-----Original Message-----
From: Roderick Bell-Irving [mailto:RoderickBell-Irving@tasekomines.com]
Sent: October 13, 2009 11:53 AM
To: Prosperity Review [CEAA]; bill.layton <email address removed>
Cc: Katherine Gizikoff
Subject: RE: Prosperity Gold-Copper Mine Project - Update to Interested Parties

Colette:

I went into the Sedar website and searched for the October 15,2007 NI 43-101 filing for the Prosperity Project. I found 2 filings for that date one related to our Gibraltar property and the other for the Prosperity Project.

I have attached the Prosperity filing for ease of reference.

I would be pleased to help further should either yourself or Mr. Layton require it.

Regards,

Rod

Roderick Bell-Irving, R.P.Bio.
Manager, Environmental Project Development Taseko Mines Limited Suite 300 - 905 West Pender Street Vancouver, B.C. V6C 1L6
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Taseko Mines Limited

TECHNICAL REPORT

EXECUTIVE SUMMARY

**FEASIBILITY STUDY OF
THE PROSPERITY GOLD-COPPER PROJECT**

BRITISH COLUMBIA, CANADA

**QUALIFIED PERSON:
SCOTT JONES, P.ENG.**

**Effective Date: September 21, 2007
Report Date: October 15, 2007**

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Abbreviation	**Unit or Description**
Ag	silver
amsl	above mean sea level
Au	gold
B.C.	British Columbia, Canada
BCEA	British Columbia Environmental Assessment Act
BE	Break Even
BH	Brook Hunt and Associates
BME	Bloomsbury Mineral Economics
ERA	Environmental Risk Assessment
C$	Canadian Dollars
CEAA	Canadian Environmental Assessment Act
CRU	Copper Research Unit
Cu	copper
DFO	Department of Fisheries
FLAC	Computer model
G&A	General and Administration
GCL	Giroux Consultants Ltd.
gpt	grams per tonne
Gwh	Gigawatt-hour
ha	hectare
IRR	internal rate of return
km	kilometre
kV	kilovolt
KP	Knight Piesold Consulting
lb	pound (weight)
m	metre
MIBC	Collector Reagent
mPa	megaPascal
Mt	million tonnes
µm	micron
NI	National Instrument 43-101
NMS	Nilsson Mine Services
NPV	net present value
NSR	net smelter return
NTS	National Topographic System
oz	Troy ounce
%	percent
PAG	Potentially Acid Generating
RA	Responsable Agency
RMR	Rock Mass Rating
SAG	Semi Autogenous Grinding
SG	specific gravity
SIBX	Collector Reagent
stpd	short tons per day
t	tonne (metric)
US$	United States Dollars
TC	Treatment Charge
TCRC	Treatment and Refining Charge
TCRCPP	Treatment, Refining and Price Participation
TWC 314	Mill Reagent
TWC 401	Mill Reagent

1. Summary

Taseko Mines Limited ("Taseko" or the "Company") has received positive results from a feasibility level study of its 100% owned Prosperity gold-copper project (the "Prosperity Project"), indicating that the property hosts proven and probable reserves of 487 million tonnes grading 0.43 gpt Au and 0.22% Cu at a C$5.25 net smelter return (NSR/t) per tonne pit-rim cut-off. This report discusses the factors that drove that study and summarizes the outcome.

The Prosperity Project is located 125 km southwest of the City of Williams Lake in the Cariboo-Chilcotin region of British Columbia, Canada.

The feasibility study was done using long term metal prices of US$1.50/lb for copper, US$575/oz for gold, and an exchange rate of US$0.80/C$1.00.

Project Highlights

- Located near existing infrastructure in south-central British Columbia
- Pre-tax net present value of C$260 million at 7.5% discount rate
- Pre-tax internal rate of return of 12% with a 6 year payback from start of production
- 20 year mine life at a milling rate of 70,000 tonnes/day
- Life of mine strip ratio of 0.8:1
- Total pre-production capital cost of C$807 million
- Operating cost of C$6.26 per tonne milled over the life of mine
- Mine site production costs net of gold credits of US$0.43/lb Cu

The mineral reserves estimated from the study are as follows:

Table 1-1
Prosperity Mineral Reserves

at C$5.25 NSR/t Pit-Rim Cut-off					
Category	**Tonnes (millions)**	**Gold (gpt)**	**Copper (%)**	**Recoverable Gold Ounces (millions)**	**Recoverable Copper Pounds (billions)**
Proven	286	0.47	0.25	3.0	1.3
Probable	201	0.37	0.18	1.7	0.7
Total	**487**	**0.43**	**0.22**	**4.7**	**2.0**

The reserve estimate takes into consideration all geologic, mining, milling, and economic factors, and is stated according to Canadian standards (NI43-101).

Taseko carried out ongoing and systematic exploration programs on the Project from 1991 – 1999, increasing drilling to 156,339 m in 470 holes, outlining a large porphyry gold-copper deposit. The Company and its consultants also carried out progressive engineering, metallurgical and environmental studies. Kilborn SNC Lavalin conducted a feasibility level study of the Prosperity Project in 2000.

Taseko re-initiated work on the Prosperity Project in late 2005. A mill redesign and project cost review was completed by SNC Lavalin in 2006. Taseko utilized information from the 2000 feasibility level study, and the 2006 revised process design and scoping level capital and operating costs to prepare a pre-feasibility study. The positive results of the pre-feasibility study were announced in a Taseko News Release dated January 11, 2007 and summarized in the 43-101 Technical Report dated February 25, 2007.

Consistent with the recommendations of the February 25, 2007 Technical Report, HATCH, Knight Piesold Consulting, and Taseko Mines Limited have completed a feasibility study incorporating the 2000 SNC Feasibility Study, 2006 SNC Lavalin Mill Redesign, additional revisions to the processing plant and infrastructure, updates to the tailings facility design and pit geotechnical analysis, and revisions to the design and scheduling of the open pit.

This report has been prepared to document the feasibility study results announced in a Taseko News Release dated September 24, 2007 in the format prescribed in National Instrument 43-101.

Pre-Production and Mine Plan

The feasibility level study incorporates activities during a pre-production period of two years which include construction of the electricity transmission line; upgrading and extension of current road access and mine site clearing; site infrastructure, processing, and tailings starter dam construction; removal and storage of overburden; and pre-production waste development.

The mine plan utilizes a large-scale conventional truck shovel open pit mining and milling operation. Following a one and a half year pre-strip period, total material moved over years 1 through 17 averages 146,000 tonnes/day at a strip ratio of 1.2:1. A declining net smelter return cut-off is applied to the mill feed, which defers lower grade ore for later processing. The lower grade ore is recovered from stockpile for the final 3 years of the mine plan. The life of mine strip ratio including processing of lower grade ore is 0.8:1.

Processing and Infrastructure

The Prosperity processing plant has been designed with a nominal capacity of 70,000 tonnes/day. The plant consists of a single 12-m diameter semi-autogenous grinding (SAG) mill, two 7.9-m diameter ball mills, followed by processing steps that include bulk rougher flotation, regrinding, cleaner flotation, thickening and filtering to produce a copper-gold concentrate. Expected life-of-mine metallurgical recovery is 87% for copper and 70% for gold, with annual production averaging 107 million pounds copper and 247,000 ounces gold over the 19 year mine life.

The copper-gold concentrate will be hauled with highway trucks to an expanded load-out facility at the Gibraltar Mines Ltd.'s existing facility near Macalister for rail transport to various points of sale, but mostly through the Port of Vancouver for shipment to smelters/refineries around the world.

Power will be supplied via a new 124 km long, 230 kV transmission line from Dog Creek on the BC Hydro Grid. Infrastructure would also include the upgrade of sections of the existing road to the site, construction of a short spur to the minesite, an on-site camp, equipment maintenance shop, administration office, concentrator facility, warehouse, and explosives facilities.

Based on this study, the project would employ up to 450 permanent hourly and staff personnel. In addition, approximately 60 contractor personnel would be employed in areas including catering, concentrate haulage, explosives delivery, and bussing.

Mineral Resources

The Proven and Probable Reserves above are included in the following Measured and Indicated Mineral Resources. The Mineral Resources are as outlined by drilling to date, and estimated at a 0.14% Cu cut-off.

Table 1-2
Prosperity Mineral Resources

at 0.14% Copper Cut-off			
Category	**Tonnes (millions)**	**Gold (gpt)**	**Copper (%)**
Measured	547.1	0.46	0.27
Indicated	463.4	0.34	0.21
Total	**1,010.5**	**0.41**	**0.24**

Conclusion and Recommendation

The Project is technically and economically viable under the assumptions of the 2007 Feasibility Study.

With respect to the February 25, 2007 Technical Report on the pre-feasibility study for the Project, there are no material changes with respect to reserves, mining method, processing method, or utilization of technology. The outcome of the base case financial model is within the range of and is consistent with the sensitivity analysis outlined in the February 25, 2007 Technical Report.

The mining claims are 100% owned by Taseko, are not subject to any royalties or carried interests and are currently in good standing until the year 2008. The property is located within territory that is the subject of an aboriginal Rights and Title case between the Tsilhqot'in National Government and the Province of British Columbia currently before the B.C. Supreme Court. The outcome and implications of this case are unknown.

A comprehensive audit and verification program of the geology and assay results in 1998 found the geological work for the Prosperity to be done in a professional manner and according to industry standards.

The project proposes to utilize mining equipment and operating practices that are tried and true in this general location. The open pit design has been based on geotechnical investigations, recommendations and design criteria, provided by Knight Piesold Consulting (KP), and incorporates adequate stability design and dewatering parameters. A revised KP recommendation to single bench in one sector of the pit should be incorporated in the final pit design prior to mining. It is recommended that the current ultimate pit shell, while adequate for definition of a reserve and a valid mine plan warrants a re-evaluation as an optimum pit using the outcomes of the 2007 feasibility study as input parameters in order to bring the optimization up to date. This would also provide the opportunity to incorporate the latest KP pit slope recommendations in the northeast sector.

The Prosperity concentrator has been designed based on suitable metallurgical testwork and incorporates proven technologies and equipment. There may be an opportunity to improve the project economics through modifications in the processing flow sheet, allowing a coarser primary grind and/or staged regrinding. It is recommended that this be undertaken with additional diamond drilling and metallurgical testwork as a first step in further optimization of the process flow sheet. Other opportunities to potentially improve the economics of the project were also identified in the course of the feasibility study as outlined in Section 20. These should be prioritized in terms of economic impact and further investigated in a stepped approach to any subsequent engineering work.

The tailings storage facility has been designed by KP incorporating adequate site characteristics, geotechnical, hydrogeological, and water management considerations for the purposes of the feasibility study. The construction of the tailings dams was based on the use of only non acid generating material. There may be the opportunity to utilize some potentially acid generating (PAG) material in water saturated upstream sections of the dams. Future pit:dam construction material balances should investigate the opportunity to reduce the quantity of PAG requiring haulage requirements.

In the opinion of the authors, the geological interpretation, resource model, mine plan, metallurgical testwork, concentrator design and supporting infrastructure are suitable for this Reserve estimate.

The project is currently in the harmonized British Columbia *Environmental Assessment Act* (EA Act) and *Canadian Environmental Assessment Act* (CEAA) review process. Based on the technical and economic viability of the project demonstrated in the feasibility study, this work should continue.

The Feasibility Study was prepared to quantify the Prosperity project's capital and operating cost parameters and to determine the project's technical and economic viability. The capital and operating cost estimates, which were used, have been developed based on detailed capital cost to production level relationships.

The following are the principal risk factors and uncertainties which, in the author's opinion, are likely to most directly affect the ultimate feasibility of the Prosperity project. The mineralized material at the Prosperity project is currently classified as a measured and indicated resources, and a portion of it qualifies under Canadian mining disclosure standards as a proven and probable reserve, but readers are cautioned that no part of the Prosperity project's mineralization is yet considered to be a reserve under US mining standards as all necessary mining permits would be required in order to classify the project's mineralized material as an economically exploitable reserve.

Although feasibility level work has been done to confirm the mine design, mining methods and processing methods assumed in the Feasibility Study, construction and operation of the mine and processing facilities depend on securing environmental and other permits on a timely basis.

Additional permits, when required, have yet to be applied for and there can be no assurance that required permits can be secured or secured on a timely basis or that third party opposition will not exist, which may delay or otherwise affect the Company's ability to secure required permits.

Although costs, including design, procurement, construction and on-going operating costs and metal recoveries have been established at a level of detail required for a feasibility study, these could be materially different from those contained in the Feasibility Study. There can be no assurance that these infrastructure facilities can be developed on a timely and cost-effective basis. Energy risks include the potential for significant increases in the cost of fuel and electricity. The Feasibility Study assumes specified, long-term prices levels for gold and copper. The prices of these metals have historically been volatile, and the Company has no control of or influence on the prices, which are determined in international markets. There can be no assurance that the price of gold and copper will continue at current levels or that these prices will not decline below the prices assumed in the Feasibility Study. Prices for gold and copper have been below the price ranges assumed in Feasibility Study at times during the past ten years, and for extended periods of time. The project will require major financing, probably a combination of debt and equity financing. Although interest rates are at historically low levels, there can be no assurance that debt and/or equity financing will be available on acceptable terms.

Other general risks include those ordinary to very large construction projects, including the general uncertainties inherent in engineering and construction cost, the need to comply with generally increasing environmental obligations, and accommodation of local and community concerns.

2. Introduction

This report has been prepared for Taseko Mines Limited to document results of a feasibility level study reported in a News Release dated September 24, 2007 in the format prescribed in National Instrument 43-101.

The Qualified Persons responsible for the content of this report are:

Scott Jones, P.Eng., General Manager of Project Development for Taseko. Mr. Jones has reviewed the methods used to determine grade and tonnage in the geological model, reviewed the long range mine plan, the capital and operating cost estimates, and directed the updated economic evaluation. He has visited the Prosperity property on four occasions in 2006: May 25, July 27, August 16, and August 30, and 4 occasions in 2007: June 21 and 22, July 4 and 10.

G.H. Giroux, P.Eng., MASc., independent consulting geological engineer. Mr. Giroux was responsible for the Resource Estimation Section completed in Vancouver during 1998 and amended 1999. He has not visited the property.

Lawrence Melis, P.Eng., consulting process engineer, working for Melis Engineering Ltd. Mr. Melis was responsible for the metallurgical testwork completed in the 1990's by Melis Engineering Ltd. He visited in the 1990's to look at core and general site conditions.

The following information has been relied upon as provided by qualified persons who have provided certificates in Section 22:

Giroux, G.H., 1998. A Resource Estimate Update for the Prosperity Project Gold-Copper Deposit. Unpublished Company Report. Taseko Mines Limited, Vancouver, British Columbia.

Giroux, G.H., 1999. Addendum to the March 12, 1998 Resource Estimate for the Prosperity Project Gold-Copper Deposit for Taseko Mines Limited. Unpublished Company Report, Taseko Mines Limited, Vancouver, British Columbia.

Melis Engineering Ltd., 1998. Prosperity Gold-Copper Project Feasibility Study, Volume 4, Appendix E, Metallurgy.

Additional sources of information used for this report are:

2007 Prosperity Feasibility Study by HATCH under the supervision of Steve McMaster, P.Eng

2007 tailings, water balance, and geotechnical studies conducted by Knight Piesold Ltd., under the supervision of Ken Brouwer, P.Eng.

Mineral reserves, mine planning and design aspects developed by Nilsson Mine Services (NMS) in conjunction with staff at Taseko Mines Limited. The Mineral Reserves are based on the 2007 Feasibility Study completed by HATCH, Knight Piesold Consulting, and Taseko Mines Limited.

The Feasibility Study relied on historical work in a number of areas:

Sampling, Analysis and Quality Assurance/Quality Control by Eric Titley, P.Geo.

Metallurgical testwork, completed in the 1990's, conducted by Lakefield Research Limited (now called SGS Lakefield) under the supervision of Melis Engineering Ltd. This work was reviewed by SGS Lakefield, SNC Lavalin, and Taseko for the purposes of the 2006 Mill Redesign and Costing Study and was accepted for the purposes of this feasibility study. 2006 mill redesign work by SNC Lavalin was supervised by Greg McCunn, P.Eng.

2000 mill process and plant design work, done in accordance with criteria provided by Melis Engineering Ltd. and completed by Kilborn SNC Lavalin under the supervision of Ross Banner, P.Eng.

All of the above persons are independent of the Company except for Mr McCunn, Mr Titley, Mr Jones, and Mr Banner

3. Reliance on Other Experts

In preparing this Technical Report, the author relied on the following information which may not be by qualified persons:

Information on History, Property, Deposit, and Mineralization was acquired from Taseko Mines Limited.

The author is not an expert on mineral tenure and has depended on the information received from the Company.

4. Property Description and Location

The Prosperity property is located approximately 125 km southwest of the City of Williams Lake in south-central British Columbia, Canada (Figures 4-1) at latitude 51 degrees 28’ N and longitude 123 degrees 37’ W.

The Property deposit is located in the Clinton Mining Division on the N.T.S. map sheet 92 O/SE. The 85 square km property is comprised of 118 mineral claims depicted in Figure 4-2. The claims are 100% owned by Taseko and are not subject to any royalties or carried interests.

The mineral claims are currently in good standing until the year 2008.

Figure 4-1
Prosperity Location

Figure 4-2
Claim Map

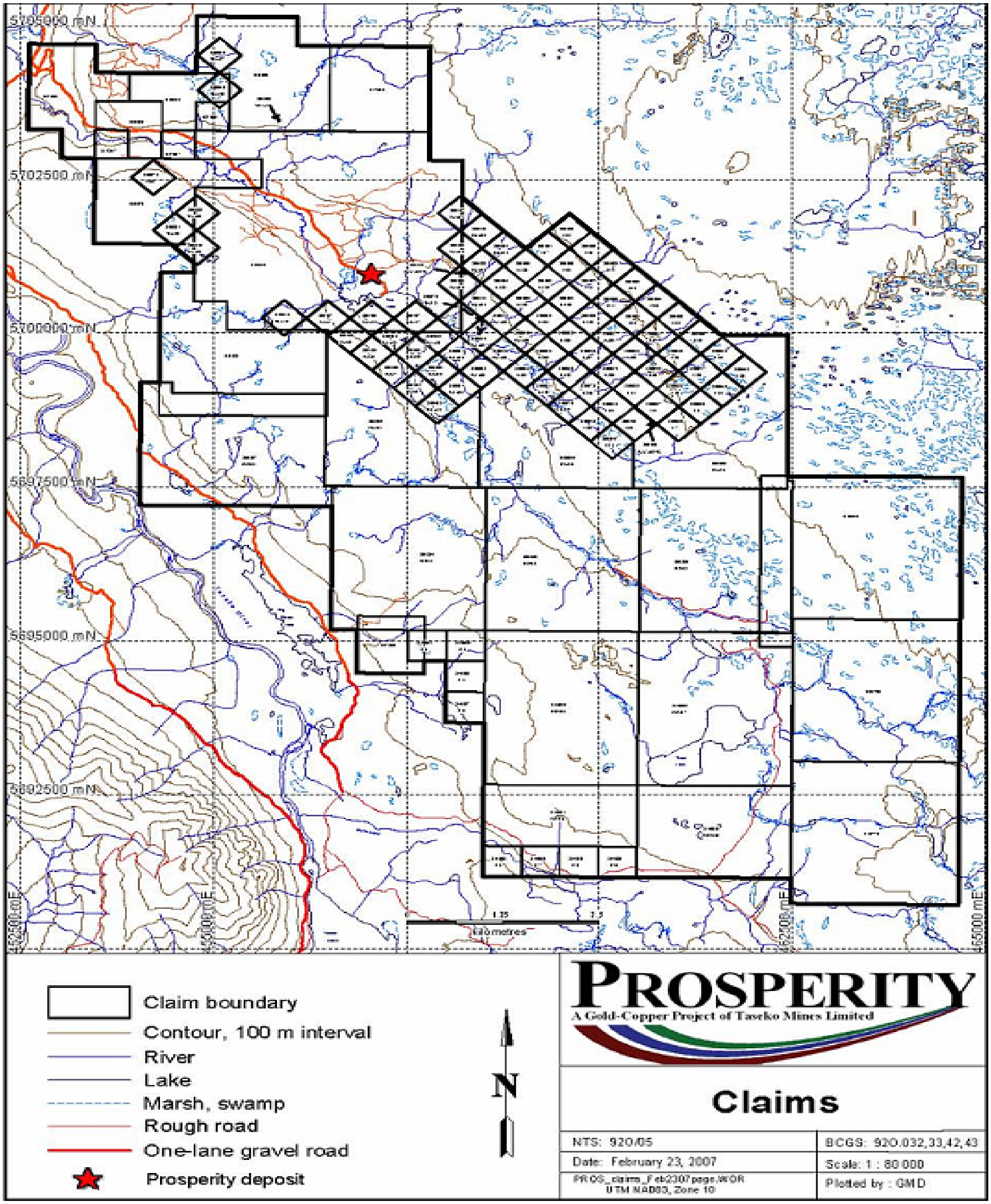


Table 4-1
Prosperity Mineral Claims

Taseko Mines Limited
(Corporate Free Miner 126450)
Prosperity Project, Clinton Mining Division
Mineral Claims

Tenure No.	Claim Name	Owner	Map No.	Good To Date	Status	Area (ha)		Tenure No.	Claim Name	Owner	Map No.	Good To Date	Status	Area (ha)
208019	BCC #5 FR.	126450 (100%)	092O	2009/feb/06	GOOD	25.0		209557	L43	126450 (100%)	092O	2009/aug/17	GOOD	25.0
208020	BCC #6 FR.	126450 (100%)	092O	2009/feb/25	GOOD	25.0		209558	L44	126450 (100%)	092O	2009/aug/17	GOOD	25.0
208024	EKO 1	126450 (100%)	092O	2009/apr/02	GOOD	500.0		209559	L45	126450 (100%)	092O	2009/aug/17	GOOD	25.0
208025	EKO 2	126450 (100%)	092O	2009/apr/02	GOOD	500.0		209560	L46	126450 (100%)	092O	2009/aug/17	GOOD	25.0
208026	EKO 3	126450 (100%)	092O	2009/apr/02	GOOD	500.0		209561	L47	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209279	TKO 2	126450 (100%)	092O	2008/jan/08	GOOD	500.0		209562	L48	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209324	FISH 1	126450 (100%)	092O	2008/feb/14	GOOD	500.0		209572	K66	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209325	FISH 2	126450 (100%)	092O	2008/feb/14	GOOD	500.0		209578	K116	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209326	FISH 3	126450 (100%)	092O	2008/feb/14	GOOD	500.0		209579	K117	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209327	FISH 4	126450 (100%)	092O	2008/feb/14	GOOD	500.0		209580	K118	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209487	BJ #1	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209581	K119	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209488	BJ #3	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209582	K120	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209489	BJ #5	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209583	K121	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209490	BJ #7	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209584	K125	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209491	BJ #9	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209585	K126	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209492	BJ #11	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209586	K127	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209496	BJ #16	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209587	K128	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209497	BJ #17	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209588	K129	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209498	BJ #18	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209589	K130	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209499	BJ #19	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209590	K131	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209500	BJ #20	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209591	K132	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209501	BJ #21	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209592	K133	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209502	BJ #22	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209593	K134	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209503	BJ #23	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209594	K135	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209504	BJ #24	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209595	K136	126450 (100%)	092O	2009/aug/17	GOOD	25.0
209509	BJ #29	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209598	TEL #75	126450 (100%)	092O	2009/apr/26	GOOD	25.0
209511	BJ #31	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209611	TK #15	126450 (100%)	092O	2009/may/28	GOOD	25.0
209512	BJ #32	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209619	TK #23	126450 (100%)	092O	2009/may/28	GOOD	25.0
209513	BJ #33	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209621	TK #25	126450 (100%)	092O	2009/may/28	GOOD	25.0
209514	BJ #34	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209622	TK #26	126450 (100%)	092O	2009/may/28	GOOD	25.0
209515	BJ #35	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209640	TK #46	126450 (100%)	092O	2009/may/28	GOOD	25.0
209516	BJ #36	126450 (100%)	092O	2009/jun/25	GOOD	25.0		209656	TK #67	126450 (100%)	092O	2009/may/28	GOOD	25.0
209517	BJ #37	126450 (100%)	092O	2009/jun/25	GOOD	25.0		314004	F 2	126450 (100%)	092O	2009/oct/15	GOOD	25.0
209519	BJ #39	126450 (100%)	092O	2009/jun/25	GOOD	25.0		314005	F 3	126450 (100%)	092O	2009/oct/15	GOOD	25.0
209520	BJ #40	126450 (100%)	092O	2009/jun/25	GOOD	25.0		314006	F 4	126450 (100%)	092O	2009/oct/16	GOOD	25.0
209521	BJ #41	126450 (100%)	092O	2009/jun/25	GOOD	25.0		314007	F 5	126450 (100%)	092O	2009/oct/16	GOOD	25.0
209522	BJ #42	126450 (100%)	092O	2009/jun/25	GOOD	25.0		314008	F 6	126450 (100%)	092O	2009/oct/16	GOOD	25.0
209535	L7	126450 (100%)	092O	2009/aug/17	GOOD	25.0		314009	F 7	126450 (100%)	092O	2009/oct/16	GOOD	25.0
209536	L8	126450 (100%)	092O	2009/aug/17	GOOD	25.0		314010	F 8	126450 (100%)	092O	2009/oct/16	GOOD	25.0
209537	L9	126450 (100%)	092O	2009/aug/17	GOOD	25.0		314025	F 9	126450 (100%)	092O	2009/oct/16	GOOD	25.0
209538	L10	126450 (100%)	092O	2009/aug/17	GOOD	25.0		314026	FISH 10	126450 (100%)	092O	2009/oct/17	GOOD	300.0
209539	L11	126450 (100%)	092O	2009/aug/17	GOOD	25.0		314028	FISH 6	126450 (100%)	092O	2009/oct/16	GOOD	500.0
209540	L12	126450 (100%)	092O	2009/aug/17	GOOD	25.0		314029	FISH 7	126450 (100%)	092O	2009/oct/17	GOOD	500.0
209541	L21	126450 (100%)	092O	2009/aug/17	GOOD	25.0		314031	FISH 9	126450 (100%)	092O	2009/oct/16	GOOD	200.0
209542	L22	126450 (100%)	092O	2009/aug/17	GOOD	25.0		516779		126450 (100%)	092O	2009/oct/17	GOOD	504.4
209543	L23	126450 (100%)	092O	2009/aug/17	GOOD	25.0		516785		126450 (100%)	092O	2009/oct/17	GOOD	504.2
209544	L24	126450 (100%)	092O	2009/aug/17	GOOD	25.0		516849		126450 (100%)	092O	2009/oct/15	GOOD	624.9
209545	L31	126450 (100%)	092O	2009/aug/17	GOOD	25.0		516915		126450 (100%)	092O	2009/may/28	GOOD	80.5
209546	L32	126450 (100%)	092O	2009/aug/17	GOOD	25.0		516926		126450 (100%)	092O	2009/feb/06	GOOD	1,047.2
209547	L33	126450 (100%)	092O	2009/aug/17	GOOD	25.0		516935		126450 (100%)	092O	2008/jan/09	GOOD	362.6
209548	L34	126450 (100%)	092O	2009/aug/17	GOOD	25.0		516970		126450 (100%)	092O	2009/sep/11	GOOD	221.5
209549	L35	126450 (100%)	092O	2009/aug/17	GOOD	25.0		516984		126450 (100%)	092O	2009/aug/17	GOOD	60.4
209550	L36	126450 (100%)	092O	2009/aug/17	GOOD	25.0		516990		126450 (100%)	092O	2009/may/28	GOOD	261.7
209551	L37	126450 (100%)	092O	2009/aug/17	GOOD	25.0		517288		126450 (100%)	092O	2009/oct/15	GOOD	80.7
209552	L38	126450 (100%)	092O	2009/aug/17	GOOD	25.0		517338		126450 (100%)	092O	2009/aug/17	GOOD	161.0
209553	L39	126450 (100%)	092O	2009/aug/17	GOOD	25.0		517347		126450 (100%)	092O	2009/aug/17	GOOD	20.1
209554	L40	126450 (100%)	092O	2009/aug/17	GOOD	25.0		517352		126450 (100%)	092O	2009/may/28	GOOD	181.2
209555	L41	126450 (100%)	092O	2009/aug/17	GOOD	25.0		537996		126450 (100%)	092O	2009/may/28	GOOD	20.1
209556	L42	126450 (100%)	092O	2009/aug/17	GOOD	25.0		537997		126450 (100%)	092O	2009/aug/17	GOOD	20.1

In the late 1990's, Kilborn SNC Lavalin undertook a spot check of claim posts on the property during the drilling verification program. The spot checks concentrated on verifying a number of Legal Corner Posts and Identification Posts. Based on site inspections combined with examination of the Mineral Titles maps and documents as well as discussions with the B.C. Mineral Titles Branch staff in Vancouver, it was Kilborn's opinion that the Taseko claim title to the project area was secure and legal.

In 2005, Taseko converted several of the ground staked legacy claims covering the mineralized area to cell claims as allowed by the amended BC Mineral Tenure Act. The legacy claim conversion consolidated the project holdings and eliminated any internal claim gaps.

The property boundaries have not been legally surveyed.

As this is a new project, there are no existing environmental liabilities on the property.

No permits are required for the feasibility work that is currently underway. For further details see Section 18.8.

The Company does not hold any surface rights.

The property is located within territory that is the subject of an aboriginal Rights and Title case between the Tsilhqot'in National Government and the Province of British Columbia currently before the B.C. Supreme Court.

5. Accessibility, Climate, Local Resources, Infrastructure, and Physiography

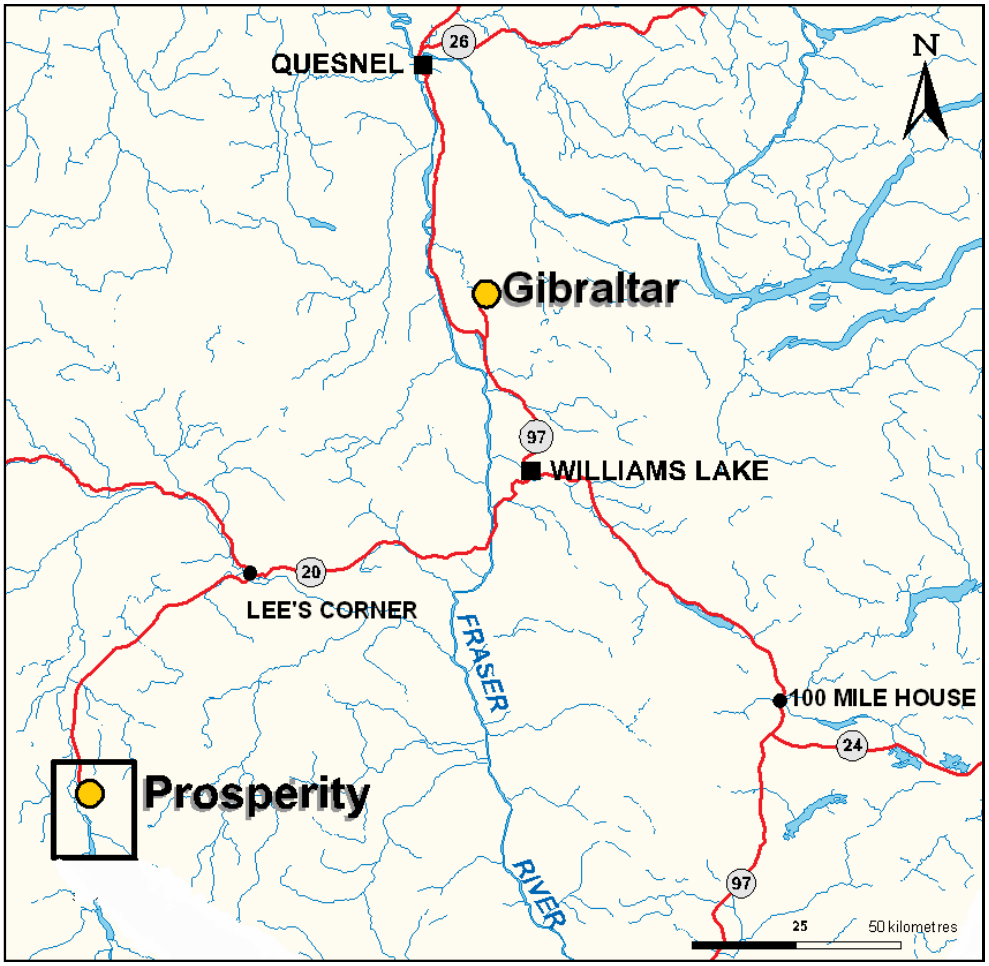
5.1 Access

At present vehicle access to the site from Williams Lake is via Provincial Highway No. 20 to Lees's Corner and forestry resource roadways see (Figure 5-1). The road between Williams Lake and the mine site (approximately 180 km) must be an all-weather road and will comprise a portion of the following:

- Provincial Highway No. 20 - 90 km of 2 lane, paved road;
- Taseko Lake (Whitewater) Logging Road - 68 km of one-lane, 5 m wide, gravel road with turnouts;
- 4500 Road (Riverside Haul Road) - 19 km of one-lane, 5 m wide, gravel road with turn-outs;
- and a new Project Site Access Road - 2.8 km of 2 lane, 8 m wide, gravel road.

This road system will serve as the principal access road both during construction and mine operation.

Figure 5-1
Prosperity Location



5.2 Resources and Infrastructure

The City of Williams Lake is sufficiently close and supplies goods and services to two operating mines in the area: Imperial Metals' Mount Polley Mine and Taseko's Gibraltar Mine.

Multiple high-voltage transmission lines from the existing Peace River hydroelectric power grid are situated 118 km east of the Prosperity Project. A 124 km conventional power line designed to connect to the existing BC Hydro electric power grid will be capable of supplying the required power to service a large mine and mill complex at the Prosperity Project site. BC Hydro has confirmed, through an Interconnection System Impact Study completed by SNC Lavalin in June, 2007, that the supply of power to the Prosperity project is technically viable through a proposed switching station at Dog Creek.

Sufficient water is available on the property for a mining operation.

The Canadian National Railway services Williams Lake and Gibraltar's existing concentrate load-out facilities near Macalister, just north of McLeese Lake, and has the ability to move copper concentrates through to the Pacific Ocean Port of Vancouver.

5.3 Physiography

The property is located on the Fraser Plateau in the Taseko Lakes region on the eastern side of the Chilcotin Mountain Range, which forms part of British Columbia's Coast Mountain Range. The landscape is characterized by the low rounded summits of the Chilcotin Range and moderately sloping upland. The property is located within the Fish Creek and Fish Lake watershed in a broad valley with slopes of moderate relief. Elevations at the site range between 1,450 m and 1,600 m above sea level. Fish Creek in the valley bottom flows into the Taseko River at the north end of the property. The local drainage pattern is typically dendritic.

5.4 Climate

Regionally, the Chilcotin Mountains record lower temperatures and more precipitation than the Fraser Plateau. The warmest and driest lands are generally in the main river valleys.

The Fish Lake area has a moderate continental climate with cold winters and warm summers. Local climatic conditions are moderated primarily by elevation, aspect, physiography, and the proximity of the area to the Chilcotin Mountains.

Climate estimates for the Prosperity project site have been determined using long-term regional data sources and short-term site-specific records.

The annual precipitation at the project site has been estimated to be 527 mm, with 44% falling as snow during the winter months. Annual pond evaporation has been estimated at 452 mm.

The annual mean temperature at the project site has been estimated to be 2 degrees Celsius. The coldest months of December and January average -10 degrees Celsius, and the warmest months of July and August average 13 degrees Celsius. The warmest temperature recorded at the site is 30 degrees Celsius. In July and August, and the coldest is –36 degrees Celsius in February.

6. History

The Prosperity deposit was originally discovered in the early 1930's by prospectors E. Calep and C. Vick, who conducted trenching of feldspar porphyritic dykes with stringers containing copper and gold values about 1.5 km east of the centre of the porphyry deposit as it is now known. In the late 1950's, George Renner did additional work on gold-silver-copper mineralized shear zones located northeast of the deposit. In 1960 Phelps Dodge Corporation located float and subcropping mineralization that indicated a porphyry environment. The company subsequently carried out a program of induced polarization (IP), geochemical and magnetic surveys, and hand trenching. In 1963-64 they conducted a small diamond drilling program comprising 8 short holes north of the presently known deposit. The results were not encouraging and the mineral claims in the area were allowed to lapse.

In 1969, Taseko Mines Limited acquired the property and drilled 12 percussion holes totaling 1,265 m and 6 diamond drill holes totaling 1,036 m immediately to the south of the area where Phelps Dodge had explored. Taseko discovered significant tonnage grading 0.25% to 0.30% copper.

In 1970, Nittetsu Mining Company optioned the property from Taseko Mines Limited and completed 236 m of core drilling in 4 holes before returning the property to Taseko. In 1972, Taseko tested the property with 2 additional diamond drill holes totaling 156 m.

Quintana Minerals Corporation optioned the property from Taseko in 1973 and completed a 23-hole diamond drill program totaling 4,705 m during 1973-74. Vertical drill hole Q73-10, collared in the center of the deposit, intersected 415 m of disseminated and stockwork copper-gold mineralization at an average grade of 0.31% Cu and 0.54 g Au/t. The drill hole was completed, at a depth of 438 m, in mineralization of similar grade.

Bethlehem Copper Corp. optioned the property in 1979 and by 1981 had completed 3,225 m of percussion drilling in 36 holes and 10,445 m of diamond drilling in 37 holes.

Following the corporate merger of Bethlehem Copper Corp. and Cominco Ltd., Cominco acquired the Bethlehem option agreement on the property. Cominco continued to drill the property, completing 1,620 m of percussion drilling in 19 holes and 3,707 m of diamond drilling in 29 holes over the period 1982 to 1989.

A summary of this historical drilling is shown in Table 6-1.

Table 6-1
Drilling Summary 1963 – 1989

Year	Company	Percussion Drilling		Diamond Drilling		All Drilling	
		No. of Holes	(m)	No. of Holes	(m)	No. of Holes	(m)
1963	Phelps-Dodge	0	0	6	611	6	611
1964	Phelps-Dodge	0	0	2	112	2	112
1969	Taseko	12	1,265	6	1,036	18	2,301
1970	Nittetsu	0	0	4	236	4	236
1972	Taseko	0	0	2	156	2	156
1973	Quintana	0	0	14	2,972	14	2,972
1974	Quintana	0	0	9	1,733	9	1,733
1979	Bethlehem	14	1,106	0	0	14	1,106
1980	Bethlehem	22	2,119	0	0	22	2,119
1981	Bethlehem	0	0	37	10,446	37	10,446
1982	Cominco	19	1,620	12	707	31	2,327
1984	Cominco	0	0	5	1,003	5	1,003
1989	Cominco	0	0	12	1,997	12	1,997
Total Drilling		67	6,110	109	21,009	176	27,119

Cominco work programs also included 50 line km of induced polarization, magnetic and soil geochemical surveys. The induced polarization survey outlined a 2 km by 3 km east-west trending zone of high chargeability. Also undertaken was a limited metallurgical testwork program which focused on achieving high copper recovery, with little emphasis on gold recovery, using a conventional copper flotation.

In 1990, Cominco Ltd. reported a drill–indicated mineral resource of 208 million tonnes at an average grade of 0.23% Cu and 0.41 gpt Au to 360 m below surface. Many of the drill holes used to estimate this resource bottomed in resource grade gold-copper mineralization.

By agreement dated August 10, 1979, the Prosperity project was optioned by Taseko to Bethlehem Copper Corporation. Cominco Ltd. acquired the Bethlehem option agreement on the property with the merger of Bethlehem and Cominco. Under that agreement, Cominco was granted an exclusive option to acquire an 80% interest in the Prosperity project by giving notice to Taseko before November 30, 1984, of Cominco's intention to proceed with commercial production from the Prosperity project. Cominco was entitled to extend its option on a yearly basis if Cominco concluded that it was not economically feasible to place the project in commercial production and if an independent consultant supported this conclusion. Cominco extended the option in 1984 and again in 1985, based on an evaluation of the Prosperity project prepared by Cominco in 1984. Cominco's extension of the option was supported by a June 1986 report from Wright Engineers Limited of Vancouver, British Columbia. That report, based on data obtained from mining and metallurgical studies provided by Cominco, confirmed Cominco's evaluation that the Prosperity project was not commercially feasible at that time.

Taseko subsequently sued Cominco, arguing that Cominco had not complied with all of the terms necessary to enable it to extend the option, and specifically had not had a proper feasibility study prepared to determine the economic viability of the Prosperity project. Cominco successfully defended its position at the trial and appeal courts. Taseko and Cominco resolved their dispute by entering into a settlement agreement dated April 25, 1991 (the "First Settlement Agreement"). Cominco entered into the First Settlement Agreement in consideration of the issuance by Taseko of 1,000,000 common shares (issued over the period May 31, 1991 to March 31, 1992), and for the grant of a general release of Cominco by Taseko from the litigation claims made by Taseko against Cominco. The First Settlement Agreement provided that Taseko had a five-year option to sell the Prosperity Project, either directly or by way of a take-over of Taseko, in which event the proceeds would be split in a certain ratio with a maximum of CDN$48 million to Cominco.

By agreement dated December 1, 1993. Taseko acquired the exclusive right to purchase from Cominco all of Cominco's residual interest in the project. Taseko acquired the balance of a 100% interest in the Prosperity Project by paying to Cominco CDN$2,000,140 from working capital and issuing to Cominco 1,636,364 common shares from treasury. Cominco sold 1,607,400 of these shares to net CDN$23 million and 28,964 shares were returned to treasury in April, 1994. As a result of the Second Settlement Agreement, Taseko acquired 100% of he Prosperity project free whatsoever of any royalties or third party interests.

There has been no production from the property.

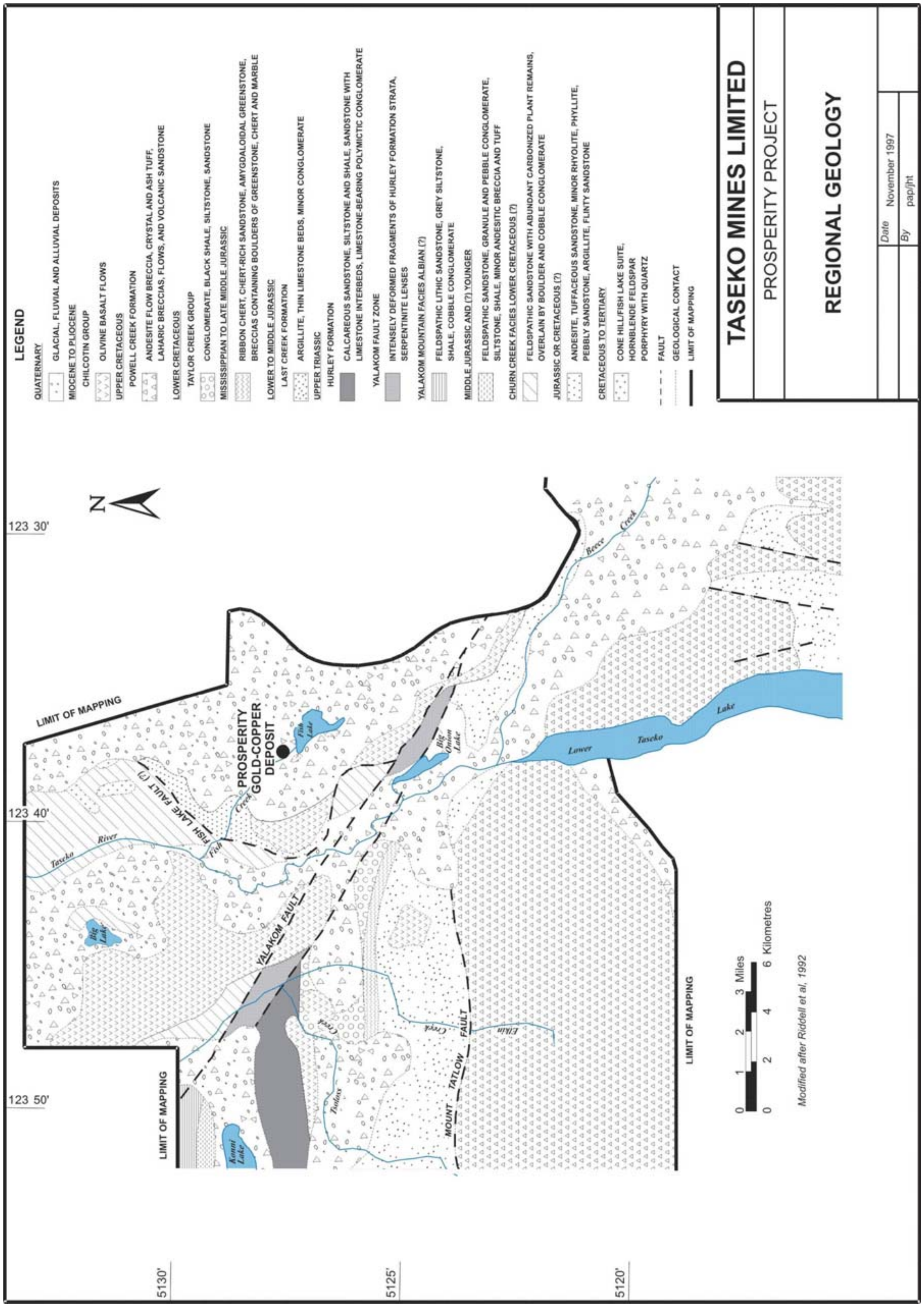
7. Geological Setting

The Prosperity project is located within the western-most portion of the Intermontaine Belt at the boundary between the Intermontaine and Coast morphologic belts. The surrounding area is underlain by poorly exposed, Late Paleozoic to Cretaceous litho tectonic assemblages which have been intruded by plutons of Mid-Cretaceous to Early Tertiary age. The main Coast Plutonic Complex is 50 km southwest of the project area (Figure 7 -1).

The Yalakom Fault is the major fault in the region and lies to the southwest of the Prosperity deposit. Estimates of Eocene dextral strike-slip offsets for the Yalakom Fault have been postulated variously as ranging from 80 to 190 km (Tipper, 1969), 125 to 175 km (Kleinspehn, 1985) or 115 km (Riddell et al., 1993). It may have imparted some related structural controls that are important to the localization of mineralization at the deposit.

Northeast of the Yalakom Fault, feldspathic lithic sandstones, conglomerates and shales comprise most of the exposed rocks. These sedimentary rocks were correlated with the Lower Cretaceous Jackass Mountain Group by Riddell et al. (1993) and Schiarizza et al. (1993). The poorly exposed andesitic volcaniclastic and volcanic rocks that host the Prosperity deposit may correlate with a succession of andesites, tuffaceous sandstones, argillites and siltstones that crop out near the mouth of Fish Creek. Here, fossils collected from shales intercalated with the volcanic rocks were assigned Hauterivian (Early Cretaceous) ages (Riddell et al., 1993). The sedimentary rocks, which have been encountered in drill holes to the south of the Prosperity deposit, are likely of similar age. Sub-horizontal Miocene plateau basalts and non-marine sedimentary rocks of the Chilcotin Group form an extensive post-mineral cover in the immediate project area.

Figure 7 -1 Regional Geology



8. Deposit Types

The Prosperity Gold-Copper deposit subcrops under a 5 to 65 m thick blanket of surficial cover at the north end of Fish Lake. Interpretation of deposit geology (Caira and Findlay, 1994 and Brommeland et al., 1998) is based on a 1963 to 1997 drill hole data base consisting of 402 diamond drill holes totaling 150,185 m and 68 percussion drill holes totaling 6,309 m as outlined in section 10.1

The Prosperity deposit is predominantly hosted in Cretaceous andesitic volcaniclastic and volcanic rocks which are transitional to a sequence of sparsely mineralized, volcanically-derived sedimentary rocks to the south (Figure 8-1). The andesitic volcaniclastics are comprised of coarse-grained crystal tuff and ash tuff, and thinly bedded tuff with lesser lapilli tuff. The upper eastern portion of the deposit is hosted by subvolcanic units of crowded feldspar porphyritic andesite and thick feldspar and hornblende porphyritic flows (Table 8-1).

In the western portion of the deposit, the multi-phase Fish Creek Stock has intruded into a thick sequence of andesite flows which overlay volcaniclastic rocks. The steeply south-dipping, oval quartz diorite stock, which is approximately 265 m wide by 800 m long, is surrounded by an east-west trending swarm of subparallel quartz-feldspar porphyritic dikes, which also dip steeply to the south. Together the stock and dikes comprise the Late Cretaceous Fish Lake Intrusive Complex that is spatially and genetically related to the deposit. Post-mineralization porphyritic diorite occurs as narrow dikes that cross-cut all units within the deposit. They represent the final intrusive phase of the emplacement of the Fish Lake Intrusive Complex.

The deposit area is overlain by a variably thick overburden cover consisting of Wisconsinian glacial till, Miocene to Pliocene basalt flows, and Tertiary colluvium and lacustrine sediments (Figure 8–2).

Figure 8–1 Geology at Overburden-Bedrock Interface

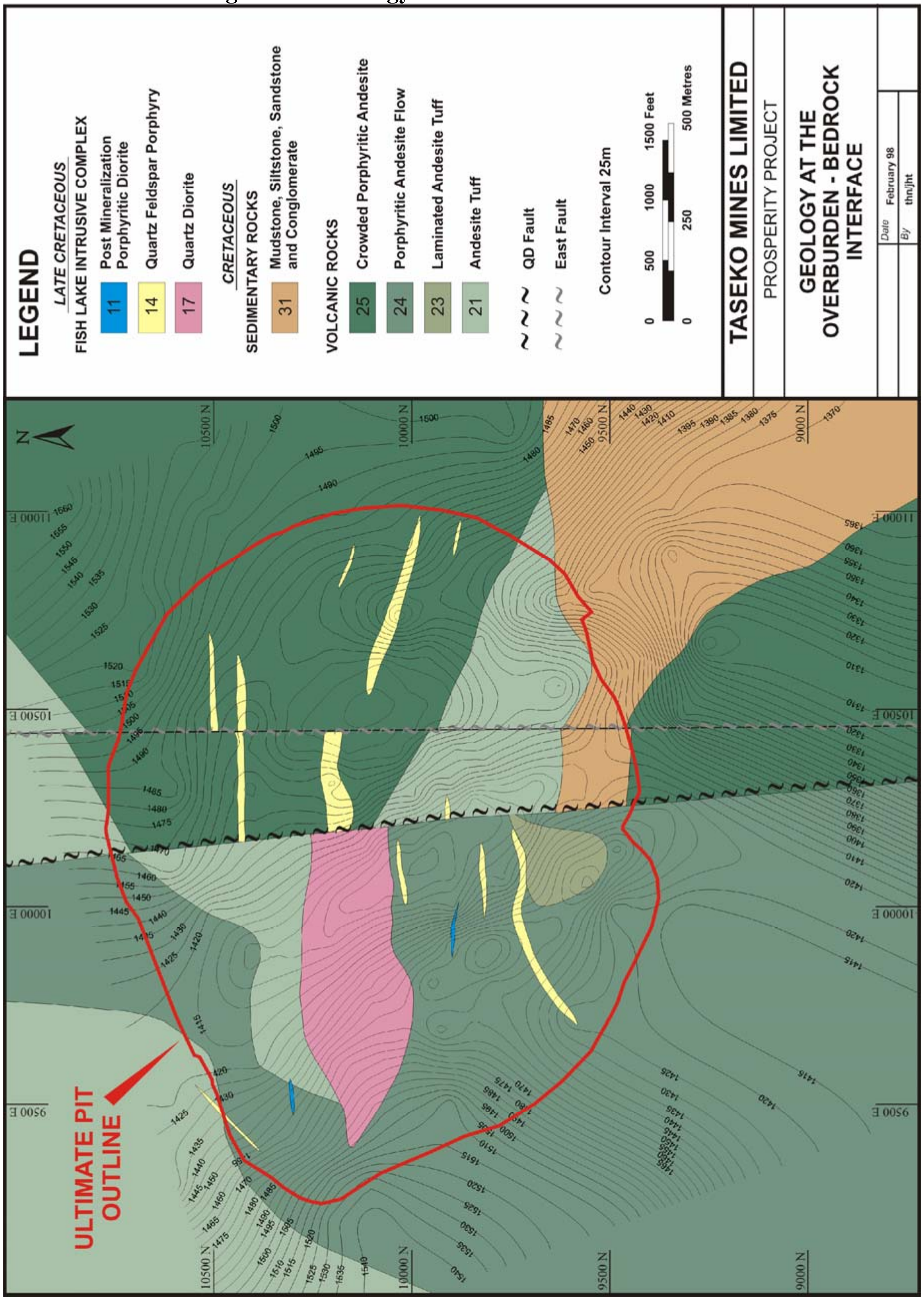
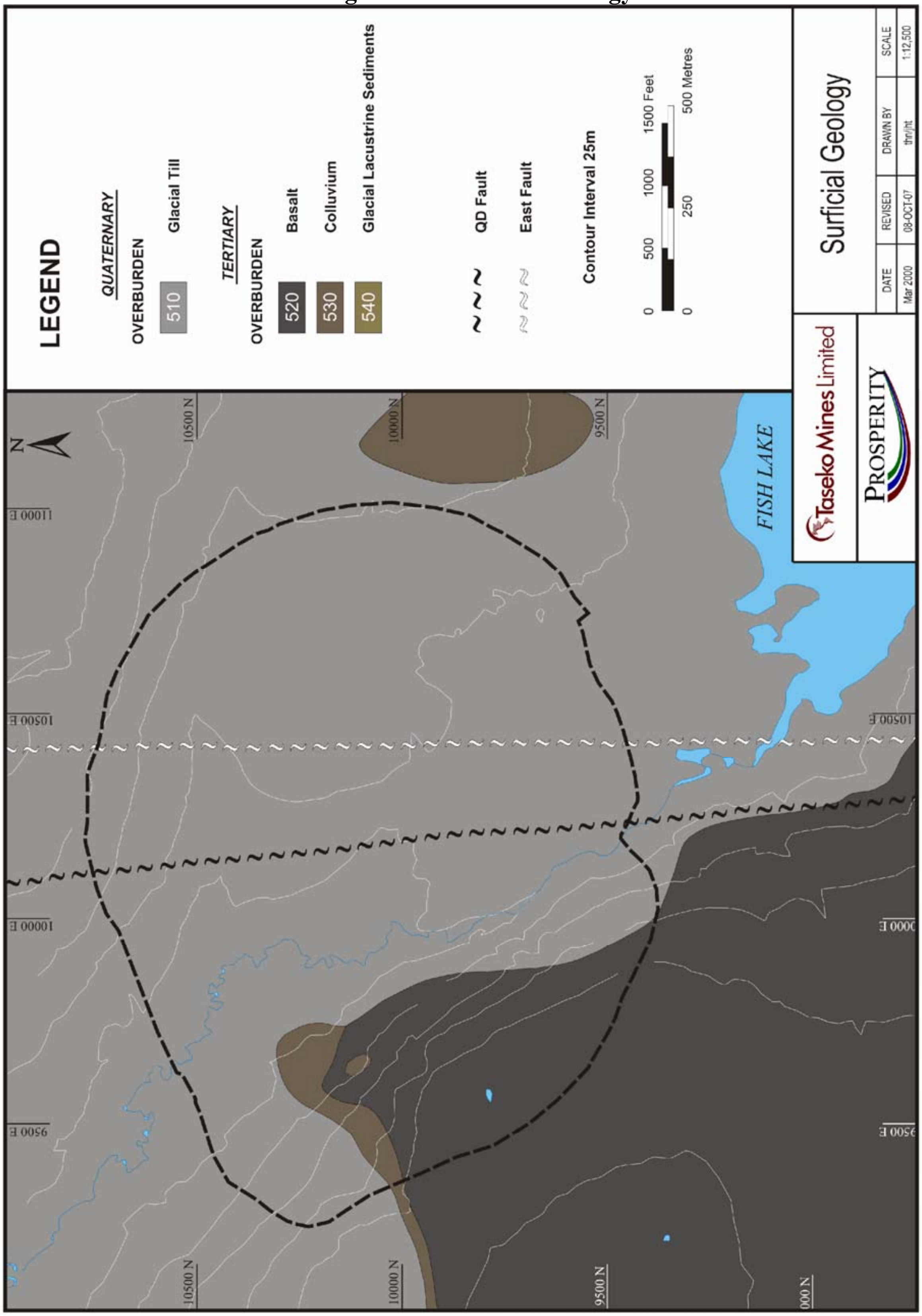


Table 8-1
Prosperity Gold-Copper Project Geology Codes

CENOZOIC	**QUATERNARY COVER**		
	Pleistocene Glacial Till		
	511	TILB	Basal Till
	512	CLAYU	Clay
	513	SICLU	Silt/Clay Mix
	514	SILTU	Silt
	515	GRAVU	Gravel
	TERTIARY COVER		
	Miocene to Pliocene Basalt Flows		
	520	BSLT	Basalt
	Colluvium		
	531	FANL	Fanglomerate – Limonitic
	532	FAN	Fanglomerate
	Glacial Lacustrine Sediments		
	541	GRAV	Gravel
	542	SICL	Silt/Clay Mix
	543	CLAY	Clay
	544	SILT	Silt
MESOZOIC	**LATE CRETACEOUS FISH LAKE INTRUSIVE COMPLEX**		
	11	PMPD	Post Mineralization Porphyritic Diorite
	12	INBX	Igneous Breccia
	13	FP	Feldspar Porphyry
	14	QFP	Quartz Feldspar Porphyry
		FISH CREEK STOCK (QD)	
	15	QD3	Subporphyritic to Equigranular Quartz Diorite
	16	QD2	Seriate Porphyritic Quartz Diorite
	17	QD1	Heterogeneous Fine Porphyritic Quartz Diorite
	CRETACEOUS SEDIMENTARY ROCKS		
	31	SEDS	Mudstone, Siltstone, Sandstone and Conglomerate
	CRETACEOUS VOLCANIC ROCKS		
	25	SUBV	Crowded Porphyritic Andesite
	24	FLOW	Porphyritic Andesite Flow
	23	BEAT	Laminated Andesite Tuff
	22	DEBF	Andesite Lapilli Tuff and Debris Flow
	21	MAT	Andesite Tuff (ash tuff)
	21	FAXT	Andesite Tuff (mainly crystal tuff)

Figure 8–2 Surficial Geology



The deposit is oval in plan and is approximately 1500 m long, 800 m wide and extends to a maximum depth of 880 m. A central potassium silicate alteration zone is co-extensive with the gold-copper mineralization. Along the deposit's eastern margin, a discontinuous zone of phyllic alteration is developed at the boundary between the potassium silicate alteration zone and the surrounding propylitically altered rocks. The latter extend outward from the deposit for several hundred metres. Late stage sericite-iron carbonate alteration forms irregular zones, particularly within the potassium silicate alteration zone. Argillic alteration is localized along fault zones and overprints earlier alteration assemblages.

Pyrite and chalcopyrite are the principal sulphide minerals in the deposit. They are uniformly distributed as disseminations, fracture-fillings and sub-vertical veinlets and may be accompanied by bornite and lesser molybdenite and tetrahedrite-tenantite. The latter results in somewhat elevated levels of arsenic, antimony, and mercury in some parts of the deposit. Native gold occurs as inclusions in, and along microfractures with, copper-bearing minerals and pyrite. Late-stage pyrite-base metal veins, up to several centimetres in width, are most abundant within the upper eastern portion of the deposit.

8.1 Surficial Geology

Regional glaciation occurred most recently during the Wisconsinian (15,000 to 18,000 years before present) during which time ice moved over the low lying and undulating surface of the West Fraser Plateau in a northerly and northeasterly radial dispersal pattern (Talisman, 1997). The hummocky topography resulting from this period of glaciation is typical of that produced by an ablating ice mass, and includes kames, eskers and kettles deposited on top of earlier lodgment or basal till.

During Wisconsinian glaciation, ice movement in the vicinity of Fish Lake was from south to north (Caira and Findlay, 1994). Recent alluvial activity has cut into, and deposited sediments on the older Wisconsinian sediments. In the proposed pit area, three main types of glacially-derived overburden were recognized: glacial till, glaciofluvial material, and glaciolacustrine material.

Prior to the most recent glaciation, Chilcotin Group flood basalts were deposited regionally across over 25,000 km^2 in the interior plateau of south central British Columbia. These flood basalts are sandwiched between the Wisconsinian sediments above and, in the immediate vicinity of the Prosperity deposit, underlying colluvial and lacustrine sediments.

In general, east of Fish Creek and north of Fish Lake the overburden consists predominantly of a patchy and variably thick sequence (less than 10 m to 65 m) of basal till (TILB) that covers colluvium (FANL, FAN) and bedrock. A prominent 750 m long esker occurs on the east side of Fish Creek and extends south to within 250 m of the outlet of Fish Lake. The west side of Fish Creek is mainly underlain by a thick sequence of basalt flows (BSLT), which can be observed in cliffs outcropping along the bank of the creek. The basal till occurs as an irregular cover up to 22 m thick over the basalt flows which in turn are in direct contact with bedrock or overlie a variably extensive and irregularly thick (8 to 70 m) layer of colluvium (FANL and FAN). Lake sediments (SILT) occur extensively in the southern portion of the deposit adjacent to Fish Lake.

Detailed geological logging of the overburden within the proposed pit indicates that there are four major types of overburden present: glacial till (TILB, CLAYU, SICLU, SILTU, and GRAVU), basalt flows (BSLT), colluvium (FANL and FAN) and glacial lacustrine sediments (GRAV, SICL, CLAY, and SILT). This overburden sequence consists of 48% basalt, 38% glacial till, 10% colluvium and 4% sediments and varies from 0 to 65 m in thickness over the deposit, but is as thick as 155 m to the south of the deposit near Fish Lake.

8.2 Volcanic and Sedimentary Rocks

Five volcanic units and one subvolcanic unit comprise the majority (78%) of the Prosperity deposit host rocks. In order of volume within the proposed pit, they are: 32% andesite crystal, ash and lapilli tuff (FAXT), 23% porphyritic andesite flow (FLOW), 21% crowded porphyritic andesite (SUBV) and 2% laminated andesite tuff (BEAT). Andesite tuffs and flows are commonly interbedded.

The volcanic rocks present in the deposit area are atypical of the surrounding area and are likely of limited regional extent. Similar volcanic rocks outcrop near the mouth of Fish Creek 3.5 km to the north and may correlate with those of the deposit.

A sparsely mineralized, volcanically-derived sedimentary unit (SEDS) occupies the upper south/southeast portion of the deposit and comprises 4% of the proposed pit. Stratigraphically, these sediments are postulated to represent a facies change in the volcanic assemblage that outcrops near the mouth of Fish Creek.

8.3 Fish Lake Intrusive Complex

The Prosperity deposit is spatially and genetically related to the Fish Lake Intrusive Complex, which is comprised of the Fish Creek Stock, quartz feldspar and lesser feldspar porphyry dikes and post-mineralization porphyritic diorite dikes.

The Fish Creek Stock is a hypabyssal lenticular east-west trending, steeply south-dipping body of porphyritic quartz diorite (QD) that has intruded a thick sequence of volcanic rocks. It is composed of three phases, the heterogeneous fine porphyritic quartz diorite, seriate porphyritic quartz diorite and subporphyritic to equigranular quartz diorite units, that together comprise 11% of the deposit's volume. These units are very similar in chemical composition, but differ in textural characteristics. The latter are commonly gradational; heterogeneous fine porphyritic quartz diorite can grade into seriate porphyritic quartz diorite and seriate porphyritic quartz diorite can grade into subporphyritic to equigranular quartz diorite over distances of several metres to tens of metres. The heterogeneous fine porphyritic quartz diorite and seriate porphyritic quartz diorite units also occur independently.

Quartz feldspar porphyry and feldspar porphyry dikes occur as an east-west trending, steeply south-dipping swarm centered east of the Fish Creek Stock. They comprise 7% of the deposit. The quartz feldspar porphyry units cross-cut all of the volcanic and sedimentary rocks identified in the deposit. The contemporaneity of the quartz feldspar porphyry dikes and the Fish Creek Stock is suggested by the occurrence of some units of transitional lithology, close to the border of the stock.

The entire suite of rocks (intrusive, volcanic and sedimentary) hosting the deposit is cross-cut by a series of barren, post-mineralization porphyritic diorite dikes (PMPD). The post mineralization porphyritic diorite unit comprises less than 1% of the deposit rocks.

Spatial distribution of the Prosperity deposit geological units is made in reference to the 1997 geology block model. The model is constructed over elevations of 547.5 to 1567.5 m above sea level, with level plans at 15 m vertical intervals. Typical plan views and vertical sections are shown in Figures 8-3 through 8-7.

Figure 8-3 Geology Plan View 1400m Elevation

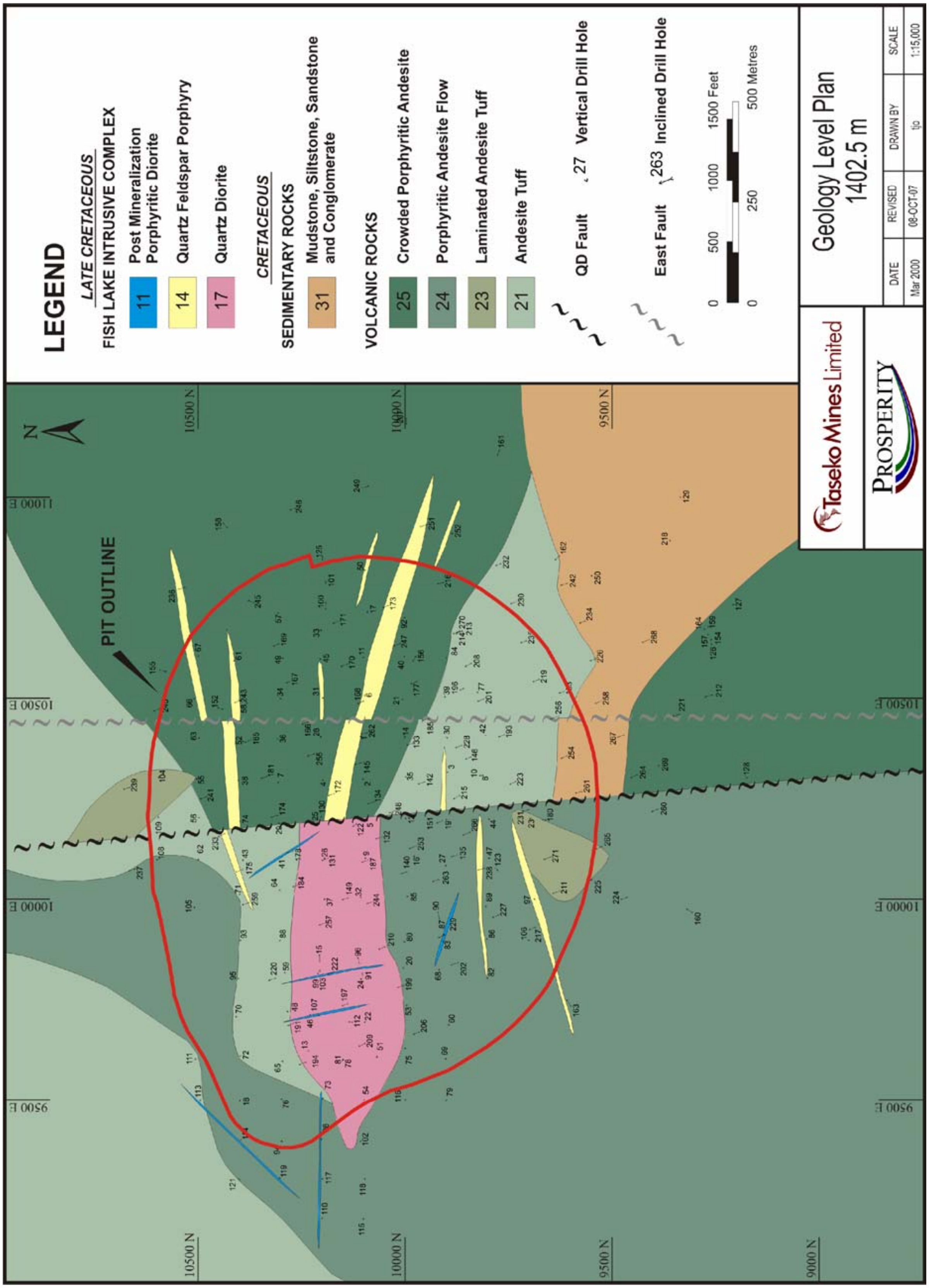


Figure 8-4 Geology Plan View 1200m Elevation

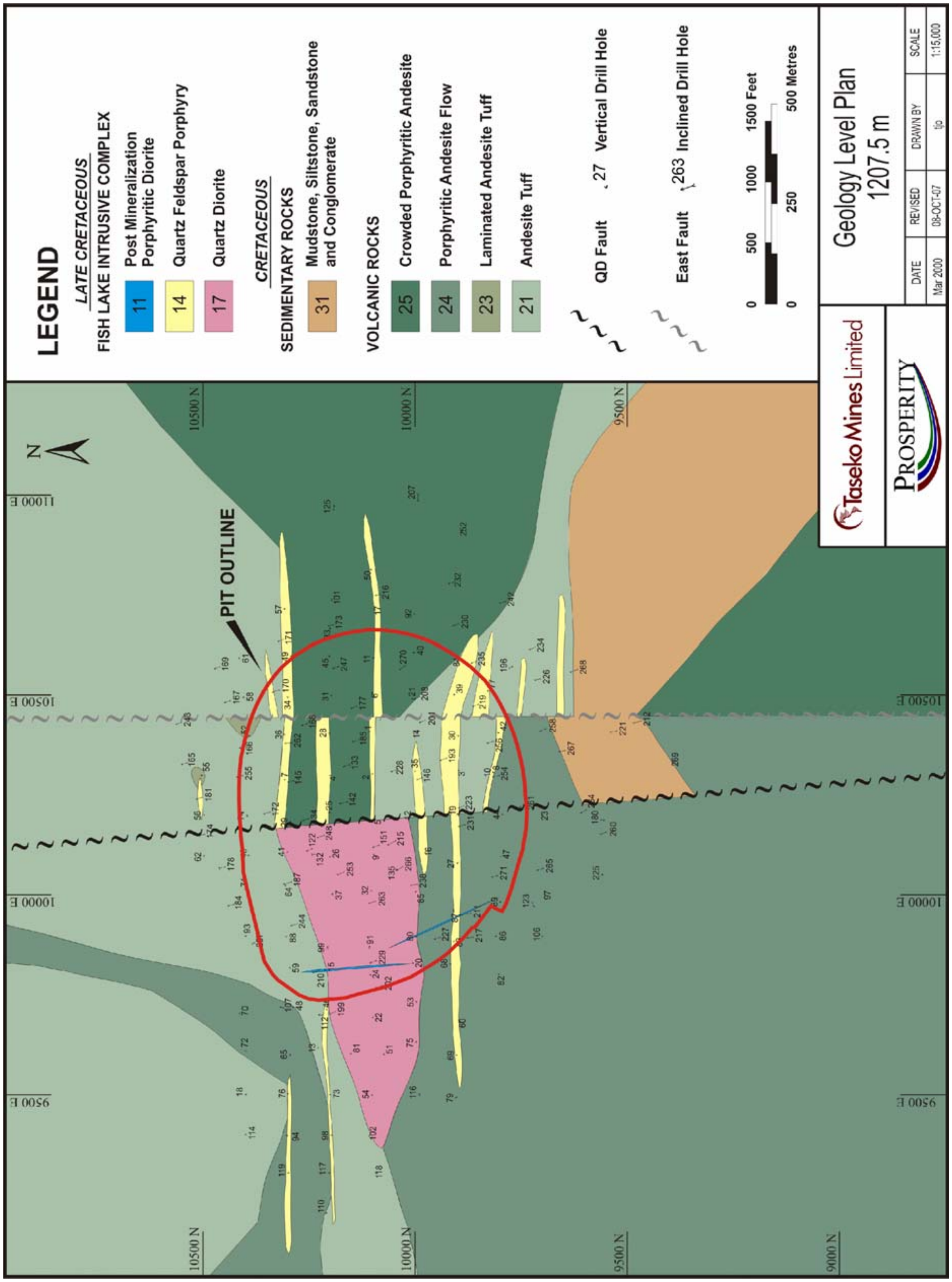


Figure 8-5 Geology Plan View 1000m Elevation

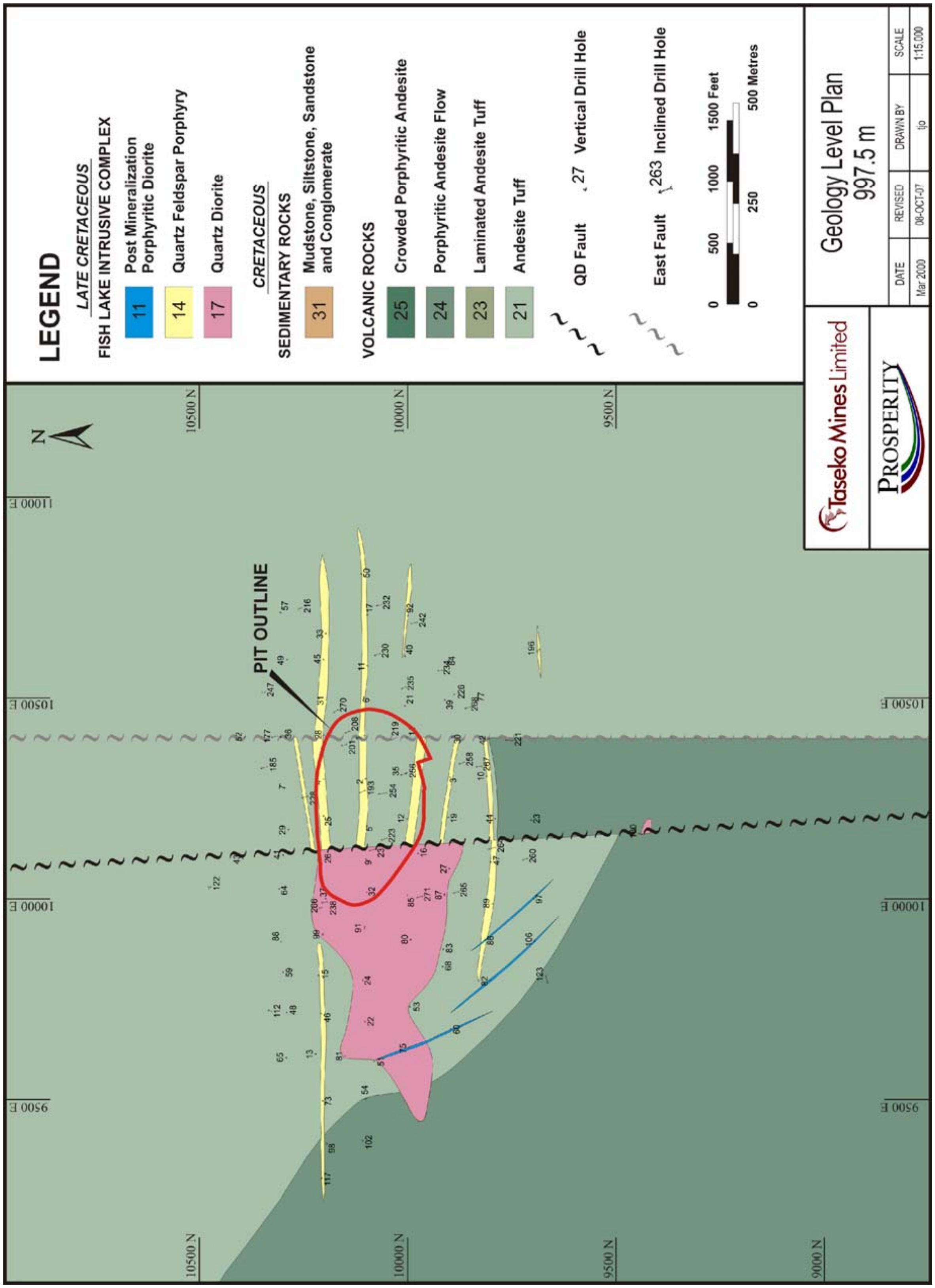


Figure 8-6 Geology Vertical Section 10100E

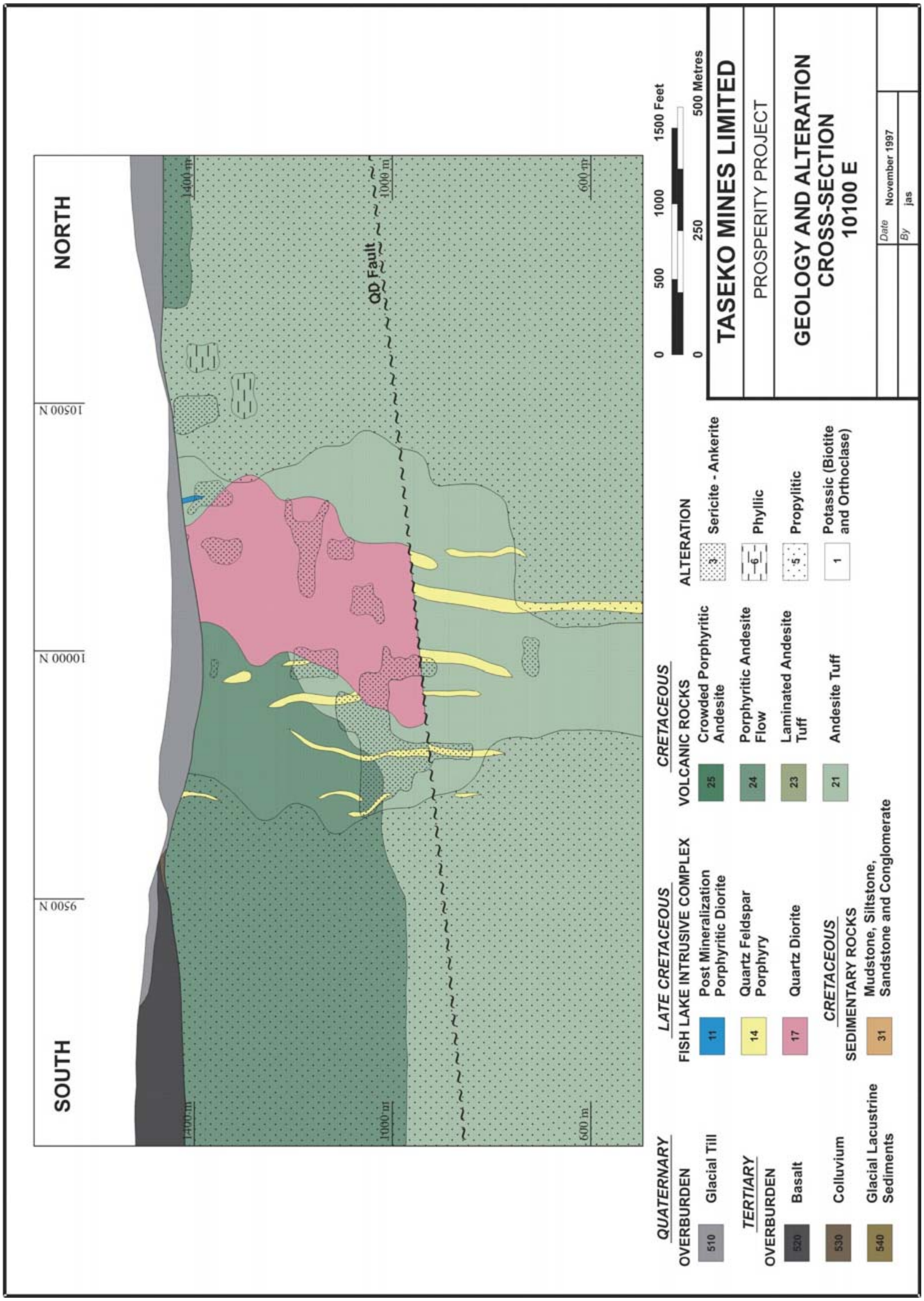
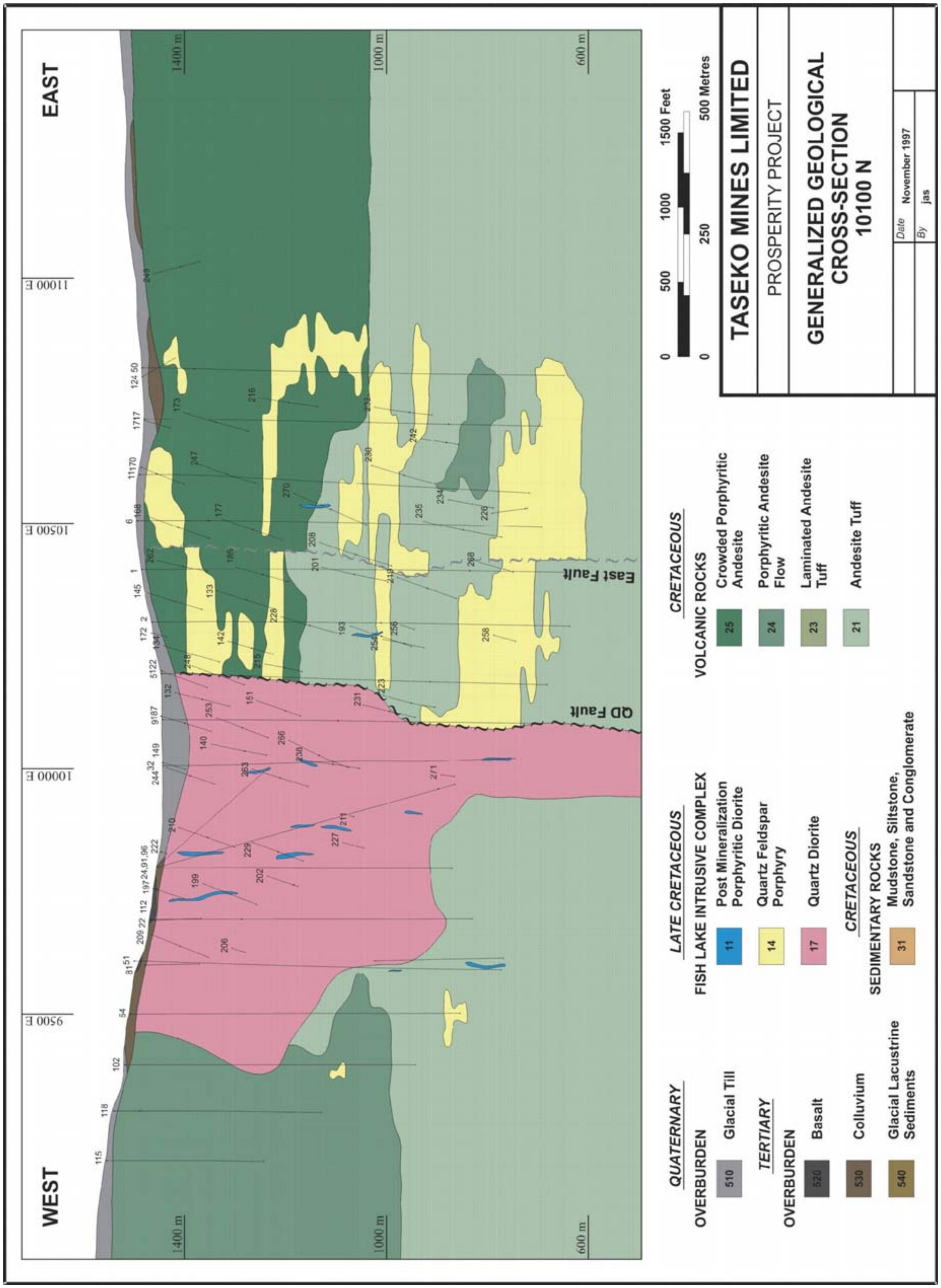


Figure 8-7 Geology Vertical Section 10100N



8.4 Alteration

Five main alteration styles have been identified at the Prosperity deposit: potassium silicate, propylitic, sericite-iron carbonate, phyllic and argillic. Alteration styles do not occur singularly in discrete zones; they commonly overlap and/or overprint each other. However, one alteration style will typically dominate over others in a given area, hence the naming of a zone specific to the dominant alteration style.

Potassium silicate alteration is the most widespread alteration within the deposit area. It forms a central east-west trending ovoid zone, which is intimately related to significant gold-copper mineralization (>0.20 gpt Au and >0.20% Cu). The zone of potassium silicate alteration is surrounded by propylitically altered rocks that extend outward for several hundred m. Along the eastern margin of the deposit a discontinuous belt of phyllic alteration is developed in proximity to the transition between the potassium silicate and propylitically altered rocks. Late stage sericite-iron carbonate alteration forms irregular zones, particularly within the central zone of potassium silicate alteration. Argillic alteration is localized along fault zones and overprints earlier alteration assemblages.

The sequence of alteration events at the Prosperity deposit commenced with the emplacement of the Fish Lake Intrusive Complex and the development of a hydrothermal cell which caused the contemporaneous infusion of potassium silicate and propylitic alteration in zones concentric about the intrusive complex. This was followed by an episode of phyllic alteration which occurred at higher levels in the system and was the result of a mixing between fluids of the hydrothermal cell and meteoric waters. Phyllic alteration overprinted both potassium silicate and propylitic alteration in certain areas. Sericite-iron carbonate and argillic alteration, the latest events in the alteration history, were the result of the migration of late stage hydrothermal fluids and meteoric waters along structural features, resulting in the formation of secondary mineral assemblages in the host rocks which overprint all other alteration styles.

8.5 Structure

Numerous faults were intersected in drill core throughout the deposit area. Faults are usually indicated by strongly broken core, gouge, sheared textures, cataclastic textures and rarely mylonitic textures. All of the aforementioned features can occur across intervals of less than 1 cm to over 20 m. Utilizing all available data, two major faults (the QD and East Faults) have been delineated.

The QD and East Faults are subparallel, strike north-south and dip steeply to the west, becoming near vertical down-dip. They cut the central portion of the deposit and are approximately 230 m apart near surface and 330 m apart at depth. The western most of the two major faults, the QD Fault, trends approximately 355º and has a steep westward dip of 82º to 86º. This fault marks the eastern boundary of the Fish Creek Stock. The eastern most of the two major faults, East Fault, strikes approximately 360º and has a steep westward dip of 85º to 87º.

9. Mineralization

Gold-copper mineralization within the Prosperity deposit is intimately related to potassium silicate alteration and a later, superimposed sericite-iron carbonate alteration. This is particularly true within a central, east-west trending ovoid zone that hosts the majority of the mineralization.

Chalcopyrite-pyrite mineralization and associated copper and gold concentrations are distributed relatively evenly throughout the host volcanic and intrusive units in the deposit. A sedimentary unit, which is located in the upper southeastern part of the mineralized zone, is sparsely mineralized. Post mineralization porphyritic dikes are essentially barren.

Pyrite and chalcopyrite are the principal sulphide minerals and are accompanied by: minor amounts of bornite and molybdenite; sparse tetrahedrite-tennantite, sphalerite and galena; and rare chalcocite-digenite, covellite, pyrrhotite, arsenopyrite and marcasite. Native gold generally occurs as inclusions in, and along microfractures with, copper sulphides and pyrite. Pyrite to chalcopyrite ratios throughout most of the proposed pit area range from 0.5:1 to 1:1 and rise to 3:1 or higher around the periphery of the deposit which coincides with the propylitic, and locally the phyllic, alteration zones.

Sulphide minerals show the thoroughly dispersed mode of occurrence characteristic of porphyry copper deposits. Sulphides occur in relatively equal concentrations as disseminations, blebs and aggregates in mafic sites, as fracture fillings and as veinlets. Disseminated sulphide mineralization is marginally more prevalent than veinlets in intrusive rocks while in volcanic rocks the reverse was noted.

Gold and copper distributions throughout the deposit are presented in Figures 9–1 through 9–3.

Figure 9–1 Au and Cu Grades at 1400m Elevation

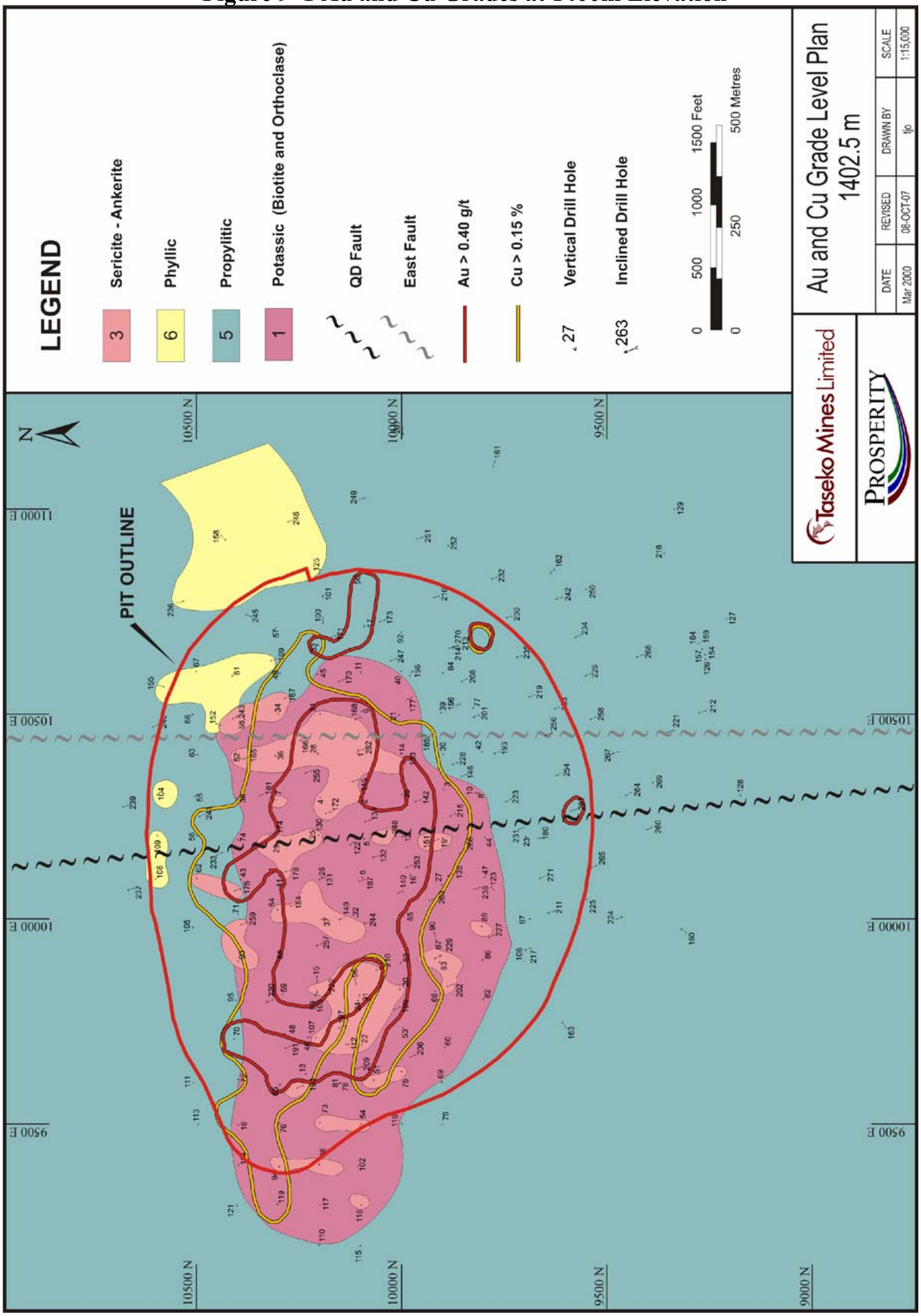


Figure 9–2 Au and Cu Grades at 1200m Elevation

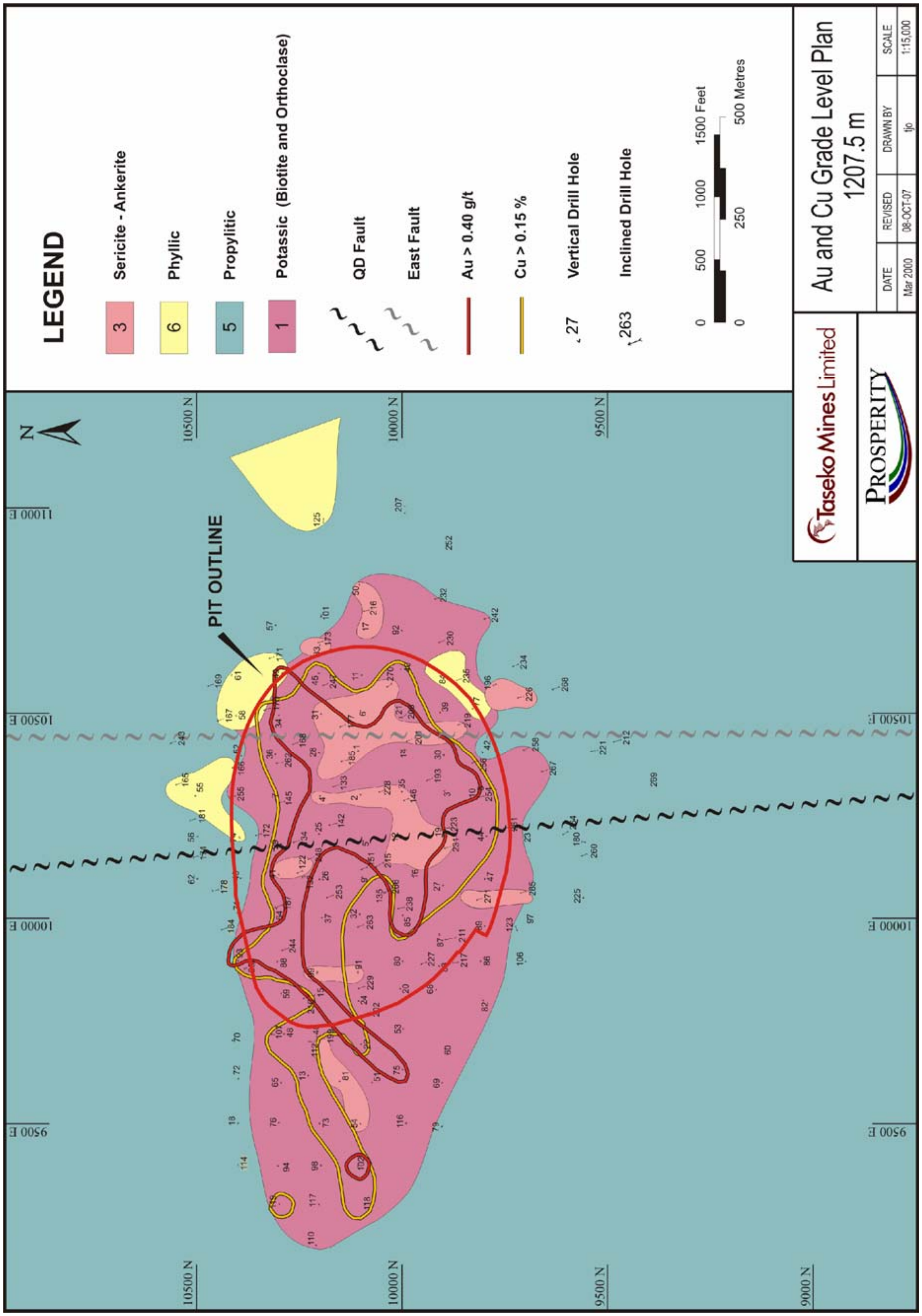
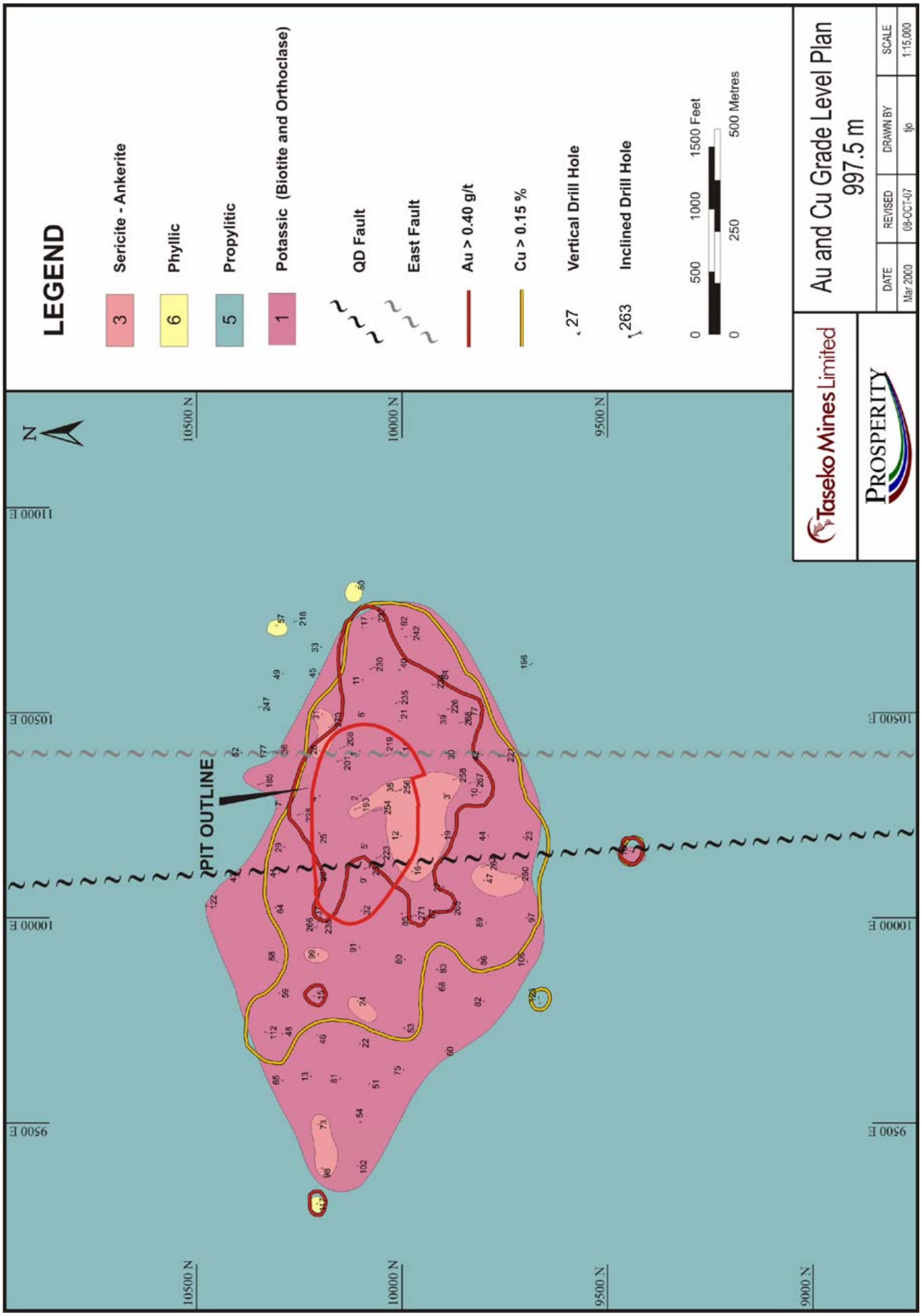


Figure 9–3 Au and Cu Grades at 1000m Elevation



10. Exploration

10.1 Extent of All Relevant Exploration

Up to 1991, exploration programs at the Prosperity Project included extensive IP, magnetic and soil geochemical surveys, and 176 percussion and diamond drill holes totaling approximately 27,100 m as outlined in Section 9. This work helped define the Prosperity Project mineralization to a depth of 200 m, and outlined a gold-copper mineralized zone approximately 850 m in diameter which Cominco estimated as a geological resource of 208 million tonnes grading 0.23% copper and 0.41 gpt Au.

In 1991 Taseko drilled 10 holes totaling 7,506 m in a “cross” pattern to test the core of the deposit over a north-south distance of 550 m. All of the holes intersected continuous significant copper and gold grades and extended the mineralization to 810m below surface. A scoping-level metallurgical testwork program was completed by Melis Engineering Ltd. The testwork demonstrated that acceptable gold and copper recoveries could be achieved by bulk sulphide flotation followed by regrinding and conventional copper flotation. Baseline environmental and monitoring studies were initiated by the Company.

Diamond drilling continued in 1992, and by the end of the year an additional 116 HQ and NQ diameter vertical drill holes totaling 60,558 m had been drilled, expanding the deposit to 1400 m east-west, 600 m north-south and to 850 m below surface. G. Giroux, P.Eng., reported mineralized material (unclassified mineral resource) of 976 million tonnes at an average grade of 0.23% Cu and 0.48 gpt Au.

Subsequent to 1993 comprehensive metallurgical tests by Melis Engineering Ltd. and a 1994 pre-feasibility report by Kilborn Engineering Pacific Ltd., the Company completed a 12 hole (4,605 m) inclined core drilling program in 1994 to investigate the distribution of fracture controlled gold and copper mineralization in the deposit. In addition, 22 holes (3,171 m) were drilled to investigate geotechnical conditions in the proposed Project development areas.

In 1996 and 1997, an additional 107 holes (49,465 m) were completed in order to upgrade the confidence limits of the deposit. Of this total, 20 holes (2,203 m) were drilled vertically and 87 holes (47,262 m) were inclined. These holes significantly increased the density of pierce points in the deposit and added to the geotechnical and geochemical characterization of he rock in the deposit.

Over the 34-year period from 1963 to 1997, a total of 154,631 m has been drilled in 452 holes on the Prosperity project. Of this total, 273 holes (83,453 m) were drilled vertically and 174 holes (71,178 m) were inclined. Sizes of cored holes have included BQ, HQ, and NQ totaling 148,322 m, with an average drill spacing of 70 m. The balance of 6,309 m is from percussion drilling. A summary of the drilling of this period is shown in Table 10-1 and Figures 10-1 and 10-2.

Table 10-1
Drilling Summary 1963 – 1998

Year	Company	Percussion Drilling No. of Holes	(m)	Diamond Drilling No. of Holes	(m)	All drilling No. of Holes	(m)
Pre 1990	Table 6.1	67	6,110	109	21,009	176	27,119
1991	Taseko	0	0	10	7,506	10	7,506
1992	Taseko	0	0	116	60,558	116	60,558
1993	Taseko	0	0	8	2,104	8	2,104
1994	Taseko	1	200	34	7,680	35	7,879
1996	Taseko	0	0	69	28,423	69	28,423
1997	Taseko	0	0	38	21,042	38	21,042
1998*	Taseko	0	0	18	1,768	18	1,768
Total Drilling		68	6,310	402	150,090	470	156,399

In 1998, G. Giroux, P.Eng., reported estimated measured and indicated mineral resources of 1.0 billion tonnes at 0.41 gpt Au and 0.24% Cu and an inferred resource of 0.2 billion tonnes grading 0.25 gpt Au and 0.21% Cu at a 0.14% copper cut-off. Giroux provided the resource at a number of different cut-offs, and since that time, the resource has also been reported at a 0.2% copper cut-off, which was also based on the Giroux 1998 estimate.

* 1998 drilling consisted of fourteen geotechnical holes and four in-pit verification holes, none of which were incorporated into the Geological model.

Figure 10-1 Drill Hole Locations 1969 to 1994

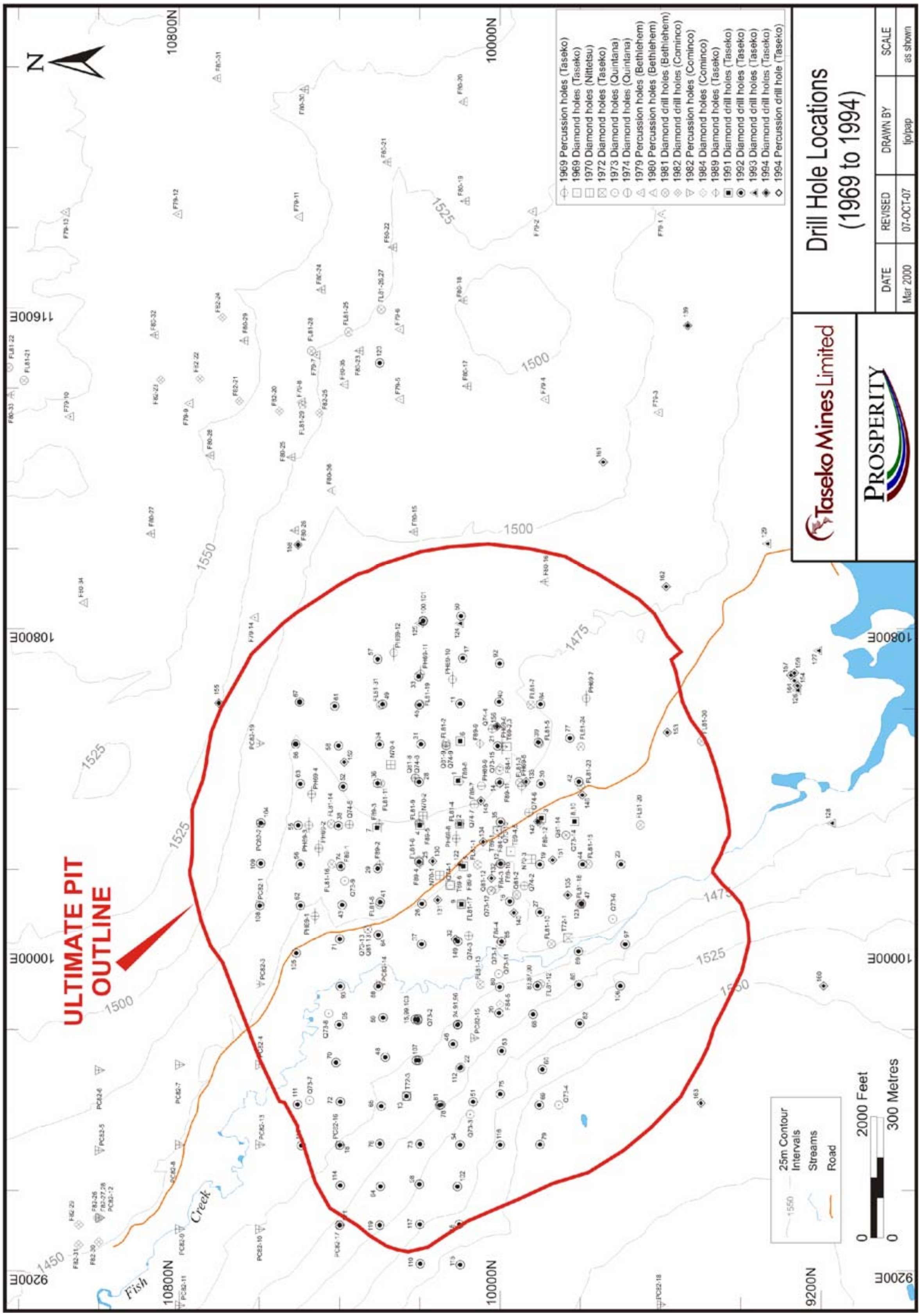
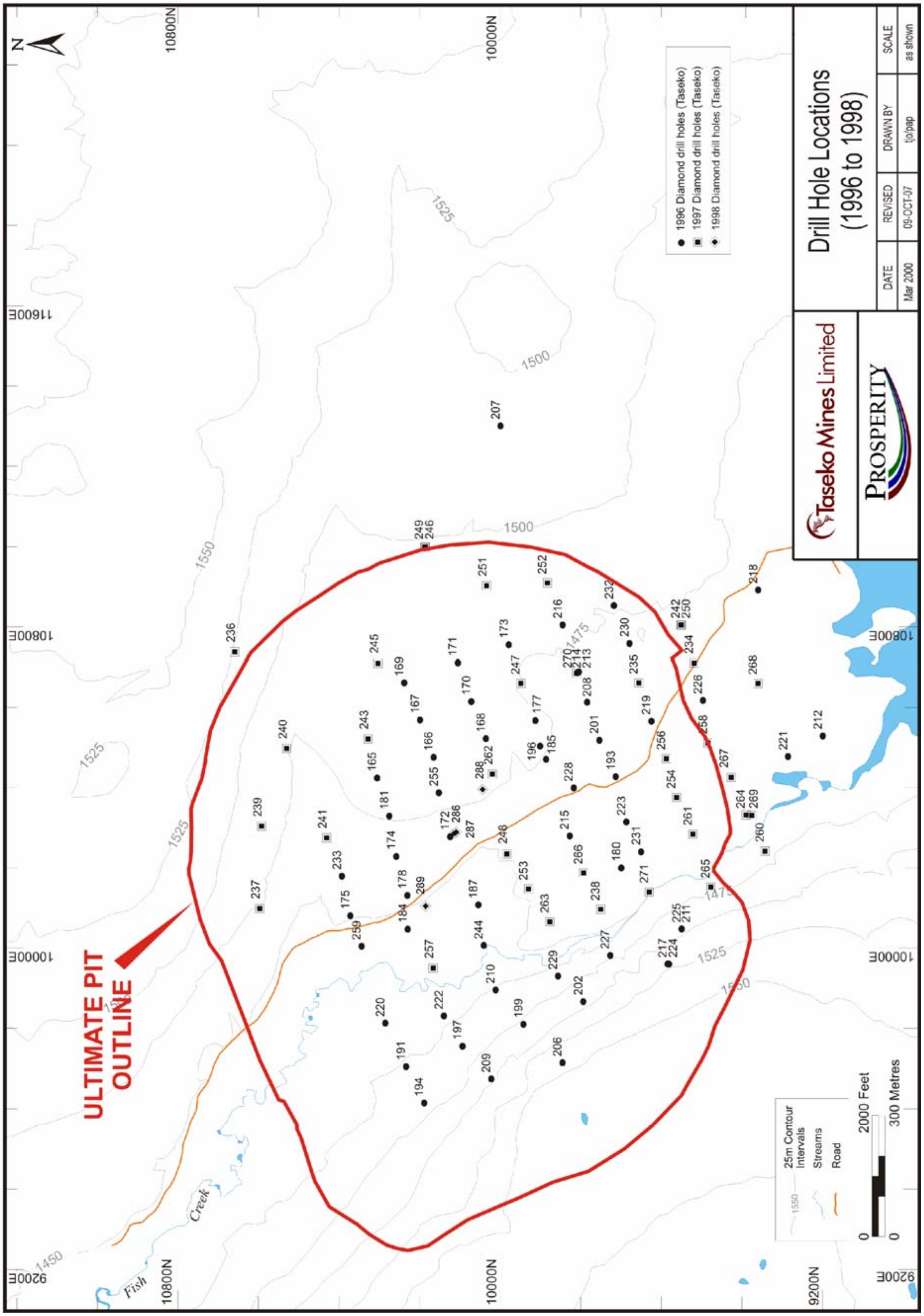


Figure 10-2 Drill Hole Locations 1996 to 1998



11. Drilling

11.1 Drilling Pre-1991

Prior to 1991, several companies unrelated to Taseko carried out mineral exploration in the Prosperity project area. The first drilling on the property was carried out by Phelps-Dodge in 1963 and 19964. Taseko Mines (Old Taseko), Nittetsu Mining Company, Quintana Minerals Corporation, Bethlehem Copper and Cominco also drilled holes between 1969 and 1991. During the 13 years of active exploration during this period, approximately 20,300 m of core drilling in 109 holes, and 6,100 m of percussion drilling in 67 holes was completed.

11.2 Drilling 1991-1994

In 1991, the current Taseko management, (New Taseko), took over the project and from 1991 through 1994 a total of 77,944 m was cored in 168 holes. Approximately 28% of the drilling was HQ (6.35 cm diameter) core size and 68% NQ (4.76 cm diameter) core. A single percussion hole 200 m in length was also drilled.

The bulk of the drilling took place in 1991 and 1992 as a series of predominantly vertical drill holes. In 1993 four drill holes inclined at –60º were drilled using oriented core methods and four vertical holes were completed. The single percussion hole and 34 core drill holes were drilled in 1994. The latter included eight holes at various orientations, drilled outwards from the centre of the deposit. Another 13 holes were drilled in the main deposit area at an azimuth of 340º with inclinations of –45º to –50 º. A further 13 holes were drilled vertically in 1994. No drilling took place in 1995.

Geotechnical data was recorded for all drill holes from 1991 to 1994. Core recovery was measured on 25,344 drill run intervals averaging 3.05 m in length. Recovery was good, with a mean value of 97.0% and a median value of 99%, for the sampled intervals measured.

11.3 Drilling 1996-1997

From June 1996 through May 1997, Taseko completed a 107 hole drill program, comprising 49,500 m of diamond bit core drilling. This included in-fill definition holes, oriented in-fill holes, oriented geotechnical pit wall holes, acid base accounting (ABA) holes, and waste rock and tailings geotechnical holes (Table 11-1). All in-fill drilling was performed at an azimuth of 340º East and an inclination of -45º, on 100 m by 100 m spacing. Of this drilling, all was sampled and assayed except 4,400 m of overburden.

JT Thomas Diamond Drilling of Smithers, BC drilled HQ and NQ core was drilled by using skid-mounted hydraulic drills and a drill modified for helicopter transport to remote sites. Geotechnical data was recorded for all but six drill holes in the 1996-1997 programs. Core recovery was measured on 17,035 drill run intervals averaging 3.05 m in length. Recovery was good, with a mean value of 94.2% and a median value of 98%, for the sampled intervals measured.

Table 11-1
Drill holes by Orientation by Year

Drill Type	Orientation	1963 to 1990	1991, 1992	1993	1994	1996, 1997	1998	All
Percussion	Vertical	67	–	–	1	–	–	68
Core	Vertical	58	115	4	13	20	18	228
Total	Vertical	125	115	4	14	20	15	293
	North	25	2	–	1	–	–	28
	Northeast	1	1	–	2	1	–	5
	East	15	6	2	1	–	–	24
	Southeast	2	–	–	1	1	–	4
Core	South	1	–	–	1	5	–	7
	Southwest	3	–	–	1	–	1	4
	West	–	2	1	1	1	–	5
	Northwest	4	–	1	–	–	–	5
	Azim. 340°	–	–	–	13	79	2	94
Total	Inclined	51	11	4	21	87	3	177
Total	**All**	**176**	**126**	**8**	**35**	**107**	**18**	**470**

11.4 Drilling 1998

Eighteen core holes were drilled in 1998 including 15 vertical holes and three holes inclined at -45º for a total of 1,768 m. Four drill holes were completed within the main porphyry, nine geotechnical holes were drilled to the south of Fish Lake, and five geotechnical holes were drilled east of the main deposit area.

In 1998 Kilborn undertook a comprehensive audit and verification program of the geology and assay results of the Prosperity project. The drilling noted above was in support of this program. The four in-pit verification diamond drill holes totaling 1150 metres were completed by Kilborn; 110 half core samples and 99 reject samples from the 1991-1992 drill programs were re-assayed. All analytical work was performed by Chemex. Based on the results of this program, it is Kilborn's opinion that the geological work for the Prosperity was done in a professional manner and according to industry standards.

12. Sampling Method and Approach

The Prosperity deposit was explored and extensively drilled by seven different companies between 1963 and 1998. A total 156,500 m of core and percussion drilling was completed in 470 drill holes during the twenty years in which active drill exploration took place (refer to sections 6 and 10). The drill hole spacing is such that no part of the deposit, as defined by the current resource model, is farther than 70 m from drill hole information, although the majority of drill holes are considerably closer.

A total of 63,937 drill core samples and 1,548 percussion samples were taken for analysis between 1969 and 1998. Sampled and assayed intervals total 136,949 m. For most holes drilled, the entire length of Cretaceous rock was sampled and assayed. Early sampling and analysis at Prosperity focused on assessing the copper mineralization visible in the rocks. Once the presence of significant gold mineralization was recognized, assaying for gold became more comprehensive. Starting in 1991, Taseko undertook multi-element analysis for 30 elements on all drill core samples, in addition to the regular assaying for copper and gold.

12.1 Core Logging

All drill holes completed from 1991 through 1998 in the main deposit area, were geotechnically logged, geologically logged, and photographed prior to sampling.

12.2 Sampling

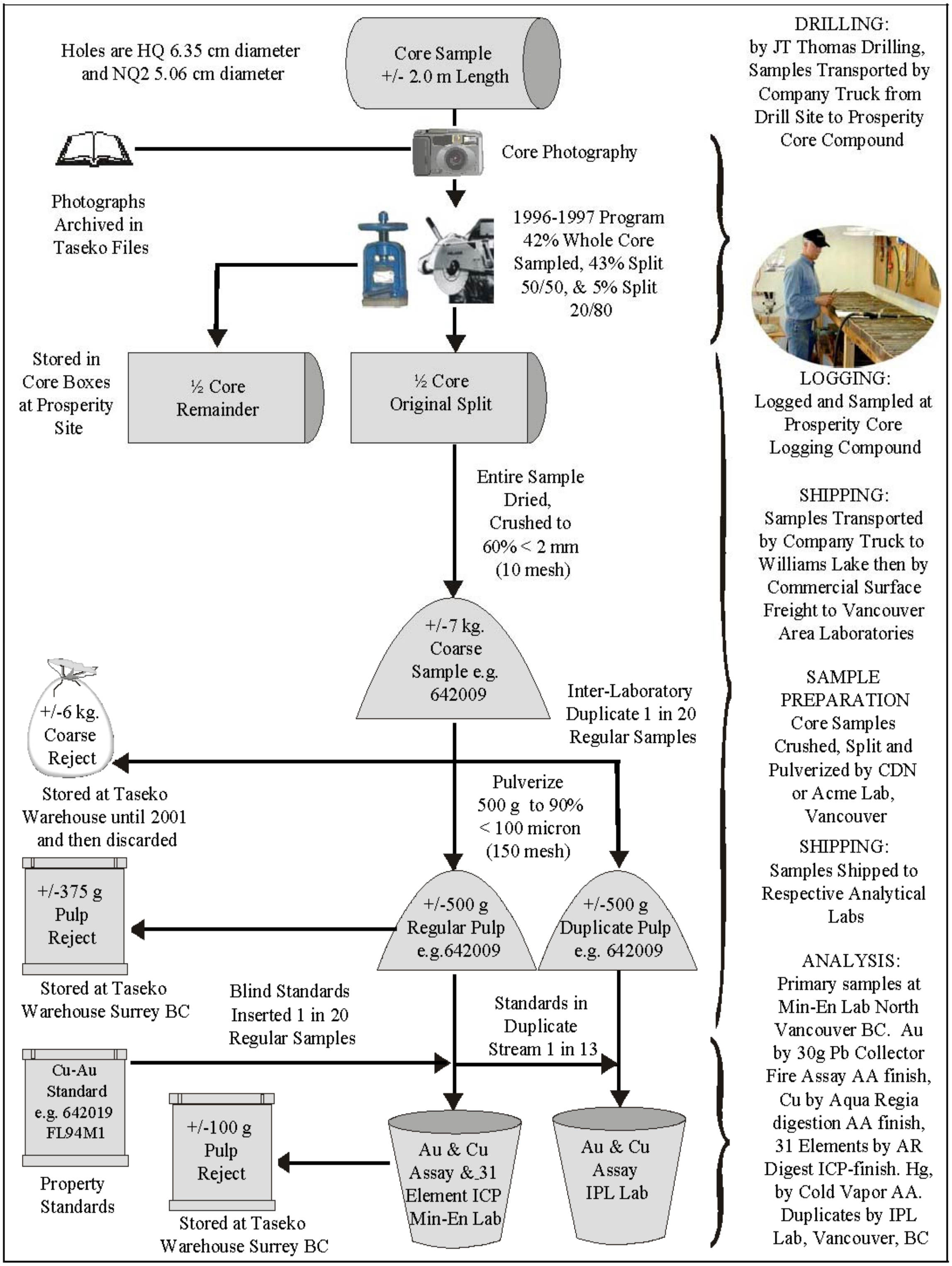
A total of 63,937 drill core samples and 1,548 percussion samples have been taken for analysis since 1969. Prior to 1991, a total of 6,905 were taken with an average length of three m. During the 1991 through 1998 drilling programs, 58,580 core samples were taken for assay. These sample intervals were generally 2 m in length, except in instances where this was impractical. No assay information exists for the eight holes Phelps- Dodge drilled in 1963-1964. Table 12-1 lists the number samples and type of analysis by year.

During the period 1991-1994, drill core was mechanically split, one half of which was submitted for preparation and analysis. Of the total meterage drilled during 1996-97, 42% was subject to whole core sampling, 44% was sampled as sawn half-core, 5% of samples comprised the larger portion of core sawn 80:20. The remaining 9% was cored overburden, which was not generally sampled, although some samples were taken for ABA studies and placer claim assessment. In 1998 the samples were half sawn core. Samples were sawn lengthwise with a diamond bladed rock saw using water to wash and lubricate the blade. The remaining sample was put back in drilling order in the core box. Figure 12-1 illustrates the sampling, sample preparation and analytical protocol for the 1996-1997 programs. Drill core remaining after sampling was returned to the core boxes, which were racked and stored at the Prosperity Site.

Table 12-1
Number of Samples by Year

Year	Au & Cu Assays	Au Assays Only	Cu Assays Only	All Assays
1969	120	–	542	662
1970	22	2	22	46
1972	37	–	–	37
1973	706	6	3	715
1974	486	–	–	486
1979	76	32	119	227
1980	442	37	90	569
1981	2,333	61	347	2,741
1982	452	53	83	588
1984	267	–	–	267
1989	564	3	–	567
1991	3,472	–	–	3,472
1992	28,700	–	–	28,700
1993	581	–	–	581
1994	2,744	–	–	2,744
1996	12,724	–	–	12,724
1997	9,606	–	–	9,606
1998	539	–	26	565
Total	**63,871**	**194**	**1232**	**65,297**

Figure 12-1
1996-1997 Drill Core Sampling, Preparation and Analytical Flow Chart



13. Sample Preparation, Analyses and Security

13.1 Security

In 1991-1998 the drill core was boxed at the drill rig and transported twice daily by company truck to the logging, sampling and sample preparation compound at the Prosperity site. The core was geologically and geotechnically logged, given quality control quality assurance (QAQC) designations, photographed and sampled under the supervision of Taseko geological and engineering staff. Samples were placed in shipping sacks and taken by company truck to Williams Lake and then shipped by commercial carriers to the Vancouver area analytical laboratories.

13.2 Sample Preparation

During the 1991-1998 programs, core samples were shipped to Vancouver laboratories for preparation, including drying at temperatures less than 65°C, and blind standards inserted by the preparation laboratory into the sample stream, crushing and pulverizing. Beginning in 1994, all samples were weighed to the nearest 10 grams and inserted blind standards into the sample stream the preparation laboratory. An average dry weight of 7.4 kg was obtained from the 24,804 weights reported for the 1994-1997 samples. The 1991-1994 samples were prepared by Mineral Environments Laboratories Ltd. (Min-En) of North Vancouver BC. In 1996-1997, the samples were prepared by either Acme Analytical Laboratories Ltd. (Acme) or CDN Resource Laboratories Ltd. (CDN) of Vancouver BC. The 1998 samples were prepared by Chemex Labs Ltd. of North Vancouver BC.

Primary comminution to approximately ¼ inch (6.4 mm) size was provided by a jaw crusher in 1991-1993. A secondary roll crusher was used to obtain minus 15 mesh material for pulverization. In 1994-1997 the dried samples were crushed in a single stage so that more than 60% passed a 10 mesh screen. Since 1991, a sub sample (the assay split), weighing a minimum of 500 grams, was riffled from the crushed material. The remaining crushed reject material was stored in Taseko's Surrey BC warehouse until 2001 then discarded.

Preparation of the assay splits involved ring and puck pulverization. The 1991-1993 laboratory specifications for pulverization were approximately 95% passing 120 mesh. In 1994-1997 the specifications for pulverization were modified to greater than 90% passing 150 mesh. Screen tests were done and reported for approximately one in fifty pulps. Additional, detailed screen analyses were done on selected samples by CDN and International Metallurgical and Environment Inc. After testing, plus and minus fractions of the screen samples were recombined, and the samples kept within the normal sample stream.

In 1991-1993 the 500 g pulp was homogenized by rolling prior to analytical aliquot selection. Starting in 1994, and continuing through the 1996-1997 programs, sample preparation was carried out separately from the assaying and analytical work, and reported on separate laboratory certificates. In these years, each 500 g pulp was rolled prior to riffle splitting, from which a 125 g analytical sub sample was obtained. This was placed in a pulp bag bearing the sample number and lot code and shipped to the analytical lab. Pulps remaining after splitting and aliquot selection were returned to Taseko and stored in the company warehouse.

13.3 Sample Analysis 1991-1997

Min-En performed the primary analytical work from 1991 through 1997. Gold was analyzed by lead collection Fire Assay, using a one assay ton (30 g) charge. After fusion, the doré bead was finished by Atomic Absorption Spectroscopy (AAS). Gold assays greater than 10 gpt were automatically re-assayed by one assay ton Fire Assay fusion with a gravimetric finish. Copper was determined by Aqua Regia digestion on a 0.5-2.0 gram sample with an AAS finish. In addition, 0.5 gram aliquots of all samples were assayed for 31 elements by Aqua Regia digestion Inductively-Coupled Plasma Atomic Emission Spectroscopy (ICP-AES). Mercury determinations were performed by Cold Vapor AAS. Chemex performed the primary analytical work for the 1998 program.

The following conversion factors were used for the analytical results:

- 1 gpt = 1,000 ppb
- 1 % = 10,000 ppm
- 1 oz/Ton = 34.2857 gpt

13.4 Quality Assurance Quality Control

Taseko Mines Limited implemented a quality control quality assurance (QAQC) program after taking over the Prosperity project in 1991. This program was in addition the QAQC procedures used internally by the analytical laboratories. The results of this program indicate that analytical results are of high quality and suitable for use in detailed modeling and resource evaluation studies. Table 13-1 describes the QAQC sample types used in this program.

Table 13-1
QAQC Sample Types Used

QC Code	Sample Type	Description	Percent of Total
MS	Regular Mainstream	• Regular samples submitted for preparation and analysis at the primary laboratory.	90%
ST	Standard Reference Material	• Mineralized material in pulverized form with a known concentration and distribution of element(s) of interest • Randomly inserted using pre-numbered sample tags	5% or 1 in 20
DP	Duplicate or Replicate	• An additional split taken from the remaining pulp reject or coarse reject. • Random selection using pre-numbered sample tags • Inter-Laboratory duplicates analyzed at a second or check laboratory (random selection) • Non-random selection, after initial assays returned	10% 1 in 10 (1991-1992) 5% 1 in 20 (1996-1998)
SD	Standard Duplicate	• Standard reference sample submitted with duplicates and replicates to the check laboratory	<1%

Table 13-2 is a summary of the regular mainstream (MS) samples and additional QAQC samples analyzed on the Prosperity Project that were submitted by Taseko in addition to the laboratory internal QAQC work. Only a limited number of QAQC samples exist prior to 1991.

Table 13-2
Drill Hole Sample QAQC Summary

Year	MS	DP	SD	ST	Total
Pre-1991	6,905	109	–	–	7,014
1991	3,472	351	–	–	3,823
1992	28,700	2,819	–	–	31,519
1993	581	–	–	–	581
1994	2,744	73	11	131	2,959
1996	12,724	677	46	636	14,083
1997	9,606	499	33	467	10,605
1998	565	284	5	25	879
ALL	**65,297**	**4,812**	**95**	**1,259**	**71,463**

Standards

In 1994, Taseko modified the sampling and analytical QAQC program, to include the random submission of project-based, bulk standard reference materials, within the mainstream and duplicate analytical streams. The insertion of standards continued over the course of the 1996-1998 drill programs. The property standards were inserted by the sample preparation laboratory approximately mid-way between duplicate samples. On average, one in twenty samples was randomly selected for duplicate analysis at a second laboratory. In addition, approximately every thirteenth standard was designated as a standard duplicate, to be submitted with the duplicate sample stream at the second laboratory.

This process involved identifying the QAQC samples at the core logging stage. Sample bags containing 'standard' tags (but no sample) were inserted at the appropriate intervals at this stage. Standard tags were numbered as part of the normal sample sequence. Quality control samples were also identified on the sample shipment notice and marked on the bags in the same fashion as regular samples.

The program employed four property-based standards: 94FLH1, 94FLM1, FL96M2 and FL96L1, with designations H, M and L corresponding to high, medium and low gold grades respectively. Standard preparation took place at CDN under the direction of Smee & Associates Consulting Ltd. (Smee) The material was crushed, pulverized, and screened to minus 150 (1994) or minus 200 mesh (1996) and then mechanically mixed. As mixing proceeded, sub-samples were periodically assayed to test for homogeneity. Depending on the standard, test results indicate that this was achieved within 2 to 5 days. In 1994 the assessment for homogeneity took place in-house, whereas in 1996, the sub-sample results were forwarded to Smee for verification.

For each standard, the sample preparation laboratory received an otherwise empty bag bearing the tag or number of the designated sample and the standard reference code. Prior to shipping the 125 g pulps prepared from the regular samples, 125 g standard pulps were inserted into the sample stream. These had the same appearance as the mainstream samples and fitted sequentially into the sample number series in order to appear anonymous to Min-En, the main assay lab.

Gold and copper analytical results were monitored when received for QAQC failures, including, results outside the control limits, or consecutive results outside the warning limits.

- Warning limits: ± 2 S.D.
- Control Limits: ± 3 S.D.

Table 13-3 lists the four standard reference samples used.

Table 13-3
Summary of Copper-Gold Standard Reference Materials Used

Standard	Times Used	Cu %	2 Std. Dev. Cu	Au gpt	2 Std. Dev. Au
94FLH1	255	0.381	0.038	0.742	0.120
94FLM1	337	0.253	0.054	0.395	0.099
FL96M2	150	0.269	0.023	0.372	0.077
FL96L1	501	0.166	0.010	0.259	0.034

Most of the larger spikes in the standard performance charts are attributable to the mislabeling or accidental insertion of a different standard by the sample preparation lab. If an assay of a standard returned outside of the set tolerance limits, the pulps of the mainstream samples bracketing the standard were re-assayed at Min-En. As a result, 438 pulp samples were re-run, the results of which are in very close agreement with the original assays. Depending on the behavior of the standard in these check batches versus the original runs, the original results were replaced in the database by the second result. In summary, the assay performance based on the standard results is within acceptable limits.

Blanks

No blanks were inserted with the samples submitted Taseko. Analysis of drill core made up entirely of material from the Post Mineral Dyke unit, which commonly occurs within mineralized porphyry rocks consistently returned values <0.05 g/t gold and <0.05% copper. Analytical blanks were regularly inserted by the assay laboratories and analyzed with each batch of samples as part of their internal QAQC protocols.

Duplicates 1991-1992

Prior to 1991, a total of 82 duplicate gold assays and 27 duplicate copper assays were completed, representing 1.4% of the total gold assays and 0.4% of the total copper assays. A more thorough check assay program of random duplicate analysis was implemented starting in 1991-1992. Every tenth two metre sample from the 121 holes drilled during this period, regardless of grade, was shipped to Chemex for riffle splitting of the coarse reject, pulverization and analysis for gold and copper. Then, for each of the duplicate samples analyzed, Chemex took another riffle split from the coarse reject, which was re-bagged, renumbered and re-submitted to Min-En for pulverization and "blind" gold-copper analysis.

Thus 3,171 samples, representing 10% of the 1991-1992 programs, were analyzed three times for gold and copper. There is good agreement in the intra-laboratory and inter-laboratory reject duplicate results for copper and gold through all grade ranges for the 1991-1992 programs.

Duplicates 1994 and 1996-1997

Similar duplicate programs were conducted in 1994 and 1996-97, although as noted above, sample preparation was not carried out at the same laboratory as the final assays. Duplicates were therefore held at the preparation laboratory until batches of twenty or more had accumulated in order to ensure that each batch shipped to the assay laboratory was accompanied by a standard. International Plasma Laboratory Ltd. (IPL) of Vancouver performed the check assay analysis in 1996-1997.

In 1994 a total of 73 duplicate and 11 standard duplicate samples were analyzed following this procedure. During the same year, 1,841 vein samples, assayed for copper and gold, were taken from the other half of the core. Consequently fewer reject duplicates were taken relative to the 1996-1997 program when a total of 1,146 reject duplicate and 79 standard duplicate samples were assayed, following the above procedure. In addition, 4,389 vein samples were assayed for copper and gold during 1996-1997.

In 1996-1997 the sample prep laboratory (Acme or CDN) a duplicate 500 g sub sample was riffled from the –10 mesh crushed reject, after the mainstream sub-sample had been taken. These duplicates were then pulverized in the same manner as mainstream samples. Duplicates were held at the preparation laboratory until batches of twenty or more had accumulated before they were sent to Chemex in 1994 and to IPL in 1996-1997 for check assay analysis. This procedure ensured that the duplicates were also accompanied by a standard.

There is good agreement in the 1996-1997 inter-laboratory reject duplicate assays for copper and gold through most of the grade ranges. Some differences in the Min-En versus IPL results appear below 0.1% copper and 0.1 gpt gold, which likely reflect different analytical detection limits at the two labs.

Results of the duplicate analysis are well within the anticipated range for inter-laboratory checks.

13.5 Specific Gravity – Bulk Density Measurements

In 1991-1992, specific gravity measurements were carried out on-site at regularly spaced 8 metre increments on 8 to 15 cm long pieces of core by the water immersion method. Additional specific gravity measurements were taken in 1996-1997. Specific gravity determinations were made from 7,687 representative field measurements. Laboratory determinations and laboratory checks were also made. Table 13-4 is a summary of the specific gravity measurements by year and method.

The model developed by Taseko accounts for variation in specific gravity with alteration type, lithologic type and geographic domain. A bulk density model was created by applying a reduction factor to the modeled specific gravity to account for the estimated bulk void space. This followed a recommendation in a 1993 Knight Piésold Ltd. memo *Report on Influence of Geotechnical Factors on Bulk Density*. Hence an upper limit reduction factor of 0.25% was applied to values above the gypsum line while a reduction factor of 0.125% was used below this level.

Table 13-4
Specific Gravity Measurements by Year and Method

Year	Field		Laboratory		Laboratory Check	
	No.	Average	No.	Average	No.	Average
1991	811	2.73	95	2.71	11	2.70
1992	6,753	2.73	430	2.71	48	2.71
1996	25	2.71	–	–	–	–
1997	98	2.74	–	–	–	–
Total	**7,687**	**2.73**	**525**	**2.69**	**59**	**2.71**

14. Data Verification

14.1 Database

In 1991 Taseko acquired the pre-1991 drill hole data from Cominco, including digital logs, and a partial set of drill hole files. In most cases, the original logs and laboratory analytical certificates were not available, although some half core material and crushed rejects were still accessible. While these were generally in such poor condition as to be unusable for the purposes of re-assaying, enough material was available from some of the pre-1991 holes for Taseko to re-log them in 1991-1992. These logs confirmed the overall geological interpretation, and the tenor of copper mineralization in these cores. Cominco's digital compilation of pre-1991 information was reviewed and routine errors and omissions were corrected. For the most part, the Cominco compilation was accepted at face value.

Since 1991, all drill logs, sampling and analytical information was compiled in an Access relational database, which has tables that are compatible with GEMS mining exploration software. Written logs were produced manually at the logging compound and site offices. The key drill data tables: header, survey, geotechnical, geology, vein, density and sample description were entered into spreadsheets at the project site and Vancouver office and imported into the database. The field data was merged with the analytical results in the Vancouver office and the compiled information was then exported to Vulcan and MineSight for further processing and modeling. A complete set of core photographs was taken; the prints and negatives are archived in the project files.

The database for the current block model and resource estimate was based on drilling completed by Taseko Mines Limited from the period 1991-1997, and also included 87 pre-1991 drill holes considered to be most reliable. This additional diamond drill data assisted with geological interpretation and provided extra control during block grade estimation.

The pre-1991 percussion drill holes were not included since the sampling lengths and procedures were quite different from diamond drill core sampling and there was no lithologic control in the sampling of percussion chips. Four holes drilled by Nittetsu in 1980 were also excluded as the distribution of both copper and gold grades for these holes were clearly different and therefore the assays were considered suspect.

A series of post-1991 holes on the edges of the deposit, drilled primarily for geotechnical information, were also incorporated in the database to define the outer boundary of the resource and the proposed pit. In these seven drill holes, yielding a total of 429 samples, copper grades were by ICP-ES analysis and gold grades were by one half assay ton (15 g) Fire Assays.

Drill hole collars at the Prosperity Project were located with reference to the Taseko Mine Grid, which was originally established for soil geochemistry and mapping purposes in the 1960's. Bethlehem surveyed several baseline hubs and claim posts in this co-ordinate system in 1979, and in 1981 McElhanney Associates tied the Mine Grid into government triangulation stations 'Tex' and 'Junior'. At this time, McElhanney also established 228 control points for photogrammetric mapping purposes, provided the first Mine Grid elevations, surveyed the locations of 13 claim posts and surveyed 111 drill hole collars. Mine Grid co-ordinates and elevations of other pre-1991 drill holes, which had not been surveyed were provided by previous operators of the project.

In 1993, the pre-1991 collar locations and elevations were checked by plotting them on the McElhanney 1:2500 scale 2.5 m contour map. The plotted elevations were then compared with the mapped contour elevations to check for discrepancies in elevation. Most differences were less than +/-2 m, however 11 holes had differences of up to +/-5 m, which were not resolved.

In 1991-1992 site surveying was undertaken by Taseko based on the control provided by the 1981 McElhanney mine-grid survey stations 1086, 1087 and 1098. These control stations were verified by McGladrey Surveys Ltd. in the spring of 1993 (Caira and Findlay, 1994). McGladrey also resurveyed 42 drill hole collars, and the co-ordinate comparison between the McGladrey and Taseko surveys was very good. McGladrey noted mean differences in northing, easting and elevation of 0.0, -0.1 and 0.2 m respectively between the two sets of collar survey results.

A Total Station survey instrument was used to survey the twenty-nine 1994 drill hole collars located within the main deposit area. The other six 1994 holes, which were drilled south of Fish Lake along the proposed tailings embankment were surveyed to +/- 1m using a differential GPS in 1997.

During the course of further photogrammetric and orthophoto work in 1996, McElhanney re-checked the key survey stations and provided a method to convert from Mine Grid to UTM NAD 83 Zone 10 co-ordinates. In 1996-1997, new drill holes located within the main deposit area were surveyed using a Nikon total station and TDS 500 Data Collector. Data was downloaded directly to a computer spreadsheet in order to eliminate transfer errors. These results are summarized in the Prosperity Site 1996-1997 Survey Report (Maguire, 1997). Outside of the main deposit area, collar coordinates were obtained with a Trimble Scoutmaster GPS. UTM coordinates were transferred manually from this instrument to a spreadsheet.

14.2 Verification

Taseko verified the post-1990 portion of the Prosperity drill hole database by manual team verification in late 1992 and early 1998. This work focused on the following areas: sample logs, assay results, laboratory measured specific gravity measurements, collar and down hole surveys and geology. In addition to this, all drill hole copper and gold assay and geologic data was plotted out in 15 m level plans and 50 m spaced cross-sections and visually validated by the project geologists.

A report on validation of drill hole orientation information from 1991-1997 was completed by Taseko in 1998. Drill hole orientation is derived from surface surveys of the collar azimuth and dip, Light Log tool downhole readings, single shot Sperry Sun magnetic down hole compass measurements, and a few downhole acid-test readings of dip. Corrections were applied to the database in the case of mis-readings. Survey records where tool malfunction was suspected were removed.

The results of the Taseko verification program indicate that the database is of good quality and acceptable for use in geological and resource modeling

As part of the feasibility study, Kilborn undertook a comprehensive audit and verification program of the geology of the Prosperity project in 1998. This included: a survey audit, a geological audit, a check assay program, a drilling, sampling and assaying program; and an additional sampling program.

As part of this program Kilborn completed a verification drilling, sampling and analytical program with logistical support provided by Taseko. Four diamond drill holes, 98-286, 98-287, 98-288 and 98-299 totaling 1150 m were drilled by Major Drilling Ltd. Drilling was done in NQ2 except one twin hole (98-289) which was partially drilled as HQ to a depth of 153 m in order to exactly replicate the Taseko hole (92-26). All drilling and sampling work was supervised by Kilborn personnel, who also delivered the samples to the assay laboratory.

The check assaying, sampling and drilling programs were designed to increase the confidence level with respect to grades and geological interpretation utilized in resource and reserve estimation for the Prosperity project in areas to be mined mostly during the payback period. Although some variation in grades and geological descriptions were noted, it was felt that they were random and not of significance. Based on the results of this program, combined with the rest of the geological audit/verification program, it was the opinion of Kilborn that the geological work for the Prosperity deposit was done in a professional manner and according to industry standards.

15. Adjacent Properties

There is no information of significance relative to the Prosperity project from adjoining property.

16. Mineral Processing and Metallurgical Testing

16.1 Introduction

Early metallurgical testwork on the Prosperity Project was conducted by various laboratories from 1973 to 1991. The early testwork was limited in scope and nature, primarily being focused on achieving high copper recovery at medium to high concentrate copper grade, with little emphasis on gold recovery to the final copper concentrate. An initial Phase I metallurgical test program was undertaken by Melis in 1991 on composites made up from upper level assay reject samples taken from a 1989 drilling program. Although the composites tested were not deemed representative of the Prosperity mineralization, this initial test program did show that acceptable copper and gold recoveries could be achieved using a bulk sulphide float at natural pH with subsequent copper-pyrite separation and concentrate cleaning at alkaline pH.

A more extensive second phase of metallurgical testing was completed on representative composites of Prosperity mineralization at Lakefield Research Limited from December 1992 to August 1993 under the supervision of Melis (Melis Project No. 265; January 25, 1994 Report). This Phase II test program included batch flotation tests and locked cycle flotation tests on composites representing different areas of the deposit to determine achievable copper and gold recoveries and provide detailed concentrate analysis, grindability assessments, tailings settling tests, environmental data, and a cursory examination of the removal of mercury, arsenic, lead and antimony from Prosperity flotation concentrate.

A third phase of metallurgical testing was a run-in pilot plant carried out on Prosperity assay rejects and half (drill) core combined composites from the 1994 drilling program at Lakefield Research Limited in November, 1996 under the supervision of Melis (Melis Project No. 333; March 25, 1997 Report). The run-in pilot plant was carried out as a precursor to the later comprehensive pilot plant; its main objectives to establish basic operating parameters and generate material for initial environmental testing. Prior to the five run-in pilot plant runs, batch flotation tests were conducted on the available composites.

The fourth phase of metallurgical testing was a pilot plant campaign carried out at Lakefield Research Limited in August, 1997 under the supervision of Melis (Melis Project No. 345, November 27, 1998 Report). This phase included batch and locked cycle tests as well as pilot plant runs carried out on composites prepared from assay rejects and half core representing different zones of the Prosperity deposit. It also included grinding testwork, detailed analysis of concentrates, generation of environmental data, tailings settling tests, and concentrate settling and filtration tests.

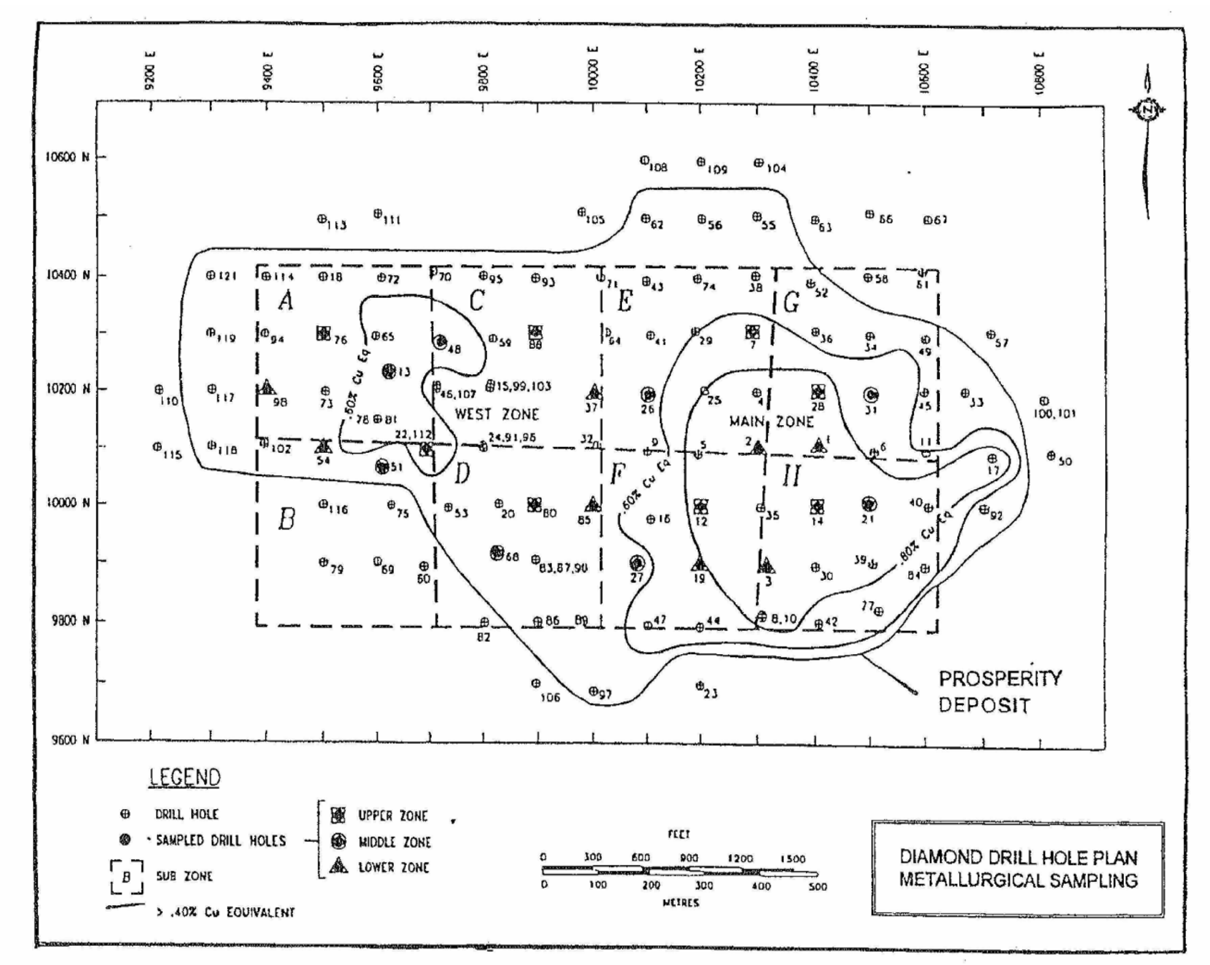
16.2 Composites

Phase II Composite Preparation

A sketch of the diamond drill hole plan used for the gathering of metallurgical samples for the Phase II test program is shown in Figure 16-1.

Individual composites were prepared by Min-En Laboratories of Vancouver, British Columbia. A total of 24 individual composites were made up from assay rejects using a 1 kg weight for each m of intersection.

Figure 16-1
Phase II Test Program – Drill Hole Plan



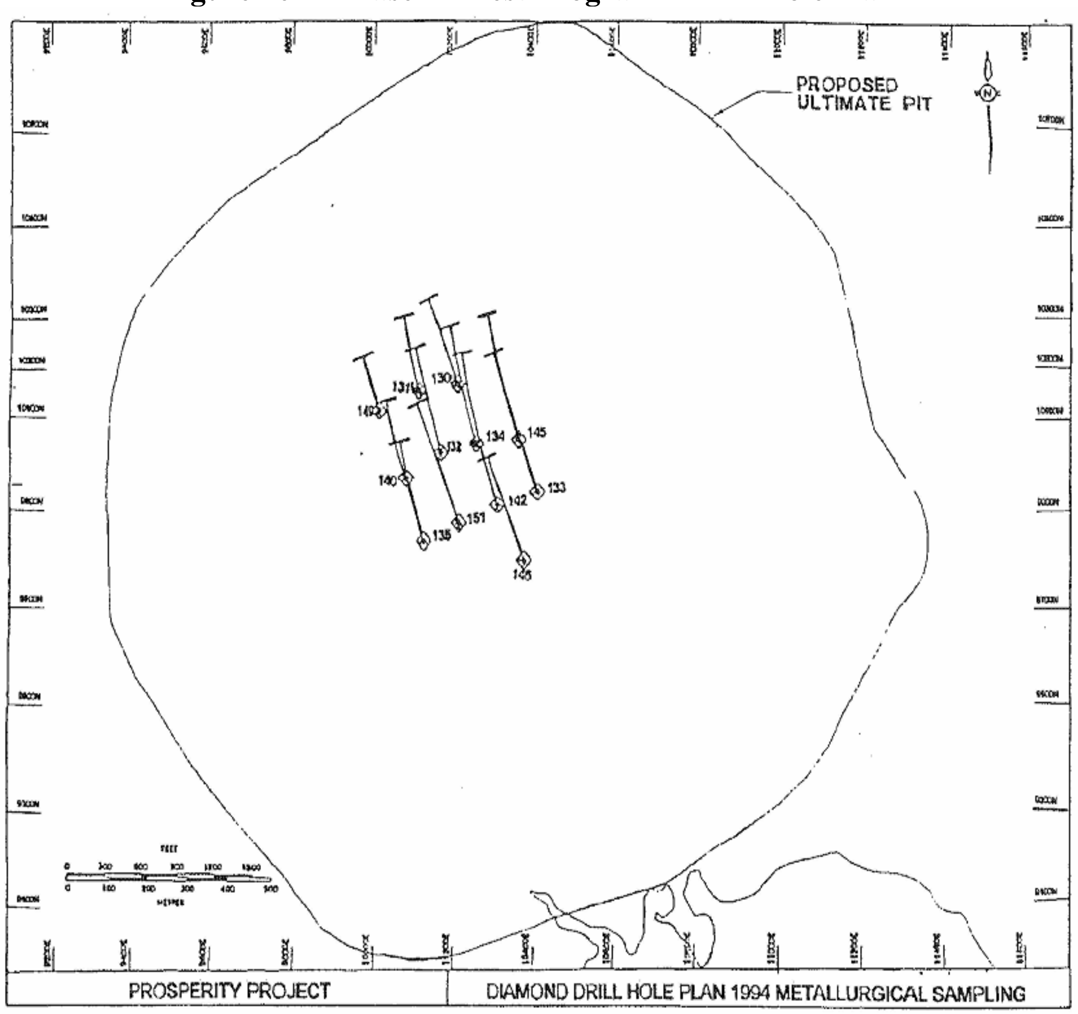
Phase III Composite Preparation

Approximately 21 tonnes of samples were received for run-in pilot plant testing, including both assay rejects and drill core from the 1994 drilling program representing the upper level (>200 m) of the main (east) area of the mineralization. Assay rejects were blended as separate intrusive and volcanic sub-composites for batch testing and one overall composite for run-in pilot plant runs RI-PP1, RI-PP2 and RI-PP3. Drill core samples were blended as separate intrusive and volcanic sub-composites for batch testing and one overall composite for run-in pilot plant runs RI-PP4 and RI-PP5.

Two kilogram test charges were prepared for five alteration composites (intrusive sericite, intrusive potassium silicate, volcanic sericite, volcanic potassium silicate and volcanic propylitic) for use in batch flotation tests to generate environmental data.

A sketch of the diamond drill hole plan of the main zone used in the 1994 metallurgical sampling for the Phase III metallurgical testwork is shown in Figure 16-2.

Figure 16-2 Phase III Test Program – Drill Hole Plan

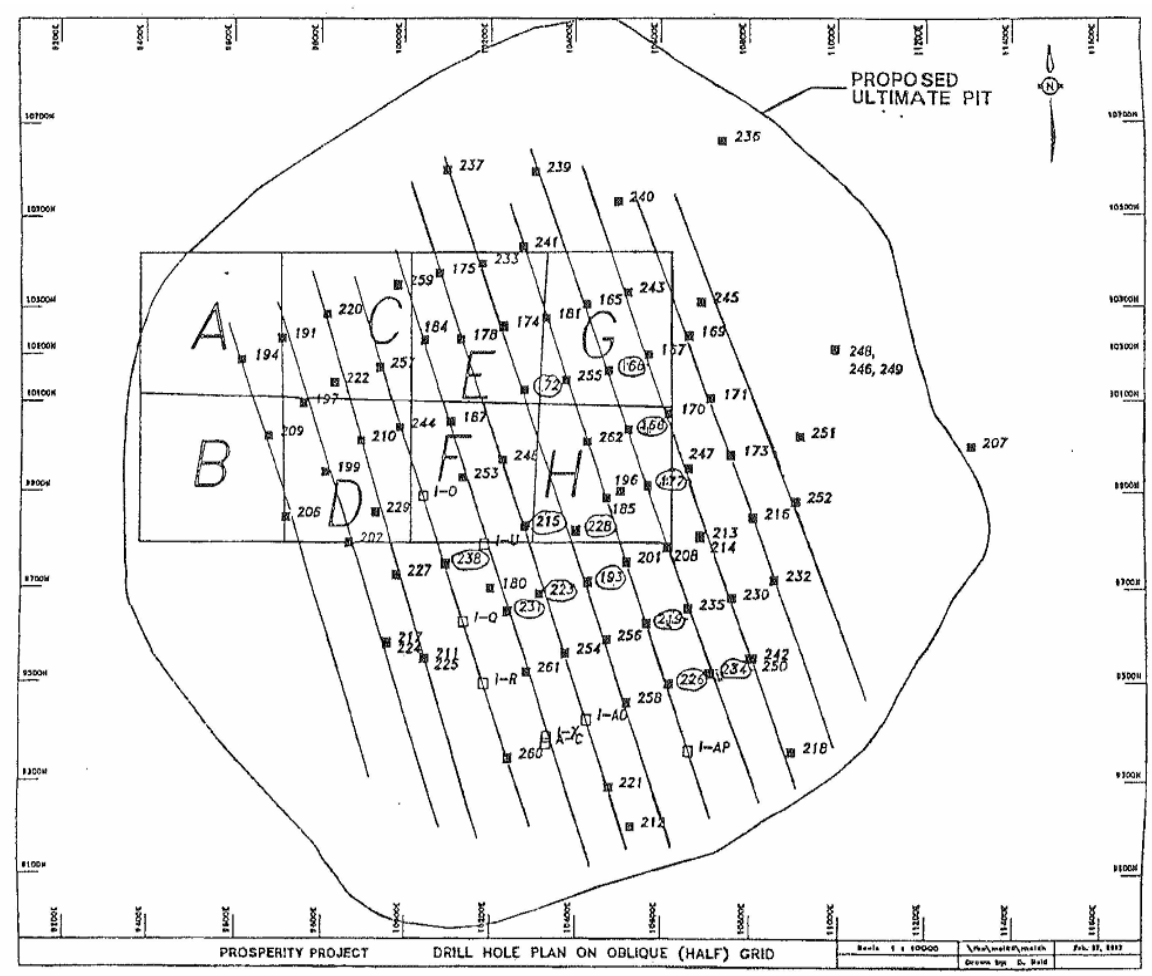


Phase IV Composite Preparation

Test composites were prepared for pilot plant testing from 58 tonnes of assay rejects and half core samples. The assay rejects samples were blended into three composites (upper, middle, and lower) for batch testing and one overall composite for locked cycle and pilot plant testing. Half core samples were blended into three composites (upper, middle and lower), which were run separately in batch tests, locked cycle tests, and pilot plant runs.

A sketch of the drill hole plan for the 1996/1997 drilling program (Phase IV), showing the locations of the drill holes in the Prosperity deposit, is shown in Figure 16-3.

Figure 16-3
Phase IV Test Program – Drill Hole Plan



16.3 Mineralogy

The major copper mineral in the Prosperity deposit is chalcopyrite ($CuFeS_2$). It occurs in multistage veins and associated fracture-controlled gangue segregation. In veins the metallic assemblage mostly includes chalcopyrite-pyrite (FeS_2) with some local occurrence of bornite (Cu_5FeS_4) -tetrahedrite [$(Cu,Fe,Zn,Ag)_{12}Sb_4S_{13}$)]/tennantite [$(Cu,Fe,Zn,Ag)_{12}As_4S_{13}$)] and minor occurrences of chalcocite (Cu_2S) or trace digenite (Cu_9S_5). Local occurrence of molybdenite (MoS_2) and gold also occur. In vein-related segregation chalcopyrite and pyrite are the main metallic minerals with minor occurrences of bornite.

Altered - impregnated {quartz(SiO_2) – sericite[$KAl_2(AlSi_3O_{10})/(OH)_2$] – albite $(NaAlSi_3O_8)_{0.9}(CaAl_2Si_2O_8)_{0.1}$} groundmass contains very minor finely disseminated chalcopyrite as loose clusters. Disseminated pyrite occurs abundantly in quartz-sericite altered rocks in a biotite [$K(Mg,Fe)_3AlSiO_{10}(OH)_2$] chlorite [$(Mg,Al,Fe)_{12}(Si,Al)_8O_{20}(OH)_{16}$] hornblende and chlorite-carbonate [calcite($CaCO_3$)/dolomite($CaMg(CO_3)_2$)] altered mafics. Significant amounts of chalcopyrite occur interstitially to and as small blebs in magnetite ($FeFe_2^{3+}O_4$) – hematite (Fe_2O_3).

Chalcopyrite, with local occurrences of bornite, tetrahedrite/tennnantite and lesser chalcocite, is commonly associated with pyrite as interstitial microveinlets and as scattered small inclusions in pyrite.

Native gold shows strong zonal distribution and commonly occurs in loose clusters of grains with strong vein control in quartz and carbonate gangue. Size ranges noted were
<2.5 µm (microns) to 95 µm. It occurs most abundantly in association with copper minerals, particularly where tetrahedrite/tennantite and chalcocite are present. It was noted as isolated grains in gangue, as isolated blebs and microveinlets in pyrite, as blebs in chalcopyrite, tetrahedrite/tennantite and chalcocite, and rarely with bornite.

Molybdenite has zonal distribution and occurs as isolated grains and loose clusters of grains. It is vein/segregation controlled and is associated with chalcopyrite in quartz/sericite – chlorite gangue.

Non-opaque gangue materials include quartz, sericite and feldspar (K, Na, $AlSi_3O_8$) with subordinate amounts of carbonate (CO_3). Opaque gangue minerals include mainly pyrite with subordinate amounts of iron or titanium oxides and iron hydrides (magnetite, hematite, rutile (TiO_2), goethite [FeO(OH)].

Gypsum ($CaSO_4 \cdot 2H_2O$) and lesser anhydride ($CaSO_4$) form late veins and open space fillings in and crossing mineralized veins, and form-filling voids in gangue segregation. There is a spatial relationship between gypsum (anhydride) and mineralization with accidental inclusion of sulphides in gypsum.

16.4 Grinding

Work Indices

The abrasion indices, bond impact crushing work indices, correlated autogenous work indices, rod mill bond work indices, and ball mill bond work indices measured for the Prosperity composites in the three test programs are listed in Table 16–1.

Table 16-1
Metric Work Indices

Composite	Abrasion Index (1)	Bond Impact Crushing Work Index	Correlated Autogenous Work Index	Rod Mill Bond Work Index	Ball Mill Work Index
Sept/91	-	-	21.7	17.4	15.9
U1	-	-	18.2	20.6	16.8
U2	-	-	16.5	19.8	15.3
M1	-	-	17.0	17.7	15.9
Int.	-	-	-	15.0	15.5
Vol.	-	-	-	16.4	16.1
VU(AGL)	0.0952	5.5	18.3	16.1	17.7
VU(BGL)	0.1832	3.3	-		
VM	0.1444	7.5	20.0	17.3	17.9
VL	0.3421	5.0	20.2	18.6	20.4
IU(AGL)	0.1339	6.7	-	16.2	16.4
IU(BGL)	0.2181	-	-		
IM	0.2638	5.4	-	15.5	17.5
IL	0.3660	5.8	-	14.7	17.6
Averages:					
All	0.2183	5.5	18.8	17.1	16.9
Volcanic	0.1912	5.3	19.5		18.0
Intrusive	0.2455	5.7	-		16.8
Upper	0.1576	5.1	17.7		16.6
Middle	0.2041	6.5	18.5		17.1
Lower	0.3541	5.4	20.2		19.0

On average the Prosperity mineralization has a low abrasion index and a relatively low impact crushing index. The measured grinding indices showed the mineralization to be of medium hardness, but with a relatively high autogenous work index.

Grinding Tests

Three batch tests were completed on the Overall Half Core Composite to analyze the effect of differing primary grind product discharge sizes on copper and gold recoveries.

Based on these three tests, a cost benefit analysis using the value of incremental copper and gold recoveries against net grinding power costs concluded that the optimum 80% cumulative passing size (K_{80}) of the primary grind was approximately 160 µm.

Four batch tests were completed to analyze the effects of regrind size distribution on copper and gold recovery to the third cleaner concentrate.

The optimum regrind K_{80} appears to be between 14 µm and 17 µm in the regrind product. Gold recovery was found to be more sensitive to the K_{80} of the regrind product than copper recovery.

16.5 Gravity Separation

A total of five gravity recovery tests were conducted during the Phase III metallurgical testwork to determine the potential for gold recovery by gravity separation. Gravity gold recovery was performed on drill core composites ground to match the flotation test grind. Processes tested were a Wilfley Table followed by a Mozley Concentrator, a Falcon Separator followed by a Mozley Concentrator, a Knelson Concentrator followed by a Mozley Concentrator, and a Falcon Separator.

Gravity gold recovery on the intrusive composite was 6.1% to 9.5%, into a relatively low grade upgraded concentrate (153 to 269 gpt Au). The single test on copper scavenger tails, although not definitive, showed there may be potential to recover gold from the copper scavenger tails by gravity.

16.6 Batch Tests

Phase II

Batch flotation tests were carried out on composites and sub-composites of the Prosperity mineralization in each of the second, third and fourth phases of metallurgical testwork.

The second phase batch tests were in two categories; development tests and variability tests. Development tests investigated recovery processes to determine those best suited to the Prosperity mineralization.

Once an acceptable flotation scheme had been identified, those conditions were used in a series of variability batch tests to obtain a measure of metallurgical variability across the deposit for the upper, middle and lower zones.

Acceptable copper and gold recoveries were achieved in these tests with cleaner concentrate grades generally being in the range of 20% Cu to 30% Cu and approximately 45 gpt Au.

Phase III

Batch flotation tests were conducted in the third phase of metallurgical testwork to determine if intrusive and volcanic drill core composites could be blended for run-in pilot plant testing. The batch tests yielded good bulk recoveries (up to 95.7% copper recovery and 79.9% gold recovery for the intrusive composite; 93.4% copper recovery and 82.4% gold recovery for the volcanic composite). Good cleaner concentrate grades were also achieved (up to 25.1% Cu and 41.9 gpt Au for the intrusive composite; 28.6% and 51.6 gpt Au for the volcanic composite). Other than head grade differences there were no metallurgical differences between the intrusive and volcanic rock types.

Phase IV

In the fourth phase of metallurgical testwork, batch flotation tests were completed on volcanic and intrusive sub-composites, and on upper, middle, and lower composites prepared from assay rejects and half core composites.

Good results were achieved with respect to both copper and gold metallurgy using a relatively coarse primary grind and a conventional bulk rougher/copper-pyrite separation float. Batch tests completed on the assay rejects composites yielded poorer copper recoveries than batch tests completed on the half core composites, and when blended with half core composite the assay rejects composite lowered the copper recovery.

From the results of these tests it was decided to mix the assay rejects samples into a single composite to run in the pilot plant, and mix separate upper, middle and lower half core composites for definitive pilot plant runs.

16.7 Locked Cycle Tests

Locked cycle tests were carried out on composites and sub-composites of the Prosperity mineralization in Phase II and Phase IV of the metallurgical testwork.

In the second phase of metallurgical testwork locked cycle tests were conducted on 11 composites to investigate the metallurgical performance of the Prosperity mineralization under conditions approaching steady state.

These locked cycle test results were used to provide estimates of copper and gold recoveries for mine block model development. The average results were 85.6% copper recovery and 72.1% gold recovery into a concentrate grading 25.2% Cu and 46.2 gpt Au from an average head grade of 0.22% Cu and 0.47 gpt Au.

In the fourth phase of metallurgical testwork, eight locked cycle tests were conducted prior to and following the pilot plant runs to confirm conditions for the pilot plant runs and provide data to increase the level of confidence regarding estimates of copper and gold recoveries. The locked cycle tests completed in Phase IV were conducted using test conditions reviewed and revised during the Phase IV batch tests.

The copper results for the eight-cycle locked cycle tests (average copper recovery of 88.7%) were slightly inferior to those from the six-cycle locked cycle tests (average copper recovery of 90.8%), but comparisons of bulk rougher recoveries show that the difference was negligible. The gold results from the eight-cycle locked cycle tests (average gold recovery of 77.1%) were superior to those from the six-cycle locked cycle tests (average gold recovery of 73.8%), both in comparison of the overall recoveries and bulk rougher recoveries. Concentrate grades achieved for the eight-cycle locked cycle tests, which averaged 26.3% Cu and 44.9 gpt Au from an average head grade of 0.22% Cu and 0.44 gpt Au, were also superior to those achieved in the six-cycle locked cycle tests, which had an average concentrate grade of 22.4% Cu and 36.6 gpt Au from an average head grade of 0.22% Cu and 0.45 gpt Au.

16.8 Run-In Pilot Plant And Main Pilot Plant Runs

The run-in pilot plant was conducted in the third phase of the metallurgical testwork. Five runs were completed, three runs on a composite of volcanic assay rejects and the final two runs on combined volcanic and intrusive drill core composites. All composites represented the upper zone (<200 m) of the main (east) area of the deposit.

Because of limited sample availability, only short pilot plant runs were possible and stable circuit conditions, especially for gold, were not reached. Consequently, generally inferior results were achieved. In spite of this, run RI-PP5 bulk rougher flotation conditions yielded good bulk copper recovery (92.1%) and acceptable (79.5%) gold recovery, even at a relatively coarse primary grind K_{80} of 157 µm.

Despite the slightly oxidized nature of the drill core composites in run RI-PP4, it was possible to achieve a respectable copper concentrate grade of 22.4% Cu and 21.5 gpt Au. The main pilot plant was conducted in Phase IV of the metallurgical testwork. Eight runs were completed, five runs on an assay rejects composite and three runs on half core composites.

The average copper recovery increased from the 82.0% recovery achieved in the run-in pilot plant to an average of 86.2% for the two pilot plant runs on the assay rejects composite in which stable steady-state operating conditions were achieved and 87.6% for the runs on the half core composites. The gold recovery, which averaged 55.6% for the run-in pilot plant increased to 79.0% for the runs on the assay rejects composite but only averaged 66.5% for the runs on half core composites.

The results of the two stable pilot plant runs on the assay rejects composite (average recovery of 86.2% for copper and 79.0% for gold) were in close agreement with the six-cycle locked cycle test result (85.6% copper recovery and 78.3% gold recovery).

The pilot plant copper recovery for the half core composites (average of 87.6%) were slightly inferior to the six-cycle locked cycle test copper recoveries (average of 90.8%). In terms of gold recovery, the upper, middle and lower pilot plant recoveries (60.4%, 73.6% and 65.5% respectively) were lower than the respective six-cycle locked cycle test recoveries (68.1%, 75.2% and 78.2%).

It was believed that an overly coarse regrind cyclone overflow in the pilot plant runs on the half core composites was the cause of the low gold recoveries. Batch and eight-cycle locked cycle tests completed after the pilot plant confirmed this, when excellent copper and gold recoveries (average of 88.7% for copper and 77.1% for gold) and concentrate grades (average of 26.3% Cu and 44.9 g/t Au) were achieved on the upper, middle and lower half core composites.

16.9 Target Concentrate Grades and Recoveries

The locked cycle tests from Phase II and Phase IV of the metallurgical testwork were used to derive target concentrate copper and gold recoveries and copper concentrate grades.

Target copper and gold concentrate grades and recoveries for two different mill feed grades (Cases A and B) are estimated in Table 16-2.

Table16-2
Target Gold and Copper Recoveries and Concentrate Grades

Zone	Case No.	Head Grade		Concentrate Grade		% Recovery	
		% Cu	gpt Au	% Cu	gpt Au	Copper	Gold
Upper				25.1	36.1	90.1	70.4
Middle/Lower	A	0.236	0.434	25.1	39.5	90.1	77.1
Upper	B	0.246	0.482	25.5	40.0	90.2	72.2
Middle Lower				25.5	43.8	90.2	79.0

Calculations to estimate copper recovery based on copper head grade were derived as shown in Table 16-3.

Table 16-3
Target Copper Recovery and Target Concentrate Copper Grade Calculations
Lower, Middle and Upper Zones

Head Grade Range (% Cu)	% Copper Recovery	% Cu in Concentrate
0.10 to 0.20	(% Cu x 93.0) + 71.3	(% Cu x 48.0) + 14.2
0.20 to 0.25	(% Cu x 6.0) + 88.7	(% Cu x 36.0) + 16.6
0.25 to 0.40	(% Cu x 6.0) + 88.7	(% Cu x 6.7) + 23.9
>0.40	91.0	26.6

Target gold recovery calculations, to estimate gold recovery based on gold head grade, were derived as show in Table 16-4 and 16-5.

Table 16-4
Target Gold Recovery Calculations
Upper Zone

Head Grade Range (gpt Au)	% Gold Recovery
0.10 to 0.40	(gpt Au x 97.0) + 30.3
0.40 to 0.65	(gpt Au x 38.0) + 53.9
0.65 to 1.00	(gpt Au x 5.7) + 74.9
>1.00	81.0

Table 16-5
Target Gold Recovery Calculations
Middle and Lower Zones

Head Grade Range (gpt Au)	% Gold Recovery
0.10 to 0.29	(gpt Au x 110.0) + 39.4
0.29 to 0.49	(gpt Au x 40.0) + 59.7
0.49 to 1.00	(gpt Au x 7.5) + 75.6
>1.00	83.0

The target concentrate gold grade can be calculated from the copper head grade, copper recovery, concentrate copper grade, gold head grade, and gold recovery as follows:

Target Concentrate Gold Grade (gpt Au) =

$$\frac{(\text{Concentrate Copper Grade, \%}) \times (\text{Gold head grade, gpt Au}) \times (\text{Gold Recovery, \%})}{(\text{Copper head grade, \%}) \times (\text{Copper Recovery, \%})}$$

Figure 16-4 displays the target copper recovery versus copper head grade.

Figure 16-4
Target Copper Recovery Vs Copper Head Grade

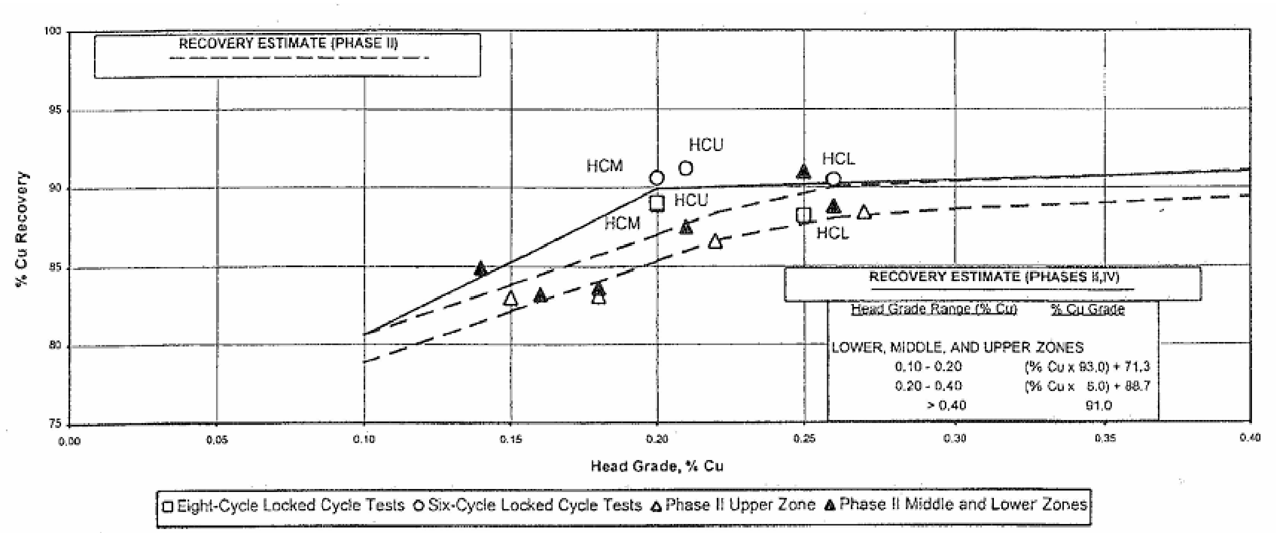


Figure 16-5 displays the target concentrate copper grade versus copper head grade:

Figure 16-5
Target Concentrate Copper Grade Vs Copper Head Grade

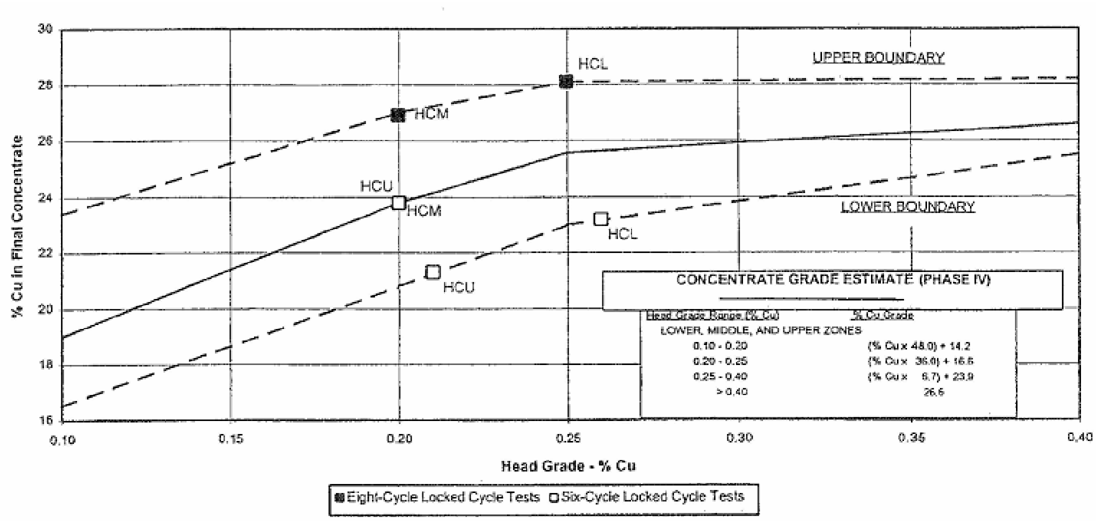
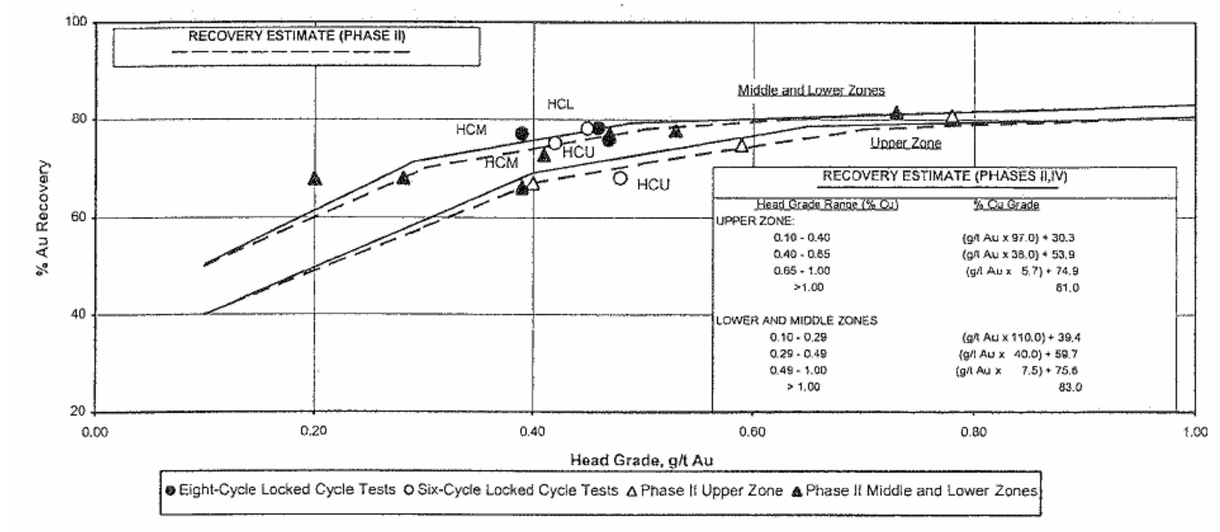


Figure 16-6 displays the target gold recovery versus gold head grade:

Figure 16-6
Target Gold Recovery Vs Gold Head Grade



16.10 Concentrate Analysis

An assessment of concentrate analyses from the locked cycle tests and pilot plant runs carried out in Phase IV of metallurgical testing yielded the values shown in Table 16-6 as typical concentrate analysis obtained in the Prosperity pilot plant program:

Table 16-6
Typical Concentrate Analysis

Zone	% Cu	gpt Au	gpt Ag	% As	ppm Hg	% Pb	% Sb	% Zn
Upper	23.1	42.8	104	0.25	144	0.36	0.38	3.57
Middle	24.4	40.1	91	0.27	105	0.19	0.37	1.02
Lower	25.3	40.1	81	0.14	68	0.06	0.29	0.55

Settling tests were completed on the bulk final concentrate from the Phase IV pilot plant run, the run carried out on the upper core composite. Results are summarized in Table 16-7.

Table 16-7
Settling Tests on Concentrate from Upper Composite

Test No.	Percol 351 Addition		Density [% Solids(w/w)]		Supernatant	Concentration Zone	
	Solids (gpt)	Liquid (mg/L)	Initial	Final	Clarity (ppm)	Settling Rate ($m^3/m^2/d$) (1)	Unit Area ($m^2/t/d$) (1)
S9	20.3	4.9	19.6	69.6	62	319.1	0.084
S10	11.3 + 5.6	4.1	19.7	71.0	2	326.4	0.084
S11	0	0	19.8	75.3	80	29.4	0.299

Pressure filtration testing by Larox Inc. achieved concentrate moistures of 8.0% to 9.1% solids (w/w). Filtration rates as high as 544 $kg/m^2/hr$ were obtained.

CERAMEC filtration tests conducted by Outokumpu Minetec USA Inc. achieved a moisture level of 14.5 to 14.8% solids (w/w) with filtration rates of 185 $kg/m^2/hr$ to 327 kg m^2/hr.

16.11 Tailings Settling Tests

Four sets of flocculant scoping tests were completed on the combined tailings produced in Locked Cycle Test No. M2 in Phase II of metallurgical testing. These tests indicated that the preferred flocculant was Percol 919, and that an addition rate of 20 to 25 gpt was satisfactory.

Following the scoping tests, fifteen variability settling tests were conducted on tailings generated in the six locked cycle tests completed on the west zone and main zone composites.

In Phase III, a series of four settling tests were performed on run-in pilot plant run RI-PP5 tailings.

In Phase IV of metallurgical testing, separate settling tests were performed on each final tailings stream from the three half core pilot plant runs. Settled tailings densities of 62 to 66% solids (w/w) were achieved with thickener unit areas varying from 0.247 to 0.351 $m^2/t/d$

16.12 Environmental Data

Tails solids and liquid from three Phase II locked cycle tests on Composites were collected and submitted to Saskatchewan Research Council (Saskatoon, Saskatchewan) for detailed low-level analysis. Tails from Phase II locked cycle tests were submitted for detailed environmental analysis. Data collected included detailed analysis of tails solids, tails liquid (total and dissolved), standard Special Waste Extraction Procedure (SWEP), simulated rainfall test, acid-base accounting, and bio-assay tests.

In Phase III, environmental data was collected for tails samples from run-in pilot plant run RI-PP5 and from five alteration composites (intrusive sericite, intrusive potassium silicate, volcanic sericite, volcanic potassium silicate, and volcanic propylitic). Phase IV environmental data was collected from tails samples from pilot plant runs PP6, PP7 and PP8. Data collected included detailed analysis of head samples, tailings solids and tailings liquid, acid-base accounting measurements on heads and tailings samples, rainbow trout and daphnia magna toxicity tests on tailings decant liquid, and separate tailings aging tests with analysis of solids and liquids for the run-in pilot plant run RI-PP5, and the pilot plant runs on the upper, middle, and lower half core composites.

Presentation of this data has been provided in the Melis Phase II, Phase III and Phase IV reports. Analyses of the environmental data have been conducted by environmental experts.

17. Mineral Resource and Mineral Reserve Estimates

17.1 Resource Modeling

The property exploration and sampling work culminated in the comprehensive geological interpretation and delineation of a major, porphyry-style gold-copper resource, based on a total of 150,090 m of core drilling in 402 holes and 6,309 m of drilling in 68 percussion holes, with an average drill hole spacing of 70 m.

In January 1998 Giroux Consultants Ltd. (GCL) commenced geostatistical analysis of data from the Prosperity Project Database. The Prosperity Project database consists of all relevant drill hole collar and downhole survey data, assay and ICP data, and geological and geotechnical data.

Capping and Compositing of Assay Results

An analysis of high-grade outlier values was conducted in order to determine a 'capping' methodology prior to compositing of assays. Respecting geological boundaries, GCL created 8,976 downhole composites 15 m in length. A total of 8 distinct geologic domains were coded in preparation for variography and ordinary kriging of block grades. An additional domain code was assigned solely to the post-mineral dikes large enough to form discrete blocks (irrespective of alteration or geographic domain). These blocks were not kriged but a global average grade was assigned, as discussed in the section on post mineral dikes below.
These composites were used to estimate the gold and copper grades of 289,638 model blocks. The dimensions of the blocks are 20 m by 20 m horizontally and 15 m thick.

Variography

GCL calculated a series of semi-variograms for geologic domains for which there was sufficient data. For domains with insufficient composite data, simplified spherical models were fitted.
For modeling purposes the various alteration, lithologic, and geographic domain information was combined into a number of unified 'geologic' domains to which the assay data could be linked. Within this block model, Cu and Au grades for each block were estimated independently by ordinary kriging, utilizing search parameters obtained from the variography analyses.

Kriging

Using the domains established above, a grade model was estimated by ordinary kriging within a three-dimensional grid comprising 20 x 20 x 15 m blocks coded by geologic domain. The block model extents in Mine Grid are shown in Table 17.1.

Table 17-1
Block Model Extents

	Easting	Northing	Elevation
Lower SW corner	8990E	8610N	540
Upper NE corner	11510E	11010N	1605
Number of blocks	126	120	71

Within the block model, Cu and Au grades for each block were estimated independently by ordinary kriging, using the search parameters obtained from the variography. Initial search ellipsoids were defined as 1/3 of the variogram range. Provided that a minimum of four composites were found within the initial search ellipsoid for the appropriate element, a grade estimate was calculated. However, if the minimum four composites were not found in the initial search, the axes of the ellipsoids were expanded by a factor of 1.5. If the requisite composites were still not found the expansion was then doubled to the full range of the variogram. If after two expansions, the minimum four composites were still not found, then the block value was not estimated. Alternatively, if more than 15 composites were found during the search procedure, the 15 closest to the block centroid were used for estimation.

From previous work it was known that a number of sporadic, narrow, high-grade gold veins existed, particularly in the propylitic rocks in the periphery of the deposit. In order to prevent these values from unduly influencing large volumes of surrounding rock it was necessary to restrict their influence. The procedure used was as follows: individual assays in propylitic rocks with gold grades in excess of 1 g/t were identified, and the corresponding composites flagged (flag = 2). If a search ellipsoid located a gold vein composite, within the block to be estimated, the composite was used in the estimation of that block. If a search ellipsoid located a flagged gold vein composite which was outside the block to be estimated, it was not used to estimate that block. A similar strategy was adopted for dealing with the narrow, low grade post mineral dikes, which essentially have the opposite effect on grade estimation of nearby blocks as the high-grade veins.

17.2 Resource/Reserve Classification

Each block within the model was classified as measured, indicated or inferred in terms of resource classification, corresponding with proven or probable respectively, in terms of reserve classification. A kriging estimation error was calculated for each estimated block for both copper and gold which takes into account the nugget effect, sill value, number of composites used in the estimate and the spatial relationships of composites relative to any anisotropy. The kriging estimation error was then used to compute a grade-linked relative kriging estimation error as follows:

Relative Estimation Error = (Kriging Estimation Error / grade) * 100%

For both Cu and Au, separate histograms of these relative estimation errors were plotted and the data divided into three populations based on the magnitude of the error. With reference to these three divisions, resource blocks were classified for each element as measured, indicated, inferred or proven, probable according to the set of rules summarized below:

Measured/Proven

- Blocks with relative estimation errors for Cu of less than 24%
- Blocks with relative estimation errors for Au of less than 40%.

Indicated/Probable

- Blocks with relative estimation errors for Cu greater than or equal to 24% and less than 49%
- Blocks with relative estimation errors for Au greater than or equal to 40% and less than 70%.

Inferred

- Blocks with relative estimation errors for Cu greater than or equal to 49%.
- Blocks with relative estimation errors for Au greater than or equal to 70%

Since the economics of the deposit depend on both gold and copper, the final classification of the (in-situ) resource block considered the combined effects of the individual block classifications. This, in turn necessitated an additional set of rules, as set out below:

- For the overall block to be classed as measured/proven, relative estimation errors for both Cu and Au had to fall in the measured category.
- If relative estimation errors for *either* Cu or Au fell into the indicated/probable category, while the other metal was classed as measured/proven, then the overall block was classed as indicated/probable.
- If relative estimation errors for both Cu and Au fell into the indicated/probable category then the overall block was classed as indicated/probable.
- If relative estimation errors for *either* Cu or Au fell into the inferred category, then the overall block was classed as inferred, irrespective of the classification of the other value.
- If during kriging, expansion of the initial search ellipsoid of 1/3 the variogram range was required to estimate either copper *or* gold, then the overall block was classed as inferred.
- Any blocks assigned a grade directly (e.g. blocks of post mineral dike) were classed as inferred.

In December 1998 the classification scheme originally applied by Giroux was re-examined, for the reasons outlined below. In particular the use of the expanded search criteria was re-evaluated. In discussions with Giroux, it was mutually agreed that within the core of the deposit as defined by the proposed ultimate pit at the time, the relative estimation error alone was sufficient for classifying the mineral resource. However, outside this boundary, the expanded search criteria in addition to the relative estimation error were applied in classifying resources.

The following is a brief clarification on the classification methodology. During estimation of the resource model, the expanded search criterion of 1/3 the variogram range was used to protect against over-classifying blocks on the margins of the deposit, which could only be estimated using an expanded search owing to a paucity of drill information and relatively long variogram ranges. The outcome of this however, was that certain blocks in the core of the deposit, that met all the other requirements for being classed as measured and/or indicated, were classed as inferred, simply because they required an expanded search during kriging. A number of these blocks even had data points within their boundaries. To accommodate this problem during classification of resources within the main deposit area, the practice of classifying every block estimated during an expanded search as inferred was dropped, and blocks within the optimized pit were re-classified, solely on the basis of relative kriging estimation errors.

Table 17-2
Prosperity Mineral Resources

at 0.14% Copper Cut-off			
Category	**Tonnes (millions)**	**Gold (gpt)**	**Copper (%)**
Measured	547.1	0.46	0.27
Indicated	463.4	0.34	0.21
Total	**1,010.5**	**0.41**	**0.24**

17.3 Mining

Optimization Methodology (2000)

The Prosperity Project Pit Optimization was completed in 2000 using Medsystem® mine planning software. A series of pit shells were created at varying discount levels for anticipated revenue and costs. These pit shells were then used as a basis to develop preliminary production schedules and discounted cash flows. The results of the cash-flow analyses were used to guide the selection of concentrator throughput and ultimate design limits used in that study.

The economic parameters used in the optimization were appropriate to 2000. These parameters were analyzed in respect of the 2007 feasibility study costs and economic assumptions, validating the pit cut-off grade parameters in the 2000 optimization.

Net Smelter Return Model

The basis for 2000 pit optimization was the Net Smelter Return (NSR) model, a calculation of insitu ore value based upon an estimate of metal content, metallurgical recovery, metal price, off property costs and the currency exchange rate. This model then provides an estimate of the NSR value for each block expressed in C$/t for all interpolated blocks in the geological resource model. The NSR value incorporates the following economic evaluation criteria:

- Variable metallurgical recovery and concentrate grade based upon head grade for copper and gold in the Upper, Middle and Lower geological domains (Melis Engineering Pilot Plant Program Report).
- A fixed silver concentrate grade based upon testwork results.
- Concentrate transportation, treatment, penalty and refining terms (Butterfield Mineral Consultants Marketing Study December 1998).
- Fixed metal price and exchange rate parameters terms (Butterfield Mineral Consultants Marketing Study December 1998).

An example of the NSR calculation is shown in Table 17-3. The metallurgical recoveries were assigned first and a gross metal value was calculated. Adjustments were then made for transportation, treatment and refining charges. The NSR value was then calculated for each block of the resource model and stored in the model.

Table 17-3 Net Smelter Return 2000

Parameters and Test Block Calculation

Test Block NSR Calculation										
Copper Head Grade	%									0.218
Gold Head Grade	g/t									0.427
Location										Lower Zone
Metallurgical Recovery										
Copper Recovery										
		Cu %		**Upper, Middle & Lower Copper Recovery**						**%**
Head Grade Range	>=	0.100	<	0.200	Cu x	93.0	+	71.3	=	
Head Grade Range	>=	0.200	<	0.400	Cu x	6.0	+	88.7	=	90.01
Head Grade Range	>=	0.400							=	91.00
Gold Recovery										
		Au g/t		**Upper Zone Gold Recovery**						**%**
Head Grade Range	>=	0.100	<	0.400	Au x	97.0	+	30.3	=	
Head Grade Range	>=	0.400	<	0.650	Au x	38.0	+	53.9	=	
Head Grade Range	>=	0.650	<	1.000	Au x	5.7	+	74.9	=	
Head Grade Range	>=	1.000							=	81.00
		Au g/t		**Lower & Middle Zone Gold Recovery**						**%**
Head Grade Range	>=	0.100	<	0.290	Au x	110.0	+	39.4	=	
Head Grade Range	>=	0.290	<	0.490	Au x	40.0	+	59.7	=	76.78
Head Grade Range	>=	0.490	<	1.000	Au x	7.5	+	75.6	=	78.80
Head Grade Range	>=	1.000							=	83.00
Metal Pricing										
Copper Price	US$/lb	$0.95								$0.95
Gold Price	US$/ounce	$325.00								$325.00
Silver Price	US$/ounce	$6.50								$6.50
USD:CDN Exchange		$0.68								$0.68
Concentrate										
Copper Concentrate Grade										
			% Cu			**Copper Concentrate Grade**				**Cu %**
Head Grade Range	>=	0.100	<	0.200	Cu x	48.00	+	14.2	=	24.66
Head Grade Range	>=	0.200	<	0.250	Cu x	36.00	+	16.6	=	24.45
Head Grade Range	>=	0.250	<	0.400	Cu x	6.70	+	23.9	=	25.36
Head Grade Range	>=	0.400							=	26.60
Gold Concentrate Grade	g/dmt	Calculated from head grade, recovery and concentrate production								
Silver Concentrate Grade	g/dmt	89.00								89.00
Moisture Content	%	8.0%								8.0%
Contained Copper	lb/dmt									538.83
Contained Gold	g/dmt									40.85
Contained Silver	g/dmt									89.00
Payable Copper	lb/dmt									516.79
Payable Gold	g/dmt									39.83
Payable Silver	g/dmt									84.55
Concentrate - Recovery Based	dmt/t ore									0.00803
Gross Value of Concentrate after Deductions										
Gross Value Concentrate	C$/dmt									**$1,359.99**
Concentrate Handling										
Truck Haul Mine to Rail	C$/wmt	$12.95								
Rail Freight	C$/wmt	$23.87								
Stevedoring	C$/wmt	$16.50								
Ocean Freight	US$/wmt	$24.84								
Total Concentrate Handling	C$/wmt	$89.85								
	C$/dmt	$97.66								**$97.66**
Treatment and Refining										
Deduction for Copper	unit	1.00	95.9%							1.00
Treatment Charges	US$/dmt	$100.00								$100.00
Gold Payment	%	97.5%								97.5%
Silver Payment	%	95.0%								95.0%
Copper Refining Cost	US$/payable lb	$0.10								$0.10
Gold Refining Cost	US$/payable oz	$7.00								$7.00
Silver Refining Cost	US$/payable oz	$0.45								$0.45
Total Treatment and Refining	C$/dmt									**$238.04**
		As	**Sb**	**Hg**	**Total**					
Penalties Upper Zone	US$/dmt	$4.50	$13.20	$18.75	**$36.45**				C$/dmt	**$53.60**
Penalties Middle Zone	US$/dmt	$3.00	$11.20	$10.35	**$24.55**				C$/dmt	**$36.10**
Penalties Lower Zone	US$/dmt	$3.00	$11.20	$10.35	**$24.55**				C$/dmt	**$36.10**
Net Smelter Return										
Net Smelter Return	C$/dmt									**$988.18**
	C$/payable lb Cu									$1.91
NSR	**NSR $/t**									**$7.93**

Metallurgical Recovery

The metallurgical recovery relationships used in the NSR calculation were based upon results of the 1997 Pilot Plant program conducted at Lakefield Research under the direction of Melis Engineering. As part of the resource modeling process the deposit was subdivided laterally and vertically into three geological domains. For the purposes of NSR calculation the Upper and Middle/Lower Domains were assumed to have quantifiable differences in gold recovery.

Metal Prices

The copper concentrate will contain three payable metals – copper, gold and silver. The NSR calculation was based upon the following metal prices quoted in US dollars:

Copper	US$0.95/lb
Gold	US$325/ounce
Silver	US$6.50/ounce

The exchange rate used for converting US$ to Cdn$ was US$0.68/Cdn$1.00.

Concentrate Grade

The copper concentrate grade for copper was calculated for each block based upon the copper grade. Four copper grade ranges were established with different copper concentrate grade formulae. Blocks below 0.1% Cu were not considered and blocks above 0.4% Cu were assigned a maximum concentrate grade of 26.6% Cu. The minimum concentrate grade possible with this calculation approach was 19.0% Cu.

The gold concentrate grade was based upon the gold head grade, estimated recovery and the copper concentrate production from the theoretical block grade.

The silver concentrate grade was fixed at 89 grams per dry metric tonne (g/dmt).

Concentrate Penalties

Concentrate penalties for arsenic, antimony and mercury were assigned based upon the location of the block in the upper, middle or lower geological domains.

Transportation Cost

The concentrate transportation costs were trucking, rail haulage, stevedoring, and ocean freight. These were identified on a wet metric tonne basis and summarized, for an 8% moisture content, in Canadian dollars per dry metric tonne of concentrate. The total transportation cost $Cdn97.66/dmt was used in the NSR calculation.

Operating Cost Basis for Optimization

The operating cost assumptions made for the 2000 pit optimization have been summarized in Table 17-4. The costs were assigned for ore, waste and location above or below the gypsum line which represents a significant change in the rock strength that is expected to impact the drilling and blasting costs. A cost increment of $0.028/tonne/bench was included to allow for increasing haulage according to the depth of pit development. Below the 1200 m elevation the cost increment was reduced to $0.015/tonne/bench.

Table 17-4 Pit Optimization Parameters

Model Block Size		
X Direction	metres	20.0
Y Direction	metres	20.0
Z Direction	metres	15.0
Volume	bcm	6,000.0
Specific Gravity		
Ore	t/bcm	variable by rock type
Waste	t/bcm	variable by rock type

Block Value

NSR Calculation

Zone Item

Blocks defined above or below gypsum line

Mining Recovery

100%

Mining Dilution

Diluted to mineable block size

Operating Costs

		Above Gypsum Line		Below Gypsum Line		
Mine Operating Costs		**Ore**	**Waste**	**Ore**	**Waste**	**Overburden**
Drilling	$/t mined	$0.021	$0.021	$0.032	$0.032	
Blasting	$/t mined	$0.097	$0.097	$0.121	$0.121	
Loading	$/t mined	$0.060	$0.060	$0.060	$0.060	
Hauling	$/t mined	$0.239	$0.289	$0.239	$0.289	
Dozing	$/t mined	$0.020	$0.020	$0.020	$0.020	
Mine Electrical	$/t mined	$0.044	$0.044	$0.044	$0.044	
Mine General	$/t mined	$0.170	$0.170	$0.170	$0.170	
Subtotal	$/t mined	**$0.651**	**$0.701**	**$0.686**	**$0.736**	
Sustaining Capital	$/t mined	$0.041	$0.041	$0.041	$0.041	
Total Base Mining	$/t mined	**$0.692**	**$0.742**	**$0.727**	**$0.777**	**$0.928**
Processing Operating Costs						
Mill Operating	$/t milled	$2.160				
Tailings	$/t milled	$0.035				
Plant Services	$/t milled	$0.210				
General & Administration	$/t milled	$0.370				
Subtotal	$/t milled	**$2.775**				
Sustaining Capital	$/t milled	$0.071				
Total Processing	$/t milled	**$2.846**				
Entrance Bench		1485	1485	1485	1485	
Cost increment to 1200	$/t/bench	$0.028	$0.028	$0.028	$0.028	
Cost increment below 1200	$/t/bench	$0.015	$0.015	$0.015	$0.015	

Pit Slope Angles

Wall Slopes for Optimization 45°

Wall Slopes

Detailed wall slope recommendations were provided in a report by Knight Piesold, "Feasibility Design of the Open Pit", dated April, 1999.

For the purposes of pit optimization a fixed wall slope of 45° was used.

Condensed Model

The pit optimization was done using the Lerch Grossman algorithm for pit limit definition. The Lerch Grossman program requires a single value condensed model. In this case a dollar model was used representing the NSR in each block.

Additional important information included in the model and required for optimization includes:

- Copper grade – used for reporting
- Gold grade – used for reporting
- Zone item indicating whether the block is above or below the gypsum line for cost calculation
- Class item indicating measured, indicated or inferred resource classification
- ABA item indicating the acid generation potential of material in the block – used for scheduling
- COSTID item which is used to store the cost data for the block based upon material type, elevation, zone and location

Use of Discounting

The internal pit phases of a mine design are generally defined based upon ranking progressively larger pit shells and smoothing these to a logical minimum operating width and development sequence. The ultimate pit selection is generally more complicated and can be relatively subjective based upon the objectives of the operator. In the case of an undeveloped mine the objectives are generally to maximize NPV while minimizing capital costs and risk. In this case the largest pit generated by a pit optimizer using fixed un-discounted costs and revenues does not always provide the optimum mine plan.

The approach taken to develop the ultimate or fourth phase of pit expansion in this study was to effectively discount both the costs and revenues by bench elevation. The assumptions and steps in the process were as follows:

- Phase 4 Mining – assumed to commence in Year 6 and be completed in Year 15
- Discounting would not be applied to upper benches above 1470 elevation
- Discounting would be applied on a factored basis to material on benches between 1035 and 975 elevation. Nine years of discounting was assumed.
- Discounting was applied to costs and NSR values prior to optimization.
- A cutoff of $4.70 NSR was applied assuming:
- Ore mining bottom bench $1.80/tonne + Processing $2.90/tonne =$4.70/tonne
- A range of discount rates were applied from 12% to 35%

The fourth phase pit designs at discount rates of 12%, 20%, 25% and 35% were used to generate production schedules at 60,000 tonnes/day to compare to an earlier Phase 4 Design. In addition to comparing all of these pits at 60,000 tonnes/day throughput rate, elevated throughputs of 70,000 tonnes/day, 80,000 tonnes/day and 90,000 tonnes/day cases were evaluated.

The final selection of a 70,000 tonnes/day throughput for the 35% discount pit was based upon the conclusion that the NPV's were very similar in the 60,000 tonnes/day and 70,000 tonnes/day cases but could be achieved more quickly in the 70,000 tonnes/day case.

Validation of Methodology (2007)

The 2000 optimization was performed under the economic assumptions appropriate at that time. That optimization provided the ultimate phase 4 shell that is the basis of the mine plan and reserves presented in this report. As of 2007 the only changes that would impact the optimization are metal price, currency exchange rate, off property costs and operating costs. It is necessary to demonstrate that the 2000 optimization used to produce the current pit design remains valid in terms of economical viability and the definition of ore reserves. This is best done by a ensuring that the increase in NSR value per block in 2007 relative to 2000 is greater than the increase in operating costs determined in the 2007 feasibility than the assumed operating costs in 2000.

Because of the variability in gold to copper ratios within the pit, this test must hold true for all Au:Cu ratios. The range of variability of Au:Cu ratios within the ultimate phase 4 pit is 0.33 to 6.7. The life of mine average is 1.96:1.

The NSR calculations for 2007 are shown in Table 17-5.

Table 17-5 2007 NSR Calculations

		Parameters and Test Block Calculation								
Test Block NSR Calculation										
Copper Head Grade	%									0.150
Gold Head Grade	g/t									0.290
Location										Lower Zone
Metallurgical Recovery										
Copper Recovery										
		Cu %		**Upper, Middle & Lower Copper Recovery**						**%**
Head Grade Range	>=	0.100	<	0.200	Cu x	93.0	+	71.3	=	85.25
Head Grade Range	>=	0.200	<	0.400	Cu x	6.0	+	88.7	=	89.60
Head Grade Range	>=	0.400							=	91.00
Gold Recovery										
		Au g/t		**Upper Zone Gold Recovery**						**%**
Head Grade Range	>=	0.100	<	0.400	Au x	97.0	+	30.3	=	
Head Grade Range	>=	0.400	<	0.650	Au x	38.0	+	53.9	=	
Head Grade Range	>=	0.650	<	1.000	Au x	5.7	+	74.9	=	
Head Grade Range	>=	1.000							=	81.00
		Au g/t		**Lower & Middle Zone Gold Recovery**						**%**
Head Grade Range	>=	0.100	<	0.290	Au x	110.0	+	39.4	=	71.30
Head Grade Range	>=	0.290	<	0.490	Au x	40.0	+	59.7	=	71.30
Head Grade Range	>=	0.490	<	1.000	Au x	7.5	+	75.6	=	
Head Grade Range	>=	1.000							=	83.00
Metal Pricing										
Copper Price	US$/lb	$1.50								$1.50
Gold Price	US$/ounce	$575.00								$575.00
Silver Price	US$/ounce	$8.00								$8.00
USD:CDN Exchange		$0.80								$0.80
Concentrate										
Copper Concentrate Grade										
			% Cu			**Copper Concentrate Grade**				**Cu %**
Head Grade Range	>=	0.100	<	0.200	Cu x	48.00	+	14.2	=	21.40
Head Grade Range	>=	0.200	<	0.250	Cu x	36.00	+	16.6	=	22.00
Head Grade Range	>=	0.250	<	0.400	Cu x	6.70	+	23.9	=	24.91
Head Grade Range	>=	0.400							=	26.60
Gold Concentrate Grade	g/dmt	Calculated from head grade, recovery and concentrate production								
Silver Concentrate Grade	g/dmt	89.00								89.00
Moisture Content	%	7.5%								7.5%
Contained Copper	lb/dmt									471.66
Contained Gold	g/dmt									34.60
Contained Silver	g/dmt									89.00
Payable Copper	lb/dmt									449.62
Payable Gold	g/dmt									33.74
Payable Silver	g/dmt									80.10
Concentrate - Recovery Based	dmt/t ore									0.00598
Gross Value of Concentrate after Deductions										
Gross Value Concentrate	C$/dmt									**$1,648.43**
Concentrate Handling										
Truck Haul Mine to Rail	C$/wmt	$28.16								
Rail Freight	C$/wmt	$22.97								
Stevedoring	C$/wmt	$24.50								
Ocean Freight	US$/wmt	$56.50								
Total Concentrate Handling	C$/wmt	$146.26								
	C$/dmt	$158.11								**$158.11**
Treatment and Refining										
Deduction for Copper	unit	1.00	95.3%							1.00
Treatment Charges	US$/dmt	$90.00								$90.00
Gold Payment	%	97.5%								97.5%
Silver Payment	%	90.0%								90.0%
Copper Refining Cost	US$/payable lb	$0.09								$0.09
Gold Refining Cost	US$/payable oz	$6.00								$6.00
Silver Refining Cost	US$/payable oz	$0.45								$0.45
Total Treatment and Refining	C$/dmt									**$172.67**
		As	**Sb**	**Hg**	**Total**					
Penalties Upper Zone	US$/dmt	$4.50	$8.40	$24.80	**$37.70**				C$/dmt	**$47.13**
Penalties Middle Zone	US$/dmt	$5.10	$8.10	$17.00	**$30.20**				C$/dmt	**$37.75**
Penalties Lower Zone	US$/dmt	$1.20	$5.70	$9.60	**$16.50**				C$/dmt	**$20.63**
Net Smelter Return										
Net Smelter Return	C$/dmt									**$1,279.90**
	C$/payable lb Cu									$2.85
NSR	**NSR $/t**									**$7.65**

The 2000 NSR cut-off for Lerch Grossman optimization was $4.70/t. This value included the mining cost on the bottom bench, all on site operating costs, and a sustaining capital cost allowance. Back-calculation of Cu and Au cut-off grades equivalent to this NSR cut-off for the range of Au:Cu ratios in the ultimate phase 4 pit provides the values shown in table 17-6. Converting these cut-off grades to 2007 NSR values shows an increase in value of $2.57/tonne to $2.95/tonne relative to 2000.

Comparing this increase in value per tonne to the lesser increase in equivalent operating costs relative to 2000 clearly shows that all blocks previously identified as ore in 2000 are classified as ore in 2007 and the ultimate phase 4 pit defined in 2000 is valid as an economically viable shell subject to capital costs and a positive cash flow.

Table 17-6
LG Cut-off Validation 2000-2007

Au:Cu Ratio	2000			2007 NSR calculated from 2000 BE grades ($/tonne)	Increase in NSR block value ($/tonne)	2000 total bottom bench op cost ($/tonne)	2007 total bottom bench op cost ($/tonne)	Increase in total bottom bench op cost ($/tonne)
	NSR used ($/tonne)	Calculated Breakeven (BE) grades						
		Cu (%)	Au (gpt)					
0.33	$4.70	0.27	0.09	$7.42	$2.72	$4.70	$6.24	$1.54
1.96		0.15	0.29	$7.65	$2.95			
6.7		0.06	0.38	$7.27	$2.57			

The 2000 NSR cut-off for a pit rim decision as to whether material was waste or destined for ultimate processing was $3.25/tonne. This value included all onsite costs except mining, with a provision for rehandle. i.e. any material with an NSR greater than $3.25 would either go directly to the mill or to stockpile, dependant on the availability of higher grade material at the time.

In the 2007 feasibility study an NSR pit rim cut-off of $5.25/tonne has been used on the basis of life-of-mine, non-mining site costs of $3.99/tonne milled plus a rehandle mining cost of $0.51/tonne milled, totaling $4.50/tonne. Allowing for reduced metallurgical recovery from stockpile this material still demonstrates a positive operating profit.

In 2007, the pit was redesigned to incorporate revised pit slopes recommended by Knight Piesold detailed in the report, “2007 Feasibility Pit Slope Design”, dated September 2007, a ramp relocation, and crest and toe definition instead of mid-bench contours as outlined in Section 18-2.

The total mineable reserve at a C$5.25 NSR /tonne cutoff grade is 487 million tonnes with an average grade of 0.22% Cu and 0.43 gpt Au. The average NSR value of this ore is C$11.59/tonne. The total overburden and waste is 399.3 million tonnes for a strip ratio of 0.82:1.

18. Additional Requirements for Technical Reports on Development Properties and Production Properties

18.1 Site Infrastructure

The ancillary facilities and services for the Prosperity Project comprise the following:

- building structures, including the service complex (administration, vehicle maintenance, tire repair, warehouse), assay laboratory and explosive plant;
- power supply from the BC Hydro grid, transmission to site, and project site distribution;
- services, including fresh water supply, fire/fresh water storage and distribution, recycled water collection/storage/distribution, fuel storage and dispensing, sewage collection and treatment, drainage and runoff settling ponds;
- housing and recreational facilities for construction personnel and operating employees;
- project site access roads;
- plant site roads, yard areas and parking, and
- security, safety, and first aid facilities

Site Access Road

At present vehicle access to the site from Williams Lake is via Provincial Highway No. 20 and forestry resource roadways. The road between Williams Lake and the mine site (approximately 180 km) must be an all-weather road and will comprise a portion of the following:

- Provincial Highway No. 20 - 90 km of 2 lane, paved road;
- Taseko Lake (Whitewater) Logging Road - 68 km of one-lane, 5 m wide, gravel road with turnouts;
- 4500 Road (Riverside Haul Road) - 19 km of one-lane, 5 m wide, gravel road with turn-outs;
- and a new Project Site Access Road - 2.8 km of 2 lane, 8 m wide, gravel road.

This road system will serve as the principal access road both during construction and mine operation. Road rehabilitation work will be required on 19 km of the 4500 Road prior to commencement of mine construction. Construction of the 2.8 km Project Site Access Road connection to the 4500 Road will be one of the first construction activities for the project.

Power Supply and Distribution

B.C. Hydro will supply power to the Prosperity Project via a 125 km, 230 kV overhead transmission line. A Switching Station will be built by B.C. Hydro at Dog Creek adjacent to the existing 230 kV transmission line, which runs south from the Soda Creek substation, near Williams Lake, to the Kelly Lake substation north of Lillooet. The Switching Station will supply the Prosperity Project 230 kV overhead line that will be built and owned by the Prosperity Project. Further details of this 230 kV overhead line is provided in the report by Ian Hayward International Ltd.

The 230 kV line will be terminated at a structure to be erected within the fenced compound of the main site substation. This will be located close to the concentrator building and will include the main 230 kV circuit breakers protecting three 75/100/125 MVA 230 kV - 25.0 kV three phase oil-filled transformers. The facility will operate on one transformer, with the second unit maintained as an off line spare.

A diesel powered standby generator plant will be used to supply emergency power. The generators will supply power via a transformer to the 25 kV, metal clad, switchgear located at the Main Substation. Initially, the generators will be used for construction power. Following the mine and process plant commissioning, they will be available to supply emergency power to selected critical loads in the process plant and camp in the event of an outage in the B.C. Hydro power supply.

Power will be distributed throughout the project site via the 25 kV switchgear located in the main substation switchgear building.

An Electrical Load Analysis has been performed based on data extracted from the project equipment list. The analysis utilized motor efficiency and power factor figures at three quarters load from a typical North American manufactures published data. For all motors, except those of the mills, maximum demand load was assumed to be 90% of the connected load. Thus motors are 10% oversized to allow for actual driven machine load variations over duty cycle, and average demand was set at 85% of the maximum demand load. For the SAG, Ball and Regrind mills, maximum demand will correspond to connected load and average demand will be 85% of the maximum. Standby units are included in the connected load but are assigned a zero demand and load factor.

Mining equipment, especially the shovels, have a power demand curve which varies considerably over the repetitive operating cycle. During the cycle there are periods when regeneration occurs. For this reason, data from a typical manufacturer has been used to represent the mining load as accurately as possible. The mining load is a significant portion of the total site load. It is important to recognize that the way that this equipment is operated will influence the overall energy requirements for the project and hence annual costs.

In order to calculate annual energy consumption, availabilities of 67% and 92% have been used for the crushing plant and concentrator, respectively. Certain equipment, such as sump pumps, overhead cranes, hoists and other similar service equipment within those plants, can be assumed to operate intermittently.

Lighting loads have been based on building floor areas in conjunction with an estimated load of 3 Watts per square foot. In the buildings, lighting is assumed to be operated on a continuous basis.

Table 18-1 contains the electrical load analysis summary for Year 6.

Table 18-1 Electrical Load Analysis – Year 6

Electrical Summary				
Area	**Description**	**Operating Power, kW (at shaft)**		**Energy Consumption, MWh/y based on a 94% motor efficiency**
		Normal Consumed	**Peak Consumed**	
131	Water Systems	265	2,013	2,467
132	Fuel Systems	47	120	436
200	Mining Equipment	3,576	8,940	33,325
250	Pit Dewatering	1,764	2,205	16,439
260	Pit Depressurization	33	41	308
310	Crushing	3,088	4,961	28,780
350	Conveyors, Stockpile Reclaim	585	1,035	5,451
405	Concentrator Building(includes 5000kW for HVAC)	2,544	4,200	23,708
410	Grinding	42,472	60,171	395,803
420	Flotation	6,355	9,631	59,225
425	Regrind	7,175	9,789	66,864
430	Concentrate Thickening	255	787	2,377
440	Concentrator Reagents	74	110	693
450	Concentrate Loadout	7	40	67
460	Process Water	616	872	5,739
470	Substation & Power Distribution	-	90	-
505	Talings Pumping	1,346	1,896	12,543
520	Water Supply	2,700	5,000	25,162
530	Water Storage & Distribution	150	208	1,396
611	Eng & Admin. Facitlites	375	500	3,495
612	Mine Service Facilities	157	268	1,461
650	Assay Laboratory	300	600	2,796
660	First Aid Room	-	-	-
710	Operations Camp	500	1,000	4,660
TOTAL PROJECT		**74,384**	**114,475**	693,193

Mill Building and Concentrate Load-out

The grinding circuit, flotation circuits, reagent mixing and concentrate dewatering will be contained in a single building with an 80 m x 75 m grinding section and a 56 m x 96 m flotation/regrind/filter section. As shown in Figure 18-2.

The general site layout site is shown in Figure 18-1.

Figure 18-1 General Site Layout

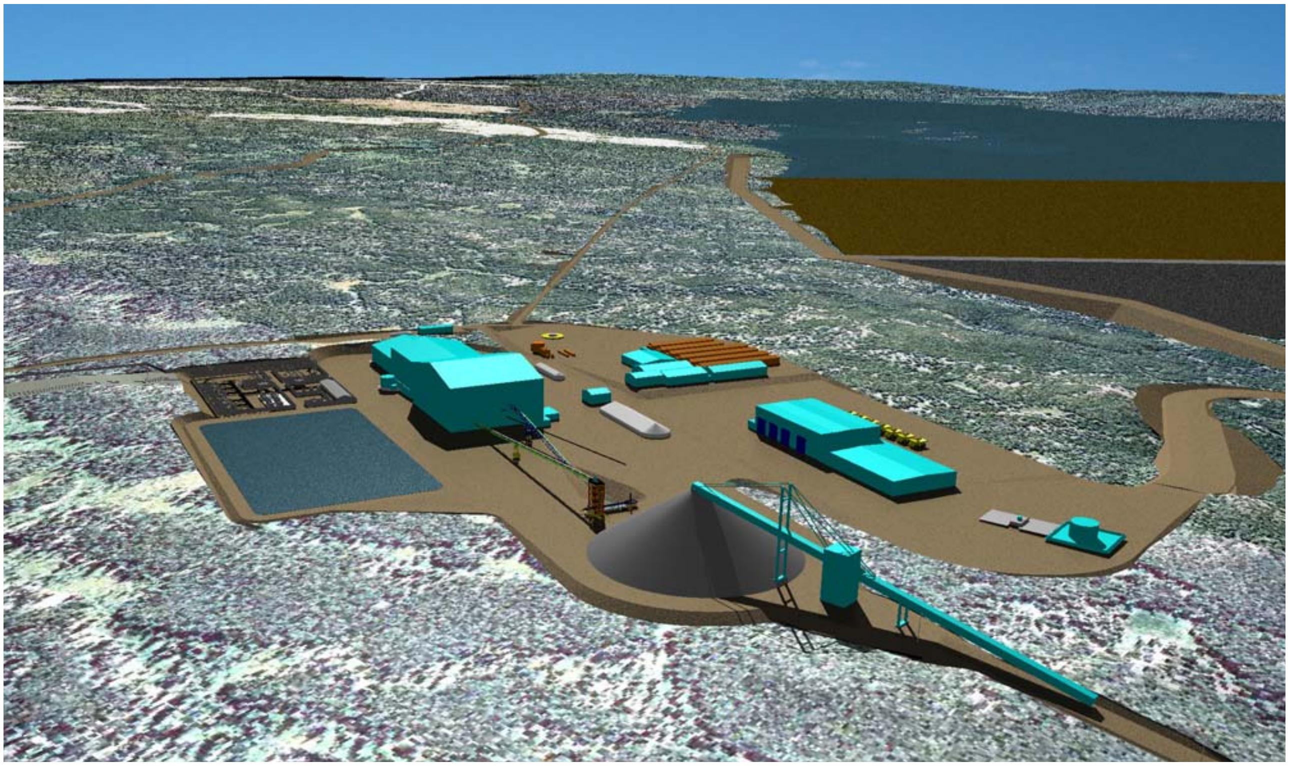
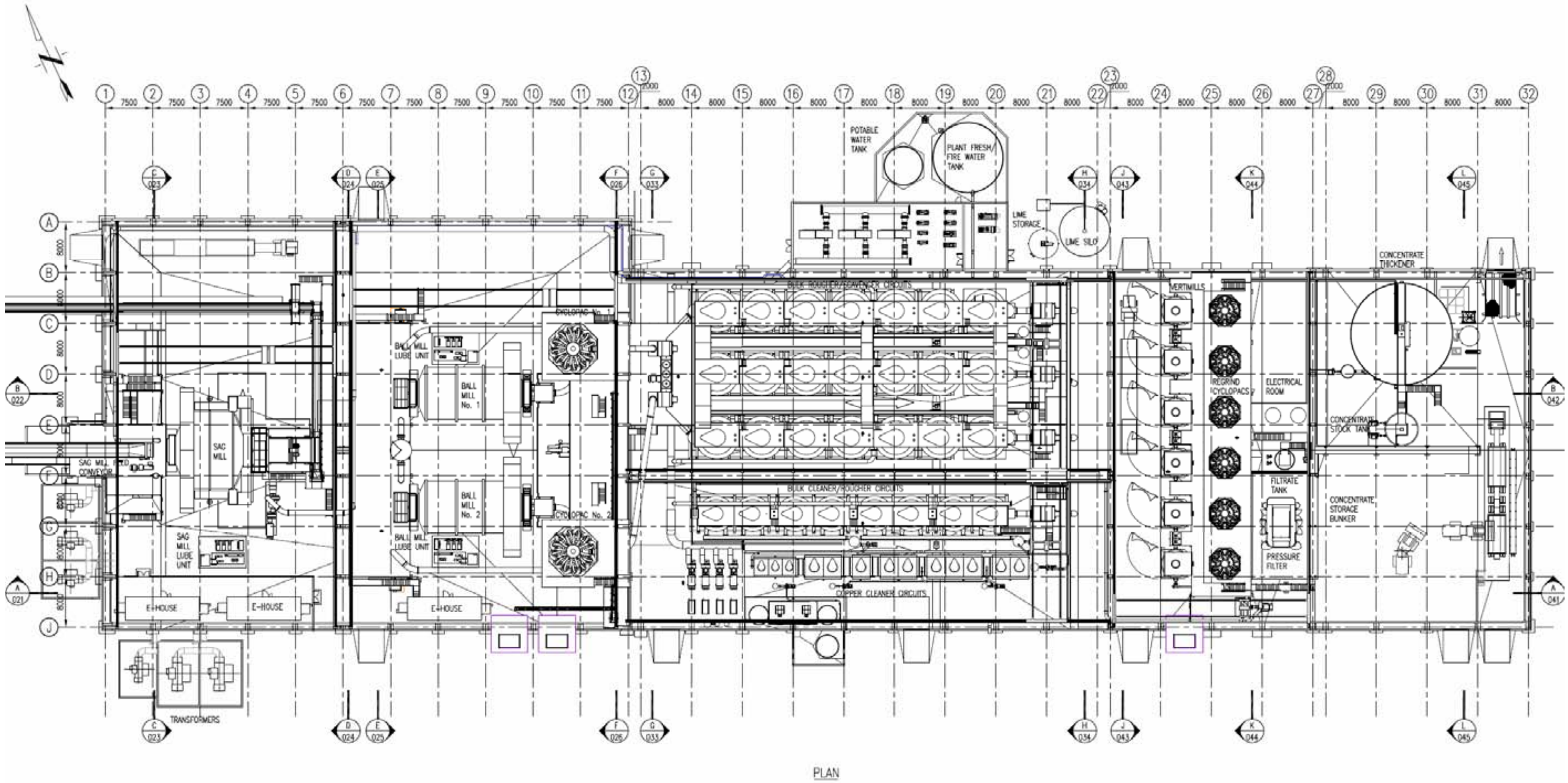


Figure 18-2
Plant Layout



A 70/10 tonne crane will service the SAG mill grinding area and an 50/10 tonne crane will service the ball mill grinding area. Additionally two 20 t cranes will be installed over the flotation/concentrate load-out areas. The large bulk flotation cells will be installed on a slab on grade, inside the mill building. The cleaner flotation cells will be located on elevated steel platforms in the flotation section. Areas with process tanks will be provided with curbs with a spill containment capacity equal to 110% of the contents of the largest tank.

The mill building will be a steel frame structure. The roof will consist of metal sheeting on steel trusses and purlins with an insulated, built-up membrane. The walls will be insulated and metal clad. An interior metal clad liner will be installed on all walls for full height to protect the insulation.

The concentrate storage and load-out areas will be contained a separate/adjoining 36 m x 30 m fabric type building (eg. Sprung), complete with a slab on grade. The concentrate load out area floors will be provided with an under-floor heating system utilizing a pumped hot glycol system to prevent ice build-up on the floor slabs. A front-end loader will load end dump concentrate trucks positioned on a truck weight scale.

The concentrate thickener and a single stock tank will be located at grade outside the load-out section.

Service Complex

The mobile equipment servicing and maintenance facilities will consist of a service complex and shop, and truck wash. The shop facilities include a small vehicle repair bay, 5 mobile equipment repair bays including lubrication provisions, 1 welding bay, and 1 tire shop bay. The major equipment bays are sized to accommodate 222 t haul trucks and other mine and ancillary mobile service equipment, with 10.5 m wide x 8.5 m high multiplex vertical lift doors. One 75/15 t crane will service two repair bays and the welding shop bay. Three15 t overhead cranes will service the other three bays.

The wash bay will be equipped with high-pressure water monitors and steam cleaning equipment. The concrete floor will be sloped towards a drain and an oil interceptor system plus waste oil tank will be included to store residual oils.

The service complex building will be a pre-engineered, steel frame structure, with metal cladding and concrete slab floors. The metal clad roof and walls will be insulated. Interior metal liner will be installed on all walls for part of their height to protect the insulation.

Ancillary Buildings and Other Structures

Materials storage includes a heated warehouse Sprung structure and a reagent storage Sprung structure. The warehouse will be 80 m x 20 m x 9 m high located adjacent to the service complex. A fenced outside storage area will provide 250 m^2 of yard storage.

A separate Sprung structure will serve as a fire, ambulance, and first aid facility.

Modular trailers will be used to provide the engineering and administration offices as well as the change-house for mine and mill operating personnel.

The assay and metallurgical laboratory will be located in a separate modular building near the service complex.

Fuel and Lubrication – Delivery, Storage and Dispensing

The diesel fuel storage facility on site will be 300,000 litres providing 3 days of fuel storage when peak consumption of approximately 100,000 liters per day occurs in the middle years of the mine life. The storage volume is considered adequate due to the proximity and the good road transportation network to Williams Lake, where commercial bulk fuel storage facilities are located.

Fire Protection

Fire water protection will be provided for the mill site area and construction/operations camp. During the project construction period, the camp will have an independent fire protection system (storage tanks and pumps) that will be incorporated into the permanent mill site system when it is operational. The primary crushers and overland conveyors will not be provided with fire water protection, because they will be remote from the mill site system and do not have a high fire risk. Fire suppression or retardant system will be provided in the primary crushing building.

Fresh, Process, and Potable Water Supply and Distribution

A suitable fresh water source will be required (non-potable) during construction and operation phases of the project.

The water sources which will be available to the project include:

- Runoff Collection Sump During Operation
- Open Pit Depressurization and Dewatering

The Process Water Pond will have a total storage capacity of 110,000 m^3 and will be supplied by three sources:

- Runoff Collection Sump Barge Pumps
- Reclaim Water from Tailings Storage Facility
- Reclaim water from the open pit

A cast-in-place concrete outlet structure in the process water pond with a manually controlled sluice gate will discharge water by gravity through a 1200 mm HDPE pipe into the concentrator building. Process pumps will boost the pipeline pressure for distribution and use in the building.

Potable water will be supplied by three proposed wells along the south perimeter of the ultimate open pit. The estimated daily potable water demand during construction will be 200 m^3 which is based on a maximum work force of 800 people. During operations, the estimated daily consumption will be 100 m^3, which is based on an average onsite work force of 400 people.

Sewage Treatment

Sewage from the mill site and camp areas will be collected by a gravity sewer system, consisting of buried PVC pipes and concrete manholes at all junctions and will be conveyed to a sewage treatment plant. For the concentrator, a sewage lift station and forcemain will be required to pump its sewage to a gravity sewer main. The lift station will be a packaged pump station with a fiberglass chamber and the forcemain will be HDPE pipe.

One sewage treatment plant (STP) will be used to service the mine during the construction phase and continue for operation. The maximum capacity of the plant will be based on a maximum workforce of 800 during construction. Sewage treatment will be by a packaged Rotating Biological Contactor (RBC) unit.

The STP will be located at the west end, low side, of the mill site, well away from the camp and other occupied areas.

Communications

Telephone and facsimile communications from the project site will be via microwave to the Telus Provincial distribution system. Distribution from the main office at the mine services area will be established across the site via buried lines. Associated equipment will be installed at the camp.

Construction Camp

The construction camp will be located south of the mill site. The construction camp will be constructed in stages in order to accommodate the build-up of workers from the early stage of construction activity to an estimated peak of 800.

On completion of the construction activities, surplus rental bunkhouse units, 1 management complex and portions of the dining and recreation buildings will be dismantled and removed. Buildings that are retained as the operations camp will be reconnected to the permanent plant site services.

Power supply to the construction camp will be from a 2 - 2200 kW diesel generator set installation. Primary usage of the power will be for heating and lighting.

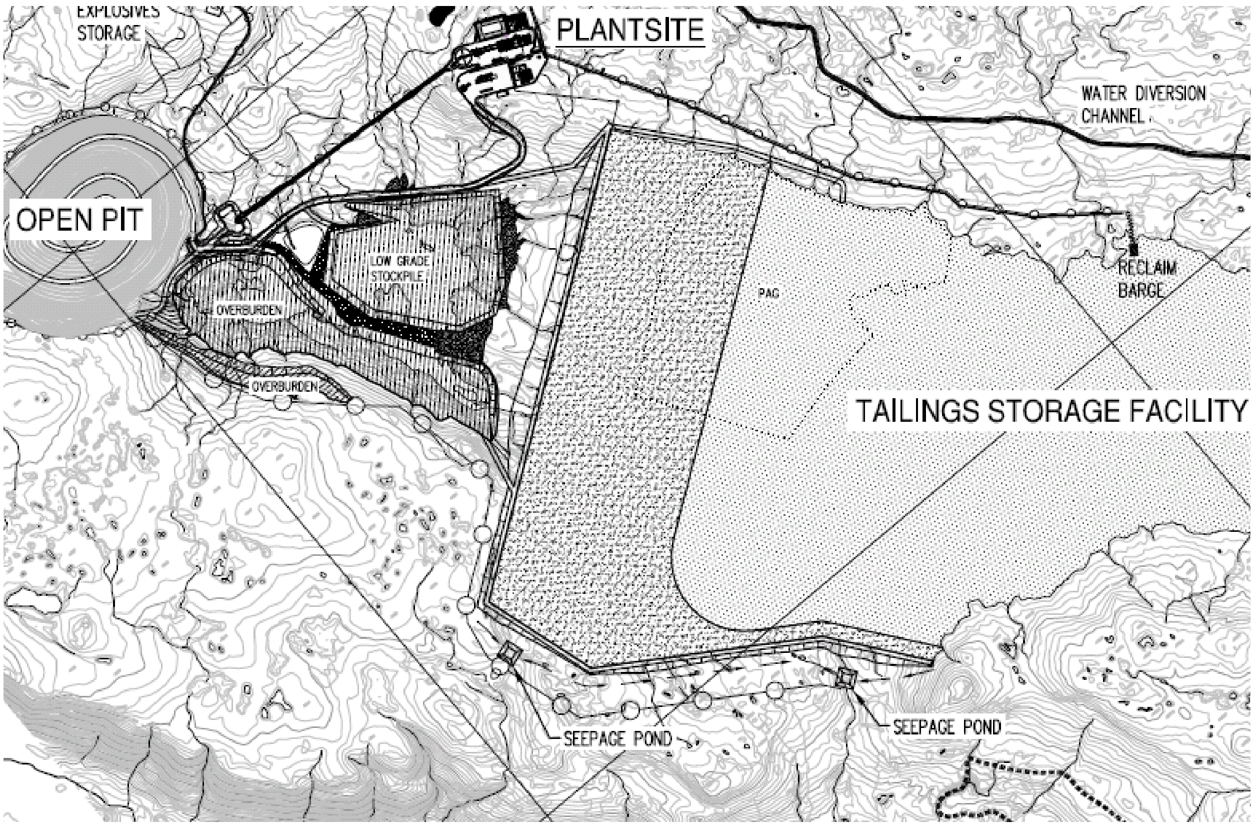
The propane system will remain in place for the life of the project.

The construction camp developed to house construction personnel will gradually be turned over to mine operations as the construction activities wind down.

Tailings Containment

Tailings from the concentrator will be collected in a tailings launder and flow by gravity to the tailings containment facility during the first few years of mine operation. A tailings pumpbox and tailings pumps will be required once the level in the containment system has reached an elevation whereby gravity flow will no longer be possible.

Figure 18-3
Tailings Containment



Tailings will be deposited in an impoundment located in the Fish Creek valley upstream from the open pit (Figure 18-3). The Tailings Storage Facility (TSF) has been designed to provide environmentally secure storage for co-disposal of approximately 480 million tonnes of tailings and 237 million tonnes of potentially reactive waste material. The remainder of mine waste generated over the life of the project will be utilized for tailing embankment construction.

The retention earthworks consist of a Main Embankment and a West Embankment. The Main Embankment will be expanded in stages across the Fish Creek Valley, and the West Embankment will be constructed along the western ridge which separates the Fish Creek drainage basin from the Big Onion Lake drainage basin.

The larger (Main Embankment) containment dam has a zoned starter water retaining dam. Once the tailings beaches have established a suitable filter, the Main Embankment will be constructed as a free draining structure that utilizes a down stream construction method with a filter and a transition zone supported by the downstream shell zone. This transition is scheduled during Year 3. The Main Embankment is shown in Figure 18-4.

By contrast, the West Embankment dam is planned to be constructed as an initial zoned starter dam in Year 3, followed by a centreline construction with a glacial till, low permeability core. Non acid generating waste rock and overburden will be used in the dam shells as shown in Figure 18-5.

The larger (Main Embankment) containment dam has a zoned starter dam. After Year 2 mine operations, the majority of the downstream section of the dam consists of clean mine waste rock forming a buttress for the use of compacted cycloned tailings in the downstream section as shown in Figure 18-4.

The discharge of tailings from the delivery pipelines into the TSF will be from a series of large diameter valved offtakes located along the Main and West Embankments. Tailings discharge will begin along the Main Embankment, and will be extended along the West Embankment starting in Year 4. The coarse fraction of the tailings are expected to settle rapidly and will accumulate closer to the discharge points, forming a gentle beach with a slope of about 1 percent.

The material requirements for dam construction are shown in Table 18-2.

Table 18-2
Dam Construction Material Requirements

STAGED EMBANKMENT FILL VOLUMES

MAIN EMBANKMENT

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Stage	Year	Min. Crest Elevation	Maximum Height	Zone B	Zone C (NR)	Zone S	Zone F	Zone G	Zone T	Wearing Coarse	Zone RF Downstream	Total Volume
		(m)	(m)	(m^3)	(m^3)	(m^3)	(m^3)	(m^3)	(m^3)	(m^3)	(m^3)	(m^3)
I	-1	1,492	34	27,817	836,750	687,186	25,646	0	0	5,000		1,582,399
II	1	1,500	42	156,014	1,682,465	606,102	260,090	0	0	12,000		2,716,671
III	2	1,508	50	67,400	348,500	365,900	37,500	0	0	22,000	1,006,700	1,848,000
IV	3	1,518	60	0	104,340	0	0	104,340	432,900		2,898,750	3,540,330
V	5	1,526	68	0	101,800	0	0	101,800	407,200		3,023,180	3,633,980
VI	7	1,534	76	0	120,760	0	0	120,760	483,040		3,644,379	4,368,939
VII	9	1,542	84	0	130,800	0	0	130,800	523,200		4,253,432	5,038,232
VIII	11	1,548	90	0	99,000	0	0	99,000	396,000		3,616,581	4,210,581
IX	15	1,554	96	0	100,140	0	0	100,140	400,560		3,995,440	4,596,280
TOTALS				251,231	3,524,555	1,659,188	323,236	656,840	2,642,900		22,438,462	31,535,412

WEST EMBANKMENT

Stage	Year	Min. Crest Elevation	Maximum Height	Zone B	Zone C (NR)	Zone S	Rock Drain	Zone G	Zone T	Wearing Coarse	Total Volume
		(m)	(m)	(m^3)	(m^3)	(m^3)	(m^3)	(m^3)	(m^3)	(m^3)	(m^3)
IV	3	1,518	13	14,540	158,449	85,401	31,900				290,290
V	5	1,526	21	38,325	261,705	175,200	16,425				491,655
VI	7	1,534	29	83,300	1,333,320	380,800	155,310				1,952,730
VII	9	1,542	37	85,050	2,517,845	388,800	269,850				3,261,545
VIII	11	1,548	43	77,500	1,054,000	372,000	139,500				1,643,000
IX	15	1,554	49	75,900	1,821,600	379,500	142,313				2,419,313
TOTALS				374,615	7,146,919	1,781,701	755,298				10,058,533

Notes:

1) Volumes are based on compacted embankment fill bulk densities of 2.1 t/m^3 for overburden and 2.4 t/m^3 for waste rock.
2) Ground level considered 1458 m at the Main Embankment and 1505 m at the West Embankment.

Figure 18-4
Main Embankment

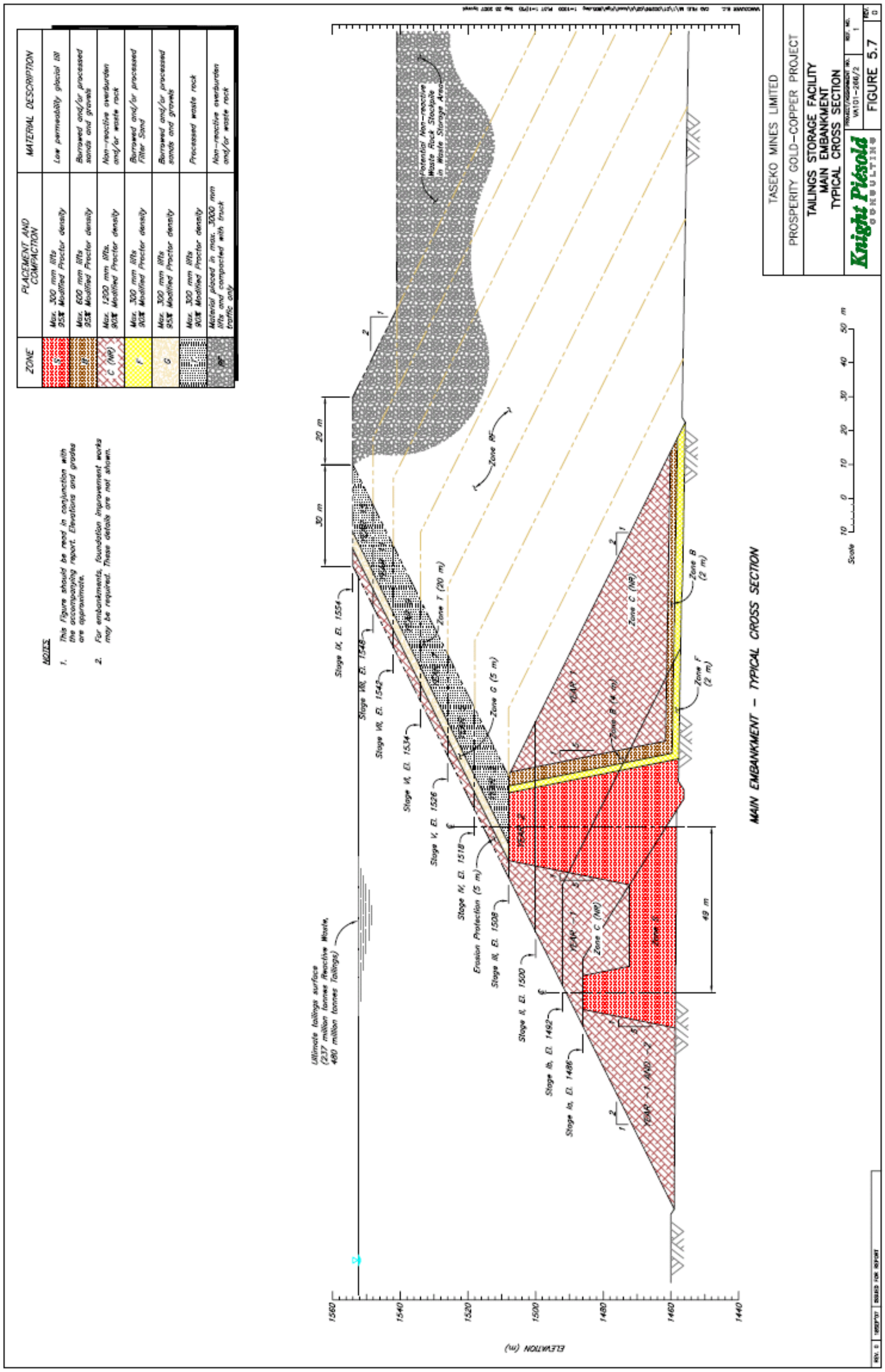
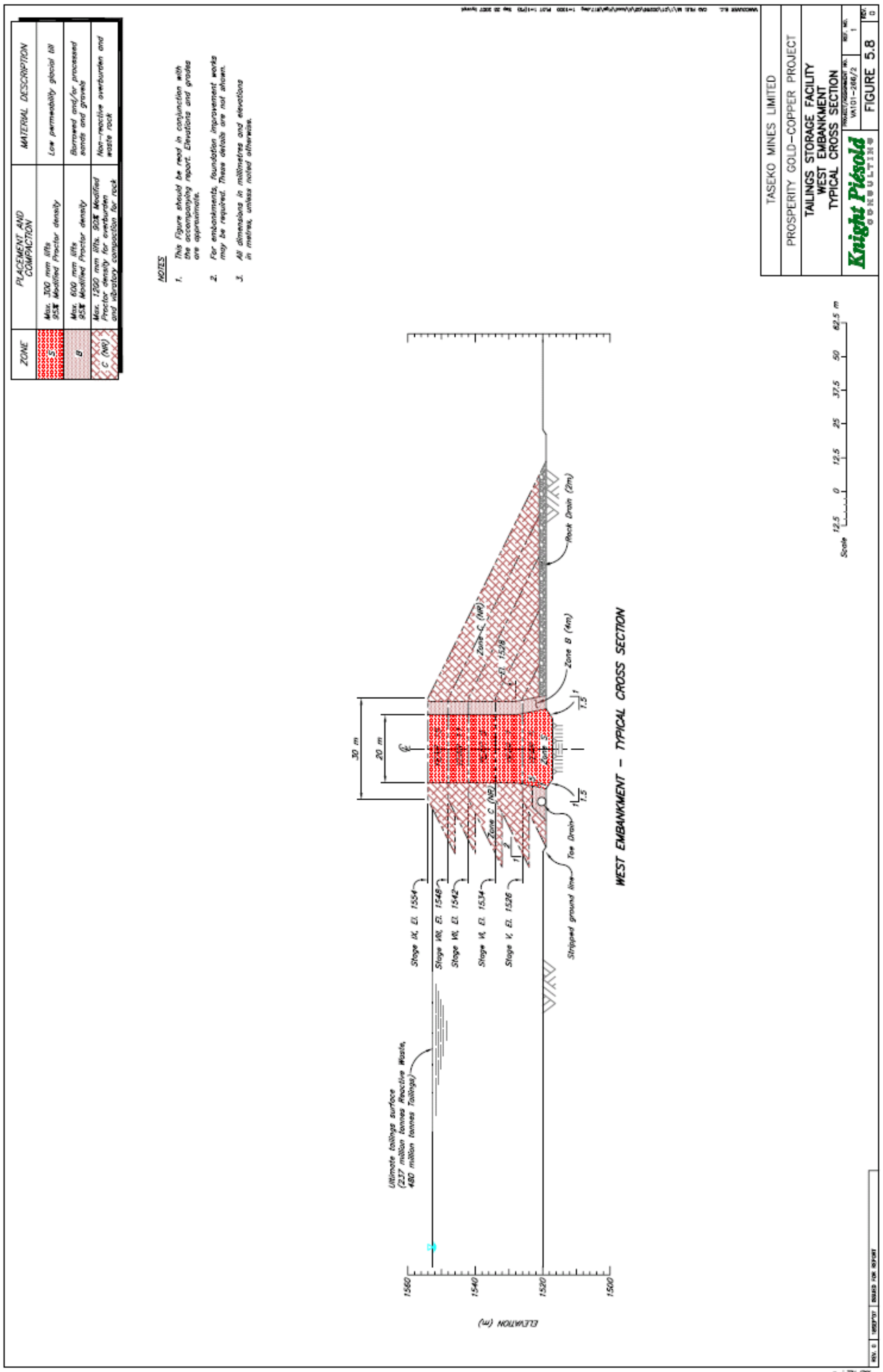


Figure 18-5
West Embankment



Seepage losses from the Main Embankment will be collected at the Water Collection Pond, which is located between the Waste Storage Area and the Open Pit. Special design provisions to minimize seepage losses include the development of extensive tailings beaches (which isolate the supernatant pond from the embankments), toe drains to reduce seepage gradients and contingency measures for groundwater recovery and recycle.

A Headwater Channel will be constructed in the pre-production years to divert runoff from the undisturbed eastern portion of the Fish Creek catchment area.

Construction of the Stage Ia Main Embankment will start approximately 18 months before mill start-up. Sufficient water will be impounded prior to start-up, and will be available for mill commissioning and early operations. Mill process water for ongoing operations will be reclaimed from the TSF and supplemented with water from the Water Collection Pond.

Details of the site characteristics, geotechnical, hydrogeological and water balance considerations for the tailings facility design, pipeworks, seepage collection and reclamation and closure are contained in the Knight Piésold "Report on Feasibility Design of the 70,000 Tonnes per Day Tailings Storage Facility", dated September 2007.

18.2 Open Pit Design

This section of the report describes the basis for the open pit design including the design parameter basis, design summary, geotechnical considerations, dewatering, and waste material types and storage method.

The open pit design has been based upon the following key considerations:

- Geotechnical recommendations and design criteria, for maximum pit slope and waste dump locations, provided by Knight Piesold Consulting (KP).
- Operating constraints of the equipment selected for mining.
- Minimum mining width defined by shovel double side loading of trucks with allowance for access ramp.
- Bench height achievable and within the safe operating reach of the primary loading unit.
- Minimum haulage road operating width and maximum effective grade within the operating limitations of the primary haulage units.
- Logical and efficient scheduling of material movement from multiple phases of pit expansion to the crusher, the stockpiles and to final waste material placement sites.

Geotechnical investigations and testing were undertaken by KP. The complete test results, findings and recommendations for the pit wall slopes, waste dumps and results of hydrological investigations are contained in the KP report, "2007 Feasibility Pit Slope Design", dated July 2007. Knight Piesold's work consisted of site reconnaissance and mapping, oriented core diamond drilling and detailed logging of fracture data, in-situ permeability testing, point load testing, uniaxial compressive and tri-axial strength tests and direct shear tests on rock joints.

Geotechnical core logging data were used to develop a rock mass classification system and rock mass model for the deposit. Mapping data were used to determine structural discontinuities and to assess the potential for wedge and plane failures in the pit walls. These assessments were the basis for stability analyses of failure modes along structural discontinuities and for evaluation of deep-seated failure.

The mine design drawings and ore reserve reporting was completed by the engineering staff of Taseko Mines Limited and Nilsson Mine Services (NMS). Mining phases were smoothed, and haulage roads, stockpiles and waste dumps were located.

Geotechnical Considerations

The Prosperity open pit will be a nominal 525 m deep when complete. Open pit wall slope stability is dependent upon the following site specific factors:

- Geological structure
- Rock alteration
- Intact rock strength
- Rock stress
- Groundwater conditions
- Discontinuity strength and orientation
- Pit geometry
- Blasting practices
- Climatic conditions
- Time

In general the rock mass quality at Prosperity ranges from fair to good. There are two major faults within the pit limit. These are referred to as the QD and East Faults. These structures are near vertical, sub-parallel and trend North-South through the center of the deposit. There do not appear to be any major structures that will adversely influence the stability of the pit slopes.

The Prosperity Deposit is centered about a diorite intrusive where potassic alteration is associated with the core of the mineralized zone. This central zone of mineralization is surrounded by a propylitic alteration zone. A retro-grade phyllic alteration is overprinted on the propylitic and potassic zones. Within the potassic zone there is a well defined vertical zonation defined by dissolution of gypsum on joint surfaces. The “gypsum” line defines the change from generally competent rock to competent rock and is used to separate structural domains for the purposes of mine design.

Intact rock strength is an important consideration, as many potential failure surfaces are not completely developed and require some failure of intact rock. The moderate to high strength of the rock at Prosperity site is beneficial due to the high stresses that are expected to develop in the pit slopes during later stages of mining. The uniaxial compressive strength, based on point load tests, varies but averages 112 megaPascals (mPa).

The rock stress conditions within the rock mass are a significant factor for high slopes. Knight Piesold has used a sophisticated finite difference computer model (FLAC) to assess the potential overstressing of the rock in the proposed pit slopes.

The predominant jointing patterns are sub-vertical and coincident with the main vein systems. Secondary veins have also been identified dipping out of the East pit slopes. KP has investigated the potential for adversely oriented structural features at depth at or near the final pit walls. The finding of this investigation was that there is a very low likelihood of adverse structures in the form of open joints. Structural features in close proximity to final walls will be primarily quartz and sulphide veins.

Based upon three structural domains the open pit has been divided by KP vertically into three major slope design sectors that correspond with:

- Sector I - Surface materials including overburden and basalt
- Sector II - Upper Zone located above the "gypsum line"
- Sector III - Lower Zone located below the "gypsum line"

These major sectors have been further subdivided in detail as shown in Figure 18-6. However, the actual design recommendations for each major sector are for the most part identical and are summarized in Table 18-3. The overburden will be mined leaving a 30° inter-ramp slope. The basalt formation on surface will be mined leaving a 45° inter-ramp slope. The Middle Zone will primarily be mined leaving a 45° inter-ramp slope and the Lower Zone inter-ramp slope will be increased to 50°.

Figure 18-6
Geotechnical Pit Slope Design Sectors Plan

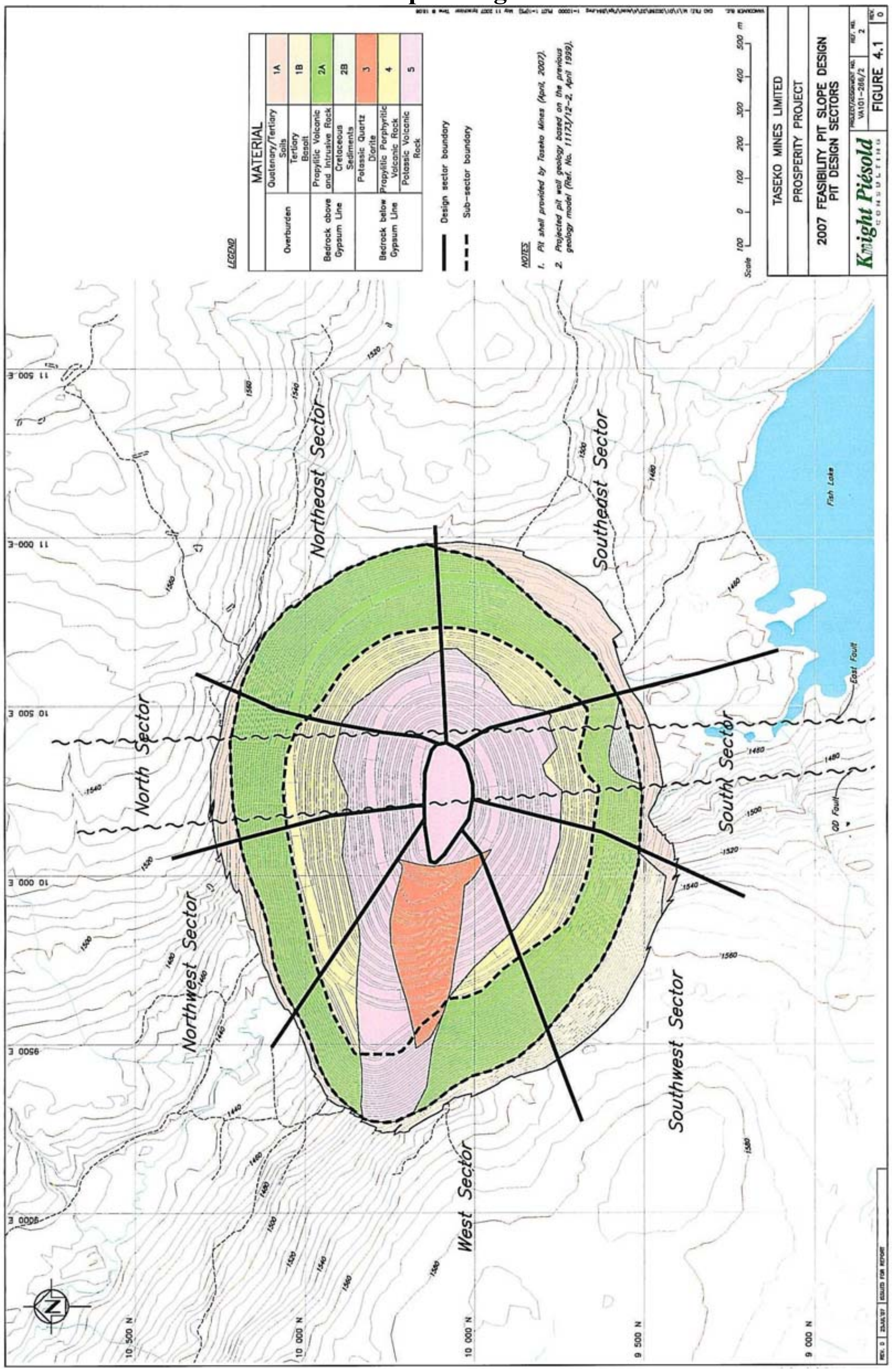


Table 18-3
Recommended Wall Slopes

Design for Near Surface Materials

Design Sector	Geologic Domain	Inter-Ramp Slope degrees	Bench Height m	Berm Interval m	Berm Width m	Interberm Slope degrees
Ia	Overburden	30.0	15.0	15.0	8.0	40.0
Ib	Overburden-Basalt	45.0	15.0	30.0	8.0	65.0

Design Above the Gypsum Line

Design Sector	Geologic Domain	Inter-Ramp Slope degrees	Bench Height m	Berm Interval m	Berm Width m	Interberm Slope degrees
IIa	Middle West Sector	45.0	15.0	15.0	8.0	65.0
IIb	Middle West Sector - Potassic	45.0	15.0	15.0	8.0	65.0
IIc	Middle Northwest Sector	45.0	15.0	15.0	8.0	65.0
IId	Middle North Sector	30.0	15.0	15.0	8.0	65.0
IIe	MiddleNortheast Sector	30.0	15.0	15.0	8.0	65.0
IIf	Middle East Sector	45.0	15.0	15.0	8.0	65.0
IIg	Middle Southeast Sector	45.0	15.0	15.0	8.0	65.0
IIh	MiddleSouth Sector	45.0	15.0	15.0	8.0	65.0
IIi	Middle Southwest Sector	45.0	15.0	15.0	8.0	65.0

Design Below the Gypsum Line

Design Sector	Geologic Domain	Inter-Ramp Slope degrees	Bench Height m	Berm Interval m	Berm Width m	Interberm Slope degrees
IIIa	Lower West Sector	45.0	15.0	15.0	8.0	65.0
IIIb	Lower Northwest Sector	50.0	15.0	30.0	11.0	65.0
IIIc	Lower North Sector	50.0	15.0	30.0	11.0	65.0
IIId	Lower Northeast Sector	45.0	15.0	15.0	8.0	65.0
IIIe	Lower Southwest Sector	50.0	15.0	30.0	11.0	65.0
IIIf	Lower South Sector	50.0	15.0	30.0	11.0	65.0
IIIg	Lower Southwest Sector	50.0	15.0	30.0	11.0	65.0

Slope stability analyses using a RMR rock mass classification basis for limit equilibrium and numerical modeling have been undertaken as part of the process of defining the design slope recommendations. These analyses have indicated a potential for instability in the zone of potassic alteration located in the central west slope. The analyses also indicated that stability was sensitive to groundwater depressurization and draw down. As a result increased drain hole density and hole lengths will be required in this area.

The climatic conditions at the Prosperity Project are typical of the British Columbia Chilcotin District with an annual average of 524 mm of rain equivalent precipitation. The seasons in this area are well defined with relatively predictable periods of “freeze up” in the fall and “break up” in the spring. The “break up” period is characterized by increased water flow from melting snow and cyclical thawing and freezing of the surface materials on pit slopes. This action results in decreased slope stability particularly at the smaller bench scale where there will be a marked increase in small face failures and raveling of rock.

The ultimate pit geometry is roughly oval and the internal pit phases expand in all directions about the Phase 1 –Starter Pit. As such during the life of the mine all internal walls are temporary and will be mined. Final walls will occur only in the Phase 4 Pit that is active for a period of 10 years between Year 6 and Year 16 of the production schedule. Phase 1 and Phase 2 pit walls will typically be exposed for 2 years and the Phase 3 walls will be exposed for 4 years prior to excavation.

Drilling and blasting near both temporary and final walls will require buffer blasting. Knight Piesold have recommended overall wall slopes of 30° in overburden, 45° above the “gypsum line” and 50° below the “gypsum line”. The KP recommendations for bench and berm configuration were based upon single benching and achieving steep inter-berm face angles up to 75°. The designs incorporated in this study assume that double benching will be possible and that shallower inter-berm angles to 64° will be allowed resulting in berm widths from 10 to 15 m width. This assumption simplifies the wall control blasting requirement and the necessity for multiple hole sizes and drill rig configurations on wall control blasts.

Buffer rows will be drilled using production blasthole drills at a reduced spacing and with adjustment to hole depth near design berm crests. Line holes will not be incorporated in the blast patterns. Walls will be scaled carefully on each bench.

Pit Dewatering

Mine dewatering has been addressed by Knight Piesold and is summarized in this section of the report in terms of how it relates to the mine operations. The water management recommendations are the basis for capital and operating cost estimates in later sections of the report.

It has been recognized that the open pit development will have significant impact on the local hydrogeological regime, as the pit will become a groundwater discharge area. The water table is currently at or near the ground surface and provisions have been made for an extensive slope depressurization system. Groundwater dewatering wells and slope depressurization will be concentrated in Sector IIIc and Sector IIIf shown in Figure 18-6.

Pit inflows will likely be dominated by localized confined aquifers in the southern area of the pit from zones of higher rock mass permeability related to major structures and from unconfined flow in the upper 150 to 300 m of fractured rock mass above the gypsum line. Inflows from good quality, low permeability rock below and peripheral to the gypsum line are expected to be low.

Depressurization systems are important in overall pit slope design. A combination of techniques including vertical wells, in-pit horizontal drains and collection systems will be implemented as a staged approach during pit development.

The open pit dewatering system has been designed to meet the combined requirements of the expected groundwater pit inflow rates and runoff from precipitation. The annual contribution of direct precipitation to the in-pit pumping requirements has been estimated for the average annual precipitation volume with a ten year return period, and the storm flow rate required to remove ponded water from the one in ten year, 24 hour storm event within 96 hours. The peak operational design capacity of the system is 400 litres/second.

General Design

The mining equipment will operate on a 15m high bench in overburden and hardrock. Wall slope design changes will be implemented by varying the berm widths and inter-berm slope angles.

Berms will be left on every bench in overburden and on alternate benches in hardrock. Berm width design will vary from 15 m to 10 m as the overall wall slope is increased from 45° to 50° in the Lower Zone. The general mine design parameters are summarized in Table 18-4.

The open pit will be mined in four phases commencing with the Phase 1 – Starter Pit. The pit will be partially pre-stripped during the preproduction development period. The starter pit will provide building materials for the tailings impoundment starter dam. The Phase 2 through Phase 4 pits are radial expansions of the mine about the Starter Pit creating a progressively deeper pit.

Table 18-4 Design Parameters

Open Pit Design

Bench and Berm Design		
Bench Height	metres	15.0
Bench Interval Overburden	metres	15.0
Bench Interval Gypsum Zone	metres	15.0
Bench Interval Hardrock	metres	30.0
Interbench Face Angle Design Sector I & II	degrees	65°
Interbench Face Angle Design Sector III	degrees	65°
Haulroad Design		
Total Road Allowance	metres	30.0
Maximum Haulroad Grade	percent	10.0%
Minimum Pushback Width		
Minimum Pushback	metres	80.0

Waste Dumps & Stockpiles

Material Properties		
Overburden Bulk Density Placed	t/pcm	1.83
Waste Rock Bulk Density Placed	t/pcm	2.04
Seismic Criteria	year	1 in 1000
Maximum Design Earthquake	g	0.1
Acceleration		
Pile Stability Criteria		
Minimum Factor of Safety During Operations		1.2
Minimum Factor of Safety for Closure		1.5
Minimum Factor of Safety for Seismic Loading		1.0
Final Slopes		
Overburden Bench Height		30.0
Overburden Berm Width		20.0
Overburden Face Slope		1.3:1
Waste Rock Bench Height		30.0
Waste Rock Berm Width		20.0
Waste Rock Face Slope		1.2:1

t/pcm = tonnes per bank cubic metre

The minimum pushback width is 80 m; however in general the expansions are in excess of 100 m width. Haul road allowances have provided at 35 m. Roads are designed at a maximum of 10% grade and are located to spiral counterclockwise into the pit bottom.

The ultimate pit features are summarized as follows:

- 1,650 m E-W by 1285 m N-S
- Total surface area 166 hectares
- Final ramp exit elevation 1500 m
- Ultimate pit bottom elevation 945 m
- Maximum wall height – 600 m in the SW quadrant with maximum elevation 1545 m
- Final overall wall slope angles in the following directions:

North Wall	45.5°
East Wall	45.2°
South Wall	43.6°
West Wall	42.4°

Waste Storage, Stockpiles, and Roads

The waste or non-ore material types included in the reserves and material movement schedules are subdivided into overburden, waste rock and stockpiled lower grade ore. These materials are further subdivided into reactive and non-reactive proportions of each respective material type if appropriate. Classification of waste materials is based on spatial remodeling of the ABA data in 2006 by Taseko, Knight Piesold and SRK. Sub-aqueous storage is proposed for reactive overburden and waste rock and the balance of the overburden and waste can be used for construction purposes or placed on surface storage sites where surface drainage is controlled and treatable as required.

A total of 886 million tonnes of material will be mined from the open pit, including 399.7 million tonnes of ore directly to the crusher, 87.1 million tonnes of stockpiled lower grade ore, 11.7 million tonnes of reactive overburden, 60 million tonnes of non-reactive overburden, 225.5 million tonnes of reactive waste and 101.9 million tonnes of non-reactive waste.

The area underlying the overburden and waste dump site is characterized by up to 20 m of glacial till, which overlies Quaternary Glaciofluvial and Glaciolacustrine units. These in turn overlie Miocene Basalt flows and a Miocene Glaciofluvial Unit followed by glacial till and colluvium. These units extend south and overlie the open pit area as well.

Overburden has been classified as reactive and non-reactive in nature. The reactive overburden contains weathered rock which includes oxidized or partially weathered sulphide minerals. This material will be placed in the tailings management facility. Non-reactive overburden will be placed in the overburden stockpile located to the south of the open pit.

Overburden piles will be developed in 30m high benches, each offset from the downstream edge by 20 m to provide an overall slope of approximately 2H:1V (2 m height to 1 m vertical). Each bench will be constructed from lifts of approximately 15 m by end dumping the material to form an angle of repose slope angle of approximately 1.3H:1V. The final overburden pile will be approximately 60 m high with a final elevation of 1515 m.

The total reactive waste identified as having potential for acid generation (PAG), is 237 million tonnes. This quantity of material will be hauled from the mine and placed in the tailings management facility. It will be dumped in lifts and dozed out into the area of active tailings deposition.

The total non-reactive waste is 101.9 million tonnes. Over the life of the mine 28.2 million tonnes will be required for construction in specific zones of the tailings dams. A downstream dam slope, of 2:1 will require up to 55.7 million tonnes of waste. The filling of Fish Lake to the 1470 elevation will require approximately 16.8 million tonnes. This will be progressively be filled in with layers of durable free draining waste rock. The balance available for road construction is approximately 1.2 million tonnes.

At closure the waste pile will be covered with a 0.7 m thick layer of topsoil and revegetated.

The total lower grade ore quantity stockpiled in the variable cutoff grade production schedule is a total of 87.1 million tonnes. This material has been reported at a $5.25/tonne NSR pit rim cutoff.

As described above the dumps and stockpiles will be constructed in lifts with berms left at 30 m intervals. Overall final slopes will be 2H:1V and crests will be contoured for reclamation. Prior to placement of overburden and waste in the stockpile areas the vegetation will be cleared, and diversion & runoff collection ditches will be constructed.

Stability analyses have been carried out for waste rock and overburden piles. The analyses were performed for static and seismic conditions. A limit equilibrium method of calculation was used. Minimum factors of safety for the static state were 1.5 for both overburden and waste rock and 1.2 for seismic conditions.

Haul roads will be required from the mine to the crusher, stockpiles, overburden spoil piles, waste dumps and the tailings management facility for construction and waste disposal. These roads will be constructed with materials derived from mine operations. The mine haul roads will be designed on a 35 m road allowance with a running surface of 3 times the total operating width of the haulage truck. The haulage roads will not exceed a design grade of 10%. They will be built with an operating surface of 30 m and additional allowance for ditches and berms where required.

18.3 Mining Operations

Production Schedule

The geometry of the ore deposit in relation to the relatively gentle topographic surface allows for flexibility in pit sequencing. Ore is easily accessible near surface in the Phase 1 – Starter Pit. Once the relatively thin layer of overburden is removed the ore is released in the Phase 1 Pit. Subsequent pit expansions are generally symmetrical about the starter pit and as such become progressively larger and deeper. The sequencing therefore becomes an exercise of balancing total production over the life of the mine to defer major stripping and associated capital equipment while effectively utilizing equipment on site.

The proposed mine development sequence is shown in Figures 18-7 through 18-11.

Figure 18-7
Prosperity Mine Development – End of Year 1

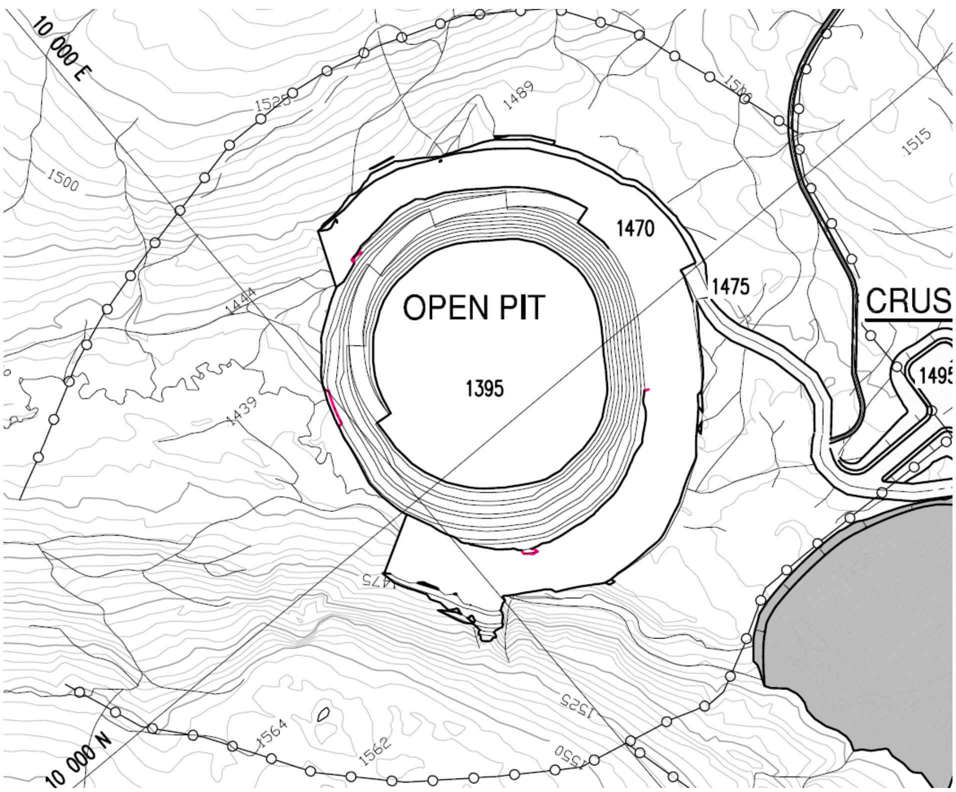


Figure 18-8
Prosperity Mine Development – End of Year 2

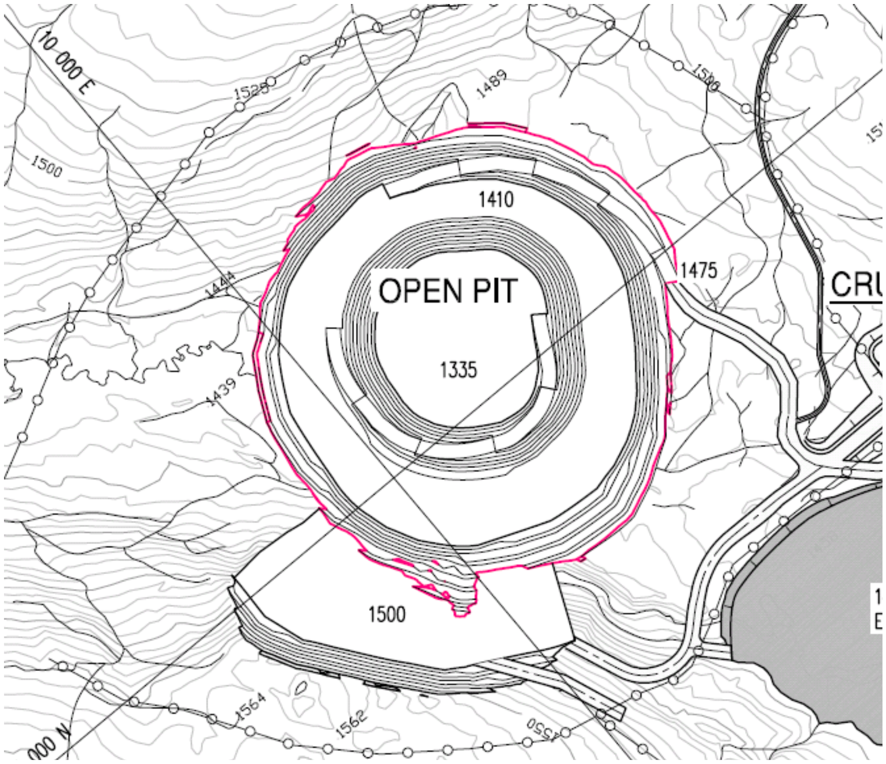


Figure 18-9
Prosperity Mine Development – End of Year 5

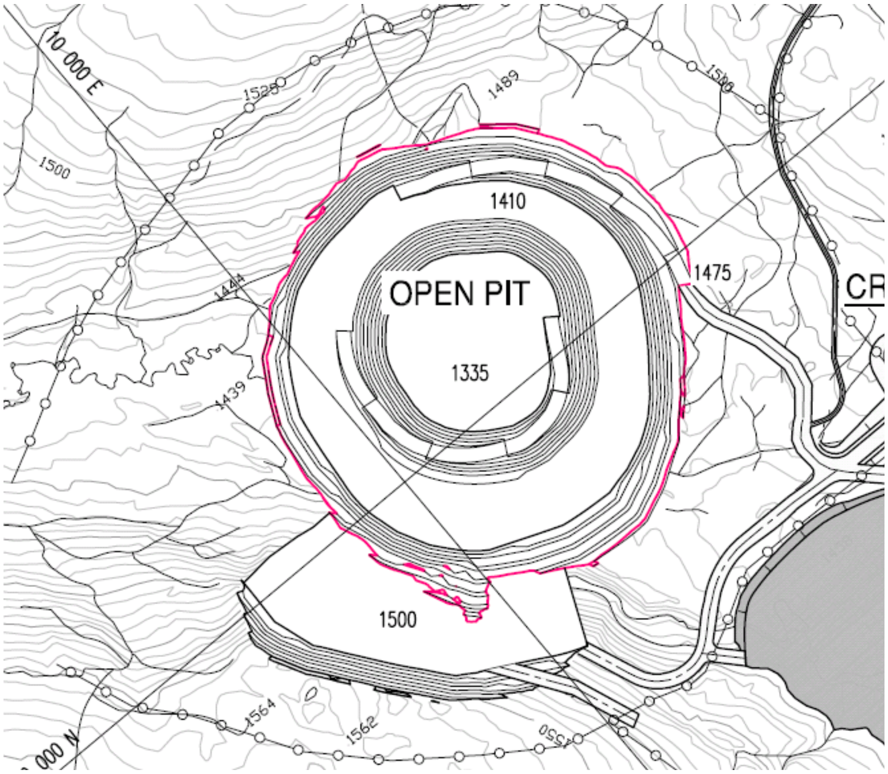


Figure 18-10
Prosperity Mine Development – End of Year 8

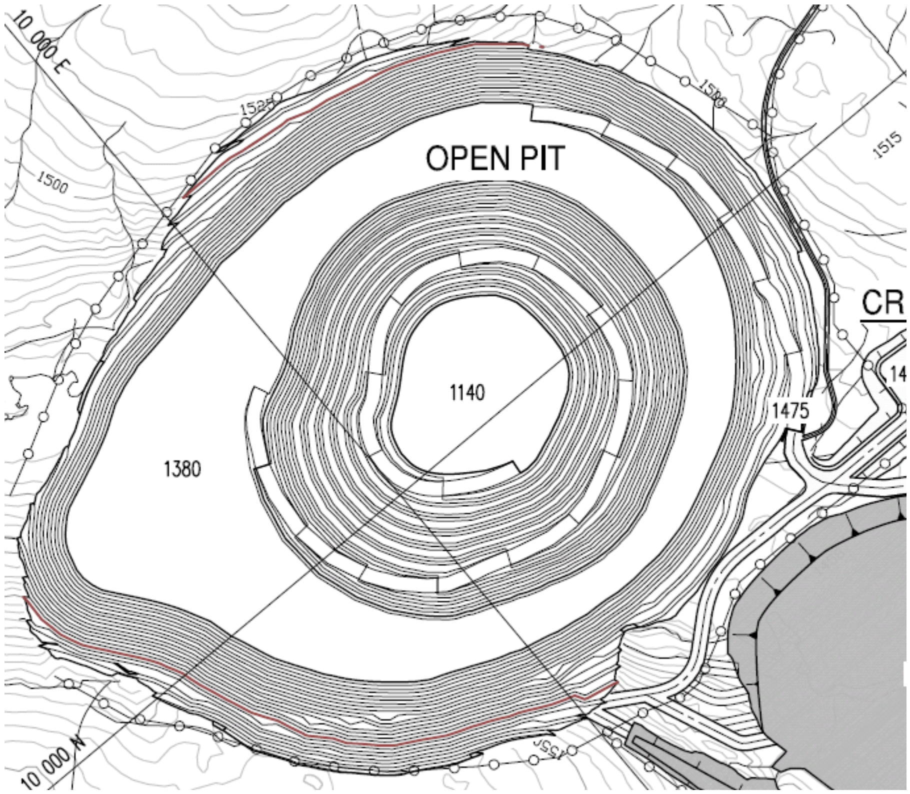
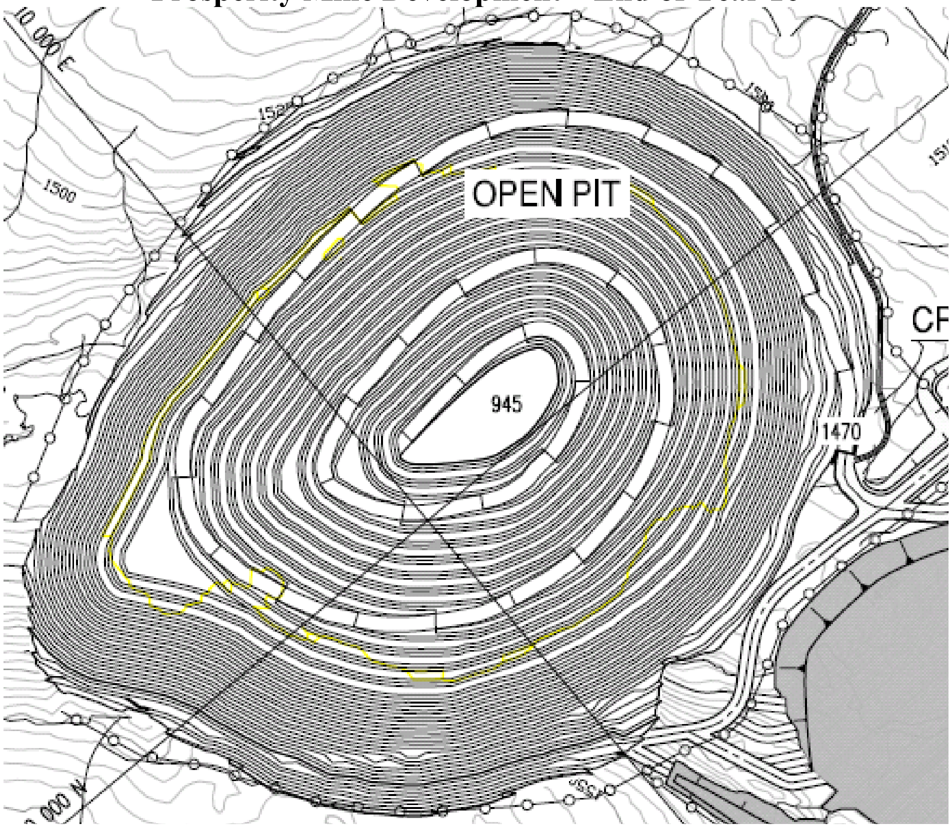


Figure 18-11
Prosperity Mine Development – End of Year 16



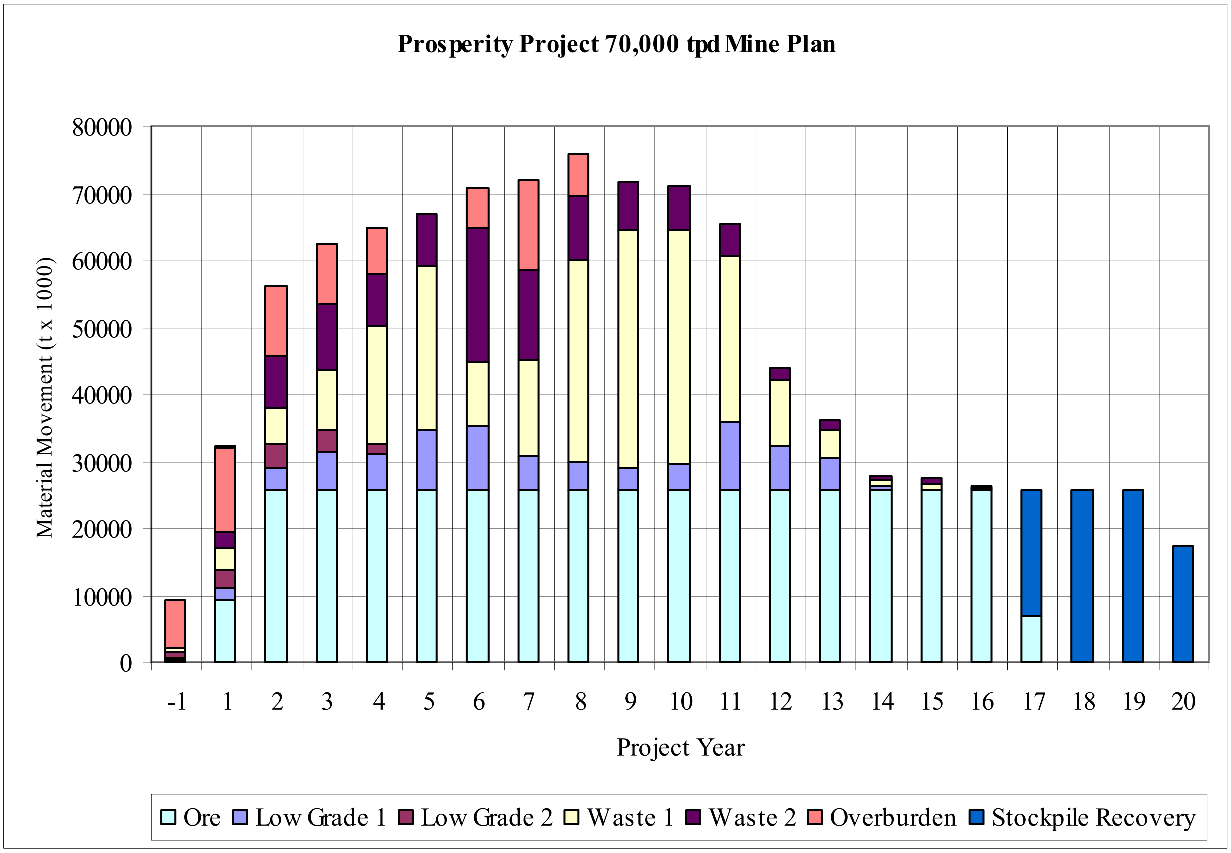
The mine production forecast has been derived by scheduling ore at a variable declining NSR cutoff. A number of preliminary iterative analyses were conducted to determine the variable cutoff NSR strategy to use for the purposes of scheduling. The objectives of the analyses were to maximize net present value of the project while smoothing the equipment requirement to achieve ore release on a consistent basis.

Ore has been scheduled to provide 25.5 million tonnes of ore to the primary crusher annually. The mine will operate 365 days per year with a nominal crusher throughput of 70,000 tonnes/day. The production schedule is shown in Table 18-5 Mine Production Forecast. This data is represented graphically in Figure 18-14 Material Movement Schedule.

Table 18-5 Mine Production Forecast

	Year		Pre-Prod'n	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	Total
Ore Production Source																								From -1
Phase 1			XXXXXX	XXXXXX	XXXXXX	XXXXXX	XXXXXX																	52,770.1
Phase 2				XXXXXX	XXXXXX	XXXXXX	XXXXXX	XXXXXX	XXXXXX	XXXXXX														60,587.3
Phase 3					XXXXXX	XXXXXX	XXXXXX	XXXXXX	XXXXXX	XXXXXX	XXXXXX	XXXXXX	XXXXXX	XXXXXX	XXXXXX									87,871.6
Phase 4						XXXXXX			XXXXXX	XXXXXX	XXXXXX	XXXXXX	XXXXXX	XXXXXX	XXXXXX	XXXXXX	XXXXXX	XXXXXX	XXXXXX	XXXXXX				198,484.3
Stockpile Activity				XXXXXX																XXXXXX	XXXXXX	XXXXXX	XXXXXX	87,431.8
Total Ore to the Mill																								487,145.1
Days			365	365	365	365	365	365	365	365	365	365	365	365	365	365	365	365	365	365	365	365	246	7,546
Production Forecast																								
Cutoff Grade ($nsr)			$9.50	$9.50	$9.50	$8.25	$8.25	$7.50	$7.50	$7.50	$7.50	$6.00	$6.00	$6.75	$7.50	$6.75	$5.25	$5.25	$5.25	$5.25	$5.25	$5.25	$5.25	
Open Pit Production																								
Ore	(t x 1000)		356.1	9,134.9	25,560.3	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	6,822.1	-	-	-	399,713.3
NSR	$/t		11.599	12.072	13.034	12.825	13.620	12.332	12.140	13.686	11.852	10.769	10.036	10.854	12.528	11.228	12.649	14.540	16.039	17.121	-	-	-	12.609
Gold	g/t		0.424	0.458	0.489	0.489	0.508	0.467	0.456	0.504	0.444	0.412	0.399	0.421	0.448	0.393	0.447	0.506	0.527	0.539	-	-	-	0.462
Copper	%		0.235	0.232	0.246	0.240	0.249	0.224	0.218	0.244	0.221	0.209	0.190	0.190	0.232	0.219	0.237	0.268	0.309	0.340	-	-	-	0.235
Low Grade $5.25 Cutoff	(t x 1000)		382.9	1,961.2	3,415.6	5,751.1	5,526.1	8,965.1	9,641.3	5,161.3	4,347.7	3,371.2	4,037.1	10,267.0	6,769.9	5,028.5	696.6	-	-	-	-	-	-	75,322.7
NSR	$/t		6.359	6.405	6.608	6.470	6.365	6.397	6.367	6.492	6.472	5.783	5.630	6.037	6.429	6.364	5.960	-	-	-	-	-	-	6.296
Gold	g/t		0.227	0.268	0.290	0.281	0.281	0.284	0.267	0.278	0.276	0.254	0.241	0.248	0.259	0.253	0.249	-	-	-	-	-	-	0.267
Copper	%		0.174	0.153	0.147	0.149	0.146	0.148	0.141	0.140	0.139	0.141	0.137	0.133	0.138	0.139	0.125	-	-	-	-	-	-	0.141
Low Grade $7.50 Cutoff	(t x 1000)		633.0	2,518.7	3,590.7	3,294.7	1,501.3	214.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	11,753.0
NSR	$/t		8.751	8.594	8.492	8.046	7.897	7.965	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	8.317
Gold	g/t		0.343	0.346	0.337	0.329	0.330	0.339	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.336
Copper	%		0.188	0.183	0.188	0.176	0.162	0.124	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.179
Reactive Overburden	(t x 1000)		-	470.6	103.4	3,285.5	549.5	-	2,328.4	4,815.9	182.7	-	-	-	-	-	-	-	-	-	-	-	-	11,736.0
Non-Reactive Overburden	(t x 1000)		7,028.0	12,095.5	10,450.7	5,577.7	6,482.3	156.4	3,488.9	8,587.1	6,163.0	44.5	-	-	-	-	-	-	-	-	-	-	-	60,074.0
Reactive Waste	(t x 1000)		606.0	3,443.2	5,425.6	8,867.8	17,527.1	24,486.9	9,564.8	14,359.0	30,170.9	35,488.2	34,818.0	24,764.5	9,747.6	3,959.9	1,012.5	929.5	381.5	-	-	-	-	225,553.0
Non-Reactive Waste	(t x 1000)		152.0	2,245.8	7,551.6	9,966.8	7,695.0	7,495.0	20,052.1	13,413.9	9,396.0	7,310.2	6,643.4	4,892.2	1,748.8	1,676.8	352.2	1,106.0	230.1	-	-	-	-	101,928.0
Total	(t x 1000)		9,158.0	31,869.8	56,097.9	62,303.6	64,841.2	66,878.1	70,635.4	71,897.3	75,820.4	71,774.2	71,058.5	65,483.7	43,826.2	36,225.2	27,621.3	27,595.5	26,171.6	6,822.1	-			886,080.0
Total Mining	tpd		25,100.0	87,300.0	153,700.0	170,700.0	177,600.0	183,200.0	193,500.0	197,000.0	207,700.0	196,600.0	194,700.0	179,400.0	120,100.0	99,200.0	75,700.0	75,600.0	71,700.0	18,700.0	-	-	-	117,400
Cumulative Low Grade			1,015.9	5,495.8	12,502.1	21,547.9	28,575.2	37,755.0	47,396.3	52,557.7	56,905.4	60,276.6	64,313.7	74,580.8	81,350.7	86,379.1	87,075.7	87,075.7	87,075.7	87,075.7	87,075.7	87,075.7	87,075.7	
Cumulative Reactive Waste & Overburden			606.0	4,519.8	10,048.8	22,202.1	40,278.7	64,765.6	76,658.8	95,833.7	126,187.3	161,675.5	196,493.6	221,258.1	231,005.6	234,965.5	235,978.0	236,907.5	237,289.0	237,289.0	237,289.0	237,289.0	237,289.0	
Cumulative Non-Reactive Overburden			7,028.0	19,123.5	29,574.2	35,151.8	41,634.1	41,790.5	45,279.4	53,866.5	60,029.5	60,074.0	60,074.0	60,074.0	60,074.0	60,074.0	60,074.0	60,074.0	60,074.0	60,074.0	60,074.0	60,074.0	60,074.0	
Cumulative Non-Reactive Waste			152.0	2,397.8	9,949.5	19,916.3	27,611.3	35,106.3	55,158.3	68,572.3	77,968.3	85,278.5	91,921.9	96,814.1	98,562.9	100,239.7	100,591.9	101,697.9	101,928.0	101,928.0	101,928.0	101,928.0	101,928.0	
Stockpile Inventory																								
Low Grade $5.25 Cutoff			382.9	2,344.1	5,759.7	11,510.8	17,036.9	26,002.0	35,643.3	40,804.7	45,152.4	48,523.6	52,560.7	62,827.8	69,597.7	74,626.1	75,322.7	75,322.7	75,322.7	68,337.8	42,777.8	17,217.8	-	
NSR			6.359	6.397	6.522	6.496	6.453	6.434	6.416	6.425	6.430	6.385	6.327	6.280	6.294	6.299	6.296	6.296	6.296	6.296	6.296	6.296	-	
Gold			0.227	0.261	0.279	0.280	0.280	0.282	0.277	0.278	0.277	0.276	0.273	0.269	0.268	0.267	0.267	0.267	0.267	0.267	0.267	0.267	-	
Copper			0.174	0.156	0.151	0.150	0.149	0.148	0.146	0.146	0.145	0.145	0.144	0.142	0.142	0.142	0.141	0.141	0.141	0.141	0.141	0.141	-	
Low Grade $7.50 Cutoff			633.0	3,151.7	6,742.4	10,037.1	11,538.3	11,753.0	11,753.0	11,753.0	11,753.0	11,753.0	11,753.0	11,753.0	11,753.0	11,753.0	11,753.0	11,753.0	11,753.0	-	-	-	-	
NSR			8.751	8.625	8.554	8.387	8.323	8.317	8.317	8.317	8.317	8.317	8.317	8.317	8.317	8.317	8.317	8.317	8.317	-	-	-	-	
Gold			0.343	0.345	0.341	0.337	0.336	0.336	0.336	0.336	0.336	0.336	0.336	0.336	0.336	0.336	0.336	0.336	0.336	-	-	-	-	
Copper			0.188	0.184	0.186	0.183	0.180	0.179	0.179	0.179	0.179	0.179	0.179	0.179	0.179	0.179	0.179	0.179	0.179	-	-	-	-	
Combined			1,015.9	5,495.8	12,502.1	21,547.9	28,575.2	37,755.0	47,396.3	52,557.7	56,905.4	60,276.6	64,313.7	74,580.8	81,350.7	86,379.1	87,075.7	87,075.7	87,075.7	68,337.8	42,777.8	17,217.8	-	
NSR			7.849	7.675	7.618	7.377	7.209	7.020	6.887	6.848	6.820	6.762	6.691	6.601	6.586	6.573	6.568	6.568	6.568	6.296	6.296	6.296	-	
Gold			0.299	0.310	0.312	0.306	0.303	0.298	0.292	0.291	0.290	0.288	0.285	0.280	0.278	0.276	0.276	0.276	0.276	0.267	0.267	0.267	-	
Copper			0.183	0.172	0.170	0.165	0.161	0.158	0.154	0.153	0.152	0.151	0.150	0.148	0.147	0.147	0.147	0.147	0.147	0.141	0.141	0.141	-	
Stockpile Recovery & Construction Borrow Material																								
Low Grade $5.25 Cutoff			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	6,985.0	25,560.0	25,560.0	17,217.8	75,322.7
NSR			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	6.296	6.296	6.296	6.296	6.296
Gold			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.267	0.267	0.267	0.267	0.267
Copper			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.141	0.141	0.141	0.141	0.141
Low Grade $7.50 Cutoff			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-		11,753.0	-	-	-	11,753.0
NSR			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	8.317	-	-	-	8.317
Gold			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.336	-	-	-	0.336
Copper			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.179	-	-	-	0.179
High Grade Stockpile - Recover in Year 1				356.1																				356.1
NSR				11.599																				11.599
Gold				0.424																				0.424
Copper				0.235																				0.235
Combined			-	356.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	18,738.0	25,560.0	25,560.0	17,217.8	87,431.8
NSR			-	11.599	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	7.563	6.296	6.296	6.296	6.589
Gold			-	0.424	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.310	0.267	0.267	0.267	0.277
Copper			-	0.235	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.165	0.141	0.141	0.141	0.147
Target Ore Production				9,135.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	25,560.0	494,775.0
Stockpile Ore Required				-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	8,342.3	
Construction Borrow Material ovb & rock			0.0	-	-	-	-	-	-	-	-	-	-	944.0	762.7	-	0.3	4,893.6	-	-				6,600.6
Total Material Moved			9,158.0	32,225.9	56,097.9	62,303.6	64,841.2	66,878.1	70,635.4	71,897.3	75,820.4	71,774.2	71,058.5	66,427.7	44,588.9	36,225.2	27,621.6	32,489.1	26,171.6	25,560.0	25,560.0	25,560.0	17,217.8	980,112.4

Figure 18-12
Material Movement Schedule



Year 1 ore production has been reduced to 9.4 million tonnes to allow for startup and commissioning. Annual waste and overburden quantities have been calculated according to the strip ratio of the scheduled benches. A total of 400 million tonnes will be mined and hauled directly to the primary crusher.

The mine preproduction development quantities are minimal. The ore deposit is overlain by a thin layer of overburden in the starter pit area. Removal of the quantity of material required for tailings dam construction in the pre-production period is adequate to release ore in Year 1.

Ore production from the open pit will cease in Year 17 of the current mine plan. Recovery of ore from stockpiles will sustain mill production into the middle of Year 20 of the mine plan. A total of 87 million tonnes will be recovered from stock pile at the end of mine life.

Equipment Selection

The mine equipment has been selected given the following considerations:

- The simultaneous distribution of multiple operating faces at several locations determined by the long range production schedule
- The necessity to minimize unit operating costs by using large scale mining equipment
- Use of well proven equipment technology and coordination of operating machines using advanced systems
- Use of equipment assembled with modular components in order to minimize onsite maintenance allowing maintenance personnel to focus on servicing and component replacements.
- Use of some available and appropriate equipment from the Gibraltar Mine during the pre-production period and transitioning into full production.

The mine will operate using electric cable shovels and rotary drills. Diesel electric trucks and a support equipment fleet will gradually be increased to match the production schedule that will peak in terms of total production in Year 7 through Year 9 at 200,000 tpd. The annual equipment requirements for the mine are summarized in Table 18-6 Equipment Requirement.

Table 18-6
Pit Equipment Requirements

Project Year	PPdn	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Major Equipment																					
Blasthole Drill		1	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	0	0	0	0
Hydraulic Drill	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0	0
Cable Shovel (4100)		1	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	1	1	1
Wheel Loader (L1850)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Haul Truck (220t)		4	12	20	22	25	25	25	28	30	30	30	25	17	16	16	16	8	4	4	4
Haul Truck (190t)	5	5	3																		
Track Dozer (D10)	4	4	4	5	7	7	7	7	7	7	7	7	5	4	4	4	3	2	2	2	2
Wheel Dozer (834)	0	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Grader (16M)	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	2
Water Truck (190t)	0	0	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Scraper (637)	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	3	3	3	1
Portable Crusher (550JG)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Support Equipment																					
Blasthole Stemmer	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	0	0	0	0
Cable Reeler (980)	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
966 Wheel Loader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
980 Wheel Loader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Compactor	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	
Excavator (325)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Low Bed (90t)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tire Manipulator (980)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lighting Plant	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Engineering 4 x 4	1	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	2
Engineering 4 x 2	1	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	3	3	3	3
Pit Services 4 x 4	4	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	8	8	8	8
Pit Services 4 x 2	0	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2	2	2	2
Pit Services Bus (20 pass)	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Pit Services Bus (32 pass)	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0
Shovel Crew Flat Deck (3t)	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Shovel Crew Hiab (5t)	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Surface Crew Hiab (10t)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Surface Crew Stinger	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel & Lube Truck	2	2	2	2	3	3	3	3	3	3	3	3	3	3	3	3	2	1	1	1	1
Blasting Truck	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	0	0	0	0
Light Repair Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Water Truck	2	2																			
Western Star	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1

In general it is expected that the major equipment will have an effective operating time of 85% corresponding with a 51 minute hour. The available versus scheduled time will be 10.2 hours available per shift or 85% of the shift. Detailed equipment productivity calculations have been made on an annual basis for drills, shovels and trucks. Support equipment operating time has been factored on an annual basis according to the total annual material movement.

Mechanical availability of the individual drills, shovels and trucks were fixed over the life of the mine. Operating costs, discussed in Section 18 have been calculated on a life cycle basis. The major mining machines including blasthole drills, shovels and trucks are expected to be useful for the life of the mine. No drill, shovel or truck replacements are anticipated.

The primary blasthole drills will be electric powered rotary machines capable of drilling 311 mm holes.

The loading fleet will consist of a total of 3- 43.0 m^3 capacity electric cable shovels. Initially, a loader will be required in preproduction, prior to availability of electric power, with shovels coming into production in Year 1. The loader will then be available to work in stockpile areas, low face conditions and where required to meet production objectives during periods of unscheduled shovel downtime.

The haulage trucks selected are 222 tonne capacity diesel electric off road end dump units. The selected truck fleet matches the loading units and overall haul profiles. Truck additions will be made as required in each year until the fleet total of 30 trucks is reached in Year 9 of the mine plan. As previously stated, the mine will be utilizing some equipment from Gibraltar. There will be 5 – 190 t trucks utilized through the pre-production period and ultimately replaced by Year 3.

The mining support equipment includes track dozers, wheel dozers, graders, water trucks, and scrapers required for road, bench and dump maintenance. Miscellaneous ancillary equipment is also required to service, maintain the major equipment and support ongoing pit operations.

Explosives will be delivered to the blasthole by a contracted supplier. The blasting crew will require support equipment to pump wet holes, deliver blasting accessories and stem holes. The bulk delivery trucks and storage facilities will be provided by the explosives contractor.

In general, the life of the project matches the expected equipment life expectancy for the cable shovels, rotary drills and haulage trucks.

Low Grade Ore

Ore which is not taken to the mill for processing during the early years of the project will be hauled by truck to a storage area above the tailings pond. This stockpile will be re-handled and shipped to the mill when the main pit has been mined out or during times of low pit ore release.

Mined Waste Rock

Mined waste rock will be classified as either reactive or non-reactive based on the potential for acid generation.

Reactive waste rock will be hauled from the mine and placed in the tailings storage facility in order to prevent oxidation by isolating it from the atmosphere through submersing.

Non-reactive waste rock will be used as construction materials in roads, dam embankments, and platforms for the low grade and overburden stockpiles.

18.4 Processing and Concentrator

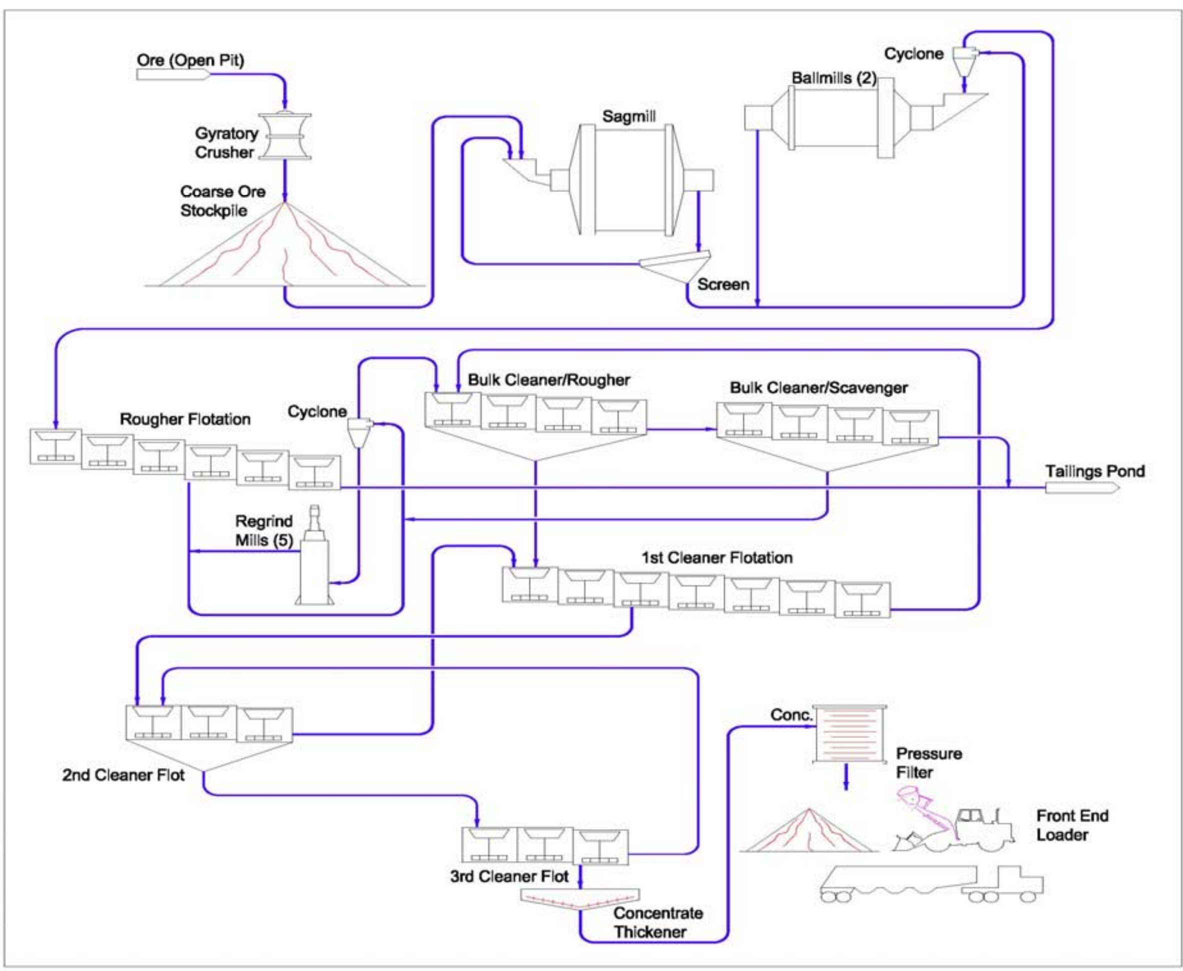
The process design criteria have been developed based on extensive metallurgical testwork and current industry operating experience in the processing of copper-gold ores. Further as a direct result of the ability of the mine to support a large production rate, the process design criteria have incorporated the latest industry proven technology for large scale comminution and recovery equipment. The benefits of utilizing large capacity process equipment will be realized in lower process operating unit costs.

Specific aspects of the design criteria are:

- Single large, industry proven primary semi-autogenous grinding mill
- Three large, industry proven secondary ball grinding mills.
- Five vertical stirred regrind mills
- Large scale flotation cells
- Bulk handling systems for concentrate
- Process automation

The process design is conventional and consists of SAG and ball mill grinding; bulk sulphide flotation, regrind and bulk rougher/scavenger cleaner flotation, cleaner flotation and concentrate dewatering. A simplified flowsheet is shown in Figure 18-13. The concentrator is designed to operate 24 hours per day, 365 days per year.

Figure 18-13
Simplified Flow Sheet



The primary SAG mill size was based on MacPherson AG (Autogenous Grinding) work indices derived from testwork. The ball mill sizes were based on standard Bond Ball Mill work indices also determined from testwork. An evaluation of the large scale processing equipment available with respect to the testwork requirements indicated that a nominal processing capacity of 70,000 tonnes/day could be achieved.

The bulk of the metallurgical testwork was performed at Lakefield Research under the supervision of Melis Engineering. Semi-autogenous grinding amenability testwork using the MacPherson test procedure was performed by Hazen Research. Bond ball mill grinding testwork was performed by Lakefield. Flotation testwork performed by Lakefield consisted of batch tests, locked cycle tests and pilot plant runs. The sample material used for the tests was either half-core or assay reject samples from the upper, middle and lower zones of the ore body.

Based on three half-core tests, a cost benefit analysis using the value of incremental copper and gold recoveries against net grinding power costs concluded that the optimum 80% cumulative passing size (K_{80}) of the primary grind was approximately 160 µm.

Five batch tests were completed to analyze the effects of regrind particle size distribution on copper and gold recoveries to the third cleaner concentrate. Based on this testwork, Melis recommended that the optimum regrind K_{80} was between 14 and 17 µm.

Kilborn's analysis of the regrind testwork results suggests the copper and gold recoveries may actually improve when the regrind becomes slightly coarser. Testwork performed by G & T Laboratories in November and December 1998 on Prosperity concentrate also indicated that a coarser grind may achieve increased recoveries, especially with respect to gold. In light of this data and taking into account regrind ball mill size considerations, Kilborn has specified a regrind K_{80} of 19 µm in the design criteria. Additional grind-recovery testwork would be required to confirm the optimum regrind K_{80}.

The copper and gold head grades in the design criteria reflect those presented in the Life of Mine production schedule (see Section 18.2). The projected recoveries presented in the design criteria have been determined by Kilborn based on it's evaluation of the testwork results and the mine head grades. Based strictly on the testwork results, Melis estimated a target recovery for copper of over 90% for all three ore zones. Kilborn analysis of the flotation testwork results generally concur with the Melis testwork. However, for actual plant operation Kilborn has projected average copper recoveries of 87.0, 88.7 and 90.2 for upper, middle and lower zones respectively, though projected recoveries will vary depending on head grade and mineralogy. Plant recoveries are approximately 2 percent lower than the testwork recoveries and reflect Kilborn industry experience where flotation plant recoveries tend to be lower than laboratory scale test recoveries. These lower plant recoveries set by Kilborn are a result of process inefficiencies realized in actual plant operating conditions.

In flotation testwork the gold recoveries showed significant variation which has made it difficult to predict the expected recovery. This difficulty is compounded by the lack of comparable tests performed under similar conditions at Lakefield. However, Kilborn analysis of the testwork indicated that the gold recovery projections presented by Melis may be conservative and Kilborn used these recoveries in the design criteria. Kilborn recommends that additional testwork be performed to more accurately predict the recovery of gold in the flotation circuit.

Predicted recoveries and grades as shown in Table 18-7 were used as a basis for the process and concentrator design.

Table 18-7
Predicted Recoveries and Grades

Zone	Head Grade		Concentrate Grade		Recovery %	
	% Cu	gpt Au	% Cu	g/t Au	Cu	Au
Upper	0.217	0.485	24.31	44.7	87.0	71.6
Middle	0.233	0.441	24.6	40.4	88.7	77.0
Lower	0.310	0.496	25.8	35.9	90.2	78.4

18.5 Processing, Crushing and Ore Reclaim

The Prosperity crushing circuit has been designed to crush on average 70,000 t/d of run-of-mine ore from minus 1,000 mm to 100% passing 350 mm and 80% passing 150 mm at a nominal rate of 4,444 tonnes/hr. The crushing system will consist of a single 1.52 x 2.79 m gyratory crusher operating at 75% utilization per 21 hour day, 365 days per year basis. The primary crusher will be located near the east corner of the ultimate pit boundary.

Product from the crusher will discharge into the surge pocket directly below. Crushed material will be drawn from the surge pocket by a variable speed apron feeder which will discharge onto a 1.8 m wide overland conveyor which will transport the crushed ore from the primary crusher 1,900 m to the coarse ore stockpile. The coarse ore stockpile is estimated to have a live capacity of 55,000 tonnes. A belt weigh scale, and a tramp iron magnet to remove tramp metal, will be installed on the conveyor.

The primary crusher will have a 10 tonne auxiliary hoist for maintenance purposes. A hydraulic rock breaker will be provided at the crusher dump pocket for breaking any oversize run-of-mine material.

Crushed ore will be reclaimed from the coarse ore stockpile via three variable speed apron reclaim feeders located in a concrete reclaim tunnel. The four apron feeders will feed directly onto the SAG mill feed conveyor.

Grinding

The primary grinding circuit will be designed to process an average of 70,000 tonnes/day at 90% utilization on a 24 hour per day, 365 day per year basis. The grinding circuit will be designed to reduce the crushed ore from 80% passing 150 mm to 80% passing 170 µm. The grinding operation will consist of a SAG mill and two ball mills.

The SAG mill will be 12.2 m in diameter (inside shell) by 6.1 m long (effective grinding length) and will be driven by a 21,000 kW variable speed wrap-around gearless drive motor. The SAG mill has been designed to operate with a nominal ball charge of 11 percent by volume. The product from the SAG mill circuit is projected to be 80% passing 2,000 µm. The SAG mill will discharge via a trommel screen into the SAG mill discharge pump box. The trommel screen oversize is returned to the SAG mill conveyor by a series of three conventional conveyors. Provision is made to install a pebble crusher in the future if required.

The SAG mill discharge is pumped by one of two installed SAG mill discharge pumps to a gravity discharge distributor located above the mill grinding floor. The flow is split into three equal parts to feed the two ball mill circuits operating in parallel.

Each of the two ball mill circuits consists of a 7900 mm diameter by 14600 mm long ball mill with dual pinion drive. The motors are 18,000 kW wrap around motors.

The ball mills each discharge into a cyclone feed pump box and a cluster of 12 cyclones are fed by a variable speed cyclone feed pump. The cyclone underflow returns to the ball mill while the cyclone overflow discharges to the flotation circuit.

The 80% passing 200 µm cyclone overflow will flow by gravity to the bulk flotation circuit. The cyclone underflow will be returned as ball mill feed. An on-line particle size indicator (PSI) will be provided for monitoring of the cyclone overflow streams and to provide grinding process control.

A weigh scale will be provided on the SAG mill feed conveyor for controlling, monitoring and recording the concentrator fresh feed rate. A weigh scale will be provided on the SAG mill screen oversize conveyor to monitor SAG mill circulating loads.

Bulk Rougher/Scavenger Flotation

The bulk flotation circuit will consist of three parallel trains of bulk rougher cells.

Cyclone overflow from each of the three parallel grinding circuits will flow by gravity into a collection box and then split into three equal portions to feed each of the three rows of seven cells. The cells in each row are 200 m^3 forced air cells, giving a total of 21 cells for the complete bulk rougher flotation area. This combination of cells provides the power and retention time required for flotation. The tailings from the last scavenger cells in each row are combined as bulk flotation tailings.

The flotation reagent distribution system will permit the addition of flotation reagents, collector, promoter and frother at each cell in the bulk sulphide flotation circuits. Flotation air for all flotation sections will be provided by 6 air blowers (5 operating, 1 stand-by), that will supply air to all flotation cells via a ring-main system. The air supply for each cell will be controlled by valves located at specific locations.

On-stream analyzers will be installed to provide on line analysis of the flotation products. This data will be monitored continuously and used to provide optimum process control for the flotation circuits.

Regrind and Bulk Cleaner Flotation

The concentrate from the bulk rougher/scavenger flotation circuit is combined with the bulk cleaner scavenger concentrate as feed to the regrind circuit. In order to provide the liberation required to achieve the design copper concentrate grade and copper and gold recovery in the concentrate, the regrind circuit P_{80} of 20 µm is required. The regrind circuit is designed with five parallel vertical stirred regrind mills. Each regrind mill circuit consists of a 1,120 kW stirred mill operating in closed circuit with a cluster of seven 250 mm cyclones. The cyclone overflow at 20% solids flows to the bulk cleaner flotation circuit.

The bulk cleaner circuit consists of a single row of four 130 m^3 forced air flotation cells followed by a further four 130 m^3 forced air bulk cleaner scavenger flotation cells. The bulk cleaner concentrate is pumped to the copper first cleaner flotation circuit while the bulk cleaner scavenger concentrate is returned to the regrind circuit.

The bulk cleaner scavenger tails are combined with the bulk rougher scavenger tailings to form the final plant tailings.

Cleaner Flotation

The bulk cleaner concentrate is cleaned in three stages of copper cleaning at an elevated pH to produce the final copper concentrate. The tailings from the copper cleaning circuit are returned to the head end of the bulk cleaner flotation circuit.

The copper first cleaner flotation uses six 30m^3 forced air flotation cells. The concentrate is pumped to three 30 m^3 forced air flotation cells. The copper first cleaner tails are pumped to the bulk cleaner circuit. The copper second cleaner concentrate is pumped to the copper third cleaners and the tailings are returned by gravity to the copper first cleaners.

The copper third cleaners consist of three 30m^3 forced air flotation cells. The third copper cleaner concentrate is the final copper concentrate, which is pumped to the concentrate thickener, and the copper third cleaner tailings are returned by gravity to the copper second cleaners.

Concentrate Dewatering and Loadout

Final copper-gold concentrate from the cleaner flotation circuit will be pumped to the concentrate dewatering and load-out circuit. The concentrate dewatering circuit will consist of thickening and filtering unit processes.

Concentrate from the cleaner flotation circuit will be pumped to the 16 m diameter high capacity concentrate thickener. The concentrate will be thickened to approximately 60% solids by weight and pumped to one of two 4.88 m diameter x 4.80 m high agitated concentrate stock tanks that will provide approximately 5 hours of total storage capacity. Overflow from the concentrate thickener will flow to a standpipe and will be pumped to the distribution points in the milling and flotation areas.

Concentrate from the stock tanks will be pumped to a 120m^2 automatic pressure filter where the moisture content of the concentrate will be reduced to an estimated 8%. Filtrate from the concentrate filter will be combined with the thickener feed and gravity flow to the concentrate thickener feed box.

Concentrate filter cake from the pressure filter will discharge (free-fall) directly onto stockpiles located on floor level below the filters. The concentrate will be reclaimed by a front end loader and loaded directly into bulk concentrate highway transport "B" trailers. The concentrate trailers will report to a weighscale to determine the concentrate load and then proceed to a truck wash station where the under side of the truck-trailer units will be spray washed to remove and recover concentrate residue and spillage prior to exiting the load-out facility. This measure is intended to minimize potential environmental concerns. The truck trailer units will the transport the concentrate to the Gibraltar Concentrate load-out facility for transfer to the CN rail transport system for transport to Vancouver.

Process, Fresh, and Fire Protection Water

The concentrator water system will consist of four water distribution systems: process, fresh, potable and fire protection.

The Prosperity water supply system has been designed to maximize the recycle of process and mine waters for re-use in the process. Process water will be made up of water recovered from the concentrate dewatering system, tailings containment pond, run-off water pond and pit depressurization wells. All process water will be stored in the $110{,}000^{m3}$ process water pond prior to being gravity fed to a header inside the concentrator building. Process water will provide the bulk of water required for process and clean-up purposes. The process water will be pumped from the header through the concentrator in a ring-main for distribution to all points of use.

Potable water will be supplied from 3 deep aquifer wells situated above the plant site. The water will be pumped to the 5.00 m diameter x 5.40 m high potable water tank. Two distribution pumps (1 operating, 1 stand-by) will deliver water to the primary crusher, concentrator and the camp.

Fresh water will be supplied from 12 shallow and 9 deep pit depressurization wells. Fresh/raw water will be stored in the 11.00 m diameter x 8.50 m high fresh water tank.

Fresh water pumps (1 operating, 1 stand-by) will supply water to the dust collector extraction fans and mill heat exchanger.

Gland water pumps (1 operating, one stand-by) will supply water to the gland seals of the process slurry pumps.

Fire water will be stored in the lower portion of the fresh water tank. Approximately $500m^3$ of water will be reserved for fire water, which will provide the required 1 hour of reserve at 340 m3 per hour consumption. Fire water will be pumped through the concentrator fire water ring-main and into the fire hydrant perimeter ring-main. An electric jockey pump will maintain fire water pressure in the fire sprinkler system. An electric fire water pump will be provided to supply the fire water distribution system. A diesel powered fire water pump will be installed to provide back up pumping capability in the event of loss of electrical power.

Reagents and Services

Reagent mixing, storage and distribution systems will be provided for the lime, primary collector, secondary collector, frother, flocculant and spare reagent.

Quick lime will be delivered in bulk to the mill site in self unloading trucks and will be off-loaded into a 280 tonne capacity lime silo via a pneumatic transfer system. The lime will be fed from the silo via a screw feeder into a 2.1 m diameter x 2.70 m long lime slaking ball mill. Lime consumption is projected to be in the range of 530 gpt.

The proposed flotation collectors, based on the metallurgical test program are SIBX (sodium isobutyl xanthate) as the primary collector and TNC 312/TNC 401 (dialkyl thionocarbamate) as the secondary collector.

The secondary collector will be pumped from the bulk container truck into the 4.50 m diameter x 4.90 m high secondary collector storage tank from where it will be distributed to the respective flotation cells by dedicated metering pumps.

The proposed frother based on the metallurgical test program is MIBC and Anionic 919 flocculant will be required for concentrate thickening.

Reagent consumption estimates are included in Section 18.10.

18.6 Recoverability

Recoverability estimation is covered in Section 16.

18.7 Markets

The 2007 Feasibility Study includes comments from a report by Neil S. Seldon & Associates Ltd. (NSA) on metal prices, concentrate refining and treatment charges, marketing, and logistics. The consultant also provided to Taseko an update of the current penalty schedule.

There have been no direct discussions with smelters and marketability comments are limited to NSA opinion.

This technical report has relied upon this report as well as other market related information and Taseko Mines Limited own market and logistics experience in its preparation.

Copper

Commodity prices in general have risen in the face of falling US dollar value, rising costs, expanding demand in face of tight supply and over the last several years, with a substantial flow of investment funds contributing to such rise. Metal prices are no exception and it is apparent that a higher plateau of long-term levels has been established. The general consensus is that the peak in the present economic cycle was reached in general terms in the second quarter of 2006.

Today there is every indication that copper prices have moved up to a higher long-term plateau. The key period for Prosperity is 2010-2020 which will cover the payback period. From the review of the CRU, BME, and BH reports, and other information the consensus is that prices will continue high through 2009. The expectation is that the high prices we have seen over the last three years or so and the expectation of continued relatively high prices will stimulate production and this could depress prices in the early stages of the 2010 to 2020 period.

While there is an argument that supply shortage brought on by a mine capacity gap in the face of continued growing demand will lead to another price spike as the second decade begins, it is more likely to be later in the decade as the present project pipeline comes in and is absorbed. There is a likelihood that this mine growth will not occur as rapidly as one would expect in a normal cycle.

A general review of long-term forecasts from various sources sees a range of $1.25 to $2.00 but most if not all such forecasts are over the period covering the next 10 years. It is noteworthy that the independent forecasters referenced here are all the lower end of this range.

Based on CRU, BME, and BH material, for the 10 year period from 2010, NSA recommends the following copper prices in constant 2007 dollars:

US$3,500 per tonne ($1.5876/lb) from 2010 to 2014
US$3,000 per tonne ($1.3608/lb) for the balance of the period

The copper price as of September was at US$3.47/lb with the 3-year trailing average 2004-2007 copper price being US$2.49/lb. As a base case, this study has selected a copper price of US$1.50/lb for the duration of the project.

Silver

Silver, like gold, has enjoyed a renaissance. The price today is well beyond expectations of recent years, but is at a level, which is unlikely to be sustained over the longer term. Producers prior to the current cycle were tending to use a planning price for the long-term of $5.00 to $5.50 per ounce in constant 2005 US dollars.

As with gold, in US dollar terms along with other commodities, a higher price plateau seems assured. Silver has found favour with investors and speculators and many analysts are forecasting higher long-term prices.

Silver has increased in price dramatically over the present cycle and closed in early September 2007 at US$12.91/oz with the 3-year trailing 2004-2007 average silver price being US$10.19/oz. As a base case, this study has selected a silver price of US$8.00/oz.

Gold

In US dollar terms along with other commodities, a higher price plateau seems assured. Production costs are rising, gold has found favour and many analysts are forecasting prices well above today's levels, but surprisingly some analysts remain below such a higher level longer term. Given the present and expected weak US dollar value, there is every reason to expect the metal to remain at a higher long term plateau and NSA suggested $575 in constant 2007 dollars as a base case.

The copper price as of September, 2007 was at US$743/oz with the 3-year trailing average 2004-2007 gold price being US$558/oz. As a base case, this study has selected a gold price of US$575/oz.

Smelter Terms

Annual Benchmark terms are not published, but are reported in various publications and indeed are not finite. They represent a consensus of the likely average base numbers negotiated by the major players in Asia. Annual terms in recent years have generally included price participation at plus/minus 10% at a basis copper price $0.90 per pound, although there has been some variation.

One method to evaluate long term charges is to look at historical levels in determining a forecast in the light of expected prices. Over the period from the early 80's to 2004 the sum total of treatment and copper refining charges together with price participation amounted to about 21.5% to 22% of the price with a theoretical 30% concentrate, depending on the information source. The copper price averaged about 93.3 cents and treatment and refining charges about $76 and 7.6 cents respectively.

Looking back at these historical TC RC PP as a percentage and at the fact that until relatively recently most TC RC forecasts for project evaluation in constant dollars were in the low 80's and 8, with PP +/- 10% at 90.

Today, despite a shift in smelting capacity to lower cost regions, assuming a continuing US dollar value, charges will eventually have to move to levels which are economic for the smelting and refining industry. While this is unlikely to happen over the next year or two, in the start up period post 2010 one would expect to see base charges in the 90's and 9's, with little or no price participation.

In discussing treatment and other charges, it needs to be recognized that various charges have an economic effect and are a matter of negotiation. Prior to the last two years or so, for long-term valuation, a treatment charge of $80 to $85 per dmt and a refining charge of $0.08 to $0.085 per pound of payable copper were generally assumed by many, with price participation of + / - 10 % from a base price of $0.90 per pound in Asian markets. However, on the assumption that prices eventually settle at a long-term higher plateau, it is likely that the base price for PP will rise with the TC and RC being adjusted in parallel.

Based on their own review, involvement, and knowledge of the concentrate market and smelter terms, Taseko have assumed charges at:

TC $90 per dmt
RC $0.09 per pound
No PP at $1.50 per pound

A summary of long-term assumptions for treatment & refining and other commercial terms for copper concentrates are presented below:

Payable Metals

Copper: Deduct 1 unit and pay for balance of content

	with refining charges of US$0.09/lb
Silver:	Pay 90%, with a refining charge of US$0.45/oz
Gold:	Pay 97.50%, with a refining charge of US$6.00/oz

Deductions

Treatment Charge (TC):	CIF FO main Asian port parity, US$90/dmt
Price Participation (PP):	None

Penalties:

Arsenic	US$3.00 per 0.1% over 0.1%
Antimony	US$3.00 per 0.1% over 0.1%
Mercury	US$0.20 per ppm over 20 ppm

Marketability

Prosperity concentrate quality is relatively low copper at 23% - 25%, relatively high gold at 35-45 grams/tonne and modest silver at 80 to 100 grams/tonne. This low copper means there will probably be some limit on the quantity that any one smelter will take as the grade is below the average smelter blend and reduces the metal output from the furnaces. Having said this, all other considerations apart, a concentrate with this level of copper, gold and silver should be readily acceptable.

An issue with Prosperity concentrate is the levels of arsenic, antimony and mercury when taken together. Individually, the arsenic in the range of plus/minus 0.2%, while above most smelter blends of 0.1%, will be acceptable.

Antimony in this concentrate at 0.3%-0.4% is well above the penalty threshold of 0.1% and while this is unlikely to affect the ability to sell, it will incur penalties and could well reduce individual smelters quantity interest.

Of these three penalty elements, mercury at 80 to 150 ppm, is probably the most significant. Mercury will incur penalties and not all smelters, even if they blend, will be prepared to take such quality.

From a logistic point of view the likely market for a major part of the production should be Asia. However, before any decision is made as to which area to market into, contact with smelters should be made. Marketing will also relate to financing and the bankability of the counterpart.

In summary Prosperity concentrate will find a market but with the prospective quantity of about 200,000 tons per annum and given the penalty considerations is likely to need to be spread around a number of smelters. While some of the major traders may well be prepared to bid for reasonably substantial tonnages, the quality problem will still be there and leads to the danger of the concentrate competing with itself.

At this time no detailed discussions have been held with potential buyers.

Logistics

For overseas markets concentrates will be moved by truck to an expanded facility at Gibraltar's existing load-out facility near Macalister, just north of McLeese Lake, B.C and then railed to the port facilities in North Vancouver or alternatively railed directly to Eastern Canada.

Transportation cost estimates for this study, based on transport to Asian markets are summarized below on a wet metric tonne (wmt) basis:

Truck mine to rail:	CDN$22.50
Transfer to Rail (Macalister)	CDN$5.66
Rail Freight to Vancouver:	CDN$22.97
Concentrate Storage and Ship Loading:	CDN$24.50
Ocean Freight:	US$55.00

Other Offsite Costs

Handling losses are estimated at 0.175%.

Marine insurance cover from the port of loading to the discharge port is assumed at a rate of 0.02% of concentrate NIV.

Supervision and assaying is estimated at US$1.50/wmt.

18.8 Contracts

No mining, concentrating, smelting, refining, transportation, handling, sales and hedging and forward sales contracts or arrangements have been negotiated to date. Rates and assumptions used within this study's economic analysis follow industry norms.

18.9 Environmental Considerations

Environmental Assessment and Permitting

Introduction and Background

In August, 1993 Taseko filed a "Pre-Application For A Mine Development Certificate" for the project, which at that time was called the "Fish Lake Project". This document outlined the technical, environmental and socio-economic aspects of the proposed project, and provided the preliminary data required to determine the Terms of Reference for an Application for a Mine Development Certificate.

In June 1995, the British Columbia Environmental Assessment Act (EA Act) was proclaimed, and the project was transferred to the new Environmental Assessment Process.

The *Canadian Environmental Assessment Act* (CEAA) was proclaimed in January 1995, establishing for the first time in a federal statute a process for conducting environmental assessments of projects involving the federal government.

Subsequently it was confirmed that the project would be reviewed under the harmonized British Columbia *Environmental Assessment Act* (EA Act) and *Canadian Environmental Assessment Act* (CEAA) review process as a Comprehensive Study.

The EA Office re-established a Project Committee in early 1997, inviting federal and provincial government agencies, local governments and First Nations to participate. The Project Committee, supported by the work of four technical subcommittees established to address water quality and acid rock drainage (ARD); geotechnical considerations; fish, fish habitat and fish compensation issues; and issues related to the transmission corridor and wildlife worked in cooperation with Taseko to identify issues and to develop environmental impact assessment terms of reference known as Draft Project Report Specifications.

In April 1998, Final Project Report Specifications were issued to Taseko along with a request to prepare and submit a Project Report for review. In 2000, largely due to economic considerations, Taseko suspended work on the Prosperity Project. In the intervening years, at Taseko's request, timeline extension orders were issued by the provincial EA Office. The last one, issued in October of 2005, stipulated that work associated with completing the Environmental Assessment Report must be substantially completed by April 30, 2007 or the current assessment of the Project would be terminated and Taseko would have to initiate a new assessment if the project was to proceed.

In April 2006 Taseko initiated work necessary to complete the Environmental Assessment Report in time to meet the April 2007 deadline. On April 5th 2007 the Executive Director of the provincial Environmental Assessment Office repealed and replaced the October 2005 Timeline Extension Order stipulating that the Environmental Assessment Report Specifications provided to date are to be considered as the basis for developing Terms of Reference and ordering that the review process be varied by referring the Project to the Minister of Environment for a determination under section 14 of the Act.

The Federal Responsible Authorities, Department of Fisheries and Oceans, Transport Canada, and Natural Resources Canada, have recommended to the Federal Minster of Environment that the project be referred to a Joint Panel Review. Provincially, the Executive Director of the Environmental Assessment Office has also referred the project to the Provincial Minister of Environment for a decision regarding a Joint Panel.

At the time of writing this report neither the provincial nor federal Ministers of Environment have announced their decision whether or not to accept the recommendation and refer the project to a panel but Taseko is actively engaged with federal and provincial regulatory agencies in the review of the Project in preparation for those decisions.

Regulatory Framework

Under the British Columbia *Environmental Assessment Act* the Project meets the criteria for a reviewable project as defined by the Reviewable Project Regulation. In the case of the *Canadian Environmental Assessment Act* there are a number of regulatory duties listed in the Law List Regulation that trigger an environmental assessment. Federal Responsible Authorities (RA) for this Project include the Department of Fisheries and Oceans (DFO), Transport Canada (TC) and Natural Resources Canada (NRCAN). Other Federal Authorities such as Health Canada and Environment Canada will provide expert advice during the assessment.

Under the Canada-British Columbia Agreement for Environmental Assessment Cooperation signed in March 2004, projects that require a review under both federal and provincial environmental assessment legislation will undergo a single, cooperative assessment, meeting the legal requirements of both governments while maintaining their respective existing roles and responsibilities.

The environmental assessment will identify all government policies, regulations and land use plans that have a bearing on the Project. It will also summarize any legal orders issued pursuant to the British Columbia *Environmental Assessment Act* relating to the review of the Project and will identify all federal and provincial legislation applicable to the Project.

Environmental Baseline Studies

The Project Report Specifications specify that in order to meet the requirements of both the EA Act and CEAA, potential effects to be considered in the environmental assessment (EA) of the proposed project should include direct and indirect effects on: environmental factors, including air and water quality, fish and other aquatic resources, wildlife, terrain and soils, and vegetation; other resource uses; economic and social factors; archaeological and other heritage and cultural resources; and health. In addition, any potential effects on the exercise of First Nations' traditional uses and activities must be identified and assessed.

Since 1993 Taseko has been undertaking baseline studies to collect biophysical and socio-economic information on the above noted environmental factors. The baseline, or without project conditions will be assessed relative to those conditions expected to exist when the project has reached it's full build out or maximum development stage, and then again at the post closure or after reclamation stage of development in the environmental assessment. Once completed, this assessment will be reported in the Environmental Assessment Report.

A summary of key parameters includes:

- Air Quality and Noise
- Water Quality
- Aquatic Ecology
- Wildlife
- Terrain and Soils
- Vegetation
- Other Resource Uses

- Socio Economic
- Archaeology, Heritage, and Cultural Resources

Environmental Impact Assessment

The Project Report Specifications (PRS) contains specific requirements regarding the identification, assessment and management of the potential environmental effects of the project. The objective of this process is to minimize (i.e. reduce or eliminate) the potential adverse effects of the project on the environment through mitigation.

Taseko is continuing to work to reduce or eliminate the potential adverse environmental effects of the project through:

- Project design;
- Construction, operation and closure planning; and
- Compensation where necessary.

This process ensures that any potential effects that remain following consideration of mitigation measures (including compensation), known as residual effects are not significant. The determination of the significance of residual effects will be a principal consideration of the agencies in deciding whether the project should be certified/approved.

Environmental Management

The environmental assessment is likely to identify a number of potential positive and adverse effects of the project on biophysical and socio-economic features. The assessment is likely to conclude that potential adverse effects can be reduced to acceptable levels or avoided altogether through effective environmental management.

Environmental management will be integrated into project design and construction, operation and closure planning and will be subdivided into the following components:

- Mitigation measures to protect biophysical and socio-economic features in the area during the construction and operation phases of the project;
- Reclamation plan to return the project area to an agreed-upon land use following mine closure;
- Compensation measures to offset unavoidable environmental effects on fish and fish habitat;
- A supervision and management program to ensure that mitigation, compensation and monitoring programs are properly implemented, that environmental liabilities are minimized, and that overall environmental performance meets or exceeds approved closure and reclamation plans; and
- Monitoring programs to verify the accuracy of the environmental assessment of the project, determine the effectiveness of mitigation and compensation measures on

environmental features, and ensure compliance with environmental permitting conditions from regulatory authorities.

Closure and Reclamation

As detailed in the Project Report Specifications a conceptual reclamation plan must be provided as part of the environmental assessment. A more detailed reclamation plan will be required as part of the BC *Mines Act* mine permit application when it is submitted.

The current conceptual reclamation plan includes reclamation planning information (maps and descriptions) corresponding to the stages of mine construction, 5-year mark, and mine closure, including interim reclamation objectives, proposed end land uses and the means by which reclamation work will achieve objectives. Upon mine closure, surface facilities will be removed in stages and full reclamation of the Tailings Storage Facility (TSF) will be initiated. General aspects of the closure plan include:

- Selective discharge of tailings around the facility during the final years of operations to establish a final tailings beach that will facilitate surface water management and reclamation;
- Dismantling and removal of the tailings and reclaim delivery systems and all pipelines, structures and equipment not required beyond mine closure;
- Construction of an outlet channel/spillway at the east abutment of the Main Embankment to enable discharge of surface water from the TSF to the open pit and ultimately to Lower Fish Creek. This full closure scenario will also work well in the event of premature closure of the mine;
- Removal of the seepage collection system at such time that suitable water quality for direct release is achieved;
- Removal and regrading of all access roads, ponds, ditches and borrow areas not required beyond mine closure;
- Long-term stabilization of all exposed erodible materials.

Permitting

The environmental assessment will identify federal and provincial licences and permits that may be required by the Project, including requirements associated with any necessary amendment to Schedule 2 of the *Metal Mine Effluent Regulations*. Relevant statutes may include most (if not all) of the following:

1. Fisheries Act (Canada)
2. Migratory Birds Convention Act (Canada)
3. Species at Risk Act (Canada)
4. Navigable Waters Protection Act (Canada)
5. Canadian Environmental Protection Act (Canada)
6. Explosives Act (Canada)

7. Transportation of Dangerous Goods Act (Canada)
8. Mines Act (BC)
9. Environmental Management Act (BC)
10. Water Act (BC)
11. Wildlife Act (BC)
12. Land Act (BC)
13. Forests Act (BC)
14. Forest Practices Code of British Columbia Act (BC)
15. Highway Act (BC)
16. Health Act (BC)
17. Heritage Conservation Act (BC)

18.10 Taxes

The economic model was run on a before tax basis. BC mining taxes were estimated and included in the cash flow model. The project will also be subject to Federal and Provincial income taxes but these rates are not fixed and it is believed that tax planning methods will be available to minimize the affect on project economics.

18.11 Capital and Operating Cost Estimates

Capital Cost – Summary

The direct and indirect capital cost to bring the Prosperity Project into production at a design ore throughput rate of 70,000 tonnes/day is estimated to be $ 807 million as summarized in Table 18-8.

Table 18-8
Pre-production Capital Cost ($ x 1000)

Description	CDN$(x1000)
Site Preparation	$12,197
Mining	$93,748
Crushing, Conveying & Stockpiling	$39,880
Concentrator	$231,487
Tailings Disposal & Reclaim Water	$18,245
Site Infrastructure	$90,241
Offsite Infrastructure & Marketing	$38,973
Total Direct Costs	**$524,771**
Total Indirect Costs	**$154,630**
Owners Costs	**$16,849**
Fish Compensation	**$8,993**
Contingency	**$101,910**
TOTAL PROJECT COSTS	**$807,153**

The capital cost estimate is expressed in constant 2nd quarter 2007 Canadian dollars.

Indirect capital costs forming part of the overall capital cost estimate for the project have been estimated to be $155 million. A summary of the indirect cost estimate by area is shown in Table 18-9.

Table 18-9
Pre-production Indirect Costs ($x1000)

Description	CDN$(x1000)
EPCM	81,697
Construction Indirects	14,733
Catering & Camp Maintenance	13,343
Diesel Generators (Fuel and Maintenance)	7,348
Vendor Reps	4,400
Critical Spares	11,582
First Fills	2,800
Freight and Transport	16,331
Mine Fleet Spares	2,397
TOTAL INDIRECTS	**154,630**

The sustaining capital cost estimate of $309 million over the life-of-mine is summarized in Table 18-10.

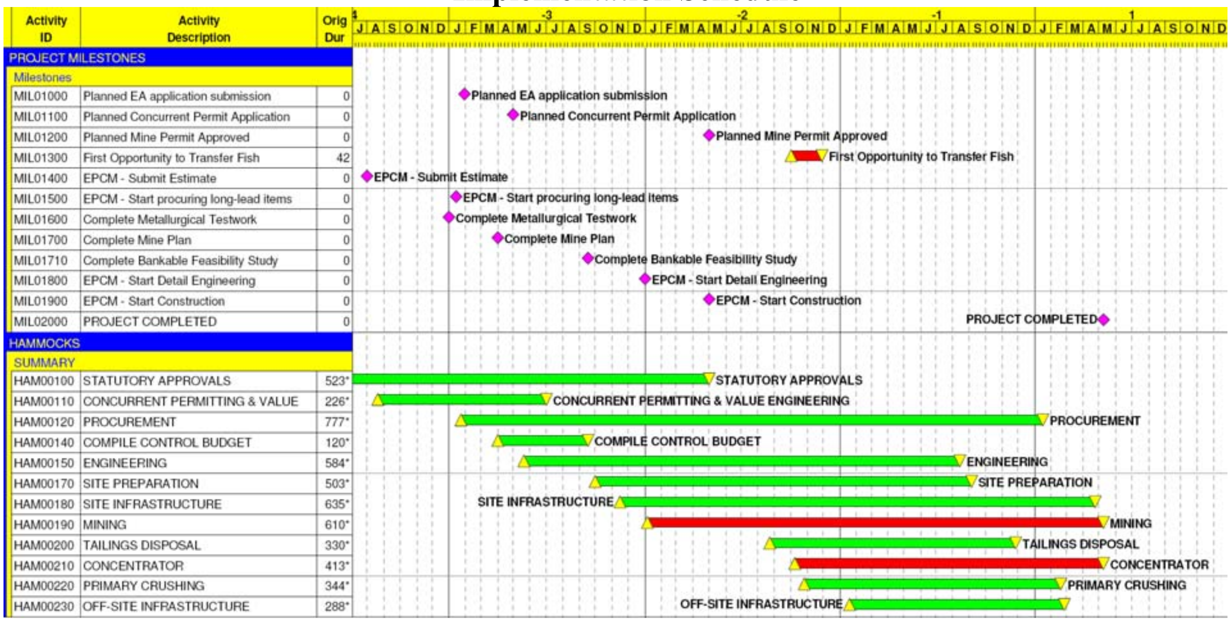
Table 18-10
Sustaining Capital Cost ($ x 1000)

Description	CDN$(x1000)
Mining	$176,335
Plant Services Equipment	$6,287
Tailings	$46,279
Pit Dewatering	$8,965
Mill	$36,837
Fish Compensation	$3,307
Closure	$30,727
TOTAL SUSTAINING CAPITAL	**$308,737**

Capital Cost Schedule

While the implementation schedule from issuance of all required permits to commissioning is approximately two years as shown in Figure 18-14, the expenditure of capital has been scheduled over a three year period, starting in the spring of year -2 and continuing through Taseko's fiscal year 1 (ending September 30).

Figure 18-14
Implementation Schedule



Pit equipment purchases, pit dewatering infrastructure, and tailings dam construction costs through year 1 have been summarized as capital while all mine, mill, and G&A costs in year 1 have been considering operating costs.

The capital expenditure schedule is summarized by work area and year in Table 18-11.

Table 18-11
Pre-production Capital Cost by Year ($ x 1000)

Description	Year -2	Year -1	Year 1
Site Preparation	6,099	6,099	
Mining	17,833	28,504	47,411
Crushing, Conveying & Stockpiling	9,970	19,940	9,970
Concentrator	57,872	115,743	57,872
Tailings Disposal & Reclaim Water	2,082	12,914	3,249
Site Infrastructure	24,218	43,369	22,654
Offsite Infrastructure & Marketing	10,174	19,487	9,313
Total Direct Costs	**128,247**	**246,056**	**150,468**
Total Indirect Costs	**35,372**	**75,785**	**43,473**
Owners Costs	**6,813**	**8,645**	**1,391**
Fish Compensation	**6,257**	**2,736**	
Contingency	**24,543**	**48,276**	**29,091**
TOTAL CAPITAL	**201,232**	**381,499**	**224,423**

Sustaining capital expenditures for the Prosperity Project were estimated on an annual basis over the operating life of the mine. A schedule of expenditures by year and area is shown in Table 18-12.

Table 18-12
Sustaining Capital Cost by Year ($ x 1000)

Description	1	2	3	4	5	6	7	8	9	10
Mining	0	60,085	51,903	10,614	12,725	590	4,592	14,453	11,102	590
Plant Services Equipment				200	1,665		1,579	1,452		382
Tailings		2,249	3,808	4,461	7,804	493	5,764	54	5,654	54
Pit Dewatering		842	533	294	968	665	1,159	301	357	401
Mill		3,633	1,993	1,996		7,435		3,713		7,811
Fish Compensation										
Closure Allowance	655	1,878	1,827	1,899	1,704	1,660	1,859	1,680	1,586	1,428
Total Sustaining Capital	**655**	**68,687**	**60,064**	**19,464**	**24,866**	**10,843**	**14,953**	**21,653**	**18,699**	**10,666**

Description	11	12	13	14	15	16	17	18	19	20
Mining	4,569	1,291	580	1,776	450	365	400	250	0	0
Plant Services Equipment	198	200	387		224					
Tailings	8,558	81		81	6,865	272	81			
Pit Dewatering	1,059	299	160	254	892	782				
Mill		3,480	2,858	651		675	1,308	1,284		
Fish Compensation									1,984	1,323
Closure Allowance	1,430	1,768	1,667	1,802	2,049	2,364	1,246	832	832	561
Total Sustaining Capital	**15,814**	**7,119**	**5,652**	**4,564**	**10,480**	**4,458**	**3,035**	**2,366**	**2,816**	**1,884**

Basis of Estimate - Direct Costs

Approximately 88% of the equipment costs, including mine fleet, mechanical, HVAC and electrical, was developed from budget level quotations.

The capital cost estimates include all direct costs, indirect costs and contingency to construct all facilities required to bring Prosperity mine and concentrator to full production. Owners cost and Fish Compensation allowances are included "below the line" (Does not attract a contingency allowance and are not included under "indirects")

The following technical documentation forms the basis of the HATCH engineered plant solution:

- Process design criteria as described in this report.
- Process Flowsheets;
- Sizing of all major equipment items.
- Mechanical Equipment List;
- General Arrangements/Layouts/Plot Plan;
- Electrical Equipment List;
- Electrical Load List;
- Civil Drawings;
- Single Line Diagrams;

These documents provide the basis for pricing of the complete works. Accordingly the direct cost estimate was derived from the following main input categories:

- Equipment Quotations
- Bulk Material and Earthworks Takeoffs
- Estimates by third parties
- Quotations on standard structures
- Labour & other unit rates developed

Mine pre-production and life of mine plan physicals and costs were developed by Nilsson Mine Services (NMS), Knight Piesold, and Taseko Mines Limited.

NMS used their in-house software and information base to carry out:

- Life of mine pit design and planning
- Equipment size and fleet requirements
- Staffing requirements
- Life of mine operating costs using labour and material cost inputs from Taseko and pit dewatering costs from Knight Piesold

Taseko provided the pre-production mining costs up to the start of year 1 using the pit pre-production requirements outlined by the plan developed by NMS and built-up equipment operating hours and costs.

Based on the above studies, mining fleet requirements were developed and a mining equipment list finalized. Budget quotations for these equipment were received by Taseko and forms the basis of the mining fleet estimate.

Hatch provided the estimate for the electrical distribution throughout the mine and pit floor dewatering facilities.

The engineering and estimating of the tailings, coffer dams, water diversion and other water management systems as well as the waste rock designs were done by Knight Piesold.

Knight Piesold provided the tailings dam construction material requirements, which were incorporated into the pit plan to utilize waste material from the pit to construct the dams. They also provided the cost estimates for dam material placement while Taseko provided the pre-production tailings dam construction material delivery costs.

The estimate also allows for tailings discharge and water reclaim infrastructure including a new barge complete with pumps for the tailings dam. The cost of the tailings pipeline to the tailings dam, as well as the return water lines are included in this estimate.

Engineering and cost estimating for the overhead transmission line from Dog Creek to Prosperity was done by Ian Hayward International Ltd. The estimate includes an upgrade of the existing switching station near Dog Creek. The IHI cost estimate reflects their in-house information base and recent budget quotes for similar work.

Current labour rates and employment conditions for remote site construction in Northern British Columbia were provided by the Christian Labour Alliance of Canada (CLAC). This organization provides qualified crafts for open site projects throughout Canada

The construction work week used for the feasibility estimate is six ten-hour days per week. Overtime over eight hours per day or forty hours per week is paid at time and one-half. The turnaround cycle is three weeks in, and one week out. The estimate includes travel compensation per turnaround. Travel pay is not included.

The estimated direct labour rates used are crew composites for each commodity and include general foremen, foremen, lead hands, journeymen and apprentices. The cost elements include:

- Base Wage Rate
- Premium overtime
- Fringe Benefits;
- Government Assessments;
- Payroll Service charge at 3% of payroll;
- Small tools and consumable supplies at $2.50 per hour worked;
- Contractor Home Office Overheads at 10% of total; and
- Contractor profit at 7.5% of total.

Basis of Estimate - Indirect Costs

The indirect cost estimate for the project includes the following items:

- EPCM based on the proposed project staffing plan, including uplift and completion bonuses, office expenses, computer fees, travel costs, and the use of additional consultants;
- Construction indirects based on built up allowances for all required services;
- Catering and camp maintenance;
- Diesel generators for construction and camp power during the construction period;
- Initial fills;
- Vendor representatives;
- Spare parts;
- Freight

Basis of Estimate - Owners Costs and Fish Compensation

Taseko Mines Limited provided the owners cost estimate to cover all owners costs excluding the mine fleet and mine preparation during the construction period.

The owner's costs were developed by Taseko to cover all owners costs excluding the mine fleet and mine preparation during the construction period. They are derived from time-based resource allocations and include:

- Salaries for owners team, pre-production supervision and support staff, and ramp up to full staffing for the beginning of year 1
- Recruitment and relocation
- Temporary accommodation, transportation, and travel
- Williams Lake office and supplies
- Insurance
- Community outreach
- Environment and assaying
- Legal costs
- Marketing
- Permits, licenses, and leases

Fish compensation costs have been estimated by Knight Piesold based on a suite of potential compensation options developed to achieve DFO's policy of No Net Loss. These have not been finalized with regulatory agencies but provide a reasonable estimate of anticipated cost.

Contingency Estimate

Contingency is sum of money added to an estimate to cover the cost of unforeseeable occurrences after the letter and intent of the scope described in the feasibility report have been duly and diligently identified, quantified and costed or otherwise provided for. Examples of contingent items include but are not limited to:

- Estimate errors and omissions;
- Design developments;
- Pricing variations;
- Unusual weather excluding extreme consequences of extreme events

The contingency capital is estimated to be 15% of the direct plus indirect capital cost, amounting to $102.0 million.

The contingency allowance is intended as a measure of the level of accuracy which can be placed on the capital cost estimate to account for unforeseen costs within the scope of the estimate as well as for unforeseen construction schedule accelerations and delays. Contingency costs may also be incurred due to undefined items of work or equipment beyond the control of the builder, or to uncertainty in some quantity estimates or unit prices for labour, equipment and materials. The contingency allowance should be expected to be spent in the normal course of events.

Sustaining Capital

The sustaining capital cost estimate is expressed in constant 2nd Quarter 2007 Canadian dollars.

The cost of ongoing mining and mining support equipment replacement has been based on the replacement and purchasing schedules developed by NMS/Taseko utilizing budgetary quotes and 2007 Gibraltar equipment purchase prices.

The ongoing cost of plantsite mobile (ancillary) equipment replacement has been based on the replacement and purchasing schedule developed by Taseko and the Kilborn 2000 feasibility study budgetary quotes inflated by 28%.

The ongoing cost of tailings embankment construction, instrumentation, pipework, reclaim water systems and seepage control has been estimated by Knight Piesold.

An annual allowance for additional sustaining capital has been calculated to achieve a minimum annual allowance of $0.13/tonne mined for years 2 through 12, decreasing annually to $0.06/tonne mined in year 18, with no additional allowance in the final two years. This allowance has been allocated to the mill.

Capital Cost Exclusions

The following costs were not included in the capital cost estimates:

- Environmental, archaeological and ecological considerations, other than those incorporated in the current design;
- Costs for acquisition of Rights-of-Way;
- The cost of producing any environmental impact statement and obtaining environmental permits and approvals from local or national authorities;
- Financing charges and interest during construction;
- Currency exchange fluctuations after Sept 1, 2007;
- All costs associated with weather interruption of construction operations;
- Costs of Public Relations activities and any costs of impacts to construction work associated with implementation of Public Relations operations;
- Escalation beyond second quarter 2007;
- Price fluctuations due to unusual market conditions;
- Provision to attract and retain qualified labour during construction;
- Value added tax;
- Owner's head office costs;
- Exploration expenses;
- Construction reclamation costs;
- Sunk costs;
- Federal goods and services taxes; and
- Import duties.

Operating Cost Estimate - Summary

The base case operating costs for the Prosperity Project, including mining, milling, and general and administrative costs have been estimated in 2nd quarter 2007 Canadian dollars and include no allowance for escalation or exchange rate fluctuations.

The average project operating cost for the 20 year life-of-mine is estimated to be \$6.26/t of ore milled.

The operating costs are presented in three major segments:

- Mining: includes the direct costs of mining, including drill, blast, load, and haul activities, roads and dumps, and a general mine expense. The general mine expense includes:

- Milling: includes all operating costs associated with the concentrator from the dump pocket of the gyratory crusher to the discharge of tailings into the tailing pump box and to the loading of concentrate trucks at the mill site. It also includes the power costs for all non-pit activities, plant services labour and equipment, and general site infrastructure and buildings.

- General and Administrative: includes salaries and wages of administrative personnel including purchasing and warehouse. It also includes all fixed costs relative to administration, warehousing, employee relations, IT services, safety and security, training, and the operations camp.

The estimated life of mine average unit costs for each major operating cost area are shown in Table 18-13.

Table 18-13
Life-of-Mine Unit Costs

Operating Category	Operating Cost ($/tonne ore processed)
Mining	$2.27
Milling	$3.55
General and Administrative	$0.44
Total	**$6.26**

Table 18-14 summarizes estimated operating costs by year for each major area.

Table18-14
Operating Costs ($x1000)

Year	1	2	3	4	5	6	7	8	9	10
Mining	25,830	45,516	57,202	66,996	73,875	75,716	77,558	85,931	91,614	89,511
Milling	46,664	87,056	87,506	86,815	88,325	88,325	88,558	88,558	88,558	88,558
G & A	10,725	11,576	11,806	11,946	12,020	11,940	11,960	12,092	12,160	11,981
Total	**83,219**	**144,148**	**156,064**	**165,757**	**174,220**	**175,981**	**178,075**	**186,581**	**192,332**	**190,050**

Year	11	12	13	14	15	16	17	18	19	20
Mining	95,744	66,637	59,666	46,323	48,371	46,794	17,660	13,057	13,031	8,942
Milling	92,492	92,492	92,492	92,492	92,413	92,229	91,426	91,064	90,956	60,701
G & A	12,038	11,257	10,925	10,343	10,375	9,944	9,049	8,608	8,028	6,300
Total	**200,274**	**170,386**	**163,083**	**149,158**	**151,159**	**148,968**	**118,136**	**112,728**	**112,105**	**75,943**

Operating Cost Basis of Estimate

Taseko developed specific operating cost estimates for mining, milling, and general administration. Other key consultants and services providers contributing operating cost estimates include:

• Gibraltar Mines Ltd. (Gibraltar)	Mining
• Nilsson Mine Services Ltd. (NMS)	Mining
• Knight Piesold Consulting (Knight Piesold)	Pit Dewatering Pit Wall Depressurization Tailings Dam Sustaining Capital

The operating cost estimates are based on the following general project data:

- Feasibility level mine designs to determine the size and makeup of the mine fleet;
- Budgetary quotations, current Gibraltar pricing and long-term Gibraltar pricing assumptions for major consumables including power, grinding media, reagents, mill and crusher liners, fuel, tires, and explosives;
- Process reagent consumption rates generated from metallurgical testwork;
- Power requirements from material testwork; and
- Power consumption requirements generated from an assessment of the mechanical equipment and service electrical loads.
- Pit water inflows and depressurization requirements from geotechnical and hydrogeological studies

Power unit cost was based on Gibraltar Mine's current cost. Mill and infrastructure power consumption was derived based on the connected load data from the mechanical equipment list. Energy consumption was based on a 94% motor efficiency. The estimated average annual energy consumption for operating years 7 to 18 (after pumping of tailings has started) is 643 GWh.
An allowance for pit related power totals an additional 50 GWh for purposes of demand but pit related power costs, including electric equipment and pit dewatering costs, have been developed independently based on specific equipment demands and hours of operation.

Operating labour wages have been based on the most recent labour contract negotiated with the Construction and Allied Workers'Union (CLAC), Local 68 at Gibraltar Mines Ltd. reflecting 2007 labour rates.

Wages include payroll burdens to cover employer costs for employee benefits, holiday pay, Canada Pension Plan contributions, Worker's Compensation assessments, employment insurance premiums, and life and long term disability insurance premiums.

Salaries for management, supervisory staff, and technical staff have been based on 2007 Gibraltar salary levels. Salaries include payroll burdens to cover employer costs for employee benefits, holiday pay, Canada Pension Plan contributions, Worker's Compensation assessments, employment insurance premiums, and life and long term disability insurance premiums.

A breakdown of operating manpower by year is summarized in Table 18-15.

Table 18-15
Summary of Estimated Operating Manpower.

	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Mining																				
Supervisory and Technical Staff	41	43	48	49	49	49	49	49	49	49	49	49	49	46	46	33	29	24	24	22
Mining Operations	60	91	121	129	143	140	144	152	158	157	159	125	108	80	87	72	30	19	19	15
Mining Maintenance	21	37	45	52	56	56	56	66	76	66	76	49	44	32	36	36	22	17	18	13
Subtotal	122	171	214	230	248	245	249	267	283	272	284	223	201	158	169	141	81	60	61	50
Milling																				
Staff	9	11	11	11	11	11	11	11	11	11	11	11	11	11	11	9	9	9	8	5
Mill Operations	32	41	41	41	41	41	41	41	41	41	41	41	41	41	41	41	41	41	41	27
Mill Maintenance	44	55	55	55	55	55	55	55	55	55	55	55	55	55	55	55	55	55	55	39
Plant Services	23	26	26	25	25	25	25	25	25	25	25	25	25	25	24	24	18	13	13	10
Subtotal	109	133	133	132	132	132	132	132	132	132	132	132	132	132	131	129	123	118	117	82
General & Administration																				
Administration Staff	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	18	17	13
Purchasing and Warehouse	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	4	3
Warehouse Hourly	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2	2
Subtotal	28	28	28	28	28	28	28	28	28	28	28	28	28	28	28	28	28	28	23	18
Operating Manpower Total	**258**	**332**	**375**	**390**	**408**	**405**	**409**	**427**	**443**	**432**	**444**	**383**	**361**	**318**	**328**	**298**	**232**	**206**	**201**	**149**

Supplies and consumables costs for mining operations include such items as explosives, blasting accessories, fuel, oil and lubricants, filters, tires, equipment wear parts, equipment mechanical and electrical component replacement parts. Costs for explosives, fuel, drill steels and bits have been based on current Gibraltar pricing and contracts, and long range forecasts. Mechanical and electrical component parts including overhaul costs are based on supplier estimates of frequency and cost.

The replacement of parts due to normal wear and tear and equipment breakdown was considered part of the operating cost. Periodic replacement of capital equipment was considered sustaining capital and not included in the operating costs. Initial stocking of spare maintenance parts and operating supplies was considered to be initial capital and therefore has not been included in the operating cost estimate.

Supplies and consumables cost for mill operations include all major items consumed during the operating of the concentrator. Consumption rates for consumable supplies are based vendor recommendations, metallurgical testwork, and Gibraltar operating experience. Unit prices for consumable items are based on Gibraltar current pricing and budget quotations from suppliers F.O.B. Williams Lake plus a $50/tonne freight surcharge.

Annual non-pit mechanical maintenance supplies, electrical maintenance supplies, and building maintenance costs are based on percentages of initial capital costs.

Mining Costs - Summary

The life-of-mine unit mining operating cost is estimated at $2.27/t ore milled or $1.14/t material mined. This operating cost is referenced to the 70,000 tpd ore production schedule over a plant operating life of 19 years

The estimated life-of-mine average mining unit cost of $2.27/t ore milled or $1.14/t mined is summarized in Table 18-16.

Table 18-16
Life-of-Mine Direct Mining Unit Cost

Area	$/t milled	$/t mined
Drill	0.06	0.03
Blast	0.30	0.15
Load	0.19	0.10
Haul	1.07	0.53
Roads and Dumps	0.27	0.14
General	0.38	0.19
Total	**2.27**	**1.14**

Included in this total, the estimated life of mine general mine expense unit cost of $0.38/t ore milled or $0.19/t material moved is summarized in Table 18-17.

Table 18-17
Life-of-Mine General Mine Expense

Area	$/t Milled	$/t Mined
Salaries and Wages	0.20	0.10
Dewatering Power & Maintenance	0.01	0.01
Consumables & Supplies	0.02	0.01
Minor Support Equipment	0.12	0.06
Crushing Screening Road Material	0.02	0.01
Ongoing Clearing & Grubbing	0.001	0.001
Total	**0.38**	**0.19**

Processing Cost Summary

The average processing cost over the life of the mine including processing of low grade stockpiles is estimated to be $3.55/t of ore.

The battery limits for operating costs associated the concentrator are from the dump pocket of the gyratory crusher to the discharge of tailings into the tailing pump box and to the loading of concentrate trucks at the mill site. The concentrator area also captures the power costs for all non-pit activities as well as the plant services labour and equipment.

Concentrator operating costs have been separated into 6 categories; labour, power, consumables, maintenance, mill general and plant services. They have been calculated on an annual and a cost “per tonne of ore processed” basis. The operating costs vary according to grinding media consumption, the requirement to pump tailings, and varying manpower levels.

Typical annual concentrator operating costs by category are shown in Table 18-18.

Processing costs include plant services which encompass tailings dam construction and general site infrastructure support. The plant services component of the milling cost includes only labour and equipment operating costs as items such as power and infrastructure materials are captured within the mill maintenance costs.

Table 18-18
Typical Processing Operating Cost Summary (Year 5-10)

Area	Annual Cost ($x1000)	Unit Cost ($/t milled)
Labour		
Staff	1,117	0.044
Operations	3,017	0.118
Maintenance	4,870	0.191
Plant Services	2,102	0.082
Subtotal	**11,106**	**0.434**
Power		
Water Systems	92	0.004
Fuel Systems	16	0.001
Crushing	1,071	0.042
Conveyors, Stockpile Reclaim	203	0.008
Concentrator Building	882	0.035
Grinding	14,724	0.576
Flotation	2,203	0.086
Regrind	2,487	0.097
Concentrate Thickening	88	0.003
Concentrator Reagents	26	0.001
Concentrate Loadout	2	0.000
Procces Water	213	0.008
Talings Pumping	467	0.018
Water Supply	936	0.037
Water Storage & Distribution	52	0.002
Eng $ Admin. Facitlites	130	0.005
Mine Service Facilities	54	0.002
Assay Laboratory	104	0.004
Operations Camp	173	0.007
Subtotal	**23,924**	**0.936**
Consumables		
Grinding Media - SAG Mill Balls	7,124	0.279
Grinding Media - Ball Mill Balls	18,528	0.725
Grinding Media - Regrind Mill Balls	5,822	0.228
Collector- Xanthate PIBX	2,410	0.094
Collector- Thionocarbamate	4,237	0.166
Frother - MIBC	172	0.007
Flocculant	30	0.001
Lime	2,384	0.093
Subtotal	**40,707**	**1.593**
Maintenance		
Crusher, SAG, Ball & Regrind Liners	5,874	0.230
Mechanical Equipment / Piping	5,187	0.203
Electrical / Instrumentation	655	0.026
Subtotal	**11,716**	**0.458**
Mill General		
Buildings	376	0.015
Subtotal	**376**	**0.015**
Plant Services		
Mobile equipment	730	0.029
Subtotal	**730**	**0.029**
TOTAL COST	**88,558**	**3.465**

The power cost allocated to the mill includes all non-pit related power. The estimated average annual energy consumption for operating years 7 to 19 (after tailings pumping is established at normal annual load) is 643 GWh.

The average power consumption per tonne of ore processed of the SAG mills and ball mills will increase as mining progresses to depths where the hardness of the ore is geologically related to the gypsum line. Since the mills are expected to operate at full power, the annual power consumption in the grinding circuit is expected to remain unchanged independent of the ore hardness.

Reagent consumption rates are based on the results from metallurgical testwork. There may be some justification for applying slightly lower consumption rates in a full scale operation than those experienced in the laboratory due to the use of reclaim water in the full scale operation which will contain some usable reagents and also due to reagent optimization that will occur in the full scale operation. However the reagent consumption rates experienced in the metallurgical testwork have not been reduced for use in this operating cost estimate.

General and Administration Costs

The estimated average life of mine cost for general and administrative (G&A) mine functions is estimated to be $0.44/t ore milled. Major categories are summarized in Table 18-18.

Administrative salaries and wages are based on an estimated G&A manpower complement while salary and wage rates including burdens have been estimated from rates at Gibraltar.

The major fixed costs in this area include property assets insurance, taxes, freight, bussing and environmental costs.

Property assets insurance costs have been based on a loss limit of $150,000,000 for any one loss with a $250,000 deductible.

As the property is not proximate to any municipalities, no municipal taxes are expected. However it is anticipated that some form of regional tax will become applicable and an allowance has been made based on a straight line depreciation of fixed assets.

Return personnel transportation from Williams Lake to the mine will be provided by a chartered bus service. The transportation cost for all operating personnel has been estimated at $500,000 annually based on correspondence with the current transportation contractor at Gibraltar.

An annual allowance of $350,000 has been made for environmental services.

Other components of fixed G&A costs have been based on current Gibraltar costs.

Table 18-19
Estimated General & Administration Costs, ($000's)

Year	1	2	3	4	5	6	7	8	9	10
Salaries and Wages										
Administration	1,529	1,529	1,529	1,529	1,529	1,529	1,529	1,529	1,529	1,529
Purchasing and Warehouse	851	851	851	851	851	851	851	851	851	851
Subtotal	**2,380**	**2,380**	**2,380**	**2,380**	**2,380**	**2,380**	**2,380**	**2,380**	**2,380**	**2,380**
Fixed Costs										
Administration	3,315	3,256	3,189	3,138	3,061	3,014	2,935	2,871	2,791	2,742
Warehouse	545	545	545	545	545	545	545	545	545	545
Employee Relations	665	782	792	795	799	799	800	804	808	805
Computer Services	323	323	323	323	323	323	323	323	323	323
Safety and Security	377	455	502	518	537	534	538	557	574	563
Training	181	217	239	246	255	254	256	265	273	267
Subtotal	**5,406**	**5,578**	**5,589**	**5,565**	**5,520**	**5,468**	**5,396**	**5,364**	**5,313**	**5,244**
Camp Costs	2,940	3,618	3,837	4,001	4,120	4,093	4,184	4,348	4,467	4,357
Subtotal	**2,940**	**3,618**	**3,837**	**4,001**	**4,120**	**4,093**	**4,184**	**4,348**	**4,467**	**4,357**
Total	**10,725**	**11,576**	**11,806**	**11,946**	**12,020**	**11,940**	**11,960**	**12,092**	**12,160**	**11,981**

Year	11	12	13	14	15	16	17	18	19	20
Salaries and Wages										
Administration	1,529	1,529	1,529	1,529	1,529	1,529	1,529	1,529	1,397	654
Purchasing and Warehouse	851	851	851	851	851	851	851	851	549	321
Subtotal	**2,380**	**2,380**	**2,380**	**2,380**	**2,380**	**2,380**	**2,380**	**2,380**	**1,946**	**975**
Fixed Costs										
Administration	2,659	2,589	2,516	2,432	2,345	2,260	2,178	2,093	2,001	1,892
Warehouse	545	545	545	545	545	545	545	545	545	545
Employee Relations	808	794	789	779	781	774	759	753	752	740
Computer Services	323	323	323	323	323	323	323	323	323	323
Safety and Security	575	510	486	440	451	419	348	320	315	210
Training	273	243	232	210	215	200	168	155	152	127
Subtotal	**5,183**	**5,004**	**4,890**	**4,729**	**4,660**	**4,521**	**4,320**	**4,189**	**4,088**	**3,836**
Camp Costs	4,476	3,874	3,655	3,235	3,335	3,043	2,350	2,039	1,994	1,489
Subtotal	**4,476**	**3,874**	**3,655**	**3,235**	**3,335**	**3,043**	**2,350**	**2,039**	**1,994**	**1,489**
Total	**12,038**	**11,257**	**10,925**	**10,343**	**10,375**	**9,944**	**9,049**	**8,608**	**8,028**	**6,300**

Housing for both site operating and ongoing contract personnel will be provided at the Prosperity camp. Camp costs at a rate of \$50/man-day have been estimated and are accounted for in the G&A area. Catering costs include the costs associated with the housing of ongoing contracted personnel for outside services, mining explosives supply, pit dewatering, horizontal drilling, mining mobile equipment erection, and guests. Camp operating costs have been estimated based on Hatch in-house data for previous projects and current budget quotations. Costs are inclusive of camp management, maintenance, housekeeping and catering.

The camp will initially be utilized to accommodate site construction personnel. Upon completion of construction, the facilities will be downsized to accommodate only the site operating personnel.

Working Capital

An amount equivalent to 3 months of first full year project operating costs (approximately $56.6 million) was charged against the project cash flow in Year 1 to provide for the production of initial product inventories. This was credited back to the project cash flow in Year 20 to reflect product inventory drawn down and recapture of receivable accounts.

18.12 Economic Analysis

Summary

Key economic indicators derived from the base case economic analysis are summarized in Table 18-20.

Table 18-20
Key Project Economic Indicators

Total copper production	929,000 tonnes/2,048,000,000 lbs
Total gold production	4,700,000 troy oz
Total silver production	348,000 kg/ 10,800,000 troy oz
Mine Life	20 years
Pre-tax rate of return (IRR)	11.7%
Net Present Value pre-Tax (7.5%)	$260,000,000
Pre-Tax payback period (from decision to proceed)	8.5 years
Pre-Tax payback period (from start of production)	6 years
Initial and ongoing project capital	$1,116,000,000
Net Smelter Return	$5,640,000,000/ $11.59 per tonne ore
LOM Average operating costs (incl. offsite costs)	$9.20/t ore processed
LOM Average operating costs (site costs only)	$6.26/t ore processed
LOM average realization cost (cash cost)	US$1.75 per lb Cu produced
LOM average cost (after by-product credits)	US$0.43 per lb Cu produced

The analysis is based on an ore reserve that processes 486,789,000 tonnes grading on average 0.22% copper and 0.43 g/t gold over a 20 year mine life, and a mill production rate of 25,560,000 t/y (70,000 t/d).

The metal prices used in the base case economic model are US$575/oz gold, US$8.00/oz silver, and US$1.50/lb copper as outlined in Section 18-6.

Unless otherwise stated all dollar amounts used in the analysis are in constant 2nd quarter 2007 CDN$.

The project has been evaluated on a "stand-alone" basis assuming one owner with 100% equity financing for the project and no external corporate structures.

An exchange rate of $0.80 USD per CDN$ has been used in the base case. Inflation factoring has not been applied.

Exchange Rate

While the US/Can dollar exchange rate in September 2007 was approaching par, it is accepted that strong commodity prices lead to strength in the Canadian dollar against its US counterpart. The value of the Canadian dollar is susceptible to commodity prices, particularly oil and metals and it is reasonable to link prediction of the US/Can dollar exchange rate to the commodity cycle.

The correlation between the value of the Canadian dollar in terms of the US/CDN dollar exchange rate and the price of copper is very evident as depicted in Figures 18-15 and 18-16.

Figure 18-15
Constant $ Copper Correlation with CDN/US Currency Exchange

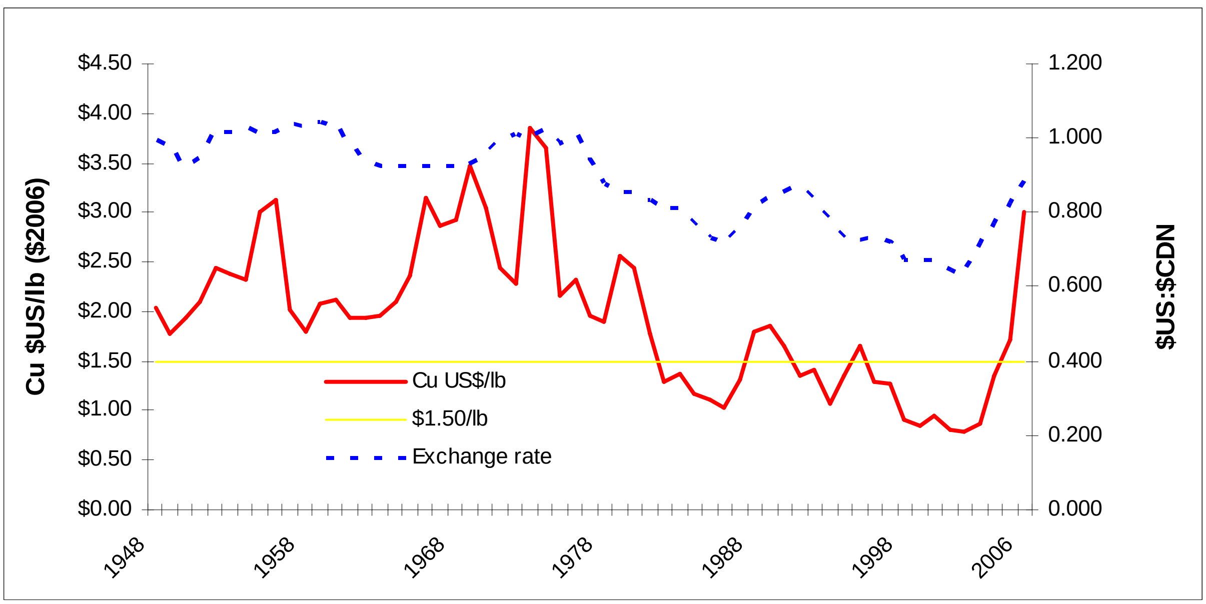
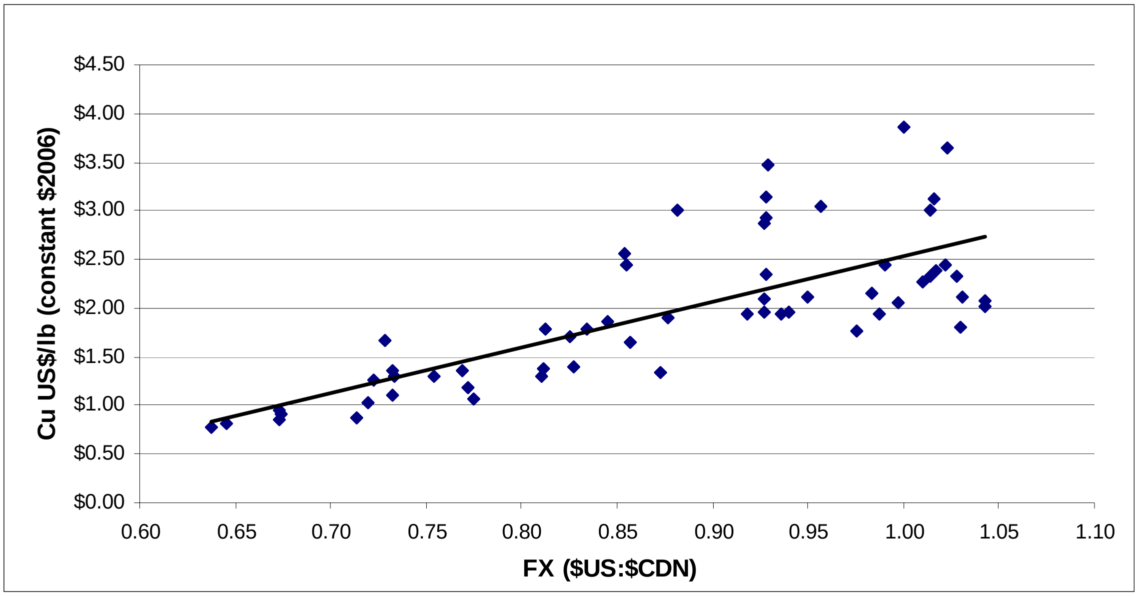


Figure 18-16
Historical Correlation – Copper and Exchange Rate 1948 to 2006



This study, using long term copper prices of US$1.50/lb reflective of the commodity cycle post the current boom, has selected an exchange rate of US$0.80:CDN$1.00.

Production Schedule

The Prosperity mine will produce an estimated 25,560,000 tonnes/year (70,000 t/d) of ore by open pit mining methods with a strip ratio of 0.8:1. An estimated 18% of all ore material will be rehandled as part of the overall ore stockpiling strategy. The economic model is based on a copper recovery to a single copper-gold concentrate with life-of-mine copper and gold recoveries of 87% and 70% respectively using conventional flotation methods.

Over the mine life, a total of approximately 220,000 tonnes of concentrate (wet basis) will be produced annually containing a life-of-mine average of 24.5% copper, 38.5 g/t gold and 89 g/t silver. Concentrate will be trucked to Gibraltar's concentrate handling facilities for transfer to rail transport to various points of sale; primarily through the Port of North Vancouver for shipment overseas.

Revenue

The project's NSR in this economic analysis has been calculated using price forecasts for copper, gold, and silver, concentrate smelter and penalty terms, and inland and ocean freight costs as outlined in Table 18-21.

Table 18-21
NSR Assumptions

Gold	US $1.50 per lb
Copper	US $575 per troy ounce
Silver	US $8.00 per troy ounce
Exchange Rate	US $0.80/CDN $
Treatment Charge	US $0.90/dmt
Copper Refining Charge	US $0.09/lb
Price Participation	none
Silver Refining Charge	US $0.45/oz
Gold Refining Charge	US $6.00/oz
Copper Payable	96.5%, minimum 1 unit
Silver Payable	90%
Gold Payable	97.5%
Penalties (Sb, As, Hg)	US $30.20/dmt
Moisture	7.5%
Mine to Port	CDN $75.63/wmt (years 1-5), CDN $71.44/wmt after year 5
Port Charges	CDN $24.50/wmt
Ocean Freight	US $55/wmt
Losses	0.175%
Insurance	$0.02/$100 NIV
Supervision/assaying	US $1.50/wmt

Deductions from revenue have been made for the presence of mercury, arsenic and antimony that exceed specified limits in the concentrate as follows:

- Arsenic - A penalty of US$3.00 for each 0.1% in excess of 0.1%
- Antimony - A penalty of US$3.00 for each 0.1% in excess of 0.1%
- Mercury - A penalty of US$0.20 for each 1ppm Hg in excess of 20 ppm

Estimated impurity content in concentrate varies with depth, with penalties decreasing from US$37.70/dmt in the upper reserve to US$16.50/dmt in the lower. US$30.20/dmt is the estimate for the middle zone and this value has been used in this analysis.

The average life-of-mine NSR per tonne has been estimated to be CDN$11.59/t milled or CDN$1,489 per DMT of concentrate produced.

Operating Costs

Operating costs were estimated in detail and are presented in Section 18.10. The operating costs for mining, milling, and general and administration are indicated in Table 18.22.

Table 18-22
Operating Unit Costs

Operating Category	Operating Cost ($/tonne ore processed)
Mining	$2.27
Milling	$3.55
General and Administrative	$0.44
Total	**$6.26**

Capital Costs

The project capital cost estimate used in this economic analysis has been estimated in detail and presented in Section 18.10. Costs have been estimated based on feasibility level engineered designs, on quantity take-offs for construction materials, on CLAC construction labour rates, on Gibraltar labour rates, and on budget level quotations for equipment and purchased packages such as pre-engineered or modular structures.

Table 18.22 provides a categorization of the estimated project capital cost in CDN$.

Table 18.23
Capital Cost

Description	C$(x1000)
Site Preparation	$12,197
Mining	$93,748
Crushing, Conveying & Stockpiling	$39,880
Concentrator	$231,487
Tailings Disposal & Reclaim Water	$18,245
Site Infrastructure	$90,241
Offsite Infrastructure & Marketing	$38,973
Total Direct Costs	**$524,771**
Total Indirect Costs	**$154,630**
Owners Costs	**$16,849**
Fish Compensation	**$8,993**
Contingency	**$101,910**
TOTAL PROJECT COSTS	**$807,154**

Ongoing Capital Expenditures

Ongoing project capital expenditures have been estimated in detail and presented in Section 18.10. Ongoing capital expenditures will be required for ongoing replacement of mobile plant and mining equipment, staged pit de-watering and well installation, staged TSF embankment construction, and final property closure and reclamation.

Ongoing capital over the life-of-mine is estimated to be $309 million. No contingency has been applied to ongoing capital.

Reclamation and post-closure monitoring costs have been estimated at approximately $30.7 million. This cost has been reflected as a cash cost of $0.015/lb copper produced.

Working Capital

An amount equivalent to 3 months of first full year project operating costs (approximately $56.6 million) was charged against the project cash flow in Year 1 to provide for the production of initial product inventories. This was credited back to the project cash flow in Year 20 to reflect product inventory drawn down and recapture of receivable accounts.

Taxes

No allowance has been made for Federal and Provincial income tax.

The only taxes calculated in this analysis are with respect to B.C. Mineral Taxes and the expectation that there will be some form of regional tax despite the fact that the Prosperity Project is well removed from any municipalities.

No allowance has been made for GST or provincial sales tax.

Life-of-Mine Cash Flow

Life-of-Mine cash flow details are presented in Table 18-23. The base case cash flow has been estimated using the following assumptions:

- Unless stated otherwise, all values are expressed in CDN$.
- Unless stated otherwise, all values are estimated in 2nd third quarter 2007 dollars.
- No escalation is applied in the evaluation for inflation.
- Values are exchanged between currencies at the following rates: (CDN$1.00 = US$0.80).
- The project is evaluated on a stand-alone basis. No external corporate structures are considered.
- The project is assumed to be financed on a 100% equity basis and 0% debt.

In the base case on a pre-tax basis the project shows an IRR of 11.7% and an estimated pre-tax payback period of 8 years from a decision to proceed with the project or 6 years from start of production.

Table 18-24
Base Case Cash Flow

PERIOD	years	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	Total
PRODUCTION																								
Total Tonnes Mined	000's	2,747	6,411	31,870	56,098	62,304	64,841	66,878	70,635	71,897	75,820	71,774	71,059	65,484	43,826	36,225	27,621	27,595	26,172	6,822	0	0	0	886,080
Total Tonnes Milled	000's			9,491	25,560	25,560	25,560	25,560	25,560	25,560	25,560	25,560	25,560	25,560	25,560	25,560	25,560	25,560	25,560	25,560	25,560	25,560	17,218	486,789
Strip Ratio	W:O			2.4	1.2	1.4	1.5	1.6	1.8	1.8	2.0	1.8	1.8	1.6	0.7	0.4	0.1	0.1	0.0					0.82
Copper Grade	% Cu			0.23%	0.25%	0.24%	0.25%	0.22%	0.22%	0.24%	0.22%	0.21%	0.19%	0.19%	0.23%	0.22%	0.24%	0.27%	0.31%	0.21%	0.14%	0.14%	0.14%	0.219%
Gold Grade	g/t Au			0.457	0.489	0.489	0.508	0.467	0.456	0.504	0.444	0.412	0.399	0.421	0.448	0.393	0.447	0.506	0.527	0.371	0.267	0.267	0.267	0.429
Concentrator Recovery	% Cu			90%	90%	90%	90%	90%	90%	90%	90%	90%	89%	89%	90%	90%	90%	90%	91%	70%	70%	70%	70%	
	% Au			71%	72%	72%	73%	72%	71%	73%	71%	70%	69%	70%	71%	68%	71%	73%	74%	66%	56%	56%	56%	
Copper Production Pounds	000's			43,698	125,227	121,833	126,633	113,621	110,647	123,951	111,979	105,710	95,171	95,302	117,856	111,126	120,125	136,577	157,624	83,050	55,479	55,479	37,372	2,048,459
Gold Production Ounces	oz			99,333	291,161	291,152	305,340	274,610	266,788	302,796	258,064	235,608	226,152	242,126	260,984	221,127	260,666	304,122	320,299	202,332	123,273	123,273	83,039	4,692,247
REVENUE																								
Copper Price	US$/lb	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	
Gold Price	US$/oz	575	575	575	575	575	575	575	575	575	575	575	575	575	575	575	575	575	575	575	575	575	575	
Silver Price	US$/oz	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	8.00	
Cdn/US Exchange Rate	$US/$CDN	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80
Gross Copper Metal Value	$CDN 000's			78,650	225,583	219,384	228,151	204,402	198,979	223,253	201,407	189,984	170,791	171,031	212,127	199,852	216,266	246,117	284,164	149,289	99,067	99,067	66,734	3,684,296
Gross Gold Metal Value	$CDN 000's			69,610	204,040	204,034	213,977	192,442	186,960	212,194	180,847	165,110	158,483	169,678	182,893	154,962	182,670	213,123	224,460	141,791	86,387	86,387	58,192	3,288,239
Gross Silver Metal Value	$CDN 000's			2,046	5,743	5,640	5,785	5,382	5,286	5,704	5,329	5,122	4,769	4,773	5,516	5,301	5,587	6,208	7,090	4,005	3,088	3,088	2,080	97,541
Total Gross Revenue	**$CDN 000's**			**150,306**	**435,366**	**429,058**	**447,913**	**402,225**	**391,224**	**441,151**	**387,582**	**360,216**	**334,043**	**345,481**	**400,536**	**360,115**	**404,523**	**465,448**	**515,714**	**295,085**	**188,542**	**188,542**	**127,006**	**7,070,076**
COST SUMMARY																								
Operating Expenditures																								
Cost to Concentrate	$CDN 000's			83,219	144,148	156,064	165,757	174,220	175,981	178,075	186,581	192,332	190,050	200,274	170,386	163,083	149,158	151,159	148,968	118,136	112,728	112,015	75,943	3,048,278
Transport, Treatment and Refining	$CDN 000's			30,352	85,592	83,966	86,383	79,792	77,301	84,082	77,890	74,536	69,074	69,281	80,801	77,144	81,892	91,484	104,434	58,482	43,809	43,809	29,511	1,429,616
Total Operating Expenditures	**$CDN 000's**			**113,571**	**229,740**	**240,029**	**252,140**	**254,012**	**253,283**	**262,157**	**264,471**	**266,868**	**259,124**	**269,555**	**251,186**	**240,227**	**231,051**	**242,643**	**253,402**	**176,618**	**156,538**	**155,824**	**105,454**	**4,477,894**
Unit Costs																								
Site Unit Costs	$CDN/t milled			8.77	5.64	6.11	6.49	6.82	6.89	6.97	7.30	7.52	7.44	7.84	6.67	6.38	5.84	5.91	5.83	4.62	4.41	4.38	4.41	6.26
Cost to Concentrate	US $/lb Cu			1.52	0.92	1.02	1.05	1.23	1.27	1.15	1.33	1.46	1.60	1.68	1.16	1.17	0.99	0.89	0.76	1.14	1.63	1.62	1.63	$ 1.19
Transport, Treatment and Refining	US $/lb Cu			0.56	0.55	0.55	0.55	0.56	0.56	0.54	0.56	0.56	0.58	0.58	0.55	0.56	0.55	0.54	0.53	0.56	0.63	0.63	0.63	$ 0.56
By-Product Credits	US $/lb Cu			1.31	1.34	1.38	1.39	1.39	1.39	1.41	1.33	1.29	1.37	1.46	1.28	1.15	1.25	1.28	1.18	1.40	1.29	1.29	1.29	$ 1.32
Total Operating Costs (After BPC)	**US $/lb Cu**			**0.77**	**0.13**	**0.20**	**0.20**	**0.40**	**0.44**	**0.29**	**0.56**	**0.73**	**0.81**	**0.80**	**0.43**	**0.58**	**0.28**	**0.14**	**0.11**	**0.30**	**0.97**	**0.96**	**0.97**	**0.43**
NET OPERATING CASHFLOW																								
Operating Profit (EBITDA)	$CDN 000's			**36,736**	**205,626**	**189,029**	**195,773**	**148,213**	**137,942**	**178,994**	**123,111**	**93,348**	**74,918**	**75,926**	**149,350**	**119,888**	**173,472**	**222,806**	**262,311**	**118,467**	**32,004**	**32,718**	**21,552**	**2,592,182**
Amortization Expense	$CDN 000's			18,051	56,227	58,799	62,596	58,001	57,324	65,628	61,326	59,727	54,816	56,610	71,080	67,967	74,462	87,969	103,897	56,561	40,150	1,408	829	1,113,428
Earnings Before Taxes (EBIT)	$CDN 000's			18,685	149,399	130,230	133,177	90,212	80,617	113,366	61,785	33,622	20,102	19,316	78,269	51,921	99,010	134,837	158,414	61,905	-8,146	31,309	20,722	1,478,754
BC Mineral Taxes	$CDN 000's			900	5,100	4,700	4,900	3,700	3,400	4,500	3,100	2,300	1,900	1,900	3,700	3,000	4,300	5,600	6,600	3,000	800	800	500	64,700
Earnings	**$CDN 000's**			**17,785**	**144,299**	**125,530**	**128,277**	**86,512**	**77,217**	**108,866**	**58,685**	**31,322**	**18,202**	**17,416**	**74,569**	**48,921**	**94,710**	**129,237**	**151,814**	**58,905**	**-8,946**	**30,509**	**20,222**	**1,414,054**
Capital Expenditures																								
Sustaining and Other capital	$CDN 000's	9,089	16,400	3,509	5,882	5,801	6,657	9,469	7,928	7,343	5,219	5,654	8,246	8,756	3,761	3,245	732	7,090	947	1,389	1,284	1,984	1,323	121,708
Mining Equipment and Pre-Production	$CDN 000's	17,676	28,347	47,411	60,927	52,436	10,908	13,693	1,255	5,751	14,754	11,459	991	5,628	1,590	740	2,030	1,342	1,147	400	250	0	0	278,734
Concentrator and Infrastructure	$CDN 000's	174,467	336,751	173,504																				684,722
Total Capital	**$CDN 000's**	**201,232**	**381,499**	**224,423**	**66,809**	**58,237**	**17,565**	**23,162**	**9,183**	**13,094**	**19,973**	**17,113**	**9,238**	**14,384**	**5,351**	**3,985**	**2,762**	**8,432**	**2,094**	**1,789**	**1,534**	**1,984**	**1,323**	**1,085,164**
Working Capital				57,435																			-57,435	
Closure Funding				655	1,878	1,827	1,899	1,704	1,660	1,859	1,680	1,586	1,428	1,430	1,768	1,667	1,802	2,049	2,364	1,246	832	832	561	30,727
Project Cashflow																								
Net Cashflow	(000) $ Cdn	-201,232	-381,499	-246,678	131,839	124,264	171,408	119,647	123,699	159,540	98,359	72,350	62,353	58,213	138,531	111,236	164,608	206,725	251,253	112,432	28,839	29,101	76,603	1,411,591
Cumulative Free Cashflow	(000) $ Cdn	-201,232	-582,731	-829,409	-697,570	-573,306	-401,898	-282,251	-158,552	988	99,347	171,697	234,050	292,263	430,794	542,031	706,638	913,363	1,164,616	1,277,048	1,305,887	1,334,988	1,411,591	
Net Present Value at 7.5% Discount	**(000) $ Cdn**	**260,150**																						

Sensitivity Analysis

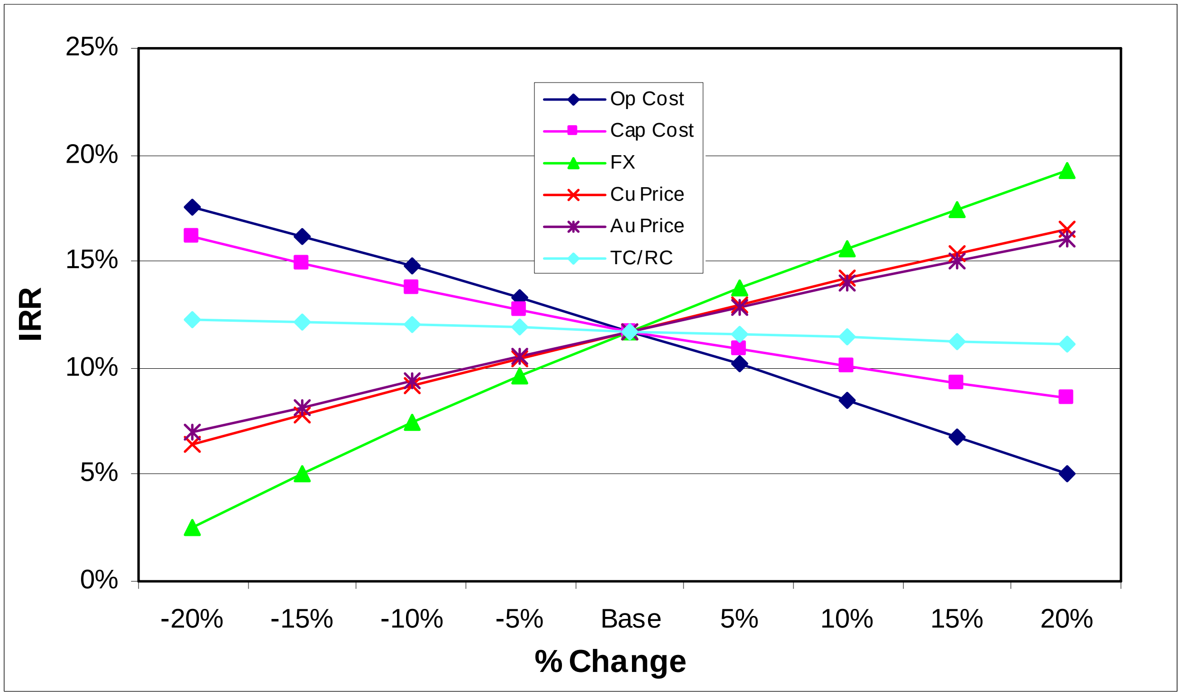
The relative sensitivity of the project to variations in revenue, capital cost and operating cost has been assessed by means of a "sensitivity analysis" which factors the above variables independently from 80% to 120% of their base case value.

The IRR and NPV sensitivity plots and tables in Figures 18-17 and 18-18 describe the relative impact of changes to the following major economic variables, namely:

Copper price;
Gold price;
Operating cost;
Capital cost;
Exchange rate; and
Smelter terms.

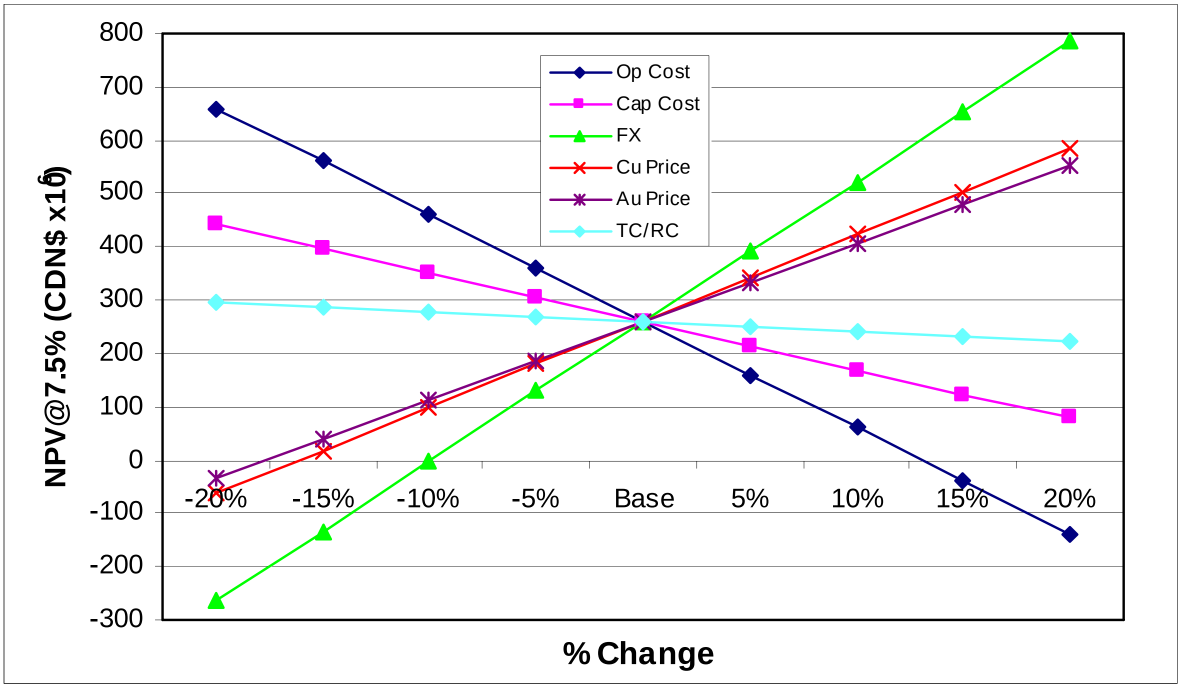
The project's economic viability as measured by IRR and NPV is most sensitive to the currency exchange rate variable followed by operating cost, and copper and gold price. The project appears to be least sensitive to variations in concentrate smelter terms and project capital cost.

Figure 18-17
IRR Sensitivity



IRR Sensitivity									
Variable	**-20%**	**-15%**	**-10%**	**-5%**	**Base**	**5%**	**10%**	**15%**	**20%**
Operating Cost	17.6%	16.2%	14.7%	13.3%	11.7%	10.2%	8.5%	6.8%	5.0%
Capital Cost	16.2%	14.9%	13.7%	12.7%	11.7%	10.9%	10.1%	9.3%	8.6%
FX	2.5%	5.1%	7.5%	9.7%	11.7%	13.7%	15.6%	17.5%	19.2%
Cu Price	6.4%	7.8%	9.2%	10.5%	11.7%	13.0%	14.2%	15.3%	16.5%
Au Price	6.9%	8.2%	9.4%	10.6%	11.7%	12.9%	14.0%	15.0%	16.1%
TC/RC's	12.3%	12.2%	12.0%	11.9%	11.7%	11.6%	11.4%	11.3%	11.1%

Figure 18-18
NPV Sensitivity



NPV @ 7.5%									
Variable	**-20%**	**-15%**	**-10%**	**-5%**	**Base**	**5%**	**10%**	**15%**	**20%**
Operating Cost	659	559	460	360	260	160	61	-39	-139
Capital Cost	442	397	351	306	260	215	169	124	78
FX	-265	-134	-2	129	260	391	523	654	785
Cu Price	-64	17	98	179	260	341	422	503	584
Au Price	-32	41	114	187	260	333	406	479	552
TC/RC's	298	288	279	270	260	251	241	232	222

19. Interpretations and Conclusions

The Prosperity Project is located 125 km southwest of the City of Williams Lake in the Cariboo-Chilcotin region of British Columbia, Canada.

Property and Access

The property is 100% owned by Taseko and is not subject to any royalties or carried interests. The mineral claims are currently in good standing until the year 2008. The property is located within territory that is the subject of an aboriginal Rights and Title case between the Tsilhqot'in National Government and the Province of British Columbia currently before the B.C. Supreme Court. The outcome and implications of this case are unknown.

Access and infrastructure is adequate for the development of a large scale open pit operation with existing road access to the property, confirmed technical viability of hydroelectric power within 120 km, adequate water available, and rail load-out services close to Williams Lake.

Exploration

Taseko carried out ongoing and systematic exploration programs on the Project from 1991 – 1999, increasing drilling to 150,090 m in 470 holes, outlining a large porphyry gold-copper deposit. The Company and its consultants also carried out progressive engineering, metallurgical and environmental studies.

Taseko re-initiated environmental and engineering work on the Prosperity Project in late 2005. No additional exploration or drilling was undertaken.

Geology and Resources

The geology of this porphyry-type gold-copper deposit is well understood. The deposit is oval in plan and is approximately 1500 m long, 800 m wide and extends to a maximum drilled depth of 880 m. Pyrite and chalcopyrite are the principal sulphide minerals in the deposit. They are uniformly distributed as disseminations, fracture-fillings and sub-vertical veinlets throughout the host volcanic and intrusive units in the deposit. Native gold occurs as inclusions in, and along microfractures with, copper-bearing minerals and pyrite.

Sampling, sample preparation, analysis and security meet industry standards. The results of the Taseko verification program indicate that the database is of good quality and acceptable for use in geological and resource modeling.

The resource modeling is well documented and the geostatistical analysis of data from the Prosperity Project Database supports the Measured and Indicated Mineral Resources listed in Table 19-1. These resources are inclusive of the stated mineral reserves.

Table 19-1
Prosperity Mineral Resources

at 0.14% Copper Cut-off			
Category	**Tonnes (millions)**	**Gold (gpt)**	**Copper (%)**
Measured	547.1	0.46	0.27
Indicated	463.4	0.34	0.21
Total	**1,010.5**	**0.41**	**0.24**

Feasibility Studies

Kilborn SNC Lavalin conducted a feasibility level study of the Prosperity Project in 2000. Taseko re-initiated work on the Prosperity Project in late 2005. A mill redesign and project cost review was completed by SNC Lavalin in 2006. Taseko utilized information from the 2000 feasibility level study, and the 2006 revised process design and scoping level capital and operating costs to prepare a pre-feasibility study. In October 2006 Taseko commissioned Hatch and Knight Piesold Consulting to carry out a 2007 Feasibility Study Update for the Prosperity Project which is the subject of this report.

The 2007 feasibility study was done using long term metal prices of US$1.50/lb for copper, US$575/oz for gold, US$8.00/oz for silver and an exchange rate of US$0.80/C$1.00.

Pre-Production and Mine Plan

The feasibility study incorporates activities during a pre-production period of two years which include construction of the electricity transmission line; upgrading and extension of current road access and mine site clearing; site infrastructure, processing, and tailings starter dam construction; removal and storage of overburden; and pre-production waste development.

The mine plan utilizes a large-scale conventional truck shovel open pit mining and milling operation. Following a one and a half-year pre-strip period, total material moved over years 1 through 16 averages 150,000 tonnes/day at a strip ratio of 1.2:1. A declining net smelter return cut-off is applied to the mill feed, which defers lower grade ore for later processing. The lower grade ore is recovered from stockpile for the final 3 years of the mine plan. The life of mine strip ratio including processing of lower grade ore is 0.8:1.

The 2000 pit optimization was found to be a valid ultimate shell with respect to encompassing reserves under 2007 economic assumptions and the mine design incorporates an appropriate level of detail with respect to design and operating parameters.

The pit wall slopes incorporated in the current design conform with all Knight Piesold recommendations with the exception of the northeast sector where Knight Piesold recommends single benching below the gypsum line to achieve an inter-ramp angle of 45 degrees. The current design incorporates double benching and an inter-ramp angle of 50 degrees. A preliminary review of this discrepancy indicates that while a revised ultimate pit shell reflecting this recommendation has the potential to result in a minor change in reserves and cash flow, it's impact will not be significant and is within the uncertainties inherent in any estimate.

Processing and Infrastructure

Exhaustive metallurgical test provided valid data to support the process design, recovery relationships and concentrate parameters. Processing incorporates proven processes and technologies.

The Prosperity processing plant has been designed with a nominal capacity of 70,000 tonnes/day. The plant consists of a single 12-m diameter semi-autogenous grinding (SAG) mill, two 7.9-m diameter ball mills, followed by processing steps that include bulk rougher flotation, regrinding, cleaner flotation, thickening and filtering to produce a copper-gold concentrate. Expected life of mine metallurgical recovery is 87% for copper and 70% for gold, with annual production averaging 107 million pounds copper and 247,000 ounces gold over the 20 year mine life.

The copper-gold concentrate would be hauled with highway trucks to an expanded load-out facility at McLeese Lake for rail transport to various points of sale, but mostly through the Port of North Vancouver for shipment to smelters/refineries around the world.

Power would be supplied via a new 124 km long, 230 kV transmission line from Dog Creek on the BC Hydro Grid. Infrastructure would also include the upgrade of sections of existing roads, construction of a short spur to the site, an on-site camp, equipment maintenance shop, administration office, warehouse, and explosives facilities.

The tailings storage facility is designed based on valid field data, current engineering standards, and has the capacity to contain the processed reserve and potentially acid generating materials identified in the mine plan.

Based on this study, the project would employ up to 450 permanent hourly and staff personnel. In addition, approximately 60 contractor personnel would be employed in areas including catering, concentrate haulage, explosives delivery, and bussing.

Key Results

- Pre-tax net present value of C$260 million at 7.5% discount rate
- Pre-tax internal rate of return of 12% with a 6 year payback from start of production
- 20 year mine life at a milling rate of 70,000 tonnes/day
- Life of mine strip ratio of 0.8:1
- Total pre-production capital cost of C$807 million
- Operating cost of C$6.26 per tonne milled over the life of mine
- Mine site production costs net of gold credits of US$0.43/lb Cu

The reserve estimate takes into consideration all geologic, mining, milling, and economic factors, and is stated according to Canadian standards (NI43-101).

The mineral reserves estimated from the study are as follows:

Table 19-2
Prosperity Mineral Reserves

at C$5.25 NSR/t Pit-Rim Cut-off					
Category	**Tonnes (millions)**	**Gold (gpt)**	**Copper (%)**	**Recoverable Gold Ounces (millions)**	**Recoverable Copper Pounds (billions)**
Proven	286	0.47	0.25	3.0	1.3
Probable	201	0.37	0.18	1.7	0.7
Total	**487**	**0.43**	**0.22**	**4.7**	**2.0**

20. Recommendations

An Environmental Assessment Certificate is currently being pursued under the harmonized British Columbia *Environmental Assessment Act* (EA Act) and *Canadian Environmental Assessment Act* (CEAA) review process. Based on the technical and economic viability of the project demonstrated in the feasibility study, this work should continue.

There may be an opportunity to improve the project economics through modifications in the processing flow sheet, allowing a coarser primary grind and/or staged regrinding. An investigation of this potential will require additional metallurgical testwork for which there is currently no sample material. A diamond drill program should be carried out to acquire core representative of approximately the first 7 years of mining with metallurgical testing to investigate the potential of this opportunity as a first step in further optimization of the process flow sheet. The assay results of this proposed drilling and the 1998 verification holes should be incorporated into the resource model.

The current ultimate pit shell, while adequate for definition of a reserve and a valid mine plan warrants a re-evaluation as an optimum pit using the outcomes of the 2007 feasibility study as input parameters. The orebody’s cylindrical shape, slow increase in grade with depth, and the fact that there is no additional geological information would suggest that there will be no significant change in the temporal distribution of head grade in the mine plan but the optimization exercise would bring the 2000 optimization process up to date. This should be completed after the incorporation of new assay into the resource model and will also provide the opportunity to incorporate the latest KP pit slope recommendations in the northeast sector.

A number of value engineering concepts were identified during the course of the feasibility study that warrant further investigation. These include but are not limited to:

- Primary Crusher Construction Methodology
- Camp & Administration Complex Optimization
- Flotation Cell Sizes

The tailings storage facility construction design was based on the use of only non acid generating material. There may be the opportunity to utilize some potentially acid generating (PAG) material in water saturated upstream sections of the dams. Future pit:dam construction material balances should investigate the opportunity to reduce the quantity of PAG requiring haulage requirements.

21. References

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22. Date and Signature Page

Signed at Vancouver, British Columbia on the 15th day of October 2007.

"Signed and Sealed"

Scott S. Jones, P.Eng.

Scott Jones, P.Eng.
Suite 1020-800 West Pender Street
Vancouver, BC V6C 2V6

I, Scott S. Jones, P.Eng., of Vancouver, British Columbia, hereby certify that:

1. I am an employee of Taseko Mines Ltd., with a business office at 1020-800 West Pender Street, Vancouver, British Columbia. In my position as General Manager, Project Development, on behalf of Taseko Mines Limited, I co-authored this technical report on the feasibility study for the Prosperity Project which was announced on September 24, 2007

2. This certificate applies to the technical report titled "Technical Report, Executive Summary of the Feasibility Study of the Prosperity Gold-Copper Project, British Columbia, Canada", dated October 15th, 2007.

3. I am a graduate of McGill University in Montreal, Quebec (B.Eng. Mining). I have practiced my profession for 22 years since graduation in 1985. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, license number 29486. As a result of my experience and qualifications, I am a Qualified Person under National Instrument 43-101.

4. I am responsible for the compilation of all sections of this report.

5. I am not independent of Taseko Mines Limited.

6. I have visited the Prosperity property on four occasions in 2006; May 25, July 27, August 16, and August 30, and four occasions in 2007; June 21 and 22, July 4, and July 10.

7. I have read National Instrument 43-101.

8. I, as of the date of the certificate and to the best of my knowledge and information, believe the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

9. I consent to the use of his Technical report for disclosure purposes of Taseko Mines Limited.

Signed at Vancouver, British Columbia on the 15th day of October 2007.

"Signed and Sealed"

Scott S. Jones, P.Eng.

Gary Giroux, PEng.
Suite 1215-675 West Hastings Street
Vancouver, BC V6B 1N2

I, G.H. Giroux, of 982 Broadview Drive, North Vancouver, British Columbia, do hereby certify that:

1) I am a consulting geological engineer with an office at #1215 - 675 West Hastings Street, Vancouver, British Columbia.
2) I am a graduate of the University of British Columbia in 1970 with a B.A. Sc. and in 1984 with a M.A. Sc., both in Geological Engineering.
3) I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia.
4) I have practiced my profession continuously since 1970. I have had over 30 years experience calculating mineral resources. I have previously completed resource estimations on a wide variety of porphyry deposits both in B.C. and around the world, many similar to Prosperity.
5) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in National Instrument 43-101.
6) This report titled "**Technical Report, Executive Summary of the Feasibility Study of the Prosperity Gold-Copper Project, British Columbia, Canada**", dated October 15th, 2007, is based on a study of the data and literature available on the Prosperity Property. I am responsible for the Resource Estimation Section completed in Vancouver during 1998 and amended 1999. I have not visited the property.
7) I have previously completed a resource estimation on this property in 1994.
8) 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.
9) I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.
10) I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 15th day of October, 2007

"signed and sealed"

G. H. Giroux, P.Eng., MASc.

Lawrence A. Melis, P.Eng.
Suite 100, 2366 Ave C North
Saskatoon SK S7L 5X5

I, Lawrence A. Melis, of 259 Egnatoff Cres., Saskatoon, Saskatchewan, do hereby certify that:

1) I am a consulting process engineer, working for Melis Engineering Ltd. with an office at 2366 Ave C North, Suite 100, Saskatoon, Saskatchewan, Canada.
2) I am a graduate of the University of Western Ontario in 1971 with a B.Sc. (Chemistry).
3) I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia (Registration No. 19398).
4) I have practiced my profession continuously since 1971. I have had over 35 years experience in process engineering for the mining industry.
5) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in National Instrument 43-101.
6) This certificate applies to the technical report titled "Technical Report, Executive Summary of the Feasibility Study of the Prosperity Gold-Copper Project, British Columbia, Canada", dated October 15th, 2007, and more specifically a review of the metallurgy section, Section 16.0, which was prepared by others based on metallurgical testwork completed in the 1990's by Melis Engineering Ltd. for which I was directly responsible.
6) I have visited the property in the 1990's to look at core and general site conditions.
8) As of the date of this certificate, to the best of my knowledge, information and belief, Section 16.0 of the technical report contains all scientific and technical information that is required to be disclosed to make the metallurgical component of the technical report not misleading.
9) I am independent of Taseko Mines Limited as defined by National Instrument 43-101.
10) I consent to the use of this Technical report for disclosure purposes of Taseko Mines Limited.

Dated this 15th day of October, 2007

"signed and sealed"

Lawrence A. Melis, P.Eng.