

TECHNICAL REPORT
on the
RED CHRIS COPPER-GOLD PROJECT

LIARD MINING DIVISION
Latitude 57° 42' North, Longitude 129° 47' West
NTS map sheet 104H/12W

For

Red Chris Development Company Ltd.
and
bcMetals Corporation

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- 1.0 List of drill holes used in the 2004 Resource Estimate
- 2.0 Review of Red Chris QA/QC Data by Dr. A.J. Sinclair

1.0 SUMMARY

This Technical Report on the Red Chris project, located in northwest British Columbia, has been prepared for Red Chris Development Company (“RCDC”) and bcMetals Corporation, Vancouver, B.C. as required under National Instrument NI 43-101. bcMetals is the 100% owner of RCDC, which in turn owns 100% of the Red Chris project.

- The Red Chris property is located about 18 km southeast of the village of Iskut and 80 km south of Dease Lake on the north-facing Todagin Plateau in northwestern British Columbia, Canada.
- The Red Chris property consists of 120 two-post, 8 fractional and 28 modified grid contiguous mineral claims for a total of 452 units. The total claim block covers approximately 110 square km.
- From 1968 to 1981, the property was explored by Conwest Exploration Ltd., Great Plains Development Co., Silver Standard Mines Ltd., Ecstall Mining Limited, and Texasgulf Canada Limited with geochemistry, geologic mapping, ground magnetics, induced polarization, trenching, and percussion and diamond drilling.
- During the 1994 field season, American Bullion completed mineral claim staking, land surveying, line cutting, soil geochemistry, geophysics (including magnetics, V.L.F. EM and induced polarization), camp and core logging facility construction, HQ and NQ diamond drilling totalling 21,417 m in 58 holes, core sample assaying, acid base accounting studies, base-line environmental studies, a mineral resource estimate, petrographic and metallurgical studies, and documentation. The programs were completed between June and November, 1994 at a cost of \$4.2 million.
- The 1995 American Bullion exploration program consisted of claim staking, survey control, soil geochemistry, geologic mapping, diamond drilling (112 holes totalling 36,770 m), geotechnical diamond drilling at three proposed tailings dam sites, acid base accounting analysis, baseline environmental studies, metallurgical studies, a resource estimate and a scoping study by Fluor Daniel Wright Ltd. American Bullion Minerals Ltd. reported the 1995 exploration program cost \$5.9 million.
- The property is situated regionally within the Stikinia Terrane of northern British Columbia. This terrane is dominated by Early Mesozoic and lesser Late Paleozoic island-arc volcanic strata and related subvolcanic intrusions that form a broad northwesterly trending belt along the centre of the province.
- The Red-Chris porphyry deposit (copper-gold mineralization), is distributed along the central axis of the pervasively altered and fractured Red Stock. At the Red-Chris deposit, the Red Stock is the predominant host of the mineralization. Mineralization and associated alteration are more intense adjacent to the ancestral en echelon fault system along the axis of the stock which controlled the emplacement of the stock and later

altering and mineralizing hydrothermal fluids – more typical of a shear-hosted copper-gold deposit. It has long been recognized that the Red-Chris copper-gold mineralization has good near-vertical and longitudinal continuity, controlled largely by post-mineral faulting superimposed on and along the ancestral, en echelon, central axis fault zone, commonly referred to as the ‘East Zone Fault’.

- It was recognized during early exploration of the Red Chris property that most of the mineralization is closely associated with individual and sheeted quartz (\pm carbonate) veining, and quartz (\pm carbonate) stockwork zones.
- Pyrite, chalcopyrite, and lesser bornite are the principal sulphide minerals of the Red-Chris deposit. Minor covellite occurs as inclusions in pyrite, and molybdenite, sphalerite and galena occur locally in trace amounts. Gold, second in economic importance to copper, occurs as electrum spatially- and genetically-associated with the copper mineralization.
- In 2003 RCDC conducted an infill drilling program consisting of 49 holes and 16,591 m targeting the core in East and Main Zones where open pit mining would be expected to take place. Based on this drilling, the Red Chris Measured, Indicated and Inferred Resources were updated and reported in the NI 43-101 Update Report dated February 16, 2004.
- The infill drill program completed in 2004 consisted of a total of 6,927 m in 25 diamond drill holes. Of these holes 10 targeted the Main Zone, 4 targeted the saddle zone between the Main and East zones, 6 tested the East zone and 5 condemnation holes were drilled to the north east of the East Zone.
- Based on the additional drilling completed in 2004, the geologic model, controlling the resource estimation at Red Chris, was adjusted from the 2003 interpretation. Within the Main Zone, the mineralized intrusive was modeled as a single unit with only post mineral faults segregated. Within the East Zone the outer core, and background intrusive domains were combined and joined to the Main Zone. The inner core remained the same as estimated in 2003. The Satellite Zone estimated in 2003 was renamed the East Zone Extension and remodeled based on 2004 drill hole data.
- The Main and East Zones were modeled and estimated separately for the various geologic domains. Blocks 20 x 20 x 15 m were estimated for copper and gold by ordinary kriging.
- The resource was classified by 43-101 standards as ‘measured’, ‘indicated’, and ‘inferred’ by using the relative kriging estimation variance. This system of classification involves both geologic and grade continuity, the number of composites found and the distance of blocks from drill holes.
- The resource estimate for all zones at a 0.2 % Cu cut-off reports 446.1 million tonnes measured plus indicated at an average grade of 0.36 % Cu and 0.29 g Au/t and an

additional 268.7 million tonnes inferred at an average grade of 0.30 % Cu and 0.27 g Au/t. In addition there are 116.0 million tonnes at an average grade of 0.32 % Cu and 0.30 g Au/t in the Far West and Gully Zones also in the Inferred resource category at 0.2 % Cu cutoff. This is unchanged from the 2003 Resource estimate. Additional drilling is required before this resource can be upgraded to either the Measured or Indicated category. However, the expectation is that this resource will extend the life of the project.

- The additional infill drilling completed during the 2004 field season has resulted in an increase in tonnes classed measured and indicated and a drop in inferred tonnes at grades similar to those estimated during the 2003 study.
- As shown by the mineral process testing conducted by Lakefield Research Limited during the period from November 1995 to February 1996 and subsequently by G&T Metallurgical Services Ltd. in 2004, the mineralization in the Red Chris deposit responds well to processing by conventional crushing, grinding and flotation to produce a commercial grade copper-gold concentrate. The recovery of copper increases directly with head grade. The recovery of copper and gold values was consistently in excess of eighty-seven percent for copper and fifty percent for gold.
- Over the course of studies on the project, possible operations of various sizes, ranging up to 90,000 t/d, have been examined. All of these studies have been based on open pit mining of at least the upper portion of the mineral deposits. The 1998 American Bullion Study proposed a different approach to mining of the East and Main zones, with the use of underground workings to excavate material from the open pit. The underground development, with raises from a main haulage level at the 1200m elevation up to the open pit, connect the mining operation to the plant site located in the valley below the mineral deposits. This approach was not considered viable by RCDC and was discarded from further consideration. With the data and information collected for rock characteristics and structural geology, the foundation is established for open pit design and the operating parameters.
- During the 2003 field season, additional geotechnical and hydrogeological data were collected for analysis in the Feasibility Study which commenced in mid January, 2004. This included additional test pitting and drilling on a potential tailings dam site and orientated core drilling for slope stability analysis. Additional base line studies were conducted in the fall of 2003 and included water quality, hydrology, fish habitat assessment, and wildlife surveys. Kinetic tests, to better understand and quantify the acid producing potential of various rock types in the area were commenced late 2003 and are ongoing. More studies were conducted in the summer of 2004 to fill identified gaps in the base line data including Terrestrial Eco-System mapping, archaeology, traditional use surveys, water quality, vegetation, metals and soils survey. Also, during the winter of 2004, additional snow course studies were undertaken.
- On January 19, 2004, the Company executed a Memorandum of Understanding with the Tahltan First Nation, the Iskut First Nation, and the Tahltan Joint Council. This

document sets out the principals for joint cooperation on the project development and operation between the Company and the First Nations.

- The additional drill information collected in 2003 and 2004 has added confidence and moved a significant tonnage into the measured plus indicated category for the Red Chris project. Based on the positive results from that Resource update, this project moved directly into a full Feasibility Study. The company awarded the Red Chris Feasibility Study to AMEC E&C Services Limited and reported the results in a News Release dated November 24, 2004.

- This Technical Report presents the following:
 1. The updated Measured, Indicated and Inferred Resources for Red Chris following the summer 2004 drilling program.

 2. The results of the Red Chris Feasibility Study Report based on conventional open pit mining, a 30,000 tpd flotation mill with the shipment of concentrates to Pacific rim smelters from the B.C. port of Stewart. The results presented herein include Proven and Probable Reserves, metallurgical performance, capital and operating cost estimates and financial analysis as performed and consolidated by AMEC E&C Services (“AMEC”), Vancouver, B.C. in the Red Chris Feasibility Study with contributions from various independent consultants as follows:
 - Giroux Consultants: preparation of the resource estimate.
 - Nilsson Mine Services Limited: preparation of Proven and Probable Reserve estimate, preparation of LOM mine plan, production schedule and mine capital and operating cost estimates.
 - Merit Consultants International Inc.: estimation and consolidation of the project capital costs.
 - G&T Metallurgical Services Limited: metallurgical test work.

2.0 INTRODUCTION AND TERMS OF REFERENCE

bcMetals Corporation acquired the Red Chris property in 2003 by completing the purchase of Red Chris Development Company Ltd. with its only material asset, the Red Chris mineral property interest. The 2003 fall drilling program was designed to confirm and update the resource model of the East and Main Zones generated from 71,472 metres of diamond drilling conducted over the period from 1974 to 1995. An updated resource was produced in November, 2004 to incorporate 2004 drill results. A total of 25 diamond drill holes were completed on the Main and East Zones during the summer of 2004 (holes numbered 04-296 to 04-320). Of these holes 10 targeted the Main Zone, 4 targeted the saddle zone between the Main and East Zones, 6 tested the East Zone and 5 condemnation holes were drilled to the north east of the East Zone. This new resource model will be used to complete a Feasibility Study which is forecast to be completed in early 2005.

Portions of this report relating to property description, work history, regional and local geology, have been taken from the American Reserve Energy Corporation, November 18, 2002 report by G.H. Giroux, J.D Blanchflower, and R. Rodger. Doug Blanchflower, P. Geo., who managed the 1994-1995 programs at Red Chris for American Bullion Minerals Ltd., completed the above sections in this report which was an accepted NI 43-101 report.

David Tenney, C.Eng. managed the summer 2004 drilling program on the Red Chris project and completed the sections on drilling, sampling method and approach, sample preparation, analysis, and security in this document

Gary Giroux, P. Eng., completed Resource estimations for American Bullion in 1994, 1996, and 1998 and for bcMetals in 2003 and has compiled the resource sections in this December 16, 2004 report.

Jay Collins, P.Eng., completed the Capital Cost Estimate for bcMetals in 2004 and compiled the Basis of Estimate in this report.

William Colquhoun, Pr Eng., was responsible for preparation of the section regarding Mineral Processing and Metallurgical Testing.

John Nilsson, P.Eng., was responsible for the open pit mine design and production schedule. This includes equipment requirements and operating cost development.

Ian Smith, B.E. (Mining), C.P. Man, reviewed the section "Additional Requirements for Technical Reports on Development Properties and Production Properties".

Michael Redfearn, P.Eng., was responsible for compilation and assembly of this 2004 Technical Report, including the recent submissions provided by the five independent Qualified Persons.

All references to currency are in Canadian dollars, unless specified otherwise.

3.0 DISCLAIMER

Not applicable.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Red Chris property is located about 20 km southeast of the village of Iskut and 80 km south of Dease Lake on the north-facing Todagin Plateau between Ealue and Kluea Lakes in north western British Columbia, Canada. The property is located within the designated area for mineral resource development in the Cassiar Iskut-Stikine Land and Resource Management Plan. A deep sea port is situated at Stewart, about 322 km to the south by road including 23 km of yet to be constructed mine access road (see Figure 1). The property is centred on latitude 57° 42' North, longitude 129° 47' West within NTS map sheet 104H/12W, Liard Mining Division.

Figure 1 - Location Map

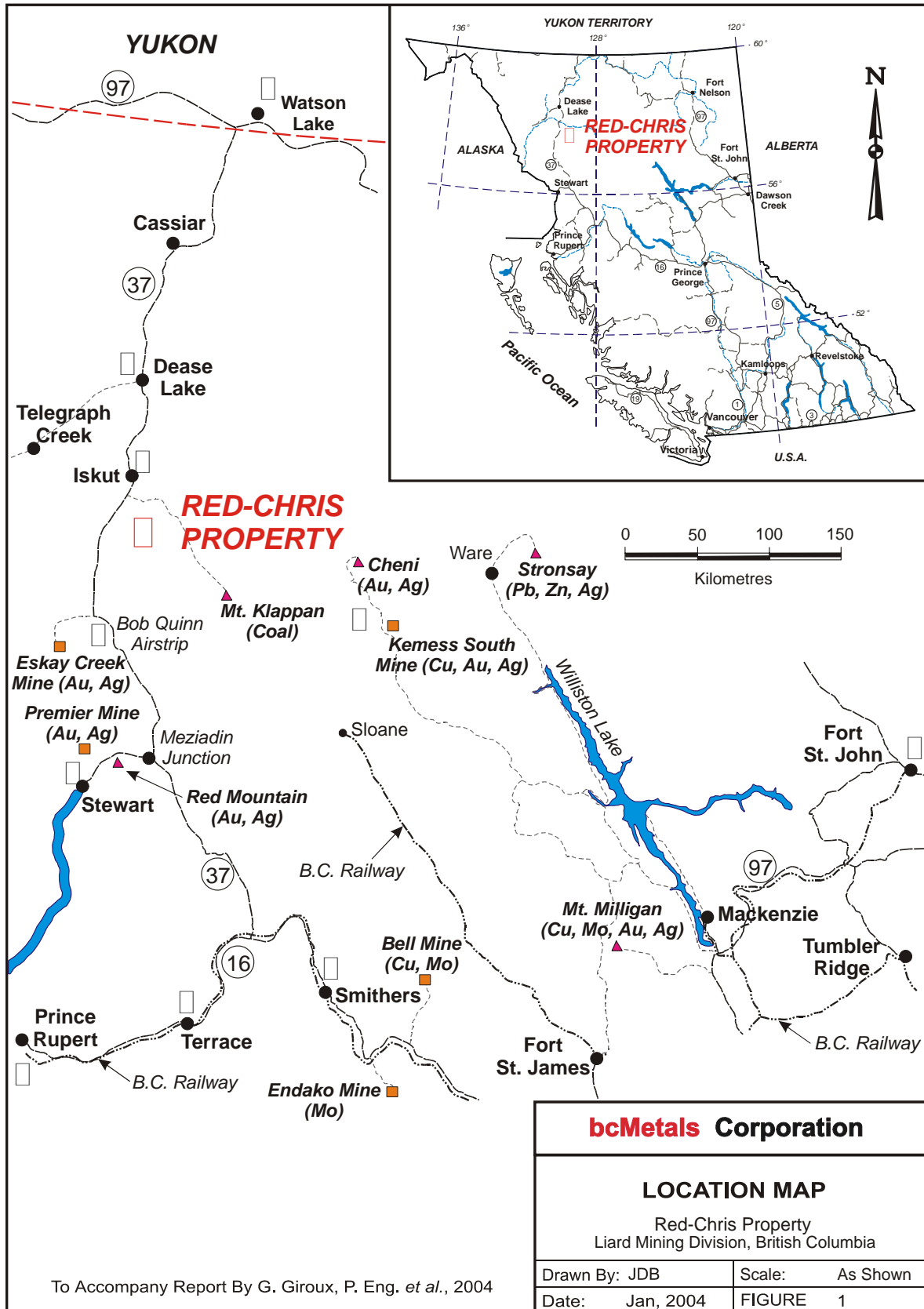
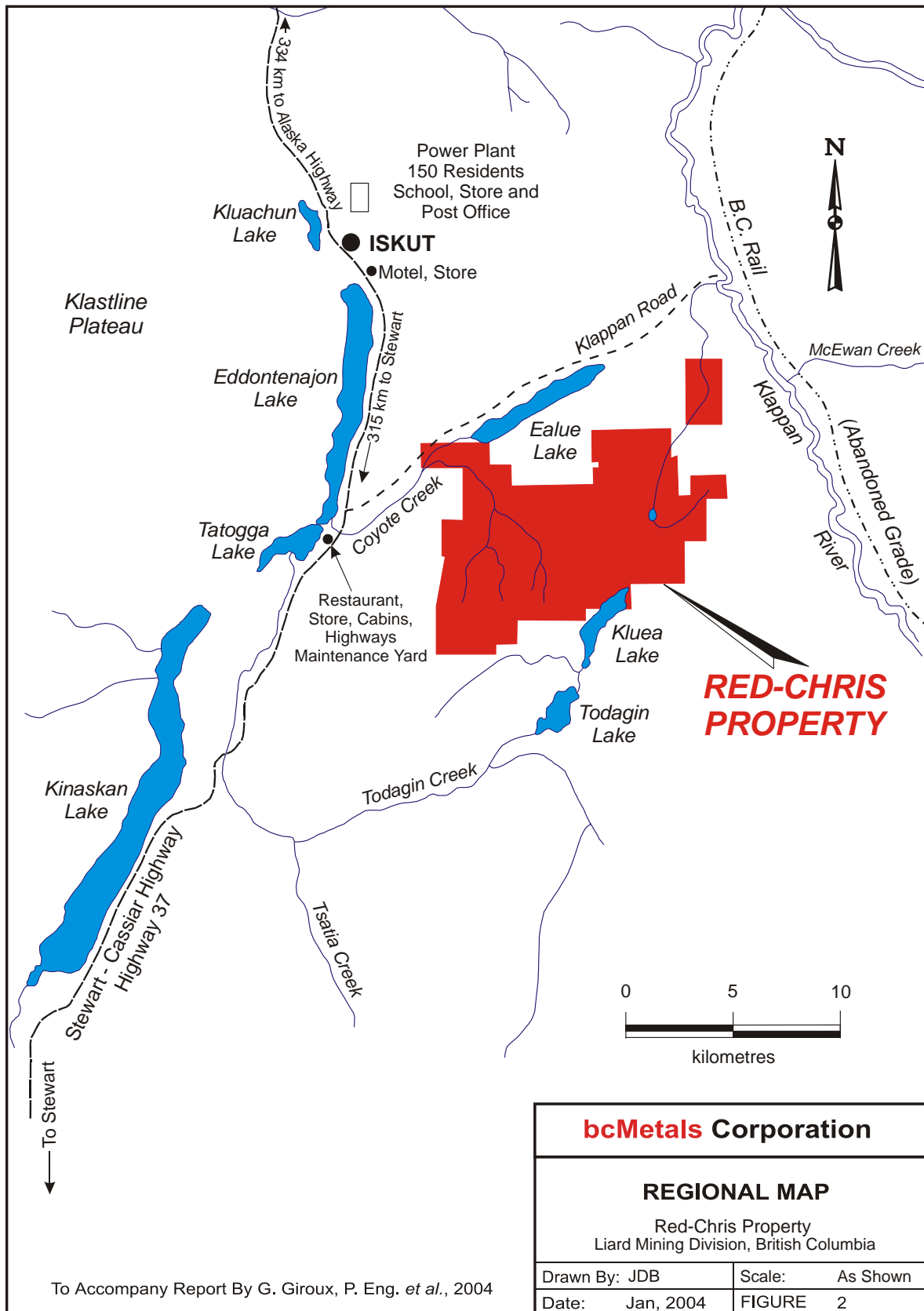


Figure 2 - Regional Map



To Accompany Report By G. Giroux, P. Eng. *et al.*, 2004

bcMetals Corporation	
REGIONAL MAP	
Red-Chris Property Liard Mining Division, British Columbia	
Drawn By: JDB	Scale: As Shown
Date: Jan, 2004	FIGURE 2

4.2 Claim Information (Mineral Tenure)

The Red Chris property consists of 120 two-post, 8 fractional and 28 modified grid contiguous mineral claims for a total of 452 units (see Table 1 and Figure 3). The total claim block covers approximately 110 square km. While limited drill hole surveys have been completed, no legal property-wide survey has been conducted. The core, two-post mineral claims were surveyed by McElhanney Associates in 1974 for the then owner Texasgulf Inc. McElhanney Consulting Services Ltd, while in the process of surveying in the 2003 drill hole collars for RCDC, re-established some of the original claim survey controls and subsequently integrated this 1974 survey into the 2003 digital format property controls.

Table 1 - Mineral Claim Information

Claim Number	UNITS	Record No.	Tenure No.	Record Date	Expiry Date
ABM-1	18	227107	330898	Sep 11, 1994	September 11, 2008
ABM-2	6	227108	330899	Sep 11, 1994	September 11, 2008
ABM-3	9	227109	330900	Sep 11, 1994	September 11, 2008
ABM-4	20	227196	330901	Sep 12, 1994	September 12, 2008
ABM-5	12	227197	330902	Sep 13, 1994	September 13, 2008
ABM-6	20	213345	330903	Sep 13, 1994	September 13, 2008
ABM-7	10	227214	337486	Jun 29, 1995	June 29, 2008
ABM-8	10	227215	337810	Jul 4, 1995	July 4, 2008
ABM-9	18	227216	337487	Jul 1, 1995	July 1, 2008
ABM-10	12	227217	337811	Jul 7, 1995	July 7, 2008
ABM-11	6	203587	337812	Jul 8, 1995	July 8, 2008
Capricorn	12	146	221682	July 7, 1976	July 7, 2008
Chris North	4	32	221642	Aug 13, 1975	August 13, 2008
Chris 01	1	31156	226748	Aug 24, 1968	August 24, 2008
Chris 02	1	31157	226749	Aug 24, 1968	August 24, 2008
Chris 03	1	31158	226750	Aug 24, 1968	August 24, 2008
Chris 04	1	31159	226751	Aug 24, 1968	August 24, 2008
Chris 05	1	31160	226752	Aug 24, 1968	August 24, 2008
Chris 06	1	31161	226753	Aug 24, 1968	August 24, 2008
Chris 07	1	31162	226754	Aug 24, 1968	August 24, 2008
Chris 08	1	31163	226755	Aug 24, 1968	August 24, 2008
Chris 09	1	31164	226756	Aug 24, 1968	August 24, 2008
Chris 10	1	31165	226757	Aug 24, 1968	August 24, 2008
Chris 11	1	31166	226758	Aug 24, 1968	August 24, 2008
Chris 12	1	31167	226759	Aug 24, 1968	August 24, 2008
Chris 13	1	31168	226760	Aug 24, 1968	August 24, 2008
Chris 14	1	31169	306684	Aug 24, 1968	August 24, 2008
Chris 15	1	31170	226761	Aug 24, 1968	August 24, 2008
Chris 16	1	31171	226762	Aug 24, 1968	August 24, 2008
Chris 17	1	31172	226763	Aug 24, 1968	August 24, 2008
Chris 18	1	31173	226764	Aug 24, 1968	August 24, 2008
Chris 19	1	31174	226765	Aug 24, 1968	August 24, 2008
Chris 20	1	31175	226766	Aug 24, 1968	August 24, 2008
Chris 21	1	31176	226767	Aug 24, 1968	August 24, 2008

Claim Number	UNITS	Record No.	Tenure No.	Record Date	Expiry Date
Chris 22	1	31177	226768	Aug 24, 1968	August 24, 2008
Chris 23	1	31178	226769	Aug 24, 1968	August 24, 2008
Chris 24	1	31179	226770	Aug 24, 1968	August 24, 2008
Cougar 1 FR	1	71985	228048	Aug 29, 1974	August 29, 2008
Cougar 2 FR	1	71986	228049	Aug 29, 1974	August 29, 2008
Cougar 3 FR	1	71987	228050	Aug 29, 1974	August 29, 2008
Cougar 4 FR	1	71988	228051	Aug 29, 1974	August 29, 2008
Cougar 5 FR	1	71989	228052	Aug 29, 1974	August 29, 2008
Cougar 6 FR	1	72180	228060	Aug 29, 1974	August 29, 2008
Cougar 7 FR	1	71990	228053	Aug 29, 1974	August 29, 2008
Cougar 8 FR	1	71991	228054	Aug 29, 1974	August 29, 2008
Money 01	1	34011	226792	Sep 30, 1968	September 30, 2008
Money 02	1	34012	226793	Sep 30, 1968	September 30, 2008
Money 03	1	34013	226794	Sep 30, 1968	September 30, 2008
Money 04	1	34014	226795	Sep 30, 1968	September 30, 2008
Money 05	1	34015	226796	Sep 30, 1968	September 30, 2008
Money 06	1	31016	226797	Sep 30, 1968	September 30, 2008
Money 07	1	34017	226798	Sep 30, 1968	September 30, 2008
Money 08	1	34018	226799	Sep 30, 1968	September 30, 2008
Money 09	1	34019	226800	Sep 30, 1968	September 30, 2008
Money 10	1	34020	226801	Sep 30, 1968	September 30, 2008
Money 11	1	34021	226802	Sep 30, 1968	September 30, 2008
Money 12	1	34022	226803	Sep 30, 1968	September 30, 2008
Money 13	1	34023	226804	Sep 30, 1968	September 30, 2008
Money 14	1	34024	226805	Sep 30, 1968	September 30, 2008
Money 15	1	34025	226806	Sep 30, 1968	September 30, 2008
Money 16	1	34026	226807	Sep 30, 1968	September 30, 2008
Money 17	1	34027	226808	Sep 30, 1968	September 30, 2008
Money 18	1	34028	226809	Sep 30, 1968	September 30, 2008
Money 19	1	34029	226810	Sep 30, 1968	September 30, 2008
Money 20	1	34030	226811	Sep 30, 1968	September 30, 2008
Money 21	1	34031	226812	Sep 30, 1968	September 30, 2008
Money 22	1	34032	226813	Sep 30, 1968	September 30, 2008
Money 23	1	34033	226814	Sep 30, 1968	September 30, 2008
Money 24	1	34034	226815	Sep 30, 1968	September 30, 2008
Money 25	1	34035	226816	Sep 30, 1968	September 30, 2008
Money 26	1	34036	226817	Sep 30, 1968	September 30, 2008
Money 27	1	34037	226818	Sep 30, 1968	September 30, 2008
Money 28	1	34038	226819	Sep 30, 1968	September 30, 2008
Money 29	1	34039	226820	Sep 30, 1968	September 30, 2008
Money 30	1	34040	226821	Sep 30, 1968	September 30, 2008
Money 32	1	34042	226822	Sep 30, 1968	September 30, 2008
Money 34	1	34044	226823	Sep 30, 1968	September 30, 2008
Money 36	1	34046	226824	Sep 30, 1968	September 30, 2008
Money 38	1	34048	226825	Sep 30, 1968	September 30, 2008
Money 40	1	34050	226826	Sep 30, 1968	September 30, 2008
Money 41	1	34051	226827	Sep 30, 1968	September 30, 2008

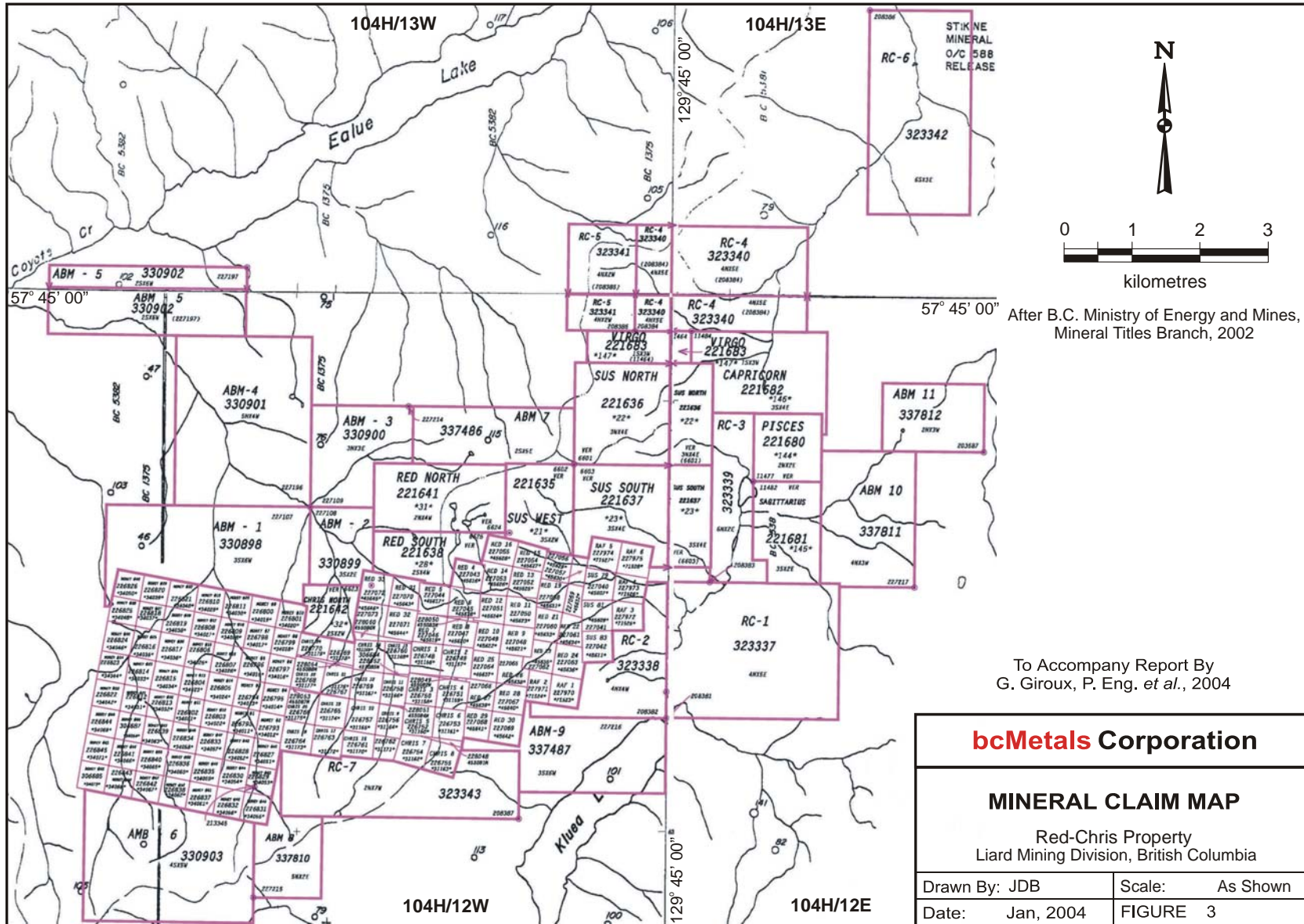
Claim Number	UNITS	Record No.	Tenure No.	Record Date	Expiry Date
Money 42	1	34052	226828	Sep 30, 1968	September 30, 2008
Money 43	1	34053	226829	Sep 30, 1968	September 30, 2008
Money 44	1	34054	226830	Sep 30, 1968	September 30, 2008
Money 45	1	34055	226831	Sep 30, 1968	September 30, 2008
Money 46	1	34056	226832	Sep 30, 1968	September 30, 2008
Money 47	1	34057	226833	Sep 30, 1968	September 30, 2008
Money 48	1	34058	226834	Sep 30, 1968	September 30, 2008
Money 49	1	34059	226835	Sep 30, 1968	September 30, 2008
Money 50	1	34060	226836	Sep 30, 1968	September 30, 2008
Money 51	1	34061	226837	Sep 30, 1968	September 30, 2008
Money 52	1	34062	226838	Sep 30, 1968	September 30, 2008
Money 53	1	34063	226839	Sep 30, 1968	September 30, 2008
Money 54	1	34064	306687	Sep 30, 1968	September 30, 2008
Money 55	1	34065	226840	Sep 30, 1968	September 30, 2008
Money 56	1	34066	226841	Sep 30, 1968	September 30, 2008
Money 57	1	34067	226842	Sep 30, 1968	September 30, 2008
Money 58	1	34068	226843	Sep 30, 1968	September 30, 2008
Money 59	1	34069	226844	Sep 30, 1968	September 30, 2008
Money 61	1	34071	226845	Sep 30, 1968	September 30, 2008
Money 63	1	34073	306685	Sep 30, 1968	September 30, 2008
Pisces	4	144	221680	July 7, 1974	July 7, 2008
Raf 1	1	71523	227970	July 31, 1974	July 31, 2008
Raf 2	1	71525	227971	July 31, 1974	July 31, 2008
Raf 3	1	71524	227972	July 31, 1974	July 31, 2008
Raf 4	1	71526	227973	July 31, 1974	July 31, 2008
Raf 5	1	71527	227974	July 31, 1974	July 31, 2008
Raf 6	1	71528	227975	July 31, 1974	July 31, 2008
RC-1	20	323337	323337	Jan 11, 1994	January 11, 2006
RC-2	16	323338	323338	Jan 14, 1994	January 14, 2005
RC-3	12	32339	323339	Jan 12, 1994	January 12, 2008
RC-4	20	323340	323340	Jan 17, 1994	January 17, 2008
RC-5	8	323341	323341	Jan 16, 1994	January 16, 2008
RC-6	18	323342	323342	Jan 18, 1994	January 18, 2005
RC-7	14	323343	323343	Jan 18, 1994	January 18, 2008
Red North	8	31	221641	Aug 13, 1975	August 13, 2008
Red South	8	28	221638	Aug 13, 1975	August 13, 2008
Red 04	1	45616	227043	Aug 5, 1970	August 5, 2008
Red 05	1	45617	227044	Aug 5, 1970	August 5, 2008
Red 06	1	45618	227045	Aug 5, 1970	August 5, 2008
Red 07	1	45619	227046	Aug 5, 1970	August 5, 2008
Red 08	1	45620	227047	Aug 5, 1970	August 5, 2008
Red 09	1	45621	227048	Aug 5, 1970	August 5, 2008
Red 10	1	45622	227049	Aug 5, 1970	August 5, 2008
Red 11	1	45623	227050	Aug 5, 1970	August 5, 2008
Red 12	1	45624	227051	Aug 5, 1970	August 5, 2008
Red 13	1	45625	227052	Aug 5, 1970	August 5, 2008
Red 14	1	45626	227053	Aug 5, 1970	August 5, 2008

Claim Number	UNITS	Record No.	Tenure No.	Record Date	Expiry Date
Red 15	1	45627	227054	Aug 5, 1970	August 5, 2008
Red 16	1	45628	227055	Aug 5, 1970	August 5, 2008
Red 17	1	45629	227056	Aug 5, 1970	August 5, 2008
Red 18	1	45630	227057	Aug 5, 1970	August 5, 2008
Red 19	1	45631	227058	Aug 5, 1970	August 5, 2008
Red 20	1	45632	227059	Aug 5, 1970	August 5, 2008
Red 21	1	45633	227060	Aug 5, 1970	August 5, 2008
Red 22	1	45634	227061	Aug 5, 1970	August 5, 2008
Red 23	1	45635	227062	Aug 5, 1970	August 5, 2008
Red 24	1	45636	227063	Aug 5, 1970	August 5, 2008
Red 25	1	45637	227064	Aug 5, 1970	August 5, 2008
Red 26	1	45638	227065	Aug 5, 1970	August 5, 2008
Red 27	1	45639	227066	Aug 5, 1970	August 5, 2008
Red 28	1	45640	227067	Aug 5, 1970	August 5, 2008
Red 29	1	45641	227068	Aug 5, 1970	August 5, 2008
Red 30	1	45642	227069	Aug 5, 1970	August 5, 2008
Red 31	1	45643	227070	Aug 5, 1970	August 5, 2008
Red 32	1	45644	227071	Aug 5, 1970	August 5, 2008
Red 33	1	45645	227072	Aug 5, 1970	August 5, 2008
Red 34	1	45646	227073	Aug 5, 1970	August 5, 2008
Sagittarius	6	145	221681	July 7, 1976	July 7, 2008
Sus North	12	22	221636	July 15, 1975	July 15, 2008
Sus South	12	23	221637	July 15, 1975	July 15, 2008
Sus West	6	21	221635	July 15, 1975	July 15, 2008
Sus 79	1	45607	227040	Aug 5, 1970	August 5, 2008
Sus 81	1	45609	227041	Aug 5, 1970	August 5, 2008
Sus 83	1	45611	227042	Aug 5, 1970	August 5, 2008
Virgo	3	147	221683	July 7, 1976	July 7, 2008

Totals

156 Claims 452 Units

Figure 3 - Mineral Claim Map



bcMetals Corporation

MINERAL CLAIM MAP

Red-Chris Property
Liard Mining Division, British Columbia

Drawn By: JDB	Scale: As Shown
Date: Jan, 2004	FIGURE 3

4.3 Permits & Agreements

The property is owned as to 80% by Red Chris Development Company (RCDC), a wholly owned subsidiary of bcMetals Corporation. Pursuant to an Option Agreement dated November 4, 2003 and amended November 26, 2003, RCDC has an option on Teck Cominco Limited's 20 % interest in the property. RCDC holds 30% of its 80% property interest in trust on behalf of American Bullion Minerals Ltd. (ABM).

By way of an agreement dated Oct. 18, 2002, the private company RCDC formed a joint venture (the "JV Agreement") with ABM such that:

- RCDC shall be owner of 70% interest in the JV while ABM will own 30%
- RCDC will be the operator
- RCDC pays American Bullion \$2,000,000 in cash in staged payments.

As a result of acquiring the option to Teck Cominco's interest, RCDC's cash payments to ABM were reduced to a total of \$1,625,000; payable as follows:

- 1 \$25,000 on signing the JV agreement; (paid);
- 2 \$225,000 on completion of a 60 day due diligence study; (paid);
- 3 \$500,000 on receipt of all necessary regulatory and shareholder approvals for the acquisition and the proposed assignment of this JV agreement ("the Approval Date" – subsequently determined to be August 20, 2003);
- 4 \$562,500 on the first anniversary of the Approval Date;
- 5 \$62,500* on the second anniversary of the Approval Date;
- 6 \$62,500* on the third anniversary of the Approval Date;
- 7 \$62,500* on the fourth anniversary of the Approval Date;
- 8 \$62,500* on the fifth anniversary of the Approval Date;
- 9 \$62,500* on the sixth anniversary of the Approval Date.

subject to an overriding requirement that the first four payments be made within 18 months of the date of the agreement, or April 18, 2004.

- * The above cash commitments were reduced by \$62,500 due to bcMetals' subsequent Option to purchase Teck Cominco Limited's interest in the property.

ABM's 30% of the JV constitutes a reversionary carried ownership interest ("RCOI") with the following terms:

- a) ABM shall receive payment under the RCOI after commercial production on the property and after RCDC has been repaid in full for all of its costs incurred on or in connection with the Property;
- b) Notwithstanding a) above, the parties recognize that ABM shall be entitled to be repaid for the \$10,000,000 it has expended on the Property and ABM shall be entitled to receive repayment of the \$10,000,000 out of commercial production from the Property, on a pro rata basis to the costs incurred by RCDC, concurrently with and on the same basis as RCDC is

- repaid for its costs;
- c) after commencement of commercial production, the RCOI shall be a 30% Working Interest in RCDC's interest;
 - d) the ROCI shall be subject to and, after commencement of commercial production on the Property, ABM shall be liable for its costs and pro rata portion of the following:
 - i) the Falconbridge Royalty;* and
 - ii) the Teck Cominco Rights**
 - e) American Bullion has the right to register its RCOI interest in the Property at any time;
 - f) ABM shall be entitled to elect to receive in kind its pro rata portion of minerals produced from the Property;
 - g) RCDC has a first right of refusal in respect to the RCOI.

* bcMetals Corporation has an option agreement with Falconbridge that allows bcMetals to purchase the 0.8% portion of the 1.8% gross overriding Falconbridge Royalty with a payment of \$1,000,000.

bcMetals has entered into an agreement to purchase the final outstanding 0.571428% carried interest owned by one of the original staking prospectors to the Red Chris claims, Mr. Jim McAusland.

** bcMetals has, pursuant to the option agreement described below, purchased the Teck Cominco Rights.

Pursuant to an Option Agreement dated November 4, 2003 and amended November 26, 2003 between bcMetals Corporation, RCDC (its wholly owned subsidiary), and Teck Cominco Limited; the Company has been granted an option to acquire all of Teck's ownership, rights and interest in and to the Red Chris porphyry copper-gold project. Teck held a 10% working interest and a 10% carried interest in the property as well as back in rights to 43.75% of RCDCs' interest.

To exercise the option Red Chris Development Company must:

1. pay \$300,000 in cash; (paid)
2. issue 250,000 share purchase warrants exercisable into 250,000 common shares of the Company at \$0.60 per share until December 31, 2006; (issued)
3. issue 500,000 common shares and 500,000 share purchase warrants on or before March 31, 2004;
4. issue 500,000 common shares and 500,000 share purchase warrants on or before July 31, 2004;
5. issue 500,000 common shares and 250,000 share purchase warrants on or before December 31, 2004; and
6. pay \$1,000,000 within one year of commencement of commercial production on the property.

The share purchase warrants have a three year term from the date of issuance and shall be exercisable at a price equal to the greater of (a) \$0.60 and (b) the average closing price of the Company's shares on the 10 trading days preceding the date of issuance of each respective

tranche of warrants, plus a 20% premium in the first year, a 40% premium in the second year, and a 50% premium in the third year.

Upon exercise of the Option, Red Chris Development Corporation will be the 100% owner of the Red Chris property, subject to American Bullion's 30% reversionary interest which becomes effective after RCDC has recovered 100% of costs from production.

5.0 ACCESSIBILITY, CLIMATE & PHYSIOGRAPHY

5.1 Accessibility

The property is helicopter accessible from Dease Lake or from several landing sites along the Stewart-Cassiar Highway (Highway 37) which is 12 km to the west of the claims centre (see Figure 2). Dease Lake is regularly serviced by scheduled commercial airline flights from either Smithers or Terrace. There is also a gravel airstrip 2 km north of the village of Iskut that could handle DC-3 aircraft. Several resorts and motels are situated along Highway 37 between Iskut and Tatogga Lake, which provide seasonal accommodation and meals.

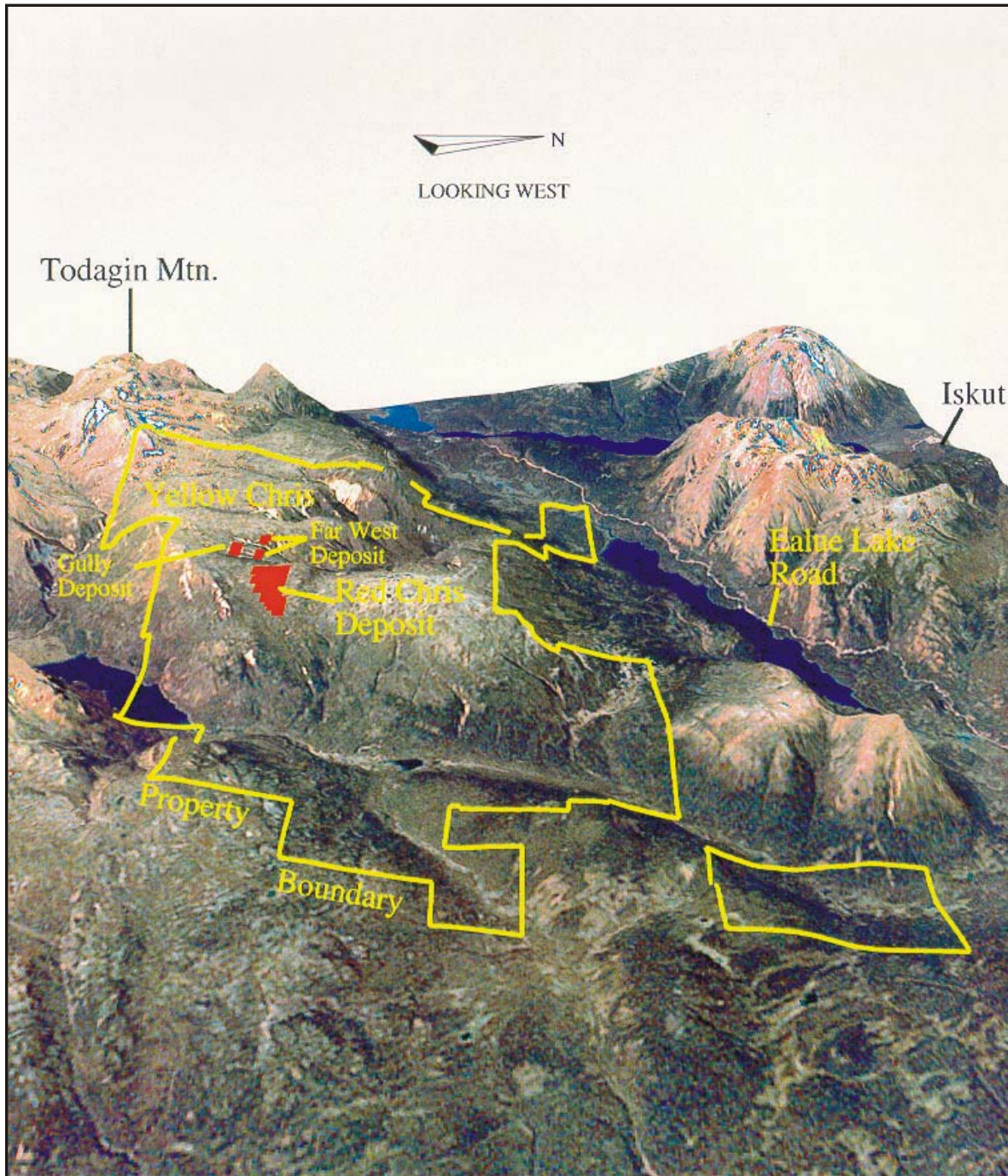
A rough tote road to the camp area leaves the Klappan Road just west of Ealue Lake. The road was constructed by Texasgulf Inc. in the 1970's and was utilized by American Bullion in the 1990's and by Red Chris Development Company in 2003 to haul drilling rigs to the project site. Because of sensitive drainage crossings and the steepness of the road, project permitting allowed only a single movement of equipment at the commencement and termination of the 2003 drilling project.

5.2 Physiography and Climate (Blanchflower *et al.*, (2002))

The Red Chris property is situated on the eastern portion of the Todagin upland plateau which forms a subdivision of the Klastine Plateau along the northern margin of the Skeena Mountains. Elevations on the property are typically $1,500 \pm 30$ m with relatively flat topography broken by several deep creek gullies. Bedrock exposure is confined to the higher-relief drainages and along mountainous ridges (see Figure 4). The majority of the property is covered by several metres of glacial till. Vegetation on the plateau consists of scrub birch and willow, grasses, and mosses. Within the creek valleys, are several varieties of conifer and deciduous trees including balsam, fir, cedar, spruce, and aspen.

The project area lies in a region of moderate annual precipitation with an average of 406 mm total annual precipitation measured over a 35 year period of record in Dease Lake. Precipitation is more or less evenly distributed throughout the year, with April to May receiving the least and August to December the most. Temperatures vary from a low of -21° C in January to a high of 9° C in July with temperature extremes ranging from -50° C to 30° C.

Figure 4 - Property Boundary and Physiography



Picture simulated from digital topography and Landsat TM. Vertical exaggeration is 2 times. (after Blanchflower, 1996)

bcMetals Corporation

PROPERTY BOUNDARY AND PHYSIOGRAPHY

Red-Chris Property
Liard Mining Division, British Columbia

Drawn By: JDB	Scale: As Shown
Date: Jan, 2004	FIGURE. 4

To Accompany Report By G. Giroux, P. Eng. *et al.*, 2004

6.0 HISTORY (Blanchflower *et al.*, (Nov. 18, 2002))

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

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

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

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

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

As a result of the Texasgulf exploration, two coalescing east-north-easterly trending zones of copper-gold mineralization named the Main and East zones were outlined. The mineralization was described as pyrite, chalcopyrite, and lesser bornite occurring spatially with zones or quartz vein stockwork near the centre of the Red intrusive stock. The estimated resource in 1976 at a

0.25% Cu cut-off was 34.4 million tonnes with an average grade of 0.51% Cu and 0.27 g/t Au to a depth of 270 m in the Main Zone and 6.6 million tonnes with average grade of 0.83% Cu and 0.72 g/t Au to a depth of 150 m in the East Zone (Newell and Peatfield, 1995).

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

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

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

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

The 1995 exploration program was designed and directed to explore, expand and delineate the mineral resources of the Red-Chris copper-gold deposit, both laterally and vertically, and to evaluate the Gully and Far West zones. Field work was carried out from April 27th to November 12th. The 1995 on- and off-site exploration work included:

- 1) relocation and reconstruction of the core logging facilities to within 125 metres of the campsite;
- 2) claim staking (ABM-7 to -11 modified grid mineral claims (56 units);

- 3) extending the survey control grid westward (20.525 line-km);
- 4) soil geochemical sample collection and analyses (412 A-, B- or C-horizon soil samples collected);
- 5) geological mapping of the East and West Gully drainages at a scale of 2:1,000 with coincident rock geochemical sampling (5 rock samples collected and analysed for copper and gold);
- 6) exploratory HQ- and NQ-core diamond drilling (112 holes totalling 36,770.46 m or 120,638 ft.);
- 7) geotechnical diamond drilling at three proposed tailings dam sites along Kluea Lake valley (3 BQTK-core diamond drill holes totalling 59.44 m or 195 ft.);
- 8) diamond drill collar and survey control grid surveying;
- 9) drill sample analyses (9,783 samples assayed for copper and gold and 1,796 samples geochemically-analysed for copper (A.A.) and gold (F.A./A.A.);
- 10) drill sample check-assaying (1,235 and 1,227 duplicate drill core samples assayed for copper and gold respectively, and 451 standard and blank samples assayed for copper and gold);
- 11) mineral characterization analyses (2,458 samples for 31-element I.C.P.);
- 12) preliminary acid base accounting analyses (123 A.B.A. analyses including 110 drill core samples and 13 duplicate samples based proportionately on major rock types and styles of mineralization);
- 13) geotechnical core samples processed by Knight and Piesold Ltd;
- 14) environmental studies (baseline monitoring programs for site hydrology, water quality and meteorology, and fish and wildlife population studies);
- 15) metallurgical testing diamond drill core rejects from selected drill holes within the Red-Chris deposit and Gully Zone;
- 16) geological resource estimation studies by G. Giroux, P. Eng., of Montgomery Consultants Ltd. and mining engineers of Fluor Daniel Wright Ltd.; and
- 17) subsequent collation, compilation and documentation of the results of the program.

The 1995 exploration program successfully increased the geological resources of the Red-Chris deposit across the width of the Red stock and over a 400-metre strike length west of the known mineralization. Significant near-surface copper-gold mineralization was also discovered at the

Gully and Far West zones. As of November, 1995, the property had been tested by a total of 244 diamond and 44 percussion drill holes, or 74,782 metres of drilling. The results of this work indicate that the Red-Chris deposit is still open both laterally and vertically, and the newly-discovered Gully and Far West zones could also host significant near-surface copper and gold resources. There are also other exploration targets on the property, such as the altered and pyritized volcanic rocks north of the Red stock that have only received minimal investigation and should be evaluated by future exploration work.

American Bullion Minerals Ltd. reported the 1995 exploration program cost CAN \$ 5.9 million.

The following is a brief summary of various resource estimates completed previous to the 2004 study that this report is based on.

Table 2 - Summary of Pre-2004 Red Chris Resource Estimates

Year	Zone	Measured Resource			Indicated Resource			Inferred Resource		
		Million Tonnes	Cu (%)	Au (g/t)	Million Tonnes	Cu (%)	Au (g/t)	Million Tonnes	Cu (%)	Au (g/t)
1976 ¹	Main				34.4	0.51	0.27			
	East				6.6	0.83	0.72			
1988 ²	Main				19.79	0.65	0.34			
	East				2.59	1.52	1.3			
1994 ⁴	Combined	11.2	0.4	0.3	169.72	0.4	0.31	139.44	0.35	0.28
1994 ⁵	Combined	2.75	0.66	0.46	38.46	0.68	0.56	19.62	0.67	0.54
1998 ⁶	Main	1.25	0.62	0.35	19.4	0.65	0.45	27.9	0.61	0.50
	East	10.5	0.88	0.82	18.1	0.72	0.70	0.25	0.60	0.55

- Notes**
- 1976¹ - Estimate reported in Newell and Peatfield, 1995 used cut-off of 0.25% Cu, Main zone taken to 270 m depth and East zone taken to 150 m.
 - 1988² - Estimate by Wrigglesworth of Falconbridge (Newell and Peatfield, 1995) taken at a 0.5 % Cu cut-off.
 - 1994³ - Estimated by Giroux, (1995) at a 0.2 % Cu cut-off, Main zone and East zone combined to form a possible large low grade pit.
 - 1994⁴ - Estimated by Giroux, (1995) at a 0.5 % Cu cut-off, Main zone and East zone combined
 - 1998⁶ - Estimated by Giroux in 1998 and Reported in Giroux *et al.*, (2002) at a 0.5% Cu cut-off

7.0 GEOLOGICAL SETTING

The following geology summary, with the exception of section 7.8, has been taken from Blanchflower *et al.*, (2002).

7.1 Regional Geology

The Stikine River area was mapped in 1957 by the Geological Survey of Canada as Operation Stikine (G.S.C. Map 9-1957). Later geological mapping by Souther, (1972) of the Telegraph Creek sheet (N.T.S. 104G, 1:250,000), and by Gabrielse and Tipper, (1984) of the Spatsizi sheet (N.T.S. 104H, 1:125,000) have been the regional geological database until quite recently. Recent geological mapping at a scale of 1:50,000 by Read, (1984) and Read and Psutka, (1990) for the eastern Ealue Lake area (104H/13E and W), and by the B.C. Ministry of Employment and Investment, Geological Survey Branch (Ash and Fraser, 1994; Ash *et al.*, 1995; Ash *et al.*, 1996a and b; Ash *et al.*, 1997) in the Tatogga Lake area have provided valuable geological information in the vicinity of the subject property. The geological setting and history of the Bowser Lake Group, which crops out south of the Red-Chris deposit, have been documented as part of the multidisciplinary Bowser Basin project (Evenchick, 1991a, b; Evenchick and Green, 1990; Evenchick and Thorkelson, 1993; Green, 1991; Poulton *et al.*, 1991; Ricketts, 1990; Ricketts and Evenchick, 1991).

The Red-Chris property geology and copper-gold mineralization have been the subject of thesis research and corporate geological studies. Detailed geological studies include those by Schink, (1977) who investigated the petrology, alteration and mineralogy of the deposit for a Master of Science thesis, and Leitch and Elliot, (1976) who mapped the detailed geology and mineralization of the property for Texasgulf Inc. Furthermore, geological reports by J. R. Forsythe, (1975; 1977a, b; Forsythe and Peatfield, 1974; Forsythe *et al.*, 1976), G. R. Peatfield, (1980, 1981) and other Texasgulf Inc. geologists have greatly contributed to the understanding of the deposit. Six more recent published geological reports on the deposit and its geological setting are by Newell and Peatfield, (1995), Ash *et al.*, (1995), Ash *et al.*, (1996a and b), Ash *et al.*, (1997), and Friedman and Ash, (1997).

The property is situated regionally within the Stikinia Terrane of northern British Columbia. This terrane is dominated by Early Mesozoic and lesser Late Paleozoic island-arc volcanic strata and related subvolcanic intrusions that form a broad northwesterly trending belt along the centre of the province from southern British Columbia into southwestern Yukon Territory, often referred to as the 'Intermontane Belt' (Woodsworth *et al.*, 1991). Stikinia terrane arc rocks have been regionally subdivided into Late Paleozoic Stikine, Late Triassic Stuhini, and Early to Middle Jurassic Hazelton Groups. The Late Triassic Stuhini Group rocks are dominated by submarine calc-alkaline basaltic volcanic rocks which are commonly augite-phyric versus those of the Hazelton Group which are dominated by subaerial volcanics that display a broad range in composition from basalt to rhyolite (Souther, 1991).

The Stikinia terrane probably developed as primarily Late Triassic and Early and Middle Jurassic oceanic island-arcs outboard of the ancient North American continental margin (Monger, 1984). Island arcs evolved along the western margin of the intervening, Late Paleozoic ocean basin in response to westerly subduction. Early Middle Jurassic arc-continent collision, related to docking of the Stikinia arc with the ancient margin, resulted in southwesterly tectonic emplacement of oceanic Cache Creek terrane rocks above the Stikinia terrane. The uplifted oceanic crust shed

clastic flysch sediments southwardly into the newly developed continental margin to form the Bowser Lake Group (Ash *et al.*, 1995).

According to Ash *et al.*, (1996a),

“The map area (Kluea Lake - 104H/12, Kinaskan Lake - 104G/9) is underlain almost entirely by Upper Triassic and Lower Jurassic arc-volcanic rocks that are overlain along their southeastern margin by Middle Jurassic Bowser Lake Group sediments. These Mesozoic volcanic rocks are divisible into three broad northeast-trending belts. The northwestern belt is dominated by Middle (?) to Upper Triassic andesitic volcanoclastics, mainly massive breccias. The central belt is underlain primarily by Upper Triassic and possibly Lower Jurassic fine to medium-grained epiclastic rocks. Lower Jurassic rocks comprise a bimodal suite of basalts and rhyolites and related subvolcanic rocks that overlie and intrude very fine to medium-grained sedimentary rocks primarily to the southeast. The younger rocks also locally intrude and overlie Triassic rocks throughout the map area.

These rocks have been affected by folding and faulting. Mesoscopic folding is generally only identified with the Lower Jurassic and older, thinly bedded sediments, mainly siltstones, and rarely in limestone. Broader warping of thicker bedded sequences is a characteristic megascopic feature commonly seen in cliff exposures. High-angle brittle faults are abundant throughout the map area and contacts are rarely exposed. As a result, it is difficult to establish continuity of contacts between individual Mesozoic volcanic units.”

Based upon fossil evidence, Ash *et al.*, (1996) found that most of the Lower Jurassic sections within the map area probably represent a short interval between 200 and 193 Ma (i.e. Sinemurian to Pliensbachian).

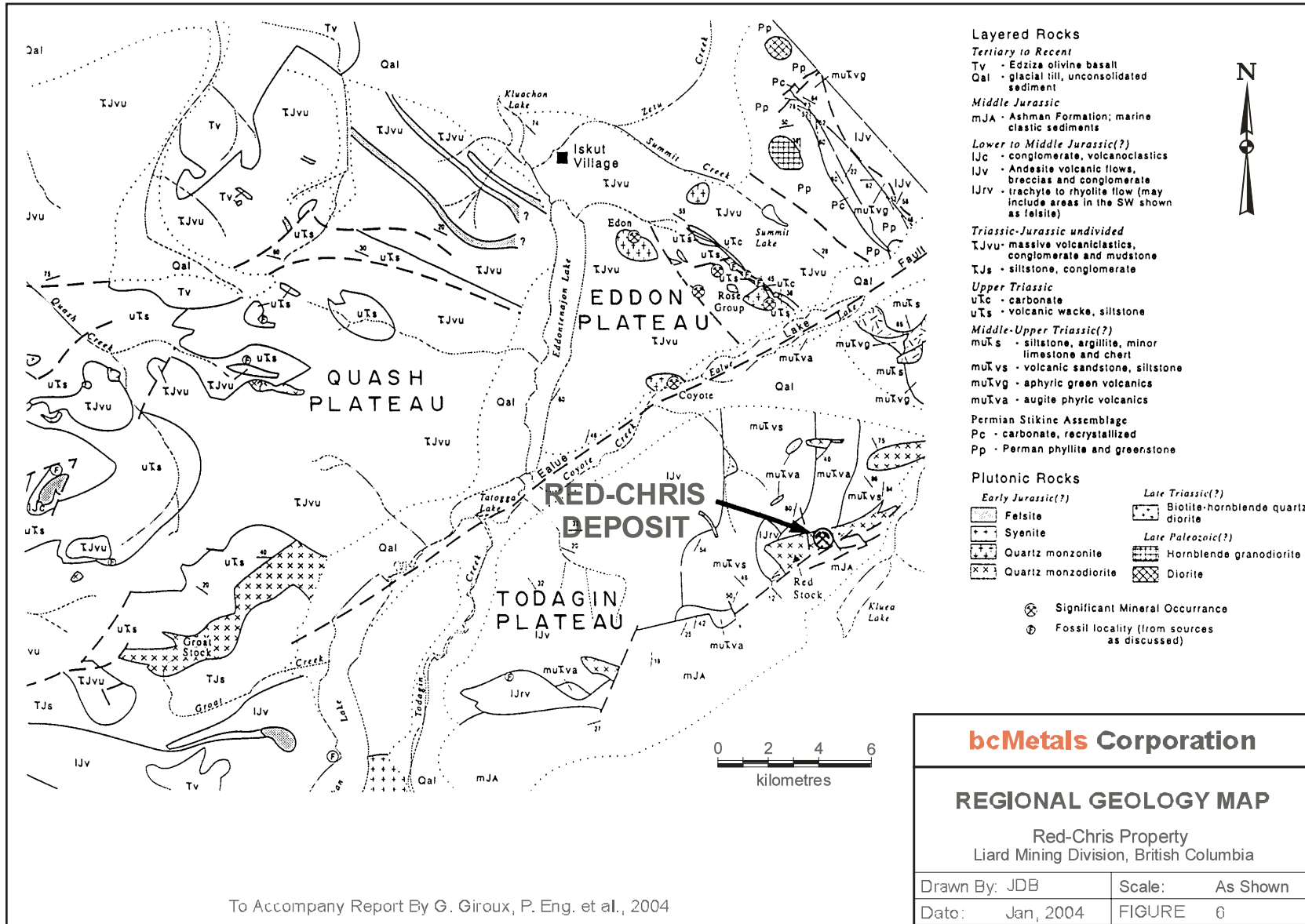
A suite of earliest Early Jurassic (195 to 205 Ma) stocks and dykes occur throughout the region. These intrusions are compositionally variable, ranging from hornblende quartz diorite to quartz monzodiorite, and are characteristically medium-grained, equigranular to porphyritic and weather a buff-white to light grey colour. The largest intrusion of this suite is the Red stock which hosts the Red-Chris deposit. It intrudes Upper Triassic massive volcanic wackes, siltstone and possibly augite-porphyritic basalt within the Red-Chris property (Ash *et al.*, 1996).

Middle Jurassic (Bathonian to Early Oxfordian) marine clastic sedimentary rocks (Gabrielse and Tipper, 1984; Poulton *et al.*, 1991) of the Bowser Lake Group, underlying the southern portion of the subject property, are assigned to the basal Ashman Formation and comprise siltstone, chert pebble conglomerate and sandstone (Evenchick and Thorkelson, 1993). Sedimentological studies indicate that Bowser Lake Group rocks become progressively younger to the south and that deposition was from the north into the tectonically active northern margin of the Bowser Basin (Ricketts, 1990; Ricketts and Evenchick, 1991; Green, 1991).

Within the region there are several isolated outcrops of olivine-phyric basalt flows, belonging to the Early Pliocene Maitland Volcanics, overlying the Stikinia terrane rocks; a few of which occur on the subject property (Ash *et al.*, 1996).

Major regional faulting has affected the local stratigraphy during Middle Cretaceous and Tertiary tectonism. The east-northeasterly trending Ealue Lake Fault is the most prominent structural feature in the vicinity of the subject property. Although not exposed, it has been projected along the Coyote Creek-Ealue Lake Valley (Ash *et al.*, 1995). Its presence is evident by contrasting lithologies and styles of alteration on either side. Zones of intense carbonatization with localized areas of ankerite flooding are widespread in rocks only south of the fault (Ash *et al.*, 1995). Also, its continuity to the east has been determined for an additional 30 kilometres where it has been designated the McEwan Creek Fault with a south side-down movement sense (Read and Psutka, 1990). There are also similarly-oriented faults along the northern contact of the Bowser Lake Group; one of which is the southside-down normal bounding fault between the Bowser Lake Group rocks and the Red stock near the centre of the property.

Figure 5 - Regional Geography Map



7.2 Local Geology

The property covers the eastern portion of a large east-northeasterly trending, stratigraphically-distinct, fault bounded upland called the 'Todagin Plateau' (Ash *et al.*, 1995). The lithologic units on the property have been described chronologically from oldest to youngest.

a) Middle to Upper Triassic Volcanic and Sedimentary Rocks (muTva and muTvs)

Recent geological mapping by Ash *et al.*, (1994 and 1995) has identified an intercalated sequence of augite-phyric volcanic and volcanically-derived sedimentary rocks cropping out between the northeastern slopes of Todagin Mountain and Ealue Lake, underlying most of the northern portion of the property.

Alkaline volcanic rocks, informally called the 'Dynamite Hill' volcanics (Leitch and Elliot, 1976), crop out immediately north and northwest of the Red stock, along the East Gully to Bowers Creek drainages north to Ealue Lake. They also reportedly occur on the southeastern side of the Red stock in fault contact with the Middle Jurassic Bowser Lake Group sedimentary rocks.

Ash *et al.*, (1995) found the volcanic rocks to be dominated by augite-phyric pillowed flows and flow breccias of basaltic composition. Leitch and Elliot, (1976) describe these rocks as massive porphyritic basic volcanics with no visible structure; however, Schink, (1977) and Forsythe, (1976) suggest that they are dominated by relatively massive flows which locally exhibit poorly developed pillow structures and flow banding. They appear on surface to be dark green-coloured, quite massive, and with varying amounts augite, hornblende and plagioclase phenocrysts in a green chloritic groundmass. Rocks observed along the intrusive contact of the Red Stock are often bleached and pyritized resulting in a pale green to buff colour, and a more felsic macroscopic colouration.

The volcanic rocks are locally intercalated with Middle to Upper Triassic volcanically-derived fine-grained sedimentary rocks (VSED), including volcanic wacke (feldspathic sandstone), siltstone and siliceous siltstone, on a scale of metres to tens of metres (Leitch and Elliot, 1976; Ash *et al.*, 1995). Volcanically-derived sedimentary rocks are much more prevalent in the western map-area. At the Gully Zone the volcanically-derived sedimentary rocks have been intersected by deep drilling and host a significant portion of the copper-gold mineralization where they occur as faulted slices and wedges within the fault-brecciated margins of the Red Stock. These rocks also occur at the Far West Zone where they host a portion of the mineralization and occur in intrusive contact with the Red stock.

b) Early Jurassic Plutonic Rocks

Several stocks and dykes of hornblende-plagioclase porphyritic quartz monzodiorite composition have been mapped within the Todagin Plateau area by Leitch and Elliot, (1976) and Ash and Fraser, (1994). These intrusions occur in close proximity to the Red stock and are very similar to it in geometry and texture. They are described by Ash *et al.*, (1995) as intrusive rocks that

weather buff-white to light grey, and have distinctive medium- to coarse-grained hornblende and plagioclase phenocrysts randomly oriented in an aphanitic grey groundmass.

Ash (1996) reports that four zircon fractions from drill core of the Red stock (i.e. DDH 94-224) have been Pb-U dated as 203.8 ± 1.3 Ma, or of earliest Early Jurassic age. This date correlates well with three dates from various other plutons throughout the Tatogga Lake map area that ranged from 199 to 205 Ma. All samples also show an Early Paleozoic inheritance at 500 Ma.

The Red stock is elongate, irregular in shape, and occupies a major east-northeasterly en echelon fault structure. It is at least 4.5 kilometres long by 300 to 1,500 metres wide, but it may also extend well beyond its exposed boundaries as a buried pluton beneath the partially eroded older volcanic and sedimentary cover. Various plutons both east and west of the main stock were identified by Leitch and Elliot, (1976) but, except for variation of pyrite and hornblende contents, they were apparently identical and are probably apophyses of a larger intrusion.

According to Leitch and Elliot, (1976), volcanic rocks in contact with the Red stock display local thermal metamorphic and metasomatic features, such as moderate hornfelsing, increased pyritization and propylitic alteration, but they have not been foliated. These features suggest that the stock was indeed emplaced hypabyssally and is probably comagmatic with the surrounding volcanic country rocks.

Two compositionally-similar phases of plutonic rocks comprise the stock and these rocks are cut by several post-mineral dykes of dioritic to monzonitic composition. The 'Main Phase' unit is a medium-grained, weakly- to intensely-altered plagioclase-hornblende porphyritic monzodiorite that hosts most of the known copper-gold mineralization and constitutes approximately seventy to eighty (70-80) percent of the stock. The 'Late Phase' unit is now thought to comprise both unaltered and barren Main Phase and post-mineral dykes with indistinct flow banded and chilled margins; all of which are remarkably similar in composition and texture to very weakly altered Main Phase rocks. However, the Late Phase unit appears to be fresher looking and less altered than the Main Phase unit, usually barren of copper-gold mineralization, and represents approximately twenty to twenty-eight (20-28) percent of the stock. The late-stage, post-mineral dykes are commonly porphyritic, range in composition from dioritic to monzonitic, are usually less than 1 to 5 metres wide; although they may attain widths of up to fifty (50) metres in the western end of the Red-Chris deposit area. These dykes comprise the remaining volume of the Red stock.

Intrusive breccia occurs throughout the Red stock; especially along the northeastern and western margins of the Red-Chris deposit and within the Gully and Far West zones. Breccia bodies may range locally in width from a few metres to 100 metres or more. Their contacts are relatively distinct; marked by a rapid increase or decrease of subangular to angular fragments of plutonic rock. These fragments can vary from less than a centimetre to several metres in diameter.

The Red stock and older country rocks are cut by several varieties of late-stage, post-mineral dykes; identified by their texture, mineralogy and appearance. There are three main varieties, from oldest to youngest: Porphyritic Feldspar-Hornblende-Biotite Dykes (DPFH), Quartz-Carbonate Amygdaloidal Dykes (DQCA), and Mafic Dykes (DMAF).

c) Lower to Middle Jurassic Volcanic Rocks (Units IJrv and IJv)

Lower to Middle Jurassic trachytic to rhyolitic flows have been mapped at the western end of the Red stock along the Bower Creek drainage (Ash *et al.*, 1995). These volcanics were also mapped by Leitch and Elliot, (1976) who classified them as intermediate to acid volcanics and minor pyroclastics. They reported that these volcanics are more varied than those underlying Dynamite Hill and that the rocks ranged from dark green andesite to orange trachyte and white rhyolite. Minor tuffaceous volcanoclastics are intercalated with the volcanics rocks. They appear to be late-stage extrusive equivalents of the Red Stock intrusion (Schink, 1977) with bedding attitudes striking 090° and dipping northward at -45° along the north side of the stock to striking north and dipping sub-vertically further to the west (Leitch and Elliot, 1976).

d) Middle Jurassic Ashman Formation (basal Bowser Lake Group; mJA)

Marine clastic sedimentary rocks of the Ashman Formation, a basal unit of the Middle Jurassic Bowser Lake Group, underlie the southern property boundary, along the ridgeline between the Red stock and Kluea Lake. The Ashman Formation is comprised of siltstone, chert-pebble conglomerate and sandstone (Evenchick and Thorkelson, 1993). Bowser Lake Group rocks young progressively to the south; indicating that deposition was from the north into the tectonically-active northern margin of the Bowser Basin (Ricketts, 1990; Ricketts and Evenchick, 1991; Green, 1991).

Massive to well-bedded chert-pebble conglomerates occur in fault contact with the southern margin of the Red stock. Repetitively-bedded laminae, varying from 5 to 15 cm thick, are defined by an up-section reduction in both size and abundance of chert clasts. Local massive conglomerates contain 40 to 60 percent sandstone clasts and/or matrix sandstone. Both laminated and massive conglomerates have subrounded, 0.5 to 3 cm diameter, light to dark grey or green chert pebbles in a tan brown to grey sandstone matrix.

e) Maitland Volcanics

Near the headwaters of the East and West Gully drainages there are small outcrops of columnar olivine-phyric basalt flows (Schink, 1977). These rocks represent the youngest rocks in the region, probably of Early Pliocene age (Gabrielse and Tipper, 1984; Ash *et al.*, 1996).

7.3 Veining and Stockwork

It was recognized during early exploration of this property that most of the mineralization is closely associated with individual and sheeted quartz (\pm carbonate) veining, and quartz (\pm carbonate) stockwork zones. Thus, considerable work has been undertaken to understand the relationship and distribution of very weak to intense quartz veining and stockwork zones with potentially economic copper-gold mineralization. Following the discovery of the Gully and Far West zones in 1995, it is now recognized that a significant portion of the mineralization also occurs as very fine- to fine-grained disseminations and fracture-fillings; resulting in visual under estimations of grades.

Quartz-carbonate veining is ubiquitous throughout the Red stock and in Middle to Upper Triassic country rocks; especially in zones of fracturing and carbonatization. Pyrite, chalcopyrite, magnetite with lesser hematite and rare molybdenite are often associated with quartz-carbonate veining as fine-grained disseminations within the vein core or as disseminations and/or fracture filling along the vein selvages.

Several discontinuous zones of intense silica flooding, accompanied by significant copper-gold mineralization, form the core of the quartz-carbonate-sulphide vein stockwork in the Red-Chris deposit. These zones are from 10 to 40 metres wide and are more common at the eastern end. They have an apparent 060° to 070° strike but cross-sectional plots show their orientation is controlled by east-west, sub-vertical splay fault structures from the larger East Zone fault structure. Geological modeling of high grade copper-gold mineralization associated with these zones shows the sheeted quartz veining to trend easterly (090°) and plunge -25° to -40° eastward. A similar orientation is indicated for the less common sheeted quartz zones in the western half of the deposit. These sheeted quartz zones have not been intersected by any recent drilling in the Gully and Far West zones.

The sheeted quartz zones are lenticular and composed of parallel to sub parallel quartz-sulphide (\pm carbonate) veins. They grade outward into an intense quartz-carbonate-sulphide vein stockwork, and are often associated with younger intense faulting that appears to be superimposed on a pre-existing zone of structural weakness through which the highly siliceous hydrothermal fluids were emplaced. Their present discontinuity appears to be a function of later faulting, rather than a primary feature. Altered Main Phase host rock fragments are locally included in the quartz sheeted zones. They have sharp boundaries with the enclosing quartz veins and abundant chalcopyrite disseminations near their margins; indicating that the sheeted quartz-sulphide veins were emplaced quite quickly without pervasive silicification (Schink, 1977).

Sheeted quartz-carbonate-sulphide zones generally host quite high grade copper-gold mineralization but the zones of weak to intense quartz-sulphide-carbonate stockwork account for most of the mineralized resources.

Quartz-sulphide stockwork veins range from 3 to 10 mm in width, rarely attain 1 cm, and form a randomly orientated network pattern with at least two generations present. They are usually symmetrical and characterized by sharp, parallel walls and regular selvages. Sulphides are usually confined to a central vein fracture or core, and to minute cross-fractures. Minor ankerite, magnetite and hematite are usually present in the vein core. Repeated episodes of fracturing and mineralization are reflected by crosscutting relationships. Alteration envelopes appear to be lacking, or they have been overprinted by later alteration facies.

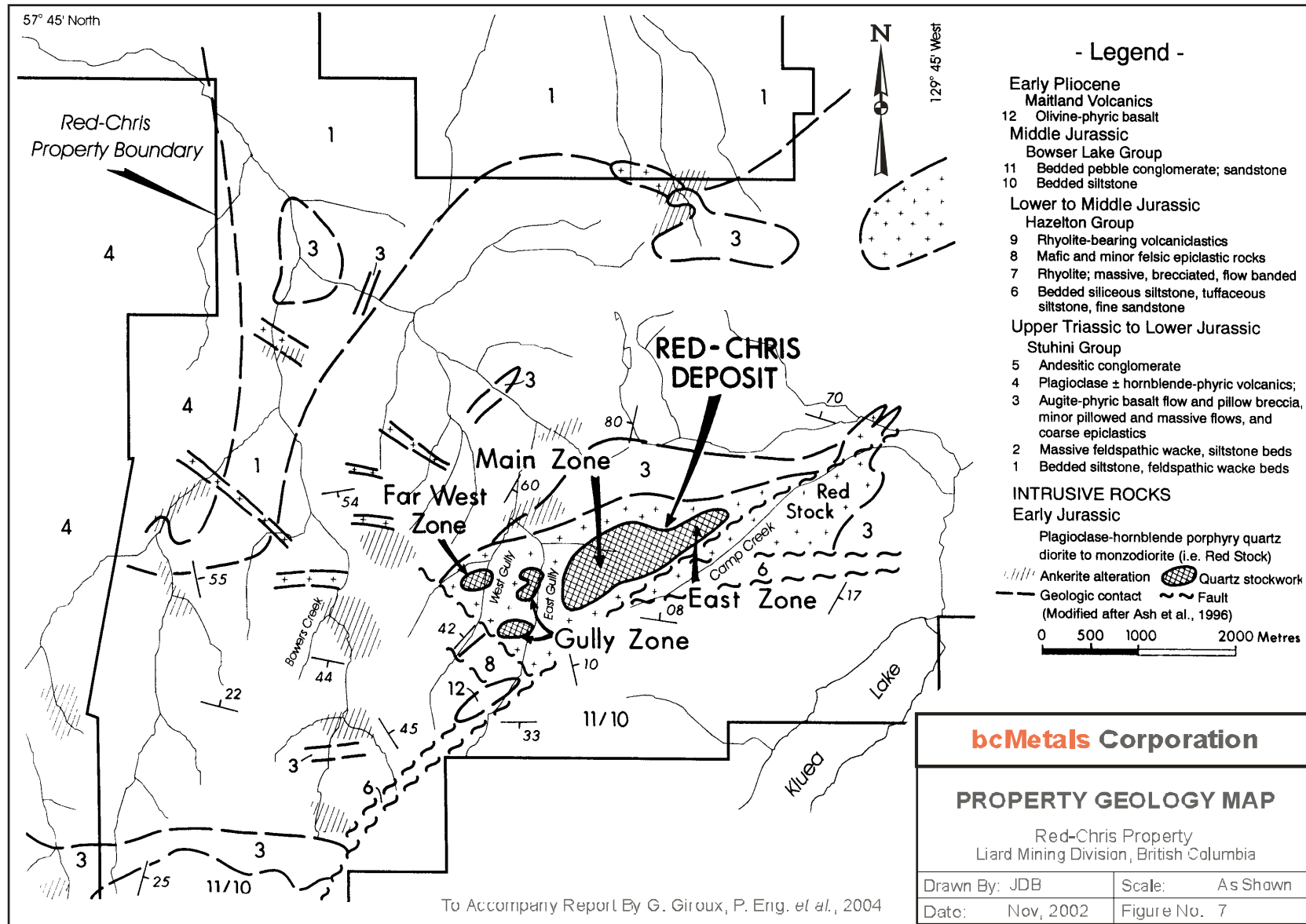
Quartz-sulphide vein stockwork is typically absent in Late Phase rocks. Trace quartz stringers or veins are occasionally observed but they usually barren of sulphides. These veins are generally less than 1 cm wide with irregular, vague boundaries and are comprised of white quartz \pm magnetite.

The grades of copper-gold mineralization are very correlative with the intensity of

quartz-sulphide stockwork veining in the Red-Chris deposit, unlike the Gully and Far West mineralization. Quartz-sulphide stockwork intensity was based upon the following arbitrary categories:

Trace	Rare vein
Very Weak	Less than 1 vein per metre
Weak	1 to 12 veins per metre
Moderate	12 to 30 veins per metre
Strong	More than 30 veins per metre

Figure 6 - Property Geology Map



It is recognized that the intensity of stockwork veining, although usually gradational, can increase or decrease rapidly across fault structures.

The majority of the mineralized resources occur in well developed quartz-sulphide (\pm carbonate) vein stockwork zones. These zones are spatially and probably genetically related to major east-northeasterly faulting in the East Zone (see Figure 7) and easterly faulting in the Main (see Figure 8), Gully and Far West zones. Although younger reactivated faults, such as the East Zone fault and its splay faults, have cut and locally displaced the quartz-sulphide stockwork zones they are distributed along the central long axis of the Red stock and dip steeply southward in the East Zone to subvertical in the Main Zone; similar to later faulting.

The quartz-sulphide (\pm carbonate) stockwork zones in the Far West and Gully zones appear to be vertical or steeply south dipping similar to the east-west reactivated faulting to the east, but they are not as intense as those in the Red-Chris deposit. Within the Gully and Far West areas, very weak to moderate quartz stockwork zones are also hosted by volcanically-derived sedimentary rocks, and there is considerable finely disseminated chalcopyrite mineralization with gold values associated with these stockwork zones; unlike similar stockwork zones within the plutonic rocks.

Irregular zones of weak to strong gypsum veining are located west and southwest of the Red-Chris deposit and in the Gully and Far West zones. Gypsum veins and fracture fillings cut all other vein types on the property, and are hosted by the Main and Late Phase units and late-stage quartz-carbonate amygdaloidal dykes (Unit DQCA) of the Red stock. Gypsum zones do not crop out but are most often intersected as irregular flat-lying features at depths of less than 10 metres to greater than 350 metres with continuous intersections over 100 metres. There are at least two periods of gypsum veining present on the property; one period either pre-dates or is contemporaneous with the emplacement of the Red stock and a second period post-dates the mineralization.

Carbonate (\pm quartz) veins and carbonatization of groundmass minerals to ankerite and iron-rich magnesite are widespread throughout the Red stock. Within structural zones the Middle to Upper Triassic volcanic and sedimentary rocks are also intensely carbonatized. Carbonate (ankerite more than calcite) veins occur as white to pale pink irregular veins averaging 2 to 7 mm wide. These veins are commonly barren of sulphides but rarely and locally host pyrite, chalcopyrite and minor sphalerite and galena. Carbonate is also common as fracture fillings and locally occurs as the matrix to tectonic breccias. Sphalerite and minor galena often occur together in pink to buff carbonate-dolomite veins cutting mineralization. Carbonate veins appear to be very late structures since they cut mineralized quartz veins and late-stage quartz-carbonate amygdale dykes; thus, they appear to post-date the main copper-gold hydrothermal mineralizing event.

7.4 Alteration

Most of the Main Phase unit of the Red stock has been repeatedly and variably altered by apparently epizonal hydrothermal fluids since its emplacement. The post-mineral Late Phase unit is usually quite fresh to only very weakly altered. A primary porphyritic texture is always observed but it may be partially obliterated by alteration around late-stage quartz-carbonate fracture fillings. None of the Bowser Lake Group rocks have been affected by any of the

pervasive alteration present in the Middle Triassic to Lower Jurassic intrusive and volcanic rocks that are situated immediately north of the South Boundary fault structure.

Six alteration facies were identified by Schink, (1977) based on petrography and the presence of ankerite.

American Bullion Minerals field personnel could not recognize six facies because both ankerite and the albitization of feldspars are only visible microscopically. The following alteration assemblages have been modified from Schink, (1977) and adapted for diamond drill core logging during the 1994 and 1995 exploration programs. RCDC geologists used the same alteration categories and alteration codes for the 2003 in-fill diamond drilling program. The following alteration descriptions described by Blanchflower (Blanchflower *et al.*, 2002) were followed in logging the 2003 drill core.

Potassic alteration is sporadic and quite limited; perhaps representing only 10 to less than 15 percent of the total altered area. It dominantly occurs in the eastern portion of the Red-Chris deposit as narrow discontinuous zones a few metres wide that have gradational to sharp contacts with zones of quartz-sericite ± hematite±kaolinite±ankerite alteration. Where the Main Phase unit has been affected by potassic alteration the rocks have a light orange-brown to salmon colour and mottled appearance. The porphyritic texture of the rock is often partially or completely destroyed and its primary mineral constituents show complete replacement. Plagioclase phenocrysts are pseudomorphed by microcrystalline sericite, hematitic albite, ankerite and quartz. Relict hornblende phenocrysts are more commonly altered to a fine grained, felted brown biotite and but may also be pseudomorphed by granular ankerite, pyrite and light coloured chlorite. Rare primary biotite phenocrysts are replaced by pseudomorphic muscovite with minor ankerite. The groundmass is flooded with secondary very fine-grained orthoclase and biotite phenocrysts, and it may also contain ankerite, sericite, kaolinite, quartz, magnetite, hematite, pyrite, and trace apatite, tourmaline and zircon in varying amounts.

Phyllic (quartz-sericite-pyrite±ankerite) alteration is pervasive and strongly developed throughout the Red-Chris deposit and western map-area. It occurs discontinuously throughout the Red stock, commonly in the Late Phase unit, and as restricted zones in the volcanic and volcanoclastic country rocks (Schink, 1977). In hand specimen, phyllic alteration is pale grey with a distinctive bleached appearance. Primary textures are only partially obliterated. Relict plagioclase phenocrysts are bleached with a pale green colour; their grain boundaries are generally preserved but the interiors are usually replaced by microcrystalline sericite and ankerite with minor quartz, dolomite and kaolinite. Groundmass feldspars are replaced by fine, anhedral quartz and interstitial sericite. Hornblende phenocrysts are typically completely destroyed; with rare remnant phenocrysts showing replacement by sericite and minor dolomite (Schink, 1977). Pervasive and abundant ankerite occurs usually in the groundmass and less commonly with quartz as vein selvages, but it is only obvious in weathered diamond drill core. It can account for 1 to 7 percent of the rock volume.

Figure 7 - East Zone Cross Section 50750 E

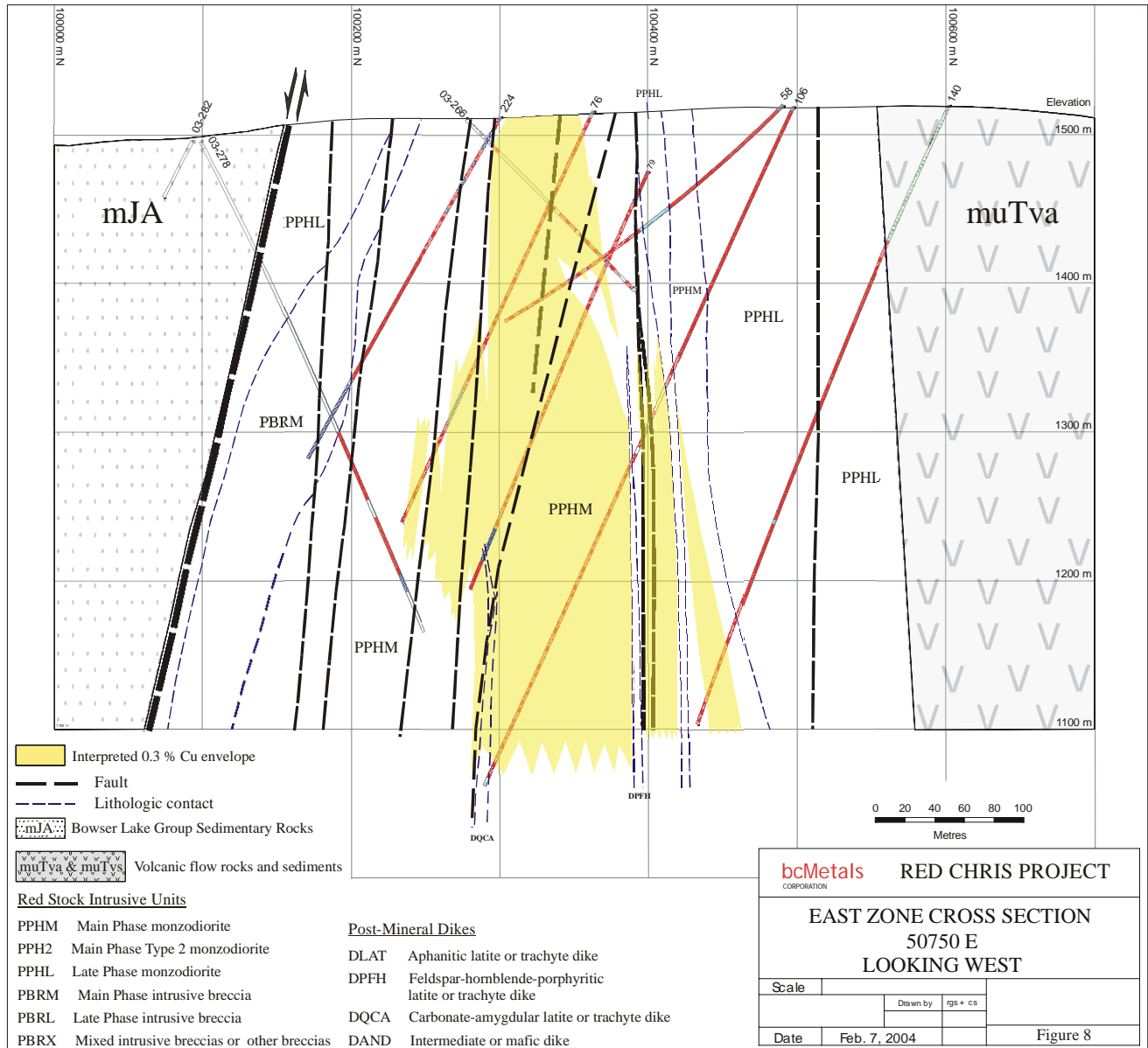
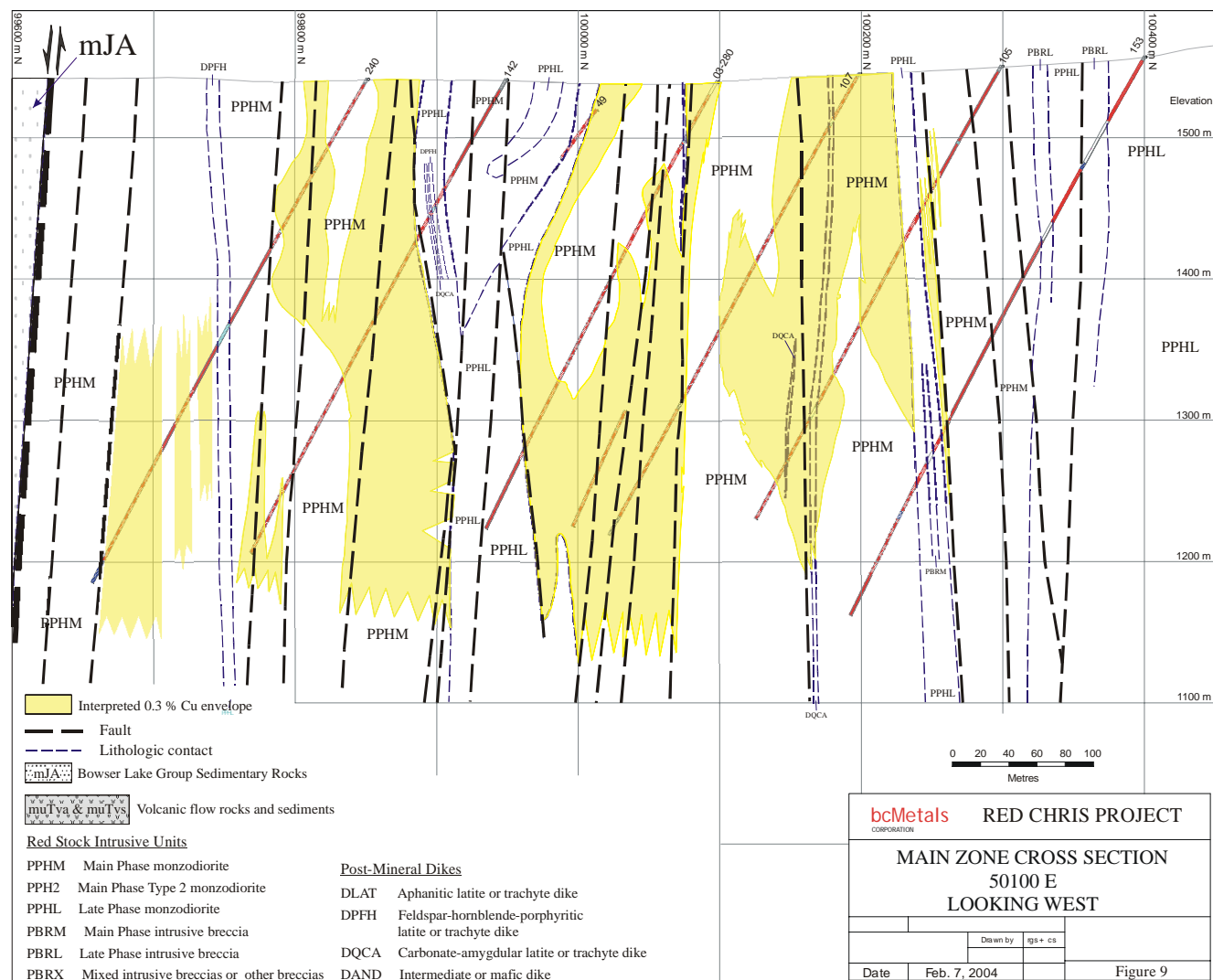


Figure 8 - Main Zone Cross Section 50100 E



Mottled phyllic alteration was thought to be a transition between argillic (quartz-sericite± hematite ±kaolinite ±ankerite) and pervasive phyllic (quartz-sericite-pyrite ± ankerite) alteration. Its inner margins may coincide with the disappearance of widespread hematite and magnetite, and with the appearance of abundant pyrite, marking the edge of the pyrite halo. It is now thought that the mottled phyllic assemblage represents a zone of alteration overprinting.

Quartz-sericite± hematite ± kaolinite± ankerite alteration is usually restricted to zones of moderate to intense quartz-sulphide stockwork veining that are developed within the Main Phase unit. This alteration facies is characterized by the presence of pale green plagioclase relics and pale brown hornblende pseudomorphs set in a pale, light to medium brown, aphanitic groundmass. It was the opinion of Schink, (1977) that this alteration facies occurred quite early during the hydrothermal process; however, since it usually occurs within zones of intense fracturing and quartz vein stockwork it is difficult to determine its temporal relationships with

other alteration facies. This alteration facies occurs with the majority of copper-gold mineralization.

Propylitic alteration is poorly developed within the Red stock. In the Main Phase unit epidote-chlorite-pyrite-calcite alteration is characterized by pseudomorphic replacement of andesine phenocrysts by hematitic albite and lesser epidote. Biotite and hornblende are replaced by chlorite and calcite (Schink, 1977) and occasional epidote. Locally some feldspar-hornblende-biotite porphyry dykes (DPFH) have epidote replacement of hornblende phenocrysts. The augite-phyrlic volcanic country rocks situated immediately north of the Red stock, underlying Dynamite Hill, are altered by this facies and host 5 percent disseminated epidote and 2 to 5 percent pyrite as disseminations and veinlets.

7.5 Structure

The structural setting of the property is dominated by east-northeasterly trending en echelon fault structures. The elongated Red stock occupies and has been displaced by at least one major east-northeasterly trending ancestral fault structure that has been repetitively reactivated during Middle Triassic to Middle Jurassic time. This fault structure and several similarly-oriented faults, such as the one bounding the northern margins of the Bowser Lake Group, are probably subsidiary or parasitic structures related to movements along the larger and east-northeasterly striking Ealue Lake Fault.

Structural evidence for the repeated reactivation of a fault zone centred on and beneath the Red stock is obvious from the shape of the intrusion, the orientation of its major rock units, and the distribution and displacements of the alteration facies, sulphide mineralization and late-stage dykes. Forsythe, (1976) and Meade (in Peatfield, 1975) both concluded that much of the faulting is normal dip-slip in character, typified by hinge movements with the south-side blocks rotating and sliding downward, and that the fault planes seem to be concave to the south. Recent deep drilling results indicate that the faulting may have a more significant lateral component and that the fault planes appear convex to the south.

The Red stock is cut by several en echelon fault zones that probably reflect the youngest tectonic event but appear to be superimposed over the inferred trace of the larger ancestral structure. The most important of these, from an exploration standpoint, is the 'East Zone Fault'. This steeply southeasterly-dipping (-75°) fault zone strikes west-southwesterly (240°) from the eastern end of the Red stock, through the middle of the East Zone, to grid coordinates 100025 North by 50300 East. At this point it appears to bend due westerly and steepen vertically. It then splays into several east-west, sub-vertical fault structures that cut through the middle of the Main Zone. Both the strong to intense quartz stockwork zones and the associated fracture filling copper-gold mineralization are spatially-related to this structure. In the East Zone, the bornite-rich mineralization has an east-west trend and moderate easterly plunge related to east-west splay faults joining the East Zone fault. On drill cross-sections this mineralization is subvertical to very steeply southerly dipping. In the Main Zone the mineralization has a similar orientation but a more moderate easterly plunge, and the majority of the Late Phase dykes appear to be similarly controlled by these east-west splay faults.

Earlier geological work by Texasgulf personnel (Leitch and Elliot, 1976; Forsythe, 1977) inferred that the East Zone fault dipped steeply north within both the East and Main zones. Most of their drilling was directed southwardly and oriented at -45° to -60° to intersect the inferred steeply north-dipping structurally-related mineralization. It now appears that, except for the eastern portion of the East Zone, most of this mineralization is vertical to sub-vertical and could be tested by either southerly or northerly directed drilling. Furthermore, despite the structural complexity of the deposit much of the youngest faulting and many of the late-stage dykes are remarkably continuous, both laterally and vertically. Less than one-metre wide faults and dykes can be readily traced from multiple drill intercepts in a vertical plane, and usually laterally, over distances of several hundreds of metres.

Another major northeasterly trending fault structure underlies much of the Camp Creek drainage, called the 'South Boundary Fault' (Newell and Peatfield, 1995). It unconformably separates the southern margins of the Red stock and the surrounding Upper Triassic volcanic strata from Middle Jurassic Bowser Lake Group (Ashman Formation) clastic sedimentary rocks. This fault is not exposed on surface; however, geological, geomorphological and drill hole evidence show that it has been responsible for down-dropping the Bowser Lake Group rocks and obliquely truncating the southwestern margin of the Red stock. Several east-west splay faults from this structure appear to cut and displace the mineralization of the Main, East and Gully zones and parallel the distribution of the Late Phase dykes in the Main and Gully zones.

The quartz stockwork zones, mineralization and some late-stage dykes in the Main Zone and eastern end of the East Zone and in the Gully and Far West zones appear to have been locally displaced by a set of north-northwesterly (340°) strike-slip faults; probably conjugate scissor structures related to transcurrent movements along the East Zone and South Boundary fault zones. Texasgulf Inc. plotted geological surface and bench plans with regular multiple sets of north-northwesterly and north-northeasterly faults to explain truncations of various geological features they encountered.

Larger fault structures occur as gouge and/or brecciated zones. Gouge zones range from a few centimetres to several metres wide. They are usually grey to black in colour and commonly contain rounded to angular fragments, usually less than 2 cm in diameter, of altered Main Phase, Late Phase and occasionally mineralized quartz stockwork fragments in a matrix of clay, quartz and carbonate and finely grained pyrite (Blanchflower *et al.*, (Nov. 18, 2002)). In the Main Zone, many of these east-west structures contain fragments of or are partially occupied by narrow late-stage dykes. Faults intersecting mineralized zones can either contain the copper values of their host rocks or appear to be dramatically diluted by faulted and sheared dyke material.

In the 2002 drilling program the large South Boundary Fault was intersected in the north orientated (-63° N) geotechnical hole DDH 03-278. This hole was collared in heavily faulted and sheared Ashman Formation argillaceous siltstones and chert-pebble conglomerates. The Red Stock contact was intersected at 81 metres and the South Boundary Fault cut between 81 and 134 metres. The fault consisted of angular fragments of intensely silicified Late Phase and Main Phase rock in a matrix of light grey intense clay-fault altered granulated wall rock. At 134.1 metres the clay alteration ends and the matrix is composed of quartz-sericite altered intrusive, 5-7% disseminated pyrite and low copper values. The estimated width of this south dipping fault is

50 to 85 metres.

The regional structures that controlled the emplacement of the Red Stock have been repetitively reactivated before, during, and after the structurally controlled mineralizing event that formed the various Red Chris mineral zones. The above described South Boundary Fault was one of the latest fault movements but essentially parallels the 'East Zone Fault' and is probably a reactivation of the original structures that controlled the emplacement of the Red Stock.

7.6 Geologic Model

The 2003, 49 hole diamond drilling program undertaken by Red Chris Development Corp, on the Red Chris property, was designed to upgrade the known drill resources and verify the previous geologic model. In December 1995, D. Blanchflower and engineers of Fluor Daniel Wright Ltd. prepared a geological model of the Red Chris deposit which encompassed the lateral and vertical limits of the known mineralization. As a large scale open pit operation was planned, only Late Phase dykes exceeding 10 metres in thickness were modelled as internal waste, and no narrow barren dykes or structural features, such as shear and fault zones, were distinguished.

The model was refined by Blanchflower and Giroux in 1998 when the then property owner, American Bullion Minerals Ltd., contracted them to identify the detailed geological and structural features that control the higher grade copper-gold mineralization in the Red Chris deposits. American Bullion had been re-evaluating the earlier resource model in order to plan a more selective and potentially lower cost innovative open pit mining operation. Blanchflower, in his November 18, 2002 report for American Reserve Energy Corporation, describes the geologic model used in this study as follows:

"It has long been recognized that the Red-Chris copper-gold mineralization has good near-vertical and longitudinal continuity, controlled largely by post-mineral faulting superimposed on and along the ancestral, en echelon, central axis fault zone, commonly referred to as the 'East Zone Fault'. Furthermore, some post-mineral dykes, such as the quartz-eye porphyry variety, are recognized as occupying the same structural features that controlled the late-stage and higher grade copper-gold mineralization. Thus, detailed modelling of the deposit incorporated the local geologic and structural features unique to each of the Main and East zones.

For modelling purposes, American Bullion provided vertical sections and plan views spaced at 50 metres and 15 metres respectively; covering the entire Red-Chris deposit. It was decided to subdivide the copper mineralization into three mineralized domains. These domains were designated as being: 'Inner Core' which comprised most copper mineralization equal to or exceeding a grade of 0.4 percent copper; 'Outer Shell' which comprised most copper mineralization ranging from 0.2 to 0.4 percent copper; and 'Main Phase' which comprised the remainder of the Red stock and included mineralization with grades dominantly less than 0.2 percent copper. The distribution of the gold mineralization was not modelled because it is intimately and proportionately associated with the copper mineralization.

The factors controlling the distribution of the three mineralized domains were

identified as being: structural features (i.e. post-mineral faults and shear zones), geological features (i.e. Main Phase host rock versus post-mineral Late Phase and other dykes) and the distribution of mineralization based upon assay results. These three factors were all considered in outlining the boundaries of the 'Inner Core', 'Outer Shell' and 'Main Phase' mineral domains."

The 2003 in-fill diamond drilling program was concentrated on the core areas of the East and Main mineral zones and the drill holes were sited between the previous 50 metre drill sections and were often orientated northerly rather than to the south, as had been the case with previous programs. The additional in-fill drilling data allowed for the construction of vertical sections on 25 metre centres along the axis of the Red Chris East and Main Zones. This work is ongoing at the time of this report writing but sufficient data have been compiled to provide an updated geologic model for the 2004 resource calculations.

It was decided to impose a hard boundary around the East Zone core mineralization similar to the "Inner Core" domain constructed by American Bullion in 1998. The 2003 drilling provided more data to incorporate into the detailed construction of the 2004 domain shells. The "Inner" shell outline, which comprises most copper mineralization equal to or exceeding a grade of 0.3 percent copper, was drawn on vertical cross sections constructed on 25 metre centres. Because of the abrupt termination in copper grade in the East Zone, this grade shell often matched the 0.4% shell used by American Bullion. The digitized, vertical section grade shell was transferred and smoothed onto plan views spaced on 15 metre levels down to the 1145 metre elevation. Below the 1145 metre level, the plans were produced on 30 metre intervals to the model base at 900 metres. A solid model of the core zone was then constructed from the level plan boundaries.

In the East Zone, a second copper grade shell was placed around the copper mineralization grading from 0.2 to 0.3 percent copper and the entire zone was constrained by the Bowser sediment contact to the south and the Dynamite Hill volcanics to the north. A small isolated 0.2 shell 'Satellite Zone' was put around drill holes DH 119 and DH 184 on section 51,000 E to restrain their influence on blocks within the 0.2 percent shell surrounding the "Inner Core" domain.

The new geologic model developed for the Main Zone differed significantly from that used in the previous 1998 study. The 2003 Main Zone drilling results indicated there was no need to define a higher grade "Core Zone" boundary or limit the copper mineralization domain with a 0.2 copper percent grade shell. Like the East Zone, the Main Zone was constrained by the Bowser sediment contact to the south and the Dynamite Hill volcanics to the north. Late Phase monzodiorite dykes and other dykes whose widths were greater than 10 metres were digitized using the new cross sections and solid models of the dykes were developed and used as hard boundaries during the block modeling. These barren dykes frequently occupy the core structures that controlled the higher grade copper-gold mineralization in the Main Zone.

The geologic model for the Main Zone remained relatively unchanged from the previous version (Giroux, *et al.*, 2004). Only minor corrections were made to the boundaries of the southern dyke complex which was intersected by four of the new drill holes. Slight adjustments were also made to the contacts of the Red Stock which were intersected by two of the 2004 drill holes.

For the 2004 estimate, the division between the Main and East zones at 50650 E was changed to a soft boundary meaning data from one side could influence blocks on the other.

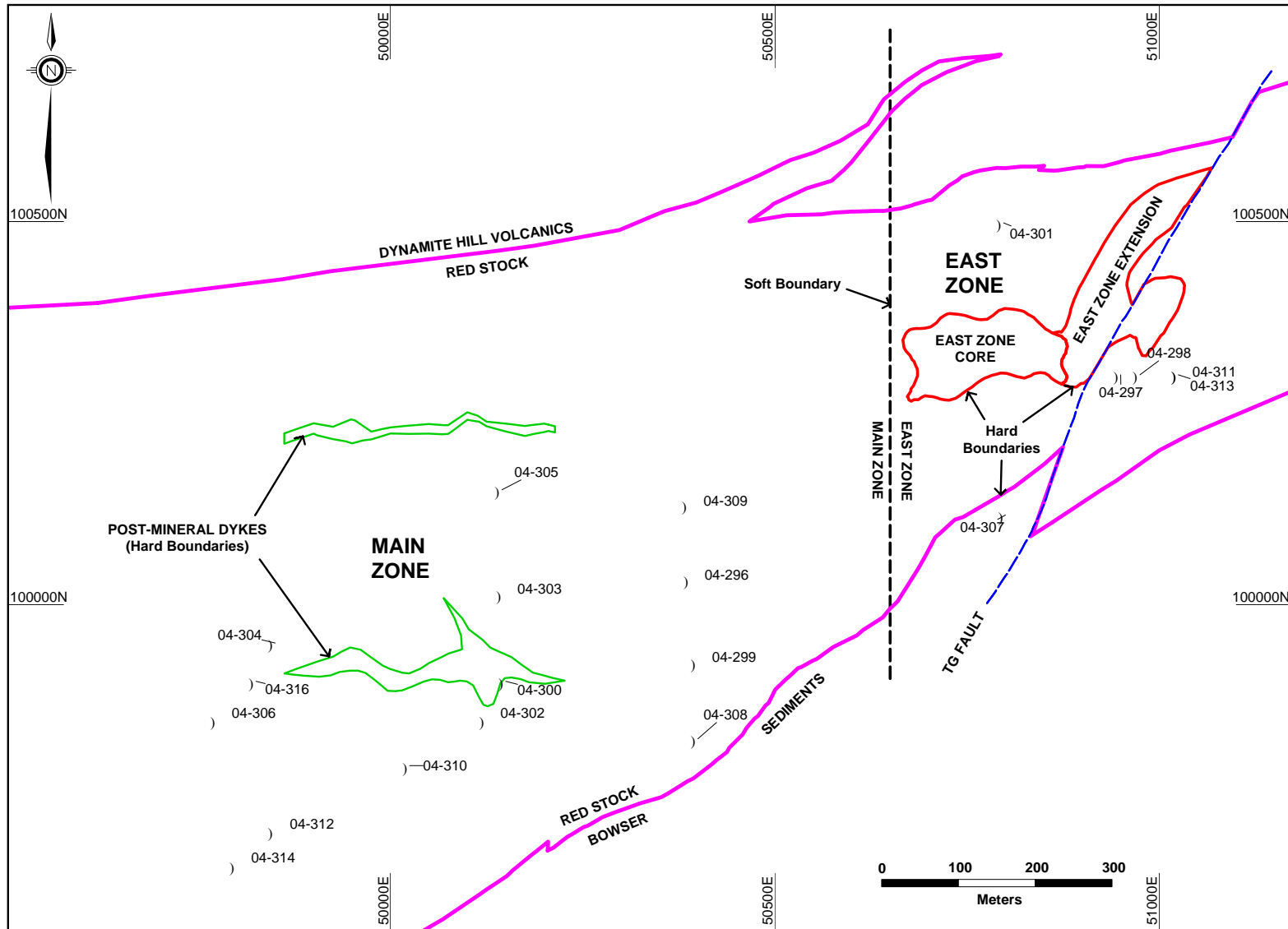
The East Zone outer core grade shell surrounding the East Zone high grade core was removed and a revised grade shell was imposed on the eastern side of the East Zone core. The reason for the removal of the lower-grade shell was the belief that stricter limitations imposed on the kriging procedure would provide a more realistic simulation of the grade distribution.

Drilling in 2004 showed that the mineralization was continuous between the former Satellite Zone and the old Outer Core model. Therefore, a new grade shell termed the 'East Zone Extension' was modeled based on an approximate 0.2% Cu cutoff and was used as a hard boundary in the resource estimation. This area is cut by a northeast-trending structure referred to as the TG Fault which was intersected by the 2004 drilling. Since two drill holes intersected continuous mineralization within and adjacent to this fault, it was not used as a hard boundary within the zone.

The East Zone Extension does contain significant drill intercepts in excess of 0.5% Cu but there is no clear division between high and lower grade material as is evident in the East Zone core area. There is very limited drill information in this area below the 1250 metre level and the boundaries of the grade shell below this level were assumed to be vertical.

No new drilling or interpretation was carried out on the Far West and Gully zones so the geologic models and resource estimates in these areas remains unchanged.

Figure 9 - Position and Type of Geologic Model Boundaries



8.0 DEPOSIT TYPES

The Red-Chris copper-gold deposit has genetic characteristics of both the alkalic and calc-alkalic suites of volcanic porphyry copper deposits in the Canadian Cordillera. The following table, modified after Schink, (1977) and Ash *et al.*, (1995), illustrates these ambiguities.

Table 3 - Porphyry Characteristics

	Alkalic Suite	Calc-Alkalic Suite	Red-Chris Deposit
Intrusive Host Rock	Diorite, Monzonite Syenite Granodiorite	Quartz Diorite	Monzodiorite
Host Rock Geochemistry	Alkalic; high K/Na ratio; high alkali/ silica ratio	Calc-alkalic; low K/Na ratio; low ratio; alkali/silica ratio	Calc-alkalic; low /Na moderate alkali/ silica ratio
Morphology of Host Intrusive	Volcanic	Plutonic, Phallic	Volcanic
Level of Intrusion	Epizonal	Mesozonal	Epizonal to hypabyssal
Country Rocks	Generally potassic volcanic rocks and volcanic rocks	Generally calc- alkalic plutonic	Sodic and potassic volcanic rocks
Alteration Types (core to rim)	Potassic, Propylitic	Potassic, Phyllic Argillic, Propylitic	Potassic, Argillic, Phyllic, Propylitic
Position of Ore in Alteration Sequence	Potassic, Propylitic	Potassic, Phyllic	Potassic, Argillic
Associated Metals	Gold, Silver	Molybdenum, Silver, minor Gold	Significant Gold; minor silver; rare molybdenum
Style of Mineralization	Sulphide fracture fillings, massive lenses and breccia	Quartz-sulphide vein stockwork breccia	Quartz-sulphide vein stockwork, silicified zone
Grade Distribution	Moderately erratic	Consistent	Moderately consistent
Relative Size of Deposit	Small to Moderate	Moderate to Large	Moderate to Large

The classification of the Red-Chris deposit, as to its genetic porphyry copper suite, remains the subject of debate. Newell and Peatfield, (1995) tend to place it in the alkalic suite of volcanic porphyry copper deposits and conclude that the calc-alkalic features are the result of secondary processes, such as the influence of oceanic waters on the hydrothermal fluids.

9.0 MINERALIZATION

Pyrite, chalcopyrite and lesser bornite are the principal sulphide minerals of the Red-Chris deposit. Minor covellite occurs as inclusions in pyrite, and molybdenite, sphalerite and galena occur locally in trace amounts. Gold, second in economic importance to copper, occurs as electrum spatially- and genetically-associated with the copper mineralization. Gold was observed in two samples by T. Fraser (Ash *et al.*, 1994). Silver values are geochemically significant but are of minor economic importance.

Pyrite occurs commonly as very fine- to fine-grained, anhedral to euhedral disseminations or fracture fillings. Within the mineralized zones it is commonly poikilitic with numerous copper sulphide and iron oxide inclusions, while elsewhere the inclusions are commonly sericite and dolomite. The pyrite content usually varies disproportionately with quartz vein stockworks. It ranges from 5 to 15 percent in Late Phase rocks, 2 to 4 percent in Main Phase rocks with very weak to weak quartz veining, and often less than 1 to 2 percent in well mineralized Main Phase rocks with moderate to intense quartz stockworks. Pyrite (\pm chalcopyrite) veins cut quartz vein stockworks, and are often associated with narrow hematite veinlets. The partial replacement of mafic phenocrysts and, to a lesser degree, plagioclase phenocrysts is occasionally seen. Pyrite occurs in the Dynamite Hill Volcanics up to 100 to 150 m from the intrusive contact, and occurs as disseminations and fracture fillings in the sedimentary country rocks up to 300 metres from the Red stock north of the Far West Zone.

Chalcopyrite is most abundant in the quartz-sulphide vein stockworks and quartz-sericite-ankerite alteration selvages. Its content is roughly proportional to the intensity of quartz vein stockwork except in the Gully and Far West zones. Beyond the quartz stockwork zones chalcopyrite occurs as disseminations, along fractures often associated with pyrite veinlets, and rarely as veinlets. In quartz veins it occurs as disseminations, aggregates, and fracture coatings and fillings both parallel to and crosscutting the quartz veins. Where quartz-sulphide vein stockwork intensity diminishes elevated copper grades remain constant due to the presence of fine-grained disseminated chalcopyrite which is associated with pyrite.

Bornite is most common as fracture fillings and fine-grained (0.5 mm) disseminations in the quartz-sulphide vein stockwork zones of the East Zone but it also occurs as fine-grained disseminations in the highly altered Main Phase rocks of the eastern Main Zone. Bornite also occurs in the Gully Zone, but is less abundant than in the Red-Chris deposit. Within quartz stockwork veins bornite occurs as disseminations and microveinlets both within their cores and as crosscutting veins. Bornite is also intimately associated with disseminations, fracture fillings and coatings of specular hematite, and with specular hematite aggregates. This association makes visual grade estimates difficult and invariably low.

Magnetite and hematite are most commonly associated with mineralized quartz stockwork zones and plagioclase-hornblende-biotite dykes where they may represent up to 10 modal percent. They usually occur as fine-grained disseminations in the veins and host rocks but they also occur as magnetite-hematite veinlets and quartz-magnetite veinlets. Magnetite typically forms fine, hexagonal grains which are usually replaced by specular and earthy hematite.

The known native gold or electrum mineralization is all microscopic. Preliminary thin section and SEM studies of the quartz-sulphide stockwork vein material discovered two grains of gold intimately associated with copper mineralization (Ash *et al.*, 1994). One subrounded gold grain occurs within a bornite grain hosted by a quartz vein and another gold grain occurs interstitially with a chalcopyrite and bornite-bearing quartz vein.

Copper to gold grade ratios (i.e. % Cu to gpt. Au) were plotted for several drill holes in the Red-Chris deposit. The results indicate that the gold-bearing mineralization is intimately associated with the copper mineralization. Copper to gold grade ratios do vary laterally in a

westward direction from 1:0.8 within the Red-Chris deposit, to 1:2 or 1:2.5 within the Gully Zone, and to 1:3 or locally 1:4 within the Far West Zone. This westward transition coincides with increased pyritization, decreased bornite versus chalcopyrite mineralization, and the dominance of phyllic versus potassic-phyllic alteration of the host rocks. Thus, it appears that the alteration and mineralization was ‘telescoped’ along the axis of the Red stock in a westward direction rather than being equidimensional like a stereotypical porphyry copper-gold deposit.

Prominent limonitic gossans occur within the East and West Gully drainages and along their steep slopes. However, in areas of low relief, such as over the Red-Chris deposit, weak limonite only extends 1 or 2 metres beneath the bedrock surface. The gravel till layer overlying the bedrock is often very limonitic or ferrocrete. Thus, it appears that Recent glaciation has removed any of the supergene mineralization that might have existed over the Red-Chris deposit. However, Great Plains Development reportedly intersected supergene chalcocite mineralization in shallow drilling near the headwaters of the East Gully drainage, and recent drilling in the vicinity has confirmed the possibility of chalcocite mineralization in near-surface fractures within the oxidized layer. Chalcocite occurs along with malachite, azurite and manganese oxides in this oxidized zone. It is possible that there may be other graben-like structures elsewhere within the property where supergene copper mineralization might have been preserved after continental and alpine glaciation.

10.0 EXPLORATION

The 2004 field exploration program was designed to infill previous work. Complete details are discussed in Section 6 History and Section 11 Drilling.

11.0 DRILLING

11.1 Exploration Drilling

11.1.1 Pre 2003 Drilling

Prior to 2003, there have been three major drilling campaigns conducted on the property; the percussion and diamond drilling undertaken by Texasgulf in the 1970's and the two diamond drilling programs conducted by American Bullion Minerals in 1994 and 1995. In addition to these major campaigns, there has been a number of much smaller drilling programs undertaken by various earlier operators. Blanchflower *et al.*, (2002) have described the previous drilling programs as follows:

Prior to American Bullion Minerals' ownership in 1994, the Red-Chris property had been drilled by:

- 1 Conwest Exploration Limited, 1956 - several short x-ray diamond drill holes;
- 2 Great Plains Development Company of Canada, Ltd., 1970- 2 diamond drill holes; 1972 - eight diamond drill holes totalling 922 metres;

- 3 Ecstall Mining Limited, 1973 - 14 percussion drill holes totalling 914 metres; and
- 4 Texasgulf Canada Limited (Texasgulf Inc.) 1974-1976 - 67 diamond drill holes (12,284 m) and 30 percussion drill holes (2,261 m); 1978 and 1980 - 7 diamond drill holes totalling 1,017 m).

Due to poor documentation, the pre-1973 drill holes could not be located accurately. Thus, the total pre-1994 drilling which could be located is as follows (Rebagliati, 1994).

Table 4 - Pre-1994 Drilling

Year	Percussion Drilling		Diamond Drilling	
	Holes	Metres	Holes	Metres
1973	14	914		
1974	10	780	16 BQ	2,265
1975	20	1,481	33 BQ	6,925
1976			18 BQ	3,094
1978			5 BQ	391
			2 BQ	626
Total	44	3,175	74 BQ	13,301

In 1994, American Bullion Minerals Ltd. contracted J. T. Thomas Diamond Drilling Ltd. of Smithers, British Columbia to provide equipment and personnel capable of completing a minimum of 15,000 metres of HQ- and/or NQ-core diamond drilling. The drill rigs, rods, and support equipment were all mobilized to the property in June via the tote trail from the Coyote Creek-Ealue Lake road. A Caterpillar D6E bulldozer and a Caterpillar 210B excavator were utilized to tow the rigs and equipment to the property. They were later used to excavate drill sites, access roads and construction sites, and reclaim those surface disturbances and many of the open trenches dating back to the early 1970's.

Due to encouraging drilling results, the diamond drilling contract was extended in early October. Thus, fifty-eight (58) drill holes were completed during the 1994 exploration program, totalling 21,417.08 metres or 70,266 feet. The first 1994 hole was labelled '75' following the last Texasgulf drill hole which was labelled '74'. Therefore, the 1994 drill holes are numbered consecutively from 75 to 132; all with a '94-' prefix.

Two Longyear skid-mounted, unitized drilling rigs, namely a 'Super 38' and a '44', were used for the entire drilling campaign. The early season drilling recovered HQ-core but after identifying substantial copper-gold mineralization beneath the 1,200 m elevation all subsequent holes were drilled deeper; often beyond the limits of a Longyear 'Super 38' drill rig to recover HQ-size core. Consequently, several of those holes that were drilled by this rig had to be reduced to NQ-size core for completion. Due to the prevailing marshy ground conditions, a Hughes 500D helicopter was used extensively to service the drilling rigs, move drilling personnel, and move the drill core to the logging and sampling site. At the end of the 1994 drilling campaign, all of the heavy equipment was stored on site for the next field season.

The 1995 diamond drilling commenced on May 5th and was completed on November 8th, 1995.

One hundred and twelve (112) HQ- and/or NQ-core exploratory diamond drill holes (36,770.46 m or 120,638 ft.) and three (3) BQTK-core geotechnical diamond drill holes (59.44 m or 195 ft.) were completed during this period, totalling 36,830.00 metres or 120,833 feet. The first 1995 hole was labelled '133' following the last 1994 drill hole which was labelled '132'. Therefore, the 1995 exploratory drill holes are numbered consecutively from 133 to 244, and the three geotechnical drill holes were labelled BH 95-1 to -3.

The 1995 diamond drilling program successfully discovered copper-gold mineralization across the width of the Red stock and over a 400-metre strike length west of the known Red-Chris deposit. Exploration drilling over a 2-kilometre strike length, west of the deposit, also discovered significant copper-gold mineralization underlying the Gully and Far West exploration targets which were identified in 1994.

Most of the 1995 diamond drilling in the Main and East zones of the Red-Chris deposit was concentrated along the northern, southern, and western margins of the deposit. In 1994, diamond drilling had shown that the Main and East zones are not discretely mineralized bodies but comprise a continuous zone of copper-gold mineralization that has been locally intruded by post-mineral dykes and slightly displaced by younger faulting. In 1995, diamond drilling tested the Red-Chris deposit from the southern to northern contacts of the Red stock and for more than 500 metres along the western strike extension of the Main Zone. It also tested the vertical continuity of the mineralization to a depth of over 750 metres.

Diamond drilling along the southern margins of the Red stock intersected copper-gold mineralization south of the previously-assumed limits of the Red-Chris deposit. More importantly, the copper (%) to gold (g/t) grade ratios of this mineralization varied locally from the deposit average of 1:0.8 to ratios of 1:1 or 1:2. These results indicate that there was probably a later structurally-controlled gold-bearing mineralizing event superimposed on the earlier more-pervasive copper-gold mineralization. Furthermore, this event was probably related to reactivation of the South Boundary fault structure since the higher grade gold-bearing mineralization appears to be spatially-related to this structure.

Copper-gold mineralization occurs throughout the Red stock but appears to decrease in grade near the northern intrusive contact of the stock; although this margin is still only sparsely tested along its strike length. There appears to be a zone of either poorly mineralized Main Phase or barren Late Phase plutonic rocks between the Red-Chris deposit and the intrusive contact of the stock with the Late Triassic Dynamite Hill volcanic strata. The width of this poorly-mineralized margin appears to vary from 50 to more than 100 metres and may be related to the proximity and distribution of pre-mineral fault structures along the axis of the stock. It is also noteworthy that propylitically-altered volcanics only occur over a very narrow width, usually less than 100 metres, along the northern margins of the intrusive contact. Beyond this narrow band the Dynamite Hill volcanic strata are only regionally metamorphosed to lower green schist facies and host less than one percent pyrite. Such a narrow alteration band indicates that the structural features controlling the alteration and mineralization of the Red-Chris deposit were largely restricted to the axis of the stock and did not pervade the older volcanic strata to the north.

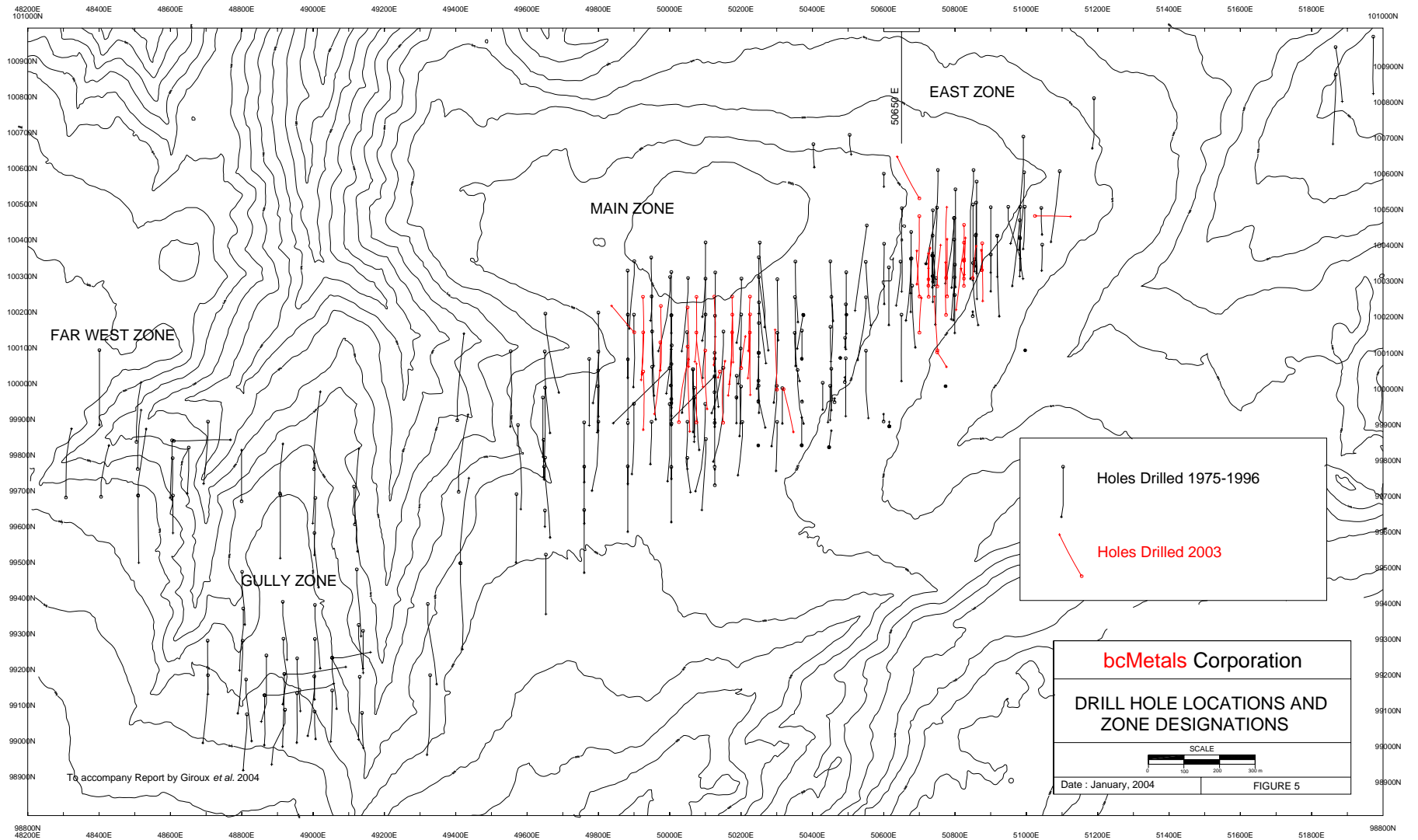
One of the most important results of the 1995 diamond drilling program was the discovery of the

western extension of the Red-Chris deposit. Diamond drilling by Texasgulf had indicated that the Main Zone might be truncated at a north-northwesterly fault structure situated near grid line 49800 East. Two 1994 drill holes (i.e. 94-123 and 94-124) tested for buried mineralization near this fault structure and found that the mineralization might have been down-dropped and displaced laterally by the fault structure. Further drilling was recommended west of this structure to test for mineralization trending northwesterly from the Main Zone (Blanchflower, 1995). This drilling discovered that the western mineralization of the Red-Chris deposit probably splits into two relatively-distinct bodies west of the fault structure and that these bodies, although displaced by westside-down, strike-slip faulting, do continue to at least grid line 49550 East. At this grid easting the mineralization is beneath grid northings 99900 and 99700, and buried from 300 to 350 metres beneath the surface. This deep copper-gold mineralization may not be readily amenable to open pit mining but the intervening nearer-surface mineralization increased the geological resources of the Red-Chris deposit (see Giroux, 1996).

Drill holes 95-140 and 95-145 were drilled in the East Zone to test the vertical continuity of its higher grade copper-gold mineralization. Drill hole 95-140 was collared at grid coordinates 100600 North by 50750 East and was finally terminated at a length of 812.90 metres or approximately 750 metres vertically beneath the surface. This hole intersected 292.61 metres of mineralization grading 0.573 percent copper and 0.565 g/t. gold from 520.29 to 812.90 metres, and the last 3.05-metre section of drill core returned a grade of 0.496 percent copper and 0.59 g/t gold. Drill hole 95-145, located 100 metres due east of DDH 95-140, was terminated at a length of 599.54 metres and it intersected 0.77 percent copper and 0.80 g/t gold over 140.2 metres from 360 to 480 metres vertically beneath the surface. These results show that the copper-gold mineralization of the deposit occurs over significant vertical distances, and that the depth of the mineralization remains to be determined.

Current drilling results indicate that there are two near-surface core zones within the Main and East zones of the Red-Chris deposit that grade greater than 0.6 percent copper and 0.6 g/t. gold and are amenable for 'starter' open pit mining. These zones are separated and surrounded by a much larger, less well delineated zone of greater than 0.25 percent copper and 0.2 g/t. gold mineralization. The strike length of the Red-Chris deposit, comprising both the Main and East zones, is now in the order of 1.7 kilometres with widths ranging from 250 to 700 metres or more (see Figure 10).

Figure 10 - Diamond Drill Hole Plan



The Gully Zone is a 700-metre long by 400-metre wide coincident geochemical and geophysical anomalies centred between the East and West Gully drainages. Exploration drilling discovered two east-west trending, subvertical zones of significant copper-gold mineralization. The northern zone is centred at grid coordinates 99800 North by 49000 East, and the southern zone is centred at 99200 North by 49000 East. Both zones, although they remain open laterally and vertically, have been tested by widely-spaced drilling over strike distances of 400 to 500 metres and widths from 200 to 300 metres.

The southern portion of the Gully Zone hosts a subvertical zone of copper-gold mineralization with a tested strike length of 500 metres and widths over 300 metres. Drill intercepts within this zone typically are more than 0.3 percent copper and 0.3 g/t gold over lengths of 15 to more than 300 metres. There are also exceptionally high grade sections within this mineralized zone, such as the one intercepted by DDH 95-168, with grades of 1.486 percent copper and 3.266 g/t. gold over 18.29 metres.

The northern portion of the Gully Zone hosts several narrower subvertical zones of copper-gold mineralization with grades generally ranging up from 0.15 to 0.40 percent copper but with significant associated gold values, usually grading 0.20 to 0.40 g/t gold. Due to the widely-spaced drilling, the distribution and delineation of this mineralization remains to be tested.

Aside from the importance of its discovery, it is important to note that the Gully Zone mineralization generally occurs with copper to gold grade ratios averaging from 1:1.5 to 1:2.5 (i.e. percent copper to grams gold per tonne); and becomes more pyritic along the western strike extensions of the Red stock.

The Far West Zone is a 600-metre by 600-metre coincident geochemical and geophysical exploration target centred at grid coordinates 99900 North by 48400 East. It was tested with widely-spaced drill holes directed at the centre of a strong high chargeability-low, resistivity geophysical anomaly. These holes intersected gold-rich pyrite-chalcopyrite mineralization in two subvertical, easterly trending structures centred at 99800 North by 48500 East. Assay results indicate that the copper to gold grade ratios are in the order of 1:3 with copper grades typically ranging from 0.2 to 0.35 percent and gold values ranging from 0.6 to 0.75 g/t. Considerably more drilling will have to be conducted within this zone to delineate the mineralized sections and their trends.

11.1.2 2003 Drilling Program

In September 2003, Red Chris Development Corporation commenced drilling on the Red Chris project using equipment and personnel supplied by Hy-Tech Drilling Ltd. of Smithers, British Columbia. Hy-Tech initially supplied two Hy-Tech 5000 drills, which along with their drill rods and support equipment, were mobilized to the property in the first week of September. A Caterpillar D6D bulldozer was used to tow the drilling rigs and rod sloops from the staging site on the Ealue Lake Road up the Coyote Creek-Red Chris property tote trail. A third HY-Tech 4000 drilling rig was mobilized to the property September 22nd to speed up the drilling progress and free up a rig for geotechnical drilling.

Fuel, extra rods, consumables, and ancillary drilling equipment were flown by helicopter from the Tatogga Lake Resort staging area to the property. All project personnel, camp support, fuel, and other project supplies were flown to the Red Chris camp using a Bell 206 helicopter supplied by Pacific Western Helicopters Ltd. of Prince George, B.C. The helicopter was used extensively to service the drilling rigs and transport drill core to the core logging and sampling site due to the poor road conditions caused by marshy ground and adverse weather during the project. The helicopter was based in camp for convenience and project safety requirements.

The drills, which were skid mounted, were moved from site to site using the D6D bulldozer and/or a tracked Caterpillar 320C excavator. The excavator was also utilized for positioning the drills, digging drill sumps, and reclaiming site and access surface disturbances. The light, helicopter-portable Hy-Tech drills were normally pulled into position without site or access construction, thereby minimizing surface disturbances.

The drill contractor, Hy-Tech Drilling, was responsible for all down-hole surveys. Two systems were used, with the Reflex digital magnetic instrument being the primary survey system and the Accushot photo system as a backup in case of Reflex breakdown or unavailability. Twenty four of the 235 down-hole surveys were deemed unacceptable due to incorrect azimuth readings and were rejected. Blocked bits and operator error were responsible for most survey failures.

The proposed drill hole sites were surveyed in by McElhanney Consulting Services Ltd. using a total station instrument and established property grid controls. The final drill hole collar locations were surveyed by McElhanney using both a total station and a survey quality Global Positioning System (GPS). The Plant Site and Waste Dump Site geotechnical drill holes were also surveyed by GPS.

The 2003 drilling campaign commenced September 7th and finished November 7th. A total of 49 holes totalling 16,591 metres were drilled into both the East and Main Zones. This drilling included nine geotechnical, orientated core holes totalling 2,499 metres. The geotechnical holes, while drilled primarily for pit slope design purposes, also provided assay information that was used in the new resource calculations. The core size for the orientated holes was NQ3 although the upper portions of some holes were cored using HQ in order to install 2 inch groundwater monitoring wells. The geotechnical drilling was supervised by personnel from Knight Piésold Consulting and the geological logging and sampling by RCDCs' personnel.

Three of the 49 Red Chris drill holes totalling 793.44 metres were drilled vertically using HQ sized core to provide material for metallurgical grinding tests. Hole 03-256A (hole 03-256 was lost at 57.16m) was drilled in the core of the East Zone and Hole 03-283 in the centre of the Main Zone. A 2 inch groundwater monitoring well was installed in the latter hole. The two holes were geologically logged and a 15 cm whole core sample was taken from each assay interval for metallurgical testing. The remaining core was then split and the half-cores sent in for assay. The analytical results were used as the basis for the January 19th, 2004 resource update.

The remaining NQ2 sized diamond drill holes were sited to in-fill drilling gaps in the Main and East Zones produced by the 1972 to 1995 drilling campaigns. The first hole in the 2003 program was labelled DDH 03-248 and follows consecutively from the last 1995 exploratory drill hole

05-244 plus the three BQTK-core geotechnical diamond drill holes BH 95-1 to -3. The last hole drilled in the 2003 program was hole DDH 03-295.

11.1.3 2004 Drilling Program

The 2004 drilling program was a similar one to that conducted the previous year. During the period June 10 to August 8, 2004, Red Chris Development Corporation completed 25 diamond drill holes with an aggregate length of 6,927.6m on the Red Chris property. Five of these holes were condemnation drilling on the site of the proposed open pit waste dump, four holes were drilled for rock mechanics studies, and the remaining sixteen holes were infill drill holes designed to increase the quantity and quality of resources within the proposed pit prior to a final feasibility study.

The condemnation drilling tested an induced polarization anomaly that lies to the east of the open pit. Four earlier holes into this anomaly failed to intersect copper mineralization, and only traces of chalcopyrite were intersected in the 2004 drilling. The lack of quartz veining and significant alteration suggests that the area is not mineralized.

One new mineral, fluorite, was noted in logging core from infill drilling in the southwest part of the Main Zone. It occurred in two locations in drill hole 04-304 as coarse grained clumps of crystals associated with gypsum veining. The gypsum veins are later than all other veining in the deposit and therefore the fluorite probably has no direct connection with the copper and gold mineralization.

The sample quality control system used was the same as that established for the 2003 work on this property.

11.2 Geotechnical Drilling

The purpose of the geotechnical drilling was to formulate the design parameters for the proposed pit slopes, Plant Site foundations, Waste Dump, Low Grade Ore Pile, and the Tailings Storage Facility prior to the commencement of a feasibility study on the Red Chris project. Knight Piésold Ltd, under contract to Red Chris Development Company Ltd. designed, supervised and documented the various geotechnical programs.

In the 2003 fall drilling program, nine geotechnical, orientated, inclined core holes totalling 2,499 metres were drilled within the proposed pits centred on the East and Main zones of the Red Chris copper-gold porphyry deposit. This geotechnical drilling was designed to:

- 1 fully log the core for geotechnical properties;
- 2 determine the locations of the contacts between the rock units;
- 3 determine the frequency, locations, and orientations of the key structural discontinuities in the rock units;
- 4 obtain samples for field and laboratory shear and strength testing;
- 5 carry out in-situ packer (Lugeon) permeability tests;
- 6 install 1" standpipe piezometers and 2" groundwater monitoring wells in selected holes;

The holes were strategically located and orientated to intersect the anticipated pit walls at different orientations and in different rock units. As these holes were also geologically logged and sampled by RCDC personnel they are numbered within the DDH 03 series of holes and were sent for assay following the routine sample shipment and analytical procedures implemented by RCDC.

Knight Piésold Ltd also designed and supervised geotechnical drilling programs in the areas of the proposed Waste Dump Site, Plant Site, and Tailings storage Facility.

The geotechnical investigation at the proposed Waste Dump site involved drilling two shallow (approximately 35 metres in total) inclined drill holes and excavating 6 test pits in an area underlain by volcanics belonging to the Middle to Upper Triassic Stuhini Group. The HQ size drill holes were drilled by a HY-Tech 5000 rig that was pulled over snow to the drill sites. Both drill holes were subsequently equipped with a 2” groundwater monitoring well. The core was geologically logged and photographed but not sampled. The geotechnical core is stored in racks adjacent to the core logging facility.

On November 5th an inclined condemnation drill hole (03-WS-3) was drilled to 18.3 metres within the proposed north east limits of the Waste Dump site. A demobilizing Hy-Tech 5000 drill rig drilled this hole under the supervision of the author. The hole encountered volcanics belonging to the Middle to Upper Triassic Stuhini Group under a thin veneer of overburden. On November 8th the author supervised the digging, using the “Cat” 320C excavator, of three condemnation test pits along the southeast margin of the proposed Waste Dump site. Two of the pits encountered Stuhini Group volcanic sediments and the third bottomed in glacial fluvial sand.

The Knight Piésold-supervised geotechnical program at the proposed Plant Site included the drilling of two shallow vertical drill holes (approximately 35 metres in total) with one of HQ size to accommodate a 2” groundwater monitoring well and the other NQ2 size to accommodate a 1” standpipe piezometer. Two test pits, in shallow overburden, were also excavated at the Plant Site. The test pits and drill holes intersected Middle Jurassic, Ashman Formation finely bedded siltstones intercalated with a few thin beds of chert-pebble conglomerate. The overburden depths were from 1 to 4 metres in the proposed Plant Site.

On October 15th a Hy-tech 3000 light weight, helicopter portable drill rig was flown to the site of the proposed Tailings Storage Facility. Four geotechnical HQ3 size diamond drill holes, totalling 210 metres, were advanced in deep overburden at sites chosen by Knight Piésold. Soil samples were recovered using a HQ3 split inner tube and minimal water pressure. None of the holes encountered bedrock (deepest hole 77.1 metres). Two 2” groundwater monitoring wells and two 1” standpipe piezometers were placed into the four geotechnical holes. In addition, a total of 16 test pits were excavated by the CAT 320C hydraulic excavator into the upper layers of overburden at selected locations at the site of the Tailings Storage Facility.

12.0 SAMPLING METHOD, APPROACH, and SECURITY

12.1 Sampling Method, Approach and Security – pre 2003

(taken from Blanchflower *et al.*, (Nov. 18, 2002)).

All of the 1994 and 1995 diamond drill core was properly handled, processed, logged and sampled on site (Blanchflower, 1995 and 1996). After the drill core had been delivered by helicopter to the logging and sampling facility its footage markers were converted to metric measurements and each box was properly labelled with its respective hole number, box number and drilling length interval. The core was then logged in detail by qualified geologists, employed by American Bullion Minerals Ltd., utilizing a ‘matrix’ coding log form. Geological data were then input into a computerized database for both on-site documentation and computer-assisted drafting (CAD). Core recovery, rock quality and specific gravity measurements were also logged and recorded. Core recoveries were generally good (>90%) to excellent (98 - 100%); except in extremely fractured near-surface rock or wider fault structures. Specific gravity measurements were recorded at 8-metre intervals. All the drill core was photographed prior to sampling.

Following the logging procedures, the drill core was split in half lengthwise using a Longyear manual splitter and sampled between drilling length blocks; usually at 3.05-metre or 10-foot intervals. A duplicate sample of every twentieth sample was inserted into the sampling sequence as a ‘blind’ check-assay sample duplicate. All of the samples were then labelled, double-bagged and flown to a secure landing and collection site at Tatogga Lake Resort, situated on Highway 37. There, the samples were hand-loaded into a secure ‘box’ van for shipping to Min-En Laboratories’ preparation facility in Smithers, British Columbia. The remaining one-half of the split core is stored at the core logging and storage facilities on the property.

12.2 Sampling Method, Approach and Security –2003 drilling

The drill core from the fall 2003 diamond drilling program was handled, logged, photographed, and sampled at the Red Chris drill camp. Most of the drill core was transported, from the drill sites to the core logging facilities, by helicopter due to the poor condition of the marshy roads and the need to prevent undue ground access disturbances. At the logging and sampling building, the core was handled and logged by qualified geologists. First core box footage markers were converted to metric, then the boxes were labelled with hole number, box number, and the contained core interval. The core recoveries and rock-quotient-density measurements (RQD) were done concurrently with the labelling. Qualified geologists, on contract to Red Chris Development Company, then geologically logged the drill core for rock type, alteration, structure, and mineralization. The logging forms were similar to those used by American Bullion in the 1995 drill program. The geologists then laid out the sample intervals with assigned assay tags, inserted the standards, and digitally photographed the core in four box groups.

The length of the sampled interval depended upon geological rock contacts, core size, and changes in mineral intensity, but generally averaged 3 metres with NQ2 core and 2 metres with HQ. As core recovery generally approached 95 to 100% there were no problems with recovery that could materially impact the accuracy or reliability of the sampling method. To check that the longitudinal half splitting of the drill core could produce an unbiased sample, 32 half-core assay

intervals from archived core drilled in the East Zone and 30 half-core intervals from the Main Zone were collected, analysed, and the results compared with the original submitted half of the core. The results, which indicated a close comparison, are discussed in the quality control section of this technical report by A.J. Sinclair, P. Eng.

During the core logging, RCDC geologists collected, for specific gravity testing, 134, 15 cm long core samples representing the various Red Chris rock types. The samples were wrapped and shipped in plastic buckets to the University of British Columbia Mining and Mineral Process Engineering Group for laboratory testing. Knight Piésold Ltd provided oversight and input into the specific gravity testing.

The standards (high copper, medium, low, and blanks) were inserted at every twentieth sample interval by the logging geologist. After being photographed with the tagged assay intervals, the core was split in half lengthwise using a Longyear manual splitter and half the sample, between the assay interval tags, was placed in labelled, tagged, double-bagged plastic sample bags. Up to 5 consecutively numbered sample bags were then placed in a white numbered and labelled “rice sack.” The consecutively numbered “rice sacks” were then sealed with a randomly numbered security tag and were flown by helicopter in sling loads of 10 to a secure landing site at Tatogga Lake Resort which is located on Highway 37.

At Tatogga Lake, an RCDC contractor took possession of the samples, palletized each sling load, wrapped them in shrink-wrap, numbered the pallets and moved the pallets into a locked 20 foot steel container using a Bobcat. Twice a week, Bandstra Trucking of Smithers would pick up the pallets, under the supervision of a RCDC representative, and ship the samples, in a closed truck, directly, via Smithers, to International Plasma Labs in Vancouver. The assay lab recorded the number and arrival time of the sample shipment and noted the security tag numbers. The total weight of samples shipped to IPL was greater than 45 tonnes. The remaining half core samples are stored in their labelled core boxes in newly constructed core storage racks located on the Red Chris property.

12.3 Sampling Method, Approach and Security –2004 drilling

The sampling method, approach and security procedures for the 2004 drilling program were identical in all respects to that outlined under the 2003 program.

13.0 SAMPLE PREPARATION and ANALYSES

13.1 Sample Preparation and Analyses –pre 2003

(taken from Blanchflower *et al.*, (Nov. 18, 2002)).

The Red-Chris exploration database (1974 to 1995 diamond drilling results) includes geological, geotechnical, geochemical and assay data for 244 diamond drill holes and 3 geotechnical bore holes (71,531.26 m). The database does not include any of the percussion drilling data because of poor documentation and undefined sampling procedures. The database contains: 4,453 geological observations, 22,731 diamond drill core samples assayed for copper (%) and gold

(grams per tonne), 6,934 specific gravity measurements, 19,669 rock quality observations, 2,458 ICP analyses, and 546 soil geochemical sample results.

A total of 5,058 copper assays were documented from the pre-1994 diamond drilling; however, 1,000 samples were not assayed for gold and many of the 1,500 gold assays were conducted on composite samples with intervals up to 15 m in length. Copper values from the various drilling programs range from less than 0.01 to 6.60 percent, and gold values ranged from 0.017 gram per tonne to 8.228 grams per tonne. Sixty-eight samples were assayed for silver and the values range from 0.686 to 10.286 g/t (Rebagliati, 1994).

A total of 19,055 diamond drill core samples were shipped to Min-En Laboratories in Smithers during the 1994 and 1995 exploration programs. Of this total, 17,031 drill core samples were assayed for both copper and gold, and copper and gold geochemical analyses were conducted on the remaining 2,024 drill core samples where no obvious copper mineralization was visible, such as within wide post-mineral dykes or Bowser Lake Group strata. However, all of the bedrock diamond drill core was either assayed or analyzed.

In Smithers, Min-En Laboratories' personnel dried each sample at 60 ° C. before crushing it to minus 1/4 inch. The crushed sample was then reduced to minus 1/8 inch size by a secondary roll crusher. The whole sample was then split on a Jones Riffle to a statistically-representative 300-gram sample pulp. This sample pulp was then pulverized in a ring pulverizer to 95 percent minus 150 mesh, rolled and bagged. All of the sample pulps were then shipped to the Min-En Laboratories facility in Vancouver, British Columbia for assay. The remaining coarse rejects from the Jones Riffle were bagged, catalogued and stored in a J. T. Thomas Diamond Drilling warehouse at Smithers, British Columbia.

All of the drill core sample pulps were assayed or analyzed initially for their copper and gold values. Min-En Laboratories' assay procedures for copper use a 0.500 to 2.00 gram sub-sample which is weighed from the sample pulp for analysis. Each batch of 70 assays has a natural standard and a reagent blank included. The samples are digested using a HNO₃ - KClO₃ mixture and when the reaction subsides, HCl is added before it is placed on a hotplate to digest. After digestion is complete the flasks are cooled, diluted to volume and mixed. The resulting solutions are analyzed on an atomic absorption spectrometer using the appropriate standard sets. The natural standard digested along with this set must be within 2 standard deviations of its known or the whole set is re-assayed. If any of the assays are more than 1 percent copper they are re-assayed at a lower weight, and 10 percent of the submitted samples are assayed in duplicate (Min-En Laboratories, 1995). During the 1994 and 1995 programs, 17,031 drill core samples, 2,143 'blind' duplicate samples and 3,252 assay laboratory standards (i.e. 813 samples of each of the AM-A, AM-B, STD and BLK laboratory standards) were assayed for copper. The 'blind' duplicate samples were inserted into the sample sequence by the field geologists.

The remaining 2,024 drill core samples that were geochemically analysed for copper were treated differently. After the samples were dried at 65° C., they were crushed by a jaw crusher and pulverized by a ceramic-plated pulverizer or ring mill pulverizer. The resultant sample was rolled and sieved to obtain a minus 80-mesh pulp for analysis. A 0.5 gram sub-sample was digested for 2 hours with an aqua regia mixture and, after cooling, the solution was diluted to

standard volume. The resultant solution was then analyzed for its copper content by atomic absorption methods. The copper values are quoted as parts per million (ppm).

Gold fire assays were conducted on 17,031 drill core samples and 2,143 'blind' duplicate samples. All gold fire assay procedures at Min-En Laboratories were conducted on one assay ton sample weights. The sub-samples were fluxed and a silver inquart was added and mixed. These sub-samples were fluxed in batches of 24 assays with a natural standard and a blank. This batch of 26 assays were carried through the whole procedure as described. After cupellation the precious metal beads were transferred into new glassware, dissolved with aqua regia solution, and diluted to volume and mixed. The resulting solutions were analyzed on an atomic absorption spectrometer using a suitable standard set. The natural standard fused along with this set must be within 2 standard deviations of its known or the whole set is re-assayed. Likewise, the blank assay must be less than 0.015 g/t. The top 10 percent of all assays per printed page were rechecked and reported along with the standard and blank. Gold values are reported in grams per tonne (g/t) with a detection limit of 0.02 g/t.

The remaining 2,024 drill core samples were geochemically analysed for their gold values. A 10.0-gram portion from each rock sub-sample was weighed and placed into a porcelain crucible, and cindered at 800° C. for 3 hours. All of the sub-samples were then transferred to beakers and digested using aqua regia, diluted to volume and mixed. Seventy-five percent of each of the diluted samples was further oxidized, treated and extracted for gold analyses using methyl iso-butyl ketone (MIBK). The MIBK solutions were then analyzed by an atomic absorption spectrometer (A.A.) using a suitable standard set and the values of gold (Au) were then reported. Gold values are reported in parts per billion (ppb) with a detection limit of 1 ppb.

After the copper and gold assay results were reported by Min-En Laboratories, 762 selected sample pulps were delivered to Chemex Labs Ltd. in North Vancouver, British Columbia for copper and gold check-assaying. These sample pulps were re-assayed using similar procedures as those undertaken at Min-En Laboratories. American Bullion Minerals Ltd. contracted Barry W. Smee, of Smee & Associates Consulting Ltd. in Vancouver, British Columbia, to prepare a report on the analytical quality of the assay data using the assay pairs (Smee, 1995 and 1996).

At monthly intervals throughout the 1994 and 1995 exploration programs every fifth and tenth drill core sample, or 2,458 samples, were analyzed for their 31-element geochemistry using inductively coupled plasma (I.C.P.) analysis techniques at Min-En Laboratories in North Vancouver, British Columbia. The purpose of these analyses was to determine if there were any other unrecognized economic or detrimental metals associated with the known copper-gold mineralization. Thus, 2,458 samples were analysed for: silver (Ag), aluminium (Al), arsenic (As), boron (B), barium (Ba), beryllium (Be), bismuth (Bi), calcium (Ca), cadmium (Cd), cobalt (Co), copper (Cu), iron (Fe), potassium (K), lithium (Li), magnesium (Mg), manganese (Mn), molybdenum (Mo), sodium (Na), nickel (Ni), phosphorus (P), lead (Pb), antimony (Sb), strontium (Sr), thorium (Th), titanium (Ti), vanadium (V), zinc (Zn), gallium (Ga), tin (Sn), tungsten (W) and chrome (Cr).

The Min-En Laboratories' I.C.P. analytical procedures require a 0.5-gram sub-sample from the original sample pulp. This sub-sample is digested for 2 hours with an aqua regia mixture. After

cooling the sample is diluted to standard volume and the solution is analyzed by a Jarrell Ash ICP computer (Inductively Coupled Plasma Spectrometer).

13.2 Sample Preparation and Analyses–2003 Drilling

The 2003 fall, 49 hole diamond drilling program, added 6,042 assayed samples to the previous exploration drilling assay database. In addition to these samples 307 assay standards (56 blanks, 81 low grade, 83 medium grade, and 89 high grade) were also shipped from the property and analyzed as part of the initial quality control program.

The assay standards used in the 2003 drilling program were prepared for RCDC by CDN Resource Laboratories Ltd. (CDN) of Delta, BC. The initial material for the preparation of the standards was collected from the remaining American Bullion 1994-1995 drilling program rejects (Min-En sample rejects) which were loosely stored near Smithers, BC. The assay intervals selected for standards were chosen to give a low, a medium, and a high copper reference assay. CDN used this material to prepare three homogenous pulps suitable for use as assay standard reference material.

The samples from Smithers were first dried, mechanically ground and screened through a 200 mesh screen. Oversize material was reground and then re-screened. The minus 200 mesh fraction was mechanically mixed for 24 hours (tumbled end-over-end in a 50 gallon drum at approximately 12 rpm). Cuts were taken from the three standard sets and assayed by Assayers Ltd (Vancouver), to test for homogeneity. In all cases assay results were deemed acceptable for purposes of homogeneity. Random splits were taken from the sample sets for round-robin analysis. Twenty sub-samples, of each pulped standard, were sent each to Acme Laboratories, ALS Chemex, Assayers Canada, and International Plasma Laboratory Ltd; for round-robin analysis for assay copper and gold. The standards were bagged in tin-top kraft bags, in lots of approximately 100 grams and were given tear-off labels. The calculated assay values for the three sets of assay standards are 0.353% Cu (0.288 gpt Au), 0.561% Cu (0.561 gpt Au), and 0.907% Cu (0.744 gpt Au).

CDN Laboratories also supplied the drilling project with blank standards. The material for the blanks was purchased “turkey grit”, which is crushed granodiorite from local Vancouver sources used to help turkeys feed. The samples were mechanically ground, pulverized and bagged in kraft bags in lots of 100 grams. The assay standards were shipped to the Red Chris project and were inserted into the sample stream by RCDC geologists.

Metallurgical composites were taken from the East Zone and the Main Zone archived drill half-cores. The 495 composite samples were taken by quarter splitting the stored half-core over predetermined assay intervals to provide representative mill feed grades as a hypothetical pit advanced down through the East and Main Zones. More than 900 kilograms of East Zone composite samples (comprising 6 sample types) and greater than 800 kilograms of Main Zone composite samples (comprising 5 sample types) were sent to G&T Metallurgical Services of Kamloops, BC, for metallurgical test work.

An additional 67 samples, totalling 300 kilograms, and representing the various property rock

types were collected from the stored half-core and were sent to BC Research in Vancouver for acid base accounting studies. Enough sample was taken over a 15 metre core length to produce either a 2 kilogram or a 5 kilogram sample.

The 6,042 diamond drill core samples were sent directly to International Plasma Laboratory Ltd. (IPL) of Vancouver for copper and gold assay. IPL is an ISO 9002 registered analytical laboratory. Bedrock diamond drill core intersecting Ashman Formation sediments were not assayed or analyzed. One in every twentieth sample with greater than 0.3% Cu were run for 30 element ICP (AqR) analysis. In total 221 samples were tested by IPL for multi element ICP analysis.

The core samples upon arrival at IPL were sorted into batches with ascending, consecutively numbered assay tags by IPL's personnel and dried at 55-60° C overnight. The samples were then crushed to minus 10 mesh using a Rhino jaw crusher. The entire sample was then split on a Jones Riffle to a statistically-representative 250-gram sample pulp. The sample pulp is then pulverized in a TM double ring and puck pulverizer to a 95 percent minus 150 mesh then rolled and bagged. All of the drill core samples submitted to IPL were assayed for copper and gold.

International plasma Laboratories' assay procedures for copper use a 50 gram sub-sample of the 250 gram assay pulp. Each laboratory batch of 40 samples is composed of thirty six assay samples, one reagent blank, one in-house standard (5, 10, or 15 ppm Cu) and two repeats. The repeats are from the 1st and 20th batch samples. IPL uses a multi-acid (HNO₃, HCL, HCLO₄, and HF) slow hot plate digestion to digest the copper. The dried sample is then re-boiled in 5% HCL acid to dissolve any soluble matter. After digestion is complete the beakers are cooled, diluted to volume, and mixed. The resulting solutions are analyzed on an atomic absorption spectrometer using the appropriate standard sets. If any copper analysis returns values higher than the in-house standard then the sample or samples are further diluted and re assayed. In addition to the 6,042 samples sent to IPL for assay there were 307 standards submitted as part of bcMetals' initial quality control program. Copper values are reported in percent and have a minimum detection limit of 0.01% Cu.

Gold fire assays were also conducted on all the submitted samples and on rechecks returned from ALS Chemex Labs. IPL conducted their fire assay procedures on a 30 gram or one assay ton sample weight. The sub-samples were fluxed and a silver inquart was added and mixed. These sub-samples were fluxed in batches of 24 assays which included one blank and one repeat sample. After cupellation the precious metal beads were transferred into new glassware, dissolved with aqua regia solution in a hot water bath, diluted to volume, and mixed. The resulting solutions were analyzed on an atomic absorption spectrometer (AA) using a suitable standard set. Any gold assays over one gram per tonne were re-run by fire assay with a gravimetric finish rather than an AA finish. Gold values are reported in grams per tonne (g/t) with a detection limit of 0.01 g/t.

As part of RCDCs' quality control and quality assurance program 125 grams of one in every twenty samples submitted to IPL were set aside and sent to ALS Chemex. ALS Chemex then split the sample in two, returned one half to IPL for re-assay, and assayed the remaining half in-house for copper and gold.

As mentioned above, one in every twentieth sample, submitted to IPL, with a greater than 0.3% Cu assay value was run for 30 element geochemistry using inductively coupled plasma (ICP) (AqR) analysis. In total 221 samples were run by IPL for multi element ICP analyses. The 30 elements analyzed in the ICP are: silver (Ag), aluminium (Al), arsenic (As), barium (Ba), bismuth (Bi), calcium (Ca), cadmium (Cd), cobalt (Co), copper (Cu), chrome (Cr), iron (Fe), mercury (Hg), potassium (K), lanthanum (La), magnesium (Mg), manganese (Mn), molybdenum (Mo), sodium (Na), nickel (Ni), phosphorus (P), lead (Pb), scandium (Sc), antimony (Sb), strontium (Sr), thallium (Tl), titanium (Ti), vanadium (V), zinc (Zn), tungsten (W) and zircon (Zr).

The IPL I.C.P. analytical procedures require a 0.5-gram sub sample from the original sample pulp. This sub-sample is digested with an aqua regia mixture for 90 minutes in a hot water bath at ~95°. After cooling the sample is diluted to standard volume and the solution is analyzed by a Jarrell Ash 6100 Inductively Coupled Plasma Spectrometer.

13.3 Sample Preparation and Analyses–2004 Drilling

The sample preparation and analyses methods used for the 2004 drilling program are the same in all respects as those used in the 2003 program.

13.4 Quality Control and Quality Assurance

13.4.1 Introduction

Quality assurance and quality control ('QA/QC') programs began on the Red Chris Project during the 1994 drill program conducted by American Bullion and have continued through to the 2003 infill program completed by RCDC. The analytical quality of the 1994 and 1995 diamond drill programs were assessed by Barry Smee, Ph.D., P.Geo., of Smee and Associates Consulting Ltd. and presented in two separate reports (Smee, 1995 and Smee, 1996). During the 2003 drill campaign RCDC retained A.J. Sinclair, Ph.D., P.Eng. to evaluate the earlier work and comment on the 2003 QA/QC procedures and results. A complete copy of Sinclair's findings is appended to this Report (Appendix 2).

13.4.2 QA/QC Procedures

During the 1995 exploration program, 1,235 'blind' duplicate samples and 1,804 assay laboratory standards (i.e. 451 samples of each of the AM-A, AM-B, STD and BLK laboratory standards) were assayed for copper. The 'blind' duplicate samples were inserted into the sample sequence by the field geologists and samplers. In addition, after the copper and gold assay results were reported by Min-En Laboratories, four hundred and ninety-five (495) selected sample pulps were delivered to Chemex Labs Ltd. in North Vancouver, British Columbia for copper and gold check-assaying. These sample pulps were re-assayed using similar procedures as those undertaken at Min-En Laboratories. Thus, by the end of the program 491 pairs of copper assays (4 cases outside scale) and 488 pairs of gold assays (7 cases outside scale) were directly comparable for bias and precision studies by Barry Smee.

The QA/QC program instituted for the 2003 drill program was as follows:

- 3 in-house standards were prepared to reflect low, medium and high grades. Expected values for the three standards were obtained by a round robin analysis between 4 laboratories. The standards were introduced into the sampling stream to monitor sample bias.
- blank samples were analyzed with all analytical batches
- pulps from approximately 1 in every 20th sample assayed at the primary lab (IPL) were shipped to a second lab (Chemex) for re-analysis. At Chemex these pulps were then renumbered with a random number sequence to produce a set of 'blind' samples that were then sent back to IPL for re-analysis.
- a set of 83 samples with Cu > 0.3% and Au > 0.3 gpt from the first hole drilled in 2003 (03-248) were sent to Acme Lab for re-analysis.
- a second sampling of half-cores for analysis at IPL was completed to provide an indication of inherent geological (short range sampling) variability.

13.4.3 Conclusions by Smee

Conclusion on the 1994 results by Smee, (1995) were as follows:

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

On completion of the 1995 drill program Smee was again retained to evaluate QA/QC results. According to Smee, (1996),

“The 1995 drill core sampling and analysis appears to be slightly better controlled than the 1994 program. The standards show that the data is accurate; no samples lie outside of acceptable limits. (However, the blank samples indicate that the 1995 analysis is free of contamination, and transcription errors should be few. However, the standards analyzed in 1995 are marginally lower in mean concentrations than they were in 1994.)

The geochemical analysis for both copper and gold show a slight rotational bias, when Min-En is compared to Chemex. This bias does not affect the validity of the data. The assay data also shows a slight rotational bias for gold. This bias may be related to the fact that Chemex reports gold analysis in ounces per ton, then uses a multiplication factor to convert to grams per tonne. This biases the data in 0.03 increments. At low concentrations, this rounding-off can bias the low concentrations by more than 10%.

The precision calculations show that the 1995 data is slightly more precise than found in 1994. The precision for both data sets is excellent at the higher concentrations; however, gold at the 0.3 gpt concentration has a precision of 30%. This must be considered when calculating ore reserves.”

13.4.4 Conclusions by A.J. Sinclair

- 1 *“The 1994 and 1995 American Bullion assay data for Au and Cu by Min-En lab are of an acceptable and consistent quality, based on a re-evaluation of quality control information summarized by Smee, (1995, 1996) and including (1) replicate analyses of three standards and (2) duplicate analyses of many pulps by an independent lab (Chemex).*
- 2 *Three in-house standards prepared for Red Chris Development Company. by CDN Resource Laboratories Ltd. in 2003, have well-established mean values for Cu and Au that make the standards useful reference materials for quality control of sampling and assaying related to the 2003 drilling program. These standards were inserted routinely with analytical batches to obtain the 2003 analytical data.*
- 3 *The principal lab for assaying samples from the 2003 drilling program is IPL Ltd. Repeat analyses of standards indicate that IPL 2003 Cu and Au analyses are of acceptable accuracy.*
- 4 *Every 20th IPL pulp was submitted to an independent lab (Chemex) in order to monitor for bias. Results indicate that for both Cu and Au the two labs agree satisfactorily. Where bias is noted, it is either negligible in magnitude or affects so few samples near the cutoff grade that the bias will have negligible impact on resource\reserve estimates.*
- 5 *Precision of IPL data is adequate, as demonstrated by independent data sets including (1) repeat analyses of standards, and (2) repeat analyses of pulps checked by Chemex.*
- 6 *Inherent geological (sampling) variability is the principal contributor to total variability within the data. For Cu the sampling variability is about 5 times the combined subsampling plus analytical variability; for Au the sampling variability is about 2.5 times the combined subsampling plus analytical variability. All these sources of error are random and will be minimized during resource/reserve estimation because many data will be used for the estimation of each block and the errors are compensating.*
- 7 *The Au/Cu ratio for various data sets is consistent, ranging from about 0.8 to 1.0.”*

13.4.5 Conclusions

Checks on standards in various grade ranges have shown acceptable accuracy at both the 1994-95 and 2003 primary analytical laboratories. Blank samples reported low values at or near the detection limit indicating the absence of contamination of material during preparation. Duplicate pulps sent to second labs have shown no significant analytical bias. The analysis of ‘blind’ duplicates by the primary lab (IPL) has shown the data are unbiased and have a moderate level of random analytical error. Re-analysis of 2nd half cores have shown sampling variability to be random and as a result should be minimized during the resource estimation. In the writers opinion the assay data base at Red Chris are both suitable and of the quality necessary to use in a Resource Estimate.

13.5 ICP Analysis

A total of 2,458 I.C.P. analyses on 20 percent of the drill core samples were taken during the 1994-95 drill campaigns. An additional 177 I.C.P. samples were taken in 2003. The results indicate that, except for copper and gold, there are no other metals of significant economic interest. Silver values are generally quite low but silver could add significant value to a copper-gold concentrate. Lead and zinc values appear to be of little economic significance, and arsenic, antimony, barium and other elements that are generally considered to be detrimental from a metallurgical standpoint are all quite low. No mercury analyses were taken in 1994-95 but the 129 samples tested in 2003 were all less than the detection limit of 3 ppm. The following Table 5 shows the analytical and statistical results for some the more commonly associated elements with porphyry copper-gold mineralization.

Table 5 - Statistics for ICP Results

Variable	Number of Samples	Mean (ppm)	Highest Value (ppm)	Lowest Value (ppm)	Standard Deviation
Silver	3819	1.14	200	0.1	3.48
Arsenic	3869	115.5	1760	1	129.7
Antimony	3869	11	1259	1	32.2
Barium	3869	133.7	2953	1	152.0
Bismuth	3869	4.4	99	1	5.5
Cadmium	3869	0.34	100	0.1	4.1
Chrome	3819	50.8	569	1	46.6
Cobalt	3819	12.8	88	1	7.0
Lead	3819	64.1	5973	1	173
Molybdenum	3818	11.4	463	1	16.3
Nickel	3695	21.9	309	1	22.5
Tungsten	3819	2.9	27	1	2.7
Zinc	3819	214.9	>10,000.0	10	515.7

The results of mineral characterization studies are very positive for the project since they indicate that a 'clean' copper-gold-silver concentrate could be produced without any penalty contaminants.

14.0 DATA VERIFICATION

The designation between the Main and East zones for Red Chris was set at the 50650 E coordinate. All information west of this boundary is referred to as Main Zone and east of the boundary as East Zone.

14.1 Assay Capping

The Main and East Zones contain a combined 261 drill holes which total 76,420 m of diamond drilling. Within this drilling, a total of 30,578 Cu assays and 29,992 Au assays have been collected. Appendix 1 lists all drill holes used in this study.

The data was initially subdivided into 4 geologic domains namely: Main Zone Inner and Outer Core, and East Zone Inner and Outer Core. Domain codes were assigned to each assay that fell within the interpreted domain solids. For each domain, copper and gold distributions were examined using lognormal cumulative frequency plots. The copper and gold distributions for each domain were evaluated with individual mineralized populations separated by partitioning the probability plots. Capping was instituted if required to reduce the effects of isolated outlier high assays. In general capping levels were set at 2 standard deviations above the mean of the highest population.

Values identified for capping were as follows:

Main Zone – Inner Core

- Cu - A capping level of 2 S.D. past the mean of the upper population (5.16%)
No assays capped
- Au - A capping level of 2 S.D. past the mean of the upper population (7.24 g/t)
One assay at 8.23 g/t was capped at 7.24 g/t

Main Zone – Outer Core

- Cu - A capping level of 2 S.D. past the mean of the upper population (1.77%)
One assay at 3.1% was capped at 1.77 % Cu
- Au - A capping level of 2 S.D. past the mean of the upper population (2.44 g/t)
No assays capped

East Zone – Inner Core

- Cu - A capping level of 2 S.D. past the mean of the upper population (4.63 %)
Two assays at 4.65 and 6.60 % were capped at 4.63%
- Au - A capping level of 2 S.D. past the mean of the upper population (5.66 g/t)
One assay at 6.86 g/t was capped

East Zone – Outer Core

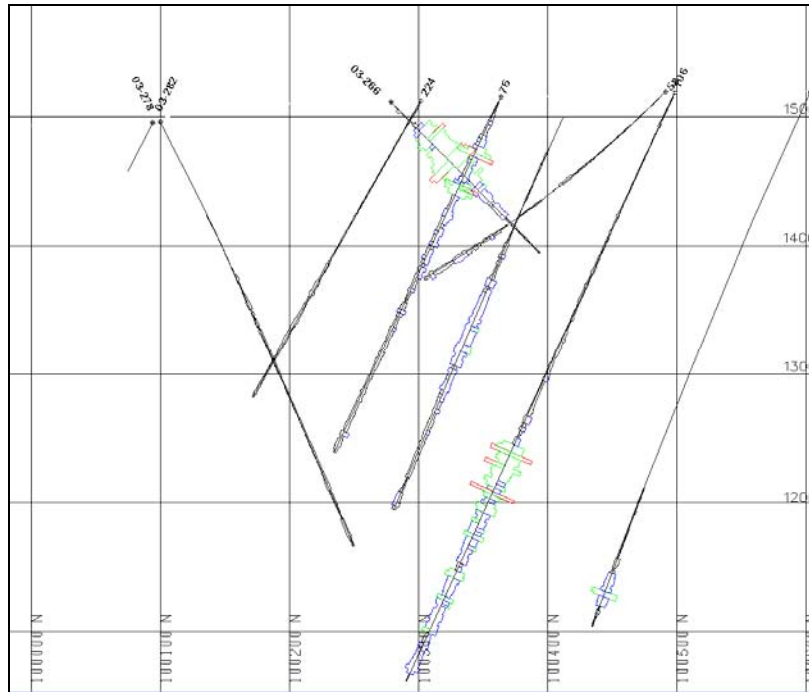
- Cu - A capping level of 2 S.D. past the mean of the upper population (4.40 %)
No assays capped
- Au - A capping level of 2 S.D. past the mean of the upper population (3.74 g/t)
No assays capped

As a further check on capping, the grade distributions for copper and gold were examined in a series of cross sections on 25 m centres through both the Main and East zones. In almost all cases, high values for copper and gold were surrounded by other high values with gradual build ups and few if any sharp spikes in grade. In addition crossing drill holes in high grade areas also showed similar grade patterns (see Figure 11).

One example of an isolated high value from the Main zone outer core is shown on section 50,450 E in Figure 12. In this case the isolated high copper value of 3.1 % near the top of the hole should be capped and in fact, it was identified by the capping scheme described above. Assays

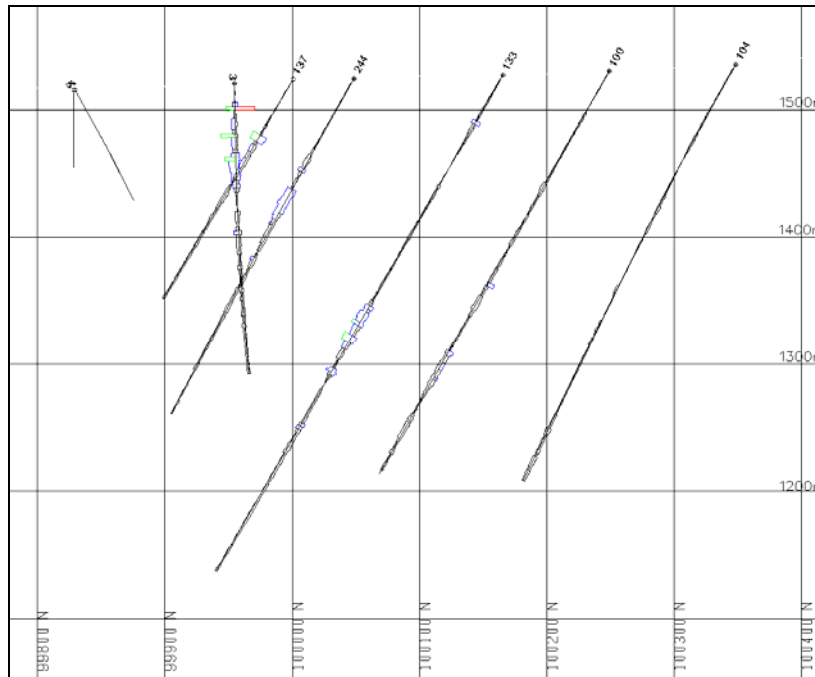
from 2004 drilling required no capping.

Figure 11 - Section 50,825 E showing drill holes with Cu and Au Histograms



Cu histogram on the right and a Au histogram on the left side of the trace of the drill stem

Figure 12 - Section 50,450 showing drill holes with Cu and Au Histograms



Cu histogram on the right and a Au histogram on the left side of the trace of the drill stem.

14.2 15 metre Composites

Down hole 15 m composites were produced to honour the domain boundaries. Composites at contacts less than 7.5 m were combined with the previous sample to produce a uniform support of 15 ± 7.5 m. Based on the additional drill hole information collected during the 2003 field season, the geologic domains, used in previous studies, were reviewed and modified.

The geologic solid models for the Main Zone and dykes were adjusted where necessary using geological information obtained in the 2004 drilling. Within the East Zone the previous interpretation of Inner Core, Outer Core and Main Phase was reinterpreted and modeled as Inner Core and East Zone Main Phase. The Main Phase solid used for the Main Zone was extended into the East Zone and the break between the Main and East zones was considered a soft boundary in 2004. That is to say, variography was completed separately for the Main and East Main Phase but drill hole composites on either side of the 50650 E divide were used to estimate blocks near the boundary. The East Satellite Zone was renamed the East Zone Extension and its boundaries were adjusted to reflect the 2004 drill information. Summary statistics for 15 m composites within each zone are presented below in Table 6.

No additional drilling was completed on the Far West or Gulley Zones, so the Resource estimates reported for these areas in the February 2004 Report, were not changed.

Table 6 - Summary of Statistics for 15m Composites by Domain

Domain	Variable	Number of Composites	Minimum	Maximum	Mean	S.D.	C. V.
Main Zone	Cu (%)	5392	0.001	3.036	0.215	0.233	1.08
	Au (g/t)	5279	0.010	3.419	0.181	0.220	1.21
Main Zone Dykes	Cu (%)	134	0.001	0.682	0.048	0.078	1.61
	Au (g/t)	134	0.001	0.352	0.052	0.050	0.97
East Zone Inner Core	Cu (%)	426	0.083	4.026	0.793	0.548	0.69
	Au (g/t)	426	0.066	3.551	0.705	0.564	0.80
East Zone Extension	Cu (%)	160	0.008	1.015	0.321	0.177	0.55
	Au (g/t)	160	0.034	0.976	0.253	0.152	0.60
Gulley Zone	Cu (%)	782	0.002	1.137	0.143	0.171	1.19
	Au (g/t)	782	0.010	1.396	0.155	0.152	0.98
Far West Zone	Cu (%)	279	0.002	0.678	0.088	0.109	1.23
	Au (g/t)	279	0.010	1.290	0.204	0.219	1.07

Where: C.V. represents the Coefficient of Variation or Standard Deviation/Mean. This measurement is a good indication of grade variability and ideally should be less than 2.

15.0 ADJACENT PROPERTIES

Not applicable.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

A comprehensive metallurgical test program was conducted in 2004 by G&T Metallurgical Services Ltd., Kamloops, British Columbia, under the direction of Peter Brown, P.Eng., Consulting Engineer, Tom Lafreniere, A.Sc.T., President, and Tom Shouldice, P.Eng., General Manager, Operations. The program was designed to develop a scheme for the treatment of Red Chris ores.

16.1 Sample Selection and Metallurgical Composites

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

Tables 7 and 8 list all samples, drill hole numbers, intervals and assay grades.

Three holes were drilled with HQ coring, #256, #256A and #283, for the purpose of obtaining a larger volume sample in each of the Main and East Zones. Segments of 256 and 256A were used in the East Zone metallurgical composites as listed in Table 8. The remainder of hole 256A and 283 were used for grinding and work index studies.

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

Table 7 - Sample Identification and Head Assays - Main Zone

Hole No.	Core Size	Sample No.	Depth (m)		Length (m)	% Cu	Au g/t
			From	To			
<u>Main Zone</u>							
MZ-1 Composite (above 1410m)							
03-258	NQ	A	118.60	139.90	21.30	0.518	0.163
03-260	NQ	B	130.80	155.20	24.40	0.543	0.193
03-263	NQ	C	60.70	85.10	24.40	0.523	0.127
03-271	NQ	D	58.70	79.70	21.00	0.557	0.286
03-276	NQ	E	99.10	114.10	15.00	0.526	0.214
03-279	NQ	F	11.00	22.90	11.90	0.502	0.219
03-280	NQ	G	28.70	43.60	14.90	0.549	0.226
MZ-2 Composite (1320-1410m)							
03-258	NQ	A	210.10	222.30	12.20	0.517	0.265
03-260	NQ	B	212.80	227.70	14.90	0.560	0.272
03-263	NQ	C	246.60	263.30	16.70	0.500	0.472
03-268	NQ	D	179.90	195.90	16.00	0.535	0.458
03-271	NQ	E	228.30	240.30	12.00	0.510	0.368
03-274	NQ	F	156.70	171.70	15.00	0.554	0.464
03-276	NQ	G	217.80	236.80	19.00	0.539	0.443
03-277	NQ	H	196.50	208.50	12.00	0.518	0.295
03-279	NQ	I	205.40	226.40	21.00	0.553	0.541
MZ-3 Composite (1230-1320m)							
03-258	NQ	A	255.80	271.00	15.20	0.707	0.343
03-265	NQ	B	295.40	320.70	25.30	0.625	0.762
03-268	NQ	C	262.00	292.00	30.00	0.604	0.457
03-274	NQ	D	276.90	294.60	17.70	0.673	0.598
03-276	NQ	E	292.00	317.40	25.40	0.578	0.541
03-277	NQ	F	263.00	284.00	21.00	0.594	0.541
MZ-4 Composite (Low grade 1365m level)							
03-258	NQ	A	4.60	17.70	13.10	0.252	0.109
03-260	NQ	B	19.80	35.40	15.60	0.257	0.116
03-260	NQ	C	50.60	59.80	9.20	0.243	0.113
03-263	NQ	D	200.20	216.20	16.00	0.274	0.234
03-265	NQ	E	72.90	91.20	18.30	0.258	0.130
03-271	NQ	F	111.80	138.10	26.30	0.226	0.098
03-274	NQ	G	177.70	194.80	17.10	0.256	0.297
03-276	NQ	H	10.60	25.60	15.00	0.264	0.345
MZ-5 Composite (Medium grade 1365m level)							
03-263	NQ	A	173.50	193.60	20.10	0.355	0.288
03-265	NQ	B	91.20	106.40	15.20	0.342	0.112
03-268	NQ	C	151.80	179.90	28.10	0.357	0.216
03-271	NQ	D	138.10	150.10	12.00	0.338	0.338
03-274	NQ	E	118.40	156.70	38.30	0.318	0.214
03-277	NQ	F	169.10	196.50	27.40	0.339	0.194

Table 8 - Sample Identification and Head Assays - East Zone

Hole No.	Core Size	Sample No.	Depth (m)		Length (m)	% Cu	Au g/t
			From	To			
East Zone							
EZ-1 Composite (1425m - surface)							
03-253	NQ	A	40.50	53.60	13.10	0.972	0.610
03-253	NQ	B	62.60	81.20	18.60	0.993	0.745
03-255	NQ	C	80.10	98.10	18.00	0.600	0.418
03-256	HQ	D	24.80	57.10	32.30	0.943	0.595
03-262	NQ	E	61.90	86.80	24.90	0.834	0.607
EZ-4 Composite (1335-1425m)							
03-251	NQ	A	180.50	188.80	8.30	0.637	0.456
03-252	NQ	B	117.00	120.10	3.10	0.745	0.532
03-253	NQ	C	158.90	170.00	11.10	0.689	0.493
03-254	NQ	D	117.1	130.0	12.90	0.723	0.472
03-254	NQ	E	135.0	144.5	9.50	0.890	0.703
03-254	NQ	F	150.0	153.0	3.00	0.600	0.490
03-255	NQ	G	98.10	119.10	21.00	0.510	0.361
03-256A	HQ	H	90.00	109.50	19.50	0.752	0.586
03-262	NQ	I	86.80	98.40	11.60	0.851	0.712
EZ-3 Composite (1245-1335m)							
03-249	NQ	A	206.40	224.80	18.40	0.496	0.498
03-253	NQ	B	247.10	256.1	9.00	0.663	0.583
03-253	NQ	C	262.10	283.20	21.10	0.753	0.690
03-254	NQ	D	170.00	208.6	38.60	0.613	0.542
03-255	NQ	E	205.50	226.20	20.70	0.495	0.392
03-256A	HQ	F	246.60	258.50	11.90	0.586	0.564
EZ-2 Composite (1140-1245m)							
03-253	NQ	A	283.20	319.30	36.10	1.035	0.952
03-253	NQ	B	361.30	376.30	15.00	0.782	0.726
03-254	NQ	C	329.00	340.30	11.30	0.618	0.494
03-255	NQ	D	319.40	346.40	27.00	0.483	0.339
03-256A	HQ	E	294.40	314.40	20.00	0.927	0.924
EZ-5 Composite (High grade)							
03-250	NQ	A	346.1	371.3	25.20	1.250	1.354
03-251	NQ	B	202.40	212.40	10.00	1.418	1.213
03-252	NQ	C	343.00	346.00	3.00	0.987	0.816
03-256A	HQ	D	109.50	161.00	51.50	1.309	1.066
EZ-6 Composite (Low grade)							
03-249	NQ	A	182.90	206.40	23.50	0.328	0.338
03-249	NQ	B	224.80	243.50	18.70	0.261	0.315
03-251	NQ	C	255.70	269.30	13.60	0.391	0.368
03-251	NQ	D	279.50	288.80	9.30	0.346	0.359
03-255	NQ	E	60.80	80.10	19.30	0.299	0.177
03-256	HQ	F	3.00	15.00	12.00	0.242	0.155
03-259	NQ	G	250.40	267.60	17.20	0.269	0.375

Portions of the drill core were blended to make up a series of composite samples for metallurgical testing. Composites MZ-1,2,3 and EZ-1,2,3,4 represent horizontal layers within each zone that would be mined sequentially. Composite EZ-5 was a high grade sample prepared to study the effect of metallurgy at higher grades. Composites MZ-4, MZ-5 and EZ-6 are lower grade, representing material that will be stockpiled during the seventeen year mining phase and processed at a later date in years 18 -25. Table 9 lists actual composite head grades used in the metallurgical testing. Over the mine life the average copper and grades of Main and East zones are expected to average about 0.42% and gold 0.30-0.39g/t respectively. In the first nine years of production the Main zone copper grade will average about 0.46% and the average grade of the Main zone metallurgical composite is about 0.47%. The average grade of the east zone composite and global sample is relatively higher than the current production plan. Additional sampling and metallurgical work has been recommended to improve this link to the new mine plan.

Table 9 - Individual Metallurgical Composite Head Grades

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

Two additional composites for metallurgical testing were prepared. The MZ-Global comprised a blend of equal portions by weight of MZ-1, MZ-2, MZ-3 and MZ-5. The EZ-Global comprised a blend of equal portions by weight of EZ-1, EZ-2, EZ-3 and EZ-4. Table 10 lists actual composite head grades used in the metallurgical testing.

Table 10 - Global Metallurgical Composite Head Grades

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

16.2 Mineralogy

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

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

Gold occurrence is higher in the East Zone, and relative to the Main Zone, it is more dominantly associated with copper sulphides than pyrite. While the basic mineralogy of occurrence is similar throughout both zones this results in better gold recoveries in East Zone. Table 11 lists the mineralogy of the metallurgical composites.

Table 11 - Metallurgical Composite Mineralogy

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

16.3 2004 Metallurgical Test Program

The program of flowsheet development studies and metallurgical response was based on the following series of objectives:

- Study the mineral composition and fragmentation characteristics of several ore composites from the Main and East Zones, representing material to be processed during the early years of the operation.

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

16.3.1 Mineral Liberation Characteristics

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

The Red Chris ore is finer grained when compared with many porphyry copper-gold deposits, and as such will require a slightly finer grind size. This is also evident in the 24 micron regrind size selected for the flotation cleaner circuit feed.

A primary grind of 105 micron K_{80} increased the recovery, however, this is more than off-set by an additional 3.5MW of power required for grinding or a reduction in throughput. A simple economic study confirmed this, using typical power costs and a net smelter revenue value for copper of 90 cents. Testing at a coarser 200 micron K_{80} resulted in a 5% recovery loss. The 150 micron K_{80} was considered the optimum level for this application. However, the economic primary grind selection should be reviewed if copper prices continue to improve.

16.3.2 Rougher and Cleaner Flotation

A total of 129 bench tests followed by 31 locked cycle flotation tests were conducted studying pulp density, pH, reagent dosage, flotation residence time and grind size effect. Table 12 lists the operating parameters required to achieve optimum performance. Although the mineralogy between the two zones is different, similar operating parameters will be used with no loss in performance.

Table 12 - Flotation Parameters

pH	Lime	Roughers	10.5
		Cleaners	12.0
Collector	Potassium Amyl Xanthate	Roughers	0.006 kg/t
		Cleaners	0.005 kg/t
Frother	MIBC		As req'd
Flotation Time	Lab Time	Roughers	9 min.
		Cleaners	9-11 min.
Flotation Density		Roughers	33-35% solids

Typical operating practice in many porphyry copper operations is to produce a 10 – 15% Cu rougher concentrate by pulling 4 – 5% mass. Current testwork indicates the Red Chris ore requires a 15 – 20 % mass pull resulting in a rougher concentrate grade of 3% Cu in order to achieve the same recovery. The regrind circuit power has been specified based on a design weight recovery of 15%.

A series of regrinding tests were conducted to investigate the possibility of preferential or distributed regrinding power, and the benefit of pulp conditioning. The tests involved by-passing part of the higher grade rougher concentrate, pre-cleaning prior to regrinding and classification prior to regrinding.

None of the modified circuits produced any improvement in results over those of the conventional regrinding circuit. Testing indicated regrinding to a relatively fine liberation sizing of $K_{80} = 24$ microns was potentially required to obtain optimum final grade and recovery, but this is mainly based on a consideration of Main Zone metallurgy. More testwork was recommended by G&T to optimize regrind size selection and power.

The duplicate tests listed in the following table indicate a slightly higher concentrate grade was achieved using the modified circuit. However, when the higher rougher concentrate feed grade effect is taken into account, the results are very similar.

Table 13 - Regrind Circuit versus Metallurgy

Test	Sample	Rough Con		Cleaner Circuit	
		% Cu	% Cu	% Cu Rec	% Au Rec
1522-21	Modified RG Circuit	15.4	27.4	92.6	65.1
1522-22	Modified RD Circuit	17.0	25.0	91.8	57.5
1522-29	Conventional RG Circuit	14.8	24.1	91.4	60.6
1522-30	Conventional RG Circuit	15.3	25.4	92.5	62.7

16.3.3 Gold Occurrence

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

Table 14 - Gold Recovery Partition

Composite	Gold Recovery Partition Coefficients			Statistic R ²
	Cu Sulphides	Pyrite	Total	
MZ-1	0.45	0.52	0.97	0.98
MZ-2	0.54	0.43	0.97	0.98
MZ-3	0.57	0.41	0.98	0.98
MZ-4	0.36	0.62	0.98	0.98
MZ-5	0.47	0.53	1.00	0.99
EZ-1	0.69	0.29	0.98	0.99
EZ-4	0.66	0.32	0.98	0.99
EZ-3	0.73	0.25	0.98	0.99
EZ-2	0.75	0.23	0.98	0.99
EZ-5	0.77	0.22	0.99	0.99
EZ-6	0.56	0.41	0.97	0.99

16.3.4 Gravity Concentration Tests

Tests were conducted on both Global composites, using a Knelson Concentrator to investigate the response of gold recovery to gravity separation. Table 15 lists the results of these tests, indicating very low recovery and up-grading effect.

Table 15 - Gold Gravity Recovery

<u>Composite</u>	<u>Feed Au g/t</u>	<u>Pan Concentrate</u>	
		<u>Au g/t</u>	<u>% Au Recovery</u>
Main Zone Global	0.29	1.04	4.5
East Zone Global	0.73	5.8	9.2

16.3.5 Work Index and Ore Hardness

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

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

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

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

16.3.6 Metallurgical Recoveries

The results of G&T's locked cycle testwork are presented in Table 16. The flotation locked cycle testing produced a copper grade-recovery profile relationship for both zones which is summarized in Table 17.

Table 16 - Locked Cycle Test Results

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

Table 17 - Copper Grade-Recovery Profile

Final Concentrate Grade % Cu	Percent Copper Recovery	
	Main Zone	East Zone
25.0	90.8	90.0
26.0	90.3	89.9
27.0	89.8	89.7
28.0	89.1	89.4
29.0	88.2	89.2

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

Based on G&T's results, and a production split of 73% and 27% between Main and East zones respectively, the overall life of mine metallurgy is estimated to be about:

Mass %	Assay - % or g/t		Recovery-%	
	Copper	Gold	Copper	Gold
1.8	26.4	11.2	88	49

This is in reasonable agreement with the overall mine plan recoveries of 87% copper and 47% gold at a 27% copper concentrate grade. Over the first five years of operation gold recovery is expected to vary in the range of 50 – 60% depending on mineralogy and grade, with an overall average of 56%.

Additional sampling and testwork is recommended to improve the correlation of metallurgy the production schedule forming the basis of this technical report. In addition the impact on recovery of storing low-grade ore for later processing should also be investigated. However this occurs only towards the end of the mine life.

16.3.7 Copper Concentrate

A pilot plant program was conducted at G&T Metallurgical Engineers with the objective of producing a final concentrate for use and testing by prospective buyers. Table 18 lists the full analysis, including minor elements, of a sample prepared in the ratio of 64% Main Zone and 36% East Zone. Hg and Sb are expected to incur penalties for being outside the limits set by some of the smelters.

Table 18 - Copper Concentrate Analysis

Element	Symbol	Unit	Analysis	Sample* Range **	Average
Copper	Cu	%	27.0		26 – 30
Gold	Au	g/t	17.3		5 – 25
Silver	Ag	g/t	42		25 – 100
Aluminum	Al	%	0.95		0.6 – 1.0
Antimony	Sb	ppm	756		400 – 900
Arsenic	As	ppm	130		80 – 250
Barium	Ba	ppm	44		12 – 45
Bismuth	Bi	ppm	<7		<8
Cadmium	Cd	ppm	16		12 – 22
CaO	CaO	%	0.66		0.4 – 0.9
Chlorine	Cl	%	0.01		0.01
Chromium	Cr	ppm	3		3 – 55
Cobalt	Co	ppm	20		10 – 25
Fluorine	F	ppm	87		40 – 190
Iron	Fe	%	28.8		23 – 33
Lead	Pb	ppm	439		200 – 1300
Manganese	Mn	ppm	213		40 – 500
Mercury	Hg	ppm	27		8 – 50
MgO	MgO	%	0.36		0.2 – 0.5
Molybdenum	Mo	ppm	166		20 – 170
Nickel	Ni	ppm	22		10 – 45
Phosphorous	P	ppm	115		40 – 225
Potassium	K	ppm	372		130 – 500
Selenium	Se	ppm	93		70 – 160
Silica	SiO ₂	%	6.6		3.5 – 7.0
Sodium	Na	ppm	274		65 – 300
Sulphur	S	%	29.8		28 – 35
Titanium	Ti	ppm	348		300 – 450
Zinc	Zn	ppm	1400		600 – 2000
Sizing	K ₈₀	µm	25		24 – 28

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

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

16.3.8 Historical Testwork

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

These earlier programs formed the basis for the 2004 program conducted on newly acquired drill core from the 2003 exploration program.

16.4 Ore Variability

A batch rougher and cleaner geometallurgical mapping program was performed to test 33 samples throughout the orebody representing the Main Zone, East Zone Outer Shell and East Zone Core. The samples were selected from the main set listed in Tables 7 and 8. The testing consisted of a mineralogical evaluation, hardness determination and metallurgical performance as outlined in Table 19. The average of these results is marginally below the projected metallurgy from earlier locked cycle testing. The reason for this is that the sub-samples used in the mapping program do not represent all the ore used in the master and global composites. As well the variability program was based on batch testing, which for recovery estimation relies on a smaller sample statistically and extrapolation of intermediate products, whereas locked cycle testing is based on processing larger samples and direct measurement of products. The locked cycle results are therefore regarded as a better estimate of recovery. The geometallurgy data indicates some moderate variability in the metallurgical performance. Additional work is recommended to align this data with the mine plan to optimize it.

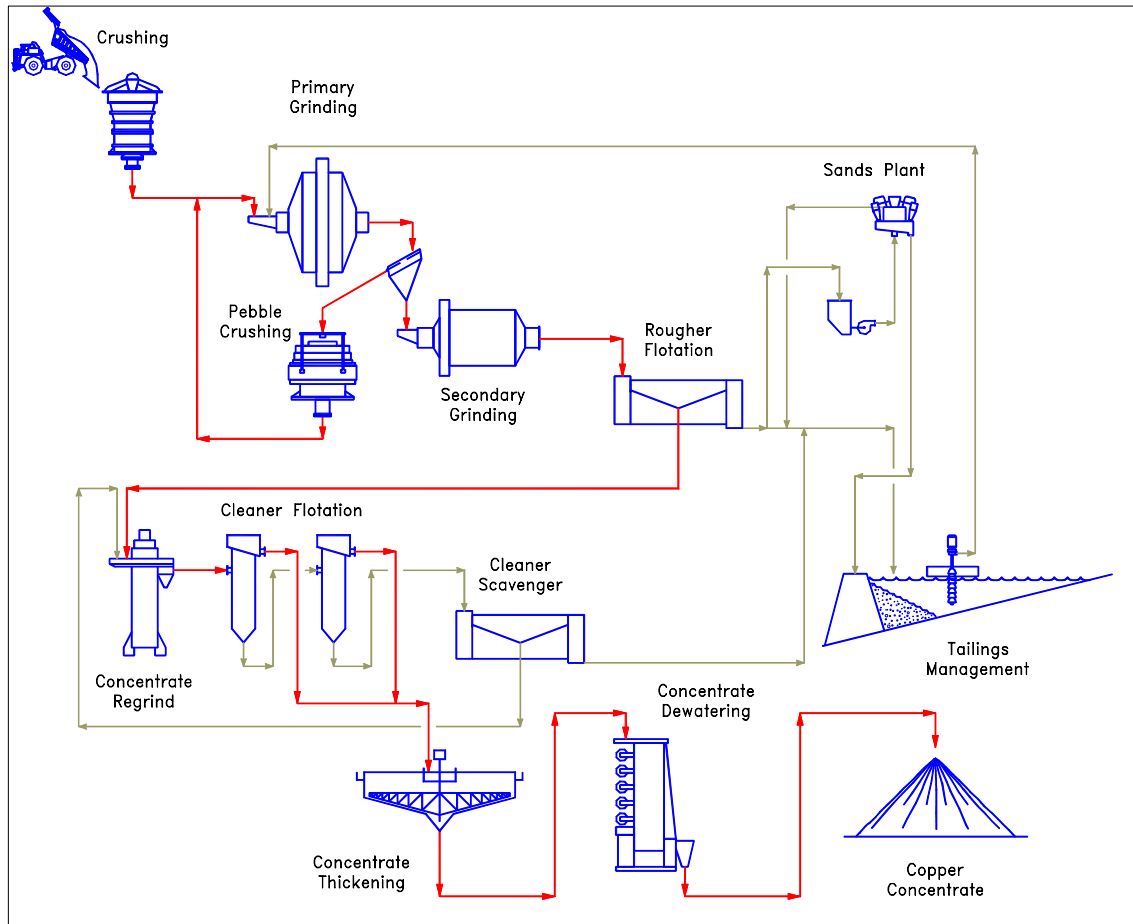
Table 19 - Geometallurgical Mapping Data

Test No.	Drill Hole No.	Sample No.	% Mineral Content			Work Index kWhrs/tonne	% Recovery *	
			Pyrite	Chalcopyrite	Bornite		Cu	Au
Main Zone								
1	258	A	8.9	1.5	0.0	13.8	80.4	27.7
13	260	B	11.0	1.5	0.0	14.0	83.6	53.5
2	271	D	11.2	1.5	0.0	13.0	85.1	43.7
14	276	E	8.5	1.5	0.0	11.6	84.7	48.6
15	279/280	F/G	15.3	1.5	0.0	11.5	84.0	30.5
16	258/260	A/B	11.9	1.5	0.0	14.5	82.7	42.2
17	263	C	12.5	1.2	0.0	14.6	87.4	54.7
3	268	D	13.7	1.4	0.0	15.4	88.2	51.2
18	271/274	E/F	14.4	1.5	0.0	15.3	90.7	46.0
4	276	G	13.1	1.6	0.0	12.8	81.9	32.9
19	279	I	15.0	1.5	0.0	16.6	90.3	56.6
5	258	A	10.8	2.0	0.0	13.7	90.6	52.7
20	265	B	15.9	1.7	0.0	14.8	86.1	45.9
6	268	C	12.7	1.7	0.0	16.2	88.3	59.3
21	274	D	13.4	1.7	0.0	16.3	87.2	42.2
7	276	E	14.5	1.8	0.0	14.5	87.6	53.3
22	277	F	11.5	1.2	0.0	16.3	88.2	51.2
East Zone Outer Shell								
8	255	C	9.5	1.6	0.0	15.8	86.0	50.2
10	249	A	12.7	1.5	0.0	15.7	84.5	44.8
12	256A	E	12.1	2.5	0.0	15.2	85.7	55.1
27	256A	H	15.7	1.9	0.0	14.0	85.6	37.3
28	253	B	11.3	2.7	0.0	13.0	91.7	74.9
31	254	D	10.3	1.4	0.0	10.8	79.5	61.7
32	251	C	8.3	1.0	0.0	16.5	82.5	43.8
33	259	G	8.9	1.5	0.0	15.2	80.2	37.3
East Zone Core								
9	262	E	2.6	1.8	0.3	18.3	85.4	66.2
23	255	D	0.1	0.4	0.5	16.4	80.6	56.9
29	253	C	0.1	0.6	0.9	15.5	90.8	66.5
11	254	D	1.1	1.0	0.3	15.2	81.2	61.2
30	255	E	0.0	1.1	0.4	17.2	83.9	66.0
24	256A	F	0.2	0.9	0.3	20.7	78.1	61.8
25	251-254	A/B/C/F	0.7	1.4	0.3	22.1	90.4	69.7
26	255	G	0.0	1.0	0.1	18.4	87.0	73.4
* Normalized to 27% Cu Grade Concentrate								

16.5 Processing

The Red Chris ore will be processed through an on-site concentrator that will produce a copper-gold flotation concentrate that will be shipped out of the Province for smelting and refining. The nominal milling rate will be 30,000 tonnes per day. A simplified flowsheet is presented in Figure 13.

Figure 13 - Red Chris Simplified Flowsheet



16.5.1 Crushing

Run-of-mine (ROM) ore will be dumped from 230 tonne mine trucks into the dump pocket of a primary 1270 mm x 1651 mm (50" x 65") gyratory crusher. The dump pocket is designed to accept two trucks at a 150 degree angle to each other. Crushed ore will discharge into a surge pocket below the primary crusher. Ore will be withdrawn from the surge pocket by a 2,134 mm wide x 25.9 m long belt feeder. The feeder will discharge onto a 1,372 mm wide x 237 m long overland conveyor. This conveyor will transport the coarse ore from the gyratory crusher to the coarse ore stockpile.

The uncovered coarse ore stockpile will have a nominal live capacity of 30,000 tonnes, or the equivalent of one day's operation at the design throughput rate. Ore will be reclaimed from the coarse ore stockpile using three 1,240 mm wide x 7.2 m long variable speed belt feeders located in a reclaim tunnel beneath the stockpile and fed to the SAG mill feed conveyor. This 1,067 mm wide x 141 m long conveyor will deliver ore to the SAG mill. Dust collection facilities will be included both at the gyratory crusher and at the stockpile reclaim tunnel to intercept and collect dust from both the crusher discharge conveyor and from the reclaim tunnel feeders under the crushed ore stockpile.

16.5.2 Grinding

Ore will be ground in two stages to produce a product suitable for flotation. The first stage will be a 10.36 m dia. x 5.49 m EGL (34 ft dia. x 18 ft EGL) semi-autogenous grinding mill driven by two (2) 5,600 kW (7,500 hp) wound rotor induction motors. Oversize pebbles extracted from the SAG Mill will be removed by screen, crushed by a 600 kW (800 hp) pebble crusher, and then returned to the SAG Mill feed. Undersize from the SAG mill discharge screens will be transferred to two parallel ball mills for secondary grinding.

The second grinding stage will consist of two parallel 5.79m dia. x 9.75m EGL (19 ft dia. x 32 ft EGL) ball mills, each driven by a 5,600 kW (7,500 hp) wound rotor induction motor. Each mill will operate in closed circuit with a cyclopac consisting of six (6) 650 mm dia. cyclones (5 operating, 1 standby). Each cyclopac will be fed by two 20" x 18" variable speed pumps (1 operating, 1 standby). The cyclone underflow will report to back to the ball mill for grinding, while the cyclone overflow, at 38 wt% solids and a P80 of 150 microns will gravitate to the rougher flotation circuit.

16.5.3 Flotation

The ground slurry from the secondary grinding circuit (cyclone overflow) will be passed through two parallel rougher/scavenger flotation circuits each consisting of 5 x 100 m³ capacity flotation cells, equivalent to approximately 18 minutes retention time. The copper-gold concentrates from all of the rougher scavenger flotation cells will be combined and directed to the regrind circuit. The tailing from the rougher scavenger cells will be directed through two additional 100 m³ parallel flotation cells designed to de-sulphurize the final mill tailings on as needed basis. The objective is to produce a de-pyritized final mill tailing to create Non-Acid Generating (NAG) tailings for use as dam construction material and for beaches along the upstream faces of each of the three tailings dams at final closure.

The copper gold concentrate from the rougher scavenger circuit and cleaner/scavenger circuit will be reground in two parallel 1120 kW (1500 hp) tower mills operating in closed circuit with a cyclopac consisting of twelve (12) 250 mm dia. cyclones (10 operating, 2 standby). Cyclone underflow will return to the regrind mills, while cyclone overflow will be pumped to two stages of cleaning using column flotation cells.

The concentrates from both column flotation cells (Stage 1 and Stage 2 Cleaner Cells) will be combined and pumped to the concentrate thickener as the final copper gold concentrate produced by the mill. The tailing from the Stage 1 column cleaner cell will be directed to the Stage 2 column cleaner cell. The tailings from the Stage 2 column cleaner cell will be directed to the cleaner scavenger flotation circuit consisting of 6 x 50 m³ flotation cells. The concentrate from the cleaner scavenger circuit will be sent back to the concentrate regrind circuit for additional liberation. The tailings from the cleaner scavenger circuit will be directed to the final mill tailings pump box where it will be combined with the rougher scavenger tailings and sent to the

tailings impoundment.

16.5.4 Concentrate Thickening and Filtration

The final copper-gold flotation concentrate from the column cleaner cells (Stage 1 and 2) will be directed to the 20 m. dia. concentrate thickener where excess water will be removed and recycled. The thickened concentrate will be drawn from the thickener underflow and sent to one of two parallel pressure filters where the water will be pressed out of the concentrate to generate a dewatered concentrate filter cake. The dewatered final copper-gold concentrate will be stockpiled at one end of the mill building from where it will be loaded by front end loader into B-Train bulk concentrate haul trucks. A scale will be provided in the load out area to ensure proper legal loading of the concentrate trucks.

16.5.5 Tailing

The process plant tailings system consists of two separate tailings pipelines: regular tailings pipeline and sands pipeline. Under normal operating practice all tailings will be sent to the tailings pond using gravity flow in the regular tailings pipeline. This line will consist of two pipelines running from the plant site to a splitter box. From the splitter box, one line will transport tailings to the North Dam and another line will transport tailings to the South Dam. Only one line will operate at a time (1 operating, 1 standby). In order to dissipate excess hydraulic head, choke stations will be installed.

During periods of tailing dam construction, the final de-pyritized rougher tailings will be sent to a cyclone sands plant, within the mill complex, where coarse tailings solids will be separated from the fines using two parallel cyclone banks to generate a de-pyritized coarse NAG tailings sand to be used in dam construction.

The sands pipeline is used to transport non-acid generating (NAG) cycloned sand from the process plant to the tailings impoundment area. This sand will be used to raise and extend the downstream shells of the North Dam and South Dam. The sands pipeline will consist of a single pipeline running in parallel with the regular tailings pipelines from the plant site to the splitter box. From the splitter box, one line will transport sand to the North Dam and another line to the South Dam. During the winter months, this system will be drained and shutdown. In order to dissipate excess hydraulic head, choke stations will be installed.

17.0 MINERAL RESOURCE ESTIMATES

17.1 Semivariogram Analysis

Based on the new geologic interpretation, the semi-variograms for Red-Chris were checked for copper and gold within the Main Phase and East Zone Inner Core Domains and not changed. The new interpretation combining East Zone outer core and Main phase required re-modeling for the East Zone Main Phase, the remaining domains had insufficient data to model.

Pairwise relative semivariograms were used to model the variables. The procedure was to first look in the directions of maximum information; namely down-hole, by modeling azimuths 0° and 180° and dip directions -60° and -45° . These four directions represent the majority of drill holes on the property. From the semivariograms produced in these four directions, the sill and nugget components of the semivariograms were established. These parameters were then applied to the models produced in the horizontal plane to determine the direction of maximum continuity. The vertical plane perpendicular to this direction was then tested to determine the directions of maximum and minimum continuity.

17.1.1 Main Zone

The Main Zone is defined as that part of the deposit west of coordinate 50,650 E and east of 49,600 E. This distinction was made in the 1996 study to account for a flexure in the mineralized zone east of this point that trends north-east. Composites within the Main Zone were modeled for copper and gold. In all cases simple spherical models were fit to the data.

Nested spherical models were fit to copper with the longest horizontal range of 200 m along azimuth 045° . The vertical plane perpendicular to this showed a maximum range of 250 m at azimuth 135° dip -75° and the minimum range of 90 m at azimuth 315° dip -15° .

Nested spherical models were also fit to gold with the longest horizontal range of 310 m along azimuth 070° . The vertical plane perpendicular to this showed a maximum range of 300 m at azimuth 160° dip -60° and the minimum range of 170 m at azimuth 340° dip -30° .

The dykes that were identified within the Main Zone did not have enough data to model with semivariograms, so the ranges from the Main Zone were used to control an inverse distance estimate.

17.1.2 East Zone

The East Zone consisted of all data east of 50650 E. Again all variables were modeled with simple spherical models. For variography, data were subdivided into 2 domains.

17.1.2.1 East Zone Main Phase

For modeling purposes the Main Phase east of 50650 E was modelled independently from the Main Phase west of 50650 E. For copper the four directions of most drill holes were modeled

first; namely azimuth 0° and 180° with dips of -60° and -45° . A sill and nugget level were established from these directions and applied to the horizontal plane. The maximum continuity in the horizontal plane was found to be 350 m along azimuth 45° . The vertical plane perpendicular to azimuth 45° was tested and the maximum continuity within this plane was 250 m at azimuth 135° and dip -75° while in the third principal direction the maximum range was 100 m at azimuth 315° dip -15° .

A similar procedure was used to model gold in this domain. A maximum range of 210 m was found in the horizontal plane at azimuth 70° and dip 0° . In the plane perpendicular to this a maximum range of 220 m was found at azimuth 160° dip -60° while a minimum of 60 m was found at azimuth 340° dip -30° .

These models (see Table 20 for parameters) were used to estimate the Main Phase East zone and the East Extension domains.

17.1.2.2 East Zone Inner Core

There were 426 composites coded East Zone inner core. As mentioned above, the strategy employed was to fit the best model possible to the 4 directions with the most data and then apply this model to the horizontal plane if possible.

For copper, the longest range of 125 m in the horizontal plane was found along azimuth 70° dip 0° . The two perpendiculars had ranges of 50 m at azimuth 340° dip 0° and 100 m at dip -90° .

Gold showed a very similar pattern to copper with maximum continuity of 90 m along azimuth 70° dip 0 . The two perpendiculars had ranges of 40 m at azimuth 340 dip 0 and 90 m at dip -90° .

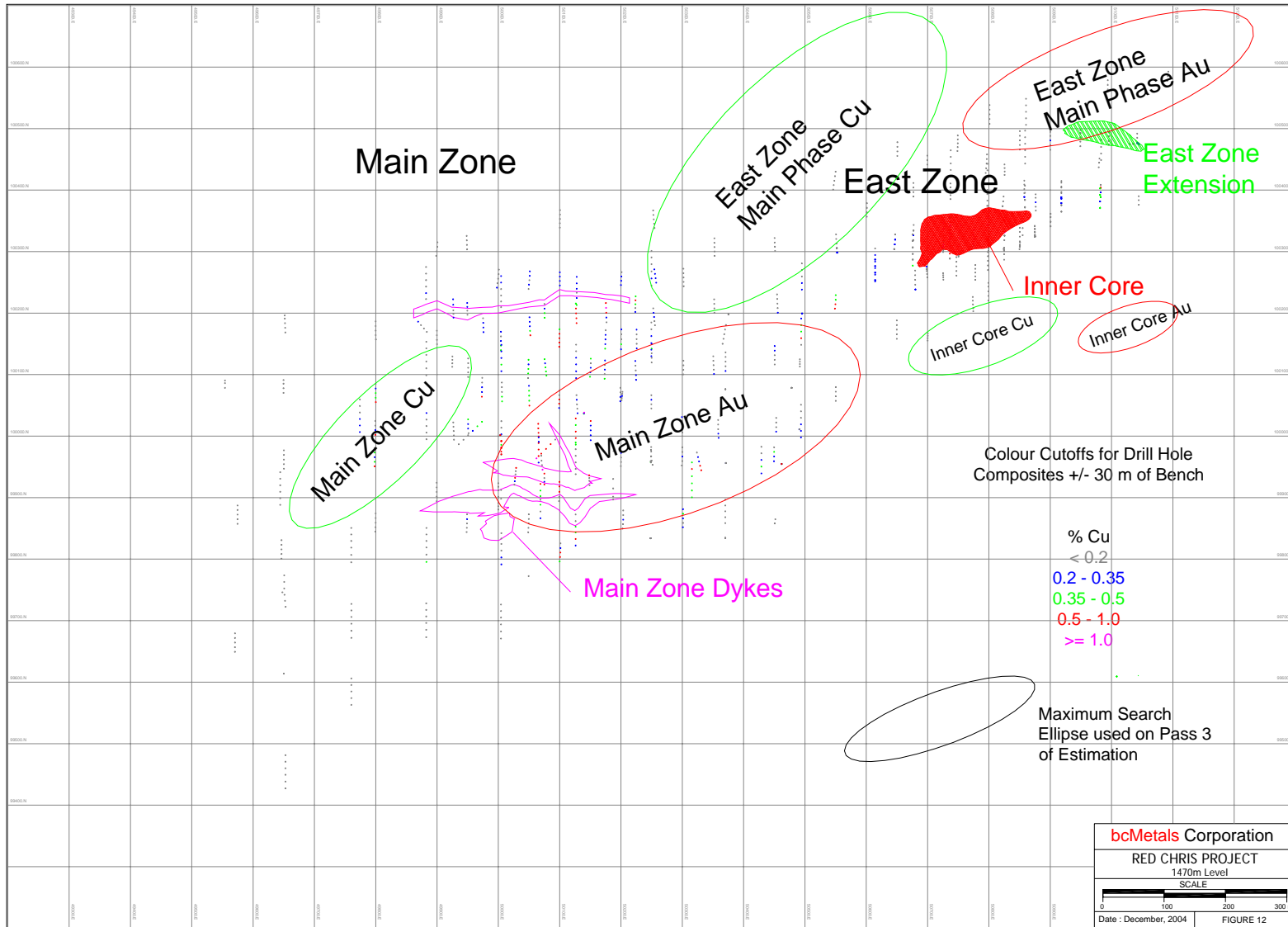
The parameters for each model within each zone are summarized below in Table 20. Figure 14 shows on the 1490 Level the various search ellipses set at the maximum semivariogram ranges, the geologic domains and the 15 m composites colour coded by copper grade and projected 30 m from each side of the section.

Table 20 - Summary of Semivariogram Parameters

Domain	Variable	Direction	C0	C1	C2	Range a1 (m)	Range a2 (m)
Main Zone	Cu	Az. 045° Dip 0	0.05	0.15	0.36	30	200
		Az. 135° Dip -75°	0.05	0.15	0.36	50	250
		Az. 315° Dip -15°	0.05	0.15	0.36	20	90
Main Zone	Au	Az. 070° Dip 0	0.02	0.23	0.30	70	310
		Az. 160° Dip -60°	0.02	0.23	0.30	70	300
		Az. 340° Dip -30°	0.02	0.23	0.30	30	170
East Zone Main Phase	Cu	Az. 045° Dip 0	0.10	0.30	0.30	200	350
		Az. 135° Dip -75°	0.10	0.30	0.30	150	250
		Az. 315° Dip -15°	0.10	0.30	0.30	60	100
East Zone Main Phase	Au	Az. 070° Dip 0	0.04	0.20	0.32	30	210
		Az. 160° Dip -60°	0.04	0.20	0.32	100	220
		Az. 340° Dip -30°	0.04	0.20	0.32	20	60
East Zone Inner Core	Cu	Az. 070° Dip 0	0.06	0.04	0.21	20	125
		Az. 340° Dip 0	0.06	0.04	0.21	20	50
		Az. 0° Dip -90°	0.06	0.04	0.21	30	100
East Zone Inner Core	Au	Az. 070° Dip 0	0.10	0.08	0.17	20	90
		Az. 340° Dip 0	0.10	0.08	0.17	20	40
		Az. 0° Dip -90°	0.10	0.08	0.17	30	90

Note: A nested semivariogram has a nugget effect (C0), combined with, in this case, two nested structures with structural component C1 over the range a1 for the short range structure and structure C2 over the range a2 for the long range structure. The sill of the semivariogram is made up of the combination of C0 + C1 + C2.

Figure 14 - Typical Level Plan showing 1998 Geologic Interpretation



17.2 Block Model Estimation

17.2.1 Introduction

The geologic model that controls the estimation process was interpreted by RCDC consulting geologists. A geologic block model was then produced with a three dimensional grid of blocks, each 20 x 20 x 15 m in dimension and coded by geologic domain. This model used partial blocks in that the proportion of a block within each domain solid was reported and stored. Blocks were also compared to the topographic surface, with the proportion of the block below this surface recorded. The parameters of the block model are shown below in Table 21. The limits of the 2004 block model were expanded to include the Far West and Gully Zones which were not estimated in 1996.

Table 21 - Parameters of block model for Red Chris

	Easting	Northing	Elevation
Lowest South West Corner of Model	48,200 E	98,800 N	900
Highest North East Corner of Model	51,500 E	100,700 N	1725
Number of blocks	165	95	55
Dimension of Block	20 m	20 m	15 m

Blocks within the Main Zone were estimated by ordinary kriging if the proportion of the block within the Main Zone solid was greater than zero. Blocks with a proportion of dyke material greater than zero were estimated using inverse distance squared and only dyke composites. For blocks with both dyke and Main Zone estimated, a weighted average was calculated for copper and gold. For example, the grade of Cu within any block was equal to the following:

$$\text{Cu} = \frac{(\text{Proportion Main Zone} * \text{Main Zone Cu}) + (\text{Proportion Dyke} * \text{Dyke Cu})}{(\text{Proportion Main Zone} + \text{Proportion Dyke})}$$

For the East zone Main Phase, the Inner Core, and East zone Extension, mineralization was all estimated by ordinary kriging, if the proportion of any of these domains within a block was greater than zero. Again, a weighted average grade was calculated for the block if two or more domains were detected. Unlike the last estimate, however, the boundary between the Main zone Main Phase and East zone Main Phase was considered soft and composites from either side of the 50650 E divide were allowed to influence blocks.

Kriging was in general completed in three passes; the first using ¼ the semivariogram range, the second using ½ the semivariogram range and the third using the full range. In a few cases, on the edges of the deposit, some blocks were still not estimated so the search ellipse was expanded to twice the range. If more than 16 composites were found, the closest 16 were used with a minimum of 4 composites required. The search ellipsoids were oriented parallel to the maximum direction of continuity for each variable with the axis of the ellipsoids a function of the range for the semivariograms in the three principal directions. The search criteria, with ranges of

semivariograms in each direction, are summarized below in Table 22.

Table 22 - Summary of Search Parameters for Main and East Zones

Variable	Domain	Major Axis	Semi Maj. Axis	Minor Axis	Dist. (m) Major Axis	Dist. (m) Semi Maj. Axis	Dist. (m) Minor Axis
Cu	Main Zone Mineralized	Az 45 Dip 0	Az 135 Dip -75	Az 315 Dip -15	200	250	90
Au	Main Zone Mineralized	Az 70 Dip 0	Az 160 Dip -60	Az 340 Dip -30	310	300	170
Cu	East Zone Inner Core	Az 70 Dip 0	Az 340 Dip 0	Az 0 Dip -90	125	50	100
Au	East Zone Inner Core	Az 70 Dip 0	Az 340 Dip 0	Az 0 Dip -90	90	40	90
Cu	East Zone Main Phase	Az 45 Dip 0	Az 135 Dip -75	Az 315 Dip -15	350	250	100
Au	East Zone Main Phase	Az 70 Dip 0	Az 160 Dip -60	Az 340 Dip -30	210	220	60

Note: Pass 1 used ¼ of the distances shown, Pass 2 used ½ while pass 3 used the entire range.

17.2.2 Specific Gravity

During the 1994 and 1995 drill campaigns, specific gravity was measured, on a routine basis, from a representative piece of drill core taken every 8 m throughout each drill hole. A total of 6,934 separate measurements of SG were taken which ranged from a low of 2.07 to a high of 3.46 and had an arithmetic average of 2.785. In 1998 this average was applied to every estimated block to determine block tonnage.

For the 2002 estimate of Red Chris resource, SG was treated as a variable and estimated into every kriged block. The 6,934 measured SG's were assigned to the appropriate 15 m composite for each drill hole. The composites were then used to interpolate an SG estimate into each block that was estimated for Cu and Au. The method of interpolation was inverse distance squared. During the 2003 drill campaign 134 samples of drill core were sent to the Department of Mining and Mineral Processing Engineering, at the University of British Columbia. The process used to measure bulk density was ASTM C92 – Standard Test Methods for Absorption and Bulk Specific Gravity of Dimension Stone– with the following deviations:

1. Several core samples received with 46 mm diameter, undersized for 2" (50.4 mm) minimum dimension for test,
2. Samples heated at 110°C for 48 hours (standard designates 60°C same time period).

The results are listed below in Table 23 for the East Zone. The minimum SG for the East Zone was 2.69 corresponding to sample [03-261 328.39 to 328.57]. The maximum SG for the East Zone was 2.87 corresponding to samples [03-264 148.13 to 148.28], [03-266 32.91 to 33.08], [03-266 145.98 to 146.13] and [03-269 92.22 to 92.36]. The average SG for the East Zone was 2.79.

Table 23 - Specific Gravity determinations for the East Zone

Hole No.	From	To	Rock Unit	Alter. Code	Alteration	Bulk Specific Gravity
03-256	263.6		PPHM		Sericite-quartz-carbonate-pyrite-altered.	2.81
03-261	239.00	239.14	PPHM-QZVZ	N.A.	Quartz-rich Zone with Hm, Mt, Py, and Cp.	2.86
03-261	298.55	298.70	PPHM	1	Weakly potassic-altered main phase.	2.70
03-261	328.39	328.57	DLAT	N.A.	Carbonate and weak clay alteration with Cb veinlets.	2.69
03-262	119.71	119.88	PBRX-QZBX		Strongly silicified brecciated intrusive with Py and Cp.	2.82
03-262	214.27	214.43	PPHM	3	Carbonate-sericite-pyrite-altered with Cb veinlets.	2.73
03-262	393.87	394.00	PBRX-QZBX		Strongly silicified brecciated intrusive with Py and Cb veinlets.	2.79
03-264	26.85	27.00	PBRM	3	Carbonate-sericite-pyrite-altered with Py veinlets.	2.73
03-264	148.13	148.28	PPHM	4	Sericite-carbonate-pyrite-altered with Hm/Mt and Cb veinlets.	2.87
03-264	192.02	192.18	PPHM	3	Sericite-pyrite-carbonate-altered with Py and Cb veinlets.	2.75
03-266	11.35	11.52	PPH2	3	Weakly sericite-carbonate-pyrite-altered.	2.72
03-266	32.91	33.08	QZVZ	N.A.	Pervasively silicified with Hm, Mt, Py, and Cp.	2.87
03-266	64.59	64.76	PPHM-QZVZ	4	Strongly quartz veined with Py and Cp.	2.85
03-266	113.69	113.86	PPHM	4	Carbonate-sericite-altered with quartz veinlets. Minor Hm is present.	2.87
03-266	145.98	146.13	DAND	N.A.	Weak sericite and clay altered with Cb veinlets.	2.81
03-267	147.3		PPHM	1	Potassic-altered.	2.76
03-267	321.6		DLAT	N.A.	Contains Cb veinlets.	2.79
03-269	92.22	92.36	QZVZ	N.A.	Strongly silicified	2.87
03-269	115.82	116.01	PPHM	4	Carbonate-sericite-hematite-pyrite-altered monzonite with Qz and Cb veinlets.	2.84
03-272	44.30		PBRX		Sericite-carbonate-pyrite-altered.	2.78
03-272	153.50		PBRL		Relatively unaltered.	2.73
03-273	131.9				Basalt	2.77
03-275	197.5				Trachytic volcanic ?	2.81

The results for the Main Zone are tabulated below in Table 24. The maximum specific gravity was recorded for sample [03-284 313.64 to 313.80] at a value of 3.07. The minimum specific gravity recorded was 2.57 corresponding to sample [03-292 11.19 to 11.33]. The average specific gravity for all Main Zone samples was 2.80.

For the November 2004 resource estimate, the 2004 measurements for specific gravity were added to the earlier data base and specific gravity was estimated for every mineralized block in the model by inverse distance squared. A search ellipse of 220 m at azimuth 70° dip 0, 150 at azimuth 340° dip 0 and 300 m azimuth 0 dip -90° was used.

Table 24 - Specific Gravity determinations for the Main Zone

Hole No.	From	To	Rock Unit	Alter. Code for interval	Alteration	Bulk Specific Gravity
03-265	43.5		DQCA	N.A.	Relatively unaltered.	2.75
03-271	27.13	27.31	PBRX	3	Sericite-carbonate-pyrite with Qz and Py veinlets.	2.81
03-271	74.22	74.38	PPHM	3	Sericite-pyrite-carbonate-altered with Py and Qz veinlets.	2.88
03-271	100.10	100.28	PPH2	3	Weak sericite and minor carbonate alteration.	2.82
03-271	159.60	159.75	PPHL	2	Weak sericite, clay and pyrite altered with Cb veinlets.	2.72
03-271	162.04	162.19	PPHL	3	Sericite-carbonate-pyrite.	2.69
03-271	175.06	175.23	DAND	N.A.	Weak carbonate and clay with hematite stain. Minor Cb and Hm veinlets.	2.78
03-271	194.46	194.63	PPHM	4	Sericite-pyrite-carbonate-altered with Py and Qz veinlets.	3.00
03-271	255.57	255.73	PPHM	4	Sericite-pyrite-quartz-altered with Py veinlets.	2.94
03-271	313.09	313.25	PPHL	2	Weak sericite-pyrite with Hm stain.	2.76
03-271	353.26	353.46	PPHM	1	Potassic alteration with small Py veinlets.	2.73
03-272	44.3		PBRX	2	Sericite-pyrite-carbonate-altered	
03-272	153.5		PBRL	N.A.	Relatively unaltered	
03-274	14.78	14.94	PPHL	2	Sericite-pyrite-altered over older carbonate; Cb and Py veinlets.	2.64
03-274	82.85	83.02	PPH2	3	Sericite-pyrite-carbonate-altered; Cb and Py veinlets.	2.74
03-274	213.06	213.21	PPHM	3	Carbonate-sericite-pyrite-altered with Py and Cb veinlets.	2.82
03-274	255.73	255.88	DOCA	N.A.	Carbonate-clay altered with Py and Cb veinlets.	2.85
03-274	281.38	281.54	PPHM	4	Strongly silicified with quartz veining. Contains Py, Cp and Cb veinlets.	2.96
03-277	67.90	68.05	DLAT	N.A.	Weak carbonate, clay, sericite and pyrite.	2.70
03-277	126.96	127.10	PPHM	3	Sericite-carbonate-pyrite with Qz and Py veinlets.	2.77
03-277	192.94	193.08	PPHM	3	Carbonate-sericite-pyrite-clay with Qz veinlets.	2.78
03-277	209.21	209.35	PPHM	3	Weak carbonate-sericite-pyrite with Cb and Py veinlets.	2.76
03-277	245.97	246.13	PPHM	1	Weak potassic - primary k-feldspar still visible.	2.73
03-277	254.20	254.35	DOCA	N.A.	Weak clay and carbonate with Cb and Py veinlets.	2.75
03-277	258.59	258.75	PPH2	2	Weak sericite-pyrite-carbonate.	2.80
03-277	270.97	271.13	PPHM	4	Carbonate-sericite-clay-pyrite with Qz and Py veinlets.	2.84
03-277	329.84	330.00	PPHM	3	Weak carbonate-sericite-pyrite with Cb and Py veinlets.	2.72
03-279	18.59	18.73	PPHM	2	Sericite-carbonate-pyrite with Cb and Py veinlets.	2.94
03-279	50.22	50.39	PPHL	3	Weak carbonate-sericite-pyrite.	2.75
03-279	86.31	86.45	PPHM	5	Propylitic with chlorite, epidote and magnetite.	2.79
03-279	136.55	136.71	DAND	N.A.	Weak clay.	2.80
03-279	215.95	216.09	PPHM	3	Carbonate-sericite-clay-pyrite with Qz and Py veinlets.	2.96
03-279	267.92	268.07	PPHM	4	Carbonate-sericite-clay-pyrite with Qz and Py veinlets.	2.99
03-279	292.15	292.30	PPHM	4	Carbonate-sericite-clay-pyrite with Qz and Py veinlets.	2.98
03-279	309.82	309.98	DAND	N.A.	Weak clay and likely sericite as well.	2.84
03-284	14.31	14.45	PPHM	2	Sericite-pyrite-altered.	2.70
03-284	26.26	26.41	PPHM	3	Carbonate-sericite-pyrite-altered.	2.66
03-284	72.24	72.38	PPHM	3	Weak sericite-carbonate-pyrite alteration.	2.75
03-284	102.03	102.20	PPHM	1-3	Weak potassic alteration (or unaltered) with carbonate and Py veinlets.	2.93
03-284	132.32	132.45	PPHM	4	Carbonate-sericite-clay-pyrite-altered.	2.72
03-284	144.72	144.90	PPHM	1	Weak potassic alteration with carbonate and Py veinlets.	2.74
03-284	207.63	207.80	DOCA	N.A.	Weak clay and likely sericite as well.	2.78
03-284	222.65	222.79	PBRX-PPH	4-1	Sericite-carbonate-pyrite-altered with Qz and Py veinlets.	2.88
03-284	269.60	269.75	PPHM	1	Potassic alteration with Qz veinlets.	2.68
03-284	313.64	313.80	PPHM	4-2	Sericite-carbonate-pyrite-clay-altered with Qz veinlets.	3.07
03-284	330.40	330.54	DOCA	N.A.	Weak clay-carbonate alteration.	2.80
03-284	339.85	339.98	PPHM	2-4	Sericite-pyrite-altered with minor clay and carbonate.	2.78
03-284	363.67	363.82	PPH2	3	Carbonate-sericite-pyrite-altered.	2.71

Table 24: Continued

Hole No.	From	To	Rock Unit	Alter. Code for interval	Alteration	Bulk Specific Gravity
03-287	14.63	14.77	PBRX-PPHM	3	Carbonate-sericite-pyrite. Sample is of PPHM.	2.85
03-287	41.43	41.58	PPH2 or PPHL	1	Weak local potassic with hematite and magnetite.	2.72
03-287	57.37	57.52	PPHM-PBRX	3	Carbonate-sericite-pyrite with To and Py-rich bands. Sample is from PBRX.	2.79
03-287	74.16	74.33	PPH2	1-4	Weakly potassic-altered then weakly sericitized and carbonate-altered.	2.74
03-287	114.25	114.41	PPHM	4	Sericite-carbonate-pyrite with trace chlorite; with Py and Qz veinlets.	2.80
03-287	133.22	133.38	PPHM	1	Potassic with K-feldspar, minor biotite, and Qz and Py veinlets.	2.74
03-287	187.00	187.15	PPHM	4	Sericite-carbonate-pyrite. Trace chlorite. Contains Py veinlets.	2.88
03-287	218.54	218.70	PPHM	2-4	Sericite-pyrite overprint of carbonate-sericite-clay-chlorite assemblage. Py and Qz veinlets.	2.85
03-287	267.92	268.08	PPHM	4	Sericite-carbonate-pyrite. With Py and Qz veinlets.	2.95
03-287	292.30	292.43	PPHM	2	Sericite-carbonate-pyrite with Py and Qz veinlets.	2.88
03-287	328.72	328.87	PPHM	3	Carbonate-sericite-pyrite with Py veinlets.	2.70
03-287	359.36	359.51	PPH2	2	Sericite-pyrite-altered with Cb veinlets.	2.81
03-287	377.65	377.80	PPHM	3	Carbonate-sericite-pyrite with Py veinlets.	2.92
03-290	43.23	43.36	DOCA	N.A.	Weak clay-carbonate-pyrite alteration.	2.79
03-290	128.92	129.06	PPHM	4	Sericite-carbonate-pyrite-altered with Qz and Py veinlets.	2.80
03-290	154.44	154.60	PPHL	3	Weak sericite-carbonate-pyrite alteration.	2.78
03-290	225.25	225.40	PPHM	4	Carbonate-sericite-pyrite-altered with trace Hm and Mt. Contains Py veinlets.	2.80
03-290	254.20	254.35	PPHM	4	Sericite-carbonate-chlorite-pyrite-altered with Py and Qz veinlets and minor Mt.	2.85
03-290	281.33	281.48	PPHM	1-4	Sericite-carbonate-chlorite-pyrite-altered with Py and Qz veinlets.	2.98
03-290	319.43	319.59	PPH2	2	Sericite-pyrite-carbonate-altered. May have been albitized.	2.76
03-290	361.35	361.49	DOCA	N.A.	Carbonate-clay-altered with Cb/Qz veinlets and Hm.	2.77
03-290	368.12	368.28	PPHM	1	Potassic-alteration with Qz and Py veinlets.	2.73
03-292	11.19	11.33	PPHL	N.A.	Weak sericite-carbonate-pyrite alteration. Local Cb veinlets and limonite stain.	2.57
03-292	98.00	98.15	PPHL	3	Weak sericite-carbonate-pyrite alteration. Local Cb veinlets.	2.68
03-292	123.93	124.07	DPFH	N.A.	Carbonate and clay altered with Cb veinlets.	2.69
03-292	188.67	188.83	DIKE	N.A.	Cb veinlets.	2.77
03-292	228.30	228.47	PPHM	4	Sericite-carbonate-pyrite. Trace chlorite. Trace hematite and magnetite. With Qz and Py veinlets.	2.79
03-292	264.87	265.02	PPHM	4	Sericite-carbonate-pyrite. Trace chlorite. Trace hematite and magnetite. With Qz and Py veinlets.	2.88
03-292	340.62	340.78	PPHM	4	Sericite-carbonate-pyrite. Trace chlorite. Trace hematite and magnetite. With Qz and Py veinlets.	2.84
03-292	352.56	352.71	DAND	N.A.	Very weak carbonate altered.	2.81
03-292	412.81	412.95	PPHM	4	Sericite-carbonate-pyrite. Trace chlorite. Trace hematite and magnetite. With Qz and Py veinlets.	2.84
03-292	437.22	437.40	PPHM	1	Potassic but partially overprinted by sericite, carbonate and chlorite. With Py and Qz veinlets.	2.76
03-294	35.58	35.75	PPH2 or PPHL	1	Weak carbonate-sericite-pyrite-altered. With Py veinlets.	2.71
03-294	106.20	106.38	PPH2	3	Sericite-carbonate-clay-altered. Formerly albitized. Contains Py veinlets.	2.82
03-294	119.25	119.42	DPFH	N.A.	Weak clay and carbonate; contains Py, To and Gy veinlets.	2.73
03-294	145.74	145.89	PPHM	4	Sericite-carbonate-hematite-pyrite-altered with Qz and Py veinlets.	2.87
03-294	149.33	149.49	PPHM	1-4	Potassic alteration is overprinted by sericite-carbonate-pyrite. Contains Py and Qz veinlets.	2.86
03-294	167.34	167.51	PPHM	4	Sericite-chlorite-carbonate-pyrite-altered. Contains Qz and Py veinlets. Minor K-Feldspar.	2.74
03-294	223.27	223.42	PPHM	4	Sericite-carbonate-hematite-pyrite-altered with Qz and Py veinlets.	2.88
03-294	256.39	256.53	PPHM	4-2	Carbonate-sericite-pyrite-hematite-altered with Qz and Py veinlets. Some late sericitic alteration.	2.80
03-294	283.10	283.26	PPHM	2	Sericite-pyrite-altered with Qz, Py, and Cb veinlets.	2.86
03-294	347.26	347.42	PPHM or PPH2	3	Carbonate-sericite-pyrite-altered with Qz, Py, and Cb veinlets.	2.84
03-294	370.45	370.63	PPHM	3	Carbonate-sericite-pyrite-altered with Qz and Py veinlets.	2.87
03-294	405.08	405.23	PPHM	4	Sericite-carbonate-pyrite-hematite-altered with Qz and Py veinlets.	2.83
03-294	435.01	435.16	DOCA	N.A.	Weak carbonate-sericite-pyrite alteration with Cb and Qz veinlets.	2.81
03-295	12.70	12.90	PPHM	3	Carbonate-sericite-pyrite-altered with Py and Cb veinlets.	2.72
03-295	35.86	36.03	PPHM	3	Carbonate-sericite-pyrite-altered with Py, To, and Cb veinlets.	2.77
03-295	72.93	73.08	PPHM	2	Sericite-pyrite-carbonate-altered with Py, To, and Qz veinlets.	2.80
03-295	92.90	93.05	PPH2	3	Carbonate-sericite-pyrite-altered.	2.70
03-295	124.17	124.32	PPHM	3	Carbonate-sericite-pyrite-altered with Py veinlets.	2.82
03-295	144.26	144.42	DLAT or DOCA	N.A.	Weak carbonate-sericite-altered.	2.85
03-295	150.02	150.21	PPHM	3	Carbonate-pyrite-sericite-altered with Qz and Py veinlets and To patches.	2.83
03-295	167.34	167.51	PPHM	3	Carbonate-sericite-pyrite-altered with Py and Cb veinlets.	2.82
03-295	201.37	201.52	PPHM	4	Sericite-carbonate-chlorite-clay-pyrite-altered with Qz and Py veinlets.	2.94
03-295	230.00	230.14	PPH2 - PBRX	3	Carbonate-sericite-pyrite-altered.	2.78
03-295	240.19	240.36	PPHL	3	Sericite-pyrite-altered. Minor carbonate in matrix. Contains Qz-Py-Cb and Cb veinlets.	2.81
03-295	252.31	252.47	PPHM	4	Sericite-carbonate-clay-pyrite-altered with Hm, Py and Qz veinlets.	2.80
03-295	277.17	277.33	PBRX	4	Sericite-carbonate-pyrite-hematite-altered with Qz and Py veinlets.	2.85
03-295	324.76	324.92	PPHL	1-2	Weak sericite-pyrite-alteration and Hm stain.	2.77
03-295	394.78	394.93	PPHL	3	Carbonate-sericite-pyrite-altered with Cb and Py veinlets.	2.77
03-295	408.13	408.28	PPHM	4	Carbonate-sericite-pyrite-altered with Py and Qz veinlets.	2.85

17.2.3 Block Tonnage Calculation

Estimated blocks were assigned a tonnage based on proportion of block below the topographic surface, the estimated specific gravity, and the volume of a block 20 x 20 x 15 m in dimension.

17.2.4 Results

Examples of the grade estimates for copper are presented as four level plans 1290, 1335, 1380 and 1425 shown as Figures 15 to 18 respectively. In each figure the blocks are colour coded by kriged copper values as shown in the Legend. The drill hole composites are also shown to demonstrate the data available to make the estimates. While gold and classification are also shown in the blocks they are not meant to be read at this scale, rather the plots are shown purely to demonstrate copper distribution.

Figure 15 - Kriged Resource Blocks 1290 Level

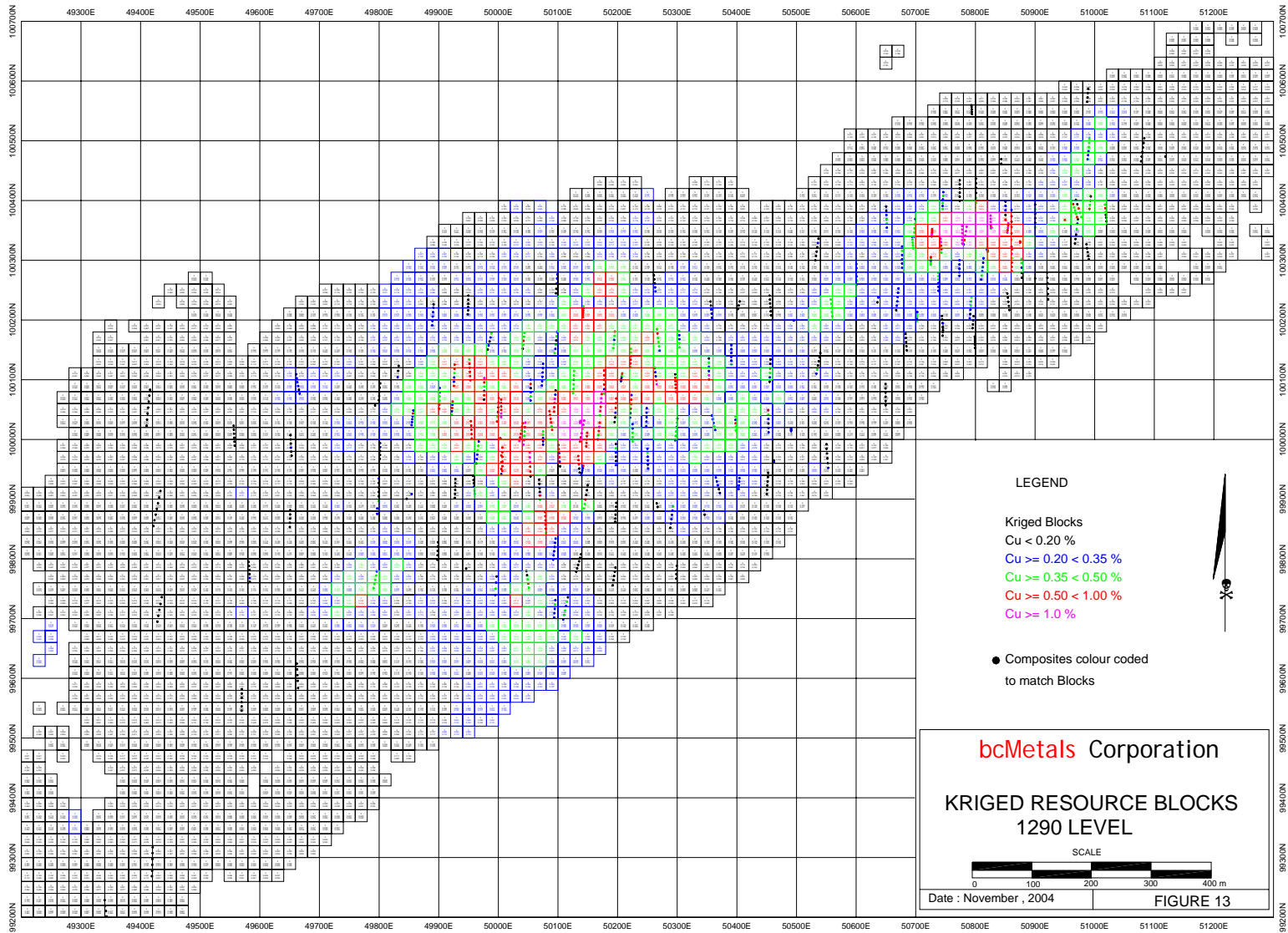


Figure 16 - Kriged Resource Blocks 1335 Level

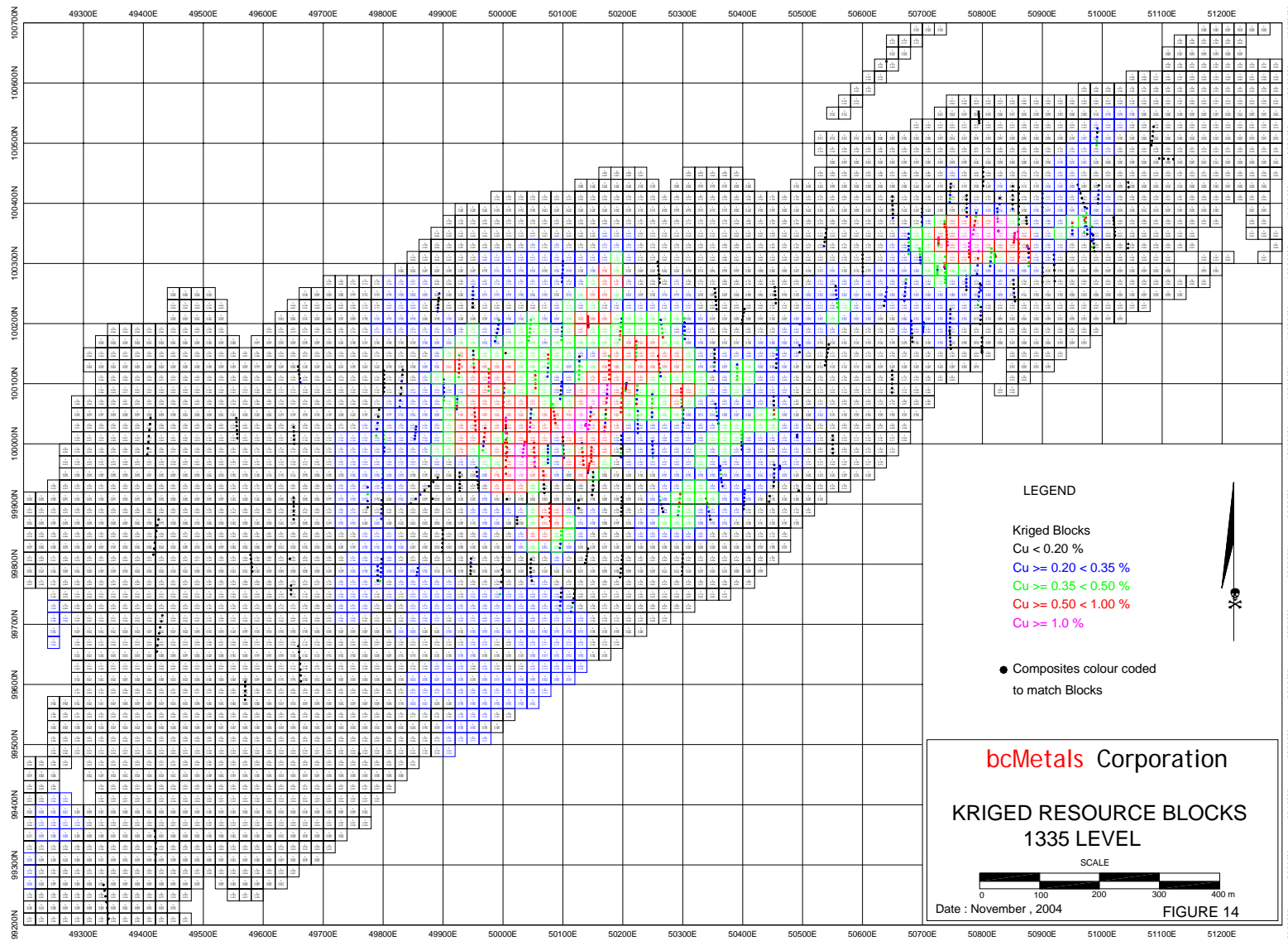


Figure 17 - Kriged Resource Blocks 1380 Level

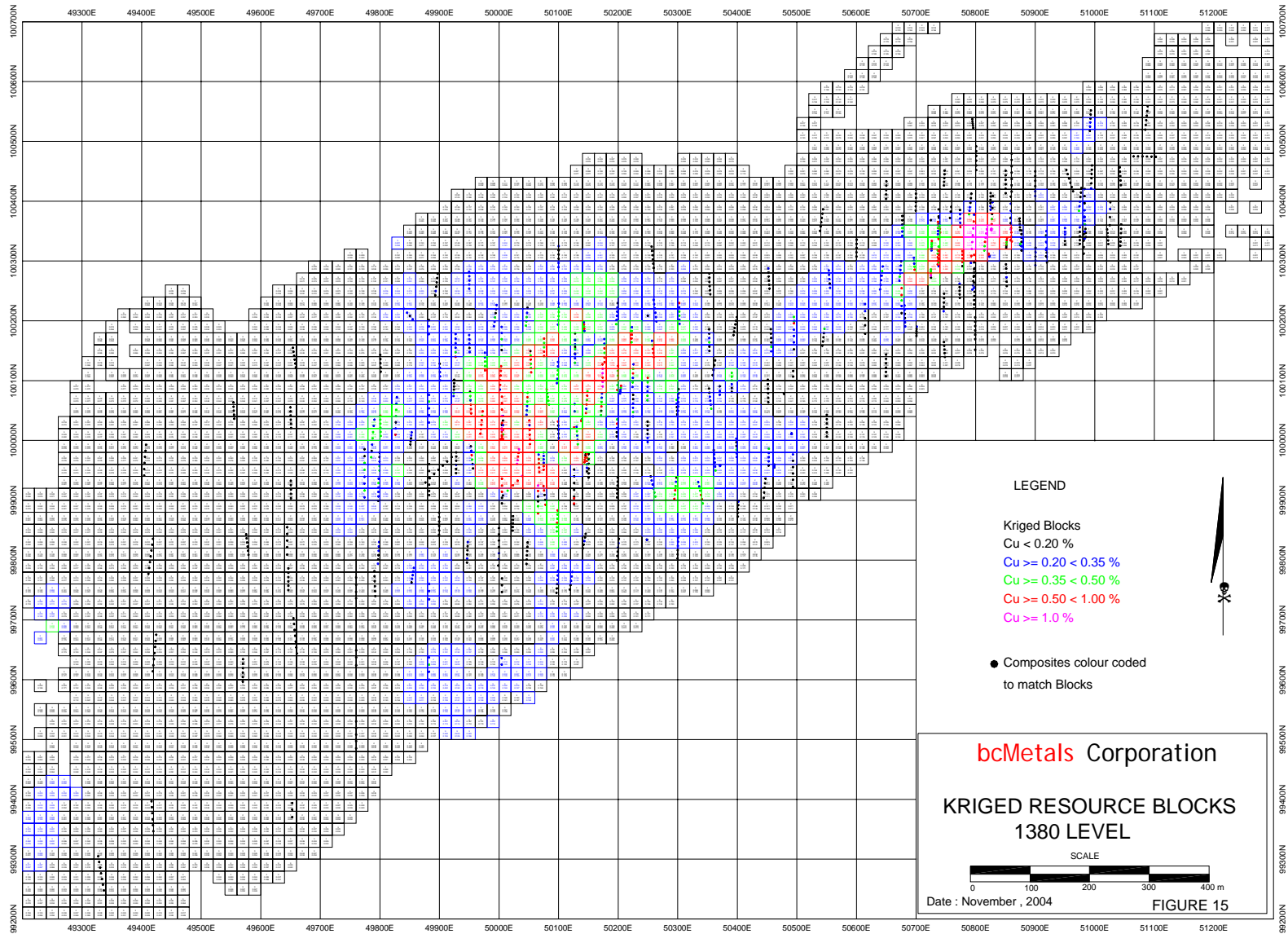
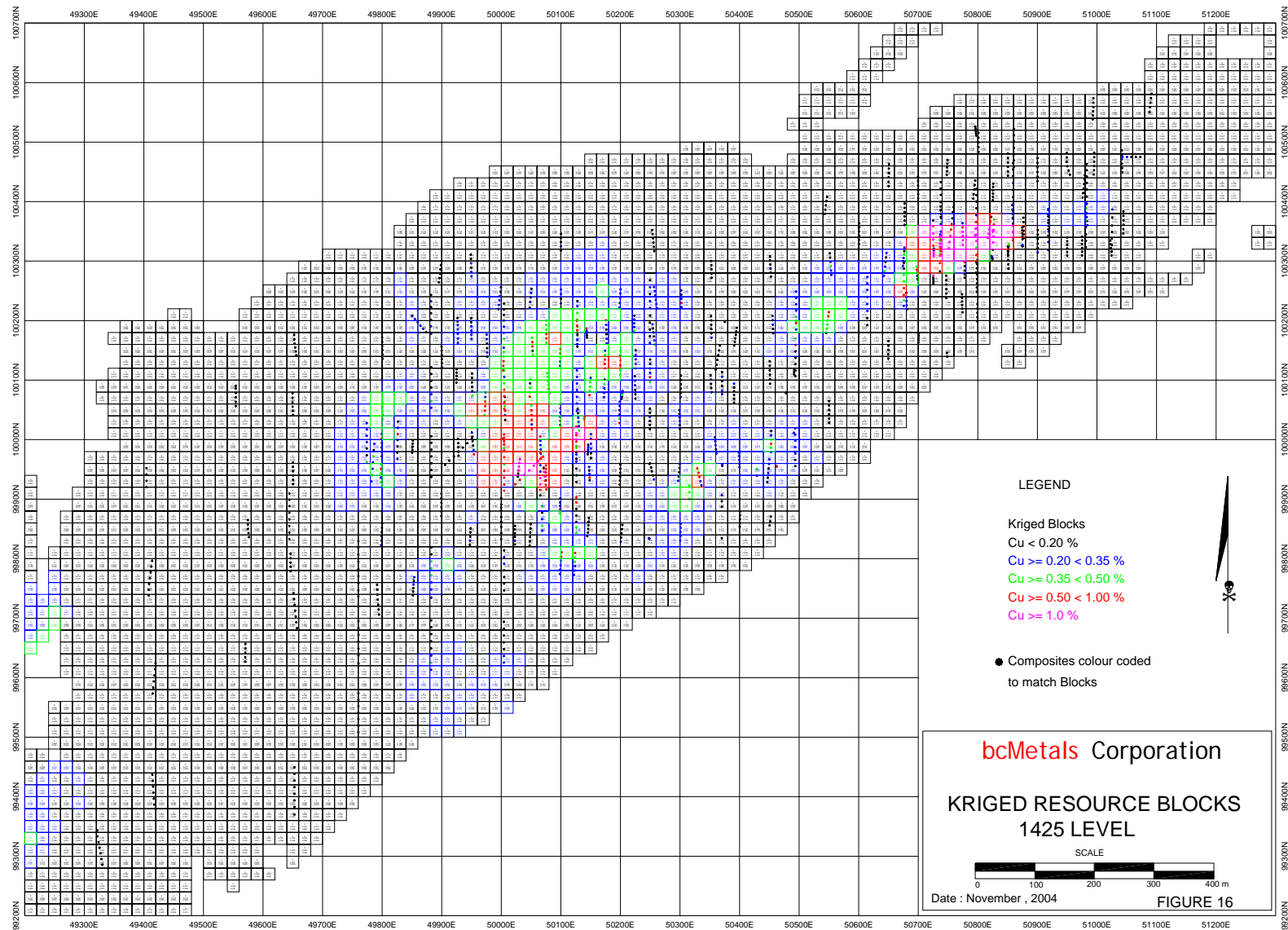


Figure 18 - Kriged Resources Blocks 1425 Level

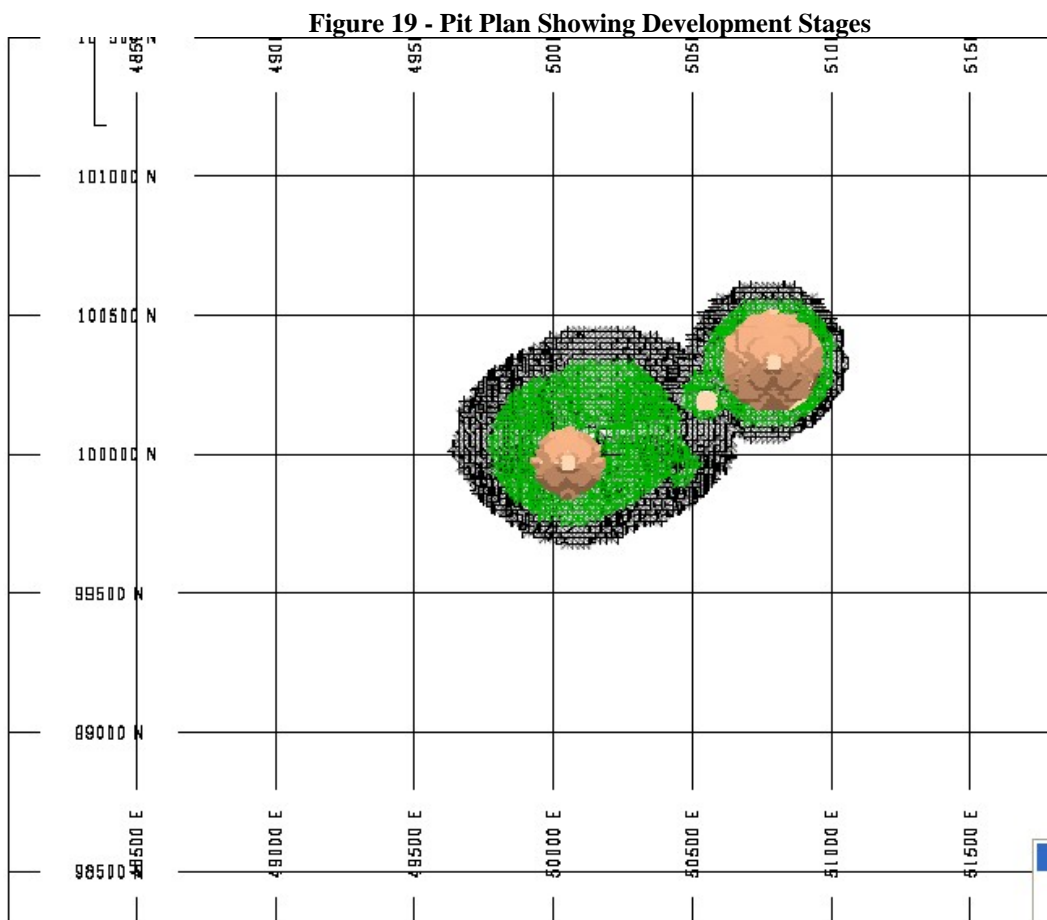


18.0 ADDITIONAL REQUIREMENTS for TECHNICAL REPORTS on DEVELOPMENT PROPERTIES and PRODUCTION PROPERTIES

18.1 Geotechnical

18.1.1 Mine Geotechnical

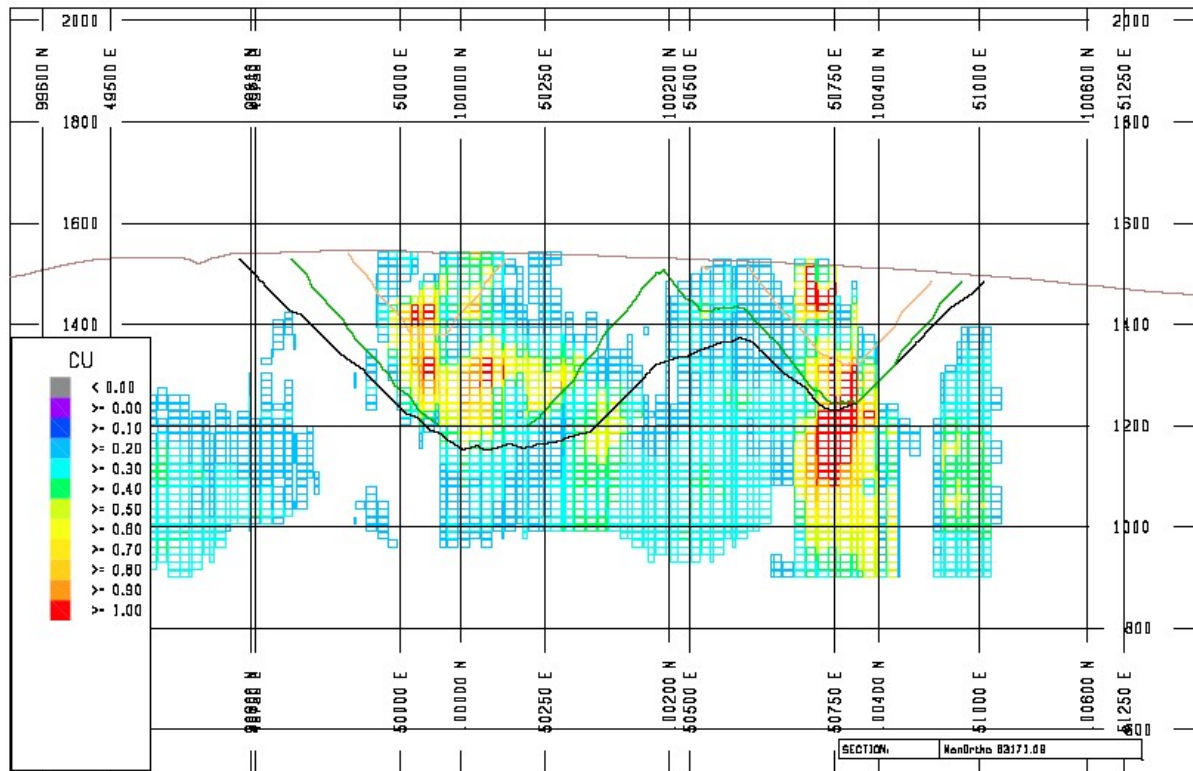
The Red Chris Open Pit encompasses about a 53 hectare area on the crest. The open pit design currently comprises a phased development with two separate pit bottoms located in the western and eastern ends. Pit floor in the eastern section is approximately 1230 m while the western end is deeper, with an expected pit floor at approximately 1150m. The pit highwall is approximately 400 m high and is to be located on the north wall. Figure 19 displays a schematic of the pit and proposed stages of development while Figure 20 displays a longitudinal section.



(Stage 1-Brown, Stage 2-Green, Stage 3-Black)

Geology within the pit walls is expected to comprise primarily of the Red Stock monzodiorite, with intersections of the Bowser Sediments and Dynamite Hill Volcanics in the southern and northern pit walls respectively. Dominant structure consists of steeply dipping shears and faults within the Red Stock that trend predominantly northeast southwest

Figure 20 - Longitudinal Section



Geotechnical investigations within the pit area were completed by American Bullion, and Knight and Piesold. Geotechnical information was gathered and analyzed from these programs and included data from both exploration drilling and geotechnical drilling. In 2004 AMEC completed four additional geotechnical drillholes into and behind the proposed pit walls in the north and south to confirm information gathered by others as well as identify structural controls that may exist behind the pit walls. These drillholes were geotechnically logged, the core was oriented and an acoustic borehole camera was used to confirm orientations and rock mass conditions in two of the four drillholes.

After review and analysis of the above geotechnical data, the open pit was divided into four geotechnical domains based on structural and lithologic characteristics for design purposes. Kinematic, probabilistic and rock mass stability analysis methods were used to determine pit slope bench face, inter-ramp and overall slope angles for the various domains and wall orientations. To optimize pit design, bcMetals is committed to implementation of a significant wall control and drainage program during pit development to reduce likelihood of damage and failure of the pit walls during development. As a result, an aggressive pit wall slope design was identified. The success of the wall control and drainage programs and subsequent back analysis and data gathering during the Stage I Pit development will be key in the performance of the Stage I Open Pit slopes and future design stages.

Table 25 - Summary of Proposed Pit Wall Designs

Domain	Wall Aspect or Dip Direction (Pit Design Sector)	Final Walls Achievable Configuration (30 m high benches)		Stage I Pit Slope Configuration**	
		Bench Face Angle/ Berm Width	Inter Ramp Angle	Bench Face Angle/ Berm Width	Inter Ramp Angle
I Eastern Red Stock (From 50400N east)	120° to 180° (300° to 000°)	60°/10 m	48°	65°/10 m	51°
	180° to 240° (000° to 060°)	60°/13 m	45°	60°/13 m	45°
	240° to 360° (060° to 180°)	60°/10 m	48°	65°/10 m	51°
III Volcanics	120° to 150° (300° to 330°)	65°/10 m	51°	65°/8 m	54°
	150° to 180° (330° to 360°)	65°/11 m	50°	65°/11 m	50°
	180° to 210° (360° to 030°)	65°/16 m	45°	70°/19 m	45°
	210° to 240° (030° to 060°)	65°/10 m	51°	70°/10 m	55°
IV Western Red Stock (From 50400N west)	330° to 120° (150° to 300°)	60°/10 m	48°	65°/10 m	51°
	120° to 210° (300° to 030°)	65°/10 m	51°	65°/8 m	54°

Notes

1. IRA is Inter Ramp Angle

2. BF is Bench Face

** Based on the design and excavation guidelines provided by C.O. Brawner – September 17, 2004.

Table 26 - Proposed Slope Designs based on Rock Mass Strengths

Domain	Wall Aspect or Dip Direction (Pit Design Sector)	Inter Ramp Slope Angle	Total Slope Height (m)	Slope Configuration (15 m high Benches)
II Bowser Sediments and Faulted Contacts	All	35°	150 m	BFA = 60° Berm Width = 13 m

18.1.2 Tailings Disposal

The tailings impoundment will be developed in the valley to the northeast of the open pit and plantsite. Drawings A1-143673-30-C-0011 through 0017, 0019 and 0026 (Figures 21 - 29) illustrate the tailings management system. The tailings impoundment will provide storage for all process tailings generated by the project. The tailings will be conveyed from the plant site to the tailings impoundment via pipeline where they will be discharged from the North and/or South Dam. Tailings will not be discharged from the smaller Northeast Dam, which is not required to be constructed until the latter years of the mine life. The tailings distribution system is designed to manage the changing requirements of the tailings impoundment development over the life of the mine. This includes provision for development of required tailings beaches and for construction of the cycloned tailings sand downstream shells of the North and South dams.

All three dams will have a central core of compacted glacial till fill of low hydraulic conductivity. The till core of each dam will be keyed into the native till that blankets most of the impoundment area and hence forms a natural “liner” that will serve to limit the rate of seepage loss from the impoundment. The starter dams for the North Dam and the South Dam will have upstream and downstream shells of compacted granular fills (sand and gravel) that are present within the impoundment area. Borrow operations for the starter dams will concentrate on obtaining granular fill from those areas that will be flooded in the first few years once the impoundment is commissioned. Subsequent raising of the North Dam and the South Dam, which will be undertaken on an annual basis, will involve upward extension of the compacted till core via the centerline raising method, with the downstream shell extended and raised using compacted non acid generating (NAG) cycloned sand, hydraulically placed. The upstream shell of the dam will also be constructed using cycloned tailings sand. Runoff and construction water draining from the hydraulic fill placement areas on the downstream shells of both these dams will report to sedimentation and seepage collection ponds, from which water will be reclaimed to the tailings pond.

The embankment zones and specifications as shown on the drawings are summarized in Table 27 below.

Table 27 - Tailing Dam Construction Material Specifications

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

The Northeast Dam will have a central core of compacted glacial till, with upstream and downstream shells of granular fill, and appropriate filter zones. The Northeast Dam will be constructed and operated as a water-retaining dam, so for closure will include riprap zones on the upstream face of the dam for erosion protection.

Tailings will be discharged into the impoundment from both the North Dam and the South Dam. As the impoundment nears closure, it will be NAG tailings that are discharged from the dams, so that PAG tailings are fully submerged at closure, and NAG above-water tailings beaches are left upstream of both dams as part of the closure configuration of the tailings impoundment.

Above-water tailings beaches will likely not form upstream of the dams until perhaps the 2nd or 3rd year of the impoundment's operating life. The reclaim pond will initially form at about the midpoint of the main north-south valley. As the impoundment level rises, the water pond will also extend into the northeast arm of the tailings impoundment. The floating reclaim barge will be maintained against the west slope of the main north-south valley. Reclaim of water from the tailings pond to the mill process will be maximized to the extent that fresh water makeup supply requirements can be optimized.

Development and maintenance of above-water tailings beaches to the upstream of the North Dam and South Dam will be greatly facilitated by the ability to produce coarse cycloned tailings sand. This allows placement in cells to effectively control beach development, and results in more steeply sloping beaches that more effectively provide physical separation between the reclaim water pond and the dam crests.

Cycloned sand will be discharged to the downstream shells of the North and South Dams for placement and compaction via hydraulic fill methods, in a manner as practiced at a number of tailings dams in British Columbia, two examples being the L-L Dam at the Highland Valley Copper Mine, and at the Kemess Mine. The sand will be discharged at a density of 65% solids by weight. Runoff water decanted from the construction cells, and draining from the sand, will report to the downstream seepage and sedimentation ponds. In these ponds, suspended solids will settle out and the water will be pumped over the dams and into the tailings impoundment. Each winter, when sand is not being placed on the downstream shells of the dams, excess tailings fines that have settled out in the downstream ponds will be trucked for dumping in the tailings impoundment, to restore sedimentation capacity for the subsequent year's downstream shell construction operations.

During the operational phase of the tailings impoundment, acid rock drainage (ARD) will not develop on the basis of the following:

- The high alkalinity of the tailings pond water will act as a buffer and prevent the development of ARD.

- As the tailings impoundment is raised, most of the tailings will be flooded, which will inhibit oxygen flux and hence also protect against ARD development. Only the sub-aerial beaches maintained upstream of the North Dam and the South Dam will not be flooded, but these will be continually buried (and flooded) as additional tailings are deposited as the impoundment level rises.
- Rougher tailings used for the production of cycloned sand for dam construction will, prior to two-stage cycloning, be subjected to an extra flotation to remove any residual sulphides. As such, the cycloned sand produced for dam construction will be NAG.
- For the last two to three years of active tailings discharge into the impoundment, the NAG tailings produced via flotation of the rougher tailings will be kept separate from the remainder of the tailings stream (sulphide-bearing cleaner tailings and the sulphides rejected from the additional float of the rougher tailings). The NAG tailings will be discharged from the North Dam and the South Dam to form above-water, NAG tailings beaches that will be in front of the dams as part of the closure configuration of the impoundment. Development of these beaches will be facilitated through use of cycloned sand, which will allow cell construction (i.e. controlled beaching), effectively “stacking” beaches against the dam. This will effectively widen the crests of the two dams, and will facilitate trafficability for subsequent growth medium placement and vegetation for reclamation of the beaches.
- During the final years of operation, when only NAG tailings are being discharged from the dams, the sulphides-bearing tailings stream will be directed to the deep portion of the reclaim pond for discharge to ensure their permanent submergence.

At closure, the alkalinity of the tailings pond water will gradually reduce. However, all PAG tailings will be submerged by water and/or a NAG tailings blanket, and hence will not generate ARD. Closure and reclamation of the tailings impoundment will comprise the following:

- NAG above-water tailings beaches will be maintained upstream of both the North Dam and the South Dam. To the extent practical (based on trafficability constraints), these beaches will be covered by a growth medium and vegetated.
- Each of the three dams will be covered with a growth medium and vegetated with appropriate local species.
- The diversion ditches will be breached, as they are no longer required at closure.
- The seepage dams will be breached, with any residual tailings fines in the seepage/sedimentation ponds covered and reclaimed.
- A closure spillway, capable of routing the Probable Maximum Flood (PMF) event, will be constructed on the right (looking downstream) abutment of the Northeast Dam.
- All pumping and piping equipment will be removed.
- Access road surfaces will be regraded and reclaimed.
- Protection for monitoring installations (required through the closure phase for the dams) will be provided.

Figure 21 - Site Plan

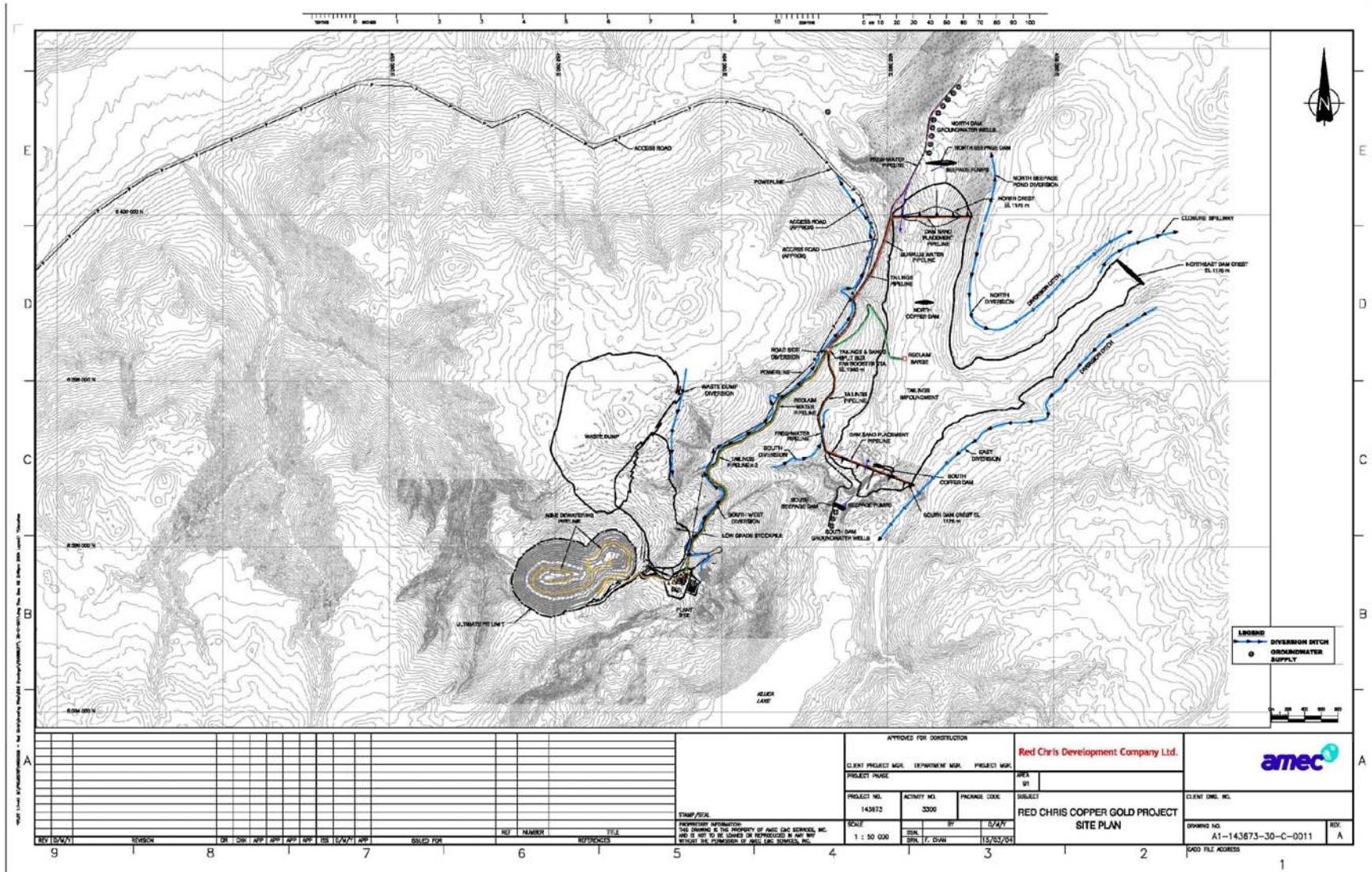


Figure 22 - Site Investigation Plan

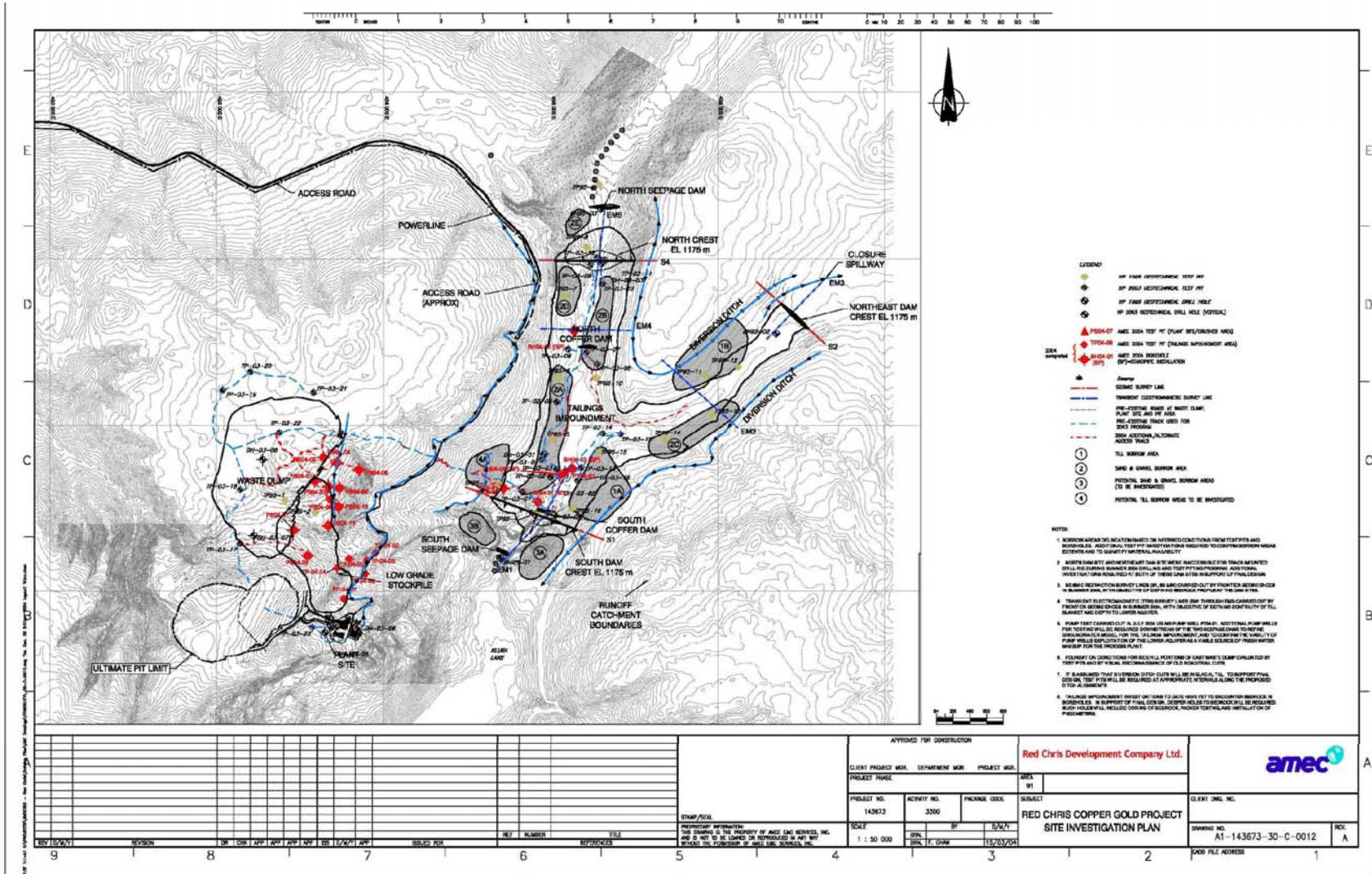


Figure 23 - Typical Dam Cross Section (North Dam)

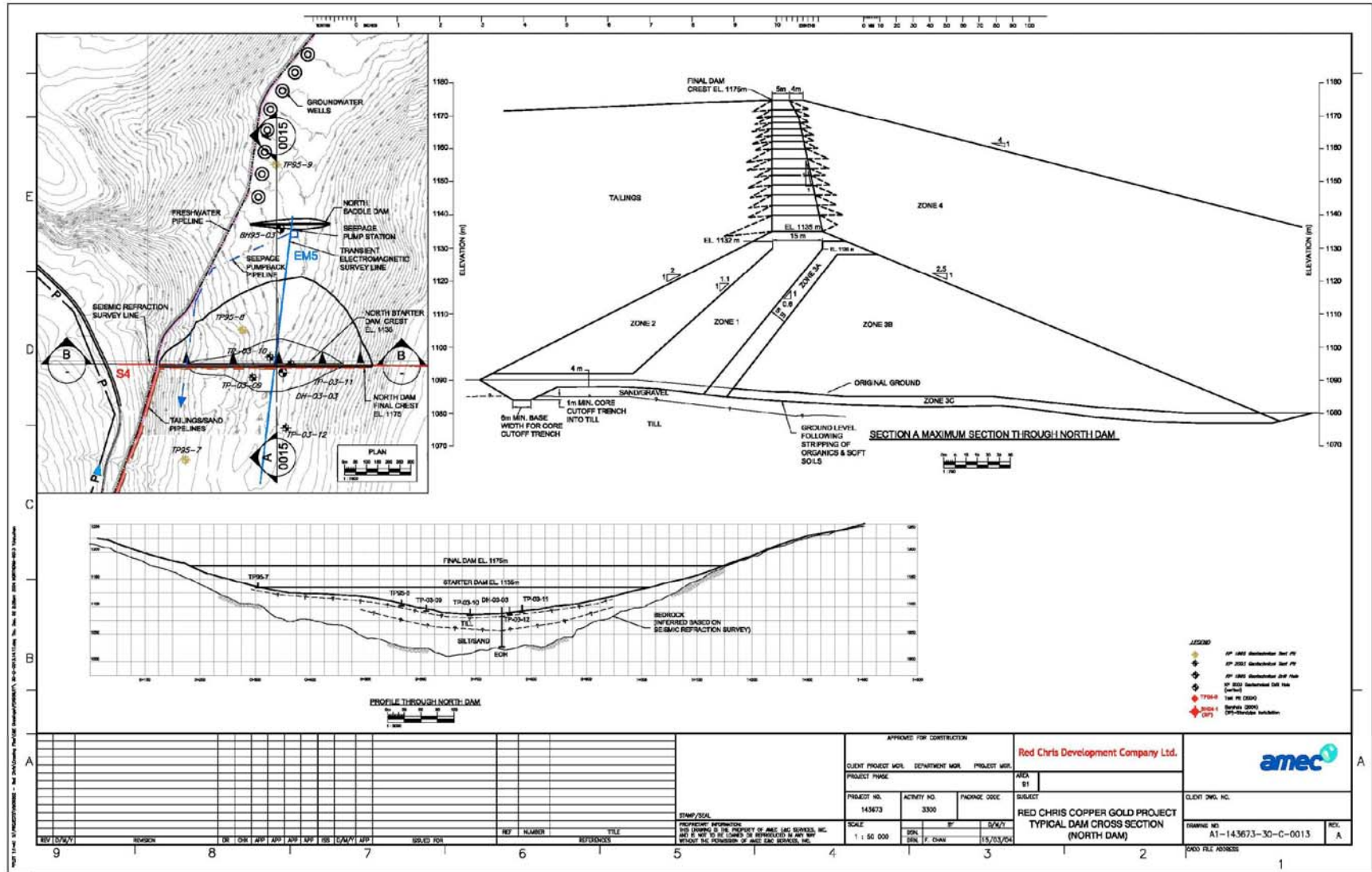


Figure 24 - Typical Dam Cross Section (South Dam)

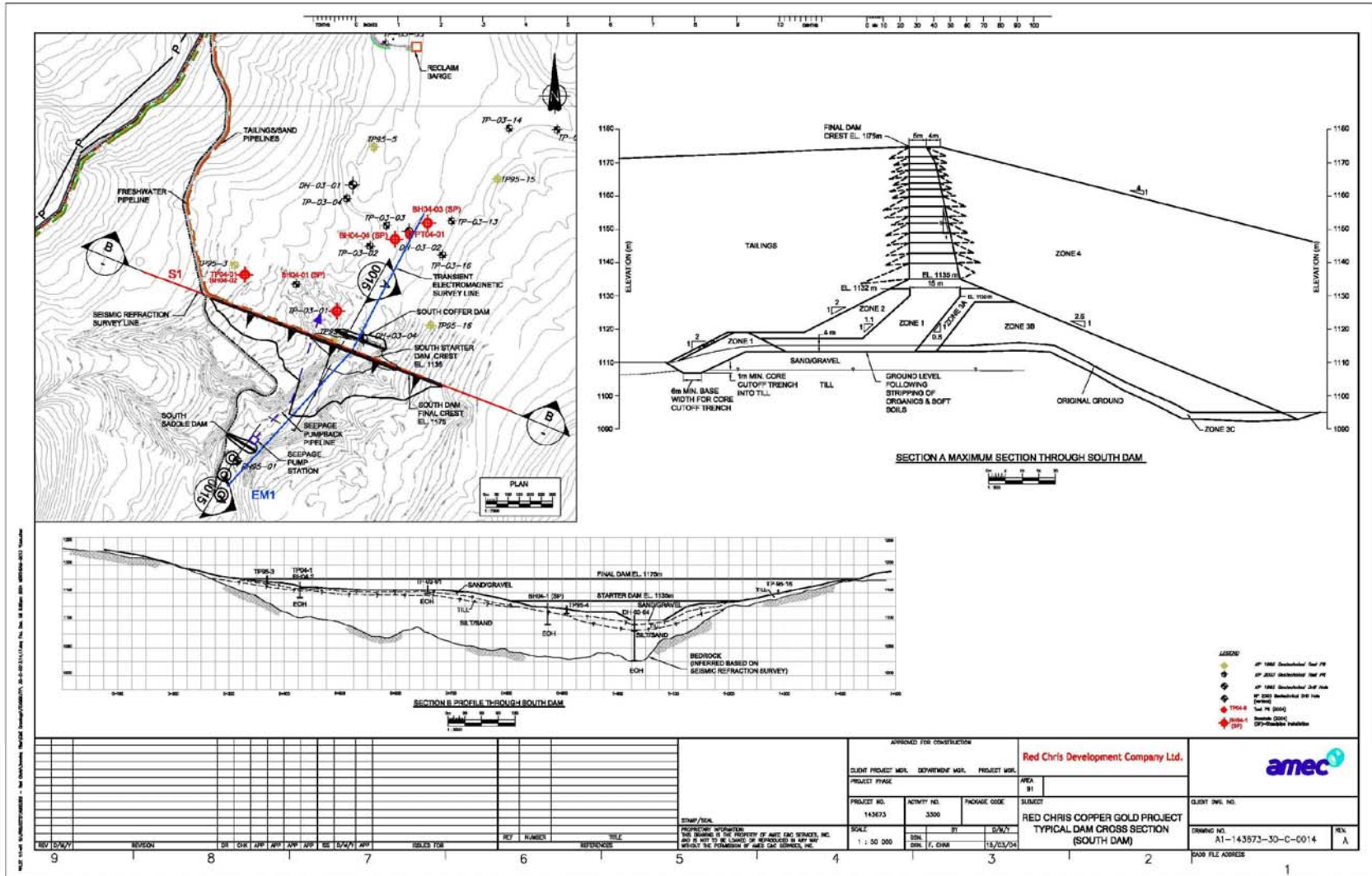


Figure 25 - Typical Dam Cross Section (North, South Dams)

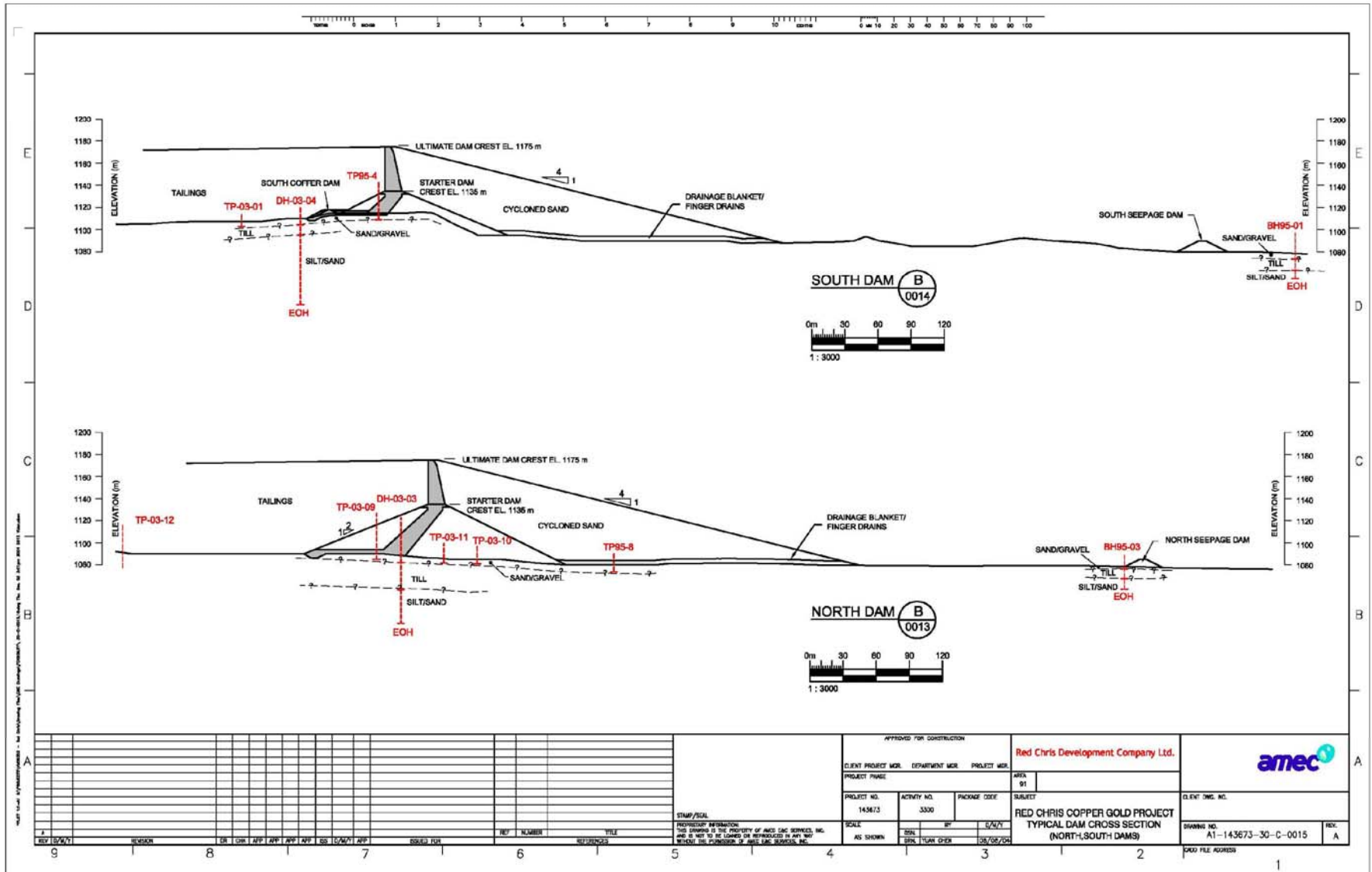


Figure 26 - Typical Dam Cross Section (Seepage Dams)

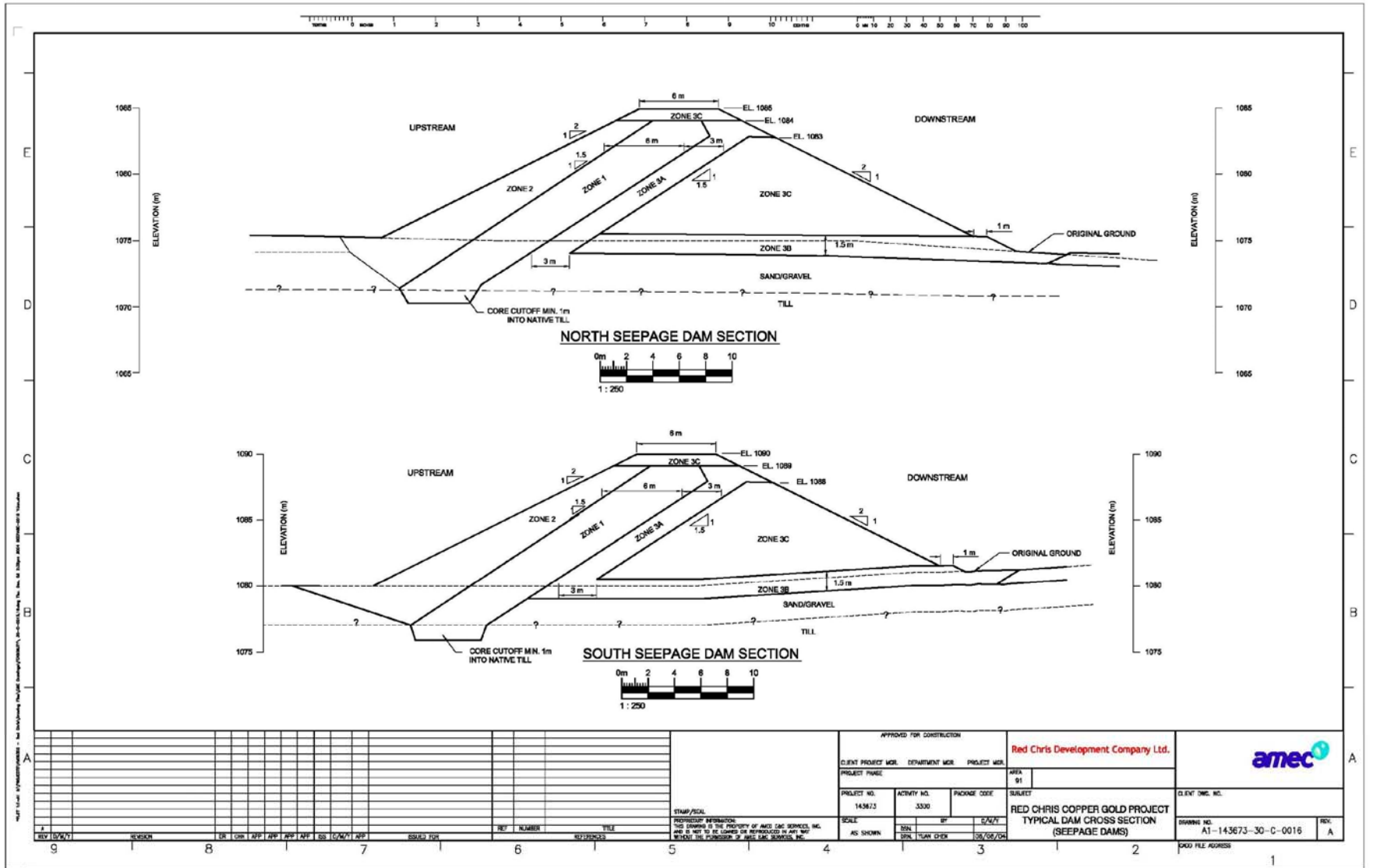


Figure 27 - Typical Dam Cross Section (Northeast Dam)

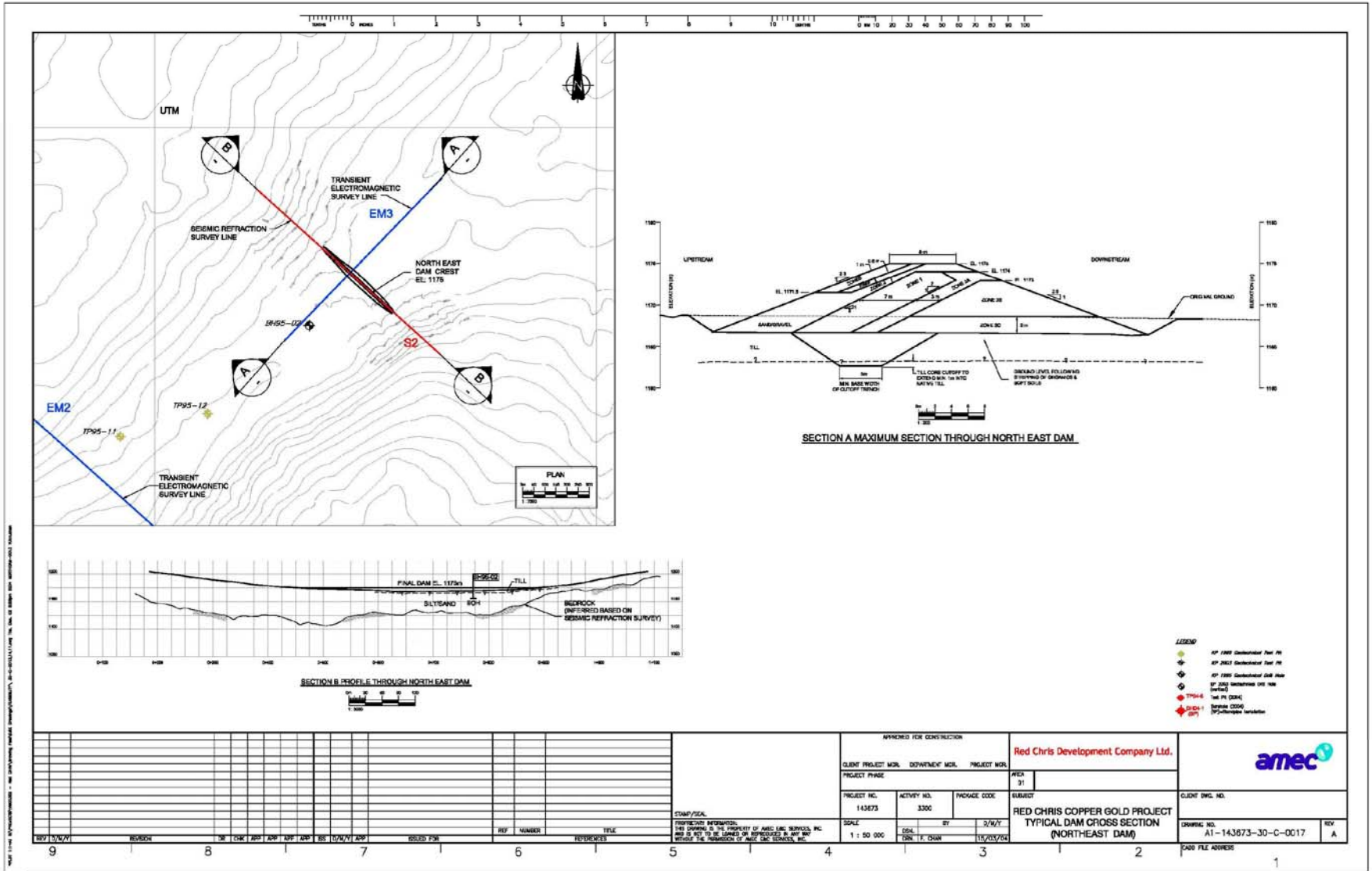


Figure 28 - North Waste Dump Final Configuration (Closure)

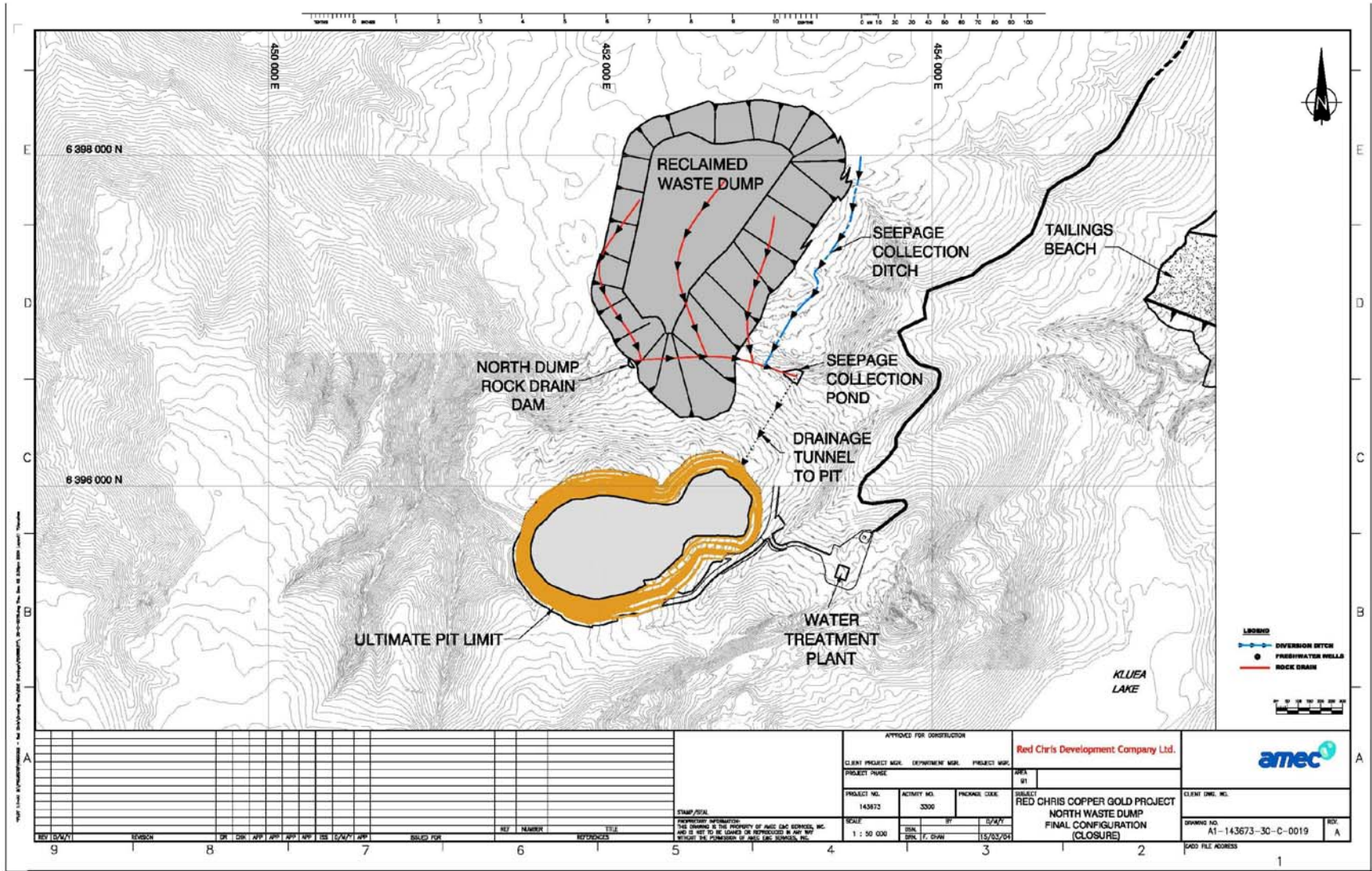
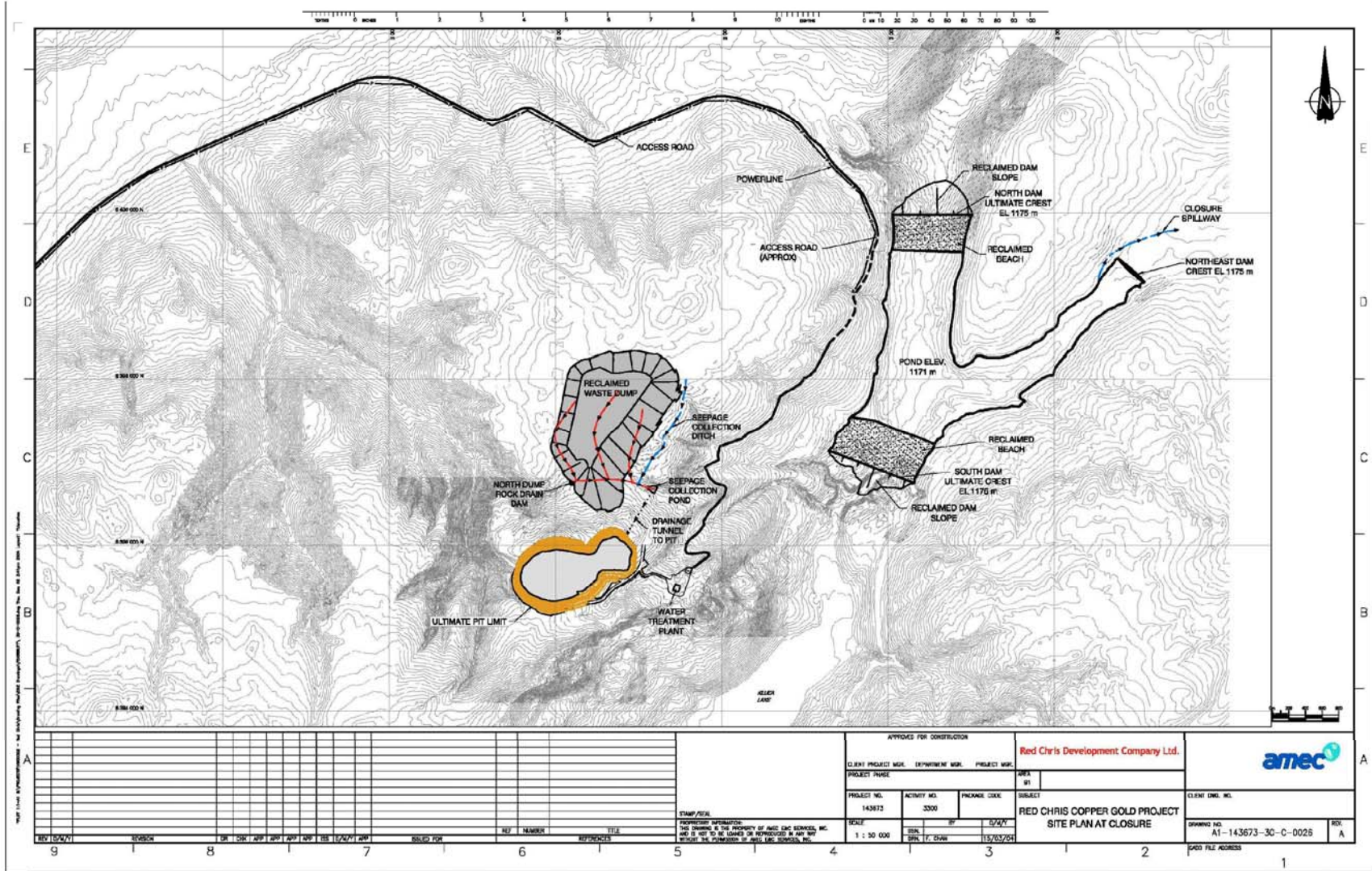


Figure 29 - Site Plan at Closure



18.2 Mining Operations

18.2.1 Introduction

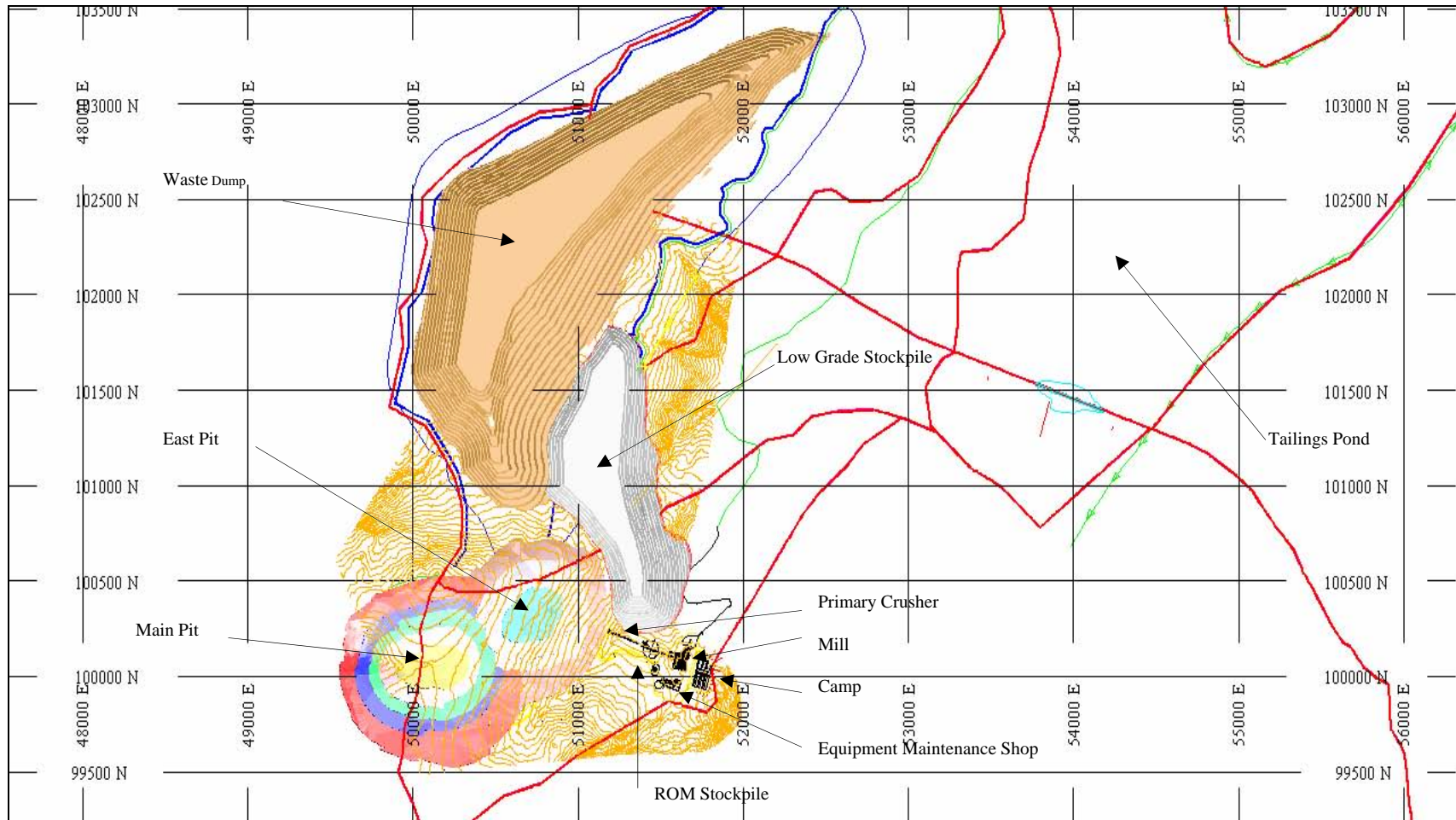
The proposed mining operation is a conventional shovel and truck open pit porphyry copper-gold mine feeding a 30,000 tonne per day processing plant using standard mineral flotation technology. The mine has been scheduled to maximise the production of high grade ore, especially during the first five years, to minimise the capital payback period. This strategy necessitates mining at a higher cut-off grade with a commensurate higher stripping ratio, i.e. waste tonnes to ore tonnes, and the stockpiling of lower grade material for recovery and processing later in the mine life. The average stripping ratio is 2.3:1 during the period of open pit production, varying from 3.0:1 in Year 1, 3.1:1 in Year 5, 2.6:1 in Year 10, 1.4:1 in Year 15 and 0.1:1 in Year 17. For the last eight years of mine life, the mill is fed from material recovered from stockpile. If this material is included in the total ore mined, the Life-of-Mine (“LOM”) stripping ratio is 1.1:1.

The mineralization at Red Chris sub-outcrops and hence the requirement for pre-production stripping is very low compared with that often required at similar open pit porphyry operations. The pre-production period lasts for three months during which approximately 5.0 million tonnes of waste and low grade material will be mined. Low grade material will be stockpiled in a segregated dump for easy recovery for processing at a later date.

The mine will utilize electric rotary blasthole drills, drilling 311mm diameter holes, 38m³ electric shovels loading 230t capacity haul trucks. The operation will be supported by standard ancillary equipment including an 18m³ front-end-loader, track-and rubber-tired-dozers, graders etc. Blastholes will be loaded with heavy ANFO bulk slurry explosive. The utilization of diesel rotary drills and hydraulic excavators is an option that will be considered at the time of equipment purchase.

The general arrangement of the mine area is shown in Figure 30 Mine Development Concept.

Figure 30 - Mine Development Concept



Ore will be hauled to a primary crusher located near the southeast pit rim. Low-grade ore will be hauled to a stockpile located to the north of the primary crusher and east of the pit limits from where it will be reclaimed for processing through the mill when mine resources have been exhausted. This low-grade material will be placed upon a base of non-acid generating (“NAG”) material. Potential Acid Generating (“PAG”) waste will be hauled to the North waste dump (north of the open pit limits) and placed upon a base of NAG waste material. Provision has been made in the design for the NAG layer to be up to 5m thick in areas where water can be expected to accumulate or flow. This dump will be reclaimed at mine closure as discussed in Section 18.5 of this report.

Mining will begin during the preproduction period (year -1) in the East Pit (East Pit Phase I) and Main Pit (Main Pit Phase I). The pit will expand in phased push backs until the ultimate pit limits are reached. In all nine phases will be mined, four in the East Pit and five in the Main Pit. The East and the Main pits eventually join to become one large open pit.

The deposit has been modelled using a 15 m bench height in both the Main Zone and the East Zone. Pit walls have been smoothed to incorporate access ramps, berms and face and inter ramp angle recommendations by design sector as provided by the geotechnical consultants.

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

18.2.2 Capacity Evaluation

Typically for porphyry deposits such as Red Chris, “bigger is better” in so far that unit operating costs decrease with increasing concentrator through-put and the utilization of larger capacity equipment. However along with increased through-put comes a commensurate increase in the capital cost. For Red Chris it was a case of balancing capacity against power requirements and the Company’s ability to raise the required debt financing.

In 1996 a Pre-Feasibility Study was completed by Fluor Daniel Wright Ltd. (“Fluor”). This study contemplated a project capacity of 90,000 tpd milled at a capital cost of \$541 million. This capital included \$88.9 million for a 287kV power line from the main Skeena sub-station near Terrace to the Red Chris mine site, a distance of approximately 390km. The high capital cost, combined with a period of falling metal prices, resulted in the project being shelved.

The Company has adopted an approach that keeps the capital within the ability of the Company to finance while maximising the use of large-scale, productive equipment to keep operating costs at a minimum. In the current Feasibility Study it is assumed that power to the junction of Highway 37 and the mine access road is provided by BC Hydro.

The Company evaluated capacities up to 50,000 tpd before fixing the capacity at the current

process design through-put rate of 30,000 tpd.

18.2.3 Mineral Resources

The Red Chris Project in-situ measured and indicated mineral inventory are based on the block model produced by Giroux Consultants Limited (“Giroux”), as discussed in the section on resource model development. For the purposes of pit optimization and estimation of Proven and Probable Reserves, the mineral inventory has been reported on an NSR cut-off basis for metal prices of \$400/ounce gold and \$1.00/lb copper. Grades were reported for copper, gold, recoverable copper, recoverable gold and recoverable equivalent copper as well as specific gravity.

Inpit resources were defined within pit limits determined using the Lerchs Grossman algorithm, complex wall slopes with input from AMEC, preliminary economic parameters provided by Nilsson Mine Services Ltd (“NMS”) and the geological block model of Giroux.

18.2.4 Geotechnical Evaluation

General

The Red Chris Open Pit encompasses about a 53 hectare area on the crest. The open pit design currently comprises a phased development with two separate pit bottoms located in the western and eastern ends. Pit floor in the eastern section is approximately 1155 m while the western end is deeper, with an expected pit floor at approximately 1125m. The pit highwall is approximately 360 m high in the East Pit and 450 m high in the Main Pit and will be located on the north wall.

Geotechnical Investigations

Boreholes and test pits were used to investigate the geotechnical properties of overburden and bedrock at the Red Chris Project. The following reports have been prepared:

- Knight Piesold Ltd, 2003 Geotechnical Investigations for Open Pit, Waste Dumps and Tailings Storage Facility – Volumes I & II; and
- AMEC, September 2004 Geotechnical Investigations
- Brawner, Spetember 2004, Geotechnical Review

Geotechnical investigations within the pit area were completed by American Bullion, and Knight and Piesold. Geotechnical information was gathered and analyzed from these programs and included data from both exploration drilling and geotechnical drilling. In 2004 AMEC completed four additional geotechnical drillholes into and behind the proposed pit walls in the north and south to confirm information gathered by others as well as identify structural controls that may exist behind the pit walls. These drillholes were geotechnically logged, the core was oriented and an acoustic borehole camera was used to confirm orientations and rock mass conditions in two of the four drillholes.

After review and analysis of the above geotechnical data, the open pit was divided into four geotechnical domains based on structural and lithologic characteristics for design purposes. Kinematic, probabilistic and rock mass stability analysis methods were used to determine pit slope bench face, inter-ramp and overall slope angles for the various domains and wall orientations. To optimize pit design, bcMetals is committed to implementation of a significant wall control and drainage program during pit development to reduce likelihood of damage and failure of the pit walls during development. As a result, an aggressive pit wall slope design was identified. The success of the wall control and drainage programs and subsequent back analysis and data gathering during the internal phase development will be key in the performance of the internal phase slopes and final design stages.

Mine Geology

Geology within the pit walls is expected to comprise primarily of the Red Stock monzodiorite, with intersections of the Bowser Sediments and Dynamite Hill Volcanics in the southern and northern pit walls respectively. Dominant structure consists of steeply dipping shears and faults within the Red Stock that trend predominantly northeast southwest.

Overburden Materials

The waste dumpsite is on a glacially shaped plateau north of the open pit area. The plateau slopes slightly to the south into an east-west valley that separates the dump and stockpile areas from the pit area.

The overburden soils consist for the most part of a thin organic surficial layer underlain by weathered rock. The depth of weathering is shallow – approximately 0.2 to 5 m depth. In the east-west drainage valley there is a sandy silty till with cobble size particles in places.

Wall Slope Design

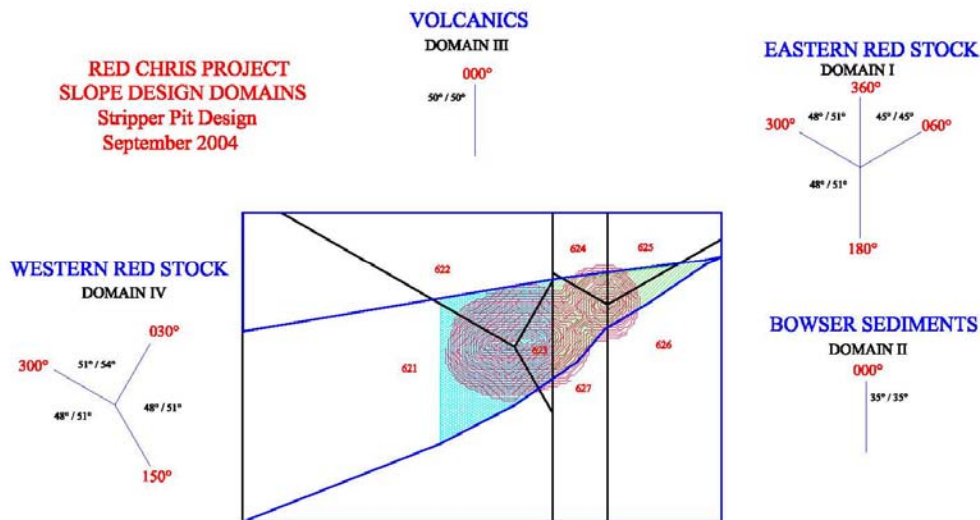
The wall slope design criteria used for the ultimate pit were based upon the assessment provided by AMEC in September 2004. As shown in Table 28 Recommended Wall Slope Angles, ten design sectors were identified within the four geotechnical domains and assigned codes in the block model. These sectors are shown in Figure 31 Slope Design Sectors. The inter-ramp slope angles were reduced by one degree for pit optimization to make allowance for haulage roads. The actual ultimate pit design input parameters were the face angle and overall inter-ramp wall slope by sector. Berm width was allowed to vary according to this geometric relationship in a double 15 m bench configuration. The internal phase pits were designed with a five degree increase in bench face angle and the specified berm width.

In addition to the design criteria shown in the table an allowance was made for a 30 m wide haulage road with 10% grade.

Table 28 - Recommended Wall Slope Angles

Domain	Wall Aspect or Dip Dir	From	To	Final Walls		Internal Walls		Less			1
				Bench Face Angle / Berm Width	Inter Ramp Angle (°)	Bench Face Angle / Berm Width	Inter Ramp Angle (°)	Perimeter	VBM	Zone	
I Eastern Red Stock (From 60400east)	000 - 120°	180	300	60°/10 m	48	65°/10 m	51	627	637	7	47
	120 - 180°	300	360	60°/10 m	48	65°/10 m	51	624	634	4	47
	180 - 240°	0	60	60°/13 m	45	60°/13 m	45	625	635	5	44
	240 - 360°	60	180	60°/10 m	48	65°/10 m	51	626	636	6	47
III Volcanics	000 - 360°	360	0	65°/11 m	50	65°/11 m	50	628	638	8	49
IV Western Red Stock (From 50400N west)	330 - 120°	150	300	60°/10 m	48	65°/10 m	51	621	631	1	47
	120 - 210°	300	30	65°/10 m	51	65°/8 m	54	622	632	2	50
	210 - 330°	30	150	60°/10 m	48	65°/10 m	51	623	633	3	47
II Bowser Sediments and Faulted Contacts	000 - 360°	360	0	60°/13 m	35	60°/13 m	35	629	639	9	34
Brawner and AMEC Within 50 m of surface				70°/9 m	46			630	640	10	45

Figure 31- Slope Design Sectors



18.2.5 Open Pit Optimization

Optimization of the open pit for the Red Chris Project was completed using the computer block model provided by Giroux and the Lerchs-Grossman, graph theory based, algorithm for pit limit definition. This program was used in conjunction with Medsystem Minesight© software. This mine planning system has been used extensively within the mining industry for deposits of similar origin and size and is generally accepted as an appropriate tool for assessing potential ultimate pit size and configuration for a given set of economic parameters.

The following key parameters were used to establish guidelines for the final pit limit using the Lerchs-Grossman based algorithm:

- Complex Wall Slope Sectors adjusted for previously located ramp locations;
- Metal prices;
- Metal recoveries;
- Operating costs;

- Concentrate transportation charges;
- Smelter terms;
- Royalties; and
- Sustaining capital cost allowance.

The metallurgical recovery and equivalence basis for the mine planning model are shown in Table 29 Recovery and Copper Equivalence Calculation. The recoverable copper and gold grades and the recoverable copper equivalent grade are carried on an individual block basis in the model. The net smelter return calculation is shown in Table 30 Net Smelter Return. The net smelter return value of each block has been calculated and is carried in the block model as a “Net Minegate Revenue” value. In other words, all off site costs associated with the production of copper and gold have been accounted and a royalty of 1.8% included. A typical bench plan showing the distribution of net smelter return values is shown in Figure 32 NSR Distribution Bench 1350.

Table 29- Recovery and Copper Equivalence Calculation

METHODOLOGY FOR UPDATED MINESIGHT MODEL

Metal Prices		Recoveries	
Cu	1.00 \$/lb	Cu	100.0 %
Au	400.00 \$/oz	Au	100.0 %

Raw CuEq	Cu +	Au *	0.583
Recov. CuEq	RCu *	1.000	+ RAu * 0.583

bcMetals filenote - May 3, 2004 M.A. Redfearn

Main Zone Copper Recovery	89.0%	Equation used in Minesight Model $5.0221 \times \ln(\text{Cu}) + 92.746$
East Zone Copper Recovery	88.0%	Equation used in Minesight Model $9.7 \times \text{Cu} + 80.224$
Gold Recovery Equation	$53.93 \times \text{feed grade} + 28.18$	

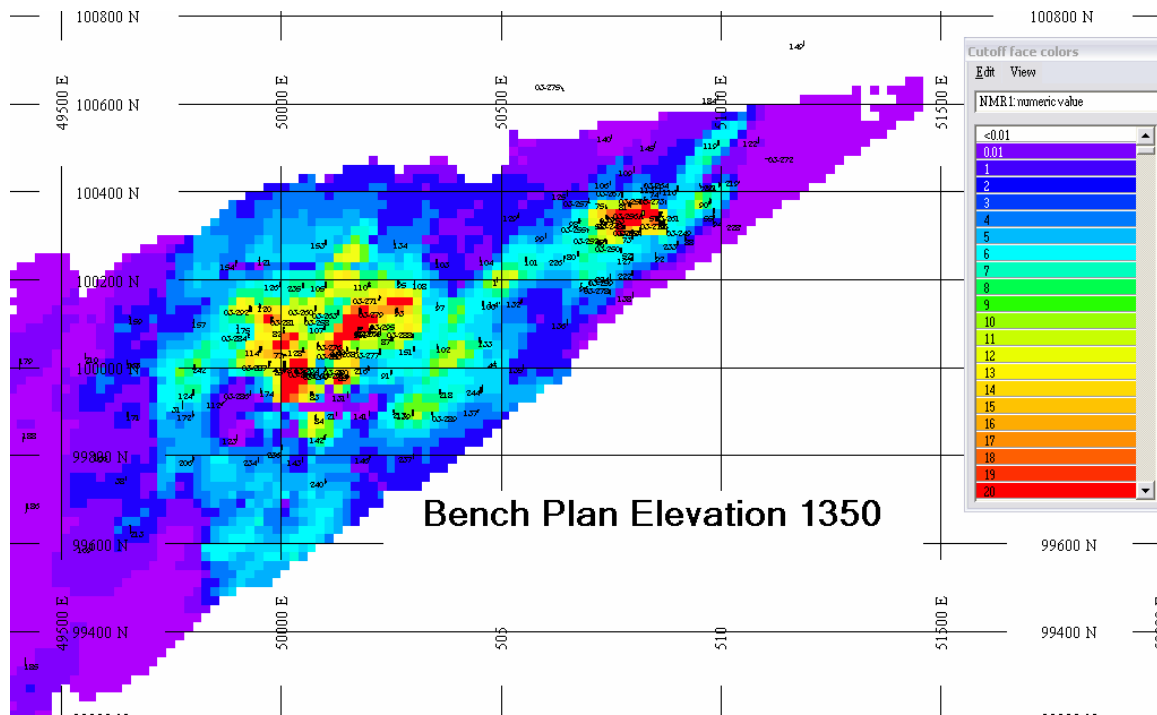
Note: Gold recovery needs to be capped at 90.20 Both Zones

Note: Copper also needs capping at 93.45 Main Zone
91.38 East Zone

Table 30 - Net Smelter Return

Test Block NSR Calculation		Parameters and Test Block Calculation		27,500 tpd
Copper Head Grade	%			0.480
Gold Head Grade	g/t			0.440
Recoverable Copper Head Grade	%			0.43
Recoverable Gold Head Grade	g/t			0.23
Metallurgical Recovery				
Copper Recovery		Capped at	93.45	Equation used in Minesight Model 5.0221xln(Cu)+92.746
		Capped at	91.38	Equation used in Minesight Model 9.7xCu+80.224
	Main Zone		ln()	0.480 Cu x 5.022 + 92.746 = 89.06
	East Zone			0.480 Cu x 9.7 + 80.224 = 84.88
Gold Recovery				
		Au g/t	90.20	Main & East Zone Gold Recovery
	Recovery	Capped at		0.440 Au x 53.93 + 28.18 = 51.91
Metal Pricing				
Copper Price	US\$/lb			\$1.00
Gold Price	US\$/ounce			\$400.00
USD:CDN Exchange				\$0.72
Concentrate				
Copper Concentrate Grade				27.00
Gold Concentrate Grade	g/dmt	Calculated from head grade, recovery and concentrate production		
Silver Concentrate Grade	g/dmt			-
Moisture Content	%			8.0%
Contained Copper	lb/dmt			595.08
Contained Gold	g/dmt			14.43
Payable Copper	lb/dmt			573.04
Payable Gold	g/dmt			13.99
Concentrate - Recovery Based	dmt/t ore			0.01583
Gross Value of Concentrate after Deductions				
Gross Value Concentrate	C\$/dmt			\$1,045.83
Concentrate Handling				
Truck Haul & Stevedoring FOB Stewart	C\$/wmt			\$44.48
Ocean Freight	US\$/wmt			\$28.00
Umpiring & Sampling	C\$/wmt			\$0.20
Marketing	C\$/wmt			\$3.00
Insurance	C\$/wmt	0.15%		\$1.71
Losses	C\$/wmt	0.25%		\$2.61
Total Concentrate Handling	C\$/wmt			\$90.89
Total	C\$/dmt			\$98.79
Treatment and Refining				
Deduction for Copper	unit	96.3%		1.00
Treatment Charges	US\$/dmt			\$55.00
Gold Payment	%			97.0%
Copper Refining Cost	US\$/payable lb			\$0.055
Gold Refining Cost	US\$/payable oz			\$5.50
Total Treatment and Refining	C\$/dmt			\$79.83
Net Smelter Return				
Net Smelter Return	C\$/dmt			\$867.21
	C\$/payable lb Cu			\$1.51
NSR Before Royalty	NSR C\$/t			\$13.73
Royalty	CDN\$/t	1.8%		\$0.25
NSR After Royalty	NSR C\$/t			\$13.48
NSR for Copper	NSR C\$/t			\$9.77
NSR for Gold	NSR C\$/t			\$3.71
Factor for Copper	\$/% Recoverable			\$22.87
Factor for Gold	\$/g Recoverable			\$16.24

Figure 32 - NSR Distribution Bench 1350



The operating cost input parameters for the pit optimization are shown in Table 31 Pit Optimization Parameters. The pit limit was determined using an operating cost basis for a 27,500 tpd mill throughput rate. As the Feasibility Study was nearing completion it became clear that the plant design will be capable of sustaining a higher throughput rate and final schedules were prepared for 30,000 tpd. Changes were also made in mine equipment and operating philosophy whereby blasthole sizes and patterns were increased and shovel sizes were also increased and electrified. This resulted in improved operating costs relative to optimization input parameters.

Table 31 - 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			
Block Value			
Block Value Discounted by Annual Advance Rate			
Zone Item			
Azimuth Sectors for Optimization			
Slope Code Values for Stripper Design			
Class Item			
Measured, Indicated & Inferred as single Integer Code CLASO - original			
Consolidated Measured and Indicated to a single Integer Code CLASR - reported			
Measured and Indicated block values calculated and compressed for dipper pits - LG			
Mining Recovery			
		100%	
Mining Dilution			
Diluted to mineable block size			
Operating Costs			
		27,500 tpd Case	
Base Mine Operating Costs		Ore	Waste
Mine General	\$/t mined	\$0.160	\$0.160
Drilling	\$/t mined	\$0.050	\$0.050
Blasting	\$/t mined	\$0.155	\$0.155
Loading	\$/t mined	\$0.160	\$0.160
Hauling	\$/t mined	\$0.260	\$0.280
Roads & Dumps	\$/t mined	\$0.140	\$0.140
Subtotal	\$/t mined	\$0.925	\$0.945
Sustaining Capital	\$/t mined	\$0.130	\$0.130
Total Base Mining	\$/t mined	\$1.055	\$1.075
Average Mining Cost		\$1.150	\$1.150
Processing Operating Costs			
Mill Operating	\$/t milled	\$2.850	
General & Administration	\$/t milled	\$0.820	
Subtotal	\$/t milled	\$3.670	
Sustaining Capital	\$/t milled	\$0.130	
Total Processing	\$/t milled	\$3.800	
Processing and Mining		\$4.855	
Entrance Bench		1485	1485
Cost increment	\$/t/bench	\$0.022	\$0.022
Pit Slope Angles			
Wall Slopes for Optimization	As per Table 25.2.1 Recommended Wall Slope Angles		

In large open pit mines it often takes many years to reach the bottom of a final development phase. Typical sinking rates in these large mines may average up to 6 benches annually. This means that in a deep pit where the ore is found at depth, waste stripping on upper level benches may take place many years ahead of the ore release at depth. For the purposes of pit limit definition at Red Chris, pit limits were evaluated for both discounted and undiscounted cost and NSR values. A discount rate of 12% was used to discount cost and revenue in 6 bench increments. The discount factor detail is shown in Table 32 Discounting & Bench Operating Cost Increment. The discounted pit limit is shown in the isometric drawing in Figure 33 Lerch Grossman Discounted Pit Limit.

Table 32 - Discounting & Bench Operating Cost Increment

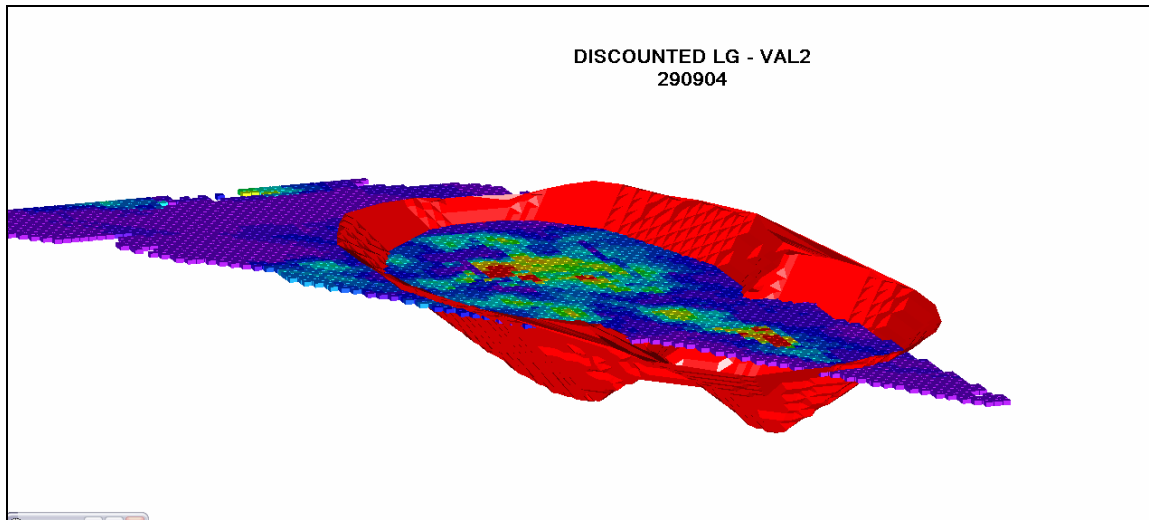
FACTORS

Discount Rate % 12.0

PERIOD	FACTOR	From	To		Undiscounted			Discounted		
					\$/t Ore	\$/t Waste	Feed	\$/t Ore	\$/t Waste	Feed
1	0.893	10	16	Step A	1.055	1.075	4.855	0.942	0.960	4.335
2	0.797	17	23	Step B	1.187	1.207	4.987	0.946	0.962	3.975
3	0.712	24	30	Step C	1.319	1.339	5.119	0.939	0.953	3.644
4	0.636	31	37	Step D	1.451	1.471	5.251	0.922	0.935	3.337
5	0.567	38	44	Step E	1.583	1.603	5.383	0.898	0.910	3.054
6	0.507	45	51	Step F	1.715	1.735	5.515	0.869	0.879	2.794
7	0.452	52	55	Step G	1.847	1.867	5.647	0.835	0.845	2.554
8	0.404									
9	0.361									
10	0.322									

	Pit	Model
Top	1575	1725
Bottom	1125	
Depth	450	
Benchs	15	
Benchs	30	
Years	5	
Annual Advance	6	

Figure 33 - Lerchs Grossman Discounted Pit Limit



The undiscounted and discounted pit limits are shown in the cross sections below in the Main Zone Pit area and the East Zone Pit area. It is clear that the high grade deep resources of the East Zone are more sensitive to the effects of discounting.

Figure 34 - Lerchs Grossman Pit Limits Main Zone Section 50,160 East

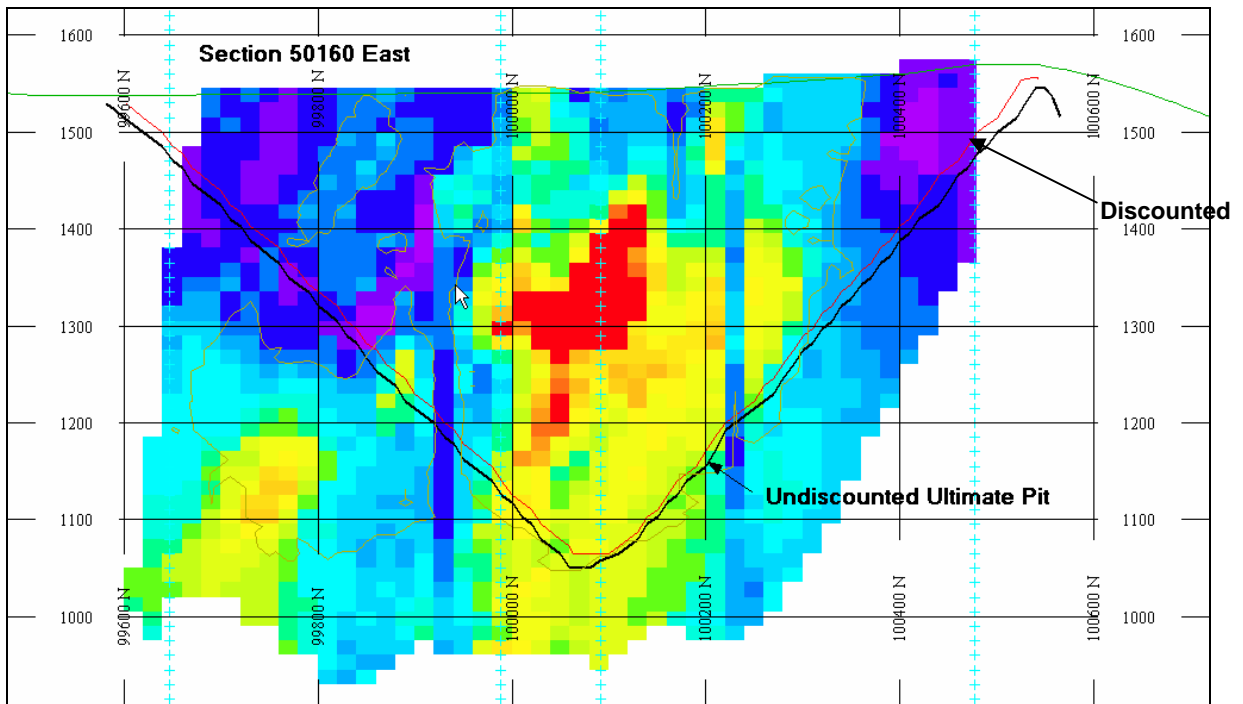
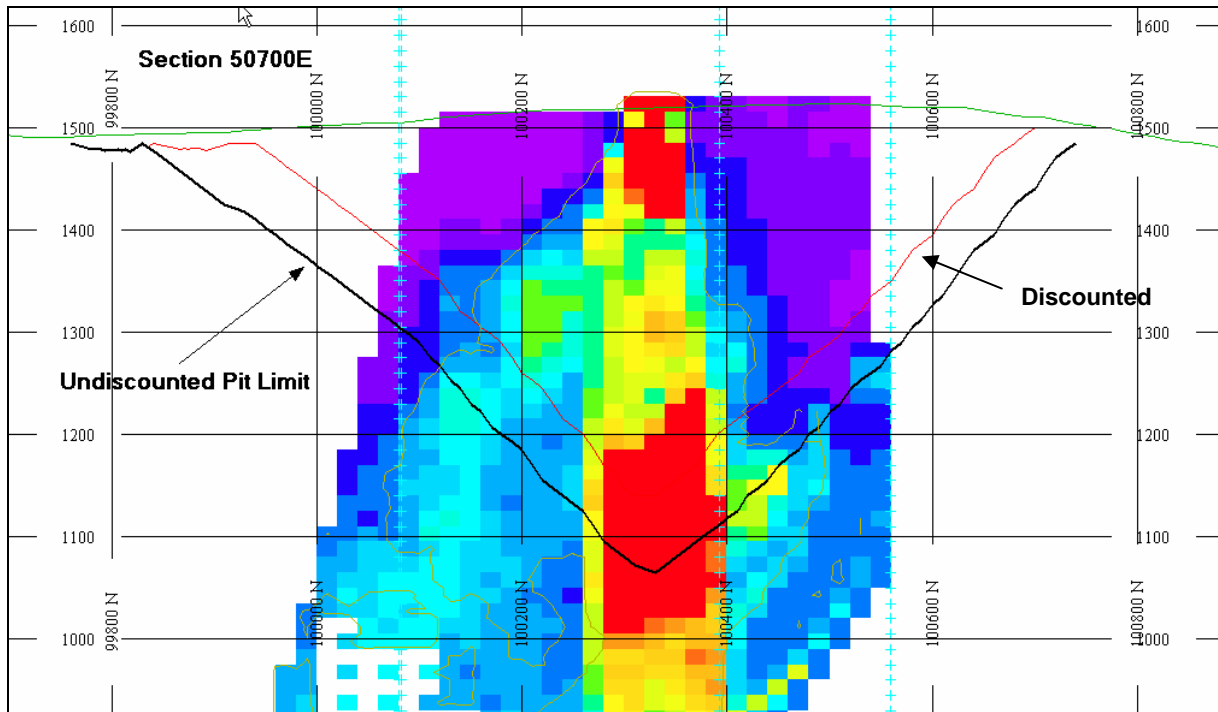


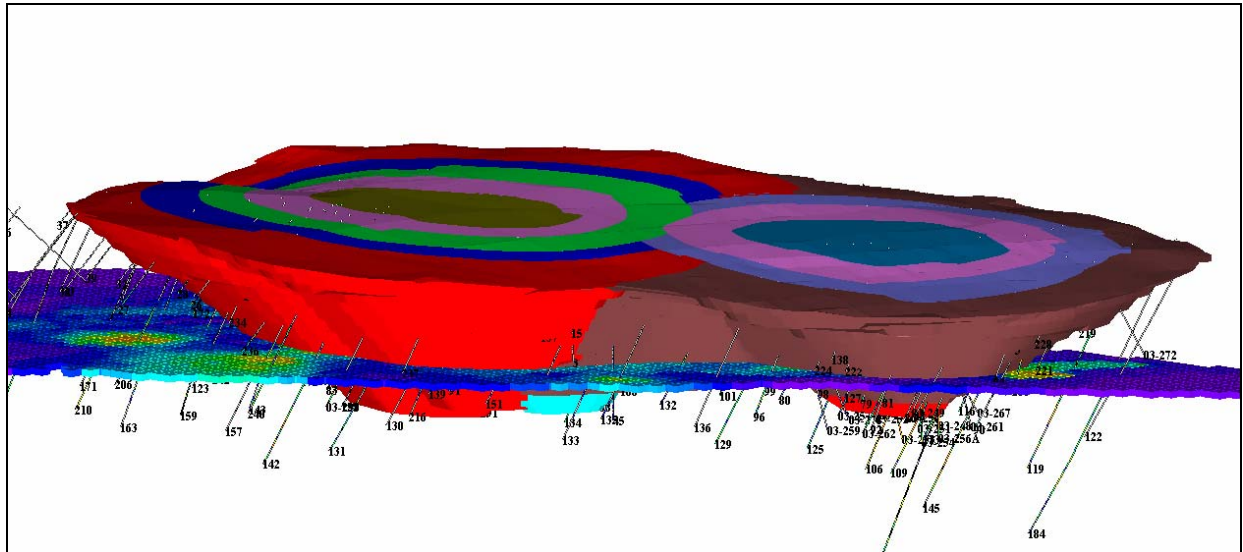
Figure 35 - Lerchs Grossman Pit Limits East Zone Section 50,700 East



18.2.6 Mine Design and Ore Reserves

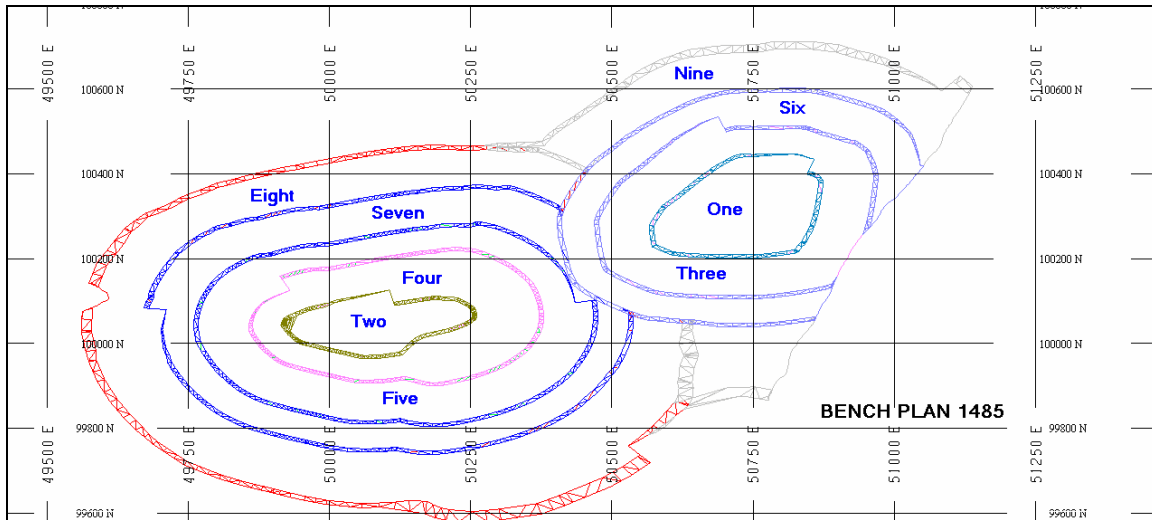
The open pit design was based upon measured and indicated resources and the economic parameters and design criteria listed in Section 18.2.5. Internal pit phases were defined on the basis of increasing ore value and geometric consideration for future expansions with a minimum mining width of 80 m. Solids models of the various phases are shown in Figure 36 Mine Development Phase Solids

Figure 36 - Mine Development Phase Solids



The mining sequence was established with the objective of mining the best grade ore material first at elevated cutoff grades while maintaining a relatively constant overall production rate. The pit optimization process identified the central areas of the East and West Zones as the most profitable and accessible for mining. For reasons of geometry and phase volume the East Zone will be mined in three phases and the Main Zone in three phases. The general sequence of phase development is shown in Figure 37 below.

Figure 37 - Mine Development Phases



The ore reserves are summarized in Table 33 Ore Reserve Summary.

Table 33 - Ore Reserve Summary

Pit Bottom Scavenging - 1003EP4S1											
	COG	ORE	MINE	NSR	CU	AU	RCU	RAU	RCUEO	SG	
	\$/t NSR	bcm x 1000	kt	\$t	%	gt	%	gt	%	tm3	
Proven	>= 8.00	239.0	672.0	31.05	0.910	0.946	0.814	0.766	1.261	2.62	
Probable	>= 7.00	20.0	55.0	13.59	0.464	0.455	0.397	0.266	0.552	2.80	
Total	>= 7.00	259.0	727.0	29.72	0.876	0.909	0.783	0.728	1.207	2.82	
Pit Bottom Scavenging - 1003EP4S2											
	COG	ORE	MINE	NSR	CU	AU	RCU	RAU	RCUEO	SG	
	\$/t NSR	bcm x 1000	kt	\$t	%	gt	%	gt	%	tm3	
Proven	>= 7.50	64.0	179.0	11.24	0.391	0.404	0.345	0.207	0.465	2.79	
Probable	>= 4.50	456.0	1,274.0	9.68	0.351	0.343	0.307	0.163	0.402	2.80	
Total	>= 4.50	520.0	1,453.0	9.87	0.356	0.351	0.312	0.169	0.410	2.80	
Pit Bottom Scavenging - Total											
	COG	ORE	MINE	NSR	CU	AU	RCU	RAU	RCUEO	SG	
	\$/t NSR	bcm x 1000	kt	\$t	%	gt	%	gt	%	tm3	
Proven	>= variable	303.0	851.0	28.89	0.801	0.832	0.716	0.649	1.093	2.81	
Probable	>= variable	478.0	1,329.0	9.83	0.355	0.349	0.311	0.168	0.409	2.80	
Total	>= variable	778.0	2,180.0	16.49	0.529	0.537	0.469	0.355	0.676	2.80	
Grand Total											
	COG	ORE	MINE	NSR	CU	AU	RCU	RAU	RCUEO	SG	
	\$/t NSR	bcm x 1000	kt	\$t	%	gt	%	gt	%	tm3	
Proven	>= 3.75	36,912.0	103,402.0	11.40	0.418	0.327	0.368	0.184	0.475	2.80	
Probable	>= 3.75	62,731.0	175,130.0	7.72	0.305	0.228	0.265	0.103	0.325	2.79	
Total	>= 3.75	99,643.0	278,528.0	9.09	0.347	0.265	0.303	0.133	0.380	2.80	
East Zone Phase I											
	COG	ORE	MINE	NSR	CU	AU	RCU	RAU	RCUEO	SG	
	\$/t NSR	bcm x 1000	kt	\$t	%	gt	%	gt	%	tm3	
Proven	>= 3.75	939.0	2,610.0	25.20	0.839	0.656	0.750	0.495	1.039	2.78	
Probable	>= 3.75	429.0	1,188.0	7.50	0.293	0.244	0.246	0.116	0.314	2.78	
Total	>= 3.75	1,367.0	3,798.0	19.66	0.668	0.527	0.593	0.376	0.812	2.78	
East Zone Phase II											
	COG	ORE	MINE	NSR	CU	AU	RCU	RAU	RCUEO	SG	
	\$/t NSR	bcm x 1000	kt	\$t	%	gt	%	gt	%	tm3	
Proven	>= 3.75	2,825.0	7,846.0	12.53	0.457	0.368	0.396	0.214	0.521	2.78	
Probable	>= 3.75	1,645.0	4,564.0	6.55	0.257	0.236	0.213	0.103	0.273	2.77	
Total	>= 3.75	4,470.0	12,409.0	10.33	0.383	0.319	0.328	0.173	0.430	2.78	
East Zone Phase III											
	COG	ORE	MINE	NSR	CU	AU	RCU	RAU	RCUEO	SG	
	\$/t NSR	bcm x 1000	kt	\$t	%	gt	%	gt	%	tm3	
Proven	>= 3.75	4,288.0	11,836.0	12.37	0.425	0.405	0.396	0.245	0.509	2.77	
Probable	>= 3.75	3,231.0	8,922.0	6.53	0.251	0.245	0.208	0.109	0.272	2.76	
Total	>= 3.75	7,499.0	20,757.0	9.86	0.350	0.337	0.298	0.187	0.407	2.77	
East Zone Phase IV											
	COG	ORE	MINE	NSR	CU	AU	RCU	RAU	RCUEO	SG	
	\$/t NSR	bcm x 1000	kt	\$t	%	gt	%	gt	%	tm3	
Proven	>= 3.75	2,876.0	8,028.0	13.01	0.435	0.441	0.375	0.273	0.534	2.79	
Probable	>= 3.75	5,748.0	15,982.0	6.90	0.266	0.257	0.222	0.112	0.288	2.78	
Total	>= 3.75	8,624.0	24,010.0	8.94	0.323	0.319	0.273	0.166	0.370	2.78	
Total East Zone											
	COG	ORE	MINE	NSR	CU	AU	RCU	RAU	RCUEO	SG	
	\$/t NSR	bcm x 1000	kt	\$t	%	gt	%	gt	%	tm3	
Proven	>= 3.75	10,908.0	30,320.0	13.68	0.471	0.427	0.409	0.266	0.564	2.78	
Probable	>= 3.75	11,052.0	30,656.0	6.78	0.261	0.250	0.218	0.110	0.282	2.77	
Total	>= 3.75	21,960.0	60,974.0	10.20	0.366	0.338	0.313	0.188	0.422	2.78	
Main Zone Phase I											
	COG	ORE	MINE	NSR	CU	AU	RCU	RAU	RCUEO	SG	
	\$/t NSR	bcm x 1000	kt	\$t	%	gt	%	gt	%	tm3	
Proven	>= 3.75	1,367.0	3,827.0	8.17	0.343	0.191	0.301	0.079	0.347	2.80	
Probable	>= 3.75	1,242.0	3,467.0	6.83	0.299	0.144	0.280	0.054	0.292	2.79	
Total	>= 3.75	2,609.0	7,294.0	7.54	0.322	0.169	0.282	0.067	0.321	2.80	
Main Zone Phase II											
	COG	ORE	MINE	NSR	CU	AU	RCU	RAU	RCUEO	SG	
	\$/t NSR	bcm x 1000	kt	\$t	%	gt	%	gt	%	tm3	
Proven	>= 3.75	4,585.0	12,893.0	9.30	0.375	0.221	0.331	0.107	0.393	2.81	
Probable	>= 3.75	3,524.0	9,858.0	7.64	0.318	0.183	0.278	0.079	0.324	2.80	
Total	>= 3.75	8,109.0	22,750.0	8.58	0.350	0.205	0.308	0.095	0.363	2.81	
Main Zone Phase III											
	COG	ORE	MINE	NSR	CU	AU	RCU	RAU	RCUEO	SG	
	\$/t NSR	bcm x 1000	kt	\$t	%	gt	%	gt	%	tm3	
Proven	>= 3.75	7,516.0	21,141.0	11.57	0.432	0.307	0.385	0.170	0.484	2.81	
Probable	>= 3.75	8,625.0	24,113.0	8.61	0.339	0.226	0.298	0.110	0.363	2.80	
Total	>= 3.75	16,141.0	45,254.0	9.99	0.383	0.264	0.339	0.138	0.419	2.80	
Main Zone Phase IV											
	COG	ORE	MINE	NSR	CU	AU	RCU	RAU	RCUEO	SG	
	\$/t NSR	bcm x 1000	kt	\$t	%	gt	%	gt	%	tm3	
Proven	>= 3.75	8,049.0	22,665.0	10.73	0.401	0.311	0.355	0.161	0.449	2.82	
Probable	>= 3.75	15,445.0	43,253.0	8.11	0.319	0.236	0.279	0.107	0.341	2.80	
Total	>= 3.75	23,495.0	65,918.0	9.01	0.347	0.262	0.305	0.126	0.378	2.81	
Main Zone Phase V											
	COG	ORE	MINE	NSR	CU	AU	RCU	RAU	RCUEO	SG	
	\$/t NSR	bcm x 1000	kt	\$t	%	gt	%	gt	%	tm3	
Proven	>= 3.75	4,184.0	11,705.0	8.71	0.336	0.257	0.293	0.124	0.365	2.80	
Probable	>= 3.75	22,387.0	62,454.0	7.60	0.302	0.223	0.262	0.098	0.330	2.79	
Total	>= 3.75	26,571.0	74,159.0	7.77	0.307	0.228	0.267	0.102	0.327	2.79	
Total Main Zone											
	COG	ORE	MINE	NSR	CU	AU	RCU	RAU	RCUEO	SG	
	\$/t NSR	bcm x 1000	kt	\$t	%	gt	%	gt	%	tm3	
Proven	>= 3.75	25,701.0	72,231.0	10.26	0.392	0.279	0.346	0.144	0.430	2.81	
Probable	>= 3.75	51,203.0	143,145.0	7.91	0.314	0.222	0.274	0.101	0.333	2.80	
Total	>= 3.75	76,905.0	215,375.0	8.69	0.340	0.241	0.298	0.115	0.366	2.80	

Table 34 - Ore Reserve Summary

	Tonnes	Cu %	Au g/t	R-Cu	R-Au	R-CuEq	NMR
Proven	93,475,785	0.423	0.327	0.374	0.185	0.482	11.554
Probable	182,524,215	0.300	0.226	0.261	0.100	0.320	7.600
Total	276,000,000	0.349	0.266	0.299	0.129	0.374	8.939

The ore reserves have been reported at a \$3.75/t net smelter return cutoff. The selection of this cutoff level was based upon a pit rim decision for recovery of processing and general & administration onsite operating costs. Low grade quantities have been segregated in the production schedule. Material between \$3.25/t net smelter return and \$3.75/t was designated as low grade one (LG1) and separated for future processing if economic conditions improve beyond the levels assumed for this study. Material with a net smelter return value above \$3.75/t but less than the applied cutoff for a particular pit phase was designate low grade two (LG2). The LG2 material was scheduled to the East Stockpile area and reclaimed for processing in Year 17 through Year 25.

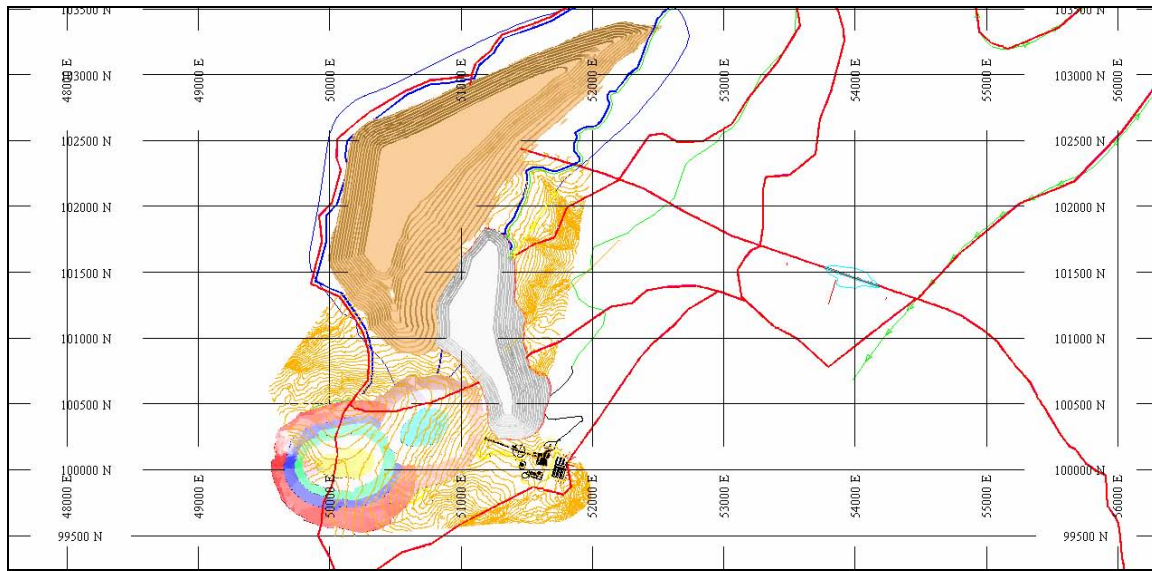
Dilution

The block model for the resource estimate was developed using 20 m x 20 m x 15 whole blocks. Where post mineral dikes could be identified they were coded into the block model and excluded from the resource estimate. The mining equipment to be employed at Red Chris is intended for a bulk mining method. Grade control will be implemented to direct the various material types to their intended destinations. It has been assumed that the resource model contains adequate inherent dilution to a selective mining unit (SMU) size of 20 m x 20 m x 15 m and that the equipment employed to drill and load this unit size has been included in the mine plan.

Development Concept

The East Zone and Main Zone pit development concept is presented in Figure 38 Mine Development Concept. The plant will be located southeast of the open pit mine area. The low grade stockpile will be located east of the mine area immediately north of the crusher. The waste dump and lower grade stockpile material will be placed on the plateau area north of the open pit within the drainage area boundary marked in red above the tailings impoundment.

Figure 38 - Mine Development Concept



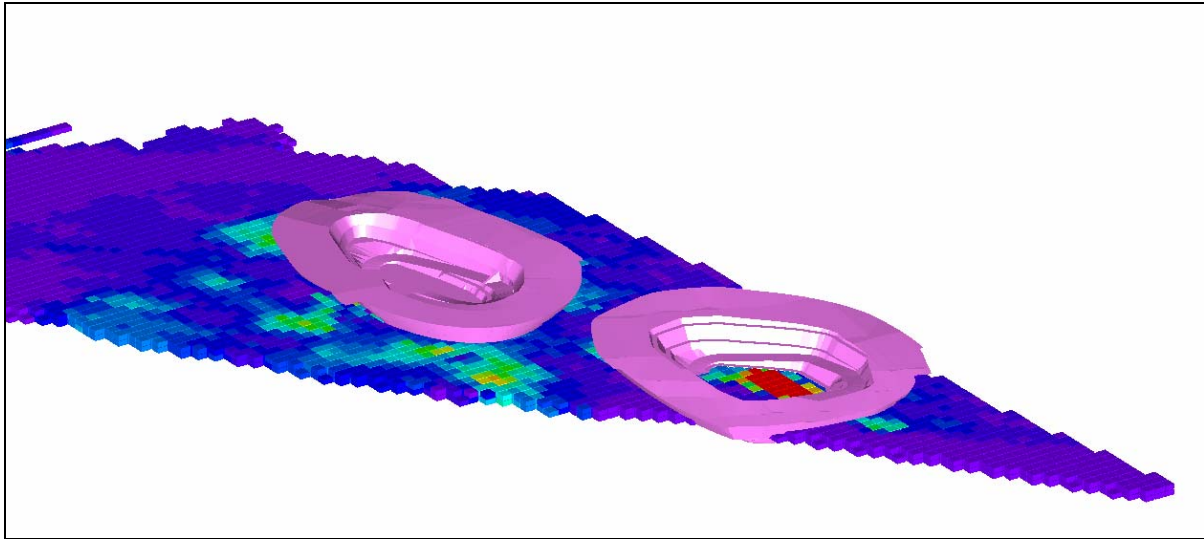
East Zone Phase I

The East Zone Phase I Pit was designed to provide in the order of 3.8 Mt of mill feed resources. This development stage also provides non-reactive (“NAG”) waste required for road and dam construction purposes in the Tailings Management facility (“TMF”). This pit will be 75 m deep and will incorporate a counterclockwise ramp that exits to the south side of the pit. This pit will be mined to completion in Year 1 of the mine plan.

East Zone Phase II

The East Zone Phase II Pit contains total of 9.4 Mt of mill feed resources at an NSR cutoff of \$5.00/t and 3.0 Mt of low-grade that will be moved to stockpile. This pit will be 180 m deep. The pit will be mined from Year 1 through Year 3.

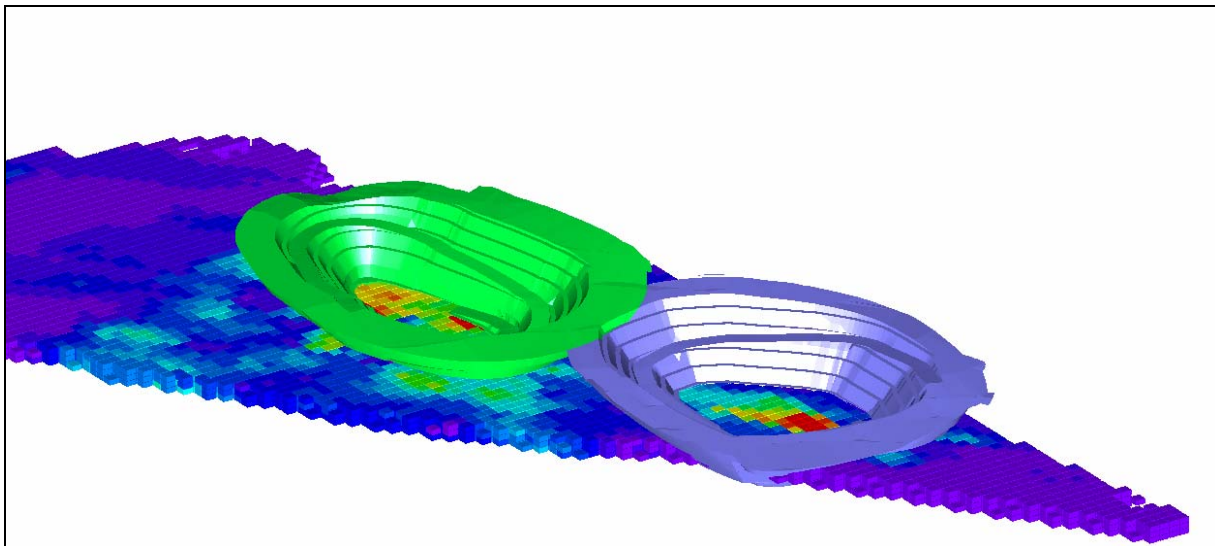
Figure 39 - East & Main Zone Phase II Development



East Zone Phase III

The East Zone Phase III Pit contains total of 13.2 Mt of mill feed resources at an NSR cutoff of \$6.00/t and 7.5 Mt of low-grade that will be moved to stockpile. This pit will be 285 m deep. The pit will be mined from Year 3 through Year 6.

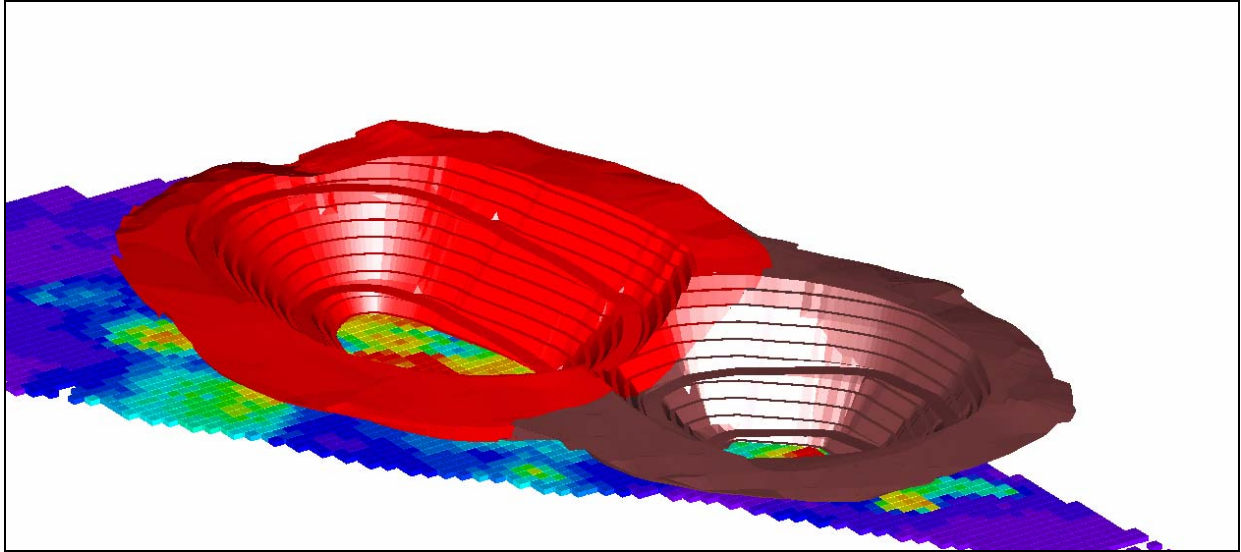
Figure 40 - East & Main Zone Phase III Development



East Zone Phase IV

The East Zone Phase IV Pit contains total of 23.4 Mt of mill feed resources at an NSR cutoff of \$4.50/t and 2.8 Mt of low-grade that will be moved to stockpile. This pit will be 360 m deep. The pit will be final phase, mined from Year 11 through Year 17.

Figure 41 - East Phase IV & Main Zone Phase V Development



Main Zone Phase I

The Main Zone Phase I Pit contains a total of 2.1 Mt of mill feed resources at an NSR cutoff of \$8.50/t and 5.2 Mt of low-grade material that will be moved to stockpile. The pit will be 75 m deep and the ramp will be directed clockwise downward with a switchback and will exit to the south side of the pit. This pit will be completed in Year 1.

Main Zone Phase II

The Main Zone Phase II Pit will be developed to provide 11.0 Mt of mill feed at an NSR cutoff of \$8.50/t and 11.8 Mt of low-grade material that will be moved to stockpile. A clockwise oriented ramp has been placed parallel to the ramp in the Phase I pit to simplify cross bench connection in the radially expanding pits of the Main Zone. The pit will be 180 m deep. It will be mined during Year 1 through Year 3.

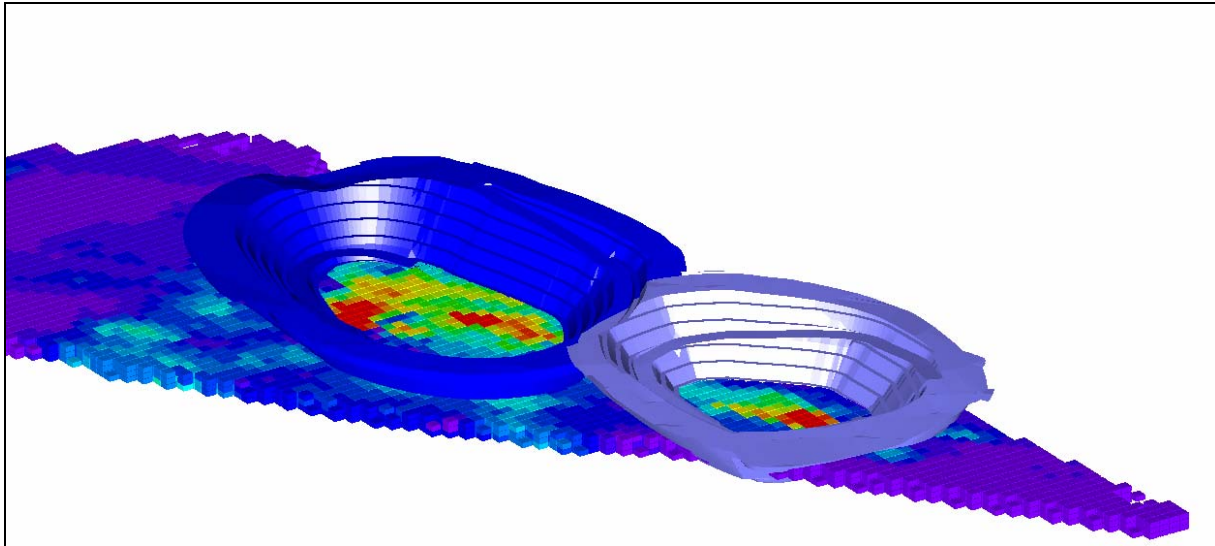
Main Zone Phase III

The Main Zone Phase III Pit will be developed to provide 24.3 Mt of mill feed at an NSR cutoff of \$7.00/t and 21.0 Mt of low-grade material that will be moved to stockpile. The pit will be 270 m deep. It will be mined during Year 2 through Year 6.

Main Zone Phase IV

The Main Zone Phase IV Pit will be developed to provide 36.6 Mt of mill feed at an NSR cutoff of \$7.00/t and 29.3 Mt of low-grade material that will be moved to stockpile. The pit will be 360 m deep. It will be mined during Year 5 through Year 10.

Figure 42 - East & Main Zone Phase IV Development



Main Zone Phase V

The Main Zone Phase V Pit will be developed to provide 61.6 Mt of mill feed at an NSR cutoff of \$4.50/t and 12.60 Mt of low-grade material that will be moved to stockpile. The pit will be 465 m deep. It will be mined during Year 6 through Year 15.

18.2.7 Mining Method

The mining method proposed for the Red Chris Project is conventional open pit shovel and truck operation using electric and/or diesel powered equipment. As described in the Section above, mining will commence simultaneously in the starter pit areas of the East Zone and Main Zone. Expansion pits will then be developed in four more phases in the Main Zone and three phases in the East Zone, for a total of nine phases. Ore will be hauled by 230t trucks to the primary crusher. Waste and low-grade ore will be hauled to dumps and temporary stockpiles as required.

Drilling & Blasting

The drilling requirements are dictated by a combination of logistics, wall control and production blasting considerations. Wall control blasting techniques will be used to maximise bench face angles and minimise stripping.

The main components of the proposed blast design are perimeter blasting and main production

blasting. Wall control buffer holes and production holes will be 311 mm in diameter. Small diameter drill holes, 127mm, will be used for pre-split blasting for pit wall control.

The main production drill hole spacing is dictated by the powder factor, bench height, sub-grade drilled and collar requirements. Good fragmentation enhances overall mine efficiency in terms of material handling and equipment availability. Using 311 mm holes, a powder factor of 0.202 kg/t, an explosives column height of 10.5 m, a collar of 6.0 m and 1.5 m sub-grade, the resultant drill equivalent spacing on a square pattern is 11 m. This equates to 5,082 t per hole blasted. Heavy ANFO slurry, density of 1.29 gm/cm³, will be used exclusively, eliminating the problem of handling wet and dry holes differently if ANFO prills were used.

A buffer row of reduced spacing and loaded length (lower powder factor) will be drilled between the production blast and final wall design. The buffer row drilling requirements are based upon the pit perimeter lengths of rock exposed on each bench in each of the phase pushbacks.

The recommended primary drilling unit is a 311 mm electric powered rotary blasthole drill capable of single pass drilling to 16.5 m with a bit loading of at least 50,000 kg. Two production drills will be required in the mine. An additional down-the-hole hammer drill, drilling 127mm vertical holes to 30m, will be used for wall control blasting.

The blasting crew will require a rubber-tired skid steer loader for blasthole stemming. An explosives supplier will deliver explosives to the boreholes, while the Red Chris blasting crew will have responsibility for priming, calculation of the quantity of slurry, stemming and blast initiation.

Grade Control

Grade control will primarily be based upon copper grade to ensure contractual commitments are met. However gold grades are also considered in determining whether a block of material goes to the crusher or the low grade stockpile. Blasthole samples will be taken by trained drillers supervised by grade control engineers and geologists.

The blasthole patterns will be determined by the blasting engineer and transmitted wirelessly to a control station on the drill. An on-board GPS locates the drill over each blast hole. Drill hole assays of the cuttings will be entered into the mineral resource database to determine the copper and gold grades for each 20m x 20m x 15m block prior to mining. The shovel operators and shift foremen will be provided with up to date digging plans, via live video monitors to ensure proper grade control to meet mill head grade requirements.

Loading and Haulage

The production rate during the first seven years of production will average approximately 120,000 tpd of material moved. This will decrease to approximately 30,000 tpd in Year 17 when the open pit is mined out and mining efforts focus on stockpile recovery.

Ore, low-grade material and waste will be loaded in 230 t capacity trucks by two 38 m³ electric

cable shovels. An 18 m³ front-end-loader will provide back up support to the electric shovels.

Waste Disposal

A total of 307.7 Mt of waste, 90.6 Mt of recoverable low-grade and 31.5 Mt of unrecoverable low-grade material will be placed in waste dumps and stockpiles. PAG and NAG classifications have been established and used to estimate the global quantities of each material type. NAG material will be used to provide “pads” upon which PAG waste and low-grade will be placed. These underliner pads will serve to provide an interface between the surface and the PAG material through which groundwater may travel. Emphasis will be on placing the NAG in drainage courses underlying the dumps and stockpiles.

Waste and low-grade material will be placed in 10 m lifts to provide an opportunity to achieve higher compaction. Berms will be left on the perimeter of each lift effectively flattening the slopes and simplifying the reclamation of the final dump surface. As recommended by O’Kane Consultants, the North Waste Dump will be sloped at 4:1 in the north, east and south directions and 2:1 in the westerly direction. The top of the dump will be sloped to the southwest and surface drainage will be directed to a constructed valley leading to the existing drainage from this area. A small till dam will be placed at the west end of the underlying saddle area beneath the dump directing drainage to the east. Any water flowing thorough the dump will be directed to the open pit.

The East Stockpile will be constructed with 2:1 slopes to reduce the overall footprint of the stockpile. The stockpile will be reclaimed.

18.2.8 Production Schedule

Summary

The mine will operate 7 days per week on a 12 hour 14 on 14 off - 28 day cycle typically found in remote British Columbia mines.

The production schedule over life of mine has been prepared with the reserves tabled by bench and by phase. The mining of the open pit phases will be overlapped to smooth the scheduled stripping and resultant equipment and manpower requirements. The mining plan will follow the following sequence; East Pit Phase I, Main Zone Phase I, East Pit Phase II, Main Pit Phases II and III, East Pit Phase III, Main Pit Phases IV and V, and East Pit Phase IV. The production schedule is summarized in Table 35 Production Schedule Summary. The annual material movement schedule is shown in Figure 43 Material Movement and the annual mill feed grades are shown in Figure 44 Head Grades.

Table 35 - Production Schedule Summary

TABLE 10.2.8 PRODUCTION SCHEDULE SUMMARY 30000 gpd MILLING

Project Year		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	TOTAL			
TRUPPER PIT (MGP) Main	Ex 1000																													
	minutes																													
Ore	Ex 1000	381.0	4,576.0																								3,337.0			
	minutes	48.0	1,178.0																								1,483			
LBI	Ex 1000		4,478.0																								800.0			
	minutes		57.0																								102.0			
LOB	Ex 1000		4,304.0																								6,586.0			
	minutes		55.0																								845.0			
WASTE	Ex 1000		1,800.0																								10,034.0			
	minutes		22.0																								1,281.0			
TRUPPER PIT (MGP) East	Ex 1000																													
	minutes																													
Ore	Ex 1000	1,790.0	6,307.0	8,941.0	7,864.0	260.0																					34,290.0			
	minutes	226.0	803.0	1,139.0	986.0	32.0																						4,285.0		
LBI	Ex 1000		1,176.0	2,273.0	475.0	17.0																					3,840.0			
	minutes		14.0	28.0	5.0	2.0																					48.0			
LOB	Ex 1000		8,099.0	7,763.0	4,873.0	648.0	1.0																				20,801.0			
	minutes		103.0	97.0	61.0	8.0	0.0																					261.0		
WASTE	Ex 1000		6,873.0	6,207.0	1,900.0	23.0																					10,099.0			
	minutes		86.0	77.0	23.0	2.0																						126.0		
TRUPPER PIT (MGP) West	Ex 1000																													
	minutes																													
Ore	Ex 1000					993.0	3,276.0	10,307.0	10,791.0	9,718.0	2,091.0																36,027.0			
	minutes					126.0	414.0	1,289.0	1,348.0	1,214.0	263.0																	4,513.0		
LBI	Ex 1000					860.0	2,667.0	1,066.0	33.0	297.0																	6,204.0			
	minutes					108.0	333.0	133.0	4.0	37.0																	780.0			
LOB	Ex 1000					2,257.0	5,237.0	2,863.0	1,533.0	188.0																	20,067.0			
	minutes					282.0	655.0	358.0	192.0	23.0																		2,517.0		
WASTE	Ex 1000					18,519.0	7,899.0	869.0	184.0																		20,040.0			
	minutes					2,326.0	987.0	107.0	23.0																			2,517.0		
PEN PIT SUMMARY	Ex 1000																													
	minutes																													
Ore	Ex 1000	454.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	100,563.0		
	minutes	58.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	13,763.0		
LBI	Ex 1000	1,810.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	2,240.0	20,400.0	
	minutes	226.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0	280.0
LOB	Ex 1000	2,980.0	11,267.0	10,960.0	11,060.0	8,061.0	7,817.0	11,060.0	12,664.0	12,664.0	8,877.0	2,964.0	1,861.0	1,796.0	874.0	842.0	1,860.0	1,860.0	1,860.0	1,860.0	1,860.0	1,860.0	1,860.0	1,860.0	1,860.0	1,860.0	1,860.0	1,860.0	1,860.0	20,400.0
	minutes	372.0	1,409.0	1,370.0	944.0	1,014.0	977.0	1,383.0	1,583.0	1,583.0	1,109.0	370.0	233.0	222.0	110.0	105.0	233.0	233.0	233.0	233.0	233.0	233.0	233.0	233.0	233.0	233.0	233.0	233.0	233.0	233.0
WASTE	Ex 1000	8,260.0	10,814.0	10,960.0	10,960.0	20,252.0	32,724.0	10,267.0	10,267.0	10,267.0	24,200.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	10,960.0	20,400.0	
	minutes	104.0	1,363.0	1,370.0	1,370.0	2,531.0	4,090.0	1,283.0	1,283.0	1,283.0	3,025.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	1,395.0	233.0	
STOCKPILE RECOVERY	Ex 1000																													
	minutes																													
Ore	Ex 1000																													
	minutes																													
LBI	Ex 1000																													
	minutes																													
LOB	Ex 1000																													
	minutes																													
WASTE	Ex 1000																													
	minutes																													
TOTAL MATERIAL MOVEMENT	Ex 1000																													
	minutes																													
TOTAL STOCKPILE RECOVERY	Ex 1000																													
	minutes																													
TOTAL MATERIAL MOVEMENT	Ex 1000		12,813.0	43,281.0	44,389.0	43,737.0	43,426.0	44,747.0	44,447.0	44,747.0	43,840.0	36,448.0	35,961.0	35,790.0	35,290.0	35,390.0	35,390.0	35,390.0	35,390.0	35,390.0	35,390.0	35,390.0	35,390.0	35,390.0	35,390.0	35,390.0	35,390.0	35,390.0	35,390.0	
	minutes		1,631.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	5,233.0	

Figure 43 - Material Movement

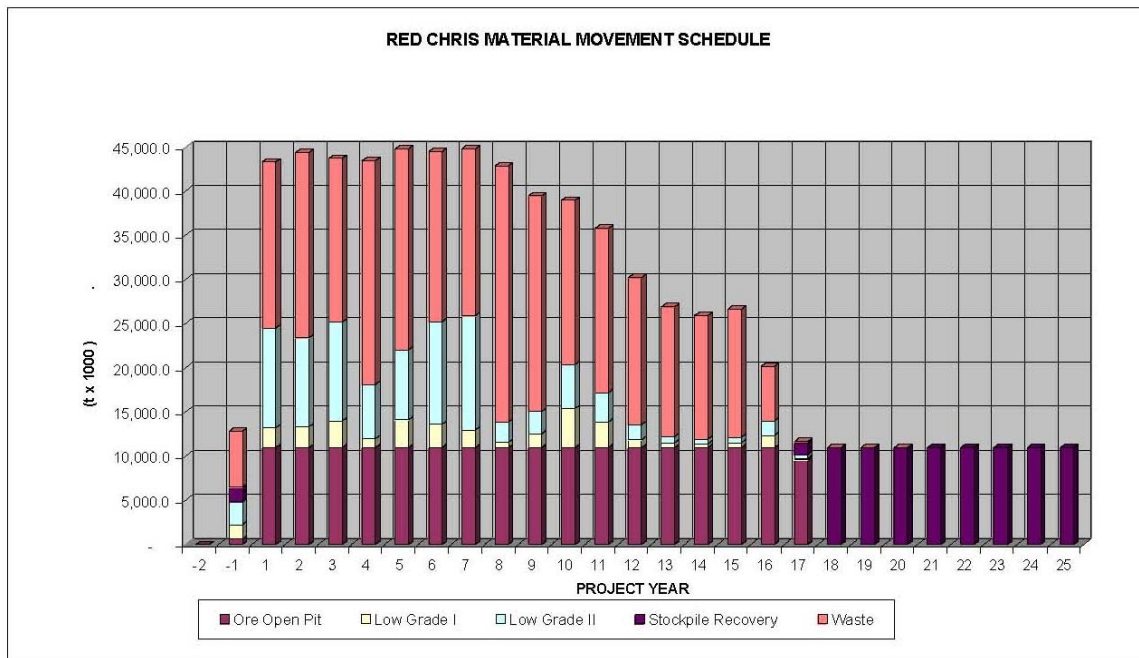
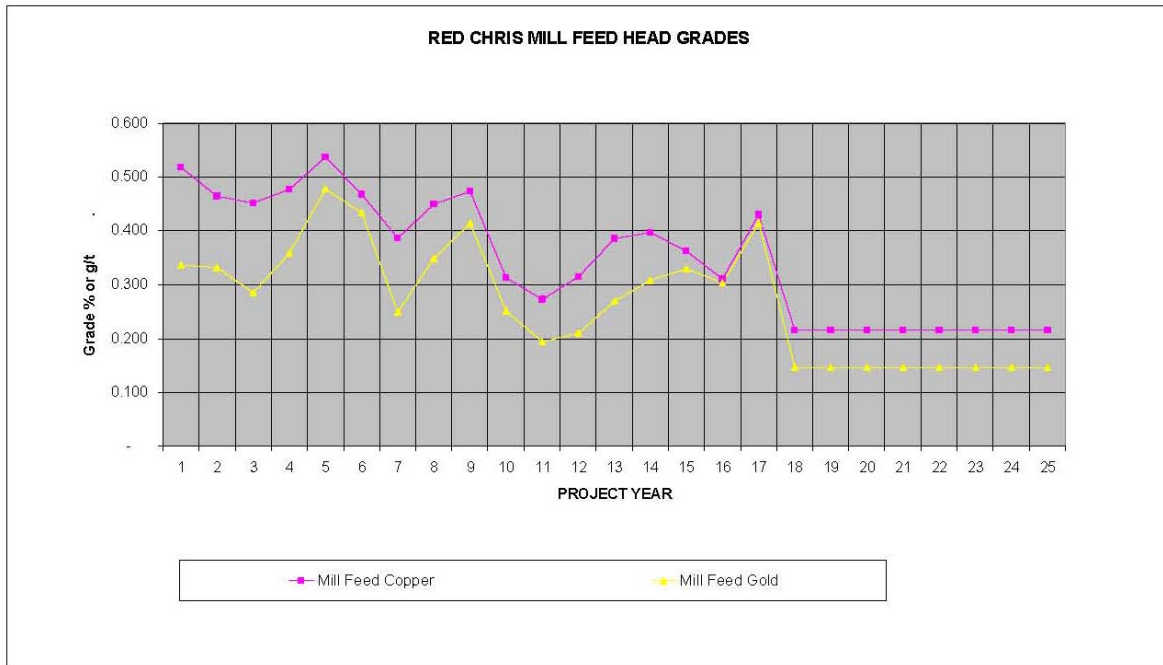


Figure 44 - Head Grades



The mine development sequence described in the sections below is presented in a series of plans Figures 45 through 50 which show the progression of the various phase advances throughout the mine life.

Figure 45 - Mine Development End of Year 1

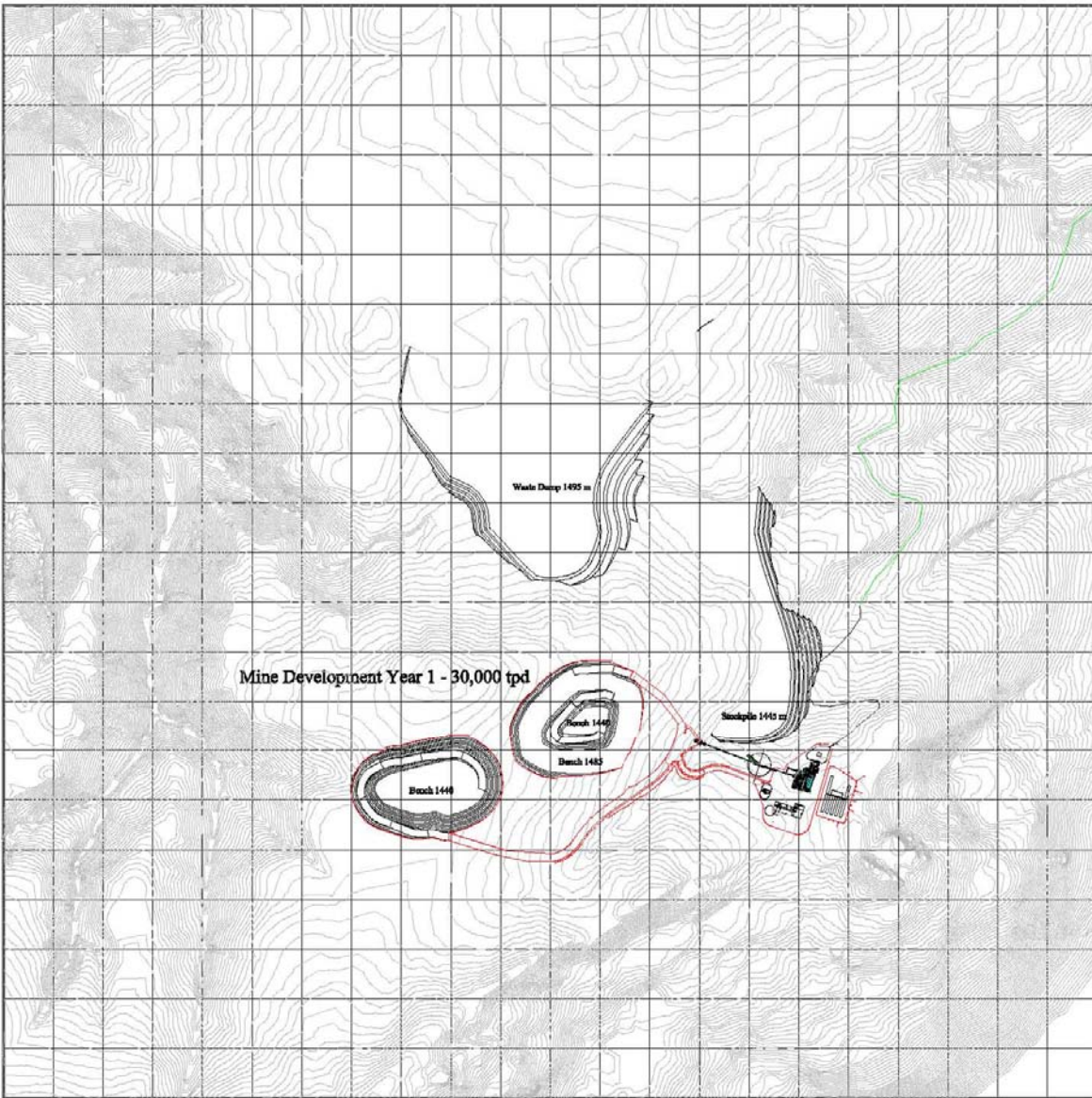


Figure 46 - Mine Development End of Year 3

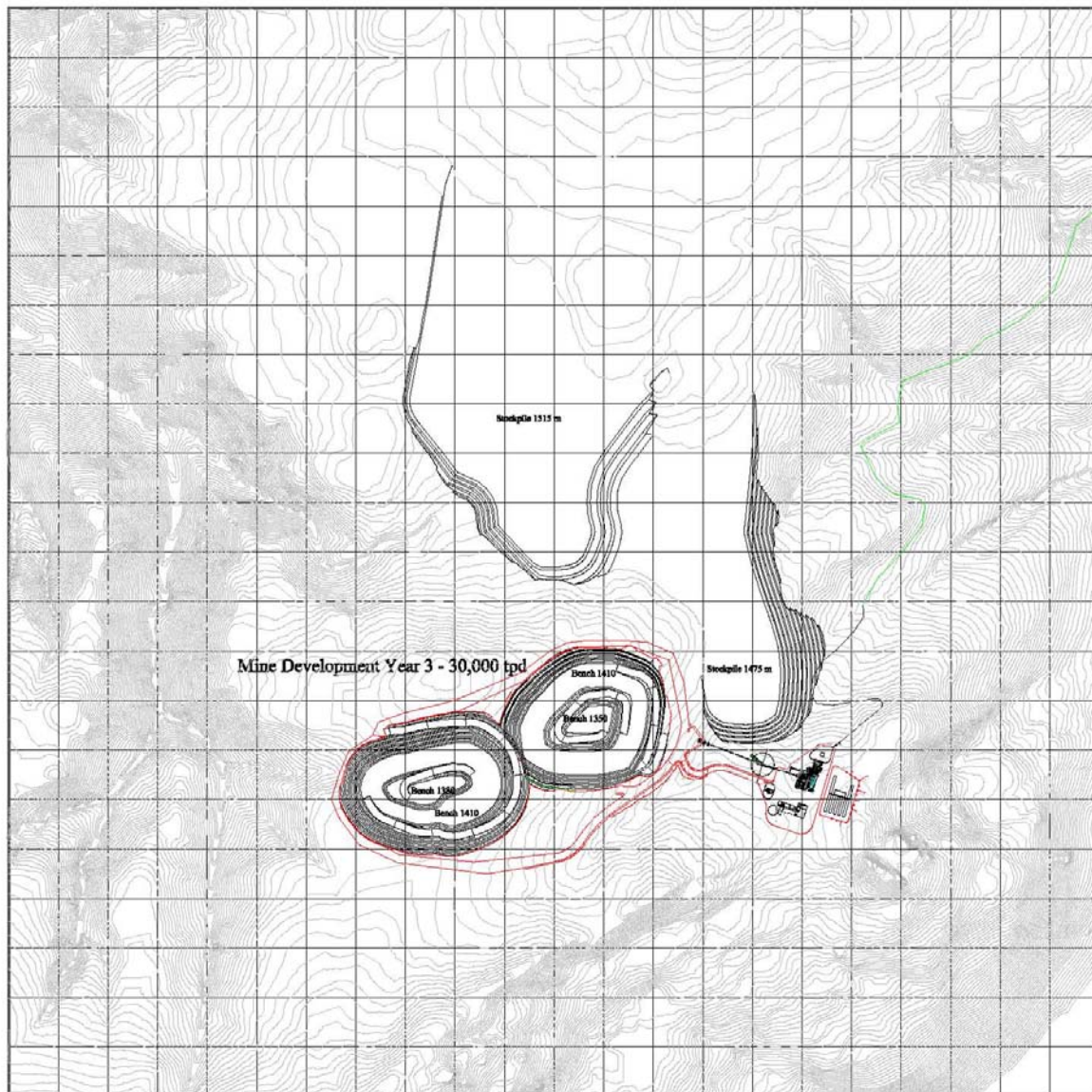


Figure 47 - Mine Development End of Year 6

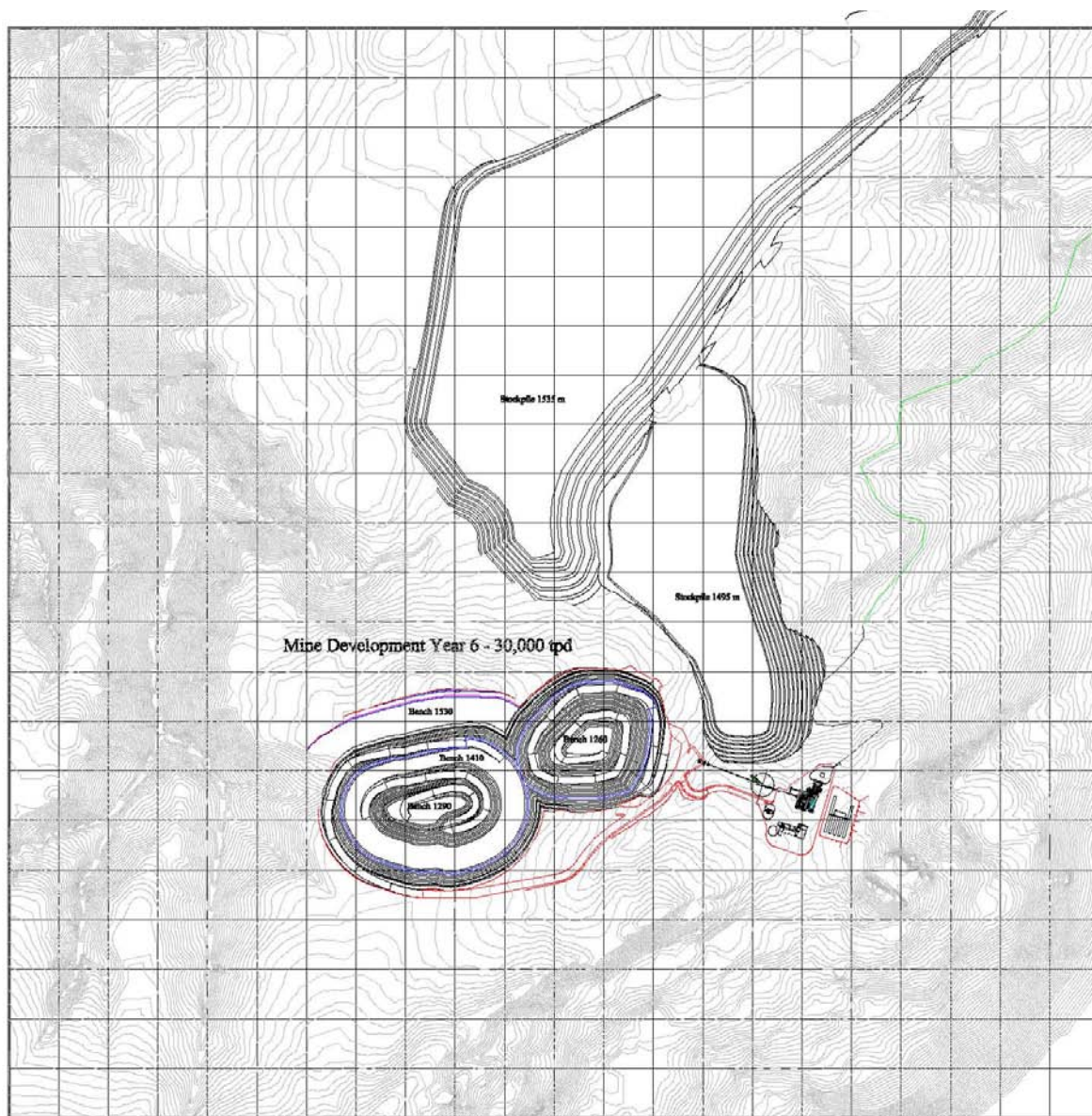


Figure 48 - Mine Development Plan End of Year 10

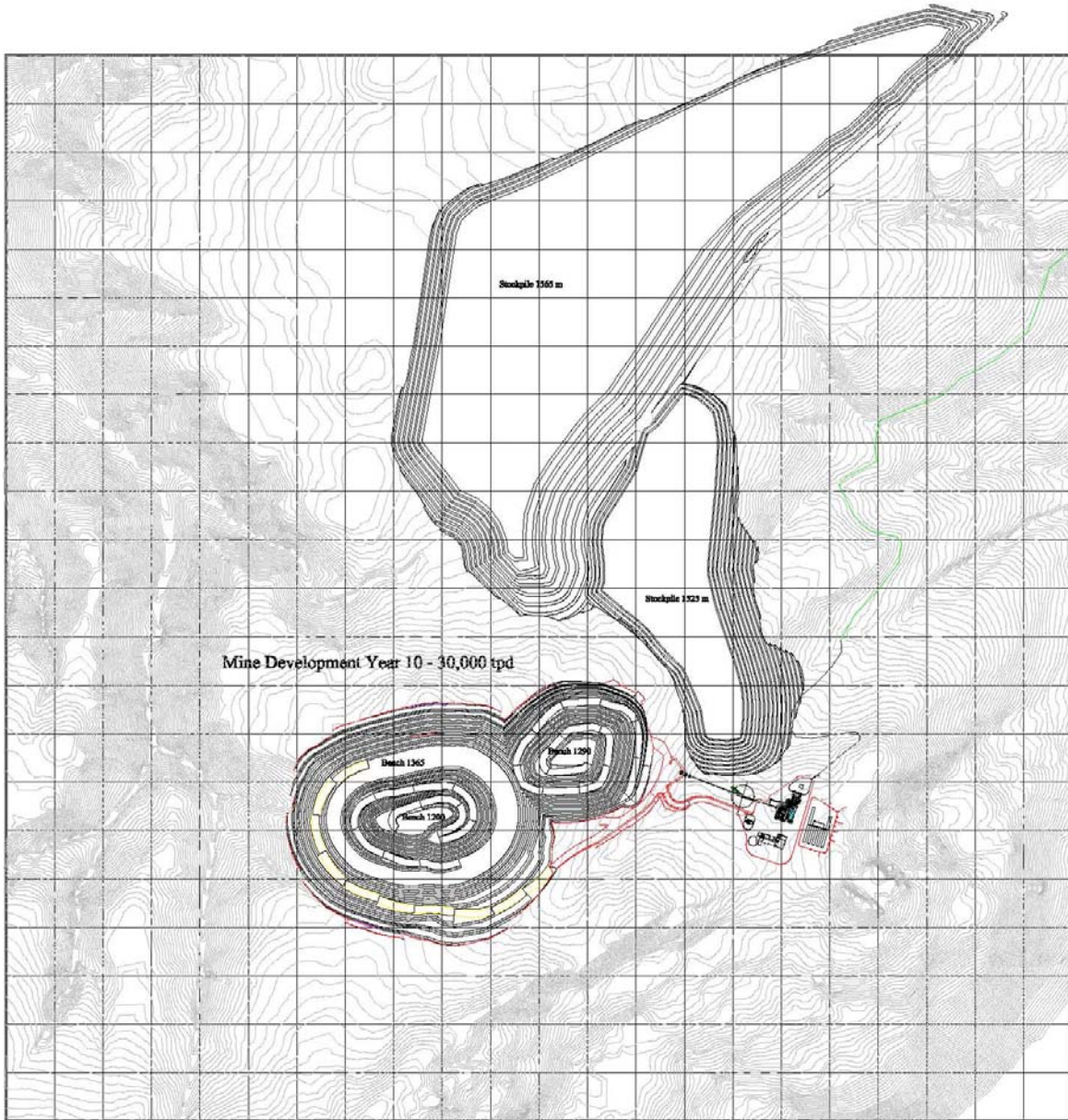


Figure 49 - Mine Development Plan End of Year 14

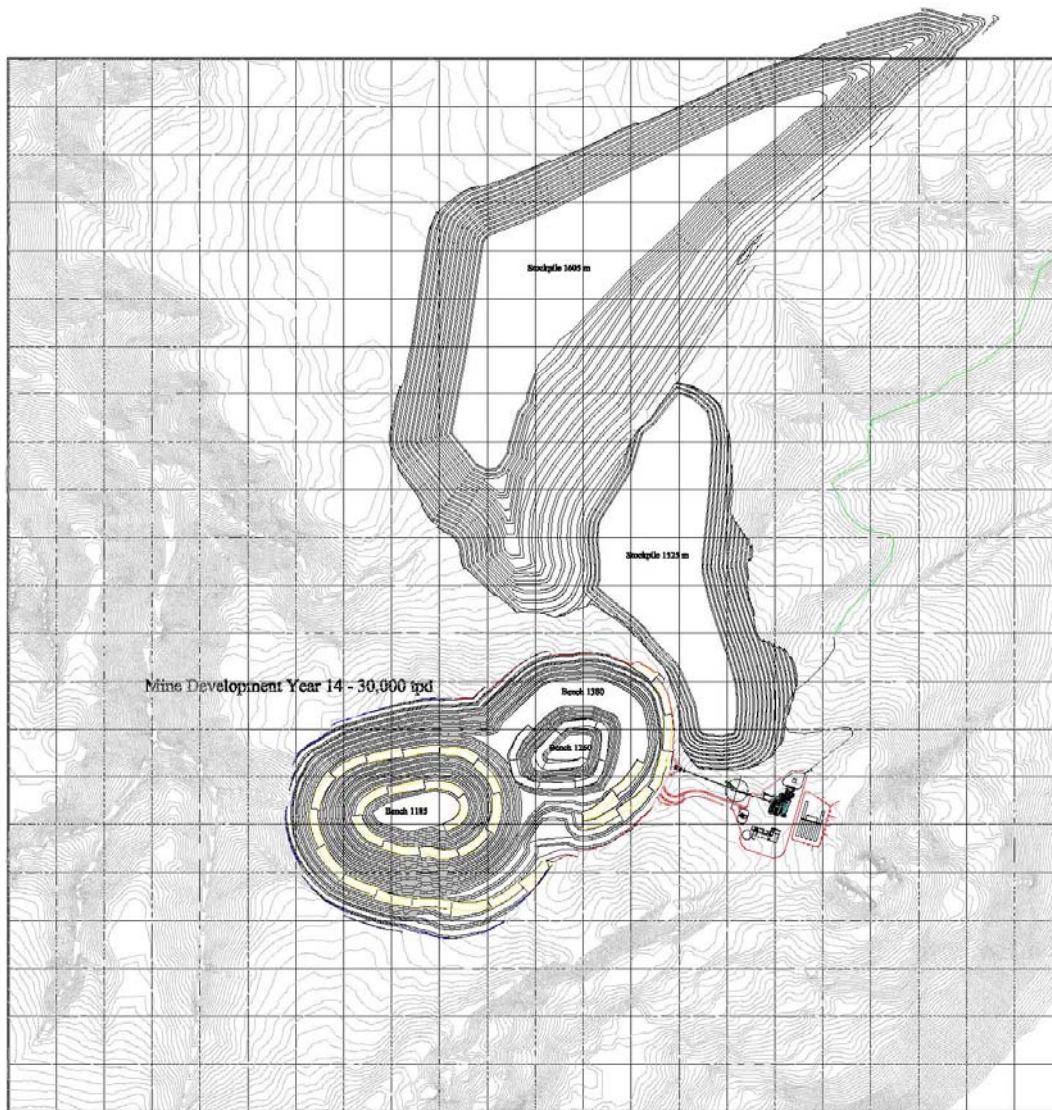
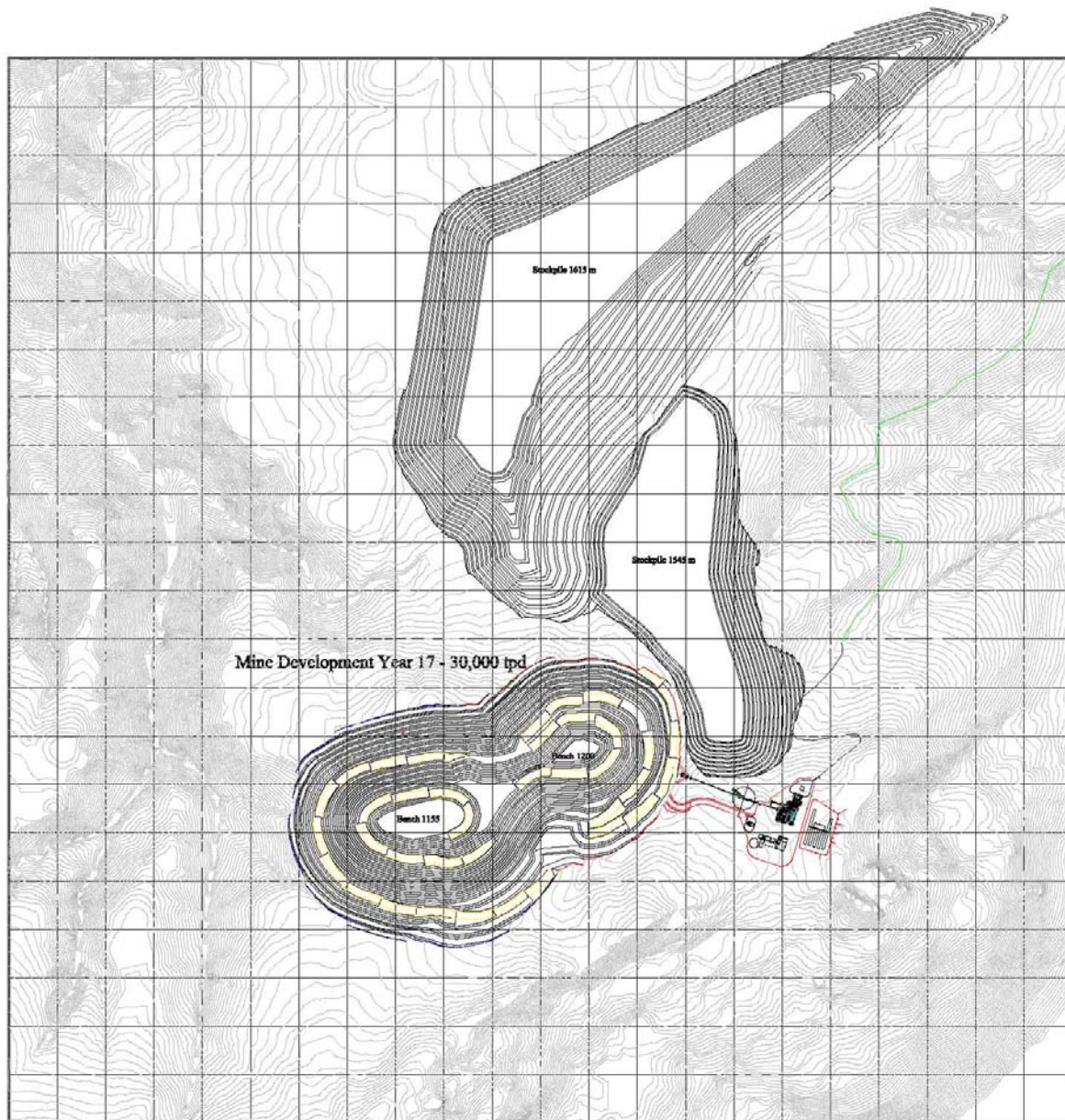


Figure 50 - Mine Development End of Year 17



Pre-production Development

A pre-production period of approximately 3 months will be required to bring the East Zone and Main Zone Phase I pits to full production. During this time, mining will open up ore in the starter pit areas and will release waste required for the primary crusher construction.

Ore mined and stockpiled during pre-production will be used during startup and commissioning of the concentrator.

Production Mining East Pit

The East Pit Phase I will be pre-stripped during the preproduction period. It is a relatively small pit and will be completed by Year 1. Overburden is so thin over the pit areas that it has not been included in the resource model. However, topsoil materials will be stockpiled for future use as reclamation cover on the waste dumps.

The East Pit Phase II mining will commence in at the same time as Phase I but will follow a few benches behind in development with completion in Year 3.

East Pit Phase III will commence in Year 3 and will be completed by Year 6. The East Pit will then remain dormant until the Year 11 when the final Phase stripping will commence. The access to the bottom of the East Pit Phase IV requires several switchbacks and mines out a portion of the final Main Zone Phase V Pit. Some scavenging of pit bottom material has been scheduled using narrow ramps and excavator extraction.

Production Mining Main Pit

The development of the Main Zone Phase I Pit will also commence during pre-production period and will be completed by Year 1. Phase II mining will commence in Year 2 and be completed by Year 6. The typical vertical separation of Phase I and Phase II will be approximately 75 m.

The Phase III pit will be mined during Year 2 through Year 5 with a one-year overlap with Phase II. The average sinking rate is approximately 5 benches per year.

The Phase IV Pit is a relatively large pit and will be mined from Year 5 through Year 10.

The Phase V pit mining will commence a year later in Year 6 and will be completed by Year 15. This phase will typically be mined 8 benches behind the Phase IV Pit.

Stockpile Recovery

Stockpile recovery will begin in Year 17 and will continue until Year 25. The mining rate will be equal the processing rate of 30,000 tpd. Although 92.5 Mt of low grade material was scheduled to be placed on the East Stockpile, the current schedule 90.6 Mt recovered. The balance is assumed to be stockpile loss.

Table 36 - Stripper Pit Table

Stripper Pit	Insitu Ore t x 1000	Grades		Insitu LG 1 t x 1000	Grades		Insitu LG 2 t x 1000	Grades		Waste tonnes t x 1000	Total tonnes t x 1000
		Cu %	Au g/t		Cu %	Au g/t		Cu %	Au g/t		
EP 1 East	3,798.0	0.668	0.527	362.0	0.144	0.133	-			6,004.0	10,164.0
EP 2 East	9,379.1	0.450	0.369	1,869.0	0.145	0.134	3,028.0	0.176	0.167	21,852.0	36,128.1
EP 3 East	13,226.3	0.436	0.426	2,320.0	0.148	0.127	7,530.9	0.199	0.179	41,269.0	64,346.2
EP 4 East	23,391.8	0.366	0.360	2,161.7	0.146	0.132	2,788.0	0.168	0.158	71,738.0	100,124.5
MP 1 Main	2,137.0	0.480	0.262	587.0	0.156	0.098	5,158.0	0.257	0.130	1,603.0	9,485.0
MP 2 Main	10,978.0	0.476	0.288	2,221.0	0.157	0.091	11,772.0	0.232	0.127	10,034.0	35,005.0
MP 3 Main	24,260.0	0.517	0.369	3,940.0	0.154	0.102	20,993.0	0.227	0.143	19,089.0	68,282.0
MP 4 Main	36,627.3	0.445	0.347	6,206.0	0.153	0.107	29,292.7	0.224	0.154	32,266.0	104,392.0
MP 5 Main	61,566.4	0.334	0.250	11,123.9	0.151	0.108	12,592.9	0.178	0.119	103,754.0	189,037.2
Total Pit	185,363.9	0.414	0.325	30,790.6	0.151	0.111	93,155.5	0.216	0.145	307,654.0	616,964.0

18.2.9 Equipment Selection

General

The initial major mining equipment together with support and auxiliary equipment required for operations are listed by year in Table 37 Equipment Requirements. Table 38 details the equipment replacement schedule.

Table 37 - Mine Equipment Requirements

Project Year			-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	
Major Equipment																													
Blasthole Drill	Lease	Rotary 311 mm	1000P	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2								
D/D1 Drill	Lease	Line Hole Drill 151 mm	Titan 500		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1								
Rope Shovel	Lease	Electric Rope Shovel 38 m ³	P&H 4100A	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Wheel Loader	Lease	Loader18 m ³	Cat 994	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Haul Truck	Lease	Diesel Electric 230 t	830E	5	6	7	9	10	12	12	12	12	12	12	12	12	12	12	12	11	7	6	5	5	5	5	5	5	
Track Dozer	Lease	425 kW	Cat D10T	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2	2	2	2	2	2	2	
Wheel Dozer	Lease	235 kW	Cat 824G	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Grader	Lease	205kW	16H	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	2	2	2	2	2	
Water Truck	Used	50 t	50 t	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Sand Truck	Used	100 t	100 t	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Spare Shovel Bucket	Used	38 m ³	38 m ³		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Support Equipment																													
Blasthole Stemmer	New		Cat 252B	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Cable Reeler	Lease	224 kW	980G	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Wheel Loader	New		996																										
Wheel Loader	Lease	224 kW	980G	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Low Bed	Rental		90 t																										
Tire Manipulator	Contractor		960G																										
Lighting Stand	New	20 kW	Amide 20 kW	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	2	2	2	2	2	2	2	
Engineering Pickup	New		4 x 4	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Engineering Pickup	New		4 x 2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1	
Pit Services Pickup	New		4 x 4	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Pit Services Pickup	New		4 x 2	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Pit Services Bus	New	F600	12 man	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2								
Pit Services Bus	n/a		20 man																										
Shovel Crew Flat Deck	New		3 t	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	
Shovel Crew Hab	New		5 t	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Surface Crew Hab	New		10 t	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Surface Crew Stinger	New		Ford F600	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Fuel & Lube Truck	New		Ford 7000	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Blasting Truck	New	F250	4 x 4	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	
Light Repair Truck	New	F350	4 x 4	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	
Mobile Crane	used		50 t	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Excavator	used	125 kW GP Bucket	Cat 325CL	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	

Table 37a - Mine Equipment Delivery

Project Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	Total
Major Equipment																											
Blasthole Drill	Rotary 311 mm	100XP	2																								2
DH Drill	Line Hole Drill 151 mm	Titan 500		1																							1
Rope Shovel	Electric Rope Shovel 38 m3	P&H 4100A	2																								2
Wheel Loader	Loader18 m3	Cat 994	1																								1
Haul Truck	Diesel Electric 230t	830E	5	1	1	2	1	2																			12
Track Dozer	425 kW	Cat D10T	4						2	2																	8
Wheel Dozer	235 kW	Cat 824G	1																								2
Grader	205kW	16H	2		1																						5
Water Truck	50 t	50 t	1								1	1															2
Sand Truck	100 t	100 t	1									1															2
Spare Shovel Bucket	38 m3	38 m3			1																						1
Support Equipment																											
Blasthole Stemmer		Cat 252B	1				1					1															3
Cable Reeler	224 kW	980G	1																								1
Wheel Loader		996																									1
Wheel Loader	224 kW	990G	1								1																2
Low Bed		90 t																									1
Tire Manipulator		980G																									1
Lighting Plant	20 kW	Amida 20 kW	5				2	2				2	2				2	2							2		19
Engineering Pickup	4 x 4	1							1				1									1					6
Engineering Pickup	4 x 2	2					2		2			2									2						12
Pit Services Pickup	4 x 4	1			1		1		1			1					1			1							12
Pit Services Pickup	4 x 2	1			1		1		1		1		1				1		1		1						12
Pit Services Bus	F600	12 man	2								1	1															4
Pit Services Bus		20 man																									2
Shovel Crew Flat Deck		3 t	2			1	1		1				1	1					1	1							11
Shovel Crew Hab		5 t	1			1																					5
Surface Crew Hab		10 t	1				1																				4
Surface Crew Stinger		Ford F800	1																								1
Fuel & Lube Truck		Ford 7000	1																								1
Blasting Truck	F250	4 x 4	2				2						2														8
Light Repair Truck	F350	4 x 4	3				3						3														15
Mobile Crane	50 t	50 t	1																								1
Excavator	125 kW GP Bucket	Cat 325CL	1									1															2
Miscellaneous																											
Radon																											
Dispatch System																											
Engineering Computers																											
GPS System																											

Table 37b – Mine Equipment Additions

Project Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	Total	
Major Equipment																												
Blasthole Drill	Rotary 311 mm	100XP	2																									2
DH Drill	Line Hole Drill 151 mm	Titan 500		1																								1
Rope Shovel	Electric Rope Shovel 38 m3	P&H 4100A	2																									2
Wheel Loader	Loader18 m3	Cat 994	1																									1
Haul Truck	Diesel Electric 230t	830E	5	1	1	2	1	2																				12
Track Dozer	425 kW	Cat D10T	4																									8
Wheel Dozer	235 kW	Cat 824G	1																									2
Grader	205kW	16H	2		1																							5
Water Truck	50 t	50 t	1								1	1																2
Sand Truck	100 t	100 t	1																									2
Spare Shovel Bucket	38 m3	38 m3			1																							1
Support Equipment																												
Blasthole Stemmer		Cat 252B	1																									1
Cable Reeler	224 kW	980G	1																									1
Wheel Loader		996																										1
Wheel Loader	224 kW	990G	1																									2
Low Bed		90 t																										1
Tire Manipulator		980G																										1
Lighting Plant	20 kW	Amida 20 kW	5																									19
Engineering Pickup	4 x 4	1							1				1									1						6
Engineering Pickup	4 x 2	2							2				2								2							12
Pit Services Pickup	4 x 4	1			1			1	1			1					1			1								12
Pit Services Pickup	4 x 2	1			1			1	1			1					1		1		1							12
Pit Services Bus	F600	12 man	2								1	1																4
Pit Services Bus		20 man																										2
Shovel Crew Flat Deck		3 t	2			1	1		1				1	1						1	1							11
Shovel Crew Hab		5 t	1			1																						5
Surface Crew Hab		10 t	1																									4
Surface Crew Stinger		Ford F800	1																									1
Fuel & Lube Truck		Ford 7000	1																									1
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Light Repair Truck	F350	4 x 4	3				3						3															15
Mobile Crane	50 t	50 t	1																									1
Excavator	125 kW GP Bucket	Cat 325CL	1				</																					

Drilling

Drilling at the Red Chris Project will encounter a variety of rock types including some sedimentary units, volcanics, intrusives and post mineral dikes with various degrees of alteration. The average compressive strength is 80 MPa. Moderate drill penetration rates and steel consumptions are to be expected.

The estimates of drill requirements are based upon the drilling of 311 mm diameter production and buffer blastholes. The primary production drills at Red Chris should be capable of single pass drilling to 16.5 m with a bit loading of up to 50,000 kg. Equipment suppliers have forecast of approximately 29 minutes for a 16.5 m hole. Based on this penetration rate and the estimated tonnage blasted per hole, only 24 holes per day are required on average to maintain production. This could easily be accomplished by one drill. However due to the phased pit design with mine development occurring simultaneously in two separate pits, two primary blasthole drills will be required.

A down-the-hole hammer, hydraulic drill, capable of drilling 127mm holes to 30m is provided for wall control drilling. This drill can also provide backup support to the primary drill units if required.

Loading

The mining of 9 open pit phases over a period of 17 years will require as many as three active work areas in certain schedule overlap years. The pit phasing has been designed to permit the mining of high-grade ore from depth to blend with lower grade material from the upper benches of the expansion pits.

The loading fleet will consist of two 38.0 m³ electric cable shovels capable of loading the 230 t haulage units with three passes. The shovels will be supported by an 18 m³ wheel loader.

When the open pit has been exhausted the shovels will be moved to the stockpile areas.

Hauling

Trucks in the 230 tonne class with 1700 kW engines are well suited to the hauls and operating conditions that will be encountered at the Red Chris Project. The selected truck fleet matches the loading units and the overall haulage profiles which are designed with adverse grades to 10%. These trucks can be fully loaded by the shovels in 3 passes and the loader in ~7 passes. Five trucks will be required in the preproduction period with additions to the fleet made as required. This increases to a maximum of 12 units in year 5 and remains at this level until active mining within the pit ceases. The increasing haul distance as the pit deepens and the height of the waste dump increases is offset by a commensurate decrease in stripping ratio, thus enabling a constant truck fleet size to be maintained.

Support Equipment

The following complement of road construction and maintenance equipment will support the mine operations:

- Three 205 kW class road graders will maintain roads on the property, including the 23km-long mine access road;
- One 100 t class water truck and one 100 t class sand truck are included;
- Four Track dozers of the 425 kW class will be used for typical dozer functions including maintenance of dumps, drill site preparation, road building, ditching, bench repair, shovel cleanup and stockpile dozing. A rubber-tired dozer of the 336 kW class is included for the majority of the lighter dozer work such as shovel cleanup and road sweeping;
- Diesel pickup trucks will be supplied for the Mine Superintendent, Maintenance Superintendent, Shift Foreman, Blaster, Engineering, Surveyors and the mechanical crew. A total of 8 units are provided for initially;
- Maintenance support vehicles and equipment will include a tire handler; flat deck trucks; fuel, water and lube trucks for servicing shovels; welding truck; and
- Miscellaneous vehicles such as an excavator, lighting towers are also provided for the support of mine operations.

Mine Dewatering

Direct precipitation of rain and snow and groundwater inflows will be the major sources of water in pit areas. Diversion ditches will be located above the open pit areas to intercept water flows to the maximum extent possible. This water will be redirected to the tailings impoundment, or diverted to existing natural drainages.

Groundwater pressures in the pit walls will be reduced, if necessary, by means of horizontal drain holes drilled from the bench floor during mining. The precise number of holes required will be determined by structural discontinuities and locations of perched water tables encountered during mining. At this stage, the requirement for pit wall dewatering/depressurization has not been determined.

Mine dewatering will be carried out using diesel powered submersible pumps installed in sumps at the bottom of the pit. Water will be pumped from the open pits and discharged into the fresh water tank for use in the mill with any excess being discharged to the TMF.

18.2.10 Mine Workforce

The anticipated mine workforce is summarized in Table 39 Mine Staffing Plan and Table 40 Mine Hourly Employees.

18.2.11 Explosives Use and Handling

The mine explosives will be stored in two separate areas close to the open pit. The explosives plant office and garage, ammonium nitrate silos and emulsion tank will be stored within a fenced compound to be located in accordance with the British Table of Distances. The storage facility will accommodate up to 20,000 kg of emulsion.

A powder magazine will house all high velocity explosives. The magazine will be bullet proof, fire resistant, theft-resistant, weatherproof, well ventilated, and conform to standards set by the department of Energy Mines and Resources Canada.

The Company will contract with an explosives supplier to provide the facilities. Bulk explosives will be purchased on an “in-the-hole” basis based on the requirements as determined by the Blasting Engineer.

18.3 Infrastructure

18.3.1 Site Development

The Red Chris property is located about 18 km southeast of the village of Iskut and 80 km south of Dease Lake on the north-facing Todagin Plateau between Ealue and Kluea Lakes in northwestern British Columbia. It is approximately 450 km north of the town of Smithers. The deep-sea port of Stewart is situated about 200 km due south, and about 320 km by road. The property is centered on latitude 57° 42' North, longitude 129° 47' West within NTS map sheet 104H/12W, Liard Mining Division.

The Red Chris property is situated on the eastern portion of the Todagin upland plateau, which forms a subdivision of the Klastine Plateau along the northern margin of the Skeena Mountains. Elevations on the property are typically 1,500 +/- 30m with relatively flat topography broken by several deep creek gullies.

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

The process facilities and ancillary facilities will be located on a benched platform located immediately to the east of the open pit. The platform will be formed by a localized sidehill cut to fill and cut to waste operation. The volumes of cut to fill and cut to waste will be in the order of 263,000 m³ and 280,000 m³ respectively. The estimate also allows for the construction of 10 km of on site service roads.

18.3.2 Services & Utilities

18.3.2.1 Fresh Water System

The fresh water system has been designed to supply up to 120 m³/hr of make up water. It will be supplied from a series of ten wells located downstream of the north seepage dam and a series of five wells located downstream of the south seepage dam, as shown on the site layout drawing.

AMEC conducted a single pump test of the aquifer in 2004 yielding approximately 20 m³/hr over a 3-day period without noticeable drawdown effects. The wells will be approximately ten metres deep, spaced at 100 metre intervals and at about elevation 1040m.

The fresh water will be pumped from these wells via a 150mm diameter pipeline, 9.9 km long, to a tank and booster pump station located part way up the hill and adjacent to the tailings and sand splitter box at elevation 1340m. From here the water will be pumped to a 9 metre diameter by 9.6 metre high fresh/fire water tank located at the millsite at elevation 1438m. The pipeline will be a buried pipeline for frost protection and will follow the planned service road between the mill and tailings impoundment.

18.3.2.2 Fire Water

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

A firewater pump skid complete with diesel driven fire pump, jockey pump and controls will be installed. A fire loop complete with hydrants will also be provided. The 100 tonne capacity mine water truck will also be available for fire fighting duties.

18.3.2.3 Potable Water

A potable water well or wells to be installed in a suitable location in close proximity to the accommodation camp and mill site;

The potable water tank, water treatment plant and distribution system have been designed based upon a maximum design population of 200 and an average daily demand per capita equal to 284 litres per day. The water treatment plant will also comprise a chlorination system. Nominal potable water use is estimated to be 2.5 m³/hour

18.3.2.4 Process Water

Process water will be used primarily in the grinding, flotation and regrind circuits. Reclaim water from the tailings impoundment will be used to provide all process water requirements. Two stages of pumping is required to reclaim water from the tailings pond to the plant site. A reclaim barge equipped with vertical turbine pumps will reclaim water from the tailings pond and pump the water to a booster station located on shore. From there, the water will be pumped using vertical pumps to the process water tank located at the plant site.

18.3.2.5 Sewage Treatment

The sewage treatment plant will be supplied with the Permanent Camp. The site sewage treatment facilities will consist of a packaged wastewater treatment plant (Filterbox LCU or similar type unit) designed for the following criteria:

Flow: 45 m³/day (174 person capacity camp)

Design BOD₅ loading is 500 mg/L up to max 25 kg/day

Design TSS loading is 500 mg/L

Design Oil and Grease 80 mg/L

Treated Effluent Criteria

CBOD₅ 20 mg/L

TSS 20 mg/L

TFC <200/100 ml

Based on 30-day average of composite samples.

The treatment process proposed is extended aeration activated sludge biological treatment. Two parallel process trains are proposed with each designed to be installed within a 10' wide x 60' long prefabricated industrial trailer.

The influent wastewater is collected within lift stations from which it is pumped to the two sewage process trains. The influent wastewater is directed to the inlet of the solids separation tank to trap settled solids as well as oil and grease. The wastewater then flows into the Anoxic/EQ tank that attenuates the peak flows and pre-mixes the influent wastewater with return activated sludge from the aeration tank. From the anoxic/EQ tank the wastewater is pumped into the aeration tank.

The main treatment is performed by the activated sludge process within the aeration tank. The extended aeration activated sludge process is a suspended growth biological treatment. The process utilizes aerobic (oxygen using) bacteria to remove organic contaminants through a process of biological oxidation. The air required to meet the oxygen demands of the system and to provide the mixing of the mixed liquor suspended solids is supplied via a central blower system and fine pore bubble diffusers. The wastewater flows from the aeration tank to a secondary clarifier that settles out the biological solids and skims off any residual oil and grease. The solids are recycled to the activated sludge system to maintain the mixed liquor suspended solids concentration in the aeration tank or pumped to the solids holding tank.

The clarified effluent flows from the clarifier to an effluent holding tank from which it is pumped through a cartridge filter for final effluent polishing prior to discharge. This provides an additional measure of process safety in case of a process upset condition. Filtered effluent is then disinfected using UV radiation. The Filterboxx LCU extended aeration package sewage treatment plant is capable of providing effluent with BOD₅ and TSS well below typical regulatory requirements of 20 mg/L BOD and 20 mg/L TSS.

The treated wastewater from the site sewage treatment plant will be pumped to the tailings impoundment by way of the final mill tailings pump box.

18.3.2.6 Fuel Storage

Three steel tanks each with a capacity of 70,400 litres (18,600 gallons) will be provided together with one tank with a capacity of 22,700 litres (6000 gallons) for the storage of small truck diesel and gasoline. All three tanks will be contained within a HDPE lined bermed compound.

18.3.2.7 Power Line

The project assumes that power would be supplied from an extension to the North American grid which currently terminates at Meziadin Junction some 230km to the south. This 138kV line would be extended north along a right-of-way following Highway 37 and initially terminating at the Iskut First Nations village, approximately 18km to the North of the Project site. The B.C. Ministry of Energy and Mines, in conjunction with BC Hydro and the British Columbia Transmission Corporation, are committed to explore every option for the construction of this line. Red Chris project capital includes the cost of building a 23km power line at 138kV, 636 ASC Orchid single conductor from Highway 37 to the mine site following the mine access road.

18.3.2.8 Access Road

The proposed access road location was selected on the basis of considering factors such as public safety, constructability, potential environmental impacts, and access control. The route will be from Highway 37 along the south side of Coyote Creek and rising up to the plateau around the north and east sides. It will be a closed radio-controlled road with a fenced compound with 24 hour, 7-day security provided at the Highway 37 junction. Only authorized personnel will be permitted into the mine area.

18.3.3 Ancillary Facilities

18.3.3.1 Accommodations Complex

A permanent accommodation camp facility will be provided and includes sleeping, dining, recreation and support facilities for the permanent workforce of approximately 250 persons, of which approximately 125 will be on site at any given time. The accommodation camp has been designed with a total of 174 rooms.

The permanent camp will consist of 3 x 32 person modular dormitory complexes (each consisting of 6 trailer units of 62' x 12') for a total of 96 rooms where each room would be a single bed facility sharing a washroom with the adjoining room. Each 32 room dormitory would include a television room, a laundry and a janitor's work and storage room.

In addition there would be 4 x 18 person dormitory units plus 1 x 6 person dormitory units for a further total of 78 rooms. The 18 person dormitory units would be single bedrooms with separate washrooms.

The dining room and recreation facilities will be designed for a 150 person camp population.

The overall camp would consist of 3 x 32 person dormitory units, 4 x 18 person management dorm units plus 1 x 6 person management dorm unit, plus the 150 person dining unit and a recreation facility all linked by covered walkways.

18.3.3.2 Maintenance Shops and Warehouse

The maintenance shops, a 78m x 36m (6 bays) building, will contain repair facilities for the mine haulage and all surface mobile equipment. Included will be a welding and machine shop bays, light vehicle, lube room, maintenance offices, mine supervision offices, ambulance and fire truck bays, mine rescue/first aid facility, lunchroom and employee change rooms. An additional part of the building (39m x 54m) will contain the warehouse and purchasing department for the property.

18.3.3.3 Assay Laboratory

The assay laboratory will contain a sample preparation area, and chemical laboratory for standard ore analyses, including gold fire assay. The environmental analysis facility will be part of this building.

18.3.3.4 Explosives Facility

The explosives buildings will be supplied and built by the supplier as part of the supply contract. It will contain the bulk emulsion, caps magazine, ammonium nitrate storage all constructed to code as well as equipment servicing and employee facilities.

18.3.3.5 Administration Office

The Administration Office will be a prefabricated modular unit of similar construction to the camp.

18.3.4 Process Facilities

18.3.4.1 Mill Complex and Structures

The mill building will comprise of a pre-engineered steel framed and steel clad insulated building founded on spread footings.

The ball mill and sag mill foundation will comprise of raft type foundations founded on soil with sufficient mass and bearing area to dampen the dynamic forces and to distribute the load. The tank foundations will comprise of ring beams founded on rock with granular in-fills. Spill containment areas comprising of slabs on grade with perimeter curbs will be provided for essentially the entire plant area. The overall footprint for the mill complex will be approximately 140 metres long by 80 metres wide.

The thickeners will comprise of vendor supplied elevated steel tanks on steel supports on spread footings.

18.3.4.2 Primary Crushing and Coarse Ore Stockpile

A twenty one metre high reinforced earth wall (hilfiker), faced with wire mesh, has been allowed for at the primary crusher. The 230 tonne dump pocket and gyratory crusher tower will comprise of an insitu concrete tower founded on soil with sufficient mass and bearing area to dampen the dynamic forces and to distribute the load.

The conveyor support structures between the primary crusher and the coarse ore stockpile will comprise of uncovered, open type, structural steel trusses with a cantilevered walkway supported on steel bents founded on spread footings. The stacker will comprise of a cantilevered structural steel frame on tied concrete columns founded on spread footings. The coarse ore stockpile structures will comprise of a concrete central chamber to house the apron feeders and multiplate type steel discharge and escape tunnels. The conveyor support structures between the coarse ore stockpile and the sag mill will comprise of uncovered, open type, structural steel trusses with a cantilevered walkway supported on steel bents founded on spread footings. The pebble transfer conveyor and pebble cone crusher will be of similar construction. The conveyor transfer towers and take up towers will comprise of unclad braced steel frames founded on spread footings.

18.3.5 Transportation

18.3.5.1 Concentrate haulage

The haulage of concentrate to the Port of Stewart will be contracted out. The trucks carrying 40-43 tonnes will be loaded and weighed inside the concentrate storage building. All trucks will be covered before leaving the building to prevent loss of product. The tires and lower trailer frame will also be sprayed with water to remove any spilled concentrate. At the design rate of 30,000 tonnes per day, there will be 12 to 14 trucks per day operating on a continuous basis. In the event of road closure for weather or maintenance, there will be five (5) days of storage in the concentrate building.

18.3.5.2 Personnel

Personnel will be transported from Dease Lake and pick up points enroute to the minesite by bus and/or van. Charter flights originating in Vancouver and Prince George will pick up employees in Smithers and/or Terrace for flights to Dease Lake.

18.4 Markets and Contracts

It is anticipated that the concentrate will be sold to markets through out the Pacific Rim countries. Discussions have been held with a number of smelters, however, no contracts have been entered into at this time.

18.5 Environmental Considerations

18.5.1 Environmental Assessment and Permitting

British Columbia's Environmental Assessment Act requires that certain large-scale project proposals undergo an environmental assessment before they can proceed.

Proposed mining developments that exceed the threshold criteria laid out in the Reviewable Project Regulation are required to obtain an environmental assessment certificate from the BC Environmental Assessment Office (BC EAO) before any development related permits, licences or approvals may be issued. The Red Chris Project's anticipated extraction of 30,000 tonnes of ore per day, or 10.95 million tonnes per annum, exceeds the threshold limit of 250,000 tonnes per year established under the Regulation and therefore will require an environmental assessment certificate.

The EA process involves six stages, starting with determining how the assessment will be conducted, through preparation, review and approval of Terms of Reference for the Application Report, preparation and submission of the Application report, review of the Application report, preparation of the EA Assessment report by the BC EAO followed by reference to the appropriate provincial ministers for a decision, culminating in a decision to either issue or not issue an EA Certificate. The decision to approve a mining project is made jointly by the Provincial Ministers of Energy and Mines, Sustainable Resource Management; and Water, Land and Air Protection.

The Canadian Environmental Assessment Act ("CEAA") is triggered by Federal involvement in a project. CEAA applies when a federal department or agency is required to make a decision on a proposed project. The Canadian Environmental Assessment Agency has made a determination that the Red Chris Project will require an environmental assessment under CEAA. The "Responsible Authorities" under CEAA will include the Department of Fisheries and Oceans and Natural Resources Canada.

The Governments of Canada and British Columbia entered into the Canada-British Columbia Agreement on Environmental Assessment Cooperation on March 11, 2004. The BC EAO and the Canadian Environmental Assessment Agency (CEAA) have indicated that it is their intention that the environmental assessment process for the Red Chris project will undergo a single, cooperative assessment as provided for in this agreement. It is intended that the Assessment Report prepared by the BC EAO will be written to serve both the needs of the Federal and Provincial environmental assessment process.

Red Chris Development Company ("RCDC") submitted its Project Description for the Red Chris project to initiate the Environmental Assessment process with the BC EAO on October 27, 2003. The Project Description provided a conceptual level outline of RCDC's plans to develop an open pit copper-gold mining and milling operation on Todagin Plateau in North-western BC near the village of Iskut with construction scheduled to begin in the first quarter of 2005 and operations by the fourth quarter of 2006.

The EAO subsequently issued an Order under Section 10(1)(c) of the BC Environmental Assessment Act S.B.C. 2002, c.43 (BCEAA) on November 26, 2003 determining that the Project constitutes a Reviewable Project under Part 3 of the Reviewable Projects Regulation (BC Reg. 370/02). As such an environmental assessment certificate will be required pursuant to the BCEAA prior to the project proceeding.

Draft Terms of Reference were developed by RCDC in accordance with the BCEAA in order to define the information requirements necessary for inclusion in an Application for an Environmental Certificate and submitted to the BC EAO in March 25, 2004. Comments on these Terms of Reference were subsequently received from a number of federal, provincial and local government agencies. Revised Terms of Reference for the Application for the project incorporating the comments received were submitted to the BC EAO on May 12, 2004. Final Terms of Reference were subsequently submitted on June 18, 2004 and approved by the BC EAO.

The BC EAO issued an Order under Section 11 of the Act on August 4, 2004 setting out the scope, procedures and methods to be used for the environmental assessment of the Red Chris Project.

RCDC subsequently compiled and submitted its Application for an Environmental Assessment Certificate to the BC EAO on September 15, 2004, initiating the 30 day screening period. Following receipt and incorporation of review comments the Final Application was submitted on November 2, 2004, initiating a 180 day review period scheduled for completion on or before April 30, 2005, at which time a recommendation will be made to the Ministers for a decision within 45 days.

Under the Concurrent Permitting Regulation, a proponent may request that some provincial approval applications be processed concurrently with the environmental assessment. In that case, the agency responsible for the approval must make a decision relating to issuing the approval within a specified time frame after certification.

In conjunction with its EA Application, RCDC applied for concurrent permitting in accordance with the Concurrent Approval Regulation (B.C. Reg. 371/2002) for licenses, permits and approvals which are key to initial construction activity scheduled to commence in early 2005, including the following:

- All licenses, permits and approvals related to the construction and operation of the mine site access road, including a Special Use Permit, a licence to cut, and a Mines Act Permit for that portion on mineral claims;
- A Mines Act Permit for pre-production site construction activity, including the development and use of a construction aggregate quarry construction of site roads, foundation excavations for the mill, crusher and maintenance facilities, diversion ditches, coffer and starter dams in the tailings impoundment facility and other initial construction related activity.

Under the Concurrent permitting regulation, these permits will be required to be issued within 60

days of the Ministers decision on certification of the project. Additional permits, licenses and approvals required for construction and operation of the mine will be applied for as development planning proceeds and additional information becomes available through detailed design and engineering.

18.5.2 Land Use Planning

The Red Chris Project lies within the area covered by the Cassiar Iskut-Stikine Land and Resource Management Plan (LRMP) which received BC Cabinet approval in October of 2000. The Cassiar Iskut-Stikine LRMP encompasses 5.2 million hectares in north-western British Columbia. The plan represents the consensus reached as a result of a three-year interest based negotiation process that involved approximately 25 public, First Nations, and provincial government representatives.

The Red Chris Project falls within the Todagin Zone of the LRMP. The Todagin zone comprises a large area that includes Todagin Plateau and Tsatia Mountain. The eastern boundary extends to the tree line of the Klappan drainage. The management intent for the Todagin Zone is to integrate management for Stone's sheep and other wildlife, recreational activities, mineral exploration, mine development and reclamation. The zone has been recommended for designation as a wildlife management area.

While the LRMP recommends that the Todagin Zone be designated as a Wildlife Management Area (WMA), the Red Chris copper/gold project was excluded from the WMA and road access was acknowledged as being a required and appropriate activity within the context of the overall management goals and objectives.

The zone provides habitat for a major Stone's sheep population. Other wildlife species include mountain goats, grizzly, wolves, moose, caribou, marmots and raptors. The area also has high recreation values for hiking, wildlife viewing and bow hunting (including guided hunting trips).

The LRMP states: "The northern portion of the zone, which encompasses Todagin Plateau, has a very high mineral potential and includes the Red Chris Project.", and "Because of its proximity to the community of Iskut, development of the Red Chris deposit has the potential to provide significant local economic benefits."

The LRMP establishes the following objective for the Todagin Zone: "To conserve Stone's sheep populations and habitat and other wildlife values integrated with mineral exploration and development."

The LRMP contains the following recommendations relating to management of the Todagin Zone:

With the exception of the Red Chris property, this zone will be designated as a Wildlife Management Area (WMA) with the following conditions:

- Mineral exploration and mine development and associated access continue to be recognized as appropriate activities;
- Fully integrate the management of wildlife, mineral exploration and mine development north of Todagin Creek. South of Todagin Creek, mineral exploration and mine development are acceptable activities, with maintenance of wildlife values as the primary consideration;
- Current approval processes will continue i.e., there will still be a one-window approach to project approval with consultation between the Ministry of Energy and Mines and the Ministry of Water, Land and Air Protection;
- Locate road access, mining camps and other infrastructure away from critical habitats. Only consider exceptions to this strategy after fairly assessing and weighing all implications (ecological, economic and safety);
- Where mineral deposits occur and mine development proceeds within mapped critical habitat;
- Design development to minimize impacts to habitat during operations; and
- Fully reclaim habitat in a timely manner (on an ongoing basis and when operations are completed).

If mine development is proposed, the following should be included as part of the approval process:

- Complete baseline inventories of wildlife population and habitat;
- Incorporate a mountain ungulate monitoring plan as part of issuing mine permits;
- Address potential for copper toxicity in Stone's sheep; and
- When operations are finished, fully rehabilitate mine sites and roads with native species ecologically suited to the area and palatable to local wildlife species.
- Locate road access, mining camps and other infrastructure away from existing tourism facilities.
- Consult fully with the public, including local residents and the LRMP Monitoring Committee, regarding any new road locations.
- Provide access controls, including staffed gates, where needed, to manage public access to the plateau area; and
- Permanently deactivate roads upon completion of operations.

RCDC's Application for an Environmental Assessment Certificate has incorporated management and mitigation measures designed so as to meet the goals and objectives of the Cassiar Iskut-Stikine LRMP.

18.5.3 First Nations

The Red Chris property is located in a geographically isolated and sparsely populated area of the Northwest Region of British Columbia, and within the Traditional Territory of the Tahltan and Iskut First Nations. The communities in the area are considered remote and include Iskut, Dease Lake and Telegraph Creek.

First Nations community members have used and continue to use the area around the Red Chris property to carry out traditional activities such as hunting, trapping and harvesting of a variety of

plant species for consumptive and medicinal purposes.

The closest community to the Red Chris property is Iskut (pop. 300), home to the Iskut First Nation, 18 km north of the property. Dease Lake (pop. 661) and Telegraph Creek (pop. 296), home to the Tahltan First Nation, are located 80 km north and 85 km to the west-northwest of the property, respectively. The total population of the Region on May 15, 2001 (Census Day) was 1,316.

The Tahltan people, particularly through the Tahltan Nation Development Corporation since its inception in 1985, have been active in mining and contracting operations within their traditional territory for many years, including at the Golden Bear and Eskay Creek mines. Through these employment and associated training opportunities, the Tahltan have developed a well-trained and experienced workforce upon which RCDC hopes to draw a significant portion of its employees.

RCDC initiated and has maintained regular communication with representatives of the Tahltan and Iskut First Nations beginning prior to the commencement of exploration activity in 2003.

On January 19, 2004 RCDC executed a Memorandum of Understanding with the Tahltan First Nation, the Iskut First Nation, and the Tahltan Central Council. This document sets out the principals for joint cooperation on the project development and operation between RCDC and the First Nations, including minimizing impacts on traditional use. Discussions have been initiated with the First Nations with regard to their involvement in the development of the project and the most appropriate means of ensuring that economic benefits accrue to the Tahltan people within whose traditional territory the mine is located.

The Tahltan and Iskut First Nations have been included throughout the Environmental Assessment Process being led by the BC Environmental Assessment Office. A series of Open Houses were conducted in the local communities of Stewart, Iskut, Dease Lake and Telegraph Creek in both May and December, 2004.

18.5.4 Reclamation and Closure

Under the B.C. Mines Act and the Health, Safety and Reclamation Code for British Columbia, the primary objective of the reclamation plan will be to return where practical all areas disturbed by mining operations to their pre-mining land use and capability.

The following goals are implicit in achieving this primary objective:

- The long-term preservation of water quality within and downstream of de-commissioned operations;
- The long-term stability of engineered structures, including the waste rock dumps, open pit and tailings impoundment;
- The removal and proper disposal of all access roads, structures and equipment that will not be required after the end of the mine life;
- The long-term stabilization of all exposed erodible materials;

- The natural integration of disturbed areas into the surrounding landscape, and the restoration of a natural appearance to the disturbed areas after mining ceases, to the best practical extent;
- The establishment of a self-sustaining cover of vegetation that is consistent with existing forestry and wildlife needs.

A conceptual Reclamation and Closure Plan was presented in Section 6.8 of the Application for an Environmental Assessment Certificate (AEAC).

The primary goals of the Reclamation and Closure Plan for the Red Chris site will be:

- to prevent degradation of the environment, and
- to enhance the natural recovery of areas affected by mining.

In order to achieve these goals, the specific objectives of the plan will be:

- to ensure that mine facilities, mine waste disposal sites and tailings areas will be closed in such a manner that the requirement for long-term maintenance and monitoring is minimized.
- to identify potential sources of contaminants and to prevent the release of contaminants or wastes to the environment.
- to return affected areas to a state compatible with original undisturbed conditions, considering such practical factors as economics, aesthetics, future productivity, and future users.

Before exploration began in the Red Chris Project area, the principal land uses were wildlife habitat that supported hunting, guide outfitting, trapping and some general outdoor recreation.

The goal of land reclamation program at Red Chris will be the creation of conditions which will promote the establishment of a self-sustaining wilderness environment that is consistent with surrounding land use potential and compatible with the original undisturbed conditions. The two major components of the restoration program will be erosion control and re-vegetation.

To the extent practical, surface drainage will be directed back to original watercourses. Where this is not practical, new channels will be excavated to supplement natural watercourses. All such excavations will be backsloped, re-contoured and rip-rapped as necessary to stabilize banks and control erosion. All works will be designed to minimize the need for future maintenance.

Where necessary, disturbed slopes will be treated to maintain an acceptable degree of stability, prevent erosion and facilitate natural re-vegetation to a state compatible with original conditions. Reclamation prescriptions will include native species ecologically suited to the area and palatable to local wildlife species.

The reclamation plan aims to achieve the following land use objectives:

- To restore the area disturbed by development of the waste rock dumps into a mixed vegetated area with similar use and productivity to that currently in existence in these areas;

- To restore the area disturbed by the plant site facilities and accommodation camp into a revegetated area with similar use and productivity to that currently in existence in these areas;
- To restore the tailings impoundment into a self sustaining shallow pond environment with broad revegetated beaches in front of the upstream face of the North and Main dams with 4H:1V revegetated slopes on the downstream dam faces.

The objective will be over time to achieve similar productivity in vegetation mix and growth rates to the productivity rates that currently existing in these areas. The end objective is to permit future wildlife use on this reclaimed land similar to that observed in the pre-development period.

The preliminary projected total disturbed area associated with the development and operation of the Red Chris mine is estimated at 995 ha, including:

- Open pit 110 ha
- Tailings impoundment 465 ha
- Waste rock dump 220 ha
- Low grade stockpile area 70 ha
- Site access roads 25 ha
- Borrow areas 80 ha
- Plant site 25 ha

Progressive reclamation will be undertaken over the life of the mine in order to minimize the outstanding reclamation liability at the end of the mine life. At the end of the mine life all buildings and structures will be removed, foundations removed or cracked and buried and disturbed areas re-contoured, capped with suitable growth medium and re-vegetated with appropriate species compatible with the surrounding environment.

A breakdown of the estimated cost to complete the anticipated reclamation and closure activity at the end of the planned mine life was included in the Addendum to the AEAC report. The Preliminary Reclamation Cost Estimate of \$10.2 million was based on the currently envisioned mine plan and concepts developed for reclamation of the various components which make up the Project.

It is anticipated that the Reclamation and Closure Plan, and the associated costs, will be refined over the mine life in response to adjustments in the mine plan, the results of reclamation research, evolving technology and other such factors. Such refinements will be presented in Annual Reclamation Reports and regular updates to the Mine and Reclamation Plan as required under the Permit Approving the Mine Plan and Reclamation Program to be issued for the mine development under the Mines Act.

18.5.5 Environmental Impact Management

RCDC's development plan and environmental impact assessment are based on an extensive database of environmental baseline studies conducted on and around the Red Chris property over the past decade. Initial fieldwork was undertaken at the property in 1994 and 1995 by Hallam Knight Piesold, and further detailed studies were continued through 1996 and 1997 by Rescan

Environmental Services. In addition to meteorology at the site, hydrology and water quality in the lakes and streams around the project area, the studies have also covered fisheries and aquatic life, wildlife and vegetation resources. The automated weather and hydrology stations continued to collect data through to 2000.

As part of the planning to initiate mining development at the Red Chris deposit, RCDC contracted specialist consultants to consolidate and synthesize the environmental baseline information previously collected in the project area. This information has been incorporated into and forms the basis for the Environmental Assessment documentation.

To compliment the extensive existing database, RCDC renewed the collection of baseline environmental data with studies commencing in the Fall of 2003 and continuing through the summer of 2004. These studies included additional wildlife surveys, water quality sampling, re-establishment of hydrology stations, collection of samples for ARD testwork, fisheries surveys, terrestrial ecosystem mapping, surface and groundwater quality sampling, collection of additional hydrometric data and reestablishment of the on-site meteorological station. The results from these additional studies were incorporated into the Application for an Environmental Assessment Certificate and the Addendum report submitted in November 2004.

The assessment of potential environmental impacts has led to the development and presentation of a series of conceptual level plans to address operational aspects of the development such as:

- Tailings Impoundment Operating Plan;
- ARD/ML Prediction and Prevention Plan;
- Materials Handling Plan;
- Reclamation and Closure Plan;
- Wildlife Management Plan;
- Fish Habitat Compensation Plan;
- Water Management and Sediment Control Plan;
- Pollution Prevention Plan;
- Spill Contingency and Emergency Response Plan; and
- Waste management Plan.

Each of these plans incorporates specific mitigation measures designed to address potential impacts associated with specific components of the development plan.

18.5.6 Fish and Fish Habitat

Baseline studies have indicated that the only fish species found to be resident within the lakes and streams within the immediate area surrounding the proposed Red Chris Project is rainbow trout. Rainbow trout are a relatively common species within the project area and the region in general. No fish species other than rainbow trout have been identified within the Trail or Quarry Creek systems proposed for the tailings storage impoundment, or in any of the other streams immediately downstream of the project site.

These studies have indicated that the upper reaches of Trail Creek where the proposed tailings

impoundment is to be located, including the small pond known as “Black Lake”, are devoid of fish populations. However, a small population of rainbow trout are present downstream of the proposed tailings impoundment within the lower reaches of Trail Creek, up to and including the proposed location of the South Dam.

RCDC commissioned additional fieldwork in the 2004 field season to assess, characterize and quantify fish habitat within creek systems draining the valley downstream of the proposed tailings impoundment. At the same time opportunities for habitat compensation were explored. The results of this work were reported in November 2004 in the addendum to the application report. This data has been used to quantify the potential for habitat alteration; disruption and/or destruction associated with the proposed project development and will form the basis for discussions of a proposed habitat compensation plan to be developed in association with DFO and WLAP.

18.5.7 ARD and Metal Leaching

An extensive geochemical database has been developed for the Red Chris Project. The results of this work have been used as a basis for geochemical characterization of all overburden, waste rock, ore and tailings materials to be generated in conjunction with the development of the Red Chris Project, and to develop prediction, prevention, mitigation and monitoring strategies for the project to deal with Acid Rock Drainage and Metal Leaching (ARD/ML) issues throughout the life of the mine and post-closure.

The ARD static database for Red Chris consists of over 1000 Acid Base Accounting analyses with associated ICP total metal analyses, 5 – 20 day leach tests and detailed mineralogical examinations. As well, kinetic tests, such as humidity cells and column tests, have been undertaken to define potential sulphide oxidation and NP depletion rates over time, and the potential chemistry of resultant drainage from waste rock dumps and the tailings disposal area. In all, a total of 14 waste rock humidity cells, 9 columns and 15 tailings humidity cells have been initiated, many of which have been running for about a year.

These programs of work have led to the development of a Material Handling Plan and an ARD Prediction and Prevention Plan designed to provide for the long term handling of all potentially acid generating materials. The testwork has demonstrated that the tailings can be produced to be Non-Acid Generating (NAG) and used in dam construction and to form upstream beaches to improve dam stability over the long term. The majority of waste rock, however, has been shown to display varying potential to oxidize and generate acidic runoff over a period of many decades and as such will require special handling consideration.

Over the mine life it is projected that a total of 307 million tonnes of waste rock and 31.5 million tonnes of ultra low-grade ore (LG1) will be placed into the North waste rock dump for permanent storage. Of this total some 47 million tonnes is expected to be non-acid generating (NAG) rock. The remainder is expected to be potentially acid generating (PAG) with an estimated time to onset of acid generation in the order of several decades.

In addition, 92.4 million tonnes of low-grade ore (LG2) will be placed to the east side of the

open pit in a separate area in close proximity to the crusher to facilitate the recovery of this material for subsequent milling once open pit mining is complete. It is planned that this low-grade ore will be reclaimed starting in year 17 of the mine life (once open pit mining ceases) and processed through the mill. It is expected that this material will be gone by the end of the planned mine life at the end of Year 25.

In order to address issues relating to long term storage of PAG waste rock, a number of design considerations have been built into the waste rock storage site. The base of the North Waste Dump will be constructed of a thin layer of NAG waste rock where necessary to fill in any natural topographic depressions and drainages in order to ensure that surface or ground water passing under the dump does not come into direct contact with the PAG rock. An allowance for a NAG layer up to approximately 5m in thickness has been allowed for in the design. PAG rock will then be placed onto the North Dump on top of the NAG rock base. By preventing direct contact of PAG rock with flowing water, the potential for flushing of oxidation products from the waste rock will be minimized.

Waste and low-grade material will be placed in 10 m lifts to facilitate higher compaction thereby further limiting water infiltration into the dump. Berms of 25 m in width will be left on the perimeter of each lift effectively flattening the slopes and simplifying the reclamation of the final dump surface. The North Waste Dump will be sloped at 4:1 in the north, east and south directions and 2:1 in the westerly direction. The top of the dump will be sloped to the southwest and surface drainage will be directed to a constructed channel leading to the existing valley draining this area. Low grade LG2 material will be placed in the East Stockpile in close proximity to the crusher in order to facilitate reclaim and processing once mining operations cease. The East Stockpile will be constructed with 2:1 slopes to reduce the overall footprint of the stockpile.

At closure the North Dump will be capped with a “store and release” vegetated soil cover (assumed to be 1 m of till) to shed as much “clean” precipitation runoff as possible from the surface of the North Dump. The “clean” runoff would be directed away from the open pit. The purpose of the cover is to minimize the amount of precipitation runoff infiltrating into the underlying dump thereby reducing the rate at which ARD and metal contaminants are flushed from the PAG rock.

The North dump has been sited immediately to the north of the open pit so that all drainage from the dumpsite will flow by gravity into the tailings impoundment area via a constructed side hill diversion ditch during the mine’s operational life. At closure all seepage from the waste dump will be directed to the open pit for containment via a constructed drainage tunnel. It is expected that it will take in the order of 80-90 years for the open pit to fill.

Prior to the water in the pit reaching a level whereby it would discharge to the environment, RCDC will control the pit lake level by pumping contaminated water from the pit to a new water treatment plant to be constructed near the proposed mill site. This treatment plant will utilize proven state-of-the-art technology, such as that which has been successfully demonstrated by Bioteq Technologies using a sulphur/sulphate reduction process, to recover metals from the pit water. RCDC will treat water from the pit lake annually, or as required, to prevent uncontrolled

discharge from the open pit until water quality within the pit reaches acceptable discharge quality. This will only occur once all the sulphides within the waste rock dump and pit walls have fully oxidized and have released all of their net acidity (expected to be many years in the future, i.e., >100 years). Provision for such a treatment plant has been built into the reclamation costing model for the project and financial security will be put aside to fund the construction and operation of such a plant.

18.6 Capital and Operating Cost Estimates

18.6.1 Capital Cost Estimates

18.6.1.1 General

Merit Consultants International Inc. (Merit) has been contracted by bcMetals Corporation to provide a Bankable Feasibility Study level of capital cost estimate that is considered to be within a 15% accuracy of actual costs.

The total estimated cost to design and build the Red Chris Copper Project described in this report is **\$214,600,000** including an owner provided leased mining fleet and self-performed pre-development. Capital cost estimate details are included in this section while supporting information is included in the Appendices. Total capital, including working capital and initial funding of a surety bond for reclamation funding at mine closure, is estimated at \$227.7 million.

All costs are expressed in fourth quarter 2004 Canadian dollars incorporating a 1.19 Cdn to 1.00 US dollar exchange (84 cents Cdn) where appropriate. There are no allowances for escalation, interest during construction, taxes or duties. All due allowances have been considered for items such as:

- Using actual productivity factors and unit prices from similar ongoing and previous projects
- Understanding of the 'open shop' contracting environment in British Columbia at this time.
- Using quantity take-offs provided by AMEC E & C Services Limited ("the engineer").
- Using capital equipment costs provided by the engineer.
- Using the geotechnical information available to identify borrow sources for the tailings dams.
- Using contractor budget costs for the power line and main access road provided by the Owner.
- Actual current costs of bulk materials such as cement and steel.
- Using experienced construction management supervision during construction.
- Using traditional methods of applying percentages for the costs associated with piping, electrical and instrumentation work.
- Using contractor submitted costs for pre-production mine work provided by the Owner.
- Using Mine fleet sizes and equipment lease costs provided by the Owner.
- Assuming that the backfill for the crusher structure is provided by the mine operations from the pit.

- Using building costs provided by reputable suppliers.
- Using quoted costs for a leased camp.
- Assuming that the explosives supplier provides the above ground facilities.
- Using the configuration of the reclaim water and surplus water system derived by the Owner.
- Using Owner's Costs established by the Owner.
- Using Engineering and Procurement costs provided by the engineer from deliverable take-offs.
- Using estimated Construction Management costs based on experience.
- Assuming a spares allowance of 5% of the equipment costs.
- Assuming a freight cost of 7% of the cost of the equipment and materials except earthwork and concrete materials.
- Understanding of remoteness of project.
- Concrete is batched on site and suitable aggregate is available locally.
- Using a contingency of 25% for the earthworks, 5% for the capital equipment and 10% for all other items including labour.

The estimate covers the direct field costs of executing this project, plus the indirect costs associated with design, procurement and construction efforts. Construction Management will be out-sourced to a competent company that specializes in such work.

18.6.1.2 Basis of Estimate

The capital cost estimate is based on the following project data:

- design criteria
- flow sheets
- general arrangement drawings
- single line electrical drawings
- equipment list
- budget quotations from vendors for major capital equipment and buildings
- quotations from BC contractors
- regional climatic data
- in-house database
- Pre-production mining undertaken by the Owner.

For the purpose of this estimate, the Owner has undertaken to carry out the pre-production work themselves using a leased fleet of equipment and new truck shop. The owner will provide the fuel storage facilities and all associated mining components.

The level of estimating accuracy and the contingency value have been based on the following premises:

- Labour rate is current and based on the combination of three British Columbian industrial contractors conversant with this kind of work.
- Confirmation by qualified contractors that the labour rate is currently valid.
- Main Access Road and Powerline work has been priced by competent contractors.
- Vendor quotations have been received for all major equipment, generally from more than one vendor. The lowest qualified vendor price has been incorporated in the estimate.
- Productivity factors are based on actual experience.
- Material costs were obtained for the work.
- Geotechnical information is available in areas of greatest earthwork risk.
- AMEC (including AEE – AMEC Earth and Environmental) were responsible for providing all bulk material quantities for the processing and infrastructure facilities.

18.6.1.3 Direct Costs

18.6.1.3.1 Quantities

The engineer provided material take-offs based project feasibility drawings other than for plant piping, electrical and instrumentation quantities which were based on percentages of mechanical equipment in each area.

Civil - All earthworks quantities – including mine pre-production stripping - have been calculated in-situ with no allowance for bulking or compaction of materials.

Concrete - Concrete quantities were determined from feasibility stage drawings and experience from previous projects of a very similar nature. The unit rates have been based on the assumption that local aggregates are available with an onsite batch plant.

Formwork - Formwork was estimated for each type of concrete classification and includes supply, form oil, accessories and shoring, and stripping. No allowances have been made for re-use of forms although this will happen and improve the concrete prices during construction.

Reinforcing Steel - Reinforcing steel was calculated based on estimated weight per cubic metre of concrete for each type of classification based on projects of a very similar nature. The unit price includes the supply of material, cutting and bending on site, accessories and installation.

Embedded Metal and Anchor Bolts - The unit price includes supply and installation of carbon steel material including sleeves and anchors.

Structural Steel - Quantities were determined from feasibility stage drawings and experience from previous projects of a very similar nature. The weights shown include allowances for connections and base plates. The unit price includes steel purchase, detailing, fabrication and erection labour.

Mechanical Equipment - Most large capital equipment is assumed to have to be imported, and was itemized and priced in accordance with the flow sheets. Budget quotations were obtained for all major items based on preliminary specifications.

Mechanical (Plate work and Tanks) - Plate work weights were calculated with allowances made for any necessary stiffeners, weirs, launders, etc. The unit prices include purchase, detailing, fabrication and installation.

Piping - Factored allowances - as used on other projects developed by the engineer, have been used for in-plant pipe and calculations for overland pipe, and includes supply (not freight), shop and field fabrication, and installation with allowances for all hangers and supports.

Electrical and Instrumentation - Allowances were based on one-line diagrams and connected loads detail provided on the flow sheets. Major electrical equipment has been assumed to be imported and prices were based on quotations. Bulk material prices were based on factored allowances provided by the engineer from other similar projects, and includes material supply (not freight) and installation. Lengths for overhead lines and high voltage cable were estimated from the overall site plan.

18.6.1.3.2 Direct Field Labour

Labour rates have been derived as a composite for a 60 hours per week, 6 days per week schedule including travel. The \$62 per hour average wage rate is a combined rate for an estimated crew mix (supervision, skilled/unskilled labour) using the rates from three industrial construction contractors experienced as general contractors in the mining industry in British Columbia. The rate includes:

- base labour wage rate
- government (payroll) burdens
- benefits including medical
- small tools and consumables allowance
- home office overheads
- contractors' profit

18.6.1.3.3 Direct Field Materials

Bulk materials will be provided from centers in Canada, primarily Vancouver and Edmonton. Freight is included as a separate indirect cost line item.

18.6.1.4 Indirect Costs

18.6.1.4.1 Temporary Construction Facilities and Services

Contractors' field distributable costs allowed for in the capital cost estimate but not included in the built-up labour rate or owner's costs are as follows:

- contractors' mobilization and demobilization
- contractors' equipment
- construction field offices, furnishings, equipment
- contractor accommodations
- temporary power supply
- temporary water supply
- temporary heating and hoarding
- warehouse and laydown costs
- temporary toilets
- temporary communications
- first aid personnel and supplies
- on-going and final clean-up
- yard maintenance
- janitorial services
- site safety personnel and training
- material testing

These indirect costs are not applicable to the major earthmoving costs, since unit prices submitted by contractors are "all-in" rates, inclusive of all direct and indirect costs.

18.6.1.4.2 Construction Equipment

It is expected that all construction equipment such as cranes, man lifts, welding machines, generators, will be leased and managed by the Owner's Construction Management group. An allowance for construction equipment has been made in the indirect accounts except for the earthworks and concrete scope where equipment is included in the contractor's unit rates with the exception of the concrete pump. Specifically, concrete supply unit prices include supply and operation of the site aggregate crushing, screening and batch plant as well as the concrete transit trucks.

18.6.1.4.3 Construction Accommodation and Catering

An allowance is included for the construction and operation of a 500 man camp based on a lease cost quoted by a reputable vendor. The camp kitchen is sized for the estimated peak man power. 42 man bunkhouses will be installed as required when the work force increases, and demobilized as it decreases at the tail end of the project. The owner has elected, for the purposes of this study, to retain a portion of the camp on a lease basis, and use it for operations. Costs for catering services have been based on quotations.

18.6.1.4.4 First Fill and Spare Parts

An allowance of 5% of the equipment purchase value has been included for spares, and an allowance commensurate with industry standards has been made for the purchase of start-up grinding media and reagents.

18.6.1.4.5 Start-up and Commissioning

An allowance has been made for retention of vendor representatives for start-up, as well as a selection of ten people from the contractor's crews and the construction management staff for a period of about 30 days.

18.6.1.4.6 Freight

A freight allowance of 7% of the material and equipment costs has been included for all materials and equipment associated with purchases.

18.6.1.4.7 Owner's Costs

Owner's costs have been developed in conjunction with the Owner and includes items such as:

- All operating costs for the three-month pre-production period.
- Insurances.
- Head Office staff assigned to the project.
- Owners allowances for field operations offices and supplies.
- Owner's travel costs during construction.

18.6.1.4.8 Engineering, Procurement and Construction Management (EPCM)

Engineering, procurement and construction management costs for the project, as described in the project description, are based on information from previous similar projects for the engineering and procurement phase separately from the construction management phase of the work. The engineering and procurement costs were derived by AMEC. The construction management costs were derived by Merit.

18.6.1.4.9 Taxes and Duties

No taxes have been included, and it is assumed that much of it is available as a refund through recent initiatives enacted by the provincial government.

18.6.1.5 Contingency

The contingency amount is an allowance added to the capital cost estimate to cover unforeseeable costs within the scope of the estimate. This can arise due to presently undefined

items of work or equipment, or to the uncertainty in the estimated quantities and unit prices for labour, equipment and materials. Contingency does not cover scope changes, project exclusions or project execution strategy changes.

Contingencies are included separately for the civil works, equipment and materials and the labour. Earthwork is considered to have the greatest risk and an allowance of 25% of the calculated costs has been assigned as a consequence.

Additional contingencies include 5% on the capital equipment, and 10% allowed on the remainder of the direct and indirect estimates.

18.6.1.6 Exchange Rates

All estimates herein are quoted in fourth quarter 2004 Canadian dollars. The CDN\$: US\$ exchange rate is $\text{CDN}\$1.19 = \text{US}\1.00 .

18.6.1.7 Assumptions

The following assumptions were made in preparing the estimate:

- Construction work is based on unit and fixed price contracts.
- Concrete aggregate and suitable backfill material are available locally and good potential areas were identified during the project team site visits.
- Actual soil bearing conditions will be adequate for the requirements of foundations included in the estimate. This has been confirmed through the surficial geology assessment, including drilling and test pitting, made by the Owner based on their intimate knowledge of the area.
- Construction activities will be carried out in a continuous program with no allowance for demobilization and remobilization.
- The tailings area contains suitable and adequate borrow sources for the dams to be constructed.

18.6.1.8 Capital Cost Exclusions

The following are excluded from the \$214.6 million project capital estimate although some items, e.g. working capital, reclamation bond funding are included in its Total Project Cost estimation.

- Escalation
- Scope changes
- Interest during construction
- Schedule delays and associated costs such as those caused by:
 - scope change
 - unidentified ground conditions

- extraordinary climatic events
- labour disputes
- permit applications
- receipt of information beyond the control of EPCM contractors
- Cost of financing
- Property costs and taxes
- Sunk costs
- Taxes
- Sustaining capital
- Permitting costs

18.6.1.9 Working Capital

Working capital of 1.5 months of Year 1 estimated mine site operating costs (\$9.0 million) has been included in the initial funding requirements. In Year 2 an additional 0.5 months (\$3.0 million) has been budgeted under sustaining capital requirements.

18.6.1.10 Reclamation Bond Funding

Initial capital includes a provision of \$1.4 million to fund a surety bond to meet estimated funding requirements for mine closure reclamation at the end of mine life currently estimated to be in Year 25. The reclamation funding plan also assumes that funds received from the salvage of the mine equipment, process plant and other infrastructure will also be applied to reclamation funding.

18.6.1.11 Sustaining Capital

Over the mine life it is estimated that an additional \$138.3 in sustaining capital will be required. This covers such items as:

- Additional and replacement mine equipment
- Other capital requirements, including construction of the centre till core of the tailings dam, expected to be performed by a contractor.

The major component of this sustaining capital estimate is for additional and replacement mine equipment as detailed in Section 18.2.

Table 41 - Capital Cost Estimate

	Total Hours	Total Labour Cost	Total Materials Cost	Total Equipment Cost	Total Sub-Cont. Cost	Total Other Costs	Total Cost
Mining	14,234	\$882,529	\$78,000	\$6,996,965	\$3,177,955		\$11,135,449
Processing, Tailings & Ancillaries	495,769	\$30,737,411	\$37,101,698	\$55,523,681	\$20,450,709	\$1,202,500	\$145,015,999
Owner's Costs						\$4,636,098	\$4,636,098
Indirects	8,000	\$434,000			\$150,000	\$33,169,153	\$33,753,153
Contingency						\$20,076,361	\$20,076,361
Total Direct & Indirect	517,999	\$32,053,940	\$37,179,698	\$62,520,646	\$23,778,664	\$59,084,112	\$214,617,060

18.6.2 Operating Cost Estimates

Data for this estimate was based on actual quotations, current regional salary rates and experience with similar operations. The cost summary in Table 42 is a comparison over the life of the operation.

Table 42 - Operating Cost Estimates

		\$/tonne Milled					
		Year 1	Year 5	Year 10	Year 17	Year 20	Life-of-Mine
G & A	G&A Labour	0.21	0.21	0.21	0.21	0.17	0.20
	Direct	<u>0.50</u>	<u>0.52</u>	<u>0.52</u>	<u>0.50</u>	<u>0.35</u>	<u>0.46</u>
	Sub-total	0.71	0.73	0.73	0.70	0.52	0.66
Mining	Sub-total	2.75	3.64	3.32	1.47	0.36	2.16
Processing	Process Labour	0.52	0.52	0.52	0.52	0.52	0.52
	Consumables	0.94	0.94	0.94	0.94	0.94	0.94
	Power	0.95	0.95	0.95	0.95	0.95	0.95
	Parts & Supplies	<u>0.47</u>	<u>0.47</u>	<u>0.47</u>	<u>0.47</u>	<u>0.47</u>	<u>0.47</u>
	Sub-total	2.88	2.88	2.88	2.88	2.88	2.88
Total Minesite		6.34	7.25	6.93	5.05	3.76	5.70
Mining only	\$/tonne moved	0.70	0.89	0.93	1.38	0.36	0.84

18.6.2.1 General and Administrative Costs

These costs include administrative and senior management, employee transportation, catering and housekeeping, taxes and insurance, telecommunications, safety and first aid, employee training, computer system maintenance, public relations, warehouse freight and expediting, and environmental. Some of the key details are as follows:

- Employee transportation is based on charter flights from several centers in the province to Dease Lake, followed by bussing to the minesite.
- All employees will work a rotational schedule of two weeks on site followed by two weeks off site.
- A full time camp, catering and housekeeping facility provided at no charge to employees.
- G&A staff in the first year will total 28 people, reducing to 22 in year 18 when pit operations cease.

The estimated direct manning levels for the operation are summarized in Table 43. It does not include personnel for catering, housekeeping, concentrate haulage or the explosives plant, which are estimated to add an additional 50 to 60 jobs.

Table 43 - Manning Levels

	Personnel On-Site					
	Year 1	Year 5	Year 10	Year 17	Year 20	Year 24
Administration	24	24	24	24	18	18
Mining	133	172	161	99	29	28
Processing	77	77	77	75	75	63
Total Property	234	273	262	198	122	109

18.6.2.2 Process Plant

The process plant operating costs cover all unit operations, including fresh, potable and process water systems.

These costs include labour, consumables, power and maintenance with budgetary figures from representative vendors.

Power is based on \$0.02787 per kW-hr plus a demand charge of 40,000 maximum kVA at \$4.73/kVA as supplied from the main BC Hydro grid with the mine tie-in at Tatogga.

Total manpower for the plant, including supervision, technical, operations and maintenance is estimated at 77 people.

18.6.2.3 Mine

The mine operating costs per tonne of material moved and per tonne of ore have been assembled from labour, consumables and maintenance for drilling & blasting, loading, material haulage, stockpiling, and property road maintenance.

The open pit will operate for approximately 17 years. Ore will be crushed while low grade and waste will be stockpiled separately. Following exhaustion of the pit resources, the mine operation will be scaled down to re-handling of the stockpiled material only, for the final 8 years.

Total manpower for the mine operation, including supervision, technical, operations and maintenance will vary from 133 in year 1 to a high of 181 in year 7 and range from 24 – 28 in

the final years of stockpile re-handle.

18.6.2.4 Realization Costs

The realization costs used in the cash flow model total US\$0.35/lb Cu and include the following:

Trucking to port, port storage, ship loading, draft survey, umpire sampling, ocean freight, treatment charge, refining charge, price participation, losses, and insurance.

18.7 ECONOMIC ANALYSIS

The **Red Chris** Project has been valued using a discounted cash flow approach. This method of valuation requires projecting yearly cash inflows, or revenues, and subtracting yearly cash outflows such as operating costs, capital costs, royalties, and taxes. The resulting net annual cash flows are discounted back to the date of valuation and totaled to determine Net Present Values (NPVs) at the selected discount rates. The Internal Rate of Return (IRR) is calculated as the discount rate that yields a zero NPV.

Section 18.7.1 summarizes the input data used in the base case cash flow analysis. The results of the base case and sensitivity economic analysis are then presented in Section 18.7.2. This analysis includes sensitivities to variation in copper and gold prices, head grades, metallurgical recoveries, operating costs, capital costs, concentrate transportation costs, and smelting and refining charges.

All amounts are presented in Canadian dollars unless otherwise specified.

18.7.1 Basis of Economic Analysis

18.7.1.1 Ore Reserves and Mine Life

The base case economic analysis includes the following life of mine ore reserves:

Total ore milled	276.0	million tonnes
Average copper grade	0.349	% Cu
Average gold grade	0.266	g/t Au
Stripping ratio	1.1:1	after reprocessing stockpiled material after year 17

These reserves will be processed at a rate of 30,000 tonnes/day over a planned mine life of approximately 17 years. Stock-piled material would be processed in years 18 to 25.

18.7.1.2 Metallurgical Balance

The base case metallurgical recoveries and copper concentrate grades for all project years are:

Copper recovery	87.2%
Gold recovery	50.3%
Copper concentrate grade	27.0%
The calculated gold concentrate grade varies by year, and averages	11.7 gr/t

Based on a review of available geologic and metallurgical data, an average silver concentrate grade of 50.0 grams/tonne has been used for the base case.

18.7.1.3 Smelter Terms

The base case incorporates the following smelter terms:

Copper concentrate pay factor	96.5% or minimum 1.0 unit deduction
Gold pay factor	97%
Silver pay factor	90%
Copper refining charge	US\$ 0.07/payable lb
Gold refining charge	US\$ 5.50/payable troy oz
Silver refining charge	US\$ 0.35/oz
Base treatment charge	US\$ 70/DMT
+ price participation	10% over US\$ 0.90/lb
Impurity penalties:	
Mercury	US\$ 0.20/ppm \geq 10ppm
Antimony	US\$ 0.50/0.01% \geq 0.05%

18.7.1.4 Concentrate Transportation Costs

The base case considers concentrate transportation charges such as:

- Truck from mine to Stewart
- Port storage and loading charges
- Ship draft surveys
- Umpire sampling
- Ocean freight
- Losses
- Insurance

Aggregate rate used representing above charges CDN \$92.97/WMT

Wet concentrate tonnages estimates are based on a moisture content of 8.0%.

18.7.1.5 Metal Prices

The following base case metal prices have been used:

Copper	US\$ 1.10/lb.
Gold	US\$ 375/troy oz.
Silver	US\$ 5.50/troy oz.

18.7.1.6 Exchange Rate

An exchange rate of CDN\$ 1.00 = US\$ 0.75 has been used in the base case economic analysis.

18.7.1.7 Operating Costs

The operating costs are summarized below:

	LOM	Years 1-5
Mining (per tonne milled)	\$ 1.76	\$ 3.19
Process (per tonne milled)	\$ 2.88	\$ 2.88
G&A (per tonne milled)	\$ 0.66	\$ 0.72
Total (per tonne milled)	\$ 5.30	\$ 6.79
Mining (per tonne mined)	\$ 0.84	\$ 0.79

18.7.1.8 Capital Costs

The estimated project capital costs are summarized as follows:

PRE-PRODUCTION CAPITAL (YEARS -1 & -2)	000s
Mine (equipment and pre-production stripping)	\$ 11,135
Site preparation	5,444
Process plant, tailings & ancillaries	122,044
Utilities	17,529
Indirects	33,753
Contingency	20,076
Owner's costs	4,636
Other equipment	2,550
Reclamation bond	1,428
Working capital	9,136
Subtotal	227,731
SUSTAINING CAPITAL	
Additional working capital	3,045
Sustaining capital	24,261
Lease/purchase, additional and replacement equipment	110,139
Site reclamation	900
Subtotal	138,345
Working capital recovery	(12,181)
NET LIFE OF MINE CAPITAL COSTS	\$ 353,895

18.7.1.9 Royalties

The base case includes \$1million payment to buy-out 0.8% royalty, \$1 million payment to Teck Cominco due in year one and a 1 % NSR royalty payable on the Red Chris production.

18.7.1.10 Taxes

The Red Chris Project will be subject to income and/or revenue taxes as follows:

Canadian federal income Tax	21.84%
British Columbia income tax	13.50%
British Columbia mineral tax	13.00%

British Columbia mineral tax is applied to an amount different than taxable income, as defined for federal and provincial income tax purposes, and is assumed to be deductible in arriving at taxable income. Federal large corporations tax is being phased out and is applied to taxable capital at the rates of 0.175% for 2005, 0.125% for 2006, 0.0625% for 2007 and 0.0% thereafter.

18.7.1.11 Financing

The base case economic analysis has been run on a basis of 100.0% equity.

18.7.1.12 Inflation

The base case economic analysis has been run with no inflation (constant dollar basis). Capital and operating costs are expressed in Fourth Quarter 2004 Canadian dollars.

18.7.1.13 Discounting and Date of Valuation

Estimated annual net cash flows have been discounted to the beginning of Project Year -2 at real discount rates of 5% and 10%.

18.7.2 Results

The results of the base case and sensitivity analysis are summarized in Table 47, and Figures 51 to 53. The estimated base case IRR based on 100.0% equity is 17.5% pre-tax. At metal prices and exchange rates prevailing at the close of business on December 14, 2004 (copper = US\$1.444/lb., gold = US\$435.90/oz., silver = US\$6.718/oz. and 1 CDN\$ = US\$0.80854), the estimated IRR increases to 33.3% pre-tax.

Table 44 - Pre-Tax Sensitivity Analysis

Pre-Tax Sensitivity Analysis Internal Rate of Return (%)														
Cu (US\$/lb.)	Au (US\$/oz.)	Foreign Exchange Rate: 1 CDN\$ = US\$												
		\$ 0.70	\$0.73	\$0.75	\$0.78	\$0.80	\$0.83	\$0.85	\$0.88	\$0.90	\$0.93	\$0.95	\$0.98	\$1.00
0.90	325	6.6	3.3	0.9	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A
0.90	350	8.0	4.8	2.5	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A
0.90	375	9.4	6.2	4.1	0.7	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A
0.90	400	10.8	7.6	5.5	2.3	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A
1.00	350	15.1	12.0	10.0	7.1	5.1	2.1	N/A	N/A	N/A	N/A	N/A	N/A	N/A
1.00	375	16.3	13.2	11.3	8.3	6.4	3.5	1.5	N/A	N/A	N/A	N/A	N/A	N/A
1.00	400	17.5	14.4	12.5	9.6	7.7	4.8	2.9	N/A	N/A	N/A	N/A	N/A	N/A
1.00	425	18.7	15.6	13.7	10.8	8.9	6.1	4.2	1.2	N/A	N/A	N/A	N/A	N/A
1.10	350	21.5	18.4	16.4	13.5	11.7	9.0	7.2	4.5	2.7	N/A	N/A	N/A	N/A
1.10	375	22.7	19.5	17.5	14.7	12.8	10.1	8.3	5.7	3.9	1.1	N/A	N/A	N/A
1.10	400	23.8	20.7	18.7	15.8	13.9	11.2	9.5	6.9	5.1	2.4	0.6	N/A	N/A
1.10	425	25.0	21.8	19.8	16.9	15.0	12.3	10.6	8.0	6.3	3.7	1.9	N/A	N/A
1.20	375	28.8	25.5	23.4	20.5	18.6	15.9	14.1	11.6	10.0	7.5	5.9	3.4	1.8
1.20	400	29.9	26.6	24.5	21.5	19.6	16.9	15.2	12.6	11.0	8.6	7.0	4.6	2.9
1.20	425	31.0	27.6	25.6	22.6	20.7	17.9	16.2	13.7	12.0	9.6	8.0	5.6	4.0
1.20	450	32.1	28.7	26.6	23.6	21.7	19.0	17.2	14.7	13.0	10.6	9.0	6.7	5.1
1.30	400	35.8	32.3	30.1	27.0	25.1	22.3	20.5	17.9	16.3	13.9	12.3	10.1	8.6
1.30	425	36.8	33.4	31.2	28.1	26.1	23.3	21.5	18.9	17.2	14.8	13.3	11.0	9.5
1.30	450	37.9	34.4	32.2	29.0	27.1	24.2	22.4	19.8	18.2	15.8	14.2	11.9	10.5
1.30	475	39.0	35.4	33.2	30.0	28.0	25.2	23.4	20.8	19.1	16.7	15.1	12.9	11.4
1.40	425	42.6	38.9	36.6	33.4	31.3	28.4	26.5	23.9	22.2	19.7	18.1	15.9	14.4
1.40	450	43.6	40.0	37.6	34.4	32.3	29.3	27.5	24.8	23.1	20.6	19.0	16.7	15.3
1.40	475	44.7	41.0	38.6	35.3	33.2	30.3	28.4	25.7	24.0	21.5	19.9	17.6	16.1
1.40	500	45.7	42.0	39.6	36.3	34.2	31.2	29.3	26.6	24.9	22.4	20.8	18.5	17.0
1.50	425	48.3	44.5	42.0	38.6	36.5	33.4	31.5	28.7	27.0	24.4	22.8	20.5	19.0
1.50	450	49.3	45.5	43.0	39.6	37.4	34.3	32.4	29.6	27.8	25.3	23.7	21.3	19.8
1.50	475	50.3	46.4	44.0	40.5	38.4	35.3	33.3	30.5	28.7	26.1	24.5	22.1	20.6
1.50	500	51.4	47.4	45.0	41.5	39.3	36.2	34.2	31.4	29.6	27.0	25.3	23.0	21.5
1.50	600	55.4	51.3	48.8	45.2	42.9	39.7	37.7	34.8	33.0	30.3	28.6	26.2	24.7

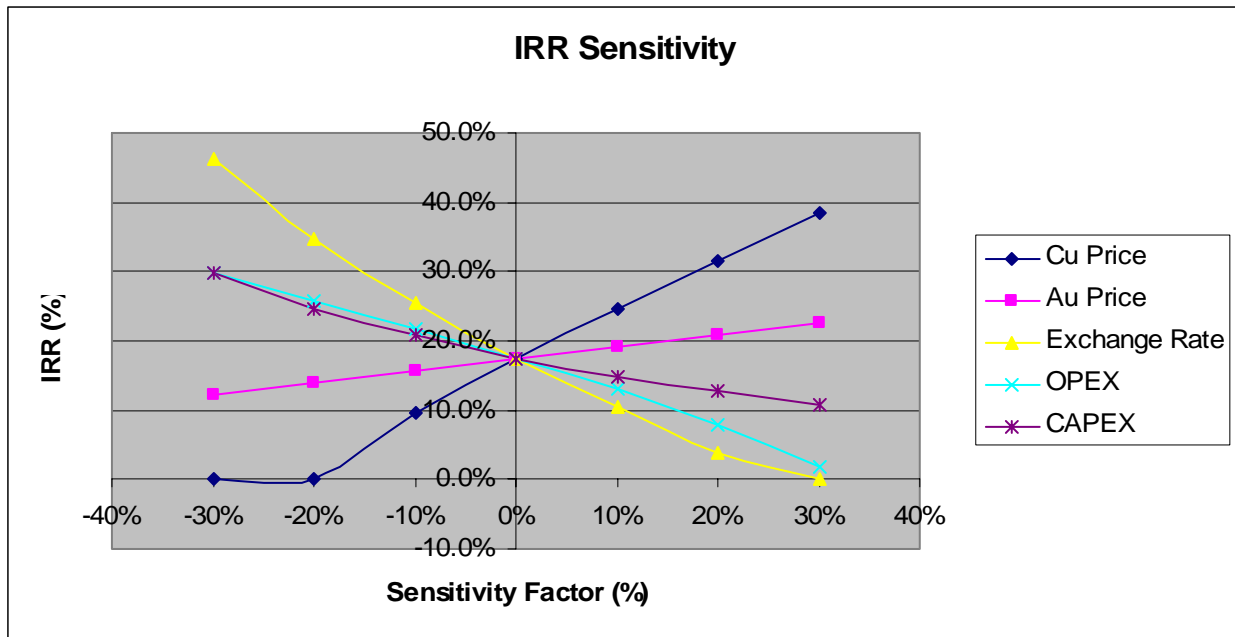
Table 45 - Pre-Tax Sensitivity Analysis - Net Present Value - 5% Discount

Pre-Tax Sensitivity Analysis														
Net Present Value - 5% Discount (CDN\$ Millions)														
Cu (US\$/lb.)	Au (US\$/oz.)	Foreign Exchange Rate: 1 CDN\$ = US\$												
		\$0.70	\$0.73	\$0.75	\$0.78	\$0.80	\$0.83	\$0.85	\$0.88	\$0.90	\$0.93	\$0.95	\$0.98	\$1.00
0.90	325	25.1	(26.0)	(57.8)	(102.4)	(130.3)	(169.7)	(194.3)	(229.2)	(251.2)	(282.4)	(302.1)	(330.2)	(347.9)
0.90	350	48.6	(3.4)	(35.8)	(81.3)	(109.7)	(149.8)	(175.0)	(210.5)	(232.9)	(264.7)	(284.8)	(313.4)	(331.5)
0.90	375	72.2	19.1	(13.9)	(60.2)	(89.2)	(130.0)	(155.6)	(191.8)	(214.6)	(247.0)	(267.5)	(296.6)	(315.0)
0.90	400	95.7	41.7	8.1	(39.1)	(68.6)	(110.2)	(136.2)	(173.1)	(196.4)	(229.3)	(250.1)	(279.8)	(298.6)
1.00	350	176.1	118.8	83.1	33.1	1.8	(42.3)	(70.0)	(109.2)	(133.8)	(168.8)	(190.9)	(222.4)	(242.3)
1.00	375	199.6	141.3	105.1	54.2	22.3	(22.5)	(50.6)	(90.5)	(115.5)	(151.1)	(173.6)	(205.6)	(225.8)
1.00	400	223.1	163.9	127.0	75.3	42.9	(2.7)	(31.3)	(71.7)	(97.2)	(133.4)	(156.2)	(188.8)	(209.3)
1.00	425	246.6	186.4	149.0	96.4	63.5	17.2	(11.9)	(53.0)	(78.9)	(115.7)	(138.9)	(172.0)	(192.9)
1.10	350	303.5	241.0	202.1	147.4	113.3	65.1	34.9	(7.8)	(34.7)	(72.9)	(97.0)	(131.3)	(153.1)
1.10	375	327.0	263.5	224.0	168.5	133.9	85.0	54.3	10.9	(16.4)	(55.2)	(79.7)	(114.5)	(136.6)
1.10	400	350.6	286.1	246.0	189.6	154.4	104.8	73.7	29.6	1.9	(37.5)	(62.3)	(97.7)	(120.1)
1.10	425	374.1	308.6	267.9	210.7	175.0	124.6	93.0	48.3	20.2	(19.8)	(45.0)	(80.9)	(103.7)
1.20	375	454.5	385.7	343.0	282.9	245.4	192.5	159.3	112.3	82.7	40.8	14.2	(23.5)	(47.4)
1.20	400	478.0	408.3	364.9	304.0	265.9	212.3	178.6	131.0	101.0	58.5	31.6	(6.7)	(30.9)
1.20	425	501.5	430.8	386.9	325.1	286.5	232.1	198.0	149.7	119.3	76.2	48.9	10.1	(14.5)
1.20	450	525.0	453.4	408.8	346.2	307.1	252.0	217.4	168.4	137.6	93.9	66.2	26.9	2.0
1.30	400	605.4	530.5	483.9	418.4	377.5	319.8	283.6	232.4	200.1	154.4	125.5	84.3	58.3
1.30	425	629.0	553.0	505.8	439.5	398.0	339.6	302.9	251.1	218.4	172.1	142.8	101.1	74.7
1.30	450	652.5	575.6	527.8	460.6	418.6	359.4	322.3	269.8	236.7	189.8	160.1	117.9	91.2
1.30	475	676.0	598.1	549.7	481.7	439.2	379.3	341.7	288.5	255.0	207.5	177.5	134.7	107.7
1.40	425	756.4	675.2	624.7	553.9	509.5	447.1	407.9	352.5	317.6	268.0	236.7	192.2	164.0
1.40	450	779.9	697.8	646.7	575.0	530.1	466.9	427.3	371.2	335.8	285.7	254.0	209.0	180.4
1.40	475	803.4	720.3	668.6	596.1	550.7	486.8	446.6	389.9	354.1	303.4	271.4	225.8	196.9
1.40	500	827.0	742.9	690.6	617.2	571.3	506.6	466.0	408.6	372.4	321.1	288.7	242.6	213.3
1.50	425	883.8	797.5	743.7	668.2	621.1	554.6	512.9	453.8	416.7	363.9	330.6	283.2	253.2
1.50	450	907.4	820.0	765.6	689.3	641.6	574.4	532.2	472.5	435.0	381.6	347.9	300.0	269.6
1.50	475	930.9	842.6	787.6	710.4	662.2	594.2	551.6	491.2	453.3	399.3	365.3	316.8	286.1
1.50	500	954.4	865.1	809.5	731.5	682.8	614.1	571.0	510.0	471.5	417.0	382.6	333.6	302.6
1.50	600	1,048.5	955.3	897.3	816.0	765.1	693.4	648.4	584.8	544.7	487.8	451.9	400.8	368.4

Table 46 - Pre-Tax Sensitivity Analysis - Net Present Value - 10% Discount

Pre-Tax Sensitivity Analysis														
Net Present Value - 10% Discount (CDN\$ Millions)														
Cu (US\$/lb.)	Au (US\$/oz.)	Foreign Exchange Rate: 1 CDN\$ = US\$												
		\$0.70	\$0.73	\$0.75	\$0.78	\$0.80	\$0.83	\$0.85	\$0.88	\$0.90	\$0.93	\$0.95	\$0.98	\$1.00
0.90	325	25.1	(26.0)	(57.8)	(102.4)	(130.3)	(169.7)	(194.3)	(229.2)	(251.2)	(282.4)	(302.1)	(330.2)	(347.9)
0.90	350	48.6	(3.4)	(35.8)	(81.3)	(109.7)	(149.8)	(175.0)	(210.5)	(232.9)	(264.7)	(284.8)	(313.4)	(331.5)
0.90	375	72.2	19.1	(13.9)	(60.2)	(89.2)	(130.0)	(155.6)	(191.8)	(214.6)	(247.0)	(267.5)	(296.6)	(315.0)
0.90	400	95.7	41.7	8.1	(39.1)	(68.6)	(110.2)	(136.2)	(173.1)	(196.4)	(229.3)	(250.1)	(279.8)	(298.6)
1.00	350	176.1	118.8	83.1	33.1	1.8	(42.3)	(70.0)	(109.2)	(133.8)	(168.8)	(190.9)	(222.4)	(242.3)
1.00	375	199.6	141.3	105.1	54.2	22.3	(22.5)	(50.6)	(90.5)	(115.5)	(151.1)	(173.6)	(205.6)	(225.8)
1.00	400	223.1	163.9	127.0	75.3	42.9	(2.7)	(31.3)	(71.7)	(97.2)	(133.4)	(156.2)	(188.8)	(209.3)
1.00	425	246.6	186.4	149.0	96.4	63.5	17.2	(11.9)	(53.0)	(78.9)	(115.7)	(138.9)	(172.0)	(192.9)
1.10	350	303.5	241.0	202.1	147.4	113.3	65.1	34.9	(7.8)	(34.7)	(72.9)	(97.0)	(131.3)	(153.1)
1.10	375	327.0	263.5	224.0	168.5	133.9	85.0	54.3	10.9	(16.4)	(55.2)	(79.7)	(114.5)	(136.6)
1.10	400	350.6	286.1	246.0	189.6	154.4	104.8	73.7	29.6	1.9	(37.5)	(62.3)	(97.7)	(120.1)
1.10	425	374.1	308.6	267.9	210.7	175.0	124.6	93.0	48.3	20.2	(19.8)	(45.0)	(80.9)	(103.7)
1.20	375	454.5	385.7	343.0	282.9	245.4	192.5	159.3	112.3	82.7	40.8	14.2	(23.5)	(47.4)
1.20	400	478.0	408.3	364.9	304.0	265.9	212.3	178.6	131.0	101.0	58.5	31.6	(6.7)	(30.9)
1.20	425	501.5	430.8	386.9	325.1	286.5	232.1	198.0	149.7	119.3	76.2	48.9	10.1	(14.5)
1.20	450	525.0	453.4	408.8	346.2	307.1	252.0	217.4	168.4	137.6	93.9	66.2	26.9	2.0
1.30	400	605.4	530.5	483.9	418.4	377.5	319.8	283.6	232.4	200.1	154.4	125.5	84.3	58.3
1.30	425	629.0	553.0	505.8	439.5	398.0	339.6	302.9	251.1	218.4	172.1	142.8	101.1	74.7
1.30	450	652.5	575.6	527.8	460.6	418.6	359.4	322.3	269.8	236.7	189.8	160.1	117.9	91.2
1.30	475	676.0	598.1	549.7	481.7	439.2	379.3	341.7	288.5	255.0	207.5	177.5	134.7	107.7
1.40	425	756.4	675.2	624.7	553.9	509.5	447.1	407.9	352.5	317.6	268.0	236.7	192.2	164.0
1.40	450	779.9	697.8	646.7	575.0	530.1	466.9	427.3	371.2	335.8	285.7	254.0	209.0	180.4
1.40	475	803.4	720.3	668.6	596.1	550.7	486.8	446.6	389.9	354.1	303.4	271.4	225.8	196.9
1.40	500	827.0	742.9	690.6	617.2	571.3	506.6	466.0	408.6	372.4	321.1	288.7	242.6	213.3
1.50	425	883.8	797.5	743.7	668.2	621.1	554.6	512.9	453.8	416.7	363.9	330.6	283.2	253.2
1.50	450	907.4	820.0	765.6	689.3	641.6	574.4	532.2	472.5	435.0	381.6	347.9	300.0	269.6
1.50	475	930.9	842.6	787.6	710.4	662.2	594.2	551.6	491.2	453.3	399.3	365.3	316.8	286.1
1.50	500	954.4	865.1	809.5	731.5	682.8	614.1	571.0	510.0	471.5	417.0	382.6	333.6	302.6
1.50	600	1,048.5	955.3	897.3	816.0	765.1	693.4	648.4	584.8	544.7	487.8	451.9	400.8	368.4

Figure 51 - IRR Sensitivity



Base case assumptions

Copper price	US\$1.10/lb.
Gold price	US\$375.00/oz.
Silver price	US\$5.50/oz.
Foreign exchange	US\$0.75 = 1 CDN\$

Figure 52 - NPV 5% Sensitivity

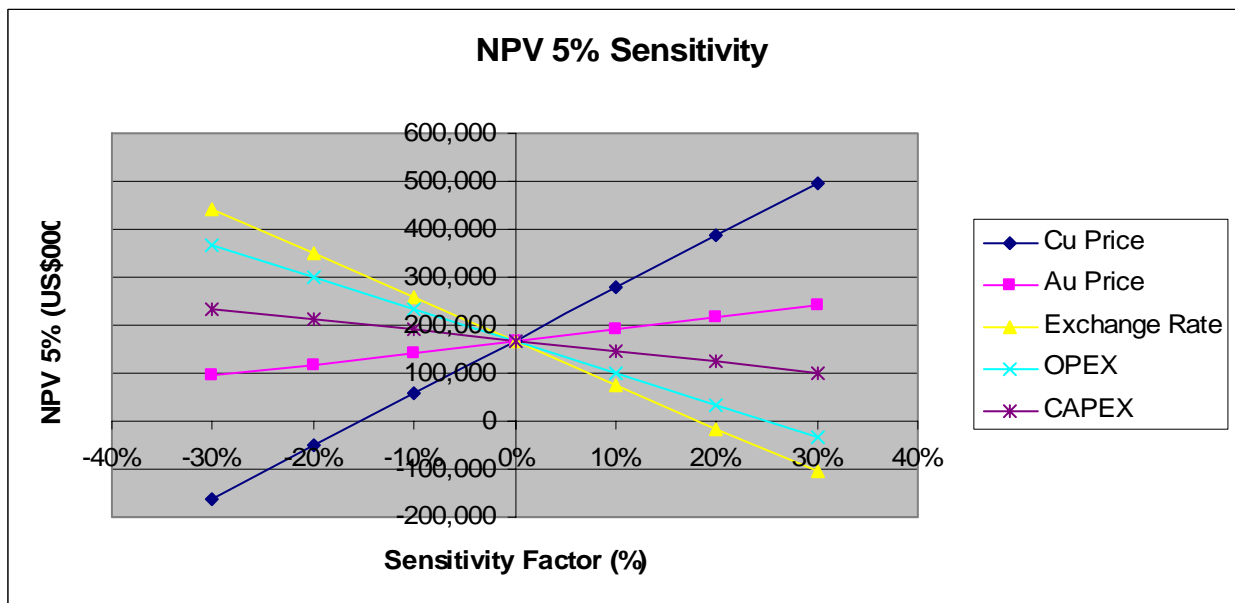
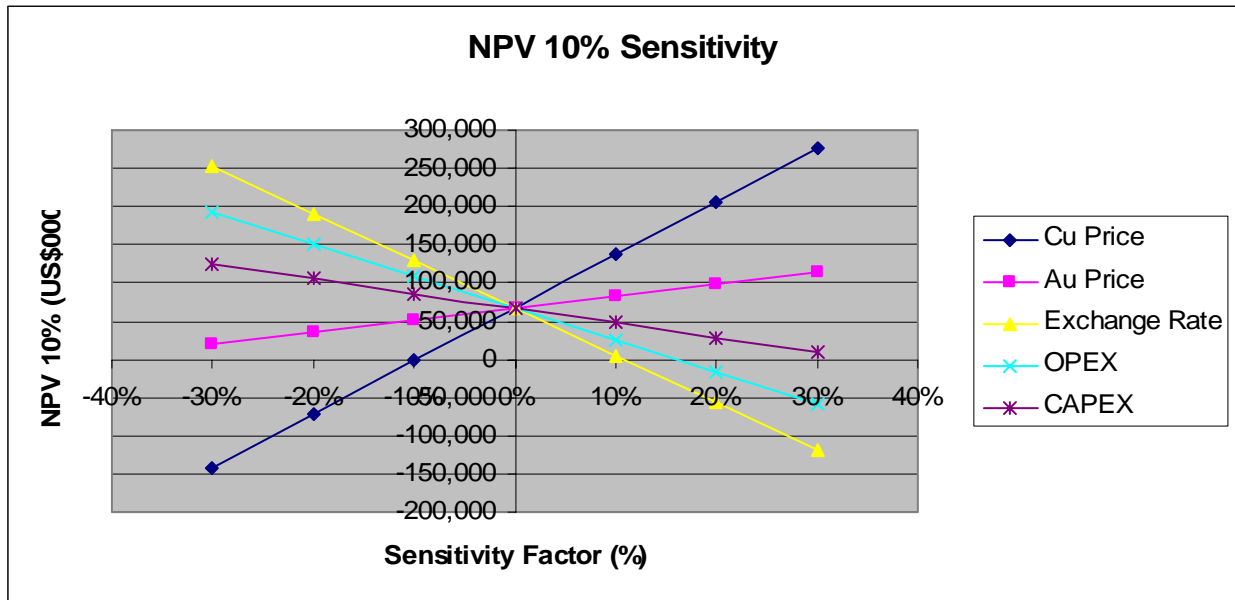


Figure 53 - NPV 10% Sensitivity



Base case assumptions

Copper price	US\$1.10/lb.
Gold price	US\$375.00/oz.
Silver price	US\$5.50/oz.
Foreign exchange	US\$0.75 = 1 CDN\$

Table 47 - Summary of Base Case Input Parameters and Results

Proven and Probable Reserves

- 185.4 Mt @ 0.414% Cu and 0.325 gpt Au, excluding processing of stock-piled material
- 276.0 Mt @ 0.349% Cu and 0.266 gpt Au, including processing of stock-piled material
- 2.123 billion pounds of Cu and 2.362 million troy ounces of Au in situ
- 1.85 billion lb. of Cu and 1.19 million ounces of Au contained in concentrate
- 1.1:1 stripping ratio, after reprocessing of stock-piled material

Production Rate and Mine Life

- 30,000 tonnes/day concentrator
- 10.95 mtpy ore
- 17 year mine life, plus 8 years of reprocessing stock-piled material

Years 1-5

- 54.75 Mt @
0.49% Cu, 523.9 million pounds recovered to concentrate (104.8 million lbs/yr Cu)
0.358 grams/tonne Au, 358,023 troy ounces recovered to concentrate (71,600 oz/yr Au)

Concentrate and Metal Production (LOM)

- 124,500 dmt/year copper concentrate production (176,000 dmt/year annual average for Years 1-5)
- 27% copper concentrate grade @ 87.2% recovery
- 11.7 grams/tonne Au @ 50.3% recovery
- 50.0 grams/tonne Ag based on metallurgical testwork
- 74.1 million lbs/year copper
- 47,500 oz/year gold

Capital Cost

- Initial capital, including phase one working capital \$227.6 million
- Sustaining and additional working and other capital \$138.3 million

LOM Operating Cost

- Mining \$1.76/tonne milled
- Processing \$2.88/tonne milled
- G&A \$0.66/tonne milled
- Total \$5.30/tonne milled

Metal Prices (US\$)

- \$1.10/lb Cu
- \$375/oz Au
- \$5.50/oz Ag

Average Operating Margin (\$/tonne ore)

	LOM	Years 1-5
• NSR	\$ 9.15	\$13.20
• Operating cost	\$ 5.30	\$ 6.79
• Head office	\$ 0.02	\$ 0.02
• Royalties	\$ 0.09	\$ 0.17
• Margin	\$ 3.14	\$ 5.96

Pre-tax Cash Flows

• Years 1-5	\$256,205
• Years 6-25	\$487,612

Pre-tax Rate of Return and Payback

• IRR (100% equity basis)	17.5%
• Payback (years, 100% equity basis)	4.6

Pre-tax Net Present Value

• 5% discount (000)	\$224,968
• 10% discount (000)	\$90,473

19.0 OTHER RELEVANT DATA & INFORMATION

19.1 Classification

19.1.1 Introduction

Based on the study herein reported, delineated mineralization of the Red Chris Project is classified as a resource according to the following definition from National Instrument 43-101:

“In this Instrument, the terms “mineral resource”, “inferred mineral resource”, “indicated mineral resource” and “measured mineral resource” have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by CIM Council on August 20, 2000, as those definitions may be amended from time to time by the Canadian Institute of Mining, Metallurgy, and Petroleum.”

*“A **Mineral Resource** is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for*

economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.”

The terms Measured, Indicated and Inferred are defined in 43-101 as follows:

*“A '**Measured Mineral Resource**' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.”*

*“An '**Indicated Mineral Resource**' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.”*

*“An '**Inferred Mineral Resource**' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.”*

19.1.2 Results

Each block at Red-Chris was classified as measured, indicated, or inferred based on geologic continuity and the proximity of drill data. The procedure for classification was as follows. For each block estimated by kriging, an estimation error was calculated for both copper and gold. These estimation errors are based on the nugget effect, sill value, number of composites used in the estimate, and where the composites are located relative to any anisotropy present. A paper by Garston Blackwell, (Blackwell, 1999) discusses the merits for such a classification method. A relative estimation error ('RKSD') was calculated for each block as follows:

$$\text{RKSD Cu} = (\text{Kr. Est. Error Cu} / \text{Kr. Cu grade})$$

$$\text{RKSD Au} = (\text{Kr. Est. Error Au} / \text{Kr. Au grade})$$

Determining the limits of relative standard errors from which to assign the various classification categories is somewhat subjective. Blackwell suggests the arbitrary limits of 0.3 and 0.5 for porphyry copper and epithermal gold deposits. For the two mineralized zones at Red Chris various level were tried and the blocks were classed as follows:

Main Zone

- Measured - blocks with copper or gold relative estimation errors < 0.30
- Indicated - blocks with copper or gold relative estimation errors less than 0.60 and not classified as measured
- Inferred - all other blocks estimated

East Zone

- Measured - blocks with copper or gold relative estimation errors < 0.35
- Indicated - blocks with copper or gold relative estimation errors less than 0.70 and not classified as measured
- Inferred - all other blocks estimated

Within the Far West and Gully zone there is insufficient geological control at this time to establish geologic continuity. As a result all blocks were classed as inferred. Results are presented in grade-tonnage tables for the entire deposit sorted by classification in Tables 48 and 49. The inferred resource calculated for the Far West and Gully Zone are shown in Tables 50 and 51 respectively.

Table 48 - Red Chris All Zones all blocks classed Measured or Indicated

Red Chris All Zones all blocks classed Measured or Indicated

All Blocks Classed Measured				All Blocks Classed Indicated		
Cutoff (Cu %)	Tonnes > Cutoff (tonnes)	Grade>Cutoff		Tonnes > Cutoff (tonnes)	Grade>Cutoff	
		Cu (%)	Au (g/t)		Cu (%)	Au (g/t)
0.05	172,100,000	0.33	0.27	761,200,000	0.21	0.18
0.10	149,600,000	0.37	0.30	589,600,000	0.25	0.21
0.15	129,300,000	0.41	0.33	459,300,000	0.29	0.24
0.20	109,800,000	0.45	0.36	336,400,000	0.33	0.27
0.25	91,600,000	0.49	0.40	233,500,000	0.38	0.31
0.30	74,200,000	0.54	0.44	164,200,000	0.42	0.34
0.35	58,700,000	0.60	0.49	110,000,000	0.47	0.38
0.40	47,200,000	0.66	0.54	72,200,000	0.52	0.41
0.45	38,200,000	0.71	0.60	46,800,000	0.58	0.46
0.50	31,200,000	0.77	0.66	31,200,000	0.63	0.51
0.55	25,900,000	0.82	0.72	21,000,000	0.68	0.57
0.60	21,100,000	0.87	0.78	14,000,000	0.74	0.64
0.65	16,700,000	0.94	0.86	9,500,000	0.79	0.70
0.70	13,600,000	1.00	0.94	6,400,000	0.85	0.78

0.75	11,100,000	1.06	1.01	4,600,000	0.90	0.85
0.80	9,500,000	1.11	1.07	3,400,000	0.94	0.91
0.85	8,100,000	1.16	1.12	2,300,000	0.99	0.97
0.90	6,700,000	1.22	1.18	1,500,000	1.06	1.04
0.95	5,600,000	1.28	1.25	1,100,000	1.11	1.13
1.00	4,800,000	1.33	1.30	700,000	1.19	1.21
1.10	3,800,000	1.40	1.38	274,000	1.40	1.45
1.20	2,800,000	1.49	1.44	172,000	1.54	1.59
1.30	2,100,000	1.57	1.52	137,000	1.62	1.67
1.40	1,500,000	1.66	1.62	120,000	1.65	1.68
1.50	1,100,000	1.73	1.67	120,000	1.65	1.68

Table 49 - All Zones All Blocks Classed Measured Plus Indicated or Inferred

Red Chris All Zones all blocks classed Measured plus Indicated or Inferred

All Blocks Classed Measured plus Indicated				All Blocks Classed Inferred		
Cutoff (Cu %)	Tonnes > Cutoff (tonnes)	Grade>Cutoff		Tonnes > Cutoff (tonnes)	Grade>Cutoff	
		Cu (%)	Au (g/t)		Cu (%)	Au (g/t)
0.05	933,200,000	0.23	0.20	565,000,000	0.21	0.19
0.10	739,200,000	0.28	0.23	452,000,000	0.24	0.22
0.15	588,600,000	0.32	0.26	360,200,000	0.27	0.24
0.20	446,100,000	0.36	0.29	268,700,000	0.30	0.27
0.25	325,100,000	0.41	0.33	193,400,000	0.34	0.29
0.30	238,300,000	0.46	0.37	126,100,000	0.37	0.31
0.35	168,700,000	0.52	0.42	67,100,000	0.41	0.33
0.40	119,400,000	0.58	0.47	27,500,000	0.46	0.32
0.45	85,000,000	0.64	0.52	10,300,000	0.52	0.31
0.50	62,400,000	0.70	0.59	5,100,000	0.57	0.34
0.55	46,900,000	0.76	0.65	3,800,000	0.60	0.35
0.60	35,100,000	0.82	0.72	1,962,000	0.61	0.33
0.65	26,200,000	0.89	0.80	33,000	0.69	0.34
0.70	20,000,000	0.95	0.88	17,000	0.72	0.29
0.75	15,800,000	1.01	0.96			
0.80	12,900,000	1.07	1.02			
0.85	10,400,000	1.12	1.09			
0.90	8,200,000	1.19	1.16			
0.95	6,700,000	1.25	1.23			
1.00	5,500,000	1.31	1.29			
1.10	4,000,000	1.40	1.39			
1.20	3,000,000	1.49	1.45			
1.30	2,300,000	1.57	1.53			
1.40	1,600,000	1.66	1.63			
1.50	1,200,000	1.73	1.67			

Table 50 - Far West Zone - All Blocks Classed Inferred

Red Chris Far West Zone - All Blocks Classed Inferred			
Cutoff (Cu %)	Tonnes > Cutoff (tonnes)	Grade>Cutoff	
		Cu (%)	Au (g/t)
0.05	141,900,000	0.13	0.26
0.10	76,800,000	0.17	0.33
0.15	38,400,000	0.22	0.38
0.20	18,800,000	0.26	0.43
0.25	9,800,000	0.30	0.49
0.30	3,100,000	0.35	0.58
0.35	1,100,000	0.42	0.63
0.40	600,000	0.46	0.67
0.45	300,000	0.49	0.72
0.50	100,000	0.52	0.67
0.55	17,000	0.55	1.08

Table 51 - Red Chris Gully Zone - All Blocks Classed Inferred

Red Chris Gully Zone - All Blocks Classed Inferred			
Cutoff (Cu %)	Tonnes > Cutoff (tonnes)	Grade>Cutoff	
		Cu (%)	Au (g/t)
0.05	391,600,000	0.16	0.17
0.10	230,300,000	0.22	0.20
0.15	145,700,000	0.28	0.24
0.20	97,200,000	0.33	0.27
0.25	69,600,000	0.37	0.29
0.30	50,800,000	0.41	0.31
0.35	37,200,000	0.44	0.33
0.40	24,300,000	0.48	0.35
0.45	14,800,000	0.51	0.36
0.50	7,800,000	0.55	0.39
0.55	2,629,000	0.61	0.45
0.60	1,200,000	0.65	0.52
0.65	500,000	0.70	0.61
0.70	200,000	0.73	0.73
0.75	50,000	0.80	0.89
0.80	17,000	0.87	1.05

Results are presented in grade-tonnage tables for the Main Zone, west of 50,650 E, sorted by classification in Tables 52 and 53. Examples of how the classified blocks relate to existing drill holes are shown as Figures 54 and 55 for the Main Zone Section 50,010 E and 50,050 E respectively. Blocks are colour coded by classification type and drill holes \pm 25 m on either side of the section are shown as solid lines.

Table 52 - Main Zone All Blocks Classed Measured or Indicated

Red Chris Main Zone all blocks classed Measured or Indicated

All Blocks Classed Measured				All Blocks Classed Indicated		
Cutoff (Cu %)	Tonnes > Cutoff (tonnes)	Grade>Cutoff		Tonnes > Cutoff (tonnes)	Grade>Cutoff	
		Cu (%)	Au (g/t)		Cu (%)	Au (g/t)
0.05	117,400,000	0.305	0.224	576,600,000	0.216	0.175
0.10	103,400,000	0.336	0.246	459,900,000	0.252	0.200
0.15	90,800,000	0.366	0.267	365,900,000	0.284	0.223
0.20	77,000,000	0.400	0.294	270,700,000	0.323	0.253
0.25	62,700,000	0.440	0.325	188,100,000	0.367	0.287
0.30	49,100,000	0.486	0.362	129,800,000	0.409	0.316
0.35	37,700,000	0.535	0.400	84,400,000	0.455	0.347
0.40	29,400,000	0.581	0.438	52,300,000	0.506	0.370
0.45	22,800,000	0.627	0.482	32,000,000	0.559	0.401
0.50	17,700,000	0.670	0.529	19,900,000	0.611	0.434
0.55	13,900,000	0.711	0.576	12,400,000	0.664	0.477
0.60	10,400,000	0.756	0.635	7,900,000	0.716	0.536
0.65	7,400,000	0.811	0.713	5,100,000	0.768	0.588
0.70	5,400,000	0.863	0.788	3,000,000	0.834	0.676
0.75	3,800,000	0.923	0.876	1,900,000	0.900	0.773
0.80	2,900,000	0.970	0.951	1,200,000	0.968	0.867
0.85	2,100,000	1.028	1.038	853,000	1.032	0.947
0.90	1,500,000	1.080	1.142	598,000	1.098	1.011
0.95	1,100,000	1.133	1.252	428,000	1.165	1.152
1.00	787,000	1.205	1.393	275,000	1.270	1.272
1.10	446,000	1.327	1.639	139,000	1.495	1.572
1.20	222,000	1.500	1.920	104,000	1.602	1.699
1.30	188,000	1.542	1.906	87,000	1.671	1.748
1.40	137,000	1.622	2.145	87,000	1.671	1.748
1.50	85,000	1.742	2.140	87,000	1.671	1.748

Table 53 - Main Zone All Blocks Classed Measured Plus Indicated or Inferred

Red Chris Main Zone all blocks classed Measured plus Indicated or Inferred

All Blocks Classed Measured plus Indicated				All Blocks Classed Inferred		
Cutoff (Cu %)	Tonnes > Cutoff (tonnes)	Grade>Cutoff		Tonnes > Cutoff (tonnes)	Grade>Cutoff	
		Cu (%)	Au (g/t)		Cu (%)	Au (g/t)
0.05	694,000,000	0.231	0.183	509,000,000	0.214	0.198
0.10	563,300,000	0.267	0.208	419,400,000	0.244	0.221
0.15	456,700,000	0.300	0.232	338,700,000	0.273	0.243
0.20	347,700,000	0.340	0.262	256,700,000	0.305	0.269
0.25	250,800,000	0.385	0.297	186,600,000	0.336	0.295
0.30	178,800,000	0.430	0.329	121,500,000	0.368	0.316
0.35	122,100,000	0.480	0.363	64,500,000	0.408	0.329
0.40	81,700,000	0.533	0.394	26,200,000	0.457	0.323
0.45	54,700,000	0.587	0.435	9,700,000	0.523	0.319
0.50	37,700,000	0.639	0.479	4,900,000	0.577	0.342
0.55	26,300,000	0.689	0.530	3,772,000	0.595	0.349
0.60	18,400,000	0.739	0.592	1,962,000	0.608	0.328
0.65	12,500,000	0.793	0.662	33,000	0.687	0.335
0.70	8,400,000	0.853	0.748	17,000	0.719	0.287
0.75	5,600,000	0.916	0.842			
0.80	4,100,000	0.970	0.926			
0.85	2,900,000	1.029	1.011			
0.90	2,100,000	1.085	1.105			
0.95	1,600,000	1.142	1.225			
1.00	1,100,000	1.222	1.362			
1.10	584,000	1.366	1.623			
1.20	326,000	1.533	1.850			
1.30	275,000	1.583	1.856			
1.40	224,000	1.641	1.991			
1.50	172,000	1.706	1.942			

Figure 54 - Classified Resource Blocks Section 50,010 East

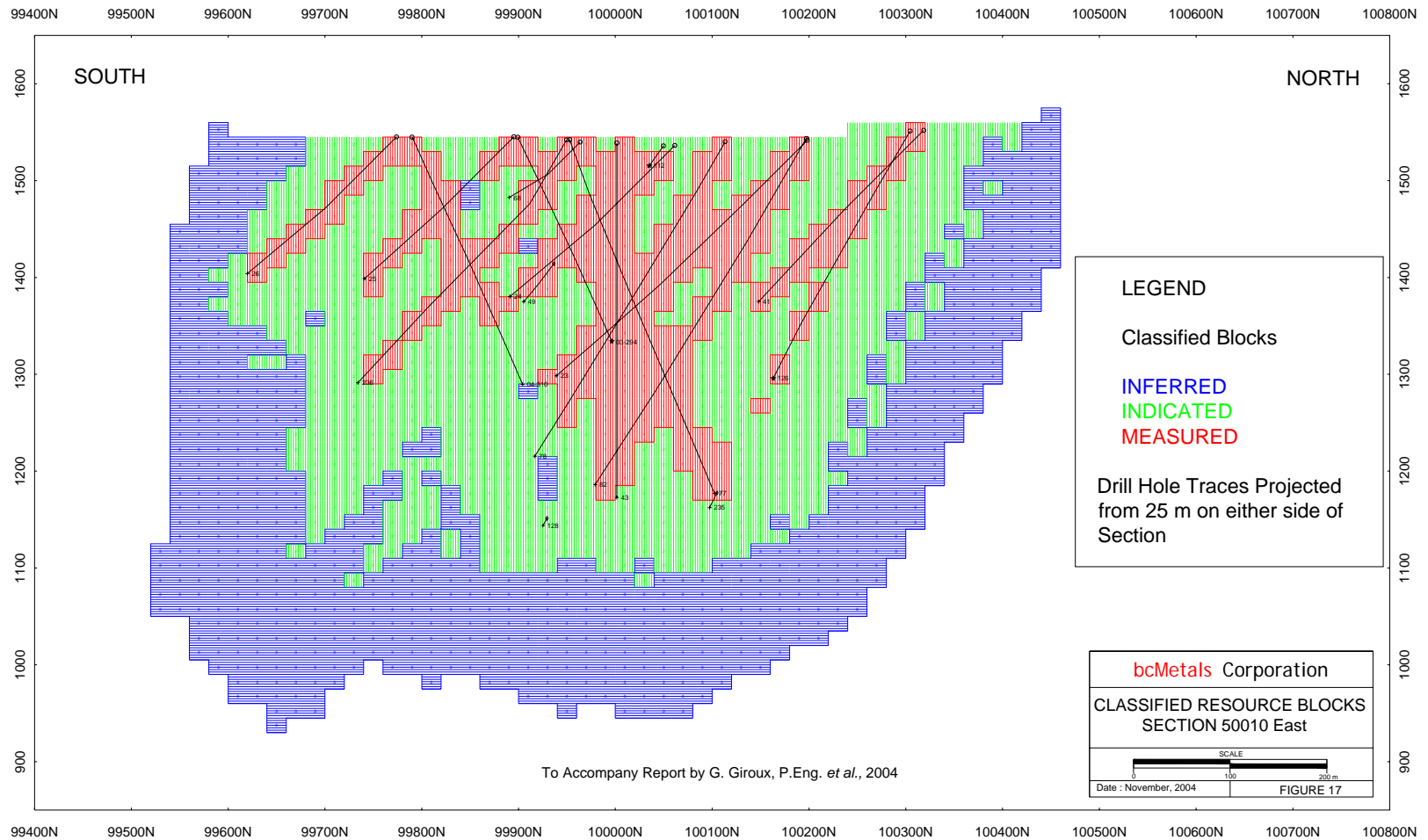
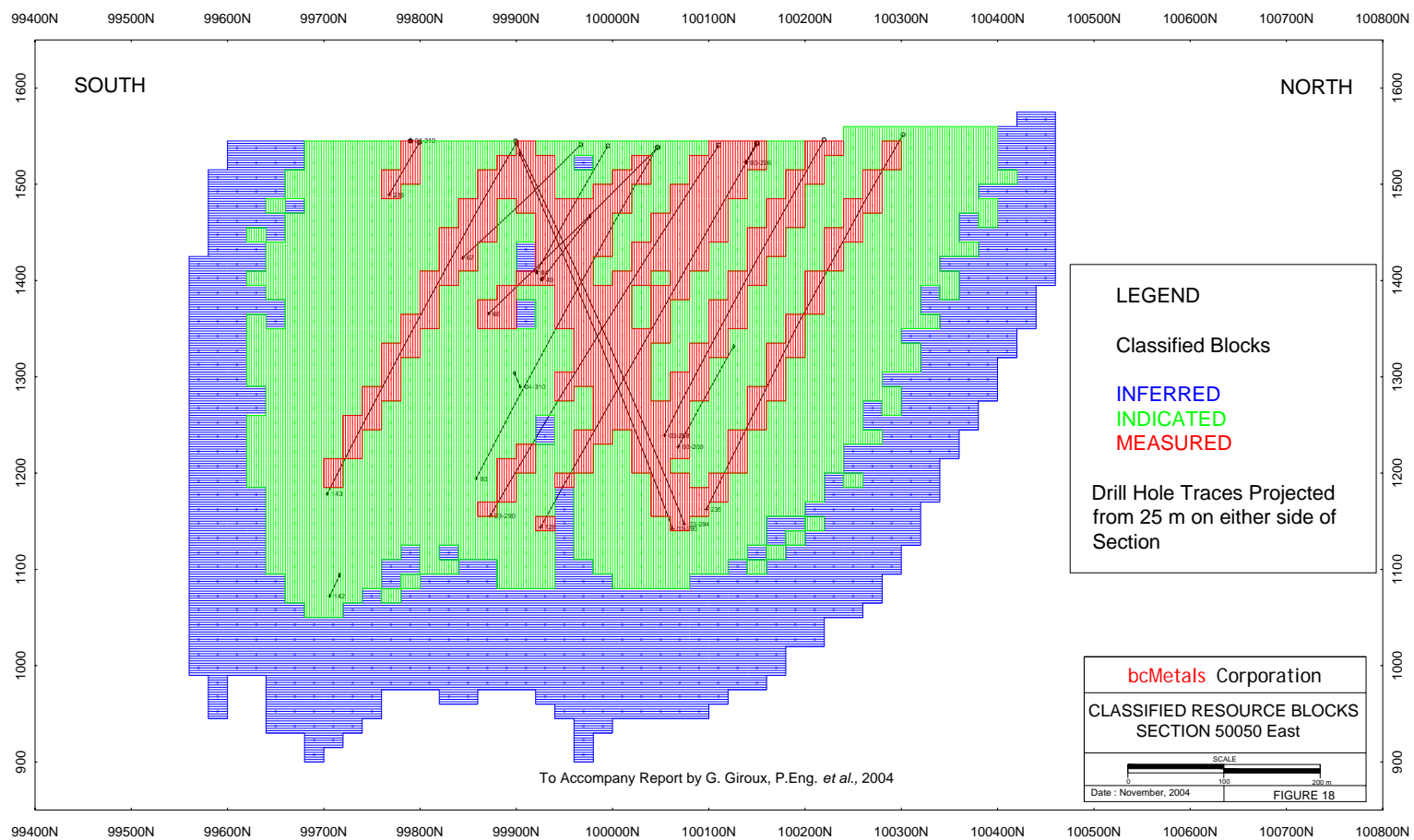


Figure 55 - Classified Resource Blocks Section 50,050 East



Results are presented in grade-tonnage tables for the East zone, east of 50,650 E, sorted by classification in Tables 54 and 55. Examples of how the classified blocks relate to existing drill holes are shown as Figures 56 and 57 for the East zone Section 50,750 E and 50,810 E. Blocks are colour coded by classification type and drill holes with ± 25 m on either side of the section are shown as solid lines.

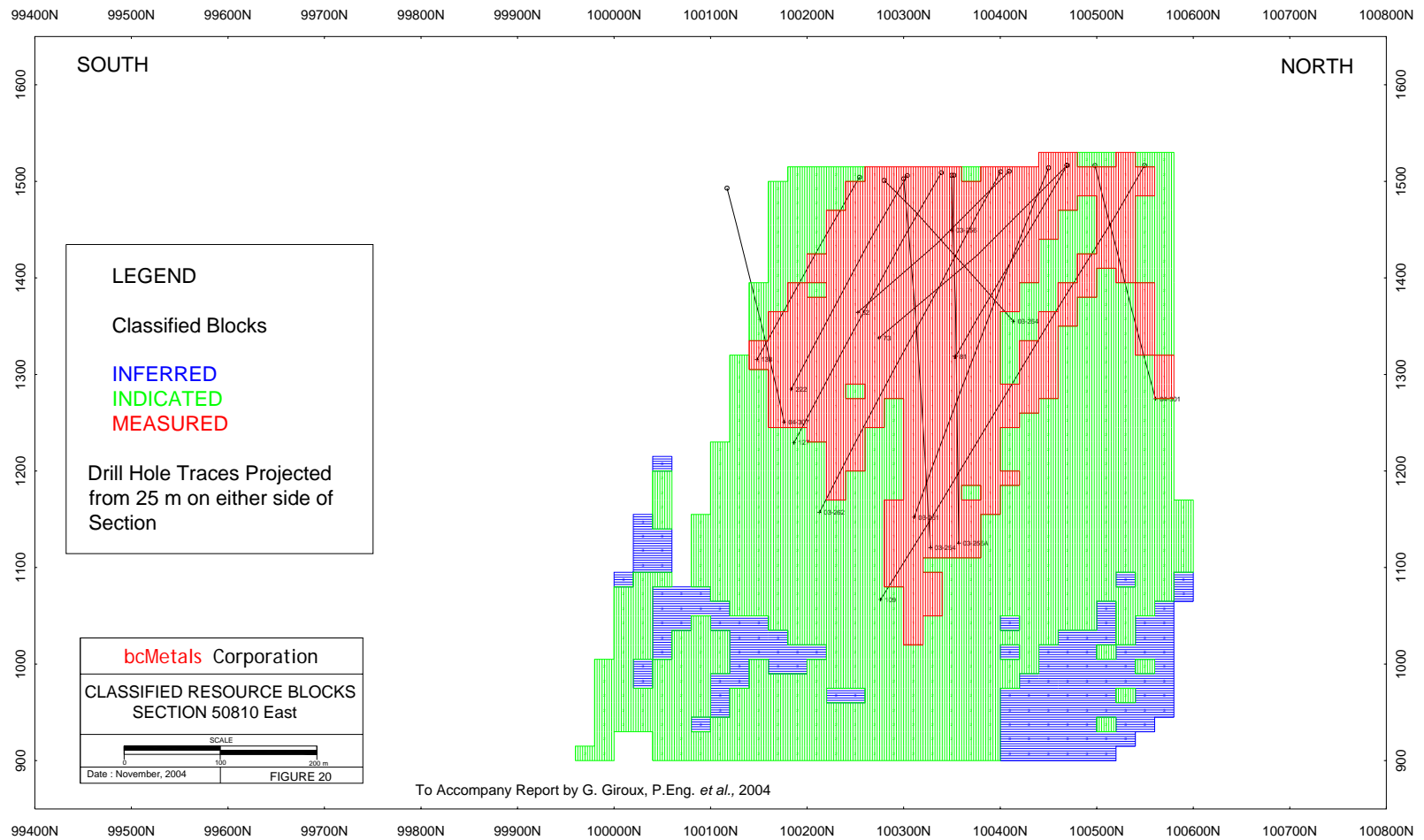
Table 54 - East Zone All Block Classed Measured or Indicated
Red Chris East Zone all blocks classed Measured or Indicated

All Blocks Classed Measured				All Blocks Classed Indicated		
Cutoff (Cu %)	Tonnes > Cutoff (tonnes)	Grade>Cutoff		Tonnes > Cutoff (tonnes)	Grade>Cutoff	
		Cu (%)	Au (g/t)		Cu (%)	Au (g/t)
0.05	54,600,000	0.384	0.356	184,600,000	0.201	0.209
0.10	46,100,000	0.441	0.405	129,800,000	0.255	0.253
0.15	38,500,000	0.504	0.461	93,400,000	0.306	0.293
0.20	32,800,000	0.563	0.513	65,700,000	0.363	0.336
0.25	28,900,000	0.609	0.555	45,400,000	0.426	0.383
0.30	25,100,000	0.658	0.599	34,400,000	0.474	0.424
0.35	21,000,000	0.723	0.658	25,600,000	0.526	0.478
0.40	17,800,000	0.787	0.719	19,900,000	0.570	0.526
0.45	15,400,000	0.843	0.774	14,900,000	0.619	0.587
0.50	13,400,000	0.897	0.829	11,300,000	0.665	0.649
0.55	12,000,000	0.943	0.877	8,600,000	0.709	0.703
0.60	10,700,000	0.988	0.922	6,000,000	0.766	0.769
0.65	9,300,000	1.042	0.980	4,400,000	0.817	0.823
0.70	8,200,000	1.092	1.033	3,400,000	0.859	0.863
0.75	7,300,000	1.133	1.076	2,800,000	0.891	0.895
0.80	6,600,000	1.172	1.116	2,200,000	0.922	0.926
0.85	6,000,000	1.206	1.148	1,500,000	0.969	0.978
0.90	5,200,000	1.260	1.197	900,000	1.034	1.062
0.95	4,500,000	1.311	1.246	600,000	1.072	1.107
1.00	4,100,000	1.348	1.279	400,000	1.130	1.161
1.10	3,300,000	1.414	1.346	136,000	1.295	1.325
1.20	2,600,000	1.485	1.402	68,000	1.441	1.429
1.30	1,900,000	1.567	1.481	50,000	1.519	1.525
1.40	1,300,000	1.662	1.570	33,000	1.593	1.484
1.50	1,000,000	1.733	1.627	33,000	1.593	1.484

Table 55 - East Zone All Blocks Classed Measured Plus Indicated or Inferred
Red Chris East Zone all blocks classed Measured plus Indicated or Inferred

All Blocks Classed Measured plus Indicated				All Blocks Classed Inferred		
Cutoff (Cu %)	Tonnes > Cutoff (tonnes)	Grade>Cutoff		Tonnes > Cutoff (tonnes)	Grade>Cutoff	
		Cu (%)	Au (g/t)		Cu (%)	Au (g/t)
0.05	239,200,000	0.243	0.243	56,000,000	0.147	0.149
0.10	175,900,000	0.304	0.293	32,500,000	0.199	0.191
0.15	131,900,000	0.364	0.342	21,500,000	0.236	0.205
0.20	98,400,000	0.429	0.395	12,000,000	0.289	0.225
0.25	74,300,000	0.497	0.450	6,800,000	0.342	0.244
0.30	59,500,000	0.552	0.498	4,600,000	0.374	0.256
0.35	46,600,000	0.615	0.559	2,600,000	0.413	0.266
0.40	37,700,000	0.672	0.617	1,200,000	0.457	0.271
0.45	30,300,000	0.733	0.682	600,000	0.491	0.215
0.50	24,700,000	0.791	0.747	200,000	0.511	0.231
0.55	20,600,000	0.845	0.804			
0.60	16,700,000	0.908	0.867			
0.65	13,700,000	0.969	0.929			
0.70	11,600,000	1.023	0.983			
0.75	10,100,000	1.067	1.026			
0.80	8,800,000	1.110	1.068			
0.85	7,500,000	1.159	1.114			
0.90	6,100,000	1.227	1.177			
0.95	5,100,000	1.281	1.229			
1.00	4,500,000	1.328	1.269			
1.10	3,500,000	1.410	1.345			
1.20	2,700,000	1.484	1.402			
1.30	2,000,000	1.566	1.482			
1.40	1,400,000	1.660	1.568			
1.50	1,000,000	1.729	1.622			

Figure 57 - Classified Resource Blocks Section 50,810 East



20.0 INTERPRETATION AND CONCLUSIONS

The 2004 infill drill program was successful in increasing both the confidence and total tonnage for the Red Chris Deposit. A total of 25 diamond drill holes were completed on the Main and East zones during the summer of 2004 (holes numbered 04-296 to 04-320). Of these holes 10 targeted the Main zone, 4 targeted the saddle zone between the Main and East zones, 6 tested the East zone and 5 condemnation holes were drilled to the north east of the East zone. The results for the 2004 Resource are compared below in a series of Tables to the Resource reported within the February 16, 2004 Resource Report (Giroux, *et al.*, 2004).

The main differences between the two studies resulted from the increased information with the addition of holes drilled in the Main Zone, in the East Zone and in the saddle zone between. As a result of the increase in information, both geological and assay, the differences are as follows:

- within the East zone, the outer shell and Main Phase were combined and joined with the Main zone for estimation purposes
- the East zone Main Phase was modeled with semivariograms separate from the Main zone
- the former Satellite zone was extended and renamed East zone Extension

The changes within the data base and within the modeling parameters resulted in the following changes in the Resource.

- Additional drill holes placed between existing drill fences increased the Measured Resource tonnage in all Cu cutoff ranges at reasonably similar grades.
- Decreases in some Indicated cutoff classes can be explained by material moving from the indicated to the measured category.
- The combined measured plus indicated table shows increases in tonnage at all cutoff ranges at virtually identical grades.
- Within the Inferred Category tonnes are decreased in all cutoff ranges due to the additional drilling and moving material into the higher categories.

Table 56 - Comparison of 2004 Results to 2002 Results in the Resource Categories

Red Chris All Zones - All Blocks Classed Measured

November 2004 Resource Estimate				16/02/04 Resource Estimate			Differences
Cutoff (Cu %)	Tonnes (x 10 ⁶)	Grade>Cutoff		Tonnes (x 10 ⁶)	Grade>Cutoff		Tonnes (x 10 ⁶)
		Cu (%)	Au (g/t)		Cu (%)	Au (g/t)	
0.20	109.8	0.45	0.36	99.3	0.46	0.36	11
0.30	74.2	0.54	0.44	68.6	0.55	0.45	6
0.35	58.7	0.60	0.49	55.5	0.61	0.49	3
0.40	47.2	0.66	0.54	44.7	0.67	0.54	3
0.50	31.2	0.77	0.66	29.8	0.78	0.66	1
0.60	21.1	0.87	0.78	20.3	0.88	0.79	1
0.70	13.6	1.00	0.94	13.1	1.01	0.95	1

Red Chris All Zones - All Blocks Classed Indicated

November 2004 Resource Estimate				16/02/04 Resource Estimate			Differences
Cutoff (Cu %)	Tonnes (x 10 ⁶)	Grade>Cutoff		Tonnes (x 10 ⁶)	Grade>Cutoff		Tonnes (x 10 ⁶)
		Cu (%)	Au (g/t)		Cu (%)	Au (g/t)	
0.20	336.4	0.33	0.27	339.0	0.33	0.27	-3
0.30	164.2	0.42	0.34	167.3	0.42	0.34	-3
0.35	110.0	0.47	0.38	112.0	0.47	0.38	-2
0.40	72.2	0.52	0.41	71.2	0.52	0.42	1
0.50	31.2	0.63	0.51	30.4	0.63	0.53	1
0.60	14.0	0.74	0.64	14.0	0.73	0.63	0
0.70	6.4	0.85	0.78	6.3	0.83	0.76	0

Red Chris All Zones - All Blocks Classed Measured plus Indicated

November 2004 Resource Estimate				16/02/04 Resource Estimate			Differences
Cutoff (Cu %)	Tonnes (x 10 ⁶)	Grade>Cutoff		Tonnes (x 10 ⁶)	Grade>Cutoff		Tonnes (x 10 ⁶)
		Cu (%)	Au (g/t)		Cu (%)	Au (g/t)	
0.20	446.1	0.36	0.29	438.2	0.36	0.29	8
0.30	238.3	0.46	0.37	235.8	0.46	0.37	3
0.35	168.7	0.52	0.42	167.5	0.51	0.42	1
0.40	119.4	0.58	0.47	115.8	0.58	0.47	4
0.50	62.4	0.70	0.59	60.1	0.70	0.59	2
0.60	35.1	0.82	0.72	34.3	0.82	0.72	1
0.70	20.0	0.95	0.88	19.4	0.95	0.89	1

Red Chris All Zones - All Blocks Classed Inferred

November 2004 Resource Estimate				16/02/04 Resource Estimate			Differences
Cutoff (Cu %)	Tonnes (x 10 ⁶)	Grade>Cutoff		Tonnes (x 10 ⁶)	Grade>Cutoff		Tonnes (x 10 ⁶)
		Cu (%)	Au (g/t)		Cu (%)	Au (g/t)	
0.20	268.7	0.31	0.29	275.8	0.31	0.29	-7
0.30	126.1	0.37	0.35	143.0	0.37	0.35	-17
0.35	67.1	0.41	0.38	83.3	0.41	0.38	-16
0.40	27.5	0.46	0.38	33.5	0.46	0.38	-6
0.50	5.1	0.58	0.44	6.1	0.58	0.44	-1
0.60	2.0	0.63	0.48	2.7	0.63	0.48	-1
0.70	0.02	0.73	0.37	0.13	0.73	0.37	-0.11

21.0 RECOMMENDATIONS

The Red Chris is a well developed project with an established resource that presents an excellent opportunity for exploitation. Over the past decade a tremendous amount of high quality work has been completed including geologic mapping and sampling, geophysics, trenching, diamond drilling, modelling, preliminary mine planning, metallurgical studies, environmental base lines studies, and ABA testing. To date, one scoping study (Fluor Daniel Wright Ltd., 1995) and two pre-feasibility studies and have been completed by American Bullion. The first feasibility study (Fluor Daniel Wright Ltd, 1996) evaluating the feasibility of a large open pit operation while the second (American Bullion Minerals Ltd., 1998) investigated possible smaller open pits using a system of ore passes and underground conveyors to move material.

Based on the results for the 2004 infill drill program completed at Red Chris and the corresponding increased confidence in the estimated resource, a full feasibility study is both warranted and recommended.

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CERTIFICATE of AUTHOR

I, Jay Collins, P.Eng., do hereby certify that:

1. I am President of:

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2. I graduated with a degree in Bachelor of Science (Commendation), Civil Engineering from the University of Portsmouth Polytechnic in 1974.

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia.

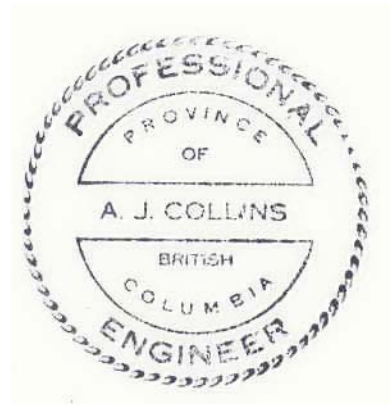
3. I have worked as a professional engineer for a total of 30 years since my graduation from university.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I am responsible for the preparation of Section 18.6.1 Capital Cost Estimate of the technical report titled *Red Chris Project Feasibility Study* scheduled to be issued on December 16th 2004 (the "Technical Report") relating to the Red Chris Copper property.
6. I have not had prior involvement with the property that is the subject of the Technical Report.
7. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

8. I am independent of the issuer applying all of the tests in section 1.5 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.
- 11.1 I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 9th Day of December, 2004



Signature of Qualified Person



Seal or Stamp
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CONSENT of AUTHOR

TO: Toronto Stock Exchange and Toronto Venture Stock Exchange

I, Jay Collins, P.Eng., do hereby consent to the filing of the written disclosure of the technical report titled "Technical Report on the Red Chris Copper-Gold Project" and dated December 16, 2004 (the "Technical Report") and any extracts from or a summary of the Technical Report under the National Instrument 43-101 of Red Chris Development Company, and to the filing of the Technical Report with the securities regulatory authorities referred to above.

I also certify that I have read the written disclosure being filed and do not have any reason to believe that there are any misrepresentations in the information derived from the Technical Report or that the written disclosure under the National Instrument 43-101 of Red Chris Development Company contains any misrepresentation of the information contained in the Technical Report.

Dated this 9th Day of December, 2004.



Signature of Qualified Person



Seal of Qualified Person

Jay Collins
Print name of Qualified Person



CERTIFICATE OF AUTHOR

William Colquhoun, Pr Eng.
111 Dunsmuir Street, Suite 400
Vancouver, BC
Tel: (604) 664-4349
Fax: (604) 664-3057
william.colquhoun@amec.com

I, William A. Colquhoun, Pr Eng., do hereby certify that:

1. I am a Professional Engineer, employed as Principal Metallurgist of AMEC Americas Limited and residing at 806-151 East Keith Ave. in the City of North Vancouver in the Province of British Columbia.
2. I am a member of the Engineering Council of South Africa. I graduated from the University of Strathclyde with a Bachelor of Science (Honours) degree in Chemical and Process Engineering in 1982.
3. I am currently a Consulting Metallurgist and have been so since September 1995.
4. I have practiced my profession continuously since 1982 and have been involved in: mine process production, research and development, engineering and construction for gold and silver, platinum group metals, uranium, chrome, coal and industrial minerals in South Africa and in process engineering and consulting for copper, zinc, gold, silver, platinum group metals, tantalum, uranium, iron ore, coal and industrial mineral properties in Canada, United States, Peru, Argentina, Chile, Ecuador, Ethiopia, Ukraine, Mongolia, Russia and the Middle East.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my experience and qualifications, I am a "qualified person" as defined in NI 43-101.
6. I am responsible for the preparation of Section 16 Mineral Processing and Metallurgical Testing of the technical report titled "Technical Report on the Red Chris Copper-Gold Project" and to be issued December 16, 2004 (the "Technical Report") relating to the Red Chris property. I have not visited this property.
7. I have not had prior involvement with the property that is the subject of the Technical Report.
8. I am not aware of any material fact or material change to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

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9. I am independent of the issuer applying all the tests in section 1.5 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.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated at Vancouver, British Columbia, this 15th day of December 2004.

A handwritten signature in black ink, appearing to read "William Colquhoun", written over a horizontal line.

William Colquhoun, Pr Eng.

**CONSENT OF AUTHOR**

William Colquhoun, Pr Eng.
111 Dunsmuir Street, Suite 400
Vancouver, BC
Tel: (604) 664-4349
Fax: (604) 664-3057
william.colquhoun@amec.com

TO: Toronto Stock Exchange and Toronto Venture Stock Exchange

I, William Colquhoun, Pr Eng., do hereby consent to the filing of the written disclosure of the technical report titled "Technical Report on the Red Chris Copper-Gold Project" and dated December 16, 2004 (the "Technical Report") and any extracts from or a summary of the Technical Report under the National Instrument 43-101 of Red Chris Development Company, and to the filing of the Technical Report with the securities regulatory authorities referred to above.

I also certify that I have read the written disclosure being filed and do not have any reason to believe that there are any misrepresentations in the information derived from the Technical Report or that the written disclosure under the National Instrument 43-101 of Red Chris Development Company contains any misrepresentation of the information contained in the Technical Report.

Dated at Vancouver, British Columbia, this 15th day of December 2004.

A handwritten signature in black ink, appearing to read "William Colquhoun", written over a horizontal line.

William Colquhoun, Pr Eng.

CERTIFICATE G.H. Giroux

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 #513 - 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.
- 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 Policy 43-101.
- 6) This report is based on a study of the data and literature available on the Red Chris Project. I am responsible for the resource estimations completed in Vancouver during 2003-04. A site visit and examination of drill core was made on October 29, 2002.
- 7) I have had prior involvement with the property completing earlier resource estimations in 1995, 1996, 1998, 2002 and 2003 as described in the Bibliography.
- 8) I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the Technical Report.
- 9) I am independent of the issuer applying all of the tests in section 1.5 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.
- 11) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public files on their websites accessible by the public.

Dated this day of December, 2004


G. H. Giroux, P.Eng., M.A.Sc.



G.H. Giroux, P.Eng., MASc.
Giroux Consultants Ltd.
Telephone: 604 684-0899
Email: gclmail@telus.net

CONSENT of AUTHOR

TO: Toronto Stock Exchange and Toronto Venture Stock Exchange

I, Gary H. Giroux, do hereby consent to the filing of the written disclosure of the technical report titled "Technical Report on the Red Chris Copper-Gold Project" and dated December 16, 2004 (the "Technical Report") and any extracts from or a summary of the Technical Report under the National Instrument 43-101 of Red Chris Development Company, and to the filing of the Technical Report with the securities regulatory authorities referred to above.

I also certify that I have read the written disclosure being filed and do not have any reason to believe that there are any misrepresentations in the information derived from the Technical Report or that the written disclosure under the National Instrument 43-101 of Red Chris Development Company contains any misrepresentation of the information contained in the Technical Report.

Dated this [insert date] Day of [insert month], 2004.



Signature of Qualified Person

G. H. GIRoux

[Print name of Qualified Person]



John Nilsson P.Eng.
Nilsson Mine Services Ltd.
Telephone: 604 465 0703
FAX:604 465 0731
Email: jnilsson@shaw.ca

CERTIFICATE of AUTHOR

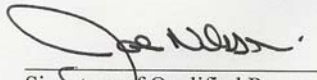
I, John Nilsson P.Eng., do hereby certify that:

1. I am President of:

Nilsson Mine Services Ltd.
20263 Mountain Place
Pitt Meadows, British Columbia
V3Y 2T9
2. I graduated with a Bachelors degree in Geology from the Queen's University in 1977. In addition, I have obtained a Masters degree in Mining Engineering from the Queen's University in 1990.
3. I am a member of the Association of Profession Engineers of British Columbia.
4. I have worked as a geologist and mining engineer for a total of 27 years since my graduation from university.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for the preparation of Section 18.2 Mining Operations of the technical report titled "Technical Report on the Red Chris Copper-Gold Project" and to be issued December 16, 2004 (the "Technical Report") relating to the Red Chris property. I visited the Red Chris property on August 5, 2004 for one day.
7. I have not had prior involvement with the property that is the subject of the Technical Report.
8. I am not aware of any material fact or material change to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
9. I am independent of the issuer applying all the tests in section 1.5 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.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 15th Day of December, 2004.


Signature of Qualified Person



John Nilsson P.Eng.

John Nilsson P.Eng.
Nilsson Mine Services Ltd.
Telephone: 604 465 0703
FAX:604 465 0731
Email: jnilsson@shaw.ca


CONSENT of AUTHOR

TO: Toronto Stock Exchange and Toronto Venture Stock Exchange

I, John Nilsson P.Eng., do hereby consent to the filing of the written disclosure of the technical report titled "Technical Report on the Red Chris Copper-Gold Project" and dated December 16, 2004 (the "Technical Report") and any extracts from or a summary of the Technical Report under the National Instrument 43-101 of Red Chris Development Company, and to the filing of the Technical Report with the securities regulatory authorities referred to above.

I also certify that I have read the written disclosure being filed and do not have any reason to believe that there are any misrepresentations in the information derived from the Technical Report or that the written disclosure under the National Instrument 43-101 of Red Chris Development Company contains any misrepresentation of the information contained in the Technical Report.

Dated this 15th Day of December, 2004.


Signature of Qualified Person



John Nilsson P.Eng.

David Tenney C.Eng.
Mine Geology Services
63, Finch Crescent
Whitehorse
Yukon Y1A 5X5

Tel: (867) 633 2759
Email: tenney@yknet.ca

CERTIFICATE of AUTHOR

I, David Tenney, C.Eng. do hereby certify that:

1. I am a consulting geologist operating from an office in Whitehorse, Yukon Territory, under the name of Mine Geology Services.
2. I graduated with a B.Sc. honours degree in geology from the University of Leicester in 1961 and was awarded a Diploma in Mineral Exploration (D.I.C. Min. Ex.) by the Royal School of Mines, London, in 1962
3. I am a member of the Institution of Mining and Metallurgy (MIMM), London.
4. I am a Chartered Engineer registered with the Engineering Council of Great Britain (Registration number 281895).
5. I have worked as a mining and exploration geologist since my graduation in 1962 to the present, a total of 42 years.
6. I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, experience, and affiliation with a professional association to be a “qualified person” for the purposes of NI 43-101.
7. I am responsible for the preparation of Section 11.1.3 2004 Drilling Program, Section 12.3 Sampling Method, Approach and Security – 2004 Drilling, and Section 13.3 Sample Preparation and Analyses – 2004 Drilling of the technical report titled “Technical Report on the Red Chris Copper-Gold Project” and to be issued December 16, 2004 (the “Technical Report”) relating to the Red Chris property. I worked at the Red Chris property from June 8, 2004 to August 14, 2004.

8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I am not aware of any material fact or material change to the subject matter of the Technical Report, covered by section 7 above, that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
10. I am independent of the issuer applying all the tests in section 1.5 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1, and the sections of the Technical Report referred to in 7 above, which have been prepared in compliance with that instrument and form.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this fifteenth Day of December, 2004.

D. Tenney,
Signature of Qualified Person

DAVID TENNEY
[Print name of Qualified Person]

David Tenney C.Eng.
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63, Finch Crescent
Whitehorse
Yukon Y1A 5X5

Tel: (867) 633 2759
Email: tenney@yknet.ca [QP's Letterhead] or

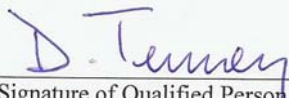
CONSENT of AUTHOR

TO: Toronto Stock Exchange and Toronto Venture Stock Exchange

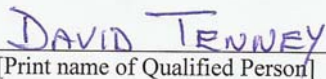
I, David Tenney, do hereby consent to the filing of the written disclosure of the technical report titled "Technical Report on the Red Chris Copper-Gold Project" and dated December 16, 2004 (the "Technical Report") and any extracts from or a summary of the Technical Report under the National Instrument 43-101 of Red Chris Development Company, and to the filing of the Technical Report with the securities regulatory authorities referred to above.

I also certify that I have read the sections of the report for which I am responsible (Section 11.1.3, section 12.3 and section 13.3) and do not have any reason to believe that there are any misrepresentation in this information, or that these sections of the written disclosure under the National Instrument 43-101 of Red Chris Development Company contain any misrepresentation.

Dated this fifteenth Day of December, 2004.



Signature of Qualified Person



[Print name of Qualified Person]