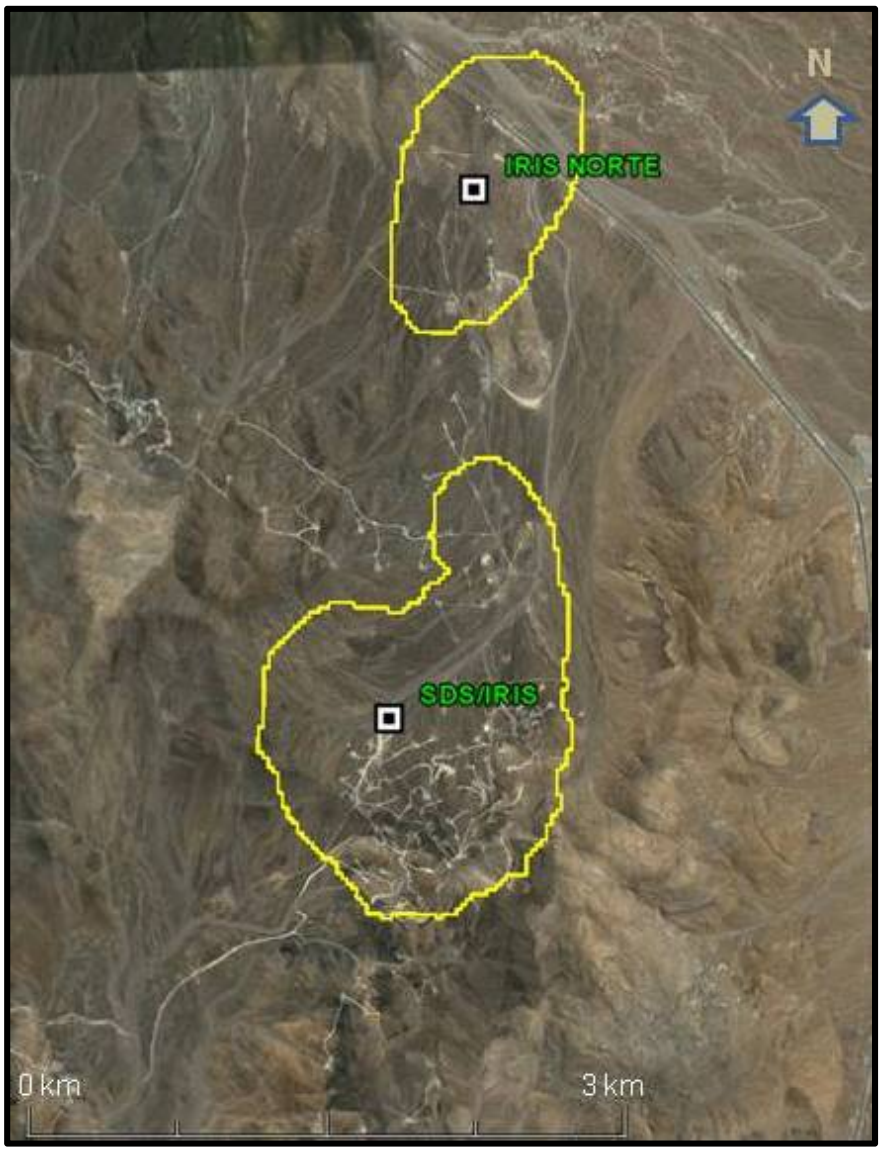


Figure 16-1: SDS/Iris and Iris Norte Location Map

The geotechnical characterization is summarized in previous studies[17] and data provided by FWM.

The work included characterization of the rock mass for the SDS/Iris open pit area based on geotechnical logging and laboratory test results from three core drilling campaigns, two of them conducted by FWM between 2006 and 2010, and one supervised by AMEC and carried out in July 2010. A total of 28 core-oriented geotechnical holes (26 with geotechnical core-logging) were completed, representing more than 7,000 m of core. Based on these data, rock mass rating (RMR) and rock quality designation (RQD) values were established to characterize the geotechnical rock mass and provide guidance in establishing the prefeasibility mine wall design parameters.

There was a more limited engineering geology database for the Iris Norte deposit. In this database the majority of the information was for the rock portion and not the overburden. While there is no extensive geotechnical information for this deposit, based upon the correlations between the geological information available (e.g., RQD, rock type etc.), characterization parameters were extrapolated from Santo Domingo data. Experience with similar deposits was used to provide further confidence in the extrapolation exercise. Given the timeframe for developing the Iris Norte deposit, >10 to 15 years after planned SDS/Iris mining, it was considered appropriate and prudent to assign design slope parameters for the PFS based upon present information, and recommend further information be gathered during the detailed feasibility stage of the project. The existing information for Iris Norte meets minimally acceptable values for a PFS-level study. This is further supported by the approved/pending commitment by FWM to gathering additional information in the near term.

One difference between the SDS/Iris and Iris Norte pits is that the former has considerably more overburden. Based upon an engineering geology evaluation of the site, there is a high likelihood for the overburden materials in the Iris Norte pit to be similar to those encountered at the plant site. This similarity in overburden materials suggests that the overburden at Iris Norte is also a very hard soil consisting of sand/gravels that are strongly cemented with salts/carbonates/calcrete, locally termed “Tertel.”

16.1.1 SDS/Iris Mine Geotechnical Design Parameters

Based on data from geotechnical and core-oriented holes, RMR and RQD values were established to assist in characterizing the geotechnical rock mass quality. Results indicated that RQD varies from 79 to 90 (good), and RMR ranges from 40 to 56 (fair rock quality) from the FWM core logging, and about 75 (good rock) from the AMEC core logging. In terms of predominant structure, there are two wide and predominant mains: Dip/DipDir 16/063 (N27°W/16°NE) and Dip/DipDir 26/240 (N30°W/26°SW).

Complementing the logging results, an extensive geomechanical laboratory testing program was carried out on 394 samples from 21 boreholes. This included developing 620 tests (unit weight, porosity, point load test, unconfined compressive strength, and triaxial), which allowed for the geomechanical characterization of five lithological/geotechnical units.

[17] a) AMEC (Aug 2010). Technical Report 40003-GG-TR-0001 Rev. 0. Geotechnical Drilling Campaign for Santo Domingo Open Pit Wall Design. b) AMEC (Feb 2011). Technical Report 40006-GG-TR-0001 Rev. 0. Estimation of Preliminary Pit Slope Angles.

The collective information was used to establish the PFS wall design parameters, which were differentiated by three sectors (Figure 16-2). The design parameters are provided in Table 16.1.

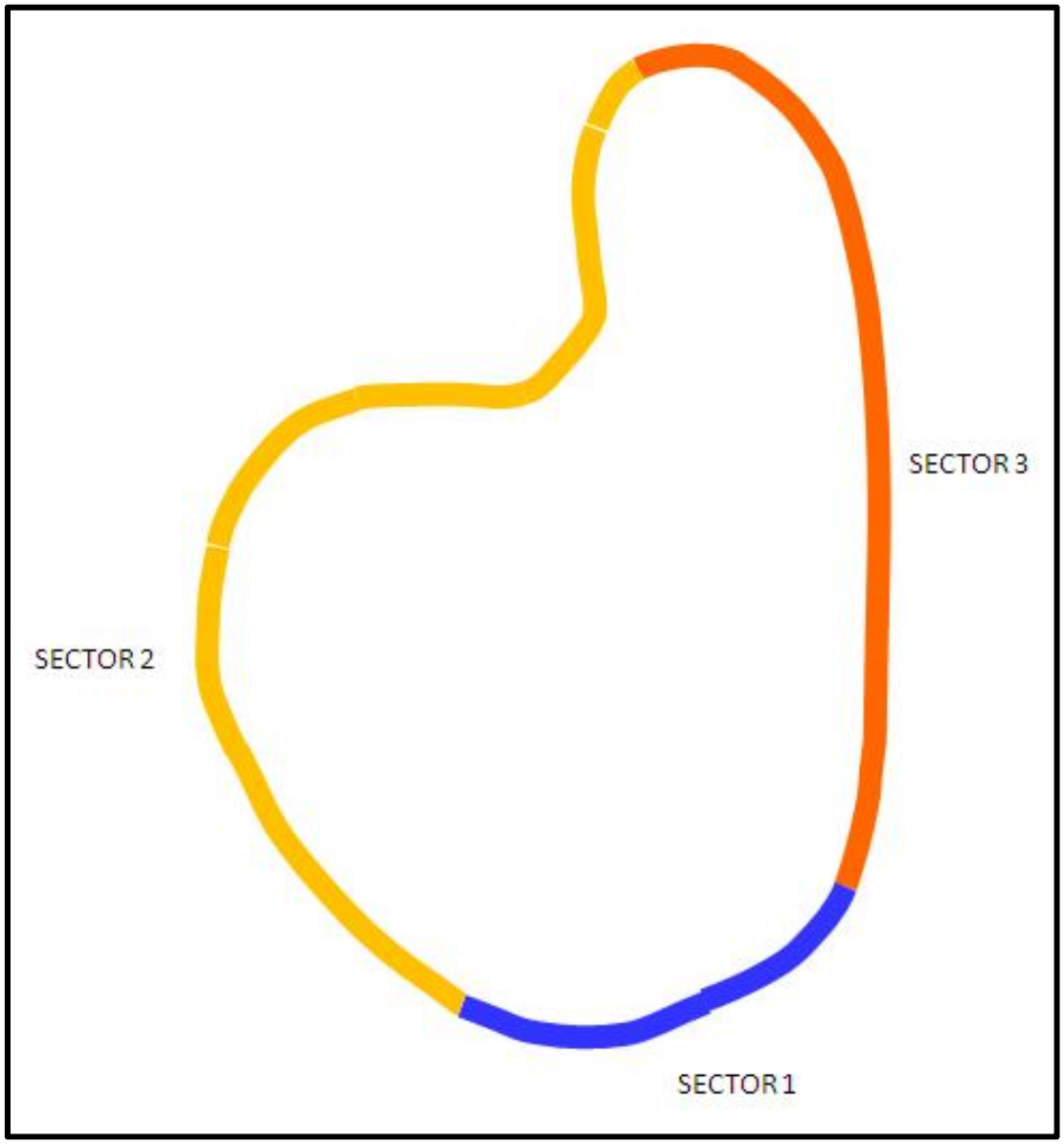


Figure 16-2: SDS/Iris Design Sectors

Table 16.1: Santo Domingo Project – Slope Design Parameters SDS/Iris

Design Parameter	Section 1 (South)	Sector 2 (West)	Sector 3 (East)
Rock Type	Bedrock	Bedrock	Bedrock
Bench Face Angle (deg.)	75	75	70
Bench Height (m)	12	12	12
Bench Configuration	Double (24 m)	Double (24 m)	Double (24 m)
Bench Width (m)	8	10	10
Inter-ramp Angle (deg)	60	56	53
Maximum Inter-ramp Stck Height (m)	200	200	100
Geotechnical Stability Berm Width (m)	35	50	50
Blasting Requirement	Pre-split	Pre-split	Pre-split
Ramp Width (m)	35	35	35
Overall Wall Angle (deg)	60	50	42

16.1.2 Iris Norte Mine Geotechnical Design Parameters

The extensive database established for SDS/Iris was used, as described above, to extrapolate through a comparison of geological databases between the two deposits. This extrapolation was only available for the rock portion of the deposit as there was no characterization of the overburden at SDS/Iris. There was, however, a good database of RQD for Iris Norte. For Iris Norte, the RQD for the rock mass varied from 60% to 100% (in 85% of cases), with an average of 84%, and the RMR varied from 20 to 59, averaging 38. These values indicated that, in general, fair to good rock conditions are present at Iris Norte, with slightly reduced rock quality than that present at SDS/Iris. However, it was judged that the rock quality present at Iris Norte would support overall wall angles comparable to SDS/Iris. The PFS slope design parameters for Iris Norte are showed in Table 16.2.

Table 16.2: Santo Domingo Project – Slope Design Parameters Iris Norte

Design Parameter	Entire Pit (One Sector)	
Rock Type	Overburden	Bedrock
Bench Face Angle (degrees)	55	75
Bench Height (m)	12	12
Bench Configuration	Single	Double (24 m)
Bench Width (m)	10	10
Inter-ramp Angle (degrees)	(1)	(1)
Maximum Inter-ramp Stack Height (m)	N/A	100
Geotechnical Stability Berm Width (m)	N/A	50
Blasting Requirement	(2)	Pre-split
Ramp Width (m)	28	28
Overall Wall Angle (degrees)	38	42

Note: (1) Take from wall geometry meeting overall wall angle. **(2)** Cushion but to be trailed prior to commitment.

16.2 Hydrology and Hydrogeology

The site is located in the Atacama region of Chile, one of the most arid regions in South America. Average annual precipitation is less than 30 mm. Site water supply and management will need to concentrate on groundwater, and thus a thorough understanding of the site and regional hydrogeology is critical to project development.

The project area contains three hydrogeological units:

- unconsolidated sediments containing two aquifers of local extension located in the Chañarcito (Chañaral Alto) ravine area
- fractured rocks underlying the sediments, which are the main aquifers in the area
- a regional bedrock that is relatively low in hydraulic conductivity, which is favourable in allowing water to sit within the fractured rocks, hence allowing access in terms of water supply.

Groundwater flow direction is strongly influenced by the morphological structures that are present, although they follow the general trend of the watershed. Groundwater levels are can be influenced by rainfall, observing increases in their levels during the wetter months.

Based on the current data, a water table level located at an elevation of 895 masl is suggested as the baseline for the SDS/Iris area.

16.2.1 Background Information

The rainfall information used in the analysis was obtained from existing meteorological stations (period of 2009-2010) in the Atacama region that are operated by the "General Directorate of Water bodies (DGA)[18].

FWM provided a database of 96 holes containing water level measurements; 72 of these holes were analyzed for the SDS/Iris pit area.

16.2.2 Hydrology

The project area is located in the Salado River Basin and has a transitional desert climate, with low and variable rainfall during winter time. In this type of climate, coastal influence is quite reduced and negligible further east, as observed with air humidity which decreases to the east. The environment temperature exhibits important oscillation when comparing day and night. Absence of permanent surface runoff is a constant condition. One potential source for aquifer recharge is weather events when maximum precipitation occurs.

[18] DGA 2009. Evaluación de Recursos Hídricos Subterráneos en Cuencas de La Región de Atacama Ubicadas Entre el Río Copiapó y Región de Antofagasta." DGA 2010. Análisis y Evaluación de los Recursos Hídricos Subterráneos de los Acuíferos Costeros ubicados entre los ríos Salado y Huasco, III región de Atacama".

From the correlation of average rainfall and altitude shown in Figure 16-3, it can be inferred that annual average rainfall in the project area should be in the order of 26 mm, which coincides with the data taken in 2010 from the meteorological station at FWM. As indicated in Figure 16-3, there is a slight orographic (elevation control) influence to average rainfall in the region.

Overall, the site is very arid, and surface runoff events will be rare. Engineering works necessary to address runoff from average events will likely be minimal, although extreme events may be significant, and confirmation of natural hazard evaluations completed in subsequent stages of the project will be required.

Water conservation in all mining activities (e.g., tailings management and water seepage through the consolidated tailings area) will be an important consideration of mine operations.

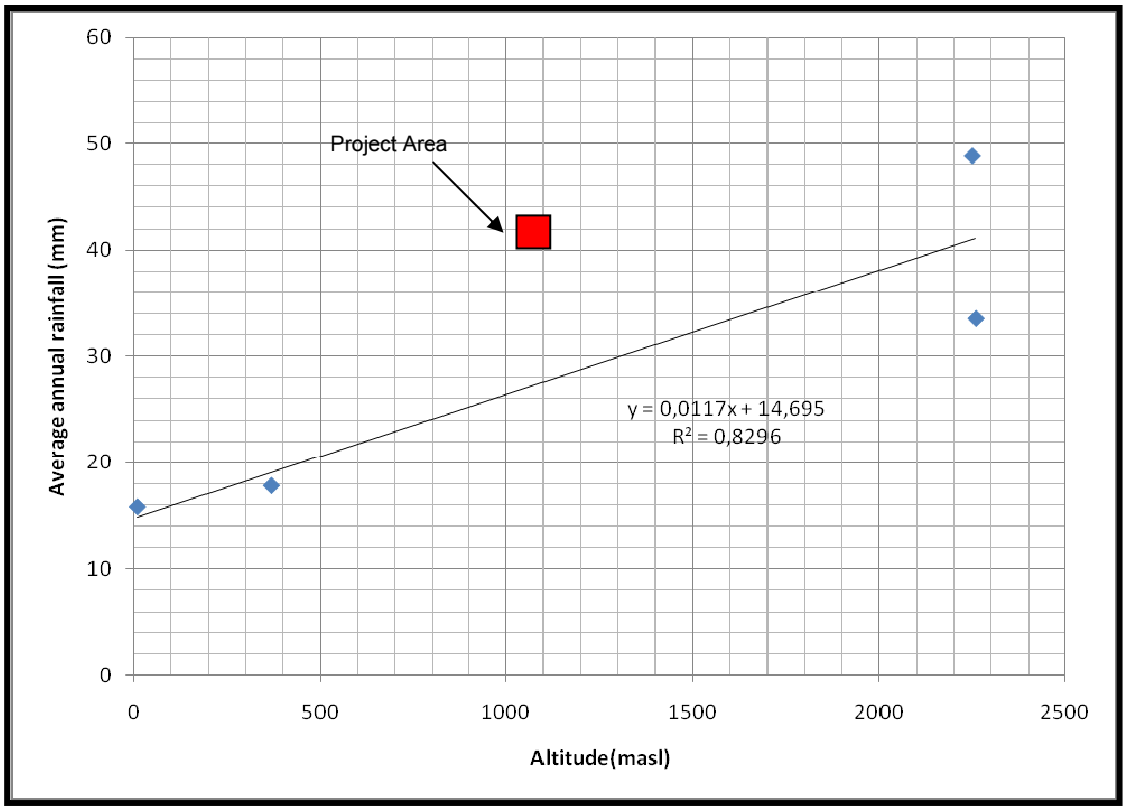


Figure 16-3: SDS/Iris Design Sectors

16.2.3 Hydrogeology

As noted above, there are three main hydrogeologic units present at the site. The estimates used for hydraulic conductivity (coefficients of permeability) for each hydrogeological unit were based on an evaluation of test pits, testing in drill holes and general experience with similar deposits. The hydraulic conductivity in the unconsolidated sediments has a high variability, with estimates ranging between 10^{-2} and 10^{-6} cm/s, decreasing with both depth and the degree of cementation present. For the fractured rock units the range is even

greater, as resistance to flow will depend on many factors including the frequency of fractures, interconnection of fractures, degree of infilling, size of openings, etc. Overall, it is estimated that the hydraulic conductivity of this units will ranges between 1 and 10^{-7}cm/s. However, average values are much less variable than that. Estimate of average hydraulic conductivity values for the fractured rock can be based on observation of the fractures from drilling carried out in the SDS/Iris area, which indicated openings generally less than 1 mm and they were largely sealed. This condition would indicate that average hydraulic conductivity for the majority of the fractured rock unit would be between 10^{-5} and 10^{-6} cm/s. The bedrock unit corresponds to relatively non-fractured rocks, which present very low hydraulic conductivities of less than 10^{-6} cm/s.

Flow direction was estimated with the aid of 3D kriging of water level results, which provided the phreatic surface shown in Figure 16-4, where the flow lines are showing the hydrologic watershed that separates the Chañarcito ravine from the areas immediately to the north.

Based on water monitoring drill hole data, a water table level located at an elevation of 895 masl was estimated, which corresponds to approximately 225 m below existing ground surface at the SDS/Iris pit area (Figure 16-5). This depth to present average water elevation indicates that groundwater would not be a pit design challenge for a considerable portion of the mining operation.

In addition to the present depth of the water table, pit development will provide considerable dewatering, further reducing the influence of water on overall wall performance. Overall, the hydrogeological characterization should be better developed at the next stage of the project, but the positive trends noted during the PFS evaluation are expected to continue as the database is enhanced.

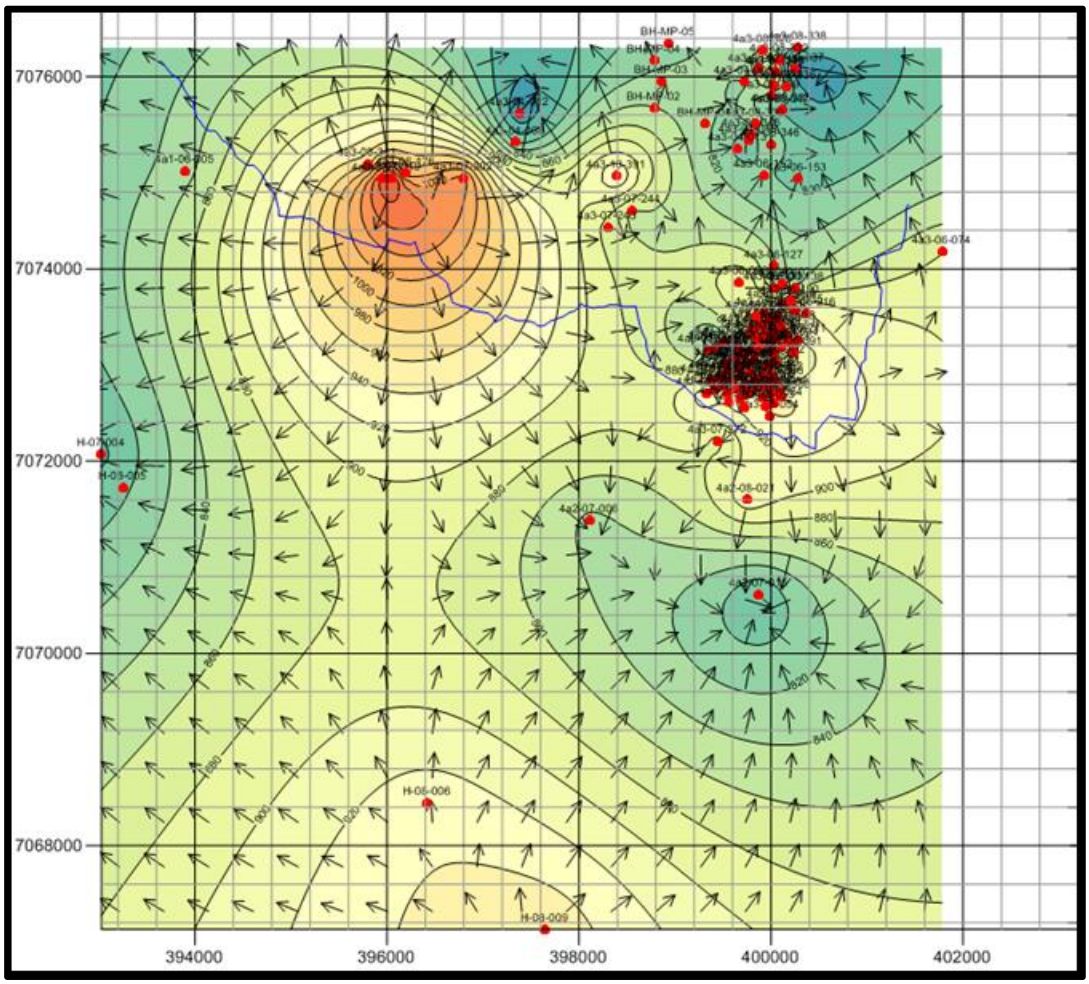


Figure 16-4: Groundwater Flow Lines

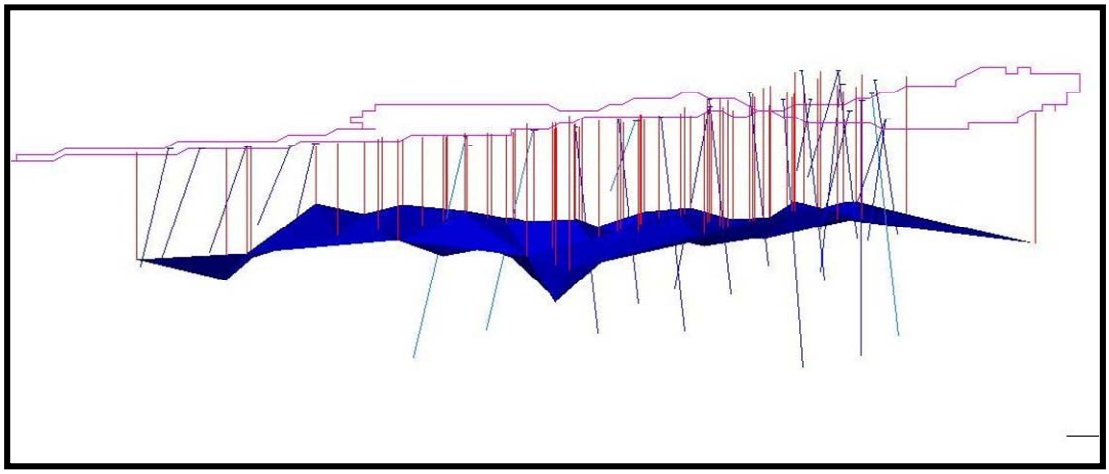


Figure 16-5: Water Table in SDS/IRIS Area

16.3 Open Pit Mine Plan

Mine planning for the Santo Domingo open pit deposits was conducted using a combination of Mintec Inc. MineSight® software and Gemcom GEMS™ and Whittle™ software. The 3-D mineral inventory models were as provided by RPA. Further NSR modelling was conducted using GEMS™. The detailed pit and stage designs were undertaken with the use of MineSight®.

The existing topography as of early 2011 was used to determine the starting point of this pre-feasibility study. Based on the thorough analysis of the Whittle™ pit shells and preliminary schedules (discussed in Mineral Reserve section of the report), base case pit shells were chosen for the various Santo Domingo deposits and used as the basis for the detailed ultimate pit designs for Santo Domingo Sur, Iris and Iris Norte, along with associated pit staging. Waste rock facilities ("WRF") were then designed to account for the waste material produced in each mining stage.

The mining sequence, which mines higher grade material early on in the schedule, begins with Santo Domingo Sur. Mining of the Santo Domingo Sur Pit will be followed by Iris, with Iris Norte mined last in the sequence.

Santo Domingo Sur and Iris form one of the pits and is divided into four stages. Iris Norte forms a separate pit and has been split into three stages. The stage tonnages and associated grades are summarized in Table 16.3, while a breakdown of material types for the OP deposits is summarized in Table 16.4.

Table 16.3: Pit Stage Tonnages and Grades

Stage	Ore (Mt)	Total Waste (Mt)	Total Material (Mt)	Strip Ratio (t:t)	Ore Grade Au (g/t)	Ore Grade Cu (%)	Contained Metal Au (Koz)	Contained Metal Cu (Mlbs)	Contained Metal Magnetite Conc. (Mt)
SDS/Iris									
SDS Stage1	71.8	154.2	226.0	2.1	0.08	0.61	193	958	11
SDS Stage2	63.7	152.4	216.1	2.4	0.06	0.41	113	574	10
SDS Stage3	170.5	493.9	664.4	2.9	0.03	0.23	173	848	32
SDS Stage4	38.8	189.7	228.5	4.9	0.05	0.36	60	304	3
Subtotal SDS/Iris	**344.8**	**990.2**	**1,335.0**	**2.9**	**0.05**	**0.35**	**539**	**2,684**	**57**
Iris Norte									
IRN Stage 1	21.4	97.2	118.7	4.5	0.03	0.23	20	108	4
IRN Stage 2	28.0	96.7	124.7	3.5	0.01	0.13	12	78	7
IRN Stage 3	23.7	93.3	117.0	3.9	0.01	0.11	8	60	5
Subtotal Iris Norte	**73.1**	**287.2**	**360.3**	**3.9**	**0.02**	**0.15**	**41**	**246**	**17**
Grand Total	**418.0**	**1,277.4**	**1,695.4**	**3.1**	**0.04**	**0.32**	**580**	**2,930**	**73**

Note: NSR cut-off of $5.79/t. Ore includes Indicated Resources only. Total Waste Mined includes Inferred Material, Overburden, Rock, and Material below cut-off. Magnetite concentrate tonnage based on average 65% iron grade.

Table 16.4: Open Pit Waste Rock Summary

Stage	Ore (Mt)	Waste Material					
		Inferred Material (Mt)	Overburden (Mt)	Rock Waste (Mt)	Low grade Material (Mt)	Total Waste (Mt)	Total Material (Mt)
SDS/Iris							
SDS Stage1	71.8	0.0	1.0	144.6	**8.6**	154.2	226.0
SDS Stage2	63.7	0.3	3.3	143.9	4.9	152.4	216.1
SDS Stage3	170.5	3.6	32.8	407.7	49.8	493.9	664.4
SDS Stage4	38.8	1.9	0.4	177.8	9.6	189.7	228.5
Subtotal SDS/Iris	**344.8**	**5.8**	**37.5**	**874.0**	**72.9**	**990.2**	**1,335.0**
Iris Norte							
IRN Stage 1	21.4	0.4	38.9	45.8	12.1	97.2	118.7
IRN Stage 2	28.0	1.5	53.9	28.2	13.1	96.7	124.7
IRN Stage 3	23.7	0.4	62.8	27.5	2.5	93.3	117.0
Subtotal Iris Norte	**73.1**	**2.3**	**155.6**	**101.6**	**27.7**	**287.2**	**360.3**
Grand Total	**418.0**	**8.2**	**193.0**	**975.6**	**100.6**	**1,277.4**	**1,695.4**

Note: NSR cut-off of $5.79/t. Ore includes Indicated Resources only. Total Waste Mined includes Inferred Material, Overburden, Rock, and Material below cut-off.

The open pit mining activities for the Santo Domingo pits were assumed to be undertaken by an owner-operator mining fleet for this pre-feasibility study. The owner-operator mining cost unit rate used in the Whittle™ optimization was US$1.21 per tonne of material for pit and dump operations, road maintenance, mine supervision, technical services and senior management costs. The mining unit rate was calculated based on equipment required to achieve a processing rate of 70 ktpd. Mining costs were developed from first principles for similar sized operations, along with estimates for local labour, fuel and power costs.

16.3.1 Mine Equipment

The major mining equipment requirements are indicated in Table 16.5 and are based on similar sized operations. The proposed plant processing rate of 70 ktpd was used to estimate the mining equipment fleet required. The fleet has an estimated maximum capacity of 300 ktpd total material, which will be sufficient for the proposed LOM plan.

Table 16.5: Mine Equipment

No. of Units	Equipment Type
6	Crawler-Mounted, Rotary Tri-Cone, 311mm Dia. Drill
1	Crawler-Mounted, Rotary Tri-Cone, 115mm Dia. Drill
4	Diesel, 26-cu-m Front Shovel
1	Diesel 25-cu-m Wheel Loader
29	220-t class Haul Truck
6	D10-class Track Dozer
3	834H-class Rubber-tired Dozer
5	16M-class Grader
2	90t class (20,00 gal.) Water Truck

16.3.2 Unit Operations

The 311 mm diameter drill will perform the majority of the production drilling in the mine, with the smaller 115 mm drill used for secondary blasting requirements. The main loading and haulage fleet consists of 220 t haul trucks, which are loaded primarily with the diesel front shovels or the large wheel loader, depending on pit conditions. As pit conditions dictate, the track dozers are used to rip and push material to the excavators, as well as maintaining the waste dumps.

The portion of the equipment listed in Table 16.5 will be used to maintain and build access roads, and to meet various site facility requirements, (including coarse mill feed stockpile maintenance and further exploration development).

The work schedule is based on two 12 hour shifts, seven days a week, 365 days per year.

16.4 Production Schedule

16.4.1 Mine Sequence and Phasing – Open Pit

The detailed pit designs for the various deposits for Santo Domingo were divided into various stages for the mine plan development to maximize grade in the early part of the schedule, balance waste stripping requirements, while providing the required mill feed production per period. The overall Santo Domingo site plan final configuration is illustrated in Figure 16-6 below.

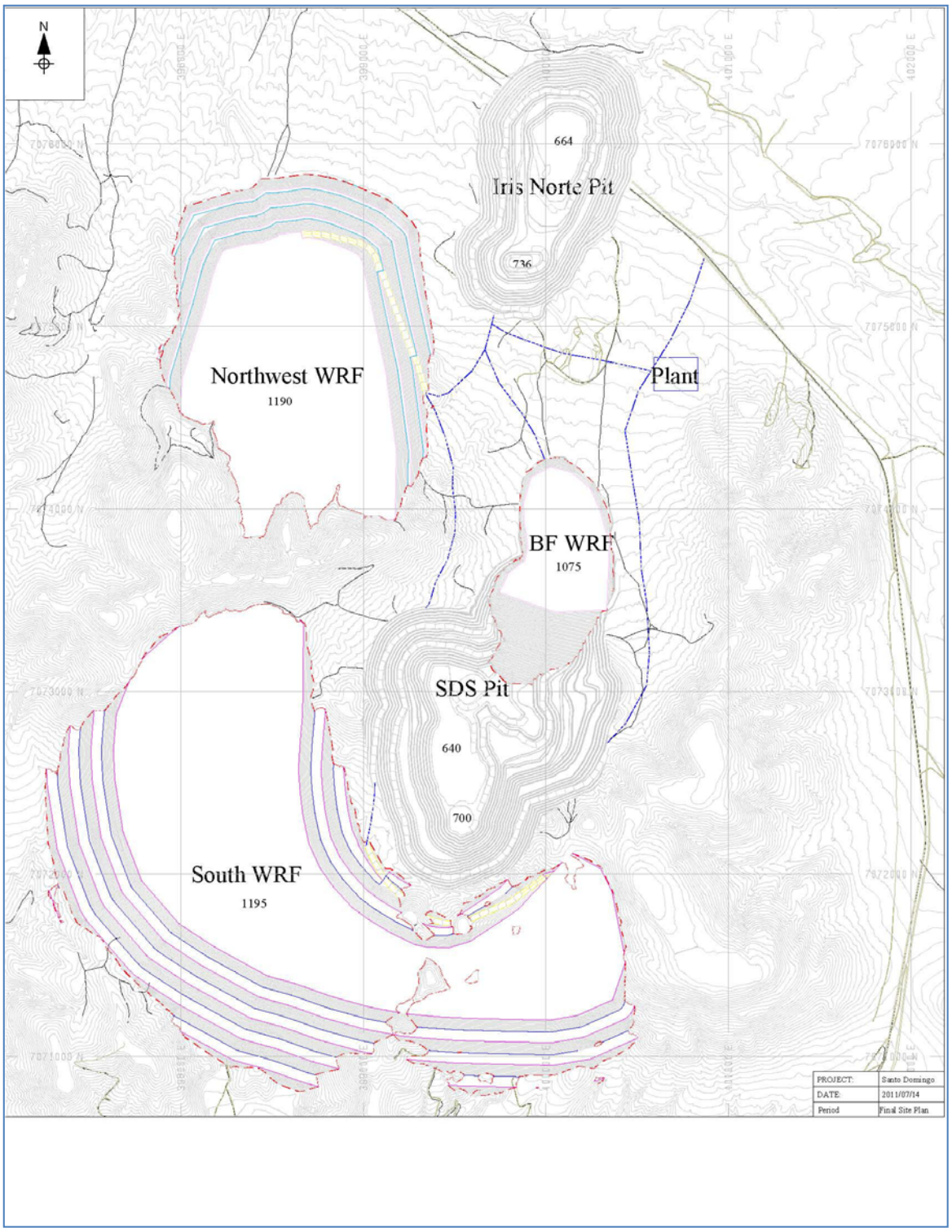


Figure 16-6: Overall Site Plan Final Configuration

The pit stages were based on the detailed pit designs. Waste material (including oxide mineralized material) will be placed in the South and Northwest waste rock facilities. Backfilling of Iris pit will also be required. All process plant feed material will be hauled directly to the plant crusher for further processing.

16.4.2 Mine Production Schedule – Open Pit

The production schedule for the Santo Domingo deposits was developed with the aid of MineSight™ software, and incorporated the open pit deposits at Santo Domingo Sur, Iris and Iris Norte. The maximum processing rate of 70 ktpd was used in the schedule. The processing rate was determined based on total iron content from the LOM schedule on an annual basis.

Open pit mining will take place sequentially with Santo Domingo Sur mined first, followed by Iris and finally Iris Norte. There will be some overlap between these pits in order to provide adequate mill feed and to balance waste stripping requirements.

The average maximum production rate from the Santo Domingo open pits is approximately 285 ktpd. Only indicated resources were used in the LOM plan.

Table 16.6 summarizes the total material movement by year for the mine production schedule.

Table 16.6: Open Pit Production Schedule – Santo Domingo Deposits

			YEAR																		
Parameter	**Unit**	**Total**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**
OPEN PIT MINING																					
ORE																					
SDS/IRIS ORE	Mt	**344.8**		15.3	25.6	25.6	25.6	25.6	25.5	24.4	23.9	24.4	24.0	24.2	21.1	19.6	18.0	11.8	10.5		
Cu	Mlbs	**2,684**		212	354	323	279	231	191	151	143	125	117	118	111	105	89	64	73		
Cu	%	**0.35**		0.63	0.63	0.57	0.50	0.41	0.34	0.28	0.27	0.23	0.22	0.22	0.24	0.24	0.22	0.25	0.32		
Magnetite Conc.	Mt	**56.6**		1.6	2.8	3.2	4.4	4.2	4.2	3.3	3.4	4.4	4.4	4.6	4.5	4.7	3.9	1.8	1.2		
Recoverable Fe to Magnetite Conc.	%	**10.67**		6.71	7.25	8.05	11.22	10.76	10.75	8.66	9.27	11.70	12.01	12.29	13.83	15.56	14.13	10.06	7.39		
Au	koz	**539**		44	71	64	54	46	38	30	28	25	24	25	24	22	18	12	13		
Au	g/t	**0.05**		0.09	0.09	0.08	0.07	0.06	0.05	0.04	0.04	0.03	0.03	0.03	0.04	0.04	0.03	0.03	0.04		
IRIS NORTE ORE	Mt	**73.1**													0.8	2.3	3.9	11.9	12.5	21.9	19.8
Cu	Mlbs	**246**													7	13	24	58	46	66	32
Cu	%	**0.15**													0.40	0.25	0.27	0.22	0.17	0.14	0.07
Magnetite Conc.	Mt	**16.5**													0.1	0.3	0.9	2.4	2.4	5.5	5.0
Recoverable Fe to Magnetite Conc.	%	**14.67**													4.31	8.30	14.98	12.82	12.53	16.40	16.32
Au	koz	**41**													1	2	4	10	8	10	5
Au	g/t	**0.02**													0.05	0.03	0.03	0.03	0.02	0.01	0.01
Total Ore Mined	**Mt**	**418.0**		**15.3**	**25.6**	**25.6**	**25.6**	**25.6**	**25.5**	**24.4**	**23.9**	**24.4**	**24.0**	**24.2**	**21.9**	**21.9**	**21.9**	**23.7**	**23.0**	**21.9**	**19.8**
Cu	**Mlbs**	**2,930**		**212**	**354**	**323**	**279**	**231**	**191**	**151**	**143**	**125**	**117**	**118**	**118**	**118**	**112**	**122**	**119**	**66**	**32**
Cu	**%**	**0.32**		**0.63**	**0.63**	**0.57**	**0.50**	**0.41**	**0.34**	**0.28**	**0.27**	**0.23**	**0.22**	**0.22**	**0.25**	**0.24**	**0.23**	**0.23**	**0.23**	**0.14**	**0.07**
Magnetite Conc.	**Mt**	**73.1**		**1.6**	**2.8**	**3.2**	**4.4**	**4.2**	**4.2**	**3.3**	**3.4**	**4.4**	**4.4**	**4.6**	**4.5**	**5.0**	**4.8**	**4.2**	**3.6**	**5.5**	**5.0**
Recoverable Fe to Magnetite Conc.	**%**	**11.37**		**6.71**	**7.25**	**8.05**	**11.22**	**10.76**	**10.75**	**8.66**	**9.27**	**11.70**	**12.01**	**12.29**	**13.48**	**14.80**	**14.28**	**11.45**	**10.19**	**16.40**	**16.32**
Au	**koz**	**580**		**44**	**71**	**64**	**54**	**46**	**38**	**30**	**28**	**25**	**24**	**25**	**26**	**25**	**22**	**23**	**22**	**10**	**5**
Au	**g/t**	**0.04**		**0.09**	**0.09**	**0.08**	**0.07**	**0.06**	**0.05**	**0.04**	**0.04**	**0.03**	**0.03**	**0.03**	**0.04**	**0.03**	**0.03**	**0.03**	**0.03**	**0.01**	**0.01**
WASTE																					
Rock Waste	Mt	**975.6**	47.6	82.9	76.2	78.4	62.2	54.1	59.7	69.8	76.7	77.3	78.5	66.6	40.8	23.0	25.1	16.9	10.2	20.8	8.8
Overburden Waste	Mt	**193.0**	1.2	1.0	3.1	0.7	12.0	14.8	4.5		0.2	0.1	0.0	6.4	32.4	27.4	26.2	17.1	29.3	16.8	
Total Waste Mined*	Mt	**1,277.4**	**49.6**	**89.6**	**81.4**	**81.3**	**78.8**	**78.6**	**78.8**	**79.8**	**80.3**	**79.9**	**80.2**	**79.8**	**82.4**	**57.0**	**58.1**	**44.4**	**44.7**	**42.4**	**10.5**
Total Material Mined	Mt	**1,695.4**	**49.6**	**104.9**	**107.0**	**106.8**	**104.3**	**104.1**	**104.2**	**104.2**	**104.2**	**104.2**	**104.2**	**104.1**	**104.3**	**78.9**	**80.0**	**68.1**	**67.7**	**64.3**	**30.3**
Strip Ratio	t:t	**3.1**		5.8	3.2	3.2	3.1	3.1	3.1	3.3	3.4	3.3	3.3	3.3	3.8	2.6	2.7	1.9	1.9	1.9	0.5
Total Material Mined/Day	t/day	232,246	135,807	287,479	293,032	292,674	285,816	285,339	285,604	285,562	285,429	285,521	285,399	285,153	285,727	216,068	219,159	186,527	185,516	176,117	82,983
Target Processing Rate																					
Target based on % Fe and Femag%	t/day			42,000	70,000	70,000	70,000	70,000	69,758	66,914	65,508	66,726	65,690	66,413	60,000	60,000	60,000	64,922	63,032	60,000	

Note: * Total Waste Mined includes Inferred material and material below cut-off.

The Santo Domingo open pits will produce 418 Mt of mill feed and 1,277 Mt of waste rock over a 19-year mine operating life (yielding an overall strip ratio of 3.1:1 (t:t). The mine schedule focuses on achieving the required plant feed production rate, mining of higher grade material early in schedule, while balancing waste stripping requirements.

Figure 16-7 summarizes mined ore production tonnages and grades by period and area.

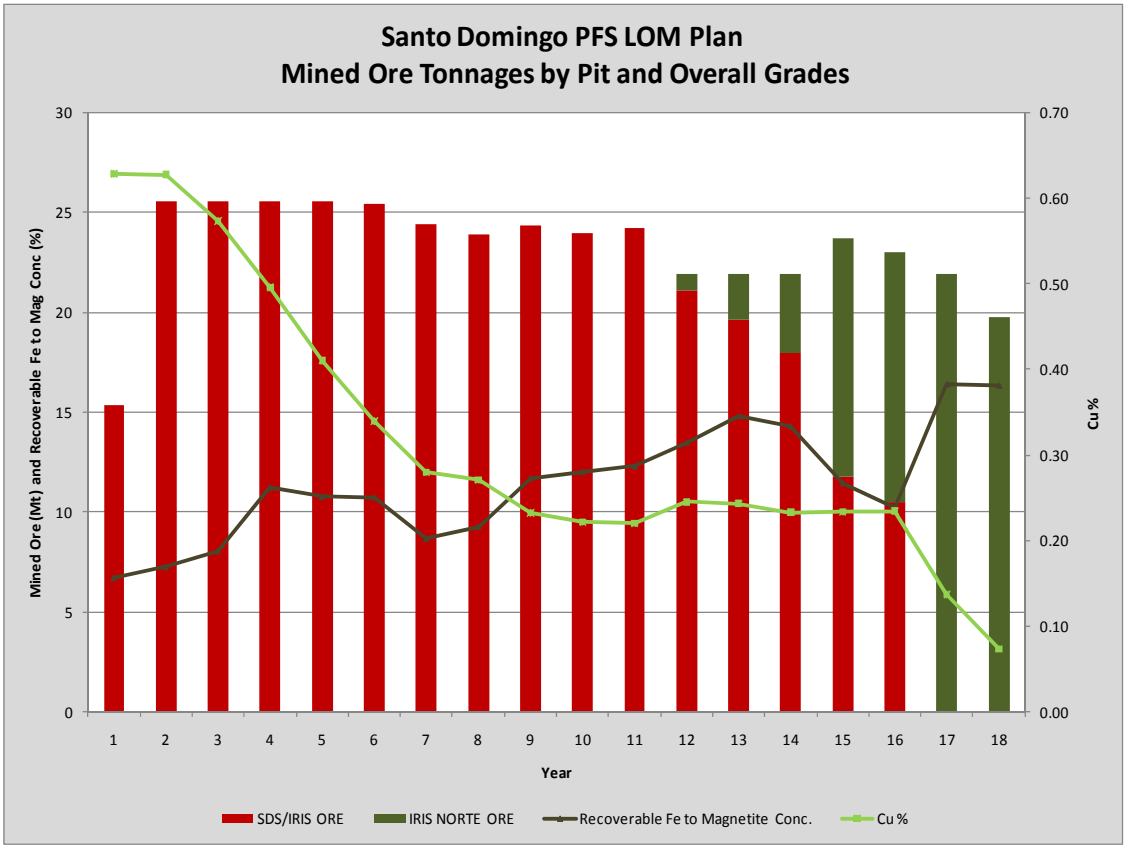


Figure 16-7: Period Ore Tonnage and Copper Grade

To further illustrate the progression of mining and processing of the Santo Domingo deposits, Figure 16-8 provides a timeline of the mine production tonnage and grade from each of the areas as well as mill feed tonnes and copper head grade. Figure 16-9 summarizes open pit annual mined benches from each stage.

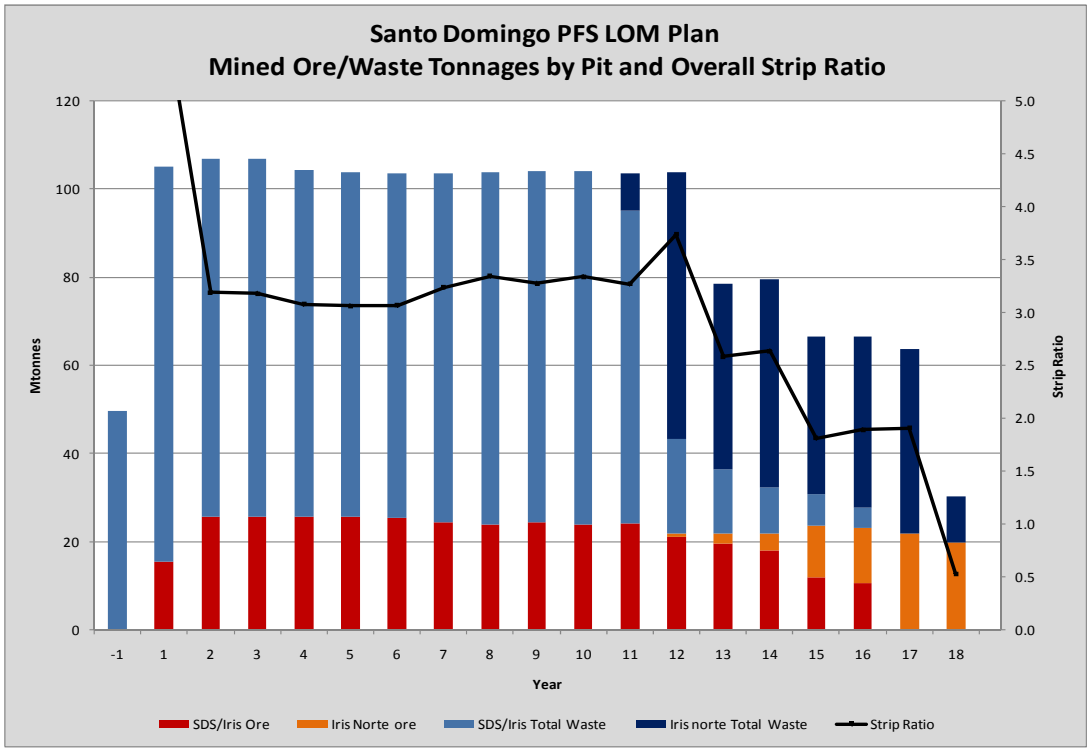


Figure 16-8: Production Timeline

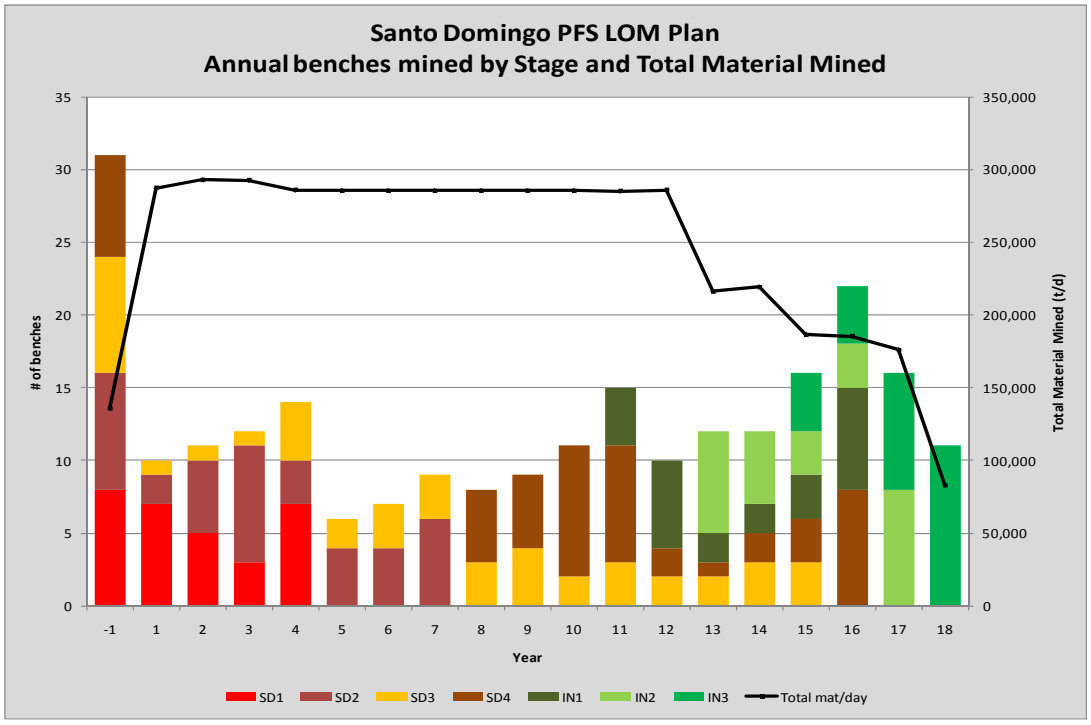


Figure 16-9: Annual Mined Benches

To further illustrate the progression of mining of the Santo Domingo deposits, Table 16.7 provides the open pit stage bottom elevation reached by the end of each period. Layout drawings with the status of the open pit configuration and waste dump advance at the end of various periods are attached in Appendix 2-3.

Table 16.7: Open Pit Development

Year	Development
Year -1	Pre-stripping of the SDS/Iris pit commences with a total of 50 Mt of wastematerial mined. Approximately 0.4 Mt of ore will be stockpiled.
Year 1	Mining continues in Stage 1 and 2 of SDS/Iris. Open pit ore production is planned to be 15.3 Mt at a strip ratio of 5.8:1 (total waste mined 90 Mt). Processing of ore commence at 60% of maximum capacity. Mined head gradeis 0.63% Cu.
Year 2	Stages 1 and 2 in SDS/Iris produce 81 Mt of waste for a 3.2:1 strip ratio. Average total mined grade is 0.63% Cu. Processing rate reaches maximum of70 kt/d.
Year 3-5	Stage 1 of SDS/Iris is completed. Mining continues in Stage 2 and commences in Stage 3. Processing mill head copper grade averages 0.49% Cuat a constant throughput rate of 70 kt/d. Average total material mined is 288 kt/dat an average strip ratio of 3.1:1.
Year 6-10	Stage 2 of SDS/Iris pit is completed during this time frame, along with continued mining in Stage 3 and 4. Mining commences at Iris Norte with pre-stripping of Stage 1. A total of122 Mt of plant feed mined in the period at an average copper grade of 0.27%Cu. Total waste tonnage is 399 Mt for an average strip ratio of 3.3:1.
Year 11-15	Stage 3 of SDS/Iris is completed with Stage 4 of SDS/Iris nearing completion. All stages in Iris Norte are active during this time period.
Year 16-19	Mill feedhead grade averages 0.24% Cu. The strip ratio averages 2.8:1 with 322 Mt ofwaste mined. Mining completed in remaining Stage 4 of SDS/Iris and three stages in Iris Norte. 65 Mt of ore mined and mill head grade decreases to0.15% Cu with a total of 98 Mt of waste mined.

Waste rock from the various open pits at Santo Domingo will be deposited in engineered waste rock facilities (“WRF”) adjacent to each of the deposits. In addition, a portion of waste rock from Iris Norte is proposed to be backfilled into the mined out Iris pit. The 31 Mt of oxide material will also be placed in these WRF to allow for potential future processing of this material.

A dump stability rating was undertaken to get a sense of what is needed in terms of the waste rock facility designs. Based on the findings, the Stability Rating is Class III, which indicates a Moderate failure hazard (according to BC Guidelines) and is based on a number of assumptions listed below.

Key assumptions are:

- maximum dump height > 200 m
- maximum dump volume > 50 Mm^3 (i.e., large dumps)
- overall slope = 22°
- foundation slope: assumed 10° to 25°
- degree of confinement: unconfined
- foundation type: assumed shallow bedrock on side slopes to deep overburden in valleys, but assumed on balance the foundation is competent
- dump material quality: assume strong durable rock with less than 10% fines
- construction method: assume bottom up but thick lifts (50 m)
- piezometric and Climatic Conditions: assumed favorable, with arid conditions
- dumping rate: assume high (110,000 to 170,000 tonnes/day)
- seismicity: high – seismic risk zone > 4
- environmental risk: assume no ARD or metal leaching issues.

Commensurate with the Stability Rating, recommendations for the WRF designs are as follows:

- build from the bottom up
- build in 50 m lifts
- build with dump face angles of 37°(or natural angle of repose)
- build in ramps to achieve a 22° (2.5:1) overall dump angle
- limit maximum dump height to 250 m

16.4.3 Waste Rock Facility Designs

South WRF

The waste rock material and oxide generated from the Santo Domingo Sur and Iris pit will be placed in the South WRF in the valley to the southwest of the pit. The dump has an overall face slope angle of 22° or 2.5:1 (toe-crest) with safety berms spaced at regular 50 m (vertical) intervals.

The maximum dump height is 235 m and has a design capacity of 365 Mm^3. Given the geotechnical constraints, the South WRF is to be built in a bottom-up approach. The maximum crest elevation is planned at 1,195 masl.

Northwest WRF

The waste rock material, and oxide, generated from the lower portions of Santo Domingo Sur and Iris pit, as well as a portion of material from Iris Norte will be placed in the Northwest WRF along the ridge to the west of the Iris Norte pit. The dump has an overall face slope angle of 22 degrees or 2.5:1 (toe-crest) with safety berms spaced at regular 50 m (vertical) intervals.

The maximum dump height is 200m and has a design capacity of 165 Mm3. Given the geotechnical constraints, the Northwest WRF is to be built in a bottom-up approach. The maximum crest elevation is planned at 1,190m.

16.4.4 Backfill Dumps

Iris pit will be partially backfilled with waste generated from subsequent mining in the Iris Norte pit. The dump has a design capacity of 110 Mm^3. The maximum crest elevation is planned at 1,075 masl.

Capacities and Sequence

Table 16.8 below summarizes the waste quantities produced by each stage of this pre-feasibility report for Santo Domingo. Material is reported in terms of type as well as tonnage and cubic metres.

Table 16.8: Waste Quantities by Stage

Stage	Overburden (Mt)	Rock Waste (Mt)	Total Waste (Mt)	Overburden (Mm3)	Rock Waste (Mm^3)	Total Waste (Mm^3)
SDS/Iris						
SDS Stage1	1	153	154	0	70	70
SDS Stage2	3	149	152	2	68	70
SDS Stage3	33	461	494	16	210	226
SDS Stage4	0	189	190	0	86	87
Subtotal SDS/Iris	**37**	**953**	**990**	**18**	**435**	**453**
Iris Norte						
IRN Stage 1	39	58	97	19	27	45
IRN Stage 2	54	43	97	26	20	45
IRN Stage 3	63	30	93	30	14	44
Subtotal Iris Norte	**156**	**132**	**287**	**75**	**60**	**135**
Grand Total	**193**	**1,084**	**1,277**	**93**	**495**	**588**

Note: Overburden density 2.70; Rock density 2.85; Swell factor 1.30.

17 RECOVERY METHODS

17.1 General

The Santo Domingo process plant design incorporates an open-air layout that minimizes the overhead cranage requirements by maximizing accessibility for mobile cranes for maintenance. This layout, shown in Figure 17-1, has taken account of the site topography and limits imposed by the preliminary locations of the pit, stockpiles, and waste dumps. Figure 17-2 shows the site general arrangement.

The concentrator will use a conventional processing flowsheet and industry standard equipment. Concentrator operation will be monitored using a control system from a centrally located control room. Sampling and stream assay monitoring will be via an automated system linked to the control system.

The process plant and associated service facilities will process ROM ore delivered to the primary crusher to produce separate copper and magnetite concentrates and tailings. The proposed process encompasses crushing and grinding of the ROM ore, copper rougher flotation, regrinding and cleaner flotation, magnetite rougher magnetic recovery on copper rougher tailings, magnetic rougher concentrate regrind, and cleaner magnetic separation. Concentrates will be thickened and stored on site prior to being pumped down a concentrate pipeline to the port for dewatering and shipping to third-party smelters.

The magnetic separator tailings will be combined with cleaner scavenger flotation tailings for thickening before placement in the TSF.

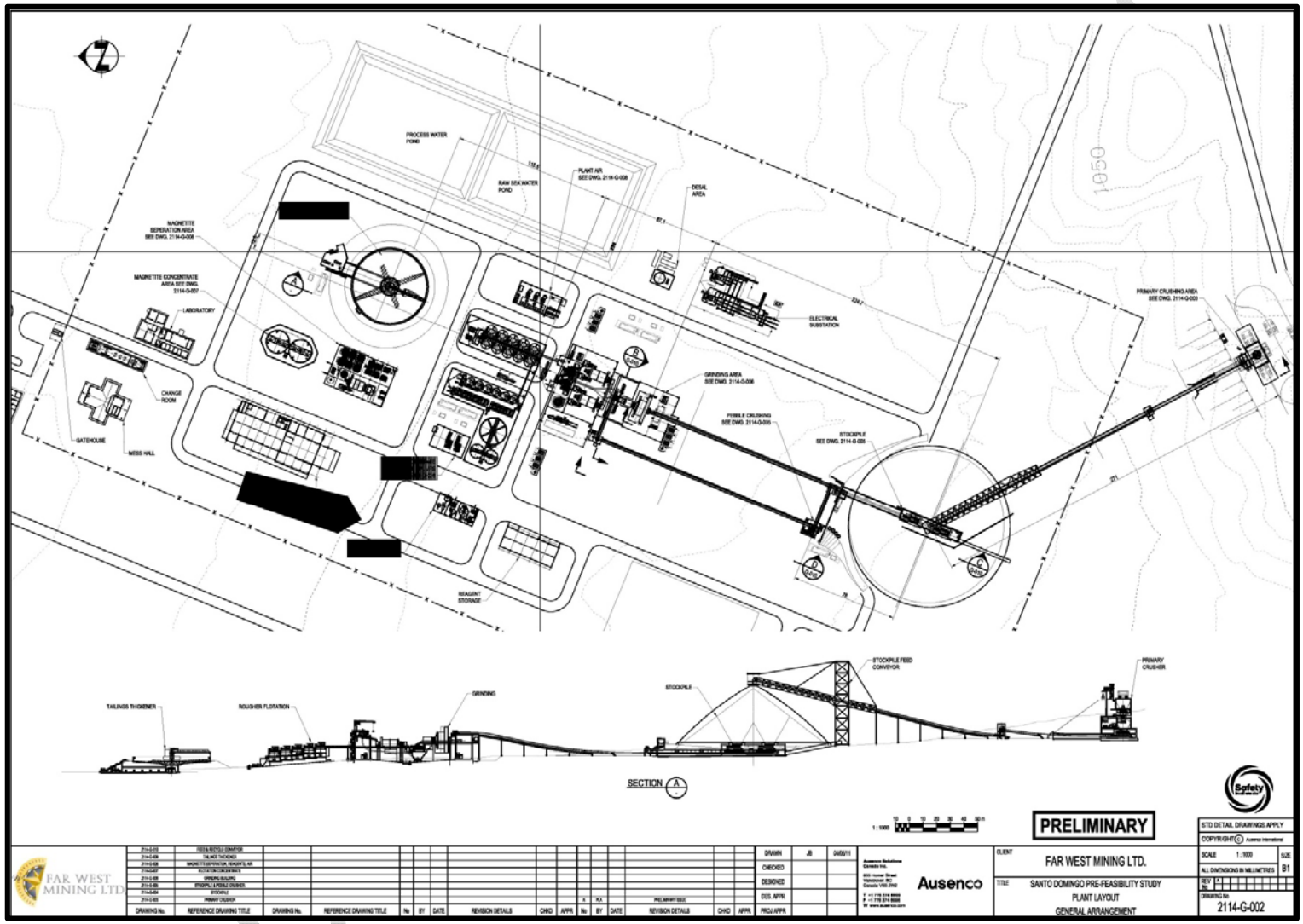


Figure 17-1: Santo Domingo Process Plant Layout

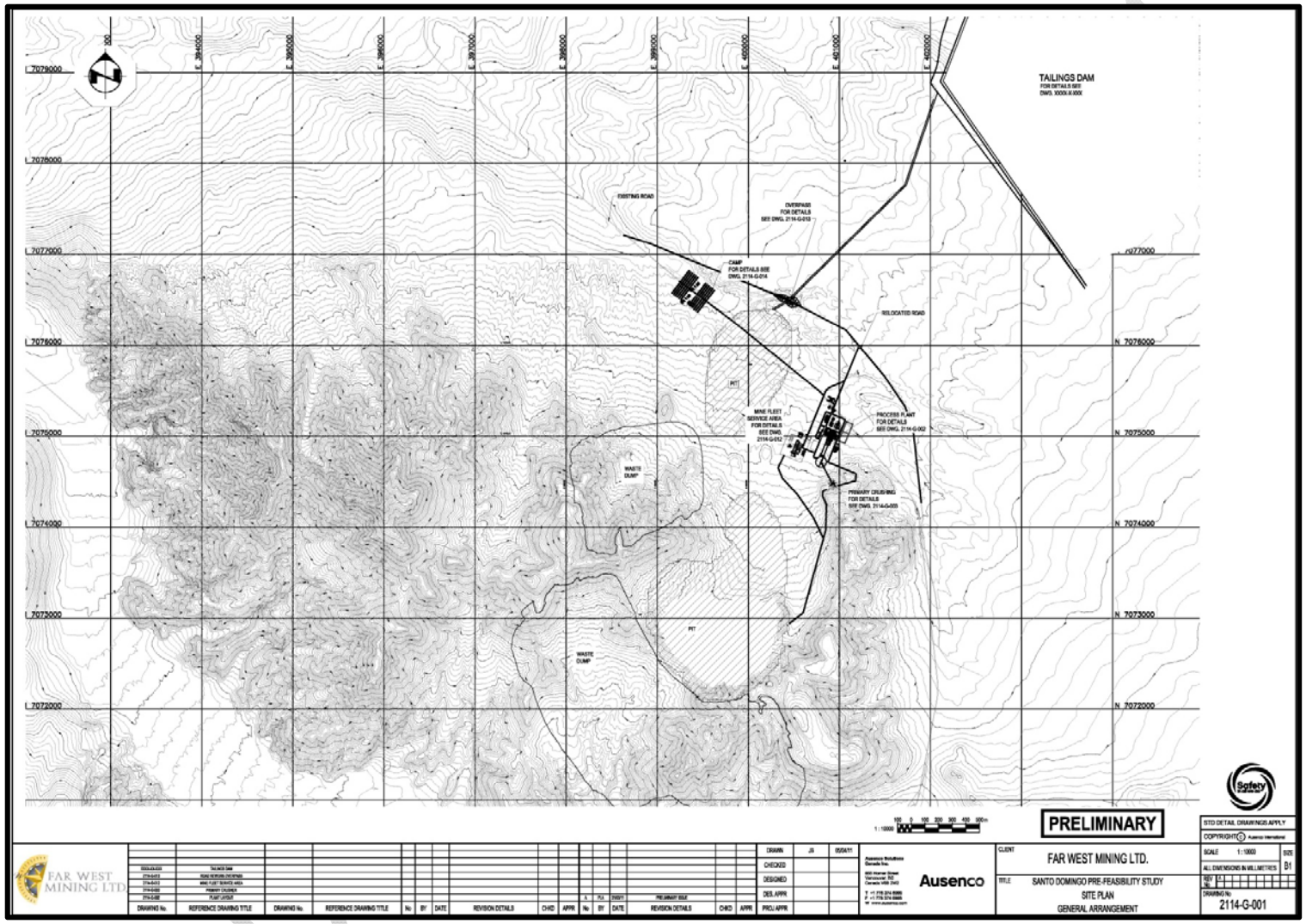


Figure 17-2: Santo Domingo Site General Arrangement

17.2 Design Criteria Summary

The overall approach was to design a robust process plant that could handle a wide range of ore variability and operating conditions and deliver good value for capital. The key project and ore-specific criteria for the plant design and operating costs are provided in Table 17.1.

Table 17.1: Summary of the Process Plant Design Criteria

Criteria		Units	Design
Crusher Feed		kt/d	60
		Mt/a	21.9
Crusher Availability		%	63.25
Crusher Throughput		t/h	3922
Crusher Selection	Size		60” x 89”
	No		1
Mill Throughput		Mt/a	21.9
Mill/Flotation Availability		%	93
Mill Throughput		t/h	2 688
Physical Characteristics	BWI	kWh/t	15.1
	RWI	kWh/t	17.0
	CWI	kWh/t	14.8
	DWI		7.0
	Specific Gravity	t/m^3	3.30
Grind Size	P_{80}	µm	180
Head Grade (Design)		% Cu	0.70
		% S	2.80
		% Fe	35.7
		g/t Au	0.11
Flotation Recovery	Copper	%	89
	Gold	%	60
Magnetite Recovery	Mass	%	20
	Iron	%	42
Cu Circuit Residence time	Roughers	min.	32.5
	Cleaner 1	min.	20
	Cleaner Scav.	min.	10
	Cleaner 2	min.	12.5
	Cleaner 3	min.	10
Magnetite Circuit Recovery	Roughers	Mass %	27
	Cleaner 1	Fe %	98.2
	Cleaner 2	Fe %	98.5
	Cleaner 3	Fe %	99.0

Criteria		Units	Design
Cu Concentrate Filtration Rate		$kg/m^2/h$	495
Concentrates Thickening Flux		$t/m^2/h$	0.25
Magnetite Concentrate Filtration Rate		$kg/m^2/h$	1000
Tailings Thickening Flux		$kg/m^2/h$	1.0
Tailings Thickener Underflow Density		% w/w	60
Collector Consumption (Aerofloat 3926)		g/t	20
Collector Consumption (Aerophine 3418A)		g/t	10
Depressant Consumption (NaHS)		g/t	0
Depressant Consumption (NaCN)		g/t	25
Frother Consumption (MIBC)		g/t	50
Lime Consumption		kg/t	0.150
Flocculant Consumption (Concentrate and tailings)		g/t	20
SAG Mill Media Consumption		kg/t	0.320
Ball Mill Media Consumption		kg/t	0.364

17.3 Plant Design Basis

The key criteria selected for the plant design are:

- a nominal plant treatment rate of 60 000 t/d (60 kt/d), with the ability to handle increased throughputs of up to 70 kt/d for softer ores
- design availability of 93% (after ramp-up), which equates to 8,147 operating hours per year, with standby equipment in critical areas
- sufficient plant design flexibility for treatment of all ore types at design throughput.

The selection of these parameters is discussed in detail below

17.4 Throughput and Availability

FWM nominated the selection of a 38 ft diameter SAG mill and pebble crushing for this plant. Ausenco believes that this circuit is suitable to achieve a throughput of 60 kt/d for design competency ore, with increased throughputs of up to 70 kt/d for softer ores. Ausenco has nominated an overall plant availability of 93% or 8,147 h/a. This is an industry standard for a large, multi-train flotation plant with moderately abrasive ore. Benchmarking indicates that similar plants have consistently achieved this level.

17.5 Processing Strategy

The main determinant of ore competency is the amount of hematite and magnetite in the ore as measured by the iron assay. This is a more reliable predictor of throughput than process design is based on treating the different sample types tested individually at the nominated

design throughput rates. Typically, the range in variability of ore parameters such as hardness and head grade during process design are considered. However, due to the preliminary nature of the mining schedule and metallurgical testwork, the most competent and hardest of the three ore types, identified by FWM have been used in the process design criteria.

17.6 Head Grade

The plant is designed to treat various tonnages of primary ore with a maximum head grade of 0.7% Cu and a magnetite concentrate mass recovery of 20% (equivalent to approximately 20% magnetite in feed).

17.7 Flowsheet Development and Equipment Sizing

The process plant flowsheet design for the Santo Domingo circuit was conceptually based on those of comparable large flotation plants. Figure 17-3 shows a process schematic for the Santo Domingo plant.

Details of the flowsheet design and selection of major equipment for the various options are discussed in the sections below.

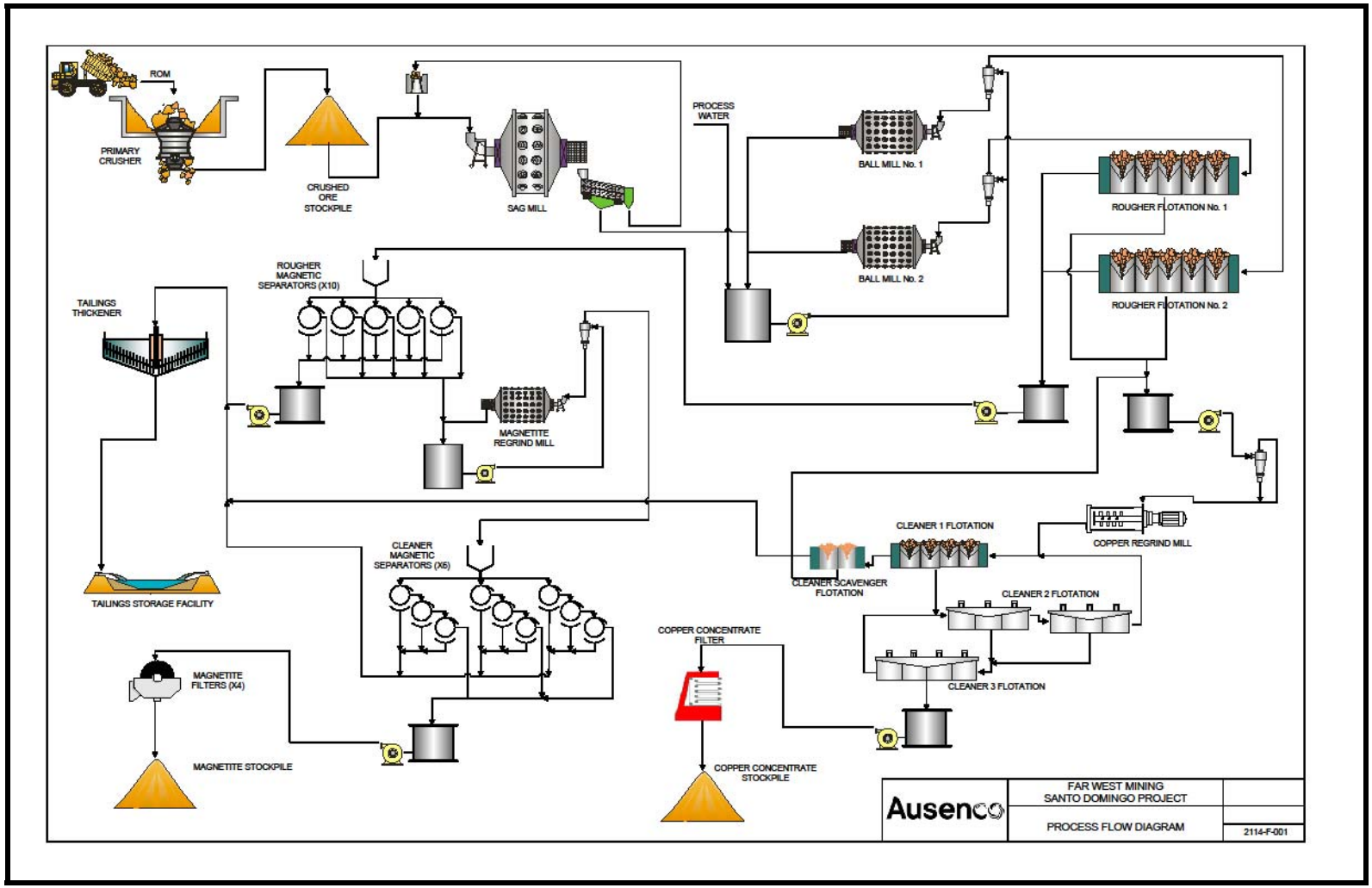


Figure 17-3: Santo Domingo Process Plant Schematic

17.8 Unit Process Selection

The process plant design is based on a flowsheet with unit process operations that are well proven in the minerals processing and sulphide flotation industries, and incorporating the following unit process operations:

- ore from the open pit is crushed using a primary gyratory crusher to a crushed product size of nominally 80% passing (P_{80}) 100 mm and fed onto the stockpile feed conveyor
- conical stockpile of crushed ore with a live capacity of 18 h, with two apron feeders, each capable of feeding 120% of the full mill throughput
- a 22 MW SAG mill, 11.58 m diameter (38 foot) with 6.6 m EGL (21.5 foot), in closed circuit with pebble crushing
- pebble crushing will comprise a single 750 kW cone crusher (with a bypass for crusher maintenance), crushing to a product size of nominally 80% passing (P_{80}) 12 mm
- two 12 MW Ball mills per grinding train, 7.32 m diameter (24 foot) with 10.97 m EGL (35.7 foot), in closed circuit with hydrocyclones, grinding to a product size P_{80} of nominally 180 µm
- rougher concentrate regrinding stage in one of 3.0 MW horizontal stirred mill, grinding to a P_{80} of 30 µm
- copper cleaner 1 and cleaner scavenger flotation consisting of six 200 m^3 forced air tank flotation cells to provide a total of 31 minutes of combined retention time
- copper cleaner 2 flotation stage consisting of six 38 m^3 trough-shaped flotation cells to provide a total of 12.5 minutes of retention time
- copper cleaner 3 flotation stage consisting of four, 38 m^3 trough-shaped flotation cells to provide a total of 10 minutes retention time
- copper concentrate thickening in a 20 m diameter high-rate thickener
- magnetic separation on flotation rougher tailings, consisting of ten 3.6 m long rougher magnetic separators for a mass recovery of 27% of separator feed (23.8% of flotation feed)
- rougher magnetite concentrate regrinding consisting of a single 6 MW mill, 6.1 m diameter and 9.75 m EGL, in closed circuit with hydrocyclones, grinding to a P_{80} of 50 µm
- cleaner magnetite magnetic separation consisting of six 3.6 m long, three-stage cleaner magnetic separators for an overall iron recovery of 96% at a concentrate grade of 65% Fe
- thickening of rougher magnetic separator tailings, cleaner magnetic separator tailings and cleaner scavenger flotation tailings, in a 60 m diameter high-rate thickener to an underflow density of 60% solids
- Tailings Storage Facility (TSF) for process tailings deposition in a conventional dam
- copper and magnetite concentrate storage on site in agitated tanks prior to pumping to the port site
- 70 km concentrate pipeline
- copper concentrate filtration at the port in a horizontal plate and frame pressure filter
- magnetite concentrate filtration at the port in four Ceramec 144 vacuum disc filters
- 70 km seawater pipeline to site to provide water

- seawater and distribution system for reticulation of seawater throughout the plant as required
- process water and distribution system for reticulation of process water throughout the plant as required. Process water is supplied from water reclaimed from the TSF and process operations, with seawater used as make-up water as required
- potable water is generated by treatment of seawater in a reverse osmosis (RO) unit at the process plant. Potable water is distributed to the plant, used to supply the town of Diego de Almagro, and for miscellaneous purposes around the site
- plant, instrument, and flotation air services and associated infrastructure.

17.9 Comminution Circuit Sizing

17.9.1 Primary Crushing

Based on the design throughput and ore characteristics, a gyratory crusher is considered the most suitable primary crusher for the duty. The primary crusher, 60" x 89", will be located at the edge of the ROM pad. A buried crusher design has been selected to minimize the requirements for expensive retaining walls. Although the design throughput of 60 kt/d would normally be considered high for this a machine of this size, the high density and moderate competency of the Santo Domingo ores will allow this machine to operate at these throughputs.

17.9.2 Stockpile Feed Conveyor and Coarse Ore Stockpile

Crusher product will be reclaimed from the crusher discharge vault by the variable-speed primary crusher discharge apron feeder, which will discharge onto the sacrificial conveyor before being transferred via a stockpile feed conveyor to the mill stockpile.

The stockpile will provide a minimum of 16 h live capacity at the design SAG mill feed rate of 60 kt/d, higher throughputs will reduce this capacity. The total capacity of the stockpile is approximately 3.5 days of nominal SAG mill feed capacity. Two apron feeders have been selected to reclaim ore from the stockpile, each able to deliver 120% of the design mill feed rate.

The stockpile will be covered to minimize fugitive dust emissions. The cover will be a circular dome with an ogival profile and double-layer, Vierendeel-type geometry. It will be 110 m in diameter and 41.5 m in height.

17.9.3 Comminution Circuit

Comminution Design Criteria

The major comminution design parameters used for this study are:

- Bond Ball mill Work Index (BWI) of 15.1 kWh/t based on the samples tested at SGS

- Drop Weigh Index (DWI) of 7.0 kWh/m^3 as measured from the SMC test (equivalent to an Axb of 47.5 at a specific gravity of 3.30)
- target grind size P_{80} of 180 µm, based on the SGS flotation testwork.

The grinding circuit was designed to meet the various requirements in tonnage outlined by FWM. FWM nominated a mill design to achieve a design throughput of 60 kt/d, with the volumetric and materials handling capability of treating up to 70 kt/d on softer ores. FWM also nominated a maximum SAG mill size of 38 ft in diameter.

Flotation testwork and mineralogy have indicated that Santo Domingo ores are moderately fine-grained, and rougher flotation recovery is relatively insensitive to grind sizes up to about 150 to 200 µm. In order to achieve the specified design metal recoveries, FWM has nominated a primary grind size target of P_{80} of 180 µm.

SAG and Ball Mill Design

The SAG mill feed weightometer will be installed on the SAG mill feed conveyor to provide feed rate data for control of the reclaim feeders. The reclaimed crushed ore will be fed at a controlled rate to an 11.58 m diameter by 6.60 m effective grinding length (EGL), 22 MW SAG mill. The SAG mill will be equipped with a gearless motor, variable-speed drive system.

Discharge from the SAG mill will gravitate through a trommel and vibrating screen, then into a common mill hydrocyclone feed hopper. Oversize pebbles from the vibrating screen (scats) will be recycled back onto the mill feed conveyor where, after magnetic separation of metal milling remnants, they will be reintroduced into the mill and crushed to below nominally 12 mm in a pebble crushing circuit. Undersize from the SAG trommel and screen will gravity flow into the hydrocyclone feed hopper.

The mills discharge slurry will be pumped via dedicated hydrocyclone feed pump to the two mill hydrocyclone clusters, each operating in a closed-circuit configuration with a single ball mill. Water is added to the hydrocyclone feed hopper as needed to achieve the required hydrocyclone feed pulp density.

Hydrocyclone underflow from each cluster will gravitate to a dedicated 12 MW ball mill, each nominally 7.32 m diameter by 10.97 m EGL, with twin 6.0 MW motors operating in parallel. Discharge from each ball mill will gravity flow into the hydrocyclone feed hopper for reclassification. Hydrocyclone overflow will gravitate to the copper flotation circuit.

The specific energy and mill sizing determined using Ausenco's in-house method for the major ore types is shown in Table 17.2.

Table 17.2: Mill Design Criteria

Criteria		Units	Design
Throughput		t/h	2,688
Mill Type			SAG Grate D/C
Shell Power required		kW	18,547
Mill Speed		% Nc	75
Ball Charge Volume	Nominal, operating	% vol	15
	Maximum for design	% vol	20
Total Charge Volume	Nominal, operating	% vol	27
	Maximum for design	% vol	30
Mill Diameter	Inside shell	m	11.58
Mill Length	EGL	m	6.60
Installed Motor power		kW	22,000
Mill Type			Ball
Grind Size	P_{80}	µm	180
Pinion Power required		kW	21,907
No. of Mills			2
Mill Speed		% Nc	75
Ball Charge Volume	Nominal, operating	% vol	30
	Maximum for design	% vol	35
Mill Diameter	Inside shell	m	7.32
Mill Length	EGL	m	10.97
Installed Motor Power		kW	12,000

Installed ball mill power of 12,000 kW incorporates the allowances for drive train losses to determine the motor power from the pinion power, as well as a 10% design contingency to account for the accuracy of the models, calculations, and testwork used to determine the expected average pinion power.

The installed motor power for the SAG mill incorporates similar allowances, as well as an additional contingency to allow adjustment in the mill operating conditions to handle ore variability. These allowances and contingencies require the installation of 22,000 kW.

17.9.4 Pebble Crushing

The upfront circuit design will incorporate pebble crushing, with conveyors returning the crushed pebbles to the SAG mill feed. The pebble crushing circuit will consist of a single 675 kW (800 hp) crusher. An electromagnet, up-stream of the pebble crusher, will be used to remove tramp metal (typically SAG mill balls detritus) to protect the crusher.

A pebble circulating load of 25% of the new feed rate has been assumed for the design of the pebble crushers, based on typical industry experience with ores of similar competency. The conveyors are designed to handle peak loads of up to 35% of new feed.

17.9.5 Mill Circuit Classification

The classification circuit has been designed for a maximum circulating load of 300%; this is a typical design value for a moderately coarse grind size, widely used within industry for SABC circuits. Due to the high volumetric flow anticipated, the SAG mill discharge slurry first passes through a trommel, then a vibrating screen with 12 mm apertures to remove pebbles; the undersize flows into the hydrocyclone feed hopper.

SAG and ball mill discharge will be combined in the hydrocyclone feed hopper and then pumped to two clusters of 760 mm diameter hydrocyclones to split the slurry at approximately P_{80} of 180 µm. Each hydrocyclone cluster will be fed with a dedicated, hydrocyclone feed pump.

Fine hydrocyclone overflow will report as flotation feed, while the coarse hydrocyclone underflow from each of the two clusters will report to a single ball mill for further grinding.

The milling circuit will require the installation of two clusters of ten 760 mm hydrocyclones, of which up to 7 per cluster will be in operation at any one time.

17.10 Flotation Circuit Design

Mineralogical examination has highlighted that the copper and magnetite mineralogy are moderately coarse grained. Flotation testwork has indicated that, while good rougher recovery can be achieved at a moderately coarse grind size, concentrate regrinding is required to achieve saleable concentrate grades.

The ores contain pyrite, but little or no talc or clay minerals. The ores will require a pyrite rejection reagent scheme and fine grind, but should be easily amenable to conventional flotation separation to produce saleable concentrate grades.

17.10.1 Circuit Type and Size

The design flowsheet selected consists of copper rougher flotation, copper concentrate regrind, cleaner 1, cleaner scavenger, cleaner 2 and cleaner 3 flotation stages; followed by magnetite recovery on the copper rougher tailings. Cleaner scavenger tailings will report directly to the final tailings thickener. Copper cleaner 3 concentrate will report to the copper concentrate thickener. The residence times and flotation parameters for the copper flotation circuit have generally been based on the testwork parameters obtained on the various Santo Domingo ores. The testwork flotation and design residence times are summarized in Table 17.3.

Table 17.3: Summary of Copper Flotation Residence Times

Flotation Time	Locked-Cycle Tests (min)	Scale Factor	Specified Design (min)	Actual Design (min)
Copper Rougher	13	2.5	32.5	37.5
Copper Cleaner 1[19]	8	2.5	20	18.4
Copper Cleaner Scavenger	4	2.5	10	12.6
Copper Cleaner 2	5	2.5	12.5	14.4
Copper Cleaner 3	4	2.5	10	38.2

17.10.2 Flotation Circuit Configuration

Hydrocyclone overflow will gravitate to parallel rougher feed boxes, where flotation reagents, frother, and dithiophosphate collector will be added. The feed boxes will flow to the copper rougher flotation cells, which are connected in series. Ten 300 m^3 forced-air tank cells have been selected to provide the required residence time for the rougher flotation. The cells will be configured as two parallel trains with a step in level between each cell. Additional dosing points for frother, dithiophosphate collector, and Aerofloat 3926 collector will be located at points along the rougher banks.

Each train consists of five individual cells. Twinning of rougher cells is a common practice used to reduce the cell capital cost and potentially reduce the total installed cost, by reducing the number of steps, and hence the structure required to support the rougher train. Single cells have metallurgical advantages over twinned cells in terms of reduced short-circuiting and better level control.

The current Santo Domingo design is based on moderately-sloping ground with a grade very close to matching that of a single-cell configuration; hence, the cell capital savings are likely to be outweighed by the structural cost savings. The metallurgical benefits of this option have not been quantified.

Concentrate from each train of the bulk rougher cells will be pumped to the regrind circuit. Each copper rougher train will have one on-duty and one standby concentrate pump. Rougher tailings will gravitate to the tailings thickener.

Regrind hydrocyclone overflow and mill discharge, as well as copper cleaner 2 tailings, will be mixed with flotation reagents (frother, dithiophosphate collector, and dithiophosphinate collector) in the cleaner 1 feed box before flowing into the copper cleaner 1 cells. The copper cleaner 1 flotation circuit consists of four individual, 200 m^3 tank flotation cells. The copper cleaner scavenger flotation circuit consists of an additional two individual 200 m^3 tank flotation cells for a total of six cells.

Concentrate from the copper cleaner 1 flotation cells will gravitate to the copper cleaner 1 concentrate hopper, flowing to the copper cleaner 2 feed pump. Tailings from the copper cleaner 1 flotation cells will gravitate to the copper cleaner scavenger flotation cells. The

[19] Combined actual residence time in Cleaner 1 and cleaner scavenger exceeds specified design residence times.

copper cleaner scavenger flotation cells will consist of two 200 m^3 forced air tank flotation cells to provide the required residence time.

Concentrate from the copper cleaner scavenger cells will gravitate to the copper cleaner scavenger concentrate hopper, flowing to the copper cleaner scavenger concentrate pump. The copper cleaner scavenger concentrate pumps deliver concentrate slurry to the regrind hydrocyclone feed hopper. Tailings from the copper cleaner scavenger cells will gravitate to the copper cleaner scavenger tailings hopper, and then be pumped to the plant tailings thickener for disposal.

The copper cleaner 2 cells will be fed with copper cleaner 1 concentrate and copper cleaner 3 tailings. The copper cleaner 2 cells will consist of six 38 m^3 forced air trough flotation cells.

Concentrate from the copper cleaner 2 cells will report to the copper cleaner 2 concentrate hopper, and will then be pumped by the copper cleaner 3 feed pump to the first copper cleaner 3 flotation cell. Tailings from the copper cleaner 2 cells will gravitate to the copper cleaner 1 distributor box.

The copper cleaner 3 cells will consist of four 38 m^3 forced air trough flotation cells. The copper cleaners 2 and 3 cells will be configures with a step in level between each bank of cells.

Concentrate from the copper cleaner 3 cells will report to the copper cleaner 3 concentrate hopper, and will then be pumped to the concentrate thickener feed box. Tailings from the copper cleaner 3 cells will gravitate to the copper cleaner 1 concentrate hopper, to be fed into the copper cleaner 2 flotation circuit.

17.11 Copper Concentrate Regrind

Metso using the jar mill test conducted copper concentrate regrind testing. The jar mill test resulted in a specific power of 7.97 kWh/t from an F_{80} of 180 µm to a P_{80} of 30 µm; this equates to a BWI of 7.4 kWh/t. As the average for the feed samples tested in the comminution testwork program is 12.0 kWh/t, this indicates that the copper mineralogy may be softer than the whole ore. The power prediction from the jar mill test is lower than would typically be expected from an ore of this competency, as is reflected by the similarity between the rougher feed and concentrate P_{80}s, both being 180 µm.

Metso applies a 65% power reduction for scale-up of the Vertimill® to full-scale operation. This reduction results in an estimated Vertimill® specific power of 5.18 kWh/t.

Copper cleaner-scavenger concentrate and copper concentrate from the rougher cells will be pumped to the regrind hydrocyclone cluster, consisting of twenty-six 250 mm hydrocyclones, with four on standby. Regrind hydrocyclone overflow gravitates directly to the copper cleaner 1 flotation bank. Cyclone underflow gravitates to the regrind mill. Both the hydrocyclone overflow and regrind mill discharge gravitate to the regrind discharge hopper, and are then pumped to the copper cleaner circuit.

Both lime for pH control and NaCN for pyrite depression will be added to the hydrocyclone feed hopper. Dithiophosphate collector will be added to the regrind discharge hopper.

Based on a design rougher concentrate mass recovery of 323 t/h (equivalent to 12% mass recovery at 60 kt/d), the required regrind net power (excluding the Metso power reduction) is calculated to be 2,560 kW. For this power requirement, a single M10 000 ISAmill™ with a 3,000 kW motor has been selected for this application. Space has been left in the layout for the potential addition of a second regrind mill.

Ausenco recommends that additional copper concentrate regrind testing should be conducted during the next phase of testwork to confirm the ISAmill™ regrind power requirements with testing by Xstrata process technology.

17.12 Magnetite Recovery

Magnetite recovery testwork has been conducted by SGA in Germany. The testwork has confirmed the recovery potential for the magnetite in the Santo Domingo ore body and the regrind requirements to achieve a suitable concentrate grade, but has not identified metallurgical operating parameters for the Santo Domingo ore.

The magnetite flowsheet has been designed based on comparison with similar operations and benchmarking, typical design and scale-up parameters, and feedback from equipment vendors.

The selected design criteria are summarized in Table 17.4.

Table 17.4: Summary of Magnetite Recovery Circuit Design Loadings

	Magnet Strength Gauss	Linear Loading t/h/m (drum)	Volumetric Loading	Magnet Configuration
Magnetite Roughers	1,000	30	110	Co-current
Magnetite Cleaner 1	750	26	105	Counter Current
Magnetite Cleaner 2	700	26	105	Counter Current
Magnetite Cleaner 3	650	26	105	Counter Current
Dewatering Magnets	1,000	50	200	Radial Pole

Copper rougher flotation tailings will be pumped to the magnetite recovery circuit. Rougher magnetite feed will be pumped to a distributor to feed the individual rougher magnetic separators. Ten of the largest (at 3.6 m length and 1.2 m diameter) commercially-proven rougher magnetic separators are required to treat the rougher tailings for the 60 kt/d plant throughput case. Space in the layout has been allowed for adding two additional units to handle higher rougher mass recoveries or throughputs.

The rougher magnets will operate in a concurrent configuration. This configuration will maximize recovery of magnetic material. After regrinding, the rougher magnetite concentrate will be cleaned in three-stage cleaners. These magnets will operate in counter-current configuration to maximize rejection of non-magnetics and maximize the concentrate grade. Dilution water will be added to the feed of each stage to facilitate separation.

Magnetite rougher tailings will be pumped to the tailings thickener for dewatering prior to disposal. Magnetite rougher concentrate will be reground in the magnetite regrind circuit prior to cleaning.

The three magnetite cleaner stages will operate as open circuits, with each tailings stream reporting to the magnetite cleaner tailings. Tailings from the magnetite cleaner circuit will report to the final tailings thickener for dewatering prior to disposal.

Cleaner 3 magnetite concentrate will be pumped to the magnetite concentrate stock tanks for storage, prior to being pumped to the port through the concentrate slurry pipeline. Two magnetite dewatering magnets are also included prior to the stock tanks to ensure that the pipeline concentrate density target of 60% w/w solids is maintained.

17.13 Magnetite Rougher Concentrate Regrind

The magnetite rougher concentrate will be diluted in the rougher concentrate hopper and then pumped to the magnetite regrind hydrocyclone cluster, consisting of sixteen 380 mm hydrocyclones, with three on standby. Regrind hydrocyclone overflow gravitates directly to the magnetite cleaning circuit. Cyclone underflow gravitates to the regrind mill. The regrind mill discharge gravitates to the hydrocyclone feed hopper regrind discharge hopper for reclassification.

The design rougher concentrate stream for regrinding is 640 t/h; at the average BWI of 12 kWh/t for the feed samples tested in the comminution testwork program, this equates to a specific power requirement of 8.0 kWh/t to achieve the nominated regrind size of P_{80} of 50 µm. On this basis, a single 6,000 kW regrind ball mill has been selected for this study.

The Metso jar mill testwork resulted in a specific energy of 11.60 kWh/t from an F_{80} of 180 µm to a P_{80} of 40 µm, which is equivalent to a BWI of 13.9 kWh/t. This suggests that the magnetite concentrate may be harder than the selected design basis. Additional testwork and investigation is needed to confirm the selection.

17.14 Copper Concentrate Thickening and Storage

Copper flotation concentrate will be thickened to approximately 60% w/w solids in a 20 m diameter aboveground high-rate thickener. The thickener has been designed based on a typical settling rate of 0.25 t/m^2/h for copper concentrates. Thickening test work conducted by Outotec has confirmed the suitability of these criteria for the selection of the concentrate thickener.

Copper concentrate from the copper flotation circuit will be pumped to a high-rate concentrate thickener via the concentrate thickener feed box. The thickener will incorporate an auto-dilution feed well and rake lift mechanism. Flocculant solution will be dosed to the thickener feed box and feed well by a variable-speed flocculant dosing pump to aid in settling of concentrate, and to provide necessary clarity in the thickener overflow.

Thickener overflow will be pumped to the process water pond for storage and re-use in the circuit, while thickener underflow will be pumped to the copper concentrate storage tank by variable-speed, concentrate thickener underflow pumps.

Twin copper concentrate storage tanks with total live volumes of 800 m^3 each will provide storage capacity ahead of the concentrate pipeline. The specification of 30 h residence time is required to allow flexibility to batch the concentrate down the pipeline once daily.

17.15 Magnetite Concentrate Thickening and Storage

Cleaner 3 magnetite concentrate will be pumped to the magnetite concentrate stock tanks for storage, prior to being pumped to the port through the concentrate slurry pipeline. Two magnetite dewatering magnets are also included prior to the stock tanks to ensure that the pipeline concentrate density target of 60% w/w solids is maintained.

The dewatering magnets also have the facility to deposit concentrate into a bunker for dry stacking on site. This facility will allow magnetite production to exceed pipeline and port capacity for extended periods of time and assist in maximizing concentrate production.

Twin magnetite concentrate storage tanks with total live volumes of 3,500 m^3 each will provide surge capacity ahead of the concentrate pipeline. The specification of 16 h residence time is required to allow flexibility to ensure pipeline and/or filter plant availability does not diminish plant operating utilization.

17.16 Tailings Disposal

Plant tailings slurry, consisting of magnetite rougher and cleaner tailings and copper cleaner scavenger tailings, will be pumped to a high-rate tailings thickener. Thickener underflow, expected to be in the range of 60 to 65% w/w solids, will be pumped to the TSF. Process water from the thickener overflow will flow by gravity to the plant process water pond.

Decant water will be pumped from the tailings decant water tank by the process water pond feed pump (one duty) to the plant process water pond. The plant process water tank feed pump (one duty) will deliver decant water from the storage dam to the plant process water pond.

Due to the preliminary nature of this study, no tailings thickening testwork has been undertaken. Ausenco has based the design on experience with similar style concentrators to derive a conceptual design. These assumptions will require testwork and confirmation during subsequent detailed design.

The design basis chosen for this level of study includes a high-rate tailings thickener, with disposal of thickened tailings in a nearby TSF and recovery of water from the TSF surface. The tailings thickener design has been based on settling specifications of 1 $t/m^2/h$. This results in the requirement of a 60 m diameter tailings thickener.

17.17 Seawater, Fresh, and Process Water

The water systems will consist of:

- high pressure/low volume plant seawater water system for the mill and rougher flotation areas (mainly hose down and spray water)
- low pressure/high volume plant process water system for the mill and rougher flotation areas
- fresh water system for reagent mixing and potable water for site needs.

The majority of the plant water requirements will be met from the plant process water system, which will be composed of recycled water streams (mainly thickener overflows and TSF return water) supplemented with seawater as required. Seawater will be sourced from the seawater storage facility and used for unit processes requiring clean water such as spray water, feed to the reverse osmosis plant, reagent mixing water, and gland water.

Plant process water will be contained in the plant process water pond. The pond will provide approximately two hours storage capacity for plant process water requirements at nominal flow rates. The process water will be reticulated throughout the plant by centrifugal pumps. The main process water consumption points will be:

- cyclone feed hopper
- SAG and ball mill inlets
- SAG and ball mill trommel sprays
- rougher feed conditioning tank
- magnetic separator dilution water
- tailings flocculant dilution.

Seawater will be pumped to site from an intake and stored on site in the seawater storage facility.

Sea water will be recovered from this water storage facility as required and pumped to the plant. The facility will provide eight hours water storage to ensure sufficient water for continued plant operations during short interruptions in the seawater pipeline operation. Seawater will be recirculated throughout the plant by centrifugal pumps. The main seawater water consumption points will be:

- flotation spray water
- gland seal water tank
- reverse osmosis plant feed
- sodium cyanide reagent mixing

- process water make-up.

Fresh water will be produced on site from the reverse osmosis (RO) plant. The RO plant will supply fresh water for reagents that cannot be mixed in seawater like flocculant, potable water for personal consumption, and safety showers. The RO plant will also supply 20 L/sec of fresh water to the town of Diego de Almagro.

17.18 Reagents

17.18.1 Collector 1 – Aerofloat 3926

The main copper collector, Aerofloat 3926, is a thiocarbamate based collector. The collector will be supplied in 1,000 L isotainers, shipped by road and offloaded by forklift. Collector will be stored in the reagent storage shed and transferred to the collector storage tank by feed pump (one on-duty) as required. The tank will have capacity for approximately seven days consumption at design flow rates. The collector will be distributed to the flotation circuit dosing points via a ring main to an elevated header tank, with tank overflow returning via gravity to the main distribution tank. A manifold with multiple dosing pumps (each dedicated to a single dosing point) will meter the collector addition to each dosing point.

17.18.2 Collector 2 – Aerophine 3418A

The secondary copper and PGM collector, Aerophine 3418A, is a sodium di(isobutyl) dithiophosphinate-based collector. The collector will be supplied in 1,000 L isotainers, shipped by road and offloaded by forklift. Collector will be stored in the reagent storage shed and transferred to the collector storage tank by feed pump (one duty) as required. The tank will have capacity for approximately seven days consumption at design flow rates. The collector will be distributed to the flotation circuit dosing points via a ring main to an elevated header tank with tank overflow returning via gravity to the main distribution tank. A manifold with multiple dosing pumps (each dedicated to a single dosing point) will meter the collector addition to each dosing point.

17.18.3 Frother – Methyl Isobutyl Carbinol

Frother will be supplied in 1,000 L isotainers, shipped by road and offloaded by forklift. Frother will be stored in the reagent storage shed and transferred to the frother storage tank by feed pump (one duty) as required. The tank will have capacity for approximately seven days consumption at design flow rates. The frother will be distributed to the flotation circuit dosing points via a ring main to an elevated header tank, with tank overflow returning via gravity to the main distribution tank. A manifold with multiple dosing pumps (each dedicated to a single dosing point) will meter the frother addition to each dosing point.

17.18.4 Copper Depressant – Sodium Cyanide

NaCN will be delivered as pillows in 1 tonne bulk boxes. Batches of sodium cyanide (NaCN) solution will be prepared on site in the NaCN mixing tank. The tank will be filled with desalinated water and the bulk bag (inside the bulk box) will be lifted onto the bag breaker and broken out into the tank. The mixing tank agitator will be started, helping to dissolve the

pellets to make a 20% w/v solution. On completion of mixing, the solution will be pumped to the NaCN storage tank using the transfer pump (one duty).

The NaCN solution will be reticulated in a ring main system to the various dosing points using the NaCN ring main pump (one duty and one standby). Line pressure will be maintained by use of an automated diaphragm valve. Dosing to each point will be by flow control valve and flow meter.

17.18.5 Flocculant – Magnaflocc 155

One flocculant mixing, storage, and dosing system will be provided. Flocculant powder will be delivered in bulk bags and loaded into hoppers. Dry powder flocculant will be mixed with raw water to make a 0.25% w/v solution in a packaged flocculant mixing system. The mixed flocculant solution will be pumped to a storage tank with 12 hours capacity at design flow rates. Flocculant solution will be dosed to each thickener by dedicated, variable-speed helical rotor pumps. Process water will be mixed into the flocculant lines to dilute solution to 0.025% w/v before addition to the thickener feed slurry.

17.18.6 pH Modifier – Hydrated Lime

Powdered hydrated lime will be delivered to site in 20 t bulk tankers. The lime will be pneumatically offloaded into a 200 m^3 lime silo.

The lime powder will be fed by a variable-speed rotary feeder to the mixing tank, where water is added. Mixed “milk-of-lime” slurry will be transferred to a lime slurry storage tank with 24 hours capacity. The lime slurry will be reticulated in a ring main system to various user points using a lime ring main pump (one duty and one standby). Line pressure will be maintained by use of a manually operated diaphragm valve. Dosing to each point will be by timer-operated pulsing diaphragm valve.

17.18.7 Grinding Media

Forged carbon steel grinding media will be delivered to site in 20 tonne containers. The balls will be unloaded into a storage hopper via a vendor-supplied, hydraulically-operated container unloader.

Overhead cranes in the primary milling and regrind areas will be used to load steel balls, into the SAG, ball and regrind mills. Steel balls will be transported from the SAG, ball, or regrind mill grinding media storage bunker in the storage yard by FEL to the grinding media hoppers located near the mill feed end. From the hoppers, balls will be added to a bottom discharging kibble, which will be hoisted by overhead crane to a position above the mill feed chute, and emptied into the mill.

17.19 Air Services

17.19.1 Low-Pressure Air

The flotation blowers will supply low-pressure air to the flotation cells at a suitable supply pressure. There will be three blowers (two duty and one standby) installed to meet flotation air requirements. Pressure control valves will be installed in the air distribution lines to meet the different air pressure requirements of different flotation cells. Multiple-stage, centrifugal type blowers will be used, equipped with variable-speed drives to adapt to fluctuations in flotation air demand.

The blowers will be housed inside an acoustic enclosure to reduce noise to an acceptable level. The enclosure will have ventilation for cooling.

17.19.2 High-Pressure Air

Two rotary screw air compressors will provide high-pressure air for plant and instrument air requirements. There will be one duty and one standby compressor operating in lead-lag mode. Plant air will be stored in the plant air receiver prior to being distributed throughout the plant.

17.20 Plant Buildings

A number of plant buildings will be required for operation and maintenance of the plant. These include the administration office, laboratory, plant workshop, warehouse, reagent stores, guardhouse, plant control room, and services.

The administration building will consist of a number of internal offices and meeting rooms. A first aid facility will be provided near this building. The warehouse will house mechanical, electrical, instrumentation, and general items in discrete areas. The warehouse structure will be contiguous to the plant maintenance workshop. Internal offices will be supplied for warehouse and maintenance staff.

The plant control room will be located at an elevated position adjacent to the hydrocyclone cluster, and will be provided with aluminum-framed windows for viewing into the process plant. In particular, stockpile, grinding, flotation, and thickening areas will be easily viewable from the control room. External windows will be double-glazed with tinted glass. Allowance has been made for security fencing around the plant site.

18 PROJECT INFRASTRUCTURE

18.1 Site Power Supply

The project requires electrical transmission, sub-transmission, and distribution infrastructure as described below:

High-voltage transmission will be achieved using a 220 kV double-circuit overhead line from Diego de Almagro. The line will be approximately 7 km long to the project main substation.

Medium-voltage power will be transmitted to the different centres of consumption via a 13.8 kV line.

Low-voltage power distribution will be at two voltage levels: 600 V for industrial systems, and 400 V for offices and ancillary systems.

The minimum equipment required is:

- 220 kV exit sections at the Diego de Almagro substation
- 220/13.8 kV transformer yard at the project main substation.

18.2 Concentrate and Seawater Pipeline

This section details the hydraulic design of the concentrate and seawater transport system from the mine site to the coastal port, including the pumping system, pipeline, and intermediate facilities.

The location of the port is still to be finalised. The location will be finalised during the Feasibility Study phase.

The mine site is at a moderate elevation (~1,000 masl). Freezing temperatures are not considered to be a significant problem in this area.

18.2.1 Concentrate Pipeline Design

The total length of the surveyed route is 74 km. There is a consistent geomorphologic zone along the entire pipeline route into Rio Salado valley.

The corridor stars from the concentrator area. The first 15 kilometers includes the bypass of Diego de Almagro city. From 15 to 34.5 km the corridor runs along the north side of the C-13 road. After road crossings at 34.5 and 34.75 km, the corridor joins with road C-209, at the

south side the El Salado city. From Km 34.75 to Km 74.0, the pipeline runs along the north side of the C-57 road and highway Route 5. The ground corresponds to gravel, clay and sand in the coast sector.

This corridor does not have high points and has a continuous slope. The principal problems are the Salvador River crossing and the town of El Salado.

Figure 18-1 shows the profile of this alternative.

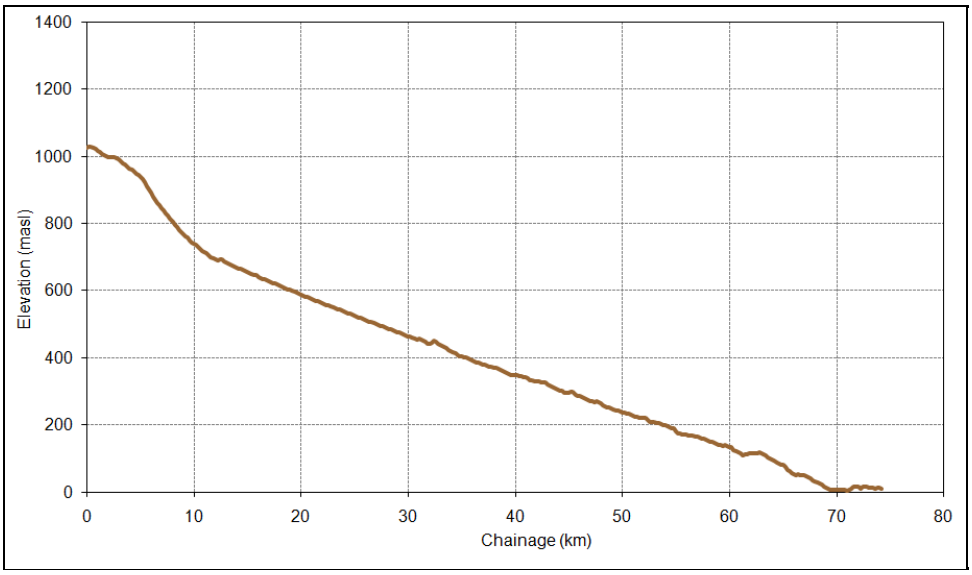


Figure 18-1: Santo Domingo Mine Site to Port - Concentrate Pipeline Corridor

Design Basis for the Concentrate Pipeline

General Parameters

The criteria for developing this design are indicated in Table 18.1.

Table 18.1: Parameters for Design

Parameter	Unit	Value
Maximum Design Solid Throughput (*)	t/h	505
Concentrate Range (Cw) (**)	%	56–63
SG solids (***)	-	4.81
Design Life	Years	20
System Availability	%	96

Notes: (*) Information by client. (**) This value assumes a system availability of 96%. (**) Transportable Range. (***) This engineering stage considers SG of iron solids.

Definition of Route

The following criteria have been considered for the concentrate pipeline platform installation:

- The platform will be 15 m wide and its longitudinal slope will not exceed ±12%.
- Concentrate pipelines will be buried in accordance with ASME B31.11.
- The seawater pipeline will be buried in the same platform as the concentrate pipeline.

Figure 18-2 and Figure 18-3 show the alignment, platform and trench design considered.

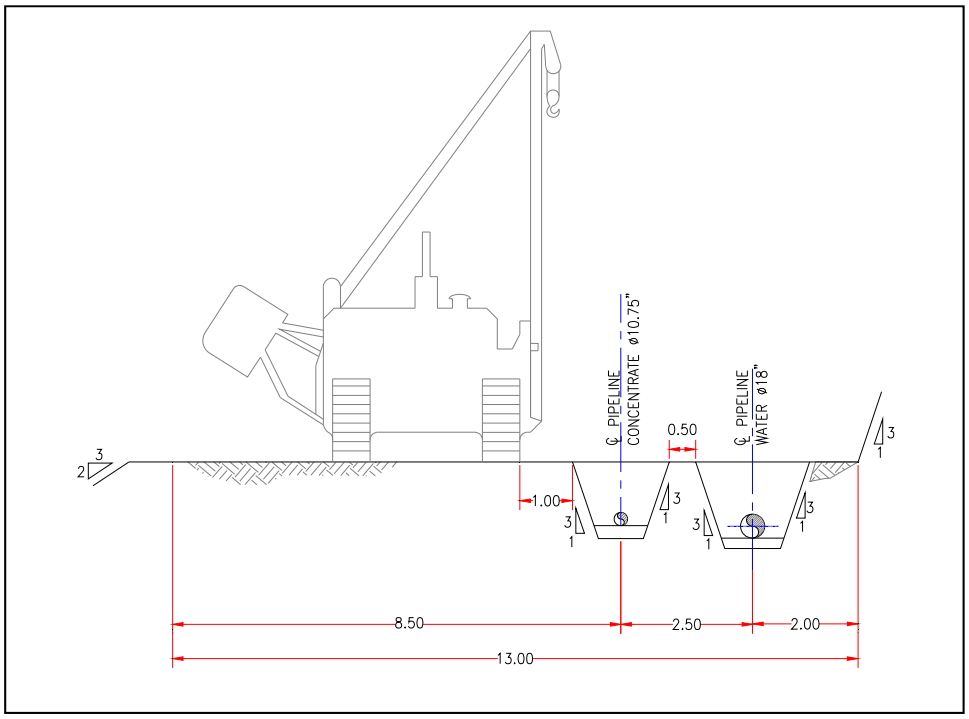


Figure 18-2: Conceptual Platform Diagram

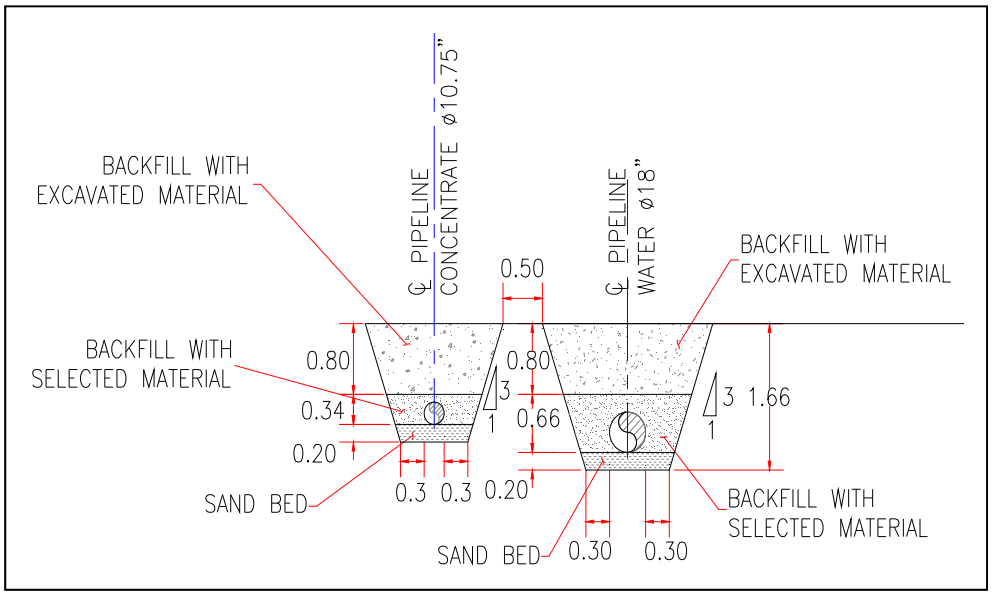


Figure 18-3: Conceptual Diagram of Trench

Pipeline Characteristics

Pipeline characteristics are defined in Table 18.2.

Table 18.2: Pipeline Characteristics

Pipe Characteristics	Santo Domingo Mine Site – Port
Maximum / Minimum Outside Diameter (inches)	12.75
Pipe Steel Specification	API 5L X 65
Maximum / Minimum Wall Thickness (inches)	0.25/0.25
Liner Thickness (inches)	0.31
Roughness (inch)	0.0008

The pipeline corridor configuration has one head station, one valve station, three pressure monitoring stations and one valve choke station/terminal. Table 18.3 presents the main system facilities for this configuration.

Table 18.3: Main Pipeline Facilities Santo Domingo Mine Site to Terminus

Facility	Location (km)	Elevation (masl)
Head pump station	0	1,000
Pressure monitoring station 1	15	656
Valve station 1	28.8	488
Pressure monitoring station 2	40	348
Pressure monitoring station 3	60	134
Terminal / valve choke station	74.81	11

18.2.2 Seawater Pipeline

The seawater pipeline follows the same route as the concentrate pipeline described above. The total length of the surveyed route is 74 km.

Figure 18-4 shows the profile from the coast to the Santo Domingo Mine Site.

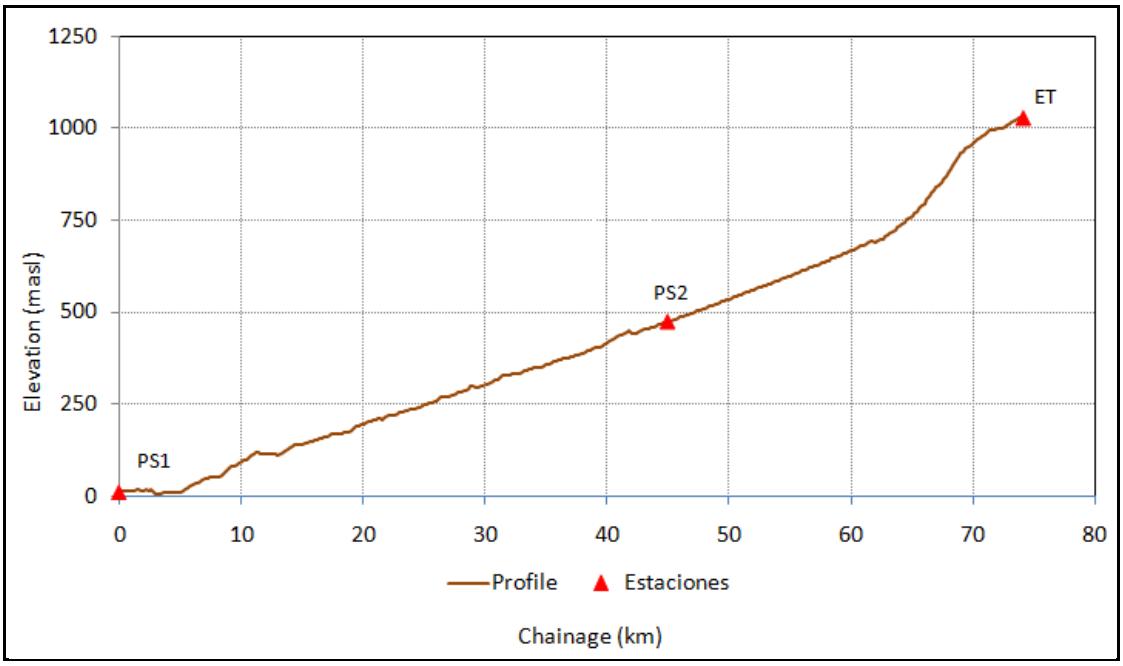


Figure 18-4: Terminus to Santo Domingo Mine Site Water Profile

Design Basis

Input data used to perform hydraulic calculations and system selection follows including Table 18.5 which details the water characteristics.

Input Data

- Flow rates:
  - Nominal – 1,390 m^3/h (Consider a 75% of maximum flow)
  - Maximum – 1,853 m^3/h
- Instantaneous flow rates (these flows assume an availability of 93.1%
  - Nominal – 1,493 m^3/h
  - Maximum – 1,990 m^3/h
- System availability: 93.1%
- Design life: 20 years

Table 18.4: Water Characteristics

Property	Value
Temperature (ºC)	15
SG	1.03
Kinematic viscosity (cSt)	1.15
Dynamic viscosity (cP)	1.18
pH	7.5
Electric conductivity (mS/cm)	43
Dissolved oxygen (mg/l)	8.30
Oxygen saturation (%)	111.4

Mechanical Design Basis

The following data summarizes the mechanical design basis used to develop the water supply system design:

- pipe material: carbon steel ASTM A53 Gr. B (in the next engineering stage pipe material must be checked according to new operational conditions)
- coating:
  - internal: HDPE liner
  - external: three-layer polyethylene[20]
- HDPE roughness: 0.0213 mm (0.00084 inches).

18.2.3 Seawater Pipeline Conclusions and Recommendations

Based on an economic and technical analysis of different pipe diameters and numbers of pump stations, a 24" pipe with two ASME Class 600 pump stations is recommended for this phase of the Santo Domingo project.

[20] Additional to cathodic protection system

Three horizontal, multistage operating pumps are considered for each pump station of the water supply system.

Two storage tanks, each one with a live volume of 995 m3, are proposed for each pumping station and each alternative. The retention time at maximum flow is 30 minutes.

A head dissipation device is required at the mine site to maintain adequate clearance between the high point (close to the mine site) and the HGL. Sizing and specification of these facilities will be defined in subsequent engineering phases after a more detailed routing shows that no feasible optimization can eliminate the high points.

Some type of air/vacuum pressure relief device is required at the high point close to PS2 in order to prevent potential water column breakage and to allow for air to be admitted into or released from the pipeline in fill and drainage operations. Sizing and specification of these facilities will be defined in subsequent engineering phases after a more detailed routing shows that no feasible optimization can eliminate the high points.

No de-aeration plant or corrosion inhibitor is considered for this system.

Since this system would be internally lined, it would allow exposure to the atmosphere. This means that the water re-oxygenation at the tanks in every pump station would not have any corrosion potential over the line.

Monitoring systems, such as for oxygen concentration, reagents residuals, and bacterial activity will be required. Regardless of the type of system or oxygen quantity, there is a high probability of micro and macro fouling formation unless parallel water treatment (biological and chemical) and/or filtering are performed prior to pumping.

18.3 Port Concentrate Handling and Shiploading Design

18.3.1 Overview

The port facility for the Santo Domingo project is suitable for receiving, storing, reclaiming, and shiploading iron ore and copper concentrates. The concentrates will be transported to the port site via a concentrate pipeline and dewatered in a filter plant.

18.3.2 Project and Service Life

The mine is expected to produce nominally 250,000 tonnes of copper concentrate and 4,000,000 tonnes of magnetite concentrate per year. The life-of-mine (LOM) is projected to be 18 years.

The design shall consider a fit-for-purpose approach, following international standards.

All new mechanical components shall be designed for an economic life of 20 years.

18.3.3 Design Vessels Information

The facilities shall be designed for the following design vessels as shown in Table 18.5.

Table 18.5: Vessels Information

	Ship Size		
Dimension	**40,000 DWT (*)**	**60,000 DWT**	**200,000 DWT**
Overall Length (m)	188	235	320
Beam (m)	29.2	32.3	55
Moulded Depth (m)	15.6	18.9	26.4
Maximum Draft (m)	11.3	12.4	18.6
Number of Hatches	5	7	9

Note: * Due to market practices, provisions are being made to enable the port to serve 40,000 DWT vessels for copper concentrate.

18.3.4 Port Facilities

The facilities at port location necessary to support the port operations include:

- the concentrate pipeline chocking station
- a filter plant for concentrate dewatering and associated settling ponds
- maintenance facilities to support the port operations
- stockyards' inloading conveying systems
- iron ore and copper concentrate stockyards
- outloading conveying systems from the stockyards to the existing shiploading system
- offshore infrastructure including trestle, shiploader, and mooring systems.

18.3.5 Stockpile Sizing

Copper Concentrate

Copper concentrate will be stored in an enclosed storage building with 40,000 tonnes capacity, providing a storage ratio of approximately 16% of annual throughput. The typical parcel size for copper concentrate shipment is 25,000 tonnes.

An A-frame-style enclosed storage building is being considered as shown below in Figure 18-5. The copper concentrate stockpile will be formed by the use of an overhead conveyor with a travelling tripper.

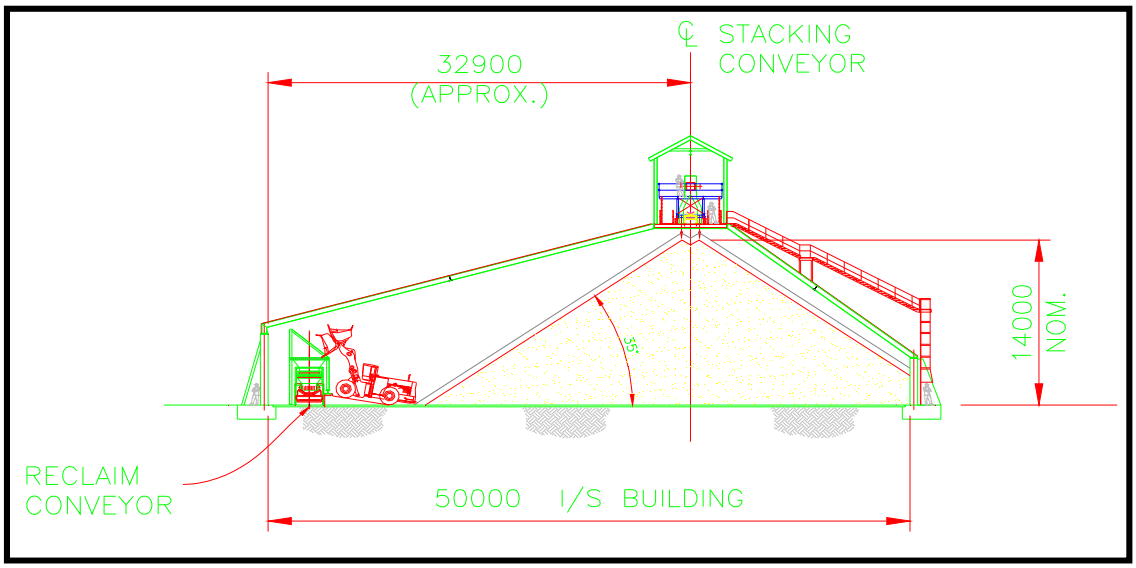


Figure 18-5: Cross-Section 1 – Copper Concentrate Storage Building

Iron Ore Concentrate

The terminal would include open stockpile storage for iron ore concentrate with a minimum capacity of 250,000 tonnes, or approximately 6.5% of annual throughput.

Figure 18-6 and Figure 18-7 show a plan and cross-sectional view of the iron ore concentrate stockpile.

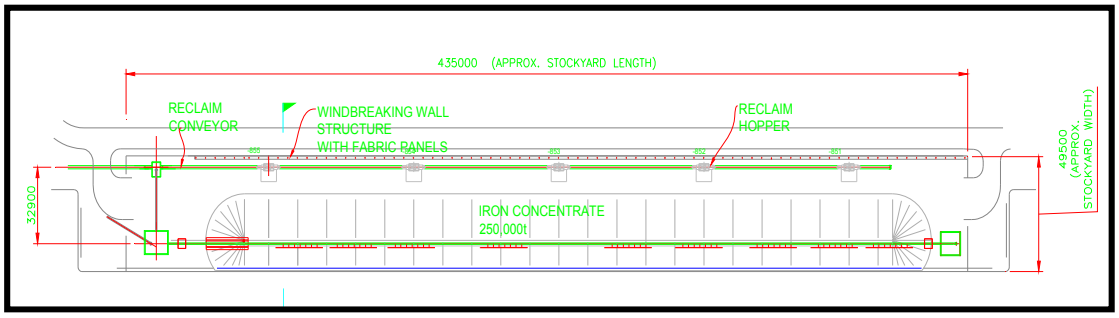


Figure 18-6: Cross-Section 2 – Copper Concentrate Storage Building

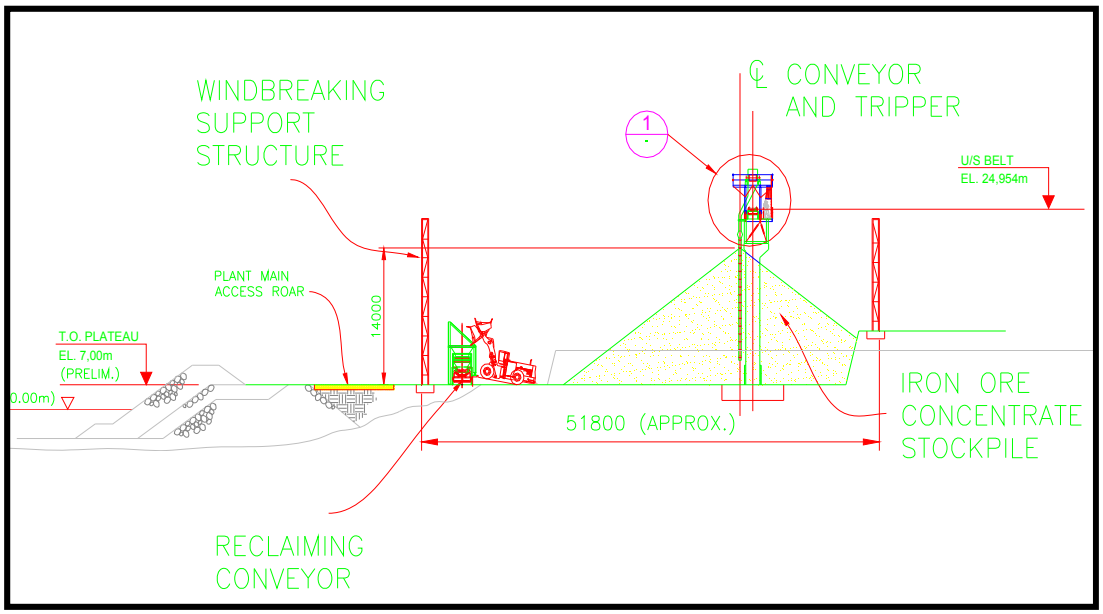


Figure 18-7: Cross-Section 3 – Copper Concentrate Storage Building

Direct Loading

Magnetite concentrate from the filter plant will be directly loaded into ships through a diverter chute located at transfer tower and bypass conveyor.

No bypass for copper concentrate direct loading is considered due to the infeeding rates.

Magnetite Concentrate Stacking – Filter Plant to Stockyard

The dewatered magnetite concentrate from the filter plant is delivered by infeed belt conveyors and stored in the stockyard using a rail-mounted travelling tripper with a cascade chute to suppress dust. The infeed belt conveyor is equipped with a belt scale for inventory. control.

The conveyors and travelling tripper leading to the magnetite concentrate stockpile will be sized to accommodate the peak rate from the filter plant, currently estimated to be 750 t/h.

The filter plant will continuously feed the travelling tripper via the stacking conveyors. Although the rate of concentrate stacking varies, a nominal capacity of 600 t/h is expected.

Stacking of the filtered product will be continuous, and ship-loading can be carried out on an on-demand basis without any interruptions to the stacking operation.

Copper Concentrate Stacking – Filter Plant to Storage Building

The dewatered copper concentrate from the filter plant is delivered by infeed belt conveyors and stored in an enclosed storage building via a travelling tripper. The infeed belt conveyor is equipped with a belt scale for inventory control.

The conveyors leading to the copper concentrate stockpile will be sized to accommodate the peak rate from the filter plant, currently assumed to be 100 t/h.

Concentrate Reclaim and Ship Loading (Stockyards to Jetty)

Concentrate will be reclaimed from the stockpile with front-end loaders (CAT 988H or equivalent) and discharged into stationary receiving hoppers with belt feeders straddling the reclaim belt conveyor. The reclaiming system will transport the concentrate through the reclaiming conveyor to the shiploader via transfer conveyors and a trestle conveyor. The products will then be loaded into bulk carriers by the shiploader.

A radial shiploader with 3,000 t/h nominal capacity is planned. The design of the shiploader and the berth will accommodate a range of vessels from 60,000 DWT to 200,000 DWT; provisions to serve 40,000 DWT vessels are being analyzed.

The vessels will have to be warped during the shiploading operations.

Shiploader

A radial type shiploader (with slewing, shuttling, and luffing capability) is proposed to support export operations.

The shiploader will be able to reach at least three hatches of the largest design vessel (200,000 DWT).

For geared vessels, usually ranging from 40,000 to 60,000 DWT, the shiploader will be positioned perpendicular to the vessel being loaded, and its boom will have to be cleared before warping the vessel.

System Operation – Concentrate Reclaim and Outloading

Magnetite concentrate will be stored at an open stockyard with protective wind fences around the perimeters before being reclaimed and loaded onto vessels.

Magnetite concentrate will be reclaimed from the stockpile by front-end loaders (FELs). Most of the time three FELs (CAT 988 H or equivalent) will be required to maintain the average reclaim rate of 2,400 t/h; with the supplemental direct loading rate of 400t/h, the total average loading rate will reach 2,800 t/h.

Copper concentrate is stored inside the storage building before being reclaimed and loaded onto vessels. Stacking and reclaiming operations are performed independently.

Copper concentrate will be reclaimed from the stockpile by FELs. Two FELs (CAT 988 H or equivalent) will be required to maintain the average reclaim rate of 1600 t/h.

The shiploader operator controls the reclaim operation and initiates the starting sequence for shiploading. The shiploader operator also initiates the shiploading system stopping sequence as required.

Concentrate will be sampled for both trace elements and ore content prior to loading onto barges. Two-stage sampling will be performed by means of a primary sweep-arm type cross belt cutter installed at the head end of the concentrate receiving conveyor. Samples extracted by the cutter at each sweep cycle will be discharged into a chute and fed by gravity onto a small inclined feeder conveyor.

The second-stage sample will be extracted from the feeder belt by a small secondary sampler of the same type. The sample from the secondary cutter will be discharged into a chute and fed by gravity into an automatic, carousel-type canister, sample collection system. The collected samples will be periodically removed and delivered to an on-site laboratory to be analyzed.

The feeder belt carries the sample reject stream back to the stockyard conveyor.

18.3.6 Dust Control and Housekeeping

Conveyor Enclosures

All belt conveyors will be completely enclosed and provided with top and side covers to avoid fugitive dust (Figure 18-8). Collecting pans will be provided at elevated sections. The belt conveyors will be cleaned using vacuum trucks.

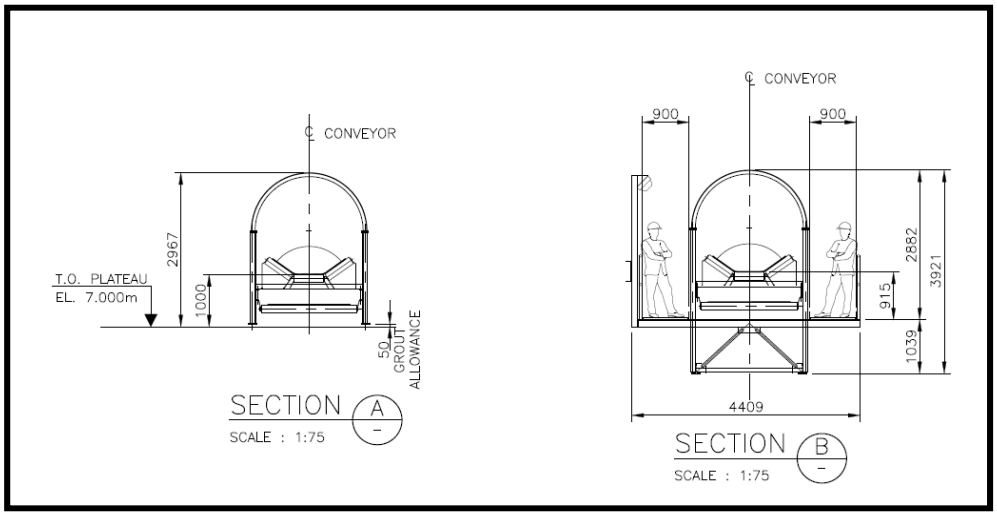


Figure 18-8: Cross-Section 4 – Copper Concentrate Storage Building

Dust Suppression System

An automated dust suppression system will be incorporated into all transfers utilizing water spray for iron concentrate handling. Stockyard dust control will be maintained by a series of sprinklers spraying settlement pond water onto the pile to suppress dust. Dust control at conveyor transfer stations will be by spray nozzles located on standpipes. General control of the dust control system will be by a dedicated control panel and weather station. Programming will allow the system to supply only the minimum required water and will compensate for temperature, humidity, and wind.

A dust collection system will be provided for the enclosed copper concentrate storage building. The primary purpose of the system is to provide a slight negative pressure to the building to help prevent fugitive dust emission.

All conveyors will be enclosed. Dust collectors will also be provided at the conveyor transfer points.

FELs will be fitted with air conditioning and filtration equipment. Operators will be required to wear appropriate respiration equipment when working in the enclosed conveyors and storage building.

Wind Fences

The iron ore stockyard will be provided with wind fences to reduce the effect of wind on the piles and avoid product dispersal affecting the surrounding environment.

18.3.7 Offshore Structures

The offshore infrastructure (see Figure 18-9) includes the trestle, shiploading structure, dolphins, and mooring buoys, and is described below. The associated materials handling equipment is addressed above.

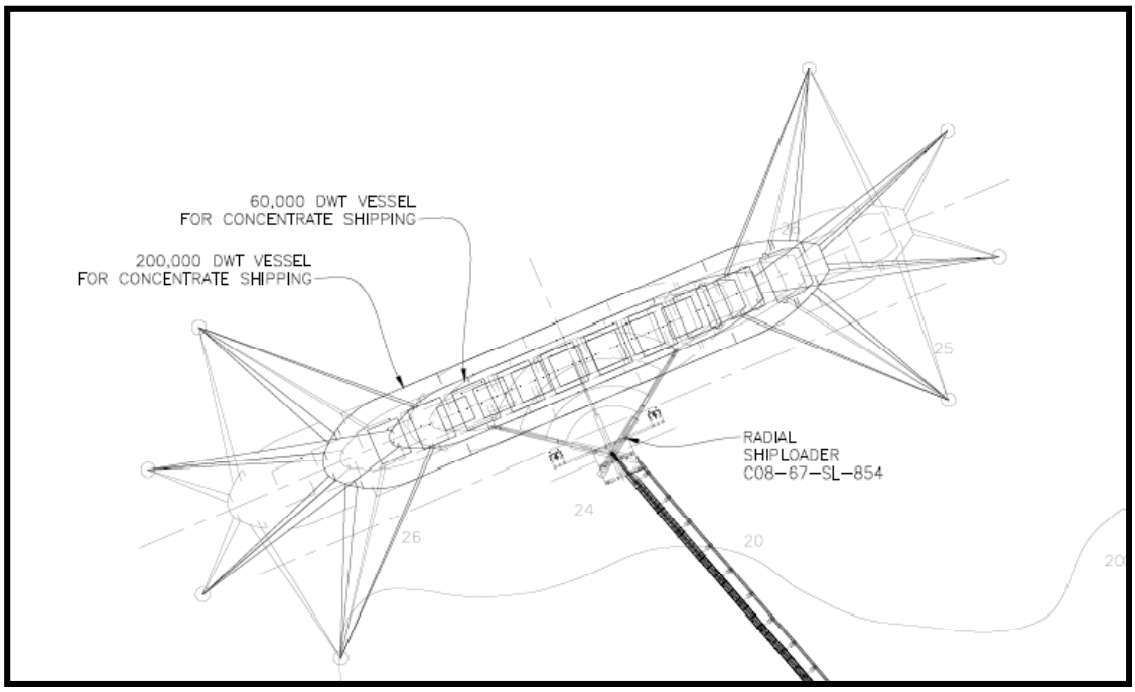


Figure 18-9: Terminal Offshore Facilities

Trestle

The trestle is designed to accommodate the conveyor belt structure and a single-lane roadway (Figure 18-10). The proposed superstructure consists of a 9 m wide concrete deck accommodating a 5.3 m wide roadway and road barriers, with the remaining 3.7 m occupied by the conveyor structure and associated cladding, which is seated directly on top of the concrete deck. Pedestrian access to the pier runs along both sides of the conveyor. The roadway is designed for transporting the shiploader operators and light vehicles for maintenance. Typical deck elevations are +7 m (SHOA).

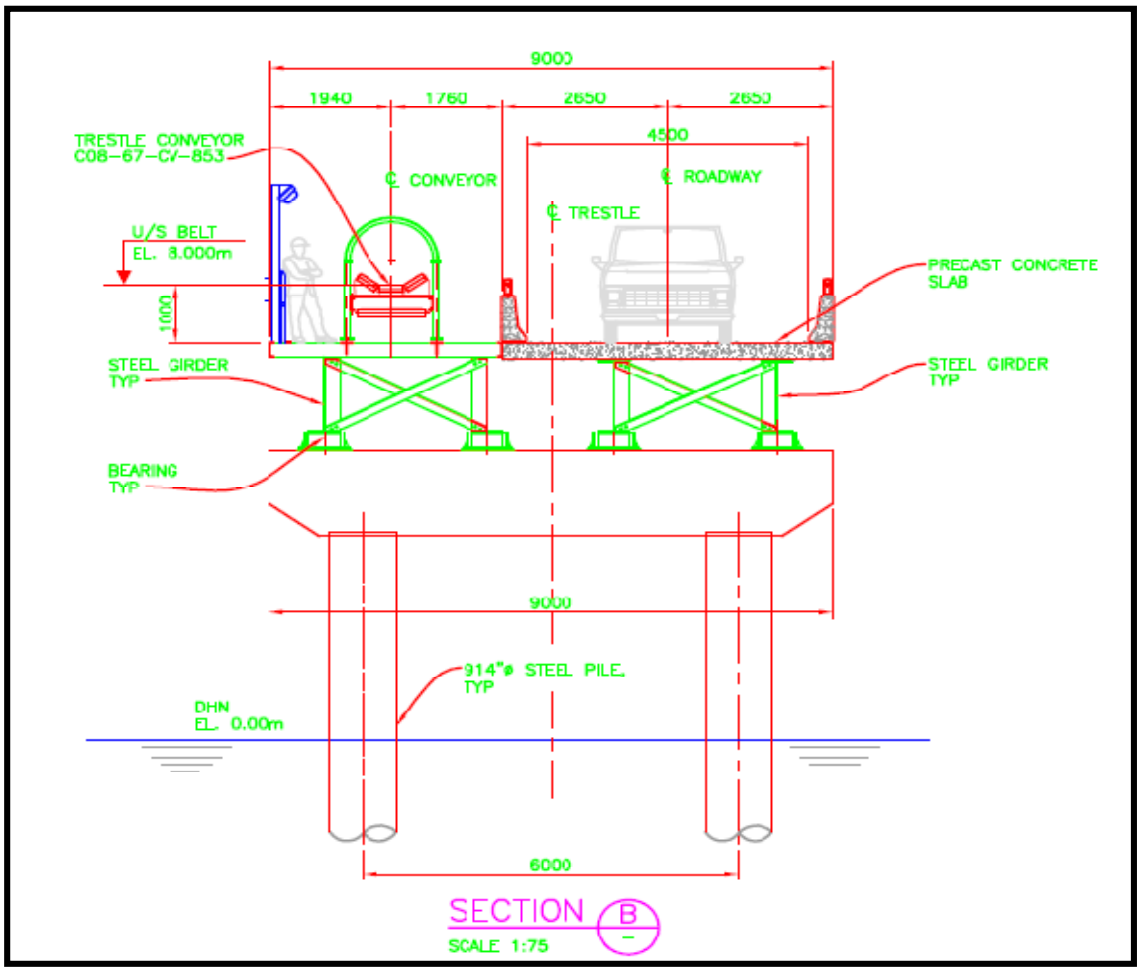


Figure 18-10: Trestle Section

Shiploader Support Structure and Dolphins

The shiploader support structure consists of a pivot platform, two emergency dolphins (to be provided with Yokohama fenders), a roadway access, and catwalks (Figure 18-11).

These dolphins are considered to provide adequate protection from ship collisions for the shiploader.

From the shiploader's platform a road access will be provided to the northeast dolphin that will be extended to enable proper maintenance of the shiploader's boom. This platform will also be used to tie down the shiploader's boom in case of storms.

Typical platform elevations are +7 m (SHOA).

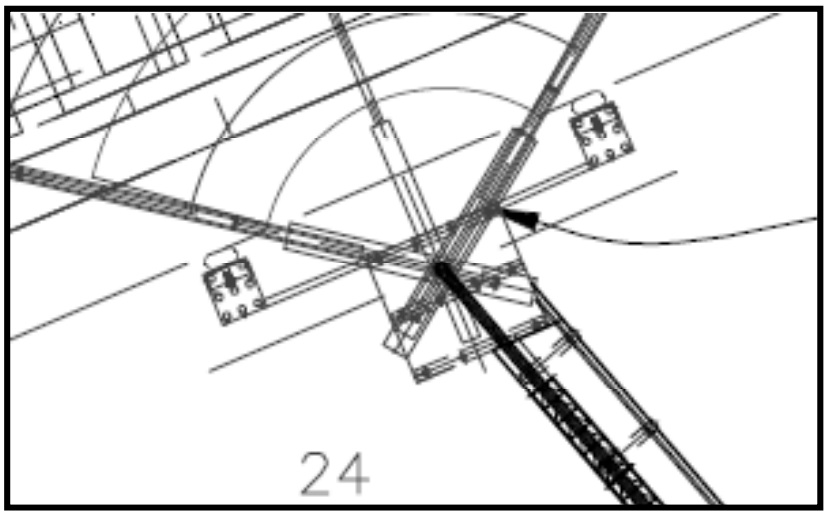


Figure 18-11: Zoom Marine Structures

18.3.8 Other Services

Since this is a relatively isolated facility, no fuel or water services will be provided. However, boat services will be required to transport pilots upon arrivals and departures of attending vessels, and to transport government and port facility officials to and from the vessel when in the mooring.

18.3.9 Concentrate Receiving and Dewatering Designs

Copper Concentrate Dewatering

Copper concentrate from the concentrate pipeline will feed the concentrate distribution box. Typically, concentrate will be fed directly into the copper concentrate storage tanks, but can be fed to the copper clarifier or into the emergency pond for short-term storage and later recovery. During changeover from pumping one concentrate to the other via the pipeline, dilute concentrate and water will typically be fed directly to the clarifier to prevent dilution of the copper concentrate filter feed.

The clarifier will incorporate an auto-dilution feed well and rake lift mechanism. Flocculant solution will be dosed to the clarifier feed box and feed well by a variable speed flocculant dosing pump to aid in settling of concentrate and to provide necessary clarity in the clarifier overflow.

Clarifier overflow will be pumped to the copper effluent high-density sludge (HDS) treatment plant for treatment prior to disposal. Clarifier underflow will be pumped to the copper concentrate filter feed tank by variable-speed concentrate clarifier underflow pumps. The copper concentrate filter feed tanks will have approximately 30 hours storage capacity.

Copper Concentrate Filtration

Copper concentrate slurry will be delivered to the copper concentrate filters from the agitated storage tanks using variable-speed centrifugal filter feed pumps. The filter press will incorporate a cake wash stage to wash chlorides present due to the salt water flotation from the filter cake.

Following cake washing the filter will reduce the moisture content of the concentrate prior to transport. Filter cake will discharge through the floor of the filter building onto a filter discharge belt conveyor. The belt conveyor will transport the cake to the concentrate stacking conveyor in the covered concentrate storage shed.

Filtrate from the filter press will gravitate to the filter air release tank and will flow back to the copper concentrate clarifier feed box. The concentrate filter press and concentrate storage stockpile will be housed in buildings that will be fully sheeted on all sides for protection from wind and rain.

The transportable moisture limit (TML) has been assumed to be higher than 10% w/w in the filter cake. In the absence of testwork, the design target for the filter duty has been set to 8% w/w.

The copper concentrate filter has been sized based on the throughput in the plant, a head grade of 0.7% Cu, a final concentrate grade of 29% Cu, and a throughput of 60 kt/d. The filter size assumes an availability of 80% while the flotation plant is operational (an annualized availability of 93%).

The filter size is based on filtration rates determined from preliminary filtration testwork at 495 kg/m^2/h. For the study, a Larox plate–and-frame filter press with 144 m^2 of filtration area and a 60 mm chamber depth has been selected.

Copper Effluent Treatment

A HDS copper effluent treatment system has been incorporated into the port flowsheet to ensure copper is not released in the concentrate wastewater stream. The system includes pH adjustment and air addition to precipitate copper with clarifiers and sand filters to remove the precipitate prior to release.

Subsequent to freezing the design and flowsheet for the port, a decision was made to relocate both the port and seawater line intake to the same location as the planned port facility. This will allow the concentrate filtrate water to be pumped to the seawater intake and returned to the plant site, removing the requirement for releasing water at the port site. The practicality of this option should be further investigated in the next phase of the project, in particular the likely impact of this water on the operation of the site's RO filtration plant.

Magnetite Concentrate Dewatering

Magnetite concentrate from the concentrate pipeline will feed the concentrate distribution box. Typically, the concentrate will be fed directly into the magnetite concentrate storage

tanks, but can be fed to the magnetite clarifier or into the emergency pond for short-term storage and later recovery. During changeover from pumping ore concentrate to the other down the pipeline, dilute concentrate and water will typically be fed directly to the clarifier to prevent dilution of the magnetite concentrate filter feed.

The clarifier will incorporate an auto-dilution feed well and rake lift mechanism. Flocculant solution will be dosed to the clarifier feed box and feed well by a variable-speed flocculant dosing pump to aid in settling of concentrate and to provide necessary clarity in the clarifier overflow.

Clarifier overflow will be pumped to the process water tank, with a bleed line for disposal to maintain a balanced water circuit. Clarifier underflow will be pumped to the magnetite concentrate filter feed tank by variable speed concentrate clarifier underflow pumps. The magnetite concentrate filter feed tanks will have approximately 32 hours storage capacity.

Magnetite Concentrate Filtration

Magnetite concentrate slurry will be delivered to the magnetite concentrate filters from the agitated storage tanks using variable-speed centrifugal filter feed pumps. Distributor dart valves are fed by slurry pumps and transfer the pulp by gravity to the disc filters. Overflow from the pulp distributors returns by gravity to the homogenization tanks.

Each of the filters is equipped with a vacuum filtrate system that discharges to a common filtrate collection tank. Filtrate water is pumped to the magnetite clarifier.

Fresh water is used for washing the ceramic filters via a set of backwash pumps and an expansion tank. A common vacuum filter backflush system enables the vacuum filters to be flushed on a regular basis. Two acid solutions (oxalic and nitric acids) are added to the backflush water that is injected into the filters.

The filters will incorporate a cake wash stage to wash chlorides present due to the salt water flotation from the filter cake.

Following cake washing, the filters will reduce the moisture content of the concentrate prior to transport. Filter cake will discharge through the floor of the filter building onto a filter discharge belt conveyor. The belt conveyor will transport the cake to the concentrate stacking conveyor in the covered concentrate storage shed.

Filtrate from the filters will gravitate to the filter filtrate tank and be pumped back to the magnetite concentrate clarifier feed box. The concentrate filters will be housed in the filter building, which will be fully sheeted on all sides for protection from wind and rain. The concentrate storage stockpiles will be uncovered and will require wind protection to minimize dust losses.

The TML has been assumed to be higher than 10% w/w in the filter cake. In the absence of testwork, the design target for the filter duty has been set to 8% w/w.

The magnetite filters have been sized based on a magnetite concentrate production of 4 Mt/a and an availability of 90%. The filter sizing is based on a filtration rate of 1,000 kg/m^2/h. Preliminary filtration testing indicates that this rate would only be achievable with a feed slurry density in excess of 65% w/w solids. Achieving this density will require replacing the current 10 m diameter clarifier with a magnetite concentrate thickener approximately 35 m in diameter. Due to the timing of the testwork, the design had been frozen prior to receipt of the preliminary results and the thickener has not been included in the PFS design. Should the final filtration test results confirm the preliminary results, the design will need to be modified to add the thickener, or additional filters will be required.

For the study, four disc type ceramic filters (Ceramec®, Larox Model CC-144 HiFlow) have been selected.

Reagents

Flocculant

Flocculant is required in the clarifiers to increase the slurry density.

The flocculant mixing system will be supplied as a vendor package. Investigations will be undertaken in the next phase of the project to determine the flocculant type (powder/emulsion) and delivery methodology.

Oxalic and Nitric Acid

The Ceramec® filters will require regular descaling to maintain operational performance. Oxalic and nitric acids will be added to the filter backwash water to maintain filter operational performance by de-scaling the capillary channels of the discs.

The oxalic acid will be delivered in crystalline form in ISO Type 2 frame-type tank containers (or by bulk tanker) and transferred into a storage silo. The oxalic acid is withdrawn from the storage silo by a screw feeder into an agitated mixing tank and diluted with water to prepare a 3% w/v solution. The oxalic acid solution is pumped to an agitated holding tank for transfer as required by metering pumps to the ceramic filter station where it is mixed with the filter wash solution.

The concentrated nitric acid, 53% w/w, will be delivered by tanker and transferred by pumps to a 20-day capacity storage tank. From the storage tank the 53% nitric acid is transferred to an agitated mixing tank and diluted with water to prepare a 1% nitric acid solution. The 1% nitric acid is then pumped to a storage tank before being transferred as required by metering pumps to the Ceramec® filter station where it is mixed with the filter wash solution.

Services

Potable, industrial, and fire water systems, and compressed air services will be installed and distributed around the site using ring mains.

High-Pressure Air

High-pressure air for the filter presses will be provided from the filter compressors (two duty, with a third providing instrumentation and plant air for the filter plant) via two dedicated, 16-bar filter high-pressure air receivers. The receivers will have sufficient volume to cater for surges in high-pressure air demand during the cake blowing stage.

A take-off from the high-pressure airline will direct air to a dedicated air filter, followed by the refrigeration-type instrument air dryer (duty/standby) to produce instrument-quality air for all pneumatic controls. A dedicated instrument air receiver will be provided.

One (duty only) pressing air compressor, via a dedicated 30-bar air receiver, will provide filter-pressing air.

18.4 Tailings Storage Facility

This section presents the tailings storage and tailings water management facility for the Santo Domingo project. The location of the TSF is north of the proposed mine. A site location study and a tailings deposition trade-off study were performed by AMEC as a part of the scoping study for the Santo Domingo project. The selected site was Alternative 5 with conventional thickened tailings deposition.

The TSF is designed to store approximately 353 Mt of thickened tailings, enough for approximately 18 years of the project life. Storage of both fresh and seawater is proposed to be in lined ponds near the plant site. No other water storage reservoir is proposed. Water make-up is by treated seawater. Based on the conventional thickened tailings disposal method, the estimated water make-up is approximately 1,445 m^3/h.

A preliminary geotechnical campaign by AMEC assessed the geotechnical and hydrogeological characteristics of the TSF area (April 2011). More detailed geotechnical work and assessment of the waste rock proposed as rock fill are required. This work can be carried out in parallel with the upcoming feasibility study.

AMEC previously identified the potential to convert the waste rock downstream raise dam to a cyclone sand raise dam. This option was not studied in this report, as it would be an optimization that could be considered at any point during operations of the TSF.

18.4.1 General Description

The location of the TSF is northeast of the mine site and consists of a rock fill downstream raise dam encompassing two drainages off of the local foothills to the east, as shown in Figure 18-12.

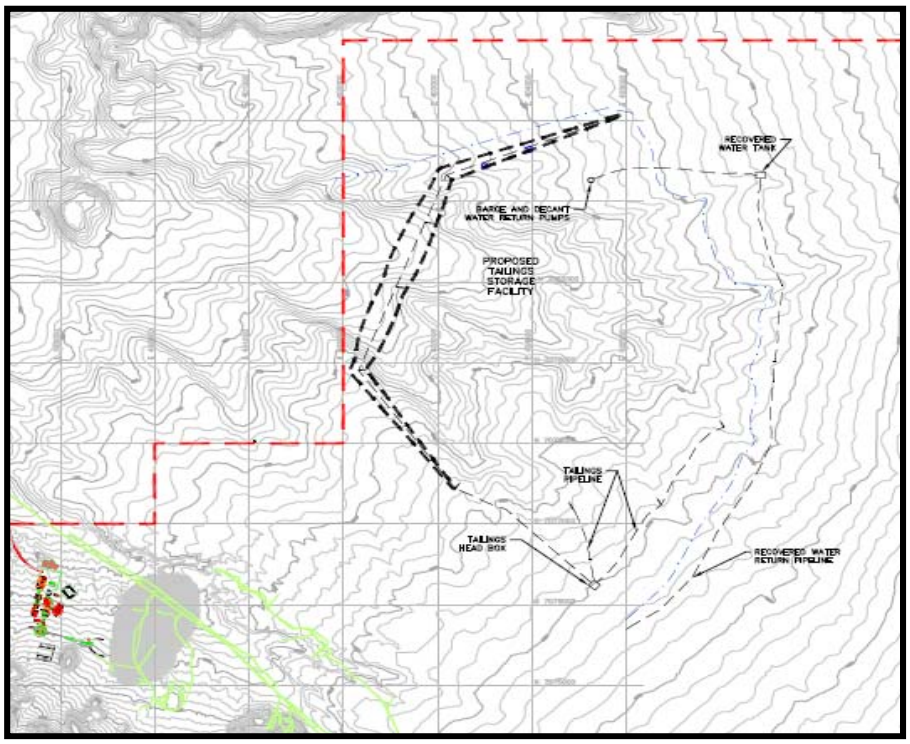


Figure 18-12: General Locations of Facilities

No water storage reservoir is planned. However, water for process use will be temporarily stored in two ponds adjacent to the plant site and the starter dam as constructed will be used to contain water to assist with the supply of water during the initial start-up purposes.

The TSF includes a starter dam for storing at least two years of thickened tailings. The starter dam crest will be raised in stages by the downstream method to contain the waste tailings within the current permitted boundary limits up to Year 18 of operations. In addition to the temporary start-up water pool, there will be backup sea water supply pond at the plant site. The base case start-up TSF is shown in Figure 18-13.

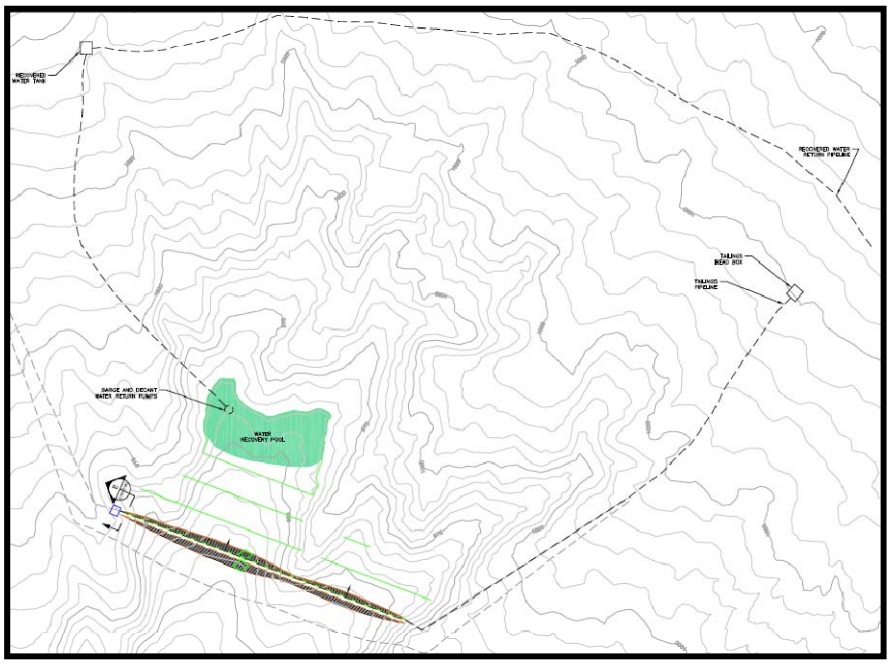


Figure 18-13: Start-up Tailings Plan

The TSF is designed as a water storage facility for the first two years of the project. For Years 3 through 20 of operation, the TSF is designed with a downstream raise embankment to provide stability in this seismically active area.

18.4.2 Design Criteria

The design criteria for the TSF are listed in Table 18.6. Assumed values were chosen on a conservative basis, and may be refined with further studies after additional testing and site investigations have been performed.

Table 18.6: TSF Design Criteria

Design Item	Criteria	Reference
Total tailings	367 Mt from copper, iron, and gold	Ausenco
Tailings production	60,000 t/d 365 d/a 21.9 Mt/a	Ausenco
Tailings solids specific gravity	3.0 – to be confirmed	Estimate
Tailings percent solids (during pipeline transport)	60% by total weight Conventional Thickened slurry	Ausenco
Expected tailings beach	0.5% (0% used in PFS)	Estimate

Design Item	Criteria	Reference
Tailings dry density	1.35 to 1.64 t/m^3 Average = 1.39 – must be verified	AMEC (2011)
Target Facility Capacity	400 Mt (288 Mm^3)	Ausenco
Facility service life	Approximately 18 years	Ausenco
Tailings start-up dam storage requirement	Approximately 2 years	Needed for start-up
Environmental Issues	No ARD, no restrictions, no issues	Ausenco
TSF dam	Criteria	Reference
Tailings dam crest width	25 m (reduced from 30 m in Scoping Study). Must be verified during Feasibility Study to be able to support the use of haul trucks placing rock fill.	Ausenco
Tailings dam upstream slope, downstream raise	1.6:1 (horizontal:vertical)	Ausenco
Tailings dam downstream slope	1.8:1 (horizontal:vertical)	Ausenco
Dry freeboard capacity	3 m above operating water pool and designed storm level	Estimate
Tailings impoundment diversion	Upstream diversion to mitigate flash floods	Requirement
Storm storage capacity	3 m freeboard easily contains limited storm events	Ausenco
Tailings impoundment storage	Beach slope away from dam at start-up transitioning to full perimeter deposition	Ausenco
Material	Waste Rock with transition zones	Ausenco

18.4.3 Site Selection

The selected site for the 18-year TSF is located to the northeast of the process plant. The proximity and relative elevation of this site to the process plant facilitates tailings disposal by minimizing the tailings transport distance and reducing the pumping head required.

18.4.4 Embankment Design

The starter dam will be constructed to crest elevation 964 masl, which will allow for approximately two years of tailings disposal. The starter dam consists of a geomembrane-lined impervious upstream face, transition zone filter layers in the area below the geomembrane, and a rock fill downstream main section.

This dam is designed for water storage to allow thickened tailings to be initially deposited in the upstream portions of the impoundment, where natural drainages will effectively seal the low-lying basin areas with relatively low permeability tailings. During this time the water pool will be located at the upstream dam face. Tailings disposal will be required from the west, south, and east sides of the impoundment, including the dam. A beach slope of approximately 0.5% is anticipated.

18.4.5 Downstream Raise Dam

The TSF expansion dam will be constructed as a downstream raise dam on top of the two-year starter dam from 964 to 1,018 masl. The expansion dam consists of a rock fill and transition zone materials similar to the starter dam. The upstream face of the expansion dam will be lined for each new raise. Figure 18-14 shows the expansion dam section and details and a plan view of the expansion dam and impoundment.

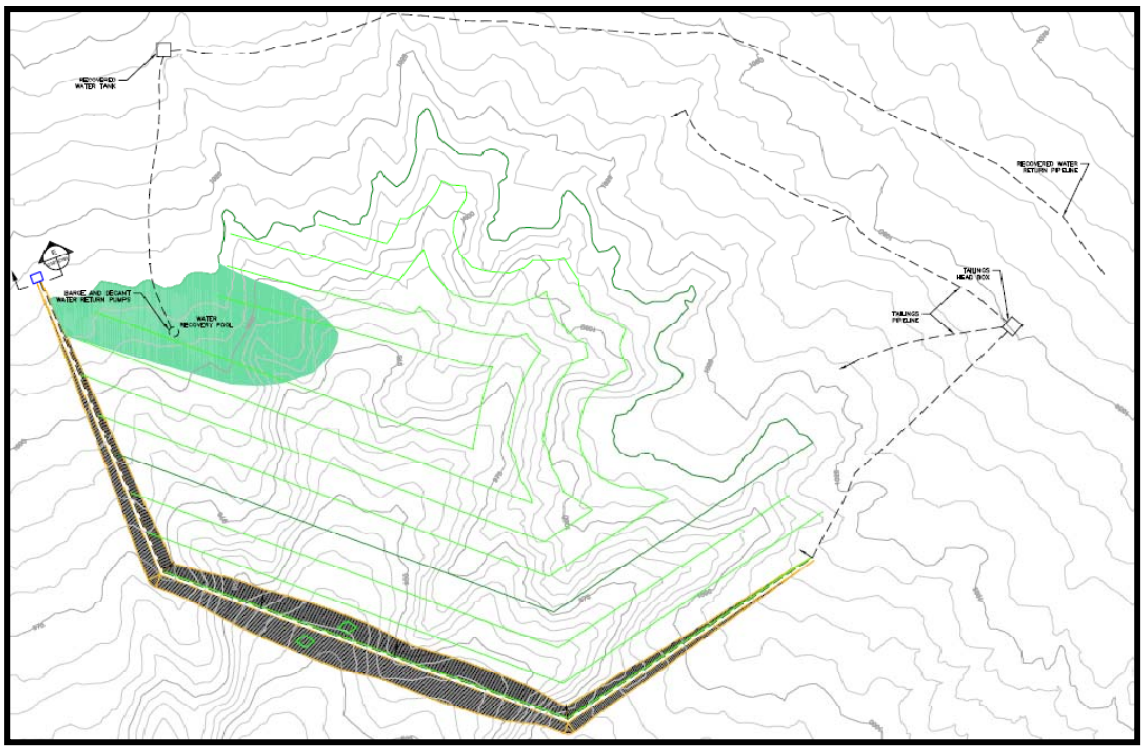


Figure 18-14: Plan View of Expansion Dam and Impoundment

18.4.6 Starter and Expansion Embankment Section

The starter dam is designed to contain the water pool at the dam face. The expansion dam uses waste rock to expand downstream and maintain tailings deposition from the dam.

Foundation Preparation

Foundation preparation will be performed for the entire footprint area for both the two-Year and the expansion embankments. There is essentially no topsoil to be removed from the area. However, there is a cemented soil 0.3 to 1.0 m thick, commonly referred to as “chusca,” which will require removal from within the dam footprint area. The chusca must be ripped and worked to provide suitable material as earthfill.

Liner System

The two-year starter dam contains a single liner system on the upstream face, consisting of a geomembrane. This provides a layer of hydrologic containment to prevent seepage from migrating through the dam face.

Internal Drains

Both the two-year dam and the expansion embankments contain a filter drain system beneath the dam, which conveys any seepage through the embankment to the blanket drain located on the valley floor. The intent of this drainage system is to provide a seepage path through the embankments and prevent seepage from saturating the downstream dam fill section for enhanced slope stability.

Seepage Cut-off

A seepage barrier is provided at the upstream toe of the starter dam. This cut-off consists of a 3 m deep trench and an injected grout curtain wall at the base of the upstream toe of the dam. In addition, this trench will serve as the lower anchor trench for the geomembrane liner on the upstream face of the dam.

18.4.7 Seepage Collection

A seepage collection pond will be located on the downstream side of the embankment. This pond will collect seepage from the TSF to prevent contaminants from leaving the TSF permit boundary limits, in addition to reducing the project water consumption.

18.4.8 Water Pool and Water Return

A water pool will be located within the TSF on the north side. Access will be provided along the east side of the TSF. This road will act as the access road to the pipeline from the impoundment water return pump. The planned alignment of the water pool access road will be determined during the FS.

18.4.9 Expansion Potential

The 18-year TSF can be expanded by raising the dam crest, which will also increase the footprint of the embankment. The design of potential expansion was not considered in detail, but appears that an additional raise of dam could provide additional storage. The dam raise will result in a larger perimeter.

18.4.10 Closure Requirements

The TSF will require a closure cap at the end of the project life. Final waste pile slopes and materials will be covered by a minimum of 5 m of inert waste fill by selective placement at the final fill levels. The tailings should be placed to accommodate long-term settlement without ponding, graded to drain to the natural drainages, and capped with a stable base of inert waste rock and a 0.5 m soil cover sourced locally. A detailed closure plan for each facility will be completed during the detailed design phase.

18.4.11 Tailings Delivery System

The tailings delivery system considered the worst case for both the start-up and expansion TSF. This included pumping tailings to 964 m for the start-up TSF and to 1,018 m for the expansion TSF.

18.4.12 Raw and Return Water Delivery System

The pumping systems for the recycle water from the TSF impoundment considered the worst case scenario. For both the TSF starter dam and ultimate dam, a floating barge of pumps will pump to a storage tank northeast of the TSF. From this storage tank the return water will be delivered back to the process plant.

19 MARKET STUDIES AND CONTRACTS

19.1 Copper

The anticipated long-term demand for copper concentrates is not easily determined. For the purpose of this study, it has been assumed that concentrate demand will continue to be strong. Gold and silver demand is also assumed to remain robust for the project period. The metal price drivers in the world economy are extremely complex and there is not any assurance that expected results will be met. Forward sales metal prices in terms of potential contracts are based on experience from actual similar operations and general knowledge.

No direct marketing has been done for the potential Santo Domingo concentrates and therefore no off-take agreements exist. Based on current industry demands it is envisioned that the copper concentrates would be best suited for smelters in Asia, namely, Japan, Korea, India or China. There is the potential for the sale of concentrate to Chilean smelters such as Las Venatanas, however, these options will be reviewed in detail when the project proceeds to the feasibility stage.

19.2 Iron Ore

The following sections are excerpts from the report “Valuation of pellet feed a report prepared for Far West Mining” CRU Strategies January 2011.

19.2.1 Overview Global Steel Industry Growth

Introduction

Over 98% of iron ore is used in the steel industry. Iron ore is one of the key raw materials in the iron making process; the other raw materials being coke and limestone in a blast furnace (BF) and natural gas in a direct reduction furnace (DRI Furnace). Iron making is the conversion of primary iron units (ore) to a product that is around 96% iron. In a blast furnace this product is known as hot metal / pig iron, and in a direct reduction furnace Direct Reduced Iron (DRI) / Hot Briquetted Iron (HBI) is produced. The next stage in the process is steel making: the refining of the products from the iron making stage into liquid steel. This process can either be accomplished in a Basic Oxygen Furnace (BOF) or an Electric Arc Furnace (EAF). The liquid steel is then cast and solidified, in this state it is referred to as crude steel.

Forecast Crude Steel Production

In the medium term, the outlook for global steel demand and in turn production, rests primarily on continued growth in China, and a slow but sustained recovery in the developed regions. Having spent their way out of the financial crisis, inflation has become a primary concern, especially amongst emerging economies, forcing policy makers to tighten monetary policy. CRU Strategies expects that China will witness brisk growth in first half of 2011, with

policy tightening measures affecting economic growth, and in turn steel production, more profoundly in the second half of the year. We expect China's GDP to grow at 9.6% and 9.7% during Q1 and Q2 2011 and then decelerate to 9.4% and 9.0% in subsequent quarters. India shall witness aggressive tightening measures during 2011 to achieve the targeted 6% inflation from the current double digit levels. This feeds into CRU's GDP growth forecast for India of 8% year¬on-year in 2011.

Over the 20110-2015 period, CRU Strategies expects global crude steel production to grow by 19% in absolute terms, and at an average rate of 4.5% per annum to reach 1.8 bn tonnes by 2015. China will remain the key driver of this growth with the country's crude steel production rising from 627 Mt in 2010 to 812 Mt by 2015, representing 50% of the global crude steel production growth. India and CIS will also be key contributors, and on a combined basis will account for 18% of the global steel production growth to 2015.

It will take until 2012 before non-Chinese steel output is back in excess of 2007 levels, although the recovery is expected to be slower in some regions (e.g. Western Europe) than in others (e.g. South America and the CIS). By 2020, global crude steel production is forecast to reach 2.1 bn tonnes.

19.2.2 Overview of Growth in Seaborne Trade of Iron Ore, 2000-2021

The seaborne iron ore market, namely iron ore shipped to other countries, totalled 992 Mt in 2010. It has experienced rapid growth in recent years, more than doubling in absolute terms, and growing at a CAGR of 8.4% per annum over 2000 to 2010. It now accounts for 45% of total iron ore consumption.

The main driver of the growth in the seaborne market has been China. The country accounted for 62% of the global seaborne market in 2010, compared to only 16% in 2000. China's reliance on seaborne iron ore grew at an average yearly rate of 24.2% over 2000 to 2010, to reach 613 Mt in 2010. Indeed, its strength of demand was the main reason that the global seaborne market grew by 8.8% year-on-year in 2009, despite concurrent declining crude steel production. Seaborne demand for iron ore outside China contracted sharply by 26% year-on-year in 2009. However, Chinese ore imports soared by 38% year-on-year, partly due to strong recovery in steel production, and also due to falling domestic ore supply in the face of lower ore prices.

Over the period of 2011-2015, CRU Strategies expects global seaborne iron ore imports to increase at a CAGR of 5.3% from 1.1bn tonnes in 2010 to reach 1.3bn tonnes by 2015. Overall, global seaborne iron ore imports will account for 55% of global iron ore demand by 2015.

The main suppliers to the seaborne market are the "Big Three" iron ore producers: Vale, BHP Billiton and Rio Tinto (which owns Hamersley Iron and the majority shares of Robe River Iron Associates (Robe), Hope Downs, and Iron Ore Company of Canada (IOC)). These three producers (including their wholly- and partly-owned subsidiaries) accounted for 58% of the seaborne iron ore market supply in 2010, around 578 Mt. Their share is expected to decline to 56% by 2015, as other producers bring supply on stream, but their share of seaborne trade will remain large and dominant.

Australia and Brazil are the leading exporters of iron ore and second and third largest producers on a gross tonnage basis. In 2010, Australia produced an estimated 422 Mt, a rise of 5.7% year-on-year, of which 414 Mt were exported (primarily to China, Japan and other countries in South East Asia). Brazil produced 307 Mt (excluding pellet feed) during the same year, a 21% rise year-on-year of which 300 Mt were exported (including pellet feed and largely to Asia and Europe). India was ranked third in 2010 with production at 200 Mt, a 2% decline compared to 2009 levels of which around half was exported (mainly to China). This was followed by the CIS at 171 Mt of which 42 Mt were exported.

CRU Strategies's projections for seaborne ore supply till 2015 include the following:-

- Australian exports will expand significantly thanks to expansions at existing producers and new entrants into the market. Over the past four years seven new entrants have commenced exporting. By 2015 we expect at least a further nine. Average annual growth in exports from 2011-2015 is expected to approach 7%.
- We project a similar rate of growth in Brazilian exports, although shipments will remain much more concentrated than in Australia with only two new entrants by 2014.
- South African exports will continue to expand, but infrastructure constraints will make it difficult to match the high growth rates of recent years.
- Strains on Indian supply are mounting as political pressures rise, internal freight constraints increase and local demand expands. Exports out of India are expected to decline progressively by 4% yearly to reach 77 Mt by 2015.

19.2.3 Spot and Contract Prices

During 2010, the mining companies managed to move the system from an annual pricing agreement to a quarterly one; effectively signalling the end of the annual benchmark system. Steelmakers conceded defeat and accepted quarterly pricing, often based on the average index price (determined from surveyed spot sales) over the previous quarter. The indexes are generally linked to daily quotations of exports of Indian ore to China. A number of indices are now published daily and some commodity exchanges are starting to clear over the counter swaps, thus creating the beginnings of a forward paper market in iron ore (the major market for OTC swaps is currently in Singapore).

Overall, the trends we see in the relationship between contract and spot prices are similar to those in other markets in that spot prices respond to short term market developments since they are one off prices for a transaction. Hence, they tend to be more volatile. Meanwhile, contract prices last for a longer duration tend to be based on previous, as well as current, market developments. The move to a quarterly system means that now iron ore contracts are negotiated more regularly the prices should be more in line with recent market developments. Moreover, the increased role of the spot market acts as a dilution of the Big Three's pricing power, although these producers are expected to maintain a dominant market position in the medium term. As a consequence, the differential between spot and contract should narrow more in the future. It also means that steelmakers outside of China are also likely to experience more volatility in their prices, and hence revenues. This may increase the attractiveness further of tools to manage this price risks, such as hedging and forward markets. It may also encourage more steelmakers to secure their own captive iron ore supply.

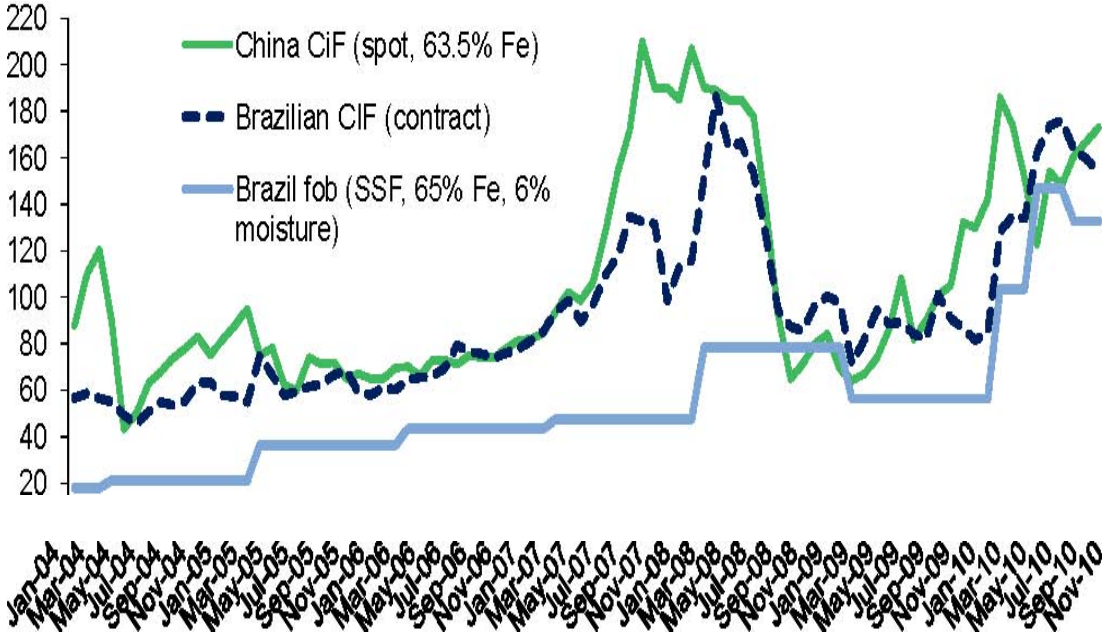


Figure 19-1: Chinese Contract Price versus Brazilian Contract Prices

19.2.4 Santo Domingo Sur Specific Price Forecast 2014 – 2021

The iron ore produced by the Santo Domingo project will be suitable to be sold as a pellet feed. A series of assumptions can be made around which the specific price forecast can be based.

- Market: the product will be sold into China as a pellet feed, most likely to one of the new world scale coastal pellet plants operated by the larger steel companies.
- Logistics: as a large bulk shipment it can be presumed that a Chinese steel mill will use a 155,000 capesize vessel to transport the ore from Chile to a port in Northern China, most likely Qingdao, a distance of 10,376 nautical miles.5 In this instance it is almost certain that any price negotiations will be based upon the steel company using a ship under a time charter agreement; the reason for a COA or owning a ship is to give the steel mill a freight advantage, this will not be ceded in price negotiaons.
- Pricing point: The large coastal pellet plants in China, which are most likely to purchase the ore from the project, will value it against pellet feed from the major supplier of imported material, in this case Brazil. Therefore, the correct benchmark price to be used will be Vale's MBR pellet feed price (fob Tubarao). The MBR pellet feed price is typically set at a 3% discount to sinter fines and this is unlikely to change in future. There is an argument to price directly against a Chinese concentrate price series, but the market for this type of product is small and in practise restricted to inland steel mills.

The specific price series is presented in the table below from 2014 until the long-term pricing point in 2021:

Table 19.1: Santo Domingo Sur specific price forecast 2014 – 2021(Nominal US c/dmtu)

	2014	2015	2016	2017	2018	2019	2020	2021
Benchmark price (Itabira Fines)	181.02	162.47	153.84	145.91	138.67	132.14	125.96	120.64
Pellet feed discount	3%	3%	3%	3%	3%	3%	3%	3%
Benchmark price (MBR Pellet Feed -Brazil)	175.60	157.60	149.24	141.54	134.51	128.18	122.19	117.03
Freight (Tubarao to Qingdao)	39.89	40.45	40.41	40.88	41.64	42.56	43.57	44.64
Benchmark price cif China	215.49	198.05	189.64	182.42	176.15	170.73	165.76	161.67
Value In Use adjusment	(0.04)	(0.03)	(0.03)	(0.03)	(0.03)	(0.03)	(0.03)	(0.04)
Freight (Qingdao to Barquito)	36.15	36.65	36.62	37.05	37.74	38.56	39.48	40.45
Netback Price (Barquito)	179.31	161.37	152.99	145.34	138.39	132.14	126.25	121.19

19.2.5 Key Findings

Findings from the CRU report are:

- The pellet feed market is growing strongly in China and should continue to do so over the next decade.
- Competition to enter this market is fierce; with scores of other pellet feed projects in a similar situation to Far West Mining.
- However, Far West Mining has an advantage in that the iron ore produced from the operation is a by-product from copper mining, rather than the sole revenue source
- Although the wider market in China is vast, obtaining a premium price for the ore may hinge on signing an off-take agreement with one or two plants.
- Many of the off-take agreements for pellet plant supply over the next three years will already have been signed; hence first mover advantage is crucial.
- Ore specifications and sizing should be acceptable to almost all pellet plants in China, but so are the specifications of many other projects. Again this reiterates the need to speak to pellet plant operators or prospective operators in China and sign some form of agreement for the plant to purchase material.
- The ore will be valued almost at parity with the MBR pellet feed product produced by Vale; this produce can be viewed as the industry standard. Hence price determination should be clear.
- Far West Mining will receive a higher fob price for their ore than competitors in Brazil due to their closer proximity to China and the associated saving on freight costs for their consumers.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Executive Summary

This section presents a summary of the prefeasibility engineering environmental study.

The objective of this study is to assess the environmental context in which the project will be carried out, and to identify: sensitive environmental aspects in the project area; relevant environmental impacts caused by the project; aspects of the project which could be improved in order to minimize the environmental impacts; and information gaps to be covered for the project's environmental assessment process. This study is also a framework within which FWM will define the environmental strategy needed to obtain the environmental permits required by the Chilean environmental assessment system (SEIA).

Primary and secondary baseline information on the project area was reviewed. It was concluded that a series of baseline studies are still required for the project in order to achieve a proper characterization of the environmental components that should be included in the Environmental Impact Study (EIA). These studies should at the least cover the entire project footprint. Among these components, water resources and marine environment characterization are critical to be developed because they require a seasonal characterization that should cover at least one year. Air quality characterization of PM25 will be required by law starting January 2012, and monthly PM10 data in the mine area are also required and should start soon in order to gather as much data as possible. Lastly, acid rock drainage (ARD) characterization of the ore, tailings, and waste should be prepared in order to address ARD-related long-term environmental effects.

The most relevant environmental impacts for the project are those related to air quality, water resources, and the social environment. Air quality impacts, although they may be considered of less magnitude than for a typical mine project given the prevailing wind conditions, are relevant to the project due to the high levels of PM10 that are currently registered in Diego de Almagro. Therefore, any increase in the PM10 levels in this city caused by the project will be a matter for discussion with Chilean health authorities. Water resources are scarce in the Atacama Region, and they would be important during the environmental impact assessment, particularly downstream of the Tailings Dam where the El Salado River has a groundwater flow of up to 70 L/sec. Proper groundwater modelling should be developed to demonstrate this and other potential aquifers will not be affected by the project's facilities at any stage. Potentially significant social environment impacts could include pressure over service levels in Diego de Almagro caused by the workers and their families, and quality of life or socioeconomic impacts during construction, operation, or closure. They should be carefully analyzed during the EIA development and discussed with the community and authorities early, especially those expected impacts for the closure and post-closure stages.

Further feasibility-level engineering and definition of the port location and other facilities will be required in order to properly assess the project's environmental impacts and prepare the EIA documentation.

From an environmental point of view, it is recommended that a thickened tailings treatment should be considered in the feasibility study and development for the project; it needs less make-up water than a conventional tailings treatment, and would therefore result in less seepage problems, reduce the impact of failure of dam structures, and reduce dust pollution. Regardless of the design selected for the TSF, FWM should demonstrate during the EIA permitting process that the TSF will not generate any significant impact to the environment and the community of Diego de Almagro.

The land and territory investigations regarding the project's current footprint indicates there would be no impact on natural parks, biodiversity conservation priority sites, or indigenous development land in the Atacama Region. However, some of these sites are close to the project and should not be impacted in order to avoid permitting difficulties.

In order to comply with International Finance Corporation (IFC) guidelines, FWM should start an early consultation process to discuss the project's features with the surrounding community and stakeholders. Stakeholders should be identified before the consultation process starts. Feedback from the community is required by the IFC. The development of the EIA document in accordance with Chilean policies and guidelines requires compliance with IFC guidelines too. A project alternative analysis and greenhouse gas emission (GHG) inventory for project operations should also be developed to comply with IFC guidelines.

An EIA-permitting schedule has been proposed in this study. If baseline studies, particularly water resources and marine environment are completed by the end of January 2012, then the EIA could be submitted to Chilean authorities in early June 2012. The environmental qualification resolution – required to move forward with the project – would be expected to be granted in September 2013.

20.2 Environmental Description of the Surroundings

This section presents the environmental characterization of the project area.

20.2.1 Physical Environment

The area where the Santo Domingo project is located has a regular desert climate, according to Köppen's classification, which is characterized by high levels of atmospheric dryness and a considerable lack of rainfall, apart from severe diurnal temperature variations.

20.2.2 Water Resources

The project is located inside the El Salado River basin, which has an area of 21,320 km^2 [21], a mostly pluvial regime, and a water resource originating in the Andean foothills. There is generally little information about the El Salado River basin, either regionally or locally.

Annual mean precipitation in the study area ranges from 10–25 mm, while annual evaporation is around 2,500 mm, indicating a serious water deficit[22].

Water Management Consultants (WMC, now Schlumberger) performed a water resource availability study in the area for FWM. Schumberger identified that the surface water resources are primarily defined by the El Salado River. Water from the El Salado River originates in the Salar de Pedernales, and is discharged to said river. It is estimated that runoff provided by the Salar de Pedernales amounts to 80 L/sec; however, the flow rate of the river upstream of Diego de Almagro amounts to 10–15 L/sec. This means some 65–70 L/sec infiltrate to the aquifer or are caught between Diego de Almagro and the Salar de Pedernales. The El Salado River, downstream of Diego de Almagro, infiltrates completely, and then reappears near the town of El Salado, due to the action of the Atacama fault.

Groundwater resources in the region, just like surface water, are linked to the El Salado River.

20.2.3 Soil

According to the information gathered during surveys in the preparation of the Santo Domingo Sur project's DIA in 2007, the study area's soils are affected by the weather's extreme aridity, which fosters the development of low-intensity genetic processes.

20.3 Biotic Environment

20.3.1 Flora and Vegetation

The *Information survey for environmental baseline study – Winter campaign* Golder Associates (Golder) report (Flora and Vegetation Report, 2010), states that all of the mine area is within the Absolute Desert sub-region, and in the Taltal's Interior Desert formation. The seawater pipeline would be located in the El Salado River valley, which extends through the Absolute Desert, the Blooming Desert, and the Coastal Desert sub-regions. In the first of these sub-regions, the seawater pipeline will be located in the Taltal's Interior Desert; in the second, the Blooming Desert of Serranías; and in the third in the Taltal's Coastal Desert formation.

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[21] http://www.sinia.cl/1292/article-26205.html

[22] General Water Administration (DGA). Chile's Water Balance, 1985.

Mine Area

In terms of richness, the Golder report concludes that most of the mine area is a denuded zone, or an area with a complete lack of vegetation and/or flora. Denuded areas are associated with ravines or depressions and alluvial cones, where most of the shrub-like specimens identified are partially senescent, and it is possible to detect a small number of specimens with low vegetative development. As well, some zones of this area have previously been affected by historic (small-scale) mining activities.

The report states that 97.3% of the flora detected in the mine area (36)species are native, of which 11 (29.7%) are non-endemic native; 9 herbs and 2 shrubs. Endemic native vegetation in the area is represented by 25 species (67.6%), of which 13 are shrubs (52%), and 12 are herbs (48%).

Of all the species detected in the mine area, none is mentioned in any of the conservation categories stated in the regulations in force[23].

Pipelines Area

As in the mine area, most of the El Salado River valley area completely lacks vegetation and/or flora. Denuded areas are associated with ravines or depressions and alluvial cones, where most of the shrub-like specimens identified are partially senescent, except in the final section near Chañaral, where specimens have good vegetative development and, in some cases, even flowers.

The survey concludes that of the total flora detected in the area, 100% are native, of which 29 (69%) are endemic native. Of these endemic native specimens, 15 (51.8%) are herbs, 13 (44.8%) are shrubs, and 1 (3.4%) is a succulent.

The baseline information analyzed does not include any comment on xerophytic formations, even when one cactus (succulent) appears in the list[24]. Regarding the *Eriosyce eriosyzoides* cactus, it is classified as a xerophytic organism pursuant to the provisions of Act 20283 and its regulations.

Of all the species detected in this area, none is mentioned in any of the conservation categories stated in the regulations in force.

20.3.2 Fauna

During the 2010 winter campaign, which covered the mine and pipelines route (El Salado River valley floor), 27 species were identified. Of these 27 species, 7 fall within some

[23] Supreme Decrees numbers 151/2007, 51/2008, 50/2008, and 23/2009, from the Presidency's General Secretariat Ministry and the Red Book of Chilean Terrestrial Flora.

[24] Unit 280, sector 309, mentioned in the "Flora and Vegetation Report" of Golder Associates, October 2010.

conservation category according to the regulations in force[25]: 1 is classified as Insufficiently Known (Chilla fox), 4 are classified as Vulnerable (Chilean lizard, lava lizard, grey gull, and Inca tern), and 2 are classified as Endangered (tree iguana and vizcacha).

Of all the species found, the 3 reptiles are endemic to the area (*Liolaemus velosoi, Microlophus atacamensis* and *Callopistes palluma*). Of the remaining species, 3 were brought to Chile: the hare (*Lepus capensis*), the rock pigeon, and the house sparrow (*Passer domesticus*). The remaining 21 species (78.8%) are native.

20.4 Marine Environment

Current background information does not include any data on the marine environment. There is only some information on the zoning of the coastal border.

20.5 Social Environment

20.5.1 Human and Built Environment

The city closest to the mine area is Diego de Almagro. Mining has been the main economic activity of the city since the beginning of the 20th century (1900).

According to the 2002 census, the area of Diego de Almagro has 18,589 inhabitants, 54% male and 46% female. Urban population accounts for 95% of the total inhabitants. The National Statistics Institute (INE) projected for 2008 a population decrease of 27.8%.

With regard to education, in the area there are 14 schools: 9 are privately subsidized, and the rest belong to the municipal corporation. All together, these 14 schools educate 4,380 students (Ministry of Education [MINEDUC], 2007).

In the Diego de Almagro area there is a hospital, an outpatient health center, and a rural emergency centre (Ministry of Health [MINSAL], 2007). Of the total inhabitants in the area, 2,810 are registered in the municipal health service (National System of Municipal Information [SINIM], 2006).

The Chilean government's official records state that there are no indigenous settlements in the project's location area.

[25] Supreme Decrees numbers 151/2007, 51/2008, 50/2008, and 23/2009, from the Presidency's General Secretariat Ministry and the Hunt Act (SAG, 2009).

Table 20.1: Main Area and Regional Indicators

Indicators	Diego de Almagro
Poverty rate	4.2%
Illiteracy rate	2.2%
School years	10.3
Workforce	61.2%
Mean income (peso/month)	670,866
% of houses in good conditions	78.8
% of houses in acceptable conditions	2.2
% of houses in poor conditions	11.8
% of people connected to the drinking water network (*)	99.4
% of people connected to the sewage network (*)	96.8
% of people connected to the public power grid (*)	99.3
Birth rate (per 1,000 inhabitants)	17.6
Child mortality rate (per 1,000 children born alive)	14.2

Source: Table prepared internally based on the information of the CASEN 2006 survey. (*): 2002 Census.

20.5.2 Cultural Heritage

In the Atacama Region there are currently certain human settlements linked to livestock breeding, agricultural, and mining activities with a longstanding cultural tradition. In recent years, this tradition has been acknowledged by the Chilean State, which has granted legal status to two ethnic groups of the Atacama Region: the Collas and the Diaguitas. This legal status goes hand-in-hand with a special acknowledgment of the Atacama region's cultural, historical, and anthropological (ethnographic) heritage, based on the pre-Hispanic archaeological heritage. It is important to mention there is no officially recognized Collas or Diaguitas native land or settlements in the project's areas.

According to the baseline studies carried out for the Santo Domingo project DIA, there are no archaeological deposits protected by Act 17288 on National Monuments, except for one found in the mine area, where lithic carved waste and historic materials were found. This finding was classified as from the historic pre-Hispanic time and it is located about 300 m west of the larger project waste dump.

It is important to mention that most of the project areas have not yet been prospected by an archaeologist.

20.6 Project's Preliminary Environmental Impact Assessment

20.6.1 Environmental Impacts Identification

The following is a preliminary Environmental Impact Assessment for the Santo Domingo project that takes into account the project's available information as well as the information on the environment.

The environmental impact assessment prepared for the project was rather generic, based on typical impacts generated by mine projects and the consultants' experience, for two reasons: first, the project being in a prefeasibility stage, some of the required data regarding works and facilities details were unavailable or pending definition; and second, baseline studies for most of the environmental components have not yet been carried out.

Nevertheless, it is possible to make an overall diagnosis of the eventual impacts that may be generated by project activities on the several environmental components, although the magnitude, duration, and degree of reversibility of such impacts are complex to estimate.

Table 20.2 shows a summary of the main impacts preliminarily identified, as well as the recommended actions to take.

Table 20.2: Summary of Main Environmental Impacts Identified

Impact	Environmental Component	Possible Impacts	Recommended Actions and/or Mitigation Measures
A1	Air	Air quality disturbance at Diego de Almagro and the camp sector (workers)	Assessment: apply air contaminant dispersion model in order to quantify the impact and verify the efficacy of emission mitigation measures. Mitigation: apply dust suppression measures, like road sprinkling, sprinklers at material transfer points, conveyor belt protection system, and others. Compensation: given the latency situation in Diego de Almagro, some emissions (to be determined) may need to be compensated for by means of reduced emissions from other sources.
A2	Air	Effects on Road Visibility	Assessment: apply air contaminant dispersion model. Mitigation: apply dust suppression measures, like road sprinkling, sprinklers at material transfer points, conveyor belt protection system, and others.
A3	Air	Effects generated by works on the coast	Assessment: apply air contaminant dispersion model. Mitigation: apply dust suppression measures in the port facility.
S1	Soil	Loss of soil due to surface occupation	Assessment: given the current soil use capacity, this impact would not be significant.
G-WR1	Geology/water resource	Generation of acid rock drainage (ARD) from mine facilities	Assessment: ARD characterization of ore, waste and tailings. If ARD is present, a long-term geochemical model assessment will be required. Mitigation: to be detailed in the mine closure plan. Design for closure.
WR1	Water	Modification of	Assessment: groundwater modelling to

Impact	Environmental Component	Possible Impacts	Recommended Actions and/or Mitigation Measures
	resources (water quantity)	groundwater flows Depression of the small aquifer and alteration of groundwater flow direction	estimate whether the scarce underground flow will be reduced by project mining works. Mitigation: minimize the surface to be occupied by the project works. Compensation: only in the event that the project generates a flow reduction will it be necessary to compensate by acquiring water rights or by directly negotiating with users affected.
WR2	Water resources (water quality)	Effect on water quality (particularly after mine closure)	Assessment: same as for G-WR1 with the addition of modelling of TSF seepage and a hazardous substances risk assessment. Mitigation implementation of spill containment measures for hazardous substances, such as retaining guard-rails/walls, substance handling areas with insulating covers, and others.
B1	Biodiversity	Intervention in flora specimens individuals under conservation category	Assessment: check for sensitive areas within the species under conservation category or endemic. Mitigation: transplant key species individuals (cactus specimens) to areas that will not be intervened.
B2	Flora and Fauna	Intervention in critical flora and fauna environments.	Assessment: as with B1. Mitigation: minimize the flora environment surface to be intervened.
B3	Flora and Fauna	Intervention of fauna individuals under conservation category	Assessment: as with B1. Mitigation: fauna rescue and relocation programs. Key habitat protection. Monitoring programs.
MS1	Social Environment	Increase in local employment rate (positive)	Foster the hiring of local labour and prioritize local service providers.
MS2	Social Environment	Introduction of resources in the local economy (positive)	Foster the hiring of local labour and prioritize local service providers.
MS3	Social Environment	Reduction in employment rate and resources in the local economy (at project closure)	Assessment: assess social effects that will take place at the project closure. Mitigation: support local development plans in other non-mining activities.
MS4	Social Environment	Alteration of the cultural heritage (archaeological sites)	Assessment: superimpose future baseline findings upon project layout. Mitigation: site protection. Compensation: archaeological enhancement strategy measures (research and publications).

20.6.2 Environmental Impact Assessment

The following is an analysis of the possible impacts the project might have on various environmental components, based on characteristics of the surroundings and the typical effects for the various planned works and activities.

Air Quality

Air quality is one of the most sensitive environmental components that may suffer a negative impact caused by mining projects, mainly due to the particulate matter emissions generated by large volumes of transported and processed materials. This impact will depend on local ventilating conditions.

The largest impact on air quality will occur in the mine area. In this area, the main identified receiving environments susceptible to air quality deterioration are the city of Diego de Almagro and the project camp, both located to the north of the mine area.

Based on the Santo Domingo project's characteristics, the main particulate matter and gas emission sources will be:

- drilling and blasting
- loading and unloading of materials
- truck and vehicle circulation
- crushing operations and conveyor belt transfers
- erosion from waste rock dumps and tailings dams
- truck, vehicle, and machine engine exhaust.

The emission rate of all sources, except for erosion-related ones, will depend on the production level at the mining site. The possible effects that might be generated by such emissions are described below.

Soils

Soils in the project site area are highly dry due to desert-like weather conditions. The soils have salt accumulation, carbonate-hardened or cemented horizons, and no vegetation. However, there is no soil usage capacity information available; nevertheless, given the weather characteristics and the scarce vegetation, it must be unfit for any agricultural use or grazing. In this scenario, the project works and facilities site would imply the loss of soil due to surface occupation, entailing a negative impact; this impact, however, would not be significant on account of the soil usage capacity mentioned above. Nevertheless, it is recommended to perform a new analysis once more baseline study data are available.

Regarding the impact on the soil that may be generated by port activities, it should be noted that works would be located in a zone already intervened; hence, no additional soil that may be affected will be used.

Water Resources

Water is the most troublesome natural resource in the region because of the difficult access and shortage. Understandably, sustainable water availability is one of the strategic goals set forth in the government of Chile's *Plan Regional de Atacama* (Atacama Regional Plan) (2010 – 2014)[26]. Project impacts on water resources would be related to the extraction and/or capture of water mostly generated by the open pit operations, and water quality deterioration that may be caused by TSF seepage and/or ARD generation. A description of the water resource impacts assessment follows below.

Flora and Fauna

Available information on flora and fauna specify that the mine and seawater pipeline areas are essentially lacking vegetation or flora. Vegetation is only found in ravine zones, and mostly senescent individuals. At the project site zone, a type of xerophytic species was identified, that is legally protected. Accordingly, it will be necessary to check for any impact on this species once more data on project engineering and the flora and vegetation baseline study are available.

Fauna information indicates that around 27 species were found in the zone, out of which 7 are under some conservation category, 2 of these classified as endangered. The project could potentially have an impact on these species, but the application of management measures such as transferring the species or installing barriers are deemed to be effective to mitigate such potential impact.

It is important to note that a better representation of key environments for endemic or conservation-status fauna should be included in the baseline studies.

Socioeconomic, Heritage, and Built Environment

Social Environment

Diego de Almagro has a long tradition of mining, which has created a mine culture among its inhabitants, thus favouring the development of mine projects. Regardless of this fact, the Santo Domingo project will necessarily generate both positive and negative impacts on its socioeconomic environment. The levels of some of these impacts will primarily depend on the decisions to be made on some issues, for instance labour sourcing and the project camp planning during the operation stage.

The fact that Diego de Almagro is located very near the project site allows for hiring at the least unskilled labour directly from its population. This would have a positive impact by increasing the local employment rate and introducing resources into the local economy, while creating no excess demand for basic services as they would be used by the same people using them at present. Moreover, Diego de Almagro's population would be encouraged since they would see the project as a potential source of employment. Nevertheless, should Diego de Almagro become very dependent on the project, the ensuing impacts will be negative at

[26] Government of Chile, Atacama's Regional Plan (2010-2014), 2010.

project closure, as the local economy would be deprived of a significant source of income. This matter should be given considerable thought.

Diego de Almagro's capacity to provide social services should also be checked, so as not to overload them. Overloading social services would result in the environmental authority requiring an investment in social infrastructure (with an ensuing increase in the project's capital costs).

Cultural Heritage

The information on cultural heritage points out that in the mine area at least one archaeological finding has been identified, and future archaeological prospecting on the project site area may increase the number of findings.

Impacts on archaeological findings should be avoided. Likewise, related sites must be identified and fenced in order to protect them. Should a site be affected, such measures as extensive excavations, rescue of materials, and surveys should be implemented.

Built Environment

The main impact the project will have on the built environment involves the intersection of the project works with Road C-17 (the open pit) and Road C-167 (the tailings dam).

Road C-17 will be intercepted by the Iris Norte pit and by the process plant. This impact would be compensated for through the construction of an authorized bypass on Road C-17, although given its proximity to the open pits and the process plant, an impact on visibility would likely be generated by particulate matter emissions from the mine.

Proposed road bypasses should be negotiated in advance with the Atacama Highway Administration (*Dirección de Vialidad de Atacama*), so as to avoid rejection of the project during its permitting process.

20.6.3 Other Environmental Impacts

Given the characteristics of the project's works and activities, other identified potential impacts are:

- Activities which would generate an impact on noise and vibration levels are mainly drilling and blasting. As a rule, this type of impact is sporadic and short. An impact on noise levels may also be generated by project-related trucks travelling on roads C-13 and C-17.
- Distribution of road traffic linked to the project should be assessed in advance in order to not cause disturbances to current traffic flows in the area. Given the robust existing highway infrastructure in the area, this impact should not be significant.
- No information on the marine environment is available. However, as seawater will be extracted, the marine environment will be affected, therefore an environmental impact would likely be generated. This impact should be assessed once related engineering

information is available. The marine environment is a sensitive issue that requires detailed study in order to avoid any issues during the environmental assessment.

- Project implementation would unavoidably generate a landscape impact on the zone; this should also be properly assessed.

20.7 Summary of Territorial Analysis of the Project Location

From analysis of the available territorial information, it is possible to make the following comments.

According to the current Communal Regulatory Plan of Diego de Almagro, the mine area is located in a rural area, and is not regulated by any territorial planning instrument. However, a new Communal Regulatory Plan is being validated and approved by the Chilean government's General Finance Office, so approval will be required to assess compatibility of the project with soil use in the sector. According to information from Diego de Almagro approval will be obtained by the end of year 2011.

The coastal area is suitable for the use proposed by the project, provided that the works are located within the ZU14 area, which allows the deployment of productive activities and complementary equipment.

20.8 Environmental Issues and Information Gaps

20.8.1 General

This section analyzes the relevant environmental aspects of the project that could become the main difficulties for an adequate assessment and thus development of the project.

20.8.2 Recommendations for Environmental Issues Management

Social Aspects

Experience shows that similar mining projects usually require countervailing actions for affected communities, generally financial or social support. Thus, a strategy should be developed to interact with the community, based on an adequate communication and trust building so that the project is regarded as a benefit for the community.

This strategy will identify key players in Diego de Almagro (stakeholders), learn their opinions regarding the project (feedback), establish good relationships with them, and consider previously assessed countervailing action proposals. The aforementioned are to be considered as requirements as per guidelines from the IFC and the World Bank related to approving the external funding of mining projects. According to these guidelines, it is necessary to conduct citizen consultation before submitting the EIA to the authorities and show that the stakeholders' opinions were taken into account in the project design.

20.8.3 Information Gaps and Additional Studies Required

The main information gaps for the environmental assessment of project are presented in Table 20.3, Table 20.4, and Table 20.5. The tables include a description of the information gaps regarding the project, the environment, and the socioeconomic environment, respectively.

Table 20.3: Project Description Information Gaps

Information Gap	Comment	Required Action/Study
Tailings dam	Further background information on the tailings dam engineering is required. The prefeasibility engineering information does not indicate if the tailings dam has a safety spillway as a hydraulic work. In similar projects, the General Water Administration (DGA) has required a safety spillway. There is no risk analysis available for the tailings dam.	Add a safety spillway to the tailings dam design. Define the seismic criteria for the tailings dam, according to the regulations in force. Conduct a risk analysis to the tailings dam that includes the environmental and social variables. Prove compliance with S.D. No. 248/2007
Concentrate pipe layout	It is necessary to consider the ravines, populations, roads, and natural risk areas it will travers.	Design the concentrate pipe layout with consideration for land restrictions. Conduct a risk analysis of the concentrated pipe layout, which includes the environmental and social variables.
Filter plant at the loading port	A filter plant would be installed at the project's port facilities.	Specify the filter plant design. Define the filter plant wastewater disposal.
Desalination plant	Desalination plant design characteristics, capacity, efficiency, etc. are not yet available.	Define desalination plant design characteristics.
Acid drainage	No study characterizing potential acid drainage generation is available; this is needed for measuring possible future environmental effects and for material management measures.	Conduct an acid drainage characterization study on the ore, waste rock, and tailings.
Closure Plan	There is no available information regarding life of mine projections, which would be used to temporarily measure closure activities.	Add closure criteria to the present engineering designs. Prepare a preliminary closure plan.
Risk Management Plan	No generalized risk analysis for the project is available (at least at the conceptual level) which would allow for identification of the main threats.	Conduct a general risk analysis of the project, which includes the environmental and social variables.

Table 20.4: Environmental Studies Information Gaps

Information Gap	Comment	Required Action/Study
Air Quality	There is a lack of information regarding: - Baseline air quality information for the Mine area and port. - Meteorology in the port area. - Source and sufficiency of water for mitigation measures.	Conduct air quality monitoring according to the regulations. Start continuous monitoring of PM2.5 and PM10 in both the port and mine areas. Complete prefeasibility or feasibility study to start emission estimations and analyze the project's effects on air quality.
Wildlife, flora, and vegetation	Studies should include a seasonal analysis of the wildlife, flora, and vegetation.	A seasonal analysis should be prepared, at least summer and winter. A spring campaign is also recommended. Studies should cover all project areas (to verify the feasibility layout).
Marine Environment	No information survey is available regarding the marine environment.	Conduct a study of the marine environment in the selected port area. Marine baseline studies should be performed according to the requirements of the Marine Territory and Merchant Navy General Agency, which has a methodological guide and includes as some of the basic studies the following: - bathymetry - coastal winds - waves and tides (direction) - structure of the water column (including temperature, salinity, and dissolved oxygen,) - water quality and marine sediments - benthonic communities.
Noise and vibrations	There is no noise and vibration baseline available, or any estimation of the project's emissions.	Baseline noise and vibration monitoring. Estimation of project noise and vibration emissions. Acoustic impact assessment.
Archaeology	There is insufficient archaeological information available to determine the existence of archaeological sites.	A larger archaeological study should be conducted in all project areas.

Table 20.5: Socioeconomic Environment Information Gaps

Information Gap	Comment	Required Action
Diego de Almagro population	There is no information identifying the main stakeholders, neighbours' associations, local authorities, NGOs or environmental groups. There is also no available information on the following issues: social perception of mining projects in Diego de Almagro current needs in Diego de Almagro population abilities the capacity of currently available services in Diego de Almagro.	Perform a social characteristics study in Diego de Almagro. Conduct baseline studies, considering the "Guidelines to assess the significant disturbance of human groups' life and customs systems in projects or activities entering the SEIA". Conduct local citizens' consultation in the towns involved in the project, once more definitive information regarding the project is available. Assess impacts to quality of life of the Diego de Almagro population (i.e., air quality, noise, increased traffic). Assess the impacts on demand of available services at Diego de Almagro (health, education, justice, recreation, etc.). Assess the impacts on employment and income during all project stages (construction, operation, and closure).
Road impact	Once the means of transport to be used by the project are defined, it will be necessary to estimate their impact on existing road infrastructure. It is recommended to start negotiations with the Chilean Roads Service in order to secure approval of the road detours required by the project.	Road impact baseline study. Modelling of the project's effect on existing road infrastructure. Start communications with the Chilean Roads Service regarding project impacts.

20.9 Applicable Regulatory Aspects

20.9.1 Introduction

The following is a preliminary analysis used to determine the environmental regulations in force that would be applicable to the Santo Domingo project. Compliance with said regulations will be enforceable within the framework of the project's environmental assessment process.

The applicable regulations were first analyzed in light of general environmental regulations – the Chilean constitutional right to live in a place free from pollution – and the Environmental Impact Assessment System (SEIA). Subsequently specific environmental regulations dealing with legal and regulatory provisions applicable to the project were analyzed, based on the project's relevant environmental components.

20.9.2 General Environmental Regulations Applicable to the Project

Table 20.6 shows the environmental regulations identified as applicable to the Santo Domingo project.

Table 20.6: General Environmental Regulations Applicable to the Project

Regulation	Description	Applicability to the Project
Political Constitution of the Republic of Chile	Sets forth the Constitutional Guarantee of the right to live in a place free from pollution.	It is the founding regulation of the environmental legislation.
Act 19300 on General Environmental Bases, modified by Act 20417	Act 19300 contains general provisions and sets forth the environmental management tools to be used by the State. Some of these are the SEIA, regulations on environmental responsibility, and environmental monitoring.	Taking into consideration the fact that the Santo Domingo project will be assessed for its environmental impact, this should be done pursuant to the provisions of Act 19300.
Supreme Decree No. 30/97, Regulations of the Environmental Impact Assessment System, modified by SD No. 95/01, both from the Presidency's General Secretariat Ministry (MINSEGPRES).	Sets forth the provisions that govern the SEIA and Community Engagement, pursuant to the contents of Act 19300 on General Environmental Bases. Among other things, it specifies the kind of projects or activities likely to have an environmental impact, at any stage, which should be submitted to the SEIA. It also defines the environmental assessment process of the projects or activities analyzed according to the SEIA by the State that are part of the assessment.	The Santo Domingo project's environmental assessment will be done pursuant to the provisions set forth in the Regulations of the SEIA.

20.9.3 Specific Environmental Regulations Applicable to the Project

Having considered the project's characteristics and location, the specific environmental regulations deemed applicable are related to the following environmental components:

- air
- noise
- water
- soil
- flora and vegetation
- fauna
- archaeology and cultural heritage
- human environment
- hazardous substances (fuel, explosives, electrical equipment)
- solid waste (hazardous and non-hazardous industrial waste)
- liquid waste
- light pollution
- transport.

20.9.4 Sectorial Environmental Permits

Projects that have to be environmentally assessed must comply with a series of sectorial environmental permits (PAS), defined in the sections of Title VII of the Regulations of the Environmental Impact Assessment System.

20.9.5 Proposed Environmental Permitting Schedule

In this section, a permitting schedule is proposed.

The EIA document strategy is to first prepare the baseline studies, with most completed by November 2011. The exceptions are those for water resources, marine environment, and noise and vibration, which require seasonal campaigns. During FS development, a task named “Environmental Design Criteria/Inputs” was included to provide for interaction between the environmental consultant, FWM, and the engineering firm in charge of the FS, with the objective of incorporating environmental variables in the design. In this way the Santo Domingo project will comply with Chilean regulations and also with the environmental, health and safety guidelines of the International Finance Corporation (IFC).

Once the FS design has been frozen, EIA document preparation will begin. Baseline information from the last (summer) campaigns will be added at that time and the environmental impact assessment will be prepared. The Rev. B will be issued for FWM’s review at the end of June 2012. This version of the EIA document will include only preliminary baseline and impact assessment results for marine environment, water resources and air quality, because baseline studies are expected to be completed by the end of July 2012. A second, complete version of the EIA document (Rev. C) will be issued by the end of September 2012 for FWM final review. Then, the EIS Rev.0 would be presented to the SEIA by the end of November, 2012.

The critical path consists of four tasks (in order of timeline):

1. development of the water resources baseline study
2. development of the marine environment baseline studies
3. development of the FS
4. development of the EIA.

21 CAPITAL AND OPERATING COST ESTIMATES

21.1 Capital Cost Estimate

The capital cost of the project has been estimated based on the scope defined in previous sections of this report. The following parties have contributed to the preparation of the capital cost estimate in specific areas:

- Ausenco
  - process plant
  - plant infrastructure and services, including road diversion, high-voltage transmission line, and high-voltage substation
  - TSF
  - concentrate and seawater pipelines
  - port facility, including associated infrastructure
  - permanent accommodation camp
  - mine workshop
  - EPCM costs relating to the process plant and infrastructure outlined above.
- NMS/SRK
  - initial mine development costs
  - mine working capital
  - haul road.
- Far West Mining
  - Owner's cost.

21.1.1 Estimate Summary

Table 21.1 provides the summary of capital costs estimated for the study. The costs are expressed in Q2 2011 US dollars. There are no allowances for escalation. The estimated costs include all mining, site preparation, process plant, pipelines, port, first fills, buildings, road works and the accommodation camp. The estimates are considered to have an overall accuracy of -20% +25% and assume the project will be developed on an EPCM basis.

The following parameters and qualifications are made:

- estimate was based on Q2 2011 prices and costs
- financing related charges (e.g., fees, consultants, etc.) are excluded
- no allowance has been made for exchange rate fluctuations.

There is no escalation added to the estimate, other than the contingency.

Data for these estimates have been obtained from numerous sources, including:

- prefeasibility level engineering design
- mine plan
- topographical information obtained from site survey
- geotechnical investigation
- budgetary equipment proposals
- budgetary unit costs from local contractors for civil, concrete, steel and mechanical works
- data from recently completed similar studies and projects
- information provided by John Nilsson/SRK.

Major cost categories (permanent equipment, material purchase, installation, subcontracts, pipelines, indirect costs and Owner's costs) were identified and analyzed. To each of these categories, a percentage of contingency was allocated based on the accuracy of the data, and an overall contingency amount was derived in this fashion. Using this method, an overall contingency of $148 M was obtained, representing 13.6% of the total capital cost, or 14.6% less mining equipment and development. The typical range for contingency on prefeasibility studies is 14% to 25% of the total project cost. The percentage of 14.6% falls within the lower range, but seems reasonable given the amount of detail that was incorporated into this study.

Table 21.1: Capital Costs by Area

Direct Costs		Permanent Equipment Imported Total (US$M)	Permanent Equipment Domestic Total (US$M)	Contractor Material (US$M)	Installation Cost (US$M)	Sub-contract (US$M)	Total Cost (US$M)
01	**MINING**						
01-30	OP Mine Development	0.0	0.0	0.0	0.0	54.3	54.3
01-60	OP Mobile Equipment	171.9	0.0	0.0	0.0	0.0	171.9
02	**CRUSHING**						
02-10	Primary Crushing	8.0	1.1	3.6	5.9	1.0	19.6
02-20	Crushed Ore Conveyor	2.2	0.0	1.4	1.5	0.0	5.1
02-40	Crushed Ore Stockpile and Reclaim	4.8	1.4	2.2	4.0	0.6	13.1
03	**PROCESS**						
03-10	Process General - Process Plant Building	0.0	0.3	0.0	0.0	0.0	0.3
03-20	Grinding Circuit General	83.0	0.8	14.0	21.3	0.8	119.9
03-30	Copper Flotation	21.8	0.4	0.6	8.4	0.4	36.8
03-40	Magnetic Separation	14.5	0.4	3.6	3.8	0.3	22.7
03-60	Copper Concentrate Storage Handling	1.2	0.4	1.2	1.6	0.1	4.5
03-70	Iron Concentrate Storage Handling	0.9	1.7	0.5	1.3	0.2	4.6
03-80	Reagents	1.6	0.2	1.4	1.6	0.5	5.4
04	**CONCENTRATE LOADOUT / PUMPING**						
04-50	Concentrate Pumping	0.1	5.2	0.6	0.8	0.3	7.0
05	**TAILINGS**						
05-10	Tailings and Reclaim Water	7.0	0.1	1.3	4.2	16.8	29.4
06	**UTILITIES**						
06-10	Air Systems	1.6	0.0	1.0	1.3	0.0	3.9
06-20	Water Systems	0.9	0.2	6.9	5.8	0.0	13.8
06-30	Sewage and Waste Water Systems	0.1	0.6	0.2	0.2	0.0	1.1
06-50	Electrical Services	19.9	0.0	0.3	1.3	1.9	23.4
06-60	Process Control System	1.2	0.0	0.0	0.0	0.0	1.2
06-70	Plant Area Lighting	0.0	0.1	0.2	0.2	0.0	0.5
07	**ONSITE INFRASTRUCTURE**						
07-10	Site Civil Infrastructure	0.0	0.0	0.6	1.1	0.2	1.9
07-30	Desalination Plant	2.0	0.1	0.1	0.3	0.4	3.0
07-50	Plant Mobile Equipment	3.3	0.0	0.0	0.0	0.0	3.3
07-60	Workshop and Ancillary Buildings	0.1	0.0	0.2	5.2	2.7	8.3
07-80	Mine Services Facilities	0.9	0.6	1.0	2.0	6.3	10.9
08	**OFFSITE INFRASTRUCTURE**						
08-10	Site Power Supply	0.0	0.0	0.0	0.0	6.1	6.1
08-40	Concentrate Pumping and Pipeline	0.0	0.0	0.0	0.0	49.3	49.3
08-50	Concentrate Filters	21.3	3.2	2.1	3.8	2.3	32.6
08-60	Port Facility	27.5	1.2	8.0	19.3	32.7	88.8
08-80	Seawater Pumping	1.0	0.0	1.9	2.9	69.9	75.7
	Total Direct Costs	**396.8**	**18.1**	**58.3**	**98.0**	**247.1**	**818.3**
09	**INDIRECT COSTS**						
09-10	Temporary Facilities and Services						7.7
09-30	Construction Camp						35.1
09-40	Cranage						4.1
09-50	Freight and Logistics						21.3
09-60	EPCM Costs						94.7
09-70	Commissioning Support						2.4
09-80	First Fills / Spares - including mining equipment						20.7
10	**OWNER'S COSTS**						
10-10	Owner's Cost – Labour						19.9
10-20	Owner's Cost – Insurance						8.2
10-30	Owner's Cost - Pre-production						18.6
10-40	Owner's Cost – Feasibility Study/In fill & geotech drilling /hydrogeology/SEIA preparation/Land Access						42.0
	TAXES AND DUTIES						**NOT INCLUDED**
	Total Indirect Costs						**274.7**
	Total Direct and Indirect Costs:						**1,093.0**
11-10	**ESCALATION**						**NOT INCLUDED**
11-10	**CONTINGENCY**					**13.6%**	**149.3**
Total Project Cost, (as at May 2011) (US$)							**1,242.3**

21.1.2 Open Pit Mine Capital Cost Estimate

The Santo Domingo LOM capital cost estimate is shown in Table 21.2.

Table 21.2: Summary of Capital Costs

Area	Unit	Cost Estimate
OP mining equipment fleet (To Year 1)	M$	**171.9**
Sustaining Capital (Year 2 onwards)	M$	149.6
Spares	M$	9.1
Contingency	M$	16.1
Salvage	M$	-18.3
TOTAL CAPITAL COST	M$	328.5

21.1.3 Open Pit Mobile Equipment

The capital cost estimate for the open pit operation is based on the scheduled plant processing throughput rates (based on total iron grades) as well as comparing to similar sized open pit operations (maximum processing throughput of 25.6 mtpa). The open pit mining activities for the Santo Domingo pits were assumed to be undertaken by an owner-operated fleet as the basis for this pre-feasibility study with the fleet having an estimated maximum capacity of 300,000 tpd total material, which will be sufficient for the proposed LOM plan.

The open pit equipment capital costs required to achieve the target processing rate is summarized in Table 21.3 below. Budgetary quotes from regional equipment suppliers, mining cost service information and factors based on experience were taken into consideration in determining the open pit capital cost estimate. Estimated freight costs to site are included. No equipment was considered as lease as the terms were not favourable when compared to purchase. Some of the open pit mobile equipment fleet will provide a salvage value as noted above.

21.1.4 Open Pit Development

Pre-stripping requirements were estimated using Year -1 total mined tonnage of 50 Mt and was considered as part of the overall open pit capital cost. Using the estimated average mining cost for the year, a capital cost of $US54.3M is allocated to pre-stripping for the various open pits.

Given that the Santo Domingo project lies in the arid Atacama Desert, and that vegetation is very sparse and hillsides and peaks are generally devoid of any vegetation no clearing and grubbing of the various pit areas and waste dumps were included in the open pit capital cost estimate.

Table 21.3: Open Pit Eqipment Capital Cost Summary

Item	Unit	Unit Cost	Initial units	Replacement units	Total units	Total
Primary						
Crawler-Mounted, Rotary Tri-Cone, 12.25-in Dia. D	US$M	$ 4.2	6	4	10	**41.8**
Crawler-Mounted, 4.5-in Dia.	US$M	$ 0.7	1	1	2	**1.4**
Diesel, 34-cu-yd Front Shovel	US$M	$ 8.8	4	2	6	**52.9**
Diesel 33-cu-yd Wheel Loader	US$M	$ 4.6	1	1	2	**9.1**
240-ton class Haul Truck	US$M	$ 3.7	25	20	45	**165.2**
D10-class 17.3' blade	US$M	$ 1.4	6	4	10	**13.8**
834H-class 15.2' blade	US$M	$ 1.1	3	2	5	**5.3**
16H-class 16' blade	US$M	$ 0.9	5	3	8	**6.9**
100 ton class (20,00 gal.)	US$M	$ 1.8	2	1	3	**5.3**
Subtotal Primary	US$M					**301.7**
Ancillary						
ANFO/Slurry Truck, 12-ton	US$M	$ 0.2	2	1	3	**0.6**
Stemming truck, 15-ton	US$M	$ 0.1	2	1	3	**0.3**
Powder Truck, 1-ton	US$M	$ 0.1	2	1	3	**0.2**
AN Storage Bin, 60-ton	US$M	$ 0.1	1	0	1	**0.1**
Powder magazine, 24-ton	US$M	$ 0.1	1	0	1	**0.1**
Cap magazine, 3.6-ton	US$M	$ 0.0	1	0	1	**0.0**
385C Excavator (backhoe), 6 cu-yd	US$M	$ 1.0	1	0	1	**1.0**
Haul Truck (road constr), 35-ton	US$M	$ 0.5	3	0	3	**1.5**
Backhoe/Loader, 3.3 cu-yd	US$M	$ 0.4	1	0	1	**0.4**
Portable Aggregate Plant,250 tph	US$M	$ 1.0	1	0	1	**1.0**
All-terrain Crane, 60-ton	US$M	$ 0.6	1	0	1	**0.6**
Transporter w/Tractor, 100-ton	US$M	$ 0.4	1	0	1	**0.4**
Fuel truck, 5000-gal	US$M	$ 0.3	2	2	4	**1.1**
Lube/Service Truck	US$M	$ 0.3	2	2	4	**1.3**
Mechanic Field Service Truck	US$M	$ 0.2	5	5	10	**1.8**
Off-Road tire handling Truck	US$M	$ 0.4	2	1	3	**1.1**
Int. Tool Carrier, 140-hp	US$M	$ 0.2	2	1	3	**0.6**
Light Plant, 6-kW	US$M	$ 0.0	10	10	20	**0.4**
Pickup Truck, 0.75-ton, 4-WD	US$M	$ 0.1	15	15	30	**1.5**
Crew Van, 1-ton, 4-WD	US$M	$ 0.1	5	5	10	**0.6**
Mobile Radio, installed	US$M	$ 0.0	97	71	168	**0.1**
Subtotal Ancillary	US$M					**14.5**
Miscellaneous						
Shop Equipment	US$M	$ 0.8	1	1	2	**1.5**
Eng & Office Equip plus Software	US$M	$ 0.7	1	1	2	**1.3**
Truck Dispatch System	US$M	$ 1.5	1	0	1	**1.5**
Radio Communications System + GPS	US$M	$ 0.5	1	1	2	**1.0**
Subtotal Miscellaneous	US$M					**5.3**
Total Equipment & Misc.						**321.5**
Spares Inventory @ 5%	US$M					**9.1**
Contingency @ 5%	US$M					**16.1**
Salavge @10%	US$M					**-18.3**
TOTAL MINE CAPITAL	**US$M**					**328.5**

21.1.5 Process Plant Estimate

Summary

The estimate has been prepared on a commodity basis (i.e., divided into earthworks, concrete, structural, etc.) and reported by area (i.e., crushing, milling, etc.).

The project is based on the purchase of new mechanical equipment, and general quantities have been assessed from first principles.

Bulk material take-offs to a prefeasibility level were developed from arrangement drawings.

Rates for civil and structural components were developed from historical unit price data obtained from similar projects in Chile and checked with input from our office in Santiago. Rates included a site productivity allowance for each type of work, plus the appropriate gang rate for the commodity and the actual cost of the permanent materials.

Labour rates are based on a 50-hour work week, which is typical for recent projects executed in this region of Chile. The average all-in labour rate for a 50-hour work week is $30/h and productivity factors were included for each commodity, ranging from 2.1 to 2.5. The productivity factors were checked against actual historical data from recent projects in Chile. Projects used to benchmark include:

- Collahuasi
- Los Pelambres
- Candelaria
- Escondida
- Los Brances

The estimate is based on the majority of the work being carried out under fixed price or re-measurable unit price contracts under a normal development schedule. No allowance is included for contracts on a cost plus or fast-track accelerated schedule basis.

Local freight associated with contractor-supplied material is included in the unit rates.

The estimate for ocean freight and Chilean inland freight were based on first principles, with appropriate percentages applied depending on where the particular equipment item originated. Overall, this figure worked out to be 5.3% of the equipment costs, which falls within the expected range.

The erection of tankage, structural, mechanical, piping, electrical, instrumentation, and civil works will be performed by experienced contractors, using a mix of national and non-national labour; the project is allowed by the Chilean government to utilize non-nationals for construction as necessary to achieve the required quality and meet the project schedule.

21.1.6 Tailings Storage Facility

The TSF consists of a two-year starter dam and an 18-year expansion dam located northeast of the process plant. Table 21.4 summarizes the approximate total contractor costs of each facility, which includes all earthworks and materials, including shipping and installation, for the embankment fills and tailings disposal pumping system.

Table 21.4: Base Case Capital Cost Estimate

Facility	Total Cost (US$M)
Starter TSF	16.8
Stage 2	9.7
Stage 3	13.4
Stage 4	18.1
Stage 5	22.7
Stage 6	28.6
Stage 7	13.6
Closure Costs	39.8

This above estimate does not include tailings pipe work or the water reclaim pumping system. This has been included in the process plant costs.

21.1.7 On-Site Infrastructure

Mobile Equipment

Table 21.5 describes the cost of the mobile fleet required to support plant operations.

Table 21.5: Mobile Equipment Costs

Description	Number	Total Cost (US$)
Run-of-mine rock breaker – Excavator with rock breaker hammer	1	500,000
Plant front-end loader – CAT 929DI	1	262,000
Skid steer loader – CAT 262C	3	141,000
Plant workshop forklift – S55FTS	2	50,000
Plant forklift truck – 18 t All terrain H450	1	240,000
Plant telehandler – 4 t capacity	1	99,700
Telescopic boom elevated work platform – 40 ft reach	1	87,900
Telescopic boom elevated work platform – 60 ft reach	1	130,600
Yard crane – 20t YB7725	1	312,000
Rough terrain crane – 80t RT890E	1	833,000
Tyre handler forklift – S55FTS	2	50,000
Elevated work platform – 7.9 m height	2	34,000
Light vehicle – 4x4 Dual Cab	4	91,500
Flatbed truck – 10 t carry capacity	1	67,000
Flatbed truck – 15 t carry capacity	1	90,700

Description	Number	Total Cost (US$)
Minibus – 21 seats	2	124,800
Fire response vehicle	1	93,000
Ambulance	1	93,000

Ancillary Buildings

The following administration buildings will be built:

- main administration building with medical centre and training room
- security office
- security gatehouse
- metallurgical office
- metallurgical laboratory
- plant mess and training room.

In addition, the following process plant buildings will be built. The capital cost for these buildings is included in the process plant cost estimate.

- reagent storage
- filtration building (port).

The ancillary buildings were costed on a lump sum basis by a Chilean-based contractor. In addition to the building structures, the cost includes the supply of the buildings electrics, fittings, and furnishing, but excludes earthworks. The cost to supply power and water services to the buildings form part of the process plant cost.

Plant Workshop and Warehouse

The plant workshop and warehouse is a fully-clad steel portal frame building. Offices are attached to the warehouse for procurement, maintenance, and planning personnel.

The workshop and warehouse buildings were costed on a lump sum basis by a Chilean-based contractor.

Mine Services Facilities

The mine service facilities include:

- in-pit dispatch control room
- heavy vehicle workshop with overhead gantry cranes
- light vehicle workshop
- mine services area bond store
- administration building
- restaurant

- gatehouse
- mine changehouse.

The buildings are fully-clad steel portal frame building, costed on a lump sum basis by a Chilean-based contractor.

Highway Diversion Cost

The access to the mine site is 6 km to the south of Diego de Almagro on Highway C-17. This section is paved and in good condition. Due to the location of the Iris Norte pit and process facility, approximately 2 km of the existing road will require diversion and an overpass. The overpass will allow vehicle access to the TSF without crossing Highway C-17.

Other improvements include a turn-off and signage at the plant road's junction with Highway C-17, and 1 km of new paved road 7 m wide.

21.1.8 Off-Site Infrastructure

Site Power Supply

The project requires electrical transmission, sub-transmission, and distribution infrastructure as described below:

High-voltage transmission will be achieved using a 220 kV double-circuit overhead line from Diego de Almagro. The line will be approximately 7 km long to the project main substation.

Medium-voltage power will be transmitted to the different centres of consumption via a 13.8 kV line.

Low-voltage power distribution will be at two voltage levels: 600 V for industrial systems, and 400 V for offices and ancillary systems.

The minimum equipment required is:

- 220 kV exit sections at the Diego de Almagro substation
- 220/13.8 kV transformer yard at the project main substation.

Seawater and Concentrate Pipeline

Bulk material and installation costs for both the seawater and concentrate pipelines from the mine site to port facility were estimated and are shown in Table 21.6 and Table 21.7. A summary of the capital costs follows.

Table 21.6: Seawater Pipeline Costs

DESCRIPTION	Total Cost (US$M)
Seawater Inlet Chambers + Intake Pump house	5.60
Pump Station #1	
Pumps (10 x 12), gear boxes, motors (3+1req)	6.48
Charge pumps	0.42
Valves and fittings	2.24
Pump Stations misc.	0.50
Pump Station #2	
Pumps (10 x 12), gear boxes, motors (3+1 req.)	6.48
Charge pumps	0.42
Valves and fittings	1.83
Pump Stations misc.	0.50
Pressure monitoring stations (3 remote)	0.48
Terminal Station	0.52
Pipe Material (24”, 312 to 562 ASTM A53 Gr. B)	16.92
Pipeline Installation (24”, 77.9 km)	30.49
Fiber optic cable/conduit (Materials)	0.39
Telecom, SCADA & leak detection	0.80
Pump Station Buildings	0.55
Pump Station Water Dams (2 x dams)	0.20
Power – HV line + Transformer+ 2 x 1.2 MW diesel Generators	0.89
Total Direct Costs	**75.71**

Table 21.7: Concentrate Pipeline Costs

DESCRIPTION	Total (US$M)
Pump Station #1	
Tanks (17.5 m x 17.5 m, 2 req) w/agitator	2.80
Pumps GEHO TZPM 2000	10.18
Charge Pumps	0.30
Valves and fittings	2.24
Pump Stations misc.	0.50
Head Station Building	0.55
Drainage at low point @ 34+400 (valves & chokes)	0.20
Pressure Monitoring stations (3 remote)	0.48
Valve Station	0.46
Choking Facilities	0.25

DESCRIPTION	Total (US$M)
Terminal Station	0.52
Pipe Material (12' 0,25 API SL X65)	5.07
Pipeline Installation (12', 77.9km)	24.93
Fiber optic cable/conduct (materials)	Incl. in Seawater Pipeline Cost
Telecom, SCADA & leak detection	0.80
Total Direct Costs	**49.28**

Port Facility

The port facility, including concentrate dewatering, stockpiling and ship load out were estimated using the principals identified in Section 21.1.1.

21.1.9 Indirect Costs and Owners Costs

Indirect costs and Owner's costs total an estimated $274.7 M, equal to 33.7% of the total direct cost.

The various cost centres that comprise the indirect costs are described in the following sections.

EPCM

For the purpose of the prefeasibility estimate, 16% of the direct costs was selected to cover the cost of EPCM services, which includes detailed engineering, procurement, project management and home office services as well as construction management. This was calculated on direct costs that excluded the mine equipment and mine development.

For the purpose of the prefeasibility study, 1.3% of the direct costs was selected to cover the cost of temporary construction facilities and services. It includes the project management team's site installation requirements for project execution, such as:

- project management team office complex supply and setup
- toilets facilities for construction
- vehicles
- general site clean-up
- refuse disposal
- site security guards
- computer, phone and communication system and office supply
- dismantling of facilities at contract's end
- commissioning support.

The estimate for first fills and spares (excluding mining equipment) is based on 5% of total mechanical equipment cost. This figure is based on actual factored historical data in the region.

The estimate for ocean freight and Chilean inland freight were based on first principles, whit appropriate percentages applied depending on where the particular equipment item originated. Overall, this figure worked out to be 5.3% of the equipment costs, which falls within the range expected.

Construction Camp

The construction camp costs include costs for the construction and operation of the camp. The camp is based on 100 beds for management and supervision, 400 beds for staff personnel, and 1,500 beds for craft personnel, bringing the camp total to 2,000 beds. The average cost per bed is $8,750. The construction of the camp is inclusive of all facilities.

The camp operating costs (all-inclusive of meals, maintenance, etc.) is based on $25 per person camp day, for a total of 700,000 camp days.

Both these rates are based on actual historical data from other camps in Chile.

Following demobilization of the construction crew, 500 beds will remain to make up the permanent camp to cover accommodation for operations personnel.

Owner's Cost

The Owner's team costs cover the costs incurred to build the operations team. The organisation structure will include the following departments:

- senior management
- mining
- process
- finance and administration
- legal and community affairs
- safety, heath, environment and community affairs
- occupational health and safety.

The selection, training and management of personnel will be undertaken in accordance with labour and recruitment and human resources management policies consistent with other major operators within Chile. Successful recruitment and training represent key issues in the successful development and operation of the project.

One percent of direct cost was calculated to cover insurances, including mining equipment and mine development.

A pre-production cost is based on the estimate operating cost for the process facilities at the mine and port for the first two months of operation.

The pre-development cost is based on estimated costs to complete the following:

- feasibility study
- high voltage power studies
- in-fill and geotechnical drilling
- hydrogeology investigations
- SEIA preparation
- land access/acquisition.

21.1.10 Contingency

The contingency reflects the potential growth in capital costs within the same scope of work. The contingency includes variations in quantities, differences between estimated and actual equipment and material prices, labour costs and site-specific conditions. It also accounts for variation resulting from uncertainties that are clarified during detail engineering, when designs and specifications of the basic engineering scope are finalized.

Contingency is an amount of money allowed in an estimate for cost which, based on past experience, are likely to be encountered, but are difficult or impossible to identify at the time the estimate is prepared. It is an amount expected to be expended during the course of the project. Contingency does not include scope changes, force majeure, labour disruptions or lack of labour availability.

Major cost categories (permanent equipment, material purchase, installation, subcontracts, pipelines, indirect costs and Owner's costs) were identified and analyzed. To each of these categories, a percentage of contingency was allocated based on the accuracy of the data, and an overall contingency amount was derived in this fashion. Using this method, an overall contingency of $148 M was obtained, representing 13.6% of the total capital cost, or 14.6% less mining equipment and development. The typical range for contingency on prefeasibility studies is 14% to 25% of the total project cost. The percentage of 14.6% falls within the lower range, but seems reasonable given the amount of detail that was incorporated into this study.

21.1.11 Duties and Taxes

Local taxes on contractor-supplied materials and installation labour are included in the estimate.

Impuesto al Valor Agregado (IVA) on process equipment, contractor-supplied material, or contractors profit are not included in the estimate. IVA has not been applied to spare or ocean freight.

21.1.12 Escalation

No escalation costs have been included.

21.1.13 Project Deferred and Sustaining Capital

Ongoing capital requirement for the mine production period totals $495.1 M over the mine life. This cost covers the phased construction of the tailings management facility. The cost also covers incorporates mine and process plant equipment to sustain the ongoing operation of the project

21.2 Operating Cost Estimates

21.2.1 Summary

This section details the estimated operating costs for the mine, process plant, port facility, general and administration (G&A), and pipeline departments for the Santo Domingo project. Costs are presented in Q2 2011 US dollars ($), unless stated otherwise. The estimate is considered prefeasibility study level with an accuracy of ±25%.

Project operating costs were determined by estimating for the following major cost centres:

- mining
- processing plant
- port facility
- general and administration
- seawater and concentrate pipelines.

The operating costs for each cost centre are summarised in Table 21.8.

Table 21.8: Overall Operating Cost Summary

Cost Centre	$M/a	$/t
Mining	107	4.62
Process plant	101	4.37
Concentrate pipeline – Port	2	0.09
Seawater pipeline – Port	10	0.43
G&A	13	0.55
Port Facility	11	0.462
Total	**244**	**10.52**

Estimates of first fill and pre-production costs, which are capitalized, are summarized in the capital cost estimate.

21.2.2 Mining Operating Cost Estimate

A summary of the Santo Domingo LOM operating cost estimates is shown in Table 21.9.

The open pit mining activities for the Santo Domingo project will be undertaken by an owner-operated mine as the basis for this prefeasibility study. The operating costs for the owner-operated scenario are presented in Q2-2011 US$ and do not include allowances for escalation or exchange rate fluctuations. Fleet replacement is addressed in sustaining capital.

Table 21.9: Summary of LOM Unit Operating Cost Estimates

Area	Unit	Cost Estimate
Open Pit Mining	US$/t mined	1.17
	US$/t milled	4.62

The mining unit rate was calculated based on equipment required for the mining configuration of the operation as described in the report, as well as a comparison to similar sized open pit operations. The open pit mining costs encompass pit and dump operations, road maintenance, mine supervision, and technical services.

The average open pit operating costs for the LOM plan are presented in Table 21.10 and Table 21.11, both by mining function and category. These costs are based on the LOM schedule presented in Section 12 and account for the material tonnages mined and their associated costs.

Table 21.10: Average LOM Open Pit Operating Cost Estimate – by Function

Cost Category	Estimated OPEX (US$/t mined)
Drilling	0.09
Blasting	0.13
Loading	0.15
Hauling	0.54
Roads/Dumps/Support Equipment	0.12
General Mine/Maintenance	0.05
Supervision & Technical	0.09
Total	**1.17**

Table 21.11: Average LOM Open Pit Operating Cost Estimate – by Category

Cost Category	Estimated OPEX (US$/t mined)
Operating Labour	0.08
Maintenance Labour	0.07
Supervision and Technician	0.08
Non-energy Consumables	0.49
Fuel	0.43
Leases, outside Services, Misc.	0.01
Total	**1.17**

Open pit mining costs are a summation of operating and maintenance labour, supervisory labour, parts and consumables, fuel, and miscellaneous operating supplies.

The open pit labour requirements and rates used for determining the overall mining cost is based on local labour rates and experience for similar operations of this size, and are divided into salaried and hourly personnel. The mine operations have, on average, a total of 205 personnel, mine maintenance 69 and supervision/technical has a total of 76 personnel.

Local labour rates are based on experience for projects in Chile for the various skill levels. Quotes from explosives suppliers and equipment suppliers were also taken into considerations. An average burden rate of 25% has been applied to the salaried and hourly labour to account for all social insurance, medical and insurance costs, pensions and vacation costs.

Parts, non-energy consumables, fuel, and miscellaneous operating costs were based on the mining fleet requirements described in the report. A diesel fuel cost of US$0.90/litre delivered to site was used as a basis in the operating cost estimate.

21.2.3 Process Plant, Seawater and Concentrate Transportation, and Port Facility

Processing plant, seawater and concentrate transport, and port facility operating costs are based on the flowsheets described in Section 12. The battery limits for the determination of process operating costs begins with the crushing facilities and end with tailings discharge into the TSF, and include plant services and seawater and concentrate transport costs. The operating costs for the port facility begin with the filter plant, receiving concentrate from the plant and end with concentrate load-out. Process plant, port facility, G&A, and piping costs have been developed separately.

The costs per tonne of ore milled ($/t) provided in this report are the average costs over the life of the mine.

Operating costs, including the process plant, port facility, pipelines, and G&A costs, are summarized in Table 21.12.

Table 21.12: Overall Operating Cost Summary

Operating Cost Summary	$M/a	$/t
Labour (including process plant, G&A, and port facility)	14	0.60
Reagents and Consumables	30	1.28
Power	56	2.41
Maintenance Consumables and Spares	13	0.54
Miscellaneous	4	0.19
General and Administration Miscellaneous	8	0.35
Seawater and Concentrate Piping	12	0.52
Total	**137**	**5.90**

Basis of Estimate

The plant, port, and G&A operating costs were determined from first principles using input from a variety of sources, including:

- process design criteria
- reagent and equipment supplier quotations
- logistics and transport costs from supplier quotations
- staffing levels for processing plant, G&A, and port facility estimated by Ausenco, and for mining by John Nilsson/SRK
- personnel salaries and overheads based on information from similar Chilean projects
- client recommendations
- seawater and concentrate pipeline operating costs were developed by Ausenco PSI;
- previous study assessments.

The following exchange rates were used in developing the operating cost estimate.

- 1 US$ = 466.35 Chilean Pesos (CLP)
- 1 US$ = 0.96 Canadian Dollars (CAD)

Inclusions

The process plant operating cost estimate includes all direct costs associated with the production of copper/iron concentrates.

Included in the Ausenco operating cost estimate are:

- labour for supervision, management, and reporting of on-site organizational and technical activities directly associated with the processing plant
- labour for operating and maintaining plant mobile equipment and light vehicles, process plant, and supporting infrastructure
- labour for port facility management, operations, and maintenance
- camp and transport costs
- costs associated with direct operation of the processing plant and port facility, including all reagents, consumables, and maintenance materials;
- fuels, lubricants, tires, and maintenance materials used in operating and maintaining the mobile equipment and light vehicles
- operation of the TSF, including tailings discharge and management and return water, excluding construction and ongoing dam raises;
- cost of power supplied to the process plant from the power grid
- operation of the water supply facilities using reverse osmosis to treat seawater
- labour and operational costs for the laboratory.

Exclusions

The plant, port facility, and G&A operating costs are exclusive of the following:

- corporate overheads
- escalation or exchange rate fluctuations
- mine operating costs other than grade control assays
- exploration labour and operating costs
- environmental permits
- contingency
- import duty and taxes
- sustaining capital
- interest and financing charges
- mine or plant closure/rehabilitation activities.

Some of the items listed above are included in the cash flow model as discrete line items.

21.2.4 Process Plant Operating Cost Summary

The plant is designed for an ore throughput of 21.9 Mt/a at an availability of 93.0%. Processing costs include labour, reagents and consumables, power, maintenance spares and consumables, mobile equipment, and ongoing metallurgical testing (Table 21.13). The estimated overall operating cost for the processing plant is $4.91/t of ore milled.

Table 21.13: Process Plant Cost Summary

Item	$M/a	$/t
Labour	7.4	0.32
Reagents and Consumables	28.8	1.24
Maintenance Consumables and Spares	9.5	0.41
Power	53.6	2.31
Miscellaneous	2.1	0.09
Total	**101.4**	**4.37**

21.2.5 Pipeline Operating Costs

Ausenco PSI provided the concentrate and seawater pipeline costs, inclusive of 15% contingency.

Table 21.14: Pipeline Costs

Pipeline	$M/a	$/t	$/t Concentrate
Concentrate Pipeline – Port	2.2	0.09	0.51
Seawater Pipeline – Port	9.9	0.43	2.31
Total	**12.1**	**0.52**	**2.82**

These costs include labour, power, maintenance, contract services, administration and miscellaneous costs associated with concentrate and seawater piping.

21.2.6 Port Facility Operating Costs

Concentrate slurry is piped to the filter plant located at the port facility. The filter plant is designed for an availability of 90%. Costs for the port facility are summarized in Table 21.15. The costs are presented in both $/t of ore processed in the plant, and $/t of concentrate. The estimated total port operating cost is $10.3 M/a. These costs do not include seawater and concentrate pipeline operating costs.

Table 21.15: Port Facility Operating Costs Summary

Port Facility Summary	$M/a	$/t	$/t Concentrate
Labour	2.0	0.08	0.46
Reagents and consumables	1.0	0.04	0.23
Power	2.4	0.10	0.55
Maintenance consumables and spares	3.0	0.13	0.71
Miscellaneous costs	2.4	0.10	0.56
Total	**10.7**	**0.46**	**2.50**

Off-site infrastructure maintenance spares and consumables, including site power supply, off-site office, camp, housing, and warehouse facilities, have been included in the port operating cost estimate. G&A costs for the port facility have been captured in the G&A operating cost summary, in Section 21.2.7.

21.2.7 G&A Operating Costs

The G&A costs include camp and transport costs, G&A personnel, and miscellaneous project costs. The estimated G&A operating cost is $12.7 M/a and is estimated to remain constant over the life of the operation.

Allowances for miscellaneous G&A costs were developed by Ausenco by benchmarking similar projects.

21.2.8 Contingency

Contingency was not included in the operating cost estimate.

22 ECONOMIC ANALYSIS

22.1 Assumptions

Economic analysis spreadsheets are included as Appendix 12 for each of the individual cases considered in this report. Variability analyses were conducted using different metal prices and varying capital and operating costs to determine the effect of these variables on the project economics. These analyses were conducted on the basis of the inputs listed in Table 22.1.

Table 22.1: Base Inputs for Economic Analysis

Parameter	Unit	Value
Plant throughput, t/d	60 - 70,000	
Life of project, years	18	
Copper Grade – % Average	0.31	
Copper Concentrate grade, %	29	
Concentrate gold grade, g/t	7.0	
Moisture of copper concentrate, %	9.0	
Mass Recovery to Magnetite Concentrate – % Average	17.3	
Magnetite Fe grade, %	65.0	
Gold Grade – g/t Average	0.04	
Moisture of magnetite concentrate, %	9.0	
Copper transport cost, US$/t conc.	- Pipeline	2.43
	- Sea	55.0
Magnetite transport cost, US$/t conc.	- Pipeline	2.43
	- Sea	3.0
Base Copper Smelter costs, US$/t conc.		60
Base Copper Refining costs, US$/lb Cu		0.06
Minimum Copper Deduction (units)		1
Gold refining costs, US$/oz Au		6.50
Payable, %	Copper	100.0
	Gold	97.0
Royalties – Prior tenement owners (% NSR)	2	
Royalties – Government (% Operational income)	5	
Interest, %	Not Applicable	
Company Taxation, %	17	
Depreciation Rate, years	5, 7, 9 and 10	

Parameter	Unit	Value
VAT, % of goods and services	19	
Customs Duty, % imported goods	6	
Amortization, %	Not Applicable	
NPV discount rate, %	8	
Base Copper price, US$/lb	2.50	
Base Magnetite price, US$/dtmu	1.00	
Base gold price, US$/oz	1000	
Base capital cost, US$M	1,242	
Total Operating Cost, US$M	**4,403**	
Sustaining capital cost, US$M	495	

22.2 Economic Analysis (Base Case)

The base case economic analysis indicates that given the current estimated mining and plant operating costs, as well as capital cost estimates, the project economics are as shown in Table 22.2. All values are calculated on both an EBITD&A (earnings before interest tax depreciation and amortization) and after-tax basis.

A third part review of the financial model was carried out by Deloitte.

Table 22.2: Base Case Economic Analysis

Economic Parameters	EBITD&A	After Tax
NPV at 8% Discount Rate (US$M @ 8%)	1,620	1,092
IRR%	29.4	22.0
Simple Payback Period (years)	2.5	3.0
Discounted Payback Period (years @ 8%)	3.0	3.8
Total Cash Cost (US$ per lb of Cu)[27]	0.11	

Table 22.3 and Figure 22-1 and Figure 22-2 highlight the pre-tax and after-tax cumulative cash flow.

[27] Total Cash Production Costs (per lb of payable Cu) are inclusive of by-product credits.

Table 22.3: Summary of Project Cash Flow, before and after Tax

ITEM	UNIT	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Years 11 to 15	Years 16 to 20	TOTAL
PRODUCTION																	
ANNUAL PRODUCTION SCHEDULE	'000 tonnes				15,330	25,550	25,550	25,550	25,550	25,462	24,424	23,910	24,355	23,977	113,637	64,661	417,956
METAL PRODUCED																	
- Copper in Concentrate	tonnes				88,411	147,172	133,714	114,619	93,767	76,453	59,668	56,458	48,865	45,727	230,525	83,294	1,178,672
- Magnetite Concentrate	'000 tonnes				1,584	2,849	3,165	4,412	4,228	4,210	3,255	3,410	4,383	4,431	23,093	14,092	73,112
- Gold Production	oz				28,095	44,375	38,409	29,791	22,890	17,204	11,469	10,230	8,563	7,683	41,343	7,534	267,585
REVENUE																	
MAGNETITE CONCENTRATE BASE PRICE	$US/dmtu Fe				1.00	1.00	1.00	1.00	1.00		1.00	1.00	1.00	1.00	1.00	1.00	1.00
COPPER BASE PRICE	$US/lb				2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50
GOLD BASE PRICE	$US/oz				1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000
COPPER CONCENTRATE																	
Copper Gross Metal Value	$USM				470	783	711	610	499	407	317	300	260	243	1,226	443	6,271
Gold Gross Metal Value	$USM				27	43	37	29	22	17	11	10	8	7	40	7	260
MAGNETITE CONCENTRATE	$USM				103	185	206	287	275	274	212	222	285	288	1,501	916	4,752
TOTAL GROSS METAL VALUE	$USM				601	1,011	954	925	796	697	540	532	553	539	2,768	1,366	11,282
OPERATING COSTS																	
AVERAGE UNIT OPERATING COSTS																	
MINING	$US/tonne ore				6.87	4.18	4.44	4.33	4.50	4.51	5.04	4.81	4.51	5.00	4.97	3.66	4.63
PLANT	$US/tonne ore				5.18	4.22	4.22	4.22	4.22	4.23	4.29	4.32	4.29	4.32	4.40	4.49	4.37
CONCENTRATE PIPELINE	$US/tonne ore				0.06	0.07	0.07	0.10	0.09	0.09	0.07	0.08	0.09	0.10	0.11	0.11	0.09
SEAWATER PIPELINE	$US/tonne ore				0.43	0.43	0.43	0.43	0.43	0.43	0.43	0.43	0.43	0.43	0.43	0.43	0.43
PORT	$US/tonne ore				0.31	0.33	0.36	0.47	0.45	0.44	0.35	0.38	0.47	0.48	0.53	0.56	0.46
GENERAL & ADMINISTRATION	$US/tonne ore				0.83	0.50	0.50	0.50	0.50	0.50	0.52	0.53	0.52	0.53	0.56	0.59	0.55
TOTAL SITE OPERATING COSTS	**$US/tonne ore**				**13.68**	**9.73**	**10.02**	**10.05**	**10.19**	**10.20**	**10.71**	**10.55**	**10.32**	**10.86**	**10.99**	**9.85**	**10.53**
REALISATION COSTS	**$US/tonne ore**				**4.20**	**4.23**	**3.94**	**3.68**	**3.12**	**2.68**	**2.18**	**2.16**	**2.11**	**2.06**	**2.23**	**1.81**	**2.61**
TOTAL UNIT OPERATING COSTS	**$US/tonne ore**				**17.88**	**13.96**	**13.96**	**13.73**	**13.31**	**12.89**	**12.88**	**12.72**	**12.43**	**12.92**	**12.97**	**14.78**	**13.14**
TOTAL OPERATING COSTS																	
TOTAL SITE OPERATING COSTS	$USM				209.8	248.6	256.0	256.8	260.4	259.8	261.5	252.3	251.4	260.4	1,249.2	636.6	4,402.8
TOTAL REALISATION COSTS	$USM				64.3	108.0	100.7	93.9	79.6	68.3	53.1	51.8	51.3	49.5	252.9	117.2	1,090.7
TOTAL OPERATING COSTS	**$USM**				**274.1**	**356.6**	**356.7**	**350.7**	**340.0**	**328.1**	**314.7**	**304.1**	**302.7**	**309.9**	**1,502.1**	**753.8**	**5,493.5**
CAPITAL COSTS																	
PLANT CAPITAL	$USM	28.6	285.6	257.0													
SEAWATER PIPELINE CAPITAL	$USM	0.0	34.5	80.5													
CONCENTRATE PIPELINE CAPITAL	$USM	0.0	22.4	52.4													
PORT CAPITAL	$USM	0.0	55.3	129.1													
MINING FLEET CAPITAL	$USM	0.0	103.3	85.8													
MINE PRE-DEVELOPMENT	$USM	0.0	0.0	82.4													
TAILINGS	$USM	0.0	0.0	25.4													
TOTAL INITIAL CAPITAL	**$USM**	28.6	501.1	712.6													
DEFERRED/SUSTAINING CAPITAL																	
PROCESS PLANT	$USM				4.78	4.78	4.78	4.78	4.78	4.78	4.78	4.78	4.78	4.78	23.92	14.35	86.1
SEAWATER PIPELINE	$USM				0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	0.76	3.79	2.27	13.6
CONCENTRATE PIPELINE	$USM				0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	0.74	3.70	2.22	13.3
PORT	$USM				2.43	2.43	2.43	2.43	2.43	2.43	2.43	2.43	2.43	2.43	12.15	7.29	43.7
MINING	$USM				0.00	12.11	0.00	7.71	0.00	11.56	15.71	53.74	36.60	20.15	0.00	0.00	157.6
TAILINGS STORAGE FACILITIES	$USM				9.69	13.37	0.00	0.00	0.00	0.00	22.65	0.00	0.00	0.00	28.63	53.34	167.7
TOTAL DEFERRED/SUSTAINING CAP.	**$USM**				**19.1**	**34.9**	**9.4**	**17.1**	**9.4**	**21.0**	**47.8**	**63.2**	**46.0**	**29.6**	**75.8**	**121.6**	**495**
TOTAL CAPITAL	**$USM**	**28.6**	**501.1**	**712.6**	**19.1**	**34.9**	**9.4**	**17.1**	**9.4**	**21.0**	**47.8**	**63.2**	**46.0**	**29.6**	**75.8**	**121.6**	**1,737**
TOTAL PROJECT CASHFLOWS																	
PROJECT PRETAX CASHFLOW	$USM	-29	-501	-713	307	620	588	558	446	348	178	165	204	199	1,190	491	4,051
CORPORATE TAXATION	$USM	0	0	0	35	86	77	73	54	42	20	19	22	31	179	87	726
PROJECT AFTER TAX CASHFLOW	$USM	-35	-611	-861	525	495	480	454	372	291	150	136	173	157	948	374	3,048
PROJECT AFTER TAX CASHFLOW @ 8% DR RATE	$USM	-35	-565	-738	416	364	326	286	217	157	75	63	74	62	298	91	1,093
PRODUCTION STATISTICS																	
NET REVENUE /TONNE ORE TREATED	$US/tonne ore				35.0	35.3	33.4	32.5	28.0	24.7	19.9	20.1	20.6	20.4	22.1	19.3	24.4
TOTAL CASH COST / lb OF PAYABLE COPPER[28]	$US/lb Cu				0.71	0.35	0.34	0.07	0.14	0.15	0.65	0.52	-0.01	0.05	-0.15	-1.10	0.11

[28] Inclusive of by-product credits.

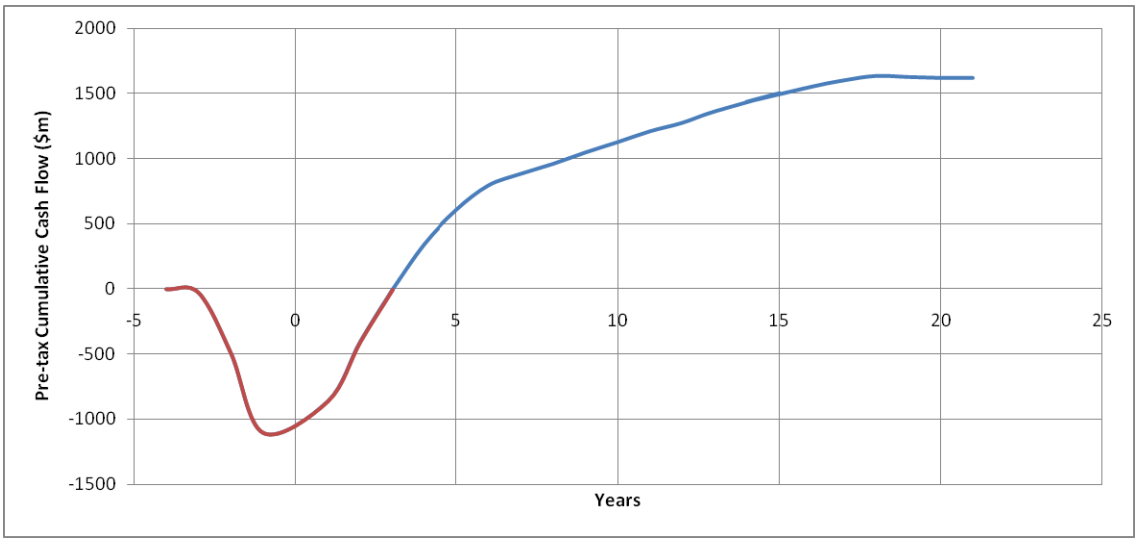


Figure 22-1: Pre-Tax Cumulative Cash Flow at 8% Discount Rate

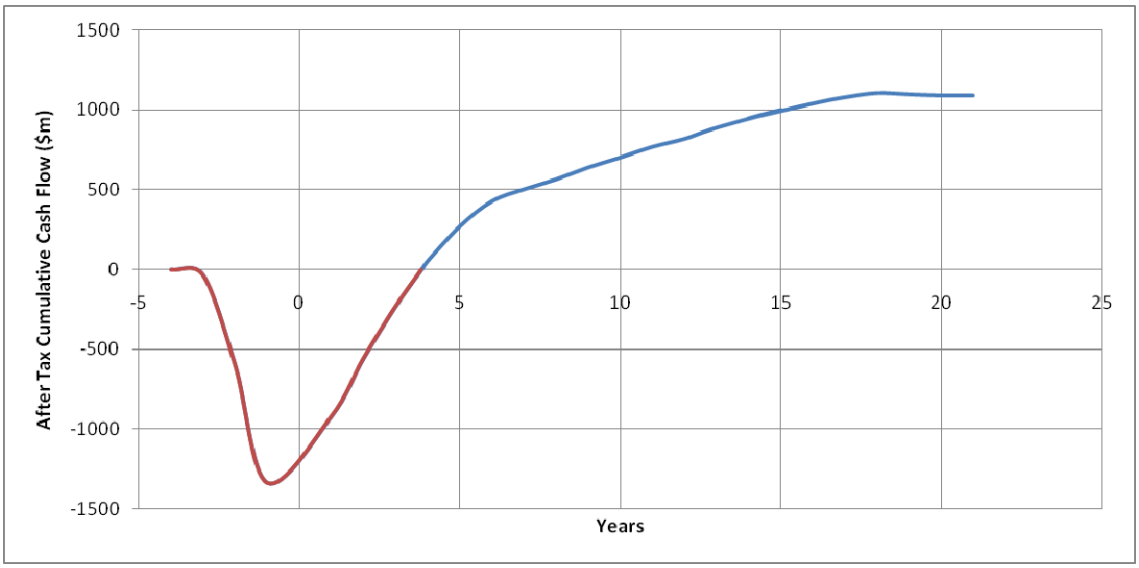


Figure 22-2: After Tax Cumulative Cash Flow at 8% Discount Rate

22.2.1 Tax Variables

The following tax allowances were applied:

- Financing by equity, no debt financing
- All net profits remain in Chile
- Corporate tax is a flat 17% on net profit
- Specific mining tax (IEAM) with variable tax rate on net taxable income (Article 64 TER)

- Five separate normal straight line depreciation models used covering 5, 7, 9, and 10 years.
- VAT at a flat 19% was applied to the total overall project cost in Years -3, -2 and -1. Costs were carried forward to Year 1 onwards and credited against VAT expense incurred
- All EPCM and construction contractors sourced within Chile
- WHT was not included
- A 6% customs duty on imported equipment and contractor materials and pre-production + sustaining capital during production.

22.3 Sensitivity Analysis (Metal Prices)

A basic sensitivity analysis was conducted on the economic effects of current (June 2011) metal prices and copper smelter treatment and refinery costs (TC/RCs). Table 22.4 shows a comparison of the various metal prices used in the scenarios analyzed.

Table 22.4: Metal Price Sensitivity

Metal	Units	Metal Prices	
		Base	Spot
Copper	US$/lb	2.50	4.00
Gold	US$/oz (troy)	1,000	1,400
Magnetite Concentrate	US$/dtmu	1.00	2.00
Smelter TCs	US$/t conc.	60	90
Smelter RCs	US$/lb	0.06	0.09

Table 22.5 summarizes the same economic criteria for the project as Table 23.2 using spot metal prices.

Table 22.5: Spot Price Economic Analysis

Economic parameters	EBITD&A	After Tax
NPV (US$M @ 8%)	5,473	3,985
IRR%	60.2	45.4
Simple Payback Period (years)	1.4	1.7
Discounted Payback Period (years @ 8%)	1.6	2.0
Total Cash Cost (US$ per lb of Cu)[29]	-1.75	

22.4 Sensitivity Analysis (Variability)

A further sensitivity analysis was conducted to ascertain the effect of variability of the following parameters:

[29] Total Cash Production Costs (per lb of payable Cu) are inclusive of by-product credits.

- copper price
- gold price
- magnetite concentrate price
- capital cost
- operating cost.

The variations in copper, gold, and magnetite concentrate prices specified are listed in Table 22.6.

Table 22.6: Metal Pricing for Sensitivity Analysis

Copper	Gold	Magnetite Concentrate
US$/lb	US$/oz	US$/dtmu
2.00	800	80
2.20	900	90
2.50 – Base case	1,000 – Base case	100 – Base case
2.75	1,100	110
3.00	1,200	120

The operating and capital costs were varied from the base costs by -20, -10, 10 and 20%. These variations were at Ausenco's discretion.

This analysis was conducted by varying one parameter at a time to determine a resulting IRR and NPV. The results of this analysis are shown in Figure 22-3 and Figure 22-4 for the base case metals prices.

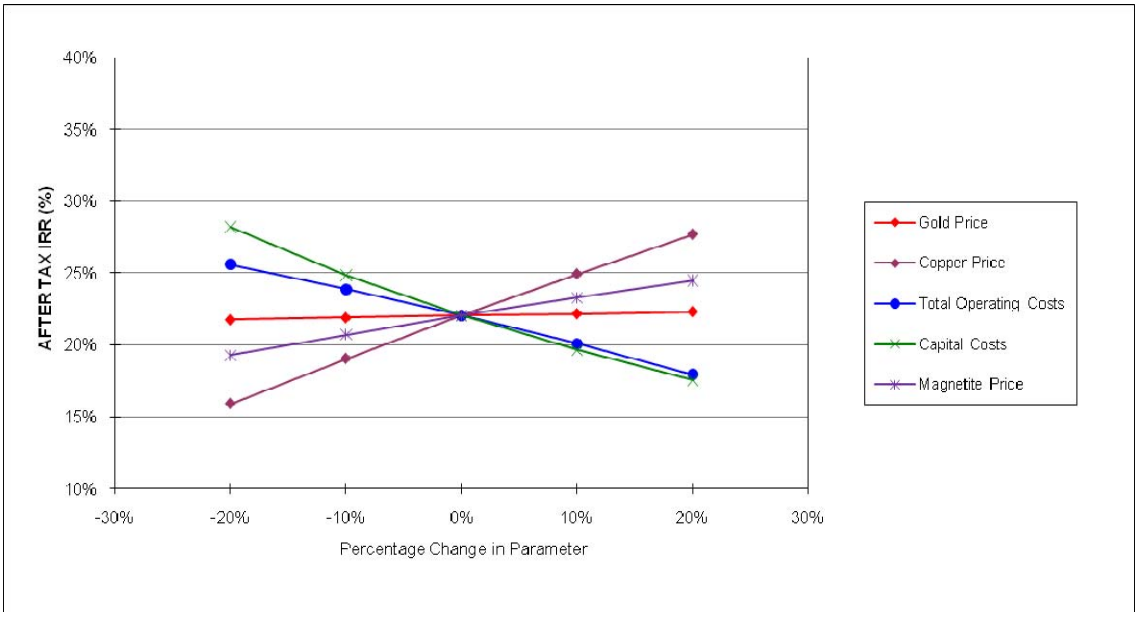


Figure 22-3: Throughput IRR Sensitivity

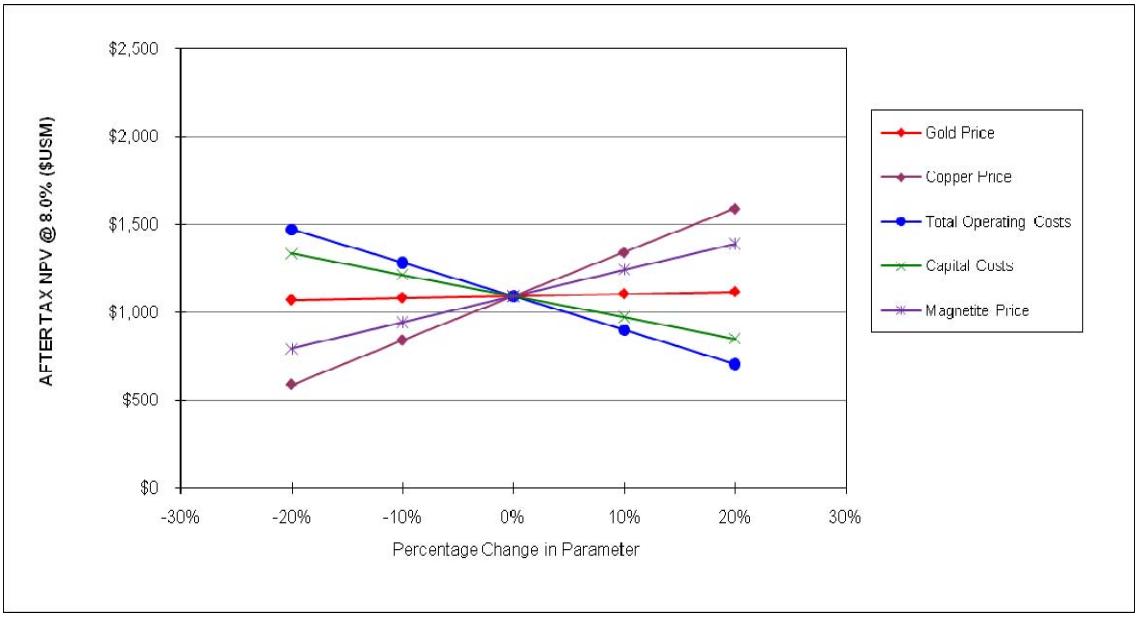


Figure 22-4: Throughput NPV Sensitivity

In Figure 22-3 and Figure 22-4, the point at which all lines meet is the base case (see assumptions above). The lines emanating out from this point show the influence of varying the different parameters from that base. It can be seen that varying the gold price causes minor variations in the IRR or NPV, as this line is relatively flat. The magnetite concentrate price has a moderate impact on the project IRR, whereas the copper price and capital and operating costs have the largest impacts. The NPV shows less sensitivity to the capital cost, but the copper price and operating cost still have large impacts.

23 ADJACENT PROPERTIES

There are no significant exploration projects on ground adjacent to FWM's holdings at the Santo Domingo Property.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Site Geotechnical

From December 2010 through May 2011, a site investigation program was completed. This program consisted of test pit excavations and geotechnically logged drill holes carried out in order to gather information in support of the PFS mine infrastructure designs. This was the first geotechnical soil characterization program at the approximate planned project facilities site[30].

Basic information was related to general soil geological-geotechnical characterization and local natural hazards. The geological-geotechnical study was developed for three specific areas:

1. plant site area
2. tailings deposition area
3. waste dump area.

A variable assemblage of soils that are frequently strongly cemented characterizes the near surface conditions at the plant site area. These soils will provide good load-bearing capacity, but there is a concern about reactions to acid/water environments. Degradation of soil quality could occur if subjected to acidic conditions. The subsurface conditions at the tailings deposition area are characterized by the presence of a range of low-plastic and non-plastic soils of moderate to high density. The north dump area was characterized by a superficial layer composed mainly of gravels and silty sands, with strongly cemented gravels at depth. At the south dump, the predominant overburden materials were observed as being poorly-graded gravels with horizons of silts, and/or well-graded sands with silts.

Figure 24-1 shows the location of the new test pits TP-01, TP-03, TP-06, and TP-07 and the bore holes in which there is information (-MP-01 to BH-MP-05 and BH-TSF-01), while Figure 24-2 shows the location of surveys in the area with limestone outcrops.

Figure 24-3 shows surveys performed at the dam area and the narrowest point of the valley, where andesitic rocks outcrop.

[30] AMEC (Apr 2011, rev. B). TECHNICAL REPORT E40010-GG-TR-0001. Geological & Geotechnical Field Investigations.

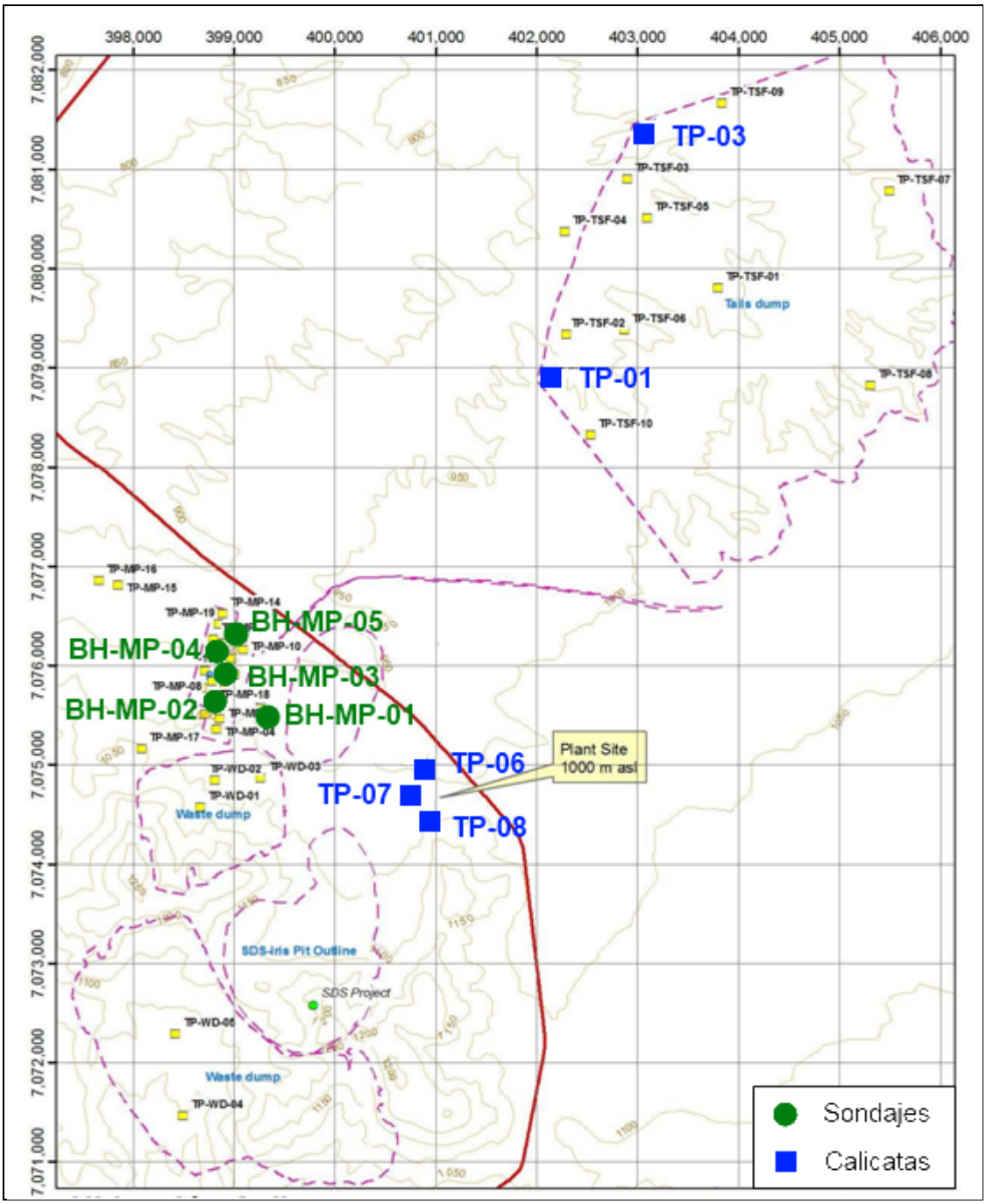


Figure 24-1: Survey and Facilities Location

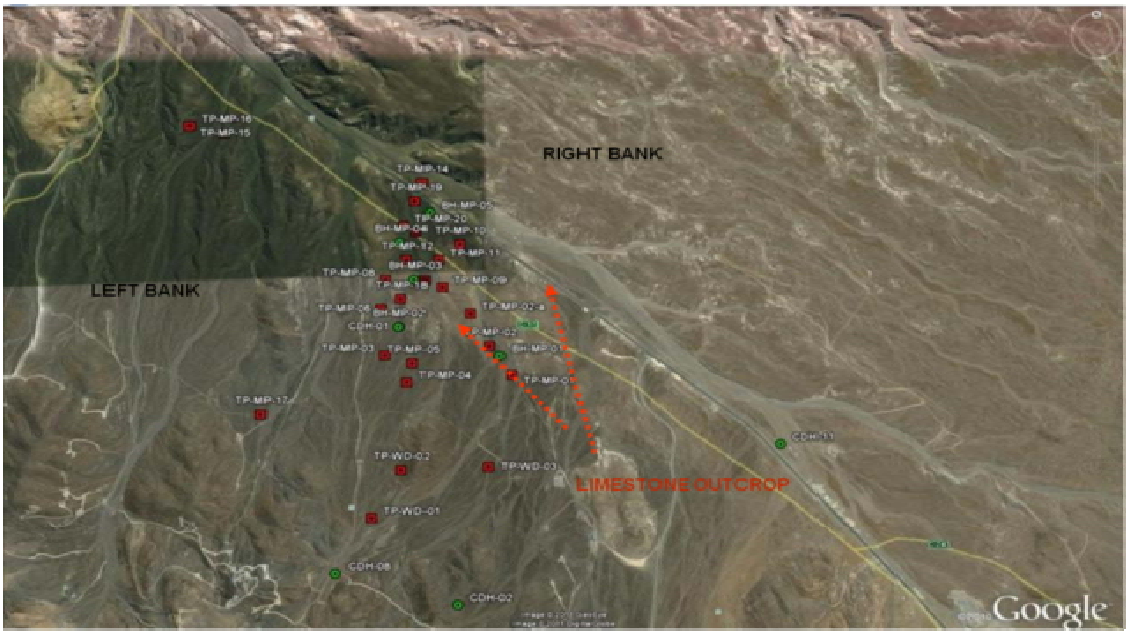


Figure 24-2: Test Pit Location of the Left Bank, Downstream of the Limestone Outcrop

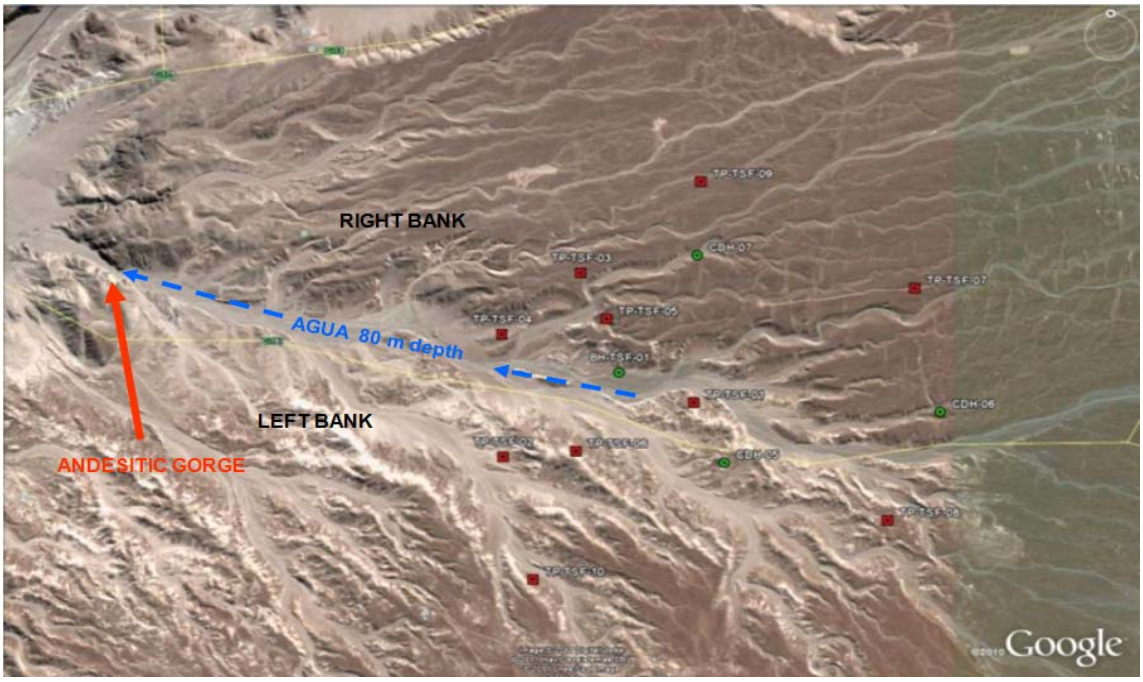


Figure 24-3: Dam Area Survey. Andesitic Gorge

24.1.1 Plant Site Area

The plant site area is characterized by soils with variable but frequently strong cementation. The sands and gravels present were strongly cemented with salts/caliche/calcrete, locally

termed “Tertel.” These cemented deposits provide good load-bearing capacity, but their strong reactivity to acidic water environments can degrade their integrity.

Direct shear testing on both sands and gravels resulted in less cohesive materials, with peak internal friction angles ranging from 42° to 52°.

California Bearing Ratio (CBR) tests performed on samples from this area indicated values higher than 94%, which qualifies these soils as excellent for use as a “base” for pavement roads.

24.1.2 Tailings Deposition Area

The subsoil presents alluvial/alluvial gravel (angular to sub rounded), with non-plastic fines with medium to dense compactness and may qualify as hard if their holes are partially or completely filled with gypsum. On the right bank, the presence of carbonates is very low.

The subsoil of the right bank in the upper horizons contains significant percentages of evaporites, mainly limestone and gypsum, which were discriminated using 10% HCL. Limestone in the form of powder and gypsum were observed as either particle or as agglomerate (temporary oversize up to 30 cm).

The subsoil has a different rheological sequence with a strong alluvial accent (more rounded particles) compared to the left border, which is more colluvial (more angular particles).

The limestone has given colluvial contributions of limestone/gypsum, which locally corresponds to the "fence", or inter-fingered alluvial gravels downstream in alternating periods. The subsoil at the right border is intermingled gravels mainly of gypsum (calcocrete) and the right border locally corresponds to gravel that increase its density as it deepens.

In the entire basin area there is a typical 15 cm silty sand overload with elongated clasts of volcanic origin.

In general, the soils are considered good foundation materials for the envisioned tailings deposition area. Overall, low plasticity indices (less than 15) were found in all soils at the tailings deposition area.

24.1.3 Waste Dump Area

A superficial layer composed mainly of gravels and silty sands, with depths varying from 0.8 to 1.2 m, characterizes the north dump area. Strongly cemented gravel was found at this depth. At the south dump, poorly-graded gravel horizons with silts and well-graded sands with silts, with depths ranging from 3.3 to 3.8 m, were observed. As at the plant site, all soils were found to be non-plastic.

Direct shear test results ranged in apparent cohesion from essentially zero to nearly 0.2 kgf/cm^2 with peak internal friction angles of approximately 50°.

24.1.4 Natural Hazards

The types of natural risks observable in the project area are mainly related to morphology, weather, and seismicity.

In general, the project area is considered not to have natural hazards are readily identified that should require the adoption of special protection measures. Good construction standards have been in use in Chile and infrastructure built using these protocols will withstand seismic activity very well. The area most exposed to alluvial risk or flooding by rising water is the planned tailings impoundment site area, but standard mitigation measures can readily address such concerns. Also, the northern portion of the projected Iris Norte pit is located on a floodplain, and as the project proceeds the location of some facilities may need to be adjusted accordingly to take this into account.

24.1.5 Borrow Materials

Potential sources of borrow materials for aggregates has been identified close to the project area. Possible availability and/or environmental restrictions on these areas were not evaluated, and no specific geotechnical investigation or testing has been carried out. Moreover, no chemical evaluation of their ability to serve as concrete aggregate (e.g., sulphate soundness) was completed as part of the PFS. The candidate borrow locations are presented in Figure 24-4. These three sites correspond to materials of alluvial-fluvial origin.

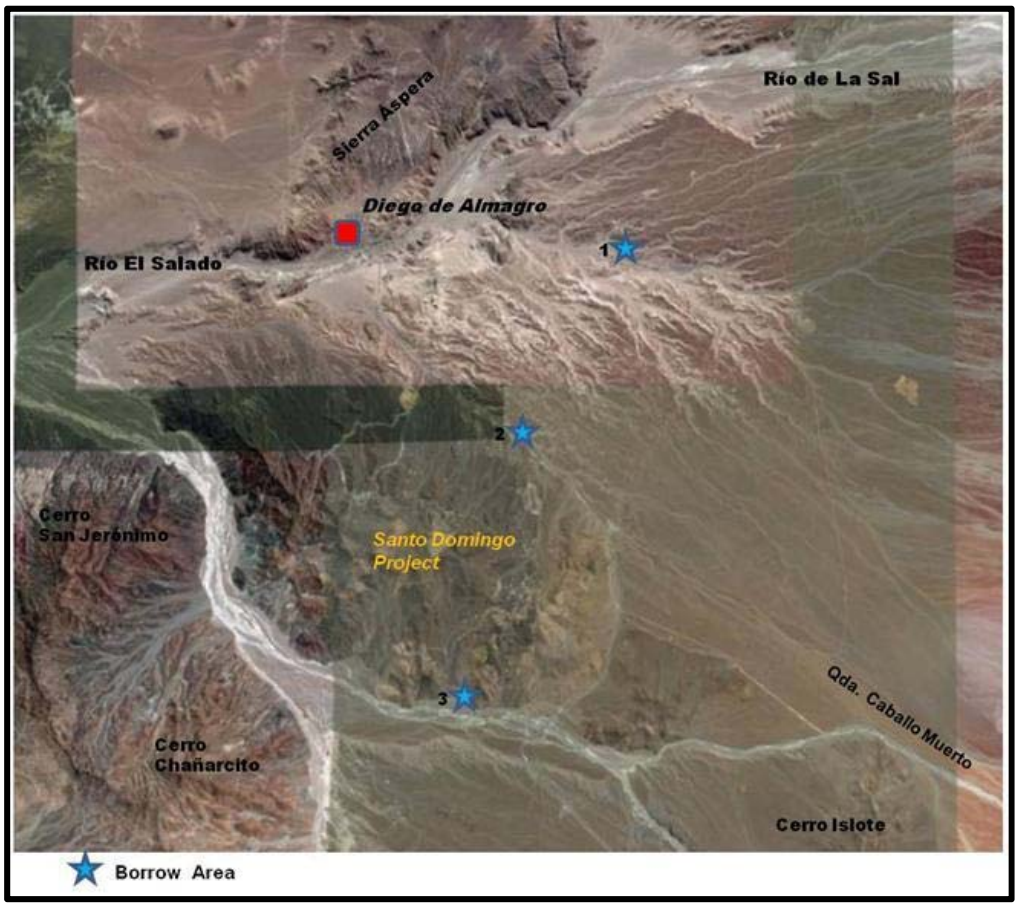


Figure 24-4: Potential Borrow Material Sources

Site 1, located in the plain in the northeast (tailings) sector of the area, consists of recent alluvium, deposited in drainage channels that scarred the ancient alluvial layer. Site 2 is located in the riverbed of the ravine that drains into the El Salado River. Site 3 consists of the lower part of an alluvial fan detached from one of the ravines that drain into the Chañaral Alto ravine. All three deposits present clear-grey colouring, characteristic of recent deposition, and are largely composed of gravel and lenses of unconsolidated coarse sands.

24.1.6 Discussion and Conclusions

The subsurface conditions at the key development sites of the plant site, tailings deposition area, and waste dump area, are all considered favourable from a geotechnical perspective for the envisioned project development. Cautions include the need to avoid having acid drainages near foundations present in the cemented soils and ensuring that concrete works are appropriately safe-guarded for sulphate soundness. Chemical analyses were performed due to the presence of layers containing soluble salts, chlorides, and sulphates. Some samples values obtained indicated potentially moderate/severe to very severe potential attack from sulphates. In relation with the chloride content and water soluble sulphates, most of samples tested exceeded the referential allowable limits.

24.2 Project Implementation Strategy

24.3 Implementation Strategy

How a project is implemented affects all aspects of the project, particularly:

- capital cost
- schedule
- risk management
- level of client input.

In respect to capital costs, this strategy is less important at a conceptual study level because the methodologies generally adopted to establish costs do not take this into account, and its impact does not affect the estimate accuracy. However, the impact of this strategy can be reflected broadly in scheduling.

Project implementation planning should start prior either to or early in the feasibility study stage so that the study can be managed to ensure that the deliverables reflect this strategy. This planning should be cognizant of the following:

- determining the level of engineering to be performed during the feasibility study – increasing the engineering level may reduce the implementation schedule, but increase the cost of the study
- a contracting strategy that minimizes the number of interfaces, particularly between site-based construction contractors, and minimizes client risk, but generally increases capital cost
- increasing the number of individual supply contracts or purchase orders may reduce project duration but increase EPCM costs.

Given the prevailing market conditions, the feasibility study should include a number of commercial and logistical evaluations to develop and optimize an appropriate implementation strategy. In particular, with fabrication workshops currently operating at or near capacity, the feasibility study should assess the cost, schedule, taxation and industrial relations implications of fabricating steel and plate work in Asia, or elsewhere overseas.

24.4 Preliminary Schedule

The preliminary project schedule is attached in Appendix 13. The current schedule shows:

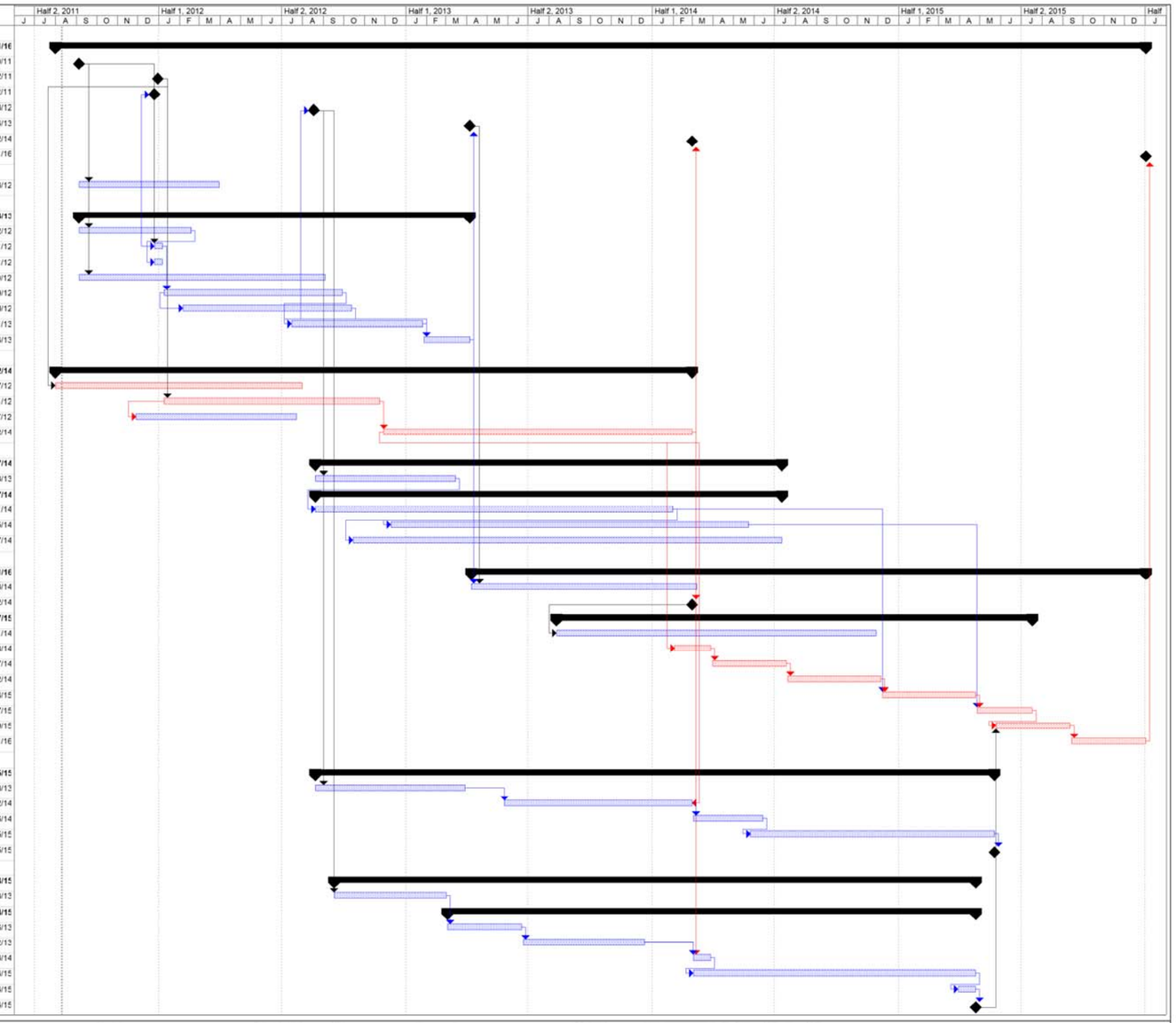
- Overall schedule duration from commencement of the Feasibility Study to the end of ore commissioning is 231 weeks, or 4 years and 5 months. Key milestone dates are described in Table 24.1.
- The duration of the schedule is driven by the completion of the Environmental Impact Study and permitting process. The EIS submission date is set at the beginning of December 2012.

- The Project Description (i.e., freezing plant location, pipeline routes and port location) will be fixed by the end of December 2011 for input into the EIS.
- Metallurgy testwork program will commence at the beginning of October 2011 with confirmation of the process flowsheet and design criteria occurring by the end of 2011. This period will also include geotechnical and infill drilling as well as engineering support ahead of the FS to assist in preparing the "Project Description" for the EIS.
- Basic engineering with a commitment to purchase major capital items like the mills, mill motors and pipelines will commence 13 months into the feasibility study.
- Tender and award of the EPCM contractor will commence immediately after completion of the FS.
- Site access is determined by EIS approval. The commissioning date is determined by site access and site works for both the SAG mill and the seawater pipeline.
- The mill erection sequence will require further review during the FS. The current schedule has the SAG and ball mills being erected at the same time.

Table 24.1: Santo Domingo Project Schedule – Key Milestone Dates

Criteria	Date	Weeks
Project Commencement - Baseline Environmental Studies	01/08/11	
Interim Engineering for EIS & Testwork Program	05/09/11	5
Port Location Frozen for EIS Commencement	30/12/11	22
Commence Engineering Feasibility Study	26/12/11	21
Basic Engineering Phase Board Approval (Long-lead items Purchase)	17/08/12	55
Complete Feasibility Study	05/04/13	88
EPCM Phase Board Approval	05/04/13	88
Site Access & Commence Construction	28/02/14	135
Complete Commissioning	01/01/16	231

ID	Task Name	Duration	Start	Finish
1				
2	**Santo Domingo Project**	**231 wks**	**Mon 01/08/11**	**Fri 01/01/16**
3	Interim Engineering for EIS & Testwork Program	0 wks	Mon 05/09/11	Mon 05/09/11
4	Port Location Frozen for EIS Commencement	0 wks	Fri 30/12/11	Fri 30/12/11
5	Commence Engineering Feasibility Study	0 wks	Mon 26/12/11	Mon 26/12/11
6	Long Lead Items Purchase (Board Approval)	0 wks	Fri 17/08/12	Fri 17/08/12
7	EPCM Phase Board Approval	0 wks	Fri 05/04/13	Fri 05/04/13
8	Site Access & Commence Construction	0 wks	Fri 28/02/14	Fri 28/02/14
9	Complete Commissioning	0 wks	Fri 01/01/16	Fri 01/01/16
10				
11	Interim Engineering for EIS & Testwork Program	30 wks	Mon 05/09/11	Fri 30/03/12
12				
13	**Feasibility Study**	**83 wks**	**Mon 05/09/11**	**Fri 05/04/13**
14	Interim Met testwork, programme design, sample ship & conduct	24 wks	Mon 05/09/11	Fri 17/02/12
15	Confirm Base Process Flowsheet and Design Criteria	2 wks	Mon 26/12/11	Fri 06/01/12
16	FS WBS and Project Controls	2 wks	Mon 26/12/11	Fri 06/01/12
17	Mine Design	52.2 wks	Mon 05/09/11	Mon 03/09/12
18	Process Engineering	38 wks	Mon 09/01/12	Fri 28/09/12
19	Design Engineering and Layouts	36 wks	Mon 06/02/12	Fri 12/10/12
20	Estimating	28 wks	Mon 16/07/12	Fri 25/01/13
21	Reporting	10 wks	Mon 28/01/13	Fri 05/04/13
22				
23	**Environmental Impact Study & Permitting**	**135 wks**	**Mon 01/08/11**	**Fri 28/02/14**
24	BASELINE STUDIES (currently ongoing)	52.4 wks	Mon 01/08/11	Tue 31/07/12
25	ENVIRONMENTAL IMPACT STUDY PREPARATION	46 wks	Mon 09/01/12	Fri 23/11/12
26	COMMUNITY AMD STAKEHOLDERS ENGAGEMENT	34 wks	Mon 28/11/11	Mon 23/07/12
27	EIS FOLLOW UP	327 days	Thu 29/11/12	Fri 28/02/14
28				
29	**Plant Project Basic Engineering**	**99 wks**	**Mon 20/08/12**	**Fri 11/07/14**
30	Engineering and Procurement	30 wks	Mon 20/08/12	Fri 15/03/13
31	**Order & Supply Long Lead Items**	**99 wks**	**Mon 20/08/12**	**Fri 11/07/14**
32	SAG Mill	76 wks	Mon 20/08/12	Fri 31/01/14
33	SAG Mill Motor	76 wks	Mon 10/12/12	Fri 23/05/14
34	Primary Ball & Regrind Mills	91 wks	Mon 15/10/12	Fri 11/07/14
35				
36	**Plant Project EPCM**	**143 wks**	**Mon 08/04/13**	**Fri 01/01/16**
37	EPCM Award & Detailed Engineering	48 wks	Mon 08/04/13	Fri 07/03/14
38	Mobilise to Site	0 wks	Fri 28/02/14	Fri 28/02/14
39	**EPCM Packages**	**101 wks**	**Mon 12/08/13**	**Fri 17/07/15**
40	EPCM Site Works Packages	68 wks	Mon 12/08/13	Fri 28/11/14
41	Bulk earthworks mobilise and batch plant operational	8 wks	Mon 03/02/14	Fri 28/03/14
42	Mills Bulk Earthworks	16 wks	Mon 31/03/14	Fri 18/07/14
43	Mills Civils Mobilise, Batch Plant & Pour	20 wks	Mon 21/07/14	Fri 05/12/14
44	Mills Mechanical Installation	20 wks	Mon 08/12/14	Fri 24/04/15
45	Mills Electrical Installation	12 wks	Mon 27/04/15	Fri 17/07/15
46	Pre commissioning	16 wks	Mon 25/05/15	Fri 11/09/15
47	Commissioning	16 wks	Mon 14/09/15	Fri 01/01/16
48				
49	**Mining Equipment and Pre Strip**	**144 wks**	**Mon 20/08/12**	**Fri 22/05/15**
50	Equipment Tender and Award Shovels & trucks	32 wks	Mon 20/08/12	Fri 29/03/13
51	Equipment Supply First Unit	40 wks	Mon 27/05/13	Fri 28/02/14
52	Supply Other Units and Commission	15 wks	Mon 03/03/14	Fri 13/06/14
53	Prestrip	52 wks	Mon 26/05/14	Fri 22/05/15
54	Ore Production	0 wks	Fri 22/05/15	Fri 22/05/15
55				
56	**Seawater & Concentrate Pipelines**	**136 wks**	**Mon 17/09/12**	**Fri 24/04/15**
57	Pipeline EPCM Contract Award & Engineering	24 wks	Mon 17/09/12	Fri 01/03/13
58	**Pipeline Construction**	**112 wks**	**Mon 04/03/13**	**Fri 24/04/15**
59	Tender and Award	16 wks	Mon 04/03/13	Fri 21/06/13
60	Pipe purchase	26 wks	Mon 24/06/13	Fri 20/12/13
61	Mobilisation	4 wks	Mon 03/03/14	Fri 28/03/14
62	Pipeline Construction	60 wks	Mon 03/03/14	Fri 24/04/15
63	Pipeline commissioning	4 wks	Mon 30/03/15	Fri 24/04/15
64	Seawater at site	0 wks	Fri 24/04/15	Fri 24/04/15



Project: Preliminary Draft Santo Doming
Date: Wed 10/08/11

Critical | Critical Progress | Split | Baseline | Baseline Milestone | Summary Progress | Project Summary | External Milestone
Critical Split | Task | Task Progress | Baseline Split | Milestone | Summary | External Tasks | Deadline

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Table 24.2: Summarised Project Schedule

The preliminary schedule considers the following broad contracting strategy and major equipment deliveries:

- SAG and ball mills: 55 to 70 weeks (FOB USA) for large mills, and GMD drives have longer lead times; primary crusher: 60 weeks; flotation cells: delivery to site commencing at 20 weeks
- tender long-lead items during the feasibility study to enable commitments to be made soon after project approval is obtained
- finalize the selection of major equipment during the feasibility study to enable vendor drawings to be obtained soon after project approval
- lump sum tendering for all major contracts and purchases
- tendering with engineering drawings at 60% complete
- award a single contract to a mill supplier for the supply, transportation, installation, and commissioning of the mills
- fabricate structural steel and free issue to structural, mechanical and piping (SMP) contractor on site
- fabricate plate work and free issue to SMP contractor on site
- all equipment purchased by EPCM engineer on behalf of the principal, and free issued to SMP contractors
- award a contract for a batching plant to supply concrete to the civil works contractors
- in the plant area:
  - one contractor for bulk earthworks, roads and drainage, and water dams
  - one or two civil contractors for detailed earthworks and concrete works. This contract would include the supply of all reinforcing bar, holding-down bolts, formwork, etc.
  - one or two SMP contractors erecting structural steel, and installing equipment, plate work, and pipe work. This contract would also include the supply of minor equipment and materials
  - one contractor for the electrical and instrumentation installation
- infrastructure:
  - one contract for the supply, transportation, and installation of the permanent Village and construction camp. This would include the leasing of the construction camp, and its removal on completion of the project
  - one contract for the supply and installation of all field piping.

Ideally, the number of site contractors should be minimized, although this may be dictated by market and commercial considerations at the time.

25 INTERPRETATIONS AND CONCLUSIONS

The following conclusions arise from the information provided in the previous sections:

- The Santo Domingo deposit, encompassing Santo Domingo Sur, Iris and Iris Norte, represents a significant ore reserve.
- It is estimated that the deposits can be economically developed by open pit methods and that, according to the assumptions of this study, this development will provide a positive NPV.
- Groundwater is not expected to impact the open pit design based on the hydrogeology work carried out to date.
- Rock classification by rock mass rating (RMR) and rock quality designation (RQD) indicated that, in general, fair to good rock conditions are present at Iris Norte, with slightly reduced rock quality than that present at SDS/Iris. Although no drill data at Iris Norte are available, it is extrapolated that the rock quality present at Iris Norte would support overall wall angles comparable to SDS/Iris.
- The subsurface conditions at the key development sites of the plant site, tailings deposition area, and waste dump area, are all considered favourable from a geotechnical perspective for the envisioned project development. Cautionary recommendations include avoiding having acid drainages near foundations in the cemented soils and ensuring that concrete works are appropriately safe-guarded for sulphate soundness.
- Some rock samples indicated potentially moderate/severe to very severe potential attack from sulphates. In relation with the chloride content and water soluble sulphates, most of samples tested exceeded the referential allowable limits.
- The mine plan has not been fully optimized and it is likely that further scheduling work will smooth out some of the grade and ore extraction variations seen in this study. The optimized mine plan may mean that higher-grade ore is available to the mill sooner in the schedule, thus having a positive effect on the discounted cash flow.
- For some ore bodies, there is a strong negative correlation between ore competency (as measured by the DWi or Axb) and the iron assay. This results in higher mill throughputs for increasing iron assays in the ore. This is quite often seen in IOCG orebodies like Santo Domingo, where increasing levels of hematite and magnetite result in both lower competency and hardness and hence higher throughputs.
- Metallurgy testwork results and subsequent scale-up factors indicate a full-scale copper recovery of 90.2% for the average head grade of 0.34% Cu is possible. This is close to the average locked cycle result of 90.7%, from the metallurgy composites.
- Elemental analysis of the copper and magnetite concentrates indicate they are of high quality, with very low penalty element levels. The copper concentrate contains attractive

gold concentrations. Both the magnetite and copper concentrates should therefore be easy to market.

- A small number of flotation tests were conducted to assess the flotation response of the oxide composite. Sequential sulphide-oxide flotation with sulphidization agents was explored. In the best of six scoping tests, 25% of the copper was recovered in a rougher concentrate grading 3.2% Cu. Mineralogical examination of the oxide composite indicated that the bulk of the copper is present at non-floatable minerals. Further work is required to assess if heap leaching this oxide material is viable.
- High grade LIMS concentrates at high magnetite recoveries were produced from the 8-year and magnetite composites. Further work is required to improve the concentrate grade from the lower grade hematite composite.
- A series of tests on variability samples showed that there is a strong correlation between Davis Tube test results and LIMS cleaner tests and between Satmagan/magnetic susceptibility head grade and Davis Tube test recovery.
- The Chilean government organisation for the coordination of electrical installations, SDEC-SIC indicates that there will be sufficient high voltage power available for the project. Specific high voltage power studies, recommended by the SDEC-SIC are required for confirmation.
- Three concentrate transportation methods (pipeline, rail, truck) were evaluated. The pipeline option presented the lowest present cost and was selected as the preferred transport method.
- The offshore installations will take advantage of the close deep waters required for the design vessel, being connected to the onshore facilities by a 420 m long trestle. At the feasibility study phase basic loading plans and a more detailed marine operations assessment will have to be developed.
- Current project definition was sufficient to identify most relevant environmental and social impacts the project will likely cause. Among these, the main environmental impacts would affect air quality, water resources, and the social environment.
- A series of baseline studies are required for proper characterization of the environmental components required for the Environmental Impact Assessment (EIA).
- Air quality impacts will be relevant to the project due to the high levels of dust currently registered in Diego de Almagro (latency condition). Therefore, any increase in dust levels in the city caused by the project will be a matter for discussion with Chilean health authorities.
- Water resources are scarce in the Atacama Region. Proper groundwater modelling should be developed to demonstrate that potential aquifers won't be affected by the project facilities.
- Further FS-level engineering and definition of the port location and other facilities will be required in order to properly assess the project's environmental impacts and to prepare the EIA document.

- A trade-off study comparing a thickened process plant tailings stream (paste) with conventional tailings treatment should be undertaken. Paste tailings could have a positive impact during the EIA review, and significant for the community's perception of the project considering the current proposed distance of the Tailings Dam from Diego de Almagro.
- Land and territory screening against the project's current footprint indicates there would be no impact on natural parks, biodiversity conservation priority sites, or indigenous development land of the Atacama Region.
- In order to comply with IFC guidelines, FWM should start an early consultation process of the project with the surrounding community and stakeholders.

26 RECOMMENDATIONS

The Investigation and analysis carried out are considered appropriate to prefeasibility level mine design. Further investigations are recommended as the project advances to more detailed levels of design.

Recommendations for future work are listed below.

- Complete a Feasibility Study that considers the following points:
  - In-fill drilling is recommended to bring the first 3 years of the mine reserves into the proven category.
  - Conduct mine optimization studies to smooth out the mill-feed grade profile and the mining schedule.
  - Evaluate alternative ramp locations in the pit stages taking advantage of changes in wall slopes.
  - Conduct further waste rock facility geotechnical engineering studies to test all assumptions made in this report.
  - Update the optimum primary grind selection based on prevailing economic parameters for a 3-year pit composite.
  - Complete a mine geotechnical drilling program to take the geological model, structural model (major features and fabric) and hydrogeological model for the SDS/Iris and Iris Norte pits to a project level status.
  - Plant and TSF geotechnical investigations including borehole drilling and test pit excavations to determine the foundation, borrow, and fill placement conditions for design.
  - Investigate paste fill tailings deposition vs conventional wet tailings dam.
  - In the tailings area, a more detailed investigation program to improve the characterization of soils and develop an approximate profile of these clayey/silty soils.
  - Review mill selection in light of the latest comminution data. Deferral of the pebble crushing circuit, twin trains of twin pinion SAG mills or optimising the current SAG mill size should be considered
  - Include a metallurgy testwork program to define the following:
    - further investigation to confirm the optimum regrind levels for the copper rougher and LIMS rougher concentrates
    - locked cycle tests to determine the effect of water recycle on the copper metallurgy
    - determine if the 3rd copper cleaning stage is warranted
    - batch copper flotation and LIMS tests on a larger suite of variability samples to improve the statistical confidence level of head grade recovery regression equations
    - hematite recovery studies

- o determining the copper and LIMS metallurgy for the first 12 quarters of mine operation
- o slurry pipeline transportation testing of copper, LIMS and hematite concentrates
- o tailings thickening testwork and rheology
- o concentrate transportable moisture limit testing
- o additional reagent optimization for copper flotation
- o additional comminution testing of the outstanding variability samples from this phase using SMC test
- o additional comminution testing on a larger suite of variability samples, focusing on ores scheduled for processing in early mine years
- o heap leach testwork on copper oxide material

- Prepare a Chilean labour rate report
- Confirm land access for port facilities.
- Continue environmental base line studies
- Prepare an EIS and commence environmental permitting process
- Identify if there is an opportunity to share port facilities with other developing projects in the region.
- Specific high voltage power studies, recommended by the SDEC-SIC are required for confirmation of High Voltage Supply.

The estimated cost in US$M to complete the following activities is listed below:

- feasibility study, 5.0
- high voltage power studies. 0.5
- in-fill and geotechnical drilling. 20.0
- hydrogeology investigations. 0.5
- SEIA preparation. 3.0
- land access/acquisition. 13.0

Totalling US$ 42.0M

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