NOVAGOLD RESOURCES INC.

UPDATED PRELIMINARY ECONOMIC ASSESSMENT

FOR THE

GALORE CREEK PROJECT

HATCH™

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NOTICE

This report was prepared for the sole and exclusive benefit of NovaGold Resources Inc. (NovaGold) by Hatch Limited (Hatch). This document is meant to be read as a whole, and sections should not be read or relied upon out of context. This document contains the expression of the professional opinion of Hatch based on information available at the time of preparation. The quality of the information, conclusions and estimates contained herein is consistent with the intended level of accuracy as well as the circumstances and constraints under which the mandate was performed. The report includes information generated or provided by other outside sources identified herein. Hatch does not warrant the accuracy or completeness of data supplied by outside sources. This report is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that this preliminary assessment will be realized.

NovaGold Resources Inc.
Galore Creek Project

Updated Preliminary Economic Assessment

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1. Summary

1.1 Overview of Updated Preliminary Economic Assessment

In August 2004 Hatch Limited (Hatch) prepared for NovaGold Resources Inc. a report entitled "Preliminary Economic Assessment for The Galore Creek Gold-Silver-Copper Project" based on resource estimates completed in April 2004, and processing 242 million tonnes (or approximately 63% if the then known resource) of ore grade material at a rate of 30,000 tonnes per day from the Central and Southwest zones.

This Updated Preliminary Economic Assessment (PEA) is based on resource estimates completed in May 2005 and processing 475 million tonnes (or approximately 43% of the known resource) of ore grade material at a rate of 65,000 tonnes per day from the Central, Southwest, Junction and West Fork zones. The costs and design incorporated in this report, with the exception of the costs of the concentrate pipeline, were prepared to pre-feasibility study quality level. Because the mine plan included inferred resources, and the concentrate pipeline work was completed to scoping study level, this overall study is classified as a PEA.

This PEA demonstrates the viability of a conventional open-pit mining operation using long-term average metal prices of US$1.00/lb Copper, US$400/oz Gold and US$6.00/oz Silver and shows that the Galore Creek project has the potential to recover 5.9 billion pounds of copper, 3.7 million ounces of gold and 40 million ounces of silver over a 20 year mine-life.

In the first 6 years the project would produce an average of over 300,000 ounces gold, 2.31 million ounces silver and 370 million pounds copper yearly at an average total cash cost of US$0.36 per pound of copper with precious metals as credits (based on long-term transportation and refining cost projections and metal prices). Net after-tax cash flow from the project would generate over US$200 million annually for the first 6 years using long-term average metal prices resulting in rapid payback of all mine capital in 5.2 years. At recent metal prices of US$1.75/lb Copper; US$450/oz Gold and US$7.00/oz Silver the project would generate an average of US$480 million in pre-tax annual cashflow and over US$350 million annually in after-tax cashflow for the first 6 years, reducing the capital payback to 2.1 years.

1.2 Ownership and Location

NovaGold Resources Inc., through its 100% owned subsidiary NovaGold Canada Inc. (formerly SpectrumGold Inc.), is acquiring the main Galore Creek property by purchasing Stikine Copper Ltd., that currently holds the claims, for payments totaling US$20.3 million over a period of eight years. NovaGold is also earning interests in two other claim areas owned by Eagle Plains Mining (containing the Copper Canyon inferred resource) and Pioneer Metals (containing mineralized zone). No resources from these properties are included in this PEA.

The Galore Creek Project is located in northwestern British Columbia approximately 1000 kilometers northwest of Vancouver, 80 kilometers northwest of the Eskay Creek Mine. Current access is mainly by helicopter and the area is characterized by cool summers, cold humid winters and relatively heavy snowfall.

1.3 Resources

The estimated mineral resources used in this study incorporate results from the 2004 drilling campaign and has benefited from a revised geological interpretation. The model integrates 106,781 m of drilling in 452 core holes with a total of 36,866 assays. Interpretation of the geological data and division of the deposits into domains for estimation was supported by extensive geological mapping, airborne and terrestrial geophysical surveys. The estimates are based on a 3-dimensional computer block model with grades interpolated into individual 25 m by 25 m by 15 m high blocks. The grade interpolation used ordinary kriging procedures and mineralization was composited on 5 m intervals with high-grade samples capped based on lognormal probability plots.

Table 1-1 summarizes the estimate for measured and indicated resources, while Table 1-2 summarizes the inferred resources, both at a cut-off grade of 0.35% copper equivalent (CuEq). The CuEq calculations use metal prices of US$375/oz for gold, US$5.50/oz for silver and US$0.90/lb for copper. Copper equivalent calculations reflect gross metal content that have been adjusted for metallurgical recoveries.

Table 1-1: Summary for Measured Plus Indicated Resource (M+I)

ZONE	Cut-off (CuEq%)	Tonnes > Cut-off (tonnes)	Grade > Cut-off				Million lbs. of Cu	Million ozs of Au	Million ozs of Ag
			Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)			
Central	0.35	423,900,000	0.61	0.30	4.68	0.78	5,739	4.12	63.8
Southwest	0.35	47,700,000	0.45	0.82	3.04	0.97	470	1.25	4.7
Junction	0.35	30,000,000	0.59	0.41	4.78	0.84	390	0.40	4.6
West Fork	0.35	15,100,000	0.58	0.38	4.79	0.80	193	0.18	2.3
TOTAL M+I		**516,700,000**	**0.59**	**0.36**	**4.54**	**0.80**	**6,792**	**5.95**	**75.4**

Table 1-2: Summary for Inferred Resource

ZONE	Cut-off (CuEq%)	Tonnes > Cut-off (tonnes)	Grade > Cut-off				Million lbs. of Cu	Million ozs of Au	Million ozs of Ag
			Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)			
Central	0.35	173,600,000	0.47	0.28	3.42	0.62	1,788	1.58	19.1
Southwest	0.35	122,900,000	0.31	0.56	2.27	0.70	851	2.20	9.0
Junction	0.35	71,600,000	0.53	0.29	3.33	0.68	840	0.66	7.7
West Fork	0.35	45,400,000	0.47	0.34	4.99	0.66	466	0.50	7.3
Copper Canyon	0.35	164,800,000	0.35	0.54	7.15	0.73	1,275	2.86	37.9
TOTAL Inferred		**578,300,000**	**0.41**	**0.42**	**4.35**	**0.68**	**5,220**	**7.79**	**81.0**

1.4 Mine Plan

The ore bodies will be mined by conventional open pit mining methods using truck and shovel units. Ore production in this study is based on an average 65,000 tpd and total material movement averaging 218,000 tpd over the 20 year life of the project. Mining in the initial years will be focused on higher grade mineralization in the Southwest zone and the South Gold Lens in the southern end of the main Central deposit.

An optimized 65,000 tpd mill feed schedule was developed for the Galore Creek mine. Detailed pit phases were engineered from the results of a Lerchs-Grossman (LG) sensitivity analysis, and yielded the phase delineated resources in Table 1-3. Phase delineated resources in Table 1-3 include a 10% dilution for blocks that contain both waste and mineralized material above cutoff grade (contact dilution). No mining losses have been taken on the assumption that after the 2005 exploration data is included in the resource modeling, most blocks above cut-off grade will be 100% mill feed. Cut-off grade for the Phase delineated resources in Table 1-3 is US$3.26/t Net Smelter Return (NSR).

Table 1-3: Galore Creek Pit Resource Summary

DEPOSITS	RUN OF MINE (kt)	WASTE (kt)	S/R	GRADES		
				CU (%)	AU (g/t)	AG (g/t)
Central Main	375,279	722,921	1.93	0.639	0.287	4.77
Central Middle Creek	9,128	4,639	0.51	0.414	0.172	0.40
South West	64,018	163,547	2.55	0.545	0.712	2.95
West Fork	10,473	28,904	2.76	0.518	0.578	9.18
Junction	34,960	95,920	2.74	0.802	0.373	6.25
Grand Total	**493,858**	**1,015,931**	**2.06**	**0.631**	**0.352**	**4.65**

1.5 Metallurgy

A comprehensive metallurgical program has been completed on samples from 2004 drilling to validate the flowsheet developed in the earlier work and to determine the metallurgy associated with the variable mineralization and head grades in the various zones of the Galore Creek deposit. The test program investigated grindability, mineralogy and minerals recovery by batch and locked cycle flotation. Coarse assay rejects were used in batch flowsheet development work while drill core composites of the primary mineral groupings were used in locked cycle tests to confirm circuit stability, metals recovery and concentrate grade. Models have also been developed to project copper, gold and silver recoveries in mining blocks and to identify gold occurrence and deportment.

Ore hardness, in terms of Bond Ball Mill Work Index, varied from 9 kWh/t to 21 kWh/t across the deposit, including “broken” and “stick” ores. A Bond Ball Mill Work Index of 16 kWh/t, corresponding to the average for the main ore types, has been used to design an SABC grinding circuit.

The current program validated the flowsheet developed in the previous work. The flowsheet will comprise rougher flotation, regrind of rougher concentrate and three stages of cleaner flotation. Gravity concentration for gold has been excluded from the flowsheet. Preliminary tests using a combined gravity concentration – flotation circuit did not produce significant incremental gold recovery over that obtained from direct flotation.

The final concentrates had relatively low penalty elements. Fluorine and selenium concentrations were variable and could be of concern. The lead content might be a concern depending on the end-use of the slag from smelters.

Copper recovery for variable Cu head grades can be estimated based on a constant tailings model and this is considered to be a reasonable model for projecting recoveries in mining blocks across the deposit. Gold and silver recoveries depend largely on copper recovery in flotation concentrate and a model has been developed to project their recoveries based on copper recovery. As such, the gold in ores with very low copper, and largely occurring within pyrite grains, would not be recovered.

1.6 Process Plant and Site

A conventional process plant based on crushing, grinding, flotation, thickening and filtration unit operations is proposed for the Galore Creek ores. A 65,000 tpd ore processing rate is expected to produce an average of 480,000 tpa of copper concentrate containing both gold and silver. Concentrate will be pumped through a 150 mm diameter pipeline approximately 140 km to a remote filter plant located near Highway 37. The concentrate pipeline will be buried alongside the road for most of its length, for protection from avalanche hazards and the weather.

Tailings and waste rock will be stored in a single dam within the Galore Creek Valley, sized to accommodate all the tailings and Potentially Acid Generating (PAG) waste rock. Non-Potentially Acid Generating (NPAG) rock will be stored in a number of dumps located along the west side of the Valley, north of the open pits and also, south of the dam. The dam will become flooded at the end of operations, and thus prevent the oxidation of residual sulphidic materials.

Management of surficial waters at the site will be a significant issue. Diversion channels will direct as much fresh water from the tailings/waste dam as practical. Diversion channels will be constructed to the west and east of the open pits and a water dam will be constructed in the East Fork Creek to divert fresh water into 7 km long diversion channel on the east side of Galore Valley that will discharge into Galore Creek downstream of the tailings dam. From current knowledge of water quality predictions, it is considered that surplus water in the tailings/waste dam can be discharged without treatment. Testwork to confirm this is on-going.

1.7 Access and Power

The two main access routes identified in the 2004 Scoping Study were further investigated during the period August 2004 to July 2005, and the Northern route selected for the project. The main access road will run almost due west from Highway 37 along the More Creek Valley, west down Sphaler Creek and then north up Scotsimpson Creek to a 4 km long road tunnel to access Galore Creek Valley. The access road is classified as a resource development road. The road will have a 6 m wide surface with intervisible turnarounds.

Electrical power will be supplied from a connection to the BCH grid near the proposed Forrest Kerr hydro power station. The 138 kV transmission line will run largely within the access road right of way. The connection to the BC grid is the subject of further study to quantify the constraints of the presently proposed BCH/Forrest Kerr system.

1.8 Schedule

A construction schedule has been developed indicating a 3 to 4 year construction period. The critical path for the schedule flows through construction of the access road and tunnel, followed by construction of the water diversion channels and dam structures. Initially, construction operations will be supported from two airstrips, one at the south side of the Porcupine River and the other near Round Lake.

1.9 Capital Costs

The capital costs for the Galore Creek Project including process plant, infrastructure and mining were estimated in 2Q, 2005 Canadian dollars and are presented in US dollars. These include no allowances for escalation or exchange rate fluctuations. The summary capital cost estimate for the Galore Creek Project is presented in Table 1-4 and reflect an accuracy level of ±20%, consistent with a pre-feasibility study level of engineering effort.

Table 1-4: Capital Cost Summary - Base Case, 65,000 tpd mill

Description	**US$M**
Process Plant	**223**
Infrastructure	**301**
Mine	**229**
Total Directs	**753**
Indirects	**205**
Contingency	**144**
Indirects & Contingency	**349**
Total Estimate	**$1,102**

1.10 Operating Costs

The base case operating cost for the Galore Creek Project including mining, general and administrative and process costs have been estimated in 2Q, 2005 Canadian dollars and presented in US dollars. These include no allowances for escalation or exchange rate fluctuations. The summary operating cost estimate is presented in Table 1-5 and reflect an accuracy level of ±15%, consistent with a PFS level of engineering effort.

Table 1-5: Operating Cost Summary

		US$/t ore
G&A	G&A Labour	**0.13**
	Fixed	**0.52**
	Total G&A	**0.65**
Mining	Life of Mine average*	**3.03**
Process	Process Labour	**0.38**
	Consumables	**1.59**
	Power	**0.72**
	Tailings	**0.02**
	Total Process	**2.70**
	Total Minesite Cash Cost	**$6.39**

* Mining costs LOM average US$1.03/t of material mined.

1.11 Economic Parameters

Capital and operating costs were developed for a 65,000 tpd operation and cashflow forecasts generated to evaluate the economic performance of the Project, testing the sensitivity to a range of metal prices, exchange rates and input prices. Key Project economic parameters are summarized in Table 1-6 below:

Table 1-6: Galore Creek – Economic Parameters Summary

	Base Case (1)	Copper	Gold	Silver
Plant Throughput (tpd)	65,000			
Mine Life (Years)	20			
Ore Tonnage (Mt)	475			
Strip Ratio	2.2:1			
First Six Years of Production:				
Average Grade		0.79%	0.56 g/t	5.18 g/t
Annual Average Metal Production		371 M lbs	302,000 ozs	2.31 M ozs
Life of Mine:				
Average Grade		0.65%	0.36 g/t	4.76 g/t
Annual Average Metal Production		296 M lbs	188,000 ozs	2.02 M ozs
Total Recovered Metal		5.9 B lbs	3.8 M ozs	40.5 M ozs

Notes:

1) Base case uses projected long-term average metal prices of copper at US$1.00/lb, a gold price of US$400/oz and US$6.00/oz for silver.
2) Cash costs include on-site and off-site operating costs, transportation and refining charges, with by-product metal credits.

The updated economic assessment is preliminary in nature, in that it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary assessment will be realized. However, approximately 80% of the in-pit resources included in the financial models are in the Measured and Indicated categories.

1.12 Financial Analysis

A financial analysis using the base case parameters indicates that, using a projected long-term average price for copper of US$1.00/lb, the Galore Creek project could generate a pre-tax rate of return (IRR) of approximately 11.1% and have an undiscounted after-tax Net Present Value (NPV) of US$783 M. Key Project financial results are summarized below in Table 1-7 below:

Table 1-7: Galore Creek – Summary Financial Results

Galore Creek Summary Financial Results		Long-Term Avg Prices	Recent Prices
After-tax Net Present Value (Undiscounted)		$783 M	$3,146 M
After-tax Net Present Value (5% Discount Rate)		$191 M	$1,495 M
Pre-tax IRR (%)		11.1	30.3
After-tax IRR (%)		8.1	23.5
After-tax Payback of Capital (years)		5.2	2.1
After-Tax Net Annual Avg Cash Flow	(Years 1-6)	**$209 M**	**$350 M**
Cumulative After-Tax Net Cash Flow	(Years 1-6)	**$1.26 B**	**$2.1 B**
Capital Costs			
Direct + Indirect Capital Cost	US$	$958 M	
Contingency	US$	$144 M	
Total Capital Cost	US$	$1,101 M	
		Copper	Gold
Total Cash Cost First 6 Years	US$	**$0.36/lb**[4]	**(-300/oz)**[5]
Total Cash Cost Life of Mine	US$	**$0.54/lb**	**(-300/oz)**
Total Co-Product Cash Cost First 6 Years[6]	US$	**$0.57/lb**	**$140/oz**
Total Co-Product Cash Cost Life of Mine	US$	**$0.68/lb**	**$187/oz**

Notes: 1) Base case uses projected long-term average metal prices of copper at $1.00/lb, a gold price of $400/oz and $6.00/oz for silver. 2) Cost estimates reflect a Pre-Feasibility level of accuracy, ±20%. 3) Total Cost Per Tonne includes minesite costs plus long-term transportation and refining cost projections. 4) Cash costs include on-site and off-site operating costs, transportation and refining charges, with by-product credits. 5) Numbers in parenthesis represent total cash costs per ounce of gold for the period calculated using the Gold Institute guidelines. 6) Total cash costs using co-product accoutning methodology.

The PEA evaluated the capital costs, operating and processing costs, taxes and treatment charge for the project. A sensitivity analysis shows that the base case rate of return is most sensitive to changes in metal prices and grades, followed by changes to the operating costs and then to changes in capital costs.

Table 1-8: Financial Analysis and Metal Price Sensitivity (US$M)

Sensitivity Matrix – Metal Prices (all equity case, all NPV and Payback Figures After-tax)						
Cu Price (US$/lb)		**Au/Ag Price (US$/oz)**				
		375/5.5	**400/6.0**	**425/6.5**	**450/7.0**	**475/7.5**
0.9	NPV @ 0% (US$M)	419	484	549	613	677
	NPV @ 5% (US$M)	-19	21	59	98	136
	Pre-tax IRR (%)	6.6	7.5	8.4	9.2	10.1
	After-tax IRR (%)	4.7	5.4	6.0	6.6	7.3
	Payback (years)	6.6	6.3	5.9	5.7	5.4
1.0	NPV @ 0% (US$M)	719	783	847	911	978
	NPV @ 5% (US$M)	152	191	228	266	303
	Pre-tax IRR (%)	10.3	11.1	11.9	12.6	13.4
	After-tax IRR (%)	7.5	8.1	8.7	9.3	9.8
	Payback (years)	5.5	5.2	5.0	4.8	4.6
1.1	NPV @ 0% (US$M)	1017	1081	1145	1209	1273
	NPV @ 5% (US$M)	320	357	395	432	469
	Pre-tax IRR (%)	135	14.2	14.9	15.6	16.3
	After-tax IRR (%)	10.0	10.5	11.1	11.6	12.1
	Payback (years)	4.6	4.4	4.3	4.1	3.9
1.25	NPV @ 0% (US$M)	1464	1528	1592	1656	1720
	NPV @ 5% (US$M)	568	605	642	678	716
	Pre-tax IRR (%)	17.7	18.3	19.0	19.6	20.2
	After-tax IRR (%)	13.3	13.8	14.3	14.8	15.2
	Payback (years)	3.7	3.5	3.4	3.3	3.2
1.5	NPV @ 0% (US$M)	2209	2273	2337	2401	2465
	NPV @ 5% (US$M)	977	1013	1050	1086	1123
	Pre-tax IRR (%)	23.7	24.2	24.8	25.3	25.9
	After-tax IRR (%)	18.0	18.5	18.9	19.3	19.7
	Payback (years)	2.7	2.6	2.6	2.5	2.5
1.75	NPV @ 0% (US$M)	2954	3018	3082	3146	3210
	NPV @ 5% (US$M)	1385	1422	1458	1495	1532
	Pre-tax IRR (%)	28.8	29.3	29.8	30.3	30.8
	After-tax IRR (%)	22.2	22.6	23.1	23.5	23.9
	Payback (years)	2.2	2.2	2.1	2.1	2.0

Note: NPV = Net Present Value using a Discounted Cash Flow Analysis; IRR = Internal Rate of Return.

1.13 Conclusions and Opportunities

The Study shows that the Galore Creek project has the scope to produce an average of 295 M lbs of copper, 188,000 ozs gold and 2.02 M ozs silver, at an estimated total cash cost of US$0.54 per pound of copper, inclusive of by-product credits, over a 20 year mine life. The study is intended to better quantify the project's cost parameters and to focus the additional exploration and detailed engineering work that will be required to advance to a Feasibility Study.

This study considers the resources defined in the Central, Southwest, West Fork and Junction deposits and does not include any upside from the definition of new resources on extensions of the Southwest and Central deposits, or the potential for definition of new resources. In June 2005, NovaGold initiated a drill campaign in excess of 60,000 m on the Galore Creek Project and is utilizing eight diamond drills with the goal to in-fill at roughly 50 x 75 m centres on the Central Zone as well as additional in-fill and step out drilling at West Fork, Southwest and Butte Zones.

Key Upside Opportunities

A number of opportunities have been identified for the project that could result in improved project economics and should be investigated further. These are discussed below.

- **Adding Further Resources.** NovaGold has been highly successful in expanding the known mineralization in and around the Galore Creek valley. Given this success and the fact that the plan laid out in this PEA envisages mining and processing less than half of the mineral resources quantified in the spring 2005 resource estimate, it is logical and recommended to continue to search for new resources in the valley and to better define those resources which are in the inferred category. Enhance project value by extending the initial period of higher grade ore available to the processing facility and deferring the processing of average or lower grade ores.
- **Refinement of Mining Schedules.** The mine schedule provided in this PEA was produced by balancing the need to mine higher grades early to support project payback and maintaining a logical sequence in the open pits given a certain sized truck and shovel fleet. Emphasis in this analysis was placed on the first 5-7 years of mining and production and as the project advances toward feasibility. It is recommended that this schedule be further refined with the objectives of improving the grade delivered to the processing facility (particularly in years 8 and 9), and of smoothing the copper concentrate tonnage level in the first 7 years to enhance the ability of the project to obtain the best terms for smelting.
- **Increased Production Rate.** The economics may be enhanced by increasing the throughput through the existing circuit, particularly in the early years when the feed material is mostly broken rock, and by adding additional equipment in order to mine and mill at a higher rate. .
- **Tailing and waste rock storage facility.** The facility is currently designed to hold all the 1.5 billion tonnes of tailings and waste rock. It is expected that further testing and modeling of the waste rock may well show that the quantity of PAG rock is significantly less than currently modeled. This would result in either a downsizing of the existing design or an increase in the total amount of material that could be mined without incurring additional storage costs.

- **Construction Cost and Schedule.** NovaGold is continuing to study ways of reducing the capital costs and improving the construction schedule. Studies are ongoing to establish more accurately all major areas of costs, and to see whether further optimizations can be made to the construction timeline.
- **Site power generation** NovaGold commissioned a conceptual level study of the feasibility of generation of electrical power on site, by running the flow from the West diversion ditch through a turbine. This would essentially be a Run of River (ROR) power plant. Initial indications are that 20 to 30 MW of power could be generated, at a capital investment in the order of US$23M to US$30M. This system could feed power back into the BCH grid at the cessation of mining operations.

This Preliminary Economic Assessment Study was completed under the direction of Paul Hosford, P.Eng., Project Manager for Hatch Ltd., an independent Qualified Person as defined by National Instrument 43-101.

2. Introduction

2.1 Terms of Reference

NovaGold Resources Inc. (NovaGold) commissioned Hatch Ltd. (Hatch) in November 2003 to carry out a PEA Study of the Galore Creek copper-gold property. That study, released in August 2004, indicated that the project merited further more detailed evaluation. In July 2004, NovaGold commissioned Hatch to carry out further engineering studies to support a Pre-Feasibility level study (PFS) for the project. During the course of the study, it became evident that Inferred resources would be included in the pit resources and consequently did not meet the CIM criteria for a PFS, which stipulated that reserves can only be estimated using Measured and Indicated resource category.

This study is therefore termed an updated Preliminary Economic Assessment (PEA), although the engineering studies and costs have been developed to PFS level. Two main project configurations relating to the North and South access corridors were defined in the 2004 PEA and these were refined and studied further as this updated study was being prepared. During the course of the PEA, the North option was selected as the project case to be carried forward into feasibility design and environmental assessment. The details of the South option are carried in the PEA, as a requirement for evaluation of alternatives for the Environmental Assessment.

The scope of this Study includes all aspects of the project including:

- resource estimation of the Central Zone, South West Zone, Junction Zone and West Fork Zone deposits,
- mine design, scheduling and reserve estimation,
- metallurgical testwork, evaluation and process design,
- assessment of access road routes,
- assessment of tailings and waste rock storage sites,
- development of a preliminary closure plan,
- assessment of power supply and powerline route options,
- assessment of concentrate transportation options,
- development of a site water balance for the project,
- assessment of the logistical and construction approaches for the development of the project,
- development of capital and operating cost estimates to an accuracy level consistent with a PFS, of approximately ±20%, for both the North and South options,
- preliminary assessment of project economics and sensitivities,
- discussion of the risks and opportunities for the project.

Hatch and NovaGold commissioned a number of experienced consultants to carry out specific technical and economic studies for the PEA. These are listed below together with their respective responsibilities:

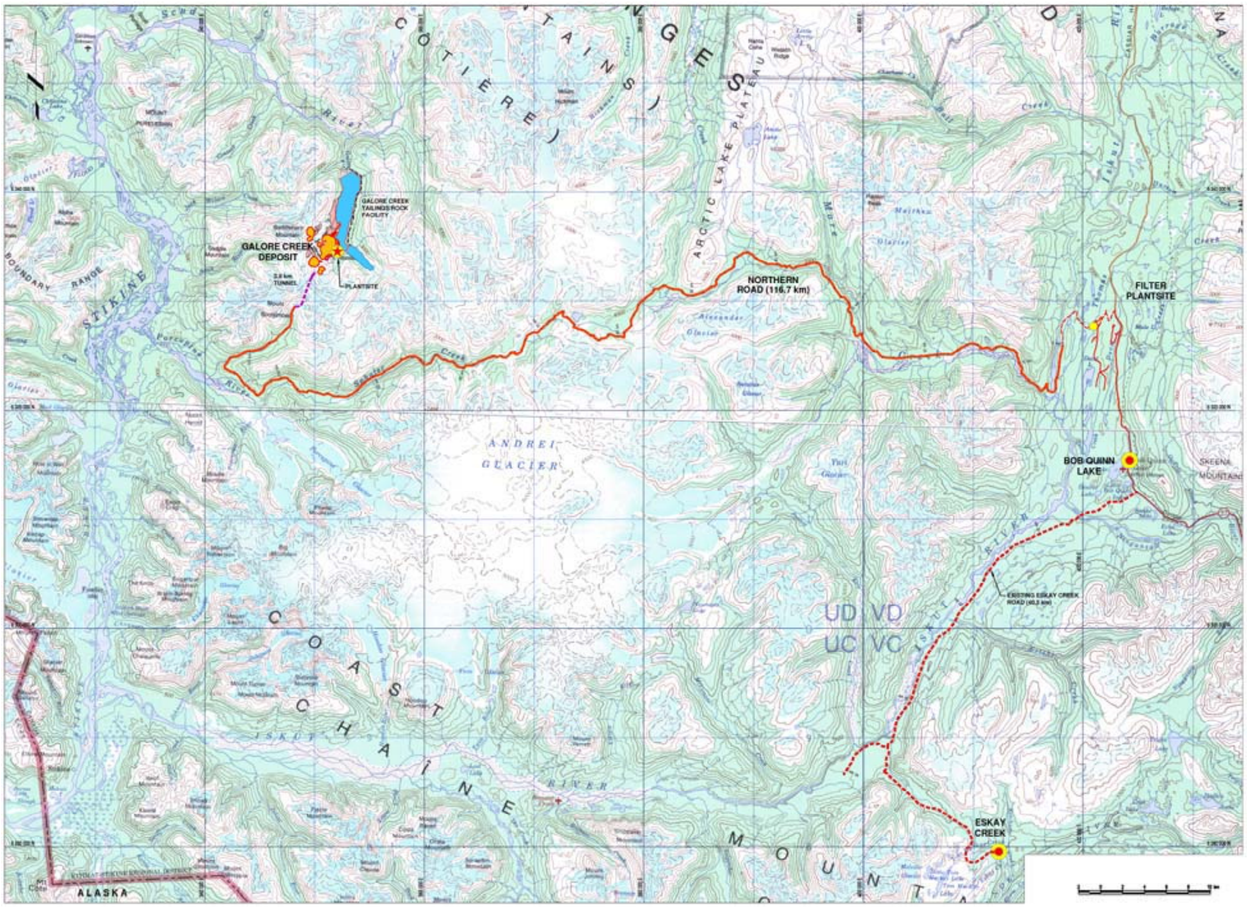
Subconsultants		**Responsibilities**
Bruce Geotechnical Consultants	BGC	Geotechnical assessment and design for tailings and waste rock storage Open pit slope stability Geohazard evaluation for road, powerline routes and site
McElhanney Engineering	ME	Assessment of access road routes and PFS design
G&T Metallurgical	G&T	Metallurgical testwork
Hatch Mott MacDonald	HMM	Assessment of access tunnel routes and PFS design
Ian Hayward & Associates	IHI	Assessment of powerline routes and PFS design
GR Technical Services Ltd.	GRTech	Reserve estimation, mine design and scheduling
Pipeline Systems Incorporated	PSI	Scoping study assessment of concentrate pumping systems
Rescan Tahltan Environmental Consultants	RTEC	Stikine River bathymetric survey and environmental permitting
Sandwell Engineering Inc.	Sandwell	Assessment of options for barging on the Stikine River
Neil Seldon & Associates	NSA	Assessment of metal prices, concentrate sales and treatment charge forecasts
Giroux Consultants Ltd.	Giroux	Resource modelling, geostatistics
MinnovEX Technologies	MinnovEX	Grinding circuit design
LPS Aviation	LPS	Aerodrome assessment
British Columbia Transmission Corporation	BCTC	System planning and performance assessment Assessment of transmission options

From June 2004, NovaGold initiated an extensive fieldwork program to develop the baseline studies for the project, using the services of RTEC. Substantial field engineering programs were carried out at site over the period June 2004 to July 2005, to generate the data to support the PFS engineering studies. This work included:

- geological core drilling
- geotechnical core drilling, trenching and testing
- aerial photography and detailed mapping
- weather data measurement
- metallurgical testing
- environmental testing of waste rock and tailings samples

2.2 Project Configuration Options

Figure 2-1: Project Access Map



The August 2004 Preliminary Economic Assessment of the Galore Creek project identified two main access routes, which were intimately tied to the locations of the project facilities. These are summarized as follows:

- North route: which ran west from the existing Highway 37, followed More Creek Valley to a plant site located near Round Lake at the headwaters of More Creek. The mine site, was accessed by a 14 km long, tunnel with a northern portal in the east fork of Galore Creek. Ore was to be conveyed approximately 25 km from a crusher station near the open pits through the tunnel to the plant site. A tailings dam site was located near the plant site in More Creek. A waste dam site was located in the Galore Valley. From the initial assessment, it was decided that a two lane concentrate haul road access west through Sphaler Creek was not feasible due to the steep terrain.
- South route: which ran west from a point on the existing Eskay Creek mine road, along the Iskut River then north along the Stikine River and finally via a 4 km long, tunnel into the Galore Valley. The plant site was located in the Galore Valley with tailings and waste stored in a single dam in the Galore Valley.

At the time the study indicated that the South route was the preferred approach for further evaluation The Study also indicated that the overall project economics were more attractive at throughput rates higher than 30,000 tpd.

Further engineering studies on the access routes were carried out in 3Q and 4Q 2004 as part of the PEA to provide an improved basis on which to select the most advantageous option with respect to overall project economics and environmental aspects. The study work included:

- fieldwork and preliminary survey by ME on both access routes, to a PFS level.
- geohazard investigations by BGC of both access routes, including assessments of possible mitigation measures.
- preliminary fieldwork to assess powerline routing options by IHI to a PFS level.
- structural geological mapping of the proposed tunnel alignment options and some testwork by HMM.
- evaluation of increased throughput.
- PFS level fieldwork and an updated scoping level assessment of the tailings and waste dam requirements for both North and South options by BGC.

The major changes to the project configuration and costs considered at that time are summarized as follows:

- North option: a shorter, 11 km tunnel alignment into Galore Valley was identified, with a south portal site located in the upper Sphaler Valley.
- South option: road relocated along south side of Porcupine to a narrower crossing upstream and selection of Scotsimpson as the most suitable location for road tunnel access into Galore Valley.
- A conservative assumption was made that 65% of the waste rock was Potentially Acid Generating (PAG), which lead to the selection of a large water retaining dam in Galore Valley so that at closure, all the waste rock and tailings would be flooded to prevent metal leaching.
- Preliminary sizing and significant costs were added to allow for water diversion of creek waters from the tailings and waste rock dams and open pits.
- Preliminary testing of rock types similar to those in the proposed tunnel alignment indicated that the tunnel rock would likely be harder, increasing the construction time and costs of the tunnel.

Prior to making a final selection of the access route, it was decided to re-examine the feasibility of extending the North road west through Sphaler Creek to connect to the proposed road tunnel site in Scotsimpson Valley. Further helicopter reconnaissance, geo-hazard assessments, mapping and engineering studies were undertaken in 1Q and 2Q 2005, which indicated that it was technically feasible and was judged most practical in combination with a concentrate pipeline system to pump concentrate from the plant site to a remote filter plant located near Bob Quinn. The pipeline will significantly reduce the traffic load on the North road route.

A scoping level study was carried out in 2Q 2005 by PSI, to examine the feasibility and costs for a system to pump concentrate from the plant site to a filter plant located near Bob Quinn. Hatch completed a trade-off study to assess the impact of the options of a concentrate pipeline routed along the North access route to a remote filter plant, compared to an all truck haul of concentrate along the South access route. The results indicate that the North concentrate pumping options have significantly higher capital cost, but lower operating costs.

This North option, with road access through Sphaler Creek and a concentrate pipeline to a filter plant near Bob Quinn, was selected by NovaGold as the basis for further feasibility engineering and environmental studies. The selection of the North over the South option resulted from the following main considerations:

- road access through Sphaler Creek was assessed as feasible, in concert with a concentrate pipeline system, which removed the previous requirement for a long tunnel to access Galore Valley,
- the First Nations peoples stated a preference for the North route,
- comparative analysis indicated that the economic performance of the North and South options were similar.

In summary, the project configurations selected for the PEA are described as follows, and shown on Figure 2-1.

The access road runs west from Highway 37, approximately 8 km north of Bob Quinn, along the More Creek Valley, west down Sphaler Creek Valley, north up Scotsimpson Creek Valley to a 4 km long, 6.8 m wide tunnel. The plant site, tailings, and waste rock storage dams are all located in Galore Creek Valley. Concentrate slurry is pumped from the plant site east along the road alignment to a filter plant located near the mine access road intersection with Highway 37. Filtered concentrate is hauled by truck from the filter plant to the port at Stewart, a distance by road of approximately 200 km.

2.3 Abbreviations, Acronyms and Units of Measure

2.3.1 Units of Measure

Unit	Abbreviation
American Dollar	US$
Canadian Dollar	$
Centigrade	°C
centimeter	cm
cubic metre	m^3
day	d
dry metric tonne	dmt
foot/feet	ft
gram	g
gram/litre	g/L

Unit	Abbreviation
gram/tonne	g/t
hour	hr
kilogram	kg
kilogram per tonne	kg/t
kilo tonne	kt
kilometre	km
kilopascal	kPa
kilovolt	kV
kilovolt amp	kVA
kilowatt	kW
kilowatt hour	kWh
litre	L
liter per second	L/s
megawatt	MW
metre	m
metre per hour	m/h
metre per second	m/s
metric tonne	t
metric tonne per day	tpd
metric tonne per hour	tph
micron	µm
milligram	mg
milligram per litre	mg/L
millimeter	mm
million	M
million tonnes	Mt
part per million	ppm
percent	%
second	s

Unit	Abbreviation
square metres	m^2
troy ounce	oz
US gallon per minute	usgpm
wet metric tonne	wmt
Work Index	WI
year	yr

2.3.2 Acronyms and Abbreviations

Abbreviation	Acronyms
ADFG	Alaska Department of Fish and Game
ADIS	Automated Digital Imaging System
Ag	Silver
ASC	Aluminium Standard
Au	Gold
BCH	British Columbia Hydro Corporation
BCTC	British Columbia Transmission Corporation
BFA	Bench Face Angle
BGC	Bruce Geotechnical Consultants
CCA	Capital Cost Allowances
CDE	Canadian Development Expenses
CEA	Cumulative Expenditure Account
CEE	Canadian Exploration Expenses
Cu	Copper
DFO	Department of Fisheries and Oceans
EIA	Environmental Impact Assessment
EMPs	Environmental Management Plans
G&T	G&T Metallurgical
Giroux	Giroux Consultants Ltd.

Abbreviation	Acronyms
GRTech	GR Technical Services Ltd.
HMM	Hatch Mott MacDonald
IHI	Ian Hayward & Associates
IPP	Independent Power Producer
IRA	Inter-Ramp Angle
IRR	Internal Rate of Return
ISO	International Organization of Standardization
ITC	Investment Tax Credit
LCT	Large Corporation Tax
LG	Lerchs-Grossman
LOM	Life of Mine
LPS	LPS Aviation
LRMP	Cassiar Iskut-Stikine Land and Resource Management Plan
MCE	Maximum Credible Earthquake
MDE	Maximum Design Earthquake
ME	McElhanney Engineering Ltd.
MII	Measured, Indicated & Inferred Resources
MinnovEX	MinnovEX Technologies
ML/ARD	Metal Leaching/Acid Rock Drainage
NBCC	National Building Code of Canada
NMFS	US National Marine Fisheries Service
NPAG	Non-potentially Acid Generating
NPC	Net Present Cost
NPV	Net Present Value
NSA	Neil Seldon & Associates
NSP	Net Smelter Metal Prices
NSR	Net Smelter Return
PAG	Potentially Acid Generating
PEA	Preliminary Economic Assessment

Abbreviation	Acronyms
PFS	Pre-Feasibility Study
PGA	Peak Ground Acceleration
PMF	Probable Maximum Flood
PSA	Pit Slope Angle
PSC	Pacific Salmon Commission
PSI	Pipeline Systems Incorporated
RIC	British Columbia Provincial Resource Inventory Committee
ROM	Run of Mine
RTEC	Rescan Tahltan Environmental Consultants
SABC	SAG, Ball Mill and Pebble Crushing circuit
SAG	Semi Autogenous Grinding
Sandwell	Sandwell Engineering Inc.
SCADA	Supervisory Control and Data Acquisition System
SPI	SAG Power Index
SPT	Standard Penetration Testing
TNDC	Tahltan Nation Development Corporation
UCS	Uniaxial Compressive Strength
VECs	Valued Ecosystem Components
vpd	Vehicles per day
vph	Vehicles per hour

2.4 Currency

The currency basis presented throughout the PEA is 2Q 2005 Canadian dollars, unless otherwise noted and shown as $.

3. Property Description, Location, History and Ownership

3.1 Introduction

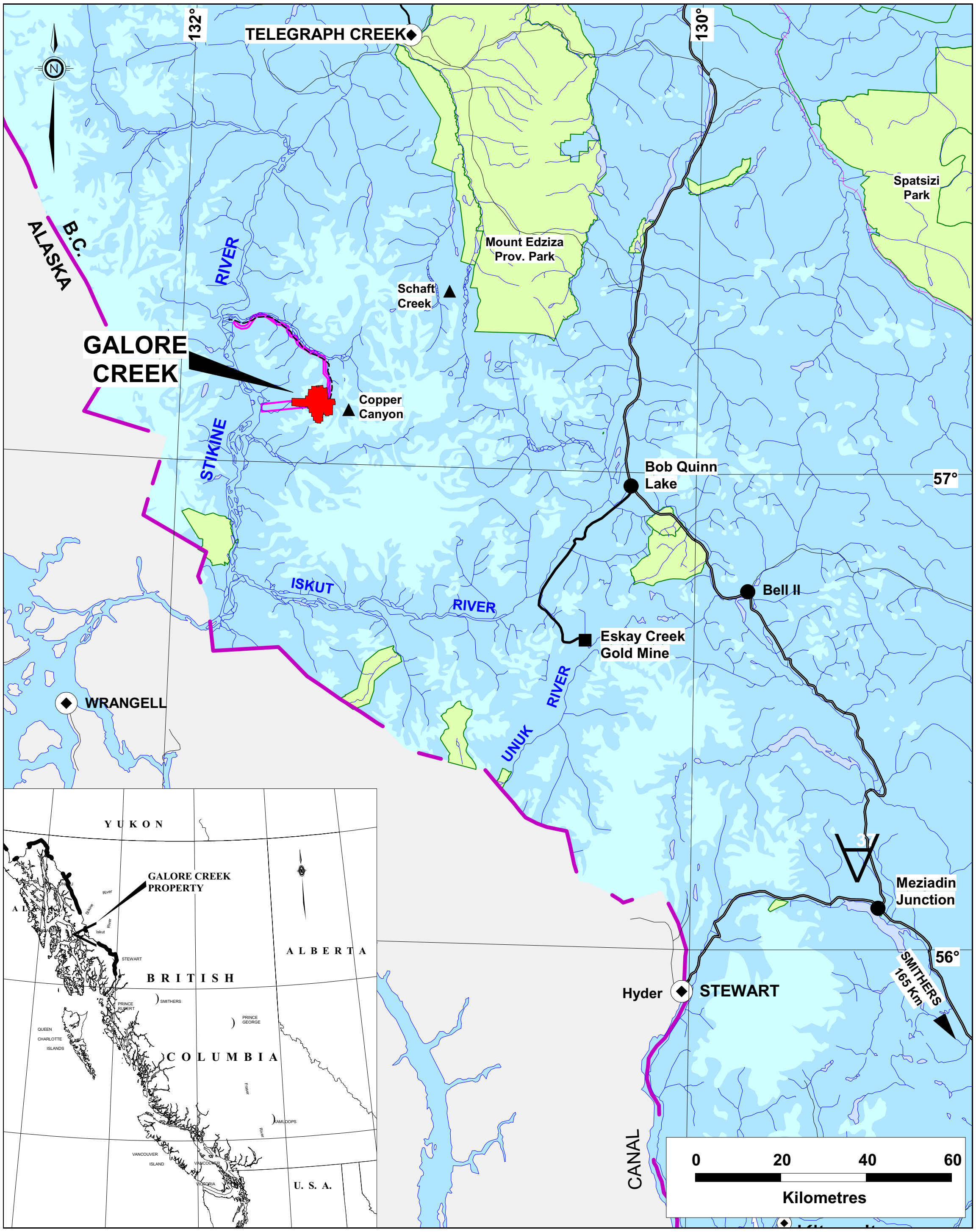
The Galore Creek property is located in mountainous terrain of northwestern British Columbia, approximately 1,000 km northwest of Vancouver, approximately 200 km north of the tidewater port of Stewart B.C. and 100 km northeast of the community of Wrangell, Alaska. The town of Smithers, 370 km southeast, is the nearest major supply centre and has an airport with regularly scheduled flights to and from Vancouver. Figure 3-1 shows the general location of the property in northwestern BC. The mining claims lie at the headwaters of Galore Creek, a tributary of the Scud River, which flows into the Stikine River. The property is centred at latitude 57° 07'30"N and longitude 131°27'W. It occurs within the Liard Mining Division and straddles the boundary between NTS map sheets 104G/3 and 104G/4.

The project falls within the boundaries of the Cassiar Iskut-Stikine Land and Resource Management Plan (LRMP), which was finalized in May 2000. The approved plan supports further exploration and development of the region's mineral resources by providing information to be considered during the permitting and impact assessment processes.

The Galore Creek claims fall within the broader traditional territory of the Tahltan First Nation. Mining has long been an important segment of the Tahltan economy. The Tahltans participated in the exploration, construction, operations and reclamation of the recently closed Golden Bear Mine north of Dease Lake. The largest mining operation within the Tahltan Traditional Territories at present is the Eskay Creek gold mine located 80 km southeast of Galore Creek. Approximately 35% of the Eskay Creek work force (including contractors) are Tahltan First Nations people.

The Tahltan Nation Development Corporation (TNDC) carries out road construction, maintenance and provides services for projects in the region. Services provided to the mining industry through TNDC include catering, trucking (through a JV with Arrow Transport) and environmental services (through a JV with Rescan Environmental Services), NovaGold is building a healthy relationship with the Tahltan and is using TNDC enterprises to support its 2005 field activities. NovaGold and the Tahltan are currently working to conclude an agreement formalizing Galore Creek's commitments to training, employment and community consultation as well as the Tahltan's support in all aspects of approval and permitting.

Figure 3-1: General Location Map



3.2 Accessibility, Local Resources, Infrastructure and Physiography

3.2.1 Location and Terrain

The terrain over the central and northern portions of the property is gentle and rolling and the surrounding topography is characterized by rugged mountains. Elevations on the property range from 500 to 2080 m above sea level with the majority of the main Central deposit starting at below 800 meters elevation. The elevation of the tree line is variable but alpine vegetation predominates above the 1100 m level. Below that, forests are made up of Balsam fir, Sitka spruce and a few cedars. Higher up the valley, the moraines are bare to sparsely overgrown by sub-alpine vegetation.

The Galore Creek Valley is a U-shaped glacially scoured valley with glacial and glaciolacustrine deposits covering the lower elevation slopes. The surrounding terrain is mountainous and at high elevations covered by glaciers and ice fields. Glaciers exist in the headwaters of the East and West forks of Galore Creek, but are currently retreating. The steep upper slopes are generally exposed bedrock.

Helicopter is the present means of access to the Galore Creek property. The Bob Quinn airstrip on Highway 37 is located approximately 75 km east of Galore Creek and is the primary staging area for project mobilization and demobilization.

Galore Creek and the Scud River are part of the tributary system of the Stikine River a waterway that drains an area of 49,000 square kilometres. In the past, the river was used by shallow draft barges and riverboats to transport goods from Wrangell, Alaska to Telegraph Creek, British Columbia, a distance of 302 kilometres. The river is navigable for this type of watercraft from mid May to October. In the 1960's Kennecott constructed 48 km of tote road from the mouth of the Scud River to the Galore Creek camp. This road would require major repair for use by the project and no plans exist to conduct this work at present.

3.2.2 Climate

In general, the project area is characterized by cold winters and short, cool, summers. More than 70% of Galore's yearly precipitation is received in the fall and winter from September through to February. The majority of this precipitation is in the form of snowfall.

Climate data for the project area was compiled from a recently installed weather station and one previous weather station. The most representative climatic data is the 67 months of data from Galore Creek Station No. 1203046. This weather station is no longer active but was located near the airstrip within the Central Zone deposit around elevation 789 m.

Average monthly temperatures at the mine site range from -13°C in January to 11°C in July. The extreme temperatures range from -18°C in January 1968 to 21°C in July 1965. Precipitation begins to fall as snow in early October and continues until the end of May. The estimated long-term average annual precipitation for the Galore Creek mine site is approximately 2300 mm rainfall equivalent. June and July tend to receive the least amount of precipitation on an annual basis (typically 40 to 60 mm of rain per month).

On average, annual total precipitation will be comprised of about 35% rainfall and 65% snowfall. Rainfall occurs typically in May through October with snowfall in October or November through March or April. April and/or October are often transitional months, which could experience both rainfall and snowfall.

3.3 History

The property history is summarized as follows:

- **1955:** Mineralization first discovered in the upper Galore Creek Valley by M. Monson and W. Bucholz while prospecting for Hudson Bay Exploration and Development Company Limited. Staking and sampling were completed in the area in 1955.
- **1959:** Kennco Explorations (Western) Limited carried out reconnaissance stream silt surveys in the Stikine River area. Results from this work prompted Kennco to stake mineral claims the following year around the 16 claims owned by Hudson Bay. Four of the original claims were subsequently optioned by Consolidated Mining and Smelting Company of Canada Limited from W. Bucholz. Late in 1962 the three companies agreed to participate jointly in future exploration work. As a result, Stikine Copper Limited was incorporated in 1963.
- **1960-1968**: Property operated by Kennco Exploration (Western) Ltd and exploration work during this period included 53,164 m of diamond drilling in 235 holes and 807 m of tunnelling in 2 adits.
- **1972-1973:** Property operated by Hudson Bay Smelting and 25,352 m of diamond drilling was completed in 111 holes; a further 5,310 m of diamond drilling was completed in 24 holes in 1976.
- **1989:** Mingold Resources Inc. (an affiliated company of Hudson Bay's) operated the property to investigate its gold potential. A further 1225 m of diamond drilling in 18 holes was done by Mingold in 1990.
- **1991:** Property operated by Kennecott and completed 18,380 m of diamond drilling in 49 holes. An airborne geophysics survey and over 90 line km in an induced polarization (IP) survey were also completed.
- **1966 (Aug) –1967 (Jan):** Haste Mining drove 799 m of 2 m x 2 m underground development into the Central Zone to collect a bulk sample. The walls of the development were chip or channel sampled and the results were compared to nearby drill hole assay results. At about the same time, a 51 m of 1.2 m x 1.2 m development was driven into the North Junction Zone.
- **1992:** Mine Reserve Associates, Inc. completed a resource model in 1992 for Kennecott Exploration.
- **2002:** Kennecott re-classified the 1992 mineral resource estimate to comply with current industry standards. Values used were $10/tonne in-situ metal value as a cut-off grade based on US$0.80/lb copper and US$320/oz gold prices. Kennecott estimated an Indicated Resource of 243.2 million tonnes grading 0.75% copper and 0.45g/t gold containing 3.6 million ounces of gold and 4.0 billion pounds of copper. In addition, an Inferred Resource was estimated to be 70.6 million tonnes grading 0.59% copper and 0.63g/t gold containing 1.4 million ounces of gold and 920 million pounds of copper. Silver was not included in the 1992 resource model. This resource estimate does not conform to NI 43-101 standards and is reported as a historical estimate only.

- **2003:** SpectrumGold Inc. (now NovaGold Canada Inc.) entered into option agreement to acquire a 100% interest in the Galore Creek property from Stikine Copper Limited, a company owned by QIT-FER et Titane Inc. and Hudson Bay Mining and Smelting Co. Limited. The claims on the property are wholly owned by Stikine Copper Ltd., controlled by QIT-Fer et Titane Inc. (55%) and Hudson Bay Mining and Smelting Co. Limited (45%).

  SpectrumGold carried out a 10 hole 2,950 m diamond drill program on the property in September and October of 2003. The work program was directed toward confirming grades of copper and gold mineralization defined by previous drilling in the Galore Creek deposit. Results from the drill program confirmed the presence of high-grade gold and copper mineralization over bulk mineable widths.

- **2004:** NovaGold completed an exploration program that was designed to confirm results from past exploration, to extend the limit of known mineralization and to advance the geological understanding of the system, included geophysics and drilling. The 2004 diamond-drilling program included 25,976 m of NQ sized core recovered from 79 drill holes. In total, the Central Zone includes 321 drill holes; 44 drill holes in the Southwest Zone; 51 holes in the Junction Zone, and 36 holes in the West Fork Zone. In 2005, a 60,000 metre program is underway to complete the drilling necessary for a Feasibility study in 2006.

3.4 Seismicity

The Galore Creek area is located in a moderately high seismic zone. The national seismic hazard map produced by the Geological Survey of Canada for use in the 1995 National Building Code of Canada, indicates that the project is located in acceleration zone 2, characterized by a peak horizontal ground acceleration (PGA) of 0.8 to 0.11g with a 10% chance of exceedance in 50 years (1 in 475). Revised seismic hazard maps for incorporation into the 2005 National Building Code of Canada show that the site has a PGA of approximately 0.1 to 0.2g with a 2% chance of exceedance in 50 years (1 in 2,475). The maximum credible earthquake (MCE) peak ground acceleration will likely be between 0.2 to 0.3 g.

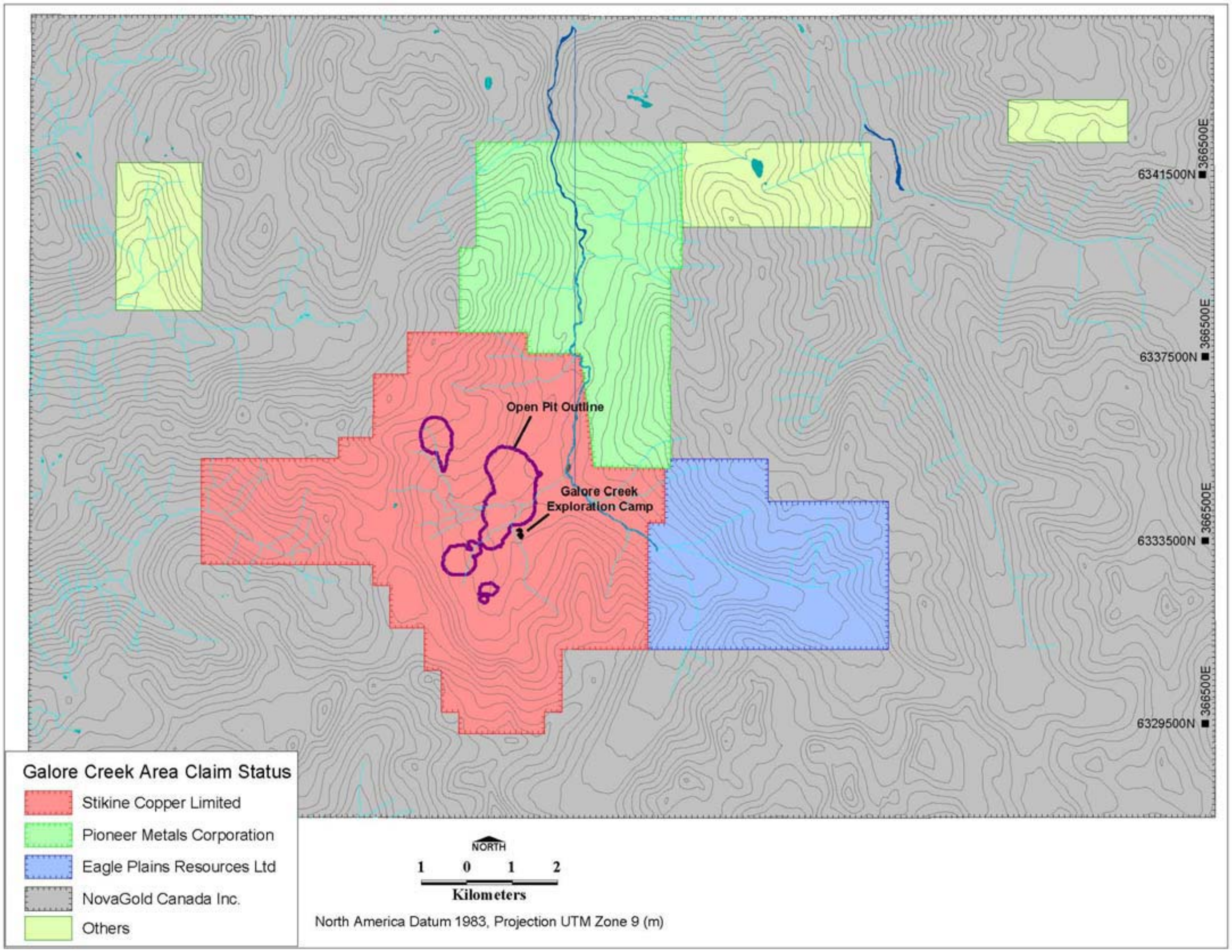
3.5 Property Ownership

The claims that cover the property are wholly owned by Stikine Copper Ltd. This company was incorporated in 1963 and is presently controlled by QIT-Fer et Titane Inc (55%) and Hudson Bay Mining and Smelting Co, Limited (45%). The property originally was comprised of 292 two-post claims and in July 2005 these legacy claims were converted to six cell claims to hold an area of 5,110 ha. In August 2003, SpectrumGold (now NovaGold Canada) entered into an option agreement to acquire a 100% interest in the project by completing a Pre-Feasibility study on the project and making payments to the parties totalling US$20.3 million within a period of 8 years. Payments of US$0.3 million in aggregate are required over the first three years of the option, with the remaining US$20 million to be paid over the following 5 years. There will be no retained interests, royalties or back-in rights on the project. In July of 2004, NovaGold acquired 100% of SpectrumGold through a plan of arrangement and SpectrumGold was renamed NovaGold Canada Inc.

NovaGold recently acquired additional claims in the Galore Creek property area through staking, purchase, and option agreements. A total of 16,850 ha of claims have been staked by NovaGold in the area to the southeast and east of the Galore Creek property. NovaGold has also acquired a 100% interest in the Jack, Sphaler Creek, and Paydirt properties, which cover a total area of 1,725 ha. In addition NovaGold has entered into an option agreement with Pioneer Metals Corporation on the Grace property contiguous with the Galore Creek Claims to the north. NovaGold has also entered into an option agreement with Eagle Plains Resources Ltd. on the Copper Canyon property to the east. Collectively, these claims cover a total of 4,275 ha.

Figure 3-2 illustrates the claim boundaries for the original claims of the Galore Creek property held as well as other regional claims controlled or managed by NovaGold as of mid-2004.

Figure 3-2: Claim Location



4. Geology and Resource Estimate

4.1 Summary

The following statement of the estimated mineral resources of the four Galore Creek zones incorporates results from the 2004 drilling campaign and has benefited from a revised geological interpretation. The model integrates 106,781 m of drilling in 452 core holes with a total of 36,866 assays. Interpretation of the geological data and division of the deposits into domains for estimation was supported by extensive geological mapping, airborne and terrestrial geophysical surveys. The estimates are based on a 3-dimensional computer block model with grades interpolated into individual 25 m by 25 m by 15 m high blocks. The grade interpolation used ordinary kriging procedures and mineralization was composited on 5 m intervals with high-grade samples capped based on lognormal probability plots. Table 4-1 summarizes the estimate for measured and indicated resources, while Table 4-2 summarizes the inferred resources.

The copper equivalent (CuEq) calculations use metal prices of US$375/oz for gold, US$5.50/oz for silver and US$0.90/lb for copper. Copper equivalent calculations reflect gross metal content that have been adjusted for metallurgical recoveries based on the following criteria: copper recovery = (%Cu - 0.06)/%Cu with a minimum of 50% and maximum of 95%; gold recovery = (Au g/t – 0.14)/Au g/t with a minimum of 30% and maximum of 80%; and silver recovery = 80%.

Effective May 2005, and based on an equivalent copper cut off grade of 0.35%, Galore Creek, an alkaline porphyry-style copper-gold-silver deposit, has a combined Measured and Indicated resource of **516.7 Mt containing 0.59% copper, 0.36g/t gold, and 4.5g/t silver**.

Figure 4-1 shows the overall grade tonnage curve for the four Galore Creek deposits combined.

Figure 4-1: Overall Grade Tonnage Curves

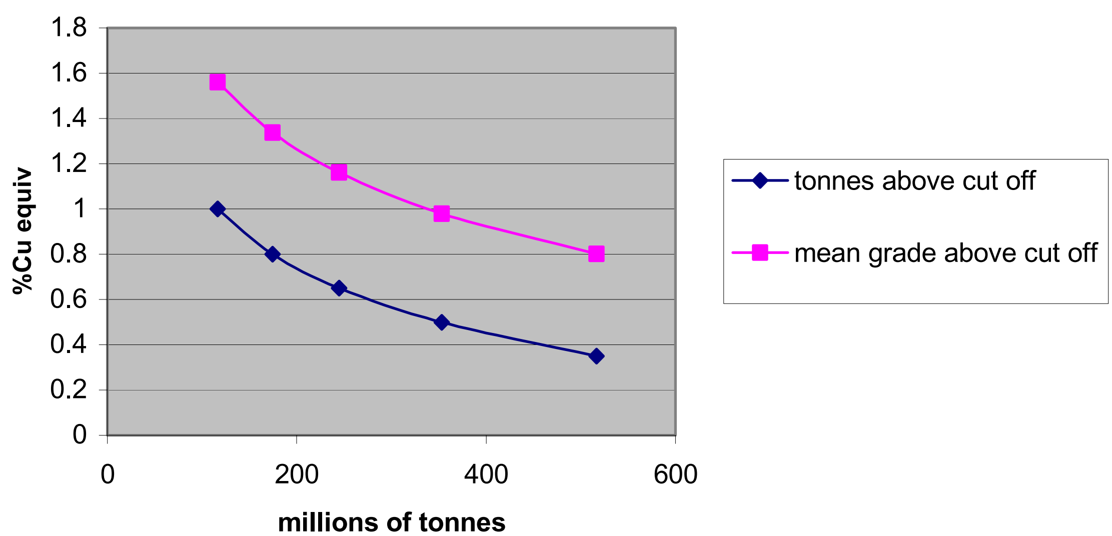


Table 4-1: Summary for Measured Plus Indicated Resource (M+I)

ZONE	Cut-off (CuEq%)	Tonnes > Cut-off (tonnes)	Grade > Cut-off				Million lbs. of Cu	Million ozs of Au	Million ozs of Ag
			Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)			
Central M+I	0.35	423,900,000	0.614	0.302	4.681	0.780	5739.05	4.12	63.80
SW M+I	0.35	47,700,000	0.447	0.818	3.041	0.972	470.15	1.25	4.66
Total Central/SW		471,600,000	0.597	0.354	4.515	0.799	6209.20	5.37	68.46
Junction M+I	0.35	30,000,000	0.589	0.412	4.777	0.837	389.62	0.40	4.61
WF M+I	0.35	15,100,000	0.580	0.379	4.793	0.798	193.11	0.18	2.33
TOTAL		**516,700,000**	**0.596**	**0.358**	**4.538**	**0.802**	**6791.94**	**5.95**	**75.39**
Central M+I	0.50	290,100,000	0.738	0.376	5.421	0.947	4720.77	3.51	50.56
SW M+I	0.50	34,100,000	0.553	1.005	3.523	1.194	415.80	1.10	3.86
Total Central/SW		324,200,000	0.719	0.442	5.221	0.973	5136.57	4.61	54.42
Junction M+I	0.50	18,300,000	0.794	0.513	6.344	1.110	320.39	0.30	3.73
WF M+I	0.50	10,800,000	0.694	0.438	5.675	0.949	165.27	0.15	1.97
TOTAL		**353,300,000**	**0.722**	**0.446**	**5.293**	**0.979**	**5622.23**	**5.06**	**60.13**
Central M+I	1.00	90,100,000	1.100	0.724	7.257	1.511	2185.38	2.10	21.02
SW M+I	1.00	16,100,000	0.843	1.477	4.787	1.738	299.27	0.76	2.48
Total Central/SW		106,200,000	1.061	0.838	6.883	1.545	2484.64	2.86	23.50
Junction M+I	1.00	7,100,000	1.347	0.713	10.617	1.795	210.88	0.16	2.42
WF M+I	1.00	3,300,000	1.118	0.631	9.930	1.514	81.35	0.07	1.05
TOTAL		**116,600,000**	**1.080**	**0.825**	**7.196**	**1.560**	**2776.88**	**3.09**	**26.98**

Table 4-2: Summary for Inferred Resource

ZONE	Cut-off (CuEq%)	Tonnes > Cut-off (tonnes)	Grade > Cut-off				Million lbs. of Cu	Million ozs of Au	Million ozs of Ag
			Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)			
Central	0.35	173,600,000	0.467	0.283	3.425	0.623	1787.62	1.58	19.12
SW	0.35	122,900,000	0.314	0.556	2.274	0.695	850.92	2.20	8.99
Total Central/SW		296,500,000	0.404	0.396	2.948	0.653	2638.54	3.78	28.10
Junction	0.35	71,600,000	0.532	0.286	3.334	0.685	839.91	0.66	7.67
WF	0.35	45,400,000	0.466	0.339	4.986	0.665	466.50	0.50	7.28
CC	0.35	164,800,000	0.351	0.539	7.154	0.735	1275.00	2.86	37.91
TOTAL Inferred		**578,300,000**	**0.409**	**0.419**	**4.354**	**0.681**	**5219.95**	**7.79**	**80.97**
Central	0.50	96,000,000	0.576	0.380	4.085	0.790	1219.28	1.17	12.61
SW	0.50	72,100,000	0.407	0.706	2.617	0.895	647.05	1.64	6.07
Total Central/SW		168,100,000	0.504	0.520	3.455	0.835	1866.33	2.81	18.67
Junction	0.50	34,600,000	0.759	0.381	4.559	0.977	579.06	0.42	5.07
WF	0.50	29,700,000	0.553	0.403	6.181	0.797	362.15	0.39	5.90
CC	0.50	116,100,000	0.408	0.641	8.300	0.870	1044.00	2.39	30.98
TOTAL Inferred		**348,500,000**	**0.501**	**0.536**	**5.411**	**0.858**	**3851.54**	**6.01**	**60.63**
Central	1.00	18,200,000	0.781	0.929	5.193	1.312	313.42	0.54	3.04
SW	1.00	18,400,000	0.714	1.092	3.245	1.445	289.68	0.65	1.92
Total Central/SW		36,600,000	0.747	1.011	4.214	1.379	603.11	1.19	4.96
Junction	1.00	10,500,000	1.334	0.588	7.533	1.686	308.85	0.20	2.54
WF	1.00	4,900,000	0.841	0.603	9.948	1.219	90.87	0.10	1.57
CC	1.00	29,200,000	0.651	1.136	13.030	1.446	419.00	1.07	12.23
TOTAL Inferred		**81,200,000**	**0.794**	**0.977**	**8.159**	**1.433**	**1421.83**	**2.55**	**21.30**

4.2 Introduction

The mineral resource estimate was completed by Qualified Persons and reported separately in accordance with National Instrument 43 –101. The following is a synopsis of the NI 43-101 Technical Report "Geological And Resource Potential Of The Galore Creek Property", 18 May 2005, prepared by Hatch, GR Tech and Giroux. The reader should consult the Technical Report if greater detail is required. The qualified persons responsible for the resource estimate and technical report are: James H. Gray, P.Eng, Robert J, Morris, P.Geo, and Garry H. Giroux, P.Eng.

4.3 Geological Setting

The Galore Creek property is an alkalic-porphyry-copper-gold-silver system located within the Stikine Arch structure in Northwestern BC. It is one of a number of Cu-Mo and Cu-Au-Ag porphyry deposits extending along the Intermountain Belt from south of the British Columbia-Washington border along the Quesnel Trough through the Stikine region and into the Whitehorse Trough, Yukon Territory. Several major alkalic porphyry deposits are known along this belt ranging in age from 175 to 201 million years including Copper Mountain-Ingerbelle, Afton, Cariboo Bell, Lorraine and Gnat Lake (Barr, D. A., et. al., CIM Spec. Vol. 15, 1976).

These deposits tend to occur in regions of fault intersections and are controlled by fractured and/or brecciated zones. Deposits typically show extensive alteration products and sulphides and often lack the classic zoning of calc-alkaline porphyries due to the absence or poorly developed nature of phyllic and argillic zones. Copper zones (i.e. chalcopyrite and minor bornite with gold and silver values) usually occur central to the alteration systems although in some cases they occur within the propylitic zone. Sulphides typically occur as fracture fillings, disseminated grains, massive lenses and pods and in breccias. Magnetite is commonly associated with these systems and may either coincide with sulphide zones or occur peripheral to the copper zones. Calc-silicate alteration products, including andradite to grossularite garnets with associated gypsum and anhydrite, occur within the potassic zones at Galore Creek, whereas, scapolite is commonly in the propylitic altered copper zones at Ingerbelle. Of the Cordilleran alkaline deposits known, Galore Creek and Ingerbelle are the largest.

4.3.1 Stratigraphy

The Galore Creek deposit is hosted by the Stuhini Group of the Stikine terrain (Figure 4-2). The Stuhini Group rocks comprise a variety of flows, tuffs, volcanic breccia and sediments, and are important host rocks to the alkaline-intrusive related gold-silver-copper mineralization at Galore Creek. They define a volcanic edifice centred on Galore Creek and represent an emergent Upper Triassic island arc characterized by shoshonitic and leucitic volcanics (de Rosen-Spence, 1985), distal volcaniclastics and sedimentary turbidites. The succession at Galore Creek was divided by Panteleyev (1976) into a submarine basalt and andesite lower unit overlain by more differentiated, partly subaerial alkali-enriched flows and pyroclastic rocks.

4.3.2 Intrusives

Three intrusive episodes have been recognized in the region. The earliest and most important is the Middle Triassic to Middle Jurassic Hickman plutonic suite that is coeval with Upper Triassic Stuhini Group volcanic flows. The Mount Hickman batholith comprises three plutons known as Hickman, Yehino and Nightout. The Schaft Creek porphyry copper deposit is associated with the Hickman stock, and is located 39 km northeast of Galore Creek. This stock is crudely zoned with a pyroxene diorite core and biotite granodiorite margins. Alkali syenites of the Galore complex like those found at the nearby Copper Canyon deposit and the pyroxene diorite bodies of the zoned Hickman pluton have been interpreted as differentiated end members of the Stuhini volcanic - Hickman plutonic suite by Southern (1972) and Barr (1966). The alkali syenites are associated with important gold-silver-copper mineralization at Galore Creek and at Copper Canyon. These rocks are believed to be at least as old as Early Jurassic in age, based on K-Ar dating of hydrothermal biotite in the syenites intruding the sequences (Allen, 1966). An Ar-Ar age of 212 Ma (Logan et al., 1989) in syenite may give the time of crystallization of the intrusive rocks at Copper Canyon, to the east of Galore Creek. More recent U-Pb dates of Galore Creek syenites have given ages ranging from 205-210 Ma (Mortensen, 1995).

At Galore Creek young post-mineral basalt and felsite dykes are abundant as a dyke swarm in the northwest part of the property. Elsewhere, Tertiary intrusions may be important in their association with small gold occurrences.

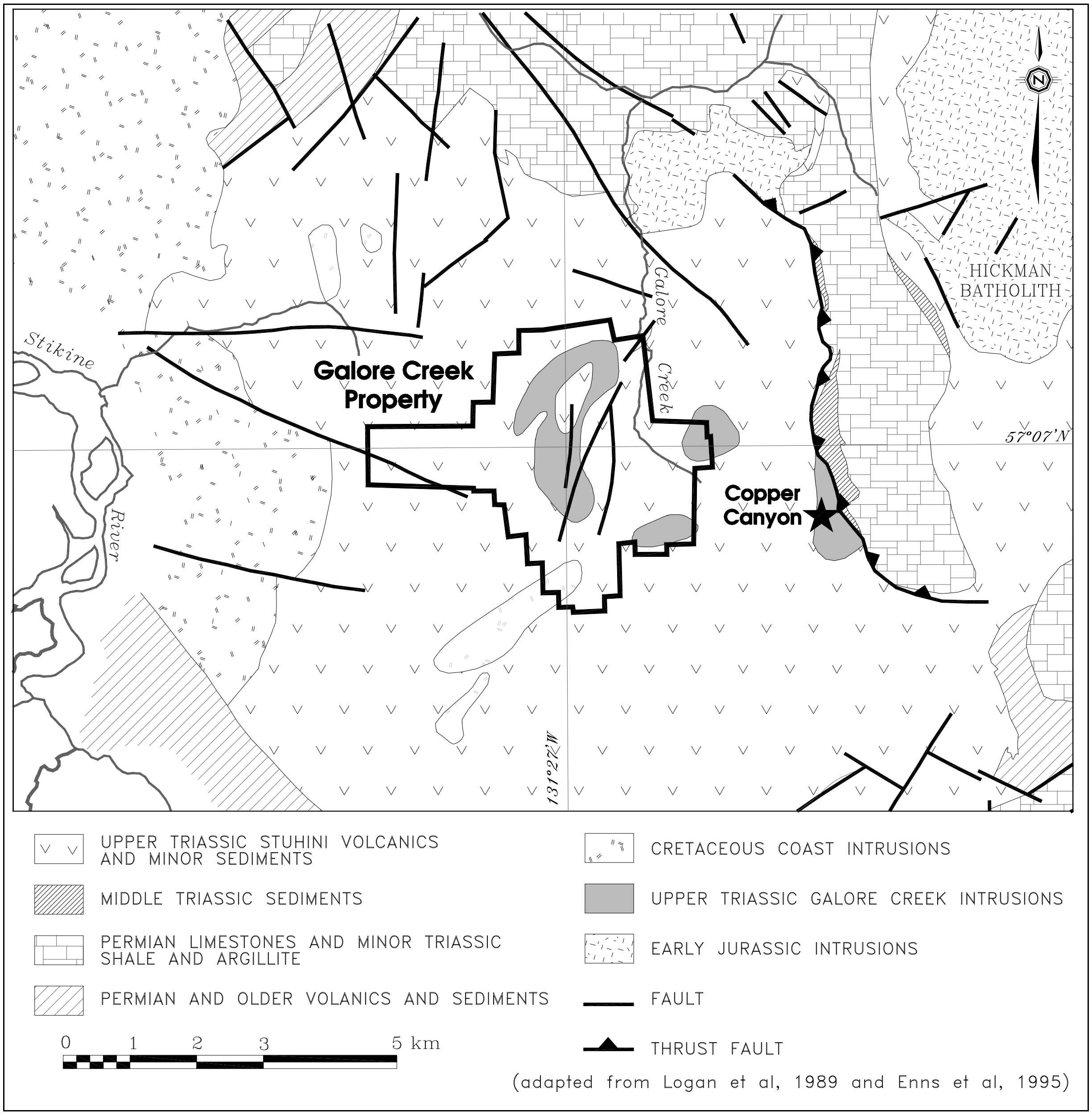
4.3.3 Structure

The regional geology has been affected by polyphase deformation and four main sets of faults. The oldest phase of folding is pre-Permian to post-Mississippian and affected the Paleozoic rocks between Round Lake and Sphaler Creek. This deformation is characterized by bedding plane parallel foliation in sediments and fragment flattening in volcaniclastics. Pre-Late Triassic folding is characterized by large, upright, tight to open folds with north to northwest trend of axial plane traces and westerly fold vergence. Metamorphism accompanying the first two phases of deformation reached greenschist facies. The third phase of folding is manifested as generally upright chevron folds with fold axes pointed west-northwesterly.

The oldest and longest-lived fault structures in the area have a north strike and sub-vertical dip. The best example occurs on the west flank of the Hickman batholith, where a major fault juxtaposes Permian limestone with a narrow belt of Stuhini Group volcanics. The second important fault type occurs at Copper Canyon as a west directed thrust fault with a north strike and east dip of 30 to 50 degrees. It juxtaposes overturned Permian limestone and Middle Triassic shale with Stuhini volcanics below. Early to Middle Jurassic syenite intrusions occupy this contact. A third important set of faults with northwest strike mark the boundary between Upper Triassic and Paleozoic rocks between Scud River and Jack Wilson Creek. The youngest faults have a northeast strike direction and are of great local importance.

At Galore Creek, some of these faults show considerable post-mineral movement of up to 200 m while others appear to control the emplacement of mineralized intrusive phases and breccia bodies.

Figure 4-2: Regional Geology (taken from Simpson, 2003)



4.4 Property Geology

Mineralization at Galore Creek occurs in upper Triassic felsic to intermediate volcanic flows and fragmental rocks. It is associated most closely with intense, pervasive Ksilicate alteration as replacement, disseminated and fracture-controlled chalcopyrite with locally abundant bornite. Higher gold values are normally associated with bornite mineralization.

Four mineralized zones of potentially economic interest have been explored at Galore Creek. These are the Central Zone, Southwest Zone; Junction Zone and West Fork Zone.

4.4.1 Central Zone

The Central Zone is the largest and most extensively explored of all the deposits and is characterized by both volcanic and intrusive geologic units. Mineralization is exposed in the southern part of the zone, but elsewhere it is covered by up to 75 m of glacial overburden. Between 80% and 90% of the gold-silver-copper occurs as sulphide replacement of the host volcanic rocks. The grade of the mineralization commonly exceeds 1% Cu, decreasing rapidly at the margin of the zone.

The Central Zone deposit has an orientation of 015° and dips steeply to the west. It is 1700 m long, 200 to 500 m wide and has been traced to a depth of 450 m and remains open. The eastern boundary of the Central Zone mineralization lies near the surface projection of a major, steeply west dipping, brittle normal fault. In the west and south, mineralization is truncated by post-mineral megaporphyry dykes. Intense SAC alteration has obliterated mineralization in the northwestern part of the Central Zone. In the north, mineralized volcanic rocks end abruptly against a thick sequence of weakly to unmineralized epiclastic sedimentary and volcanic rocks.

The Central Zone exhibits considerable internal variations in both mineralization and alteration. Hydrothermal alteration changes from Ca-K-silicate in the core region to intense K-silicate alteration toward the north and south parts of the zone. In terms of gold-silver-copper replacement mineralization, the most favourable volcanic lithologies are the pseudoleucite-bearing volcanic rocks in the north and the dark crystal tuffs in the south. Augite-bearing units in the north are low to moderate in copper content and the core of the deposit hosts a mineralized orthomagmatic breccia. Gold values are highest in the northern and southern portions of the Central Zone where significant disseminated bornite, magnetite and hematite are present. Lower gold grades correlate with the intense Ca-K-Silicate altered core region. Chalcopyrite is the most important copper mineral and occurs as replacements, disseminations and fracture fillings throughout the zone. Supergene copper mineralization is minor and occurs primarily as malachite, azurite and chrysocolla on fractures within 60 m of surface. Pyrite increases in abundance to the east of the Central Zone reaching concentrations of up to 5%.

4.4.2 Southwest Zone

The Southwest Zone is located about 600 m southwest of the south end of the Central Zone and contains some of the highest grade near surface gold mineralization. Drilling has outlined an elongate pod shaped body that trends roughly east-west and dips approximately 60° to the south. The zone is up to 400 m long and may be as wide as 140 m; however, the 1991 drilling suggests that the zone narrows at both the eastern and western ends of the deposit. The Southwest Zone is still open at depth. Primary hosts for the Southwest mineralization are a diatreme breccia and an early syenite phase intrusive. Localization of high-grade gold-silver-copper mineralization within the diatreme appears to relate to a combination of structural traps and mineralizing faults.

4.4.3 Junction Zones

The Junction and North Junction zones lie about 2 km northwest of the Central Zone and about 460 m higher in elevation. They are a series of irregular, flat-lying manto-shaped bodies plunging about 20° to the northeast. Together these two zones are interpreted to have been a single deposit that was originally 1400 m in length before it was faulted. Width of the zones varies from 50 to 150 m. The mineralization, consisting of disseminated chalcopyrite and bornite, is hosted by fine to coarse lapilli tuff and feldspar phyric flows. Higher gold and copper grades correlate with the presence of bornite in the North Junction zone. K-silicate alteration consisting of pervasive hydrothermal biotite and K-feldspar flooding is associated with the mineralization. A large mass of late-mineral megaporphyry truncates the zone on the west.

4.4.4 West Fork Zone

The West Fork zone lies in the valley floor less than 1 km south of the Central Zone and less than 50 m higher in elevation. West Fork contains two adjacent but distinctly different styles of mineralization; disseminated sulphide replacements similar to portions of the Central Zone, and massive veining. The disseminated mineralization is characterized by disseminated chalcopyrite and bornite hosted by a combination of intrusive and volcanic rocks. Higher grade disseminated zones appear to have a possible structural control locally though distinct veining is absent.

The Opulent Vein, which consists of massive chalcopyrite, bornite and magnetite, defines the other style of mineralization at West Fork. Sulphide textures indicate fissure-style fillings of open space, but associated calc-silicate gangue minerals, possibly tremolite, indicate replacement. The known extent of the Opulent Vein is limited within a breccia mass and strikes approximately 355° with a steep west dip.

4.5 Exploration

4.5.1 Exploration and Drilling

Since mineralization was discovered at Galore Creek in 1955, the property has been explored by an extensive range of exploration techniques which support the interpretation of Galore Creek mineralized zones and the domains within them. The mineral resource estimates quoted in this report are based exclusively on assays of samples obtained by diamond drilling and there are no trench or grab samples in the database.

Drilling, logging, sampling, sample preparation, storage, assaying and data management procedures have evolved over time and with successive operators, however, the majority of the drilling produced NQ or

HQ size core. Core was logged under the direction of qualified geologists and sample intervals were determined by a geologist. The average sample length is 2.9 m and overall average core recovery is good (averaged about 80% in 2004).

Very little is known about the earliest drilling campaign in 1961, however, records of the remainder are available and have been checked by earlier workers and in the course of this study. A QA/QC program in line with currently accepted standards was introduced in 2003-4. The data has been verified in five separate campaigns and verification is now a continuous process. In the opinion of the Qualified Persons responsible for this resource estimate, the sampling, assay and data verification programs have been carried out with the reasonable care and skill expected of the profession.

4.6 Mineral Resource and Mineral Reserve Estimates

The Galore Creek Project consists of 4 separate mineralized zones namely: Central, Southwest, West Fork and Junction. Each of these zones was treated as a standalone deposit to estimate the resource.

A total of 563 specific gravity measurements were made on the Galore Creek property during the 1966-67 drill campaign by measuring the weight of the sample and dividing by the amount of water it displaced. In April 1992 a total of 96 specific gravity determinations were made from drill core. The SG determinations were made from small pieces of core at recorded distances down the hole. Using these distances the specific gravity values were compiled with the assay from-to intervals that contained them. In a number of instances more than one SG determination was made on the same from-to interval. In these cases the SG values were averaged. As a result 631 assays had specific gravity determinations.

Table 4-3 shows average values for the various lithologies coded. In most cases, average specific gravity values for selected rock types were assigned to domains.

Table 4-3: Summary of Specific Gravity measurements from Galore Creek Property

	VOLC	INTR	BREC	DYKE	SEDS	FAUL
Number	435	140	8	7	5	3
Mean S.G.	2.67	2.63	2.71	2.67	2.59	2.59
S.D.	0.182	0.156	0.095	0.077	0.066	0.114
Minimum	1.69	1.91	2.56	2.50	2.34	2.43
Maximum	3.40	3.30	2.86	2.76	3.04	2.69

4.6.1 Central Zone Estimates

4.6.1.1 Data Analysis

NovaGold geologists subdivided the various lithologies present at Galore Creek Central Zone into three main domains:

- **Group 1** - Main Mineralized Zone –Volcanics with some intrusive intervals that were too small to model. (Further subdivided by the East Fault into west of fault (201) and east of fault (202))
- **Group 2** - I4 intrusives, I8 intrusives and some I3 Dykes (further subdivided by the East Fault into west of fault (337) and east of fault (338))
- **Group 3** – I5, I9, I10 and I11 Intrusives (further subdivided by East Fault into west of the East fault (357) and east of the East fault (358))

Simple statistical parameters calculated for each group found that the coefficients of variation ranged between 1.33 and 6.55, indicating a fair degree of variability. Lognormal cumulative probability plots were produced for each variable within each of the six main geologic domains. Each plot was partitioned and examined to establish individual capping policies for each population (summarized in Table 4-4).

Table 4-4: Summary of Capping Levels

Domain	Variable	Capping Level	Number Capped
Group 1	Cu (%)	6.90% Cu	5
	Au (g/t)	16.59 g Au/t	5
	Ag (g/t)	53.4 g Ag/t	8
Group 2	Cu (%)	None Capped	
	Au (g/t)	3.85 g Au/t	8
	Ag (g/t)	None Capped	
Group 3	Cu (%)	None Capped	
	Au (g/t)	4.0 g Au/t	5
	Ag (g/t)	27.72 g Ag/t	4

4.6.1.2 Composites

The statistics for composites are summarized in Table 4-5. The combination of capping and smoothing over 5 m has reduced all coefficients of variation to below 2.88.

Table 4-5: Summary of Statistical Parameters for 5 m Composites

Domain	Variable	Number	Mean	S.D.	Minimum	Maximum	Coef. of Var.
Group 1	Cu (%)	11,111	0.416	0.624	0.001	6.46	1.50
West	Au (g/t)	11,049	0.207	0.594	0.001	16.59	2.88
(201)	Ag (g/t)	11,053	3.17	4.92	0.001	49.37	1.55
Group 1	Cu (%)	2,693	0.303	0.410	0.001	3.13	1.35
East	Au (g/t)	2,669	0.099	0.192	0.001	3.70	1.94
(202)	Ag (g/t)	2,669	2.61	4.22	0.001	29.65	1.62
Group 2	Cu (%)	1,191	0.309	0.472	0.001	3.62	1.53
West	Au (g/t)	1,188	0.170	0.358	0.001	3.85	2.10
(337)	Ag (g/t)	1,189	2.50	3.83	0.001	26.40	1.53
Group 2	Cu (%)	56	0.318	0.381	0.001	1.38	1.20
East	Au (g/t)	55	0.057	0.082	0.001	0.35	1.44
(338)	Ag (g/t)	55	1.89	2.37	0.001	6.86	1.26
Group 3	Cu (%)	2,158	0.123	0.295	0.001	4.73	2.39
West	Au (g/t)	2,156	0.081	0.198	0.001	4.00	2.46
(357)	Ag (g/t)	2,156	1.14	2.50	0.001	25.47	2.20
Group 3	Cu (%)	285	0.043	0.104	0.001	0.85	2.43
East	Au (g/t)	285	0.032	0.061	0.001	0.38	1.92
(358)	Ag (g/t)	285	0.74	1.98	0.001	13.71	2.68

4.6.1.3 Variography

Pairwise relative semi-variograms were produced for each variable in Group 1 east and west of the East Fault, in Group 2 west of the East Fault and in Group 3 west of the East Fault. There was insufficient data to model variography east of the East Fault in Groups 2 and 3 and the models generated west of the fault were used. Models were generated in the vertical direction and in 4 main directions within the horizontal plane (Az. 90, 0, 45 and 135). Anisotropic nested models were fitted to the data in all but Group 3 silver where an omnidirectional spherical nested model was fitted. The parameters for all models are shown below in Table 4-6.

Table 4-6: Parameters for semi-variogram models at Galore Creek Central Zone

Domain	Variable	Direction	C0	C1	C2	Range a1 (m)	Range a2 (m)
Group 1 Volcanics West of Fault (201)	Cu	Az. 30° Dip 0	0.20	0.25	0.45	30	200
		Az. 300° Dip -45	0.20	0.25	0.45	40	100
		Az. 120° Dip -45	0.20	0.25	0.45	60	100
	Au	Az. 45° Dip 0	0.10	0.40	0.50	20	200
		Az. 315° Dip 0	0.10	0.40	0.50	20	120
		Az. 0° Dip -90	0.10	0.40	0.50	30	120
	Ag	Az. 45° Dip 0	0.10	0.45	0.65	20	120
		Az. 315° Dip 0	0.10	0.45	0.65	20	60
		Az. 0° Dip -90	0.10	0.45	0.65	30	200
Group 1 Volcanics East of Fault (202)	Cu	Az. 30° Dip 0	0.10	0.30	0.42	40	130
		Az. 300° Dip -45	0.10	0.30	0.42	40	150
		Az.120° Dip -45	0.10	0.30	0.42	40	90
	Au	Az. 90° Dip 0	0.15	0.25	0.40	50	70
		Az. 0° Dip 0	0.15	0.25	0.40	20	60
		Az. 0° Dip -90	0.15	0.25	0.40	40	100
	Ag	Az. 90° Dip 0	0.05	0.30	0.75	40	80
		Az. 0° Dip 0	0.05	0.30	0.75	10	20
		Az. 0° Dip -90	0.05	0.30	0.75	20	120
Group 2 Intrusives I4, I8, I3 West of Fault (337)	Cu	Az. 0° Dip 0	0.10	0.30	0.60	15	100
		Az. 270° Dip 0	0.10	0.30	0.60	20	50
		Az. 0° Dip -90	0.10	0.30	0.60	50	150
	Au	Az. 135° Dip 0	0.20	0.35	0.55	40	150
		Az. 45° Dip 0	0.20	0.35	0.55	35	100
		Az. 0° Dip -90	0.20	0.35	0.55	60	110
	Ag	Az. 135° Dip 0	0.10	0.60	0.70	20	100
		Az. 45° Dip 0	0.10	0.60	0.70	40	60
		Az. 0° Dip -90	0.10	0.60	0.70	90	200
Group 3 Intrusives I5, i9, i10, i11 West of Fault (357)	Cu	Az. 90° Dip 0	0.10	0.40	0.50	20	40
		Az. 0° Dip 0	0.10	0.40	0.50	20	40
		Az. 0° Dip -90	0.10	0.40	0.50	20	80
	Au	Az. 135° Dip 0	0.20	0.40	0.50	50	160
		Az. 45° Dip 0	0.20	0.40	0.50	30	120
		Az. 0° Dip -90	0.20	0.40	0.50	70	120
	Ag	Omnidirectional	0.05	0.30	0.85	40	60

4.6.1.4 Bulk Density

For this resource estimation Group 1 was assigned the average specific gravity of volcanics or 2.67 while Group 2 and Group 3 were made up of intrusives with an average specific gravity of 2.63 (refer to Table 4-6 above). These values were used for blocks below the broken/intact rock interface. Material above this boundary was severely broken and as a result a general 5% reduction in specific gravity was applied. Blocks completely above the surface were given a specific gravity of 2.54 in Group 1 domains and 2.50 in Group 2 and 3 domains. Blocks straddling the boundary were given a weighted average specific gravity based on the percentage of volume above and below the broken rock surface.

4.6.1.5 Block Interpolation

Ordinary kriging was completed in 3 passes for each variable in each domain. A minimum 4 and maximum 16 composites were used for each pass.

- All blocks with some proportion below the bedrock surface and proportion of primary solid greater than 0.0 were estimated for each variable in separate runs using the appropriate composites for Domains with 3 separate passes in each run. Results to Cu1, Au1 and Ag1 fields.
- All blocks with some proportion below bedrock and proportion of (BDRK – PCORE1) > 0.0 were estimated for each variable using appropriate composites with 3 separate passes. Results to Cu2, Au2 and Ag2 fields.
- The grades were then combined to form weighted averages for each block based on the proportion of each domain in the block.

4.6.1.6 Classification

The estimated mineral resource was classified in accordance with the definitions contained in NI 43-101.

Geologic continuity on the Central Zone has been established through surface mapping and diamond drilling over a number of years. Grade continuity has been measured through the use of semi-variograms. Using the ranges from the semi-variograms for both copper and gold, a classification scheme was devised for the Central Zone.

- Measured - Blocks estimated in Pass 1 (using 1/4 of the semi-variogram ranges) for both Cu and Au.
- Indicated - Blocks not classified and estimated in at least pass 2 (using 2/3 of the semi-variogram ranges) for both Cu and Au.
- Inferred - All other blocks estimated. In addition all blocks below the 200 m elevation were assigned inferred status regardless of when they were estimated.

Table 4-7 summarize the results for three particular cut-offs.

Table 4-7: Grade Tonnage for Central Zone

Cut-off (CuEq%)	Tonnes > Cut-off (tonnes)	Grade > Cut-off				Million lbs. of Cu	Million ozs of Au	Million ozs of Ag
		Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)			
Classed Measured Plus Indicated								
0.35	423,900,000	0.614	0.302	4.681	0.780	5739.05	4.12	63.80
0.50	290,100,000	0.738	0.376	5.421	0.947	4720.77	3.51	50.56
1.00	90,100,000	1.100	0.724	7.257	1.511	2185.38	2.10	21.02
Classed Inferred								
0.35	173,600,000	0.467	0.283	3.425	0.623	1787.62	1.58	19.12
0.50	96,000,000	0.576	0.380	4.085	0.790	1219.28	1.17	12.61
1.00	18,200,000	0.781	0.929	5.193	1.312	313.42	0.54	3.04

4.6.1.7 Inverse Distance Squared Test

As a test of the interpolation procedure, a second run was made using inversed distance squared (IDS), utilizing the same elliptical search parameters used for kriging, with minimum 4 and maximum 16 composite limits. As with kriging, the various domains were estimated in a series of passes with expanding search ellipses. The results are very similar to those obtained by kriging. Differences occurred due to the weighting techniques employed by the different interpolation methods.

4.6.2 Southwest Zone

4.6.2.1 Data Analysis

NovaGold geologists subdivided the various lithologies present at Galore Creek Southwest zone into two main domains:

- Group 352 – I9 (Medium Grained Orthoclase Syenite Megaporphyry)
- Group 400 – SW Breccia

Simple statistical parameters were calculated for each group. The coefficients of variation range between 1.80 and 2.8, indicating a fair degree of variability, which is also evident in the maximum values present.

The methodologies utilized in analysis of the Central Zone analysis were also applied to the Southwest Zone.

4.6.2.2 Classification

Geologic continuity on the Southwest Zone has been established through surface mapping and diamond drilling over a number of years. Grade continuity has been measured through the use of semi-variograms. Using the ranges from the semi-variograms for both copper and gold, a classification scheme was devised for the Southwest Zone.

- Measured - Blocks estimated in Pass 1 (using 1/4 of the semi-variogram ranges) for both Cu and Au.
- Indicated - Blocks not classified and estimated in at least pass 2 (using 2/3 of the semi-variogram ranges) for both Cu and Au.
- Inferred - All other blocks estimated.

Table 4-8 summarizes the results for three CuEq cut-offs.

Table 4-8: Grade Tonnage for Southwest Zone

Cut-off (CuEq%)	Tonnes > Cut-off (tonnes)	Grade > Cut-off				Million lbs. of Cu	Million ozs of Au	Million ozs of Ag
		Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)			
Classed Measured Plus Indicated								
0.35	47,700,000	0.447	0.818	3.041	0.972	470.15	1.25	4.66
0.50	34,100,000	0.553	1.005	3.523	1.194	415.80	1.10	3.86
1.00	16,100,000	0.843	1.477	4.787	1.738	299.27	0.76	2.48
Classed Inferred								
0.35	122,900,000	0.314	0.556	2.274	0.695	850.92	2.20	8.99
0.50	72,100,000	0.407	0.706	2.617	0.895	647.05	1.64	6.07
1.00	18,400,000	0.714	1.092	3.245	1.445	289.68	0.65	1.92

4.6.3 Junction Zone

4.6.3.1 Data Analysis

A total of 55 drill holes were provided for analysis containing 248 down hole survey measurements and 4,645 assays. These codes for lithologies[1] present at Junction have been simplified to the following list.

- Sediments (Code 160)
- Volcanics V3 (Code 230)
- Volcanics V4, V4F, V4H, V5, V5A, V5D, V5H V5h (Codes 240-258)

[1] There are 107 lithological codes

- Intrusives i, i5, i9, i9B, i6, i8 (Codes 300 – 362)
- Syenite (Codes 352-355)
- Junction Porphyry (Codes 367-368)
- Breccia (Codes 400-430)
- Dykes (Codes 510-540)

Drilling has been completed in several campaigns by different property operators. There were many ambiguities within the supplied database with 0.000 values and -2.00 values for Cu, Au and Ag. As a result, each drill hole was examined using drill logs to determine the following:

- Overburden or ice – assays were not taken and values set to blank
- Areas of no core recovery – assays not taken and values set to blank
- Areas of post mineral dykes – assays not taken and value set to 0.001
- Areas with Cu assay but no gold or silver – early drill holes were reassessed in later programs and gold and silver assays were taken where there was sufficient drill core left. Samples with insufficient core for gold or silver assay were set to blank.
- Assays with 0.000 represented below detection value and were set to 0.001.

As a result of this audit a total of 4,510 samples had copper assays while 3,954 had gold assays.

The methodologies utilized for the Central Zone analysis were also applied to the Junction Zone.

4.6.3.2 Classification

Geologic continuity on the Junction zone has been established through surface mapping and diamond drilling. Grade continuity has been measured through the use of semi-variograms. Using the ranges from the semi-variograms for both copper and gold, a classification scheme was devised for the Junction zone. In addition, for this zone a shell of geologic continuity was imposed, by NovaGold geologists, on the block model outside of which, blocks could only be classed Inferred regardless of what pass they were estimated in.

- Measured - Blocks estimated in Pass 1 (using ¼ of the semi-variogram ranges) for both Cu and Au within the geologic continuity shell.
- Indicated - Blocks not classified and estimated in at least pass 2 (using ½ of the semi-variogram ranges) for Cu and Au within the geologic continuity shell.
- Inferred - All other blocks estimated and all those outside the geologic continuity shell.

Table 4-9 summarizes the results for three CuEq cut-offs.

Table 4-9: Grade Tonnage for Junction Zone

Cut-off (CuEq%)	Tonnes > Cut-off (tonnes)	Grade > Cut-off				Million lbs. of Cu	Million ozs of Au	Million ozs of Ag
		Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)			
Classed Measured Plus Indicated								
0.35	30,000,000	0.589	0.412	4.777	0.837	389.62	0.40	4.61
0.50	18,300,000	0.794	0.513	6.344	1.110	320.39	0.30	3.73
1.00	7,100,000	1.347	0.713	10.617	1.795	210.88	0.16	2.42
Classed Inferred								
0.35	71,600,000	0.532	0.286	3.334	0.685	839.91	0.66	7.67
0.50	34,600,000	0.759	0.381	4.559	0.977	579.06	0.42	5.07
1.00	10,500,000	1.334	0.588	7.533	1.686	308.85	0.20	2.54

4.6.4 West Fork Zone

4.6.4.1 Data Analysis

Assay samples were subdivided into lithologic groups, based on NovaGold geologists logging of drill core. Lithologies present at West Fork are listed as follows:

Code	Description
200-250	Volcanics including Orthoclase-bearing (V3), crystal lithic tuff (V3h), and intermediate undivided (V5)
300-352	Intrusives including Pseudoleucite Porphyry (i1), dark orthoclase syenite (i4), dark and orthoclase syenite (i4), fine grained orthoclase syenite megaporphyry (i5), medium 371-380 grained orthoclase syenite megaporphyry (i9), fine grained syenite porphyry (i7), medium grained syenite porphyry (i11) and plagioclase syenite porphyry (i10a)
362	Medium Grained Syenite (i8)
369	West Fork Porphyry (WFP)
400 -435	Breccia including Diatreme (B1), Heterolithic Diatreme (B1b), Hydrothermal (B2), Hydrothermal Monolithic (B2b), Orthomagmatic (B3), Orthomagmatic Heterolithic (B3b)
500-540	Dykes including lamprophyre (D1), intermediate dykes (D3) and felsic dykes (D4)

The Opulent Vein was not divided out in the above tabulations leading to the high copper values and averages in Volcanics and Breccias. NovaGold geologists tried to model each lithology in three-dimensional space and found many could not be individually represented. It appears from the statistics that volcanics, intrusives and breccias are all mineralized in this area and could be lumped together if necessary. The four units that modeled well were the Opulent Vein, the I8 intrusive, the Lamprophyre dyke and the West Fork Porphyry. The remaining rock units northeast of the Opulent Vein were combined to form a geologic domain that appeared to dip steeply while those units southwest of the Opulent Vein were combined into a more tabular Geologic Domain.

These geologic domains were examined for copper, gold and silver using lognormal cumulative frequency plots to determine if capping was required and if so at what levels. The Opulent Vein, which was highly mineralized and the Lamprophyre Dyke which was not mineralized were separated out while the remaining mineralized zones were combined.

The mineralized zones of West Fork Steep, West Fork Flat, I8 and West Fork Porphyry were combined into one data set called West Fork Mineralized Zone. For copper the highest population with a mean of 1.638% represents 108 samples and cannot be considered erratic. A reasonable capping level for this population would be at 2 standard deviations past the mean of population 1 or a value of 2.58% Cu. A single value at 2.78 was capped at 2.58%. For gold the upper population 1 represented 0.41% if the data or 13 samples. This population could be considered erratic and an effective cap would be at 2 standard deviations past the mean of population 2, a value of 2.56 g Au/t. A total of 8 samples were capped at 2.56 g Au/t. The highest population for silver had a mean of 35.54 g Ag/t and represented 0.61 of the data or 19 samples. A reasonable cap on silver would be at 2 standard deviations past the mean of population 2, a value of 39.4 g Ag/t. A total of 5 silver assays were capped at 39.4 g Ag/t.

Cap levels for the Opulent Vein were set at 2 standard deviations above the mean of the upper population and no samples required capping. No samples in the Lampropyre Dyke required capping.

The methodologies utilized in the Central Zone analysis were also applied to the West Fork Zone.

4.6.4.2 Classification

Geologic continuity on the West Fork property has been established through surface mapping and diamond drilling. Grade continuity has been measured through the use of semi-variograms. Using the ranges from the semi-variograms for both copper and gold, a classification scheme was devised for the West Fork Deposit.

- Measured Blocks estimated in Pass 1 (using ¼ of the semi-variogram ranges) for both Cu and Au.
- Indicated - Blocks not classified and estimated in at least pass 2 (using ½ of the semi-variogram ranges) for Cu and Au.
- Inferred - All other blocks estimated.

Table 4-10 summarizes the results for three CuEq cut-offs.

Table 4-10: Grade Tonnage for West Fork Zone

Cut-off (CuEq%)	Tonnes > Cut-off (tonnes)	Grade > Cut-off				Million lbs. of Cu	Million ozs of Au	Million ozs of Ag
		Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)			
Classed Measured Plus Indicated								
0.35	15,100,000	0.580	0.379	4.793	0.798	193.11	0.18	2.33
0.50	10,800,000	0.694	0.438	5.675	0.949	165.27	0.15	1.97
1.00	3,300,000	1.118	0.631	9.930	1.514	81.35	0.07	1.05
Classed Inferred								
0.35	45,400,000	0.466	0.339	4.986	0.665	466.50	0.50	7.28
0.50	29,700,000	0.553	0.403	6.181	0.797	362.15	0.39	5.90
1.00	4,900,000	0.841	0.603	9.948	1.219	90.87	0.10	1.57

5. Mining

5.1 Summary

An optimized 65,000 tpd mill feed schedule was developed for the Galore Creek mine. Detailed pit phases were engineered from the results of a Lerchs-Grossman (LG) sensitivity analysis, and yielded the phase delineated resources in Table 5-1. Phase delineated resources in Table 5-1 include a 10% dilution for blocks that contain both waste and mineralized material above cutoff grade (contact dilution). No mining losses have been taken on the assumption that after the 2005 exploration data is included in the resource modeling, that most blocks above cut-off grade will be 100% mill feed. Cut-off grade for the Phase delineated resources in Table 5-1 is US$3.26/t Net Smelter Return (NSR).

Table 5-1: Galore Creek Pit Resource Summary

MINE PHASE	PIT NAME	RUN OF MINE (kt)	WASTE TOTAL (kt)	ROM S/R	ROM GRADES			
					NSR ($/t)	CU (%)	AU (g/t)	AG (g/t)
Central								
Phase 1	P615	56,881	91,396	1.61	16.03	0.839	0.274	5.949
Phase 2	P625	62,469	167,672	2.68	14.94	0.632	0.648	3.845
Phase 3	P635B	58,397	71,534	1.22	12.76	0.684	0.219	5.224
Phase 4	P645	79,298	226,941	2.86	11.83	0.624	0.250	4.301
Phase 5	P655	63,097	101,526	1.61	10.42	0.570	0.183	4.656
Phase 6	P695	55,138	63,851	1.16	8.71	0.490	0.137	4.953
Central Total		375,279	722,921	1.93	12.44	0.639	0.287	4.774
South West								
Phase 1	SW601a	22,655	18,377	0.81	19.25	0.684	1.088	3.502
Phase 2	SW611	41,363	145,169	3.51	10.57	0.469	0.506	2.563
South West Total		64,018	163,547	2.55	13.64	0.545	0.712	2.895
Middle Creek								
Phase 1	W611	9,128	4,639	0.51	7.41	0.414	0.172	0.403
Middle Creek Total		9,128	4,639	0.51	7.41	0.414	0.172	0.403
West Fork								
Phase 1	WF611	10,473	28,904	2.76	12.65	0.518	0.578	9.184
West Fork Total		10,473	28,904	2.76	12.65	0.518	0.578	9.184
Junction								
Phase 1	JN6	30,018	83,143	2.77	18.24	0.817	0.415	6.692
Phase 2	JN5	4,942	12,777	2.59	13.47	0.708	0.118	3.580
Junction Total		34,960	95,920	2.74	17.57	0.802	0.373	6.252
Grand Total		493,858	1,015,931	2.06	12.87	0.631	0.352	4.648

5.2 Introduction

The entire mine planning for the Galore Creek mineral property is based on work done with MineSight©, a suite of software that includes the resource model, pit optimization (MSEP), detailed pit design, and optimized production scheduling (MSSP).

In addition to the geological information used for the block model, other data used for the mine planning includes the base economic parameters, mining cost data derived from supplier estimates and historical data, geotechnical slope design parameters, hydrology and geohydrology flow rates, metallurgical recoveries, and project design plant costs and throughput rates.

The climatic conditions of this site are factored into the design costing and operational considerations for the open pit mining operations. The comparatively high snow precipitation and the mountainous terrain at the mine site will require active snow control and extra road maintenance/traction control for roads and mining benches.

5.3 Project Production Rate Considerations

A number of factors are considered in establishing an appropriate mining and processing rate, the key ones are discussed below in relations to Galore Creek:

- **Resource size:** Typically, a "reserve tail" of at least 50% is preferred (i.e. the mine is projected to continue for 50% beyond the projected payback period). For a base metal mine, this usually requires a minimum 15 to 20 year life of mine.
- **Unit Capacity:** Generally, unit operating costs are lower using the largest possible equipment for a single train. In the case of SAG mills, this was about 38 foot diameter, although at least two 40 foot diameter units have now been proven in operations elsewhere. Depending on ore hardness, and considering a single primary mill, throughputs of up to 80,000 tpd are possible depending on final grind size selection.
- **Operational Constraints:** Practical considerations with respect to the number of operating mining faces required to achieve a production rate in relation to the pit geometry.
- **Construction Constraints:** Physical size and weight of equipment and shipping limits can determine the maximum size of available units.
- **Project Financial Performance:** Generally, economies of scale can be realized at higher production rates and lead to reduced unit operating costs. These are tempered to the above mentioned physical constraints and generally higher capital requirements for higher tonnage throughputs.

Determining the optimal production rate is an iterative exercise. The Galore Creek Scoping Study – 2004 considered the above factors and selected a production rate of 30,000 tpd ore as a base case for project design and costing. Economics were significantly enhanced by the updated Resource estimate and by higher throughput scenarios. This study has subsequently been based on throughput of 65,000 tpd ore. Further testing and evaluation of grind size and recovery will be undertaken in the feasibility study to confirm the most economic production rate.

5.4 Economic Pit Limits, Mine Plan

The economic pit limit is determined using the ePit optimization routines in MineSight which is based on the LG algorithm which runs against the 3D Block model, evaluating the costs and revenues of the blocks within potential pit shells. Two separate models have been generated for the Junction and the Central Area. Economic pit limit studies have been broken into these two respective areas.

5.4.1 Economic Pit Limits – Central Area

5.4.1.1 LG Pit Cases Central Area

Three series of LG pits are developed. At this stage, exploration targets within an economic pit limit are still being drilled in the 2005 exploration program, therefore revenues from resource classes below the confidence of Measured and Indicated have been included in the study and consequently this study is considered a PEA. Resource classes used for the pits were as follows:

- ***Series A*** - LG Pits created using economics from classes 1,2,3 &4
- ***Series C*** - LG Pits created using economics from classes 1 & 2
- ***Series D*** - LG Pits created using economics from classes 1,2 & 3

where:

- Class 1 = Measured
- Class 2 = Indicated
- Class 3 = Inferred based on full variogram range
- Class 4 = Inferred based on twice the full variogram range

LG pit shells are determined for each series by keeping mining costs constant and varying the estimated net smelter metal prices (NSP).

5.4.1.2 Central Area LG Economic Pit Limit Summary

The economic pit limits have been determined using Lerchs-Grossman optimization based on Measured, Indicated, and Inferred resources to give a target for the 2005 exploration program and overall scope for the current engineering and planning studies. The pit design parameters are pre-feasibility level estimates; key parameters have been varied to test the pit limit sensitivity. The following points summarize the pit limit results for Central Zone.

- LG economic pit limits are significantly sensitive to complex slopes, but not sensitive to incremental bench cost or bench time value discounting within the resource model limits.

5.4.2 Economic Pit Limits – Junction

5.4.2.1 LG Pit Cases Junction Area

The Junction area economic pit limit is a much simpler evaluation than the Central area due to the smaller size and limited mineralized zone. The economic pit design is more determined by the design parameters (costs, prices and slope parameters) than the extent of the mineralized zone.

5.4.2.2 Junction Area LG Economic Pit Limit Summary

The economic pit limits have been determined using LG optimization for the Junction model. Many of the pit design parameters are rough estimates at this stage, to be confirmed during the feasibility study. From this analysis, the 100% case has been chosen to base ongoing detailed designs and 2005 field work. Two pit stages result from the Junction economic pit limit: one centered on the South Junction (SJ1) zone and the other on the two North Junction zones (NJ2 & NJ3). These are used as the basis of the detailed design, which in turn are used in production scheduling and costing. The following points summarize the pit limit results for the Junction area.

- The LG economic pit limit indicates that most of the modeled resource (MII) falls within the economic pit limit. The pit size is relatively insensitive around the parameters used, so higher prices and costs will likely not change the results for Junction.
- The 2005 exploration will need to upgrade the high proportion of Inferred resource within the economic pit shell.
- The pit limit for Junction is sensitive to the slope design parameters because of the geometry of the mineralized zone and the depth of cover. It is possible to minimize the effect of the high wall ramps since the upper benches are accessible from the hill side and the lower benches from a potential access slot in the Dendritic Creek gully.

5.4.3 Detailed Pit Designs

5.4.3.1 Selection of Pushbacks

Central Pit

The Central area was sub-divided into pushbacks or pit phases due to the size of the LG ultimate pit to enable a more favorable (even) material flow during the production scheduling of the mine and to minimize the pre-stripping required to release the mill feed at start-up. Properly sized and sequenced phases improve the project cashflow while meeting the mill feed targets and keeping the mine loading and hauling fleet at a consistent number of units. Progression from the highest value pit phase to the lowest value pit phase, provides a scheduling sequence that minimizes the payback period, and maximizes the net present value and project rate of return.

Generally, the LG cases less than the 50% case indicated a quicker release of mill feed from mining at the south end of Central area. Therefore the first pit phase is designed at the south end of the 50% LG and the second phase at the north. The initial benches of the south end are also closer to the valley floor than the north end, requiring simpler initial bench access during the pre-production period. The third phase mines deeper at the north end to target a higher grade gold zone (NGL). Phases beyond these are then sequenced based on logistics and access requirements.

Note: - The 2004 scoping studies examined mining the NGL as a starter phase. The high-grade ore associated with the north gold lens is located below a steep slope on the west side of Central Pit and is west of the 50% LG pit shell being used to locate the starter pit phases. The large initial strip ratio with associated capital required to expose sufficient quantities of the deeper 'gold lens' for startup production causes this to be less attractive economically than the current mining sequence.

Southwest

The same need for phasing or pushbacks is required for Southwest pit to even out stripping requirements to release the mill feed from the deeper benches. The upper benches of the model in the Southwest pit area, contain higher gold values than the lower benches. To mine the deeper mineralized material requires pushing the walls back to reach the bottom benches within the economic pit shell. From a scheduling and economic prospective it is better to mine a smaller pit inside the design limits, followed by a deeper pit, than mine to the final economic pit limit from top down in a single pass

Junction, West Fork, and Middle Creek

For the smaller mining areas such as Junction, Middle Creek, and Southwest physical requirements such as minimum bench size and access dictate that these pit be mined as one phase.

5.4.3.2 Detailed Design – Pit Slope Parameters

Geotechnical Studies for Pit Slope Design

Pre-feasibility level pit slope design parameters have been provided by BGC based on historical reports and field data collected in the 2004 field season. The current data sources and resultant information are:

- NovaGold geology interpretation from the drill hole database used to create the mineralized zones and host rock types.
- structural mapping to model regional faults, local bedding, sills, intrusions, and contacts.
- geology logs and core samples for geomechanical and rock strength testing.
- oriented core measurements for down hole structural mapping.
- piezometer installations for groundwater monitoring.
- hydraulic test (rising head, falling head, and constant flow packer tests) for ground water flow.

Angles for the upper (broken rock) and lower (stick rock) slopes for wet and dry conditions are provided. BGC's work takes into account structurally possible modes of instability identified by the surface mapping and sub-surface discontinuity mapping, as well as potential rock mass instability based on the information listed above. From this, preliminary over-all slope design parameters are provided for the detailed pit design work.

Pit Slope Parameters

BGC provided design angles for both dry and saturated slopes. Subsequent design has assumed dry slopes and dewatering wells and drains have been included in the detailed design and costing for the mine operations. The overall slope design parameters for the different sectors of the pit wall, assuming dry slopes, and are listed in Table 5-2.

Table 5-2: Pit slope design parameters

Achievable Pit Slopes - Broken Rock	Pit Slope
Slope Segment 356º - 056º	40º
Slope Segment 056º - 098º	45º
Slope Segment 098º - 164º	40º
Slope Segment 164º - 228º	45º
Slope Segment 228º - 272º	45º
Slope Segment 272º - 332º	45º
Slope Segment 332º - 356º	40º
Achievable Pit Slopes - Stick Rock	**Pit Slope**
Slope Segment 356º - 056º	45º
Slope Segment 056º - 098º	50º
Slope Segment 098º - 164º	45º
Slope Segment 164º - 228º	50º
Slope Segment 228º - 272º	50º
Slope Segment 272º - 332º	50º
Slope Segment 332º - 356º	45º

The MineSight pit design software is used for the detailed design of the pit phases using the input overall angles for stick rock as above plus a bench face angle (BFA) of 70˚ and a minimum berm width of 8 m. on a 30 m vertical interval (double benching).

The ultimate pit wall intersects several streams, which must be diverted. A design for diversion of Galore Creek around the east side of the ultimate pit limit is included in the pre-production capital works. Interim diversion of Dendritic Creek is included as well as a ditch on the west wall of the detailed pit phase designs.

5.4.3.3 Pit Phases

Central Detailed Pit Phase Designs

Five pit phases are included in the detailed pit phase designs, along with a sixth (ultimate) phase included for the Central area. The sixth phase follows the surface of the LG shell for the 110% case with complex slopes.

A comparison of the resources within the phase designs shows that each phase contains similar tonnes of mill feed. Phase 1 has a low strip ratio for early ore release, and the NSR starts at higher values, decreasing to the lowest value for the last phase.

NSR is calculated from the metal grades in each block, base case metal prices, plant recoveries for copper, gold and silver, $US/$CDN exchange rate, offsite concentrate transportation, smelting and refining charges.

Table 5-3: Comparison of Pit Resources Central Pit Phases

Phase	Pit	Bottom Elevation Description	Mill Feed (kt)	Waste (kt)	SR (t/t)	NSR ($/t)
1	C615	600 L-G Pit 40* = 50% price shell – southern portion	56,881	91,396	1.6	16.0
2	C625	480 L-G Pit 40* = 50% price shell – northern portion	62,469	167,672	2.7	14.93
3	C635B	435 L-G Pit 40* = 50% price shell – middle portion	58,397	71,534	1.2	12.80
4	C645	405 West wall to ultimate pit limit	79,298	226,941	2.9	11.87
5	C655	375 Incremental east wall phase	63,097	101,526	1.6	10.40
6	C695	270 L-G Pit 06** = 110% price shell (w/complex slopes)	55,138	63,850	1.2	8.67

* Designated internal pit number in MineSight model GC0513.ep1
** Designated internal pit number in MineSight model GC0513.ep2

Pit phase one (C615) is where the LG pits indicate that mining should be initiated, because it contains the highest value ore with the best stripping ratio. The LG pits also indicate that pit phase two (C625) in the north area is the next location that should be mined because it contains the highest gold values and the waste rock necessary for the dam construction. This phase contains the second highest NSR, but it does require higher waste stripping.

The LG Pit 40 shows a joining of the north and south phases (i.e., phases 1 and 2). Pit phase three (C635B), which contains the next highest NSR and has a low strip rate, is designed to join the first two phases. There is an interim highwall ramp in the west wall of this third pit phase requiring the next phase (pit C645) to mine to the ultimate pit's west wall or future access to the final (ultimate) pit wall to the west will be cut off. The mine production schedule requires that the high waste stripping for phase 4 is done before phase 3 mining cuts off the bench access.

The fifth pit phase (C655) is the logical progression to the ultimate pit on the east side (and to depth) of the previous phases. The final increment to the ultimate pit (C695) contains the lowest value material and strip ratio.

All pit phase ramps, except for the ultimate pit ramps, exit at elevation 690 on the central east side of the pit and are interim until the final haul road is built in the last / ultimate sequence.

Junction, South West, West Fork and Middle Creek

Table 5-4 summarizes the pits not included in the main Central area. These auxiliary mining areas can be mined independently from the Central phases and can be scheduled as needed to improve the mill feed grade and optimize the waste stripping fleet. The only dependency for these outside pits is that phase 2 must follow phase 1 for Junction and Southwest. The higher NSR values and lower strip ratio for Southwest Phase 1 (see Table 5-4) indicates this pit should be mined early in the production schedule.

Middle creek was modeled as part of the Central zone, but considered as a separate pit for scheduling purposes.

Table 5-4: Comparison of Pit Resources Auxiliary Pits

Area/Phase	Pit	Mill Feed kt	Waste kt	SR t/t	NSR* $/t
Southwest P1	SW601a	22,655	18,377	0.8	19.33
Southwest P2	SW611	41,363	145,169	3.5	10.67
West Fork	WF611	10,473	28,904	2.8	12.67
Middle Creek	W611	9,128	4,639	0.5	7.47
North Junction	JN6	30,018	83,143	2.8	18.27
South Junction	JN5	4,942	12,777	2.6	13.47

5.4.3.4 Phase Designs

The resultant 6 cumulative pit phase designs for Central and the 6 auxiliary pits are illustrated in Figure 5-1 to Figure 5-12. These are shown relative to the truck shop/ROM stockpile ahead of the crusher to the northeast of the drawings (at approximately 705 m. elevation) and the plant site to the southeast. Topography contours are shown every 15 m. at bench grade elevations.

Figure 5-1: Central Pit – Phase 1 (P615)

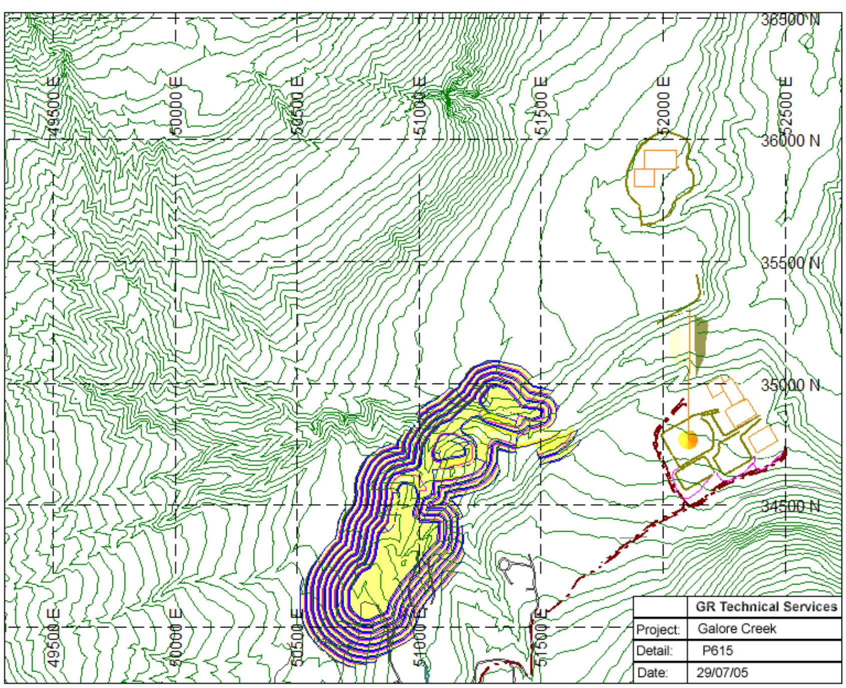


Figure 5-2: Central Pit – Phase 2 (P625)

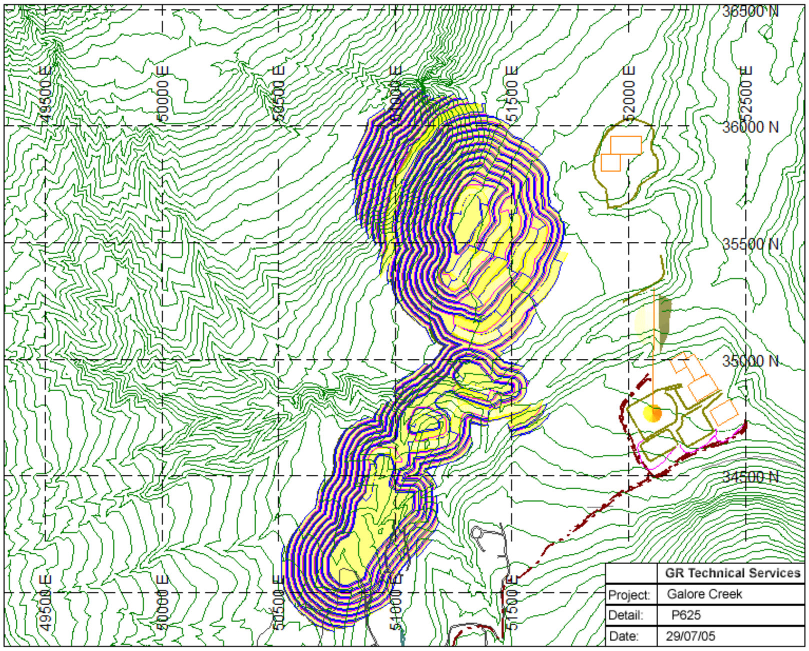


Figure 5-3: Central Pit – Phase 3 (P635B)

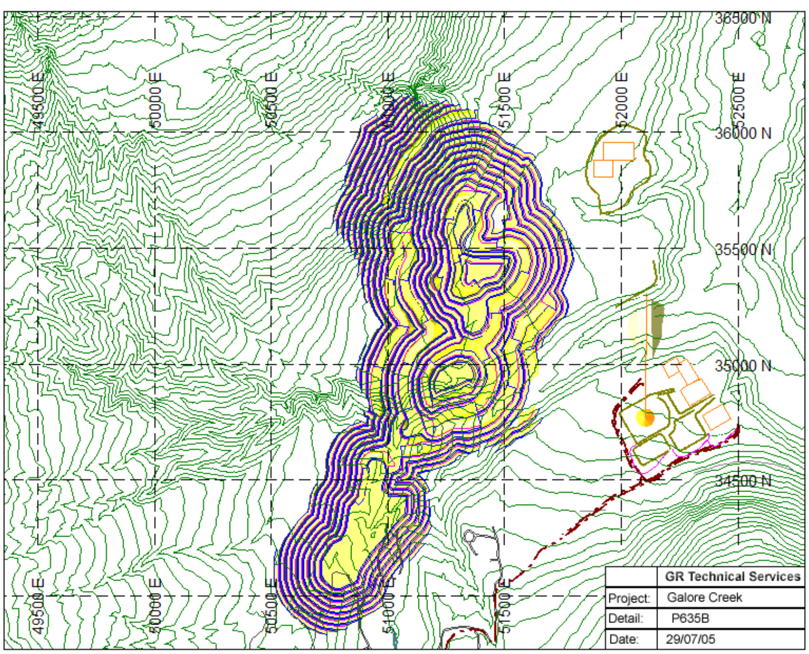


Figure 5-4: Central Pit – Phase 4 (P645)

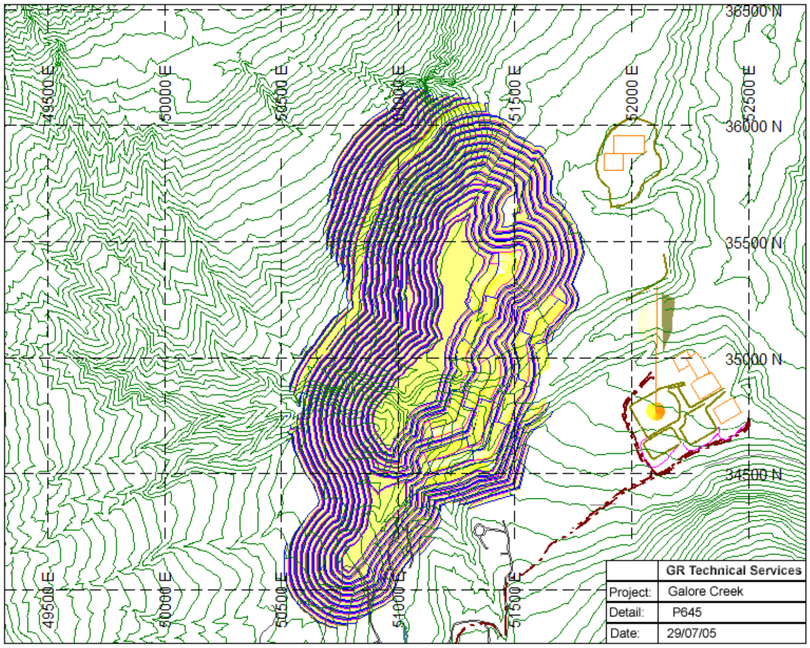


Figure 5-5: Central Pit – Phase 5 (P655)

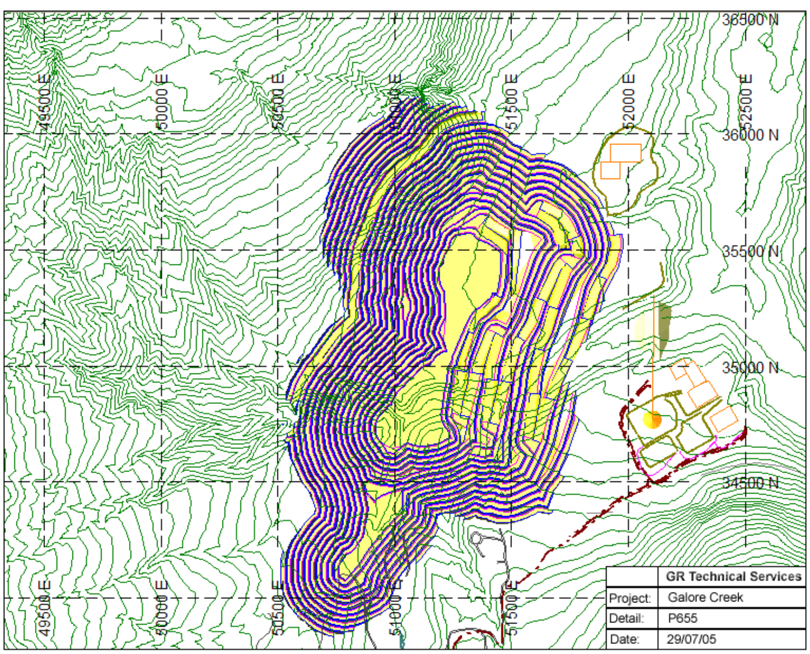


Figure 5-6: Central Pit – Phase 6 (P695)

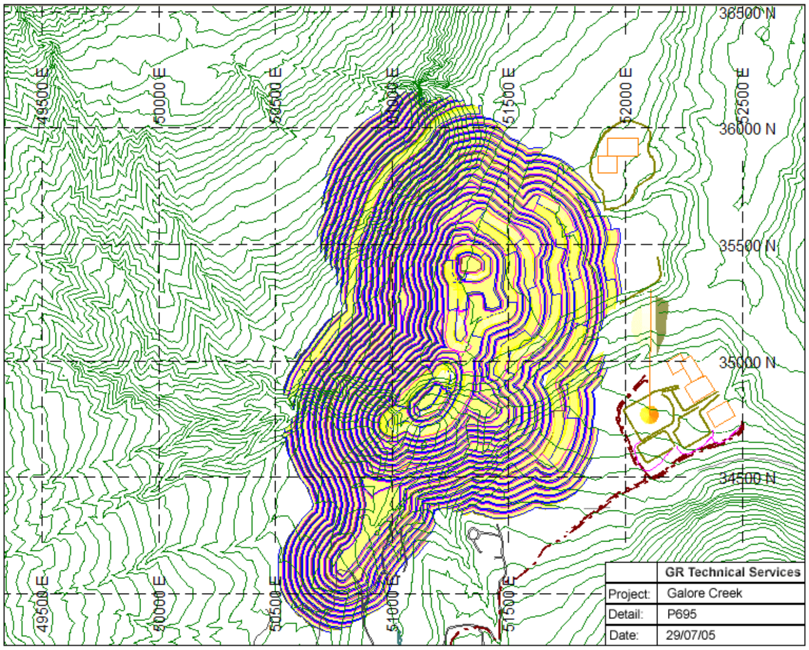


Figure 5-7: Southwest Pit – Phase 1 (SW601a)

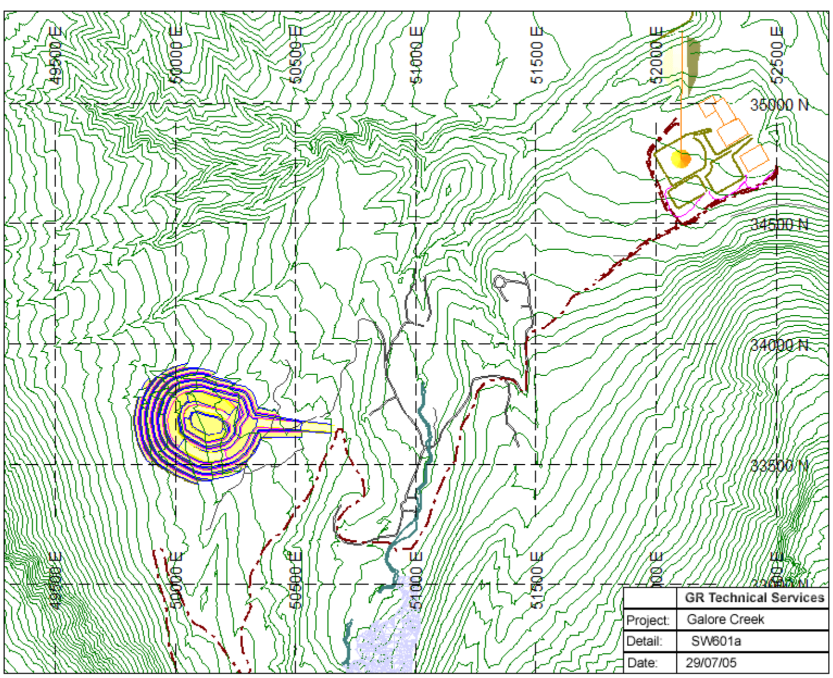


Figure 5-8: Southwest Pit – Phase 2 (SW611)

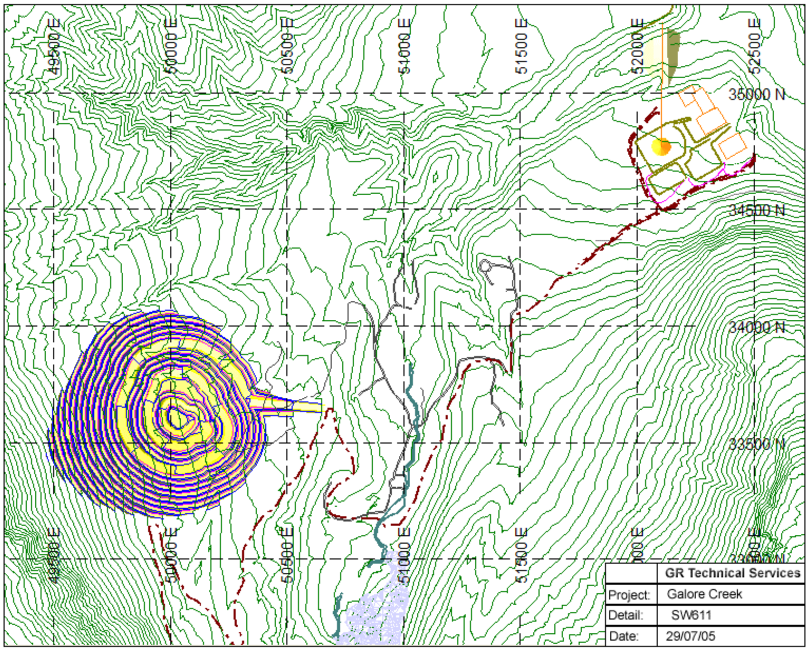


Figure 5-9: Middle Creek Pit (W611)

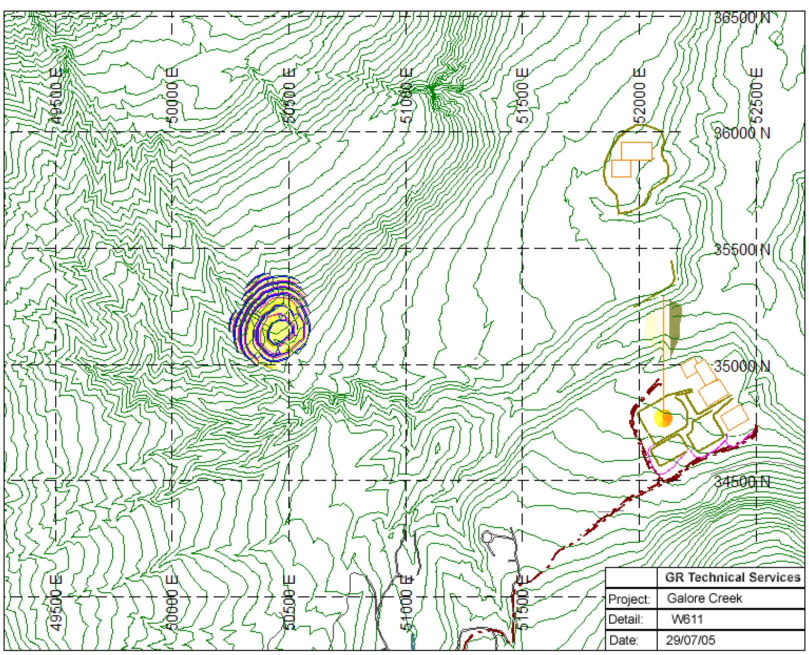


Figure 5-10: West Fork Pit (WF611)

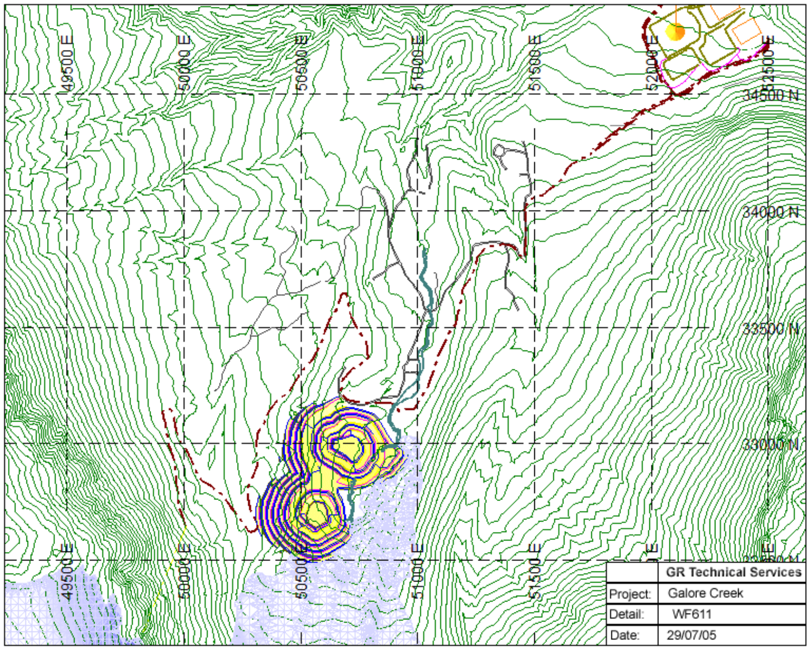


Figure 5-11: Junction Pit – Phase 1 (JN6)

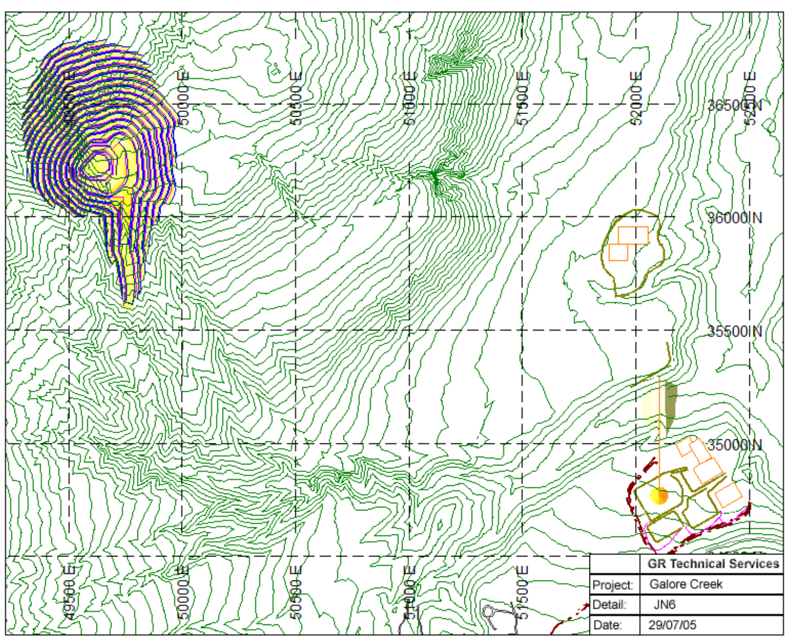
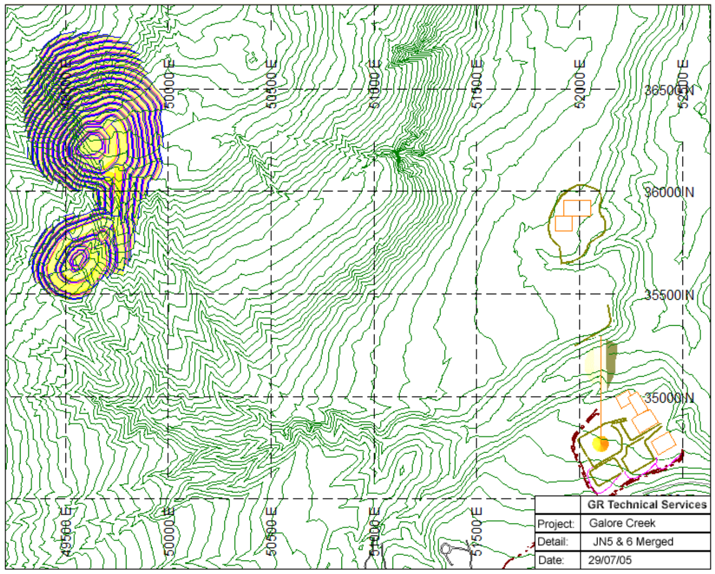


Figure 5-12: Junction Pit – Phase 2 (JN5 & 6 Merged)



5.4.3.5 Grade Bins by NSR Cutoff

The pit delineated resource reports used in this study have the mill feed broken into grade bins for production schedule optimization. These grade bins are based on the NSR value of each block, which is calculated from the metal grades in each block, the net smelter metal price, and the plant recovery for gold, copper and silver. The net smelter price is based on base case metal prices, offsite transportation, smelting, and refining charges, etc. Table 5-5 describes the NSR cutoffs used in the pit delineated resource reports.

Table 5-5: NSR Grade cutoff bins for Pit delineated resources

Grade Bin ($/t ore)	Label	Description
3.26	Ore/Waste Cutoff	Material below this is wasted
4.04	Sub Grade Cutoff	Assuming a 20% reduced mill recovery from stockpile, if this material is stockpiled it is wasted.
4.93	Low Grade	Covers Process & Infrastructure, + stockpile re-handle + Process G&A, and potential 20% lost revenue if stockpiled.
6.22	Mid Grade	Median between low grade and high grade.
> 7.50	High Grade	

5.4.3.6 Detailed Design Pit Resources

Pit resources for the detailed pit designs are calculated from the 3D block model using the MineSight Pitres routine. The resource calculation methodology includes a 10% dilution for blocks that contain both waste and mineralized material above cutoff grade (contact dilution). No mining losses have been taken on the assumption after the 2005 exploration data is included in the resource modeling, that most blocks above cut-off grade will be 100% mill feed. The mining operations will then include all blocks within large ROM feed grade boundaries. This will need to be confirmed in future studies, with detailed engineering of typical operating benches.

Table 5-1 provides a Pit Resource Summary for Galore Creek.

5.4.4 Production Schedule

5.4.4.1 Basic Criteria

The production schedule for Galore Creek has been developed to meet the input mill feed targets, while balancing waste stripping requirements with the given truck and shovel fleet. Partial bench mining of 2 benches per year is allowed. The various grade bins are evaluated to improve the relative project NPV. All production will be from the owner's fleet. Pre-strip begins in 'Year –2' and full production begins at the beginning of 'Year 1'. The project mill feed target is an average 65,000 tpd or 23,725,000 tpa.

Southwest Phase 1 (SW601a) has the highest grade and is selected as the startup phase. Most of the ore requirement in Year 1 is mined from Southwest Starter Phase (SW601a). Sub-grade material becomes uneconomical if stockpiled and is wasted in the startup period to prevent mill feed grade dilution. A pre-strip waste production volume of 75 Mt is required to ensure suitable material is available for start-up dam construction. Galore Creek production schedule is presented in Table 5-7 showing the first 5 years of production and LOM data.

The proportions of pit delineated resource classification in Year 5 and Year 10 are estimated in Table 5-6 using end of period shapes for those years.

Table 5-6: Summary of proportions of scheduled pit delineated resource classification

Year	Measured %	Indicated %	Measured + Indicated %	Inferred %
Year 1 - 5	38.1%	42.9%	81.0%	19.0%
Year 1 - 10	33.3%	40.8%	74.1%	25.9%
LOM	26.4%	49.2%	75.6%	24.4%

Table 5-7: Summarized Galore Creek Production Schedule

PRODUCTION		Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	LOM
ROM Mill Feed									
ROM Mill Feed mined to crusher	tonnes	-	-	23,735,018	23,735,028	23,734,976	23,734,992	23,735,000	416,414,573
Cu	%	-	-	0.731	0.939	0.962	0.626	0.715	
Au	g/t	-	-	1.072	0.316	0.344	0.664	0.480	
Ag	g/t	-	-	3.976	6.270	6.756	3.690	4.536	
ROM Mill Feed mined to stockpiles	tonnes	61,129	61,129	1,952,410	2,624,628	2,780,071	5,764,682	2,792,971	58,227,732
Total ROM Mill Feed Mined	tonnes	61,129	61,129	25,687,428	26,359,656	26,515,047	29,499,675	26,527,971	474,642,306
ROM Mill Feed retreived from stockpiles	tonnes	-	-	-	-	-	-	-	57,130,000
Cu	%	-	-	-	-	-	-	-	
Au	g/t	-	-	-	-	-	-	-	
Ag	g/t	-	-	-	-	-	-	-	
Total Stockpile Inventory	0	61,129	122,258	2,074,669	4,699,296	7,479,367	13,244,050	16,037,021	
Total ROM Mill Feed to Mill	tonnes	0	0	23,735,018	23,735,028	23,734,976	23,734,992	23,735,000	473,544,573
Cu	%	-	-	0.731	0.939	0.962	0.626	0.715	
Au	g/t	-	-	1.072	0.316	0.344	0.664	0.480	
Ag	g/t	-	-	3.976	6.270	6.756	3.690	4.536	
Sub Grade to Waste	tonnes	48,211	48,211	2,005,984	2,651,625	3,203,052	4,861,323	2,815,308	19,215,394
Waste Mined	tonnes	37,389,525	37,389,525	41,041,618	59,722,766	84,011,896	44,372,848	49,390,511	1,015,937,962
Total Waste Mined	tonnes	37,437,736	37,437,736	43,047,602	62,374,391	87,214,947	49,234,170	52,205,819	1,035,153,356
Waste Types:									
Till	tonnes	10,654,989	10,654,989	17,193,520	16,875,780	31,066,715	5,191,076	9,111,231	166,117,378
Broken NPAG	tonnes	258,836	258,836	12,733,734	9,138,005	10,277,417	7,042,877	16,099,452	180,306,378
Broken PAG	tonnes	25,671,525	25,671,525	10,769,786	31,200,173	32,287,318	13,648,787	11,509,365	227,328,043
Stick NPAG	tonnes	-	-	344,577	486,668	2,145,260	6,157,664	9,604,237	180,900,018
Stick PAG	tonnes	804,175	804,175	-	2,022,140	8,235,186	12,331,968	3,044,081	165,365,728
Junction (Un Zoned)	tonnes	-	-	-	-	-	476	22,146	95,920,417
Waste Hauled for Main Dam Construction	tonnes	13,289,391	13,289,391	11,430,428	12,519,762	12,519,762	12,519,762	12,519,762	227,876,181
Strip Ratio1 (Waste/Total ROM Mill Feed Mined)		612.4	612.4	1.7	2.4	3.3	1.7	2.0	2.2
Strip Ratio2 (Waste/ Total ROM Mill Feed to Mill)		-	-	1.8	2.6	3.7	2.1	2.2	2.2
Strip Ratio3 (Waste/ ROM Mill Feed Mined to Crusher)		-	-	1.8	2.6	3.7	2.1	2.2	2.5
Total Material *Mined*	tonnes	37,498,865	37,498,865	68,735,030	88,734,047	113,729,994	78,733,845	78,733,791	1,509,795,662
Total Material *Moved*	tonnes	37,498,865	37,498,865	68,735,030	88,734,047	113,729,994	78,733,845	78,733,791	1,566,925,662

5.4.5 End of period maps

End of period maps are attached as drawings SKAO-C-002, SKAO-C-003, SKAO-C-004.

5.5 Mine Startup and Construction

5.5.1 Introduction

The construction of the tailings dam and water diversion ditches required for the Galore Creek mine will require significant earthmoving operations, which will be integrated into the startup of the mining operation. Details of the design basis and geotechnical studies undertaken are discussed in Section 8 of this PEA.

Development of the supporting infrastructure such as mine access, power and communication structures, equipment erection & maintenance facilities (fuel, service, repair) must be implemented well in advance of actual construction and mining activity. Ensuring key management objectives for environmental protection and capital stewardship are met while applying a 'common sense' approach to mine construction are overriding principles guiding the development of the startup plan.

Ore grade material is available near surface, and from a mill supply perspective, requires very little pre-stripping to feed the plant at design rates. Mining activity in the startup period is being driven by infrastructure construction requirements and will ultimately be further influenced by the design characteristics of the material required for these features. Details of the design criteria for catchment ditches, dams, and spillways are discussed in Section 8. The main mining related site features being constructed during startup are the main tailings dam, the freshwater bypass dam, catchment ditches and spillway, and mining haul roads.

An open ditch and temporary culvert on the west side of the Galore Valley will be constructed to divert water from the Dendritic Creek watershed to the freshwater catchment ditch on the east side of the Galore Valley during construction. The culvert will be maintained through the construction of the main tailings dam, carrying Dendritic Creek past the dam footprint above the elevation of the starter dam. Once the starter dam is complete, the culvert will be recovered and Dendritic Creek flow will report to the main tailings impoundment.

5.5.1.1 Source Materials

Mine pit waste sources will provide most of the required construction materials. Phases 1 to 4 of Central Pit will provide all materials (till and ballast) for the East freshwater dam, all haul road capping material and fills, and Main tailings dam ballast. Local (i.e. near the starter dam) borrow sources were selected for the Main dam impervious clay core. The specific locations for the till borrow pits will be determined following 2005 field investigations.

5.5.2 Winter Operations

A study of the most economic and practical methods of managing the mining operation in this climate was undertaken to provide a preliminary review of possible winter mining conditions and solutions for the Galore Creek property.

The property falls within a region of substantial snowfall and moderate temperatures ranging from 20°C to -20°C. The snow study addressed the impact of snow and ice on mine operations. The findings and recommendations were used as a starting point for planning purposes. This study will be updated as climate data is reviewed and pit areas, haul distances, and other relevant data are firmed up during the subsequent feasibility study.

5.5.2.1 Operating Scenarios

Five operating scenarios were evaluated:

- **Scenario 1 Seasonal Operations:** Seasonal operations refers to no winter support equipment requiring complete shutdown of all mine operations, except for feed to the mill, during the winter season (December till March).
- **Scenario 2 Shutdown during Winter Storm Events:** No winter support equipment has been added to the mining equipment fleet. This means operating during the winter snow storm season but suspending operations during almost any snowfall event due to icing problems on the haul roads.
- **Scenario 3 Partial Shutdowns during Most Storms:** Basic winter support equipment including a waste rock crusher has been added to the fleet. This small winter support fleet would still require some mining operations to be consolidated during every winter storm event to take advantage of limited support equipment.
- **Scenario 4 Partial Shutdown Only During Severe Storms**: Mine crusher and winter support equipment would be sized to allow full production during winter storms with production shutdown of some pit areas only during the most severe storms. At no time would all operations cease.
- **Scenario 5 Full Operations Year Round:** No allowance for shutdowns, even during the most extreme storm. Keeping all roads open and hauling during all weather conditions would require a number of pieces of support equipment that would only be utilized 1-2% of the year.

5.5.2.2 Cost Evaluation And Analysis For Winter Operation

Each of the scenarios 1 through 5 carries a capital and operational cost with it. These costs are summarized in Table 5-8.

Table 5-8: Summary Of Capital and Operating Costs For Snow Fleet Scenarios

Scenario	Capital ($M)	Op Cost ($/t ore)
1. Seasonal	117	0.03
2. Down for all Storms	53	0.13
3. Down for Severe Storms Only	27	0.08
4. Partial Op During Severe Storms	12	0.04
5. Full Operations	16	0.05

The most cost effective operating scenario is Scenario 4 Partial Operations during severe storms. In addition to the economics of this scenario, there is a strategic advantage of never completely shutting down mining operations. Having an attitude that weather may slow but never stop operations encourages innovation within the operation.

Based on the potential of weather forcing the operation towards Scenario 3, it is also prudent to have a contingency plan for Scenario 3 down time. Since Scenario 3 requires an additional loading unit and trucks to make up lost production, additional contingency is built into the production fleet.

5.5.2.3 Snow Fleet Recommendations

The adopted scenario in this PEA is a combination of Scenarios 3 and 4. A snow support fleet sized for Scenario 4 (15 lost days) and stockpile, fuel storage and additional production sized for Scenario 3 (30 lost days). The additional production is made up with the loader and trucks. The snow support fleet is made up of crusher, scrapers, additional grader and miscellaneous blades and rakes. Visibility issues will be addressed with road lighting and mobile equipment GPS. Other winter issues such as pole stands, blast hole covers, outdoor storage covers, etc. is covered by a 5% contingency.

The important issues during winter mining in a heavy snow zone are the effects of icing and reduced access from snow cover. Planning for each situation and ensuring the tools are available to respond to the changing weather conditions can minimize each of these issues. Costs and strategies for managing in this environment were developed following a detailed review of operating practices in other open pit mines (in BC and worldwide) which experience heavy winter snowfall events.

5.5.3 Stockpiling

As part of the process to smooth out stripping requirements, medium and low grade mill feed mined during the early years of the mine life is sometimes stockpiled for later processing. Besides creating mill feed inventory for future high strip ratio years in the mine, this strategy has the effect of improving project revenues in the early years of the project.

The mill stockpile pad will be located immediately north of the crusher area on waste placed early in the mine life.

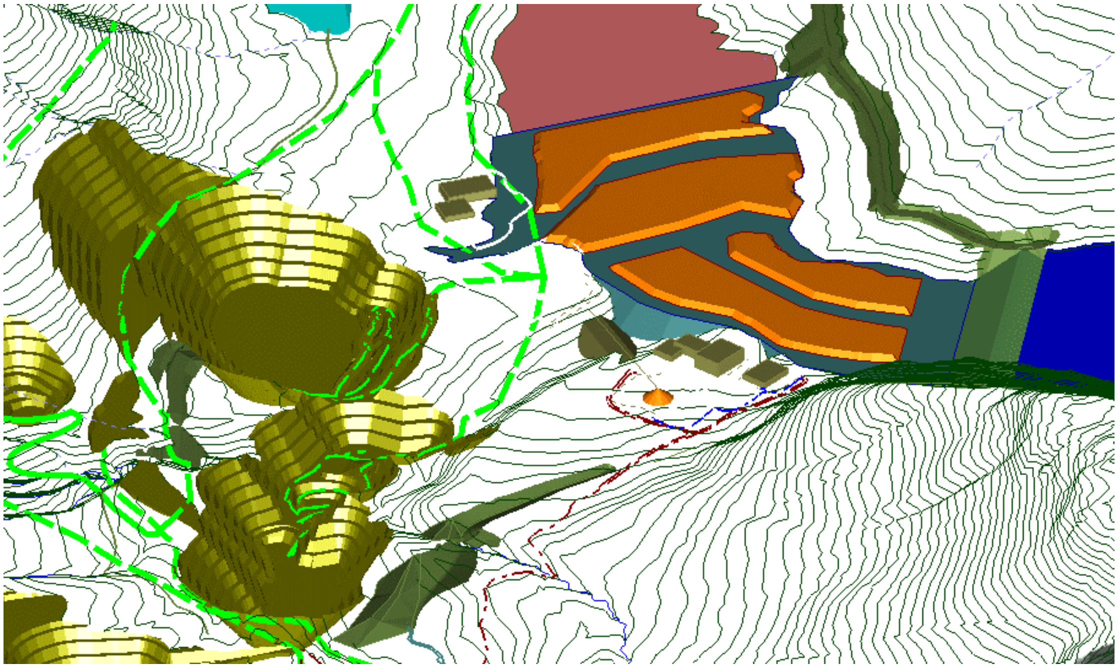
Table 5-9 shows the required waste volumes to generate increasing ROM ore storage capacity.

Table 5-9: Construction Requirements for ROM Stockpile

ROM Stockpile Phase	Waste Required (t)	Ore Capacity (t)
1	42,425	6,396
2	84,152	13,765
3	115,558	22,935

The construction of the piles will allow multiple stockpile faces to be reclaimed to control grade. To accomplish this objective the mill stockpile will be built in a series of cells.

Figure 5-13: Proposed Layout of Stockpile Pad at the end of Year 5



Intermediate stockpiles of low grade ore will be constructed between the pit and crusher on an 'as required' basis to provide operating flexibility during poor weather and when the mill demand is for ore other than that available in the immediate shovel face. It is estimated typical intermediate stockpile capacity will need to be approximately 1,000 kt at any given point in the mine life to provide two weeks of mill feed.

5.5.4 Waste Rock Storage Strategy

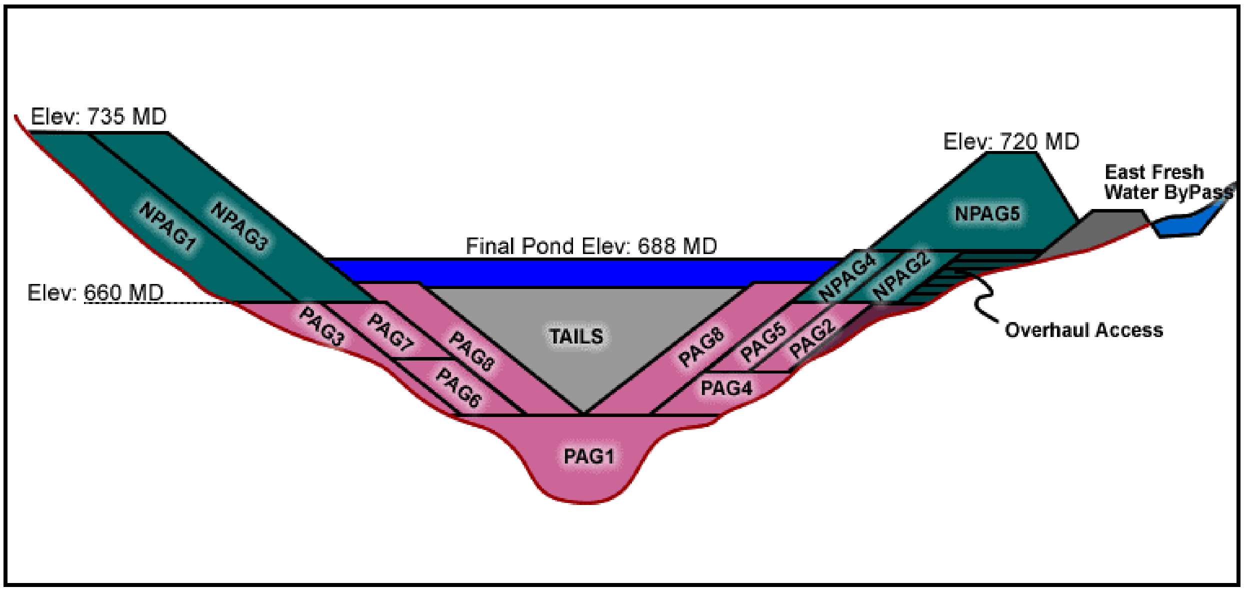
All mining materials in the Galore Creek deposit have been assigned a waste classification code on the basis of their potential for acid generation and their 'degree of fracture' (Broken and Stick).

Figure 5-14 shows the conceptual layout of waste storage for the Galore mine. Principals to govern waste placement were used as follows:

- It is recognized that PAG waste stored inappropriately could have negative environmental impacts and therefore must be managed to minimize the risks associated with its storage. To accomplish this task the waste management strategy for Galore Creek employs selective and sequenced placement of waste types.
- Waste storage sequencing.

- Till is placed between the freshwater bypass dam and the tailings impoundment area to prevent backflow from reservoirs to mining areas.
- Till is considered an NPAG waste.
- PAG waste will be submerged by end of mine life.
- NPAG waste does not require submergence and will be the predominant material in engineered structures such as the tailings dam and freshwater bypass dams.
- Waste will not be 'co-mingled' with tailings. Waste lifts are constructed with 'bottom up' precedence in the Galore Valley.

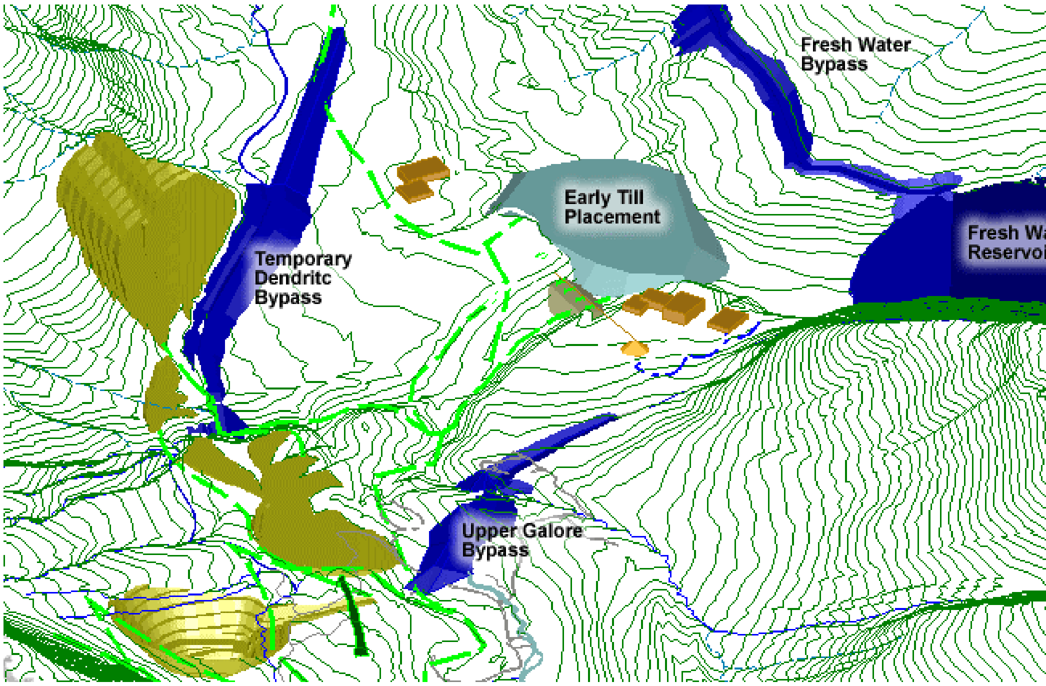
Figure 5-14: Conceptual Schematic of Waste Sequencing



5.5.4.1 Till Storage

The Figure 5-15 shows the initial placement of till volumes early in mine development. Initially, till will be placed from the crusher pad elevation, creating a platform to be used for the stockpiling of ore, however NPAG rock may be required as a cap to these dumps to ensure trafficability. Till forms a large portion of the total waste removed in the first few years. As pits are developed deeper it becomes a less prominent waste component.

Figure 5-15: Initial Till Placement

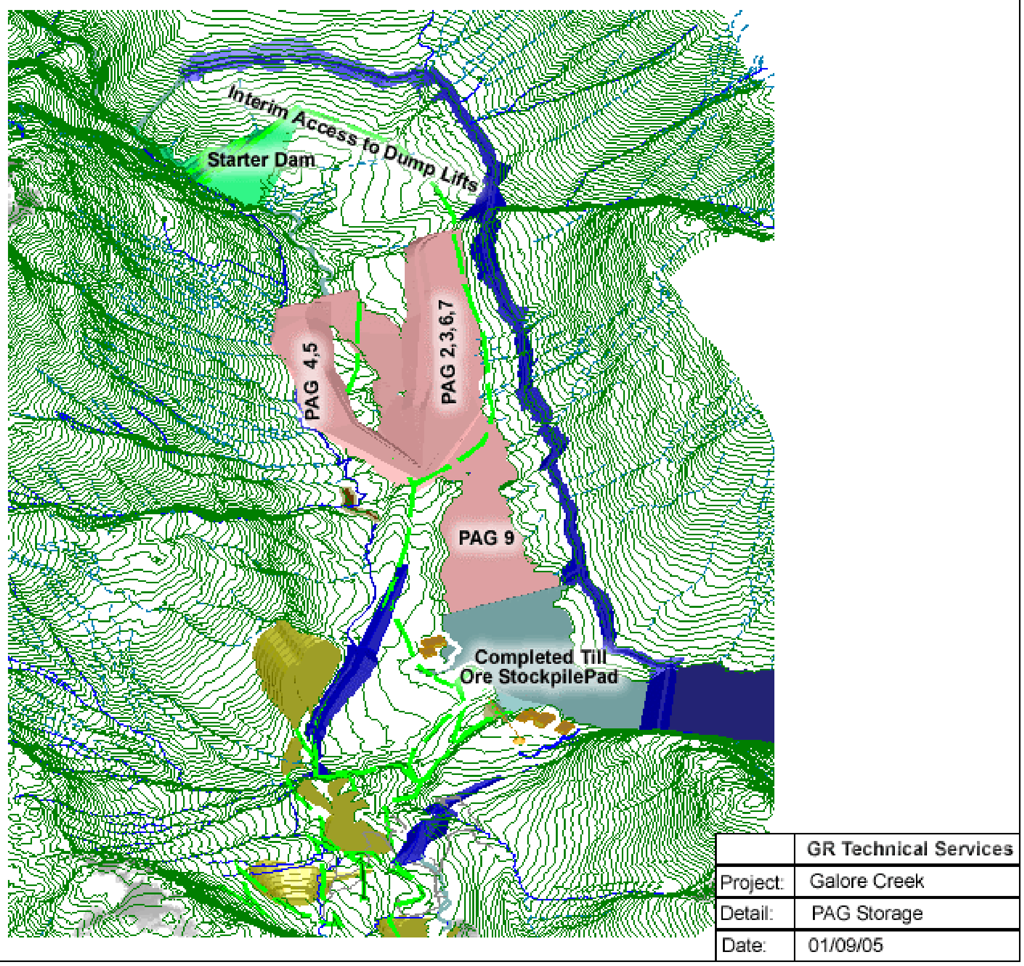


	GR Technical Services
Project:	Galore Creek
Detail:	Initital Till Placement
Date:	01/09/05

5.5.4.2 PAG Storage

PAG will be dumped in advance of NPAG on the valley flanks and in advance of the tailings beach advance in the center of the valley as shown in Figure 5-16.

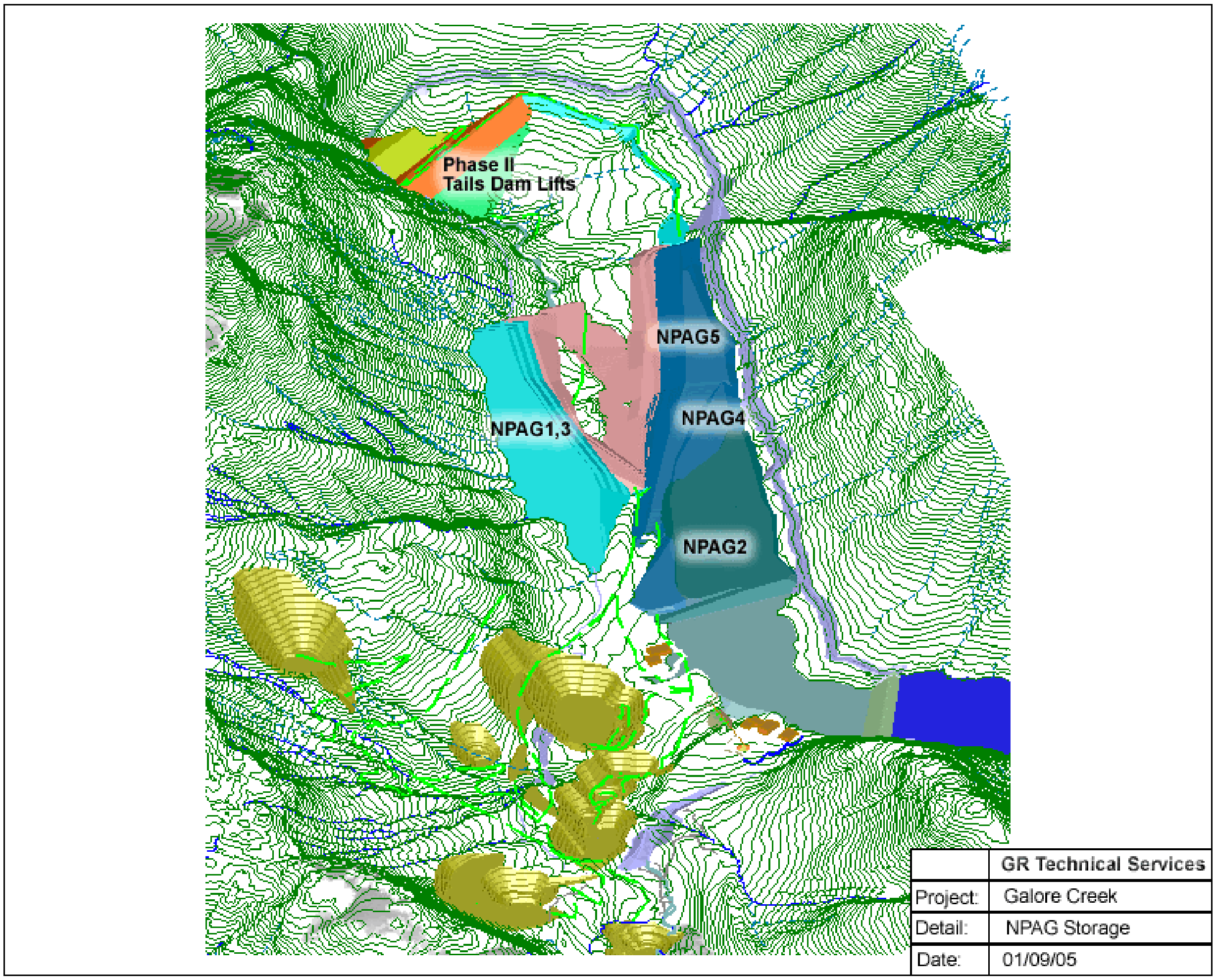
Figure 5-16: PAG Storage Configuration



5.5.4.3 NPAG Storage

Figure 5-17 presents the approximate pond elevations for every five years of mine life and the proposed downstream lift concept. Shape 1 represents the starter dam, 2 the downstream access for future NPAG areas (shown in shades of blue).

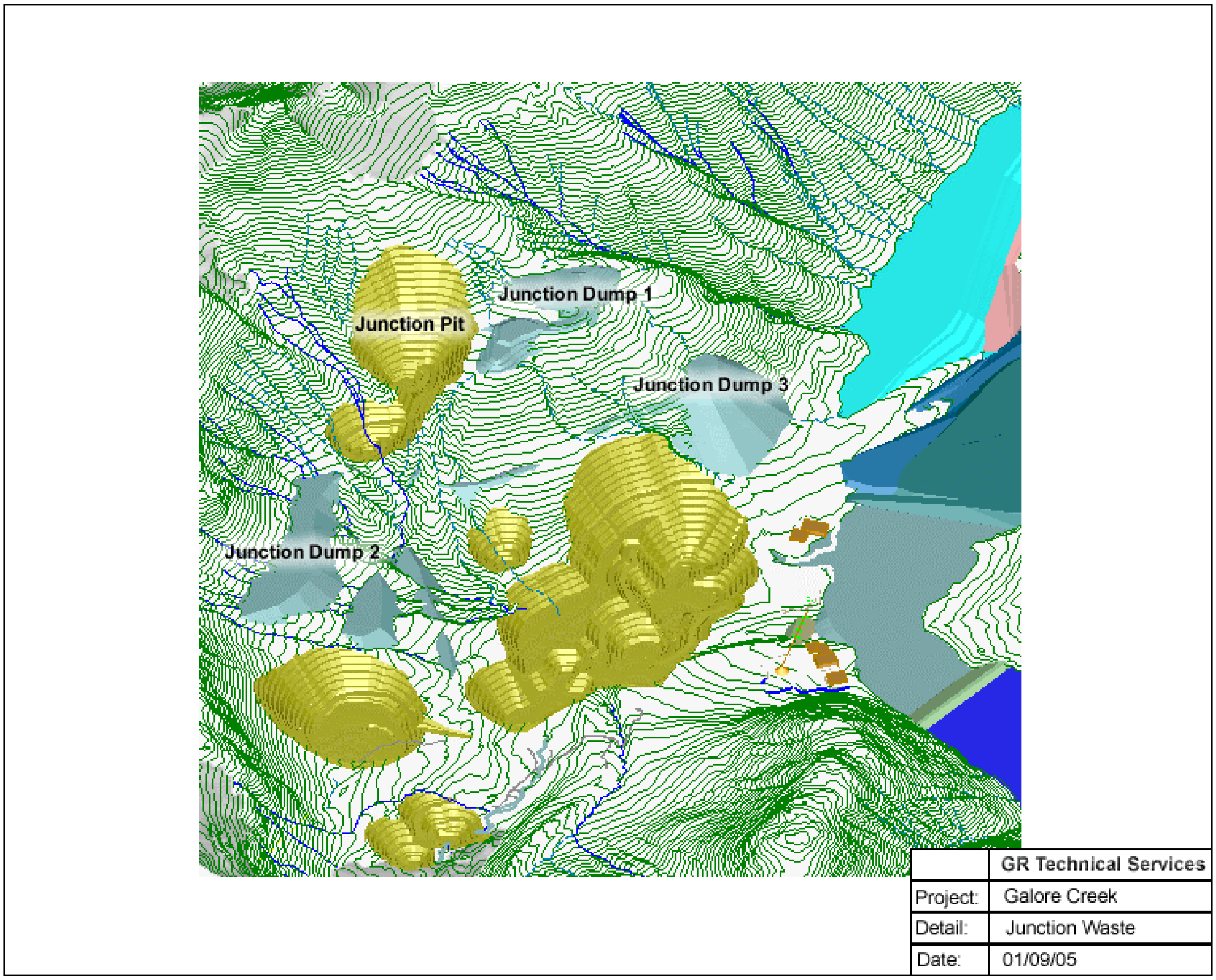
Figure 5-17 NPAG Storage Configuration



5.5.4.4 Junction Waste

These shapes are ex-pit NPAG areas for Junction waste (See Figure 5-18). There is more than enough volume in these shapes to accept all of the NPAG from Junction pits. The dump shapes will be built to complete design, presuming the foundation conditions are acceptable.

Figure 5-18 Junction NPAG waste dumps



6. Metallurgy

6.1 Summary

A comprehensive metallurgical program has been completed on samples from 2004 drilling to validate the flowsheet developed in the earlier work and to determine the metallurgy associated with the variable mineralization and head grades in the various zones of the Galore Creek deposit. The test program investigated grindability, mineralogy and minerals recovery by batch and locked cycle flotation. Coarse assay rejects were used in batch flowsheet development work while drill core composites of the primary mineral groupings were used in locked cycle tests to confirm circuit stability, metals recovery and concentrate grade. Models have also been developed to project copper, gold and silver recoveries in mining blocks and to identify gold occurrence and deportment.

Copper occurs predominantly as chalcopyrite and chalcopyrite-bornite in a mixed silicate host. Pyrite occurrence is variable, with pyrite-copper sulphide mass ratio averaging 3:1. Gold particles are fine at nominally 10 microns. Approximately 69% of the gold occurs with chalcopyrite, while the majority of the remainder occurs as inclusions in pyrite.

Ore hardness, in terms of Bond Ball Mill Work Index, varied widely from 9 kWh/t to 21 kWh/t across the deposit, including "broken" and "stick" ores. Similarly, the hardness, measured as SAG Power Index (SPI), ranged from 13 min. to 213 min. The results indicated that the plant throughput could be SAG mill limiting when treating certain ores within the deposit while it could be ball mill limiting with other ores. Further work is required to determine if these ores can be differentiated for production scheduling. A Bond Ball Mill Work Index of 16 kWh/t, corresponding to the average for the main ore types, has been used to design a SABC grinding circuit. The vast majority of ore types mined and processed in this PEA are estimated to have a Bond Index between 14 and 18 kWh/t.

The pre-feasibility level work program which provided the basis for this study validated the flowsheet developed in the previous scoping level work. The flowsheet will comprise rougher flotation, regrind of rougher concentrate and three stages of cleaner flotation. Gravity concentration for gold has been excluded from the flowsheet. Preliminary tests using a combined gravity concentration – flotation circuit did not produce significant incremental gold recovery over that obtained from direct flotation.

Locked cycle flotation showed that high copper and gold recoveries could be achieved in a concentrate grading 26% Cu to 32% Cu, depending on the head grade, from the more dominant chalcopyrite and chalcopyrite-bornite ores.

For chalcopyrite-bornite ores with 1% Cu head grade, recoveries of 93% Cu, 76% Au and 84% Ag were achieved in a concentrate grading 32% Cu. At a lower head grade of 0.4% Cu, the recoveries were lower at 91% for Cu, 71% for Au and 57% for Ag in a concentrate grading 27% Cu.

For chalcopyrite ores with 1% Cu head grade, the recoveries were 96% Cu, 81% Au and 79% Ag in a concentrate grading 26% Cu. With a 0.4% Cu head grade, the recoveries dropped to 86% Cu, 61% Au and 59% Ag in a concentrate grading 28% Cu.

Significant amounts of calcite, biotite and chlorite, which adversely affected flotation, were found in two samples (4% of total) from the Central and Copper Canyon zones. (Note: Copper Canyon is not part of the currently defined mine schedule.)

The final concentrates appear readily marketable and had relatively low penalty elements. Fluorine and selenium concentrations were variable and could slightly increase the cost of processing at some smelters. The lead content might be a concern depending on the end-use of the slag from smelters.

Copper recovery for variable Cu head grades can be estimated with reasonable accuracy based on a constant tailings model. This model would tend to over-estimate the recovery at high Cu head grades (i.e. greater than 0.8% copper) and under-estimate at lower head grades (below 0.8% copper). At the pre-feasibility stage, this is considered to be a valid and reasonable model for projecting recoveries in mining blocks across the deposit.

Gold and silver recoveries depend largely on copper recovery in flotation concentrate and a model has been developed to project their recoveries based on copper recovery. As such, the gold in ores with very low copper, and largely occurring within pyrite grains, would not be recovered.

6.2 Introduction

The initial work by Kennecott Corporation in the 1960's focused on the Central Zone and identified the principal rock types and minerals as syenite, leucodiorite, tactite and diorite. The sulphidic minerals within the rock types were identified as chalcopyrite, sphalerite and pyrite.

Work continued in 1991 on the Central Zone and the newly identified Southwest Zone. Chalcopyrite or pyrite was identified as the predominant mineral depending on the location in the deposit. Bornite was identified as a minor constituent. Potassium feldspar and quartz were the most abundant gangue minerals followed by albite, biotite and calcite.

The 1991 work did not replicate the 1960's results in that the copper recovery and concentrate grade were generally lower at similar process conditions. This was attributed to the presence of "talc" which was observed in the 1991 samples but not in the 1960's.

In 2003, NovaGold initiated a drilling program on the Central and Southwest Zones to verify previous results and to further understand the mineralogy and variability in the two zones. The accompanying test program, as part of a scoping study, investigated the metallurgy of selected high grade drill core samples under the current industrial flotation practices and reagent scheme for comparison with the previous results at lower grades. The work validated the flowsheet developed in the 1960's, producing comparable recovery and concentrate grade.

Exploration activity continued in 2004 for this PEA, expanding understanding of the variability of mineralization in the Central and Southwest Zones and identifying additional ore zones designated as Junction, West Fork and Copper Canyon. As a result, a comprehensive metallurgical program has been conducted to determine the metallurgy associated with the variable mineralization. The samples tested included coarse-crushed assay rejects and drill cores from discrete intervals, drill hole composites and composites of drill holes by mineral grouping. They represented ore at various head grades, mineralogy and geological classification of "broken" or "stick" ore. The program also included a grinding test program on drill core samples for grinding circuit design.

The 2004 metallurgical test program has been conducted in two phases:

- **Phase I:** program included grindability tests, mineralogical analysis, preliminary batch flotation tests to validate the previously developed flowsheet and locked-cycle flotation. A model was also developed to identify gold occurrence and deportment.
- **Phase II:** program attempted to characterize the metallurgical responses of the various ore zones by relating metals recovery to head grade.

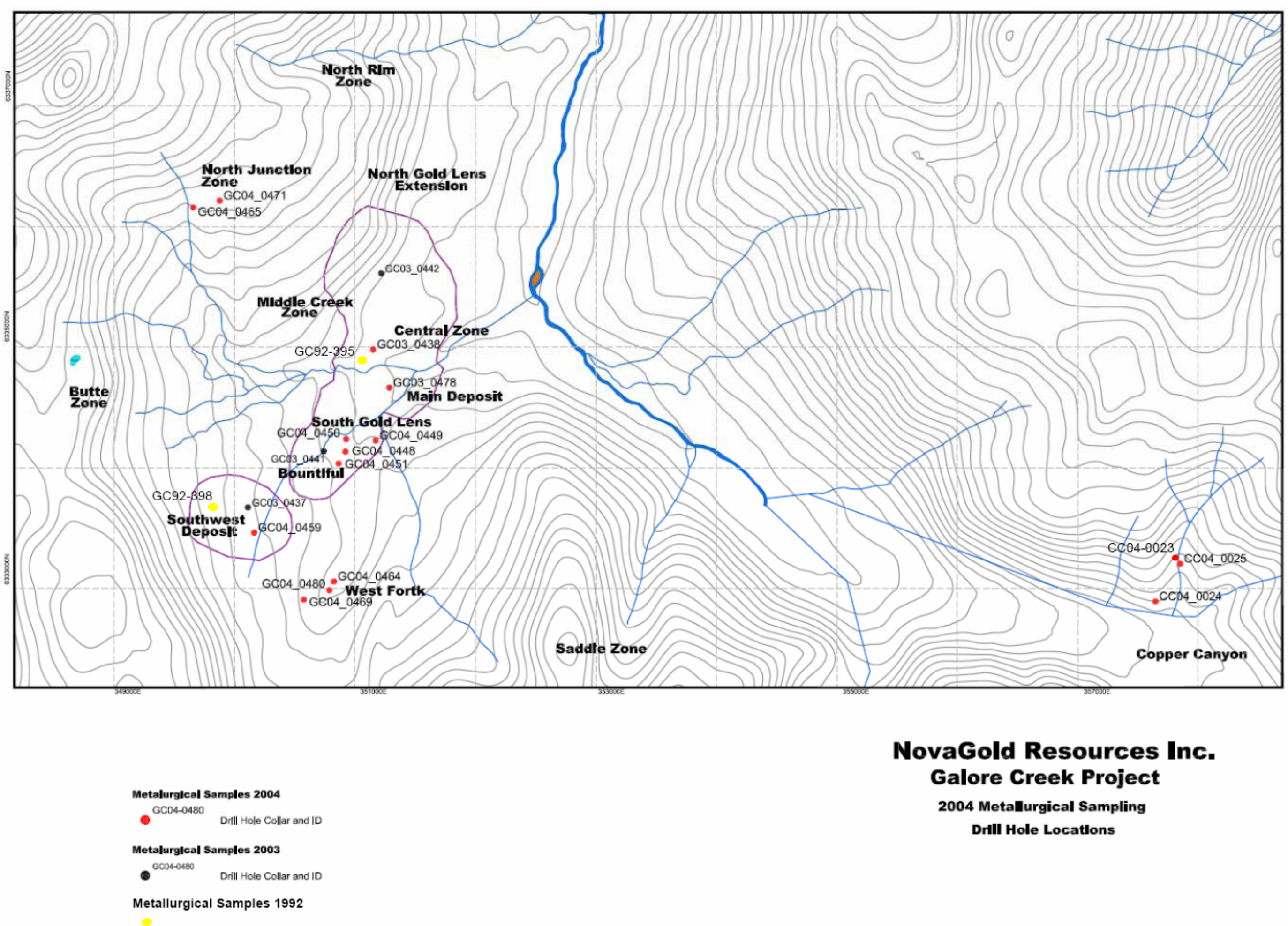
6.3 2004 Metallurgical Testwork Summary

All flotation and standard Bond Ball Mill Work Index testwork were conducted at G&T Metallurgical Services Ltd (Kamloops, BC, Canada), while additional grindability tests were run at MinnovEX Technologies Inc. (Toronto, ON, Canada).

6.3.1 Phase I Program

Laboratory batch open-circuit and locked-cycle flotation tests and grindability tests have been conducted on samples from individual drill holes and drill hole composites. The samples for the test programs have been selected based on geological mineralization groupings, ore zones, copper head grade, and locations in the deposit, with input from NovaGold geologists. The site drill hole locations are shown in Figure 6-1.

Figure 6-1: 2004 Drill Hole Map



6.3.2 Phase II Program

In the Phase II program, twenty-eight assay reject samples from discrete drill hole intervals of different head Cu grades were selected for the variability test. Comparative Bond Ball Mill work Index and single batch cleaner flotation tests were run on each of the samples to investigate grade-recovery relationships.

6.4 2004 Phase I Program

The samples from the different ore zones were characterized in terms of mineralogy, hardness and mineral fragmentation then floated in batch open-circuit and locked cycle runs to determine the metallurgy associated with the zones. Correlations have been developed to identify gold occurrence and deportment and to project metal recoveries.

6.4.1 Ore Characterization

The principal copper mineral in the deposit is chalcopyrite in a mixed silicate host. Some of the samples showed chalcopyrite-bornite mineralization. Pyrite occurrence was variable, with a pyrite to copper sulphide mass ratio averaging 3:1 in the suite of samples listed above. Ore hardness, in terms of Bond Ball Mill Work Index, was determined for each of the drill hole composites and the mineral groupings listed above, and shown in Table 6-1.

There was no distinction between “broken” and “stick” ores according to the work index measurements. A Bond Work Index of 16 kWh/t has been selected as the criteria for mill sizing based on the data in Table 6-1 and the dominance of chalcopyrite and chalcopyrite-bornite ores in the deposit.

Table 6-1: Ore Hardness of Samples by Mineral Zones

Sample ID	Ore Classification	Bond Work Index kWh/t
GC04-0451 SGL	Broken	14.9
GC04-0459 SW	Stick	14.7
GC04-0448 Bountiful	Stick	16.0
GC04-0465 NJ	Broken	13.7
GC04-0469 WF	Stick	17.0
CC04-0025 Cu Fluid	Broken	17.5
GC04-0451 Cu Ox	Broken	15.8
GC04-0478 Rep Cu	Stick	16.5
Average		**15.8**
Chalcopyrite-Bornite		16.2
Chalcopyrite		15.4
Pyrite-Gold		15.0
Cu Canyon		20.2

6.4.2 Flowsheet Validation

Batch open-circuit rougher and cleaner flotation tests have been conducted based on the flowsheet developed in the earlier work and using the current industry standard reagents. The flowsheet consisted of rougher flotation, regrind of rougher concentrate and three-stage cleaner flotation. Assay rejects with head grades of 0.10% Cu to 0.54% Cu were used for the tests. The work validated the flowsheet.

6.4.3 Locked Cycle Flotation

Locked cycle flotation tests have been carried out to determine the stability of the circuit and metals recovery and concentrate grade achievable in a continuous circuit. The samples for the tests were selected by NovaGold to represent the main ore types in the deposit. These samples were grouped as follows:

- Chalcopyrite-bornite ore, high and low Cu head grades, composited from North Gold Lens and Bountiful in the Central Zone, North Junction and West Fork Zones drill holes.
- Chalcopyrite ore, high and low Cu head grades, composited from Central Zone drill holes.
- Pyrite-gold ore from Southwest Zone
- Copper oxide ore
- Copper-gold fluids from Copper Canyon

The results for all the ores have been tabulated in Table 6-2.

Table 6-2: Locked-Cycle Flotation Results

Mineral Group	Head			Final Recovery, %			Concentrate
	% Cu	g/t Au	g/t Ag	Cu	Au	Ag	% Cu
Chalcopyrite-bornite	1.12	0.9	8.8	92.7	75.3	83.1	32.6
	1.07	0.8	8.4	92.6	75.7	84.1	32.0
	0.38	0.2	3.5	90.1	79.7	40.5	27.7
	0.38	0.3	2.3	90.6	70.7	57.4	26.7
Chalcopyrite	1.06	0.3	12.7	95.8	81.0	78.9	25.7
	0.96	0.2	10.9	89.0	57.2	66.6	32.7
	0.46	0.1	8.1	88.3	63.8	53.6	28.6
	0.41	0.1	5.9	86.3	60.8	59.1	27.6
Pyrite-Gold	0.68	1.41	5	84.1	55.9	53.8	28.4
Copper Oxide	0.58	0.4	3.1	64.6	62.7	61.1	33.6
	0.56	0.3	2.9	64.9	72.9	62.0	31.6
Copper Canyon	0.72	1.0	15.7	77.6	61.8	65.8	25.8
	0.73	1.1	18.2	69.1	56.0	42.5	25.5

6.4.4 Gold Behaviour Model

A gold behaviour model has been developed to identify its occurrence and deportment to the various flotation products. The model has been based on direct searches for gold occurrences using the Automated Digital Imaging System (ADIS) and a statistical analysis of flotation data.

The ADIS searches identified gold occurrences as fine particles of nominally less than 10 µm. Approximately 69% occurred as liberated gold and gold locked with chalcopyrite. These occurrences are generally recovered in flotation to the copper concentrate. The majority of the remaining gold occurred as fine inclusions in relatively large pyrite particles, which will not be recovered by flotation because the pyrite would be depressed during the upgrading of the final copper concentrate. This fraction of gold would unlikely be recovered by gravity concentration, but this should be investigated further.

A statistical analysis of the data from copper flotation, separate pyrite flotation to investigate gold behaviour, and sulphur balance have produced a correlation of gold recovery with copper sulphide, pyrite and gangue recovery. Gold recovery was relatively consistent at approximately 65% of copper sulphide recovery, regardless of ore type. The gold-pyrite and gold-gangue recovery correlations, however, were variable. The gold-pyrite recovery appeared to be impacted by the pyrite content in the feed. The overall gold recovery in concentrate would be dependent on the copper and pyrite contents in the feed and their deportment to concentrate. This indicated that the gold recovery would be low in ores with low copper and relatively high pyrite contents as the majority of the gold would occur with pyrite and would be rejected in flotation. Similarly, the small fraction of gold associated with gangue would largely be rejected in flotation.

6.4.5 Gravity Concentration

Flotation tests, with and without prior gravity concentration, have been run to determine the potential for improving overall gold recovery with the addition of gravity concentration. Gravity concentration has been effective in operations where some of the gold is not recoverable by flotation.

The tests on various mineralogical groups indicated only a small increase in overall gold recovery in a few samples in a gravity-flotation circuit. Overall gold recovery by flotation only was generally just as high in most of the samples, indicating that a gravity concentration circuit would not be required.

6.4.6 Prefloat

Of the 46 samples tested in this PEA, only two were observed to generate slimes that interfered with flotation. Similar observation was reported in the 1991 study where the poor metallurgical responses were attributed to "talc". There was no work, however, to confirm the causes. It is noted that samples from other drill holes around these did not yield the problematic characteristics. It appears then that the occurrence of the interfering minerals is spotty in the deposit. Further work is required to identify the extent of the occurrences of these minerals and their significance on the overall metallurgy of the deposit.

6.4.7 Magnetite Recovery

Geological data from NovaGold has shown the occurrence of magnetite in the Galore Creek deposit. This was supported by the mineralogical analysis of the ore samples, which showed that some of the samples contained approximately 3% magnetite by weight. Consequently, scoping magnetic separation tests using a Davis Tube were run on rougher tailings to investigate the viability of magnetite recovery.

The Davis Tube recovered 2% to 3% by weight of magnetics from the rougher tailings. A preliminary assessment of the magnetic concentrate showed a low quality of approximately 35% iron and 25% silica. This would require multiple stages of magnetic separation and perhaps other processes to upgrade the magnetite to saleable quality. It would not be economical to include this magnetite recovery circuit in the copper flotation plant considering the low mass recovery and substantial processing requirements.

6.4.8 Concentrate and Tailings Settling

Preliminary standard bench settling tests have been made on the copper concentrate and final tailings to estimate their thickener requirements. Concentrate thickening would be required prior to filtration. Tailings thickening would be an option if it was necessary to recycle process water immediately and decrease the pumping of tailings to the pond and reclaim water from the pond.

Flocculant may not be required for concentrate settling given the small reduction in thickener requirement. Flocculant addition to the final tailings, however, would be necessary to decrease the thickener size by approximately four times using a nominal dosage of 10 g/t.

Table 6-3: Final Tailings Thickener Requirements

Thickener feed slurry density	25% solids
Thickener underflow density	60% solids

Ore Type	Floc Dosage g/t	Unit Area m^2/tpd
Chalcopyrite-bornite		
low grade	0	0.231
	10	0.062
high grade	0	0.227
	10	0.072
Chalcopyrite		
low grade	0	0.278
	10	0.069
high grade	0	0.177
	10	0.063
Pyrite-Gold	0	0.185
	10	0.045
Cu Oxide	0	0.393
	10	0.090
Cu Canyon	0	0.498
	10	0.199

6.4.9 Concentrate Quality

The quality of the concentrates produced in the locked cycle tests for the various ore types has been tabulated in Table 6-4 .

The concentrates are relatively “clean” with relatively low penalty elements. Fluorine and selenium contents are variable and may be of concern. All the lead in the concentrates occurred as galena and there was no non-sulphidic lead. The deportment of lead to concentrate was variable and it appeared to be affected by the copper and pyrite contents in the ore. The lead occurrence and deportment indicated that the proposed flotation circuit would be relatively ineffective to depress the lead.

The spatial distribution and occurrence of lead in the Galore Creek deposit needs to be defined to assess the potential impact on concentrate quality, and the actual need for further separation.

Table 6-4: Assay of Locked-Cycle Concentrates

Ore Type	Cu %	Fe %	Au g/t	Ag g/t	Pb %	Zn %	Ni ppm	Co ppm	As ppm	Sb ppm	Bi ppm	Mo %	MgO %	Se ppm	SiO2 %	Hg ppm	Pt ppm	Pd ppm	Rh ppm	F ppm	Cl %	Te ppm
Chalcopyrite-bornite																						
high grade ore	32.6	26.6	22	231	0.254	0.182	38	84	135	289	<9	0.002	0.41	114	6.97	2	0.08	1.48	0.09	440	0.02	15.78
low grade ore	26.7	28.9	16	102	0.222	0.320	48	162	17	181	<9	0.002	0.26	170	8.29	3	0.12	2.2	<.01	340	<.01	13.8
Chalcopyrite																						
high grade ore	25.7	33.7	6	252	0.311	0.92	58	404	28	48	<9	0.006	0.07	140	2.28	2	0.05	0.37	0.02	290	<.01	9.51
low grade ore	27.6	33.3	6	270	0.173	1.11	46	164	29	60	<9	0.003	0.08	118	1.89	1	0.13	0.89	<.01	230	<.01	7.04
Pyrite-gold	28.4	29.8	38	123	0.044	0.135	24	54	<10	48	<9	0.001	0.57	128	6.78	2	0.05	0.71	<.01	640	<.01	12.54
Copper oxide	33.6	22.8	22	168	0.398	1.36	34	54	609	289	<9	0.013	0.19	189	6.85	2	<.01	0.79	<.01	330	<.01	7.66
Copper Canyon	25.8	30.1	28	477	0.182	2.66	30	250	332	145	<9	0.025	0.34	159	4.75	2	0.01	0.11	0.01	1230	<.01	11.42

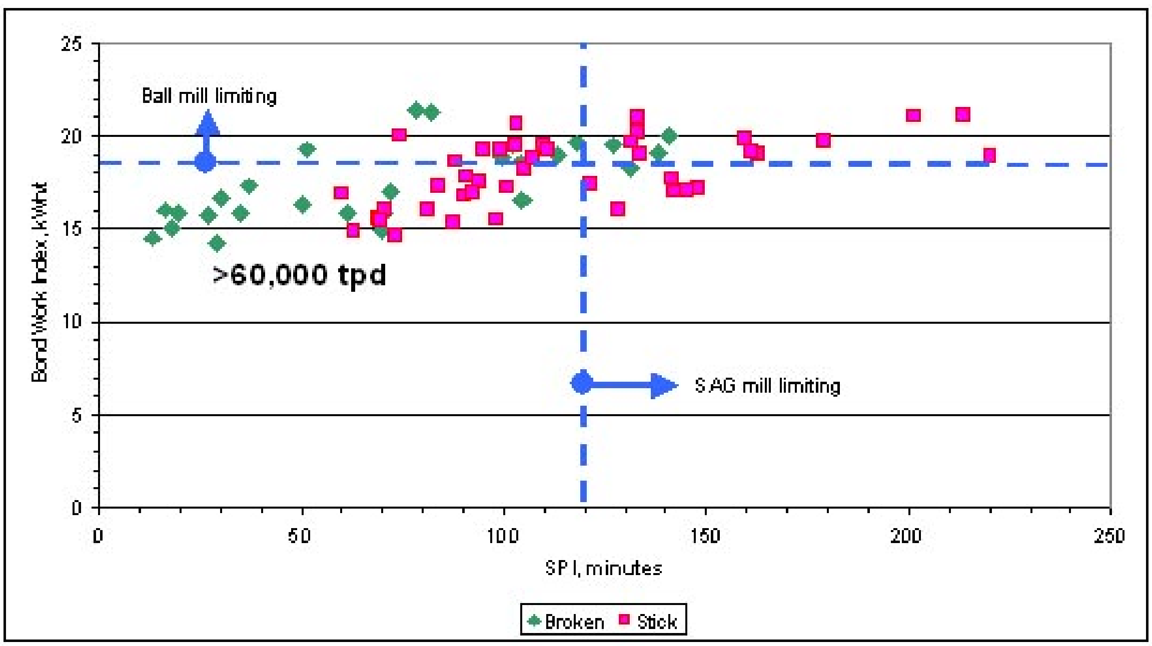
6.4.10 Grinding Program (MinnovEX)

Sixty-nine drill core samples, discrete intervals and composited by drill hole, were submitted to MinnovEX for grinding study and grinding circuit design using their CEET technology.

The SPI cumulative frequency distribution obtained has shown that the hardness of the Galore Creek ore is within the range observed for other operating copper porphyry mines. Its distribution, however, is narrower than that evidenced at other mines and is an indication that more samples need to be tested to ensure proper representation of the deposit.

The MinnovEX CEET analysis also indicated that the SAG mill would be limiting plant throughput when processing ores with an SPI of greater than 120 min., while the ball mills would be limiting when treating ores with a Bond Ball Mill Work Index of greater than 18.5 kWh/t. Figure 6-2 illustrates the variability of ore hardness and suggests that the circuit could process more than 65,000 tpd ore if its SPI is less than 120 min and the Bond Index is less than 18.5 kWh/t. It also shows that it is possible for “broken” ore to limit SAG mill capacity. More testwork is required to assess if the deposit could be characterized by its mill-limiting tendencies to assist production scheduling. Other comminution options and strategies will also be investigated in subsequent study.

Figure 6-2: Galore Creek Ore Grindability – 2004 Drill Cores



6.5 2004 Phase II Program

The 2004 Phase II program involved a series of characterization tests to investigate the metallurgical variability across the deposit. The objectives of this test phase were to assess the impact of the variable mineralization on flotation metallurgy based on a common flowsheet and to develop a method to project metallurgical responses for mining blocks based on head grades.

Comparative Bond Work Index and batch cleaner flotation tests, based on the flowsheet developed in Phase I, were conducted on 28 assay reject samples of drill hole intervals covering a range of copper head grades and ore types. Assay rejects were used for the program because drill cores were not available and the initial tests on other assay reject samples had shown sufficient integrity for the purpose. Flotation parameters were not varied to suit the variability in the mineralization.

6.5.1 Ore Hardness

Ore hardness was quite variable between intervals within the same drill hole and between drill holes. The Bond Work Index ranged from 8.8 kWh/t to 20.8 kWh/t. The average of the sample suite was 13.5 kWh/t. There was no correlation between ore hardness and feed copper grade which varied from 0.1% Cu to 1.6% Cu. The Copper Canyon samples were the hardest (20.8 kWh/t) and here again it is important to note that no ores from Copper Canyon were included in the mine schedule in this PEA.

The average Bond Work Index was lower than the 16 kWh/t obtained initially for the main mineral zones and the dominant chalcopyrite and bornite ores. Consequently, the 16 kWh/t index has conservatively been retained for pre-feasibility mill sizing. This will be confirmed or possibly reduced in the feasibility study.

Table 6-5: Ore Hardness Variability

Sample ID	**Work Index kWh/t**
GC04-0464 West Fork – 109820	13.1
GC04-0464 West Fork – 109810	12.8
GC04-0480 West Fork Opulent 110387+110391	8.8
GC04-0480 West Fork Opulent 110394	9.2
GC04-0469 West Fork 110070	14.8
GC04-0469 West Fork 110062+110064	13.2
GC04-0469 West Fork 110050+110051	18.7
GC04-0469 West Fork 110052	13.6
GC04-0469 West Fork 110068	14.8
GC04-0450 Bountiful 101722+101723	16.4
GC04-0450 Bountiful 101730	10.7
GC04-0450 Bountiful 101732	12.8
GC04-0449 Bountiful 102721	10.3
GC04-0450 SGL 101875	13.8
GC04-0450 SGL 101869	18.7
GC04-0465 Junction 112682+112684	11.3
GC04-0465 Junction 112674+112678	11.2
GC04-0465 Junction 112679+112681	9.4
GC04-0471 Junction 112844+112852	10.1
GC04-0459 SW 112541+112550	10.2
GC04-0462 SiO2 Pyrite-Au	18.0
GC04-0478 111163+111175	10.6
GC04-0478 111177+111178	14.7
GC04-0478 111172+111182	16.1
GC04-0478 111188+111190	15.8
GC04-0024 Cu Canyon 103368	18.1
GC04-0024 Cu Canyon 103376+103384	20.8
Average	**13.5**

6.5.2 Chemical and Mineralogical Analysis

An assessment of the sulphur analysis showed that approximately half of the total sulphur occurred as sulphide sulphur, and virtually all the copper occurred as a sulphide. There was insignificant amounts of non-sulphide copper based on the analysis of total copper [Cu(t)] and sulphide copper [Cu(s)]. Chalcopyrite was the dominant copper mineral. Small amounts of bornite were found in only four of the samples. The sulphide minerals occurred within a mixed silicate host.

The analysis also indicated the presence of minor amounts of pyrite in the majority of the samples and would likely have little impact on copper and gold metallurgy in most of the samples.

Table 6-6: Chemical Composition of Variability Samples

Sample ID	Cu(t) %	Cu(s) %	Fe %	S(t) %	S(SO4) %	S(s) %	Ag g/t	Au g/t
GC04-0464 West Fork - 109820	0.35	0.33	4.5	1.5	0.9	0.6	2	0.90
GC04-0464 West Fork - 109810	0.23	0.22	3.3	2.0	1.6	0.4	2	0.13
GC04-0480 West Fork Opulent 110387+110391	4.24	4.16	29.1	6.7	2.9	3.8	6	1.35
GC04-0480 West Fork Opulent 110394	8.60	8.31	41.8	6.9	2.6	4.3	62	1.54
GC04-0469 West Fork 110070	1.00	0.98	3.7	2.7	1.2	1.5	2	0.27
GC04-0469 West Fork 110062+110064	0.50	0.48	2.1	3.3	3.1	0.2	2	0.47
GC04-0469 West Fork 110050+110051	0.63	0.62	4.7	3.3	1.0	2.2	10	0.97
GC04-0469 West Fork 110052	2.34	2.28	6.3	6.4	0.5	5.9	56	3.41
GC04-0469 West Fork 110068	1.44	1.40	3.1	2.5	1.7	0.8	6	0.72
GC04-0450 Bountiful 101722+101723	0.47	0.46	7.1	4.0	0.9	3.1	8	0.16
GC04-0450 Bountiful 101730	0.83	0.81	7.9	4.4	1.3	3.0	18	0.52
GC04-0450 Bountiful 101732	0.33	0.32	6	3.8	1.0	2.8	8	0.29
GC04-0449 Bountiful 102721	0.21	0.21	9.5	3.8	1.8	2.0	4	0.13
GC04-0450 SGL 101875	0.20	0.20	3.7	5.1	2.5	2.6	2	0.05
GC04-0450 SGL 101869	0.06	0.06	3.5	2.6	2.2	0.4	2	0.03
GC04-0465 Junction 112682+112684	0.21	0.20	5.4	5.0	3.3	1.7	2	0.53
GC04-0465 Junction 112674+112678	1.15	1.13	2.2	4.6	3.7	0.9	10	2.20
GC04-0465 Junction 112679+112681	0.54	0.54	3	4.9	4.3	0.6	6	1.56
GC04-0471 Junction 112844+112852	0.39	0.38	6.2	1.2	1.0	0.3	2	0.13
GC04-0459 SW 112541+112550	0.07	0.07	8.5	8.4	2.9	5.6	6	1.14
GC04-0462 SiO2 Pyrite-Au	0.04	0.04	3.5	2.4	0.2	2.2	8	3.72
GC04-0478 111163+111175	0.65	0.64	9.8	4.3	1.7	2.6	14	0.07
GC04-0478 111177+111178	1.70	1.67	7.8	5.6	1.3	4.3	10	0.14
GC04-0478 111172+111182	0.52	0.51	7.3	3.5	1.9	1.6	4	0.05
GC04-0478 111188+111190	0.33	0.32	7.7	6.6	0.9	5.7	2	0.06
GC04-0024 Cu Canyon 103368	0.04	0.04	4.6	4.5	0.2	4.3	2	0.70
GC04-0024 Cu Canyon 103376+103384	0.10	0.10	4.8	4.3	0.1	4.1	258	2.07

6.5.3 Metallurgical Variability Testing

The same flotation protocol was used in single open circuit batch cleaner flotation test on each sample to assess the metallurgical variability in the deposit. With the limited sample sizes, only single flotation tests were conducted and primary grinding time was established for each sample to target a K80 of 150 µm, based on a single grind calibration and the comparative Bond Work Index. The results for the variability program has been tabulated in Table 6-7.

As may be seen, initial baseline tests on assay rejects from the main ore zones and locked cycle tests on drill core sample produce lower and more variable metal recoveries and concentrate grades. This may be due to variability in mineral locking and surface chemistry and/or insufficient regrinding and mineral liberation in the test procedure. The lower recoveries may also be the result of alterations to the sulphide surfaces considering that the samples had been crushed and exposed to the atmosphere for a prolonged period of time prior to testwork. Further variability testing will be conducted on fresh drill core samples as part of the Feasibility Study program.

Any sulphide surface alterations would not affect ore hardness and the grindability results would still be valid.

As a result of the anomalous metallurgical responses compared with all previous testwork, a head grade versus recovery correlation could not be developed for projecting recoveries in mining blocks. Instead, an alternate method has been developed based on the locked cycle testwork.

Table 6-7: Metallurgical Responses from Variability Samples

Sample ID	Head			Conc Grade	Concentrate Recovery, %			
	% Cu	g/t Au	g/t Ag	% Cu	Mass	Cu	Au	Ag
GC04-0464 West Fork - 109820	0.34	0.88	2.7	15.4	1.4	63.9	30.4	42.8
GC04-0464 West Fork - 109810	0.23	0.13	1.9	16.0	0.9	61.3	47.6	31.5
GC04-0480 West Fork Opulent 110387+110391	4.59	1.47	6.1	32.2	12.1	85.0	47.4	39.6
GC04-0480 West Fork Opulent 110394	9.27	1.95	64.3	55.7	13.3	79.9	59.9	75.3
GC04-0469 West Fork 110070	1.08	0.36	1.9	27.1	3.4	84.1	74.1	45.0
GC04-0469 West Fork 110062+110064	0.51	0.55	3	42.3	0.9	79.5	68.5	62.7
GC04-0469 West Fork 110050+110051	0.63	1.05	12	10.7	5.1	85.9	77.2	71
GC04-0469 West Fork 110052	2.38	3.67	35.8	18.1	10.7	81.3	78.5	67.9
GC04-0469 West Fork 110068	1.49	0.75	6.4	36.4	3.4	82.5	78.0	78.5
GC04-0450 Bountiful 101722+101723	0.47	0.14	8.1	11.7	3.2	80.4	55.7	36.4
GC04-0450 Bountiful 101730	0.86	0.45	16.4	21.7	2.7	68.9	55.8	54.5
GC04-0450 Bountiful 101732	0.31	0.33	8.8	7.1	2.2	49.5	38.3	31.0
GC04-0449 Bountiful 102721	0.2	0.08	3.7	12.1	1.1	67.6	54.2	35.5
GC04-0450 SGL 101875	0.19	0.06	2.4	4.6	2.6	65.4	48.3	37.7
GC04-0450 SGL 101869	0.04	0.03	1.9	6.2	0.5	70.0	53.3	26.6
GC04-0465 Junction 112682+112684	0.19	0.56	1.7	5.0	2.2	58.9	42.7	31.4
GC04-0465 Junction 112674+112678	1.20	2.82	10.4	28.0	2.8	66.1	50.5	48.8
GC04-0465 Junction 112679+112681	0.55	1.35	6	27.2	1.3	63.2	50.9	32.7
GC04-0471 Junction 112844+112852	0.37	0.14	1.3	21.5	0.5	27.7	37.4	19.8
GC04-0459 SW 112541+112550	0.07	1.11	2.3	0.9	4.0	53.9	32.4	27.0
GC04-0462 SiO2 Pyrite-Au	0.05	3.76	6.5	1.5	2.1	65.9	49.7	45.3
GC04-0478 111163+111175	0.64	0.12	5.1	14.7	3.3	76.8	56.5	49.3
GC04-0478 111177+111178	1.57	0.17	9.8	21.7	5.9	81.5	44.3	63.9
GC04-0478 111172+111182	0.53	0.10	3.9	16.3	2.8	85.4	63.7	45.5
GC04-0478 111188+111190	0.31	0.08	2.5	11.6	2.0	75.8	37.2	42.0
GC04-0024 Cu Canyon 103368	0.03	0.73	3.8	1.1	0.4	14.7	19.0	6.4
GC04-0024 Cu Canyon 103376+103384	0.06	1.86	197	1.5	0.4	9.5	19.6	6.8

6.6 Metals Recovery Projection

6.6.1 Summary

A pre-feasibility level metallurgical test program has been completed for the PEA. The program included preliminary flotation tests to identify a common flowsheet for the mineralized groups in the various ore zones using coarse assay reject samples, followed by locked-cycle flotation tests on drill cores of the main ore types to confirm the flowsheet, metals recovery and concentrate grade.

All the samples for the metallurgical program were selected according to the database of mineralized groups by drill holes provided by NovaGold Resources. Tests have been run on discrete drill hole intervals, composites of individual drill holes and composites of various drill holes, depending on the objectives in the program.

In the preliminary test phase, the tests were run on individual drill hole composites from the following zones: Central/Main, Junction, Bountiful, Southwest, West Fork, and Copper Canyon. Copper Canyon testwork is summarized in this section although no materials from the Copper Canyon area are included in the PEA mine plan. A sample from South Gold Lens identified as copper oxide (oxides are infrequent in the overall deposit) was also tested for completeness.

In the locked-cycle test phase, the ore types tested were chalcopyrite, chalcopyrite-bornite, pyrite-gold, copper oxide, and mixed copper-gold fluids (Copper Canyon). The chalcopyrite and chalcopyrite-bornite ores represent by far the more abundant ore types in the overall deposit and, consequently, more tests were run on these ores covering low and high copper head grades, and their results have been used to project copper, gold and silver recoveries.

An analysis of all the test results, particularly from the locked cycle tests, have shown that it is reasonable to project copper recovery for a range of Cu head grade based on constant Cu assay in tailings. The use of constant tailings assay would tend to over-estimate somewhat the recovery at high Cu head grades (i.e. >0.8%) and under-estimate at lower head grades (below 0.8%).

The analysis has also indicated a significant dependence of gold and silver recoveries on copper recovery. A gold behaviour model, based on mineralogical analysis, supports the metallurgical data in that approximately 65% of the gold occurrence reported with Cu while the majority of the remaining Au reported with pyrite. Furthermore, in Au-rich ores but with very low Cu, the Au was observed to largely occur within pyrite grains and would not be recovered in the Cu concentrate.

From the data analysis, Cu, Au and Ag recoveries may be estimated using the following equations for a 28% Cu grade concentrate, based on a constant tailings assay of 0.06% Cu. The recovery projection for full-scale plant operation will be re-evaluated in the Feasibility Study.

Cu Recovery

$$\%\ \text{Cu recovery} = 100.21 \times \left(1 - \frac{0.06}{F}\right)$$

where F = % Cu in feed

Au Recovery

% Au recovery = $15.76e^{0.0166x}$

where x = % Cu recovery

Ag Recovery

% Ag recovery = $6.433e^{0.0258x}$

where x = % Cu recovery

The equations apply to sulphidic ores (the vast majority of all ore types) but not to Cu oxide ores. As expected, Cu oxide samples have yielded lower recoveries under the same conditions used for floating the sulphidic ores. Hence, the locations of infrequent and highly localized oxide occurrences across the Galore Creek deposit will be identified and accounted for in Feasibility recovery estimates.

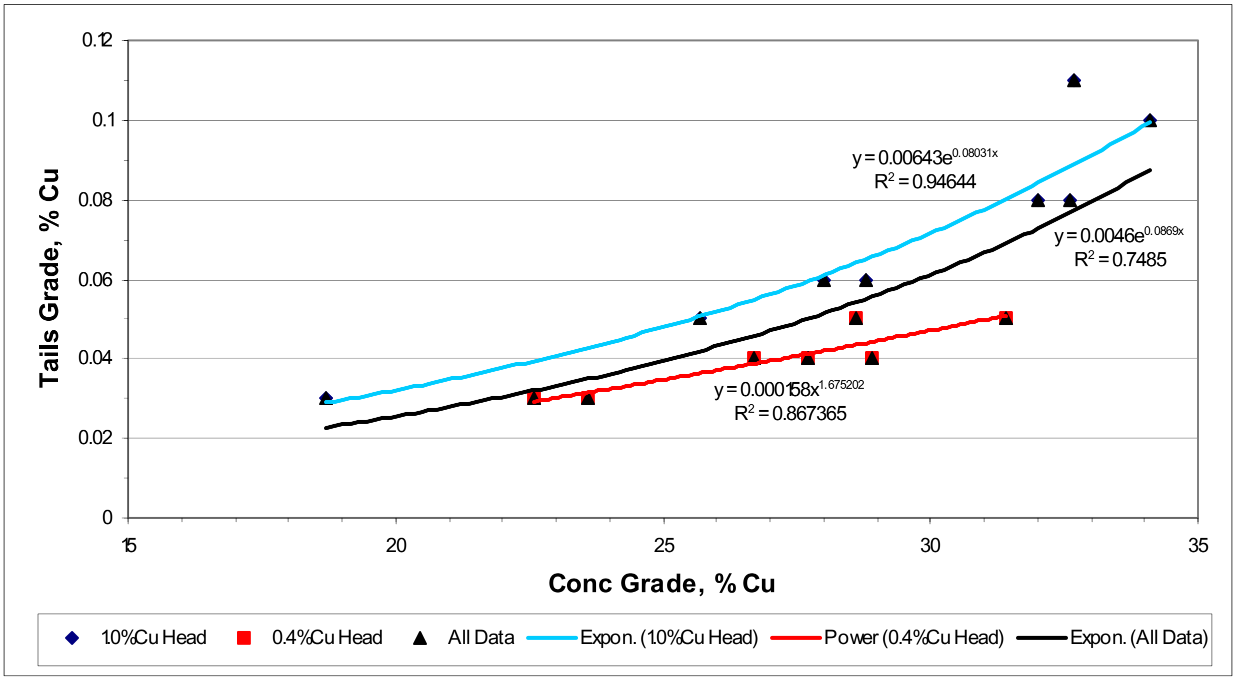
6.7 Data Analysis

NovaGold has identified chalcopyrite and chalcopyrite-bornite as the major ore types across the deposit. As such, high grade (1% Cu) and low grade (0.4% Cu) drill core composites of these ore types were selected for the majority of the locked cycle flotation tests. The results for these ore types have been used to develop the correlation for estimating recovery across the major ore zones in the deposit.

6.7.1 Copper Recovery Projection

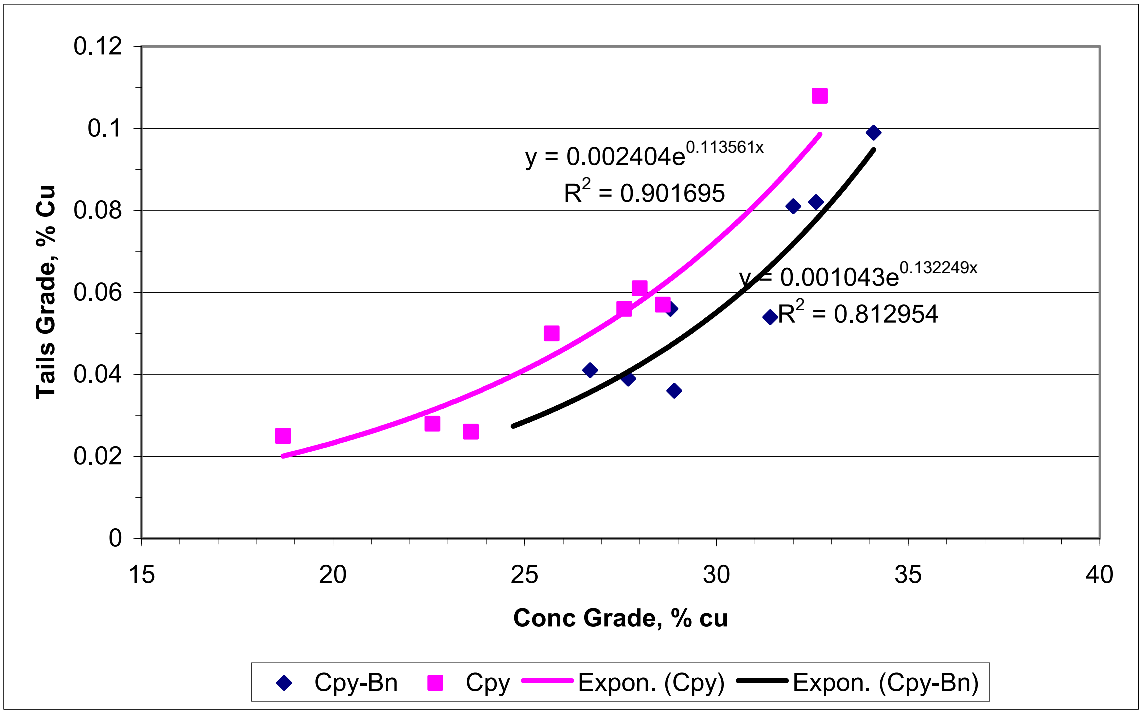
Test results have shown that rougher Cu recovery is high resulting in low Cu content in tailings. The Cu assay of the final tailings, which includes first cleaner scavenger tailings, would be impacted more by the losses in the cleaner scavenger tailings. The losses, in turn, would be impacted by the concentrate grade achieved. The analysis, shown in Figure 6-3, indicated good correlation between concentrate grade and tailings grade by head grades of combined chalcopyrite and chalcopyrite-bornite ores. Figure 6-4 also shows a reasonable correlation when all the data were combined. It would be reasonable to assume that this overall correlation represented the average ore grade of 0.7% Cu and could be used to estimate recovery for the overall deposit. Using this correlation, the projected final tailings grade would be 0.052% Cu at a 28% Cu concentrate grade.

Figure 6-3: Galore Creek Concentrate vs. Tails Grade *



As shown in Figure 6-4, good correlations between concentrate grade and tailings grade have also been obtained by ore types, for feed grades of 0.4% Cu and 1.0% Cu. For a 28% Cu concentrate, the analysis projected tailings grades of 0.058% Cu from chalcopyrite ore and 0.043% Cu from chalcopyrite-bornite ores. Since a good correlation is achieved irrespective of feed grades, it would be reasonable to project Cu recovery based on constant tailings assay for a given concentrate grade over a range of feed grades. With chalcopyrite being more dominant than chalcopyrite-bornite in the major ore zones, its correlation is used to project Cu recovery, based on a constant 0.058% Cu tailings assay at 28% Cu concentrate over the range of expected feed grade.

Figure 6-4: Galore Creek Concentrate vs. Tails Grade (By Ore Type)



Consequently, the equation for Cu recovery at 28% Cu concentrate grade has been derived according to the two-product formula as follows:

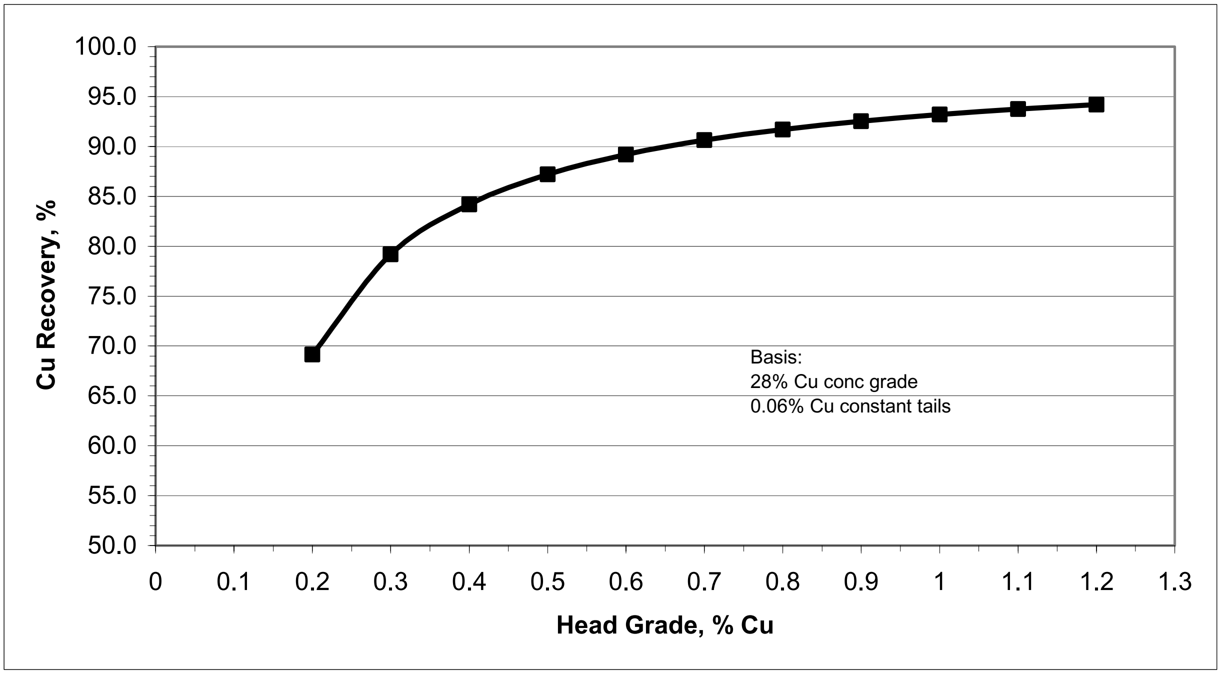
$$\% \text{ Cu recovery} = 100.21 \times \left(1 - \frac{0.06}{F}\right)$$

where F = % Cu in feed

The Cu recovery projection has been plotted in Figure 6-5.

As can be evidenced in Figure 6-4, Cpy-Bn ore types will generate higher concentrate grades, in the range of 30% Cu.

Figure 6-5: Galore Creek Cu Recovery Projection



6.7.2 Gold Recovery Projection

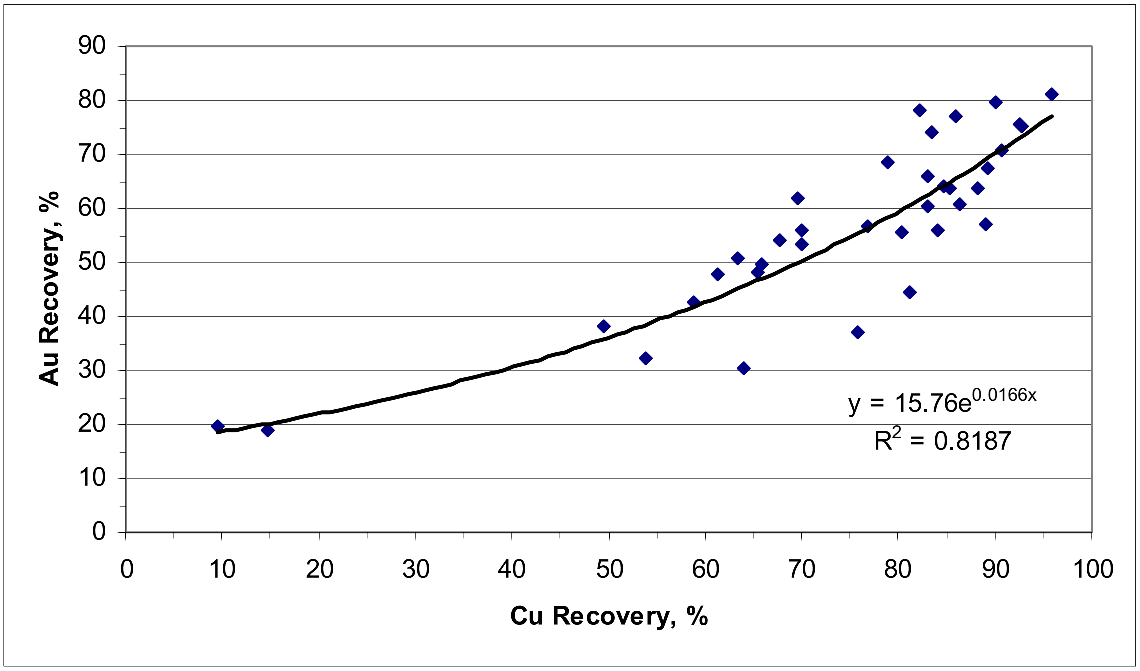
Mineralogical analysis during the test program has determined that Au occurred with Cu sulphides and pyrite, and that approximately 65% of Au followed Cu deportment to concentrate. In low Cu ores, the Au generally occurred within pyrite grains and was not recovered into a high grade copper concentrate. Accordingly, as shown in Figure 6-6, there is a good correlation between Au recovery and Cu recovery in concentrate.

Once the Cu recovery has been estimated, the Au recovery may be estimated based on the following correlation:

$$\%\ \text{Au recovery} = 15.76e^{0.0166x}$$

where x = % Cu recovery

Figure 6-6: Galore Creek Cu vs. Au Recovery (All Test Data)



6.7.3 Silver Recovery Projection

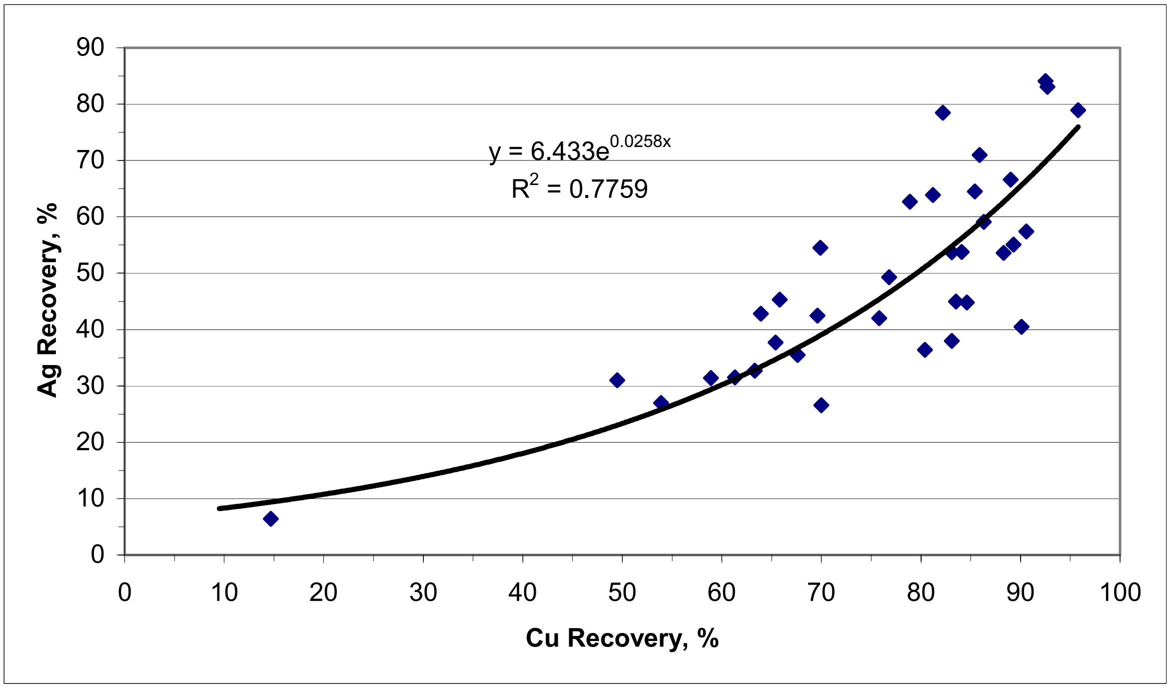
As shown in Figure 6-7, the test data has produced a good correlation between Ag recovery and Cu recovery.

Accordingly, the Ag recovery may be projected based on the following correlation:

% Ag recovery = $6.433e^{0.0258x}$

where x = % Cu recovery

Figure 6-7: Galore Creek Cu vs. Ag Recovery (All Test Data)



7. Processing

7.1 Summary

A conventional crushing, grinding and flotation process plant is proposed for the project, utilizing standard unit processes and equipment. The process design criteria developed for the PEA is based on a combination of testwork results on Galore Creek ores, experience from other similar operations and Industry standards. No significant technical complexities are foreseen in processing the ore. A number of trade-off studies were carried out to investigate process or equipment layout alternatives.

7.2 Process Description

7.2.1 General

The Galore Creek mill will process a blend of open pit ore from the Central Zone, Southwest Zone, Junction, and West Fork. Approximately 80% of the ore will be from the Central Zone. The same flowsheet can be used for all ore types.

The Galore Creek plant has been designed to process 65,000 tpd of ore containing copper, gold and silver to produce a high quality copper concentrate. The plant will consist of the following unit operations.

- Ore storage
- Primary crushing
- Two-stage grinding
- Copper flotation
- Rougher concentrate regrinding
- Concentrate thickening and pumping
- Concentrate filtration
- Concentrate storage
- Tailings handling
- Water reclaim

The plant will operate 24 hours per day, 365 days per year with scheduled downtime for equipment maintenance and will process 65,000 tonnes of ore per day.

7.2.2 Crushing

The crushing facilities will comprise of a rock breaker, a gyratory crusher, coarse ore bin, apron feeder, and collection conveyors.

Ore will be trucked from the open pit mining operation to the primary crushing facilities and dumped directly into the dump pocket. A rock breaker will be installed over the dump pocket to break large pieces of ore and avoid plugging the crusher. The ore will be crushed in the gyratory crusher to nominally 80% passing 150 mm and conveyed to a 600 t coarse ore surge bin. From the bin, the ore will be withdrawn with an apron feeder onto a coarse ore conveyor and conveyed to a stockpile having a live capacity of 40,000 t and total capacity of about 240,000 t. Dust collectors with pickups around the crusher, conveyors and ore transfer points will be installed to minimize fugitive dust in the facilities.

7.2.3 Grinding

The grinding circuit will consist of: one SAG mill with a trommel, vibrating screen, two ball mills with trommels, two cyclopacs, two pebble cone crushers and pebble recycle conveyors.

Ore will be reclaimed from the coarse ore stockpile via three apron feeders and conveyed to the two-stage grinding circuit. The apron feeders will be controlled by a weightometer on the SAG mill conveyor to achieve a nominal operating rate of 2632 tph. The ore will be ground to a final product size of 80% passing 170 µm (approximately) using a 11.58 m diameter by 6.10 m, 19,400 kW SAG mill and two 7.03 m diameter by 10.98 m, 11,190 kW ball mills operating in parallel.

The SAG mill will operate in closed circuit with a trommel and vibrating screen. The trommel oversize will feed the vibrating screen while the vibrating screen oversize will be recycled to SAG mill feed via conveyors and two pebble cone crushers operating in parallel. A magnet and metal detector will be installed over the pebble recycle conveyor to remove ball chips and other metallic objects and protect the cone crushers from damage. The trommel and vibrating screen undersize will be pumped and distributed to the two ball mill discharge sumps.

The two ball mills will operate independently, each in closed circuit with a cyclopac. The combined SAG mill and ball mill discharges will be pumped to the corresponding cyclopac. The cyclopac underflow will be recycled to the ball mill while the overflow will be directed to the rougher flotation circuit.

Sampling systems and particle size monitors will be installed on the cyclone overflows to control the grinding mills to achieve the target grind.

7.2.4 Flotation and Regrind

The flotation and regrind circuit will consist of the following:

- Two parallel banks of large diameter rougher flotation tank cells
- Single bank of each of the three-stage cleaner flotation cells
- Two regrind vertimills with a cyclopac

Each of the ball mill cyclone overflows will be fed to a conditioner tank and a bank of five 200 m^3 rougher flotation cells. The concentrate from the two banks will be combined and pumped to the regrind circuit while the tailings will be pumped to the tailings pond.

The rougher concentrate will be ground in two 1,119 kW vertimills operating in closed circuit with a cyclopac to achieve a product grind of 80% passing 45 µm (approximately). The cyclopac underflow will be split to the two vertimills while the overflow will feed the cleaner circuit. A sampling system and particle size monitor will be installed on the cyclone overflow to control the vertimill operation and achieve the target grind.

The cyclone overflow will be upgraded in a three-stage cleaner flotation circuit. The first cleaner will operate in open-circuit while the second and third cleaners will operate in closed circuit.

The first cleaner concentrate will be pumped to the second cleaner flotation cells while the tailings will flow into the first cleaner scavenger cells. The first cleaner scavenger concentrate will be reground and recycled to the first cleaner while the tailings will be pumped to the final tailings pump box then pumped to the tailings pond together with the rougher tailings. The first cleaner will have four 50 m^3 cells while the cleaner scavenger will have two 50 m^3 cells.

The second and third cleaner stages will be arranged as a single bank such that the third cleaner tailings will flow by gravity to the adjoining second cleaner cells. Each stage will have four 20 m^3 cells. The second cleaner concentrate will be pumped to the third cleaner and the third cleaner concentrate will be the final concentrate feeding the concentrate dewatering circuit. The second cleaner tailings will be pumped back to the first cleaner circuit.

The flotation reagents, lime, PAX and MIBC, will be added to the circuit to achieve the desired metals recovery and concentrate grade.

Sampling systems will be installed on key flotation streams to collect samples for process monitoring and control, and for metallurgical accounting purposes.

7.2.5 Concentrate Dewatering

The concentrate dewatering circuit will incorporate thickening and filtration and will consist of the following equipment: concentrate thickener, filter feed tank, two pressure filters and concentrate conveyor and storage.

Two concentrate stock tanks will be installed at the mine site to provide additional surge capacity for the concentrate pumping system. Concentrate slurry will be pumped from the two concentrate stock tanks at mine site via a concentrate pipeline and an intermediate pumping station to a storage tank in the filtration plant located near BC Highway 37 (Bob Quinn). The concentrate production rate will vary due to variable ore grades and dictates that the slurry will be batch-pumped in order to maintain the required line velocity. The pipeline will be flushed with water in between batches of concentrate to avoid plugging the pipe.

All water received at the Highway 37 filter plant, either as transport water with the concentrate slurry or as pipeline flush, will be treated and discharged. Consequently, the filtration plant filter will include water treatment facilities to ensure that the discharge water is clean and will readily meet discharge criteria. The plant will consist of the following: concentrate stock tank, two pressure filters, filtrate thickener, reactor clarifier, two sand filters and flocculant and lime systems. The filter plant will also house the equipment for the necessary utilities such as power, compressed air, fire water, heating.

Concentrate slurry will be received in an agitated stock tank, then filtered through the two pressure filters. The filter cake will be stored in a shed until collected for transportation by truck to the tidewater port of Stewart, BC. The combined filtrate, manifold flush and cloth wash will be pumped from the filtrate tank to the filtrate thickener. Flocculant will be added at the thickener to settle the solids. The thickener underflow will be recycled to the concentrate stock tank.

The thickener overflow will be treated with lime addition in a reactor clarifier to precipitate minor amounts of dissolved metals. The reactor clarifier underflow will be purged, as required, to the concentrate stock tank while the overflow will be filtered through sand filters to remove traces of suspended solids and ensure a compliant discharge to the environment. The treated effluent will be used for filter manifold flush, filter cloth wash, reagent make-up and gland seals. Clean water will be discharged to the environment in accordance with permit conditions.

When flush water is pumped down the pipeline, the water will bypass the stock tank into the filtrate thickener. The thickener overflow will then be treated and filtered through the sand filters and clean water will be discharged to the environment.

7.2.6 Tailings and Water Handling

The final combined tailings from rougher flotation and first cleaner scavenger flotation will be pumped to the tailings pond. Following solids settling and consolidation, the pond water, comprising a combination of site drainage and tailings water, will be discharged. Ongoing monitoring defined by regulatory and permit conditions will confirm that this discharge meets or exceeds design criteria. Concentrate thickener overflow will be recycled within the plant as process water via the process water tank. Water for all process requirements will be obtained from the East Fork water dam.

7.3 Process Design Criteria

A standard flowsheet comprising single stage crushing, two-stage SAG-ball mill grinding and three-stage flotation with regrinding will be used for the Galore Creek plant. Single-stage crushing with SAG milling will be suitable for high tonnage operation for this ore and location. A SABC grinding circuit with pebble crushers has been selected in view of the relatively hard ore and the possibility of pebble generation. Membrane pressure filters will be used to achieve minimum concentrate cake moisture.

The process design criteria has been tabulated in Table 7-1, indicating the source of the data.

Table 7-1: Process Design Criteria

PROCESS DESIGN CRITERIA			SOURCE		
			1 - CLIENT		
PROJECT	: GALORE CREEK PRE-FEASIBILITY STUDY		2 - VENDOR		
PROJECT NO	: 317882		3 - EXPERIENCE		
CLIENT	: NOVAGOLD		4 - CALCULATION		
DATE	: 26 AUGUST, 2005		5 - MASS BALANCE		
REV	: A		6 - TESTWORK		
			Nominal Prod.	**Max Prod.**	SOURCE
A	**GENERAL**				
	DAILY THROUGHPUT, NOMINAL	tpd	**65,000**	**65,000**	1
	HOURLY THROUGHPUT	tph	2,851	2,851	5
	OVERALL PLANT AVAILABILITY, NOMINAL	%	**95**	**95**	1
	ORE SOLIDS DENSITY	t/m^3	**2.70**	**2.70**	1
	ORE MOISTURE	%	**5.00**	**5.00**	1

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		1 - CLIENT
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REV	: A	6 - TESTWORK

			Nominal Prod.	**Max Prod.**	SOURCE
	ORE COPPER GRADE, AVERAGE	%	**0.70**	**0.95**	1
	ORE GOLD GRADE, AVERAGE	g/t	**0.47**	**0.70**	1
	ORE SILVER GRADE, AVERAGE	g/t	**5.26**	**6.86**	1
	CONCENTRATE MASS RECOVERY	%	2.3	3.0	6
	CONCENTRATE PRODUCTION	dry tph	65.5	84.9	5
	CONCENTRATE PRODUCTION	dry tpd	1493	1936	5
	CONCENTRATE MOISTURE	%	**8.0**	**8.0**	3
	CONCENTRATE COPPER GRADE	%	**28.0**	**30.0**	1
	CONCENTRATE GOLD GRADE	g/t	14.6	17.4	4
	CONCENTRATE SILVER GRADE	g/t	154	163	4
	TAILINGS COPPER GRADE	%	**0.06**	**0.06**	6
	COPPER RECOVERY	%	91	93	6
	GOLD RECOVERY	%	71	74	6
	SILVER RECOVERY	%	67	71	6
B	**CRUSHING**				
	OVERALL CRUSHING PLANT AVAILABILITY	%	**75**	**75**	3
	CRUSHER OPERATING SCHEDULE	hrs/day	**24**	**24**	1
	PRIMARY CRUSHING				
	CRUSHER TYPE		Gyratory	Gyratory	2
	HOURLY THROUGHPUT	tph	3611	3611	5
	DISCHARGE P80, NOMINAL	mm	**150**	**150**	3
	COARSE ORE STOCKPILE CAPACITY, LIVE	t	**40,000**	**40,000**	3
	MOTOR SIZE	kW	**600**	**600**	2
C	**MILLING**				
	OVERALL PLANT AVAILABILITY	%	**95**	**95**	1
	OVERALL HOURLY THROUGHPUT	tph	2,851	2,851	5
	FEED SIZE, F80	mm	**150**	**150**	3
	PRODUCT SIZE, P80	microns	**170**	**170**	6
	BALL MILL BOND WORK INDEX	metric	**16.0**	**16.0**	6

PROCESS DESIGN CRITERIA		SOURCE
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			Nominal Prod.	**Max Prod.**	SOURCE
	SAG MILL (PRIMARY)				
	NUMBER OF MILLS		**1**	**1**	4
	SAG MILL DIAMETER	m	**11.59**	**11.59**	4
	SAG MILL LENGTH	m	**6.10**	**6.10**	4
	SAG MILL MOTOR	kW	**19,500**	**19,500**	4
	SAG MILL FEED RATE	tph	2851	2851	5
	TRANSFER SIZE, K80	microns	**1200**	**1200**	4
	PEBBLE CRUSHER TYPE		Cone	Cone	3
	NUMBER OF PEBBLE CRUSHERS		**2**	**2**	5
	PEBBLE CRUSHER MOTOR SIZE	kW	**298**	**298**	2
	PEBBLE RECYCLE RATE	%	**20**	**20**	3
	BALL MILL (SECONDARY)				
	NUMBER OF BALL MILLS		**2**	**2**	4
	BALL MILL DIAMETER	m	**7.93**	**7.93**	4
	BALL MILL LENGTH	m	**10.98**	**10.98**	4
	BALL MILL MOTOR	kW	**11,190**	**11,190**	4
	BALL SIZE	mm	**75**	**75**	4
	CIRCULATING LOAD	%	**300**	**300**	3
	BALL MILL CYCLONES				
	NUMBER OF CYCLOPAC		**2**	**2**	4
	NUMBER OF CYCLONES/CYCLOPAC		**12**	**12**	2
	NUMBER OF CYCLONES OPERATING		**10**	**10**	2
	CYCLONE SIZE	mm	**840**	**840**	2
	CYCLONE PRESSURE DROP	kPa	**62**	**62**	2
	CYCLONE OVERFLOW, P80	microns	**170**	**170**	6
	CYCLONE FEED DENSITY	% solids	**50.0**	**50.0**	5
	CYCLONE UNDERFLOW DENSITY	% solids	**70.0**	**70.0**	5
	CYCLONE OVERFLOW DENSITY	% solids	**26.9**	**26.9**	5
D	**FLOTATION**				
	ROUGHER CONDITIONER				
	NUMBER OF TANKS		**2**	**2**	

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DATE	: 26 AUGUST, 2005	5 - MASS BALANCE
REV	: A	6 - TESTWORK

		Nominal Prod.	Max Prod.	SOURCE
CONDITIONER FEED PER TANK	m^3/h	**4397**	**4397**	5
TANK VOLUME	m^3	**200**	**200**	1
RESIDENCE TIME	min	2.7	2.7	6
ROUGHER FLOTATION				
CELL TYPE		Tank cell	Tank cell	2
NUMBER OF BANKS		**2**	**2**	2
NUMBER OF CELLS PER BANK		**5**	**5**	2
CELL VOLUME	m^3	**200**	**200**	2
FEED FLOWRATE PER BANK	m^3/h	4397	4397	5
FEED SLURRY DENSITY	% solids	**26.9**	**26.9**	5
RESIDENCE TIME PER BANK	min	13.6	13.6	6, 4
BATCH RESIDENCE TIME	min	**6**	**6**	6
CONCENTRATE MASS PULL	%	**8.0**	**10.0**	6
CONCENTRATE MASS PULL	tph	228	285	5
REGRIND MILL				
TYPE OF MILL		Vertimill	Vertimill	
NUMBER OF MILLS		**2**	**2**	4
MILL MOTOR	kW	**1,119**	**1,119**	4
BALL SIZE	mm	**25**	**25**	4
CIRCULATING LOAD	%	**250**	**250**	3
REGRIND CYCLONES				
NUMBER OF CYCLOPAC		**1**	**1**	4
NUMBER OF CYCLONES/CYCLOPAC		**4**	**4**	2
NUMBER OF CYCLONES OPERATING		**3**	**3**	2
CYCLONE SIZE	mm	**500**	**500**	2
CYCLONE PRESSURE DROP	kPa	**117**	**117**	2
CYCLONE OVERFLOW, P80	microns	**45**	**45**	6
CYCLONE FEED DENSITY	% solids	**45.0**	**45.0**	5
CYCLONE UNDERFLOW DENSITY	% solids	**70.0**	**70.0**	5
CYCLONE OVERFLOW DENSITY	% solids	**23.8**	**23.8**	5

PROCESS DESIGN CRITERIA		SOURCE
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CLIENT	: NOVAGOLD	4 - CALCULATION
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REV	: A	6 - TESTWORK

		Nominal Prod.	**Max Prod.**	SOURCE
CLEANER CONDITIONER				
CONDITIONER FEED	m^3/h	**796**	**919**	5
RESIDENCE TIME	min	5.4	4.7	6, 4
TANK DIAMETER	m	**4.5**	**4.5**	4
TANK HEIGHT	m	**5.0**	**5.0**	4
FIRST CLEANER FLOTATION				
CELL TYPE		Tank cell	Tank cell	2
NUMBER OF BANKS		**1**	**1**	2
NUMBER OF CELLS PER BANK		**4**	**4**	2
CELL VOLUME	m^3	**50**	**50**	2
FEED FLOWRATE, TOTAL	m^3/h	**1173**	**1363**	5
FEED SLURRY DENSITY	% solids	**21.4**	**21.1**	5
RESIDENCE TIME, TOTAL	min	10.2	8.8	6, 4
BATCH RESIDENCE TIME	min	**4**	**4**	6
CONCENTRATE MASS PULL	%	**3.8**	**6**	6
CONCENTRATE MASS PULL	tph	108	157	5
FIRST CLEANER SCAVENGER FLOTATION				
CELL TYPE		Tank cell	Tank cell	2
NUMBER OF BANKS		**1**	**1**	2
NUMBER OF CELLS PER BANK		**2**	**2**	2
CELL VOLUME	m^3	**50**	**50**	2
FEED FLOWRATE	m^3/h	**984**	**1134**	5
FEED SLURRY DENSITY	% solids	**16.9**	**16.3**	5
RESIDENCE TIME, TOTAL	min	6.1	5.3	6, 4
BATCH RESIDENCE TIME	min	**2**	**2**	6
CONCENTRATE MASS PULL	%	**0.8**	**1.0**	6
CONCENTRATE MASS PULL	tph	23	29	5
SECOND CLEANER FLOTATION				
CELL TYPE		Tank cell	Tank cell	2
NUMBER OF BANKS		**1**	**1**	2
NUMBER OF CELLS PER BANK		**4**	**4**	2
CELL VOLUME	m^3	**20**	**20**	2

PROCESS DESIGN CRITERIA		SOURCE
		1 - CLIENT
PROJECT	: GALORE CREEK PRE-FEASIBILITY STUDY	2 - VENDOR
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			Nominal Prod.	Max Prod.	SOURCE
	FEED FLOWRATE	m^3/h	**447**	**538**	5
	FEED SLURRY DENSITY	% solids	**24.0**	**23.7**	5
	RESIDENCE TIME, TOTAL	min	10.7	8.9	6, 4
	BATCH RESIDENCE TIME	min	**3**	**3**	6
	CONCENTRATE MASS PULL	%	**3.0**	**4.5**	6
	CONCENTRATE MASS PULL	tph	86	128	5
	THIRD CLEANER FLOTATION				
	CELL TYPE		Tank cell	Tank cell	2
	NUMBER OF BANKS		**1**	**1**	2
	NUMBER OF CELLS PER BANK		**4**	**4**	2
	CELL VOLUME	m^3	**20**	**20**	2
	FEED FLOWRATE	m^3/h	**277**	**341**	5
	FEED SLURRY DENSITY	% solids	**25.0**	**25.0**	5
	RESIDENCE TIME, TOTAL	min	17	14	6, 4
	BATCH RESIDENCE TIME	min	**2**	**2**	6
	CONCENTRATE MASS PULL	%	2.3	**3.8**	6
	CONCENTRATE MASS PULL	tph	66	108	5
E	**CONCENTRATE DEWATERING**				
	THICKENER TYPE		High Rate	High Rate	3
	NUMBER OF THICKENERS		1	1	4
	FLOCCULANT ADDITION	g/t	5	5	6
	THICKENER UNIT AREA, TEST	m^2/tpd	0.043	0.043	6
	THICKENER UNIT AREA, DESIGN	m^2/tpd	0.125	0.125	est., 3
	THICKENER AREA, DESIGN	m^2	187	309	4
	THICKENER DIAMETER, CALCULATED	m	15	20	4
	THICKENER DIAMETER, INSTALLED	m	17	17	3
	THICKENER UNDERFLOW DENSITY	% solids	60	60	5
F	**CONCENTRATE FILTRATION**				
	CONCENTRATE SLURRY FLOWRATE	m^3/h	**62.7**	**81.4**	5
	CONCENTRATE PRODUCTION	dry tph	65.5	84.9	5
	STOCK TANK CAPACITY	hr	12.5	9.6	4
	STOCK TANK DIAMETER	m	**10.0**	**10.0**	4

PROCESS DESIGN CRITERIA		SOURCE
		1 - CLIENT
PROJECT	: GALORE CREEK PRE-FEASIBILITY STUDY	2 - VENDOR
PROJECT NO	: 317882	3 - EXPERIENCE
CLIENT	: NOVAGOLD	4 - CALCULATION
DATE	: 26 AUGUST, 2005	5 - MASS BALANCE
REV	: A	6 - TESTWORK

			Nominal Prod.	Max Prod.	SOURCE
	STOCK TANK HEIGHT	m	**10.0**	**10.0**	4
	FILTER TYPE		**Larox**	**Larox**	2
	NUMBER OF FILTERS		**2**	**2**	2
	FILTER FEED DENSITY	% solids	**56**	**56**	5
	FILTRATION RATE	kg/m^2-h	**580**	**580**	est, 2
	FILTER CYCLE TIME	min	**9**	**9**	2
	FILTER AREA REQUIRED	m^2	113	146	4
	FILTER AREA, PER FILTER	m^2	**72**	**72**	4
	FILTER CAKE MOISTURE	%	**8**	**10**	2, 3
G	**REAGENT SYSTEMS**				
	PAX				
	ADDITION TO ROUGHER	g/t	9	9	6
	ADDITION TO CLEANERS	g/t	0	0	6
	ADDITION TO CLEANER-SCAVENGER	g/t	5	5	6
	SOLUTION CONCENTRATION	%	10	10	3
	MIX TANK CAPACITY	day	7	7	4
	MIX TANK DIAMETER	m	4.5	4.5	4
	MIX TANK HEIGHT	m	5.0	5.0	4
	DAY TANK DIAMETER	m	2.5	2.5	4
	DAY TANK HEIGHT	m	3.0	3.0	4
	MIBC				
	ADDITION TO ROUGHER	g/t	16	16	6
	ADDITION TO CLEANERS	g/t	22	22	6
	ADDITION TO CLEANER-SCAVENGER	g/t	0	0	6
	SOLUTION CONCENTRATION	%	100	100	3
	HOLDING TANK CAPACITY	day	28	28	4
	HOLDING TANK DIAMETER	m	4.5	4.5	4
	HOLDING TANK HEIGHT	m	5.0	5.0	4
	DAY TANK DIAMETER	m	2.5	2.5	4
	DAY TANK HEIGHT	m	3.0	3.0	4

PROCESS DESIGN CRITERIA		SOURCE
		1 - CLIENT
PROJECT	: GALORE CREEK PRE-FEASIBILITY STUDY	2 - VENDOR
PROJECT NO	: 317882	3 - EXPERIENCE
CLIENT	: NOVAGOLD	4 - CALCULATION
DATE	: 26 AUGUST, 2005	5 - MASS BALANCE
REV	: A	6 - TESTWORK

		Nominal Prod.	**Max Prod.**	SOURCE
LIME				
ADDITION TO GRINDING	g/t	300	300	6
ADDITION TO REGRINDING	g/t	100	100	6
ADDITION TO CLEANERS	g/t	140	140	6
SLURRY CONCENTRATION	% solids	20	20	3
HOLDING TANK CAPACITY	day	1	1	4
HOLDING TANK DIAMETER	m	6.0	6.0	4
HOLDING TANK HEIGHT	m	6.0	6.0	4

7.3.1 Site Layout Considerations

The area available for the plant and support infrastructure is fairly constrained to a 500 m x 500 m area immediately south of the confluence of the East and West Forks of Galore Creek by:

- the open pits to the west and north.
- prospective ore zone to the north of the control pit.
- waste rock and tailings dumps to the north.
- East Fork dam to the east.
- steep topography to the south.
- avalanche hazards to the south and west.

Limited geotechnical investigations have been carried out in the area of the plantsite during the PEA, to delineate the depth to bedrock. Geotechnical drilling underway as part of the 2005 field program will confirm the attributes of this location prior to the completion of the Feasibility report next year. With this new information, the plantsite and concentrator layout will be optimized in relation to the topographic and foundation conditions, but due to the above-mentioned constraints, it is unlikely that the plantsite will be relocated.

8. Tailings and Waste Rock Storage and Water Management Plan

8.1 Summary

Bedrock at the Galore Valley tailings/waste rock storage site is generally competent. However, sub-horizontal sheet fractures have been observed in the upper 150 to 200 m in the Galore Creek bedrock. The fracture spacing is in the order of a few millimetres to a few centimetres. In general, these well-developed closely spaced fractures are believed to be the result of a combination of stress relief from glacier retreat and the presence of anhydrite/gypsum, which was introduced during ore emplacement. The presence of such fractures is not uncommon, however, and practically, any risk they may appear to introduce can be managed if required by grouting and other established construction practices.

The area has been glaciated during the Pleistocene era. The ice has left deposits of till overlying bedrock. The till and other surficial deposits in this area are complex and result from fluvial erosion and deposition, and glaciolacustrine deposition. The upper slopes are gullied bedrock partially overlain with a thin veneer of colluvium. The site is located in a moderately high seismic zone.

The Galore Creek project site will experience a high annual precipitation of approximately 1650 – 2300 mm throughout its anticipated mine life. Rain and snow, as well as water-discharged from nearby icefields and glaciers, have been incorporated into the design of the proposed mine. The catchment area around Galore Creek is large (120 km^2) which means that large volumes of freshwater require diversion to facilitate mining and minimize environmental impacts and these diversions have been incorporated into the site designs and cost estimates for tailings and waste dump(s) at a pre-feasibility level in this PEA.

Water quality in the tailings and waste dump facilities is currently being assessed. At this time, it is predicted based on pre-feasibility level testwork that the water quality will be suitable for discharge, but this will be confirmed by further investigation now underway. Given the high rainfall, large catchments and the assumption that water which comes in contact with mine facilities may require some form of treatment, surface water diversion works have been designed to limit catchment areas and direct clean non-contact water away from the facilities. The diversions will include the east fork of Galore Creek, the eastern portion of the catchment above the tailings/waste rock impoundment, and areas of undisturbed natural ground above the Central and Southwest pits and the plant site.

To support a safe and environmentally sound long term closure plan, the primary tailings and waste facility in Galore Creek will be submerged to form a lake approximately 8 km long and 2 km wide upon closure. This area will include the following:

- three water retaining dams (Main Dam, West Dam and East Dam);
- seepage recovery dam and/or recovery wells;
- large freshwater pond; and,
- open channel diversions (including an aqueduct access to the waste dump).

The intent will be to leave all the PAG waste in a flooded condition for perpetuity. The tailings and waste rock will be primarily contained behind the Main Dam.

The majority of the waste rock from the open pit will be trucked and dumped into the facility using the shortest possible haul. However, a portion of the waste rock will have to be hauled to the north end to be used as construction material for the Main dam. The dams will be built of non-acid generating rock. Any potentially acid-generating (PAG) rock kept in the interior of the waste dump will be flooded sequentially throughout the mine life. Approximately two thirds of the waste rock is assumed to be PAG in the pre-feasibility level design although this estimate is believed to be conservative based on ongoing testwork.

A large natural catchment area exists around the tailings impoundment. In order to minimize the volume of water, which enters the PAG rock and tailings storage facility, open channel diversions are proposed on the eastern slope above the impoundment and around the proposed open pits to divert freshwater. In order to pond water and divert it into the diversion ditch, an impervious dam, referred to as the East Dam, has been designed and included in cost estimates. The dam will be impervious to prevent freshwater from seeping into the impoundment As a result of the East Dam, a large freshwater pond will be created in the East Fork of Galore.

To prevent water flowing toward the open pit areas, a diversion structure will be required in the West Fork of Galore Creek. The elevation of this structure will be controlled by the water elevation within the impoundment. All of the water dams and structures, the Main, East and West will be built sequentially.

The storage capacity and design criteria for the tailings and PAG waste rock facility has included an allowance for storing extreme precipitation event(s). At the pre-feasibility level, the impoundment has been designed to store precipitation from the 200-year flood.

The Main Dam is proposed to be a rockfill dam with an impervious till core. The upstream shell will be constructed of compacted PAG and/or NPAG waste rock and the downstream shell of compacted NPAG waste rock. The rockfill will provide stability and resistance to earthquake forces and the compacted till core will provide an impermeable barrier to retain water. The transition from the fine grained till core to the downstream rockfill shell will be through two filter zones of processed material. Four metre thick filter zones have been allowed for to account for potential deformation during earthquake events.

Pre-feasibility level investigations indicate there are sources of borrow for dam construction within the Galore Creek Valley. Glacial deposits are widespread throughout the valley and in some locations are up to 80 m thick on the lower western slopes. Till borrow will be collected upstream, downstream and within the footprint of the tailings dam for the impervious core. Material for the sand and gravel filter zones and blanket drain, will likely come from screened glacial deposits from the valley slopes. No attempt has been made, at this time, to optimize the use of till from the pit prestripping.

Preliminary seepage analyses for the proposed tailings embankments suggest seepage rates of between 150 to 300 l/s at Galore Creek. Seepage recovery systems, possibly in the form of groundwater recovery wells have been allowed for in cost estimates.

For Galore Creek, one 8 km long, 20 m wide (at the base), 5 m deep open diversion channel is proposed along the eastern slope above the tailings/waste dump. It has been assumed that the channel foundation conditions (soil or rock) are of low permeability to prevent excessive seepage out of the channel. If the channel is cut in pervious soil or highly fractured rock, a liner will be required. Construction and operating parameters identified and considered in all design, capital and operating cost estimates for the proposed open diversion channel include measures to mitigate or compensate for steep terrain, climate, geohazards and emergency flooding.

Pre-feasibility level cost estimates have been developed for earthworks to manage the tailings, waste rock and mine site drainage within the Galore Creek Valley.

The Main Starter Dam has been sized to contain the first two years of tailings production, a probable maximum flood (PMF) and 3 m of freeboard. The ultimate dam height is sized to contain all the tailings, a PMF, 3 m of freeboard, and also flood all PAG waste rock at mine closure. The required maximum height of the Main Tailings Dam is 265 m.

No costs for closure have been estimated at this time, pending the completion of testwork underway to confirm water quality models.

No detailed work has been undertaken for plant site design at this time, although a preferred plant site area has been identified in Galore Creek. The plant site is located above and east of the Central Pit area. This area is underlain by shallow metavolcanic bedrock. No detailed geotechnical assessment has been specifically undertaken in this area as part of the PEA, although 2004 exploration drilling suggested that the rock will likely be fractured to depths of up to 100 m. Free digging of the rock will likely be possible for excavating much of the site, although some blasting should be anticipated. A fault zone is known to exist in the valley in along the pit wall. The influence if this fault on the plant site area is not known at this time. Field programs being completed during the summer of 2005 will provide feasibility level information in this regard.

8.2 General Site Conditions

8.2.1 Location and Terrain

The Galore Creek Valley is a U-shaped glacially scoured valley with thick glacial and glaciolacustrine deposits covering the lower elevation slopes. The material has been reworked by fluvial action and then overridden in places by colluvium. The surrounding terrain is mountainous and covered by glaciers and icefields. Glaciers exist in the East and West forks of Galore Creek, but are currently retreating. The steep upper slopes are generally exposed bedrock.

8.2.2 Permafrost

Within the Galore Creek project area, permafrost is present only in an active rock glacier on an east-facing slope due west of the existing mine camp and at the head of Dendritic Creek. The toe of this rock glacier is at an elevation of 1320 m and its head at approximately 1500 m with a steep, light-coloured actively ravelling toe slope and well defined unvegetated compression ridges across its surface indicating recent activity. Some stagnant ice may also exist below the large lateral moraines on the north side of the Galore Creek East Fork valley. To determine if permafrost is present in the areas of any proposed structures, BGC installed one thermistor in the East Fork of Galore Creek at an elevation of 705 m. The ground temperatures were measured in early September and found to be above freezing.

None of the currently proposed mine-related facilities at either the Galore Creek are likely to encounter permafrost according to the vegetation criterion or the thermistor strings. For pre-feasibility design, therefore, it has been assumed that permafrost will not exist within the foundations of the plant site, tailings dams, and waste dumps. Based on open pit outlines of the Central and Southwest open pits, the likelihood of encountering permafrost in the pit walls would also be low.

8.2.3 Bedrock Geology

8.2.3.1 General

The study area lies within the Intermontane Belt and is underlain by Paleozoic rocks of the Stikine Assemblage and Mesozoic rocks of the Stuhini Group. These rocks are typically well stratified sedimentary, volcanic and plutonic rocks. The tectonic setting is that of a volcanic island arc that has been intruded by younger Mesozoic intrusions such as the Galore Creek intrusions. Structurally, the area is dominated by brittle deformation and faulting. Some ductile deformation exists, but it is limited to discrete reverse and thrust fault zones. The dominant structures are two approximately orthogonal folds, trending to the west and to the north. There is a northeast striking penetrative foliation that has been deformed by the west trending folds. In addition, there are reverse faults and northwest trending kink folds.

8.2.4 Surficial Geology

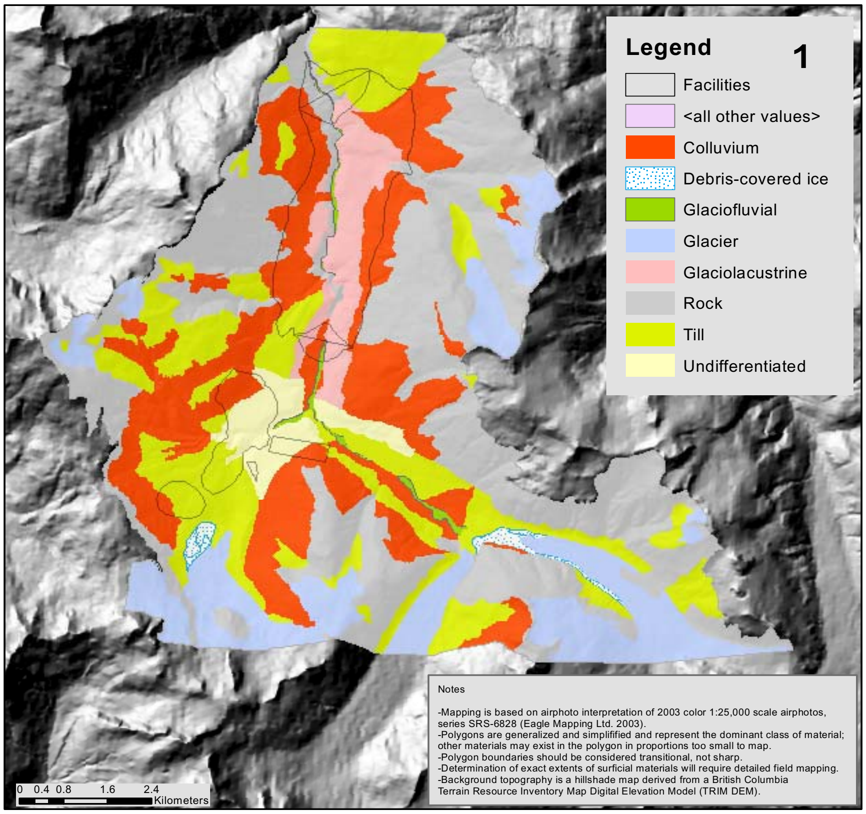
8.2.4.1 Galore Surficial Deposits

The Galore Creek Valley was glaciated during the Pleistocene era, followed by periods of glacial advance and interglacial fluvial erosion.

During the maximum extent of glaciation, glaciers covered the entire mountain range with only the highest rock peaks extending above the glacier surface. At one time, glaciers extended down the Galore Creek Valley to the Scud River confluence, scouring a U-shaped valley profile that has subsequently been downcut in the lower basin by Galore Creek. During more recent ice advances, glaciers extended approximately 1.5 km downvalley of the present glacier limits, scouring previous deposits and depositing new morainal sediments in the upper valley area. Fluvial erosion has since incised the deposits downstream of the existing glacial limits.

Figure 8-1 is simplified surficial geology map for upper and mid Galore Creek with approximate outlines of the proposed open pits and ultimate tailings/waste rock storage area (500 Mt tailings + 1000 Mt PAG waste rock) included for reference.

Figure 8-1: Simplified Surficial Geology Map of Galore Creek



In the Upper Galore basin, the open pit complex, consisting of exposed bedrock and colluvium, spans Dendritic Creek and the West Fork of Galore. On the upper steeper slopes south of Dendritic Creek the proposed open pit area consists of exposed fractured bedrock overlain in places by colluvium and morainal soils ranging from 1 to 10 m thick. In flatter areas, glacial till (up to 40 m thick) overlies bedrock. North of Dendritic Creek, surficial deposits are complex with glacial till interbedded with glaciolacustrine clays and silts up to 130 m thick.

In the East Fork coarse bouldery debris (rock avalanche deposit) overlies till interbedded with glaciolacustrine clays and silts. Overburden is up to 40 m thick on these lower slopes. The upper slopes are gullied bedrock partially overlain with a thin veneer (less than one metre thick) of colluvium. In addition, two debris-flow prone watersheds draining into the area from the north and south have created bouldery debris flow fans at their outlets.

In mid-Galore Valley, where a tailings/waste facility is proposed, the narrow valley bottom is covered with fluvial sand and gravel deposits ranging from 4-8 m thick. Higher on the slopes but below 600 m the overburden is typically complex gravely tills interbedded with finely bedded (or stratified) glaciolacustrine clay, silt, and fine sand deposits. The glacial lake deposits are interpreted to have formed due to ice damming of lower Galore Creek in the Late Pleistocene. The mid slope soils are up to 80 m thick, based on drill hole information provided by NovaGold and interpreted seismic refraction surveys conducted in summer 2004. The mid to upper slopes may also have a surficial blanket (greater than 10 m) of gravely, bouldery colluvium deposited during rockfall, debris flow and snow avalanche events. Lastly, the steep upper hillslopes are either bare bedrock or have a veneer (less than one metre thick) of colluvium and/or till.

North of the proposed tailings/waste storage area, Galore Creek incises deeply into Stikine Assemblage limestone. Thin colluvium partially mantles steep rockslopes above the creek, and treed slopes above the canyon are covered in veneers (less than one metre) of colluvium and till.

8.3 Pre-feasibility 2004 Field Investigations

8.3.1 Galore Creek Geotechnical Investigations

8.3.1.1 Geotechnical Drilling

Nine geotechnical boreholes were drilled within the Galore Creek Valley: five holes within the footprint of the proposed waste dump/tailings impoundment; three holes in the East Fork of Galore; and one hole in the proposed central pit. The boreholes varied in depth from 15 to 47 m.

8.3.1.2 Geologic Mapping

Outcrop mapping was conducted in the central pit area. At each outcrop, the rock type was noted and structural measurements were taken.

8.3.1.3 Permeability Testing

Thirteen standpipe piezometers were installed in the 2004 Galore drill holes (six in soil and seven in rock) to assess the permeability of the sub-surface ground conditions and determine groundwater surface elevations. Rising head or falling head tests were conducted in the standpipe piezometers with the exception of two where strongly artesian pressures observed in these holes prevented conducting any reasonable permeability testing.

8.3.2 Groundwater Characterization

Groundwater monitoring within Galore Creek Valley is based on thirteen piezometers in nine drill holes throughout the Galore Creek Valley, installed by BGC during 2004, a number of holes drilled by NovaGold and subsequently converted to ground water monitoring wells and, qualitative data from the drilling of holes GC76-0353 and CC90.

8.3.2.1 Geophysical Surveying

Between October 6^{th} and 14^{th}, 2004 Frontier carried out a seismic refraction investigation around the Galore Creek Valley. The primary purpose of these surveys was to determine the depth to bedrock. Six separate seismic traverses, a total of approximately 6.1 km, were completed within the Galore Creek Valley.

8.4 Site Hydrology

8.4.1 Site Area Streamflow Data

The hydrological regime of the watersheds in the Galore Creek area is strongly influenced by the surrounding glaciers. For large watersheds the hydrological year can be divided into four main flow periods:

1. Winter: low to negligible streamflows.
2. Spring Freshet: high flows due to snow melt; this period typically contains the annual peak flow.
3. Summer: moderate to high flows due to glacial melt water.
4. Fall: moderate to low flows with some rain fed storm events.

Preliminary streamflows reported for Galore Creek (Station GAL-1A, 24.4 km^2 catchment area) indicated a total catchment area runoff depth of about 2100 mm over the May to October 2004 period. This suggests a total annual precipitation over the period from fall 2003 to fall 2004 of approximately 2400 - 2500 mm, depending on actual evaporation and evapotranspiration losses.

8.5 Waste Storage Options

8.5.1 Introduction

The presently proposed project considers 500 Mt of milled tailings and 1,000 Mt of PAG waste rock to be generated over the 20 year mine life (assuming a production rate of 65,000 tpd). Tailings and waste rock storage is located in Galore Valley.

In order to undertake the preliminary design of the tailings impoundments and waste dumps a number of assumptions were made, and had to be taken into account during the tailings/waste design process.

8.5.2 Geotechnical Design Criteria

8.5.2.1 General Assumptions

The following general criteria were used to design the tailings impoundment facility and waste dump:

- Tailings will be stored in a cross-valley impoundment.
- The tailings dam will be designed using the downstream method of construction.
- All water-retaining dams will be rockfill dams with an upstream sloping impervious till core and filters. The dam shells will be built of waste rock hauled and placed by mine fleet.
- The ultimate size of the tailings dam will allow storage of the total volume of tailings and total volume of PAG waste rock assuming that two-thirds of the waste generated by the open pit is PAG. Operating pond will hold an estimate volume of the 200 year flood storage water and freeboard. Dam crest elevations were determined assuming a flooded tailings impoundment.
- Freeboard consists of an emergency surcharge to pass a PMF through the emergency spillway plus an operating freeboard (e.g. wave run-up, dam settlement, etc).

- Volume elevation curves for each location were based on a 10 m contour interval topographic map.
- Tailings dam earthworks volumes were determined assuming no beaching losses.
- Water within the tailings pond and runoff from the waste dump is assumed to be acceptable for direct release based on models completed to date by expert consultants.
- 65% of all the waste rock (or 650 Mt) is potentially acid generating (PAG) waste rock and 35% (or 350 Mt) is non-potentially acid generating (NPAG) waste rock. The PAG and NPAG can be separated for the purpose of using NPAG for dam construction.
- PAG waste rock must be submerged within five years of placement in order to mitigate ARD.
- Given the low sulphur content of the ores, it has been assumed that lime will not be added to the tailings line prior to it being released into the impoundment.
- A dry settled density of 1.5 t/m^3.

Additional design criteria and assumptions are summarized below in Table 8-1.

Table 8-1: Summary of Additional Design Criteria for Tailings/Waste Rock Design

Design Criteria	Value
Total Mineral Resource	500 Mt
Mill Production	65,000 tonnes per day
Mine Design Life	20 – 25 years
Total Mass of Tailings to be Contained	500 Million tonnes
Total Mass of Waste Rock to be Contained	1,000 Million tonnes
Settled Dry Density of Tailings	1.5 t/m^3
Dry Density of Waste Rock	1.9 t/m^3
Specific Gravities	2.7 - Waste Rock 2.45 - Broken Rock
Processing	conventional copper flotation based on crushing, grinding & flotation unit operations (no cyanide); lime added to grinding and flotation; lower grade ore stockpiled and processed at a later period
Mining Method	conventional open pit mining methods using truck & shovel units
Product	copper concentrate shipped to Stewart sea port
Operation	year-round (12 months)
Primary Grind Grain Size	80% passing 100 to 120 μm & 50% passing 60 μm
Slurry Solids Transport Density	25% solids
Reclaim Water from Tailings	85% maximum recycle of slurry transport water

8.5.3 Seismic Design Criteria

8.5.3.1 Tailings Dam(s)/Seepage Recovery Dams

The project site is located in a moderately high seismic zone. Based on the CDA 1999 Guidelines, the downstream consequence of failure of a tailings dam located in Galore Creek is considered to be very high due to potential socio-economic, financial and environmental losses. The rating is the same for all stages during the life of the dam: construction, operation and closure. With this very high consequence rating, the dam has been diligently designed for the maximum design earthquake (MDE), which in this case will be the MCE or the 1:10,000 return period earthquake.

The tailings dam will be designed to a minimum factor of safety of 1.5 under steady state seepage and maximum water level for the downstream slope. Rapid drawdown is not considered likely due to the continuous tailings filling operation. The dam will be designed to allow some deformations. The design will allow for and prevent overtopping under the maximum credible earthquake and will prevent loss of freeboard under the operating design event. The seepage recovery dams, located downstream of the tailings and waste dumps, will be designed to the same seismic criteria as the tailings dam.

8.6 Water Management

8.6.1 General

The Galore Creek project site will experience a high annual precipitation of approximately 1650 – 2300 mm throughout its anticipated mine life. Rain and snow, as well as water-discharged from nearby icefields and glaciers, have been incorporated into the design of the proposed mine. The catchment area around Galore Creek is large (120 km^2) which means that large volumes of freshwater require diversion to facilitate mining and minimize environmental impacts and these diversions have been incorporated into the site designs and cost estimates for tailings and waste dump(s) at a pre-feasibility level in this PEA. .

Water quality in the tailings and waste dump facilities is currently being assessed. At this time, it is predicted based on pre-feasibility level testwork that the water quality will be suitable for discharge, but this will be confirmed by further investigation now underway. Seepage recovery dams, constructed downstream of the tailings dam and waste dumps, have been designed to capture seepage and local runoff downstream of the tailings dam(s) for the purpose of returning the water to the tailings pond. Recovery wells may also be considered for deeper seepage recovery.

Sedimentation ponds will likely be needed to remove suspended solids and to remove glacially derived sediments on the upstream end of the waste facilities.

In order to minimize the amount of freshwater entering the mine and waste containment areas, diversion ditches and channels are designed and included in all capital and operating cost estimates to intercept and divert freshwater.

The hydrologic design criteria proposed for each of these facilities is based primarily on the CDSA Dam Safety Guidelines for water storage facilities, CDA (1999) Guidelines, Province of BC regulatory requirements, and the consequences of failure of a given structure.

8.7 Water Balance

Given the high rainfall, and large catchments, it was considered important to develop a preliminary water balance and a prediction of water volumes that could potentially require treatment.

A continuous simulation spreadsheet water balance model with monthly time steps was created to evaluate the Galore Creek storage and potential water treatment requirements.

All runoff and drainage water from the open pit(s), the plant site and the waste rock dumps were assumed to be transferred to the tailings impoundment. Tailings impoundment water (contact water) is not assumed to require treatment prior to release to the environment based on the water balance model. Data to confirm this assumption for feasibility is being gathered as part of the 2005 field program at Galore Creek.

8.7.1 Drainage Areas

Surface water diversion works have been designed and included in all operating and capital cost estimates to limit catchment areas and direct clean non-contact water away from the facilities. The diversions include the East Fork of Galore Creek, the eastern portion of the catchment above the tailings/waste rock impoundment and areas of undisturbed ground above the Central and Southwest pits and the plant site. Diversions along the western side of the tailings impoundment were not considered due to steep valley wall slopes. The total remaining catchment area draining to the tailings impoundment was estimated to be 1860 ha, including tailings pond, waste dumps, undisturbed ground on the western side above the impoundment and below diversions on the eastern side. The year-to-year variation in each sub-area over the mine life was estimated and included in the water balance model. Ranges of the sub-areas are shown in Table 8-2.

Table 8-2: Catchment Area Variations (ha)

Area	Catchment Areas (ha)
	Galore Creek
Tailings Impoundment	
Total	1860
Tailings Pond	50 to 200
Disturbed Ground	100 to 450
Undisturbed Ground	1710 to 1210
Waste Dumps	
Total	200 to 400
Disturbed	50 to 300
Undisturbed	150 to 100 (incl. above)
Open Pits	
Total	160 to 520
Disturbed	50 to 410
Undisturbed	110 (approx.)
Plant Site	36

8.7.2 Other Assumptions

Major releases of water from the system to the receiving environment were assumed to occur during the snowmelt and during the open water season (April to October). Maximum allowable monthly release rates from the system were assumed as 6.5 Mm^3/month into Galore Creek.

8.7.3 Water Balance Results

The water balance model was run for the Galore Creek site for a mill throughput of 60,000 tpd and for a range of annual precipitation return periods. Output from the model runs included annual volumes of excess water in the system. The model was also used to assess tailings impoundment storage requirements in the spring freshet period.

Table 8-3 summarizes annual release requirements for Galore Creek for annual precipitation return periods from 10 year dry conditions to 200 year wet conditions. These volumes include the following:

- Net inflow to the tailings impoundment including local precipitation and net runoff plus net process inflows (slurry transport water minus tailings void losses minus reclaim)
- Net waste dump runoff = local precipitation and net runoff minus waste rock moisture losses
- Runoff from the open pit(s) area and from the plant site area

The total excess is the sum of all the net runoff volumes.

Table 8-3: Galore Creek Area – Annual Release Requirements (million m^3)

	Initial (1-2 years)	Ultimate
10 year Dry Precipitation		
Net Inflow to Impoundment	27.0	24.0
+ Net Waste Dump Runoff	1.3	4.4
+ Plant site Area Runoff	0.6	0.6
+ Open Pit(s) Runoff	2.4	8.0
= Total Excess	**31.2**	**36.9**
Average Precipitation		
Net Inflow to Impoundment	32.0	28.6
+ Net Waste Dump Runoff	2.0	5.6
+ Plant site Area Runoff	0.7	0.7
+ Open Pit(s) Runoff	2.9	9.7
= Total Excess	**37.6**	**44.5**
10 year Wet Precipitation		
Net Inflow to Impoundment	37.3	33.2
+ Net Waste Dump Runoff	2.7	7.0
+ Plant site Area Runoff	0.8	0.8
+ Open Pit(s) Runoff	3.4	11.4
= Total Excess	**44.2**	**52.5**
50 year Wet Precipitation		
Net Inflow to Impoundment	41.5	36.9
+ Net Waste Dump Runoff	3.2	8.0
+ Plant site Area Runoff	0.9	0.9
+ Open Pit(s) Runoff	3.8	12.8
= Total Excess	**49.4**	**58.6**

	Initial (1-2 years)	Ultimate
100 year Wet Precipitation		
Net Inflow to Impoundment	43.1	38.2
+ Net Waste Dump Runoff	3.4	8.6
+ Plant site Area Runoff	0.9	0.9
+ Open Pit(s) Runoff	4.0	13.3
= Total Excess	51.4	61.0
200 year Wet Precipitation		
Net Inflow to Impoundment	44.7	39.6
+ Net Waste Dump Runoff	3.6	9.0
+ Plant site Area Runoff	1.0	1.0
+ Open Pit(s) Runoff	4.1	13.8
= Total Excess	**53.4**	**63.4**

Assuming that the excess water will have to be stored in the tailings pond prior to discharge, Table 8-4 summarizes Galore Creek water storage requirements for the 200 year wet year snowmelt. The total water storage includes the minimum operating pond plus the maximum pond volume during the snowmelt period. The storage volumes assume a sustained release rate from the pond of 6.5 Mm^3/month (approximately 9030 m^3/hr) during the snowmelt period.

Table 8-4: Galore Creek Area – Water Storage Requirements (million m^3)

Year	Minimum Pond Operating Volume	Maximum Snowmelt Month Volume	Total Storage Volume
Initial	1.0	27.1	28.1
Ultimate	5.8	41.8	47.6

8.8 Waste Storage in Galore Creek Valley

8.8.1 General

To support a safe and environmentally sound long term closure plan, the primary tailings and waste facility in Galore Creek will be submerged to form a lake approximately 8 km long and 2 km wide upon closure. This area will include the following

- three water retaining dams (Main Dam, West Dam and East Dam);
- seepage recovery dam and/or recovery wells;
- treatment plant, if required;
- large freshwater pond; and
- open channel diversions.

The intent will be to leave all the waste in a flooded condition for perpetuity. The tailings and waste rock will be primarily contained behind the Main Dam. Tailings will be generated from a conventional copper floatation process, piped to the northern end of the impoundment and spigotted into the facility off the Main Dam or from other locations depending on the filling arrangement and desired pond location.

The majority of the waste rock from the open pit will be trucked and dumped into the facility, however, a portion of the waste rock will be hauled to the north end for use as dam construction material. The dams will be built of non-acid generating rock. Any PAG rock kept in the interior of the waste dump will be flooded sequentially throughout the mine life. Optimizing of waste dump heights and flood elevations with haul distances will be a critical operating criteria for this dump.

A large natural catchment area exists around the tailings impoundment. To minimize water inflow to the impoundment, open channel diversions are designed on the eastern slope above the impoundment and around the proposed open pits to divert freshwater. An impervious dam (East Dam) has been designed and included in capital and operating cost estimates to pond and divert water into the diversion ditch. The dam must be impervious to prevent freshwater from seeping into the impoundment. A large freshwater pond will be created in the East Fork of Galore Creek. To prevent water flowing toward the open pit areas, a diversion structure will be required in the West Fork of Galore Creek, the elevation of which will be controlled by the water elevation within the impoundment. All of the water dams, the Main, East and West will be built sequentially.

Seepage out of the impoundment bypassing the East Dam, will be captured by a seepage recovery dam and/or recovery wells located immediately downstream of the Main Dam.

8.8.2 Waste and Water Storage

The dams and waste dumps areas are scheduled to be raised incrementally. An average mill throughput of 65,000 tpd was used to develop filling schedules for the tailings system. It has been assumed at this stage that the total tonnage of tailings and waste rock is evenly distributed over the life of the mine as laid out in Table 8-5. In general, there is twice the tonnage of waste rock to tailings and the waste rock is placed in the facility at a higher density than the tailings.

Table 8-5: Required Tailings and Waste Rock Storage versus Time

Mine Life	Tailings (M m^3)	Waste Rock (M m^3)	Total Solids (M m^3)
-1	29.2	23.1	52.3
1	29.2	46.1	75.3
2	43.8	69.2	113.0
3	58.4	92.2	150.6
4	73.0	115.3	188.3
5	87.6	138.3	225.9
6	102.2	161.4	263.6
7	116.8	184.4	301.2
8	131.4	207.5	338.9
9	146.0	230.5	376.5
10	160.6	253.6	414.2
11	175.2	276.6	451.8
12	189.8	299.7	489.5
13	204.4	322.7	527.1
14	219.0	345.8	564.8
15	233.6	368.8	602.4
16	248.2	391.9	640.1
17	262.8	414.9	677.7
18	277.4	438.0	715.4
19	292.0	461.1	753.1
20	306.6	484.1	790.7
21	321.2	507.2	828.4
22	333.3	526.3	859.6
23	333.3	526.3	859.6

The estimated grain size information from the metallurgical testwork tailings indicates that 80% will be finer than 100 to 120 microns and 50% finer than 60 µm. This indicates the tailings are likely 50% sand and 50% silt/clay. Tailings characterization is being conducted to confirm the grain size and other properties such as dry density.

The initial design basis included an allowance for storing extreme precipitation event(s). For this pre-feasibility level assessment, the impoundment has been designed to store precipitation from the 200-year flood. After this extreme event is impounded, the flood storage capacity will be restored as the pond is brought back down to regular operating levels by controlled discharge in accordance with regulatory and permit conditions. Testwork and modelling is on-going to improve the definition and prediction of effluent water quality.

In addition to the flood storage described above, an allowance has been made for an operating pond to facilitate the reclaim of water for treatment.

8.9 General Layout

The preferred locations of the three water retaining dams are shown in plan view on insert Drawing 317882-SKA0-C-002. This location of the Main Tailings Dam, located just upstream of a deposit of lower Permian karstic limestone, has been optimized to achieve the maximum storage capacity with the least amount of embankment fill material. The West Dam is located east of the proposed Central pit with some room left for future expansion. The East Dam is located near the confluence of the West and East Forks to leave room for future waste storage.

Freshwater diversion channels around the impoundment and open pits are also shown on Drawing 317882-SKA0-C-002. The main diversion channel is on the eastern slope above the tailings/waste dump facility. Freshwater from the open pit area will also be diverted. It is expected that these ditches will require reconstruction several times, at different elevations and areas and costs have been included throughout the operating life to account for this progression. The open pit diversions show an example of what these channels could look like. In general, clean water diverted from the pits will either be transported to the freshwater pond behind the East Dam or, directly into the main diversion channel.

8.10 Freshwater Diversion

The catchment around the proposed Galore tailings/waste dump facility is estimated to be 120 km^2. The installation and maintenance of water channels and diversions included in the operating and capital costs in this PEA is estimated to reduce the total catchment area flowing toward the tailings/waste dump from 120 km^2 to 18.6 km^2.

All proposed open channel diversions for Galore, have the following typical design features:

- Channels will be designed for peak flows from a 200-year return period event.
- Channel longitudinal slopes will range from 0.3% or steeper to prevent sediment deposition along the channel. Steeper slopes (> 0.3%) in soil must have a 1.5 m thick layer of riprap overlying a geomembrane and a 0.5 m thick layer of sand and gravel bedding. No lining was considered for steeper slopes (> 0.3%) in rock.
- Cut slopes were assumed to be 2H:1V in soil and 1H:1V in rock.
- Energy dissipation structures and sediment traps are included cost estimates for areas where the diversion channel is intercepted by larger creeks or gullies. These installations could include: a small berm, pond, outlet and inlet structures, and/or suitable erosion protection measures.
- Bed protection and stabilization, as well as energy dissipation measures are included in designs and cost estimates where the diversion channel flows into the downstream creek.
- For the purpose of channel maintenance (i.e. snow clearing), a 10 m wide running surface was assumed alongside the channel. It was assumed this surface could be constructed from fill from channel excavation. Operating costs, labour and capital for added equipment have been provided to maintain these channels open year around.
- Channel base width has to be a minimum 20 m width to allow snow clearing equipment.

8.10.1 Main Diversion Channels

For Galore Creek, an 8 km long, 20 m wide (at the base), 5 m deep open diversion channel is proposed along the eastern slope above the tailings/waste dump at an elevation of between 700 to 724 m. The channel will collect water from the eastern slopes and from the pond created behind the East Dam, transport it around the facility past the Main tailings dam and into Galore Creek.

Near the Main Tailings Dam, the diversion channel would connect with the emergency spillway on the right abutment to reduce costs for constructing a second channel down to the creek. This section of the channel will likely have a steeper slope; however, it is expected that this channel will be excavated in relatively steep intact limestone. At the bottom of the channel, a settling pond is assumed to prevent any excess sediment from entering the creek. A road will be required to access the downstream ponds and dam toe for maintenance.

The main diversion channel will be constructed in rock and soil. To minimize rock excavation a cut and fill channel design has been adopted, which utilizes bulk waste rock as part of the fill slope. Rock and soil excavated out during channel construction will be placed and compacted downslope to create a 10 m wide road surface for vehicles to provide a means of cleaning the channel.

It has been assumed that the channel foundation conditions (soil or rock) are of low permeability to prevent excessive seepage out of the channel. If the channel is cut in pervious soil or highly fractured rock, a liner will be used.

8.10.2 Open Pit Diversion Channels

Open pit channels are also included in operating and capital cost estimates to divert water around the open pits and past the waste dump and tailings pond. These channel locations will move depending on where mining is occurring. Drawing SK0A-C-002 shows two typical open pit channels: Channel 1 in the southern portion of the deposit, near the G1 (West Fork) glacier and Channel 2 constructed to divert water from Dendritic Creek. Channel 1 is approximately 3000 m in length and transports clean water from West Fork Creek to the east of the Central pit and into the freshwater pond behind the East dam. Channel 2 is approximately 3800 m in length and transports clean water to the north across the high wall in the Central pit to the Main dam. Due to the steep nature of the slope above the pit and the presence of glaciers, it is not considered possible to divert water around the south and southwest pit area. Precipitation falling in this area will report to the pit and will be collected in a sump and costs for pumping are included in the PEA. For cost estimation it has been assumed that the channels will be 5 m deep and 5 m wide at the invert and will require lining, as the bedrock near surface may be pervious.

8.11 Construction Issues and Geohazards

Potential challenges identified and considered in the design, layout, and construction and operating cost estimates of all the proposed open channels are as follows:

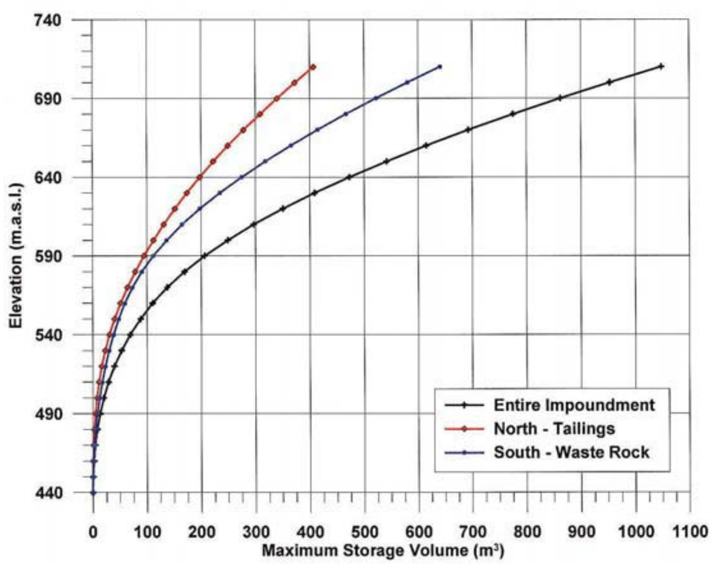
- *Steep Terrain* – areas of steep terrain have been avoided where possible or cost allowances have been made to reflect difficult construction conditions.
- *Climate* – The channels will be cleared of ice & snow (from precipitation or avalanches) on a continual basis and costs and equipment have been included in this PEA to allow for this.

- *Geohazards* – In addition to avalanches, other geohazards such as debris flows could temporarily block or destroy portions of the channel. Provision has been made in operating and maintenance costs for periodic events such as this.
- *Emergency Flood Conditions* - When the tailings pond exceeds the 200-year flood, tailings pond water will flow through the emergency spillway.

8.12 Construction and Storage Requirement

Tailings and waste rock will be stored separately, but adjacent to each other in Galore Creek Valley. PAG waste rock will be placed in the lower elevations of the valley so that the waste will be flooded within five years of placement. The potential storage capacity in the proposed tailings/waste rock basin has been estimated for the dam alignments and waste dump plans. Volume vs. elevation relationships were defined using a 10 m contour interval map and are shown in Figure 8-2.

Figure 8-2: Galore Volumetric Storage Capacity



8.12.1 Dam Heights

The estimated dam crest elevation for each year of the mine life was determined assuming a production rate of 60,000 tpd, waste tonnages shown in Table 8-5 and the volume – elevation curves presented above. Table 8-6 shows the heights of the three water retaining dams and the aqueduct for Year -1, 10 and end of mine life (Year 23).

Table 8-6: Galore Required Crest Elevations and Maximum Dam Heights

Dam Name	Height	Crest Elevation	Valley Bottom Elevation
Main Dam			
Year -1	138 m	578 m	440 m
Year 10	210 m	650 m	"
Ultimate	265 m	705 m	"
East Dam			
Year -1	44 m	674 m	630 m
Year 10	99 m	674 m	"
Ultimate	99 m	729 m	"
West Dam			
Year -1	0 m	-	610 m
Year 10	40 m	650 m	"
Ultimate	95 m	705 m	"

The Main Starter Dam has been sized to contain the first two years of tailings production, a probable maximum flood and 3 m of freeboard. The ultimate dam height is sized to contain all the tailings, a PMF, 3 m of freeboard, and also flood all PAG waste rock at mine closure. The required maximum height of the Main Tailings Dam is 265 m.

In Year -1, the East Dam must be constructed to a height such that freshwater can be diverted through the initial Main channel diversion on the east valley slope. The crest elevation of the initial aqueduct will be controlled by the elevation of the initial channel diversions. The elevation of the West Dam is controlled by the water level in the pond so in Year -1 when the water level is below elevation 610 m, the West Dam is not yet required.

8.13 Closure Concepts

Based on preliminary analyses of acid rock drainage potential, it has been conservatively assumed that two thirds of the waste rock will be acid generating and will require flooding in perpetuity. It has further been assumed that the ARD will not develop for up to five years so flooding of the waste rock may be delayed without correcting acidity.

Upon closure, flooding of the PAG waste and the lower pit slopes is planned to maintain good water quality. It is envisaged that all the tailings and all the PAG waste rock will be flooded in perpetuity behind the Main Dam creating an artificial lake. A final spillway capable of passing a probable maximum flood will be excavated in rock and the dam will be armoured with waste rock. It is expected that only nominal costs for upgrading the final spillway will be incurred.

No costs for closure have been estimated at this time pending feasibility level assessment of water quality.

9. Infrastructure – Off Site

9.1 Summary

Development of the Galore Creek deposit will require the construction of a 135 km main access corridor (including road, power and concentrate pipeline) which will run west from Highway 37 at a point approximately 8 km north of Bob Quinn, rising up along the More Creek Valley, down Sphaler Creek valley to the Porcupine River then north up Scotsimpson Creek to a tunnel through to Galore Creek Valley. All costs for this access corridor are included in the capital and operating cost estimate for this PEA and have been developed to a pre-feasibility standard and are based upon detailed field mapping performed by consultants familiar with design and construction of access roads in similar terrains.

Highway 37 and new Galore Road will provide the main access for transport of mining equipment and consumables, construction equipment, process plant supplies, personnel and food during construction and plant operations. Additional access requirements have been addressed with criteria set out for the Galore Creek Deposit access road and bridge crossings.

Inexpensive hydroelectric power will be provided to the Galore Creek project by connecting to the B.C. grid using a 138 kV transmission line at a point near Bob Quinn Lake on highway 37 and continuing west along the Galore Creek access road. The location of the Galore Creek project relative to the existing B.C. hydroelectric grid presents certain technical challenges for the economical provision of electrical power to the site. To address these challenges, three independent studies have been conducted to determine the technical feasibility and cost of grid connection to supply power to the site. These studies have confirmed that 80 MW of power may be delivered to site by a series capacitor compensated 138 kV line. A definitive study and detailed engineering will be undertaken as part of the feasibility work program.

Access to the minesite will be by way of a 4 km long tunnel, sized to accommodate both construction and operations traffic. The mine access tunnel is to extend north from the headwaters of Scotsimpson Creek to immediately south of the most southern proposed open pit in Galore Creek. Prior to the 2005 field season, limited field geotechnical investigations had been completed along the tunnel alignment. However, Provincial government regional mapping indicates that the tunnel will encounter two main bedrock units: medium grained quartz diorite to hornblende grandiorite intrusive rocks in the southern 1.6 km; and undivided volvanics and sediments in the northern 2.4 km. As well, a limited number of shear faults can be expected to intersect the tunnel alignment.

All capital costs associated with this access tunnel have been prepared to a pre-feasibility level of confidence using tunnelling contractors and EPCM groups familiar with developing sites in mountainous terrain. While overall rock mass quality and expected tunnel conditions are expected to be good over most of the tunnel, some rock support will has been provided for during tunnelling. As well, allowances have been made for pre-excavation grouting, spilling and support due to expected occasional fractured bedrock conditions. Minor groundwater inflows are expected to accumulate along the tunnel alignment and designs and costs have been adjusted to reflect this. Due to the potential of moderate in situ stress levels, additional allowances have been made for increased rock support over the central region.

A trade-off evaluation was conducted to select the most appropriate and economical case for concentrate transportation. The option which resulted in the lowest operating cost and least reliance on increasing diesel costs is based on pumping the concentrate slurry through a pipeline to a filter plant near Highway 37, where the filtered concentrate will be hauled to the terminal at Stewart for export to overseas markets. A conventional B-train style commercial truck haulage operation will transport filtered concentrate from the plant to the terminal at Stewart, which has two concentrate storage sheds. There is also space and approval for the construction on an additional shed, if required. Excess water from the filtration plant near Highway 37 will be treated and discharged in accordance with best practices and permit conditions.

Moving concentrate by slurry through pipelines is common practice in many large copper mines in Latin America and, in the case of Galore, the pipeline is designed to operate continuously with a design throughput of 2000 tpd. To accommodate variation in daily concentrate production the pipeline is designed to allow a 60% turndown through batch mode operation. To mitigate any risk of freezing during the winter months, the concentrate pipeline will be buried along the access road and insulated where it must be exposed to air.

Access road construction will be supported by temporary air strips located in two places along the Galore Creek road (Round Lake and Porcupine). The strip near the Porcupine River may be maintained during mine operation to facilitate crew change.

9.1.1 Road Engineering

9.1.1.1 Route Selection

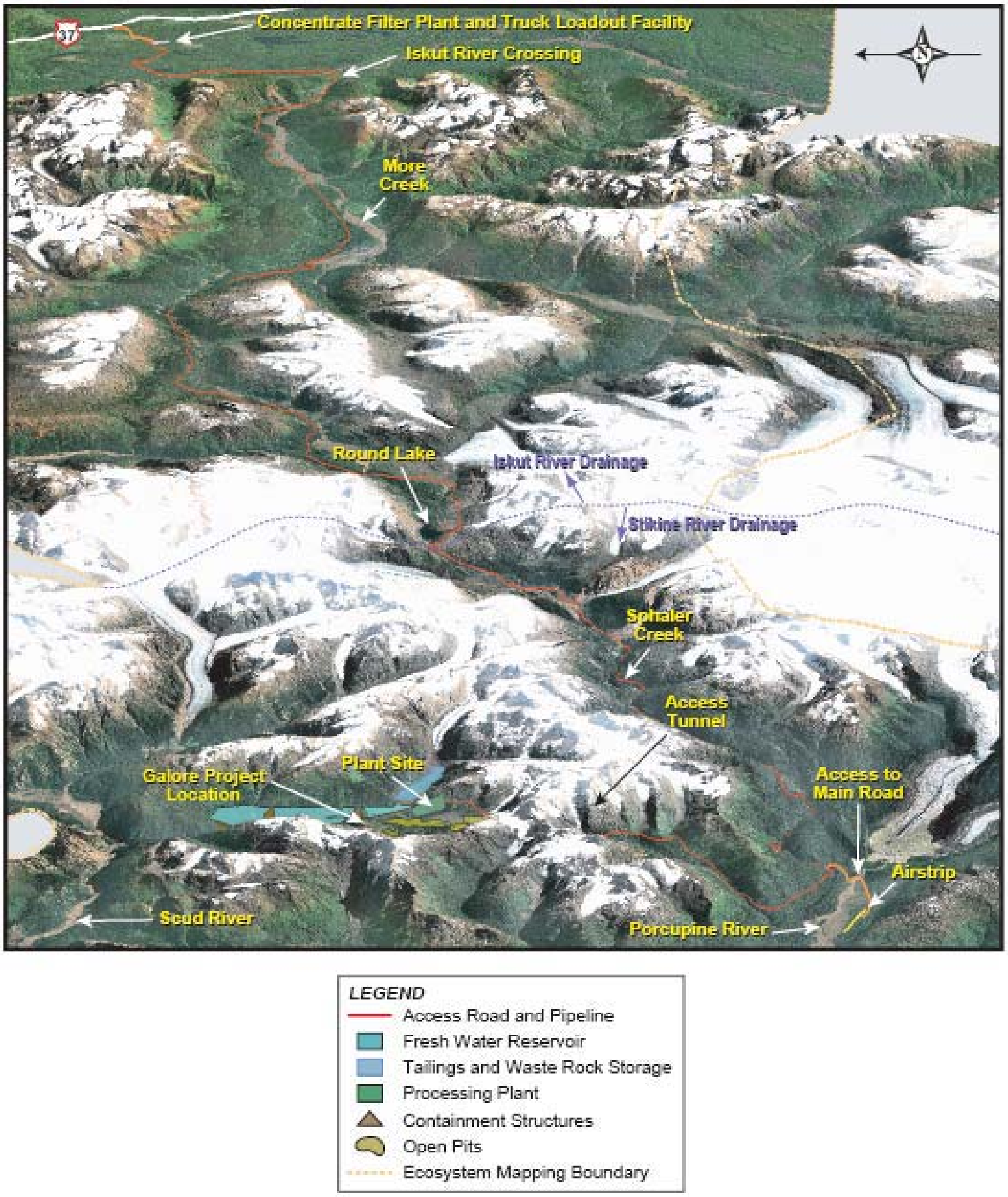
Aerial photographs at 1:20,000 and 1:30,000 scales along with 1:20,000 TRIM mapping and 1:50,000 topographic maps were used to identify all potential access routes to the Galore deposit. A comprehensive review of these resources, complemented by selective air reconnaissance provided important guidance for the ground reconnaissance work, which followed.

The North route (More Creek/Sphaler Creek) was marked on air photos and maps. A detailed ground reconnaissance was conducted by expert consultants familiar with designing and building roads in British Columbia. This field work involved walking each route using hand held GPS units and recording control points, terrain features and major creek crossings. Often several preliminary lines were investigated in order to achieve a feasible road location.

9.1.2 Detailed Route Description

The primary objective of field investigations was to locate and GPS the most appropriate road alignment from Highway 37 near Bob Quinn Lake into the Galore Creek deposit based on standards established by the project team and on information available at the time from field environmental and geotechnical studies. The optimal route was selected by balancing design standards, environmental sensitivity and field investigation of certain geologic and avalanche hazard areas.

Figure 9-1: North Access Alignment



9.1.3 Estimated Road Traffic

9.1.3.1 Construction Period

Traffic on Highway 37 and the Galore Road during the construction period will include mining equipment and consumables, road and tunnel construction equipment and consumables, as well as construction and mining personnel, and supplies.

9.1.3.2 Traffic During Mining Operations

Traffic on Highway 37 and the Galore access road during the plant operations will include mining consumables, process plant operating and maintenance supplies and consumables, concentrate (highway 37 only), as well as operations and maintenance personnel and supplies. The traffic frequency including the maximum daily trips is summarized below in Table 9-1.

Table 9-1: Traffic Frequency Estimate

Truck Type	Load	Delivery Frequency		Total Annual Trips	Maximum Daily Trips
20,000 L tanker	MIBC	1	Two weeks	26	1
40 tonne Flat Deck Trucks	PAX, Flocculant, Maintenance Supplies	1	Two weeks	26	1
40 tonne Tandem Trucks	Lime	5	Week	260	1
	Grinding Balls	10	Week	520	2
	Explosive	2	Day	500	2
	Concentrate	30	Day	7500	50
40 ft Trailers	Food	3	Week	156	1
Super B Train	Diesel	2	Day	500	2
50-person Buses	People	2	Week	104	1
Small Vehicles	People	7	Day	2500	10
Total Highway 37				**12,100**	**71**
Total Mine Access Road				**4,620**	**21**

9.1.4 Road Design Concepts

The Galore Creek Deposit access road is classified as a resource development road and the terms of reference adopted for the project specify a finished road width of 6 m with inter-visible turnouts and an average design speed of 60 km/h capable of carrying the legal load limit for trucks on British Columbia highways on a year round basis.

9.1.4.1 Drainage Structures

The following design criteria were established by the project team for the design of stream crossings:

- all bridges over 50 m length to be designed for the 200 year flood plus 1.5m debris clearance;
- all fish bearing streams shall be bridged or have open bottom arches installed to protect the stream channel;
- all culverts to be designed for the 100 year flood and cross drain culverts to be placed not more than 250 m apart.
- all major bridge crossings shall meet the clearance requirements of the Navigable Waters Protection Act.

Major bridge crossings are required for the Iskut River, More Creek and Porcupine Creek and field investigations have confirmed the suitability and cost estimates incorporated for these structures.

9.2 Power Supply

9.2.1 Summary

Technically feasible transmission line routes to supply 93 MW of electrical power to the Galore Creek mining project have been identified. For the purposes of this PEA, it is assumed that electrical power will be supplied from the BC Hydro (BCH) grid from an interconnection point near highway 37 and Coast Mountain Power Corp's (CMPC) Forrest Kerr run-of-river hydropower plant (Forrest Kerr). At this time it is uncertain what the maximum power supply capacity of this system is due to the system configuration. From preliminary evaluations by BCTC, it appears that the maximum practical power that could be supplied to the mine site at 138 kW is about 80 MW, with the addition of significant line compensation. Engineering and System Impact Studies will be commissioned by NovaGold with CMPC and BCH to confirm the system requirements to provide adequate north bound power to the project.

The powerline route will consist of 138 kV woodpole construction with "Arbutus" 37/0 ASC, 795 kcmil conductor. Some areas will require special steel poles for avalanche protection and costs in this PEA reflect this understanding. A Few areas need steel towers and "Drake" ASCR conductors for river crossings and long spans and cost estimates reflect this as well.

Avalanche frequency in the Scotsimpson Creek section (9 kilometres in length) may be best addressed by using an underground cable in this area. A more detailed study will be undertaken as part of the project's feasibility.

9.2.2 Introduction

The Galore Creek project has the strategic advantage of being able to rely upon low cost electric power from the B.C. hydroelectric grid and, because of this, is significantly less sensitive to changes in fuel costs than other large open pit operations. However, the location of the Galore Creek site and the limited capacity of the existing utility infrastructure represent technical challenges, which have been examined extensively to form the basis for this report. The main technical issues which have been identified for ongoing examination are:

1. The large distance (approximately 250 km) between the site and the existing utility infrastructure.

2. The 138 kV voltage rating of the existing infrastructure, which limits the uncompensated power transfer over a long distance.

It was recognized at the outset of the PEA that grid connection would be essential and that the technical and economic feasibility of the connection should be evaluated in detail. To this end, three independent system studies have been undertaken to determine the technical feasibility of the grid connection:

1. A preliminary interconnection study performed by Hatch to assess at a high level the feasible compensation strategies for delivering power to site at 138 kV.
2. An interconnection study performed by Coast Mountain Hydro/Siemens (CMH/Siemens) to assess the feasibility of "piggy-backing" the Galore Creek interconnection on the proposed Forrest Kerr Hydro Project 138 kV interconnection.
3. An interconnection study performed by BC Hydro/British Columbia Transmission Corporation (BCH/BCTC) to assess the feasibility of a series capacitor compensated extension of the 138 kV transmission line from the existing Meziadin substation to Galore Creek.

9.2.3 Power Supply Interconnections

9.2.3.1 Preliminary Interconnection Study

Hatch performed a preliminary interconnection study to assess which basic configuration would be feasible for delivering power to site given the existing 138 kV utility infrastructure. Of these options, only the series capacitor compensated 138 kV line is capable of delivering the design basis 80 MW site power requirement.

9.2.3.2 CMPC Interconnection Study

Coast Mountain Power Corporation (CMPC) is in the process of developing a run-of-river hydro project at the confluence of the Forrest Kerr and Iskut rivers in the proximity of Bob Quinn Lake. The project is scheduled to come online in 2008. The peak summer output of the project is in excess of 100 MW however this tapers off to less than 10 MW in the winter months. Consequently, the hydro project could not meet Galore Creek's power demand year round. This project will, however, result in the extension of the B.C. grid northward along highway 37 to a point near Bob Quinn where the Galore Creek line is assumed to interconnect.

9.2.3.2 BCH/BCTC Interconnection Study

In the event that the CMPC project does not proceed, BCH/BCTC has been retained to perform an interconnection study based on an AC connection from the minesite to Meziadin. BCH/BCTC had previously investigated the prospect of aggregate 100 MW load distributed in the vicinity of Bob Quinn Lake with encouraging results, however they had not considered a single concentrated load of the size and type of Galore Creek.

Power transfer at 138 kV must consider the distance (about 520 km) between the backbone grid (Skeena 500/287/138 kV substation) and the mine site. In order to increase the power transfer capability of the line, BCTC has proposed series capacitor compensation of the line in the order of approximately 50%. The maximum practical power transfer is 90 MW (80 MW mine site plus 10 MW area loads).

BCTC's conclusion is that maximum practical load that can be supplied to the mine site is 80 MW with 50% series capacitor compensation (partitioned among the three line segments) and shunt compensation (synchronous condenser) at Meziadin and the mine site.

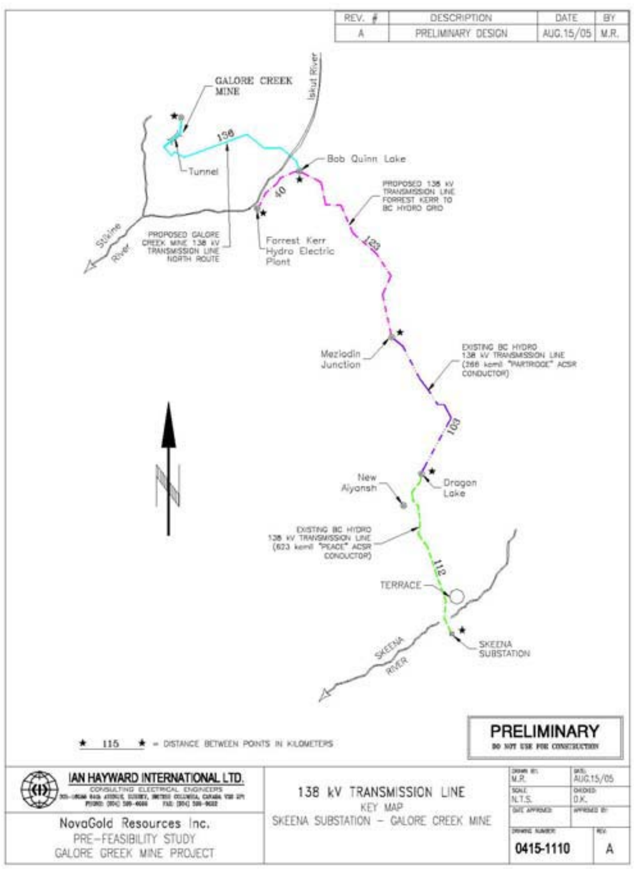
The planning level ±50% estimate for the grid system upgrades proposed by BCTC assuming that CMPC does not proceed is $60M. This includes the following:

1. Addition of a second 287/138 kV, 90 MVA transformer at Skeena.
2. Re-conductoring the 138 kV line segment between Aiyansh and Meziadin.
3. Two (2) 50% series capacitor stations at Aiyansh and Meziadin respectively.
4. 60 MVAr shunt compensation at Meziadin.

The shunt compensation estimate is based on a Statcom, not a synchronous condenser. Note that the planning level estimate does not include cost of the line along the Galore Creek road nor equipment at the Galore Creek mine site (included in all capital estimates as a separate line item).

A map of the existing grid and the proposed connection configuration is shown in Figure 9-2.

Figure 9-2: Existing Grid Map



9.2.3.3 North-West Area Transmission Options

The BCTC Report is a general assessment of the options for servicing a number of mining loads and IPPs in the Northwest Region from the Skeena substation. This is in contrast to the above-described study, which assumes that Galore Creek is the sole development in the area. This report has been produced at the request of the B.C. Government to allow it to examine the cost and possible benefits of extending the B.C. grid northward along highway 37 to a point well north of the Galore Road turn off.

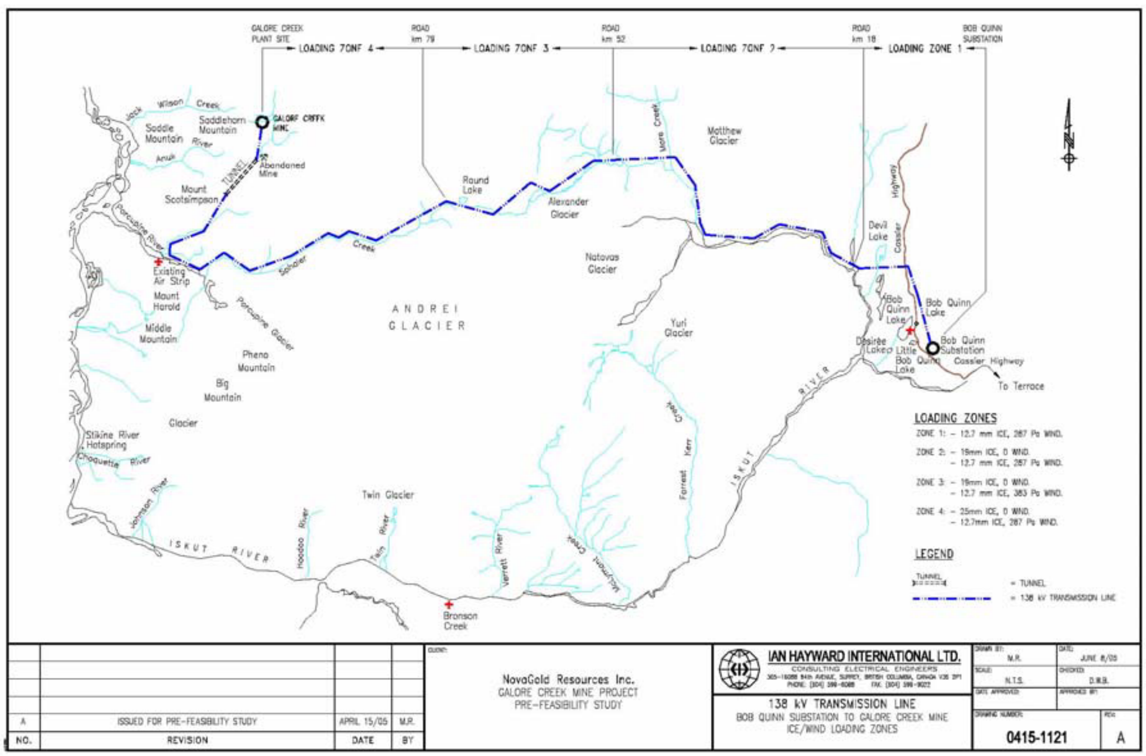
The maximum load served for the 138 kV option (110 MW) exceeds that recommended in the Galore Creek interconnection study and would be more than adequate for the project described in this PEA.

9.2.4 Transmission Line Route Selection

9.2.4.1 Route Selection

This study considers that a route for the 138 kV transmission line to the Galore Creek Mine will be from Bob Quinn Lake along the mine access road corridor, as shown in Figure 9-3.

Figure 9-3: 138 kV Transmission Line



9.3 Access Road Tunnel

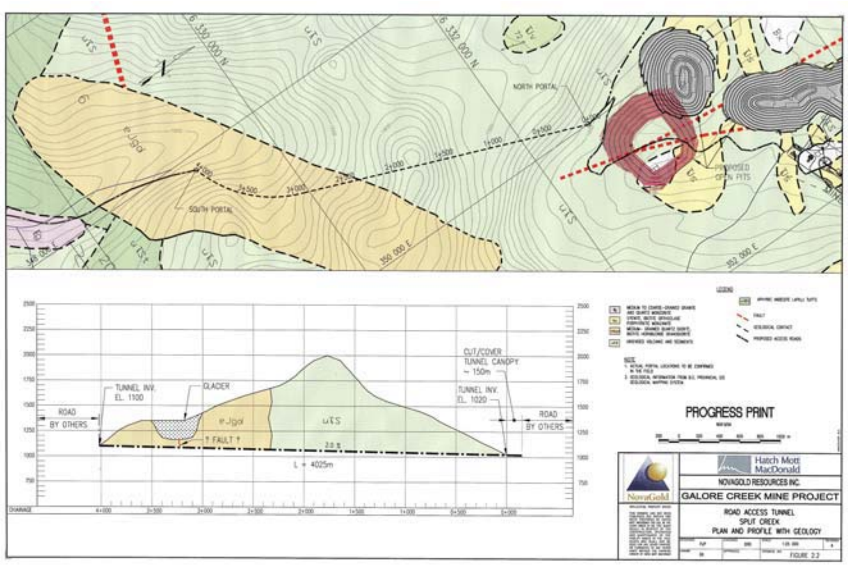
9.3.1 Summary

Access to the Galore Creek Valley and minesite will be via a 4 km long tunnel. The tunnel will be sized to accommodate both construction and operations traffic. Considerable effort was expended to identify the minimum sizes of equipment disassembled for road transportation, resulting in a finished tunnel cross-sectional dimension of approximately 6.8 m wide x 6.5 m high (excavated dimensions 7.3 m wide x 7 m high). It is proposed that the tunnel will be constructed using conventional drill and blast methods. The proposed mine access tunnel for the Galore Creek Mine is not unprecedented in size, length, depth of cover or geological conditions. Numerous similar sized and length tunnels have been constructed around the world.

9.3.2 Tunnel Layout

The mine access tunnel is envisaged to extend north from the headwaters of Scotsimpson Creek (at the end of the proposed road access route) to immediately south of the most southern proposed open pit in Galore Creek. Current portal locations indicate that the access tunnel will have a length of just over 4 km. Figure 9-4 shows the horizontal and vertical alignments of the tunnel, as well as geological information. The south portal elevation is estimated at approximately El. 1100 m, while the north portal is anticipated at approximately El. 1020m. All tunnel groundwater inflows will drain into the open pit due to the 2.0% gradient. The tunnel will possess a maximum cover of approximately 600 to 900 m along a 700 m long portion of its central region.

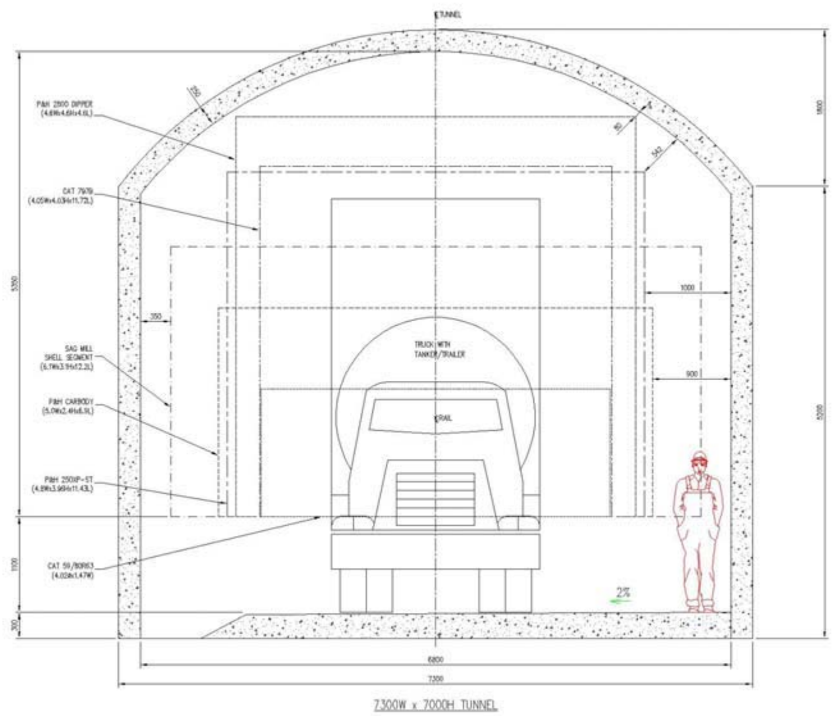
Figure 9-4: Mine Access Tunnel Plan and Profile



9.3.3 Access Tunnel Sizing

The criteria for the sizing of the access tunnel is to allow passage for all of the equipment and supplies for initial construction and on-going operations. These units would be broken down into the smallest practical size for transportation. The limitations to further size reductions were largely due to concerns over the effectiveness of site welding and consequent warrantee/liability issues and to significant non-standard engineering and manufacturing practices. It may be possible, with further study, to be able to further reduce the number of large size components although there is obviously additional costs associated with component size reduction, relating to product engineering and site erection. The tunnel cross-section is shown in Figure 9-5.

Figure 9-5: Road Access Tunnel – Clearances Sketch – Base Case



9.3.4 Expected Geological Conditions

Limited geotechnical or geological investigations were completed along the tunnel alignment prior to the 2005 field season. The BC Provincial GIS Geological mapping system was utilized to review expected bedrock units along the tunnel alignment as part of the design and cost estimation process.

Provincial government regional mapping information indicates that the tunnel will encounter two main bedrock units. The southern 1.6 km of the tunnel is expected to be excavated through medium grained, quartz diorite to hornblende granodiorite intrusive rocks. The northern remaining 2.4 km of the tunnel will be excavated through what are termed undivided volcanics and sediments belonging to the Stuhini

Group. The volcanic portion of the Stuhini Group is likely to comprise of basaltic to andesitic tuffs, flows and fragmentals that are observed to be very strong and massive in adjacent areas. The sedimentary assemblage within the Stuhini group is more likely to consist of well-bedded siltstones and sandstones, which can be expected to be weaker and thinly bedded. It is currently unknown which of the volcanic or sedimentary facies will dominate the Stuhini Group bedrock unit in the region of the tunnel. However, based on exposures elsewhere in the region, it would appear that the volcanic assemblage is more prevalent. Site investigation program activities underway during 2005 will more closely characterize the geological and geotechnical properties of the bedrock.

While provincial mapping has not identified any regional fault structures that cross the tunnel alignment, a limited number of fault/shear zones can be expected to intersect the tunnel alignment. The geological contact between the intrusive and volcanic sequences mid-way along the tunnel alignment may also have similar conditions to those commonly expected with fault and/or shear zones.

9.3.5 Expected Tunnelling Conditions

Based on inspection of adjacent areas of similar rock types, the overall rock mass quality and expected tunnelling conditions are expected to be good over most of the length of the tunnel. Rock support consisting of pattern rock bolts and / or shotcrete will be required to provide initial support during tunnelling. Once the access road has reached the south portal, final support, consisting of additional shotcrete (and possible rock bolts) will be applied.

Fractured bedrock conditions can be expected at and near the intersection of geological contacts and fault/shear zones that may include significant groundwater inflows. These fault zones will vary in width and rock quality but in general, will represent poorer tunnelling conditions that will slow advance rates and require the installation of steel sets and application of thicker shotcrete. Methods and procedures for managing tunnelling in such areas are well established. For costing and schedule purposes, a set number of faults and widths has been assumed and additional allowances have been made for pre-excavation grouting, spilling and rock support.

Minor groundwater inflows are expected to accumulate along the tunnel alignment except at intersection of fault zones where moderate to high inflows could be encountered. Using assumed hydraulic conductivities for different bedrock units, depth of cover and fault zones, steady state groundwater inflows after completion of the tunnel are estimated at 50 litres per second. Flush flows up to 35 litres per second can also be expected at individual higher permeability features such as fault zones. These estimates will be confirmed once results from future site investigations are completed.

The tunnel is located beneath a glacier at the extreme southern end. The depth of the glacier is currently unknown. In order to avoid a possible deeply eroded valley, the alignment has been curved into the mountain to increase the depth of cover below the glacier. A fault has been assumed to coincide with the glacial valley.

There exists the potential for moderate in situ stress levels to be present along the tunnel alignment given the thrust fault environment of the project area. Over-stressing conditions could be expected to occur within the central region of the tunnel given that it possesses both high ground cover and may encounter the weaker sedimentary rock facies of the Stuhini Group. Overstressing of bedrock is a key concern for tunnel construction due to the impact to schedule (slower advance) and cost (greater support). Additional allowances have been made for increased rock support over this central region (~700m) of the tunnel.

The quality, and in particular, the strength of these rock types are uncertain. Rock quality and rock strength represent key parameters for tunnelling in terms of excavation advance rates and assessing the potential for overstressing that effects tunnel support requirements, both of which have a significant influence on construction cost. To compensate for this uncertainty conservative cost and advance rate assumptions have been incorporated into this PEA.

9.4 Concentrate Handling

9.4.1 Summary

Concentrate transportation was the subject of an extensive trade-off evaluation to select the most appropriate and economic case relative to the project access options. The base case carried in the PEA is based on pumping the concentrate slurry to a filter plant located near Highway 37, from which filtered concentrate will be hauled by truck to the terminal at Stewart. Excess water will be cleaned and discharged into the Iskut River in accordance with permit and regulatory conditions. This option results in the lowest operating cost and the least reliance upon volatile diesel prices, although the pumping system does increase the capital cost.

PSI was commissioned to carry out a scoping level study to assess the technical feasibility of the concentrate piping system, and to develop scoping level cost estimates for the system. Prior to the 2005 field season, the location of the remote filter plant has not yet been ground truthed and no testwork had specifically been carried out on concentrate or filtrate to confirm process parameters. The key results of the scoping study which is incorporated into this PEA are summarized below in Table 9-2. The capital cost estimate for the slurry pipeline system is US$81M, while the annual operating cost estimate is US$4.07/t per tonne concentrate.

Table 9-2: Pipeline Systems Summary

Design Parameter	
Design Throughput	87.7 tph
Design Flow Rate	90 m^3/h
Concentration	56 wt%
Pipe Length	125 km
Outside Diameter	168 mm
Average Wall Thickness	6.1 mm
Flange Rating	900/150 psi
Pipe Steel Tonnage	3,229 t
Agitated Tanks at Concentrator Plant	2 tanks, 8 x 8 m
Agitated Tanks at Terminal	1 tank, 8 x 8 m
Number of pump Stations	2
Piston Diaphragm Pumps per station	1 Op + 1 Std by
Batch Water Volume per Year	299,606 m^3
Total Water Use per Year	661,644 m^3

The concentrate terminal at Stewart was visited in the preparation of this study and it has been assessed that minimal upgrades will be required at the port, assuming that existing large concentrate sheds will be available to the project at the time. The terminal owner has space and approval to construct additional, suitable storage capacity if required.

9.4.2 Concentrate Pipeline System

Hatch retained PSI to perform a conceptual study of the slurry pipeline transportation system to assess technical feasibility and develop order of magnitude cost estimates for the systems. Hatch requested the study of two alternate pipeline routes. A site visit was performed by a PSI pipeline construction specialist, in April 2005.

9.4.2.1 Cold Weather Engineering

Depending on the depth of snow on the ground (snow actually insulates the soil from harsh air conditions), seasonal frost penetration along the Galore Creek access corridor could be several meters below ground. It is, therefore, envisaged that the pipeline will be buried below the frost line wherever possible and an allowance has been included in the cost estimate for this purpose.

Experience on similar projects indicates that the risk of a pipeline freezing, even in such harsh conditions, is primarily a function of power reliability. Mineral slurries typically arrive at the pipeline agitated storage tanks at elevated temperatures due to the energy input of grinding. Friction in the pipe helps sustain, and can increase, the slurry temperature depending on ambient soil conditions.

Detailed thermal modeling will be completed as part of the Galore Creek feasibility and is expected to demonstrate that the pipeline should be initially commissioned when the ground is not frozen. Additionally, heat input from the pipe to the soil will likely keep it from freezing unless an extended shutdown occurs. In case of an unexpected and extended pipeline shut down, pipeline flushing with water and then with air or nitrogen will be required as an emergency procedure to eliminate risk of freeze. The thermal model will calculate the allowable duration of the shutdown, which will be compared to power reliability to determine the final design. It could be that emergency generators will be justified, but the heating of the slurry at each station is likely not required.

9.4.2.2 General Operating and Control Philosophy

The concentrate slurry pipeline is designed to operate continuously with a design throughput of 2000 tpd. The pipeline must also accommodate a 60% turndown (i.e. 1200 tpd) through batch mode operation for periods with reduced concentrate production.

The design of the pipeline is such that shutdowns will not occur as a normal operating practice. Short-term shutdowns with slurry in the line are possible, however, and do not require complete pipeline flushing with water. Maximum shutdown duration will be established during system start-up testing and every effort will be made to avoid multiple shutdowns with the same batch of slurry in the pipeline.

The pipeline primary control strategy implies that the pipeline throughput will scheduled based on the concentrator production. When the concentrator production is below the pipeline minimum allowable flow rate, the pipeline will operate in batch mode.

9.4.3 Concentrate Truck Haulage

A conventional B-train style commercial truck haulage operation will transport filtered concentrate from the filter plant to the terminal at Stewart, similar in nature to that currently in operation at Eskay Creek, Kemess and Highland Valley mines.

Two local haulage contractors experienced with the area and with mineral concentrate haulage were contacted to provide costs for the proposed operations. ME was commissioned to carry out a traffic study to determine the cycle times for each route, to provide an alternative assessment of the likely costs.

The concentrate haul for the route is largely a BC highway haul, without significant difficult grades and kept open year around by Provincial road crews. The round trip time for this route is approximately 6 to 8 hours, and will enable the drivers to be based in Stewart.

Table 9-3 lists travel times for trucks on each section of roadway, as well as the estimated overall cycle times.

Table 9-3: Heavy Vehicle Travel Times

Direction	Section	Distance	Time
Inbound	Hwy 37A – Stewart to Meziadin junction	60.4 km	49 min.
	Hwy 7 – Meziadin jtn to route intersection	144.4 km	103 min.
	Total	**205 km**	**152 min.**
Outbound	Hwy 37 – Route intersection to Meziadin jtn	144.4 km	142 min.
	Hwy 37A – Meziadin junction to Stewart	60.4 km	52 min.
	Total	**205 km**	**194 min.**
Total Route Length (Round trip)		**410 km**	
Cycle Time[1]			**346 min.**

Note: Loading/unloading times and other delays at Port in Stewart or at the Galore Creek Mine are not included

9.4.4 Concentrate Storage and Port

The existing terminal at Stewart has two concentrate storage sheds, of 25,000 t and of 15,000 t storage capacity. The Eskay mine, which currently uses the port, has announced that it will cease production within 2 years. Various plans to transport concentrate from Kemess and BC Metals and coal from Klappan have been suggested, but there are no definite plans as yet.

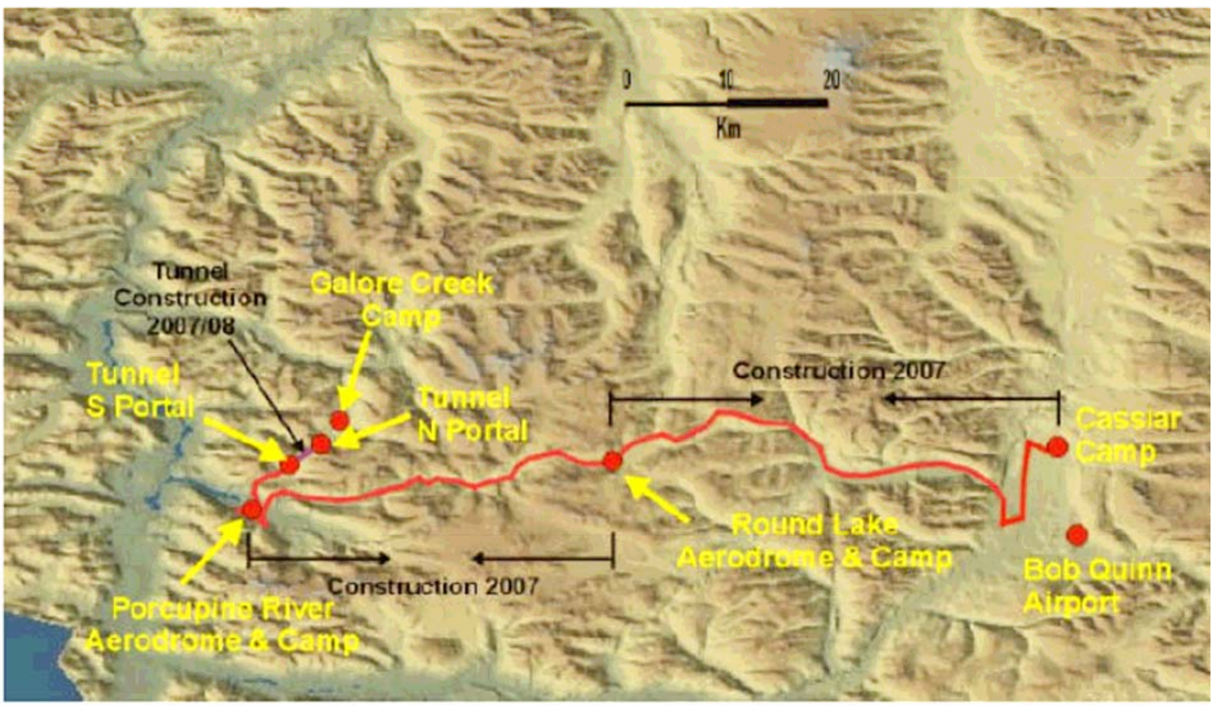
On this basis, it has been assumed that it will not be necessary to construct additional concentrate storage facilities at Stewart. Preliminary discussions with the terminal Owner indicate that there is space and approval to construct an additional shed if required. It is likely that additional truck wheel wash and containment facilities will be installed prior to Galore's startup.

Preliminary discussions with Stewart Municipality indicate that they have considered establishing a truck route to bypass the town centre. It has been assumed at this stage that these works, if required, will be carried out by the Provincial Government, at no cost to the Galore project.

9.5 Airstrip

Air strips at two locations (Round Lake and Porcupine River) will be required for the construction and operation of the Galore Creek mine, access roads and the tunnel. Figure 9-6 shows general locations of the two proposed air strips.

Figure 9-6: Air Strip Locations



10. Infrastructure – On Site

On-site infrastructure that is required to support the mining and processing activities includes the permanent camp, truckshop, fuel storage, cold storage buildings, assay laboratory, administration offices and water supply and distribution system. These facilities are briefly described below.

10.1 Truckshop, Warehouse and Offices

The truckshop, warehouse and office complex will be a pre-engineered steel building. There will be four truck maintenance bays, which will be suitable for the 345 t haul trucks, a truck wash bay and a tire shop. The workshop/warehouse/office complex will be located on the west end of the building, together with two small truck service bays. The workshop will provide a general repair area and welding and electrical shops, and will be fully equipped. The offices will be situated on the second floor above the warehouse/machine shop and dry and provide offices for mine and maintenance planning.

10.2 Fuel and Lubricant Storage and Distribution

Diesel fuel will be delivered to the site by tanker truck. Due to road conditions during the winter months, two 1244 m^3 (330,000 gal) storage tanks will be required, each containing approximately two weeks fuel storage.

Diesel fuel requirements for the mining equipment, process and ancillary facilities will be supplied from diesel fuel storage tanks located near the truckshop. The diesel fuel storage tanks will be 12 m diameter x 11 m high, installed within a lined containment area sized to contain 110% of the capacity of the storage tank.

Lubricants will be delivered to the site in drums. The drums will be stored in a secure area. The lubricants will be distributed to hose reels in the truckshop service bay with barrel pumps.

10.3 Site Power Distribution and Motor Control

The 138 kV transmission line will terminate at the main mine substation. At this substation, two identical step-down transformers will each supply half the plant load during normal operating conditions, with each transformer rated for 75/100 MVA ONAN/ONAF operation. The rating will allow one transformer to supply the entire plant load if required.

Substation compensation will consist of a shunt reactor. The reactor will have switching capability, allowing it to regulate voltage when desirable during light plant loading. Additional compensation will consist of power factor correction and harmonic filtering banks, which primarily compensate for the mill drive cycloconverters.

For a plant running load of 85-100 MW, 34.5 kV was selected for the primary distribution voltage, as standard 2000 A switchgear allows for a capacity of up to 120 MW. The two step-down transformers supply the plant 34.5 kV switchgear line-up. All feeder circuit breakers are 1200 A rated.

In the concentrator building, large motors (225 kW and up) and large variable speed drives (450 kW and up) are supplied at 4.16 kV via four 7.5/10 MVA transformers.

Smaller motors, variable speed drives (VSDs) and other power loads are supplied at 600 V. There are two main 34.5 kV circuit breaker feeds for 600 V distribution, and each feed is split between three 1.5/2 MVA transformer supplies via fused disconnect switches.

Each transformer supplies a 600 V 2000A distribution board comprising 1200A feeders that feed a range of MCCs, VSD line-ups and power distribution centres (PDCs). The MCCs supply all 600 V motors and contain all VSDs rated 75 kW and below. Larger VSDs are supplied separately in standalone panels. Control system and building service power is supplied from PDCs, comprising several 100 A and 200 A fused switches, and several 30 kVA transformer units supplying 208/120 V. The PDCs are independent of process motor loads. This allows motor faults to clear at the MCCs reducing the likelihood of interruption to the control system and building services power supplies.

Site power distribution is supplied via three separate 34.5 kV overhead line systems. The three circuits are mining, process loads and infrastructure/utilities, and each is fed from a separate 34.5 kV switchgear circuit breaker. This configuration allows for separate fault clearing for each system. Transmission line services to three separate pits have been allowed.

Two separate standby generation systems will serve the site. These systems consist of several containerized diesel engine driven alternators. Each system is able to be synchronized with the grid supply for routine testing purposes.

The first system is at 4.16 kV and will provide standby power to the main 34.5 kV network via a step-up transformer. The standby system is intended to supply power in the event of site incoming power loss, for safe shut down of plant operations while providing power for essential services (excluding camp facility). An allowance for four 1.5 MW units has been made.

The second system is dedicated to providing standby power to the camp facility 600 V system. The allowance for the camp generation is two 500 kW units.

Site locations external to the concentrator building will have power delivered by the 34.5 kV transmission line network, which will supply skid mounted substations and containerized switchgear. Skid mounted substations will distribute power to the movable mine site equipment, and are inclusive of all switchgear and transformers. There will be three different types of skid-mounted substation, one type for mining equipment, another for mine & borehole dewatering, and a third type for borehole dewatering only. The first two types will be present at all mine sites, and the additional borehole dewatering substation will be present at the Central pit. ‘Fixed’ equipment locations, including the crushing area and several remote pumping stations along the dam, will have power distributed by local containerized switchgear.

10.4 Water Supply

A number of water circuits will be required for the operation, as follows:

- process water
- fresh water

- potable water
- fire water
- tailings reclaim water
- tailings effluent discharge

Fresh water for fire water, process make-up and gland seal water will be supplied from the East Fork water dam. A barge mounted pumpstation on the East Fork dam will pump water to the 8 m diameter x 11 m high, steel fresh water tank, from where it will be distributed. The lower section of the tank is dedicated to the fire water pumps. The upper section will provide the supply for the gland service and fresh water system.

Process water will be made up from fresh water, concentrate supernatant and tailings dam reclaim water, if required. Process water will be stored in a 17 m diameter x 19 m high steel tank and supplied to the distribution points by pumps.

Potable water for domestic use will be supplied from a wellfield, estimated at this time as 2 wells, located near the plantsite. The pumps will supply water to a 5 m x 5 m steel potable water tank, from where water will be pumped through a small ozonation treatment plant, to the potable water distribution circuit.

The fire water system will consist of the supply pumps, including a self-contained diesel powered unit, and a system of hydrants and fire hoses distributed throughout the plantsite.

Surplus water in the tailings pond will be pumped from the tailings pond by a series of vertical turbine pumps into the west diversion ditch. Pumped flowrates will be controlled to match the natural stream flowrates in the ditch.

10.5 Site Storage Facilities

Three unheated, pre-engineered, fabric covered buildings approximately 50 m long x 20 m wide will provide bulk, covered storage for winter operations. These structures can also be utilized for maintenance and laydown of large components during winter conditions.

A 48 m long x 12 m wide, heated, pre-engineered steel building, will house maintenance shops, offices and storage or working space.

10.6 Administration Offices

A 29 m long x 17 m wide, pre-fabricated office complex will house the main administrative functions for the operation. Sixteen offices and two open plan work areas are provided for senior management and administration.

10.7 Assay Laboratory

A 30 m long x 10.5 m wide, pre-fabricated building will house the assay laboratory, fully equipped to support the mining, metallurgical and environmental operations.

10.8 Permanent Camp

The permanent camp facility for the operation will consist of:

- a pre-fabricated, 2 storey, modular structure with 280 single occupancy rooms.
- 100 rooms with private bathrooms and the balance with shared dormitory style washrooms.
- kitchen and dining facilities.
- lounge and fitness rooms.

The structure will be designed with peaked roofs to handle the high snow loads in this location.

It is proposed to install the camp during the early phase of construction to house construction workers, together with additional, temporary camp facility at that time.

10.9 Sewage and Refuse Disposal

Domestic sewage generated from the camp and offices will be treated in rotary bioreactors and effluent discharged into a tilefield. Sludge will be periodically pumped and disposed of in a lined disposal site.

Domestic garbage will be incinerated and/or buried in a lined disposal site.

10.10 Communication System

The communication system for the mine site includes a satellite telephone system, PC LAN and the fibre optic cabling connecting the various sites.

The fibre optic cable between the crusher station, concentrator, truckshop, East Fork water pumping station and reclaim water pumping stations will be strung along the electrical transmission lines.

11. Concentrate Marketing and Sales

Copper data includes statistics provided by Bloomsbury Minerals Economics Limited (BME) with respect to both metal prices, concentrates balances and treatment charges. Ocean transportation costs comments from load port to market are derived from NSA market knowledge and NSA discussions with shipbrokers.

11.1 Copper

The price in 2005 continues to be strong with prices rising well above US$1.50/lb level. High demand for copper together with the fact that many smelters postponed maintenance shutdowns, which were not affordable during 2002/2003 and were rescheduled to the first half of 2005, are factors supporting continued high prices.

On the supply side, global mine production rose from 13.7 Mt in 2003 to 14.5 Mt in 2004 about 6% (limited by large production losses at the Grasberg open pit in Indonesia), while further along the production chain, primary refined production growth was limited to well under 0.1 Mt in 2004. Although refined production from scrap increased sharply, the overall increase in refined production was only 0.5 Mt (under 4%) from 15.3 Mt in 2003 to 15.8 Mt in 2004. With refined consumption growing over twice as fast as production in 2004, a large supply deficit arose – around 1.1 Mt. During 2004, exchange stocks were drawn down by about 0.7 Mt. The entire Codelco 0.2 Mt strategic stockpile was sold and the Chinese strategic stockpile was drawn down by over a million tonnes. By the end of 2004 remaining copper stocks had fallen to very low levels.

BME is forecasting about 4.5% increase in mine production in 2005 but an increase of over 10% in refined production to 17.42 Mt and an increase in refined consumption of about 4% to 17.32 Mt.

A small production-consumption surplus for 2005 is forecasted at around 0.1 Mt. In addition, BME reports the Chinese bought copper in early 2005 for their strategic stockpile, but in early May were a seller of stocks. BME is forecasting an average LME cash price of about US$1.40/lb in 2005.

The prices shown in Table 11-1 indicate expected levels relative to expected consumption/production forecasts.

Table 11-1: Global Production-Consumption Balance over 2002 to 2010

Global Production-Consumption Balance (Thousand tonnes refined copper)										
	2001	**2002**	**2003**	**2004**	**2005**	**2006**	**2007**	**2008**	**2009**	**2010**
Copper Production	15,602	15,308	15,288	15,819	17,421	18,475	19,530	20,422	21,083	22,389
Copper Consumption	14,680	15,033	15,566	16,959	17,324	18,172	19,537	20,149	20,454	21,966
Balance Surplus/-Deficit	922	275	-278	-1,140	97	303	-6	272	629	423
Inv. End Year	3,865	4,140	3,862	2,722	2,819	3,121	3,115	3,387	4,021	4,439
Copper Price										
US c/lb	71.6	70.7	80.7	129.1	140.9	127.5	126.8	118.8	106.5	102.5
	Source: Bloomsbury Minerals Economics Ltd ©									

Source: Bloomsbury Minerals Economics Ltd ©
Note: The average Copper Price over 2001-2010 is 102.6 US cents per pound, basis actual prices and estimates shown.

Copper supply from copper mines grew sharply in the second half of 2004. This trend will continue in 2005 but will slow down considerably in 2006. Growth beyond 2006 looks to be slow and will depend on the uncommitted projects still on the drawing board. If the copper price continues to be strong, the incentive to move forward on these projects will be strong. One caveat is the long lead time required for mining and milling equipment in the current market, together with increasing project capital costs which will likely delay some projections and expansions.

Much of the increased supply (tonnes of copper in concentrates) in 2004 and 2005 is the result of mine expansions. These include Escondida (125,000), Collahuasi (90,000) and El Teniente (105,000) in Chile and Telfer (25,000) in Australia. Sossego in Brazil (140,000 per year) went into production in 2004 and is now running at full capacity.

With the recent price increases, mines that closed for economic reasons are reopening. The Tintaya mine in Peru has re-started adding 90,000 -100,000 tpa. Gibraltar and Mt. Polley in Canada and Robinson in the USA opened during the fourth quarter of 2004 and the first quarter of 2005 respectively. Freeport's mine in Indonesia, which experienced major production difficulties in 2004, is back at full production in 2005.

Although there are no significant projects yet to be announced there are several junior mining companies around the world studying smaller projects. If prices continue to hold at present levels, some of these will likely be developed.

There has been a dramatic change in the concentrate market in the past 18 months. Smelters that were worrying about having to curtail production due to lack of material are now asking mines to delay shipments because of high concentrate inventories. The main causes of this situation were mine expansions, delayed expansions at smelters, smelter failures (Freeport) and rescheduling of smelter maintenance shutdowns. However, there are several expansions in China (Tongling, Yunnan, Huludao to name a few) due to be completed in 2005. Codelco is planning a major expansion of their facilities over the period 2008-2010. In addition there are considerable "de-bottlenecking" projects planned at a number of smelters, which will add substantially to capacity. This issue is somewhat low key but in private discussion with many smelters, it is apparent that a significant increase in capacity may result.

A key factor in commodities has been the fall in the value of the dollar and across the whole commodity spectrum, prices have risen strongly in US dollar terms but less so in other currencies. The BME forecast over the period 2005 to 2014 is an average in constant 2005 dollars of just over US$0.95/lb. As a comparison, in dollars of the day, copper over the period 2002 to 2004 averaged US$0.94/lb and over the period 2002 to 2005 to date averages about US$1.08/lb. For the purposes of this PEA, a long term copper price of US$1.00/lb has been assumed.

11.1.1 Silver, Gold and the Dollar

11.1.1.1 Silver

Statistically silver has shown an apparent primary supply deficit for many years, with about two-thirds of the world's demand coming from mine production and the balance from various stockpile sales and silver scrap. With the advent of digital photography, there has been concern about a fall in photographic silver usage. However, new industry demand is encouraging, with increasing usage in the electronic industry.

With a declining dollar stimulating interest, silver boomed in price over 2004. The price today is well beyond expectations of recent years, but is at a level, which is unlikely to be sustained over the longer term. Producers tend to use a planning price for the long-term of US$5.00 to US$6.00/oz in constant 2004 US dollars. For the purposes of this PEA, a long term price of US$6.00/oz has been assumed.

As a comparison, in dollars of the day, silver over the period 2002 to 2004 averaged US$5.40/oz and over the period 2002 to 2005 to date averages about US$5.60/oz.

11.1.1.2 Gold

Gold is enjoying a much more positive climate than has been the case in recent years. With the price quoted in US dollars, gold as a commodity faced an uphill price battle, while this currency remained strong. However, with the decline in the dollar, the metal has come back into favour.

Gold averaged just over US$363/oz in 2003, US$409/oz in 2004 and is about US$440/oz to date in 2005.

While the US dollar still remains to some extent a haven for value and wealth, gold has regained favour. A most likely long-term scenario is for a gold price in the range US$350 - US$400/oz in constant 2004 dollars, on the assumption of the US dollar remaining relatively weak. For the purposes of this PEA, a long term price of US$400/oz has been assumed.

As a comparison, in dollars of the day, gold over the period 2002 to 2004 averaged US$361/oz and over the period 2002 to 2005 to date averaged US$369/oz.

11.1.2 US Dollar Exchange Rate

One key feature of 2003/4 commodity price increases has been the fall in value of the US dollar against its world counterparts and this is reflected in higher dollar prices of many commodities. However, for many mine producers around the world, this price increase has been offset to a large degree by currency change. As already mentioned, the world's smelters and refineries in hard currency countries are very adversely affected. The USA economy is susceptible to debt and trade imbalance and the dollar outlook remains uncertain.

Commodity price increases in US$ dollar terms, have been offset to a lesser or greater extent in various commodities and currency realignments. Mines and smelters are not seeing the full benefit of price rises and the situation varies from country to country. Smelters in the Euro area as well as in Japan were under extreme pressure in 2004 with treatment charges expressed in US dollars and the country currencies strong.

11.1.3 Smelter Terms

11.1.3.1 Copper

Annual contract terms for 2005 between mines and Asian smelters settled out at US$85/dmt and US$0.085/lb again with full Price Participation.

With the dramatic change in the concentrate balance to a large surplus over the last year and a half, there has been an unprecedented increase in treatment and refining charges. Spot prices (one-off deals) have gone through the roof with business being done at US$200/dmt and US$0.20/lb where only 18 months ago the same deals were US$20/dmt and US$0.02/lb.

Negotiations for mid-year to mid-year 2005 annual contracts are currently ongoing and are expected to settle out around US$110/dmt and US$0.11/lb with normal Price Participation. It is likely that while there is currently a surplus, smelters should begin to erode their stocks as the year progresses. Key to this is the increased Indian capacity and completion of major maintenance shutdowns. However, this is not expected to have a material effect on smelter charges for 2006 but by 2007 a downtrend is likely.

For copper concentrates, the largest impact variables in the terms on a year-to-year basis are the TC, the copper RC and price participation. Historically, the sum of these three charges represented about 22% of the price. Annual benchmark terms are not published numbers, but represent a consensus of the average base numbers negotiated by the major players. Over the period 1984 to 2004, an average TC of approximately US$75/dmt and US$0.075/lb for copper RC has applied. Annual terms generally in recent years have included price participation at ±10% based on a copper price of US$0.90/lb, although there has been some variation.

For PFS purposes, a TC of US$75 to US$80/dmt concentrate and a RC of US$0.075 to US$0.08/lb of payable copper is recommended, along with PP of ±10% from a base copper price of US$0.90/lb. In the economic analysis and optimization of Galore Creek, a long term TC of US$75/dmt and an RC of US$0.075/lb has been applied along with 10% price participation when copper prices exceed US$0.90/lb.

11.1.4 Assumptions for Smelter Terms

The following is a summary of terms applicable for an evaluation of copper concentrates. Based on the information available at the time of this report preparation, it is unlikely penalties will be applicable but for the record, some typical penalties are included. This, however, is subject to review once more assays are available. It should also be noted that penalties vary with the market and the capability of particular plants.

In discussing TCs, mention was made that various charges having an economic effect are a matter of negotiation and therefore the charges shown below are typical as to what may be expected over the longer term. PP has for many years been quoted at ±10% at US$0.90/lb.

Another point of interest in looking at terms, is the question of payment for PGMs, specifically platinum and palladium in copper concentrates. Generally, most copper smelters do not pay for low levels of 1 - 2 g/t which would be payable, if it was gold. At certain smelters, where there is recovery, some indirect recognition in other terms could be on the table. Some smelters have been looking for concentrates with levels of say 50 to 100 g/t, for which case payment, around 70 to 80% might be achievable.

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

Payable Metals

Copper	Deduct 1 unit and pay for balance of content with refining charges of US$0.075/lb
Silver	If over 30 g/dmt, pay 90%, with a refining charge of US$0.40/oz
Gold	< 1 g/dmt, no payment 1 to 3 g/dmt, pay 90% 3 to 5 g/dmt, pay 93% 5 to 7 g/dmt, pay 95% 7 to 10 g/dmt, pay 96.5% 10 to 20 g/dmt, pay 97% Over 20 g/dmt, pay 97.5%
	Refining Charge US$5.50/oz

Deductions

TC	CIFFO main Asian port parity, US$75/dmt	
PP	±10% basis US$0.90/lb	
Penalties	Arsenic	US $3.00 per 0.1% over 0.2%
	Antimony	US $3.00 per 0.1% over 0.1%
	Lead	US $3.00 per 1% over 2%
	Zinc	US $3.00 per 1% over 4%
	Mercury	US $0.20 per ppm over 20 ppm
	Bismuth	US $5.00 per 0.1% over 0.05%

Selenium	US $3.00 per 0.01% over 0.05%
Fluorine	US $0.10 per 10 ppm over 300 ppm

Payment

Provisional	90% on arrival of an ocean vessel, which for average tonne deemed to be 45 days after production. This deemed time may vary once production volume and shipment size and frequency is determined. Final 10% balance when all facts known deemed to be 150 days after production

11.1.5 Marketability

Based on assays received and shown in Table 11-2 below, the concentrates are relatively clean except for the fluorine content. This may attract some penalty from time to time.

The expected target market area for the output is Asia given the west coast location and the accessibility of west coast ports.

Table 11-2: Galore Creek Concentrate Analyses

Concentrate Sample	Elements for Assay															By Global Lab	
	Pb	Zn	Ni	Co	As	Sb	Bi	Mo	MgO	Se	SiO2	Hg	Pt	Pd	Rh	F	Cl	Te
	%	%	ppm	ppm	ppm	ppm	ppm	%	%	ppm	%	ppm	ppm	ppm	ppm	ppm	%	ppm
KM1547-82 Cu Con IV,V	0.25	0.182	38	84	135	289	<9	0	0.41	114	6.97	2	0.08	1.48	0.09	440	0.02	15.8
Comp 1A - HG Cpy/Bn																		
KM1547-85 Cu Con III,IV	0.311	0.92	58	404	28	48	<9	0.01	0.07	140	2.28	2	0.05	0.37	0.02	290	<.01	9.51
Comp 2A - HG Cpy																		
KM1547-87 Cu Con II,IV	0.04	0.14	24	54	<10	48	<9	0	0.57	128	6.78	2	0.05	0.71	<.01	640	<.01	12.5
Comp 3 - Py/Au																		
KM1547-88 Cu Con II-IV	0.4	1.36	34	54	609	289	<9	0.01	0.19	189	6.85	2	<.01	0.79	<.01	330	<.01	7.66
Comp 4 - CuOx																		
KM1547-90 Cu Con III,IV	0.18	2.66	30	250	332	145	<9	0.03	0.34	159	4.75	2	0.01	0.11	0.01	1230	<.01	11.4
Cu Canyon - CC04-023																		
KM1547-91 Cu Con II-IV	0.22	0.320	48	162	17	181	<9	0	0.26	170	8.29	3	0.12	2.2	<.01	340	<.01	13.8
Comp 1B - LG Cpy/Bn																		
KM1547-93 Cu Con II-IV	0.173	1.11	46	164	29	60	<9	0	0.08	118	1.89	1	0.13	0.89	<.01	230	<.01	7.04
Comp 2B - LG Cpy																		
AVERAGE CONCENTRATE																		
Average of 1A, 1B, 2A, 2B	0.24	0.63	48	204	52	145	<9	0.003	0.21	136	4.86	2	0.10	1.24		325	<.01	12
Average of 1A, 1B, 2A, 2B	Cu	Au	Ag	Fe														
Head Grade (% or g/t)	0.73	0.38	7	5.2														
Concentrate Grade (% or g/t)	29.1	12	223	30.5														
Recovery (%)	90.7	70.5	65.4	13.9														

11.1.6 Logistics

For overseas markets concentrates will be moved by truck to Stewart, BC. Stewart is BC's most northerly ice free port and is capable of accommodating large ocean going vessels.

Sales contract quantities and the number of buyers and required delivery frequency will determine parcel size. With this sales volume it is likely that contracts will be approximately 65,000 t to 100,000 t for larger sales down to approximately 30,000 t to 40,000 t. Generally the intent will be to spread deliveries to a smelter evenly over the year. Given these volumes two to four ships per month would have to be scheduled in order to move production. Parcel size for the shipments could vary from 6,000 t to 8,000 t, 12,000 t to 14,000 t or 18,000 t to 26,000 t depending on the number of holds utilized. It is expected that actual vessel sizes will be either around 25,000 / 30,000 DWT with about 9 m draft or 45,000 / 55,000 DWT with about 13 m draft.

With these volumes Galore will need about 30,000 to 60,000 t of storage at the port approximately one months production. At Stewart, there is an existing concentrate loader and two storage sheds capable of holding over 30,000 t. This storage is presently used for Eskay Creek and Huckleberry productions, both of which currently have limited remaining mine life. It is highly likely such storage will become available about the time of Galore start-up.

It must be noted there are other projects in BC, which are potential shippers through Stewart. There is enough room to construct additional warehouse facilities capable of handling the volume needed for the Galore project. One caveat is the current environmental state at the facilities.

Ocean freight has been discussed with Simpson Spence and Young (SSY). This is the Canadian arm of the world's largest privately owned shipbrokers in the world with branches around the globe.

Over the last 24 months or so, ocean freight rates have risen to levels, which are two to three times those seen in recent years. For parcels around 10,000 t until 2002, rates have averaged about US $20/wmt off the west coast of North America to Japan. Today rates of US $55/t to US $60/t are prevalent.

All shipyards are full and new vessel supply is ongoing. However, conventional wisdom is that rates will stay high through 2005 and into 2006. However, shippers who are ready today to contract for approximately three to five years are able to achieve rates substantially below today's levels.

SSY suggest that for average parcels sizes in the 12,000 t to 14,000 t range, then rates out of Stewart are likely to be in the US $35t to US $45/wmt range (basis one port load and one port discharge). Rates will vary with load port conditions and destination and the number of discharge ports.

11.1.7 Other Offsite Costs

All other offsite costs including losses, insurance, assaying and selling amount to approximately US$5 – US$7/dmt.

12. Environmental Assessment and Socio-economics

12.1 Summary

NovaGold commissioned Rescan Environmental Services and the Rescan-Tahltan Environmental Consultants (RTEC) to develop and implement an extensive environmental assessment of the project area and to initiate the environmental permitting process. The main environmental considerations for the project area are stream and river crossings of potential fish bearing waters, extensive habitat for bears, mountain goats, and migratory birds and the naturally elevated background levels of suspended solids and some metals in the mine area.

The environmental application process has commenced with a revised CEAA Project Description submitted in mid-2005. It is expected that the permitting will continue through 2006 with final issuance of all permits and licences by the end of January 2007.

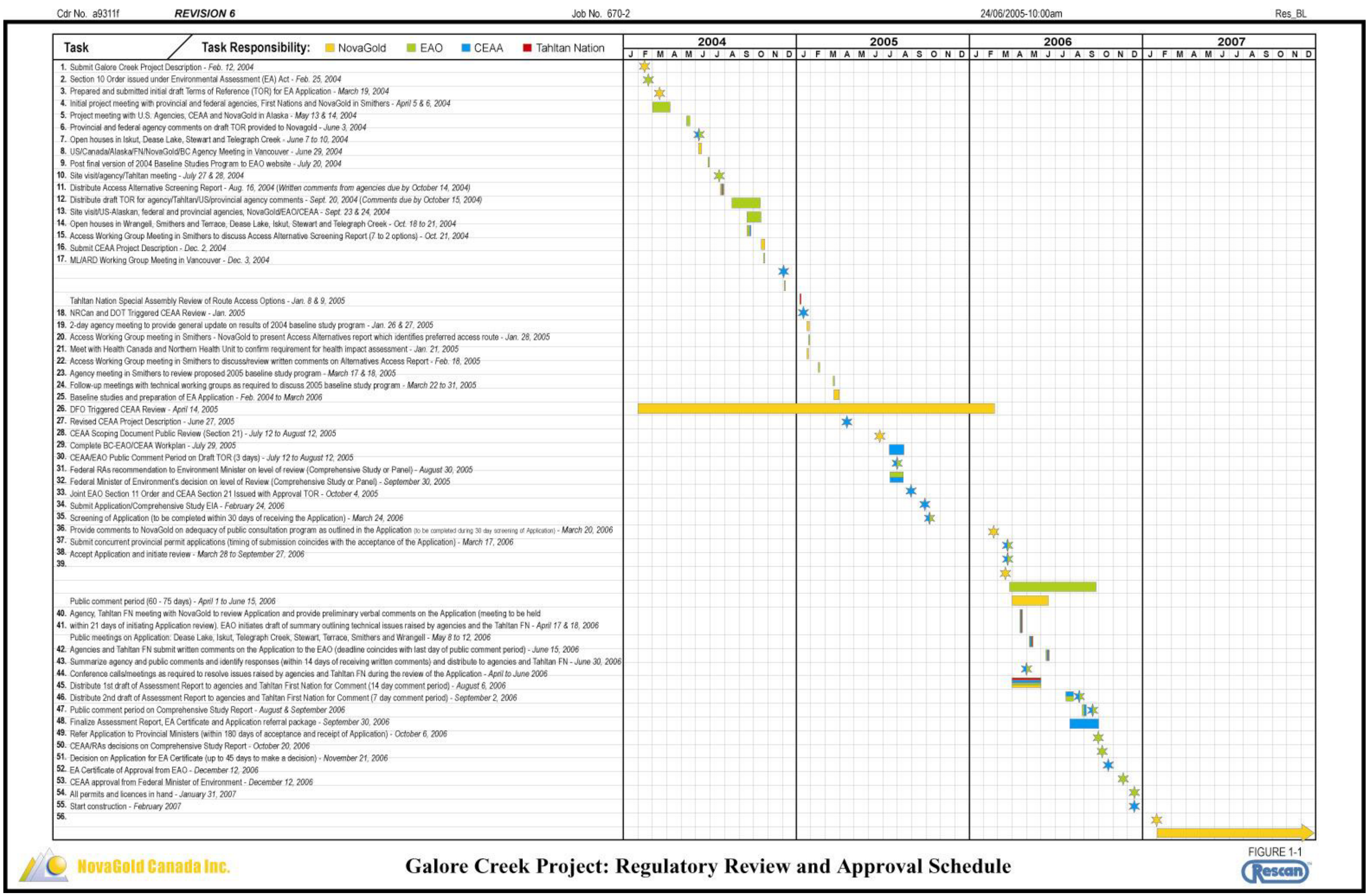
12.2 Regulatory Approval Process

The Galore Creek Project was initiated into the British Columbia environmental assessment process through the issuance of a Section 10 Order on February 25, 2004. Figure 12-1 presents a schedule of the review and approval process to January 2007. Pursuant to the schedule, the combined Application/Comprehensive Study Report is to be completed and submitted to the EAO and the Agency in early 2006.

Following the acceptance of the report, the EAO and the Agency have a legislative timeline of approximately 180 days to review and summarize the document for their respective Ministers. This effort should be completed by August or September 2006. The decision on the Application EA Certificate should be made by the responsible British Columbia Ministers by September/October 2006. The federal CEAA approval of the Comprehensive Study by the Minister of Environment should be made by October/November 2006.

By December 2006, the Galore Creek Project should have an EA approval from both the Government of Canada and the Province of British Columbia and be finalizing the permits and licences to construct the project. Major permits will be applied for under BC's concurrent permitting process and required licences should be completed by February 2007. Construction commitments for the Galore Creek Project could start in the late fall of 2006. Field construction would likely start in the spring of 2007.

Figure 12-1: Regulatory Review and Approval Schedule



12.2.1 British Columbia Authorizations, Licences and Permits

NovaGold intends to proceed with concurrent permitting. The agency responsible for the approval of specific permits must make a decision relating to issuing the approval within a specified timeframe. However, under no circumstance can an authorization to construct or operate the mine be issued until the environmental assessment has been completed and an Environmental Assessment Certificate has been granted.

The "Concurrent Approval Regulation" sets out the provisions related to concurrent permit approvals. With a concurrent approval, applications for statutory authorizations such as licences and permits are reviewed while the environmental assessment review process is ongoing. Any major project authorization is eligible for concurrent review.

The concurrent permits that NovaGold will be seeking are the essential authorizations required to start construction in 2007. These authorizations will include the following:

- all licences, permits and approvals related to the construction and operation of the mine site access road and powerline right of way;
- all licences, permits and approvals related to the development and construction of borrow pits and aggregate quarries including *Mines Act* permits and *Forest Act* licences to cut;
- *Mines Act* permit for initial site construction activity including construction of site roads, water diversion structures, excavation for major buildings such as mill, crusher and maintenance facilities, tailings dam and other construction related activities;
- all licences, permits and approvals related to operation of temporary construction camps for mine, plant site and road including permits required under the *Health Act*, *Drinking Water Protection Act* and *Environmental Management Act* (sewage, incinerator, refuse disposal and waste generator); and
- all water licences for diversion, use and storage.

12.2.2 Federal Approvals and Authorizations

The federal approvals include an authorization from the federal Minister of Environment approving the combined Application/Comprehensive Study Report for the Galore Creek Project. Major authorizations will be required from Fisheries and Oceans under the *Fisheries Act*. Approvals for water crossings will also be required under the *Navigable Waters Protection Act*. An explosive factory license will be required under the *Explosives Act*. The Metal Mining Effluent Regulation under the *Fisheries Act* and administered by Environment Canada is not likely to require a Schedule II authorization because the areas proposed for the tailings impoundment do not contain fish. Other federal requirements such as those in respect of radio communication and aviation matters will need licences. The following Table 12-1 lists some of the federal approvals required.

Table 12-1: List of Federal Approvals and Licences Required to Develop Galore Creek Mine

Federal Government Approvals & Licenses	Enabling Legislation
CEAA Approval	*Canadian Environmental Assessment Act*
Metal Mining Effluent Regulations (MMER)	*Fisheries Act*/Environment Canada
Fish Habitat Compensation Agreement	*Fisheries Act*
Section 35(2) Authorization	*Fisheries Act*
Navigable Water: Stream Crossings Authorization	*Navigable Waters Protection Act*
Explosives Factory License	*Explosives Act*
Ammonium Nitrate Storage Facilities	*Canada Transportation Act*
Radio Licences	*Radio Communication Act*
Radioisotope License (Nuclear Density Gauges/X-ray analyzer)	*Atomic Energy Control Act*

12.3 Environmental Baseline Studies

12.3.1 Regional Studies

Detailed environmental surveys of the Stikine-Iskut basin did not occur until the late 1970's when BC Hydro began to study the hydroelectric generating potential of the Stikine and Iskut Rivers. Between 1977 and 1983, hydroelectric power investigations supported a wide range of baseline environmental studies before that project was abandoned. The environmental studies conducted during those seven years formed an important starting point for the field baseline studies being conducted by RTEC on behalf of NovaGold Canada Inc. in support of the Galore Creek Project.

In 1984, the Department of Fisheries and Oceans (DFO) published a catalogue of Pacific salmon spawning locations and escapement estimates for the Stikine and Iskut Rivers (Hancock and Marshall, 1984). It remains the only published report on spawning locations, although escapement data has been compiled every year since then (Etherton, 2004). A report on DFO's sockeye salmon investigations in the Stikine River was published 3 years later (Wood *et al*., 1987). The Pacific Salmon Commission (PSC) is currently responsible for co-ordinating management planning of salmon harvests on the U.S. and Canadian sides of the Stikine Watershed (*e.g*., PSC, 2003). The US National Marine Fisheries Service (NMFS) and the Alaska Department of Fish and Game (ADFG) have also conducted studies of the Stikine estuary.

In 2000, the Cassiar Iskut-Stikine Land and Resource Management Plan (LRMP) was released by the BC Government (MSRM, 2000). The LRMP is a sub-regional, integrated resource plan for land use and resource management. The plan contains summaries of some of the environmental setting of the area.

12.3.2 Project Studies

In the fall of 2003, RTEC conducted a review and gap analysis of existing environmental data collected as part of previous exploration efforts in the area. A subsequent aerial reconnaissance survey of the two potential mine access corridors and the Galore Creek valley itself in the winter of 2004 resulted in the development of a Baseline Studies Field Program Plan for the project. The plan was based on extensive consultations with provincial and federal scientists, regulatory officials and Tahltan Nation representatives.

Studies were subsequently initiated in the following areas:

- Atmospheric Environment
- Hydrology and Water Management
- Aquatic Environment:
  - Surface and Groundwater Quality
  - Sediment Quality
  - Limnology
  - Fish Habitat and Community
- Wetland Resources
- Ecosystem Mapping and Reclamation
- Terrestrial Wildlife
- Metal Leaching/Acid Rock Drainage (Rock Geochemistry)
- Archaeology

Environmental baseline studies have been progressing steadily over the past eighteen months. The following list provides a chronology of important project milestones reached to date:

2003	November	Review and Gap Analysis of Existing Environmental Data
2004	January	Field Reconnaissance of Access Corridors and Mine Site
	February	British Columbia Environmental Assessment Process Initiated
	April	Baseline Study Plan Submitted to Regulatory Agencies
	April	Meetings with Federal and Provincial Regulatory Agencies
	May	Meetings with U.S. Agencies
	May	Initiated Intensive Baseline Study Program
	June	Community Open Houses
	July	Regulatory Agency Site Visit and Tahltan Meetings
	September	Additional Regulatory Agency Site Visit
	November	Development of 2005 Field Program Plan
2005	January	Winter Environmental Baseline Studies
	January	Canadian Environmental Assessment Process Triggered
	January	Environmental Working Group Meetings in Vancouver
	March	Meetings with Canadian and U.S. Regulatory Agencies

April	Continue Environmental Baseline Studies
May/June	Community meetings in Dease Lake and Iskut
June	Adjust Studies to Focus on the Modified North Access Corridor

Figure 12-2 illustrates the initial baseline study area selected in 2004 along with the relevant sampling stations for each study component. In June of 2005, NovaGold Canada Inc. gave direction to Rescan and RTEC to focus remaining baseline assessment on a modified northern alternative: *Highway 37 to the Galore Creek Project Site along More and Sphaler Creeks*.

Remaining baseline studies were therefore re-focussed on the modified northern access option as illustrated in Figure 12-3. Table 12-2 provides a summary of baseline studies completed to date for the Project.

Figure 12-2: 2004 Environmental Baseline Study Area and Sampling Locations

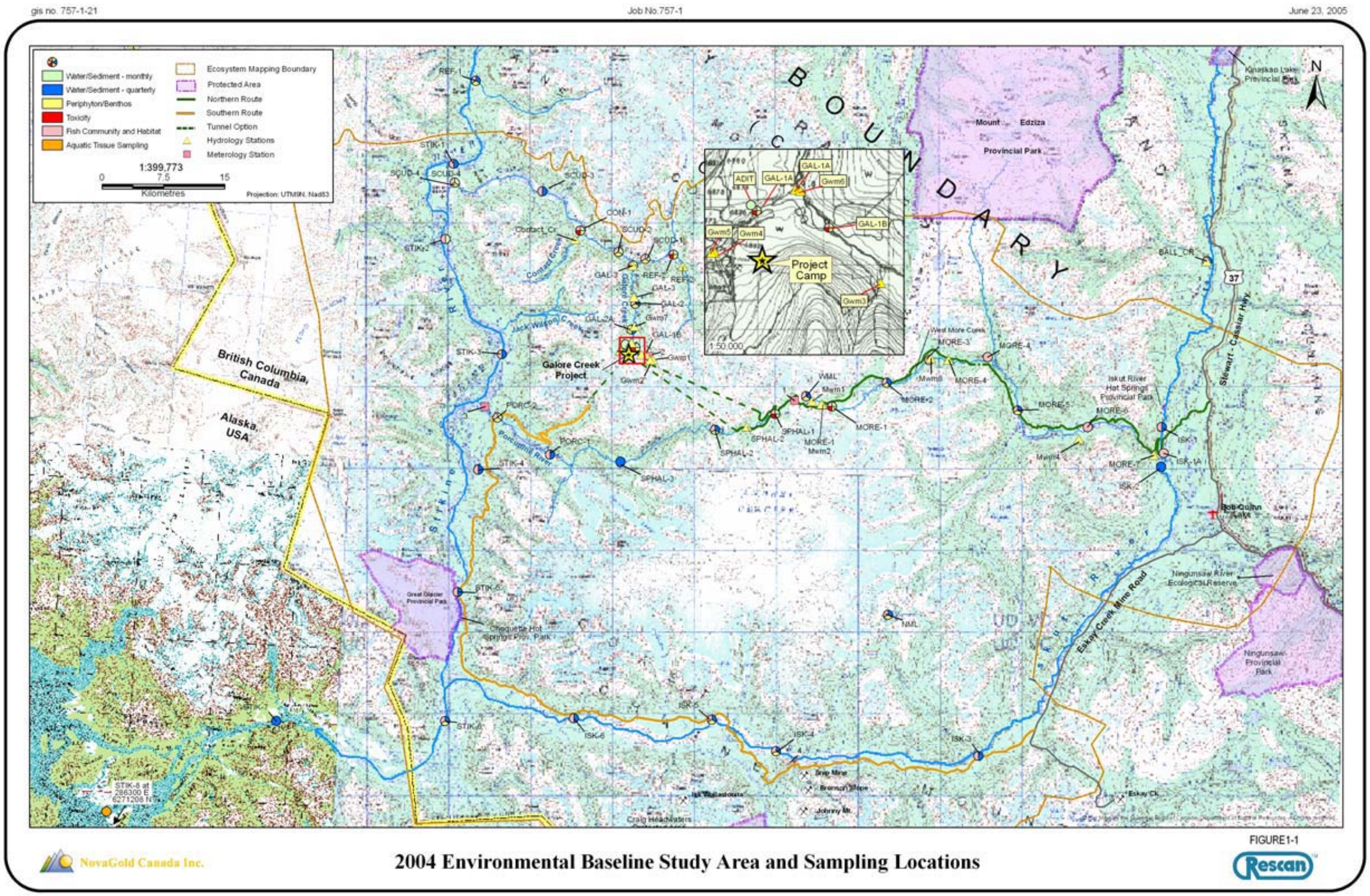


Figure 12-3: Modified Northern Access Alignment

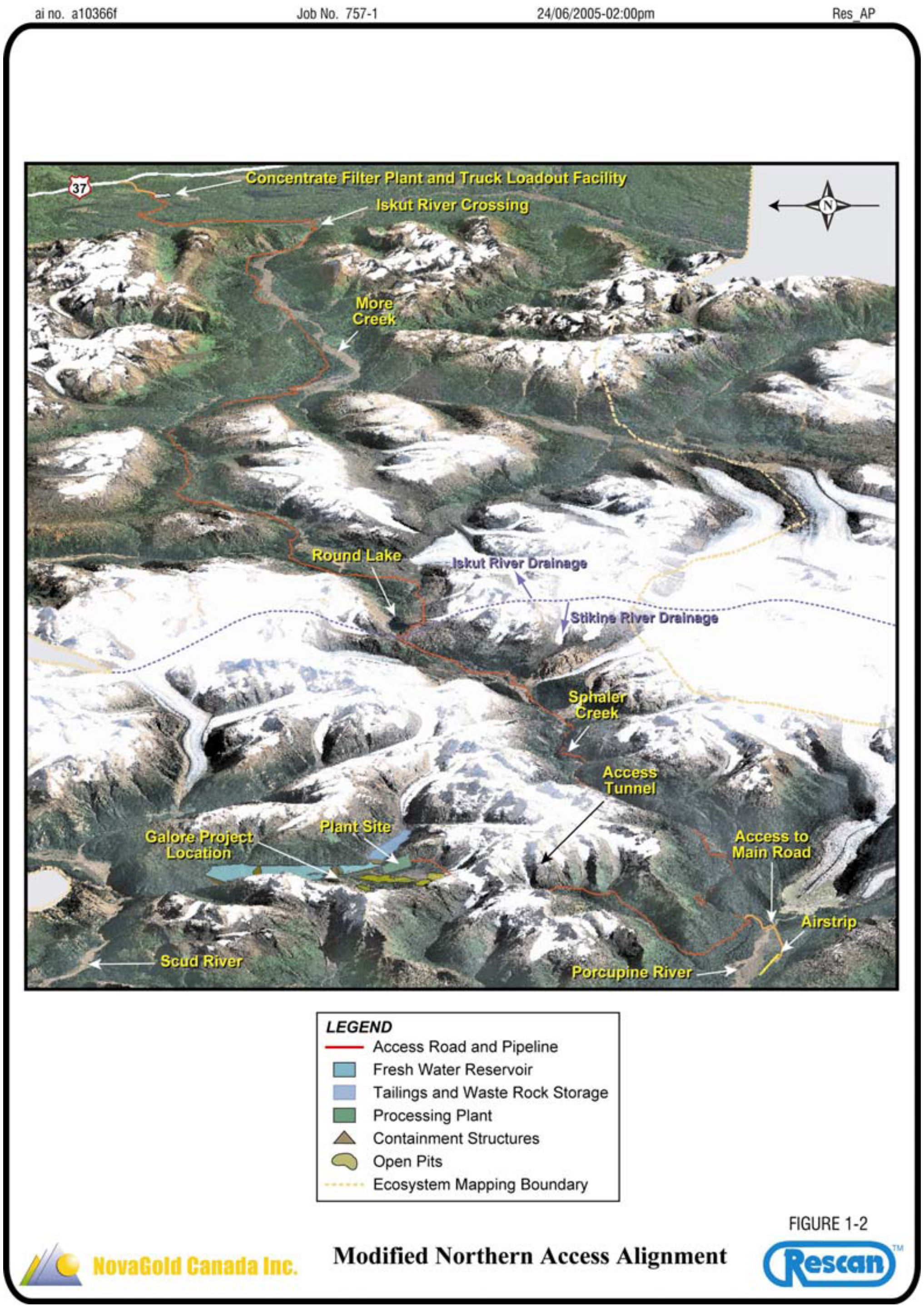
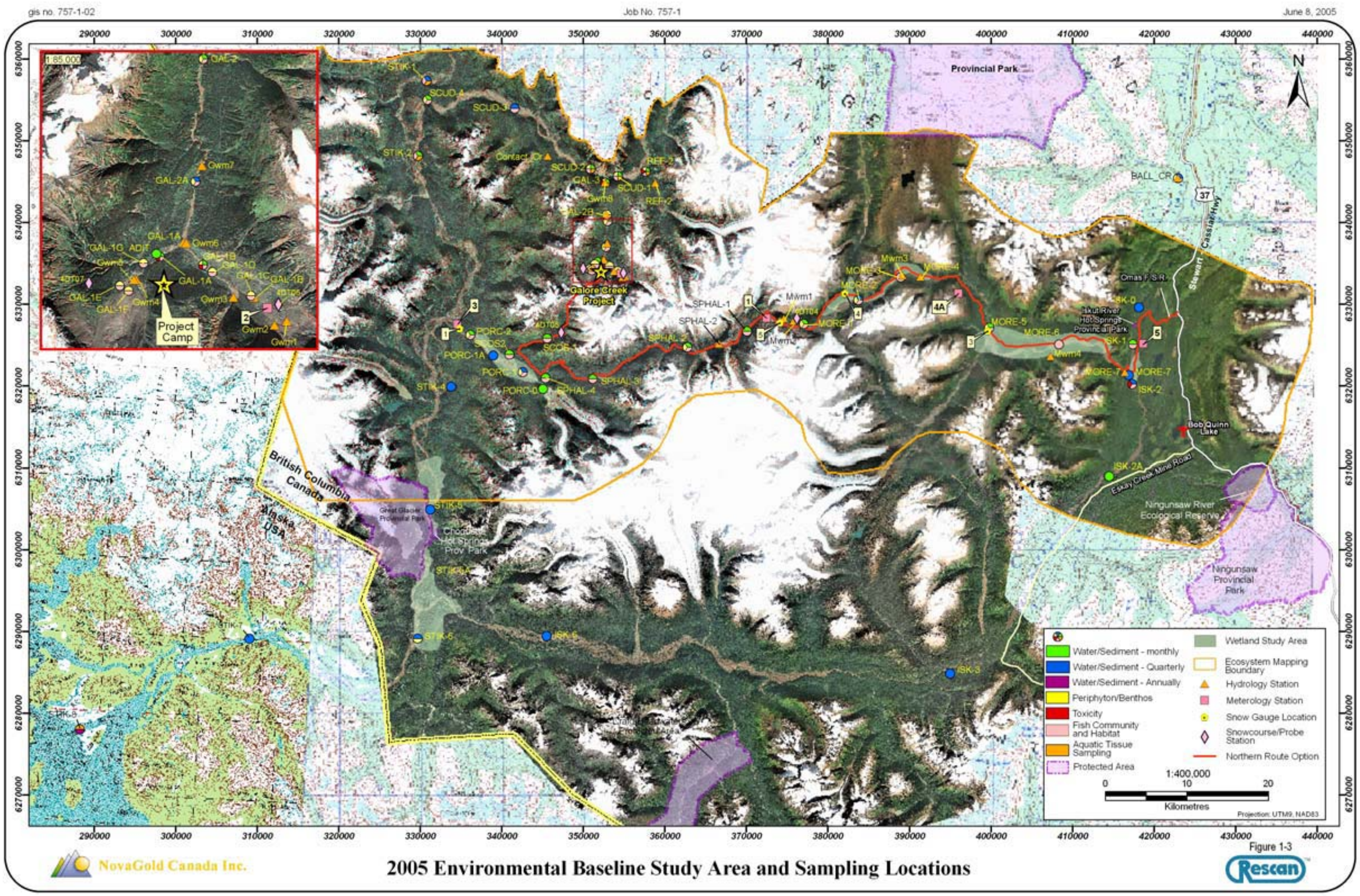


Table 12-2: Summary of Baseline Studies

Study Component	**Status**
Atmospheric Environment	4 fully automated stations established. One each in Galore Valley, confluence of Stikine and Porcupine rivers, Iskut Valley near Bronson Slope and More Valley. Snow surveys were conducted during the first quarter of 2005
Surface Hydrology	11 stations established in Galore Valley, 7 stations in More Valley with an additional 4 stations located within reference and tributary drainages.
Groundwater Quality	4 groundwater quality wells installed, Quarterly sampling on-going
Aquatic Environment	Water and Sediment Quality: 1 annual site, 16 quarterly sites, 20 monthly sites and 6 toxicity sites established. Primary/Secondary Producers: 19 annual sampling locations established. Fish Community and Habitat: 37 bi-annual sampling locations established.
Wetland Assessment	Wetland assessment of northern corridor study area underway.
Soil Survey and Salvage Assessment	Soil surveys initiated along each of the potential mine access corridors and within the proposed mine site (Galore Valley).
Rock Geochemistry Characterization	Phase I geochemical assessment of historic Central and Southwest zone drill core completed. Phase II program completed with over 1,000 samples of 2004 drill core analyzed. Phase III kinetic testing program presently underway.
Vegetation Assessment and Ecosystem Mapping	Vegetation surveys completed along each of the potential mine access corridors and within the proposed mine site (Galore Valley). Ecosystem Mapping using field study and predictive techniques underway.
Terrestrial Wildlife	Raptor and Waterfowl surveys initiated along each of the potential mine access corridors and within the proposed mine site (Galore Valley). Goat, stone's sheep and grizzly bear surveys/habitat suitabilities conducted within study area blocks. Summarized trapping returns for 12 traplines, which at least partially overlap the potential area of influence. Winter moose and goat surveys completed. Riverine bird surveys completed for both access corridors. Spring grizzly bear survey completed.
Archaeology, Heritage and Cultural Assessment	Completed aerial reconnaissance over potential access options and mine site area. Heritage and cultural assessment program underway concurrent with community consultation.

Figure 12-4: 2005 Environmental Baseline Study Area and Sampling Locations



12.3.3 Galore Valley

12.3.3.1 Watershed Hydrology

The Galore Creek valley is subjected to high intensity rain and snowfall events with major occurrences mainly taking place in the spring and fall periods. Annual average precipitation for Galore Creek is approximately 2,200 mm with 65% falling as snow primarily during October through February period.

The hydrological regime of the watersheds in the Galore Creek area is strongly influenced by glaciers and for large watersheds the hydrological year can be divided into four main flow periods:

- Winter – characterized by ice covered streams with low to negligible stream flow
- Spring/freshet – characterized by high flows due to snowmelt. This is often the period which contains the annual peak flow
- Summer – characterized by steady moderate to high flows supplied by glacial melt water. Flows are characterized by diurnal variations reflecting the impact of temperature on glacial melting rates. Rainstorms can result in short-term peaks
- Fall – characterized by generally moderate to low flows, but interrupted by rain fed storm events, which can have peak flows in excess of freshet flows

In the spring of 2004, a series of hydrology stations along with a fully automated climate monitoring station were installed within the Galore Creek valley in support of a larger hydrological modelling exercise. Data from these stations is being continuously gathered and forms the basis for all pre-feasibility level estimates included in this PEA. Data gathered during this season will be used to further develop the water management plan and for feasibility level design of dams and related water conveyance and retaining structures.

12.3.3.2 Aquatic Resources

Recent glacial retreat within the Cordillera of North America has exposed surfical geologic materials, including the headwaters of the Galore Creek valley, to the erosional forces of nature. Large areas previously covered by tonnes of snow and ice are now subjected to precipitation events that mobilize rock and sediments into the aquatic environment.

A series of surface water and groundwater quality monitoring locations have been established within the Galore Creek valley as part of the baseline study program. Samples are collected on a regular basis to establish background water quality for comparison with Provincial as well as Federal water quality guidelines. The background water quality assessment has determined that the levels of some metals such as copper are generally elevated over reference monitoring results from outside the Galore Creek valley. The results from the ongoing water quality monitoring program will be used in feasibility design and in the development of discharge permit limits as part of the Project licencing program.

In addition to water quality sampling, sampling of benthic invertebrates is also undertaken to establish community diversity and provide benchmarks for long-term monitoring of aquatic effects during the mine operation. The locations and frequency of mine operation-phase sampling will also be developed as part of the licensing stage of the Project.

To date, aquatic assessment within Galore Creek has not identified fish presence or use of the system upstream from the confluence with the Scud River. Extremely steep terrain in the lower reaches of Galore Creek generate high velocity flows that are thought to present a natural barrier to fish passage to Galore Creek from the Scud River. Additional assessment is planned in the summer of 2005 to confirm the 2004 assessment results.

Aquatic assessment of the Scud River downstream of Galore Creek confirmed the presence of Coho, Chinook and Dolly Varden. DNA testing of selected samples detected one Bull Trout capture. Bull Trout are a species of conservation concern in British Columbia and are thus Blue listed. They are typically used as an indicator of the health of a watershed as they are widely distributed in the province and known to be sensitive to habitat changes. Additional DNA testing is on-going to determine the presence and extent of Bull Trout within the Scud River system.

12.3.3.3 Terrestrial Wildlife

The Galore Creek valley provides habitat features for a range of terrestrial wildlife including grizzly bear and mountain goat. Terrestrial ecosystem assessment initiated in the spring of 2004 is on-going and includes mapping of habitat types combined with species-directed population studies for both grizzly bear and mountain goat.

Grizzly bear are a species of conservation concern in British Columbia and require special assessment consideration. The population inventory for the Galore Creek project uses non-invasive DNA-based sampling techniques to assess the study area population. The 2004 sampling program detected a total of 119 individuals within the entire study area, which includes both the Galore valley and a large range along the access corridor (Figure 12-2). In addition to vegetation and small mammal browse, grizzly bear in the region depend on adult salmon during the spawning season for a sizable proportion of their summer and fall diet. Given that Galore Creek itself does not directly support salmon, grizzly bear that use the valley are thought to mainly depend on other food sources such as alpine flora and small mammals. The results from additional field studies underway presently will be used to develop a long-term grizzly bear monitoring and protection program as part of the EIA application.

Population estimates of mountain goat have also been undertaken within the Project study area. The summer 2004 survey detected a total of 550 goats. The 2005 winter survey was conducted over a smaller study area with a total of 289 goats observed. A DNA-based program is planned for the Galore Valley in the summer of 2005 in an effort to help monitor the population long-term and aid in impact mitigation planning as part of the EIA.

12.3.4 Mine Access Corridor

The Porcupine River is the only salmon-bearing waterway the northern access corridor would encroach upon. More and Sphaler creeks are each non-salmon bearing. The 2004 environmental field studies identified fish passage barriers on both More and Sphaler creeks indicating that species such as Dolly Varden inhabit the lower reaches of these drainages only. DNA-based studies are on-going to determine any presence of Bull Trout within the waterways adjacent to the modified northern access alignment.

The northern access corridor provides valuable habitat for large mammal species such as grizzly bear and mountain goat with generally higher habitat values associated with the southern corridor. The higher elevation of the northern access corridor distinguishes this area from the previously considered southern option in that there is a larger percentage of alpine environment. Forested coastal systems differ from alpine environments in the timing and extent of available habitat. Alpine habitats generally become important later in the season than lower elevation systems as they take longer to loose their snow and ice covers and begin the process of green-up. The alpine areas of West More Creek corridor provide habitat for grizzly bear in the spring and summer periods. Rugged, high elevation areas such as those at the headwaters of Sphaler Creek are important habitats for mountain goat. Alpine areas are also home to a variety of small mammals that will require consideration during the access construction and operational phases.

12.4 Environmental Management at Galore Creek

12.4.1 Mitigation Strategies

Development and operation of the mine and associated access corridor will involve passage through a range of both aquatic and land-based habitat types. The protection of water and air quality will be maintained through the use of engineering controls such as water treatment and air filtration.

Wildlife species within the Project area that have been identified as sensitive to the Project development include:

1. Grizzly bear;
2. Mountain goat; and
3. Fish (including salmon, Dolly Varden and bull trout-Blue Listed).

Detailed impact mitigation strategies for the Project will be included in the EIA application based upon Provincial management guidelines for species at risk as well as other species that have been identified as sensitive to Project development and operation.

The most significant source of potential adverse environmental impact from Project development and operation is from un-authorized access to the Project area. Without access restrictions in place, road developments through pristine areas have been shown to have measurable adverse impacts to both big game and fish populations. For safety and wildlife protection reasons, NovaGold Canada Inc. will restrict entry to the mine access road during both the construction and operational phases of the Project. A gatehouse will be installed near the entrance of the alignment and will be staffed by an access control officer at all times.

The purposes of the access control point are:

1. Allow passage of authorized haul and supply trucks.
2. Perform spot checks of vehicles for the presence of restricted materials such as firearms and illegal drugs.
3. Communicate access management issues to the mine manager and support staff as required.

12.5 Waste and Ore Geochemistry Characterization

12.5.1 Metal Leaching and ARD Potential

The historical and recent phases of ML/ARD testing on rock samples have shown consistent results as the size of the sampling programs has increased and work incorporated in this PEA is consistent with pre-feasibility level estimation of costs and design criteria. Further testwork is underway in 2005 as part of the feasibility data collection program. In the Central, West Fork and Southwest Zones, sulphide sulphur concentrations (representing acid potential) are typically less than 1% but vary up to near 4%. Sulphate sulphur in the form of gypsum is ubiquitous at Galore Creek.

All rock types in all five zones contain significant acid-buffering capacity as carbonate minerals such that the potential for net acid generation varies. In all except the Southwest Zone, screening level assessment of the Phase 2 static test results – used in pre-feasibility design and cost estimation - showed that a least 50% of samples would be classified as not potentially acid generating (not PAG), and less than 30% were classified as potentially acid generating (PAG). The balance of samples (typically to 20 to 25%) were classified as have uncertain potential for net acid generation. In the Southwest Zone, 32% of samples were classified as not PAG and 50% were classified as PAG samples (11%).

Testing for other parameters indicated elevated concentrations of several elements known to be associated with specific minerals in the deposit, including copper (chalcopyrite), zinc (sphalerite), lead (galena) and fluorine (fluorite).

Initial results from kinetic tests have shown that the most parameters leach at low rates. Copper, cadmium, fluoride, manganese, selenium, sulphate and zinc were leached at concentrations greater than typical water quality criteria.

Static testing of tailings has shown that tailings from processing of different types of ores may vary from PAG to not-PAG, for example, tailings from bornite ores are expected to be not-PAG whereas tailings from pyritic ores are expected to be PAG. Separate testing of rougher tailings indicates that they are expected to be not-PAG whereas cleaner tailings are expected to be PAG.

12.5.2 Implications to Waste Management

The static test results have shown that components of the waste rock (and possibly by extension parts of the pit walls) are PAG and will need to be managed to be address potential for release of ARD. Initial assessment of the spatial distribution of potential for acid generation has shown that PAG rock occurs in large blocks in well-defined areas. It is therefore possible that waste rock can be segregated based on acid generation potential. Subaqueous disposal is being assumed for PAG waste rock to address acid generation potential as part of the pre-feasibility design and cost estimation process. Subaqueous disposal will also address elements released from sulphide minerals under non-acidic conditions (including cadmium, selenium and zinc) and low level release of metals such as copper.

Due to the presence of elevated levels of carbonate minerals, net acid generation in PAG materials will be delayed. Based on experience, this delay may be of the order of decades, and will allow time for management of PAG materials to prevent the onset of ARD.

The presence of elevated levels of sulphate-sulphur and fluorine indicates that these parameters will probably leach at levels controlled by the dissolution of gypsum and fluorite respectively. Subaqueous disposal will not control leaching of these minerals. Leaching of these elements will occur under both aerial and subaqueous conditions.

Characterization of tailings indicates that it may be necessary to manage whole tailings to prevent acid generation and this has formed the basis of the pre-feasibility level design incorporated into this PEA. The distinctive characteristics of the rougher and cleaner tailings indicate that only the pyritic PAG cleaner tailings would need subaqueous disposal to address ARD potential. Rougher tailings are expected to have lower reactivity and could be disposed sub-aerially.

12.5.3 Further Investigations

Ongoing investigations as part of the 2005 feasibility work program include kinetic testing, mineralogical characterization and characterization of the deposit gossan to understand longer term weathering. These studies are designed to provide input to waste management planning and water quality predictions for impact assessment.

12.6 Mine Reclamation and Closure

Mine development and operation at Galore Creek will incorporate techniques to both minimize surficial disturbance as well as progressively reclaim mining-impacted areas. Progressive reclamation is the cornerstone to both maintaining a healthy environment and reducing closure-related capital costs at the cessation of mining activities. The reclamation plan for Galore Creek will be designed to ensure that the operational and post-closure phases of the project are compatible with the surrounding natural environment.

There are three main goals of the reclamation program at Galore Creek:

1. Provision of Stable Landforms
2. Re-establishment of Productive Land Use
3. Protection of Aquatic Resources

Design of the open pits, containment dams, waste rock and tailings management areas has been undertaken as part of this pre-feasibility level examination to ensure long-term stability both before and after mine closure. Post-closure landforms will be contoured so as to enhance wildlife use of mining areas long after mine closure.

A wide range of terrestrial and aquatic wildlife species depend on the Stikine River and associated valleys to support their life cycles. Galore Creek itself is one of dozens of valleys located within the lower Stikine system supporting terrestrial mammals such as grizzly bear and mountain goat among others. Historic hunting and trapping in the region by Tahltan First Nation peoples is also an important component when considering area land use and these needs will be incorporated into the closure plan being developed with input from the Tahltan. .

Mine operations will ensure that the receiving aquatic environment is protected on an on-going basis both during mine operations as well as post-closure. Concurrent with the initiation of mining, an aquatic effects monitoring program will be implemented to measure any potential changes in the downstream environment.

12.7 Socio-Economics

Development of the Galore Creek Copper Gold Project will have significant beneficial impacts on the economic development of the northwestern region of British Columbia in general and the community of Stewart in particular. In conjunction with the British Columbia Environmental Assessment Office (BCEAO) and the Tahltan First Nation, NovaGold initiated a public information and consultation program within the local communities at the early stages of Project planning and this process is well established and ongoing. The company has made a substantial commitment to working with residents of the region to ensure that the effects of the Project are positive leading to sustainable long term benefits.

NovaGold Canada also entered into consultations with the Tahltan First Nation at the early stages of the project in the fall of 2003. A draft Tahltan-NovaGold Participation Agreement is currently being reviewed by the parties. NovaGold continues to work closely with the Tahltan on an on-going basis as project engineering and environmental assessment studies progress. NovaGold has committed to a training program with the Tahltan community that will ultimately lead to increased skill capacity within the community and facilitate long term benefits to both the project and Band members.

12.7.1 Employment and Population Effects

The primary social and economic effects of the Project will result from the direct and indirect employment opportunities created by development of the project. Development of the Galore Creek Copper Gold Project would result in the expenditure of about US$1100M, which will lead to employment opportunities directly and indirectly for people in the local communities as well as throughout the Province.

Once operational, the Galore Creek Project will employ an average of about 500 to 600 workers in full-time, permanent positions. It is also estimated that requirements for supplies, services and demands by NovaGold employees for housing, food, clothing, and other consumer goods and services as well as government services will create additional job opportunities in the region.

The Galore Creek Copper Gold Project will provide long-term, steady employment to many area residents and it offers some measure of diversification for the local economy. Economic activity generated by the Project and employees will increase the revenues of local governments and increase levels of government services, social amenities, and commercial services. This general economic development will lead to general improvement in the standard of living for all area residents.

12.7.2 Economic Effects

The economic effect of the increased incomes resulting from the Project will be significant to the region. The provincial government will also enjoy significant tax revenue increases from the development of the Galore Creek Copper Gold deposit. It is estimated that the project will, at long term metal prices of US$1.00/lb copper and US400/oz gold generate about US$560M in income taxes to the Federal and Provincial Governments over the life of the project, approximately US$20M to US$30M annually in direct and indirect Federal and Provincial tax payments and several million dollars annually in local municipal tax payments with the Municipality of Stewart, the Kitimat-Stikine Regional District, the Cassiar and Stewart School districts, and the Regional Hospital District being the largest beneficiaries.

12.7.3 Impact Management

Based on the projected settlement patterns of immigrant workers, it is anticipated that all regional communities will be able to accommodate new residents. Stewart has ample infrastructure capacity for the projected increase. There is also potential for increased assimilative capacity within the communities of Iskut, Telegraph Creek and Dease Lake.

The Galore Creek Project does not represent a short-term expansion for the region. Copper and gold resources at Galore Creek can potentially support operations far beyond the present planning period. NovaGold is confident that with well considered socio-economic and human resource policies and plans, the Galore Creek Project can make a very positive, long-term, stabilizing contribution to the social and economic well being of the region.

13. Project Implementation Schedule

Development of the Galore mine presents some significant construction and logistical challenges, largely due to the lack of access to the area. The critical steps for the development of the project are:

- Progression of project evaluation to Feasibility Study level, and successful project financing.
- Commencing engineering activities early, so that construction contracts and long delivery items are available at the necessary time without delaying construction.
- Construction of the access road, bridges and tunnels to the minesite. Powerline construction will lag road construction.
- Construction of the mine, plant and site facilities.
- Prestripping of the mine and the use of this material for construction of the tailings and waste rock dam, water reservoir and water diversion ditches.

A number of discussions were held with road and tunnel contractors to better determine the likely approach and time for construction, as this component is the key driver of the construction schedule. This is discussed below. The detailed development schedule for the project is shown in Figure 13-1.

The access road, tunnel and tailings dam are the critical path activities. The general approach that has been developed is summarized below.

- The 4 km tunnel into the Galore Valley, which forms the permanent access to the site, will be constructed based on air support for both the north and south headings. The south portal will be developed and driving will commence as soon as the project is approved. The north portal access requires installation of protection from avalanche and therefore it may be necessary to install a short section of tunnel culvert so that the tunnel program can be established as soon as possible. It is anticipated the tunnel will require 388 days to establish, constructed from both headings.
- Airstrips will be developed at both Porcupine River and at Round Lake as soon as practical after project approval. Both of these development operations will be based on air support.
- A glacier crossing route could possibly be developed from the south east of the Galore Valley, to be used as a construction access or temporary tunnel bypass route to protect the construction schedule should delays occur on the tunnel driving operation, and to support limited schedule sensitive construction activities.
- The initial road development from the Bob Quinn area will commence as soon as practical in 2007 and the road will be pushed through as a pioneer trail as fast as possible to Round Lake.
- Road headings will be started from Round Lake towards Bob Quinn and from Porcupine Creek to the south tunnel portal using the limited construction equipment that can be transported and supported via the air operations. Once the road access to the tunnel portal has been completed, road construction will continue up Sphaler Creek towards the Round Lake base.
- Installation of the pipeline pumping stations will commence in the spring of 2008. Pipeline installation will be completed in 2009 with the closing section through the tunnel being the final section in mid-summer.

- The power transmission lines to site will be installed along with the road construction operation to provide power to a location adjacent to the south tunnel portal by the end of 2007.
- Completion of the tie-in to the construction substation will coincide with the completion of the tunnel construction.
- Temporary power may have to be generated at Bob Quinn until grid power is available. Tie-in to the substation at Bob Quinn with permanent power is anticipated in late 2008.

The critical path flows through the construction of the access road and tunnel to access Galore Valley, for construction of the tailings/waste dam and water diversion ditches.

13.1 Mine Development

Once the main access road and the tunnel are established, major equipment and construction equipment will be mobilized to the site. Generally the mine development approach will be managing water flowing into the valley followed by facility installations. This means that the water reservoir and the diversion ditching will take first priority. The generalized approach to construction is as follows:

- Commence scaling operation and key installation for the water reservoir as soon as possible in 2008.
- Development of the borrow pits including local diversion of water away from the borrow areas during the winter and early spring of 2008/2009.
- Construct the water reservoir dam.
- Install diversion ditches on both the west side and east side of the valley to minimize the water flowing into the area in which the dams are to be constructed. This will be accomplished through the winter of 2008/2009 to facilitate spring start for the mine development.
- Develop the access road from the mine site to the north portal for the tunnel alignment as soon as equipment is available.
- Strip the area, scale as required, and cut the key for the tailings dam installation in the spring of 2009.
- Develop the plant site platform area in 2008/2009.
- Install foundations for mill concentrators and flotation cells (onsites).
- Install foundations for pump houses and slurry prep area (offsites).
- Install buildings and structures 2008/2009.
- Install substation 2009.
- Install mills crushers and other equipment completing end of 2009.
- Install piping and complete installation in 2010.

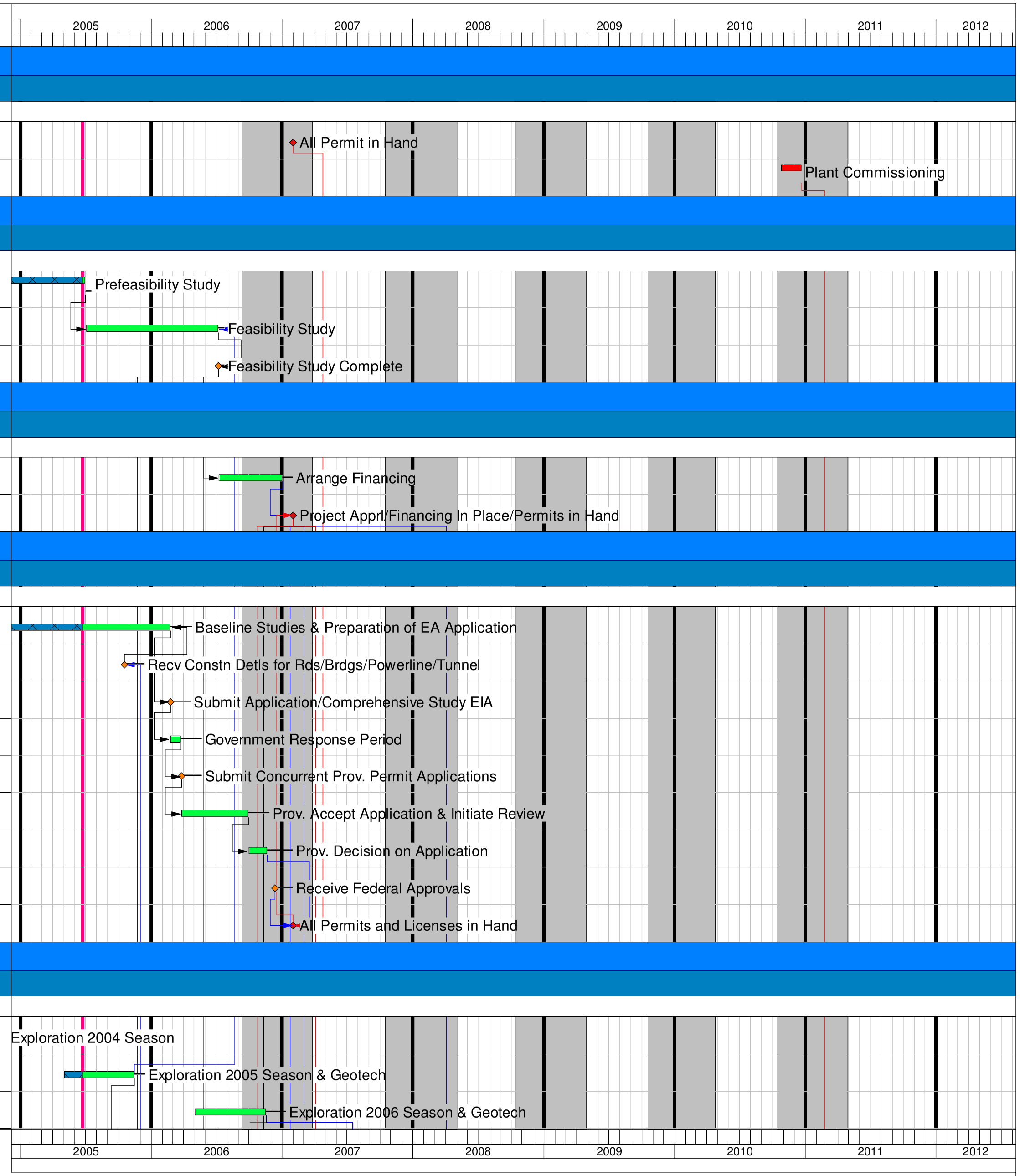
A more detailed description of the mine pre-stripping and construction of the tailings/waste and water dam structures is presented in Section 4.

13.2 Opportunities

A number of opportunities were identified, that require further evaluation, could provide improvements in the construction schedule. These are approaches that would provide additional risk management benefits, cost, and/or schedule benefits.

- Development of a conveying system for the placement of the ballast on the tailings dam. This would involve creating borrow pits at an elevation that would allow for downhill transportation of ballast materials over conveyors and thus reduce the overall requirement for trucks.
- The early provision of power to the site would displace the requirement for diesel fuel and a number the heavy power users and the construction camp could then be electric driven/supported. Installation of a temporary power, should this be acceptable by the permitting authority, would provide displacement of a significant amount of fuel during the early construction activities and speed the construction build-up.

Act ID	Activity Description	Orig Dur	Early Start	Early Finish
MILESTONES				
01	All Permit in Hand	0		31JAN07
02	Plant Commissioning	40	25OCT10	20DEC10
FEASIBILITY STUDIES				
20	Prefeasibility Study	210	06SEP04 A	30JUN05
25	Feasibility Study	254	05JUL05	07JUL06
30	Feasibility Study Complete	0		07JUL06
OWNER				
35	Arrange Financing	120	10JUL06	02JAN07
40	Project Apprl/Financing In Place/Permits in Hand	0		31JAN07
PERMITTING/ENVIRONMENTAL				
50	Baseline Studies & Preparation of EA Application	199	02FEB04 A	24FEB06
55	Recv Constn Detls for Rds/Brdgs/Powerline/Tunnel	0		18OCT05
60	Submit Application/Comprehensive Study EIA	0		23FEB06
65	Government Response Period	21	24FEB06	24MAR06
75	Submit Concurrent Prov. Permit Applications	0	27MAR06	
85	Prov. Accept Application & Initiate Review	129	27MAR06	29SEP06
95	Prov. Decision on Application	53	30SEP06	21NOV06
100	Receive Federal Approvals	0		12DEC06 *
110	All Permits and Licenses in Hand	0		31JAN07 *
EXPLORATION				
120	Exploration 2004 Season	100	04MAY04 A	26NOV04 A
125	Exploration 2005 Season & Geotech	133	04MAY05 A	15NOV05
130	Exploration 2006 Season & Geotech	137	03MAY06 *	17NOV06



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NOVAGOLD
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DETAILED SCHEDULE

HATCH
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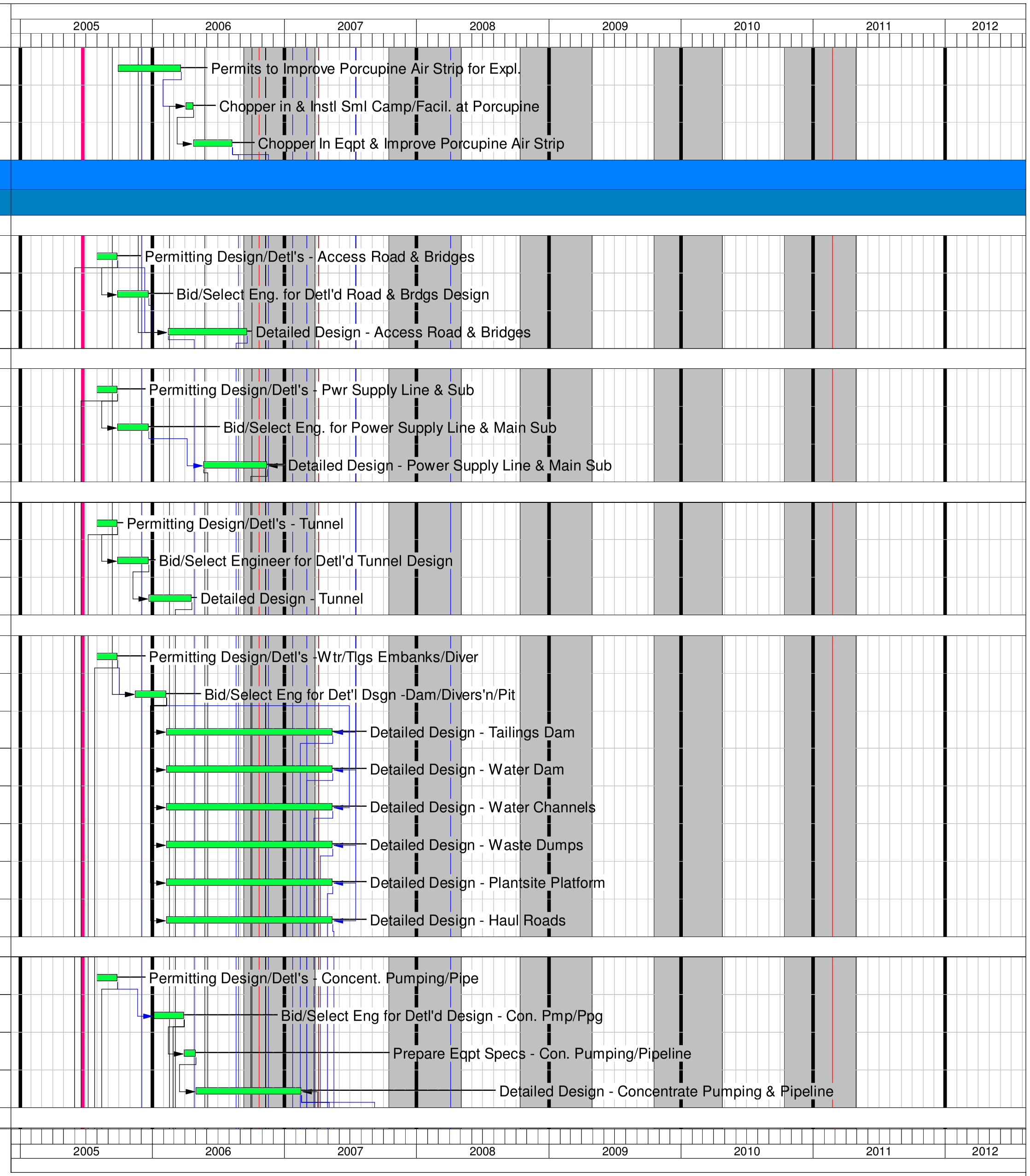
Act ID	Activity Description	Orig Dur	Early Start	Early Finish
135	Permits to Improve Porcupine Air Strip for Expl.	121	29SEP05 *	22MAR06
136	Chopper in & Instl Sml Camp/Facil. at Porcupine	21	04APR06 *	24APR06
137	Chopper In Eqpt & Improve Porcupine Air Strip	108	25APR06	10AUG06
ENGINEERING				
Roads, Brdgs, Tunnel, Pipe/Pwrlines, Embankmnts				
Access Road & Bridge Design				
160	Permitting Design/Detl's - Access Road & Bridges	40	02AUG05 *	27SEP05
162	Bid/Select Eng. for Detl'd Road & Brdgs Design	60	28SEP05	22DEC05
165	Detailed Design - Access Road & Bridges	150	14FEB06	19SEP06
Power Line Design				
170	Permitting Design/Detl's - Pwr Supply Line & Sub	40	02AUG05 *	27SEP05
172	Bid/Select Eng. for Power Supply Line & Main Sub	60	28SEP05	22DEC05
175	Detailed Design - Power Supply Line & Main Sub	121	23MAY06 *	14NOV06
Tunnel Design				
180	Permitting Design/Detl's - Tunnel	40	02AUG05 *	27SEP05
182	Bid/Select Engineer for Detl'd Tunnel Design	60	28SEP05	22DEC05
185	Detailed Design - Tunnel	80	23DEC05	19APR06
Major Earthworks Design (Dams/Diversions/Mining)				
190	Permitting Design/Detl's -Wtr/Tlgs Embanks/Diver	40	02AUG05 *	27SEP05
192	Bid/Select Eng for Det'l Dsgn -Dam/Divers'n/Pit	60	16NOV05	09FEB06
195	Detailed Design - Tailings Dam	314	10FEB06	14MAY07
225	Detailed Design - Water Dam	314	10FEB06	14MAY07
235	Detailed Design - Water Channels	314	10FEB06	14MAY07
245	Detailed Design - Waste Dumps	314	10FEB06	14MAY07
255	Detailed Design - Plantsite Platform	314	10FEB06	14MAY07
265	Detailed Design - Haul Roads	314	10FEB06	14MAY07
Concentrate Pumping & Pipeline				
200	Permitting Design/Detl's - Concent. Pumping/Pipe	40	02AUG05 *	27SEP05
205	Bid/Select Eng for Detl'd Design - Con. Pmp/Ppg	60	06JAN06 *	30MAR06
210	Prepare Eqpt Specs - Con. Pumping/Pipeline	20	31MAR06	01MAY06
212	Detailed Design - Concentrate Pumping & Pipeline	200	02MAY06	16FEB07



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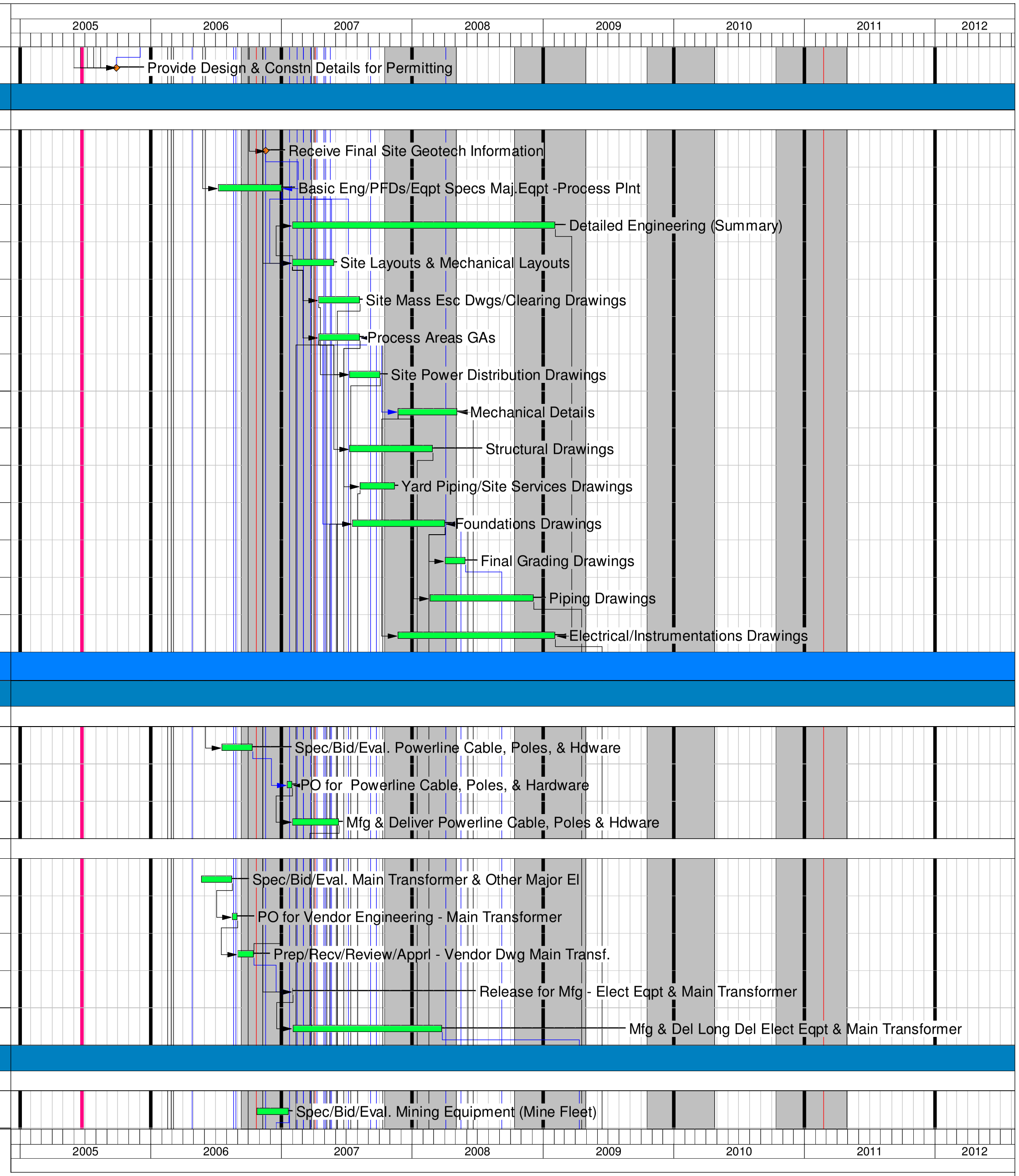
Act ID	Activity Description	Orig Dur	Early Start	Early Finish
215	Provide Design & Constn Details for Permitting	0		27SEP05
Process Plant				
300	Receive Final Site Geotech Information	0		17NOV06
305	Basic Eng/PFDs/Eqpt Specs Maj.Eqpt -Process Plnt	120	10JUL06	02JAN07
310	Detailed Engineering (Summary)	503 *	01FEB07	04FEB09
315	Site Layouts & Mechanical Layouts	80	01FEB07	28MAY07
320	Site Mass Esc Dwgs/Clearing Drawings	80	16APR07	08AUG07
325	Process Areas GAs	80	16APR07	08AUG07
330	Site Power Distribution Drawings	60	11JUL07	04OCT07
335	Mechanical Details	112	23NOV07	05MAY08
340	Structural Drawings	160	11JUL07	28FEB08
345	Yard Piping/Site Services Drawings	66	09AUG07	13NOV07
350	Foundations Drawings	175	19JUL07	01APR08
355	Final Grading Drawings	40	02APR08	28MAY08
360	Piping Drawings	200	20FEB08	05DEC08
365	Electrical/Instrumentations Drawings	300	23NOV07	04FEB09
PROCUREMENT - CAPTIAL EQUIPMENT				
Main Power Supply Line & Substation				
Overhead Power Supply Line				
366	Spec/Bid/Eval. Powerline Cable, Poles, & Hdware	60	19JUL06	13OCT06
368	PO for Powerline Cable, Poles, & Hardware	10	18JAN07	31JAN07
369	Mfg & Deliver Powerline Cable, Poles & Hdware	90	01FEB07	11JUN07
Main Substation				
370	Spec/Bid/Eval. Main Transformer & Other Major El	60	24MAY06	17AUG06
375	PO for Vendor Engineering - Main Transformer	10	18AUG06	31AUG06
377	Prep/Recv/Review/Apprl - Vendor Dwg Main Transf.	30	01SEP06	16OCT06
380	Release for Mfg - Elect Eqpt & Main Transformer	1	01FEB07	01FEB07
390	Mfg & Del Long Del Elect Eqpt & Main Transformer	417	02FEB07	24MAR08
Pre-Production Mining & Site Development				
Mining Equipment				
400	Spec/Bid/Eval. Mining Equipment (Mine Fleet)	60	25OCT06 *	22JAN07



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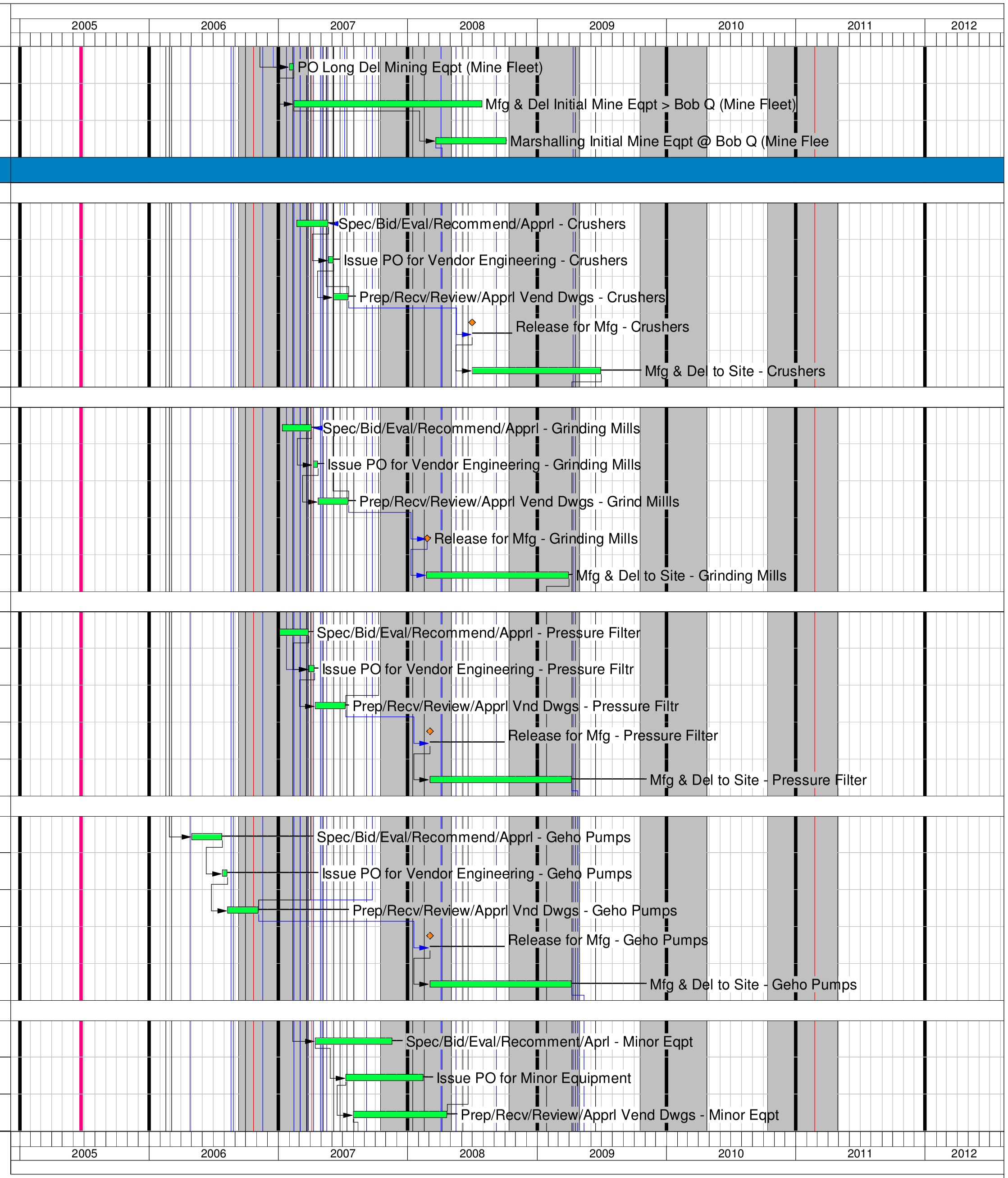
Act ID	Activity Description	Orig Dur	Early Start	Early Finish
405	PO Long Del Mining Eqpt (Mine Fleet)	10	01FEB07	14FEB07
410	Mfg & Del Initial Mine Eqpt > Bob Q (Mine Fleet)	531	15FEB07	29JUL08
415	Marshalling Initial Mine Eqpt @ Bob Q (Mine Flee	200	21MAR08	06OCT08
Process Plant				
Gyratory/Cone Crushers				
440	Spec/Bid/Eval/Recommend/Appri - Crushers	60	23FEB07	22MAY07
445	Issue PO for Vendor Engineering - Crushers	10	23MAY07	05JUN07
450	Prep/Recv/Review/Appri Vend Dwgs - Crushers	30	06JUN07	18JUL07
455	Release for Mfg - Crushers	0	02JUL08	
460	Mfg & Del to Site - Crushers	365	01JUL08	30JUN09
Grinding Mills				
480	Spec/Bid/Eval/Recommend/Appri - Grinding Mills	60	12JAN07	05APR07
485	Issue PO for Vendor Engineering - Grinding Mills	10	10APR07	23APR07
490	Prep/Recv/Review/Appri Vend Dwgs - Grind Millls	60	24APR07	18JUL07
495	Release for Mfg - Grinding Mills	0	25FEB08	
498	Mfg & Del to Site - Grinding Mills	401	24FEB08	30MAR09
Pressure Filter				
500	Spec/Bid/Eval/Recommend/Appri - Pressure Filter	60	04JAN07	28MAR07
505	Issue PO for Vendor Engineering - Pressure Filtr	10	29MAR07	13APR07
510	Prep/Recv/Review/Appri Vnd Dwgs - Pressure Filtr	60	16APR07	10JUL07
515	Release for Mfg - Pressure Filter	0		03MAR08 *
520	Mfg & Del to Site - Pressure Filter	401	04MAR08	08APR09
Positive Displacement Pumps (Gehos)				
530	Spec/Bid/Eval/Recommend/Appri - Geho Pumps	60	02MAY06	26JUL06
535	Issue PO for Vendor Engineering - Geho Pumps	10	27JUL06	10AUG06
540	Prep/Recv/Review/Appri Vnd Dwgs - Geho Pumps	60	11AUG06	06NOV06
545	Release for Mfg - Geho Pumps	0		03MAR08 *
550	Mfg & Del to Site - Geho Pumps	401	04MAR08	08APR09
Minor Equipment				
688	Spec/Bid/Eval/Recomment/Aprl - Minor Eqpt	150	16APR07	19NOV07
698	Issue PO for Minor Equipment	150	11JUL07	14FEB08
708	Prep/Recv/Review/Appri Vend Dwgs - Minor Eqpt	180	01AUG07	21APR08



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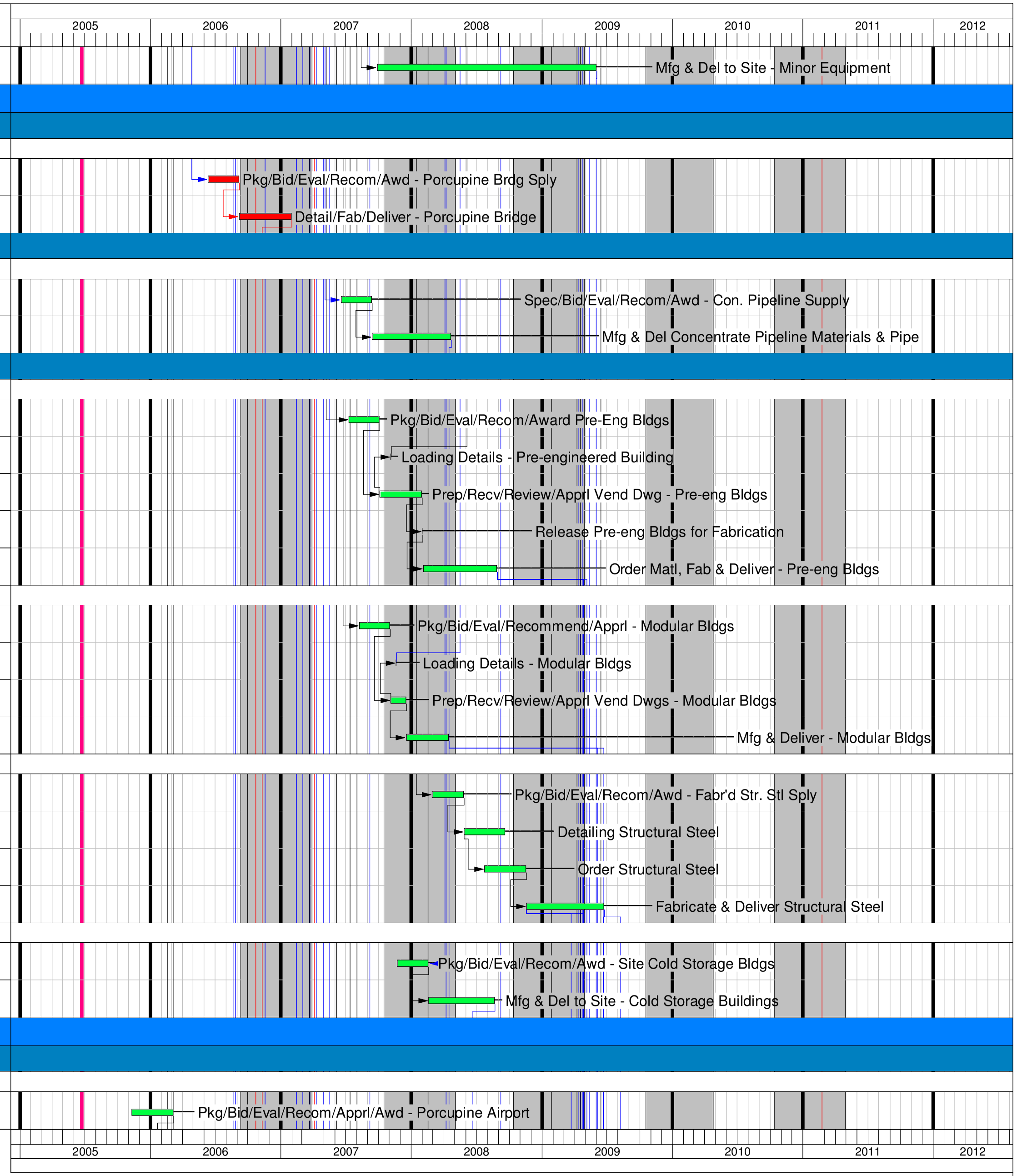
Act ID	Activity Description	Orig Dur	Early Start	Early Finish
718	Mfg & Del to Site - Minor Equipment	421	28SEP07	03JUN09
PROCUREMENT - FABRICATED ITEMS				
Access Road				
Porcupine Creek Bridge				
740	Pkg/Bid/Eval/Recom/Awd - Porcupine Brdg Sply	60	12JUN06	06SEP06
745	Detail/Fab/Deliver - Porcupine Bridge	100	07SEP06	31JAN07
Concentrate Pumping & Pipeline				
Concentrate Pumping & Pipeline				
746	Spec/Bid/Eval/Recom/Awd - Con. Pipeline Supply	60	19JUN07 *	13SEP07
747	Mfg & Del Concentrate Pipeline Materials & Pipe	150	14SEP07	21APR08
Process Plant				
Pre-Engineered Buildings Supply				
750	Pkg/Bid/Eval/Recom/Award Pre-Eng Bldgs	60	11JUL07	04OCT07
760	Loading Details - Pre-engineered Building	1	05NOV07	05NOV07
770	Prep/Recv/Review/Apprl Vend Dwg - Pre-eng Bldgs	80	05OCT07	31JAN08
775	Release Pre-eng Bldgs for Fabrication	1	01FEB08	01FEB08
780	Order Matl, Fab & Deliver - Pre-eng Bldgs	143	04FEB08	27AUG08
Modular Buildings Supply				
830	Pkg/Bid/Eval/Recommend/Apprl - Modular Bldgs	60	09AUG07	02NOV07
840	Loading Details - Modular Bldgs	1	20NOV07	20NOV07
850	Prep/Recv/Review/Apprl Vend Dwgs - Modular Bldgs	30	05NOV07	17DEC07
860	Mfg & Deliver - Modular Bldgs	80	18DEC07	14APR08
Fabricated Structural Steel Supply				
790	Pkg/Bid/Eval/Recom/Awd - Fabr'd Str. Stl Sply	60	29FEB08	27MAY08
800	Detailing Structural Steel	80	28MAY08	19SEP08
810	Order Structural Steel	80	24JUL08	18NOV08
820	Fabricate & Deliver Structural Steel	150	19NOV08	23JUN09
Cold Storage Buildings				
870	Pkg/Bid/Eval/Recom/Awd - Site Cold Storage Bldgs	60	22NOV07	18FEB08
880	Mfg & Del to Site - Cold Storage Buildings	129	19FEB08	22AUG08
PROCUREMENT - CONSTRUCTION CONTRACTS				
Access Road				
Procupine Airport Improvement Contract				
900	Pkg/Bid/Eval/Recom/Apprl/Awd - Porcupine Airport	80	10NOV05	06MAR06



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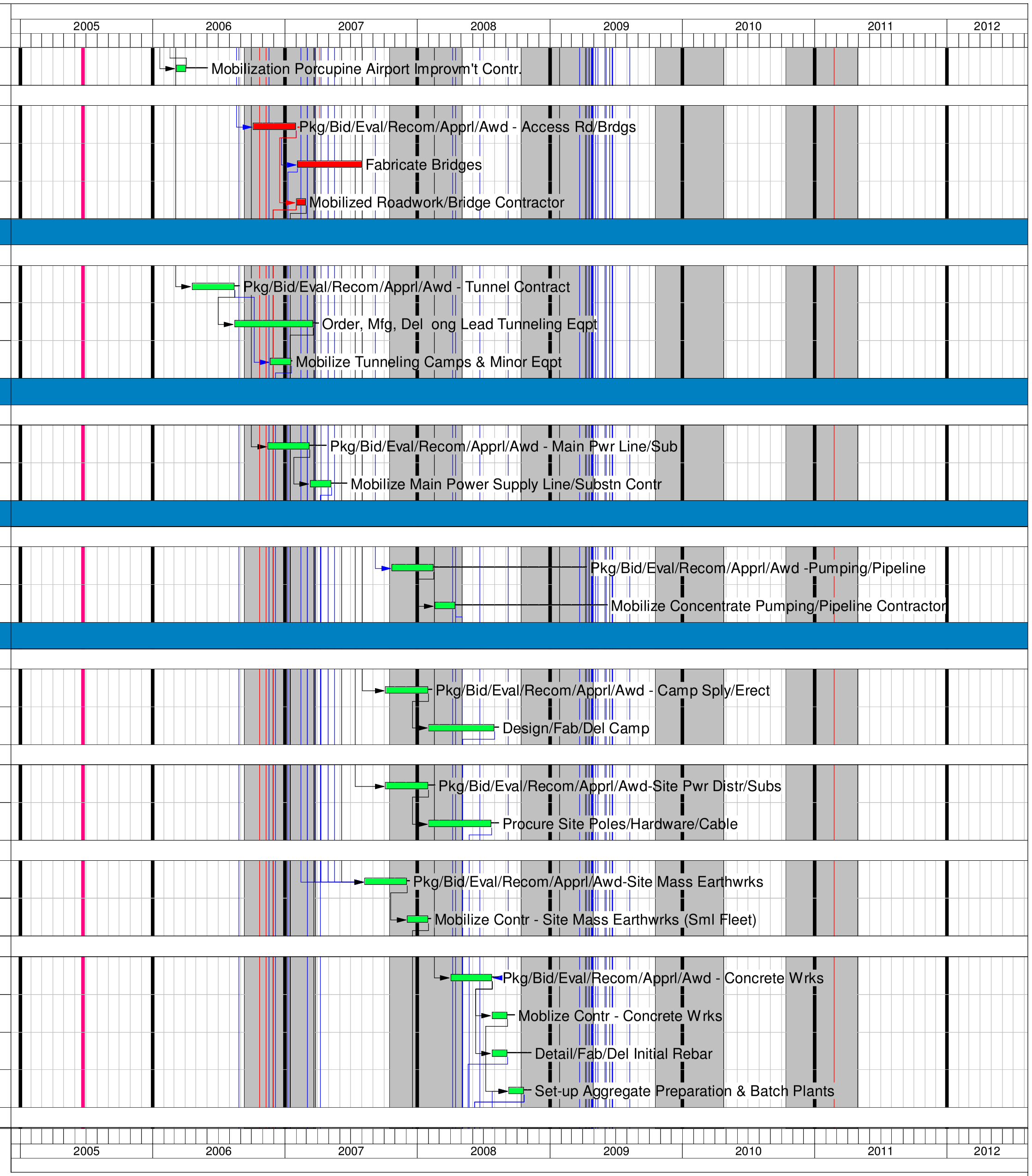
Act ID	Activity Description	Orig Dur	Early Start	Early Finish
905	Mobilization Porcupine Airport Improvm't Contr.	20	07MAR06	03APR06
Access Road & Bridge Constr. Contract				
910	Pkg/Bid/Eval/Recom/Apprl/Awd - Access Rd/Brdgs	80	05OCT06	31JAN07
915	Fabricate Bridges	125	05FEB07	02AUG07
920	Mobilized Roadwork/Bridge Contractor	20	01FEB07	28FEB07
Tunnel				
930	Pkg/Bid/Eval/Recom/Apprl/Awd - Tunnel Contract	80	20APR06	14AUG06
935	Order, Mfg, Del ong Lead Tunneling Eqpt	150	15AUG06	20MAR07
940	Mobilize Tunneling Camps & Minor Eqpt	40	22NOV06	19JAN07
Main Power Supply Line & Substation				
960	Pkg/Bid/Eval/Recom/Apprl/Awd - Main Pwr Line/Sub	80	15NOV06	09MAR07
970	Mobilize Main Power Supply Line/Substn Contr	40	12MAR07	08MAY07
Concentrate Pumping & Pipeline				
980	Pkg/Bid/Eval/Recom/Apprl/Awd -Pumping/Pipeline	80	23OCT07 *	15FEB08
990	Mobilize Concentrate Pumping/Pipeline Contractor	40	18FEB08	15APR08
Process Plant				
Camp Supply/Installation Contract				
1000	Pkg/Bid/Eval/Recom/Apprl/Awd - Camp Sply/Erect	80	05OCT07	31JAN08
1010	Design/Fab/Del Camp	126	01FEB08	31JUL08
Site O/H Lines & Power Distribution				
1015	Pkg/Bid/Eval/Recom/Apprl/Awd-Site Pwr Distr/Subs	80	05OCT07	31JAN08
1025	Procure Site Poles/Hardware/Cable	120	01FEB08	23JUL08
Mass Earthworks (Small Fleet)				
1030	Pkg/Bid/Eval/Recom/Apprl/Awd-Site Mass Earthwrks	80	09AUG07	03DEC07
1035	Mobilize Contr - Site Mass Earthwrks (Sml Fleet)	40	04DEC07	31JAN08
Concrete Including Detl Earthworks				
1050	Pkg/Bid/Eval/Recom/Apprl/Awd - Concrete Wrks	80	02APR08	24JUL08
1055	Moblize Contr - Concrete Wrks	30	25JUL08	05SEP08
1060	Detail/Fab/Del Initial Rebar	30	25JUL08	05SEP08
1065	Set-up Aggregate Preparation & Batch Plants	30	09SEP08	21OCT08
Structural/Mech/Piping Instln				



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Act ID	Activity Description	Orig Dur	Early Start	Early Finish
1070	Pkg/Bid/Eval/Recom/Apprl/Awd Str/Mech/Ppg Contr	80	07NOV08	04MAR09
1075	Pipe Spooling/Materials/Fabrication	121	05MAR09	26AUG09
1080	Mobilize Str/Mech/Ppg Instln Contractor	20	20MAR09	17APR09
Electrical/Instrumentation Instln				
1085	Pkg/Bid/Eval/Recom/Apprl/Awd Elect/Instr Contr	80	08JAN09	30APR09
1090	Contractor Supplied Matls - Elect/Instr Contr	100	01MAY09	23SEP09
1095	Mobilize Contr- Electrical/Instrumentation Contr	20	28SEP09 *	26OCT09
CONSTRUCTION				
Access Road				
Road Heading #1				
1005	Mobilize Road Building Eqpt to Bob Q.	60	01FEB07	01APR07
1012	Constr Road/Brdgs -Starting @ Bob Q. - Access Rd	375	02APR07	10APR08
1016	Rd Sufficiently Compl for Eqpt Movement to Site	0		10APR08
1021	Road Improvements/Widening/Surfacing	105	11APR08	10SEP08
1100	Prep Site, Set-up Road Camp & Facilities @ Bob Q	60	01FEB07	01APR07
Road Heading #2				
1026	Chopper Temp Camp/Facil.at Intermediate Location	10	01MAR07	10MAR07
1031	Chopper Road Eqpt/Fuel to Intermediate Location	10	11MAR07	20MAR07
1036	Assemble Road Equipment	20	16MAR07	04APR07
1041	Drive Road & Const Brdgs	366	26MAR07	25MAR08
Road Heading #3				
1046	Chopper Temp Camp/Facil.at Intermediate Location	10	21MAR07	30MAR07
1051	Chopper Road Eqpt/Fuel to Intermediate Location	10	31MAR07	09APR07
1056	Assemble Road Equipment	20	05APR07	24APR07
1061	Drive Road & Const Brdgs	330	25APR07	19MAR08
Road Heading #4				
1066	Air Transport Roadwrk Eqpt/Fuel to Porcupine	80	12OCT06	11JAN07
1071	Air Transport Bridge Materials to Porcupine	20	12JAN07	31JAN07
1076	Instl Bridge over Porcupine Creek	60	01FEB07	01APR07
1081	Drive Road to Tunnel Portal	26	02APR07	27APR07
1091	Drive Road Towards More Creek	172	28APR07	16OCT07
Tunnel				
South Portal				

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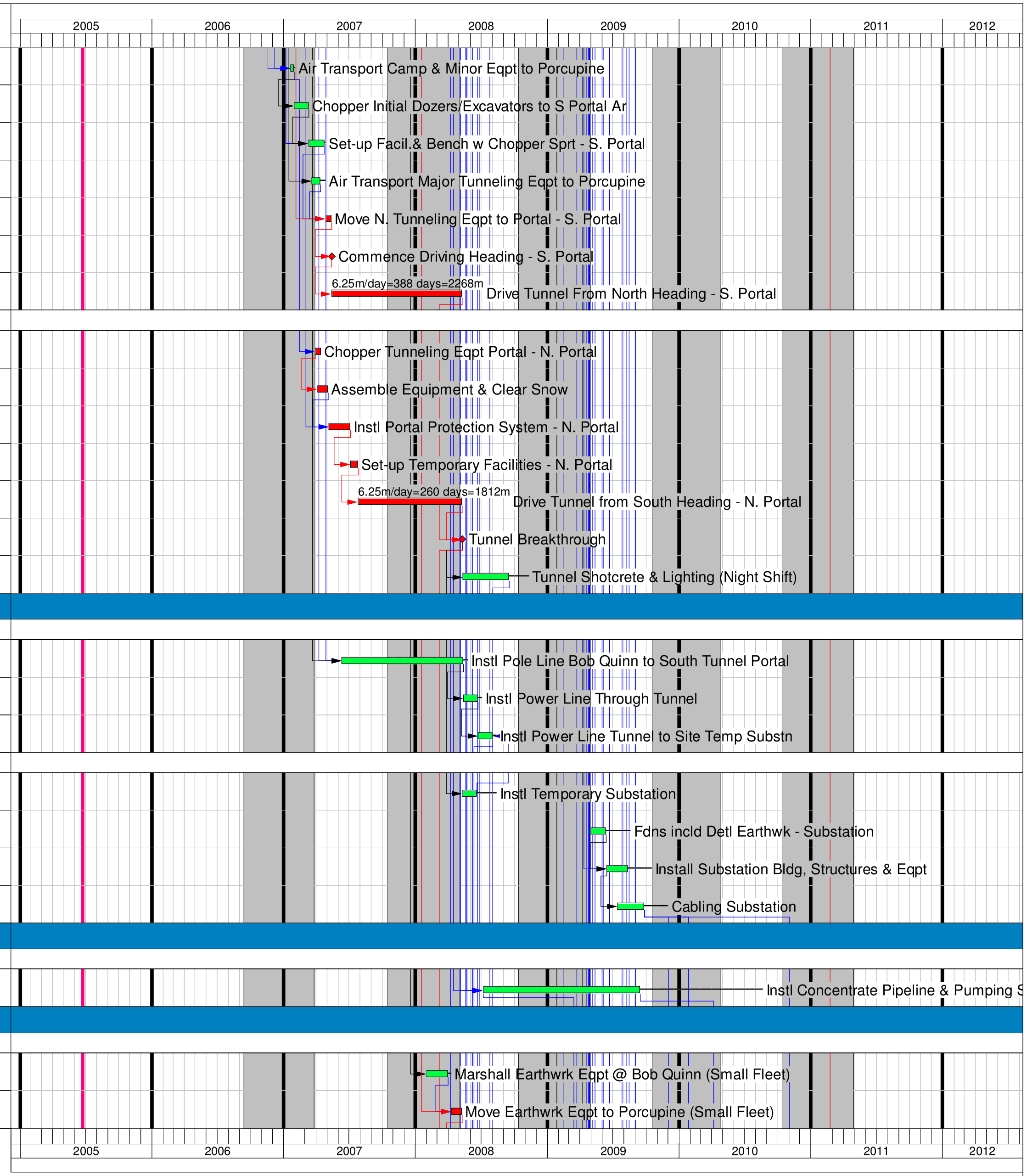
Act ID	Activity Description	Orig Dur	Early Start	Early Finish
1205	Air Transport Camp & Minor Eqpt to Porcupine	10	21JAN07	30JAN07
1210	Chopper Initial Dozers/Excavators to S Portal Ar	40	31JAN07	11MAR07
1215	Set-up Facil.& Bench w Chopper Sprt - S. Portal	45	12MAR07 *	25APR07
1220	Air Transport Major Tunneling Eqpt to Porcupine	15	21MAR07	12APR07
1225	Move N. Tunneling Eqpt to Portal - S. Portal	16	28APR07	13MAY07
1230	Commence Driving Heading - S. Portal	0		13MAY07
1235	Drive Tunnel From North Heading - S. Portal	363	14MAY07	10MAY08
North Portal				
1240	Chopper Tunneling Eqpt Portal - N. Portal	15	31MAR07	14APR07
1245	Assemble Equipment & Clear Snow	20	05APR07	04MAY07
1250	Instl Portal Protection System - N. Portal	60	07MAY07 *	05JUL07
1255	Set-up Temporary Facilities - N. Portal	20	06JUL07	25JUL07
1260	Drive Tunnel from South Heading - N. Portal	290	26JUL07	10MAY08
1265	Tunnel Breakthrough	0		10MAY08
1270	Tunnel Shotcrete & Lighting (Night Shift)	88	12MAY08	16SEP08
Main Power Supply Line & Substation				
Overhead Power Supply Line				
1305	Instl Pole Line Bob Quinn to South Tunnel Portal	337	12JUN07	13MAY08
1315	Instl Power Line Through Tunnel	40	14MAY08	22JUN08
1325	Instl Power Line Tunnel to Site Temp Substn	40	23JUN08	01AUG08
Main Substation				
1360	Instl Temporary Substation	40	11MAY08	19JUN08
1370	Fdns incld Detl Earthwk - Substation	30	01MAY09 *	12JUN09
1380	Install Substation Bldg, Structures & Eqpt	40	15JUN09	11AUG09
1390	Cabling Substation	52	14JUL09	25SEP09
Concentrate Pumping & Pipeline				
1400	Instl Concentrate Pipeline & Pumping Stations	300	07JUL08 *	15SEP09
Pre-Production Mining & Site Development				
2000	Marshall Earthwrk Eqpt @ Bob Quinn (Small Fleet)	60	01FEB08	31MAR08
2010	Move Earthwrk Eqpt to Porcupine (Small Fleet)	30	11APR08	10MAY08



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Act ID	Activity Description	Orig Dur	Early Start	Early Finish
2020	Move Earthwrk Eqpt Porcupine>Site (Small Fleet)	40	11MAY08	19JUN08
2080	Const Wrk Ar./Borrows/Initl Haul Rd (Sml Fleet)	60	21MAY08	19JUL08
2090	Instl & Provision Fuel Storage/Dispensing	20	31MAY08	19JUN08
2260	Move 15,000 Mine Eqpt Bob Q>Site (Mine Fleet)	79	10JUN08	27AUG08
2270	Assemble Mine Initial Eqpt (Mine Fleet)	91	20JUN08	18SEP08
2500	Const Initial Haul Roads	50	10JUL08	28AUG08
2510	Initial Mine Stripping & Pre-Production	751	29AUG08	18SEP10
2520	Const Haul Road to Waste Dump & Tlngs Facility	115	29AUG08	21DEC08
Water Diversion (East Water Channel)				
4000	Construct East Water Channel	248	05JUL08	09MAR09
East Water Dam (Fresh Water Reservoir)				
4110	Strip & Scale Alignment/Cut Keyway - East Dam	149	21MAY08	16OCT08
4120	Construct Temporary Water Pass Thru - East Dam	30	21MAY08	19JUN08
4130	Testing of Geotech Prior to Placement of Core	60	20JUN08	18AUG08
4140	Place Dam Core (Small Fleet)	201	19AUG08	07MAR09
4150	Install Ballast (Small Fleet)	57	08SEP08	03NOV08
4160	Install Ballast (Mine Fleet)	138	04NOV08	21MAR09
4170	Block Temporary Pass Thru & Run Wtr to East Chnl	14	22MAR09	04APR09
Spillway				
4200	Stripping & Scaling - Diversion Spillway	40	05JUL08	13AUG08
4210	Const Spillway - Diversion Spillway	207	14AUG08	08MAR09
4220	Stipping & Scaling - Tailings Dam Spillway	40	15MAY09	23JUN09
4230	Const Spillway - Tailings Dam Spillway	100	24JUN09	01OCT09
Effluent Pumping				
4300	Construct Temp. Infiltration Pumping Platfrm	60	22DEC08	19FEB09
4310	Temp. Instl Effluent Pumps & Pipeline	60	31JAN09	31MAR09
4320	Commence Water Pumping (Power Reqd)	1	01APR09	01APR09
Tailings Facility				

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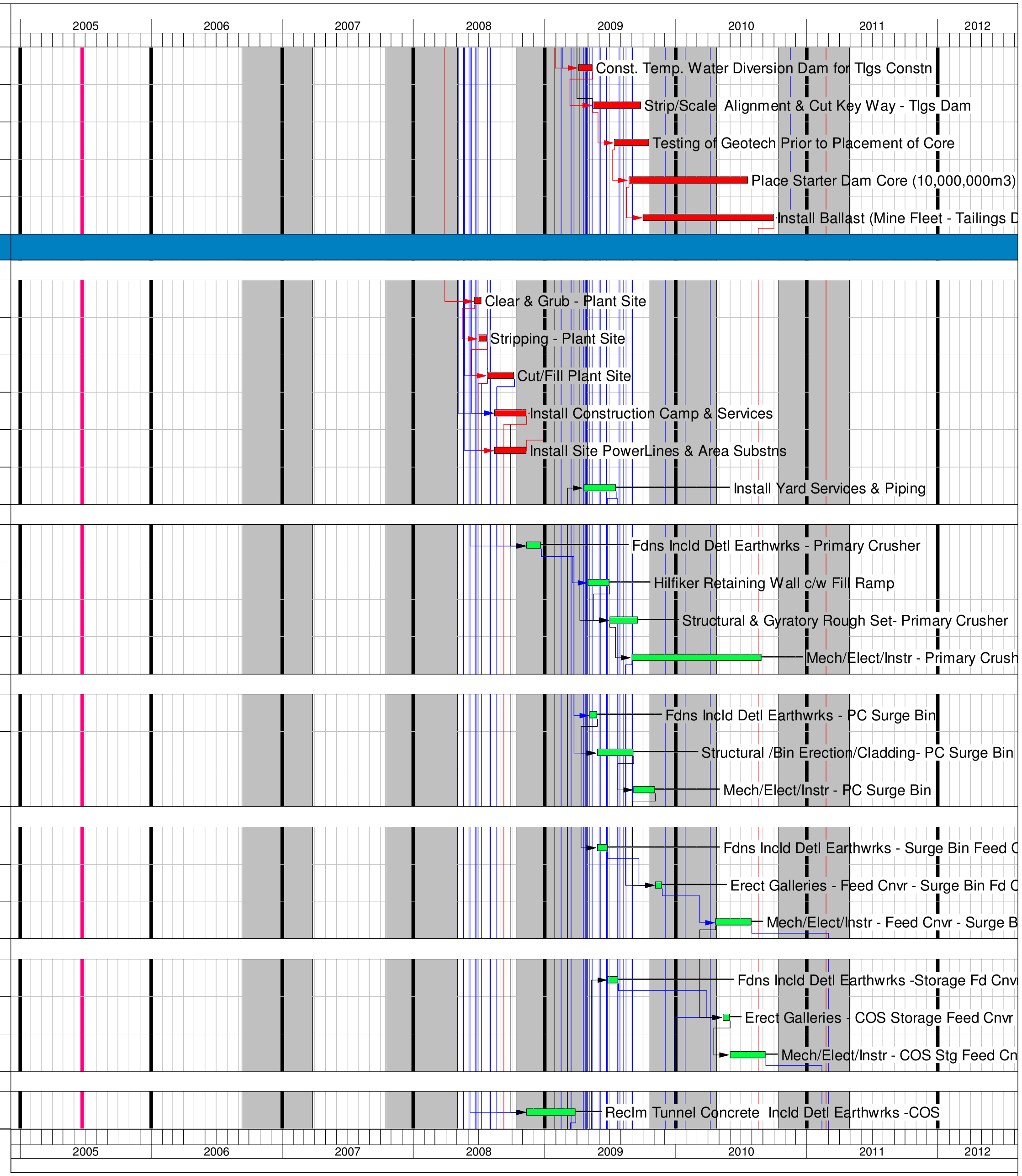
Act ID	Activity Description	Orig Dur	Early Start	Early Finish
4400	Const. Temp. Water Diversion Dam for Tlgs Constn	40	05APR09	14MAY09
4410	Strip/Scale Alignment & Cut Key Way - Tlgs Dam	135	15MAY09	26SEP09
4420	Testing of Geotech Prior to Placement of Core	97	14JUL09	18OCT09
4430	Place Starter Dam Core (10,000,000m3) - Tlgs Dam	332	23AUG09	20JUL10
4440	Install Ballast (Mine Fleet - Tailings Dam	365	02OCT09	01OCT10
Process Plant				
Plantsite				
6020	Clear & Grub - Plant Site	20	20JUN08	09JUL08
6025	Stripping - Plant Site	25	30JUN08	24JUL08
6030	Cut/Fill Plant Site	75	25JUL08	07OCT08
6035	Install Construction Camp & Services	90	14AUG08	11NOV08
6045	Install Site PowerLines & Area Substns	90	14AUG08	11NOV08
6050	Install Yard Services & Piping	90	20APR09	18JUL09
Primary Crushing				
6200	Fdns Incld Detl Earthwrks - Primary Crusher	40	12NOV08	21DEC08
6205	Hilfiker Retaining Wall c/w Fill Ramp	60	02MAY09 *	30JUN09
6210	Structural & Gyratory Rough Set- Primary Crusher	80	01JUL09	18SEP09
6215	Mech/Elect/Instr - Primary Crusher	249	31AUG09	27AUG10
Primary Crusher Surge Bin				
6250	Fdns Incld Detl Earthwrks - PC Surge Bin	20	07MAY09 *	26MAY09
6260	Structural /Bin Erection/Cladding- PC Surge Bin	102	27MAY09	05SEP09
6270	Mech/Elect/Instr - PC Surge Bin	60	06SEP09	04NOV09
Feed Conveyor - PC Surge Bin				
6280	Fdns Incld Detl Earthwrks - Surge Bin Feed Cnvr	30	27MAY09	25JUN09
6290	Erect Galleries - Feed Cnvr - Surge Bin Fd Cnvr	20	05NOV09	24NOV09
6350	Mech/Elect/Instr - Feed Cnvr - Surge Bin Fd Cnvr	100	22APR10 *	30JUL10
Feed Conveyor - COS Storage				
6325	Fdns Incld Detl Earthwrks -Storage Fd Cnvr - COS	20	26JUN09	24JUL09
6330	Erect Galleries - COS Storage Feed Cnvr - COS	20	12MAY10	31MAY10
6340	Mech/Elect/Instr - COS Stg Feed Cnvr - COS	100	01JUN10	08SEP10
Coarse Ore Storage				
6300	Reclm Tunnel Concrete Incld Detl Earthwrks -COS	95	12NOV08	27MAR09



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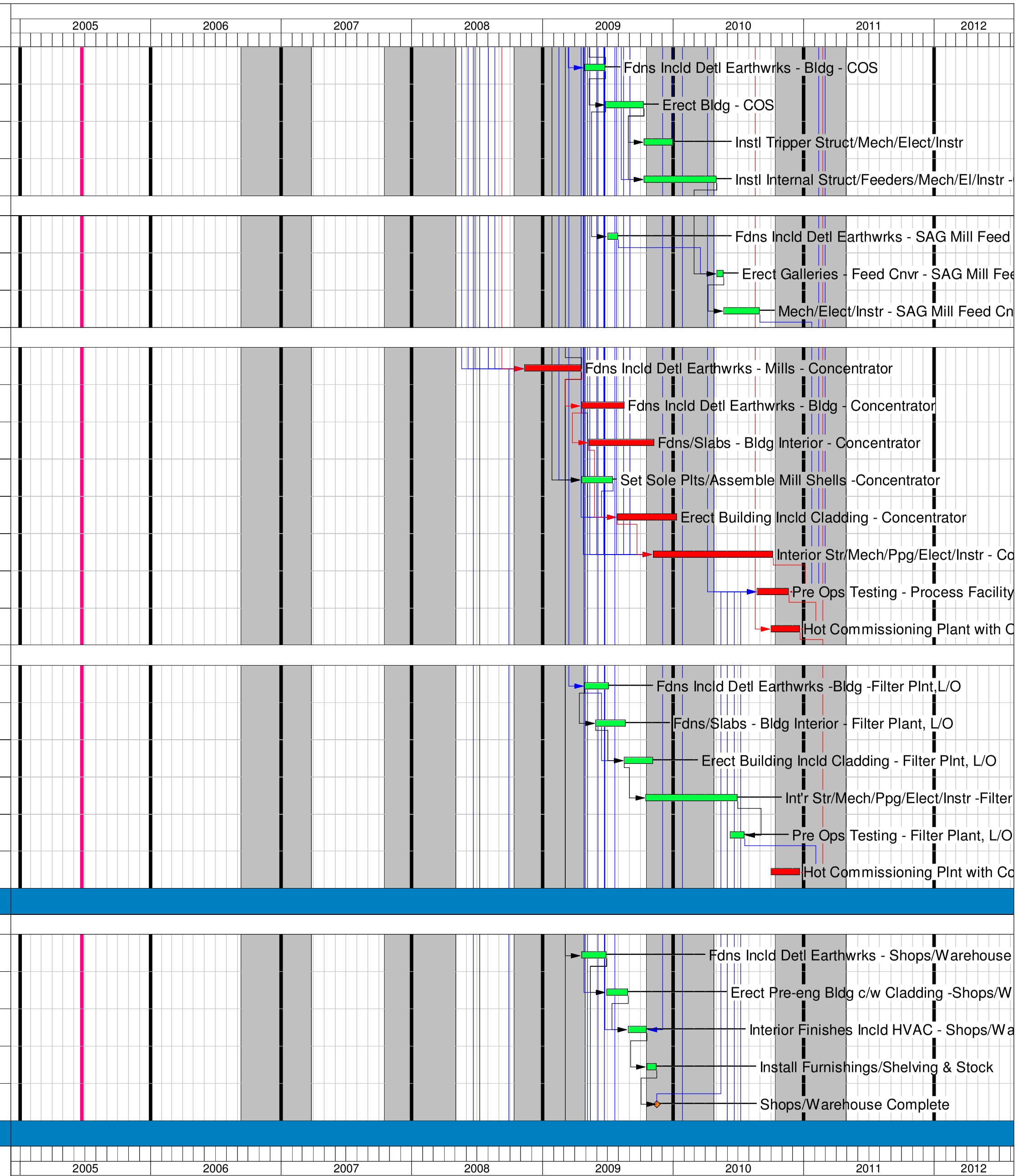
Act ID	Activity Description	Orig Dur	Early Start	Early Finish
6305	Fdns Incld Detl Earthwrks - Bldg - COS	59	28APR09 *	25JUN09
6310	Erect Bldg - COS	108	26JUN09	11OCT09
6315	Instl Tripper Struct/Mech/Elect/Instr	80	12OCT09	30DEC09
6320	Instl Internal Struct/Feeders/Mech/El/Instr -COS	203	12OCT09	02MAY10
SAG Mill Feed Conveyor				
6400	Fdns Incld Detl Earthwrks - SAG Mill Feed Cnvr	30	02JUL09	31JUL09
6410	Erect Galleries - Feed Cnvr - SAG Mill Feed Cnvr	20	03MAY10 *	22MAY10
6420	Mech/Elect/Instr - SAG Mill Feed Cnvr	100	23MAY10	30AUG10
Concentrator				
6450	Fdns Incld Detl Earthwrks - Mills - Concentrator	159	12NOV08	19APR09
6460	Fdns Incld Detl Earthwrks - Bldg - Concentrator	121	20APR09	18AUG09
6470	Fdns/Slabs - Bldg Interior - Concentrator	183	10MAY09	08NOV09
6480	Set Sole Plts/Assemble Mill Shells -Concentrator	88	20APR09 *	16JUL09
6490	Erect Building Incld Cladding - Concentrator	167	29JUL09	11JAN10
6500	Interior Str/Mech/Ppg/Elect/Instr - Concentrator	335	06NOV09	06OCT10
6510	Pre Ops Testing - Process Facility	90	23AUG10	20NOV10
6520	Hot Commissioning Plant with Ore -Process Facil.	80	02OCT10	20DEC10
Filter Plant (Bob Quinn)				
6600	Fdns Incld Detl Earthwrks -Bldg -Filter Plnt,L/O	67	29APR09 *	04JUL09
6605	Fdns/Slabs - Bldg Interior - Filter Plant, L/O	85	29MAY09	21AUG09
6610	Erect Building Incld Cladding - Filter Plnt, L/O	82	17AUG09	06NOV09
6615	Int'r Str/Mech/Ppg/Elect/Instr -Filter Plnt, L/O	257	16OCT09	29JUN10
6620	Pre Ops Testing - Filter Plant, L/O	40	10JUN10	19JUL10
6630	Hot Commissioning Plnt with Con.- Filtr Plnt,L/O	80	02OCT10	20DEC10
Shops & Warehouse				
7000	Fdns Incld Detl Earthwrks - Shops/Warehouse	70	20APR09	28JUN09
7010	Erect Pre-eng Bldg c/w Cladding -Shops/Warehouse	60	29JUN09	27AUG09
7020	Interior Finishes Incld HVAC - Shops/Warehouse	52	28AUG09	18OCT09
7030	Install Furnishings/Shelving & Stock	20	19OCT09	16NOV09
7080	Shops/Warehouse Complete	0		16NOV09
Truck Shop				



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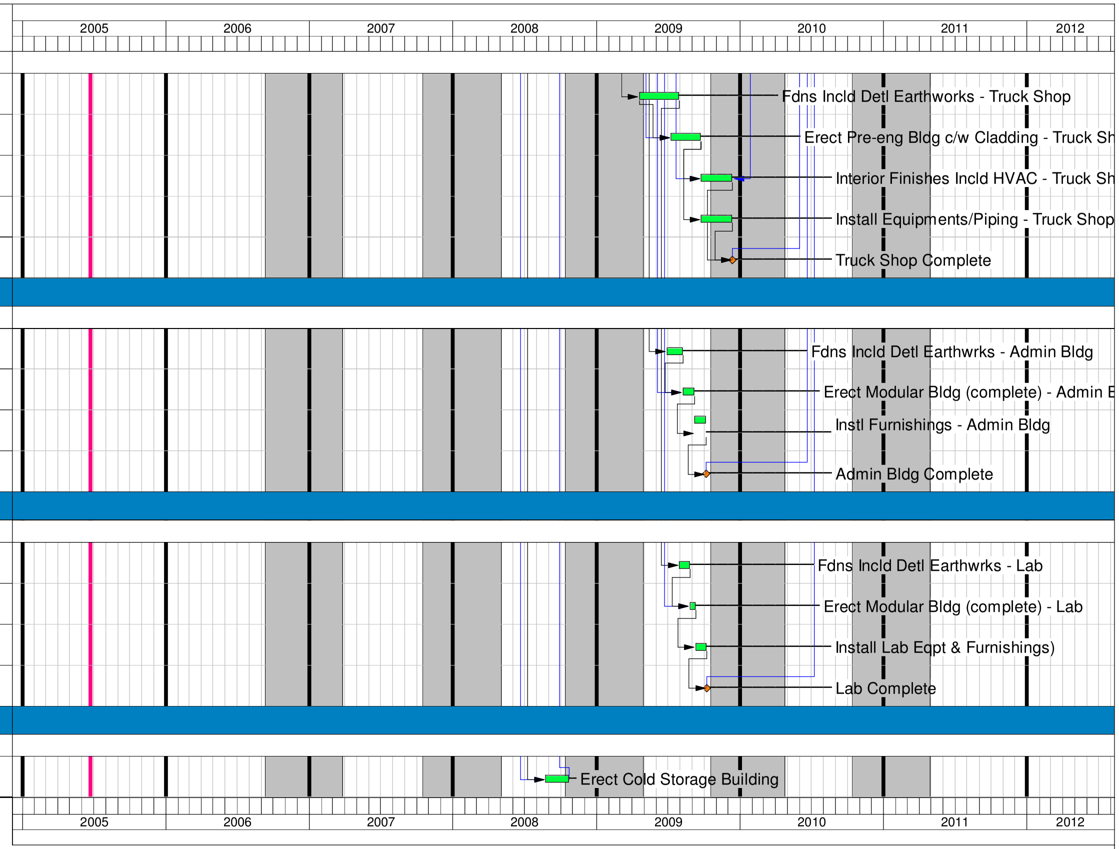
Act ID	Activity Description	Orig Dur	Early Start	Early Finish
7500	Fdns Incld Detl Earthworks - Truck Shop	101	20APR09	29JUL09
7510	Erect Pre-eng Bldg c/w Cladding - Truck Shop	77	09JUL09	23SEP09
7520	Interior Finishes Incld HVAC - Truck Shop	80	24SEP09	12DEC09
7530	Install Equipments/Piping - Truck Shop	80	24SEP09	12DEC09
7540	Truck Shop Complete	0		12DEC09
Admin Bldg				
7610	Fdns Incld Detl Earthwrks - Admin Bldg	40	29JUN09	07AUG09
7620	Erect Modular Bldg (complete) - Admin Bldg	30	08AUG09	06SEP09
7630	Instl Furnishings - Admin Bldg	20	08SEP09	05OCT09
7640	Admin Bldg Complete	0		05OCT09
Lab				
7710	Fdns Incld Detl Earthwrks - Lab	27	30JUL09	25AUG09
7720	Erect Modular Bldg (complete) - Lab	15	26AUG09	09SEP09
7730	Install Lab Eqpt & Furnishings)	20	10SEP09	07OCT09
7740	Lab Complete	0		07OCT09
Cold Storage Buildings				
7750	Erect Cold Storage Building	60	24AUG08	22OCT08



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14. Capital Cost Estimate

14.1 Summary

The capital costs for the Galore Creek Project including process plant, infrastructure and mining were estimated in 2Q, 2005 Canadian dollars and are presented in US dollars. These include no allowances for escalation or exchange rate fluctuations.

The summary capital cost estimate for the Galore Creek Project are presented in Table 14-1 and reflect an accuracy level of ±20%, consistent with a pre-feasibility study level of engineering effort.

Table 14-1: Capital Cost Summary - Base Case, 65,000 tpd mill

Description	**US$M**
Process Plant	**223**
Infrastructure	**301**
Mine	**229**
Total Directs	**753**
Indirects	**205**
Contingency	**144**
Indirects & Contingency	**348**
Total Estimate	**1102**

The base case capital cost estimate for the project options examined are presented in detail in Table 14-2. The detailed capital cost schedules are presented in Appendix B. (Note the costs in Appendix are expressed in CDN$.)

Table 14-2: Capital Cost Breakdown - Base Case - 65,000 tpd mill

Area	US$M
Process Plant	
Excavation & Backfill	**2.7**
Primary Crushing	**9.3**
Coarse Ore Reclaim	**10.6**
Concentrate Electrical	**12.4**
Grinding	**81.9**
Flotation	**24.9**
Concentrate Pumping and Concentrate Pipeline	**55.7**
Concentrate Dewatering	**17.7**
Reagent Handling	**1.3**
Concentrate Loadout	**3.7**
Plantsite Utilities, Comms	**2.4**
PLC & Software	**0.5**
Total Process Plant Cost	**223.1**
Infrastructure	
Shop & Warehouse	**2.2**
Truck Shop	**9.0**
Administrative Building & Laboratory	**3.8**
Plant Access Roads, Tunnels & Bridges	**97.2**
Power Supply	**42.3**
Water Supply	**6.8**
Waste Rock/Tailings Storage/Water Diversion	**72.5**
Water Management	**46.1**
Camp	**12.6**
Airstrip	**3.0**
Other Buildings	**1.9**
Plant Mobile Equipment	**3.2**
Total Infrastructure Cost	**300.6**
Mine	
Haul Roads (Includes Plant Site Roads)	**4.7**
Prestripping	**78.8**
Mine Equipment	**133.7**
Mine Dewatering	**7.5**
Mine Electrical	**2.9**
Magazine	**0.2**

Area	US$M
Fuel Storage, Dispensing and Magazine	1.3
Total Mine Cost	**229.1**
Total Project Direct Cost	**752.8**
Indirect Costs	
EPCM	65.3
Construction Indirects	81.4
Commissioning, Start-up, & Vendor Reps	1.7
Spares	12.3
First Fill	4.0
Freight	17.6
Owners Costs	22.5
Total Project Indirect Costs	**204.8**
Contingency	**144.1**
Total Project Costs	**1101.7**

14.1.1 Scope of Estimate

The capital cost estimates include all the direct and indirect costs and appropriate project estimating contingencies for all the facilities required to bring the Galore Creek mine into production, as defined by a pre-feasibility level of engineering effort. The scope includes:

- the supply, delivery and installation of the mechanical, electrical equipment, instrumentation, piping, steelwork, buildings and concrete works for the mine, process plant and support facilities in Galore Valley.
- the construction of all major civil earthworks for the dams and water diversion structures.
- the construction of a powerline to the mine, the main substation, and connection to the BC Hydro grid from a point near Bob Quinn Lake.
- the construction of the main access road to the minesite, including bridges, culverts and basic avalanche mitigation structures where appropriate.
- the supply, delivery and installation of the mechanical and electrical equipment, instrumentation, piping, steelwork, buildings and concrete works for the concentrate pipeline, pump stations and remote filter plant.
- the construction of the 4 km long access tunnel to the minesite, including ventilation systems.
- no allowance for the refurbishment of a concentrate storage shed at Stewart.
- engineering, procurement and construction management (EPCM) for the project.
- construction indirects including the provision and operation of construction camps for the various sites, construction equipment, crew rotations, communications, supplies, etc.

- owner's costs to include the implementation team, training, etc.
- first fill of reagents and supplies.
- freight and initial spares.
- contingency, assessed by item, activity and level of engineering definition. The contingency included is an estimating contingency and is expected to be spent, and reflects the pre-feasibility level of engineering effort..

The scope of the estimate does not include the following:

- escalation beyond 2Q 2005 for equipment, material or labour costs.
- upgrades to BCH power transmission line up to the location of the Galore Creek tie-in.
- exchange rate fluctuations.

14.2 Basis of Estimate

14.2.1 General

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

- preliminary process design criteria, developed from fairly extensive metallurgical testwork conducted in 2004, limited testwork in 2003, review of previous testwork carried out in the period 1960 – 1992 and experience.
- preliminary process flowsheets identifying all major unit operations.
- preliminary sizing of all major equipment items.
- plantsite layout and development of general arrangement drawings for the major facilities and buildings.
- preliminary mine design, mine scheduling and mine fleet sizing and selection.
- no detailed geotechnical studies have been carried out in the area of the plantsite. A soil bearing pressure of 3000 psf has been assumed for the area.
- limited geotechnical study has been carried out in the areas of tailings/waste and water dams and structures.
- no detailed assessment or mapping of borrow materials suitable for dam construction has been completed to date. It has largely been assumed that suitable till core material can be sourced within 2 km of the main dam site.
- factors were applied for piping and instrumentation, based on experience with similar projects.
- engineering cost studies for the access roads, tunnel and powerline, based on preliminary engineering design, field inspections, digital topographic maps and preliminary geotechnical investigations.
- engineering and cost studies for the concentrate pipeline were based on site inspection, digital topographic maps and extensive previous experience of PSI.

This estimate has been prepared using a combination of quoted, estimated and factored costs to provide a level of accuracy of ±20%. All costs herein are shown in 2Q2005, US dollars.

14.3 Contingency

The intent of the contingency is to cover items that cannot be accurately estimated and it should be assumed that the contingency would be spent. The contingency allowance included here specifically excludes scope changes, growth allowances, technical project risks, exchange rate variance and escalation.

The contingency allowance is evaluated by activity and for each area based on the perceived level of accuracy of the estimate. Generally, the level of uncertainty associated with costs for major earthworks and tunnelling etc. is significantly higher than for equipment. Hatch considers a contingency level of 15 - 20% appropriate for this level of study.

14.3.1 Capital Cost Exclusions

The following costs have been specifically excluded from the scope of the capital cost estimate and are considered to be the owner's cost risk unless otherwise noted:

- project financing costs
- schedule acceleration costs
- sunk costs
- license fees and property payments
- environmental permitting costs
- escalation costs
- exchange rate hedging costs

14.4 Sustaining Capital

Estimates for the sustaining capital requirements through the project life were developed. The significant contributors to these costs are:

- additional mining equipment required according to the ore/waste mining schedule
- mining equipment rebuilds
- tailings dam construction through the mine life
- allowances for periodic process plant upgrades

The estimate for the sustaining capital schedule is summarized in Table 14-3.

Table 14-3: Sustaining Capital Schedule (US$M)

	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	Total
Mining equipment		11	17					3		9	3										44
Tailings/waste rock dam			9	9	9	9	9	9	9	9	9	9	9	9	9	9	9				133
Dam spillways			8		8		8		8		8		8								45
Plant upgrades			2			2			2			2			2			2			9
Totals		**$11**	**$35**	**$9**	**$17**	**$11**	**$17**	**$12**	**$17**	**$18**	**$20**	**$11**	**$17**	**$9**	**$12**	**$9**	**$9**	**$2**	**$0**	**$0**	**$230**

14.4.1 Tailings/Waste Rock Dam and Water Management

The starter tailings/waste rock dam construction is sufficient for the first two years of operations. As the dam fills, it will be necessary:

- to raise the Main dam height, and
- to construct the West dam, to prevent water ingress into the control pit.

In order to minimize the impact on the truck haul fleet, it is assumed that tailings dam construction will take place each year.

The material quantities for dam construction have been estimated by BGC and overhaul rates applied on the basis that the Owner's mining fleet will be utilized for this task.

14.4.2 Plant Upgrades

A nominal allowance has been included to cover general upgrades to the control, analysis and instrumentation systems within the plant.

15. Operating Cost Estimate

15.1 Summary

The base case operating cost for the Galore Creek Project including mining, general and administrative and process costs have been estimated in 2Q, 2005 Canadian dollars and presented in US dollars. These include no allowances for escalation or exchange rate fluctuations.

The summary operating cost estimate is presented in Table 15-1 and reflect an accuracy level of ±15%, consistent with a PFS level of engineering effort.

Table 15-1: Operating Cost Summary

		US$/t ore
G&A	G&A Labour	**0.13**
	Fixed	**0.52**
	Total G&A	**0.65**
Mining	Life of Mine average*	**3.03**
Process	Process Labour	**0.38**
	Consumables	**1.59**
	Power	**0.72**
	Tailings	**0.02**
	Total Process	**2.70**
	Total Minesite Cash Cost	**$6.39**

* Mining costs LOM average US$1.03/t of material mined.

15.2 Scope of Estimate

The operating cost estimates include all the costs associated with the mining, processing and infrastructure activities for a large scale mining operation in this location. The scope includes:

- mining labour and consumables for operations, maintenance and technical support services
- process labour and consumables for operations and maintenance for crushing, grinding, flotation and concentrate dewatering
- concentrate pumping and pipeline operation
- site water management, including effluent pumping from the tailings dam to discharge
- filtrate water treatment costs, prior to discharge

- general and administration costs for the operations including road, tunnel and powerline maintenance, crew rotation by air and property taxes, etc.
- supply of electrical power, including demand charges and an estimate for line losses, to BCH's point of service at Meziadin.

The scope of the estimate does not include:

- escalation beyond 2Q, 2005 for labour or material prices
- exchange rate fluctuations

The detailed operating cost estimates are shown in Appendix C. (Note: the costs in Appendix C are shown in CDN$.)

15.3 Basis of Estimate

15.3.1 General

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

- preliminary mine designs to determine the size and makeup of the mine fleet.
- budgetary quotations for all major consumables, including grinding media, reagents, tires, fuel and explosives.
- budgetary quotations for chartered air services for crew rotation.
- process reagent consumption rates generated from metallurgical testwork and from experience.
- grinding power requirements generated from material testwork and from experience.

15.3.2 Electrical Power

Electrical power consumption was estimated from an assessment of the operating power draws of the units noted in the equipment list. Additional allowances were made for each of the buildings, support services and for HVAC.

Power costs were estimated as the sum of:

- Consumption charges. The current BCH Industrial power tariff of C$0.028/kWh (US$0.021/kWh) was applied to the estimated annual power consumption for the operation.
- Demand charges. These are a function of operations and power management and were estimated as 28% of consumption costs, typical of a similar large copper concentrator in BC.
- Wheeling charges. These were assumed to be a net zero cost to Galore Creek and considered to be part of the BCH rate.

15.3.3 Manpower

The estimated direct manning levels for the operation are summarized below in Table 15-2, and do not include personnel for operating the camp, concentrate haulage, periodic tailings dam expansion or mill relining, all of which are assumed to be contracted services.

Table 15-2: Manning Levels

Area	Function	Number
G&A	Administration	12
	Information Systems	3
	Materials Management	10
	Human Resource	3
	Safety	12
	Site Services	6
	Environmental	5
	S/Total G&A	51
Process	Operations	57
	Maintenance Services	53
	Technical	4
	Warehouse	6
	S/Total Process	120
Mine	Operations	190
	Maintenance	114
	Technical	31
	S/Total Mining	335
Total Direct Mine Operations		**506**

Labour rates for each job category are based on Hatch's experience of similar functions in BC mines. An average burden rate of 40% has been applied to base salaries to include all statutory Canadian and BC, social insurance, medical and insurance costs, pension and vacation costs.

15.3.4 Consumables

Budgetary quotations were obtained for all major consumables from suppliers, including:

- fuel
- explosives
- grinding media

- flotation reagents
- lime
- mill and crusher liners.

Costs for all these items are currently at historic high levels, and in the case of diesel fuel and grinding, media, we have based the costs for the PEA on an assessment of "long term" prices. For these commodities, we have assumed as follows:

- diesel fuel - US$0.50/l, which is equivalent to a US$42/barrel crude oil price
- grinding media - US$675/t

15.3.5 Mine Operating Costs

Mine operating costs are derived from a combination of supplier quotes and historical data. This includes the labour, maintenance, major component repairs, fuel, and consumables costs.

The current fleet hourly operating costs are used as a constant basis over the schedule periods and estimates are input as sustaining/replacement capital for major rebuilds. From the basic operating capacities of the equipment, the travel speed characteristics of the trucks, and the haul road profiles, the equipment productivities for the shovels and trucks are calculated from the MineSight production scheduling program. The truck speeds and cycle times for the various haul cycles were calculated by Finning. The equipment productivity and the scheduled production are used in the scheduling program to calculate the required equipment operating hours. These are multiplied by the hourly operating cost to calculate the total equipment operating costs for each time period.

Labour factors in terms of man hours/operating hour were also assigned to each of the equipment. Labour costs were calculated by multiplying the labor factor by the equipment operating hours. Each major part replacement was calculated from the expected life of the major part, the cost of the part, and the fleet size for that equipment. The same parameters were used for equipment replacement cost calculations.

Blasting costs were based on a blasting study.

The mine operating costs have been broken down into 3 components in the production and cost schedules. These are: G&A Labour, Operating Labour, and Equipment Operating costs. The labour rates are based on current salaries for G&A employees and negotiated hourly rates for mine operations and maintenance personnel. The LOM average mining cost is US$1.03/t mined or US$3.03/t mill feed.

15.3.6 Concentrate Transportation to the Port

Unit rates for concentrate haulage by truck were obtained from a number of haulage contractors with this experience. An average haulage rate of US$0.11/tkm was used, and applied to the total haulage distance from the minesite to the port of Stewart. No backhaul credits were included at this time, although this represents an opportunity. Loading and storage costs of US$6 to US$7 per tonne of concentrate have been included in the cashflow forecast model.

16. Financial Analysis

16.1 Introduction

Preliminary economic analyses were prepared on the base case, 65,000 tpd project with the North access corridor.

The updated economic assessment is preliminary in nature, in that it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary assessment will be realized. However, approximately 80% of the in-pit resources included in the financial models are in the Measured and Indicated categories.

The detailed cashflow schedule is presented in Appendix D.

16.2 Summary

The results of the economic analyses for the Galore Creek Project is summarized in Table 16-1, expressed in US$ using an exchange rate of US$0.75/CDN$.

Table 16-1: Project Financial Analysis Summary

Copper price	US$/lb	**1.00**
Gold price	US$/oz	**400**
Silver price	US$/oz	**6.00**
Exchange rate	US$/CDN$	**0.75**
Ore production rate	tpd	**65,000**
Total Capital cost:	US$M	**$1,102**
Total Copper Production	Billions lbs	**5.9**
Total Gold Production	Mozs	**3.7**
Total Silver Production	Mozs	**40.4**
Minesite total cash cost	US$/t ore	**6.39**
Total cash cost (incl. credits)	US$/lb copper	**$0.54**
Annual production/cash flow, yrs 1-6		
Copper	Mlb/yr	**371**
Gold	oz 000's/yr	**308**
Silver	oz 000's/yr	**2,254**
After-Tax Cash Flow	US$M/yr	**$209**
Cumulative After-Tax Cash Flow (yrs 1-6)	US$M	**$1,230**
Annual production, LOM, average		
Copper	Mlb/yr	**295**
Gold	oz 000's/yr	**188**
Silver	oz 000's/yr	**2,023**
Cash Flow pre-tax, LOM	US$M	**$1,343**
Cash Flow post-tax, LOM	US$M	**$783**
NPV @5%, pre-tax	US$M	**$455**
NPV @5%, post-tax	US$M	**$191**
IRR, pre-tax	%	**11.1**
IRR, post-tax	%	**8.1**
Payback period	years	**5.2**

16.3 Sensitivity Analysis

Sensitivity analysis to metal prices were carried out, on the results of which are summarized in Table 16-2.

Table 16-2: Financial Analysis and Metal Price Sensitivity (US$M)

Sensitivity Matrix – Metal Prices (all equity case, all NPV and Payback Figures After-tax)						
Cu Price (US$/lb)		Au/Ag Price (US$/oz)				
		375/5.5	400/6.0	425/6.5	450/7.0	475/7.5
0.9	NPV @ 0% (US$M)	419	484	549	613	677
	NPV @ 5% (US$M)	-19	21	59	98	136
	Pre-tax IRR (%)	6.6	7.5	8.4	9.2	10.1
	After-tax IRR (%)	4.7	5.4	6.0	6.6	7.3
	Payback (years)	6.6	6.3	5.9	5.7	5.4
1.0	NPV @ 0% (US$M)	719	783	847	911	978
	NPV @ 5% (US$M)	152	191	228	266	303
	Pre-tax IRR (%)	10.3	11.1	11.9	12.6	13.4
	After-tax IRR (%)	7.5	8.1	8.7	9.3	9.8
	Payback (years)	5.5	5.2	5.0	4.8	4.6
1.1	NPV @ 0% (US$M)	1017	1081	1145	1209	1273
	NPV @ 5% (US$M)	320	357	395	432	469
	Pre-tax IRR (%)	135	14.2	14.9	15.6	16.3
	After-tax IRR (%)	10.0	10.5	11.1	11.6	12.1
	Payback (years)	4.6	4.4	4.3	4.1	3.9
1.25	NPV @ 0% (US$M)	1464	1528	1592	1656	1720
	NPV @ 5% (US$M)	568	605	642	678	716
	Pre-tax IRR (%)	17.7	18.3	19.0	19.6	20.2
	After-tax IRR (%)	13.3	13.8	14.3	14.8	15.2
	Payback (years)	3.7	3.5	3.4	3.3	3.2
1.5	NPV @ 0% (US$M)	2209	2273	2337	2401	2465
	NPV @ 5% (US$M)	977	1013	1050	1086	1123
	Pre-tax IRR (%)	23.7	24.2	24.8	25.3	25.9
	After-tax IRR (%)	18.0	18.5	18.9	19.3	19.7
	Payback (years)	2.7	2.6	2.6	2.5	2.5
1.75	NPV @ 0% (US$M)	2954	3018	3082	3146	3210
	NPV @ 5% (US$M)	1385	1422	1458	1495	1532
	Pre-tax IRR (%)	28.8	29.3	29.8	30.3	30.8
	After-tax IRR (%)	22.2	22.6	23.1	23.5	23.9
	Payback (years)	2.2	2.2	2.1	2.1	2.0

Note: NPV = Net Present Value using a Discounted Cash Flow Analysis; IRR = Internal Rate of Return.

Sensitivity analysis to exchange rate, capital and operating cost variations are presented in Table 16-3.

Table 16-3: Sensitivity Analysis (IRR%)

IRR% (after tax)	Variation from Base Case						
	-15%	-10%	-5%	0	+5%	+10%	+15%
Exchange rate, construction	8.9	8.6	8.4	8.1	7.8	7.6	7.4
Exchange rate, operations	9.5	9.0	8.6	8.1	7.6	7.1	6.6
Capital cost	10.1	9.4	8.7	8.1	7.5	7.0	6.4
Operating cost	10.4	9.7	8.9	8.1	7.2	6.3	5.4
Diesel price	8.3	8.3	8.2	8.1	8.0	7.9	7.8
Concentrate haulage cost	8.2	8.2	8.1	8.1	8.0	8.0	7.9
TC/RC	8.9	8.6	8.4	8.1	7.8	7.5	7.2
Ocean freight, concentrate	8.3	8.3	8.2	8.1	8.0	7.9	7.8

The results indicate that the Galore Creek project is most sensitive to metal prices (and hence also recoveries and head grade), operating costs, capital costs and exchange rate during operations. The project appears to be relatively insensitive to diesel price and concentrate haulage and ocean freight costs. Overall, the project is more sensitive to operating than capital costs.

16.4 Basis of the Cashflow Analysis

The cashflow analyses, presented in Appendix D, are determined assuming mining production schedules based on the resources and grades and capital and operating cost estimates described in this report. In addition, the following key assumptions form the basis of the analyses:

Metal Prices

- Copper — US$1.00/lb
- Gold — US$400/oz
- Silver price — US$6.00/oz

Smelter Terms

- Treatment Charge — US$75/dmt
- Copper Accountability — Deduct 1 unit pay balance
- Gold Accountability — 97%
- Silver Accountability — 90%
- Copper Refining Charge — US$0.075/lb
- Price Participation — ± 10% at $0.90/lb

• Gold Refining Charge	US$5.5/oz
• Silver Refining Charge	US$0.40/oz
• Provisional Payment 90%	45 days after production
• Final Payment	150 days after production
• Ocean Freight	US$35 per dmt

Exchange Rate 0.75 US$/CD$

The average exchange rate for the Canadian dollar to US dollars over the 10 years to July 2005 was US$0.70 per CDN$ according to the Bank of Canada. The exchange rate at July 31, 2005 was US$0.81 per CDN$. Since the basis of metal prices and capital costs for the economic modeling has been chosen to reflect long term prices, an exchange rate of US$0.75 per CDN$ has been used throughout.

Costs	expressed in second quarter 2005 dollars
Escalation	excluded
Financing Charges	excluded
Working Capital	4 months of operating costs
Constant Dollars	All prices and costs are expressed in constant 2005 dollars.
Commissioning and the Start of Production:	It is assumed that start-up will occur in 4Q of 2010 (final construction year), and that revenue during this start-up phase will cover the costs of commissioning.

16.4.1 Taxation

Canadian mining operations are subject to an essentially 3-tiered tax system:

- **Federal income tax** is levied on an operations taxable income, generally net of operating expenses, depreciation on capital assets and deduction of exploration and pre-production costs.
- **Provincial income taxes** are based on the same taxable income.
- **Provincial mining taxes**, duties or royalties are levied on a separate measure of production profits or revenues.

17. Project Opportunities, Risks and Recommendations

During the development of the PEA, a number of opportunities to potentially improve the project performance in terms of economics, environmental mitigation and construction were identified, and these are discussed below, together with the potential risks to the project and recommendations to be investigated in subsequent study.

17.1 Opportunities

A number of opportunities have been identified for the project that could result in improved project economics and should be investigated further. These are discussed below.

17.1.1 Continued exploration and delineation of resources

NovaGold has been highly successful in expanding the known mineralization in and around the Galore Creek valley. Given this success and the fact that the plan laid out in this PEA envisages mining and processing less than half of the mineral resources quantified in the spring 2005 resource estimate, it is logical and recommended to continue to search for new resources in the valley and to better define those resources which are in the inferred category. Success in this area may:

- Enhance project value by extending the initial period of higher grade ore available to the processing facility and deferring the processing of average or lower grade ores.
- Allow for a significant extension to the current 20 year mine life.
- Provide sufficient resources to support a major expansion in processing rate.

It is noted that this study ignores the inferred resource at Copper Canyon entirely. This area, which remains open in several directions, alone contains 164 million tonnes of inferred mineralization at an average grade of 0.35% copper, 0.54 g/t gold, and 7.1 g/t silver at a copper-equivalent cut-off of 0.35%.

17.1.2 Refinement of Mining Schedules

The mine schedule provided in this PEA was produced by balancing the need to mine higher grades early to support project payback and maintaining a logical sequence in the open pits given a certain sized truck and shovel fleet. Emphasis in this analysis was placed on the first 5-7 years of mining and production and as the project advances toward feasibility, it is recommended that this schedule be further refined with the objectives of:

- Smoothing the copper concentrate tonnage level in the first 7 years. This refinement will enhance the ability of the project to sell a fixed portion of its product under long term contracts early in the mine life.
- Improving the grade delivered to the processing facility in years 8 and 9. The current production schedule witnesses a sharp drop in grade in these years as stockpiles are processed while stripping is carried out to access deeper ore zones. A logical alternative to this approach (which should

enhance the cash flow in these years) would be to add a few trucks to accelerate the mining rate ahead of this period or to hire contract miners ahead of year 8 and 9 to support a higher mining rate. Either alternative should allow higher grades to be mined in 8 and 9 and smooth the volatility of cash flow during this period.

17.1.3 Increased Production Rate

Increasing the rate of metal production could possibly be achieved by:

- Increasing the throughput through the existing circuit. It may be possible to further increase the throughput through the present milling circuit, possibly in the early years when the feed material is mostly broken rock and possibly at a marginally coarser grind. The issues of grind vs. recovery and further hardness evaluation of the orebody will be investigated in the FS metallurgical program. It is not uncommon for copper concentrators to be operating in excess of the nameplate production rate, and an additional 5 to 10% additional throughput may be possible.
- Mine and mill at a higher rate by adding additional equipment. It may be more economic to consider a significantly higher production rate, say 80,000 tpd. On the basis of the existing pit delineated resource estimates this would result in a shorter mine life of approximately 15 years, however, it is reasonable to expect that some of the resources not currently in the mine plan would be added to maintain or increase the mine life over the current 20 years. A conceptual level assessment of this possibility, suggests that capital costs for this case would likely increase in the range of US$120M to US$270M.

17.1.4 Tailing and waste rock storage facility sizing and construction

Two potential opportunities have been identified, as follows:

- The facility is currently designed to hold all the 1.5 billion tonnes of tailings and waste rock and at the end of mine life the facility will be entirely flooded. It is expected that further testing and modeling of the waste rock may well show that the quantity of PAG rock is significantly less than currently modeled. Testwork is currently ongoing to improve the characterization of the waste rock geochemistry. This would then provide two areas of economic improvement:
  - If less of the current material mined needs to be impounded within the storage facility more material may be processed without incurring additional storage costs, potentially significantly improving the current economics.
  - Alternatively the overall height, size and cost of the Main dam could be reduced with significant direct saving. In addition, if a smaller dam can be designed, it may also be possible to use alternative less expensive materials in place of the till core, at least for the starter dam.
- It may be possible to utilize a bitumen geomembrane material as an alternative material to clay till for the starter Main dam. This would be less expensive and could be applied more rapidly than the construction of a till core. Subsequent construction of the Main dam would incorporate a till core, but an initial cost saving could be realized without compromising the integrity of the design concept.

17.1.5 Construction Cost and Schedule

NovaGold is continuing to study ways of reducing the capital costs and improving the construction schedule. Studies are ongoing to establish more accurately all major areas of costs, and to see whether further optimizations can be made to the construction timeline.

17.1.6 Site power generation

NovaGold commissioned a conceptual level study of the feasibility of generation of electrical power on site, by running the flow from the West diversion ditch through a turbine. This would essentially be a Run of River (ROR) power plant. Initial indications are that 20 to 30MW of power could be generated, at a capital investment in the order of US$23M to US$30M. This system could feed power back into the BCH grid at the cessation of mining operations.

17.1.7 Backhaul of operating supplies

Many mining operations are able to reduce the transport costs of their supplies by using the empty concentrate haul trucks to transport supplies to the mine on the return trip.

17.2 Risks

The main areas of risk to the project have been identified in the Updated PEA as design, construction and operations. Most of these risks are common to many projects in northern Canada and no risks have been identified that would reasonably be expected to jeopardize the successful development and operation of the project, based on the assumptions in this study. These are discussed briefly below.

17.2.1 Design Risks

The main areas of design risk for the project are as follows:

- Geotechnical – design changes to the dam and facility structures and their location may result from less favourable geotechnical than assumed in those areas. Similarly, open pit slopes angles may be reduced from the current design as a result of the FS geotechnical program underway in that area.
- Water management – more detailed review of the ground and surficial water regime may result in design changes to the size of dam and diversion channel structures. Improved characterization of tailings dam effluent quality and predicted effluent quality may also lead to changes in the design discharge rate and possibly result in the addition of water treatment prior to effluent discharge.
- Electrical power supply capacity – this study assumes a connection in the vicinity of Bob Quinn to the proposed Coast Mountain Power Project power line. Studies to date indicate a north bound power flow of 80 MW can be delivered through this proposed system. In the absence of Coast Mountain Power, the project may need to connect directly to the BC grid at Meziadin. A study made by BCTC indicates that 80MW can be made available at this location.

17.2.2 Construction Risks

Typical of major mining developments most places in the world there are a number of risks associated with the construction of this project. All of these risks are manageable and they largely impact capital cost and schedule. The planning and modelling has recognized each of these risks and certain provision have been made for potential delays and changes. The main risks are as follows:

- Adverse weather. This probably constitutes the greatest uncertainty, as the potential impacts are numerous:
  - delay in the start of construction activities could result in a shortened initial summer construction period and consequently an extended completion date.
  - shorter construction seasons would lead to more work in winter conditions, at reduced efficiencies, additional costs for heating and boarding.
  - greater requirements for active avalanche control and road maintenance.
  - delay in accessing the tunnel portal sites and reduced progress for portal construction.
  - extreme wet spring/summer season could significantly slow progress of the access road construction.
- Construction materials for earthworks – during large scale construction and mining operations, it may become evident that the quality of specific materials are not suitable. The quantity and moisture content of the clay till core for the Main dam is a risk, as is the quantity of NPAG waste rock.
- Difficult construction area – there are a number of areas of difficult construction which could lead to extended construction periods. Sphaler Creek is difficult terrain for road and bridge construction, and it is possible that this section will take longer than planned to construct.
- Construction labour availability and cost – the current construction market in BC and Alberta has lead to significant shortage of skilled construction labour in BC and consequent increasing costs. Much of this activity is focussed on the Oil Sands projects in Alberta and civil infrastructure projects in BC associated with the 2010 Olympics, all of which are likely to persist over the periods during which Galore would be constructed.
- Construction material cost – current prices for steel, mining and processing equipment are high and reflect the prevailing market conditions. Steel prices, in particular, have increased 25% to 30% over those one year previously.
- Exchange rate fluctuations – many components for equipment and materials are manufactured in the USA or Europe and consequently the prices of the finished product are affected to some degree by exchange rate fluctuations, which could lead to increased capital costs.

17.2.3 Operations Risks

The key risks for operations are largely associated with the consequence of adverse weather conditions.

- Reduced mine production – the frequency and severity of winter storms could result in reduced production from the mine.
- Water management in the mine – may prove more difficult, resulting in higher costs and loss of production time.
- Geohazards – the severity and frequency of avalanches and debris flows pose significant risks to road, pipeline and powerline, which can be mitigated to a large extent by appropriate design.

17.3 Recommendations

The following is a summary of the recommendations to be carried out in the subsequent study to support a FS.

17.3.1 General

- Rationalize the topographic data and ensure one common system used for FS.

17.3.2 Site Investigation Program

During 2005 Feasibility studies, a site investigation program will be carried out to confirm practical portals, further define the expected tunnelling conditions, and confirm constructability issues. The site investigation program shall consist of the following:

- Obtain the most current available digital aerial (ortho) photos and topography.
- Site inspection to ground-proof bedrock and overburden conditions at both the north and south portals.
- At the north portal, a seismic survey to best determine the depth and profile of the overburden in the portal area. The thickness of overburden will ultimately determine the optimized elevation of the rock portal and associated surface excavations for the protected canopy structure.
- Reconnaissance geological mapping along the tunnel alignment to assist in characterizing geological unit types, contacts and structural orientations.
- A seismic refraction survey (and possibly other geophysical surveys) should be located to best define the profile (longitudinal and valley section) of the valley bottom in the area of the southern portal.
- Rock samples from the mapping and drilling program for strength and petrographic analyses.

17.3.3 Mining

The following items should be addressed in the subsequent feasibility study:

- Dam construction material waste classification. Material for tailings dam construction was scheduled using waste types identified from the current ABA model. Most of the waste generated during preproduction is till or broken PAG which is not suitable for dam construction. A full evaluation of the ABA characteristics of the pre-production waste mined for dam construction is required in order to ensure sufficient NPAG material.
- Geotechnical parameters. Geotechnical and geohydrology parameters used in this study were of a pre-feasibility level. Geotechnical design assumptions should be updated with more definitive engineering design parameters in order to complete the feasibility study. Overburden and stick rock slope drainage need to be addressed and hydraulic transmissivity of the major structure and groundwater draw down need to be addressed.
- SG factor for broken zone. The SG factor applied in the broken zone requires testing and confirmation.
- Ore classification upgrade. Ore classification should upgrade indicated and inferred classes based on close spaced drilling from the 2005 exploration program.
- Mining factor. Delineated resource calculations were based on a 100% mining factor. This assumption should be verified by quantifying what mining factor can reasonably be expected with the resource model and using large mining equipment.
- Economic pit limits. The economic pit limits optimization needs to be redone based on the resource model updated with the 2005 exploration data, using only Measured and Indicated resource class material, and also use updated costs, recoveries and geotechnical parameters from the results of ongoing work.
- Mine waste management plan. The waste dump quantities need to be updated to accommodate revised ABA modeling information. Mine waste dump face advances need to be matched to the production schedule for more accurate haul cycle calculations. Pit backfilling opportunities need to be determined.
- ABA modeling in resource model. The ABA designation of material in the resource model needs to be updated with 2005 field data, and the results of ABA testing.
- Water diversion structures. Detailed construction parameters need to be developed for stream diversions located on, or in close proximity to designed high walls.
- Avalanche control structures. Rock embankments, release zone fences, avalanche locations, and an avalanche management plan need to be developed to protect critical roads and facilities, and maintain safe operating conditions.

17.3.4 Metallurgy

The following issues have been identified in the PEA, which need to be better defined in the Feasibility Study from the drilling planned for 2005 and a metallurgical program. It is recommended that fresh drill core samples, rather than assay rejects, be used for the metallurgical program.

- Mineralogical, grinding and metallurgical variability across the deposit
- Flowsheet optimization and grinding circuit design
- Concentrate dewatering requirements
- Copper, gold and silver recovery correlations and projections
- Occurrence of gold and its recovery particularly in pyritic, low copper ores
- Occurrences of talc-like minerals and concentrate penalty elements, such as lead, selenium and fluoride, and their process requirements

It is recommended that the metallurgical program include:

- Ore hardness testing on a minimum of 50 discrete ore samples covering the various ore zones and spatial distributions to add to the current database for a statistical analysis of the grinding circuit.
- Further assessment of gold recovery from pyritic ores
- Ore characterization using batch flotation tests on a minimum of 50 samples from across the deposit. To confirm the flowsheet and develop metals recovery correlations and projections for mining blocks. This should cover a range of head grades and mineralization.
- Metallurgical and production assessment of operating year groupings, such as for Years 1+2+3, Years 4+5 and Years 6 to 10, using batch and locked cycle tests.
- Pilot plant campaign on the composite ore representing the first 2 to 3 operating years to generate concentrate for marketing purposes, liquid-solid separation tests and materials for environmental evaluations.

17.3.5 Process

- Reassess the benefits for a tailings thickener in light of effluent discharge requirements.
- Re-evaluate coarse ore stockpile configuration.

17.3.6 Power Supply

- A rigorous interconnection study for the Forrest Kerr and BCH grid has to be carried out to determine the north bound power limits for the HVDC system and investigate the options for debottlenecking.

17.3.7 Concentrate Handling

- Carry out testwork to define process parameters for the concentrate pumping system.
- Carry out a detailed study of the winter operation for the pipeline and carry out HAZOP analysis of the system during the FS.
- Solicit quotations from local experienced haulers for the concentrate haul route.
- Carry out traffic studies for the concentrate traffic through Stewart.

17.3.8 Construction and Logistics

- Carry out detailed FS level evaluation of integrated construction plan and schedule for the project, to include identifying marshalling areas and route plans, and confirm most suitable road camp areas.

17.3.9 Access Road and Tunnel

- Carry out FS level design and costing for the facilities, discuss with contractors to increase the confidence in the cost estimates.

Appendix A

Mining Schedules

GR Technical Services

Series 5.10 Pits - S23 - Aug 31 2005 : 65,000tpd

Galore Creek Project

SUMMARY TABLES

DIRECT COSTS		Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
PRODUCTION												
ORE												
Ore mined to crusher	tonnes	-	-	-	23,735,018	23,735,028	23,734,976	23,734,992	23,735,000	23,735,029	23,735,005	12,853,028
CuKrg	%	-	-	-	0.731	0.939	0.962	0.626	0.715	0.790	0.803	0.629
AuKrg	g/t	-	-	-	1.072	0.316	0.344	0.664	0.480	0.500	0.371	0.374
AgKrg	g/t	-	-	-	3.976	6.270	6.756	3.690	4.536	5.864	7.341	5.596
Ore mined to stockpiles	tonnes	0	61,129	61,129	1,952,410	2,624,628	2,780,071	5,764,682	2,792,971	2,912,743	425,766	1,504,665
Total Ore Mined	tonnes	0	61,129	61,129	25,687,428	26,359,656	26,515,047	29,499,675	26,527,971	26,647,772	24,160,771	14,357,693
Ore retreived from stockpiles	tonnes	-	-	-	-	-	-	-	-	-	-	10,882,000
CuKrg	%	-	-	-	-	-	-	-	-	-	-	0.332
AuKrg	g/t	-	-	-	-	-	-	-	-	-	-	0.182
AgKrg	g/t	-	-	-	-	-	-	-	-	-	-	2.485
Total Stockpile Inventory	0	-	61,129	122,258	2,074,669	4,699,296	7,479,367	13,244,050	16,037,021	18,949,764	19,375,529	9,998,194
Total Ore to Mill	tonnes	0	0	0	23,735,018	23,735,028	23,734,976	23,734,992	23,735,000	23,735,029	23,735,005	23,735,028
CuKrg	%	-	-	-	0.731	0.939	0.962	0.626	0.715	0.790	0.803	0.493
AuKrg	g/t	-	-	-	1.072	0.316	0.344	0.664	0.480	0.500	0.371	0.286
AgKrg	g/t	-	-	-	3.976	6.270	6.756	3.690	4.536	5.864	7.341	4.170
Sub Grade to Waste	tonnes	-	48,211	48,211	2,005,984	2,651,625	3,203,052	4,861,323	2,815,308	2,999,910	581,771	0
Waste Mined	tonnes	0	37,389,525	37,389,525	41,041,618	59,722,766	84,011,896	44,372,848	49,390,511	49,085,283	53,989,082	56,494,257
Total Waste Mined	tonnes	0	37,437,736	37,437,736	43,047,602	62,374,391	87,214,947	49,234,170	52,205,819	52,085,192	54,570,853	56,494,257
Waste Types:												
Till	tonnes	-	10,654,989	10,654,989	17,193,520	16,875,780	31,066,715	5,191,076	9,111,231	4,722,415	11,083,950	4,763,297
Broken NPAG	tonnes	-	258,836	258,836	12,733,734	9,138,005	10,277,417	7,042,877	16,099,452	2,665,646	2,596,328	7,595,524
Broken PAG	tonnes	-	25,671,525	25,671,525	10,769,786	31,200,173	32,287,318	13,648,787	11,509,365	3,309,672	27,062,828	21,724,625
Stick NPAG	tonnes	-	-	-	344,577	486,668	2,145,260	6,157,664	9,604,237	8,620,731	640,917	2,886,690
Stick PAG	tonnes	-	804,175	804,175	-	2,022,140	8,235,186	12,331,968	3,044,081	4,999,445	738,509	11,487,533
Junction (Un Zoned)	tonnes	-	-	-	-	-	-	476	22,146	24,767,374	11,866,550	8,036,588
Waste Hauled for Main Dam Construction	tonnes	-	13,289,391	13,289,391	11,430,428	12,519,762	12,519,762	12,519,762	12,519,762	12,519,762	12,519,762	12,519,762
Strip Ratio1 (Waste/Total Ore Mined)		-	612.4	612.4	1.7	2.4	3.3	1.7	2.0	2.0	2.3	3.9
Strip Ratio2 (Waste/ Total Ore to Mill)		-	-	-	1.8	2.6	3.7	2.1	2.2	2.2	2.3	2.4
Strip Ratio3 (Waste/ Ore Mined to Crusher)		-	-	-	1.8	2.6	3.7	2.1	2.2	2.2	2.3	4.4
Total Material *Mined*	tonnes	-	37,498,865	37,498,865	68,735,030	88,734,047	113,729,994	78,733,845	78,733,791	78,732,964	78,731,625	70,851,950
Total Material *Moved*	tonnes	-	37,498,865	37,498,865	68,735,030	88,734,047	113,729,994	78,733,845	78,733,791	78,732,964	78,731,625	81,733,950

DIRECT COSTS	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	TOTAL
PRODUCTION														
ORE														
Ore mined to crusher	15,152,282	19,571,872	23,735,042	22,738,346	19,434,221	23,734,985	23,735,007	11,803,302	23,735,020	23,734,993	17,723,428	12,317,999	0	416,414,573
CuKrg	0.560	0.654	0.682	0.644	0.572	0.662	0.746	0.824	0.512	0.494	0.487	0.651	-	
AuKrg	0.264	0.292	0.260	0.229	0.182	0.247	0.303	0.434	0.130	0.186	0.513	0.594	-	
AgKrg	3.146	4.589	4.758	4.081	4.568	5.448	6.179	6.587	4.853	5.045	2.749	3.761	-	
Ore mined to stockpiles	3,404,720	2,865,985	4,075,518	1,459,643	3,327,925	4,311,865	1,785,219	1,092,177	6,040,080	4,251,172	3,634,396	1,098,838	0	58,227,732
Total Ore Mined	18,557,003	22,437,857	27,810,560	24,197,989	22,762,146	28,046,849	25,520,226	12,895,480	29,775,100	27,986,165	21,357,824	13,416,837	0	474,642,306
Ore retreived from stockpiles	8,583,000	4,163,000	-	997,000	4,301,000	-	-	11,932,000	-	-	6,012,000	10,260,000	-	57,130,000
CuKrg	0.328	0.351	-	0.346	0.345	-	-	0.342	-	-	0.337	0.317	-	
AuKrg	0.189	0.142	-	0.143	0.141	-	-	0.138	-	-	0.155	0.222	-	
AgKrg	2.538	1.394	-	2.610	2.562	-	-	3.029	-	-	3.240	2.744	-	
Total Stockpile Inventory	4,819,915	3,522,900	7,598,418	8,061,061	7,087,986	11,399,850	13,185,069	2,345,247	8,385,327	12,636,498	10,258,894	1,097,732	1,097,732	
Total Ore to Mill	23,735,282	23,734,872	23,735,042	23,735,346	23,735,221	23,734,985	23,735,007	23,735,302	23,735,020	23,734,993	23,735,428	22,577,999	0	473,544,573
CuKrg	0.476	0.601	0.682	0.632	0.531	0.662	0.746	0.582	0.512	0.494	0.449	0.499	-	
AuKrg	0.237	0.266	0.260	0.225	0.174	0.247	0.303	0.285	0.130	0.186	0.423	0.425	-	
AgKrg	2.926	4.029	4.758	4.020	4.205	5.448	6.179	4.798	4.853	5.045	2.873	3.299	-	
Sub Grade to Waste	0	0	0	0	0	0	0	0	0	0	0	0	0	19,215,394
Waste Mined	51,595,277	56,133,959	60,921,701	38,538,302	48,293,398	30,686,839	28,212,727	23,906,802	54,630,426	40,747,383	58,546,356	10,837,482	0	1,015,937,962
Total Waste Mined	51,595,277	56,133,959	60,921,701	38,538,302	48,293,398	30,686,839	28,212,727	23,906,802	54,630,426	40,747,383	58,546,356	10,837,482	0	1,035,153,356
Waste Types:														
Till	795,510	250	7,896,058	6,645,212	197,812	4,022,952	5,834,944	6,105,847	5,102,027	7,102,303	1,096,501	-	-	166,117,378
Broken NPAG	6,893,579	915,036	6,605,879	9,397,393	4,254,852	2,000,293	8,505,829	16,040,337	14,865,198	20,547,409	21,613,920	-	-	180,306,378
Broken PAG	4,456,352	521,367	10,942	6,192,789	5,318,923	712,152	710,447	133,450	2,486,660	1,499,803	2,429,555	-	-	227,328,043
Stick NPAG	9,035,823	21,281,916	17,664,459	5,769,663	19,515,956	12,750,186	5,545,906	192,522	21,369,879	8,803,735	21,506,497	6,576,734	-	180,900,018
Stick PAG	25,599,429	33,415,390	17,214,868	1,198,724	9,519,474	3,204,223	984,981	-	10,806,662	2,794,134	11,899,883	4,260,748	-	165,365,728
Junction (Un Zoned)	4,814,584	-	11,529,495	9,334,522	9,486,383	7,997,033	6,630,621	1,434,646	-	-	-	-	-	95,920,417
Waste Hauled for Main Dam Construction	11,358,737	11,358,737	11,358,737	11,358,737	11,358,737	11,358,737	11,358,737	11,358,737	11,358,737	0	0	0	0	227,876,181
Strip Ratio1 (Waste/Total Ore Mined)	2.8	2.5	2.2	1.6	2.1	1.1	1.1	1.9	1.8	1.5	2.7	0.8	-	2.2
Strip Ratio2 (Waste/ Total Ore to Mill)	2.2	2.4	2.6	1.6	2.0	1.3	1.2	1.0	2.3	1.7	2.5	0.5	-	2.2
Strip Ratio3 (Waste/ Ore Mined to Crusher)	3.4	2.9	2.6	1.7	2.5	1.3	1.2	2.0	2.3	1.7	3.3	0.9	-	2.5
Total Material *Mined*	70,152,279	78,571,815	88,732,261	62,736,291	71,055,544	58,733,688	53,732,953	36,802,282	84,405,526	68,733,548	79,904,181	24,254,319	-	1,509,795,662
Total Material *Moved*	78,735,279	82,734,815	88,732,261	63,733,291	75,356,544	58,733,688	53,732,953	48,734,282	84,405,526	68,733,548	85,916,181	34,514,319	-	1,566,925,662

Appendix B

Capital Cost

![HATCH]

Project No.: 317882
Client: NOVAGOLD

GALORE CREEK PROJECT STUDY
NORTHERN CORRIDOR OPTION - 65mtpd
CAPITAL COST - PREFEASIBILITY STUDY

September 6, 2005
Rev.B

Exchange Rate US to Canadian is at x 1.25

Area	Com-mod-ity	Tag No.	Description	Qty	UOM	Labor Unit MH	Labor MH	Labor Rate	Labor Cost CAD Dollars	Material Unit Cost	Material Cost CAD Dollars	Other Unit Cost	Other Cost CAD Dollars	TOTAL COST CAD Dollars
			DIRECT COSTS - AREA SUMMARY:											
90			**PLANT SITE**											
	91		Excavation & Backfill				0		$0		$0		$3,574,750	$3,574,750
100			**PROCESS PLANT**											
	101		Primary Crushing				46295		$2,653,775		$9,205,258		$534,902	$12,393,935
	102		Coarse Ore Reclaim (40,000 tonne conical single stockpile)				69397		$3,994,735		$9,529,524		$569,032	$14,093,291
	110		Concentrator Electrical				77486		$4,417,017		$10,294,765		$1,845,521	$16,557,303
	111		Grinding				295255		$15,447,355		$78,463,393		$15,222,919	$109,133,667
	112		Flotation				114759		$5,839,428		$18,709,756		$8,689,095	$33,238,278
	113a		Concentrate Pumping & Concentrate Pipeline				451509		$1,445,954		$15,242,902		$57,536,078	$74,224,934
	113		Concentrate Dewatering				89477		$3,974,651		$10,830,849		$8,808,546	$23,614,047
	114		Reagent Handling				7089		$403,698		$1,227,420		$139,287	$1,770,405
	115		Concentrate Loadout				22831		$1,320,169		$2,239,385		$1,404,229	$4,963,784
	120		Plant site Utilities, Communications & Fire Detection				21227		$1,352,923		$1,738,409		$112,080	$3,203,412
	121		PLC & SOFTWARE				0		$0		$0		$700,000	$700,000
			Sub Total				1,195,325							**$297,467,806**
200			**INFRASTRUCTURE**											
	201		Shop & Warehouse				13200		$610,141		$708,867		$1,605,062	$2,924,071
	202		Truck Shop				62941		$3,466,291		$3,821,043		$4,727,928	$12,015,262
	203		Lab				6328		$167,782		$396,165		$946,180	$1,510,127
	204		Admin Building				11518		$316,475		$706,445		$2,509,656	$3,532,576
	205		Mine Office (Located in Truck Shop or Admin Bldg)				0		$0		$0		$0	$0
	206		Plant Access Roads, Tunnels & Bridges				6253		$413,290		$812,840		$128,430,035	$129,656,165
	207		Power Supply				48898		$3,173,463		$17,766,061		$35,465,704	$56,405,227
	208		Water Supply				22388		$1,101,599		$6,831,577		$1,137,606	$9,070,782
	209		Waste Rock/Tailings Storage				91523		$4,879,661		$16,723,440		$75,083,140	$96,686,241
	210		Water Management (Diversion Ditches/Structures etc)				0		$0		$0		$61,531,613	$61,531,613
	211		Camp				0		$0		$0		$16,748,552	$16,748,552
	212		Airstrip				0		$0		$0		$3,935,800	$3,935,800
	213		Other Buildings - Cold Storage & Scale House				13801		$847,923		$1,636,511		$56,542	$2,540,975
	214		Plant Mobile Equipment				0		$0		$0		$4,286,128	$4,286,128
	215		Port Site				0		$0		$0		$0	$0
			Sub Total											**$400,843,519**
300			**MINE**											
	300		Haul Roads (Inclds Plant Site Roads)				0		$0		$0		$6,238,373	$6,238,373
	301		Pre-stripping				0		$0		$0		$104,996,822	$104,996,822
	302		Mine Equipment				200		$10,070		$178,295,579		$0	$178,305,649
	303		Mine Dewatering (Pit Sumps & Deep Well)				0		$0		$0		$10,000,000	$10,000,000
	304		Mine Electrical				2066		$136,563		$3,727,900		$0	$3,864,463
	305		Explosives Factory				0		$0		$0		$0	$0
	306		Magazine				925		$0		$0		$250,000	$250,000
	307		Fuel Storage & Dispensing				11326		$603,639		$899,981		$289,459	$1,793,079
			Sub Total											**$305,448,385**
			TOTAL PROJECT DIRECT COST (CAD DOLLARS)				2682017		$56,576,603		$389,808,070		$557,375,038	**$1,003,759,711**

GALORE CREEK PROJECT STUDY
NORTHERN CORRIDOR OPTION
CAPITAL COST - PREFEASIBILITY STUDY

Project No.: 317882
Client: NOVAGOLD

Area	Com-mod-ity	Tag No.	Description	Qty	UOM	Labor Unit MH	Labor MH	Labor Rate	Labor Cost CAD Dollars	Material Unit Cost	Material Cost CAD Dollars	Other Unit Cost	Other Cost CAD Dollars	TOTAL COST CAD Dollars
			INDIRECT COSTS - AREA SUMMARY:											
900			**INDIRECT COSTS**											
920			**EPCM**				0		$0		$0		$87,083,392	$87,083,392
930			**CONSTRUCTION INDIRECTS**				486832		$25,487,897		$200,000		$82,850,702	$108,538,599
940			**COMMISSIONING, START-UP, & VENDOR REPS**				16770		$1,115,040		$9,000		$913,391	$2,037,431
950			**SPARES**				0		$0		$16,442,330		$0	$16,442,330
960			**FIRST FILL**				0		$0		$5,248,072		$0	$5,248,072
970			**FREIGHT**				0		$0		$0		$23,599,670	$23,599,670
980			**OWNERS COSTS**				0		$0		$0		$30,000,000	$30,000,000
990			**TAXES/DUTIES - NOT INCLUDED**				0		$0		$0		$0	$0
			TOTAL PROJECT INDIRECT COST (CAD DOLLARS)				503602		$26,602,937		$21,899,401		$224,447,155	**$272,949,493**
			TOTAL PROJECT DIRECTS & INDIRECTS											**$1,276,709,204**
999			**TOTAL CONTINGENCY (% OF DIRECT & INDIRECT COSTS)**	**14.9%**			0		$0		$0		$190,560,976	**$190,560,976**
			TOTAL PROJECT COST (CAD DOLLARS)											**$1,467,270,180**

Appendix C

Operating Costs

<u>SUMMARY OPERATING COSTS</u> **(North Route with concentrate pipeline)**

Avg daily throughput	**65,000**	tpd
Annual Throughput	23,725,000	tpa
Avg availability	92%	
Avg strip ratio, LOM	2.4	
Exchange rate, U$/C$	0.75	

		$/t moved	C$pa	C$/t ore	U$pa	U$/t ore
G&A	labour		4,140,259	0.17	3,105,194	0.13
	fixed		16,361,863	0.69	12,271,397	0.52
	s/total		20,502,122	0.86	15,376,591	0.65
Mining	mining (LOM) *		95,924,920	4.04	71,943,690	3.03
Process & Infrastructure	labour		12,010,155	0.51	9,007,616	0.38
	consumables		50,018,300	2.11	37,513,725	1.58
	power		22,787,183	0.96	17,090,387	0.72
	tailings		711,750	0.03	533,813	0.02
	s/total		85,527,388	3.60	64,145,541	2.70
Contingency	0%		-	0.00	0	0.00
Mine site			**201,954,430**	**8.51**	**151,465,822**	**6.38**
Conc	**transport to portsite**		**13,175,647**	**0.56**	**9,881,735**	**0.42**

Notes: * $/t ore milled is based on Mining Model Version 23

Appendix D

Cashflow Schedule

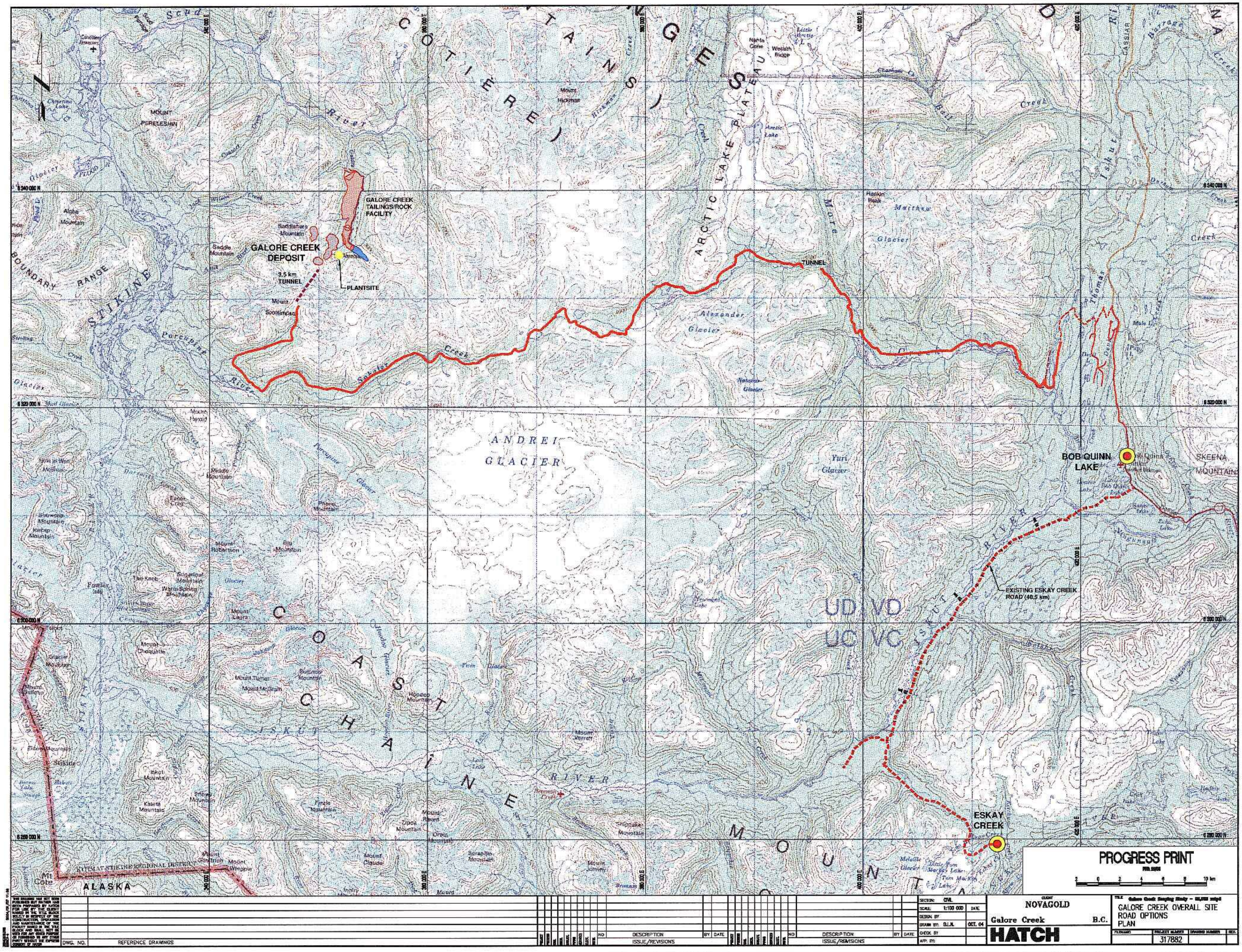
Galore Creek Mine
Northern Option: 65,000 tpd

	Pre-Tax	Post -Tax
IRR	11.11%	8.10%
0% NPV	$1,343.3	$783.0
5% NPV	$455.0	$190.6
Payback Yr	5.2	

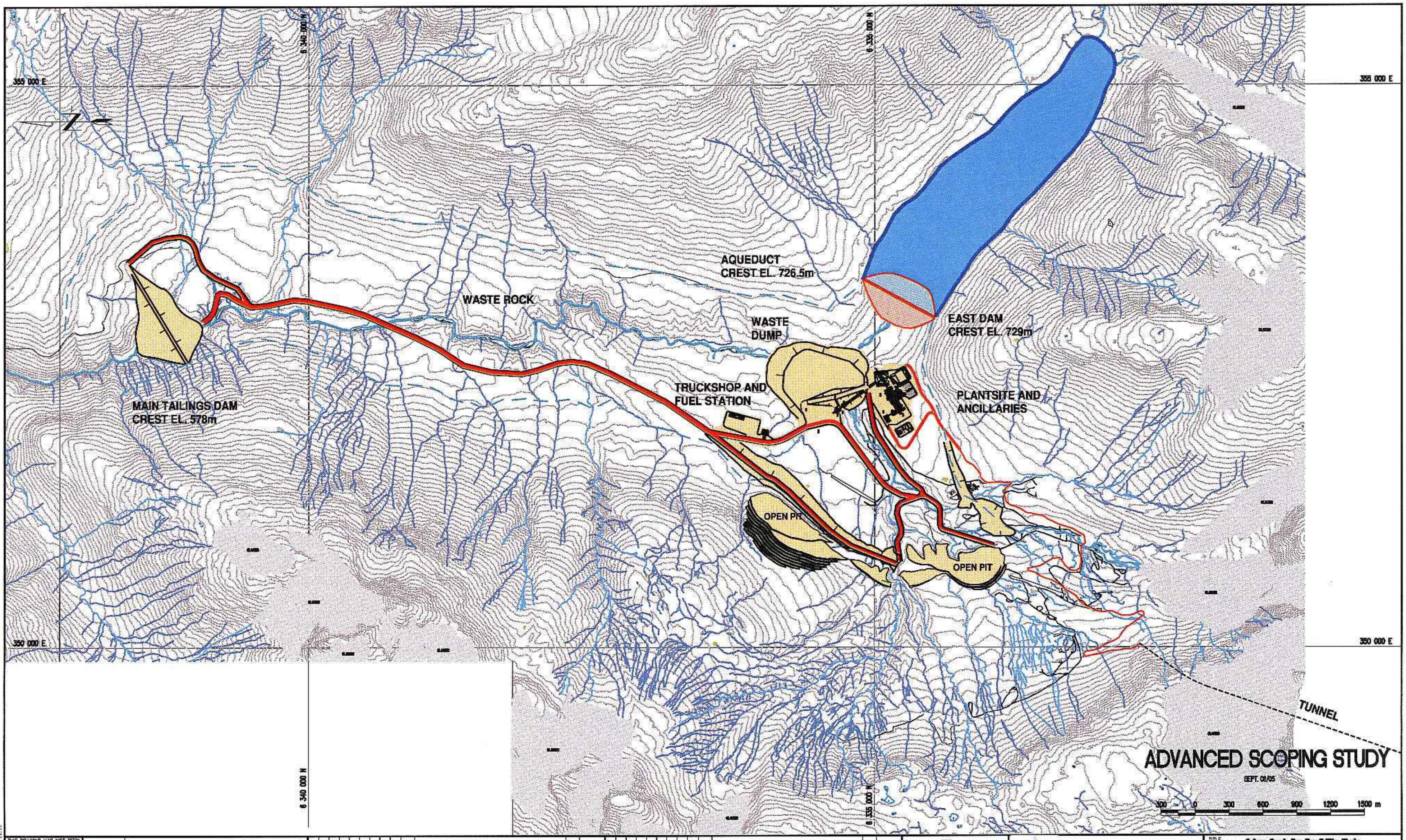
				-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	TOTAL
Metal Prices	GOLD	US$ / oz	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400
	COPPER	US$ / lb	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00
	SILVER	US$ / oz	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00	6.00
	US/C$	0.75	0.75																										
Production Schedule																													
Average Milling Rate		t / day						65,027	65,027	65,027	65,027	65,027	65,027	65,027	65,027	65,028	65,027	65,028	65,028	65,028	65,027	65,027	65,028	65,027	65,027	65,029	61,858	-	
Tons of Ore Milled		'000 tpa						23,735,018	23,735,028	23,734,976	23,734,992	23,735,000	23,735,029	23,735,005	23,735,028	23,735,282	23,734,872	23,735,042	23,735,346	23,735,221	23,734,985	23,735,007	23,735,302	23,735,020	23,734,993	23,735,428	22,577,999	-	473,544,573
Mining Strip Ratio	2.2							1.81	2.63	3.67	2.07	2.20	2.19	2.30	2.38	2.17	2.37	2.57	1.62	2.03	1.29	1.19	1.01	2.30	1.72	2.47	0.48	-	2.19
Tonnes - Ore mined		tonnes				61,129	61,129	25,687,428	26,359,656	26,515,047	29,499,675	26,527,971	26,647,772	24,160,771	14,357,693	18,557,003	22,437,857	27,810,560	24,197,989	22,762,146	28,046,849	25,520,226	12,895,480	29,775,100	27,986,165	21,357,824	13,416,837	-	474,642,306
Tonnes - Ore to SP						61,129	61,129	1,952,410	2,624,628	2,780,071	5,764,682	2,792,971	2,912,743	425,766	1,504,665	3,404,720	2,865,985	4,075,518	1,459,643	3,327,925	4,311,865	1,785,219	1,092,177	6,040,080	4,251,172	3,634,396	1,098,838	-	58,227,732
Tonnes - Ore from SP						-	-	-	-	-	-	-	-	-	10,882,000	8,583,000	4,163,000	-	997,000	4,301,000	-	-	11,932,000	-	-	6,012,000	10,260,000	-	57,130,000
Tonnes - Waste mined		tonnes				37,437,736	37,437,736	43,047,602	62,374,391	87,214,947	49,234,170	52,205,819	52,085,192	54,570,853	56,494,257	51,595,277	56,133,959	60,921,701	38,538,302	48,293,398	30,686,839	28,212,727	23,906,802	54,630,426	40,747,383	58,546,356	10,837,482	-	1,035,153,356
Total Tonnes Mined		tonnes				37,498,865	37,498,865	68,735,030	88,734,047	113,729,994	78,733,845	78,733,791	78,732,964	78,731,625	70,851,950	70,152,279	78,571,815	88,732,261	62,736,291	71,055,544	58,733,688	53,732,953	36,802,282	84,405,526	68,733,548	79,904,181	24,254,319		1,509,795,662
Total Tonnes Moved		tonnes				37,498,865	37,498,865	68,735,030	88,734,047	113,729,994	78,733,845	78,733,791	78,732,964	78,731,625	81,733,950	78,735,279	82,734,815	88,732,261	63,733,291	75,356,544	58,733,688	53,732,953	48,734,282	84,405,526	68,733,548	85,916,181	34,514,319	-	1,566,925,662
Cu Grade (mill)								0.731%	0.939%	0.962%	0.626%	0.715%	0.790%	0.803%	0.493%	0.476%	0.601%	0.682%	0.632%	0.531%	0.662%	0.746%	0.582%	0.512%	0.494%	0.449%	0.499%		0.647%
Contained Copper		tonnes						173,621	222,900	228,230	148,599	169,647	187,548	190,686	117,000	112,979	142,620	161,892	149,945	125,979	157,231	177,091	138,057	121,432	117,224	106,510	112,669	-	3,061,861
Copper Recovery		%						91.8%	93.6%	93.8%	90.4%	91.6%	92.4%	92.5%	87.8%	87.4%	90.0%	91.2%	90.5%	88.7%	90.9%	92.0%	89.7%	88.3%	87.9%	86.6%	88.0%		91%
Recovered Copper		tonnes						159,380	208,659	213,989	134,358	155,406	173,307	176,445	102,759	98,738	128,379	147,651	135,704	111,738	142,990	162,850	123,816	107,191	102,983	92,268	99,122		2,777,734
Conc. Copper Grade		0.70%						30%	30%	30%	28%	30%	30%	30%	28%	28%	28%	28%	28%	28%	28%	30%	28%	28%	28%	28%	28%		
Conc. Produced, Dry		tonnes						531,267	695,531	713,298	479,850	518,020	577,690	588,151	366,995	352,636	458,496	527,325	484,656	399,065	510,680	542,833	442,200	382,825	367,797	329,530	354,008		9,622,852
Conc. Produced, Wet		tonnes						576,424	754,651	773,928	520,637	562,052	626,794	638,144	398,190	382,610	497,468	572,148	525,851	432,985	554,088	588,974	479,787	415,365	399,060	357,540	384,099		10,440,795
Payable Copper		%						96.7%	96.7%	96.7%	96.4%	96.7%	96.7%	96.7%	96.4%	96.4%	96.4%	96.4%	96.4%	96.4%	96.4%	96.7%	96.4%	96.4%	96.4%	96.4%	96.4%		96.5%
Payable Copper		tonnes						154,067	201,704	206,856	129,559	150,226	167,530	170,564	99,089	95,212	123,794	142,378	130,857	107,747	137,884	157,422	119,394	103,363	99,305	88,973	95,582	-	2,681,506
Gold Grade		g / tonne						1.072	0.316	0.344	0.664	0.480	0.500	0.371	0.286	0.237	0.266	0.260	0.225	0.174	0.247	0.303	0.285	0.130	0.186	0.423	0.425		0.360
Contained Gold		ozs						818,122	241,016	262,290	506,474	366,356	381,605	283,433	218,399	181,035	202,719	198,712	171,681	132,976	188,856	231,461	217,704	99,377	141,975	322,604	308,468		5,475,262
Gold Recovery		%						72.3%	74.5%	74.7%	70.7%	72.1%	73.1%	73.2%	67.7%	67.2%	70.2%	71.6%	70.8%	68.7%	71.3%	72.5%	69.8%	68.2%	67.7%	66.4%	67.9%		71.0%
Recovered Gold		ozs						591,793	179,668	196,012	358,057	264,162	278,841	207,536	147,906	121,723	142,362	142,328	121,544	91,361	134,685	167,876	152,049	67,799	96,187	214,171	209,418		3,885,479
Payable Gold		ozs						574,040	174,278	190,131	347,315	256,237	270,476	201,310	143,469	118,072	138,091	138,059	117,898	88,620	130,645	162,840	147,488	65,765	93,301	207,746	203,136		3,768,915
Silver Grade		g / tonne						3.98	6.27	6.76	3.69	4.54	5.86	7.34	4.17	2.93	4.03	4.76	4.02	4.20	5.45	6.18	4.80	4.85	5.05	2.87	3.30		4.76
Contained Silver		ozs						3,034,143	4,784,908	5,155,710	2,815,874	3,461,147	4,474,868	5,601,814	3,182,107	2,232,768	3,074,500	3,630,561	3,067,318	3,208,684	4,157,086	4,715,382	3,661,467	3,703,528	3,850,136	2,192,642	2,394,789		72,399,434
Silver Recovery								63.8%	66.8%	67.0%	61.6%	63.5%	64.8%	65.0%	57.8%	57.2%	61.0%	62.9%	61.8%	59.0%	62.5%	64.1%	60.5%	58.4%	57.8%	56.1%	58.0%		62%
Recovered Silver		ozs						1,936,098	3,194,873	3,455,322	1,735,836	2,197,988	2,899,243	3,640,743	1,838,687	1,276,241	1,876,326	2,282,511	1,894,902	1,894,693	2,596,554	3,021,017	2,216,175	2,163,876	2,225,986	1,229,539	1,388,898		44,965,507
Payable Silver		ozs						1,742,488	2,875,385	3,109,790	1,562,253	1,978,189	2,609,319	3,276,668	1,654,819	1,148,617	1,688,694	2,054,260	1,705,412	1,705,223	2,336,899	2,718,915	1,994,557	1,947,489	2,003,387	1,106,586	1,250,009		40,468,957
Copper Eq (Cu +Au +Ag)%								**1.37%**	**1.16%**	**1.21%**	**1.03%**	**1.02%**	**1.12%**	**1.07%**	**0.68%**	**0.63%**	**0.78%**	**0.86%**	**0.79%**	**0.66%**	**0.84%**	**0.96%**	**0.78%**	**0.62%**	**0.63%**	**0.70%**	**0.76%**		**0.88%**
				US$'000	US$'000	US$'000	US$'000	US$'000	US$'000	US$'000	US$'000	US$'000	US$'000	US$'000	US$'000	US$'000	US$'000	US$'000	US$'000	US$'000	US$'000	US$'000	US$'000	US$'000	US$'000	US$'000	US$'000		
Cashflow Schedule																													
Gross Revenue																													
Copper								339,657	444,677	456,036	285,627	331,188	369,337	376,025	218,451	209,904	272,916	313,886	288,487	237,540	303,978	347,052	263,216	227,874	218,928	196,150	210,720		5,911,648
Gold								229,616	69,711	76,053	138,926	102,495	108,190	80,524	57,387	47,229	55,236	55,223	47,159	35,448	52,258	65,136	58,995	26,306	37,320	83,098	81,254		1,507,566
Silver								10,455	17,252	18,659	9,374	11,869	15,656	19,660	9,929	6,892	10,132	12,326	10,232	10,231	14,021	16,313	11,967	11,685	12,020	6,640	7,500		242,814
Total Gross Revenue								579,728	531,640	550,747	433,926	445,552	493,183	476,209	285,767	264,024	338,284	381,435	345,879	283,220	370,258	428,501	334,178	265,864	268,269	285,888	299,475		7,662,028
Royalties	0%							0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0		-
Treatment / Refining Charges								72,570	92,071	94,550	62,802	69,203	77,252	78,491	47,544	45,398	59,020	67,811	62,201	51,290	65,792	72,195	57,147	49,222	47,508	42,973	46,079		1,261,121
Total Net Revenue								507,157	439,569	456,197	371,124	376,349	415,931	397,718	238,223	218,626	279,264	313,624	283,678	231,929	304,465	356,306	277,031	216,643	220,761	242,915	253,396		6,400,907
Cash Operating Costs								**176,975**	**194,844**	**207,043**	**182,664**	**185,218**	**194,787**	**193,456**	**184,776**	**180,159**	**192,552**	**211,711**	**180,965**	**188,130**	**189,811**	**187,795**	**174,732**	**188,559**	**177,339**	**177,673**	**140,619**		**3,709,807**
Cash Operating Costs/ Tonne Ore		US$/t						$7.46	$8.21	$8.72	$7.70	$7.80	$8.21	$8.15	$7.78	$7.59	$8.11	$8.92	$7.62	$7.93	$8.00	$7.91	$7.36	$7.94	$7.47	$7.49	$6.23		$7.83
Project Operating Cost/ Tonne Ore		US$/t						$10.51	$12.09	$12.71	$10.34	$10.72	$11.46	$11.46	$9.79	$9.50	$10.60	$11.78	$10.24	$10.09	$10.77	$10.95	$9.77	$10.02	$9.47	$9.30	$8.27		$10.50
Total Cash Cost Per Copper Lb. (Au,Ag as Credits)								$0.03	$0.45	$0.45	$0.34	$0.42	$0.40	$0.46	$0.76	$0.82	$0.68	$0.68	$0.64	$0.82	$0.62	$0.51	$0.61	$0.88	$0.80	$0.67	$0.46		$0.54
Net Minesite Cashflow		$M						402,752	336,796	343,704	251,262	260,334	298,397	282,754	100,991	83,865	145,732	169,724	164,914	95,090	180,446	240,706	159,446	77,306	90,930	108,214	158,856		3,952,220
Net Cash Flow before tax		$M						**330,182**	**244,725**	**249,154**	**188,460**	**191,131**	**221,145**	**204,262**	**53,447**	**38,467**	**86,712**	**101,913**	**102,713**	**43,800**	**114,654**	**168,511**	**102,299**	**28,084**	**43,422**	**65,242**	**112,777**		**2,691,100**
Construction Capital				168,056	173,173	267,330	493,820																						1,102,379
Sustaining Capital									14,925	36,770	11,810	11,810	13,460	11,810	15,935	13,310	23,735	15,973	13,490	11,960	11,810	13,460	11,810	11,810	1,500	0	0		245,378
Working capital								50,000																			-50,000		0
Total CapEx				168,056	173,173	267,330	493,820	50,000	14,925	36,770	11,810	11,810	13,460	11,810	15,935	13,310	23,735	15,973	13,490	11,960	11,810	13,460	11,810	11,810	1,500	0	-50,000	0	1,347,756
Net Cashflow, pre-tax				-168,056	-173,173	-267,330	-493,820	280,182	229,800	212,384	176,650	179,321	207,685	192,452	37,512	25,157	62,977	85,941	89,223	31,840	102,844	155,051	90,489	16,274	41,922	65,242	162,777		1,343,343
Cumulative cashflow, pre-tax				-168,056	-341,228	-608,559	-1,102,379	-822,197	-592,397	-380,013	-203,363	-24,042	183,643	376,095	413,608	438,765	501,742	587,683	676,906	708,745	811,589	966,640	1,057,129	1,073,403	1,115,325	1,180,566	1,343,343	1,343,343	1,343,343
Tax paid,total				0	0	0	0	0	11,498	4,894	3,858	3,769	6,190	136,753	28,860	9,199	12,435	56,258	38,780	14,092	12,626	82,423	56,286	6,620	6,620	23,445	35,682	10,014	560,303
Net Cashflow, after tax				-168,056	-173,173	-267,330	-493,820	280,182	218,302	207,490	172,792	175,552	201,495	55,700	8,652	15,958	50,542	29,683	50,443	17,748	90,218	72,627	34,203	9,654	35,302	41,796	127,095	-10,014	783,040
Cumulative cashflow, after tax				**-168,056**	**-341,228**	**-608,559**	**-1,102,379**	**-822,197**	**-603,895**	**-396,405**	**-223,613**	**-48,061**	**153,434**	**209,133**	**217,786**	**233,744**	**284,286**	**313,968**	**364,411**	**382,159**	**472,377**	**545,004**	**579,208**	**588,862**	**624,163**	**665,959**	**793,055**	**783,040**	**783,040**
Operating Costs																													
Mining Cost		US$Mpa				39,341	39,351	59,787	66,021	76,961	69,117	68,967	74,310	72,238	79,223	75,622	80,519	94,801	67,077	80,305	74,081	69,787	63,852	81,884	71,729	74,773	37,828		1,438,883
Total Concentrator		US$Mpa						64,185	64,185	64,185	64,185	64,185	64,185	64,185	64,185	64,185	64,184	64,185	64,185	64,185	64,184	64,185	64,185	64,184	64,185	64,185	62,341		1,281,850
Administration Cost		US$Mpa						15,375	15,375	15,375	15,375	15,375	15,375	15,375	15,375	15,375	15,375	15,375	15,375	15,375	15,375	15,375	15,375	15,375	15,375	15,375	15,375		307,500
Total Plantsite Cash Cost		US$Mpa						139,346	145,581	156,521	148,677	148,527	153,870	151,798	158,782	155,182	160,078	174,361	146,637	159,864	153,640	149,347	143,412	161,444	151,288	154,333	115,545		3,028,233
Total Plantsite Cash Unit Cost								5.87	6.13	6.59	6.26	6.26	6.48	6.40	6.69	6.54	6.74	7.35	6.18	6.74	6.47	6.29	6.04	6.80	6.37	6.50	5.12		
Road, Port & Marketing Cost		US$Mpa						17,454	22,851	23,435	15,765	17,019	18,979	19,323	12,057	11,585	15,063	17,325	15,923	13,111	16,778	17,834	14,528	12,577	12,084	10,826	11,630		316,147
Sea Freight		US$Mpa						20,175	26,413	27,087	18,222	19,672	21,938	22,335	13,937	13,391	17,411	20,025	18,405	15,154	19,393	20,614	16,793	14,538	13,967	12,514	13,443		365,428
Total Operating Cost		US$Mpa						176,975	194,844	207,043	182,664	185,218	194,787	193,456	184,776	180,159	192,552	211,711	180,965	188,130	189,811	187,795	174,732	188,559	177,339	177,673	140,619		3,709,807
Total Operating (Site & Transp)		US$/t ore						7.46	8.21	8.72	7.70	7.80	8.21	8.15	7.78	7.59	8.11	8.92	7.62	7.93	8.00	7.91	7.36	7.94	7.47	7.49	6.23		7.83

Appendix E

Drawings



GALORE CREEK DEPOSIT
GALORE CREEK TAILINGS/ROCK FACILITY
PLANTSITE
3.5 km TUNNEL
TUNNEL
BOB QUINN LAKE
ESKAY CREEK
EXISTING ESKAY CREEK ROAD (60.5 km)
ANDREI GLACIER
ARCTIC LAKE PLATEAU
STIKINE
BOUNDARY RANGE
ISKUT RIVER
ALASKA
SKEENA MOUNTAINS
PROGRESS PRINT
NOVAGOLD
Galore Creek
B.C.
HATCH
GALORE CREEK OVERALL SITE
ROAD OPTIONS
PLAN
317882
REFERENCE DRAWINGS
DESCRIPTION
ISSUE/REVISIONS



THIS DRAWING HAS NOT BEEN PUBLISHED BUT RATHER HAS BEEN PREPARED BY HATCH FOR USE BY THE CLIENT NAMED IN THE TITLE BLOCK SOLELY IN RESPECT OF THE CONSTRUCTION, OPERATION AND MAINTENANCE OF THE FACILITY NAMED IN THE TITLE BLOCK AND SHALL NOT BE USED FOR ANY OTHER PURPOSE OR FURNISHED TO ANY OTHER PARTY WITHOUT THE EXPRESS CONSENT OF HATCH

DWG. NO.	REFERENCE DRAWINGS

NO	DESCRIPTION	BY	DATE
	ISSUE		

NO	DESCRIPTION	BY	DATE
A	ISSUED FOR ADVANCED SCOPING STUDY	DN	05/09/08
	REVISIONS		

SECTION:	CIVIL	
SCALE:	1:15 000	DATE
DESIGN. BY		
DRAWN BY:	D.L.N.	AUG. 05
CHECK. BY		
APP. BY:		

NovaGold

Galore Creek B.C.

HATCH

TITLE Galore Creek Pre Feasibility Study

MINE SITE AREA
YEAR AT STARTUP
PLAN

FILENAME:	PROJECT NUMBER	DRAWING NUMBER	REV.
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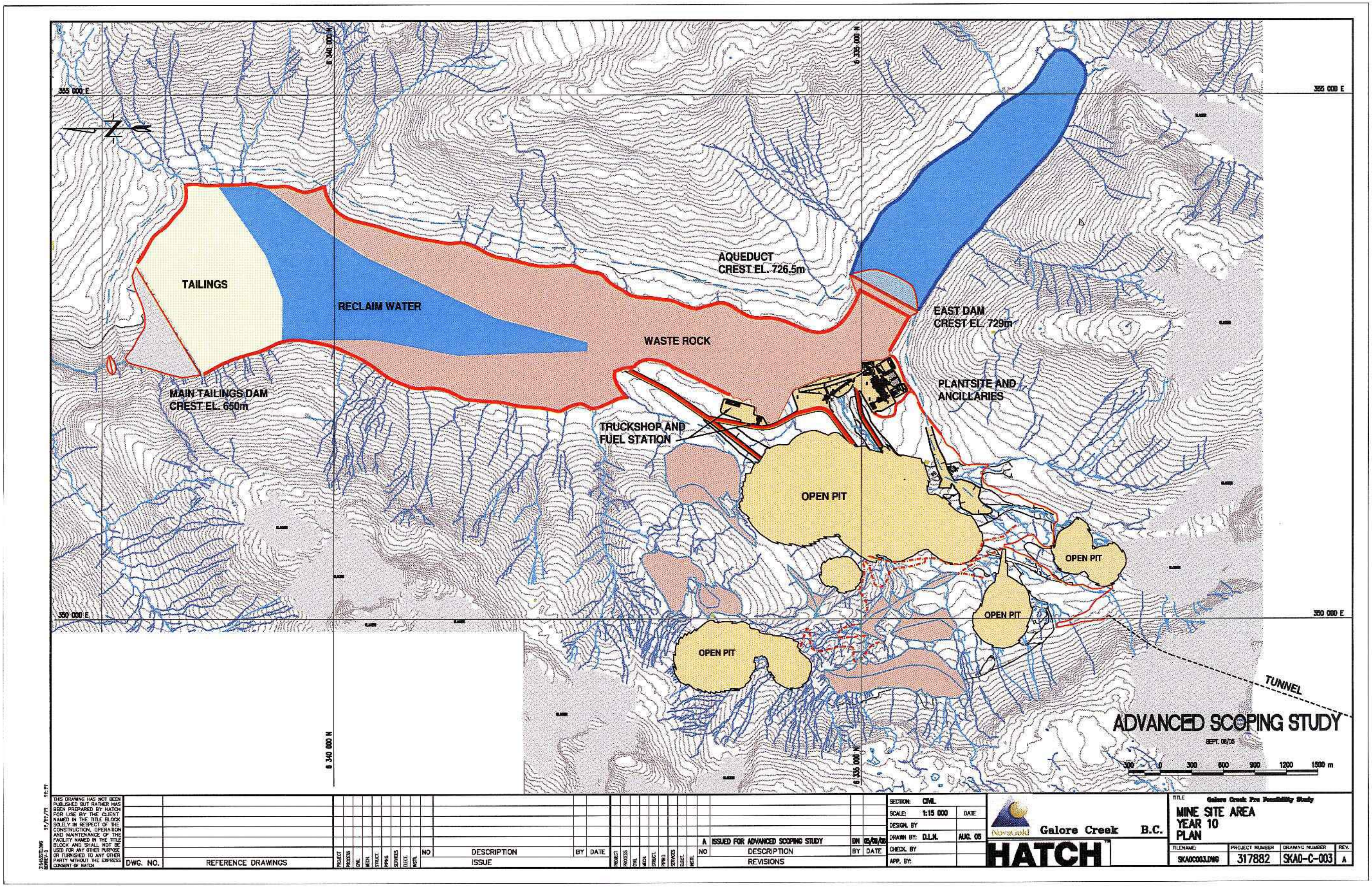
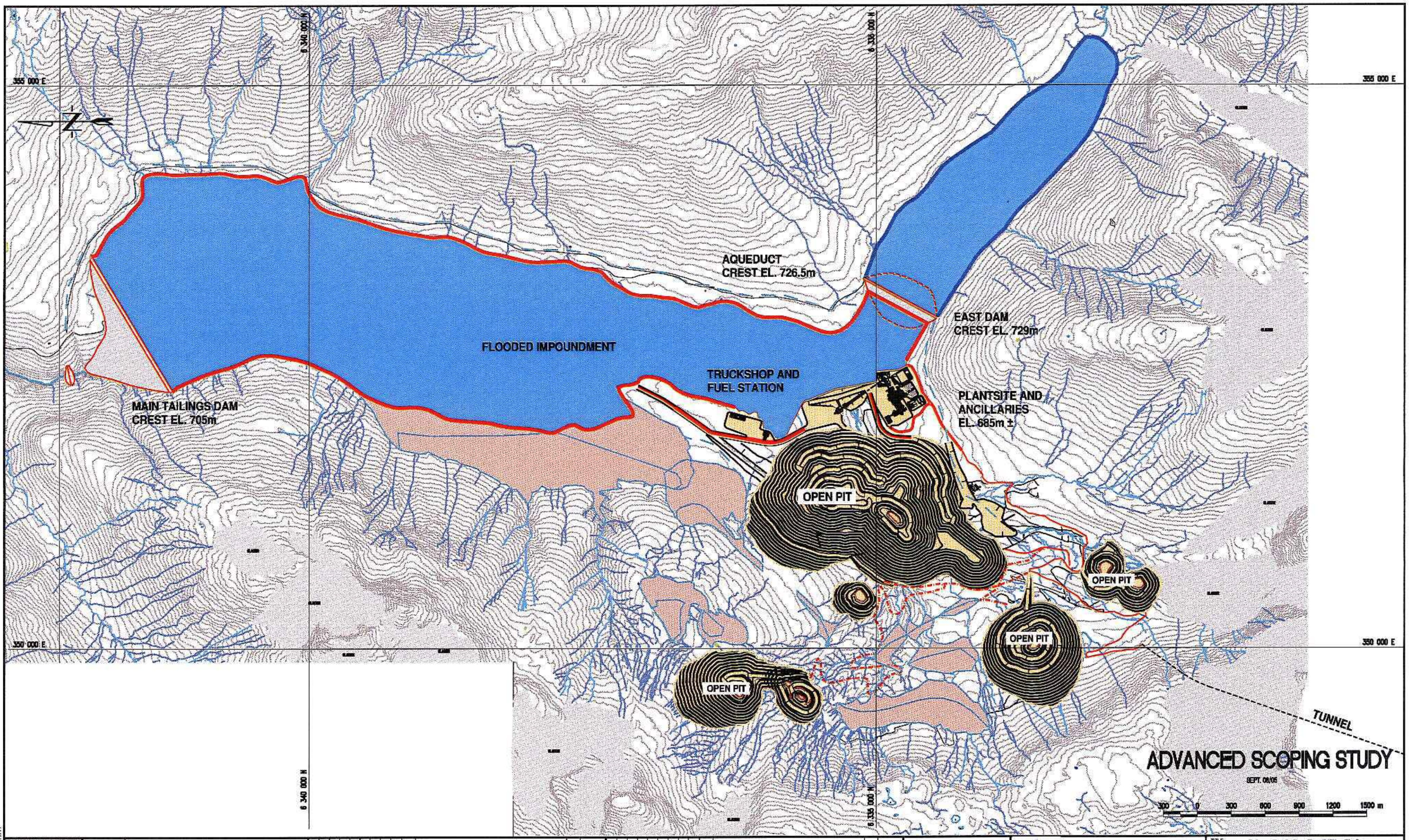

TAILINGS
RECLAIM WATER
WASTE ROCK
AQUEDUCT
CREST EL. 726.5m
EAST DAM
CREST EL. 729m
PLANTSITE AND
ANCILLARIES
MAIN TAILINGS DAM
CREST EL. 650m
TRUCKSHOP AND
FUEL STATION
OPEN PIT
OPEN PIT
OPEN PIT
OPEN PIT
TUNNEL
355 000 E
350 000 E
6 340 000 N
6 335 000 N
ADVANCED SCOPING STUDY
SEPT. 08/05
300 0 300 600 900 1200 1500 m
THIS DRAWING HAS NOT BEEN PUBLISHED BUT RATHER HAS BEEN PREPARED BY HATCH FOR USE BY THE CLIENT NAMED IN THE TITLE BLOCK SOLELY IN RESPECT OF THE CONSTRUCTION, OPERATION AND MAINTENANCE OF THE FACILITY NAMED IN THE TITLE BLOCK AND SHALL NOT BE USED FOR ANY OTHER PURPOSE OR FURNISHED TO ANY OTHER PARTY WITHOUT THE EXPRESS CONSENT OF HATCH
DWG. NO.
REFERENCE DRAWINGS
NO
DESCRIPTION
ISSUE
BY
DATE
A
ISSUED FOR ADVANCED SCOPING STUDY
DN
05/08/05
DESCRIPTION
REVISIONS
SECTION: CIVIL
SCALE: 1:15 000
DESIGN. BY
DRAWN BY: D.L.H.
AUG. 05
CHECK. BY
APP. BY:
NovaGold
Galore Creek B.C.
HATCH
TITLE
Galore Creek Pre Feasibility Study
MINE SITE AREA
YEAR 10
PLAN
FILENAME: SKA0C003.DWG
PROJECT NUMBER 317882
DRAWING NUMBER SKA0-C-003
REV. A



THIS DRAWING HAS NOT BEEN PUBLISHED BUT RATHER HAS BEEN PREPARED BY HATCH FOR USE BY THE CLIENT NAMED IN THE TITLE BLOCK SOLELY IN RESPECT OF THE CONSTRUCTION, OPERATION AND MAINTENANCE OF THE FACILITY NAMED IN THE TITLE BLOCK AND SHALL NOT BE USED FOR ANY OTHER PURPOSE OR FURNISHED TO ANY OTHER PARTY WITHOUT THE EXPRESS CONSENT OF HATCH

DWG. NO.	REFERENCE DRAWINGS

NO	DESCRIPTION	BY	DATE
	ISSUE		

NO	DESCRIPTION	BY	DATE
A	ISSUED FOR ADVANCED SCOPING STUDY	DN	05/08/08
	REVISIONS		

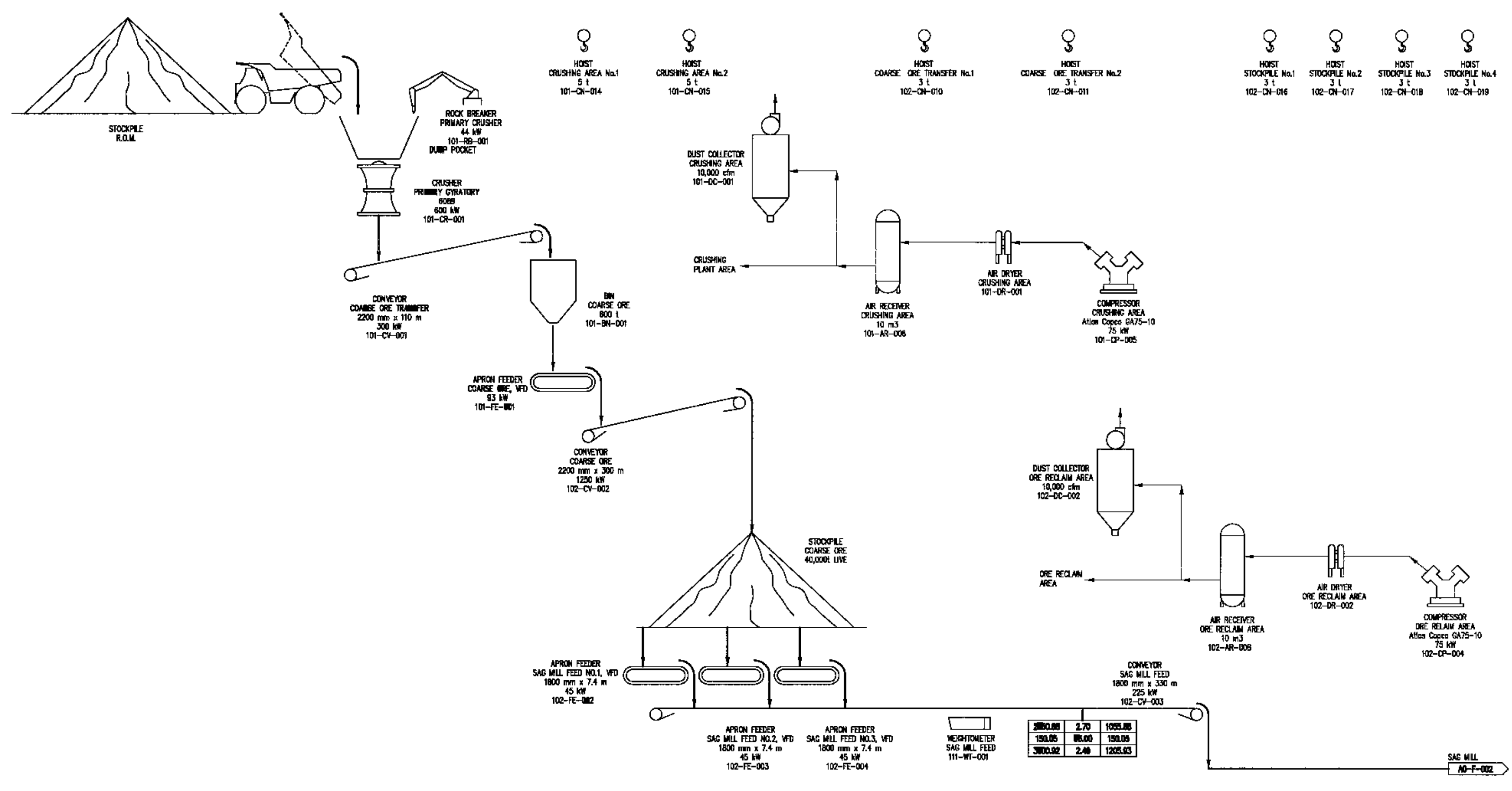
SECTION: CIVIL
SCALE: 1:15 000
DESIGN. BY
DRAWN BY: D.L.N. DATE AUG. 05
CHECK. BY
APP. BY:

NovaGold Galore Creek B.C.

HATCH™

TITLE: Galore Creek Pre Feasibility Study
MINE SITE AREA
YEAR END (20 YRS)
PLAN

FILENAME:	PROJECT NUMBER	DRAWING NUMBER	REV.
SKA0C004.DWG	317882	SKA0-C-004	A



ADVANCED SCOPING STUDY

THIS DRAWING HAS NOT BEEN PUBLISHED BUT RATHER HAS BEEN PREPARED BY HATCH FOR USE BY THE CLIENT NAMED IN THE TITLE BLOCK SOLELY IN RESPECT OF THE CONSTRUCTION, OPERATION AND MAINTENANCE OF THE FACILITY NAMED IN THE TITLE BLOCK AND SHALL NOT BE USED FOR ANY OTHER PURPOSE OR FURNISHED TO ANY OTHER PARTY WITHOUT THE EXPRESS CONSENT OF HATCH

DWG. NO.	REFERENCE DRAWINGS	NO	DESCRIPTION	BY	DATE	NO	DESCRIPTION	BY	DATE
			ISSUE				REVISIONS		

SECTION: GENERAL
SCALE: NONE
DESIGN BY: MT — OCT/04
DRAWN BY: JSL — OCT/04
CHECK. BY:
APP. BY:

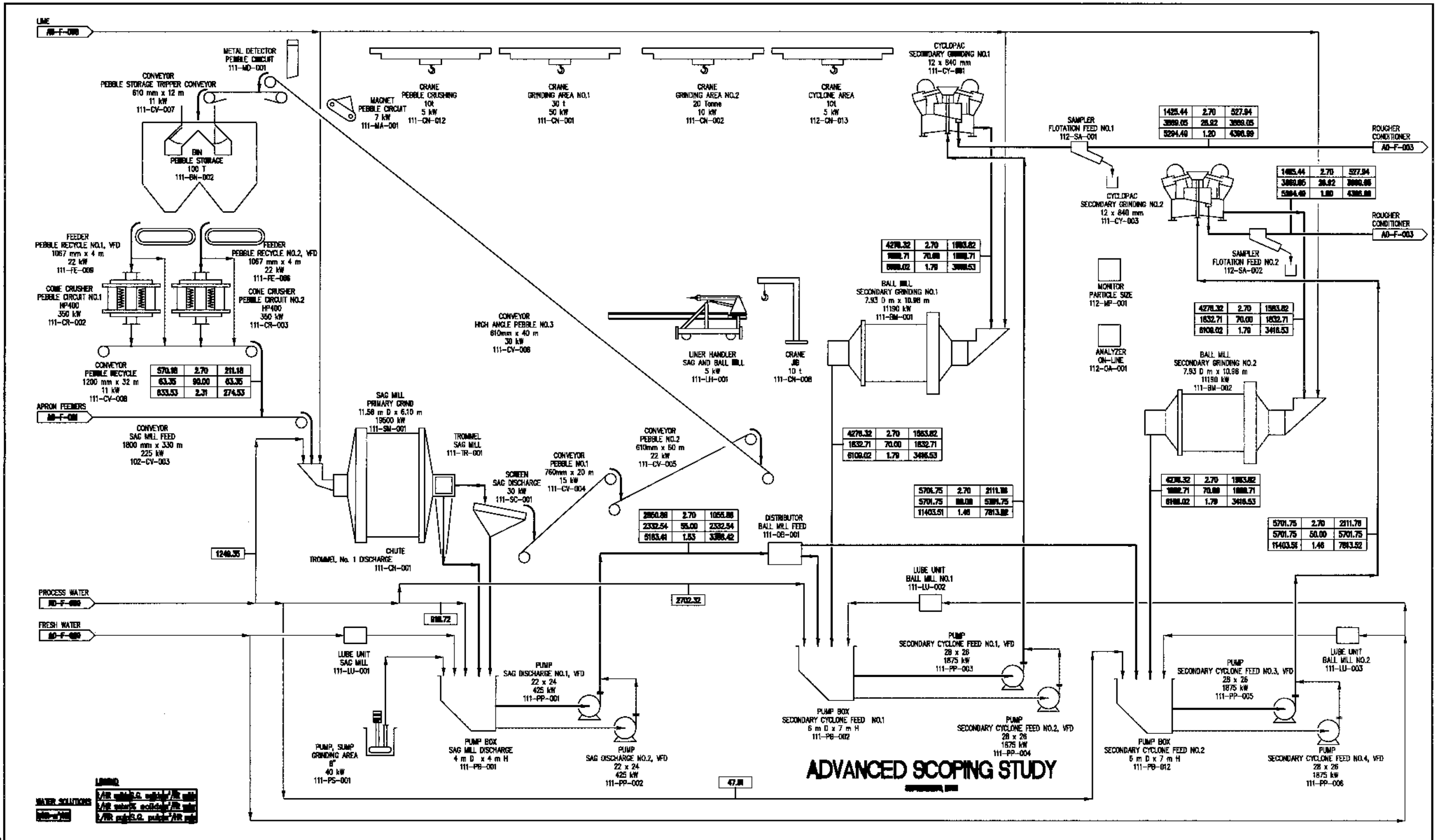
Galore Creek B.C.

HATCH

TITLE: GENERAL CRUSHING FLOWSHEET NO. 1

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
	317882	AO-F-001	/

2005/07/11 14:15



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3869.05	26.92	3869.05
5294.49	1.20	4396.99

570.18	2.70	211.18
63.35	90.00	63.35
633.53	2.31	274.53

4276.32	2.70	1583.82
1832.71	70.00	1832.71
6109.02	1.79	3416.53

5701.75	2.70	2111.76
5701.75	50.00	5701.75
11403.51	1.46	7813.52

2850.88	2.70	1055.88
2332.54	55.00	2332.54
5183.41	1.53	3388.42

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DWG. NO.	REFERENCE DRAWINGS	NO	DESCRIPTION	BY	DATE	NO	DESCRIPTION	BY	DATE
			ISSUE				REVISIONS		

SECTION:	GENERAL	
SCALE:	NONE	DATE
DESIGN. BY	HT	OCT/04
DRAWN BY:	JSL	OCT/04
CHECK. BY		
APP. BY:		

NovaGold Galore Creek B.C.

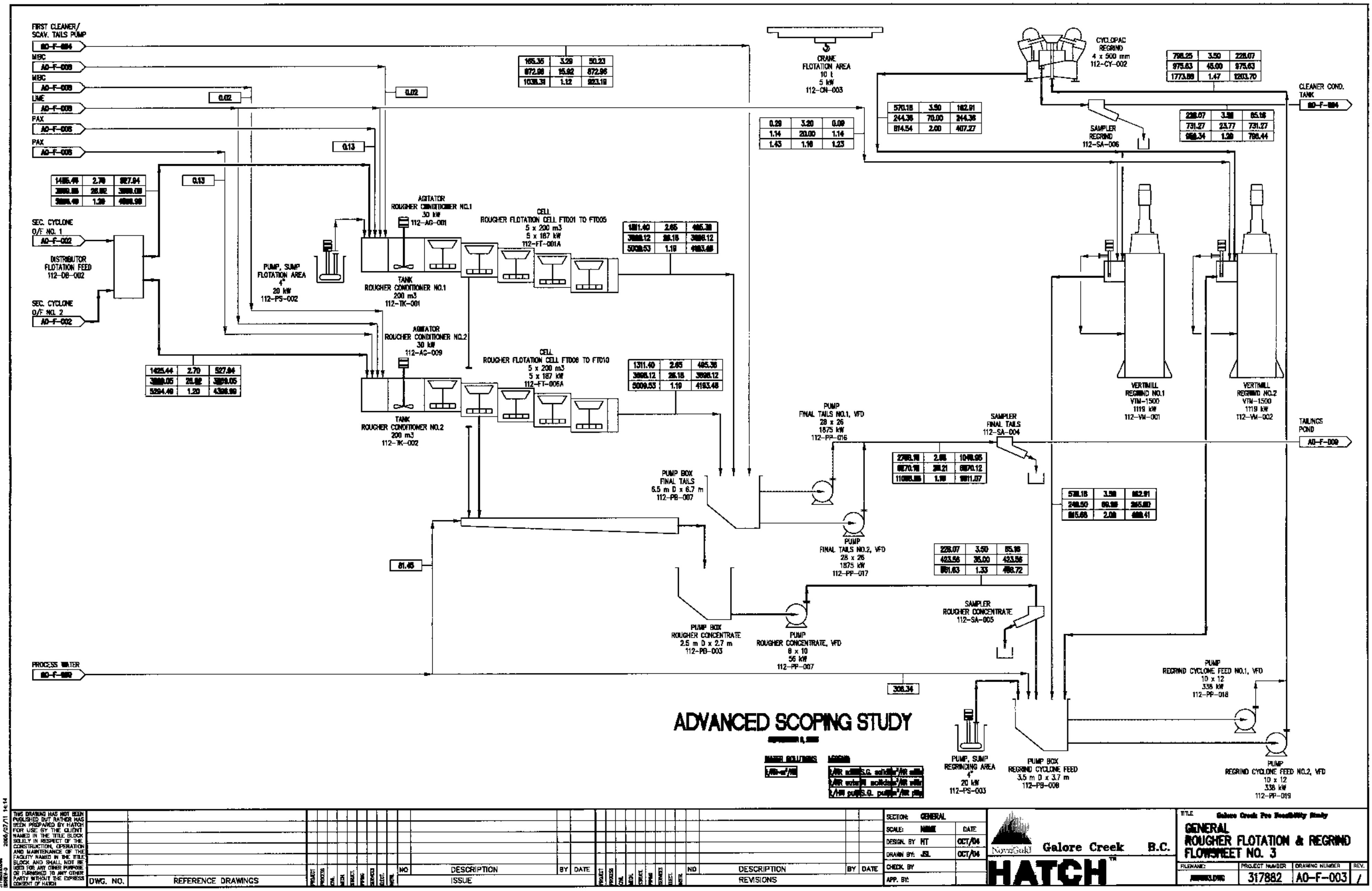
HATCH

TITLE Galore Creek Pre Feasibility Study

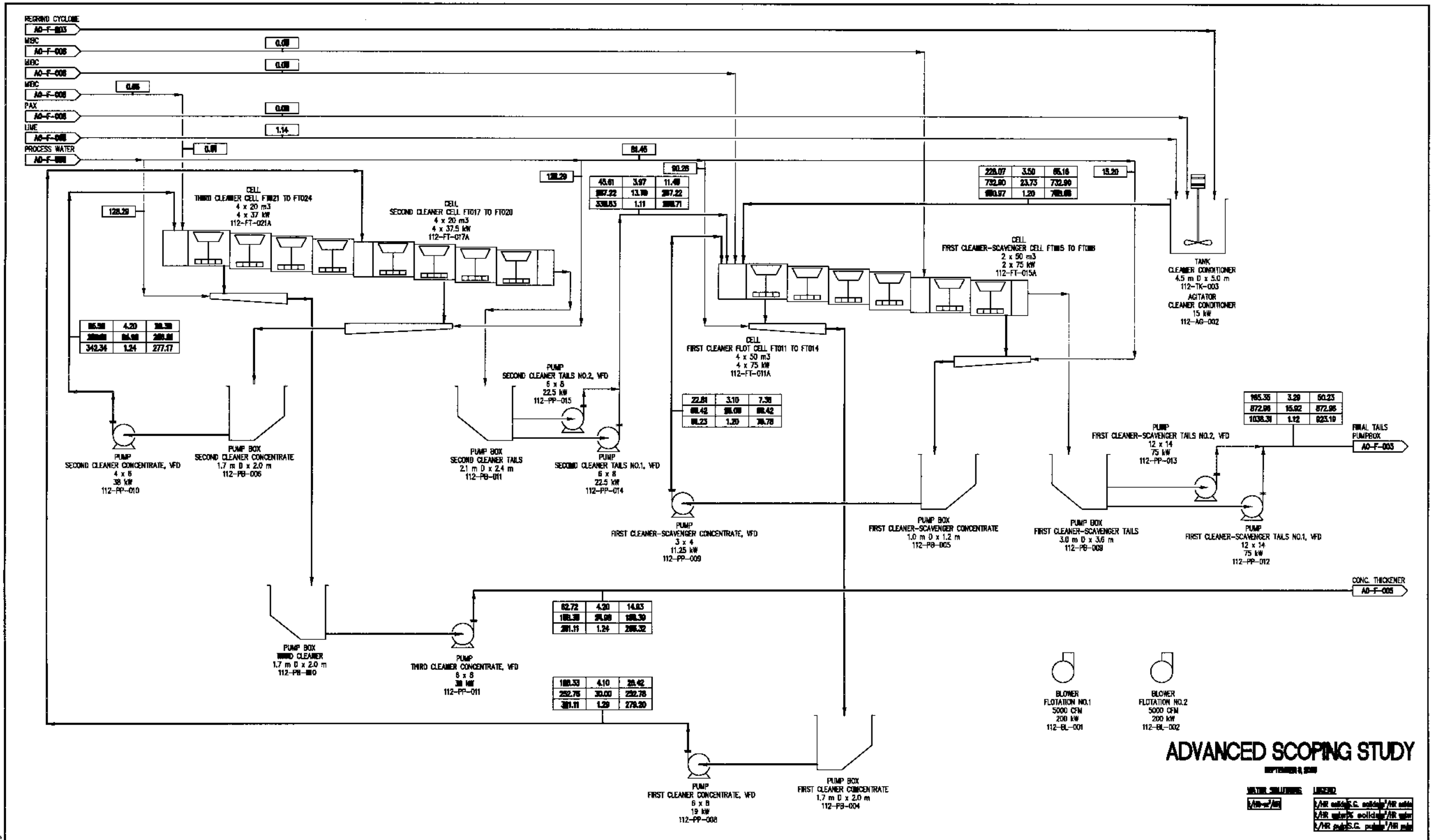
GENERAL GRINDING FLOWSHEET NO. 2

FILENAME:	PROJECT NUMBER	DRAWING NUMBER	REV.
A0F002.DWG	317882	AO-F-002	/

2005/07/11 14:15



FIRST CLEANER/ SCAV. TAILS PUMP
MIBC
LIME
PAX
SEC. CYCLONE O/F NO. 1
AO-F-002
DISTRIBUTOR FLOTATION FEED 112-DB-002
SEC. CYCLONE O/F NO. 2
AGITATOR ROUGHER CONDITIONER NO.1 30 kW 112-AG-001
CELL ROUGHER FLOTATION CELL FT001 TO FT005 5 x 200 m3 5 x 187 kW 112-FT-001A
PUMP, SUMP FLOTATION AREA 4" 20 kW 112-PS-002
TANK ROUGHER CONDITIONER NO.1 200 m3 112-TK-001
AGITATOR ROUGHER CONDITIONER NO.2 30 kW 112-AG-009
CELL ROUGHER FLOTATION CELL FT006 TO FT010 5 x 200 m3 5 x 187 kW 112-FT-006A
TANK ROUGHER CONDITIONER NO.2 200 m3 112-TK-002
CRANE FLOTATION AREA 10 t 5 kW 112-CN-003
CYCLOPAC REGRIND 4 x 500 mm 112-CY-002
CLEANER COND. TANK
SAMPLER REGRIND 112-SA-006
VERTIMILL REGRIND NO.1 VTM-1500 1119 kW 112-VM-001
VERTIMILL REGRIND NO.2 VTM-1500 1119 kW 112-VM-002
TAILINGS POND
AO-F-009
PUMP FINAL TAILS NO.1, VFD 28 x 26 1875 kW 112-PP-016
SAMPLER FINAL TAILS 112-SA-004
PUMP BOX FINAL TAILS 6.5 m D x 6.7 m 112-PB-007
PUMP FINAL TAILS NO.2, VFD 28 x 26 1875 kW 112-PP-017
PUMP BOX ROUGHER CONCENTRATE 2.5 m D x 2.7 m 112-PB-003
PUMP ROUGHER CONCENTRATE, VFD 8 x 10 56 kW 112-PP-007
SAMPLER ROUGHER CONCENTRATE 112-SA-005
PROCESS WATER
PUMP REGRIND CYCLONE FEED NO.1, VFD 10 x 12 336 kW 112-PP-018
PUMP, SUMP REGRINDING AREA 4" 20 kW 112-PS-003
PUMP BOX REGRIND CYCLONE FEED 3.5 m D x 3.7 m 112-PB-008
PUMP REGRIND CYCLONE FEED NO.2, VFD 10 x 12 336 kW 112-PP-019
0.02
0.13
81.45
306.34
ADVANCED SCOPING STUDY
DWG. NO.
REFERENCE DRAWINGS
NO
DESCRIPTION
BY
DATE
ISSUE
REVISIONS
SECTION: GENERAL
SCALE:
DATE
DESIGN. BY HT OCT/04
DRAWN BY: JSL OCT/04
CHECK. BY
APP. BY:
Galore Creek B.C.
NovaGold
HATCH
TITLE
GENERAL
ROUGHER FLOTATION & REGRIND
FLOWSHEET NO. 3
FILENAME:
PROJECT NUMBER 317882
DRAWING NUMBER AO-F-003
REV.
2005/07/11 14:14



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DWG. NO.	REFERENCE DRAWINGS	NO	DESCRIPTION	BY	DATE
			ISSUE		

NO	DESCRIPTION	BY	DATE
	REVISIONS		

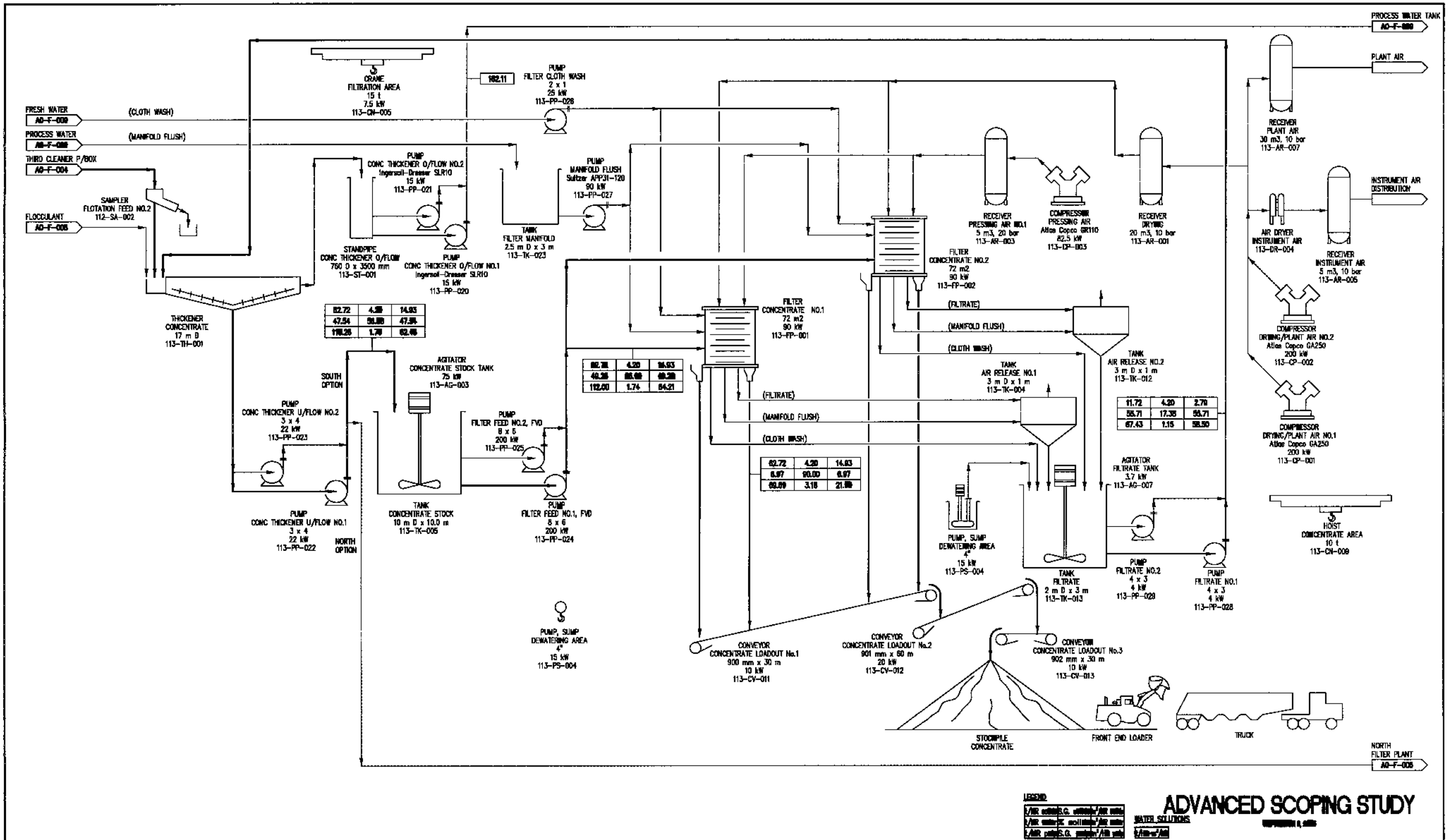
SECTION:	GENERAL	
SCALE:	NONE	DATE
DESIGN. BY	HT	OCT/04
DRAWN BY:	JSL	OCT/04
CHECK. BY		
APP. BY:		

NovaGold Galore Creek B.C.

HATCH™

TITLE
GENERAL
CLEANER FLOTATION
FLOWSHEET NO. 4

FILENAME:	PROJECT NUMBER	DRAWING NUMBER	REV.
A0F004.DWG	317882	AO-F-004	/



THIS DRAWING HAS NOT BEEN PUBLISHED BUT RATHER HAS BEEN PREPARED BY HATCH FOR USE BY THE CLIENT NAMED IN THE TITLE BLOCK SOLELY IN RESPECT OF THE CONSTRUCTION, OPERATION AND MAINTENANCE OF THE FACILITY NAMED IN THE TITLE BLOCK AND SHALL NOT BE USED FOR ANY OTHER PURPOSE OR FURNISHED TO ANY OTHER PARTY WITHOUT THE EXPRESS CONSENT OF HATCH.

DWG. NO.	REFERENCE DRAWINGS	NO	DESCRIPTION	BY	DATE	NO	DESCRIPTION	BY	DATE
			ISSUE				REVISIONS		

SECTION: GENERAL
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DESIGN. BY: RT
DRAWN BY: JSL — OCT/04
CHECK. BY:
APP. BY:

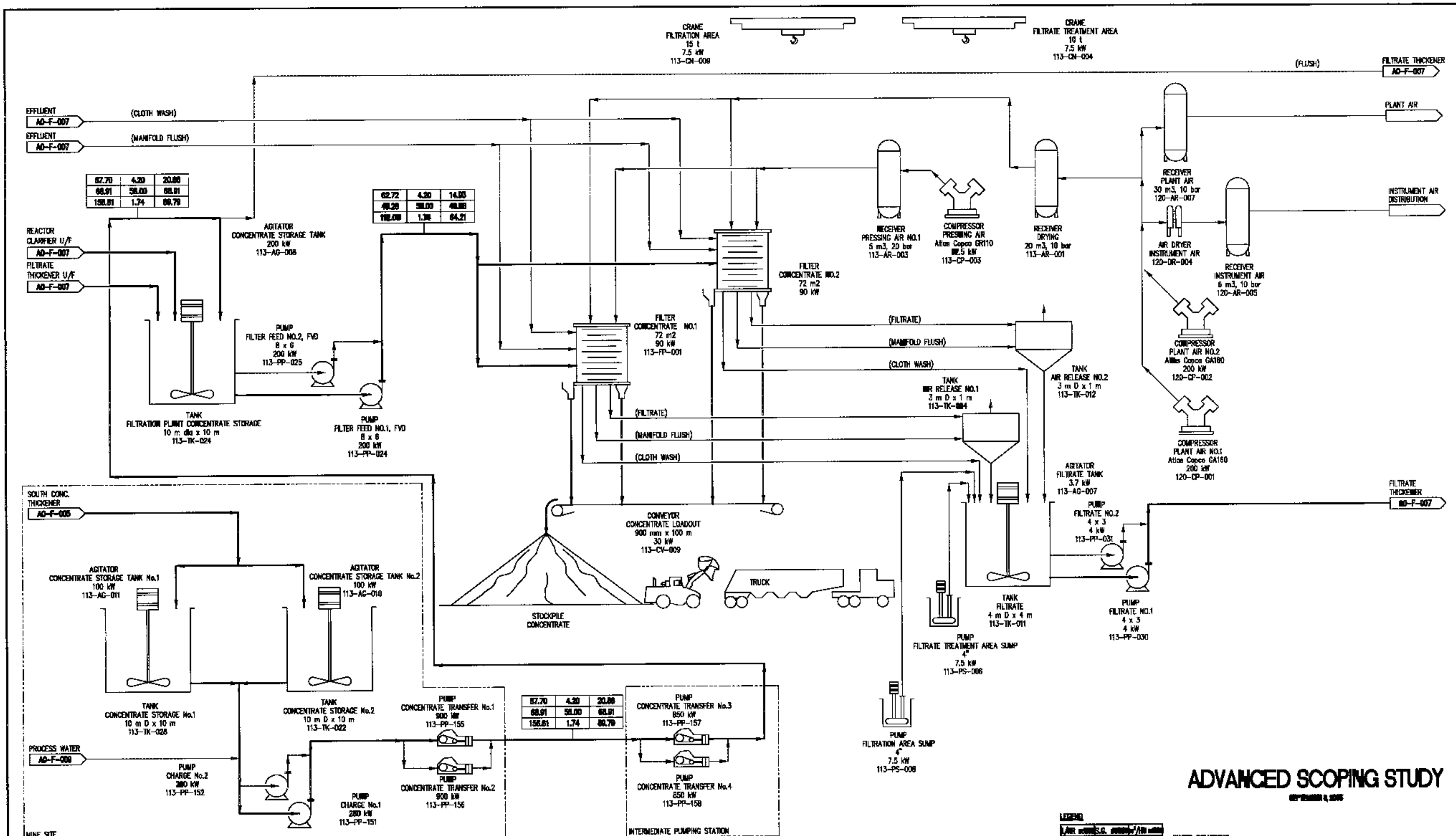
Galore Creek B.C.

HATCH

TITLE: Galore Creek Pre-Feasibility Study
GENERAL – SOUTH OPTION
CONC. DEWATERING & LOADOUT
FLOWSHEET NO. 5

PROJECT NUMBER	DRAWING NUMBER	REV.
317882	AO-F-005	/

2005/07/11 14:13



87.70	4.20	20.88
68.91	58.00	68.91
156.61	1.74	89.79

62.72	4.20	14.93
49.28	58.00	49.88
112.00	1.74	64.21

87.70	4.20	20.88
68.91	58.00	68.91
156.61	1.74	89.79

ADVANCED SCOPING STUDY

2006/07/11 14:11

DWG. NO.	REFERENCE DRAWINGS	NO	DESCRIPTION	BY	DATE	NO	DESCRIPTION	BY	DATE
			ISSUE				REVISIONS		

SECTION: GENERAL
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DATE
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DRAWN BY: JSL OCT/04
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APP. BY:

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TITLE
GENERAL – NORTH OPTION
CONC. FILTRATION
FLOWSHEET NO. 6

FILENAME:	PROJECT NUMBER	DRAWING NUMBER	REV.
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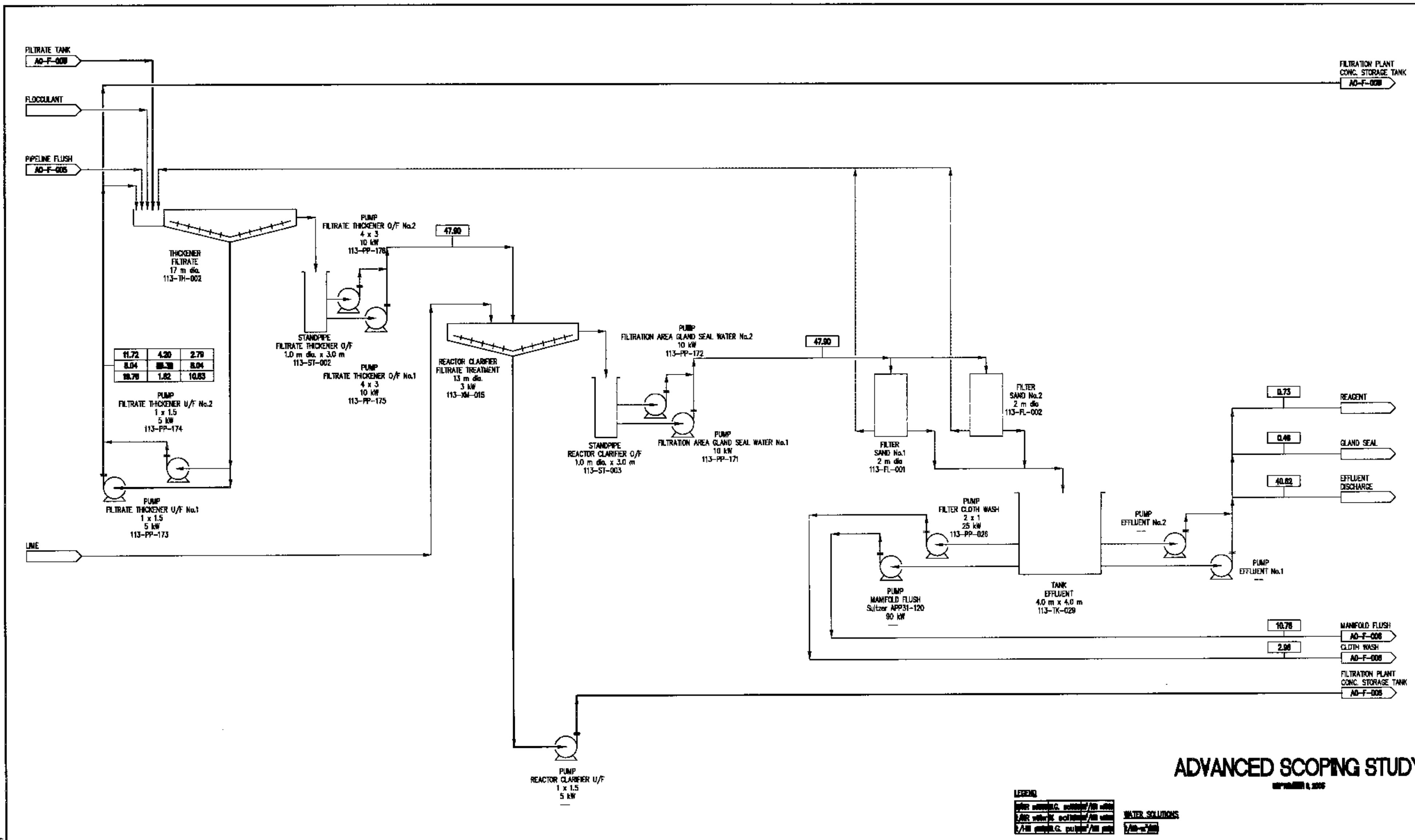

11.72	4.20	2.79
8.04	[illegible]	8.04
19.76	1.82	10.83

ADVANCED SCOPING STUDY

LEGEND

WATER SOLUTIONS

DWG. NO.	REFERENCE DRAWINGS

NO	DESCRIPTION	BY	DATE
	ISSUE		

NO	DESCRIPTION	BY	DATE
	REVISIONS		

SECTION:	GENERAL	
SCALE:	NONE	DATE
DESIGN. BY	HT	OCT/04
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TITLE Galore Creek Pre Feasibility Study

GENERAL – NORTH OPTION
WATER TREATMENT
FLOWSHEET NO. 7

FILENAME:	PROJECT NUMBER	DRAWING NUMBER	REV.
	317882	AO-F-007	/

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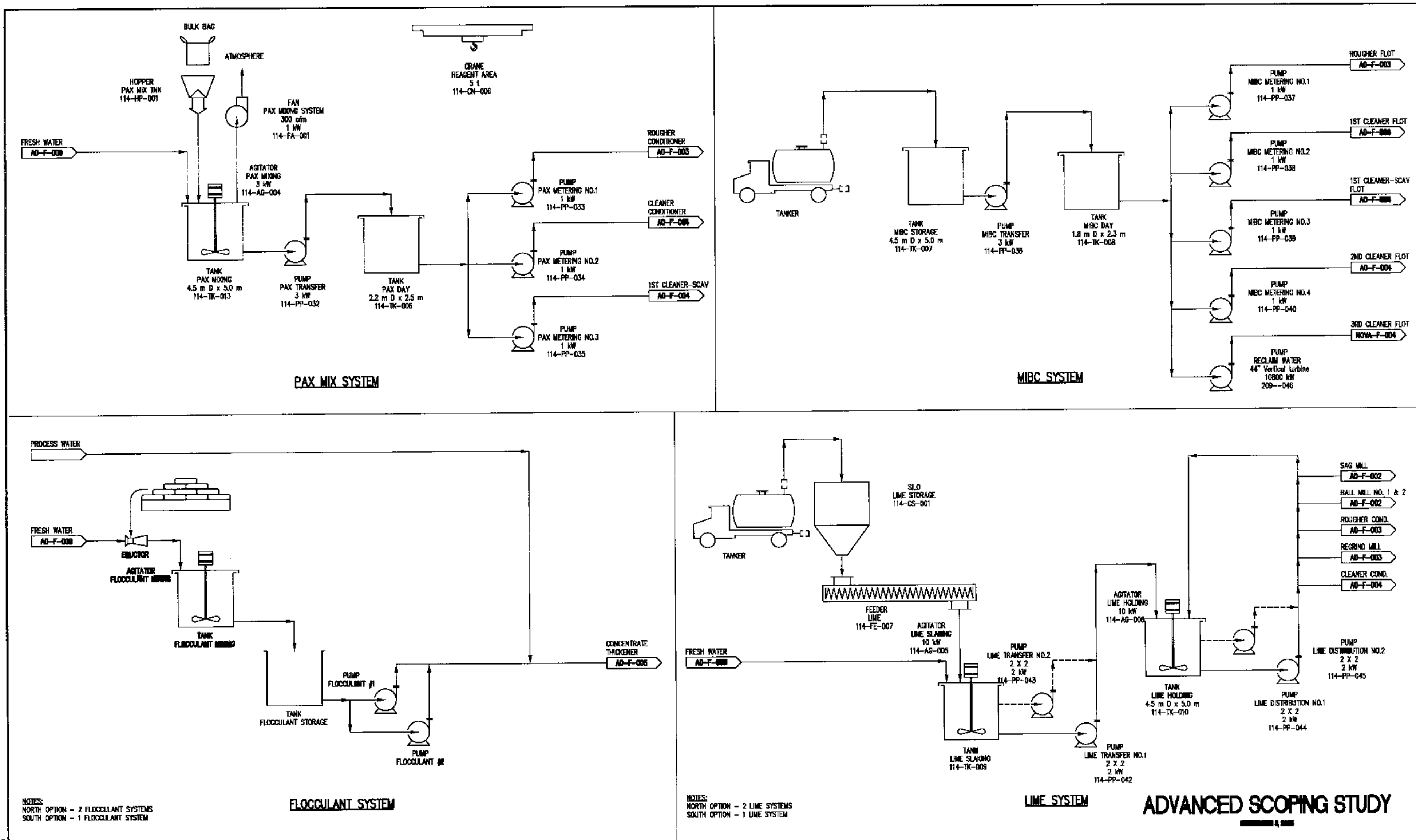

ADVANCED SCOPING STUDY

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2006/07/11 14:10

DWG. NO.	REFERENCE DRAWINGS

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	ISSUE		

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	REVISIONS		

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DRAWN BY:	JSL	OCT/04
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APP. BY:		

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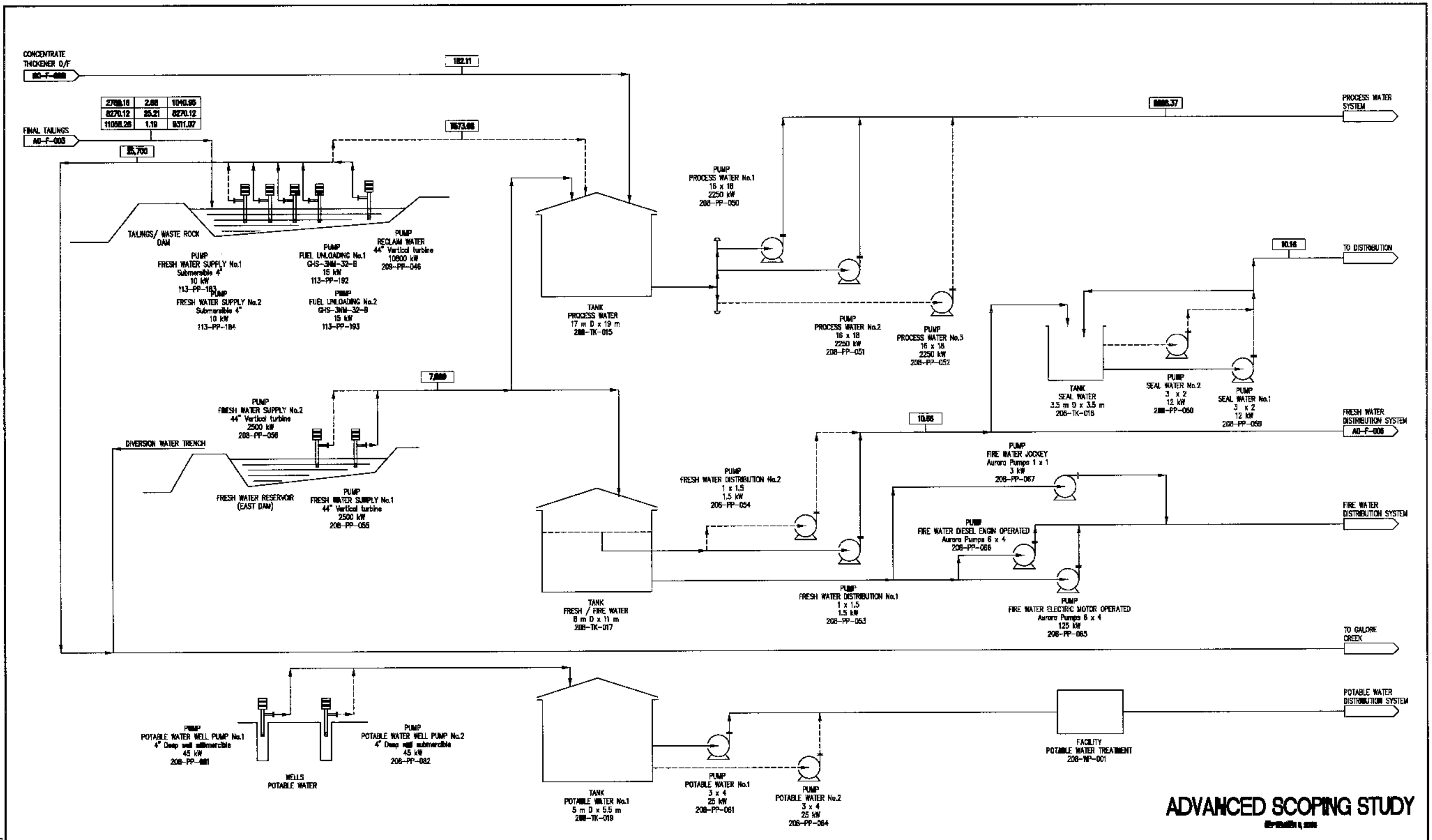
TITLE

GENERAL
REAGENT SYSTEMS
FLOWSHEET NO. 8

FILENAME:	PROJECT NUMBER	DRAWING NUMBER	REV.
	317882	AO-F-008	/



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THIS DRAWING HAS NOT BEEN PUBLISHED BUT RATHER HAS BEEN PREPARED BY HATCH FOR USE BY THE CLIENT NAMED IN THE TITLE BLOCK SOLELY IN RESPECT OF THE CONSTRUCTION, OPERATION AND MAINTENANCE OF THE FACILITY NAMED IN THE TITLE BLOCK AND SHALL NOT BE USED FOR ANY OTHER PURPOSE OR FURNISHED TO ANY OTHER PARTY WITHOUT THE EXPRESS CONSENT OF HATCH

DWG. NO.	REFERENCE DRAWINGS	NO	DESCRIPTION	BY	DATE	NO	DESCRIPTION	BY	DATE
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SECTION:	GENERAL	
SCALE:	NONE	DATE
DESIGN. BY	HT	OCT/04
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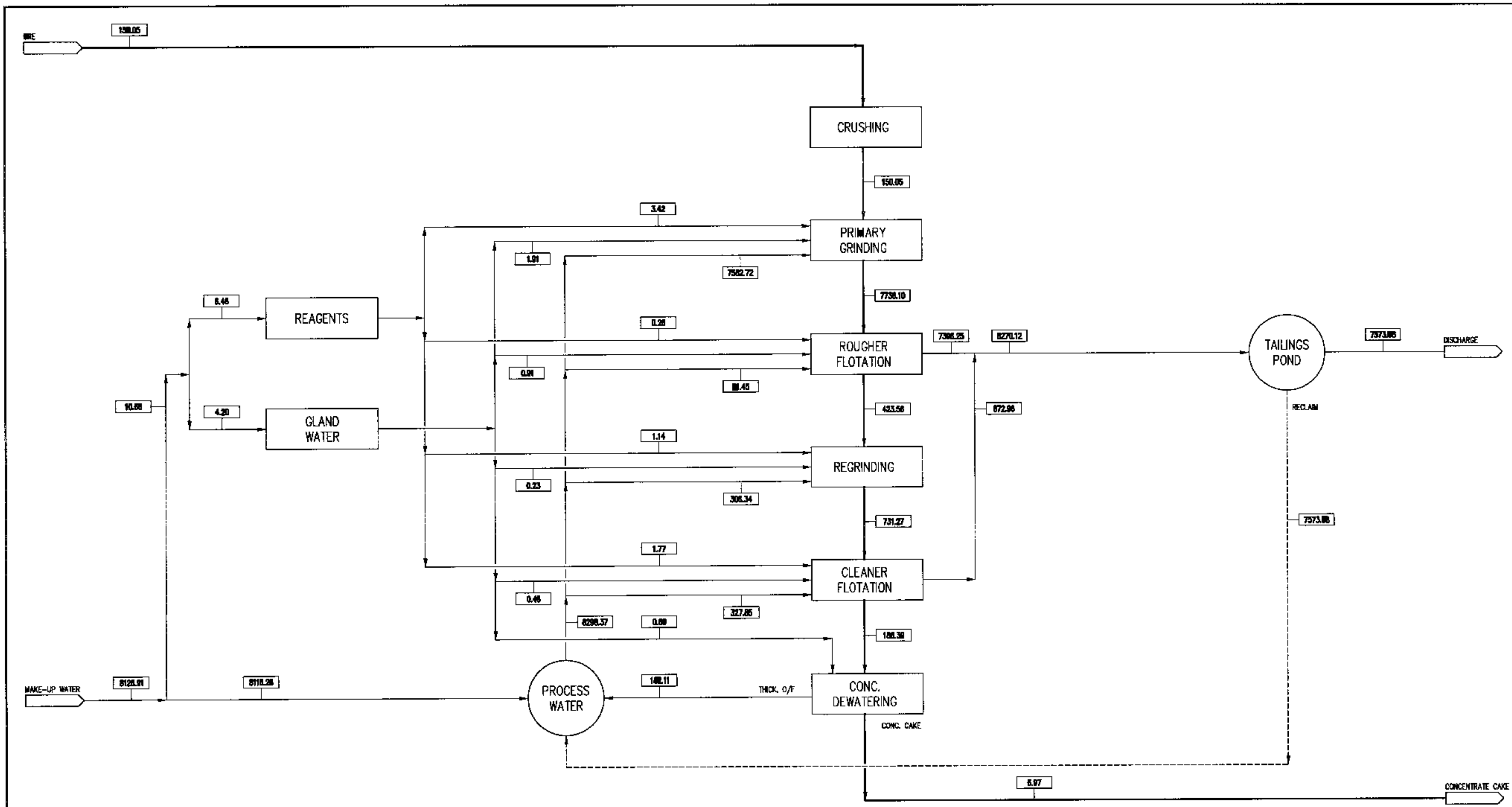
TITLE Galore Creek Pre Feasibility Study

GENERAL WATER SYSTEM FLOWSHEET NO. 9

FILENAME	PROJECT NUMBER	DRAWING NUMBER	REV.
AOF009.DWG	317882	AO-F-009	/

2005/07/11 14:18



ADVANCED SCOPING STUDY

DWG. NO.	REFERENCE DRAWINGS

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SECTION:	GENERAL	
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DRAWN BY:	JSL	OCT/04
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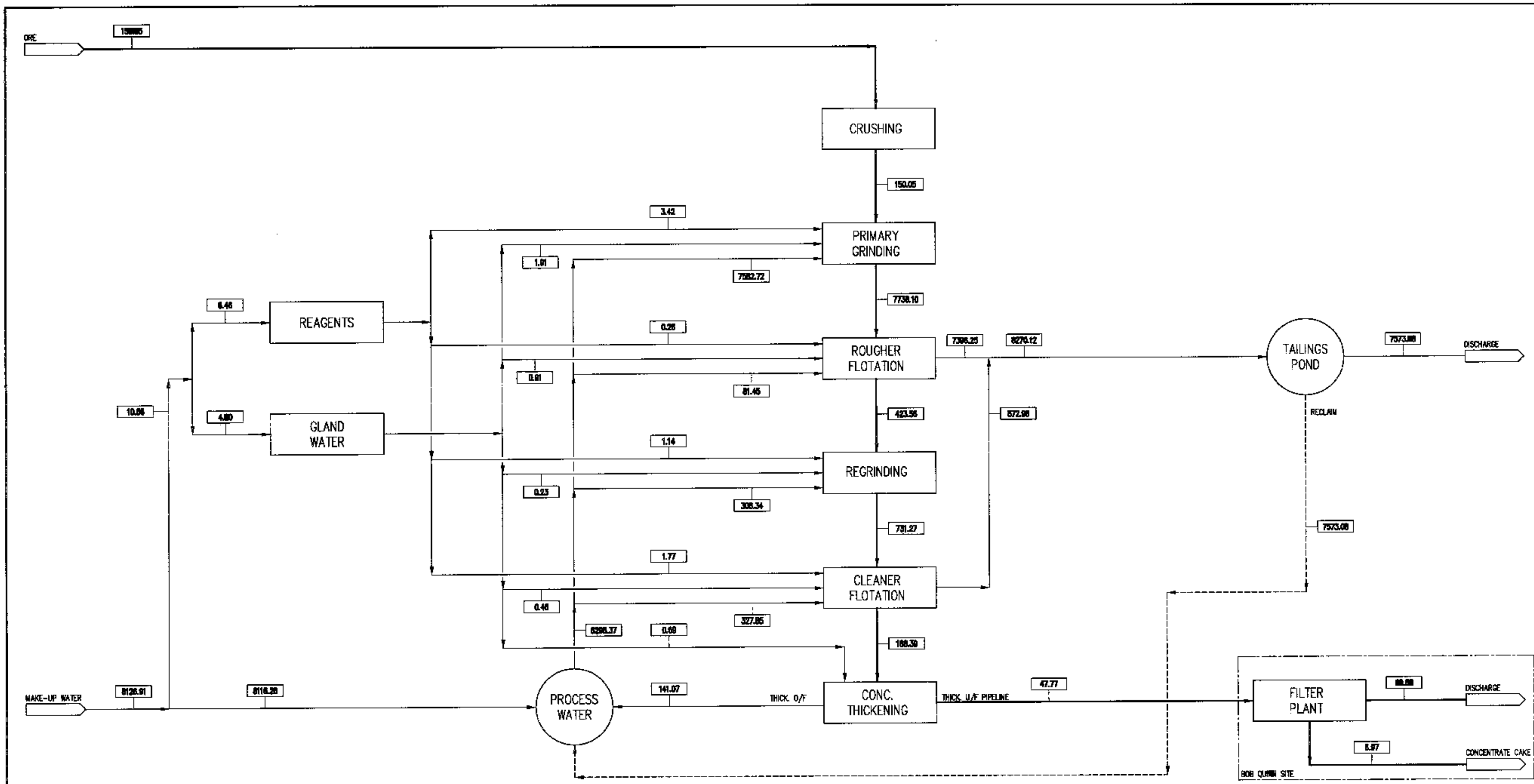
HATCH

TITLE
GENERAL – SOUTH OPTION
WATER BALANCE
FLOWSHEET NO. 10

FILENAME:	PROJECT NUMBER	DRAWING NUMBER	REV.
A0F010.DWG	317882	A0-F-010	/



ADVANCED SCOPING STUDY

SECTION:	GENERAL
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Galore Creek B.C.

HATCH

GENERAL -- NORTH OPTION
WATER BALANCE
FLOWSHEET NO. 11

FILENAME:	PROJECT NUMBER	DRAWING NUMBER	REV.
A0F011.DWG	317882	A0-F-011	/

DWG. NO.	REFERENCE DRAWINGS

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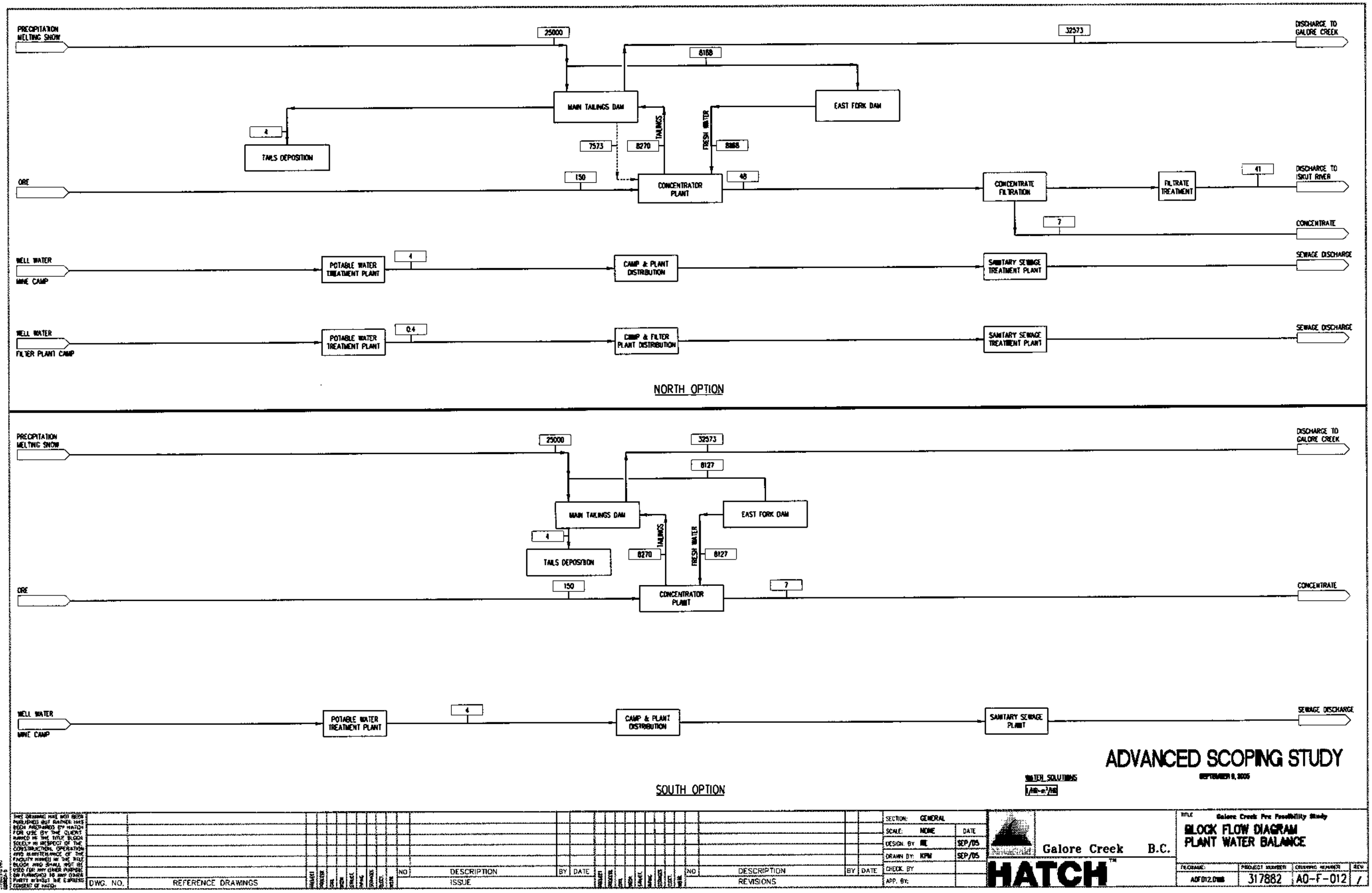
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25000
32573
DISCHARGE TO GALORE CREEK
8188
MAIN TAILINGS DAM
EAST FORK DAM
4
TAILS DEPOSITION
7573
8270
TAILINGS
FRESH WATER
8188
ORE
150
CONCENTRATOR PLANT
48
CONCENTRATE FILTRATION
FILTRATE TREATMENT
41
DISCHARGE TO ISKUT RIVER
7
CONCENTRATE
WELL WATER
MINE CAMP
POTABLE WATER TREATMENT PLANT
4
CAMP & PLANT DISTRIBUTION
SANITARY SEWAGE TREATMENT PLANT
SEWAGE DISCHARGE
WELL WATER
FILTER PLANT CAMP
POTABLE WATER TREATMENT PLANT
0.4
CAMP & FILTER PLANT DISTRIBUTION
SANITARY SEWAGE TREATMENT PLANT
SEWAGE DISCHARGE
NORTH OPTION
PRECIPITATION MELTING SNOW
25000
32573
DISCHARGE TO GALORE CREEK
8127
MAIN TAILINGS DAM
EAST FORK DAM
4
TAILS DEPOSITION
8270
TAILINGS
FRESH WATER
8127
ORE
150
CONCENTRATOR PLANT
7
CONCENTRATE
WELL WATER
MINE CAMP
POTABLE WATER TREATMENT PLANT
4
CAMP & PLANT DISTRIBUTION
SANITARY SEWAGE PLANT
SEWAGE DISCHARGE
SOUTH OPTION
ADVANCED SCOPING STUDY
SEPTEMBER 9, 2005
WATER SOLUTIONS
l/HR-m³/HR
DWG. NO.
REFERENCE DRAWINGS
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SEP/05
DRAWN BY: KPM
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CHECK BY
APP. BY:
Galore Creek B.C.
HATCH™
TITLE Galore Creek Pre Feasibility Study
BLOCK FLOW DIAGRAM
PLANT WATER BALANCE
PROJECT NUMBER
317882
DRAWING NUMBER
AO-F-012
REV
/

Appendix F

Certificates

STATEMENT OF QUALIFICATIONS

I, Paul Anthony John Hosford, P.Eng., do hereby certify that:

1. I am currently employed as Manager, Metallurgy by:
   Hatch Ltd.,
   Suite 2200, 1066 West Hastings Street,
   Vancouver, British Columbia,
   CANADA V6E 3X2

2. I graduated with the degree of B.Sc. Chemical Engineering (Honours) from the University of Edinburgh, Scotland in 1982.

3. I am a member of the Association of Professional Engineers and Geoscientists of the Province of British Columbia.

4. I have worked as a metallurgical engineer for 21 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43- 101") and certify that by reason of my education, affiliation with a professional association (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 have overall responsibility for the coordination of the preparation of the technical report entitled "Updated Preliminary Economic Assessment for the Galore Creek Project" dated 21 October 2005 (the "Technical Report"). I have visited the site.

7. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

8. I am independent of the issuer applying all 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 website accessible by the public, of the Technical Report.

Dated this 21st day of October 2005

__"P. Hosford"_________
PAJ Hosford, P.Eng.
Manager, Metallurgy
Hatch Vancouver

STATEMENT of QUALIFICATIONS

I, James H Gray. P.Eng. do hereby certify that:

1. I am a Principal of GR Technical Services Ltd., 2767 Evercreek Bluffs Way SW Calgary, Alberta Canada T2Y 4P6.

2. I graduated with a Bachelor of Applied Science in Mining Engineering from the University of British Columbia in 1975.

3. I am a member of The Association of Professional Engineers and Geoscientists of the Province of British Columbia, registration number 11919, and the Association of Professional Engineers, Geologists and Geophysicists of Alberta (M47177).

4. I have worked as a Professional Engineer for over 25 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (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 the Mine Planning and Pit Delineated Resource aspects of the technical report entitled "Updated Preliminary Economic Assessment for the Galore Creek Project" dated 21 October 2005 (the "Technical Report"). I have visited the project site.

7. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

8. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

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.

Dated this 21th day of October, 2005.

Signed "J H Gray PEng."

James H. Gray PEng.
President and Principal Mining Engineer
GR Technical Services Ltd.