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Casino Project

Form 43-101F1 Technical Report

Feasibility Study

Yukon, Canada

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DATE AND SIGNATURES PAGE

This report is current as of 25 January 2013.

"Conrad Huss, P.E."	25 January 2013
Signature	Date

The Certificates of Qualified Persons are included in Appendix A.

CASINO PROJECT
FORM 43-101F1 TECHNICAL REPORT
FEASIBILITY STUDY

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1 SUMMARY

This report has been prepared by M3 Engineering & Technology Corporation (M3) of Tucson, Arizona to summarize the work performed in the preparation of a feasibility study, supplementary to the Canadian Standard NI 43-101, previously issued for the development of the Casino property (the "Property") in the Yukon Territory in northern Canada for Casino Mining Corporation ("CMC"), a 100% owned subsidiary of Western Copper and Gold Corporation ("WCGC").

1.1 KEY DATA

The key details about this project are as follows:

1. Casino is primarily a copper and gold project that is expected to process 120,000 dry tonnes of material per day (t/d) or 43.8 million dry tonnes per year (t/y). Metals to be recovered are copper, gold, molybdenum and silver.

2. There are a total of 965 million tonnes of proven and probable mill ore reserves and 157 million tonnes of proven and probable heap leach reserves. Based on the economic analysis, the Property will produce the following over the life of the mine from heap leach and flotation:

    a. Gold – 5.72 million ounces
    b. Silver – 30.26 million ounces
    c. Copper – 3.58 billion pounds
    d. Molybdenum – 325 million pounds

3. The process will include a conventional single-line SAG mill circuit (Semi-Autogenous Ball Mill Crushing, or SABC) followed by conventional flotation to produce concentrate for sale. In addition to the concentrator, there will be a separate carbon-in-column facility to recover precious metals from oxide ore. Gold and silver bullion (doré) produced will be shipped by truck to metal refineries.

4. The Property will require the construction of a power plant and will generate its own electrical power using LNG to fuel the generator drivers. Additionally, the mine haulage vehicles and over-the-highway tractors which haul concentrates and LNG will utilize LNG as fuel.

5. The Property has several routes of access, including by the Yukon River, by aircraft, winter roads, and existing trails. A network of paved highways provides access to the region from the Port of Skagway, Whitehorse and northern British Columbia. Paved roads to the Property currently exist up to Carmacks. A new, all weather, gravel road will be constructed by the project to connect Casino to Carmacks via the existing Freegold Road. The new access road will, in general, follow the existing Casino Trail that will be upgraded to support trucking from Carmacks to Casino.

6. Fresh water will be sourced from the Yukon River.

7. The major milestones in the schedule are as follows:

    a. Full Notice to Proceed and construction start-up – first quarter 2016
    b. Heap Leach operation start-up – fourth quarter 2017
    c. Mill operation start-up – fourth quarter 2019
    d. Commercial Production – first quarter 2020

1.2 PROPERTY DESCRIPTION AND OWNERSHIP

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 northwest trending Dawson Range mountains, 300 km northwest of the territorial capital of Whitehorse.

The area around Casino has been subject to increasing staking and exploration activity over the past few years. Two properties have defined reserves, the Carmacks Copper Project and the Minto Mine, both of which are discussed in Section 23, Adjacent Properties.

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. The proposed access road crosses into Little Salmon Carmacks First Nation traditional territory to the south.

The Casino Property lies within the Whitehorse Mining District and consists of 723 full and partial Quartz Claims and 85 Placer Leases acquired in accordance with the Yukon Quartz Mining Act. The total area covered by Casino Quartz Claims is 13,339.17 ha. The total area covered by Casino Placer Leases is 747.45 ha. CMC is the registered owner of all claims, although certain portions of the Casino property remain subject to royalty agreements. The claims covering the Casino property are discussed further in Section 4 of this document.

Figure 1-1 at the end of this section shows the site's location in Yukon Territory as well as other points of interest relevant to this Report. Figure 1-2 and Figure 1-3 show the roadway paths from the Yukon River to the proposed site facilities.

1.3 GEOLOGY

The geology of the Casino deposit is typical of many porphyry copper deposits. The deposit is centered on an Upper Cretaceous-age (72-74 Ma), east-west elongated porphyry stock, the Patton Porphyry, which intrudes Mesozoic granitoids of the Dawson Range Batholith and Paleozoic schists and gneisses of the Yukon Crystalline Complex. Intrusion of the Patton Porphyry into the older rocks caused brecciation of both the intrusive and the surrounding country rocks along the northern, southern and eastern contact of the stock. Brecciation is best developed in the eastern end of the stock where the breccia can be up to 400 metres wide in plan view. To the west, along the north and south contact, the breccias narrow gradually to less than 100 metres. The overall dimensions of the intrusive complex are approximately 1.8 by 1.0 kilometres.

The main body of the Patton Porphyry is a relatively small, locally mineralized, stock measuring approximately 300 by 800 metres and is surrounded by a potassically-altered Intrusion Breccia in contact with rocks of the Dawson Range. Elsewhere, the Patton Porphyry forms discontinuous dikes ranging from less than one to tens of metres wide, cutting both the Patton Porphyry Plug and the Dawson Range Batholith. The overall composition of the Patton Porphyry is rhyodacite, with phenocrysts falling into a dacite composition and the matrix being of quartz latite composition. It is more commonly made up of distinct phenocrysts of abundant plagioclase and lesser biotite, hornblende, quartz and opaques.

The Intrusion Breccia surrounding the main Patton Porphyry body consists of granodiorite, diorite, and metamorphic fragments in a fine-grained Patton Porphyry matrix. It may have formed along the margins, in part, by the stoping of blocks of wall rocks. An abundance of Dawson Range inclusions are prominent at the southern contact of the main plug, Wolverine Creek metamorphic rocks increase along the northern contact, and bleached diorite increases at the eastern contact of the main plug. Strong potassic alteration locally destroys primary textures.

1.4 MINERALIZATION

Primary copper, gold and molybdenum mineralization was deposited from hydrothermal fluids that exploited the contact breccias and fractured wall rocks. Better grades occur in the breccias and gradually decrease outwards away from the contact zone both towards the centre of the stock and outward into the granitoids and schists. The main mineralization types are:

- **Leached Cap Mineralization (CAP) –** This oxide gold zone is gold-enriched and copper-depleted due to supergene alteration processes as well as the lower specific gravity of this zone relative to the other supergene zones. Weathering has replaced most minerals with clay. The weathering is most intense at the surface and decreases with depth.
- **Supergene Oxide Mineralization (SOX)** – This zone is copper-enriched, with trace molybdenite. It generally occurs as a thin layer above the Supergene Sulphide zone. Where present, the supergene oxide zone averages 10 metres thick, and can contain chalcanthite, malachite and brocanthite, with minor azurite, tenorite, cuprite and neotocite.
- **Supergene Sulphide Mineralization (SUS) –** Supergene copper mineralization occurs in an up to 200 metre-deep weathered zone below the leached cap and above the hypogene. It has an average thickness of 60 metres. Grades of the Supergene sulphide zone vary widely, but are highest in fractured and highly pyritic zones, due to their ability to promote leaching and chalcocite precipitation. The copper grades in the Supergene Sulphide zone are almost double the copper grades in the Hypogene (0.43% Cu versus 0.23% Cu).
- **Hypogene Mineralization –** Hypogene mineralization occurs throughout the various alteration zones of the Casino Porphyry deposit, as mineralized stock-work veins and breccias. Significant Cu-Mo mineralization is related to the potassically-altered breccia surrounding the core Patton Porphyry, as well as in the adjacent phyllically-altered host rocks of the Dawson Range Batholith. The pyrite halo in this mineralization is host to the highest Cu values on the property.

Native gold can occur as free grains in quartz (50 to 70 microns) and as inclusions in pyrite and/or chalcopyrite grains (1 to 15 microns). High grade smoky quartz veins with numerous specks of visible gold are also reported to exist.

1.5 EXPLORATION STATUS

In 2009, Quantec Geoscience Limited of Toronto, Ontario performed Titan-24 Galvanic Direct Current Resistivity and Induced Polarization (DC/IP) surveys as well as a Magnetotelluric Tensor Resistivity (MT) survey over the entire grid. Magnetotelluric Resistivity results in high resolution and deep penetration (to 1 km) and the Titan DC Resistivity & Induced polarization provides reasonable depth coverage to 750 m.

In 2010, all Pacific Sentinel's historic drill core stored at the Casino Property was re-logged. The purpose of the re-logging was to provide data for the new lithology and new alteration models.

In 2011 and 2012, WCGC focused on geotechnical, metallurgical and baseline environmental studies, however certain of the drill holes were also drilled, logged and sampled for exploration purposes. In 2011, the program involved 41 drill holes for a total of 3,163.26 m. In 2012, six holes (228.07 m) were drilled for geotechnical purposes and 5 holes (1,507.63 m) were drilled for metallurgical sampling.

1.6 DEVELOPMENT AND OPERATIONS

1.6.1 Mining and Processing

A mine plan was developed to supply ore to a conventional copper sulphide flotation plant with the capacity to process ore at a nominal rate of 120,000 tonnes per day, or 43.8 million tonnes per year. Actual annual throughput will vary depending on the ore hardness encountered during the period. The mine is scheduled to operate two 12 hour shifts per day, 365 days per year.

Both sulphide copper-molybdenum ore and oxide gold ore will be processed. Copper-molybdenum ore will be transported from the mine to the concentrator facility and oxide gold ore will be transported from the mine to a crushing facility ahead of a heap leaching facility and a gold recovery facility.

Copper-molybdenum ore will be processed by crushing, grinding, and flotation to produce copper and molybdenum sulphide mineral concentrates. Copper concentrate will be loaded into highway haul trucks and transported to the Port of Skagway for ocean shipment to market. Molybdenum concentrate will be bagged and loaded onto highway haul trucks for shipment to market.

Oxide gold ore will be leached with an aqueous leach solution. Gold in the enriched (or pregnant) leach solution which will extracted by using carbon absorption technology to produce gold doré bars. The enriched leach solution will also be treated to recover copper and cyanide and produce a copper sulphide precipitate. The copper sulphide precipitate will be bagged and loaded onto highway haul trucks for shipment to market. Recovery methods are discussed more in depth in Section 17.

1.6.2 Metallurgical Testing and Metal Recoveries

Recent test work 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. This process has been adopted for this feasibility study.

Flotation testing by G&T Metallurgical from 2008 to 2012 indicated that copper concentrate grades of 28% copper could be routinely achieved at good copper recoveries with a primary grind size of 80% passing 200 µm and a regrind of 80% passing 25 µm. Gold and silver will be recovered with the copper concentrate. Molybdenum will be recovered to a molybdenum concentrate in a separate flotation circuit.

The average metal recoveries expected from mill processing following the planned mill feed schedule are noted below:

- Copper recovery to copper concentrate, percent 86
- Gold recovery to copper concentrate, percent 67
- Silver recovery to copper concentrate, percent 53
- Molybdenum recovery to molybdenum concentrate, percent 71

The metal recoveries expected from oxide cap heap leach processing are based on:

- Gold recovery, percent 66
- Silver recovery, percent 26
- Copper recovery to SART precipitate, percent 18

1.6.3 Infrastructure

The region is serviced by paved all-weather roads connecting the towns of Carmacks and Whitehorse in the Yukon with the Port of Skagway Alaska. With the completion of the 132 km Casino access road, the project will have an all-weather access route through Carmacks to Whitehorse (approx. 380 km) and to the Port of Skagway (550 km). The Port of Skagway has existing facilities to store and load-out concentrates as well as facilities to receive bulk commodity shipments, fuels and connection to the Alaska Marine Highway. The Port of Skagway is developing plans to expand these facilities to better serve the expanding mining activity in the Yukon and Alaska.

The City of Whitehorse is the government, financial and commercial hub of the Yukon with numerous business and service entities to support the project, and represents a major resource to staff the project. Whitehorse has an international airport and provides commercial passenger and freight services for the region. The proposed new access road alignment is shown in Section 18.2 of this report.

A new airstrip will be constructed at the mine to accommodate appropriately sized aircraft. The existing airstrip will be razed in preparation for grading for process facilities.

1.6.3.1 Power

Electrical power to the mine will be developed in two phases. An initial power plant designated the Supplementary Power Plant will be constructed near the main workforce housing complex to provide power to the camp, to construction activities, and to the heap leach facilities that go into operation before the concentrator facility is operational.

The Supplementary Power Plant will consist of three internal combustion engines (ICE), dual fuel driven generators (capable of using both LNG and diesel) with a combined power output of 20.1 MW.

A Main Power Plant will be constructed at the Casino concentrator complex and will supply the electrical energy required for operation of the mine, concentrator, oxide ore treatment facilities and all infrastructure facilities. The primary electrical power generation will be provided by two gas turbine driven generators and a steam generator, operating in combined cycle mode (CCGT) to nominally produce 125 MW. Two internal combustion engine (ICE) driven generators will provide another 18.6 MW of power for black start capability, emergency power, and to complement the gas turbine generation when required. The gas turbines and the ICEs will be fueled by natural gas (supplied as liquefied natural gas, or LNG).

LNG will be transported to the site from Fort Nelson, British Columbia via tanker trucks and stored on-site in a large, 10,000 m^3 site-fabricated storage tank to supply the power plant. In addition to providing fuel for the power plants, the LNG facility will provide fuel for the mine haulage fleet and over the highway tractors used for hauling concentrates and LNG tankers. Distribution to the vehicles will be by two portable fueling stations and two mobile refuelers.

1.6.3.2 Water

The main fresh water supply will be supplied from the Yukon River. The water will be collected in a riverbank caisson and radial well system (Ranney Well) and pumped through an above-ground insulated 36” diameter by 17.4 km long pipeline with four booster stations to the 22,000 m^3 capacity freshwater pond near the concentrator. The design capacity of the fresh water collection and transfer system will be approximately 3,400 cubic meters per hour.

The fire water requirement is 341 m^3/h for two hours. This demand is satisfied by a fire reserve capacity of 682 m^3 in the lower portion of the freshwater pond that will be unavailable for other uses.

Potable water will be produced by filtering and chlorinating fresh water and will be stored and distributed separately.

Process water reclaimed from the tailings pond and from the plant will be collected in a 63,700 m^3 process water pond. The total process water supply to the plant required at design tonnage is 11,191 m^3/h. The total feed to the pond consists of 7,122 m^3/h combined thickener overflow, 3,228 m^3/h reclaim water from the tailings pond and 841 m^3/h fresh water makeup. Assuming the reclaim water system is not operating the plant can run for 19.7 hours with all other feed streams operating at the design rates.

1.6.4 Permits

The *Yukon Socioeconomic Assessment Board* (YESAB) will assess the Casino Project for environmental and socio-economic effects under the *Yukon Environmental and Socioeconomic Assessment Act* (YESAA). At the conclusion of the assessment, YESAB issues its recommendations to a decision body for consideration. For the Casino Project, it is expected that the designated decision body will be the Yukon Government.

After assessment, the project must secure certain permits and licenses. Mining related projects in the Yukon are regulated through territorial and federal legislation by various agencies, including government departments and independent quasi-judicial boards. The regulatory permitting and licensing processes are separate from YESAA. There is no single-window application process for regulatory permitting. Separate applications and information packages are required for each authorization and agency. The main pieces of legislation that will govern mining related activities for the Casino properties include:

- *Quartz Mining Act;*
- *Waters Act;*
- *Territorial Lands (Yukon) Act;*
- *Environment Act;*
- *Highways Act;*
- *Dangerous Goods Transportation Act;*
- *Fisheries Act;*
- *Yukon Occupational Health and Safety Act.*

Work on these permits is in progress. Environmental study results are detailed in Section 20 of this report.

1.7 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

1.7.1 Metal Pricing

Table 1-1 shows a summary of metal pricing that has been used for this report. Long-term metal prices used in the financial model were converted from US dollars using a long term exchange rate of C$:US$ = 0.95 and are shown in Canadian dollars.

Table 1-1: Summary of Metal Pricing

	Copper	Gold	Molybdenum	Silver
Resources	$2.00/lb	$875.00/oz	$11.25/lb	$11.25/oz
Reserves	$2.75/lb	$1,300.00/oz	$14.50/lb	$23.00/oz
Financial Model	$3.16/lb	$1,473.68/oz	$14.74/lb	$26.32/oz

1.7.2 Mineral Resource

Table 1-2 summarizes the mineral resources for the Casino Project.

Table 1-2: Mineral Resource-Inclusive of Mineral Reserve

Supergene and Hypogene Zones (Mill Resource)	Cutoff CuEq (%)	Ore Mtonnes	Copper (%)	Gold (g/t)	Moly (%)	Silver (g/t)	CuEq (%)
Measured Mineral Resource							
Supergene Oxide	0.25	25	0.28	0.52	0.026	2.38	0.77
Supergene Sulphide	0.25	36	0.39	0.41	0.029	2.34	0.84
Hypogene	0.25	32	0.32	0.38	0.026	1.94	0.73
Total Measured Resource	**0.25**	**93**	**0.34**	**0.43**	**0.027**	**2.21**	**0.78**
Indicated Mineral Resource							
Supergene Oxide	0.25	36	0.23	0.21	0.019	1.44	0.46
Supergene Sulphide	0.25	216	0.24	0.22	0.019	1.72	0.50
Hypogene	0.25	711	0.17	0.21	0.023	1.65	0.45
Total Indicated Resource	**0.25**	**963.6**	**0.19**	**0.21**	**0.022**	**1.66**	**0.46**
Measured/Indicated Mineral Resource							
Supergene Oxide	0.25	61	0.25	0.34	0.022	1.83	0.59
Supergene Sulphide	0.25	252	0.26	0.25	0.021	1.81	0.55
Hypogene	0.25	743	0.17	0.22	0.023	1.66	0.46
Total Measured/Indicated Resource	**0.25**	**1057**	**0.20**	**0.23**	**0.022**	**1.71**	**0.49**
Inferred Mineral Resource							
Supergene Oxide	0.25	26	0.26	0.17	0.010	1.43	0.41
Supergene Sulphide	0.25	102	0.20	0.19	0.010	1.49	0.38
Hypogene	0.25	1568	0.14	0.16	0.020	1.36	0.37
Total Inferred Resource	**0.25**	**1696**	**0.14**	**0.16**	**0.019**	**1.37**	**0.37**
Leached Cap/Oxide Gold Zone (Heap Leach Resource)	Cutoff Gold (g/t)	Ore Mtonnes	Copper (%)	Gold (g/t)	Moly (%)	Silver (g/t)	CuEq (%)
Measured Mineral Resource	0.25	31	0.05	0.52	0.025	2.94	N.A.
Indicated Mineral Resource	0.25	53	0.03	0.33	0.017	2.36	N.A.
Measured/Indicated Resource	**0.25**	**84**	**0.04**	**0.40**	**0.020**	**2.57**	N.A.
Inferred Mineral Resource	0.25	17	0.01	0.31	0.008	1.93	N.A.
CuEq is based on metal prices of US$2.00/lb copper, $US875/oz gold, US$11.25/lb molybdenum, and US$11.25/oz silver and assumes 100% metal recovery.							

The supergene oxide, supergene sulphide, and hypogene zones are mill resources and are tabulated at a 0.25% copper equivalent cutoff grade. Measured and indicated supergene and hypogene resources amount to 1.06 billion tonnes at 0.20% copper, 0.23 g/t gold, 0.022% molybdenum, and 1.71 g/t silver. Inferred resources are an additional 1.7 billion tonnes at 0.14% copper, 0.16 g/t gold, 0.019% molybdenum, and 1.37 g/t silver.

The leach cap contains potential heap leach ore and is tabulated at a 0.25 g/t gold cutoff grade. Measured and indicated heap leach ore amounts to 84.0 million tonnes at 0.04% copper, 0.40 g/t gold, and 2.57 g/t silver. Inferred resources are an additional 17 million tonnes at 0.01% copper, 0.31 g/t gold, and 1.93 g/t silver.

1.7.3 Mineral Reserve

The mill ore reserve amounts to 965.2 million tonnes at 0.204% copper, 0.240 g/t gold, 0.0227% molybdenum, and 1.74 g/t silver. Heap leach reserve is an additional 157.5 million tonnes at 0.292 g/t gold and 0.036% copper. Table 1-3 presents the mineral reserve for the Casino Project.

Table 1-3: Mineral Reserve

Mill Ore Reserve:	Ore Ktonnes	Tot Cu (%)	Gold (g/t)	Moly (%)	Silver (g/t)
Proven Mineral Reserve:					
Mill Ore	91,602	0.336	0.437	0.0275	2.23
Probable Mineral Reserve:					
Mill Ore	729,777	0.203	0.235	0.0240	1.78
Low Grade Stockpile	143,828	0.122	0.139	0.0133	1.19
Total Probable Reserve	873,605	0.190	0.219	0.0222	1.68
Proven/Probable Reserve					
Mill Ore	821,379	0.218	0.258	0.0244	1.83
Low Grade Stockpile	143,828	0.122	0.139	0.0133	1.19
Total Mill Ore Reserve	965,207	0.204	0.240	0.0227	1.74
Heap Leach Reserve:	**Ore ktonnes**	**Gold (g/t)**	**Tot Cu (%)**	**Moly (%)**	**Silver (g/t)**
Proven Mineral Reserve	31,760	0.480	0.051	N/A	2.79
Probable Mineral Reserve	125,694	0.244	0.032	N/A	2.06
Total Heap Leach Reserve	157,454	0.292	0.036	N/A	2.21

1.8 COSTS AND FINANCIAL DATA

1.8.1 Capital Cost Estimate

The initial capital investment for complete development of the project is estimated to be $2.456 billion total direct and indirect cost. Table 1-4 shows the capital cost breakdown.

Table 1-4: Capital Cost Estimate Summary

	(millions)
Direct Costs	
Mining Equipment & Mine Development	$454
Concentrator (including related facilities)	$904
Heap Leach Operation	$139
Camp	$70
Sub-Total	**$1,566**
Indirect Costs	$295
Infrastructure Costs	
Power Plant	$209
Access Road	$99
Airstrip	$24
Subtotal Infrastructure	$332
Contingency	**$218**
Owner's Costs	$44
Grand Total	**$2,456**

In addition to the above, the total life of mine sustaining capital is estimated to be $361.7 million. This capital will be expended during a 22 year period.

1.8.2 Operating Cost Estimate

Life of mine average operating cost is $8.52 per tonne for sulphide ore, which includes mining, concentrator plant and general and administrative costs. The life of mine average operating cost is $4.04 per tonne for oxide ore which includes processing only.

1.8.3 Financial Analysis

Net Income after Tax amounts to $6.7 billion for the life of the mine. The base case economic analysis (Table 1-5) indicates that the project has an Internal Rate of Return (IRR) of 20.1% after taxes with a payback period of 3.0 years.

Table 1-5 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 metal sales price has the most impact, while variation of operating cost, variation of mill recovery, and variation of capital cost are approximately equal.

Table 1-5: Sensitivity Analysis (After tax figures)

	NPV @ 0%	NPV @ 5%	NPV @ 8%	NPV @ 10%	IRR	Payback Years
Base Case (LTP)	$6,651	$2,986	$1,830	$1,296	20.1%	3.0
SEC Prices	$7,848	$3,621	$2,287	$1,669	22.5%	2.7
Spot Prices*	$7,744	$3,597	$2,282	$1,672	22.7%	2.6
Base-Case Sensitivities						
Metals Price +10%	$8,157	$3,786	$2,407	$1,768	23.1%	2.6
Metals Price -10%	$5,146	$2,186	$1,253	$824	16.7%	3.5
Capex +10%	$6,499	$2,840	$1,689	$1,158	18.4%	3.2
Capex -10%	$6,804	$3,133	$1,972	$1,434	22.1%	2.7
Opex +10%	$6,103	$2,705	$1,631	$1,135	19.0%	3.1
Opex -10%	$7,200	$3,268	$2,029	$1,457	21.1%	2.9
Mill Recovery +5%	$7,304	$3,329	$2,075	$1,495	21.3%	2.8
Mill Recovery -5%	$5,998	$2,644	$1,585	$1,096	18.7%	3.1
$ in millions						

*Spot prices are on the last day of October 2012	Base Case		SEC Prices		Spot Prices*	
	Copper	$3.16	Copper	$3.67	Copper	$3.57
	Molybdenum	$14.74	Molybdenum	$14.67	Molybdenum	$11.80
	Gold	$1,473.68	Gold	$1.487.85	Gold	$1,657.50
	Silver	$26.32	Silver	$28.80	Silver	$29.95

1.9 CONCLUSIONS AND RECOMMENDATIONS

The Casino mineral occurrence can be successfully and economically exploited by proven and conventional mining and processing methods under the conditions and assumptions outlined in this report. Overall, the main risk and key metric for financial success is metal pricing.

Opportunities exist to enhance the project economics including:

- Conversion of some of the inferred resource into measured and indicated.
- Sharing of infrastructure development costs with other parties.
- Optimize the process during the basic and detailed engineering phases.

To further enhance the project, M3 recommends that CMC perform the following:

- CMC 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 (approximately $2 million required).

- CMC should continue with the environmental studies and permitting efforts now underway (approximately $5 million required).

- CMC should continue with the engineering effort in support of permitting (approximately $1 million required).

- CMC should continue to monitor developments in the Yukon, northern British Columbia, and Alaska to be in a position to share infrastructure development (approximately $200,000 required).

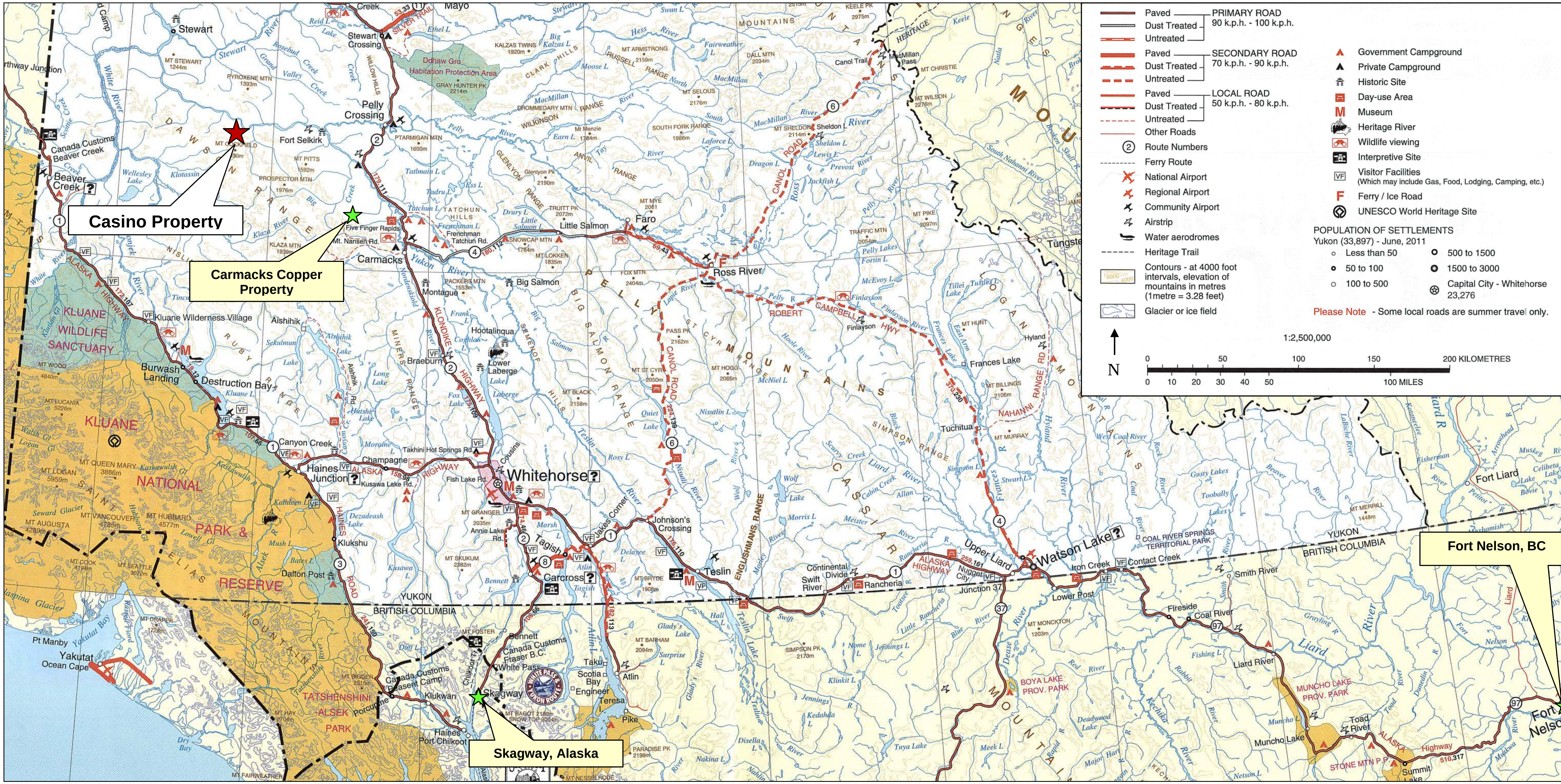


Figure 1-1: Casino Property Location

(Source: Yukon Highway Map, Yukoninfo.com)

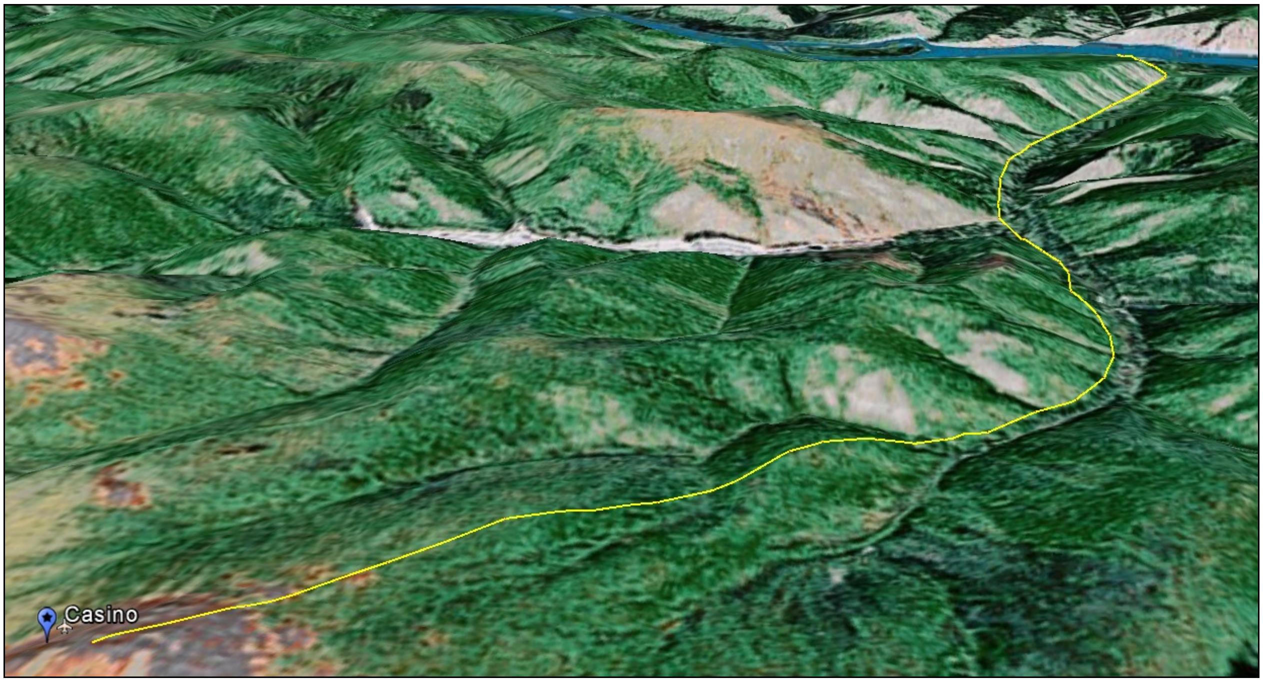


Figure 1-2: Road to Casino Concentrator Site from the Yukon River

(Source: M3)

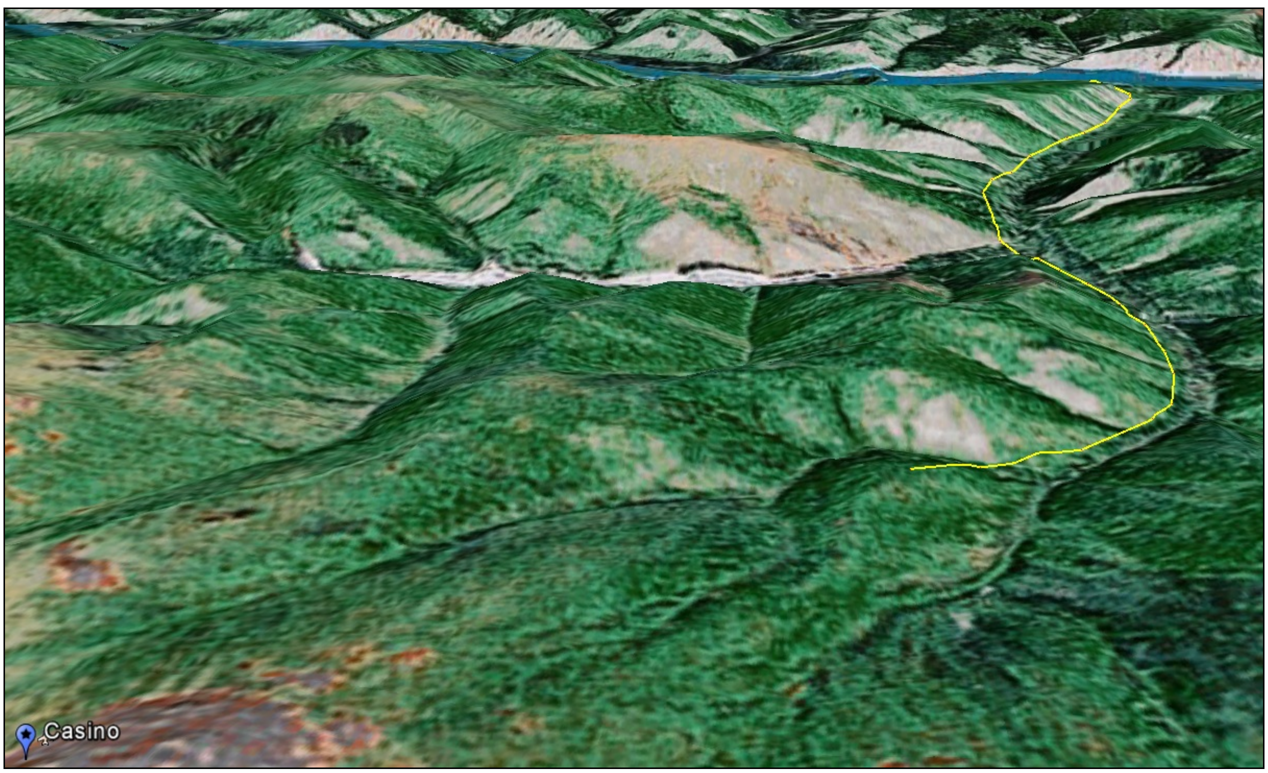


Figure 1-3: Path from Yukon River to Proposed Fresh Water Pond

(Source: M3)

2 INTRODUCTION

2.1 ISSUER AND PURPOSE OF ISSUE

This Report was prepared by M3 Engineering & Technology Corporation for Casino Mining Corporation ("CMC"), a wholly-owned subsidiary of Western Copper and Gold Corporation ("WCGC").

The purpose of this report is to provide an updated independent technical report and reserve estimate of the ore present on the Casino property. The estimate of mineral reserves contained in this report conforms to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Mineral Resource and Mineral Reserve definitions (November, 2010) referred to in National Instrument (NI) 43-101, Standards of Disclosure for Mineral Projects.

M3 provided engineering services to design a process plant, ancillary, and support facilities to bring the Casino Project into full production. This report supersedes the "Casino Project Pre-Feasibility Study Update, Yukon Territory, Canada" M3 report, dated April 29th, 2011.

Based on both in-house designs and contributions by others consultants, M3 prepared capital and operating cost estimates and performed an economic analysis to assess the economic viability of the project.

2.2 SOURCES OF INFORMATION

This Report builds upon prior data, including the following reports:

- 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."

- M3, 2008. "Casino Project Pre-Feasibility Study, Yukon Territory, Canada." Prepared by M3 Engineering & Technology Corporation.

- M3, 2011. "Casino Project Pre-Feasibility Study Update, Yukon Territory, Canada". Prepared by M3 Engineering & Technology Corporation.

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 as well as reports supplied by WCGC and CMC area are listed in references section.

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

- Associated Engineering ("AE"): Main access road and port facility capital and operating quantities and 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 and operating costs.

A summary of the Qualified Persons ("QPs") responsible for the content of this report is shown in Table 2-1.

Table 2-1: Dates of Site Visits and Areas of Responsibility

QP Name	Company	Qualification	Site Visit Date	Area of Responsibility
Conrad Huss	M3 Engineering & Technology Corporation – Tucson, AZ	P.E.	12-Jun-2012	Sections 1, 2, 3, 4, 5, 6, 13, 17, 18, 19, 21, 22, 23, 24, 25, 26 & 27.
Tom Drielick	M3 Engineering & Technology Corporation – Tucson, AZ	P.E.	N/A	Sections 13 and 17.
Jeff Austin	International Metallurgical and Environmental Inc.	P. Eng.	N/A	Section 13.
Gary Giroux	Giroux Consultants Ltd.	P.Eng.	N/A	Section 14.
Scott Casselman	Casselman Geological Services Ltd.	P. Geo	2008-2010	Sections 4, 6, 7, 8, 9, 10, 11, 12, and 14.
Graham Greenaway	Knight-Piésold Consulting	P. Eng.	N/A	Section 18.
Michael G. Hester	Independent Mining Consultants, Inc.	F Aus IMM	22-Jul-2008	Sections 15 and 16.
Jesse Duke	Ibex Valley Environmental Consulting, Inc.	P. Geo	9-May-2008, 2009, 2010	Section 20.

2.3 PERSONAL INSPECTIONS

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

- 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.

- Jesse Duke, Ibex Valley Environmental Consulting, first inspected the site on May 9, 2008, and has made several visits in 2009 and 2010.

- Ray Korpela, Associated Engineering Group, Ltd., performed a helicopter oversight of the proposed transportation routes and walked several of the sections on September 25th and 26th, 2007.

- Scott Casselman, Casselman Geological Services Ltd., was a Site Project Manager at Casino for the 2008, 2009 and 2010 exploration programs.

- Conrad Huss, M3 Engineering & Technology Corporation, current Project Manager for the Casino Project, 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 his trip to site on June 12, 2012.

2.4 UNITS

This report generally uses the SI (metric) system of units, including metric tonnes. The term “tonne” rather than “ton” is commonly used to denote a metric ton, and is used throughout the report. Units used and abbreviations are listed in Table 2-2.

Table 2-2: 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
Hypogene	HYP
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
Metric Tonne (1000 kg)	Tonne
Millimeters	mm
Overburden	OVB
Oxide Gold/Leached Cap	CAP
Parts per million	ppm
Specific gravity	S.G.
Square meters	m^2
Supergene sulphide	SUS
Supergene oxide	SOX
Temperature Celsius	°C
Temperature Fahrenheit	°F
Tonnage factor or specific volume	m^3/tonne
Tonnes per day	t/d
Tonnes per year	t/y
Volts	V
Watts	W

3 RELIANCE ON OTHER EXPERTS

In cases where the study authors have relied on contributions of other qualified persons, 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 CMC and M3. 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.

M3 relied upon WCGC for project ownership data. M3 did not verify ownership or any underlying agreements. Mining is a risky business. The risk must be borne by the Owner. M3 does not assume any liability other than performing this technical study to normal professional standards.

The following sections describe additional information that this report relies upon beyond that which was provided by the QPs listed in Section 2.2.

3.1 METALLURGY AND PROCESS ENGINEERING

Outside reports that were relied upon included the following:

- ALS Metallurgy (formally G&T Metallurgical Services) of Kamloops, BC, performed numerous metallurgical testing to advance the flotation process design. Tom Shouldice is the official contact. International Metallurgical and Environmental and CMC managed and oversaw this work with input from FLSmidth.

- Starkey and Associates of Oakville, Ontario, performed a grinding circuit study. John Starkey is the official contact for Starkey and Associates. CMC managed and oversaw this work with input from FLSmidth.

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

- METCON Research of Tucson, AZ, USA, performed metallurgical testing to advance design of the gold heap leach. Rodrigo Carneiro is the official contact for METCON Research. CMC managed and oversaw METCON's work.

M3 staff and consultants reviewed and evaluated metallurgical testing results from the tests listed above. In addition to supervising the effort, M3's Tom Drielick also reviewed and approved design criteria, flow sheets and equipment lists for the metallurgical processes.

3.2 TRANSPORTATION

Associated Engineering (B.C.) Ltd. assisted by Lauga & Associates Consulting, Ltd. performed updates of the studies of transportation options including selection and design of the access road route. Associated Engineers and Lauga also prepared a report on port facility options. Ray Korpela is the official contact for Associated Engineers, and Tom Lauga, P. Eng. is the official contact for Lauga and Associates.

The transportation costs for concentrates and bulk commodities used in the estimate are based on information from Trimac. The concentrate storage and load out was based on criteria developed by Alaska Industrial Development Authority (AIDA). M3 performed the research for and developed costs for handling and freight.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Casino porphyry copper-gold-molybdenum deposit is located at latitude 62° 44'N and longitude 138° 50'W (NTS map sheet 115J/09, 10 and 15), in west central Yukon, in the north-westerly trending Dawson Range mountains, 300 km northwest of the territorial capital of Whitehorse. Figure 1-1 in Section 1 is a map showing the location of Casino property in relation to the Yukon, British Columbia and the Northwest Territories. The property covers a total area of 13,124 ha.

The Yukon has a population of approximately 36,000 people. Whitehorse is the nearest commercial and population center to the project property, with a population of approximately 27,500 people. Whitehorse is 380 km from the mine site via Carmacks. 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.

The Arctic Circle is 430 km to the north. The Yukon River flows about 16 km north of the site. Yukon Highway 1, the Alaskan Highway, is about 110 km west at the nearest point. Yukon Highway 2, the Klondike Highway, is about 100 km to the east at the nearest point. No year-round roads reach the property.

The international border and Alaska are about 111 km to the west at the nearest point. British Columbia is south approximately 300 km. The closest port is Skagway, Alaska.

The area around Casino has been subject to increasing staking and exploration activity over the past few years. Over 100 mining companies are now actively working in the general region. Two properties have defined reserves, the Carmacks Copper Project and the Minto Mine.

Exploration projects in the area include the following:

- To the west, Kaminak Resources is exploring for gold on their Coffee Creek property. In 2012, Kaminak had an extensive drill program and announced the intent to produce a resource estimate in 2013.
- To the north, Kinross Gold Corporation has acquired and explored the White Gold property.
- To the east, Ethos Gold Corp. conducted reverse circulation drilling on their Betty Property in 2012. Also, further east, Northern Freegold Resources, Ltd. undertook a small drill campaign on their Freegold Mountain claims in 2012.
- To the south Dehua International, a Chinese company, has acquired mineral claims.

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. The proposed access road crosses into Little Salmon Carmacks First Nation traditional territory to the south.

4.2 LAND POSITION AND STATUS

4.2.1 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 1,200 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 Casino Project area is estimated to be -2.7 °C, with minimum and maximum monthly temperatures of -18.1 °C and 11.1 °C occurring in January and July, respectively. The mean monthly temperature values are presented in Table 5-1 in Section 5. The Mean Annual Precipitation (MAP) for the Casino Project area is estimated to be 500 mm, with 65% falling as rain and 35% falling as snow.

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.

4.2.2 Environmental

See Section 20 for a list of permits either obtained or in progress. No environmental liabilities are expected to impact the Project.

4.2.3 Mineral Tenure

The Casino Property lies within the Whitehorse Mining District and consists of 723 full and partial Quartz Claims and 85 Placer Leases acquired in accordance with the Yukon Quartz Mining Act. The total area covered by Casino Quartz Claims is 13,339.17 ha. The total area covered by Casino Placer Leases is 747.45 ha. The claims are registered in the name of, and are

100%-owned by CMC, a wholly-owned subsidiary of WCGC. A list of claims is provided in Appendix B.

The historical claims held by prior owners of the project and transferred as part of Western Copper's plan of arrangement with Lumina Resources Corp. ("Lumina") consist of 83 Casino "A" claims covering an area of 1,143 ha, 55 Casino "B" claims covering an area of 924 ha, and 23 claims in the "JOE" block covering an area of 322 ha.

The 188 VIK mineral claims, covering an area of 3,416 ha, were staked in June 2007 by CRS. In June 2008, an additional 94 "CC" claims, covering an area of 1,933 ha, and 63 "BRIT" claims covering an area of 1,223 ha were staked by CRS Copper Resources Corp. ("CRS"). In October, 2009, CRS staked 136 AXS mineral claims, covering an area of 2,845 ha. In May of 2010, CRS staked an additional 63 AXS claims, covering an area of 1,318 ha. In 2011, CRS staked 18 FLY claims covering 327 ha.

4.2.4 Option Agreements

On August 9, 2007, WCG exercised its option to acquire the 161 claims that comprise the Casino "A", "B", and "JOE" claims north and east of the Casino deposit in exchange for a $1 million cash payment to Great Basin Gold Ltd. ("GBG").

The Casino "A" and "B" claims are subject to an option agreement (the "Casino B Option") with Cariboo Rose Resources Ltd. ("Cariboo Rose").

Pursuant to the Casino B Option, Cariboo Rose agrees to maintain the Casino "A" and "B" claims in good standing until May 2, 2020. In exchange, Cariboo Rose has the right to acquire the Casino "B" claims for $1 each, payable on May 2, 2020. Cariboo Rose may acquire the Casino "B" claims at any time prior to May 2, 2020 by making a $200,000 payment to CMC. Should Cariboo Rose make the $200,000 payment, it will be relieved of any further maintenance obligations respecting the Casino "A" claims.

4.2.5 Agreements and Royalties

Certain portions of the Casino property remain subject to certain royalties and production payments, as follows:

WCGC is required to make a $1 million payment to GBG within 30 days of a production decision involving either the Casino "A", "B", or "JOE" claims.

All Casino claims other than the Casino "B" claims are subject to a 2.75% Net Smelter Returns (NSR) Royalty in favour of 8248567 Canada Limited. WCGC has the option to repurchase 0.75% of the NSR Royalty (resulting in a 2.00% remaining NSR Royalty) for the following amount:

a. US$39 million, if the amount is paid on or prior to December 31, 2013; or
b. US$59 million, if the amount is paid after January 1, 2014 but on or before December 31, 2017.

Should WCGC or CMC maintain title to any Casino B claims after the period covered by the Casino B Option or acquire Casino B claims in any way; the 5% Net Profits Royalty in favour of 8248567 Canada Limited will be suspended and the NSR Royalty will apply to such claims.

The Casino "B" claims are subject to:

- 5% Net Profits Royalty in favour of 8248567 Canada Limited;
- 5% Net Profits Royalty in favour of GBG; and
- 5% Net Profits Royalty in favour of Archer Cathro & Associates Ltd.

4.2.6 Placer Claims

There are 28 active placer claims held by others that are staked around Canadian Creek that overlap the Casino property mineral claims. There are no pre-existing agreements relating to these overlapping placer claims.

In the summer of 2010, Western Copper staked a 5-mile Placer Lease along Casino Creek and a 3-mile Placer Lease along Britannia Creek. In 2011, these leases were converted to claims. WCGC, through CMC, has 30 placer claims on Britannia Creek and 55 placer claims on Casino Creek.

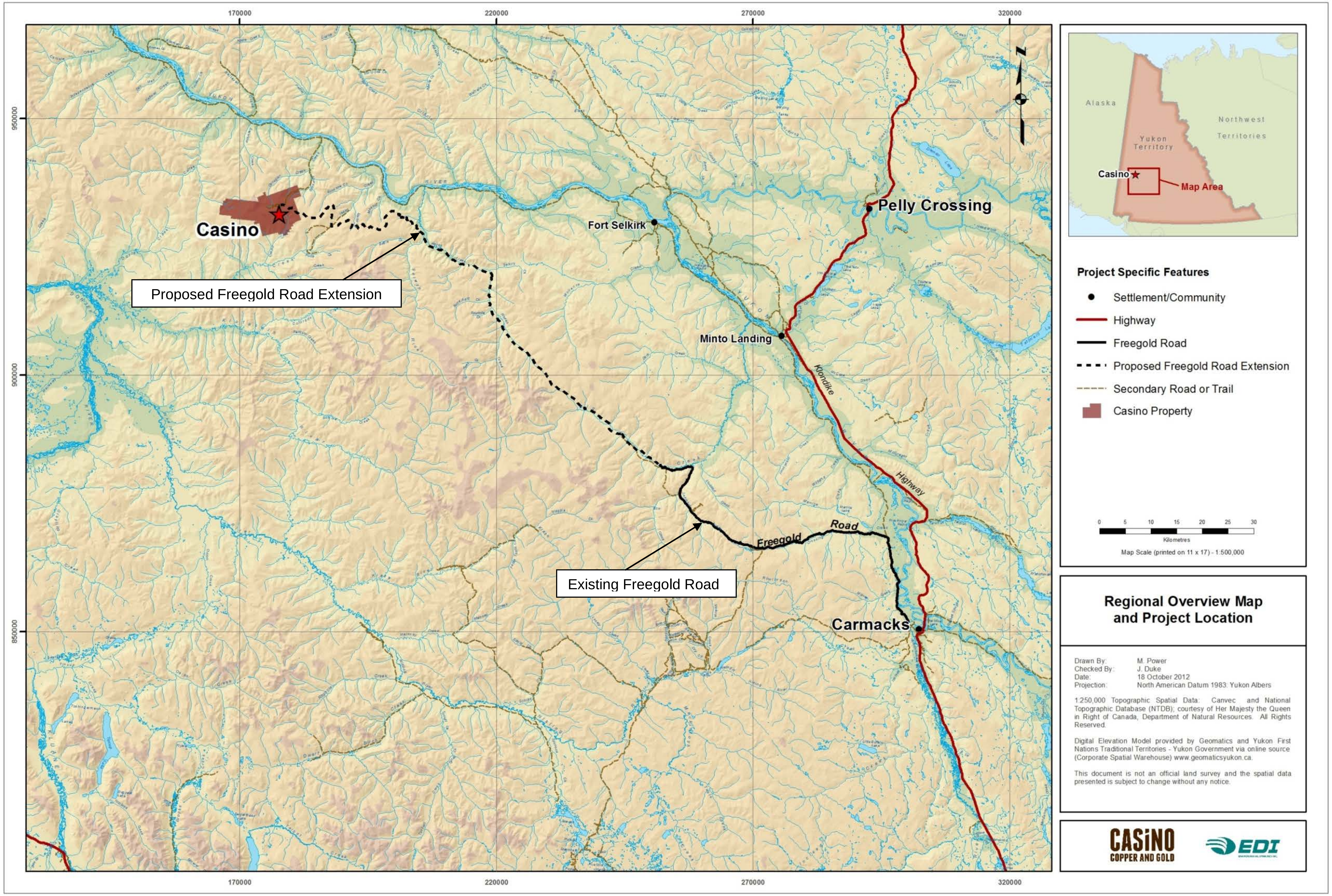


Figure 4-1: Project Road Access Map

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Casino Mine is located in Central Yukon, roughly 150 km due northwest of Carmacks, at approximately N62° 44’ 25”, W138° 49’ 32”. Current site access is by small aircraft using the existing 760 m airstrip by winter road and from the Yukon River.

Associated Engineering examined various route options for a year-round access road. The selected option, which appears to have the least environmental impact and the greatest stakeholder support, is a new 132 km unpaved road from the end of the Freegold Road, approximately 70 km northwest of the village of Carmacks. This road is further described in Section 18.2.

Either road or barge service will provide early access for construction equipment, camp construction and initial equipment. A new barge landing area at Britannia Creek and the Yukon River was prepared in 2010 and the lower 10 km of the 23 km road from the landing to the site was realigned.

The project plan includes a new airstrip, described in Section 18.1.6.

5.2 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, at about 1,250 m and deeply incised by mature dendritic drainages. Although the Dawson Range escaped Pleistocene continental glaciation, minor alpine glaciation has produced a few small cirques and terminal moraines.

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. The southern part of the property flows southward via Casino Creek to Dip Creek to the Donjek River and northward to the Yukon River.

Outcrop is rare on the property. 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.

5.3 CLIMATE

The climate in the Dawson Range is subarctic. Permafrost is widespread on north-facing slopes, and discontinuous on south-facing slopes. WCGC installed an automated weather station at the site in 2009 and collected a certain amount of data.

The climate at the Casino Project area can generally be described as continental and cold. Winters are long, cold and dry, with snow generally on the ground from September through June.

Summers are short, mild and wet, with the greatest monthly precipitation falling in July. The climate and hydrology at the Project site have been assessed based on both short term site data and longer-term regional data. Site data are available from a program operated from 1993 to 1995 and from the current program that was initiated in 2008. A summary of the hydrometeorological parameters is described below and the detailed analyses are presented in KPL report "Hydrometeorology Report" (Ref. No. VA101-325/3-1, June 15, 2010).

The mean annual temperature for the Casino Project area is estimated to be -2.7 °C, with minimum and maximum monthly temperatures of -18.1°C and 11.1°C occurring in January and July, respectively. The mean annual precipitation (MAP) for the Casino Project area is estimated to be 500 mm, with 65% falling as rain and 35% falling as snow. The mean monthly temperatures and precipitation are presented in Table 5-1.

Table 5-1: Mean Monthly Temperature and Precipitation Values

	Parameter	
Month	**Precipitation (mm)**	**Temperature (°C)**
Jan	25	-18.1
Feb	19	-14.2
Mar	16	-8.2
Apr	15	-0.1
May	42	5.7
Jun	74	9.8
July	103	11.1
Aug	65	9.1
Sept	49	4.4
Oct	35	-3.3
Nov	31	-12.7
Dec	26	-16.5
Annual	500	-2.7

The estimated average annual lake evaporation is 308 mm at the Heap Leach Facility, based on climate data collected at site and used in conjunction with long-term regional climate data.

Based on the estimated MAP of 500 mm and a rain/snow ratio of 0.65/0.35, the annual snowfall value for Casino was estimated to be 175 mm. This is generally consistent with the 140 mm mean annual maximum snowpack value (snow water equivalent, SWE) recorded in the Project area at the Casino Creek snow course station (09CD-SC01) operated by the Yukon Department of Environment (1977-2009), Water Resources Branch.

Based on the complete years of snowpack data, the average monthly snowmelt distribution for the Casino Project area was estimated to be 40% in April and 60% in May, although there is considerable variation from year to year.

5.4 TRANSPORTATION AND SHIPPING

Associated Engineering examined seven alternative routes for an all-weather road access to the mine. The analysis included consideration of the ports of Skagway and Haines as seawater ports for the mine. Their report also evaluated alternative modes of transportation include barge, pipeline, rail, air, and truck.

It was concluded that trucking presents the most reliable means of transporting concentrate and supplies to and from the Mine. The selected access road route connects the site to the existing Freegold road, and is discussed in Section 18.2. Associated Engineering also examined port options and determined that the Skagway port provides the most advantageous port for both concentrate exports and mine supply imports. The port at Skagway offers a dedicated terminal and storage facility for receiving and load-out of concentrates.

Highway-capable trucks will carry inbound and outbound materials and supplies. No substantial rail cargo service exists within the Yukon Territory.

5.5 WATER RIGHTS

The project economics have been based on the assumption that rights can be obtained for withdrawal of water from the Yukon River.

5.6 POWER AVAILABILITY

There is no utility power available to serve site. The Project will need to generate its own power. See Section 18.8.

5.7 MINING PERSONNEL

There will be approximately 600 to 700 permanent mining personnel on site during production with an additional 100 to 200 contractor support personnel. Yukon Territory has a long mining history, so it is the intent to employ as many people from the Yukon Territory as possible, including from First Nations. However, it is recognized that additional personnel from outside the territory may be necessary to fully staff the site. Since the project site is in a remote location, mining personnel employed at the site from outside the territory will be flown between Casino and Whitehorse on a rotation basis. At Whitehorse, personnel will use commercial carriers to fly to and from areas outside the territory. See Sections 18.1.6 and 18.2 for information on the airstrip and the access road.

5.8 AVAILABLE LAND AND RIGHTS OF WAY

Western Copper Corporation's land position includes claim boundaries extending to the north along the proposed water pipeline alignment and to the south along the proposed access road alignment as far as Dip Creek.

CMC has sufficient rights and available land at the Project site for the mine, tailing storage areas, waste disposal areas, heap leach pad areas and process plant areas.

Rights-of-way and permits will be required from various agencies for the access road.

6 HISTORY

The Casino Property has had a long and varied exploration history. The first documented placer claims in the immediate area were recorded in April 1911, following a placer gold discovery on Canadian Creek by J. Britton and C. Brown. In 1917, D.D. Cairnes, of the Geological Survey of Canada, recognized huebnerite ($MnWO4$) in the heavy-mineral concentrates of the placer workings. He suggested that the gold and tungsten mineralization was derived from an intrusive complex on Patton Hill (which is now recognized as the core of the Casino porphyry deposit). The total placer gold production is unknown; the most recent work (1980-1985) yielded about 50 kg (1,615 troy ounces) of gold. During the Second World War, a small amount of tungsten was recovered.

The first mineral claims at Casino were staked by N. Hansen in 1917. In 1936 silver-lead-zinc veins were discovered by J. Meloy and A. Brown approximately 3 km south of the Canadian Creek placer workings. Over the next several years the Bomber and Helicopter vein systems were explored by hand trenches and pits. The Helicopter claims were staked in 1943 and the Bomber and Airport groups in 1947.

From 1948 to 1967 the focus of exploration on the property was for lead-silver mineralization at the Helicopter and Bomber veins. The property was optioned to Noranda in 1948 and then to Rio Tinto in 1963. During this time trenching, mapping and sampling were conducted.

In 1963, L. Proctor purchased the claims and formed Casino Silver Mines Limited to develop the silver-rich veins. Between 1965 and 1980, the silver-bearing veins were explored and developed intermittently by underground and surface workings. In total, 372.5 tonnes of hand-cobbed argentiferous galena, assaying 3,689 g/t Ag, 17.1 g/t Au, 48.3% Pb, 5% Zn, 1.5% Cu and 0.02% Bi, were shipped to the smelter at Trail, British Columbia.

B. Hestor noted that the area had porphyry deposit potential in 1963, but his observations did not become generally known. In 1967, the porphyry potential was recognized again, this time by A. Archer and separately by G. Harper. Archer's evaluation led to the acquisition of Casino Silver Mines Limited by the Brynelsen Group, and from 1968 to 1973 exploration was directed jointly by Brameda, Quintana, and Teck Corporation towards a porphyry target. Exploration included extensive geophysical and trenching program, but it was mainly thanks to the soil geochemistry, that the porphyry deposit was discovered in 1969.

Following the porphyry discovery, various parties including Brameda Resources, Quintana Minerals and Teck Corporation drilled the property. During this period (between 1969 and 1973), 5,328 m of reverse circulation drilling in 35 holes and 12,547 m of diamond drilling in 56 holes was completed.

In 1991, Archer, Cathro & Associates (1981) Ltd. optioned the property and assigned the option to Big Creek Resources Ltd. A drill program in 1992 consisting of 21 HQ (63.5 mm diameter) holes totalling 4,729 m systematically assessed the gold potential in the core of the deposit for the first time. The larger-sized core gave better recovery and more reliable assays than earlier drilling.

In 1992, Pacific Sentinel Gold Corp. (PSG) acquired the property from Archer Cathro and commenced a major exploration program. The 1993 program included surface mapping and 50,316 m of HQ and NQ (47.6 mm diameter) drilling in 127 holes. All but one of the 1992 drill holes were deepened in 1993.

In 1994, PSG drilled an additional 108 drill holes totalling 18,085 metres. This program completed the delineation drilling set out in 1993 and investigated various geological, geotechnical, structural, and environmental aspects of the project. In addition, PSG performed a considerable amount of metallurgical, geotechnical and environmental work and completed a scoping study in 1995. The scoping study envisioned a large-scale open pit mine, conventional flotation concentrator that would produce a copper-gold concentrate for sale to Pacific Rim smelters.

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. In the fall of 2011, Western Copper Corporation spun out all other assets except the Casino Deposit and changed its name to Western Copper and Gold Corporation.

In 2007, Western Copper conducted an evaluation of the Bomber Vein System and the southern slope of the Patton Hill by VLF-EM and Horizontal Loop EM survey and soil geochemistry. Environmental baseline studies were also initiated in 2007.

In 2008, Western Copper reclaimed the old camp site, constructed a new exploration camp next to the Casino airstrip and drilled three drill holes (camp water well and two exploration holes) totalling 1,163 m. The main purpose of the drilling was to obtain fresh core samples for the metallurgical and waste characterization tests. Both exploration holes twinned PSG's holes to confirm historic copper, gold and molybdenum grades. Later that year, M3 Engineering produced a pre-feasibility study for Western Copper.

In 2009, Western Copper completed 22.5 km of DC/IP surveying and MT surveying using the Quantec Geosciences Ltd. Titan system. As well, the company drilled 10,943 meters in 37 diamond drill holes. 27 holes were infill holes drilled to convert inferred and non-defined material to measured and indicated. Infill drilling covered the north slope of the Patton Hill that was mapped as "Latite Plug" on PSG maps. Drilling has identified supergene Cu mineralization and Mo mineralization in this area. The remaining 10 holes, totalling 4,327 m, were drilled to test geophysical targets.

In 2010, infill and delineation drilling continued with most of the drilling done to the North and West of the deposit as outlined by PSG. The drilling program also defined hypogene mineralization at the southern end of the deposit. In addition, the company drilled a series of geotechnical holes at the proposed tailings embankment area and within the pit and several holes for hydrogeological studies. The geotechnical drilling continued in 2011 (41 holes, 3,163 m) and 2012 (6 holes, 228 m).

A breakdown of drilling by WCGC from 2010 to 2012 is as follows:

- 47 exploration holes for 12,200.11 m
- 11 combined hydrogeological holes for 1,689.58 m
- 53 geotechnical holes in the tailings embankment, heap leach pad, plant site, waste rock storage, airstrip, access road and water well areas, for 3,786.54 m
- 5 holes, 1,570.63 m for metallurgical sample

The total meterage drilled by Western Copper from 2008 to 2012 is 31,353.19 m.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 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 7-1 summarizes the major units with isotopic ages. All isotopic dates are based on U-Pb ratios in zircons analyzed by J.R. Mortensen.

The Yukon Crystalline Terrane in the Dawson Range area is represented by the Devono-Mississippian Wolverine Creek Metamorphic Suite (Johnston, 1995) made up of sedimentary and igneous protoliths (Tempelman-Kluit, 1974; Payne et al., 1987). These meta-sedimentary rocks consist mainly of quartz-feldspar-mica schist and gneiss, quartzite, and micaceous quartzite, while the meta-igneous unit includes biotite-hornblende-feldspar gneiss and other orthogneisses, as well as hornblende amphibolite (Selby & Nesbit, 1997).

During the mid-Cretaceous, Wolverine Creek Metamorphic rocks of this area were intruded by the Dawson Range Batholith and subsequent Casino Intrusions (Selby et al., 1999). The Dawson Range Batholith is the main country rock of the Casino Property and is represented by a relatively homogeneous, medium- to coarse-grained, hornblende-bearing, potassic quartz diorite to granodiorite; and lesser fine- to medium-grained diorite and quartz monzonite veins, dykes, and plugs (Tempelman-Kluit, 1974).

The Casino Intrusions, better termed the Casino Plutonic Suite, were said to be composed of quartz monzonite stocks up to 18 kilometres across (Hart and Selby, 1998) trending westnorthwest parallel to the Big Creek Lineament and its northwestern extension. Mapping by Tempelman-Kluit (1974), and successively by Payne et al. (1987), associates this Casino Plutonic Suite with the mid-Cretaceous Dawson Range Batholith. Subsequently, Johnston (1995) grouped the intrusions with the late-Cretaceous Prospector Mountain Plutonic Suite based largely on field relationships that show stocks of the Casino Plutonic Suite cutting the Dawson Range Batholith. Later age determination by Mortensen and Hart in 1998, as well as geochemistry provided by Selby et al. (1999), placed the Casino Intrusions back into the mid-Cretaceous as fractionated magmas of the Dawson Range Batholith. Recent field relationships have proven that the ‘quartz monzonites’ of the Casino property, once thought to be separate intrusions, are actually intensely altered and recrystallized diorites of the Dawson Range Batholith.

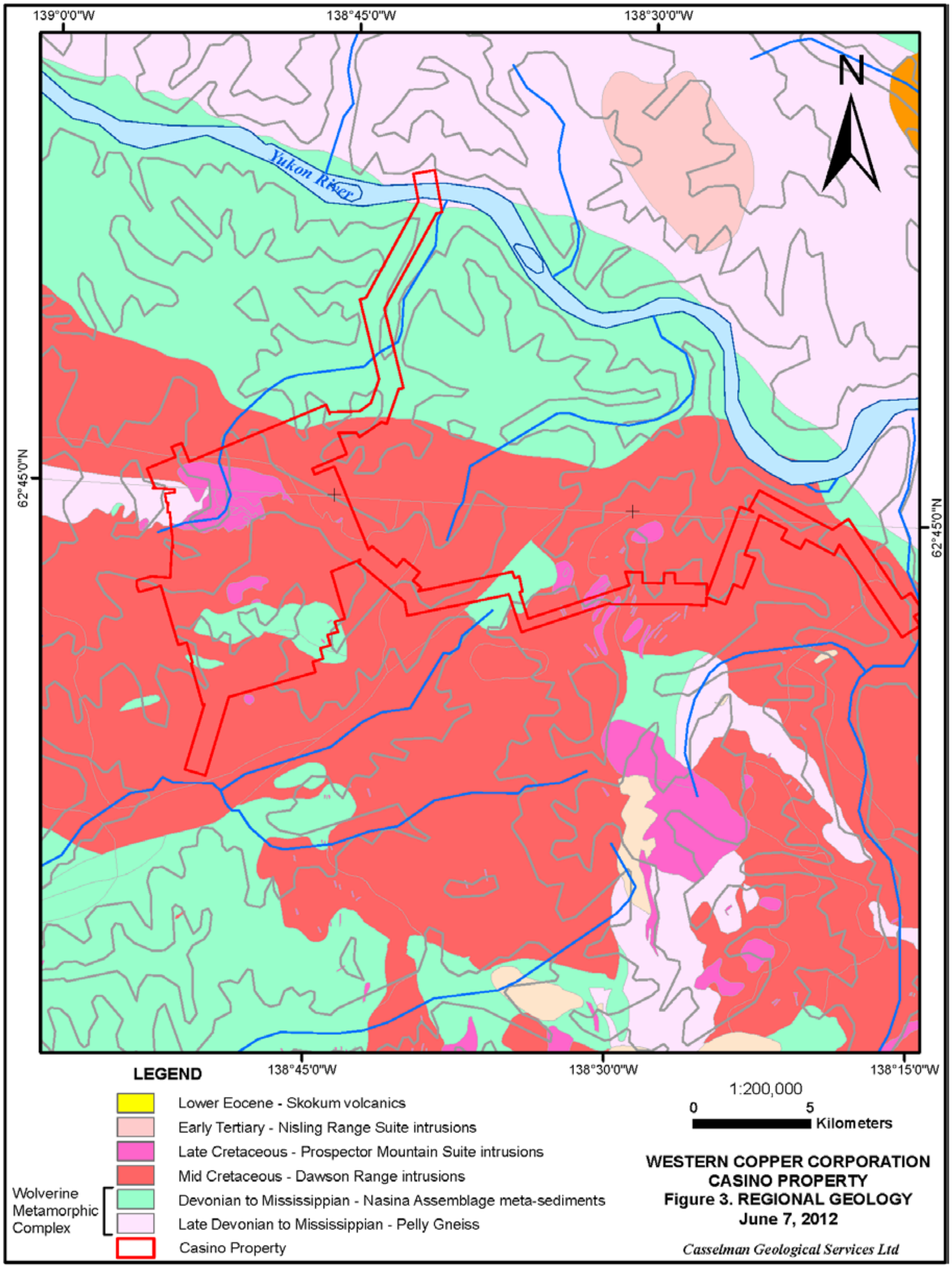


Figure 7-1: Regional Geology

In the late Cretaceous, the Prospector Mountain Plutonic Suite intruded as stocks and apophyses into the Dawson Range Batholith (Johnston, 1995; Selby et al, 1999). In the Casino area, this suite is represented by Patton Porphyry: small, biotite-bearing, feldspar-porphyritic, hypabyssal rhyodacite to dacite intrusions near the center of the deposit and discontinuous centimeter- to metre-wide dikes northwest of the property. Here, early phases the Patton Porphyry grade into a mineralized intrusive breccia. Later, unaltered dykes of similar rock cut surrounding hydrothermally altered and mineralized rocks (Payne et al., 1987) suggesting there are multiple phases of this unit (Bower, 1995; Selby and Creaser, 2001). Hydrothermal alteration and mineralization occur in and adjacent to some of these late Cretaceous intrusions.

The Casino Property is sandwiched between parallel west-northwest-trending faults that form contacts between rocks of the Wolverine Creek Metamorphic Suite and the Dawson Range Batholith. In Figure 7-2 (Below), the fault furthest to the northeast is an extension of the Big Creek Fault thought to be dextrally offset by 20 to 45 km. A parallel fault 8 km to the southwest forms the southwest boundary of a sliver of Wolverine Creek Metamorphic Suite rocks and contains outcroppings of ultramafic rocks in a similar fashion to those seen along the Big Creek Fault.

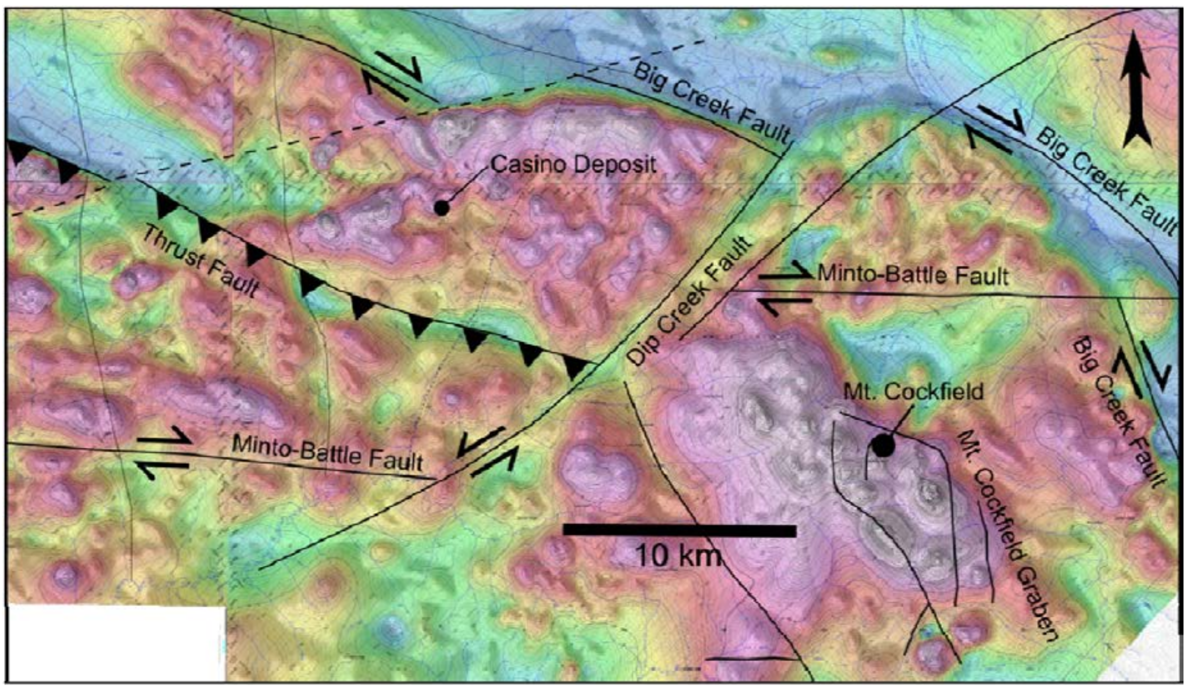


Figure 7-2: Regional structures overlain on recent aeromagnetic survey

The Casino Property is bounded to the southeast by a northeast-trending regional structure known as the Dip Creek Fault. The left-lateral displacement of the Yukon River well east of the Casino Property is a reflection of sinistral movement along this fault. The east-trending Minto-Battle Fault is also sinistrally translated by the Dip Creek Fault (Johnston, 1999). This dextrally offset fault lies east of the Casino Property on the opposite side of Dip Creek with its extension lying south and southwest of the Casino Property.

7.2 MINERALIZATION

7.2.1 Hydrothermal Alteration

Crystallization and exsolution of hydrothermal fluids from Patton Porphyry magmas produced porphyry style Cu-Mo-Au mineralization. Therefore, the Patton Porphyry, and associated Intrusive Breccia, is genetically related to the Cu-Mo-Au mineralization of the deposit.

Hydrothermal alteration at the Casino property consists of a potassic core centered on and around the main Patton Porphyry body, in turn bordered by contemporaneous, strongly developed and fracture controlled phyllic zone, a weak propylitic zone, and a secondary discontinuous argillic overprint. Mineralized stockwork veins and breccias on the Casino Property are closely associated with the hydrothermal alteration.

Potassic alteration minerals include texturally destructive K-feldspar, biotite, magnetite and quartz with lesser hematite, purple anhydrite and gypsum. Biotite is generally felted and pseudomorphic after hornblende. Occasionally, magnetite may form braided veinlets. In drill core potassic alteration is represented by dark brown to black biotite alteration and/or by pink potassium feldspar alteration.

The texturally destructive phyllic zone is found peripheral to, and locally overprints, the potassic zone of alteration. It has a distinctive 'bleached' appearance and can be structurally controlled. Phyllic alteration minerals include quartz, pyrite, sericite, muscovite (after biotite), and abundant tourmaline, as well as minor hematite and or magnetite towards the potassic zone. Quartz and sericite are generally alteration minerals after potassic and plagioclase feldspars. Biotite alters to muscovite or titanite and hornblende alters to chlorite, calcite, quartz and biotite. Tourmaline forms radiating disseminations and veinlets. Sulphides are typically high, with pyrite ranging from 5-10% throughout as disseminated blebs or cores to quartz 'd' veins.

Where intense phyllic overprints potassic alteration relict textures are destroyed and minerals are recrystallized, commonly to equal portions of quartz, plagioclase, and K-feldspar; up to 10 percent biotite; and trace apatite and titanite. Strongly zoned plagioclase and locally kinked biotite form subhedral lathes, surrounded by K-feldspar, locally strained quartz, and biotite. The colour is pale pink overall.

Propylitic alteration is rare on surface, but forms a wide halo around the deposit in gradational contact with the inner potassic alteration. Alteration minerals include epidote, chlorite and calcite, with lesser carbonate, clay, sericite, pyrite and albite. Hornblende and biotite are completely chloritized whereas feldspars look relatively fresh, and textures are generally well-preserved.

In typical porphyry copper deposits, advanced argillic alteration will occur above the phyllic alteration. It appears that on the Casino property, all evidence of advanced argillic alteration has been eroded or destroyed.

Secondary argillic alteration is closely associated with the supergene zone and may appear locally as patches or pockets within potassic and phyllic alteration. It is poorly developed, appears bleached or pale green, and contains abundant clays (kaolinite, montmorillionite), and possible chlorite and/or carbonate. In drill core, this unit may be recognized by distinctive "pock-marks" along the surface of the core.

7.2.2 Supergene Mineralization

The Casino deposit is unusual among Canadian porphyry deposits as it has a substantially preserved, outcropping oxide gold leached cap; an upper well-developed, copper enriched supergene zone; and a lower copper-gold hypogene zone. Table 7-1 summarizes the main minerals identified in the Leached Cap and Supergene zones.

Leached Cap Mineralization (CAP)

The Leached Cap (oxide gold zone) is gold-enriched and copper-depleted due to supergene alteration processes as well as the lower specific gravity of this zone relative to the other supergene zones. It averages 70 metres thick and is characterized by boxwork textures filled with jarosite, limonite, goethite, and hematite. This weathering has completely destroyed rock textures and replaces most minerals with clay. The resulting rock is pale gray to cream in colour and is friable to the touch, and the clay is often stained yellow, orange, and/or brown by iron oxides. The weathering is most intense at the surface and decreases with depth.

Supergene Oxide Mineralization (SOX)

The poorly defined Supergene Oxide zone is copper-enriched, with trace molybdenite. It exists as a few perched bodies within the leached cap likely due to more recent fluctuations in the water table. This zone is thought to be related to present day topography, and is best developed where oxidation of earlier secondary copper sulphides occur above the water table, on well drained slopes. Where present, the supergene oxide zone averages 10 metres thick, and can contain chalcanthite, malachite and brocanthite, with minor azurite, tenorite, cuprite and neotocite. Where present, the supergene copper oxide zone grades into the better-defined supergene copper sulphide zone.

Supergene Sulphide Mineralization (SUS)

Supergene copper mineralization occurs in an up to 200 metre-deep weathered zone below the leached cap and above the hypogene. It has an average thickness of 60 metres and is positively correlated with high grade hypogene mineralization, high permeability, and phyllic and/or outer potassic alteration. Grades of the Supergene sulphide zone vary widely, but are highest in fractured and highly pyritic zones, due to their ability to promote leaching and chalcocite precipitation. Thus, secondary enrichment zones are thickest along contacts of the potassic and phyllic alteration; accordingly, the copper grades in the Supergene Sulphide zone are almost double the copper grades in the Hypogene (0.43% Cu versus 0.23% Cu). Grain borders and fractures in chalcopyrite, bornite and tetrahedrite may be altered to chalcocite, diginite and/or covellite. Chalcocite also locally coats pyrite grains and clusters, and may extend along fractures deep into the hypogene zone. Molybdenite is largely unaffected by supergene processes, other than local alteration to ferrimolybdite.

In drill-core, the SUS zone is generally broken with decreasing clay alteration and weathering and is 'stained' dark blue to gray.

Table 7-1: 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

7.2.3 Hypogene Mineralization

Mineralization of the Casino Cu-Au-Mo deposit occurs mainly in the steeply plunging, in-situ contact breccia surrounding the Patton Porphyry intrusive plug by crystallization and exsolution of hydrothermal fluids from late Cretaceous magmas of the Casino Plutonic Suite. The breccia forms an ovoid band around the main porphyry body with dimensions up to 250 metres, and has an interior zone of potassic alteration surrounded by discontinuous phyllic alteration, typical of some porphyry deposits.

Hypogene mineralization occurs throughout the various alteration zones of the Casino Porphyry deposit, as mineralized stock-work veins and breccias. Field relationships show that the potassic alteration came first as mineralized quartz veins of the phyllically altered zones, cut those of the potassically altered zones; Re-Os age dating showed that the dates of the potassic and phyllic alteration are contemporaneous at around 74.4 +/- 0.28 Ma. Significant Cu-Mo mineralization is related to the potassically- altered breccia surrounding the core Patton Porphyry, as well as in the adjacent phyllically-altered host rocks of the Dawson Range Batholith.

Mineralization in the potassic zone is mainly finely disseminated pyrite, chalcopyrite, molybdenite as well as trace sphalerite and bornite. The phyllic zones have increased gold, copper, molybdenite, and tungsten values concentrated in disseminations and veins of pyrite, chalcopyrite, and molybdenite along the inner side of the pyrite halo. The pyrite halo follows the potassic-phyllic contact, within the phyllic zone, and discontinuously surrounds the main breccia body. It is host to the highest Cu values on the property.

Chalcopyrite commonly occurs as veins, disseminations and irregular patches. In breccia and granodiorite west of the Casino Fault, disseminated chalcopyrite is more abundant than chalcopyrite in veins and veinlets, whereas to the east of the fault, chalcopyrite is controlled by brittle deformation and found in fractures and open space fillings. Pyrite to chalcopyrite ratios range from less than 2:1 in the core of the deposit, to greater than 20:1 in the outer phyllic zones. Locally, bornite and tetrahedrite can be coarsely intergrown with chalcopyrite.

Molybdenite is not generally intergrown with other sulphides and occurs as selvages in early, high temperature, potassic quartz veins and as discrete flakes and disseminations.

Native gold can occur as free grains in quartz (50 to 70 microns) and as inclusions in pyrite and/or chalcopyrite grains (1 to 15 microns). High grade smoky quartz veins with numerous specks of visible gold are also reported to exist.

Late-stage, commonly vuggy, polymetallic veins (like those of the Bomber Vein) follow roughly parallel, steeply dipping fractures trending 150 to 170 degrees. Metallic mineralogy includes abundant sphalerite and galena, with less abundant tetrahedrite, chalcopyrite (commonly intergrown with tetrahedrite), and bismuth bearing minerals, and are geochemically anomalous in any or all of Ag, As, Bi, Cu, Cd, Mn, Pb, Sb, Zn, and locally W.

In drill-core, the hypogene zone is un-weathered and un-oxidized.

8 DEPOSIT TYPES

8.1 GEOLOGY OF THE CASINO DEPOSIT

The geology of the Casino deposit is typical of many porphyry copper deposits. The deposit is centered on an Upper Cretaceous-age, east-west elongated tonalite porphyry stock that intrudes Mesozoic granitoids of the Dawson Range Batholith and Paleozoic schists and gneisses of the Yukon Crystalline Complex. Intrusion of the tonalite stock into the older rocks caused brecciation of both the intrusive and the surrounding country rocks along the northern, southern and eastern contact of the stock. Brecciation is best developed in the eastern end of the stock where the breccia can be up to 400 metres wide in plan view. To the west, along the north and south contact, the breccias narrow gradually to less than 100 metres. Little drilling has been done at the western end of the tonalite stock and it is not known if the breccia is present along this contact. Intruded into the tonalite stock and surrounding granitods and metamorphic rocks are younger, non-mineralized dykes of similar composition to the older tonalite stock and a late diatreme, which forms both pipe-like body in the west and a dyke-like body in the east. The overall dimensions of the intrusive complex are approximately 1.8 by 1.0 kilometres.

Primary copper, gold and molybdenum mineralization was deposited from hydrothermal fluids that exploited the contact breccias and fractured wall rocks. Better grades occur in the breccias and gradually decrease outwards away from the contact zone both towards the centre of the stock and outward into the granitoids and schists. A general zoning of the primary sulphides occurs with chalcopyrite and molybdenite occurring in the tonalite and breccias grading outward in to pyrite dominated mineralization in the surrounding granitoids and schists. Alteration accompanying the sulphide mineralization consists of an earlier phase of potassic alteration and a later overprinting of phyllic alteration. The potassic alteration typically has secondary biotite, K-feldspar as pervasive replacement and veins, stockworks of quartz and anhydrite veinlets. Phyllic alteration consists of sericite and silicification in the form of replacements and veins.

The Casino Copper deposit is somewhat unique amongst Canadian porphyry copper deposits in having a well-developed secondary enriched blanket of copper mineralization similar to those found in deposits in Chile and the Southwest United States such as Escondida and Morenci. Unlike other porphyry deposits in Canada, the Casino deposit's enriched copper blanket was not eroded by the glacial action of ice sheets during the last ice age. At Casino, weather during the Tertiary Period leached the copper from the upper 70 metres of the deposit and re-deposited it lower in the deposit. This created a layer- like sequence consisting of an upper leached zone up to 70 metres thick where all sulphide minerals have been oxidized and copper removed leaving behind a bleached, iron oxide leached cap containing residual gold. Beneath the leached cap is a zone up to 100 metres thick of secondary copper mineralization consisting primarily of chalcocite and minor covellite with a thin, discontinuous layer of copper oxide minerals at the upper contact with the leach cap. The copper grades of the enriched, blanket-like zone can be up to twice that of the underlying non weathered primary copper mineralization. Beneath the secondary enriched mineralization the primary mineralization consists of pyrite, chalcopyrite and lesser molybdenite. The primary copper mineralization is persistent at depth and is still present at the bottoms of the deepest drill holes over 600 metres from surface.

8.2 LOCAL GEOLOGY

Historical descriptions of the geology of the Casino Property are somewhat variable and inconsistent. Systematic re-logging efforts of drill core from the entire property have recently simplified the geologic interpretation, shown in Figure 8-1 and Figure 8-2. The following table outlines the major geological units with relative and isotopic ages (where available) based on work done by Selby et al. (2001), and references therein.

Table 8-1: Stratigraphic Column

	Geological Unit	Isotopic Age
Late Cretaceous	**PROSPECTOR MOUNTAIN PLUTONIC SUITE:**	
	Explosion Breccia *Heterolithic; fine-grained matrix; angular clastic*	
	Heterolithic Intrusion Breccia *Heterolithic; patton porphyry/potassic matrix; autobrecciated fragments*	
	Patton Porphyry *Plag-Bi Porphyry; Kf +/- Qz megacrystic porphyry*	72.4 +/-0.5 Ma
Mid-Cretaceous	**DAWSON RANGE BATHOLITH:**	
	Granodiorite *bi-hbld granodiorite*	104.0 +/-0.5 Ma
	Diorite *Hbld-Bi-Qtz diorite; hbld-bi diorite*	104.0 +/-0.5 Ma
Devono-Mississippian	**WOLVERINE CREEK METAMORPHIC SUITE:**	
	Meta-sedimentary *Micaceous Quartzite*	
	Meta-igneous *Qtz-Bi-Plag-Microcline Gneiss; KF-Qtz-Bi Gneiss; Amphibolite*	

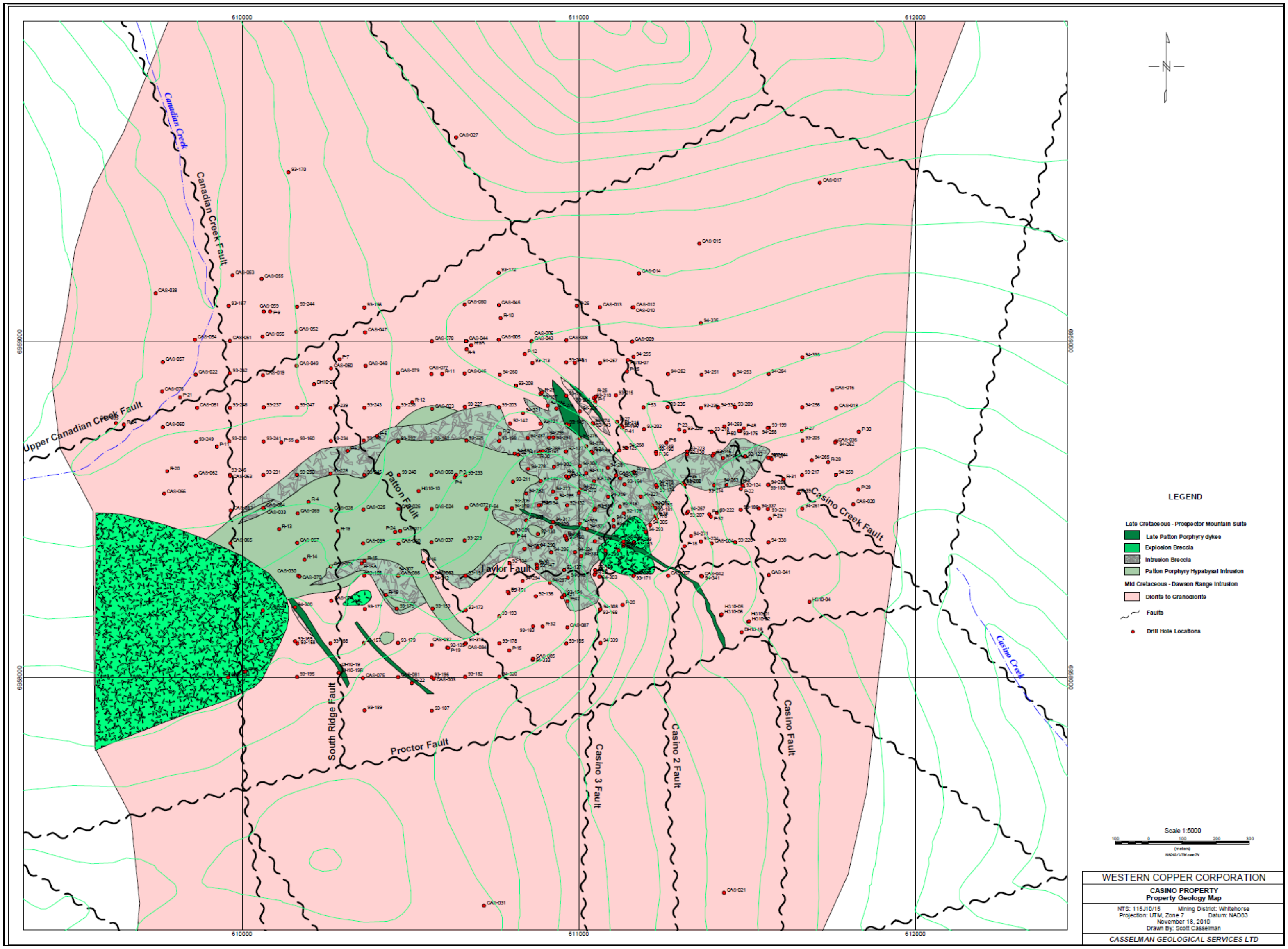


Figure 8-1: Geology of the Casino Deposit

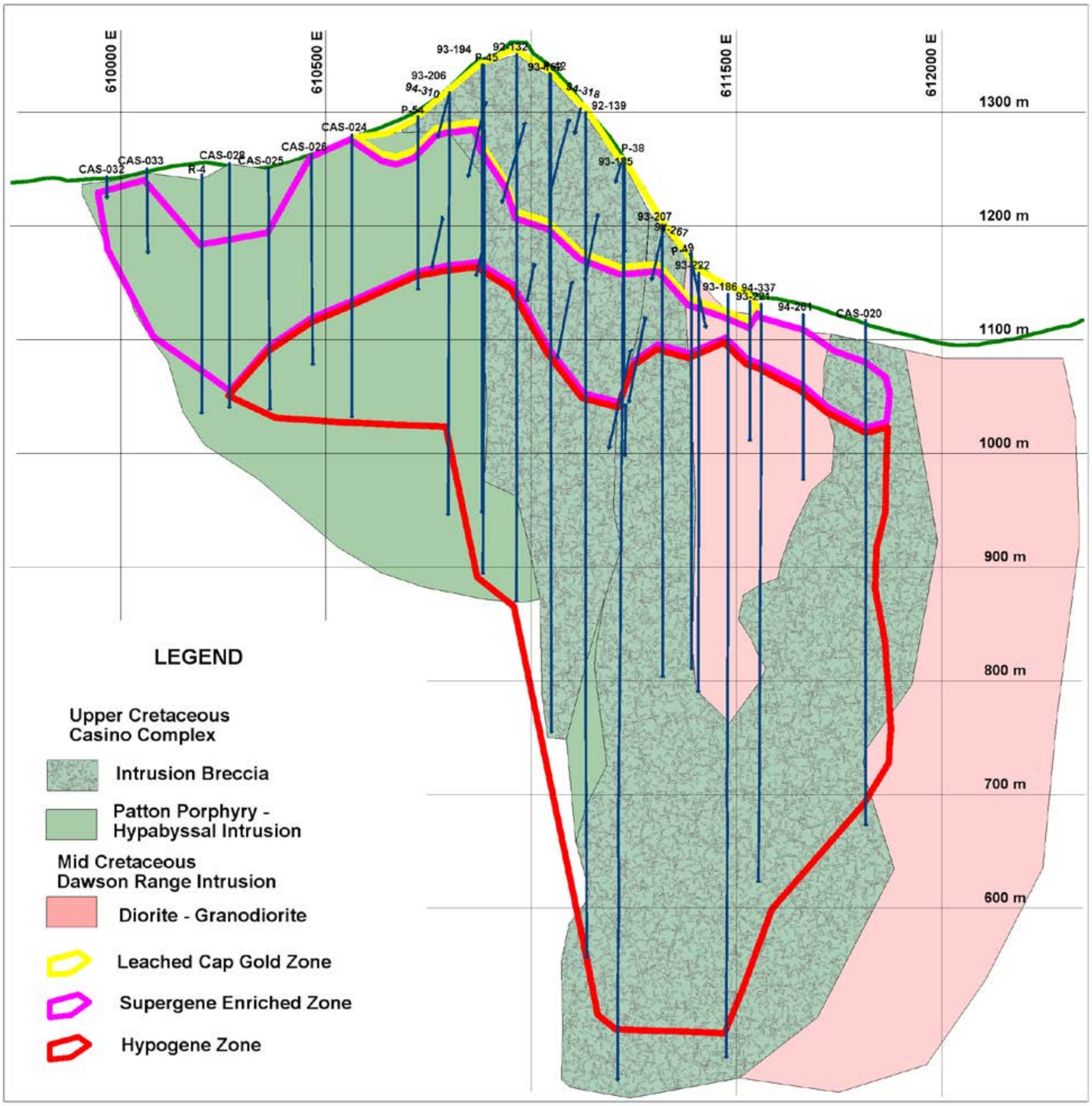


Figure 8-2: Casino Property Geology – Cross Section

Casino property geology has been described in detail by Godwin (1975) and Payne et al. (1987), and was later summarized by Bower et al. (1995). Although groupings have changed, the majority of rock descriptions have not; therefore the following sections borrow significantly from all three reports.

8.2.1 Yukon Metamorphic Complex, Wolverine Creek Metamorphic Suite (YM)

The Wolverine Creek Metamorphic Suite rocks include meta-sedimentary and meta-igneous rocks (Tempelman-Kluit, 1974; Payne et al. 1987; Johnston, 1995) of Devono-Mississippian age

(Johnston, 1995). They occur mainly in the northern and northeastern parts of the Casino deposit, as fragments in intrusion breccias and local roof pendants/screens throughout the Dawson Range Batholith (Bower et al., 1995). More common rock types in the deposit area are biotite-hornblende-feldspar diorite schist and gneiss. Less abundant types include meta-diorite/ amphibolite, quartz-rich and intermediate gneiss, quartzite, and micaceous quartzite.

8.2.2 Dawson Range Batholith

The mid-Cretaceous Dawson Range Batholith is the main country rock of the deposit and is characterized by hornblende-biotite-quartz diorite, hornblende-biotite diorite, and biotite-hornblende granodiorite (Payne et al. 1987). Hornblende-biotite bearing phases are common throughout the deposit, and lesser biotite-hornblende bearing phases are generally north of Patton Hill (Godwin, 1975). Diorite is concentrated north and northeast of the deposit, particularly east of Casino Fault, and is considered to be the earliest phase of the batholith.

Casino diorites are typically dark gray to brown, locally inequigranular, and texturally similar to the meta-diorite of the Wolverine Metamorphic Suite. Average grain size is less than 1 millimetre, dominated by locally aligned and/or zoned plagioclase; hornblende; and interstitial, anhedral quartz. In places, primary biotite is more abundant than hornblende. Accessory minerals include up to 1 percent apatite and trace titanite. Some intrusions show foliation and increased mafic content near their margins, particularly north of the deposit and in the block east of the Casino Fault (Bower et al., 1995). Locally, mafic diorites are cut by later, more felsic phases of the Dawson Range Batholith (Johnston and Shives, 1995).

Granodiorite is generally pale gray, medium to coarse grain and equigranular to porphyritic. They can be distinguished by scattered, subhedral hornblende phenocrysts averaging 0.5 to 1.2 centimetres long; poikilitic K-feldspar; zoned plagioclase; and 10 to 20 percent mafic minerals, which may be layered. Plagioclase shows minor myrmekitic rims when in contact with K-feldspar. Anhedral quartz and K-feldspar are interstitial to earlier subhedral plagioclase, hornblende and biotite. Locally, quartz forms interlocking aggregates of slightly, to moderately strained grains. Accessory minerals include honey-coloured titanite and apatite to 1 percent each.

Rocks of the Dawson Range commonly display in-situ/crackle to intensely deformed cataclastic brecciation where in contact with the Patton porphyry intrusive plug. Elsewhere, this unit may be truncated by the late Cretaceous dykes and associated explosive breccias (modified from Bower, 1995).

8.2.3 Prospector Mountain Plutonic Suite – Casino Plutonic Suite

Late-Cretaceous igneous activity of the Prospector Mountain Plutonic suite is locally represented by the Patton Porphyry intrusive and associated breccias of the Casino Plutonic Suite.

8.2.3.1 Patton Porphyry (PP)

The main body of the Patton Porphyry (72-74 Ma) is a relatively small locally-mineralized stock measuring approximately 300 by 800 metres and is surrounded by a potassically-altered Intrusion Breccia in contact with rocks of the Dawson Range. Elsewhere, the Patton Porphyry forms discontinuous dikes ranging from less than one to tens of metres wide, cutting both the Patton Porphyry Plug and the Dawson Range Batholith (Bower et al., 1995). Contacts between the Patton Porphyry and breccias are variable and range from sharply intrusive to gradational and brecciated. It has therefore been suggested by Bower et al. (1995) and Selby and Creaser (2001) that this suite consists of two or more episodes of high-level intrusions.

Godwin (1975) determined that the Patton Porphyry has an overall composition of rhyodacite, with phenocrysts falling into a dacite composition and the matrix being of quartz latite composition. It is more commonly made up of distinct phenocrysts of abundant plagioclase and lesser biotite, hornblende, quartz and opaques (Godwin, 1975). Phenocrysts average 4 millimetres in size, and can comprise up to 50 percent of the rock. Lathes of plagioclase are euhedral and zoned, and range in size from 2 to 7 millimetres, with some up to 2.5 centimetres in length (Bower et al., 1995). Biotite lathes range from 2-3 millimetres across, and make up 1-5 percent of the rock. They are kink-banded, subhedral, and locally chloritized. Hornblende phenocrysts are difficult to recognize due to their alteration, but have generally been replaced by chlorite and other opaques, and can be recognized by their diamond cross-section. Quartz phenocrysts are not always present but can be anhedral, embayed, and 3-5 millimetres in size. K-feldspar phenocrysts are rare but the mineral is abundant in the commonly medium to dark green, microcystalline matrix.

Smaller, possibly more evolved, discontinuous plugs of Patton Porphyry exist where K-feldspar and/or quartz megacrysts range from 3 to 20 mm in size, displaying ragged boundaries and intergrowths with surrounding grains (Godwin, 1975). Contacts between the main Patton plug and this unit are generally gradational or masked by alteration. Currently, the age-relationship between the megacrystic variety and the main plug is unknown.

Later Patton dykes in the south-central part of the deposit somewhat resemble the main Patton Porphyry body and contain 2 to 5 percent quartz phenocrysts and up to 35 percent plagioclase phenocrysts in an aphanitic latite groundmass (Bower et al., 1995). These dykes intruded after the main hydrothermal event and contain only minor base- and precious-metal mineralization, as well as locally abundant disseminated pyrite (Godwin, 1975). These dykes are of latitic to dacitic composition and are generally steeply dipping, striking between 130 and 160 degrees (Bower et al., 1995). On the Casino property, they are generally pale to light green with abundant plagioclase and lesser hornblende phenocrysts in a very fine- to extremely fine-grained matrix of plagioclase and K-feldspar (Payne et al., 1987). Wider versions of the dyke are coarser grained and may contain scattered quartz and/or biotite phenocrysts to 3 millimetres along with plagioclase and hornblende. Narrow versions with or without chilled dyke margins can be dark green with a glassy groundmass, and may show flow banding and/or lenticular structures near contacts (Bower et al., 1995). Outcrop of this unit can be mapped on surface trending northwest along Proctor Gulch.

8.2.3.2 Intrusion Breccia (IX)

The Intrusion Breccia surrounding the main Patton Porphyry body consists of granodiorite, diorite, and metamorphic fragments in a fine-grained Patton Porphyry matrix. It may have formed along the margins, in part, by the stoping of blocks of wall rocks (Bower et al., 1995). The unit is rhyodacitic in composition and is inherently related to the Patton Porphyry intrusive (Godwin, 1975 and Payne et al., 1987). Local quartz grains are generally 1 to 2 millimetre unstrained crystals and crystal-fragments, and are texturally similar to quartz phenocrysts of the Patton Porphyry (Bower, 1995). Eroded fragments, ranging in size from less than one centimeter to greater than a few metres, are found proximal to their associated wall rocks, and therefore indicate limited transport and/or mixing (after Bower et al., 1995). For example, an abundance of Dawson Range inclusions are prominent at the southern contact of the main plug, Wolverine Creek metamorphic rocks increase along the northern contact, and bleached diorite increases at the eastern contact of the main plug. Strong potassic alteration locally destroys primary textures (Bower et al., 1995).

8.2.3.3 Explosion Breccia (MX)

Abundant fragments of the Patton Porphyry and its Intrusion Breccia are present in a late Cretaceous Explosion Breccia pipe. Godwin (1975) concluded that this pipe most likely represents a sub volcanic neck, brecciated from explosions caused by the rapid expansion of hot water (hydrothermal solutions) by vessiculation of rhyolitic magmas, and that any extrusive volcanic related to this event may have since been weathered away. This unit indicates multiple episodes of brecciation (Bower, 1995) as it contains 5 to 50 percent ragged fragments of altered intrusive breccia and host rock, with lesser fragments of late often quartz-phyric Patton Porphyry. Locally, the groundmass has a very fine-grained interlocking igneous texture; elsewhere it resembles milled rock flour (Bower, 1995) with up to 10 percent plagioclase and lesser quartz phenocrysts. Godwin also noted large angular cavities being a distinctive quality of this unit measuring up to 10 centimetres in size.

8.2.4 Property-Scale Structure

The largest fault affecting the known mineralized portion of the Casino Property, the Casino Creek Fault, trends at 310 degrees, forming an acute angle to regional structures (Figure 8-3). The right-lateral displacement of this fault has offset the eastern part of the Casino deposit by some 200 metres (Bower, et al., 1995). Similarly-directed faults are repeated throughout the Casino Property including one heading south from Canadian Creek, traveling through Patton Gulch, and forming a col in Patton Hill. This fault is aptly named the Patton Fault. The fault nearly bisects the known Casino Deposit but little movement has been noted or described. The West Fault, on the west side of the property is thought to be similarly-oriented. Most of these shear zones are denoted by brittle to gougy drill-core intercepts and may show increased amounts of hydrothermal veining including tourmaline, pyrite, magnetite, and gangue infilling fractures.

At the eastern part of the Casino deposit near the Casino Fault: a narrow zone of steeply-dipping, south-directed faults and conductors are interpreted from ground geophysics. This orientation may have weak repetitions throughout the deposit and is considered to be a mineralized, early structure predating the Casino Creek Fault system.

West-directed fault systems have been noted in some early workings of the Casino 'C' anomaly (southeast along the Casino Creek Fault), have been suggested by airborne magnetics south of the deposit, and have been interpreted in air photos west of the deposit. In addition, this orientation proved to be prominent in veins in Pacific Sentinels study of oriented core in 1994. A suture between Dawson Range diorite and granodiorite along this orientation has locally incorporated older metamorphic rocks. This orientation is parallel to the Minto-Battle Fault and may have had a hand in dictating the overall shape of the main Patton stock and resultant mineralization.

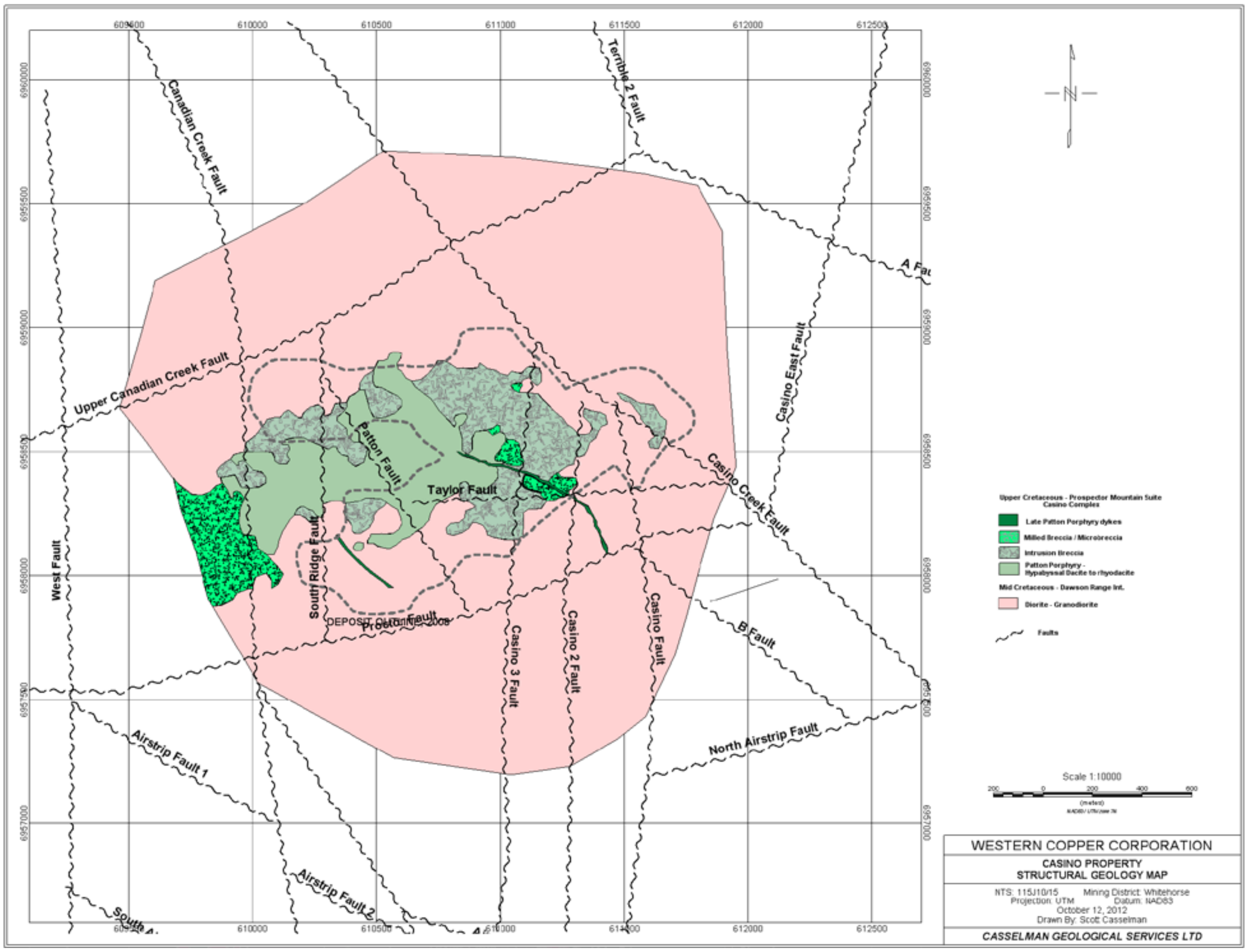


Figure 8-3: Casino Deposit Structural Geology Map

9 EXPLORATION

9.1 EXPLORATION TECHNIQUES

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

Later in the project history, 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. This section discusses exploration programs in general. Drilling is specifically discussed in Section 10.

9.2 2008-2012 EXPLORATION PROGRAM

This section describes Western Copper's exploration programs from 2008 through 2012 at the Casino Property. Prior exploration activities are described in the History Section of this report.

9.2.1 2008 Drilling Program

In 2008, Western Copper drilled three drill holes (camp water well and two exploration holes) totalling 1,163 m. The main purpose of the drilling was to obtain fresh core samples for the metallurgical and waste rock characterization tests. Both exploration holes twinned PSG's holes and their role was also to confirm historic copper, gold and molybdenum grades.

9.2.2 2009 Titan TM Geophysical Survey

Between July 16 and 29, 2009, Quantec Geoscience Limited of Toronto, Ontario performed Titan-24 Galvanic Direct Current Resistivity and Induced Polarization (DC/IP) surveys as well as a Magnetotelluric Tensor Resistivity (MT) survey over the entire grid. Magnetotelluric Resistivity results in high resolution and deep penetration (to 1 km) and The Titan DC Resistivity & Induced polarization provides reasonable depth coverage to 750 m.

The survey grid was centered on the Casino deposit covering 2.4 km by 2.4 km area. The grid consisted of nine (9) lines, spaced 300 m apart, each 2.4 km long and at an azimuth of approximately 64 degrees (perpendicular to Casino Creek Fault).

Results of Titan survey were used by Quantec to identify a series of drill targets within the survey grid and adjacent to the known mineralization. 10 holes, totalling 4,327 m, were drilled to test geophysical targets. Several distal PB-Zn veins and arsenopyrite rich veins were intercepted during this drilling, but porphyry copper mineralization wasn't found.

Results of Titan survey were later used to interpret structures within and around the Casino deposit.

9.2.3 2009 Drilling Program

In 2009, the company drilled 10,943 meters in 37 diamond drill holes. 27 holes (6,616 m) were infill holes drilled to upgrade inferred and non-defined material to measured and indicated classes. Holes CAS-004 and CAS-007 were designed to test inferred grades at the bottom of the proposed pit. Most of the infill drilling was done on the north slope of Patton Hill that was mapped as "Latite Plug" on PSG maps. Drilling has identified supergene Cu mineralization and Mo mineralization in this area.

9.2.4 2010 Drilling Program

In 2010, infill and delineation drilling continued with most of the drilling done to the North and West of the deposit. The drilling program also defined hypogene mineralization at the south end of the deposit. In addition, the company drilled a series of geotechnical holes at the proposed tailings embankment area and within the pit. Several hydrogeological holes were also drilled in 2010.

A breakdown of 2010 drilling by Western Copper is as follows:

- 46 exploration holes for 12,046.19 m
- 4 combined geotech/Hydrogeo holes for 1,170.50 m
- 6 geotech holes in the tailings embankment area for 395.21 m
- 6 Hydrogeological holes for 519.08 m
- 1 water well for 153.92 m

9.2.5 2010 Re-logging Program

From April to June 2010 all Pacific Sentinel's core stored at the Casino Property was re-logged. The purpose of the re-logging was to provide data for the new lithology and new alteration models. All previously logged lithologies were grouped in four generalized lithological groups:

- WR – Dawson Range batholith granodiorite, diorite and earlier metamorphic rocks
- PP – Patton porphyry
- IX – intrusion breccia formed between Patton porphyry and Dawson range rocks
- MX – post-mineral explosion breccia

Alteration assemblages were re-logged according to the following criteria:

- Potassic – K-spar or biotite altered rocks
- Phyllic – Sericitised, Qz-Ser and silicified rocks

- Propylitic – marked with the first appearance of secondary epidote and chlorite
- Argillic – argillic as the result of supergene weathering, not as hydrothermal alteration.

In most cases argillic Cu mineralization zones were also reinterpreted: in Western Copper holes weak assay leach assay Cu-AA05 and CN leach assay Cu-AA17a were used to define supergene zones.

- SOX = Cu grade > 0.10% and Cu-AA05 > 25% of the total Cu
- SUS = any Cu grade and Cu-AA17a >20% of the total Cu

Pacific Sentinel utilized different assay methods to determine oxide and supergene sulphide copper, as reported in 2008 Pre-Feasibility Study (M3 Engineering, 2008):

> *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).*

Since the analytical procedures used in 1994 are not commonly available, Western Copper did a comparison of the modern supergene copper assay and historic supergene copper assays. Ninety (90) samples from different parts of supergene mineralization zone were assayed for copper after:

- 3% sulphuric acid leach
- 5% sulphuric acid leach
- 5% sulphuric acid + 2% ferric sulphate leach
- Cyanide leach

Plots have shown that CuW corresponds to currently utilized Cu-AA05 and that sum of CuW and CuM factored by 1.67 corresponds to CN leach copper.

CuW = Cu-AA05
(CuW+CuM) x 1.67 = Cu-AA17a

9.2.6 2011 Exploration / Geotechnical Program

The 2011 exploration program on the Casino Project was primarily focused on geotechnical, and baseline environmental studies, however certain of the drill holes were drilled, logged and sampled for exploration purposes. The program commenced on May 16 and was completed on October 11 and involved 41 drill holes for a total of 3,163.26 m.

9.2.7 2012 Metallurgical Sampling and Geotechnical Program

In 2012, the drilling program on the property focussed on metallurgical and geotechnical studies. The program commenced on May 15 and was completed by July 11. Six holes (228.07 m) were drilled for geotechnical purposes and 5 holes (1,507.63 m) were drilled for metallurgical sampling. The program also included ongoing environmental baseline studies.

10 DRILLING

This section describes the various drilling programs developed on the Property.

10.1 1992-1994 DRILLING PROGRAM

Drilling prior to 1992 (Figure 10-1) 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 Ltd. by Archer Cathro, followed by PSG, the drilling is well documented.

During the intense campaigns from 1992 through 1994, (Figure 10-2) 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 and 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.

10.2 2008 TO 2012 DRILLING

The drilling for 2008 to 2012 exploration programs was contracted to Kluane Drilling Ltd. from Whitehorse, Yukon. Up to three hydraulic diamond drills were utilized for these programs.

Water for the drilling was pumped from Canadian Creek (Canadian Creek bend, location of the old placer camp) and from Casino Creek.

Drilling was carried out from March through November. Conditions in the late winter and fall required winter-type drilling equipment. The main challenges during the winter drilling were water supply due to the low water level in both creeks and freezing of long water lines.

All drilling was done using “thin wall” drill rods. Holes CAS-001 to CAS-007 were HTW size (core diameter 70.92 mm) and the remainder of the drilling was done primarily in NTW core size (core diameter 56.00 mm). For deeper holes, the hole would be collared in HTW and reduce to NTW generally around 200 to 300 m. In few cases holes were reduced to BTW core size (core diameter 42.00 mm).

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.

Down-hole orientation surveying was performed using an Icefield Tools MI3 Multishot Digital Borehole Survey Tool for holes CAS-002 to CAS-076. For holes CAS-077 to CAS-092 and the geotechnical and hydrogeological holes a Reflex Instruments downhole survey instrument was used.

Western Copper continued the drilling pattern established by PSG, utilizing a vertical drill hole orientation, for the most part and a nominal 100 metre grid spacing. Later in the program, Western Copper drilled a series of inclined holes in the northern part of the deposit. Several inclined holes were also drilled in the Western part of the deposit to establish contacts with the post-mineral explosion breccia and to confirm orientation of proposed N-S structure.

Geostatistics done in 2010 have shown that the 100 m spacing was sufficient for supergene mineralization which prevails in this area. The same studies have shown that the 100 m drill hole spacing is only marginally sufficient for hypogene copper mineralization.

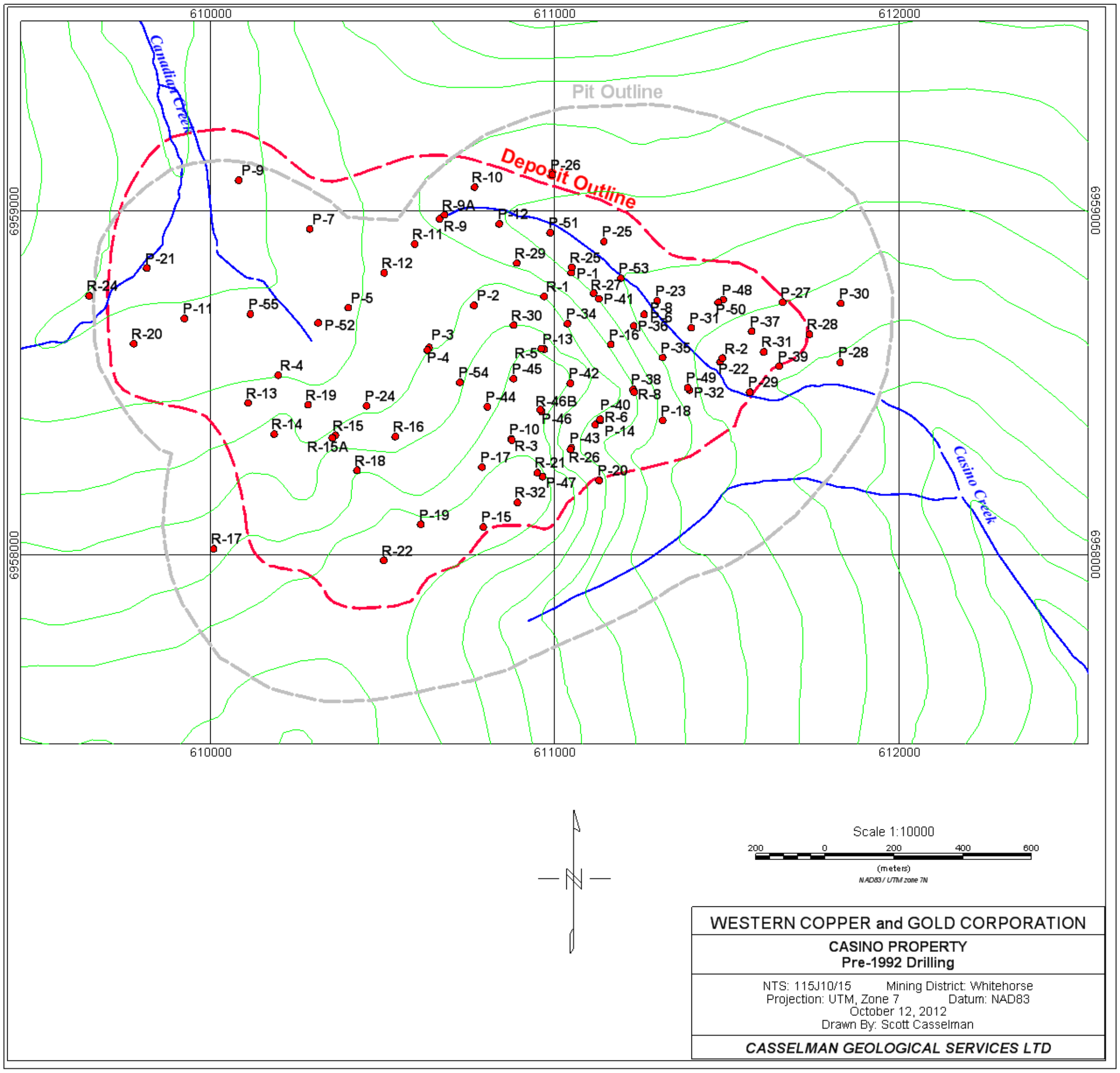


Figure 10-1: Casino Property Drilling Pre-1992

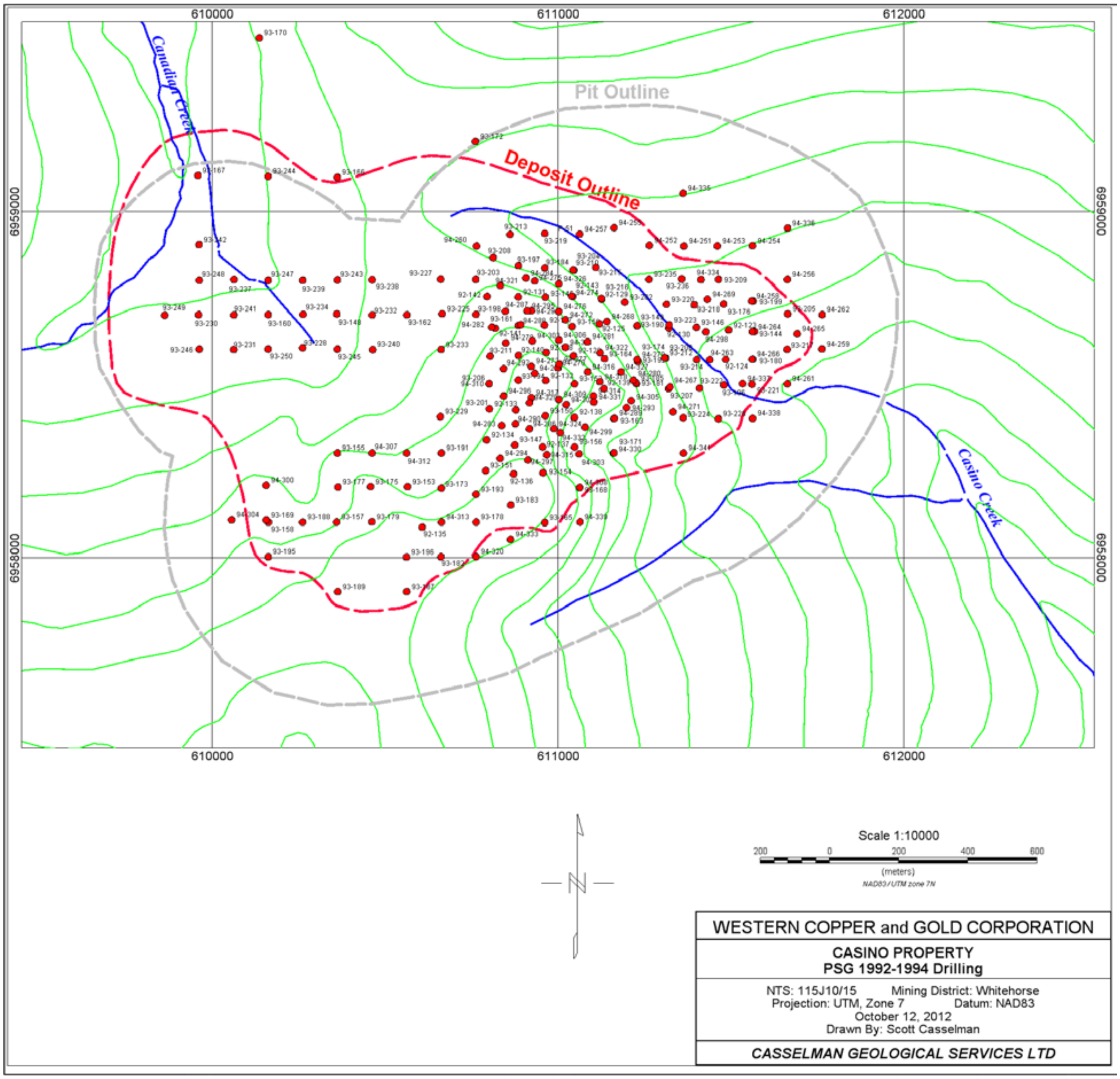


Figure 10-2: Casino Property Drilling 1992 – 1994

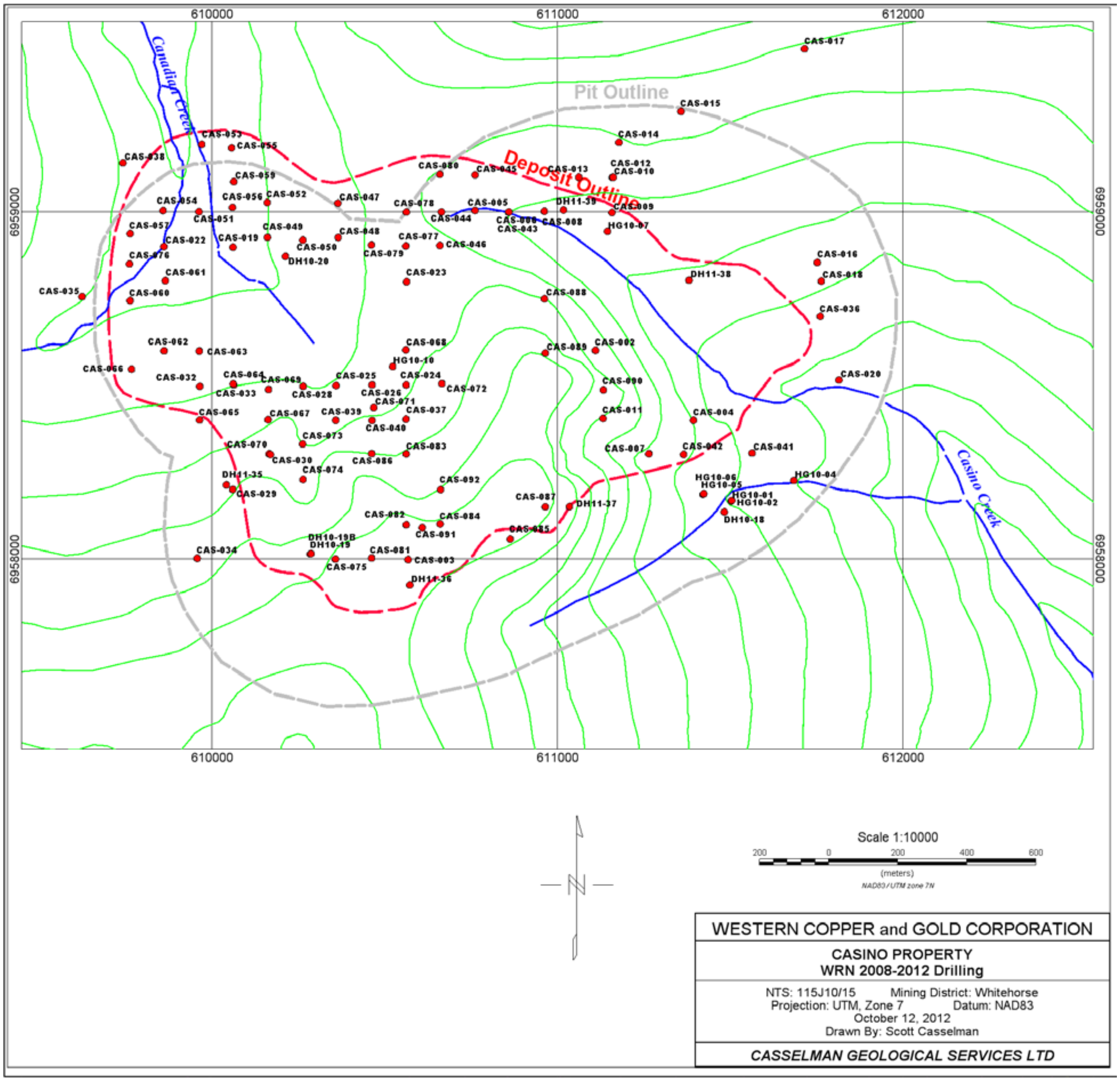


Figure 10-3: Casino Project Drilling from 2008 to 2012

10.3 SURVEY DATA-PHOTOGRAMMETRY

In April 1993 McElhanney Consulting Services Ltd. of Vancouver, BC, 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 conducted in July 1993, by Lamerton & Associates of Whitehorse. The area they covered was mapped by Eagle Mapping Services Ltd. of Port Coquitlam, BC. 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:

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.

10.4 COLLAR COORDINATES

1992 to 1994 Collar co-ordinates (Northing, Easting and elevation) were surveyed using a total station Nikon C-100. Surveying of the 1992 and 1993 drill holes was undertaken by Lamerton & Associates. 1994 holes were surveyed by Z. Peter, Surveyor from Burnaby, B.C. It should be stressed that all Pacific Sentinel's collar coordinates were surveyed in local grid coordinates.

2008-2012 drill collars were surveyed by Yukon Engineering Services from Whitehorse. The survey was done using Differential GPS units and the results are reported in UTM NAD83 coordinates.

Twenty eight (28) of Pacific Sentinel's drill hole collars were also surveyed by Yukon Engineering for comparison purposes. Those were entered in data base with their new UTM NAD83 collar coordinates.

10.5 SPERRY SUN SURVEYS

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. Tests were normally performed every 152 m (500 ft.) down hole on vertical holes and every 76 m (250 ft.) down hole on angle holes. In the shallower angle hole program of 1994, Sperry Sun tests were taken at the bottom of the hole as well as half way up.

The Sperry Sun surveys taken in the 123 vertical holes drilled or deepened in 1993 had an average dip reading of 89.03°. Since the average deviation observed in the 123 vertical holes of the 1993 program was less than one degree, it was decided not to survey the vertical holes of the 1994 program.

10.6 LIGHT-LOG SURVEY SYSTEM

A Light-Log directional drill hole survey system, developed by H.J. Otte & Co., 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. This system recorded, on film, the bending of the unit caused by the existent curvatures in a drill hole. The instrument's timer activated the camera and advanced the film at pre-set time intervals, allowing time to lower the instrument between pictures (normally every 3 m). On completion of the survey, the film was developed. The values observed on the film were converted by a computer program to provide co-ordinates, dip, and azimuth at every three metre downhole interval.

10.7 ACID DIP TESTS

In the 1994 program, acid dip tests were performed in the vertical holes while Sperry Sun surveys were continued in the angle holes.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 SAMPLING METHOD AND APPROACH

11.1.1 Core Processing

At the drill site, core was placed into wooden core boxes directly upon emptying the core tube. A wooden block marked with the depth in feet and meters was placed in the core box upon the completion of each drill run, which in good conditions was 10 feet. Core boxes were transported by drilling company truck less than 5 km to the core logging facility adjacent to the Casino Airstrip. There, core boxes were laid out in sequence on elevated tables in the core shack.

Boxes were labelled with black felt tip pens and embossed steel tags containing hole number, box number and interval of core within the box. Geotechnical data including core recovery, RQD, hardness and natural breaks were recorded for each drill run, as marked by the wooden core run blocks. This information was recorded by the geologist or trained logging assistant under direct supervision of a geologist. Logging of the geotechnical data followed codes and format outlined in a project specific manual prepared by Knight Piésold.

The geologist recorded key geologic information including lithology, zone, mineralization and alteration. The data was entered directly into a Microsoft Excel table for each drill hole. The codes and logging form followed as close as possible the format outlined in the Pacific Sentinel's logging procedures. The 4-letter lithology codes, 3-letter copper mineralization zone codes and 4-digit alteration codes used in 2008-2012 drilling programs were all developed by Pacific Sentinel and modified by Western Copper.

Core was photographed after the geology log was completed and after the sampling intervals were marked.

11.1.2 Core Sampling

Sampling and analytical protocols in use prior to the 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%. Sample intervals were marked by the geologist on the core generally at 3-meter-long intervals or at geological contacts. 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.

In 2008, all samples were split using a conventional core splitter. In 2009, about 150 samples were split with a core splitter at the beginning of the program. From then on, in 2009 through 2012, all samples were cut with a core saw. All samples were split or cut on site and placed in individually labelled plastic sample bags with the unique sample number selected by the geologist logging the hole. The core samples were split lengthwise with half of the core placed in the sample bag, the other half returned to the core box. The samples were sent to ALS Chemex Labs in North Vancouver for analysis.

In 2008, 422 drill core samples were collected; in 2009, 3,832 drill core samples were collected; in 2010, 4,768 drill core samples were collected; in 2011, 387 drill core samples were collected; and in 2012, 533 drill core samples were collected.

11.2 SAMPLE PREPARATION

All original samples were sent to ALS Chemex Labs in North Vancouver for analysis. The standard analytical request for all samples was for preparation by procedure Prep-31A. This processed involved logging the sample into the tracking system, weighing, drying and crushing the entire sample to better than 70% -2 mm. A 250 gram split of the crushed material was then collected by riffle splitter and it was pulverized to better than 85% passing 75 microns. The resultant pulp was then sent for analysis.

Duplicate samples were sent to Acme Analytical Laboratories Ltd in Vancouver for analysis. At Acme, preparation procedure R150 was used. This processed involved logging the sample into the tracking system, weighing, drying and crushing the entire sample to better than 70% passing 10 mesh. A 250 gram split of the crushed material was then collected by riffle splitter and it was pulverized to better than 85% passing 200 mesh. The resultant pulp was then sent for analysis.

Sample Standards and Blanks arrived at the labs in pulverized form. These samples skipped the crushing and pulverizing stage and went to the analysis stage.

11.3 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.

The analytical processes used at ALS Chemex and for the sample duplicates at Acme Analytical Laboratories were similar.

11.3.1 Gold Analysis

At ALS Chemex gold assays were run using 30 gram sample of the pulp with fire assay and AA finish to a 0.005 ppm detection limit according to procedure Au-AA23. Results were reported in parts per million (ppm).

At Acme gold assays were run by using a 30 gram sample of the pulp with fire assay and ICP-ES finish to a 2 ppb detection limit according to procedure Group 3B. Results were reported in parts per billion (ppb).

11.3.2 Copper, Molybdenum and Silver Assay

Samples that returned over-limits for copper, molybdenum or silver in the ICP analysis were assayed by process OG62 at ALS Chemex. This process involved a four acid digestion and analysis by Inductively Coupled Plasma-Atomic Emission Spectroscopy or Inductively Coupled Plasma-Atomic Absorption Spectroscopy. Results were reported in percent (%).

11.3.3 ICP Analysis

Samples sent to ALS Chemex were analyzed for multiple elements, including copper, molybdenum and silver by process ME-ICP61. This process involved a four acid "Near Total" digestion of 1.0 grams of sample pulp with Mass Emission-Inductively Coupled Plasma Spectroscopy for the analysis. This process returned results for: Ag (ppm), Al (%), As (ppm), Ba (ppm), Be (ppm), Bi (ppm), Ca (%), Cd (ppm), Co (ppm), Cr (ppm), Cu (ppm), Fe (%), Ga (ppm), K (%), La (ppm), Mg (%), Mn (ppm), Mo (ppm), Na (%), Ni (ppm), P (ppm), Pb (ppm), S (%), Sb (ppm), Sc (ppm), Sr (ppm), Th (ppm), Ti (%), Tl (ppm), U (ppm), V (ppm), W (ppm), and, Zn (ppm).

Samples sent to Acme were analyzed for multiple elements, including copper, molybdenum and silver by process Group 1EX. This process involved a four acid digestion of 0.25 grams of sample pulp with Mass Emission-Inductively Coupled Plasma Spectroscopy for the analysis. This process returned results for: Ag (ppm), Al (%), As (ppm), Au (ppm), Ba (ppm), Be (ppm), Bi (ppm), Ca (%), Cd (ppm), Co (ppm), Cr (ppm), Cu (ppm), Fe (%), K (%), La (ppm), Mg (%), Mn (ppm), Mo (ppm), Na (%), Nb (ppm), Ni (ppm), P (%), Pb (ppm), Sb (ppm), Sc (ppm), Sn (ppm), Sr (ppm), Th (ppm), Ti (%), U (ppm), V (ppm), W (ppm), Y (ppm), Zn (ppm) and, Zr (ppm).

11.3.4 Acid Soluble Copper Analysis

In 2008 and 2009, following receipt of the copper analyses, samples were selected for "non-sulphide" or "acid soluble" copper analysis. The criteria for "non-sulphide" selection was any sample that contained >100 ppm Cu in the Leached Cap, Supergene Zone, or top 50 m of the Hypogene Zone. A list of these samples was presented to ALS Chemex. ALS Chemex then retrieved the pulps and analyzed it by 5% sulphuric acid leach and AAS finish (procedure Cu-AA05).

In 2010 to 2012, selected samples for "acid soluble" copper analyses were identified by the geologist logging the core and the request for this analysis was submitted when the samples were originally sent to the lab. The samples identified by the geologist were generally from the top of the hole down through the top 50 m of the hypogene zone. On a few occasions, after receiving the geochemical results additional samples were identified for "non-sulphide" copper analyses and ALS Chemex was requested to pull these sample pulps and perform the analysis.

11.3.5 Cyanide Soluble Copper Analysis

In 2010, a large number of samples from the 2008, 2009 and 2010 were identified for cyanide soluble copper analyses. These samples were selected to aid with identification of the Supergene Sulphide – Hypogene metallurgical boundary. The selected samples were analyzed by cyanide leach with AAS finish (ALS Chemex procedure Cu-AA17a). For samples that had already been received and processed at the lab, ALS Chemex retrieved the pulps and analyzed this material. For samples not yet sent to the lab, the geologist would identify the Supergene Sulphide – Hypogene boundary visually and samples 30 m on either side of the boundary were identified for cyanide leach copper analysis. On a few occasions, after receiving the geochemical results additional samples were identified for cyanide soluble copper analyses and ALS Chemex was requested to pull these sample pulps and perform the analysis.

11.3.6 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).

Samples were shipped in rice bags with uniquely-numbered, non-re-sealable security tags. Each sample shipment was shipped from the Casino Property via air to Whitehorse. The samples were received at the airport by the project expediter and trans-shipped to the appropriate lab from there. In 2008 and early 2009, all shipments were sent by Byers Transport to North Vancouver. Later in 2009 and early 2010, samples for ALS Chemex were shipped by Byers Transport to the ALS Chemex preparation facility in Terrace, BC, where they were crushed and pulverized. The pulps were then shipped by ALS Chemex to North Vancouver for analyses. In May of 2010, ALS Chemex opened a preparation facility in Whitehorse. From then on, all samples were delivered to the Whitehorse preparation lab by the project expediter. The samples were crushed and pulverized in Whitehorse and the pulps were shipped to North Vancouver for analysis.

If a shipment was received with a broken security tag, the lab would notify the project manager to determine if it the shipment was tampered with, or if the tag was accidentally damaged during shipping.

11.3.7 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.

Figure 11-1 and Figure 11-2 are flow charts illustrating the processing of drill core and quality control procedures from 1992 to 1994.

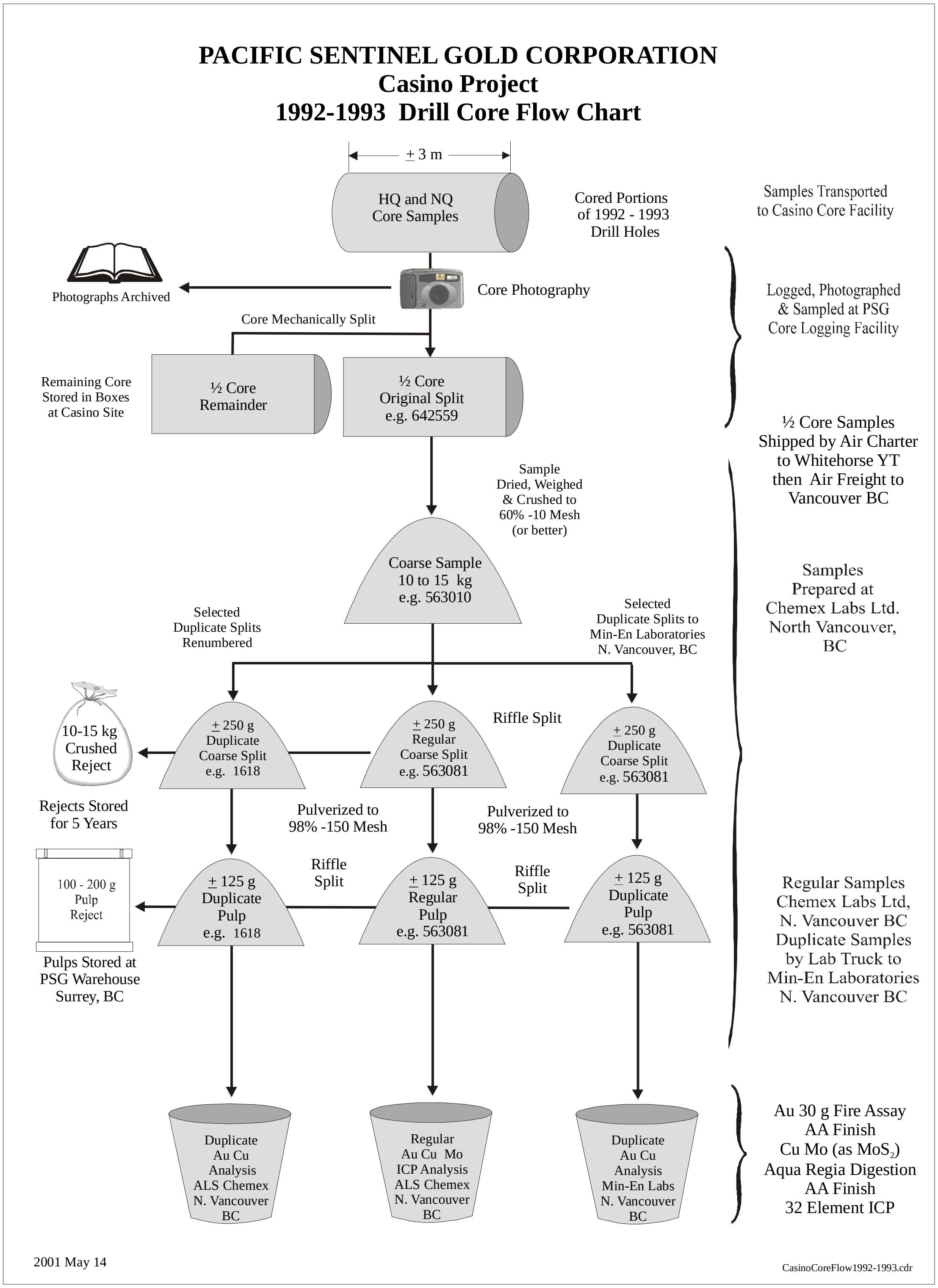


Figure 11-1: Casino Drill Core Processing and Quality Control Procedures, 1992-93

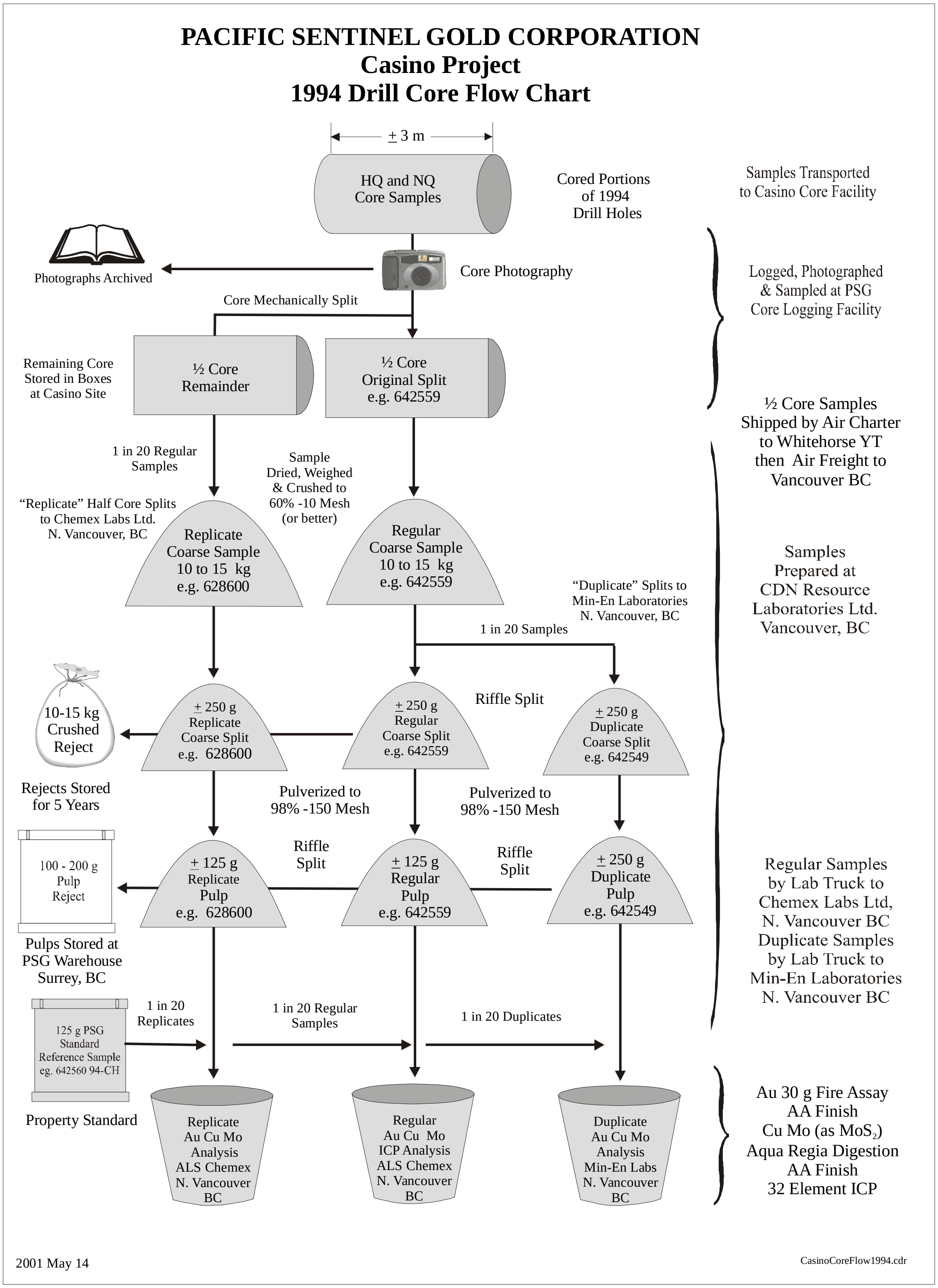


Figure 11-2: Casino Drill Core Processing and Quality Control Procedures 1994

During the 2008 through 2012 drilling programs at Casino, standards and half-core duplicates 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. The standards and blanks were prepared by CDN Resource Laboratories Ltd. of Delta, BC.

11.3.8 Sample Standards

The sample standard used in 2008, 2009 and 2010 was prepared by CDN Resource Laboratories Ltd. of Delta, BC. The standard is a gold-copper-molybdenum standard, CDN-CM-4. It is certified by Duncan Sanderson, Licensed BC Assayer with independent certification by Dr. Barry Smee, Ph.D., geochemist. Round robin assaying for the standard was performed at 12 independent laboratories. CDN reports the recommended values and the "Between Lab" Two Standard Deviations of the standard values of:

Gold: 1.18 + 0.12 g/t
Copper: 0.508 + 0.025 %
Molybdenum: 0.032 + 0.004 %

In 2008, 8 sample standards were submitted regularly with the sample shipments; in 2009, 81 standards were submitted; and in 2010, 86 standards were submitted (approximately 1 every 50th sample). ALS Chemex analyzed the standards along with the drill core samples by gold, copper and molybdenum assay, as well as multi-element ICP as described above.

The results from sample standard CDN-CM-4 for 2008, 2009 and 2010, for gold, copper and molybdenum analyses are plotted below.

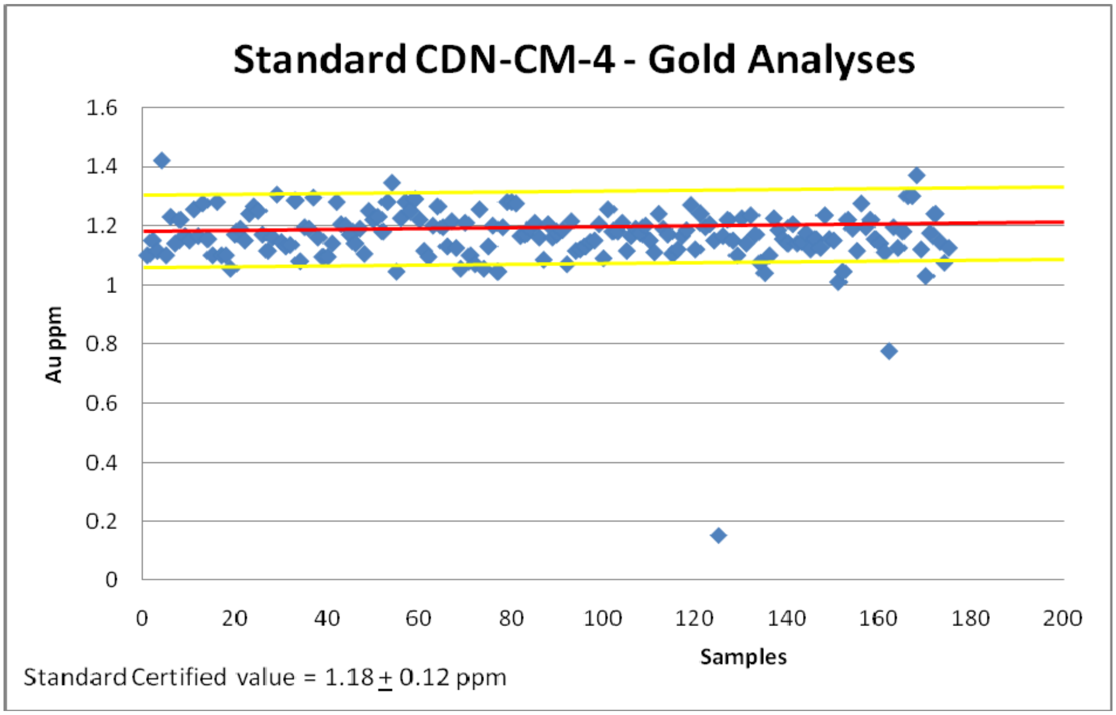


Figure 11-3: Sample Standard CDN-CM-4 Gold Assay Results

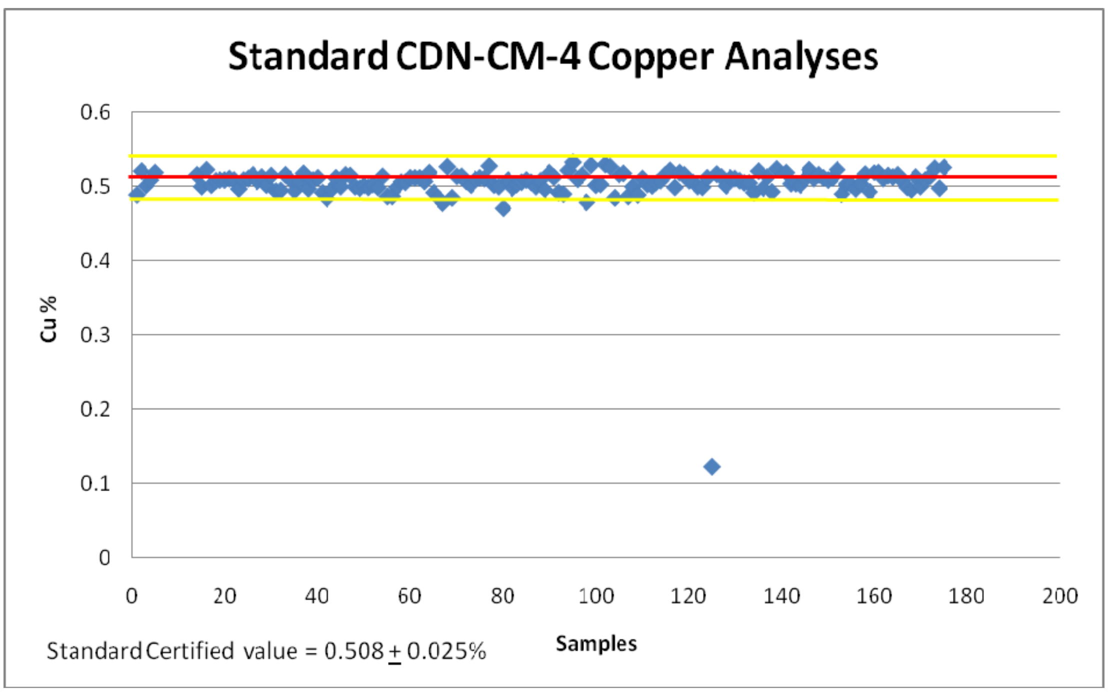


Figure 11-4: Sample Standard CDN-CM-4 Copper Assay Results

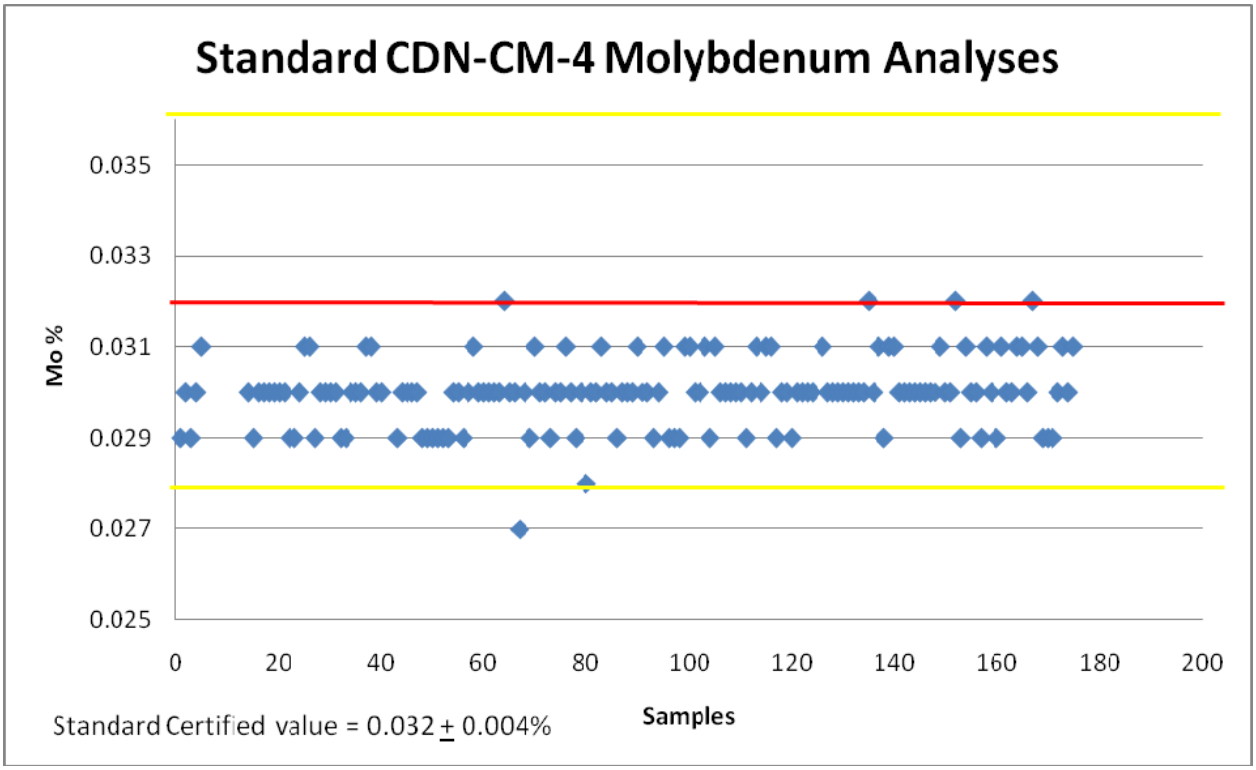


Figure 11-5: Sample Standard CDN-CM-4 Molybdenum Assay Results

The three plots demonstrate that with very few exceptions (9 for gold, 2 for copper, and one for molybdenum), the values plot within the acceptable range of the certified standard. The plots also demonstrate that there is a reasonable spread of values within the recommended value range of 2 standard deviations as provided by CDN Resource Laboratories Ltd. There does not appear to be any systematic bias.

Later in 2010, a second sample standard (CDN-CM-7) was purchased from CDN Resource Laboratories Ltd. because they had run out of standard CDN-CM-4. This sample is also certified by Duncan Sanderson and Dr. Barry Smee. CDN reports the recommended values and the "Between Lab" Two Standard Deviations of this standard as:

Gold: 0.427 + 0.042 g/t
Copper: 0.445 + 0.027 %
Molybdenum: 0.027 + 0.002 %

Fifteen of these standards were submitted in 2010. ALS Chemex analyzed these standards in the same manner as standard CDN-CM-4, described above.

The results from sample standard CDN-CM-7 for 2010, for gold, copper and molybdenum analyses are plotted below:

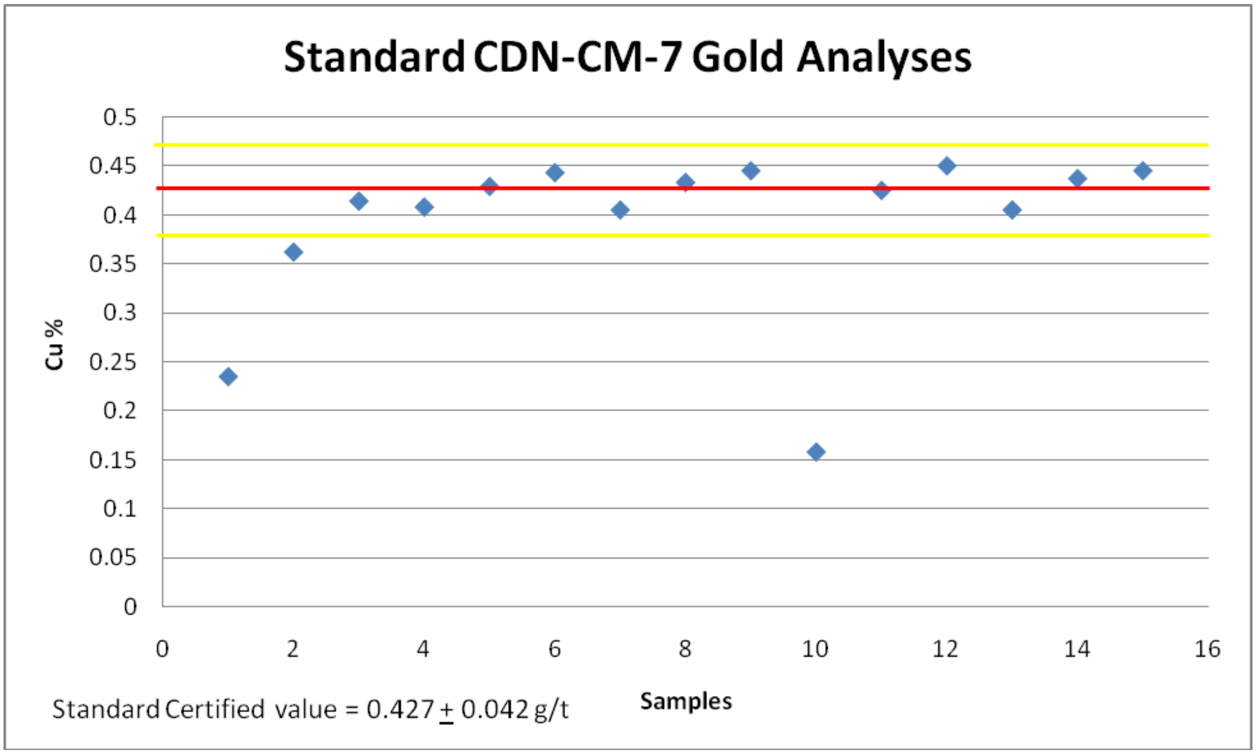


Figure 11-6: Sample Standard CDN-CM-7 Gold Assay Results

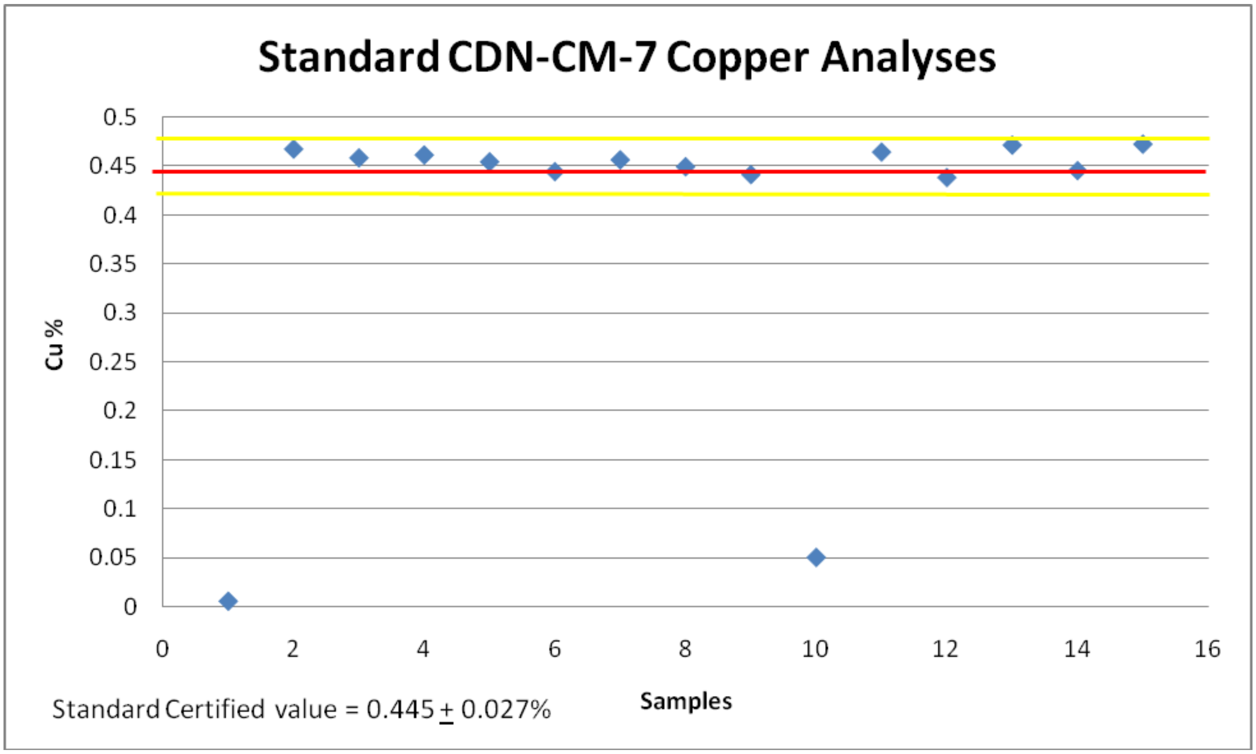


Figure 11-7: Sample Standard CDN-CM-7 Copper Assay Results

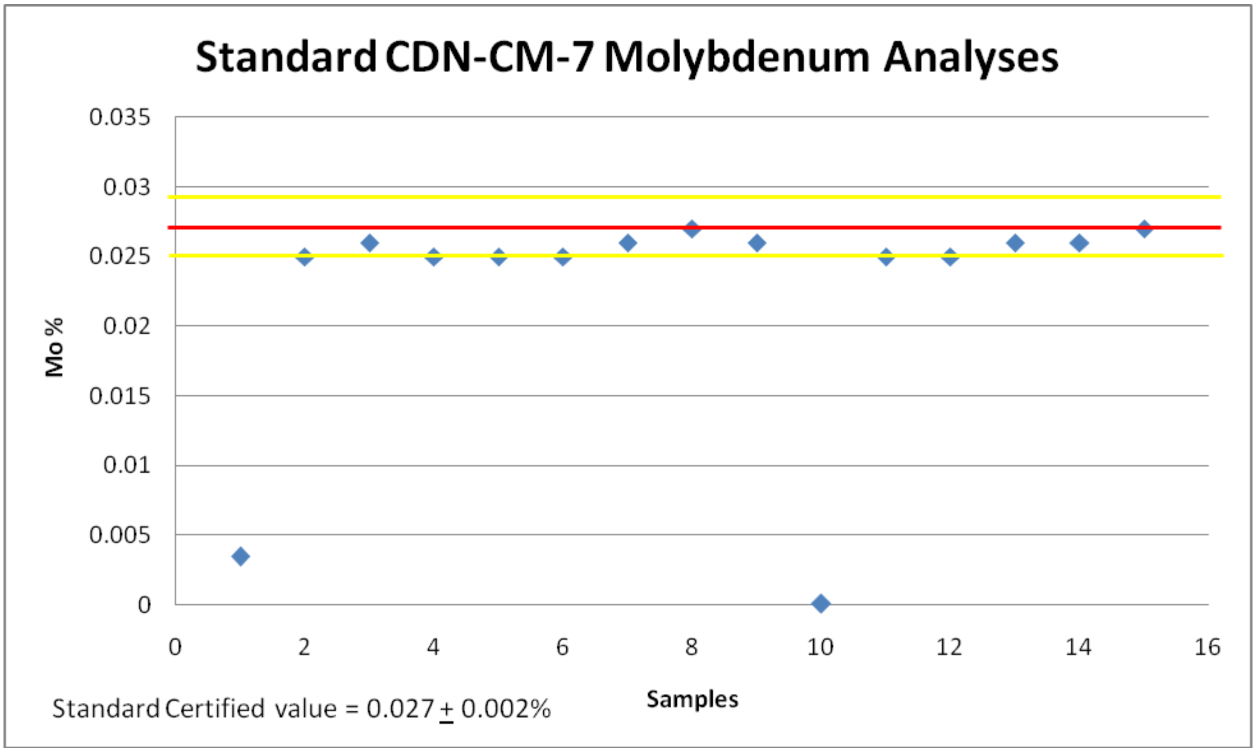


Figure 11-8: Sample Standard CDN-CM-7 Molybdenum Assay Results

The three plots show good precision with the exception of samples 1 and 10 which are well below the expected values as certified by CDN. After checking the ALS Chemex internal standards for these batches and the sample duplicates from these batches of analyses there did not appear to be a systemic error in the batches. The suspicion is that an error may have occurred when the sample standards were inserted in the field, or when the standard was originally placed in the geochemical run at the lab. These anomalous errors are not considered significant considering that, in general, the great majority of standards returned analysis within the expect range.

11.3.9 Blanks

Commencing in 2010 sample Blanks were regularly inserted into the sample stream. Blanks are included as a check of the lower limit of the analytical range and to ensure that at all stages in the process, the equipment and instruments are thoroughly cleaned prior to running subsequent samples. This is particularly important for precious metals. A total of 75 blanks were submitted during the 2010 program, nominally one every 50 samples.

The Blank was also prepared by CDN Resource Laboratories Ltd (CDN-BL-6). It is certified for gold, platinum and palladium. The recommended values for these elements are:

Gold: <0.01 g/t
Platinum: <0.01 g/t
Palladium: <0.01 g/t

Since the reported recommended gold values by CDN are less than detection it is not included in a plot. The gold values of the Blanks analyzed ranged from below detection (<0.005 g/t) to a maximum of 0.046 g/t. The silver values ranged from <0.5 to 0.8 ppm; copper values ranged from 48 to 85 ppm; and molybdenum from 1 to 10 ppm.

11.3.10 Field Duplicate Drill Core Analysis

Field duplicates are separate samples taken in the same manner and at the same core interval as the original sample. They are used to measure inherent variability in metal content from a single location and sample medium, and give an idea of sample reproducibility in the field. Core duplicates were collected from the ½ core that remained following the collection of the original sample. The duplicate was collected by sawing the ½ core in half longitudinally, so that ¼ of the original core was collected. Duplicates were collected, nominally every 20th sample. Where duplicates were collected, only ¼ of the core remains stored in the core box on the property.

In 2008, 21 core duplicate pairs were collected; in 2009, 199 core duplicate pairs were collected; in 2010, 245 core duplicate pairs were collected. The original ½ core sample was shipped to ALS Chemex for analysis for gold, copper and molybdenum assay, as well as multi-element ICP as described above. The duplicate ¼ core sample was shipped to Acme Labs for analysis for gold, copper and molybdenum assay, as well as multi-element ICP in a manner identical to that performed at ALS Chemex, as described above. The results for the duplicate analyses for gold, silver, copper and molybdenum are demonstrate in comparison plots between the Acme and ALS Chemex values below:

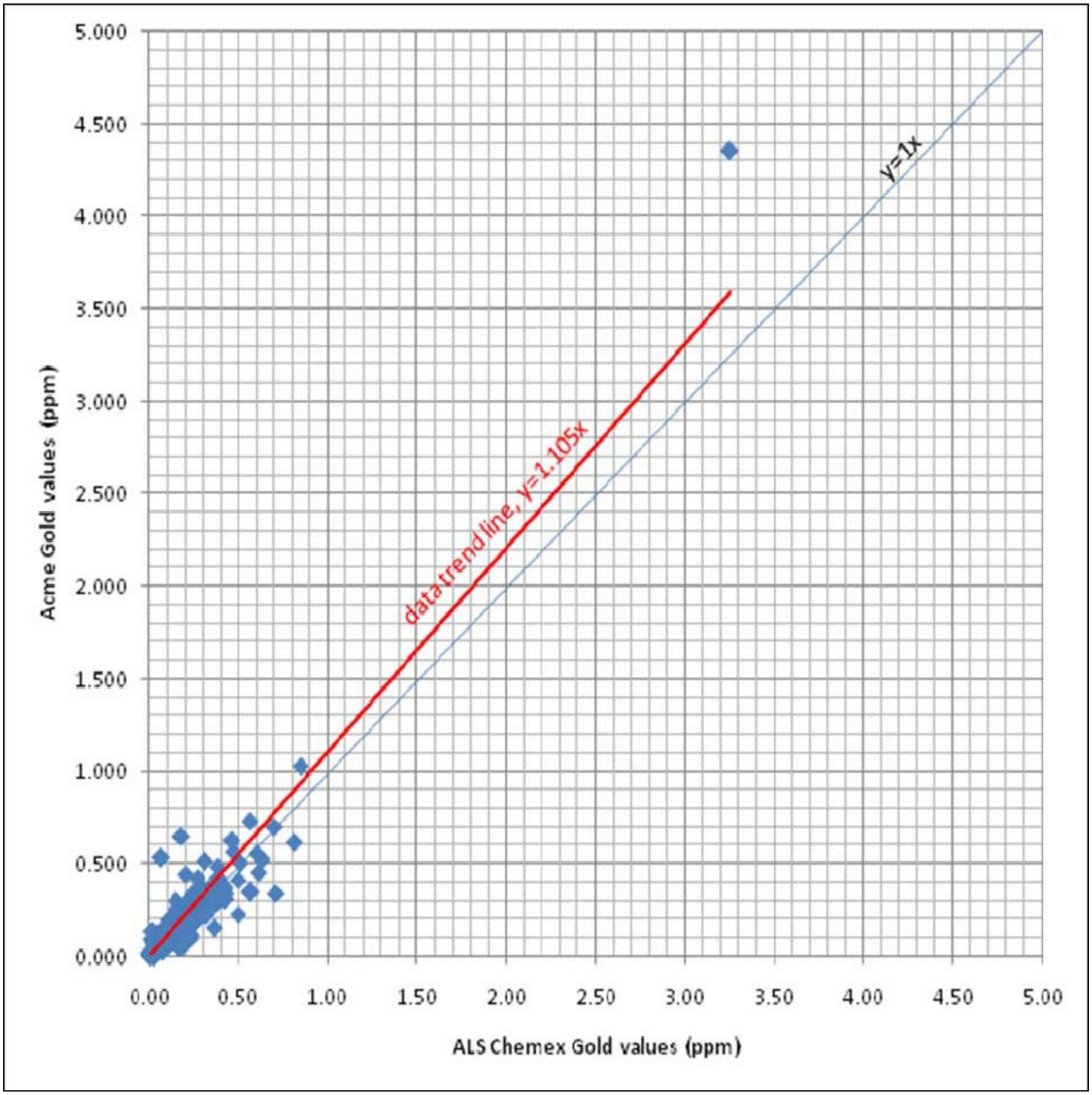


Figure 11-9: Plot of ALS Chemex gold assay versus Acme Labs gold assay for field duplicate samples (2008, 2009 and 2010 data)

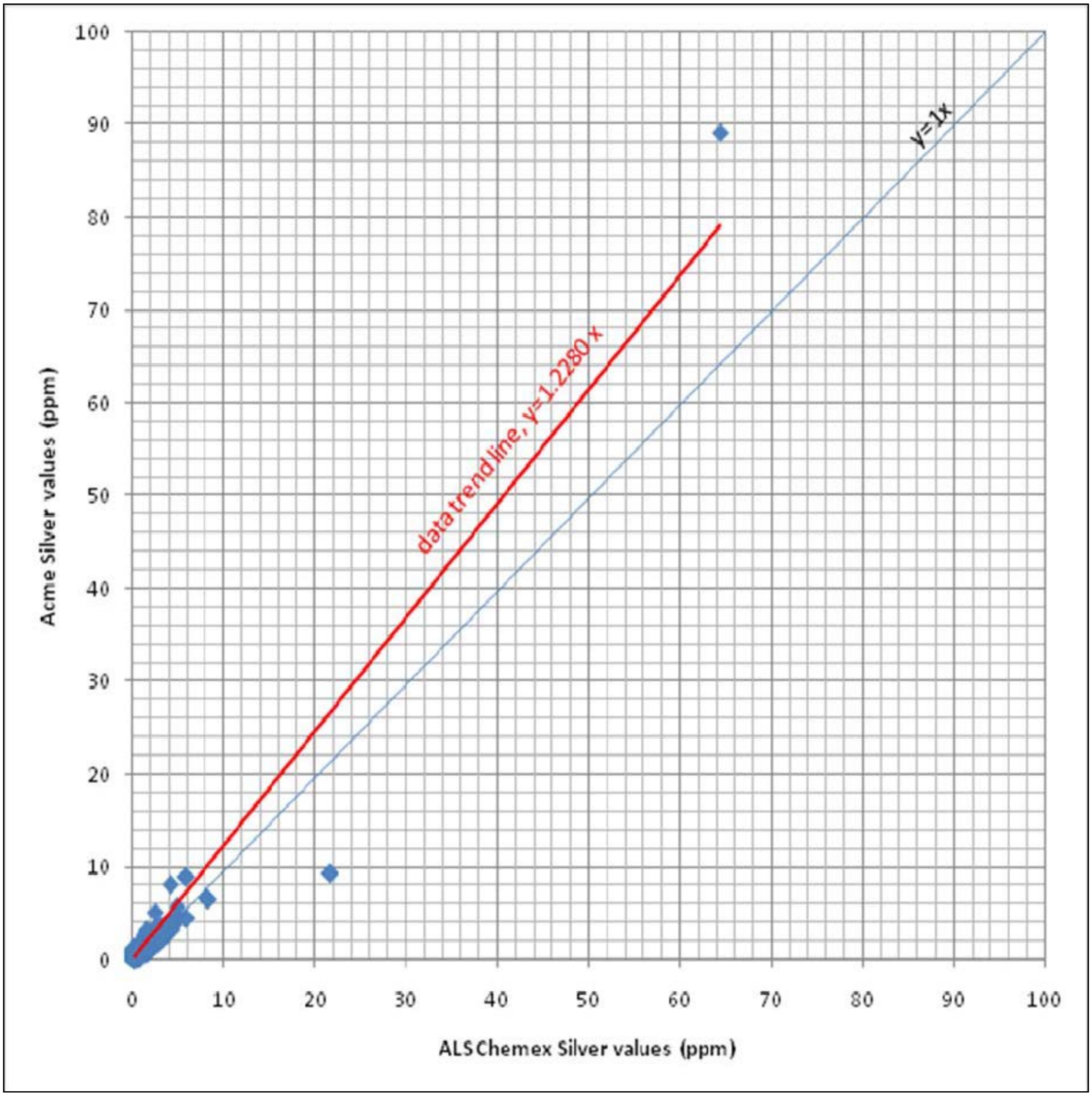


Figure 11-10: Plot of ALS Chemex silver analyses versus Acme Labs silver analyses for field duplicate samples (2008, 2009 and 2010 data)

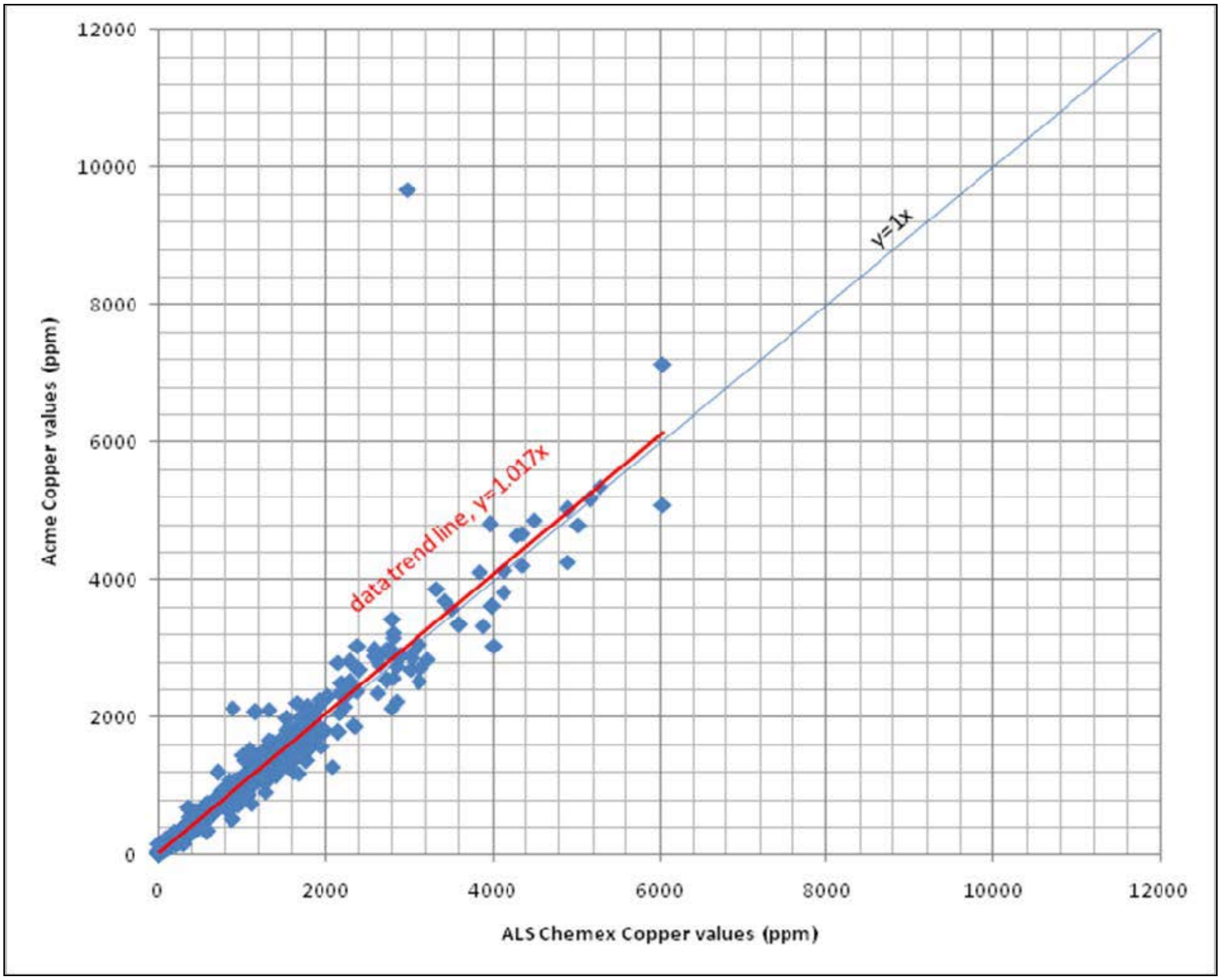


Figure 11-11: Plot of ALS Chemex copper assay versus Acme Labs copper assay for field duplicate samples (2008, 2009 and 2010 data).

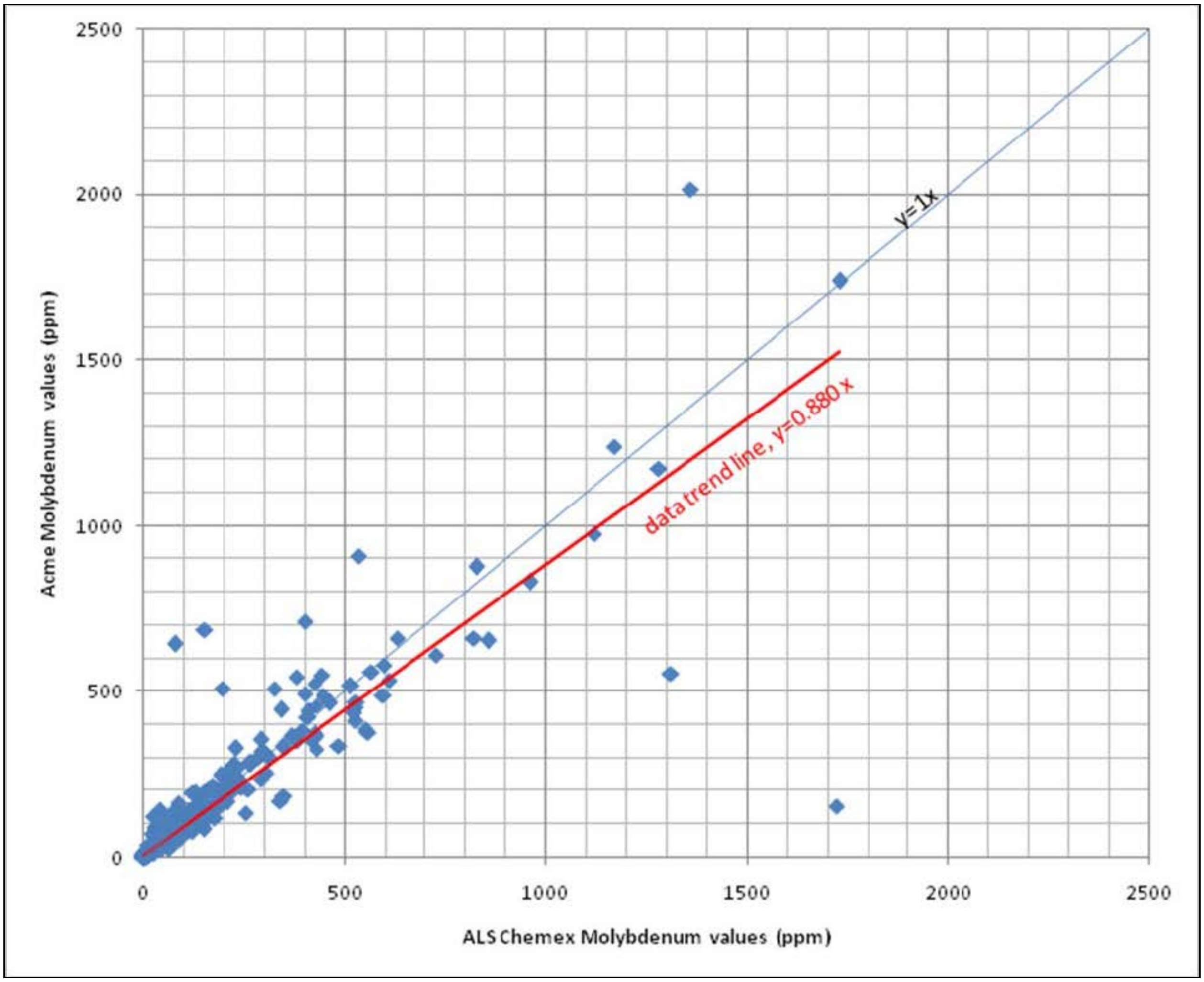


Figure 11-12: Plot of ALS Chemex molybdenum assay versus Acme Labs molybdenum assay for field duplicate samples (2008, 2009 and 2010 data).

The plots generally show good correlation between ALS Chemex and Acme Labs for all four elements of interest.

Often the "nugget effect" associated with gold and silver content will produce widely divergent values, which would plot as highly scattered data points. However, the gold and silver results from the duplicate samples show good correlation.

Ideally, a trend line of y=1x would show perfect reproducibility. This is rarely, if ever, the case due to the difference of mineral content between duplicate samples. The data trend line for gold returned y=1.105x. This demonstrates that Acme Lab results, as a whole, are 10.5% higher than ALS Chemex results. All samples cluster in close proximity to the trend line which indicates no strong "nugget effect" and good reproducibility.

The data trend line for silver is y=1.228x. This demonstrates that Acme Lab analyses, as a whole, are 22.8% higher than ALS Chemex values. In general, the points cluster well around the trend line with the exception of one sample. This also demonstrates good reproducibility.

The results for duplicate analyses for copper demonstrate excellent reproducibility. The data trend line returned y=1.017x. The copper data clusters tightly around trend line with the exception of one value. In general, the Acme results are very slightly higher (1.7%) than the ALS Chemex results.

The molybdenum plot demonstrates slightly more scattered results with 8 points plotting far off the trend line (y=0.880x). The trend line indicates that, in general, the Acme results for molybdenum are 12% lower than ALS Chemex results. Overall, however the duplicate results show good correlation. Molybdenite mineralization was observed in quartz veins in the drill core and it is possible that the 8 erratic values are reflecting a molybdenum "nugget effect", where there is a variability of molybdenite concentration between samples.

The results of analyses from the sample standards, blanks and duplicates provide for acceptable Quality Assurance and Quality Control (QA-QC) for the geochemical program at Casino for 2008 through 2010. The results also indicate that there is no evidence of tampering during the sample collection process, shipping or at the laboratory. There is also no evidence of systemic errors in the sample preparation and analytical processes.

12 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 and verification for historic data used in the updated resource estimate was documented in the Casino Project Pre-Feasibility Study, Yukon Territory, Canada, prepared by M3 Engineering and Technology Corp in 2008. Data entry and verification for the drilling programs conducted by Western Copper Corporation from 2008 through 2010 is documented below. The Western Copper database was verified on an ongoing basis by the Scott Casselman, Project Manager, 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 coordinates and down hole surveys, for the 2008, 2009 and 2010 diamond drill holes.

12.1 DATA ENTRY

Drill hole logging, sampling and geotechnical data was entered directly by the geologist or geotechnical logger working on the core in a Microsoft Excel spreadsheet. Upon completion of each hole these files was submitted to the Project Manager for checking. Upon receipt of analytical data from the lab, the data was merged with the sample intervals by the Project Manager and the data was checked.

Down hole survey information was recorded digitally by the down hole survey instruments and the dumped data was submitted by the operator to the Project Manager. This data was checked by the Project Manager. Hole collar surveying was performed by surveyors from Yukon Engineering Services Ltd and this data was provided to the Project Manager for inclusion into the project data base. This data was checked by the Project Manager.

Geotechnical data such as Specific Gravity measurements, Rock Quality Determinations (RQD), and core box intervals was first recorder on paper then transcribed into the computer.

Sample interval data was entered directly into the computer with the hole number and interval being also written in the sample booklets. The booklets were checked in the event of a discrepancy in sample intervals.

All the data was brought into Microsoft Excel for Windows spreadsheets where the data was organized into a standardized format. Once the data was checked it was posted on the Western Copper FTP site for use in the office. The data was merged into Geosoft Target software database for creation of drill plans and sections and for 3D modelling.

Original 1992 and 1993 field data was entered by Archer Cathro and Assoc. and by Nowak and Assoc., both of Vancouver, BC. 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.

12.2 VERIFICATION PROCEDURE

The data verification process was performed under the supervision of the Project Manager. When errors were observed in geological, geotechnical or sample intervals, the Project Manager and geologist or technician would go back to the core and/or original notes or sample booklets and sort out the error and make appropriate corrections. Data verification was performed on an ongoing basis and kept as up to date as possible. At times where data were first recorded on paper, then transcribed to the computer, original copies of the hand notes were kept for future reference. All original data is kept in the Project Files.

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' side logs, synoptic logs, drillers' time sheets, and/or core photographs were referred to and a decision was made by a geologist familiar with logging and sampling techniques.

12.3 ERRORS

The errors encountered during the data verification process varied depending upon the data being checked. Overall there were very few errors in the database. The most common error observed was in geological or sample intervals, where the "To" recording of a previous sample did not match the "From" recording of the subsequent sample. These were generally easy to sort out by the geologist or geotechnical logger.

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.

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. Additionally, 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. Since the re-logging of the historic core in 2010, these errors associated with geological, mineralogical or alteration, have been eliminated.

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 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 original or in transcription. Wherever possible computer checks were done on the data; several types of errors were detected this way.

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.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

The Casino Project will produce copper flotation concentrates with contained gold and silver values, and molybdenite flotation concentrates. 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.

13.1 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 2008 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.

Samples for the G&T Metallurgical Services test program reported in early 2011 were split from fresh core from the 2010 drill program.

Samples for the comminution testing performed by Starkey and Associates, and comminution and flotation testing by G&T Metallurgical reported in early 2012 were retrieved from the 1993 to 2010 drill programs and consisted of split core.

A drill program to retrieve fresh hypogene core was completed in early 2012 and split core from this drilling campaign was used for the flotation tests reported by G&T Metallurgical in December 2012.

13.2 LEACHING TESTS

13.2.1 Kappes, Cassiday and Associates

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. 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.

13.2.2 METCON

METCON ran two column tests on a composite sample blended to create gold and copper concentrations similar to the average reserve concentrations.

The ore was crushed coarsely to -3.8 cm (-1.5 inch), placed in 15 cm by 6 meter 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.

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.

Cyanide consumptions were similar based on titrations and the amount of cyanide added to the system for the locked cycle column at ~0.5 kg/t. Lime consumptions were similar to the bottle roll test work at approximately 3 kg/t.

Table 13-1: 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

13.3 SART COPPER RECOVERY

SART stands for Sulphidization, Acidification, Recycling 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 removing the gold with carbon and recycling the treated fluid to the column. The SART results are summarized in Table 13-2.

Table 13-2: SART Results

Pregnant Solution				Barren Solution after SART & Carbon				Copper	Reagent Consumption (g/L solution treated)		
Free NaCN (g/L)	Cu (ppm)	Au (ppm)	Ag (ppm)	Free NaCN (g/L)	Cu (ppm)	Au (ppm)	Ag (ppm)	Removal (%)	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

13.4 COMMINUTION TESTING

SGS Lakefield, under the direction of SGS Minnovex, performed a comprehensive comminution study. Fifty (50) split drill core samples, representing the first 6 years of production were sent to SGS and subjected to the several tests.

A summary of the grinding results appears in Table 13-3. As SGS reports, 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 13-3: 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

Additional comminution testing was performed in 2012 under the direction of FLSmidth and Starkey and Associates at G&T Metallurgical Services and FLSmidth laboratories. This program tested 11 composites of ore representing a combination of different zones, lithologies and alterations. The 11 composites represent over 80% of the material that will be processed through the mill.

Ore composite types that were not tested were mapped to similar composites that were tested by CMC geologists.

The 11 comminution composites were subjected to a series of tests at G&T Metallurgical's laboratory and FLSmidth's laboratory. The test results are summarized in the following tables.

Table 13-4: Summary of SMC Tests and JK Parameters

Sample ID	DWi, kWh/m³	DWi, %	Mia, kWh/t	Mih, kWh/t	Mic, kWh/t	A	B	SG	t_a
Composite 1	4.90	39	15.5	10.8	5.6	56.7	0.95	2.64	0.53
Composite 2	4.35	32	14.1	9.6	5.0	56.3	1.07	2.63	0.59
Composite 3	6.05	55	18.6	13.5	7.0	61.8	0.70	2.60	0.43
Composite 4	6.62	63	19.8	14.6	7.6	62.3	0.64	2.63	0.39
Composite 5	6.69	64	19.9	14.7	7.6	63.4	0.62	2.64	0.39
Composite 6	3.92	26	13.3	8.8	4.6	62.9	1.05	2.58	0.66
Composite 7	5.75	51	18.1	13	6.7	66.4	0.67	2.58	0.45
Composite 8	5.60	49	16.9	12.1	6.2	64.1	0.75	2.69	0.46
Composite 9	5.00	40	16.1	11.3	5.8	67.9	0.76	2.58	0.52
Composite 10	9.63	90	26.3	20.9	10.8	91.3	0.30	2.67	0.27
Composite 11	5.69	50	17.6	12.6	6.5	66.4	0.69	2.62	0.46

Table 13-5: Summary of SAGDesign Results and Crushed Bond Test Results

Sample ID	DML SAGDesign Test Results			G&T Crushed Bond Test Results			
	Relative Density	Calc W_{SAG} to 1.7 mm (kWh/t)	SAG Dis. Bond BWi (kWh/t)	BWi (kWh/t)	RWi (kWh/t)	Ai (g)	CWi
Composite 1	2.66	8.19	16.18	13.5	12.9	0.162	9.41
Composite 2	2.60	6.78	17.26	14.1	12.3	0.176	10.00
Composite 3	2.66	9.39	15.70	14.1	14.5	0.198	13.62
Composite 4	2.72	12.41	18.36	15.5	15.5	0.199	13.84
Composite 5	2.64	9.56	18.26	15.3	14.6	0.156	11.20
Composite 6	2.67	5.05	16.37	13.7	10.4	0.118	10.22
Composite 7	2.69	7.45	16.12	13.4	12.4	0.155	14.57
Composite 8	2.82	7.71	17.82	15.2	14.1	0.170	12.27
Composite 9	2.57	6.48	14.35	12.9	11.4	0.158	11.03
Composite 10	2.71	11.68	18.93	16.6	14.9	0.161	13.23
Composite 11	2.67	8.50	17.23	15.1	13.5	0.170	10.33
Average	**2.67**	**8.47**	**16.96**	**14.5**	**13.3**	**0.166**	**11.79**

A circuit consisting of one 40 ft diameter (12.2 m) SAG mill and two 28 ft diameter (8.2 m) ball mills in closed circuit with three pebble crushers was selected, based on discussions with M3 and FLSmidth, as a circuit that would likely meet the design tonnage. This circuit was modeled by FLSmidth using the parameters developed by SGS, G&T Metallurgical, and FLSmidth. The results of this exercise are shown in Table 13-6.

Table 13-6: Predicted Production Rate

Project Sample Number	Client Sample Information	BWi G&T (kWh/t)	Production Rate (mtpd)
1	Composite 1	13.5	133,805
2	Composite 2	14.1	128,064
3	Composite 3	14.1	128,064
4	Composite 4	15.5	116,582
5	Composite 5	15.3	118,018
6	Composite 6	13.7	131,818
7	Composite 7	13.4	134,798
8	Composite 8	15.2	118,790
9	Composite 9	12.9	139,987
10	Composite 10	16.6	108,854
11	Composite 11	15.1	119,674
Average		**14.5**	**125,314**

13.5 FLOTATION

13.5.1 2008 G&T Metallurgical Work

In 2008 Western Copper and G&T Metallurgical reviewed the previous metallurgical work and developed a new flotation program. In order to prevent oxidation, the program used split drill core rather than assay rejects as had been done for the previous work.

The new work 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. Assays for these composites are shown in Table 13-7.

Table 13-7: G&T Flotation Composite Assays

	Cu(%)			Mo (%)		Fe	Au
	Total	WAS	CNS	Total	AS	(%)	(g/t)
Oxide Composite	0.275	0.132	0.042	0.019	0.006	3.225	0.345
Sulphide Composite	0.260	0.016	0.032	0.021	0.002	3.525	0.255

13.5.1.1 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. Based on the poor performance of the oxide flotation, no further testing on the oxide composite was performed.

13.5.1.2 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.

Copper recoveries of 70-82% were obtained into concentrates grading from 26.8 to 32.2% copper in cleaner tests. Good recovery of copper was obtained with both a primary grinds with K80's of 147 and 121 µm, and regrinds with K80's less than 22 µm. A coarser grind with a K80 of 209 µm was examined in rougher tests and shown to be less favourable than the finer particle sizes selected for cleaner testing.

13.5.1.3 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 indicate that a grind with a K80 of 147 µm, 85.6% copper can be recovered into a 28.5% copper concentrate. Molybdenum recovery was variable and ranged from 26.5% to 69.4%. Gold recovery was more consistent and averaged 64.0%.

13.5.1.4 Variability Testing

A total of 63 individual split drill core intervals were tested for variability. These samples were chosen to primarily represent the first 6 years of production and covered a broad range of total copper, acid soluble copper, molybdenum and gold values. Each of these samples was individually ground and floated in a cleaner test with regrind under the conditions determined from the locked cycle tests.

13.5.2 2009 - 2011 G&T Metallurgical Work

13.5.2.1 2009 Fresh Core Tests

In 2009, a new drilling campaign was initiated which included two holes in the middle of the deposit – CAS-002 and CAS-003. A composite from CAS-002 had 92% copper recovery into a concentrate grading about 28% copper in cleaner tests. Similarly, a composite from CAS-003 had 87% of the copper in the feed recovered into a concentrate grading 26% copper. Moly recoveries were high in both tests at approximately 90%.

13.5.2.2 2010 Supergene Sulphide Composite Tests

The material tested in the 2010 test program (reported at the beginning of 2011) was a composite of supergene material that was obtained from the drilling campaigns in 2009 and 2010. This material represented ore that will be fed to the mill in the later years of the operation. The feed grade averaged 0.30% copper and 0.037% molybdenum.

One of the main objectives of the 2010 test program was to evaluate coarser grinds than were tested in the 2008 test program. Results of this evaluation indicate that copper flotation response is virtually unaffected by primary grind size between 142 and 253 µm for this composite. Molybdenum flotation recovery to the bulk rougher concentrate was lower at grinds coarser than 179 µm. Molybdenum recovery was also reduced at elevated pH levels.

13.5.2.3 2010 Supergene Sulphide Composite Locked Cycle Tests

Locked cycle tests at primary grind K80's of 142 µm and 222 µm were performed. The results from these tests are presented in Table 13-8. The effect of regrind size on bulk concentrate copper grade and the effect of primary grind and regrind size on moly recovery are indicated in the table.

Table 13-8: Locked Cycle Test Results

Test	P. Grind K80 µm	Regrind K80 µm	Cycle	Assay - percent or g/t				Distribution - percent			
				Cu	Mo	Fe	Au	Cu	Mo	Fe	Au
KM2721-33	222	19	IV	30.8	1.6	23.6	20.2	82.9	34.7	5.2	71.9
KM2721-33	222	19	V	28.2	1.4	26.5	19.9	81.6	34.2	6.2	69.7
KM2721-34	222	20	IV	26.1	1.6	25.7	17.8	88.6	48.8	7.0	68.4
KM2721-34	222	20	V	25.7	1.5	26.6	19.9	86.6	45.1	7.5	64.7
KM2721-35	142	19	IV	26.3	1.9	27.8	18.6	87.3	57.1	7.1	71.4
KM2721-35	142	19	V	25.1	1.7	27.6	16.1	86.4	54.3	7.6	66.1
KM2721-36	222	37	IV	17.8	1.4	31.1	13.1	81.7	55.7	9.6	67.0
KM2721-36	222	37	V	18.8	1.4	30.4	10.1	82.8	51.2	10.9	61.5
KM2721-37	222	31	IV	21.2	1.4	31.1	11.7	83.2	54.1	9.2	62.4
KM2721-37	222	31	V	20.8	1.7	31.3	11.7	83.9	59.7	9.9	65.7

13.5.2.4 Pyrite Flotation

Pyrite flotation was examined as a process to produce tailings samples that had low levels of residual sulphur, and thus could be deemed "not acid generating".

The locked cycle tests outlined in Table 13-8 included a pyrite rougher to reduce the sulphide concentration of the tailings. Pyrite flotation tailings from these tests obtained tailings averaging less than 0.08% sulphur.

13.5.3 2011 – 2012 G&T Metallurgical Work

WCGC retained International Metallurgical and Environmental to assist in the metallurgical testing, and continued to perform the testing at G&T Metallurgical Services (name changed to ALS Metallurgy in late 2012).

13.5.3.1 New Flowsheet Development

In previous testing campaigns, in order to achieve acceptable recoveries from the conventional copper flotation flowsheet's tested, 15-20% of the feed material needed to be reground. The focus of the new flowsheet development was to reduce the material sent to the regrind mills.

The new flowsheet development centered on a flowsheet where rougher concentrate was sent to the first cleaning stage *prior* to regrinding, the first cleaner *concentrate* went to regrinding and the second and third cleaner tails were returned to the first cleaner. By utilizing this flowsheet, the amount of feed material that needed to be reground dropped from 15-20% to 3 to 5%.

Locked cycle test results from the composites tested using this flowsheet are shown in Table 13-9. The results show similar recoveries to previous test work using a conventional copper flotation flowsheet.

Table 13-9: Locked Cycle Test Results

Composite	Tests	P. Grind K80 µm	Regrind K80 µm	Assay - percent or g/t Cu	Mo	S	Au	Distribution - percent Cu	Mo	S	Au
HYP1	38, 42	218	19	26.0	1.98	33.1	24.6	82.1	64.9	24.9	61.1
HYP2	39, 43	216	16	26.3	1.31	32.8	23.9	81.7	37.1	14.6	56.1
SUS1	44, 46	192	17.5	21.8	1.77	33.9	23.6	77.7	59.5	24.9	75.9
SUS2	47	190	14	24.1	0.85	38.1	28.3	62.8	32.8	20.4	64.4

13.5.3.2 Tests using Fresh Core

While supergene flotation tests were performed on fresh core obtained during the 2010 campaign, no flotation tests had been performed on fresh hypogene core with the exception of a limited number of tests performed in 2009.

In 2012, a drilling campaign was executed to obtain fresh hypogene core from the first years of mining that represented the predominate mineralization that would be fed to the mill. In total five holes were drilled (CAS-088 to CAS-093), and from these five holes, three composites were made representing lithologies: Patton porphyry (PP), Intrusion breccia (IX), and Dawson range batholith (WR).

Table 13-10: Hypogene Composites

	Cu(%) Total	WAS	CNS	Mo (%)	Fe (%)	Au (g/t)
PP Composite	0.14	0.004	0.008	0.030	2.95	0.22
IX Composite	0.17	0.006	0.012	0.071	2.39	0.22
WR Composite	0.19	0.005	0.013	0.019	2.50	0.18

Locked Cycle Recoveries using these fresh composites were significantly better than previous testing on oxidized core and are shown in Table 13-11. Note that the primary grind size for these tests was also higher than the target of 200 µm, in some cases significantly, so it would be expected that actual plant recovery would be better than these tests indicate.

Table 13-11: Locked Cycle Test Results

Composite	Tests	P. Grind K80 µm	Regrind K80 µm	Assay - percent or g/t Cu	Mo	Ag	Au	Distribution - percent Cu	Mo	Ag	Au
PP	23	234	31	18.6	7.5	126	15.6	89.9	77.9	46.5	57.3
IX	24	254	32	24.6	4.3	107	24.3	87.2	78.6	46.0	55.4
WR	25	211	31	17.5	1.50	82	13.5	91.9	89.4	53.8	67.2

13.5.3.3 Pilot Plant Testing and Copper/Molybdenum Separation

A pilot plant was performed on hypogene and supergene composites taken from the drilling campaign to produce representative tailings for environmental testing, geotechnical testing and thickener testing and to produce sufficient copper/molybdenum concentrate for copper moly separation tests. Unfortunately, there was not sufficient feed material to obtain operating information from the pilot plant.

Although suitable copper/molybdenum concentrate was produced to perform several copper/molybdenum separation tests, only one cleaner test was performed as the results from this test were sufficiently good to warrant no further testing. The results from this test are shown in Table 13-12.

Table 13-12: Copper/Molybdenum Separation Cleaner Test

Cumulative Product	Cum. Weight		Assay - percent or g/t				Distribution - percent			
	%	grams	Cu	Mo	Fe	S	Cu	Mo	Fe	S
Final Conc.	3.1	31.2	0.39	57.4	0.8	37.9	0.1	94.1	0.1	2.6
Second Conc.	3.5	35.5	2.38	51.3	3.8	37.6	0.5	95.7	0.4	3.0
Rougher Conc.	6.1	62.5	9.04	29.7	15.3	38.0	3.5	97.4	2.9	5.3
Tails	93.9	953.8	16.5	0.05	33.9	44.3	96.5	2.6	97.1	94.7
Feed	**100.0**	**1016.3**	**16.0**	**1.87**	**32.8**	**43.9**	**100**	**100**	**100**	**100**

13.5.4 Interpretation of Flotation Test Results

The most current work at G&T Metallurgical has shown good copper recovery to copper concentrates that routinely achieve 28% or greater for various drill core samples from the deposit using the reagent scheme developed. The conclusions from this work are unambiguous and will be used as the basis of this study.

13.5.4.1 Supergene – Copper

It was difficult to achieve good copper concentrate grades from supergene oxide material that had copper oxide concentrations greater than 25-30% of the total copper. For this reason, during operation of the mill, supergene oxide ore should be blended in with the other ore to achieve an oxide copper percentage less than 25%.

The supergene ore contains a certain percentage of oxide copper minerals (this is what defines it as being supergene material). Oxide copper minerals will be poorly recovered by the flotation process, so in the interpretation of the results, it is important to examine the recovery of *sulphide copper* to a copper concentrate. Sulphide copper can be calculated by subtracting the concentration of oxide copper from the total copper. Supergene mineralization at Casino has been assayed for *weak acid soluble copper*, which is approximately equal to the amount of oxide copper in the sample assayed, but may under or over represent the amount of oxide copper present depending on the specifics of the mineralization.

Sulphide copper recovery as a function of and sulphide copper grade is shown in Table 13-13 for the supergene locked cycle tests by G&T Metallurgical. Recovery appears to be fairly consistent.

Table 13-13: Supergene Locked Cycle Recoveries to Concentrate

	Feed Assays					Recovery to Concentrate			
	Cu (%)			Au	Mo	Total	Sulphide		
Test	**Total**	**WAS**	**Sulphide**	**(g/t)**	**(%)**	**Cu**	**Cu**	**Au**	**Mo**
KM2721									
33	0.3	0.03	0.27	0.25	0.036	82.3	91.4	70.7	34.5
34	0.3	0.03	0.27	0.25	0.036	87.6	97.3	66.4	46.9
35	0.3	0.03	0.27	0.25	0.036	86.8	96.4	68.8	55.7
36	0.3	0.03	0.27	0.25	0.036	82.3	91.4	64.4	53.3
37	0.3	0.03	0.27	0.25	0.036	83.5	92.8	64.1	57
KM3134									
44	0.3	0.056	0.244	0.37	0.022	79.9	98.2	75.5	64.6
46	0.3	0.056	0.244	0.47	0.022	75.6	93.0	76.1	54.6
47	0.3	0.094	0.206	0.47	0.028	62.8	91.5	64.4	32.8

Averaging the locked cycle tests results indicates that an average of 94% of the sulphide copper was recovered to a copper concentrate. This result also closely mirrors the variability results. Thus the overall copper recovery for the supergene material will be:

$$\text{Cu Recovery} = 94 \times (Cu_{total} - Cu_{WAS})/(Cu_{total})$$

13.5.4.2 Supergene – Gold

Averaging the gold recovery from Table 13-13, an average gold recovery of 69% to copper concentrate is obtained:

Au Recovery = 69%

13.5.4.3 Supergene – Molybdenum

In most of the tests, no attempt was made to optimize the molybdenum recovery. For this reason, the molybdenum recovery is quite variable.

Examining the locked cycle tests in Table 13-13, an average molybdenum recovery of 55% to copper concentrate was chosen, which represents the average molybdenum recovery when the two low outliers are removed.

Recovery of molybdenum from the copper-molybdenum concentrate to a molybdenum concentrate was not specifically tested for the supergene material, but it is expected to be similar to that obtained in hypogene tests that achieved approximately 95% molybdenum recovery to a molybdenum concentrate. Molybdenum recovery throughout the plant is equal to the recovery to the copper-molybdenum concentrate multiplied by recovery to a molybdenum concentrate and is shown below:

Mo Recovery = 52.25%

13.5.4.4 Supergene – Silver

Unfortunately, silver recovery was not determined in all test programs. The 2011 test program followed silver. Averaging the silver recovery from these locked cycle tests indicates that a silver recovery of 60% should be achievable:

Ag Recovery = 60%

13.5.4.5 Hypogene

Hypogene recoveries are based on the December 2012 flotation work performed by ALS Metallurgy on "fresh" core that had been drilled earlier specifically for flotation test work.

The following table shows cleaner circuit recoveries for both copper and molybdenum for all three locked cycle tests with hypogene material. Copper concentrate grades have been corrected to reflect the removal of molybdenum and represent final concentrate grades in terms of copper.

Table 13-14: Cleaner Circuit Recoveries for Locked Cycle Test Results

Test and Cycle no.	Cu Con Grade %Cu	Cu Recovery %	Mo Recovery %	Au Recovery %
WR Composite				
Cycle 4	17.8	96.4	95.0	86.0
Cycle 5	17.9	96.9	95.6	88.0
IX Composite				
Cycle 4	22.8	96.9	81.7	83.3
Cycle 5	21.2	96.7	80.8	80.6
PP Composite				
Cycle 4	26.1	97.1	90.4	88.0
Cycle 5	26.5	97.1	91.1	84.9

Copper, molybdenum and gold recovery, when a primary grind size of 200 to 220 µm is used is summarized in Table 13-15 and is based on both locked cycle testing and open circuit rougher flotation tests. Molybdenum recovery was variable and the higher grade molybdenum sample (IX) had the lowest molybdenum recovery indicating that reagent conditions could possibly improve this recovery. Within the cleaning circuit copper and gold recoveries were very constant, irrespective of the final copper concentrate grade.

Table 13-15: Predicted Recoveries to Copper/Molybdenum Concentrate

Process Stream	Cu	Mo	Au
Rougher Circuit Recovery	95	92	78
Cleaner Circuit Recovery	97	90	85
Metal Recovery	**92.15**	**82.8**	**66**

13.5.4.6 Hypogene – Copper-Molybdenum Separation

One test was performed to determine how well molybdenum could be separated from a copper/molybdenum concentrate, which indicated that approximately 95% molybdenum recovery could be achieved. Thus the overall recovery of molybdenum will be equal to:

Mo Recovery = 78.6%

13.5.4.7 Hypogene – Silver Recovery

Hypogene silver recovery was followed in the last set of tests on fresh core. Reviewing these recoveries a silver recovery of 50% was chosen.

Ag Recovery = 50.0%

13.6 DEWATERING TESTS

Flotation tailing from the 2008 test program piloting were submitted to Outotec for dynamic high rate thickening tests. Results were favourable and a thickener underflow of over 55 percent solids was achieved. Flocculant addition was 22 g/t. The solids loading rate of 1.05 t/m^2h was demonstrated. Rheology on the thickened material was low.

13.7 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 based on M3's experience were used:

Table 13-16: Heap Leach Operational Parameters

Parameter	Value	Units
Gold recovery	66	Percent
Copper recovery	18	Percent
Silver recovery	26	Percent
Crush size	-1	inch
Irrigation rate	12	$L/h/m^2$
Lift height	8	m
Reagent consumptions		
NaHS	0.025	kg/t ore
Sulfuric acid	0.328	kg/t ore
Hydrochloric acid	0.010	kg/t ore
Lime (CaO)	3.270	kg/t ore
Sodium hydroxide	0.130	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
Flocculent	0.350	g/t ore
Primary crusher liners	0.040	kg/t
Secondary crusher liners	0.085	kg/t

Table 13-17: Flotation Operational Parameters

Parameter	Value	Units
Copper recovery		
Supergene	Recovery = 94 x (Cu_{total} – Cu_{WAS})/(Cu_{total})	percent
Hypogene	92.15	percent
Gold recovery		
Supergene	69	percent
Hypogene	66	percent
Molybdenum recovery (final conc)		
Supergene	52.25	percent
Hypogene	78.6	percent
Silver recovery		
Supergene	60	percent
Hypogene	50	percent
Bond work index	14.5	kWh/t
Primary grind size (P80)	200	µm
Regrind size (P80)	25	µm
Reagent consumptions		
Lime		
Supergene	2.5	kg/t ore
Hypogene	1.0	kg/t ore
Aerophine 3418A		
Supergene	8.4	g/t ore
Hypogene	4.0	g/t ore
Aerofloat 208		
Supergene	16.7	g/t ore
Hypogene	8.0	g/t ore
MIBC	10	g/t ore
Fuel Oil	7.44	g/t ore
PAX	40	g/t ore
NaSH	0.053	kg/t ore
Flocculent	25.4	g/t ore
SAG Mill – Liners	0.040	kg/t ore
Ball Mill – Liners	0.048	kg/t ore
SAG Mill – Balls	0.400	kg/t ore
Ball Mill – Balls	0.400	kg/t ore
Regrind – Balls	0.0410	kg/t ore

14 MINERAL RESOURCE ESTIMATES

14.1 KEY RESULTS

Table 14-1 summarizes the mineral resources for the Casino Project.

Table 14-1: Mineral Resources

Supergene and Hypogene Zones (Mill Resource)	Cutoff CuEq (%)	Ore Mtonnes	Copper (%)	Gold (g/t)	Moly (%)	Silver (g/t)	CuEq (%)
Measured Mineral Resource							
Supergene Oxide	0.25	25	0.28	0.52	0.026	2.38	0.77
Supergene Sulphide	0.25	36	0.39	0.41	0.029	2.34	0.84
Hypogene	0.25	32	0.32	0.38	0.026	1.94	0.73
Total Measured Resource	**0.25**	**93**	**0.34**	**0.43**	**0.027**	**2.21**	**0.78**
Indicated Mineral Resource							
Supergene Oxide	0.25	36	0.23	0.21	0.019	1.44	0.46
Supergene Sulphide	0.25	216	0.24	0.22	0.019	1.72	0.50
Hypogene	0.25	711	0.17	0.21	0.023	1.65	0.45
Total Indicated Resource	**0.25**	**963.6**	**0.19**	**0.21**	**0.022**	**1.66**	**0.46**
Measured/Indicated Mineral Resource							
Supergene Oxide	0.25	61	0.25	0.34	0.022	1.83	0.59
Supergene Sulphide	0.25	252	0.26	0.25	0.021	1.81	0.55
Hypogene	0.25	743	0.17	0.22	0.023	1.66	0.46
Total Measured/Indicated Resource	**0.25**	**1057**	**0.20**	**0.23**	**0.022**	**1.71**	**0.49**
Inferred Mineral Resource							
Supergene Oxide	0.25	26	0.26	0.17	0.010	1.43	0.41
Supergene Sulphide	0.25	102	0.20	0.19	0.010	1.49	0.38
Hypogene	0.25	1568	0.14	0.16	0.020	1.36	0.37
Total Inferred Resource	**0.25**	**1696**	**0.14**	**0.16**	**0.019**	**1.37**	**0.37**
Leached Cap/Oxide Gold Zone (Heap Leach Resource)	Cutoff Gold (g/t)	Ore Mtonnes	Copper (%)	Gold (g/t)	Moly (%)	Silver (g/t)	CuEq (%)
Measured Mineral Resource	0.25	31	0.05	0.52	0.025	2.94	N.A.
Indicated Mineral Resource	0.25	53	0.03	0.33	0.017	2.36	N.A.
Measured/Indicated Resource	**0.25**	**84**	**0.04**	**0.40**	**0.020**	**2.57**	N.A.
Inferred Mineral Resource	0.25	17	0.01	0.31	0.008	1.93	N.A.
CuEq is based on metal prices of US$2.00/lb copper, US$875/oz gold, US$11.25/lb molybdenum, and US$11.25/oz silver and assumes 100% metal recovery.							

The supergene oxide, supergene sulphide, and hypogene zones are mill resources and are tabulated at a 0.25% copper equivalent cutoff grade. Measured and indicated supergene and hypogene resources amount to 1.06 billion tonnes at 0.20% copper, 0.23 g/t gold, 0.022% molybdenum, and 1.71 g/t silver. Inferred resources is an additional 1.7 billion tonnes at 0.14% copper, 0.16 g/t gold, 0.019% molybdenum, and 1.37 g/t silver.

The leach cap contains potential heap leach ore and is tabulated at a 0.25 g/t gold cutoff grade. Measured and indicated heap leach ore amounts to 84.0 million tonnes at 0.04% copper, 0.40 g/t gold, and 2.57 g/t silver. Inferred resource is an additional 17 million tonnes at 0.01% copper, 0.31 g/t gold, and 1.93 g/t silver.

Copper equivalent (CuEq) is determined using the following metal prices: Cu - US$2.00/lb, Au - US$875.00/oz, Ag - US$11.25/oz and Mo - US$11.25/lb and is calculated as follows:

CuEq % = (Cu %) + (Au g/t x 28.13/44.1) + (Mo % x248.06/44.1) + (Ag g/t x 0.36/44.1)

The copper equivalent calculation reflects gross metal content and does not apply any adjustment factors for difference in metallurgical recoveries of gold, copper, silver and molybdenum.

The current resource estimation for Casino was based on 305 drill holes, 34 of which were completed in 2009 and an additional 56 completed in 2010. In addition, WCGC geologists re-interpreted the geologic model during the 2010 field season, re-logging older Pacific Sentinel drill holes from drilling in the 90's. Finally the collar coordinates previously reported in Mine Grid Units were converted by Yukon Engineering Services to NAD83 UTM coordinates.

Analysis of the drill data with contact plots showed copper values within the leached cap, supergene horizons and hypogene needed to be evaluated independently while Au, Ag and Mo could be combined. In addition the lithologies Intrusive Breccia, Patton Porphyry and Dawson Range Granodiorites had higher grades for all variables than the post mineral explosion breccia and were estimated separately. Erratic high values for all variables were capped based on their grade distributions within the various mineral domains. Uniform down-hole 15 m composites were produced that honored the domain boundaries.

14.2 DATA ANALYSIS

A total of 305 drill holes within the area of interest and totaling 95,655 m were provided by WCGC for analysis. Assays reported as less than some detection limit were assigned a value of ½ that limit. Assays reported as not sampled (NS) were left blank. A total of 348 gaps in the assay record at surface (OB) or within the hole were assigned a nominal value of Cu = 0.001%, Au= 0.002 g/t, Mo= 0.0001% and Ag = 0.1 g/t.

WCGC geologists reassessed the Casino geologic model during the 2010 field season. The lithologic model and oxidation zone models were digitized from cross sections and three dimensional surfaces were created in GemCom software. The contacts were joined to produce three-dimensional geologic solids to control the grade estimation procedure.

14.3 COMPOSITES

Uniform down hole composites 15 m in length were produced for each of the mineralized Domains. Intervals less than 7.5 m at the domain contacts were combined with adjoining samples to produce a uniform support of 15 ± 7.5 m. The composite statistics are tabulated below.

Table 14-2: 15 m Composite Statistics Sorted by Mineral Domain

Variable	Domain	Number	Mean	S.D.	Min.	Max.	C.V.
Cu (%)	CAPCU1	1,027	0.038	0.032	0.001	0.217	0.85
	WAS CU	666	0.012	0.016	0.001	0.167	1.28
	CAPCU2	52	0.014	0.021	0.003	0.113	1.44
	WAS CU	20	0.006	0.005	0.001	0.022	0.94
	SUPCU1	1,697	0.266	0.188	0.003	1.750	0.71
	WAS CU	1,644	0.055	0.060	0.001	0.919	1.09
	SUPCU2	33	0.106	0.074	0.017	0.267	0.70
	WAS CU	33	0.017	0.011	0.002	0.050	0.67
	HYPCU1	3,307	0.168	0.128	0.002	1.513	0.76
	HYPCU2	114	0.056	0.043	0.005	0.318	0.77
Au (g/t)	ALLREST	6,034	0.249	0.213	0.003	3.328	0.85
	ALLMX	196	0.127	0.125	0.009	0.629	0.98
Mo (%)	ALLREST	6,034	0.020	0.025	0.0001	0.521	1.22
	ALLMX	196	0.005	0.007	0.0001	0.040	1.40
Ag (g/t)	ALLREST	6,034	1.70	2.51	0.10	80.80	1.47
	ALLMX	196	1.48	0.89	0.30	5.13	0.60

Note: WAS Cu refers to Weak Acid Soluble Cu assays taken in Leached Cap and Supergene zones

14.4 VARIOGRAPHY

A variogram study using pairwise relative semivariograms was completed for all the domains west of the Casino Fault. The similar units east of the fault did not have enough data to determine variography and as a result the west models were used for all east domains. In each case semivariograms were produced in the four principal directions of E-W, N-S, SW-NE and NW-SE. These were used to determine the maximum continuity in the horizontal plane. Vertical or down hole semivariograms were produced to establish the nugget effect and the vertical continuity. In almost all cases the maximum continuity was in the east-west direction. In all cases nested spherical models were fit to the data. The semivariogram parameters are tabulated below.

Table 14-3: Semivariogram Parameters for Cu, Au, Mo and Ag in Domains

DOMAIN	VARIABLE	AZ / DIP	C_0	C_1	C_2	Short Range (m)	Long Range (m)
CAPCU1 Leached Cap	Cu	080 / 0	0.05	0.10	0.35	40	180
		350 / 0	0.05	0.10	0.35	40	50
		0 / -90	0.05	0.10	0.35	60	140
SUPCU1 Supergene	Cu	012 / 0	0.05	0.06	0.14	30	150
		282/ 0	0.05	0.06	0.14	30	80
		0 / -90	0.05	0.06	0.14	30	50
HYPCU1 Hypogene	Cu	090 / 0	0.05	0.05	0.10	80	100
		0 / 0	0.05	0.05	0.10	50	80
		0 / -90	0.05	0.05	0.10	100	200
ALL REST	Au	155 / 0	0.08	0.04	0.10	40	90
		65/ 0	0.08	0.04	0.10	30	40
		0 / -90	0.08	0.04	0.10	50	140
	Mo	155 / 0	0.08	0.05	0.18	40	84
		65/ 0	0.08	0.05	0.18	20	34
		0 / -90	0.08	0.05	0.18	40	140
	Ag	135 / 0	0.10	0.05	0.08	30	50
		45 / 0	0.10	0.05	0.08	20	40
		0 / -90	0.10	0.05	0.08	40	74

14.5 BULK DENSITY

An extensive specific gravity data base has been compiled for Casino over the years with a total of 11,600 measurements taken on drill core by the Wt. in Air-Wt. in Water method. The results are tabulated below as a function of oxidation zone.

Table 14-4: Bulk Density

Domain	Number	Minimum SG	Maximum SG	Average SG
Leached Cap	1,986	1.62	3.59	2.52
Supergene Oxide	797	1.61	3.74	2.58
Supergene Sulphide	3,442	1.46	3.85	2.62
Hypogene	5,375	1.26	4.67	2.65
Total	11,600	1.26	4.67	2.62

Specific gravity increases with depth in general and as a function of alteration. As a result, the average specific gravity for each alteration zone was used to determine a weighted average for each block in the model.

*Block SG = ((%Cap*2.52) + (%SOX*2.58) + (%SUS*2.62) + (%HYP*2.65))/ %Below Topo - % OVB*

14.6 BLOCK MODEL

A block model with blocks 20 x 20 x 15 m was superimposed over the geologic solids. The block model was coded for lithology on a bench by bench basis by WCGC using the majority rule for each block. Once the model was coded for lithology it was compared to the various surfaces namely: surface topography, overburden, leached cap, supergene oxide, supergene sulphide and hypogene with the percentage of each recorded for each block. The block mode origin was as follows:

Lower Left Corner		
609492.73 E	Column size = 20 m	138 Columns
6957332.60 N	Row size = 20 m	113 Rows
Top Model		
1530 Elevation	Level size = 15m	72 Levels
No Rotation		

14.7 GRADE INTERPOLATION

Grades for Cu, Au, Mo and Ag were interpolated into blocks by Ordinary Kriging. Copper was estimated for Leached Cap, Supergene and Hypogene zones for both the combined IX, PP and WR lithologies and the MX lithology in separate runs. For each domain hard boundaries were used so that, for example, in the Leached Cap portion of blocks within the MX lithology, only Leached Cap Composites within MX were used. The kriging exercise was completed in each Domain in four passes. The maximum number of composites used, from one hole, was set to 3, insuring that for all estimated blocks a minimum of two drill holes were required. In all passes, if more than 16 composites were found, the closest 16 were used. The copper grade for any given block was the weighted average of Leached Cap Cu, Supergene Cu and Hypogene Cu.

An estimate for weak acid soluble copper in the Leached Cap and Supergene zones was also made. Weak acid soluble Cu assays were used along with the variograms for total Cu. A similar kriging procedure above was used.

The estimation of gold, molybdenum and silver was completed for the combined IX, PP and WR lithologies and then for the MX lithology. A similar strategy using 4 passes with expanding search ellipses was used. Again a minimum of 4 composites and maximum of 16 composites were required to estimate a block with a maximum of 3 from one hole allowed. Due to shorter ranges in Au, Mo and Ag, a larger fifth pass was required to estimate grades into blocks with a Cu grade estimated.

Table 14-5 through Table 14-9 show tabulations of the mineral resource at copper equivalent cutoffs of 0.20%, 0.25%, and 0.30% for supergene and hypogene zones, and at gold cutoff grades of 0.25 g/t, 0.30 g/t, 0.35 g/t, and 0.40 g/t gold for the leached cap. The CuEq is determined using the following metal prices: Cu - US$2.00/lb, Au - US$875.00/oz, Ag - US$11.25/oz and Mo - US$11.25/lb as follows:

$$CuEq\ \% = (Cu\ \%) + (Au\ g/t \times 28.13/44.1) + (Mo\ \% \times 248.06/44.1) + (Ag\ g/t \times 0.36/44.1)$$

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

The gold-dominant Leached Cap zone resource estimates remained unchanged at a gold cutoff grade of 0.40 g/t Au (Table 14-5); copper equivalent values are presented for this zone for comparative purposes.

Table 14-5: Leached Cap/Oxide Gold Zone

Class	Cutoff Au (g/t)	Tonnes (million)	Cu (%)	Au (g/t))	Mo (%)	Ag (g/t)	Weak Acid Sol. Cu (%)	CuEQ (%)
Measured	0.25 g/t	30.6	0.05	0.52	0.025	2.94	0.016	0.56
Indicated	0.25 g/t	53.2	0.03	0.33	0.017	2.36	0.010	0.37
Inferred	0.25 g/t	17.1	0.01	0.31	0.008	1.93	0.004	0.27
Total Measured + Indicated		**83.8**	**0.04**	**0.40**	**0.020**	**2.57**	**0.012**	**0.44**
Measured	0.30 g/t	27.9	0.05	0.55	0.025	3.04	0.016	0.58
Indicated	0.30 g/t	29.7	0.03	0.38	0.016	2.52	0.010	0.39
Inferred	0.30 g/t	9.0	0.01	0.35	0.006	1.92	0.004	0.28
Total Measured + Indicated		**57.6**	**0.04**	**0.46**	**0.020**	**2.77**	**0.013**	**0.48**
Measured	0.35 g/t	25.5	0.06	0.57	0.026	3.14	0.016	0.60
Indicated	0.35 g/t	15.2	0.03	0.44	0.016	2.75	0.011	0.43
Inferred	0.35 g/t	3.4	0.01	0.39	0.006	2.01	0.004	0.31
Total Measured + Indicated		**40.7**	**0.05**	**0.52**	**0.022**	**2.99**	**0.014**	**0.54**
Measured	0.40 g/t	23.0	0.06	0.59	0.025	3.23	0.015	0.61
Indicated	0.40 g/t	9.0	0.04	0.48	0.017	2.88	0.012	0.47
Inferred	0.40 g/t	1.1	0.01	0.44	0.006	2.09	0.004	0.34
Total Measured + Indicated		**32.0**	**0.05**	**0.56**	**0.023**	**3.13**	**0.015**	**0.57**

Table 14-6: Supergene Oxide Zone

Class	Cutoff CuEQ (%)	Tonnes (million)	Cu (%)	Au (g/t))	Mo (%)	Ag (g/t)	Weak Acid Sol. Cu (%)	CuEQ (%)
Measured	0.20 %	25.0	0.28	0.52	0.026	2.38	0.094	0.77
Indicated	0.20 %	40.4	0.22	0.20	0.018	1.37	0.058	0.44
Inferred	0.20 %	31.3	0.24	0.16	0.008	1.35	0.087	0.38
Total Measured + Indicated		**65.4**	**0.24**	**0.32**	**0.021**	**1.76**	**0.072**	**0.56**
Measured	0.25 %	25.0	0.28	0.52	0.026	2.38	0.094	0.77
Indicated	0.25 %	36.4	0.23	0.21	0.019	1.44	0.060	0.46
Inferred	0.25 %	26.1	0.26	0.17	0.010	1.43	0.097	0.41
Total Measured + Indicated		**61.4**	**0.25**	**0.34**	**0.022**	**1.83**	**0.074**	**0.59**
Measured	0.30 %	24.9	0.28	0.52	0.026	2.39	0.094	0.77
Indicated	0.30 %	31.7	0.24	0.23	0.021	1.54	0.063	0.49
Inferred	0.30 %	22.3	0.27	0.18	0.010	1.46	0.104	0.44
Total Measured + Indicated		**56.6**	**0.26**	**0.36**	**0.023**	**1.91**	**0.077**	**0.61**

Table 14-7: Supergene Sulphide Zone

Class	Cutoff CuEQ (%)	Tonnes (million)	Cu (%)	Au (g/t))	Mo (%)	Ag (g/t)	Weak Acid Sol. Cu (%)	CuEQ (%)
Measured	0.20 %	36.4	0.39	0.41	0.029	2.34	0.062	0.83
Indicated	0.20 %	223.6	0.24	0.22	0.019	1.70	0.034	0.49
Inferred	0.20 %	129.0	0.18	0.18	0.009	1.40	0.031	0.34
Total Measured + Indicated		**260.0**	**0.26**	**0.25**	**0.020**	**1.79**	**0.038**	**0.54**
Measured	0.25 %	36.3	0.39	0.41	0.029	2.34	0.062	0.84
Indicated	0.25 %	216.0	0.24	0.22	0.019	1.72	0.034	0.50
Inferred	0.25 %	102.1	0.20	0.19	0.010	1.49	0.034	0.38
Total Measured + Indicated		**252.3**	**0.26**	**0.25**	**0.021**	**1.81**	**0.038**	**0.55**
Measured	0.30 %	36.0	0.39	0.41	0.029	2.34	0.062	0.84
Indicated	0.30 %	200.1	0.25	0.23	0.020	1.78	0.035	0.52
Inferred	0.30 %	82.1	0.21	0.19	0.011	1.54	0.036	0.40
Total Measured + Indicated		**236.1**	**0.27**	**0.26**	**0.022**	**1.86**	**0.039**	**0.57**

Table 14-8: Hypogene Zone

Class	Cutoff CuEQ (%)	Tonnes (million)	Cu (%)	Au (g/t))	Mo (%)	Ag (g/t)	CuEQ (%)
Measured	0.20 %	32.7	0.31	0.38	0.025	1.94	0.72
Indicated	0.20 %	779.3	0.16	0.20	0.022	1.59	0.43
Inferred	0.20 %	1,967.7	0.13	0.15	0.018	1.30	0.34
Total Measured + Indicated		**812.0**	**0.16**	**0.21**	**0.022**	**1.61**	**0.44**
Measured	0.25 %	32.4	0.32	0.38	0.026	1.94	0.73
Indicated	0.25 %	710.6	0.17	0.21	0.023	1.65	0.45
Inferred	0.25 %	1,568.2	0.14	0.16	0.020	1.36	0.37
Total Measured + Indicated		**743.0**	**0.17**	**0.22**	**0.023**	**1.66**	**0.46**
Measured	0.30 %	32.1	0.32	0.38	0.026	1.95	0.73
Indicated	0.30 %	620.5	0.18	0.22	0.025	1.71	0.47
Inferred	0.30 %	1,144.7	0.15	0.17	0.023	1.43	0.40
Total Measured + Indicated		**652.6**	**0.18**	**0.23**	**0.025**	**1.72**	**0.49**

Table 14-9: Combined Supergene and Hypogene Zones

Class	Cutoff CuEQ (%)	Tonnes (million)	Cu (%)	Au (g/t))	Mo (%)	Ag (g/t)	CuEQ (%)
M+I	0.20 %	1,137.5	0.19	0.23	0.021	1.65	0.47
M+I	0.25 %	1,056.9	0.20	0.23	0.022	1.71	0.49
M+I	0.30 %	945.6	0.21	0.25	0.024	1.77	0.51
Inferred	0.20 %	2,128.1	0.13	0.16	0.017	1.39	0.34
Inferred	0.25 %	1,696.4	0.14	0.16	0.019	1.37	0.37
Inferred	0.30 %	1,249.1	0.16	0.18	0.022	1.44	0.40

14.8 MODEL VERIFICATION

To determine if any bias exists, the mean grades of the composites used to estimate the blocks were compared to the mean grades of the blocks estimated. When a comparison is done for all blocks estimated, the kriged grades are similar, but lower in all categories. This is most likely caused by the large number of inferred blocks estimated on the periphery and a depth in the deposit where the composites used are the lower grades on the edge of the mineralized system. When the comparison is restricted to just measured and indicated blocks, which are tighter to the data, the comparison is much closer. Comparisons between estimated results and drill holes on cross sections were also made and the results were satisfactory.

15 MINERAL RESERVE ESTIMATES

15.1 MINERAL RESERVE

The mine and plant production schedules defines the mineral reserve for a mining project. Table 15-1 presents the mineral reserve for the Casino Project based on the production schedule presented in Section 16.

The mill ore reserve amounts to 965.2 million tonnes at 0.204% copper, 0.240 g/t gold, 0.0227% molybdenum, and 1.74 g/t silver. Heap leach reserve is an additional 157.5 million tonnes at 0.292 g/t gold and 0.036% copper.

For this reserve estimate, measured mineral resource was converted to proven mineral reserve and 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. The parameters used to determine mineral reserves are shown in the following sections.

Table 15-1: Mineral Reserve

Mill Ore Reserve:	**Ore Ktonnes**	**Tot Cu (%)**	**Gold (g/t)**	**Moly (%)**	**Silver (g/t)**
Proven Mineral Reserve:					
Mill Ore	91,602	0.336	0.437	0.0275	2.23
Probable Mineral Reserve:					
Mill Ore	729,777	0.203	0.235	0.0240	1.78
Low Grade Stockpile	143,828	0.122	0.139	0.0133	1.19
Total Probable Reserve	873,605	0.190	0.219	0.0222	1.68
Proven/Probable Reserve					
Mill Ore	821,379	0.218	0.258	0.0244	1.83
Low Grade Stockpile	143,828	0.122	0.139	0.0133	1.19
Total Mill Ore Reserve	965,207	0.204	0.240	0.0227	1.74
Heap Leach Reserve:	**Ore ktonnes**	**Gold (g/t)**	**Tot Cu (%)**	**Moly (%)**	**Silver (g/t)**
Proven Mineral Reserve	31,760	0.480	0.051	N/A	2.79
Probable Mineral Reserve	125,694	0.244	0.032	N/A	2.06
Total Heap Leach Reserve	157,454	0.292	0.036	N/A	2.21

IMC does not know of any mining, metallurgical, infrastructure or other factors that might materially affect the mineral reserve. The mineral reserve is consistent with current CIM and NI 43-101 guidelines.

15.2 DESIGN ECONOMICS

15.2.1 Economic Parameters and NSR Calculations

15.2.1.1 Commodity Prices

Table 15-2 summarizes the economic parameters for mine design and scheduling. The commodity prices used for design are $2.75 per pound copper, $1,300 per ounce gold, $14.50 per pound molybdenum, and $23.00 per ounce silver. These were specified by WCGC personnel. Also, for this study, it is assumed that the US$ to C$ exchange rate is 1:1.

15.2.1.2 Mining

The base mining cost of $1.60 per total tonne was estimated by IMC. The mining cost estimate from the March 2011 Preliminary Feasibility Study (PFS) was updated with revised equipment parts, fuel and blasting agent costs. The estimated mine operating cost for the 2011 PFS was $1.51 per total tonne, so this estimate is about a 6% increase.

15.2.1.3 Flotation Ore Processing

Flotation ore refers to the supergene oxide, supergene sulphide, and hypogene zones of the orebody. The base unit costs for ore processing and G&A are estimated at $6.54 and $0.36 per ore tonne respectively, based on the 2011 PFS study. For flotation ore, the plant recoveries are estimated as follows:

Copper recovery = 100%(Cut% - Cuw% - 0.022%) / Cut%
Gold Recovery = 66%
Molybdenum Recovery = 57%
Silver Recovery = 50%

Where,

Cut% = Total copper grade
Cuw% = Weak acid soluble copper grade

These recoveries and recovery equations were provided by WCGC personnel.

The copper, gold, and silver payable percentages shown on Table 15-2 are typical terms for copper concentrates, assuming a clean concentrate with a copper concentrate grade of 28% copper or greater. Table 15-3 shows the details of the off-site costs for the copper concentrate. The 2011 PFS study was based on smelter treatment terms of $80 per tonne concentrate and refinery terms of $0.080 per pound recovered copper. The land freight, terminal handling charge, and ocean freight charge amounts to $158.11 per dry metric tonne (about $145 per wet metric tonne) based on the parameters used for the 2011 PFS. IMC assumed moisture content of 8.5% and 0.5% concentrate loss during shipping. Copper smelting, refining, and freight charges amount to $0.480 per pound payable copper. This does not include any charges for marketing or insurance. Gold and silver refining is estimated at $6.00 per ounce gold and $0.50 per ounce silver respectively.

Due to the 0.5% concentrate loss estimate payable percentages at the mine site are 99.5% of the smelter payables, which is factored into the NSR calculation.

The 2011 PFS assumed an 85% payable percentage for molybdenum in concentrate. This is assumed to be the net payable after treatment and transportation charges. This is applicable to a clean moly concentrate with a moly grade of about 50% or greater.

15.2.1.4 Heap Leach Ore

Heap leach ore refers to ores in the leach capping of the orebody. Heap leach ore processing is estimated at $3.70 per ore tonne for crushed ore. It is assumed that all G&A costs are charged to the flotation ore.

Heap leach recoveries are estimated at 65% for gold, 20% for copper, and 20% for silver. Gold and silver from the heap leach will report to a typical doré which will be sent to a refinery. Typical terms are shown on Table 15-2. The payable percentage is estimated at 98% for gold and silver. The transportation and refining charges are estimated at $1.611 per ounce gold and $0.661 per ounce silver. The transportation portion of this cost is estimated as $10,000 per tonne doré, or about $0.311 per ounce gold and silver.

It is also assumed that the heap leach process will produce a copper concentrate with a grade of about 60% copper. Smelting, refining, and freight terms are assumed the same as for the flotation concentrate. This results in a smelting, refining, and freight charge of about $0.267 per pound copper. Details of this calculation are shown in Table 15-4.

Table 15-2: Economic Parameters for Mine Design (C$)

Parameter	Flotation Ore	Heap Leach
Commodity Prices:		
Copper Price Per Pound	2.75	2.75
Gold Price Per Ounce	1300.00	1300.00
Silver Price Per Ounce	23.00	23.00
Molybdenum Price Per Pound	14.50	N.A.
Mining Cost Per Total Tonne:		
Mining Cost	1.600	1.600
Processing and G&A Per Ore Tonne		
Processing	6.540	3.700
G&A	0.360	0.000
Total Processing and G&A	6.900	3.700
Average Plant Recoveries:		
Copper Recovery	Note 1	20.0%
Gold Recovery	66.0%	65.0%
Silver Recovery	50.0%	20.0%
Molybdenum Recovery	57.0%	N.A.
Refinery Payables:		
Copper Payable	96.5%	96.5%
Gold Payable	97.5%	98.0%
Silver Payable	97.5%	98.0%
Molybdenum Payable	85.0%	N.A.
Payable Concentrate (0.5% Conc Loss)	99.5%	Cu Only
Offsite Costs:		
Copper SRF Cost Per Pound	0.480	0.267
Gold Refining Per Ounce (Note 4)	6.000	1.611
Silver Refining Per Ounce (Note 4)	0.500	0.661
Molybdenum Freight/Treatment Per Pound	Note 2	N.A.
NSR Factors (Apply to Recovered Grades):		
Copper Factor (Note 3)	48.06	52.57
Gold Factor (Note 3)	40.36	40.91
Silver Factor (Note 3)	0.702	0.704
Moly Factor (Note 3)	270.36	N.A.
NSR Cutoff Grades:		
Breakeven Cutoff (C$/t)	8.50	5.30
Internal Cutoff (C$/t)	6.90	3.70
Stockpile Cutoff (C$/t) ($1.00 Rehandle)	7.90	N.A.

Note 1: Cu Recovery = 100%(Cut% - Cuw% - 0.022%)/Cut%
Note 2: Moly offsite costs are accounted in payable percentage
Note 3: NSR factors are applied to recovered grades
Note 4: Includes dore transport of $0.311 per ounce ($10,000/t)

Table 15-3: Copper SRF Cost Per Pound – Basis: 1 Tonne of Concentrate at the Destination

Parameter	(units)	Value		Comments
Copper Concentrate Grade	(%Cu)	28		WC assumption
Moisture Content	(%)	8.50%		IMC assumption
Concentrate Loss	(%)	0.50%		IMC assumption
Payable Percentage	(%)	96.50%		Typical for clean concentrate with 28% or greater Cu grade.
Payable Copper/Tonne	(lbs)	595.7		Concentrate grade x payable % x 22.046
Treatment Cost Per Tonne	(US$)	80.00		2011 Pre-feasibility Parameter. Section 1.25.8.1
Treatment Cost Per Lb	(US$)	0.134		
Refining Cost Per Lb	(US$)	0.080		2011 Pre-feasibility Parameter. Section 1.25.8.1
Land Freight per WMT	(US$)	40.00		2011 Pre-feasibility Parameter. Section 1.25.8.1
Terminal Charge per WMT	(US$)	27.00		2011 Pre-feasibility Parameter. Section 1.25.8.1
Ocean Freight per WMT	(US$)	78.00		2011 Pre-feasibility Parameter. Section 1.25.8.1
Total Freight Per DMT (Note 1)	(US$)	158.11		Includes paying for lost concentrate
Total Freight Per Lb	(US$)	0.265		
Total SRF Cost Per Lb	(US$)	0.480		Smelting, refining,and freight
Total SRF Cost Per Lb	(C$)	0.480		

Table 15-4: Copper SRF Cost Per Pound – Leach Ore – Basis: 1 Tonne of Concentrate at the Destination

Parameter	(units)	Value		Comments
Copper Concentrate Grade	(%Cu)	60.0		Arcadis
Moisture Content	(%)	8.50%		IMC assumption
Concentrate Loss	(%)	0.50%		IMC assumption
Payable Percentage (Note 1)	(%)	96.50%		Typical for clean concentrate with 28% or greater Cu grade.
Payable Copper/Tonne	(lbs)	1276.5		Concentrate grade x payable % x 22.046
Treatment Cost Per Tonne	(US$)	80.00		2011 Pre-feasibility Parameter. Section 1.25.8.1
Treatment Cost Per Lb	(US$)	0.063		
Refining Cost Per Lb	(US$)	0.080		2011 Pre-feasibility Parameter. Section 1.25.8.1
Land Freight per WMT	(US$)	40.00		2011 Pre-feasibility Parameter. Section 1.25.8.1
Terminal Charge per WMT	(US$)	27.00		2011 Pre-feasibility Parameter. Section 1.25.8.1
Ocean Freight per WMT	(US$)	78.00		2011 Pre-feasibility Parameter. Section 1.25.8.1
Total Freight Per DMT (Note 2)	(US$)	158.11		Includes paying for lost concentrate
Total Freight Per Lb	(US$)	0.124		
Total SRF Cost Per Lb	(US$)	0.267		Smelting, refining,and freight
Total SRF Cost Per Lb	(C$)	0.267		

NSR values were calculated for each model block to use to classify blocks into ore and waste. For the heap leach ore:

NSR_au = ($1300 – $1.611) x 0.65 x 0.98 x gold / 31.103
NSR_cu = ($2.75 - $0.267) x 0.20 x 0.965 x 0.995 x copper x 22.046
NSR_ag = ($23.00 - $0.661) x 0.20 x 0.98 x silver / 31.103
NSR = NSR_au + NSR_cu + NSR_ag

The internal NSR cutoff grade for heap leach ore is the processing cost of $3.70 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 $5.30 per tonne (mining plus processing). For flotation ore, the NSR values are calculated as:

NSR_cu = ($2.75 – $0.480) x rec_cu x 0.965 x 0.995 x 22.046

NSR_au = (\$1300 – \$6.00) x 0.66 x 0.975 x 0.995 x gold / 31.103
NSR_mo = \$14.50 x 0.57 x 0.85 x 0.995 x moly x 22.046
NSR_ag = (\$23.00 - \$0.50) x 0.20 x 0.975 x 0.995 x silver / 31.103
NSR = NSR_cu + NSR_au + NSR_mo + NSR_ag

where,

rec_cu = 1.00 x (cut – cuw – 0.022%)

The internal NSR cutoff grade is the processing plus G&A cost of \$6.90. Breakeven NSR cutoff is \$8.50. The stockpile re-handle cutoff grade is estimated at \$7.90 per ore tonne which covers processing plus G&A costs plus mining re-handle estimated at about \$1.00 per tonne.

It is important to note that the parameters presented in this section are initial estimates used for the mine design process and are not the final economic parameters developed for the financial analysis of the project.

15.2.2 Slope Angles

Slope angles recommendations were developed by Knight Piesold Ltd. (KP) and documented in the report "Open Pit Geotechnical Design", dated October 12, 2012. Table 15-5 shows the recommended angles by design sector and Figure 15-1 shows the design sectors.

Forty five degree inter-ramp angles were recommended for most of the slope sectors. The north sectors of the main pit and west pit were recommended to be designed at 42 degree inter-ramp angles. For the small amount of overburden on the north wall the recommended angle was 27 degrees. The slope angle recommendations also specified that there be no more than 200 m of vertical wall at the inter-ramp angle without an extra wide catch bench (16 m instead of 8 m).

IMC used an overall slope angle of 42 degrees in the floating cone runs to approximate overall slope angles with the KP inter-ramp angles and also to account for the location of haulage roads.

Table 15-5: Recommended Slope Angles (Knight-Piesold)

Design Sector	Slope Height	Wall Geology	Bench Face Angle	Bench Height	Bench Width	Inter-ramp Angle	Max Inter-ramp Slope Height	Overall Slope Angle
	M		°	m	m	°	m	°
M-North	30	Overburden	40	5	4	**27**	200	39
	600	DRB	60	15	8	**42**		
M-Northeast	30	Overburden	40	5	4	**27**	100[1]/200	**40**
	600	DRB	65	15	8	**45**		
M-South	540	PMS, DRB	65	15	8	**45**	200	**42**
Central	210	PMS	65	15	8	**45**	200	**N/A**
W-North	285	DRB	60	15	8	**42**	200	39
W-South	480	DRB	65	15	8	**45**	200	**42**
W-Southwest	345	PMS	65	15	8	**45**	200	**42**
W-West	210	DRB	65	15	8	**45**	200	**42**

NOTES:

1. A 100-m high inter-ramp slope is recommended for slopes developed in weathered bedrock. The maximum height for the inter-ramp slopes in fresh rock is 200 m.

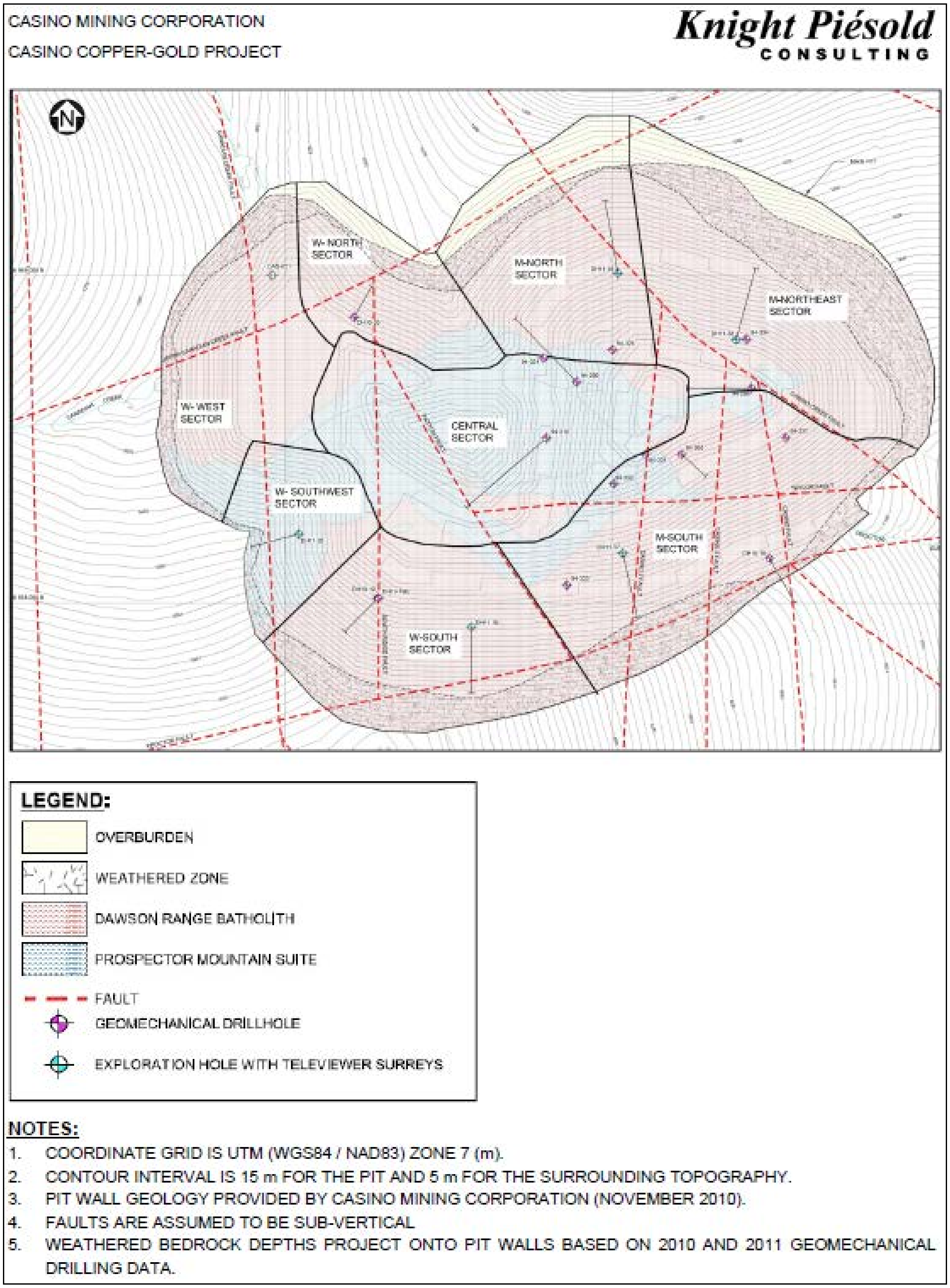


Figure 15-1: Open Pit Design Sectors (Knight-Piesold)

16 MINING METHODS

16.1 INTRODUCTION

Independent Mining Consultants, Inc. (IMC) was commissioned by WCGC to provide mine planning services for the Casino Copper-Gold Project, Yukon Territory, Canada. The work was done to support a Feasibility Study (FS) conducted by M3 Engineering and Technology Corporation (M3).

IMC's scope of work is summarized as follows:

- Develop a mine plan and mine production schedule;
- Determine mine equipment and mine labor requirements;
- Estimate the mine capital and mine operating costs.

IMC also provided these services for a Preliminary Feasibility Study (PFS), also conducted by M3, and completed during April 2011. The mining plan for this current study is similar to the 2011 study. The plan represents a commercially viable development, but has not necessarily been optimized. Accordingly, as the project develops, the percentage of waste and ore may change to accommodate earlier start-up and corresponding better financial metrics.

This report is in metric units. Ktonnes is an abbreviation for 1,000 metric tonnes. Linear measurements are expressed in meters or kilometers and liquid volumes in liters or cubic meters. Gold and silver grades are expressed in grams per metric tonne (g/t). Quantities of gold metal are often abbreviated as koz for 1000 troy ounces; klbs and mlbs are abbreviations for one thousand (1000) and one million (1,000,000) US pounds of copper metal, respectively.

Currency is in Canadian dollars as of the 4th quarter of 2012 and the exchange rate for mining is assumed to be US$ 1 = C$ 1.

16.2 RESOURCE MODEL

See Section 14 for details on the Resource Model. The resource block model was provided to IMC by WCGC. The model was developed by Giroux and Associates (Giroux) during 2010. The block model was based on regular 20m by 20m by 15m high blocks. The main variables included in the block model were:

- The percentage of the block in the various lithology domains.
- The percentage of the block in the various ore type domains (leach cap, supergene oxide, supergene sulphide, hypogene).
- Grades of total copper and weak acid soluble copper by ore type domain.
- Gold, silver and molybdenum grades.
- A resource classification code to denote measured, indicated, and inferred blocks.
- Bulk density.

IMC did validate ore tonnages and grades for each domain with resource tabulations published by Giroux. IMC did not audit the resource block model.

Subsequent to completion of the 2011 PFS additional geologic data was incorporated into the model. These items included:

- An expanded lithologic interpretation to better define rock types in the waste zones, and
- An interpretation of alteration types to better define potential ore types for metallurgical characterization.

Block grades were not re-calculated; they are the same as the original model.

16.3 OPEN PIT DESIGN

A mine plan was developed to supply ore to a conventional copper sulphide flotation plant with the capacity to process approximately 124,000 tonnes per day, or 45.4 million tonnes per year, though actual annual throughput is based on ore hardness. The mine is scheduled to operate two 12 hour shifts per day, 365 days per year. This will require four mining crews.

The final pit design was based on a floating cone shell at $2.75 per pound copper, $1300 per ounce gold, $14.50 per pound molybdenum, and $23.00 per ounce silver.

Five mining phases were also developed for the study. The phase designs include haul roads and adequate working room for large mining equipment. The roads are 36m wide at a maximum grade of 10%. The width will accommodate trucks up to the 360 tonne class, such as the Caterpillar 797F truck.

Figure 16-1 shows the final pit design. As was discussed in Section 15.2.2, slope angle recommendations were provided by Knight Piesold Ltd. (KP).

Figure 16-1: Final Pit Design

16.4 MINE PRODUCTION SCHEDULE

The top section of Table 16-1 shows the proposed plant production schedule. Total mill ore is 965.2 million tonnes at 0.204% copper, 0.240 g/t gold, 0.0227% moly, and 1.74 g/t silver. The average NSR value of this ore is $18.53 per tonne. For Years 2 through 21, full production years, ore throughput varies from a low of 44,695 ktonnes in Year 11 to a high of 46,249 ktonnes in Year 13, depending on ore hardness.

The Bond Work Index and mill throughput rate have been assigned to model blocks based on rock type, ore type, and alteration based on the recommendations in the FLSmidth report "Casino Project Circuit Design Basis – Update on Testwork and Mill Sizing/Selection". The ore schedule has been developed based on plant hours, so throughput varies by year. It was reported to IMC that all necessary efficiency factors were incorporated in throughput rates so IMC has based the schedule on 8,760 plant hours per year.

The table also shows the average Bond Work Index and throughput rate (hours per ktonne). The throughput units are somewhat unconventional, but for the model a parameter that can be weight averaged by tonnes is necessary. Copper recovery was also incorporated into the model on a block by block basis, based on total and soluble copper grades. The average recovered copper grade is 0.167%, indicating an average copper recovery of 82.1%.

The table also shows the various components of the mill ore. Direct feed ore is ore that is scheduled to be processed the same year it is mined. This amounts to 789.0 million tonnes at 0.216% total copper, 0.248 g/t gold, 1.81 g/t silver, and 0.0244% moly. This is about 82% of total ore. The average NSR value of this ore is $19.76 per tonne. Note that Year 1 ore production is 34.5 million tonnes, about 75% of nominal capacity (125,000 tpd) and is made up of ore mined during preproduction and Year 1.

As was done for the PFS, the SOX ore in mining phase 1 that was mined during preproduction and Year 1 is stockpiled and processed during Years 4 through 12 at the rate of 3.6 million tonnes per year. This is done to maintain the ratio of weak soluble copper to total copper at less than 25%. This ore amounts to 32.4 million tonnes at 0.260% total copper, 0.486 g/t gold, 2.35 g/t silver, and 0.0250% moly. The NSR value for this ore is $24.84 per tonne. For the PFS this ore amounted to 35.8 million tonnes and was processed over ten years instead of nine years. The reduction in the size of this stockpile is due to the following: 1) 1.2 million tonnes of SOX mined during Year 2 was processed that year, instead of being stockpiled, and 2) the cutoff grade for the SOX stockpile was raised, sending the more marginal grade SOX to the low grade stockpile to defer processing to the end of open pit operations.

The operating schedule also results in a significant amount of low grade ore that is stockpiled and processed at the end of the mine life during Years 19 through 22. This amounts to 143.8 million tonnes at 0.122% total copper, 0.139 g/t gold, 1.19 g/t silver, and 0.0133 g/t moly.

The reclaim schedules for both the SOX ore and low grade are on a last-in-first-out (LIFO) basis, consistent with stockpiles build up in lifts and reclaimed in reverse order.

Based on the schedule the commercial life of the project is 21.5 years after an approximate 3 year preproduction period.

Table 16-2 shows the mine production schedule. The upper section of the table shows the direct feed ore. 4.9 million tonnes of this is mined during preproduction and stockpiled near the crusher to be part of the Year 1 ore feed.

For economic purposes, an NSR value was calculated for each ore block to classify blocks into ore and waste. The NSR is in $US (=$C) per tonne and reflects gross revenue less concentrate freight, concentrate treatment charges, and refining. Process recoveries and treatment payable amounts are incorporated into the NSR calculation as are allowances for concentrate loss in shipping and marketing costs. The schedule shows that in Year 2, the NSR cutoff grade is high, $15.50 per tonne and declines to the internal cutoff grade of $6.90 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.

The second section of the table shows the SOX mill stockpile ore that is mined during preproduction and Year 1. The average soluble copper to total copper ratio for this ore is 33.8%. The third section of the table shows low grade ore produced by year. This is material with an NSR cutoff grade between $7.90 per tonne and the operating cutoff for the year.

The bottom of Table 16-2 also shows the schedule of gold ore mined from the leach cap ore zone by year. It is assumed that this is processed by crushing and heap leaching. Gold ore is defined as leach cap ores with an NSR above $3.70 with leach economics and total copper less than 0.1%. This ore amounts to 157.5 million tonnes at 0.292 g/t gold, 2.21 g/t silver, and 0.036% total copper. This is considerably more ore (about double) what was considered for the PFS. The PFS was limited to 81.6 million tonnes of ore.

The bottom of Table 16-2 summarizes tonnages. It can be seen that life of mine total material from the pit is 1.78 billion tonnes. Preproduction is 70.0 million tonnes staged over three years. Year 1 total material is scheduled at about 91.2 million tonnes after which the peak material movement of 100 million tonnes per year is maintained over much of the mine life. Total waste is 657.9 million tonnes so the waste to ore ratio is about 0.58 if mill ore (including SOX), low grade, and gold ore are all counted as ore.

The upper section of Table 16-3 shows a proposed stacking schedule for the gold ore. This is based on the ability to crush and stack 9,125 ktonnes per year (25,000 tpd for 365 days/year). The second section of the table shows mine production of gold ore. The third and fourth sections show up to 9,125 ktonnes of mined ore as direct crusher feed and the excess going to a stockpile. Both are shown at average grades for the year. The bottom of the table shows the stockpile reclaim on a last-in-first-out basis (LIFO). The stockpile gets to a maximum size of almost 40 million tonnes with this scenario.

Table 16-1: Proposed Mill Production Schedule

	(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	TOTAL
Proposed Mill Schedule:																											
Ore Ktonnes	(kt)				34,500	45,928	45,814	45,656	45,253	45,205	45,176	45,745	46,091	44,775	44,695	45,861	46,249	45,798	45,187	45,064	44,772	44,934	45,591	45,053	44,985	22,875	965,207
NSR Value	(C$/t)				27.24	28.88	27.26	23.29	19.21	19.86	20.88	20.86	20.92	20.35	20.25	18.20	17.48	17.04	15.60	13.06	14.09	16.83	12.11	10.29	10.77	11.44	18.53
Total Copper	(%)				0.327	0.347	0.287	0.273	0.231	0.209	0.212	0.211	0.209	0.240	0.244	0.192	0.184	0.193	0.173	0.137	0.137	0.152	0.124	0.112	0.123	0.168	0.204
Weak Soluble Copper	(%)				0.060	0.039	0.008	0.028	0.029	0.009	0.008	0.009	0.009	0.028	0.025	0.014	0.006	0.006	0.004	0.000	0.000	0.000	0.004	0.008	0.009	0.032	0.014
% Cuw/Cut	(%)				18.4%	11.2%	2.8%	10.1%	12.4%	4.4%	3.9%	4.3%	4.4%	11.8%	10.4%	7.2%	3.3%	3.1%	2.3%	0.0%	0.0%	0.0%	2.8%	6.8%	7.7%	19.1%	7.1%
Gold	(g/t)				0.423	0.384	0.370	0.320	0.276	0.291	0.286	0.268	0.260	0.238	0.232	0.228	0.202	0.196	0.177	0.167	0.175	0.199	0.146	0.136	0.146	0.150	0.240
Molybdenum	(%)				0.0214	0.0273	0.0286	0.0216	0.0164	0.0189	0.0242	0.0290	0.0313	0.0275	0.0259	0.0258	0.0264	0.0218	0.0210	0.0164	0.0214	0.0307	0.0196	0.0149	0.0129	0.0100	0.0227
Silver	(g/t)				2.59	2.00	1.90	2.04	1.85	1.87	2.18	1.67	1.82	1.91	1.79	1.81	1.57	1.48	1.69	1.65	1.76	1.56	1.37	1.14	1.33	1.20	1.74
Recovered Copper	(%)				0.245	0.286	0.257	0.223	0.180	0.177	0.182	0.180	0.178	0.189	0.197	0.156	0.156	0.165	0.147	0.114	0.115	0.130	0.099	0.083	0.092	0.114	0.167
Bond Work Index	(Kwh/t)				14.4	14.4	14.4	14.5	14.6	14.6	14.6	14.4	14.3	14.7	14.7	14.3	14.3	14.4	14.6	14.6	14.7	14.7	14.4	14.6	14.6	14.6	14.5
Hours Per Ktonne	(hr/kt)				0.19043	0.19074	0.19119	0.19188	0.19359	0.19379	0.19392	0.19151	0.19006	0.19563	0.19598	0.19101	0.18941	0.19127	0.19386	0.19438	0.19566	0.19495	0.19213	0.19445	0.19472	0.19427	0.19293
Mill Hours	(hours)				6,570	8,760	8,759	8,760	8,760	8,760	8,760	8,761	8,760	8,759	8,759	8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,759	8,760	8,760	4,444	186,212
Copper Recovery	(%)				74.85%	82.42%	89.55%	81.51%	78.10%	85.04%	85.73%	85.28%	85.05%	79.05%	80.57%	81.29%	84.78%	85.49%	84.97%	83.21%	83.94%	85.53%	79.45%	73.50%	74.87%	67.76%	82.10%
Gold Recovery	(%)				66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%
Moly Recovery	(%)				57.00%	57.00%	57.00%	57.00%	57.00%	57.00%	57.00%	57.00%	57.00%	57.00%	57.00%	57.00%	57.00%	57.00%	57.00%	57.00%	57.00%	57.00%	57.00%	57.00%	57.00%	57.00%	57.00%
Silver Recovery	(%)				50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%
Direct Feed Mill Ore:																											
Ore Ktonnes	(kt)				34,500	45,928	45,814	42,056	41,653	41,605	41,576	42,145	42,491	41,175	41,095	42,251	46,249	45,798	45,187	45,064	44,772	44,934	14,676				788,969
NSR Value	(C$/t)				27.24	28.88	27.26	23.12	18.69	19.39	20.50	20.49	20.58	19.97	19.86	17.78	17.48	17.04	15.60	13.06	14.09	16.83	18.16				19.76
Total Copper	(%)				0.327	0.347	0.287	0.270	0.224	0.200	0.204	0.203	0.206	0.244	0.249	0.193	0.184	0.193	0.173	0.137	0.137	0.152	0.178				0.216
Weak Soluble Copper	(%)				0.060	0.039	0.008	0.021	0.022	0.001	0.000	0.001	0.003	0.025	0.022	0.010	0.006	0.006	0.004	0.000	0.000	0.000	0.000				0.012
Gold	(g/t)				0.423	0.384	0.370	0.311	0.263	0.279	0.274	0.254	0.239	0.208	0.201	0.201	0.202	0.196	0.177	0.167	0.175	0.199	0.195				0.248
Molybdenum	(%)				0.0214	0.0273	0.0286	0.0210	0.0154	0.0181	0.0239	0.0291	0.0319	0.0281	0.0264	0.0264	0.0264	0.0218	0.0210	0.0164	0.0214	0.0307	0.0310				0.0244
Silver	(g/t)				2.59	2.00	1.90	2.04	1.83	1.86	2.19	1.64	1.76	1.82	1.69	1.75	1.57	1.48	1.69	1.65	1.76	1.56	2.00				1.81
Recovered Copper	(%)				0.245	0.286	0.257	0.226	0.180	0.177	0.182	0.180	0.181	0.197	0.205	0.161	0.156	0.165	0.147	0.114	0.115	0.130	0.156				0.182
Bond Work Index	(Kwh/t)				14.370	14.400	14.400	14.500	14.600	14.600	14.600	14.400	14.300	14.800	14.800	14.400	14.300	14.400	14.600	14.600	14.700	14.700	14.300				14.521
Hours Per Ktonne	(hr/kt)				0.19043	0.19074	0.19119	0.19234	0.19420	0.19442	0.19456	0.19194	0.19051	0.19671	0.19710	0.19181	0.18941	0.19127	0.19386	0.19438	0.19566	0.19495	0.19062				0.19302
Mill Hours	(hours)				6,570	8,760	8,759	8,089	8,089	8,089	8,089	8,089	8,095	8,100	8,100	8,104	8,760	8,760	8,760	8,760	8,760	8,760	2,798				152,290
SOX Mill Ore Stock (LIFO):																											
Ore Ktonnes	(kt)							3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,610											32,410
NSR Value	(C$/t)							25.25	25.25	25.25	25.25	25.25	24.92	24.65	24.65	23.09											24.84
Total Copper	(%)							0.308	0.308	0.308	0.308	0.308	0.244	0.190	0.190	0.178											0.260
Weak Soluble Copper	(%)							0.104	0.104	0.104	0.104	0.104	0.083	0.065	0.065	0.060											0.088
Gold	(g/t)							0.430	0.430	0.430	0.430	0.430	0.512	0.581	0.581	0.549											0.486
Molybdenum	(%)							0.0281	0.0281	0.0281	0.0281	0.0281	0.0241	0.0207	0.0207	0.0190											0.0250
Silver	(g/t)							2.03	2.03	2.03	2.03	2.03	2.53	2.95	2.95	2.56											2.35
Recovered Copper	(%)							0.182	0.182	0.182	0.182	0.182	0.139	0.103	0.103	0.097											0.150
Bond Work Index	(Kwh/t)							14.000	14.000	14.000	14.000	14.000	13.891	13.800	13.800	13.641											13.903
Hours Per Ktonne	(hr/kt)							0.18648	0.18648	0.18648	0.18648	0.18648	0.18471	0.18324	0.18324	0.18171											0.18503
Mill Hours	(hours)							671	671	671	671	671	665	660	660	656											5,997
Low Grade Mill Stock (LIFO):																											
Ore Ktonnes	(kt)																						30,915	45,053	44,985	22,875	143,828
NSR Value	(C$/t)																						9.24	10.29	10.77	11.44	10.40
Total Copper	(%)																						0.098	0.112	0.123	0.168	0.122
Weak Soluble Copper	(%)																						0.005	0.008	0.009	0.032	0.012
Gold	(g/t)																						0.122	0.136	0.146	0.150	0.139
Molybdenum	(%)																						0.0141	0.0149	0.0129	0.0100	0.0133
Silver	(g/t)																						1.07	1.14	1.33	1.20	1.19
Recovered Copper	(%)																						0.071	0.083	0.092	0.114	0.088
Bond Work Index	(Kwh/t)																						14.5	14.6	14.6	14.6	14.6
Hours Per Ktonne	(hr/kt)																						0.19285	0.19445	0.19472	0.19427	0.19416
Mill Hours	(hours)																						5,962	8,760	8,760	4,444	27,926

Table 16-2: 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	TOTAL
Direct Feed Mill Ore:																											
NSR Cutoff Grade	(C$)		11.00	11.00	11.00	15.50	12.75	14.00	12.50	13.75	13.25	12.50	12.50	12.25	11.75	11.25	11.25	11.00	10.50	9.50	9.50	6.90	6.90				
Ore Ktonnes	(kt)		603	4,326	29,571	45,928	45,814	42,056	41,653	41,605	41,576	42,145	42,491	41,175	41,095	42,251	46,249	45,798	45,187	45,064	44,772	44,934	14,676				788,969
NSR Value	(C$/t)		19.52	21.33	28.26	28.88	27.26	23.12	18.69	19.39	20.50	20.49	20.58	19.97	19.86	17.78	17.48	17.04	15.60	13.06	14.09	16.83	18.16				19.76
Total Copper	(%)		0.232	0.228	0.344	0.347	0.287	0.270	0.224	0.200	0.204	0.203	0.206	0.244	0.249	0.193	0.184	0.193	0.173	0.137	0.137	0.152	0.178				0.216
Weak Soluble Copper	(%)		0.055	0.043	0.063	0.039	0.008	0.021	0.022	0.001	0.000	0.001	0.003	0.025	0.022	0.010	0.006	0.006	0.004	0.000	0.000	0.000	0.000				0.012
Gold	(g/t)		0.359	0.403	0.427	0.384	0.370	0.311	0.263	0.279	0.274	0.254	0.239	0.208	0.201	0.201	0.202	0.196	0.177	0.167	0.175	0.199	0.195				0.248
Molybdenum	(%)		0.0121	0.0125	0.0229	0.0273	0.0286	0.0210	0.0154	0.0181	0.0239	0.0291	0.0319	0.0281	0.0264	0.0264	0.0264	0.0218	0.0210	0.0164	0.0214	0.0307	0.0310				0.0244
Silver	(g/t)		1.88	2.41	2.63	2.00	1.90	2.04	1.83	1.86	2.19	1.64	1.76	1.82	1.69	1.75	1.57	1.48	1.69	1.65	1.76	1.56	2.00				1.81
Recovered Copper	(%)		0.155	0.163	0.259	0.286	0.257	0.226	0.180	0.177	0.182	0.180	0.181	0.197	0.205	0.161	0.156	0.165	0.147	0.114	0.115	0.130	0.156				0.182
Bond Work Index	(Kwh/t)		14.7	14.8	14.3	14.4	14.4	14.5	14.6	14.6	14.6	14.4	14.3	14.8	14.8	14.4	14.3	14.4	14.6	14.6	14.7	14.7	14.3				14.5
Hours Per Ktonne	(hr/kt)		0.1959	0.1961	0.1895	0.1907	0.1912	0.1923	0.1942	0.1944	0.1946	0.1919	0.1905	0.1967	0.1971	0.1918	0.1894	0.1913	0.1939	0.1944	0.1957	0.1950	0.1906				0.1930
Mill Hours	(hours)		118	848	5,604	8,760	8,759	8,089	8,089	8,089	8,089	8,089	8,095	8,100	8,100	8,104	8,760	8,760	8,760	8,760	8,760	8,760	2,798				152,290
SOX Mill Ore Stock:																											
NSR Cutoff Grade	(C$)	11.00	11.00	11.00	11.00																						
Ore Ktonnes	(kt)	20	2,806	9,950	19,634																						32,410
NSR Value	(C$/t)	23.29	22.65	24.65	25.25																						24.84
Total Copper	(%)	0.165	0.175	0.190	0.308																						0.260
Weak Soluble Copper	(%)	0.059	0.058	0.065	0.104																						0.088
Gold	(g/t)	0.577	0.540	0.581	0.430																						0.486
Molybdenum	(%)	0.0202	0.0185	0.0207	0.0281																						0.0250
Silver	(g/t)	2.2300	2.4500	2.95	2.03																						2.35
Recovered Copper	(%)	0.084	0.095	0.103	0.182																						0.150
Bond Work Index	(Kwh/t)	13.1	13.6	13.8	14.0																						13.9
Hours Per Ktonne	(hr/kt)	0.17418	0.18133	0.18324	0.18648																						0.18503
Mill Hours	(hours)	3	509	1,823	3,661																						5,997
Low Grade Mill Ore:																											
NSR Cutoff Grade	(C$)			7.90	7.90	7.90	7.90	7.90	7.90	7.90	7.90	7.90	7.90	7.90	7.90	7.90	7.90	7.90	7.90	7.90	7.90						
Ore Ktonnes	(kt)			184	839	7,187	3,396	10,772	16,513	22,234	11,972	6,775	8,299	5,575	4,944	9,461	9,029	6,670	8,620	7,142	4,216						143,828
NSR Value	(C$/t)			10.24	10.07	11.98	10.54	11.53	10.47	11.01	10.72	10.34	10.53	10.68	10.27	9.83	9.78	9.56	9.39	8.75	8.74						10.40
Total Copper	(%)			0.139	0.137	0.188	0.134	0.169	0.138	0.118	0.105	0.103	0.108	0.142	0.127	0.104	0.107	0.112	0.104	0.085	0.080						0.122
Weak Soluble Copper	(%)			0.036	0.045	0.034	0.020	0.034	0.021	0.004	0.000	0.001	0.004	0.020	0.014	0.009	0.008	0.010	0.006	0.001	0.000						0.012
Gold	(g/t)			0.169	0.160	0.158	0.167	0.140	0.130	0.158	0.146	0.135	0.132	0.127	0.128	0.141	0.140	0.137	0.118	0.114	0.105						0.139
Molybdenum	(%)			0.0078	0.0127	0.0068	0.0084	0.0123	0.0130	0.0120	0.0155	0.0164	0.0174	0.0134	0.0137	0.0141	0.0128	0.0112	0.0143	0.0153	0.0179						0.0133
Silver	(g/t)			1.80	1.40	1.05	1.11	1.31	1.26	1.39	1.30	1.05	1.13	1.19	1.10	1.16	1.07	1.01	1.08	1.09	1.09						1.19
Recovered Copper	(%)			0.081	0.070	0.132	0.092	0.113	0.095	0.093	0.083	0.080	0.082	0.100	0.090	0.073	0.077	0.080	0.076	0.062	0.058						0.088
Bond Work Index	(Kwh/t)			14.0	14.5	14.6	14.5	14.7	14.7	14.6	14.6	14.5	14.7	14.9	15.0	14.4	14.4	14.5	14.7	14.4	14.4						14.6
Hours Per Ktonne	(hr/kt)			0.1866	0.1926	0.1942	0.1925	0.1951	0.1956	0.1943	0.1942	0.1929	0.1954	0.1986	0.1988	0.1920	0.1908	0.1930	0.1960	0.1909	0.1917						0.1942
Mill Hours	(hours)			34	162	1,396	654	2,102	3,229	4,320	2,325	1,307	1,622	1,107	983	1,816	1,722	1,287	1,689	1,363	808						27,926
Gold Ore:																											
NSR Cutoff Grade	(C$)	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70									
Ore Ktonnes	(kt)	6,580	14,197	21,654	18,667	17,579	12,879	7,809	495	1,072	8,773	13,111	13,352	18,235	1,263	625	925	238									157,454
NSR Value	(C$/t)	9.29	12.66	12.15	9.66	8.09	7.00	5.78	5.28	6.59	6.96	6.89	6.12	6.05	5.19	5.66	5.17	4.72									8.45
Gold	(g/t)	0.330	0.445	0.423	0.333	0.284	0.236	0.190	0.171	0.231	0.243	0.239	0.208	0.201	0.172	0.186	0.167	0.149									0.292
Silver	(g/t)	1.79	2.80	3.09	2.17	1.76	2.30	1.92	1.520	2.740	2.560	2.190	1.860	1.660	1.000	1.110	1.180	1.110									2.21
Total Copper	(%)	0.025	0.040	0.046	0.046	0.027	0.037	0.044	0.049	0.005	0.014	0.021	0.031	0.045	0.045	0.053	0.053	0.059									0.036
Weak Soluble Copper	(%)	0.008	0.011	0.013	0.016	0.008	0.010	0.013	0.015	0.005	0.010	0.009	0.008	0.008	0.010	0.012	0.011	0.012									0.011
Molybdenum	(%)	0.0088	0.0132	0.0182	0.0209	0.0098	0.0114	0.0115	0.0099	0.0209	0.0271	0.0255	0.0255	0.0249	0.0102	0.0096	0.0053	0.0045									0.0181
Tonnage Summary:																											
Direct Feed Mill Ore	(kt)	0	603	4,326	29,571	45,928	45,814	42,056	41,653	41,605	41,576	42,145	42,491	41,175	41,095	42,251	46,249	45,798	45,187	45,064	44,772	44,934	14,676	0	0	0	788,969
SOX Stockpile	(kt)	20	2,806	9,950	19,634	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	32,410
Low Grade Ore	(kt)	0	0	184	839	7,187	3,396	10,772	16,513	22,234	11,972	6,775	8,299	5,575	4,944	9,461	9,029	6,670	8,620	7,142	4,216	0	0	0	0	0	143,828
Gold Ore	(kt)	6,580	14,197	21,654	18,667	17,579	12,879	7,809	495	1,072	8,773	13,111	13,352	18,235	1,263	625	925	238	0	0	0	0	0	0	0	0	157,454
Total Material	(kt)	8,082	20,542	41,376	91,223	100,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000	100,000	95,491	91,636	90,140	69,230	53,200	19,608	0			1,780,528
Waste Material	(kt)	1,482	2,936	5,262	22,512	29,306	37,911	39,363	41,339	35,089	37,679	37,969	35,858	35,015	52,698	47,663	43,797	42,785	37,829	37,934	20,242	8,266	4,932	0	0	0	657,867
Stockpile Rehandle	(kt)				4,929			3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,610							30,915	45,053	44,985	22,875	181,167
	(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	TOTAL
Waste by Mineral Type:																											
Overburden	(kt)	336	690	1,206	4,614	2,311	818	993	1,345	336	353	617	138	1,450	943	723	596	35	0								17,504
Leach Cap	(kt)	1,043	2,187	3,960	16,879	23,787	32,290	25,694	7,534	4,516	14,007	12,718	13,345	13,072	24,510	8,220	4,704	1,127	429								210,022
Supergene Oxide	(kt)	0	9	39	681	919	338	6,724	6,979	6,399	1,407	2,167	3,083	1,679	6,016	6,989	2,712	914	1,948	210	0	0	0				49,213
Supergene Sulfide	(kt)	0	0	0	255	2,014	3,311	5,348	14,727	9,721	11,269	10,027	10,114	12,390	13,166	16,448	15,443	9,337	4,068	353	0	0	0				137,991
Hypogene	(kt)	0	0	0	5	159	943	573	10,690	14,096	10,579	12,401	8,952	6,369	7,931	15,273	20,341	31,372	31,379	37,370	20,242	8,266	4,932				241,873
Waste - Unclassified	(kt)	103	50	57	78	116	211	31	64	21	64	39	226	55	132	10	1	0	5	1							1,264
Undefined	(kt)																										0
TOTAL WASTE	(kt)	1,482	2,936	5,262	22,512	29,306	37,911	39,363	41,339	35,089	37,679	37,969	35,858	35,015	52,698	47,663	43,797	42,785	37,829	37,934	20,242	8,266	4,932	0	0	0	657,867

Table 16-3: Gold Ore Stacking Schedule

	(Units)	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	TOTAL
Leach Pad Stacking Schedule:																				
NSR Cutoff Grade	(C$)	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	
Ore Ktonnes	(kt)	6,580	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	4,874	157,454
NSR Value	(C$/t)	9.29	12.66	12.15	9.66	8.09	7.00	5.96	7.65	8.91	7.06	6.89	6.12	6.05	5.93	6.33	9.96	11.97	12.66	8.45
Gold	(g/t)	0.330	0.445	0.423	0.333	0.284	0.236	0.197	0.265	0.309	0.246	0.239	0.208	0.201	0.197	0.216	0.344	0.416	0.445	0.292
Silver	(g/t)	1.79	2.80	3.09	2.17	1.76	2.30	1.97	1.891	2.135	2.545	2.190	1.860	1.660	1.569	1.891	2.459	3.032	2.800	2.21
Total Copper	(%)	0.025	0.040	0.046	0.046	0.027	0.037	0.043	0.031	0.036	0.015	0.021	0.031	0.045	0.045	0.031	0.044	0.046	0.040	0.036
Weak Soluble Copper	(%)	0.008	0.011	0.013	0.016	0.008	0.010	0.013	0.009	0.013	0.010	0.009	0.008	0.008	0.008	0.009	0.013	0.013	0.011	0.011
Molybdenum	(%)	0.0088	0.0132	0.0182	0.0209	0.0098	0.0114	0.0115	0.0102	0.0181	0.0269	0.0255	0.0255	0.0249	0.0229	0.0243	0.0187	0.0177	0.0132	0.0181
As Produced From Mine:																				
NSR Cutoff Grade	(C$)	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70	3.70		
Ore Ktonnes	(kt)	6,580	14,197	21,654	18,667	17,579	12,879	7,809	495	1,072	8,773	13,111	13,352	18,235	1,263	625	925	238		157,454
NSR Value	(C$/t)	9.29	12.66	12.15	9.66	8.09	7.00	5.78	5.28	6.59	6.96	6.89	6.12	6.05	5.19	5.66	5.17	4.72		8.45
Gold	(g/t)	0.330	0.445	0.423	0.333	0.284	0.236	0.190	0.171	0.231	0.243	0.239	0.208	0.201	0.172	0.186	0.167	0.149		0.292
Silver	(g/t)	1.79	2.80	3.09	2.17	1.76	2.30	1.92	1.520	2.740	2.560	2.190	1.860	1.660	1.000	1.110	1.180	1.110		2.21
Total Copper	(%)	0.025	0.040	0.046	0.046	0.027	0.037	0.044	0.049	0.005	0.014	0.021	0.031	0.045	0.045	0.053	0.053	0.059		0.036
Weak Soluble Copper	(%)	0.008	0.011	0.013	0.016	0.008	0.010	0.013	0.015	0.005	0.010	0.009	0.008	0.008	0.010	0.012	0.011	0.012		0.011
Molybdenum	(%)	0.0088	0.0132	0.0182	0.0209	0.0098	0.0114	0.0115	0.0099	0.0209	0.0271	0.0255	0.0255	0.0249	0.0102	0.0096	0.0053	0.0045		0.0181
Direct to Crusher:																				
Ore Ktonnes	(kt)	6,580	9,125	9,125	9,125	9,125	9,125	7,809	495	1,072	8,773	9,125	9,125	9,125	1,263	625	925	238		100,780
NSR Value	(C$/t)	9.29	12.66	12.15	9.66	8.09	7.00	5.78	5.28	6.59	6.96	6.89	6.12	6.05	5.19	5.66	5.17	4.72		8.13
Gold	(g/t)	0.330	0.445	0.423	0.333	0.284	0.236	0.190	0.171	0.231	0.243	0.239	0.208	0.201	0.172	0.186	0.167	0.149		0.280
Silver	(g/t)	1.79	2.80	3.09	2.17	1.76	2.30	1.92	1.520	2.740	2.560	2.190	1.860	1.660	1.000	1.110	1.180	1.110		2.17
Total Copper	(%)	0.025	0.040	0.046	0.046	0.027	0.037	0.044	0.049	0.005	0.014	0.021	0.031	0.045	0.045	0.053	0.053	0.059		0.035
Weak Soluble Copper	(%)	0.008	0.011	0.013	0.016	0.008	0.010	0.013	0.015	0.005	0.010	0.009	0.008	0.008	0.010	0.012	0.011	0.012		0.010
Molybdenum	(%)	0.0088	0.0132	0.0182	0.0209	0.0098	0.0114	0.0115	0.0099	0.0209	0.0271	0.0255	0.0255	0.0249	0.0102	0.0096	0.0053	0.0045		0.0179
To Gold Ore Stockpile:																				
Ore Ktonnes	(kt)		5,072	12,529	9,542	8,454	3,754					3,986	4,227	9,110						56,674
NSR Value	(C$/t)		12.66	12.15	9.66	8.09	7.00					6.89	6.12	6.05						9.03
Gold	(g/t)		0.445	0.423	0.333	0.284	0.236					0.239	0.208	0.201						0.312
Silver	(g/t)		2.80	3.09	2.17	1.76	2.30					2.190	1.860	1.660						2.27
Total Copper	(%)		0.040	0.046	0.046	0.027	0.037					0.021	0.031	0.045						0.039
Weak Soluble Copper	(%)		0.011	0.013	0.016	0.008	0.010					0.009	0.008	0.008						0.011
Molybdenum	(%)		0.0132	0.0182	0.0209	0.0098	0.0114					0.0255	0.0255	0.0249						0.0186
Stockpile Reclaim:																				
Ore Ktonnes	(kt)							1,316	8,630	8,053	352				7,862	8,500	8,200	8,887	4,874	56,674
NSR Value	(C$/t)							7.00	7.78	9.22	9.66				6.05	6.38	10.50	12.16	12.66	9.03
Gold	(g/t)							0.236	0.270	0.319	0.333				0.201	0.218	0.364	0.423	0.445	0.312
Silver	(g/t)							2.30	1.913	2.055	2.170				1.660	1.948	2.603	3.084	2.800	2.27
Total Copper	(%)							0.037	0.030	0.041	0.046				0.045	0.029	0.043	0.046	0.040	0.039
Weak Soluble Copper	(%)							0.010	0.009	0.014	0.016				0.008	0.008	0.014	0.013	0.011	0.011
Molybdenum	(%)							0.0114	0.0103	0.0178	0.0209				0.0249	0.0254	0.0202	0.0181	0.0132	0.0186

16.5 WASTE MANAGEMENT

Total waste in the IMC mine plan amounts to 657.9 million tonnes. This material is disposed in the tailing storage facility. The material will be placed by trucks and dozers and the pond water level will cover the material soon after it is placed. This material, by domain, is as follows:

- 17.5 million tonnes of overburden.
- 210.0 million tonnes of leach cap material.
- 49.2 million tonnes of supergene oxide material.
- 138.0 million tonnes of supergene sulphide material.
- 241.9 million tonnes of hypogene material.
- And, 1.3 million tonnes that is not classified.

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

- The heap leach pad which at the end of the project will contain 157.5 million tonnes of spent, non-reactive ore.
- A low grade stockpile, amounting to 143.8 million tonnes, which will be processed at the end of the mine life.
- There will also be supergene oxide (SOX) ore stockpile south of the pit to store phase 1 SOX ore. It will be reclaimed during mining years 4 through 12. The maximum size of this facility is estimated at 32.4 million tonnes.
- There will be a stockpile for gold leach ore east of the pit. This is expected to reach a maximum size of 40 million tonnes at the end of Year 3 and will be reclaimed by the end of Year 15.

Figure 16-2 shows the maximum extent of these various facilities. They are all constructed in lifts from the bottom up. The low grade stockpile, gold ore stockpile, and SOX stockpile are designed with 30m lifts at angle of repose with a 20m setback between lifts to make the overall slope angle about 2H:1V. This is assumed to be adequate since these are not permanent facilities.

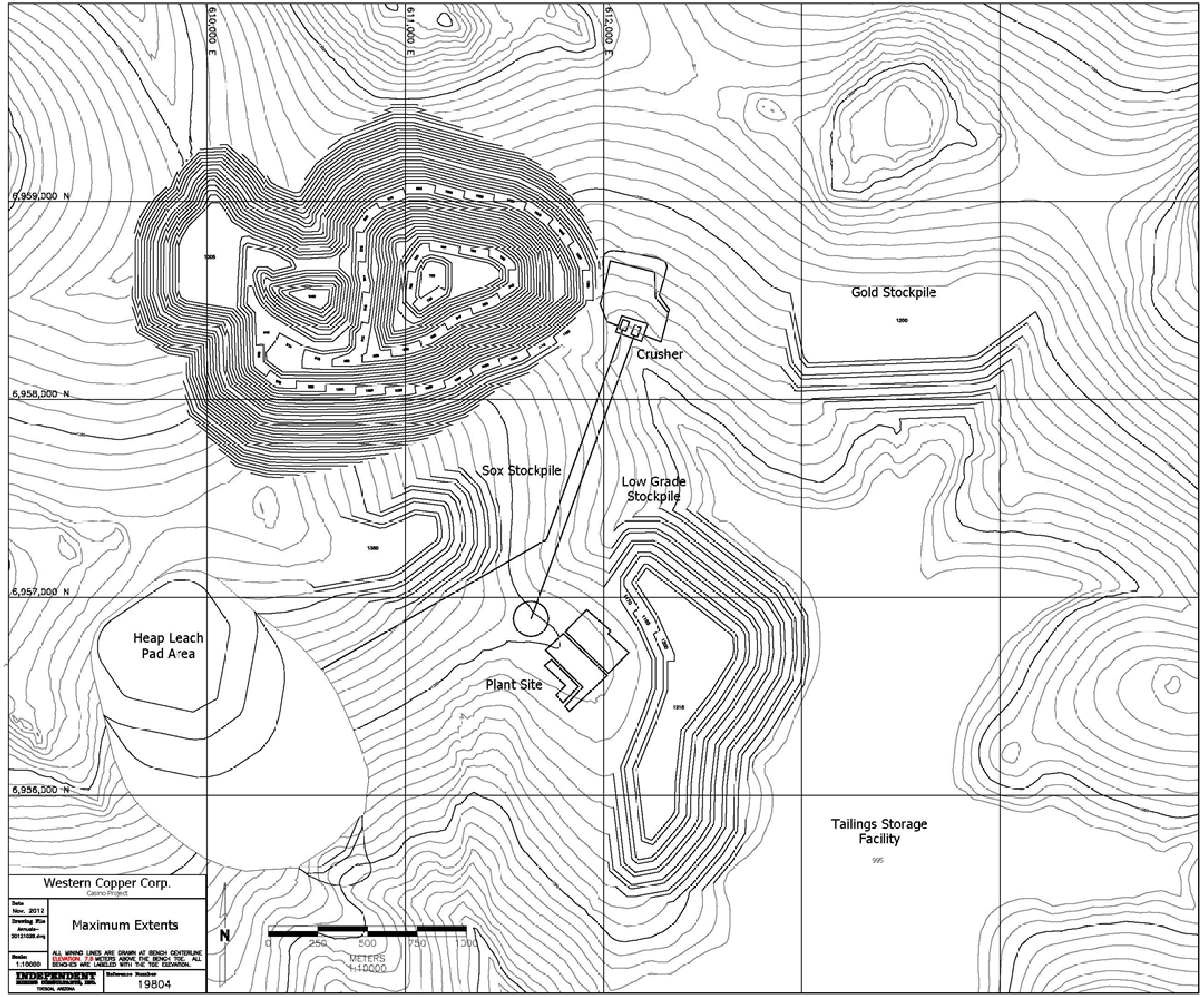


Figure 16-2: Maximum Extent of Pit and Stockpiles

16.6 MINE EQUIPMENT

Mine equipment requirements were sized and estimated on a first principles basis, based on the mine production schedule, the mine work schedule, and estimated equipment shift productivity rates. The size and type of mining equipment is consistent with the size of the project, i.e. peak material movements of 100 million tonnes per year. The mine equipment estimate is based on owner operation and assumes a well-managed mining operation with a well-trained labor pool, and that all the equipment is new at the start of the operation.

Table 16-4 shows the major equipment requirements by time period for Casino. This represents the equipment required to perform the following duties:

- Develop access roads from the mine to the crusher, various stockpiles, and the waste storage area.
- Mine and transport mill ore and gold leach ore to the crushers.
- Mine and transport ore to various stockpiles as required.
- Reclaim stockpiled ore and transport it to the crushers.
- Mine and transport waste to the co-disposal site.
- Maintain the haul roads and stockpiles and truck dumping sites at the co-disposal site.

Table 16-4: Mine Major Equipment Fleet Requirement

Equipment Type	Capacity/ Power	Time Period																								
		-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
P&H 320XPC Drill	(457 mm)	1	1	1	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	1	0	0	0
P&H 4100XPC Cable Shovel	(67.6 cu m)	1	1	1	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	1	1	2
Cat 994F Wheel Loader	(17 cu m)	0	0	0	1	0	0	1	2	2	1	1	1	1	2	2	1	2	1	0	0	0	0	0	0	0
Cat 797F Truck	(360 mt)	4	5	8	17	20	23	22	18	19	24	24	24	22	24	19	21	23	20	19	15	15	12	6	6	6
Cat D11T Track Dozer	(634 kw)	1	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	1	1	1
Cat D10T Track Dozer	(433 kw)	1	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	1	1	1
Cat 854K Wheel Dozer	(597 kw)	1	1	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	2	2
Cat 24M Motor Grader	(397 kw)	1	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	1	1	1
Water Truck - 30,000 gal	(113,562 l)	1	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	1	1	1
Cat 345D Excavator	(2.7 cum)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Atlas Copco ECM 720 Drill	(140 mm)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0
Cat 992K Wheel Loader	(10.7 cu m)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Cat 777F Truck	(90 mt)	3	3	3	5	3	3	6	10	10	6	6	6	6	10	10	7	7	5	3	3	2	2	2	2	2
TOTAL		17	22	26	46	47	50	53	54	55	55	55	55	53	60	55	53	56	50	46	40	34	30	17	17	18

16.7 MINE CAPITAL COSTS

The estimated mine capital cost developed by IMC includes the following items:

- Mine major equipment
- Mine support equipment and initial spare parts
- Mine preproduction development expense

The estimated cost of the following mining facilities was developed by others and is included in the infrastructure capital budget:

- The mine shop and warehouse
- Fuel and lubricant storage facilities
- Explosive storage facilities
- Office facilities

Table 16-5 summarizes the mine capital cost by category for initial and sustaining capital. The initial capital period is considered to be the four year period from Years -3 through Year 1, as these are the years of significant capital build-up.

Table 16-5: Mining Capital – Mine Equipment and Mine Development (US$x1000)

Category		Initial Capital by Time Period				Initial Capital	Sustaining Capital	Total Capital
		Yr -3	Yr -2	Yr -1	Year 1			
Major Equipment		85,281	7,675	21,781	141,770	256,507	160,070	416,577
Support Equipment @	15.00%	12,792	1,151	3,267	21,265	38,476	24,011	62,487
Equipment Subtotal		98,073	8,827	25,048	163,035	294,983	184,081	479,064
Equipment Contingency @	10.0%	9,807	883	2,505	16,304	29,498	0	29,498
Mine Development		23,537	55,306	76,952	0	155,795	0	155,795
TOTAL MINE CAPITAL		131,418	65,015	104,504	179,339	480,276	184,081	664,357
Exclusions: Mine shop and warehouse, fuel and lubricant storage, explosives storage, and offices.								

Mine preproduction development of $155.8 million is based on the estimated mine operating costs during the preproduction period. The cost estimate is based on owner operating costs with large equipment plus a contingency to provide additional allowance for additional road construction or other site preparation and subcontracting portions of the mining, etc. Table 16-6 shows the components of the cost during mine development by year. Total preproduction development is estimated as 70.0 million tonnes, so the unit rate amounts to $2.23 per tonne.

Table 16-6: Mine Development Direct Costs Plus Contingency

Item	Year -3	Year -2	Year -1	Total
Owner Operating Cost – Large Equip	19,037	47,406	69,352	135,795
Mine Development Contingency	4,500	7,900	7,600	20,000
Total Mine Development Cost	23,537	55,306	76,952	155,795
%Contingency	23.6%	16.7%	11.0%	14.7%

16.8 MINE OPERATING COSTS

Table 16-7 summarizes the mine operating costs. Total cost, the cost per total tonne, and cost per ore tonne are shown by various time periods. The $155.8 million preproduction development cost is the source of the mine development capital cost reported on Table 16-5 in the previous section. During commercial production the unit costs for mining are $1.513 per total tonne and $3.054 per ore tonne.

Table 16-7: Summary of Total and Unit Mining Costs

Category	Total Material (kt)	Ore (kt)	Total Cost (US$)	Cost Per Total Ton(ne) (US$/t)	Cost Per Ore Ton(ne) (US$/t)
Mine Development (Years -3 to -1)	70,000	0	155,795	2.226	0.000
Commercial Production (Years 1 to 22)	1,948,369	965,207	2,947,482	1.513	3.054
All Time Periods	2,018,369	965,207	3,103,277	1.538	3.215
Commercial Production Years 1 - 5	513,298	217,151	759,870	1.480	3.499
Commercial Production Years 6 - 10	526,405	226,992	828,778	1.574	3.651
Commercial Production Years 11 - 15	532,660	227,790	821,638	1.543	3.607
Commercial Production Years 16-18	212,570	134,770	364,228	1.713	2.703
Commercial Production Years 19-22 (Stockpile)	163,436	158,504	172,968	1.058	1.091

Table 16-7 also breaks out the mining cost by several time periods. Years 19 to 22 are low grade stockpile re-handle.

The estimate is based on assumed prices for commodities such as fuel, explosives, parts, etc. that are subject to wide variations depending on market conditions. It is also assumed that the truck fleet, including the water trucks, is fueled by a combination of liquid natural gas (LNG) and diesel fuel. The current estimate is based on the following estimated prices for key commodities:

- Diesel fuel delivered to the site for $1.041 per liter,
- LNG/diesel for trucks delivered at $0.48 per liter diesel equivalent,
- Electrical power at $0.10 per kwh,
- Bulk blasting agents at $0.75 per kg delivered to the site, and
- Tires at approximately 75% of US list prices.

16.9 SUMMARY OF MINE CAPITAL AND OPERATING COSTS

Table 16-8 summarizes the Casino mine capital and operating costs for the life of the project.

Table 16-8: Summary of Mine Capital and Operating Costs

MINE CAPITAL COSTS:	Units	Yr -3	Yr -2	Yr -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	TOTAL
Initial Equipment Capital	($x1000)	98,073	8,827	25,048	163,035	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	294,983
Sustaining Equipment Capital	($x1000)	0	0	0	0	29,193	22,531	1,943	14,526	0	8,369	0	13,774	11,344	18,007	0	6,663	36,703	0	21,028	0	0	0	0	0	0	184,081
Contingency (Equipment Only)	($x1000)	9,807	883	2,505	16,304	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	29,498
Mine Development	($x1000)	23,537	55,306	76,952	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	155,795
Mine Infrastructure	($x1000)																										0
TOTAL MINE CAPITAL COST	($x1000)	131,418	65,015	104,504	179,339	29,193	22,531	1,943	14,526	0	8,369	0	13,774	11,344	18,007	0	6,663	36,703	0	21,028	0	0	0	0	0	0	664,357
MINE OPERATING COST:																											
Total Material	(kt)	0	0	0	96,152	100,000	100,000	104,916	112,230	111,653	103,952	103,600	103,600	103,600	111,462	112,110	108,200	104,378	96,510	90,140	69,230	53,200	50,523	45,053	44,985	22,875	1,948,369
Total Ore	(kt)	0	0	0	34,500	45,928	45,814	45,656	45,253	45,205	45,176	45,745	46,091	44,775	44,695	45,861	46,249	45,798	45,187	45,064	44,772	44,934	45,591	45,053	44,985	22,875	965,207
Total Operating Cost	($x1000)	0	0	0	136,887	148,085	157,705	160,138	157,055	159,557	167,700	168,580	170,677	162,263	174,492	160,018	162,709	169,993	154,427	142,980	121,961	99,288	74,930	40,353	39,142	18,542	2,947,482
Cost Per Total Tonne	(US$/t)	0.000	0.000	0.000	1.424	1.481	1.577	1.526	1.399	1.429	1.613	1.627	1.647	1.566	1.565	1.427	1.504	1.629	1.600	1.586	1.762	1.866	1.483	0.896	0.870	0.811	1.513
Cost Per Ore Tonne	(US$/t)	0.000	0.000	0.000	3.968	3.224	3.442	3.507	3.471	3.530	3.712	3.685	3.703	3.624	3.904	3.489	3.518	3.712	3.418	3.173	2.724	2.210	1.644	0.896	0.870	0.811	3.054

17 RECOVERY METHODS

17.1 PROCESS DESCRIPTION

The Casino process plant will consist of two processing facilities, one for sulphide ore and one for oxide ore.

The sulphide ore processing facility will produce mineral concentrates of copper and molybdenum using conventional flotation technology. The copper concentrate will be dewatered and transported as a filtered cake by highway trucks. The molybdenum concentrate will be dewatered and packaged in super sacks for transport. Gold and silver contained in the sulphide ore will be recovered as a fraction of the copper concentrate.

The oxide ore processing facility will produce gold and silver Doré bars via heap leach and carbon adsorption technology. Copper contained in the oxide ore will be recovered as a copper sulphide precipitate using SART technology.

Figure 17-1 is a simplified schematic of the overall process for processing both the sulphide ore and the oxide ore. This provides the basis for the process description that follows.

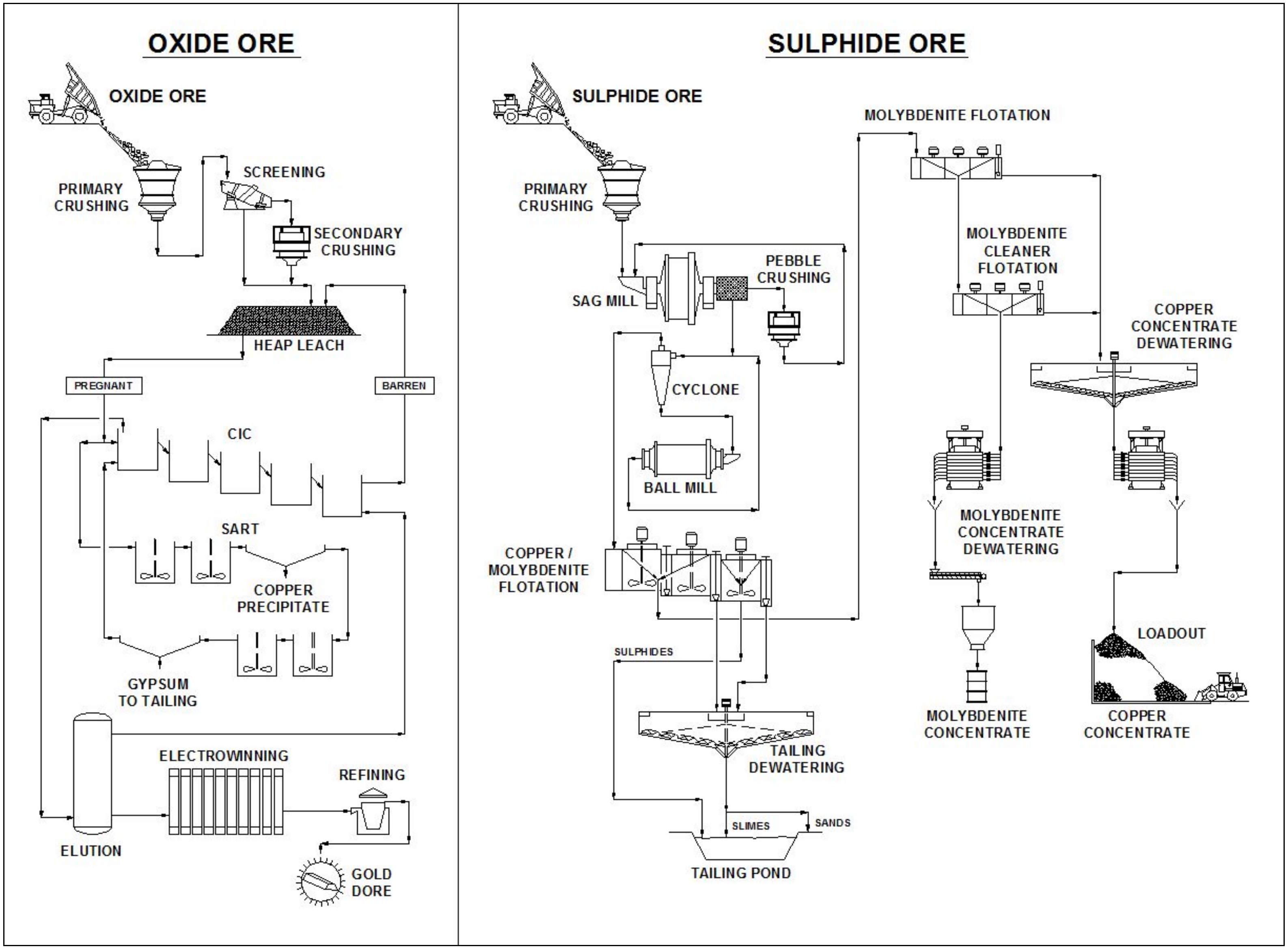


Figure 17-1: Overall Process Flowsheet

17.2 SULPHIDE ORE PROCESS PLANT DESCRIPTION

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

- Size reduction of the run-of-mine (ROM) to minus 200 mm.
- Stockpiling primary crushed ore and then reclaiming with feeders and a belt conveyor.
- Size reduction of the ore in a semi-autogenous (SAG) mill - ball mill grinding circuit with pebble crushing.
- Concentration and separation of the copper and molybdenum sulphide minerals by froth flotation to produce a bulk (copper/molybdenum) concentrate.
- Separation of the bulk concentrate into separate copper and molybdenum concentrates.
- Final copper concentrate will be thickened, filtered, and loaded in highway haul trucks for shipment.
- Final molybdenum concentrate will be filtered, dried, and packaged in bags for shipment.
- Concentration of the bulk flotation tailing in a pyrite flotation circuit. Pyrite flotation circuit tailing will have a low sulphide sulfur concentration.
- Subaqueous deposition of the pyrite concentrate in the tailing storage facility.
- Flotation tailing will be thickened and transported by a gravity pipeline to a tailing impoundment area. The tailing will be cycloned with underflow recovered as sand for tailing dam construction and overflow reporting to the tailing disposal impoundment site
- Storing, preparing, and distributing reagents used in the sulphide ore process.
- 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.

17.2.1 Crushing and Coarse Ore Stockpile

Run of Mine (ROM) sulphide ore will be trucked from the mine to the primary crusher, and fed to the crusher via a dump pocket. The primary crusher will be a 1,524 mm x 2,870 mm (60”x113”), or equivalent, gyratory crusher, with an open side setting of 200 mm. The crushed ore will drop into a discharge bin equipped with an apron feeder. The apron feeder will discharge onto a belt conveyor that will discharge the primary crushed ore to a covered, conical ore stockpile.

Primary crushed ore will be stockpiled on the ground in a covered, conical ore stockpile. A reclaim tunnel will be installed beneath the stockpile. The stockpile will contain approximately 75,000 tonnes of “live” ore storage. Ore will be moved from the “dead” storage area to the “live” storage area by front-end loader or bulldozer.

Ore for the single grinding line will be withdrawn from the coarse ore reclaim stockpile by variable speed, apron feeders. The feeders will discharge to a conveyor belt which will provide the new feed to the SAG mill in the primary grinding circuit.

17.2.2 Grinding and Classification

Ore will be ground to rougher flotation feed size in two stages: first, a primary semi-autogenous (SAG) mill circuit and, second, a ball mill circuit. The SAG mill will operate in closed circuit with a pebble trommel screen and pebble crushers. The ball mills will operate in closed circuit with hydrocyclones.

The SAG mill, 12.2 m diameter x 7.9 m Effective Grinding Length (EGL), will be equipped with a 29 MW gearless wrap-around drive. SAG mill product will discharge through a trommel screen. Trommel undersize will flow by gravity to the primary cyclone feed sump where it will combine with the discharge of the ball mills. Trommel oversize will be transported by belt conveyors to the pebble crushing circuit.

The pebble crushing circuit will consist of a surge bin, belt feeders and three short-head, cone type crushers, each equipped with 750 kW, or equivalent, drives. The cone crushers will discharge onto the SAG feed conveyor. Pebbles may bypass the pebble crushing circuit via a diverter gate, ahead of the pebble crusher surge bin, to the SAG feed conveyor.

Secondary grinding will be performed in two ball mills operated in parallel. Each ball mill, 8.5 m diameter x 13.4 m EGL, will be equipped with a 22 MW gearless wrap-around drive. Each ball mill will operate in closed circuit with two parallel clusters of hydrocyclones. Discharge from both ball mills will be combined with the undersize from the trommel in the primary cyclone feed sump and will be pumped to the hydrocyclone clusters via variable speed horizontal centrifugal slurry pumps. Hydrocyclone underflow will return by gravity to the ball mills. Hydrocyclone overflow (final grinding circuit product), with a target particle size distribution of 80 percent finer than 200 microns, will flow by gravity to the flotation circuit.

17.2.3 Flotation

17.2.3.1 Bulk (Copper/Molybdenum) Flotation

Hydrocyclone overflow will flow by gravity to the bulk (copper-moly) flotation circuit. The copper-moly flotation circuit will consist of two rows of mechanical rougher flotation cells, two rows of mechanical first cleaner flotation cells, two concentrate regrind mills operated in closed circuit with hydrocyclones, one row of mechanical second cleaner flotation cells, and four copper-moly third cleaner flotation column cells.

Rougher flotation concentrate will flow by gravity to a sump and will be pumped by variable speed, horizontal centrifugal pumps to the first cleaner flotation circuit. Tailing from the rougher flotation cells will flow by gravity to the pyrite scavenger flotation circuit.

Pyrite concentrate will join the first cleaner tailing at the pyrite thickener. Tailing from the pyrite flotation circuit (final tailing) will flow by gravity to the tailing thickeners.

First cleaner flotation concentrate will flow by gravity to the regrind cyclone feed sump. Tailing from the first cleaner flotation cells will be combined with the concentrate from the pyrite flotation section in the pyrite thickener.

Copper-moly concentrate regrinding will be performed in two vertical tower mills operated in parallel. The vertical mills will operate in closed circuit with hydrocyclones. Vertical mill discharge will be combined with copper-moly first cleaner flotation concentrate in the regrind cyclone feed sump and will be pumped by variable speed, horizontal centrifugal slurry pump to hydrocyclone clusters. Hydrocyclone underflow will be split by dedicated hydrocyclones for each regrind mill and will report back to the regrind mills. Hydrocyclone overflow (final regrind circuit product), with a target particle size distribution of 80 percent finer than 25 microns, will flow by gravity to the copper second cleaner flotation circuit.

Second cleaner concentrate will flow by gravity to the third cleaner feed sump and will be pumped by horizontal centrifugal pumps to the third cleaner column for upgrading. Tailing from the second cleaner flotation cells will return to the first cleaner flotation circuit.

Third cleaner concentrate will flow by gravity to the molybdenum separation circuit. Tailing from the third cleaner flotation columns will return to the second cleaner flotation circuit.

The quantity and size of the flotation cells that will be installed in the bulk flotation circuit are shown in Table 17-1.

Table 17-1: Bulk Flotation Cells

STAGE	QUANTITY OF CELLS	SIZE OF CELLS (m^3)
Copper Rougher	14	300
Pyrite Scavenger	8	300
Copper 1st Cleaner	12	100
Copper 2nd Cleaner	6	100
Copper 3rd Cleaner	4	3.05 m dia. column

Flotation reagents will be added at several points in the bulk flotation circuit as required.

17.2.3.2 Molybdenite Flotation

Concentrate from the final cleaner of the bulk flotation circuit with report to a copper-moly thickener. Thickened copper-moly concentrate will be pumped by variable speed, horizontal centrifugal slurry pumps to the molybdenite (moly) flotation circuit.

The moly flotation circuit will consist of two agitated rougher conditioning tanks, one row of separation (rougher) flotation cells, one row of first cleaner flotation cells, a concentrate regrind circuit, one second cleaner flotation column, one third cleaner flotation column, and one fourth cleaner flotation column.

Concentrate from the moly rougher cells will be pumped to the moly first cleaner flotation cells. Tailing from the moly rougher cells, which will be the final copper concentrate flotation product, will flow by gravity to the copper concentrate thickener.

Concentrate from the moly first cleaner cells will flow by gravity to the moly concentrate regrind circuit. Tailing from the moly first cleaner flotation cells will flow by gravity to the copper concentrate thickener.

In order to reduce consumption of NaHS and nitrogen, the rougher and first cleaner cells will be hooded and the flotation gas will be recycled.

Moly concentrate regrinding will be performed in vertical mill, operated in open circuit. First cleaner moly concentrate will flow by gravity to a regrind sump and be pumped by a variable speed, horizontal centrifugal slurry pump through the mill. Reground moly first cleaner concentrate will be pumped to the moly second cleaner flotation column.

Concentrate from the moly second cleaner column will flow by gravity to the moly third cleaner flotation circuit. Tailing from the moly second cleaner flotation column will be pumped to the moly first cleaner flotation circuit.

Concentrate from the moly third cleaner column will flow by gravity to the moly fourth cleaner flotation circuit. Tailing from the moly third cleaner flotation column will be recycled to the moly second cleaner flotation circuit.

Concentrate from the moly fourth cleaner column, which will be the final moly concentrate flotation product, will flow by moly concentrate dewatering circuit. Tailing from the moly fourth cleaner column will be recycled to the moly third cleaner flotation circuit.

The quantity and size of the flotation cells that will be installed in the moly flotation circuit are shown in Table 17-2.

Table 17-2: Moly Flotation Cells

STAGE	QUANTITY OF CELLS	SIZE OF CELLS (m^3)
Moly Rougher	7	8.50
Moly 1st Cleaner	7	2.83
Moly 2nd Cleaner	1	2.0 m dia. column
Moly 3rd Cleaner	1	1.25 m dia. column
Moly 4th Cleaner	1	1.0 m dia. column

Flotation reagents will be added at several points in the moly flotation circuit as required.

17.2.4 Concentrate Dewatering and Storage

17.2.4.1 Copper Concentrate Dewatering

Moly rougher flotation tailing (copper concentrate) and moly first cleaner flotation tailing (copper concentrate) will flow by gravity to a copper concentrate thickener. Thickened copper concentrate will be filtered in three tower type copper concentrate pressure filters. Filter cake will discharge to conveyor belt that will discharge to a covered copper concentrate stockpile.

Copper concentrate will be reclaimed by front-end loader onto highway haulage trucks. The loaded haul trucks will proceed to a wash station and be cleaned before exiting the concentrate load out area. This procedure will ensure against tracking of concentrate from the facility.

17.2.4.2 Molybdenite Concentrate Dewatering

Moly concentrate from the final moly cleaner flotation circuit will flow by gravity to an agitated filter feed tank. Moly concentrate slurry will be filtered in one tower type moly concentrate pressure filter. Filter cake will discharge to a Holo-Flite type dryer. The dryer will discharge to the moly concentrate storage bin.

Moly concentrate will be withdrawn from the dried moly concentrate storage bin by a packaging system and will be bagged in super-sacks for shipment by trucks to market.

17.2.5 Pyrite Concentrate Deposition

Pyrite concentrate and tailing from the copper first cleaner flotation circuit will collect in a pyrite thickener. Thickened pyrite concentrate will be flow by gravity to the Tailings Management Facility ("TMF") for subaqueous deposition.

17.2.6 Tailing Dewatering

Tailing from the pyrite scavenger flotation will flow by gravity to two tailing thickeners operated in parallel. Overflow solution from the tailing thickeners will be pumped by horizontal centrifugal pumps to the process water pond for reuse in the mill. Thickened tailing will flow by gravity to the TMF.

17.2.7 Reagents

Reagent storage, mixing, and distribution will be provided for all of the reagents used in the sulphide processing circuits. The following is a list of the reagents used in the process plant:

- Sodium-diisobutyl dithiophosphinate (Aerophine 3418A Promoter)
- Sodium diethyl dithiophosphate/sodium di-secondary butyl dithiophosphate (Aerofloat 208 Promoter)
- Methyl Isobutyl Carbinol (MIBC, frother)
- Pebble Lime (CaO, pH modifier)
- Fuel Oil (#2 Diesel fuel, moly collector)
- Sodium Hydrosulfide (NaHS, copper mineral depressant)
- Flomin D-910 (copper mineral depressant)
- Flocculant
- Potassium amyl xanthate (PAX – pyrite flotation)

17.2.8 Water System

17.2.8.1 Fresh Water

The water source for Casino is the Yukon River, 17 km away.

Fresh water will be pumped from the fresh water intake tank through a series of booster stations, pumps, and pipelines to the plant site area. Fresh water will collect in a fresh/fire water pond at the process site. The design capacity of the fresh water collection and transfer system will be approximately 3,400 cubic meters per hour.

Fresh water and fire water will be supplied from the fresh/fire water pond to the facility by gravity. Fresh water will be distributed to:

- A chlorinator package and subsequent potable water tank, for use in offices, laboratory, dry, and rest rooms
- A mine water tank for mine road dust control
- A gland seal water tank to supply seal water for mechanical equipment
- The ADR Fresh/Fire water tank
- The process water pond
- The fire water distribution system in the mill site area

17.2.8.2 Process Water

Water reclaimed from the Tailing Management Facility (TMF), designated as process water, will be pumped from water barges at the TMF pond through a series of booster tanks, booster pumps, and pipelines to the plant site area. Process water will collect in a process water pond at the process site. Overflow solution from the tailing, bulk concentrate, and copper concentrate thickeners will also be pumped to the process water pond. Process water will be distributed from the process water pond by gravity flow through a pipeline to mill process water usage points.

17.3 OXIDE ORE PROCESS PLANT DESCRIPTION

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

- Size reduction of the run-of-mine (ROM) ore to minus 200 mm using a primary gyratory crusher.
- Size reduction of the primary crushed ore to minus 50 mm through screening and a secondary cone crusher.
- Stacking crushed ore by overland conveyors and a stacker onto a heap leach pad and, subsequently, leaching the ore with cyanide solution.
- Recovering gold and silver from the pregnant leach solution on activated carbon in carbon in column tanks (CIC).
- Recovering copper from the pregnant leach solution by the Sulphidization, Acidification, Recycling and Thickening (SART) process.
- Treating gold and silver loaded carbon recovered from the CIC circuit by acid washing, cold stripping with cyanide solution to remove copper, hot stripping with caustic solution to remove gold, and thermal reactivation of the 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.

17.3.1 Crushing, Conveying, and Stacking

The Oxide ore will have a primary crusher and conveyor system separate from the Sulphide ore.

Run of mine (ROM) oxide ore will be trucked from the mine to the primary crusher apron feeder. Alternatively, ROM may be stockpiled in the ROM Stockpile if the primary crusher is down for maintenance. A front end loader will reclaim ore from the stockpile and dump onto the primary crusher apron feeder. The apron feeder will provide the feed to the primary crusher. The primary crusher will be a 1,372 mm x 1,905 mm (54”x75”), or equivalent, gyratory crusher. The primary crusher will discharge onto a secondary screen feed belt conveyor.

Secondary screen feed conveyor will discharge onto the secondary screen. Screen undersize will discharge onto the fine ore transfer conveyor. Screen oversize will discharge onto secondary screen discharge belt conveyor, which discharges into secondary crusher feed bin.

Secondary crusher belt feeder will draw ore from the bin and provide feed to the secondary cone crusher, which will be equipped with a 750 kW, or equivalent, drive. Cone crusher discharge will combine with undersize from the secondary screen on the fine ore transfer conveyor. Lime will be added to this conveyor, which discharges onto the intermediate transfer conveyor.

The intermediate transfer conveyor discharges onto a series of overland transfer conveyors, with the last overland conveyor discharging onto the telescoping stacker feed conveyor. The telescoping stacker feed conveyor discharges onto the leach pile stacking conveyor, which places crushed ore onto the heap leach pile.

17.3.2 Heap Leaching

The heap leach pad will consist of liners and a low-permeability soil liner. A perforated pipe drainage system will enhance drainage of leach solutions away from the liner, reduce hydrostatic head, and facilitate pregnant solution recovery. A 600 mm thick layer of sized river gravel or crushed and screened rock will be placed on the liner. This layer is necessary to protect the liner from crushed ore stacking and to allow unimpeded leach solution flow to the drainage pipe system.

Barren process solution will be applied to ore lots. Solution will be applied with drip emitters to minimize evaporation losses. When an ore lot has completed the primary leach cycle, solution application will be stopped and another ore lift (or layer) will be placed on top of the previous lift. Leach solution application will resume. The process of layering and leaching the ore will repeat for a maximum of eight ore lifts or layers on the leach pad. When the last process leach cycle is completed on the last lift, the ore heaps will be rinsed with freshwater to recover the remaining gold and rinse the residue.

Pregnant solution discharging from the ore heaps will be collected in a network of pipe placed throughout the overliner material that will direct the solution to the in-heap collection area.

Pregnant solution will be pumped from the in-heap collection area using horizontal, centrifugal pregnant solution pumps. The pump discharge pipes will be combined in a single pipeline to the carbon-in-column (CIC) / SART circuit for recovery of gold and copper.

An events pond will be installed to handle any overflow that might occur during a large precipitation event. Water that accumulates in the events pond will be periodically pumped by a submersible pump to the barren solution system feeding leach solution to the ore heaps.

The pond system has been sized to contain normal operating solutions and stormwater. Ponds will be fenced to reduce the risk of danger to wildlife.

17.3.3 Carbon ADR Plant/SART

Gold and silver will be recovered from heap leach pregnant solution by adsorption of their ions on activated carbon followed by desorption and electrowinning of a gold and silver solid product. Copper will be recovered from the pregnant solution by the SART process where the copper will be precipitated by the addition of sulphide to produce a copper sulphide product.

The process steps required to recover gold and silver by the carbon adsorption method include:

- loading gold and silver on activated carbon in a carbon in column (CIC) circuit
- acid washing of the carbon to remove water scale and acid soluble copper
- cold stripping of carbon (elution) to remove copper
- stripping gold and silver from the carbon using a hot caustic solution
- electrowinning gold and silver from the stripping solution in a precious metal sludge using an electrolytic cell
- reactivating stripped carbon by thermal regeneration
- melting the precious metal sludge in a crucible furnace to produce Doré bars.

The process steps required to recover copper by the SART method include:

- bleeding a portion of the pregnant solution to the SART process
- adding sodium hydro-sulphide to the solution
- decreasing the pH of the solution with acid, thereby precipitating copper
- removing the copper precipitate from the solution by thickening, filtration, and drying
- increasing the pH of the solution with lime, thereby precipitating gypsum
- removing the gypsum from the solution by thickening
- shipping the filtered copper sulphide product to a smelter for refining

17.3.3.1 Carbon-in-Column (CIC) Circuit

Gold will be recovered from pregnant leach solution in a two-train, five-stage carbon-in-column (CIC) circuit. Pregnant leach solution will pass through a stationary screen to remove trash prior to being introduced into the CIC tank line. The majority of the pregnant solution will pass directly to the CIC circuit, and the remainder will be processed for copper recovery by the SART process and then be returned to the CIC circuit.

The CIC circuit will consist of two parallel trains of five CIC tanks operated in series. Each CIC tank will be a flat bottom tank with an internal distribution plate between the bottom and the carbon charge. It will be possible to bypass any tank by using a manually operated dart-valve.

Solution (barren solution) will exit the CIC lines, be combined, and pass through a safety, single deck, vibrating screen to capture carbon unintentionally washed from the carbon columns. Screen oversize (carbon) will be transferred by gravity to a carbon quench tank.

Carbon will be advanced through the circuit by a series of recessed impeller, horizontal centrifugal pumps. Loaded carbon advanced from the first CIC vessel will be pumped by the first carbon transfer pump from each train to a carbon distributor tank. Carbon will be advanced through from tank to tank by additional, dedicated carbon transfer pumps – one pump for each CIC vessel.

Regenerated and new carbon will be sized by screening on single-deck, vibrating screens. Screen undersize will flow by gravity to a carbon fines tank. Screen oversize will flow by gravity to the last carbon column on either of the CIC lines when carbon is advanced.

Barren solution will discharge from the CIC tank line and flow into a barren solution tank. Vertical turbine pumps will pump barren solution from the barren solution tank to the heap leach barren solution distribution system. The pump discharge lines will be combined to a single pipeline to the heap leach area.

Cyanide solution will be added to the barren solution tank and/or the first CIC columns.

17.3.3.2 SART Circuit

A portion of the heap leach pregnant solution, approximately 150 cubic meters per hour, will be pumped to the SART process. Sodium hydrosulphide will be added to the solution through an inline mixer. Downstream of this in-line mixer, sulfuric acid will be added to the solution prior to mixing in the precipitation reactor.

The discharge from the precipitation reactor will flow by gravity to a covered copper sulphide thickener. Most of the underflow from this thickener will be recirculated to the precipitation reactor, while the balance of the thickener underflow will be advanced to a copper sulphide neutralization tank. Overflow solution from the copper sulphide thickener will collect in a neutralization feed tank.

Sodium hydroxide will be added to the copper thickener underflow neutralization tank. Neutralized copper sulphide, thickened slurry will be fed to the copper sulphide filter press. The filter cake will discharge onto the copper sulphide filter cake belt conveyor, which discharges into a bin. The filtered cake will pass through a dryer. Dried copper sulphide will collect in a bin, be loaded into supersacks, and transported by flatbeds trucks to market.

Copper sulphide thickener overflow will collect in a neutralization feed tank. Neutralization feed pump will transfer the overflow solution to the first neutralization reactor. Lime and recirculated gypsum thickener underflow will be added to this reactor, which overflows into the second neutralization reactor.

Neutralized solution will flow by gravity to a covered, gypsum thickener. A portion of the thickener underflow will be recirculated to the first neutralization reactor. The balance of the underflow will be pumped to a gypsum holding tank.

Gypsum thickener overflow will collect in a treated solution tank, where an anti-scalant will be added. This treated solution will be returned to the CIC circuit.

A scrubber will collect fumes from the precipitation reactor, the copper sulphide thickener, the copper sulphide neutralization tank, the neutralization feed tank, the neutralization reactors, the gypsum thickener, and the sodium hydro-sulphide storage tank. The pH of the scrubber will be adjusted by NaOH. Some scrubber discharge will be recirculated, but net scrubber discharge will report to the first neutralization reactor.

17.3.3.3 Carbon Acid Wash

Loaded carbon will be acid washed in a column. A dilute hydrochloric acid solution will be pumped into the bottom of the acid wash tank, flow up through the vessel, and overflow to the acid tank. This pump will either be operated to circulate solution through the carbon or the carbon may be left to soak. After completion of the acid wash solution cycle, the batch of spent acid solution will be pumped to the acidifying reactor tank in the SART circuit.

A caustic (basic) solution will be pumped into the bottom of the acid wash tank at the end of its cycle, flow up through the vessel, and overflow to the acid tank. The caustic solution pump will either be operated to circulate solution through the carbon or to fill the acid wash tank and allow the carbon to soak. Upon completion of the acid wash cycle, the caustic solution will be used to neutralize the acid solution to pH 8 to 10. Neutralized solution will be pumped to the barren tank.

17.3.3.4 (Copper) Cold Stripping

The acid washed loaded carbon will be cold stripped to remove copper before hot stripping to remove gold.

Water will be added to the acid washed carbon and the carbon and water slurry will be pumped from the acid wash vessel to one of the two strip columns. Water transferred with the carbon (carbon transfer water) will be drained through an internal screen in the bottom of the cold strip tank and will drain to a carbon fines tank.

Cyanide solution will be pumped into the top of the cold strip tank, flow down through the vessel, and discharge through the internal screen. The carbon will then be soaked for the cold strip cycle. Upon completion of the cold strip cycle, the strip solution will be transferred to the SART circuit.

The drained carbon will be rinsed with barren solution to remove the copper strip solution. The rinse solution will also be transferred to the carbon fines tank.

17.3.3.4.1 Carbon Elution Circuit

Elution will be by pressure Zadra techniques. Barren strip solution, containing sodium hydroxide (caustic) will be heated by a solution heating system and circulated through the bottom of the elution vessel by a horizontal centrifugal pump. The solution will flow up through the column, exit the top as pregnant solution, and flow through a heat recovery heat exchanger. The cooled solution will flow to the electrowinning (EW) barren return tank.

After completion of the elution cycle, the strip solution will be drained through an internal screen in the bottom of the elution tank and will be transferred to the carbon fines tank.

17.3.3.5 Electrowinning

Gold and silver will be recovered from pregnant strip solution by the electrowinning process.

Pregnant strip solution will be pumped from the EW barren return tank to an electrowinning cell. Knitted stainless steel mesh cathodes will be used in the cells for deposition of the gold and silver. The stripped solution will discharge from the cell and flow back to the EW barren return tank.

When the electrowinning cycle has been completed, the solution will be pumped from the EW barren return tank to the barren strip solution tank. A high pressure, water wash system will remove the metal sludge from the cathodes and wash it into an electrowinning wash/ sludge tank. Sludge slurry will be pumped by a diaphragm pump to a plate and frame, pressure filter. Filter cake will be cleaned from the filters by hand and placed in filter cake boats for refining.

17.3.3.6 Smelting

Filter cake will be mixed with fluxing materials and charged to an electric, crucible furnace. The melted charge will be poured into conical molds. Doré (gold and silver) will sink to the bottom of the mold, and slag glass containing fused fluxes and impurities, will float to the top of the mold. Doré will be sampled for gold content using vacuum tube samples.

After cooling and solidifying, the molds will be dumped and the slag will be knocked off the Doré buttons by hand. A vertical drill press will be available as a back-up sample method for Doré buttons.

Buttons will be cleaned under a water stream using a needle gun, weighed, and stamped with an identification number and weight. Doré buttons will be the final product of the operation and will be stored in a safe until shipment.

Slag will be crushed and screened to recover high-grade chips that will be returned to the melting furnace. Remaining slag will be returned to the heap leach.

Fumes from the melting furnace will be collected through ductwork and cleaned in a bag house dust collector system before discharging to atmosphere.

17.3.3.7 Carbon Regeneration

Barren, stripped carbon will be sized on a single deck, vibrating screen. Screen undersize will flow by gravity to the carbon fines tank. Screen oversize will flow by gravity to a kiln feed tank.

Carbon will be withdrawn from the kiln feed tank by a screw conveyor feeder. The feeder will discharge to a horizontal, propane gas fired, carbon regeneration kiln.

The carbon regeneration kiln will discharge into carbon quench tank. Quenching will be done in a conical bottom tank. Regenerated carbon will be transferred from this tank to an activated carbon storage tank. New carbon, after being soaked and attrition agitated in a separate tank, will be added as required. Carbon and water slurry will be pumped from this tank by recessed impeller, horizontal, centrifugal pump to carbon sizing screens for use in the CIC circuit.

17.3.4 Reagents

Reagent storage, mixing, and distribution will be provided for all of the reagents used in the oxide processing circuits. The following is a list of the reagents used in the process plant:

- Sodium Cyanide (NaCN)
- Caustic (sodium hydroxide, NaOH)
- Pebble Lime (CaO)
- Hydrochloric Acid (HCl)
- Sodium Hydro-Sulphide (NaHS)
- Sulphuric acid (H_2SO_4)
- Activated Carbon
- Antiscalant
- Flocculant

17.3.5 Water System

Water will be supplied from the fresh water pond on-site to an ADR fresh/fire water tank. Water from this tank will supply a variety of destinations to meet process, mine, and fire water requirements.

17.4 PRIMARY PROCESS DESIGN CRITERIA

The design of the Casino ore processing facilities is based on the following criteria. All units of mass are in dry metric tonnes unless otherwise specified.

17.4.1 Primary Process Design Criteria – Sulphide Circuit

17.4.1.1 Run-of-Mine Ore Characteristics

Maximum mine-run ore size, mm	900
Ore specific gravity	2.7
Ore bulk density, mine-run sulphide ore, t/m^3	1.68
Ore bulk density, mine-run waste material, t/m^3	1.68
Ore bulk density, primary crushed, t/m^3	1.68
Ore assay (life of mine average)	
Percent Copper	0.202
Percent Molybdenum	0.0229
Gold, grams per tonne	0.238
Silver, grams per tonne	1.740
Ore assay (for mill design; first five years average)	
Percent Copper	0.293
Percent Molybdenum	0.0236
Gold, grams per tonne	0.357
Silver, grams per tonne	2.050
Ore abrasion index, Bond, (A_i)(g)	0.265
Ore work indices:	
Crushing index, SGS Method, (CI)	29.2
Rod mill work index, Bond, (RW_i) kWh/t	9.9
Ball mill work index, Bond, (BW_i) kWh/t	14.5
Modified Ball Mill Wi, SGS Method, (ModBWi) kWh/t	14.3
SAG power index, SPI (minutes)	52.9
Ore moisture content, %, design	3

17.4.1.2 Production Schedule

Ore Crushing and Milling Rate, average, t/y	43,800,000
Ore Crushing and Milling Rate, average, t/d	120,000

17.4.1.3 Primary Crushing Operating Schedule

Days per year	365
Hours per day	24
Shifts per day	2
Hours per shift	12
% Utilization (excluding start-up)	80
Ore crushing rate, flow sheet design, t/h	6,250
Ore crushing rate, maximum operating, t/h	8,100

17.4.1.4 Grinding, Flotation, and Concentrate Handling Operating Schedule

Days per year	365
Hours per day	24
Shifts per day	2
Hours per shift	12
% Availability (excluding start-up)	93
Ore processing rate, flow sheet design, t/h	5,376.3
Ore processing rate, maximum operating, t/h	6,183

17.4.1.5 Metal Production Schedule Based on Mill Feed Schedule

17.4.1.5.1 Supergene sulphide (SUS) ores, and mixed Supergene (SUS/SOX) ores

Copper recovery to copper concentrate, percent	94% of sulfide copper
Copper concentrate grade, percent copper	TBD
Gold recovery to copper concentrate, percent	69%
Molybdenum recovery to molybdenum concentrate, percent	52.3%
Molybdenum concentrate grade, percent molybdenum	50%
Silver recovery to copper concentrate, percent	60%

17.4.1.5.2 Hypogene (HYP) ores

Copper recovery to copper concentrate, percent	92.1%
Copper concentrate grade, percent copper	TBD
Gold recovery to copper concentrate, percent	66%
Molybdenum recovery to molybdenum concentrate, percent	78.6%
Molybdenum concentrate grade, percent molybdenum	TBD
Silver recovery to copper concentrate, percent	50%

17.4.2 Primary Process Design Criteria – Oxide Circuit

17.4.2.1 Run-of-Mine Ore Characteristics

Mine-run maximum ore size, millimeters	900
Mine-run ore specific gravity	2.65
Mine-run particle size distribution, p80, mm	400
Mine-run ore moisture content, %, design	2
Ore angle of repose, degree	37
Ore angle of draw down, degree	60
Ore gold assay, g/t, design	0.45
Ore silver assay, g/t, design	3.00
Ore copper grade, %, design	0.05
Ore stacked density, t/m^3	1.75

17.4.2.2 Production Schedule

Crushing, Conveying, and Stacking Rate, DTPY	9,125,000
Crushing, Conveying, and Stacking Operating Schedule	
Days per year	300
Hours per day	24
% Availability (excluding start-up)	75
Ore Leaching Rate, DTPY	9,125,000
Leaching and Gold Recovery Operating Schedule	
Days per year	365
Hours per day	24
% Availability (excluding start-up)	98

17.4.2.2.1 Heap Leach Pile

Leach Operating Schedule:	
Days per year	365
Ore placement days per year	300
Hours per day	24
Leach Operating Cycle Days:	
Stack leach pile section	n/a
Cure leach pile	n/a
Leach solution application, primary	60
Leach solution application, secondary	through subsequent ore lifts
Drain solution from leach pile	n/a
Total cycle	n/a
Leach Pile:	
Pad ore storage capacity, tonnes	157,500,000
Pile slope	2.5:1
Maximum ore thickness, meters	150
Number of ore lifts	n/a
Thickness of ore lift, meters	8.0
Ore pile setback, meters	9
(to be used for design of benches between ore lifts and setback from solution collection channels.)	

Leach Pad Liner:
In ponding areas, the liner system will be a double layer composite system, consisting of an 80-mil LLDPE liner, a compacted, low-permeability soil liner, and a 60-mil LLDPE liner. In non-ponding areas, the liner system will be a single-liner composite system, consisting of an 80-mil LLDPE liner and a compacted, low-permeability soil liner. A leak detection layer underlies the low permeability soil liner for both the ponding and non-ponding systems. The leak detection layer consists of a sand layer and perforated collection pipes. The leachate collection system will be installed

within the overliner layer. The collection system will consist of a perforated pipe network, which enhances drainage of the leachate away from the liner, reduces hydrostatic head, and facilitates pregnant solution recovery.

Leach Solution:	
Application method	distribution network with emitters
Application rate, $L/h/m^2$	12
Precipitation, mm/year	500
Pan Evaporation rate, mm/year	308
Retained solution, leach pile % moisture (wet)	10
Operating solution, leach pile % moisture (wet)	12
Barren Solution Flow Rate:	
Average, balanced flow, m^3/h	1,312
Pregnant Solution Flow Rate:	
Average, m^3/h	1,223

17.4.2.2.2 Events Pond

Pond Size:	
Storage capacity, total m^3	112,900

Total storage capacity of the stormwater pond will contain all rainfall over the lined leach pad area with 100 percent runoff from an estimated 100-year, 24-hour storm event.

Pond side slopes	3:1
Max pond depth, meters	20
Freeboard at design containment, meters	0.3

Pond Liner:
The events pond liner system will be a triple layer composite system, consisting of a 60-mil HDPE liner, a compacted, low-permeability soil liner, and a 60-mil HDPE liner. A leak detection layer, consisting of a synthetic 'geonet' drainage layer, underlies the low-permeability soil layer.

18 PROJECT INFRASTRUCTURE

The Project site is located about 300 km northwest of Whitehorse, about 190 km west of Minto, about 200 km northwest of the Village of Carmacks, and about 18 km southwest of the Yukon River, near Patton Hill. The approximate elevation of the mine is 1,300 m.

The mine site is remote from significant population centers, but the site is accessible by several existing winter roads. About 180 km of the roads from Whitehorse to the site are paved. The paved roads include the Alaska Highway and the Klondike Highway. Both highways pass through Whitehorse.

Based on the Port Study by Associated Engineering, the Skagway port is the optimal port to utilize in terms of both concentrate exports and mine supply imports. Whitehorse has an international airport with daily flights to Vancouver. The mine site has an airstrip used to access the site by light aircraft.

The Yukon electrical grid is at a significant distance from the mine and at the current time the utility does not have enough generating capacity to fulfill the requirements for the Casino mine. The cost effective solution to provide power for the project is to build and operate an on-site power plant.

18.1 SITE LAYOUT AND ANCILLARY FACILITIES

18.1.1 General

Project facilities will lie east of Patton Hill. The open pit mine is located between the headwaters of Casino Creek and Canadian Creek and will occupy an area of more than 300 ha. The mine waste rock storage area (WRSA) will cover nearly 210 ha and will be located south of the pit.

A small valley about 1 km south of the pit (within the Tailings Management Facility (TMF) area) will be filled with oxide ore to form the heap leach pad. An earthen embankment at the eastern end of the pad will provide structural support for the heap leach. A spill and runoff control collection pond termed the "Events Pond" will be built directly downhill from the heap leach pad. The heap leach pad, events pond, and embankments will cover about 101 ha in total. The metallurgical recovery plant (gold plant) will lie immediately southeast of the leach heap.

The Tailings Management Facility (TMF) will be located southeast of the pit and plant site within the valley formed by headwaters of Casino Creek. It will retain an ultimate volume of 974 Mt of tailings together with 598 Mt of potentially reactive waste rock. The TMF will cover about 1,120 ha. The TMF will include a water reclaim system to collect process water and return it to the process.

There are two primary crushers: one for the oxide ores and one for the sulphide ores. The primary gyratory crusher pad where each primary crusher will be located will be constructed east of the pit. An overland conveyor belt will carry the sulphide ore 1.6 km to the southwest from the crusher to the coarse ore stockpile, which is located immediately northwest of the processing plant. The crushed oxide ore will be carried to the heap leach by a separate overland conveyor.

The concentrator consisting of the grinding mills, flotation circuits, reagent mixing, storage and distribution systems, concentrate filter facility and thickeners will be located on an area of relatively flat terrain about 2 km south of the pit and crusher.

Low-grade ore will be placed in several temporary stockpiles adjacent to the plant site and crusher for processing in later years. Figure 18-1 shows the overall project site layout.

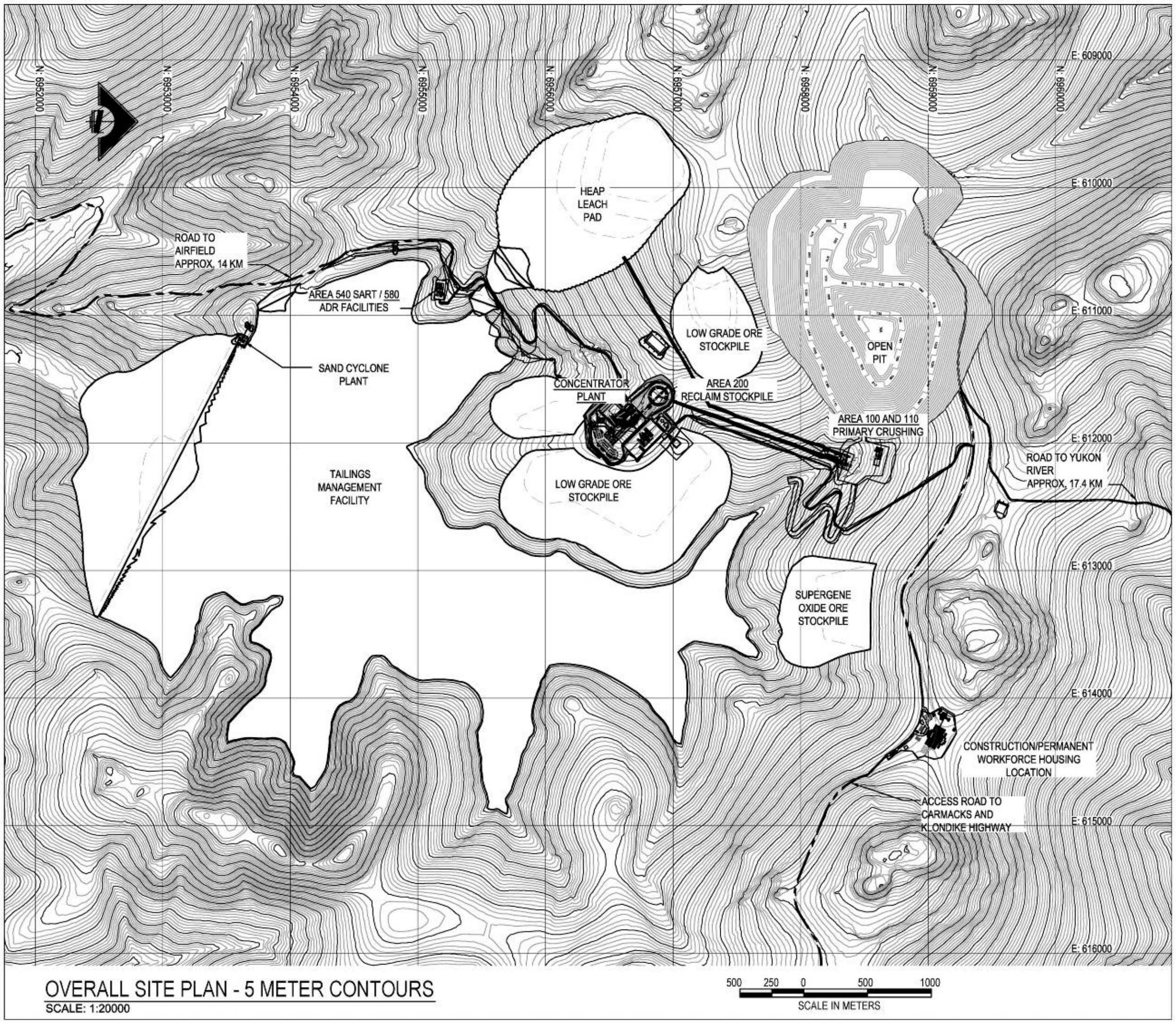


Figure 18-1: Overall Site Plan

18.1.2 Truck Shop

A truck shop and associated facilities will occupy about 2.4 ha adjacent to the open pit mine's eastern exit.

18.1.3 Residence Camp

A nominal 1,000-man capacity camp will be installed to support construction with additional camps for off-site construction provided by the contractor(s). The construction camp will be converted to serve as residence for the operations staff as construction activities wind down.

18.1.4 Operational Support Facilities

The majority of ancillary buildings are proposed to be located at the processing plant site. The Administration building will provide office space for both the construction effort and operations. The change house (mine dry) and laboratory buildings will be located near the mill and flotation buildings. A warehouse and laydown area will be provided for receiving and storage of parts and supplies, and for maintenance of plant mechanical and electrical equipment. A light vehicle maintenance building is proposed at the plant site apart from the truck shop. All of these buildings will be pre-engineered steel structures.

18.1.5 Guard Shed/Scale House

A Guard Shed/Scale House at the facility entrance will house a guard around the clock. The guard will also operate the truck scale. The guard shed will be of modular construction.

18.1.6 Airstrip

The Casino Mine site is remote and the required workforce is expected to be 1,000 during construction and 600 to 700 during operations. Access to the site for this number of personnel will be best served by aircraft. The project plan includes a new 2,000 m airstrip and pre-engineered building for employee transport to and from the site. The airstrip is planned to be located about 15 km southwest of the mine on a flat area downstream of the TMF embankment adjacent to Dip Creek.

Figure 18-2 shows the current Casino airstrip and the mill plant site.

Figure 18-2: Current Casino Airstrip

WCGC intends to replace the existing small airstrip with a larger facility that permits all-season operations. Two possible aircraft have been considered to fly to the site. These are the Hawker-Sidley HS 748 turbo-prop aircraft which can be configured with 40-58 seats and the Bombardier Dash 8-100 or 200 series turbo-prop aircraft which can be configured with 37-39 seats.

Given the size of the workforce, it is expected that flights will originate from Whitehorse connecting with scheduled flights from Vancouver or other major centres.

The Dip Creek Valley airstrip site poses few aeronautical challenges and provides safe aircraft operations during all visible weather conditions. The airstrip design criteria have been developed to conform to the Transport Canada Aerodrome Standards and Recommended Practices (TP 312). The airstrip consists of a Code 3C non-instrument runway generally oriented northeast to southwest. It is 2,000 m long with 60 m overrun beyond the thresholds at each end for a total length of 2,120 m. The required runway width is 30 m and the total graded width is 80 m. The airstrip embankment is raised a minimum of 1.5 m above the existing ground to provide a stable base and to prevent any ground ice from melting. There is a taxiway at the northeast end connecting to an apron for aircraft unloading and a parking area at the start of the airstrip access road. Figure 18-3 shows the location of the proposed new airstrip.

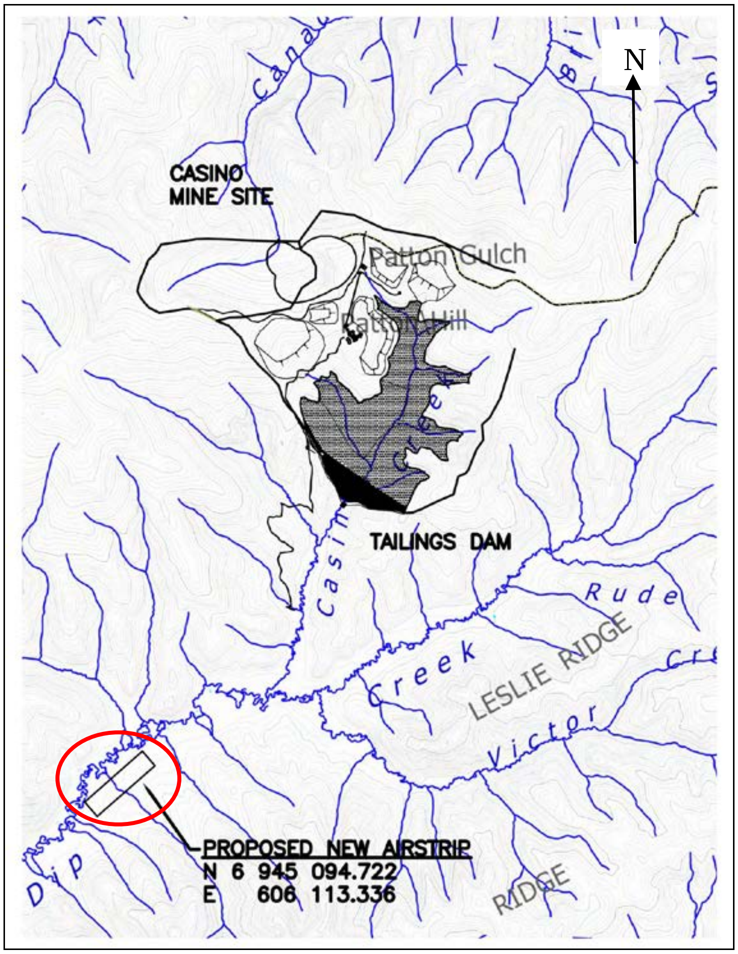


Figure 18-3: Proposed New Airstrip Location

18.1.7 Airstrip Access Road

The airstrip access road consists of approximately 14 km of single lane gravel road starting from the airstrip in the Dip Creek Valley and climbing to connect with the tailings dam access road at the mine site. The road design criteria satisfies the guidelines in the BC Ministry of Forests and Range Forest Road Engineering Guidebook (2nd Edition, 2002) for a 30 km/h design speed. Road construction methods are similar to those of the main access road with the road raised on a fill embankment in the wet valley bottoms and built with both cut and fill as it climbs the slopes out of the valleys. The lower design speed and narrower cross-section allow for more flexibility in the alignment and profile. The road follows more closely to the natural topography.

18.2 BYPASS AND ACCESS ROADS

18.2.1 Service Roads

Service roads will connect the various mine and process facilities together with the ancillaries. The roads will be constructed with a minimum 4 m wide all-weather gravel surface. In general, the maximum grade for the road will be 10%.

An existing road leads northward from the mine facility to the Yukon River along Britannia Creek. The new fresh water pipeline will roughly follow this road. The roadway will be graded and improved as a service road to facilitate construction, access, inspection, and maintenance of the pipeline.

18.2.2 Access Road

A winter access road from Kluane Lake near the Alaska Highway has been used in the past to service the Casino Mine. Historically, access to the area has been available via the Casino Trail, and WCGC has used this access in recent years.

Access to the site is also available from a barge landing at the junction of Britannia Creek and the Yukon River was prepared in 2010 and the lower 10 km of the 23 km road to the site realigned to avoid all but one creek crossing at Canadian Creek. That creek crossing was replaced by a 15 m bridge in 2011.

Associated Engineering examined various route options for a year-round access road to Casino. The selected option which appears to have the least environmental impact and the greatest stakeholder support is a new road from the end of the existing Freegold Road approximately 70 km northwest of the village of Carmacks. The Casino Mine access road is a 132 km, two-lane, gravel resource road designed to accommodate B-Train Double (BTD) and Tridem trucks. The road design criteria satisfies the guidelines in the BC Ministry of Forests and Range Forest Road Engineering Guidebook (2nd Edition, 2002) for a 70 km/h design speed with some 50 km/h sections where road geometry is limited by the terrain.

In order to maximize the design speed and avoid unstable terrain, the route is located as much as possible in valley bottoms. The road surface elevation is designed to be 2.0 m above existing ground in these areas, with the fill material placed over undisturbed soils. This embankment height stabilises the road against washouts and protects against permafrost degradation under the road.

Where the road climbs out of the valley bottoms, the road construction method includes both cut and fill. Permafrost rich areas may require buttressing of cut slopes with a layer of angular rock fill on top of filter fabric. This will prevent permafrost degradation and act as a retaining structure to improve slope stability.

Most of the fill required for road construction will be developed from borrow pits located along the road alignment and then hauled to where it is needed. The section of road from the Selwyn River to the mine is located in soil that is mainly suitable for road embankment construction and can be utilized for fill. Further soil testing may reveal other locations with borrow suitable for road construction which will result in shorter haul distances and reduced road construction costs.

There are 18 major bridge crossings located along the main access road. These include crossings of Bow Creek, Big Creek, Hayes Creek, and Selwyn River along with several tributaries and side channels. Associated Engineering completed Hydro-technical analysis and topographic site surveys to provide the necessary information to then complete conceptual bridge designs for each crossing. There are also 71 major culvert crossings that have been identified for the main access road with estimated diameters ranging from 1,500 mm to 2,400 mm. Detailed hydro-technical analysis has not been completed for the culvert crossings. Culverts in fish bearing streams will be embedded to provide a simulated natural stream bottom.

Smaller 500 mm and 600 mm culverts will be used to control road drainage with the culvert spacing determined by road gradient and natural depressions in the topography. Figure 18-4 shows the proposed mine access road route.

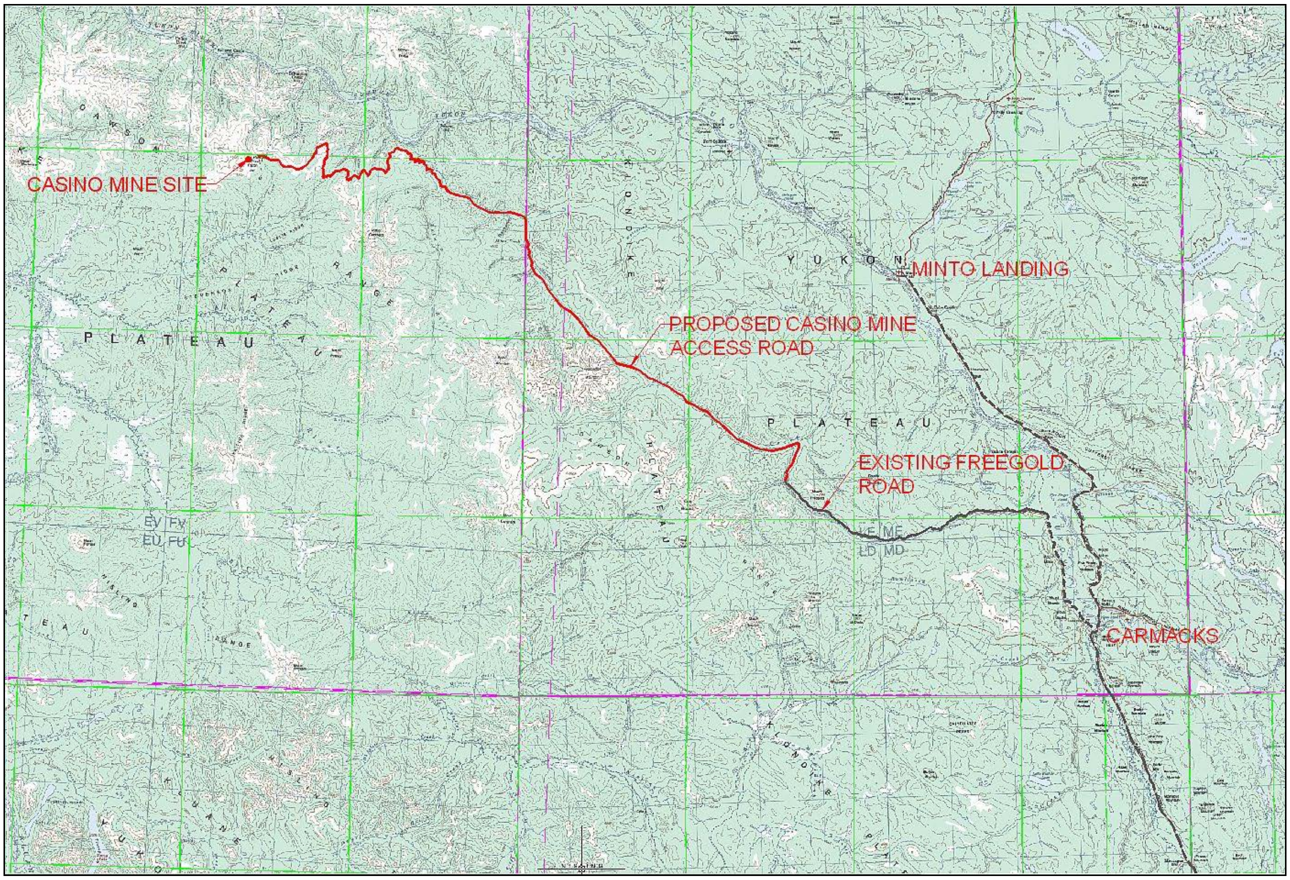


Figure 18-4: Proposed New Access Road

18.3 PORT FACILITIES

The Port of Skagway is located 560 km from the Casino site and has been selected as the port of export for the project. The port has historically exported up to 600,000 tonnes annually of lead and zinc concentrates and currently exports the copper concentrates from Capstone Mining, Minto operation. An engineering evaluation has determined that the existing concentrate storage and handling facilities at Skagway can be economically up-graded to serve the Casino export requirements. The Port of Skagway and AIDEA have expressed interest in providing concentrate storage and load-out services to Casino in their to be expanded facilities, consistent with the conceptual design prepared by WC&G. The feasibility study operating cost reflects exporting of copper concentrate through the Port of Skagway facilities. Figure 18-5 shows a conceptual rendering of the proposed port facilities expansion at Skagway.

Figure 18-5: Conceptual Rendering of Skagway Port Facilities

18.4 PROCESS BUILDINGS

18.4.1 Crushing Plant

There will be two primary crushers. One crusher will do the initial crushing of the sulphide ore, and the other will do the initial crushing of the oxide ore. Each primary crusher will be housed within 38 m and 34.5 high concrete structures, respectively, placed only a few meters below existing grade in order to minimize blasting of bedrock. These structures will be surrounded by “U”-shaped Mechanically Stabilized Earth (Hilfiker or equivalent) retaining walls to form the truck maneuvering area around the dump hopper, roughly at the elevation of the exit from the pit.

18.4.2 Gold Recovery & SART Building

The oxide crushing facility and overland conveyor will feed the heap leach pad, and the pregnant solution from there will feed the ADR/SART Gold Recovery facility. The ADR (Adsorption, Desorption and Recovery) and SART (Sulphidization, Acidification, Recycling, and Thickening) facilities will be located in a single structure.

18.4.3 Grinding

The Grinding Building (Mill) will house the 40 ft. diameter SAG mill and two 28 ft. diameter ball mills. The SAG mill and ball mill areas will each have a 90-tonne overhead bridge crane for maintenance.

18.4.4 Flotation/Reagent Storage & Mixing

The flotation circuits will be located within a structurally independent building adjacent and connected to the grinding building. Reagent storage and mixing facilities will be fully enclosed and placed adjacent to the flotation building. The 8,000 t lime silo will be located apart from the Mill/flotation building.

18.5 TAILINGS MANAGEMENT FACILITY

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 approximately 956 million tonnes of tailings (including cyclone sand tailings used as embankment fill) and co-disposal of up to 658 million tonnes of potentially reactive waste rock and overburden materials.

The embankments will be developed in stages throughout the life of the project using a combination of suitable non-reactive overburden, cyclone sand and waste rock materials from the plant site, open pit and/or local borrow sources. The Main Embankment will be expanded in stages across the Casino Creek valley.

A preliminary water balance model was developed for the TMF to assess the various flows to and from the facility, to predict the variation in supernatant pond volume during operations and to estimate make up water requirements. The water balance indicates that the operation requires a make-up water supply system to supplement natural run-off captured by the TMF to meet the project water demand.

Seepage water losses from the TMF will be collected in seepage collection systems constructed downstream of the embankments. The seepage will be pumped back into the TMF.

Details of the site characteristics, geotechnical and water management considerations for the tailings facility design, tailings distribution systems, seepage collection and reclamation and closure are contained in the Knight Piésold “Report on Feasibility Design of the Tailings Management Facility” (Ref. No. VA101-325/8-10).

18.6 WASTE ROCK STORAGE AND STOCKPILES

A total of approximately 658 million tonnes of waste rock and overburden will be mined from the open pit. The results of an updated waste characterization study indicate that almost all of the waste material is PAG and ML. The current plan is to store this potentially reactive material subaqueously within the Tailings Management Facility (TMF).Approximately 157.5 million tonnes of gold ore will be processed at the Heap Leach Facility (HLF). This gold ore will be stored in a temporarily stockpile near to the crusher and will report to the heap leach facility over a period of about 18 years, starting in the pre-production years (3 years prior to mill start-up) and continuing up to Year 15 of mill operations. HLF operations will commence during pre-production stripping of the open pit.

During mine operations approximately 144 million tonnes of Low Grade ore and 32 million tonnes of Supergene Oxide (SOX) ore will be stored in temporary stockpiles. The SOX ore will be stockpiled during the pre-production years and in Year 1 of mill operations, and report to the mill during Years 4 to 12 together with direct feed mill ore. Low Grade ore will be stockpiled up to Year 17 and milled during the last four years of mine operations (Years 19 to 22).

It is anticipated that approximately 2 to 3 million tonnes of Non-Acid Generating (NAG) waste rock (leach cap material) will be produced during pre-production mining. This material will be used in the construction of the TMF Starter (Stage 1) embankment. Any NAG waste rock that is either unsuitable for use as a construction material or in excess of construction material requirements will be placed in the Waste Storage Area or an adjacent surface dump.

Details of the site characteristics, geotechnical conditions, Waste Storage Area and stockpiles design, stability assessment, water management, and reclamation and closure are contained in the Knight Piésold report “Waste Storage Area and Stockpiles Feasibility Design” (Ref. No. VA101-325/8-12).

18.7 LEACH PAD

Information provided by site investigations and design studies indicate that the construction and operation of a Heap Leach Facility (“HLF”) is possible at this site. However, the cold climate and presence of extensive permafrost requires special design and operational considerations.

The proposed facility is located on a southeast facing hill-slope approximately one kilometre south of the deposit area. HLF operations will commence during pre-production stripping of the open pit. The design was carried out for an ore tonnage of 157.5 million tonnes. The heap leach pad will be stacked with ore and leached from Year -3 through Year 15 of mine operations. The ore will be leached at a nominal rate of 9.125 million tonnes per year. The following is a summary of the 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 southeast of the plant site area.
- Excavation of the pad foundation down to competent bedrock in areas with permafrost to eliminate potential settlement and instability resulting from thawing of 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 has been estimated as approximately two metres deep.

- An events pond for temporary storage of storm runoff and pregnant solution overflow during shut down will be constructed at the foot of the HLF confining embankment. The events pond will be included as part of the pre-production schedule.

- A composite liner system comprising a LLDPE liner, compacted soil liner and leak 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 LLDPE liners, a compacted soil liner and a geotextile layer will also be constructed in the lower portion of the leach pad (potential ponding area) and events pond and will include a leak detection and recovery system for intercepting and collecting any leakage through the inner liner.

- Borrow materials for the confining embankment, events pond and soil liner construction may utilize suitable overburden (residual and colluvial soils) and weathered bedrock along well drained, non-frozen, south-facing slopes east or west of the HLF. Suitable non-reactive mine waste rock may also be used for HLF construction.

- The heap leach pad will be developed in five 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 eight metre lifts constructed at repose bench face angles of approximately 1.4H:1V. Bench widths approximately nine metres wide will be left at the toe of each lift to establish a final overall slope angle of 2.5H:1V.

- Operations will involve the irrigation of a weak cyanide solution over the ore lift and the recovery of pregnant solution by means of solution collection pipes and pumps. The solution will be pumped to the gold extraction plant for metal extraction and recycled for re-use in the leaching process. The irrigation lines will be buried to prevent freezing during winter conditions.

The final heap leach pad required for an ore tonnage of 157.5 million tonnes will have a surface area of approximately 1,501,000 m^2. 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 1,067,000 m^3 of embankment and drainage fill for construction. The required storage capacity for the events pond is approximately 110,000 m^3. The total quantity of geosynthetic liner required for the heap leach pad and confining embankment is approximately 1,587,000 m^2. The total quantity of geosynthetic liner for the events pond is approximately 50,300 m^2.

18.8 POWER GENERATION AND DISTRIBUTION

18.8.1 Power Generation

Electrical power generation for the project will be developed in two phases. An initial power plant designated the Supplementary Power Plant will be constructed in the vicinity of the main workforce housing complex to provide power to the camp, for construction activities, and to oxide crushing, conveying and heap leach facilities that go into operation before main power plant is operational.

The Supplementary Power Plant will consist of three internal combustion engines (ICE), dual fuel driven generators (capable of using both LNG and diesel) with a combined power output of 20.1 MW.

A Main Power Plant will be constructed at the Casino main mill and concentrator complex to supply the electrical energy required for operations throughout the mine site. The Supplementary Power Plant will supplement, and to a limited extend provide backup for, the Main Power Plant if one of the gas turbines is out of commission. The primary electrical power generation will be provided by two gas turbine driven generators and a steam generator, operating in combined cycle mode (CCGT) to produce up to 125 MW. Two internal combustion engine (ICE) driven generators will provide another 18.6 MW of power for black start capability, emergency power, and to complement the gas turbine generation when required. The gas turbines will be fueled by natural gas (supplied as liquefied natural gas, or LNG). The internal gas combustion engines will have LNG fuel capability only.

18.8.2 LNG Receiving, Storage and Distribution Facilities

LNG will be transported to the site from Fort Nelson, British Columbia via tanker trucks and stored on-site in a large 10,000 m^3 site-fabricated storage tank to provide fuel for the power plant. An LNG receiving station is provided to unload the LNG tankers and transfer the LNG into the storage facility. A LNG vaporization facility is provided to convert the LNG into gas at a suitable supply pressure to operate the power generation equipment. In addition to providing fuel for the power plants, the LNG facility will provide fuel for the mine haulage fleet, and fuel for over-the-highway tractors hauling concentrates, lime, grinding media, and LNG from Fort Nelson. Distribution to the on-site vehicles will be by two portable fueling stations, and two mobile refuelers.

18.8.3 Power Distribution

The power system for the Casino project consists of two generating stations and the distribution system.

The main generating station will consist of a combined cycle plant with two 43 megawatt (MW) gas turbines (GT) and a 29 MW steam turbine (ST). There will also be two 9 MW natural gas powered reciprocating engine generators at the main power plant. These units will all generate power at 13.8 kilovolts (kV) which will be stepped up to 34.5 kV through four (4) 13.8 kV to 34.5 kV transformers for the distribution system.

The second generating station will be located at the main/construction camp site and will consist of three 6.7 MW natural gas/diesel powered reciprocating engine generators. These units will generate power at 13.8 kV and will connect to the 34.5 kV distribution system through one 13.8 kV to 34.5 kV transformer. This station will be the first installed and will provide power for the project construction.

The 34.5 kV distribution systems will radiate from a 34.5 kV switchgear line-up with feeders to the SAG mill, Ball Mill #1, Ball Mill #2, and feeders to the mill and flotation areas in cable tray using insulated copper conductors. Overhead line feeder circuits with aluminum conductor steel reinforced (ACSR) will be provided for the tailings reclaim water, fresh water from the Yukon River, crushing/conveying and SART/ADR, camp site and two feeders to the pit loop.

Electric power utilization voltages will be 4,160 volts for motors 300 horsepower (hp) and above, 575 volts for three-phase motors 250 hp and below. For lighting, small loads and building services 480/277 or 208/120 volts will be the utilization voltage.

18.9 WATER SUPPLY AND DISTRIBUTION

18.9.1 Oxide Ore Processing

Initial water requirements for the gold plant and oxide ore heap leach operations will be met by pumping water retained behind a temporary coffer dam located at or near an elevation of 825 m within the TMF catchment area along Casino Creek. The coffer dam will provide a source of fresh water for the process. Over time, water usage from the coffer dam will be replaced by reclaim water supplied from the TMF.

18.9.2 Sulphide Ore Processing

The main fresh water supply will be supplied from the Yukon River. The water will be collected in a riverbank caisson and radial well system (Ranney Well) and pumped through an above ground insulated 36 inch diameter by 17.4 km long pipeline with four booster stations. The design flow rate of the system is 3,400 m^3/h.

The fire water requirement is 341 m^3/h for two hours. This demand is satisfied by designing the storage pond with a fire reserve capacity of 682 m^3 in the lower portion of the pond that will be unavailable for other uses.

Potable water will be produced by filtering and chlorinating fresh water and will be stored and distributed separately.

Process water reclaimed from the tailings pond and from the plant will be collected in a 63,700 m^3 process water pond.

18.10 WASTEWATER DISPOSAL

Packaged sewage treatment plant systems will accept and treat all sanitary wastewater.

18.11 COMMUNICATIONS

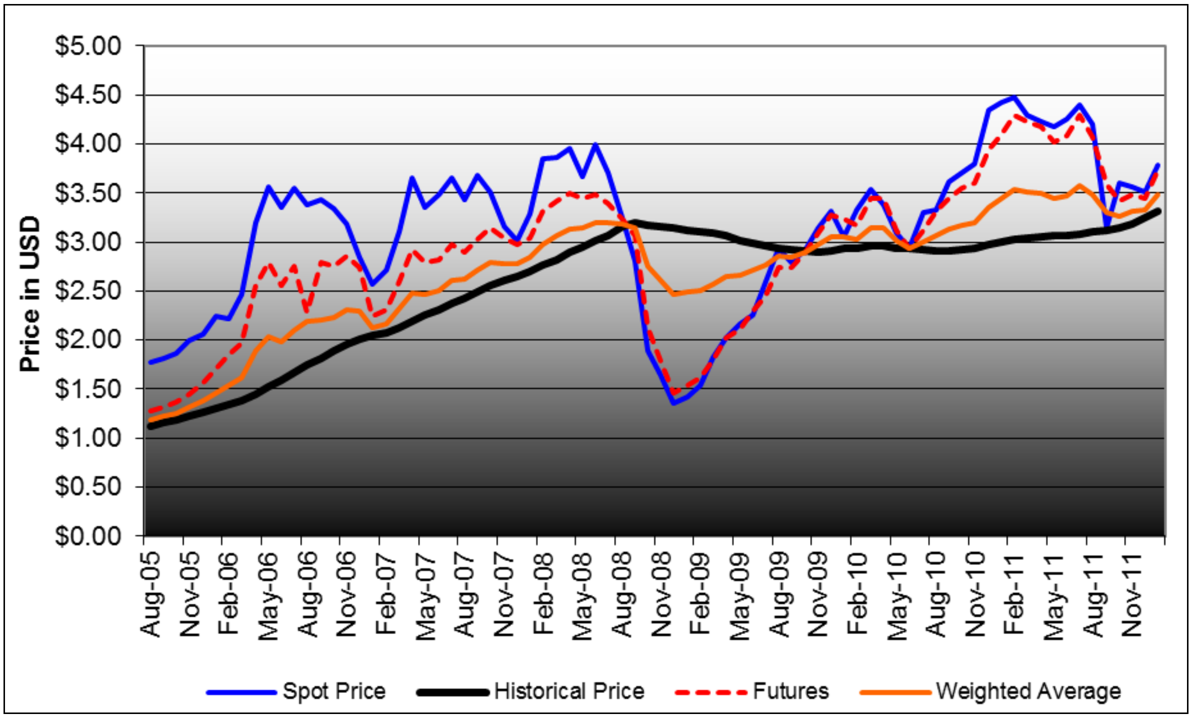
No communications infrastructure exists in the area of the site. The project will develop communications infrastructure to meet construction and operations requirements.

19 MARKET STUDIES AND CONTRACTS

No contracts are yet in place for the sale of concentrates to smelters/refineries. Transport and process costs have been included based on prevailing conditions at worldwide smelters.

Markets for copper and silver remain strong, with the market for molybdenum being more volatile. Figure 19-1 shows copper prices from August 2005 through January 2012.

The author reviews metal pricing and warehouse inventories for the various metals on a continual basis. Figure 19-2 demonstrates Kitco's 5-year LME copper warehouse stock levels.



Note: These prices are NOT adjusted for inflation. They are strictly a historical record.

Figure 19-1: Price of Copper

- Spot Price = Price that a commodity is actually selling for at a recorded instant.
- Historical Price = A 36-month average that is typically used for US SEC filings.
- Futures Price Forecast = A 24-month average indicative of where the price may be headed.
- Weighted Average = A price based on 36 months of historical data plus 24 months of futures.

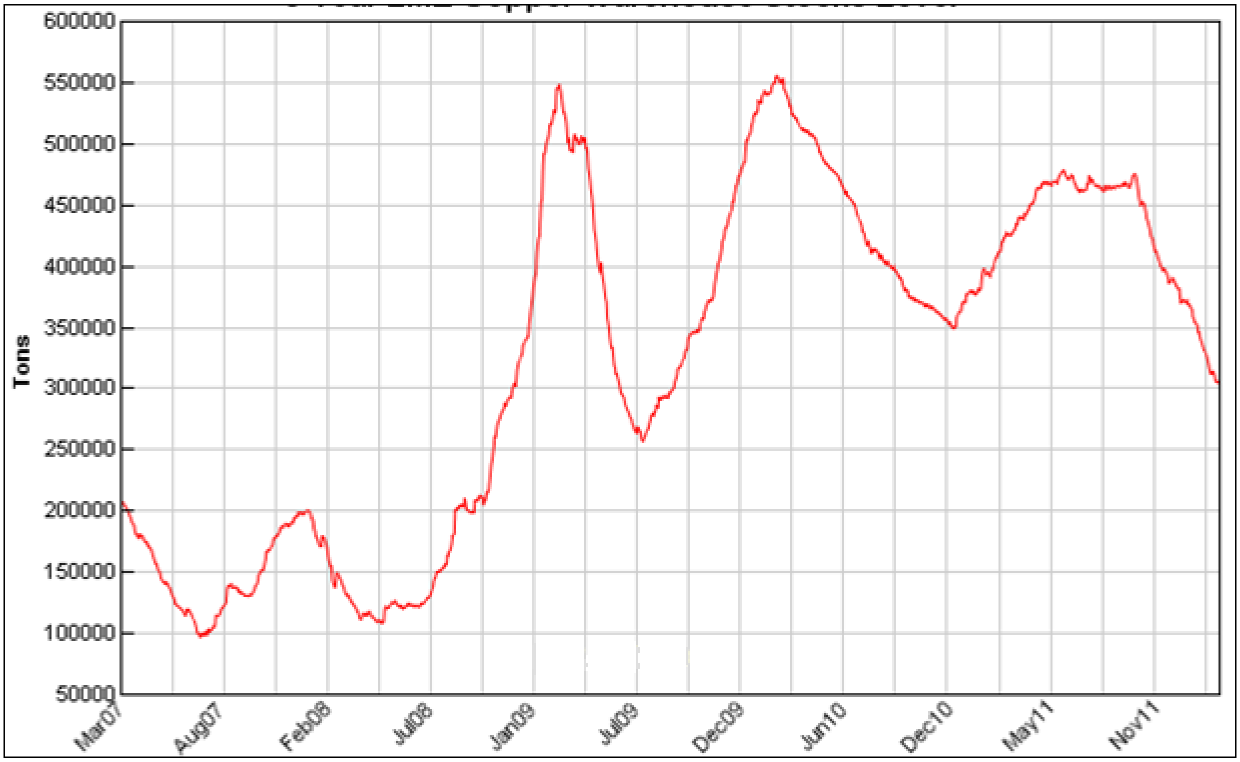


Figure 19-2: 5 Year LME Copper Warehouse Stock Level

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

WCGC, through their subsidiary Casino Mining Corporation ("CMC") has prepared a feasibility study of the Casino property located 300 km north of Whitehorse in the Dawson Range (see Figure 1-1). To support this study, this section provides further details on the Yukon Government's environmental/socio-economic assessment process and regulatory approvals required to bring this property into production.

20.2 SITE ENVIRONMENTAL CONDITIONS

20.2.1 Location and Terrain

The property in located in the Dawson Range, an area that 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 1,200 m ASL, which is deeply incised by the mature drainage of the Yukon River watershed. The property lies in the Klondike Plateau ecozone (172) within the Boreal Cordillera Ecoregion. Initial data collected by CMC show the mean annual temperature for the area is approximately -2.7°C with a summer mean of approximately 10°C and a winter mean of -16.3°C. Mean annual precipitation has been measured as 500 mm.

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 where the air strip will be located 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.

Widespread and discontinuous permafrost occurs throughout the area with cryoplanation terraces, peat plateaus, pingos, palsas, polygons and tussock fields common.

20.2.2 Surface and Groundwater Quality and Hydrology

The property is within the Yukon River drainage basin. The Yukon River is the longest river in Yukon and Alaska and the third longest river in North America. It flows northwest from the Coastal Range Mountains of northern British Columbia, through the Yukon Territory and Alaska to the Bering Sea. The watershed's total drainage area is approximately 840,000 km^2 (323,800 km^2 in Canada).

Surface and groundwater in the Casino property area has elevated metal concentrations due to naturally occurring acid rock drainage likely caused by natural oxidation of the large mineral deposit. Groundwater quality data indicates elevated concentrations of dissolved metals including aluminum, arsenic, cadmium, cobalt, copper, iron, uranium and zinc, as well as elevated concentrations of fluoride and total dissolved solids (TDS) and low pH. Similar to groundwater quality, baseline surface water quality in Casino Creek is affected by a small alpine stream, Proctor Gulch, which is acidic and contains elevated levels of metals. This stream drains into upper Casino Creek, causing elevated levels of total aluminum, cadmium, chromium, copper, iron, and zinc. Other surface water quality parameters that are affected include acidic pH levels and high sulphate concentrations. The majority of the creeks in the area observe peak discharges during the month of May, and daily fluctuations in discharge due to thawing of the permafrost active layer and rainfall events from June to September.

20.3 POLITICAL ANALYSIS

20.3.1 Yukon Political Environment

Yukon is presently enjoying a strong resurgence in mining and exploration, which has generally received good local press and public support. Yukon completely escaped the 2008-2009 economic downturn experienced by the rest of Canada and the world. This is fuelled in a large part by exploration success in the Dawson Range copper-gold belt (where Casino is located) and the move to commercial production by three Yukon mining operations; Alexco's Bellekeno silver mine, Yukon Zinc's polymetallic Wolverine mine and Sherwood Copper's Minto mine. In addition, two other proposed mines are well advanced in the permitting process – the Cantung tungsten project owned by North American Tungsten and Victoria Gold's Eagle project.

Over the last 10 years, the Yukon has experienced a period of political stability as the majority conservative Yukon Party government carries out its mandate with the continued support of a pro-industry (Conservative) federal government in Ottawa. Yukon Party government has been in power since 2002, and is currently in the second year of a third term (five years) winning a majority of seats in the Territorial Legislative Assembly. Land and resources administration and control have been transferred to the territorial government where most of the assessment and permitting is conducted).

The federal conservative government has been announcing a number of significant policies and programs designed to improve the investment climate in Canada's north as part of their "Northern Strategy." (See: http://www.northernstrategy.gc.ca/cns/cns-eng.asp) This includes actions towards further improvements to the regulatory system, the creation of a "*Northern Major Projects Office*" to coordinate the participation of federal departments in project reviews, and changes to resource revenue sharing agreements between the federal and Yukon governments, so that more of the tax benefits from mining stay in the Yukon.

The new revenue sharing agreement with the Yukon Government recently announced will see a larger share of the royalties from Yukon mining projects flow to Yukon First Nations that have settled land claims. (Casino is in Selkirk Traditional Territory. The agreement applies to Selkirk First Nation and Casino as the area is subject to settled claims.)

The overall economic outlook for Yukon is very good and linked to success in the mineral sector, which is strongly supported by the current conservative Yukon Party government. Our current assessment of the permitting timeline for Casino would place final government decisions on the project during the term of the current government. Given the strong and growing dependence on the mining industry for Yukon's current economic prosperity, any new government would likely continue to offer an attractive investment climate for mining development.

20.4 YUKON GOVERNMENT APPROACH TO MINE DEVELOPMENT

Yukon is in a period of sustained growth driven by the mining sector. The Conference Board of Canada's Territorial Outlook, winter, 2012 recently reported that Real GDP in Yukon will increase by 3.7 percent in 2012 and the pace of growth is forecast to accelerate in both 2013 and 2014 (See *Conference Board of Canada Centre for the North:* http://www.centreforthenorth.ca/blogs/latestnews/miningpotentialgivestheterritoriesbrighteconomicpr). They project that over the next decade, several new mines will come into production. They go on to say between 2013 and 2020, mining output in Yukon will grow by an average compound rate of 10.7 per cent per year. Casino was not considered in the forecast as this project was still at the pre-feasibility stage. Casino will result in much higher impact on GDP and significant upward revision of the economic outlook for the territory as it moves towards production.

The Casino project will have far-reaching effects on the Yukon, and will move the territory to a new level of economic prosperity that will transform the Yukon Government's ability to provide services and fund programs in support of a growing and dynamic population. A report completed for CMC in 2012 by Myers, Norris Penny describes the economic impacts of the Casino project on government revenues based on the Pre-Feasibility Study Update released in May 2011 (*Economic Impacts of the Casino Mine Project*, Myers, Norris Penny LLP, January 2012. Unpublished internal report prepared for Casino Mining Corporation.). Total direct, indirect and induced economic impacts over the life of mine for the project include output of $16.4 billion; GDP of $9.8 billion; wages and salaries of $2.8 billion; federal tax of $950 million; Yukon tax of $1.8 billion; and additional taxes of $67 million. While this report has not been updated to reflect this feasibility study, the analysis provides an important benchmark.

20.4.1 Selkirk First Nation APPROACH to Mine Development

The Selkirk Traditional Territory, which includes the Casino property in the Dawson Range, encompasses the project area in the central portion of the Yukon. The Selkirk First Nation concluded a land claims agreement with the federal government in 1997. This agreement was negotiated under the Umbrella Final Agreement (UFA) that was used as the framework or template for individual agreements for all Yukon land claims that have been settled to date. The land claims agreement includes provisions related to ownership of lands by Selkirk ("Settlement Land") and management of these lands and resources. Discussions with Selkirk First Nation have been ongoing since 2008. Negotiations towards a formal agreement between the company and the First Nation are underway.

20.5 ENVIRONMENTAL ASSESSMENT PROCESS

As of November 2005, the Yukon Socioeconomic Assessment Board (YESAB) must assess most projects in Yukon for environmental and socio-economic effects under the *Yukon Environmental and Socioeconomic Assessment Act* (YESAA).

20.5.1 Role of the Yukon Environmental Assessment Board

YESAB is an independent Board established under YESAA to administer the assessment process under YESAA. YESAA establishes a number of obligations and responsibilities of the Board with respect to the conduct of reviews, including:

- Determining the adequacy of the proposal;
- Seeking information and views from agencies and First Nations that have expressed an interest;
- Determining the scope of the project;
- Consulting with First Nations;
- Notifying other agencies and governments that are decision bodies as defined by YESAA;
- Establishing rules for the conduct of evaluations;
- Determining whether or not a project will have an adverse socio-economic or environmental impact;
- Recommending to decision bodies that a project proceed subject to specific terms and conditions if negative effects can be mitigated by those terms and conditions; and
- Establishing a Panel of the Board for further review under certain circumstances.

For more information see the official website of the Yukon Socioeconomic Assessment Board: http://www.yesab.ca/index.html.

A project is assessed by YESAB at one of three levels. Smaller projects are *evaluated* at one of YESAB's six designated offices (located in six communities around the Yukon). The three-member Executive Committee of YESAB *screens* projects of greater magnitude, with presumably greater potential for significant effects. The Executive Committee may make a recommendation for the project to receive a further level of assessment: *a panel review*.

Development of Casino into a fully operational mine triggers an environmental assessment under YESAA as all activities related to the construction, operation, modification or closure of a mine are listed as assessable activities in *Schedule 1 of the Activities Regulations*. Furthermore, the level of assessment for the Casino project would be at the Executive Committee screening level as several of the activities meet or exceed the activity thresholds listed in *Schedule 3 of the Activities Regulations*.

Work on the environmental baseline data package required to support a formal application for assessment and licenses in nearing completion. The baseline programs were initiated in 2008 and will continue through 2013 and provide detailed and comprehensive information about the project.

Information requirements from government agencies and the scope of the review are well understood. This has been done through a series of early meetings with review agencies. All components of the baseline program have been subject to early review and discussion in meetings with Yukon Government departmental staff, Water Board staff, First Nations Government representatives, Environment Canada and the Department of Fisheries and Oceans.

There have also been a number of formal sessions with staff and Executive Committee members of the Yukon Environmental and Socioeconomic Assessment Board (YESAB). These discussions are continuing and are helping to build an understanding of the proposed project by review agencies in advance of a complex submission.

For assessment under YESAA, the scope of the Casino project is expected to include the proposed Freegold Road Extension and may include necessary upgrading of the existing Freegold Road as this would be the primary means of access to the main highway system during construction and operation of the mine. Baseline data has also been collected to support the assessment of the access road.

Timelines for recent major mining projects in Yukon suggest that the YESAB screening process should be completed within 2 years. One of the critical factors that influences the time required to complete the screening is whether or not the information provided in the project proposal is deemed to be adequate for the purposes of completing the assessment. This is why the early meetings with review agencies are taking place to appropriately define the scope of the project and information requirements at an early stage so that baseline studies and other information gathering is relevant and in sufficient detail for the assessment.

20.5.2 Consideration of Recommendations and Issuance of Decision Documents

At the conclusion of the screening, the Executive Committee issues its recommendations to a Decision Body for consideration. For all Executive Committee screenings, the designated Decision Body for Yukon Government is the Development Assessment Branch within the Government of Yukon Executive Council Office. (The Development Assessment Branch is also responsible for coordinating the Yukon Government's input to the review process). Recommendations issued by YESAB may be accepted, rejected or varied in the decision document, or a recommendation may be referred back to the Executive Committee for reconsideration. The Decision body has 60 days to issue a decision document, or 45 days in which to refer the recommendations back to the Executive Committee for reconsideration. Regulatory permitting, discussed in the following section, would follow on the heels of a positive decision document being issued.

20.6 LICENSING

Mining related projects in the Yukon are regulated through territorial and federal legislation by various agencies, including government departments and independent boards. The regulatory permitting and licensing processes are separate from the *Yukon Environmental and Socio-economic Assessment Act* (YESAA). There is no single-window application process for regulatory permitting. Separate applications and information packages are required for each authorization and agency.

The main pieces of legislation that will govern mining related activities for the Casino properties include:

- *Quartz Mining Act;*

- *Waters Act;*
- *Territorial Lands (Yukon) Act;*
- *Environment Act;*
- *Highways Act;*
- *Dangerous Goods Transportation Act;*
- *Fisheries Act;*
- *Yukon Occupational Health and Safety Act.*

Authorizations are processed and issued following completion of the YESAA screening and issuance of a decision document by the decision bodies. However, some agencies will conduct preliminary reviews of applications or begin drafting authorizations prior to the issuance of a positive YESAA decision document. Timeframes for authorizations vary from agency to agency.

A brief overview of the legal framework for issuing the two most important licenses follows.

20.6.1 Quartz Mining Act

The purpose of the *Quartz Mining Act* is to regulate hard rock exploration and mining-related activities in the Yukon. The Act includes a specific requirement for a proponent to secure a "*Quartz Mine License*" (Section 135) for mining projects. The *Quartz Mining Act* and associated Regulations are administered by the Yukon Government, Energy Mines and Resources, Mineral Resources Branch. The *Security Regulation* defines the form and amount of security acceptable to the Minister of EMR, as well as mechanisms to adjust the amount of security held during the course of the operation.

The Quartz Mine License is staged. It has been designed with sufficient flexibility to allow adjustments as may be required as the project evolves. Part 1 covers mine development, and provides a mechanism to approve initial construction activities that do not require a water license. Part 2 covers mine construction and operation, and covers specific mine operational plans and environmental protection plans.

20.6.2 Waters Act

The purpose of the *Waters Act* is to regulate the use of water and deposition of waste into water in the Yukon Territory. This territorial statute outlines provisions for the development of regulations for prescribing types of undertakings and licensing criteria, application information requirements, and financial security. This act also establishes the Yukon Water Board. The *Waters Act* defines provisions for public hearings, compensation to affected parties, and allows the Board to make rules related to administrative procedures. Under the Act, quartz mining activities are regulated and classified according to water use and waste disposal criteria.

A producing mine at the Casino properties will require a Type "A" Water Use License issued by the Yukon Water Board. This license will include conditions related to water use and waste disposal, water control and diversion structures, the submission of studies and plans, monitoring and surveillance, and the modification and construction of water-related structures.

20.7 PROJECT ASSESSMENT AND PERMITTING TIMELINE

Baseline environmental work was initiated in 2008 and will be complete in 2013. This includes wildlife, fisheries and terrestrial studies, archaeological and heritage resource investigations, socioeconomic studies, First Nation and community consultation, geochemistry, hydrology, hydrogeology, air quality and weather data collection. Geotechnical work around the tailings management facility will be completed in 2013. This work will form the basis of a project proposal to be submitted for YESAB screening and the subsequent authorizations which are required. Additional work underway includes negotiations with the Department of Fisheries for a habitat compensation agreement, closure planning and water management modeling.

Preliminary engineering and baseline environmental work for the 132 km access road that was initiated in 2010 is also nearly complete. The access road will be part of the scope of the YESAB review. This work includes wildlife and terrestrial surveys, archaeological investigations and continued fisheries and hydrology data collection at key watercourse crossings in support of engineering to refine the route alignment.

20.8 BONDING, RECLAMATION AND CLOSURE PLANS

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 detailed plan is expected to be required as a condition for 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. Both the *Yukon Waters Act* and the *Yukon Quartz Mining Act* have provisions for security to be held by government.

21 CAPITAL AND OPERATING COSTS

21.1 BASIS OF CAPITAL COST ESTIMATE

In general, M3 based this capital cost estimate on its knowledge and experience of similar types of facilities and work in similar locations. To assist in the estimating, M3 used quantity estimates, and in some cases, costs supplied by specialist sub-consultants, such as the following:

- Associated Engineering (AE): Main access from Carmacks to site, the new airfield, and the road to the airfield.
- Knight Piésold (KP): Geotechnical quantities associated with the Heap Leach Facility, Waste Rock Storage Area, Water Supply and the Tailing Management Facility.
- Independent Mining Consultants (IMC): Mine capital and operating costs.

"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 from end of 3rd quarter 2012 Canadian dollars except as noted otherwise. The values in this report assume an exchange rate of 1:1 between U.S. and Canadian dollars.

Based on the level of engineering completed and definition of scope, M3 estimated the contingency at 10% of the direct and indirect costs (Contracted Cost). The contingency was estimated on an area and cost category basis. The costs provided by others also contain a suitable contingency.

21.1.1 Assumptions

The capital costs are based on this project being executed by experienced EPCM contractor(s) in the hard rock mining industry with a recent record of bringing projects on budget or under budget. In addition, it is assumed that all contracts and subcontracts are based on a lump-sum basis or a competitively bid unit cost basis, such as per cubic yard of concrete placed. In particular, no time and material contracts are anticipated nor should they be allowed in order to ensure this budget is best maintained. In addition, it is assumed that at least two sufficiently sized self-performing local contractors are in place for all trades, such as civil, concrete, steel, architectural, mechanical, electrical, instrumentation and controls, and process piping. Certain contractors will have multiple trade capabilities.

WCGC will order major material supplies (i.e., structural and mechanical steelwork) as well as bulk orders (i.e., piping and electrical). These will be issued to construction contractors on site using strict inventory control.

Budgetary quotes were obtained for the mining equipment, crusher, mills, cyclones, and other large equipment. Negotiation of budget quotes in final purchasing may result in better capital costs.

All equipment is priced as new. Opportunity may exist at the time of project execution to obtain suitable used equipment for capital cost savings. All costs to date by Owner are considered as sunk costs.

Any costs incurred for the scoping study, the pre-feasibility study and the completion of a feasibility study (including field drilling and lab testing) are not included.

21.1.2 Exclusions and Qualifications

This capital cost estimate excludes the following items.

1. Future escalation from the end of 4th quarter of 2012 is excluded.
2. The cost of all prior studies; including this feasibility study, are excluded.
3. Start-up and initial operating expenses subsequent to Owner’s acceptance of the plant ready to accept feed are excluded.
4. Future foreign currency exchange variation from the estimate is excluded.
5. The cost to provide insurance coverage for the duration of the project is excluded.
6. Environmental and ecological considerations, other than those already incorporated in the estimate basis, are excluded.
7. Any future changes in regulations, codes and standards are excluded.
8. Geotechnical conditions inconsistent with the current project geotechnical investigation results and upon which the estimate is based are excluded.
9. Any disposal of toxic or contaminated materials encountered during the execution of the project that arise from pre-existing conditions are excluded.
10. Financial charges and interest during the course of construction.
11. Salvage value of all residual construction materials, vehicles, camps, and temporary buildings not specifically identified in the estimate are excluded.
12. Training of operations personnel and the cost to prepare training manuals.

This capital cost estimate depends on the following qualifications:

1. Environmental permits and licenses (such as the permit for Yukon River water) required to operate the facilities are obtained in a timely manner.
2. Construction permits are available to support the project schedule.
3. Unfettered access to the project site is assured for the duration of the project development and operation.

21.1.3 Estimate Accuracy

The accuracy of this estimate for those items identified in the scope-of-work is estimated to be well within the range of plus 15% to minus 10%.

Total mine and plant mechanical equipment cost is estimated at $629,826,996. Most of that came directly from escalated 2012 vendor quotes. Much of the electrical equipment estimates were derived from recent quotes as well.

Outside consultants provided estimates for the main access road, and airfield. These estimates are stated to be accurate to ±25%.

21.1.4 Contingency

Based on the level of engineering completed and definition of scope, M3 calculates its contingency at 10% of the direct and indirect costs (Contracted Cost). Contingency is intended to cover unallocated costs from lack of detailing in scope items. It is a compilation of aggregate risk from all estimated cost areas. The Capital Cost Estimate spreadsheet contains a contingency calculation matrix.

21.1.5 Documents

Documents available to the estimators include the following:

a)	Design Criteria	(Yes)
b)	Equipment List	(Yes)
c)	Equipment Specifications	(Duty Only)
d)	Construction Specifications	(Partial)
e)	Flowsheets	(Yes)
f)	P&ID's (Preliminary)	(Yes)
g)	General Arrangements	(Yes)
h)	Architectural Drawings	(Yes)
i)	Civil Drawings	(Partial)
j)	Concrete Drawings	(No)
k)	Structural Steel Drawings	(No)
l)	Mechanical Drawings	(No)
m)	Main Electrical One-Lines	(Yes)
n)	Electrical Schematics	(No)
o)	Electrical Physicals	(No)
p)	Instrumentation Schematics	(No)
q)	Instrument Log	(No)
r)	Piping Drawings/Line List	(Partial)
s)	Piping/Valve Takeoff	(Yes)
t)	Valve List	(No)
u)	Cable and Conduit Schedule	(No)
v)	Electrical Takeoffs	(Yes)
w)	Structural Takeoffs	(Yes)
x)	Pump Calculations	(Preliminary)
y)	Reclamation Program	(Yes)

21.1.6 Construction Labour

21.1.6.1 Labour Rate

In 2007 M3 canvassed construction contractors in the Yukon and elsewhere in Northern Canada for wage rates for the various construction trades. Based on this survey, Table 21-1 presents a breakdown of the average hourly pay rate for construction trades that is used in this study. The hourly rate includes overtime based on a 70-hour week. Overtime is applied by the contractor against a base wage plus certain other fees and taxes. Adjustment was made current for 2012 rates.

Table 21-1: Average Hourly Pay for Construction Trades

Job	Hourly Rate
Labourer	$54.44
Carpenter	$74.82
Electrical	$74.82
Structural	$74.82
Operator	$74.82
Mechanical	$74.82
Concrete Finisher	$74.82
Piping	$74.82
Foreman	$80.94
General Foreman	$87.06

21.1.6.2 Camp Cost

The camp rate is $85.00 per day based on a prior study for anther Yukon project which determined a camp rate of $75.00 per day. The Casino rate was increased to $85.00 to account for higher expenses to bring camp labour in from population centers. This rate is consistent with information collected from a canvas of large construction projects in Northern Canada.

21.1.6.3 Transportation

The study assumes construction labour will be drawn from population centers in the Whitehorse area. Additional labour will be sourced from locations in Canada such as Vancouver, Edmonton, or Calgary. Where appropriate, employees will be flown into camp by air charter. The study assumes air charter rates will be competitive with typical airfares over similar distances by commercial air service. The reason for the assumption is that the guarantee of a full plane and a normal schedule will encourage competition among charter carriers. The estimate reflects this approach.

21.1.6.4 Productivity

The estimate assumes that US and Canadian productivity rates are comparable. In order to maintain productivity during winter months, work will be carried out inside heated buildings. During that time, care will be taken to minimize any work that must be done outside in inclement weather.

Project construction was scheduled to minimize outdoor winter work. Certain special design elements and expenses, such as the 90-tonne overhead cranes in the grinding areas, are included in project plans to allow construction to proceed indoors during the winter.

21.1.7 Direct Costs

21.1.7.1 Sitework

Civil earthwork quantities for the major civil structure (Heap Leach Facility, Waste Rock Storage Area and Tailings Management Facility) are based on preliminary designs by KP.

Site earthworks are generally classified as follows:

- Clearing & Grubbing
- Spread & compact
- Haul Material and/or equipment
- Excavation-rock by hand
- Site excavation soils
- Site backfill not compacted
- Site excavation rock
- Site backfill mechanical compacted
- Structural excavation
- Structural backfill
- Cut to fill
- Rock backfill hand compact
- Cut to waste
- Soil backfill hand compact
- Borrow to fill

Other civil works are:

- Crushing
- ABC (base course)
- Riprap dumped
- Polyvinyl & Polyethylene Liners
- Subgrade preparation

Specific labour, material and equipment costs apply to each category. Costs are based on standard worker productivity figures for construction trades derived from cost estimating handbooks and equipment productivity information obtained from manufacturers. M3 sometimes adjusts these factors according to M3 cost estimator experience and judgement.

In general, the estimate assumes overburden and suitable (non-acid-generating) mine waste will be used as the bulk fill material for the tailings and heap leach embankments and other large fills.

Therefore, mine haulage cost estimates cover excavation, transportation and dumping of material in major structural fills.

Tailings dam and heap leach embankment core and filter material are borrowed and compacted; shell material is only compacted as the tailings shell material is exclusively mine waste.

General site excavation, grading and backfill quantities for the process plant buildings and ancillary structures were calculated using Autodesk's Land Development program for AutoCAD applied to preliminary facility layouts by M3 and foundation depths specified by KP (i.e., "generally encountered at shallow depths of from 1 to 3 metres"). Site platforms were benched and sloped according to general arrangements for the facilities and to minimize earthmoving where possible.

Likewise, ponds were located and arranged to take advantage of existing land contours where possible and to minimize earthmoving.

Service roads are designed according to M3 standard road designs. Quantities were calculated by AutoCAD.

The study assumes the existing network of exploration roads will suffice for construction roads. The cost of a winter road is based on an estimate supplied by AE. AE also supplied the estimated costs for the access road to the plant area as described below. Haul road estimates were supplied by IMC.

21.1.7.2 Concrete

Concrete quantities were based on either parametric factors collated from constructed projects and current construction designs, or from direct material takeoffs from drawings. M3 applied parametrics to estimate concrete quantities for the crushing plant foundations, coarse ore (overland) conveyor footings, coarse ore stockpile building footings, reclaim tunnel and feeders, grinding and flotation sections, concentrate handling and tailings thickener walls and floor. M3 used takeoffs for simpler structures such as building foundations. Concrete costs were estimated based upon an informal survey of current and recent projects in the Yukon.

21.1.7.3 Structural Steel

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 take-offs from drawings of conceptual designs.

Steel costs were based upon a recent large steel purchase for a mine plant of similar scale. Competitive bids were collected from the US, Canada, and Mexico. M3 considers the resulting bid prices to be representative of world structural steel prices for the purposes of this study.

21.1.7.4 Architectural

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.

21.1.7.5 Electrical

Electrical equipment prices are recent historical prices from projects of similar scale with suitable escalation applied.

21.1.7.6 Piping

Piping in most areas is estimated as a percentage of the mechanical equipment cost added to an adjusted labour cost. Pipe quantities for the heap leach were developed by KP. M3 applied costs from historical data to KP's quantities.

21.1.7.7 Instrumentation

Instrumentation in most areas is estimated as a percentage of the mechanical equipment cost added to an adjusted labour cost. These allowances are based on M3 cost estimator experience and judgment. Allowances range from 1.5% to 5%.

21.1.7.8 Demolition

Demolition costs for any existing facilities have been included. These are presented as end of mine life costs and are not in initial capital.

21.1.7.9 Mine Operations

The estimated cost of the mining equipment and facilities was developed by IMC and is included in Section 16 of this report.

21.1.7.10 Heap Leach and Gold Recovery Plant

<u>Heap Leaching</u>

Leach heap earthwork is the highest cost element of the gold recovery plan. Quantities were estimated by KP for the heap embankment, events pond and water supply system. Unit costs were applied by M3 according to the methods described above in the discussion of civil and earthwork.

- Pipe and pump costs were estimated using M3 historical information.
- Mobile equipment costs were copied from the mine equipment quotes.

Tailings embankment construction requires placement of a coffer dam upstream. The coffer dam will be built during early construction stages so collected water can supply the heap leach operation. The cost of the coffer dam therefore falls within Area 675 – Heap Leach & Gold

Recovery Water System. The costs were estimated according to the method described above in the Civil and Earthworks discussion.

Sulphidization–Acidification–Recycling–Thickening (SART)

The sulphidizing column and the two thickeners were priced based on vendor quotes. The remaining equipment is tankage, agitators and other standard process equipment estimated based on M3's data library with appropriated scaling and escalation.

Civil quantities were based on AutoCAD calculations. Construction materials were based on material takeoffs.

Historical information was the basis for electrical equipment estimates.

Piping and instrumentation estimates were based on allowances set according to M3 cost estimator experience and judgment.

Adsorption Desorption Recovery (ADR)

Summit Valley Engineering, a major worldwide provider of such plants, supplied a vendor quote for the ADR plant installed equipment including piping. This number was supplemented with equipment quotes as the area was better defined. M3 estimated the cost of civil works based on AutoCAD calculations and an engineered building with foundation based upon the plant square footage. M3 applied an allowance for instrumentation.

21.1.7.11 Sulphide Flotation Plant

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 was based on M3's historical records, including budgetary and equipment purchase pricing from similar projects. Some historic records were scaled to correct for size or capacity difference.

All equipment is priced as new. Installation costs are based on allowances for materials and the cost estimator's judgment and experience for labour.

M3 requested updates for a number of vendor quotes for some commonly used equipment for which historic information was available.

Quantities for concrete, and structural steel were developed based on parametric factors collated from constructed projects or current construction designs, and applied on a unit area or unit volume basis. M3 applied parametrics to estimate costs for the crushing plant, coarse ore (overland) conveyor footings, coarse ore stockpile building, reclaim tunnel and feeders, grinding and flotation sections, concentrate handling and tailings thickener walls and floor. M3 engineers made certain adjustments to accommodate the support of the overland conveyor head by the cover building and the use of Grade 50 rather than A36 steel.

Ancillary building 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.

Process piping costs are factored from mechanical equipment costs, except for major pipe lines such as the tailings and reclaim lines and the fresh water supply line, including service piping.

Allowances for instrumentation within the plant have been factored as a percent of permanent equipment cost. This allowance is based on experience with similar installations and the judgement of the cost engineer.

Construction equipment costs were estimated according to the tasks performed and the crew hours involved. Construction equipment is included as a part of the direct cost. Owner will not supply any construction equipment (such as forklifts and crane for unloading or water trucks for dust suppression or dozers) to the project.

21.1.7.12 Power Generation

Costs for the gas turbine power generation plant were estimated on a cost per kWh basis. These costs were based on consultations with vendors and on budget quotes for equipment. The design criteria used in this study were provided by WCGC.

21.1.7.13 Main Access Road

AE selected the main access route and designed the 132 km gravel road which is an extension of the Freegold Road from the town of Carmacks to the site. AE's estimate included all construction costs, including the road construction crew support camp.

21.1.8 Indirect Costs

21.1.8.1 Freight

A freight allowance 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 not included in the capital cost but is accounted for in the financial model.

Sustaining Capital is not included in the initial 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 financial model. Sustaining costs are included in the financial model.

A construction camp with executive, supervisor, and labourer quarters to hold the anticipated maximum construction manpower of 980 people is included in the estimate. Additional camp housing for contractors engaged in developing the site prior to the construction camp being erected, is provided by those contractors and is included within the civil costs. A camp operating cost of $85 per day per person is also included.

Labour costs were estimated by applying hourly quantities to common construction metrics. M3 cost estimators use in-house data vendor information, published cost estimating guides and experience and judgment to determine the hourly quantities. Construction trade labour rates average approximately $71.83 per hour.

21.1.8.2 Temporary Facilities

The construction camp is included in the estimate and will serve also as the operations camp. The cost of the camp is based on a budgetary quote from The Britco Group, an established supplier of this type of facility in Northern Canada. Sanitary facility and water supply costs are separate from the overall cost of the camp.

Also separate from the camp cost is the supplemental power plant that will provide power to the camp, and to the construction site until the main power plant comes online. The supplemental plant will consist of three dual-fueled (LNG and diesel) generators. The same generator will later supply standby power for operations. The cost for the supplemental power plant was based on a cost per kWh basis developed from vendor budgetary quotes.

Water for construction and initial operation of the camp will be provided from a well and contractors will be responsible for their own connections. Contractors' temporary facilities, office, storage and warehousing are assumed to be covered in the contractors' overhead and profit, which is included in the labour rate.

21.1.8.3 Other Indirect Field Costs

Mobilization and demobilization is estimated to be 0.5% of direct field cost. Contractors' home office supervision is included in the contractors' fee and hence is included in the labour rate. Overall site safety and first aid is a function of the EPCM contractor and is covered in the EPCM costs.

21.1.8.4 Commissioning Costs

Costs are included for plant acceptance and initiation of operations as per the following:

- Mechanical completion – by Contractor
- Pre-Operational Testing – by Contractor
- Initial fills – procurement by Owner
- Start-up – by Owner with Contractor support
- Demonstration test if needed – by Owner

21.1.8.5 Taxes

The following taxes have been considered for this estimate:

- Sales tax is 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

21.1.8.6 Working Capital

Operating working capital will vary by year depending on sales revenue. Operating working capital is allowed at six months of sales revenue to provide cash to meet operating expenses prior to receipt of sales revenue. All the working capital is recaptured at the end of the mine life and the final value of the account is $0.

21.1.8.7 Construction Capital Start-up Spares

Construction capital start-up spares are estimated at $1 million.

21.1.8.8 Sustaining Capital

The major components of sustaining capital are expansion of leach pad, additions to the mine equipment fleet in later years, and tailings pond expansions including return water pump station relocation. Estimated costs are applied in future years in the economic model.

21.1.9 Owner's Costs

<u>Costs Included in Estimate</u>

Owner's costs prior to the start of project engineering and construction are deemed sunk and not included in this estimate. Owner's costs incurred during the project development include:

- First fills and consumables
- Owner's team
- Project insurance
- Bonding
- Permitting
- Land acquisition
- Consultants other than EPCM
- Letters of credit
- Office equipment, furniture and hardware

21.2 MINE CAPITAL

See Section 16.7 for a discussion of capital costs for mining.

21.3 CAPITAL COST ESTIMATE TABULATION

The initial capital investment for complete development of the project is estimated to be $2.456 billion total direct and indirect cost. Table 21-2 shows the summary of the cost breakdown.

Table 21-2: Capital Cost Estimate Summary

	(millions)
Direct Costs	
Mining Equipment & Mine Development	$454
Concentrator (including related facilities)	$904
Heap Leach Operation	$139
Camp	$70
Sub-Total	**$1,566**
Indirect Costs	$295
Infrastructure Costs	
Power Plant	$209
Access Road	$99
Airstrip	$24
Subtotal Infrastructure	$332
Contingency	**$218**
Owner's Costs	$44
Grand Total	**$2,456**

21.4 OPERATING COSTS

This section addresses the following costs:

- Process Plant Operating & Maintenance Cost
- General and Administrative Costs

The operating and maintenance costs for the Casino operations are summarized by areas of the plant and shown in Table 21-3. Cost centers include mine operations, process plant operations, and the General and Administration area. Process operating costs were determined on year by year of operations shown below is (Year 2) for illustration, based on an annual mill ore tonnage of 45.9 million tonnes that will produce approximately 474,000 tonnes of copper concentrates and 14,000 tonnes of molybdenum concentrates. The heap leach plant costs are based on gold ore processed of 9.13 million tonnes and production of approximately 55,000 ounces of gold, 134,000 ounces of silver, and 978 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 $8.52 per tonne for sulphide ore, which includes mining, concentrator plant and general and administrative costs. The life of mine average operating cost is $4.04 per tonne for oxide ore which includes processing only.

Table 21-3: Operating Cost – Mine Site Cost Summary

Concentrator Processing Units Base Rate (tonnes/year ore)	45,928,000	
Heap Leach Processing Units Base Rate (tonnes/year ore)	9,125,000	
Total Tons Mined	119,634,000	
	Year 2	
	Cost	$/Mill Ore
Mining Operations		
Drilling	$8,745,347	$0.07
Blasting	$24,896,712	$0.21
Loading	$10,812,550	$0.09
Hauling	$63,209,823	$0.53
Roads and Dumps	$24,843,919	$0.21
Mine Services	$10,496,884	$0.09
Mine Administration	$5,080,065	$0.04
Subtotal Mining	$148,085,301	$1.24
Processing Operations		
Concentrator		$/Mill Ore
Primary Crushing & Stockpile Feed	$15,760,839	$0.34
Grinding, Classification & Pebble Crushing	$135,099,648	$2.94
Flotation & Regrind	$55,927,236	$1.22
Concentrate Thickening/Filtration	$6,414,688	$0.14
Tailings Dewatering & Disposal	$18,786,502	$0.41
Fresh Water/Plant Water	$1,384,185	$0.03
Flotation Reagents	$996,442	$0.02
Ancillary Services	$4,259,720	$0.09
Subtotal Concentrator	$238,629,260	$5.20
Supporting Facilities		$/Mill Ore
Laboratory	$1,493,255	$0.03
General and Administrative	$12,980,264	$0.28
Subtotal Supporting Facilities	$14,473,519	$0.32
Total Mill Ore Cost	$401,188,080	$8.74
Heap Leach		$/Heap Leach Ore
Heap Leach - Gold Ore	$11,947,477	$1.31
ADR/SART - Gold Ore	$24,938,440	$2.73
Subtotal Heap Leach	$36,885,917	$4.04

21.5 PROCESS PLANT OPERATING & MAINTENANCE COSTS

The process plant (concentrator and heap leach) operating costs are summarized by cost elements of labour, power, reagents, maintenance parts and supplies and services. The process plant operating costs are shown by cost element in Table 21-4 and Table 21-5.

21.5.1 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.

21.5.2 Power

Power costs were based on obtaining power from a LNG fuelled power plant at a rate of $C0.095 per kWh. Power consumption was based on the equipment list connected kW, discounted for operating time per day and anticipated operating load level. A summary of the power cost and consumption are shown at the end of this section.

21.5.3 Reagents

Reagents for the process plants are shown below:

- Copper Flotation Reagents
  - Lime
  - Fuel Oil
  - 3418A
  - A208
  - MIBC
  - Flocculent
  - PAX
- Moly Flotation Reagents
  - NaSH
  - FloMin
  - Flocculent
  - Fuel Oil
- Heap Leach Reagents
  - NaSH
  - Sulfuric Acid
  - Hydrochloric Acid
  - Lime
  - Sodium Hydroxide
  - Sodium Cyanide (NaCN)
  - Carbon
  - Anti-scalant

Consumption rates were determined from the metallurgical test data or industry practice. Budget quotations were received for reagents supplied to Skagway, AK, or from local sources where available with allowance for freight to site.

21.5.4 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 $17.4 million for the concentrator and $0.6 million for the heap leach plants.

21.5.5 Process Supplies & Services

Allowances were provided in concentrator, heap leach process plants for outside consultants, outside contractors, vehicle maintenance, and miscellaneous supplies. The allowances were estimated using historical information from other operations and projects. Operating costs for the process plants are summarized in Table 21-4 and Table 21-5.

21.6 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 21-4: Operating Cost – Concentrator Cost Summary – Typical Year of Operation

Processing Units Base Rate (tonnes/year ore)	45,928,000	
	Year 2 Annual Cost	
Primary Crushing & Stockpile Feed		
Labor and Fringes	$1,880,617	$0.04
Power	$4,337,849	$0.09
Liners	$5,961,487	$0.13
Maintenance	$3,283,346	$0.07
Supplies & Services	$297,540	$0.01
Subtotal Primary Crushing & Stockpile Feed	$15,760,839	$0.34
Grinding, Classification & Pebble Crushing		
Labor and Fringes	$1,888,598	$0.04
Power	$59,641,303	$1.30
Liners	$13,060,044	$0.28
Grinding Media	$49,718,262	$1.08
Maintenance	$10,269,551	$0.22
Supplies and Services	$521,890	$0.01
Subtotal Grinding, Classification & Pebble Crushing	$135,099,648	$2.94
Flotation & Regrind		
Labor and Fringes	$2,197,068	$0.05
Power	$12,106,688	$0.26
Reagents	$37,775,148	$0.82
Maintenance	$3,730,342	$0.08
Supplies and Services	$117,990	$0.00
Subtotal Flotation & Regrind	$55,927,236	$1.22
Concentrate Thickening/Filtration		
Labor and Fringes	$2,795,731	$0.06
Power	$1,703,291	$0.04
Maintenance	$1,130,776	$0.02
Supplies and Services	$784,890	$0.02
Subtotal Concentrate Thickening/Filtration	$6,414,688	$0.14
Tailings Dewatering & Disposal		$0.00
Labor and Fringes	$1,982,129	$0.04
Power	$12,956,737	$0.28
Maintenance	$2,516,858	$0.05
Supplies and Services	$1,330,777	$0.03
Subtotal Tailings Dewatering & Disposal	$18,786,502	$0.41
Fresh Water/Plant Water		
Labor and Fringes	$495,416	$0.01
Power	$517,277	$0.01
Maintenance	$284,282	$0.01
Supplies and Services	$87,210	$0.00
Subtotal Fresh Water/Plant Water	$1,384,185	$0.03
Flotation Reagents		
Labor and Fringes	$495,416	$0.01
Power	$185,547	$0.00
Maintenance	$238,529	$0.01
Supplies and Services	$76,950	$0.00
Subtotal Flotation Reagents	$996,442	$0.02
Ancillary Services		
Labor and Fringes	$846,540	$0.02
Power	$644,306	$0.01
Maintenance	$2,691,924	$0.06
Supplies and Services	$76,950	$0.00
Subtotal Ancillary Services	$4,259,720	$0.09
Total Process Plant	$238,629,260	$5.20

Table 21-5: Operating Cost – Heap Leach Cost Summary – Typical Year of Operation

Processing Units Base Rate (tonnes/year ore)	9,125,000	
Processing Units Base Rate (doré oz/year) - Gold Ore	86,164	
Heap Leach		
Labor and Fringes	$984,192	$0.11
Power	$2,976,920	$0.33
Liners	$3,706,324	$0.41
Maintenance	$2,905,200	$0.32
Supplies & Services	$1,374,840	$0.15
Subtotal Heap Leach	$11,947,477	$1.31
ADR/SART (Gold Ore)		
Labor and Fringes	$2,071,038	$0.23
Power	$1,339,918	$0.15
Reagents	$19,983,647	$2.19
Maintenance	$964,148	$0.11
Supplies and Services	$579,690	$0.06
Subtotal ADR/SART (Gold Ore)	$24,938,440	$2.73
Total Process Plant	$36,885,917	$4.04

22 ECONOMIC 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 and silver credits, molybdenum concentrate, and gold & silver doré. The estimates of capital expenditures and site production costs have been developed specifically for this project and have been presented in earlier sections of this report.

All amounts are in Canadian dollars with an exchange rate of 1:1 with the US dollar except as noted otherwise.

22.1 MINE PRODUCTION STATISTICS

Mine production is reported as supergene oxide ore, supergene sulfide ore, hypogene 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 Table 22-1.

Table 22-1: Life of Mine Ore, Waste Quantities and Ore Grade

	Tonnes (000')	Copper %	Moly %	Gold g/t	Silver g/t
Supergene Oxide Ore	60,010	0.235%	0.022%	0.340	1.806
Supergene Sulphide Ore	255,864	0.258%	0.021%	0.250	1.811
Hypogene Ore	649,332	0.179%	0.024%	0.227	1.702
Gold Ore	157,454	0.036%		0.292	2.209
Waste	657,867				
Total Material Mined	**1,780,527**	**0.180%**	**0.020**	**0.247**	**1.803**

22.2 PLANT PRODUCTION STATISTICS

In the current plan, all the Mill Ore and Gold Ore is processed directly, with the Low Grade Mill 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 and silver doré and a copper precipitate. The estimated production over the life of the heap leach is 975,000 ounces of gold, 2,907,000 ounces of silver, and 22.6 million pounds of copper.

Production from the flotation plant will produce a copper-gold and silver concentrate and a molybdenum concentrate. The estimated copper concentrate production for the life of the flotation plant is 6.06 million tonnes containing 3.7 billion pounds of copper and 5.0 million ounces of gold and 28.8 million ounces of silver. The estimated molybdenum concentrate production for the life of the flotation plant is 277,000 tonnes containing 341.9 million pounds of molybdenum.

22.3 CAPITAL EXPENDITURE

22.3.1 Initial Capital

The base case financial indicators have been determined on the basis of 100% equity financing of the initial capital. The base case assumes LNG is used to fuel the power plant from initial start-up of the concentrator and the haulage units throughout the life of the mine. 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 $2.456 billion in Canadian dollars and is expended over a five-year period (4 years of construction plus one additional year for finalizing invoices and other miscellaneous items). 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.

22.3.2 Sustaining Capital

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 $361.7 million. This capital will be expended during a 22 year period.

22.3.3 Working Capital

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 Heap Leach 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.

22.3.4 Salvage Value

A $31.5 million allowance for salvage value has been included in the cash flow analysis.

22.4 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 long term prices that were based on analyst consensus of C$3.16 per pound of copper, C$1,473.68 per oz of gold, C$26.32 per oz of silver and C$14.74 per pound of molybdenum with an exchange rate of US$0.95 per C$. Two other price cases were used; the SEC and a spot price at the end of December 2012. Prices used are as shown in Table 22-2.

Table 22-2: Metal Prices Used in Financial Model

	Base Case Long Term Price	SEC	Spot
Copper (C$/lb)	3.16	3.67	3.57
Molybdenum (C$/lb)	14.74	14.67	11.80
Gold (C$oz)	1,473.68	1,487.85	1,657.50
Silver (C$oz)	26.32	28.80	29.95

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

22.5 TOTAL CASH OPERATING COST

The sulphide average cash operating costs are $10.44 per tonne of mill ore processed. Included in these costs are the mining operations, concentrator, general administration cost and smelter and transportation cost. Heap Leach process operating cost is $4.10 per tonne of leach ore. Included in these costs are the heap leach operations, ADR/SART operations and transportation and refining.

Specifics of operating costs are discussed in Section 21.

22.6 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 $12.58 per tonne of mill ore processed. The Heap Leach costs remain the same at $4.10 per tonne of leach ore as the sulphide process bears these additional costs alone.

22.6.1 Royalty

A NSR royalty will be paid using a rate of 2.75% totalling $604.6 million over the life of the mine.

It is estimated that $1.357 billion will be paid in Yukon mining royalties. Yukon mining royalties are based on a sliding scale of 12% of revenues less operating expenses, depreciation and pre-production expenses.

Corporate income taxes paid is estimated to be $2.878 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.

22.6.2 Property Tax

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

22.6.3 Reclamation & Closure

$125.9 million spread over four years (one year of production and three years of post-production) was allowed for post-closure reclamation & final closeout.

22.7 TOTAL PRODUCTION COST

Total Production Cost is the Total Cash Cost plus depreciation. Depreciation is calculated by the 25% Declining Balance method starting with the first year of production. The last year of production is the catch-up year if the assets are not fully depreciated by that time. An additional deduction for the initial capital is being taken in the early years until the initial capital is fully depreciated.

22.8 PROJECT FINANCING

It is assumed the project will be all equity financed.

22.9 NET INCOME AFTER TAX

Net Income after Tax amounts to $6.7 billion for the life of the mine.

22.10 NPV AND IRR

The base case economic analysis (Table 22-3) indicates that the project has an Internal Rate of Return (IRR) of 20.1% after taxes with a payback period of 3.0 years.

Table 22-3 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 22-3: Sensitivity Analysis (After tax figures)

	NPV @ 0%	NPV @ 5%	NPV @ 8%	NPV @ 10%	IRR	Payback Years
Base Case (LTP)	$6,651	$2,986	$1,830	$1,296	20.1%	3.0
SEC Prices	$7,848	$3,621	$2,287	$1,669	22.5%	2.7
Spot Prices*	$7,744	$3,597	$2,282	$1,672	22.7%	2.6
Base-Case Sensitivities						
Metals Price +10%	$8,157	$3,786	$2,407	$1,768	23.1%	2.6
Metals Price -10%	$5,146	$2,186	$1,253	$824	16.7%	3.5
Capex +10%	$6,499	$2,840	$1,689	$1,158	18.4%	3.2
Capex -10%	$6,804	$3,133	$1,972	$1,434	22.1%	2.7
Opex +10%	$6,103	$2,705	$1,631	$1,135	19.0%	3.1
Opex -10%	$7,200	$3,268	$2,029	$1,457	21.1%	2.9
Mill Recovery +5%	$7,304	$3,329	$2,075	$1,495	21.3%	2.8
Mill Recovery -5%	$5,998	$2,644	$1,585	$1,096	18.7%	3.1
$ in millions						
*Spot prices are on the last day of October 2012	Base Case Copper $3.16 Molybdenum $14.74 Gold $1,473.68 Silver $26.32		SEC Prices Copper $3.67 Molybdenum $14.67 Gold $1.487.85 Silver $28.80		Spot Prices* Copper $3.57 Molybdenum $11.80 Gold $1,657.50 Silver $29.95	

Ore produced during the first four years is substantially higher in copper, gold, silver, and molybdenum than the life-of-mine average. The results are more robust cash flow during those years allowing payback in 3.0 years under the base case. Table 22-4 illustrates the difference during these early years.

Table 22-4: Project Cash Flow

	Years 1-4	Life of Mine
Average Annual Pre-Tax Cashflow ($ thousand)	$773,371	$531,211
Average Annual After-Tax Cashflow ($ thousand)	$682,163	$400,386
Average NSR (sulphide ore)	$31.59	$22.59
Average Annual Mill Feed Grade		
Copper (%)	0.307%	0.204%
Gold (g/t)	0.371	0.240
Silver (g/t)	2.103	1.74
Molybdenum (%)	0.025%	0.023%
Average Concentrate Production		
Copper (dry ktonnes)	395	275
Molybdenum (dry ktonnes)	12	13
Average Annual Metal Production		
Copper & Molybdenum Concentrate (M lb)		
Copper (Mlbs)	245	171
Gold (kozs)	399	260
Silver (kozs)	1,777	1,425
Molybdenum (Mlbs)	15.3	15.5
Gold/Silver Doré		
Gold (kozs)	52	54
Silver (kozs)	157	162
Copper Precipitate		
Copper (Mlbs)	1.4	1.3

The Financial Model is shown on the following page in Table 22-5.

Table 22-5: Financial Model

		Total	-4	-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
Mining Operations																													
Supergene Oxide Ore	Beginning Inventory (kt)	60,010		60,010	59,990	57,184	47,203	27,182	23,920	22,290	14,201	8,065	6,850	6,850	6,115	4,659	3,197	2,954	2,163	679	43	-	-	-	-	-	-	-	-
	Mined (kt)	60,010		20	2,806	9,901	20,021	3,262	1,630	8,089	6,136	1,215	-	735	1,456	1,462	243	791	1,484	636	43	-	-	-	-	-	-	-	-
	Ending Inventory (kt)	-		59,990	57,184	47,203	27,182	23,920	22,290	14,201	8,065	6,850	6,850	6,115	4,659	3,197	2,954	2,163	679	43	-	-	-	-	-	-	-	-	-
	Copper Grade (%)	0.235%		0.165%	0.175%	0.190%	0.305%	0.272%	0.160%	0.212%	0.201%	0.146%	0.000%	0.151%	0.184%	0.225%	0.171%	0.225%	0.225%	0.149%	0.134%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
	Molybdenum Grade (%)	0.022%		0.0202%	0.0185%	0.0207%	0.0278%	0.0103%	0.0126%	0.0189%	0.0209%	0.0036%	0.0000%	0.0359%	0.0321%	0.0213%	0.0323%	0.0180%	0.0099%	0.0094%	0.0106%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%
	Gold Grade (g/t)	0.340		0.577	0.540	0.580	0.425	0.208	0.238	0.172	0.127	0.163	-	0.224	0.212	0.187	0.086	0.127	0.131	0.121	0.107	-	-	-	-	-	-	-	-
	Silver Grade (g/t)	1.806		2.230	2.450	2.947	2.015	0.990	1.338	1.408	0.840	0.680	-	1.595	1.767	1.758	0.835	0.935	0.788	0.709	0.710	-	-	-	-	-	-	-	-
	Contained Copper (klbs)	311,294		73	10,826	41,777	134,454	19,534	5,735	37,785	27,179	3,919	-	2,448	5,904	7,257	915	3,927	7,351	2,084	127	-	-	-	-	-	-	-	-
	Contained Molybdenum (klbs)	28,716		9	1,144	4,545	12,289	739	454	3,366	2,824	96	-	582	1,030	686	173	314	324	131	10	-	-	-	-	-	-	-	-
	Contained Gold (kozs)	656		0	49	186	274	22	12	45	25	6	-	5	10	9	1	3	6	2	0	-	-	-	-	-	-	-	-
	Contained Silver (kozs)	3,484		1	221	946	1,297	104	70	366	166	27	-	38	83	83	7	24	38	15	1	-	-	-	-	-	-	-	-
Supergene Sulfide Ore	Beginning Inventory (kt)	255,864		255,864	255,864	255,261	251,230	224,103	191,350	180,528	156,297	130,503	126,726	126,701	126,425	121,599	87,753	64,288	34,938	26,015	13,519	1,730	-	-	-	-	-	-	-
	Mined (kt)	255,864		-	603	4,031	27,127	32,753	10,822	24,231	25,794	3,777	25	276	4,026	33,846	33,465	19,350	8,923	12,496	11,789	1,730	-	-	-	-	-	-	-
	Ending Inventory (kt)	-		255,864	255,261	251,230	224,103	191,350	180,528	156,297	130,503	126,726	126,701	126,425	121,599	87,753	64,288	34,938	26,015	13,519	1,730	-	-	-	-	-	-	-	-
	Copper Grade (%)	0.258%		0.000%	0.232%	0.221%	0.349%	0.361%	0.279%	0.237%	0.214%	0.157%	0.047%	0.123%	0.178%	0.250%	0.255%	0.207%	0.214%	0.212%	0.179%	0.141%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
	Molybdenum Grade (%)	0.021%		0.0000%	0.0121%	0.0122%	0.0228%	0.0266%	0.0250%	0.0125%	0.0146%	0.0156%	0.0028%	0.0130%	0.0196%	0.0246%	0.0240%	0.0248%	0.0165%	0.0135%	0.0159%	0.0169%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%
	Gold Grade (g/t)	0.250		-	0.359	0.303	0.413	0.359	0.294	0.252	0.226	0.229	0.269	0.232	0.215	0.193	0.108	0.175	0.162	0.178	0.171	0.148	-	-	-	-	-	-	-
	Silver Grade (g/t)	1.811		-	1.880	2.395	2.583	1.839	1.746	1.932	1.672	1.822	0.600	2.005	2.037	1.798	1.697	1.613	1.325	1.318	1.436	1.149	-	-	-	-	-	-	-
	Contained Copper (klbs)	1,455,414		-	3,084	19,616	208,988	260,859	66,489	126,828	121,917	13,039	26	747	18,905	186,432	187,897	88,300	42,030	58,333	46,533	5,390	-	-	-	-	-	-	-
	Contained Molybdenum (klbs)	116,897		-	161	1,088	13,635	19,242	5,960	6,678	8,284	1,296	2	79	2,087	18,337	17,715	10,583	3,254	3,730	4,124	646	-	-	-	-	-	-	-
	Contained Gold (kozs)	2,057		-	7	50	380	378	102	197	187	28	0	2	33	210	202	109	47	71	65	8	-	-	-	-	-	-	-
	Contained Silver (kozs)	14,894		-	36	310	2,252	1,937	608	1,505	1,387	221	0	18	316	1,957	1,825	1,003	380	530	544	64	-	-	-	-	-	-	-
Hypogene Ore	Beginning Inventory(kt)	649,332		649,332	649,332	649,332	648,884	645,988	628,888	592,131	571,624	545,389	486,543	433,019	385,110	340,600	329,158	316,828	285,257	240,386	201,050	159,075	108,598	59,610	14,676	-	-	-	-
	Mined (kt)	649,332		-	-	448	2,896	17,100	36,757	20,507	26,235	58,846	53,524	47,909	44,510	11,442	12,330	31,571	44,871	39,336	41,975	50,477	40,988	44,934	14,676	-	-	-	-
	Ending Inventory (kt)	-		649,332	649,332	648,884	645,988	628,888	592,131	571,624	545,389	486,543	433,019	385,110	340,600	329,158	316,828	285,257	240,386	201,050	159,075	108,598	59,610	14,676	-	-	-	-	-
	Copper Grade (%)	0.179%		0.000%	0.000%	0.264%	0.259%	0.267%	0.281%	0.270%	0.185%	0.173%	0.102%	0.190%	0.191%	0.182%	0.185%	0.157%	0.161%	0.175%	0.157%	0.129%	0.132%	0.152%	0.178%	0.000%	0.000%	0.000%	0.000%
	Molybdenum Grade (%)	0.024%		0.0000%	0.0000%	0.0134%	0.0226%	0.0232%	0.0285%	0.0273%	0.0135%	0.0162%	0.0220%	0.0273%	0.0305%	0.0322%	0.0277%	0.0239%	0.0261%	0.0228%	0.0211%	0.0162%	0.0211%	0.0307%	0.0310%	0.0000%	0.0000%	0.0000%	0.0000%
	Gold Grade (g/t)	0.227		-	-	0.492	0.512	0.371	0.379	0.346	0.247	0.239	0.245	0.237	0.223	0.215	0.207	0.201	0.200	0.194	0.167	0.160	0.169	0.199	0.195	-	-	-	-
	Silver Grade (g/t)	1.702		-	-	2.340	2.850	2.105	1.900	2.038	1.843	1.710	1.991	1.556	1.610	1.566	1.444	1.670	1.543	1.456	1.631	1.590	1.702	1.560	2.000	-	-	-	-
	Contained Copper (klbs)	2,564,779		-	-	2,607	16,536	100,781	227,664	125,706	107,245	224,895	214,643	201,084	187,483	45,957	50,419	109,348	159,446	151,332	145,007	143,796	142,662	150,575	57,592	-	-	-	-
	Contained Molybdenum (klbs)	330,163		-	-	132	1,443	8,750	23,098	12,320	7,012	21,076	26,016	28,061	29,941	8,117	7,525	16,658	25,818	19,809	19,510	18,047	22,787	30,412	10,030	-	-	-	-
	Contained Gold (kozs)	4,733		-	-	7	48	204	448	228	209	453	422	365	320	79	82	204	288	245	226	260	266	287	92	-	-	-	-
	Contained Silver (kozs)	35,529		-	-	34	265	1,157	2,245	1,343	1,554	3,235	3,427	2,396	2,304	676	572	1,695	2,226	1,841	2,201	2,580	2,681	2,254	944	-	-	-	-
Total Mill Ore	Beginning Inventory(kt)	965,206	-	965,206	965,186	961,777	947,317	897,273	844,158	794,949	742,122	683,957	620,119	566,570	517,650	466,868	420,108	374,070	322,358	267,080	214,612	160,805	108,598	59,610	14,676	-	-	-	-
	Mined (kt)	965,206	-	20	3,409	14,460	50,044	53,115	49,209	52,827	58,165	63,838	53,549	48,920	50,792	46,750	46,038	51,712	55,278	52,468	53,807	52,207	48,988	44,934	14,676	-	-	-	-
	Ending Inventory (kt)	-		965,186	961,777	947,317	897,273	844,158	794,949	742,122	683,957	620,119	566,570	517,650	466,868	420,108	374,070	322,358	267,080	214,612	160,805	108,598	59,610	14,676	-	-	-	-	-
	Copper Grade (%)	0.204%	0.000%	0.165%	0.185%	0.201%	0.326%	0.326%	0.276%	0.249%	0.200%	0.172%	0.182%	0.189%	0.190%	0.233%	0.236%	0.177%	0.171%	0.183%	0.162%	0.130%	0.132%	0.152%	0.178%	0.000%	0.000%	0.000%	0.000%
	Molybdenum Grade (%)	0.023%	0.000%	0.020%	0.017%	0.018%	0.025%	0.025%	0.027%	0.019%	0.015%	0.016%	0.022%	0.027%	0.030%	0.026%	0.025%	0.024%	0.024%	0.020%	0.020%	0.016%	0.021%	0.031%	0.031%	0.000%	0.000%	0.000%	0.000%
	Gold Grade (g/t)	0.240	-	0.577	0.508	0.522	0.424	0.353	0.366	0.276	0.225	0.237	0.245	0.237	0.222	0.198	0.193	0.190	0.192	0.189	0.168	0.160	0.169	0.199	0.195	-	-	-	-
	Silver Grade (g/t)	1.737	-	2.230	2.349	2.774	2.371	1.872	1.847	1.893	1.661	1.697	1.991	1.559	1.655	1.740	1.624	1.637	1.487	1.414	1.587	1.575	1.702	1.560	2.000	-	-	-	-
	Contained Copper (klbs)	4,331,487	-	73	13,910	64,001	359,978	381,173	299,888	290,318	256,341	241,853	214,669	204,280	212,293	239,646	239,232	201,576	208,827	211,749	191,667	149,186	142,662	150,575	57,592	-	-	-	-
	Contained Molybdenum (klbs)	483,776	-	9	1,305	5,765	27,367	20,731	29,512	22,364	10,920	22,468	26,017	29,522	33,057	27,140	25,412	27,555	29,395	23,670	23,644	10,693	22,787	30,412	10,030	-	-	-	-
	Contained Gold (kozs)	7,445	-	0	56	243	682	604	563	469	421	487	423	373	363	298	285	316	341	319	290	268	266	287	92	-	-	-	-
	Contained Silver (kozs)	53,906	-	1	257	1,290	3,814	3,198	2,923	3,215	3,107	3,482	3,427	2,452	2,703	2,616	2,404	2,722	2,643	2,385	2,746	2,644	2,681	2,254	944	-	-	-	-
Gold Ore	Beginning Inventory(kt)	157,454		157,454	150,874	136,677	115,023	96,356	78,777	65,898	58,089	57,594	56,522	47,749	34,638	21,286	3,051	1,788	1,163	238	-	-	-	-	-	-	-	-	-
	Mined (kt)	157,454		6,580	14,197	21,654	18,667	17,579	12,879	7,809	495	1,072	8,773	13,111	13,352	18,235	1,263	625	925	238	-	-	-	-	-	-	-	-	-
	Ending Inventory (kt)	-		150,874	136,677	115,023	96,356	78,777	65,898	58,089	57,594	56,522	47,749	34,638	21,286	3,051	1,788	1,163	238	-	-	-	-	-	-	-	-	-	-
	Copper Grade (%)	0.036%		0.0250%	0.0400%	0.0460%	0.0460%	0.0270%	0.0370%	0.0440%	0.0490%	0.0050%	0.0140%	0.0210%	0.0310%	0.0450%	0.0450%	0.0530%	0.0530%	0.0590%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%	0.0000%
	Gold Grade (g/t)	0.292		0.330	0.445	0.423	0.333	0.284	0.236	0.190	0.171	0.231	0.243	0.239	0.208	0.201	0.172	0.186	0.167	0.149	-	-	-	-	-	-	-	-	-
	Silver Grade (g/t)	2.209		1.790	2.800	3.090	2.170	1.760	2.300	1.920	1.520	2.740	2.560	2.190	1.860	1.660	1.000	1.110	1.180	1.110	-	-	-	-	-	-	-	-	-
	Contained Copper (klbs)	125,601		3,627	12,520	21,960	18,931	10,464	10,506	7,575	535	118	2,708	6,070	9,125	18,091	1,263	730	1,081	310	-	-	-	-	-	-	-	-	-
	Contained Gold (kozs)	1,477		70	203	294	200	161	98	48	3	8	69	101	89	118	7	4	5	1	-	-	-	-	-	-	-	-	-
	Contained Silver (kozs)	11,181		379	1,278	2,151	1,302	995	952	402	24	94	722	923	790	973	41	22	35	8	-	-	-	-	-	-	-	-	-
Waste	Beginning Inventory(kt)	657,867		657,867	656,385	653,449	648,187	625,675	596,369	558,458	519,095	477,756	442,667	404,988	367,019	331,161	296,146	243,448	195,785	151,988	109,203	71,374	33,440	13,198	4,932	-	-	-	-
	Mined (kt)	657,867		1,482	2,936	5,262	22,512	29,306	37,911	39,363	41,339	35,089	37,679	37,969	35,858	35,016	52,698	47,663	43,797	42,785	37,829	37,934	20,242	8,266	4,932	-	-	-	-
	Ending Inventory (kt)	-		656,385	653,449	648,187	625,675	596,369	558,458	519,095	477,756	442,667	404,988	367,019	331,161	296,146	243,448	195,785	151,988	109,203	71,374	33,440	13,198	4,932	-	-	-	-	-
	Total Material Mined (kt)	1,700,527		8,082	20,542	41,376	91,223	100,000	99,999	99,999	99,999	99,999	100,001	100,000	100,002	100,000	99,999	100,000	100,000	95,491	91,636	90,141	69,230	53,200	19,608	-	-	-	-
	Waste to Ore Ratio	0.59		0.22	0.17	0.15	0.33	0.41	0.61	0.65	0.70	0.54	0.60	0.61	0.56	0.54	1.11	0.91	0.78	0.81	0.70	0.73	0.41	0.18	0.34	-	-	-	-
	Mine Re-handle (kt)	181,167		-	-	-	4,929	-	-	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,600	3,610	-	-	-	-	-	-	30,915	45,053	44,985	22,875	-
Process Plant Operations - Concentrator																													
Supergene Oxide Ore	Beginning Ore Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Mined Ore to Concentrator (kt)	60,010		-	-	-	-	1,228	888	7,950	6,416	3,606	3,600	4,228	4,780	4,782	3,738	4,004	717	30	-	-	-	-	1,011	1,570	4,429	7,033	-
	Mined Ore - Processed (kt)	60,010		-	-	-	-	1,228	888	7,950	6,416	3,606	3,600	4,228	4,780	4,782	3,738	4,004	717	30	-	-	-	-	1,011	1,570	4,429	7,033	-
	Ending Ore Inventory	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Copper Grade (%)	0.235%		0.000%	0.000%	0.000%	0.000%	0.377%	0.181%	0.271%	0.276%	0.308%	0.308%	0.286%	0.233%	0.201%	0.190%	0.190%	0.280%	0.181%	0.000%	0.000%	0.000%	0.000%	0.156%	0.151%	0.165%	0.179%	0.000%
	Molybdenum Grade (%)	0.022%		0.0000%	0.0000%	0.0000%	0.0000%	0.0200%	0.0124%	0.0236%	0.0274%	0.0281%	0.0281%	0.0299%	0.0271%	0.0214%	0.0212%	0.0185%	0.0101%	0.0046%	0.0000%	0.0000%	0.0000%	0.0000%	0.0097%	0.0154%	0.0127%	0.0132%	0.0000%
	Gold Grade (g/t)	0.340		-	-	-	-	0.351	0.299	0.311	0.308	0.430	0.430	0.400	0.442	0.487	0.563	0.512	0.155	0.198	-	-	-	-	0.114	0.117	0.122	0.131	-
	Silver Grade (g/t)	1.006		-	-	-	-	1.620	1.620	1.811	1.595	2.028	2.030	1.962	2.364	2.666	2.874	2.407	0.860	1.100	-	-	-	-	0.702	1.097	0.673	0.979	-
	Contained Copper (klbs)	311,294		-	-	-	-	10,206	3,543	47,557	39,034	24,472	24,445	26,688	24,559	21,151	15,664	16,814	4,426	120	-	-	-	-	3,472	5,237	16,102	27,804	-
	Contained Molybdenum (klbs)	28,716		-	-	-	-	541	243	4,129	3,875	2,233	2,230	2,788	2,851	2,255	1,750	1,632	160	3	-	-	-	-	216	532	1,236	2,041	-
	Contained Gold (kozs)	656		-	-	-	-	14	9	79	64	50	50	54	68	75	68	66	4	0	-	-	-	-	4	6	17	30	-
	Contained Silver (kozs)	3,484		-	-	-	-	64	46	463	329	235	235	267	363	410	345	310	20	1	-	-	-	-	23	55	96	221	-
	Recovery Copper (%)	65.52%		0.00%	0.00%	0.00%	0.00%	77.29%	76.86%	66.69%	64.26%	62.26%	62.26%	62.44%	64.99%	67.02%	62.02%	61.64%	63.11%	79.98%	0.00%	0.00%	0.00%	0.00%	69.32%	69.86%	67.26%	69.84%	0.00%
	Recovery Molybdenum (%)	52.25%		0.00%	0.00%	0.00%	0.00%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	0.00%
	Recovery Gold (%)	69.00%		0.00%	0.00%	0.00%	0.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	0.00%
	Recovery Silver (%)	60.00%		0.00%	0.00%	0.00%	0.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	0.00%
	Copper Concentrate (kt)	330		-	-	-	-	13	4	61	41	25	25	27	26	23	16	17	5	0	-	-	-	-	4	6	18	31	-
	Copper Concentrate Grade (%)	20.00%		0.00%	0.00%	0.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	0.00%
	Recovered Copper (klbs)	203,950		-	-	-	-	7,889	2,724	31,716	25,082	15,237	15,219	16,664	15,962	14,176	9,714	10,364	2,793	96	-	-	-	-	2,407	3,669	10,830	19,418	-
	Recovered Gold (kozs)	453		-	-	-	-	10	6	55	44	34	34	37	47	52	47	45	2	0	-	-	-	-	3	4	12	20	-
	Recovered Silver (kozs)	2,090		-	-	-	-	38	28	278	197	141	141	160	218	246	207	186	12	1	-	-	-	-	14	33	57	133	-
	Molybdenum Concentrate (kt)	12		-	-	-	-	0	0	2	2	1	1	1	1	1	1	1	0	0	-	-	-	-	0	0	1	1	-
	Molybdenum Concentrate Grade (%)	56.00%		0.00%	0.00%	0.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	0.00%
	Recovered Molybdenum (klbs)	15,004		-	-	-	-	283	127	2,157	2,025	1,166	1,165	1,457	1,490	1,178	914	853	83	2	-	-	-	-	113	278	646	1,067	-
Supergene Sulfide Ore	Beginning Ore Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Mined Ore to Concentrator (kt)	255,864		-	-	-	31,156	28,786	9,179	18,498	19,113	1,894	-	132	3,089	29,872	30,269	16,054	7,591	10,775	8,954	1,171	-	-	5,744	13,081	8,377	12,149	-
	Mined Ore - Processed (kt)	255,864		-	-	-	31,156	28,786	9,179	18,498	19,113	1,894	-	132	3,089	29,872	30,269	16,054	7,591	10,775	8,954	1,171	-	-	5,744	13,081	8,377	12,149	-
	Ending Ore Inventory	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Copper Grade (%)	0.258%		0.000%	0.000%	0.000%	0.335%	0.385%	0.301%	0.250%	0.239%	0.189%	0.000%	0.175%	0.213%	0.263%	0.267%	0.223%	0.225%	0.224%	0.199%	0.162%	0.000%	0.000%	0.124%	0.137%	0.139%	0.173%	0.000%
	Molybdenum Grade (%)	0.021%		0.0000%	0.0000%	0.0000%	0.0215%	0.0293%	0.0282%	0.0134%	0.0148%	0.0186%	0.0000%	0.0194%	0.0250%	0.0263%	0.0253%	0.0268%	0.0176%	0.0138%	0.0157%	0.0162%	0.0000%	0.0000%	0.0146%	0.0122%	0.0136%	0.0086%	0.0000%
	Gold Grade (g/t)	0.250		-	-	-	0.414	0.385	0.320	0.285	0.261	0.307	-	0.243	0.244	0.201	0.194	0.188	0.170	0.185	0.191	0.172	-	-	0.115	0.132	0.130	0.154	-
	Silver Grade (g/t)	1.811		-	-	-	2.664	1.930	1.860	2.100	1.900	2.420	-	2.120	2.310	1.870	1.760	1.720	1.380	1.390	1.580	1.330	-	-	0.930	1.233	1.064	1.288	-
	Contained Copper (klbs)	1,455,414		-	-	-	229,828	244,329	60,911	105,215	100,707	7,892	-	509	14,412	173,202	178,173	78,926	37,654	53,211	39,283	4,182	-	-	15,646	39,379	25,734	46,219	-
	Contained Molybdenum (klbs)	116,897		-	-	-	14,747	18,594	5,707	5,465	6,236	777	-	56	1,691	17,320	16,883	9,485	2,945	3,278	3,099	418	-	-	1,850	3,526	2,506	2,312	-
	Contained Gold (kozs)	2,057		-	-	-	414	356	94	169	160	19	-	1	24	193	189	97	41	64	55	6	-	-	21	56	35	60	-
	Contained Silver (kozs)	14,894		-	-	-	2,568	1,786	549	1,249	1,160	147	-	9	228	1,796	1,703	888	337	402	455	50	-	-	172	519	287	503	-

		Total	-4	-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
	Recovery Copper (%)	80.62%		0.00%	0.00%	0.00%	75.46%	80.08%	83.07%	81.61%	79.45%	83.56%	0.00%	76.27%	80.32%	82.21%	83.79%	84.30%	81.47%	83.51%	85.03%	84.72%	0.00%	0.00%	79.48%	79.25%	77.69%	76.36%	0.00%
	Recovery Molybdenum (%)	52.25%		0.00%	0.00%	0.00%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	52.25%	0.00%
	Recovery Gold (%)	69.00%		0.00%	0.00%	0.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	69.00%	0.00%
	Recovery Silver (%)	60.00%		0.00%	0.00%	0.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	0.00%
	Copper Concentrate (kt)	1,901		-	-	-	281	317	82	139	130	11	-	1	19	231	242	108	50	72	54	6	-	-	20	51	32	57	-
	Copper Concentrate Grade (%)	28.00%		0.00%	0.00%	0.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	0.00%
	Recovered Copper (klbs)	1,173,329		-	-	-	173,431	195,667	50,599	85,869	80,010	6,594	-	388	11,575	142,382	149,292	66,539	30,676	44,436	33,400	3,543	-	-	12,435	31,209	19,993	35,293	-
	Recovered Gold (kozs)	1,419		-	-	-	286	246	65	117	111	13	-	1	17	133	130	67	29	44	38	4	-	-	15	38	24	41	-
	Recovered Silver (kozs)	8,936		-	-	-	1,541	1,072	329	749	701	88	-	5	137	1,078	1,022	533	202	289	273	30	-	-	103	311	172	302	-
	Molybdenum Concentrate (kt)	49		-	-	-	6	8	2	2	3	0	-	0	1	7	7	4	1	1	1	0	-	-	1	1	1	1	-
	Molybdenum Concentrate Grade (%)	56.00%		0.00%	0.00%	0.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	0.00%
	Recovered Molybdenum (klbs)	61,079		-	-	-	7,705	9,716	2,982	2,855	3,258	406	-	29	884	9,050	8,821	4,956	1,539	1,713	1,619	219	-	-	967	1,843	1,309	1,208	-
Hypogene Ore	Beginning Ore Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Mined Ore to Concentrator (kt)	649,332		-	-	-	3,344	15,914	35,747	19,207	19,724	39,705	41,576	41,385	38,243	10,121	10,688	25,803	37,941	34,993	36,233	43,893	44,772	44,934	38,836	30,404	32,177	3,692	-
	Mined Ore - Processed (kt)	649,332		-	-	-	3,344	15,914	35,747	19,207	19,724	39,705	41,576	41,385	38,243	10,121	10,688	25,803	37,941	34,993	36,233	43,893	44,772	44,934	38,836	30,404	32,177	3,692	-
	Ending Ore Inventory			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Copper Grade (%)	0.179%		0.000%	0.000%	0.000%	0.260%	0.276%	0.286%	0.288%	0.209%	0.201%	0.204%	0.204%	0.205%	0.192%	0.198%	0.173%	0.174%	0.184%	0.166%	0.136%	0.137%	0.152%	0.123%	0.100%	0.113%	0.129%	0.000%
	Molybdenum Grade (%)	0.024%		0.0000%	0.0000%	0.0000%	0.0214%	0.0243%	0.0291%	0.0285%	0.0145%	0.0181%	0.0239%	0.0290%	0.0323%	0.0339%	0.0294%	0.0264%	0.0284%	0.0243%	0.0223%	0.0164%	0.0214%	0.0307%	0.0206%	0.0161%	0.0127%	0.0084%	0.0000%
	Gold Grade (g/t)	0.227		-	-	-	0.509	0.385	0.384	0.359	0.280	0.278	0.274	0.254	0.240	0.229	0.220	0.210	0.209	0.200	0.174	0.167	0.175	0.199	0.151	0.138	0.154	0.176	-
	Silver Grade (g/t)	1.702		-	-	-	2.782	2.160	1.920	2.080	1.860	1.830	2.190	1.640	1.710	1.650	1.520	1.770	1.620	1.500	1.710	1.660	1.760	1.560	1.449	1.106	1.491	1.351	-
	Contained Copper (klbs)	2,564,779		-	-	-	19,144	96,833	225,392	121,951	90,881	175,944	186,985	186,126	172,838	42,841	46,655	98,412	145,543	141,949	132,601	131,604	135,226	150,575	105,580	66,853	80,380	10,466	-
	Contained Molybdenum (klbs)	338,163		-	-	-	1,575	8,525	22,933	12,068	6,305	15,844	21,907	26,459	27,233	7,564	6,928	15,018	23,755	18,747	17,813	15,870	21,123	30,412	17,605	10,786	9,006	687	-
	Contained Gold (kozs)	4,733		-	-	-	55	197	441	222	178	355	366	338	295	75	76	174	255	225	203	236	252	287	189	135	159	21	-
	Contained Silver (kozs)	35,529		-	-	-	299	1,105	2,207	1,284	1,180	2,336	2,927	2,182	2,103	537	522	1,468	1,976	1,688	1,992	2,343	2,533	2,254	1,809	1,081	1,543	160	-
	Recovery Copper (%)	92.15%		0.00%	0.00%	0.00%	92.15%	92.15%	92.15%	92.15%	92.15%	92.15%	92.15%	92.15%	92.15%	92.15%	92.15%	92.15%	92.15%	92.15%	92.15%	92.15%	92.15%	92.15%	92.15%	92.15%	92.15%	92.15%	0.00%
	Recovery Molybdenum (%)	78.60%		0.00%	0.00%	0.00%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	78.60%	0.00%
	Recovery Gold (%)	66.00%		0.00%	0.00%	0.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	66.00%	0.00%
	Recovery Silver (%)	50.00%		0.00%	0.00%	0.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	50.00%	0.00%
	Copper Concentrate (kt)	3,829		-	-	-	29	145	336	182	136	263	279	278	258	64	70	147	217	212	198	196	202	225	158	100	120	16	-
	Copper Concentrate Grade (%)	28.00%		0.00%	0.00%	0.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	0.00%
	Recovered Copper (klbs)	2,363,444		-	-	-	17,641	89,231	207,699	112,378	83,747	162,133	172,307	171,515	159,270	39,478	42,992	90,687	134,118	130,806	122,192	121,273	124,611	138,755	97,292	61,605	74,070	9,645	-
	Recovered Gold (kozs)	3,124		-	-	-	36	130	291	146	117	234	242	223	195	49	50	115	168	149	134	156	166	190	125	89	105	14	-
	Recovered Silver (kozs)	17,764		-	-	-	150	553	1,103	642	590	1,168	1,464	1,091	1,051	268	261	734	988	844	996	1,171	1,267	1,127	904	540	771	80	-
	Molybdenum Concentrate (kt)	215		-	-	-	1	5	15	8	4	10	14	17	17	5	4	10	15	12	11	10	13	19	11	7	6	0	-
	Molybdenum Concentrate Grade (%)	56.00%		0.00%	0.00%	0.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	0.00%
	Recovered Molybdenum (klbs)	265,796		-	-	-	1,238	6,701	18,026	9,486	4,956	12,453	17,219	20,797	21,405	5,945	5,445	11,804	18,672	14,735	14,001	12,474	16,603	23,904	13,837	8,478	7,079	540	-
Total Mill Ore	Beginning Ore Inventory (kt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Mined Ore to Concentrator (kt)	965,206	-	-	-	-	34,500	45,928	45,814	45,655	45,253	45,205	45,176	45,745	46,092	44,775	44,695	45,861	46,249	45,798	45,187	45,064	44,772	44,934	45,591	45,055	44,983	22,874	-
	Mined Ore - Processed (kt)	965,206	-	-	-	-	34,500	45,928	45,814	45,655	45,253	45,205	45,176	45,745	46,092	44,775	44,695	45,861	46,249	45,798	45,187	45,064	44,772	44,934	45,591	45,055	44,983	22,874	-
	Ending Ore Inventory		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Copper Grade (%)	0.204%	0.000%	0.000%	0.000%	0.000%	0.327%	0.347%	0.287%	0.273%	0.231%	0.209%	0.212%	0.212%	0.208%	0.240%	0.244%	0.192%	0.184%	0.193%	0.173%	0.137%	0.137%	0.152%	0.124%	0.112%	0.123%	0.168%	0.000%
	Molybdenum Grade (%)	0.023%	0.000%	0.000%	0.000%	0.000%	0.021%	0.027%	0.029%	0.022%	0.016%	0.019%	0.024%	0.029%	0.031%	0.027%	0.026%	0.026%	0.026%	0.022%	0.021%	0.016%	0.021%	0.031%	0.020%	0.015%	0.013%	0.010%	0.000%
	Gold Grade (g/t)	0.240	-	-	-	-	0.423	0.384	0.370	0.321	0.276	0.291	0.286	0.267	0.261	0.238	0.231	0.229	0.202	0.196	0.177	0.167	0.175	0.199	0.146	0.136	0.146	0.150	-
	Silver Grade (g/t)	1.737	-	-	-	-	2.585	2.001	1.902	2.041	1.839	1.870	2.177	1.671	1.818	1.905	1.789	1.808	1.569	1.474	1.684	1.651	1.760	1.560	1.367	1.143	1.331	1.203	-
	Contained Copper (klbs)	4,331,487	-	-	-	-	248,972	351,369	289,847	274,723	230,623	208,308	211,430	213,323	211,809	237,194	240,492	194,153	187,624	195,280	171,884	135,786	135,226	150,575	124,697	111,468	122,215	84,490	-
	Contained Molybdenum (klbs)	483,776	-	-	-	-	16,322	27,661	28,883	21,662	16,417	18,853	24,137	29,304	31,775	27,140	25,560	26,135	26,860	22,028	20,912	16,288	21,123	30,412	19,671	14,845	12,748	5,040	-
	Contained Gold (kozs)	7,445	-	-	-	-	469	567	544	471	401	423	416	393	387	342	332	337	300	289	258	242	252	287	214	197	212	111	-
	Contained Silver (kozs)	53,906	-	-	-	-	2,867	2,955	2,802	2,996	2,676	2,719	3,162	2,458	2,694	2,743	2,571	2,666	2,333	2,170	2,447	2,393	2,533	2,254	2,003	1,655	1,925	885	-
	Recovery Copper (%)	86.36%	0.00%	0.00%	0.00%	0.00%	76.74%	83.33%	90.05%	83.71%	81.88%	88.31%	88.69%	88.40%	88.20%	82.65%	83.99%	86.32%	89.32%	89.79%	90.52%	91.92%	92.15%	92.15%	89.92%	86.55%	85.83%	76.17%	0.00%
	Recovery Molybdenum (%)	70.67%	0.00%	0.00%	0.00%	0.00%	54.79%	60.37%	73.17%	66.93%	62.37%	74.39%	76.17%	76.04%	74.83%	59.59%	59.39%	67.39%	75.55%	74.67%	74.69%	77.92%	78.60%	78.60%	75.83%	71.40%	70.87%	55.84%	0.00%
	Recovery Gold (%)	67.09%	0.00%	0.00%	0.00%	0.00%	68.65%	67.96%	66.57%	67.59%	67.67%	66.49%	66.36%	66.42%	66.71%	68.35%	68.32%	67.45%	66.45%	66.67%	66.64%	66.08%	66.00%	66.00%	66.35%	66.94%	66.74%	68.43%	0.00%
	Recovery Silver (%)	53.41%	0.00%	0.00%	0.00%	0.00%	58.96%	56.26%	52.12%	55.71%	55.59%	51.41%	50.74%	51.12%	52.19%	58.04%	57.97%	54.49%	51.53%	52.22%	51.86%	50.21%	50.00%	50.00%	50.97%	53.47%	51.99%	58.19%	0.00%
	Copper Concentrate (kt)	6,060	-	-	-	-	310	474	423	373	306	298	304	305	303	318	327	271	271	284	252	202	202	225	182	156	170	104	-
	Copper Concentrate Grade (%)	28.00%	0.00%	0.00%	0.00%	0.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	28.00%	0.00%
	Recovered Copper (klbs)	3,740,723	-	-	-	-	191,072	292,787	261,021	229,963	188,838	183,963	187,526	188,567	186,808	196,035	201,999	167,590	167,587	175,338	155,592	124,816	124,611	138,755	112,133	96,472	104,894	64,356	-
	Recovered Gold (kozs)	4,995	-	-	-	-	322	385	362	318	272	281	276	261	258	234	227	227	199	193	172	160	166	190	142	132	141	76	-
	Recovered Silver (kozs)	28,791	-	-	-	-	1,690	1,663	1,460	1,669	1,488	1,398	1,605	1,256	1,406	1,592	1,490	1,453	1,202	1,133	1,269	1,201	1,267	1,127	1,021	885	1,001	515	-
	Molybdenum Concentrate (kt)	277	-	-	-	-	7	14	17	12	8	11	15	18	19	13	12	14	16	13	13	10	13	19	12	9	7	2	-
	Molybdenum Concentrate Grade (%)	56.00%	0.00%	0.00%	0.00%	0.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	56.00%	0.00%
	Recovered Molybdenum (klbs)	341,879	-	-	-	-	8,943	16,700	21,134	14,498	10,239	14,025	18,384	22,283	23,778	16,174	15,181	17,613	20,294	16,449	15,621	12,692	16,603	23,904	14,917	10,599	9,034	2,815	-
Heap Leach	Beginning Ore Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Mined Ore to Heap Leach (kt)	157,454		2,300	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,154	-	-	-	-	-	-	-	-
	Mined Ore - Processed (kt)	157,454		2,300	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,125	9,154	-	-	-	-	-	-	-	-
	Ending Ore Inventory (kt)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Copper Grade (%)	0.036%		0.025%	0.040%	0.046%	0.046%	0.027%	0.037%	0.043%	0.039%	0.035%	0.015%	0.021%	0.031%	0.045%	0.039%	0.039%	0.039%	0.038%	0.038%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
	Gold Grade (g/t)	0.292		0.330	0.445	0.423	0.333	0.284	0.236	0.214	0.344	0.339	0.247	0.239	0.208	0.201	0.279	0.288	0.283	0.292	0.296	-	-	-	-	-	-	-	-
	Silver Grade (g/t)	2.209		1.790	2.800	3.090	2.170	1.760	2.300	1.989	2.354	2.442	2.554	2.190	1.860	1.660	2.008	2.097	2.069	2.142	2.170	-	-	-	-	-	-	-	-
	Contained Copper (klbs)	125,601		1,268	8,047	9,254	9,254	5,432	7,443	8,700	7,914	7,004	3,009	4,225	6,236	9,053	7,784	7,791	7,892	7,692	7,604	-	-	-	-	-	-	-	-
	Contained Gold (kozs)	1,477		24	131	124	98	83	69	63	101	100	73	70	61	59	82	85	83	86	87	-	-	-	-	-	-	-	-
	Contained Silver (kozs)	11,181		132	821	907	637	516	675	584	691	716	749	642	546	487	589	615	607	628	639	-	-	-	-	-	-	-	-
	Recovery Copper (%)	18%		18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	18%	0%	0%	0%	0%	0%	0%	0%	0%
	Recovery Gold (%)	66%		66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	66%	0%	0%	0%	0%	0%	0%	0%	0%
	Recovery Silver (%)	26%		26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	26%	0%	0%	0%	0%	0%	0%	0%	0%
	Copper Precipitate (kt)	17		0	1	1	1	1	1	1	1	1	0	1	1	1	1	1	1	1	1	-	-	-	-	-	-	-	-
	Copper Precipitate Grade	60%		60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	60%	0%	0%	0%	0%	0%	0%	0%	0%
	Recovered Copper (klbs)	22,608		228	1,448	1,666	1,666	978	1,340	1,566	1,425	1,261	542	760	1,123	1,629	1,401	1,402	1,421	1,385	1,369	-	-	-	-	-	-	-	-
	Recovered Gold Dore - Heap Leach (kozs)	975		16	86	82	64	55	46	41	67	66	48	46	40	39	54	56	55	57	57	-	-	-	-	-	-	-	-
	Total Recovered Gold Dore (kozs)	975		16	86	82	64	55	46	41	67	66	48	46	40	39	54	56	55	57	57	-	-	-	-	-	-	-	-
	Recovered Silver Dore (kozs)	2,907		34	214	236	166	134	175	152	180	186	195	167	142	127	153	160	158	163	166	-	-	-	-	-	-	-	-
Payable Metals	Copper Concentrate																												
	Payable Copper (klbs)	3,609,798	-	-	-	-	184,384	282,539	251,886	221,914	182,229	177,525	180,962	181,967	180,269	189,174	194,929	161,725	161,722	169,201	150,146	120,447	120,250	133,898	108,208	93,096	101,222	62,104	-
	Payable Gold (kozs)	4,870	-	-	-	-	314	376	353	310	265	274	269	255	252	228	221	222	194	188	167	156	162	185	138	128	138	74	-
	Payable Silver (kozs)	27,351	-	-	-	-	1,606	1,580	1,387	1,586	1,413	1,328	1,524	1,194	1,336	1,512	1,416	1,380	1,142	1,077	1,205	1,141	1,203	1,070	970	841	951	489	-
	Molybdenum Concentrates																												
	Payable Molybdenum (klbs)	290,597	-	-	-	-	7,602	14,195	17,964	12,323	8,703	11,922	15,626	18,941	20,212	13,748	12,904	14,971	17,250	13,982	13,277	10,788	14,112	20,318	12,679	9,009	7,679	2,393	-
	Gold/Silver Dore																												
	Payable Metal Gold (kozs)	955		16	84	80	63	54	45	41	65	64	47	45	39	38	53	55	54	55	56	-	-	-	-	-	-	-	-
	Payable Metal Silver (kozs)	2,849		34	209	231	162	132	172	149	176	183	191	164	139	124	150	157	155	160	163	-	-	-	-	-	-	-	-
	Copper Precipitate																												
	Payable Metal (klbs)	21,817		220	1,398	1,607	1,607	943	1,293	1,511	1,375	1,217	523	734	1,083	1,572	1,352	1,353	1,371	1,336	1,321	-	-	-	-	-	-	-	-

Income Statement ($000)

	Total	-4	-3	-2	-1	1	2	3	4	5
Copper ($/lb.)	$ 3.16		$ 3.16	$ 3.16	$ 3.16	$ 3.16	$ 3.16	$ 3.16	$ 3.16	$ 3.16
Molybdenum ($/lb)	$ 14.74		14.74	14.74	14.74	14.74	14.74	14.74	14.74	14.74
Gold ($/oz)	$ 1,473.68		1,473.68	1,473.68	1,473.68	1,473.68	1,473.68	1,473.68	1,473.68	1,473.68
Silver ($/oz)	$ 26.32		26.32	26.32	26.32	26.32	26.32	26.32	26.32	26.32
Revenues										
Copper Concentrates - Cu	$ 11,399,362		-	-	-	582,266	892,230	795,428	700,781	575,460
Copper Concentrates - Au	$ 7,177,544		-	-	-	462,694	553,799	520,607	457,011	390,376
Copper Concentrates - Ag	$ 719,775		-	-	-	42,258	41,567	36,510	41,733	37,194
Molybdenum Concentrates	$ 4,282,484		-	-	-	112,028	209,184	264,733	181,609	128,259
Dore' - Gold	$ 1,407,971		23,260	124,440	118,287	93,120	79,418	65,995	59,735	96,145
Dore' - Silver	$ 74,974		888	5,508	6,079	4,269	3,462	4,524	3,914	4,631
Copper Precipitate	$ 68,896		695	4,414	5,076	5,076	2,979	4,083	4,772	4,341
Total Revenues	$ 25,131,006		24,843	134,361	129,442	1,301,711	1,782,639	1,691,881	1,449,556	1,236,406
Operating Cost										
Mining	$ 2,947,482		-	-	-	136,887	148,085	157,705	160,138	157,055
Concentrator	$ 4,955,181		-	-	-	188,830	238,629	230,879	236,722	235,541
Heap Leach - Gold Ore	$ 636,527		9,376	36,886	36,886	36,886	36,886	36,886	36,886	36,886
General Administration	$ 327,186		5,242	5,779	5,779	13,468	14,474	14,474	14,474	14,474
Treatment & Refining Charges										
Copper Concentrates										
Treatment Charges	$ 484,790		-	-	-	24,763	37,945	33,828	29,803	24,473
Copper Refining Charges	$ 299,258		-	-	-	15,286	23,423	20,882	18,397	15,107
Gold Refining Charges	$ 29,972		-	-	-	1,932	2,313	2,174	1,908	1,630
Silver Refining Charges	$ 14,396		-	-	-	845	831	730	835	744
Transportation	$ 951,395		-	-	-	48,596	74,466	66,387	58,487	48,028
Road Maintenance	$ 69,063		-	-	-	3,139	3,139	3,139	3,139	3,139
Molybdenum Concentrate										
Treatment Charges	$ -		-	-	-	-	-	-	-	-
Transportation	$ -		-	-	-	-	-	-	-	-
Gold Dore										
Gold Refining Charges	$ 1,267		21	112	106	84	71	59	54	87
Silver Refining Charges	$ 872		10	64	71	50	40	53	46	54
Transportation	$ 1,207		16	93	99	72	59	69	60	77
Copper Precipitate										
Treatment Charges	$ 1,367		14	88	101	101	59	81	95	86
Refining Charges	$ 1,809		18	116	133	133	78	107	125	114
Transportation	$ 2,934		30	188	216	216	127	174	203	185
Total Cash Operating Cost	$ 10,724,707		14,726	43,326	43,391	471,286	580,625	567,626	561,372	537,679
Royalty	$ 604,561		680	3,484	3,364	31,771	42,490	40,684	35,182	30,446
Property Tax	$ 2,500		100	100	100	100	100	100	100	100
Salvage Value	$ (31,491)		-	-	-	-	-	-	-	-
Reclamation & Closure	$ 125,942		-	-	-	-	-	-	-	-
Total Cash Production Cost	$ 11,426,218		15,506	46,910	46,855	503,157	623,215	608,411	596,654	568,224
Operating Income	$ 13,704,788	$ -	9,337	87,451	82,587	798,554	1,159,423	1,083,470	852,902	668,182
Yukon Mining Royalty	$ 1,357,718		-	6,993	6,395	54,341	98,135	87,904	60,185	38,707
Net Income before Depreciation	$ 12,347,070	$ -	9,337	80,458	76,192	744,212	1,061,289	995,566	792,717	629,475
Capital Cost Depreciation	$ 1,250,812		9,337	80,458	76,192	572,485	387,250	125,090	-	-
Additional Deduction (Class 41A Assets)	$ 1,205,115		-	-	-	168,453	661,392	375,269	-	-
Sustaining Capital Depreciation	$ 361,739				-	3,274	12,646	21,976	19,833	20,358
Total Depreciation	$ 2,817,666		9,337	80,458	76,192	744,212	1,061,289	522,335	19,833	20,358
Net Income After Depreciation	$ 9,529,403		-	-	-	-	-	473,231	772,884	609,118
Tax Loss Carry Forward Applied	$ (30,000)		-	-	-	-	-	(30,000)	-	-
Net Income After Tax Loss Carry Forward	$ 9,499,403		-	-	-	-	-	443,231	772,884	609,118
Taxable Income	$ 9,499,403		-	-	-	-	-	443,231	772,884	609,118
Taxes at 30%	$ 2,878,158		-	-	-	-	-	132,969	231,865	182,735
Net Income After Taxes	$ 6,651,245		-	-	-	-	-	340,262	541,019	426,383
Cash Flow										
Net Income before Depreciation plus Tax Loss Carry F	$ 12,347,070		9,337	80,458	76,192	744,212	1,061,289	995,566	792,717	629,475
Working Capital										
Account Recievable	$ -		(4,084)	(18,003)	809	(192,702)	(79,057)	14,919	39,834	35,038
Accounts Payable	$ -		1,210	2,351	5	35,169	8,987	(1,068)	(514)	(1,947)
Inventory - Parts, Supplies	$ -		(500)	-	(5,800)	-	-	-	500	-
Total Working Capital	$ -		(3,373)	(15,652)	(4,986)	(157,532)	(70,070)	13,851	39,820	33,091
Capital Expenditures										
Initial Capital										
Mine	$ 480,277	$ -	131,417	65,016	104,505	179,339	-	-	-	-
Concentrator	$ 1,792,539	60,762	527,379	723,325	453,492	27,582	-	-	-	-
Heap Leach	$ 138,720	68,720	70,000	-	-	-	-	-	-	-
Owners Cost	$ 44,391	1,776	11,986	18,644	9,766	2,220	-	-	-	-
Sustaining Capital										
Mine	$ 184,081		-	-	-	-	29,193	22,531	1,943	14,526
Process Plant	$ 177,659		-	-	-	13,096	11,569	27,436	11,460	7,406
Total Capital Expenditures	$ 2,817,666	131,258	740,781	806,985	567,763	222,236	40,762	49,966	13,402	21,932
Cash Flow before Taxes	$ 9,529,403	(131,258)	(734,818)	(742,179)	(496,556)	364,444	950,457	959,451	819,135	640,634
Cumulative Cash Flow before Taxes		(131,258)	(866,076)	(1,608,255)	(2,104,811)	(1,740,367)	(789,911)	169,540	988,675	1,629,309
						1.0	1.0	0.8	-	-
Taxes										
Income Taxes	$ 2,878,158	$ -	-	-	-	-	-	132,969	231,865	182,735
Cash Flow after Taxes	$ 6,651,245	(131,258)	(734,818)	(742,179)	(496,556)	364,444	950,457	826,481	587,269	457,899
Cumulative Cash Flow after Taxes		(131,258)	(866,076)	(1,608,255)	(2,104,811)	(1,740,367)	(789,911)	36,571	623,840	1,081,739
						1.0	1.0	1.0	-	-

	6	7	8	9	10	11	12	13	14
Copper ($/lb.)	3.16	3.16	3.16	3.16	3.16	3.16	3.16	3.16	3.16
Molybdenum ($/lb)	14.74	14.74	14.74	14.74	14.74	14.74	14.74	14.74	14.74
Gold ($/oz)	1,473.68	1,473.68	1,473.68	1,473.68	1,473.68	1,473.68	1,473.68	1,473.68	1,473.68
Silver ($/oz)	26.32	26.32	26.32	26.32	26.32	26.32	26.32	26.32	26.32
Revenues									
Copper Concentrates - Cu	560,604	571,461	574,634	569,272	597,391	615,565	510,709	510,700	534,318
Copper Concentrates - Au	404,454	396,668	375,383	371,050	336,331	325,945	326,707	286,444	277,109
Copper Concentrates - Ag	34,938	40,117	31,411	35,150	39,798	37,256	36,319	30,051	28,334
Molybdenum Concentrates	175,687	230,282	279,126	297,855	202,595	190,158	220,623	254,210	206,048
Dore' - Gold	94,885	69,147	66,834	58,165	56,208	77,944	80,634	79,086	81,668
Dore' - Silver	4,803	5,024	4,308	3,659	3,265	3,950	4,125	4,071	4,214
Copper Precipitate	3,842	1,650	2,317	3,421	4,966	4,270	4,274	4,329	4,219
Total Revenues	1,279,213	1,314,348	1,334,014	1,338,572	1,240,554	1,255,087	1,183,392	1,168,891	1,135,909
Operating Cost									
Mining	159,557	167,700	168,580	170,677	162,263	174,492	160,018	162,709	169,993
Concentrator	227,897	227,121	228,587	230,618	237,987	237,577	240,240	236,618	231,124
Heap Leach - Gold Ore	36,886	36,886	36,886	36,886	36,886	36,886	36,886	36,886	36,886
General Administration	14,474	14,474	14,474	14,474	14,474	14,474	14,474	14,474	14,474
Treatment & Refining Charges									
Copper Concentrates									
Treatment Charges	23,841	24,303	24,438	24,210	25,406	26,179	21,719	21,719	22,723
Copper Refining Charges	14,717	15,002	15,085	14,945	15,683	16,160	13,407	13,407	14,027
Gold Refining Charges	1,689	1,656	1,568	1,549	1,404	1,361	1,364	1,196	1,157
Silver Refining Charges	699	802	628	703	796	745	726	601	567
Transportation	46,788	47,694	47,959	47,512	49,858	51,375	42,624	42,623	44,594
Road Maintenance	3,139	3,139	3,139	3,139	3,139	3,139	3,139	3,139	3,139
Molybdenum Concentrate									
Treatment Charges	-	-	-	-	-	-	-	-	-
Transportation	-	-	-	-	-	-	-	-	-
Gold Dore									
Gold Refining Charges	85	62	60	52	51	70	73	71	74
Silver Refining Charges	56	58	50	43	38	46	48	47	49
Transportation	78	75	66	57	51	64	67	66	68
Copper Precipitate									
Treatment Charges	76	33	46	68	99	85	85	86	84
Refining Charges	101	43	61	90	130	112	112	114	111
Transportation	164	70	99	146	211	182	182	184	180
Total Cash Operating Cost	530,247	539,120	541,726	545,167	548,477	562,947	535,165	533,941	539,249
Royalty	31,527	31,997	31,723	31,852	29,363	29,699	28,306	27,954	27,084
Property Tax	100	100	100	100	100	100	100	100	100
Salvage Value	-	-	-	-	-	-	-	-	-
Reclamation & Closure	-	-	-	-	-	-	-	-	-
Total Cash Production Cost	561,874	571,218	573,549	577,119	577,939	592,746	563,570	561,995	566,433
Operating Income	717,339	743,131	760,465	761,452	662,614	662,341	619,822	606,896	569,476
Yukon Mining Royalty	44,484	61,008	90,383	90,939	79,084	78,744	73,693	72,267	67,413
Net Income before Depreciation	672,855	682,123	670,081	670,513	583,531	583,596	546,129	534,629	502,063
Capital Cost Depreciation	-	-	-	-	-	-	-	-	-
Additional Deduction (Class 41A Assets)	-	-	-	-	-	-	-	-	-
Sustaining Capital Depreciation	18,753	22,830	19,850	22,756	23,251	22,673	19,546	16,791	24,601
Total Depreciation	18,753	22,830	19,850	22,756	23,251	22,673	19,546	16,791	24,601
Net Income After Depreciation	654,102	659,293	650,231	647,757	560,279	560,923	526,583	517,838	477,462
Tax Loss Carry Forward Applied	-	-	-	-	-	-	-	-	-
Net Income After Tax Loss Carry Forward	654,102	659,293	650,231	647,757	560,279	560,923	526,583	517,838	477,462
Taxable Income	654,102	659,293	650,231	647,757	560,279	560,923	526,583	517,838	477,462
Taxes at 30%	196,231	197,788	195,069	194,327	168,084	168,277	157,975	155,351	143,239
Net Income After Taxes	457,871	461,505	455,162	453,430	392,196	392,646	368,608	362,487	334,223
Cash Flow									
Net Income before Depreciation plus Tax Loss Carry F	672,855	682,123	670,081	670,513	583,531	583,596	546,129	534,629	502,063
Working Capital									
Account Recievable	(7,037)	(5,776)	(3,233)	(749)	16,113	(2,389)	11,785	2,384	5,422
Accounts Payable	(611)	729	214	283	272	1,189	(2,284)	(101)	436
Inventory - Parts, Supplies	-	-	-	-	-	-	-	-	-
Total Working Capital	(7,648)	(5,046)	(3,019)	(466)	16,384	(1,200)	9,502	2,283	5,858
Capital Expenditures									
Initial Capital									
Mine	-	-	-	-	-	-	-	-	-
Concentrator	-	-	-	-	-	-	-	-	-
Heap Leach	-	-	-	-	-	-	-	-	-
Owners Cost	-	-	-	-	-	-	-	-	-
Sustaining Capital									
Mine	-	8,369	-	13,774	11,344	18,007	-	6,663	36,703
Process Plant	13,941	26,689	5,669	6,620	9,707	17,574	6,748	5,064	2,870
Total Capital Expenditures	13,941	35,058	5,669	20,394	21,051	35,580	6,748	11,727	39,574
Cash Flow before Taxes	651,267	642,018	661,394	649,653	578,864	546,816	548,883	525,185	468,347
Cumulative Cash Flow before Taxes	2,280,575	2,922,594	3,583,987	4,233,641	4,812,505	5,359,321	5,908,205	6,433,390	6,901,737
	-	-	-	-	-	-	-	-	-
Taxes									
Income Taxes	196,231	197,788	195,069	194,327	168,084	168,277	157,975	155,351	143,239
Cash Flow after Taxes	455,036	444,230	466,324	455,326	410,780	378,539	390,908	369,834	325,108
Cumulative Cash Flow after Taxes	1,536,775	1,981,005	2,447,330	2,902,656	3,313,436	3,691,976	4,082,884	4,452,718	4,777,826
	-	-	-	-	-	-	-	-	-

	15	16	17	18	19	20	21	22	23
Copper ($/lb.)	3.16	3.16	3.16	3.16	3.16	3.16	3.16	3.16	-
Molybdenum ($/lb)	14.74	14.74	14.74	14.74	14.74	14.74	14.74	14.74	-
Gold ($/oz)	1,473.68	1,473.68	1,473.68	1,473.68	1,473.68	1,473.68	1,473.68	1,473.68	-
Silver ($/oz)	26.32	26.32	26.32	26.32	26.32	26.32	26.32	26.32	-
Revenues									
Copper Concentrates - Cu	474,146	380,360	379,735	422,837	341,711	293,987	319,649	196,117	-
Copper Concentrates - Au	246,732	229,909	238,885	272,629	203,755	189,084	203,161	108,809	-
Copper Concentrates - Ag	31,723	30,033	31,668	28,171	25,526	22,123	25,022	12,873	-
Molybdenum Concentrates	195,668	158,987	207,970	299,428	186,853	132,764	113,159	35,258	-
Dore' - Gold	83,002	-	-	-	-	-	-	-	-
Dore' - Silver	4,282	-	-	-	-	-	-	-	-
Copper Precipitate	4,171	-	-	-	-	-	-	-	-
Total Revenues	1,039,724	799,289	858,258	1,023,065	757,845	637,959	660,991	353,057	-
Operating Cost									
Mining	154,427	142,980	121,961	99,288	74,930	40,353	39,142	18,542	-
Concentrator	229,160	225,974	224,927	225,263	229,169	231,029	230,192	131,096	-
Heap Leach - Gold Ore	36,977	-	-	-	-	-	-	-	-
General Administration	14,476	13,792	13,792	13,792	13,792	13,792	13,792	11,536	-
Treatment & Refining Charges									
Copper Concentrates									
Treatment Charges	20,164	16,176	16,149	17,982	14,532	12,503	13,594	8,340	-
Copper Refining Charges	12,447	9,985	9,969	11,100	8,971	7,718	8,391	5,148	-
Gold Refining Charges	1,030	960	998	1,138	851	790	848	454	-
Silver Refining Charges	634	601	633	563	511	442	500	257	-
Transportation	39,572	31,745	31,693	35,290	28,519	24,536	26,678	16,368	-
Road Maintenance	3,139	3,139	3,139	3,139	3,139	3,139	3,139	3,139	-
Molybdenum Concentrate									
Treatment Charges	-	-	-	-	-	-	-	-	-
Transportation	-	-	-	-	-	-	-	-	-
Gold Dore									
Gold Refining Charges	75	-	-	-	-	-	-	-	-
Silver Refining Charges	50	-	-	-	-	-	-	-	-
Transportation	70	-	-	-	-	-	-	-	-
Copper Precipitate									
Treatment Charges	83	-	-	-	-	-	-	-	-
Refining Charges	109	-	-	-	-	-	-	-	-
Transportation	178	-	-	-	-	-	-	-	-
Total Cash Operating Cost	512,591	445,351	423,261	407,557	374,415	334,302	336,277	194,883	-
Royalty	24,918	19,290	20,634	24,375	18,193	15,352	15,792	8,400	-
Property Tax	100	100	100	100	100	100	100	100	-
Salvage Value	-	-	-	-	-	-	-	(31,491)	-
Reclamation & Closure	-	-	-	-	-	-	-	31,485	31,485
Total Cash Production Cost	537,609	464,741	443,995	432,032	392,708	349,754	352,170	203,377	31,485
Operating Income	502,115	334,547	414,263	591,033	365,137	288,205	308,822	149,680	(31,485)
Yukon Mining Royalty	59,210	38,367	48,497	70,617	42,899	33,704	36,715	17,034	-
Net Income before Depreciation	442,904	296,181	365,766	520,417	322,238	254,501	272,106	132,646	(31,485)
Capital Cost Depreciation	-	-	-	-	-	-	-	-	-
Additional Deduction (Class 41A Assets)	-	-	-	-	-	-	-	-	-
Sustaining Capital Depreciation	16,278	19,680	15,748	13,342	7,354	6,111	7,447	6,642	-
Total Depreciation	16,278	19,680	15,748	13,342	7,354	6,111	7,447	6,642	-
Net Income After Depreciation	426,627	276,501	350,018	507,075	314,884	248,390	264,659	126,004	(31,485)
Tax Loss Carry Forward Applied	-	-	-	-	-	-	-	-	-
Net Income After Tax Loss Carry Forward	426,627	276,501	350,018	507,075	314,884	248,390	264,659	126,004	(31,485)
Taxable Income	426,627	276,501	350,018	507,075	314,884	248,390	264,659	126,004	(31,485)
Taxes at 30%	127,988	82,950	105,005	152,122	94,465	74,517	79,398	37,801	-
Net Income After Taxes	298,639	193,551	245,012	354,952	220,419	173,873	185,261	88,203	(31,485)
Cash Flow									
Net Income before Depreciation plus Tax Loss Carry F	442,904	296,181	365,766	520,417	322,238	254,501	272,106	132,646	(31,485)
Working Capital									
Account Recievable	15,811	39,524	(9,694)	(27,092)	43,598	19,707	(3,786)	50,619	58,037
Accounts Payable	(2,191)	(5,527)	(1,816)	(1,291)	(2,724)	(3,297)	162	(11,621)	(16,018)
Inventory - Parts, Supplies	-	-	-	-	-	-	-	5,800	-
Total Working Capital	13,620	33,997	(11,509)	(28,382)	40,874	16,410	(3,624)	44,798	42,019
Capital Expenditures									
Initial Capital									
Mine	-	-	-	-	-	-	-	-	-
Concentrator	-	-	-	-	-	-	-	-	-
Heap Leach	-	-	-	-	-	-	-	-	-
Owners Cost	-	-	-	-	-	-	-	-	-
Sustaining Capital									
Mine	-	21,028	-	-	-	-	-	-	-
Process Plant	3,077	2,961	3,690	304	937	389	304	147	-
Total Capital Expenditures	3,077	23,990	3,690	304	937	389	304	147	-
Cash Flow before Taxes	453,447	306,188	350,566	491,730	362,175	270,522	268,178	177,297	10,534
Cumulative Cash Flow before Taxes	7,355,184	7,661,372	8,011,938	8,503,668	8,865,843	9,136,365	9,404,544	9,581,840	9,592,374
	-	-	-	-	-	-	-	-	-
Taxes									
Income Taxes	127,988	82,950	105,005	152,122	94,465	74,517	79,398	37,801	-
Cash Flow after Taxes	325,459	223,237	245,561	339,607	267,710	196,005	188,781	139,495	10,534
Cumulative Cash Flow after Taxes	5,103,285	5,326,523	5,572,084	5,911,691	6,179,401	6,375,406	6,564,187	6,703,682	6,714,216
	-	-	-	-	-	-	-	-	-

Economic Indicators before Taxes		
NPV @ 0%	0%	$ 9,529,403
NPV @ 5%	5%	$ 4,428,256
NPV @ 8%	8%	$ 2,824,144
NPV @ 10%	10%	$ 2,083,172
IRR		24.0%
Payback		2.8

Economic Indicators after Taxes		
NPV @ 0%	0%	$ 6,651,245
NPV @ 5%	5%	$ 2,986,189
NPV @ 8%	8%	$ 1,830,180
NPV @ 10%	10%	$ 1,295,677
IRR		20.1%
Payback	Years	3.0
NSR $/Sulphide Ore		$ 22.59

23 ADJACENT PROPERTIES

Numerous quartz mineral claim blocks registered to other owners are staked adjacent to and in the general vicinity of CMC's 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. 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 CMC'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 WCGC 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.

23.1 CARMACKS PROJECT

Copper North Mining Corporation is actively developing the oxide copper deposit at Carmacks located about 115 km ESE of the Casino deposit and 200 km NW of Whitehorse. Carmacks will be open pit mining with crushing, heap leaching, and copper recovery by solvent extraction/ electrowinning to produce about 14,500 t/y of cathode copper over eight years.

23.2 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.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT EXECUTION PLAN

The Project Execution Plan describes, at a high level, how the project will be carried out. This plan contains an overall description of what the main work focuses are, project organization, the estimated schedule, and where important aspects of the project will be carried out. The plan is to start operations with the Heap Leach Facility, to be followed by the concentrator. Key milestones include the following:

- Full Notice to Proceed and construction start-up – first quarter 2016
- Heap Leach operation start-up – fourth quarter 2017
- Concentrator start-up – fourth quarter 2019
- Commercial Production – first quarter 2020

24.1.1 Focus

The proposed project execution plan incorporates an integrated strategy for engineering, procurement and construction management (EPCM). The primary objective of the execution methodology is to deliver the project at the lowest capital cost, on schedule, and consistent with the project standards for quality, safety, and environmental compliance.

The majority of mechanical and electrical equipment required for the project will be procured within North America. Concrete, building construction materials and timber products will be sourced primarily in the Yukon. Structural and miscellaneous steel, piping, tanks, electrical and miscellaneous process equipment will be largely sourced within Canada, and to the extent practical, within the region. Some commodities, such as structural steel, may be sourced out of country.

24.1.2 Engineering

The project will enter the basic engineering phase in the fourth quarter of 2013. Basic Engineering would be followed by Detailed Engineering in the third quarter of 2014. It is particularly important to identify priorities for long lead procurement and priorities for early construction that support the overall construction schedule. Engineering must be completed to the point that key procurement and construction activities have been decided contractually prior to the project's Notice to Proceed. Some funding may need to be committed to achieve this status.

24.1.3 Procurement

Equipment and bulk material suppliers will be selected via a competitive bidding process. Similarly, construction contractors will be selected through a pre-qualification process followed by a competitive bidding process. The project will employ a combination of lump sum and unit price contracts as appropriate for the level of engineering and scope definition available at the time contracts are awarded.

Procurement of long lead equipment and materials will be scheduled with their relevant engineering tasks. This will ensure that the applicable vendor information is incorporated into the design drawings and that the equipment will be delivered to site at the appropriate time, as well as support the overall project schedule. Particular emphasis will be placed on procuring the material and contract services required to establish the temporary construction infrastructure required for the construction program.

As mentioned before, it is the objective of WCGC and M3 to have the construction documents completed for bidding, bids offered and received, and contractors accepted and prepared to begin work well before the "Full Notice to Proceed." Contractors will be selected, and a hold will be put on their contracts awaiting the release of funds and the notice to proceed. January 25, 2016 is the anticipated "Full Notice to Proceed."

24.1.4 Project Services

The EPCM contractor will be responsible for management and control of the various project activities and will ensure that the team has appropriate resources to accomplish WCGC's objectives. For example, M3 has a completely integrated project services system from construction documents through procurement, cost control, project accounts, warehousing, and start-up.

24.1.5 Construction Management

The construction program is scheduled to start after the full notice to proceed, identified at this time as 25 January 2016. The completion of one lane of the all-weather road will be of the highest priority. It will necessary to get the large earth-moving equipment onto the site to begin major work. This work will include the clearing and grubbing of the airfield and approaches, the access road to the airfield, the mine site including the main plant building pad, the heap leach pad, the north and south access roads within the mine site, and the Tailings Management Facility. In particular, the clearing of timber will begin in the first quarter of 2016. Completion of an all-weather road will allow mine pre-stripping and clearing and grubbing for heap leach construction to begin in the second quarter of 2016.

Erection of the construction camp will be divided into two phases. The first phase, scheduled to start in June 2016, will be the relocation of the existing camp from the new mill plant site and the construction of the new pioneer camp, which will include three Worker's dorms, one Supervisor's dorm, and a kitchen/diner/recreation unit for approximately 264 personnel. The first phase should be completed by September 2016. The second phase will begin the following construction season with further site preparation and construction of the foundations. The second phase will expand the construction camp by approximately 684 personnel for a total of approximately 948 personnel. It will include seven additional Worker's dorms, one additional Supervisor's dorm, and two new Executive dorms. It will also include additional kitchen/dining facilities, and recreation facilities. The entire construction camp will be completed in May 2018.

Though the concrete plant is not expected to be able to be transported to the site until 2017, it is anticipated that sand and aggregate, as well as cement if appropriate storage can be assured, can be staged in 2016 by utilizing the limited access road from Carmacks as well as barges during the four months that barge traffic is available (June, July, August, and September). Additionally, it is anticipated that a low-volume batch or portable concrete mixer will be utilized in 2016 to facilitate the construction schedule.

Processing construction will begin with the Oxide Primary Crushing Building and the ADR/SART Facility in in the second quarter of 2017. Processing construction will finish with full commercial production of the concentrator in the first quarter of 2020. The objective is to have sufficient structures enclosed by the third quarter of 2017 so that mechanical and electrical work can continue during the colder months.

24.1.6 Contracting

Contracting is an integral function in the project's overall execution. Contracting for the Casino Project will be done in full accord with the provisions of the WCGC EPCM contract.

A combination of vertical, horizontal, and design construction contracts will be employed as best suits the work to be performed, and as best suits the degree of engineering and scope definition available at the time of award. A site-installed concrete batch plant will supply concrete to all construction contractors. The Owner-furnished construction camp will be utilized by all construction contractors. Camp operations will be provided by the EPCM through 2017, and will be taken over by WCGC-contracted service providers in 2018 when production starts. Early earthwork contractors will be expected to provide their own camps, as they will be on site prior to the erection of the site camp.

The mass earthwork contract will cover all mine pre-stripping, clearing, grubbing, bulk excavation, and leach pad preparation. The estimate assumes a single contractor performs this work, to achieve economies of scale, and the elimination of interfacing issues. The contractor will require only one major mobilization for all work.

24.1.7 Labour

The labour market in northwestern Canada at this time is scarce. Many contractors in the Yukon are open shop. Construction labour will likely need to be imported from major population centers by air, including possibly Alaska.

24.1.8 Construction Completion and Turn-over Procedure

The Construction Completion Procedure is part of the Construction Quality Plan. Contractors are to enter into contractual agreements with WCGC to perform certain portions of the work, which includes quality control of their work. Facilities will be verified and accepted in a stepwise documented process of mechanical completion and pre-operational testing. The main steps are as follows:

- Mechanical completion of components,
- Pre-commissioning of instrumentation,
- Pre-operational testing of overall systems,
- Start-up by Owner, and

- Full commercial production.

24.1.9 Quality Plan

A project-specific Quality Plan will be developed and implemented. The Quality Plan is a management tool for the EPCM contractor, through the construction contractors, to maintain the quality of construction and installation on every aspect of a project. The plan, which consists of many different manuals and subcategories, will be developed during the engineering phase and available prior to the start of construction. The Quality Plan ensures compliance with various technical and accounting activities that will take place.

24.1.10 Health and Safety Plan

The Health and Safety Plan (HASP) will be established for the construction of the Casino Project and any other authorized work at the project site. The HASP covers both contractor personnel and operational personnel working at the project, and any on any other authorized work for the project.

The HASP specifies regulatory compliance requirements, training, certifications and medical requirements necessary for Contractors to complete the project. Along with the Operations Procedures, the HASP is to be followed by all Contractor personnel working at the site.

24.1.11 Camp Transition

The camp is being built as a construction camp. However, it is expected that this camp will also become the permanent operations personnel camp. The camp usage will transition from construction to operations during the latter stages of construction prior to start-up.

24.1.12 Project Schedule

At the present time, the study has developed a sequence of efforts that should be followed as well as an estimated schedule through which the project will likely proceed. The schedule is shown in Figure 24-1.

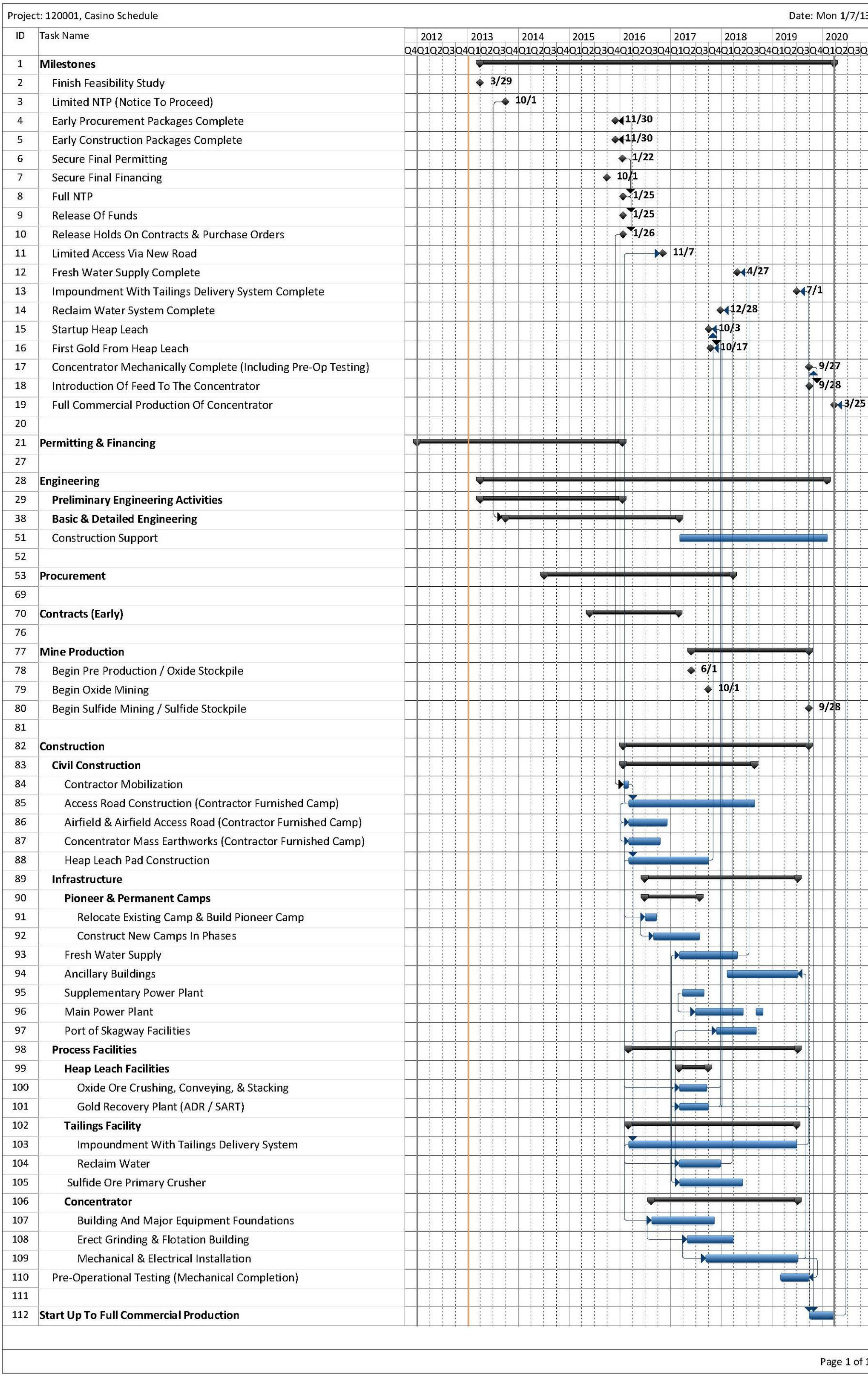


Figure 24-1: Overall Project Schedule

24.2 GEOTECHNICAL STUDY

Knight Piésold Ltd. (KPL) conducted an updated foundation and geotechnical assessment for the proposed plant site at the Casino Project, a copper-gold mine located in west central Yukon, Canada. This report is the result of previous geotechnical site investigation programs and laboratory testing carried out in 1993, 1994 and 2010 by KPL.

24.3 RECLAMATION

Preliminary closure requirements for the waste storage areas will include on-going monitoring of surface and groundwater quality and flow rates, and periodic inspection of the waste rock pile, as required.

Reclamation will be carried out in conjunction with on-going environmental monitoring to ensure that sediment control and water quality objectives are met. The final waste rock bench crests will be rounded to provide long-term stability. The bench tops of the final waste rock pile will be covered with a suitable topsoil layer and re-vegetated.

24.4 TAILING FACILITY SITE GEOTECHNICAL CONDITIONS

Several geotechnical site investigation programs were conducted by KPL in the area of the TMF from 1993 to 1994 and recently in 2010. The programs included drillholes and test pits to investigate the geotechnical characteristics and foundation conditions, and to evaluate the geological factors affecting design of a TMF. The geotechnical data has been used to evaluate the tailings basin and embankment foundation conditions.

The site investigations conducted in the area of the proposed TMF site included the following:

- Preliminary surficial geotechnical investigations were carried out at the Casino Project during the period of June 4 to October 1, 1993. The investigations were carried out as part of the Placer Lease Agreement Program to provide a preliminary overview of the surficial materials and foundation conditions for the various potential mine-site components, including the Open Pit, Plant site, Heap Leach Pad, Waste Rock Storage and Tailings Impoundment sites. The investigation program included the excavation of geotechnical trenches for visual description and characterization of surficial materials, and for the collection of representative samples for laboratory testwork. The laboratory testwork was carried out on-site by KPL and included particle-size analyses, natural moisture contents, Atterberg limits, compaction tests and permeability tests. The results of the investigations are included in KPL report “Report on Preliminary Surficial Geotechnical Investigations” (Ref. No. 1831/1, March, 1994).

- Geotechnical and hydrogeological investigations conducted in 1994 resulted in 15 drillholes for visual inspection and characterization of surficial materials. Representative rock core and test pit samples were collected for laboratory testwork. The laboratory test work included particle-size analyses, natural moisture contents, Atterberg limits, and Proctor compaction tests. In situ packer and falling head permeability tests and shut-in pressure tests were conducted within the drillholes and groundwater monitoring wells and thermistors were installed. Point load testing was conducted on site by KPL for select rock core samples. The results of the investigations are included in KPL report "Data Compilation Report on 1994 Geotechnical and Hydrogeotechnical Investigations" (Ref. No. 1832/2, February 22, 1995).

- Geotechnical and hydrogeological investigations conducted in 2010 resulted in nine drillholes throughout the project area. The investigation also included the excavation of test pits for visual inspection and characterization of surficial materials. Representative test pit soil samples were collected for laboratory test work. The laboratory testwork included natural moisture content, particle size distribution, hydrometer analysis, Atterberg limits, specific gravity, standard proctor compaction, triaxial shear, and flexible wall permeability tests. In situ packer and falling head permeability tests were conducted within the drillholes and groundwater monitoring wells were installed. The results of the investigations are included in KPL report "2010 Geotechnical Site Investigation Data Report" (Ref. No. VA101-325/3-4, November 2, 2010).

25 INTERPRETATION AND CONCLUSIONS

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

25.1 OPPORTUNITIES

Opportunities exist to enhance the project economics including:

- Conversion of some of the inferred resource into measured and indicated.
- Sharing of infrastructure development costs with other parties.
- Optimize the process during the basic and detailed engineering phases.

25.2 RISKS

Extreme cold weather can be a problem, but it is an anticipated problem with established solutions available.

This report and its costing have utilized prevailing rates based on available research on contractors in the Yukon. Although much of the procurement is international in nature, some materials and all labor are anticipated to be supplied from North America. The various rates in the estimates have reflected the cost impact. As such, the project becomes more sensitive to global economics than local economics. Metal pricing becomes the key sensitivity factor for project financial success.

26 RECOMMENDATIONS

To further enhance the project, M3 recommends that CMC perform the following:

- CMC 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 (approximately $2 million required).

- CMC should continue with the environmental studies and permitting efforts now underway (approximately $5 million required).

- CMC should continue with the engineering effort in support of permitting (approximately $1 million required).

- CMC should continue to monitor developments in the Yukon, northern British Columbia, and Alaska to be in a position to share infrastructure development (approximately $200,000 required).

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APPENDIX A: FEASIBILITY STUDY CONTRIBUTORS AND PROFESSIONAL QUALIFICATIONS

CERTIFICATE of QUALIFIED PERSON

Conrad E. Huss

I, Conrad E. Huss, P.E., Ph.D., do hereby certify that:

1. I am Senior Vice President and Chairman of the Board of:

   M3 Engineering & Technology Corporation
   2051 W. Sunset Rd., Suite 101
   Tucson, Arizona 85704
   U.S.A.

2. I graduated with a Bachelor's of Science in Mathematics and a Bachelor's of Art in English from the University of Illinois in 1963. I graduated with a Master's of Science in Engineering Mechanics from the University of Arizona in 1968. In addition, I earned a Doctor of Philosophy in Engineering Mechanics from the University of Arizona in 1970.

3. I am a Professional Engineer in good standing in the State of Arizona in the areas of Civil (No. 9648) and Structural (No. 9733) engineering. I am also registered as a professional engineer in the States of California, Illinois, Maine, Minnesota, Missouri, Montana, New Mexico, Oklahoma, Texas, Utah, and Wyoming.

4. I have worked as an engineer for a total of forty three years since my graduation from the University of Illinois. I have taught at the University level part-time for five years and as an assistant professor for one year.

5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am the principal author for the preparation of the technical report titled "Casino Project, Form 43-101F1 Technical Report, Feasibility Study, Yukon, Canada" (the "Technical Report"), dated January 25, 2013, prepared for Western Copper and Gold Corporation; and am responsible for Sections 1 through 6, 13, 17, 18, 19, and 21 through 27. I have visited the project site on 12 June 2012.

7. 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 required to be disclosed to make the Technical Report not misleading.

8. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 25 day of January, 2013.

“Signed and Sealed”
Signature of Qualified Person

Registered Professional Engineer
CERTIFICATE NO.
9648
CONRAD
E. HUSS
25 JAN 2013

Conrad E. Huss, P.E., Ph.D.
Print Name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

Thomas L. Drielick

I, Thomas L. Drielick, P.E., do hereby certify that:

1. I am currently employed as Sr. Vice President by:

   M3 Engineering & Technology Corporation
   2051 W. Sunset Road, Ste. 101
   Tucson, Arizona 85704
   U.S.A.

2. I am a graduate of Michigan Technological University and received a Bachelor of Science degree in Metallurgical Engineering in 1970. I am also a graduate of Southern Illinois University and received an M.B.A. degree in 1973.

3. I am a:

   - Registered Professional Engineer in the State of Arizona (No. 22958)
   - Registered Professional Engineer in the State of Michigan (No. 6201055633)
   - Member in good standing of the Society for Mining, Metallurgy and Exploration, Inc. (No. 850920)

4. I have practiced metallurgical and mineral processing engineering and project management for 41 years. I have worked for mining and exploration companies for 18 years and for M3 Engineering and Technology, Corporation for 23 years.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of Section 13 "Mineral Processing and Metallurgical Testing" and Section 17 "Recovery Methods", of the technical report titled "Casino Project, Form 43-101F1 Technical Report, Feasibility Study, Yukon, Canada"," dated January 25, 2013 (the "Technical Report").

7. 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 required to be disclosed to make the report not misleading.

8. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 25 day of January, 2013.

Signature of Qualified Person

Thomas L. Drielick
Print name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Jeffrey B. Austin, P.Eng.

I, Jeffrey B. Austin, P.Eng., do hereby certify that:

1. I am a Consulting Engineer and President of International Metallurgical and Environmental Inc., residing at 906 Fairway Crescent, Kelowna, B.C., Canada.

2. This certificate applies to the technical report titled "Casino Project, Form 43-101 Technical Report, Feasibility Study, Yukon, Canada", dated January 25, 2013 (the "Technical Report").

3. I fulfill the requirements of a qualified person for the purposes of NI 43-101 based on my academic qualifications, professional membership and relevant experience, as set out below:

   a. I hold the following academic qualifications:

BASc.	University of British Columbia	1984

   b. I am a member in good standing of the following professional and technical associations:

Association of Professional Engineers and Geoscientists of BC	15708

   c. I have worked in the minerals industry as a Consulting Process Engineer continuously since 1987, a period of 26 years.

4. I have not personally inspected the property.

5. I am responsible for section 13 of the Technical Report.

6. I am independent of Western Copper and Gold Corp. and Casino Mining Corp. as defined in section 1.5 of NI 43-101.

7. My prior involvement with the property includes metallurgical test work management for the Hunter Dickenson Group, the previous owners and developers of the project.

8. I have read and am familiar with NI 43-101 and the sections of the Technical Report for which I am responsible. To the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible have been prepared in compliance with 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 the Technical Report not misleading.

Dated this 25th day of January, 2013

"Signed and Sealed"

____

Jeffrey B. Austin, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

[Gary H. Giroux]

I, Gary H. Giroux of 982 Broadview Drive, North Vancouver, B.C., do hereby certify that:

1. I am a consulting geological engineer with an office at #1215 - 675 West Hastings Street, Vancouver, British Columbia.

2. This certificate applies to the technical report titled "Casino Project, Form 43-101 Technical Report, Feasibility Study, Yukon, Canada", dated January 25, 2013 (the "Technical Report").

3. I fulfill the requirements of a qualified person for the purposes of NI 43-101 based on my academic qualifications, professional membership and relevant experience, as set out below:

   a. I hold the following academic qualifications: [*Insert additional rows as required*]

B.A.Sc. – Geological Engineering	University of British Columbia	1970
M.A.Sc. – Geological Engineering	University of British Columbia	1984

   b. I am a member in good standing of the following professional and technical associations:

Association of Professional Engineers and Geoscientist of B.C.	8814
Professional Engineers and Geoscientist Newfoundland & Lab.	06651

   c. 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.

4. I have not visited the property.

5. I am responsible for section (14) (Mineral Resource Estimate) of the Technical Report.

6. I am independent of Western Copper and Gold Corp. and Casino Mining Corp. as defined in section 1.5 of NI 43-101.

7. My prior involvement with the property includes: a resource estimate on Casino for Pacific Sentinel Gold Corp. in 1995.

8. I have read and am familiar with NI 43-101 and the sections of the Technical Report for which I am responsible. To the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible have been prepared in compliance with 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 the Technical Report not misleading.

Dated this 25th day of January 2013

G H Giroux

Gary H. Giroux, P.Eng. MASc.

PROFESSIONAL
PROVINCE OF
G. H. GIROUX
BRITISH COLUMBIA
ENGINEER

CERTIFICATE OF QUALIFIED PERSON

[Name of Qualified Person]

I, Scott Casselman do hereby certify that:

1. I am geologist with Casselman Geological Services Ltd, of 33 Firth Road, Whitehorse, Yukon.

2. This certificate applies to the technical report titled "Casino Project, Form 43-101 Technical Report, Feasibility Study, Yukon, Canada", dated January 25, 2013 (the "Technical Report").

3. I fulfill the requirements of a qualified person for the purposes of NI 43-101 based on my academic qualifications, professional membership and relevant experience, as set out below:

   a. I hold the following academic qualifications:

Bachelor of Science, Geology	Carleton University, Ottawa, Ontario	1985

   b. I am a member in good standing of the following professional and technical associations:

Association of Professional Engineers and Geoscientists of BC	Member No. 20032

   c. I have worked in the minerals industry as a geologist continuously since 1985, a period of 27 years.

4. I have most recently inspected the property on May 18, 2012 for two (2) weeks.

5. I am responsible for section(s) 4, 6, 7, 8, 9, 10, 11, 12, and 14 of the Technical Report.

6. I am independent of Western Copper and Gold Corp. and Casino Mining Corp. as defined in section 1.5 of NI 43-101.

7. My prior involvement with the property includes: Managing geological exploration on the site, including exploration drilling programs in 2008, 2009, 2010 and 2011.

8. I have read and am familiar with NI 43-101 and the sections of the Technical Report for which I am responsible. To the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible have been prepared in compliance with 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 the Technical Report not misleading.

Dated this 25 day of January, 2013

S Cassel

Scott Casselman
B.Sc., P.Geo.

CERTIFICATE OF QUALIFIED PERSON

Graham Rowland Greenaway P.Eng.

I, Graham Rowland Greenaway do hereby certify that:

1. I am a Specialist Geotechnical Engineer / Project Manager with Knight Piesold Ltd., Suite 1400 – 750 West Pender Street, Vancouver, B.C., V6C 2T8.

2. This certificate applies to the technical report titled "Casino Project, Form 43-101 Technical Report, Feasibility Study, Yukon, Canada", dated Friday January 25th, 2013 (the "Technical Report").

3. I fulfill the requirements of a qualified person for the purposes of NI 43-101 based on my academic qualifications, professional membership and relevant experience, as set out below:

   a. I hold the following academic qualifications:

B.Eng. Civil Engineering (Honours)	University of Sheffield, U.K.	1987
M.Sc. Earthquake Engineering	University of London, Imperial College, U.K.	1988

   b. I am a member in good standing of the following professional and technical associations:

Association of Professional Engineers of Yukon (APEY)	Registration No. 1757
Association of Professional Engineers and Geoscientists of British Columbia (APEGBC)	Registration No. 20948
Canadian Geotechnical Society (CGS)	Membership No. 062671
American Society of Civil Engineers (ASCE)	Member ID 430964
Earthquake Engineering Research Institute (EERI)	Member ID 11946

   c. I have worked in the minerals industry as an Engineering Consultant continuously since 1991, a period of 21 years. Work experience has included numerous mine waste management studies, tailings dam and heap leach designs (varying from conceptual design level to detailed design) as well as studies and reviews for operating mine facilities. Specialisations include development of mine waste and water management strategies, tailings and waste rock characterisation, consolidation and seepage assessments, tailings embankment and leach pad stability analyses, and seismic risk analyses.

4. I have not personally inspected the property

5. I am responsible for sections 18.5 (Tailings Management Facility), 18.6 (Waste Rock Storage and Stockpiles), 18.7 (Leach Pad) of the Technical Report.

6. I am independent of Western Copper and Gold Corp. and Casino Mining Corp. as defined in section 1.5 of NI 43-101.

7. My prior involvement with the property includes:

   - Pre-Feasibility Design studies carried out in 2008 (Tailings Management Facility, Waste Storage Area and Stockpiles and Leach Pad)

   - Revised Pre-Feasibility Design studies in 2011 (Tailings Management Facility, Waste Storage Area and Stockpiles and Leach Pad)

8. I have read and am familiar with NI 43-101 and the sections of the Technical Report for which I am responsible. To the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible have been prepared in compliance with 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 the Technical Report not misleading.

Dated this 25th day of January, 2013

PROFESSIONAL
YUKON
GRAHAM R. GREENAWAY
TERRITORY
ENGINEER

Graham R Greenaway P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Michael G. Hester

I, Michael G. Hester, do hereby certify that:

1. I am currently employed as Vice President and Principal Mining Engineer by Independent Mining Consultants, Inc. (IMC) of 3560 E. Gas Road, Tucson, Arizona, 85714, USA, phone number (520) 294-9861.

2. This certificate applies to the Technical Report titled "Casino Project, Form 43-101 Technical Report, Feasibility Study, Yukon, Canada" (the "Technical Report"), dated January 25, 2013.

3. I fulfill the requirements of a "Qualified Person" for the purposes of NI 43-101 based on my academic qualifications, professional membership, and relevant experience.

4. I hold the following academic qualifications:

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

5. 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)

6. I have worked in the minerals industry as an engineer continuously since 1979, a period of 33 years. 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.

7. I have most recently inspected the property on July 22, 2008 for a period of one day.

8. I am responsible for Section 15, Mineral Reserve Estimates, and Section 16, Mining Methods.

9. I am independent of Western Copper and Gold Corp. and Casino Mining Corp. as defined in Section 1.5 of NI 43-101.

10. My prior involvement with the property includes work on the Preliminary Feasibility Studies conducted by Western Copper Corporation dated April 2011 and August 2008. I also worked on the mining portion of a Scoping Study for Casino for Pacific Sentinel Gold Corporation dated September 1995.

11. I have read and am familiar with NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with the instrument.

12. 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.

13. 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 25th day of January 2013, at Tucson, Arizona.

Michael G. Hester, FAusIMM

CERTIFICATE OF QUALIFIED PERSON

Jesse Lynn Duke P. Geo.

I, Jesse Lynn Duke do hereby certify that:

1. I am President of Ibex Environmental Consulting Inc. P.O. Box 40165, Mile 932 Old Alaska Highway, Whitehorse, Yukon, Y1A 6M9.

2. This certificate applies to the technical report titled "Casino Project, Form 43-101 Technical Report, Feasibility Study, Yukon, Canada", dated January 25, 2013 (the "Technical Report").

3. I fulfill the requirements of a qualified person for the purposes of NI 43-101 based on my academic qualifications, professional membership and relevant experience, as set out below:

   a. I hold the following academic qualifications:

B.Sc. Geology	University of Alaska	1986

   b. I am a member in good standing of the following professional and technical associations:

Association of Professional Engineers and Geoscientists of British Columbia	20417

   c. I have worked in the minerals industry as a professional geologist continuously since 1986 a period of 27 years including 10 years as a consultant specializing in the field of environment, government relations and permitting continuously since 2003.

4. I have most recently inspected the property on September 5, 2012 for 1 day.

5. I am responsible for section 20 of the Technical Report.

6. I am independent of Western Copper and Gold Corp. and Casino Mining Corp. as defined in section 1.5 of NI 43-101.

7. My prior involvement with the property includes: Qualified Person for the environmental component of the *Casino Project Technical Report Pre-Feasibility Study Update. Yukon Territory, Canada*, released in May, 2011 and *Casino Project Pre-Feasibility Study, Yukon Territory, Canada*, released in 2008.

8. I have read and am familiar with NI 43-101 and the sections of the Technical Report for which I am responsible. To the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible have been prepared in compliance with 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 the Technical Report not misleading.

Dated this 25 day of January, 2013

Jesse L. Duke P. Geo.

APPENDIX B: LIST OF CLAIMS

Table 1.26-1: Casino Property – List of Active and Pending Mineral Claims

Grant Number	Claim Name	Staking Date	Expiry Date
4252	HELICOPTER	9/4/1943	3/25/2024
56979	#1 BOMBER GROUP	8/7/1947	3/25/2024
56980	#3 BOMBER GROUP	8/7/1947	3/25/2024
56981	#5 BOMBER GROUP	8/7/1947	3/25/2024
56983	#1 AIRPORT GROUP	8/7/1947	3/25/2024
56984	#3 AIRPORT GROUP	8/7/1947	3/25/2024
56985	#5 AIRPORT GROUP	8/7/1947	3/25/2024
56987	#2 BOMBER GROUP	8/7/1947	3/25/2024
56988	#6 BOMBER GROUP	8/7/1947	3/25/2024
56990	#2 AIRPORT GROUP	8/7/1947	3/25/2024
56991	#4 AIRPORT GROUP	8/7/1947	3/25/2024
56992	#6 AIRPORT GROUP	8/7/1947	3/25/2024
56993	#8 AIRPORT GROUP	7/7/1947	3/25/2024
92201	CAT 1	6/29/1965	3/25/2024
92202	CAT 2	6/29/1965	3/25/2024
92203	CAT 3	6/29/1965	3/25/2024
92204	CAT 4	6/29/1965	3/25/2024
92205	CAT 5	6/29/1965	3/25/2024
92206	CAT 6	6/29/1965	3/25/2024
92207	CAT 7	6/29/1965	3/25/2024
92208	CAT 8	6/29/1965	3/25/2024
92209	CAT 9	6/29/1965	3/25/2024
92210	CAT 10	6/29/1965	3/25/2024
92211	CAT 11	6/29/1965	3/25/2024
92212	CAT 12	6/29/1965	3/25/2024
92213	CAT 13	6/29/1965	3/25/2024
92214	CAT 14	6/29/1965	3/25/2024
92215	CAT 15	6/30/1965	3/25/2024
92216	CAT 16	6/30/1965	3/25/2024
92217	CAT 17	6/30/1965	3/25/2024
92218	CAT 18	6/30/1965	3/25/2024
92219	CAT 19	6/30/1965	3/25/2024
92220	CAT 20	6/30/1965	3/25/2024
92221	CAT 21	6/30/1965	3/25/2024
92222	CAT 22	6/30/1965	3/25/2024
92764	CAT 23	9/10/1965	3/25/2024
92765	CAT 24	9/10/1965	3/25/2024
92766	CAT 25	9/10/1965	3/25/2024
92776	CAT 35	9/11/1965	3/25/2024
92777	CAT 36	9/11/1965	3/25/2024
92778	CAT 37	9/11/1965	3/25/2024
92779	CAT 38	9/11/1965	3/25/2024
92780	CAT 39	9/12/1965	3/25/2024
92781	CAT 40	9/12/1965	3/25/2024
92782	CAT 41	9/12/1965	3/25/2024
92783	CAT 42	9/12/1965	3/25/2024
95724	CAT 47	12/2/1965	3/25/2024
95725	CAT 48	12/2/1965	3/25/2024
95726	CAT 49	12/2/1965	3/25/2024
95727	CAT 50	12/2/1965	3/25/2024
95728	CAT 51	12/2/1965	3/25/2024

Grant Number	Claim Name	Staking Date	Expiry Date
95729	CAT 52	12/2/1965	3/25/2024
95730	CAT 53	12/2/1965	3/25/2024
95731	CAT 54	12/2/1965	3/25/2024
95732	CAT 55	12/2/1965	3/25/2024
95733	CAT 56	12/2/1965	3/25/2024
95734	CAT 57	12/2/1965	3/25/2024
95735	CAT 58	12/2/1965	3/25/2024
95736	CAT 59	12/2/1965	3/25/2024
95737	CAT 60	12/2/1965	3/25/2024
95738	CAT 61	12/2/1965	3/25/2024
95739	CAT 62	12/2/1965	3/25/2024
95740	CAT 63	12/5/1965	3/25/2016
95741	CAT 64	12/5/1965	3/25/2016
95742	CAT 65	12/5/1965	3/25/2016
95743	CAT 66	12/5/1965	3/25/2016
95744	CAT 67	12/5/1965	3/25/2016
95745	CAT 68	12/5/1965	3/25/2016
95746	CAT 69	12/5/1965	3/25/2016
95747	CAT 70	12/5/1965	3/25/2016
Y 10693	JOE 89	9/24/1966	3/5/2024
Y 10694	JOE 90	9/24/1966	3/5/2024
Y 10695	JOE 91	9/24/1966	3/5/2024
Y 10696	JOE 92	9/24/1966	3/5/2024
Y 10697	JOE 93	9/24/1966	3/5/2024
Y 10698	JOE 94	9/24/1966	3/5/2024
Y 10699	JOE 95	9/24/1966	3/5/2024
Y 10700	JOE 96	9/24/1966	3/5/2024
Y 10701	JOE 97	9/24/1966	3/25/2024
Y 10702	JOE 98	9/24/1966	3/5/2024
Y 10703	JOE 99	9/24/1966	3/5/2024
Y 10704	JOE 100	9/24/1966	3/25/2024
Y 10705	JOE 101	9/24/1966	3/5/2024
Y 10706	JOE 102	9/24/1966	3/5/2024
Y 10707	JOE 103	9/24/1966	3/5/2024
Y 10708	JOE 104	9/24/1966	3/5/2024
Y 35192	MOUSE 1	6/4/1969	3/5/2024
Y 35193	MOUSE 2	6/4/1969	3/5/2024
Y 35194	MOUSE 3	6/4/1969	3/25/2016
Y 35195	MOUSE 4	6/4/1969	3/25/2016
Y 35196	MOUSE 5	6/4/1969	3/25/2016
Y 35197	MOUSE 6	6/4/1969	3/25/2016
Y 35198	MOUSE 7	6/4/1969	3/25/2016
Y 35199	MOUSE 8	6/4/1969	3/25/2016
Y 35200	MOUSE 9	6/4/1969	3/25/2016
Y 35201	MOUSE 10	6/4/1969	3/25/2016
Y 35202	MOUSE 11	6/4/1969	3/25/2016
Y 35203	MOUSE 12	6/4/1969	3/25/2016
Y 35204	MOUSE 13	6/4/1969	3/25/2016
Y 35205	MOUSE 14	6/4/1969	3/25/2016
Y 35206	MOUSE 15	6/4/1969	3/25/2016
Y 35207	MOUSE 16	6/4/1969	3/25/2016
Y 35483	MOUSE 89	6/22/1969	3/25/2016

Grant Number	Claim Name	Staking Date	Expiry Date
Y 35484	MOUSE 90	6/22/1969	3/25/2016
Y 35491	MOUSE 97	6/22/1969	3/25/2016
Y 35492	MOUSE 98	6/22/1969	3/25/2016
Y 35517	MOUSE 123	6/22/1969	3/25/2016
Y 35518	MOUSE 124	6/22/1969	3/25/2016
Y 35519	MOUSE 125	6/22/1969	3/25/2016
Y 35520	MOUSE 126	6/22/1969	3/25/2016
Y 35521	MOUSE 127	6/22/1969	3/25/2016
Y 35522	MOUSE 128	6/22/1969	3/25/2016
Y 35582	MOUSE 161	6/25/1969	3/25/2024
Y 35583	MOUSE 162	6/25/1969	3/25/2024
Y 35584	MOUSE 163	6/25/1969	3/25/2024
Y 35585	LOST FR. 1	6/25/1969	3/25/2024
Y 35586	LOST FR. 2	6/25/1969	3/25/2024
Y 35587	LOST FR. 3	6/25/1969	3/25/2024
Y 36686	CAT 22	8/12/1969	3/25/2024
Y 36687	CAT 47	8/12/1969	3/25/2024
Y 36688	CAT 48	8/12/1969	3/25/2024
Y 36689	CAT 57	8/12/1969	6/5/2024
Y 36690	CAT 62	8/12/1969	3/25/2024
Y 39601	CAT 3	10/23/1969	3/25/2024
Y 39602	CAT 4	10/23/1969	3/25/2024
Y 39603	CAT 23	10/23/1969	3/25/2024
Y 51846	CAT 1	3/29/1970	3/25/2024
Y 51847	CAT 2	3/29/1970	3/25/2024
Y 51849	CAT 26	3/30/1970	3/25/2024
Y 51850	JOE 91	3/29/1970	3/5/2024
Y 51851	JOE 92	3/29/1970	3/5/2024
Y 51852	JOE 93	3/29/1970	3/5/2024
Y 51853	JOE 94	3/29/1970	3/5/2024
Y 51854	JOE 95	3/29/1970	3/5/2024
Y 51855	JOE 96	3/29/1970	3/5/2024
YB36618	CAS 31	11/28/1991	3/25/2016
YB36619	CAS 32	11/28/1991	3/25/2016
YB36620	CAS 33	11/28/1991	3/25/2016
YB36621	CAS 34	11/28/1991	3/25/2016
YB36622	CAS 35	11/28/1991	3/25/2016
YB36623	CAS 36	11/28/1991	3/25/2016
YB37242	E 23	9/1/1992	3/25/2016
YB37243	E 24	9/1/1992	3/25/2016
YB37244	E 25	9/1/1992	3/25/2016
YB37246	E 27	9/1/1992	3/25/2016
YB37247	E 28	9/1/1992	3/25/2016
YB37248	E 29	9/1/1992	3/25/2016
YB37249	E 30	9/1/1992	3/25/2016
YB37250	E 31	9/1/1992	3/25/2016
YB37251	E 32	9/1/1992	3/25/2016
YB37278	F 27	8/30/1992	3/25/2016
YB37279	F 28	8/30/1992	3/25/2016
YB37280	F 29	8/30/1992	3/25/2024
YB37282	F 31	8/30/1992	3/25/2024
YB37284	F 33	8/30/1992	3/25/2024

Grant Number	Claim Name	Staking Date	Expiry Date
YB37640	I 1	9/9/1992	3/25/2016
YB37641	I 2	9/9/1992	3/25/2016
YB37642	I 3	9/9/1992	3/25/2016
YB37643	I 4	9/9/1992	3/25/2016
YB37658	I 19	9/9/1992	3/25/2016
YB37659	I 20	9/9/1992	3/25/2016
YC64893	VIK 1	5/27/2007	3/5/2024
YC64894	VIK 2	5/27/2007	3/5/2024
YC64895	VIK 3	5/27/2007	3/5/2024
YC64896	VIK 4	5/27/2007	3/5/2024
YC64897	VIK 5	5/27/2007	3/5/2024
YC64898	VIK 6	5/27/2007	3/5/2024
YC64899	VIK 7	5/27/2007	3/5/2024
YC64900	VIK 8	5/27/2007	3/5/2024
YC64901	VIK 9	5/27/2007	3/5/2024
YC64902	VIK 10	5/27/2007	3/5/2024
YC64903	VIK 11	5/27/2007	3/5/2024
YC64904	VIK 12	5/27/2007	3/5/2024
YC64905	VIK 13	5/27/2007	3/5/2024
YC64906	VIK 14	5/27/2007	3/5/2024
YC64907	VIK 15	5/27/2007	3/5/2024
YC64908	VIK 16	5/27/2007	3/5/2024
YC64909	VIK 17	5/27/2007	3/5/2024
YC64910	VIK 18	5/28/2007	3/5/2024
YC64911	VIK 19	5/28/2007	3/5/2024
YC64912	VIK 20	5/28/2007	3/5/2024
YC64913	VIK 21	5/28/2007	3/5/2024
YC64914	VIK 22	5/25/2007	3/5/2024
YC64915	VIK 23	5/25/2007	3/5/2024
YC64916	VIK 24	5/25/2007	3/5/2024
YC64917	VIK 25	5/25/2007	3/5/2024
YC64918	VIK 26	5/25/2007	3/5/2024
YC64919	VIK 27	5/25/2007	3/5/2024
YC64920	VIK 28	5/25/2007	3/5/2024
YC64921	VIK 29	5/25/2007	3/5/2024
YC64922	VIK 30	5/25/2007	3/5/2024
YC64923	VIK 31	5/25/2007	3/5/2024
YC64924	VIK 32	5/25/2007	3/5/2024
YC64925	VIK 33	5/25/2007	3/5/2024
YC64926	VIK 34	5/25/2007	3/5/2024
YC64927	VIK 35	5/25/2007	3/5/2024
YC64928	VIK 36	5/25/2007	3/5/2024
YC64929	VIK 37	5/25/2007	3/5/2024
YC64930	VIK 38	5/25/2007	3/5/2024
YC64931	VIK 39	5/25/2007	3/5/2024
YC64932	VIK 40	5/25/2007	3/5/2024
YC64933	VIK 41	5/31/2007	3/5/2024
YC64934	VIK 42	5/31/2007	3/5/2024
YC64935	VIK 43	6/5/2007	3/5/2024
YC64936	VIK 44	6/5/2007	3/5/2024
YC64937	VIK 45	6/5/2007	3/5/2024
YC64938	VIK 46	6/5/2007	3/5/2024

Grant Number	Claim Name	Staking Date	Expiry Date
YC64939	VIK 47	6/5/2007	3/5/2024
YC64940	VIK 48	6/5/2007	3/5/2024
YC64941	VIK 49	5/25/2007	3/5/2024
YC64942	VIK 50	5/25/2007	3/5/2024
YC64943	VIK 51	5/25/2007	3/5/2024
YC64944	VIK 52	5/25/2007	3/5/2024
YC64945	VIK 53	5/25/2007	3/5/2024
YC64946	VIK 54	5/25/2007	3/5/2024
YC64947	VIK 55	5/25/2007	3/5/2024
YC64948	VIK 56	5/24/2007	3/5/2024
YC64949	VIK 57	5/24/2007	3/5/2024
YC64950	VIK 58	5/24/2007	3/5/2024
YC64951	VIK 59	5/24/2007	3/5/2024
YC64952	VIK 60	5/24/2007	3/5/2024
YC64953	VIK 61	5/24/2007	3/5/2024
YC64954	VIK 62	5/24/2007	3/5/2024
YC64955	VIK 63	5/24/2007	3/5/2024
YC64956	VIK 64	5/24/2007	3/5/2024
YC64957	VIK 65	5/26/2007	3/5/2024
YC64958	VIK 66	5/24/2007	3/5/2024
YC64959	VIK 67	5/24/2007	3/5/2024
YC64960	VIK 68	5/24/2007	3/5/2024
YC64961	VIK 69	5/24/2007	3/5/2024
YC64962	VIK 70	5/24/2007	3/5/2024
YC64963	VIK 71	5/24/2007	3/5/2024
YC64964	VIK 72	5/24/2007	3/5/2024
YC64965	VIK 73	5/24/2007	3/5/2024
YC64966	VIK 74	5/24/2007	3/5/2024
YC64967	VIK 75	5/24/2007	3/5/2024
YC64968	VIK 76	5/24/2007	3/5/2024
YC64969	VIK 77	5/24/2007	3/5/2024
YC64970	VIK 78	5/24/2007	3/5/2024
YC64971	VIK 79	5/24/2007	3/5/2024
YC64972	VIK 80	5/27/2007	3/5/2024
YC64973	VIK 81	5/25/2007	3/5/2024
YC64974	VIK 82	5/26/2007	3/5/2024
YC64975	VIK 83	5/26/2007	3/5/2024
YC64976	VIK 84	5/26/2007	3/5/2024
YC64977	VIK 85	5/26/2007	3/5/2024
YC64978	VIK 86	5/26/2007	3/5/2024
YC64979	VIK 87	5/26/2007	3/5/2024
YC64980	VIK 88	5/26/2007	3/5/2024
YC64981	VIK 89	5/26/2007	3/5/2024
YC64982	VIK 90	5/26/2007	3/5/2024
YC64983	VIK 91	5/26/2007	3/5/2024
YC64984	VIK 92	5/26/2007	3/5/2024
YC64985	VIK 93	5/26/2007	3/5/2024
YC64986	VIK 94	5/26/2007	3/5/2024
YC64987	VIK 95	5/26/2007	3/5/2024
YC64988	VIK 96	5/28/2007	3/5/2024
YC64989	VIK 97	5/28/2007	3/5/2024
YC64990	VIK 98	5/28/2007	3/5/2024

Grant Number	Claim Name	Staking Date	Expiry Date
YC64991	VIK 99	5/28/2007	3/5/2024
YC64992	VIK 100	5/28/2007	3/5/2024
YC64993	VIK 101	5/28/2007	3/5/2024
YC64994	VIK 102	5/28/2007	3/5/2024
YC64995	VIK 103	5/28/2007	3/5/2024
YC64996	VIK 104	5/26/2007	3/5/2024
YC64997	VIK 105	5/26/2007	3/5/2024
YC64998	VIK 106	5/26/2007	3/5/2024
YC64999	VIK 107	5/26/2007	3/5/2024
YC65000	VIK 108	5/26/2007	3/5/2024
YC65001	VIK 109	5/26/2007	3/5/2024
YC65002	VIK 110	5/26/2007	3/5/2024
YC65003	VIK 111	5/26/2007	3/5/2024
YC65004	VIK 112	5/26/2007	3/5/2024
YC65005	VIK 113	5/26/2007	3/5/2024
YC65006	VIK 114	5/26/2007	3/5/2024
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YC65008	VIK 116	5/26/2007	3/5/2024
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YC65010	VIK 118	5/26/2007	3/5/2024
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YC65016	VIK 124	5/26/2007	3/5/2024
YC65017	VIK 125	5/26/2007	3/5/2024
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YC65040	VIK 148	5/29/2007	3/5/2024
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YC65042	VIK 150	5/29/2007	3/5/2024

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YC65043	VIK 151	5/29/2007	3/5/2024
YC65044	VIK 152	5/29/2007	3/5/2024
YC65045	VIK 153	5/29/2007	3/5/2024
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YC65048	VIK 156	5/29/2007	3/5/2024
YC65049	VIK 157	5/29/2007	3/5/2024
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YC65053	VIK 161	5/29/2007	3/5/2024
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YC65072	VIK 180	5/30/2007	3/5/2024
YC65073	VIK 181	5/30/2007	3/5/2024
YC65074	VIK 182	5/30/2007	3/5/2024
YC65075	VIK 183	5/30/2007	3/5/2024
YC65076	VIK 184	5/28/2007	3/5/2024
YC65077	VIK 185	5/28/2007	3/5/2024
YC65078	VIK 186	5/28/2007	3/5/2024
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YC65080	VIK 188	5/29/2007	3/5/2024
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YC81318	BRIT 3	6/10/2008	3/5/2022
YC81319	BRIT 4	6/10/2008	3/5/2022
YC81320	BRIT 5	6/10/2008	3/5/2022
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YC81322	BRIT 7	6/10/2008	3/5/2022
YC81323	BRIT 8	6/10/2008	3/5/2022
YC81324	BRIT 9	6/10/2008	3/5/2022
YC81325	BRIT 10	6/10/2008	3/5/2022
YC81326	BRIT 11	6/10/2008	3/5/2022
YC81327	BRIT 12	6/10/2008	3/5/2022
YC81328	BRIT 13	6/10/2008	3/5/2022
YC81329	BRIT 14	6/10/2008	3/5/2022

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YC81330	BRIT 15	6/10/2008	3/5/2022
YC81331	BRIT 16	6/10/2008	3/5/2022
YC81332	BRIT 17	6/10/2008	3/5/2022
YC81333	BRIT 18	6/10/2008	3/5/2022
YC81334	BRIT 19	6/10/2008	3/5/2022
YC81335	BRIT 20	6/10/2008	3/5/2022
YC81336	BRIT 21	6/10/2008	3/5/2022
YC81337	BRIT 22	6/10/2008	3/5/2022
YC81338	BRIT 23	6/10/2008	3/5/2022
YC81339	BRIT 24	6/10/2008	3/5/2022
YC81340	BRIT 25	6/10/2008	3/5/2022
YC81341	BRIT 26	6/10/2008	3/5/2022
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YC81343	BRIT 28	6/10/2008	3/5/2022
YC81344	BRIT 29	6/10/2008	3/5/2022
YC81345	BRIT 30	6/10/2008	3/5/2022
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YC81347	BRIT 32	6/10/2008	3/5/2022
YC81348	BRIT 33	6/10/2008	3/5/2022
YC81349	BRIT 34	6/10/2008	3/5/2022
YC81350	BRIT 35	6/10/2008	3/5/2022
YC81351	BRIT 36	6/10/2008	3/5/2022
YC81352	BRIT 37	6/10/2008	3/5/2022
YC81353	BRIT 38	6/10/2008	3/5/2022
YC81354	BRIT 39	6/10/2008	3/5/2022
YC81355	BRIT 40	6/10/2008	3/5/2022
YC81356	BRIT 41	6/10/2008	3/5/2022
YC81357	BRIT 42	6/10/2008	3/5/2022
YC81358	BRIT 43	6/10/2008	3/5/2022
YC81359	BRIT 44	6/10/2008	3/5/2022
YC81360	BRIT 45	6/10/2008	3/5/2022
YC81361	BRIT 46	6/10/2008	3/5/2022
YC81362	BRIT 47	6/10/2008	3/5/2022
YC81363	BRIT 48	6/10/2008	3/5/2022
YC81364	BRIT 49	6/10/2008	3/5/2022
YC81365	BRIT 50	6/10/2008	3/5/2022
YC81366	BRIT 51	6/10/2008	3/5/2022
YC81367	BRIT 52	6/10/2008	3/5/2022
YC81368	BRIT 53	6/10/2008	3/5/2022
YC81369	BRIT 54	6/10/2008	3/5/2022
YC81370	BRIT 55	6/10/2008	3/5/2022
YC81371	BRIT 56	6/10/2008	3/5/2022
YC81372	BRIT 57	6/10/2008	3/5/2022
YC81373	BRIT 58	6/10/2008	3/5/2022
YC81374	BRIT 59	6/10/2008	3/5/2022
YC81375	BRIT 60	6/10/2008	3/5/2022
YC81376	BRIT 61	6/10/2008	3/5/2022
YC81377	BRIT 62	6/10/2008	3/5/2022
YC81378	BRIT 63	6/10/2008	3/5/2022
YC81379	CC 1	6/12/2008	3/5/2022
YC81380	CC 2	6/12/2008	3/5/2022
YC81381	CC 3	6/12/2008	3/5/2022

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YC81382	CC 4	6/12/2008	3/5/2022
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YC81384	CC 6	6/12/2008	3/5/2022
YC81385	CC 7	6/11/2008	3/5/2022
YC81386	CC 8	6/11/2008	3/5/2022
YC81387	CC 9	6/11/2008	3/5/2022
YC81388	CC 10	6/11/2008	3/5/2022
YC81389	CC 11	6/11/2008	3/5/2022
YC81390	CC 12	6/11/2008	3/5/2022
YC81391	CC 13	6/11/2008	3/5/2022
YC81392	CC 14	6/11/2008	3/5/2022
YC81393	CC 15	6/11/2008	3/5/2022
YC81394	CC 16	6/11/2008	3/5/2022
YC81395	CC 17	6/11/2008	3/5/2022
YC81396	CC 18	6/11/2008	3/5/2022
YC81397	CC 19	6/11/2008	3/5/2022
YC81398	CC 20	6/11/2008	3/5/2022
YC81399	CC 21	6/12/2008	3/5/2022
YC81400	CC 22	6/12/2008	3/5/2022
YC81401	CC 23	6/12/2008	3/5/2022
YC81402	CC 24	6/12/2008	3/5/2022
YC81403	CC 25	6/12/2008	3/5/2022
YC81404	CC 26	6/12/2008	3/5/2022
YC81405	CC 27	6/12/2008	3/5/2022
YC81406	CC 28	6/12/2008	3/5/2022
YC81407	CC 29	6/12/2008	3/5/2022
YC81408	CC 30	6/11/2008	3/5/2022
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YC81410	CC 32	6/11/2008	3/5/2022
YC81411	CC 33	6/11/2008	3/5/2022
YC81412	CC 34	6/11/2008	3/5/2022
YC81413	CC 35	6/11/2008	3/5/2022
YC81414	CC 36	6/11/2008	3/5/2022
YC81415	CC 37	6/11/2008	3/5/2022
YC81416	CC 38	6/11/2008	3/5/2022
YC81417	CC 39	6/11/2008	3/5/2022
YC81418	CC 40	6/12/2008	3/5/2022
YC81419	CC 41	6/12/2008	3/5/2022
YC81420	CC 42	6/12/2008	3/5/2022
YC81421	CC 43	6/12/2008	3/5/2022
YC81422	CC 44	6/12/2008	3/5/2022
YC81423	CC 45	6/12/2008	3/5/2022
YC81424	CC 46	6/12/2008	3/5/2022
YC81425	CC 47	6/12/2008	3/5/2022
YC81426	CC 48	6/12/2008	3/5/2022
YC81427	CC 49	6/12/2008	3/5/2022
YC81428	CC 50	6/12/2008	3/5/2022
YC81429	CC 51	6/12/2008	3/5/2022
YC81430	CC 52	6/12/2008	3/5/2022
YC81431	CC 53	6/12/2008	3/5/2022
YC81432	CC 54	6/12/2008	3/5/2022
YC81433	CC 55	6/12/2008	3/5/2022

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YC81434	CC 56	6/12/2008	3/5/2022
YC81435	CC 57	6/11/2008	3/5/2022
YC81436	CC 58	6/11/2008	3/5/2022
YC81437	CC 59	6/11/2008	3/5/2022
YC81438	CC 60	6/11/2008	3/5/2022
YC81439	CC 61	6/11/2008	3/5/2022
YC81440	CC 62	6/11/2008	3/5/2022
YC81441	CC 63	6/12/2008	3/5/2022
YC81442	CC 64	6/12/2008	3/5/2022
YC81443	CC 65	6/12/2008	3/5/2022
YC81444	CC 66	6/12/2008	3/5/2022
YC81445	CC 67	6/12/2008	3/5/2022
YC81446	CC 68	6/12/2008	3/5/2022
YC81447	CC 69	6/12/2008	3/5/2022
YC81448	CC 70	6/12/2008	3/5/2022
YC81449	CC 71	6/12/2008	3/5/2022
YC81450	CC 72	6/12/2008	3/5/2022
YC81451	CC 73	6/12/2008	3/5/2022
YC81452	CC 74	6/12/2008	3/5/2022
YC81453	CC 75	6/12/2008	3/5/2022
YC81454	CC 76	6/12/2008	3/5/2022
YC81455	CC 77	6/12/2008	3/5/2022
YC81456	CC 78	6/12/2008	3/5/2022
YC81457	CC 79	6/12/2008	3/5/2022
YC81458	CC 80	6/11/2008	3/5/2022
YC81459	CC 81	6/11/2008	3/5/2022
YC81460	CC 82	6/11/2008	3/5/2022
YC81461	CC 83	6/13/2008	3/5/2022
YC81462	CC 84	6/13/2008	3/5/2022
YC81463	CC 85	6/13/2008	3/5/2022
YC81464	CC 86	6/13/2008	3/5/2022
YC81465	CC 87	6/13/2008	3/5/2022
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YC81467	CC 89	6/13/2008	3/5/2022
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YC81469	CC 91	6/13/2008	3/5/2022
YC81470	CC 92	6/13/2008	3/5/2022
YC81471	CC 93	6/13/2008	3/5/2022
YC81472	CC 94	6/13/2008	3/5/2022
YD17559	AXS 1	10/5/2009	3/25/2015
YD17560	AXS 2	10/5/2009	3/25/2015
YD17561	AXS 3	10/5/2009	3/25/2015
YD17562	AXS 4	10/5/2009	3/25/2015
YD17563	AXS 5	10/5/2009	3/25/2015
YD17564	AXS 6	10/5/2009	3/25/2015
YD17565	AXS 7	10/5/2009	3/25/2015
YD17566	AXS 8	10/5/2009	3/25/2015
YD17567	AXS 9	10/5/2009	3/25/2015
YD17568	AXS 10	10/5/2009	3/25/2015
YD17569	AXS 11	10/6/2009	3/25/2015
YD17570	AXS 12	10/6/2009	3/25/2015
YD17571	AXS 13	10/6/2009	3/25/2015

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YD17572	AXS 14	10/6/2009	3/25/2015
YD17573	AXS 15	10/6/2009	3/25/2015
YD17574	AXS 16	10/6/2009	3/25/2015
YD17575	AXS 17	10/6/2009	3/25/2015
YD17576	AXS 18	10/6/2009	3/25/2015
YD17577	AXS 19	10/5/2009	3/25/2015
YD17578	AXS 20	10/5/2009	3/25/2015
YD17579	AXS 21	10/5/2009	3/25/2015
YD17580	AXS 22	10/5/2009	3/25/2015
YD17581	AXS 23	10/5/2009	3/25/2015
YD17582	AXS 24	10/5/2009	3/25/2015
YD17583	AXS 25	10/5/2009	3/25/2015
YD17584	AXS 26	10/5/2009	3/25/2015
YD17585	AXS 27	10/5/2009	3/25/2015
YD17586	AXS 28	10/5/2009	3/25/2015
YD17587	AXS 29	10/5/2009	3/25/2015
YD17588	AXS 30	10/5/2009	3/25/2015
YD17589	AXS 31	10/5/2009	3/25/2015
YD17590	AXS 32	10/5/2009	3/25/2015
YD17591	AXS 33	10/5/2009	3/25/2015
YD17592	AXS 34	10/5/2009	3/25/2015
YD17593	AXS 35	10/5/2009	3/25/2015
YD17594	AXS 36	10/5/2009	3/25/2015
YD17595	AXS 37	10/5/2009	3/25/2015
YD17596	AXS 38	10/5/2009	3/25/2015
YD17597	AXS 39	10/5/2009	3/25/2015
YD17598	AXS 40	10/5/2009	3/25/2015
YD17599	AXS 41	10/5/2009	3/25/2015
YD17600	AXS 42	10/5/2009	3/25/2015
YD17601	AXS 43	10/5/2009	3/25/2015
YD17602	AXS 44	10/5/2009	3/25/2015
YD17603	AXS 45	10/5/2009	3/25/2015
YD17604	AXS 46	10/5/2009	3/25/2015
YD17605	AXS 47	10/5/2009	3/25/2015
YD17606	AXS 48	10/5/2009	3/25/2015
YD17607	AXS 49	10/6/2009	3/25/2015
YD17608	AXS 50	10/6/2009	3/25/2015
YD17609	AXS 51	10/6/2009	3/25/2015
YD17610	AXS 52	10/6/2009	3/25/2015
YD17611	AXS 53	10/6/2009	3/25/2015
YD17612	AXS 54	10/6/2009	3/25/2015
YD17613	AXS 55	10/5/2009	3/25/2015
YD17614	AXS 56	10/5/2009	3/25/2015
YD17615	AXS 57	10/5/2009	3/25/2015
YD17616	AXS 58	10/5/2009	3/25/2015
YD17617	AXS 59	10/5/2009	3/25/2015
YD17618	AXS 60	10/5/2009	3/25/2015
YD17619	AXS 61	10/5/2009	3/25/2015
YD17620	AXS 62	10/5/2009	3/25/2015
YD17621	AXS 63	10/5/2009	3/25/2015
YD17622	AXS 64	10/5/2009	3/25/2015
YD17623	AXS 65	10/5/2009	3/25/2015

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YD17624	AXS 66	10/5/2009	3/25/2015
YD17625	AXS 67	10/5/2009	3/25/2015
YD17626	AXS 68	10/5/2009	3/25/2015
YD17627	AXS 69	10/6/2009	3/25/2015
YD17628	AXS 70	10/6/2009	3/25/2015
YD17629	AXS 71	10/6/2009	3/25/2015
YD17630	AXS 72	10/6/2009	3/25/2015
YD17631	AXS 73	10/6/2009	3/25/2015
YD17632	AXS 74	10/6/2009	3/25/2015
YD17633	AXS 75	10/5/2009	3/25/2015
YD17634	AXS 76	10/5/2009	3/25/2015
YD17635	AXS 77	10/5/2009	3/25/2015
YD17636	AXS 78	10/5/2009	3/25/2015
YD17637	AXS 79	10/5/2009	3/25/2015
YD17638	AXS 80	10/5/2009	3/25/2015
YD17639	AXS 81	10/5/2009	3/25/2015
YD17640	AXS 82	10/5/2009	3/25/2015
YD17641	AXS 83	10/5/2009	3/25/2015
YD17642	AXS 84	10/5/2009	3/25/2015
YD17643	AXS 85	10/5/2009	3/25/2015
YD17644	AXS 86	10/5/2009	3/25/2015
YD17645	AXS 87	10/6/2009	3/25/2015
YD17646	AXS 88	10/6/2009	3/25/2015
YD17647	AXS 89	10/6/2009	3/25/2015
YD17648	AXS 90	10/6/2009	3/25/2015
YD17649	AXS 91	10/6/2009	3/25/2015
YD17650	AXS 92	10/6/2009	3/25/2015
YD17651	AXS 103	10/7/2009	3/25/2015
YD17652	AXS 102	10/7/2009	3/25/2015
YD17653	AXS 101	10/7/2009	3/25/2015
YD17654	AXS 100	10/6/2009	3/25/2015
YD17655	AXS 99	10/6/2009	3/25/2015
YD17656	AXS 98	10/6/2009	3/25/2015
YD17657	AXS 97	10/6/2009	3/25/2015
YD17658	AXS 96	10/6/2009	3/25/2015
YD17659	AXS 95	10/6/2009	3/25/2015
YD17660	AXS 94	10/6/2009	3/25/2015
YD17661	AXS 93	10/6/2009	3/25/2015
YD17662	AXS 104	10/7/2009	3/25/2015
YD17663	AXS 105	10/7/2009	3/25/2015
YD17664	AXS 106	10/7/2009	3/25/2015
YD17665	AXS 107	10/7/2009	3/25/2015
YD17666	AXS 108	10/7/2009	3/25/2015
YD17667	AXS 109	10/7/2009	3/25/2015
YD17668	AXS 110	10/7/2009	3/25/2015
YD17669	AXS 111	10/7/2009	3/25/2015
YD17670	AXS 112	10/7/2009	3/25/2015
YD17671	AXS 113	10/6/2009	3/25/2015
YD17672	AXS 114	10/6/2009	3/25/2015
YD17673	AXS 115	10/6/2009	3/25/2015
YD17674	AXS 116	10/6/2009	3/25/2015
YD17675	AXS 117	10/6/2009	3/25/2015

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YD17676	AXS 118	10/6/2009	3/25/2015
YD17677	AXS 119	10/6/2009	3/25/2015
YD17678	AXS 120	10/6/2009	3/25/2015
YD17679	AXS 121	10/6/2009	3/25/2015
YD17680	AXS 122	10/6/2009	3/25/2015
YD17681	AXS 123	10/6/2009	3/25/2015
YD17682	AXS 124	10/6/2009	3/25/2015
YD17683	AXS 125	10/6/2009	3/25/2015
YD17684	AXS 126	10/6/2009	3/25/2015
YD17685	AXS 127	10/6/2009	3/25/2015
YD17686	AXS 128	10/6/2009	3/25/2015
YD17687	AXS 129	10/6/2009	3/25/2015
YD17688	AXS 130	10/6/2009	3/25/2015
YD17689	AXS 131	10/6/2009	3/25/2015
YD17690	AXS 132	10/6/2009	3/25/2015
YD17691	AXS 133	10/6/2009	3/25/2015
YD17692	AXS 134	10/6/2009	3/25/2015
YD17693	AXS 135	10/6/2009	3/25/2015
YD17694	AXS 136	10/6/2009	3/25/2015
YD60030	AXS 137	5/11/2010	3/25/2016
YD60031	AXS 138	5/11/2010	3/25/2016
YD60032	AXS 139	5/11/2010	3/25/2016
YD60033	AXS 140	5/11/2010	3/25/2016
YD60034	AXS 141	5/11/2010	3/25/2016
YD60035	AXS 142	5/11/2010	3/25/2016
YD60036	AXS 143	5/11/2010	3/25/2016
YD60037	AXS 144	5/11/2010	3/25/2016
YD60038	AXS 145	5/11/2010	3/25/2016
YD60039	AXS 146	5/11/2010	3/25/2016
YD60040	AXS 147	5/11/2010	3/25/2016
YD60041	AXS 148	5/11/2010	3/25/2016
YD60042	AXS 149	5/11/2010	3/25/2016
YD60043	AXS 150	5/11/2010	3/25/2016
YD60044	AXS 151	5/11/2010	3/25/2016
YD60045	AXS 152	5/11/2010	3/25/2016
YD60046	AXS 154	5/11/2010	3/25/2016
YD60047	AXS 153	5/11/2010	3/25/2016
YD60048	AXS 155	5/11/2010	3/25/2016
YD60049	AXS 156	5/11/2010	3/25/2016
YD60050	AXS 157	5/11/2010	3/25/2016
YD60051	AXS 158	5/11/2010	3/25/2016
YD60052	AXS 159	5/11/2010	3/25/2016
YD60053	AXS 160	5/11/2010	3/25/2016
YD60054	AXS 161	5/11/2010	3/25/2016
YD60055	AXS 162	5/11/2010	3/25/2016
YD60056	AXS 163	5/11/2010	3/25/2016
YD60057	AXS 164	5/11/2010	3/25/2016
YD60058	AXS 165	5/11/2010	3/25/2016
YD60059	AXS 166	5/11/2010	3/25/2016
YD60060	AXS 167	5/11/2010	3/25/2016
YD60061	AXS 168	5/11/2010	3/25/2016
YD60062	AXS 169	5/11/2010	3/25/2016

Grant Number	Claim Name	Staking Date	Expiry Date
YD60063	AXS 170	5/11/2010	3/25/2016
YD60064	AXS 171	5/11/2010	3/25/2016
YD60065	AXS 172	5/11/2010	3/25/2016
YD60066	AXS 173	5/11/2010	3/25/2016
YD60067	AXS 174	5/11/2010	3/25/2016
YD60068	AXS 175	5/11/2010	3/25/2016
YD60069	AXS 176	5/11/2010	3/25/2016
YD60070	AXS 177	5/11/2010	3/25/2016
YD60071	AXS 178	5/11/2010	3/25/2016
YD60072	AXS 179	5/11/2010	3/25/2016
YD60073	AXS 180	5/11/2010	3/25/2016
YD60074	AXS 181	5/11/2010	3/25/2016
YD60075	AXS 182	5/11/2010	3/25/2016
YD60076	AXS 183	5/11/2010	3/25/2016
YD60077	AXS 184	5/11/2010	3/25/2016
YD60078	AXS 185	5/11/2010	3/25/2016
YD60079	AXS 186	5/11/2010	3/25/2016
YD61120	AXS 187	5/11/2010	3/25/2016
YD61121	AXS 188	5/11/2010	3/25/2016
YD61122	AXS 189	5/11/2010	3/25/2016
YD61123	AXS 190	5/11/2010	3/25/2016
YD61124	AXS 191	5/11/2010	3/25/2016
YD61125	AXS 192	5/11/2010	3/25/2016
YD61126	AXS 193	5/11/2010	3/25/2016
YD61127	AXS 194	5/11/2010	3/25/2016
YD61128	AXS 196	5/11/2010	3/25/2016
YD61129	AXS 195	5/11/2010	3/25/2016
YD61130	AXS 197	5/11/2010	3/25/2016
YD61131	AXS 198	5/11/2010	3/25/2016
YD61132	AXS 199	5/11/2010	3/25/2016

Grant Number	Claim Name	Staking Date	Expiry Date
YD04375	FLY 1	2/2/2011	3/26/2016
YD04376	FLY 2	2/2/2011	3/25/2016
YD04377	FLY 3	2/2/2011	3/25/2016
YD04378	FLY 4	2/2/2011	3/25/2016
YD04379	FLY 5	2/2/2011	3/25/2016
YD04380	FLY 6	2/2/2011	3/25/2016
YD04381	FLY 7	2/2/2011	3/25/2016
YD04382	FLY 8	2/2/2011	3/25/2016
YD04383	FLY 9	2/2/2011	3/25/2016
YD04384	FLY 10	2/2/2011	3/25/2016
YD04385	FLY 11	2/2/2011	3/25/2016
YD04386	FLY 12	2/2/2011	3/25/2016
YD04387	FLY 13	2/2/2011	3/25/2016
YD04388	FLY 14	2/2/2011	3/25/2016
YD04399	FLY 15	2/2/2011	3/25/2016
YD04400	FLY 16	2/2/2011	3/25/2016
YD04401	FLY 17	2/2/2011	3/25/2016
YD04402	FLY 18	2/2/2011	3/25/2016