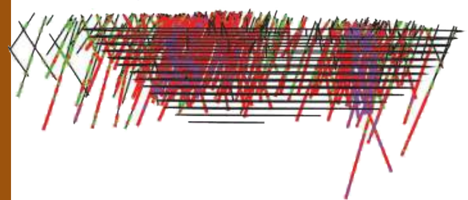
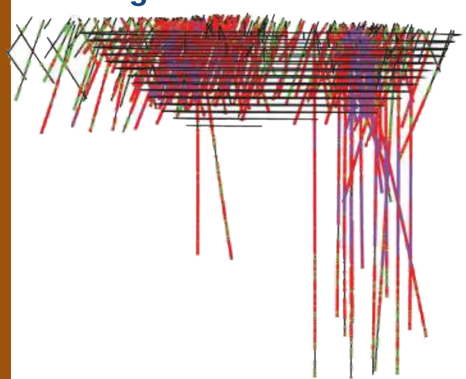
RED CHRIS

2012 Technical Report on the Red Chris Copper-Gold Project

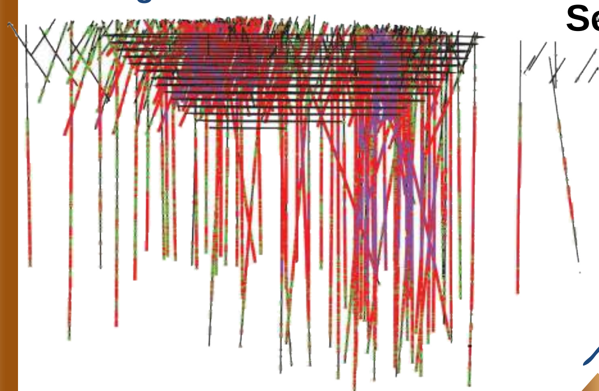
Drilling to the End of 2006



Drilling to the End of 2009



Drilling to the End of 2011



Red Chris Deposit
British Columbia
Liard Mining Division
Latitude 57° 42' North,
Longitude 129° 47' West
NTS map sheet 104H/12W

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Report Date:
February 14, 2012
Amended and Restated Date:
September 30, 2015

Effective Date for
the Reserve/Resource Estimate:
February 2, 2012

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1 Summary

1.1 Summary Introduction

This report ("2012 Technical Report") was originally issued on February 4, 2012 and was amended and restated on September 30, 2015. The effective date for the resource/reserve statement remains the same at February 2, 2012, as do all of the project economics and mine parameters of the original report. This amended report adds some clarity to several sections, as listed below

- Sec 3.) Removed reference to other qualified persons
- Sec 6.) Removed the wording about historic estimates not conforming to 43-101 regulations
- Sec 14.) Clarified QP`s position on sample validity
- Sec 17.) Included details on open pit/block cave constrained estimated resource
- Sec 31.) Clarified reference to budgets for recommendations
- Revised the certificates of QPs responsible for the report to more implicitly follow the wording of 43-101 regulatory act.
- Paul Sterling. P. Eng. was added as an Author and QP for section 16. Mr. Sterling authored this section in the original release of this report, but was only listed as a contributor in the original report.

This report was prepared for Imperial Metals Corporation to update the 2011 Technical Report on the Red Chris Copper-Gold Project ("2011 Technical Report”). This report updates all drilling and exploration activities conducted on the property to the end of 2011 and reports a new mineral resource for the property.

A block model was completed in January of 2012 to account for all the new deep drilling conducted in 2010 and 2011. The new resource statement presented in this report is based on this 2012 model.

The mine plan and mineral reserve presented in this report remain unchanged from the 2011 report and were based on the May 2010 model. The in pit reserve was re-run in January of 2012 with the new 2012 model as a check. The results showed an in-pit increase in tonnage of 8.3% compared to the values presented in this report with about the same grades.

All past Technical reports on the Red Chris Property can be found on SEDAR. http://www.sedar.com

Note: On SEDAR, the 2004 Technical Report is available under Company Documents for bcMetals Corporation. The 2010 and 2011 Technical Reports are under Company Document for Imperial Metals.

The four authors of this report (Greg Gillstrom P.Eng., Raj Anand P.Eng., and Stephen Robertson P.Geo., Paul Sterling, P.Eng.) are all employees of Imperial Metals Corporation.

Below is a list of contributing consultants.

- **Merit Consultants International Inc. -** Project capital costs
- **Swallow Services Ltd.** - Updating the financial model
- **AMEC Earth and Environmental (AMEC E&E) --** Geotechnical Investigation and detailed design for the Tailings Storage Facilities, and water management plan
- **Clearwater Consultants Ltd. (CCL)** - Updating precipitation estimates
- **Landmark Consultants -** Design and construction management for the temporary access road and preliminary engineering for the permanent access road.
- **SRK Consultants** - Metal Leaching and ARD Management.

Table 1.1 Summary Fact Sheet (as of Feb 14, 2012)

PROPERTY	Red Chris, 68 Mineral Claims, totaling 16,994.00 hectares
LOCATION	18 km southeast of town of Iskut and 80 km south of Dease Lake in northwest British Columbia. The property is centered on latitude 57° 42' north, longitude 129° 47' west within NTS map sheet 104H/12W, Liard Mining Division.
OWNERSHIP	100% owned and operated by Red Chris Development Company Ltd., a wholly owned subsidiary of Imperial Metals Corporation, subject to a 1.8% net smelter return royalty (NSR) held by Xstrata Canada Corporation (successor to Falconbridge Limited) on all or portions of 32 mineral claims.
OPERATOR	Red Chris Development Company Ltd. 580 Hornby Street, Suite 200. Vancouver. BC, V6C 3B6 (604) 669 8959 Contacts: Raj Anand. Manager Project Development: rajanand@imperialmetals.com Steve Robertson, Exploration Manager : srobertson@imperialmetals.com
ACCESS	A 17 km long temporary access road from the Ealue Lake Road side. For permanent use the existing road will be upgraded and extended to form a 23 km long access from Highway 37.
GEOLOGY	Copper/gold porphyry deposit. Chalcopyrite and bornite, in intrusive host.
METALS	Copper, Gold and Silver.
MINEABLE RESERVES	Approximately 301.6 million tonnes at 0.36 % copper and 0.27 g/t gold.
MINE	Conventional shovel, truck and open pit mine Average mining rate of about 30 million tonnes with maximum mining rate of about 40 million tonnes per annum.
PROCESSING	Conventional crushing, grinding and floatation technology to produce copper and gold concentrate for shipment overseas through the Port of Stewart.
LIFE OF MINE	28.3 years
ENVIRONMENTAL APPROVALS AND PERMITTING	Provincial Environmental Assessment completed and EA Certificate obtained in August 2005 and extended in July 2010. Federal Environmental Assessment completed and EA Certificate obtained in May 2006. Permitting application process for project development is in progress. Development is contingent on availability of power. Federal and Provincial governments have committed to fund building of a 335 km long, 287 kV power transmission line from Terrace to Bob Quinn for which the construction has now commenced. RCDC will need to build approximately 115 km long power transmission line extension from Bob Quinn to the mine.

1.2 Location and Background

The Red Chris property is located about 18 km southeast of the village of Iskut and 80 km south of Dease Lake on the Todagin Plateau in northwestern British Columbia, Canada (see figure 1.3).

The Red Chris Project is comprised of two components, the Red Chris claims and the Red claims. The Red Chris claims consist of 50 mineral tenures containing 528 cells and covering 9,688.98 hectares. Red Chris Development Company Ltd. ("RCDC") has a 100% interest in the Red Chris claims and all or portions of 32 claims are subject to a 1.8% net smelter return royalty ("NSR") held by Xstrata Canada Corporation (successor to Falconbridge Limited). The 1.8% NSR may be reduced to 1% at any time prior to commencement of commercial production in consideration of a payment to Xstrata of $1,000,000. RCDC is a wholly owned subsidiary of Imperial. The Red claims cover the northern part of the project and consist of 18 mineral claims containing 428 cells and covering 7,305.02 hectares. Imperial Metals Corporation owns 100% of the Red claims (see Figure 4.2). A total of five mining lease applications comprised of 25 mineral claims and covering 5,141.26 hectares have been filed with the Mineral Titles Branch following the completion and approval of legal surveys over certain mineral claims.

The Red Chris porphyry copper-gold deposit is distributed along the central axis of the pervasively altered and fractured formation called the Red Stock. At the Red Chris deposit, the Red Stock is the predominant host of the mineralization. Mineralization and associated alteration are more intense adjacent to the ancestral en echelon fault system along the axis of the stock which controlled the emplacement of the stock and later altering and mineralizing hydrothermal fluids which is more typical of a shear-hosted copper-gold deposit. The Red-Chris copper-gold mineralization has good near-vertical and longitudinal continuity, controlled largely by post-mineral faulting superimposed on and along the ancestral, en echelon, central axis fault zone. Pyrite, chalcopyrite, bornite, with minor chalcocite are the principal sulphide minerals of the shallower portions of the Red Chris deposit. Minor covellite occurs as inclusions in pyrite, and molybdenite, sphalerite and galena occur locally in trace amounts. Gold, second in economic importance to copper, occurs spatially and genetically associated with the copper mineralization. In the newly discovered deep East Zone pyrite becomes significantly decreased and in places bornite becomes the dominant copper mineral.

From 1968 to 2006, the property was explored by Conwest Exploration Ltd., Great Plains Development Co., Silver Standard Mines Ltd., Ecstall Mining Limited, Texasgulf Canada, American Bullion and bcMetals Corporation (via its wholly owned subsidiary RCDC).

In 2003 and 2004 RCDC conducted an infill drilling program targeting the core in East and Main Zones where open pit mining would be expected to take place. Based on this drilling, the Red Chris Measured, Indicated and Inferred Resources were updated and RCDC commissioned AMEC to complete a Feasibility Study of an open pit mine at Red Chris. The Technical Report was published in December, 2004 and the Feasibility Study in March of 2005.

On September 8, 2006 Imperial's subsidiary CAT-Gold launched an all cash takeover bid of bcMetals Corporation, at $0.95 per/share. bcMetals responded by adopting a poison pill which limited potential ownership of the company to 20%. Upon termination of the initial takeover bid on November 8, 2006, Imperial owned approximately 17% of bcMetals. On November 23, Taseko Mines made an offer for all outstanding shares of bcMetals, to which Imperial responded with a friendly offer of $1.10/share, representing a 4.8% premium over Taseko's offer. A bidding war ensued, which Imperial eventually won with a final bid of $1.70/share submitted on February 2, 2007 for total cost of $68.4 million.

In June 2011 RCDC completed the acquisition of all the common shares of American Bullion Minerals Ltd. not already owned by RCDC and now beneficially owns or holds all of the issued and outstanding common shares of American Bullion.

1.3 Exploration

Since acquiring the Red Chris Property Imperial has conducted an aggressive exploration and drilling program, consisting of diamond drilling, geological mapping, and geophysics. During the period from 2007 to 20011 Imperial drilled 20 holes targeting deep mineralization in the East, Main, and Gully Zone's. Figure 1.1 shows a comparison of the drilling at the end of 2006 with the drilling to the end of 2009 and to the end of 2011.

The highlights from the program were holes 07-335 and 09-350. Hole 07-335 was collared vertically in the core of the East zone, and graded 1.01% copper, 1.26 g/t gold its entire length of 1,024.1 metres. This extended high-grade mineralization in the East Zone down another 270m from its previously-known extent. Hole RC09-350 was collared approximately 170 metres northeast of drill hole 07-335 and returned 432.5 metres grading 2.00% copper and 3.80 g/t gold which included a 152.5m zone grading 4.12% copper and 8.83 g/t gold starting at a depth of 540.0 metres (see figures 1.2 and 11.3).

Figure 1.1 Red Chris Drilling Progression 2006 to 2009 to 2011

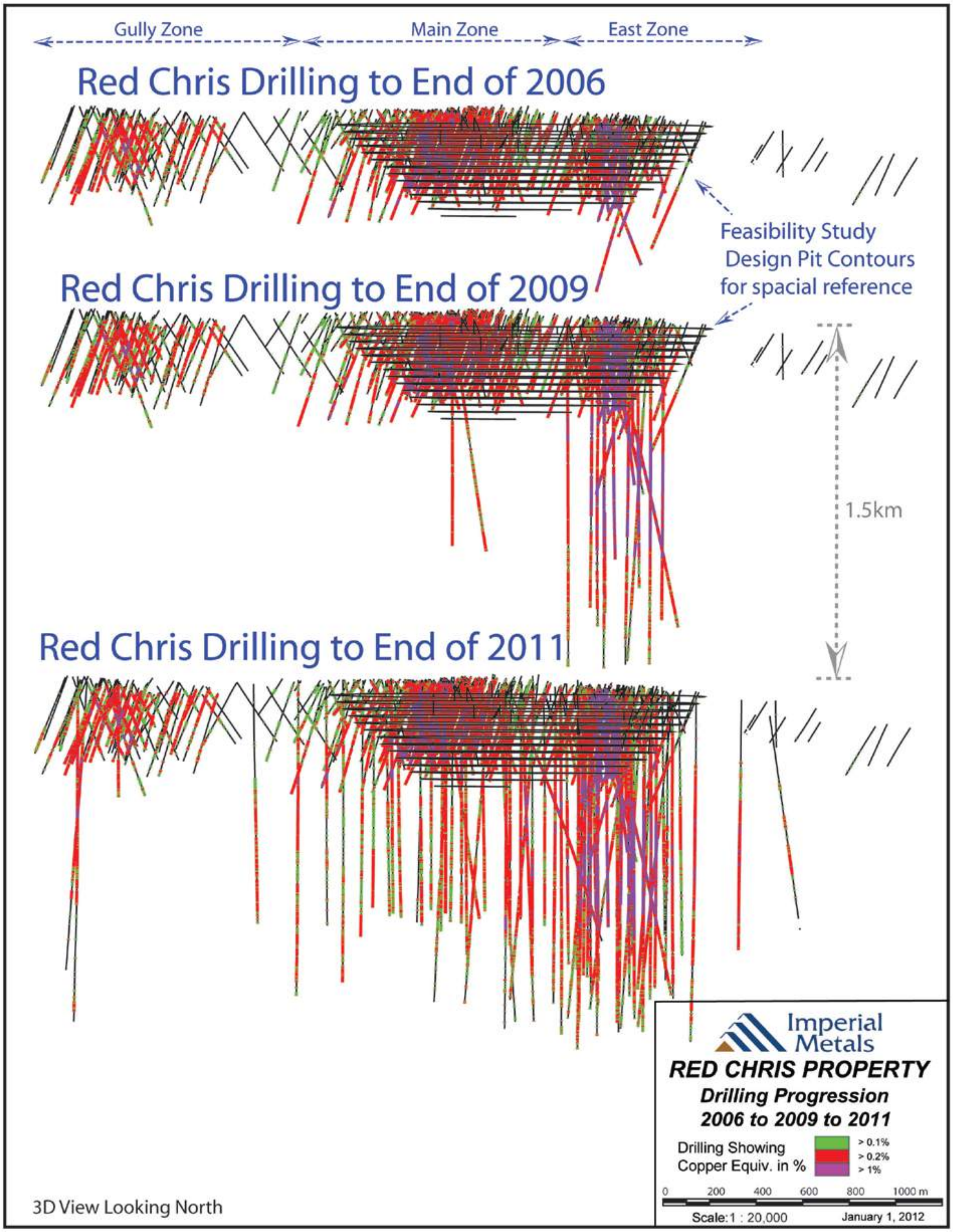
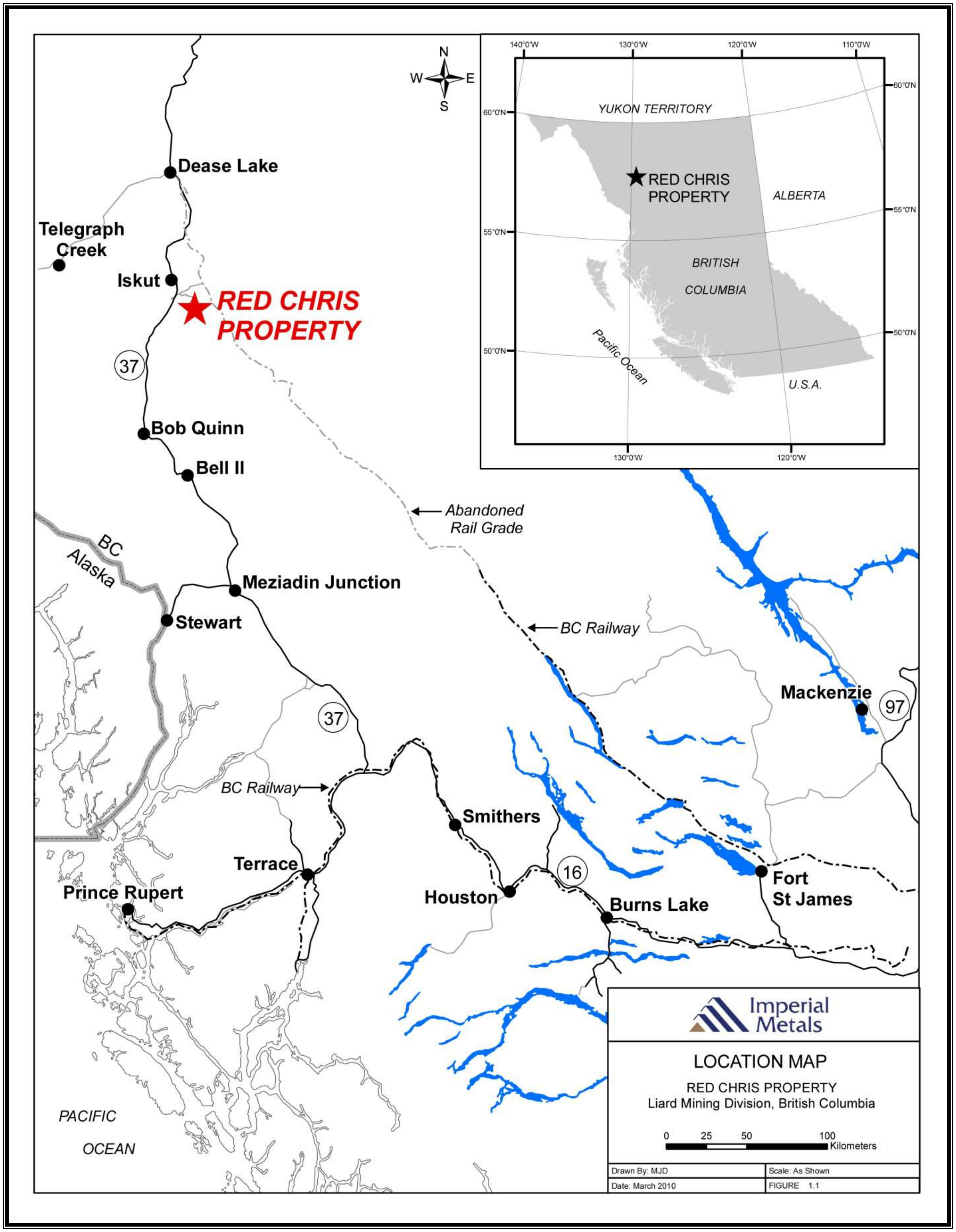


Figure 1.2 Drill Core from Drill Hole RC09-350

Figure 1.3 Red Chris General Location Map



1.4 2012 Red Chris Resources

A new block model for the Red Chris deposit was completed in January of 2012 to include the new deep drilling done in 2010 and 2011 (see Table 1.2 and 1.3). This model included all new drilling up to hole RC11-565 (62 new diamond drill holes, totaling over 69 thousand additional metres completed since the May 2010 resource update). The resource was originally published at various copper equivalent cutoffs, constrained to a wireframe shell within the model domains. The amended and restated resource below shows the resource within the model domains, constrained to a combination of Open Pit and Block Cave wire frame digital solids. Tables 1.2 show a summary of the resource values. A detailed breakdown of the economics used for both mining methods is available in Chapter 17.

Table 1.2 Red Chris 2012 Open Pit/Block Cave Constrained Mineral Resource

Red Chris 2012 Total Open Pit/Block Cave Constrained Resource								
Material		Ore	Mill Head	Insitu Grades				
Class		millions	Value	Copper Equiv.	Copper	Gold	Silver
		Tonnes	$/tonne	(%)	(%)	(g/t)	(g/t)
MEASURED		830.7	$25.13	0.57	0.36	0.36	1.17
INDICATED		203.0	$18.55	0.47	0.30	0.29	1.01
M&I		**1,034.7**	**$23.84**	**0.56**	**0.35**	**0.35**	**1.14**
INFERRED		**787.1**	**$18.65**	**0.48**	**0.29**	**0.32**	**1.04**
WASTE		1.2	-$6.40				

**Mill Head Value is a calculation of the value of material mined, in Canadian dollars per metric tonne, once it reaches the Crusher Pocket. This includes all downstream costs from the crusher forward, including: Milling / Concentrate handling and transportation / Treatment and refining / Royalties / Sustaining capital / Administration and head office overhead costs. Large capital costs associated with expansions, such as mining fleet additions, or replacements are not included. See table 17.24 for metal recovery formulas, costs and parameters used to calculate this value*

***Copper Equivalent % = [Copper Grade (%) + (.60415 * Gold Grade (g/t))]; based copper/ gold price ratio at Copper - $3.50 /lb, Gold $ 1450/oz*

1.5 Feasibility Study

Imperial completed a Red Chris Feasibility Study Update at the end of 2010. The major findings were published by Imperial in a News Release dated November 16, 2010. The details of that study are presented herein. In general, the pit and plant design remain virtually the same as the 2004 Study but the resource estimate, operating and capital costs have been revised.

The project is based on conventional shovel and truck open pit copper/gold mine with a 30,000t/d processing plant using standard mineral flotation technology, to produce an average of 337t/d of concentrate over LOM. Mine life is estimated to be approximately 28.3 years.

1.6 Financial Conclusions

Financial calculations are predicated on the following base case assumptions:

Table 1.4 Base Case Price and Exchange Rate Assumptions

Description	Assumption
Copper price	US$2.20/lb
Gold price	US$900.00/oz
Silver price	US$12.00/oz
Cdn$	**US$0.90**

Table 1.5 shows the key economic findings derived from the financial model. This table describes the project's after tax financial performance using base case assumptions and January 2012 monthly average metal prices and exchange rates.

Table 1.5 Financial Results

Financial Results (100% Equity Basis) Using Base Case and January 2012 Average Financial Parameters	BASE CASE		JANUARY 2012AVERAGE	
	Copper Price	US$2.20/lb	Copper Price	US$3.6485/lb
	Gold Price	US$900/oz	Gold Price	US$1,656.095/oz
	Silver Price	US$12/oz	Silver Price	US$30.769/oz
	Cdn$ =	US$0.90	Cdn$ =	US$0.986911676
Post Tax IRR	15.7%		38.8%	
Post Tax NPV				
@0%	Cdn$1.214 B		Cdn$3.675 B	
@5%	Cdn$423.2 M		Cdn$1.571 B	
@10%	Cdn$133.9 M		Cdn$772.1 M	
Project Payback	4.6 yrs		1.81 yrs	
LOM Production Cost of per lb of Cu with credits from Au and Ag	US$1.22		US$0.96	

Table 1.6 Major Input Parameters for Financial Analysis

Major Input Parameters		
Total Mineable Reserves Copper grade Gold grade		**301.549 M tonnes** 0.359% (avg) 0.274 g/t (avg)
Metal - In Situ	- Copper	2.380 B lbs
	- Gold	2.67 M oz
Metal - In Concentrate	- Copper	2.080 B lbs
	- Gold	1.32 M oz
Production Rate and Mine Life		
Production rate	-ore milled	10.95 Mt/a
Life of mine (LOM): including reprocessing of stockpiled material in years 24 to 28.3		28.3 years
LOM strip ratio:		1.25:1
Copper Recovery - LOM average Gold Recovery - LOM average		87.0% 49.3%
Concentrate grade		27% Copper
Concentrate production – daily - LOM average		337 dmt/d
Concentrate production – annual – peak		183,000 dmt/a
Concentrate production – annual - average		123,000 dmt/a
Capital Costs		
Initial capital costs, including Indirect costs and contingency @10%.		$ 443.6 M
Working capital		$ 26.8 M
Sustaining capital (LOM)		$ 238.3 M
Net Revenue at Mine Gate	**Cdn $/t ore**	**$17.79**
Less Mining Cost	$/t ore	$4.45
Less Milling Cost	$/t ore	$3.87
Less Royalty	$/t ore	$0.18
Less Reclamation Expenditure	$/t ore	$0.04
Less Project Overhead	$/t ore	$1.09
Less Head Office costs **Total Operating Costs**	$/t ore **$/t ore**	$0.33 **$9.96**
Net Current Proceeds before Capital cost	**Cdn $/t ore**	**$7.84**

Base case economic analysis has been run with no inflation (constant dollar basis). Capital and operating costs are expressed in 2010 Canadian dollars, unless otherwise noted.

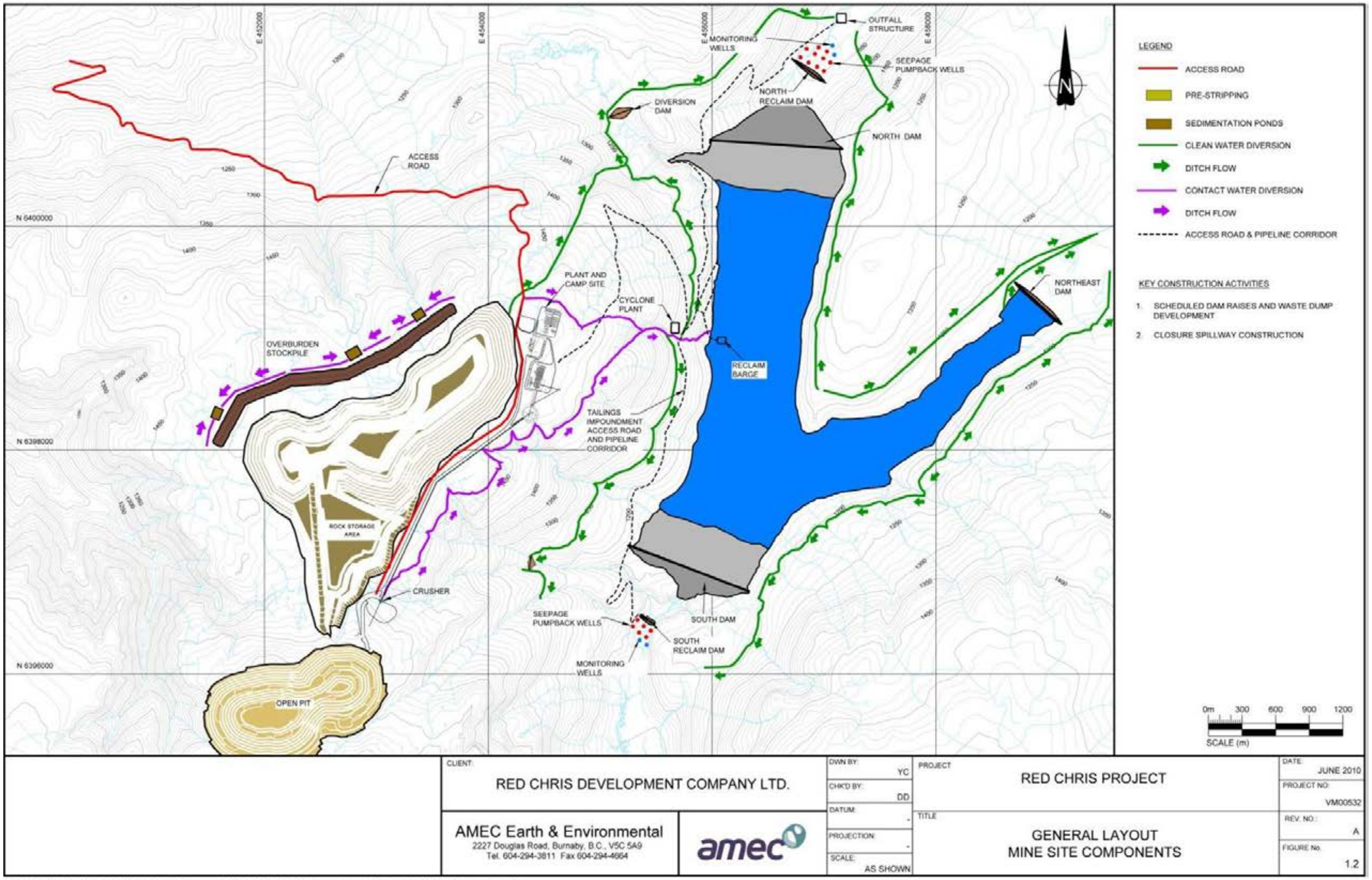
1.7 Key Updates since the 2005 Feasibility Study (to Feb. 14, 2015)

- Provincial Environmental Assessment process completed and Environmental Assessment Certificate obtained in August 2005, and duly extended in July 2010.
- Federal approval for the Red Chris project under the Canadian Environmental Assessment Act (CEAA) completed in May 2006.
- SRK completed a due diligence study on the earlier published Feasibility Study of 2005 on behalf of potential financiers of the Red Chris project, then owned and operated by bcMetals. SRK's due diligence study was completed in March 2006.
- Ownership changed from bcMetals to Imperial as of February 2007.
- Continued onsite exploration since 2007, updated the mineral resources at varying cut-off grades as reported in Technical Report: 2010 Exploration, Drilling and Mineral Resources Update.
- As a follow-up on SRK findings of 2006, completed additional field geotechnical investigation for the plant site and for the tailings impoundment area and accordingly revised the plant site location and overall site layout.
- With the continued on-site climatic data gathering, revised the average precipitation and other climatic considerations.
- Updated mineable mineral reserves within the pit design to reflect the current economic environment.
- Mineable mineral reserves are now based on mill head values rather than based on NSR.
- Capital cost updated to reflect the current economic environment and also to include the construction of a power transmission extension line from Bob Quinn to the Mine site.
- Operating costs updated to reflect the current economic environment and also with references to other mines owned and operated by Imperial.
- Updated the financial model with the revised capital and operating costs.
- Provincial and Federal environmental assessment for the NTL project has been approved and as of January 2012, construction of the Northwest Transmission Line has commenced.
- Red Chris Project at the present is in the advanced stage of permitting for on-site construction to commence in spring 2012.

1.8 Ownership and Agreements

- RCDC will operate the Red Chris mine.
- RCDC was previously a wholly owned subsidiary of bcMetals. Imperial acquired bcMetals in February 2007. bcMetals and RCDC were then merged and continued as RCDC which is now a wholly owned subsidiary of Imperial.
- RCDC has a 100% interest in the Red Chris Claims (through ownership of ABML).
- A 1.8% NSR royalty is held by Xstrata Canada Corporation (successor to Falconbridge Limited).
- The 1.8% NSR may be reduced to 1% at any time prior to commencement of commercial production in consideration of the payment to Xstrata of $1,000,000.

Figure 1.4 Overall Site Layout



1.9 Feasibility Mineral Reserves

The table below shows as summary of reserves used in this report.

Table 1.7 Proven and Probable Reserves

ITEM DESCRIPTION	P&P Reserves
Tonnes milled	301.549 M
Copper	0.359%
Gold	0.274 grams/tonne
Contained pounds of copper	2.380 B
Contained ounces of gold	2.67 M
LOM Strip Ratio	1.25

Projected mine life with the currently defined mineable reserves is approximately 28.3 years. Maximum mining production rate will be about 40 million tonnes per annum.

1.10 Mining

Initially mining will be conducted in two open pits, the Main and East Pits. Later in the mine life these two pits will be merged into one large pit.

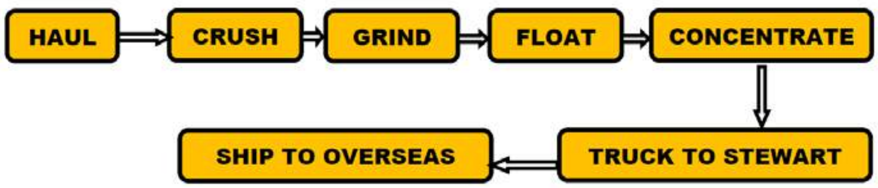
The mining operation is a conventional shovel and truck, open pit porphyry copper/gold mine, feeding a 30,000 tpd processing plant using standard mineral flotation technology. The mine has been phased and scheduled to maximize the production of high-grade ore, especially during the first five years, to minimize the capital payback period.

Waste rock, depending on the neutralizing potential to acid generating potential (NP/AP) ratio, will be stockpiled as potentially acid generating (PAG) or non-acid generating waste rock (NAG). NAG waste rock will generally be used as base below the PAG rock storage area, as road topping material or as general construction material. Depending on the available quantity of NAG rock, the base below the PAG waste will be designed from one to five metres thick.

1.11 Processing

The ore will be processed through a concentrator that will produce a copper/gold flotation concentrate. The concentrate will be trucked to concentrate storage and ship-loading facilities at the Port of Stewart and shipped to smelters overseas. The nominal milling rate will be 30,000 tonnes per day. Processing will be based on a conventional copper/gold porphyry flowsheet, with a semi-autogenous grinding (SAG) mill, pebble crusher and ball mill grinding, rougher/cleaner flotation and concentrate dewatering.

Figure 1.5 A simplified process flow-sheet is shown below:



1.12 Tailings Storage Facility and Associated Structures

The site of the proposed TSF is in a Y-shaped valley, approximately 3.5 km northeast of the Red Chris ore deposit (see figure 11.12). The TSF straddles the watershed divide between Trail Creek and Quarry Creek. Construction of three dams for the TSF will be required with one located at each of the south, north and northeast arms of the valley. These dams are designated as the South, North, and Northeast Dams respectively. The South and Northeast Dams will not be required at the start-up of the operation.

The proposed TSF will affect the headwaters of Trail Creek, which is considered a natural water body frequented by fish. After consideration of options for placement of the TSF dams to avoid or minimize impact to fish habitat, the TSF dams were located as described above. This option includes a Fish Habitat Compensation Plan (FHCP) with the compensation works exceeding a 2:1 ratio to compensate for fish habitat losses in Trail Creek.

1.13 Ancillary Facilities

On-site components will include the open pit mine, process plant, TSF, rock storage area, low grade ore stockpile, mine camp and associated works, new access/haul roads and related infrastructure including upgrades to existing access roads from mine site to Highway 37, water supply and associated works, power supply and related infrastructure from the mine site to Highway 37, maintenance shop, explosives storage and/or manufacturing facility and other related works. All these facilities are located on RCDC mineral tenures.

Off-site components will include the power transmission line extension along Highway 37, concentrate storage and ship-loading facilities at the Port of Stewart, and the fisheries compensation works.

1.14 Permitting Status

- The Red Chris BC Environmental Assessment Act (BCEAA) Certificate (M05-02) was issued in August 2005. The certificate extension was obtained on July 9, 2010. Federal approval for the Red Chris project under the *Canadian Environmental Assessment Act* (CEAA) was received in May 2006. The Federal approval was subsequently challenged by a third party, however a subsequent decision by the Supreme Court of Canada, on January 21, 2010, upheld the Federal approval, which has allowed mine permitting and development to proceed.
- A joint Mines' Act and Environmental Management Act Permit Application was submitted on July 23, 2010 to the North West Mine Development Review Committee (NWMDRC).
- Considering the proposed coordinated permitting approach by the NWMDRC it is anticipated that all permits for the project development and construction will be obtained by late spring / early summer of 2011.
- Development of the Red Chris mine will be contingent upon the availability of electric power from BC Hydro at Highway 37 near Tatogga at standard industrial rates, or other form feasible for the viability of the project, or an alternative viable power source including all necessary approvals as may be required under the BCEAA.
- The Provincial and Federal governments have committed to provide funding for a 287 kV power transmission line from Terrace to Bob Quinn, the Northwest Transmission Line (NTL), which now after required provincial and federal environmental assessment has been approved and as of January 2012 construction has commenced.
- Assuming that the NTL is constructed, RCDC will be responsible for extending powerline service, sufficient to meet its needs, from Bob Quinn to Tatogga and from there to the Red Chris mine. The power transmission line extension along Highway 37 requires approval through an amendment of the Red Chris Environmental Assessment (EA) Certificate and will also require a permit from the BC Ministry of Transportation and Infrastructure (MOTI) to allow development within the Highway 37 Right-of-Way (ROW). RCDC has applied for an amendment of the Red Chris EA Certificate in this regard and this Application is under review.

1.15 First Nations

The Red Chris property is located in a sparsely populated area of the northwest region of British Columbia, and within the Tahltan Nation traditional territory. The communities in the area are considered remote and include Iskut, Dease Lake, and Telegraph Creek.

The District Municipality of Stewart (SDM) is included in the study as concentrate from the mine will be trucked to the Port of Stewart for shipment overseas. SDM is within the Kitimat-Stikine Regional District.

Two First Nations bands, each with an elected council, represent the Tahltan people. The Tahltan Band has its office in Telegraph Creek and the Iskut Band has its office in Iskut. The Tahltan Central Council (TCC) represents both bands, and non-resident band members, on issues of joint concern and is comprised of an elected executive, Chiefs of both bands, and representatives of the traditional Tahltan families. The TCC has its office in Dease Lake.

RCDC has a Memo of Understanding with the TCC indicating their willingness to cooperate in the development of the Red Chris project. Several community meetings in Iskut, Dease Lake and Telegraph Cree have been held during EA process and now during the permitting process. Tahltan, through their Tahltan Heritage Resources Environmental Assessment Team (THREAT) are active participating members in reviewing the permitting for Red Chris.

1.16 Capital Cost Summary

Capital costs have been estimated to reflect the current economic environment.

All costs are in 2010 Canadian dollars.

Table 1.8 Direct Capital Costs

Direct Capital Costs	X $1,000 (Cdn)
Mine Preproduction Development	2,646
Mine Equipment	72,185
Mine Dewatering	538
Crushing	6,990
Conveyors	18,758
Coarse Ore Storage	5,060
Grinding	41,915
Flotation and Regrinding	21,607
Concentrate Dewatering	4,306
In Plant Tailings System	2,018
Reagents Handling and Storage	4,061
Process Building	16,430
General Site Development	5,939
Access Roads	1,761
Power Generation or Supply	29,713
Power Distribution	8,308
Fuel Supply, Storage & Distribution	531
Fresh Water Supply & Distribution	3,625
Fire Protection and Prevention	612
Waste Disposal	59
Control and Communications Systems	1,850
Warehouse and Maintenance	13,279
Administration Building	1,997
Laboratory Facilities	2,112
Cold Storage Building	284
Gate House	59
Plant Mobile & Utility Equipment	4,583
Explosives Handling and Storage	100
Tailings Management Facility	30,738
Tailings and Sands Pipelines	8,093
Reclaim System	10,688
Total Direct Costs	**$320,845**

Table 1.9 Indirect Capital Costs

Indirect Capital Costs	X $1,000 (Cdn)
RCDC's Costs including allowance for a concentrate storage shed	11,450
Engineering Procurement Construction Mgmt	25,294
Construction Indirects	12,996
Construction Camp and Catering with allowance for conversion to permanent camp	13,735
Capital Spares	5,587
Freight	12,106
Vendor Reps	150
Start-up & Commissioning	1,140
Total Indirects	**$82,458**
Total Capital Cost without Contingencies	**$403,303**
Contingencies @10%	40,330
Total Capital Cost with Contingencies	**$443,633**

1.17 Operating Cost Summary

The operating cost update is based on RCDC operating the mine itself including all mining, processing and maintenance. Cost estimates include experience drawn from Imperial's open pit operations located in British Columbia. The average operating costs over the LOM are shown in the following table.

Table 1.10 Operating Cost Summary

Area	Per Tonne Milled (LOM)
Mining	$4.45
Milling	$3.87
Royalty	$0.18
Reclamation Expenditure	$0.04
Project Overhead (G&A)	$1.09
Head Office Costs	$0.33
TOTAL OPERATING COST	**$9.96**

1.18 Schedule

Red Chris mine's development is contingent on the approval and construction of the NTL project which has now completed its EA and permitting process. As of early January 2012 the NTL construction has commenced starting from the Bob Quinn end. The NTL is anticipated to be operational by May 2014. RCDC will continue monitoring the NTL's construction schedule and plans to complete mine development and installation of the required powerline extension to Bob Quinn to coincide with the NTL powerline completion. Based on completion of NTL by the end of May 2014, RCDC proposes the following permitting, development, construction and operation schedule:

Table 1.11 Project Schedule

Activity	Target Date
Apply for the Mines' Act Permit	July 2010
Request DFO to initiate MMER Schedule 2 Process	June 2010
Feasibility Study Update	November 2010
Site Clearing for the Plant Site, Mobile Equipment Shop Area and the Camp Area	Early 2012
Commence Detailed Engineering and Procurement	November 2010
Complete MMER Schedule 2 Process and Obtain DFO Authorization for TSF construction commencement	2012
Commence Foundation Preparation for the North Dam (TSF)	Summer of 2012
Commence Major Construction at Red Chris	Early 2012
Commence Construction of Power line Extension	Late 2012
Complete Construction	April 2014
Power-line hook-up at Bob Quinn	May 2014
Commission and Start-up	June 2014
Production Start-up	Mid 2014
Production Operation	2014 to 2041
Commence Closure and Reclamation	Early 2042
Complete Closure and Reclamation	Late 2043
Post-closure Monitoring and Surveillance	2043 to 2048

1.19 Financial Analysis

1.19.1 Valuation Methodology

The Red Chris project has been valued using a discounted cash flow approach.

The analysis includes sensitivity to variation in copper and gold prices, head grades, metallurgical recoveries, operating costs, capital costs, concentrate transportation costs, and smelting and refining charges.

The economic analysis has been run with no inflation (constant dollar basis). Capital and operating costs are expressed in 2010 Canadian dollars, unless otherwise noted.

Financial model parameters incorporated in the analysis, include:

- ore reserves
- metallurgical balance
- smelter terms
- concentrate transportation costs
- markets and contracts
- metal prices
- exchange rate
- operating costs: mine, process and G&A
- head office costs
- capital costs over LOM
- royalties
- taxes

Estimated annual net cash flows were discounted to the beginning of project Year -4 at real discount rates of 5% and 10%.

1.19.2 Sensitivity Analysis

The results of the base case and sensitivity analysis are summarized in Table 1.12, and Figures 1.6 and 1.7.

The estimated base case IRR based on 100.0% equity is 15.7% after tax and the project undiscounted cashflows a total of $1.214 billion, after capital costs and taxes. Payback of the total construction cost of $443.6 million is achieved within 4.6 years of start up at the base case prices. LOM cost per pound of copper, for the base case, taking silver and gold as a credit is US$1.22 per pound of copper.

At January 2012 monthly average metal prices and exchange rate (Cu = US$3.6485/lb, Au = US$1,656.095/oz, Ag = US$30.769/oz, and exchange rate as Cdn$1 = US$0.9869), the estimated

IRR increases to 38.8% post tax and the project undiscounted cash flows is $3.675 billion, after capital costs and taxes. Payback of the total construction cost of Cdn$443.6 million is achieved within 1.81 years of start up at these prices. LOM cost per pound of copper, at January 2012 monthly average metal prices, taking silver and gold as a credit is US$0.96 per pound of copper.

Table 1.12 Financial Analysis: Base Case- January 2012 Metal Prices

Economic Parameters	BASE CASE		January 2012 Price	
	Copper Price	US$2.20/lb	Copper Price	US$3.6485/lb
	Gold Price	US$900/oz	Gold Price	US$1,656.095/oz
	Silver Price	US$12/oz	Silver Price	US$30.769/oz
	Cdn$ =	US$0.90	Cdn$ =	US$0.986911676
Capital Cost	Cdn$443.6 M		Cdn$443.6 M	
Post Tax IRR	15.7%		38.8%	
Post Tax NPV				
@0%	Cdn$1.214 B		Cdn$3.675 B	
@5%	Cdn$423.24 M		Cdn$1.571 B	
@10%	Cdn$133.92 M		Cdn$772.1 M	
Project Payback	4.6 yrs		1.81 yrs	
LOM Production Cost of per pound of Copper with credits from Gold and Silver	US$1.22		US$0.96	

Figure 1.6 Project Sensitivity to Operating Cost Variables

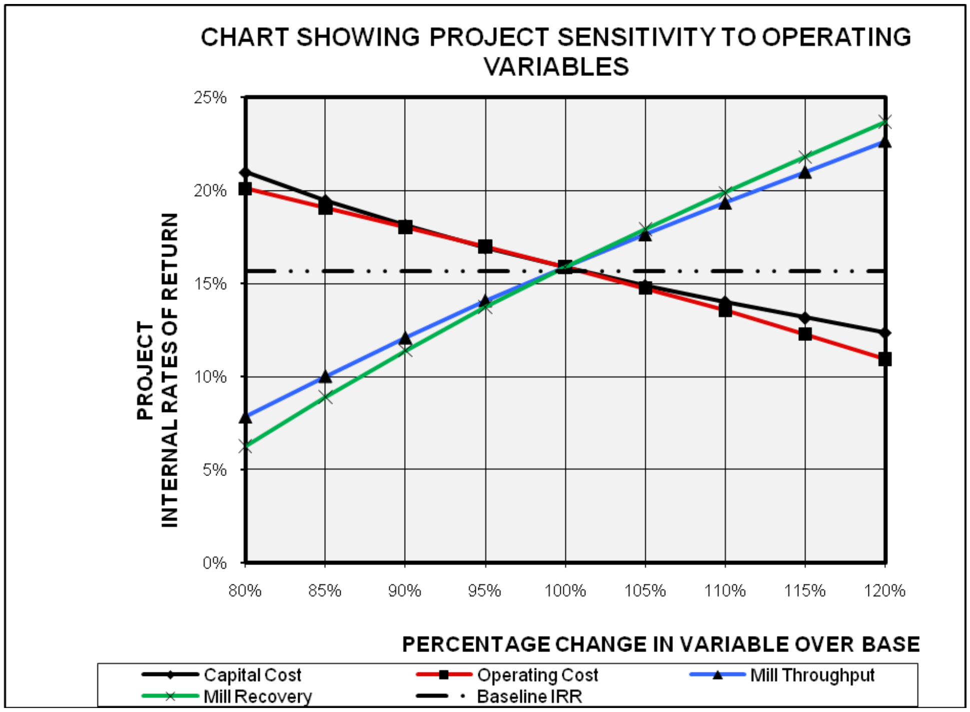
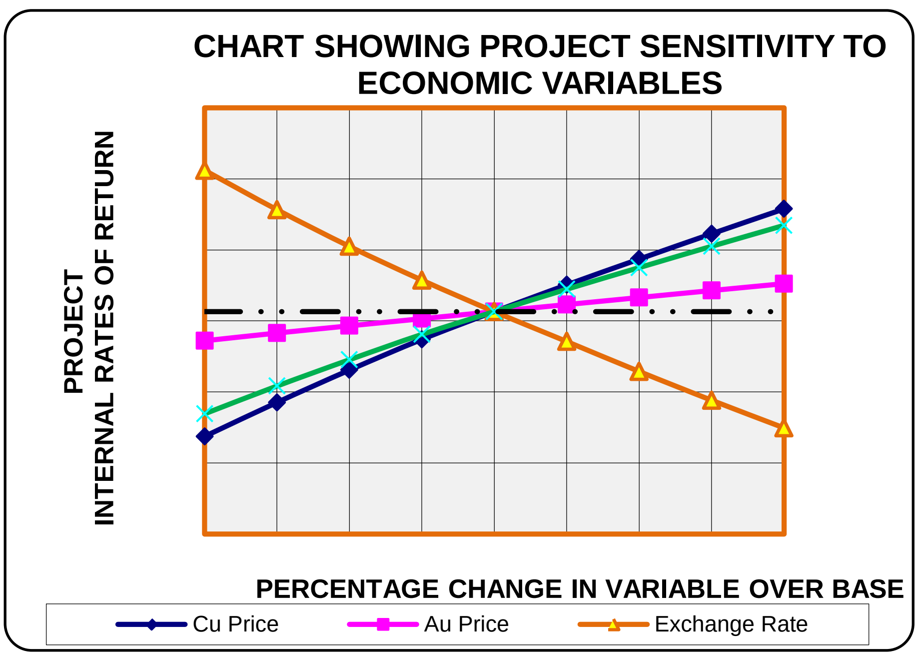


Figure 1.7 Project Sensitivity to Economic Variables



1.20 Recommendations

Based on the findings of this 2012 Technical Report, it is concluded that the project has robust economic viability. The Provincial and Federal governments have committed to provide funding for a powerline from Terrace to Bob Quinn (Northwest Transmission Line ("NTL") Project), for which the provincial and federal environmental assessment process has been completed and the project is now under construction. Assuming that the NTL project is constructed, Imperial will be responsible for extending powerline service sufficient to meet its needs from Bob Quinn to Tatogga and from there to the Red Chris Mine. Target completion date for the NTL Project is now May 2014. It is recommended that RCDC continue with detailed engineering, procurement, construction, and commissioning to target commercial production by mid 2014, contingent on the completion of the NTL in May 2014.

2 Introduction and Terms of Reference

This report was reissued on Sept 1, 2015, with corrections for several typographical errors and clarification on some sections. The Certificated of the Authors were also updated with clarifications. Sections with changes are as follows:

This amended report adds some clarity to several sections, as listed below

- Sec 3.) Removed reference to other qualified persons
- Sec 6.) Removed the wording about historic estimates not conforming to 43-101 regulations
- Sec 14.) Clarified QP`s potion on sample validity of pre-Imperial data
- Sec 17.) Included details on pit constrained/block cave constrained estimated resource
- Sec 31.) Clarified reference to budgets for recommendations
- Revised the certificates of QPs responsible for the report to more implicitly follow the wording of 43-101 regulatory act. Paul Sterling. P. Eng. was also added as an Author and QP for section 16. Mr. Sterling authored this section in the original release of this report, but was only listed as a contributor in the original report.

This report was prepared for Imperial Metals Corporation to update the 2011 Technical Report on the Red Chris Copper-Gold Project ("2011 Technical Report”). This report updates all drilling and exploration activities conducted on the property to the end of 2011 and reports a new mineral resource for the property.

A new block model was completed in January of 2012 to include all the new deep drilling conducted in 2010 and 2011. The resource statement presented in this report is based on this 2012 model.

The scope of work for this report includes the following:

- A review of the work done on the Red Chris property previous to the 2007 Imperial Metals Corporation ownership.
- Details of all drilling and exploration conducted from 2007 to the end of 2011.
- Details of the block model completed in January of 2012 and the new mineral resource results. This section was amended and restated in September of 2015 as a Pit constrained/block caved constrained resource.
- Details of the 2010 reserve estimate, mine plan and financial results.
- Recommendations and conclusions.

Table 2.1 List of Standard Abbreviations

Above mean sea level	amsl
Ampere	A
Annum (year)	a
Billion years ago	Ga
Centimetre	cm
Cubic centimetre	cm^3
Cubic feet per second	ft^3/s or cfs
Cubic foot	ft^3
Cubic metre	m^3
Day	d
Days per week	d/wk
Degree	°
Degrees Celsius	°C
Dry metric ton	dmt
Foot	ft
Gallons per minute (US)	gpm
Gram	g
Grams per litre	g/L
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Horsepower	hp
Hour	h (*not* hr)
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per hour	kg/h
Kilograms per square metre	kg/m^2
Kilojoule	kJ
Kilometre	km
Kilometres per hour	km/h
Kilonewton	kN
Kilopascal	kPa
Kilovolt	kV
Kilovolt-ampere	kVA
Kilovolts	kV
Kilowatt	kW
Kilowatt hour	kWh
Kilowatt hours per tonne (metric ton)	kWh/t
Kilowatt hours per year	kWh/a
Less than	<
Litre	L

Litres per minute	L/m
Measured and Indicated	M&I
Megabytes per second	Mb/s
Megapascal	MPa
Megavolt-ampere	MVA
Megawatt	MW
Metre	m
Metres above sea level	masl
Metres per minute	m/min
Metres per second	m/s
Micrometre (micron)	µm
Milliamperes	mA
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million	M
Million tonnes	Mt
Minute (plane angle)	'
Minute (time)	min
Month	mo
Ounce	oz
Parts per billion	ppb
Parts per million	ppm
Percent	%
Percent moisture (relative humidity)	% RH
Phase (electrical)	Ph
Pound(s)	lb
Second (plane angle)	"
Second (time)	s
Specific gravity	SG
Square centimetre	cm^2
Square foot	ft^2
Square kilometre	km^2
Square metre	m^2
Thousand tonnes	kt
Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Volt	V
Week	wk
Wet metric ton	wmt

3 Reliance on Other Experts

The report has been written to conform to the specification outlined in NI 43-101F1, for the Standards of Disclosure for Mineral Projects as required in National Instrument 43-101. This NI 43-101 Technical Report has not relied on any information provided by nonqualified people.

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4 Property Description and Location

4.1 Location and Claim Status

The Red Chris property is located in northwest British Columbia, approximately 18 km southeast of the Iskut village, 80km south of Dease Lake, and 12 km east of the Stewart-Cassiar Highway 37 (see Figure 4.1). The nearest gravel airstrip is located in Iskut. During summer Northern Thunderbird Air regularly services the Dease Lake airport. Helicopter and fix winged charters can also be secured from the Dease Lake airport.

The Red Chris exploration site is accessible year round via a 17 km gravel access trail, which was constructed in 2008. This trail branches off at the 6km marker on the Ealue Lake road, making it a 23 km trip from Red Chris camp to Highway 37 (0 km on the Ealue Lake road). The access trail is predominantly travelled by pick-up trucks, but is also navigated by larger fuel trucks, flatbed trucks and semi-trailers. Access to the property via this trail has significantly reduced exploration expenditures associated with helicopter reliance and has made working around the property safer (see Figure 10.5).

4.2 Claim Information (Mineral Tenure)

The Red Chris Property is comprised of the Red Chris Claims and the Red Claims, which are described below. The Red Chris Deposit which has been approved for development under the British Columbia Environmental Assessment Process is located on the Red Chris Property.

The Red Chris Property consists of 50 mineral tenures containing 528 cells and covering 9,688.98 hectares. Red Chris Development Company Ltd. (“RCDC”) has a 100% interest in the Red Chris Claims and all or a portion of 32 claims are subject to a 1.8% net smelter return royalty (“NSR”) held by Xstrata Canada Corporation (successor to Falconbridge Limited). The 1.8% NSR may be reduced to 1% at any time prior to commencement of commercial production in consideration of the payment to Xstrata of $1,000,000. Four mining lease applications comprised of 22 mineral tenures and covering 3,887.26 hectares have been filed with the Mineral Titles Branch.

The Red claims consist of 18 mineral tenures containing 428 cells and covering 7,305.02 hectares. Imperial owns 100% of the Red claims. One mining lease application comprised of three mineral tenures and covering 1,254.00 hectares has been filed with the Mineral Titles Branch.

Refer to tables 4.1 and 4.2 for a list of the mineral tenures and Figure 4.2 for exact locations of all claims

Figure 4.1 Red Chris Property Map

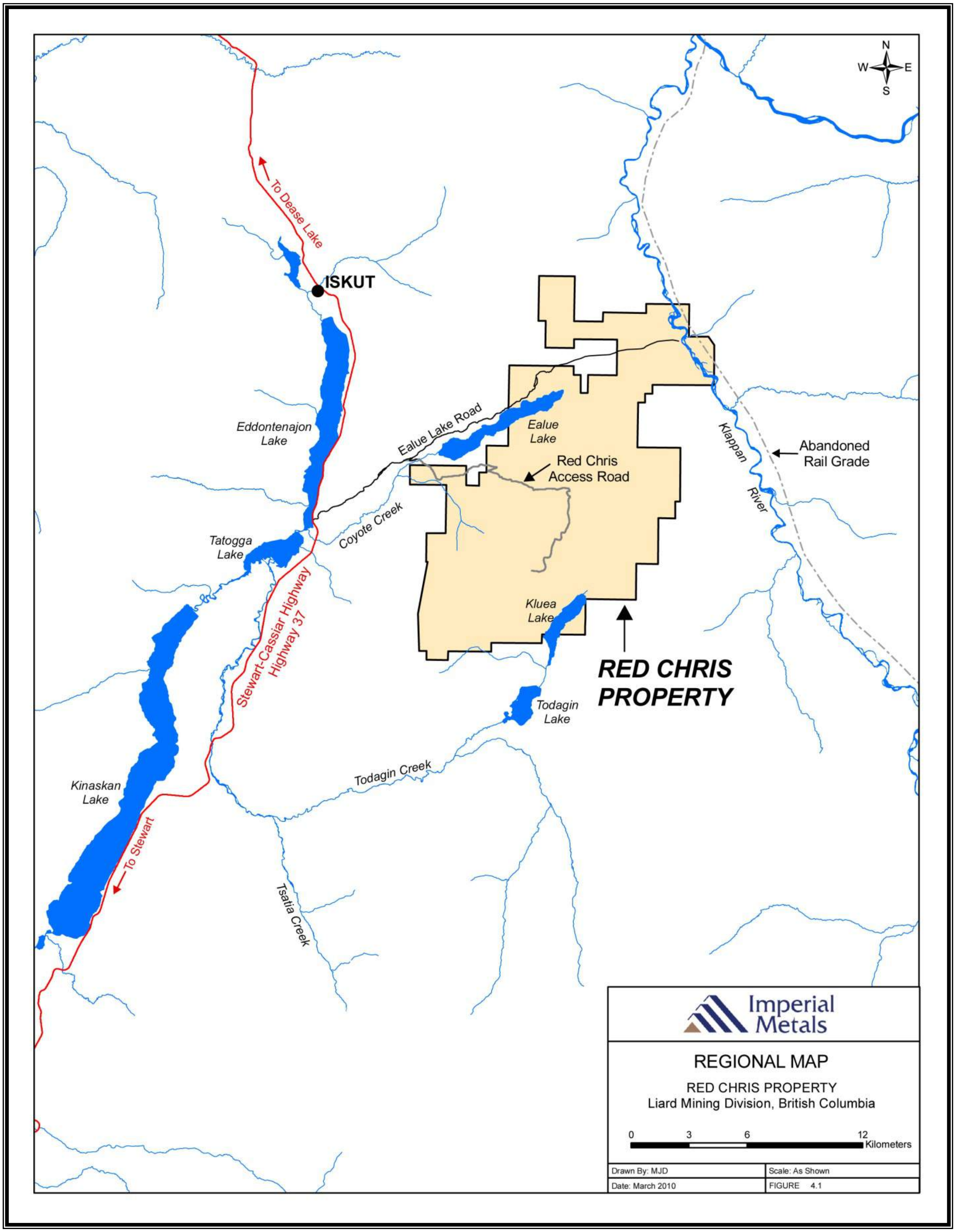


Table 4.1 Red Chris Claims

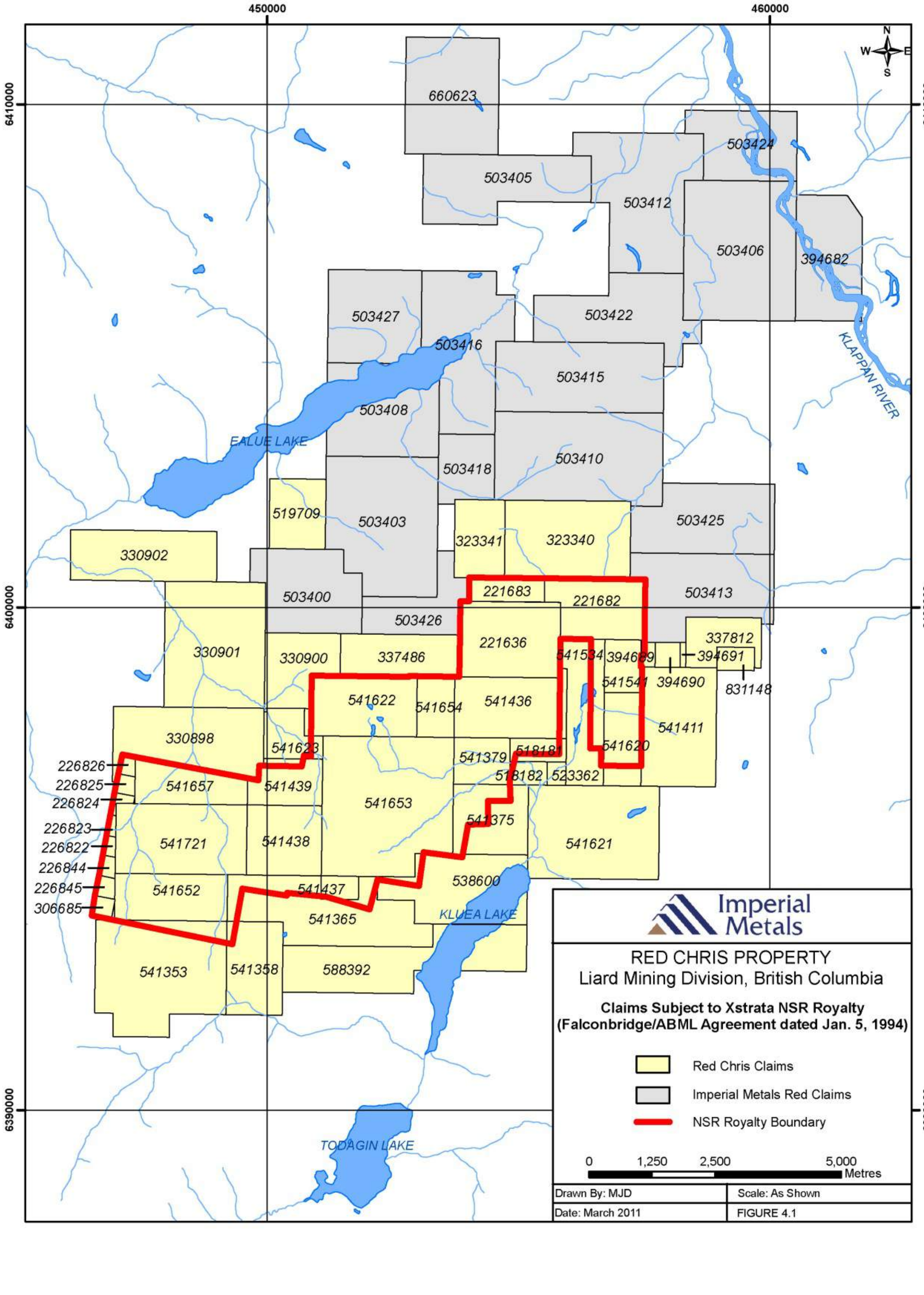
Tenure No.	Claim Name / Plan #	Tenure Sub Type	Issue Date	Good To Date	Lease App. Date	Area (ha)
221636	Plan EPC 564	Claim	1975/jul/15	2019/jan/11	2010/aug/06	300.00
221682	Plan EPC 562	Claim	1976/jul/07	2019/jan/11	2010/aug/31	219.00
221683	Plan EPC 562	Claim	1976/jul/07	2019/jan/11	2010/aug/31	71.00
226822	MONEY #32	Claim	1968/sep/30	2021/oct/31		25.00
226823	MONEY #34	Claim	1968/sep/30	2021/oct/31		25.00
226824	MONEY #36	Claim	1968/sep/30	2021/oct/31		25.00
226825	MONEY #38	Claim	1968/sep/30	2021/oct/31		25.00
226826	MONEY #40	Claim	1968/sep/30	2021/oct/31		25.00
226844	MONEY #59	Claim	1968/sep/30	2021/oct/31		25.00
226845	MONEY #61	Claim	1968/sep/30	2021/oct/31		25.00
306685	MONEY #63	Claim	1968/sep/30	2021/oct/31		25.00
323340	Plan EPC 562	Claim	1994/jan/17	2019/jan/11	2010/aug/31	387.00
323341	RC-5	Claim	1994/jan/16	2021/oct/31		200.00
330898	ABM - 1	Claim	1994/sep/11	2021/oct/31		450.00
330900	ABM - 3	Claim	1994/sep/11	2021/oct/31		225.00
330901	ABM-4	Claim	1994/sep/12	2021/oct/31		500.00
330902	ABM - 5	Claim	1994/sep/13	2021/oct/31		300.00
337486	Plan EPC 561	Claim	1995/jun/29	2019/jan/11	2010/aug/06	198.00
337812	Plan EPC 562	Claim	1995/jul/08	2019/jan/11	2010/aug/31	150.00
394689	Plan EPC 562	Claim	2002/jun/17	2019/jan/11	2010/aug/31	12.00
394690	Plan EPC 562	Claim	2002/jun/17	2019/jan/11	2010/aug/31	24.70
394691	Plan EPC 562	Claim	2002/jun/17	2019/jan/11	2010/aug/31	5.46
518181	Plan EPC 562	Claim	2005/jul/22	2019/jan/11	2010/aug/31	51.80
518182	Plan EPC 562	Claim	2005/jul/22	2019/jan/11	2010/aug/31	34.60
519709	EALUE	Claim	2005/sep/06	2021/oct/31		155.40
523362	Plan EPC 562	Claim	2005/dec/02	2019/jan/11	2010/aug/31	17.30
538600		Claim	2006/aug/03	2021/oct/31		346.00
541353		Claim	2006/sep/15	2021/oct/31		536.52
541358		Claim	2006/sep/15	2021/oct/31		207.69
541365		Claim	2006/sep/15	2021/oct/31		415.25
541375	Plan EPC 562	Claim	2006/sep/15	2019/jan/11	2010/aug/31	207.00
541379	Plan EPC 562	Claim	2006/sep/15	2019/jan/11	2010/aug/31	104.00
541411	Plan EPC 562	Claim	2006/sep/15	2019/jan/11	2010/aug/31	353.00
541436	Plan EPC 564	Claim	2006/sep/15	2019/jan/11	2010/aug/06	273.00
541437		Claim	2006/sep/15	2021/oct/31		34.60
541438		Claim	2006/sep/15	2021/oct/31		207.56

541439		Claim	2006/sep/15	2021/oct/31		138.33
541534	Plan EPC 562	Claim	2006/sep/18	2019/jan/11	2010/aug/31	225.00
541541	Plan EPC 562	Claim	2006/sep/18	2019/jan/11	2010/aug/31	77.30
541620	Plan EPC 562	Claim	2006/sep/19	2019/jan/11	2010/aug/31	138.00
541621		Claim	2006/sep/19	2021/oct/31		484.27
541622	Plan EPC 561	Claim	2006/sep/19	2019/jan/11	2010/aug/06	258.00
541623		Claim	2006/sep/19	2021/oct/31		155.59
541652		Claim	2006/sep/19	2021/oct/31		207.61
541653	DL 7356, EPC98	Claim	2006/sep/19	2019/jan/11	2006/oct/24	691.00
541654	Plan EPC 561	Claim	2006/sep/19	2019/jan/11	2010/aug/06	90.10
541657		Claim	2006/sep/19	2021/oct/31		207.49
541721		Claim	2006/sep/20	2021/oct/31		363.21
588392		Claim	2008/jul/17	2021/oct/31		432.64
831148	RC-10-1	Claim	2010/aug/05	2021/oct/31		34.56
Subtotal	**50 tenures**				**Area (ha)**	**9,688.98**

Table 4.2 Red Claims

Tenure No.	Claim Name / Plan #	Tenure Sub Type	Issue Date	Good To Date	Lease App. Date	Area (ha)
394682	RED 10	Claim	2002/jun/18	2021/oct/31		375.00
503400		Claim	2005/jan/14	2021/oct/31		397.36
503403		Claim	2005/jan/14	2021/oct/31		569.87
503405		Claim	2005/jan/14	2021/oct/31		379.25
503406		Claim	2005/jan/14	2021/oct/31		620.81
503408		Claim	2005/jan/14	2021/oct/31		414.20
503410	Plan EPC 563	Claim	2005/jan/14	2019/jun/10	2010/aug/06	596.00
503412		Claim	2005/jan/14	2021/oct/31		517.19
503413	Plan EPC 563	Claim	2005/jan/14	2019/jun/10	2010/aug/06	339.00
503415		Claim	2005/jan/14	2021/oct/31		465.90
503416		Claim	2005/jan/14	2021/oct/31		465.82
503418		Claim	2005/jan/14	2021/oct/31		155.37
503422		Claim	2005/jan/14	2021/oct/31		379.50
503424		Claim	2005/jan/14	2021/oct/31		275.77
503425	Plan EPC 563	Claim	2005/jan/14	2019/jun/10	2010/aug/06	319.00
503426		Claim	2005/jan/14	2021/oct/31		259.17
503427		Claim	2005/jan/14	2021/oct/31		345.00
660623	LIMY	Claim	2009/oct/27	2021/oct/31		430.79
Subtotal	**18 tenures**				**Area (ha)**	**7,305.02**
TOTAL	**68 Tenures**				**Area (ha)**	**16,994.00**

Figure 4.2 Claim Map



4.3 Permits and Agreements

Red Chris commenced its environmental assessment process by submitting an application in 2004 to the government of British Columbia and that of Canada. The Provincial Environmental Assessment process follows the British Columbia Environmental Assessment Act ("BC EAA") and the Federal Environmental Assessment follows the Canadian Environmental Assessment Act ("CEAA") process.

The Red Chris (BCEAA) Certificate was issued in August 2005. Federal approval for the Red Chris Project under the CEAA was received in May 2006. The federal approval was subsequently challenged by a third party; however a decision by the Supreme Court of Canada on January 21, 2010 upheld the federal approval which has allowed mine permitting and development to proceed. An extension to the BCEAA Certificate was granted in July 2010.

4.3.1 Exploration Permits

Since the acquisition of bcMetals by Imperial Metals Corporation in 2007, work at the Red Chris site (exploration and geotechnical investigations for site and tailings impoundment) has been conducted under mineral exploration permits issued by the Ministry of Energy Mines and Petroleum (MEMPR). In 2008, an access trail to the project site was also constructed under the exploration permit.

The exploration permit was obtained to conduct the following work:

- Exploration drilling as per submitted yearly work program
- Building an exploration trail for safer and environmentally improved access for drilling
- Geotechnical investigation for the plant site and the tailings impoundment areas.

4.3.2 Project Development, Operation and Closure Permits

RCDC submitted its draft Terms of Reference for the Mines' Act Permit Application in October 2009 and met with Northwest Mine Development Review Committee in December 2009 and in March 2010. The Terms of Reference ("TOR") were finalized on March 31, 2010.

The BC Government is in the process of streamlining the overall permitting process and is proceeding with asynchronous permitting approach. The new system will have all permits required from the various ministries of the Province to be synchronized with the Mines Act Permit Application. RCDC submitted its Joint Application for the Mines' Act Permit and Environmental Management Act Permit in July 2010 and a review meeting with the NWMDRC was held in September 2010. In June 2011 RCDC submitted its response to the comments received from all NWMDRC related agencies.

The following permits, approvals, licenses and leases will be required for the Red Chris Mines' development, operation and closure:

BC MEMPR

- Mines' Act Permit
- Permit pursuant to *Mining Right of Way Act*
- Mining Lease

BC MOE

- Effluent Discharge Permit, Refuse, Air, Special Waste, Sewage Registration
- Water License(s)
- Permit pursuant to *Wildlife Act*

MOFR

- Occupant License to Cut
- Special Use Permit (for the access road segment off the mineral claims)
- Burning

ILMB

- License of Occupation (for the power line off the mineral claims)

MOTI

- Permit to Connect to a Public Highway
- Permit to install the power line if located in a road right of way

TCA, Archaeology Branch

- Alteration Permit (Section 12, Heritage Conservation Act) OR Systematic Data Recovery (Section 14, HCA) in some areas

Northern Health Authority

- Permits pursuant to the Health Act, Food Premise Regulation, Industrial Camps Health Regulation
- Permit pursuant to *Drinking Water Protection Act*

Federal Authorizations

- DFO – Federal *Fisheries Act* Authorization
- EC – MMER Schedule 2 Designation for the Tailings Impoundment Area
- NRCan – *Explosives Act* Authorization (for explosives manufacturing)

4.3.3 Agreements

The Red Chris Development Company Ltd. (RCDC) will operate the Red Chris mine.

RCDC was previously a wholly owned subsidiary of bcMetals. Imperial acquired bcMetals in February 2007. bcMetals and RCDC were then merged and continued as RCDC which is now a wholly owned subsidiary of Imperial. RCDC holds a 100% interest in the Red Chris Property as follows:

On July 1, 1974, Silver Standard Mines Limited ("Silver Standard"), until then the sole, beneficial and legal owner, subject to prospector's interests, of certain of the claims now comprising the Red Chris Property (the "Original Claims") entered into an option agreement (the "Red Chris Option Agreement") with Ecstall Mining Limited ("Ecstall"), and Great Plains Development Company of Canada Ltd. ("Great Plains") for the exploration and development of the Original Claims together with certain other claims owned by Great Plains (the "Pooled Claims"). Under the Red Chris Option Agreement, Ecstall acquired the right to a 60% ownership interest in the Original Claims and the Pooled Claims (the "Acquired Claims"), with Silver Standard and Great Plains each holding a 20% interest in the Acquired Claims.

The interests of Ecstall, Silver Standard and Great Plains in the Acquired Claims were ultimately assigned to Falconbridge Limited (now Xstrata Canada Corporation, "Xstrata"), Teck Corporation ("Teck") and Norcen Energy Resources Limited ("Norcen").

In January 1994 ABML purchased a 60% interest in the Acquired Claims from Falconbridge Limited (now Xstrata) and a 20% interest in the Acquired Claims from Norcen. Xstrata retained a 1.8% net smelter return royalty in the Acquired Claims, which can be reduced to 1% in consideration of a $1 million payment prior to commencement of commercial production. The purchase agreement with Norcen provided for the payment of additional consideration to Norcen prior to commencement of commercial production in the form of 150,000 free trading common shares of ABML.

In March 1994, ABML and Teck entered into an Option Joint Venture Agreement governing the exploration and development of the Acquired Claims (the "Teck Agreement"). The Teck Agreement terminated the Red Chris Option Agreement and established the following respective joint venture interests in, and share of expenditures on certain programs ("Cost Share") on, the Red Chris Property (see Table 4.3).

Table 4.3 Agreements 1: ABML and Teck

Party	Interest	Cost Share
ABML	80%	90%
Teck	20%	10%

In addition, ABML granted to Teck, subject to the terms of the Teck Agreement, the exclusive right and option to earn 43.75% of ABML's 80% interest (the "Teck Back-in Right") by proceeding at its sole cost to prepare a final feasibility report on the Red Chris Property and arranging production financing (the "Production Financing") sufficient to place the Red Chris Property into commercial production whereupon the interests and Cost Shares of the parties would be as shown in table 4.4 below.

Table 4.4 Agreements 2: ABML and Teck

Party	Interest	Cost Share
ABML	45%	45%
Teck	55%	55%

The Production Financing arrangement gave Teck the right to charge the entire interest of Teck and ABML together with their respective shares of mineral production as security for repayment by Teck and ABML of the Production Financing, but as concerns ABML, was to be limited in terms of recourse to the 45% interest retained by ABML.

On October 18, 2002 ABML entered into a joint venture agreement with RCDC (the "JV Agreement"), for the development of the Red Chris Property. The JV Agreement resulted in RCDC becoming the owner of a 70% interest and ABML becoming the owner of a 30% reversionary carried ownership interest ("RCOI"), in ABML's 80% interest of the Red Chris Property such that the net interests of ABML, RCDC and Teck in the Red Chris Property prior to exercise of the Teck Back-in Right would be as shown in table 4.5 below.

Table 4.5 Agreements 3: ABML, Teck and RCDC

Party	Interest	Cost Share
ABML	24%	0%
Teck	20%	10%
RCDC	56%	90%

At the time the JV Agreement was entered into, RCDC was a wholly-owned subsidiary of bcMetals Corporation ("bcMetals"). In the event that Teck exercised the Teck Back-in Right the net interests of ABML, RCDC and Teck in the Red Chris Property would be as shown in table 4.6 below.

Table 4.6 Agreements 4: ABML, Teck and RCDC

Party	Interest	Pre-production Cost Share	Post-production Cost Share
ABML	13.5%	0%	13.5%
Teck	55%	100%	55%
RCDC	31.5%	0%	31.5%

The JV Agreement contemplated on November 4, 2003, Teck, bcMetals and RCDC entered into an option agreement (the "bcMetals Option Agreement"), wherein Teck agreed to grant an option to bcMetals to earn all of Teck's ownership, rights and interest in and to the Red Chris Property and the JV Agreement.

In June 2004 ABML, by way of its Receiver-Manager, entered into an agreement with bcMetals to settle claims against ABML. The agreement terminated ABML's right to participate with RCDC in the joint acquisition of Teck's interest in the Red Chris Property. The agreement was validated by the Supreme Court of British Columbia on June 29, 2004 and concluded shortly thereafter.

On December 10, 2004 RCDC completed the purchase of Teck's interest in the Red Chris Property

In the event that RCDC exercises the Back-in Right acquired from Teck the net interests of ABML and RCDC in the Red Chris Property would be as shown in table 4.7 below.

Table 4.7 Agreements 5: ABML and RCDC

Party	Interest	Pre-production Cost Share	Post-production Cost Share
ABML	13.5%	0%	13.5%
RCDC	86.5%	100%	86.5%

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Red Chris exploration site is accessible year round via a 17 km gravel access trail, which was constructed in 2008. This trail branches off at the 6 km marker on the Ealue Lake road, making it a 23 km trip from camp to Highway 37 (0 km on the Ealue Lake road). The access trail is predominantly travelled by pick-up trucks, but can also be navigated by larger fuel trucks, flatbed trucks and semi-trailers. Access to the property via the trail has significantly reduced exploration expenditures associated with helicopter support and has made access to the property safer.

Figure 5.1 Air Photo Showing the Red Chris Camp and Local Terrain

In summer Northern Thunderbird Air regularly services the Dease Lake airport. Aside from the regular flight schedule, helicopter and fix winged charters can be secured from the Dease Lake airport. The Red Chris property is located approximately 18 km southeast of the Iskut gravel airstrip located in the village of Iskut (see Figure 4.1).

The Red Chris property is situated on the eastern portion of the Todagin upland plateau which forms a subdivision of the Klastine Plateau along the northern margin of the Skeena Mountains. Elevations on the plateau are typically 1,500 ± 30 m with relatively flat topography broken by several deep creek gullies (see figure 5.1). Bedrock exposure is confined to the higher-relief drainages and along mountainous ridges. The majority of the property is covered by several metres of glacial till. Vegetation on the plateau consists of scrub birch and willow, grasses, and mosses. Within the creek valleys, are several varieties of conifer and deciduous trees including balsam, fir, cedar, spruce, and aspen. The project area lies in a region of moderate annual precipitation with an average of 530 mm total annual precipitation which is more or less evenly distributed throughout the year, with April to May receiving the least and August to December the most. Temperatures vary from a low of -21^{o} C in January to a high of 9^{o} C in July with temperature extremes ranging from -50^{o} C to 30^{o} C.

5.1 Northwest Transmission Line Project

Development of the Red Chris Mine is contingent upon the availability of electric power by Tatogga on Highway #37. The Provincial and Federal governments have committed to provide funding for a powerline from Terrace to Bob Quinn (Northwest Transmission Line ("NTL") Project), for which the provincial and federal environmental assessment process has been completed and the project is approved. BC Hydro has retained a Design –Build Contractor for the NTL Project. As of January 2012 the construction has commenced for the NTL project. Asuming that the NTL project is constructed, RCDC will be responsible for extending powerline service sufficient to meet its needs from Bob Quinn to Tatogga and from there to the Red Chris Mine. Target completion date for the NTL Project is May 2014.

The powerline extension along Highway 37 requires approval through an amendment of the Red Chris EA Certificate and will also require a permit from the BC Ministry of Transportation and Infrastructure (MOTI) to allow development within the Highway 37 Right-of-Way (see figure 1.2 and 4.1). RCDC has applied for an amendment to its EA Certificate to allow RCDC to build the required powerline extension from Bob Quinn and this application is in review.

6 History

Below is a cumulative history of all the exploration that has occurred on the Red Chris Property to 2009.

Table 6.1 Red Chris Cumulative Exploration Summary

Owner	Geochemical Samples	Geophysical Samples	Drilling/Trenching	References/ARIS #
Conwest Exploration Ltd.			X-Ray drilling program not defined	B.C.M.M. Annual Report, 1956.
Great Plains Development Company of Canada Ltd.	Survey not well defined, roughly 534 B-horizon soil, and 8 rock samples			Reynolds (1969) / 02164&02165
Great Plains Development Company of Canada Ltd.			2 DDH (309m), 70-1 to 70-2; trenching program not defined	Referenced in Giroux et al. 2002, Work conducted in 1970
Silver Standard Mines Ltd.	B-horizon samples, survey not defined		Trenching (457m)	Referenced in Giroux et al. 2002, Work conducted in 1971
Great Plains Development Company of Canada Ltd.		12km IP survey	8 DDH (922m), 72-1 to 72-8	Referenced in Panteleyev 1973, Work conducted in 1972
Texasgulf Canada Ltd.	Overburden program not defined	IP survey not defined, 6km proton magnetometer survey	67 DDH (12,284m) 23/49-67 core logs; 44 PDH (3,173m), holes 1-44; trenching (558m)	Leitch, Phil and Newell (1976) / 06111, Work conducted further referenced in Forsythe (1977) / 06489. Cumulative work from 1973-1976
Texasgulf Canada Ltd.	153 overburden samples	20km IP survey		Forsythe (1977) / 06489
Texasgulf Canada Ltd.			5 DDH (391m), 68-72 core logs	Newell (1978) / 06872
Texasgulf Canada Ltd.			2 DDH (626m), 73-74 core logs	Peatfield (1980) / 08994
Dryden Resource Corp.	92 B-horizon soil, 78 silt and 24 rock samples			Mehner (1991) / 21204
Dryden Resource Corp.	228 B-horizon soil, 26 silt and 5 rock samples			Mehner (1991) / 21957
Dryden Resource Corp.	170 B-horizon soil, 12 silt, and 15 rock samples		Minor hand trenching program not defined	Tupper (1993) / 22909
American Bullion Minerals Ltd.			13 DDH (4,562m), 75-87 core logs	Roberts (1994) / 23534
American Bullion Minerals Ltd.	547 B-horizon soil samples	74km ground mag., 72km IP and 26km EM	45 DDH (16,855m) 88-99 core logs	Blanchflower (1995) / 23834

American Bullion Minerals Ltd.	290 A, B or C soil and 5 rock samples		112 DDH (36,830m), 3 (59m) geotechnical holes	Blanchflower (1996) / 24453
BC Metals			49 DDH (16,591m), 24 geotechnical test pits Appendix D	Bellamy (2004) / 27479
BC Metals		4.6km seismic and 6.5km EM survey	25 DDH (6,927m), 296-320 holes	Referenced in Ferreira 2008 / 29900. Work conducted in 2004.Hillmand and Yarham 2004 (Geophysics)
BC Metals			14 DDH (4,679m), 321-334 holes	Referenced in Ferreira 2008 / 29900. Work conducted in 2006.
Imperial Metals Corp.		Proton magnetometer survey over deposit	6 DDH (4,835m), see report 335-340 core logs	Ferreria (2008) / 29900
Imperial Metals Corp.	11 rock samples		3 DDH (2,220m) see report 341-343 core logs	Ferreira (2009) / Not Released, see internal report
Imperial Metals Corp.	83 rock samples, 23 mot mass samples, 491 ICP composite samples (comprised of individual 2,500 samples)	30km Titan IP survey; 1,295 km Aeroquest airborne magnetic survey; extensive proton magnetometer surveys, 19,434 m acoutstic televiewer survey over 61 holes	76 Exploration DDH (87,112m), 235 Bobcat DDH (1,580m), 84 Geotech DDH (3,097m), 23 condemnation DDH (5,343m), 5 Water Wells (328m), 80 test pits	Gillstrom and Robertson (2010), Gillstrom, Anand, and Robertson (2011) / SEDAR filings

6.1 Exploration History

This section has been largely taken from the Giroux et al, 2002 and 2004 reports on the Red Chris Project.

The first recorded exploration of the project area occurred in 1956 when Conwest Exploration Limited staked the Windy claims to cover prominent limonitic gossans on the Todagin Plateau. The showings reported (B.C.M.M. Annual Report, 1956) consisted of a large oxidized area with small amounts of azurite and malachite. Work consisted of a limited amount of open-cutting and pack-sack X-Ray drilling.

In September 1968, Great Plains Development Co. of Canada staked the Chris and Money claims to cover the headwaters of a stream in the western portion of the present project area, based on a strong copper anomaly in stream sediments. Over the next 2 years Great Plains conducted geological (8 rock samples) and geochemical (534 B-horizon samples) surveys followed by two diamond drill holes in 1970 totalling 309 m. One of the holes (70-2) intersected 0.25 % Cu over 73 metres. During the next two years, additional surveys were completed including geologic mapping, ground magnetic and induced polarization surveys, followed by the drilling of eight diamond drill holes in 1972, totalling 922 m. These holes intersected weak pervasive (hypogene) alteration controlled by fracturing with low supergene copper mineralization near surface (Panteleyev, 1973).

In 1970, Silver Standard Mines Ltd. staked the Red and Sus claims to the north and east of the Chris claim group. In 1971, Silver Standard conducted geologic mapping and soil geochemical surveys over the claims and tested anomalies with bulldozer trenches (457m) near the common boundary between the Red and Chris claims. Two trenches exposed low-grade copper mineralization in intrusive rocks. Ecstall Mining Limited (which later became Texasgulf Canada Limited, the Canadian subsidiary of Texasgulf Inc.), optioned the Silver Standard claims in 1973 and drilled 14 percussion holes totalling 914 m, of which half intersected low grade copper mineralization.

In 1974, Texasgulf Canada Ltd. formed an agreement with Silver Standard and Great Plains to acquire an option on 60 per cent of the combined Red and Chris groups of claims and paying 80% of costs with Silver Standard and Great Plains both retaining 20 per cent.

During the years from 1974 to 1976, Texasgulf drilled a total of 67 diamond drill holes (12,284 m) and 30 percussion holes (2,261 m). During the 1978 and 1980 field seasons, Texasgulf drilled an additional 7 shallow core holes totalling 1,017 m to test for near-surface copper-gold mineralization. (Newell and Peatfield, 1995). Property-wide geological, geochemical (153 overburden samples), and geophysical surveys (20km IP) were also completed during this time. An overburden drill was utilized to test bedrock geochemistry in poorly exposed areas of the property. The results of this program outlined an area 3.4 km long, striking east-northeast, with multiple anomalies greater than 500 ppm copper. This anomalous copper zone effectively outlines the limits of the Red intrusive stock. Magnetometer surveys delineated the northern intrusive contact of the Red Stock with volcanics but could not discriminate between the various intrusive lithologies or the Bowser Lake Group of clastics to the south.

As a result of the Texasgulf exploration, two coalescing east-north-easterly trending zones of copper-gold mineralization named the Main and East zones were outlined. The mineralization was described as pyrite, chalcopyrite, and lesser bornite occurring spatially with zones or quartz vein stockwork near the centre of the Red intrusive stock. The estimated resource in 1976 at a 0.25% Cu cut-off was 34.4 million tonnes with an average grade of 0.51% Cu and 0.27 g/t Au to a depth of 270 m in the Main Zone and 6.6 million tonnes with average grade of 0.83% Cu and 0.72 g/t Au to a depth of 150 m in the East Zone (Newell and Peatfield, 1995).

No exploration was done on the property in the period 1981 to 1994. A series of corporate takeovers and reorganizations in January, 1994 resulted in the ownership of the property divided amongst Falconbridge (60%), Norcen Energy (20%), and Teck Corporation (20%). American Bullion Minerals Ltd. acquired an 80% interest in the property in early 1994 with Teck Corporation retaining the remaining 20%. American Bullion retained Mark Rebagliati to review and evaluate the exploration completed by previous owners. Rebagliati estimated a possible resource at a 0.20 % Cu cut-off of 136 million tonnes averaging 0.38 % Cu and 0.25 g Au/t. He estimated a higher grade core containing 37 million tonnes averaging 0.67% Cu and 0.45 g Au/t. Rebagliati recommended 15,000 m of diamond drilling to upgrade and expand the higher grade core zones and explore the remainder of the property (Rebagliati, 1994).

During the 1994 field season, American Bullion completed mineral claim staking, land surveying, line cutting, soil geochemistry (547 B-horizon), geophysics (including 74km ground magnetic, 26km V.L.F. EM, and 72km induced polarization surveys), camp and core logging facility construction, HQ and NQ diamond drilling totalling 21,417 m in 58 holes, core sample assaying, acid base accounting studies, base-line environmental studies, a mineral resource estimate, petrographic and metallurgical studies, and documentation. The programs were completed between June and November, 1994 at a cost of CAN $4.2 million.

Drilling completed in 1994 extended the lateral dimensions for mineralization in a north-south direction and extended the known copper-gold mineralization over vertical distances of up to 400 m. Geochemical and geophysical surveys extended the mineralization to the west to include the 600 by 600 m Far West zone and the 700 by 400 m Gully zone.

Based on the additional 1994 drill data the measured + indicated resource was estimated at 181 million tonnes averaging 0.4% Cu and 0.31 g Au/t at a 0.2% Cu cut-off (Giroux, 1995). In this report, terms of proven, probable, and possible were used that under 43-101 Guidelines would conform to Measured, Indicated, and Inferred. An additional 139 million tonnes averaging 0.35% Cu and 0.28 g Au/t at the 0.2% Cu cut-off was classed as inferred. This resource, estimated by ordinary kriging of 30 x 30 x 15 m blocks, was compiled and estimated within a 1,300 x 200 m area to depths of between 1,050 to 1,530 m A.M.S.L.

The 1995 exploration program (112 holes totalling 36,770m) successfully increased the geological resources of the Red Chris deposit across the width of the Red stock and over a 400-metre strike length west of the known mineralization. Significant near surface copper-gold mineralization was also discovered at the Gully and Far West zones. As of November, 1995, the property had been tested by a total of 244 diamond and 44 percussion drill holes, or 74,782 metres of drilling. Drill program was supplemented by a 290 sample geochemical survey.

In 2003, bcMetals conducted an infill drilling program totalling 16,591m in 49 drillholes. This resulted in updated measured, indicated, and inferred resourced calculations which were released in the NI 43-101 Update Report dated February 16, 2004.

The infill drill program completed in 2004 consisted of a total of 6,927 m in 25 diamond drill holes. Of these holes 10 targeted the Main Zone, 4 targeted the saddle zone between the Main and East zones, 6 tested the East zone and 5 condemnation holes were drilled to the north east of the East Zone (RCDC Technical Report, 2007). This resulted in a reinterpretation of the geologic model upon which the resource estimation was based. As a result, the mineralized unit was re-modeled as a single unit, whereas prior to 2004, the Main Zone and East Zone had been separated, with inner and outer mineralized shells. Drill program was supplemented by a 4.6km seismic and 6.5km EM geotechnical program designed to further investigate the tailing impoundment area.

Exploration in 2006 consisted of 14 drillholes for a total length of 4679m. This consisted of 7 holes in the Gully Zone and 2 geotechnical holes 300m to 600m northeast of the pit limit, in the vicinity of the then-proposed mill site. In addition, 5 holes were drilled within the East and Main zones for due diligence and verification purposes under the terms of a joint venture agreement between bcMetals and the Global International Jiangxi Copper Company Ltd, which had recently been announced for the development of Red Chris.

On September 8, 2006 Imperial's subsidiary CAT-Gold launched an all cash takeover bid of bcMetals Corporation at $0.95 per share. bcMetals responded by adopting a poison pill which limited potential ownership of the company to 20%. Upon termination of the initial takeover bid on November 8, 2006, Imperial owned approximately 17% of bcMetals. On November 23, 2006 Taseko Mines Limited made an offer to purchase all outstanding shares of bcMetals, to which Imperial responded with a friendly offer of $1.10/share, representing a 4.8% premium over Taseko's offer. A bidding war ensued, which Imperial eventually won with a final bid of $1.70/share submitted on February 2, 2007 for total cost of $68.4 million.

Imperial conducted a deep drilling program in 2007 which was successful in revealing the potential for high grades beneath the current pit design. Exploration tested the vertical continuity of mineralization in the East and Main zones with encouraging results from six deep holes. The mineralized intersection in hole RC07-335, the longest in the Imperial's history, graded 1.01% copper, 1.26 g/t gold and 3.92 g/t silver over 1,024.1 metres, and bottomed in strong mineralization. The mineralization in RC07-335, drilled in the core of the East Zone, is continuous from bedrock surface and extends at least 679 vertical meters below the currently designed open pit. In general, the horizontal area of the high grade mineralization in the East Zone and the gold to copper ratio of the East Zone both appear to increase with depth.

In 2008 Imperial installed a 30-meter free span bridge over Coyote Creek. This was followed by the construction of a 17-kilometer road allowing vehicles and equipment access to the Red Chris camp, eliminating the need for helicopter support, providing for safer working conditions, an extended working season and lower exploration costs. Following road completion in late 2008, a deep drilling program was initiated to follow up on the 2007 drill results, designed to explore the depth of mineralization in the East Zone. Twelve holes were planned to test down to 1,500 metres below surface, although by year end only three holes totaling 2,220 metres were drilled due to adverse ground conditions and other drilling difficulties. Only one hole (RC09-343) was successful in penetrating into the target area below previously known mineralization, and was collared approximately 165 metres northwest of RC07-335 in an area known to be barren near surface. Drill hole RC08-343 intersected the mineralized zone at 840.3 metres, returning 362.2 metres grading 0.40% copper, 0.53 g/t gold and 1.27 g/t silver, including a 97.5 metre interval of 0.63% copper, 0.96 g/t gold and 1.89 g/t silver. A snapped drill rod prevented the hole from reaching the target depth of 1,500 metres (1,273.2 metre final depth).

In 2009, Imperial implemented a comprehensive exploration program including drilling, geophysics, and geological mapping. Nine diamond drillholes were completed (11,528 metres) in or proximal to the East Zone to test the depth extent of mineralization, with most holes reaching or approaching their 1,500 metre target depth. The results showed the continuation of copper-gold mineralization beneath the projected open pit, and in some cases significantly exceeding the average grade of the shallow deposit, most notably in hole RC09-350 which intersected 325.0 metres of 2.24% copper, 4.52 g/t gold and 5.28 g/t silver starting at 390.0 metres, including a sub-interval of 152.5 metres grading 4.12% copper, 8.83 g/t gold and 10.45 g/t silver, starting at 540.0 metres.

A Titan-24 deep imaging IP-MT geophysical survey was done in July 2009, consisting of thirteen NNW-oriented lines perpendicular to the structural grain of the Red Stock, and extending into country rocks or structurally overlapping rock units, and resulting in high quality resistivity and chargeability imaging of the subsurface. A property wide aeromagnetic survey was done in the fall, and field crews ran extensive proton ground magnetometer surveys over the Titan grid and throughout the Todagain Plateau. Geological mapping and prospecting led to some important map revisions, as did a program of low-impact overburden drilling, which completed 235 short holes on the Todagin plateau. The resulting geological, geochemical, and mineralogical sample data was used for research into the hydrothermal alteration zoning, and modeling of the deposit.

7 Geological Setting

7.1 Introduction

Information on the regional geological setting of Red Chris is available in provincial and federal geological survey reports and open files, or references therein, and other government database resources. Such data was used for the regional geology perspective, and to guide geological mapping and stratigraphic interpretations on the Red Chris property. The most recent published work relevant to Red Chris is by Ash and others who produced 1:50,000 scale maps of the geology and mineral occurrences in the Tatogga Lake area for the British Columbia Geological Survey, along with accompanying reports (Ash *et al*., 1995, 1996, 1997a,b). Geological Survey of Canada open files by Read (1984, 1990) are also relevant to the area. A comprehensive report on the northern Bowser Basin by Evenchick and Thorkelson (2005) is the best recent reference for this element of the regional geological framework.

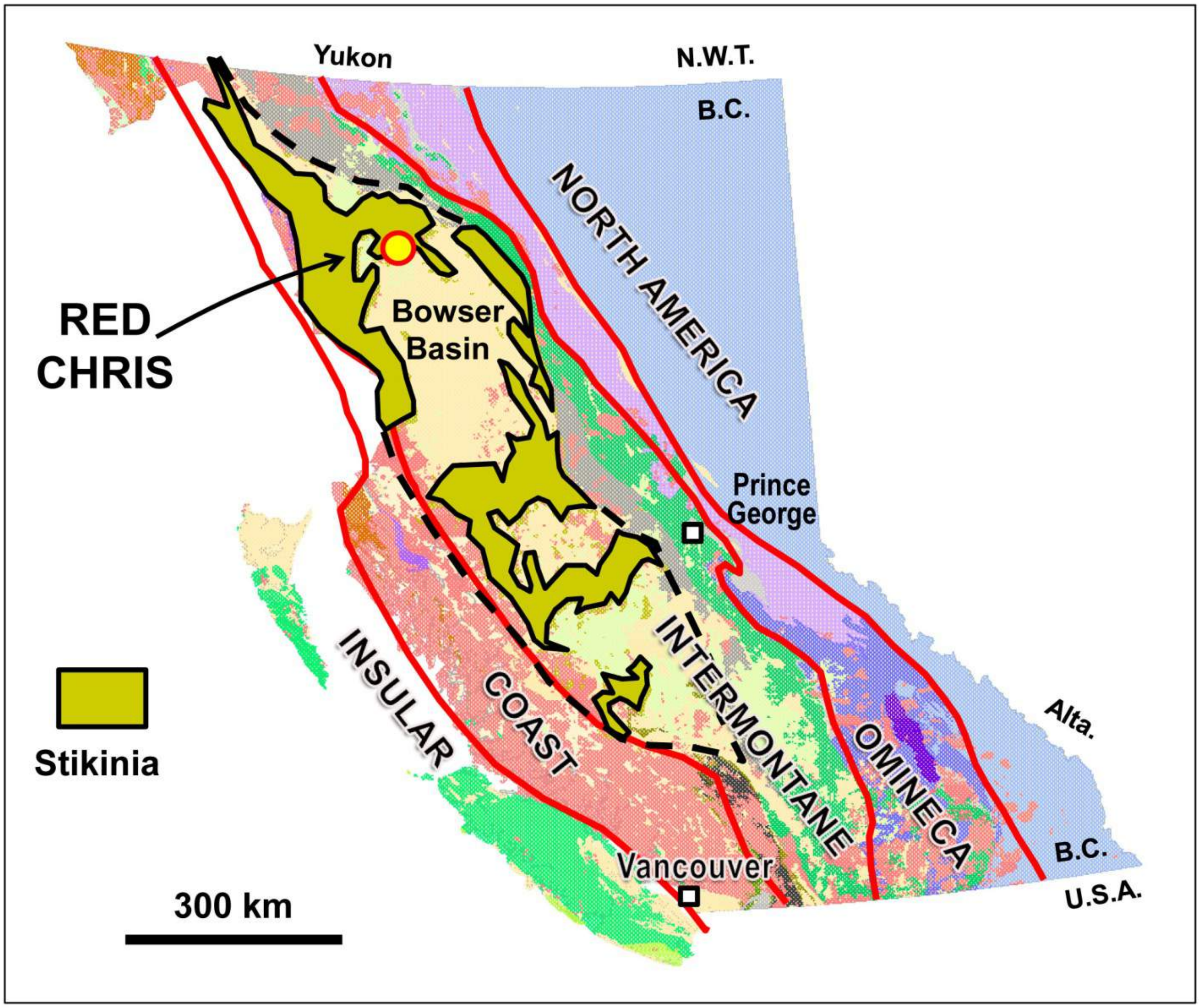
A report on the Red Chris deposit by Giroux *et al*. (2002) should be acknowledged for laying the groundwork for current geological interpretations. Other notable work on Red Chris includes publications by Newell and Peatfield (1995) and Baker *et al*. (1997). Descriptions and interpretations on Red Chris geology, alteration and mineralization in Sections 7, 8 and 9 are derived from internal company reports generated by consultants Brock Riedell and John Proffett, with contributions from Imperial's exploration staff.

7.2 Tectonic Setting

Red Chris is situated in northern British Columbia within the accreted terrane of Stikinia (Fig. 7.1). This terrane forms a broad northwesterly trending area in the centre of the Canadian Cordillera from southern British Columbia into southwestern Yukon, and forms a major part of the 'Intermontane Belt'. Stikinia is dominated by early Mesozoic island arc volcanic strata and related intrusions, overlying a basement of Late Paleozoic metasedimentary and metavolcanic rocks known as Stikine Assemblage. In northern B.C., the Mesozoic arc rocks are the Late Triassic Stuhini Group, and the Early to Middle Jurassic Hazelton Group. Both the Stuhini and Hazelton assemblages formed in oceanic arcs outboard of the North American paleocontinental margin (now represented by the Omineca Belt), in response to east-directed, and possibly west-directed, subduction. Hazelton arc(s) construction included an episode of interarc extension and subsidence related to the migration and docking of the Stikinia microplate against the North American margin in the Early to Middle Jurassic.

The Stuhini Group consists of submarine basaltic to andesitic volcanics and volcaniclastics, and related sedimentary rocks. The Hazelton Group is a diverse assemblage of bimodal, basaltic to rhyolitic subaerial and submarine volcanic rocks and related sediments, and may be a composite of two subparallel arcs and an intervening rift-basin(s) (Evenchick and Thorkelson, 2005). There was a deformation event in the latest Triassic to earliest Jurassic. Regionally, both the Stuhini and Hazelton groups host significant mineral deposits related to Late Triassic and/or Early to Middle Jurassic intermediate intrusions, or to volcanogenic hydrothermal activity.

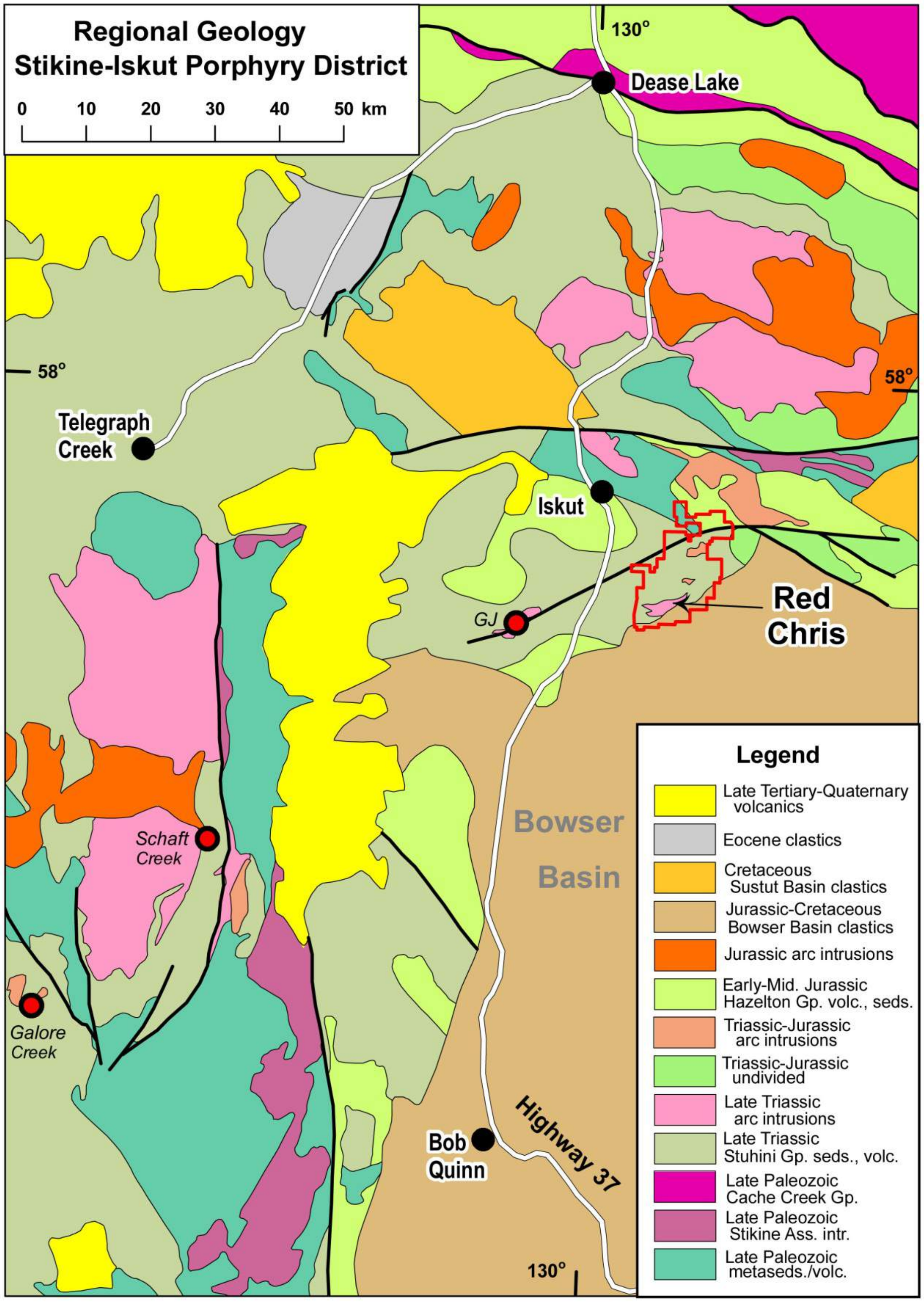
Figure 7.1 Tectonic Setting for the Red Chris Deposit



Stikinia's accretion involved the inboard trapping of ocean floor rocks represented by the Cache Creek terrane, now interposed between Stikinia and the Omineca Belt. Ensuing crustal shortening thrust the Cache Creek terrane southwestwards onto Stikinia, producing a marine to non-marine flexural basin, the Bowser Basin, on its southwestern front into which chert clast-rich sediments (derived from the Cache Creek Group) were deposited, accumulating as the Middle Jurassic to Early Cretaceous Bowser Lake Group. All units were affected by Early to Late Cretaceous tectonism of the Skeena Fold Belt, although this is best displayed in fold and thrust structures in the well bedded strata of the Bowser Basin (see Evenchick and Thorkelson, 2005).

Red Chris is in a Late Triassic stock intruding the Stuhini Group, just beyond the northern edge of the Bowser Basin (Fig. 7.2).

Figure 7.2 Stikine-Iskut Porphyry District in Northwestern British Columbia



7.3 Property Geology

The southern half of the Red Chris property (Fig. 7.3) lies on a broad physiographic upland called the Todagin plateau. This is underlain mainly by the Stuhini Group, and numerous Late Triassic to Early Jurassic intrusions. The southern edge of the plateau in the Red Chris area is marked by a ridge composed of Middle Jurassic, upper Hazelton Group and Bowser Lake Group sedimentary rocks, which were originally deposited unconformably on the Red stock and the Stuhini Group after Early Jurassic uplift and erosion. In places the northern edge of the unconformity is truncated by a high-angle fault.

The northern half of the property is largely in topographically lower ground of the Ealue Lake valley and the plain containing the Klappan River, where the geology is less well known due to limited rock exposure. At higher elevations, a mountain east of Ealue Lake consists of Stuhini Group and an Early Jurassic intrusion (Ealue Stock). Slopes to the north and northeast of the lake are underlain by Stuhini/Hazelton rocks and Stikine Assemblage, respectively. A swarm of Early Jurassic granitic dikes north of the lake trends NW-SE. A northeast-trending fault is inferred to follow the trace of Coyote Creek and the Ealue Lake valley. It continues to the east for an additional 30 kilometres where it has been designated the McEwan Creek Fault with a south side-down movement sense.

Intrusive rocks on the Red Chris property range from large stocks to dikes a few metres in thickness. They range in age from Late Triassic to Early Jurassic, and are typically plagioclase and hornblende-phyric monzodiorite in composition. The largest intrusion is the Red Stock, which hosts the Red Chris deposit, and is described in more detail under 'Local Geology', below. Friedman and Ash (1997) reported that four zircon fractions from drill core have been dated at 203.8 ±1.3 Ma by U-Pb on zircon, which is taken as Late Triassic (*i.e.* assuming the boundary with the Jurassic is *ca*. 200 Ma). A smaller stock, 4 km to the northeast, is lithologically very similar but somewhat younger, at 197.9 Ma (U-Pb, zircon). Dikes belonging to this suite in the area generally trend between ENE and WNW, and dip steeply. In the southwestern corner of the property is an unnamed unit of green to maroon andesitic volcanic and volcaniclastic rocks of uncertain Late Triassic-Early Jurassic age; the rocks overlie the Stuhini Group, and may be the volcanic equivalents of the Triassic-Jurassic intrusive suite.

Bedding attitudes in the Stuhini Group on the property are typically moderate to steep, dipping and younging to the east or northeast. This is believed to be due to Late Triassic deformation which pre-dated intrusion of the Red stock and related dikes, mainly because the intrusions are relatively upright and less deformed than their Stuhini host rocks. Unconformably overlying Jurassic strata of the upper Hazelton Group and the Bowser Lake Group have gentle dips between southeast and southwest. Metamorphic grade in the Stuhini is very low (subgreenschist) although there is hornfelsing around intrusions locally. The Stikine Assemblage is in greenschist facies due to deformation and regional metamorphism in the Late Paleozoic.

Figure 7.3 Red Chris Property Geology Map

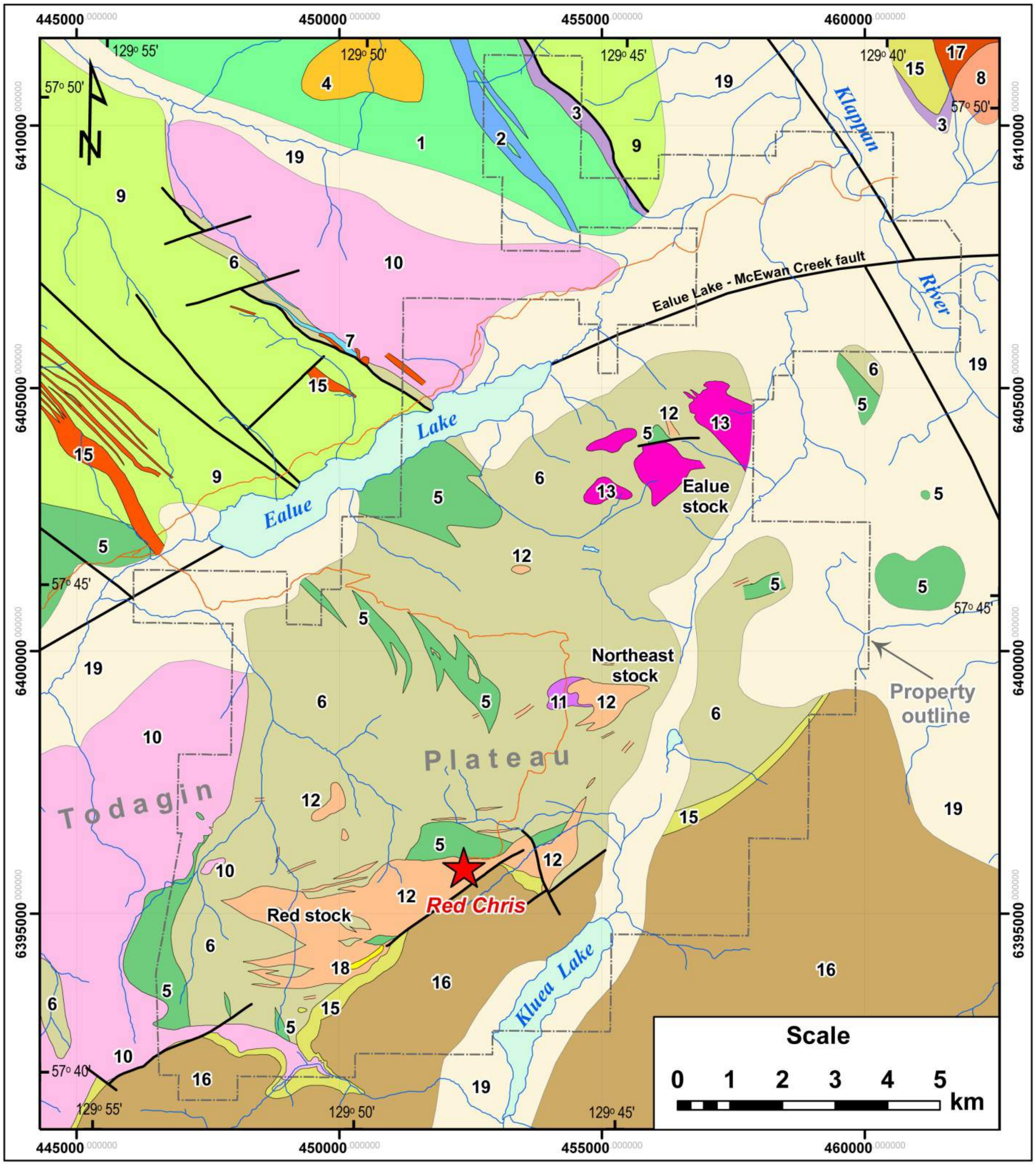


Figure 7.3 (continued) Legend for the Red Chris Geological Map

STRATIFIED ROCKS

QUATERNARY

19 *Glacial and post-glacial sediments*

MIOCENE-PLIOCENE

Maitland Volcanics

18 *Olivine basalt*

MIDDLE JURASSIC

Bowser Lake Group

16 *Sandstone, siltstone, conglomerate*

EARLY TO MIDDLE JURASSIIC

Hazelton Group

14 *Siltstone, sandstone, minor limestone*

LATE TRIASSIC - EARLY JURASSIC

10 *Andesitic volcanics, volcaniclastics*

9 *Basaltic to andesitic volcanics, sediments, volcaniclastics*

LATE TRIASSIC

Stuhini Group

7 *Limestone*

6 *Sandstone, siltstone*

5 *Basaltic-andesitic volcanics, volcaniclastics*

LATE PALEOZOIC

Stikine Assemblage

3 *Mafic volcanics, greenstone*

2 *Limestone, dolostone*

1 *Metavolcanics, metasediments*

INTRUSIVE ROCKS

JURASSIC - TERTIARY

17 *Leucogranite, quartz monzonite*

EARLY JURASSIIC

15 *Alkali granite*

LATE TRIASSIC - EARLY JURASSIC

13 *Equigranular diorite, monzodiorite*

12 *Porphyritic diorite, monzodiorite, monzonite*

11 *Diorite*

LATE TRIASSIC

8 *Monzodiorite*

EARLY MISSISSIPPIAN

4 *Granodiorite, diorite*

SYMBOLS

Contact

Fault

Gravel road/trail

7.4 Local Geology

7.4.1 Red Stock setting

The Late Triassic Red Stock (Fig. 7.4), which hosts the Red Chris deposit, is an ENE-elongate intrusion about 6.5 kilometres long by up to 1.5 km wide at surface. The intrusion tapers to fingers of dikes in the northeast and in the far west. The stock's northern contact is steep, generally dipping around 80° to the north (Fig. 7.5), and may be faulted in part. Basaltic volcanics and volcaniclastics are the predominant Stuhini Group country rocks along the northern margin in the centre of the stock. Elsewhere along the stock's margin, it is in contact with Stuhini Group feldspathic sandstone and siltstone. Drilling and mapping in these marginal rocks reveals recrystallization to siliceous and quartz-biotite-K-feldspar hornfels due to contact metamorphism, and zones of hydrothermal microbreccia locally.

Figure 7.4 Red Stock and Red Chris Mineral Zones, with Surrounding Geology

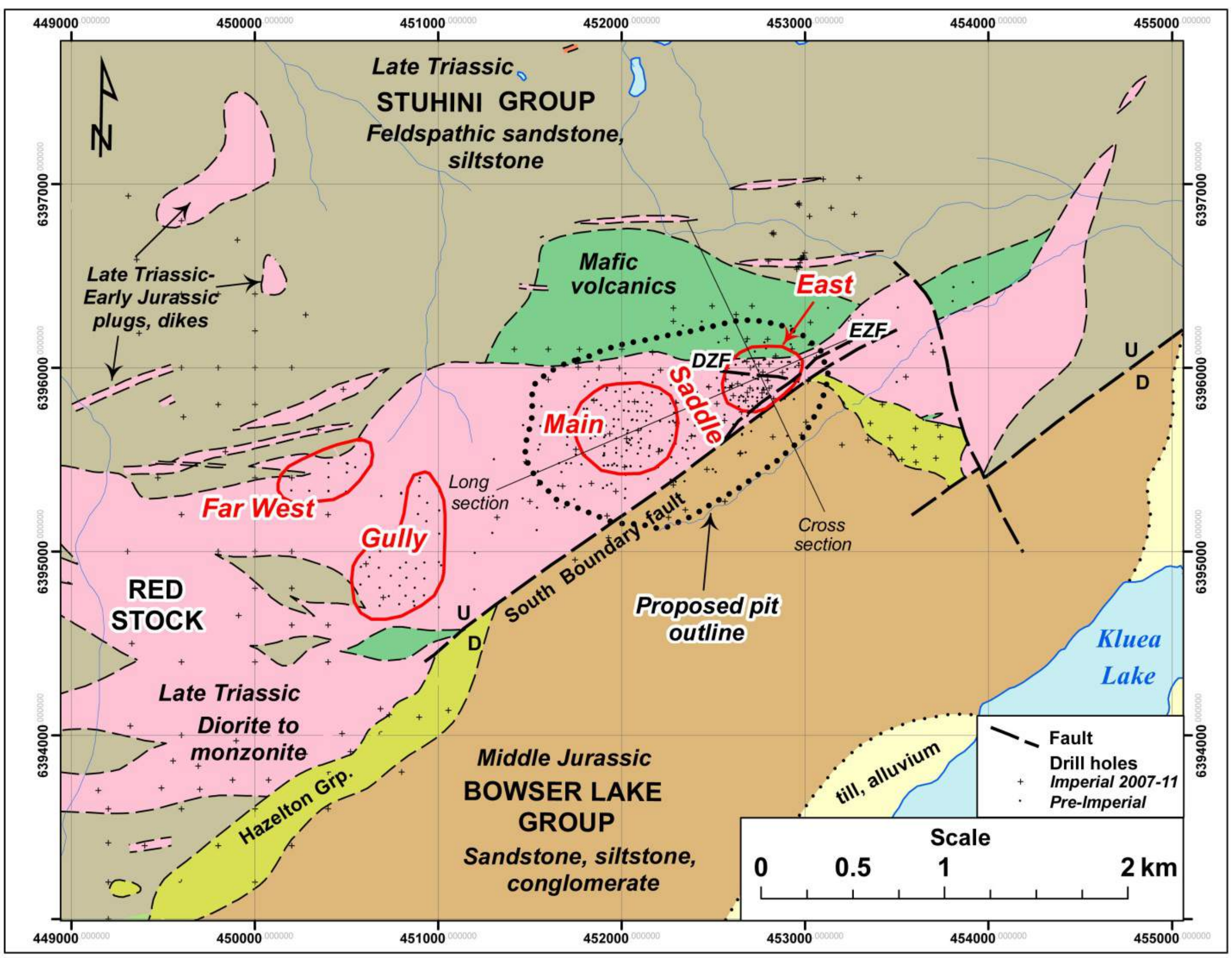
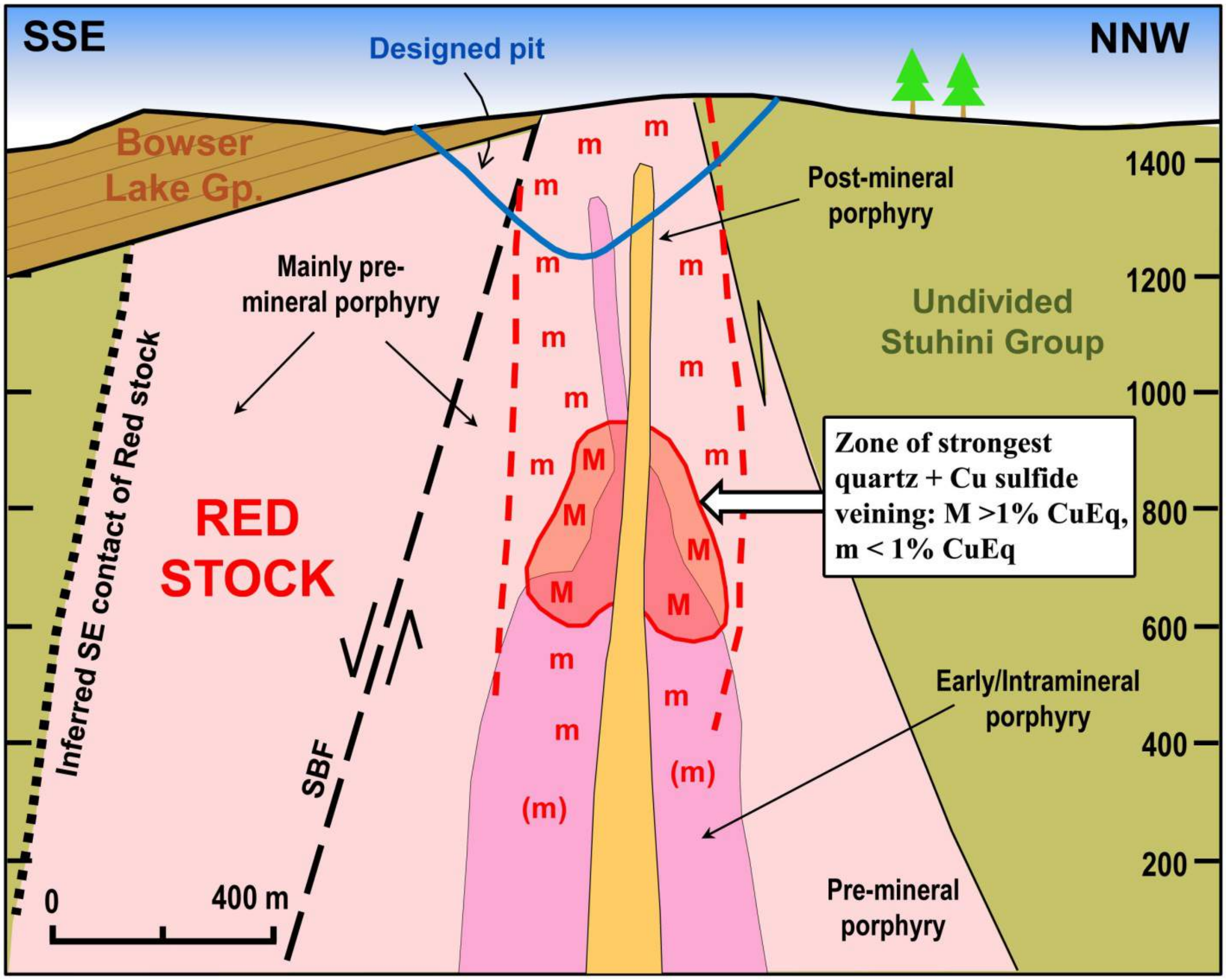


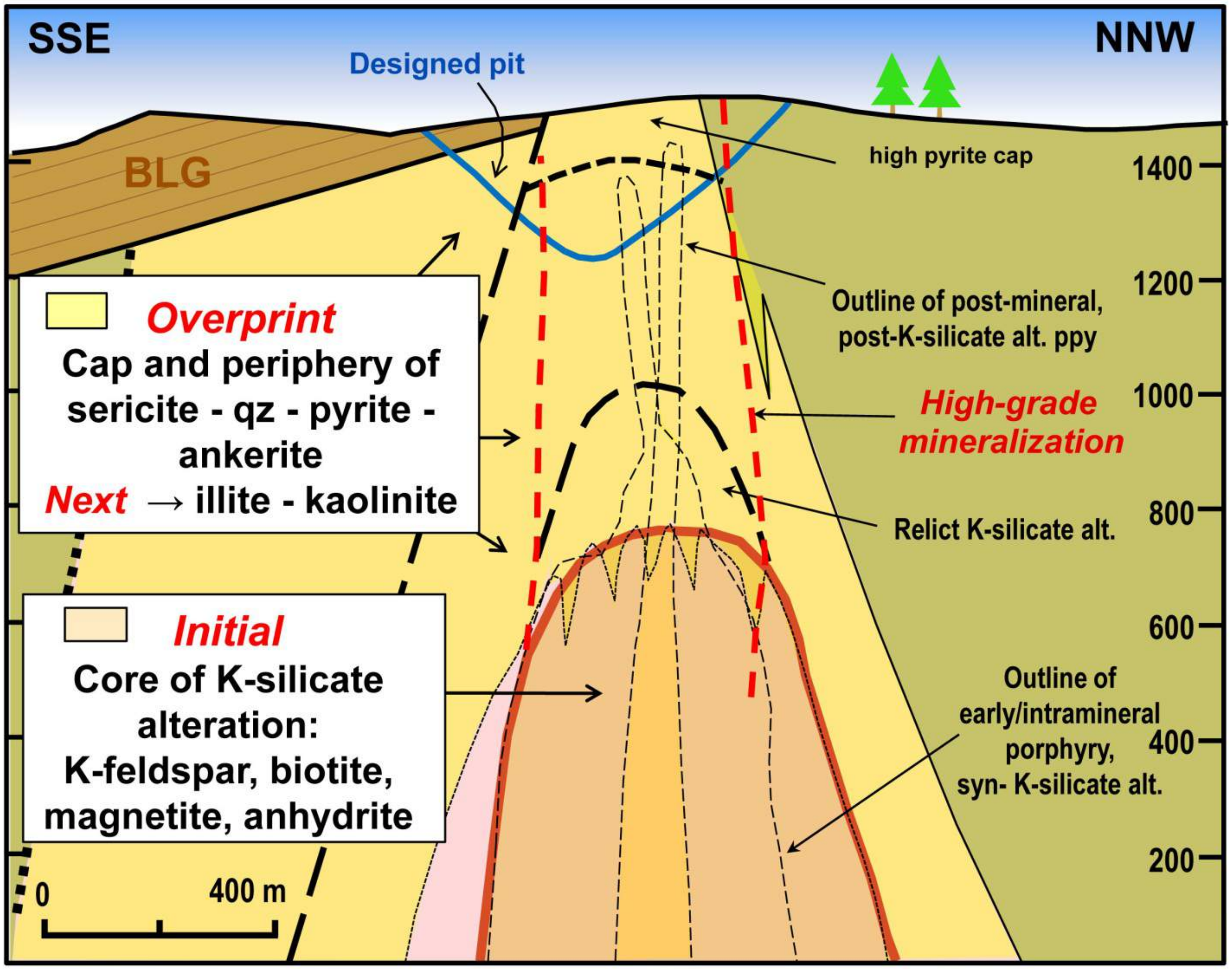
Figure 7.5 Schematic Cross-Section showing East Zone Phase Geometry of the Red Stock



The southeastern margin of the Red stock is generally hidden due to overlying or fault-bounded Hazelton and Bowser Lake Group strata. From the base, these Early to Middle Jurassic units comprise grey, feldspathic and locally calcareous sandstone, overlain by grey and pink-grey, thinly bedded and laminated siliceous siltstones of the Quock Member of the Spatsizi Formation of the Hazelton Group. The Hazelton is paraconformably overlain by the Bowser Lake Group. The latter comprises grey-brown, thinly bedded shaly siltstone with minor cm-scale claystone beds and shell-rich siltstone, chert-grain rich sandstone, and lenses of resistive chert-pebble conglomerate. Formerly known as the Ashman Formation of the basal Bowser Lake Group, the rocks are now termed Todagin Assemblage (Evenchick and Thorkelson, 2005), and interpreted as marine slope facies, with local channels or submarine fans filled with pebble conglomerate.

In the central part of the Red stock, it is bounded on the southeastern side by a NE-trending, steeply SE-dipping, south-side-down fault which places the stock against the Hazelton and Bowser Lake groups (Figs. 7.4 and 7.5). The fault is known as the South Boundary fault (SBF), and is assumed to be mainly dip slip. It is Late Jurassic or Cretaceous in age. The unconformity contact with the underlying Red stock on the hangingwall side has been intersected in several

Figure 7.6 Schematic Cross-Section showing the Alteration Profile of the Deposit



Schematic cross-section through the eastern part of the East zone showing the characteristic alteration profile of the deposit. A core of early potassic alteration is overprinted by later, lower temperature sericitic and intermediate argillic alteration. The upper part of the stock is marked by intense pyritization.

drill holes; however, the dip-slip separation on the fault cannot be calculated confidently, as Hazelton-Bowser rocks have been completely eroded from the footwall side, but it is at least 150 metres. Mapping and subsurface evidence indicates that the unconformity dips gently south overall, but is also gently folded along NNW-trending axes. The SBF may actually be the southeastern limit of a broader fault zone which can be traced laterally into the core of the stock, where it affects mineralization and is known as the East zone fault (more below under 'Red stock structure').

A linear ridge of olivine-phyric alkali basalt is present within the western half of the Red stock (see Fig. 7.3), conspicuous by its well-developed columnar jointing. It is not known if these volcanics represent an isolated outlier of a flow sheet, or alternatively were independently fed by a small fissure pipe. The rocks are assigned to the early Pliocene Maitland Volcanics unit of Evenchick and Thorkelson (2005). They are compositionally similar to the Mount Edziza volcanics, and are within the age range (5.7-4.9 Ma, K-Ar) of that volcanic event, and are thus correlated.

7.4.2 Red Stock geology

The Red Stock is a composite intrusion with multiple internal contacts between constituent porphyries. Alteration (described in a later section) and deformation have obscured primary mineralogy and textures to varying degrees, especially in the upper levels of the stock. The interpretations given below benefitted from the perspective provided by deep drilling done from 2007 onwards into somewhat less altered levels of the stock, where the porphyry sequence could be clarified with detailed logging.

Individual porphyry phases and age relations can be identified by subtle to distinct textural variations in phenocryst and groundmass characteristics, supported by micro-petrographic study. A binocular microscope and graduated hardness picks to identify quartz can be used at the logging stage, and the process is aided by features at porphyry contacts such as igneous breccia, vein truncations and chill margins.

In general, the Red stock porphyries are characterized by plagioclase, hornblende and minor biotite phenocrysts; a trachytoid alignment of both plagioclase and hornblende phenocrysts due to flow banding may be present. Groundmass texture ranges from aplitic to aphanitic. Compositional variations are mainly due to the amounts of primary K-feldspar and quartz in the groundmass. The present interpretation is that the earliest porphyry phases are dominated by diorite to monzodiorite, and were not accompanied by mineralization (*'pre-mineral porphyries'*). These porphyries form the host- or wall rocks to multiple, intermediate-stage porphyries which are quartz monzonite in composition and visually distinguished by longer hornblende phenocrysts (≤8 mm) and groundmass quartz (>5%). They appear to be the principal mineralizing intrusions (*'early- and intra-mineral porphyries'*), although some are more mineralized than others. Still younger porphyries also have 'coarse' hornblende but are deficient in groundmass quartz, are monzonitic in composition, and are only sparsely mineralized (*'late mineral porphyries'*). These latter porphyries plot petrochemically as alkalic, whereas the preceding wall-rock and mineralizing porphyries are calc-alkalic. Late-stage, post-mineral dikes are mafic to intermediate, fine grained and commonly contain small phenocrysts (plagioclase ± hornblende ± biotite) and/or amygdules (carbonate ± quartz).

7.4.3 Red stock structure

The ENE linear trend of the Red Stock suite suggests that it was intruded along a syn-arc structure. This trend is also close to the orientation of important structures within the Red stock such as the already-mentioned NE-trending South Boundary fault (SBF), and the quasi-parallel East zone fault (EZF) (Fig. 7.4). [The ENE trend (present coordinates) is quite prominent in the region, and may represent a deep-seated structural 'grain' in this part of Stikinia that exerted its influence on igneous intrusion and tectonics throughout the Mesozoic.]

The EZF is a sub-vertical to steeply SE-dipping cataclastic zone, 50 to 100 metres thick, consisting of sericite- and clay-altered intrusives in breccia and minor gouge. Porphyroclasts in the fault breccias range from granule to cobble-size, and include mineralized rocks and vein material, indicating at least some displacement occurred after Red Chris mineralization. The EZF does not host undeformed late-stage dikes, suggesting that it post-dates all Red stock intrusive activity. Relative movement on the EZF is believed to be oblique, or sinistral normal-slip (SE-side down), based on separations of sulfide zones and magnetic susceptibility. The vertical separation is in the order of 200 metres; the horizontal separation component may be up to 1,500 metres or more, but is not confidently constrained.

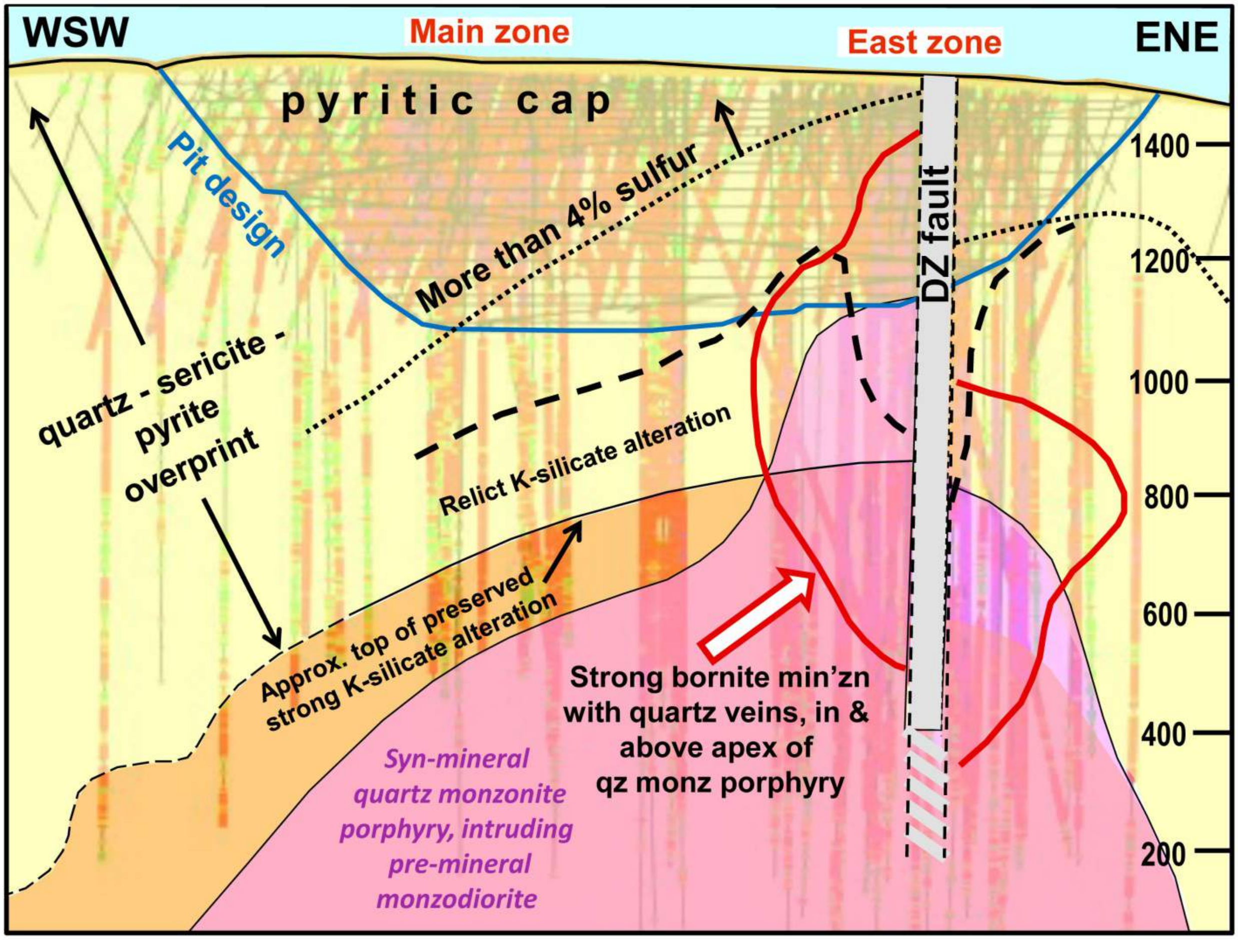
The SBF converges with the EZF from the SW at a very acute angle (Fig. 7.4), and may be part of the same faulting, forming the uppermost surface of a broad, and NE-trending fault zone. The SBF is post-Middle Jurassic because it involves Bowser Lake Group. It is unclear if the near coincidence of the EZF means that it is the same age, or alternatively whether the EZF could be somewhat older (but still post-Red stock), with the later SBF exploiting the inherent weakness of the EZF.

The DZ fault zone (DZF) is another important fault in the east-centre of the Red stock (Figs. 7.4, 7.7), and is characterized by semi-ductile shearing and fault breccia over a 30-50 metre thickness. Its 280° trend differs from the EZF and SBF, and it dips vertically or very steeply north. It also differs in being an older fault, active during the latter stages of Red stock formation as it localized the intrusion of barren to weakly mineralized late-phase porphyries, igneous breccia and dikes. The DZF is late- to post mineralization due to offsets of sulfide zones and copper-gold grade, and it also appears to be truncated by the EZF. The clearest sense of relative movement on the DZF is a large, 300-400 metre, down to the north separation of Cu-Au grade and sulfur contours (Fig. 7.7). It may have a dextral strike-slip component as well.

Other zones of brittle or semi-brittle deformation occur in the Red stock, including healed (~syn-intrusion) tectonic breccia, shear zones and fault/gouge zones. They have a variety of geometries, but most are high-angle and trend parallel to the E-W or ENE axis of the intrusion. Also, numerous steeply dipping transverse faults deform the Red stock, striking NNW to NNE. The geometry and separations related to these faults can be assessed by constructing level plans using drill hole data. However, many geological features can be traced from section to section, and to depth, implying that most of the transverse faulting is probably minor and does not radically affect continuity along the stock's axis.

Bulk tilting of the Red stock has probably occurred. Interpreting the available evidence, the stock has been tilted about 15° to the SSE. This is based on two lines of reasoning: (1) the regional dip of bedding in the Bowser Lake Group, which is approximately 15° to 20° to the SSE; and (2) the observation that on average the long axes of prismatic hornblende phenocrysts plunge about 75° to the north or northwest, and the assumption that this records the paleovertical flow direction during emplacement. While a *ca*. 15° post-mineral bulk tilt of the Red Chris deposit is not insignificant, it would not have a major influence on modeling of the deposit.

Figure 7.7 Schematic Section showing Sulfide Zones, Alteration and Porphyry Morphology



Schematic long-section through the East and Main zones in the centre of the Red stock, against a muted background of composite drilling. Lines depicting sulfide zones, alteration and porphyry morphology are approximate, illustrating features described in the report text. Note offsets due to late- to post-mineral DZ fault.

7.5 Deposit hydrothermal alteration

Hydrothermal alteration in the Red stock (Figs. 7.6 and 7.7) is characterized by an early high-temperature, 'potassic' alteration stage of K-feldspar (orthoclase)-biotite-magnetite-anhydrite, superimposed by lower temperature alterations beginning with sericite-quartz-pyrite(-ferrocarbonate), followed by an intermediate argillic overprint dominated by illite and lesser kaolinite. Quartz veining is associated with the early potassic phase whereas carbonate (+/-quartz) veins mainly accompany the later alterations.

The earlier 'potassic' phase is preserved mainly at deeper levels in the eastern part of the stock (Figs. 7.6, 7.7), in intrusives or in lenses of Stuhini rocks. Biotite replaces primary hornblende and locally forms haloes around quartz veins; later, retrograde chlorite may replace this hydrothermal biotite. Secondary K-feldspar replaces plagioclase and mafic minerals within vein haloes, and where most intense forms a semi-pervasive replacement of the groundmass. The distinction between primary and secondary K-feldspar in the groundmass is difficult.

The potassium silicate alteration is variably overprinted by sericite-quartz-pyrite and intermediate argillic alteration which intensifies upwards and becomes virtually exclusive in the upper levels of the stock, at least in the main part of the hydrothermal system. Where strongly developed, the sericitic alteration replaced primary and secondary feldspars as well as mafic minerals, and can be completely texture-destructive, and variably grade-destructive. Sulfidation during this event is manifested by the replacement of early bornite by chalcopyrite, and by the widespread deposition of pyrite in the sericitic halo and periphery of the mineralization. Even at depth in the (metastable) potassic alteration zone, the sericitic overprint is present in fracture zones or around larger veins where lower temperature and more acidic hydrothermal fluids were able to penetrate and alter the fracture/vein walls.

At shallow levels of the deposit, the sericitic alteration is itself strongly overprinted by illite and kaolinite, especially in zones of stronger deformation. This intermediate argillic alteration is locally intense and texture-destructive, but it is most commonly marked by the replacement of plagioclase phenocrysts by pale milky green illite. In places in the mineralized zones, late stage bornite replaced chalcopyrite. Magnetite (both primary as well as that formed in the potassic alteration stage) is preserved only at depth; elsewhere, magnetite was replaced by dark specular hematite, earthy red hematite, and /or pyrite.

A characteristic of the Red Chris hydrothermal system is the presence of significant ferrocarbonate or ankerite, accompanying quartz in veins and open-space fillings. Some may have formed in the potassic alteration, but it is mainly associated with the lower temperature sericitic and intermediate argillic alteration (Baker *et al*., 1997). After weeks of exposure, the ferrocarbonate in drill core gradually oxidizes to an orange-brown colour, clearly distinguishing it from grey quartz and silicate minerals.

Evidence of propylitic alteration within the stock is limited to minor epidote, which is most common in the deep levels where other alteration is only weakly developed. Propylitic minerals such as epidote and chlorite are better developed in marginal Stuhini Group rocks, at all levels.

Figure 7.8 Field Mapping at Red Chris

8 DEPOSIT TYPE

8.1 Classification

The Red Chris deposit displays characteristics of both alkalic and calc-alkalic porphyry copper deposits.

Alkalic features are:

- the paucity of quartz in the pre-mineral monzodiorite and late-mineral porphyry phases,
- the relatively high K_2O, Na_2O lithogeochemistry of the Stuhini Group volcanics,
- its copper-gold metal signature (*versus* copper-molybdenum for most calc-alkalic systems),
- magnetite-bearing potassic alteration,
- the unusually high copper-gold grades, at least in part of the system (in contrast, calc-alkalic deposits tend to be bulk mineable lower grade ore bodies).

Calc-alkalic features are:

- the large tonnage of the deposit,
- the relatively simple and centralized alteration pattern,
- the intense quartz-sericite-pyrite or 'phyllic' alteration, which is generally underdeveloped in alkalic systems, especially in British Columbia,
- the strong association of copper sulfides with quartz veins.

Whole rock analyses from early and intermediate stage porphyries in the Red stock fall marginally on the calc-alkaline side of the alkaline/subalkaline divider, suggesting that Red Chris should be assigned to the high-K calc-alkalic category of Lang *et al*. (1994). However, late-mineral stage porphyries plot in the alkalic field. Although there remains ambiguity in the petrochemical affinity of Red Chris, perhaps the most telling observation is that the mineralizing (intermediate-stage) quartz monzonite porphyries plot in the high-K calc-alkalic field.

In Seedorff *et al*.'s (2005) classification scheme, Red Chris conforms with their 'monzonitic Cu-(Mo-Au) class' of porphyry copper deposits. This class features an uncommon association of Cu with both Au and Mo credits, although at Red Chris the Mo relationship with Cu-Au is irregular.

Proffett (2009) recognizes two types of porphyry copper systems, based on differences in mineralization style (copper in A-type quartz veins *versus* copper in early-formed fracture halos), and the corresponding genetic correlation with a particular porphyry magma phase (a high *versus* a low correlation, respectively). He attributes the differences to depth of formation, and whether a two-phase or a single-phase fluid was responsible for sulfide behaviour. Red Chris has the hallmarks of an 'A-vein type' system, and probably formed at a relatively shallow level in the crust from a two-phase mineralizing fluid.

Finally, it should be noted that mineralization related to magmatic-hydrothermal brecciation is not evident at Red Chris.

8.2 Exploration model

Prior to Imperial Metals' acquisition of Red Chris in early 2007, historic exploration had established significant copper-gold resources in the Red stock, but since the objective had been open-pit feasibility, drilling was limited to a standard depth of about 400 metres from surface, forcing many drill holes to terminate in mineralization. Two deeper drill holes in the eastern part of the deposit had gone down to between 600 and 800 metres below surface, showing that mineralization did indeed continue substantially below the projected pit, but this did not lead to a change in exploration strategy at the time.

Under Imperial's ownership, the immediate objective remained the permitting and mining of the shallow part of the deposit, but there was obviously a need to test the lower levels of the system to determine if deeper, ore-grade mineralization could be built into the long-term mine plan. Deep drill holes to 1000 metres depth in 2007 and 2008, and to 1,500 metres depth in 2009 and 2010-2011, were successful in finding significant copper-gold mineralization to over 1,000 metres below surface. These programs led to positive recalculations of the total resource at Red Chris in updated technical reports in 2010/2011 and in the present report, and also to a better understanding of the constitution of the Red stock and the nature of the deposit.

The present concept is that the Red stock evolved as a series of high level (< 4-5 km paleodepth) plagioclase+hornblende porphyritic intrusions. The early diorite-monzodiorite phases were essentially barren. The next phase of quartz monzonite porphyries, accompanied by potassium silicate alteration, were richer in quartz both as a groundmass phase and in silica-rich solutions. On crystallization of this magma, exsolved fluids carrying brine-complexed metals and sulfur precipitated as A-type quartz veins with bornite, magnetite and anhydrite. The quartz vein stockworks were concentrated in the apical regions of the mineralizing quartz monzonite porphyries, and copper-gold grade is strongly correlated with vein density. Mineralization extends for several hundred metres (<500-800 metres) from the fertile porphyries into the early, pre-mineral porphyry wall rocks, and into septa of Stuhini Group rocks trapped within the porphyry complex, especially on the northern flank of the Red stock. Late- to post-mineral porphyries are monzonitic and contributed a minor amount of quartz veins and mineralization. As the system cooled, sericitic and argillic alteration migrated downwards and laterally, a common feature among porphyry copper deposits, especially the calc-alkalic type. At Red Chris, the phyllic-argillic alteration is probably associated with widespread sulfidation of bornite to chalcopyrite, some grade destruction and re-distribution of copper, and the formation of a pyrite-rich cap (>4% sulfur) which characterizes the shallower levels of the preserved Red stock.

Previous interpretations held that structure was the controlling factor behind the development of mineralized quartz vein stockworks, but faulting and fracturing may have played only a secondary role; instead, the location of the mineralizing quartz monzonite porphyries is believed to be the primary control on the copper-gold grade distribution in the Red stock. This is a characteristic of A-vein type porphyry systems. As a result, more recent detailed logging and deposit modeling have focused on the geometry of these particular porphyries. The recent (2011) discovery of significant lengths of strong mineralization at intermediate levels in the

Gully zone, 1.5 km west of the present Red Chris ore reserve, has been linked to the presence of a coarse-hornblende porphyry in that area, a possible indication of another mineralizing centre.

Since the coarse-hornblende quartz monzonite 'mineralizing' porphyries tend to be most evident at intermediate levels of the drill-tested Red stock (beneath about 4-500 metres depth from the present surface), it is likely that historic drilling did not penetrate deep enough to test what is apparently the richest part of the system, at least in the East zone.

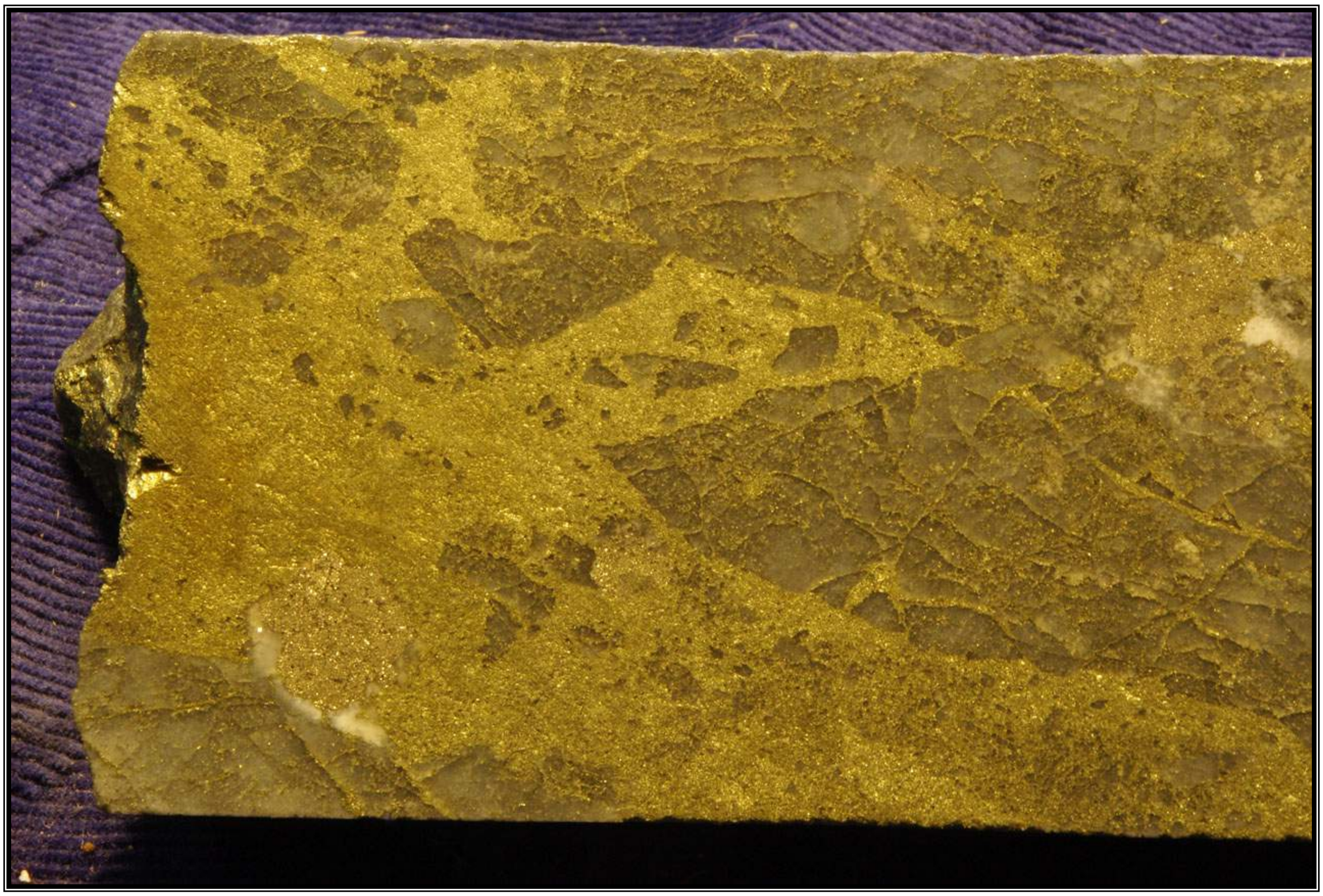


Figure 8.1 Red Chris Core: High Grade Chalcopyrite

9 Mineralization

9.1 Mineral zones

Prospecting and drilling in the Red stock has delineated several zones of significant copper-gold mineralization in the central part of the stock. The two principal zones targeted for mine development currently are the East zone, and the Main zone, centred 800 metres to the west (Figs. 7.4 and 7.7). The intervening area has been termed the Saddle zone, which is somewhat less well mineralized. At surface, East zone and Main zone mineralization extends at least 1,500 metres along the stock's east-northeast axis; in width, it ranges from at least 200 metres in the East zone to 500 metres in the Main zone. The depth of significant mineralization is over 1,000 metres in the East zone and about 1,000 metres in the centre of the Main Zone.

A further 1.5 km to the west of the Main zone are the Gully and Far West zones (Fig. 7.4). The Gully zone footprint is approximately 400-500 metres across, east-west. The Far West zone has a smaller footprint and has seen less drilling than the other zones. Some mineralization in the Far West zone is hosted in Stuhini Group country rocks close to the stock's northern contact.

Prominent limonitic gossans and natural kill zones occur within the steep slopes and drainages over the Gully and Far West zones. However, in areas without drainage relief such as over the East and Main zones, weak limonite extends only 1 or 2 metres beneath the top of the bedrock. The overlying gravel till layer is often very limonitic or composed of ferricrete. Thus, it appears that glaciation has removed any of the supergene mineralization that might have existed over the Red Chris deposit. However, supergene chalcocite mineralization has reportedly been intersected in shallow drilling near the headwaters of the East Gully drainage. Chalcocite occurs along with malachite, azurite and manganese oxides in this oxidized zone.

9.2 Mineralization textures and veins

Copper sulfide mineralization is disseminated, occurs in quartz veins, and in microfractures. Quartz veins range from microveinlets about a millimetre thick to domains of quartz flooding several centimetres thick. The veins can be wavy and anastomosing, or form intricate stockworks, and most may be characterized as A-veins. In mineralized A-veins, quartz forms fine and 'sugary' interlocking grains, hosting fine specks or coarser blebs of bornite and/or chalcopyrite, +/- magnetite. Zones of higher quartz vein density may represent composite A-veins due to repeated fissuring along vein margins or other zones of weakness. Some of these may have the appearance of 'B-veins' due to a central seam of bornite or chalcopyrite, but most are probably A-veins as they lack the vein-normal crystal growths and terminations associated with true B-veins; some B-veins have been recognized, but they are comparatively minor. In addition to the dominant quartz vein-hosted and disseminated mineralization, a minor amount of moderate- to high-grade copper is associated with K-feldspar or biotite-altered halos around early fractures (*early dark micaceous* or 'EDM' veins; see Proffett, 2009).

Bornite and chalcopyrite frequently occupy mafic mineral sites due to the partial or complete replacement of hornblende or biotite. Bornite can be very finely disseminated and easily mistaken for dark red or blue-black specular hematite. Gold is present in microscopic inclusions in bornite and chalcopyrite. Fine molybdenite occurs locally in veinlets of various stages around the strongest Cu-Au zones, especially on the north side of the East zone. Sphalerite and galena occur very locally, usually in trace amounts, in veins or fracture coatings.

Pyrite occurs commonly as very fine- to coarse-grained, anhedral to euhedral disseminations, fracture fillings, and veins. Within the mineralized zones, pyrite content increases upwards, generally concomitant with a decrease in bornite. It ranges from less than 1% to over 10%, with the highest abundance in a halo peripheral to the higher grade copper mineralization (where it is not truncated by faulting). Pyrite is abnormally low or absent in the high-grade East zone except near the surface, but it is quite ubiquitous throughout the Main zone ore and elsewhere in the Red stock. Late-stage pyrite (± minor chalcopyrite) veins cut quartz vein stockworks, and are particularly common in the upper levels of the mineralized system in the sericitic-argillic alteration, and these represent classic D-veins. Pyrite commonly replaces hornblende sites, even in distal parts of the Red Chris deposit.

Figure 9.1 Red Chris Core Sample: Chalcopyrite in Quartz Stockwork

10 Exploration

Since acquiring the Red Chris Property in 2007 Imperial Metals has conducted an aggressive exploration and drilling program and in 2010, exploration drilling at the project was the largest program of its kind in the province of British Columbia. The most recent program consisted of diamond drilling, geological mapping, condemnation drilling, geotechnical drilling and the construction of a new camp. From 2007 to the end of 2010 Imperial has completed 56 holes targeting deep mineralization in the East zone and Main zone beneath the open pit area. Including Imperials work there has been a total of 594,653 metres of exploratory drilling on the Red Chris project since the 1950's when mineralization was first discovered.

10.1 Geophysical Surveys

10.1.1 Titan 24 Geophysical Survey

A Titan-24 geophysical survey was conducted in 2009 as an exploration tool to delineate potential porphyry style mineralization at depth within and surrounding the Red Chris deposit. The survey consisted of 13 parallel lines with 400m line separation with station spacing of 100m. DC, IP, and MT measurements were completed along each line. The survey line length was approximately 2.4km plus additional current injections up to 500m beyond the ends of the survey line for the DC/IP measurements. The DC/IP measurements were completed along a total of 30km (40.2km with current extension). The MT surveys were carried out along a total length of 30km. Final survey analysis and interpretations remains ongoing at the time of publication.

10.1.2 Airborne Magnetic Survey

A 1,295km Aeroquest airborne magnetic survey was flow over the Red Chris deposit between October 13th and October 15th, 2009. The survey was conducted with a helicopter stinger-mounted cesium vapour magnetometer. Ancillary equipment included a GPS navigation system, radar altimeter, digital video recording system and a base station magnetometer.

10.1.3 Proton Magnetometer Survey

A Proton Magnetometer program was established in 2009 with the use of a GSM 19T Series Magnetometer V7 unit. The program commenced August 15th and concluded September 31st. The extensive program included traverses over the entire Titan geophysical grid, road and trail networks as well as concentrated traverses in areas of interest. The purpose of the program was to identify covered geological contacts and faults, as well as to add to the growing geophysical database for vectoring purposes within the Red Chris deposit.

10.1.4 Acoustic Televiewer Survey

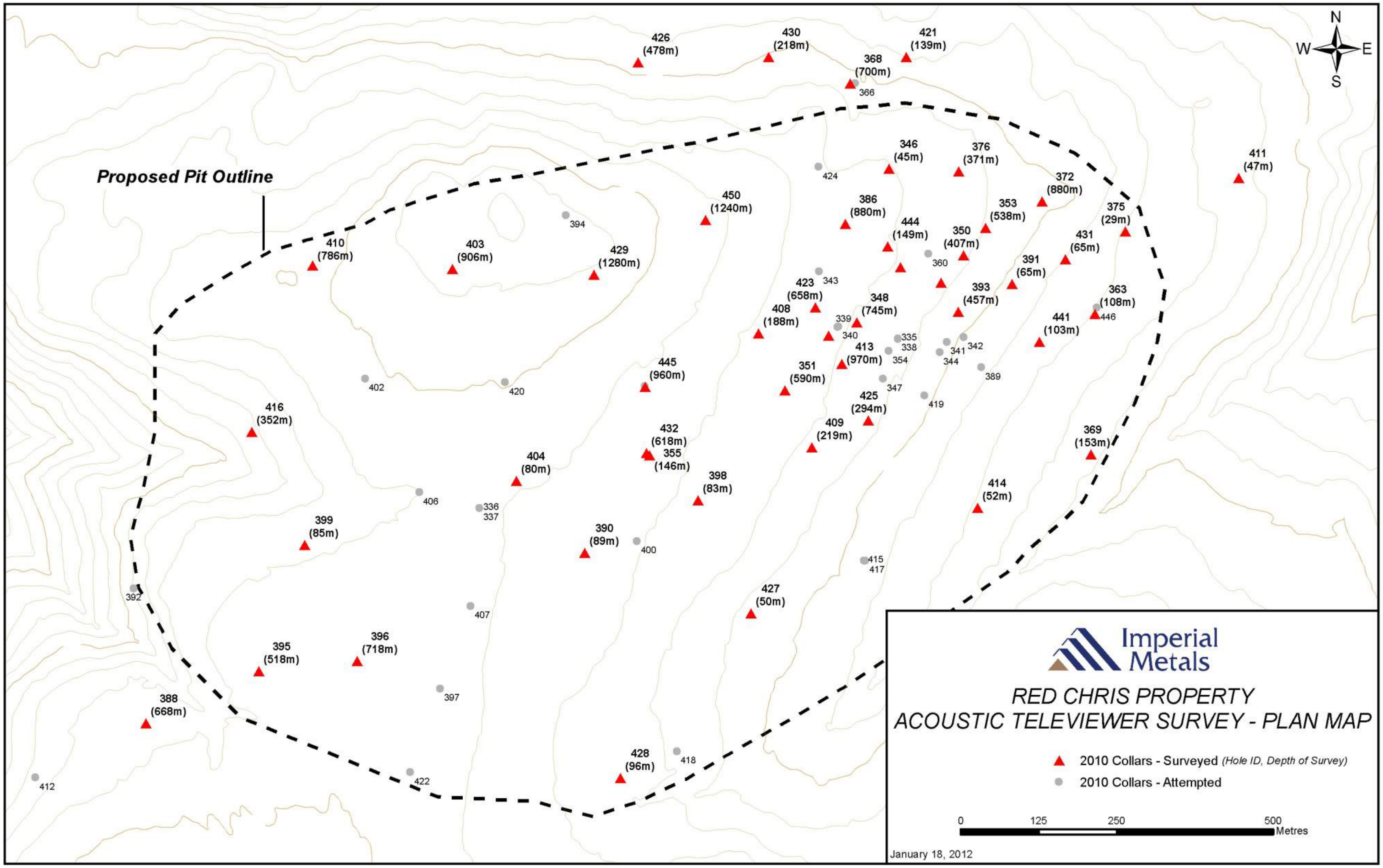
In the summer of 2011 Imperial initiated a downhole acoustic televiewer survey in order to map the geotechnical stability of both the projected pit walls and the geology at depth. In total 61 holes were surveyed, totaling 19,434 metres. Imperial contracted Golder Associates to operate Imperial's in house acoustic televiewer, which consists of an AB140 Televiwer, 3WCA-1000 winch control and a 4WNA winch with 1,500 metres of 4 component cable. The equipment was installed in the back of a canopied Dodge 3500, and surveys were conducted from the pre-existing drill pads. Due to drillhole instability, multiple surveys were terminated prematurely in order to avoid damaging or losing the tool. Data analysis and interpretations remains an ongoing process at the time of publication. See Table number 10.1 for a list of the drill holes surveyed, and Figure 10.1 for a map of the collar locations.

Table 10.1 Acoustic Televiewer Survey Coordinates

Drill Hole	Zone	Easting	Northing	Azimuth	Dip	Survey Depth (m)
RC09-345	East	6395999.76	452702.05	0	-90	131
RC09-346	East	6396157.00	452684.00	0	-77	45
RC09-348	East	6395912.18	452632.55	0	-90	745
RC09-349	East	6395975.58	452767.43	0	-90	364
RC09-350	East	6396018.89	452803.18	0	-90	407
RC09-351	East	6395803.48	452517.74	0	-90	590
RC09-352	East	6395891.05	452587.24	0	-90	681
RC09-353	East	6396063.00	452838.00	0	-90	538
RC10-355	Saddle	6395700.00	452301.10	0	-90	146
RC10-363	East	6395925.00	453013.00	0	-90	108
RC10-368	East	6396293.00	452622.16	155	-80	700
RC10-369	East	6395701.00	453007.10	0	-90	153
RC10-372	East	6396105.00	452928.60	0	-90	880
RC10-375	East	6396057.00	453061.50	0	-90	29
RC10-376	East	6396153.00	452795.00	0	-90	371
RC10-386	East	6396069.00	452614.50	0	-90	880
RC10-388	Main	6395272.00	451496.60	0	-90	668
RC10-390	East	6395544.00	452197.50	0	-90	89
RC10-391	East	6395973.00	452880.30	0	-90	65
RC10-393	East	6395929.03	452794.60	0	-90	457
RC10-395	Main	6395354.74	451676.70	0	-90	518
RC10-396	Main	6395371.29	451834.04	0	-90	718
RC10-398	Saddle	6395628.15	452379.05	0	-90	83
RC10-399	Main	6395556.62	451749.90	0	-90	85
RC10-403	Main	6395997.00	451986.20	155	-65	906

RC10-404	Main	6395658.60	452088.70	0	-90	80
RC10-405	East	6495532.00	452644.50	0	-90	38
RC10-408	East	6395894.44	452475.63	0	-90	188
RC10-409	East	6395712.08	452560.52	0	-90	219
RC10-410	Main	6396003.00	451763.00	155	-65	786
RC10-411	East	6396142.00	453242.80	0	-90	47
RC10-413	East	6395845.74	452608.37	0	-90	970
RC10-414	East	6395616.00	452825.80	335	-76	52
RC10-416	Main	6395737.00	451665.90	0	-90	352
RC10-421	East	6396335.05	452711.21	155	-65	139
RC10-423	East	6395936.00	452566.10	0	-90	658
RC10-425	East	6395755.11	452650.63	0	-90	294
RC10-426	East	6396327.00	452283.00	155	-63	478
RC10-427	Saddle	6395447.06	452463.26	335	-70	50
RC10-428	Main	6395184.33	452254.76	0	-90	96
RC11-429	Saddle	6395987.97	452212.24	0	-90	1280
RC11-430	East	6396335.42	452491.63	155	-65	218
RC11-431	East	6396012.64	452965.96	0	-90	65
RC11-432	Saddle	6395703.40	452296.58	65	-65	618
RC11-441	East	6395880.81	452923.95	0	-90	103
RC11-444	East	6396032.90	452681.88	0	-90	149
RC11-445	Saddle	6395809.00	452294.00	335	-76	960
RC11-450	East	6396075.00	452391.00	0	-90	1240

Figure 10.1 Acoustic Televiewer Survey Collar Locations



10.2 Geochemical Sampling

10.2.1 Composite Sampling Program

Eighteen drillholes from all the deposit zones were selected for multi-element analysis to assist in targeting deeper mineralization in peripheral areas. Core from the selected drill holes was quartered and bagged individually according to original sample tag number (eg 94851). Samples were then composited into groups of four to six sequential samples (eg 94851-94855). In the case that a tag number within a composite sample represented a standard, duplicate or blank, that tag was eliminated from the composite sample. The four to six samples that yielded one composite were placed together in a rice sack, and shipped to Acme Analytical Laboratories in Smithers. A total of 2,500 quarter cut samples comprising 486 composite samples were cut and shipped.

The samples were crushed in Smithers using a Terminator Jaw Crusher to 80% passing 10 mesh sieves. The samples were then shipped to Vancouver and pulverized using a ring and puck to 85% passing 200 mesh sieves. The representative portion of the pulverized material was then weighed and separated to be homogenized with the remaining samples. The composite sample was then sent analyzed using a 4-acid, 1EX ICP-MS package. The remainders of the pulps were sent via commercial vehicle transport to Imperial Metal's storage facility at Mt. Polley Mine.

10.2.2 Peggy Mineral Showing

Geological mapping in 2011 uncovered the Peggy showing 5 kilometers north of the proposed open pit. The sericitized main phase monzodiorite showing is comprised of 1-2% disseminated pyrite with lesser pyrite and malachite fracture fill. Chalcopyrite mineralization is hosted in sporadic 2-10mm A-B type quartz veins. With encouraging hand sample results from the showing (442.9 ppm copper, 0.047 g/t gold from sample 486428) Imperial initiated a grass roots reconnaissance program. The program consisted of 188 B-horizon soil samples and a 15 kilometres proton magnetometer survey. The majority of the soil samples yielded background values, with a few sporadic anomalies. The magnetic survey failed to identify any geological anomalies or support the anomalous soil sample results. Geochemical and geophysical results are summarized in Figure 10.2 and 10.3.

Figure 10.2 Peggy Soil Sample Results

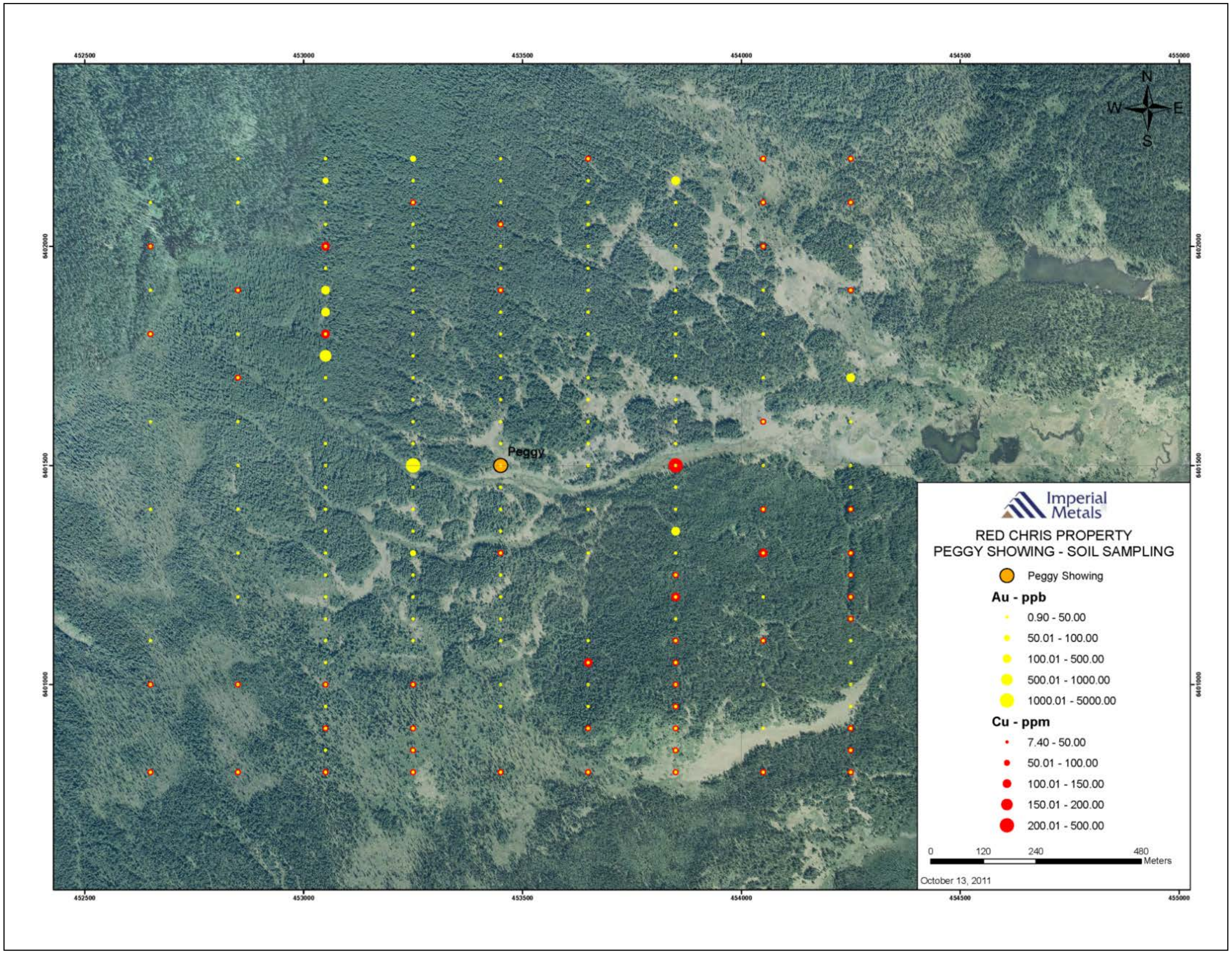
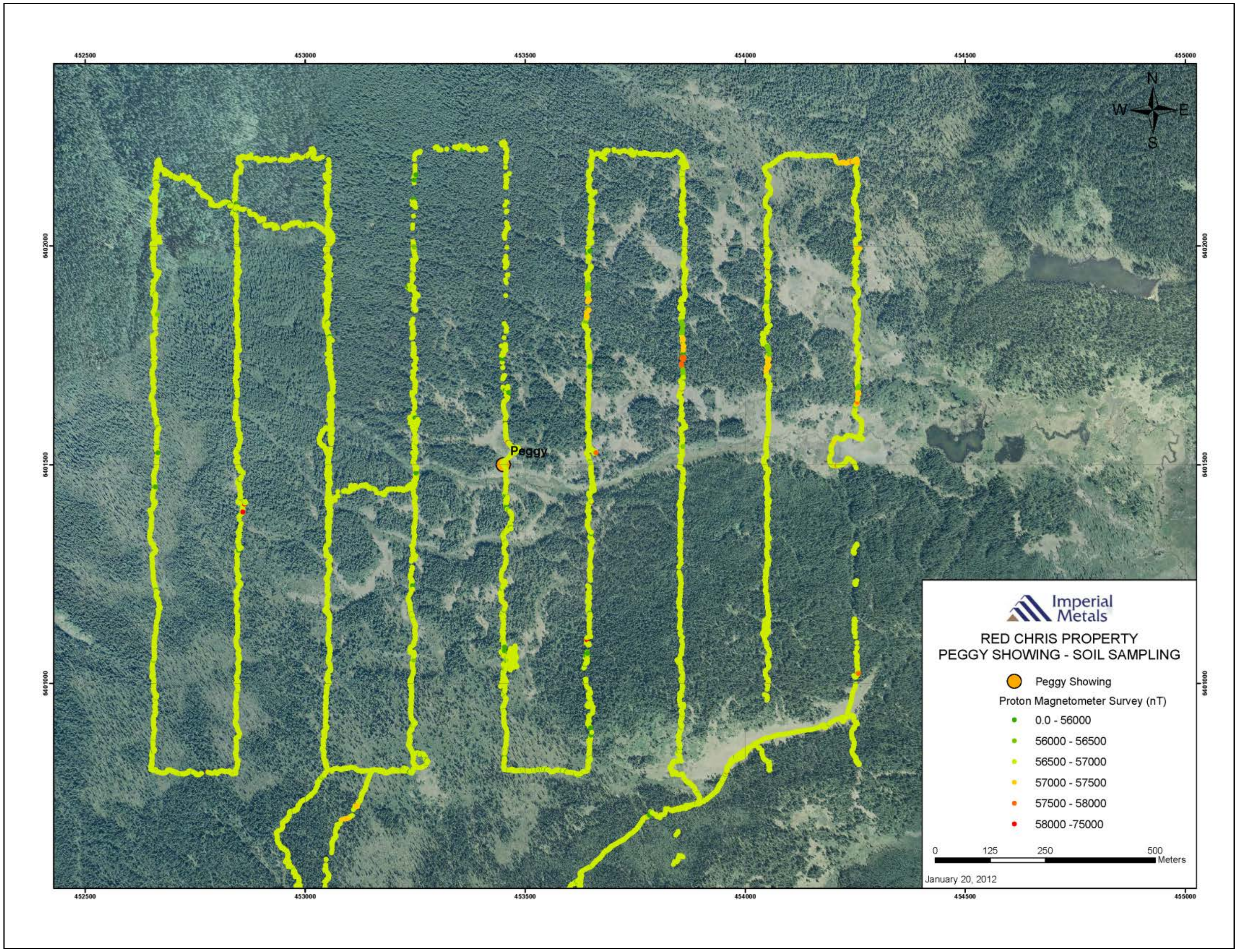


Figure 10.3 Peggy Proton Magnetometer Survey



10.3 Camp and Infrastructure Improvements

The 2007 to 2009 exploration program utilized the long since established Red Chris camp, with additional resources and improvements made to accommodate increasingly larger programs and winter operations. The established 25 man camp was complete with a kitchen, dining area, dry, showers, flush toilets, washer/dryer, office, maintenance sheds, and individual 4-man tent cabins. There is also a dedicated first aid shack, compliant with Health and Safety requirements. Water to the camp is derived from a 100m well located 100m east of the camp.

Due to an increase in the program scope and year round exploration requirements, four trailer sleeper units were installed in late 2008. The four new units (equipped with 7 individual flush toilets and 6 showers) provide an additional 32 man capacity in camp. The trailers were arranged in rows of two lengthwise, with the space in between the sets of trailers enclosed from the elements adding additional dry/storage facilities. Additional camp improvements included insulating the tent cabins (office and sleeping units) and core shack with thermal batt insulation. Further to that, the core shack was upgraded with the addition of an additional logging area, a core cutting (two rock saws) and sample preparation rooms. Additional diesel drip heaters were installed throughout the camp in preparation for winter operations.

During the 2010 exploration season, a new camp was constructed approximately 4 kilometres to the north of the old camp. The old camp was located within the open pit boundary; therefore it was not practical to invest in additional facilities at that location. The new camp (see Figure 10.4) has capacity for 60 people in a fully connected complex with a kitchen, dining area, dry, showers, flush toilets, washer/dryer, exercise room, recreation room, employee’s telephone booth and a dedicated first room compliant with Health and Safety requirements. A separate office, a 40’ X 80’ steel building core shack, two maintenance sheds and a spare office trailer are outbuildings at the new camp. The facilities are serviced by a water well and a mound-type septic disposal field. The old camp was decommissioned and the site was reclaimed in the summer of 2011. All of the drill core that was stored at the old camp was relocated to the new camp.

Improvements to the 2010-2011 drill operations included the installation of a 750 metres waterline from Border Creek southward, surfacing along the periphery of the East Zone. A 3ft by 10ft perforated culvert was installed vertically along the periphery of Border Creek, and enclosed by a monitoring/pumping station. The waterline was buried in a 7-8ft deep trench to ensure its utility during the winter months. A similar setup has been constructed from an unnamed creek to the west of Border Creek, to the Main zone. Adequate water capacity is available to run 5 drills, 12 months of the year.

Figure 10.4 New 2010 Red Chris Camp and Core Storage

10.4 Property Access Trail

The 17km gravel access trail, which branches off at the 6km marker of the Ealue Lake road, was completed in early 2009. Further upgrades to the trail included the installation of two eco barriers, to prevent potential slope failure, and multiple culverts, to prevent washouts. (see Figure 10.5).

Figure 10.5 New Red Chris 17km Gravel Access Trail

11 Drilling

Upon the acquisition of the Red Chris project, Imperial conducted a helicopter-based diamond drilling program in the summer of 2007 to test for higher-grade material below the bottom of the planned pit, proposed in the 2005 feasibility study. Initial results were rewarding, prompting the company to construct an access road into the project area prior to resumption of the deep drilling. The campaign of deep exploration resumed in the fall of 2008 and then from July through December of 2009. Encouraged by positive drill results, Imperial initiated year round drilling in 2010 and 2011 to fully delineate the mineralization beneath and peripheral to the proposed pit (see Figure 11.4 for collar locations). Figure 11.1 shows a comparison of the drilling at the end of 2006 and 2009, with the drilling to the end of 2011.

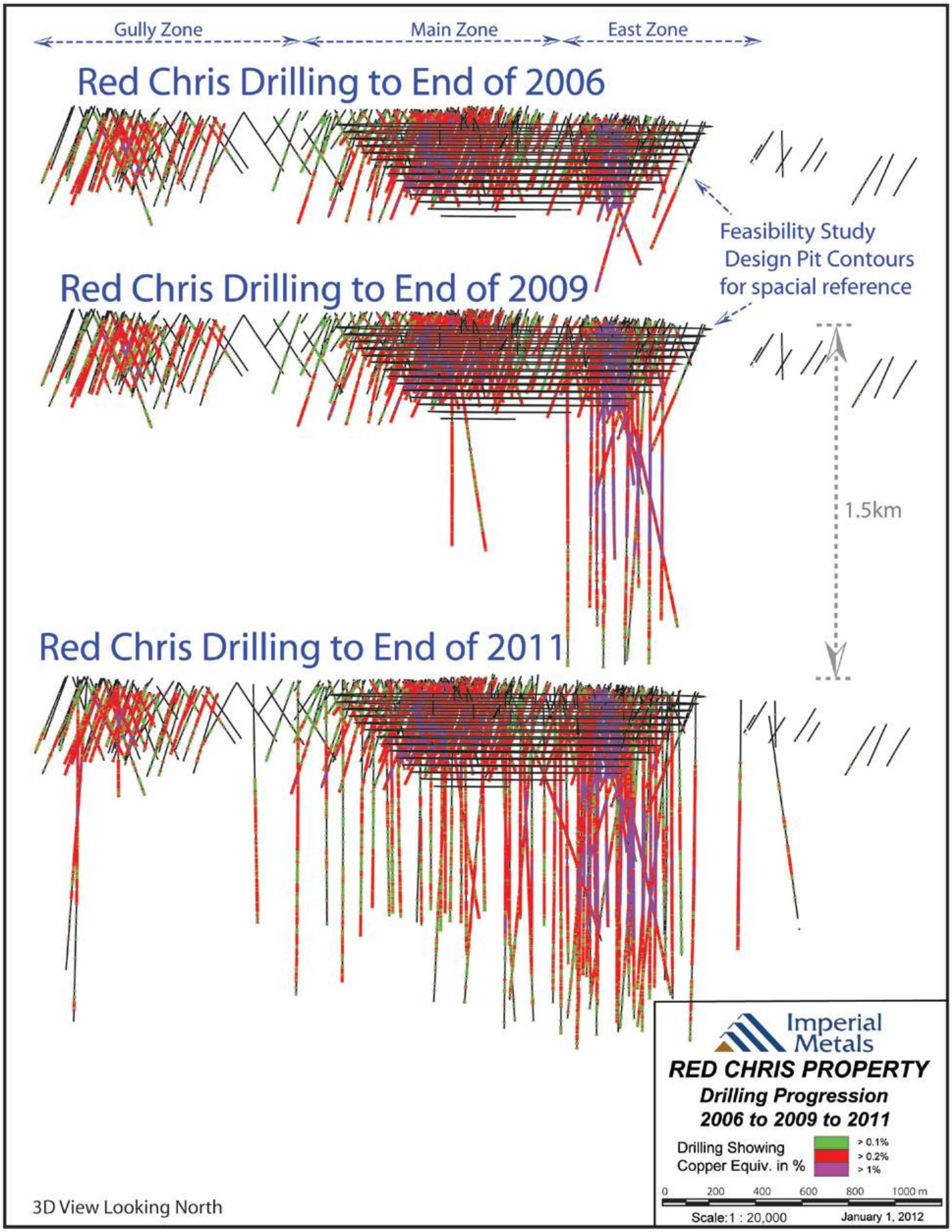
11.1 Deep Exploration Drilling - 2007

Historical results from drillholes 140 and 06-324 indicated the presence of grades in excess of 1% CuEq up to 400 metres below the bottom of the pit design (see Figure 11.1). Imperial completed six holes in 2007 for a total length of 4,835 metres. The most significant drillhole was 07-335, collared vertically in the core of the East Zone, 07-335 graded 1.01% copper, 1.26 g/t gold and 3.92 ppm silver over its entire length in bedrock of 1,024.1 metres, extending high-grade mineralization in the East Zone down another 270 metres from its previously-known extent. Drillhole 07-336, collared in the middle of the Main Zone, returned a vertical intercept of 996.4 metres grading 0.40% copper, 0.38 g/t gold, and 1.34 ppm silver. Drillhole 07-338, collared in the East Zone, was drilled from the same collar location as 07-335, but was angled to the north at a -78° dip. The final 46.5 metres of this drillhole graded 1.05% copper, 1.67g/t gold, and 1.41ppm silver, before being abandoned at 721.5 metres due to stuck rods. The final drillhole of 2007, 07-340, was collared in the East Zone and drilled at 095° azimuth. 07-340 returned 753.9 metres grading 0.60% copper, 0.56 g/t gold and 2.07ppm silver before terminating in a fault at 760.0 metres depth. Collar coordinates and orientations are summarized below.

Table 11.1 2007 Drillhole Coordinates

Drill Hole	Zone	Easting	Northing	Azimuth	Dip	Total Length (m)
07-335	East	6395888	452699	0	-90	1029.0
07-336	Main	6395615	452026	0	-90	1000.7
07-337	Main	6395615	452026	0	-75	1050.7
07-338	East	6395888	452699	0	-78	721.5
07-339	East	6395904	452602	95	-66	217.1
07-340	East	6395904	452602	95	-66	815.9

Figure 11.1 Red Chris Drilling Progression 2006 to 2009 to 2011



11.2 East Zone Deep Exploration Drilling – 2008

In 2008 three vertical holes, totaling 2,220 metres were drilled in the East zone, targeting the deep mineralization that was discovered in the 2007 program holes 07-335 and 07-338. The first two holes of 2008 were abandoned above their target depth due to adverse ground conditions and technical difficulties with drilling. The third hole, RC08-343, collared 165m northwest of 07-335 was completed to 1273 metres, and encountered 433 metres of mineralization between 840.3 metres and 1273.2 metres, grading 0.36% copper, 0.46 g/t gold and 1.13 ppm silver. Within this intersection was a higher grade interval of 97.5m grading 0.63% copper, 0.96 g/t gold and 1.89 ppm silver. The drill program was supplemented by geological mapping survey to further refine geological contacts (8 rock samples). To reduce helicopter reliance a 17 km access trail was constructed to camp, branching off at the 6 kilometre marker on the Ealue Lake Road. Collar locations and orientations are summarized below.

Table 11.2 2008 Drillhole Coordinates

Drill Hole	Zone	Easting	Northing	Azimuth	Dip	Total Length (m)
RC08-341	East	452775	6395881	0	-90	435.0
RC08-342	East	452803	6395890	0	-90	511.8
RC08-343	East	452572	6395995	0	-90	1273.2

Figure 11.2 Red Chris Drilling in Progress

11.3 2009 Deep Exploration Drilling

The 2009 exploration program consisted of 11 diamond drillholes totaling 14,172 metres (see Figure 11.3 for highlights). The East Zone drilling program was a continued initiative from the 2007/2008 exploration programs, which discovered the extension of high grade Cu-Au mineralization past the previously established pit outline, down to over a kilometer in depth (RC07-335 graded 1.01% copper, 1.26 g/t gold and 3.92 g/t silver over 1,024.1 metres, ending in high grade mineralization). HQ/NQ diameter core was drilled with the use of a modified Bolyles 56 and a LF-230. Collar coordinates and orientations are summarized below.

Table 11.3 East Zone Drillhole Coordinates

Drill Hole	Zone	Easting	Northing	Azimuth	Dip	Total Length (m)
RC09-344	East	452765	6395864	0	-90	331.3
RC09-345	East	452702	6396000	0	-90	1501.1
RC09-346	East	452684	6396157	155	-77	1503.9
RC09-347	East	452675	6395822	0	-90	1315.5
RC09-348	East	452633	6395912	0	-90	1501.8
RC09-349	East	452769	6395976	0	-90	1150.6
RC09-350	East	452803	6396019	0	-90	1477.4
RC09-351	East	452518	6395803	0	-90	1501.1
RC09-352	East	452587	6395891	0	-90	1245.0
RC09-353	East	452838	6396063	0	-90	1287.8
RC09-354	East	452683	6395868	39	-71	1357.34

Note: holes RC09-335 and 354 were started in 2009 and were completed in 2010 upon return from winter break up.

The 2009 drilling program successfully expanded the previously known mineralization in the Deep East Zone, both laterally and to depth. All drill holes encountered significant intercepts of copper and gold mineralization, and added to the volume of the deep resource. The highlight of the program was Hole RC09-350 which was collared approximately 170 metres northeast of drill hole 07-335 and returned 432.5 metres grading 2.00% copper, 3.80 g/t gold and 4.42 g/t silver which included a 152.5 metres interval grading 4.12% copper, 8.83 g/t gold and 10.46 g/t silver starting at a depth of 540.0 metres. The significant results from the 2009 drill holes are provided in Table 11.3.

Figure 11.3 Red Chris 2009 Drilling Cross Section: Showing Assay Highlights

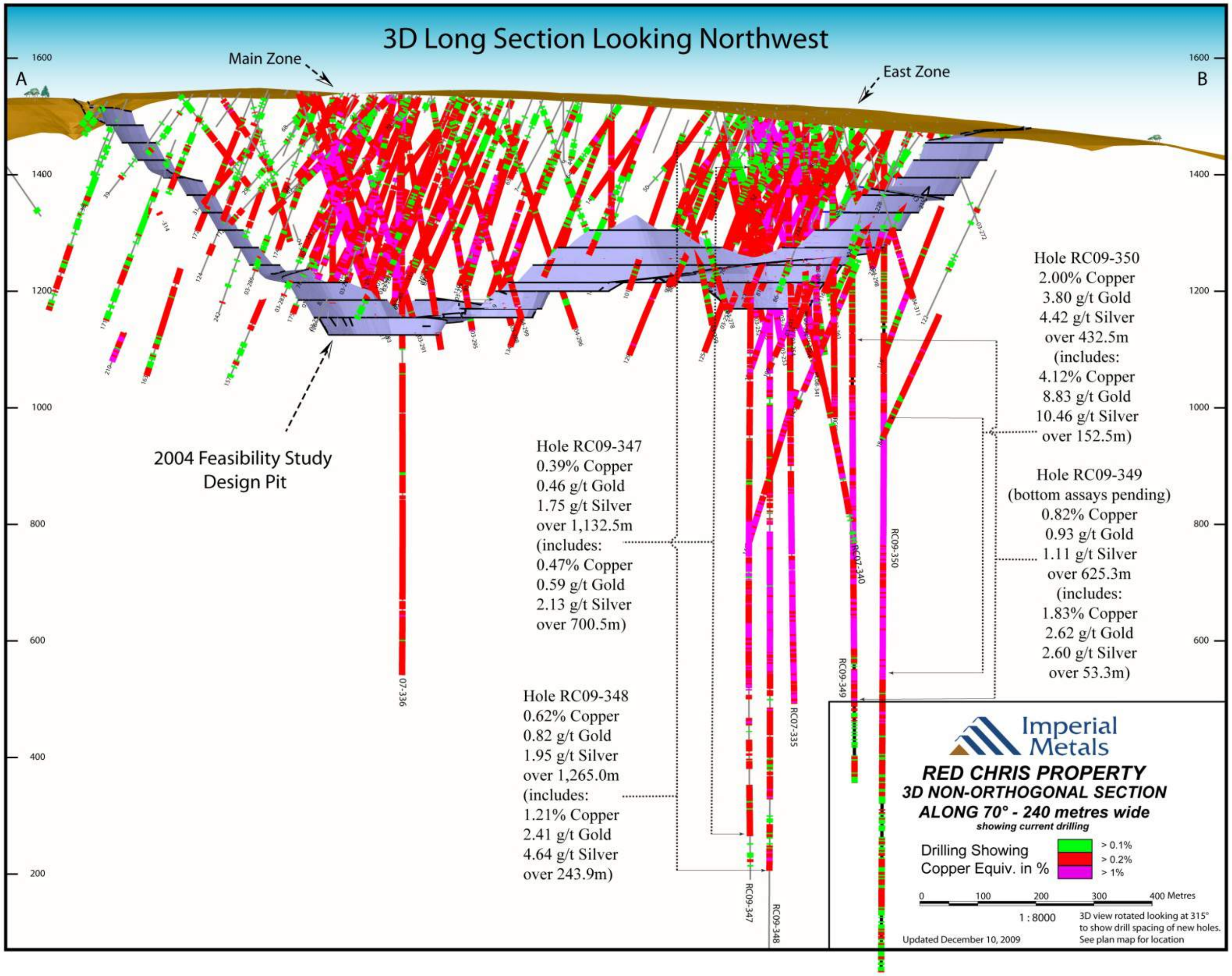


Table 11.4 2009 East Zone Drill Intercepts

	Total	Metre Interval		Interval	Copper	Gold	Silver
Drill Hole	**Length (m)**	**from**	**to**	**Length**	**%**	**g/t**	**g/t**
RC09-344	331.3	3.7	331.3	327.7	0.37	0.40	1.22
incl.		125.0	331.3	206.3	0.49	0.50	1.71
incl.		132.5	211.7	79.2	0.77	0.77	2.64
RC09-345	1501.1	426.1	952.5	526.4	0.77	1.37	1.68
incl.		475.0	518.9	43.9	2.22	3.22	3.35
incl.		833.8	950	116.2	1.06	3.11	4.09
and		1035.0	1090	55.0	0.40	0.41	1.02
RC09-346	1503.9	530.0	1043.6	513.6	0.61	0.91	1.16
incl.		615.0	806.1	191.1	0.92	1.53	1.20
incl.		995.0	1043.6	48.6	1.01	1.8	3.61
		1107.5	1277.5	170.0	0.21	0.16	0.97
RC09-347	1315.5	108.0	1240.5	1132.5	0.39	0.46	1.75
incl.		875.0	990.5	115.5	0.63	0.88	2.50
RC09-348	1501.8	42.5.0	1307.5	1265	0.62	0.82	1.95
incl.		725.4	969.3	243.9	1.21	2.41	4.64
RC09-349	1150.6	390.0	1015.3	625.3	0.82	0.93	1.11
incl.		545.0	582.5	37.5	1.51	1.89	1.54
incl.		866.6	919.9	53.30	1.83	2.62	2.60
RC09-350	1477.4	420.0	1067.5	647.5	1.50	2.68	3.22
incl.		530.0	962.5	432.5	2.00	3.8	4.42
incl.		540.0	692.5	152.5	4.12	8.83	10.46

Table 11.4 continued 2009 East Zone Drill Intercepts

	Total	Metre Interval		Interval	Copper	Gold	Silver
Drill Hole	Length (m)	from	to	Length	%	g/t	g/t
RC09-351	1501.1	237.5	732.5	495.0	0.46	0.59	1.21
incl.		287.5	545.0	257.5	0.54	0.69	1.38
incl.		425.7	530.0	104.3	0.66	0.96	1.62
and		1092.5	1222.5	130.0	0.30	0.26	0.91
RC09-352	1245.0	115	852.9	737.9	0.54	0.6	1.61
incl.		300	852.9	552.9	0.62	0.73	1.85
incl.		407.5	530.0	122.5	0.82	0.67	1.94
incl.		680	850.0	170.0	0.84	1.41	3.33
RC09-353	1287.8	67.5	1202.5	1135.0	0.50	0.59	0.71
including		570	1020.0	450.0	0.80	1.07	1.03
including		632.5	705.0	72.5	1.16	1.88	1.32
including		912.3	952.5	40.2	0.96	1.14	1.19
RC09-354	1357.6	6.7	304.1	297.4	1.77	1.69	5.23
including		6.7	120.0	113.3	2.44	1.87	7.38
and		480	1297.5	817.5	0.55	0.47	0.82

11.4 2010 Deep Exploration Drilling

The 2010 deep exploration diamond drilling program consisted of 47 completed holes totaling 52,810 metres, as well as 4 partial drill holes that were collared in late 2010, but completed in early 2011 yielding an additional 5,576 metres (see Figure 11.4 for collar locations). Drilling began in the first week of February, building up to five drills by June and concluded in mid-December. Both HQ/NQ core was drilled using three modified Boyles 56's, two LF-230's and a Boyles 37. The deep drilling program was designed to delineate the extent of mineralization at depth in the open pit area. Holes were typically drilled to a minimum depth of 1,000 metres, but were extended to more than 1,500 metres where alteration and mineralization continued to show promise. Collar coordinates and orientations are summarized below.

Table 11.5 2010 Deep Drillhole Coordinates

Drill Hole	Zone	Easting	Northing	Azimuth	Dip	Total Length (m)
RC10-355	Saddle	452301	6395700	0	-90	1315.53
RC10-360	East	452747	6396021	0	-90	1426.52
RC10-363	East	453014	6395924	0	-90	1219.61
RC10-366	East	452622	6396293	150	-80	317.53
RC10-368	East	452622	6396293	150	-80	1312.77
RC10-369	East	453004	6395701	0	-90	1010.41
RC10-372	East	452928	6396105	0	-90	1139.04
RC10-375	East	453061	6396057	0	-90	1486.23
RC10-376	East	452796	6396154	0	-90	1065.00
RC10-386	East	452612	6396069	0	-90	1452.98
RC10-388	Main	451499	6395272	0	-90	1355.45
RC10-389	East	452829	6395840	0	-90	1486.82
RC10-390	Saddle	452200	6395544	0	-90	1167.78
RC10-391	East	452877	6395973	0	-90	1324.97
RC10-392	Main	451477	6395487	0	-90	1026.76
RC10-393	East	452795	6395930	0	-90	1318.26
RC10-394	Saddle	452168	6396083	155	-65	1501.14
RC10-395	Main	451677	6395355	0	-90	1300.38
RC10-396	Main	451834	6395371	0	-90	983.59
RC10-397	Main	451967	6395327	0	-90	1001.88
RC10-398	Saddle	452379	6395628	0	-90	1120.14
RC10-399	Main	451750	6395557	0	-90	1011.22
RC10-400	Main	452282	6395363	0	-90	1014.28
RC10-401	Saddle	452297	6395809	0	-90	1014.28
RC10-402	Main	451847	6395822	0	-90	1089.50
RC10-403	Main	451986	6395997	155	-65	1522.48

RC10-404	Main	452088	6395658	0	-90	1043.94
RC10-405	East	452644	6395531	0	-90	1001.88
RC10-406	Main	451934	6395641	0	-90	986.64
RC10-407	Main	452016	6395459	0	-90	1053.69
RC10-408	Main	452476	6395894	0	-90	1206.09
RC10-409	Main	452561	6395712	0	-90	1361.54
RC10-410	Main	451763	6396003	155	-65	1493.20
RC10-411	East	453243	6396142	0	-90	1075.03
RC10-412	Main	451320	6395185	0	-90	1029.31
RC10-413	Main	452608	6395847	0	-90	1468.22
RC10-414	East	452825	6395623	335	-76	1357.27
RC10-415	East	452647	6395536	120	-45	236.52
RC10-416	Main	451666	6395737	0	-90	1001.88
RC10-417	East	452647	6395536	335	-45	166.42
RC10-418	Main	452345	6395226	335	-45	166.42
RC10-419	East	452741	6395795	0	-90	1212.19
RC10-420	Main	452071	6395816	0	-90	1001.88
RC10-421	East	452711	6396335	155	-65	1527.30
RC10-422	Main	451919	6395193	0	-90	1101.02
RC10-423	East	452566	6395936	0	-90	1257.91
RC10-424	East	452572	6396160	0	-90	1001.88
RC10-425	East	452651	6395755	0	-90	1550.52
RC10-426	East	452280.52	6396325.10	155	-63	1510.28
RC10-427	Main	452463.08	6395445.90	335	-70	1498.70
RC10-428	Main	452255.02	6395184.00	0	-90	1016.51

Note holes RC10-425, 426, 427 and 428 were started in 2010 and were completed in 2011 upon return from winter break up.

The 2010 deep exploration drilling program successfully expanded the known limits of mineralization beneath the planned open pit. Initial drilling was concentrated in the East Zone, and was successful in increasing the volume of high grade mineralization at depth. The most notable intercepts include hole RC10-360 (57 metres west of RC09-350) which intersected 671.2 metres of 1.03% copper, 1.66 g/t gold and 1.94 g/t silver, as well as hole RC10-393 which intersected 1,112.5 metres of 0.54% copper, 0.61 g/t gold and 1.96 g/t silver, one of the longest mineralized intervals on the property to date. Mineralization was extended east of the East Zone with drill hole RC10-375, which intersected 861.7 metres grading 0.41% copper, 0.38 g/t gold and 0.94 g/t silver including a 302.5 meter section grading 0.56% copper, 0.54 g/t gold and 1.03 g/t silver.

Hole RC10-388, one of the first holes in the deep Main Zone since 2007, tested the western edge of the zone and intersected five intervals of copper-gold mineralization including 380.0 metres grading 0.34% copper, 0.50 g/t gold and 0.77 g/t silver. Subsequently two holes were collared near the western rim of the proposed open pit and added significantly to Main Zone dimensions. RC10-392 returned 232.5 metres grading 0.39% copper, 0.48 g/t gold and 1.63 g/t silver, and RC10-395 returned multiple high grade intercepts over its 1,300.58 metre length. The significant results of all 2010 drill holes are provided in Table 11.6.

Table 11.6 2010 Drill Intercepts

	Total	Metre Interval		Interval	Copper	Gold	Silver
Drill Hole	Length (m)	from	to	Length	%	g/t	g/t
RC10-355	1489.00	335.40	1092.50	757.10	0.38	0.48	1.27
including		1007.50	1065.00	57.50	0.93	1.56	4.19
RC10-360	1426.52	473.80	1145.00	671.20	1.03	1.66	1.94
including		560.00	1087.50	527.50	1.19	1.98	2.16
including		607.50	962.50	355.00	1.41	2.59	2.40
including		700.00	755.80	55.80	2.00	4.22	3.25
including		802.50	952.50	150.00	1.83	3.34	3.35
RC10-363	1219.61	630.00	655.00	25.00	0.26	0.25	2.68
and		670.00	695.10	25.10	0.36	0.50	0.65
and		728.30	779.60	51.30	0.34	0.69	1.84
and		837.10	849.50	12.40	0.37	0.50	1.26
RC10-366	317.53	NSI					
RC10-368	1312.77	732.50	1287.50	555.00	0.28	0.26	0.67
including		915.40	1202.50	287.10	0.34	0.35	0.83
including		1010.00	1130.00	120.00	0.41	0.35	1.03
RC10-369	1010.41	NSI					
RC10-372	1139.04	505.00	682.50	177.50	0.35	0.36	0.71

RC10-375	1486.23	297.50	1159.20	861.70	0.41	0.38	0.94
including		530.00	832.50	302.50	0.56	0.54	1.03
including		870.00	923.40	53.40	0.55	0.42	1.18
including		962.50	1152.50	190.00	0.29	0.20	0.72
and		1222.50	1410.00	187.50	0.20	0.13	0.51
RC10-376	1065.89	612.40	777.50	165.10	0.26	0.20	0.53
RC10-386	1452.98	757.50	1352.50	595.00	0.32	0.35	0.72
including		757.50	1015.00	257.50	0.41	0.48	0.77
RC10-388	1355.45	250.00	1297.30	1047.30	0.22	0.34	0.89
including		352.50	732.50	380.00	0.34	0.50	0.77
including		872.50	910.00	37.50	0.29	0.55	3.83
including		965.00	1050.00	85.00	0.21	0.31	0.83
including		1092.50	1230.00	137.50	0.20	0.33	0.92
RC10-389	1156.72	425.00	732.50	307.50	0.47	0.36	1.64
including		492.50	552.50	60.00	0.64	0.50	1.60
and		839.30	1062.50	223.20	0.40	0.32	1.52
and		840.00	927.50	87.50	0.56	0.52	2.29
RC10-390	1167.78	57.50	72.50	15.00	0.33	0.32	0.62
and		117.50	195.00	77.50	0.28	0.24	0.82
and		233.60	259.90	26.30	0.32	0.35	0.38
and		302.50	365.00	62.50	0.19	0.22	2.40
and		452.50	911.60	459.10	0.28	0.20	0.85
and		936.10	977.50	41.40	0.24	0.22	2.79
and		1095.00	1152.50	57.50	0.17	0.15	4.12
RC10-391	1324.97	162.50	1165.00	1002.50	0.50	0.38	0.99
including		512.50	680.00	167.50	0.75	0.58	0.97
including		910.00	932.00	22.00	1.04	1.19	2.61
RC10-392	1026.76	247.50	480.00	232.50	0.39	0.48	1.63
including		345.00	432.50	87.50	0.61	0.70	2.73
and		705.00	782.50	77.50	0.11	0.35	0.66
RC10-393	1318.30	55.00	1167.50	1112.50	0.54	0.61	1.96
including		617.50	935.00	317.50	1.08	1.46	4.28
including		617.50	652.00	34.50	1.45	1.93	5.39
including		666.50	694.60	28.10	1.18	1.82	4.88
including		722.20	847.50	125.30	1.40	1.87	5.23
including		880.00	937.50	57.50	1.26	1.74	5.84
RC10-394	1501.10	472.10	1207.50	735.40	0.32	0.32	1.54

including		567.50	595.80	28.30	1.16	1.12	2.48
RC10-395	1300.58	200.00	245.00	45.00	0.32	0.16	0.54
and		837.50	877.50	40.00	0.21	0.21	1.03
and		1097.20	1172.50	75.30	0.25	0.28	1.68
and		1232.50	1297.50	65.00	0.23	0.18	0.74
RC10-396	983.59	732.50	983.59	251.09	0.33	0.30	1.05
RC10-397	1001.90	27.50	62.50	35.00	0.37	0.18	0.39
and		130.00	537.50	407.50	0.43	0.42	0.93
including		240.00	270.00	30.00	1.13	0.55	1.14
including		387.50	427.50	40.00	0.79	1.09	1.63
and		635.00	675.00	40.00	0.23	0.24	0.59
RC10-398	1120.14	112.50	155.00	42.50	0.27	0.31	0.81
and		185.00	425.00	240.00	0.25	0.23	0.98
and		597.20	758.90	161.70	0.30	0.15	3.82
and		600.00	630.00	30.00	0.27	0.19	13.39
and		800.50	1045.00	244.50	0.36	0.36	2.27
RC10-399	1011.22	549.30	570.00	20.70	0.23	0.23	1.04
and		602.50	615.00	12.50	0.37	0.39	1.00
RC10-400	1014.28	90.00	102.50	12.50	0.41	0.17	1.09
and		557.50	582.50	25.00	0.25	0.19	0.80
RC10-401	1038.45	347.50	985.00	637.50	0.42	0.39	1.31
including		352.50	437.50	85.00	0.55	0.50	1.13
RC10-402	1089.50	102.50	150.00	47.50	0.26	0.13	0.49
and		277.50	309.50	32.00	0.30	0.16	0.42
and		660.00	751.50	91.50	0.29	0.22	0.74
and		825.00	847.50	22.50	0.23	0.23	0.55
RC10-403	1522.48	337.50	1237.50	900.00	0.25	0.22	0.95
including		337.50	563.70	226.20	0.29	0.28	0.46
including		670.00	775.00	105.00	0.23	0.25	0.80
including		877.50	1237.50	360.00	0.33	0.25	1.63
including		1015.00	1077.50	62.50	0.59	0.35	1.79
including		1125.00	1162.50	37.50	0.52	0.34	3.15
RC10-404	1043.94	55.00	257.50	202.50	0.45	0.44	1.37
including		122.50	177.50	55.00	0.94	0.95	2.27
including		162.50	177.50	15.00	2.43	2.50	2.82
and		324.30	456.80	132.50	0.67	0.76	1.17
including		387.50	415.00	27.50	1.16	1.52	1.71

and		478.40	665.00	186.60	0.20	0.20	0.72
and		687.50	775.00	87.50	0.17	0.21	0.45
and		805.80	830.40	24.60	0.22	0.28	1.41
and		867.50	880.00	12.50	0.31	0.33	0.77
and		925.00	997.50	72.50	0.22	0.23	0.79
RC10-405	1001.88	530.00	667.50	137.50	0.29	0.31	0.61
and		827.50	942.50	115.00	0.32	0.48	0.60
RC10-406	986.64	15.00	265.80	250.80	0.43	0.31	0.86
and		15.00	607.50	592.50	0.33	0.26	0.75
including		196.20	212.50	16.30	0.92	1.06	1.11
including		292.30	305.00	12.70	0.30	0.39	0.40
including		345.90	607.50	261.60	0.30	0.27	0.80
and		875.00	915.00	40.00	0.21	0.23	0.49
RC10-407	1053.69	4.00	60.00	56.00	0.39	0.20	0.92
and		4.00	1030.00	1026.00	0.22	0.20	0.97
including		77.50	135.00	57.50	0.70	0.31	1.14
including		214.30	271.00	56.70	0.57	0.22	4.63
including		500.00	540.00	40.00	0.32	0.49	0.59
including		591.30	732.50	141.20	0.29	0.39	1.02
including		770.60	785.00	14.40	0.36	0.35	0.53
including		865.00	1030.00	165.00	0.26	0.29	1.92
RC10-408	1206.09	206.50	235.00	28.50	0.26	0.12	0.38
and		302.50	370.00	67.50	0.24	0.15	0.62
and		502.50	650.00	147.50	0.32	0.33	0.98
and		687.50	754.40	66.90	0.37	0.59	1.33
and		781.20	1060.00	278.80	0.44	0.57	1.34
and		1072.50	1110.00	37.50	0.22	0.17	0.47
RC10-409	1361.54	297.50	1304.50	1007.00	0.30	0.30	1.44
RC10-410	1493.20	535.00	669.30	134.30	0.30	0.23	0.96
and		707.40	820.00	112.60	0.28	0.33	0.44
and		986.20	1000.00	13.80	0.17	0.32	1.12
and		1085.00	1115.00	30.00	0.17	0.58	1.09
RC10-411	1075.03	437.50	1047.50	610.00	0.43	0.41	1.27
RC10-412	1029.31	530.00	612.50	82.50	0.27	0.32	0.76
and		530.00	974.30	444.30	0.22	0.27	1.63
including		612.50	643.90	31.40	0.11	0.55	12.64
including		659.00	974.30	315.30	0.23	0.24	0.83

RC10-413	1468.22	45.00	532.20	487.20	0.49	0.40	1.73
including		395.00	500.00	105.00	0.84	0.77	2.76
RC10-414	1357.27	557.90	625.00	67.10	0.32	0.26	1.59
and		641.30	1130.40	489.10	0.50	0.66	2.33
including		792.50	813.40	20.90	1.20	1.65	3.57
including		907.50	957.50	50.00	0.62	1.03	2.47
and		1160.00	1185.00	25.00	0.29	0.26	0.89
and		1207.50	1254.70	47.20	0.21	0.24	1.37
and		1296.50	1357.50	61.00	0.30	0.14	0.94
RC10-414	1357.27	532.50	567.50	35.00	0.22	0.18	0.77
and		602.50	926.80	324.30	0.52	0.44	1.59
and		1075.00	1160.60	85.60	0.31	0.26	2.98
and		1178.20	1203.60	25.40	0.13	0.09	8.21
RC10-415	236.52	NSI					
RC10-416	1001.88	422.50	486.40	63.90	0.31	0.21	1.07
and		530.00	605.00	75.00	0.24	0.23	1.01
and		795.00	862.50	67.50	0.22	0.21	0.49
and		887.50	930.00	42.50	0.23	0.19	0.46
RC10-417	166.42	125.00	137.50	12.50	0.26	0.51	0.26
RC10-418	218.24	NSI					
RC10-419	1212.19	82.50	167.50	85.00	0.19	0.21	0.32
and		190.40	950.20	759.80	0.45	0.57	1.61
including		392.50	444.90	52.40	0.60	1.61	1.73
including		617.50	637.20	19.70	0.89	0.64	2.10
including		882.50	927.50	45.00	0.73	0.80	2.97
and		1033.90	1051.60	17.70	0.42	0.38	1.15
RC10-420	1001.88	5.20	125.00	119.80	0.38	0.13	0.45
and		300.00	322.50	22.50	0.40	0.28	0.70
and		345.00	355.00	10.00	0.25	0.29	0.47
and		412.50	945.00	532.50	0.22	0.25	0.73
RC10-421	1527.30	600.00	1240.00	640.00	0.57	0.67	1.03
including		695.00	815.00	120.00	0.88	1.44	1.49
including		950.00	985.00	35.00	0.72	0.81	1.21
RC10-422	1101.02	207.50	220.00	12.50	0.23	0.19	0.88
and		500.00	515.00	15.00	0.26	0.19	0.92
and		570.00	592.50	22.50	0.23	0.23	0.50
and		817.50	837.50	20.00	0.34	0.27	0.78

RC10-423	1257.91	385.00	730.90	345.90	0.68	0.94	2.16
including		500.00	515.00	15.00	1.05	0.82	3.52
including		537.50	557.50	20.00	0.70	1.15	2.32
including		627.50	730.90	103.40	0.97	1.88	3.47
including		697.50	712.50	15.00	2.07	5.11	7.03
and		820.40	1037.30	216.90	1.15	2.44	5.79
including		820.40	879.80	59.40	2.93	6.85	15.44
including		860.00	875.00	15.00	3.65	12.59	18.67
and		1050.60	1165.00	114.40	0.39	0.44	1.30
RC10-424	1001.88	NSI					
RC10-425	1550.52	122.50	239.00	116.50	0.36	0.42	0.82
and		302.50	1065.00	762.50	0.33	0.37	1.40
and		1197.50	1260.00	62.50	0.42	0.20	1.09
and		1457.50	1470.00	12.50	0.36	0.14	1.02
RC10-426	1513.30	814.90	975.00	160.10	0.74	0.76	1.49
including		835.00	847.50	12.50	1.46	1.82	1.67
including		865.00	890.00	25.00	1.08	1.03	2.20
and		982.50	995.00	12.50	0.21	0.17	0.47
and		1010.00	1042.50	32.50	0.24	0.20	0.61
and		1077.80	1117.50	39.70	0.23	0.18	0.98
and		1135.00	1147.50	12.50	0.28	0.25	0.83
RC10-427	1498.70	190.00	205.00	15.00	0.23	0.50	0.98
and		302.50	382.50	80.00	0.20	0.23	0.95
and		605.00	1044.10	439.10	0.29	0.27	1.20
and		1060.00	1079.60	19.60	0.17	0.39	4.37
and		1197.50	1209.10	11.60	0.13	2.11	25.99
RC10-428	1016.51	NSI					

11.5 Deep Exploration Drilling – 2011

The 2011 deep exploration diamond drilling program consisted of 12 completed holes totaling 12,685 metres (see Table 11.7 for collar locations). Note that RC10-425, 426, 427 and 428 were completed in 2011, reference section 11.4 for collar location and footage summary. Drilling began in the second week of January 2011 with three drills. Upon completing the grid delineation program in early spring, all three drills were sequentially demobilized in April and May. In early September a single deep drill was re-mobilized to test the Gully Zone for high grade mineralization at depth. Exploration drilling consisted of both HQ/NQ diameter core and was accomplished with the use of modified Boyles 56's and LF-230's. Collar locations and orientations are summarized in below.

Table 11.7 2010 Drill Intercepts

Drill Hole	Zone	Easting	Northing	Azimuth	Dip	Total Length (m)
RC11-429	Saddle	452212.24	6395987.97	0	-90	1251.81
RC11-430	East	452491.63	6396335.42	155	-65	1470.66
RC11-431	East	452965.96	6396012.64	0	-90	1312.78
RC11-432	Main	452296.58	6395703.40	65	-65	1189.78
RC11-441	East	452922.52	6395881.89	0	-90	1183.23
RC11-444	East	452681.88	6396032.90	0	-90	151.49
RC11-445	Saddle	452294.21	6395809.58	335	-76	1482.85
RC11-446	East	453016.45	6395935.09	0	-90	151.49
RC11-450	East	452390.95	6396075.37	0	-90	1243.89
RC11-477	Gully	450700.93	6394750.03	0	-90	1482.85
RC11-539	Gully	450605.06	6394931.12	155	-75	1263.40
RC11-564	Gully	450870.41	6394816.40	0	-90	501.40

Deep exploration drilling from January to May 2011 marked the conclusion of the deep delineation program beneath the open pit. The most notable exploration development was the discovery of high grade mineralization beneath the barren Stuhini sediments along the northern margins of the projected pit (see figure 11.3 for cross section). This was demonstrated with drill hole RC10-429 which intersected 402.5 metres of 0.37% copper, 0.42 g/t gold and 0.70 g/t silver, between 750.0 and 1152.5 metres, including a 70 metre interval of 0.71% copper, 0.91 g/t gold and 1.08 g/t silver.

In early September through to late November Imperial was successful in tracing high grade mineralization to depth in the Gully Zone. As with the East and Main Zones, the Gully Zone had only previously been drilled to depths of 400 metres, with many holes ending in mineralization and quartz veins. The first deep drill hole RC11-477 intersected 807.5 metres of 0.31% copper and 0.29 g/t gold, including a 101.4 metre interval of 0.58% copper and 0.45 g/t gold. The second drill hole RC11-539 scissored RC11-477 and confirmed the lateral continuity of mineralization at depth. RC11-539 intersected 587.1 metres of 0.41% copper and 0.41 g/t gold. High grade mineralization remains open in all directions at depth. The significant results of the 2011 drill holes are provided in Table 11.8.

Table 11.8 2010 Drill Intercepts

	Total	Metre Interval		Interval	Copper	Gold	Silver
Drill Hole	Length (m)	from	to	Length	%	g/t	g/t
RC11-429	1251.81	750.00	1152.50	402.50	0.37	0.42	0.70
including		797.50	867.50	70.00	0.71	0.91	1.08
and		1202.50	1251.80	49.30	0.23	0.21	0.69
RC11-430	1470.66	667.50	855.00	187.50	0.26	0.26	0.48
and		1064.60	1227.50	162.90	0.34	0.54	1.54
including		1067.50	1097.50	30.00	0.61	1.30	2.61
and		1247.50	1257.50	10.00	0.42	0.50	1.20
and		1282.50	1360.00	77.50	0.26	0.33	1.08
RC11-431	1312.78	172.50	1250.00	1077.50	0.44	0.29	0.84
RC11-432	1189.78	175.00	1144.90	969.90	0.40	0.54	2.07
including		595.00	615.00	20.00	1.01	1.85	5.57
including		1117.50	1142.50	25.00	0.80	1.36	15.15
RC11-441	1183.23	480.00	662.50	182.50	0.30	0.37	1.05
and		682.50	1005.00	322.50	0.30	0.21	1.72
RC11-444	151.49	NSI					
RC11-445	1482.85	177.50	237.50	60.00	0.25	0.21	0.39
and		252.50	287.50	35.00	0.28	0.23	0.57
and		337.50	1100.00	762.50	0.43	0.48	0.84
and		1175.00	1282.50	107.50	0.24	0.21	0.72
RC11-446	151.49	NSI					
RC11-450	1243.89	NSI					
RC11-477	1482.85	172.50	980.00	807.50	0.31	0.29	N/A
including		230.00	657.50	427.50	0.39	0.35	N/A
including		333.60	435.00	101.40	0.58	0.45	N/A
RC11-539	1263.40	275.40	862.50	587.10	0.41	0.41	N/A
including		347.50	362.50	15.00	0.66	1.01	N/A
and		917.30	952.50	35.20	0.27	0.25	N/A
RC11-564	501.40	18.30	422.50	404.20	0.39	0.39	N/A
including		18.30	282.30	264.00	0.47	0.51	N/A

Figure 11.4 2007 to 2011 Drillhole Collar Location Plan Map

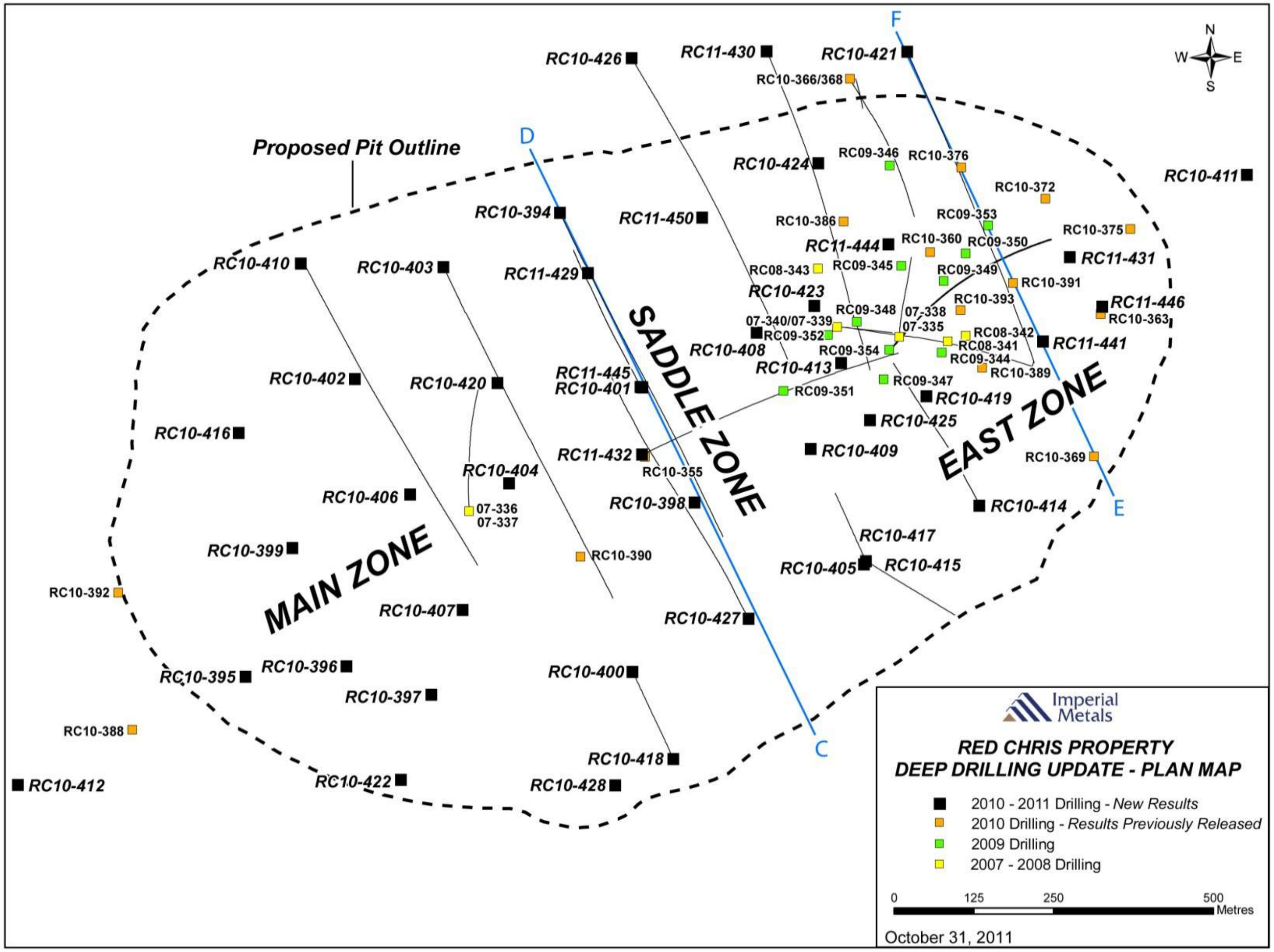


Figure 11.5 Red Chris Drilling Cross Section C-D: Showing Assay Highlights

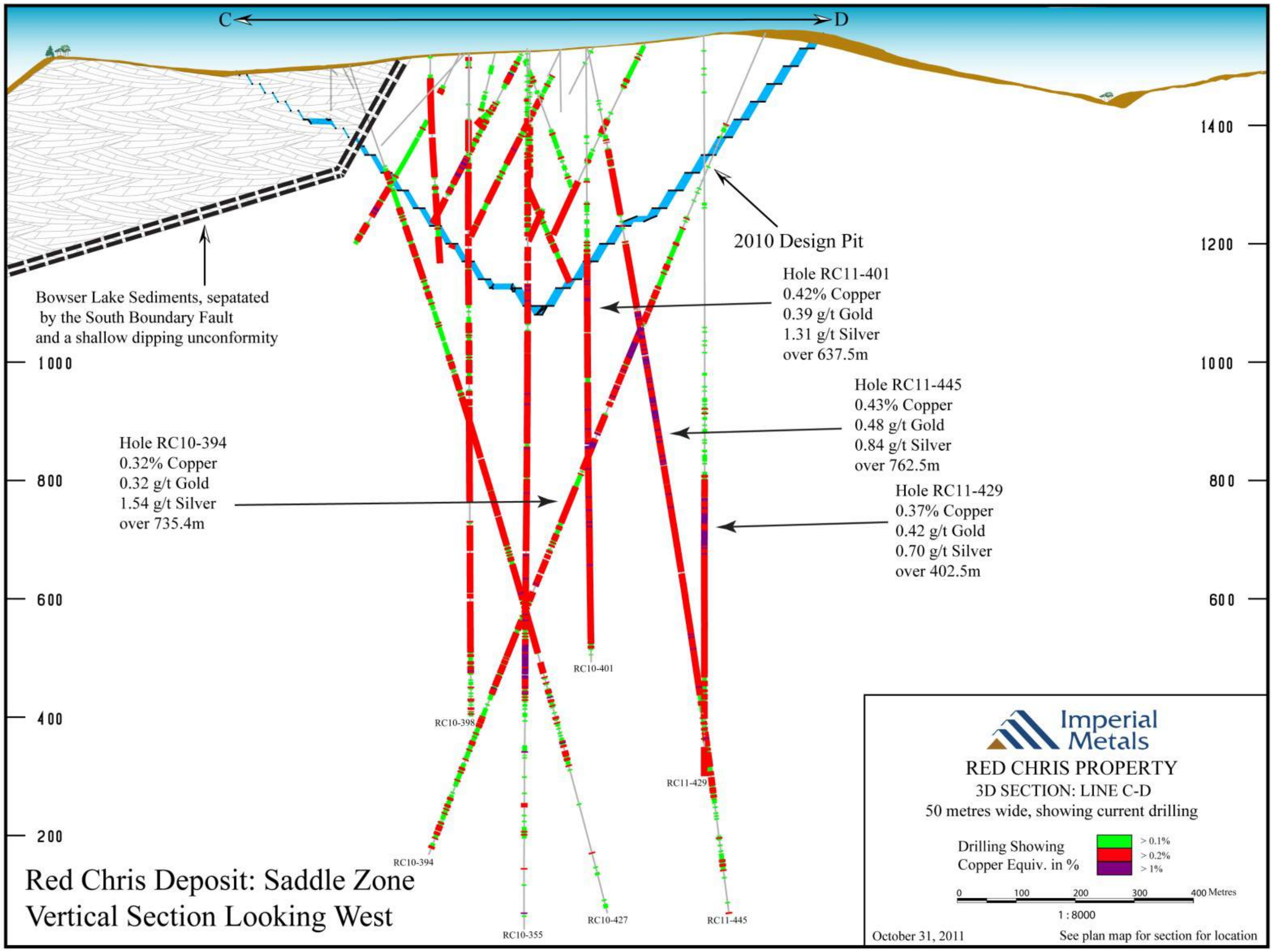
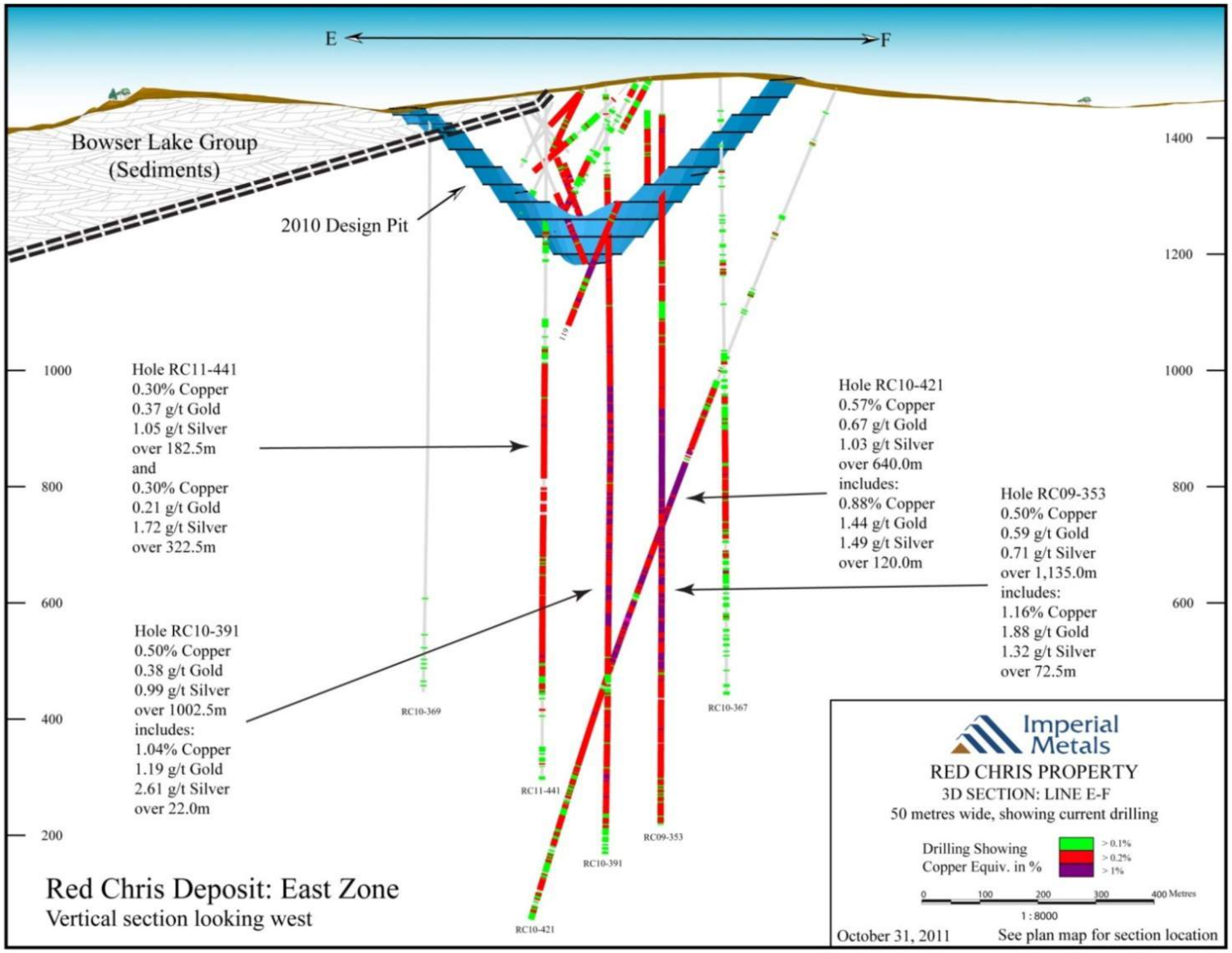


Figure 11.6 Red Chris Drilling Cross Section E-F: Showing Assay Highlights



11.6 Footprints/Bobcat Drilling Programs

In the summer of 2009 Imperial sponsored and participated in the University of British Columbia's (MDRU) and Oregon State University's Porphyry Footprint Project. The purpose of the project was to analyze and identify the regional geochemical signatures of large porphyry deposits. Aside from the geochemical investigation, the high density of drillholes also served to update and refine the local geology, which is masked by overburden. The project was concentrated on the plateau north of the pit outline and consisted of 133 diamond drill holes totaling 710 metres. The program was accomplished with the aid of a low impact Bobcat mounted Hydra Core drill (BQTK rods) and a track mounted Komatsu crawler carrier.

Due to the Bobcats ability to quickly survey the covered bedrock geology, Imperial implemented an in house Bobcat drill mapping program in the summer 2011. The mapping program was concentrated on the plateau west of the Gully Zone and consisted of 102 holes, totaling 870 metres.

Figure 11.7 Bobcat Mounted Hydra Core Drill

11.7 Geotechnical Drilling

Since acquiring the property in 2007, Imperial has had three main areas of geotechnical focus which are the tailing impoundment area, the plant site and the primary crusher. Cumulatively these areas have been investigated with 84 drill holes totaling 3,097 metres. Drilling has been conducted with the use of site specific drills, and to date has included a low impact Bobcat Mounted Hydra Core, a Boyles 37, a Boyles 56, and a Fraste MDXL drill. In all instances core and overburden samples were collected with the use of split spoons in order to obtain maximum recovery and to prevent any additional mechanical fracturing. See Table 11.9 for a list of drill collars, Figure 11.12 for map geotechnical sites, and Figures 11.8-11.11 for maps of site specific collar locations. AMEC was contracted in 2010 to manage the geotechnical operations, analysis and final reporting of the geotechnical drilling program.

Table 11.9 2010 Geotechnical Drillhole Coordinates

Drill Hole	Zone	Easting	Northing	Azimuth	Dip	Total Length (m)
09-B01	TIA	456143	6397920	0	-90	64.30
09-B02	TIA	456716	6398038	0	-90	53.60
09-C01	Crusher	452799	6396580	0	-90	24.50
09-C02	Crusher	453028	6396247	0	-90	18.60
09-P01	Plant Site	453097	6397027	0	-90	21.60
09-P02	Plant Site	453293	6397032	0	-90	20.10
09-P03	Plant Site	453142	6396870	0	-90	20.10
09-P04	Plant Site	453268	6396834	0	-90	17.10
BH-10-001	TIA	456090	6400815	270	-70	27.48
BH-10-002	TIA	456388	6400750	0	-90	45.10
BH-10-003/003-2	TIA	456605	6400720	0	-90	93.57
BH-10-004	TIA	456800	6400692	0	-90	53.34
BH-10-005	TIA	457087	6400693	110	-70	25.91
BH-10-006	TIA	456576	6400778	0	-90	68.58
BH-10-007	TIA	456720	6401059	0	-90	93.90
BH-10-008	TIA	456893	6401393	0	-90	50.29
BH10-01	Plant Site	454437	6399145	0	-90	35.66
BH10-02	Plant Site	454404	6398941	0	-90	32.61
BH10-03	Plant Site	454570	6398893	0	-90	33.83
BH10-04	Plant Site	454375	6398746	0	-90	33.53
BH10-05	Plant Site	454545	6398719	0	-90	32.61
BH10-06	Plant Site	454721	6398745	0	-90	34.14
BH10-07	Plant Site	454445	6398658	0	-90	34.14
BH10-08	Plant Site	454335	6398540	0	-90	35.66

BH10-09	Plant Site	454492	6398505	0	-90	35.66
BH-10-101	TIA	458655	6399564	0	-90	15.24
BH-10-102	TIA	458810	6399410	0	-90	41.20
BH-10-103/103A	TIA	458931	6399301	0	-90	18.29
BH-10-104	TIA	459034	6399490	0	-90	103.00
BH-10-201	TIA	455243	6397148	300	-70	38.10
BH-10-202	TIA	455354	6397095	0	-90	64.80
BH-10-203	TIA	455510	6397040	0	-90	110.34
BH-10-204	TIA	456048	6396846	0	-90	57.30
BH-10-205	TIA	456259	6396773	110	-70	42.60
BH-10-206/206-2	TIA	455777	6396698	0	-90	93.73
BH-10-207	TIA	455938	6396864	0	-90	48.77
BH-10-208	TIA	455915	6396874	0	-90	53.34
BH-10-301	TIA	456099	6399940	270	-70	35.51
BH-10-302	TIA	456099	6399940	0	-90	112.17
BH-10-303	TIA	456523	6399816	0	-90	103.63
BH-10-304/304A	TIA	456601	6400375	0	-90	67.10
BH-10-401	TIA	456209	6399600	0	-90	15.24
BH-10-402	TIA	456209	6399600	0	-90	15.24
BH-10-403	TIA	456500	6399426	0	-90	15.24
BH-10-404	TIA	457898	6398848	0	-90	15.24
BH-10-405	TIA	456122	6398422	0	-90	15.24
BH-10-406	TIA	457717	6397931	0	-90	15.24
BH-10-407	TIA	455257	6397524	0	-90	10.67
BH-10-408	TIA	456576	6397263	0	-90	15.24
RC11-433	Crusher	452829	6396735	0	-90	51.97
RC11-434	Crusher	452828	6396725	0	-90	51.82
RC11-435	Crusher	452819	6396731	0	-90	49.00
RC11-436	Plant Site	454470	6398735	0	-90	35.05
RC11-437	Plant Site	454462	6398729	0	-90	36.88
RC11-438	Plant Site	454462	6398746	0	-90	35.36
RC11-439	Plant Site	454454	6398740	0	-90	35.36
RC11-440	Plant Site	454435	6398710	0	-90	33.53
RC11-442	Plant Site	454429	6398705	0	-90	35.36
RC11-443	Crusher	453024	6396823	0	-90	35.36
RC11-447	Crusher	452967	6396885	0	-90	35.36
RC11-448	Crusher	452968	6396895	0	-90	35.97

RC11-449	Crusher	452959	6396892	0	-90	33.83
RC11-457	Crusher	452998	6396625	0	-90	14.33
RC11-458	Crusher	452990	6396610	0	-90	14.43
RC11-459	Crusher	452980	6396592	0	-90	12.41
RC11-460	Crusher	452972	6396571	0	-90	13.72
RC11-461	Crusher	452963	6396554	0	-90	16.15
RC11-462	Crusher	452954	6396536	0	-90	7.32
RC11-463	Crusher	452993	6396600	0	-90	39.62
RC11-464	Crusher	452983	6396596	0	-90	41.14
RC11-465	Crusher	452974	6396594	0	-90	42.67
RC11-567	Crusher	452985	6396593	0	-90	43.00
RC11-568	Crusher	452983	6396603	0	-90	41.50
RC11-569	Plant Site	454507	6398789	0	-90	35.00
RC11-570	Plant Site	454491	6398796	0	-90	35.00
RC11-571	Plant Site	454474	6398803	0	-90	35.00
RC11-572	Plant Site	454507	6399023	0	-90	21.00
RC11-573	Plant Site	454424	6398959	0	-90	21.00
RC11-574	Plant Site	454347	6398946	0	-90	21.00
RC11-575	Plant Site	454490	6398876	0	-90	19.00
RC11-576	Plant Site	454482	6398824	0	-90	19.00
RC11-577	Plant Site	454359	6399046	0	-90	19.00
RC11-578	Plant Site	454557	6398801	0	-90	18.00
RC11-579	Plant Site	454415	63988790	0	-90	19.00

Figure 11.8 Geotechnical Drilling

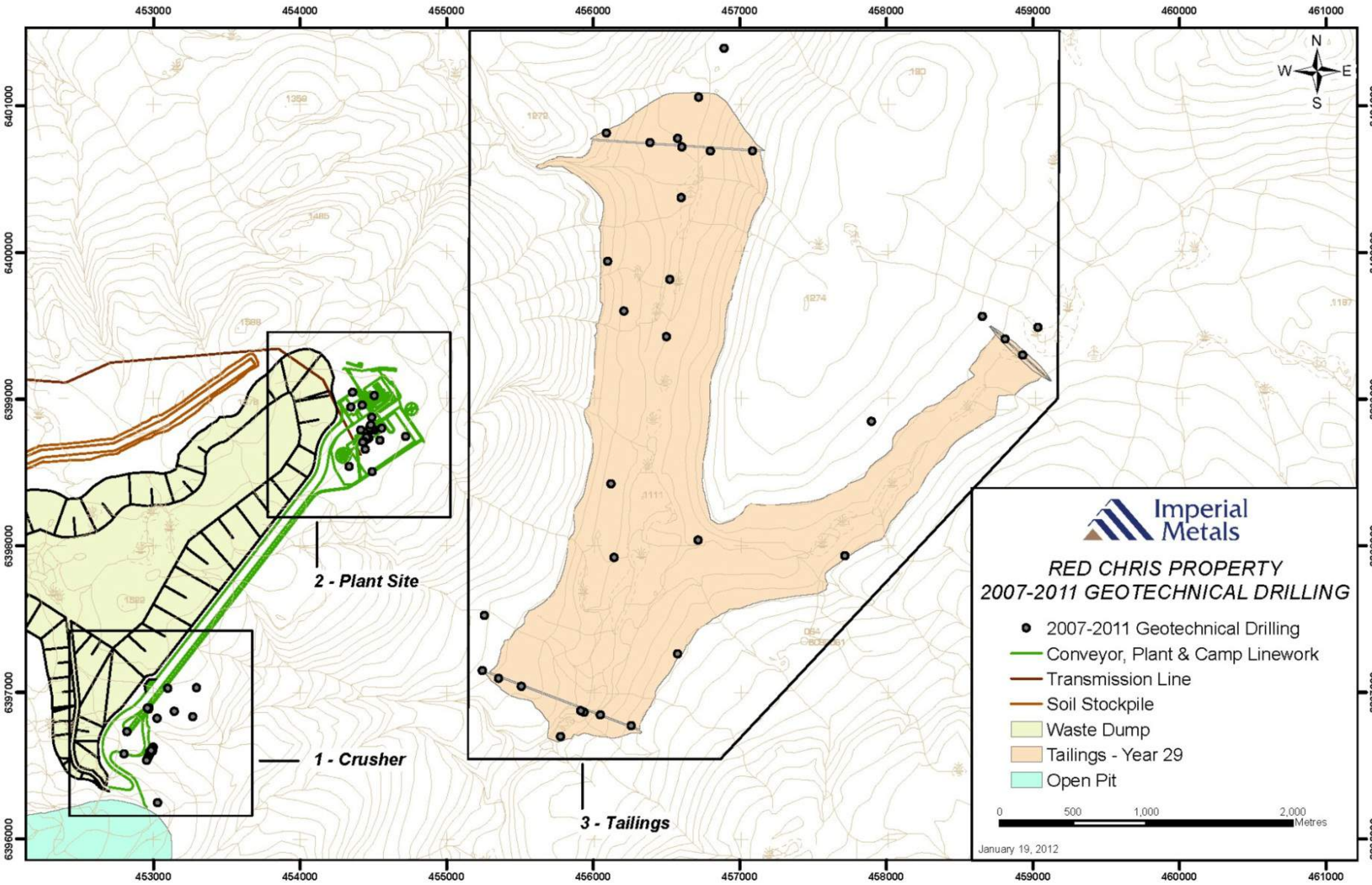


Figure 11.9 Plant Site Geology

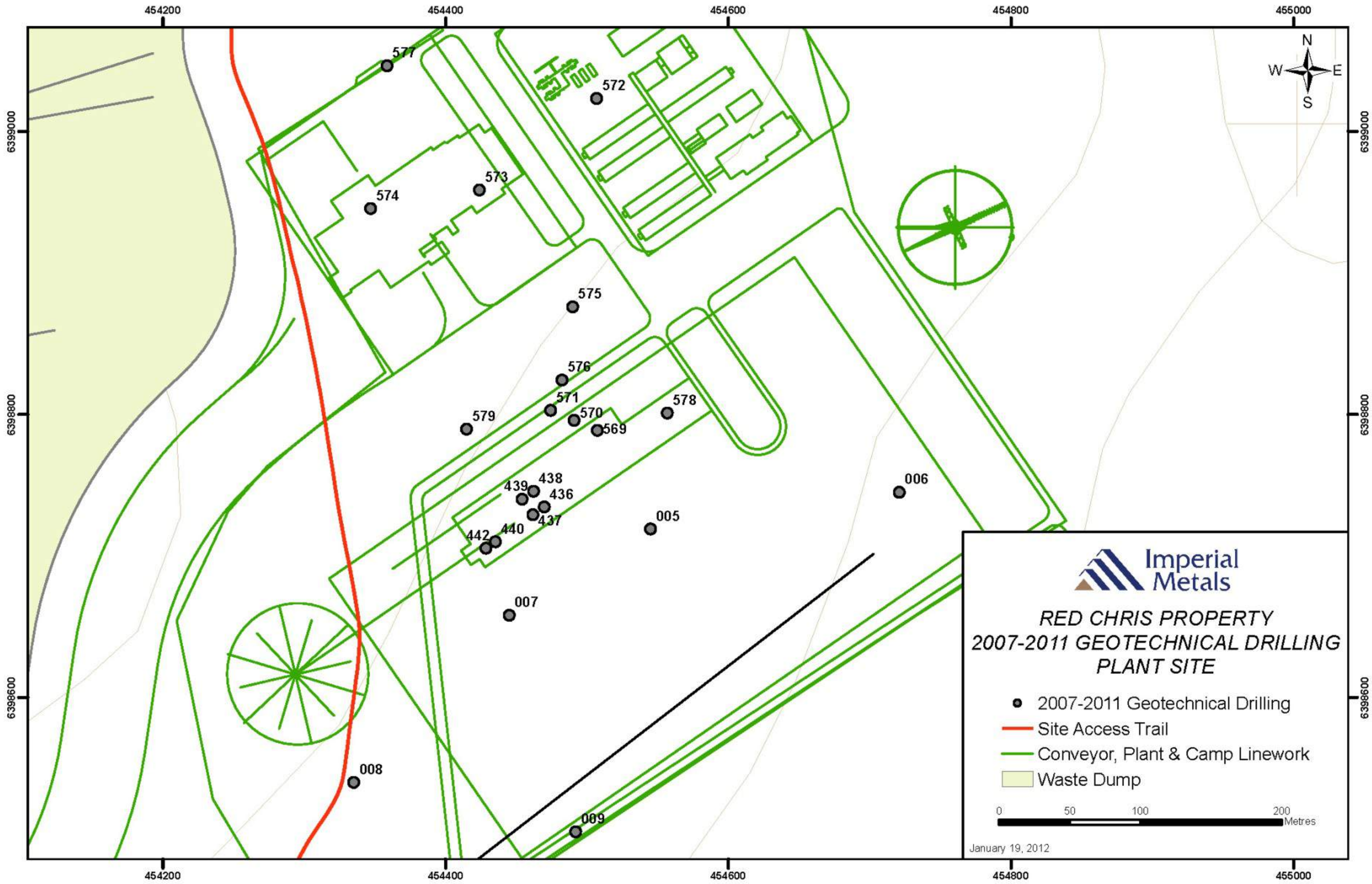


Figure 11.10 Geology for the Crusher Area

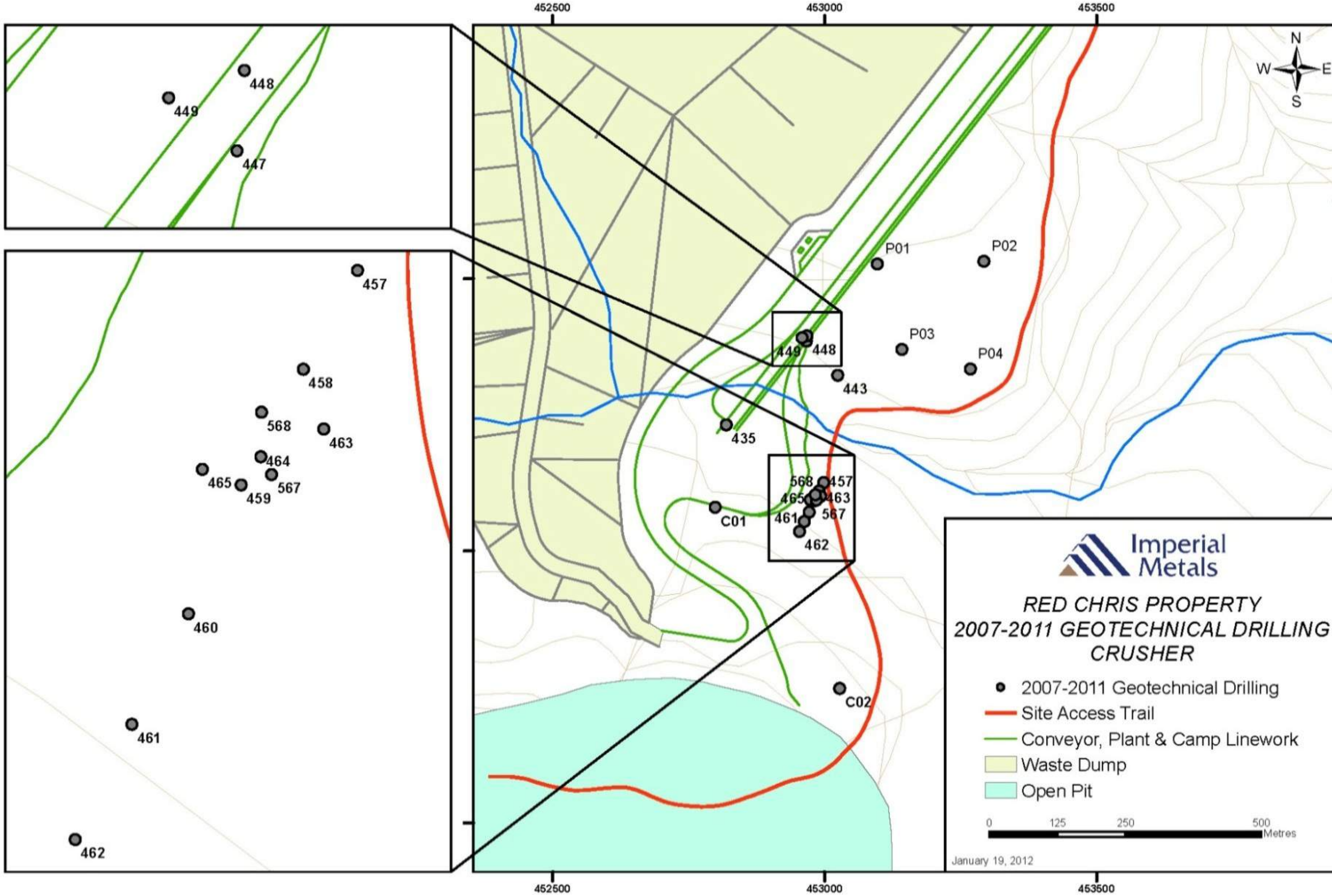
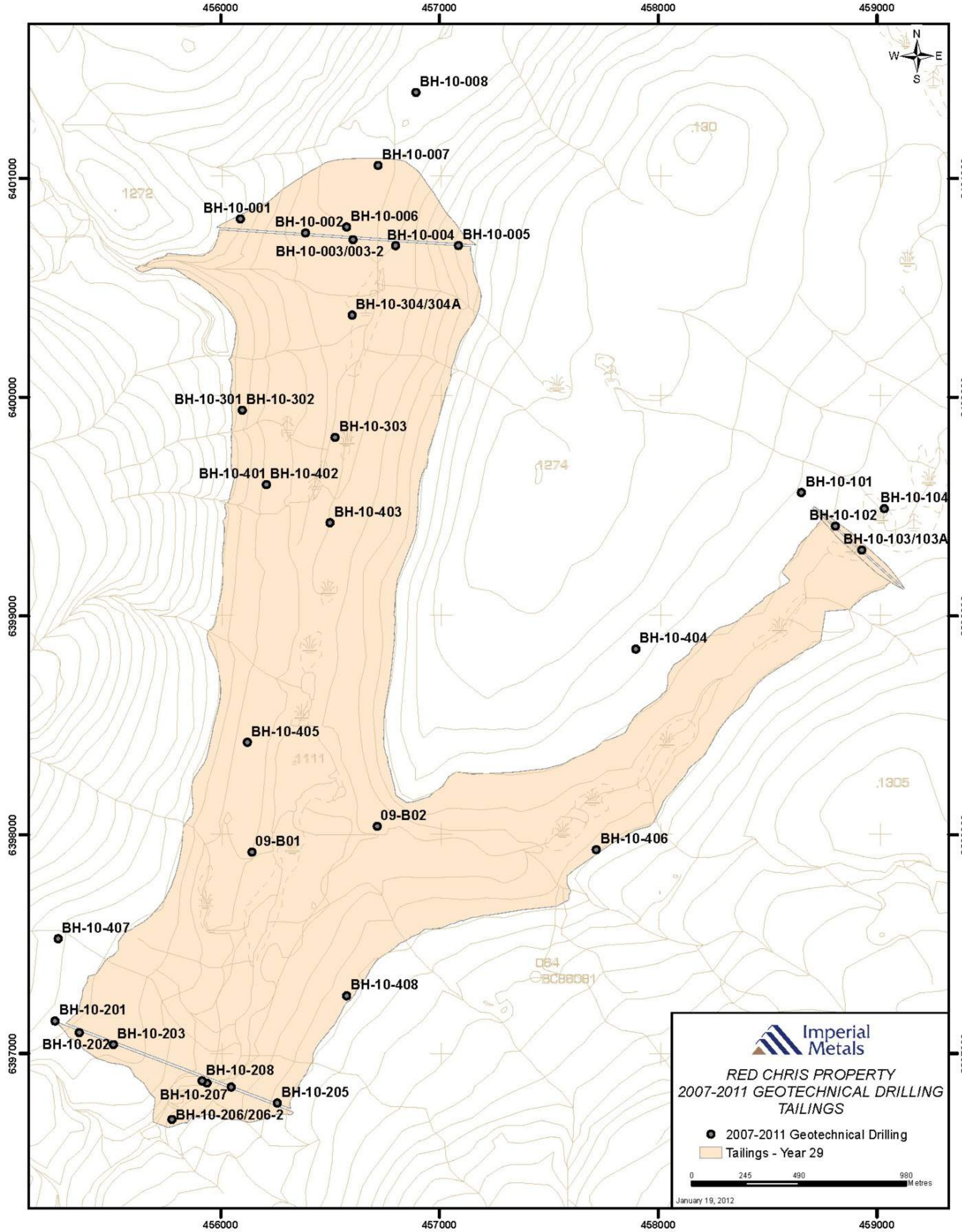


Figure 11.11 Tailings Impoundment Condemnation Collar Locations



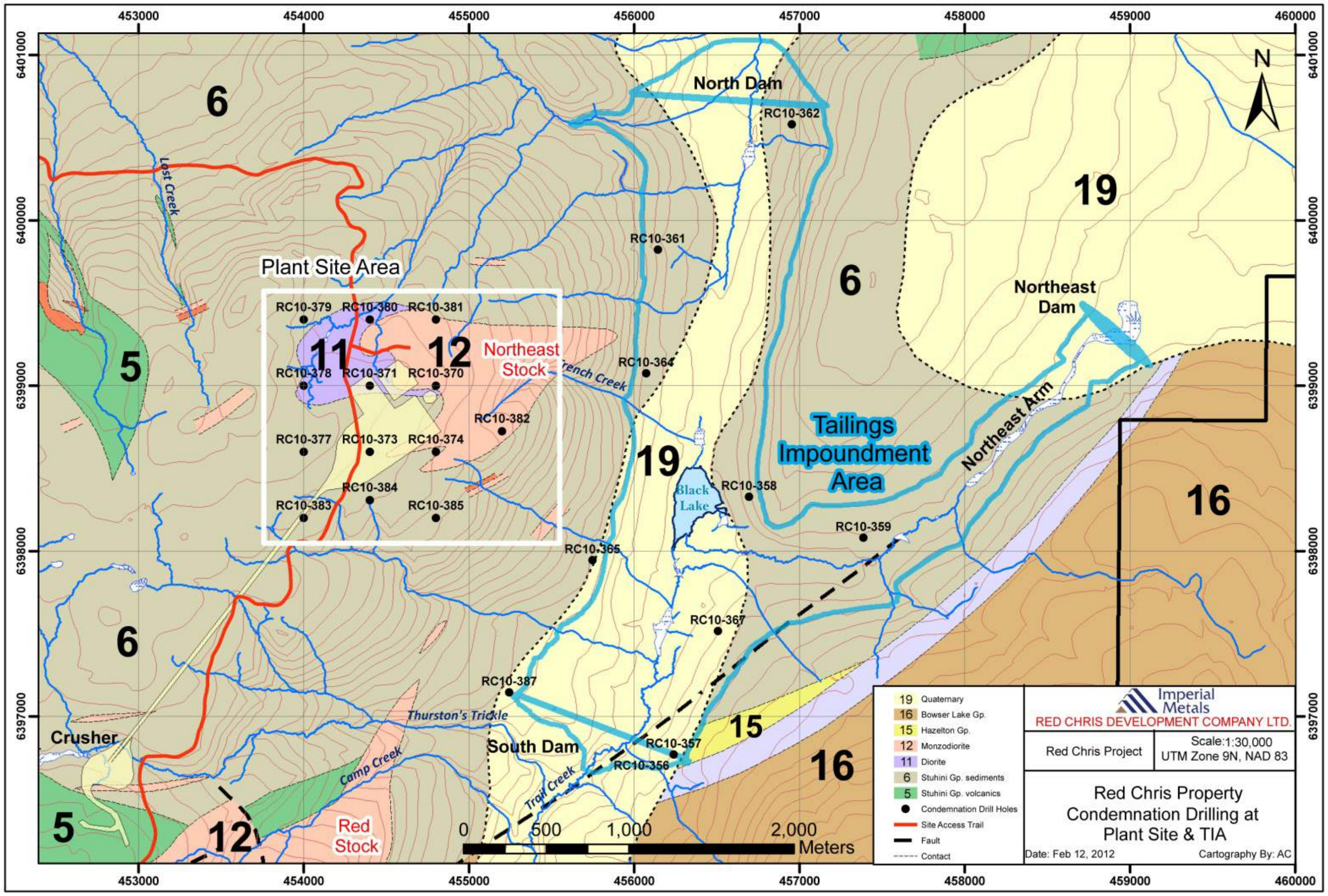
11.8 Condemnation Drilling

Condemnation holes were drilled in the plant site and TIA. The strategy was to test for copper and gold potential in the Northeast Stock, and in any peripheral dikes that might intrude the surrounding Stuhini Group. The plant site contains intrusive rocks of the Northeast stock, lithologically similar to the Red Stock which hosts the Red Chris ore body 4km to the southwest. However, the Northeast stock lacks the alteration intensity and vein characteristics associated with Red Chris mineralization, and holes completed in and around the stock failed to discover any meaningful evidence of mineral potential. In the TIA, nine holes were drilled in the main north-south valley, and one hole in the Northeast Arm of the TIA, totaling 3,556 metres of drilling. The TIA is underlain mainly by Stuhini Group sedimentary rocks, with a few dikes of intermediate composition representing the Late Triassic to Early Jurassic intrusive suite characteristic of the region. No samples showed visual evidence of mineralization and alteration is distinctly lacking in the drill core. The Northeast Stock does not appear to extend into the northern TIA, and the Red Stock does not appear to extend into the southern TIA.

Table 11.10 2010 Condemnation Drillhole Coordinates

Name	Zone	Easting	Northing	Elevation (m)	AZM	Dip	TD (m)
RC10-356	TIA	456238.00	6396770.00	1176.00	290.00	-45.00	559.92
RC10-357	TIA	456238.00	6396770.00	1176.00	80.00	-45.00	142.34
RC10-358	TIA	456694.00	6398328.00	1163.00	270.00	-45.00	352.65
RC10-359	TIA	457386.00	6398081.00	1125.00	130.00	-45.00	338.37
RC10-361	TIA	456143.00	6399823.00	1153.00	90.00	-45.00	317.79
RC10-362	TIA	456954.00	6400581.00	1115.00	270.00	-45.00	358.93
RC10-364	TIA	456072.00	6399075.00	1166.00	250.00	-45.00	620.88
RC10-365	TIA	455747.00	6397945.00	1199.00	110.00	-45.00	314.55
RC10-367	TIA	456507.00	6397517.00	1138.00	55.00	-50.00	153.01
RC10-387	TIA	455243.00	6397146.00	1190.00	290.00	-50.00	398.37
RC10-370	Plant	454800.00	6399000.00	1510.00	0.00	-90.00	99.67
RC10-371	Plant	454400.00	6399000.00	1490.00	0.00	-65.00	127.10
RC10-373	Plant	454400.00	6398600.00	1500.00	180.00	-65.00	124.05
RC10-374	Plant	454800.00	6398600.00	1461.00	0.00	-90.00	127.27
RC10-377	Plant	454000.00	6398600.00	1530.00	180.00	-65.00	127.10
RC10-378	Plant	454000.00	6399000.00	1516.00	0.00	-65.00	130.15
RC10-379	Plant	454000.00	6399400.00	1511.00	180.00	-65.00	127.10
RC10-380	Plant	454400.00	6399400.00	1480.00	180.00	-65.00	130.15
RC10-381	Plant	454800.00	6399400.00	1465.00	180.00	-65.00	127.43
RC10-382	Plant	455200.00	6398725.00	1437.00	0.00	-70.00	325.22
RC10-383	Plant	454000.00	6398200.00	1505.00	180.00	-65.00	121.01
RC10-384	Plant	454400.00	6398308.00	1480.00	180.00	-65.00	115.00
RC10-385	Plant	454800.00	6398200.00	1426.00	180.00	-65.00	130.15

Figure 11.12 Tailings Impoundment and Plant Site Condemnation Drill Collar Locations



12 Sampling methods and Approach

12.1 Drilling Core Handling Procedures

Drill core handling begins at the drill where the core tube is retrieved from the downhole origin of the drilling, by way of an overshot and wireline. The wireline is used to hoist the core tube to surface where the drill crew extracts the drillcore from the core tube. The drill core is placed into 1.3 metre long core boxes, laid out so that it is in the same order that it was retrieved in. The core placement reflects English reading with the top of the hole in the top left hand corner of the core box and the deepest core located in the bottom right hand corner of the core box. Permanent marker is used to label box with information on the hole number, footage and box number. Wooden blocks labeled with the appropriate metreage are placed between each drill run. Once full, the core box is secured for transport to the logging facility.

The Red Chris site has a dedicated facility for handing and logging drill core, known as 'the Core Shack' (see Figure 12.1). Once drillcore is received into the Core Shack, the core is washed and logged (geotechnically and geologically). Then the core is separated into 2.5m sample intervals by a geologist. The 2.5m sample length can be split in two, if needed to conform to a geological contact. Geotechnical data collected included core recovery, RQD, fracture counts, core strength, and overall ratings, with special attention paid to the occurrence of slickensides and fault gouge.

Each core box is permanently labeled with aluminum tags which show information reflecting hole name, box number and metreage at the top and bottom of the core stored in the box. Sample tags are stapled into the box at the beginning of each sample interval.

The core is logged with a KT-9 magnetic susceptibility meter over every sample interval. Ten susceptibility reading are taken for each sample, and then averaged. Geology data is recorded into Lagger (Northface Software), a database program designed for exploration drilling. Sample tags are placed at each sample contact by a geologist. Standards, duplicates, and blanks are randomly inserted within every 17 consecutive core samples.

The marked and tagged core is then photographed. Three additional whole rock samples are collected every 100m for geotechnical analysis. The first sample consists of a 10cm drill hole core plug which is sent to ACME for specific gravity analysis. The remaining two core samples are selected to test the axial (4cm NQ and 6cm HQ) and diametral (5cm NQ and 10cm HQ) strength of the rock. This is conducted onsite with a point load testing device.

Samples are returned to the core boxes after tested. The core is subsequently cut longitudinally using diamond bladed rock saws. Cut core is placed into clear poly-ore bags with the sample tag and zap-strapped. The other half-core is left in the core box, with the sample tag stub stapled to the start of the appropriate sample interval. Archived core is stored on-site in wooden racks. Sample bags are placed into white plastic rice bags, labeled, and zap-strapped with red numbered ties. The rice bags of samples are driven to Iskut and stored on pallets at the locked Bandstra Depot. Twice a week samples are shipped via Bandstra to ACME's preparation Lab in Smithers.

12.2 Down Hole Survey and Collar Coordinates

Downhole surveys were periodically conducted on the drillholes to measure their deviation. This is facilitated during bit changes and hole shutdowns by using a Reflex EZ-Trac downhole probe. Measurements were taken every 9.14m (three rods), with the probe suspended by aluminum running gear 7m beyond the drill-bit. The EZ-Trac is manufactured such that a handheld computer is synchronized to the probe, and measurements can quickly be obtained during the pulling of rods. Magnetic interference of the EZ-Trac is negligible at Red Chris due to the low amount of magnetite. Data recorded at each survey station included azimuth, dip, temperature, and magnetic field strength. Drillhole collars were surveyed with a handheld Garmin 60csx GPS, with accuracy to 3m.

Figure 12.1 The Red Chris Core shack

13 Sampling Preparation, Analyses and Security

13.1 Sampling Method, Approach and Security – 2007 to 2011

Drill core from the 2007, to 2011 drilling programs was processed, logged and sampled on site by Imperial Metals employees. Delivery of the core from the drill to the logging and sampling facility was via helicopter (2007 program) or truck (2008 to 2011 programs). Once the core was delivered to the core shack, it was treated as described in section 12.1.

During the 2007 and 2008 drilling programs, the samples were flown out in canvas mega-bags via helicopter to a staging area at Tatogga Lake Resort. Here, the samples were placed in a locked container by Imperial Metals employees. The samples were shipped approximately once a week via Canadian Freightways Ltd. to the ACME Analytical Laboratories preparation lab in Smithers, British Columbia. Beginning in 2009, the rice bags were driven by truck to Iskut by an Imperial Metals employee and hand-loaded onto pallets in a secure location at the Bandstra Trucking Depot. Twice a week, the samples were delivered via Bandstra Transportation Systems to ACME's preparation lab in Smithers.

ACME preparation lab personnel recorded the shipment number, arrival time and security tag numbers for each sample shipment. The samples were dried at 60°C before being crushed using a jaw crusher to 80% passing. Samples were then split and pulverized in a ring pulverizer to 85% minus 200 mesh, rolled and bagged. ACME then arranged for the pulp samples to be delivered to the main ACME Analytical Laboratory in Vancouver, British Columbia for assaying. The remaining coarse reject samples were bagged and labeled with the appropriate sample number and sent via commercial vehicle transport to Imperial Metal's storage facility at the Mount Polley Mine in Likely, British Columbia. ACME Analytical Laboratories Ltd. is an ISO 9001 registered analytical laboratory.

All samples were analyzed for gold, copper, iron and a 36-element geochemistry suite. Gold analysis was completed through fire assay fusion by ICP-ES (inductively coupled plasma) on a 30g sample. Copper and iron were analyzed by ICP-ES with an aqua-regia digestion. In addition, all samples were analyzed using ICP-MS with an aqua-regia digestion for a 36-element suite. The 36 elements analyzed in the ICP suite were: silver (Ag), aluminum (Al), arsenic (As), gold (Au), boron (B), barium (Ba), bismuth (Bi), calcium (Ca), cadmium (Cd), cobalt (Co), copper (Cu), chrome (Cr), iron (Fe), gallium (Ga), mercury (Hg), potassium (K), lanthanum (La), magnesium (Mg), manganese (Mn), molybdenum (Mo), sodium (Na), nickel (Ni), phosphorus (P), lead (Pb), sulphur (S), scandium (Sc), antimony (Sb), selenium (Se), strontium (Sr), thorium (Th), thallium (Tl), titanium (Ti), vanadium (V), zinc (Zn), tungsten (W). Rhenium (Re) was added to the suite part way through the 2010 drilling program.

Beginning in November 2011, core samples from the final stages of the 2011 drilling program were shipped via Bandstra Transportation Systems to Imperial Metals' Mount Polley mine laboratory for analysis. This was done in an attempt to decrease the turnaround time of the analytical results. A total of 1181 samples were processed at the Mount Polley exploration facility. Samples were dried to 60°C, crushed using a jaw crusher to 80% passing, split and pulverized. The remaining coarse rejects were bagged and labeled with the appropriate sample number and stored in wooden crates on site.

The pulp samples were analyzed for copper, gold and total iron at the Mount Polley mine laboratory. The copper and iron analyses were completed with a solution produced from 0.5 gram sample splits treated with aqua-regia digestion and diluted to 50 ml. Assay was by Atomic Absorption (AA). Gold was obtained by Fire Assay Fusion – AA finish on a 15g sample. Upon completion of the analyses, the pulp samples were packaged and delivered via Van-Kam Freightways Ltd. to ACME Analytical Laboratories Ltd. in Vancouver to be analyzed for a 36-element geochemistry suite. This analysis was performed using ICP-MS with an aqua-regia digestion.

14 Data Verification

14.1 Pre-2007 Data

Quality assurance and quality control ('QA/QC') programs began on the Red Chris Project during the 1994 drill program conducted by American Bullion and continued through to the 2003-2006 programs. The analytical quality of the 1994 and 1995 diamond drill programs were assessed by Barry Smee, Ph.D., P.Geo., of Smee and Associates Consulting Ltd. and presented in two separate reports (Smee, 1995 and Smee, 1996). During the 2003 drill campaign RCDC retained A.J. Sinclair, Ph.D., P.Eng. to evaluate the earlier work and comment on the 2003 QA/QC procedures and results. Both consultants had favourable conclusions on the QA/QC programs.

"*The analytical data for the Red Chris Project is well controlled. Standards prove that the data is accurate to within acceptable limits. Duplicates show a very small rotational bias between the two laboratories, with Min-En being slightly high on copper, and slightly low on gold, when compared with Chemex. However, the differences are not significant and do not impact the validity of analysis.*" Smee, (1995)

"The writer is in agreement with Smee that the accuracy and precision of the 1994 and 1995 Red Chris assay data for Cu and Au, obtained for American Bullion Minerals Ltd., meets quality levels that are generally acceptable throughout the mining industry. An exhaustive quality control program incorporated into the 2003 Red Chris drill program by RCDC demonstrates that the Cu and Au assay data obtained in 2003 is of acceptable accuracy and precision." Sinclair, (2004)

Mr. Sinclair was hired by Imperial in 2007 to update his 2004 report to reflect his review of the data up to the end of the 2006 program, when Imperial took over the QA/QC, and his conclusions were equally supportive of the analytical data received for that work.

"The quality control program implemented for the 2006 Red Chris drilling program has been successful in demonstrating that the quality of the data obtained are within limits widely accepted within the mining industry." Sinclair, (2007)

In the Author's opinion all the past drilling programs at the Red Chris Project have well documented procedures for quality assurance and quality control. Checks on standards in various grade ranges have shown acceptable accuracy at both the primary analytical laboratories. Blank samples reported low values at or near the detection limit indicating the absence of contamination of material during preparation. Duplicate pulps sent to second labs have shown no significant analytical bias. The analysis of 'blind' duplicates by the primary lab (IPL) has shown the data are unbiased and have a moderate level of random analytical error. Re-analysis of 2nd half cores have shown sampling variability to be random and as a result should be minimized during the resource estimation. In the author's opinion the pre-2007 assay data at Red Chris are both suitable and of the quality necessary to use in a Resource Estimate.

Collar location of all in pit and near pit pre-2007 exploration holes were resurveyed by Imperial personal in 2007/2008. A comparison with the original surveyed location showed no major discrepancies.

14.2 2007-2011 Quality Assurance and Quality Control Program

The QA/QC program from 2007 to 2011 involved the random placement of a duplicate (DUP), blank (BLK) and standard reference samples (STD) within every 17 consecutive core samples. Three standards were used to reflect low, medium and high grades. Table 14.1 summarizes the regular mainstream samples (MS) and QA/QC samples (duplicates, standards and blanks) analysed during the 2007, to 2011 drilling programs at Red Chris.

Table 14.1 QA/QC Sample Summary of Drilling Programs by Year

YEAR	MS	DUP	STD	BLK	TOTAL
2007	1939	114	114	114	2281
2008	917	54	53	54	1078
2009	4835	284	282	281	5682
2010	24891	1460	1465	1454	29270
2011	5741	336	341	339	6757
ALL	**38323**	**2248**	**2255**	**2242**	**45068**

The standard reference material used during the 2007 drilling program was prepared by CDN Labs of Surrey, British Columbia. The custom standards were originally prepared for use during Imperial Metals' 2007 Mount Polley exploration drilling program. The material used to prepare the standards was collected from the remaining rejects from the 2005 and 2006 drilling programs at Mount Polley. The rejects were selected based on assay intervals that would yield a low, medium and high copper and gold reference assay. CDN Labs prepared and packaged three homogeneous standards for use as assay standard reference material. The standards were certified by Barry Smee, Ph.D., P.Geo. of Smee and Associates Consulting Ltd.

The 2008 to 2009 drilling programs utilized the same low grade standard (MP06LG) used in the 2007 drilling program. Pre-processed and certified medium and high grade copper/gold standards were purchased from CDN Labs. The pre-made standards purchased from CDN Labs were also certified by Barry Smee, Ph.D., P.Geo. of Smee and Associates Consulting Ltd.

Standard reference material used during the 2010 and 2011 drilling programs consisted of custom-made and pre-processed standards prepared by CDN Labs. Both the custom standards and pre-processed standards were certified by Barry Smee, Ph.D., P.Geo. of Smee and Associates Consulting Ltd. Two batches of custom standard material were used. One batch was prepared in 2009 using the coarse reject material from the 2007 and 2008 drilling programs at Mount Polley. Three homogeneous standards (high, medium and low grades) were prepared and packaged for use as standard reference material. In early 2010, CDN Labs prepared and packaged three homogeneous standards (high, medium and low grades) using coarse reject material from the 2009 Red Chris drilling program. These standards were inserted into the sampling stream in the fall of 2010.

Copper and gold assays for the standard reference material were monitored for bias and precision. Standards met QA/QC requirements if the assayed values were within 3 standard deviations of the mean calculated standard value, as stated in the reference material certification. To monitor for bias, any two consecutive standard assay values could not be above two standard deviations on the same side of the mean calculated standard value. Failure to meet these requirements resulted in a re-assay of the failed standard, along with at least ten sequential samples above and below that standard. Re-assays were generally completed within three weeks of receipt of the original certificate.

Duplicate samples, taken to measure the precision of analysis, were randomly inserted into the sampling sequence within 17 consecutive core samples. These samples were made from quartering the half-core sample at the time of cutting or splitting. The assay results for the duplicate samples met QA/QC requirements if they assayed within an acceptable limit of +/-20%. Samples failing to meet this requirement were investigated and re-assayed, along with at least ten sequential samples above and below the failed sample.

Blank samples were randomly inserted into the sampling stream by Imperial geologists. The blank material consisted of crushed rock from a highways gravel pit located along the Likely highway in the central interior of BC. This material was bagged in poly ore bags in one kilogram samples. If a blank sample returned copper and gold assay values over a pre-determined threshold (0.05% Cu and 0.05g/t Au), the blank reject along with at least ten sequential core rejects (above and below the blank) would be re-processed and re-assayed. If the re-processed reject failed to meet the QA/QC requirements, the half-core was quartered and new samples in the affected range were re-submitted to the lab for processing and assaying. The majority of the blank samples assayed at or below the detection limit. Out of the 449 blank samples assayed between 2007 and 2009, 5 samples failed to meet QA/QC requirements. Out of 1793 blank samples assayed between 2010 and 2011, 13 samples failed to meet QA/QC requirements. Each case was investigated and if necessary, the affected range of samples was re-processed and re-submitted to the lab for assaying.

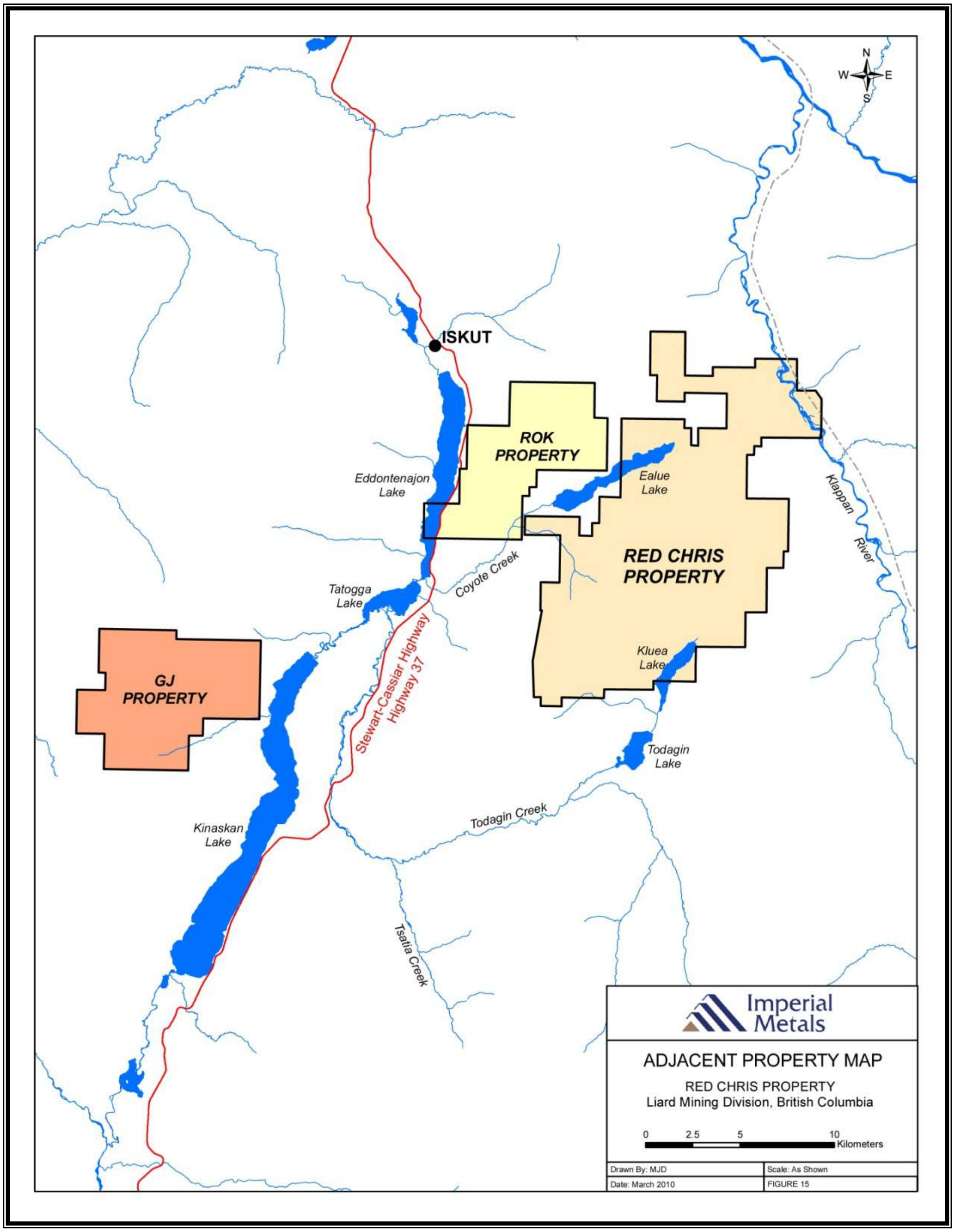
15 Adjacent Properties

In the immediate area of the Red Chris deposit the BC government mineral inventory database (Minfile) records a few mineral showings, most of which are low level geochemical anomalies or small showings with limited tonnage potential. Immediately to the west is the Gin Property where historical work was conducted to look for Eskay Creek style targets due to the presence of prospective stratigraphy, but no mineralization has been identified. To the east, the Eldorado/Bonanza property has received similar grassroots style exploration for porphyry targets, no significant mineralization has been identified. To the south, work was completed on the "Red Chris South" project to the southwest of Red Chris in 2011 but no results of significance have been released.

The two properties of significance in the area are the GJ owned by NGEX Resources and Rok which is owned by Firesteel Resources Inc. The GJ property is located on the southern end of the Klastline Plateau, over 30 km to the west of Red Chris. The property is underlain by Stuhini Group, and overlain by Lower Jurassic, Hazelton Group. Intruding the volcano-sedimentary sequence are numerous small plugs and sills of diorite to monzonite composition. The largest of these is the Groat Stock which hosts porphyry copper-gold mineralization in at least four areas; the Donnelly, North Donnelly, GJ and North zones. GJ has a 43-101 compliant measured and indicated resource, at a cut-off of 0.20% copper, of 153.3 million tonnes grading 0.321% copper and 0.369 g/t gold. An additional inferred resource, at a cut-off of 0.20% copper, is 23 million tonnes grading 0.26% copper and 0.31 g/t gold (NGEX Resources website).

The Rok property is north of Coyote Creek, to the northwest of Red Chris, with a relatively small area of mineralization exposed at surface. Much of the prospective area is covered by a thick package of Toodogonne volcanics thought to be younger than the age of mineralization emplacement, so the extent of the copper-gold mineralization is not known. The mineralization observed occurs in structurally-controlled quartz-vein stockworks, however there is no evidence of a large intrusive body to host significant porphyry style deposit. A blind and untested magnetic anomaly in the valley bottom provides encouragement that a large hydrothermal system may be present (see Figure 15.1 for locations).

Figure 15.1 Red Chris Adjacent Property Map



16 Mineral Processing, Recovery and Metallurgical Testing

Mineral processing test work conducted by Lakefield Research Limited in 1995 and 1996 and earlier work by G&T Metallurgical Services Ltd., indicated the Red Chris deposit responds well to processing by conventional crushing, grinding and flotation to produce a commercial grade copper-gold concentrate.

These earlier programs formed the basis for the 2005 Feasibility Study program conducted on newly acquired drill core from the 2003 exploration program. A comprehensive metallurgical test program was completed for the 2005 Feasibility Study.

The test work was done principally at Lakefield and G&T Metallurgical Services, utilizing both laboratory and pilot scale tests. The tests were conducted in logical sequence to determine an acceptable commercially viable process. These test are summarized in section 16.1. Results from this work form the basis for the plant design and performance estimates.

In January, 2010 Imperial Metals contracted G&T to initiate a scoping level study to examine the metallurgical characteristic of the deeper areas of the East Zone. This study used core from the 2008/2009 drilling programs.

16.1 2005 Feasibility Study Test work

The 2005 Feasibility Study comprehensive test work programs was performed at G&T using fresh drill core from the 2003 exploration program. This material was selected to be representative of the material to be mined in the open pit contemplated in the Feasibility Study. The G&T metallurgical program was carried out in following three phases:

- Phase 1: Flowsheet Development;
- Phase 2: Recovery Variability; and
- Phase 3: Pilot Plant Concentrate Production

The results of the G&T 2004 metallurgical programs were used as the basis for the design and consumption parameters for the Red Chris concentrator. The G&T 2004 programs confirmed previous test work that the mill flow sheet for Red Chris would utilize conventional processing techniques for a porphyry copper-gold flotation plant using:

- SAG and ball mill grinding to produce a nominal 150µ product to rougher flotation;
- Rougher flotation with an 18 minute retention time;
- Regrind of the rougher flotation concentrate to a nominal 25µ product to cleaner flotation;
- Two stages of cleaner flotation to produce a final copper-gold concentrate; and
- Thickening and dewatering of the final copper-gold concentrate for transport to off-site smelting facilities.

The proposed flow sheet is expected to produce an average 27% copper concentrate at a copper recovery of 87%. Test work indicates that the gold will track the chalcopyrite, pyrite and gangue in near equal portions throughout the process resulting in a gold concentrate ranging from 5g/t to 25g/t of concentrate at an average recovery of 50%.

16.1.1 Sample Selection and Metallurgical Composites

Drill core from the 2003 exploration program was used for all 2004 metallurgical testing. A ¼ split of selected sections from 23 holes was assembled into composites representing possible mining sequences.

Three holes were drilled with HQ coring, #256, #256A and #283, for the purpose of obtaining a larger volume sample in each of the Main and East Zones. The remaining core from holes 256A and 283 was used for grinding and work index studies.

The samples selected for metallurgical testing cover the majority of production within the East and Main ore zones with emphasis spatially on zones in the first nine years of the seventeen year mining production phase. The original sampling strategy was based on a mine plan and production level that has been superseded by the one forming the basis of this technical report. Additional sampling and metallurgical testing may be required to more fully characterize the metallurgical response of the entire deposit and link it to the current mine plan. Over the mine life, the East Zone will account for about 27% of mine production, but will average about 40% in the first six years of operation.

Portions of the drill core were blended to make up a series of composite samples for metallurgical testing. Composites MZ-1, -2, -3 and EZ-1, -2, -3, -4 represent horizontal layers within each zone that would be mined sequentially. Composite EZ-5 was a high grade sample prepared to study the effect of metallurgy at higher grades. Composites MZ-4, MZ-5 and EZ-6 are lower grade, representing material that will be stockpiled during the seventeen year mining phase and processed at a later date in years 18 through 25. Table 16.1 lists actual composite head grades used in the metallurgical testing. Over the mine life the average copper and grades of Main and East zones are expected to average about 0.42% and gold 0.30-0.39g/t respectively. In the first nine years of production the Main zone copper grade will average about 0.46% and the average grade of the Main zone metallurgical composite is about 0.47%. The average grade of the East Zone composite and global sample is relatively higher than the current production plan. Additional sampling and metallurgical work has been recommended to test grades closer to the mine plan for this zone.

Table 16.1 Individual Metallurgical Composite Head Grades

Composite Designation	Zone	Description	% Cu	Au g/t
MZ-1	Main	1410m – Surface	0.55	0.22
MZ-2	Main	1320-1410m	0.54	0.45
MZ-3	Main	1230-1320m	0.61	0.54
MZ-4	Main	Low Grade	0.28	0.18
MZ-5	Main	Medium Grade	0.36	0.25
EZ-1	East	1425m – Surface	0.84	0.53
EZ-4	East	1335-1425m	0.68	0.52
EZ-3	East	1245-1335m	0.58	0.57
EZ-2	East	1140-1245m	0.82	0.79
EZ-5	East	High Grade	1.24	1.19
EZ-6	East	Low Grade	0.32	0.30

Two additional composites for metallurgical testing were prepared. The MZ-Global comprised a blend of equal portions by weight of MZ-1, MZ-2, MZ-3 and MZ-5. The EZ-Global comprised a blend of equal portions by weight of EZ-1, EZ-2, EZ-3 and EZ-4. Table 16.2 lists actual composite head grades used in the metallurgical testing.

Table 16.2 Global Metallurgical Composite Head Grades

Composite Designation	Zone	Description	% Cu	Au g/t
MZ-Global	Main	MZ-1,MZ-2, MZ-3, MZ-5	0.50	0.42
EZ-Global	East	EZ-1, EZ-2, EZ-3, EZ-4	0.74	0.69

16.2 Mineralogy

Chalcopyrite and lesser bornite are the principal copper sulphide minerals in the Red Chris deposit. Minor covellite occurs as inclusions in pyrite, and molybdenite, sphalerite and galena occur locally in trace amounts. Gold, second in economic importance to copper, occurs as native and electrum, genetically-associated with the copper and pyrite mineralization

The Main Zone mineralogy consists predominantly of chalcopyrite and pyrite with an average pyrite: chalcopyrite ratio of 10:1. The East Zone mineralogy has an average pyrite: chalcopyrite ratio of 4:1, with significant amounts of bornite present. The non-sulphide gangue minerals include a mixture of sericite, quartz, ankerite, dolomite, illite and magnesite. There are no oxide copper minerals present in the material tested.

Gold occurrence is higher in the East Zone, and relative to the Main Zone, it is more dominantly associated with copper sulphides than pyrite. While the basic mineralogy of occurrence is similar throughout both zones this results in better gold recoveries in East Zone. Table 16.3 lists the mineralogy of the metallurgical composites.

Table 16.3 Metallurgical Composite Mineralogy

	Percent Mineral Content (by weight)				Pyrite/Chalcopyrite
Composite	Chalcopyrite	Pyrite	Bornite	Non-sulphide Gangue	Ratio
MZ-1	1.6	12.0	-	86.4	7.5
MZ-2	1.5	14.2	-	84.3	9.5
MZ-3	1.7	14.0	-	84.4	8.2
MZ-4	0.7	11.1	-	88.1	15.9
MZ-5	1.0	13.3	-	85.7	13.3
MZ-Global	1.5	13.4	-	85.2	8.9
EZ-1	2.0	10.1	0.2	87.6	5.1
EZ-4	1.8	7.9	0.1	90.2	4.4
EZ-3	0.8	2.1	0.5	96.6	2.6
EZ-2	0.8	3.3	0.8	95.0	4.1
EZ-5	2.6	8.9	0.5	88.0	3.4
EZ-6	0.8	8.5	<0.1	90.7	10.6
EZ-Global	1.4	5.9	0.4	92.4	4.2

16.2.1 2004 Metallurgical Test Program

The program was guided the following series of objectives:

- Study the mineral composition and fragmentation characteristics of several ore composites from the Main and East Zones, representing material to be processed during the early years of the operation.
- Devise a set of common treatment parameters for processing the Red Chris ore types, including flotation feed sizing, regrind product sizing, reagent regime and flowsheet configurations.
- Conduct a series of work index tests to determine ore hardness variation for mill sizing and power requirements.
- Conduct modal assessments on groups of cycle test products to determine if further enhancements in metallurgical performance of the ores would be technically feasible.
- Assess the concentrate quality with regard to mineral composition and minor element concentrations.
- Using optimum treatment parameters, perform a series of standard tests on a variety of samples throughout the deposit to determine the variation in expected metallurgy.

16.2.2 Mineral Liberation Characteristics

A primary grind of 150 micron K_{80} was determined as optimum for the Red Chris ore. At this feed sizing, 50% of the chalcopyrite and bornite particles are liberated, along with 90% pyrite liberation and 95% non-sulphide gangue minerals. These liberation figures are within the typical range of standard industry practice. The average liberation of minerals in the flotation feed of twenty two porphyry copper-gold deposits in G&T's data base at a primary grind size of 185 microns K_{80} was 55% for copper sulphides, 65% for pyrite and 92% for non-sulphide gangue minerals.

The Red Chris ore is finer grained when compared with many porphyry copper-gold deposits, and as such will require a slightly finer grind size. This is also evident in the 24 micron regrind size selected for the flotation cleaner circuit feed. A primary grind of 105 micron K_{80} increased the recovery, however, this is more than off-set by an additional 3.5MW of power required for grinding or a reduction in throughput. A simple economic study confirmed this, using typical power costs and a net smelter revenue value for copper of 90 cents. Testing at a coarser 200 micron K_{80} resulted in a 5% recovery loss. The 150 micron K_{80} was considered the optimum level for this application. However, the economic primary grind selection could be lowered as copper prices have improved significantly since the feasibility report was written.

16.2.3 Rougher and Cleaner Flotation

A total of 129 bench tests followed by 31 locked cycle flotation tests were conducted studying pulp density, pH, reagent dosage, flotation residence time and grind size effect. Table 16.4 lists the operating parameters required to achieve optimum performance. Although the mineralogy between the two zones is different, similar operating parameters will be used with no loss in performance.

Table 16.4 Flotation Parameters

pH	Lime	Roughers	10.5
		Cleaners	12
Collector	Potassium Amyl Xanthate	Roughers	0.006 kg/t
		Cleaners	0.005 kg/t
Frother	MIBC		As req'd
Flotation Time	Lab Time	Roughers	9 min.
		Cleaners	9-11 min.
Flotation Density		Roughers	33-35% solids

Typical operating practice in many porphyry copper operations is to produce a 10 – 15% Cu rougher concentrate by pulling 4 – 5% mass. Current testwork indicates the Red Chris ore requires a 15 – 20 % mass pull resulting in a rougher concentrate grade of 3% Cu in order to achieve the same recovery. The regrind circuit power has been specified based on a design weight recovery of 15%. A series of regrinding tests were conducted to investigate the possibility of preferential or distributed regrinding power, and the benefit of pulp conditioning. The tests involved by-passing part of the higher grade rougher concentrate, pre-cleaning prior to regrinding and classification prior to regrinding.

None of the modified circuits produced any improvement in results over those of the conventional regrinding circuit. Testing indicated regrinding to a relatively fine liberation sizing of K_{80} = 24 microns was potentially required to obtain optimum final grade and recovery, but this is mainly based on a consideration of Main Zone metallurgy. More testwork was recommended by G&T to optimize regrind size selection and power. The duplicate tests listed in the table 16.5 indicate a slightly higher concentrate grade was achieved using the modified circuit. However, when the higher rougher concentrate feed grade effect is taken into account, the results are very similar.

Table 16.5 Regrind Circuit versus Metallurgy

		Rough Con	Cleaner Circuit		
Test	Sample	% Cu	% Cu	% Cu Rec	% Au Rec
1522-21	Modified RG Circuit	15.4	27.4	92.6	65.1
1522-22	Modified RD Circuit	17	25	91.8	57.5
1522-29	Conventional RG Circuit	14.8	24.1	91.4	60.6
1522-30	Conventional RG Circuit	15.3	25.4	92.5	62.7

16.2.4 Gold Occurrence

Gold occurs with both the copper sulphides and pyrite. Very little is associated with the non-sulphide gangue minerals as illustrated in the Gold Recovery Partition Table 16.6. The differences recorded in gold recoveries for the East and Main Zone composites are entirely attributable to a much larger proportion of gold tracking the pyrite in the Main Zone composites.

Table 16.6 Gold Recovery Partition

Composite	Gold Recovery Partition Coefficients			Statistic
	Cu Sulphides	Pyrite	Total	R^2
MZ-1	0.45	0.52	0.97	0.98
MZ-2	0.54	0.43	0.97	0.98
MZ-3	0.57	0.41	0.98	0.98
MZ-4	0.36	0.62	0.98	0.98
MZ-5	0.47	0.53	1	0.99
EZ-1	0.69	0.29	0.98	0.99
EZ-4	0.66	0.32	0.98	0.99
EZ-3	0.73	0.25	0.98	0.99
EZ-2	0.75	0.23	0.98	0.99
EZ-5	0.77	0.22	0.99	0.99
EZ-6	0.56	0.41	0.97	0.99

16.2.5 Gravity Concentration Tests

Gravity concentration testwork was performed on global composites for both ore zones using a Knelson concentrator to examine the potential for gold recovery. This test work resulted in minimal gold recovery and no further work was performed and the inclusion of a gravity stage was not recommended in the flow sheet.

16.2.6 Work Index and Ore Hardness

A total of 64 ball mill / rod mill grinding work index tests were performed by five separate laboratories. G&T Metallurgical Engineers performed 49 of the tests using both the Bond Work Index ("WI") method and the Comparative Work Index method. Four tests were conducted to confirm the validity of the Comparative Work Index method. On a Main Zone composite the Bond WI was 14.0 kW-hrs/tonne compared with 13.5 kW-hrs/tonne using the Comparative WI method. Similarly, on an East Zone composite, the Bond WI was 14.1 kW-hrs/tonne compared with 14.2 kW-hrs/tonne using the Comparative WI method.

Thirty three samples were tested in the Geometallurgical Ore Mapping Program designed to establish the metallurgical variation within the two zones. All samples were subjected to the Comparative Work Index test. The work index of the Main Zone averaged 14.8 kW-hrs/tonne with a range of 11.5 – 16.6 kW-hrs/tonne. The East Zone averaged 16.4 kW-hrs/tonne with a range of 10.8 – 22.1 kW-hrs/tonne.

During the first year of mining, the weighted calculation of 60% Main Zone and 40% East Zone provides a work index of 14.0 kW-hrs/tonne. In year 2, the average work index increases slightly to 14.3 with the East Zone contributing 31% of the feed. For years 3 to 5 inclusive, the average work index increases to 15.5 kW-hrs/tonne due to the gradual hardness increase at depth. Beyond year 7, test work indicates a further increase in hardness.

An average 16.1 kW-hrs/tonne ball mill work index was used to calculate power requirements for mill sizing at 30,000 tonnes per day.

16.2.7 Metallurgical Recoveries

The results of G&T's locked cycle testwork are presented in Table 16.7. The flotation locked cycle testing produced a copper grade-recovery profile relationship for both zones which is summarized in Table 16.8.

Table 16.7 Locked Cycle Test Results

Ore Zone	Sample &	Mass %	Assay - % or g/t		Recovery-%	
Main	**Product**		**Copper**	**Gold**	**Copper**	**Gold**
MZ-1	Feed	100.0	0.54	0.2	100	100
	Concentrate	1.9	26.1	4.34	89	40
MZ-2	Feed	100.0	0.53	0.42	100	100
	Concentrate	1.8	26	10.8	90	47
MZ-3	Feed	100.0	0.58	0.53	100	100
	Concentrate	2.0	26.2	12.98	88	48
MZ-4	Feed	100.0	0.25	0.16	100	100
	Concentrate	0.8	25.9	5.59	84	29
MZ-5	Feed	100.0	0.32	0.21	100	100
	Concentrate	1.4	21	6.33	89	41
MZ-Global	Feed	100.0	0.5	0.34	100	100
	Concentrate	1.7	26.6	9.27	90	46
MZ Average	Feed	100.0	0.45	0.31	100	100
	Concentrate	1.6	25.4	8.59	89	44
East						
EZ-1	Feed	100.0	0.84	0.59	100	100
	Concentrate	3.0	24.8	12.2	89	63
EZ-2	Feed	100.0	0.81	0.76	100	100
	Concentrate	1.9	37.0	26.2	88	66
EZ-3	Feed	100.0	0.58	0.51	100	100
	Concentrate	1.4	35.2	22.6	84	61
EZ-4	Feed	100.0	0.67	0.54	100	100
	Concentrate	2.2	26.6	14.1	88	58
EZ-5	Feed	100.0	1.27	1.23	100	100
	Concentrate	3.7	31.8	23.7	91	70
EZ-Global	Feed	100.0	0.74	0.59	100	100
	Concentrate	2.5	26.0	15.1	89	65
EZ Average	Feed	100.0	0.74	0.65	100	100
	Concentrate	2.2	29.2	18.31	88	63

Table 16.8 Copper Grade-Recovery Profiles

Final Concentrate Grade	Percent Copper Recovery	
% Cu	Main Zone	East Zone
25	90.8	90
26	90.3	89.9
27	89.8	89.7
28	89.1	89.4
29	88.2	89.2

The recoveries of copper to concentrate ranged from 85 to 90 percent while gold recoveries varied between 45 and 65 percent. Gold recovery in both zones was dependent on the head grades and pyrite content and higher in East zone.

16.2.8 Copper Concentrate

A pilot plant program was conducted at G&T Metallurgical Engineers with the objective of producing a final concentrate for use and testing by prospective buyers. Table 16.9 lists the full analysis, including minor elements, of a sample prepared in the ratio of 64% Main Zone and 36% East Zone. Hg and Sb are expected to incur penalties for being outside the limits set by some smelters.

Table 16.9 Copper Concentrate Analysis

Element	Symbol	Unit	Analysis	Range
Copper	Cu	%	27	26-30
Gold	Au	g/t	17.3	5-25
Silver	Ag	g/t	42	25-100
Aluminum	Al	%	0.95	0.6-1
Antimony	Sb	ppm	756	400-900
Arsenic	As	ppm	130	80-250
Barium	Ba	ppm	44	12-45
Bismuth	Bi	ppm	<7	<8
Cadmium	Cd	ppm	16	12-22
CaO	CaO	%	0.66	0.4-0.9
Chlorine	Cl	%	0.01	0.01
Chromium	Cr	ppm	3	3-55
Cobalt	Co	ppm	20	10-25
Fluorine	F	ppm	87	40-190
Iron	Fe	%	28.8	23-33

Lead	Pb	ppm	439	200-1300
Manganese	Mn	ppm	213	40-500
Mercury	Hg	ppm	27	8-50
MgO	MgO	%	0.36	0.2-0.5
Molybdenum	Mo	ppm	166	20-170
Nickel	Ni	ppm	22	10-45
Phosphorous	P	ppm	115	40-225
Potassium	K	ppm	372	130-500
Selenium	Se	ppm	93	70-160
Silica	SiO2	%	6.6	3.5-7
Sodium	Na	ppm	274	65-300
Sulphur	S	%	29.8	28-35
Titanium	Ti	ppm	348	300-450
Zinc	Zn	ppm	1400	600-2000
Sizing	K80	µm	25	24-28

* Sample Analysis refers to the composite concentrate sample generated from the G&T Metallurgical Engineers pilot plant exercise, based on an ore mixture of 64% Main Zone and 36% East Zone

** Average Range is based on results of concentrates from the various Main Zone and East Zone composites tested during the metallurgical program.

16.3 Deep East Zone Test Work (2010)

In late 2009, two composite samples of deep East zone mineralization were put together to complete some preliminary metallurgical test work on these materials. The focus of this initial test work was not to optimize the flowsheet for these composites but to see how the samples would perform in the flowsheet proposed in the 2005 Feasibility Study. The primary objective of the study was to assess the mineralogical and metallurgical response of a new mineralized zone recently located during drilling at depth. The assessment was to be conducted with respect to the previously developed process for treating Red Chris mineralization.

Approximately 859 kilograms of sample, in the form of quarter drill core, were received for use in this study. From this material inventory, the two composites were constructed for use in this program, using approximately 135 kilograms of the drill core inventory.

The two samples collected were labeled, High Au to Cu, and High Bornite, and each was selected from intervals in recently drilled holes that had a high gold to copper ratio for the first sample and a high percentage of its copper mineralization in the form of Bornite in the second.

The following points summarized some of the key findings to date:

- The copper and gold grades of the composites selected for construction were considerably higher than previously tested samples.

- The samples contained much less pyrite than previously studied East zone samples. The copper sulphide liberations were between 64% and 69% percent when measured a size fraction of 80% passing 150 microns. These values were much higher than observed for other samples and should result in efficient rougher flotation recovery of copper sulphides.

- One composite had similar hardness as previously measured samples at 14.6kWh/tonne. The Bornite East Composite was hard, averaging 16.7 kWh/tonne.

- Using the previously developed process flowsheet, metallurgical performance for copper and gold far exceeded the historical levels from previous programs. The flowsheet and test conditions remain un-optimized and simplifications to the process should be considered for this mineralization to reduce operating costs.

- Mineralogical analyses, specifically for gold, indicated most of the gold in the samples was associated with copper sulphides, or was liberated. These forms of gold were well recovered to the copper concentrate. There was evidence to suggest the small amount of gold lost to tailings occurred as tiny inclusions in non-sulphide gangue.

The chemical and mineral contents of the composites were determined using standard analytical techniques and QEMSCAN particle mineral analysis (PMA). The results of QEMSCAN analysis provided quantitative mineral content values for the array of sulphide minerals observed in the sample. A composite sample from the historic testing of shallower East zone mineralization is included on the Table 16.10 that highlights the differences in the deeper samples. The relevant data summary is presented in Table 16.11.

Table 16.10 Chemical and Mineral Contents

Element or Mineral	Symbol	Units	East Zone	East Zone	East Zone Global
			High Au-Cu	High Bornite	KM1428
Copper	Cu	%	2.9	1.4	0.7
Iron	Fe	%	5.9	4.6	7.1
Sulphur	S	%	3.7	0.85	-
Silver	Ag	g/t	6.0	6.4	-
Gold	Au	g/t	4.74	1.84	0.7
Weak Acid Soluble Copper	Cu Ox	%	0.022	0.024	-
Cyanide Soluble Copper	CuCN	g/t	0.073	0.94	-
Mineral					

Chalcopyrite	Cp	%	8.5	0.6	1.4
Bornite	Bn	%	<0.1	2.0	0.4
Chalcocite/Covellite	Ch/Cv	%	0.1	<0.1	<0.1
Pyrite	Py	%	2.6	0.2	5.9
Ankerite	Ank	%	7.5	16	-
Quartz	Qz	%	61	40	-
Micas	Mic	%	10	21	-
Other Gangue	Gn	%	10.4	20.4	92.4

Both samples contained relatively high values of copper and gold compared to previously studied East Zone samples, as shown in Table 16.3. The new composite samples also display relatively low levels of pyrite compared to the amount of copper sulphides.

The rougher tails from the locked cycle testing of the high Au-Cu East Zone were tested for Acid Base Accounting. The two tests showed the rougher tails to have a neutralization potential ratio (NPR) of 26.9 and 85.0.

The High Au-Cu East Zone composite exhibited almost perfect copper metallurgical performance. Copper recoveries of 97 to 98 percent were achieved with a rougher mass recovery of about 20 percent. Similarly, gold was also well recovered to the rougher concentrate. On average, between 91 and 93 percent of the gold in the feed was recovered into a rougher concentrate containing 20 percent of the feed mass. Both copper and gold rougher flotation performances were well above average for deposits of this type. Clearly this composite could be processed at a much coarser feed sizing.

The best cleaner flotation response for the high Au-Cu East composite was achieved in Test 11. In this test, additional collector (3418A) was added along with Potassium Amyl Xanthate. The superior performance of this test would suggest that the previous tests may have had insufficient collector dosages. Overall the results were spectacular, copper was 96 percent recovered into a concentrate grading about 33 percent copper. Similarly, gold was 88 percent recovered into the copper concentrate. The gold grade in the concentrate was about 50 g/tonne.

The Bornite East Zone composite also performed well. Copper in the feed was 93 to 95 percent recovered into the rougher concentrate at a rougher mass recovery of about 10 percent. Gold recoveries to the rougher concentrate were lower; averaging about 88 percent at 10 percent feed mass recovery.

The cleaner test results for the Bornite East Zone were more consistent. On average, about 90 percent of the copper in the feed was recovered into a concentrate grading about 45 percent copper. Gold performance was more variable, but on average, about 75 percent of the gold was recovered into the copper concentrate. The gold grade was about 60 g/tonne in the concentrate.

In locked cycle testing, copper recovery to the final concentrate, from the High Gold Copper Ratio East Zone sample was 98 percent, with a copper concentrate grade of 29 percent. Gold recovery to the final concentrate was 90 percent, grading 47 g/tonne gold. Copper from the High Bornite East Zone sample was 93 percent recovered into a final concentrate, grading 49 percent copper. Gold was 85 percent recovered into the final concentrate, grading at a high 69 g/tonne gold.

The observed performance was superior to results achieved on previously tested East Zone samples. The high grade of copper and gold, low pyrite content and favorable fragmentation properties of the composites make these composites respond well at the previously developed Red Chris process design.

16.4 Mineral Processing Description

The ore will be processed through an on-site concentrator that will produce a copper/gold flotation concentrate. The concentrate will be trucked to concentrate storage and ship-loading facilities at the Port of Stewart and shipped to smelters and refiners overseas. The nominal milling rate will be 30,000 tonnes per day. Processing will be based on a conventional copper/gold porphyry flowsheet, with a semi-autogenous grinding (SAG) mill, pebble crusher and ball mill grinding, cleaner flotation and concentrate dewatering. A simplified process flow-sheet is shown in Figure 16.1 below.

Process associated facilities include:

- Primary Crushing
- Coarse Ore Stockpile and Reclaim Tunnel
- Mill Complex and Structures, including Pebble Crushing and Concentrate load-out

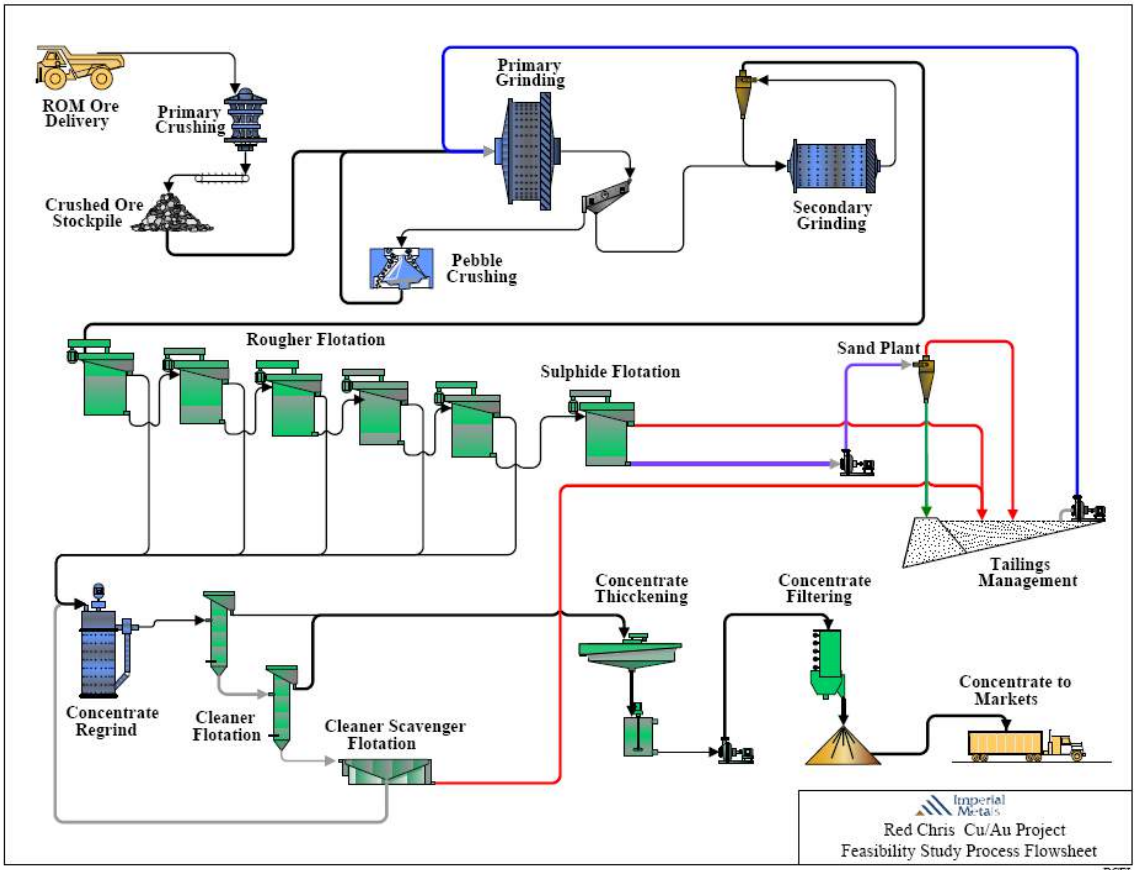
16.4.1 Process Design Criteria

Table 16.11: Summary Process Design Criteria

Description/Phase (year)	Units	Nominal
Production		
Operating Days/Year	(d)	365
Tonnes Milled/Calendar Day	(t/d)	30,000
Tonnes Milled/Year	(kt/a)	10,950
Ore Characteristics		
Ore Specific Gravity	SG	2.8
Ore Bulk Density	(t/m$^{3)}$	1.6
Ore Moisture Content	(%)	2.0
Work Indices (metric)		
Impact WI (avg)	(kWh/t)	8.0
Rod Mill (avg)	(kWh/t)	15.4
Ball Mill (avg)	(kWh/t)	15.3
Ball Mill (annual max)	(kWh/t)	17.2
% Cu (avg)	%	0.334
% Cu (annual max)	%	0.527
% Cu (annual min)	%	0.173
Gold (avg)	g/t	0.252
Recoveries		
Copper Recovery (avg)	%	86.5
Gold Recovery	%	46.9
Concentrate Grade	% Cu	27.0

16.5 Flowsheet Showing Process Streams

Figure 16.1 Simplified Process Flowsheet



Using mine haul trucks, run-of-pit ore will be hauled to a primary gyratory crusher, where it will be crushed and then fed by conveyor to a 90,000 tonne outdoor crushed ore stockpile. Ore will be drawn on demand from the crushed ore stockpile by means of feeders located in a reclaim tunnel beneath the stockpile and fed by conveyor into the mill grinding circuit. Dust collection/scrubbing facilities will be included the stockpile reclaim tunnel to intercept and collect dust from the reclaim tunnel feeders under the crushed ore stockpile.

Crushed ore will be conveyed on demand to the mill primary grinding circuit, which will consist of a single SAG Mill operating in closed circuit with a pebble crusher.

Primary grinding will be wet, that is with water addition to the ore. Oversize pebbles extracted from the SAG Mill will be removed by screen, crushed and then returned into the SAG Mill feed. Undersize from the SAG mill discharge screens will be transferred to a ball mill for secondary grinding. The ball mill will operate in closed circuit with a bank of cyclone sizing devices.

The ground slurry from the secondary grinding circuit (cyclone overflow) will be passed through a rougher/scavenger flotation circuit consisting of 5 x 200 m3 capacity flotation cells. The copper/gold concentrate from all of the rougher scavenger flotation cells will be combined and directed to the regrind circuit. The tailings from the rougher scavenger cells will be directed through an additional 200 m3 flotation cell designed to de-sulphurize the final mill tailings on as needed basis. The objective is to produce a de-pyritized final mill tailing to create Non-Acid Generating (NAG) tailings for use as dam construction material and for beaches along the upstream faces of each of the three tailings dams at final closure.

The effectiveness of this process was ascertained through metallurgical testwork and geochemical characterization of these materials. The pyrite concentrate from these cells can either be directed to combined with the rougher scavenger concentrate or separately directed to the tailings impoundment where it will be discharged sub-aqueously within the impoundment. In this way the PAG sulphide mineralized concentrate can be permanently stored in a fully saturated condition, while at the same time providing for NAG above water tailings beaches in front of the South Dam and the North Dam, which enhances their long-term stability and safety.

The copper/gold concentrate from the rougher scavenger circuit will be reground in a ball mill acting in series with a tower mill operating in closed circuit with a set of cyclones. The objective is to liberate the copper mineralization to allow concentrate grades to be improved through the cleaner flotation circuit.

The cyclone overflow from the regrind circuit will be pumped to two stages of cleaning using column flotation cells. The concentrates from both column flotation cells (Stage 1 and Stage 2 Cleaner Cells) will be combined and pumped to the concentrate thickener as the final copper/gold concentrate produced by the mill. The tailings from the Stage 1 column cleaner cell will be directed to the Stage 2 column cleaner cell. The tailings from the Stage 2 column cleaner cell will be directed to the cleaner scavenger flotation circuit consisting of 5 x 50 m^3 flotation cells. The concentrate from the cleaner scavenger circuit will be sent back to the concentrate regrind circuit for additional liberation. The tailings from the cleaner scavenger circuit will be directed to the final mill tailings pump box where it will be combined with the rougher scavenger tailings and sent to the tailings impoundment.

The final copper/gold flotation concentrate from the column cleaner cells (Stage 1 and 2) will be directed to the concentrate thickener where excess water will be removed and recycled. The thickened concentrate will be drawn from the thickener underflow and sent to one of two parallel pressure filters where the water will be pressed out of the concentrate to generate a dewatered concentrate filter cake. The dewatered final copper/gold concentrate will be stockpiled within concrete bins within the mill building (under the filters) from where it will be loaded by front end loader into highway trucks for delivery to the Port of Stewart.

The final de-pyritized mill tailings will be sent to a cyclone sands plant, near the TSF or at the dam abutments, where coarse tailings solids will be separated from the fines using two parallel cyclone banks to generate a de-pyritized NAG tailings sand to be used in tailings dam construction and a tailings fines product.

Water will be reclaimed from the tailings impoundment for use within the milling process as process water. Fresh make up water will be obtained from groundwater pump wells, downstream of the North Reclaim Dam and/or the South Reclaim Dam, for use in mixing reagents and for pump glands and fire protection water within the mill.

16.6 Major Process Equipment

- One Gyratory Crusher 54 in x 80 in (Open side setting = 6.5 in); 500 hp motor
- One SAG Mill 10.36m dia. X 4.65 m EGL (=34 ft dia x 15.25 ft EGL); 2x6000 hp motors
- One Ball Mill 7.32m dia x 12.65 m EGL (=24 ft dia x 41.5 ft EGL); 2x9383 hp motors
- One Pebble Cone Crusher HP800
- One 3000 hp Regrind Ball Mill plus one 1500 hp Tower Mill VTM-1500
- One Bank of Rougher Circuit Cells; Five Cells ; Each Cell: 200 m3
- One Sulphide Removal Cell; 200 m3
- Five Cleaner Circuit Cells; Each Cell: 50 m3

16.7 Process Reagents

Reagents used within the mill will include potassium amyl xanthate and Aerophine 3418A (di(isobutyl) dithiophosphinate), both conventional flotation collectors; MIBC (Methyl Isobutyl Carbinol), a conventional flotation frothing agent, a flocculant to be used to enhance concentrate settling and quicklime for pH modification on an as required basis in the primary and regrind circuits.

Potassium Amyl Xanthate (PAX) will be shipped in dry pellet form to the mine site in drums. The drums will be stored within the mill in a reagent storage area in close proximity to the reagent mixing facilities. The contents of the drums will be emptied directly into the xanthate mix tank. PAX will be mixed with freshwater taken from ground water wells. Mixed xanthate will be transferred in batch lots from the mix tank to the xanthate day tank and then pumped to a head tank from where it will be distributed by gravity pipeline to the points of use.

Collector 3418A will be shipped to site in liquid form in drums. The drums will be stored within the mill in a reagent storage area in close proximity to the reagent mixing facilities. The drums will be emptied directly into the 3418A mix tank. 3418A will be mixed with freshwater. Mixed 3418A will be transferred in batch lots from the mix tank to a day tank and then pumped by a series of metering pumps to the points of use.

MIBC frother will be shipped as a liquid to the site in drums. The MIBC will be transferred from the drums into a mix tank where it will be diluted with freshwater. MIBC frother will be transferred from the mix tank into a day tank on a batch basis and then transferred by metering pump to the points of use.

16.8 Hazardous Waste Products

Used oil, antifreeze, batteries, tires, solvents, etc. are considered hazardous waste products on the mine site. Typically all hazardous wastes outside of tailings and waste rock will be segregated at the point of generation, placed into appropriate storage containers and then shipped off site to an appropriate recycling or disposal facility. A lined storage facility will be constructed within or near the site fuel storage facilities to store the hazardous waste held in segregation pending periodic off-site shipment. Specifically hazardous wastes will be handled as follows:

- Used oil – from heavy equipment and stationary milling equipment will be transferred to a used oil storage tank to be located within the truck maintenance shop area. This oil will be filtered and then burned in a packaged waste oil burner unit to generate supplemental heat for heating of the truck maintenance shop in the winter months. Any excess used oil not consumed in this manner will be shipped off site using a licensed used oil disposal firm for recycling off-site. Every attempt will be made to dispose of used oil on site as a supplemental heat supply.
- Used antifreeze, solvents and grease will be collected and stored in appropriate drums for regular shipment off site to a licensed recycle or disposal facility.
- Used batteries – spent vehicle batteries will be collected, placed on pallets for regular shipment off site for disposal at a battery recycling facility.
- Used Tires – tires will be collected and those not used on site to provide vehicle protection barriers will be disposed through burial within an active section of the North rock storage area
- Hydrocarbon Contaminated Soil – A landfarm will be constructed utilizing bio-remediation to treat petroleum contaminated soil that is likely to accrue during the mine's operational life. The landfarm will be constructed near the proposed non-hazardous waste on-site landfill, on a compacted till or other suitable liner. Hydrocarbon contaminated soil will be transferred into the landfarm, spread out over the surface in thin lifts and treated with fertilizer to promote bio-remediation. The soils will be routinely turned over and sampled until it can be demonstrated that the hydrocarbon contamination has been reduced to acceptable standards. "Clean" soils will be stockpiled for use in progressive reclamation projects. Water collected within the landfarm will be run through an oil-water separator with the clean water discharged into the tailings impoundment.

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17 Mineral Resource Estimate

17.1 2012 Open Pit/Block Cave Mineral Resource Estimate

A new block model for the Red Chris deposit was completed in January of 2012 to include the new deep drilling done in 2010 and 2011 This model included all new drilling up to hole RC11-565 (62 new diamond drill holes, totaling over 69 thousand additional metres completed since the May 2010 resource update). The original resource estimate table published on Feb. 14, 2012 was constrained by a series of Copper Equivalent grade shells, within a wire frame digital solid constructed around the three mineralized deposit domains. The resource has now been amended and restated here as a combination of an Open Pit and Block Cave constrained Resource to demonstrate “reasonable prospects of economic extraction” as referred to in Instrument NI 43-101. Table 17.1 summarizes the total Mineral Resources, both open pit and underground, for the Red Chris Project. Although the authors believe there is a reasonable prospect of economic extraction there can be no assurance that these Mineral Resources will be eventually upgraded to Mineral Reserves. The results are shown in Tables 17.1, to 17.4.

Table 17.1 Red Chris 2012 Total Open Pit/Block Cave Mineral Resource Estimate

Red Chris 2012 Total Open Pit/Block Cave Resource Estimate								
Material		Ore	Mill Head	Insitu Grades				
Class		millions	Value	Copper Equiv.	Copper	Gold	Silver	
		Tonnes	$/tonne	(%)	(%)	(g/t)	(g/t)	
MEASURED		830.7	$25.13	0.57	0.36	0.36	1.17	
INDICATED		203.0	$18.55	0.47	0.30	0.29	1.01	
M&I		**1,034.7**	**$23.84**	**0.56**	**0.35**	**0.35**	**1.14**	
INFERRED		**787.1**	**$18.65**	**0.48**	**0.29**	**0.32**	**1.04**	

**Mill Head Value is a calculation of the value of material mined, in Canadian dollars per metric tonne, once it reaches the Crusher Pocket. This includes all downstream costs from the crusher forward, including: Milling / Concentrate handling and transportation / Treatment and refining / Royalties / Sustaining capital / Administration and head office overhead costs. Large capital costs associated with expansions, such as mining fleet additions, or replacements are not included. See table 17.24 for metal recovery formulas, costs and parameters used to calculate this value*

***Copper Equivalent % = [Copper Grade (%) + (.60415 * Gold Grade (g/t))]; based copper/ gold price ratio at Copper - $3.50 /lb, Gold $ 1450/oz*

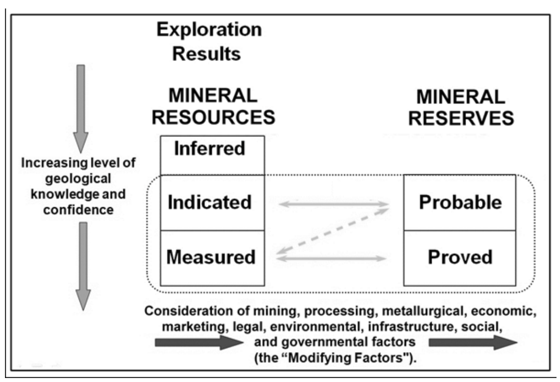
17.2 Mineral Resources Details

The Mineral Resources lies within the three adjacent mineralized domains (zones) and has been calculated to include both open pit and underground block cave mining methods. A longitudinal cross section of the mineralized deposit, Figure 17.1 below, illustrates Mineral Resource Zones and the proposed mining methods.

Table 17.1 lists the Total Open Pit Mineral and Block Cave Constrained Resource tonnages by Classification. Both the open pit resource and underground block resource statements contain Inferred Resource tonnages and the reader should note that under CIM guidelines Inferred Resources can be included in a resource statement, but has the lowest category of confidence. The CIM definition defines an Inferred Mineral Resource as:

"*that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.*"

Figure 17.1 CIM Resource and Reserve Confidence Chart (from CIM)



Most of the Deep Red Chris Ore Body is classified as the inferred, due to the limited drilling density below the 1150 elevation. The underground block cave resource is very conceptual at this time. In addition to First Nations consultation and various permitting requirements a great deal of geotechnical work and in-depth engineering studies regarding the nature of the cave-ability of the Red Chris deep ores need to be completed. These studies will likely be done in conjunction with the "fill-in" drilling required to upgrade the resource classification. For this reason there is no assurance that the proposed Block Cave will be realized.

Since the current Mineral Reserve (and permitted mine) have over 25 years of mill feed at the current milling rates there is no immediate need to incur the costs of fill-in drilling and future mining studies at this time.

17.2.1 Mill Head Value

Mill Head Value is a calculation of the value of material mined in Canadian dollars per metric tonne, at the Crusher Pocket. This includes all downstream costs from the crusher forward, including: Milling / Concentrate handling and transportation / Treatment and refining / Royalties / Sustaining capital / Administration and head office overhead costs. Large capital costs associated with expansions, such as mining fleet additions, or replacements are not included.

Except for metal prices, the methodology, metal recovery formulas and all cost items used to calculate the Mineral Resources Mill Head Values are identical to those used to define the Mill Head Value in the Mineral Reserves. Table 17.25 provides a detailed in-depth example on how the Mill Head Value is calculated for a sample mineralized block for the Mineral Reserve.

Mill Head Values are used as cutoff parameters for evaluation of both the open Pit and Underground mining methods discussed below (shown in tables 17.1 to 17.3). This method has been used successfully in Imperial Metal's other operating copper mines as a standard financially based ore valuation tool.

The key economic inputs into defining both open pit and underground mineral resources, that differ from those used in the reserve statement are metals pricing and the US$/Can$ exchange rate:

- Metals pricing in US$: Copper - $3.50 /lb, Gold $ 1450/oz., Silver $25/oz.
- US / Can $ Exchange Rate: $.95 US = $1.00 Can

Mining method specific operating costs used to define the mineral resources are discussed in subsequent sections.

17.2.2 Open Pit Resources Details

The open pit constrained Mineral Resources were defined utilizing MineSight's Computer Software Lerch-Grossman pit optimization routine. MineSight's computer programs and the Lerch-Grossman algorithm are acknowledged within the mining industry as creditable tools for this purpose. Key pit specific inputs into the Lerch-Grossman program were:

- Pit slope angle = 42 degrees - which is the average pit slope of the currently approved pit.
- Waste Mining costs of $ 1.872 per tonne for the 1470 elevation bench.
- An additional cost of $.052 per tonne was added to for each 15 meter bench below the 1470 elevation for increased haulage costs.
- Ore Mining Costs of $ 1.787 per tonne for the 1470 elevation bench.
- An additional cost of $.044 per tonne was added to for each 15 meter bench below the 1470 elevation for increased haulage costs.
- No capital costs were included for replacement or additional mine equipment fleet purchases.
- A portion of the East side of the pit was constrained by approximately 100 meters to preserve the current crusher installation.

All classes of resources, including inferred, were utilized in the Lerch-Grossman algorithm.
Since the Mineral Resource Lerch -Grossman constrained pit(s) are significantly larger than the currently permitted mine additional consultation with First Nations, environmental studies, waste rock disposal storage area designs, tailings storage, geotechnical reviews and other numerous works will have to be completed and approved prior to obtaining the required permitting of such a large mine expansion.

The Open Pit sourced Feed with a Mill Head Value greater than $1.50 per tonne would be processed immediately. Feed with Mill Head values between $1.50 and $0.00 are stock piled for processing at the end of the mines life. The use of the Mill Head Value classification in this fashion assures a minimum return of $1.50 per tonne on all processed mill feed. The average Mill Head Value of all Open Pit Measured & Indicated tonnes is $18.14/t, and Open Pit Inferred is $13.54/t. (see Table 17.1)

The 2012 Mineral Reserves in table 18.1 are completely contained within the Lerch Grossman Pit constrained Mineral Resources and are inclusive within the Resource Statement.

Table 17.2 Red Chris 2012 Open Pit Mining Constrained Resource Estimate

Red Chris 2012 Upper Resource from Open Pit								
Material	Material	*Cut-Off	Ore	*Mill Head	Insitu Grades			
Class	Type	Mill Head	Millions	Value	**Copper	Copper	Gold	Silver
		Value ($)	Tonnes	$/tonne	Equivalent (%)	(%)	(g/t)	(g/t)
Measured	Stockpile	$0.00	6.0	$0.95	0.15	0.11	0.07	0.48
	Mill Feed	$1.50	676.4	$19.10	0.48	0.32	0.27	1.04
		Sub Total	682.4	$18.94	0.48	0.31	0.27	1.04
Indicated	Stockpile	$0.00	0.8	$0.93	0.15	0.11	0.07	0.46
	Mill Feed	$1.50	164.7	$14.91	0.42	0.27	0.24	0.91
		Sub Total	165.5	$14.84	0.41	0.27	0.24	0.90
Inferred	Stockpile	$0.00	20.1	$0.65	0.16	0.08	0.14	1.06
	Mill Feed	$1.50	377.2	$14.23	0.41	0.25	0.26	0.92
		Sub Total	397.3	$13.54	0.40	0.24	0.26	0.93
MEASURED			682.4	$18.94	0.48	0.31	0.27	1.04
INDICATED			165.5	$14.84	0.41	0.27	0.24	0.90
M&I			**847.9**	**$18.14**	**0.47**	**0.31**	**0.27**	**1.01**
INFERRED			**397.3**	**$13.54**	**0.40**	**0.24**	**0.26**	**0.93**
	Waste Rock		**2,407.3**					
	Over Burden		**121.0**					
	Strip ratio		**2.0**					

**Mill Head Value is a calculation of the value of material mined, in Canadian dollars per metric tonne, once it reaches the Crusher Pocket. This includes all downstream costs from the crusher forward, including: Milling / Concentrate handling and transportation / Treatment and refining / Royalties / Sustaining capital / Administration and head office overhead costs. Large capital costs associated with expansions, such as mining fleet additions, or replacements are not included. See table 17.24 for metal recovery formulas, costs and parameters used to calculate this value*

***Copper Equivalent % = [Copper Grade (%) + (.60415 * Gold Grade (g/t))]; based copper/ gold price ratio at Copper - $3.50 /lb, Gold $ 1450/oz*

17.2.3 Underground Block Cave Resource Details

The vertical orientation of the Red Chris Deposit, coupled with its very large size makes the Deep Red Chris Mineralization attractive to mining by underground block caving methods. Three blocks economically favorable to underground mining by block caving were identified as shown Figure 17.1. The associated Mineral Resource Statement for these blocks can be found in Table 17.2. The three blocks are clipped to the bottom of the open pit discussed above.

The key mining parameters used to define those underground mineral resource blocks which have a reasonable prospect of economic extraction are:

- An all-in mine development capital cost of $ 7.94 per tonne
- An operating cost of $8.96 per tonne.

Therefore the targeted mineralization was required to have:

- A Mill Head Value greater than $16.90 per tonne for the chosen block cave volumes
- A Mill Head Value greater than $8.96 per tonne operating cut-off grade at the draw points

The average Mill Head Value of all Block Cave Measured & Indicated tonnes is $49.86/t, and Block Cave Inferred tonnes is $23.85/t. (see Table 17.2)

Due to the unselective nature of a block cave mine all material within the designed block, regardless of grade or class will be recovered as a mixture or blend. It is very unlikely that any of the lower grades or waste can be separated.

Table 17.3 Red Chris 2012 Block Cave Constrained Resource Estimate

Red Chris 2012 Lower Resource from Block Cave Including Planed Dilution								
Material Class	Material Type	Cut-Off Mill Head Value ($)	Ore Millions Tonnes	*Mill Head Value $/tonne	Insitu Grades **Copper Equivalent (%)	Copper (%)	Gold (g/t)	Silver (g/t)
Measured	Mineralized Dilution	$0.00	2.9	$6.83	0.28	0.17	0.17	0.82
	Draw Point Cut Off	$8.96	20.6	$13.60	0.40	0.25	0.25	1.10
	Targeted Ore	$16.90	124.8	$61.33	1.12	0.61	0.85	1.91
		Sub Total	148.4	$53.62	1.00	0.55	0.75	1.78
Indicated	Mineralized Dilution	$0.00	0.6	$7.50	0.29	0.19	0.17	0.84
	Draw Point Cut Off	$8.96	7.3	$13.94	0.41	0.25	0.26	1.17
	Targeted Ore	$16.90	29.6	$40.73	0.83	0.48	0.57	1.59
		Sub Total	37.5	$34.98	0.74	0.43	0.50	1.50
Inferred	Waste Dilution	-$6.40	64.6	-$4.81	0.04	0.02	0.03	0.22
	Mineralized Dilution	$0.00	18.7	$6.15	0.27	0.16	0.17	0.73
	Draw Point Cut Off	$8.96	63.2	$13.38	0.40	0.25	0.25	0.98
	Targeted Ore	$16.90	243.4	$35.52	0.76	0.45	0.52	1.47
		Sub Total	389.8	$23.85	0.56	0.33	0.38	1.15
MEASURED			148.4	$53.62	1.00	0.55	0.75	1.78
INDICATED			37.5	$34.98	0.74	0.43	0.50	1.50
M&I			**185.8**	**$49.86**	**0.95**	**0.53**	**0.70**	**1.72**
INFERRED			**389.8**	**$23.85**	**0.56**	**0.33**	**0.38**	**1.15**

**Mill Head Value is a calculation of the value of material mined, in Canadian dollars per metric tonne, once it reaches the Crusher Pocket. This includes all downstream costs from the crusher forward, including: Milling / Concentrate handling and transportation / Treatment and refining / Royalties / Sustaining capital / Administration and head office overhead costs. Large capital costs associated with expansions, such as mining fleet additions, or replacements are not included. See table 17.24 for metal recovery formulas, costs and parameters used to calculate this value*

***Copper Equivalent % = [Copper Grade (%) + (.60415 * Gold Grade (g/t))]; based copper/ gold price ratio at Copper - $3.50 /lb, Gold $ 1450/oz*

Figure 17.2 Red Chris 2012 Open Pit/Block Cave Resource Constrains: Plan Map

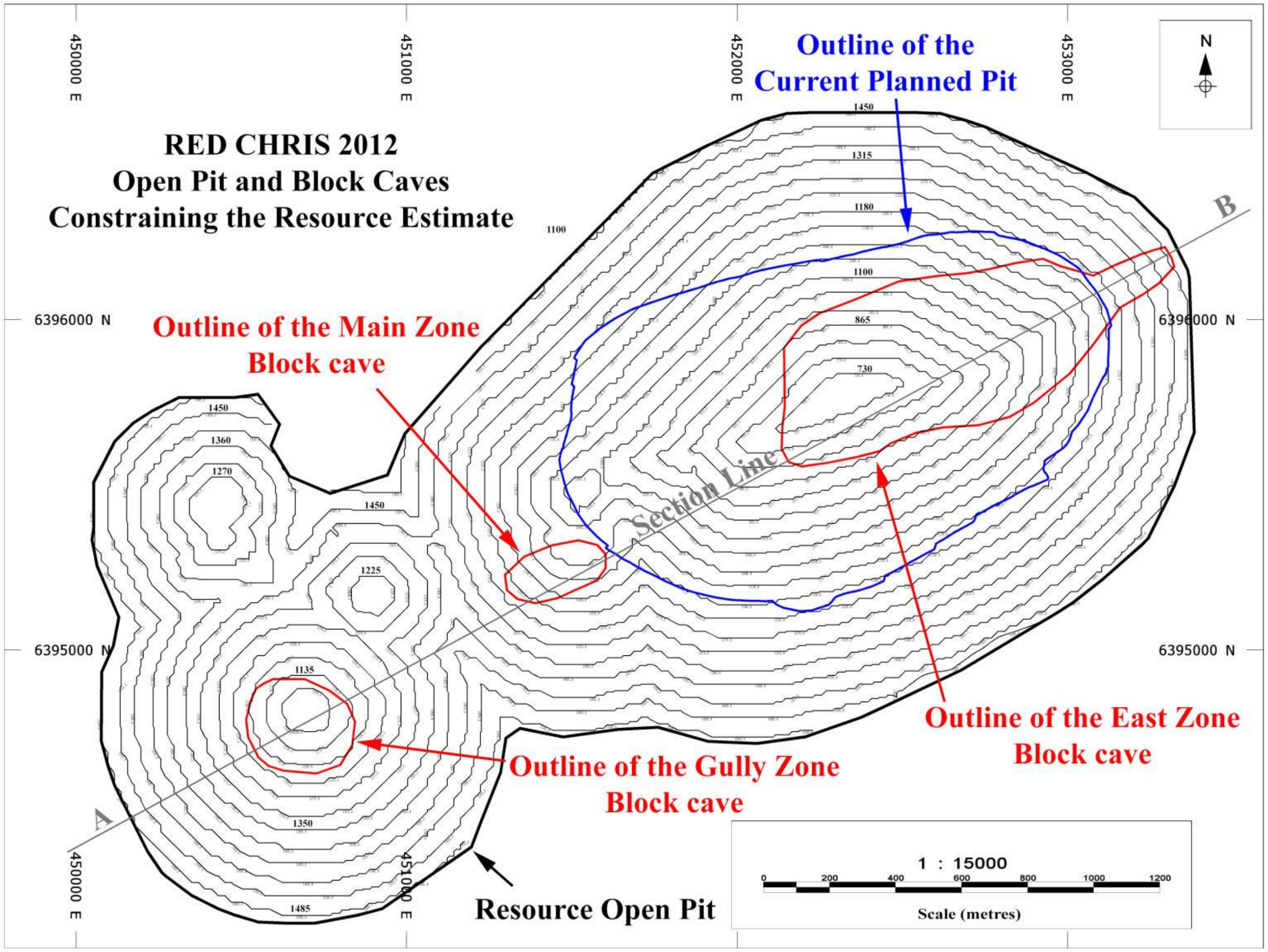
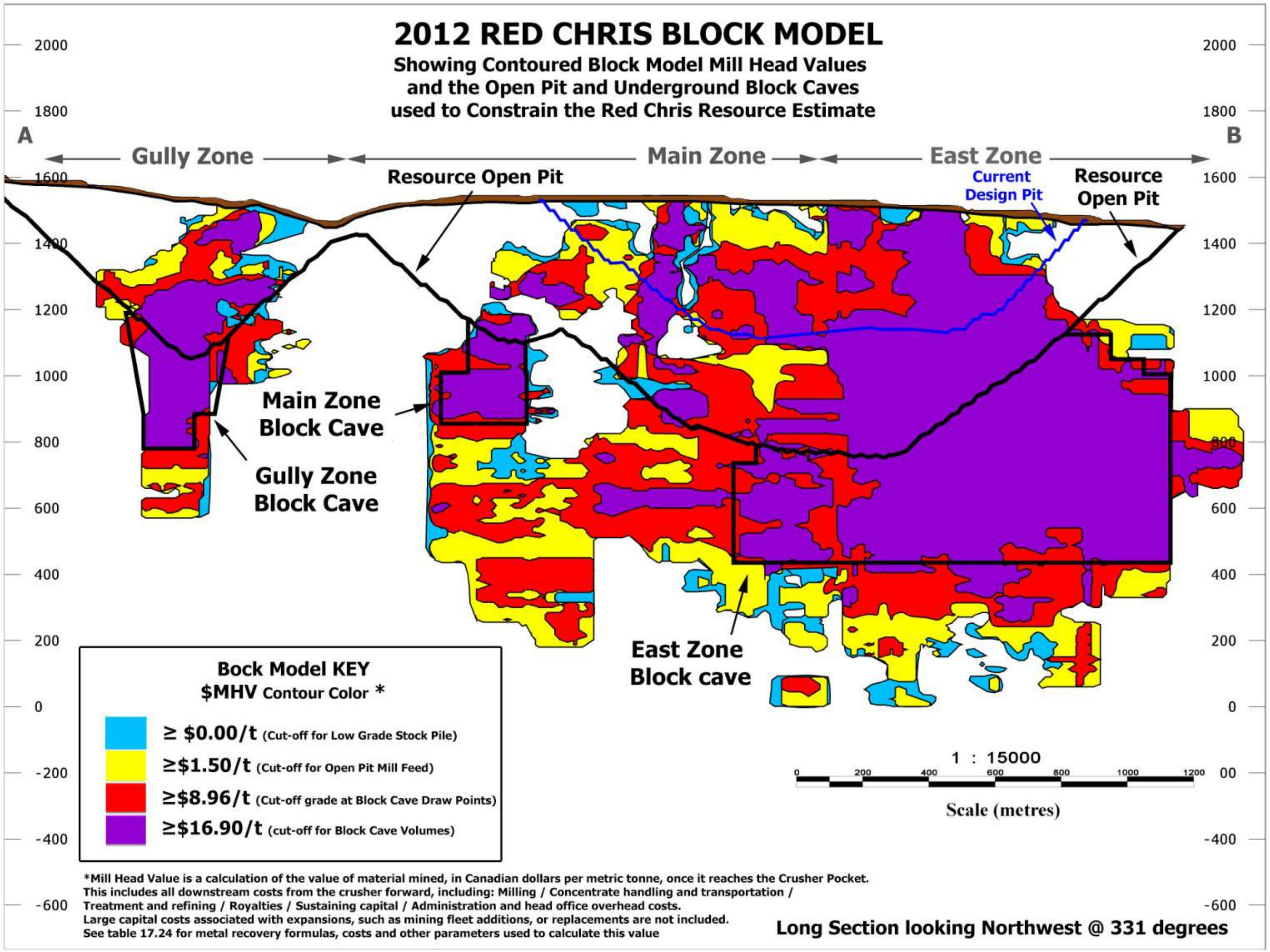


Figure 17.3 Red Chris 2012 Open Pit/Block Cave Resource Constrains: Long Section



17.3 Modeling Methodology and Resource Classification

Resource and Reserve classification were done in accordance to guidelines set out in the CIM Definition Standards for Mineral Resources and Mineral Reserves.

This Red Chris Deposit was divided into three geological Domains (Zones) for statistical modeling purposes as follows (see Figure 17.3).

Figure 17.4 Red Chris 2012 BLOCK Model Domains (Zones)

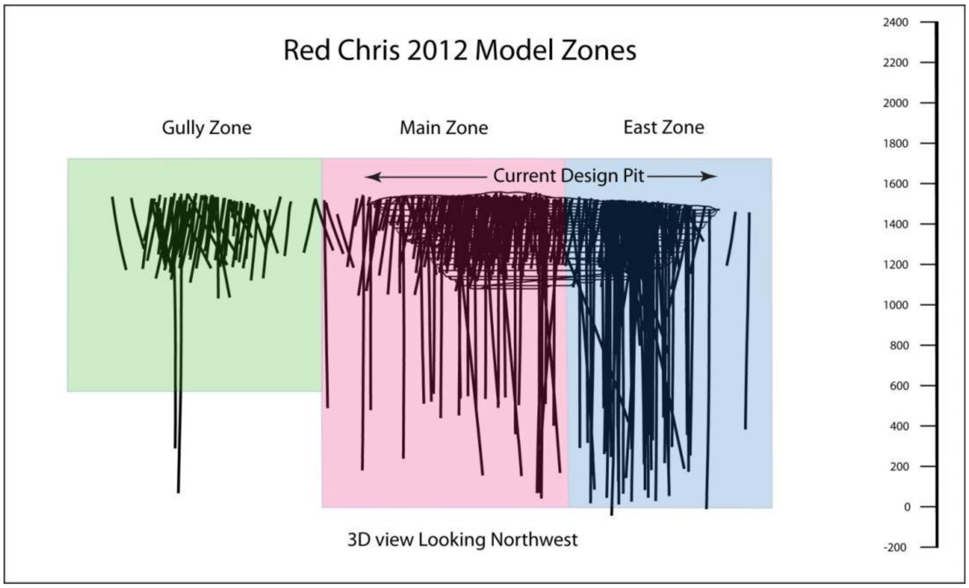


Table 17.4 Red Chris Model Domains (Zones)

Red Chris model Zones		
Zone #	Name	Description
1	Main	Largest Central Zone
2	East	East of Main Zone
3	Gully	West of Main Zone

A new block model for the Red Chris deposit was completed in 2012 utilizing Mintec's MineSight software for the modeling. The model is in UTM Coordinates (NAD83) with the lower SW corner (origin) at: 450,000N, 6,394,000E and 0m Elevation. The upper NE corner is at: 454,500N, 6,397,500E and 1725 Elevation. The block dimensions are 20m north by 20m east and 15m high and are considered appropriate for the size and nature of the deposit.

Table 17.5 Red Chris Model Dimensions

Red Chris 2012 Model Dimensions				
	UTM Coordinate		Block	Number
	Min	Max	Size (m)	of Blocks
East	450,000	454,500	20	225
North	6,394,000	6,397,500	20	175
Elevation	0	1,725	15	115

The drilling data set used comprises of 388 drill holes with an associated copper and gold assays sampled every 2.5m for the entire hole and composited to 15m for modeling. Sixty-Two (62) of the holes have been added to the data base since the previous resource update completed in May 2010. Holes outside the deposit zone or ones drilled for geotechnical or condemnation purposes were not included.

As mentioned above the deposit was divided into 3 geological Domains for statistical modeling purposes as seen in table 17.16. The model methodology listed here is the same for all zones.

- Wire Frame models were constructed for the three Model Domains.
- All diamond drill holes are loaded into MineSight drillhole files where each assay interval is assigned a zone code matching the zone they fall within.
- Model blocks are coded to the same Zone (Domain) codes as the drilling.
- Drill holes are composited to 15M bench composites except if their dip is less than 50 degree in which case they are composited to 15M down hole lengths. Composites are not allowed to span Zones.
- Statistics were carried out on the drillhole composites by zone to identify any outlier grades and to explore the grade correlation between copper, gold and silver grades. The zones show strong grade correlation. Outlier grades are identified using Log Probability plots by zone and are capped to the grade selected.

Table 17.18 Red Chris Grade Capping Logic by Domain (Zone)

Red Chris Grade Capping logic by Zone			
Zone	Max Grade	Capped	# Capped
Copper %			
Main	2.94	2.00	4
East	5.16	4.00	6
Gully	1.23	1.00	2
Gold g/t			
Main	3.80	3.00	3

East	14.29	10.00	5
Gully	2.32	1.50	1
Silver g/t			
Main	15.23	15.00	2
East	35.51	20.00	1
Gully	6.59	5.00	4

- A code is interpolated to define an interpolation zone around each drill hole 100M along strike direction and 60M across and vertical to prevent over extrapolation of grades in areas of wide drill hole spacing.
- Indicator kriging is used to prevent the smoothing of high grades across well-defined zones of waste. An indicator is assigned as 0 or 1 to the composites based on copper grade using a 0.10% cut-off (<0.10 = 0 and >=0.10 = 1). Variograms are run on the indicator by zone. See attached table.
- The indicators are kriged into a probability item in the block model resulting in each block having a value from 0.0 to 1.0 representing the probability of the block being high grade, see attached table of kriging parameters. Based on this probability an indicator is set in the block identifying the block as 0 = lowgrade/waste or 1 = highgrade.
- The composite indicators are then re-set to match the indicator in the block that they fall within. As a result waste grade composites within the ore zone are used in the ore zone interpolation (internal waste dilution) and ore composites in the waste zone are used in the waste zone interpolation (Note: these composites have their range of influence limited).
- Copper and gold variography was performed for each model zone ore (indicator = 1) zone, Sample for Main Zone Copper shown in Figure 17.3.
- Copper and gold grades both un-capped and capped are interpolated using zone and indicator matching. Two pass kriging is used in the ore zone. The first pass uses the full variogram range and the estimates are only used to provide an estimate of inferred grades. The second pass limits the search radius to 67% of the variogram range with the vertical range further limited to +-22m (bench above and bench below). This pass provides the measured and indicated grades. The waste zone is interpolated using inverse distance squared. Any composites in the waste zone have their range of influence limited to 40M.
- Silver grades are interpolated using inverse distance to the 3^{rd} power with a maximum search distance of 250M with the vertical search limited to 45M. Zone and indicator matching is also used. Since the early drill holes were not analyzed for silver not all blocks received an interpolated silver estimate. The missing blocks received a silver grade based on the strong correlation between copper and silver.
- The ore is classified using the following logic.

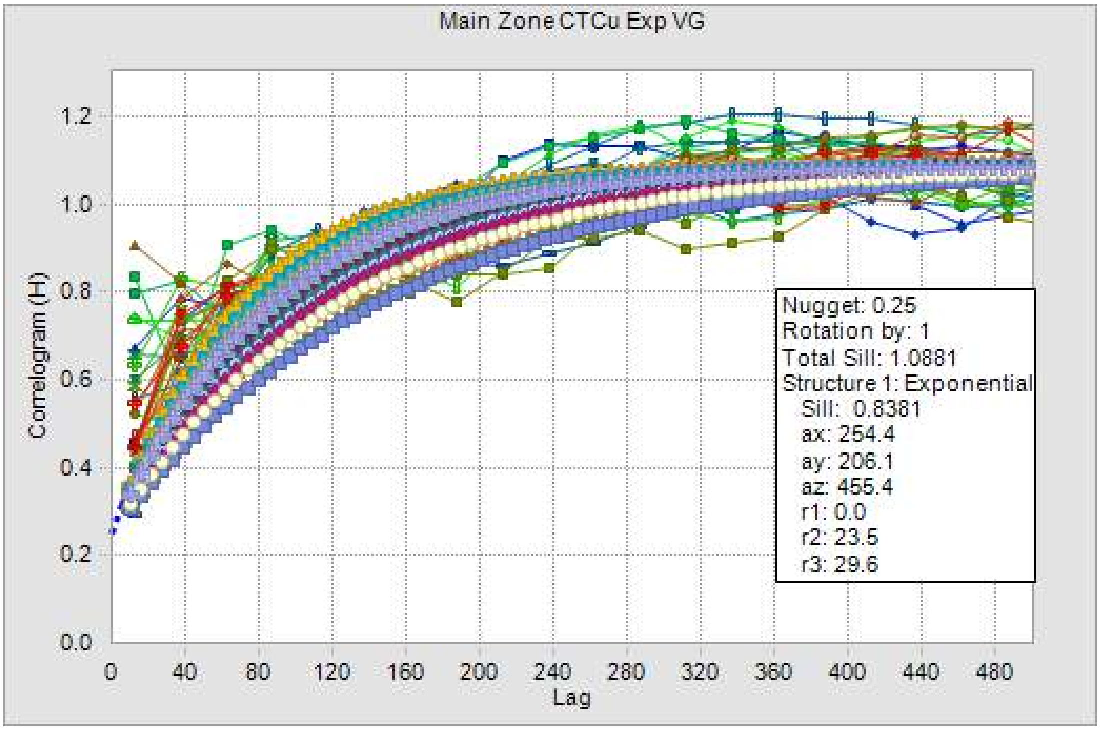
Table 17.19 Red Chris Model Classifications

Resource Classification				
	Minimum # Of Drill holes Used	Minimum # of Composites	Max Distance to Nearest Composite	Maximum Search Distance
Category				
Inferred	1	3	130	VG range
Indicated/ Probable	2	6	60	67% VG Range
Measured/ Proven	2	6	45	67% VG Range

Table 17.20 Red Chris Model Variography Results by Domain (Zone)

Red Chris Variography									
			Structure: Exponential in GSLIB logic						
Domain	Code	Nugget	Sill 1	Range Y	Range X	Range Z	Rot Z	Rot X	Rot Y
Ore Zone Indicator									
Main	1	0.250	0.733	152	297	263	40	14	42
East	2	0.150	0.843	99	192	380	0	7	23
Gully	3	0.195	0.870	215	160	500	91	-32	5
Copper									
Main	1	0.250	0.838	206	254	455	0	24	30
East	2	0.150	0.893	121	56	435	83	3	-2
Gully	3	0.280	0.793	350	244	79	78	58	45
Gold									
Main	1	0.210	0.816	124	344	167	0	39	-26
East	2	0.1500	0.863	128	71	316	70	-8	-3
Gully	3	0.270	0.749	50	147	350	112	-8	25

Figure 17.5 Main Zone 3D Copper Variogram



Note: Nugget set from downhole variogram

Table 17.21 Red Chris Model Interpolation Search Parameters by Domain (Zone)

Red Chris Interpolation Search Parameters								
Zone	Code	Logic	Range Y	Range X	Range Z	Rot Z	Rot X	Rot Y
Mineralized Zone Indicator, 0.1% Cu								
Main	1	80% VG range	122	238	210	40	14	42
East	2		79	154	304	0	7	23
Gully	3		172	128	400	91	-32	5
Ore Zone Grades								
Pass 1								
Copper								
Main	1	100% VG range	206	254	455	0	24	30
East	2		121	56	435	83	3	-2
Gully	3		350	244	79	78	58	45
Gold								
Main	1	100% VG range	124	344	167	0	39	-26
East	2		128	71	316	70	-8	-3
Gully	3		50	147	350	112	-8	25
Pass 2								
Copper								
Main	1	67% VG range & 22m Vertical Search	137	170	304	0	24	30
East	2		81	37	290	83	3	-2
Gully	3		233	163	53	78	58	45
Gold								
Main	1	67% VG range & 22m Vertical Search	83	229	111	0	39	-26
East	2		85	47	210	70	-8	-3
Gully	3		33	98	233	112	-8	25
Waste Zone Grades								
Copper and Gold								
All Zones	1,2,3	Inverse distance squared with a spherical 200M range						
		Range of influence if grade > .2 limited to 40M						

Table 17.22 Red Chris Model Interpolation Search Rules

In all cases the following rules were applied:	Pass 1	Pass 2
Maximum distance to nearest composite	130.0	60.0
Maximum composites used	9.0	9.0
Maximum Composites per drill hole	3.0	3.0
Minimum number of composites Pass 2	3.0	4.0
No interpolation across zone boundaries		
Indicator matching for all but indicator interpolation		
A code was placed in the model to limit the extrapolation of grades around drillholes to 100M along zone, 60M across zone, and 60M vertical		

17.4 Specific Gravity Modeling Calculations

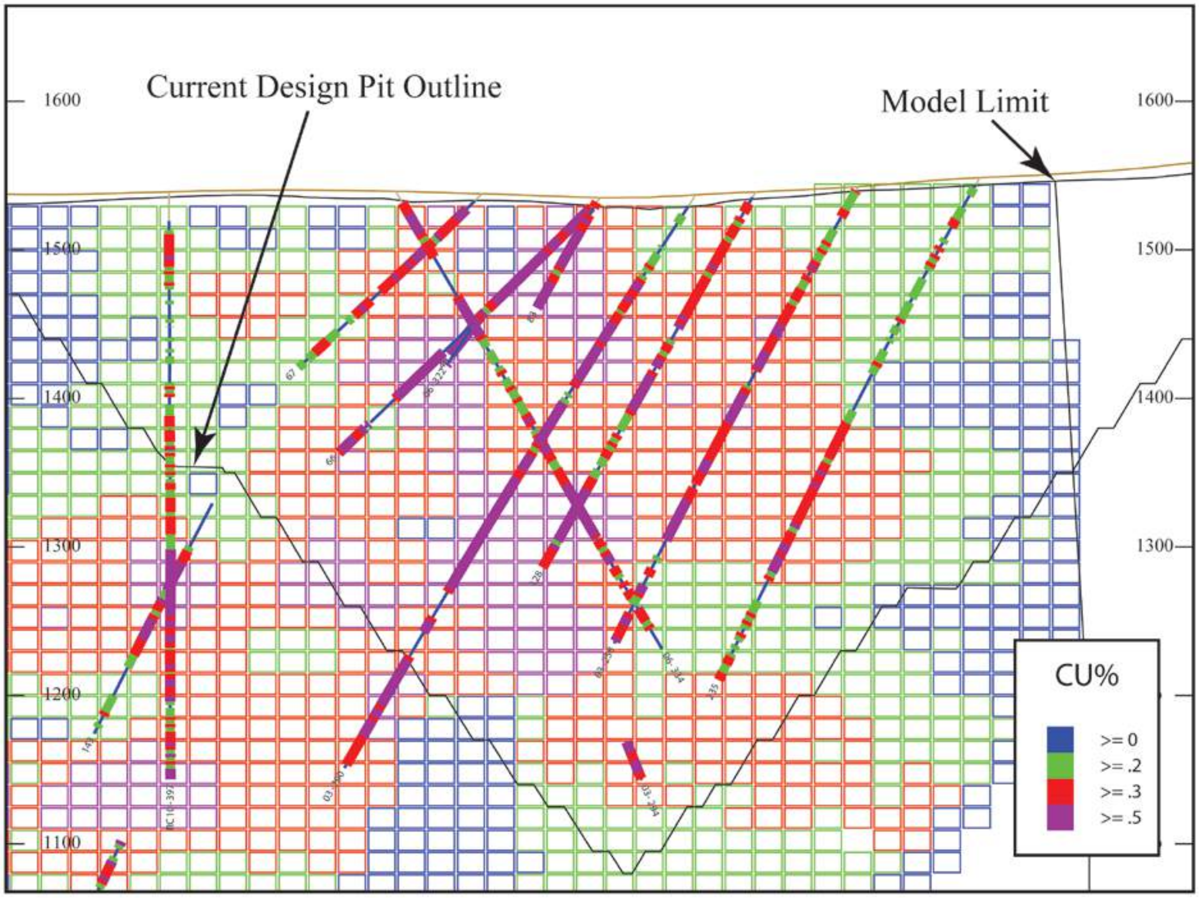
Every insitu block in the model has a specific gravity assigned to it to enable a tonnage to be calculated. The data set collected from core samples comprised of a total of 3,172 composited SG measurements loaded into the composite drillhole file. The arithmetic average of all the composite samples was 2.78.

The SG modeling process began with the assignment of the average, 2.78, to every insitu block in the model. This was followed interpreting grades into the blocks with a three pass spherical inverse distance to the 3rd power calculation. The resulting SG's were verified visually by comparing the block model to the drill data. No significant discrepancies were noted. The 2012 model mineralized blocks had an average SG of 2.79.

17.5 Model Validation

The modeling results were validated both visually and numerically. Using the 3D Viewer in the MineSight program, the grades and coding in the modeled blocks were manually compared to the drilling in cross section view (both North/South and East/West). This was also done in plan view for every model bench level. No significant discrepancies in grade or coding were noted between the block model and the drillhole assay composites. Figure 17.4 below shows (Section 451960E) the Main Zone Domain; with drillhole copper assays against interpolated block model copper grades. As shown below the model grade interpolation to the south and north is constrained by the zone wire frame solid.

Figure 17.6 Cu Drillhole Assay Grades vs Interpolated Block Model Cu Grades



Section 451960E Looking West

17.6 Comparing the 2012 Block Model with the 2010 Block Model

Historically block model calculations for the Red Chris deposit were completed providing only copper and gold grades. The 2010 block model was reported with copper only cut-offs, so a comparison was made with the 2012 Model using copper only cut-offs (see Tables 17.3-5). The new model shows an increase of 103% in tonnes of measured and indicated blocks using a 0.1 Cu cut-off, with both models constrained to the wire frame model domains from the May 2010 model.

Table 17.6 2012 Measured + Indicated Model Statistics (Cu Cut-offs)

2012 MEASURED + INDICATED MODEL STATISTICS (Copper Only Cut-Off)							
Cu Cut-Off	M Tonnes		Copper %	Gold g/t	Silver g/t	M lbs Copper	K oz Gold
>=0.1	1,258.5		0.321	0.319	1.102	8,892.1	12,903
2012 INFERRED MODEL STATISTICS (Copper Only Cut-Off)							
Cu Cut-Off	M Tonnes		Copper %	Gold g/t	Silver g/t	M lbs Copper	K oz Gold
>=0.1	1,266.3		0.267	0.279	0.991	7,442.7	11,359

Table 17.5: 2010 Measured + Indicated Model Statistics (Cu Cut-offs)

Cu Cut-Off	M Tonnes		Copper %	Gold g/t	M lbs Copper	K oz Gold
>=0.1	617.4		0.38	0.36	5,139.8	7,162

2010 Inferred Model Statistics (Cu Cut-offs)

Cu Cut-Off	M Tonnes		Copper %	Gold g/t	M lbs Copper	K oz Gold
>=0.1	619.1		0.30	0.32	4,120.7	6,429

For comparison purposes, this table above provides the 2010 block model statistics:

17.7 Sliver Resource

Historic drilling was not consistently assayed for silver, however all drilling completed by Imperial since 2007 has also been assayed for silver. Using these new silver assays, a calculation of the silver resource contained in the Red Chris deposit was completed. The Red Chris Open Pit/Block Cave Constrained Resource includes 37.9 million ounces silver in the Measured and Indicated category and 26.3 million ounces silver in the Inferred category.

17.8 Detailed 2012 Block Model Statistics by Zone

The calculated block model statistics within the three block model digital wire frame Domains (Zones) are summarized in tables 17.6 to 17.15. The tables are shown at increasing copper-equivalent grade cut-offs within the three deposit Domains. Figures 17.2 and 17.4 show the location of the Model Domains. The copper equivalent used in the tables below is based on the same copper to gold metal price ratio used in the resource estimation.

*Copper Equivalent % = [Copper Grade (%) + (.60415 * Gold Grade (g/t))]; based on a copper/ gold price ratio at Copper - $3.50 /lb, Gold $ 1450/oz*

Figure 17.7 Red Chris 2012 Block Model Grade Shell @ 0.20% Copper Equivalent

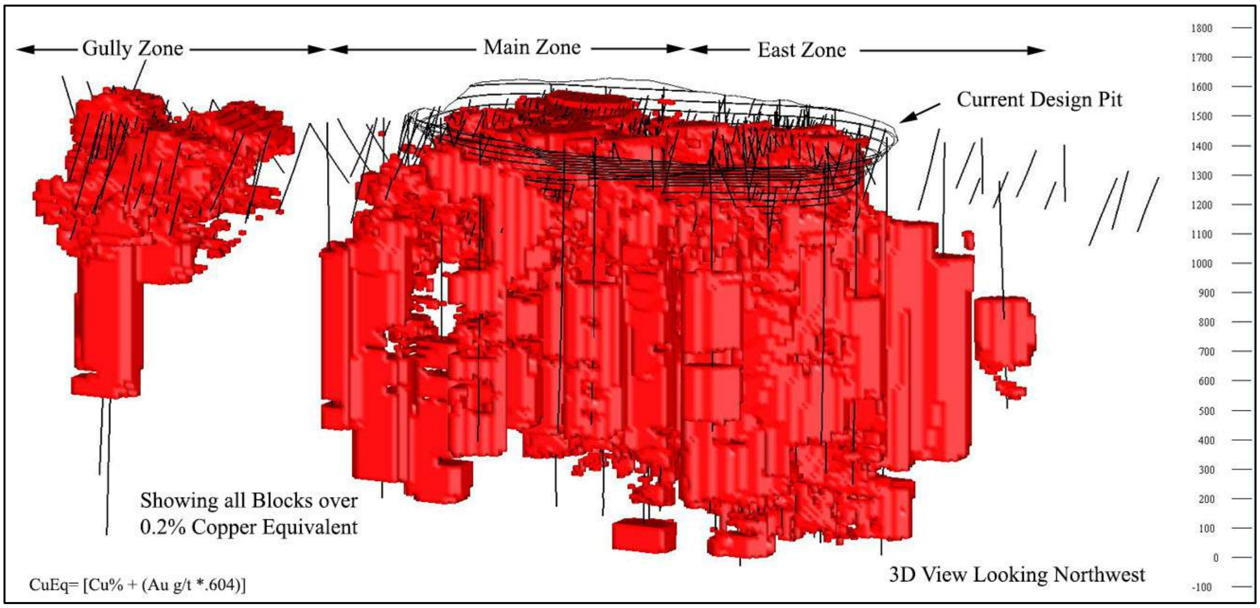


Table 17.7 2012 Red Chris Block Model Statistics: Total M&I Model Blocks

Total Deposit: M&I					
CuEq Cut-Off	M Tonnes	CuEq %	Copper %	Gold g/t	Silver g/t
>=0.1	1264.9	0.513	0.321	0.319	1.101
>=0.2	1217.2	0.527	0.328	0.329	1.116
>=0.3	934.8	0.609	0.375	0.387	1.227
>=0.4	643.3	0.727	0.442	0.472	1.378
>=0.5	436.8	0.859	0.514	0.571	1.527
>=0.6	297.4	1.006	0.591	0.687	1.678
>=0.7	205.9	1.165	0.671	0.819	1.851
>=0.8	146.7	1.335	0.753	0.963	2.036
>=0.9	112.2	1.485	0.823	1.096	2.227
>=1	86.5	1.644	0.895	1.240	2.432
>=1.1	67.7	1.810	0.968	1.392	2.651
>=1.2	55.1	1.962	1.034	1.535	2.858
>=1.3	46.0	2.103	1.097	1.666	3.018
>=1.4	38.4	2.253	1.163	1.804	3.193
>=1.5	32.5	2.398	1.225	1.943	3.356

Table 17.7 2012 Red Chris Bock Model Statistics: Total Inferred Model Blocks

Total Deposit Inferred					
CuEq Cut-Off	M Tonnes	CuEq %	Copper %	Gold g/t	Silver g/t
>=0.1	1622.2	0.369	0.222	0.243	0.895
>=0.2	1212.2	0.449	0.272	0.293	1.022
>=0.3	869.7	0.526	0.315	0.350	1.139
>=0.4	552.5	0.630	0.375	0.422	1.269
>=0.5	356.1	0.732	0.431	0.498	1.389
>=0.6	223.9	0.841	0.488	0.584	1.492
>=0.7	137.1	0.963	0.547	0.689	1.629
>=0.8	80.1	1.118	0.613	0.836	1.804
>=0.9	50.0	1.282	0.678	1.001	2.032
>=1	32.5	1.464	0.743	1.194	2.306
>=1.1	23.7	1.619	0.799	1.359	2.511
>=1.2	18.8	1.746	0.847	1.487	2.681
>=1.3	15.4	1.856	0.890	1.599	2.823
>=1.4	12.9	1.953	0.932	1.690	2.971
>=1.5	10.9	2.047	0.970	1.782	3.095

Table 17.8 2012 Red Chris Bock Model Statistics: Total Measured Model Blocks

Total Deposit Measured					
CuEq Cut-Off	M Tonnes	CuEq %	Copper %	Gold g/t	Silver g/t
>=0.1	989.1	0.535	0.334	0.333	1.131
>=0.2	949.9	0.550	0.342	0.344	1.149
>=0.3	741.0	0.633	0.390	0.402	1.259
>=0.4	524.5	0.751	0.457	0.487	1.409
>=0.5	366.0	0.883	0.528	0.586	1.563
>=0.6	254.5	1.030	0.605	0.704	1.722
>=0.7	180.1	1.188	0.684	0.835	1.893
>=0.8	131.4	1.352	0.763	0.976	2.067
>=0.9	101.9	1.498	0.830	1.106	2.253
>=1	79.5	1.653	0.900	1.246	2.448
>=1.1	62.6	1.817	0.973	1.397	2.667
>=1.2	50.9	1.971	1.040	1.541	2.879
>=1.3	42.4	2.117	1.104	1.676	3.049
>=1.4	35.2	2.272	1.173	1.819	3.235
>=1.5	29.8	2.420	1.235	1.962	3.400

Table 17.9 2012 Red Chris Bock Model Statistics: Total Indicated Model Blocks

Total Deposit: Indicated					
CuEq Cut-Off	M Tonnes	CuEq %	Copper %	Gold g/t	Silver g/t
>=0.1	275.8	0.435	0.273	0.269	0.993
>=0.2	267.3	0.443	0.277	0.275	0.999
>=0.3	193.8	0.514	0.317	0.327	1.102
>=0.4	118.8	0.620	0.376	0.403	1.241
>=0.5	70.8	0.737	0.441	0.490	1.345
>=0.6	42.9	0.862	0.508	0.587	1.417
>=0.7	25.7	1.008	0.580	0.708	1.554
>=0.8	15.4	1.185	0.668	0.856	1.772
>=0.9	10.3	1.354	0.749	1.001	1.968
>=1	7.0	1.544	0.835	1.174	2.244
>=1.1	5.2	1.718	0.910	1.338	2.454
>=1.2	4.2	1.851	0.968	1.463	2.598
>=1.3	3.6	1.948	1.013	1.547	2.660
>=1.4	3.1	2.043	1.057	1.632	2.722
>=1.5	2.7	2.150	1.105	1.730	2.859

Table 17.10 2012 Red Chris Bock Model Statistics: East Zone M&I Model Blocks

East Zone: M&I					
CuEq Cut-Off	M Tonnes	CuEq %	Copper %	Gold g/t	Silver g/t
>=0.1	353.1	0.764	0.450	0.520	1.388
>=0.2	345.1	0.777	0.457	0.530	1.408
>=0.3	303.6	0.849	0.497	0.584	1.511
>=0.4	252.0	0.951	0.550	0.663	1.665
>=0.5	205.8	1.064	0.608	0.755	1.825
>=0.6	166.4	1.186	0.668	0.857	1.984
>=0.7	133.3	1.319	0.734	0.969	2.136
>=0.8	107.0	1.460	0.801	1.091	2.285
>=0.9	89.8	1.577	0.855	1.196	2.421
>=1	73.4	1.717	0.917	1.323	2.597
>=1.1	59.7	1.871	0.985	1.466	2.806
>=1.2	49.9	2.013	1.047	1.598	2.993
>=1.3	42.6	2.144	1.106	1.718	3.134
>=1.4	36.2	2.285	1.170	1.846	3.292
>=1.5	31.1	2.423	1.229	1.976	3.441

Table 17.11 2012 Red Chris Bock Model Statistics: East Zone Inferred Model Blocks

East Zone: Inferred					
CuEq Cut-Off	M Tonnes	CuEq %	Copper %	Gold g/t	Silver g/t
>=0.1	717.4	0.449	0.275	0.288	0.889
>=0.2	576.4	0.524	0.320	0.338	1.012
>=0.3	444.3	0.606	0.365	0.399	1.129
>=0.4	337.9	0.687	0.410	0.459	1.250
>=0.5	241.3	0.782	0.458	0.537	1.382
>=0.6	168.7	0.883	0.507	0.624	1.491
>=0.7	113.2	0.999	0.561	0.725	1.636
>=0.8	71.4	1.147	0.623	0.866	1.835
>=0.9	47.0	1.302	0.684	1.024	2.052
>=1	31.8	1.473	0.745	1.204	2.313
>=1.1	23.6	1.622	0.799	1.361	2.507
>=1.2	18.7	1.747	0.848	1.488	2.680
>=1.3	15.4	1.856	0.890	1.599	2.823
>=1.4	12.9	1.953	0.932	1.690	2.971
>=1.5	10.9	2.047	0.970	1.782	3.095

Table 17.12 Red Chris Bock Model Statistics: Main Zone M&I Model Blocks

Main Zone: M&I					
CuEq Cut-Off	M Tonnes	CuEq %	Copper %	Gold g/t	Silver g/t
>=0.1	801.1	0.413	0.271	0.235	0.888
>=0.2	762.9	0.425	0.279	0.243	0.897
>=0.3	543.5	0.494	0.321	0.287	0.982
>=0.4	333.6	0.587	0.378	0.345	1.077
>=0.5	198.1	0.682	0.437	0.406	1.131
>=0.6	113.9	0.784	0.497	0.475	1.135
>=0.7	64.3	0.892	0.560	0.550	1.168
>=0.8	36.4	1.004	0.627	0.623	1.270
>=0.9	21.1	1.120	0.700	0.696	1.392
>=1	12.5	1.240	0.774	0.772	1.467
>=1.1	7.8	1.359	0.848	0.845	1.479
>=1.2	5.0	1.478	0.918	0.927	1.543
>=1.3	3.3	1.597	0.985	1.013	1.563
>=1.4	2.1	1.727	1.060	1.105	1.559
>=1.5	1.4	1.873	1.137	1.219	1.524

Table 17.13 Red Chris Bock Model Statistics: Main Zone Inferred Model Blocks

Main Zone: Inferred					
CuEq Cut-Off	M Tonnes	CuEq %	Copper %	Gold g/t	Silver g/t
>=0.1	662.0	0.305	0.184	0.200	0.842
>=0.2	485.7	0.369	0.225	0.239	0.955
>=0.3	314.8	0.431	0.259	0.284	1.063
>=0.4	144.5	0.534	0.321	0.353	1.177
>=0.5	74.5	0.619	0.368	0.415	1.210
>=0.6	32.3	0.715	0.426	0.478	1.235
>=0.7	14.3	0.804	0.477	0.541	1.348
>=0.8	6.0	0.888	0.523	0.604	1.497
>=0.9	2.1	0.978	0.578	0.662	1.737
>=1	0.6	1.070	0.629	0.730	2.051
>=1.1	0.1	1.179	0.681	0.825	3.366
>=1.2	0.0	1.229	0.726	0.833	3.320
>=1.3	0.0	0.000	0.000	0.000	0.000
>=1.4	0.0	0.000	0.000	0.000	0.000
>=1.5	0.0	0.000	0.000	0.000	0.000

Table 17.14 Red Chris Bock Model Statistics: Gully Zone M&I Model Blocks

Gully Zone: M&I					
CuEq Cut-Off	M Tonnes	CuEq %	Copper %	Gold g/t	Silver g/t
>=0.1	110.7	0.439	0.264	0.289	1.728
>=0.2	109.2	0.442	0.266	0.292	1.720
>=0.3	87.8	0.487	0.292	0.322	1.757
>=0.4	57.8	0.557	0.334	0.369	1.868
>=0.5	32.8	0.641	0.394	0.408	2.050
>=0.6	17.2	0.727	0.456	0.448	2.300
>=0.7	8.3	0.816	0.519	0.491	2.559
>=0.8	3.3	0.919	0.572	0.575	2.425
>=0.9	1.4	1.034	0.610	0.703	2.325
>=1	0.6	1.157	0.645	0.848	2.221
>=1.1	0.3	1.280	0.681	0.991	2.095
>=1.2	0.2	1.334	0.684	1.077	1.879
>=1.3	0.1	1.400	0.702	1.156	1.783
>=1.4	0.1	1.473	0.726	1.237	1.852
>=1.5	0.0	1.540	0.770	1.275	1.880

Table 17.15 Red Chris Bock Model Statistics: Gully Zone Inferred Model Blocks

Gully Zone: Inferred					
CuEq Cut-Off	M Tonnes	CuEq %	Copper %	Gold g/t	Silver g/t
>=0.1	242.8	0.309	0.169	0.231	1.053
>=0.2	150.1	0.420	0.241	0.297	1.278
>=0.3	110.5	0.478	0.274	0.338	1.397
>=0.4	70.0	0.554	0.322	0.385	1.550
>=0.5	40.2	0.635	0.383	0.417	1.757
>=0.6	22.8	0.703	0.437	0.439	1.863
>=0.7	9.5	0.783	0.489	0.486	1.980
>=0.8	2.8	0.877	0.548	0.544	1.661
>=0.9	0.9	0.951	0.601	0.580	1.662
>=1	0.1	1.039	0.633	0.671	1.601
>=1.1	0.0	0.000	0.000	0.000	0.000
>=1.2	0.0	0.000	0.000	0.000	0.000
>=1.3	0.0	0.000	0.000	0.000	0.000
>=1.4	0.0	0.000	0.000	0.000	0.000
>=1.5	0.0	0.000	0.000	0.000	0.000

17.9 Mineral Reserve Estimate and Open Pit Optimization

17.9.1 Review of 2005 Feasibility Study

The 2005 Feasibility Study ore reserves for the ultimate pit design were reported at a $3.75/t net smelter return internal cut-off value and are summarized in Table 17.23. This cut-off value was calculated as a pit rim cut-off equal to the value of processing plus general and administration onsite operating costs.

Table 17.23 Ore Reserve Summary from 2004 Technical Report

	M Tonnes	Cu %	Au g/t	R-Cu %	R-Au g/t	R-CuEq %	NSR $/t
Proven	93.5	0.423	0.327	0.374	0.185	0.482	11.554
Probable	182.5	0.300	0.226	0.261	0.100	0.320	7.600
Total	276.0	0.349	0.266	0.299	0.129	0.374	8.939

17.9.2 2010 Reserve Estimate

Note: The Reserve Estimate below and the following mine plan were based on the May 2010 model and is the same as published in March of 2011. The in pit reserve was re-run in January of 2012 with the new 2012 model as a check. The results showed an in pit increase in tonnage of 8.3% compared to the values below with the same grades.

Table 17.24 2010 Open Pit Ore Reserve Summary @ MHV>$1.50/t

	M Tonnes	Cu %	Au g/t	R-Cu %	R-Au g/t	R-CuEq %	MHV $/t
Proven	262.3	0.38	0.29	0.33	0.15	0.47	11.62
Probable	25.5	0.29	0.21	0.25	0.09	0.35	6.72
Total	287.7	0.37	0.28	0.32	0.14	0.46	11.19

The 2010 Reserve Estimate uses the same ultimate pit shell developed in 2004 with minor changes to eliminate irregular bench features and more economic ramp designs. The block model used to calculate reserves is based on the 2010 Mineral Resource Estimate.

The ore reserve block model was updated in 2010 to include economic cost estimates for each block. These cost estimates are given in Table 17.25 and 17.26 in this case for a mineable test block chosen from the ore reserve model. The balance of revenues and deductions 'at the crusher pocket' determined the millhead value (MHV) ($/dmt) of the test block. The positive MHV of $14.69/dmt for the test block would indicate ore. The MHV cut-off is used during the scheduling process to differentiate between ore to be processed, ore to be stockpiled, low grade to be stockpiled should economic conditions improve and storage. Reserve classification of proven and probable was based on the parameters shown in table 17.5

The proven and probable ore reserves at a $1.50/t MHV are summarized in following table. Low-grade material is below a MHV of $1.50/t and therefore is not included in this table.

Table 17.25 Millhead Value Block Model Economic Parameters

Millhead Value Calculation – Sample Calculation on a test block					
Test Block Millhead Value Calculation - Main Zone Block			Parameters and Test Block Calculation		
Cu Head Grade		(%)			0.480
Au Head Grade		(g/t)			0.263
Recoverable Cu Head Grade		(%)			0.427
Recoverable Au Head Grade		(g/t)			0.111
Metallurgical Recovery					
Cu Recovery				Equation used in MineSight Model	
	Main Zone; Capped at	(%)	93.45	5.0221 x ln(Cu) + 92.746	89.06
	East Zone; Capped at	(%)	91.38	9.7 x Cu + 80.224	84.88
Au Recovery					
	Both Zones; Capped at	(%)	90.20	53.93 x Au + 28.18	42.36
Metal Pricing					
	Cu Price	($US/lb)	2.20		
	Au Price	($US/oz)	900		
	Exchange Rate	($US/$Cdn)	0.90		
Concentrate					
	Cu Concentrate Grade	(%)			25.00
	Au Concentrate Grade	(g/t)		Calculated Based on Cu Recovery	
	Moisture Content	(%)			8.00
	Contained Cu	(lb/dmt)			551.16
	Contained Au	(g/dmt)			6.50
	Payable Cu	(lb/dmt)			529.11
	Payable Au	(g/dmt)			6.17
	Concentrate – Recovery Based	(dmt/t ore)			0.017
	Gross Value of Concentrate Before Deductions				
	Gross Value Concentrate	($Cdn/dmt)			1,491.87

Table 17.26 Ore Reserve Block Model Economic Parameters – Cont.

Mill Head Value Calculations continued:					
Concentrate Handling Costs					
	Minesite Loadout	($Cdn/wmt)			1.00
	Inland Freight	($Cdn/wmt)			39.50
	Port Charges	($Cdn/wmt)			13.36
	Insurance	($Cdn/wmt)			0.65
	Ocean Freight	($Cdn/wmt)			72.98
	Representation	($Cdn/wmt)			1.50
	Assays	($Cdn/wmt)			1.00
	Total Concentrate Handling	($Cdn/wmt)			129.99
	Total Concentrate Handling	**($Cdn/dmt)**			**141.29**
Treatment and Refining Costs					
	Treatment Charges	($US/dmt)			60.00
	Cu Payment after 1 Unit Deduction	(%)			96.0%
	Au Payment	(%)			95.0%
	Cu Refining Cost	($US/payable lb)			0.060
	Au Refining Cost	($US/payable oz)			5.000
	Total Treatment and Refining	**($Cdn/dmt)**			**107.35**
Royalty Costs					
	Total Royalty (after .8% buyout)	($Cdn/dmt)	1.00%		12.48
Site Costs					
	Milling	($Cdn/dmt milled)	4.00		
	Camp & Travel	($Cdn/dmt milled)	0.40		
	General / Admin. / Head Office	($Cdn/dmt milled)	1.00		
	Sustaining Capital	($Cdn/dmt milled)	1.00		
	Total Site Deduction to Conc Value	($Cdn/dmt)			374.71
Net Value of Concentrate After Mine Gate Deductions					
	Net Smelter Return Before Royalty	($Cdn/dmt)			1243.23
	Net Smelter Return After Royalty	($Cdn/dmt)			1230.75
Millhead Value - Net Value of Ore After Site Costs and Concentrate Handling, Treatment & Refining					
	Ore Value at the Crusher Pocket	($Cdn/dmt milled)			14.69

Notes: Main Zone Test Block Coordinates; N6395630, E451850, Elev1312.5 (20m x 20m x 15m)

18 Mining Method

18.1 Introduction

The proposed mining operation is a conventional shovel and truck open pit porphyry copper/gold mine feeding a 30,000 t/d processing plant. The planned mine life is 28.3 years with a minable Reserve of 301.6 Million tonnes @ 0.359% Copper and 0.274 g/t gold. The pit has been phased into minable pushbacks and scheduled to maximize the production of high-grade ore, especially during the first five years.

This strategy necessitates mining at a higher cut-off grade, stockpiling lower grade material for later in the mine life. The average mine life stripping ratio will be 1.25:1.

The pre-production mining period lasts for four months during which approximately 1.8 Mt of waste and incidental ore will be mined. The mine will use electric rotary blasthole drills, drilling 311 mm diameter holes, 28 m³ electric hydraulic shovels loading 230 t capacity haul trucks from 15m benches. The operation will be supported by standard ancillary equipment including an 18 m³ front-end loader, track and rubber-tired-dozers, and graders. Dry blastholes will be loaded with bulk ammonium nitrate prill and fuel oil (ANFO); the wet blastholes will be loaded with a combination of emulsion and ANFO from bulk mix trucks. The utilization of large electric rope shovels is an option that will be considered at the time of equipment purchase. The general arrangement of the mine area is shown in Figure 18.1.

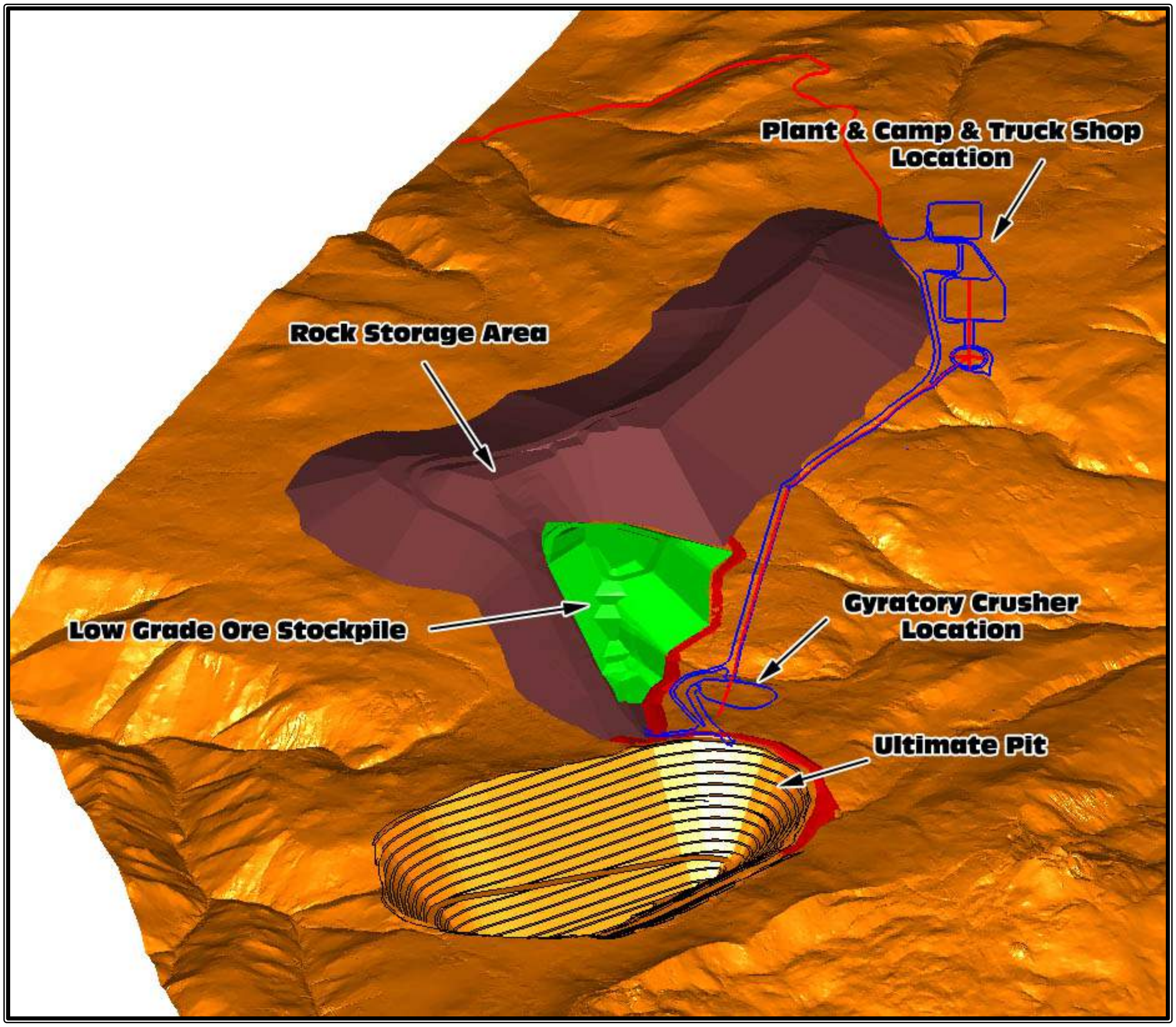
Ore will be hauled to a primary crusher located northeast of the pit rim. Low grade ore will be hauled to a stockpile located to the north of the primary crusher from where it will be reclaimed for processing in Years 21 to 28.

Low-grade material will be stockpiled on a base of non-acid generating (NAG) material. Potential acid generating (PAG) waste will be hauled to the north rock storage area (north of the open pit limits) and placed upon a base of NAG waste material. Provision has been made in the design for the NAG layer to be up to 5 m thick in areas where water can be expected to accumulate or flow and to progressively cover dump to limit infiltration of water. This dump will be reclaimed at mine closure as discussed in Section 22 of this report.

Mining will begin during the preproduction period (Year -1) in the East pit (East pit Phase I) and Main pit (Main pit Phase I). The pit will expand in phased pushbacks until the ultimate pit limits are reached. Six phases will be mined encompassing the East and Main pits. Each pit phase will expose a wide section of both ore bodies facilitating control of mill head grade by blending ore types. Access has been maintained between the East and Main pits permitting easy equipment movement between zones. The deposit has been modelled using a 15 m bench height in both the Main zone and East zone to match the pit designs. Pit walls have been designed to incorporate access ramps, berms, face and inter ramp consistent with recommendations made by geotechnical consultants.

The mine will operate around the clock, 7 days per week, using a 14 days on, 14 days off rotation schedule, with 12-hour shifts.

Figure 18.1 General Arrangement of the Mine Area



18.2 Development Concept

The plant, truck shop and camp will be located northeast of the north rock storage. The low grade ore stockpile will be located between the north rock storage and the Ultimate Pit. The rock storage and stockpiles will be placed on the plateau area north of the open pit, within the drainage catchment area of the tailings impoundment.

18.3 Mine Production Schedule

The production schedule over LOM has been prepared with the reserves tabled by bench and by phase. A MHV cut-off analysis determined the optimum MHV cut-off value for each phase to maximize the NPV of the project. Six phases or pushbacks were scheduled.

The mining of the open pit phases will be overlapped to smooth the scheduled stripping and resultant equipment and manpower requirements. During the first 10 years of operation, material greater than $0.01/t MHV and less than $1.50/t MHV will be stockpiled for future processing between the Years 11-15 and near the end of mine life. The ore stockpiled in the early years of mining, between the MHV $1.50/t and the phase MHV cut-off will be processed in the later years of the mine plan. The East zone constituent of the total ore processed will be less than 50% above the 1350m bench. The annual mining rate will not exceed 40 Mt/yr. At the end of the mine life; 25 million tonnes of 0.165% copper and 0.118 g/t gold remain in the low grade stockpile for processing should the economic conditions improve beyond the levels assumed for this study.

The proven and probable reserves scheduled for processing are given in Table 18.1. The production schedule for the Mine, Mill and Concentrate are summarized in Appendix A. The location of the ultimate pit, rock storage area and low grade stockpile was shown in the previous Figure 18.1. The annual material movement schedule is graphically presented in Figure 18.2 and the annual mill feed grades are shown in Figure 18.3.

Table 18.1 2012 Mineable Reserves Scheduled for Processing from the Planned Open Pit

	M Tonnes	Cu %	Au g/t
Proven and Probable ($1.50MHV or greater)	287.7	0.37	0.28
Proven and Probable (Greater than $0.01/t MHV & less than $1.50/t MHV)	13.8	0.18	0.14
Total Processed	301.6	0.36	0.27

The Proven and Probable Reserves in Table 18.1 are inclusive of the Open Pit constrained Measured and Indicated Mineral Resources stated in Table 17.1

Figure 18.2 Material Movement

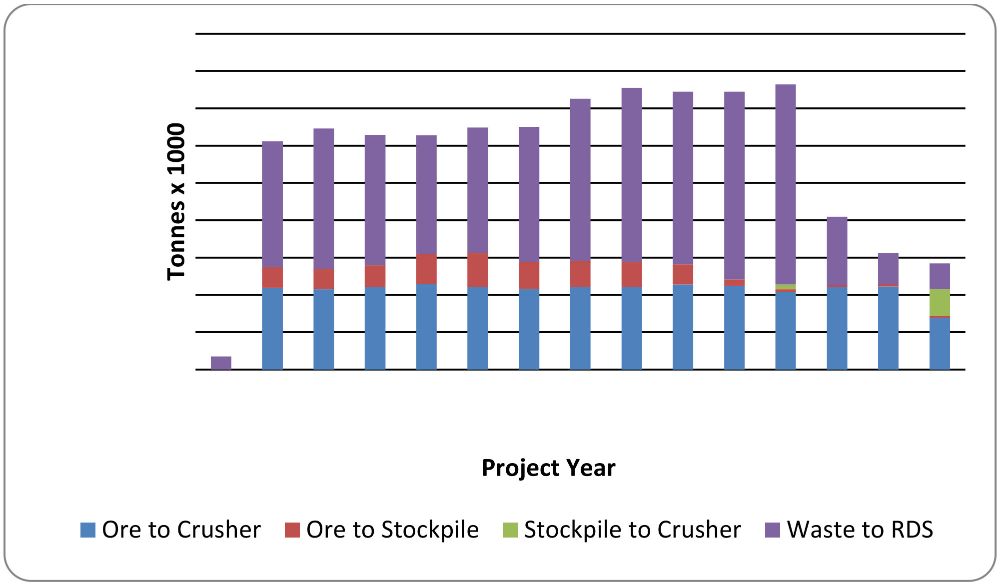
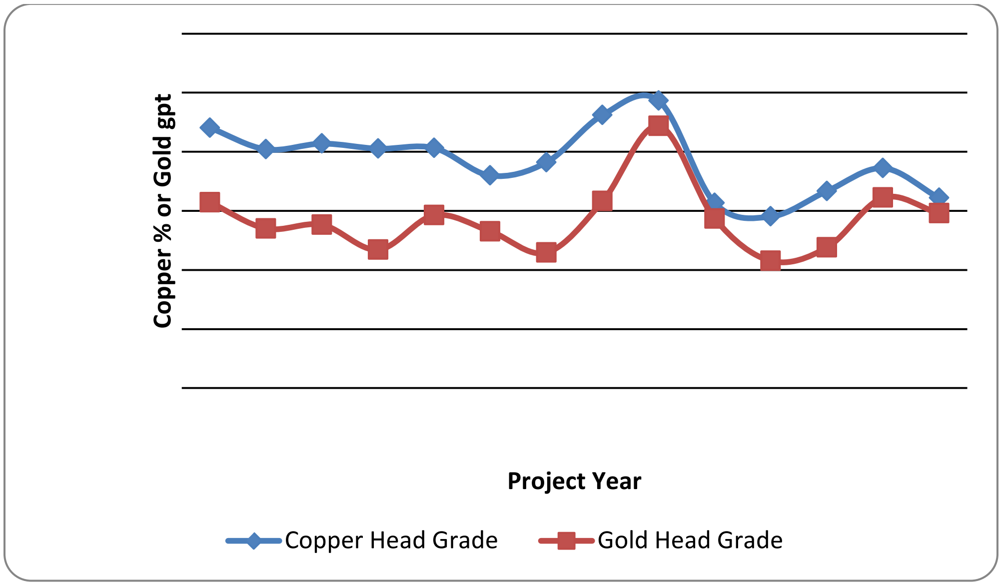


Figure 18.3 Head Grades



18.4 Mining Equipment

The mining method will be conventional shovel / truck operation utilizing electric and diesel powered equipment. The tentative equipment list is as shown in Table 18.2. Make and model are provided to give an indication of size only.

Table 18.2 Tentative Equipment List

Major Fleet	Make	Model
Production Rotary Drill (12 ¼" dia.)	Atlas Copco	PV351
Wall Control Rotary Drill (6 ½" dia.)	Atlas Copco	PV271
Hydraulic / Electric Shovel (28 m3 / 58 mt)	P&H	2800XPB
Hydraulic / Diesel Shovel (22 m3 / 46 mt)	Komatsu	PC4000
High Lift Loader (18 m3 / 34 mt)	Caterpillar	994H
Haultrucks (231 mt)	Caterpillar	793F
Haultrucks (136 mt)	Caterpillar	785D
Dozers Tracked (391 kw / 525 hp)	Caterpillar	D10T
Dozers Wheeled (362 kw / 485 hp)	Komatsu	WD600-3
Grader	Caterpillar	16M
Support Fleet		
Hydraulic Excavator (40 mt)	Caterpillar	345DL
Yard Loader w/ Cable Reeler	Caterpillar	980
Tire Manipulator	Caterpillar	980
Concentrate Shed Loader	Caterpillar	950
Blasthole Stemming Loader (3 m3)	Caterpillar	IT38
Blaster's Powder Truck (3 mt)	Ford	F350
Water Wagon (86 mt)	Ground Force	777
Sand and Gravel Truck (40 mt)	Caterpillar	740 Ejector
Low Bed Truck (86 mt)	Caterpillar	777
Fuel & Lube Truck (40 mt)	Ground Force	740
Fuel & Lube Island	Ground Force	Skid
Mechanics Service Trucks	Ford	3t
Bus Crummy	Ford	20 p
Crane Wire Rope (150 mt)	Bucyrus	150t
Crane Hydraulic (40 mt)	Grove	40t
Cable Repair Shed	CWS	CR450
Cable Reels	CWS	CR450
Trailing Cable (7,200v, 3/0, 2.7" dia.)	UEE	3/0
Electrical Substation (13.8 / 7.2 / 0.6 kv, switchgear)	UEE	Mobile Sub
Light Plants	Caterpillar	3 light
Submersible Pumps	Flygt	88 hp
Submersible Pumps	Flygt	150 hp
Pickups	Ford	F250

18.5 Infrastructure

The Red Chris property is located on the eastern portion of the Todagin Upland Plateau, which forms a subdivision of the Klastine Plateau along the northern margin of the Skeena Mountains. Elevations on the property are typically 1500 m ± 30 m with relatively flat topography broken by several deep creek gullies. The property is centred on latitude 57° 42' north, longitude 129° 47' west within NTS map sheet 104H/12W, Liard Mining Division.

Bedrock exposure is confined to the higher-relief drainages and along mountainous ridges. A thin layer of glacial till covers the majority of the property.

18.6 Plant Site Location

18.6.1 Background

In the 2005 Feasibility Study the process facilities and ancillary facilities were located on a cut/fill benched platform located immediately to the east of the open pit. SRK completed a due diligence report in March 2006, on behalf of potential financiers, and identified stability concerns for the location of the plant site and ancillary facilities.

Further geotechnical investigations were completed in 2006, 2009 and 2010. The plant site and other infrastructure have now been relocated further northeast of the open pit, as shown in Figure 18.1.

18.6.2 Access

At present vehicle access to the property is via access road from the Ealue Lake Road, constructed in 2008 to assist continued exploration and geotechnical investigations. Permanent access will be from the EA approved access road from Highway 37. When operating, mine personnel will be transported from Dease Lake and pick up points enroute to the mine site by bus and/or van. Charter flights originating in Vancouver and Prince George will pick up employees in Smithers and/or Terrace for flights to Dease Lake.

18.6.3 Roads

Either the existing access road will be upgraded to serve as permanent access road or it will be upgraded and extended to form a new 23 km all-weather access road, connecting BC Highway 37 with the mine site. The proposed access road location was selected considering factors such as public safety, constructability, potential environmental affects, and access control. The proposed route will be from Highway 37 along the south side of Coyote Creek and rising up to the plateau around the north and east sides. It will be a restricted access radio-controlled road with 24 hour, 7-day security provided at the Highway 37 junction. Only authorized personnel will be permitted into the mine area.

Construction of approximately 10 km of on-site service roads is also planned.

18.6.4 Process Facilities

Process facilities include:

- primary crushing
- coarse ore stockpile and reclaim tunnel
- mill complex and structures, including pebble crushing and concentrate load-out.

18.6.5 Ancillary Facilities

Ancillary facilities include:

- permanent accommodations complex
- maintenance shops and warehouse
- assay laboratory
- administration, security, and first aid offices.

18.7 Power

18.7.1 Power Supply

Development of the Red Chris mine will be contingent upon the availability of electric power from BC Hydro at Highway 37 near Tatogga at standard industrial rates, or other form feasible for the viability of the project, or an alternative viable power source including all necessary approvals as may be required under the BC *Environmental Assessment Act*. The Provincial and Federal governments have committed to provide funding for a 287 kV power transmission line from Terrace to Bob Quinn, the NTL, which now, after Provincial and Federal environmental assessment, has been approved. Site clearing for construction of the NTL began in January of 2012. RCDC will be responsible for extending powerline service sufficient to meet its needs from Bob Quinn to Tatogga and from there to the Red Chris mine. The power transmission line extension along Highway 37 requires approval through an amendment of the Red Chris EA Certificate and will also require a permit from the BC Ministry of Transportation and Infrastructure (MOTI) to allow development within the Highway 37 Right-of-Way (ROW). RCDC has applied for an amendment of the Red Chris EA Certificate and this application is now under review.

18.7.2 Utility Tie Switching Station or Sub-station

Power supply from BC Hydro at Bob Quinn will be at the 287 kV level. Depending on the voltage of the power transmission extension line from Bob Quinn to Red Chris Mine site either a switching station or a sub-station with transformation will be built at Bob Quinn.

18.7.3 Site Power Sub-station and Distribution

A step down utility tie sub-station on the property will be installed. Plant power will be distributed on the property at 25 kV, with three local exceptions where 4.16 kV will be used.

18.7.4 Emergency Power

A diesel-powered emergency generator will be installed near the process building to provide an alternate power supply to the automatic transfer switch for critical heating, lighting, fire protection, and process loads.

18.8 Services and Utilities

18.8.1 Fresh Water System

The fresh water system has been designed to supply up to 120 m3/h of make-up water. It will be supplied from a series of ten wells located downstream of the north TMF seepage dam and a series of five wells located downstream of the south seepage dam.

18.8.2 Fire Water

The fresh fire water tank at the plant has been designed to store a two-hour firewater demand at 350 m3/h.

18.8.3 Potable Water

A potable water well is to be installed in a suitable location in close proximity to the accommodation camp and mill site.

18.8.4 Process Water

Process water will be used primarily in the grinding, flotation, and regrind circuits. Reclaim water from the tailings impoundment will be used to provide all process water requirements.

18.8.5 Sewage Treatment

A sewage treatment system will be constructed for the permanent camp. After start-up of operations sewage may be diverted to the TSF.

18.8.6 Fuel Storage

Three steel enviro-tanks, each with a capacity of 75,000 L, will be provided together with one tank with a capacity of 22,700 L for small truck diesel and gasoline. All tanks will be contained within a HDPE lined and bermed compound.

18.9 Communications

A broadband Satellite Communications System and Office MIS Network with software will be installed.

19 Tailings Management

Key updates to the Tailings Management section from the 2005 Feasibility Study are as follows:

- Site selected for the North Dam has been revised.
- Detailed geotechnical, geophysical, and hydrogeologic investigations were undertaken in 2009 and 2010 for the facility area, including an update on site characterization.
- Revised description of the TSF and associated infrastructure.
- Revised layout of the tailings facilities at various stages of the project's development to illustrate the evolution of these facilities through the mine life, from pre-production through closure.
- Revised closure and reclamation plan for the tailings impoundment.
- AMEC E&E is completing the detailed design and construction drawings for the Tailings Management facility.

Most of this section is excerpted from the Joint Application for the Mines' Act Permit and Effluent Discharge Permit Application submitted to the North West Mine Development Review Committee in July 2010 with input from AMEC E&E.

19.1 Introduction

The site of the proposed tailings impoundment is in a Y-shaped valley approximately 3.5 km northeast of the Red Chris ore deposit (see Figure 18.1, general site plan). Construction of three dams will be required at the south, north, and northeast arms of the valley. The tailings impoundment will provide storage for all process tailings generated by the project. The tailings will be conveyed from the plant site through pipelines to the tailings impoundment, where they will be discharged from the North and/or South Dam. Tailings discharge is not required from the Northeast Dam, which is not required to be constructed until the later years of the mine life. The tailings distribution system is designed to manage the changing requirements of the tailings impoundment development over the life of the mine. This includes provision for development of required tailings beaches and for construction of the cycloned tailings sand downstream shells of the North and South Dams.

The TSF is planned for development in a Y-shaped valley in part tributary to Kluea Lake that has been divided into three areas for the purpose of the discussion below: the North Valley, the South Valley and the Northeast Valley. Earth fill embankments are planned for each of these valleys to contain the impoundment. The North and South Valleys are essentially the two ends of a main approximately north-south trending valley, the bottom of which is located approximately 400 m below the proposed rock storage area. To the east, a north-northeast trending ridge rises approximately 150 m above the North-South Valley bottom separating this valley from the Northeast Valley. To the southeast, moderate slopes extend approximately 300 m above the South Valley.

19.1.1 North Valley

The North Dam area is proposed to be located in the north portion of the North-South Valley, north of a local drainage divide which separates surface waters flowing toward Kluea Lake and toward the Klappan River. Moderate slopes increasing upwards to locally steep near the crest and gentle slopes extend upwards from the valley bottom to the west and east, respectively.

The terrain in the valley was interpreted to consist of a discontinuous glacial till blanket along the lower sides of the valley overlain by younger alluvial sands and gravels and organic deposits. The presence of glaciofluvial terrace remnants to the south near Kluea Lake suggests that the valley was a meltwater channel during deglaciation and there are glaciofluvial sand or sand and gravel deposits at depth below the valley bottom, underlying the till. This general stratigraphic model has been confirmed by the various site investigations undertaken at the site.

The predominant deposit on the upper valley slopes is a till mantle (variable thickness from a thin veneer to likely several metres in thickness), which thins towards the upper parts of the slopes, where bedrock is either exposed or thinly mantled with colluvium. Boreholes drilled in 1995, 2003 and 2010 in the general area of the North Dam area encountered organic deposits, alluvium, glacial till (a discontinuous unit at relatively shallow depth, and another unit, also apparently discontinuous, at greater depth) and glaciofluvial channel deposits. The depth to bedrock is up to about 90 m in the valley bottom, with most of the overburden soils thickness comprising glacifluvial sands and gravels.

Slope activity identified on airphotos in the North Valley area included potentially active gullies on the valley side slopes to the west of the proposed North Dam site. These gullies, although moderately well developed, did not exhibit signs of active erosion such as fresh unvegetated areas or linear scars along the gully sidewalls. Thus the potential for debris flows and sediment generation may be less along these gullies compared to elsewhere in the area, subject to further examination in the field. A significant, deeply incised gully exists immediately to the west of the site selected for the North Dam. However, bedrock is exposed within much of the steep section of this gully near the dam abutment.

19.1.2 South Valley

The proposed South Dam area is located south of the confluence of the Northeast and North Valleys. Immediately south of the drainage divide between the North and South valleys, is a swamp, referred to as Black Lake, likely infilled with fine-grained organic sediments of at least several meters in thickness. Black Lake drains into a swampy area to the south, which then outlets into the well-defined creek channel that bisects the proposed South Dam alignment. On the west side of the main north-south arm of the TSF the valley slopes are moderate and become steeper at higher elevations. On the southeast side of the main arm, the valley slopes are also moderate.

The terrain along the valley bottom is swampy, possibly because of groundwater seepage or due to low slopes along the valley bottom or a combination of the two. There is also beaver activity in the South Valley area. Beneath the organic deposits, geologically recent fluvial deposits of sands and gravels overlie a relatively thin layer and apparently discontinuous glacial till layer underlain in turn by thick glaciofluvial deposits. There are also surficial deposits of clay and silt in the swampy area upstream of the proposed South Dam alignment. The discontinuous shallow till unit appears to thicken on the east side of the creek. Extensive eroded glaciofluvial terrace remnants occur on the sides of the valley, particularly the west side, where the shallow till unit appears absent except at higher abutment elevations.

Similar to the geomorphology to the north in this valley, the upper valley slopes are covered by a till mantle, which thins towards the upper portions of the slopes. A geomorphological interpretation prepared by BGC (ABML, 1998) suggested the lower portion of the South Valley was underlain by glaciolacustrine deposits including silts and deltaic sands, which were subsequently incised by glaciofluvial flows through the South Valley. The presence of such sediments in the area upstream of the South Dam was confirmed by geotechnical drilling carried out under AMEC's supervision in 2004. Drilling and additional sampling undertaken in 2010 along the proposed dam alignment however is more indicative of a glaciofluvial and/or deltaic depositional environment, rather than glaciolacustrine, given the lack of silt layers. Further, as discussed subsequently, these units are all dense to very dense.

On the east slopes above the South Dam site, several curvilinear features were mapped based on airphoto analysis that resembled possible slide scarps. In addition, possibly bulged terrain occurs on the lower part of the slope. It was also thought that these could be conglomerate beds based on a geological map by Evenchick and Green (1995). Preferential erosion of weaker sediments between near vertically-dipping conglomerate beds could produce the surface expression noted on the 1:60,000 airphotos. When this area was ground-truthed, no signs of modern slope activity were noted such as fresh steep soil or rock exposures, disturbed trees or elongate wet areas with young vegetation as would be evident in areas experiencing movement. The low areas between the ridges were swampy and followed gentle subdued contours between the rock exposures. The bedrock encountered on the elongate ridges was a moderately weathered, very thickly bedded, reddish-brown, moderately porous, strong (R4) chert-pebble conglomerate of the Bowser Lake Group. The bedding strike (055) could only be determined at one of the field sites where a series of ridges were exposed, the dip could not be determined with the weathered and rounded exposures, however it appears to be sub-vertical in this area.

To the south of the open pit, and well away from the proposed tailings impoundment, is an area identified as the Kluea Lake landslide. Aerial and ground reconnaissance of this area was undertaken by AMEC in July 2004. The area around the slide was investigated for tension cracks and any other indicators that might suggest whether the slide may still be active. The slide escarpment above Kluea Lake extends approximately 6 kilometres from just east of the two peaks to the headwaters of an unnamed creek, which flows to a fan dividing Kluea and Todagin Lakes. This unnamed creek appears to have been pushed over by the slide causing toe erosion to the slope on the opposite side of the creek resulting in a very steep slope. The slide escarpment cuts obliquely across bedding exposing predominantly clay shales of the Bowser Lake Group. Tension cracks were observed while walking the areas north and west of the peaks south of the existing exploration camp. The cracks were well developed, several metres deep with a V-notch shape and a gentle convex slope extending away from the crack towards the slide. Most of the tension cracks did not have any mature vegetation in them, only grasses and moss. The rocks exposed on the peaks south of the camp are strong, chert-pebble conglomerate of the Bowser Lake Group.

There are no indications of such slide activity in the vicinity of the South Dam, the South Seepage Dam, and the seepage pumpback well field.

19.1.3 Northeast Valley

The proposed Northeast Dam area is located near the headwaters of the Northeast Valley; a valley with predominately gentle side slopes at approximately 10 to 15 degrees. A drainage divide at the headwaters of the valley separates surface waters destined for Kluea Lake via the Northeast Valley from flows towards the Klappan River. The alignment of the proposed Northeast Dam is at this drainage divide.

The terrain in the Northeast Valley includes discontinuous organic deposits in the valley bottom overlying or adjacent to fluvial deposits, which appear (based on a single borehole near the proposed Northeast Dam site) to overlie glacial till and glaciofluvial deposits. The upper valley slopes have till mantle deposits, which thin towards the upper portions of the slopes. Based on borehole and test pit information from investigations in 1995 and 2010, the valley is infilled with surficial materials including alluvium, glaciofluvial sediments, glacial till, and pre-glacial fluvial deposits indicating multiple glacial advances. In general, the shallow till unit appears both thicker and more continuous in the vicinity of the Northeast Dam alignment than is the case in the North and South valleys, although the shallow till is either absent, or present at depths of greater than 15 m, further to the southwest. A thick (about 50 m), deep till unit was encountered below the alignment of the Northeast Dam. Till was also encountered to a thickness of about 22 m in a borehole (09-B02) at the junction between the main north-south valley and the Northeast Valley.

The depth to bedrock along the Northeast Dam alignment was inferred, on the basis of a seismic refraction survey undertaken in 2004, to be up to about 70 m below the west abutment, to about 25 m below the east abutment. However, 2010 borehole BH-10-104 encountered overburden (predominantly till) to a depth of 103 m without contacting bedrock. Given the very dense state of the overburden soils, the high seismic velocity was misinterpreted in 2004 as bedrock.

Well defined gullies exist on the south side of the Northeast Valley near the confluence of the Northeast and South Valleys. As mentioned in the South Valley section above, these gullies do not show any obvious indications of sidewall instability and ravelling such as fresh unvegetated areas or linear scars in gully sidewalls and the upper slopes appear to be covered with vegetation. As outlined previously, the potential for debris flows and sedimentation appears to be lower along these gullies than elsewhere in the area. The lineaments described above in the South Valley section are also located upslope of the lower reach of the Northeast Valley.

19.2 Geotechnical and Hydrogeologic Site Investigations

Several geotechnical, geophysical, and hydrogeologic site investigation campaigns have been undertaken within the proposed TSF. Investigations undertaken up to and including 2004 were documented within the EA submission and also in the Feasibility Study of 2005. Subsequent investigations (2009 and 2010) are documented in the Joint Application for the Mines' Act Permit and Effluent Discharge Permit Application on the basis of an interim factual report that will be updated upon completion of the summer test pit programs, and further analysis of the data.

Table 19.1 Summary of Tailings Facility Investigations

Year	Geotech. Boreholes	Condemnation Boreholes	Pump Wells	Pump Tests	Test Pits	Seismic Refraction Lines	Transient EM Survey Lines
1995	3				14		
2003	4				16		
2004	5		1	1	1	3	5
2009	2						
2010	28	8	2	2		6	
Totals	*42*	*8*	*3*	*3*	*31*	*9*	*5*

Note: A further test pit program was completed during the summer of 2010

19.2.1 1995 Investigations

Knight Piesold Ltd. carried out the initial site investigation in 1995. The objective of this investigation was to obtain preliminary information on surficial materials and foundation conditions at the proposed plant site location and the TSF. Three boreholes and 14 test pits were excavated at the tailings facility and groundwater monitoring wells were installed in each of the boreholes. The results of this investigation indicated that there are extensive deposits of glaciofluvial sand and gravels within the valleys and along terrace flats adjacent to the valleys. Glacial till was found to lie along the slopes at higher elevations and at the base of slopes adjacent to the valley.

19.2.2 2003 Investigations

The 2003 investigation, also performed by Knight Piesold Ltd., focused on obtaining more detailed information on the geotechnical conditions at the proposed rock storage area dump site and the tailings facility. Two drill holes and six test pits were excavated at the rock storage area site and 4 boreholes and 16 test pits were excavated at the TSF location.

The overburden soils at the TSF were identified and classified into four soil general stratigraphic units on the basis of the 2003 program:

- Surficial organic layer
- Alluvial and Glaciofluvial Sand and Gravel
- Glacial Till
- Deep Sand with Silt

The majority of the near surface overburden in the valley was identified to comprise alluvial and glaciofluvial sand and gravel underlain and occasionally interbedded with low permeability glacial till. The glacial till generally lies below the sand and gravel deposit but was exposed at surface in some areas (e.g. east valley slope to the upstream of the South Dam). The deep sand with silt was characterized as a consolidated, moderately dense to very dense layer (as confirmed by Standard Penetration Testing (SPT) carried out in the 2003 drilling program) that was encountered in each of the boreholes. The layer was only fully penetrated in one drill hole (DH-03-03, south of the proposed North Dam alignment) where an underlying till layer was encountered at a depth of 60m.

19.2.3 2004 Investigations

The 2004 site investigation, undertaken under the supervision of AMEC, involved:

- Two geotechnical boreholes upstream of the South Dam alignment (the initial intent of drilling on the alignment was not achieved due to site access constraints). The boreholes were between 20 m and 30 m in depth.
- Three geotechnical boreholes within the impoundment area, 26 m to 33 m in depth.
- One pumping well was installed into the lower aquifer in the vicinity (upstream) of the South Dam. A pump test was conducted using this well to assess the hydraulic conductivity and transmissivity of the upper sand and gravel aquifer, and to assess the connectivity of the upper and lower aquifers, an indicator of the continuity of the till layer.
- One hoe-excavated test pit at the South Dam site within the impoundment area.
- Seismic refraction geophysical surveys (lines S1, S2 and S3) along the axes of each of the dams, carried out by Frontier Geosciences of North Vancouver, BC. The Frontier report is included within AMEC (2010).
- Transient electromagnetic geophysical surveys (conducted by Frontier) transverse to the dams axes (EM1, EM3 and EM5) and at two cross-valley transects (EM2 and EM4) within the TSF, to evaluate the continuity and extent of the glacial till layer that, depending upon its continuity, would function to some extent as a natural low hydraulic conductivity "liner" within the tailings impoundment.

The original scope of the 2004 investigation for the TSF included 12 boreholes and 18 test pits in the tailings basin. However, due to the complexity and challenges of developing access trails to the borehole and test pit sites on valley slopes, particularly at the north and northeast dams, and exacerbated by particularly wet weather, it was decided to carry out the investigation of these sites in a subsequent investigation These investigations were undertaken in the winter of 2010.

The 2004 investigations confirmed the general overburden stratigraphic model, with an upper aquifer of alluvial sands and gravels, overlying a discontinuous, low hydraulic conductivity till unit of variable thickness, overlying in turn a predominantly sand aquifer.

19.2.4 2009 Investigations

The 2009 investigations within the TSF area comprised two boreholes drilled under the direction of Knight Piesold. These boreholes, 09-B01 and 09-B02, were drilled within the impoundment area to improve characterization of potential borrow areas. Borehole 09-B01, drilled to the west of Black Lake, encountered sands and gravelly sands to the borehole termination at 64 m depth. Borehole 09-B02, drilled to the east of Black Lake at the junction of the main north-south and the northeast valleys, encountered 10 m of sand and gravel underlain by about 22 m of glacial till, underlain in turn by about 22m of sand.

19.2.5 2010 Investigations

The investigations undertaken in 2010 are to support final design of the tailings facilities (dams, seepage collection facilities, runoff diversions, and spillways). The 2010 investigations were undertaken in two phases: winter and summer. Both of these field programs are now complete. The summer program involved approximately 30-50 test pits, with subsequent laboratory testing of selected samples.

Data gaps from previous investigations specifically targeting in the winter phase of the 2010 investigations were as follows:

- Obtain additional SPT data to confirm density of the overburden soils and provide greater confidence in conclusion from previous investigations that the granular overburden soils are sufficient dense as to preclude any potential for seismic liquefaction.
- Increase drilling coverage at the North Dam, and confirm the optimal alignment for that dam.
- Increase drilling coverage at the Northeast Dam, where previously only a single borehole had been completed.
- Undertake additional pump tests to improve characterization of impoundment area hydrogeology, for use in updating of the MODFLOW groundwater model developed for the project during the EIA.
- Confirm depths to bedrock (indicated previously by the interpretations of the 2004 seismic refraction surveys) along the dam alignments via drilling in the valley bottom, previous drilling programs having been unable to reach bedrock.
- Undertake drilling and coring in the abutment areas where bedrock is relatively near surface, including packer hydraulic conductivity testing to estimate hydrogeologic parameters for the bedrock.
- Undertake additional seismic refraction surveys along the alignments of the two seepage dams (one downstream of each of the North and South Dams), and along additional alignments in the North Dam area.

- Undertake additional seismic refraction surveys within the Northeast Valley that likely represents a significant borrow area for dams construction.
- Obtain better definition as to the thickness, extent, and continuity of till units within the impoundment area.
- Provide confirmation as to the lack of glaciolacustrine silts/clays in the foundation soils which, if present, with clay varves of high plasticity, would represent a potential low strength foundation unit controlling dam stability.
- Estimate shear wave velocity profiles in the overburden soils via multi-spectral analysis of surface waves (MASW), the data obtained from the seismic refraction surveys.

All of these objectives were achieved.

The objectives for the summer phase of the 2010 program comprised a test pit program, as follows:

- Determine geotechnical conditions along proposed runoff diversion alignments.
- Further define borrow areas, and obtain additional samples for laboratory testing.
- Obtain additional data as to presence of the shallow till unit.

19.2.6 Key Findings from 2009/2010 Investigations

Key findings from that program, and from the two boreholes completed in the impoundment area in 2009, are summarized as follows:

- The general overburden stratigraphic model for the impoundment area is essentially the same as that defined in previous programs, with the exceptions that two discontinuous till units are present: the shallow, and relatively thin, upper till unit identified in previous site investigations, and a deeper, somewhat thicker lower till unit identified in the Winter 2010 drilling program.
- A deep silt unit, likely of glaciolacustrine origin, was encountered in at depth, in one borehole, below the lower aquifer.
- Apart from isolated areas within a few meters of existing ground surface in the immediate vicinity of the active creeks, the soils are in a dense to very dense state and are non-susceptible to liquefaction given the seismicity of the site. The shear wave velocities derived from the MASW profiling confirm the dense to very dense state of the foundation soils as indicated by the SPT data.
- The discontinuous nature of the upper glacial till unit has been confirmed. The unit is absent in significant areas along the alignments of the North and South Dams. It does appear continuous in the area of the Northeast Dam. As such, the option of keying a till core cutoff into this unit is infeasible (where the upper till is absent), and of minimal benefit (where the upper till unit exists). It is achievable for the Northeast Dam, and remains the design basis for that dam.
- While a deep silt of apparent glaciolacustrine origin (could also be till, but SPT sampling was not possible given the depth so logging was via drilling cuttings) appears to have been encountered at a depth of about 93 m in the valley bottom upstream of the North Dam alignment, this unit was not encountered below the North

Dam itself. Even if the unit includes clay varves, at such depths such features would be immaterial to the stability of the dam.

- The Northeast Valley appears the most suitable and viable location for till borrow.
- Bedrock at shallow depths on the dam abutments is generally of good quality, and packer testing indicates moderate to low hydraulic conductivities, particularly on the abutments of the selected alignment of the North Dam. The packer testing (and inspection of the core from the condemnation boreholes) confirms that bedrock will not be a preferred or even significant seepage pathway. Seepage will be dominated by the upper and lower aquifer units as the principal pathways.

The updated general stratigraphic model for the impoundment area is summarized in Table 19.2. Interpreted geologic profiles are provided on Figures 19.1 to 19.5. Given that all of the foundation units are of high shear strength, the definition of the stratigraphic units was made with a focus on hydrogeology, with aquifers (sand and gravel units) and aquitards (glacial till units, and deep silt unit) governing the delineation of the units. This approach represents a considerable simplification of the actual site geology which is clearly complex, and does not account for the variability in gradation of the units, for example the lower aquifer which ranges in gradation from a sand with trace silt, to gravel and cobbles with sand, and varies in fines content (% by dry weight finer than 0.074 mm) from less than 5% to up to 25%. The upper till unit, which in terms of seepage represents the most significant of the two till units, has been represented as discontinuous on the interpreted sections and profiles.

Table 19.2: General Stratigraphic Model

Unit	Occurrence	Geotechnical Parameters	Comments
Organic and swamp / lacustrine deposits	Poorly drained valley bottom areas, within the main north-south valley, and within the lower elevations within the Northeast Valley. Will also exist within Black Lake. The thickness of this unit ranges from one to perhaps several meters in thickness (Black Lake).	Not investigated, other than visual classifications of soils from test pits upstream of the South Dam. Additional investigations planned for Summer 2010. These soils are however of relatively low hydraulic conductivity.	Strength properties of this unit is not relevant to design of the dams (such soils would be removed from dam alignments as part of foundation preparation. However, these deposits will, particularly once consolidated under a significant depth of tailings, be of low hydraulic conductivity and limit the rate of seepage from tailings into the underlying aquifers.

Upper aquifer (sand and gravel)	The upper aquifer is present essentially everywhere within the lower elevations of the valleys, below about El. 1160 m to 1170 m. In a few isolated areas, this unit is absent with the upper till unit exposed essentially at surface, notably in the area of the Northeast Dam.
	
	The thickness of this unit varies from about 1 – 20 m in thickness. Where the upper till is absent, it is difficult to identify the demarcation between the upper and lower aquifers.
		Based on 2003 Knight Piesold lab testing, and 2004/2010 AMEC lab testing:
		• Specific gravity = 2.68 to 2.8
		• Grain size distribution as follows:
		• Gravel (> #4 sieve) – 40% to 70%
		• Sand (#200 to #4 sieve) – 20% to 60%
		• Silt and clay (finer than #200 sieve) – 0.1% to 17%
		• Effective shear strength:
		  o Effective cohesion (c') 0 to 10.5 kPa
		  o Effective friction angle (ϕ') 38.1 to 40.2°
		• Hydraulic conductivity (k) = 10^{-5} m/sec
		SPT blowcounts: typically > 20, most > 30, but some values < 10 within several meters of original ground surface, likely where there has been some reworking due to fluvial action.
			This unit is generally of high hydraulic conductivity. Where the upper till unit is absent, this unit will recharge the underlying, thicker, and more extensive lower aquifer.
			
			Apart from upper 2-3 meters in some areas (most notably along the creek channel at the South Dam), this unit is not susceptible to seismic liquefaction given the site seismicity. Looser near surface materials can be removed from below the dam footprints.
			
			This unit represents a good borrow source for general dam fill (for starter dams), and for under-drainage/finger drains. This unit also represents a potential aggregate source for concrete production.

Upper till	The upper till unit is discontinuous. It was not encountered in many of the test pits although given excavator limit of 5 m, this might mean that at some locations the thickness of the upper aquifer exceeds the reach of the excavator. Notable areas of absence include much of the alignment of the South Dam (particularly west of the creek), in general the west slope of the valley (at lower elevations), and the North Dam alignment, again primarily to the west side. The thickness of this unit ranges from 1 to 4 m in thickness. However, in some areas significantly higher thicknesses were encountered. For example, downstream of the Northeast Dam, borehole BH010-04 encountered till from surface to a depth of 55 m. Borehole 09-B02 encountered the unit to a thickness of 10 m.	Based on 2003 Knight Piesold lab testing, and 2004/2010 AMEC lab testing: • Specific gravity = 2.75 • Grain size distribution: • Gravel (> #4 sieve) – 15% to 40% • Sand (> #200 sieve, < #4 sieve) – 26% to 38% • Silt (< #200 sieve, > 0.002 mm) – 13% to 32% • Clay (< 0.002 mm) – 11% to 20% • Standard Proctor Test results: o Maximum dry density = 2119 kg/m^3 o Optimum moisture content = 8.4% • Effective shear strength parameters: o c' = 8.6 kPa o ϕ' = 40.3º • Hydraulic conductivity – 10^{-8} m/sec SPT blowcounts: > 30, some reached refusal.	Seismic refraction and Transient Electromagnetic (TEM) surveys have proven ineffective in delineation of the upper till unit, due to the lack of seismic velocity contrast with the underlying lower aquifer, and given that the upper till unit is relatively thin. This unit is not potentially liquefiable. It is dense to very dense. This unit is significant from a hydrogeologic perspective, in that: • Where present, it greatly reduces the rate of recharge from the upper aquifer to the lower aquifer. • Its overall extent limits the area of direct contact between the upper and lower aquifers. This unit represents a borrow source for construction of the low hydraulic conductivity core zones for the North and Northeast Dams.
Lower aquifer (sand with silt)	This unit appears continuous within the impoundment area, below about El. 1150 m. The unit is up to 80 m in thickness. For the most part, and especially in the area of the North Dam, this unit is predominantly sand with trace to some silt. However, there are also areas where there is significant gravel and cobble content at depth. Further, fines contents, and therefore hydraulic conductivity, in this unit are quite variable	Based on 2003 Knight Piesold and 2004.2010 AMEC lab testing: • Specific gravity = 2.71 to 2.78 • Moisture contents = 9-25%, with 3 of the 5 samples yielding moisture contents in the range of 24-25% • Grain size distribution: o Gravel (> #4 sieve) – typically < 3% o Sand (< #4 sieve, > #200 sieve) – 40% to 93%, with 4 of the 5 tests greater than 70% o Silt and clay (< #200 sieve) – 5% to 30% • Effective shear strength parameters: • c' = 13.9-19.2 kPa • ϕ' = 38.2-38.4º • Hydraulic conductivity: 10^{-5} m/sec SPT blowcounts: > 30, most > 50.	This unit represents the principal aquifer and pathway for seepage. The unit is not potentially liquefiable. It is dense to very dense.

Lower till	The lower till unit is discontinuous, and was not encountered in many of the boreholes. It lies within the overall lower aquifer.	Properties as per the upper till unit.	This unit is likely not significant from a hydrogeologic perspective, given its apparent lack of continuity, particularly given that it appears to occur as discontinuous lenses within the overall lower aquifer.
Glaciolacustrine silt	This unit was encountered at a depth of 93 m only in BH-10-302, upstream of the North Dam.	Properties likely approximate those of the till units.	May be deep till unit, but split spoon sampling could not be undertaken to confirm owing to the depth.
Bedrock		Packer tests in abutment holes yielded K values in the range of 10^{-7} m/sec to 10^{-10} m/sec.	

Figure 19.1 Interpreted Geologic Profiles: North Dam Area

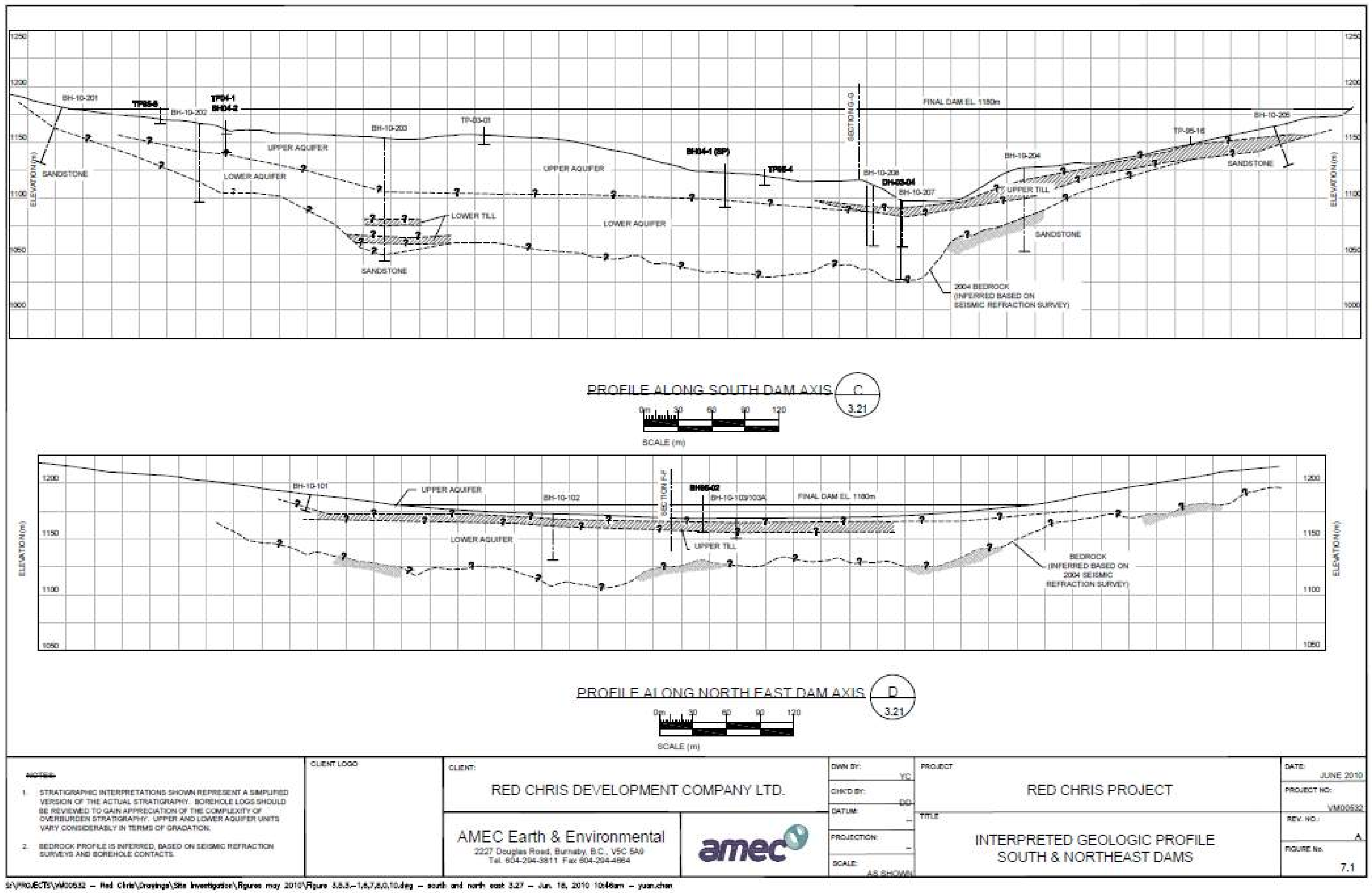


Figure 19.2 Interpreted Geologic Profiles: South and Northeast Dams

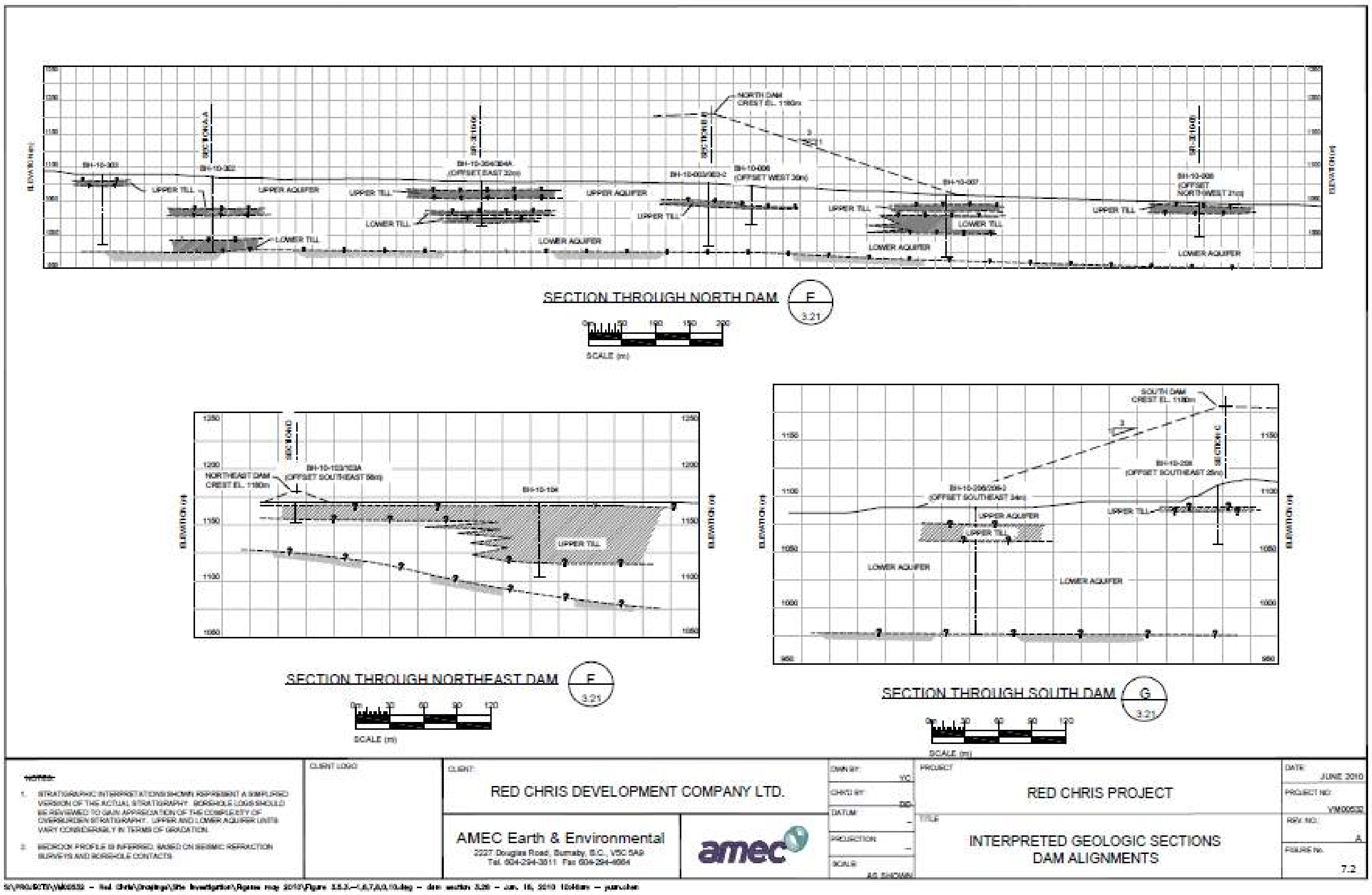


Figure 19.3 Interpreted Geologic Sections: Dam Alignments

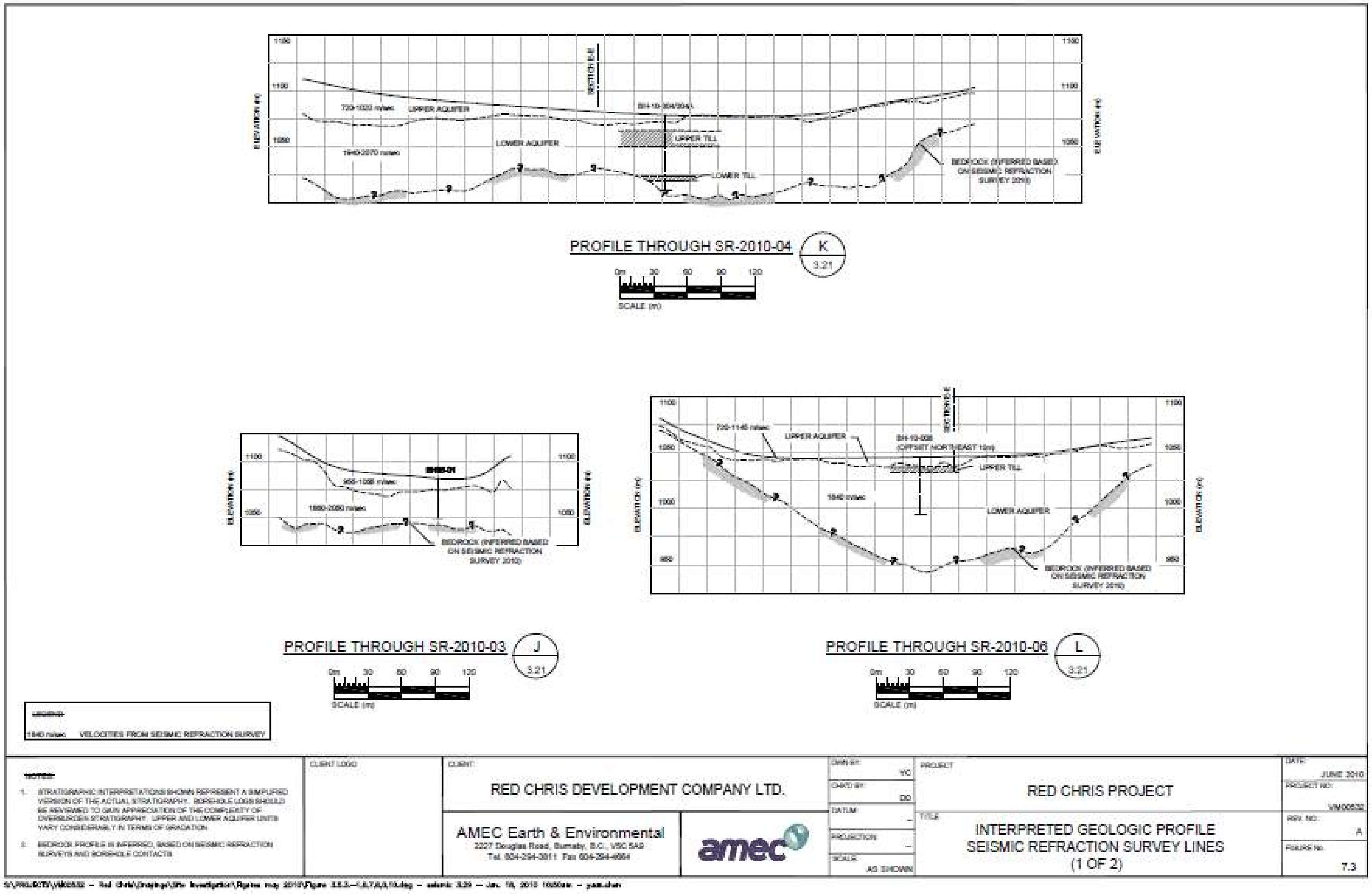


Figure 19.4 Interpreted Geologic Profiles: Seismic Refraction Survey Lines (1 of 2)

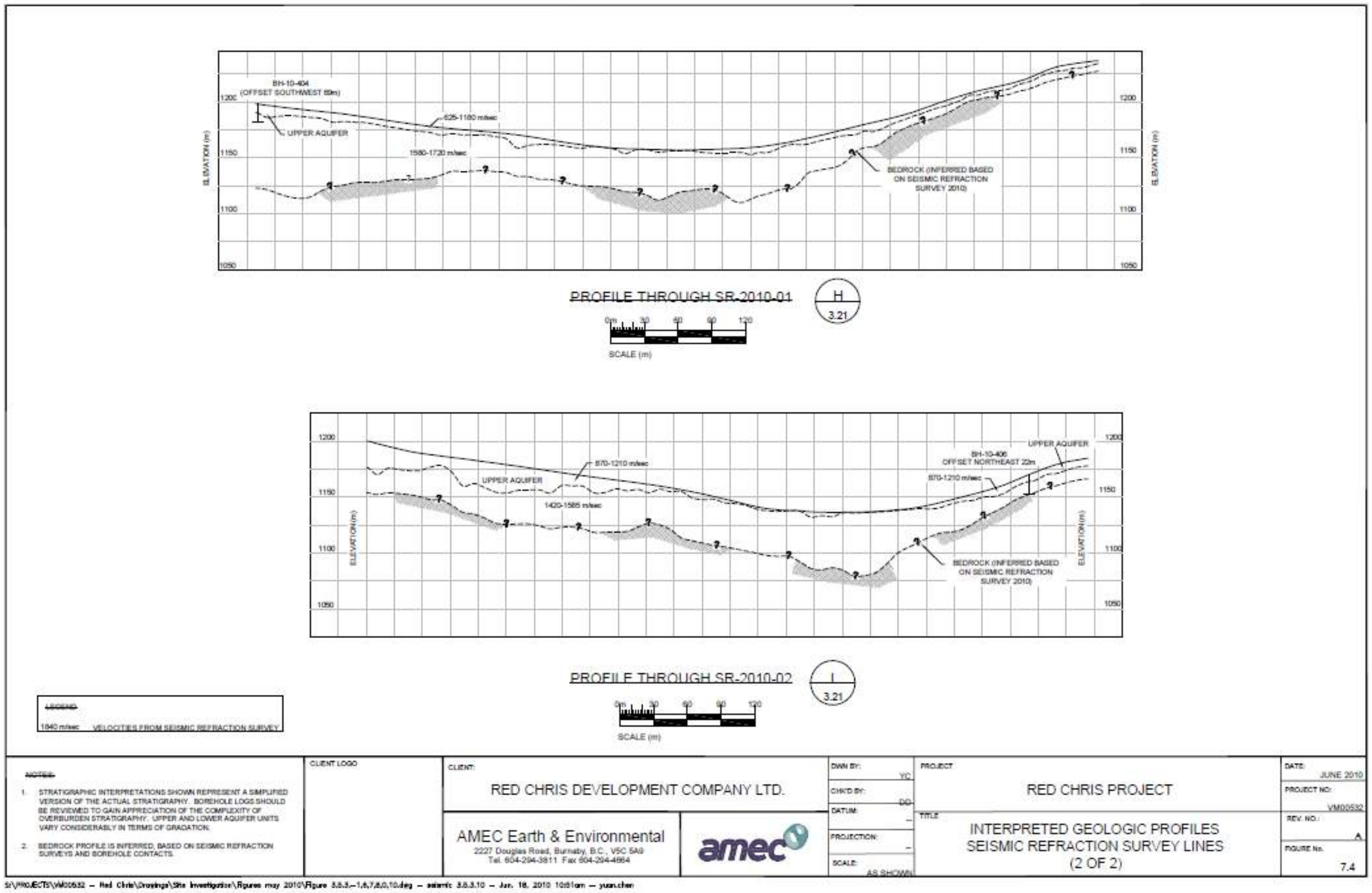
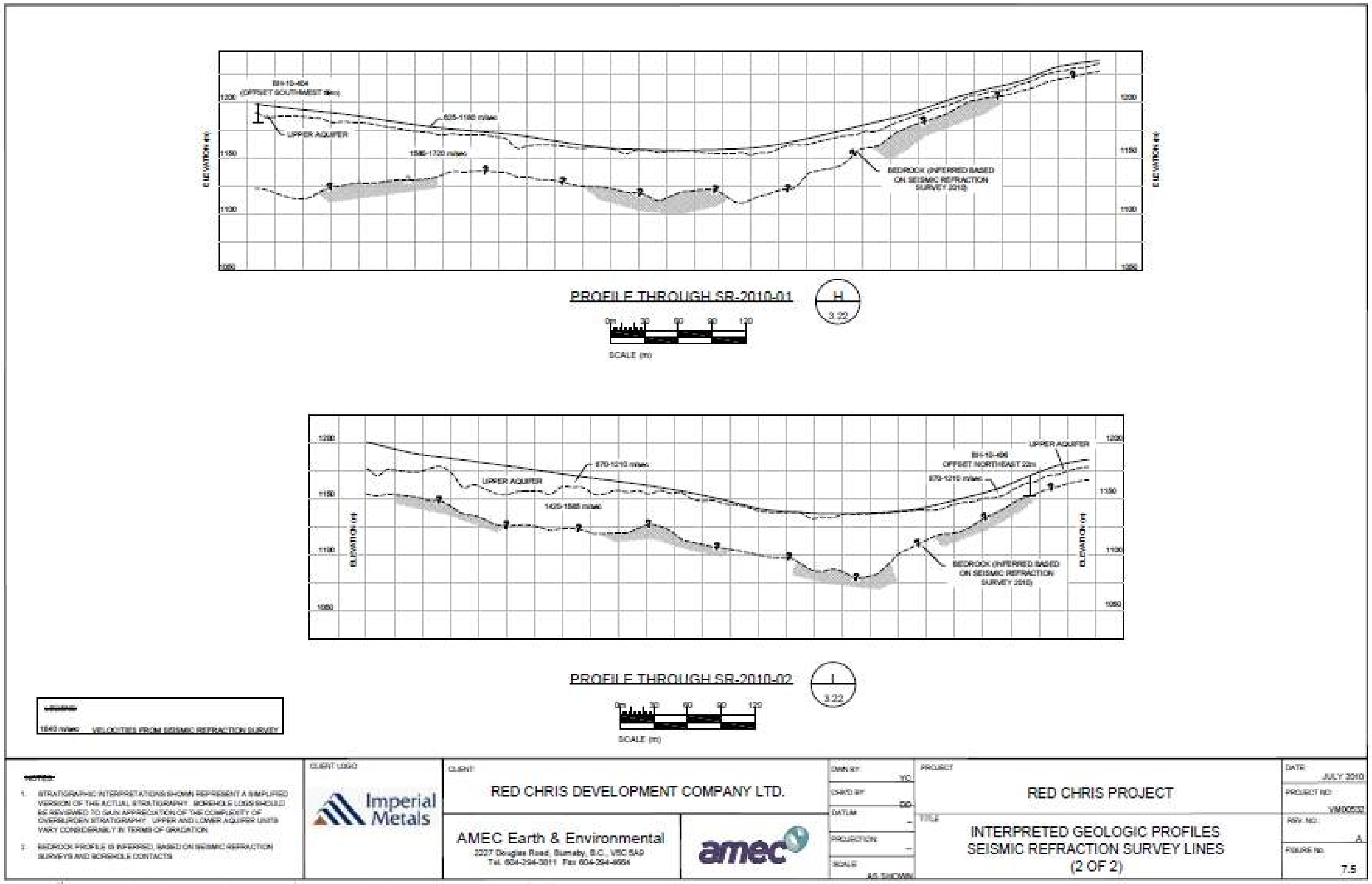


Figure 19.5 Interpreted Geologic Profiles: Seismic Refraction Survey Lines (2 of 2)



19.3 Tailings Facilities and Associated Infrastructure

Figure 1.4 shows an overall site plan that includes all key elements of the tailings facilities and associated infrastructure. These are summarized in Table 19.3.

Table 19.3 Tailings Storage Facilities and Infrastructure

Component	Function	Comments
North Dam	Tailings containment, and process water containment for start-up	Central till core, upstream liner, and downstream shell of cyclone sand and under-drainage. Ultimate height about 100 m.
Saddle Dam	Provide for 2-3 years of tailings storage in North Valley, deferring South Dam construction	Upstream till section and shell comprised of sand and gravel. The dam will be about 15 m in height.
South Dam	Tailings containment	Cyclone sand dam with sand/gravel starter dam, with under-drainage. The ultimate dam height will be about 80 m.
Northeast Dam	Water pond containment	Till core and cutoff, with upstream riprap and downstream shell of sand and gravel. Ultimate height about 10 m.
North Sedimentation Pond, South Sedimentation Pond	Settle out fines from hydraulic sand fill placement	Sand and gravel berms to provide primary settling of suspended fines from sand prior to water reaching Reclaim Dam ponds.
North Reclaim Dam, South Reclaim Dam	Retain cyclone sand drainage, dam seepage, and runoff for reclaim to the tailings pond	Till core, sand/gravel shells, and possibly a liner within the pond area, depending on findings of Summer 2010 test pit program.
North Seepage Pumpback Wells, South Seepage Pumpback Wells	Capture seepage to supplement reclaim water during early years before tailings blanket in impoundment serves to reduce seepage.	Preliminary plans are for 8 pumpback wells downstream of North Dam, and 6 downstream of South Dam. To be refined as detailed design proceeds.
North Monitoring Wells, South Monitoring Wells	Allow for groundwater sampling downstream of the seepage pumpback wells	Wells likely to be installed to depths of 50 m.
North Reclaim Pumps and Pipeline, South Reclaim Pumps and Pipeline	Return water from the downstream reclaim ponds to the tailings impoundment.	Discharge will be immediately upstream of the North and South Dams.
Reclaim Water Barge and Pumps	Reclaim process water from the tailings pond to the mill.	Pumps on reclaim barge will also be used for discharge of surplus water to Quarry Creek watershed.
Reclaim Water Pipeline	Pipeline from Reclaim Barge to the mill	
Surplus Water Discharge Pipeline	Pipeline extending from Reclaim Water Barge to outfall downstream of North Reclaim Dam	
Surplus Water Outfall Structure	Armouring and stilling basin(s) as appropriate for surplus water discharge	
Rougher Tailings Pipeline	Pipeline for de-pyritized rougher tailings from mill to splitter box	There may also be a standby line. Rougher tailings will be not potentially acid generating (NAG).
Cyclone sand plant	Separation of de-pyritized rougher tailings to cyclone sand underflow (coarse fraction) and overflow (fine fraction). Underflow used for dams construction.	Located approximately midway between the North and South Dams, below the process plant area, at a grade sufficient to provide for gravity flow underflow and overflow to the dams
Underflow splitter box and pipelines	To apportion cyclone underflow tailings to North and South Dams	Sand used for downstream shells construction and upstream beaching as needed. Flow by gravity.
Overflow splitter box and pipelines	Apportion overflow tailings flow to North and South Dams	Overflow used for upstream beaching. Flow by gravity.
Cleaner Tailings Pipeline	Transport potentially acid generating	For cleaner tailings and sulphides rejected from

	(PAG) cleaner tailings stream from mill to central portion of impoundment	additional flotation of rougher tailings. May also be a standby line.
Runoff Diversions	Divert non-contact water from the tailings impoundment so as to minimize the net annual water balance surplus requiring discharge from the tailings pond.	Discussed in Section 8.
Dams Instrumentation	To monitor pore pressures and displacements of the dams.	Required for the three main dams, and the two reclaim dams.
Operational Spillways	Allow for discharge of extreme runoff inflow events to prevent overtopping of dams.	Required for the two reclaim dams. The tailings impoundment will be sized to accommodate, without release, the runoff from a 30-day duration PMF, although a spillway will be provided for the starter impoundment.
Closure Spillway	On abutment of Northeast Dam to allow for release from closed tailings impoundment.	
Reclamation Cover	Covering over the North, South, and Northeast Dams to support revegetation and to provide erosion protection.	To include armouring where cyclone sand downstream shells in contact with valley slopes.

19.4 Key Design Criteria

19.4.1 Consequence Classification per CDA Guidelines

The tailings dams and the two reclaim dams will be designed and constructed in conformance with the 2007 Canadian Dam Association (CDA) guidelines.

Table 19.4 provides the consequence classification scheme per the CDA (2007) guidelines. On the basis of this scheme, the setting of the Red Chris tailings impoundment, and given that a portion of the impounded tailings will be potentially acid generating, the dams are assigned the following consequence classifications:

Table 19.4 Classification of Impoundment areas as per CDA guidelines

Area	Classification
North Dam	Very high
South Dam	Very high
Northeast Dam	High
North Reclaim Dam	Low
South Reclaim Dam	Low

19.4.2 Dam Stability

Per the CDA guidelines, the minimum limit equilibrium factors of safety (FoS) for differing loading conditions will be as follows:

Table 19.5 CDA guidelines for factors of safety

CDA guidelines limit equilibrium factors of safety	
Static loading, long term conditions:	> 1.5
Short term, construction conditions:	> 1.3
Design earthquake loading conditions:	
Pseudostatic	> 1.0
Post-earthquake	> 1.2

19.4.3 Design Earthquake Parameters

Given the "very high" consequence classification for the North and South Dams, the CDA guidelines require that these dams be designed to accommodate the loading associated with a 1 in 5,000 year return period seismic event. However, the design event selected during the EIA was the Maximum Credible Earthquake (MCE), which can be approximated probabilistically as the 1 in 10,000 year return period event. The MCE is retained as the design earthquake for the design of the Red Chris tailings impoundment.

The Red Chris site is located within the Northern British Columbia (NBC) source zone in the GSC-R seismic source zonal models developed by the Geological Survey of Canada for the recently updated National Building Code of Canada-NBCC 2005 (Adams and Halchuk, 2003). The results of a seismic hazard calculation based on the NBCC (2005) seismic hazard models (see http://earthquakescanada.nrcan.gc.ca/hazard-alea/interpolat/index-eng.php) are given in Figure 19.6, along with the EZFRISK (see http://www.ez-frisk.com/) analysis results for the GSC R and H source zone models. These analyses indicate a median (50th percentile) PGA of about 0.11g for a 10,000 year recurrence interval, taken to represent the MCE. This is lower than the value of 0.19g that was obtained during the EIA, on the basis of the previous seismic hazard models, carried out by the Pacific Geoscience Center, Geological Survey of Canada.

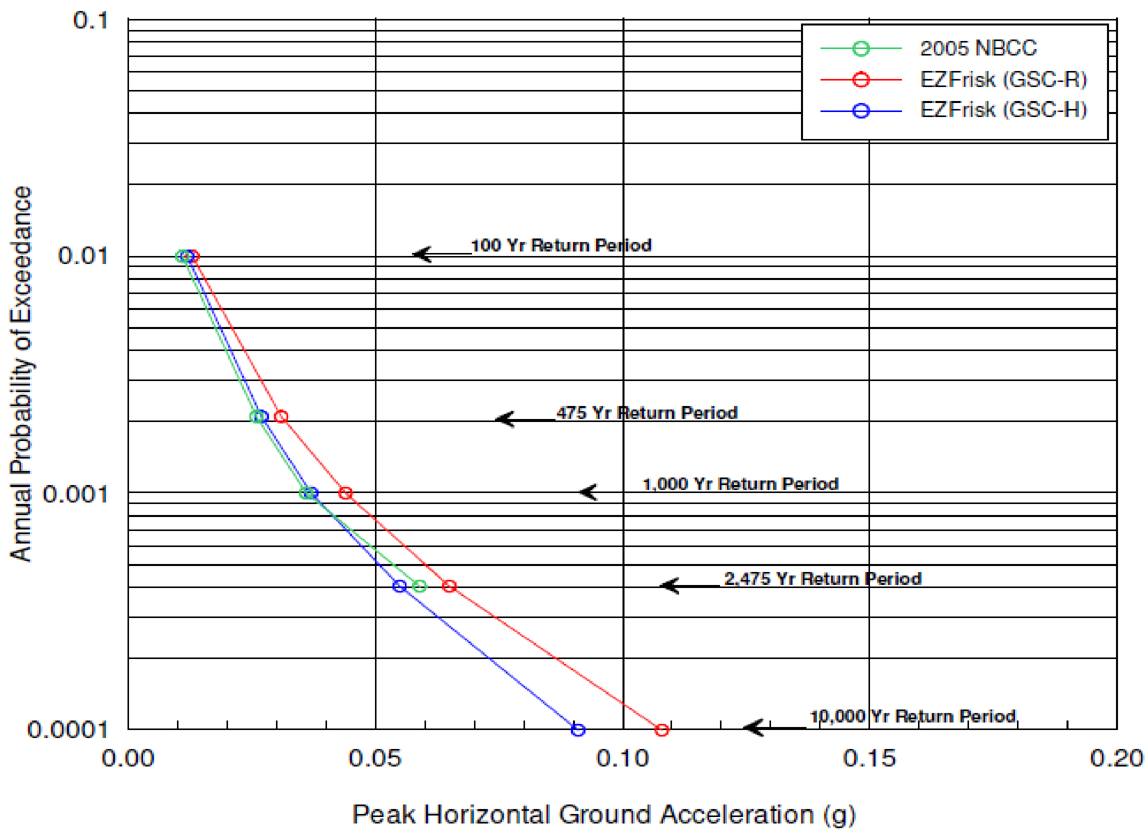
The Red Chris site is located within an area of low historical seismicity, as illustrated on Figure 19.7.

The design earthquake parameters selected for the tailings dams design during the EIA were as follows:

- Moment magnitude 6.0
- PGA 0.19g

Conservative parameters given above from the EIA have been used in seismic analysis of the tailings dams. Seismic design parameters will be updated and incorporated into the analyses presented in the detailed design report for the tailings facility. Further site seismicity is not a driver of the design of the dams, and the modelled behaviour of the dams under the conservative MCE parameters given above is acceptable.

Figure 19.6 NBCC (2005) and EZFRISK Probabilistic Seismicity Assessments



Annual Probability of Exceedance	Return Period (years)	Peak Ground Acceleration (g)			
		NBCC 2005	EZ Frisk (mean)		
			GSC-H	GSC-R	Robust[a]
0.0100	100	0.011	0.012	0.013	0.013
0.0021	475	0.026	0.027	0.031	0.031
0.0010	1000	0.036	0.037	0.044	0.044
0.0004	2475	0.059	0.055	0.065	0.065
0.0001	10000	-	0.091	0.108	0.108

[a] Robust Model value is simply the higher value determined using the GSC-R and GSC-H Models

19.4.4 Inflow Design Flood Event for Tailings Impoundment

For dams with a "very high" consequence classification, the CDA (2007) guidelines specify that the Inflow Design Flood (IDF) should lie within a range two-thirds between an annual exceedance probability of 1 in 1,000 to the Probable Maximum Flood (PMF) event. For the Red Chris tailings impoundment, it is the PMF that will apply.

Two PMF events will govern design, depending on whether or not each stage of the dam raising incorporates an emergency spillway.

- **No spillway –** for each stage raise without a spillway, there must be sufficient storage within the tailings impoundment to store the full PMF without overtopping of the dams. A long duration PMF is appropriate for such circumstances, and the 30-day PMF has been selected as the appropriate event, given the ability to use the surplus water discharge system that will be in place at all times. This is a long duration, but relatively low intensity, inflow event, during which the diversion ditches would remain functional. The governing 30-day PMF is the May-June event coincident with a 1:100 year snowmelt event. For the feasibility-level design of the tailings impoundment, this was estimated to represent a total 1124 mm of runoff. Given the subsequent reassessment of site hydrology, this will represent an overestimate of the 30-day PMF. Reanalysis of the revised site hydrology will be undertaken to finalize the runoff associated with this event for incorporation into the detailed design of the facility. The no spillway case will also be checked for the 24-hour duration PMF, with failure of the diversion ditches at the onset of the event being assumed.
- **With spillway –** for each stage raise where a spillway is in place, and in particular for the closure configuration of the impoundment, the 24-hour duration PMF event will apply. This event is estimated as 226 mm of runoff, which includes a 100-year snowmelt component.

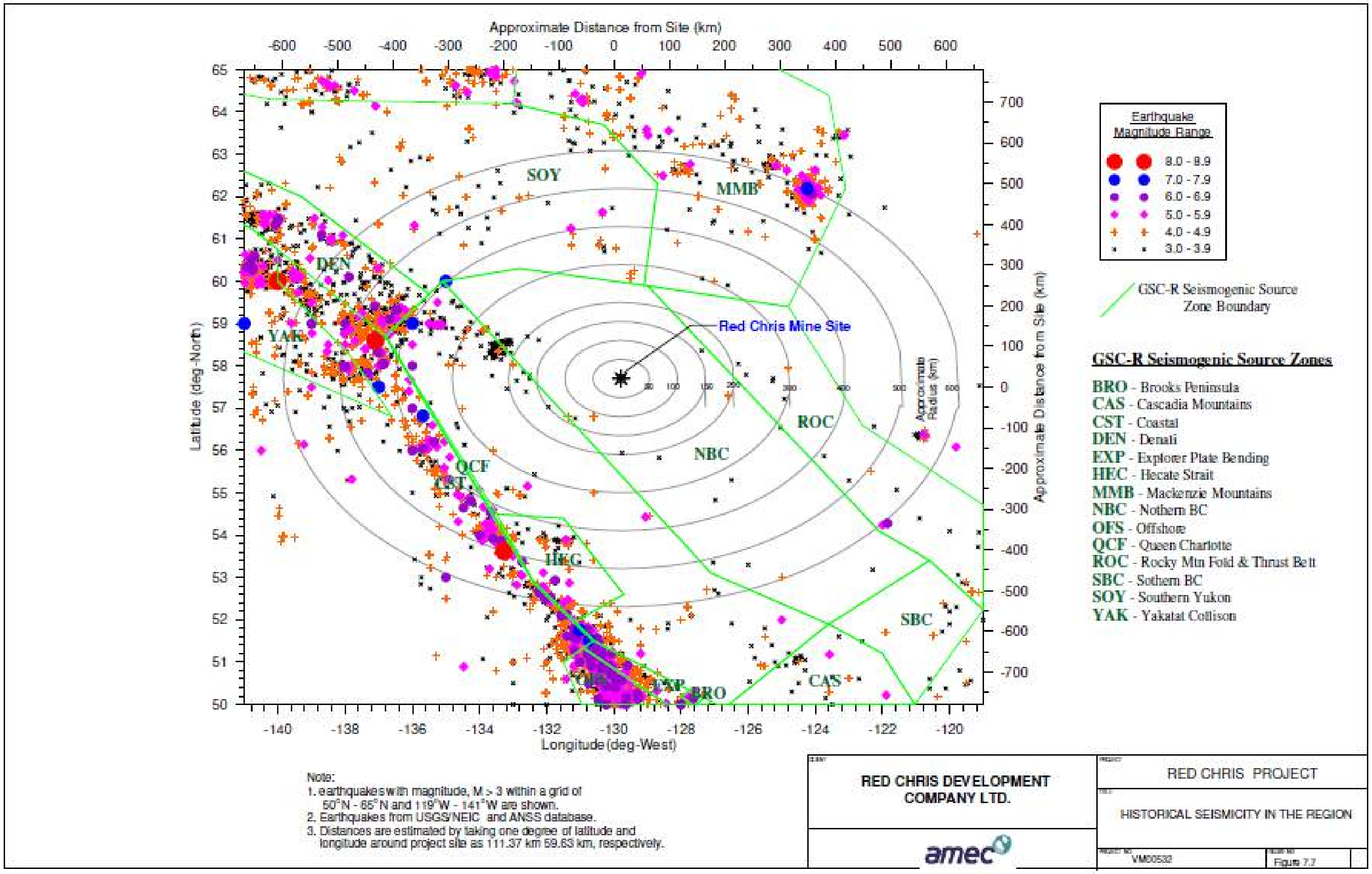
Table 19.6 CDA (2007) Consequence Classification Scheme

Dam Class	Population at Risk [note 1]	Incremental Losses		
		Loss of Life [note 2]	**Environmental and Cultural Values**	**Infrastructure and Economics**
Low	None	0	Minimal short-term loss No long-term loss.	Low economic losses; area contains limited infrastructure or services.
Significant	Temporary only	Unspecified	No significant loss or deterioration of fish or wildlife habitat. Loss of marginal habitat only. Restoration or compensation in kind highly possible.	Losses to recreational facilities, seasonal workplaces, and infrequently used transportation routes.
High	Permanent	10 or fewer	Significant loss or deterioration of *important* fish or wildlife habitat. Restoration or compensation in kind highly possible.	High economic losses affecting infrastructure, public transportation, and commercial facilities.
Very High	Permanent	100 or fewer	Significant loss or deterioration of *critical* fish or wildlife habitat. Restoration or compensation in kind possible but impractical.	Very high economic losses affecting important infrastructure or services (e.g. highway, industrial facility, storage facilities for dangerous substances).
Extreme	Permanent	More than 100	Major loss of critical fish or wildlife habitat. Restoration or compensation in kind impossible.	Extreme losses affecting critical infrastructure or services (e.g. hospital, major industrial complex, major storage facilities for dangerous substances).

Note 1. Definitions for population at risk:
None – There is no identifiable population at risk, so there is no possibility of loss of life other than through unforeseeable misadventure.
Temporary – People are only temporarily in the dam-breach inundation zone (e.g. seasonal cottage use, passing through on transportation routes, participating in recreational activities).
Permanent – The population at risk is ordinarily located in the dam-breach inundation zone (e.g. as permanent residents); three consequence classes (high, very high, extreme) are proposed to allow for more detailed estimates of potential loss of life (to assist in decision-making if the appropriate analysis is carried out).

Note 2. Implications for loss of life:
Unspecified – The appropriate level of safety required at a dam where people are temporarily at risk depends on the number of people, the exposure time, the nature of their activity, and other conditions. A higher class could be appropriate, depending on the requirements. However, the design flood requirement, for example, might not be higher if the temporary population is not likely to be present during the flood season.

Figure 19.7 Historical Seismicity in the Region



19.4.5 Freeboard for Tailings Impoundment

Freeboard (elevation difference between the operating water pond level and the crests of the dams) must accommodate the routed (or stored) inflow from the IDF, and additional freeboard above that PMF pond level to prevent overtopping by waves. The CDA (2007) guidelines recommend the following wind and wave parameters for embankment dams, which are more vulnerable to erosive failure due to overtopping than for example concrete dams:

- No overtopping by 95% of the waves caused by the most critical wind when the reservoir is at its maximum extreme level during passage of the IDF.
- Annual exceedance probability for wind frequency (generating the waves) = 1/2

These criteria will apply for each of the North, South, and Northeast Dams. The analyses and resultant freeboard requirements will be provided in the detailed design report for the tailings facility.

19.4.6 Design Flood Event for Reclaim Dam Ponds

Given the "low" consequence classification assigned for the North Reclaim Dam and the South Reclaim Dam, the CDA (2007) guidelines would prescribe a 1 in 100 annual exceedance probability for the IDF. However, it is more typical practice for the design of such facilities associated with tailings impoundments to design for to contain the runoff from the 1 in 200 year return period, 24-hour storm event, without spilling to the environment (the Environmental Design Flood – EDF), and to accommodate up to the 1 in 1,000 year, 24-hour storm inflow (i.e. the IDF) without overtopping, via emergency spillway routing. Freeboard of 0.5 m will be allowed for over the maximum routed water level during the IDF. Note that events in excess of the 1 in 200 year return period, 24-hour precipitation event would overwhelm the diversion ditches, and thus the IDF for the Reclaim Pond Dams must include both diverted and undiverted catchment.

19.4.7 Runoff Diversion Ditches

The runoff diversion ditches will be capable of routing, without overtopping, the flows corresponding to the 1 in 200 year return period, 24-hour precipitation event, with 0.3 m of freeboard.

19.4.8 Tailings Facility Stewardship

An important design criterion will be incorporation of monitoring program into the construction, operation, and closure of the tailings facility to confirm that the dams are performing in accordance with design assumptions and intent, both in terms of physical stability, and in terms of its environmental design criteria. In general terms, monitoring will include:

- Instrumentation within the dam and its foundation for performance monitoring, including:
- Piezometers at each of the three tailings dam sites, within dam fills and foundation units.
- A lesser degree of piezometer coverage in the two seepage dams.
- Means for monitoring deformations of the dams, most likely slope indicators, although given overburden depths, survey monuments may be more viable.
- Groundwater monitoring wells downstream of the seepage pumpback wells, and downstream of the Northeast Dam.
- Seepage weirs downstream of each of the three dams for quantitative seepage flows measurement.
- An Operations, Maintenance and Surveillance (OMS) Manual in accordance with CDA guidelines and guidelines published by the Mining Association of Canada (MAC). This will include programs for instrumentation and visual inspections.
- In addition to the regular inspections to be carried out by the Operators in accordance with the requirements set out on the OMS manual, the facility designer shall visit and inspect the tailings facilities no less frequently than once per year as part of preparation of annual review reports as required by the BC Ministry of Mines.
- Surveys and soundings of the tailings impoundment will be required on an annual basis for use in calibration of the mass/water balance for the tailings impoundment.
- Construction quality assurance and quality control (QA/QC) will be required for all construction, from the development of the starter facilities, annual raising of the dams, through implementation of closure earthworks, as specified by the facility designer.
- Dam safety reviews (DSR's) at frequencies commensurate with the CDA (2007) guidelines. For "very high" consequence dams, DSR's are required at 5 year intervals.

19.4.9 Tailings Impoundment Closure

The following general criteria apply for the closure of the tailings impoundment:

- The impoundment shall have a sufficient combination of flood storage and routing capacity (via a robust closure spillway) to route the 24-hour duration PMF, with sufficient freeboard, to avoid any potential for overtopping of the dams. Flood routing analyses for the closed impoundment will be undertaken to confirm that the 24-hour PMF, plus snowmelt component, is in fact the governing design event, and will be presented in the detailed design report for the tailings facility.

- An appropriate settlement allowance will be factored into the final dam heights to accommodate potential settlements induced by an MCE event. The dams shall have ample stability, and sufficiently limited deformation, under the MCE loading.
- The dams shall have a minimum factor of safety of 1.5 under static loading conditions.
- All potentially acid generating materials shall be submerged (saturated) within the tailings impoundment to prevent oxidation and development of acid rock drainage (ARD).
- Above-water tailings beaches, comprised of non-acid generating (NAG) tailings, will separate the North and South Dams from the closure water pond. The Northeast Dam will be constructed as a water retaining dam and hence will not require an above water tailings beach separating it from the water pond.
- The downstream slopes of the dams, and the exposed above-water beaches separating the North and South Dams from the closure water pond, shall be vegetated and reclaimed in such a manner that they are well-protected from erosion and a minimum of long term maintenance is required. Armouring at abutments, where runoff will tend to be concentrated, will be incorporated as required.
- Runoff diversion ditches will be breached.
- The North Reclaim Dam and the South Reclaim Dam will be breached and recontoured.

19.5 Dam Construction Materials and Borrow Areas

A key objective of the Summer 2010 test pit program was to obtain delineation and quantification of the borrow areas.

- Topsoil and other overburden judged to be unsuitable as structural fill will be stripped from the borrow pits and hauled to an acceptable stockpile for subsequent use in reclamation of the site.
- The borrow areas will be developed such that groundwater inflow and precipitation runoff are directed in a controlled manner to a designated sump area (or areas) of the site, and then removed as required. External surface water runoff shall be prevented from flowing into the borrow materials area by construction of diversion ditches or small berms as required. Sedimentation ponds will be provided.

The following zones are specified in the designs of the tailings dams and the reclaim dams:

- Zone 1: Compacted glacial till core
- Zone 2: Compacted sand and gravel (starter dam upstream shell)
- Zone 3A: Compacted 3” minus sand and gravel (chimney filter for starter dam)
- Zone 3B: Compacted sand and gravel (starter dam downstream shell)
- Zone 3C: Compacted sand and Gravel (underdrainage blanket for starter dam and final dam)
- Zone 4: Compacted cycloned sand (downstream shell for all subsequent dam raising, and filter protection for the Zone 1 till core).
- Zone 5: Fine riprap transition
- Zone 6: Coarse riprap

The material types function and specifications for Zones 1 through 4 are provided in Table 19.7, and are further described below. Gradation specifications are given in Figure 19.8 for those zones to be constructed of sand and gravel (Zones 2, 3A, 3B and 3C). Gradation specifications for the Zone 1 till core are given on Figure 19.9. Cycloned sand is to have maximum fines content (% by dry weight finer than 0.074 mm) of 10%.

As shown on the figures, the specified envelopes were designed to maximize use of available materials and to minimize processing requirements.

Gradation specifications for Zones 5 and 6 (required for the Northeast Dam only) will be developed upon completion of wind and wave analyses for that dam, which will determine the riprap sizing required, and the gradation of any transition zone(s).

19.5.1 Zone 1 compacted Till

The basal till borrow materials approved for construction are to be well graded, organic-free mineral soils, having moisture contents near their optimum for compaction with a minimum of 25% by weight passing the No. 200 sieve. The optimum moisture content range of the borrow soils is to be determined by Standard Proctor moisture-density relationship testing. A general guideline for allowable moisture contents for the Zone 1 structural fill is 1% wet and 2% dry of optimum moisture content as determined by the Standard Proctor test. However, typical experience is that if haul trucks can traffic on the till surface without excessive rutting and trafficability issues, then provided the density specification is achieved, moisture content specifications can be relaxed.

The approved Zone 1 structural fill is to be spread in lift thicknesses not exceeding 500 mm. The Zone 1 till fill shall be spread with a dozer and then compacted by a heavy sheepsfoot roller, or other compaction equipment capable of working (kneading) the soil during compaction. The indentations or pad marks left by the heavy sheepsfoot roller in the compacted Zone 1 fill will form the scarified surface upon which subsequent lifts of borrow soil will be placed and compacted.

The Zone 1 fill is to be compacted to a minimum of 98% of the Standard Proctor maximum dry density.

19.5.2 Zone 2 Upstream Compacted Sand and Gravel

The Zone 2 upstream sand and gravel is to be pit run, organic-free mineral soil with a maximum of 15% by weight passing the No. 200 sieve. Zone 2 material will be placed in maximum lift thicknesses of 500 mm, and will be compacted by a minimum 10 tonne vibratory roller.

19.5.3 Zone 3A Sand and Gravel Filter

The Zone 3A sand and gravel filter material is to be a clean
75 mm (3 in) minus organic-free mineral soil. This material appears readily available and

unlikely to require processing to fall within the specified gradation envelope. Zone 3A material will have a maximum of 5% by weight passing the No. 200 sieve.

The Zone 3A material will be placed in maximum lift thicknesses of 300 mm, watered as necessary to achieve compaction and compacted by vibratory roller. The Zone 3A fill is to be compacted to a minimum of 98% of the Standard Proctor maximum dry density.

19.5.4 Zone 3B Downstream Compacted Sand and Gravel

The Zone 3B sand and gravel material is to be pit run, organic-free mineral soil with a maximum of 10% by weight passing the No. 200 sieve. Zone 3B material will be placed in maximum lift thicknesses of 500 mm, and will be compacted by a minimum 10 tonne vibratory roller.

19.5.5 Zone 3C Sand and Gravel Drainage Blanket

The Zone 3C sand and gravel material is to be pit run, organic-free mineral soil with a maximum of 5% by weight passing the No. 200 sieve. Processing of native sand and gravels is not expected to be required to produce this material. Zone 3C material will be placed in maximum loose lift thicknesses of 500 m, and will be compacted by vibratory roller.

Table 19.7 Summary of Starter Facility Embankment Construction Materials

Zone	Description	Zone Function	Material	Borrow Source(s) Presently Identified	Placement Specifications	Compaction Specifications
1	Compacted Till	Minimize rate of seepage through the dam.	Glacial till.	Borrows 1a,1b	Placed in maximum loose lift thicknesses of 500 mm with allowable moisture contents of 1% wet and 2% dry of optimum moisture content Scarification required to ensure bonding between successive lifts.	Minimum 98% of maximum standard Proctor density, compaction with pneumatic or sheepsfoot equipment.
2	Upstream Compacted Sand and Gravel	1. Provide upstream structural support for the Zone 1 till core.	Pit run sand and gravel with less than 15% fines by weight (material passing the No. 200 sieve). .	Borrows 2a through 2f	Placed in maximum 500 mm loose lifts, watered as necessary to achieve compaction, and compacted with a minimum 10 tonne vibratory roller.	Minimum 98% of maximum standard Proctor density, compacted with vibratory smooth drum roller.
3A	Sand and Gravel Filter	Provide critical downstream filter for Zone 1 till core, and provide for effective drainage of seepage through the core.	Clean, 75mm minus sand and gravel with less than 5% fines content (% by dry weight passing No. 200 sieve).		Placed in maximum 300 mm loose lifts, watered as necessary to achieve compaction, and compacted with a minimum 10 tonne vibratory roller.	Compaction to minimum of 98% standard Proctor density.
3B	Downstream Compacted Sand and Gravel	Provide downstream structural support for the dam.	Pit run sand and gravel with less than 10% fines content. .		Placed in maximum 500 mm loose lifts, watered as necessary to achieve compaction, and compacted with a minimum 10 tonne vibratory roller.	Compaction to minimum of 98% standard Proctor density.
3C	Sand and Gravel Drainage Blanket / Finger Drains	Provides drainage for the filter	Pt run sand and gravel with less than 5% fines by weight (material passing the No. 200 sieve). .		Maximum loose lift thickness 500 mm and compacted with a minimum 10 tonne vibratory roller.	Compaction to minimum of 98% standard Proctor density.
4	Compacted Cycloned Sand	Downstream shell of North Dam, and bulk fill for South Dam	Cycloned, NAG tailings, maximum fines content of 10%.	Cyclone Plant	Placed hydraulically, and spread and compacted via dozers.	Compaction to minimum of 98% standard Proctor density.
5	Coarse Rip Rap	Prevents erosion of ditch surface, upstream face of Northeast Dam		Processed from non-acid generating rock from open pit		N/A
6	Fine Rip Rap			Processed from non-acid generating rock from open pit		N/A

Figure 19.8 Sand and Gravel Borrow Material Gradations and Specifications

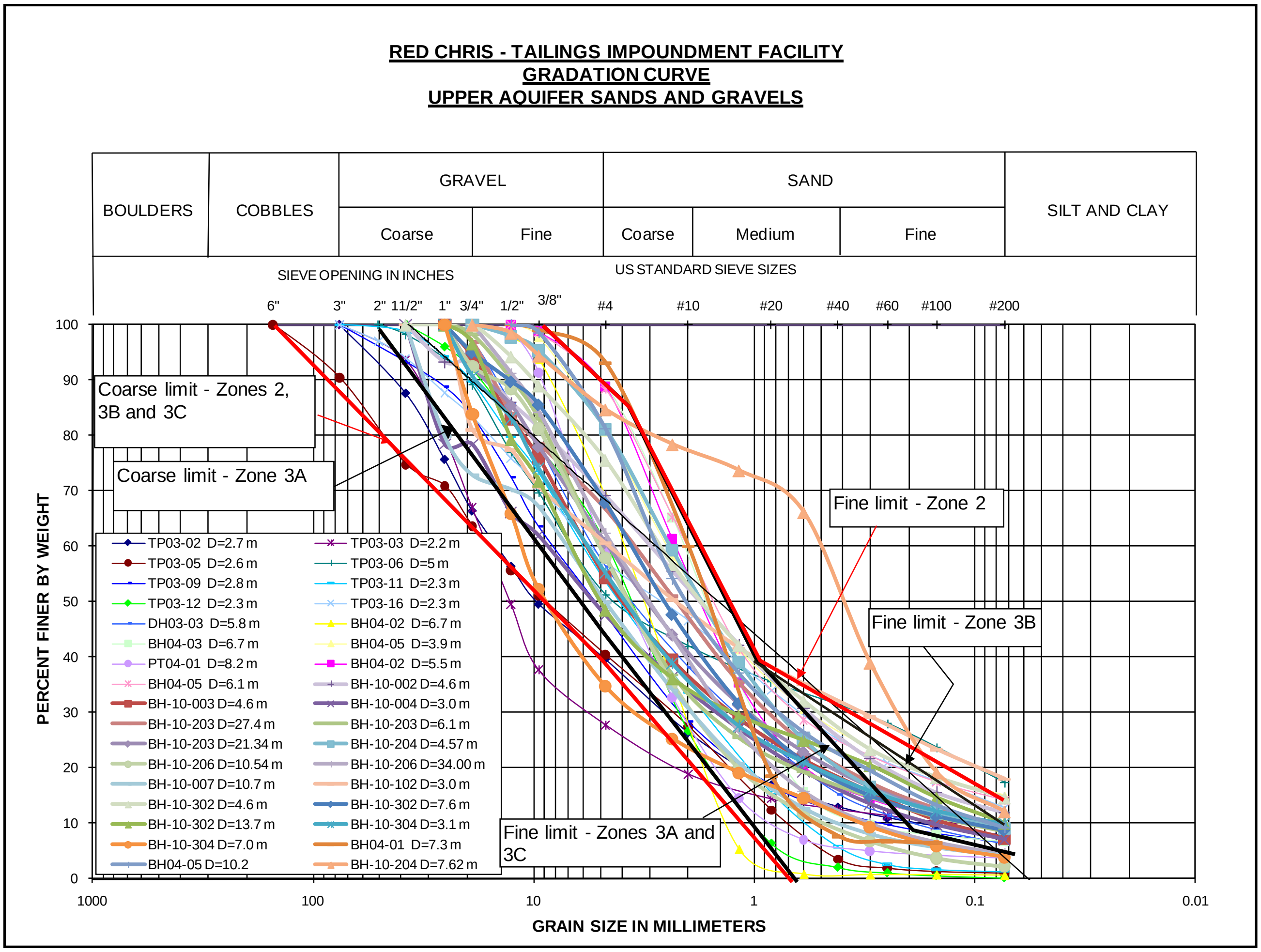
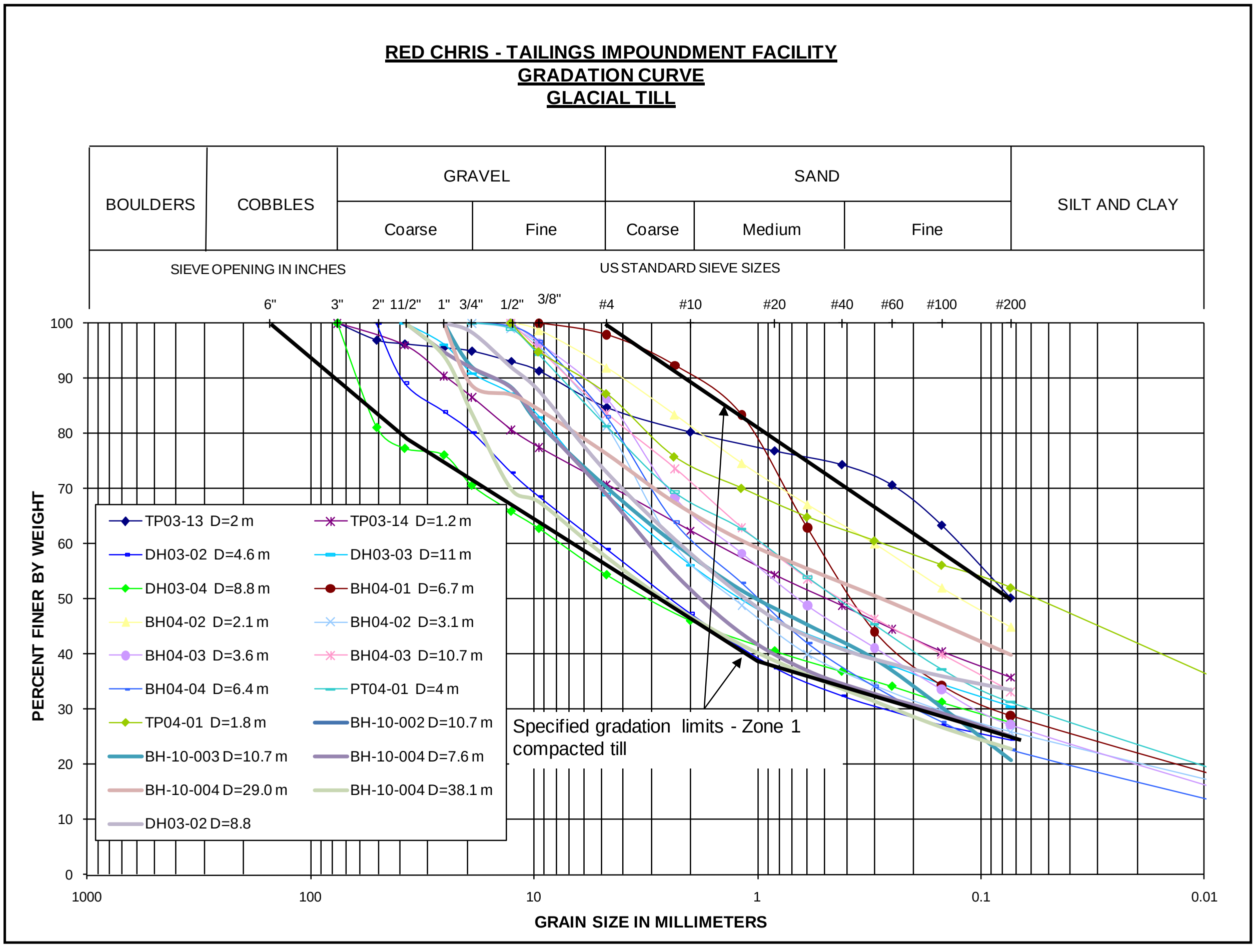


Figure 19.9 Till Borrow Material Gradations and Specifications



19.6 Design of Tailings Dams

19.6.1 North Starter Dam

The design of the North Dam, including the starter dam portion, is shown on Figure 19.10. The starter dam will incorporate a compacted till upstream facing to reduce seepage through the dam, with a downstream filter zone and shell comprised of sand and gravel. Given the discontinuous nature of the upper till unit, rather than a cut-off trench, the low hydraulic conductivity core will be extended upstream, within approximately 50% of the limits of the start-up water pond (with a volume of about 5Mm^3) via a geomembrane liner. The objective of the liner is to reduce seepage losses from the start-up water pond and thus reduce reliance on the seepage pumpback wells for process water supply during the initial months of operation. Once the tailings deposit upstream of the starter dam develops and begins to consolidate, this will serve to limit seepage, and the liner will eventually become redundant. Seepage modelling indicates that seepage from the start-up pond would be limited to about 20 litres/sec, a rate readily manageable by the planned pumpback wells system, with only the downstream 50% of the start-up water pond being lined. The analyses were based on conservative assumptions in terms of the hydraulic conductivity of the liner.

The design section for the starter dam includes an upstream slope buttress of Zone 2 (sand and gravel, allowable fines content 15%), the objective of which is to:

- Provide a platform for the initial tailings pipeline
- Provide protection for the liner where it is keyed into the Zone 1 till core

Additional analyses will be required to determine stresses on the liner, and the potential need for textured liner, which may require modification (possibly slope flattening) of the Zone 2 buttress. Note that additional analyses will also be carried out, and included within the detailed design report, to confirm the detail of the liner key-in with the till core.

19.6.2 North Dam

Above its starter configuration, the North Dam will be raised via centerline construction as shown on Figure 19.10. The till core will be raised in annual stages, with upstream support provided by cyclone sand prior to beach raising. The downstream cyclone sand shell will be constructed hydraulically, with dozer compaction of the sand per operations at Highland Valley Copper, Kemess, and other such examples. The Zone 4 cycloned sand shell will be underlain by a blanket/finger drain comprising Zone 3C sand and gravel, with a maximum fines content of 5% to facilitate good under-drainage. The inclusion of coarse rock drains (separated from the cyclone sand by a suitable filter sequence) will be considered as detailed design proceeds. The installation of liners below such collector drains will be considered as a means of maximizing capture of under-drainage and routing it to the North Reclaim Dam pond, rather than allowing infiltration which would then be managed by the north seepage pumpback wells.

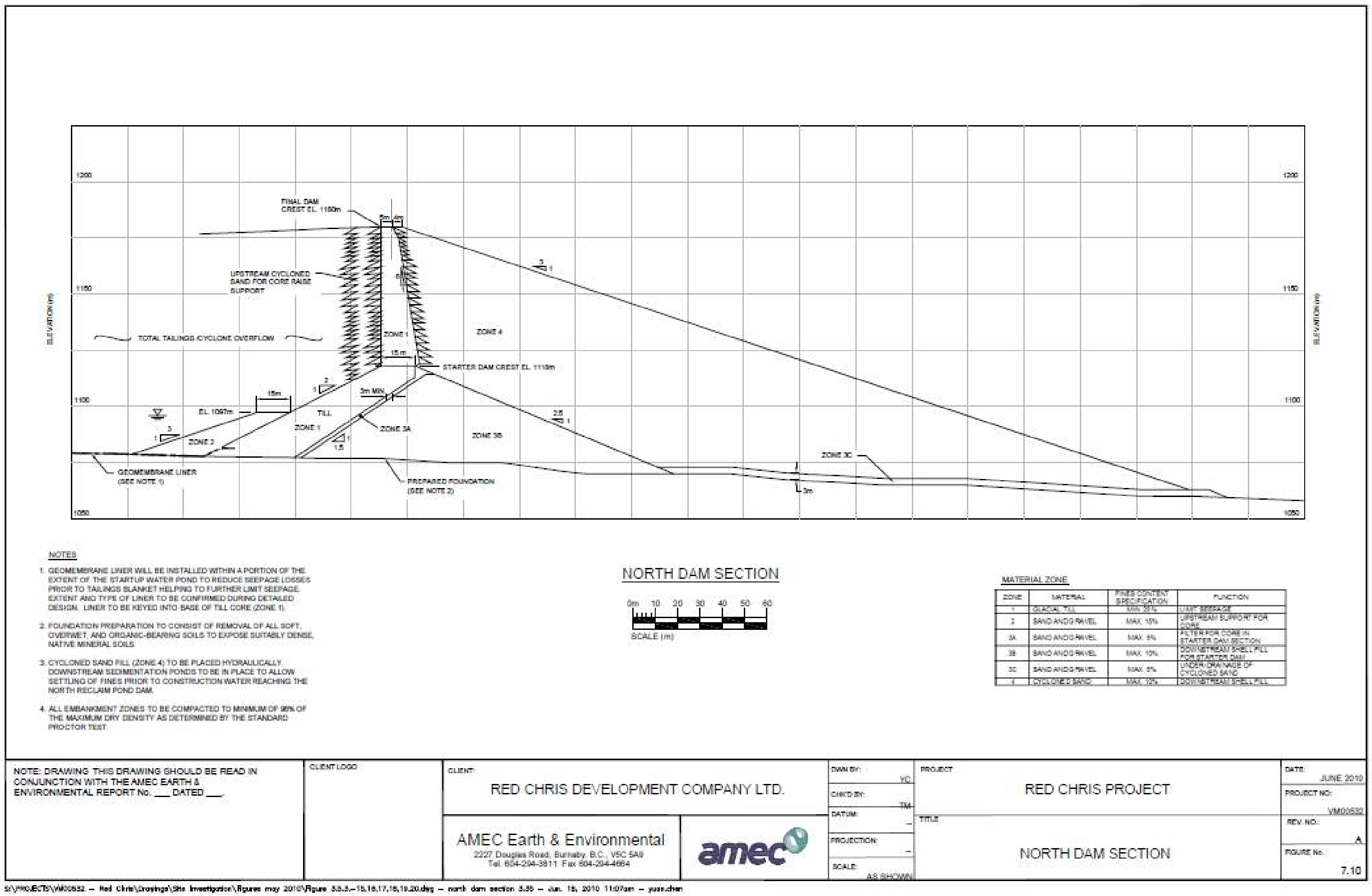
The Zone 4 cyclone sand fill will provide a good downstream filter for the Zone 1 till core. The maximum fines content for Zone 4 cyclone sand will be 10%. A fines content of 15% may be designated above a certain elevation, and/or within a certain offset from the Zone 1 till core, to maximize use of sand and given that it is in the lower and more downstream portion of the cyclone sand shell where a high hydraulic conductivity is most important.

All embankment zones will be compacted to a minimum of 98% of the maximum dry density as determined from the Standard Proctor Test. Experience indicates however that, for dozer compacted, hydraulically-placed cyclone sand, field densities typically exceed 100% of the Standard Proctor maximum dry density.

The cyclone sand downstream shell of the North Dam will require appropriate erosion protection, both during operations and post-closure. During operations, experience at other sites (e.g. Kemess) has indicated that the abutment contacts are particularly prone to erosion due to concentration of runoff there. Temporary diversion ditches will be considered to mitigate this. Placement of the sand and gravel drainage blanket between the abutment and the sand shell will encourage infiltration rather the concentration of runoff that can erode the sand. At closure, a filtered armour sequence will be provided for along the abutment contact, essentially a diversion ditch, along both abutments. The sand shell itself will be covered and reclaimed. Experience at

Kemess with covering of the cyclone sand shell (at a 2H:1V slope rather than the 3H:1V slope proposed for Red Chris) indicates that providing for a rough surface in the reclamation medium represents an effective means of reducing erosion prior and subsequent to vegetation becoming established.

Figure 19.10 North Dam Section



19.6.3 Saddle Dam

The design of the Saddle Dam is illustrated on Figure 19.11. The Saddle Dam is a temporary structure, and will eventually be buried within the tailings deposit. Its function is to allow for discharge of tailings for the first two years' of mill production into the Northern Valley, deferring the development of the southern portion of the impoundment, including the South Dam, South Reclaim Dam, and south seepage pumpback wells, into the first and second year of mine operation. This approach also permits the deferral of construction of some of the runoff diversion ditches into the first few years of mine operation and, via limitation of the overall catchment of the tailings impoundment, reduces the net annual water balance surplus that necessitates discharge.

The Saddle Dam includes a Zone 1 till core section, with upstream Zone 2 sand and gravel (for erosion protection), and a downstream shell/filter zone and drainage blanket (Zones 3B and 3C, respectively). Due to the discontinuity and depth of the upper till unit, a till blanket will be extended 15 m upstream of the upstream toe of the Saddle Dam, and will tie into the Zone 1 till core. This will lengthen the seepage path below the dam and thus reduce seepage. Analyses being undertaken in support of detailed design will determine estimated seepage rates (the Saddle Dam will be a water-retaining structure), and if downstream seepage pumpback wells will be required until the south portion of the tailings impoundment is commissioned, at which time the Saddle Dam becomes redundant. Additional analyses will also determine if the Saddle Dam can be raised to a higher elevation to defer development of the southern impoundment by one additional year, and/or forestall the need for discharge of surplus water for as long as possible.

All embankment zones will be compacted to a minimum of 98% of the maximum dry density as determined from the Standard Proctor Test.

19.6.4 South Dam Starter Dam

The design of the South Dam, including its starter dam, is illustrated on Figure 19.12. The starter dam will be a Zone 3B sand and gravel embankment, providing a platform for upstream tailings discharge, and for downstream shell construction via hydraulic sandfill placement (Zone 4).

Figure 19.11 Saddle Dam Section

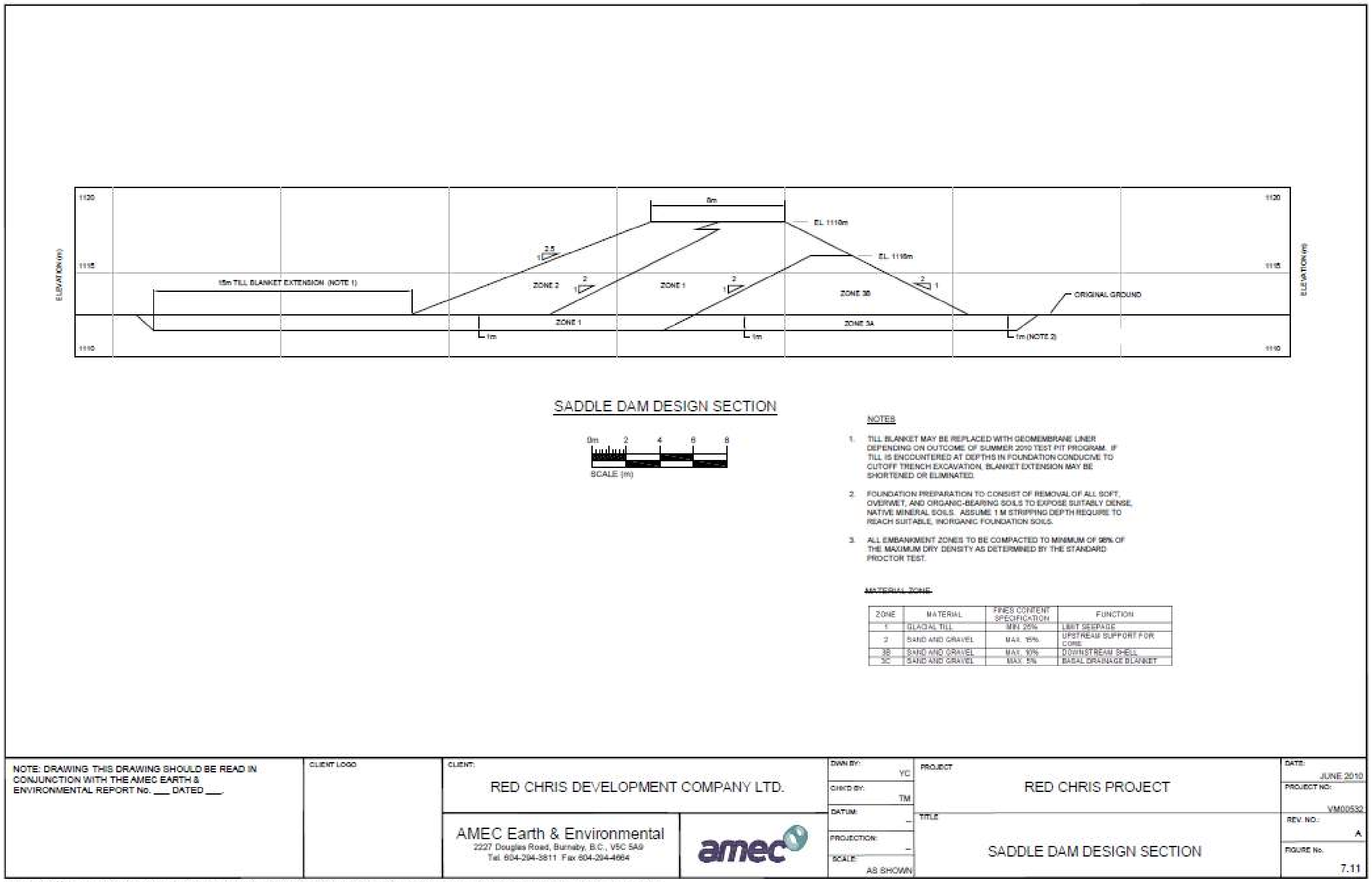
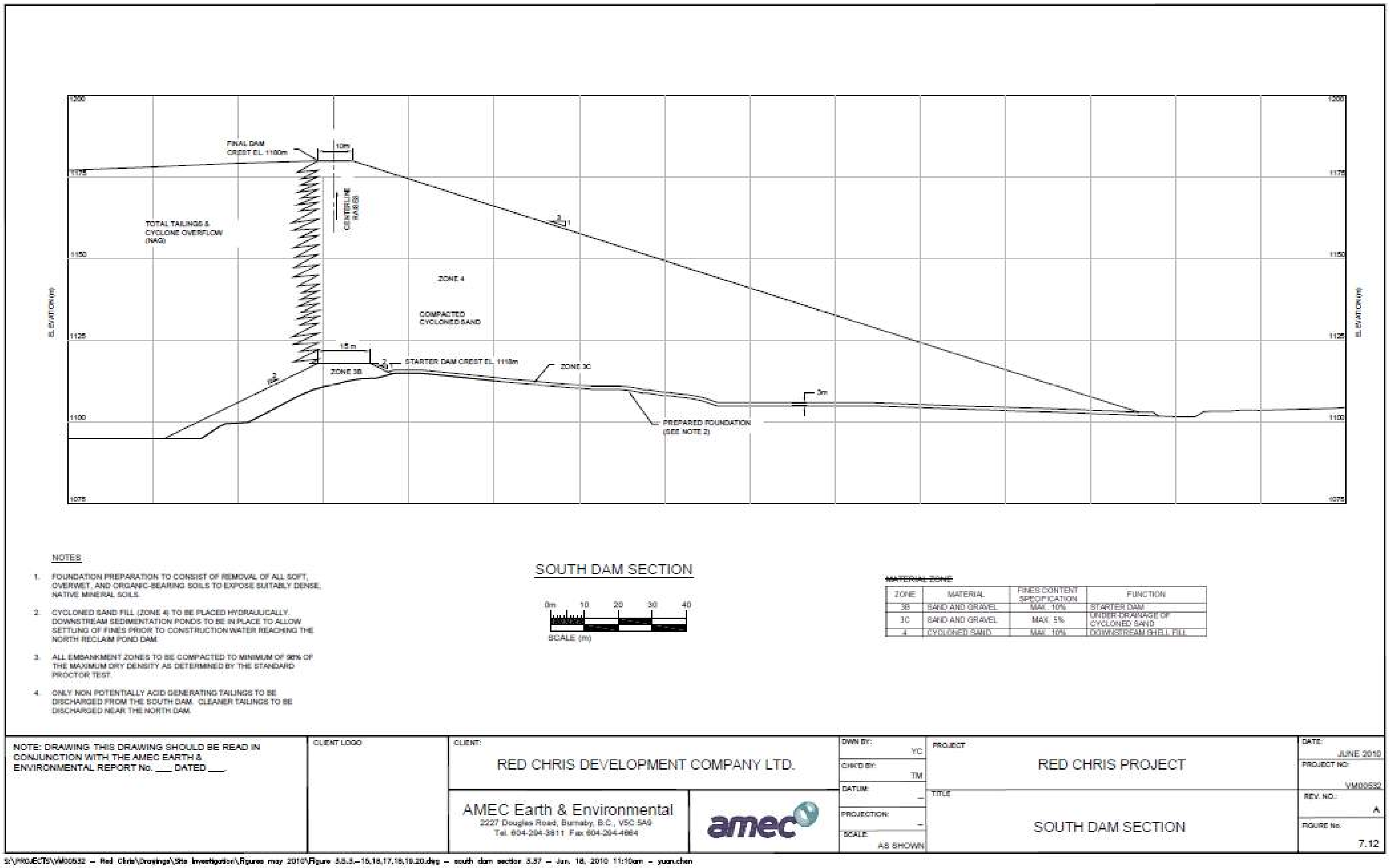


Figure 19.12 South Dam Section



19.6.5 South Dam

The South Dam will be a centerline-raised, compacted cyclone sand dam. A till core is not judged necessary for incorporation into the design of this dam for the following reasons:

- The South Dam starter embankment is not required to retain process water. The process water reclaim pond will be located well to the upstream of the dam.
- For a significant portion of the South Dam alignment, including the west abutment area, the upper till unit is absent. As such, a till core cutoff trench to key into a native unit of low hydraulic conductivity is not viable.
- Groundwater modeling, using a three-dimensional MODFLOW model, indicates:
- For the closure configuration, the predicted rate of seepage loss from the tailings impoundment is essentially no different for the case with a till core than is the case without. It is the wide tailings beach, the head loss from downward seepage through the low hydraulic conductivity tailings, and the valley bottom aquifers that govern seepage through and below the dam, rendering a till core redundant.
- The predicted rate of seepage loss (up to about 50 litres/sec) from the tailings impoundment in its closure configuration, including recharge into the underlying aquifers, and seepage through the South Dam, is immaterial in terms of the closure water balance, thus there is no concern associated with maintaining a closure water pond (even in dry year scenarios), nor in maintaining saturation/flooding of PAG cleaner tailings, providing these are discharged into the northern half of the tailings impoundment.
- A two-dimensional SEEP/W model of the South Dam and pond configuration at closure, incorporating boundary conditions derived from the MODFLOW modeling, indicates a zone of unsaturated tailings only to a distance of about 300 m upstream of the South Dam. Provided that deposition of PAG tailings within this area is prevented, readily achievable as these tailings will be in the cleaner tailings pipeline only, this is of no consequence in terms of water quality at closure.

All embankment zones will be compacted to a minimum of 98% of the maximum dry density as determined from the Standard Proctor Test.

The maximum fines content for Zone 4 cyclone sand will be 10%. A fines content of 15% may be designated above a certain elevation, to maximize use of sand and given that it is in the lower portion of the cyclone sand shell where a high hydraulic conductivity is most important.

For the South Dam, it is the upstream tailings (total tailings and cyclone plant overflow) that will serve as the seepage reduction measure.

Under-drainage for Zone 4 cyclone sand will be provided by a drainage blanket of Zone 3C sand and gravel (maximum 5% fines). It is likely that, as detailed design proceeds, this drainage blanket will be supplemented by rock finger drains (separated from the cyclone sands fill by a suitable filter sequence) to further enhance under-drainage, maintain a low phreatic surface, and maximize efficiency of collection of under-drainage water from hydraulic fill operations and routing of that water to the South Reclaim Dam pond for pumpback to the tailings impoundment.

The installation of liners below such collector drains will be considered as a means of maximizing capture of under-drainage and its routing to the South Reclaim Dam pond, reducing the extent to which such drainage infiltrates the groundwater system and reports to the pumpback wells downstream of the South Reclaim Dam.

The development of the South Dam as a pervious, cyclone sand structure imposes constraints on the deposition of tailings within the impoundment. Specifically, the sulphide-bearing cleaner tailings stream (which includes the cleaner tailings and the sulphides rejected from the additional flotation of the rougher tailings), must be discharged into the impoundment such that they will be permanently submerged (below the water pond) or below the phreatic surface (where underlying subaerial beaches rather than the water pond).

As is the case for the downstream shell of the North Dam, the cyclone sand downstream shell of the South Dam will require erosion protection measures during operations and post-closure. The measures described above for the North Dam will be adopted at the South Dam.

19.6.6 Northeast Dam

The Northeast Dam design section is shown on Figure 19.13. This will be a water retaining structure, with no tailings against the upstream face. The dam will include a Zone 1 till core, keyed into the upper till unit that, on the basis of the winter 2010 drilling program, appears continuous throughout the area. Upstream of the Zone 1 till core will be Zone 2 sand and gravel, which will also provide support for the riprap zones (5 and 6) that will be required to provide for long term erosion protection. The sizing of these zones will be driven by wind and wave analyses that have yet to be undertaken but will be included as part of the ongoing detailed design effort. Zone 5 will be the coarse riprap, while Zone 6 will be the fine riprap transition likely to be required to provide an acceptable filter sequence between Zones 2 and 5.

Downstream of the Zone 1 till core will be a filter zone of Zone 3A sand and gravel, and a downstream shell of Zone 3B sand and gravel, which will be underlain by a Zone 3C drainage blanket to maintain a low phreatic surface through the dam section.

All embankment zones will be compacted to a minimum of 98% of the maximum dry density as determined from the Standard Proctor Test.

19.6.7 Design of Reclaim Dams

The design of the North and South Reclaim Dams is illustrated in section on Figure 19.14. The dams will comprise a till core, with upstream shells of sand and gravel, a downstream filter against the core, and a drainage blanket. The till core will be extended to key into the upper till unit (if present and at reachable depth), or else extended upstream via a geomembrane liner to lengthen the seepage path, and to reduce seepage from the reclaim ponds that must be dealt with by the seepage pumpback wells. Alternatively, a compacted till blanket might be used to lengthen the seepage path if a continuous cut-off to the upper till unit is not viable. The Summer 2010 test pit program will determine which design approach is adopted for each of the two dams.

All embankment zones will be compacted to a minimum of 98% of the maximum dry density as determined from the Standard Proctor Test.

The crest elevations of the Reclaim Dams will be finalized in detailed design, on the basis of storage required to contain the runoff from the 1 in 200 year return period, 24-hour storm event, without spilling to the environment (the Environmental Design Flood – EDF), and to accommodate up to the 1 in 1,000 year, 24-hour storm inflow (i.e. the IDF) without overtopping, via emergency spillway routing. Pumping capacity and backup requirements for operational contingencies (e.g. power failure for 24 hours, etc.) will also factor into required storage capacity and hence dam crest elevations.

Figure 19.13 Northeast Dam Section

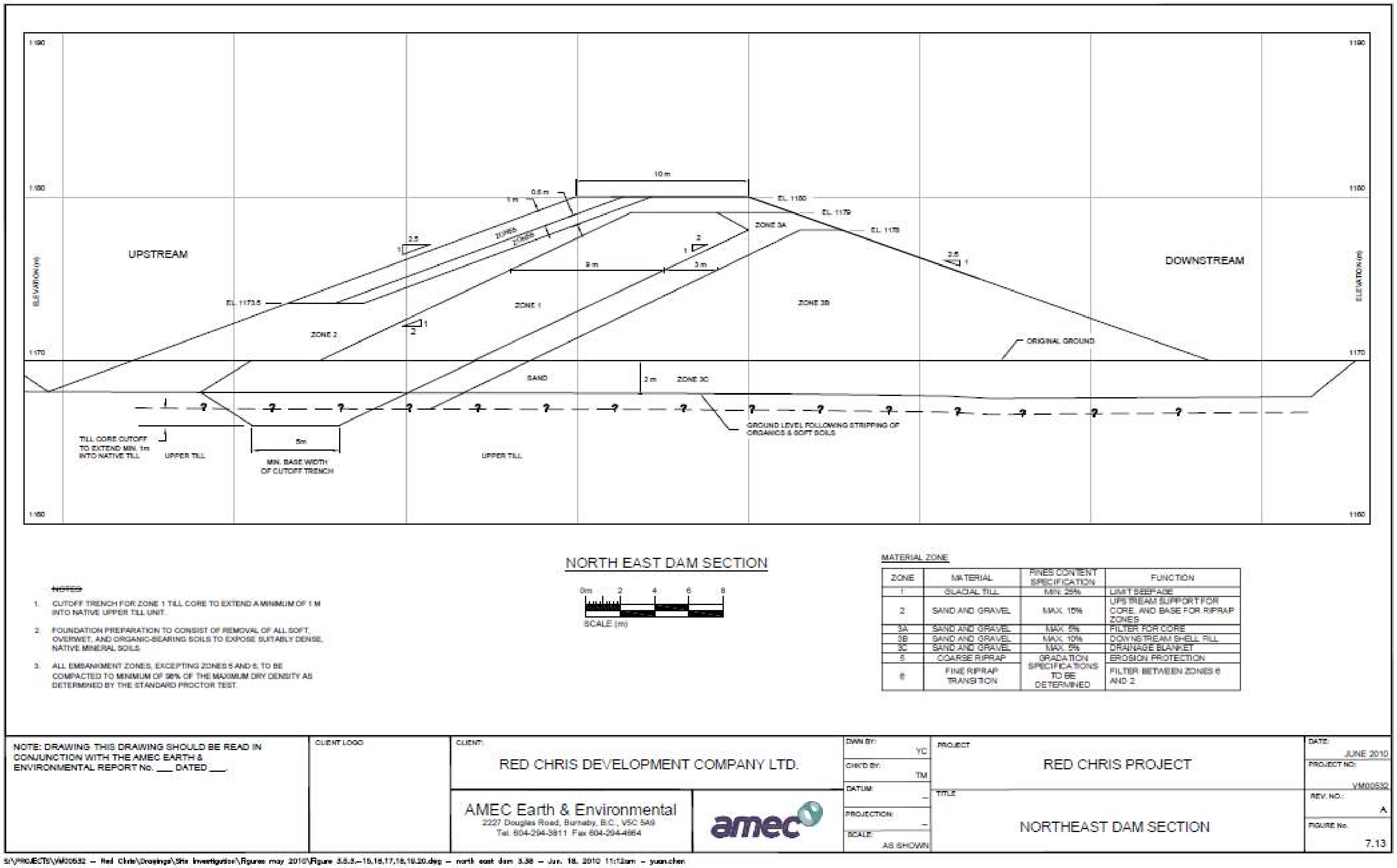
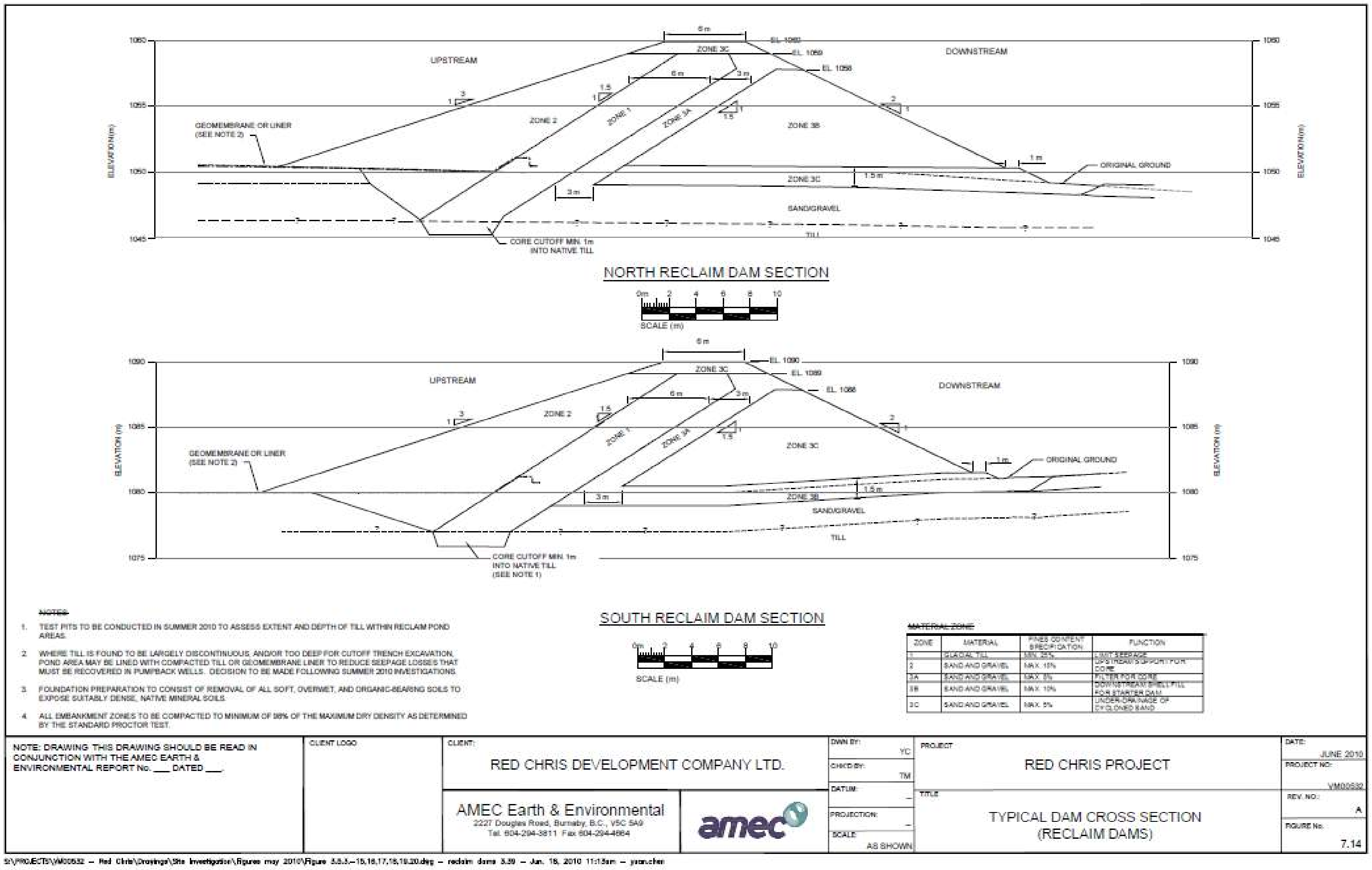


Figure 19.14 Reclaim Dams Sections



19.6.8 Runoff Diversion Ditches

Runoff diversion ditches will be constructed around the tailings impoundment to reduce runoff inflows, and thereby reduce the predicted net annual water balance surplus that will require discharge of surplus water from the impoundment.

As stated previously, the runoff diversion ditches will be designed to accommodate the runoff from a 1 in 200 year return period, 24-hour storm event, with 0.3 m freeboard. The conceptual design of the runoff diversion ditches is illustrated in Figure 19.15. Ditch alignments, grades, and erosion protection requirements will be finalized as detailed design proceeds.

19.6.9 Seepage Pumpback Well System

Until significant thicknesses of tailings are deposited upstream of the North and South Dams, significant seepage is anticipated within the hydraulically-conductive upper and lower aquifer units, the upper till unit being discontinuous. At the North Dam, the rate of seepage will be significantly reduced by the liner that will be placed within a portion of the limits of the start-up water pond. At the South Dam, seepage will be high initially, although infiltration is likely to be dominated by drainage water from the hydraulically-placed cyclone sand. Accordingly, a field of seepage pumpback wells will be installed downstream of both the North Reclaim Dam and the South Reclaim Dam. The wells will be installed within the lower aquifer to a depth of about 50 m, although MODFLOW analyses are ongoing for optimization of the design (number of wells, and depths) of the pumpback well fields. Based on MODFLOW (three-dimensional) and SEEP/W (two-dimensional) analyses completed to date, it appears that a conservative approach would be to design the well fields to each be capable of pumping up to 50 litres/sec.

It is not expected that pumping will be required from all of the wells at any given time. It is likely that the number of wells being pumped will decline as the impoundment develops and the tailings deposit gradually reduces seepage into the underlying aquifers. Further, monitored water quality is expected to be such that pumping from the wells is deemed not to be required, in which case the seepage would not be pumped back into the tailings impoundment.

19.6.10 Operational Spillways

The dams will be raised on an annual basis, and given the lack of bedrock at lower elevations along the dam alignments and abutments, it is expected that, for most of the operating life of the tailings facility, no emergency overflow spillway will be provided, and the impoundment will be sized to store the most critical of:

- 24 hour duration PMF, with failure of diversion ditches assumed at the onset of the event.
- 30-day PMF, with the diversion ditches remaining operational.

There will be certain stages of the development of the tailings impoundment when operational spillways can be provided. Such a spillway will be provided for the initial impoundment in the northern valley, with the spillway discharging to the south, around the Saddle Dam. An emergency spillway can also be provided once the water pond rises approximately to existing ground elevation at the alignment of the Northeast Dam, which is not required to be constructed until the latter years of the mine life.

The system for discharge of surplus water from the impoundment due to the net annual water balance surplus will also be advantageous in terms of managing extreme inflow events. As detailed design proceeds, contingencies for the use of this system for handling extreme inflow events will be developed and may govern the pumping and pipeline capacity installed.

19.6.11 Tailings Impoundment Closure Spillway

The closure spillway will be on the west abutment of the Northeast Dam. As indicated previously, the spillway will be capable of routing the critical duration PMF event, assumed at present to be a 24-hour event with snowmelt component, although hydraulic modelling will be carried out to confirm this is the critical design event.

19.6.12 Cycloned Sand Production and Distribution

Use of cycloned sand produced from the Red Chris tailings for raising of the tailings dams is considered economically attractive for the project. A number of tailings dams in British Columbia are constructed using clean sand produced by cycloning of the tailings stream. One example of this practice is the L-L Dam at the Highland Valley Copper Mine. This dam started as a conventional earthfill dam, with a central till core and upstream/downstream shells of compacted sand and gravel (i.e. the same basic design proposed for the Red Chris starter dams), and later converted to a cycloned sand dam, raised via the centerline method (as proposed for Red Chris), with a central till core, downstream shell of compacted cycloned sand (hydraulically placed and compacted via dozers), and upstream support provided by the tailings beach. The raising of the Kemess Mine tailings dam, beginning in 2002, incorporated the use of compacted cycloned sand for construction of the upstream and downstream shells (while retaining the central till core). To achieve this, sulphides flotation is carried out to produce a non-acid generating sand suitable for use in dam construction, and the same approach will be taken for Red Chris.

For the Red Chris project, additional flotation cells will be added at the end of the scavenger circuit to remove pyrite from the rougher tailings. The removed pyrite will be added to the cleaner tailings stream, which will be PAG. The de-pyritized rougher tailings will then be subjected to two stages of cycloning to produce a clean fine sand that is NAG and has about 10% finer than 0.074 mm.

Pilot plant testwork was carried out at G&T Metallurgical in Kamloops, BC in May 2004 on Red Chris ore samples (from East zone, Main zone, and combined for the two ore types). This program yielded samples of rougher tailings, cleaner tailings, and de-pyritized rougher tailings following the additional flotation. ABA testing was carried out on the de-pyritized rougher tailings samples, and demonstrated that a NAG rougher tailings with high neutralization potential ratio (NPR) values could be produced. The NAG rougher tailings will be used for the production of cyclone sand and for the formation of the above-water beaches upstream of the North and

South Dams for closure. Further, only rougher total tailings, and cyclone plant overflow, will be discharged from the South Dam. Cleaner tailings will be well to the north.

Figure 19.16 shows the depyritized rougher tailings gradation, and the predicted overflow and underflow gradations of the 1st stage cycloning of this material. Figure 19.17 shows the underflow/overflow gradations predicted for the 2nd stage cycloning, while Figure 19.18 shows a summary of the rougher tailings gradation, the combined first and second stage overflow, and the secondary cyclone underflow (the product used for downstream shell construction).

Figure 19.15 Runoff Diversion Ditches: Conceptual Design Section

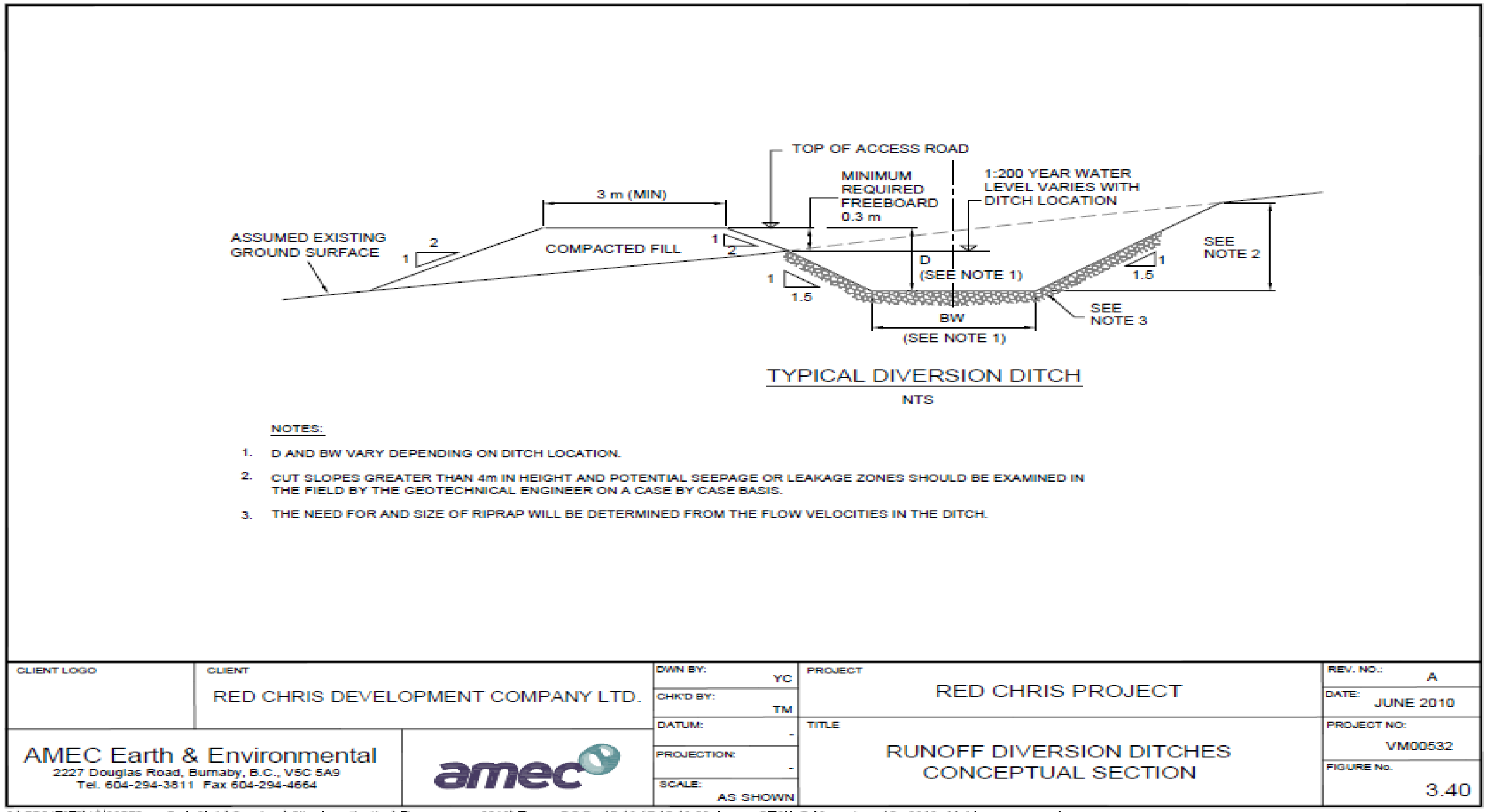


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Figure 19.16 Depyritized Rougher Tailings and 1st Stage Cyclone Sand Gradations

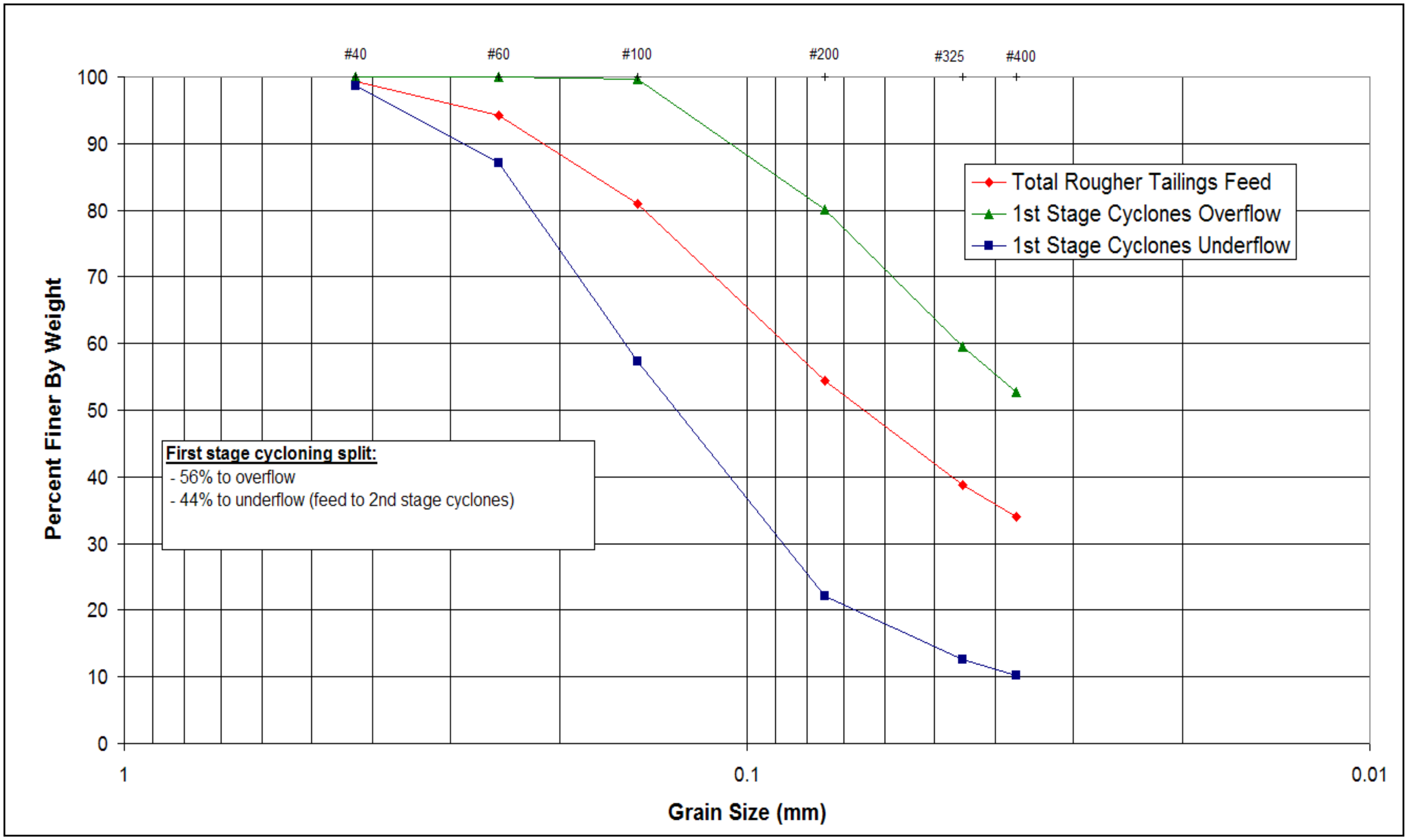


Figure 19.17 2nd Stage Cyclone Gradations

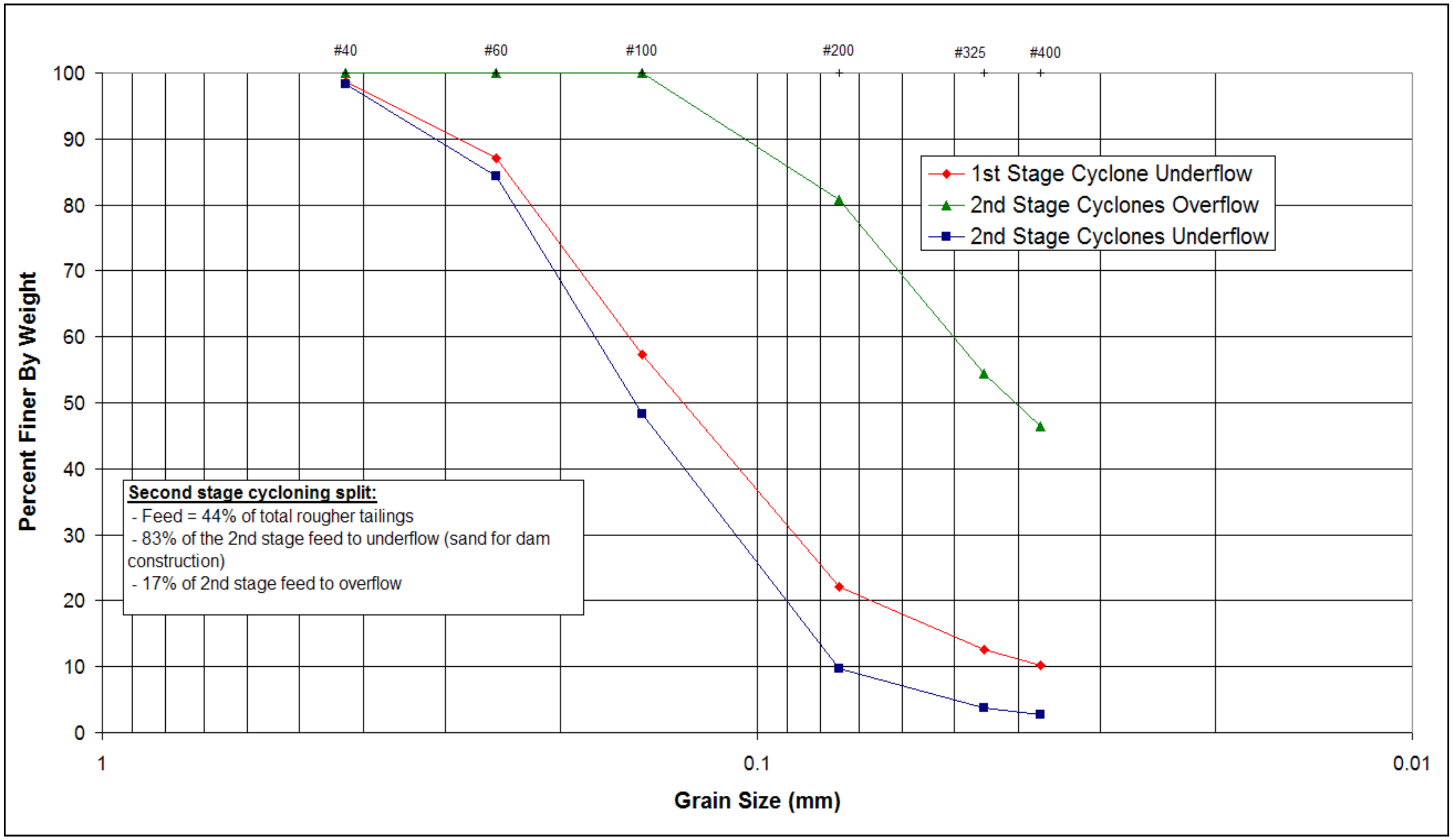
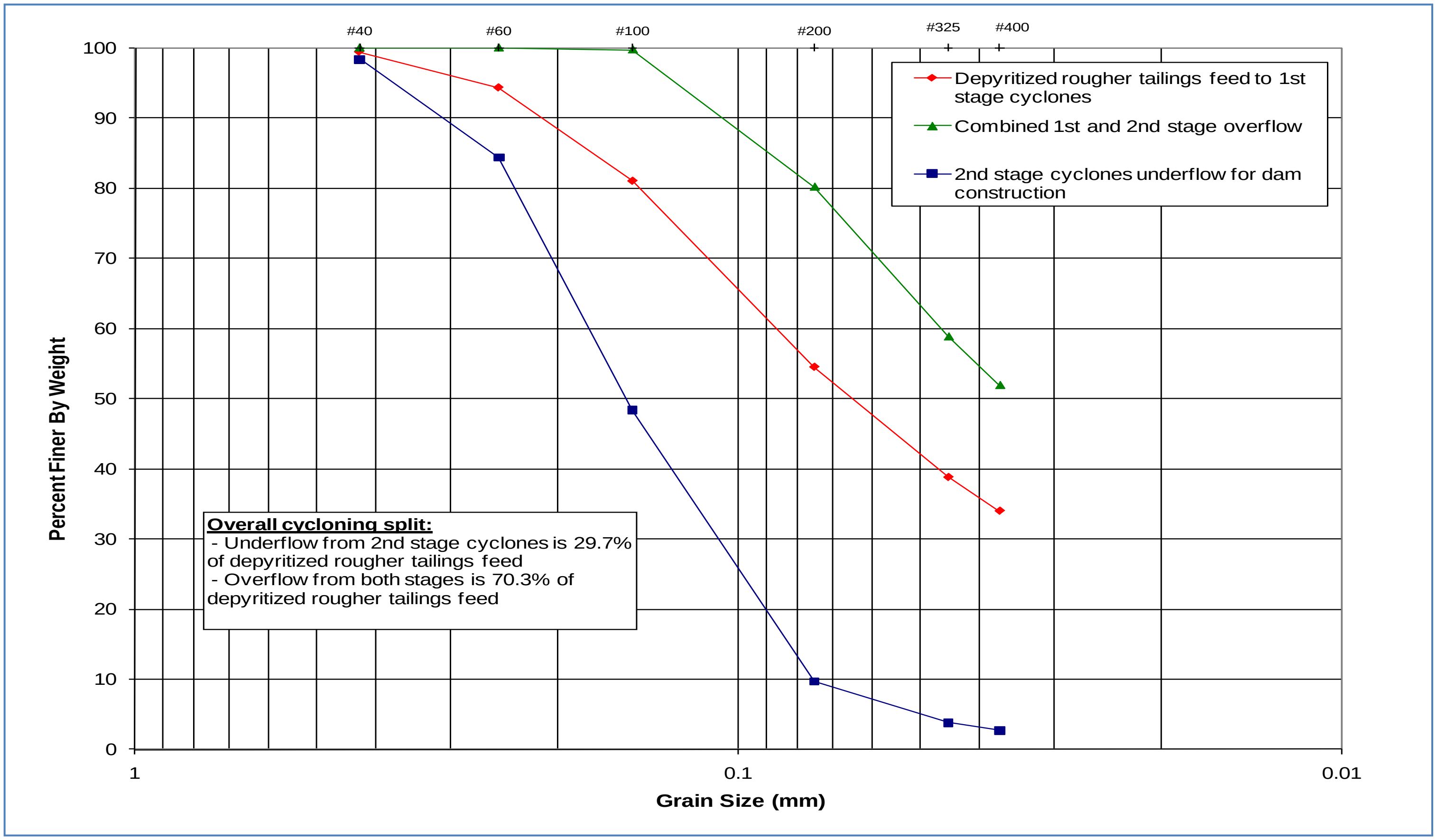


Figure 19.18 Cycloning Summary



Cycloned sand availability for dam construction at Red Chris is estimated based on the following assumptions:

- Cycloning period = 6 months per year (note that Kemess operates successfully year round unless temperatures drop to below -20ºC) – June through November.
- During cycloning period, cyclone plant availability = 92% plant efficiency x 90% sands plant efficiency = 83% overall efficiency during the 6-month timeframe.
- The de-pyritized rougher tailings stream comprises 85% of the total tailings stream (i.e. 85% x 30,000 tpd = 25,500 tpd). Cleaner tailings are not suitable for production of cycloned sand due to concentrated pyrite content.
- ABA tests on the de-pyritized rougher tailings samples produced from the May 2004 program at G&T confirm the viability of flotation of the rougher tailings as a means of producing a reliable and consistent NAG product.
- It is projected that about 30% of the de-pyritized rougher tailings stream will be recovered as cycloned sand from a two-stage cycloning process, based on the computer simulations carried out by Krebs (see Figure 19.18). On this basis, sand availability is about 6,200 tonnes/day when the sand plant is in operation and accounting for plant availability factors.

Based on the above assumptions, the estimated sand availability per year (assuming 6 months/year of downstream cycloned sand production) for downstream shell extension and raising is about 1.13 million tonnes. At a compacted in-place dry density of 1.6 tonnes/m^3 (based on Kemess experience) the resulting sand volume per year available for dam construction would be about 700,000 m^3.

The estimated sand fill volumes required for the North and South Dams (to crest El. 1180 m) are 10.6 million m^3 and 3.8 million m^3, respectively, which in total averages 480,000 m^3 per year over the projected 30-year mine life. In turn, that amounts to an average of about 4 months of cycloning each year over the 30-year mine life, meaning that there is an excess of sand production capacity available, which affords the following opportunities to be considered as detailed design proceeds:

- The Zone 4 sand shells, or portions of them, can be completed in advance of the end of milling operations, allowing progressive reclamation during the mine life.
- The sand shells can be overbuilt and/or flattened, and incorporate features (such as berms) to facilitate access for reclamation media placement, erosion control, slope drainage, and instrumentation access for post-closure monitoring.

19.6.13 Tailings Deposition Management

Discharge of tailings into the impoundment will occur from the North Dam, the South Dam, and from the west side of the valley, below the plant area. Tailings discharge locations will be as summarized in Table 19.8.

Table 19.8 Tailings Deposition: Discharge Locations

Tailings Stream	Discharge Locations
Cleaner tailings + sulphides from flotation of rougher tailings (~ 15% of total tailings stream)	North Dam crest (until impoundment expands south beyond Saddle Dam). Thereafter, cleaner tailings stream to be discharged from west valley slope at least 200 m upstream of the North Dam centerline, but sufficiently far north of the Reclaim Barge to avoid process water quality issues. Over the course of the final year or two of tailings discharge, sub-aqueous discharge of the cleaner tailings stream will be required, so as to achieve permanent submergence within the closure water pond. This may involve the construction of a pipeline causeway extending partially into the pond, or the use of a floating pipeline.
Rougher tailings	North Dam, South Dam, and west abutments of both dams when active dam raising prevents this from the dam crests.
Cyclone plant overflow	North Dam, South Dam, and west abutments of these dams. Effort to be made to extend discharge of cyclone overflow across these dams as much as possible, particularly at the South Dam, as these fine tailings will serve to reduce seepage.
Cyclone plant underflow	Zone 4 sand shell for North and South Dams, and upstream cells for Zone 1 till core raise support for North Dam

No tailings will be discharged from the Northeast Dam, which will function as a water-retaining structure.

Given the pervious nature of the South Dam, an important tailings deposition requirement will be prevention of PAG tailings deposition within the immediate vicinity of the South Dam. This will be achieved via one or more of the following strategies:

- Discharge of the PAG tailings stream into the northern portion of the reclaim water pond within the impoundment.
- Maintenance of an above-water tailings beach at least 300 m wide upstream of the South Dam at all times, via discharge of de-pyritized rougher total tailings, and via discharge of cyclone overflow.

The second strategy will not be achievable until several years into the operation. However, the first strategy should be achievable throughout the mine life, and relies upon the reclaim water pond serving to limit the southward migration of PAG tailings. Furthermore, as indicated by the seepage analyses, tailings deposited immediately upstream of the South Dam in the earlier years of the mine operation will be permanently saturated in any case.

19.6.14 Tailings Pipelines

The tailings pipelines system has yet to be designed. It is anticipated, however, that the system would be roughly as follows:

- Dedicated pipeline for PAG tailings (cleaner + rougher flotation sulphides reject)
- Dedicated pipeline for rougher (NAG) tailings, with gravity flow from plant to cyclone station, and splitter box, for use in distribution of rougher tailings into delivery pipelines for gravity flow of overflow to the Tailings Impoundment
- Splitter box and pipelines for gravity flow of cyclone underflow from cyclone plant to North and South Dams
- Emergency (backup) tailings pipeline for combined cleaner and rougher tailings when one or more of the above lines is temporarily out of service. Separate backup lines for the rougher and the cleaner tailings may be provided.

Flows in all tailings pipelines will be via gravity.

The tailings pipelines shall be located, relative to runoff diversions, such that any release from a ruptured tailings pipeline will report to the tailings impoundment, and will not enter runoff diversion ditches intended for non site contact water.

Additional spare pipelines may be added as detailed design proceeds. At present the tailings pipeline system as described above is conceptual.

19.6.15 Process Water Reclaim

Process water will be reclaimed from the pond within the tailings impoundment, via the floating reclaim barge. Cyclone sand drainage water and seepage reporting to the North Reclaim Dam pond and the South Reclaim Dam pond will be pumped back into the impoundment, upstream of the North and South Dams, respectively. Water pumped from the seepage pumpback wells will be pumped into the North and South Reclaim Dam ponds, and from there to the tailings pond.

19.6.16 Analyses

Design analyses are ongoing as part of the development of the detailed design of the tailings storage facilities. The following sections describe briefly the key analyses completed to date, and those still in progress as of issue of this document.

19.7 Seepage Modeling

A three-dimensional MODFLOW model was constructed for seepage modelling as part of the EIA process. That model has been refined and updated on the basis of the two additional pump tests undertaken in the Winter 2010 site investigation program, the additional stratigraphic information obtained from the drilling program, and calibrated against additional piezometric data and the updated site hydrology characterization.

For context, Table 19.9 summarizes the results of the MODFLOW-predicted seepage flows within the aquifers underlying the tailings impoundment, pre-mine and for the closed tailings impoundment. Given the reduced estimated precipitation relative to that incorporated in the original modelling, the “infiltration from precipitation” numbers were approximately double relative to current estimates. Note that seepage from the tailings deposit was estimated during the EIA to lie within a range of approximately 25 to 50 litres/sec, thus constituting a minor component of overall flow in the aquifers, the majority being natural recharge.

Table 19.9 MODFLOW results from EIA

Flow Component	Volumetric Flow Rate (L/s)			
	Pre-Mine		Closure	
	Buried Channel	Isotropic	Buried Channel	Isotropic
Infiltration from Precipitation	214	214	214	214
Recharge from Tailings	0	0	43	25
Modeled Total Inflow	214	214	257	239
Outflow to Drains	215	215	259	241

The updated MODFLOW model was similarly used to predict rates of infiltration from the tailings into the underlying aquifers for the closure configuration of the tailings impoundment, with above water beaches (about 300 m in width) separating the North and South Dams from the closure water pond, and the water pond in direct contact with the Northeast Dam. This analysis yielded an estimated seepage rate in the range of 10 to 50 litres/sec infiltrating into the aquifers. A slightly lower rate of seepage was estimated assuming the presence of a geomembrane liner within the limits of the start-up water pond against the North Dam starter dam, but for closure it is prudent to discount that liner, which per previous discussions is required only to control seepage loss from the start-up water pond prior to the development of the tailings deposit.

A MODFLOW analysis was also undertaken to estimate seepage from the impoundment at approximately half of its ultimate height. This analysis yielded essentially the same result as per the closure configuration (up to about 40 litres/sec), not including drainage of construction water from hydraulic sandfill placement operations that will be collected in the Reclaim Dam ponds), indicating that once the tailings deposit is well-developed, the low hydraulic conductivity of the tailings will be the controlling factor in terms of seepage rates.

The MODFLOW analyses included allowance for areas where the closure water pond would be in direct contact with native soils around the pond periphery, which would provide for a more hydraulically-conductive flow path into the valley bottom aquifer units.

Additional MODFLOW analyses in progress include scenarios involving a till core within the South Dam, and partial seepage cut-offs (e.g. deep soil mixing, soil-cement-bentonite cut-off walls) below the till cores of the North and South Dams. Initial results from such simulations indicate that the partial cut-offs achieve negligible to minimal benefit at high cost, and to be effective any such cut-off walls would have to extend to depths representing over half of the aquifers thickness (i.e. depths of > 50 m), and that the seepage reduction afforded by such cut-offs is minimal once the tailings deposit is developed and serves to limit the rate of recharge of the underlying aquifers.

MODFLOW analyses are also being undertaken to estimate the efficiency of seepage capture by the pumpback wells (i.e. percentage of impoundment seepage recovered versus percentage bypassing seepage capture), and will be reported in the detailed design report for the tailings facility.

19.7.1 2-D SEEP/W Modelling

Two-dimensional finite element seepage modelling using SEEP/W was also carried out for the North and South Dams. Specifically, analyses completed as of the issue of this document include:

- The South Dam in its closure configuration;
- The North Dam starter dam with the start-up water pond, prior to deposition of tailings into the impoundment; and
- The North Dam in its closure configuration.

The two-dimensional SEEP/W finite element seepage model of the South Dam, at the deepest valley section (i.e. greatest thickness of the underlying aquifers) was undertaken to:

- Estimate the rate of seepage through the South Dam at closure for a range of assumed beach widths and tailings hydraulic conductivity scenarios;
- Obtain a comparison in terms of seepage rate with the MODFLOW modelling; and
- Estimate the position of the steady state phreatic surface through the South Dam (and the upstream tailings beach) for the closure configuration of the tailings impoundment.

The analysis results indicate the following:

- The combination of the low hydraulic conductivity of the tailings upstream of the South Dam, the wide beach (300 m) separating the South Dam from the closure water pond, and the free-draining nature of the cyclone sand fill, combine to produce a low phreatic surface through the South Dam.
- Seepage rates through the South Dam, taking the unit seepage rate per metre length of dam, and multiplying this by a length of 800 m representing the width of the deep aquifer reasonably represented by the 2-dimensional section, range from 5 to 25 litres/sec, depending on the assumed hydraulic conductivity of the upstream tailings.
- The zone of desaturation predicted within the impounded tailings is indicated and, given the aforementioned restrictions imposed on the locations for discharge of PAG cleaner tailings, will be of no concern as PAG tailings would not be deposited within this zone.

The analysis results indicate that inclusion of a till core within the South Dam design is not required, as the core would, for the closure configuration, be essentially redundant.

A two-dimensional SEEP/W model was also constructed for the North Dam starter dam and the start-up water pond (with a volume of about 5 million m^3), to estimate:

- Seepage losses from the pond prior to the initiation of tailings discharge, which if excessive would prevent the dam from impounding the requisite volume for process plant supply;
- Seepage pressures in the dam foundation.
- The benefit of seepage reduction measures, such as an upstream till blanket extension, or liner extension, of the till core to reduce seepage to manageable levels prior to the tailings deposit serving to limit infiltration into the hydraulically conductive upper aquifer unit.

The results of the SEEP/W analyses, along with a MODFLOW model simulation of the same start-up situation, are provided in Table 19.10.

Table 19.10 Modelled Seepage from North Dam Startup Water Pond

Upstream extent of liner relative to extend of startup water pond (%)	2-D SEEP/W analysis	3-D MODFLOW analysis
	Predicted seepage (litres/sec)	
0	55	49
25	26	
50	20	19
	17	
100	7	6

Notes:

1. *Hydraulic conductivity of upper aquifer assumed = 10^{-5} m/sec.*
2. *Liner modelled as 1 m thick zone with K = 10^{-6} m/sec, a very conservative (high) number, that would be representative of a compacted till blanket with many defects. A geomembrane installed to a moderate standard would have a significantly lower effective hydraulic conductivity. For a liner K = 10^{-9} m/sec, essentially the same seepage rates were obtained, indicating that, provided the liner is an order of magnitude or more lower in hydraulic conductivity than the upper aquifer, the seepage rate is insensitive to its modelled hydraulic conductivity. This indicates that a liner of high quality, with minimal construction defects, is not required to achieve the desired objective.*
3. *Seepage rates marked with * assume a hydraulic conductivity of 10^{-4} m/sec for the upper aquifer.*

Based on these results, it would be sufficient to provide for a liner extension of the till core of the North Dam that extends over the downstream 25-50% of the extent of the start-up water pond. Additional analyses are being undertaken to optimize the extent of the liner and the design of the seepage pumpback wells system.

SEEP/W analysis was also undertaken for the North Dam in its closure configuration.

19.8 Dam Stability Analyses

Two-dimensional limit equilibrium stability analyses have been completed for the ultimate configurations of the North Dam and the South Dam (crest elevations 1180 m). The stability analysis sections are illustrated on Figures 19.19 and 19.20 for the North and South Dams, respectively, along with the assumed material parameters and piezometric conditions. Given the high density and shear strength of the various foundation units, the foundation soils were modelled as having a uniform shear strength (i.e. same strength for till and aquifer units), and sensitivity analyses were undertaken to derive the factor of safety as a function of the foundation soils shear strength. The results of this analysis are illustrated on Figures 19.21 and 19.22 for the North and South Dams, respectively. The results indicate more than ample factors of safety even for unrealistically low estimates of the foundation soils shear strength (effective friction angle, ϕ', of 32^o represents an underestimate of foundation soils' strength based on laboratory testing and Standard Penetration Test results). It should be noted that the stability analysis for the South Dam assumed a significantly higher phreatic surface than predicted on the basis of the MODFLOW and SEEP/W analyses, and would be more representative of an end-of-construction case than the long term steady state condition.

Given that the dam fills and the foundation soils are non-susceptible to liquefaction given the low to moderate seismicity at the site, the seismic stability of the dams under the design earthquake loading (MCE, moment magnitude 6 and PGA = 0.19g), a pseudostatic analysis was undertaken. The seismic coefficient used was 50% of the PGA. The impounded tailings were assigned a liquefied residual shear strength for these analyses. The results are illustrated, again in terms of factor of safety versus foundation shear strength, on Figure 19.23. More than acceptable factors of safety are indicated.

Stability analyses were not carried out for the Northeast Dam, or for the two Reclaim Dams, as these are very low structures, to be constructed on the same competent foundation soils as the North and South Dams. Analyses for these dams will be included in the detailed design report.

19.9 Water Balance – Operations

The water balance for the tailings impoundment during its operational phase is discussed in Section 8. In summary, the tailings impoundment, even with runoff diversions, is projected to be in a net annual water balance surplus, except for the initial years of operation (start-up impoundment), which will require discharge of surplus water from the impoundment on an annual basis.

19.10 Water Balance - Closure

The monthly water balance for the tailings impoundment was developed for closure conditions, meaning all diversions were considered to be breached, and the establishment of steady state seepage conditions. As indicated above, the 3-dimensional MODFLOW groundwater model for the ultimate tailings impoundment, and the SEEP/W analyses for the North and South Dams, suggest a steady state seepage rate for the closed impoundment configuration of about 50 litres/sec, as compared to about 40 litres/sec per the EIA analyses). This is incorporated into the impoundment's closure water balance, summarized on a monthly basis for annual average hydrologic conditions in Figure 19.24, and for a 1 in 200 year return period dry year scenario in Figure 19.25. It is important to note that neither of these figures include the inflows to the closed tailings impoundment from water treatment plant discharge (treating open pit water and rock storage area infiltration that is routed to the open pit) in the post-closure period. The closure water balance model is simplistic in that any water inflow that would result in a rise of the closure pond level above the spillway invert elevation (assumed at El. 1175 m) would be discharged (i.e. reservoir routing is not accounted for).

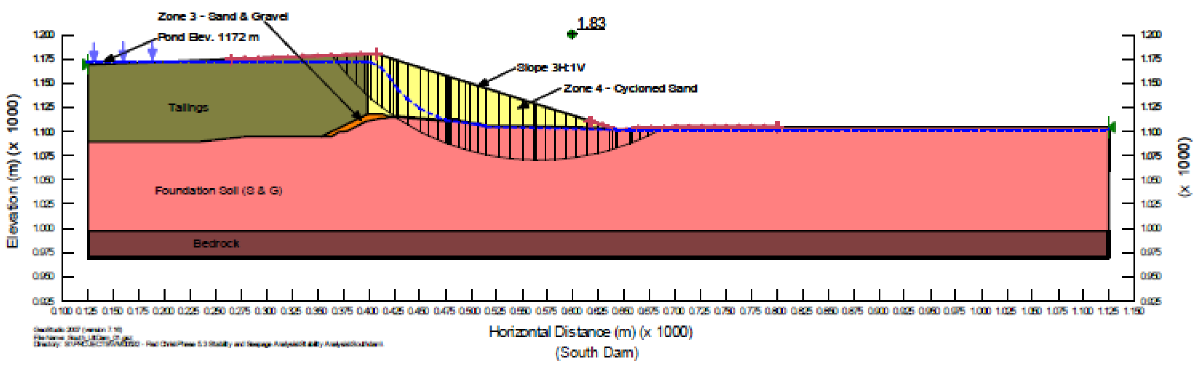
The results indicate a predicted spillway discharge for every month of the year for the average annual scenario. The month of lowest discharge is April, and only if the seepage rate were 160 litres/sec would the impoundment be in balance (i.e. no spillway discharge) for that month, with inflows equalling seepage outflows.

For the 1 in 200 year return period dry year (423 mm annual precipitation, 300 mm annual evaporation), for a seepage rate of 50 litres/sec, spillway discharge is also projected for every month of the year, although the volumes are approximately 30-50% of those projected for the average year. For the 1 in 200 year return period dry year scenario, only if the seepage rate were about 110 litres/sec or higher would the pond be in balance with no spillway discharge, for the months of April, July and August.

In summary, the closure water balance model projections for the tailings impoundment indicate a monthly water balance surplus for all months of the year, including for a 1 in 200 year return period dry year. As such, a water pond can be readily maintained within the impoundment, with no potential for desaturation of PAG tailings provided these are discharged into the impoundment well upstream of the South Dam.

Figure 19.19 Ultimate North Dam: Stability Analysis Section

Material Description	γ_b kN/m^3	ϕ' Degrees	c kPa	Piezometric Condition
Zone 3 - Sand & Gravel	17	35	0	Piezometric line 1
Zone 4 - Cycloned Sand	16.5	35	0	Piezometric line 1
Tailings	13	$S_u/\sigma'_v = 0.25$		Piezometric line 1
Foundation Soil	17	Varies	0	Piezometric line 1
Bedrock		Harder Layer		Piezometric line 1



amec	AMEC Earth & Environmental 2227 Douglas Road, Burnaby BC V5C 5A9 Tel: 604-294-3811/Fax: 604-294-4664	**Project:**	Red Chris Copper Project - Ultimate Dam Stability Analyses
		Title:	Ultimate South Dam: stability analysis section
		Client:	Red Chris Development Company Ltd. **Job no.:** VM00532 **Fig. 7.19**

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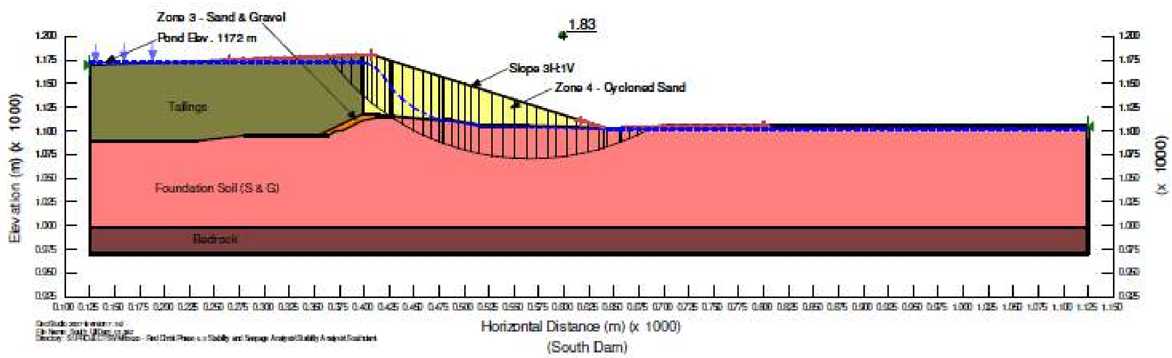
Figure 19.20 Ultimate South Dam: Stability Analysis Section

Material Description	γ_b kN/m^3	ϕ' Degrees	c kPa	Piezometric Condition
Zone 3 - Sand & Gravel	17	35	0	Piezometric line 1
Zone 4 - Cycloned Sand	16.5	35	0	Piezometric line 1
Tailings	13	$S_u/\sigma_v' = 0.25$		Piezometric line 1
Foundation Soil	17	Varies	0	Piezometric line 1
Bedrock	Harder Layer			Piezometric line 1



	AMEC Earth & Environmental 2227 Douglas Road, Burnaby BC V5C 5A9 Tel: 604-294-3811/Fax: 604-294-4664	**Project:**	Red Chris Copper Project - Ultimate Dam Stability Analyses			
amec		**Title:**	Ultimate South Dam: stability analysis section			
		Client:	Red Chris Development Company Ltd.	**Job no.:**	VM00532	**Fig. 7.20**

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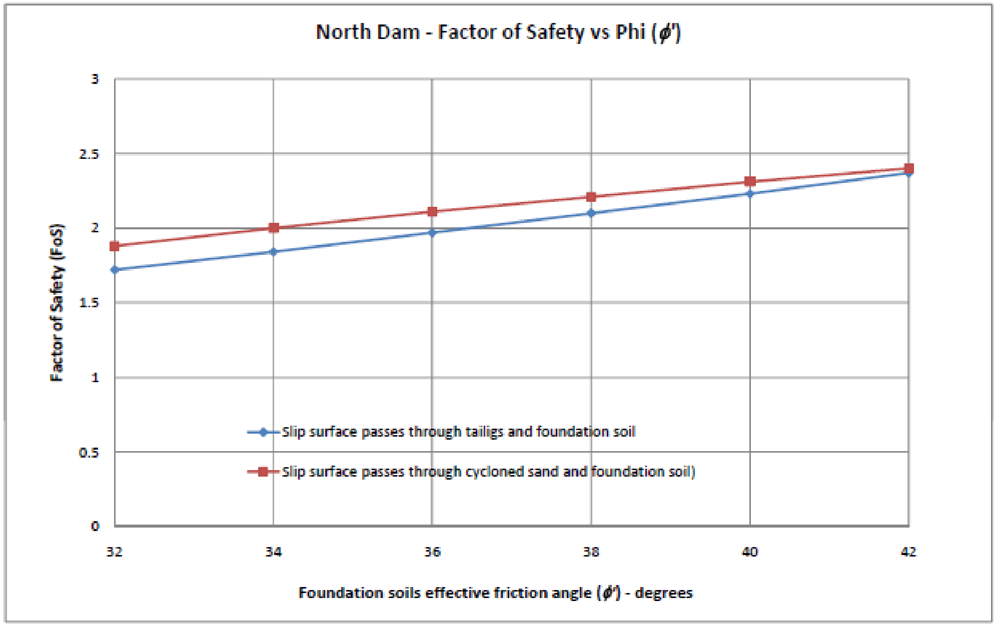
Figure 19.21 North Dam: Stability Versus Foundation Shear Strength



amec	AMEC Earth & Environmental 2227 Douglas Road, Burnaby BC V5C 5A9 Tel: 604-294-3811/Fax: 604-294-4664	**Project:**	Red Chris Project - Ultimate Dam Stability Analyses			
		Title:	North Dam: stability versus foundation shear strength			
		Client:	Red Chris Development Company Ltd.	**Job no.:**	VM00532	**Fig. 7.21**

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Figure 19.22 South Dam: Stability Versus Foundation Shear Strength

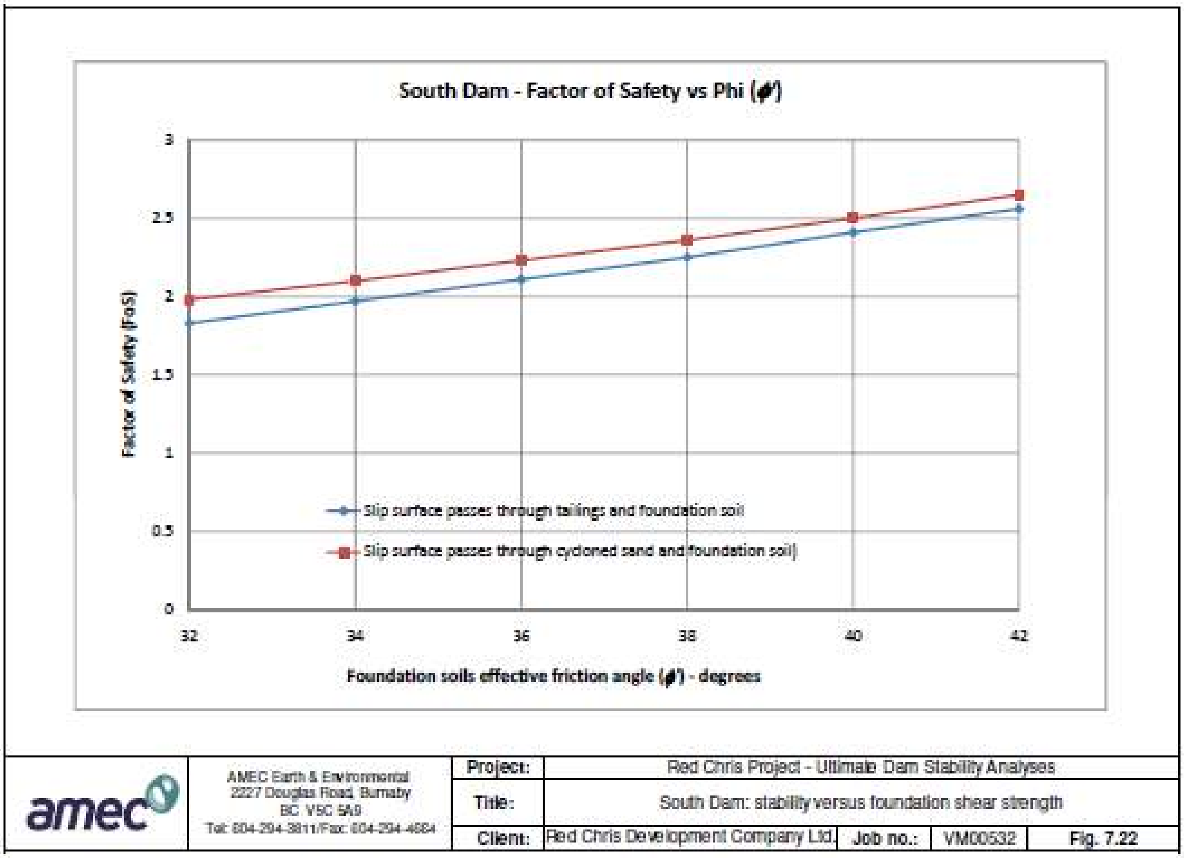

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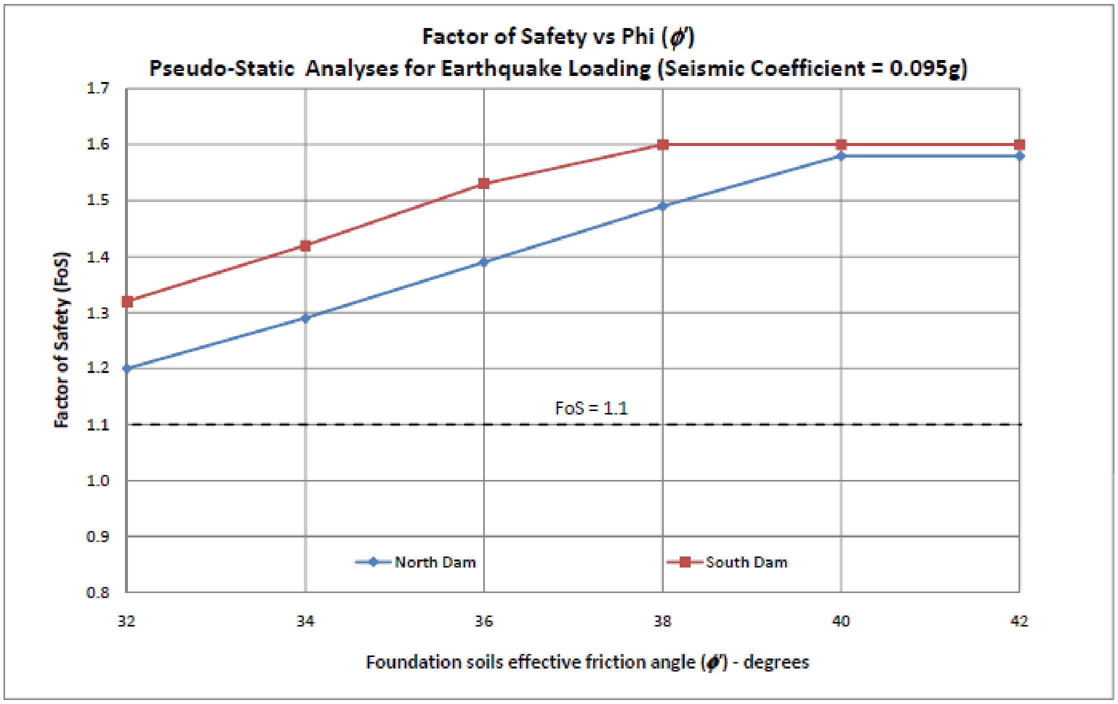
Figure 19.23 Pseudostatic Stability Analysis Results



amec	AMEC Earth & Environmental 2227 Douglas Road, Burnaby BC V5C 5A9 Tel: 604-294-3811/Fax: 604-294-4664	**Project:**	Red Chris Project - Ultimate Dam Stability Analyses			
		Title:	Pseudostatic Stability Analysis Results			
		Client:	Red Chris Development Company Ltd.	**Job no.:**	VM00532	**Fig. 7.23**

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Figure 19.24 Annual Tailings Impoundment Closure Water Balance: Average Year

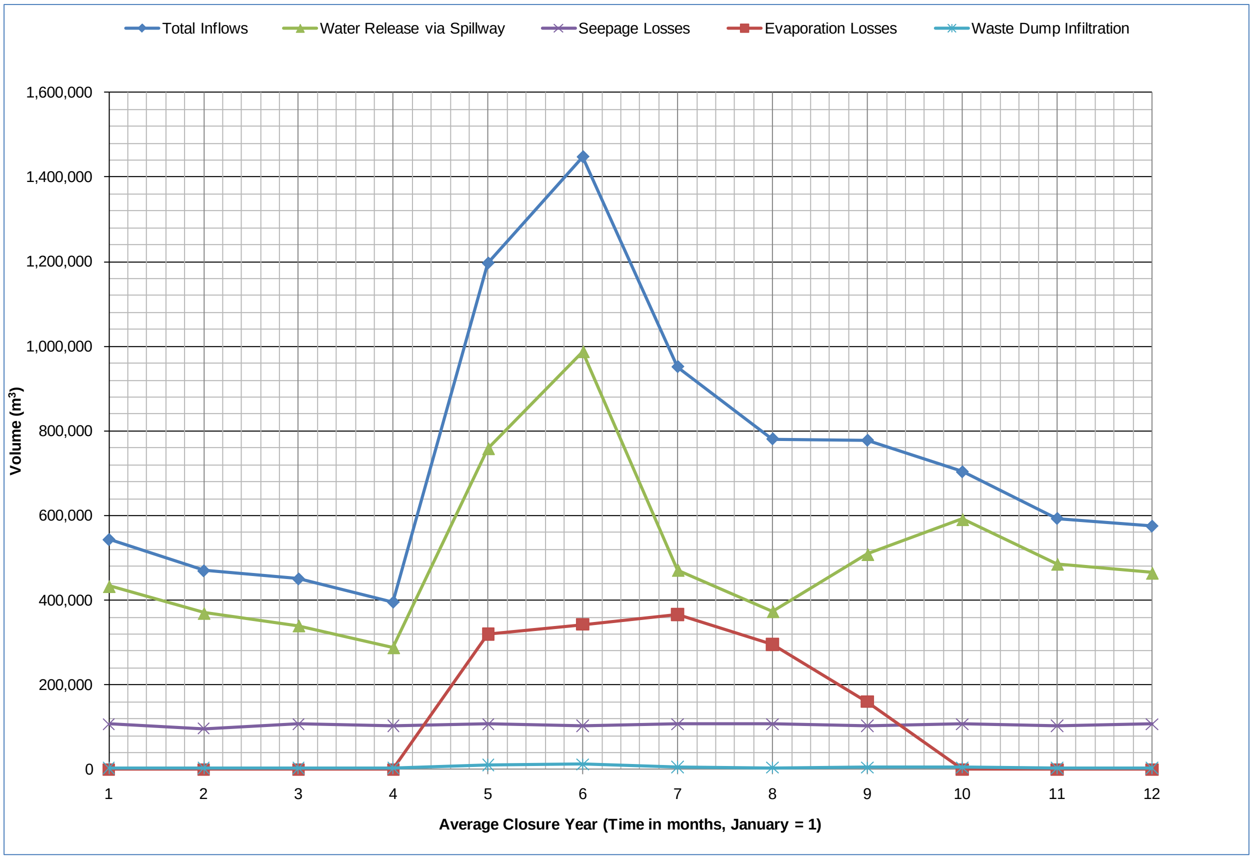
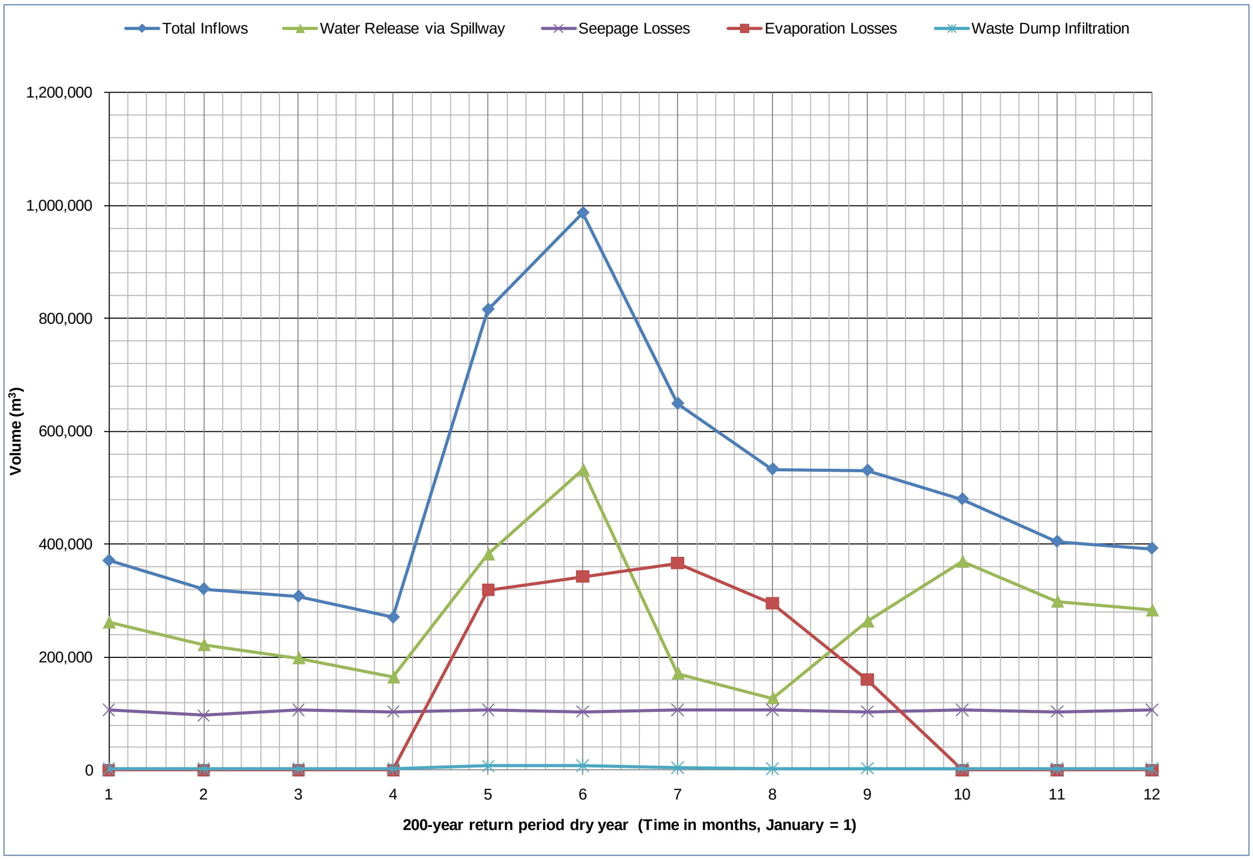


Figure 19.25 Annual Tailings Impoundment Closure Water Balance: 1/200 Dry Year



20 Water Management Plan

This section has been updated from the 2005 Feasibility Study to incorporate the following additional data and changes to the proposed operating regime:

- Overall climatic data on the basis of continued data gathered on-site has been updated and accordingly the Water Balance Model has been updated.
- The proposal to take make up water from the Klappan River pump house for water supply is no longer being considered for the project. Improved water balance figures indicate that, as mine development proceeds, the project has surplus water and needs to discharge water from the site. As a consequence, it is intended to accelerate construction of the TSF early in the mine life to allow this excess water to be stored in the TSF; such that sufficient water is available to operate the Mill year around, without additional make up water from the Klappan River.
- LOM has been increased to approximately 28.3 years and accordingly the Water Management Plan has been modified.

20.1 Introduction

The goal of the Water Management Plan is to minimize the impact of the Red Chris project on the local and regional aquatic ecosystems, to which end the following strategies will be used:

- Intake of fresh water from local aquifers and streams will be minimised by recycling and reusing water to the greatest extent practical. All process water will be recycled from the TSF and the seepage pumpback systems. There will be no external fresh water make up source.
- Isolation of the site and reduction in the volume of site contact water to the greatest extent practical, via runoff and diversion ditches, which reduces the net annual water balance surplus for the TSF.
- Collect and route all contact water from the open pit, rock storage area, surface runoff from site infrastructure and camp sewage to the TSF during operations. The TSF will therefore represent the repository of all site-affected water.
- Manage PAG materials with regard to acid rock drainage and metal leaching (ARD/ML) and other water quality concerns.
- Manage and use NAG materials in order to minimise metal leaching and other water quality concerns.
- Monitor the quantity and quality of all discharge.
- Adjust water collection, transfer, treatment and/or disposal practices through an adaptive management program. If monitoring results indicate discharge quantity or quality is not meeting discharge criteria, adapt the Plan.

20.2 The Water Management Plan

- During the pre-production period, surface runoff from the plant and camp sites, crusher and conveyor, overburden stockpile and rock storage area will be collected in sumps, ponds and ditches, transferred to sedimentation ponds, for removal of suspended solids and particulate/colloidal phase contaminants and routed to ambient receiving water bodies. The quantity and quality of runoff from these regions will be monitored, to ensure it complies with applicable direct discharge limits. Best management practices will also be followed to help minimize erosion of roads, crusher, plantsite and camp site areas and thus reduce sediment loadings to ambient receiving water bodies and/or associated mitigative treatment costs.
- Surface runoff from the access roads over the project life will be collected in a series of ditches and released to ambient receiving waters. If necessary, especially on steep slopes where erosion may be problematic, small ponds that function as both equalization basins (to attenuate peak flows released to ambient receiving water bodies) and sediment traps, will be installed within the road runoff collection systems.
- Surface runoff from haul roads will be intercepted by a contact water diversion ditch and routed to the TSF, for use as reclaim water in the process plant.
- Pre-production camp sewage will be managed via approved septic fields, or if necessary stored in approved vessels and removed from site as required.
- During operations camp sewage will be transferred to the TSF.
- Staged collection ditching, around the rock storage area, will minimize net runoff from the exposed plan area of this facility, thus minimizing contact water that must be routed to the TSF (during operations).
- Following completion of the North Dam starter dam in pre-construction Year -02, runoff from a reduced portion of the respective contributing catchments, upstream of either tailings dam option, will be allowed to report to the reservoir formed by this dam, which will be the water source for mill start-up. The remainder of the natural surface runoff, in the catchments upstream of the starter dam, will be diverted around the TSF via a series of ditches and allowed to report to ambient receiving waters in Quarry Creek to the north or Trail Creek to the south. Water stored in the TSF will be recycled for use within the process plant, dust suppression and for fire suppression.
- During the operational period from Year 01 through Year 28, surface runoff within the open pit and rock storage area will be transferred to the contact water diversion ditch and routed to the TSF.
- As the TSF expands to the south in the early years of operation, additional runoff diversions will be constructed around the eastern portion of the TSF.

- Drainage and excess water from the hydraulically-placed cyclone sand fill that will be used construct the North and South Dams will be collected and pumped back into the TSF. Seepage pumpback wells will be installed downstream of the North and South Dams and used for pumping this seepage back to the TSF, if water quality and/or water balance considerations require this.
- Despite the runoff diversions, the recycle of process water and the storage of water in the voids of the deposited tailings, the TSF will operate under a net annual water balance surplus, of approximately 2 million m^3 per year (under average annual hydrologic conditions). The surplus water will be discharged from the impoundment, via pumping, a pipeline and an outfall downstream of the north seepage pumpback wells.
- At closure, an open channel closure spillway will be constructed around the left (looking downstream) abutment of the Northeast Dam. The closure spillway will be suitable for routing of the Probable Maximum Flood event.
- A vegetated cover will be constructed over the rock storage area. Runoff from the cover, constructed over the rock storage area, will be released to the environment. Water that infiltrates through the cover will be collected in a system of engineered under-drains and routed to the open pit.
- At closure the open pit will be allowed to naturally re-flood. The topography of the pit is such that the pit will flood to an elevation of approximately 1452 m at which point the pit would overflow at a topographic low point located at the east end of the pit. It is currently projected for the pit to take approximately 128 years to flood to this level.
- Water quality modelling suggests that the pit water, which will include infiltration collected from the rock storage area, will not be suitable for discharge to the environment at this time, (year 128 post closure,) due to oxidation of sulphide minerals in the wall rock and waste rock, resulting in elevated metal levels. As a result, pit water will be treated on a seasonal basis, to generate water of suitable quality for discharge to the environment. The water treatment plant discharge will be routed to the TSF, through the TSF and then to the environment via the closure spillway. The treatment plant will be operated in such a way as to prevent an uncontrolled discharge of poor quality water.

Table 20.1 presents a summary of the key project components and the water management plan for each during the pre-production, operations and closure phases.

Table 20.1 Summary of Water Management Plan

	Pre-Production	Operations	Closure
Open Pit	Not developed	Run-on minimized by non-contact water ditches Direct precipitation collected in an in-pit sump, pumped to	Allowed to flood, predicted that water treatment will be required to generate water of suitable quality for discharge. Treated water will be

		contact water ditch and conveyed to TSF.	transferred to the TSF
Site Infrastructure (plantsite, campsite, conveyor, crusher)	Run-on minimized by non-contact water diversion ditches. Contact water collected by ditches and directed to sedimentation ponds for removal of suspended solids and then allowed to report to ambient receiving water bodies.	Run-on minimized by non-contact water diversion ditches. Contact water collect by ditches and conveyed to the TSF	Site infrastructure will be decommissioned and to the extent possible removed from site. Disturbed areas will be reclaimed. Sediment control measures will be employed until such a time as the runoff quality from the reclaimed areas is deemed acceptable
Rock Storage Area	Not developed	Run-on minimized by non-contact water ditches Direct precipitation and contact water collected by contact water ditches and conveyed to TSF.	Closure plan for rock storage area includes a soil cover that will minimize infiltration of direct precipitation and maximize non-contact water. Non-contact water will be allowed to drain to the natural receiving catchments. Contact water that seeps from the storage is expected to be of poor quality and will be directed to the open pit for treatment
TSF	North dam starter dam constructed in Year -02, non-contact water diverted via temporary	Runoff from dam downstream slopes will be collected by the North and South Reclaim Dams and recycled to the	Spillway constructed in the left abutment of the Northeast Dam will carry excess impoundment water into the Nea' creek

	diversions. A sedimentation pond will be constructed downstream of the tailings dam construction site to collect construction runoff until of suitable quality for discharge	impoundment. Seepage may be pumped back to the impoundment via pumpback wells installed downstream of the North and South Reclaim Dams. Excess impoundment water suitable for direct discharge will be pumped from the reclaim barge via a pipeline along the western flank of the TSF valley beyond the North Dam and reclaim dam and allow to discharge into the Quarry Creek system	Downstream shells of the tailings dams will be vegetated, sediment control measures will be employed until such a time as the runoff quality from the dam shells is deemed acceptable
Overburden Stockpile	Contact water collected by ditches and directed to sedimentation ponds for removal of suspended solids and then allowed to report to ambient receiving water bodies.	Contact water collected by ditches and directed to sedimentation ponds for removal of suspended solids and then allowed to report to ambient receiving water bodies.	Overburden stockpiles will be used during the reclamation process, after implementation of the closure and remediation plan the overburden stockpiled will have been exhausted

20.3 Mine Water Source and Use

20.3.1 Understanding Water Sources and Usage at the Mine

- Potable Water: Sourced from a fresh water well or a series of wells located near the camp. The daily demand for this water will be based on a 174 man camp. The demand is estimated at a nominal 2.5 m^3/hr or 60 m^3/day. A potable water storage tank will be installed in close proximity of the camp complex.
- Process Water / Fire Demand Water will be recycled and reclaimed from the TSF as explained in detail in this section.
- Fresh Water – to be used for the mine ablutions, reagents mixing, gland water, air conditioned coolers, etc. This will be sourced from fresh water wells and stored in a storage tank. Approximate demand for this fresh water is estimated at 45 m^3 per hour.
- Seepage Pump Back Water: Approximately 25% of seepage from the TSF through the foundations of the North Dam and South Dams will be recovered in seepage ponds and or wells, located immediately downstream of the dams and pumped back into the TSF.

Water for the mill process will be recycled from the TSF. Process water will be drawn from the TSF by floating barge mounted reclaim pump and pumped to a separate process water tank, to be located adjacent to the mill. Process water will be used in the grinding and flotation circuits.

Given the net annual water balance surplus projected, 100% of the mill process water supply will be via recycle, direct from the TSF, supplemented if necessary via the seepage pumpback wells. Water from the wells may be required for reagent mixing, grinding mill cooling water, pump glands and compressor seal water and will be pumped through a common pipeline to the freshwater storage tank, to be located adjacent to the mill. The pipeline will follow the planned service road between the mill and TSF.

The amount of reclaim water used within the mill will be maximized to the greatest extent practical, i.e. minimizing the use of freshwater within the milling process. However, within the first year of start-up it is expected that the pond, of water, within the TSF, available for reclaim to the mill, will be insufficient to operate the mill without additional make up water pumped from the TSF Seepage wells.

As the volume of water enclosed in the TSF becomes deeper and larger in volume allowing reclaim water quality to improve, RCDC will also move to replace clean seepage pumpback water with reclaim water from the TSF, for such uses as pump gland water, grinding mill cooling water and other uses with the objective of minimizing the net annual water balance surplus requiring discharge to the environment.

Freshwater will be pumped to a fire/freshwater supply tank to be located adjacent to the mill. The top of the tank will be used to store freshwater requirements for the Red Chris project, while the bottom of the tank would act to store the required freshwater reserve for site fire protection, as established by insurance requirements.

Potable water for use across the Red Chris project site will be stored within a potable water storage tank. Potable water will be obtained from a separate freshwater well installed in close proximity to the plant site and accommodation complex.

20.4 Water Balance

A monthly site water balance model has been developed for the Red Chris project. The model takes into account the various inflows and outflows from the TSF and incorporates both average years (in terms of precipitation, evaporation and runoff) and wet and dry years of varying return periods. The water balance model serves the following purposes:

- It forms the basis for the prediction of impoundment levels (average tailings level and pond levels and volumes).
- It provides a monthly projection of reclaim pond water volume and so determines whether the water pond is in surplus or in deficit on a monthly and an annual basis.

A simplified schematic of the water balance model is shown in Figure 20.1 presenting the overall site water balance during mining operations.

While the North and South Dams are being raised and extended, de-pyritized rougher tailings will be sent to a cyclone sand plant which will produce cyclone underflow for use in the construction of the dams, and cyclone overflow which will be deposited into the TSF. Contact water, comprising runoff from the rock storage area, building areas and the open pit will be collected and transported to the impoundment via ditches or pumping. Several non-contact diversion ditches have been proposed to divert clean water away from the impoundment to existing natural channels downstream. The efficiency of the ditches can be adjusted within the model.

The site water balance simulates Mine Year -2 through to Mine Year 28 and accounts for the various areas of contact versus non-contact runoff as these evolve over the life of the mine, from pre-production, impounding of the initial process water supply behind the North Dam starter dam, the southward expansion of the TSF, to the end of operations.

The following summarizes the available data and assumptions used in the site water balance.

20.4.1 Mill

- Tailings production over the life of the mine 300,000,000 tonnes.
- The cyclone sand plant will operate typically from June through October, although longer cycloning seasons are feasible.
- Tailings Solids Characteristics:
- Tailings deposited dry density (average) = 1.3 t/m^3
- Tailings slurry density = 34.4%
- Tailings solids specific gravity = 2.70
- Cyclone sand dry density = 1.6 t/m^3
- Cyclone sand residual degree of saturation (following drainage) = 35%

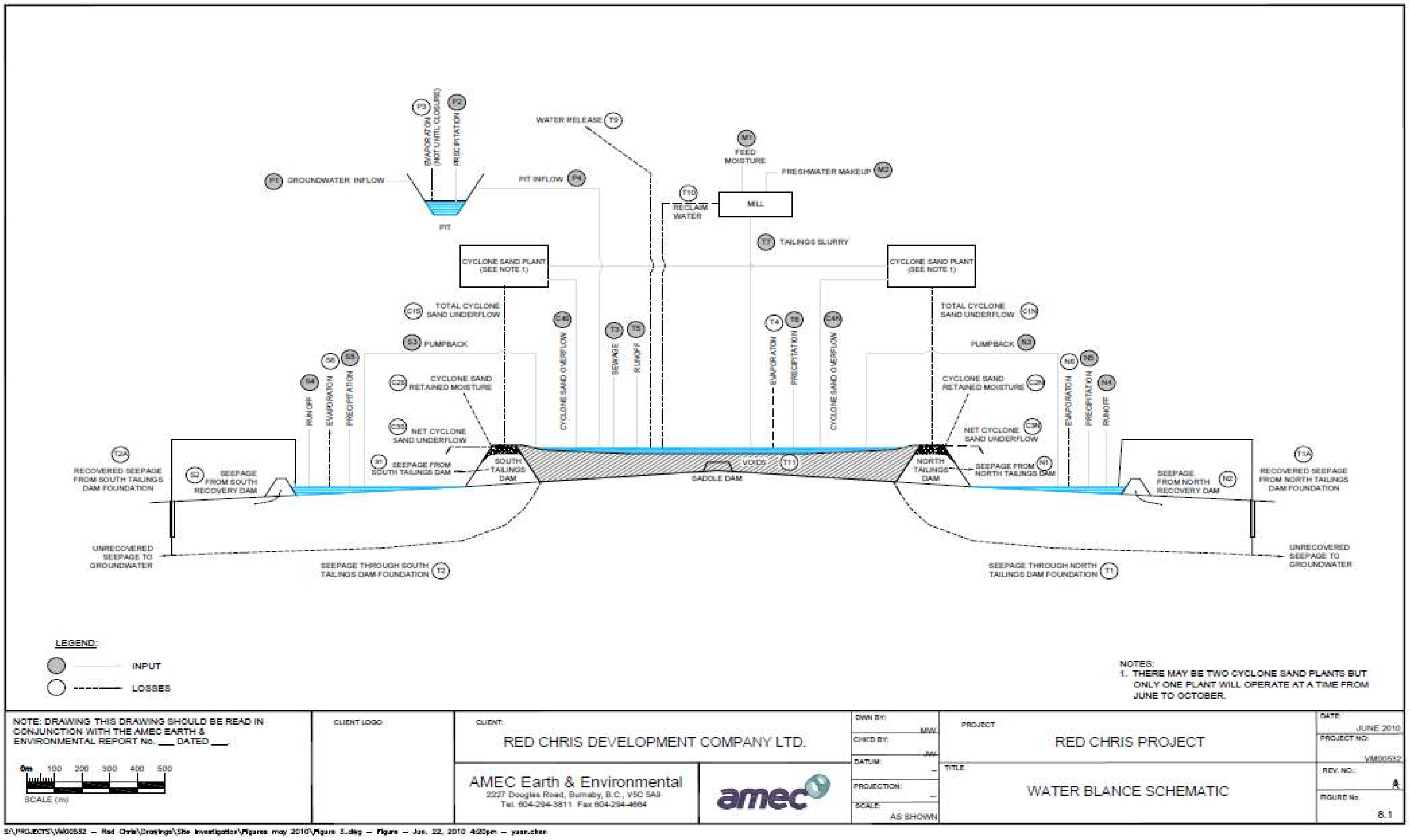
20.4.2 Pit

- The final pit area has been estimated to be 146.3 ha. For the site water balance the pit has been assumed to grow about 0.45 ha per month during mining operations.
- Pit groundwater inflow has been assumed to start at 1 litre/sec and increase on an annual basis by 5 litres/sec.
- All water collected in the pit will be directed to the TSF during operations.

20.4.3 Tailings Storage Facility

- Feed moisture is assumed to be 25 m^3/hr and starts and ends with mining operations.
- Sewage inflow rate is assumed to be 45 m^3/day and starts once tailings discharge commences.
- A start-up water pond designed to store approximately 3 months of mill process water (~5 Mm^3) will be developed against the North Dam starter dam.
- Seepage through the North and South Dam foundations has been assumed to start when the dams are operational and increase on a yearly basis. Seepage flow will be captured by the Reclaim Dam ponds downstream of the North and South Dams, or intercepted in downstream collection wells and be pumped back to the impoundment.
- Seepage through the North and South Seepage Pond Dams has been assumed to be intercepted in downstream collection wells and be pumped back to the impoundment.
- Maximum amount of water to be stored in the impoundment is equivalent to three months mill process water requirement (5 Mm^3).
- Surplus water will be discharged via additional pumping capacity (and pipeline) installed on the floating reclaim water barge. The excess water will be discharged typically over a minimum of 7 months per year. To take advantage of the greater dilution available during the freshet period, additional discharge capacity may be developed for these periods. The excess water will be discharged into the Quarry Creek system.

Figure 20.1 Water Balance Schematic



20.4.4 Catchment Areas

- Prior to Mine Year -2 (Oct), no water will be captured within the impoundment.
- From Mine Year -2 (Oct) until Mine Year -1 (Oct), the north impoundment will receive water from runoff around the impoundment and from water being diverted from the rock storage area, plant and campsite area.
- From Mine Year -1 (Oct) until Mine Year 2 (Oct), the north water diversion will be in place, therefore the north impoundment will only receive water from direct precipitation, runoff from below the diversion ditches and water being diverted from the rock storage area, plant and campsite area.
- Mine Year 2 (Oct) and beyond, all diversions will be in place so the north and south impoundment will only receive water from direct precipitation, runoff from below the diversion ditches and water diverted from the rock storage area, plant and campsite area.

Figure 20.2 illustrates the filling schedule for the TSF. Figure 20.3 plots, on a monthly basis, the total inflows and outflows for the TSF, along with the surplus water discharged to maintain a water pond volume of about 5 million m3. Note that the volumes shown on Figure 20.4 are based on average annual year runoff conditions. This figure also summarizes the estimated annual discharges of surplus water from the TSF during operations, again assuming average annual runoff conditions. The first discharge of surplus water occurs immediately before startup, prior to the first discharge of tailings into the impoundment, which reduces the volume of the startup water pond sufficiently, so as to avoid the first discharge of surplus water from the active TSF until Year 3.

Figure 20.2 Impoundment Filling Schedule

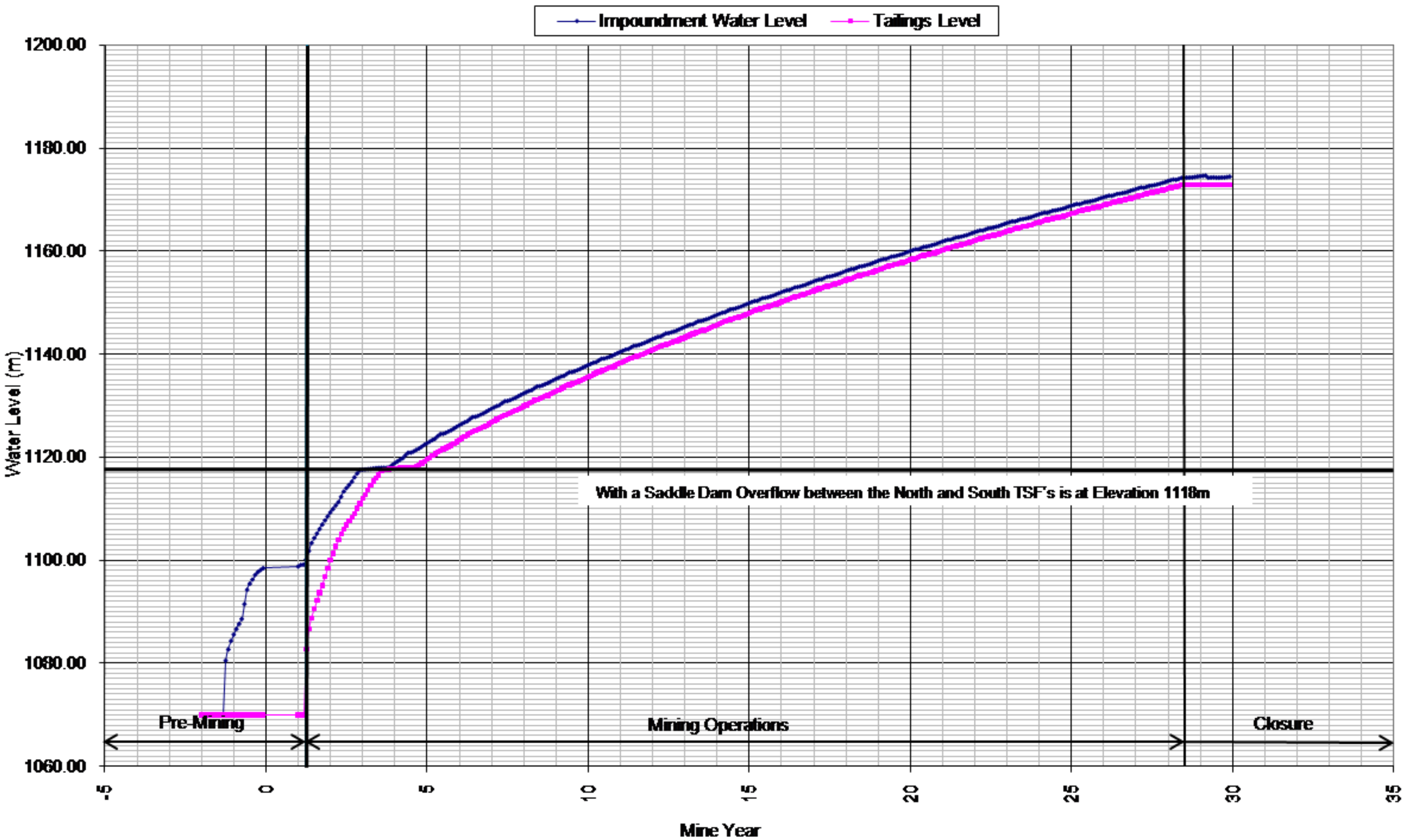


Figure 20.3 Monthly Water Balance: Key Monthly Volumes Summary

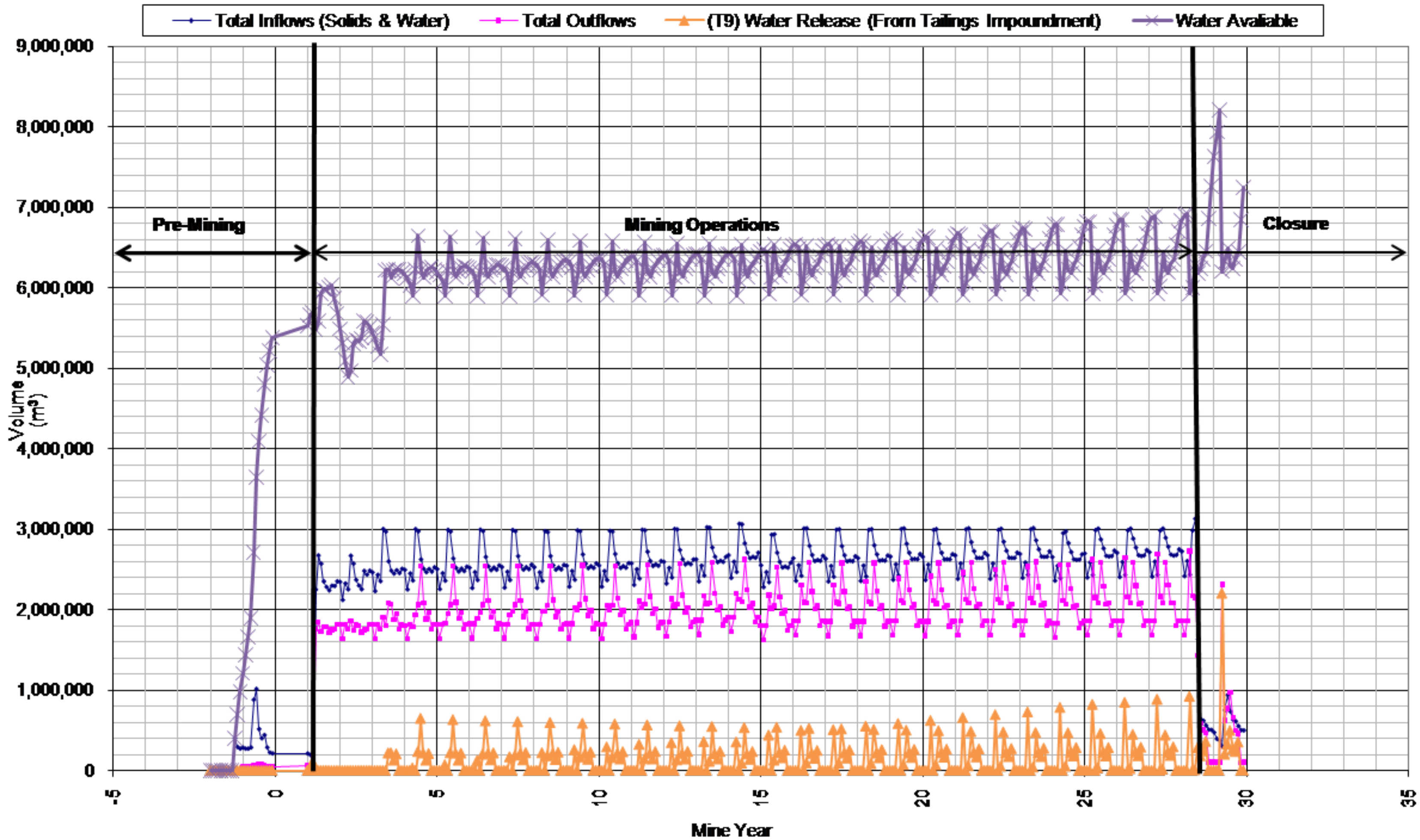
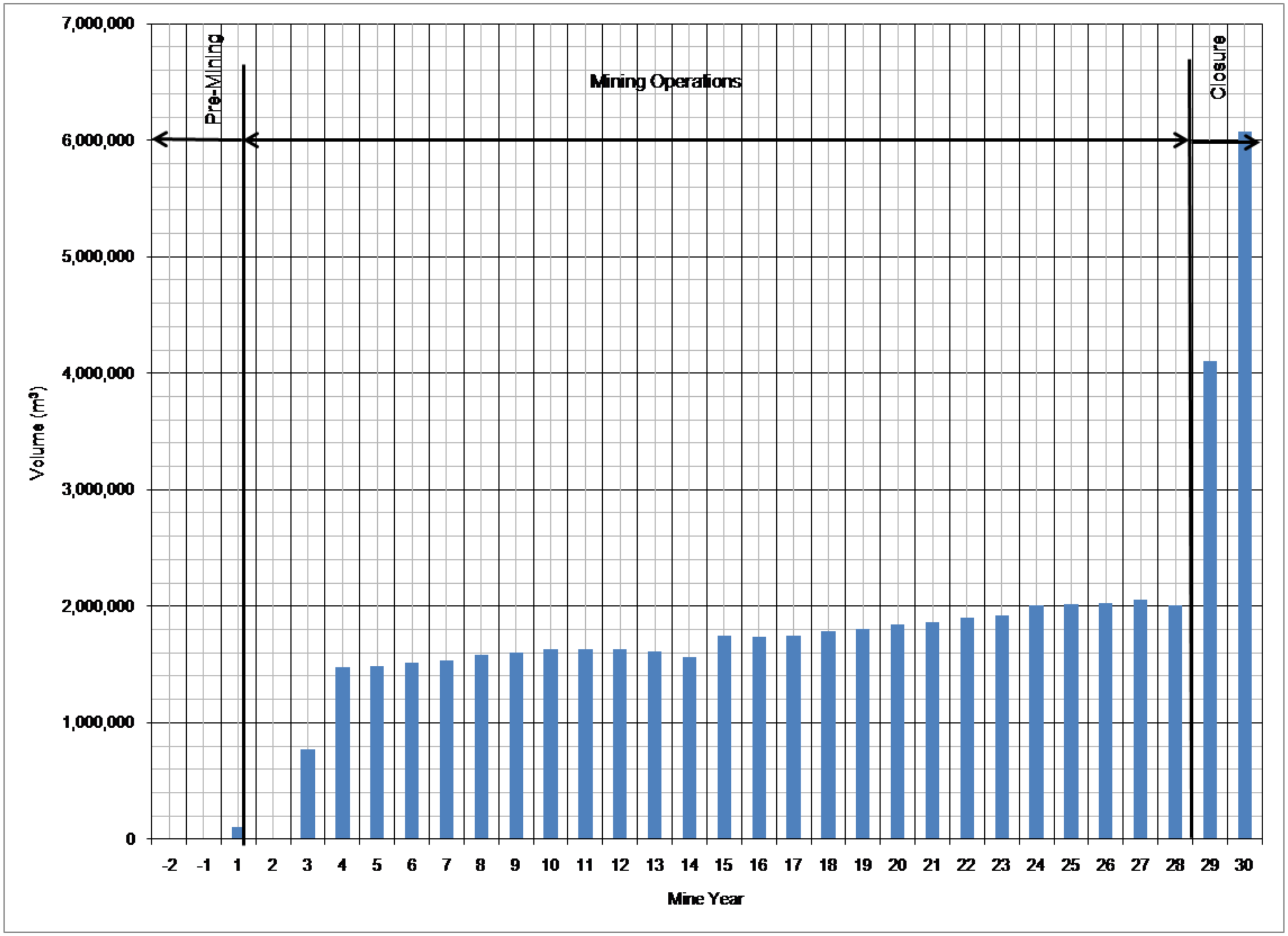


Figure 20.4 Project Annual Surplus Water Discharge Volumes



20.5 Diversion Structures and Control Ponds

20.5.1 Overview

The water management system consists of both the infrastructure and practices that are designed to manage site water. The design of the system is founded on the water management objectives, strategies and standards discussed in Section 8.1 and 8.2, and also based on sound environmental and engineering considerations. Contingencies for unexpected events and emergencies will be built into the system and an adaptive management approach will be taken in order to best respond to and address, expected changes in the mine plan and processing operations over the project life.

The sediment control plan will provide for proven management practices to control runoff water and sediment during construction and operation of the mine. The plan discusses diversion ditch and sedimentation pond design for the major mine components.

20.5.2 Sediment Control Pond Designs

This section presents the sediment control plan for the Red Chris Mine project (construction and operational phases) including design criteria for probable sediment control structures. Design drawings and finalization of the plan will require detailed engineering and will be provided in the detailed design report.

The objective of sediment control for the Red Chris project will be to provide a settling pond on main drainages receiving runoff from areas that will be disturbed during construction and operation of the mine. These ponds will also act as seepage control ponds during operations. The settling ponds will be designed with as large an area as is practical, given the steep gradients at the site. Where necessary, the main settling ponds will be supplemented by smaller scale measures, particularly along the main access road during construction. These smaller scale measures will include installation of small settling ponds within diversion ditches and localized sediment control using geotextile silt fences installed in small drainages immediately downstream of areas of active disturbance.

Temporary sediment control ponds will be established downstream of areas of development that are scheduled to take place prior to the establishment of the north basin of the TSF in Year -02. Such activities are scheduled to include the development of the plantsite and camp, crusher areas and overburden stockpile.

Small sediment control ponds may be required during construction of the TSF access road, to prevent excess sedimentation downslope of the road. Once the centreline of the road has been established, the need for such ponds will be assessed. Access road sediment ponds that are no longer required after construction will be reclaimed and revegetated.

Sediment control ponds will be established downstream of areas of tailings dam construction activities. The North and South Reclaim Dams will function as well as sediment control ponds and will be constructed prior to the main North and South Dams, to trap any sediment generated during construction. The Northeast Dam is in an area that slopes toward the TSF and therefore a sediment pond will not be required at this location. Sediment will be removed from the ponds as required to maintain adequate volume for storm runoff or at the end of the construction period when the ground has stabilized at the latest prior to operation of the TSF.

The catchment area for the North Dam Reclaim Pond will be relatively small and once the North Dam is constructed, about one-third again as small. The South Reclaim Dam, on the other hand has a relatively large catchment area. Construction of the diversion ditches prior to commencement of construction of the TSF will reduce the size requirement for the pond. The actual dimensions of the sediment control ponds will be determined on an individual basis, to provide adequate sediment control for each drainage area.

During operations, cyclone sand will be discharged to the downstream shells of the North and South Dams for placement and compaction via hydraulic fill methods. Runoff water decanted from the construction cells and draining from the sand, will report to the downstream sedimentation ponds and from there to the ponds formed by the North and South Reclaim Dams. In these ponds, suspended solids will settle out and the water will be pumped over the dams and into the TSF. Each winter, when sand is not being placed on the downstream shells of the dams, excess tailings fines that have settled out in the sedimentation ponds will be trucked back, for dumping in the TSF, to restore sedimentation capacity for the subsequent year's downstream shell construction operations.

The water quality of the drainage water from the cycloned sand construction will essentially mimic that of the process water, with some additional dilution due to runoff from the catchment, reporting to the seepage and sedimentation ponds.

20.5.3 Diversion Structure Designs

Diversion ditches will be designed to collect and divert surface runoff to carry the design event. Appropriate channel lining will be specified depending on channel gradients and velocities.

Ditches and culverts will be sized to convey the 1-in-200 year maximum instantaneous discharge with a minimum of 0.3m freeboard. The designs will provide riprap erosion protection, controlled drop structures and/or excavations into bedrock where it is required based on bed material and channel velocities; and similarly, the designs will provide for access along ditches for inspection and maintenance.

Site-specific sections depend on topography, excavation material (till or bedrock) and design discharge. Channel section designs will be based on peak storm event discharges, estimated by catchment drainage models employing an SCS Type II rainfall distribution. A minimum 3 m wide bench, typically on the downslope side, is proposed for access and maintenance, except for road ditches where the road provides access, or where steeply sloping terrain precludes this.

Different material gradations may be used for armouring depending upon the situation and availability of material sizes. On steep slope sections, rock weir drop structures are proposed. Spacing of the drops will vary depending upon the slope required; however, the top of the downstream drop should be higher than the bottom of the next upstream drop.

Diversion plugs will be required at the numerous streams intercepted by the diversion ditches. In many instances these will be excavated into the slope to effectively intercept seepage. Along the access road, stream convergences with ditches will be armoured as required to prevent erosion. As required, the downslope surfaces will also be armoured to prevent scour. Culverts will be designed for the 1:200 year return period 24-hour runoff event as per the ditches.

The layout of the proposed diversion ditches will be finalized as part of the mine water management plan in support of the detailed design of the TSF.

20.5.4 Use of Existing Drainages

Natural drainages located in the area of the proposed rock storage area will be lined with up to 5 m of NAG waste rock prior to the placement of Potentially Acid Generating (PAG) waste rock. Existing drainages will be used to the maximum extent possible. On some reaches, steep channel gradients may be required resulting in velocities exceeding safe limits for the native till material. The riprap channel armouring will be sized during the detailed engineering phase.

20.5.5 Structure Maintenance

Regular inspection and periodic maintenance following major runoff events should be anticipated for all water management structures. Periodic removal of accumulated sediments from sedimentation ponds will be required to ensure adequate storage capacity is maintained and ponds remain functional.

20.6 Site Investigations and Additional Studies

Previous site investigations have provided information regarding the soil conditions of the plateau area of the main mine development, the western flank of the TSF and the tailings basin area. An additional geotechnical site investigation program was completed out in August of 2010, to better characterize the ground conditions along the main impoundment diversions. Final ditch alignments and typical design sections for various ground conditions encountered will be developed from the August 2010 site investigation program.

20.7 Water Quality Modeling

A number of changes in the mine plan required remodelling of water quality in the Tailings TSF and receiving environment. The model was kept as close as possible to the EIA model to facilitate comparison of results for background on the EIA model. The water quality model used for this assessment and that for the EIA is a mass balance model. Mass balance models make the conservative assumption that all water quality parameters modelled are conservative, that is, do not undergo any changes from physical or biological processes along their path lengths.

Scenarios modelled during mining and closure and include:

- TSF water chemistry during all mine phases (construction through post-closure)
- Quarry Creek at the upper limit of fish migration and a baseline monitoring site where groundwater investigations suggest all seepage from the North Dam will have surfaced; operations and closure scenarios were modelled
- Kluea Lake on closure

- Unnamed creek downslope from the Northeast Dam on post-closure after open pit release to the TSF.

Results of the 2010 model scenarios are similar to the 2004 model scenarios with some increases in metals predicted but with most parameters at downstream sites predicted to be near or below CCME guidelines.

21 Project Infrastructure, Site Access and Transportation

Updates to this section from the 2005 Feasibility Study are the following:

- A temporary road from the Ealue Lake Road to the site was built in 2008 to assist in continued exploration and for project development work. For permanent use the present temporary road will either be upgraded for permanent use, or it will be upgraded and extended to join with the EA approved road from the Highway 37.
- The allowable payload for the haul trucks on highways in northwest BC has been revised from 40-43t to approximately 50 tonnes.
- This section provides only a summary for any further details please refer to the 2004 Technical Report Study.

21.1 Summary

The Red Chris property is located approximately 450 km north of the town of Smithers. The deep-sea Port of Stewart is situated about 320 km to the south. CN Rail is expected to provide freight unloading capability at the town of Kitwanga, which is located at the junction of Highways 37 and 16.

Site access and transportation options to the Red Chris property for construction, operating freight and personnel, are summarized as follows:

- Inter-Provincial road access is principally from the south by Highway 16.
- Regional road transportation to site will be by Highway 37 and a new radio controlled, restricted access mine access road.
- The Port of Prince Rupert will provide general cargo unloading and storage facilities.
- Year-round tidal access is by port facilities at the Port of Stewart. This facility will be used primarily for concentrate export to Pacific Rim based smelters.
- Access to the North American freight transportation rail network by CN Rail at Kitwanga.
- Commercial airline flights to Dease Lake, from Smithers or Terrace, BC. There is also an approximately 1,100 m gravel airstrip located 2 km north of the village of Iskut.
- Helicopters based in Dease Lake or from several helicopter landing sites along BC Highway 37 can be deployed.
- Bus transportation will be arranged to and from Dease Lake Airport and local communities.

21.2 Truck Transport

21.2.1 Concentrate

The hauling of concentrate produced at the proposed Red Chris mine will be to the Port of Stewart using B-Train style rigs.

Trucks will carry revised payload of approximately 50 t (revised from 40-43 t) of moist copper concentrate and will be loaded and weighed inside the Red Chris concentrate storage building. At the design mill feed rate of 30,000 t/d, and concentrate production will average 337 dmt/d, and will require an average of 7 to 10 trucks per day to deliver concentrate to the Port of Steward. Depending on the weather and soil conditions load restrictions may be applicable during the period from late March to mid May.

21.2.2 Fuel

Diesel fuel for mine production and maintenance vehicles will arrive by fuel contractor road tanker from supply depots in Terrace or Smithers. Average annual diesel consumption is estimated at approximately 14.7 ML/yr. This corresponds to 6 road tankers per week, each with 50,000 L capacity, to supply fuel for the mine.

21.2.3 Propane

Liquid propane (LPG) for building heating will be provided by contractor road tanker, from supply depots in Terrace or Smithers.

21.3 Road Maintenance

Mine access road from Highway 37 or from Ealue Lake Road will be maintained and operated by RCDC.

Maintenance of Highway 37 is funded by the Provincial government. Local contractors undertake proactive and effective year-round road maintenance. Examination of road usage logs going back to 1990 revealed the longest period of road closure was approximately 100 hours, for proactive avalanche control.

22 Reclamation and Closure

Updates to this section from the 2005 Feasibility Study are with respect to the relocated plant site, infrastructure, a revised footprint of the rock storage area, a revised location of the North Dam for the TSF, and the extended life of the mine from 25 years to 28.3 years. The reclamation cost estimates for the first five years and for the closure have been revised and are detailed in Section 28.

22.1 Introduction

The reclamation plan for the Red Chris project was developed to meet the requirements for a conceptual plan as set out in “Application Requirements for a Permit Approving the Mine Plan and Reclamation Program Pursuant to the Mines Act R.S.B.C. 1996, C.293”, by the BC Ministry of Energy and Mines. The current level of engineering design of the various mine components is consistent with that normal at the feasibility stage of a project.

Detailed engineering is in progress. The reclamation and closure plan presented here is conceptual in nature and deals with the mine facilities as currently proposed at the current level of project engineering. It is recognized that the degree of detail for the reclamation and closure plan will improve as the project moves through permitting and into operations.

22.2 Reclamation Objectives

Under the BC Mines Act and the Health, Safety, and Reclamation Code for British Columbia, the primary objective of the reclamation plan will be to return, where practical, all areas disturbed by mining operations to their pre-mining land use and capability. Before exploration began in the Red Chris project area, the principal land uses were wildlife habitat that supported hunting, guide outfitting, trapping, and some general outdoor recreation.

22.3 Land Use and Productivity

Mine operations will disrupt existing land uses especially in the immediate area of the open pit, rock storage area, tailings impoundment, plant site and under the proposed roadways. In some cases it will not be possible to restore current land use due to the physical changes that will occur as a result of mine development, such as in the area of the open pit where mining will permanently alter the pre-development landscape and, consequently, the land use. In other areas, restoration of pre-development land use will be possible, such as in the plant site and within the tailings impoundment although the land form will be permanently changed by the mine development activity.

There is currently no commercial forestry in the Red Chris project area. Guide outfitting activities focused on bow-hunting for Stone’s sheep take place to the southeast of the project area on the south slope of Todagin Mountain, but typically do not extend directly into the mine site and tailings impoundment areas.

22.4 Open Pit

At the end of the mine life the open pit will be allowed to naturally flood. The watershed area feeding the post-closure pit is small; consequently the amount of surface water and groundwater entering the post-closure pit is expected to be low, resulting in a flooding time of approximately 128 years. At the cessation of mining, all manmade equipment and materials such as mining equipment, piping, pumps, electrical cables will be removed from the open pit and either removed from site for their salvage value or disposed of in an approved landfill for materials with no salvage value. All equipment and materials with no salvage value will be cleaned of potentially hazardous materials before being disposed of in an approved landfill. Access into the open pit will be blocked by installation of a rock boulder barrier across the access ramp(s) into the pit.

22.5 Rock Storage Area

Over the mine life it is projected that a total of 408 Mt of waste rock and 36.7 Mt of low-grade ore (LG) will be placed into the rock storage area for permanent storage. Of this total, a limited amount is expected to be NAG rock. The remainder is expected to be PAG with an estimated time to onset of acid generation in the order of several decades. NAG rock will be incorporated into the base and toe areas of the storage area to minimize direct contact of PAG rock with near surface runoff and water reporting down the storage area slopes. A till cover will then be placed over top of the rock storage area to provide an infiltration barrier. The cover would then be seeded and fertilized to provide a self-sustaining vegetative cover.

22.6 Tailings Impoundment

At mine closure, the proposed tailings impoundment would be reclaimed to provide a central water pond with wide NAG beaches upstream of the North and South Dams, and a permanent spillway around the Northeast Dam. In this manner, the potentially acid generating component of the tailings would remain water-saturated in perpetuity to prevent sulphide oxidation, while above water NAG beaches keep the water pond away from the North and South Dams to improve the long-term stability and safety of these dams. The central water pond will be designed to provide a water cover over top of all of the PAG tailings stored within the proposed tailings impoundment.

22.7 Processing Plant

At mine closure, all stockpiles of ore adjacent to the crusher will be removed and processed through the mill. Similarly, the crushed ore stockpile and underlying dead stockpile base load will be dozed into the reclaim tunnel feeders and processed through the mill. Once all of the ore and low-grade ore stockpiles have been milled, the process plant will be decommissioned and buildings and equipment removed from the site.

22.8 Other Infrastructure

At mine closure the accommodation buildings, assay lab, heavy equipment maintenance shop, explosives manufacturing plant, fuel farm, power distribution system, warehouse, and office building will be removed, the site re-graded and re-vegetated as will the landfill, land-farm and borrow areas.

22.9 Water Courses

All culverts will be removed from road crossings and water courses will be returned to their natural channels to the extent practical, or new, stable water courses established. All crossings will be back-sloped and rip-rapped as appropriate to minimize potential for erosion. All ditches and channels will be designed to pass the appropriate flood flows consistent with their purpose and long-term closure considerations.

22.10 Progressive Reclamation and Research

RCDC will reclaim areas where mining and associated activity has been completed wherever possible during the mine operating life. RCDC will conduct an ongoing reclamation research program during the mine life.

23 Project Execution Plan

23.1 Summary

Red Chris mine's development is contingent and dependent on the construction of the NTL project for which BC Hydro has retained a Design-Build Contractor.. According to BC Hydro the NTL target date for completion and becoming operational at Bob Quinn is May 2014. RCDC will continue monitoring the NTL's approval and construction schedule.

Detailed engineering and procurement (EP) work is in progress with AMEC. RCDC's own construction management (CM) team is proposed for the project. Pre-engineered buildings and modular construction will be used where possible, to accelerate construction and reduce on-site labour costs.

23.2 Project Management

An EP management team, headed by a Project Manager, will be responsible for detailed engineering, procurement, and the preparation of both a control budget and a project execution schedule. The Owner's CM team, headed by a Construction Manager, will run the project and will be responsible for safety, environmental compliance, budget planning scheduling, logistics, and project organization.. The team leader will be located initially at the design offices, and will relocate to the construction site as development begins.

Both the Project Manager and the Construction Manager will report to an appointed RCDC representative, who will also be located initially in the design offices and then at the project site.

23.3 Project Management System

The EP and CM teams will employ a RCDC approved project-proven computer-based project management system. Effective decision making on a project is predicated on the availability of timely and accurate information.

23.4 Engineering

The in progress detailed engineering follows the concepts and mine plan developed in the 2005 Feasibility Study and this Update. Initially, a basic engineering phase will be executed, to review, confirm and freeze key design documentation. A comprehensive mine plan has been developed as part of this study. RCDC operations personnel will undertake any remaining mine design as part of the overall mobilization, pre-strip, commissioning, and start-up activities. There are three principal areas of detail design:

- ore processing systems
- on-site infrastructure systems and ancillary buildings
- external infrastructure systems including access road and power supply.

23.5 Design Standards

Mine systems will be designed to BC and North American standards and include a maximum of pre-assembly and modularization of components.

The project will be designed on the Metric (SI) system of units.

23.6 Procurement and Logistics

Equipment and materials for the Red Chris project will be sourced on a world-wide basis. Procurement personnel and design leads will develop a bid list based on the initial equipment selections made for this study. A procurement plan will be put in place detailing what equipment will be tendered long-lead equipment, critical vendor information, and equipment that will require special inspection and expediting services. The procurement plan will tie all purchases to the master implementation schedule to ensure that critical delivery items are ordered in a timely manner.

Equipment and materials for the Red Chris project can be transported to site either by road via the North American road system, by ship from the Ports of Prince Rupert or Stewart, or by rail to the siding at Kitwanga, and then by road.

23.7 Safety, Health, and Environmental Management

In choosing the EP and building the Owner's CM teams RCDC conducted rigorous pre-qualification reviews to ensure those invited to submit final proposals have proven Safety, Health, and Environmental (SHE) programs and cultures. Submission of satisfactory construction safety statistics will be mandatory for pre-qualification assessment.

The contractors will develop and implement a health and safety program that will be integrated into all work practices. The health and welfare of employees will be a top management priority.

The contractors will support the project's environmental program during construction and develop a site-specific environmental plan for the construction phase, which satisfies RCDC & Imperial's corporate goals and requirements.

23.8 Construction

Several plants of this type have been constructed in British Columbia and a number of contracting companies with this type of experience are likely to bid on this work. The Red Chris mine will be built under standard Canadian northern climate conditions. Many of the ancillary buildings will be pre-fabricated and trucked to site to reduce costlier on-site construction activities.

The existing temporary road from the Ealue Lake Road will be used for commencing the on-site construction. Likewise use of some of the existing RCDC owned 60 person exploration camp will be made available for the early construction crews. The temporary road will either be upgraded for permanent operational use or it will be upgraded and extended to the EA approved route from Highway 37.

23.9 Construction Camp

The camp will consist of pre-assembled self-contained units, with electricity provided initially by the project's emergency generators. A bus/truck service will be established and operated to transport personnel, supplies, and light materials to site.

23.10 Mine Pre-production Development

RCDC staff and equipment will be used for all activities relating to mine pre-production development. The Construction Manager and RCDC's Mine Superintendent and Environmental Coordinator will co-develop a mining implementation plan for inclusion in the overall Project Execution Plan.

23.11 Commissioning and Start-up

A commissioning team will handle commissioning and start-up with assistance from major equipment suppliers, such as SAG and ball mill manufacturers. The team will include representatives from both the EP and Owner's CM teams and RCDC operations personnel (plant area operators and lead maintenance staff). The commissioning team will assist in familiarizing the RCDC personnel with the unit operations and types of equipment that will be in the plant.

RCDC's production supervisors and plant operators will play a key functional role in plant start-up. It will be necessary for key operations personnel to be hired well before the end of construction, to provide adequate time for orientation.

23.12 Training

RCDC will establish a training program for mine and process equipment operators. The EP and CM contractors will provide simulated process control and plant operating training using the actual PLC/HMI hardware and software purchased for the project. Manufacturers' representatives will provide journeymen specialist training in the preventive maintenance and repair of major equipment items.

In conjunction with RCDC, the EP and CM contractors will also prepare and assemble operator and training manuals, both web-based and conventional.

23.13 Project Implementation Schedule

The development of the Red Chris mine is contingent on the construction of the NTL project. Now that EA Certificate for the NTL has been approved and BC Hydro has retained the NTL Design-Build Contractor the target completion of the NTL project is anticipated to be operational by May 2014. RCDC will continue monitoring the NTL's construction schedule and plans to complete mine development to coincide with the powerline completion. Based on completion of NTL by May 2014, RCDC proposes the following permitting, development, construction and operation schedule, (Table 23-1)

Table 23-23.1 Project Schedule

Activity	Target Date
Apply for the Mines' Act Permit	July 2010
Request DFO to initiate MMER Schedule 2 Process	June 2010
Feasibility Study Update	November 2010
Site Clearing for the Plant Site, Mobile Equipment Shop Area and the Camp Area	Early 2012
Commence Detailed Engineering and Procurement	November 2010
Complete MMER Schedule 2 Process and Obtain DFO Authorization for TSF construction commencement	2012
Commence Foundation Preparation for the North Dam (TSF)	Summer of 2012
Commence Major Construction at Red Chris	Early 2012
Commence Construction of Power line Extension	Late 2012
Complete Construction	April of 2014
Power-line hook-up at Bob Quinn	May 2014
Commission and Start-up	June 2014
Production Start-up	Mid 2014
Production Operation	2014 to 2041
Commence Closure and Reclamation	Early 2042
Complete Closure and Reclamation	Late 2043
Post-closure Monitoring and Surveillance	2043 to 2048

24 Environmental Studies and Permitting

24.1 Environmental Assessments

The Red Chris BC Environmental Assessment Act (BCEAA) Certificate (M05-02) was issued in August 2004. The certificate extension was obtained on July 9, 2010.

Federal approval for the Red Chris project under the Canadian Environmental Assessment Act (CEAA) was received in May 2006. The Federal approval was subsequently challenged by a third party, however a subsequent decision by the Supreme Court of Canada on January 21, 2010 upheld the Federal approval which has allowed mine permitting and development to proceed.

24.2 Permitting

RCDC submitted its draft Terms of Reference for the Mines' Act Permit Application in October 2009 and met with the Northwest Mine Development Review Committee (NWMDRC) in December 2009 and in March 2010 when the Terms of Reference (TOR) were finalized.

The BC Government is in the process of streamlining the overall permitting process and is proceeding with asynchronous permitting approach. The new system will have all permits required from the various ministries of the Province to be synchronized with the Mines Act Permit Application.

On the basis of finalized TOR, RCDC submitted a joint Mines' Act and Environmental Management Act Permit Application on July 23, 2010 and met with the NWMDRC on September 9, 2010 to present the Joint Application and also to get initial review comments. All comments are being collated for RCDC's response. Several Working Group Committees have been formulated to address key areas such as Fisheries, Wildlife, EMA related Permit Applications, and Archaeology. Responses to received comments from various agencies were submitted in June 2011. Considering the proposed coordinated permitting approach by the NWMDRC it is anticipated that all permits for project development and construction will be obtained by the spring of 2012.

The following permits, approvals, licenses and leases will be required for the development, operation and closure of the Red Chris mine:

BC Ministry of Mines, Energy and Petroleum (MEMPR)

- Mines' Act Permit
- Permit pursuant to *Mining Right of Way Act*
- Mining Lease

BC Ministry of Environment (MOE)

- Effluent Discharge Permit, Refuse, Air, Special Waste, Sewage Registration
- Water License(s)
- Permit pursuant to *Wildlife Act*

BC Ministry of Forest Resources (MOFR)

- Occupant License to Cut
- Special Use Permit (for the access road segment off the mineral claims)

- Burning

BC Integrated land Management Bureau (ILMB)

- License of Occupation (for the power line off the mineral claims)

BC Ministry of Transportation and Infrastructure (MOTI)

- Permit to Connect to a Public Highway
- Permit to install the power line if located in a road right of way

TCA, Archaeology Branch

- Alteration Permit (Section 12, Heritage Conservation Act) OR Systematic Data Recovery (Section 14, HCA) in some areas

Northern Health Authority

- Permits pursuant to the Health Act, Food Premise Regulation, Industrial Camps Health Regulation
- Permit pursuant to *Drinking Water Protection Act*

Federal Authorizations

- DFO – Federal *Fisheries Act* Authorization
- EC – MMER Schedule 2 Designation for the Tailings Impoundment Area
- Rican – *Explosives Act* Authorization (for explosives manufacturing)

24.3 Fisheries Habitat Compensation Plan and Review Process

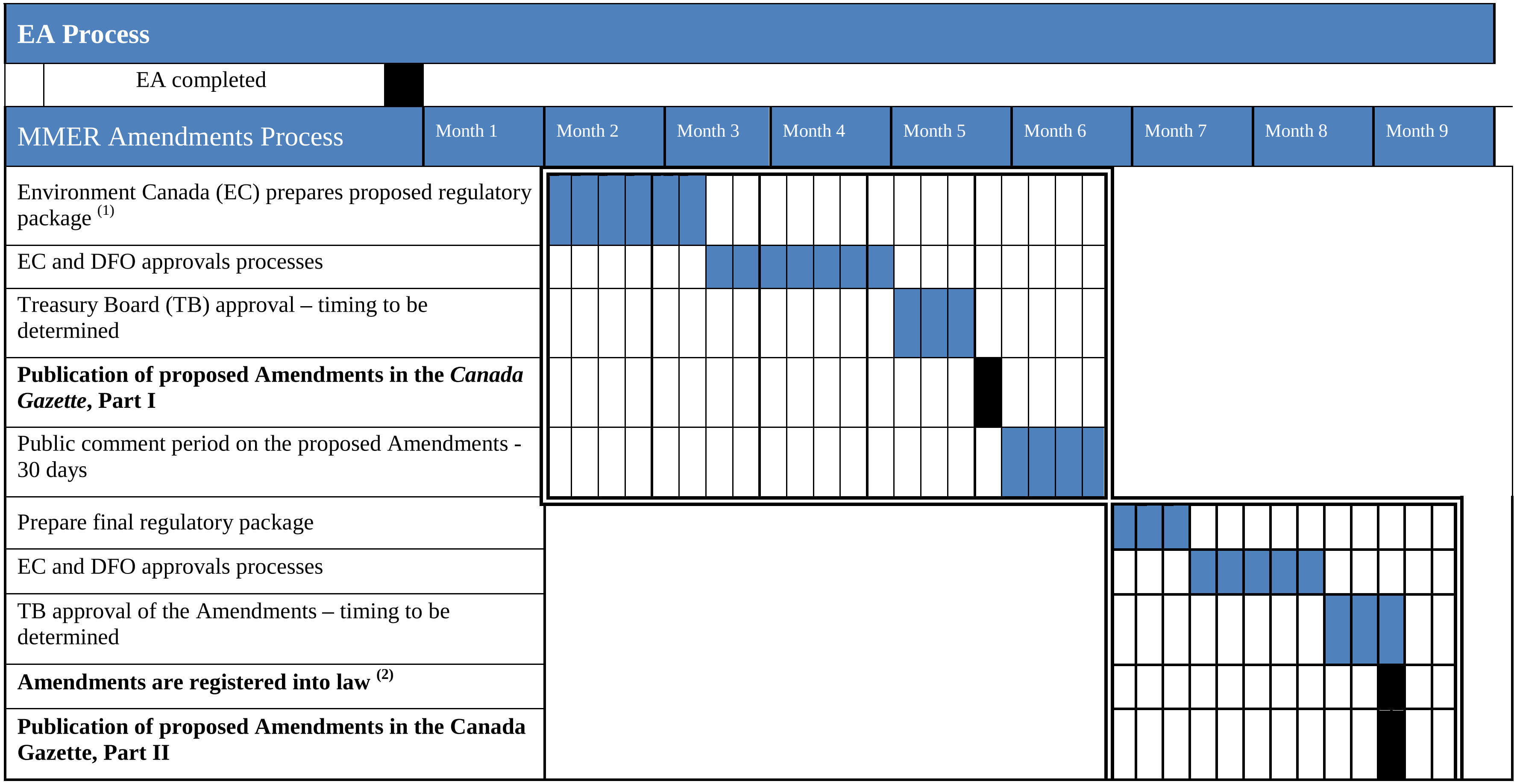
A detailed Fisheries Habitat Compensation Plan (FHCP), on the basis of the earlier approved FHCP on the Coyote Creek, was prepared and submitted to the DFO in May 2010. Review comments received suggested that the plan be split into two parts, one to address the loss of fish habitat in Trail Creek resulting from the construction of the TSF and a second addressing the potential impact on Trail Creek downstream of the TSF due to the potential for reduced flows in the Trail Creek after the construction of the TSF. As a result, separate plans to address each of these issues will be prepared.

These two plans were completed and submitted for comments, which resulted in considering some other options for providing the required fisheries habitat compensation. Snapper Creek on Highway 37 has two culverts which appear to act as major fish barriers for any fish habitat upstream of the culverts. The fisheries habitat compensation plan consists of replacing these two culverts by a single clear span bridge. This plan has evolved with joint consultation with the local Tahltan First Nation and has also been agreed to by the Ministry of Transportation and Infrastructure (MoTI). The FHCP has been submitted to the DFO for its review. The DFO will consult with the local First Nations as part of its review. If the DFO is satisfied, it will recommend to Environment Canada that Environment Canada initiate the Schedule 2 Designation Process so that Tailings Impoundment Area can be designated as a Tailing Impoundment Area (TIA) under the Metal Mines Effluent Regulations (MMER).

24.4 Schedule 2 of the MMER Process

Timelines for the Schedule 2 designation process are as shown in figure 24.1.

Figure 24.1Amendments to the *Metal Mining Effluent* Regulations (MMER)



NOTE: Targets are achievable under ideal circumstances. Delays may occur when the Treasury Board is not in regular session, e.g., during the summer or during a federal election.

25 First Nations

25.1 Property Location

The Red Chris property is located in a sparsely populated area of the northwest region of British Columbia, and within the asserted Tahltan Nation traditional territory. Two First Nations bands, as recognized by Indian & Northern Affairs Canada, each with an elected council, represent the Tahltan people. The Tahltan Band has its office in Telegraph Creek and the Iskut Band has its office in Iskut. The Tahltan Central Council (TCC) represents both bands on issues of joint concern and is comprised of Chiefs of both bands as well as representative of the traditional families. The TCC has its office in Dease Lake.

It is understood that the Tahltan First Nation is not negotiating in British Columbia's treaty process, but according to the Government of British Columbia, the Province is currently engaged in negotiations with the TCC at a reconciliation table. The work of this table deals with a variety of resource sectors including mining, oil and gas, and land use planning.

25.2 Tahltan and Nearby Communities

The communities in the area are Dease Lake, Telegraph Creek, Iskut, and Stewart

Dease Lake, Iskut and Telegraph Creek communities are considered remote. Dease Lake ,Iskut and Telegraph Creek are within the Kitimat-Stikine Regional District. The District Municipality of Stewart (SDM) is included in the study as concentrate from the proposed mine will be trucked to the Port of Stewart for shipment overseas. SDM is within the Kitimat-Stikine Regional District.

They communities within a corridor 100 km wide centred on Highways 37 and 37A, starting at Stewart and ending at Dease Lake, and including Telegraph Creek. Highway 37 is the major transportation corridor south to the rest of the province and north to Yukon. Dease Lake is served by regularly scheduled air service during the summer. Dry goods, such as groceries, fuel, and mail, are transported to the region by road.

25.3 Dease Lake

The community of Dease Lake is located on Highway 37, 83 km north of Iskut and 234 km south of the Alaska Highway. It is also located at the junction of a 119 km gravel road that leads to Telegraph Creek.

Dease Lake is the largest settlement on Highway 37 and is the service and government centre for the residents of the region. It has food and accommodation, automotive, health, banking and other services. It also serves as a staging area for wilderness tourists visiting the parks in the area. A modest airport with a paved runway exists near to the community (IATA: YDL, ICAO: CYDL). The facility is capable of handling smaller commercial aircraft and a scheduled service from Smithers is operated in spring and summer months.

Tahltan Reserve No. 9 is located in Dease Lake as are the offices of the Tahltan Nation Development Corporation (TNDC) and the TCC. Four ministries of the British Columbia Government are represented in the community. They include the Ministry of Forests, Mines & Lands, the Ministry of Citizen Services, the Ministry of Transportation, and the Ministry of Children and Family Development. The Ministry of Citizen Services provides access to all British Columbia Government programs such as insurance, licenses, social assistance, and permits, and acts as the mineral claims recording office, accepts payments for phone and hydro, and serves the communities of Lower Post, Good Hope Lake, Telegraph Creek and Iskut (Jacobs, Pers. Comm., 2004). Other public offices include School District No. 87, Northern Lights College, an RCMP detachment, a health clinic, a British Columbia Ambulance Unit, and, during the summer, a provincial forest fire fighting base.

Although Dease Lake is unincorporated and does not belong to any regional district, its residents are currently considering some form of community governance, in part because the Ministry of Forests, Mines & Lands is understood to be withdrawing funding for community fire protection.

Recreation facilities include a community hall, a public library, an outdoor skating rink, and a school gym.

25.4 Iskut

Iskut is located 83 km south of Dease Lake and 163 km from Eskay Creek mine and 18 km from the Red Chris mine site. The majority of the approximate 330 population of Iskut are members of the Iskut First Nation. Iskut has a post office, gas station/grocery store, a school, fire station, community health facility and a Band office. The community serves as a staging area for the Mount Edziza, and Spatsizi Plateau wilderness parks.

25.5 Telegraph Creek

This Tahltan community of Telegraph Creek is situated beside the Stikine River at the foot of the Stikine River Canyon. Telegraph Creek is a remote community, located 112 km. from Dease Lake. Most of the Tahltan reserves are located around Telegraph Creek (north and east toward Dease Lake). The population of Telegraph Creek is about 350. Telegraph Creek has a Nursing Station, while the nearest doctor is in Dease Lake. The nearest full service hospital is in Terrace, over 700 km south by road.

The main industry of Telegraph Creek is guiding, with some mining exploration. It’s the only town on the 600-km long (372 miles) Stikine River. Telegraph Creek has an airstrip, store and café, post office, police station, school, church and a small museum.

25.6 Regulatory Process

The Province is obliged to consider First Nations’ interests in relation to an environmental assessment and development and operational permitting. The Province, including the Environmental Assessment Office (EAO) and regulatory agencies, consults with First Nations whose interests may be affected by a proposed project to ensure First Nations issues and concerns are identified, and that adequate efforts are made to address those issues and concerns.

The project proponent is also required to undertake consultations with First Nations. The EAO monitors and evaluates the proponent’s consultations and may direct the proponent to undertake further measures.

The EAO and regulatory agencies may provide funding to assist First Nations to participate in the environmental assessment process and in the permitting process.

25.7 First Nation Traditional Land Use in the Project Area

The Red Chris property lies within the area claimed by the Tahltan people as their traditional territory. The total area of the Tahltan traditional territory in northwest British Columbia covers some 93,600 km2. The Tahltan people have historically traveled extensively throughout their territory in search of migratory game and other wildlife, and to trade with other native groups.

Iskut Band members' land use in the proposed mine and in adjacent areas includes camping, hunting, access trails, plant and berry harvesting and trapping. It is also reported that these lands contain an important migratory route and wintering habitat for the various animals hunted and trapped by the Iskut band members, particularly in areas to the west and northwest of the mine site.

The local people are also active users of the Stikine Country Protected Areas and are interested in commercial opportunities within the protected areas systems. The communities of Iskut, Telegraph Creek, and Dease Lake are the main staging areas for visitors accessing the Stikine Country Protected Areas.

25.8 Subsistence Economy

Hunting, trapping, and fishing are important activities for recreation and the economy in the study area and are an integral component of the economic, social, and cultural life of First Nations and non-aboriginal residents. Many residents hunt, fish, and gather berries. The Tahltan also gather plants for medicinal uses. Generally wild game provides the bulk of the diet for Iskut families and hunting continues to be the most important subsistence activity.

It is understood that members of the Iskut Band regularly hunt moose, caribou, sheep, goats, and ground hogs in the area of the mine site. It is understood to be a bow-hunting area for non-aboriginals.

Subsistence fishing is extremely important for members of the Tahltan Nation in Telegraph Creek, as it provides a significant amount of their food. Families have particular family fishing sites and camps along the banks of the Stikine where they go to catch and preserve their catch.

25.9 Native Land Claims

The Tahltan territories include a large section of northwest British Columbia, with approximate borders of Alaska to the west, Treaty Creek to the south and Yukon border to the north. Tahltan traditional areas include the region encompassed by the entire drainage basin of the upper Stikine River, and headwaters of the streams that flow into the Taku, Nass, Skeena, and Yukon rivers. The total area of the Tahltan land claim in northern British Columbia is 93,600 km2. The first formal declaration of sovereign right to all territories occupied by the Tahltan First Nation was in the "Declaration of the Tahltan Tribe signed at Telegraph Creek, British Columbia, [the] eighteenth day of October, nineteen hundred and ten [1910], by: Nanok, Chief of the Tahltans; Nastulta (alias Little Jackson); George Assadza, Kenetl (alias Big Jackson); and eighty other members of the tribe."

An overlap of land claims between the Tahltan First Nation and the Taku Tlingit First Nation exists in the northern portion of the territory. It is understood that land claim boundaries to the south of the Tahltan territory have recently been settled by the Nisga'a.

25.10 First Nations Consultation, Memorandum of Understanding and Agreements

bcMetals, the previous owner of the Red Chris property initiated contact with the Tahltan Band Council and Iskut First Nations prior to the bcMetals becoming publicly traded and prior to the onset of site exploration activity in September 2003. Introductory meetings outlined the bcMetals’ plans for the upcoming 2003 field season as well as for longer-term development of the property.

Contact with the First Nations groups has continued on a regular basis since that time with the focus of discussions being on Tahltan involvement in the project and the mitigation of potential environmental and land use consequences, including the current use of lands and resources for traditional purposes, associated with development activities. An extensive log of these of consultations/contact with the Tahltan was provided in the Environmental Assessment Application and also in the 2005 Feasibility Study.

Discussions with Tahltan and Iskut leaders resulted in signing a Memorandum of Understanding (MOU) on 19 January 2004. The MOU outlines a set of principles under which the company and the First Nations will work together in the development of the Red Chris mine and protection of the environment. The MOU foresees the parties working towards a more comprehensive Participation Agreement. A Negotiations Agreement was signed between Tahltan First Nation and bcMetals to finalize the terms for the Participation Agreement. These terms were finalized in September 2006. bcMetals also held open houses in the three Tahltan communities. Public reaction to the project at the open houses was predominantly positive. Public comment from outside the local communities has been limited to date.

After acquisition of bcMetals by Imperial, Imperial has honoured all the earlier concluded MOU and Agreements. RCDC signed a Traditional Knowledge (TK) Agreement so that this knowledge could be applied to additional Archaeological and other studies after the EA Certificate was obtained.

Several meetings with Tahltan representatives have occurred since 2007. At the direction of the Tahltan, Participation Agreement related negotiations are now in abeyance.

At the direction of the TCC in June 2010, RCDC provided funding for facilitated internal meetings within the three communities so that Tahltan concerns, needs, desires could be well defined. As a follow-up of these internal meetings RCDC provided an organized site tour to many residents of Telegraph Creek, Dease Lake and Iskut on September 13 and further followed up with facilitated meetings with the Tahltan in all the three communities on September 14th to 16th 2010. Further several community consultation meetings were held in Iskut, Telegraph Creek, Dease Lake, Terrace, Smithers, Prince George, Fort St John and Vancouver. Additional meetings are expected as suggested by TCC.

RCDC is currently providing considerable capacity funding to the Tahltan socio cultural and environmental agency, Tahltan Heritage Resources Environmental Assessment Team (THREAT), so that THREAT may actively participate in the NWMDRC. Funding is provided in increments as work is carried out.

RCDC continues to act as a “good neighbour” by supporting a range of community events and needs within the local First Nations community.

Most recently RCDC, through Imperial Metals, is supporting a Tahltan led process to create an effective list of resident and non-resident First Nation members in an effort to ensure all Tahltan, regardless of location, are properly informed of activity on Tahltan traditional territory. This work is ongoing.

25.11 Revenue Sharing by the Province

In October 2008, the Province authorized its Provincial negotiators to include revenue sharing with First Nations on new mining projects. Revenue Sharing Agreements for two new mining projects were concluded in the Province in 2010. Revenue sharing is agreed on a project by project basis. For concluded agreements sharing was up to approximately 37.5% of the Minerals Tax revenue by the Province. For the purposes of Revenue Sharing Red Chris will be considered as a new mining project when it goes into production. The Integrated Land Use Management Branch (ILMB) (now incorporated into the Ministry of Aboriginal Affairs and the Ministry of Natural Resource Operations) has informed RCDC that it will be beginning discussions with the Tahltan for the purposes of reaching agreement in this matter.

26 Socio Economic Impact

26.1 Social Setting

26.1.1 Population Trends

The combined population of the communities and rural areas with the study area (the unincorporated region of Stikine) was approximately 1,130 in 2006, of which 44% was Aboriginal. The population is currently estimated at 1200. The region has experienced net population outflows for the past twenty years and the population is aging. BC Stats, the provincial government statistical agency, projects population growth in the region is likely to be moderately positive for the next 15 years primarily due to in-migration. On the other hand, the closest community to the project, Iskut First Nation, has seen considerable relative population growth, increasing from 255 to 330 persons from 1996 to 2006, a 29.4% increase.

26.1.2 Health Indicators

Health indicators in the region indicate a slightly less healthy population as compared to British Columbia averages. Life expectancy for males and females is approximately three years less than provincial averages. Infant mortality per 1000 persons is 4.9 for the region as compared to 4.2 for the province. Cancers, circulatory diseases, respiratory diseases, unintentional injuries and suicides all compare unfavourably to provincial averages. Access and use of physicians is comparable to provincial averages.

Health Services are highly dispersed due to the small population and large area. The nearest health facility to the mine site is Iskut Valley Health Services (IVHS) located in the community of Iskut. IVHS provides the services of a community health nurse, a home care program, mental health program, basic dental services, patient travel services and walk-in health services.

A larger medical facility, the Stikine Health Centre, is located in Dease Lake while the nearest hospitals are the Bulkley Valley District Hospital in Smithers and the Mills Memorial Hospital in Terrace. Both approximately seven hours drive from Iskut.

26.1.3 Level of Education

Levels of education attainment are generally lower than provincial averages. According to the most recent census (2006), 17.8% of the population aged 25-64 does not have a high school diplomas, certificate or degree; 19.4% have only completed high school. Only 20% of the population has a university degree compared to an average of 30% for the province. Conversely a high percentage of region residents, compared to provincial averages, have apprenticeships, trade or college certificates and diplomas at 42.6%.

26.1.4 Transportation and Communications

Two main highways serve the area, Highway 37, often referred to as the Stewart-Cassiar Highway, is the main route in the region and runs from the junction of Highway 16 in the south to the Alaska

Highway in the north, near the Yukon community of Watson Lake. From Highway 37 at Meziadin Junction, Highway 37A extends westward to the coastal port community of Stewart in Canada and Hyder in the Alaska panhandle.

There are airstrips in the areas which are open year-round and regularly maintained. The first, Dease Lake Airport, (IATA: YDL, ICAO: CYDL), is located 1.5 NM (2.8 km; 1.7 mi) south of community of Dease Lake and is operated by the Dease Lake Airport Society. The second is the Bob Quinn Lake airstrip, (IATA: YBO, TC LID: CBW4), located 191 km south of Dease Lake. The Bob Quinn airstrip is unpaved.

Northern Thunderbird Air currently provides limited scheduled services from Smithers to Dease Lake in spring and summer months. Helicopters services are provided by Pacific Western Helicopters and others.

Land based telephone service in the communities is provides by Northwestel, which operates throughout northern British Columbia. Mobile telephone services are not available in the region except through local wi-fi. Northwestel also provides high and low capacity digital radio in Dease and Iskut respectively as well as a satellite earth station in Telegraph Creek. All communities have access to high-speed internet.

26.1.5 Housing and Health Services

Housing is considerably more affordable in the region. Approximately 10% of rental housing consumes over 30% of household income (a provincial measure of affordability). Province wide over one-third of rental housing consumes this level of income. Eight percent of owner occupied housing in the region consumes 30% of household income compared to 23% province wide. Monthly rent in the region for 2006 was $579 compared to a provincial average of $828. Value of dwellings was $157,605 compared to a provincial average of $418,703. 48% of dwellings are owned and 37% are rented. Approximately 16% of all housing in the Stikine region is “Band Housing” in that it belongs to an Indian Reserve.

26.2 Land Use and Land Tenure

26.2.1 Land Use

Current land use around the proposed project area consists of backcountry wilderness tourism and recreation, hunting, trapping, guide-outfitting, and mineral exploration. The existence of the Red Chris mineral claims are recognized within the Long Range Management Plan (LRMP) for the 131,000 ha Todagin Area Specific Resource Management Zone. Mineral exploration, mine development, and associated access continue to be recognized within the LRMP as appropriate activities.

26.2.2 Land Tenure

The Red Chris Property is comprised of the Red Chris Claims and the Red Claims. The Red Chris claims consist of 50 mineral claims covering 10,218 hectares and the Red Claims are 18 mineral claims covering 7,501.51 hectares, with a combined total of 17,719 hectares, located in the Liard Mining Division of northwest British Columbia.

There are three private lots and two residential leases on the north side of Ealue Lake. In addition to Ealue Lake, there are several other land tenures on crown land surrounding Tatogga, Eddontenajon, Kinaskan, Todagin and Kluea Lakes near the Red Chris project.

26.3 Economic Setting

26.3.1 Regional Economy

The main economic activities of the area are mining and forestry. Considerable mining exploration has taken place in the region in recent years and is resulting in spin-off support and service sectors. Forestry activity does exist but it is restricted as there are no processing facilities in the region. Some commercial fishing licenses are held on the Stikine River but these too are limited.

Tourism opportunities, particularly related to back-country tourism and fish & game, do exist, but tourism revenues have dropped off considerably from a high of over $11 million in 2007 to approximately $9.8 million in 2009.

The Cassiar Iskut-Stikine LRMP states that the area is endowed with provincially to globally significant mineralization and rich energy values. The region has had several productive operating mines, most recently Eskay Creek which closed in 2008 and boasts over 30 developed prospects with proven geological reserves and additional geological units. It is one of the most attractive areas for mineral exploration and development in the province. Of $154 million in exploration spent in British Columbia in 2009 some $65 million was spent in the northwest.

26.3.2 Employment

Unemployment rates as of 2006 were 10.1% although current estimates place the rate at approximately 11.1%. Unemployment for males under the age of 24 is estimated to exceed 60%.

Aboriginal unemployment is generally considered higher across the province, but labour force participation rates by the Iskut First Nation and the Tahltan Band compare favourably. Iskut labour force participation by population over 15 years of age was 70.2% in 2006, while Tahltan Band participation was 58.7%. This compares to a 57.1% provincial average for on-reserve aboriginals and a 65.6% participation rate for the provincial population as a whole.

26.3.3 Trained Workforce

Several mines, such as Eskay Creek Mine, Snip Mine, Golden Bear Mine and earlier Cassiar Mine operated in the Tahltan traditional territory. A number of experienced and qualified operators from these mining operations are available within the three traditional Tahltan communities. As indicated in the section on Socio-Economic factors, a higher than average percentage of the region's population have apprenticeships, trade and college certificates and diplomas, indicating an industrial workforce.

26.4 Corporate Policy

RCDC management will actively recruit employees from the local area. When Eskay Creek was operating some 35% of the workers at the mine were Tahltans.

As per the Memorandum of Understanding with the Tahltan Band Council, the Iskut Band Council and the Tahltan Central Council, RCDC will source qualified employees from the local First Nations communities where possible. RCDC and the Tahltan share a common goal of maximizing employment opportunities for Tahltan members at the project, subject to skills and suitability.

RCDC will encourage its contractors to follow the same philosophy with regard to recruitment, training, safety, and environmental responsibility.

Typically, contracts for goods and services, required in association with mine development and operation, will be tendered using industry accepted practices with award made based on price, quality, value, availability, and schedule. Letting of contracts for the provision of materials, labour, equipment, services, or construction will be conducted in a competitive, businesslike, morally, and ethically responsible manner. Tahltan businesses will enjoy competitive access to such economic opportunities provided always that RCDC is satisfied that the terms of the contract are competitive in all respects as to price, delivery, capability, performance, and quality.

RCDC will not permit its employees or contractors to carry unauthorized firearms while at work for the company. Employees will not be permitted to hunt on the property. Possession or use of alcohol and drugs will be grounds for dismissal.

27 CAPITAL COST

27.1 General

In 2010, Merit was contracted and retained by RCDC to provide a capital cost estimate within a 15% accuracy. Merit updated capital costs to include the impact of plant site and tailings management facilities relocation, other changes of scope as well as the following additional pricing provided by RCDC and other contributors:

- Mine Pre-production – Pricing by RCDC
- Mine Equipment - Pricing by RCDC
- Overland Conveyor – Pricing by local fabricator
- Access Roads - Pricing as provided by RCDC
- Site Substation - Pricing by Merit
- RCDC's Costs - Pricing by RCDC

The capital cost estimate has been updated for the following items:

- Pre-production mine work costs as provided by the RCDC.
- Mine fleet sizes and actual equipment costs as provided by the RCDC,
- Shop tools allowance based on 3% of total mine equipment cost has been added.
- Overland conveyor and reclaim conveyor:
- Conveyor 210-CV-201 Stockpile Feed was deleted and replaced by the new overland conveyor.
- Conveyor 220-CV-202 sag mill feed was deleted and replaced by the new reclaim conveyor.
- Civil/concrete works costs were updated using prorated quantities from original estimate.
- Conveyor access road allowance has been added.
- New conveyors equipment supply costs were updated with pricing provided by local fabricator. Installation costs were also updated.
- Electrical and Instrumentation allowances have been updated accordingly.
- Access Road costs as provided by the RCDC and priced by a local contractor.
- Dam construction new quantities as provided by AMEC were included in the Capex and estimated using our revised unit rates provided in December 2009. A 15% increase has been added to some of the units to cover for extra hauling distances.
- Tailings site construction roads and development of borrow area allowance has been added.
- Outfall structure allowance has been added.
- Seepage pumps at northern dam have been updated.
- Seepage pumps at south dam have been deleted.
- Monitoring wells allowance has been added.
- Plant Mobile Equipment list as provided by the RCDC and pricing updated by Merit.
- RCDC's costs updated.
- Indirect Costs were analyzed and adjustments made accordingly.

- Contingency was adjusted accordingly and, as indicated by RCDC, a 10% contingency has now been considered for all estimated costs.

The total estimated cost to design and build the Red Chris project described in this report is $443.6 million, including a 10% contingency. Total capital, including working capital and initial funding of a reclamation bond for reclamation funding at mine closure, is estimated at $473.5 million.

A summary overview of the estimate by area is presented in Table 27.1. All costs are expressed in fourth quarter 2009 Canadian dollars with the exception of the pricing of the overland conveyor and site substation, which are both expressed in third quarter 2010 Canadian dollars. The capital cost estimation is based on the Canadian and US dollar being at par. There is no allowance for escalation, interest during construction, taxes, or duties. All allowances have been considered for items such as:

- actual productivity factors and unit prices from similar ongoing and previous projects.
- understanding of the open shop contracting environment in British Columbia at this time.
- quantity take-offs provided by AMEC.
- capital equipment costs provided by the engineer and updated by Merit in December 2009.
- geotechnical information available to identify borrow sources for the tailings dams.
- contractor budget costs for the power line.
- contractor budget costs for the main access road provided by RCDC.
- actual current costs of bulk materials such as cement and steel.
- batching of concrete on site and locally available suitable aggregate.
- experienced construction management supervision during construction.
- traditional methods of applying percentages for the costs associated with piping, electrical, and instrumentation work.
- pre-production mine work costs provided by RCDC.
- mine fleet sizes and equipment costs provided by RCDC.
- assuming that the backfill for the Crusher structure is provided by the mine operations from the pit.
- building costs provided by reputable suppliers.
- quoted costs for a leased to purchase camp.
- assuming that the explosives supplier provides the above ground facilities.
- RCDC 's Costs established by RCDC.

- Indirect Costs allowances based on calculated factors from recent northern mine project studies and current operations of RCDC.
- understanding of remoteness of project

The estimate covers the direct field costs of executing this project, plus the indirect costs associated with design, procurement, and construction efforts.

Table 27.27.1 Summary of Project Capital Costs

Area	Cdn$1000
Direct Costs	
Mining	75,369
Processing	121,144
Site Preparation	7,701
Utilities	44,698
Ancillaries	22,414
Tailings	49,519
Total Direct Costs	**320,845**
Indirect Costs	
RCDC's Costs	11,450
Indirects	71,008
Total Indirect Costs	**82,458**
Subtotal	**403,303**
Contingency – 10%	**40,330**
Total Project	**443,633**

27.2 Basis of Estimate

The capital cost estimate is based on the following project data:

- design criteria
- flow sheets
- general arrangement drawings
- single line electrical drawings
- equipment list
- budget quotations from vendors for major capital equipment and buildings
- quotations from British Columbia contractors
- regional climatic data
- in-house database

27.2.1 Pre-Production Mining

For the purpose of this estimate, the RCDC will carry out the pre-production work themselves using a new mine fleet of equipment and new Truck Shop. The RCDC will provide the fuel storage facilities and all associated mining components.

The level of estimating accuracy and the contingency value have been based on the following premises:

- labour rate is current and based on local and regional construction.
- main access road and power line work has been priced by competent contractors.
- vendor quotations have been updated for all major equipment.
- productivity factors are based on actual experience.
- material costs were obtained for the work.
- geotechnical information is available in areas of greatest earthwork risk.
- AMEC E&E were responsible for providing all bulk material quantities for the processing and infrastructure facilities.

27.3 Direct Costs

27.3.1 Quantities

The engineer provided material take-offs based on project feasibility drawings other than for plant piping, electrical, and instrumentation quantities, which were based on percentages of mechanical equipment in each area.

- Civil - All earthworks quantities, including mine pre-production stripping - have been calculated in situ with no allowance for bulking or compaction of materials.
- Concrete - Concrete quantities were determined from feasibility stage drawings and experience from previous projects of a similar nature. The unit rates have been based on the assumption that local aggregates are available with an onsite batch plant.
- Formwork - Formwork was estimated for each type of concrete classification and includes supply, form oil, accessories, shoring and stripping. No allowances have been made for re-use of forms although this will happen and improve the concrete prices during construction.
- Reinforcing Steel - Reinforcing steel was calculated based on estimated weight per cubic metre of concrete for each type of classification and based on projects of a similar nature. The unit price includes for the supply of material, cutting and bending on site, accessories, and installation.
- Embedded Metal and Anchor Bolts - The unit price includes supply and installation of carbon steel material including sleeves and anchors.
- Structural Steel - Quantities were determined from feasibility stage drawings and experience from previous projects of a similar nature. The weights shown include allowances for connections and base plates. The unit price includes steel purchase, detailing, fabrication, and erection labour.

- Mechanical Equipment - Most large capital equipment is assumed to have to be imported, and was itemized and priced in accordance with the flow sheets. Budget quotations were obtained for all major items based on preliminary specifications.
- Mechanical (Plate Work and Tanks) - Plate work weights were calculated with allowances made for any necessary stiffeners, weirs, and launders. The unit prices include purchase, detailing, fabrication, and installation.
- Piping - Factored allowances, as used on other projects developed by the engineer, have been used for in-plant pipe and calculations for overland pipe, and includes supply (not freight), shop and field fabrication, and installation with allowances for all hangers and supports.
- Electrical and Instrumentation - Allowances were based on one-line diagrams and connected loads detail provided on the flow sheets. Major electrical equipment has been assumed imported and prices were based on quotations. Bulk material prices were based on factored allowances provided by the engineer from other similar projects, and include material supply (not freight) and installation. Lengths for overhead lines and high voltage cable were estimated from the overall site plan.

27.3.2 Direct Field Labour

The project construction has been assumed to be an Open-Shop site where union and non-union labour co-exist. The labour rates have been provided by local and regional construction contractors. The rates used in this study have been derived from extensive research combining strategy of labour pool research, and input from experienced union (CLRA) and alternative union (CLAC) contractors with rates that reflect the most likely situation where a combination work force would be invited to bid on the work. Rates have been identified against each trade type and not just input as one average rate for all trades.

Labour rates have been derived as a composite for 70 hr work week, 21 days on 7 days off schedule including travel. The $80/h average wage rate is a combined rate for an estimated crew mix (supervision, skilled/unskilled labour) from industrial construction contractors experienced as general contractors in the mining industry in British Columbia. The rate includes:

- base labour wage rate
- government (payroll) burdens
- benefits including medical
- small tools and consumables allowance home office overheads
- Contractors’ profit.

27.3.3 Direct Field Materials

Bulk materials will be provided from centres in Canada, primarily Vancouver and Edmonton. Freight is included as a separate indirect cost line item.

27.4 Indirect Costs

27.4.1 Temporary Construction Facilities and Services Including Permanent Camp

Contractors' field distributable costs allowed for in the capital cost estimate, but not included in the built-up labour rate or owner's costs, are as follows:

- Contractors' mobilization and demobilization
- contractors' equipment
- construction field offices, furnishings, equipment
- contractor accommodations temporary power supply temporary water supply temporary heating and hoarding warehouse and laydown costs temporary toilets
- temporary communications first aid personnel and supplies on-going and final clean-up
- yard maintenance
- janitorial services
- site safety personnel and training material testing.

These indirect costs are not applicable to the major earthmoving costs, since unit prices submitted by contractors are "all-in" rates, inclusive of all direct and indirect costs.

27.4.2 Construction Equipment

It is expected that all construction equipment such as cranes, man lifts, welding machines, and generators will be leased and managed by the RCDC's construction management group.

Costs for fully maintained construction equipment have been determined by using contractor historical data on similar types of projects and developing an hourly cost for construction equipment per direct man hour.

Construction equipment indirect costs are not applicable to the major earthmoving, access road, power line and concrete supply since unit prices submitted by contractors are "all-in" rates which include contractor's construction equipment. Specifically, concrete supply unit prices include supply and operation of the site aggregate crushing, screening, and batch plant as well as the concrete transit trucks.

27.4.3 Construction Accommodation and Catering

An allowance is included for the construction and operation of up to 500-man camp based on a lease cost quoted by a reputable vendor. The camp kitchen is sized for the estimated peak labour force. Forty-two man bunkhouses will be installed as required when the work force increases, and demobilized as it decreases at the tail end of the construction project. RCDC has elected, for the purposes of this study, to retain a portion of the camp on a lease to purchase basis, and use it for operations. Costs for catering services have been based on quotations.

27.5 First Fill and Spare Parts

An allowance of 3% of the equipment purchase value has been included for spares, and an allowance commensurate with industry standards has been made for the purchase of start-up grinding media and reagents.

27.6 Start-up and Commissioning

An allowance has been made for retention of vendor representatives for start-up, as well as a selection of fifteen people from the contractor's crews and the construction management staff for a period of about 60 days.

27.7 Freight

A freight allowance of 7.5 % of the material and equipment costs has been included for all materials and equipment associated with purchases with the exception of concrete materials and mining equipment where freight is included in the price provided.

27.8 RCDC's Costs

RCDC's costs have been developed from manpower estimates of pre-operations personnel required for the engineering and construction phases of the project. Salary rates and benefits have been estimated from experience with similar northern Canada mining operations, current salary survey and market reviews. Transportation, materials and supplies for this group are included in RCDC's costs.

An allowance has been made for the construction of a concentrate shed at Stewart to handle the new concentrate from Red Chris. This would be constructed in Year -1.

27.9 Engineering, Procurement, and Construction Management (EPCM)

Costs for the engineering procurement and construction management includes for transportation, supplies and communication components required for these services. A factor of 10.5 % of the direct costs has been applied, except mining pre-production and mine equipment, based on a calculated factor from a recent northern mine project study.

27.10 Taxes and Duties

No taxes have been included as with the HST currently in place the sales tax is refunded.

27.11 Contingency

The contingency amount is an allowance added to the capital cost estimate to cover unforeseeable costs within the scope of the estimate. This can arise due to presently undefined items of work or equipment, or to the uncertainty in the estimated quantities and unit prices for labour, equipment, and materials. Contingency does not cover scope changes, project exclusions, or project execution strategy changes.

27.12 Exchange Rates

The Cdn$:US$ exchange rate is Cdn$1.00 = US$1.00 for the purpose of this capital cost estimate only. A different exchange rate has been adopted for financial analysis of the project.

27.13 Assumptions

The following assumptions were made in preparing the estimate:

- construction work is based on unit and fixed price contracts
- concrete aggregate and suitable backfill material are available locally, and good potential areas were identified during the project team site visits
- actual soil bearing conditions will be adequate for the requirements of foundations included in the estimate; this has been confirmed through the surficial geology assessment, including drilling and test pitting, made by RCDC based on their intimate knowledge of the area.
- construction activities will be carried out in a continuous program with no allowance for demobilization and remobilization
- the tailings area contains suitable and adequate borrow sources for the dams to be constructed.

27.14 Capital Cost Exclusions

The following are excluded from the $443.6 million project capital estimate.

- escalation
- interest during construction

Schedule delays and associated costs such as those caused by:

- scope change
- unidentified ground conditions
- extraordinary climatic events
- labour disputes
- permit applications
- receipt of information beyond the control of EPCM contractors
- cost of financing property costs and taxes
- sunk costs
- permitting costs

27.15 Working Capital

Working capital equal to 1.5 months of year 1 estimated mine site operating costs ($26.8 million) has been included in the financial model.

27.16 Reclamation Bond Funding

Initial capital includes a provision of $3.1 million to fund a surety bond to meet estimated funding requirements for ongoing mine disturbance over the first five years of operation. Additional amounts are included in the financial model, to cover mine reclamation at the end of mine life, currently estimated to be in year 28. The reclamation funding plan also assumes that funds received from the salvage of the mine equipment, process plant, and other infrastructure will also be applied to reclamation funding.

27.17 Sustaining Capital

The major component of this sustaining capital estimate is for additional and replacement mine equipment, as detailed in Section 4.

Over the mine life, it is estimated that an additional $238 million in sustaining capital will be required. This covers such items as: additional and replacement mine equipment and other capital requirements, including construction of the centre till core of the tailings dam, expected to be performed by a contractor.

28 OPERATING COST ESTIMATES

This section presents the updated operating cost estimates for the Red Chris Project. These costs form the basis of the financial analysis of the project.

28.1 Summary

On-site operating costs are estimated to be $9.96/t of ore milled including mining, milling, royalty payments, reclamation expenditures, G&A, and head office costs. The unit costs, summarized in Table 28.1, are based on annual mill throughput of approximately 11 Mt/a (30,000 t/d).

Table 28.1 Operating Cost Estimates Per Tonne Milled (LOM)

Area	Per Tonne Milled (LOM)
Mining	$4.45
Milling	$3.87
Royalty	$0.18
Reclamation Expenditure	$0.04
Project Overhead (G&A)	$1.09
Head Office Costs	$0.33
TOTAL OPERATING COST	**$9.96**

28.2 Basis of Estimate

Data for this estimate was based on current regional salary rates, and experience from Imperial's Mount Polley Mine and Huckleberry Mine operations.

28.2.1 Mining

Unit mining costs used for the annual tonnages mined in the shovel and truck open pit operations are based on Imperial's current operating experience. The distribution of costs has been identified as direct mining (drilling, blasting, loading, hauling, and road maintenance) and general mine expense (supervision, engineering, geology, pit dewatering, and mine electrical and mechanical services). Table 28-2 shows the distribution of unit costs in each area as below:

Table 28.2 Mining Operating Costs by Area

	Per tonne Mined (LOM)	Per tonne Milled (LOM)
Drilling	$0.14	$0.34
Blasting	$0.20	$0.49
Shovel Operations	$0.12	$0.29
Trailing Cable	$0.01	$0.02
Loader Operations	$0.11	$0.27
Stockpile Rehandle	$0.01	$0.02
Pit Road & Services	$0.27	$0.66
Mine Dispatch System & Computers	$0.01	$0.02
Truck Haulage	$0.62	$1.51
Drainage & Dewatering	$0.03	$0.07
Pit Electrics	$0.02	$0.05
Yards & Access Roads	$0.02	$0.05
Supervision	$0.07	$0.17
Ground Monitoring	$0.01	$0.02
Engineering	$0.03	$0.07
Sub Total Mine Operations	$1.67	$4.06
Mine Maintenance Supervision & Shop	$0.16	$0.39
Sub-Total Mine Operations + Maintenance	$1.83	$4.45
TOTAL MINE OPERATING COSTS	**$1.83**	**$4.45**

28.2.2 Milling

The average annual processing cost for the 30,000 t/d concentrator is estimated to be $3.87/t milled. Table 28.3 provides a summary of estimated processing costs.

Table 28.3 Processing Costs by Area

Area	Unit Cost per Tonne Milled (LOM)
Labour	$0.68
Materials & Supplies	$0.45
Power	$1.16
Consumables - Reagents rods and Balls	$1.29
Other	$0.29
TOTAL PROCESSING COSTS PER TONNE MILLED	**$3.87**

28.2.3 Royalty

The royalty includes a $1 million payment to buy-out 0.8% of the 1.8% royalty to Falconbridge, leaving a 1% NSR royalty payable on the Red Chris production.

28.2.4 Reclamation Expenditures

Estimated Reclamation expenditures at years 5, 10, 15, 20, 25 and at year 28 in today's dollars have been unitized over the LOM milled tonnes. Estimated expenditures are shown in the Table 28.4 as below:

Table 28.4 Reclamation Expenditures

Year of Operation	Estimated Expenditure
5	$0.16 million
10	$0.43 million
15	$1.58 million
20	$1.90 million
25	$2.20 million
28	$4.91 million
Total	**$11.17 million**

28.2.5 Project Overhead (G&A)

On the basis of experience from Imperial's operations, G&A related costs have been estimated at $1.09/t milled over the LOM.

28.2.6 Head Office Costs

Head Office Costs as experienced from Mount Polley Mine have been estimated at $0.33/t milled over the LOM.

28.3 Work Force

The estimated direct workforce totals for the operation are summarized in Table 28.5. It does not include personnel for catering, housekeeping, concentrate haulage, or the explosives plant, which are normally contracted out and are estimated to add an additional 50 to 60 jobs.

Table 28.5 Workforce Totals

			Personnel On Site					
	Year -1	**Year 1**	**Year 5**	**Year 10**	**Year 15**	**Year 20**	**Year 25**	**Year 28**
Administration	28	28	28	28	28	28	28	28
Mining	98	166	171	194	181	160	160	156
Processing	0	93	93	93	93	93	93	93
Total Property	136	**287**	**292**	**315**	**302**	**281**	**281**	**277**

29 ECONOMIC ANALYSIS

The Red Chris project has been valued using a discounted cash flow approach. This method of valuation requires projecting yearly cash inflows, or revenues, and subtracting yearly cash outflows such as operating costs, capital costs, royalties, and taxes. The resulting net annual cash flows are discounted back to the date of valuation and totalled to determine net present values (NPVs) at the selected discount rates. The internal rate of return (IRR) is defined as the discount rate that yields a zero NPV on the date of valuation.

All amounts are presented in Canadian dollars unless otherwise specified.

29.1 Financial Model Parameters

29.1.1 Ore Reserves and Mine Life

The economic analysis is based on the following LOM statistics:

- Total ore milled: 301,549,000 tonnes milled.
- Average copper grade milled: 0.359%
- Average gold grade milled: 0.274 g/t
- Overall Striping ratio LOM: 1.25 tonnes ore to 1.0 tonnes waste.

These reserves will be processed at a rate of 30,000 t/d over a planned mine life of approximately 28.3 years. (*Note: The 301.6M tonnes milled includes treatment of stockpiled low grade material.)*

29.1.2 Metallurgical Balance

The base case metallurgical recoveries and copper concentrate grades for the LOM are:

- Copper recovery: 87.04%
- Gold recovery: 49.27%
- Copper concentrate grade: 27%

Calculated gold concentrate grade varies by year and averages 11.68 g/t Au over the LOM.

Based on a review of available geologic and metallurgical data, an average silver concentrate grade of 42.0 g/t has been used for the base case.

29.1.3 Smelter Terms

The base case financial evaluation model incorporates the following smelter terms outlined in Table 29.1 below:

Table 29.29.1 Smelter Terms

Copper payment	*Lesser of*	*96.50%*	*of full Copper content*
	and	100.00%	of full Copper content
	less	1	unit of Copper
Price Participation	No Price Participation		
Treatment Charge	US$ per DMT	$60.00	$65.00
Penalty Allowance	US$ per dmt	$5.00	$5.00
Refining Charge Payable Cu	US$ per lb	$0.060	$0.065
Gold Payments	Pay 95% of Contained gold		
Gold Refining Charge	US$ per Oz	$5.00	Payable Gold
Silver Payments	Pay 90% Contained Ag		
Silver Refining Charge	US$ per Oz	$0.35	Payable Silver

29.2 Concentrate Transportation Costs

The base case considers aggregate concentrate transportation charges to the smelter of US$114.91/wmt consisting of the following listed in Table 29.2 below:

Table 29.2 Concentrate Transportation Costs

Item		Cost/tonne
Trucking Mine Site to Stewart	Can$/tonne	$40.50
Port Charges (warehouse and ship loading)	Can$/tonne	$13.50
Ocean Freight (inc. insurance)	US$/tonne	$66.44
Total Shipping Cost per WMT	Can$/tonne	$127.68
Total Shipping Cost per WMT	**US$/tonne**	**$114.91**

Wet concentrate tonnage estimates are based on moisture content of 8.0%

29.3 Markets and Contracts

RCDC anticipates that the concentrate will be sold to markets throughout the Pacific Rim countries. Marketing discussions have been held with a number of interested parties however, no sales contracts have been entered into at this time.

29.4 Metal Prices

The following long-term base case metal prices have been used:

- Copper: US$2.20/lb
- Gold: US$900/troy oz
- Silver: US$12.00/troy oz.

These compare with the average metal prices and exchange rate for the month of January 2012 of:

- Copper: US$3.6485/lb
- Gold: US$1,656.095/troy oz
- Silver: US$30.769/troy oz.

29.5 Exchange Rate

An exchange rate of Cdn$1.00 = US$0.90 has been used in the base case economic analysis except for the estimated capital cost where the Canadian and US dollar where assumed to be at par. The January 2012 average exchange rate was Cdn$1.00 = US$0.986911676.

29.6 Operating Costs

The LOM operating costs are summarized in 2010 dollars as shown in Table 29.3 below:

Table 29.3 LOM Mine Operating Costs

Area	Per Tonne Milled (LOM)
Mining	$4.45
Milling	$3.87
Royalty	$0.18
Reclamation Expenditure	$0.04
Project Overhead (G&A)	$1.09
Head Office Costs	$0.33
TOTAL OPERATING COST	**$9.96**

29.7 Capital Costs

The estimated project direct capital costs (provided by Merit & RCDC) are summarized in Table 29.4 and the estimated Indirect Costs, are shown in Table 29.5.

Table 29.4 Direct Capital Costs

Item	Cdn$1,000
Mine Preproduction Development	2,646
Mine Equipment	72,185
Mine Dewatering	538
Crushing	6,990
Conveyors	18,758
Coarse Ore Storage	5,060
Grinding	41,915
Flotation and Regrinding	21,607
Concentrate Dewatering	4,306
In Plant Tailings System	2,018
Reagents Handling and Storage	4,061
Process Building	16,430
General Site Development	5,939
Access Roads	1,761
Power Generation or Supply	29,713
Power Distribution	8,308
Fuel Supply, Storage & Distribution	531
Fresh Water Supply & Distribution	3,625
Fire Protection and Prevention	612
Waste Disposal	59
Control and Communications Systems	1,850
Warehouse and Maintenance	13,279
Administration Building	1,997
Laboratory Facilities	2,112
Cold Storage Building	284
Gate House	59
Plant Mobile & Utility Equipment	4,583
Explosives Handling and Storage	100
Tailings Management Facility	30,738
Tailings and Sands Pipelines	8,093
Reclaim System	10,688
Total Direct Capital Costs	**$320,845**

Table 29.5 Indirect Capital Costs

Indirect Capital Costs	Cdn$1,000
RCDC's Costs	11,450
EPCM	25,294
Construction Indirects	12,996
Construction Camp and Catering with allowance for conversion to permanent camp	13,735
Capital Spares	5,587
Freight	12,106
Vendor Reps	150
Start-up & Commissioning	1,140
Total Indirect Capital Costs	**82,458**
Total Direct + Indirect	**403,303**
Contingencies @ 10%	**40,330**
TOTAL CAPITAL COSTS	**$443,633**

29.8 Royalty

The royalty includes a $1 million payment to buy-out 0.8% of the 1.8% royalty to Falconbridge, leaving a 1% NSR royalty payable on the Red Chris production.

29.9 Taxes

The Red Chris project will be subject to income and revenue taxes as follows:

BC Mineral Tax Act	Stage I	2.00%	Tax on net current Proceeds from operation
	Stage II	13.00%	Tax on cumulative net Revenues less capital costs

Federal Income Tax rate	15.0%
Provincial Income Tax rate	10.0%

British Columbia mineral tax is applied to an amount different from taxable income, as defined for Federal and Provincial income tax purposes, and is deductible in arriving at taxable income for Federal and Provincial income tax purposes.

Federal and Provincial Tax rates are changing in the near future and have been incorporated in the model as follows:

Table 29.6 Federal and Provincial Tax rates

	2010	2011	2012	2013
Federal rates:	18.0%	16.5%	16.5%	15.0%
Provincial rates:	10.5%	10.0%	10.0%	10.0%

29.10 Financing

The base case economic analysis has been presented on a 100.0% equity basis. However, the analysis of the project indicates that it will support debt financing.

29.11 Inflation

The base case economic analysis has been presented with no inflation (constant dollar basis).

29.12 Discounting and Date of Valuation

Estimated annual net cash flows have been discounted to the beginning of project Year -4 at real discount rates of 5% and 10%.

29.13 Sensitivity Analysis

The results of the base case and sensitivity analysis are summarized in Tables 29.7 and Figures 29.1 and 29.2. The estimated base case IRR based on 100.0% equity is 15.7% post tax with an undiscounted project cashflow of Cdn$1.214 billion, after capital costs and taxes. Payback of the total construction cost of Cdn$443.6 million is achieved within 4.6 years of start-up.

LOM cost per pound of copper, for the base case, taking silver and gold as a credit is US$ 1.22 per pound of copper. At January 2012 monthly average metal prices and exchange rates (Cu = US$3.6485lb, Au = US$1,656.095/oz, Ag = US$30769 /oz, and Cdn$1 = US$0.986911676), the estimated IRR increases to 38.8% post tax and the undiscounted project cashflow totals of Cdn$3.675 billion, after capital costs and taxes. Payback of the total construction cost of Cdn$443.6 million is achieved within 1.81 years of start up at these prices.

LOM cost per pound of copper, at the average of January 2012metal prices, taking silver and gold as a credit is US$0.96 per pound of copper.

Table 29.7 Financial Analysis: Base Case and January 2012 Prices

Economic Parameters	BASE CASE		JAN 2012Prices	
	Copper Price	US$2.20/lb	Copper Price	US$3.6485/lb
	Gold Price	US$900/oz	Gold Price	US$1,656.095/oz
	Silver Price	US$12/oz	Silver Price	US$30.769/oz
	Cdn$ =	US$0.90	Cdn$ =	US$0.986911676
Capital Cost	Cdn$443.633 M		Cdn$443.633 M	
Post Tax IRR	15.7%		38.8%	
Post Tax NPV				
@0%	Cdn$1.214 B		Cdn$3.675 B	
@5%	Cdn$423.24 M		Cdn$1.571 B	
@10%	Cdn$133.92 M		Cdn$772.1 M	
Project Payback	4.58 yrs		1.81 yrs	
LOM Production Cost of per pound of Copper with credits from Gold and Silver	US$1.22		US$0.96	

Figure 29.29.1 Project Sensitivity to Operating Cost Variables

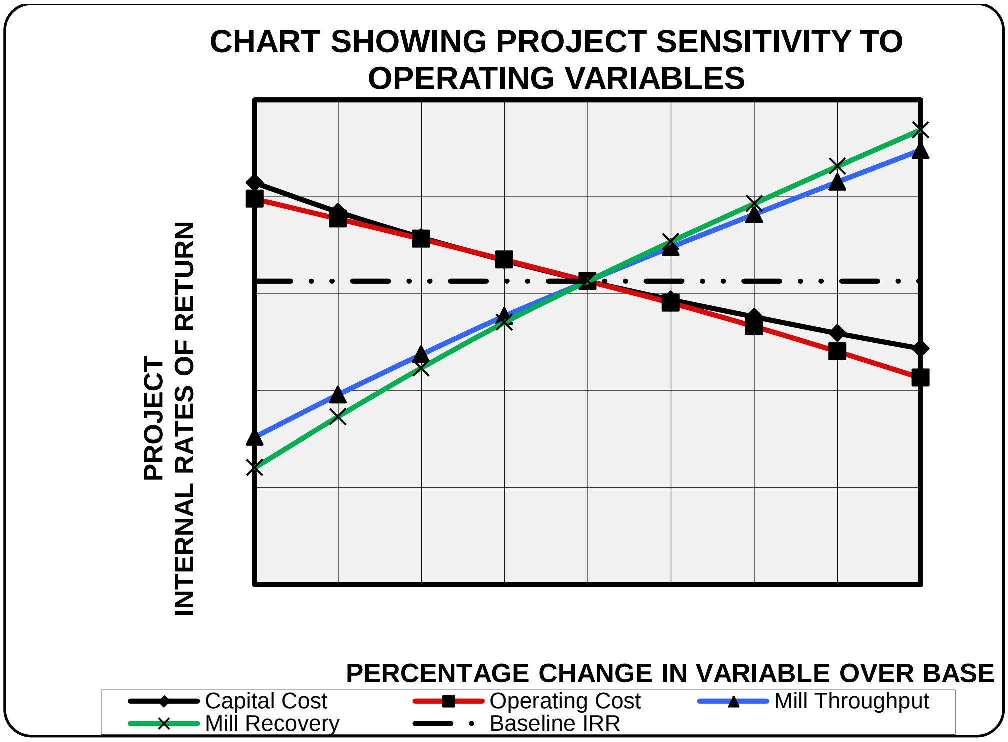
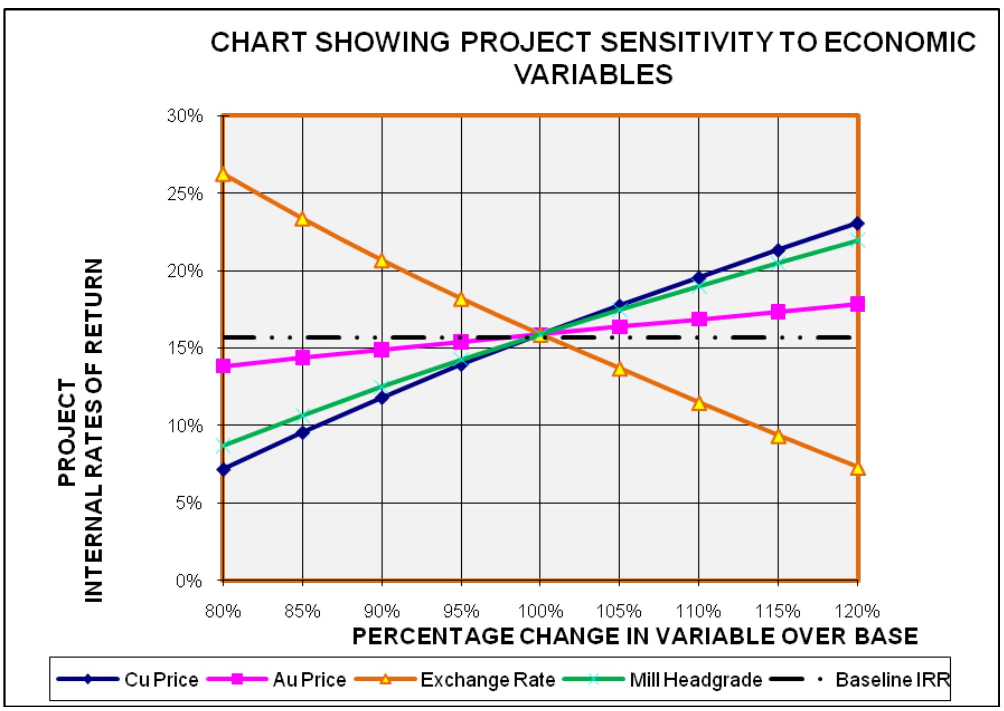


Figure 29.2 Project Sensitivity to Economic Variables



30 Interpretation and Conclusions

Key financial conclusions are predicated on the following base case assumptions as shown in Table 30.1.

Table 30.1 Base Case Price and Exchange Rate Assumptions

Description	Assumption
Copper price	US$2.20/lb
Gold price	US$900.00/oz
Silver price	US$12.00/oz
Cdn$	**US$0.90**

Table 30.2 shows the key economic findings derived from the financial model. The financial model uses the information on reserves, mining costs, capital costs, and base case assumptions developed in this report. This table also demonstrates the financial performance of the project after tax at January 2012 monthly average metal prices and exchange rates.

Table 30.2 Financial Analysis for Base Case and January 2012 Metal Prices & Rates

Economic Parameters	BASE CASE		January 2012 AVERAGE	
	Copper Price	US$2.20/lb	Copper Price	US$3.6485/lb
	Gold Price	US$900/oz	Gold Price	US$1,656.095/oz
	Silver Price	US$12/oz	Silver Price	US$30.769/oz
	Cdn$ =	US$0.90	Cdn$ =	US$0.986911676
Capital Cost	Cdn$443.633 M		Cdn$443.633 M	
Post Tax IRR	15.7%		38.8%	
Post Tax NPV				
@0%	Cdn$1.214 B		Cdn$3.675 B	
@5%	Cdn$423.24 M		Cdn$1.571 B	
@10%	Cdn$133.92 M		Cdn$772.1 M	
Project Payback	4.58 yrs		1.81 yrs	
LOM Production Cost of per lb of Cu with credits from Au and Ag	US$1.22		US$0.96	

Table 30.3 Summary: Financial, Reserve, Operating, and Capital Information

Project Finance (100% Equity)		
Internal rate of return (post-tax):		15.7%
Net present value	- 0% disc.:	$1.214 B
	- 5% disc.:	$ 423.24 B
	- 10% disc.:	$133.92 M
Payback:		4.58 years
Total Mineable Reserves Copper grade Gold grade		301.549 M tonnes 0.359% (avg) 0.274 g/t (avg)
In situ metal	- Copper	2.380 B lbs
	- Gold	2.67 M oz
In Concentrate metal	- Copper	2.080 B lbs
	- Gold	1.32 M oz
Production Rate and Mine Life		
Production rate	-ore milled	10.95 Mt/a
Life of mine (LOM): including reprocessing of stockpiled material in years 24 to 28.3		28.3 years
LOM strip ratio:		1.25:1
Copper Recovery Gold Recovery		87.0% (avg) 49.3% (avg)
Concentrate grade		27% Copper
Concentrate production		337 dmt/d (avg)
Concentrate production		183,000 dmt/a peak
Concentrate production		123,000 dmtt/a (avg)
Capital Costs		
Initial capital costs, including Indirects and contingency @10%.		$ 443.633 M
Working capital		$ 26.75 M
Sustaining capital (LOM)		$ 238.25 M
Net Revenue at Mine Gate	**Cdn $/t ore**	**$17.79**
Less Mining Cost	$/t ore	$4.45
Less Milling Cost	$/t ore	$3.87
Less Royalty	$/t ore	$0.18
Less Reclamation Expenditure	$/t ore	$0.04
Less Project Overhead	$/t ore	$1.09
Less Head Office costs **Total Operating Costs**	$/t ore **$/t ore**	$0.33 **$9.96**
Net Current Proceeds before Capital cost	**Cdn $/t ore**	**$7.84**

Base case economic analysis has been run with no inflation (constant dollar basis). Capital and operating costs are expressed in 2010 Canadian dollars, unless otherwise noted.

A review of the past exploration programs at the Red Chris Project shows a tremendous amount of high quality work has been completed including geologic mapping and sampling, geophysics, trenching, diamond drilling, modelling, preliminary mine planning, metallurgical studies, environmental base lines studies, and ABA testing.

The drilling, logging, and sampling protocols employed at the Red Chris Project in both the past and current drilling programs are appropriate for the deposit type and are being carried out in a fashion that meets or exceeds common industry standards. Assaying was carried out using industry standard methods and QA/QC protocols. The QA/QC data indicate that the assays are acceptable for use in Mineral Resource estimation.

The 2007 to 2012 drilling programs were highly successful in achieving their goal of increasing the size depth and understanding of the deposit.

The original resource estimate table published on Feb. 14, 2012 was constrained by a series of Copper Equivalent grade shells, within a wire frame digital solid constructed around the three mineralized deposit domains. The resource has now been amended and restated here (September 30 1015) as a combination of an Open Pit and Block Cave constrained Resource to demonstrate “reasonable prospects of economic extraction” as referred to in Instrument NI 43-101. Table 17.3 summarizes the total Mineral Resources, both open pit and underground, for the Red Chris Project. Although the authors believe there is a reasonable prospect of economic extraction there can be no assurance that these Mineral Resources will be eventually upgraded to Mineral Reserves. The results are shown in Tables 17.1, 17.2 and 17.3.

31 Recommendations

31.1 Exploration Recommendations

The deep drilling program at Red Chris completed since the last update in early 2010 has confirmed a larger area of continuous, deep mineralization below the East, Saddle and Main zones and also provided important information on mineralization, alteration, structure and geology both at the East Ridge and Gully zones.

The program was designed to complete grid drilling, at 200 metre spacing, beneath the proposed open pit presented in the November 2010 feasibility study. The results of the drilling proved that almost that entire area is mineralized to some degree below the bottom of the 425 metre deep pit. Mineralization continues to be open for expansion along the outer edges of this drilling in several areas and additional drilling to the north, east, south and west of the current drilling is recommended. Most notably, hole RC11-429 displays strong alteration and mineralization that could indicate it is near a major pathway of metal laden hydrothermal fluids which were responsible for one or more pulses of mineralization in the Saddle and East zones. It is possible that some infill drilling in the Saddle zone could result in an increase in the size of the high grade core of mineralization defined there.

There are large areas to the south of current drilling where Red Stock intrusive is believed to exist and although some of that phase of Red Stock is known to be barren of copper – gold mineralization, the strong alteration does provide evidence that hydrothermal activity was pervasive in the area. Extension of the existing mineralization or perhaps even an entirely separate intrusive source may exist in that area, so additional drilling is warranted.

Mineralization continues to the east of the open pit at depth and although only one of the two East Ridge zone drill holes intersected a zone of significant grade, the volume of unexplored and altered intrusive in that area will require significant amount of diamond drilling to fully assess it. Similar or better potential exists to the west of the proposed open pit as the most westerly holes in the Main zone are strongly mineralized and/or altered and the deep drilling at Gully zone a kilometer to the west has now proven the extent strength of mineralization there is much greater than previously realized. The resulting conclusion is that the status of the intervening area is high prospective. The topography between Main and Gully zones will make drill setups difficult so long angle holes will likely be the approach to investigate the deep potential.

It is recommended that the future exploration continue to pursue some of the detailed geologic work that has been so important in helping the exploration team build a detailed model of the mineralizing system. This work included detailed petrographic work, phase logging, and structural modeling. These studies are ongoing, and will be conducted by the onsite geological staff. Costs for these ongoing studies are included in the Mining Costs in the Life of Mine Operating Costs in Table 29.3.

31.2 Feasibility Study Recommendations

Based on the findings of this Technical Report, it is concluded that the project has robust economic viability. The Provincial and Federal governments have committed to provide funding for a powerline from Terrace to Bob Quinn (Northwest Transmission Line ("NTL") Project), for which the provincial and federal environmental assessment process has just been completed and the project is now approved and as of January 2012 construction on the NTL project has commenced. Assuming that the NTL project is constructed, Imperial will be responsible for extending powerline service sufficient to meet its needs from Bob Quinn to Tatogga and from there to the Red Chris Mine. Target completion date for the NTL Project is December 2013. It is recommended that RCDC proceed with detailed engineering, procurement, construction, and commissioning as per the budget set out in Chapter 27 to 29 this report, and to target commercial production in the first quarter of 2014, contingent on the completion of the NTL in December 2013.

Key Updates Since the Feasibility Study of 2005:

- Provincial Environmental Assessment process completed and Environmental Assessment Certificate obtained.
- Federal approval for the Red Chris project under the Canadian Environmental Assessment Act (CEAA) completed.
- SRK completed a due diligence study, in March of 2006, on the earlier published Feasibility Study of Dec 2004 on behalf of potential financiers of the Red Chris project, then owned and operated by bcMetals.
- Ownership changed from bcMetals to Imperial as of February 2007.
- Continued onsite exploration since 2007, updated the mineral resources at varying cut-off grades as reported in Technical Report: 2010 Exploration, Drilling and Mineral Resources Update.
- As a follow-up on SRK findings of 2006, completed additional field geotechnical investigation for the plant site and for the tailings impoundment area and accordingly revised the plant site location and overall site layout.
- With the continued on-site climatic data gathering, revised the average precipitation and other climatic considerations.
- Updated mineable mineral reserves within the pit design to reflect the current economic environment.
- Mineable mineral reserves are now based on mill head values rather than based on NSR.
- Capital cost updated to reflect the current economic environment and also to include the construction of a power transmission extension line from Bob Quinn to the Mine site.
- Operating costs updated to reflect the current economic environment and also with references to other mines owned and operated by Imperial.
- Updated the financial model with the revised capital and operating costs.
- Provincial and Federal environmental assessment for the NTL project has been approved and construction has commenced.
- Detailed engineering and procurement for the Red Chris Project is in progress with AMEC.
- Red Chris Project is in advanced stage of permitting. Construction is anticipated to commence in spring 2012.

Figure 31.1 Drilling on the Red Chris Property

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33 Certificate of Authors and Signatures

All Certificates are Signed, Dated and Sealed on the Original Version of this Document.

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1. I, Stephen Robertson P.Geo., am a Registered Professional Geologist and am currently employed in the position of Vice President Corporate Affairs with Imperial Metals Corporation of Suite 200 at 580 Hornby Street, in the City of Vancouver, in the Province of British Columbia, Canada.
2. I am a member of the Association of Professional Engineers and Geoscientists of British Columbia.
3. That I am one of the contributing authors of the report dated February 14, 2012 entitled "2012 Technical Report on the Red Chris Copper-Gold Project" and re-issued on September 30, 2015 to which this Certificate applies.
4. I graduated from the University of Alberta with a Bachelor of Science degree in Geology in 1989. I have practiced my profession continuously since 1989. I have 26 years relevant experience in the mining industry mostly with copper, gold and copper/gold porphyry deposits. I have knowledge and experience in mineral exploration, geologic mapping, prospecting, exploration drilling, assay management and quality control, open pit and underground mining, milling and metallurgical testing, project planning and budgeting, project management, environmental monitoring and permitting. Prior to joining Imperial Metals Corporation, I worked from 1989 to 1992 with Corona Corporation as part of a team exploring for copper/gold porphyry systems within British Columbia. The majority of my applicable experience with copper/gold porphyry deposits was gained during my employment with Imperial Metals where I have been employed as a geologist since 1993. I was a mine geologist and chief geologist at the Goldstream copper/zinc mine from 1993 until 1996 and then an exploration geologist working mostly on Imperials copper and gold properties from 1996 until 2013, at which time I moved to my current position. I designed and supervised the exploration of copper/gold porphyry mineralization at Polley Mine from 2003 until 2006. I designed and supervised the exploration of copper/gold porphyry mineralization at the Red Chris project from 2007 until 2012, at which time exploration was suspended. Other relevant experience was gained while working on numerous exploration and drilling and mining projects during my 26 years as a geologist.
5. As a result of my education, professional qualifications and experience, I am a Qualified Person as defined in NI 43-101.

6. I frequently visited the Red Chris property during the period of 2007 through 2012, a period during which I managed onsite activities including the exploration program. My last visit prior to the original release of this report was for one day on January 9, 2012. My most recent visit to the Red Chris property was a single day visit on August 19, 2015.

7. I am responsible for all items related to the following chapters in the technical report: Sections 5 through 15 inclusive.

8. I am not independent of Imperial Metals Corporation in reference to Section 1.5 of National Instrument 43-101. I hold the position of Vice President Corporate Affairs with the company.

9. I supervised a claim staking program for Imperial Metals Corporation, adjacent to the Red Chris property in 2004. That is the only direct involvement with the Red Chris project previous to the acquisition by Imperial Metals Corporation in 2007.

10. I have read National Instrument 43-101, Form 43-101F1 and the Companion Policy 43-101 CP and this technical report has been prepared in compliance with NI 43-101, Form 43-101F1 and 43-101 CP.

11. As of the date of this Certificate, 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 at Vancouver, British Columbia, this day of Sept 30, 2015.

____

Signature

Stephen Robertson, P.Geo.

Original is Dated, Signed and Sealed

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I, Raj Anand, M.Eng; P.Eng., am a Registered Professional Engineer and am currently employed in the position of Manager, Project Development with Imperial Metals Corporation of Suite 200 at 580 Hornby Street, in the City of Vancouver, in the Province of British Columbia, Canada.

1. I am a Professional Engineer, registered with the Association of Professional Engineers and Geoscientists of the Province of British Columbia, and with the Association of Professional Engineers and Geoscientists of the Province of Alberta.

2. That I am one of the contributing authors of the report dated February 14, 2012 entitled "2012 Technical Report on the Red Chris Copper-Gold Project" and re-issued on September 30, 2015 to which this Certificate applies.

3. I am a 1983 Masters in Engineering (Civil/Structural) Degree holder from the University of Alberta in Edmonton with a Bachelors Equivalent Degree, 1970, from the Institution of Engineers, India. I have been employed in mining in Canada since 1978 and have continuously practiced my profession as Professional Engineer since 1985 with the Consulting Engineering Houses and Operating Companies. I have been working on the Red Chris Project since 2007. I have been responsible for the ongoing activities towards Red Chris Mine development including updating the Feasibility Study of 2005 in 2010, and detailed engineering, procurement and permitting related activities for Red Chris. I have thirty years of professional experience as Professional Engineer including the last over eight years with Imperial Metals as below:

June 2007 to present	Imperial Metals Corporation: Manager, Project Development, responsible for bringing the Red Chris Project into Production - permitting, feasibility study updating, and engineering and construction.

4. As a result of my, professional qualifications and experience, I am a Qualified Person as defined in NI 43-101.

5. I have been working on the Red Chris Project since 2007. I have been responsible for the ongoing activities towards Red Chris Mine development including updating the Feasibility Study of 2005 in 2010, and detailed engineering and permitting related activities for Red Chris Project. My last visit to the property, prior to the original release of this report, was for four days, January 9 to 12, 2012.

6. I am responsible for all items related to Section 4 and Sections 18 through 29, inclusive in this Technical Report.

7. I am not independent of Imperial Metals Corporation in reference to Section 1.5 of National Instrument 43-101. I hold the position of Manager, Project Development with the company.

8. I have had no prior involvement with the Red Chris project except when I joined Imperial Metals in June 2007 as Manager, Project Development.

9. I have read National Instrument 43-101, Form 43-101F1 and the Companion Policy 43-101 CP and this technical report has been prepared in compliance with NI 43-101, Form 43-101F1 and 43-101 CP.

10. As of the date of this Certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Vancouver, British Columbia, September 30, 2015.

____

Signature

Raj Anand, M.Eng; P.Eng.

Original is Dated, Signed and Sealed

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1. I, Paul Martin Sterling, P.Eng, am a Registered Professional Engineer and am currently employed in the position of Corporate Metallurgist with Imperial Metals Corporation of Suite 200 at 580 Hornby Street, in the City of Vancouver, in the Province of British Columbia, Canada.
2. I am a member of the Association of Professional Engineers and Geoscientists of British Columbia.
3. That I am one of the contributing authors of the report dated February 14, 2012 entitled "2012 Technical Report on the Red Chris Copper-Gold Project" and re-issued on September 30, 2015 to which this Certificate applies.
4. I graduated from the University of British Columbia with a Bachelor of Applied Science degree in Chemical Engineering in 1984 and I have practiced my profession continuously since 1984. My experience includes the following:

   a. 2006 - Present : Corporate Metallurgist – Imperial Metals Corp., Vancouver
   b. 2001 - 2006: Consulting Metallurgical Engineer, Summerland
   c. 1998 – 2001: Consulting Metallurgical Engineer, Reno, Nevada
   d. 1993 - 1998: Kappes, Cassiday and Associates – Reno, Nevada
   e. 1991 - 1993: MK Gold, Yuma, Arizona
   f. 1990 - 1991: Chief Metallurgist, Bethlehem Resources Corp.

5. As a result of my education, professional qualifications and experience, I am a Qualified Person as defined in NI 43-101.
6. I visited the property previous to the original release date of this report on August 18th, 2011 for one day. I visited the site on May 12th, 2012, August 12th to 14thth, 2012, September 24th, 2012, May 20th to 22nd, 2013, September 12th, 2013, November 26th, 2013, January 28th to 31st, 2014, and finally May 26th to 28th, 2014.
7. I am responsible for all items related to the following chapter in the technical report: Section 16.
8. I am not independent of Imperial Metals Corporation in reference to Section 1.5 of National Instrument 43-101. I hold the position of Corporate Metallurgist with the company.

9. I have had no prior involvement with the Red Chris project previously to the acquisition by Imperial Metals Corporation in 2007.

10. I have read National Instrument 43-101, Form 43-101F1 and the Companion Policy 43-101 CP and this technical report has been prepared in compliance with NI 43-101, Form 43-101F1 and 43-101 CP.

11. As of the date of this Certificate, 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 at Vancouver, British Columbia, this day of September 30, 2015

____
Signature

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12. I, Greg O. Gillstrom P.Eng, am a Registered Professional Engineer and am currently employed in the position of Senior Geological Engineer with Imperial Metals Corporation of Suite 200 at 580 Hornby Street, in the City of Vancouver, in the Province of British Columbia, Canada.

13. I am a member of the Association of Professional Engineers and Geoscientists of British Columbia.

14. That I am one of the contributing authors of the report dated February 14, 2012 entitled "2012 Technical Report on the Red Chris Copper-Gold Project" and re-issued on September 30, 2015 to which this Certificate applies.

15. I graduated from the University of British Columbia with a Bachelor of Applied Science degree in Geological Engineering in 1990, and a Technologist Diploma in Electrical Engineering from BCIT in 1984. I have practiced my profession continuously since 1990. I have 25 years relevant experience in the mining industry mostly with copper, gold and copper/gold porphyry deposits. I have comprised knowledge and experience in mineral exploration, exploration drilling, assay management and quality control, open pit and underground ore control, milling and metallurgical testing programs, mine surveying, mine design, long and short range mine planning, operations supervision, mine economics engineering management principles, environmental monitoring and geologic modeling and ore reserve estimation using advanced mining software. I have taken additional software courses over the past 10 years in the use of MineSight software, the software used to calculate the estimations and mining parameters in this report. These include, MineSight 3D Block modeling, MineSight for Modeling and Geostatistics, Advanced MineSight for Modeling and Geostatistics, Advanced MineSight for Short Term Mine Planning, and Customized MineSight for Long Term Mine Planning. The majority of my applicable experience with copper/gold porphyry deposits was gained during my employment with Imperial metals, where I have been the company's Senior Geological Engineer (2004 - present), previously as Chief Mine Geologist at the companies Mount Polley Mine (1998-2002) similar copper/gold porphyry and at the Goldstream Copper/Zinc mine (1994- 1996). Other relevant experience was gained while working on numerous exploration and drilling and mining projects.

16. As a result of my education, professional qualifications and experience, I am a Qualified Person as defined in NI 43-101.

17. I had not visited the property previous to the original release date of this report. My last visit to the Red Chris property was on September 9th 2013, for two days.

18. I am responsible for all items related to the following chapters in the technical report: Sections (1-3, 17 and 30-37).

19. I am not independent of Imperial Metals Corporation in reference to Section 1.5 of National Instrument 43-101. I hold the position of Senior Geological Engineer with the company.

20. I have had no prior involvement with the Red Chris project previously to the acquisition by Imperial Metals in 2007.

21. I have read National Instrument 43-101, Form 43-101F1 and the Companion Policy 43-101 CP and this technical report has been prepared in compliance with NI 43-101, Form 43-101F1 and 43-101 CP.

22. As of the date of this Certificate, 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 at Vancouver, British Columbia, this day of Sept 30, 2015.

Signature

Greg O. Gillstrom

Original is Dated, Signed and Sealed

APPENDIX

34 Appendix A - Production Schedules

Mining Schedule – Summary to Period 7								
Period	-1	1	2	3	4	5	6	7
Mill Feed from Pit (ktonnes =kt)	0	10,952	10,744	11,033	11,463	11,063	10,798	11,048
Mill Feed from Skpl (Rehandle) (kt)	0	0	0	0	0	0	0	0
Total Mill Feed (kt)	0	10,952	10,744	11,033	11,463	11,063	10,798	11,048
Pit to Low Grade Ore Stockpile kt	0	2,769	2,724	2,914	3,995	4,529	3,598	3,503
Pit to Rock Storage (kt)	1,765	16,886	18,848	17,501	15,937	16,846	18,132	21,717
Total to the Skpl/Storage Area (kt)	1,765	19,655	21,572	20,415	19,932	21,376	21,730	25,220
Total Material Movement (kt)	1,765	30,607	32,316	31,447	31,396	32,439	32,528	36,269
Ore & LG to Mill & Stockpiles (kt)	0	13,721	13,468	13,946	15,459	15,592	14,396	14,551
Waste to Dump (kt)	1,765	16,886	18,848	17,501	15,937	16,846	18,132	21,717
Total Production (w/o Skpl Rhdl) (kt)	1,765	30,607	32,316	31,447	31,396	32,439	32,528	36,269
Strip Ratio (Waste Mined/Ore Mined)	N/A	1.23	1.4	1.25	1.03	1.08	1.26	1.49
Strip Ratio (Waste Mined/Ore Milled)	N/A	1.54	1.75	1.59	1.39	1.52	1.68	1.97
Weighted Ave Bench Location (m)	1500	1493.5	1459.1	1448.7	1454	1434.6	1443.9	1428.9

Mining Schedule – Summary Period 8 to 28								
Period	8	9	10	11 to 15	16 to 20	21 to 25	26 to 28.3	Total
Mill Feed from Pit (ktonnes =kt)	11,062	11,393	11,189	51,720	55,169	55,627	16,441	289,700
Mill Feed from Skpl (Rehandle) (kt)	0	0	0	3,296	0	0	8,553	11,849
Total Mill Feed (kt)	11,062	11,393	11,189	55,016	55,169	55,627	24,994	301,549
Pit to Low Grade Ore Stockpile kt	3,386	2,706	896	1,968	1,614	1,627	481	36,711
Pit to Rock Storage (kt)	23,317	23,155	25,166	134,197	45,544	20,813	8,174	408,000
Total to the Skpl/StorageArea (kt)	26,704	25,861	26,062	136,166	47,158	22,441	8,656	444,711
Total Material Movement (kt)	37,766	37,253	37,251	191,181	102,327	78,067	33,649	746,260
Ore & LG to Mill & Stockpiles (kt)	14,448	14,099	12,084	53,688	56,783	57,254	16,922	326,411
Waste to Dump (kt)	23,317	23,155	25,166	134,197	45,544	20,813	8,174	408,000
Total Production (w/o Skpl Rhdl) (kt)	37,766	37,253	37,251	187,886	102,327	78,067	25,096	734,411
Strip Ratio (Waste Mined/Ore Mined)	1.61	1.64	2.08	2.5	0.8	0.36	0.48	1.25
Strip Ratio (Waste Mined/Ore Milled)	2.11	2.03	2.25	2.44	0.83	0.37	0.33	1.35
Weighted Ave Bench Location (m)	1419	1361.2	1372.7	1422.7	1312.6	1222.7	1139.8	

Milling Schedule – Summary to Period 7								
Period	-1	1	2	3	4	5	6	7
Mill Production								
Mill Feed (kt)	0	10,952	10,744	11,033	11,463	11,063	10,798	11,048
Copper From Pit (t)		48,265	43,418	45,658	46,453	44,949	38,844	42,226
Copper From Stockpile (t)		0	0	0	0	0	0	0
Mill Feed Copper (t)		48,265	43,418	45,658	46,453	44,949	38,844	42,226
000s lbs		106,405	95,720	100,659	102,410	99,095	85,636	93,093
Gold From Pit (kg)		3,444	2,902	3,049	2,685	3,237	2,865	2,533
Gold From Stockpile (kg)		0	0	0	0	0	0	0
Mill Feed Gold (kg)		3,444	2,902	3,049	2,685	3,237	2,865	2,533
troy oz		110,736	93,308	98,037	86,324	104,057	92,104	81,450
Copper Head Grade %		0.441	0.404	0.414	0.405	0.406	0.36	0.382
Gold Head Grade g/t		0.314	0.27	0.276	0.234	0.293	0.265	0.229
Copper Equivalent Head Grade %		0.569	0.505	0.512	0.476	0.516	0.457	0.451
Copper Recovery%		87.50%	87.10%	87.10%	88.10%	86.90%	87.40%	88.30%
Gold Recovery %		60.00%	54.50%	51.90%	44.60%	54.80%	53.60%	44.50%

Milling Schedule – Summary Periods 8 to 28								
Period	8	9	10	11 to 15	16 to 20	21 to 25	26 to 28.3	Total
Mill Production								
Mill Feed (kt)	11,062	11,393	11,189	55,016	55,169	55,627	24,994	301,549
Copper From Pit (t)	51,131	55,424	35,054	152,972	183,934	207,099	63,253	1,058,679
Copper From Stockpile (t)	0	0	0	6,758	0	0	17,164	23,921
Mill Feed Copper (t)	51,131	55,424	35,054	159,729	183,934	207,099	80,417	1,082,600
000s lbs	112,725	122,188	77,281	352,143	405,505	456,576	177,289	2,386,724
Gold From Pit (kg)	3,499	5,054	3,209	11,342	13,133	17,925	6,071	80,949
Gold From Stockpile (kg)	0	0	0	505	0	0	1,321	1,826
Mill Feed Gold (kg)	3,499	5,054	3,209	11,847	13,133	17,925	7,392	82,775
troy oz	112,491	162,476	103,181	380,897	422,225	576,316	237,668	2,661,271
Copper Head Grade %	0.462	0.486	0.313	0.29	0.333	0.372	0.322	0.359
Gold Head Grade g/t	0.316	0.444	0.287	0.215	0.238	0.322	0.296	0.274
Copper Equivalent Head Grade %	0.567	0.663	0.409	0.362	0.407	0.481	0.422	0.453
Copper Recovery%	89.10%	89.20%	83.80%	85.80%	87.30%	87.50%	86.60%	87.20%
Gold Recovery %	49.40%	59.40%	47.00%	47.90%	45.00%	49.40%	49.40%	49.80%

Concentrate Production to Period 7								
Period	-1	1	2	3	4	5	6	7
Copper Produced - Pit (t)		42,250	37,823	39,754	40,942	39,064	33,936	37,277
Copper Produced - Skpl (t)		0	0	0	0	0	0	0
Copper Produced (t)		42,250	37,823	39,754	40,942	39,064	33,936	37,277
		93,144	83,385	87,642	90,262	86,121	74,817	82,181
Gold Produced - Pit (kg)		2,066	1,583	1,582	1,197	1,773	1,537	1,128
Gold Produced - Skpl (kg)		0	0	0	0	0	0	0
Gold Produced (kg)		2,066	1,583	1,582	1,197	1,773	1,537	1,128
troy oz		66,423	50,890	50,854	38,474	57,015	49,407	36,279
Con Produced @ 27% Cu (t)		156,482	140,086	147,238	151,639	144,682	125,691	138,063

Concentrate Production Periods 8 to 28								
Period	8	9	10	11 to 15	16 to 20	21 to 25	26 to 28.3	Total
Copper Produced - Pit (t)	45,533	49,425	29,362	131,384	160,537	181,201	55,163	923,650
Copper Produced - Skpl (t)	0	0	0	5,613	0	0	14,513	20,126
Copper Produced (t)	45,533	49,425	29,362	136,997	160,537	181,201	69,676	943,776
	100,382	108,963	64,732	302,026	353,923	399,481	153,610	2,080,669
Gold Produced - Pit (kg)	1,728	3,002	1,507	5,491	5,907	8,860	3,163	40,523
Gold Produced - Skpl (kg)	0	0	0	190	0	0	488	678
Gold Produced (kg)	1,728	3,002	1,507	5,681	5,907	8,860	3,651	41,201
troy oz	55,547	96,518	48,444	182,638	189,925	284,856	117,368	1,324,637
Con Produced @ 27% Cu (t)	168,641	183,058	108,750	507,401	594,587	671,125	258,065	3,495,507

35 Appendix B - Geotechnical Flowsheet Map