Technical Report

Casino Project
Pre-Feasibility Study
Yukon Territory, Canada

Volume I

Prepared for:

WESTERN COPPER CORPORATION

M3 Engineering & Technology Corp.

M3-PN07078
5 August 2008

TABLE OF CONTENTS

SECTION **PAGE**

VOLUME I: EXECUTIVE SUMMARY (NI 43-101)

LIST OF TABLES

TABLE NO. **DESCRIPTION**

LIST OF FIGURES

LIST OF APPENDICES

Responsibility	Qualified Person	Registration	Company
Feasibility Study Manager	Timothy S. Oliver	P. Eng.	M3
Resource Model	Gary Giroux	P. Eng.	Giroux Consultants Ltd.
Mine Planning	Mike Hester	F Aus IMM	IMC
Reserves	Mike Hester	F Aus IMM	IMC
Geology	C.M. Rebagliati	P.Eng.	Rebagliati Geological Consulting Ltd.
Metallurgical Testing	Jerry Hanks(sulphide)	PE	M3
Metallurgical Testing	Tom Drielick (oxide)	PE	M3
Flow Sheets	Jerry Hanks(sulphide)	PE	M3
Flow Sheets	Tom Drielick (oxide)	PE	M3
Geotechnical	Bruno Borntraeger	P.Eng.	Knight Piésold Consulting
Tailings	Bruno Borntraeger	P.Eng.	Knight Piésold Consulting
Environmental & Permitting	Jesse Duke		Gartner Lee Consultants

1 EXECUTIVE SUMMARY

1.1 TITLE PAGE

This report is prepared in accordance with the Canadian Standard NI 43-101. The first two items of this 26-item outline are the Title Page and Table of Contents. For ease of cross-referencing during review, the first two subsections of this report (1.1 and 1.2) are incorporated into the format for this report.

1.2 TABLE OF CONTENTS

See discussion in subsection 1.1.

1.3 SUMMARY (SYNOPSIS)

This report has been prepared by M3 Engineering & Technology Corporation of Tucson, Arizona to summarize the work performed in the preparation of a pre-feasibility study for the development of the Casino property in the Yukon Territory in northern Canada. The study contemplates the development of the property as a conventional truck-shovel, open pit mine, initially processing the gold bearing oxide cap as a heap leach operation. Sulphide ore processing would commence approximately 2.5 years later at a nominal rate of 90,000 tpd in a concentrator, which would produce copper concentrate and molybdenum concentrate. Gold values will report in the copper concentrate.

1.3.1 Financial Analysis

The base case for development of the Casino deposit will provide a pre-tax Internal Rate of Return ("IRR") of 20.4% and an undiscounted Net Present Value ("NPV") of $7.5 billion, based on 100% equity. The after-tax IRR is 14.9% with an undiscounted NPV of $4.5 billion. The base case financial evaluation uses LME three-year historical rolling average commodity prices as of the end of May 2008. The commodity prices used in the financial analysis are:

- Copper $2.95/lb.
- Molybdenum $30.97/lb.
- Gold $647.40/oz.

This approach is considered to be an industry standard and consistent with the guidance of the United States Securities and Exchange Commission. The values in this report are in Canadian currency, and assume an exchange rate of one to one between U.S. and Canadian dollars.

Table 1.3-1 shows production and financial estimates for the life of the mine and for to the first six years of operation. Higher ore grades and greater concentrate production during the initial six years of operation provide an accelerated cash flow during this period and achieves capital payback in 3.8 years.

Table 1.3-1: Production and Financial Estimates

	Years 1-6	**Life of Mine**
Average Annual Pre-tax Cashflow	$571 million	$219 million
Average Annual After-tax Cashflow	$448 million	$132 million
Average NSR (sulphide ore)	$29.66/tonne	$21.54/tonne
Average Annual Mill Feed Grade		
Copper (%)	0.325	0.212
Gold (g/t)	0.380	0.237
Molybdenum (%)	0.028	0.024
Average Annual Concentrate Production		
Copper (dry ktonnes)	313	201
Molybdenum (dry ktonnes)	11	9
Average Annual Metal Production from Flotation Plant		
Copper (M lb)	193	124
Gold (k oz)	263	158
Molybdenum (k lb)	13,415	10,899

1.3.2 Author's Recommendations

Based upon the encouraging financial performance indicated by the pre-feasibility study, M3 recommends Western Copper Corporation consider proceeding to full feasibility evaluation of the Casino property.

Western Copper Corporation should continue to further define the resource through exploration drilling, particularly in the more sparsely drilled area west of the main zone and deep drilling adjacent to the microbreccia pipe.

Western Copper Corporation should continue with the environmental studies and permitting efforts now underway.

Western Copper Corporation should continue with the engineering effort in support of permitting and to advance efforts toward preparation of a full feasibility study.

Western Copper Corporation should continue to monitor developments in the Yukon, northern British Columbia and Alaska to be in a position to participate in infrastructure development that might be beneficial to the advancement of the Casino project.

1.3.3 Author's Conclusions

The Casino mineral occurrence can be successfully and economically exploited by proven mining and processing methods under the conditions and assumptions outlined in this report.

Opportunities exist to enhance the project's economics, including:

- Inclusion of revenue from the recovery of silver
- Conversion of inferred resources into measured and indicated
- Increasing the overall resource
- Sharing of infrastructure development costs with other parties
- Refined engineering during the feasibility study
- Investigation of local sources of lime

1.3.4 Property Location

The Casino porphyry copper-gold-molybdenum deposit is located at latitude 62° 44'N and longitude 138° 50'W (NTS map sheet 115J/10), in west central Yukon, in the north-westerly trending Dawson Range mountains, 300 km northwest of the territorial capital of Whitehorse. Figure 1.3-1 is a map showing the location of Casino property in relation to the Yukon, British Columbia and the Northwest Territories.

The region has at least two other extensively developed exploration properties, the Carmacks Copper Project and the Minto Mine, both of which are discussed in Section 1.17, Adjacent Properties.

Other exploration projects in the area are:

- BC Gold's Carmacks Copper-Gold Project (not to be confused with Western Copper's Carmacks Project) where, according to the company's website, "In June, 2007 BCGold confirmed significant "Carmacks-style" copper oxide and gold mineralization (the ICE Zone) occurs on the Company's ICE claims, situated seven kilometres southward of Western Copper Corporation's Carmacks (Williams Creek) deposits. The properties under exploration are from 70-120 km SE of Casino."
- Northern Freegold Resources, Ltd. is actively drilling on their Freegold Mountain claims located about 100 km SE of Casino with four diamond drill rigs planned for the 2008 drilling campaign.
- Cariboo Rose Resources holds 253 contiguous claims adjacent to and west of the Casino claims and drilled five holes on the Canadian Creek property in 2007.

The project is located on Crown land administered by the Yukon Government and is within the Selkirk First Nation traditional territory and the Tr'ondek Hwechin First Nation traditional territory lies to the north.

1.3.5 Property Description

The Dawson Range forms a series of well-rounded ridges and hills that reach a maximum elevation of 1,675 m above mean sea level (ASL). The ridges rise above the Yukon Plateau, a peneplain at approximately 1200 m ASL, which is deeply incised by the mature drainage of the Yukon River watershed.

The characteristic terrain consists of rounded, rolling topography with moderate to deeply incised valleys. Major drainage channels extend below 1,000 m elevation. Most of the project lies between the 650 m elevation at Dip Creek and an elevation of 1,400 m at Patton Hill. The most notable local physical feature is the Yukon River, which flows to the west about 16 km north of the project site.

The mean annual temperature for the area is approximately -5.5°C with a summer mean of 10.5°C and a winter mean of -23°C. Mean annual precipitation ranges between 300-450 mm. Summers are warm, with very long, cold winters.

Figure 1.3-1: Property Location Map

1.3.6 Mineral Tenure, Royalties and Agreements

The Casino property presently consists of 349 full and partial active quartz mineral claims in good standing and 157 quartz mineral claims of pending status staked in June 2008. The total area covered is 8,961 ha. CRS Copper Resources Corp. (“CRS”), a 100% subsidiary of Western Copper Corporation (“WCC”), is the registered owner of all claims. The claims covering the property are shown on the map in Figure 1.3-2.

The historical claims held by prior owners of the project and transferred as part of WCC’s plan of arrangement with Lumina Resources Corp. (“Lumina”) consist of the Casino “A”, “B” and “JOE” claims. The 188 VIK mineral claims were staked in June 2007 by WCC and registered to CRS. The 94 CC claims and the 63 BRIT were staked in late June 2008 by WCC and as of July 14, 2008 active status is pending standard review of staking details by the Yukon Mining Recorder. The addition of the VIK, CC and BRIT claims have expanded the area of the Casino property significantly to the north, east and south.

On the basis of prior agreements of CRS, certain portions of the Casino property remain subject to royalty agreements in favour of Archer, Cathro and Associates (1981) Ltd. (“Archer Cathro”), and to an option agreement with Wildrose Resources Ltd. (“Wildrose”).

The royalties and agreements are:

- A 5% Net Profits Royalty (“NPR”) in favour of Archer Cathro on the Casino “A” and “B” Claims, and the “JOE” block of claims;

- The Casino “B” Claims are subject to an agreement between CRS Copper Resources Corp. and Wildrose (through the Option Agreement between CRS and Great Basin Gold, Ltd. (“GBG”) exercised on August 9, 2007) whereby Wildrose agrees to maintain the Casino “A” and “B” claims in good standing. In exchange, Wildrose has the right to acquire the Casino “B” claims for $1 each until May 2, 2020 upon a $200,000 payment to CRS, and subject to CRS reserving a 10% Net Profits Interest in the Casino “B” Claims.

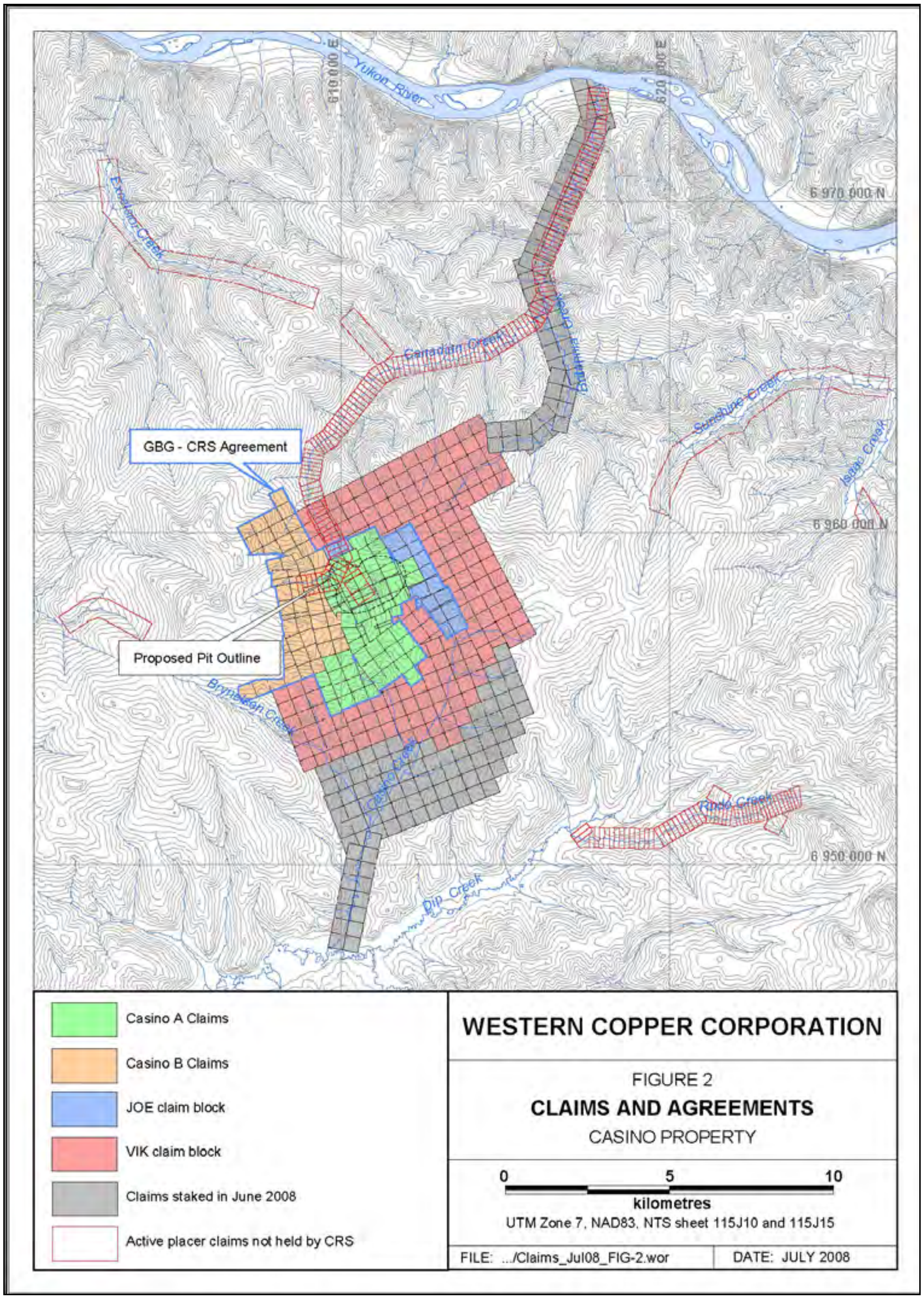


Figure 1.3-2: Project Claim Map

1.3.7 Geology and Mineralization

The Casino deposit occurs in an overlapping zone of the northern Yukon Cataclastic Terrane and the southern Yukon Crystalline Terrane, which contains the Dawson Range Batholith. The Dawson Range Batholith is the main country rock of the deposit and is mostly granodiorite and lesser diorite and quartz monzonite. There are two phases of granodiorite: a hornblende-bearing phase to the west, south and east and a biotite-hornblende bearing phase to the north. The granodiorite is brecciated along the south margin of the deposit with the breccia ranging from crackle breccia to intensely deformed cataclastic breccia. In some places the cataclastic breccia grades into the microbreccia of the core of the deposit. The intrusive microbreccia forms an irregularly shaped, subvertical pipe approximately 400 m in diameter in the core of the deposit and a series of smaller irregular bodies to the southwest.

The deposit is weathered to an average depth of 70 m, producing a well defined leached cap ("oxide gold") zone that is relatively gold enriched and copper depleted due to supergene alteration processes. With depth, the supergene alteration erratically grades from a poorly defined supergene oxide zone (upper "copper oxide" zone) to a better-defined supergene sulphide zone (lower "copper sulphide" zone). The average thickness of the copper oxide zone and copper sulphide zone are 10 m and 60 m respectively.

Copper-bearing minerals in the supergene oxide zone include chalcanthite, malachite and brochantite and minor azurite, tenorite, cuprite, and neotocite. In the supergene sulphide zone, chalcopyrite, bornite, and tetrahedrite are slightly to moderately supergene altered along grain borders and fractures to chalcocite, digenite, and/or covellite. Chalcocite also forms coatings on pyrite grains and pyrite clusters.

The supergene copper sulphide zone is characterized by a high pyrite content in the phyllic alteration, which promotes leaching. Thus, secondary enrichment zones are thickest along the contact of the potassic and phyllic alteration and accordingly the copper grades in the supergene sulphide zone are almost double the copper grades in the underlying hypogene zone (0.43% Cu versus 0.23% Cu). Generally molybdenite was unaffected by the supergene process.

The bulk of the sulphide mineralization occurs in and adjacent to the intrusion breccia and microbreccia. Molybdenite is concentrated moderately in the core of the deposit, and chalcopyrite is concentrated moderately towards the periphery, just inside the potassic / phyllic alteration contact.

Molybdenite occurs as discrete flakes, clusters of flakes, and selvages in early quartz veins, and is not generally intergrown with other sulphides. Native gold occurs as 50 to 70 micron free grains in quartz and as 1 to 15 micron inclusions in fractures in pyrite and chalcopyrite grains.

1.3.8 Exploration and Sampling

Exploration included multi-element soil geochemistry, geological mapping, geophysical surveys, trenching and drilling. Drill targets were located primarily on the basis of coincident copper-molybdenum geochemical anomalies.

Geophysical tools were used as target confirmation. Induced polarization surveys showed an area of high chargeability coincident with the phyllic halo of the porphyry deposit and an area of high resistivity coincident with the thickest part of the Leached Cap and the milled breccia pipe. Intermediate resistivity and low to moderate chargeability values over the deposit and broad areas to the north and west coincide with copper-gold-molybdenum soil geochemical anomalies. The zone of potassic alteration produced a strong ground-magnetic high, as did an area of magnetite-bearing diorite to the northwest.

Drilling to test the geochemical and geophysical anomalies centred on Patton Hill partly defined a bulk tonnage porphyry deposit. Further potential below and around the main area drilled was indicated.

1.3.9 Mineral Resource Estimate

Pacific Sentinel Gold Corp. staff geologists and consultants undertook an extensive geological interpretation of the Casino Deposit in 1994 and 1995 and a block model was developed to include lithologic, metallurgical and structural controls prior to establishing grade estimates.

Ordinary kriging of assay composites was utilized to interpolate grade into each block using variogram models developed by G. Giroux, P. Eng. in May 1995. In January 2004 a copper equivalent value was calculated for each block in this model using the grades for copper, gold, and molybdenum estimated by Giroux. In April 2008 measured and indicated blocks were re-categorized by Giroux for the purposes of this study.

Information provided by the 214 HQ (6.35 cm diameter) and NQ (4.76 cm diameter) core holes of the 1992 through 1994 drill campaigns was used in this study. These holes provide extensive coverage of the deposit and tend to have better core recovery and more reliable analytical results than the drilling done prior to 1992.

The gold, total copper and molybdenum data from the 70,276 m of core drilling from 214 holes was composited into 4,787 nominal 15 m down-hole composites based on mineralogical zone, domain and lithology. The composite lengths were modified to honour actual lithologic and mineralogical boundaries in the drill core. The composites ranged from 7.5 to 22.5 m in length, and averaged 14.68 m.

No cutting or capping was applied to the original assays, composites or final estimated values.

The statistical characteristics of the composited assay data for the different geologic units and mineralogical zones were examined and 31 variogram models were developed for the purposes of kriging the deposit. Ordinary kriging was used to interpolate grade into each block using the appropriate variogram model and composite file in May 1995.

Resource classification criteria are based on a combination of the gold relative estimation error, the domain code, and the presence of both Cu and Au estimates in each block. A further refinement to the classification of measured and indicated resources was introduced in April 2008 whereby blocks in the measured category by the above criteria were downgraded to indicated category blocks in the more sparsely drilled area west of the main zone.

The deposit is estimated to host measured and indicated supergene plus hypogene resources of 964 Mt grading 0.22 % copper, 0.24 g/t gold and 0.02 % molybdenum at a 0.30 % copper equivalent cutoff grade, containing an estimated 3.6 billion pounds of copper, 5.7 million ounces of gold, and 515 million pounds of molybdenum. In the overlying oxide cap, the deposit is estimated to host a measured and indicated resource of 38 Mt grading 0.57 g/t gold, 0.07% copper and 0.02 % molybdenum at a 0.40 g/t gold cutoff grade, containing an estimated 696,000 ounces of gold. This mineral resource estimate is inclusive of the mineral reserve estimate presented in the following section.

1.3.10 Mineral Reserve Estimate

Table 1.3-2 presents the mineral reserve for the Casino Project. The mill ore reserve amounts to 913.5 million tonnes at 0.212% copper, 0.237 g/t gold, and 0.0236% molybdenum. The heap leach reserve is an additional 77.9 million tonnes at 0.427 g/t gold and 0.062% copper.

For this reserve estimate, the measured mineral resource was converted to proven mineral reserve and the indicated mineral resource was converted to probable mineral reserve, with one exception. The low grade mill ore stockpile is considered a probable mineral reserve regardless of the original classification of the in-situ material.

Table 1.3-2: Mineral Reserve

Mill Ore Reserve:	**Ore ktonnes**	**Tot Cu (%)**	**Gold (g/t)**	**Moly (%)**
Proven Mineral Reserve:				
Direct Mill Feed	143,804	0.284	0.354	0.0311
Probable Mineral Reserve:				
Direct Mill Feed	613,571	0.208	0.225	0.0255
Low Grade Stockpile	156,171	0.163	0.177	0.0095
Total Probable Reserve	769,742	0.199	0.215	0.0223
Proven/Probable Reserve				
Direct Mill Feed	757,375	0.222	0.249	0.0266
Low Grade Stockpile	156,171	0.163	0.177	0.0095
Total Mill Ore Reserve	913,546	0.212	0.237	0.0236
Heap Leach Reserve:	**Ore ktonnes**	**Gold (g/t)**	**Tot Cu (%)**	**Moly (%)**
Proven Mineral Reserve	34,498	0.545	0.074	n.a.
Probable Mineral Reserve	43,419	0.333	0.053	n.a.
Total Heap Leach Reserve	77,917	0.427	0.062	n.a.

1.3.11 Mining

A mine plan was developed to supply ore to a conventional copper sulphide flotation plant with the capacity to process 32.0 to 36.4 million tonnes per year, depending on the mix of supergene and hypogene ores. The mine is scheduled to operate two 12 hour shifts per day, 365 days per year. This will require four mining crews. The crews will operate 7 days on-7 days off from a fly in-fly out camp.

Mining is by conventional open pit methods with drilled and blasted rock loaded onto rigid frame haul trucks by large electric shovels.

1.3.12 Metallurgical Testing

Early metallurgical testwork indicated that the oxide cap material could be leached with cyanide to recover gold, but at increasing copper grades, gold recovery and cyanide consumption were negatively affected. Flotation testing indicated that good copper concentrate grades were difficult to achieve.

Recent testwork by METCON Research on the oxide cap material showed that good recoveries of gold and acceptable cyanide consumptions could be obtained by integrating the cyanide heap leach with the SART process.

Flotation testing by G&T Metallurgical indicated that copper concentrate grades of 28% copper could be routinely achieved at copper recoveries averaging 84% with a primary grind size of 80% passing 147 µm and a regrind of 80% passing 22 µm. Gold will be recovered with the copper concentrate. Molybdenum will be recovered to a molybdenum concentrate in a separate flotation circuit.

1.3.13 Processing Flowsheet

Sulphide ore will be transported from the mine to the primary crusher by off-highway haulage trucks. Mineral concentrates of copper and molybdenum will be produced by mineral flotation technology.

The sulphide concentrator will consist of one SAG mill followed by two ball mills. This will be followed by a conventional flotation circuit consisting of tank and column cells with a separate circuit for molybdenum. Copper concentrate will be thickened and filtered and sent by on-highway haul trucks to nearby ports. Molybdenum concentrate will be dried and placed in supersacks for transport.

The design basis for the sulphide ore processing facility is 94,770 dry tonnes per day (dt/d). Design ore grade to the sulphide process plant is estimated to average 0.212% copper and 0.0236% molybdenum.

Oxide ore will be transported from the mine to a run of mine heap leaching facility by off-highway haulage trucks. Gold bullion produced from the oxide gold ore will be shipped by truck to metal refiners.

The design basis for oxide ore processing is 25,000 t/d. The overall oxide ore grade is estimated to average 0.427 grams/metric tonne (g/t) of gold and 0.0624% copper. Copper will be recovered, as a precipitate, by the SART process to control the quality of the leach solution. This precipitate will be shipped to smelters.

1.3.14 Metal Recoveries

The average metal recoveries expected from sulphide ore processing are based on:

- Copper recovery to copper concentrate, percent 84
- Copper concentrate grade, percent copper 28
- Gold recovery to copper concentrate, percent 66
- Molybdenum recovery to molybdenum concentrate, percent 66
- Molybdenum concentrate grade, percent molybdenum 55

The metal recoveries expected from oxide ore processing are based on:

- Gold recovery, percent 50
- Copper recovery to SART precipitate, percent 20
- Copper precipitate grade, percent copper 75

1.3.15 Process Reagents

Reagents that will be used in the sulphide ore process circuit are:

- Aerophine 3418A 9.6 g/t
- Aerofloat 208 36.7 g/t
- MIBC 79 g/t
- Pebble Lime (CaO) 3.0 kg/t
- #2 Diesel Fuel 67 g/t
- Sodium Hydrosulphide 0.033 g/t
- Flomin D-910 0.006 g/t
- Flocculant, 0.55 g/t

Reagents that will be used in oxide ore process circuit are:

- Sodium Cyanide 0.5 kg/t
- Caustic 0.13 kg/t
- Pebble Lime (CaO), 4.27 kg/t
- Hydrochloric Acid 0.01 kg/t
- Hydrogen Sulphide 0.039 kg/t
- Sulphuric Acid 0.38 kg/t
- Activated carbon, 0.5 kg/t
- Antiscalant 0.003 kg/t

1.3.16 Power

Kerr Wood Leidal Associates Ltd. evaluated options for the project power supply. The highest ranked option, and the one considered in this study, is an onsite 100 MW coal-fired circulating fluidized bed (CFB) power plant. A 100 MW diesel-fired combustion turbine plant will back up the CFB when it is unavailable; about 10% of the time. A 10 MW diesel generator will support the construction effort and supply power to the camp and the gold heap leach and recovery operations during construction of the mine, processing plant and CFB.

The base case assumes coal will be imported through a new coal receiving facility at Haines, Alaska. Coal will be hauled on a backhaul with trucks hauling concentrate to the port.

1.3.17 Water

The study anticipates the Yukon River will supply fresh and process makeup water. An infiltration gallery will collect water for pumping some 17 km to the site through a 750 mm diameter buried and insulated steel pipeline. A series of four booster stations are necessary to lift the water about 925 m to the plant from the river.

The nominal water makeup rate is 2,000 m^3/h.

A process return water system made up of a pump barge, booster station and pipeline will return tailings decant water to the process at a nominal rate of 2,500 m^3/h. This system will also collect meteoric water that accumulates within the basin. At times, the accumulated water and decant return water may be sufficient for all process requirements, eliminating much of the pumping required from the river.

1.3.18 Permits

The Yukon Environmental and Socio-economic Assessment Board (YESAB) assesses projects in Yukon for environmental and socio-economic effects under the *Yukon Environmental and Socio-economic Assessment Act* (YESAA). The YESAA screening process for the Casino Project has been assumed to take 24 to 30 months from the date the Project Proposal is accepted for review.

A positive YESAA Screening Report will lead to the three major permit approval procedures. The three major permits are:

- The Quartz Mining License, including a reclamation and closure plan, from the Yukon Government Energy, Mines and Resources/ Minerals Management Branch, which could follow the Screening Report within a matter of weeks.
- The Type A Water Use License from the Yukon Water Board, which could follow the Screening Report by about 12 months.
- The Metal Mining Effluent Regulation Schedule 2 Amendment from the Canada Department of Fisheries and Oceans/Environment, which requires about 15 months to obtain.

The Yukon government will determine the form and amount of security, or bond, to cover the full amount of outstanding mine reclamation and closure liability.

1.3.19 Operating Costs

Operating costs were determined as average costs over the life of the mine. The annual production basis for the sulphide concentrator is 33.4 million tonnes of ore producing approximately 371,000 tonnes of copper concentrates and 16,000 tonnes of molybdenum concentrates. The annual production basis for the oxide leach area is 9.7 million tonnes of ore producing approximately 53,000 ounces of gold and 1,000 tonnes of copper precipitates. The sulphide milling operation bears all mining costs since the premise of the heap leach is that were it not for the heap leach, the material would be waste. Table 1.3-4 summarizes sulphide operating costs. Heap leach/SART/ADR costs are $3.19 per tonne of leach material.

Table 1.3-3: Sulphide Operating Costs

Sulphide Operations	C$ per tonne ore
Mining	$3.24
Processing	$6.01
General and Administration	$0.47
Total	**$9.72**

1.3.20 Capital Cost Estimate

The initial capital investment for complete development of the project is estimated to be $2.1 billion total direct and indirect cost. Of this figure, $1.56 billion are direct and indirect costs for mining, concentrator and infrastructure including access road and port infrastructure. The remaining $550 million is the cost of a complete mine site power plant as estimated by Kerr Wood Leidal Associates Ltd.

Table 1.3-4: Capital Costs

	(millions)
Mine (including pre-stripping)	$340
Mill & Flotation	$541
Tailings	$96
Heap Leach	$51
Direct Cost	**$1,028**
Engineering & Management	$137
Camp	$65
Contingency	$165
Owner's Costs	$41
Total Capital Cost	**$1,437**
Power Plant (includes Heap power)	$548
Access Road	$91
Port	$36
Total	**$2,112**

The life-of-mine sustaining capital for the processing plant is estimated at $420 million and for the mine is estimated at $440 million.

1.4 INTRODUCTION

1.4.1 Purpose

The purpose of this report is to provide an independent Technical Report and Reserve Estimate of the copper ore present on the Casino property, in conformance with the standards required by NI 43-101 and Form 43-101F. The estimate of mineral reserves contained in this report conforms to the CIM Mineral Resource and Mineral Reserve definitions (December, 2005) referred to in National Instrument (NI) 43-101, Standards of Disclosure for Mineral Projects.

M3 provided engineering services to design a process plant and all ancillary and support facilities needed to bring the Casino Project into full production at an optimum rate. The design work included a number of engineering tradeoff studies culminating in a workable and efficient plant design.

Based on the designs and contributions by others as detailed in Section 1.5, M3 prepared capital and operating cost estimates and performed an economic analysis to assess the economic viability of the project.

1.4.2 Sources of Information

The Casino Project was the subject of a 1995 scoping-level study titled "Pacific Sentinel Gold Corp., A Development Plan for Feasibility, Casino Project, Scoping Study and Overview Report, 25,000 TPD Open Pit Mine and Concentrator Complex."

The study cited five factors that "collectively contributed towards a diminished economic outcome" for the project as studied. The five factors were:

- High preproduction stripping
- Low metallurgical recoveries for copper and gold
- High freight charges
- High infrastructure costs for road access and electrical power, and
- High power costs

Each of these factors is addressed in the current report and resolved satisfactorily.

Other reports supplied by Western Copper included:

- Pacific Sentinel Gold Corp, Casino Project, 1994 Exploration and Geotechnical Drilling Program on the Casino Porphyry Copper-Gold-Molybdenum Deposit, by B. Bower, B.Sc., T. Case, B.Sc., C. DeLong, B.Sc., and M. Rebagliati, P. Eng., May 16, 1995.
- Summary Report, Casino Project, Yukon Territory, Brameda Resources Ltd., April 1, 1969 – March 31, 1970, R.J. Cathro, P. Eng. and M.P. Phillips, May 31, 1970

M3 and others also relied on a large body of metallurgical, geological, geotechnical and environmental data reports compiled by prior owners over the years. These reports are listed in References.

1.4.3 Personal Inspections

Various members of the project team conducted on-site inspections of the property. Among those visiting the site were the following:

- Tim Oliver, M3 Engineering and Technology, Inc., conducted a walking tour of the open pit area, processing plant, heap leach, tailings management facility and a helicopter oversight of the proposed all-weather road, the air strip, the fresh water withdrawal point and the water pipeline route during two trips to site during September 25, 2007 and July 21, 2008.
- Mike Hester, Independent Mining Consultants, conducted a walking tour of the open pit area and an examination of the core stored at the site on July 22, 2008.
- C.M. Rebagliati, Rebagliati Geological Consulting Ltd, visited the mine pit area and viewed core stored at the site on July 22, 2008.
- Bruno Borntraeger, Knight Piésold Ltd., conducted a walking tour of the open pit area, processing plant, heap leach, and tailings management facility on July 21, 2008.
- Jesse Duke, Gartner Lee Ltd., inspected the site on May 9, 2008.
- Ray Corpela, Associated Engineering Group, Ltd., performed a helicopter oversight of the proposed transportation routes on September 25th and 26th, 2007.
- John Collings, Collings Johnson Inc., performed a helicopter oversight of the proposed transportation routes on September 25th and 26th, 2007.

1.4.4 Units

This report generally uses the SI (metric) system of units. Exceptions are some common uses such as pounds of copper or use of inches for piping sizes. All engineering calculations are conducted using the SI system. The term “tonne” rather than “ton” is used to denote a metric ton, and is used throughout the report. Units used and their abbreviations are listed in the table below.

Table 1.4-1: Abbreviations Used in This Document

Units	Abbreviations
Amperes	A
Cubic meters	m^3
Cubic meters per hour	m^3/h
Current density	A/m^2
Density	t/ m^3
Hectares	ha
grams/liter	g/L
Kilo (1000)	k
Kilogram	kg
Kilometer	km
Kilotonnes	ktonnes
Liters	L
Liters per second	L/s
Mega (1,000,000)	M
Meters	m
Millimeters	mm
Parts per million	ppm
Specific gravity	S.G.
Square meters	m^2
Temperature Celsius	°C
Temperature Fahrenheit	°F
Tonnage factor or specific volume	m^3/tonne
Tonnes per day	t/d
Volts	V
Watts	W

1.5 RELIANCE ON OTHER EXPERTS

In cases where the study authors have relied on contributions of the qualified persons listed below, the conclusions and recommendations are exclusively the qualified persons' own. The results and opinions outlined in this report that are dependent on information provided by qualified persons outside the employ of M3 are assumed to be current, accurate and complete as of the date of this report.

Draft copies of reports received from other experts have been reviewed for factual errors by Western Copper Corporation. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statement and opinions expressed in these documents are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of these reports.

1.5.1 Geology and Resource Definition

Geological characterization of the Casino deposit, in addition to the estimation of NI43-101 compliant resources, was performed by E.D. Titley, P. Geo. and C.M. Rebagliati, P. Eng., Rebagliati Geological Consulting Ltd. in February 2004. A non-material revision to the resource estimate was made by G. H. Giroux, P. Eng., MASc. in April 2008.

Messrs. C.M. Rebagliati and G.H. Giroux are the qualified persons responsible for the geology and resources portion of the pre-feasibility study.

1.5.2 Reserves and Mine Engineering

Resources were converted to reserves by Independent Mining Consultants, Inc. (IMC). IMC also executed the mine engineering, scheduling and planning for the project.

Mr. Michael G. Hester, F Aus IMM of IMC is the qualified person responsible for the reserves and mine engineering portion of the pre-feasibility study.

1.5.3 Metallurgy and Process Engineering

G&T Metallurgical Services of Kamloops, BC, performed numerous metallurgical testing to advance the flotation process design. Tom Shouldice is the official contact for G&T Metallurgical Services. Western Copper managed and oversaw G&T's work.

SGS Lakefield Research Limited of Lakefield, Ontario, performed a grinding circuit study. Mr. Carlos Lozano is the official contact for SGS Lakefield. Western Copper managed and oversaw SGS's work.

METCON Research of Tucson, AZ, USA, performed metallurgical testing to advance design of the gold heap leach. Francisco Garcia is the official contact for METCON Research. Western Copper managed and oversaw METCON's work.

Jerry T. Hanks, PE, and Tom Drielick, PE, under M3 supervision, reviewed and evaluated metallurgical testing results from the tests listed above. Mister Hanks and Mr. Drielick also reviewed and approved design criteria, flow sheets and equipment lists for the metallurgical processes.

1.5.4 Environmental and Permitting

Environmental baseline studies were performed by Hallam Knight Piésold during the 1990's. The studies are now outdated for the most part.

Gartner Lee Ltd. of Whitehorse, YT, is directing the present campaign of environmental baseline studies and provided permitting evaluations and analysis for this report. Mr. Jesse Duke is the official contact for Gartner Lee. Western Copper and M3 jointly managed and oversaw Gartner Lee's work.

1.5.5 Geotechnical

Knight Piésold Ltd. of Vancouver, BC, provided geotechnical consulting for the heap leach, tailings management, plant foundations, and water supply. Graham Greenaway is the official contact for Knight Piésold. Knight Piésold performed the work under M3 supervision. Bruno Borntraeger, P. Eng., is the qualified person on behalf of Knight Piésold.

1.5.6 Power Supply

Kerr Wood Leidal Associates, Ltd. assisted by W.N. Brazier Associates Inc. performed a power supply study for the project. Ron Monk is the official contact for Kerr Wood Leidal. W. Neil Brazier is the official contact for W. N. Brazier Associates. Kerr Wood Leidal and Brazier performed the work under M3's supervision.

1.5.7 Transportation

Associated Engineering (B.C.) Ltd. assisted by Collings Johnston Inc. and Lauga & Associates Consulting, Ltd. performed a study of transportation options including selection of an access road route. Associated Engineers and Lauga also prepared a report on port facility options. Martin Jobke is the official contact for Associated Engineers, John Collings is the official contact for Collings Johnson, and Tom Lauga is the official contact for Lauga and Associates. The transportation consultants worked under M3's supervision.

1.6 PROPERTY DESCRIPTION & LOCATION

1.6.1 Location

The Casino porphyry copper-gold-molybdenum deposit is located at latitude 62° 44'N and longitude 138° 50'W (NTS map sheet 115J/10), in west central Yukon, in the north-westerly trending Dawson Range mountains, 300 km northwest of the territorial capital of Whitehorse. Figure 1.6-1 is a map showing the location of Casino property in relation to the Yukon, British Columbia and the Northwest Territories.

Whitehorse is the nearest commercial and population center. No human settlements can be described as “local.” The village of Carmacks is about 150 km ESE and Pelly Crossing is about 115 km ENE. Beaver Creek, a tourist stop on the Alaskan Highway, is about 112 km WSW. Fairbanks, Alaska is 500 km WNW.

1.6.2 Property Description

The Dawson Range forms a series of well-rounded ridges and hills that reach a maximum elevation of 1,675 m above mean sea level (ASL). The ridges rise above the Yukon Plateau, a peneplain at approximately 1200 m ASL, which is deeply incised by the mature drainage of the Yukon River watershed.

The characteristic terrain consists of rounded, rolling topography with moderate to deeply incised valleys. Major drainage channels extend below 1,000 m elevation. Most of the project lies between the 650 m elevation at Dip Creek and an elevation of 1,400 m at Patton Hill. The most notable local physical feature is the Yukon River which flows to the west about 16 km north of the project site.

The mean annual temperature for the area is approximately -5.5°C with a summer mean of 10.5°C and a winter mean of -23°C. Mean annual precipitation ranges between 300-450 mm. Summers are warm, with very long, cold winters.

The Selkirk First Nation Traditional Territory encompasses the project area in the central portion of the Yukon.

Characteristic wildlife in the region includes caribou, grizzly and black bear, Dall sheep, moose, beaver, fox, wolf, hare, raven, rock and willow ptarmigan, and golden eagle.

The tops of hills and ridges are sparsely covered, most vegetation lies at the bottom and on the slopes of valleys. Vegetation consists of black and white spruce forests with aspen and occasionally lodgepole pine. Black spruce and paper birch prevail on permafrost slopes. Balsam poplar is common along floodplains. Scrub birch and willow form extensive stands in subalpine sections from valley bottoms to well above the tree line.

1.6.3 Mineral Tenure

The Casino property presently consists of 349 full and partial active quartz mineral claims in good standing and 157 quartz mineral claims of pending status staked in June 2008. The total area covered is 8961 ha. CRS Copper Resources Corp. ("CRS") has 100% ownership of these claims and is itself a 100% subsidiary of Western Copper Corporation ("WCC"). The claims covering the property are shown on the map in Figure 1.6-2.

The claims on the property are comprised of (a) 83 Casino "A" claims covering an area of 1,143 ha, (b) 55 Casino "B" claims covering an area of 924 ha, (c) 23 claims in the "JOE" block covering an area of 322 ha, (d) 188 claims in the "VIK" block covering an area of 3,416 ha, (e) 94 "CC" ("Casino Creek") claims covering an area of 1,933 ha, and (f) 63 "BRIT" ("Britannia Creek") claims covering an area of 1,223 ha. A listing of all Casino property claims indicating status and ownership details as of July 14, 2008 is provided in Table 1.6-1.

The historical claims held by prior owners of the project and transferred as part of WCC's plan of arrangement with Lumina Resources Corp. ("Lumina") consist of the Casino "A", "B" and "JOE" claims. The 188 VIK mineral claims were staked in June 2007 by WCC and registered to CRS. The 94 CC claims and the 63 BRIT were staked in late June 2008 by WCC and as of July 14, 2008 active status is pending standard review of staking details by the Yukon Mining Recorder. The addition of the VIK, CC and BRIT claims have expanded the area of the Casino property significantly to the north, east and south.

1.6.4 Option Agreements

The property interests of CRS in the Casino "A", "B" and "JOE" mineral claims (Figure 1.6-2) were assumed by WCC as a result of a plan of arrangement with Lumina Resources Corp. in November 2006.

On August 9, 2007, CRS, as a wholly owned subsidiary of Western Copper Corporation, exercised its option to acquire the 161 mineral claims that comprise the Casino "A" and "B" Claims, and the block of 23 "JOE" claims north and east of the Casino deposit in exchange for a $1 million cash payment to Great Basin Gold Ltd. ("GBG").

In the event that a production decision is made to put the optioned portion of the Casino property into commercial production, CRS will pay Great Basin Gold Ltd. an additional $1,000,000.

1.6.5 Agreements and Royalties

On the basis of prior agreements of CRS certain portions of the Casino property remain subject to royalty agreements in favour of Archer, Cathro and Associates (1981) Ltd. ("Archer Cathro"), and to an option agreement with Wildrose Resources Ltd. ("Wildrose").

The royalties and agreements are:

- A 5% Net Profits Royalty ("NPR") in favour of Archer Cathro on the Casino "A" and "B" Claims, and the "JOE" block of claims;

- The Casino "B" Claims are subject to an agreement between CRS and Wildrose (through the Option Agreement between CRS and GBG exercised on August 9, 2007) whereby Wildrose agrees to maintain the Casino "A" and "B" claims in good standing. In exchange, Wildrose has the right to acquire the Casino "B" claims for $1 each until May 2, 2020 upon a $200,000 payment to CRS, and subject to CRS reserving a 10% Net Profits Interest in the Casino "B" Claims.

1.6.6 Placer Claims

There are 28 active placer claims staked around Canadian Creek that overlap the Casino property mineral claims (Figure 1.6-2). These claims are held by others and are in good standing. There are no pre-existing agreements relating to the overlapping placer claims.

Figure 1.6-1: Casino Property Location Map

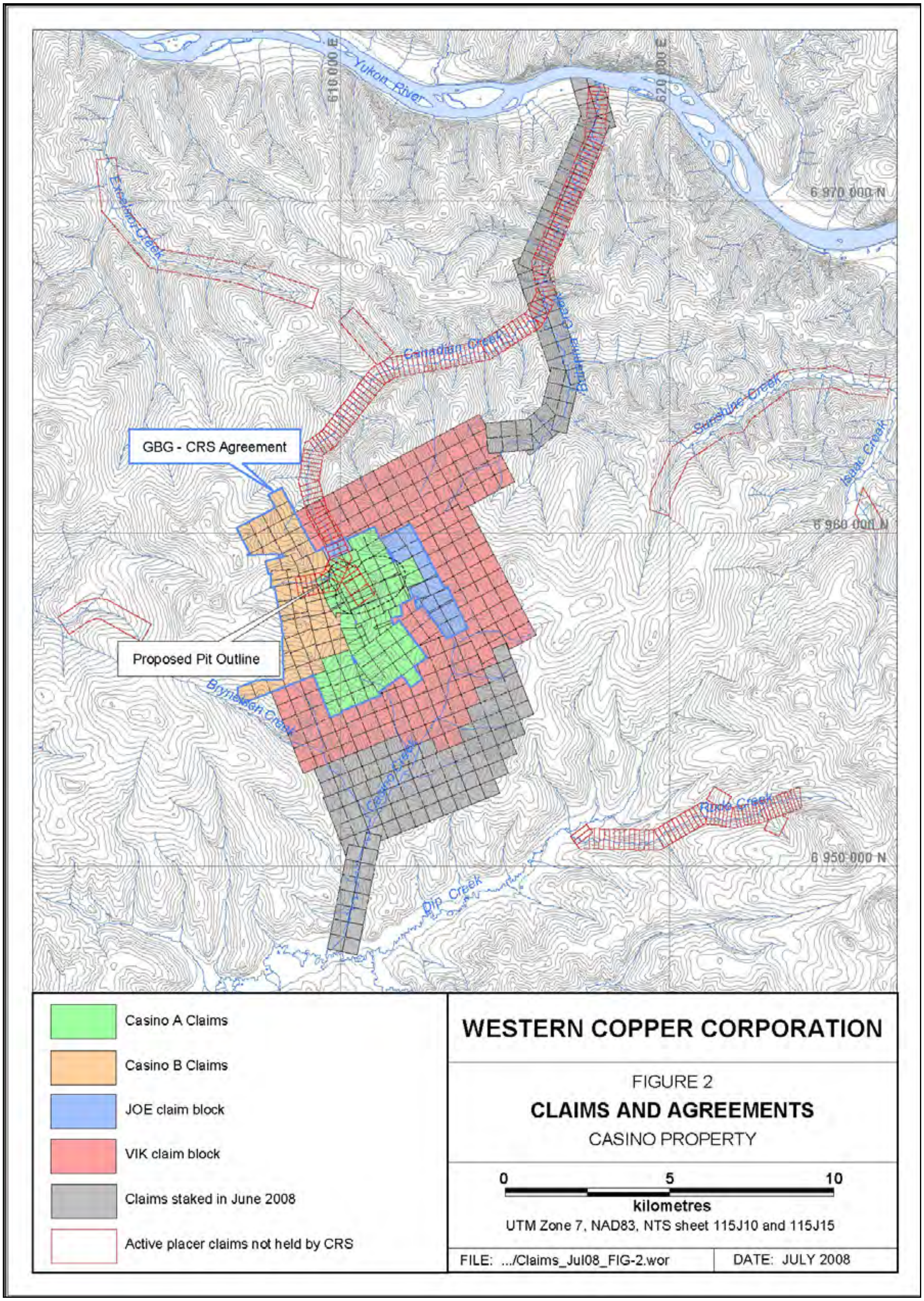


Figure 1.6-2: Casino Property Claims and Agreements

Table 1.6-1: Casino Property – List of Active and Pending Mineral Claims

"VIK" MINERAL CLAIMS (188) - WHITEHORSE MINING DISTRICT

No.	Grant Number	Claim Name	Claim No.	Expiry Date	Owner	NTS Map Sheet	Status
1	YC64893	VIK	1	5-Jun-2012	CRS - 100%.	115J10	Active
2	YC64894	VIK	2	5-Jun-2012	CRS - 100%.	115J10	Active
3	YC64895	VIK	3	5-Jun-2012	CRS - 100%.	115J10	Active
4	YC64896	VIK	4	5-Jun-2012	CRS - 100%.	115J10	Active
5	YC64897	VIK	5	5-Jun-2012	CRS - 100%.	115J10	Active
6	YC64898	VIK	6	5-Jun-2012	CRS - 100%.	115J10	Active
7	YC64899	VIK	7	5-Jun-2012	CRS - 100%.	115J10	Active
8	YC64900	VIK	8	5-Jun-2012	CRS - 100%.	115J10	Active
9	YC64901	VIK	9	5-Jun-2012	CRS - 100%.	115J10	Active
10	YC64902	VIK	10	5-Jun-2012	CRS - 100%.	115J10	Active
11	YC64903	VIK	11	5-Jun-2012	CRS - 100%.	115J10	Active
12	YC64904	VIK	12	5-Jun-2012	CRS - 100%.	115J10	Active
13	YC64905	VIK	13	5-Jun-2012	CRS - 100%.	115J10	Active
14	YC64906	VIK	14	5-Jun-2012	CRS - 100%.	115J10	Active
15	YC64907	VIK	15	5-Jun-2012	CRS - 100%.	115J10	Active
16	YC64908	VIK	16	5-Jun-2012	CRS - 100%.	115J10	Active
17	YC64909	VIK	17	5-Jun-2012	CRS - 100%.	115J10	Active
18	YC64910	VIK	18	5-Jun-2012	CRS - 100%.	115J10	Active
19	YC64911	VIK	19	5-Jun-2012	CRS - 100%.	115J10	Active
20	YC64912	VIK	20	5-Jun-2012	CRS - 100%.	115J10	Active
21	YC64913	VIK	21	5-Jun-2012	CRS - 100%.	115J10	Active
22	YC64914	VIK	22	5-Jun-2012	CRS - 100%.	115J10	Active
23	YC64915	VIK	23	5-Jun-2012	CRS - 100%.	115J10	Active
24	YC64916	VIK	24	5-Jun-2012	CRS - 100%.	115J10	Active
25	YC64917	VIK	25	5-Jun-2012	CRS - 100%.	115J10	Active
26	YC64918	VIK	26	5-Jun-2012	CRS - 100%.	115J10	Active
27	YC64919	VIK	27	5-Jun-2012	CRS - 100%.	115J10	Active
28	YC64920	VIK	28	5-Jun-2012	CRS - 100%.	115J10	Active
29	YC64921	VIK	29	5-Jun-2012	CRS - 100%.	115J10	Active
30	YC64922	VIK	30	5-Jun-2012	CRS - 100%.	115J10	Active
31	YC64923	VIK	31	5-Jun-2012	CRS - 100%.	115J10	Active
32	YC64924	VIK	32	5-Jun-2012	CRS - 100%.	115J10	Active
33	YC64925	VIK	33	5-Jun-2012	CRS - 100%.	115J10	Active
34	YC64926	VIK	34	5-Jun-2012	CRS - 100%.	115J10	Active
35	YC64927	VIK	35	5-Jun-2012	CRS - 100%.	115J10	Active
36	YC64928	VIK	36	5-Jun-2012	CRS - 100%.	115J10	Active
37	YC64929	VIK	37	5-Jun-2012	CRS - 100%.	115J10	Active
38	YC64930	VIK	38	5-Jun-2012	CRS - 100%.	115J10	Active
39	YC64931	VIK	39	5-Jun-2012	CRS - 100%.	115J10	Active
40	YC64932	VIK	40	5-Jun-2012	CRS - 100%.	115J10	Active
41	YC64933	VIK	41	5-Jun-2012	CRS - 100%.	115J10	Active

"VIK" MINERAL CLAIMS (188) - WHITEHORSE MINING DISTRICT

No.	Grant Number	Claim Name	Claim No.	Expiry Date	Owner	NTS Map Sheet	Status
42	YC64934	VIK	42	5-Jun-2012	CRS - 100%.	115J10	Active
43	YC64935	VIK	43	5-Jun-2012	CRS - 100%.	115J10	Active
44	YC64936	VIK	44	5-Jun-2012	CRS - 100%.	115J10	Active
45	YC64937	VIK	45	5-Jun-2012	CRS - 100%.	115J10	Active
46	YC64938	VIK	46	5-Jun-2012	CRS - 100%.	115J10	Active
47	YC64939	VIK	47	5-Jun-2012	CRS - 100%.	115J10	Active
48	YC64940	VIK	48	5-Jun-2012	CRS - 100%.	115J10	Active
49	YC64941	VIK	49	5-Jun-2012	CRS - 100%.	115J10	Active
50	YC64942	VIK	50	5-Jun-2012	CRS - 100%.	115J10	Active
51	YC64943	VIK	51	5-Jun-2012	CRS - 100%.	115J10	Active
52	YC64944	VIK	52	5-Jun-2012	CRS - 100%.	115J10	Active
53	YC64945	VIK	53	5-Jun-2012	CRS - 100%.	115J10	Active
54	YC64946	VIK	54	5-Jun-2012	CRS - 100%.	115J10	Active
55	YC64947	VIK	55	5-Jun-2012	CRS - 100%.	115J10	Active
56	YC64948	VIK	56	5-Jun-2012	CRS - 100%.	115J10	Active
57	YC64949	VIK	57	5-Jun-2012	CRS - 100%.	115J10	Active
58	YC64950	VIK	58	5-Jun-2012	CRS - 100%.	115J10	Active
59	YC64951	VIK	59	5-Jun-2012	CRS - 100%.	115J10	Active
60	YC64952	VIK	60	5-Jun-2012	CRS - 100%.	115J10	Active
61	YC64953	VIK	61	5-Jun-2012	CRS - 100%.	115J10	Active
62	YC64954	VIK	62	5-Jun-2012	CRS - 100%.	115J10	Active
63	YC64955	VIK	63	5-Jun-2012	CRS - 100%.	115J10	Active
64	YC64956	VIK	64	5-Jun-2012	CRS - 100%.	115J10	Active
65	YC64957	VIK	65	5-Jun-2012	CRS - 100%.	115J10	Active
66	YC64958	VIK	66	5-Jun-2012	CRS - 100%.	115J10	Active
67	YC64959	VIK	67	5-Jun-2012	CRS - 100%.	115J10	Active
68	YC64960	VIK	68	5-Jun-2012	CRS - 100%.	115J10	Active
69	YC64961	VIK	69	5-Jun-2012	CRS - 100%.	115J10	Active
70	YC64962	VIK	70	5-Jun-2012	CRS - 100%.	115J10	Active
71	YC64963	VIK	71	5-Jun-2012	CRS - 100%.	115J10	Active
72	YC64964	VIK	72	5-Jun-2012	CRS - 100%.	115J10	Active
73	YC64965	VIK	73	5-Jun-2012	CRS - 100%.	115J10	Active
74	YC64966	VIK	74	5-Jun-2012	CRS - 100%.	115J10	Active
75	YC64967	VIK	75	5-Jun-2012	CRS - 100%.	115J10	Active
76	YC64968	VIK	76	5-Jun-2012	CRS - 100%.	115J10	Active
77	YC64969	VIK	77	5-Jun-2012	CRS - 100%.	115J10	Active
78	YC64970	VIK	78	5-Jun-2012	CRS - 100%.	115J10	Active
79	YC64971	VIK	79	5-Jun-2012	CRS - 100%.	115J10	Active
80	YC64972	VIK	80	5-Jun-2012	CRS - 100%.	115J10	Active
81	YC64973	VIK	81	5-Jun-2012	CRS - 100%.	115J10	Active
82	YC64974	VIK	82	5-Jun-2012	CRS - 100%.	115J10	Active
83	YC64975	VIK	83	5-Jun-2012	CRS - 100%.	115J10	Active

"VIK" MINERAL CLAIMS (188) - WHITEHORSE MINING DISTRICT

No.	Grant Number	Claim Name	Claim No.	Expiry Date	Owner	NTS Map Sheet	Status
84	YC64976	VIK	84	5-Jun-2012	CRS - 100%.	115J10	Active
85	YC64977	VIK	85	5-Jun-2012	CRS - 100%.	115J10	Active
86	YC64978	VIK	86	5-Jun-2012	CRS - 100%.	115J10	Active
87	YC64979	VIK	87	5-Jun-2012	CRS - 100%.	115J10	Active
88	YC64980	VIK	88	5-Jun-2012	CRS - 100%.	115J10	Active
89	YC64981	VIK	89	5-Jun-2012	CRS - 100%.	115J10	Active
90	YC64982	VIK	90	5-Jun-2012	CRS - 100%.	115J10	Active
91	YC64983	VIK	91	5-Jun-2012	CRS - 100%.	115J10	Active
92	YC64984	VIK	92	5-Jun-2012	CRS - 100%.	115J10	Active
93	YC64985	VIK	93	5-Jun-2012	CRS - 100%.	115J10	Active
94	YC64986	VIK	94	5-Jun-2012	CRS - 100%.	115J10	Active
95	YC64987	VIK	95	5-Jun-2012	CRS - 100%.	115J10	Active
96	YC64988	VIK	96	5-Jun-2012	CRS - 100%.	115J10	Active
97	YC64989	VIK	97	5-Jun-2012	CRS - 100%.	115J10	Active
98	YC64990	VIK	98	5-Jun-2012	CRS - 100%.	115J10	Active
99	YC64991	VIK	99	5-Jun-2012	CRS - 100%.	115J10	Active
100	YC64992	VIK	100	5-Jun-2012	CRS - 100%.	115J10	Active
101	YC64993	VIK	101	5-Jun-2012	CRS - 100%.	115J10	Active
102	YC64994	VIK	102	5-Jun-2012	CRS - 100%.	115J10	Active
103	YC64995	VIK	103	5-Jun-2012	CRS - 100%.	115J10	Active
104	YC64996	VIK	104	5-Jun-2012	CRS - 100%.	115J10	Active
105	YC64997	VIK	105	5-Jun-2012	CRS - 100%.	115J10	Active
106	YC64998	VIK	106	5-Jun-2012	CRS - 100%.	115J10	Active
107	YC64999	VIK	107	5-Jun-2012	CRS - 100%.	115J10	Active
108	YC65000	VIK	108	5-Jun-2012	CRS - 100%.	115J10	Active
109	YC65001	VIK	109	5-Jun-2012	CRS - 100%.	115J10	Active
110	YC65002	VIK	110	5-Jun-2012	CRS - 100%.	115J10	Active
111	YC65003	VIK	111	5-Jun-2012	CRS - 100%.	115J10	Active
112	YC65004	VIK	112	5-Jun-2012	CRS - 100%.	115J10	Active
113	YC65005	VIK	113	5-Jun-2012	CRS - 100%.	115J10	Active
114	YC65006	VIK	114	5-Jun-2012	CRS - 100%.	115J10	Active
115	YC65007	VIK	115	5-Jun-2012	CRS - 100%.	115J10	Active
116	YC65008	VIK	116	5-Jun-2012	CRS - 100%.	115J10	Active
117	YC65009	VIK	117	5-Jun-2012	CRS - 100%.	115J10	Active
118	YC65010	VIK	118	5-Jun-2012	CRS - 100%.	115J15	Active
119	YC65011	VIK	119	5-Jun-2012	CRS - 100%.	115J15	Active
120	YC65012	VIK	120	5-Jun-2012	CRS - 100%.	115J15	Active
121	YC65013	VIK	121	5-Jun-2012	CRS - 100%.	115J15	Active
122	YC65014	VIK	122	5-Jun-2012	CRS - 100%.	115J15	Active
123	YC65015	VIK	123	5-Jun-2012	CRS - 100%.	115J15	Active
124	YC65016	VIK	124	5-Jun-2012	CRS - 100%.	115J15	Active
125	YC65017	VIK	125	5-Jun-2012	CRS - 100%.	115J15	Active

"VIK" MINERAL CLAIMS (188) - WHITEHORSE MINING DISTRICT

No.	Grant Number	Claim Name	Claim No.	Expiry Date	Owner	NTS Map Sheet	Status
126	YC65018	VIK	126	5-Jun-2012	CRS - 100%.	115J15	Active
127	YC65019	VIK	127	5-Jun-2012	CRS - 100%.	115J15	Active
128	YC65020	VIK	128	5-Jun-2012	CRS - 100%.	115J15	Active
129	YC65021	VIK	129	5-Jun-2012	CRS - 100%.	115J15	Active
130	YC65022	VIK	130	5-Jun-2012	CRS - 100%.	115J15	Active
131	YC65023	VIK	131	5-Jun-2012	CRS - 100%.	115J15	Active
132	YC65024	VIK	132	5-Jun-2012	CRS - 100%.	115J15	Active
133	YC65025	VIK	133	5-Jun-2012	CRS - 100%.	115J15	Active
134	YC65026	VIK	134	5-Jun-2012	CRS - 100%.	115J15	Active
135	YC65027	VIK	135	5-Jun-2012	CRS - 100%.	115J15	Active
136	YC65028	VIK	136	5-Jun-2012	CRS - 100%.	115J15	Active
137	YC65029	VIK	137	5-Jun-2012	CRS - 100%.	115J15	Active
138	YC65030	VIK	138	5-Jun-2012	CRS - 100%.	115J15	Active
139	YC65031	VIK	139	5-Jun-2012	CRS - 100%.	115J15	Active
140	YC65032	VIK	140	5-Jun-2012	CRS - 100%.	115J15	Active
141	YC65033	VIK	141	5-Jun-2012	CRS - 100%.	115J15	Active
142	YC65034	VIK	142	5-Jun-2012	CRS - 100%.	115J15	Active
143	YC65035	VIK	143	5-Jun-2012	CRS - 100%.	115J15	Active
144	YC65036	VIK	144	5-Jun-2012	CRS - 100%.	115J15	Active
145	YC65037	VIK	145	5-Jun-2012	CRS - 100%.	115J15	Active
146	YC65038	VIK	146	5-Jun-2012	CRS - 100%.	115J15	Active
147	YC65039	VIK	147	5-Jun-2012	CRS - 100%.	115J15	Active
148	YC65040	VIK	148	5-Jun-2012	CRS - 100%.	115J15	Active
149	YC65041	VIK	149	5-Jun-2012	CRS - 100%.	115J15	Active
150	YC65042	VIK	150	5-Jun-2012	CRS - 100%.	115J15	Active
151	YC65043	VIK	151	5-Jun-2012	CRS - 100%.	115J15	Active
152	YC65044	VIK	152	5-Jun-2012	CRS - 100%.	115J15	Active
153	YC65045	VIK	153	5-Jun-2012	CRS - 100%.	115J15	Active
154	YC65046	VIK	154	5-Jun-2012	CRS - 100%.	115J15	Active
155	YC65047	VIK	155	5-Jun-2012	CRS - 100%.	115J15	Active
156	YC65048	VIK	156	5-Jun-2012	CRS - 100%.	115J15	Active
157	YC65049	VIK	157	5-Jun-2012	CRS - 100%.	115J15	Active
158	YC65050	VIK	158	5-Jun-2012	CRS - 100%.	115J15	Active
159	YC65051	VIK	159	5-Jun-2012	CRS - 100%.	115J15	Active
160	YC65052	VIK	160	5-Jun-2012	CRS - 100%.	115J15	Active
161	YC65053	VIK	161	5-Jun-2012	CRS - 100%.	115J15	Active
162	YC65054	VIK	162	5-Jun-2012	CRS - 100%.	115J15	Active
163	YC65055	VIK	163	5-Jun-2012	CRS - 100%.	115J15	Active
164	YC65056	VIK	164	5-Jun-2012	CRS - 100%.	115J15	Active
165	YC65057	VIK	165	5-Jun-2012	CRS - 100%.	115J15	Active
166	YC65058	VIK	166	5-Jun-2012	CRS - 100%.	115J15	Active
167	YC65059	VIK	167	5-Jun-2012	CRS - 100%.	115J15	Active

"VIK" MINERAL CLAIMS (188) - WHITEHORSE MINING DISTRICT

No.	Grant Number	Claim Name	Claim No.	Expiry Date	Owner	NTS Map Sheet	Status
168	YC65060	VIK	168	5-Jun-2012	CRS - 100%.	115J15	Active
169	YC65061	VIK	169	5-Jun-2012	CRS - 100%.	115J15	Active
170	YC65062	VIK	170	5-Jun-2012	CRS - 100%.	115J15	Active
171	YC65063	VIK	171	5-Jun-2012	CRS - 100%.	115J15	Active
172	YC65064	VIK	172	5-Jun-2012	CRS - 100%.	115J15	Active
173	YC65065	VIK	173	5-Jun-2012	CRS - 100%.	115J15	Active
174	YC65066	VIK	174	5-Jun-2012	CRS - 100%.	115J15	Active
175	YC65067	VIK	175	5-Jun-2012	CRS - 100%.	115J15	Active
176	YC65068	VIK	176	5-Jun-2012	CRS - 100%.	115J15	Active
177	YC65069	VIK	177	5-Jun-2012	CRS - 100%.	115J15	Active
178	YC65070	VIK	178	5-Jun-2012	CRS - 100%.	115J15	Active
179	YC65071	VIK	179	5-Jun-2012	CRS - 100%.	115J15	Active
180	YC65072	VIK	180	5-Jun-2012	CRS - 100%.	115J15	Active
181	YC65073	VIK	181	5-Jun-2012	CRS - 100%.	115J15	Active
182	YC65074	VIK	182	5-Jun-2012	CRS - 100%.	115J15	Active
183	YC65075	VIK	183	5-Jun-2012	CRS - 100%.	115J15	Active
184	YC65076	VIK	184	5-Jun-2012	CRS - 100%.	115J15	Active
185	YC65077	VIK	185	5-Jun-2012	CRS - 100%.	115J15	Active
186	YC65078	VIK	186	5-Jun-2012	CRS - 100%.	115J15	Active
187	YC65079	VIK	187	5-Jun-2012	CRS - 100%.	115J15	Active
188	YC65080	VIK	188	5-Jun-2012	CRS - 100%.	115J15	Active

"JOE" MINERAL CLAIMS (23) - WHITEHORSE MINING DISTRICT

No.	Grant Number	Claim Name	Claim No.	Expiry Date	Owner	NTS Map Sheet	Status
1	Y 51849	CAT (F)	26	25-Mar-2012	CRS 100%	115J10	Active
2	Y 10693	JOE	89	5-Jun-2012	CRS 100%	115J10	Active
3	Y 10694	JOE	90	5-Jun-2012	CRS 100%	115J110	Active
4	Y 51850	JOE (F)	91	5-Jun-2012	CRS 100%	115J10	Active
5	Y 10695	JOE	91	5-Jun-2012	CRS 100%	115J10	Active
6	Y 10696	JOE	92	5-Jun-2012	CRS 100%	115J10	Active
7	Y 51851	JOE (F)	92	5-Jun-2012	CRS 100%	115J10	Active
8	Y 10697	JOE	93	5-Jun-2012	CRS 100%	115J10	Active
9	Y 51852	JOE (F)	93	5-Jun-2012	CRS 100%	115J10	Active
10	Y 10698	JOE	94	5-Jun-2012	CRS 100%	115J10	Active
11	Y 51853	JOE (F)	94	5-Jun-2012	CRS 100%	115J10	Active
12	Y 10699	JOE	95	5-Jun-2012	CRS 100%	115J10	Active
13	Y 51854	JOE (F)	95	5-Jun-2012	CRS 100%	115J10	Active
14	Y 10700	JOE	96	5-Jun-2012	CRS 100%	115J10	Active
15	Y 51855	JOE (F)	96	5-Jun-2012	CRS 100%	115J10	Active
16	Y 10702	JOE	98	5-Jun-2012	CRS 100%	115J10	Active

"JOE" MINERAL CLAIMS (23) - WHITEHORSE MINING DISTRICT

No.	Grant Number	Claim Name	Claim No.	Expiry Date	Owner	NTS Map Sheet	Status
17	Y 10703	JOE	99	5-Jun-2012	CRS 100%	115J10	Active
18	Y 10705	JOE	101	5-Jun-2012	CRS 100%	115J10	Active
19	Y 10706	JOE	102	5-Jun-2012	CRS 100%	115J10	Active
20	Y 10707	JOE	103	5-Jun-2012	CRS 100%	115J15	Active
21	Y 10708	JOE	104	5-Jun-2012	CRS 100%	115J15	Active
22	Y 35192	MOUSE	1	5-Jun-2012	CRS 100%	115J10	Active
23	Y 35193	MOUSE	2	5-Jun-2012	CRS 100%	115J10	Active

CASINO "A" MINERAL CLAIMS (83) - WHITEHORSE MINING DISTRICT

No.	Grant Number	Claim Name	Claim No.	Expiry Date	Owner	NTS Map Sheet	Status
1	Y 10701	JOE	97	25-Mar-2012	CRS 100%	115J10	Active
2	Y 10704	JOE	100	25-Mar-2012	CRS 100%	115J10	Active
3	56983	#1 AIRPORT GROUP	-	25-Mar-2012	CRS 100%	115J10	Active
4	56990	#2 AIRPORT GROUP	-	25-Mar-2012	CRS 100%	115J10	Active
5	56984	#3 AIRPORT GROUP	-	25-Mar-2012	CRS 100%	115J10	Active
6	56991	#4 AIRPORT GROUP	-	25-Mar-2012	CRS 100%	115J10	Active
7	56985	#5 AIRPORT GROUP	-	25-Mar-2012	CRS 100%	115J10	Active
8	56992	#6 AIRPORT GROUP	-	25-Mar-2012	CRS 100%	115J10	Active
9	56993	#8 AIRPORT GROUP	-	25-Mar-2012	CRS 100%	115J10	Active
10	56979	#1 BOMBER GROUP	-	25-Mar-2012	CRS 100%	115J10	Active
11	56987	#2 BOMBER GROUP	-	25-Mar-2012	CRS 100%	115J10	Active
12	56980	#3 BOMBER GROUP	-	25-Mar-2012	CRS 100%	115J10	Active
13	56981	#5 BOMBER GROUP	-	25-Mar-2012	CRS 100%	115J10	Active
14	56988	#6 BOMBER GROUP	-	25-Mar-2012	CRS 100%	115J10	Active
15	Y 35585	LOST FR.	1	25-Mar-2014	CRS 100%	115J10	Active
16	Y 35586	LOST FR.	2	25-Mar-2013	CRS 100%	115J10	Active
17	Y 35587	LOST FR.	3	25-Mar-2013	CRS 100%	115J10	Active
18	92201	CAT	1	25-Mar-2012	CRS 100%	115J10	Active
19	Y 51846	CAT (>F)	1	25-Mar-2012	CRS 100%	115J10	Active

CASINO "A" MINERAL CLAIMS (83) - WHITEHORSE MINING DISTRICT

No.	Grant Number	Claim Name	Claim No.	Expiry Date	Owner	NTS Map Sheet	Status
20	92202	CAT	2	25-Mar-2012	CRS 100%	115J10	Active
21	Y 51847	CAT (F)	2	25-Mar-2012	CRS 100%	115J10	Active
22	92203	CAT	3	25-Mar-2012	CRS 100%	115J10	Active
23	Y 39601	CAT (F)	3	25-Mar-2012	CRS 100%	115J10	Active
24	92204	CAT	4	25-Mar-2012	CRS 100%	115J10	Active
25	Y 39602	CAT (F)	4	25-Mar-2012	CRS 100%	115J10	Active
26	92205	CAT	5	25-Mar-2012	CRS 100%	115J10	Active
27	92206	CAT	6	25-Mar-2012	CRS 100%	115J10	Active
28	92207	CAT	7	25-Mar-2012	CRS 100%	115J10	Active
29	92208	CAT	8	25-Mar-2012	CRS 100%	115J10	Active
30	92209	CAT	9	25-Mar-2012	CRS 100%	115J10	Active
31	92210	CAT	10	25-Mar-2012	CRS 100%	115J10	Active
32	92211	CAT	11	25-Mar-2012	CRS 100%	115J10	Active
33	92212	CAT	12	25-Mar-2012	CRS 100%	115J10	Active
34	92213	CAT	13	25-Mar-2012	CRS 100%	115J10	Active
35	92214	CAT	14	25-Mar-2012	CRS 100%	115J10	Active
36	92215	CAT	15	25-Mar-2012	CRS 100%	115J10	Active
37	92216	CAT	16	25-Mar-2012	CRS 100%	115J10	Active
38	92217	CAT	17	25-Mar-2012	CRS 100%	115J10	Active
39	92218	CAT	18	25-Mar-2012	CRS 100%	115J10	Active
40	92219	CAT	19	25-Mar-2012	CRS 100%	115J10	Active
41	92220	CAT	20	25-Mar-2012	CRS 100%	115J10	Active
42	92221	CAT	21	25-Mar-2012	CRS 100%	115J10	Active
43	92222	CAT	22	25-Mar-2012	CRS 100%	115J10	Active
44	92764	CAT	23	25-Mar-2012	CRS 100%	115J10	Active
45	Y 36686	CAT (F)	22	25-Mar-2013	CRS 100%	115J10	Active
46	Y 39603	CAT (F)	23	25-Mar-2012	CRS 100%	115J10	Active
47	92765	CAT	24	25-Mar-2012	CRS 100%	115J10	Active
48	92766	CAT	25	25-Mar-2012	CRS 100%	115J10	Active
49	92776	CAT	35	25-Mar-2012	CRS 100%	115J10	Active
50	92777	CAT	36	25-Mar-2012	CRS 100%	115J10	Active
51	92778	CAT	37	25-Mar-2012	CRS 100%	115J10	Active
52	92779	CAT	38	25-Mar-2012	CRS 100%	115J10	Active
53	92780	CAT	39	25-Mar-2012	CRS 100%	115J10	Active
54	92781	CAT	40	25-Mar-2012	CRS 100%	115J10	Active
55	92782	CAT	41	25-Mar-2012	CRS 100%	115J10	Active
56	92783	CAT	42	25-Mar-2012	CRS 100%	115J10	Active
57	95724	CAT	47	25-Mar-2012	CRS 100%	115J10	Active
58	Y 36687	CAT (F)	47	25-Mar-2012	CRS 100%	115J10	Active
59	95725	CAT	48	25-Mar-2012	CRS 100%	115J10	Active
60	Y 36688	CAT (F)	48	25-Mar-2012	CRS 100%	115J10	Active
61	95726	CAT	49	25-Mar-2012	CRS 100%	115J10	Active
62	95727	CAT	50	25-Mar-2012	CRS 100%	115J10	Active
63	95728	CAT	51	25-Mar-2012	CRS 100%	115J10	Active

CASINO "A" MINERAL CLAIMS (83) - WHITEHORSE MINING DISTRICT

No.	Grant Number	Claim Name	Claim No.	Expiry Date	Owner	NTS Map Sheet	Status
64	95729	CAT	52	25-Mar-2012	CRS 100%	115J10	Active
65	95730	CAT	53	25-Mar-2012	CRS 100%	115J10	Active
66	95731	CAT	54	25-Mar-2012	CRS 100%	115J10	Active
67	95732	CAT	55	25-Mar-2012	CRS 100%	115J15	Active
68	95733	CAT	56	25-Mar-2012	CRS 100%	115J15	Active
69	95734	CAT	57	25-Mar-2012	CRS 100%	115J10	Active
70	Y 36689	CAT (F)	57	5-Jun-2012	CRS 100%	115J10	Active
71	95736	CAT	59	25-Mar-2012	CRS 100%	115J10	Active
72	95735	CAT	58	25-Mar-2014	CRS 100%	115J10	Active
73	95737	CAT	60	25-Mar-2012	CRS 100%	115J10	Active
74	95738	CAT	61	25-Mar-2012	CRS 100%	115J10	Active
75	Y 36690	CAT (F)	62	25-Mar-2012	CRS 100%	115J10	Active
76	95739	CAT	62	25-Mar-2012	CRS 100%	115J10	Active
77	Y 35582	MOUSE (F)	161	25-Mar-2012	CRS 100%	115J10	Active
78	Y 35583	MOUSE (F)	162	25-Mar-2013	CRS 100%	115J10	Active
79	Y 35584	MOUSE (F)	163	25-Mar-2013	CRS 100%	115J10	Active
80	YB37280	F	29	25-Mar-2012	CRS 100%	115J10	Active
81	YB37282	F	31	25-Mar-2012	CRS 100%	115J10	Active
82	YB37284	F	33	25-Mar-2012	CRS 100%	115J10	Active
83	4252	HELICOPTER	-	25-Mar-2012	CRS 100%	115J10	Active

CASINO "B" MINERAL CLAIMS (55) - WHITEHORSE MINING DISTRICT

No.	Grant Number	Claim Name	Claim No.	Expiry Date	Owner	NTS Map Sheet	Status
1	YB36618	CAS	31	25-Mar-2012	CRS 100%	115J10	Active
2	YB36619	CAS	32	25-Mar-2012	CRS 100%	115J10	Active
3	YB36620	CAS	33	25-Mar-2012	CRS 100%	115J15	Active
4	YB36621	CAS	34	25-Mar-2012	CRS 100%	115J15	Active
5	YB36622	CAS	35	25-Mar-2012	CRS 100%	115J15	Active
6	YB36623	CAS	36	25-Mar-2012	CRS 100%	115J15	Active
7	95740	CAT	63	25-Mar-2012	CRS 100%	115J10	Active
8	95741	CAT	64	25-Mar-2012	CRS 100%	115J10	Active
9	95742	CAT	65	25-Mar-2012	CRS 100%	115J10	Active
10	95743	CAT	66	25-Mar-2012	CRS 100%	115J10	Active
11	95744	CAT	67	25-Mar-2012	CRS 100%	115J10	Active
12	95745	CAT	68	25-Mar-2012	CRS 100%	115J10	Active
13	95746	CAT	69	25-Mar-2012	CRS 100%	115J10	Active
14	95747	CAT	70	25-Mar-2012	CRS 100%	115J10	Active
15	YB37242	E	23	25-Mar-2012	CRS 100%	115J15	Active
16	YB37243	E	24	25-Mar-2012	CRS 100%	115J15	Active
17	YB37244	E	25	25-Mar-2012	CRS 100%	115J15	Active
18	YB37246	E	27	25-Mar-2012	CRS 100%	115J10	Active

CASINO "B" MINERAL CLAIMS (55) - WHITEHORSE MINING DISTRICT

No.	Grant Number	Claim Name	Claim No.	Expiry Date	Owner	NTS Map Sheet	Status
19	YB37247	E	28	25-Mar-2012	CRS 100%	115J10	Active
20	YB37248	E	29	25-Mar-2012	CRS 100%	115J15	Active
21	YB37249	E	30	25-Mar-2012	CRS 100%	115J15	Active
22	YB37250	E	31	25-Mar-2012	CRS 100%	115J15	Active
23	YB37251	E	32	25-Mar-2012	CRS 100%	115J15	Active
24	YB37278	F	27	25-Mar-2012	CRS 100%	115J10	Active
25	YB37279	F	28	25-Mar-2012	CRS 100%	115J10	Active
26	YB37640	I	1	25-Mar-2012	CRS 100%	115J10	Active
27	YB37641	I	2	25-Mar-2012	CRS 100%	115J10	Active
28	YB37642	I	3	25-Mar-2012	CRS 100%	115J10	Active
29	YB37643	I	4	25-Mar-2012	CRS 100%	115J10	Active
30	YB37658	I	19	25-Mar-2012	CRS 100%	115J10	Active
31	YB37659	I	20	25-Mar-2012	CRS 100%	115J10	Active
32	Y 35194	MOUSE	3	25-Mar-2012	CRS 100%	115J10	Active
33	Y 35195	MOUSE	4	25-Mar-2012	CRS 100%	115J10	Active
34	Y 35196	MOUSE	5	25-Mar-2012	CRS 100%	115J10	Active
35	Y 35197	MOUSE	6	25-Mar-2012	CRS 100%	115J10	Active
36	Y 35198	MOUSE	7	25-Mar-2012	CRS 100%	115J10	Active
37	Y 35199	MOUSE	8	25-Mar-2012	CRS 100%	115J10	Active
38	Y 35200	MOUSE	9	25-Mar-2012	CRS 100%	115J10	Active
39	Y 35201	MOUSE	10	25-Mar-2012	CRS 100%	115J10	Active
40	Y 35202	MOUSE	11	25-Mar-2012	CRS 100%	115J10	Active
41	Y 35203	MOUSE	12	25-Mar-2012	CRS 100%	115J10	Active
42	Y 35204	MOUSE	13	25-Mar-2012	CRS 100%	115J10	Active
43	Y 35205	MOUSE	14	25-Mar-2012	CRS 100%	115J10	Active
44	Y 35206	MOUSE	15	25-Mar-2012	CRS 100%	115J10	Active
45	Y 35207	MOUSE	16	25-Mar-2012	CRS 100%	115J10	Active
46	Y 35483	MOUSE	89	25-Mar-2012	CRS 100%	115J10	Active
47	Y 35484	MOUSE	90	25-Mar-2012	CRS 100%	115J10	Active
48	Y 35491	MOUSE	97	25-Mar-2012	CRS 100%	115J10	Active
49	Y 35492	MOUSE	98	25-Mar-2012	CRS 100%	115J10	Active
50	Y 35517	MOUSE	123	25-Mar-2012	CRS 100%	115J10	Active
51	Y 35518	MOUSE	124	25-Mar-2012	CRS 100%	115J10	Active
52	Y 35519	MOUSE	125	25-Mar-2012	CRS 100%	115J10	Active
53	Y 35520	MOUSE	126	25-Mar-2012	CRS 100%	115J10	Active
54	Y 35521	MOUSE	127	25-Mar-2012	CRS 100%	115J10	Active
55	Y 35522	MOUSE	128	25-Mar-2012	CRS 100%	115J10	Active

"CC" MINERAL CLAIMS (94) - WHITEHORSE MINING DISTRICT

No.	Grant Number	Claim Name	Claim No.	Expiry Date	Owner	NTS Map Sheet	Status
1	YC81379	CC	1	20-Jun-2009	CRS 100%	115J10	Pending
2	YC81380	CC	2	20-Jun-2009	CRS 100%	115J10	Pending
3	YC81381	CC	3	20-Jun-2009	CRS 100%	115J10	Pending
4	YC81382	CC	4	20-Jun-2009	CRS 100%	115J10	Pending
5	YC81383	CC	5	20-Jun-2009	CRS 100%	115J10	Pending
6	YC81384	CC	6	20-Jun-2009	CRS 100%	115J10	Pending
7	YC81385	CC	7	20-Jun-2009	CRS 100%	115J10	Pending
8	YC81386	CC	8	20-Jun-2009	CRS 100%	115J10	Pending
9	YC81387	CC	9	20-Jun-2009	CRS 100%	115J10	Pending
10	YC81388	CC	10	20-Jun-2009	CRS 100%	115J10	Pending
11	YC81389	CC	11	20-Jun-2009	CRS 100%	115J10	Pending
12	YC81390	CC	12	20-Jun-2009	CRS 100%	115J10	Pending
13	YC81391	CC	13	20-Jun-2009	CRS 100%	115J10	Pending
14	YC81392	CC	14	20-Jun-2009	CRS 100%	115J10	Pending
15	YC81393	CC	15	20-Jun-2009	CRS 100%	115J10	Pending
16	YC81394	CC	16	20-Jun-2009	CRS 100%	115J10	Pending
17	YC81395	CC	17	20-Jun-2009	CRS 100%	115J10	Pending
18	YC81396	CC	18	20-Jun-2009	CRS 100%	115J10	Pending
19	YC81397	CC	19	20-Jun-2009	CRS 100%	115J10	Pending
20	YC81398	CC	20	20-Jun-2009	CRS 100%	115J10	Pending
21	YC81399	CC	21	20-Jun-2009	CRS 100%	115J10	Pending
22	YC81400	CC	22	20-Jun-2009	CRS 100%	115J10	Pending
23	YC81401	CC	23	20-Jun-2009	CRS 100%	115J10	Pending
24	YC81402	CC	24	20-Jun-2009	CRS 100%	115J10	Pending
25	YC81403	CC	25	20-Jun-2009	CRS 100%	115J10	Pending
26	YC81404	CC	26	20-Jun-2009	CRS 100%	115J10	Pending
27	YC81405	CC	27	20-Jun-2009	CRS 100%	115J10	Pending
28	YC81406	CC	28	20-Jun-2009	CRS 100%	115J10	Pending
29	YC81407	CC	29	20-Jun-2009	CRS 100%	115J10	Pending
30	YC81408	CC	30	20-Jun-2009	CRS 100%	115J10	Pending
31	YC81409	CC	31	20-Jun-2009	CRS 100%	115J10	Pending
32	YC81410	CC	32	20-Jun-2009	CRS 100%	115J10	Pending
33	YC81411	CC	33	20-Jun-2009	CRS 100%	115J10	Pending
34	YC81412	CC	34	20-Jun-2009	CRS 100%	115J10	Pending
35	YC81413	CC	35	20-Jun-2009	CRS 100%	115J10	Pending
36	YC81414	CC	36	20-Jun-2009	CRS 100%	115J10	Pending
37	YC81415	CC	37	20-Jun-2009	CRS 100%	115J10	Pending
38	YC81416	CC	38	20-Jun-2009	CRS 100%	115J10	Pending
39	YC81417	CC	39	20-Jun-2009	CRS 100%	115J10	Pending
40	YC81418	CC	40	20-Jun-2009	CRS 100%	115J10	Pending
41	YC81419	CC	41	20-Jun-2009	CRS 100%	115J10	Pending
42	YC81420	CC	42	20-Jun-2009	CRS 100%	115J10	Pending
43	YC81421	CC	43	20-Jun-2009	CRS 100%	115J10	Pending
44	YC81422	CC	44	20-Jun-2009	CRS 100%	115J10	Pending
45	YC81423	CC	45	20-Jun-2009	CRS 100%	115J10	Pending
46	YC81424	CC	46	20-Jun-2009	CRS 100%	115J10	Pending
47	YC81425	CC	47	20-Jun-2009	CRS 100%	115J10	Pending

"CC" MINERAL CLAIMS (94) - WHITEHORSE MINING DISTRICT

No.	Grant Number	Claim Name	Claim No.	Expiry Date	Owner	NTS Map Sheet	Status
48	YC81426	CC	48	20-Jun-2009	CRS 100%	115J10	Pending
49	YC81427	CC	49	20-Jun-2009	CRS 100%	115J10	Pending
50	YC81428	CC	50	20-Jun-2009	CRS 100%	115J10	Pending
51	YC81429	CC	51	20-Jun-2009	CRS 100%	115J10	Pending
52	YC81430	CC	52	20-Jun-2009	CRS 100%	115J10	Pending
53	YC81431	CC	53	20-Jun-2009	CRS 100%	115J10	Pending
54	YC81432	CC	54	20-Jun-2009	CRS 100%	115J10	Pending
55	YC81433	CC	55	20-Jun-2009	CRS 100%	115J10	Pending
56	YC81434	CC	56	20-Jun-2009	CRS 100%	115J10	Pending
57	YC81435	CC	57	20-Jun-2009	CRS 100%	115J10	Pending
58	YC81436	CC	58	20-Jun-2009	CRS 100%	115J10	Pending
59	YC81437	CC	59	20-Jun-2009	CRS 100%	115J10	Pending
60	YC81438	CC	60	20-Jun-2009	CRS 100%	115J10	Pending
61	YC81439	CC	61	20-Jun-2009	CRS 100%	115J10	Pending
62	YC81440	CC	62	20-Jun-2009	CRS 100%	115J10	Pending
63	YC81441	CC	63	20-Jun-2009	CRS 100%	115J10	Pending
64	YC81442	CC	64	20-Jun-2009	CRS 100%	115J10	Pending
65	YC81443	CC	65	20-Jun-2009	CRS 100%	115J10	Pending
66	YC81444	CC	66	20-Jun-2009	CRS 100%	115J10	Pending
67	YC81445	CC	67	20-Jun-2009	CRS 100%	115J10	Pending
68	YC81446	CC	68	20-Jun-2009	CRS 100%	115J10	Pending
69	YC81447	CC	69	20-Jun-2009	CRS 100%	115J10	Pending
70	YC81448	CC	70	20-Jun-2009	CRS 100%	115J10	Pending
71	YC81449	CC	71	20-Jun-2009	CRS 100%	115J10	Pending
72	YC81450	CC	72	20-Jun-2009	CRS 100%	115J10	Pending
73	YC81451	CC	73	20-Jun-2009	CRS 100%	115J10	Pending
74	YC81452	CC	74	20-Jun-2009	CRS 100%	115J10	Pending
75	YC81453	CC	75	20-Jun-2009	CRS 100%	115J10	Pending
76	YC81454	CC	76	20-Jun-2009	CRS 100%	115J10	Pending
77	YC81455	CC	77	20-Jun-2009	CRS 100%	115J10	Pending
78	YC81456	CC	78	20-Jun-2009	CRS 100%	115J10	Pending
79	YC81457	CC	79	20-Jun-2009	CRS 100%	115J10	Pending
80	YC81458	CC	80	20-Jun-2009	CRS 100%	115J10	Pending
81	YC81459	CC	81	20-Jun-2009	CRS 100%	115J10	Pending
82	YC81460	CC	82	20-Jun-2009	CRS 100%	115J10	Pending
83	YC81461	CC	83	20-Jun-2009	CRS 100%	115J10	Pending
84	YC81462	CC	84	20-Jun-2009	CRS 100%	115J10	Pending
85	YC81463	CC	85	20-Jun-2009	CRS 100%	115J10	Pending
86	YC81464	CC	86	20-Jun-2009	CRS 100%	115J10	Pending
87	YC81465	CC	87	20-Jun-2009	CRS 100%	115J10	Pending
88	YC81466	CC	88	20-Jun-2009	CRS 100%	115J10	Pending
89	YC81467	CC	89	20-Jun-2009	CRS 100%	115J10	Pending
90	YC81468	CC	90	20-Jun-2009	CRS 100%	115J10	Pending
91	YC81469	CC	91	20-Jun-2009	CRS 100%	115J10	Pending
92	YC81470	CC	92	20-Jun-2009	CRS 100%	115J10	Pending
93	YC81471	CC	93	20-Jun-2009	CRS 100%	115J10	Pending
94	YC81472	CC	94	20-Jun-2009	CRS 100%	115J10	Pending

"BRIT" MINERAL CLAIMS (63) - WHITEHORSE MINING DISTRICT

No.	Grant Number	Claim Name	Claim No.	Expiry Date	Owner	NTS Map Sheet	Status
1	YC81316	BRIT	1	20-Jun-2009	CRS 100%	115J15	Pending
2	YC81317	BRIT	2	20-Jun-2009	CRS 100%	115J15	Pending
3	YC81318	BRIT	3	20-Jun-2009	CRS 100%	115J15	Pending
4	YC81319	BRIT	4	20-Jun-2009	CRS 100%	115J15	Pending
5	YC81320	BRIT	5	20-Jun-2009	CRS 100%	115J15	Pending
6	YC81321	BRIT	6	20-Jun-2009	CRS 100%	115J15	Pending
7	YC81322	BRIT	7	20-Jun-2009	CRS 100%	115J15	Pending
8	YC81323	BRIT	8	20-Jun-2009	CRS 100%	115J15	Pending
9	YC81324	BRIT	9	20-Jun-2009	CRS 100%	115J15	Pending
10	YC81325	BRIT	10	20-Jun-2009	CRS 100%	115J15	Pending
11	YC81326	BRIT	11	20-Jun-2009	CRS 100%	115J15	Pending
12	YC81327	BRIT	12	20-Jun-2009	CRS 100%	115J15	Pending
13	YC81328	BRIT	13	20-Jun-2009	CRS 100%	115J15	Pending
14	YC81329	BRIT	14	20-Jun-2009	CRS 100%	115J15	Pending
15	YC81330	BRIT	15	20-Jun-2009	CRS 100%	115J15	Pending
16	YC81331	BRIT	16	20-Jun-2009	CRS 100%	115J15	Pending
17	YC81332	BRIT	17	20-Jun-2009	CRS 100%	115J15	Pending
18	YC81333	BRIT	18	20-Jun-2009	CRS 100%	115J15	Pending
19	YC81334	BRIT	19	20-Jun-2009	CRS 100%	115J15	Pending
20	YC81335	BRIT	20	20-Jun-2009	CRS 100%	115J15	Pending
21	YC81336	BRIT	21	20-Jun-2009	CRS 100%	115J15	Pending
22	YC81337	BRIT	22	20-Jun-2009	CRS 100%	115J15	Pending
23	YC81338	BRIT	23	20-Jun-2009	CRS 100%	115J15	Pending
24	YC81339	BRIT	24	20-Jun-2009	CRS 100%	115J15	Pending
25	YC81340	BRIT	25	20-Jun-2009	CRS 100%	115J15	Pending
26	YC81341	BRIT	26	20-Jun-2009	CRS 100%	115J15	Pending
27	YC81342	BRIT	27	20-Jun-2009	CRS 100%	115J15	Pending
28	YC81343	BRIT	28	20-Jun-2009	CRS 100%	115J15	Pending
29	YC81344	BRIT	29	20-Jun-2009	CRS 100%	115J15	Pending
30	YC81345	BRIT	30	20-Jun-2009	CRS 100%	115J15	Pending
31	YC81346	BRIT	31	20-Jun-2009	CRS 100%	115J15	Pending
32	YC81347	BRIT	32	20-Jun-2009	CRS 100%	115J15	Pending
33	YC81348	BRIT	33	20-Jun-2009	CRS 100%	115J15	Pending
34	YC81349	BRIT	34	20-Jun-2009	CRS 100%	115J15	Pending
35	YC81350	BRIT	35	20-Jun-2009	CRS 100%	115J15	Pending
36	YC81351	BRIT	36	20-Jun-2009	CRS 100%	115J15	Pending
37	YC81352	BRIT	37	20-Jun-2009	CRS 100%	115J15	Pending
38	YC81353	BRIT	38	20-Jun-2009	CRS 100%	115J15	Pending
39	YC81354	BRIT	39	20-Jun-2009	CRS 100%	115J15	Pending
40	YC81355	BRIT	40	20-Jun-2009	CRS 100%	115J15	Pending
41	YC81356	BRIT	41	20-Jun-2009	CRS 100%	115J15	Pending
42	YC81357	BRIT	42	20-Jun-2009	CRS 100%	115J15	Pending
43	YC81358	BRIT	43	20-Jun-2009	CRS 100%	115J15	Pending
44	YC81359	BRIT	44	20-Jun-2009	CRS 100%	115J15	Pending
45	YC81360	BRIT	45	20-Jun-2009	CRS 100%	115J15	Pending

"BRIT" MINERAL CLAIMS (63) - WHITEHORSE MINING DISTRICT

No.	Grant Number	Claim Name	Claim No.	Expiry Date	Owner	NTS Map Sheet	Status
46	YC81361	BRIT	46	20-Jun-2009	CRS 100%	115J15	Pending
47	YC81362	BRIT	47	20-Jun-2009	CRS 100%	115J15	Pending
48	YC81363	BRIT	48	20-Jun-2009	CRS 100%	115J15	Pending
49	YC81364	BRIT	49	20-Jun-2009	CRS 100%	115J15	Pending
50	YC81365	BRIT	50	20-Jun-2009	CRS 100%	115J15	Pending
51	YC81366	BRIT	51	20-Jun-2009	CRS 100%	115J15	Pending
52	YC81367	BRIT	52	20-Jun-2009	CRS 100%	115J15	Pending
53	YC81368	BRIT	53	20-Jun-2009	CRS 100%	115J15	Pending
54	YC81369	BRIT	54	20-Jun-2009	CRS 100%	115J15	Pending
55	YC81370	BRIT	55	20-Jun-2009	CRS 100%	115J15	Pending
56	YC81371	BRIT	56	20-Jun-2009	CRS 100%	115J15	Pending
57	YC81372	BRIT	57	20-Jun-2009	CRS 100%	115J15	Pending
58	YC81373	BRIT	58	20-Jun-2009	CRS 100%	115J15	Pending
59	YC81374	BRIT	59	20-Jun-2009	CRS 100%	115J15	Pending
60	YC81375	BRIT	60	20-Jun-2009	CRS 100%	115J15	Pending
61	YC81376	BRIT	61	20-Jun-2009	CRS 100%	115J15	Pending
62	YC81377	BRIT	62	20-Jun-2009	CRS 100%	115J15	Pending
63	YC81378	BRIT	63	20-Jun-2009	CRS 100%	115J15	Pending

1.7 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

1.7.1 Accessibility

The site is currently accessible year-round only by air. Fixed wing aircraft can use the existing 760 m airstrip.

Access in the past was either by a 225 km winter road from Burwash Landing or a slightly shorter winter route along the Freegold Road from Carmacks. Access has also been gained in the past by river barge on the Yukon River from Minto or other communities along the river. The barge landed at a point 19 km from the site near the mouth of Britannia Creek.

Associated Engineers examined various route options for a year-round access road. All the options are considered viable however the selected option for the purpose of this study was a new, 187 km, unpaved road from the Alaskan Highway at a point about 48 km northwest of Burwash Landing. The route is termed the “Onion Creek” route.

Either a winter road or barge service will provide early access for construction equipment, camp construction and initial equipment.

The project plan includes a new airstrip and building for employee transport to and from the site. The airstrip is planned to be located about 35 km southeast of the mine on a flat area adjacent to the proposed access road.

1.7.2 Climate

The mean annual temperature for the area is approximately -5.5°C with a summer mean of 10.5°C and a winter mean of -23°C. Mean annual precipitation ranges between 300-450 mm.

Snow survey data for the years 1977 to 1994 (based on information from Hallam Knight Piésold, Casino Project, Data Report 1993-1995, March 1997) showed the maximum snow depth was 97 cm containing the equivalent of 225 mm of water in April 1991. Average depths (equivalent H_2O) by month were: February 1: 52 cm (73 mm), March 1: 62 cm (107 mm), April 1: 65 cm (126 mm), May 1: 55 cm (128 mm), and May 15: 27 cm (74 mm).

Actual climate data is sparse. No site measurements exist for evaporation, frost depth or wind speed and direction. Design specifications for evaporation used 400 mm which is 90% of measured lake evaporation from a station at Williams Creek, Yukon, near Carmacks. Western Copper plans to install an automated weather station at the site in the third quarter of this year.

Other design specifications came from the Canadian National Building Code, Dawson Creek.

1.7.3 Local Resources

No local commercial or human resources exist to support mine development. Whitehorse, the Capital of the Yukon Territory, is about 300 km due southeast of the property. It is the nearest commercial center, the point of departure for access by commercial aircraft and the location of lodging and other commercial service.

The village of Pelly Crossing on the Klondike Highway is the nearest community, some 115 km to the east of the property. The village of Carmacks, also on the Klondike highway, is approximately 150 km to the southeast.

1.7.4 Infrastructure

Other than the airstrip, little infrastructure currently exists.

Mine development will require development of water, sanitation, transportation, power, and communications resources and systems. Figures 1.7-1 and 1.7-2 in the Illustrations section (1.25) at the end of this report show an overall site plan.

1.7.5 Physiography

The Casino property is located in the Dawson Range, a north-westerly trending belt of well rounded ridges and hills that reach a maximum elevation of about 1,675 m. The hills rise above the Yukon Plateau, most of which as been peneplaned at about 1,250 m and deeply incised by mature dendritic drainages that are part of the Yukon River watershed. Although the Dawson Range escaped Pleistocene continental glaciation, minor alpine glaciation has produced a few small cirques and terminal moraines.

Local elevations range from 1,580 m on a ridge along the western property boundary to 430 m on the banks of the Yukon River at Britannia Creek.

The deposit area is situated on a small divide. The northern part of the property drains to Canadian Creek and Britannia Creek into the Yukon River. Drainage from the southern part of the property flows southward via Casino Creek to Dip Creek to the Donjek River and thence northward to the Yukon River.

Outcrop is rare on the property being restricted to a few widely spaced tors on ridge crests. Soil development is variable ranging from coarse talus and immature soil horizons at higher elevations to a more mature soil profile and thick organic accumulations on the valley floors.

1.8 HISTORY

The first known recording of a lode mineral claim in the area was in 1917. Discovery of silver-lead veins 1930's led to further exploration and subsequent discovery of the large copper and molybdenum porphyry deposit on Patton Hill in 1969.

Following the porphyry discovery, various parties including Brameda Resources, Quintana Minerals and Teck Corporation drilled the property. In 1991 Archer, Cathro and Associates, Ltd. acquired the property and did more drilling.

In 1992 Pacific Sentinel Gold Corp. acquired the property and commenced a major drilling program from 1992 to 1995. In addition, they performed a considerable amount of metallurgical, geotechnical and environmental work and completed a scoping study in 1995.

First Trimark Resources and CRS Copper Resources obtained the property and using the Pacific Sentinel Gold data published a Qualifying Report on the property in 2003 to bring the resource estimate into compliance with National Instrument 43-101 requirements. The two firms combined to form Lumina Copper Corporation in 2004. An update of the Qualifying Report was issued in 2004.

Western Copper Corporation acquired Lumina Copper Corporation, and the Casino Deposit, in November 2006.

1.9 GEOLOGICAL SETTING

The Casino deposit occurs in an overlapping zone of the Yukon Cataclastic Terrane to the north and the Yukon Crystalline Terrane to the south (Templeman-Kluit, 1976). An elongate band of ultramafic rocks 1 km north of the Casino deposit may occur along a major tectonic suture, which separates the two terranes. The southern terrane contains the Dawson Range Batholith with scattered roof-pendants and blocks of the Yukon Metamorphic Complex. The northern terrane is dominated by rocks of the Yukon Metamorphic Complex with scattered intrusions of the Coffee Creek Suite which are petrographically distinct from the Dawson Range Batholith. The latter are similar to rocks near Mount Nansen dated at 104 Ma.

The regional geology is illustrated in Figure 1.9-1.

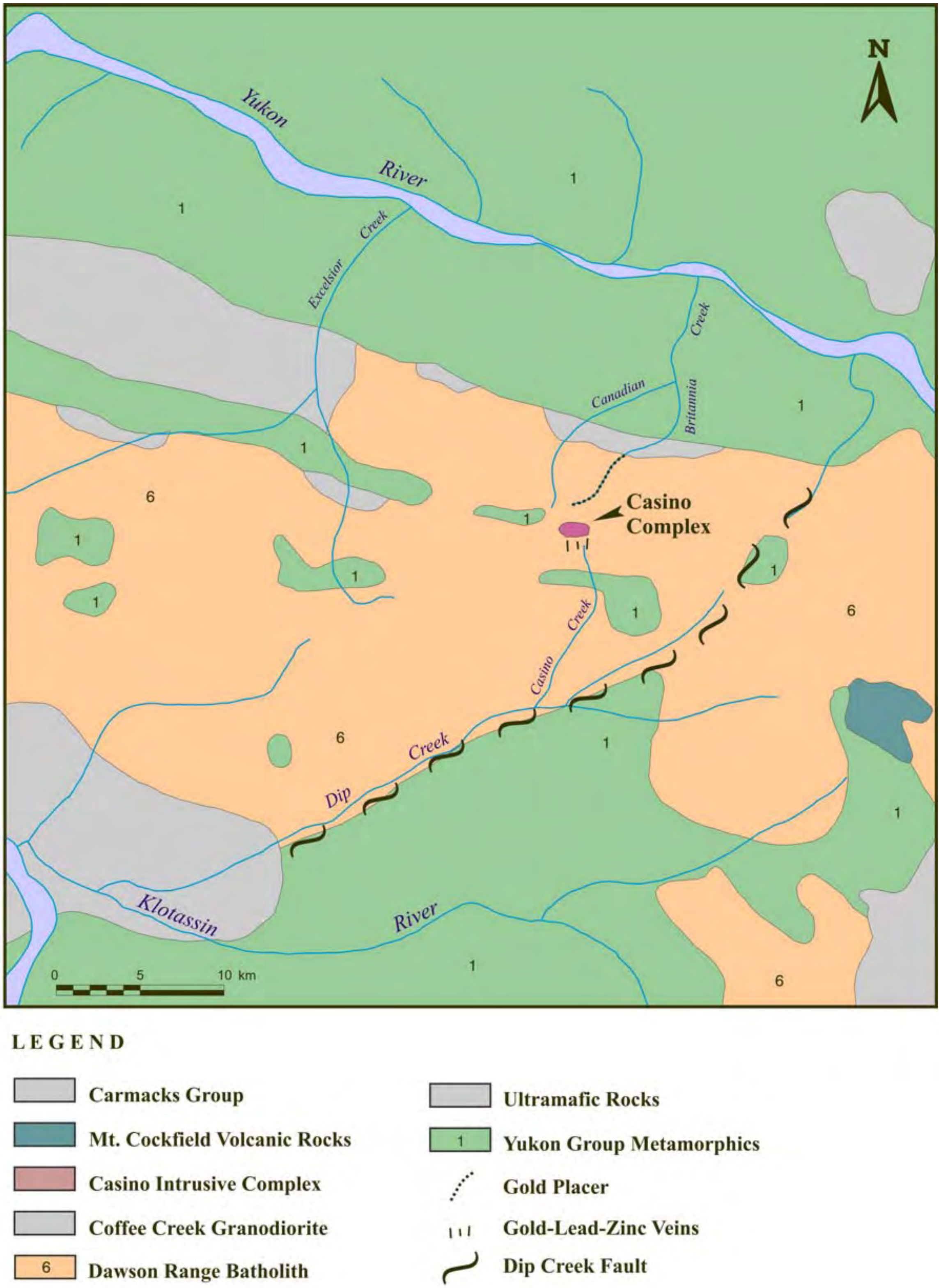


Figure 1.9-1: Regional Geology

Table 1.9-1 summarizes the major geological units with isotopic ages. All isotopic dates are based on U-Pb ratios in zircons analyzed by J.R. Mortensen.

Table 1.9-1: Major Geological Units

	Isotopic Age
Latite (Tertiary/Upper Cretaceous)	
CASINO INTRUSIONS (Upper Cretaceous)	
Microbreccia	
Patton Porphyry	(72-74 Ma)
Intrusion Breccia	
Quartz Monzonite	
DAWSON RANGE BATHOLITH (Middle) (Cretaceous)	
Quartz Diorite, Granodiorite	(102-104 Ma)
YUKON METAMORPHIC COMPLEX (Paleozoic)	
Gneiss, meta-diorite/amphibolite, quartzite	

Rocks of the Yukon Metamorphic Complex are dominated by quartz-biotite-plagioclase gneiss, metamorphosed granodiorite to quartz diorite, and less abundant quartzite and metadiorite / amphibolite. All are complexly deformed. Original deformation probably was during the Late Triassic to Early Jurassic but may have been as early as Late Paleozoic era. Weaker deformation probably occurred during the intrusion of the west-northwest trending Dawson Range Batholith.

The Dawson Range Batholith is dominated by medium to coarse grained quartz diorite to granodiorite, with much less fine-to locally medium-grained diorite. It contains scattered to locally abundant, centimetre sized, meta-diorite xenoliths of the Yukon Metamorphic Complex.

The Casino Intrusions are stocks up to 5 km across which intrude the Dawson Range Batholith and are concentrated in a west-northwest trending belt along the Big Creek Lineament and its northwestern extension. Most Casino intrusions are dominated by biotite-bearing to leucocratic granodiorite to quartz monzonite; some are hypabyssal porphyritic dacite (Patton Porphyry). Many types of hydrothermal alteration and sulphide precious-metal mineralization occur in and adjacent to the genetically related intrusions.

1.10 DEPOSIT TYPES

Geology of the Casino Deposit

The Casino deposit is centred in a complex of quartz monzonites, intrusion breccias, shallow porphyritic intrusions, and a central breccia to microbreccia body. Microbreccia refers to this central breccia and other smaller satellite bodies that are characterised by a microbrecciated groundmass with broken quartz crystals. All of the Casino Intrusions are of Upper Cretaceous age (J. R. Mortensen, pers. comm., 1993). The complex is hosted by the Middle Cretaceous Dawson Range Batholith.

The geology of the deposit is illustrated in plan view (Figure 1.10-1) and in cross-sections looking north and west (Figure 1.10-2).

Yukon Metamorphic Complex

Rocks of the Yukon Metamorphic Complex occur mainly as fragments in intrusion breccias in the northern and northeastern parts of the Casino deposit and locally as roof pendants in the Dawson Range Batholith in the same area. The most common rock type of this complex is mafic gneiss, dominated by plagioclase, biotite, and quartz. Less abundant types include meta-diorite/amphibolite, quartz-rich and intermediate gneiss, and quartzite.

Dawson Range Batholith

The Dawson Range Batholith is the main country rock to the deposit, and is dominantly granodiorite in composition. Quartz monzonite and diorite are less abundant. The two main phases of granodiorite are: a hornblende-bearing phase, mainly to the west, south, and east of the deposit, and a biotite-hornblende-bearing phase to the north. Early formed diorite is concentrated in the north and northeast, particularly east of Casino Fault. Some intrusions are foliated near their margins, particularly in the north and in the block east of the Casino Fault.

Granodiorite is characterized by scattered hornblende phenocrysts averaging 0.5 to 1.2 cm long. It commonly contains 10 to 20% mafic minerals, with hornblende more abundant than biotite except in the biotite-hornblende phase to the north. Subhedral to euhedral plagioclase commonly show prominent compositional growth-zoning from andesine cores to oligoclase rims. Quartz and K-feldspar are interstitial to plagioclase. Quartz forms interlocking aggregates of slightly to moderately strained grains. Minor patches of myrmekite occur in plagioclase grains adjacent to those of K-feldspar. Quartz monzonite contains less abundant mafic minerals (2-10%), lacks hornblende phenocrysts, and commonly contains biotite in more abundance than hornblende.

The diorite is dominated by andesine and hornblende in subequal proportions; however, in places, primary biotite is more abundant than hornblende. Some finer-grained diorite is similar texturally to meta-diorite of the Yukon Metamorphic Complex.

On the southern margin of the deposit granodiorite commonly is brecciated. Breccia ranges from crackle breccia to more intensely deformed cataclastic breccia. The latter contains 5 to 50% ragged fragments of the host rock ranging from 0.3 to 5 cm in size in an extremely fine-to very fine-grained, granulated groundmass dominated by plagioclase and quartz. Fragment size generally decreases with increasing degree of brecciation. In places, the cataclastic breccia grades into the microbreccia core of the deposit; elsewhere, it is truncated by the microbreccia.

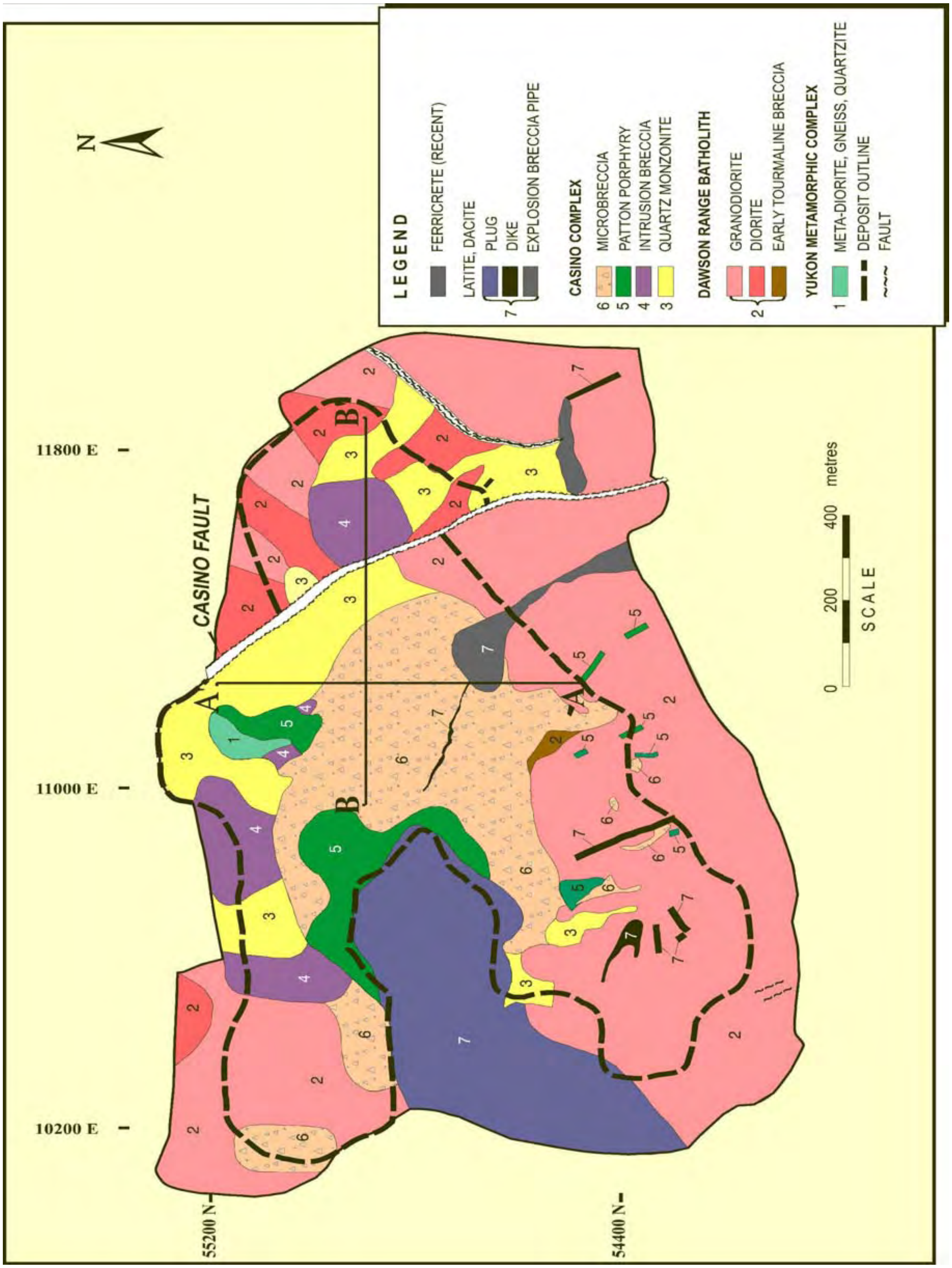


Figure 1.10-1: Casino Property Geology - Plan

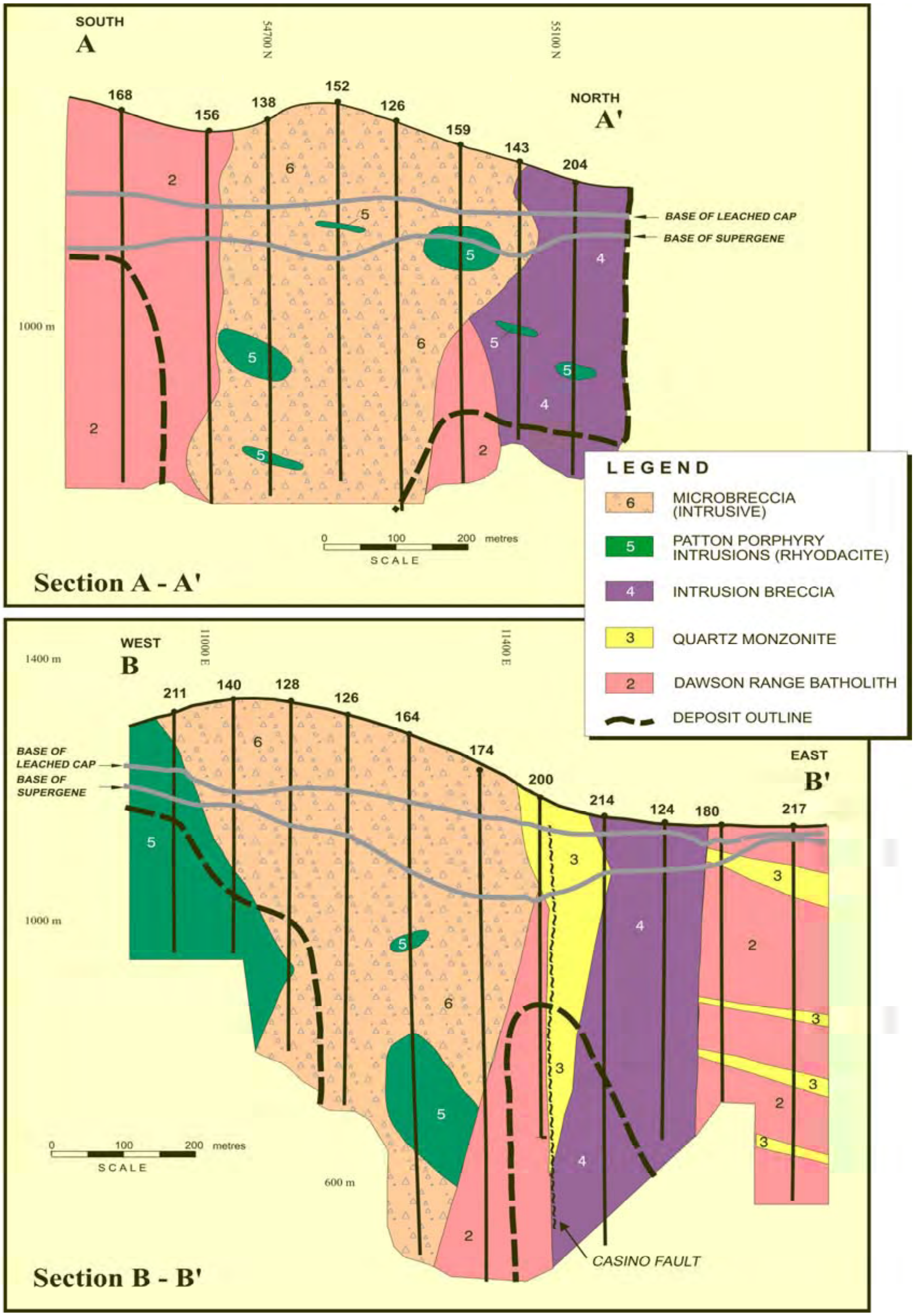


Figure 1.10-2: Casino Property Geology - Cross Section

Casino Intrusive Complex

Leucocratic quartz monzonite is prominent just west of the Casino Fault. The dominant phase is fine-to locally medium- grained, with approximately equal amounts of quartz, plagioclase, and K-feldspar. These minerals form aggregates of anhedral grains. Locally, this unit has aplitic to graphic textures that suggest the rock crystallized from a near eutectic liquid. The aplite contains minor zones of pegmatite. Medium to coarse-grained quartz monzonite forms scattered small bodies with a similar texture.

Heterolithic intrusion breccia occurs in the northern and eastern part of the deposit. It contains fragments of Dawson Range Batholith and Yukon Metamorphic rocks in a matrix of fine-grained, leucocratic quartz monzonite. A few fragments are of Patton Porphyry, suggesting the presence of an early variety of that rock type. The amount of groundmass ranges from 2 to 40%. Fragments range widely in size from a few centimetres to a few metres. In drill core, intervals of diorite and granodiorite cut by scattered quartz monzonite dikes may be very large rafts in the breccia or may represent country rock cut by abundant dikes of quartz monzonite. The intrusion breccia was formed along the margin of the quartz monzonite intrusion by stoping of blocks of wall rocks. Along the contact, intrusion breccia grades into quartz monzonite containing 1 to 5% xenoliths, mainly less than 10 cm in size.

The Patton Porphyry (72-74 Ma) represents two or more episodes of high-level intrusion of porphyritic hypabyssal dacite to rhyodacite. The main body of Patton Porphyry, a few hundred metres across, occurs northwest of Patton Hill. Abundant Patton Porphyry fragments occur in the adjacent microbreccia. The contacts between the Patton Porphyry and the microbreccia are variable and range from sharply intrusive to gradational. Elsewhere, Patton Porphyry forms many discontinuous dikes ranging from a few cm to 20 m wide; these cut Casino quartz monzonite and less commonly rocks of the Dawson Range Batholith.

Most types of Patton Porphyry are porphyritic with plagioclase and biotite phenocrysts set in an aphanitic, commonly medium to dark green groundmass of latite to rhyodacite composition. Plagioclase phenocrysts normally comprise 10 to 30% of the rock and range in size from 2 to 7 mm, with some up to 2.5 cm in length. Biotite phenocrysts (1 to 5%) range from 2 to 3 mm across. Quartz phenocrysts (0 to 5%) are usually 3 to 5 mm in size. Some chloritic aggregates may be altered hornblende grains.

At least two ages of dominantly plagioclase- and biotite-phyric Patton Porphyry are present; the earlier type contains less abundant, commonly ragged plagioclase phenocrysts in a more chloritic groundmass and the later type contains more abundant, commonly subhedral to euhedral plagioclase phenocrysts in a vitreous, less chloritic groundmass. Episodic Patton Porphyry magmatism probably occurred before, during, and after the main stage of hydrothermal alteration and mineralization. Much of the Patton Porphyry is poorly mineralized, but commonly is strongly potassically altered, suggesting that it formed during and after the main stage of sulphide mineralization in the core of the deposit.

One Patton Porphyry intrusion intersected at depth in drill holes in the southeastern part of the core of the deposit contains 10 to 15% K-feldspar and 5 to 7% quartz phenocrysts in a rhyodacitic groundmass. It is well mineralized, and may be closely associated genetically with the hydrothermal fluids.

The intrusive microbreccia forms an irregularly shaped, subvertical pipe approximately 400 m in diameter in the core of the deposit and also forms a series of smaller irregular bodies to the southwest. It contains angular fragments of quartz grains and wall rocks in an extremely fine-grained groundmass of rhyodacitic to rhyolitic composition but of uncertain genetic origin. In places the groundmass of the microbreccia has a very fine-grained, interlocking igneous texture and elsewhere it resembles milled rock flour. Locally strong potassic alteration destroyed primary textures. Quartz forms 10 to 20%, generally unstrained crystals and crystal fragments commonly 1 to 2 mm in size. A few have subhedral outlines, suggesting that they may represent early- formed quartz phenocrysts in a silica-saturated magma. They are similar texturally to quartz phenocrysts in some of the Patton Porphyry intrusions.

Lithic fragments in the microbreccia, ranging in size from 0.5 to 5 cm, commonly belong to the adjacent wallrock type; Dawson Range Batholith in the south, Patton Porphyry in the northwest, mixed fragments of heterolithic intrusion breccia and Yukon Group metamorphic rocks in the north and minor fragments of quartz monzonite in the east. This suggests only limited fragment transport and mixing in the microbreccia. Large blocks of non-brecciated Patton Porphyry, some exceeding 100 m in size, occur in the main body of microbreccia. Many lithic fragments are earlier breccias; some clasts contain mineralized quartz veins and some host sulphides indicating multiple episodes of brecciation and mineralization.

Late dikes, sills, and an explosion-breccia pipe, of probable late Cretaceous/Tertiary age, are latitic to dacitic in composition. Most dikes are steeply dipping and strike between 130° and 160°. Some dikes are of slightly porphyritic latite containing 5-10% plagioclase phenocrysts, mainly less than 1.0 mm in size, in an aphanitic to very fine-grained groundmass dominated by plagioclase and K-feldspar. Some borders of the dikes are chilled and some show flow-banded or lenticular structures near the contact. On the east side of Casino Fault a latite dike shows vesiculation, with quartz- filled amygdules stretched parallel to steep contacts. A large plug of porphyritic latite intrudes the western portion of the Casino Intrusive Complex.

Some late dikes in the south-central part of the deposit, which somewhat resemble the Patton Porphyry in texture, contain 2 to 5% coarse quartz phenocrysts and 1 to 3% plagioclase phenocrysts in an aphanitic latite groundmass.

The explosion-breccia is an irregular, vertical pipe up to 100 m across near the southeast margin of the central microbreccia body. It contains subrounded fragments, mainly of altered microbreccia, and less commonly of late, quartz-phyric dikes and Dawson Range Batholith in a groundmass, that ranges from rock flour (probably formed mainly by granulation of the latite) to very fine-grained latite. The breccia grades into a large latite dike which shows strong sheeting parallel to its contacts. The late dikes, sills and breccias were intruded after the main hydrothermal event and contain only minor base- and precious- metal mineralization, most of which is contained in mineralized and altered clasts of microbreccia.

1.11 MINERALIZATION

Leached Cap and Mineralization

The deposit is weathered to a depth of up to 300 m producing a well-defined Leached Cap (Figure 1.10-2). The Leached Cap averages 70 m thick and is characterized by boxwork textures partly filled by earthy limonite, jarosite, goethite and hematite. This (leached) copper-oxide zone is closely related to present topography and is best developed on well drained slopes, where oxidation of earlier secondary copper sulphides occurs above the water table.

In the Leached Cap copper grades range from 0.01 to 0.15% and average 0.08%. Much of this residual copper occurs in goethite in boxwork cavities, which represent leached sulphides. Locally, some goethite contains up to 5% copper (Coe, 1994). Where copper in the Leached Cap exceeds 0.1%, grades are caused by small pods of perched oxide, Supergene or Hypogene mineralization.

Gold is enriched in the Leached Cap relative to the Supergene Sulphide and Hypogene zones, thus it is often referred to as the Oxide Gold zone. This gold enrichment is in part caused by the lower specific gravity of the Leached Cap relative to the other zones. However, the reduced specific gravity does not account fully for the higher gold grades, which must also be somewhat attributed to supergene processes.

Supergene Alteration and Mineralization

Supergene alteration at Casino consists of a poorly defined upper copper oxide zone that overlies a better-defined copper sulphide zone.

The poorly defined Supergene copper oxide zone averages 10 m in thickness (Figure 1.10-2), separates the Leached Cap from the Supergene Sulphide zone and forms a few perched bodies in the Leached Cap.

Copper-bearing minerals in the Supergene Oxide zone include chalcanthite, malachite and brochantite and minor azurite, tenorite, cuprite, and neotocite.

Thickness and grade of the Supergene copper sulphide zone varies widely, in part controlled by the distribution of pyrite, fracture density, and the erosion surface at the time of leaching. The Supergene Sulphide zone averages 60 m in thickness and is characterized by a high pyrite content of the phyllic zone, which promotes leaching. Thus, secondary enrichment zones are thickest near the contact of the potassic and phyllic alteration zones.

The Supergene Sulphide zone has an overall copper enrichment that is almost double the grade of the Hypogene zone (0.43% Cu versus 0.23% Cu). Chalcopyrite, bornite, and tetrahedrite are slightly to moderately supergene altered along grain borders and fractures to chalcocite, digenite, and/or covellite. Chalcocite also forms coatings on pyrite grains and pyrite clusters. Locally, chalcocite/digenite extends on large fractures well into the Hypogene zone. In late fractures of the Supergene and upper part of the Hypogene zones, native copper forms flakes, coatings and discrete grains up to 1.5 cm across. Generally, molybdenite was unaffected by supergene processes, but locally was replaced by ferrimolybdite.

Table 1.11-1 summarizes the main minerals identified in the Leached Cap and Supergene zones.

Table 1.11-1: Leached Cap & Supergene Minerals

Zone	Minerals Present	Average Thickness
Leached Cap	jarosite, goethite, hematite, ferrimolybdite	70 metres
Supergene Oxide	chalcanthite, brochantite, malachite, azurite, tenorite, cuprite, neotocite, copper WAD native copper, copper-bearing goethite	10 metres
Supergene Sulphide	digenite, chalcocite, minor covellite, bornite, copper-bearing goethite	60 metres

Hypogene Mineralization

The bulk of the sulphide mineralization occurs in and adjacent to the intrusion breccia and microbreccia. Molybdenite is concentrated moderately in the core of the deposit, and chalcopyrite is concentrated moderately towards the periphery, just inside the potassic / phyllic alteration contact.

Chalcopyrite occurs in veins, disseminations, and irregular patches. On the west side of the Casino Fault disseminated chalcopyrite is more abundant than chalcopyrite in veins and veinlets in microbreccia and granodiorite. On the east side of Casino Fault the distribution of chalcopyrite is more strongly controlled by brittle deformation. In this zone, the dominant style of mineralization consists of fracture controlled chalcopyrite, open space chalcopyrite fillings, and chalcopyrite with ankeritic carbonate and gypsum, as breccia matrix. The pyrite/chalcopyrite ratio ranges widely from less than 2:1 in much of the potassic zone to greater than 20:1 in the phyllic zone.

Bornite and tetrahedrite are rare primary minerals occurring as coarse inter-growths in chalcopyrite.

Molybdenite occurs as discrete flakes, clusters of flakes, and selvages in early quartz veins, and is not generally intergrown with other sulphides.

Native gold occurs as 50 to 70 micron free grains in quartz and as 1 to 15 micron inclusions in fractures in pyrite and chalcopyrite grains.

Within the Hypogene zone late stage, commonly vuggy, polymetallic base- metal veins were introduced into roughly parallel, steeply dipping fractures trending 150° to 170°. The veins vary widely in mineralogy. Many contain abundant sphalerite, galena, and calcite/ankerite, less abundant quartz, tetrahedrite, and chalcopyrite, and minor bismuth-bearing minerals. Commonly tetrahedrite and chalcopyrite are intimately intergrown. Some veins are dominated by chalcopyrite and/or pyrite. Typically they are geo-chemically anomalous in any or all of Ag, As, Au, Bi, Cu, Cd, Mn, Pb, Sb, and Zn, and locally in W.

1.12 EXPLORATION

Exploration included multi-element soil geochemistry, geological mapping, geophysical surveys, trenching and drilling. Drill targets were located primarily on the basis of coincident copper-molybdenum geochemical anomalies (Archer and Main, 1971).

Geophysical tools were used as target confirmation. Induced polarization surveys showed an area of high chargeability coincident with the phyllic halo of the porphyry deposit and an area of high resistivity coincident with the thickest part of the Leached Cap and the milled breccia pipe. Intermediate resistivity and low to moderate chargeability values over the deposit and broad areas to the north and west coincide with copper-gold-molybdenum soil geochemical anomalies. The zone of potassic alteration produced a strong ground-magnetic high, as did an area of magnetite-bearing diorite to the northwest. Drilling to test the geochemical and geophysical anomalies centred on Patton Hill partly defined a bulk tonnage porphyry deposit. Further potential below and around the main area drilled was indicated.

1.13 DRILLING

Drilling prior to 1992 (Figure 1.13-1) consisted of reverse circulation drilling and NQ diamond drilling. There is little documentation that specifically focused on this early drilling, its specifications or challenges. After the acquisition of Casino Silver Mines Limited by Archer Cathro & Associates, followed by Pacific Sentinel Gold Corp., the drilling is well documented.

During the intense campaigns from 1992 through 1994, (Figure 1.13-2) drilling was contracted to E. Caron Drilling Ltd. of Whitehorse. Up to six diamond drills were utilized. The 1994 drilling program fulfilled a variety of purposes: infill, delineation, geotechnical, structural and waste rock characterization. Infill drilling involved a program of angle and vertical holes designed to outline and more fully define the Leached Cap, (Oxide Gold zone) and Supergene Copper zones. Delineation drilling to the north, northeast, east, and southeast outlined the extent of the deposit area. Four oriented angle holes were drilled in the deposit area for geotechnical information regarding rock strength, structure and geological information regarding vein-set orientations.

Five vertical holes were drilled on the periphery of the deposit area for waste rock characterization studies. Seven vertical holes were drilled in the periphery of the deposit area for geotechnical information. Eighteen vertical holes were drilled outside the deposit area for geotechnical and geological information regarding potential site development.

The combined drilling from 1992 through 1994 consisted of 71,437.59 m of HQ core from 236 holes.

Core recoveries were consistently in the 80% to 90% range in the Leached Cap and Supergene zones and 90% to 100% in the Hypogene zone.

Drilling can be carried out at Casino from March through November with minor logistical challenges, while conditions in the spring and fall require winter type drilling equipment.

The use of a water supply truck is necessary with very cold weather conditions. Three reliable water supply sites exist on the property and can all be utilized during multiple drill rig programs.

Surveying

In April 1993 McElhanney Consulting Services Ltd. of Vancouver, B.C., produced a map of the Casino area based on 1985 air photos provided by the Department of Energy, Mines and Resources.

New aerial photography was flown in July 1993, by Lamerton & Associates of Whitehorse. The area they covered was mapped by Eagle Mapping Services Ltd. of Port Coquitlam, B.C. Eagle Mapping utilized two government UTM co-ordinates systems, NAD83 and WGS84, in the derivation of the deposit grid co-ordinates at photo target station #11. The following transformation parameters were used to convert from UTM coordinates to Property Grid:

Table 1.13-1: Transformation Parameters

ROTATION:	-0° 00' 05"
SCALE:	1.000453652
TRANSLATION:	-6703701.92 N
	-499861.96 E
ELEVATION SHIFT:	-8.32 m

The contours on McElhanney and Eagle Mapping Services maps compare to within approximately five metres and often closer. Generally, Eagle Mapping contours are smoother, having more gradual changes in direction.

Collar co-ordinates (Northing, Easting and elevation) for each drill hole were surveyed using a total station Nikon C-100. Surveying of the 1992 and 1993 drill holes was undertaken by Lamerton & Associates. Pacific Sentinel Gold did all the 1994 surveying.

In the 1993 drilling program, all drill holes, including deepened 1992 holes, were down-hole surveyed by a Sperry Sun magnetic compass tool to determine azimuth and dip.

In the 1994 drilling program, only angle holes were Sperry Sun surveyed. Acid dip tests were performed in the vertical holes. A Light-Log directional drill hole survey system was used for sixteen angle holes at Casino, starting at hole 94-285 and continuing for most of the angle holes through the remainder of the 1994 drilling program.

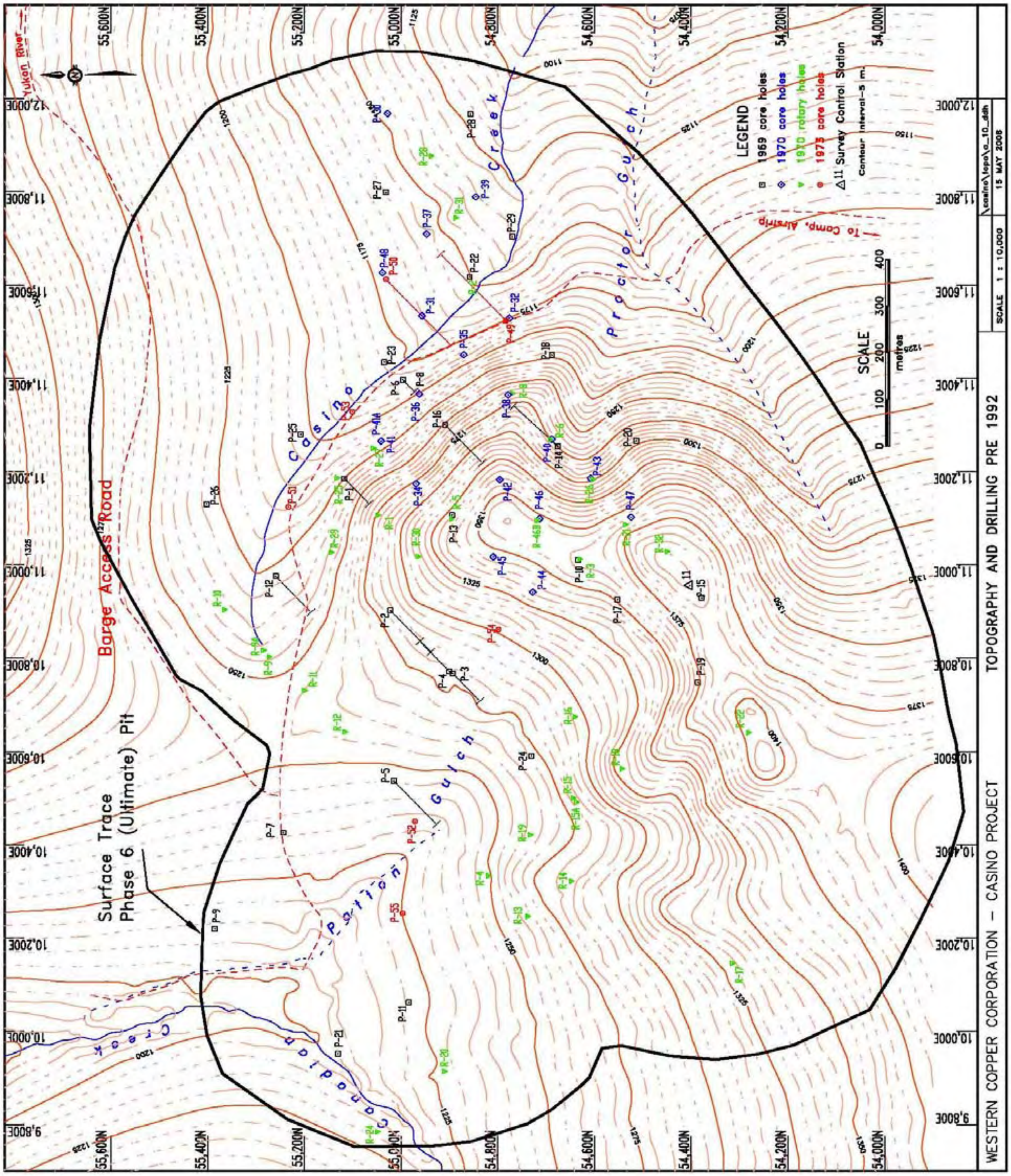


Figure 1.13-1: Casino Property Drilling Pre -1992

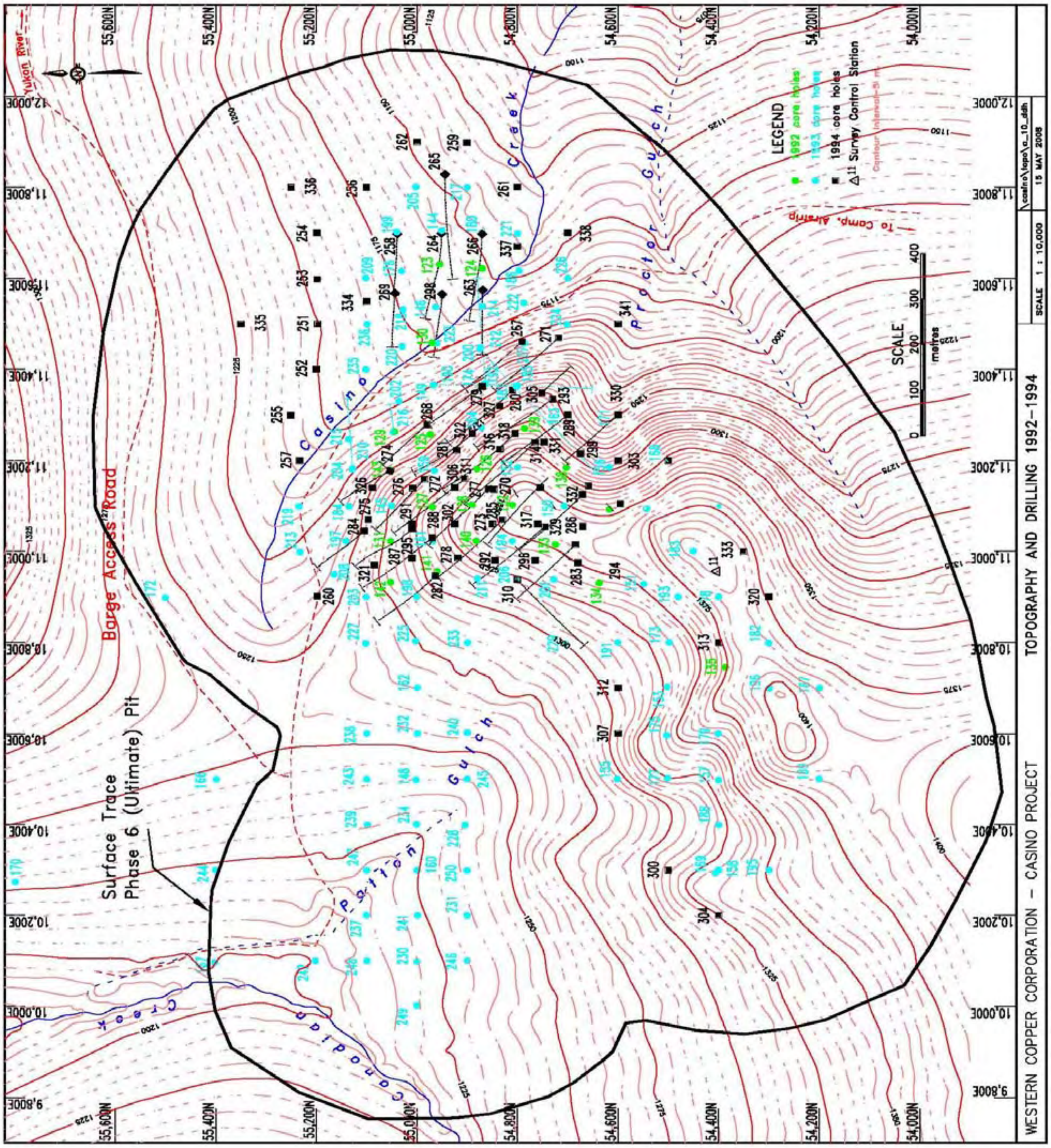


Figure 1.13-2: Casino Property Drilling 1992 - 1994

1.14 SAMPLING METHOD AND APPROACH

Core Sampling

Sampling and analytical protocols in use prior to the Pacific Sentinel Gold ("PSG") diamond drill programs are not well documented. In June 1992 core from 22 old holes was re-sampled by Archer Cathro, and the new assay results for all metals were compared to the originals. The results indicated 14 holes (64%) had identical results, while five holes (23%) had higher re-assays and three were inconclusive. When results of the old holes were compared with new holes drilled in the same locations, the results were similar to the re-sampling tests. Archer Cathro surmised that the higher gold results in the new holes were due to a combination of: improved drilling techniques that resulted in better core recovery, and advanced laboratory techniques that provided lower analytical detection limits.

The PSG core sampling followed rigorous procedures that were well documented and standardized throughout the drilling programs. In the 1992, 1993 and 1994 programs, exploration targets were sampled by HQ (63.5 mm diameter) core drilling; on occasion this was reduced to NQ (47.6 mm). The boxed core samples were transported by truck less than 5 km to a core logging facility adjacent to the Casino Airstrip, for geotechnical logging, sample selection quality control designation and sampling by PSG personnel. The average core recovery for all PSG core was 94%, with Hypogene averaging 96%, Supergene 92% and the Leached Cap (Oxide Gold zone) averaging 89%. Drill core was generally sampled over geologically defined intervals, the average width of which was 3 m, corresponding with the length of the average drill run. Core intervals were sampled by mechanical splitting. The remaining half core was returned to the boxes and stored in racks at the site. The average lengthwise half-split provided 10 to 15 kg of material, which was transported by charter aircraft (primarily DC-3) directly from the core sampling facility to Whitehorse, and then by commercial air freight to Vancouver for delivery to the sample preparation laboratory.

Figures 1.14-1 and 1.14-2 below are flow charts illustrating the processing of drill core and quality control procedures from 1992 to 1994.

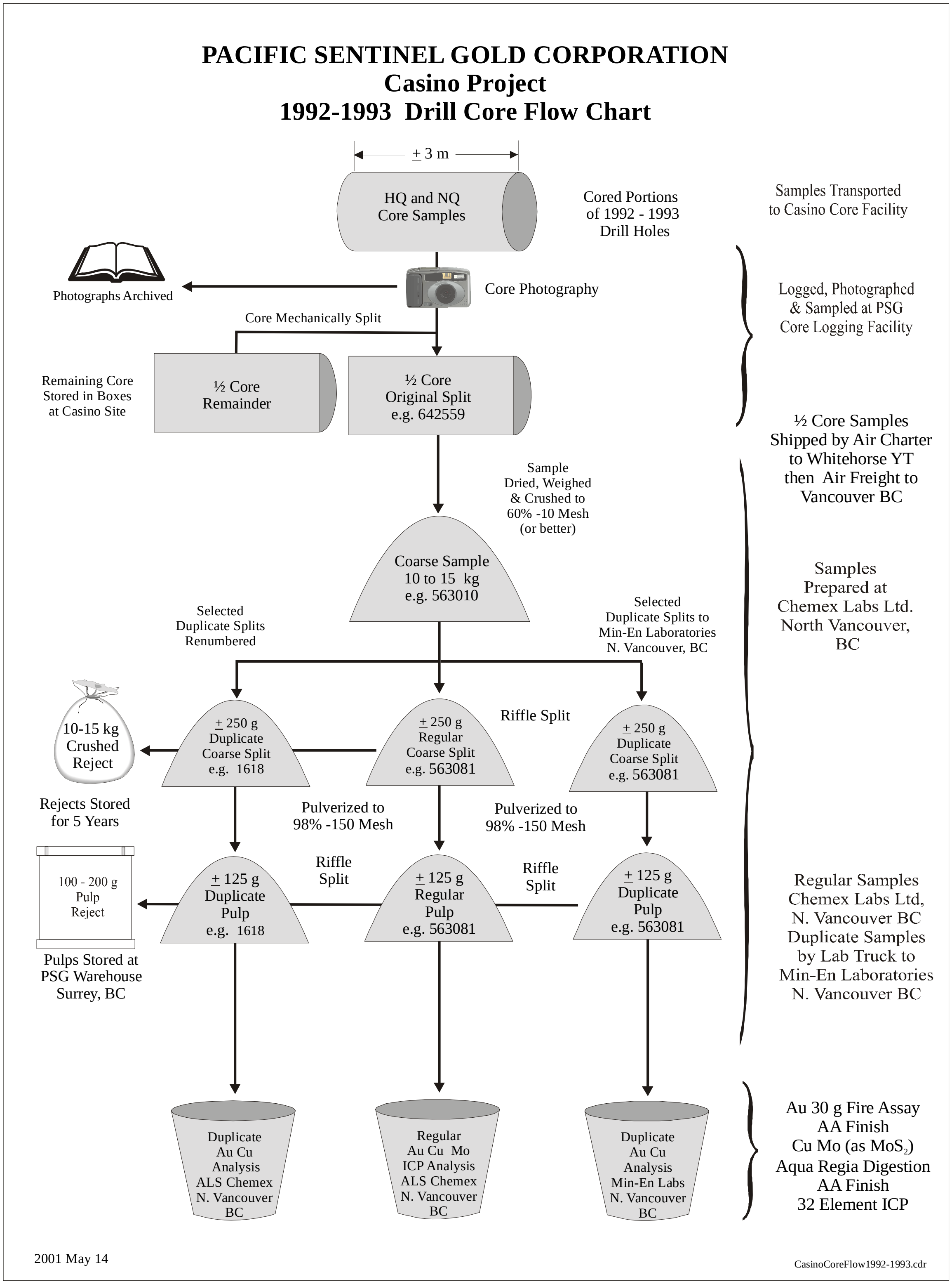


Figure 1.14-1: Casino Project Drill Core Processing and Quality Control Procedures, 1992-93

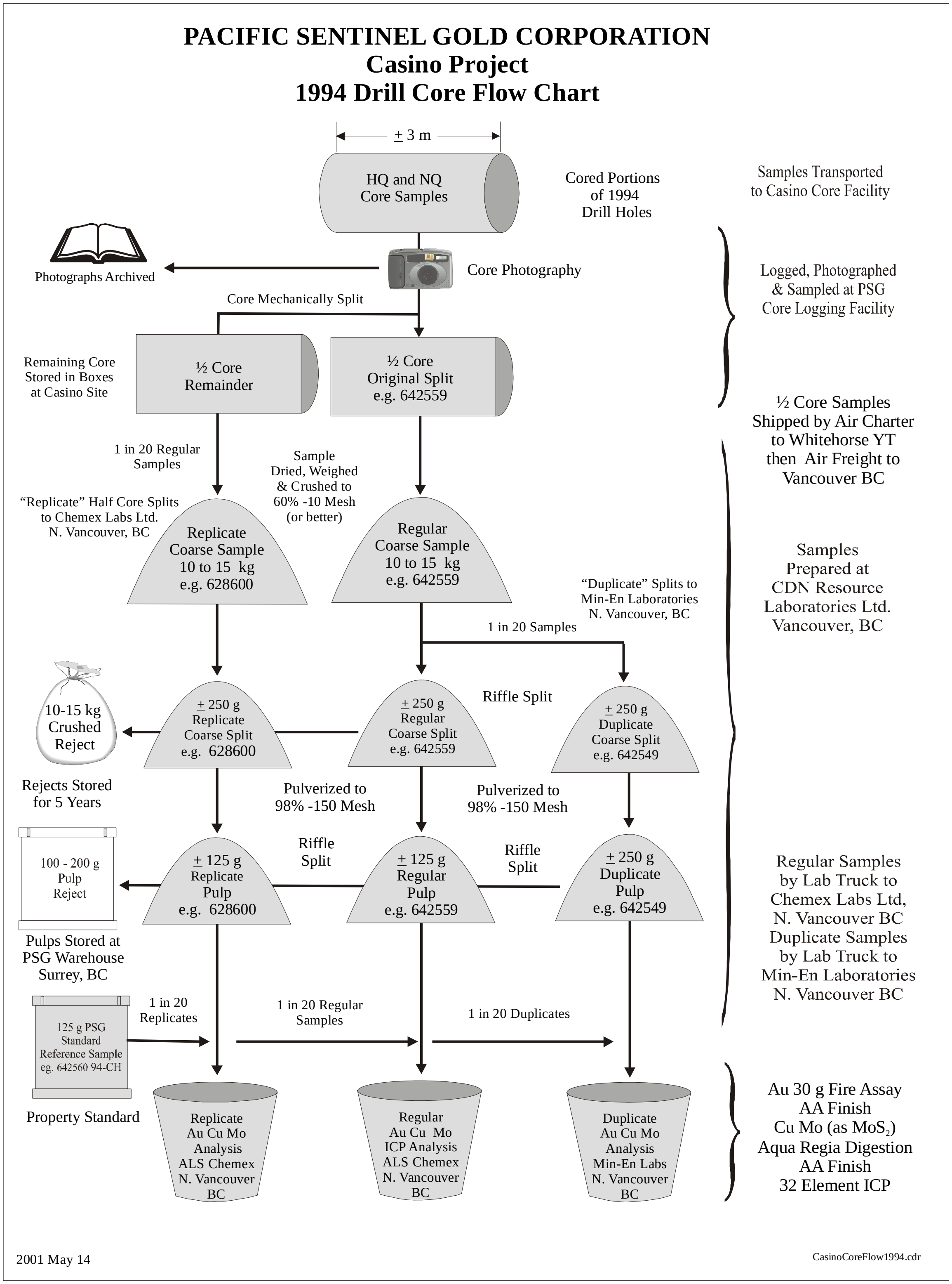


Figure 1.14-2: Casino Project Drill Core Processing and Quality Control Procedures, 1994

1.15 SAMPLE PREPARATION, ANALYSIS AND SECURITY

Sample Preparation

In 1992-1993 the samples were prepared at Chemex Labs Ltd. (now ALS Chemex) of North Vancouver, B.C. In 1994, all sample preparation was performed by CDN Resource Laboratories Ltd., of Vancouver, B.C. Samples arrived as half splits of HQ or NQ drill core, weighing an average of 15 and 10 kilograms respectively. At CDN the samples were dried and then weighed to the nearest 10 g. The dried samples were crushed to better than 60% passing 10 mesh. A sub-sample (mainstream sample), with a nominal weight of 250 g, was riffled from the crushed material for analysis while the remaining crush (reject) was sent to warehouse storage for safe keeping and possible future analysis.

Screen analyses were done on approximately one in every fifty, 250 g crushed core subsample, in order to check the crushing consistency. The results of these screen analyses were favourable, with only 3% of the 139 analyses less than 60% -10 mesh. In 1994, CDN also inserted the Pacific Sentinel Gold standard reference samples and took duplicate riffle splits from designated crushed rejects for analysis at a second laboratory.

All mainstream and duplicate 250 g crush samples were pulverized to better than 98% -150 mesh. Screen tests were done on one in every fifty pulps, and again the results were favourable with only 2% of 139 analyses below the target 98% -150 mesh. Each 250 g pulp was riffle split into two approximate 125 g pulp samples and bagged. One of each of the 125 g pulp samples was sent to warehouse storage, except those retained for duplicate analysis, for future use. The other 125 g pulp sample was sent for assay. At this point the duplicate pulps were separated from the mainstream pulps. Mainstream pulps were sent to Chemex Labs, while duplicates were sent to Min-En Labs, both of Vancouver, B.C.

Assay Analysis

Chemex analyzed all 1992-1994 regular (mainstream) samples, 1992-1993 selected duplicate samples and 1994 random ½ core replicate samples. Min-En Laboratories, of North Vancouver, BC, analyzed the selected duplicate samples from 1992 and 1993, and random duplicate samples from 1994. The analytical procedures used prior to 1992 are unknown.

Immediately prior to selecting each pulp's analytical aliquot, each pulp sample was passed through a -20 mesh screen to eliminate lumps of agglomerated clay minerals.

Gold Analysis

Samples were assayed for gold by 1 Assay Ton (AT) lead collection fire assay (FA) fusion with an atomic absorption spectroscopy (AAS) finish, results reported in grams per tonne.

Copper and Molybdenum Analysis

Total copper content (CuT) was determined by Aqua Regia digestion with an AAS finish. Copper results were reported in percent. Molybdenum content was determined by Aqua Regia digestion with an AAS finish. The Mo results were multiplied by 1.6681 and reported as MoS_2% to correspond with the format of the pre-1992 data.

Supergene Copper Analysis

Supergene Copper analysis utilized a multi step procedure to determine the amount of oxide (weak acid soluble), secondary sulphide (moderate acid soluble) and sulphide (insoluble) copper.

In 1994, selected samples in the Leached Cap, Supergene zone and the upper 50 m of Hypogene zone were composited two to one and subjected to a two-stage leach process. The filtered solution from a weak (3%) sulphuric acid initial leach was analyzed for copper by AAS. This result was designated weak acid soluble (CuW). This leach digested all copper oxide minerals (including neotocite, tenorite, malachite, azurite, chalcanthite, and brochantite), with the exception of cuprite. A second, stronger leach of (5%) sulphuric acid combined with 2% ferric sulphate. This leach digested from 25 to 50 percent of the copper present as supergene sulphide minerals such as chalcocite, digenite and covellite, without digesting a significant amount of the copper present as chalcopyrite and/or bornite. This leachate was analyzed for Cu by AAS and was designated moderate acid soluble (CuM). Copper remaining in the residue was un-dissolved chalcopyrite and/or bornite which was reported as insoluble sulphide copper or strong acid soluble copper (CuS).

ICP Analysis

For every sample, 1.0 gram was taken from the screened pulp and analyzed by Aqua-Regia digestion with an Inductively Coupled Plasma – Atomic Emission Spectroscopy finish (ICP-AES), for 32 elements as follows: Ag (ppm), Al (%), As (ppm), Ba (ppm), Be (ppm), Bi (ppm), Ca (%), Cd (ppm), Co (ppm), Cr (ppm), Cu (ppm), Fe (%), Ga (ppm), Hg (ppm), K (%), La (ppm), Mg (%), Mn (ppm), Mo (ppm), Na (%), Ni (ppm), P (ppm), Pb (ppm), Sb (ppm), Sc (ppm), Sr (ppm), Ti (%), Tl (ppm), U (ppm), V (ppm), W (ppm), and, Zn (ppm).

Security

During the drilling campaigns at Casino, the rigours of "chain of custody" were not as stringent as presently required. The remoteness of the Casino site provided a large degree of security as air traffic into the project was closely monitored. Further, the Casino gold grades were low and any metal contamination or grade enhancement would be quickly and easily identified. However, good sample handling procedures were in place during the PSG programs. Geologists supervised the sampling, and the samples were kept in a secure impoundment prior to shipping. The best vigilance on the samples was the attention to results, and in that regard PSG maintained a thorough quality assurance/quality control program (QA/QC).

Quality Assurance and Quality Control

Exploration sampling and analysis prior to 1992 was not subjected to the rigours required of modern regulatory requirements, but work conducted by major companies, like Quintana and Teck Corporation generally followed geologically accepted good sampling practices.

However, details of the sampling and analytical methodology are unknown. Moreover, analytical quality, particularly with respect to the determination of gold in the less than 1.0 g/t range, has improved considerably since the pre-1992 work was done. It is for this reason that the assay result from these old holes was not used in this study.

During the 1993 and 1994 Pacific Sentinel Gold drilling programs at Casino, standards, reject duplicates, and half-core replicates were assayed at regular intervals in order to check the security of the samples as well as the quality and accuracy of the laboratory analyses. Further, in-house laboratory standards, duplicates, and blanks were also run and reported as normal assays on certificates.

1.16 DATA VERIFICATION

Verification of the 1992 through 1994 diamond drilling investigation results was conducted on a full-time basis by one to two teams during the entire PSG exploration. Verification was done as the data was received from the field or the assay labs. The objective was to obtain an error free computer database of all the geological logs, geotechnical logs, assay results, ICP results, drill collar co-ordinates and down hole surveys.

Data Entry

Original 1992 and 1993 field data was entered by Archer, Cathro and Assoc. and by Nowak and Assoc., both of Vancouver, B.C. Entry of the 1994 field data to a database was performed on site and in the Vancouver office, by PSG personnel. All data was placed into a digital database. Assay, ICP, copper leach data, check assays and specific gravities were downloaded from the Chemex Labs computer bulletin board. Pacific Sentinel Gold Corp. personnel entered the down hole surveys and the collar surveys and were responsible for making corrections from the data verification process.

Verification Procedure

The data verification process was performed under the supervision of a geologist familiar with the site logging procedures for consistency. Working in teams of two, one person read the original certificate, information sheet or logging form out loud while the other visually scanned the database printouts. Differences between the two were noted and corrected on the printout and the digital database. When required, a second pass was done on selected data.

The procedure for correcting discrepancies was to highlight the value in question and to write the correct value beside it. Occasionally, in verification of field logs, when it was unclear which value was correct, or it seemed both may be in error, other sources such as the geologists' logs, synoptic logs, drillers' time sheets, and/or core photographs were referenced and a decision was made by a geologist familiar with logging and sampling techniques.

Errors

The errors encountered during the data verification process varied depending upon the data being checked. There were very few errors that appeared to have occurred during the data entry process.

Discrepancies with the assay, ICP and copper leach data involved values below the detection limit. Occasionally less than signs (<) were misplaced for the lower detection limit values. Anomalously high ICP values were occasionally rounded off differently in the assay certificates than in the assay data downloaded from the computer bulletin board.

Errors detected in the field data of the geological logs, geotechnical logs, synoptic logs, specific gravity logs and down hole survey data were often a result of human error in recording the originals.

The geological logs, especially the *Geolog* format of the 1992 and 1993 logs, had some errors introduced when the data entry personnel were unclear of the recording method the geologist was using. In addition, changing definitions of many of the lithologic types required re-logging of many of the holes in the 1992 and 1993 programs. The process of combining the information from the old and new logs introduced some errors into the database. Due to the number of discrepancies encountered in the *Geolog* data of the 1992 and 1993 programs, a second verification of lithologies and alteration was performed after the errors detected in the first pass were corrected.

The geotechnical logs were checked by the computer to find intervals with combinations of parameters that were suspect. These intervals were extracted from the database and the suspect values were checked against the originals and against other available information, such as core photos, to determine if they were in error. A large majority of the extracted parameters were correct and considered to be caused by normal variance of geotechnical characteristics.

Errors found in the specific gravity data were due to the geotechnician assigning the wrong sample number to the interval from which the specific gravity was taken. These errors were detected by a computer check and confirmed by the data verification personnel.

1.17 ADJACENT PROPERTIES

Numerous quartz mineral claim blocks registered to other owners are staked adjacent to and in the general vicinity of Western Copper Corporation's Casino claim block. No known mineral exploration activity is being pursued on these claims at the present time.

Placer claims have been staked on Canadian and Britannia Creeks, some of which overlap with mineral claims on the Casino claim block (Figure 1.6.2). During a visit on September 25, 2007, members of the project team observed evidence of recent activity on the Canadian Creek placer claims.

Some of the placer claims overlap the Casino claims in the area of the pit. There is no priority in circumstances where placer and quartz claims overlap and negotiations with the holders of the placer claims will be necessary to determine an appropriate resolution to this situation.

The Bomber and Helicopter Ag-Pb-Zn, +/- Au polymetallic vein occurrences located approximately 2 km S-SW of the Casino porphyry deposit are within Western Copper Corporation's claim block. These occurrences became known in the late 1920's, preceding the discovery of the Casino porphyry deposit by several decades. The majority of work has been undertaken on the Bomber showing where locally high grade silver, sphalerite and galena mineralization occurs in a series of four sub-parallel, northwest trending quartz barite shears that dip steeply to the west and cut Dawson Range batholith granodiorite. The Bomber vein was last mined from 1978 to 1980 when 49 m of drifting, 55 m of raising and extensive surface trenching was carried out. In 2007 Western Copper ran VLF, TMF and soil sampling grids to identify additional exploration targets and possible extensions of the vein shears. A follow-up exploration program is planned for this area.

1.17.1 Carmacks (Williams Creek) Project

Western Copper is actively developing the oxide copper deposit at Carmacks located about 115 km ESE of the Casino deposit and about 200 km NW of Whitehorse. The project anticipates open pit mining with crushing, heap leaching and copper recovery by solvent extraction/electrowinning to produce about 14,500 tonnes per year of cathode copper over about eight years.

1.17.2 Minto Mine

In October 2007 Sherwood Copper Corp. commenced operations at the Minto Mine about 240 km NW of Whitehorse and about 80 km east of the Casino Deposit.

The Minto mine is an open pit mine and flotation concentrator presently treating about 2,400 tonnes per day of ore, with a planned expansion to 3,500 tonnes per day.

The Carmacks and Minto deposits share some geologic and mineralogical similarities. Neither one is similar to the Casino deposit and therefore hold no value in terms of leading to a better understanding of Casino.

1.18 MINERAL PROCESSING AND METALLURGICAL TESTING

The Casino Project will produce copper flotation concentrates with contained gold values, and molybdenum flotation concentrates, both from sulphide ore. Gold in the form of doré, and a high grade copper sulphide product will also be produced from an oxide ore heap leach. All products will be shipped offsite for sale or further processing.

1.18.1 Metallurgical Samples

In the testwork commissioned by Pacific Sentinel Gold in the mid 90's, all of the samples used were assay rejects that were nominally -10 mesh in particle size. These assay rejects were combined to prepare a number of composites that were sent to Lakefield Research for flotation and other testing under the direction of Melis Engineering, Ltd., to Brenda Process Technology for flotation testing, and to Kappes, Cassiday and Associates for copper and gold leaching.

The source of samples for the current work was split HQ core that was retrieved from site in September 2007. The core had been at site since it was drilled in 1993 and 1994, but was stored under cover and well preserved by the cold temperatures of the Yukon. Recent oxide copper analysis of the core confirmed that oxidation was insignificant.

1.18.2 Leaching Tests

Kappes, Cassiday and Associates performed two studies in 1995 on the leaching of the oxide cap and supergene material. In the first study they leached a selection of oxide cap material with cyanide and in the second study they examined pre-leaching both oxide cap and supergene material with acid followed by cyanidation of the residue.

Gold extraction was affected by the amount of copper leached during cyanidation and ranged from 10-97.4%. Average gold extraction was 79.9%.

Lime consumption during cyanidation averaged 3.9 kg/t without the acid pre-leach, and 4.1 kg/t with the acid pre-leach. Cyanide consumption was significant, averaging 5.5 kg/t without the acid pre-leach. There was not a significant difference between the lime consumption for the oxide copper composites and copper oxide composites.

In the current leaching testwork, two column tests were run by METCON Research, Inc. on a composite sample blended to create similar gold and copper concentrations as the average reserve.

The ore was crushed coarsely to -3.8 cm (-1.5 inch), placed in 15 cm by 6 m columns, and irrigated at 12 L/h/m^2. One column was leached "open cycle" – a 0.5 g/L NaCN solution was fed to the top of the column and the pregnant solution was collected and assayed. The second column was "locked cycle" and solution was recycled. In the locked cycle column when the copper concentration in solution exceeded 50 mg/L, the solution was treated through a SART pilot plant discussed in the next section, and the gold was recovered on activated carbon. Results from the tests are shown in Table 1.18-1.

The gold, silver, and copper extractions from the open and locked cycle tests compare favourably. Although the gold extraction was slightly higher for the open cycle test, both tests produced good gold recovery considering the coarse crush size.

Table 1.18-1: Extractions and Reagent Consumptions from Open Cycle and Locked Cycle Cyanidation

	Assays (calculated head) (g/t)			Percent Extraction			Reagent Consumption (kg/t)		
	Au	Ag	Cu	Au	Ag	Cu	NaCN*	NaCN**	CaO
Open	0.47	1.92	693	69.52	25.14	17.4	0.39		2.83
Locked	0.42	1.61	654	65.79	27.31	18.2	0.48	0.54	3.06

*based on titrations
**based on additions

1.18.2.1 SART copper recovery

SART stands for Sulfidation, Acidification, Recovery and Thickening. In this process, a cyanide solution containing copper is treated to remove copper – gold is not affected.

In the locked cycle test described previously, the pregnant leach solution from the column was treated using a SART pilot plant several times before having the gold removed using carbon and being recycled to the column. Average SART results are summarized in Table 1.18-2.

Table 1.18-2: SART Results

Pregnant Solution				Barren Solution after SART & Carbon				Copper	Reagent Consumption		
NaCN (g/L)	Cu (ppm)	Au (ppm)	Ag (ppm)	NaCN (g/L)	Cu (ppm)	Au (ppm)	Ag (ppm)	Removal (%)	(g/L solution treated) S^{2-}	H_2SO_4	CaO
0.25	81	0.21	0.30	0.39	6.8	0.04	0.02	91.3	0.024	0.64	0.37

1.18.3 Comminution Testing

A limited amount of comminution tests were performed on the composite samples tested by Melis Engineering, Ltd., and Brenda Process Technology in prior testwork.

SGS Mineral Services performed a comminution study for this present study. 50 split drill core samples, representing the first 6 years of production were sent to SGS and subjected to several tests. The results from these tests were then analyzed by SGS using SGS MinnovEx CEET2 technology.

A summary of the grinding results appears in Table 1.18-3. As per the SGS report, the samples tested were characterized as medium in hardness from the perspective of semi-autogenous milling and of medium in hardness with respect to ball milling.

Table 1.18-3: Summary of Comminution Results

Test Name	CEET CI	SPI (min)	RWI (kWh/t)	BWI (kWh/t)	MBWI (kWh/t)	AI (g)
Average	29.2	52.9	9.9	14.5	14.30	0.265
Std. Dev.	13.9	20.8	5.6	2.6	1.60	0.046
Rel. Std. Dev.	47.5	39.3	56.5	18.1	11.30	17.0
Minimum	13.5	12.6	0.0	11.2	11.40	0.226
10th Percentile	15.3	31.4	4.4	12.1	12.50	0.232
25th Percentile	19.1	37.4	11.1	13.3	13.00	0.242
Median	24.1	50.3	12.5	14.1	14.10	0.252
75th Percentile	38.0	63.4	13.0	15.9	15.60	0.275
90th Percentile	52.3	82.5	13.0	17.3	16.30	0.309
Maximum	66.9	114.1	13.0	18.2	18.30	0.332

A circuit consisting of one 12.2 m SAG mill and two 8.2 m ball mills in closed circuit with two pebble crushers was selected as a circuit that would likely meet the design tonnage. This circuit was modeled by CEET2 technology for the first six years of production based on the results from the comminution testwork to produce a primary grind size of 80% passing 147 µm.

The throughput as a function of year was correlated with the ore type per year from the preliminary mining schedule. A simple correlation was found that is outlined in Table 1.18-4. These throughput per ore type values in Table 1.18-4 were used for the final mining schedule developed by IMC.

Table 1.18-4: Throughput Based on Ore Type

Ore Type	Throughput (t/h)
CAP	4469
SUP	4469
HYP	3933

1.18.4 Flotation

Previous testwork in 1994 and 1995 consisted of both Melis and Brenda (later International Metallurgical and Environmental, Inc.) performing flotation tests on the oxide cap, supergene and hypogene mineralization. These tests were unable to routinely produce good concentrate grades with reasonable copper recoveries. A high pyrite concentration in the hypogene material and the presence of oxide copper in the supergene material were cited as the key issues.

The present work at G&T Metallurgical focused on two composites at two different levels of oxide copper – an "oxide composite" and a "sulphide composite". The composites were prepared to be close to the average grade of ore received for the first 5 years.

Oxide Composite

Copper recovery and grade from the oxide composite was very poor. Various combinations of sulphidizing the ore, changing grind size, using different reagents were attempted. Poor concentrate grades and low metal recoveries were obtained in all cases leading to the conclusion that this material be treated as waste in this study.

Sulphide Composite

Copper recovery from the sulphide composite was much better than that achieved for the oxide composite. Copper concentrate grades greater than 28% were routinely achieved.

Cleaner copper recoveries of 70-82% were obtained into concentrates grading from 26.8 to 32.2% copper at primary grinds with K80's of 147 and 121 µm, and regrinds with K80's less than 22 µm.

Molybdenum cleaner recovery appeared to be somewhat independent of the copper concentrate grade at a K80 of 147 µm, with an average recovery of 65% obtained.

Gold recovery decreased slightly with increasing copper grade in the concentrate. Gold recoveries at a K80 of 147 µm were 57% when the concentrate copper grade was greater than 25%.

Locked Cycle Tests

Duplicate locked cycle tests at both primary grind K80's of 121 µm and 147 µm were performed as well as one locked cycle at a primary K80 of 209 µm. The results from these tests are presented in Table 1.18-5.

Table 1.18-5: Locked Cycle Test Results with Comparative Cleaner Tests

P. Grind	Regrind	Cycle	Assay - percent or g/t				Distribution - percent			
K80 µm	K80 µm		Cu	Fe	Mo	Au	Cu	Fe	Mo	Au
121	18	IV	30.3	27.0	1.5	28.9	85.2	5.9	59.0	64.8
121	18	V	27.5	27.4	1.9	28.1	86.0	7.0	66.5	73.5
121	18	IV	31.3	25.3	1.0	30.3	83.3	4.7	34.7	68.5
121	18	V	30.6	25.2	0.5	27.7	84.7	5.1	27.8	71.1
147	20	IV	31.7	26.8	0.5	27.2	82.6	4.7	26.5	59.3
147	20	V	31.0	27.8	0.6	25.7	85.3	5.5	37.1	62.6
147	20	IV	24.5	28.2	1.4	20.6	87.0	7.7	63.9	67.1
147	20	V	26.8	28.3	1.8	22.6	88.5	7.5	69.4	66.8
147	20	avg.	28.5	27.8	1.1	24.0	85.8	6.3	49.2	64.0
209	20	IV	26.1	26.2	1.5	19.0	80.1	5.9	62.3	53.0
209	20	V	31.7	27.4	1.6	21.6	81.3	5.3	63.7	54.5
121	20	clnr	30.8	25.7	2.3	22.3	80.7	4.8	59.9	43.8
147	20	clnr	28.9	26.5	1.6	38.8	83.4	5.3	64.0	57.1
147	20	clnr	31.7	26.2	1.6	42.1	80.7	4.9	57.8	55.1
147	20	avg.	30.3	26.4	1.6	40.5	82.0	5.1	60.9	56.1

Variability Testing

The results from the variability tests were normalized to a 28% copper concentrate grade for analysis. To ensure that inaccurate extrapolation did not occur, all flotation tests that did not achieve a concentrate grade of at least 25% copper were not extrapolated to the 28% grade. These results were used to determine the recovery correlations below.

Copper

Copper recovery correlated well with the copper *sulphide* head grade of the sample – that is the total copper minus the weak acid soluble copper grade. The equation for copper recovery was found to be:

$$\text{Cu Recovery} = 100 \times (\text{Cu}_{\text{total}} - \text{Cu}_{\text{WAS}} - 0.022\%)/(\text{Cu}_{\text{total}})$$

Molybdenum

The molybdenum recovery was also shown to be a function of the molybdenum head grade. The equation for molybdenum recovery to a molybdenum concentrate, assuming a 90% molybdenum recovery from the copper/moly concentrate was found to be:

$$\text{Mo Recovery} = 76.0 \times (\text{Mo}_{\text{total}} - 0.003\%)/(\text{Mo}_{\text{total}})$$

Gold

Gold recovery was not found to be related to head grade or any other parameter. The gold recovery was averaged over the tests and found to be:

Au Recovery = 66.0%

Reagent Consumptions

Table 1.18-6 shows the average reagent consumptions for the variability tests where the weak acid soluble copper as a percentage of the total is less than 15% as will be typical in the plant.

Table 1.18-6: Average Reagent Consumptions for Variably Flotation Tests

	Consumption g/t
Lime:	3,300
No. 2 Diesel Fuel:	66.6
Aerophine 3418A:	9.6
Aerofloat 208:	36.7
MIBC	79.0

1.18.5 Determination of Recoveries and Reagent and other Consumable Consumptions

As described in the preceding sections, the following recoveries, reagent and other consumable consumptions will be used. Where values were unknown, typical values were used:

Heap Leach

Table 1.18-7: Heap Leach operational parameters.

Parameter	**Value**	**Units**
Gold recovery	50	Percent
Copper recovery	20	Percent
Crush size	ROM	
Irrigation rate	12	$L/h/m^2$
Lift height	6	m
Reagent consumptions		
Hydrogen sulphide	0.039	kg/t ore
Sulphuric acid	0.380	kg/t ore
Hydrochloric acid	0.010	kg/t ore
Lime	4.270	kg/t ore
Sodium hydroxide	0.013	kg/t ore
Sodium cyanide (NaCN)	0.500	kg/t ore
Activated carbon	0.500	g/t ore
Anti-scalant	0.003	kg/t ore

Flotation

Table 1.18-8: Flotation Operational Parameters

Parameter	Value	Units
Copper recovery	Recovery = 100 x (Cu_{total} – Cu_{WAS} – 0.022%)/(Cu_{total})	percent
Molybdenum recovery	Recovery = 76.0 x (Mo_{total} – 0.003%)/(Mo_{total})	percent
Gold recovery	66.0	percent
Bond work index	14.5	kWh/t
Primary grind size (P80)	147	µm
Regrind size (P80)	22	µm
Reagent consumptions		
Lime	2.0	kg/t ore
Fuel Oil	66.6	g/t ore
Aerophine 3418A	9.6	g/t ore
Aerofloat 208	36.7	g/t ore
MIBC	79.0	g/t ore
NaSH	0.033	kg/t ore
Flomin D-910	0.006	kg/t ore
Flocculent	0.550	g/t ore
No. 2 Diesel Fuel	0.440	g/t ore
Flocculent	3.0	g/t ore
SAG Mill - Liners	0.040	kg/t ore
Ball Mill - Liners	0.085	kg/t ore
SAG Mill - Balls	0.500	kg/t ore
Ball Mill - Balls	0.600	kg/t ore

1.18.6 Sulphide Ore Process Plant

Sulphide ore will be transported from the mine to the concentrator facility by off-highway haulage trucks. Copper concentrate produced at the concentrator facility will be loaded into highway haul trucks and transported to port for shipment to a concentrate smelter and metal refinery. Molybdenum concentrate produced at the concentrator facility will be bagged and loaded onto trucks for shipment to market.

The design basis for the sulphide ore processing facility is 94,770 dry tonnes per day (t/d) or 34,591,000 dry tonnes per year (t/y). Sulphide ore is available for approximately 23.5 years of milling. Lower grade ore will be reclaimed from stockpile and milled in the last five years of the mill’s 28 year life.

Design ore grade to the sulphide process plant is estimated to average 0.212% copper and 0.0236% molybdenum. The process plant design allows for sustained metal recovery of both copper and molybdenum.

1.18.7 Oxide Ore Process Plant

Oxide ore will be transported from the mine to a run of mine heap leaching facility by the off-highway haulage trucks. Gold bullion produced from the oxide gold ore will be shipped by truck or air to metal refiners.

The design basis for oxide ore processing is 25,000 dry tonnes per day (t/d) or 9,125,000 t/y. Oxide ore is available to provide enough oxide ore for approximately 7 years of leaching.

The overall oxide ore grade is estimated to average 0.427 grams/metric ton (g/t) of gold and 0.0624% copper. Annual average ore gold grades are projected to range from 0.300 to 0.538 g/t. Annual average copper grades are projected to range from 0.032 to 0.082 percent. The heap leach and recovery plant operating budget indicates a sustained metal production rate of 148,000 troy ounces of gold in the highest production year and an overall average of 85,000 ounces of gold per year throughout the seven years of gold production. Copper will be recovered by the SART process to control the quality of the leach solution.

1.18.8 Process Flow Sheets

The following items summarize the process operations required to extract copper and molybdenum from the sulphide ore:

- Crushing the ore in a primary gyratory crusher to reduce the ore size from run-of-mine to minus 150 mm.
- Stockpiling primary crushed ore and then reclaiming by feeders and conveyor belt.
- Size reduction of the ore in a semi-autogenous (SAG) mill - ball mill grinding circuit prior to processing in a flotation circuit. The SAG mill will operate in closed circuit with a pebble crushing circuit. The ball mills will operate in closed circuit with hydrocyclones.
- The flotation circuit will consist of copper and molybdenum flotation circuits. The copper and molybdenum minerals will be concentrated into a bulk copper/molybdenum concentrate. Molybdenum will then be separated from the copper minerals in a molybdenum flotation circuit. The bulk (copper-moly) flotation circuit will consist of rougher flotation, concentrate regrind, cleaner flotation, and cleaner scavenger flotation circuits. The molybdenum flotation circuit will consist of a copper-moly concentrate thickener, molybdenum rougher flotation, rougher cleaner flotation, concentrate regrind, second cleaner flotation, and third cleaner flotation circuits.
- Final copper concentrate will be thickened, filtered, and loaded in trucks for shipment.
- Final molybdenum concentrate will be filtered, dried, and packaged into shipping containers for shipment.
- Flotation tailing will be thickened and transported by a gravity pipeline to a tailing impoundment area at the mill site.
- Water from tailing and concentrate dewatering will be recycled for reuse in the process. Plant water stream types include: process water, fresh water, potable water, and fire water.

- Storing, preparing, and distributing reagents used in the sulphide ore process.

The following items summarize the process operations required to extract gold from the oxide gold ore.

- Stacking run of mine ore by a truck dump method onto a heap leach pad.
- Leaching the stacked ore with cyanide solution.
- Recovering gold from the pregnant leach solution on activated carbon in column tanks (CIC).
- Recovering copper from the pregnant leach solution by the Sulphidization, Acidification, Recycling and Thickening (SART) process.
- Treating gold loaded carbon recovered from the CIC circuit by acid washing, cold stripping with cyanide solution to remove copper, and hot stripping with caustic solution to remove gold.
- Reactivating the stripped carbon.
- Recovering gold from the pregnant carbon stripping solution as cathode sludge on stainless steel mesh cathodes in an electrowinning cell.
- Melting the cathode sludge with fluxes to produce a gold-silver Doré bar, the final product of the ore processing facility.
- Storing, preparing, and distributing reagents to be used in the process.

1.18.9 Extraction Rates

The average metal recoveries expected from sulphide ore processing are based on:

Copper recovery to copper concentrate, percent:	84
Copper concentrate grade, percent copper:	28
Gold recovery to copper concentrate, percent:	66
Molybdenum recovery to molybdenum concentrate, percent:	66
Molybdenum concentrate grade, percent molybdenum:	55

The metal recoveries expected from oxide ore processing are based on:

Gold recovery, percent:	50
Copper recovery to SART precipitate, percent:	20
Copper precipitate grade, percent copper:	75

1.18.10 Process Reagents

Sulphide ore process reagents that will require handling, mixing, and distribution systems include:

- Sodium-diisobutyl dithiophosphinate (Aerophine 3418A promoter), 9.6 g/t
- Sodium diethyl dithiophosphate/sodium di-secondary butyl thiophosphate (Aerofloat 208 promoter), 36.7 g/t
- Methyl Isobutyl Carbinol (MIBC, frother), 79 g/t

• Pebble Lime (CaO, pH modifier),	3.0 kg/t
• No. 2 Diesel Fuel (collector),	67 g/t
• Sodium Hydrosulphide (NaHS, copper mineral depressant),	0.033 g/t
• Flomin D-910 (copper mineral depressant),	0.006 g/t
• Flocculant, Superfloc A-130	0.55 g/t

Oxide ore process reagents that will require handling, mixing, and distribution systems include:

• Sodium Cyanide (NaCN),	0.5 kg/t
• Caustic (sodium hydroxide, NaOH),	0.13 kg/t
• Pebble Lime (CaO),	4.27 kg/t
• Hydrochloric Acid (HCl),	0.01 kg/t
• Hydrogen Sulphide (H2S),	0.039 kg/t
• Sulphuric Acid (H2SO4),	0.38 kg/t
• Activated carbon,	0.5 kg/t
• Antiscalant,	0.003 kg/t

1.19 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

1.19.1 Mineral Resource Estimate

Pacific Sentinel Gold Corp. staff geologists and consultants undertook an extensive geological interpretation of the Casino Deposit in 1994 and 1995 and a block model was developed to include lithologic, metallurgical and structural controls prior to establishing grade estimates. Ordinary kriging of assay composites was utilized to interpolate grade into each block using variogram models developed by G. Giroux, P. Eng. in May 1995. In January 2004 a copper equivalent value was calculated for each block in this model using the grades for copper, gold, and molybdenum estimated by Giroux. In April 2008 measured and indicated blocks were re-categorized by Giroux for purposes of this study.

1.19.2 Data

Information provided by the 214 HQ (6.35 cm diameter) and NQ (4.76 cm diameter) core holes of the 1992 through 1994 drill campaigns was used in this study. These holes provide extensive coverage of the deposit and tend to have better core recovery and more reliable analytical results than the drilling done prior to 1992. Therefore, the analytical data from the pre-1992 drilling campaigns was not composited, or used in the statistical analysis, variography, grade block modelling or resource estimation because of concerns with its overall quality. However, the geological information from the old holes was used in the geological modelling process. A further 22 holes in 1994 were drilled outside of the main deposit area for a variety of purposes, including exploration, engineering and environmental studies.

The drill holes used in the resource estimate are as follows: 92-123 through 92-143, 93-144 through 93-250 and 94-251 through 94-341 (except 94-319, 323, 328 and 340). The gold, total copper and molybdenum data from the 70,276 m of core drilling from 214 holes was composited into 4,787 nominal 15 m down-hole composites based on mineralogical zone, domain and lithology. The composite lengths were modified to honour actual lithologic and mineralogical boundaries in the drill core. The composites ranged from 7.5 to 22.5 m in length, and averaged 14.68 m. The weak acid soluble copper (CuW), moderate acid soluble copper (CuM) and strong acid soluble copper (CuS) data from 33,017 m of drilling in 210 drill holes (as above, but excluding 93-177, 187, 189, 195), was composited into 2,239 composites using the same criteria as above for an average composite length of 14.75 m.

No cutting or capping was applied to the original assays, composites or final estimated values.

1.19.3 Variography

Gary Giroux, P.Eng, examined the statistical characteristics of the composited assay data for the different geologic units and mineralogical zones, and developed 31 variogram models for the purposes of kriging the deposit. These are summarized in Table 1.19-1.

Table 1.19-1: Summary of Variogram Models

Parameter	Mineralogical Zone	Lithology	Models
Gold	All	IX, MB, QM, WR, OT	5
Total Copper	Hypogene	IX, MB, QM, WR, OT	5
Weak Acid Soluble Copper	Leached Cap	MB, OT	2
Weak Acid Soluble Copper	Supergene	MB, OT	2
Moderate Acid Soluble Copper	Leached Cap	MB, OT	2
Moderate Acid Soluble Copper	Supergene	MB, OT	2
Strong Acid Soluble Copper	Leached Cap	MB, OT	2
Strong Acid Soluble Copper	Supergene	MB, OT	2
Molybdenum	Leached Cap	MB, OT	2
Molybdenum	Supergene	MB, OT	2
Molybdenum	Hypogene	IX, MB, QM, WR, OT	5
TOTAL			31

Code OT, stands for *other* rock types, i.e. LT or PP or QP or QX or YM.

1.19.4 Geological Model

PSG geologists created digital geological models in the fall of 1994 and spring of 1995. The lithologic (rock) and domain models were digitized from cross sections and transferred to level plans. The contacts were joined to produce three-dimensional geologic models to control the grade estimation procedure. The lithologic units and domain codes are listed in Table 1.19-2 and Table 1.19-3 respectively. A model of the mineralogical zones (Leached Cap, Supergene, Hypogene) was created in cross section and on 15 m level plans and digitized. The mineralogical zone codes are listed in Table 1.19-4. The surface topography model was created from the Eagle Mapping project 93-75 digital base-map and the overburden surface was modeled in 1994 based on the drill data. The various geologic models and surfaces were combined in the block model. The three-letter codes used in the block model are represented by the lithologic (first digit), domain (second digit) and mineralogical (third digit) codes shown in Tables 1.19-2, 1.19-3 and 1.19-4.

Table 1.19-2: Lithology (Rock) Codes

First Digit	Lithology Code	Description
2	YM	Yukon Group Metamorphic
3	WR	Dawson Range
4	PP, QP	Patton Porphyry
5	QM, QX	Quartz Monzonite
6	IX	Intrusion Breccia
7	MB	Intrusive Milled Breccia
8	LT, LP	Latite Breccia, Latite Dykes

Table 1.19-3: Domain Codes

Second Digit	Domain Code	Description
0	West	West of Casino Creek Fault
1	East	East of Casino Creek Fault
2	Extended	Outside Main Mineralized Area
9	Undefined	Undefined Rock/Domain Code

Table 1.19-4: Mineralogical Zone Codes

Third Digit	Zone Code	Description
1	OVB	Overburden
2	CAP	Oxide Gold/Leached Cap
3	SUP	Supergene Oxide/Supergene Sulphide
4	HYP	Hypogene

1.19.5 Block Model

A block model was created in 1994-1995 with the following configuration:

- Bottom left hand corner of the model = 9990 E, 53890 N, 442.5 m Elev.
- Top right hand corner of the model = 12010 E, 55890 N, 1522.5 m Elev.
- Rows = 100 x 20 m = 2000 m
- Columns = 101 x 20 m = 2020 m
- Levels = 72 x 15 m = 1080 m

The lithologic, domain and mineralogical zone information was transferred to each block with the three-digit codes described previously based on the block centroid. The blocks were further coded as to metallurgical (copper) zone for the purposes of modelling.

Table 1.19-5 lists the metallurgical (copper) codes, and the criteria used to define them.

Table 1.19-5: Metallurgical (Copper) Zone Codes

Model Code	Zone	Description
1	Oxide Gold/Leached Cap	CuT = 0 & CuC < 0.12
2	Supergene Oxide	CuT = 0 & CuC > 0.12 & CuR > 0.4
3	Supergene Sulphide	CuT = 0 & CuC > 0.12 & CuR > 0.15 & CuR < 0.4
4	Hypogene (Calculated)	CuT = 0 & CuC > 0.12 & CuR < 0.15
5	Hypogene (Field)	CuT > 0

- CuT = Total Copper Assay
- CuW = Weak Acid Soluble Copper (soluble in 3% H_2SO_4) = Oxide Cu
- CuM = Moderate Acid Soluble Copper (soluble in 5% H_2SO_4 + 2% $Fe_2(SO_4)_3$) = Supergene Sulphide Cu

- CuS = Strong Acid Soluble Copper Assay (insoluble in 5% H_2SO_4 + 2% $Fe_2(SO_4)_3$) = Hypogene Cu
- CuC = Calculated Total Copper from Soluble Cu Assays = CuW + CuM + CuS
- CuR = Copper Ratio = (CuW + CuM)/CuC

1.19.6 Resource Estimation

Ordinary kriging was used to estimate the block model on May 10, 1995. An additional component was added on May 31, 1995. Gold and molybdenum were kriged in all zones. Weak, moderate and strong acid soluble copper grades were kriged in the Leached Cap / Oxide Gold, Supergene Oxide and Supergene Sulphide zones, while copper was kriged in the Hypogene zone. The domains (east or west of the Casino fault) were considered hard estimation boundaries. Estimation boundaries for lithologic (rock) and mineralogical zones were the same as for the variography.

1.19.7 Resource Classification

Resource classification is based on a combination of gold relative estimation error (AuREE), (calculated by Giroux during kriging), the domain code, and the presence of both Cu and Au estimates in each block.

- Measured = AuREE <40%

- Indicated = AuREE >= 40% and <68%

- Inferred = AuREE >= 68% or if Domain Code = 2 “Extended”

- Unclassified = If either Au g/t or Cu % or both was not estimated

Figure 1.19-1 is a histogram of the AuREE with the measured, indicated and inferred boundaries plotted on it.

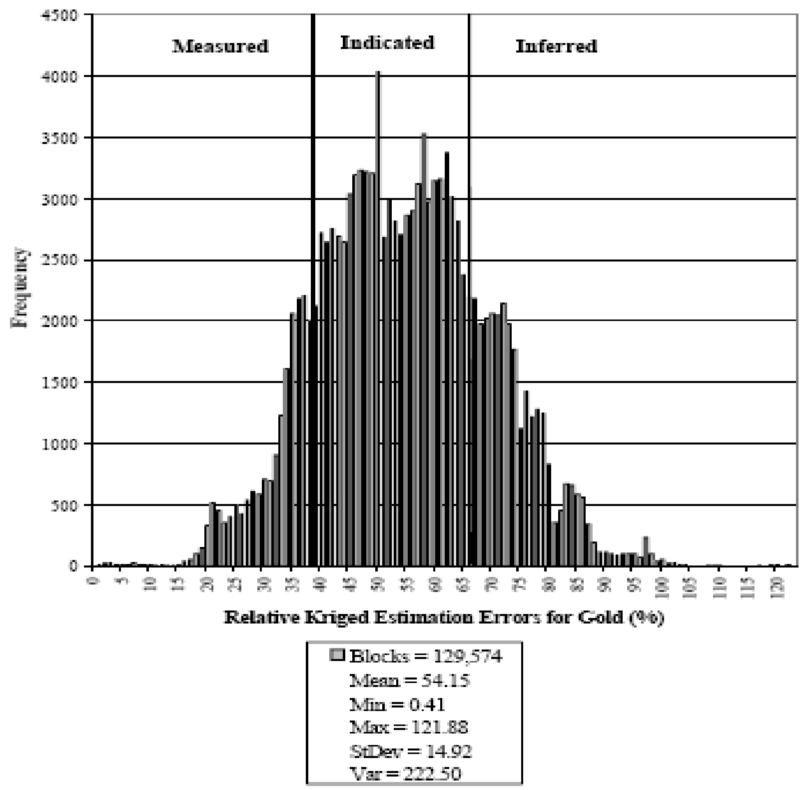


Figure 1.19-1: Histogram of Relative Estimation Errors for Gold (AuREE)

A further refinement to the classification of measured and indicated resources was introduced in April 2008 whereby blocks in the measured category by the above criteria were downgraded to indicated category blocks in the more sparsely drilled area west of the main zone (Giroux, 2008). This adjustment affected 9258 blocks, re-classifying approximately 2.7 Mt in the Leached Cap Zone at a 0.40 g/t cut-off, and 3.6 Mt, 14.8 Mt and 68.1 Mt in the supergene oxide, supergene sulphide and hypogene zones respectively, at a 0.30 % Cu EQ cut-off.

1.19.8 Resource Estimate Tabulations

The mineral resources were tabulated based on the estimated grades and a tonnage factor, which was applied to each lithologic/mineralogical type from 8,864 drill hole measurements of specific gravity (SG). The average SG of all results was 2.61, for Hypogene 2.64, for Supergene 2.61 and for Leached Cap 2.52. The estimated tonnage and grade for the Leached Cap/Oxide Gold zone, Supergene Oxide zone, Supergene Sulphide zone and Hypogene zone are presented in Tables 1.19-6, 1.19-7, 1.19-8, 1.19-9, 1.19-10, 1.19.11 and 1.19-12 respectively. In January 2004 a copper equivalent was calculated for each block based on metal prices of US$0.80/lb copper, US$350/oz for gold and US$4.50/lb for molybdenum as follows:

$$\text{Cu EQ \%} = (\text{Cu \%}) + (\text{Au g/t} \times 11.25 / 17.64) + (\text{Mo \%} \times 99.21 / 17.64)$$

The copper equivalent calculations reflect gross metal content and do not apply any adjustment factors for difference in metallurgical recoveries of gold, copper and molybdenum. This information can only be derived from definitive metallurgical testing which has yet to be completed.

The resource estimates for the Supergene Oxide, Supergene Sulphide and Hypogene zones were re-tabulated in April 2008 and stated in terms of two copper equivalent cut-off grades, 0.25% Cu EQ and 0.30% Cu EQ. The gold-dominant Leached Cap zone resource estimates remained unchanged at a gold cutoff grade of 0.40 g/t Au; copper equivalent values are presented for this zone for comparative purposes.

The mineral resources presented in the following tables are inclusive of the mineral reserve presented in Section 1.19.9.

Table 1.19-6: Leached Cap / Oxide Gold Zone at 0.40 g/t Au Cut-Off

Class	Cut-off Category	Tonnes (Million)	Au g/t	Cu %	Mo %	Cu EQ %
Inferred	>0.40 g/t Au	1	0.45	0.10	0.01	0.44
Indicated	>0.40 g/t Au	10	0.48	0.06	0.01	0.44
Measured	>0.40 g/t Au	28	0.60	0.08	0.03	0.60
Total Measured and Indicated		38	0.57	0.07	0.02	0.56

Table 1.19-7: Supergene Oxide Zone at 0.25 % Cu EQ Cut-Off

Class	Cut-off Category	Tonnes (Million)	Au g/t	Cu %	Mo %	Cu EQ %
Inferred	>0.25 % Cu EQ	9	0.18	0.26	0.01	0.45
Indicated	>0.25 % Cu EQ	25	0.19	0.27	0.01	0.46
Measured	>0.25 % Cu EQ	21	0.49	0.36	0.02	0.82
Total Measured and Indicated		46	0.33	0.31	0.02	0.63

Table 1.19-8: Supergene Oxide Zone at 0.30 % Cu EQ Cut-Off

Class	Cut-off Category	Tonnes (Million)	Au g/t	Cu %	Mo %	Cu EQ %
Inferred	>0.30 % Cu EQ	8	0.19	0.28	0.01	0.49
Indicated	>0.30 % Cu EQ	21	0.21	0.29	0.01	0.50
Measured	>0.30 % Cu EQ	21	0.49	0.36	0.02	0.82
Total Measured and Indicated		42	0.35	0.33	0.02	0.66

Table 1.19-9: Supergene Sulphide Zone at 0.25 % Cu EQ Cut-Off

Class	Cut-off Category	Tonnes (Million)	Au g/t	Cu %	Mo %	Cu EQ %
Inferred	>0.25 % Cu EQ	23	0.14	0.21	0.01	0.38
Indicated	>0.25 % Cu EQ	100	0.25	0.29	0.02	0.56
Measured	>0.25 % Cu EQ	33	0.47	0.39	0.03	0.87
Total Measured and Indicated		133	0.31	0.31	0.02	0.63

Table 1.19-10: Supergene Sulphide Zone at 0.30 % Cu EQ Cut-Off

Class	Cut-off Category	Tonnes (Million)	Au g/t	Cu %	Mo %	Cu EQ %
Inferred	>0.30 % Cu EQ	19	0.15	0.22	0.02	0.41
Indicated	>0.30 % Cu EQ	91	0.26	0.30	0.02	0.58
Measured	>0.30 % Cu EQ	32	0.47	0.39	0.03	0.88
Total Measured and Indicated		124	0.32	0.32	0.02	0.66

Table 1.19-11: Hypogene Zone at 0.25 % Cu EQ Cut-Off

Class	Cut-off Category	Tonnes (Million)	Au g/t	Cu %	Mo %	Cu EQ %
Inferred	>0.25 % Cu EQ	200	0.18	0.15	0.02	0.38
Indicated	>0.25 % Cu EQ	795	0.20	0.18	0.02	0.44
Measured	>0.25 % Cu EQ	111	0.27	0.22	0.03	0.55
Total Measured and Indicated		906	0.21	0.19	0.02	0.45

Table 1.19-12: Hypogene Zone at 0.30 % Cu EQ Cut-Off

Class	Cut-off Category	Tonnes (Million)	Au g/t	Cu %	Mo %	Cu EQ %
Inferred	>0.30 % Cu EQ	152	0.19	0.16	0.02	0.42
Indicated	>0.30 % Cu EQ	693	0.21	0.19	0.02	0.46
Measured	>0.30 % Cu EQ	106	0.28	0.22	0.03	0.56
Total Measured and Indicated		799	0.22	0.20	0.02	0.48

1.19.9 Mineral Reserve Estimate

1.19.9.1 Mineral Reserve

Table 1.19-13 presents the mineral reserve estimate for the Casino Project. The mill ore reserve amounts to 913.5 million tonnes at 0.212% copper, 0.237 g/t gold, and 0.0236% molybdenum. The heap leach reserve is an additional 77.9 million tonnes at 0.427 g/t gold and 0.062% copper.

For this reserve estimate, the measured mineral resource was converted to proven mineral reserve and the indicated mineral resource was converted to probable mineral reserve, with one exception. The low grade mill ore stockpile is considered probable mineral reserve regardless of the original classification of the in-situ material.

Table 1.19-13: Mineral Reserve

Mill Ore Reserve:	**Ore ktonnes**	**Tot Cu (%)**	**Gold (g/t)**	**Moly (%)**
Proven Mineral Reserve:				
Direct Mill Feed	143,804	0.284	0.354	0.0311
Probable Mineral Reserve:				
Direct Mill Feed	613,571	0.208	0.225	0.0255
Low Grade Stockpile	156,171	0.163	0.177	0.0095
Total Probable Reserve	769,742	0.199	0.215	0.0223
Proven/Probable Reserve				
Direct Mill Feed	757,375	0.222	0.249	0.0266
Low Grade Stockpile	156,171	0.163	0.177	0.0095
Total Mill Ore Reserve	913,546	0.212	0.237	0.0236
Heap Leach Reserve:	**Ore Ktonnes**	**Gold (g/t)**	**Tot Cu (%)**	**Moly (%)**
Proven Mineral Reserve	34,498	0.545	0.074	n.a.
Probable Mineral Reserve	43,419	0.333	0.053	n.a.
Total Heap Leach Reserve	77,917	0.427	0.062	n.a.

1.19.9.2 Economic Parameters for Mine Design

The mineral reserve presented in the previous section is based on a pit design and mine production schedule developed by IMC. Table 1.19-14 summarizes the economic parameters for mine design and scheduling. The commodity prices used for design are $2.79 per pound copper, $667.42 per ounce gold, and $14.00 per pound molybdenum and were specified by Western Copper personnel. It is important to note that these are preliminary estimates used to start the mine design process, and are not the final parameters developed for the economic analysis of the project.

The base unit costs for mining and flotation ore processing (including G&A) were estimated at $1.35 per total tonne and $6.15 per ore tonne respectively. Heap leach ore processing was estimated at $2.90 per ore tonne for a run-of-mine (ROM) leach process. A sustaining capital allowance of $0.15 per total tonne was also added to the mining cost for all material.

Heap leach recoveries were estimated at 50% for gold and 20% for copper by Western Copper personnel. Heap leach ore refers to ores in the leach capping of the ore body. Flotation ore refers to the supergene oxide, supergene sulphide, and hypogene zones of the orebody. For flotation ore the gold recovery in the plant was estimated at 67%. Copper and molybdenum recoveries in the flotation plant are estimated using the following constant tail grade relationships, based on preliminary flotation results:

$$\text{Copper recovery} = 99\%(Cuc\% - Cuw\% - 0.02\%) / Cuc\%$$

Molybdenum recovery = 76%(Mo% - 0.003%) / Mo%

Where,

Cuc% = Total copper grade
Cuw% = Weak acid soluble copper grade
Mo% = Molybdenum grade

The copper and molybdenum plant recoveries shown on Table 1.19-14 as 84.2% and 66.4% respectively reflect life of mine averages from the mine production schedule, including low grade ores.

Off –site costs were developed based on information in the following reports:

- "Casino Mine Project – Project Transportation Scoping Study" by Associated Engineering, dated March 2008.
- "Casino Project – Marketing Certain Key Revenue Assumptions for Copper Concentrates and Molybdenum" by Neil S. Seldon & Associates Ltd., dated February 2008.

Table 1.19-14: Economic Parameters for Mine Design (C$)

Parameter	Flotation Ore	Heap Leach
Commodity Prices:		
Copper Price Per Pound	2.79	2.79
Gold Price Per Ounce	667.42	667.42
Molybdenum Price Per Pound	14.00	N.A.
Mining Cost Per Total Tonne:		
Base Mining Cost	1.350	1.350
Sustaining Capital Allowance	0.150	0.150
Total Mining Cost	1.500	1.500
Processing and G&A Per Ore Tonne	6.150	2.900
Average Plant Recoveries:		
Copper Recovery	84.2%	20.0%
Gold Recovery	67.0%	50.0%
Molybdenum Recovery	66.4%	N.A.
Refinery Payables (Net of Conc Loss):		
Copper Payable	96.0%	80.0%
Gold Payable	97.0%	100.0%
Molybdenum Payable	98.5%	N.A.
Offsite Costs:		
Copper SRF Cost Per Pound	0.621	0.00
Gold Refining Per Ounce	5.000	5.00
Molybdenum Freight/Treatment Per Pound	2.382	0.00
NSR Factors:		
Copper Factor	45.91	49.21
Gold Factor	20.66	21.30
Moly Factor	252.29	N.A.
NSR Cutoff Grades:		
Breakeven Cutoff (C$/t)	7.50	4.25
Internal Cutoff (C$/t)	6.15	2.90
Stockpile Cutoff (C$/t)	6.75	N.A.
Note 1: Refinery Payables Reflect 0.5% Concentrate Loss		
Note 2: Offsite Costs for Copper in Leach Ore is Accounted in Payable Percentage		
Note 3: NSR Factors are Applied to Recovered Grades		

The copper off-site smelting, refining, and freight costs (SRF costs) and payable terms for the copper concentrate based on a concentrate grade of 28% and no known penalties. The Seldon study recommended $100 per tonne concentrate plus $0.10 per pound treatment and refining terms for long range forecasts, which is assumed to include price participation terms.

Based on the Associated Engineering (AE) study, the land freight cost for copper concentrate was estimated at $91.95 per tonne. IMC assumed $80 per tonne for ocean freight, based on an average of Seldon and AE estimates. IMC also assumed $0.02 per pound finished copper for marketing and insurance. All the smelting, refining, freight, insurance and marketing costs come to $0.62 per pound copper.

Off-site costs and payable terms for the molybdenum concentrate are based on a concentrate grade of 50%. Treatment charges are estimated at 15% of sales price which is in line with the Seldon study. Freight costs per tonne have been assumed to be the same as copper since it is uncertain where the material would be going. Note that given the high moly concentrate grade the transportation costs are relatively small. With an allowance for marketing and insurance this amounts to $2.38 per pound.

A cost of $5.00 per ounce for gold refining and a payable of 97.5% at the smelter (97.0% out of the plant) were also assumed by IMC.

To facilitate economic calculations, recovered copper, recovered gold, and recovered molybdenum grades were calculated on a block by block basis and stored in the block model. Note that these are recovered grades, not recovery percentages. For heap leach ore these are:

Rec_cu = 0.2 x cuc
Rec_au = 0.5 x au
Rec_mo = 0.0

And for flotation ore these are:

Rec_cu = 0.99 x (cuc – cuw – 0.02%)
Rec_au = 0.67 x au
Rec_mo = 0.76 x (mo – 0.003%)

The recovered copper and gold grades are only calculated for blocks with sulphide copper grades (cuc – cuw) above 0.02% and recovered moly is only calculated for moly above 0.003% and sulphide copper above 0.02%.

Then, NSR values were calculated for each block to use to classify blocks into ore and waste.

For the heap leach ore:

NSR_au = ($667.42 – $5.00) x rec_au x 1.0 / 31.103
NSR_cu = $2.79 x rec_cu x 0.8 x 22.046
NSR = NSR_au + NSR_cu

The internal NSR cutoff grade for heap leach ore is the processing plus G&A cost of $ 2.90 per ore tonne since all the recoveries and refining costs are accounted for in the NSR calculation. The breakeven NSR cutoff grade for heap leach ore is $4.25 per tonne (mining plus processing plus G&A costs).

And for flotation ore NSR values, as of the plant are:

NSR_cu = ($2.79 – $0.621) x rec_cu x 0.96 x 22.046
NSR_au = ($667.42 – $5.00) x rec_au x 0.97 / 31.103

NSR_mo = ($14.00 – $2.382) x rec_mo x 0.985 x 22.046
NSR = NSR_cu + NSR_au + NSR_mo

The internal NSR cutoff grade is the processing plus G&A cost of $6.15. Breakeven NSR cutoff is $7.50. The stockpile re-handle cutoff grade is estimated at $6.75 per ore tonne which covers processing plus G&A costs plus mining re-handle estimated at about half the nominal mining cost.

1.19.9.3 Slope Angles

Slope angles recommendations were developed by Knight Piésold Ltd. (KP) and documented in the letter report “Casino Copper-Gold Project – Preliminary Open Pit Slope Design”, dated January 24, 2008.

Forty five degree interramp angles were recommended for almost all slopes except for two areas on the west side of the pit for which a 42 degree interramp angle was recommended.

IMC used an overall slope angle of 42 degrees in the floating cone run to define the final pit. This was based on reviewing a preliminary floating cone and adjusting the overall angle for likely haulage roads.

1.19.9.4 Pit and Mining Phase Design

Floating cones were run at copper prices from $3.00 per pound down to $1.00 per pound to define the final pit limits and to guide the design of mining phases. Gold prices and molybdenum prices were varied at the same rate of change as the copper price for the various cones. A floating cone pit shell at prices of $2.79 copper, $667 gold, and $14.00 molybdenum was used as the basis for the final pit design. Figure 1.19-2 shows the base case floating cone.

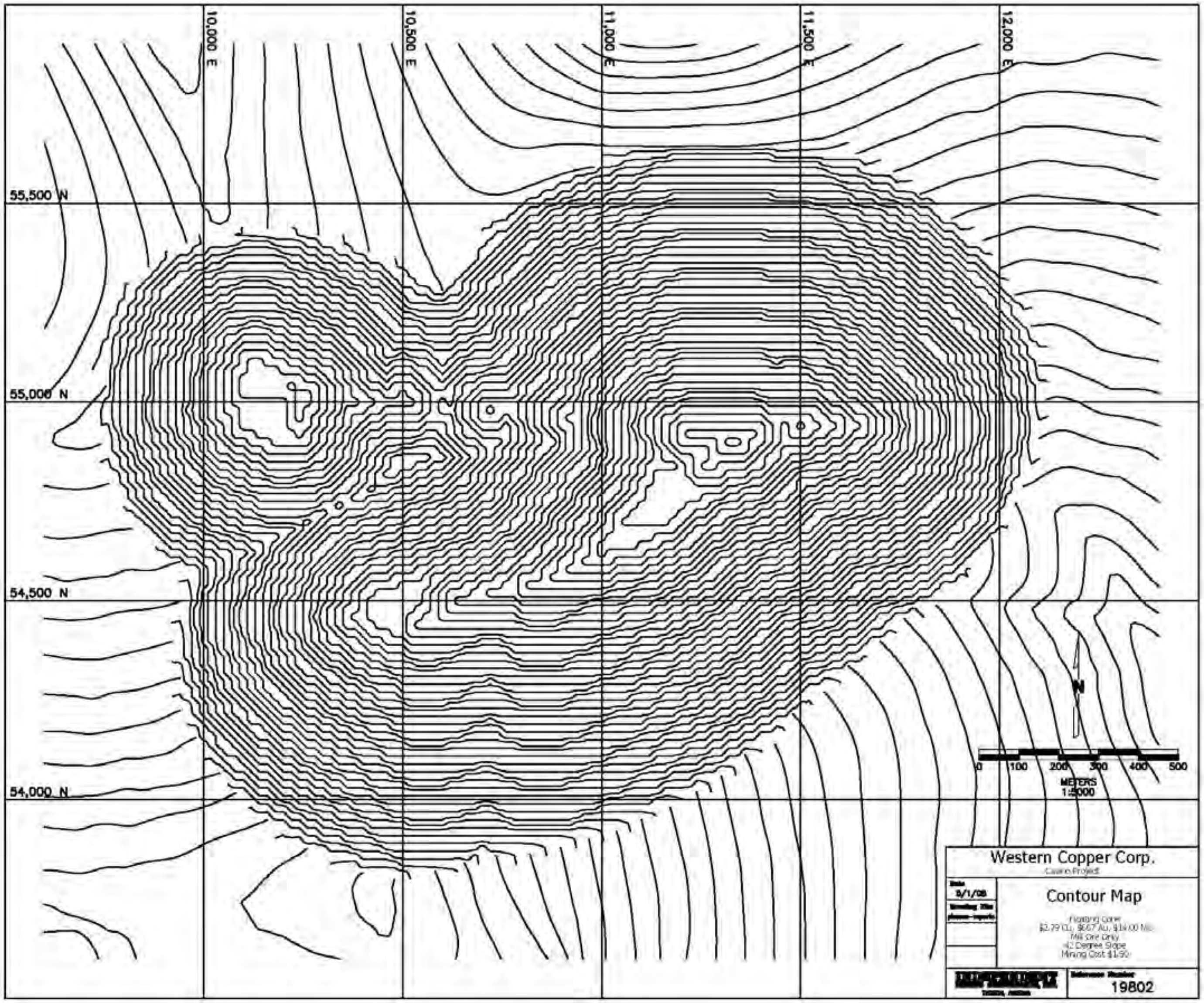


Figure 1.19-2 Floating Cone for Final Pit Design

Six mining phases were also developed for the study.

- Phase 1, the starter pit, is based on the $1.50 copper floating cone;
- Phase 2 is a C shape to the south, west, and north, mining out to the approximate $1.875 copper cone;
- Phase 3 is a push to the southwest to the $2.00 copper cone limits;
- Phase 4 mines the west satellite pit to the $2.00 copper limits with independent access.
- Phase 5 mines the north wall, including the west satellite pit, the final $2.79 copper cone limits.
- Phase 6 mines the south wall to the final pit limits. This defines the final pit for this study.

The phase designs include haul roads and adequate working room for large mining equipment. The roads are 32m wide at a maximum grade of 10%. The width will accommodate trucks up to the 220 tonne class, such as the Caterpillar 793D truck. The truck ultimately chosen for this study was a 345 tonne truck such as the Caterpillar 797D truck. For the next study the road width needs to be increased to approximately 36m to accommodate trucks of this size.

Figure 1.19-3 shows the final pit design.

1.19.9.5 Mining Production Schedule

Table 1.19-15 shows the mine production schedule. The schedule is based on three years of preproduction followed by 23.5 years of commercial mine production, though processing the low grade stockpile extends commercial production to just over 28 years. Note that the three year preproduction is significantly longer than is strictly required for the preproduction development of the copper mine. Western Copper desires to mine and process gold ores from the leach cap by conventional run-of-mine heap leaching during the preproduction period.

The upper section of the table shows direct feed mill ore by year. About 8.5 million tonnes of mill ore will be mined during preproduction and will be stockpiled and re-handled to make up part of the commercial Year 1 ore feed. Year 1 plant production is assumed to be 27,000 ktonnes, or about 75% of the nominal supergene capacity. It can also be seen that ore tonnes varies by year. The schedule is based 8,147 mill hours per year (365 x 24 x 93% availability) and supergene and hypogene processing rates of 4,469 tph and 3,933 tph respectively. These rates also amount to about 36,400 ktpy for supergene ore and 32,040 ktpy for hypogene ore.

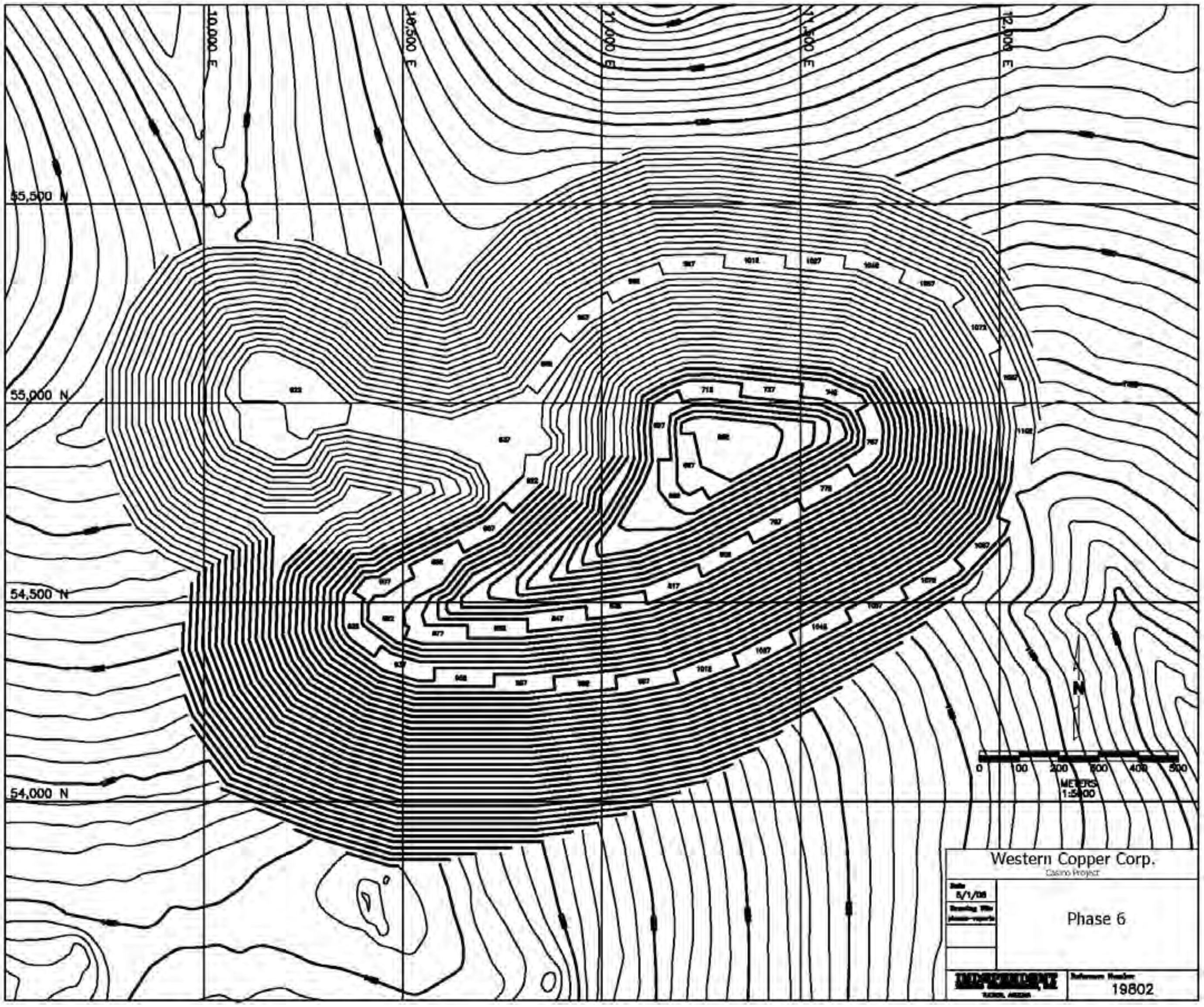


Figure 1.19-3: Final Pit Design

The schedule shows that in early years the NSR cutoff grade is high, $15.00 to $15.50 per tonne and declines to the internal cutoff grade of $6.15 per ore tonne toward the end of the mine life. The declining cutoff grade balances mine and plant capacities over the mine life to maximize the amount of metal produced given the mine total material rate limit. Along with the NSR value, grades of total copper, weak soluble copper, gold and molydenum are shown for each time period. The table also shows *plant recovered* copper, gold, and molydenum grades based on the recovery information provided to IMC. The recovered grades were calculated on a block by block basis and stored in the block model. There is also a throughput rate value reported in units of hours per ktonne since a variable was needed in scheduling that could be appropriately weighted by ore tonnages.

Total direct mill feed ore tonnes is 757,375 ktonnes at 0.223% total copper, 0.249 g/t gold and 0.0266% moly. The average life of mine NSR value is $16.68 per ore tonne for direct feed ore.

The operating schedule results in a significant amount of low grade ore that is assumed to be stockpiled and processed at the end of the mine life. The portion of the table under “Low Grade Mill Ore” shows this material. It is material with an NSR cutoff grade between $6.75 per tonne and the operating cutoff for the year. This material amounts to 156,171 ktonnes at 0.163% total copper, 0.177 g/t gold and 0.0095% moly. Recovered grades are also shown for that material.

The middle part of the table shows a proposed plant schedule. Year 1 production is 27,000 ktonnes and is made up of ore mined during Years -3, -2, -1, and 1. The schedule also incorporates the processing of the low grade at the end.

The table also shows a schedule for seven years production of gold ore between Years -3 and 4. This results in 77,917 ktonnes of ore at 0.427 g/t gold. It is assumed that this is processed by ROM heap leaching. The amount of gold ore is constrained by the allowable area of the leach pad. Recall that Table 3-5 shows up to 130.9 million tonnes of potential leach ore is available.

The bottom of the table summarizes tonnages. It can be seen that life of mine total material from the pit is 1.99 billion tonnes. Preproduction is 59.3 million tonnes staged over three years. Year 1 total material is scheduled at 59.5 million tonnes after which the peak material movement of 90 million tonnes per year is maintained over much of the mine life. Total waste is 995.2 million tonnes, almost exactly half of the total material. The waste excludes mill ore, low grade, and gold ore.

Table 1.19-15 Mine Production Schedule

Table 1-1: Mine Production Schedule - All Ore Types																																		
	(Units)	-3	-2	-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	26	27	28	29	TOTAL
Mill Ore:																																		
NSR Cutoff Grade	(C$)	15.00	15.00	15.00	15.00	15.50	14.50	13.25	12.50	12.50	11.50	11.50	10.50	10.50	9.50	9.00	9.00	8.25	8.75	9.00	8.75	7.50	7.75	7.75	6.15	6.15	6.15	6.15						
Ore Ktonnes	(kt)	34	2,121	6,334	18,511	35,400	33,370	33,300	34,550	32,125	32,404	32,439	33,600	32,225	32,200	32,250	32,100	32,000	32,000	32,050	32,201	32,100	32,170	32,000	32,025	32,025	32,025	13,816						757,375
NSR Value	(C$/t)	15.67	16.79	17.85	22.34	26.11	26.27	23.83	19.07	16.77	16.51	18.19	16.87	16.47	16.68	15.09	14.67	13.53	11.87	12.33	12.79	13.03	13.16	15.25	14.38	13.52	15.53	16.00						16.68
Total Copper	(%)	0.137	0.165	0.245	0.340	0.429	0.352	0.329	0.296	0.229	0.199	0.211	0.252	0.203	0.209	0.201	0.188	0.166	0.150	0.149	0.159	0.150	0.147	0.187	0.189	0.182	0.216	0.220						0.223
Weak Soluble Copper	(%)	0.031	0.046	0.072	0.087	0.075	0.019	0.012	0.028	0.001	0.005	0.005	0.020	0.002	0.002	0.003	0.001	0.000	0.000	0.000	0.002	0.001	0.002	0.000	0.000	0.000	0.000	0.000						0.011
Gold	(g/t)	0.573	0.611	0.554	0.555	0.419	0.396	0.355	0.291	0.291	0.250	0.244	0.243	0.220	0.229	0.201	0.202	0.193	0.170	0.170	0.169	0.182	0.176	0.234	0.201	0.178	0.201	0.190						0.249
Molybdenum	(%)	0.0228	0.0231	0.0197	0.0242	0.0298	0.0373	0.0311	0.0227	0.0202	0.0297	0.0362	0.0232	0.0301	0.0292	0.0249	0.0254	0.0251	0.0218	0.0246	0.0251	0.0272	0.0291	0.0260	0.0235	0.0222	0.0231	0.0253						0.0266
Recovered Copper	(%)	0.086	0.098	0.152	0.231	0.331	0.310	0.295	0.245	0.206	0.173	0.184	0.210	0.179	0.185	0.176	0.165	0.144	0.128	0.127	0.135	0.128	0.125	0.166	0.167	0.161	0.194	0.198						0.190
Recovered Gold	(g/t)	0.384	0.409	0.371	0.372	0.281	0.265	0.238	0.195	0.195	0.167	0.164	0.163	0.148	0.154	0.135	0.135	0.129	0.114	0.114	0.113	0.122	0.118	0.156	0.135	0.119	0.135	0.127						0.167
Recovered Moly	(%)	0.0150	0.0153	0.0127	0.0161	0.0203	0.0261	0.0214	0.0150	0.0130	0.0203	0.0252	0.0153	0.0206	0.0199	0.0167	0.0170	0.0168	0.0143	0.0164	0.0168	0.0184	0.0198	0.0175	0.0156	0.0146	0.0153	0.0170						0.0179
Throughput Rate	(hr/kt)	0.2407	0.2416	0.2442	0.2437	0.2474	0.2625	0.2632	0.2534	0.2727	0.2703	0.2700	0.2607	0.2717	0.2718	0.2716	0.2726	0.2735	0.2735	0.2733	0.2719	0.2729	0.2723	0.2735	0.2735	0.2735	0.2735	0.2735						0.2681
Plant Hours	(hrs)	8	477	1,438	4,195	8,145	8,146	8,151	8,142	8,147	8,146	8,145	8,146	8,143	8,139	8,146	8,138	8,139	8,139	8,146	8,143	8,147	8,147	8,139	8,146	8,146	8,146	3,514						188,805
Low Grade Mill Ore:																																		
NSR Cutoff Grade	(C$)	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75	6.75										
Ore Ktonnes	(kt)	33	1,465	2,558	6,420	11,718	10,268	6,699	15,335	18,424	8,120	4,143	10,627	13,726	6,901	5,289	5,725	3,895	4,215	6,308	5,188	2,216	3,725	3,173										156,171
NSR Value	(C$/t)	14.40	13.76	12.73	11.81	11.39	11.20	10.58	9.94	10.12	9.42	9.49	8.74	8.64	8.24	8.00	7.94	7.65	7.76	7.86	7.75	7.16	7.27	7.23										9.46
Total Copper	(%)	0.126	0.200	0.199	0.193	0.229	0.205	0.178	0.188	0.156	0.162	0.170	0.158	0.156	0.146	0.121	0.124	0.125	0.117	0.115	0.115	0.123	0.133	0.122										0.163
Weak Soluble Copper	(%)	0.028	0.051	0.052	0.069	0.040	0.014	0.013	0.028	0.003	0.012	0.018	0.016	0.015	0.009	0.002	0.001	0.000	0.000	0.001	0.003	0.012	0.010	0.000										0.016
Gold	(g/t)	0.568	0.412	0.381	0.310	0.192	0.185	0.208	0.170	0.187	0.190	0.177	0.178	0.165	0.136	0.127	0.135	0.128	0.125	0.125	0.118	0.107	0.134	0.154										0.177
Molybdenum	(%)	0.0184	0.0145	0.0119	0.0175	0.0083	0.0073	0.0088	0.0094	0.0108	0.0070	0.0080	0.0065	0.0074	0.0085	0.0120	0.0102	0.0089	0.0116	0.0126	0.0131	0.0109	0.0068	0.0052										0.0095
Recovered Copper	(%)	0.078	0.128	0.125	0.103	0.167	0.170	0.144	0.139	0.131	0.129	0.131	0.121	0.119	0.116	0.098	0.102	0.104	0.096	0.093	0.091	0.090	0.102	0.101										0.125
Recovered Gold	(g/t)	0.380	0.276	0.255	0.208	0.128	0.124	0.139	0.114	0.125	0.127	0.119	0.119	0.110	0.091	0.085	0.090	0.086	0.083	0.084	0.079	0.072	0.090	0.103										0.118
Recovered Moly	(%)	0.0117	0.0087	0.0067	0.0110	0.0042	0.0034	0.0044	0.0049	0.0060	0.0035	0.0041	0.0028	0.0035	0.0042	0.0069	0.0055	0.0044	0.0065	0.0074	0.0077	0.0062	0.0030	0.0018										0.0050
Throughput Rate	(hr/kt)	0.2407	0.2411	0.2413	0.2432	0.2504	0.2633	0.2608	0.2604	0.2724	0.2565	0.2508	0.2557	0.2629	0.2676	0.2719	0.2733	0.2735	0.2735	0.2719	0.2680	0.2665	0.2679	0.2735										0.2626
Plant Hours	(hrs)	7	328	574	1,452	2,729	2,514	1,625	3,714	4,667	1,937	966	2,527	3,356	1,717	1,337	1,455	991	1,072	1,595	1,293	549	928	807										38,143
Proposed Mill Schedule:																																		
NSR Cutoff Grade	(C$)	15.00	15.00	15.00	15.00	15.50	14.50	13.25	12.50	12.50	11.50	11.50	10.50	10.50	9.50	9.00	9.00	8.25	8.75	9.00	8.75	7.50	7.75	7.75	6.15	6.15	6.15	6.15	6.75	6.75	6.75	6.75	6.75	
Ore Ktonnes	(kt)				27,000	35,400	33,370	33,300	34,550	32,125	32,404	32,439	33,600	32,225	32,200	32,250	32,100	32,000	32,000	32,050	32,201	32,100	32,170	32,000	32,025	32,025	32,025	32,788	33,360	33,360	33,360	33,360	3,761	913,546
NSR Value	(C$/t)				20.84	26.11	26.27	23.83	19.07	16.77	16.51	18.19	16.87	16.47	16.68	15.09	14.67	13.53	11.87	12.33	12.79	13.03	13.16	15.25	14.38	13.52	15.53	12.22	9.46	9.46	9.46	9.46	9.46	15.45
Total Copper	(%)				0.304	0.429	0.352	0.329	0.296	0.229	0.199	0.211	0.252	0.203	0.209	0.201	0.188	0.166	0.150	0.149	0.159	0.150	0.147	0.187	0.189	0.182	0.216	0.187	0.163	0.163	0.163	0.163	0.163	0.212
Weak Soluble Copper	(%)				0.080	0.075	0.019	0.012	0.028	0.001	0.005	0.005	0.020	0.002	0.002	0.003	0.001	0.000	0.000	0.000	0.002	0.001	0.002	0.000	0.000	0.000	0.000	0.009	0.016	0.016	0.016	0.016	0.016	0.012
Gold	(g/t)				0.559	0.419	0.396	0.355	0.291	0.291	0.250	0.244	0.243	0.220	0.229	0.201	0.202	0.193	0.170	0.170	0.169	0.182	0.176	0.234	0.201	0.178	0.201	0.183	0.177	0.177	0.177	0.177	0.177	0.237
Molybdenum	(%)				0.0231	0.0298	0.0373	0.0311	0.0227	0.0202	0.0297	0.0362	0.0232	0.0301	0.0292	0.0249	0.0254	0.0251	0.0218	0.0246	0.0251	0.0272	0.0291	0.0260	0.0235	0.0222	0.0231	0.0161	0.0095	0.0095	0.0095	0.0095	0.0095	0.0236
Recovered Copper	(%)				0.202	0.331	0.310	0.295	0.245	0.206	0.173	0.184	0.210	0.179	0.185	0.176	0.165	0.144	0.128	0.127	0.135	0.128	0.125	0.166	0.167	0.161	0.194	0.156	0.125	0.125	0.125	0.125	0.125	0.179
Recovered Gold	(g/t)				0.375	0.281	0.265	0.238	0.195	0.195	0.167	0.164	0.163	0.148	0.154	0.135	0.135	0.129	0.114	0.114	0.113	0.122	0.118	0.156	0.135	0.119	0.135	0.122	0.118	0.118	0.118	0.118	0.118	0.159
Recovered Moly	(%)				0.0152	0.0203	0.0261	0.0214	0.0150	0.0130	0.0203	0.0252	0.0153	0.0206	0.0199	0.0167	0.0170	0.0168	0.0143	0.0164	0.0168	0.0184	0.0198	0.0175	0.0156	0.0146	0.0153	0.0101	0.0050	0.0050	0.0050	0.0050	0.0050	0.0157
Copper Recovery	(%)				66.46%	77.16%	88.07%	89.67%	82.77%	89.96%	86.93%	87.20%	83.33%	88.18%	88.52%	87.56%	87.77%	86.75%	85.33%	85.23%	84.91%	85.33%	85.03%	88.77%	88.36%	88.46%	89.81%	83.40%	76.91%	76.91%	76.91%	76.91%	76.91%	84.17%
Gold Recovery	(%)				67.01%	67.06%	66.92%	67.04%	67.01%	67.01%	66.80%	67.21%	67.08%	67.27%	67.25%	67.16%	66.83%	66.84%	67.06%	67.06%	66.86%	67.03%	67.05%	66.67%	67.16%	66.85%	67.16%	66.88%	66.90%	66.90%	66.90%	66.90%	66.90%	67.00%
Moly Recovery	(%)				66.09%	68.12%	69.97%	68.81%	66.08%	64.36%	68.35%	69.61%	65.95%	68.44%	68.15%	67.07%	66.93%	66.93%	65.60%	66.67%	66.93%	67.65%	68.04%	67.31%	66.38%	65.77%	66.23%	62.36%	52.95%	52.95%	52.95%	52.95%	52.95%	66.43%
Throughput Rate	(hr/kt)				0.2436	0.2474	0.2625	0.2632	0.2534	0.2727	0.2703	0.2700	0.2607	0.2717	0.2718	0.2716	0.2726	0.2735	0.2735	0.2733	0.2719	0.2729	0.2723	0.2735	0.2735	0.2735	0.2735	0.2672	0.2626	0.2626	0.2626	0.2626	0.2626	0.2671
Plant Hours	(hrs)				6,118	8,145	8,146	8,151	8,142	8,147	8,146	8,145	8,146	8,143	8,139	8,146	8,138	8,139	8,139	8,146	8,143	8,147	8,147	8,139	8,146	8,146	8,146	8,148	8,148	8,148	8,148	8,148	918	226,947
Gold Ore:																																		
NSR Cutoff Grade	(C$)	2.90	2.90	2.90	2.90	2.90	2.90	2.90																										
Ore Ktonnes	(kt)	4,154	13,376	14,833	19,919	4,746	9,731	11,158																										77,917
NSR Value	(C$/t)	4.37	5.85	6.42	5.66	4.04	3.91	3.64																										5.16
Gold	(g/t)	0.361	0.488	0.538	0.460	0.303	0.338	0.300																										0.427
Total Copper	(%)	0.053	0.066	0.069	0.077	0.082	0.032	0.045																										0.062
Weak Soluble Copper	(%)	0.012	0.012	0.014	0.024	0.018	0.009	0.013																										0.016
Molybdenum	(%)	0.0093	0.0132	0.0192	0.0286	0.0209	0.0042	0.0066																										0.0165
Recovered Gold	(g/t)	0.181	0.244	0.269	0.230	0.152	0.169	0.150																										0.213
Recovered Copper	(%)	0.011	0.013	0.014	0.015	0.016	0.006	0.009																										0.012
Tonnage Summary:																																		
Mill Ore	(kt)	34	2,121	6,334	18,511	35,400	33,370	33,300	34,550	32,125	32,404	32,439	33,600	32,225	32,200	32,250	32,100	32,000	32,000	32,050	32,201	32,100	32,170	32,000	32,025	32,025	32,025	13,816						757,375
Low Grade Ore	(kt)	33	1,465	2,558	6,420	11,718	10,268	6,699	15,335	18,424	8,120	4,143	10,627	13,726	6,901	5,289	5,725	3,895	4,215	6,308	5,188	2,216	3,725	3,173	0	0	0	0						156,171
Gold Ore	(kt)	4,154	13,376	14,833	19,919	4,746	9,731	11,158	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0						77,917
Total Material	(kt)	8,212	20,786	30,301	59,520	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	82,857	87,021	55,261	49,386	42,667	20,624						1,986,635
Waste Material	(kt)	3,991	3,824	6,576	14,670	38,136	36,631	38,843	40,115	39,451	49,476	53,418	45,773	44,049	50,899	52,461	52,175	54,105	53,785	51,642	52,611	55,684	46,962	51,848	23,236	17,361	10,642	6,808						995,172
Waste to Embankment	(kt)	3,209	2,981	5,126	10,402	13,634	12,605	17,074	14,800	17,553	18,745	12,641	10,045	4,534	9,067	8,157	8,117	8,329	10,530	12,926	13,028	10,088	7,050	0	0	0	0	0						230,641
Waste to Waste Pile	(kt)	782	843	623	1,214	7,435	6,478	6,120	5,935	4,345	14,626	12,150	8,015	7,061	958	3,851	2,156	1,373	96	645	0	122	109	2,272	140	213	366	903						88,832
Waste to TMF Co-Dispo	(kt)	0	0	827	3,054	17,067	17,548	15,649	19,380	17,553	16,105	28,627	27,713	32,454	40,874	40,453	41,902	44,403	43,159	38,071	39,583	45,474	39,803	49,576	23,096	17,148	10,276	5,905						675,699
Waste to Ore Ratio	(none)	0.95	0.23	0.28	0.33	0.74	0.69	0.76	0.80	0.78	1.22	1.46	1.03	0.96	1.30	1.40	1.38	1.51	1.49	1.35	1.41	1.62	1.31	1.47	0.73	0.54	0.33	0.49						1.004
Mine Re-handle	(kt)			8,489																								18,972	33,360	33,360	33,360	33,360	3,761	164,660

1.20 OTHER RELEVANT DATA AND INFORMATION

1.20.1 Geotechnical – Site Conditions and Foundation Design

Knight Piésold Ltd. (KP) prepared a preliminary foundation assessment for the proposed plant site based on results of geotechnical investigation programs and laboratory testing conducted by KP in 1993 and 1994.

1.20.1.1 Site Conditions

The investigation results indicate that the surficial materials generally comprise of a thin layer of topsoil (peats and organics) underlain by a well graded colluvium veneer and a thicker layer of residual soil (sands with some gravel and trace silt). Bedrock lies at shallow depths of from 1 to 3 meters and comprises blocky, hard fresh to heavily weathered grandiorite. Permafrost is prevalent in valley bottoms, in shaded draws and on north facing slopes. Although permafrost was recorded in some areas of the plant site, the majority of the area was found to be well drained and ice free. Permafrost issues can be avoided by constructing the plant site facilities away from shaded areas on the south facing slopes of the plant site area, or by excavating down to the bedrock.

Based on the laboratory test results, the average unit weight of the residual soil is 19.6 kN/m³, with a corresponding moisture content of 8%. It is classified as sand with a trace to some gravel and trace fines (silt and clay). The colluvium was found to be at greater a natural moisture content of 13%. It is classified as well graded gravelly, silty sand.

1.20.1.2 Foundation Assessment Recommendations

Knight Piésold's preliminary foundation recommendations are that footings for critical plant site structures be constructed directly on the bedrock due to the expectation that the depth of freeze exceeds the depth to bedrock. Bedrock is generally encountered at shallow depths of from 1 to 3 meters and will provide a solid foundation that is not susceptible to frost effects. Strip footings for plant site structures should be at least 1.0 m wide. Negligible settlement should result if the footings exert no more than a pressure of 500 kPa on weathered bedrock.

Overburden materials should be removed at a 2:1 slope. The site should be kept dry and well drained during and after construction.

1.20.2 Tailings Design

Tailings and reactive waste rock will be deposited in an impoundment located in the Casino Creek valley southeast from the open pit. The Tailings Management Facility (TMF) has been designed to provide sufficient capacity to store 974 million tonnes of tailings (696 million m³ at an average dry density of 1.4 t/m³) and co-disposal of up to 598 million tonnes of reactive waste rock.

Specific overall features of the TMF are illustrated on Figure 1.20-1 and are listed below:

- Two earth-rockfill, zoned embankments, referred to as the Main Embankment and West Saddle Embankment;
- Tailings distribution system;
- Reclaim water system;
- Reactive waste storage area;
- Supernatant (surface) water pond;
- Seepage collection ditches and ponds/sumps.

The embankments will be developed in stages throughout the life of the project using a combination of suitable non-reactive overburden and waste rock materials from stripping operations at the open pit and local borrow sources. The mine waste scheduling has been designed accordingly.

The Main Embankment will be expanded in stages across the Casino Creek valley. The Stage I (Starter) embankment is designed to accommodate tailings and reactive waste rock production for 1 year of operations. The West Saddle Embankment will be constructed at the southwestern corner of the TMF, as shown on Figure 1.20-1 in the Illustrations section at the end of this report, and is required by year 20 of operations. The embankments will be constructed as water-retaining zoned structures with a low permeability core zone and appropriate filter and transition zones to prevent downstream migration of fines. The core zone will include a seepage cut-off keyed into competent rock in the foundation. Staged expansions of the embankments will utilize the centerline method of construction. Typical sections through the Main Embankment and West Saddle Embankment that illustrate their staging are shown on Figure 1.20-2 in the Illustrations section at the end of this report.

The Stage I (starter) Main Embankment will be constructed in pre-production years concurrent with open pit stripping, with construction scheduled to provide enough time to impound sufficient surface water runoff for mill start-up. Mill process water for ongoing operations will be reclaimed from the TMF Water for the process will be reclaimed from the supernatant pond and returned to the mill head tank using floating barge mounted pumps, on-shore booster pump-stations and a high pressure pipeline. Initially reclaim will be from the northwest arm of the TMF and will be routed directly to the head tank. The on-going encroachment of the Reactive Waste Storage Area into this initial reclaim area and tailings deposition form the Main Embankment will reduce the availability of clean water from this location. At an appropriate time a second floating pump-station will be installed on the east side of the facility, in the central arm of the surface pond. A new reclaim line will be routed north and west around the TMF to the reclaim head tank. It will include two booster pump-stations.

The new line will not cross the Main Embankment, thereby eliminating any risk to the embankment dam in the event of pipeline leakage.

A preliminary site wide annual water balance, completed for average precipitation conditions indicates that the site is in a water deficit for the majority of the mine's operating life. To maintain a minimum one meter water cover over the tailings, a make-up water supply system will be required for the milling process after approximately Year 5 of operations. It is estimated that a supernatant pond volume of approximately 10 million m³ is required to maintain a one meter water cover at closure.

Seepage water losses from the TMF will be collected in seepage interception systems constructed downstream of the embankments. Intercepted seepage will be conveyed to collection sumps/ponds from where it will be pumped back into the TMF. Special design provisions to minimise seepage losses include the installation of a seepage cut-off beneath the Main Embankment and West Saddle Embankment, a seepage interception and recovery system and contingency measures for groundwater recovery and recycle.

Details of the site characteristics, geotechnical and water management considerations for the tailings facility design, pipeworks (tailings and reclaim systems), seepage collection and reclamation and closure are contained in the Knight Piésold "Report on Pre-Feasibility Design of the Tailings Management Facility" (Ref. No. VA101-325/1-2).

1.20.3 Waste Rock Storage

A total of 995 million tonnes of waste rock will be mined from the open Pit. A preliminary waste characterization study indicates that approximately 598 million tonnes of the waste rock is potentially acid generating material. The current plan is to store this potentially reactive material subaqueously within the Tailings Management Facility (TMF).

Non-reactive mine waste material will be used for embankment construction at the TMF. Knight Piésold estimates approximately 230 million tonnes of nonreactive waste rock and the 16 million tonnes of non-reactive overburden may be used for embankment fill. The remaining 151 million tonnes of non-reactive waste rock not used in the TMF embankment will be stored within a Waste Storage Area. There is also potential to use some of this non-reactive waste rock for construction of the heap leach facility and other purposes. However, for this study the Waste Storage Area has been sized to provide potential storage for all of the non-reactive mine waste rock. The quantity of mine waste requiring storage in the Waste Storage Area will be dependent on the actual construction fill requirements for the TMF and heap leach facility, and also on the suitability of the materials as construction fill.

Low grade ore, amounting to 156.2 Mt, is stockpiled near the plant (Figure 1.25-1) for processing at the end of the mine life.

Geotechnical conditions generally match those discussed in Section 1.20.1 above.

Where permafrost is present, surface vegetation will be stripped well in advance of waste or low grade ore deposition to allow thaw.

During operations the waste rock piles will be developed with 30 m high benches, each offset from the downstream edge by 20 m berms to provide an overall slope of approximately 2H:1V. This will maintain on-going stability of the pile and will facilitate final grading to establish a uniform slope for reclamation. Each bench will be constructed from lifts of approximately 15 m by end-dumping the waste material to form repose slope angles of approximately 1.3H:1V. The waste pile slope configuration is shown schematically on Figure 4.2.2. The final maximum elevation for the waste rock pile is approximately 1,382 m. At closure the crest of the waste pile will be rounded to provide long term stability.

1.20.4 Leach Pad

The facility is located on a southwest facing hill-slope approximately 1,500 m east of the deposit area.

While initial construction will be smaller, the final heap leach pad required for an ore tonnage of 75 Mt will have a surface area of approximately 935,000 m². An events pond will provide storage for excess leachate and storm water runoff. The heap leach pad confining embankment and events pond will require approximately 280,500 m³ of embankment fill for construction. The required solution storage for the events pond is approximately 335,000 m³. The total quantity of geosynthetic liner required for the heap leach pad and confining embankment is approximately 945,000 m².

The total quantity of geosynthetic liner for the events pond is approximately 77,000 m². The following design features, operational requirements and construction methods which will be required for the proposed facility:

- Pumping of pregnant solution to a gold extraction plant, located west of the heap leach pad.
- It may be required to excavate the foundation down to competent bedrock in areas with permafrost to eliminate potential settlement and instability resulting from thawing ice-rich overburden. These materials will require containment and sediment control upon thawing. The extent of ice-rich overburden throughout the heap leach pad area will be confirmed through additional site investigations. The estimate assumes excavation to bedrock at a depth of 3 meters.
- A confining embankment will be constructed along the toe of the pad to provide confinement and stability. Pre-production earthworks will also include the construction of an events pond for temporary storage of storm runoff and overflow of pregnant solution during shut-down.
- A composite liner system comprising a geomembrane, compacted soil liner and leachate detection and recovery system to maximize leachate collection and minimize seepage losses will be constructed over the upper portion of the leach pad. A double composite liner system comprising two geomembrane liners, a compacted soil liner and two geotextiles will be constructed in the lower portion of the leach pad (potential ponding area) and events pond and will include a leachate detection and recovery system for intercepting and collecting any leakage through the inner liner.

- Suitable non-reactive mine waste rock will be used for heap leach facility construction. Borrow materials may be required to augment the mine waste and overburden in some applications. The estimate assumes sufficient mine waste rock will be available and no borrow will be required.
- The heap leach pad will be developed in stages by loading in successive lifts upslope from the confining embankment. This will provide initial stability and minimize initial capital costs. The pad will be developed in 8 m lifts constructed at repose bench face angles of approximately 1.4H:1V. Bench widths approximately 9 m wide will be left at the toe of each lift to establish a final overall slope angle of 2.5H:1V.

1.21 INTERPRETATION AND CONCLUSIONS

The Casino mineral occurrence can be successfully and economically exploited by proven mining and processing methods under the conditions and assumptions outlined in this report.

Substantial opportunities exist to enhance the project economics including:

- Inclusion of revenue from the recovery of silver
- Conversion of inferred resource into measured and indicated
- Increasing the overall resource
- Sharing of infrastructure development costs with other parties
- Refined engineering during feasibility study
- Investigation of local sources of lime

1.22 RECOMMENDATIONS

Western Copper Corporation should continue to further define the resource through exploration drilling, particularly in the more sparsely drilled area west of the main zone and deep drilling adjacent to the microbreccia pipe.

Western Copper Corporation should continue with the environmental studies and permitting efforts now underway.

Western Copper Corporation should continue with the engineering effort in support of permitting and to advance efforts toward preparation of a full feasibility study.

Western Copper Corporation should continue to monitor developments in the Yukon, northern British Columbia and Alaska to be in a position to share infrastructure development.

1.23 REFERENCES

Archer Cathro & Associates (1981) Ltd., Williams Creek Copper Oxide Project, Preliminary Submission Prepared for the Regional Environmental Review Committee, Yukon Territory, August 1991.

Associated Engineering (BC), Casino Mine Project, Project Transportation Scoping Study, Unpublished Company Report, Western Copper Corporation, April 2008.

Bower, B., Case, T., DeLong, C., and Rebagliati, C.M., Casino Project, 1994 Exploration and Geotechnical Drilling Program on the Casino Porphyry Copper-Gold-Molybdenum Deposit, Unpublished Company Report, Pacific Sentinel Gold Corp., May 1995.

Bower, B., DeLong, C., Payne, J., and Rebagliati, C.M., The Oxide-Gold, Supergene and Hypogene Zones at the Casino Gold-Copper-Molybdenum Deposit West Central Yukon, Special Volume 46, Canadian Institute of Mining, Metallurgy and Petroleum, 1995.

Brenda Process Technology, Preliminary Flotation Testing for Copper, Gold and Molybdenum, Unpublished Company Report, Pacific Sentinel Gold Corp., March 1994.

Giroux, G.H., A Geostatistical Resource Evaluation of the Casino Cu-Au-Mo Deposit, Yukon, Unpublished Company Report, Pacific Sentinel Gold Corp., September 1995.

Giroux Consultants Ltd., Memorandum Re. Resource Classification for Casino Cu-Au-Mo Project, April 28, 2008, Memo prepared for Western Copper Corporation

G&T Metallurgical Services Ltd., Metallurgical Assessment of the Casino Deposit, Yukon Territory, Unpublished Company Report, Western Copper Corporation, May 2008.

Hallam Knight Piésold, Casino Project – Data Report 1993 – 1995, Unpublished Company Report, Pacific Sentinel Gold Corp., March 1997

Heap Leach/Carbon Adsorption Study for Oxide Gold Zone, Unpublished Company Report, Pacific Sentinel Gold Corp., April 1995.

Kappes, Cassiday & Associates, Cyanide Bottle Roll Tests, Unpublished Company Report, Pacific Sentinel Gold Corp., October 1994.

Kappes, Cassiday & Associates, Acid Leach and Cyanide Leach Bottle Roll Tests, Unpublished Company Report, Pacific Sentinel Gold Corp., March 1995.

Kerr Wood Leidal Associates Limited, Scoping-Level Power Study for Casino Mineral Deposit, Yukon, Final Report, Unpublished Company Report, Western Copper Corporation, June 2008.

Knight Piésold Ltd., Casino Copper-Gold Project – Preliminary Open Pit Slope Design, Unpublished Company Report, Western Copper Corporation, January 24, 2008.

Knight Piésold Ltd., Casino Copper-Gold Project – Preliminary Plant Site Foundation Assessment, Unpublished Company Report, Western Copper Corporation, February 15, 2008.

Knight Piésold Ltd., Casino Copper-Gold Project – Waste Storage Area and Low Grade Ore Stockpiles, Unpublished Company Report, Western Copper Corporation, April 16, 2008.

Knight Piésold Ltd., Report on Conceptual Design of Heap Leach Facility (Ref. No. 1833/1), Unpublished Company Report, Pacific Sentinel Gold Corp., January 1995.

Knight Piésold Ltd., Western Copper Corporation Casino Copper-Gold Project, Pre-feasibility Design of the Heap Leach Facility, Unpublished Company Report, Western Copper Corporation, April 23, 2008.

Knight Piésold Ltd., Western Copper Corporation Casino Copper-Gold Project, Pre-feasibility Design of the Tailings Management Facility, Unpublished Company Report, Western Copper Corporation, May 30, 2008

Lauga and Associates Consulting Ltd. in Association with Associated Engineering, Ltd., Casino Mine Project, Conceptual Analysis of Port Development Options, Unpublished Company Report, Western Copper Corporation, May 5, 2008.

Lynden Incorporated, Intermodal Transportation Study, Unpublished Company Report, Pacific Sentinel Gold Corp., March 1995.

Melis Engineering Ltd., Metallurgical Test Program, Unpublished Company Report, Pacific Sentinel Gold Corp., Unpublished Company Report, Pacific Sentinel Gold Corp., July 1995.

METCON Research, Casino Project – Agitated and Column Cyanide Leach Study, Unpublished Company Report, Western Copper Corporation, July 2008.

Minto Explorations Ltd., Annual Report 1999, Quarterly Reports March 2000 through March 2002.

Neil S. Seldon & Associates Ltd., Casino Project – Marketing Certain Key Revenue Assumptions for Copper Concentrates and Molybdenum, Unpublished Company Report, Western Copper Corporation, February 2008.

Pacific Sentinel Gold Corp., Casino Project Internal Files.

Rebagliati, C.M. and Banner R.H., Qualifying Report Casino Property Yukon Territory, January 23, 2003, Report prepared for CRS Copper Resources Corp. and First Trimark Ventures.

Rebagliati, C.M. and Titley, E.D., Report on the Revised Resource Estimate Casino Property Yukon Territory, February 27, 2004, Report prepared for Lumina Copper Corp.

Scoping Study and Overview Report, 25,000 TPD Open Pit Mine and Concentrator Complex, Unpublished Company Report, Pacific Sentinel Gold Corp., November 1995.

SGS Lakefield Research, Pre Feasibility Grinding Circuit Study for Western Copper Corporation – Casino Project based on Small-Scale Tests Data, Unpublished Company Report, Western Copper Corporation, February 2008.

Titley, E.D., Memorandum Re. Casino Sampling and Analytical Procedures, May 14, 2001, Memo prepared for Shari Gardiner and Company File.

1.24 DATE AND SIGNATURES

The information in this report is current as of 5 August 2008.

1.25 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

1.25.1 Mine Operations

A mine plan was developed to supply ore to a conventional copper sulphide flotation plant with the capacity to process 32.0 to 36.4 million tonnes per year, depending on the mix of supergene and hypogene ores. The mine is scheduled to operate two 12 hour shifts per day, 365 days per year. This will require four mining crews. The crews will operate 7 days on-7 days off from a fly in-fly out camp.

Mining is by conventional open pit methods with drilled and blasted rock loaded onto rigid frame haul trucks by large shovels. Table 1.25-1 summarizes the major mining equipment required.

Table 1.25-1: Summary of Mine Major Equipment Requirements

Equipment Type	Size	Year -1	Year 2	Peak
P&H 3200 XPC Drill	(311 mm)	1	3	3
P&H 4100C Shovel	(50.8 m^3)	1	3	3
Caterpillar 797D Truck	(345 t)	8	20	25
Caterpillar D11T Dozer	(634 kW)	2	3	3
Caterpillar D10T Dozer	(433 kW)	2	3	3
Caterpillar 834H Dozer	(372 kW)	3	3	3
Caterpillar 16M Grader	(198 kW)	2	3	3
Caterpillar 777 Water Truck	(90,000 L)	2	3	3
Caterpillar 992K Loader	(11.5 m^3)	1	1	1
Caterpillar 777D Truck	(91 t)	4	4	4
Caterpillar 345C Backhoe	(2.3 m^3)	1	1	1
Atlas Copco ECM660 Drill	(114 mm)	1	1	1

Total waste in the IMC mine plan amounts to 995.2 million tonnes, almost exactly half of the total material. This waste material is disposed as follows:

- 230.6 million tonnes of leach cap, supergene oxide, and non-reactive hypogene waste is used to build the tailings embankment.
- 88.9 million tonnes of leach cap, supergene oxide, and non-reactive hypogene waste is placed in the waste rock pile south of the pit.
- 675.7 million tonnes of supergene sulphide and hypogene waste will be co-disposed with tailings in the tailings management facility.

Additional rock storage facilities during the life of the project include:

- The heap leach pad which at the end of the project will contain 77.9 million tonnes of spent, non-reactive ore.
- Low grade stockpiles, amounting to 156.2 million tonnes, which will be processed at the end of the mine life.
- There will also be an embankment construction stockpile, south of the waste rock pile, which will be a temporary repository for waste used to build the embankment. Depending on the construction schedule, this could get a large as 67 million tonnes.

Figure 1.25-1 shows these various facilities at the end of mining. The waste rock pile is constructed in lifts, from the bottom up. Each 30 m lift will have a 20 m setback to make the overall angle 2H:1V, appropriate for final reclamation. Stockpiles are crest dumped at the angle of repose, since they will be rehandled.

The clean waste dump contains 88.8 million tonnes. This includes the small amounts of leach cap and supergene oxide not used for the embankment. Unclassified waste material, i.e. material that was not classified as leached cap, supergene, hypogene, etc. amounted to 447.5 million tonnes. For this study, IMC assumed that 1/3 of the unclassified waste is non-reactive and 2/3 is reactive waste. Of the assumed non-reactive portion, 73 million tonnes was used for embankment construction and 76.7 million tonnes was routed to the clean waste dump. The actual size of the clean waste dump is conjectural at this time. This study shows that the waste classification will need to be addressed more completely during the feasibility study to more correctly identify acceptable construction material and waste that will have to be disposed in the tailings embankment.

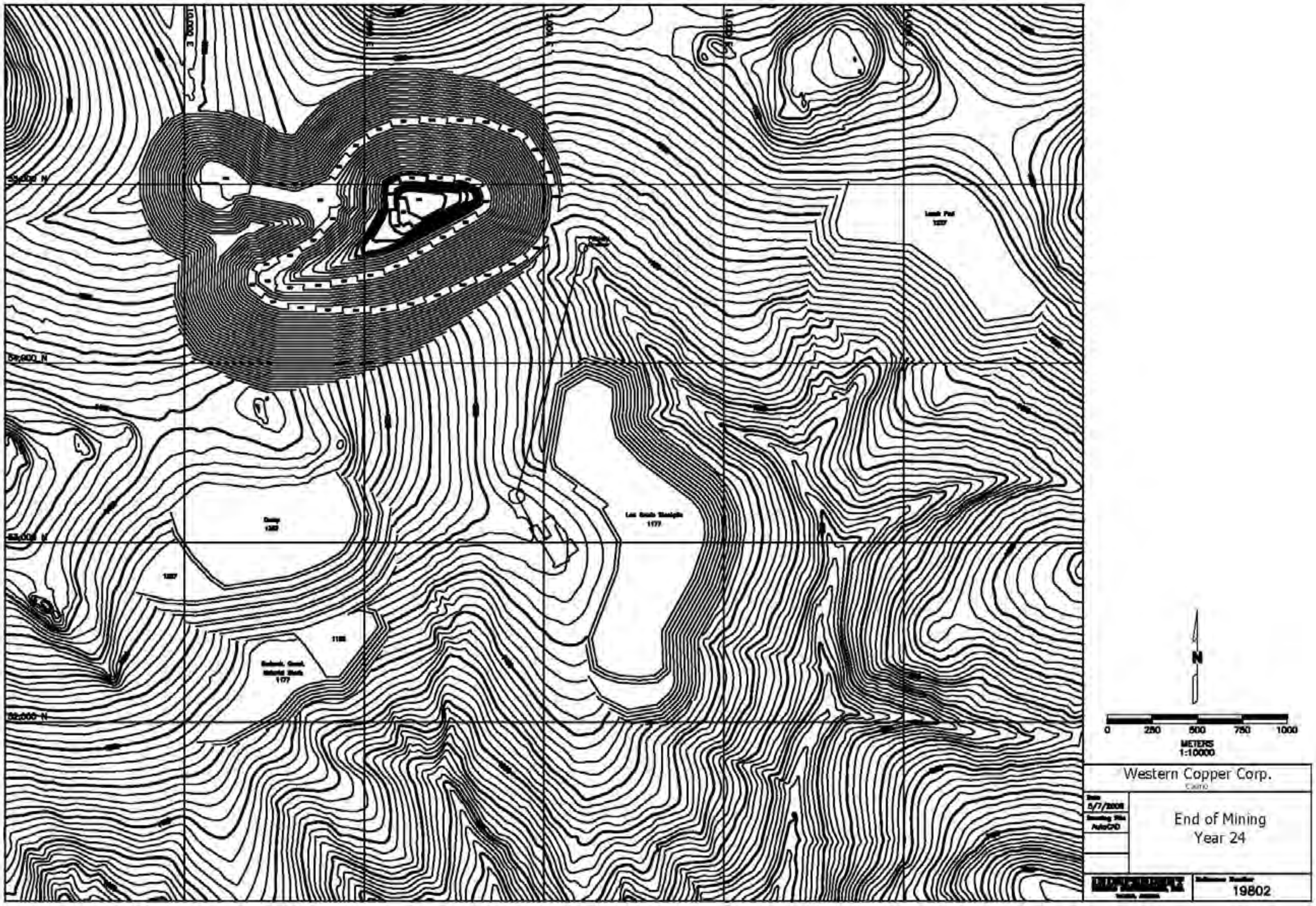


Figure 1.25-1: End of Mining, Year 24

1.25.2 Power and Transportation

1.25.2.1 Power

M3 retained Kerr Wood Leidal Associates Ltd. (KWL) of Vancouver, BC, to examine alternatives for electric power supply. KWL and their consultants, W. N. Brazier Associates, used project power demand data supplied by M3. M3's load estimate was for an average load of 96.1 MW, a running load of 101.8 and a peak load of 115.8 MW. KWL was also to recommend a method to supply power for early project development, including the heap leach/gold recovery plant, and project construction including the residence camp. These requirements were estimated to be 10 MW. The 10 MW power supply would be available for emergency standby once construction of the primary power supply is compete.

KWL's scope was to evaluate all power supply options considering: adequacy to meet demand, costs (capital, O&M, unit energy), practicality, lead time, risks, environmental issues, and regulatory issues.

The study evaluated all reasonable alternatives including coal, fuel oil, natural gas, wind, hydro, transmission by utilities, solar, biomass and geothermal.

KWL found coal and fuel oil to be the only viable primary power options.

The option used in this study is a coal-fired circulating fluidized bed (CFB) plant with 100% backup capacity provided by a fuel oil-fired turbine generator to allow continuation of operations during the estimated 10% downtime required for maintenance of CFB plants. At an estimated delivered coal price of $116 per tonne, the estimated power cost for the CFB option is $0.0945 per kWh, exclusive of capital amortization. This figure has been used in the estimation of operating costs for the mine. The costs of operating the standby unit for 10% of the time are included in the estimated power cost.

The estimated capital cost for the CFB plant and standby unit is $548 million inclusive of contingency.

At an estimated delivered fuel oil price of $0.913 per litre, the estimated power cost for the fuel oil option is $0.22 per kWh exclusive of capital amortization.

In spite of recent price volatility for both fuels, the cost advantage of coal remains compelling. KWL's cost estimate assumes prices will stabilize near historic levels in the coming years.

KWL recommended a modular diesel genset for early operations and emergency standby.

KWL notes that the federal government of Canada is considering a surcharge for greenhouse gas emissions that could add an estimated $0.017 per kWh. M3 did not include this surcharge in the cost analysis since it is speculative at this point.

1.25.2.2 Roads

M3 retained Associated Engineering, Ltd. (AE) of Vancouver to evaluate transportation options for the project. AE evaluated a range of methods and routes for bringing supplies to the project and hauling mine products to points of shipment for markets or further processing.

Seven alternative routes for an all-weather, year round access road were evaluated. The ports of Skagway and Haines, Alaska, USA, were considered for concentrated shipping and fuel and supply receiving. Figure 1.25-2 in the Illustrations section at the end of this report shows the seven alternatives including the selected route.

The route selected for the study begins at Haines, follows the Haines/Alaska Highway about 400 km to a point just north of Burwash Landing where it heads north on new ground for about 187 km to the mine site. The route is termed the "Onion Creek Route." This route requires the least amount of new construction and is aligned to minimize river crossings and construction in low-lying swampy areas. AE specified a road with a 7 m wide gravel surface.

The estimated cost of access road construction is about $91 million.

Annual maintenance costs for the road of $3.1 million are included in the cashflow model.

It should be noted that other routes still represent viable alternatives and potential cost sharing arrangements with other users may affect the final decision on the route actually developed.

1.25.2.3 Port

AE's subconsultant, Lauga and Associates Consulting Ltd., prepared a conceptual analysis of port development options including a cost estimate. The analysis concluded that Haines, Alaska, offers substantial advantages over Skagway, Alaska, in terms of available area for development, cost, and public acceptance. The port facility includes general cargo, coal receiving and concentrate loading to ships.

The estimated cost of port construction is about $36 million.

1.25.3 Water

Initial water requirements for the gold plant and heap leach operations will be met by pumping water retained behind a temporary dam located at or near an elevation of 825 m within the TMF catchment area along Casino Creek. The dam should remain a viable source for water supply for the heap leach operations while a fresh water pipeline is being constructed (see below) and for several years of sulphide ore processing until tailings deposition overtakes the dam.

Water from the dam will be lifted 285 meters through a 2.2 km, 8-inch diameter carbon steel pipe by two pumps (one operating, one standby) to the ADR Fresh/Fire Water Tank located above the gold plant in order to supply water via gravity flow to the plant. Each pump will have a design flow capacity of 139 m^3/h.

Water from the ADR Water Tank will also be pumped to the Mine Yard Fresh/Fire Water Tank to be located near the future Truck Shop. Incidental fresh water needs at the gold plant are expected to be provided by a new local well.

The main fresh water requirements will be supplied to the mine site from the Yukon River through a 17.3 km buried 28-inch diameter pipeline. Water will be withdrawn from the river by constructing an adjacent infiltration gallery. Five vertical turbine pumps at the gallery and at each of four booster stations will lift the water approximately 925 meters to the fresh/fire water pond located on a ridge above the mine site. The pipeline alignment will follow an existing access road along Britannia Creek.

Each pump will have a flow capacity of 831 m^3/h. With all five pumps operating, total maximum design flow capacity will be 4,155 m^3/h. Two pumps are expected to meet fresh water needs under normal operating conditions.

The fresh/fire water pond will provide pressurized fresh water by gravity flow to the plant and ancillary uses. The pond will also serve a separate underground fire water piping loop with hydrants. The fire loop will be insulated and heat-traced.

1.25.4 Permits

As of November 2005, the Yukon Environmental and Socio-economic Assessment Board (YESAB) must assess projects in Yukon for environmental and socio-economic effects under the *Yukon Environmental and Socio-economic Assessment Act* (YESAA). The Act includes two regulations: The *Yukon Activity and Project Regulation* and the *Timelines/Decision Bodies Coordination Regulation.*

Development of the Casino properties into a fully operational mine will trigger an environmental assessment under YESAA as all activities related to the construction, operation, modification or closure of a mine are listed as assessable activities. Furthermore, the level of assessment for the Casino project will be at the Executive Committee screening level as the key activities meet or exceed the applicable activity thresholds.

The YESAA screening process for projects submitted to the Executive Committee is estimated to take between 24 to 30 months to complete. For planning purposes, given the scale and complexity of the project, this process including document preparation and response to additional information requests, will likely take close to 30 months. For the Casino project this process will include consultation with the Selkirk First Nation community as a whole, and the Selkirk First Nation Chief and Council in particular in advance of submission of the proposal for a project. Additional consultation with other First Nations is likely also required, depending on the preferred access route chosen.

Western Copper will require a number of authorizations from various agencies to bring the Casino properties into production or perform further work on the property. The regulatory permitting and licensing processes are separate from the environmental and socio-economic assessment process (YESAA), and are generally initiated following the issuance of a positive YESAA Screening Report and Yukon Government decision document.

Authorizations are processed and issued following completion the YESAA assessment. However, some agencies will conduct preliminary reviews of applications or begin drafting authorizations prior to the issuance of a positive YESAA decision document.

Table 1.25.4-1 lists the major and minor permits required.

Table 1.25.4-1: Major Permits

Agency Responsible/web link	License/ Authorisation	Description	Term/Timing	Comments
ASSESSMENT STAGE				
Yukon Socioeconomic Assessment Board http://www.yesab.ca	Not a license, recommendations only.	Administers the Yukon Socioeconomic Assessment Act. (YESAA)	2.5 years.	Conducts a review and provides recommendations to government agencies (Decision Bodies)
Decision Bodies: the federal, territorial, or First Nation having authority to determine whether a project may proceed.	Decision Documents (Not a License) Licenses cannot contravene Decision Documents.	Decision Bodies respond to recommendations by the Executive Committee of YESAB. A Decision Body may accept, vary or reject a recommendation, and must state its decision in Decision Documents.	Defined timelines depend on whether the Decision Body accepts, varies or rejects YESAA recommendations.	Agencies cannot issue licenses until a Decision Document has been issued.
MAJOR LICENSES				
YG Energy, Mines and Resources/ Minerals Management Branch	Quartz Mining License (Production License)	Establishes conditions that authorize specific mining activities, reclamation and closure, security, requirements for design approval, reporting requirement, and advance notification requirements for certain activities.	Discretionary	Broad discretion to establish license terms and conditions and require security.
Yukon Water Board	Type A Water Use License	License conditions include: Quantity of water for camp use and mine processing, sets discharge water quality standards and establishes Provisions for the collection of security	Expect licensing process to take up to one year. Can issue licenses for up to 25 years.	The Water board will not initiate their process until Decision Bodies have issued Decision Documents.
Department of Fisheries and Oceans/Environment Canada	Metal Mining Effluent Regulation Schedule 2 Amendment	Addition to listing of Tailings Impoundment Areas.	Up to 2 year process after environmental assessment.	Requires a federal Order-In-Council.

1.25.5 Operating Costs

The operating and maintenance costs for the Casino operations are summarized by areas of the plant, and shown in Table 1.25.5-1. Cost centers include mine operations, process plant operations, and the General and Administration area.

Process operating costs were determined for a typical year of operations, based on an annual mill ore tonnage of 33.4 million tonnes that will produce approximately 371,000 tonnes of copper concentrates and 16,000 tonnes of molybdenum concentrates. The ADR/SART plant costs are based on gold ore production of 9.7 million tonnes that production of approximately 53,000 ounces of gold and 1,000 tonnes of copper precipitates. These costs were adjusted on a year by year basis and incorporated into the financial model.

Life of mine average operating cost is $9.72 per tonne for sulphide ore, which includes mining, concentrator plant and general and administrative costs. The life of mine average operating cost is $3.19 per tonne for oxide ore which includes processing only.

1.25.5.1 Mine Operating Costs

Mining operating costs were developed on an annual basis for several cost centres, namely, drilling, blasting, loading, hauling, roads and dumps, general mine, general maintenance and mining administration. Cost centres were estimated on the basis of the cost of operating equipment, providing labour and supplying maintenance parts and operating consumables as required to mine the tonnage specified by the mining schedule. The cost of operating mining equipment was derived on an hourly basis prorated to the number of operating minutes per shift and then to the number of shifts required per year. The cost of labour was determined on the basis of an hourly burdened rate for each job classification and the man-hours required for performing operating and maintenance functions. Parts and consumables costs were estimated on budget estimates and average consumptions for fuel, lubes, tires, ground engaging tools, and repair parts.

Table 1.25.5-1 shows the total mining cost, the average unit cost per total tonne, and the average unit cost per ore tonne for Years 1 – 23 (Open Pit Operations), Years 24 – 29 (Stockpile Re-handling Operations) and for Years 1 – 29, the overall LOM. The total mining cost of production over the LOM amounts to $2.96 billion and the average unit costs for mining are $1.393 per total tonne and $3.235 per ore tonne.

Table 1.25.5-1: Summary of Total and Unit Mining Operating Costs

Category	Total Cost (C$x1000)	Cost Per Total Tonne	Cost Per Ore Tonne
Commercial Production Years 1 to 23	2,768,373	1.446	3.723
Commercial Production Years 24 to 29	187,089	1.059	1.101
Commercial Production Years 1 to 29	2,955,462	1.393	3.235

The costs are in 1st quarter 2008 Canadian dollars. This estimate is based on assumed prices for commodities such as fuel, explosives, parts, etc. that are subject to wide variations depending on market conditions.

Mining Manpower

Table 1.25.5-2 shows the mining manpower requirements and the labour cost component of the mining cost for both supervisory (salaried) staff and hourly personnel. The salaried staff requirements were estimated by IMC based on open pit mining operations of similar scale, and the hourly personnel requirements were estimated based on annual equipment operating and maintenance hours required and the number of hours worked per man-year.

The annual rates include a burden of 30% and are considered all-in rates that include vacation, overtime, and travel time (to and from the property) allowances. Mining labour costs do not include personnel camp costs and transportation costs to and from the property.

Average life of mine, labour costs amount to about $0.39 per total tonne or about 28% of the total mine operating cost.

Table 1.25.5-2: Mining Manpower Complement and Labour Costs

SALARIED STAFF	Min	Max	Avg	Total C$ x 1000	C$ per total tonne
Mine Administration	2	2	2	5,788	0.003
Mine Supervision	3	9	9	25,458	0.012
Mine Maintenance	5	16	15	42,264	0.020
Mine Engineering	9	19	17	45,055	0.021
Salaried Staff Total	29	46	43	118,565	0.055

HOURLY PERSONNEL	Min*	Max	Avg	Total C$ x 1000	C$ per total tonne
Mine Operations	0	159	116	384,004	0.179
Mine Maintenance	0	126	92	313,621	0.146
Hourly Total	0	285	208	697,625	0.324
Mine Total	29	331	250	816,190	0.379

* No hourly personnel required during contract mining operations in Year -3.

1.25.5.2 Process Plant Operating & Maintenance Costs

Process Labour & Fringes

Process labour costs were derived from a staffing plan and based on prevailing daily or annual labour rates referenced from an industry survey for Canadian wages and benefits. Labour rates and fringe benefits for employees include all applicable social security benefits as well as all applicable payroll taxes. The staffing plan summary and gross annual labour costs are shown in Table 1.25.5-3.

Table 1.25.5-3: Operating Cost – Labour Cost Summary (Typical Year of Operation)

Department	Number Of Personnel	Total Labor ($)/year
General & Administrative		
Administration	5	551,088
Controller	5	572,720
Health, Safety & Environment	20	2,240,855
Purchasing	10	1,079,868
Concentrator	190	23,272,003
Heap Leach and ADR Plant	42	5,212,747
Total Labor Cost	272	32,929,280
-Includes holidays, vacation allowance camp and flight charges		

Power

Power costs were based on obtaining power from a steam/diesel power plant at a rate of $C0.0945 per kWh. Power consumption was based on the equipment list connected kW, discounted for operating time per day and anticipated operating load level.

Reagents

Reagents for the process plants are listed in Section 1.3.15. Consumption rates were determined from the metallurgical test data or industry practice. Budget quotations were received for reagents supplied to Skagway, AK, Haines, AK, or from local sources where available with allowance for freight to site.

Maintenance Wear Parts and Consumables

Grinding media and part consumption are based on industry practice for the crusher and grinding operations. An allowance was made to cover the cost of maintenance of all items not specifically identified and the cost of maintenance of the facilities. The allowance made was 5.0% of the direct capital cost of equipment, which totalled approximately $11.7 million for the concentrator and $2.0 million for the ADR & SART plants.

Process Supplies & Services

Allowances were provided in concentrator, ADR and SART process plants for outside consultants, outside contractors, vehicle maintenance, and miscellaneous supplies. The allowances were estimated using historical information from other operations and projects.

1.25.5.3 General Administration

General and administration costs include labour and fringe benefits for the administrative personnel, human resources, and accounting. Also included are office supplies, communications, insurance, employee transportation and camp, and other expenses in the administrative area.

Labour costs for G&A are based on a staff of 40. Labour rates are based on a daily rate and include benefits.

All other G&A costs were developed as allowances based on historical information from other operations and other projects.

Laboratory costs estimates are based on labour and fringe benefits, power, reagents, assay consumables, and supplies and services.

All other laboratory costs were developed as allowances based on historical information from other operations and other projects.

Note that laboratory services are likely to be contracted out. This estimate retains the costs under the assumption that contract and in-house laboratory services costs are equal.

Table 1.25.5-4: Life of Mine Operating Costs by Area – Mining and Concentrator

Concentrator Processing Units Base Rate (LOM ktonnes ore)	**913,546**		
Total Ore and Waste Mined (LOM ktonnes)	**1,986,635**		
Cost Area	**Annual Cost - $000**	**$/tonne milled**	**$/tonne mined**
Mining Operations			
Drilling	172,591	0.19	0.09
Blasting	323,487	0.35	0.16
Loading	230,219	0..25	0.11
Hauling	1,486,277	1.63	0.75
Road & Dumps	389,256	0.43	0.20
Mining General	353,632	0.39	0.18
Subtotal Mining	$ 2,955,462	$ 3.24	$ 1.49
Concentrator Operations			
Primary Crushing & Stockpile Feed	198,606	0.22	
Grinding, Classification & Pebble Crushing	3,451,037	3.77	
Flotation & Regrind	1,276,857	1.40	
Concentrate Thickening/Filtration	107,760	0.12	
Tailings Dewatering & Disposal	259,918	0.28	
Fresh Water/Plant Water	134,318	0.15	
Flotation Reagents	24,845	0.03	
Ancillary Services	36,810	0.04	
Subtotal Concentrator	$ 5,490,151	$6.01	
Supporting Facilities			
Laboratory	39,938	0.04	
General and Administrative	393,274	0..43	
Subtotal Supporting Facilities	$ 433,212	$0.47	
Total Operating Costs	**$ 8,878,825**	**$ 9.72**	

Table 1.25.5-5: Life of Mine Operating Costs by Area – Heap Leach

ADR/SART Processing Units Base Rate (LOM ktonnes)	**77,917**	
Cost Area	**Annual Cost - $000**	**$/tonne leached**
ADR & SART		
Heap Leach	22,976	0.29
ADR/SART	225,890	2.90
Total Operating Cost ADR & SART	**$248,866**	**$3.19**

1.25.6 Capital Costs

Table 1.25.6 presents a detailed capital cost summary.

To assist in the estimating, M3 used quantity estimates, and in some cases costs, supplied by specialist subconsultants:

- Associated Engineering (AE): Main access road and port facility
- Kerr Wood Leidal (KWL): Project power supply capital and operating costs,
- Knight Piésold (KP): Geotechnical quantities associated with the Heap Leach Facility, Waste Rock Storage Area, Water Supply and the Tailing Management Facility, and
- Independent Mining Consultants (IMC): mine capital

"Initial Capital" is defined as all capital costs through to the end of construction or the end of Year 1 of the mine life. Capital costs predicted for later years are carried as sustaining capital in the financial model.

All costs are in 1st quarter 2008 Canadian dollars with the exception of steel prices. Steel prices rose substantially during the second quarter such that using Q1 steel costs was considered unrealistic. The study used Q2 2008 steel prices. The values in this report assume an exchange rate of one to one between U.S. and Canadian dollars.

The accuracy of this estimate for those items identified in the scope-of-work is estimated to be well within the range of ±25%.

Outside consultants provided estimates for the coal-fired power plant, access road, and port facility. These consultants reported that their estimates are compatible with the overall estimate accuracy of ±25%.

Based on the level of engineering completed and definition of scope, M3 estimated the contingency at 13% of the direct and indirect costs (Contracted Cost). The contingency was estimated on an area and cost category basis.

1.25.6.1 Direct Costs

Sitework quantities were estimated by Knight Piésold for the tailings pond, heap leach and mine waste rock piles. M3 applied unit cost factors based on experience and judgement.

Other sitework quantities were estimated using Autodesk's Land Development program for AutoCAD applied to preliminary facility layouts prepared by M3 and using foundation depths suggested by Knight Piésold.

Structural steel and concrete quantities for the process plant buildings were estimated using parametric factors collated from constructed projects and current construction designs for projects of a similar size and nature. Other areas were estimated from direct material takeoffs from drawings of conceptual designs.

Concrete costs were estimated based upon an informal survey of current and recent projects in the Yukon.

Steel costs were based upon a recent large steel purchase for a mine plant of similar scale. Competitive bids were collected from the US, Mexico and Chile. M3 considers the resulting bid prices to be representative of world structural steel prices during the second quarter of 2008 as these costs reflect a substantial increase on first quarter pricing.

Major items of chutework, skirting, liners etc. were estimated from parametric factors collated from similar, constructed projects and current designs. The cost of the steel was based on quotes for recently constructed projects.

Architectural costs are based on M3 records of similar sized projects for the major buildings. Yukon unit rates are applied to the quantities previously calculated.

Vendor quotes supplied cost data for major mine equipment. Small support equipment cost was estimated by an allowance of 5%.

M3 prepared a comprehensive Equipment List based on the Flowsheets developed for the project. Major process plant equipment such as the primary crusher, grinding mills, pebble mills, large capacity conveyor belts, cyclone clusters, large-capacity pumps, and major electrical components were priced from vendor budgetary quotations. Other equipment prices were based on M3's historical records including budgetary and equipment purchase pricing from recent, similar projects. Some historic records were scaled to correct for size or capacity difference. Installation costs are based on allowances for materials and M3's judgment and experience for labour. About 86%, of total mine and plant mechanical equipment cost came directly from 2008 vendor budgetary quotes for this or other projects.

Electrical equipment prices are historical prices from two projects of similar scale collected over the past two years.

Piping in most areas is estimated as a percentage of the mechanical equipment cost. But the fresh water pipeline from the Yukon River, the reclaim water system from the tailings pond, the fresh water system for the heap leach, and the piping to and from the heap leach was estimated based on an M3 conceptual design.

Instrumentation in most areas is estimated as a percentage of the mechanical equipment cost.

1.25.6.2 Indirect Costs

A freight allowance of 10% is included on equipment and bulk material cost

The following taxes have been considered for this estimate:

- Sales tax is not included in the cost of equipment.
- GST is not included as it is a pass through cost.
- Payroll taxes are included in the labour rates

Working capital is accounted for in the financial model. See Section 1.25.7.

Sustaining Capital is not reflected in this Capital Cost Estimate but has been estimated on the same basis as the initial capital. The major components of sustaining capital are expansion of Heap Leach Facility pad, additions to the mine equipment fleet in later years, and Tailings Management Facility expansions including return water pump station relocation and installation of a second pump station in later years. Estimated costs are applied in future years in the economic model. See Section 1.25.7.

A construction camp with executive, supervisor, and labourer quarters to hold the anticipated maximum construction manpower of 1,200 people is included in the estimate. A camp operating cost of $85 per day per person is also included.

All other indirect costs are specifically listed in Table 1.25.6-1.

Labour costs were estimated by applying hourly quantities to certain tasks or materials such as per tonne of concrete, per cubic meter of excavation, etc. M3 cost estimators use vendor information, published cost estimating guides and experience and judgment to determine the hourly quantities. Construction trade labour rates average $70.00 per hour.

Table 1.25.6-1: Capital Cost Estimate Summary

	(millions)
Mine (including pre-stripping)	$340
Mill & Flotation	$541
Tailings	$96
Heap Leach	$51
Direct Cost	**$1,028**
Engineering & Management	$137
Camp	$65
Contingency	$165
Owner's Costs	$41
Total Capital Cost	**$1,437**
Power Plant (includes Heap power)	$548
Access Road	$91
Port	$36
Total	**$2,112**

1.25.7 Financial Analysis

The financial evaluation presents the determination of the Net Present Value (NPV), payback period (time in years to recapture the initial capital investment), and the Internal Rate of Return (IRR) for the project. Annual cash flow projections were estimated over the life of the mine based on the estimates of capital expenditures and production cost and sales revenue. The sales revenue is based on the production of copper concentrate with gold credits, molybdenum concentrate and gold doré.

1.25.7.1 Mine Production Statistics

Mine production is reported as mill ore, low grade mill ore, gold ore and waste material from the mining operation. The annual production figures were obtained from the mine plan as reported earlier in this report. The financial model reflects the stockpiling of low grade ore and its subsequent processing at the end of mine life.

The life of mine ore and waste quantities and ore grade are presented in the table below.

Table 1.25.7-1: Life of Mine Ore, Waste Quantities, and Ore Grade

	Tonnes (000')	**Copper %**	**Moly %**	**Gold g/t**
Mill Ore	757,375	0.223%	0.0266%	0.249
Low Grade Mill Ore	156,171	0.163%	0.0095%	0.177
Gold Ore	77,917	0.062%		0.427
Waste	995,172			
Total	1,986,635			

1.25.7.2 Plant Production Statistics

In the current plan, all the Mill Ore and Gold Ore is processed directly, with the Low Grade Ore being stockpiled and processed at the end of mine life. The Gold Ore will begin to be processed in year -3, that is, three years before mill ore processing begins.

The gold ore processing by heap leaching will produce to two products, a gold doré and a copper precipitate. The estimated production over the life of the heap leach is 536,000 ounces of gold and 21.4 million pounds of copper.

Production from the flotation plant will produce a copper-gold concentrate and a molybdenum concentrate. The estimated copper concentrate production for the life of the flotation plant is 5.8 million tonnes containing 3.6 billion pounds of copper and 4.6 million ounces of gold. The estimated molybdenum concentrate production for the life of the flotation plant is 256,000 tonnes containing 316.1 million pounds of molybdenum.

1.25.7.3 Capital Expenditure

The base case financial indicators have been determined on the basis of 100% equity financing of the initial capital. Any acquisition cost or expenditures prior to the full project production decision have been treated as "sunk" cost and have not been included in the analysis.

The total capital carried in the financial model for new construction is $2.1 billion in Canadian dollars and is expended over a five year construction period. The cash flow for the new construction is shown being expended in the years before production and ending in the first year of production. The initial capital includes Owner's costs and contingency and the capital for the power plant.

A schedule of capital cost expenditures during the production period was estimated and included in the financial analysis under the category of sustaining capital. The total life of mine sustaining capital is estimated to be $862 million. This capital will be expended during a 30 year period, starting in Year -1 with heap leach sustaining capital and ending in Year 27.

Accounts receivable for sale of the metals vary by year depending on sales revenue. Operating working capital is allowed at two months of sales revenue to provide cash to meet operating expenses prior to receipt of sales revenue. In addition, working capital for accounts payable is being allowed for 30 days, also an allowance for plant consumable inventory is estimated in year -3 for the ADR/SART plant and -1 for the process plant. All the working capital is recaptured at the end of the mine life and the final value of the account is $0.

A $25 million allowance for salvage value has been included in the cash flow analysis.

1.25.7.4 Revenue

Annual revenue is determined by applying selected metal prices to the annual payable metal contained in the concentrates and doré estimated for each operating year. Sales prices have been applied to all life of mine production without escalation or hedging. The base case financial evaluation uses LME three-year historical rolling average prices as of the end of May 2008. This approach is considered to be an industry standard and consistent with the guidance of the United States Securities and Exchange Commission. Prices used are:

- Copper $2.95/lb.
- Molybdenum $30.97/lb.
- Gold $647.40/oz.

The revenue is the gross value of payable metals sold before transportation and smelting charges.

1.25.7.5 Total Cash Operating Cost

The sulphide milling operation bears all mining costs since the premise of the heap leach is that were it not for the heap leach, the material would be waste. The sulphide average operating costs are $9.72 per tonne of mill ore processed. Heap leach/SART/ADR costs are $3.19 per tonne of leach ore.

Specifics of operating costs are discussed in Section 1.25.5 above.

Shipping, smelting and refining charges are discussed in Section 1.25.8 below.

1.25.7.6 Total Cash Production Cost

Total Cash Production Cost is the Total Cash Operating Cost plus royalties, reclamation & closure, property tax and salvage income. The sulphide average cash production cost including these items totals $10.18 per tonne of mill ore processed. The heap leach/SART/ADR costs remain the same at $3.19 per tonne of leach ore as the sulphide process bears these additional costs alone.

1.25.7.7 Royalties and Taxes

Archer Cathro and Associates will receive a royalty of 5% net profits totalling $343.6 million over the life of the mine.

Great Basin Gold has a one time payment of C$1.0 million, due once approval to proceed with the project is obtained.

It is estimated that $962 million will be paid in Yukon mining royalties. Yukon mining royalties are based on an assumption that the Yukon legislature will follow through with plans to cap the mining royalty at 12% of revenues less operating expenses, depreciation (no more than 15% of the initial capital cost balance at the beginning of the year) and taxes paid.

Corporate income taxes paid is estimated to be $2.0 billion for the life of the mine based on a 30% combined federal and territorial corporate income tax rate of taxable income. A deduction of depreciation for class 41A assets is being taken which results in no income tax being paid until initial capital is fully depreciated. These deductions against income are applied each year, but cannot create a loss.

An allowance of $100,000 per year was included in the cash flow to account for property tax.

1.25.7.8 Reclamation and Closure

$100 million spread over the final four years of production was allowed for post-closure reclamation & final closeout.

1.25.7.9 Project Financing

It is assumed the project will be all equity financed.

1.25.7.10 Net Income after Tax

Net Income after Tax amounts to $4.5 billion for the life of the mine.

1.25.7.11 NPV and IRR

The base case economic analysis (Table 1.25.7-2) indicates that the project has an Internal Rate of Return (IRR) of 14.9% after taxes with a payback period of 3.8 years.

Table 1.25.7-2 compares the base case project financial indicators with the financial indicators for other cases when the sales price, the amount of capital expenditure, operating cost, and copper recovery are varied from the base case values. By comparing the results of this sensitivity study, it can be seen that the project IRR's sensitivity to variation in sales price has the most impact, while variation of operating cost, variation of mill recovery, and variation of capital cost are approximately equal.

Table 1.25.7-2: Sensitivity Analysis

	NPV @ 0%	NPV @ 5%	NPV @ 8%	NPV @ 10%	IRR	Payback Years
Base Case	$4,481	$1,661	$867	$518	14.9%	3.8
Spot Price*	$7,150	$2,978	$1,802	$1,281	21.4%	2.8
Metals Price +10%	$5,816	$2,317	$1,330	$894	18.1%	3.2
Metals Price -10%	$3,110	$975	$377	$116	11.2%	4.9
Capex +10%	$4,276	$1,484	$703	$363	13.2%	4.3
Capex -10%	$4,686	$1,839	$1,030	$673	17.0%	3.4
Opex +10%	$3,910	$1,391	$679	$366	13.5%	4.1
Opex -10%	$5,052	$1.931	$1,054	$669	16.2%	3.6
Mill Recovery +5%	$5,903	$1,961	$1,078	$689	16.4%	3.5
Mill Recovery -5%	$3,869	$1,360	$654	$345	13.3%	4.2
$ in millions						

* Spot Prices are from May 31, 2008.
Copper $3.67/lb.
Molybdenum $33.70/lb.
Gold $885.75/oz.

Ore produced during the first six years is substantially higher in copper, gold and molybdenum than the life-of-mine average. The results are more robust cash flow during those years allowing payback in only 3.8 years under the base case. Table 1.25.7-3 illustrates the difference during these early years.

Tables 1.25.7-5 through 1.25.7-10 are detailed NPV matrices showing a broad range of pre-tax and post-tax NPV sensitivities for gold, copper, and molybdenum prices.

Table 1.25.7-3: Project Cash Flow

	Years 1-6	Life of Mine
Average Annual Pre-tax Cashflow	$571 million	$219 million
Average Annual After-tax Cashflow	$448 million	$132 million
Average NSR (sulphide ore)	$29.66/tonne	$21.54/tonne
Average Annual Mill Feed Grade		
Copper (%)	0.325	0.212
Gold (g/t)	0.380	0.237
Molybdenum (%)	0.028	0.024
Annual Concentrate Production		
Copper (dry ktonnes)	313	201
Molybdenum (dry ktonnes)	11	9
Average Annual Metal Production		
Copper (M lb)	193	124
Gold (k oz)	263	158
Molybdenum (k lb)	13,415	10,899

Table 1.25.7-4: Tax Price Matrix Legend

NPV 0% (CAN$ millions)
NPV 8% (CAN$ millions)
IRR (%)
Payback (years)

Table 1.25.7-5: Pre-Tax Price Matrix, Molybdenum at $15/lb

		Copper Price (US$/lb)						
		$2.00	$2.50	$3.00	$3.50	$4.00	$4.50	$5.00
Gold Price (US$/oz)	$600	($38)	$1,634	$3,302	$4,970	$6,639	$8,307	$9,976
		($650)	($86)	$472	$1,029	$1,585	$2,142	$2,697
		na	7.2%	12.0%	16.0%	19.6%	22.8%	25.8%
		25.5	6.9	4.6	3.7	3.2	2.9	2.6
	$700	$443	$2,112	$3,780	$5,448	$7,117	$8,785	$10,454
		($464)	$96	$654	$1,210	$1,767	$2,323	$2,879
		2.6%	8.9%	13.5%	17.4%	20.9%	24.1%	27.0%
		10.1	5.7	4.2	3.5	3.0	2.7	2.5
	$800	$922	$2,589	$4,258	$5,926	$7,595	$9,263	$10,932
		($281)	$278	$836	$1,392	$1,948	$2,504	$3,060
		4.9%	10.6%	15.0%	18.8%	22.2%	25.3%	28.3%
		7.9	4.9	3.8	3.3	2.9	2.6	2.4
	$900	$1,399	$3,067	$4,736	$6,404	$8,073	$9,741	$11,409
		($98)	$460	$1,017	$1,573	$2,129	$2,685	$3,241
		7.0%	12.2%	16.5%	20.2%	23.5%	26.6%	29.5%
		6.6	4.4	3.6	3.1	2.8	2.5	2.3
	$1,000	$1,877	$3,545	$5,214	$6,882	$8,551	$10,219	$11,887
		$85	$642	$1,199	$1,755	$2,311	$2,866	$3,422
		8.9%	13.8%	17.9%	21.5%	24.8%	27.9%	30.7%
		5.4	3.9	3.3	2.9	2.6	2.4	2.2
	$1,100	$2,355	$4,023	$5,692	$7,360	$9,029	$10,697	$12,365
		$267	$824	$1,380	$1,936	$2,492	$3,048	$3,603
		10.7%	15.4%	19.4%	22.9%	26.1%	29.1%	32.0%
		4.7	3.7	3.1	2.8	2.5	2.3	2.1

Table 1.25.7-6: Pre-Tax Price Matrix, Molybdenum at $25/lb

Gold Price (US$/oz)		Copper Price (US$/lb)						
		$2.00	$2.50	$3.00	$3.50	$4.00	$4.50	$5.00
	$600	$2,533	$4,201	$5,870	$7,538	$9,207	$10,875	$12,543
		$186	$744	$1,300	$1,857	$2,413	$2,969	$3,524
		9.6%	13.8%	17.4%	20.8%	23.8%	26.7%	29.4%
		6.1	4.3	3.6	3.1	2.8	2.6	2.4
	$700	$3,011	$4,679	$6,348	$8,016	$9,685	$11,353	$13,020
		$368	$925	$1,482	$2,038	$2,594	$3,150	$3,705
		11.1%	15.2%	18.8%	22.0%	25.1%	27.9%	30.6%
		5.2	3.9	3.4	3.0	2.7	2.5	2.3
	$800	$3,489	$5,157	$6,826	$8,494	$10,163	$11,831	$13,498
		$550	$1,107	$1,663	$2,219	$2,775	$3,331	$3,886
		12.6%	16.5%	20.1%	23.3%	26.3%	29.1%	31.8%
		4.6	3.7	3.2	2.8	2.6	2.4	2.2
	$900	$3,967	$5,635	$7,304	$8,972	$10,641	$12,309	$13,976
		$732	$1,288	$1,845	$2,401	$2,957	$3,512	$4,067
		14.1%	17.9%	21.4%	24.6%	27.5%	30.3%	32.9%
		4.1	3.4	3.0	2.7	2.5	2.3	2.1
	$1,000	$4,445	$6,113	$7,782	$9,450	$11,118	$12,786	$14,453
		$914	$1,470	$2,026	$2,582	$3,138	$3,693	$4,248
		15.6%	19.3%	22.7%	25.8%	28.8%	31.5%	34.1%
		3.8	3.2	2.8	2.6	2.4	2.2	2.0
	$1,100	$4,923	$6,591	$8,260	$9,928	$11,596	$13,264	$14,930
		$1,095	$1,652	$2,207	$2,763	$3,319	$3,874	$4,429
		17.0%	20.7%	24.0%	27.1%	30.0%	32.7%	35.3%
		3.5	3.0	2.7	2.5	2.3	2.1	2.0

Table 1.25.7-7: Pre-Tax Price Matrix, Molybdenum at $35/lb

		Copper Price (US$/lb)						
		$2.00	$2.50	$3.00	$3.50	$4.00	$4.50	$5.00
Gold Price (US$/oz)	$600	$5,101	$6,769	$8,437	$10,106	$11,774	$13,442	$15,109
		$1,015	$1,572	$2,128	$2,684	$3,240	$3,795	$4,350
		15.3%	18.7%	21.9%	24.8%	27.6%	30.2%	32.7%
		4.1	3.5	3.0	2.8	2.5	2.4	2.2
	$700	$5,579	$7,247	$8,915	$10,584	$12,252	$13,919	$15,586
		$1,197	$1,753	$2,310	$2,865	$3,421	$3,976	$4,531
		16.6%	20.0%	23.1%	26.0%	28.7%	31.4%	33.8%
		3.8	3.3	2.9	2.6	2.4	2.3	2.1
	$800	$6,057	$7,725	$9,393	$11,062	$12,730	$14,397	$16,063
		$1,378	$1,935	$2,491	$3,047	$3,602	$4,157	$4,712
		17.9%	21.2%	24.3%	27.2%	29.9%	32.5%	35.0%
		3.6	3.1	2.8	2.5	2.3	2.2	2.0
	$900	$6,535	$8,203	$9,871	$11,540	$13,208	$14,875	$16,541
		$1,560	$2,116	$2,672	$3,228	$3,783	$4,338	$4,893
		19.3%	22.5%	25.5%	28.4%	31.1%	33.7%	36.1%
		3.3	2.9	2.7	2.4	2.3	2.1	2.0
	$1,000	$7,013	$8,681	$10,349	$12,018	$13,685	$15,352	$17,018
		$1,741	$2,297	$2,853	$3,409	$3,964	$4,519	$5,073
		20.6%	23.8%	26.8%	29.6%	32.3%	34.8%	37.2%
		3.1	2.8	2.5	2.3	2.2	2.0	1.9
	$1,100	$7,491	$9,159	$10,827	$12,496	$14,163	$15,829	$17,496
		$1,923	$2,479	$3,035	$3,590	$4,145	$4,700	$5,254
		21.9%	25.1%	28.0%	30.8%	33.5%	36.0%	38.4%
		2.9	2.7	2.4	2.2	2.1	1.9	1.8

Table 1.25.7-8: Post-Tax Price Matrix, Molybdenum at $15/lb

		Copper Price (US$/lb)						
		$2.00	$2.50	$3.00	$3.50	$4.00	$4.50	$5.00
Gold Price (US$/oz)	$600	($492)	$636	$1,793	$2,877	$3,961	$5,045	$6,080
		($884)	($467)	($53)	$330	$711	$1,089	$1,435
		na	3.1%	7.5%	10.9%	14.0%	16.8%	19.2%
		30.0	10.2	6.5	4.7	3.9	3.3	3.0
	$700	($282)	$1,006	$2,131	$3,215	$4,299	$5,347	$6,377
		($745)	($317)	$90	$471	$852	$1,207	$1,550
		na	4.8%	8.8%	12.2%	15.2%	17.8%	20.1%
		30.0	8.5	5.5	4.3	3.6	3.2	2.9
	$800	$124	$1,384	$2,469	$3,553	$4,614	$5,645	$6,676
		($591)	($153)	$232	$613	$978	$1,321	$1,666
		0.8%	6.4%	10.2%	13.5%	16.3%	18.7%	21.0%
		22.5	7.0	4.8	3.9	3.4	3.0	2.8
	$900	$539	$1,722	$2,806	$3,882	$4,912	$5,942	$6,976
		($426)	($9)	$373	$749	$1,093	$1,436	$1,782
		3.1%	7.9%	11.5%	14.7%	17.3%	19.6%	21.9%
		9.4	5.8	4.3	3.6	3.2	2.9	2.7
	$1,000	$926	$2,060	$3,144	$4,179	$5,210	$6,240	$7,276
		($269)	$133	$514	$864	$1,207	$1,550	$1,898
		5.0%	9.4%	12.9%	15.7%	18.2%	20.6%	22.8%
		7.7	4.9	3.9	3.6	3.0	2.8	2.6
	$1,100	$1,308	$2,398	$3,447	$4,477	$5,507	$6,537	$7,577
		($111)	$275	$633	$978	$1,321	$1,665	$2,014
		6.8%	10.8%	14.0%	16.7%	19.2%	21.5%	23.8%
		6.1	4.4	3.7	3.2	2.9	2.7	2.5

Table 1.25.7-9: Post-Tax Price Matrix, Molybdenum at $25/lb

		Copper Price (US$/lb)						
		$2.00	$2.50	$3.00	$3.50	$4.00	$4.50	$5.00
Gold Price (US$/oz)	$600	$1,239	$2,359	$3,444	$4,528	$5,602	$6,633	$7,671
		($281)	$123	$505	$886	$1,258	$1,601	$1,950
		5.4%	9.1%	12.2%	15.1%	17.7%	19.9%	22.1%
		8.7	5.9	4.5	3.7	3.3	3.0	2.8
	$700	$1,578	$2,698	$3,782	$4,866	$5,900	$6,930	$7,974
		($139)	$266	$647	$1,027	$1,373	$1,716	$2,068
		6.7%	10.3%	13.4%	16.3%	18.6%	20.8%	23.0%
		7.5	5.1	4.0	3.5	3.1	2.9	2.7
	$800	$1,951	$3,036	$4,119	$5,167	$6,198	$7,230	$8,276
		$24	$407	$789	$1,144	$1,487	$1,832	$2,185
		8.2%	11.6%	14.7%	17.2%	19.5%	21.7%	23.9%
		6.3	4.6	3.7	3.5	3.0	2.8	2.6
	$900	$2,289	$3,373	$4,435	$5,465	$6,495	$7,530	$8,578
		$168	$549	$915	$1,258	$1,602	$1,948	$2,303
		9.6%	12.9%	15.7%	18.2%	20.4%	22.6%	24.8%
		5.3	4.1	3.5	3.1	2.9	2.7	2.4
	$1,000	$2,627	$3,702	$4,732	$5,762	$6,793	$7,833	$8,881
		$309	$685	$1,030	$1,373	$1,716	$2,066	$2,420
		10.9%	14.1%	16.7%	19.1%	21.4%	23.6%	25.7%
		4.6	3.8	3.3	3.0	2.8	2.6	2.4
	$1,100	$2,965	$4,000	$5,030	$6,060	$7,090	$8,135	$9,185
		$450	$800	$1,144	$1,487	$1,831	$2,183	$2,537
		12.2%	15.1%	17.7%	20.0%	22.3%	24.5%	26.6%
		4.1	3.5	3.1	2.9	2.7	2.5	2.3

Table 1.25.7-10: Post-Tax Price Matrix, Molybdenum at $35/lb

		Copper Price (US$/lb)						
		$2.00	$2.50	$3.00	$3.50	$4.00	$4.50	$5.00
Gold Price (US$/oz)	$600	$2,926	$4,009	$5,095	$6,155	$7,185	$8,229	$9,279
		$299	$680	$1,061	$1,424	$1,767	$2,119	$2,474
		10.5%	13.4%	16.1%	18.6%	20.7%	22.8%	24.9%
		5.5	4.3	3.6	3.2	2.9	2.7	2.5
	$700	$3,265	$4,349	$5,423	$6,453	$7,483	$8,531	$9,583
		$442	$823	$1,195	$1,538	$1,882	$2,236	$2,593
		11.7%	14.6%	17.2%	19.4%	21.5%	23.7%	25.7%
		4.8	3.9	3.4	3.0	2.8	2.6	2.4
	$800	$3,602	$4,686	$5,720	$6,750	$7,786	$8,834	$9,888
		$583	$964	$1,310	$1,653	$2,000	$2,354	$2,711
		12.9%	15.8%	18.1%	20.3%	22.4%	24.6%	26.6%
		4.3	3.6	3.2	3.0	2.7	2.5	2.3
	$900	$3,940	$4,988	$6,018	$7,048	$8,088	$9,136	$10,192
		$725	$1,081	$1,424	$1,768	$2,117	$2,472	$2,828
		14.1%	16.7%	19.0%	21.2%	23.3%	25.5%	27.4%
		3.9	3.4	3.1	2.8	2.6	2.4	2.3
	$1,000	$4,255	$5,285	$6,315	$7,346	$8,390	$9,441	$10,496
		$851	$1,195	$1,539	$1,882	$2,234	$2,590	$2,945
		15.2%	17.6%	19.9%	22.1%	24.2%	26.3%	28.3%
		3.6	3.2	2.9	2.7	2.5	2.3	2.2
	$1,100	$4,553	$5,583	$6,613	$7,644	$8,692	$9,745	$10,801
		$967	$1,310	$1,653	$1,997	$2,352	$2,707	$3,063
		16.1%	18.6%	20.8%	23.0%	25.2%	27.2%	29.2%
		3.4	3.1	2.8	2.6	2.4	2.2	2.1

1.25.8 Production and Sales Information

The Casino Project will produce four products. Shipping, treatment charges and/or marketing for each are discussed below.

1.25.8.1 Copper Concentrate

Concentrate production will average 313,000 tonnes per year at 28% copper and 26.2 g/tonne gold. The concentrate is assumed to be hauled to the port at Haines, Alaska, at $80.16/tonne. That cost assumes a back haul of coal from the port to the mine site. The cost estimate was provided by Associated Engineering, Ltd as part of the road study.

A new concentrate terminal will be built at Haines for concentrate shipping and coal and cargo receiving. Terminal charges at the port for outgoing concentrate are estimated at $10.00/tonne by Lauga and Associates in the Port study.

Ocean shipping to an undetermined Asian smelter is estimated to be $60/tonne. That cost is based on M3's experience and judgement in overseas shipping for other projects.

The study used a concentrate treatment (smelting) charge of $85.00/tonne and a refining charge of $0.085/lb of copper metal. The payable percentages are 96.5% for copper and 97.5% for gold. The treatment charges are averages of future charges estimated by Neil S. Sheldon and Associates in an analysis performed for this project.

1.25.8.2 Molybdenum Concentrate

The project will produce 11,000 tonnes or molybdenum concentrate per year grading 56% molybdenum. The concentrate will be dried and packaged on-site. The study assumes a typical arrangement where the molybdenum customer takes possession of the concentrate at the production site and pays the market price for the contained molybdenum at a payable percentage of 85% to cover treatment charges, transportation charges, etc.

1.25.8.3 Copper Precipitate

The SART process will produce 13,000 tonnes of copper sulphide precipitate during years -3 through 4. The precipitate will be dried and sacked and shipped by truck to port for shipment to an overseas smelter. Charges are the same as for concentrate except the trucking fee is $91.95 since there is no backhaul.

1.25.8.4 Gold Doré

The ADR plant will produce 536,000 ounces (16,670 kg) of gold doré during years -3 through 4. Transportation to market by truck is estimated to cost $226.00/tonne. Refining charges are estimated at $1.30 per ounce at 98% of the metal payable.

1.25.9 Mine Life

Based on the mine production schedule developed for this study, the mine life is three years of preproduction followed by 23.5 years of commercial pit operations. Processing of low grade ores will extend the commercial life to just over 28 years.

The potential exists to convert 150-200 million tonnes of inferred resource into the indicated or measured categories with targeted drilling to depth adjacent to the Casino fault, and more shallowly in areas west of the main zone. A large proportion of this inferred material is contained within the ultimate pit design and upon conversion has the potential to extend mine life.

1.25.10 Reclamation and Closure

A reclamation and closure plan must be prepared by the mine owner and submitted for review and approval by the government prior to receiving a Quartz Mining License. The reclamation and closure plan must be updated periodically throughout the operating mine life (minimum every five years). A conceptual plan will be expected to support the environmental assessment process, while a more detailed plan is expected to be required as a condition in the Quartz Mine License. Provisions for changes and updates as mining progresses are also expected.

The Yukon Government will require the company to post security for this project. The Yukon government will determine the form and amount of security to be provided by the mine operator to cover the full amount of outstanding mine reclamation and closure liability.

The overall objectives of the mine reclamation and closure plan will be:

- Protection of public health and safety by restricting access to potentially dangerous locations such as the pit;
- Prevent, minimize or mitigate adverse environmental impacts;
- Reclaim the site such that, in time, the reclaimed land is comparable visually and for land use purposes to the undisturbed surrounding land by removing mine facilities to ground level and revegetating cleared areas, and
- Ensure long term stability of the tailings, heap leach and waste rock storage area and site water quality by recontouring, covering and revegetating as appropriate.

The plan will be designed to achieve the above objectives in a “walkaway” scenario, that is, one in which there will be no further requirements for monitoring and maintenance. It is envisaged that a period of post reclamation “active care” will be required until it has been satisfactorily demonstrated from the results of site monitoring that reclamation measures have achieved the required outcomes and are self sustaining.

Table 1.25.8-1: Simplified Cash Flow

Pre-Feasibility Casino Project
Financial Model
7/31/2008

		Total	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Mining Operations																					
Mill Ore	Mined (kt)	757,375		34	2,121	6,334	18,511	35,400	33,370	33,300	34,550	32,125	32,404	32,439	33,600	32,225	32,200	32,250	32,100	32,000	32,000
Low Grade Mill Ore	Mined (kt)	156,171		33	1,465	2,558	6,420	11,718	10,268	6,699	15,335	18,424	8,120	4,143	10,627	13,726	6,901	5,289	5,725	3,895	4,215
Gold Ore	Mined (kt)	77,917		4,154	13,376	14,833	19,919	4,746	9,731	11,158	-	-	-	-	-	-	-	-	-	-	-
Waste	Mined (kt)	995,172		3,991	3,824	6,576	14,670	38,136	36,631	38,843	40,115	39,451	49,476	53,418	45,773	44,049	50,899	52,461	52,175	54,105	53,785
	Total Material Mined (kt)	1,986,635		8,212	20,786	30,301	59,520	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000	90,000
	Waste to Ore Ratio	1.00		0.95	0.23	0.28	0.33	0.74	0.69	0.76	0.80	0.78	1.22	1.46	1.03	0.96	1.30	1.40	1.38	1.51	1.49
	Mine Re-handle (kt)	164,662		-	-	8,489	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Process Plant Operations																					
Concentrator	Recovered Copper (klbs)	3,596,133		-	-	-	119,955	259,105	228,797	216,571	187,377	145,896	122,874	131,589	155,558	127,169	131,329	125,134	116,768	101,589	90,301
	Recovered Gold (kozs)	4,590		-	-	-	320	315	280	251	213	198	172	168	173	150	156	138	138	131	115
	Recovered Molybdenum (klbs)	316,065		-	-	-	9,073	15,896	19,178	15,678	11,404	9,258	14,496	18,045	11,372	14,632	14,135	11,834	12,048	11,849	10,080
ADR/SART Plant	Recovered Copper (klbs)	21,442		971	3,893	4,513	6,763	1,716	1,373	2,214	-	-	-	-	-	-	-	-	-	-	-
	Recovered Gold Dore (kozs)	536		24	105	129	148	23	53	54	-	-	-	-	-	-	-	-	-	-	-
Payable Metals																					
Copper Concentrate	Payable Copper (klbs)	3,470,268					115,756	250,036	220,789	208,991	180,819	140,790	118,574	126,983	150,113	122,718	126,733	120,755	112,681	98,033	87,141
	Payable Gold (kozs)	4,476					312	307	273	245	208	193	168	164	169	147	153	134	134	128	113
Molybdenum Concentrates	Payable Molybdenum (klbs)	268,655					7,712	13,512	16,301	13,327	9,693	7,869	12,322	15,338	9,666	12,437	12,015	10,059	10,240	10,072	8,568
Gold Dore	Payable Metal (kozs)	525		24	103	126	145	23	52	53	-	-	-	-	-	-	-	-	-	-	-
Copper Precipitate	Payable Metal (klbs)	20,691		937	3,756	4,355	6,526	1,656	1,325	2,136	-	-	-	-	-	-	-	-	-	-	-
Income Statement ($000)																					
Revenues	Copper Concentrates - Cu	10,237,291		-	-	-	341,482	737,606	651,328	616,523	533,416	415,330	349,792	374,601	442,835	362,017	373,861	356,226	332,408	289,198	257,065
	Copper Concentrates - Au	2,897,493		-	-	-	202,224	198,669	176,996	158,338	134,665	125,213	108,505	106,016	109,360	94,957	98,765	86,824	86,850	82,722	72,864
	Molybdenum Concentrates	8,320,250		-	-	-	238,847	418,452	504,845	412,722	300,208	243,713	381,608	475,021	299,363	385,185	372,104	311,517	317,147	311,925	265,348
	Gold Dore	340,002		15,337	66,759	81,615	93,710	14,707	33,638	34,235	-	-	-	-	-	-	-	-	-	-	-
	Copper Precipitate	61,039		2,763	11,081	12,847	19,252	4,885	3,909	6,302	-	-	-	-	-	-	-	-	-	-	-
	Total Revenues	21,856,075		18,100	77,840	94,462	895,514	1,374,319	1,370,716	1,228,121	968,288	784,256	839,905	955,637	851,558	842,160	844,731	754,566	736,405	683,845	595,276
Operating Cost	Mining	2,955,462		-	-	-	91,073	118,456	123,202	132,866	127,667	122,773	128,075	134,903	122,486	120,602	124,868	128,801	130,445	128,531	127,969
	Concentrator	5,490,151		-	-	-	164,100	211,152	199,781	199,389	206,391	192,808	194,370	194,566	201,070	193,368	193,228	193,508	192,668	192,107	192,107
	ADR/SART Plant	248,866		15,110	42,130	46,399	61,301	16,845	31,450	35,631	-	-	-	-	-	-	-	-	-	-	-
	General Administration	433,212		3,628	3,628	5,080	14,513	14,513	14,513	14,513	14,513	14,513	14,513	14,513	14,513	14,513	14,513	14,513	14,513	14,513	14,513
	Treatment & Refining Charges																				
	Copper Concentrates	1,864,192	-	-	-	-	63,289	130,801	115,883	109,771	95,375	75,149	63,815	68,022	79,689	65,771	67,827	64,706	60,645	53,238	47,665
	Gold Dore	700	-	32	138	168	193	30	69	71	-	-	-	-	-	-	-	-	-	-	-
	Copper Precipitate	5,151	-	233	935	1,084	1,625	412	330	532	-	-	-	-	-	-	-	-	-	-	-
	Total Cash Operating Cost	10,997,735		19,003	46,831	52,731	396,094	492,209	485,229	492,773	443,946	405,243	400,774	412,004	417,757	394,254	400,436	401,528	398,271	388,389	382,255
	Royalty	343,553		-	-	-	-	-	-	26,031	22,980	17,630	18,742	24,086	18,079	20,004	17,097	14,675	10,948	9,983	7,466
	Property Tax	3,300		100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100
	Salvage Value	(25,000)		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Reclamation & Closure	100,000		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Total Cash Production Cost	11,419,588		19,103	46,931	52,831	396,194	492,309	485,329	518,904	467,026	422,973	419,616	436,190	435,936	414,358	417,633	416,303	409,318	398,472	389,821
	Operating Income	10,436,487		(1,003)	30,909	41,631	499,321	882,010	885,387	709,217	501,262	361,283	420,290	519,448	415,621	427,802	427,098	338,263	327,087	285,373	205,456
	Total Depreciation	2,974,111		12,920	15,318	27,191	522,253	766,845	808,030	67,228	30,922	23,325	26,359	23,175	33,815	24,472	29,083	28,270	40,658	50,412	47,240
	Net Income After Depreciation	7,462,376		(13,923)	15,591	14,440	(22,933)	115,164	77,357	641,989	470,340	337,958	393,931	496,273	381,806	403,330	398,015	309,993	286,429	234,961	158,216
	Tax Loss Carry Forward Applied	(57,924)		-	(13,923)	-	-	(44,002)	-	-	-	-	-	-	-	-	-	-	-	-	-
	Net Income After Tax Loss Carry Forward	7,404,452		(13,923)	1,668	14,440	(22,933)	71,163	77,357	641,989	470,340	337,958	393,931	496,273	381,806	403,330	398,015	309,993	286,429	234,961	158,216
	Mining Royalty	961,773		-	1,668	2,836	21,069	71,163	77,357	60,724	39,226	25,778	34,738	49,055	37,966	41,479	41,658	32,811	31,300	27,604	19,200
	Taxes at 30%	1,961,522				3,481	-	-	-	174,379	129,334	93,654	107,758	134,165	103,152	108,555	106,907	83,154	76,539	62,207	41,705
	Net Income After Taxes	4,481,157		(13,923)	-	8,123	(44,002)	-	-	406,885	301,780	218,526	251,435	313,053	240,689	253,296	249,450	194,027	178,591	145,150	97,311
	Cash Flow																				
	Operating Income	10,436,487		(1,003)	30,909	41,631	499,321	882,010	885,387	709,217	501,262	361,283	420,290	519,448	415,621	427,802	427,098	338,263	327,087	285,373	205,456
	Total Working Capital	-		(1,913)	(7,533)	(7,248)	(103,458)	(70,808)	19	24,560	38,699	27,071	(9,515)	(18,101)	17,582	(387)	85	14,911	2,718	7,828	14,055
	Total Capital Expenditures	2,974,111	75,700	483,439	864,392	589,580	131,330	96,013	50,217	27,390	25,413	533	29,453	12,767	34,149	6,245	60,594	26,624	87,785	68,093	44,406
	Cash Flow before Taxes	7,462,376	(75,700)	(486,355)	(841,016)	(555,196)	264,532	715,189	835,188	706,387	514,548	387,821	381,322	488,579	399,054	421,170	366,590	326,550	242,020	225,107	175,105
	Cummulative Cash Flow before Taxes		(75,700)	(562,055)	(1,403,072)	(1,958,268)	(1,693,736)	(978,547)	(143,358)	563,029	1,077,577	1,465,398	1,846,720	2,335,299	2,734,353	3,155,523	3,522,113	3,848,663	4,090,683	4,315,790	4,490,895
							1.0	1.0	1.0	0.2	-	-	-	-	-	-	-	-	-	-	-
	Taxes	2,981,219	-	-	15,591	6,317	21,069	115,164	77,357	235,104	168,560	119,432	142,496	183,220	141,118	150,034	148,565	115,966	107,839	89,811	60,904
	Cash Flow after Taxes	4,481,157	(75,700)	(486,355)	(856,607)	(561,514)	243,463	600,025	757,831	471,284	345,988	268,389	238,826	305,359	257,936	271,136	218,024	210,584	134,181	135,296	114,200
	Cummulative Cash Flow after Taxes		(75,700)	(562,055)	(1,418,663)	(1,980,176)	(1,736,713)	(1,136,688)	(378,857)	92,426	438,415	706,804	945,630	1,250,989	1,508,925	1,780,061	1,998,085	2,208,670	2,342,851	2,478,148	2,592,348
							1.0	1.0	1.0	0.8	-	-	-	-	-	-	-	-	-	-	-

Economic Indicators before Taxes		
NPV @ 0%	0%	$ 7,462,376
NPV @ 5%	5%	$ 3,066,433
NPV @ 8%	8%	$ 1,825,018
NPV @ 10%	10%	$ 1,277,296
IRR		20.4%
Payback		3.2

Economic Indicators after Taxes		
NPV @ 0%	0%	$ 4,481,157
NPV @ 5%	5%	$ 1,661,178
NPV @ 8%	8%	$ 866,786
NPV @ 10%	10%	$ 517,636
IRR		14.9%
Payback	Years	3.8

Pre-Feasibility Casino Project
Financial Model
7/31/2008

		16	17	18	19	20	21	22	23	24	25	26	27	28	29	30
Mining Operations																
Mill Ore	Mined (kt)	32,050	32,201	32,100	32,170	32,000	32,025	32,025	32,025	13,816	-	-	-	-	-	-
Low Grade Mill Ore	Mined (kt)	6,308	5,188	2,216	3,725	3,173	-	-	-	-	-	-	-	-	-	-
Gold Ore	Mined (kt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste	Mined (kt)	51,642	52,611	55,684	46,962	51,848	23,236	17,361	10,642	6,808	-	-	-	-	-	-
	Total Material Mined (kt)	90,000	90,000	90,000	82,857	87,021	55,261	49,386	42,667	20,624	-	-	-	-	-	-
	Waste to Ore Ratio	1.35	1.41	1.62	1.31	1.47	0.73	0.54	0.33	0.49						
	Mine Re-handle (kt)	-	-	-	-	-	-	-	-	18,972	33,360	33,360	33,360	33,360	3,761	-
Process Plant Operations																
Concentrator	Recovered Copper (klbs)	89,736	95,838	89,876	87,235	116,404	117,907	112,965	136,970	112,410	91,615	91,615	91,615	91,615	10,320	-
	Recovered Gold (kozs)	116	115	124	120	159	137	121	137	127	125	125	125	125	14	-
	Recovered Molybdenum (klbs)	11,599	11,924	13,016	14,068	12,332	11,000	10,302	10,785	7,215	3,610	3,610	3,610	3,610	407	-
ADR/SART Plant	Recovered Copper (klbs)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Recovered Gold Dore (kozs)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Payable Metals																
Copper Concentrate	Payable Copper (klbs)	86,595	92,483	86,730	84,182	112,330	113,780	109,011	132,176	108,476	88,409	88,409	88,409	88,409	9,959	-
	Payable Gold (kozs)	113	113	121	117	155	133	118	133	124	122	122	122	122	14	-
Molybdenum Concentrates	Payable Molybdenum (klbs)	9,859	10,135	11,063	11,958	10,482	9,350	8,757	9,168	6,133	3,068	3,068	3,068	3,068	346	-
Gold Dore	Payable Metal (kozs)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Copper Precipitate	Payable Metal (klbs)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Income Statement ($000)																
Revenues	Copper Concentrates - Cu	$ 255,455	$ 272,826	$ 255,854	$ 248,336	$ 331,373	$ 335,652	$ 321,582	$ 389,919	$ 320,004	$ 260,806	$ 260,806	$ 260,806	$ 260,806	$ 29,380	$ -
	Copper Concentrates - Au	$ 72,978	$ 72,890	$ 78,251	$ 75,836	$ 100,295	$ 86,218	$ 76,352	$ 86,218	$ 80,148	$ 79,106	$ 79,106	$ 79,106	$ 79,106	$ 8,918	$ -
	Molybdenum Concentrates	$ 305,344	$ 313,884	$ 342,632	$ 370,339	$ 324,627	$ 289,568	$ 271,205	$ 283,918	$ 189,931	$ 95,024	$ 95,024	$ 95,024	$ 95,024	$ 10,704	$ -
	Gold Dore	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
	Copper Precipitate	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
	Total Revenues	$ 633,777	$ 659,600	$ 676,737	$ 694,510	$ 756,295	$ 711,437	$ 669,140	$ 760,054	$ 590,083	$ 434,936	$ 434,936	$ 434,936	$ 434,936	$ 49,001	$ -
Operating Cost	Mining	$ 132,316	$ 134,669	$ 133,309	$ 125,859	$ 122,694	$ 89,822	$ 85,870	$ 81,118	$ 63,581	$ 29,245	$ 31,163	$ 27,710	$ 29,352	$ 6,038	$ -
	Concentrator	$ 192,387	$ 193,233	$ 192,668	$ 193,060	$ 192,107	$ 192,247	$ 192,247	$ 192,247	$ 196,519	$ 199,724	$ 199,724	$ 199,724	$ 199,724	$ 33,925	$ -
	ADR/SART Plant	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
	General Administration	$ 14,513	$ 14,513	$ 14,513	$ 14,513	$ 14,513	$ 14,513	$ 14,513	$ 14,513	$ 14,513	$ 14,513	$ 14,513	$ 14,513	$ 14,513	$ 14,513	$ -
	Treatment & Refining Charges															
	Copper Concentrates	$ 47,392	$ 50,353	$ 47,510	$ 46,205	$ 60,597	$ 61,192	$ 58,700	$ 70,446	$ 58,467	$ 48,362	$ 48,362	$ 48,362	$ 48,362	$ 8,234	$ -
	Gold Dore	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
	Copper Precipitate	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
	Total Cash Operating Cost	386,608	392,769	388,000	379,637	389,911	357,774	351,330	358,324	333,080	291,844	293,762	290,309	291,951	62,710	-
	Royalty	$ 7,764	$ 7,711	$ 12,924	$ 12,032	$ 16,318	$ 14,829	$ 14,173	$ 16,771	$ 11,446	$ 5,392	$ 6,321	$ 5,111	$ 5,040	$ -	$ -
	Property Tax	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100
	Salvage Value	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ (25,000)	$ -
	Reclamation & Closure	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 31,000	$ 31,000	$ 31,000	$ 7,000
	Total Cash Production Cost	$ 394,472	$ 400,580	$ 401,024	$ 391,769	$ 406,329	$ 372,703	$ 365,604	$ 375,195	$ 344,626	$ 297,337	$ 300,183	$ 326,520	$ 328,090	$ 68,810	$ 7,100
	Operating Income	$ 239,305	$ 259,021	$ 275,712	$ 302,742	$ 349,966	$ 338,734	$ 303,536	$ 384,860	$ 245,456	$ 137,599	$ 134,753	$ 108,415	$ 106,845	$ (19,809)	$ (7,100)
	Total Depreciation	$ 54,285	$ 59,927	$ 50,401	$ 44,731	$ 39,778	$ 32,619	$ 22,093	$ 24,601	$ 20,388	$ 11,863	$ 13,100	$ 5,746	$ 6,171	$ 10,894	$ -
	Net Income After Depreciation	$ 185,019	$ 199,094	$ 225,312	$ 258,011	$ 310,188	$ 306,115	$ 281,443	$ 360,258	$ 225,069	$ 125,736	$ 121,653	$ 102,670	$ 100,674	$ (30,703)	$ (7,100)
	Tax Loss Carry Forward Applied	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Net Income After Tax Loss Carry Forward	185,019	199,094	225,312	258,011	310,188	306,115	281,443	360,258	225,069	125,736	121,653	102,670	100,674	(30,703)	(7,100)
	Mining Royalty	$ 23,411	$ 26,010	$ 30,093	$ 32,806	$ 39,500	$ 37,955	$ 34,241	$ 43,685	$ 27,491	$ 14,238	$ 14,372	$ 11,249	$ 11,089	$ -	$ -
	Taxes at 30%	48,482	51,925	58,566	67,562	81,206	80,448	74,161	94,972	59,273	33,449	32,184	27,426	26,876	-	-
	Net Income After Taxes	113,125	121,159	136,653	157,644	189,481	187,712	173,041	221,601	138,304	78,048	75,096	63,995	62,710	(30,703)	(7,100)
	Cash Flow															
	Operating Income	$ 239,305	$ 259,021	$ 275,712	$ 302,742	$ 349,966	$ 338,734	$ 303,536	$ 384,860	$ 245,456	$ 137,599	$ 134,753	$ 108,415	$ 106,845	$ (19,809)	$ (7,100)
	Total Working Capital	$ (5,971)	$ (3,739)	$ (3,209)	$ (3,609)	$ (9,312)	$ 4,732	$ 6,423	$ (14,370)	$ 25,866	$ 22,114	$ 158	$ (284)	$ 135	$ 44,600	$ 7,901
	Total Capital Expenditures	$ 68,378	$ 86,505	$ 59	$ 41,324	$ 428	$ 19,028	$ -	$ 22,527	$ 487	$ 20,909	$ 285	$ 59	$ -	$ -	$ -
	Cash Flow before Taxes	$ 164,955	$ 168,777	$ 272,445	$ 257,809	$ 340,226	$ 324,439	$ 309,959	$ 347,963	$ 270,835	$ 138,805	$ 134,625	$ 108,073	$ 106,980	$ 24,791	$ 801
	Cummulative Cash Flow before Taxes	$ 4,655,850	$ 4,824,627	$ 5,097,072	$ 5,354,880	$ 5,695,106	$ 6,019,545	$ 6,329,504	$ 6,677,467	$ 6,948,302	$ 7,087,107	$ 7,221,732	$ 7,329,805	$ 7,436,785	$ 7,461,576	$ 7,462,376
		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Taxes	$ 71,894	$ 77,935	$ 88,659	$ 100,367	$ 120,707	$ 118,403	$ 108,402	$ 138,657	$ 86,765	$ 47,688	$ 46,557	$ 38,675	$ 37,964	$ -	$ -
	Cash Flow after Taxes	$ 93,061	$ 90,841	$ 183,786	$ 157,441	$ 219,519	$ 206,035	$ 201,558	$ 209,306	$ 184,071	$ 91,117	$ 88,069	$ 69,398	$ 69,016	$ 24,791	$ 801
	Cummulative Cash Flow after Taxes	$ 2,685,409	$ 2,776,251	$ 2,960,037	$ 3,117,478	$ 3,336,997	$ 3,543,033	$ 3,744,590	$ 3,953,896	$ 4,137,967	$ 4,229,084	$ 4,317,153	$ 4,386,550	$ 4,455,566	$ 4,480,357	$ 4,481,157
		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Economic Indicators before Taxes															
	NPV @ 0%															
	NPV @ 5%															
	NPV @ 8%															
	NPV @ 10%															
	IRR															
	Payback															
	Economic Indicators after Taxes															
	NPV @ 0%															
	NPV @ 5%															
	NPV @ 8%															
	NPV @ 10%															
	IRR															
	Payback															

1.26 ILLUSTRATIONS

This section contains all illustrations which are not imbedded in other sections of this report.

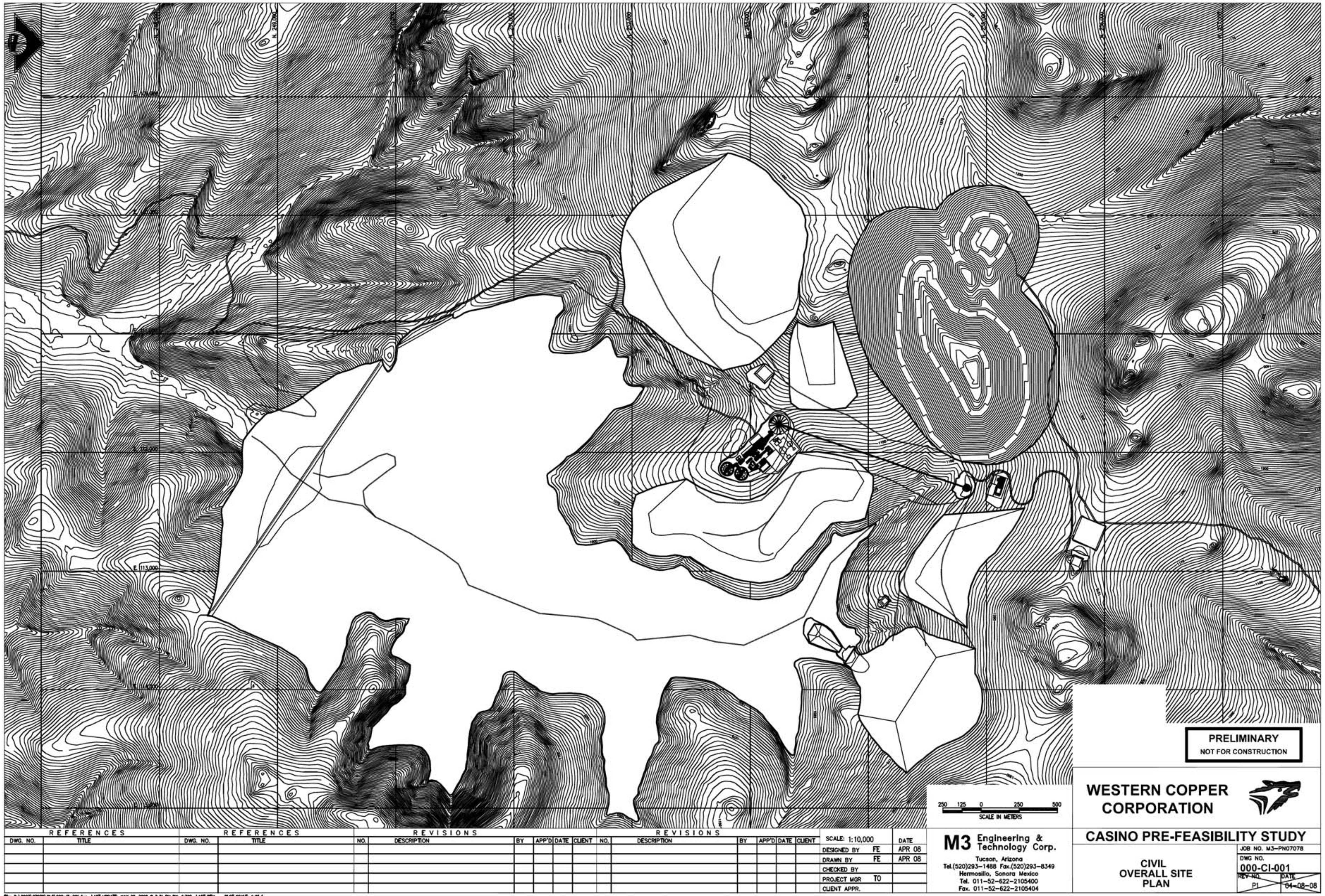


Figure 1.7-1: Overall Site Plan

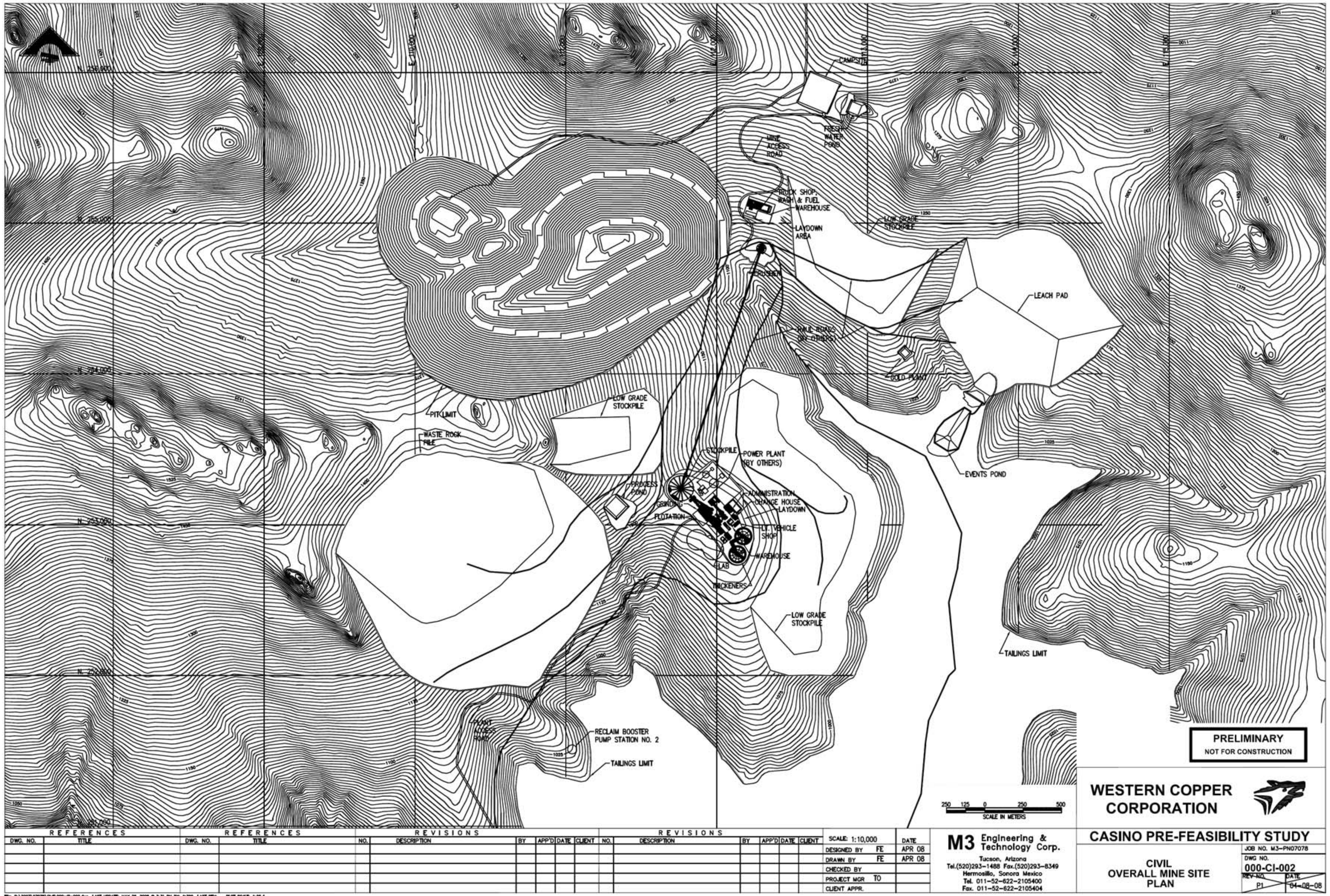
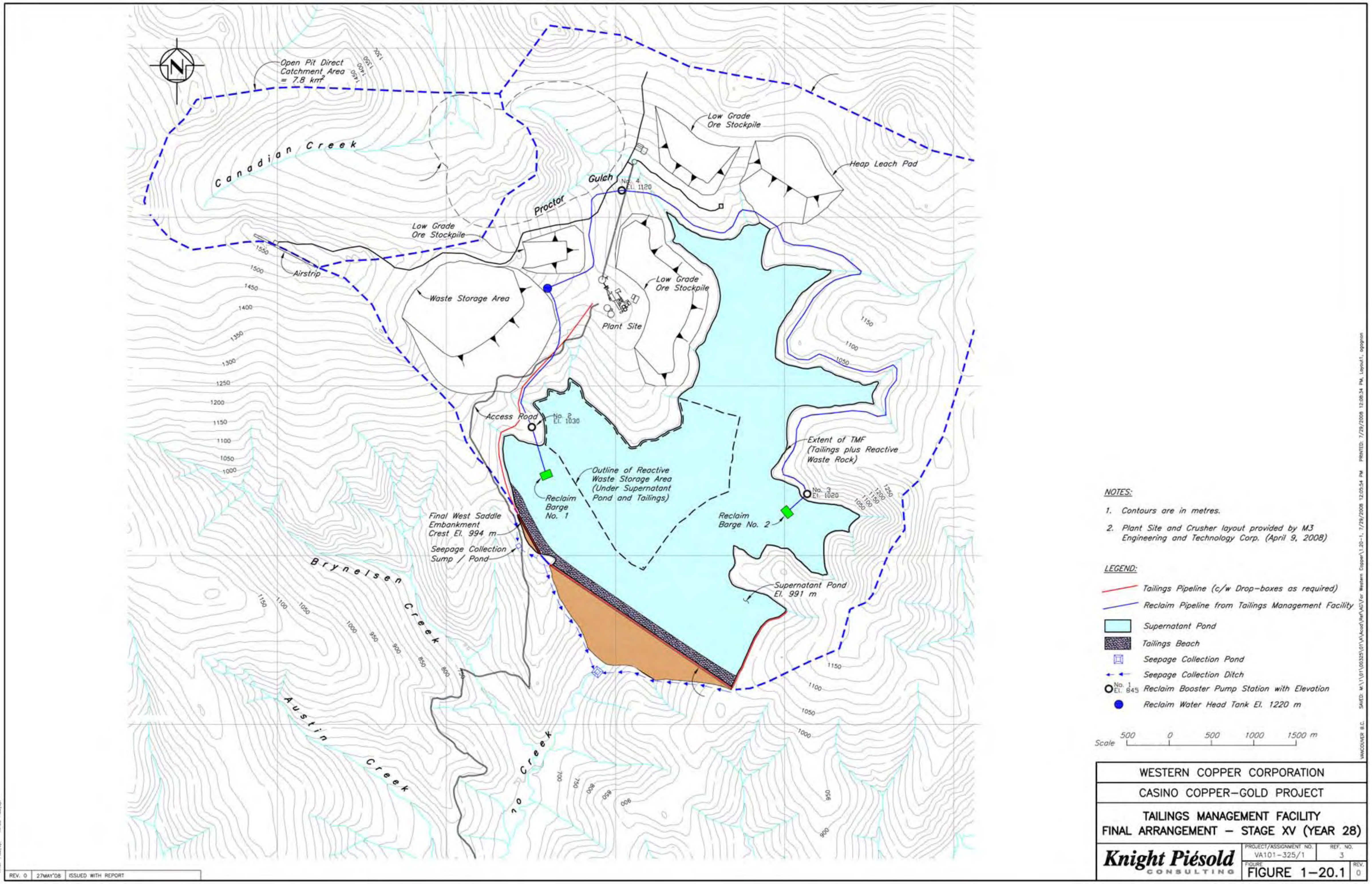
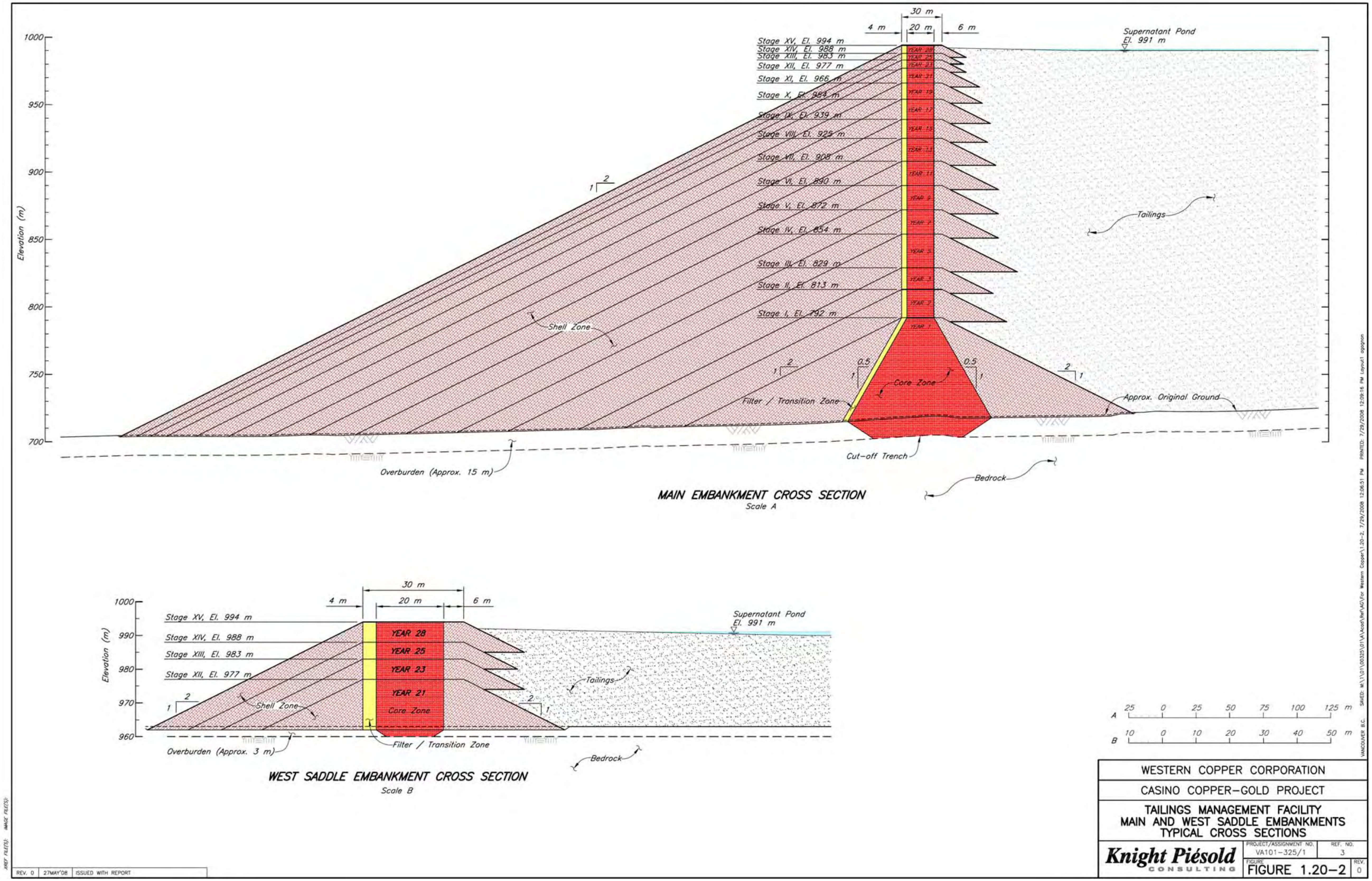
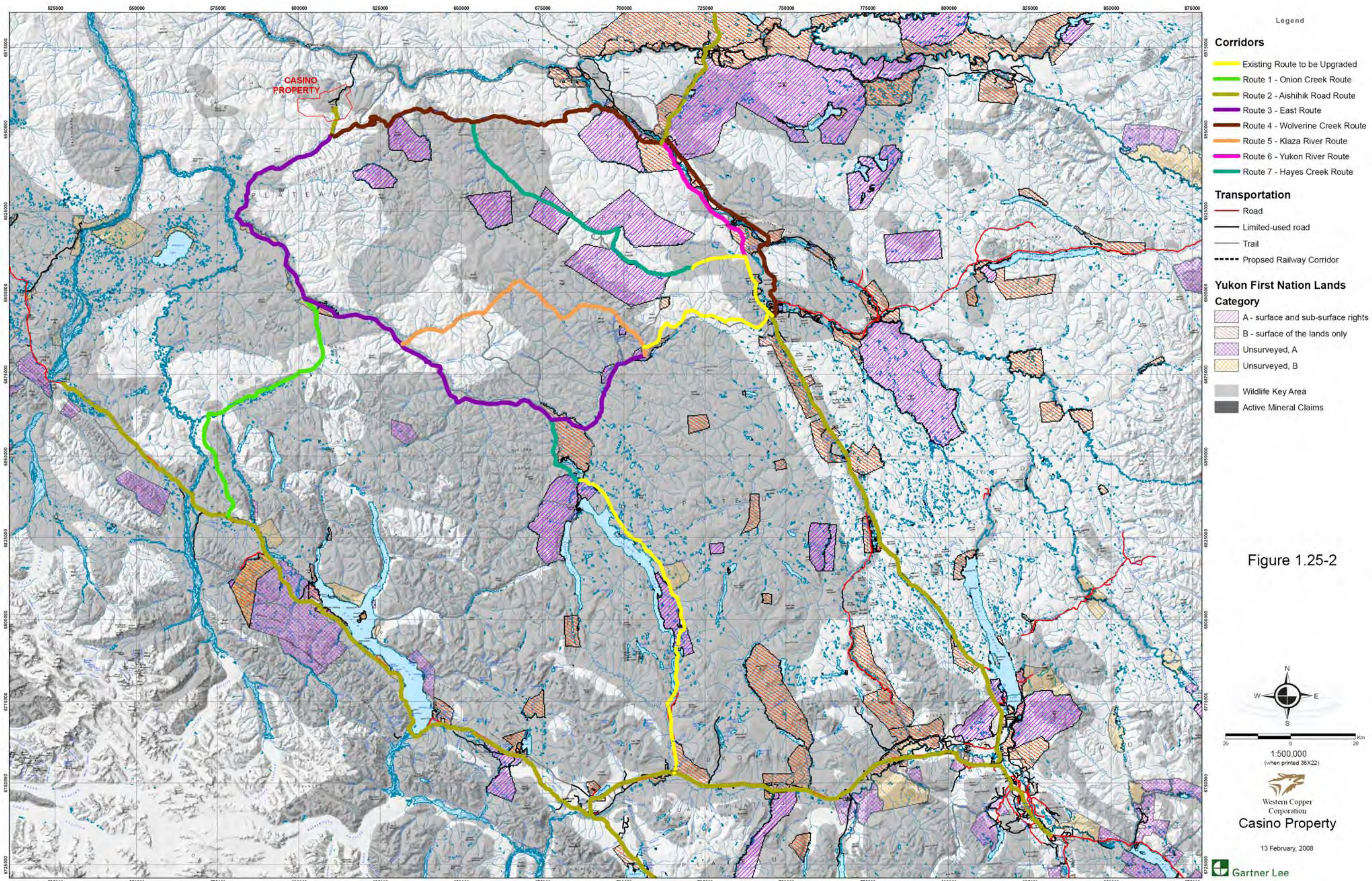
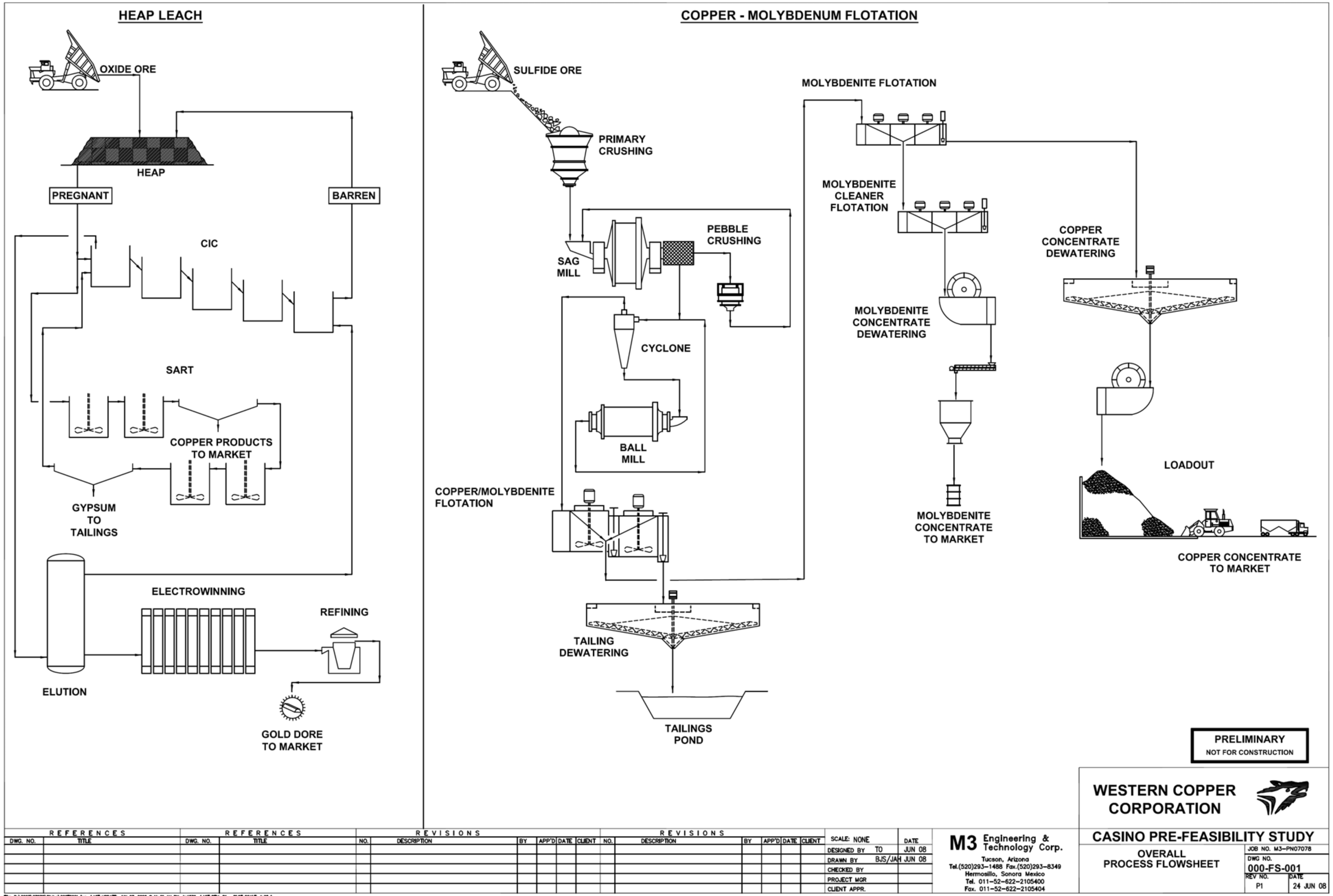


Figure 1.7-2: Overall Site Plan

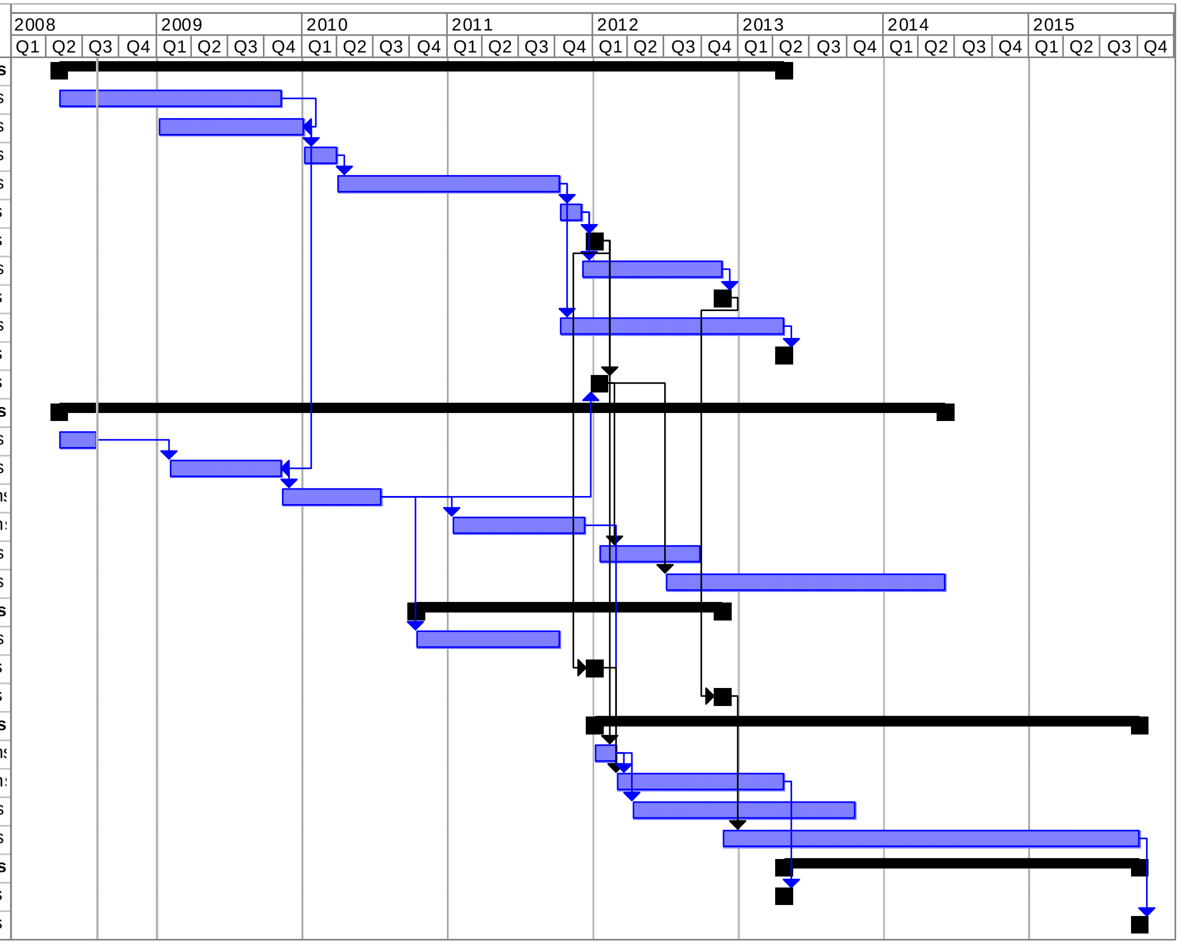

Open Pit Direct Catchment Area = 7.8 km²
Canadian Creek
Proctor Gulch
Low Grade Ore Stockpile
Heap Leach Pad
Airstrip
Waste Storage Area
Plant Site
Access Road
Outline of Reactive Waste Storage Area (Under Supernatant Pond and Tailings)
Extent of TMF (Tailings plus Reactive Waste Rock)
Reclaim Barge No. 1
Reclaim Barge No. 2
Final West Saddle Embankment Crest El. 994 m
Seepage Collection Sump / Pond
Supernatant Pond El. 991 m
Brynelsen Creek
Austin Creek
NOTES:
1. Contours are in metres.
2. Plant Site and Crusher layout provided by M3 Engineering and Technology Corp. (April 9, 2008)
LEGEND:
Tailings Pipeline (c/w Drop-boxes as required)
Reclaim Pipeline from Tailings Management Facility
Supernatant Pond
Tailings Beach
Seepage Collection Pond
Seepage Collection Ditch
Reclaim Booster Pump Station with Elevation
Reclaim Water Head Tank El. 1220 m
Scale 500 0 500 1000 1500 m
WESTERN COPPER CORPORATION
CASINO COPPER-GOLD PROJECT
TAILINGS MANAGEMENT FACILITY
FINAL ARRANGEMENT – STAGE XV (YEAR 28)
Knight Piésold CONSULTING
VA101-325/1
FIGURE 1-20.1
REV. 0 27MAY'08 ISSUED WITH REPORT


MAIN EMBANKMENT CROSS SECTION
Scale A
WEST SADDLE EMBANKMENT CROSS SECTION
Scale B
Supernatant Pond
El. 991 m
Shell Zone
Core Zone
Filter / Transition Zone
Tailings
Cut-off Trench
Bedrock
Overburden (Approx. 15 m)
Overburden (Approx. 3 m)
Approx. Original Ground
Elevation (m)
WESTERN COPPER CORPORATION
CASINO COPPER–GOLD PROJECT
TAILINGS MANAGEMENT FACILITY
MAIN AND WEST SADDLE EMBANKMENTS
TYPICAL CROSS SECTIONS
Knight Piésold
CONSULTING
VA101-325/1
FIGURE 1.20–2


CASINO PROPERTY
Legend
Corridors
Existing Route to be Upgraded
Route 1 - Onion Creek Route
Route 2 - Aishihik Road Route
Route 3 - East Route
Route 4 - Wolverine Creek Route
Route 5 - Klaza River Route
Route 6 - Yukon River Route
Route 7 - Hayes Creek Route
Transportation
Road
Limited-used road
Trail
Propsed Railway Corridor
Yukon First Nation Lands
Category
A - surface and sub-surface rights
B - surface of the lands only
Unsurveyed, A
Unsurveyed, B
Wildlife Key Area
Active Mineral Claims
Figure 1.25-2
N
W
E
S
1:500,000
Western Copper Corporation
Casino Property
13 February, 2008
Gartner Lee


HEAP LEACH
OXIDE ORE
HEAP
PREGNANT
BARREN
CIC
SART
COPPER PRODUCTS TO MARKET
GYPSUM TO TAILINGS
ELECTROWINNING
REFINING
ELUTION
GOLD DORE TO MARKET
COPPER - MOLYBDENUM FLOTATION
SULFIDE ORE
PRIMARY CRUSHING
PEBBLE CRUSHING
SAG MILL
CYCLONE
BALL MILL
COPPER/MOLYBDENITE FLOTATION
TAILING DEWATERING
TAILINGS POND
MOLYBDENITE FLOTATION
MOLYBDENITE CLEANER FLOTATION
MOLYBDENITE CONCENTRATE DEWATERING
MOLYBDENITE CONCENTRATE TO MARKET
COPPER CONCENTRATE DEWATERING
LOADOUT
COPPER CONCENTRATE TO MARKET
PRELIMINARY
NOT FOR CONSTRUCTION
WESTERN COPPER CORPORATION
CASINO PRE-FEASIBILITY STUDY
OVERALL PROCESS FLOWSHEET
JOB NO. M3-PN07078
DWG NO. 000-FS-001
REV NO. P1
DATE 24 JUN 08
M3 Engineering & Technology Corp.
Tucson, Arizona
SCALE: NONE
DESIGNED BY TO JUN 08
DRAWN BY BJS/JAH JUN 08
REFERENCES
REVISIONS

ID	Task Name	Duration
1	**Permitting**	**260 wks**
2	Baseline Monitoring	80 wks
3	Prepare & Submit Project Proposal	52 wks
4	YESAB Review Proposal for Adequacy	12 wks
5	YESSA Review	80 wks
6	YTG Issue Decision Document	8 wks
7	Quartz Mining License Issued	0 wks
8	WUL Appln Review & Hearings	50 wks
9	Water License Issued	0 wks
10	Schedule II Amendment Hearings	80 wks
11	Amendment issued	0 wks
12	WCC Production Decision	0 wks
13	**Engineering**	**318 wks**
14	Pre-Feasiblity	14 wks
15	Permit & Interim FS Engineering	40 wks
16	Feasiblity Study	9 mons
17	Basic/Interim Eng & Procmnt	12 mons
18	Detailed Eng & Proc for Heap Leach	36 wks
19	Detailed Eng & Proc for Mill	100 wks
20	**Financing**	**110 wks**
21	Negotiate Financing Terms	52 wks
22	Release Funds for Heap Leach	0 wks
23	Release Financing of Milling Operation	0 wks
24	**Construction**	**196 wks**
25	Build Winter Road	2 mons
26	Gold Heap Leach	15 mons
27	All Season Road Construction	80 wks
28	Flotation Mill	150 wks
29	**Operation**	**128 wks**
30	Heap Leach Operation Starts	0 wks
31	Milling Operation Starts	0 wks



Appendix A: Professional Qualifications

TO: Western Copper Corporation
British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Securities Commission
Manitoba Securities Commission
Ontario Securities Commission
Authorité des marchés financiers du Québec
Nova Scotia Securities Commission
New Brunswick Office of the Administrator of Securities
PEI Provincial Affairs & Attorney General
Department of Government Services and Lands, Government of Newfoundland and Labrador
The Toronto Stock Exchange

I, Bruno Borntraeger, do hereby consent to the public filing by Western Copper Corporation (the "Company") of the technical report entitled "Technical Report, Casino Project Pre-Feasibility Study, Yukon Territory, Canada", dated August 5, 2008 (the "Technical Report") prepared for the Company by M3 Engineering & Technology Corporation, and a summary of the Technical Report contained in the news release (the "News Release") of the Company dated June 24, 2008 relating to the subject matter of the Technical Report, with the above securities regulatory authorities.

I confirm that I have read the written disclosure in the News Release and the News Release fairly and accurately represents the information in the Technical Report.

Dated as of the 5th day of August, 2008.

Bruno Borntraeger

Signature of Qualified Person

PROFESSIONAL YUKON BRUNO BORNTRAEGER TERRITORY ENGINEER

Aug 5 2008

BRUNO BORNTRAEGER

Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Bruno Borntraeger

As a co-author of this report on certain mineral properties of Western Copper Corp., in the Yukon Territory of Canada, I, Bruno Borntraeger of **12158 59 Ave Surrey BC Canada, V3X 3L4, Geological Engineer** do hereby certify that:

1. I am employed by, and carried out this assignment for Knight Piésold Ltd 1400-750- West Pender Street Vancouver, BC V6C 2T8 Canada, phone number (604) 685-0543;

2. This certificate applies to the Technical Report titled "Technical Report, Casino Project Pre-Feasibility Study, Yukon Territory, Canada", dated August 5, 2008;

3. I hold the following academic qualifications:

    I am a graduate of the University of British Columbia (Bachelor of Applied Science, Geological Engineering, 1990)

4. I am a:

    Professional Engineer in good standing of the association of Professional Engineers and Geoscientists of BC (20926)

    Professional Engineer in good standing of the association of Professional Engineerss of Yukon Territory (1219)

5. I have worked in the minerals industry as an engineer continuously since **1990** a period of **18** years;

6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfill the requirements of a Qualified Person as defined in NI 43-101.

7. I have read NI 43-101 and this technical report has been prepared in compliance with the instrument;

8. I am responsible for the **Geotechnical, Tailings Facility and Heap Leach Pad and Pond** sections presented in this report. In addition, I visited the Property on July 21, 2008.

9. I am independent of the parties involved in the transaction for which this report is required using the tests in section 1.4 of NI 43-101;

10. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make this technical report not misleading;

11. I consent to the filing of this report with any Canadian stock exchange or securities regulatory authority, and any publication by them of the report.

Dated this 5th day of August, 2008

Bruno Borntraeger

Bruno Borntraeger, P. Eng.

PROFESSIONAL ENGINEER
YUKON
BRUNO BORNTRAEGER
TERRITORY

Aug 5, 2008

TO: Western Copper Corporation
British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Securities Commission
Manitoba Securities Commission
Ontario Securities Commission
Authorité des marchés financiers du Québec
Nova Scotia Securities Commission
New Brunswick Office of the Administrator of Securities
PEI Provincial Affairs & Attorney General
Department of Government Services and Lands, Government of Newfoundland and Labrador
The Toronto Stock Exchange

I, Thomas L. Drielick, P.E., do hereby consent to the public filing by Western Copper Corporation (the "Company") of the technical report entitled "Technical Report, Casino Project Pre-Feasibility Study, Yukon Territory, Canada", dated August 5, 2008 (the "Technical Report") prepared for the Company by M3 Engineering & Technology Corporation, and a summary of the Technical Report contained in the news release (the "News Release") of the Company dated June 24, 2008 relating to the subject matter of the Technical Report, with the above securities regulatory authorities.

I confirm that I have read the written disclosure in the News Release and the News Release fairly and accurately represents the information in the Technical Report.

Dated as of the 5th day of August, 2008.

Signature of Qualified Person

Thomas L. Drielick P.E

Print Name of Qualified Person

Registered Professional Engineer (Mining)
CERTIFICATE NO
22958
THOMAS L
DRIELICK
Date Signed 8/5/08
ARIZONA U.S.A.

Expires 9/30/08

CERTIFICATE OF QUALIFIED PERSON

Thomas L. Drielick

As a co-author of this report on certain mineral properties of Western Copper Corp., in the Yukon Territory of Canada, I, Thomas L. Drielick, P.E., of M3 Engineering & Technology Corporation 2440 W, Ruthrauff Road, Tucson, Arizona, 85705, Metallurgical Engineer, do hereby certify that:

1. I am employed by and carried out this assignment for M3 Engineering & Technology Corporation.

2. This certificate applies to the Technical Report titled "Technical Report, Casino Project Pre-Feasibility Study, Yukon Territory, Canada", dated August 5, 2008.

3. I hold the following academic qualifications:

   B.S. Metallurgical Engineering, Michigan Technological University, and
   MBA, Southern Illinois University

4. I am a member of the Society for Mining, Metallurgy and Exploration, Inc. (SME) and the American Association of Cost Engineers (AACE).

5. I have worked in the minerals industry as an engineer continuously since 1971, a period of 37 years;

6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfill the requirements of a Qualified Person as defined in NI 43-101.

7. I have read NI 43-101 and this technical report has been prepared in compliance with the instrument.

8. I am responsible for the Section 7, Process Plant, presented in this report.

9. I am independent of the parties involved in the transaction for which this report is required using the tests in section 1.4 of NI 43-101.

10. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make this technical report not misleading;

11. I consent to the filing of this report with any Canadian stock exchange or securities regulatory authority, and any publication by them of the report.

Dated this 5th day of August, 2008

Thomas L. Drielick, P.E.

Registered Professional Engineer (Mining)
CERTIFICATE NO
22958
THOMAS L
DRIELICK
Date Signed 8/5/08
ARIZONA U.S.A.
Expires 9/30/08

TO: Western Copper Corporation
British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Securities Commission
Manitoba Securities Commission
Ontario Securities Commission
Authorité des marchés financiers du Québec
Nova Scotia Securities Commission
New Brunswick Office of the Administrator of Securities
PEI Provincial Affairs & Attorney General
Department of Government Services and Lands, Government of Newfoundland and Labrador
The Toronto Stock Exchange

I, Jesse Duke, do hereby consent to the public filing by Western Copper Corporation (the "Company") of the technical report entitled "Technical Report, Casino Project Pre-Feasibility Study, Yukon Territory, Canada", dated August 5, 2008 (the "Technical Report") prepared for the Company by M3 Engineering & Technology Corporation, and a summary of the Technical Report contained in the news release (the "News Release") of the Company dated June 24, 2008 relating to the subject matter of the Technical Report, with the above securities regulatory authorities.

I confirm that I have read the written disclosure in the News Release and the News Release fairly and accurately represents the information in the Technical Report.

Dated as of the 5th day of August, 2008.

Jesse L. Duke

Signature of Qualified Person

Jesse L. Duke

Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Jesse L. Duke

As a co-author of this report on certain mineral properties of Western Copper Corp., in the Yukon Territory of Canada, I, Jesse L. Duke of Mile 932 Old Alaska Highway, P.O Box 40165, Whitehorse, Yukon, Geologist do hereby certify that:

1. I am employed by, and carried out this assignment for Gartner Lee, Ltd. 2251 2nd Ave., Whitehorse Yukon Y1A 5W1, Canada, phone number (867) 633-6474 ;

2. This certificate applies to the Technical Report titled "Technical Report, Casino Project Pre-Feasibility Study, Yukon Territory, Canada", dated August 5, 2008;

3. This certificate applies to the Technical Report titled "Technical Report, Casino Project Pre-Feasibility Study, Yukon Territory, Canada", dated August 5, 2008;

4. I hold the following academic qualifications:

   Bachelors of Science Degree in Geology, issued by the University of Alaska.

5. I am a member is good standing with the Association of Professional Engineers and Geoscientists of British Columbia (License Number 20417).

6. I have worked in the industry and government in a capacity directly related to the minerals industry continuously since 1986, a period of 22 years.

7. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfill the requirements of a Qualified Person as defined in NI 43-101.

8. I have read NI 43-101 and this technical report has been prepared in compliance with the instrument.

9. I am responsible for sections 1.3.18, 1.25.4, AND 1.25.10 presented in this report.

10. I am independent of the parties involved in the transaction for which this report is required using the tests in section 1.4 of NI 43-101;

11. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make this technical report not misleading.

12. I consent to the filing of this report with any Canadian stock exchange or securities regulatory authority, and any publication by them of the report.

Dated this 5th day of August, 2008

Jesse L. Duke P. Geo.

TO: Western Copper Corporation
British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Securities Commission
Manitoba Securities Commission
Ontario Securities Commission
Authorité des marchés financiers du Québec
Nova Scotia Securities Commission
New Brunswick Office of the Administrator of Securities
PEI Provincial Affairs & Attorney General
Department of Government Services and Lands, Government of Newfoundland and Labrador
The Toronto Stock Exchange

I, Gary H. Giroux, do hereby consent to the public filing by Western Copper Corporation (the "Company") of the technical report entitled "Technical Report, Casino Project Pre-Feasibility Study, Yukon Territory, Canada", dated August 5, 2008 (the "Technical Report") prepared for the Company by M3 Engineering & Technology Corporation, and a summary of the Technical Report contained in the news release (the "News Release") of the Company dated June 24, 2008 relating to the subject matter of the Technical Report, with the above securities regulatory authorities.

I confirm that I have read the written disclosure in the News Release and the News Release fairly and accurately represents the information in the Technical Report.

Dated as of the 5th day of August, 2008.

Signature of Qualified Person

G.H. GIROUX
Print Name of Qualified Person

PROFESSIONAL ENGINEER
PROVINCE OF BRITISH COLUMBIA
G. H. GIROUX

CERTIFICATE OF QUALIFIED PERSON

Gary H. Giroux

As a co-author of this report on certain mineral properties of Western Copper Corp., in the Yukon Territory of Canada,

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

1) I am a consulting geological engineer with an office at #1215 - 675 West Hastings Street, Vancouver, British Columbia.
2) I am a graduate of the University of British Columbia in 1970 with a B.A. Sc. and in 1984 with a M.A. Sc., both in Geological Engineering.
3) I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia.
4) I have practiced my profession continuously since 1970. I have had over 30 years experience calculating mineral resources. I have previously completed resource estimations on a wide variety of porphyry Cu deposits, including Red Chris, Copper Mt., Prosperity, Schaft Cr., Zaldivar and Kemess South.
5) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in National Instrument 43-101.
6) This report titled "**Technical Report, Casino Project Pre-Feasibility Study, Yukon Territory, Canada**" dated August 5, 2008, is based on a study of the data and literature available on the Casion Property. I am responsible for the Mineral Resource Estimate section. I have not visited the property.
7) I have previously worked on this property in 1995 producing a resource estimation for Pacific Sentinel Gold Corp.
8) As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
9) I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.
10) I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 5th day of August, 2008

G H Giroux

G. H. Giroux, P.Eng., MASc.

PROFESSIONAL
PROVINCE OF
G. H. GIROUX
BRITISH

TO: Western Copper Corporation
British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Securities Commission
Manitoba Securities Commission
Ontario Securities Commission
Authorité des marchés financiers du Québec
Nova Scotia Securities Commission
New Brunswick Office of the Administrator of Securities
PEI Provincial Affairs & Attorney General
Department of Government Services and Lands, Government of Newfoundland and Labrador
The Toronto Stock Exchange

I, Jerry Hanks, do hereby consent to the public filing by Western Copper Corporation (the "Company") of the technical report entitled "Technical Report, Casino Project Pre-Feasibility Study, Yukon Territory, Canada", dated August 5, 2008 (the "Technical Report") prepared for the Company by M3 Engineering & Technology Corporation, and a summary of the Technical Report contained in the news release (the "News Release") of the Company dated June 24, 2008 relating to the subject matter of the Technical Report, with the above securities regulatory authorities.

I confirm that I have read the written disclosure in the News Release and the News Release fairly and accurately represents the information in the Technical Report.

Dated as of the 5th day of August, 2008

Registered Professional Engineer (Metallurgical)
CERTIFICATE NO.
21106
JERRY T.
HANKS
Date signed 8-5-08
ARIZONA, U.S.A.
Expires 3-31-09

____
Signature of Qualified Person

Jerry Hanks
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Jerry T. Hanks, P.E.

As a co-author of this report on certain mineral properties of Western Copper Corp., in the Yukon Territory of Canada, I, Jerry Hanks of 7307 West Mesquite River Drive, Tucson AZ 85743, a self-employed Metallurgical / Mineral Processing Engineer doing business as "Jerry T. Hanks, P.E." do hereby certify that:

1. I have carried out this assignment for M3 Engineering & Technology Corporation of Tucson AZ.
2. This certificate applies to the Technical Report titled "Technical Report, Casino Project Pre-Feasibility Study, Yukon Territory, Canada", dated August 5, 2008;
3. I hold the following academic qualifications: Graduated the Colorado School of Mines with the degree of Metallurgical Engineer (1063).
4. I am a Registered Professional Engineer in Colorado (P.E. No. 10042) Arizona (Metallurgical, Reg. No. 21106. I am also a member in good standing of the Society of Mining Engineers (SME).
5. I have worked in the minerals industry as an engineer continuously since 1963, a period of 45 years,
6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfill the requirements of a Qualified Person as defined in NI 43-101.
7. I have read NI 43-101 and this technical report has been prepared in compliance with the instrument;
8. I am responsible for Section 1.18 of the report.
9. I am independent of the parties involved in the transaction for which this report is required using the tests in section 1.4 of NI 43-101;
10. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make this technical report not misleading;
11. I consent to the filing of this report with any Canadian stock exchange or securities regulatory authority, and any publication by them of the report.

Dated this 5th day of August, 2008

Registered Professional Engineer (Metallurgical)
CERTIFICATE NO.
21106
JERRY T.
HANKS
Date signed 8-5-08
ARIZONA, U.S.A.

Jerry T. Hanks

Expire 3-31-2009

TO: Western Copper Corporation
British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Securities Commission
Manitoba Securities Commission
Ontario Securities Commission
Authorité des marchés financiers du Québec
Nova Scotia Securities Commission
New Brunswick Office of the Administrator of Securities
PEI Provincial Affairs & Attorney General
Department of Government Services and Lands, Government of Newfoundland and Labrador
The Toronto Stock Exchange

I, Michael G. Hester, do hereby consent to the public filing by Western Copper Corporation (the "Company") of the technical report entitled "Technical Report, Casino Project Pre-Feasibility Study, Yukon Territory, Canada", dated 5 Aug, 2008 (the "Technical Report") prepared for the Company by M3 Engineering & Technology Corp., and a summary of the Technical Report contained in the news release (the "News Release") of the Company dated June 24, 2008 relating to the subject matter of the Technical Report, with the above securities regulatory authorities.

I confirm that I have read the written disclosure in the News Release and the News Release fairly and accurately represents the information in the Technical Report.

Dated as of the 5th of August, 2008.

____
Signature of Qualified Person

Michael G. Hester
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Michael G. Hester

As a co-author of this report on certain mineral properties of Western Copper Corp., in the Yukon Territory of Canada, I, Michael G. Hester, Vice President and Principal Mining Engineer of Independent Mining Consultants, Inc. (IMC) of 3560 E. Gas Road, Tucson, Arizona, 85714, USA, phone number (520) 294-9861, do hereby certify that:

1. This certificate applies to the Technical Report titled "Technical Report, Casino Project Pre-Feasibility Study, Yukon Territory, Canada", dated August 5, 2008;

2. I hold the following academic qualifications:

B.S. (Mining Engineering)	University of Arizona	1979
M.S. (Mining Engineering)	University of Arizona	1982

3. I am a Fellow of the Australian Institute of Mining and Metallurgy (FAusIMM #221108), a professional association as defined by National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101". As well, I am a member in good standing of the following technical associations and societies:

   Society for Mining, Metallurgy, and Exploration, Inc. (SME Member #1423200)
   The Canadian Institute of Mining, Metallurgy and Petroleum (CIM Member # 100809)

4. I have worked in the minerals industry as an engineer continuously since 1979, a period of 29 years;

5. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfill the requirements of a Qualified Person as defined in NI 43-101. I am a founding partner, Vice President, and Principal Mining Engineer for Independent Mining Consultants, Inc. (IMC), a position I have held since 1983. I have also been employed as an Adjunct Lecturer at the University of Arizona (1997-1998) where I taught classes in open pit mine planning and mine economic analysis. I was also employed as a staff engineer for Pincock, Allen & Holt, Inc. from 1979 to 1983.

6. I have read NI 43-101 and this technical report has been prepared in compliance with the instrument;

7. I am responsible for the mineral reserve statement presented in this report and the development of the mine plan on which it was based (Sections 1.19.9 and 1.25.1). I also developed the mine capital and operating costs presented in the study (portions of Sections 1.25.5 and 1.25.6). I visited the Casino site for one day on July 22, 2008 to review conditions at the site. I also developed mine plans and cost estimates for the project for a scoping study done by Pacific Sentinel in 1995.

8. I am independent of the parties involved in the transaction for which this report is required using the tests in section 1.4 of NI 43-101;

9. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make this technical report not misleading;

10. I consent to the filing of this report with any Canadian stock exchange or securities regulatory authority, and any publication by them of the report.

    Dated this 5th day of August, 2008

    "Michael G. Hester"

TO: Western Copper Corporation
British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Securities Commission
Manitoba Securities Commission
Ontario Securities Commission
Authorité des marchés financiers du Québec
Nova Scotia Securities Commission
New Brunswick Office of the Administrator of Securities
PEI Provincial Affairs & Attorney General
Department of Government Services and Lands, Government of Newfoundland and Labrador
The Toronto Stock Exchange

I, Timothy S. Oliver, do hereby consent to the public filing by Western Copper Corporation (the "Company") of the technical report entitled "Technical Report, Casino Project Pre-Feasibility Study, Yukon Territory, Canada", dated August 5, 2008 (the "Technical Report") prepared for the Company by M3 Engineering & Technology Corp., and a summary of the Technical Report contained in the news release (the "News Release") of the Company dated June 24, 2008 relating to the subject matter of the Technical Report, with the above securities regulatory authorities.

I confirm that I have read the written disclosure in the News Release and the News Release fairly and accurately represents the information in the Technical Report.

Dated as of the 5th day of August, 2008.

Signature of Qualified Person

Timothy S. Oliver
Print Name of Qualified Person

PROFESSIONAL ENGINEER ALBERTA
TIMOTHY S. OLIVER
LICENSED TO PRACTICE

CERTIFICATE OF QUALIFIED PERSON

Timothy S. Oliver

As a co-author of this report on certain mineral properties of Western Copper Corp., in the Yukon Territory of Canada, I, Timothy S. Oliver of 1856 N Cheyenne Trail, Tucson, Arizona 85750, **Environmental Engineer,** do hereby certify that:

1. I am employed by, and carried out this assignment for M3 Engineering and Technology Inc., 2440 W. Ruthrauff Road, Tucson, Arizona 85705, USA, phone number (520) 293-1488;

2. This certificate applies to the Technical Report titled "Technical Report, Casino Project Pre-Feasibility Study, Yukon Territory, Canada", dated August 5, 2008;

3. I hold the following academic qualifications:

   Bachelor of Science, Environmental Engineering, New Mexico Institute of Mining and Technology, 1976

4. I am and have been a member of the Society of Mining Engineers of AIME since 1980. I am and have been a member of the Association of Professional Engineers, Geologists, and Geophysicists of Alberta since 2008.

5. I have worked in the minerals industry as an engineer continuously since **1977,** a period of **31** years;

6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration; I fulfill the requirements of a Qualified Person as defined in NI 43-101.

7. I have read NI 43-101 and this technical report has been prepared in compliance with the instrument;

8. I am responsible for the **overall preparation of the report titled** "Technical Report, Casino Project Pre-Feasibility Study, Yukon Territory, Canada", dated August 5, 2008.

9. I am independent of the parties involved in the transaction for which this report is required using the tests in section 1.4 of NI 43-101;

10. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make this technical report not misleading;

11. I consent to the filing of this report with any Canadian stock exchange or securities regulatory authority, and any publication by them of the report.

Dated this 5th day of August, 2008

Timothy S. Oliver, P. Eng.

PROFESSIONAL ENGINEER ALBERTA
8/5/2008
TIMOTHY S.
OLIVER
LICENSED TO PRACTICE

TO: Western Copper Corporation
British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Securities Commission
Manitoba Securities Commission
Ontario Securities Commission
Authorité des marchés financiers du Québec
Nova Scotia Securities Commission
New Brunswick Office of the Administrator of Securities
PEI Provincial Affairs & Attorney General
Department of Government Services and Lands, Government of Newfoundland and Labrador
The Toronto Stock Exchange

I, Mark Rebagliati, do hereby consent to the public filing by Western Copper Corporation (the "Company") of the technical report entitled "Technical Report, Casino Project Pre-Feasibility Study, Yukon Territory, Canada" dated August 5th, 2008 (the "Technical Report") prepared for the Company by M3 Engineering & Technology Corp., and a summary of the Technical Report contained in the news release (the "News Release") of the Company dated June 24th, 2008 relating to the subject matter of the Technical Report, with the above securities regulatory authorities.

I confirm that I have read the written disclosure in the News Release and the News Release fairly and accurately represents the information in the Technical Report.

Dated as of the 5th day of August, 2008.

Signature of Qualified Person

C. M. REBAGLIATI
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Mark Rebagliati

As a co-author of this report on certain mineral properties of Western Copper Corp., in the Yukon Territory of Canada, I, Mark Rebagliati of 317 – 2200 Highbury Street, Vancouver, BC. V6R 4N8, Geological Engineer, do hereby certify that:

1. I am employed by, and carried out this assignment for Rebagliati Geological Consulting, Ltd., 317 – 2200 Highbury Street, Vancouver, BC. V6R 4N8 phone number (604) 662 – 7487;

2. I hold the following academic qualifications:

   Bachelor of Science, Geological Engineering. Michigan Technological University 1969.

3. I am a member of Association of Professional Engineers and Geoscientist of British Columbia; Canadian Institute of Mining and Metallurgy; Society of Economic Geology; American Society for Mining, Metallurgy and Exploration.

4. I have worked in the minerals industry as an engineer continuously since 1969, a period of thirty nine years.

5. I am familiar with NI 43 – 101 and, by reason of education, experience and professional registration; I fulfill the requirements of a Qualified Person as defined in NI 43 – 101.

6. I have read NI 43 – 101 and this technical report has been prepared in compliance with the instrument;

7. I am responsible for the sections on Geological Setting, Deposit Types and Mineralization presented in this repot.

8. I am independent of the parties involved in the transaction for which this report is required using the test in section 1.4 of NI 43 – 101.

9. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make this technical report not misleading;

10. I consent to the filing of this report with any Canadian stock exchange or securities regulatory authority, and any publication by them of the report.

Dated this 5th day of August, 2008

Mark Rebagliati