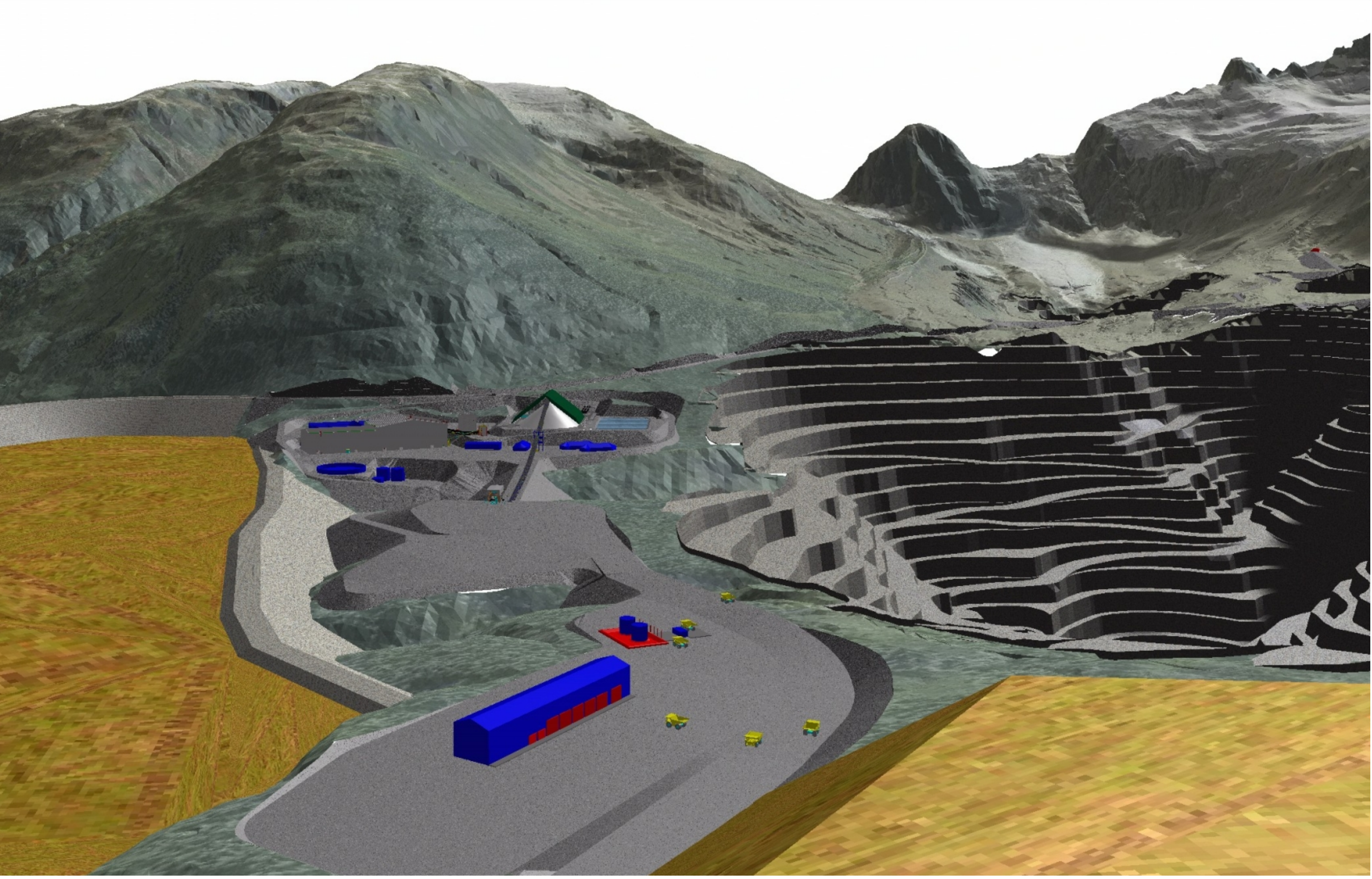
NovaGold Canada Inc.

Galore Creek Project Feasibility Study

Northwestern British Columbia



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3. Summary

Hatch Ltd., an independent engineering services company located in Vancouver, B.C., Canada, together with a number of specialized consultants, has completed the Feasibility Study for NovaGold's Galore Creek project in northwestern British Columbia. The Galore Creek Feasibility Study was completed under the direction of Bruce Rustad, P.Eng., Director of P&CM/Project Manager for Hatch and an independent Qualified Person as defined by National Instrument 43-101 ("NI 43-101"). This NI43-101 technical report has been compiled to disclose the findings of the Feasibility Study. This report is intended to be read as a whole, and sections should not be read or relied upon out of context.

The mineral resource estimate in this Feasibility study was completed by Mr. Michael J. Lechner,RPG from Resource Modeling Incorporated an independent Qualified Person as defined by NI 43-101 and is the subject of a technical report entitled "Updated Galore Creek Mineral Resources, Northwestern British Columbia", dated September 7, 2006 and posted to SEDAR on September 12, 2006.

3.1 Overview of Feasibility Study

Section 3.1 was prepared by Mr. Bruce Rustad, P.Eng., Hatch

NovaGold Resources Inc., through its 100% owned subsidiary, NovaGold Canada Inc. (NovaGold.), is engaged in the exploration of the Galore Creek Project in British Columbia, Canada. The Galore Creek deposit is located in a mountainous area of Northwestern British Columbia approximately 1,000 kilometres northwest of Vancouver and 80 kilometres northwest of the Eskay Creek Mine. The project is located 200 kilometres north of the tidewater port of Stewart, British Columbia and 100 kilometres northeast of the community of Wrangell, Alaska. Current access is by helicopter, although fixed-wing aircraft, as well as, a combination of barge and road access have been used in the past to transport personnel and equipment to the site. Typical of northwest coastal areas of British Columbia, the area is characterized by cool summers and cold humid winters and relatively heavy snowfall. The area is rugged and mountainous and a 135 km access road including a 4.3 km long tunnel is required to connect the mine site to Highway 37 near Bob Quinn Lake.

Figure 1: General Location Map

The proposed project will be a large tonnage open pit operation with an estimated 540 million tonnes of ore. The majority of this will need to be stored as tailings following metal extraction. In addition to the estimated 540 million tonnes of tailings, approximately 840 million tonnes of waste rock (including overburden pre-stripping) will also require storage. The process plant will be a conventional milling/flotation concentrator with a pipeline transferring concentrate to a remote filter plant and concentrate truck loading facility near Highway 37.

Mineralization was discovered on the property in the mid 1950s, and extensive exploration work was carried out by major mining companies during the 1960s, 70s and 90s. This potential resource has not been developed to date for a number of reasons, including the difficulty in accessing the mine. SpectrumGold Inc., a predecessor company of NovaGold Canada Inc., optioned the property in 2003 and initiated extensive exploration, engineering and environmental assessment programs.

The design of the mine and related facilities takes into account the potential effects on the remote wilderness environment, the presence of wildlife, and the social and economic well-being of local residents. Project plans are consistent with the management direction developed in 2000 under the Cassiar Iskut-Stikine Land and Resource Management Plan (CIS-LRMP).

The capital cost of the project is estimated at CA$2.23 billion using 2nd quarter 2006 Canadian Dollars. This estimate assumes access to the provincial electric power grid at Bob Quinn. Using the Economic Analysis Base Case Assumptions including US$1.50/lb Cu, US$ 525/oz Au, US$8.00/oz Ag and a US$ / CA$ exchange rate of 0.81/1.00 the payback period will be 4.0 years. If recent high metal prices continue, then the payback period will be considerably shorter. The mine will pay approximately CA$1.2 billion in taxes over its 22-year life.

Testwork suggests that about 41% of the waste rock will be potentially acid generating (PAG). The PAG waste rock and the tailings from the concentration process will be stored in a flooded impoundment to prevent oxidation and metal leaching. ARD testwork indicates that the PAG material must be placed underwater within 20 years, however, all PAG material has been conservatively placed in areas that will flood within five years. An extensive system of diversion channels and ditches will be required to direct natural surface waters away from the waste rock and tailings areas, sedimentation pond and other mine facilities. These diversion channels and structures will be constructed from mine waste rock during the pre-operational construction period.

The construction of the project will involve two phases. The first phase will be to establish access to the Galore valley and the second phase will be to construct the mine and facilities. Phase 1 will involve road, bridge and tunnel construction using helicopter-supported construction activities. The second phase will be constructed using the mine fleet of equipment.

The project is expected to create approximately 900 to 1,000 jobs during the construction phase and require approximately 500 direct employees during the operations phase. Additional contract employees will be required for many ongoing and intermittent tasks, including camp operation, concentrate hauling, tailings dam expansion and mill relining. Construction is expected to take four years and operations are currently planned for an additional 22 years at a nominal production rate of 65,000 tpd.

The mine is expected to produce an average of approximately 450,000 tonnes of copper/precious metals concentrate per year. The concentrate will be hauled by truck along Highway 37 to the deep-sea port at Stewart for transfer to ocean-going freighters. The destination for this concentrate will most likely be smelters in Asia.

NovaGold is targeting production from the Galore Creek Project by the 1st quarter of 2011. This date assumes that the environmental assessment process, initiated in 2004, will be completed and the major required permits issued by early 2007. This date also assumes that tunnel breakthrough will occur in the 3rd quarter of 2008. Planning and detailed engineering will continue in the interim. Phase 1 construction is expected to take at least eighteen months and consequently the construction of the on-site mine area process facilities will not be able to start until the 3rd quarter of 2008.

3.2 History and Ownership

Section 3.2 was prepared by Mr. Michael J. Lechner,RPG. Resource Modeling Incorporated

In August 2003, SpectrumGold Inc. (now NovaGold Canada Inc.) entered into an option agreement to acquire a 100% interest in the Galore Creek property by acquiring Stikine Copper Limited, a company owned by QIT-FER et Titane Inc. and Hudson Bay Mining and Smelting Co. Limited. NovaGold must complete a pre-feasibility study on the project and make payments to the owners totalling US$20.3 million within a period of eight years. Stikine Copper will have no retained interests, royalties or back-in rights on the project. The Galore Creek property consisted of 292 two-post claims, 39 of which were fractions, all held in the name of Stikine Copper Limited.

In 2004 SpectumGold Inc. entered into an option agreement with Eagle Plains Resources Ltd. giving NovaGold the exclusive right to earn up to an 80% interest in the Copper Canyon Property comprised of 4 located claims. The effective date of the agreement is October 1, 2003. During the first option period, NovaGold must issue 296,296 shares to Eagle Plains on or before February 26, 2007 and incur property expenditures of CA$3 million on or before October 1, 2013 to earn a 60% interest. To earn another 20% interest, NovaGold must make a payment of CA$1 million within 90 days of exercising the first option and complete a feasibility study within eight years of the agreement effective date. In addition, NovaGold assumed the commitments of the underlying Eagle Plains option agreement dated May 28, 2002 with Bernard Kreft that included payments totalling CA$250,000 and a 2% NSR.

In 2004, SpectrumGold Inc. purchased 11 two-post claims from Silver Standard Resources Inc. and Teck-Cominco Limited. In June 2005, NovaGold transferred its held 100% interest in the eleven two-post claims (Bik 1, Bik 2, Bik 3 and eight Penny claims) to Eagle Plains Resources Ltd. As required under the Eagle Plains option agreement.

In March 2004, SpectrumGold Inc. entered into an option agreement (Grace Option Agreement) with Pioneer Metals Corporation (Pioneer) giving NovaGold the exclusive right to earn up to a 60% interest in the Grace Property comprised of five located claims. To exercise the option, NovaGold must incur property expenditures of CA$5 million and subscribe for 3.92 million units of Pioneer on or before March 24, 2009. In October 2005, NovaGold received a writ of summons from Pioneer Metals Corporation to rescind the Grace Option Agreement. NovaGold is defending its position and a court date has been set for the third quarter of 2007 to hear Pioneer's claim.

Pioneer holds subsurface mineral rights on the Grace property subject to an option granted to NovaGold. The British Columbia government owns the surface, and has the sole and exclusive right to grant surface rights to third parties.

On June 21, 2006, NovaGold filed an application with the British Columbia government to obtain a surface lease over the Grace property. NovaGold intends to build a tailings and waste rock storage facility over a portion of the Grace property to facilitate operations at Galore Creek as shown Figure 3-1.

In 2005, NovaGold reviewed the status of all Galore Creek property mineral claims and recommended that legacy claims be converted to cell claims as allowed by the amended British Columbia Mineral Tenure Act. The legacy claim conversion would consolidate the project holdings, eliminate any internal claim gaps, secure tenure ownership and reduce legal surveying costs for mining lease application. All parties agreed to this conversion and signed the Galore Creek Legacy Claim Cell Conversion Agreement dated June 30, 2005.

Between July 6 -11, 2005, NovaGold converted the Galore claims with the exception of claims located adjacent to third party cell claims. These claims were not converted, since a portion of their held area would be surrendered to the adjacent cell claim holder on conversion. The option agreement land schedules were revised and in early 2006 were presented to Stikine Copper Limited, Eagle Plains, Bernard Kreft and Pioneer Metals for approval.

On November 28, 2005 NovaGold applied drilling expenditures incurred on the Galore Creek and Copper Canyon properties as assessment work to advance all claims contiguous with the Galore Creek property to the claim expiry year to 2015, the maximum allowed under the Mineral Tenure Act. Under the new online system, access to file assessment work on claims must be granted by the registered owners of the tenures. On November 29, 2005 a request for NovaGold to be appointed agents to file assessment work was forwarded to Pioneer Metals. As there was no response to this request, no assessment work was filed on the Pioneer tenures.

3.3 Geology and Resources

Section 3.3 was prepared by Michael J. Lechner. Resource Modeling Incorporated

The Galore Creek property is characterized as a large copper-gold porphyry system consisting of a number of mineralized zones including the Central Replacement Zone, the Southwest Zone, the Junction Zone, Middle Creek, North and South Gold Lenses, Bountiful, Copper Canyon, and the West Fork Zone.

Four mineralized zones of potentially economic interest have been explored at Galore Creek. These are briefly summarized as follows:

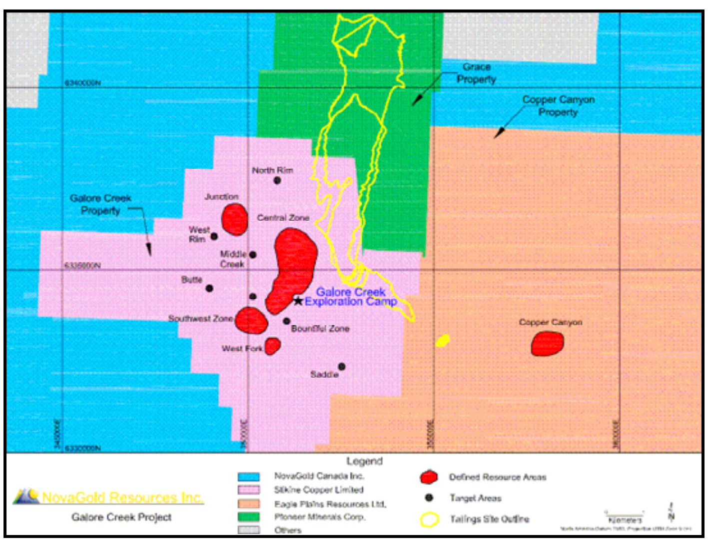


Figure 3-1: Galore Valley Mine Site

- The Central Zone is the largest and most extensively explored of all the deposits and is characterized by fairly complex geology. 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 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 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 copper-gold-silver mineralization within the diatreme appears to relate to a combination of structural traps and mineralizing faults.
- The 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. 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.
- 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 property has been tested with 758 diamond drill holes totaling about 187,267 metres; the assay database for the property contains about 60,000 assay records. A large amount of data has been collected from the property since the early 1960's. In addition to diamond drilling, this data includes soil, stream sediment, rock geochemistry programs, helicopter airborne magnetic and radiometric surveys, ground based IP/resistivity surveys, and seismic refraction surveys.

Since 2003, NovaGold has conducted a series of diamond drilling campaigns to further define and delineate the known mineralized zones within the Galore Creek area. NovaGold Resources Inc. (NovaGold) engaged Resource Modeling Incorporated (RMI) to provide an independent review of the geological data, to review the estimation parameters and the resultant estimate of Mineral Resources, and to prepare a Technical Report for the Galore Creek Project. The current resource model is based on all data through the 2005 exploration season. The resource estimate is based on a 3-dimensional computer block model with copper, gold, and silver block grades estimated into 25 m by 25 m by 15 m high blocks using 5-metre-long drill hole composites. Prior to compositing the drill hole grades, high-grade outlier values were cut based on an analysis of cumulative probability plots. The grade models were validated by visual and statistical methods and are deemed to be globally unbiased. The blocks were then classified into Measured, Indicated, and Inferred Mineral Resource categories using the number of data and distance to data method.

The Galore Creek deposit contains about 749 million tonnes of Measured and Indicated Mineral Resources at a 0.25% copper equivalent cut-off grade with a grading of 0.52% copper, 0.30 g/t gold, and 4.9 g/t silver, Michael Lechner is the qualified person within the meaning of NI43-101 responsible for the resource estimate. The Measured, Indicated, and Inferred Mineral Resources as of September 7, 2006 are summarized in
Table 3-1.

Table 3-1: Summary of Mineral Resources – 0.25% CuEq Cutoff

Resource Category	Tonnes (millions)	Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)	Cu Pounds (billions)	Au Ounces (millions)	Ag Ounces (millions)
Measured	263.6	0.62	0.35	5.9	0.81	3.6	3.0	50.0
Indicated	485.3	0.46	0.28	4.3	0.63	4.9	4.4	67.1
Measured + Indicated	748.9	0.52	0.30	4.9	0.69	8.5	7.4	117.1
Inferred	300.1	0.37	0.21	3.7	0.51	2.4	2.0	35.7

Note: CuEq calculation is based on net smelter return and uses metal prices of US$1.25/lb, US$450/oz, US$7.00/oz for copper, gold and silver respectively.

Excludes inferred resources at Copper Canyon.

3.4 Mine Plan

Section 3.4 was prepared by Mr. Jim Gray,P.Eng. GR Technical Services.

An optimized feasibility level 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 produced the reserves in the Table 3-2 below. The reserves include a 3.6% dilution for all material above cutoff grade, and mining losses of 2.4%.

Cut-off grade for the reserves in Table 3-3 below is CA$3.82/t Net Smelter Return (NSR).

CIM Standards On Mineral Resources And Reserves Definitions And Guidelines define 'Probable Mineral Reserve' as " the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified."

Proven and Probable Reserves at Galore Creek are summarized in the table below.

Table 3-2: Proven and Probable Reserves at Galore Creek

		Diluted Grades			Contained Metal		
Classification	**Ore (Mt)**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**Cu Pounds (billions)**	**Au Ounces (millions)**	**Ag Ounces (millions)**
Proven	239.5	0.625	0.343	6.01	3.30	2.64	46.28
Probable	301.3	0.503	0.271	4.78	3.34	2.63	46.30
Total	540.7	0.557	0.303	5.32	6.64	5.27	92.58

3.5 Metallurgy

Section 3.5 was prepared by Mr. Hoe Teh, P.Eng., Hatch

A comprehensive metallurgical program has been completed on fresh drill core samples from 2005 drilling to further validate the flowsheets 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 using CEET and JKSimMet methodologies, mineralogy, and minerals recovery by batch and locked cycle flotation. Models have been developed to project copper, gold and silver recoveries in mining blocks for each pit. Pilot plant campaigns have also been completed, primarily to generate concentrate samples for dewatering tests and marketing purposes, and tailings samples for dewatering tests and environmental purposes.

The 2005 drill cores confirm investigations from previous campaigns that copper occurs predominantly as chalcopyrite and chalcopyrite-bornite in a mixed silicate host. Pyrite occurrence is variable, with pyrite-copper sulphide mass ratio averaging less than the 3:1 ratio observed in the samples in the Preliminary Economic Assessment (PEA) study. At a grind of 80% passing 150 microns, 50% to 60% of copper sulphides are liberated resulting in good overall copper recovery by flotation. The gold particles are fine at 8 to 12 microns nominally, and are contained largely within the sulphide matrix. Although unlikely to be recoverable by gravity concentration, they exhibit a high recovery by flotation into the rougher copper concentrate. A primary grind of 80% passing 200 microns could be employed to achieve the same metals recovery. The metallurgical response deteriorates as the grind approaches 300 microns.

Ore hardness, in terms of Bond Ball Mill Work Index, varied between 13 kWh/t and 21 kWh/t over the various pits. The average hardness in the dominant Central Pit was 16.5 kWh/t, similar to that determined in the PEA work. The hardness, measured as SAG Power Index (SPI), ranged from 20 minutes to 141 minutes across the deposit. The MinnovEX CEET model indicated that the proposed mill circuit would be SAG mill limiting when treating ores with SPI greater than 115 minutes.

For 65,000 tpd ore throughput, a SABC grinding circuit comprising a 40 ft (12.2 m) x 24 ft (7.3 m) SAG mill, two 26 ft (7.9 m) by 36 ft (11 m) ball mills and two 600kW pebble crushers has been proposed based on the CEET and JKSimMet models and optimal equipment design. The concentrator is able to process 71,500 dry tpd in those areas (e.g. central broken zone) which have a hardness of 16.5 kWh/t or less. The "stick" rock is generally harder and more abrasive than the "broken" rock.

The feasibility program has further validated the flowsheet developed in the previous work. The flowsheet will comprise of crushing, primary grinding, rougher flotation, regrind of rougher concentrate and three stages of cleaner flotation using a simple reagent scheme that utilizes PAX as the primary collector and MIBC as the frother. The use of 3418A, a more selective dithiophosphinate collector, instead of PAX, might produce slightly higher concentrate grade at similar recovery. Occasionally, a guar gum carboxymethyl cellulose reagent will be required to disperse talc-like materials and minimize their adverse impact on flotation responses. Variable amounts and occurrences of these talc-like materials have been observed in the drill cores from certain areas of the deposit. The talc-like mineral species have not been identified.

The program also confirmed that chalcopyrite and bornite ores from various pits have similar metallurgical responses.

Models have been developed for each pit to project copper recovery from variable head grades at constant concentrate grade and to project gold and silver recoveries from copper recovery. Using a head grade of 0.7% copper for each pit, the projected recoveries are as follows:

- Central Pit – 92% Cu, 76% Au, 71% Ag at 28% Cu concentrate grade
- Southwest Pit – 88% Cu, 68% Au, 57% Ag at 26% Cu concentrate grade
- North Junction Pit – 88% Cu, 70% Au, 62% Ag at 28% Cu concentrate grade
- West Fork Pit – 91% Cu, 70% Au, 68% Ag at 28% Cu concentrate grade.

A model has also been developed for projecting copper recovery from ores containing variable amounts of non-sulphide copper (as a small fraction of the ore near surface has localized oxidization). Copper recovery will be lower in these few zones and will vary with the proportion of non-sulphide copper content while the gold and silver recoveries will correlate with copper recovery. Using a head grade of 0.7% copper and assuming 20% of the total copper occurring as a non-sulphide, the model projects recoveries of 71% copper, 55% gold and 51% silver at a 28% Cu concentrate grade.

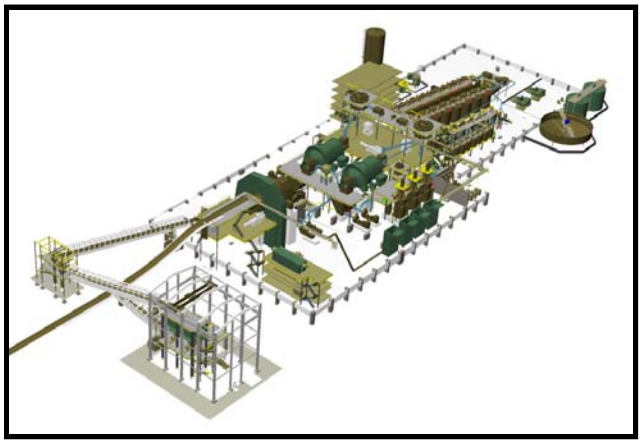
Since gold and silver recoveries largely follow copper recovery, the gold and silver in ores with very low copper, and largely occurring within pyrite grains, would not be recovered.

The concentrates are relatively clean, with the exception of variable fluorine content, as indicated in the Neil Seldon marketing report. This may attract some penalty although the dollar amount is not likely to be significant.

3.6 Process and Plant Site

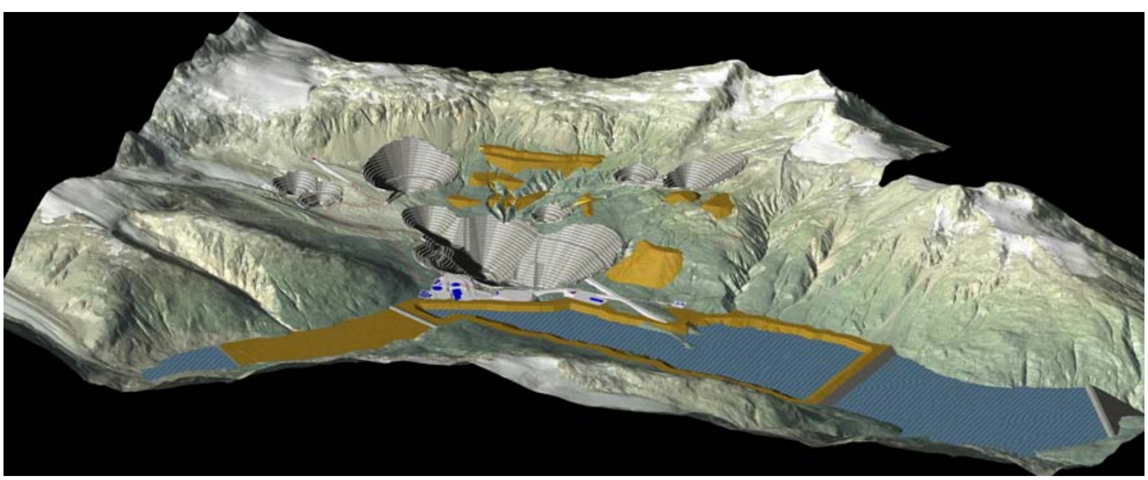
Section 3.6 was prepared by Mr. Hoe Teh, P.Eng., Hatch

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 450,000 tpa of copper concentrate (600,000 tpa over the first ten years) containing both gold and silver. Concentrate will be pumped through a lined 175 mm diameter pipeline approximately 135 km to a 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.

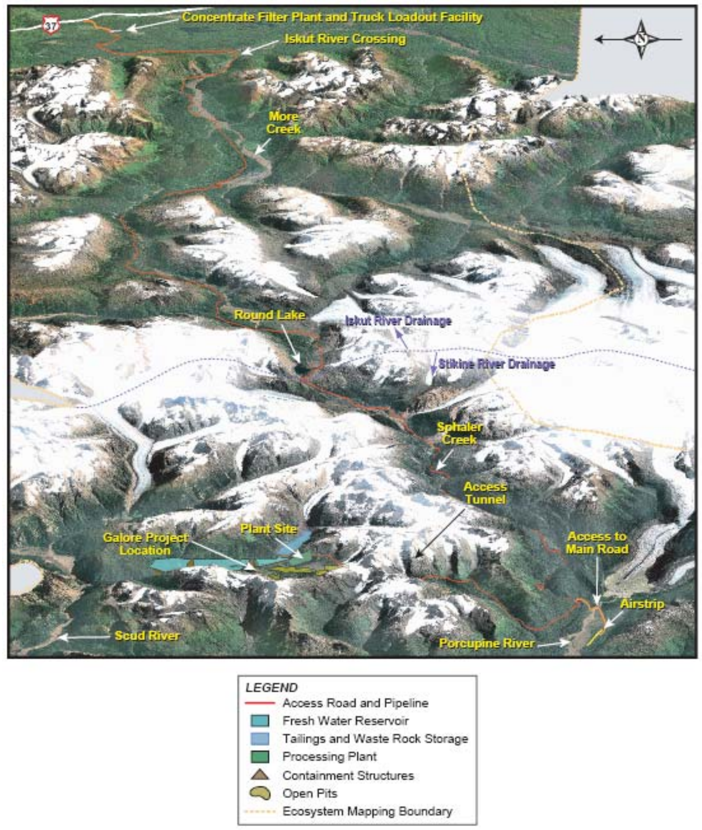


Tailings and waste rock will be stored in a single dam within the Galore Creek Valley, sized to accommodate all the tailings and Potentially Acid Generating (PAG) waste rock. Non-Potentially Acid Generating (NPAG) rock will be stored in a number of dumps located along the west side of the Valley, north 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 for both construction and operations. 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 an aqueduct will be constructed in the East Fork Creek to divert fresh water into 12 km long diversion channel on the east side of Galore Valley that will discharge into Galore Creek downstream of the tailings dam. Testwork and water quality modeling has predicted that surplus water in the tailings/waste dam can be discharged without treatment.



3.7 Access and Power Supply



The Galore Creek feasibility scope includes all the infrastructure associated with providing an access supply corridor for goods and services. The access corridor 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.

Elements of the access corridor include:

- Mine Access Road
- Power Transmission Line
- Mine Access Road Tunnel
- Concentrate / Diesel Pipeline

The Galore Creek Project is currently only accessible by helicopter. In order to gain access a road (the Access Road) must be constructed complete with a 4.3 km tunnel (the Access Tunnel). 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 the access tunnel to access Galore Creek Valley. The access road is classified as a resource development road and is similar to roads developed throughout Northern British Columbia for logging and mine development. The access road will have a 6 m wide surface with line-of-site turnarounds.

Power will be supplied from a connection to the BC Hydro 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 will be bidirectional to ensure power availability off of the main grid.

3.8 Schedule

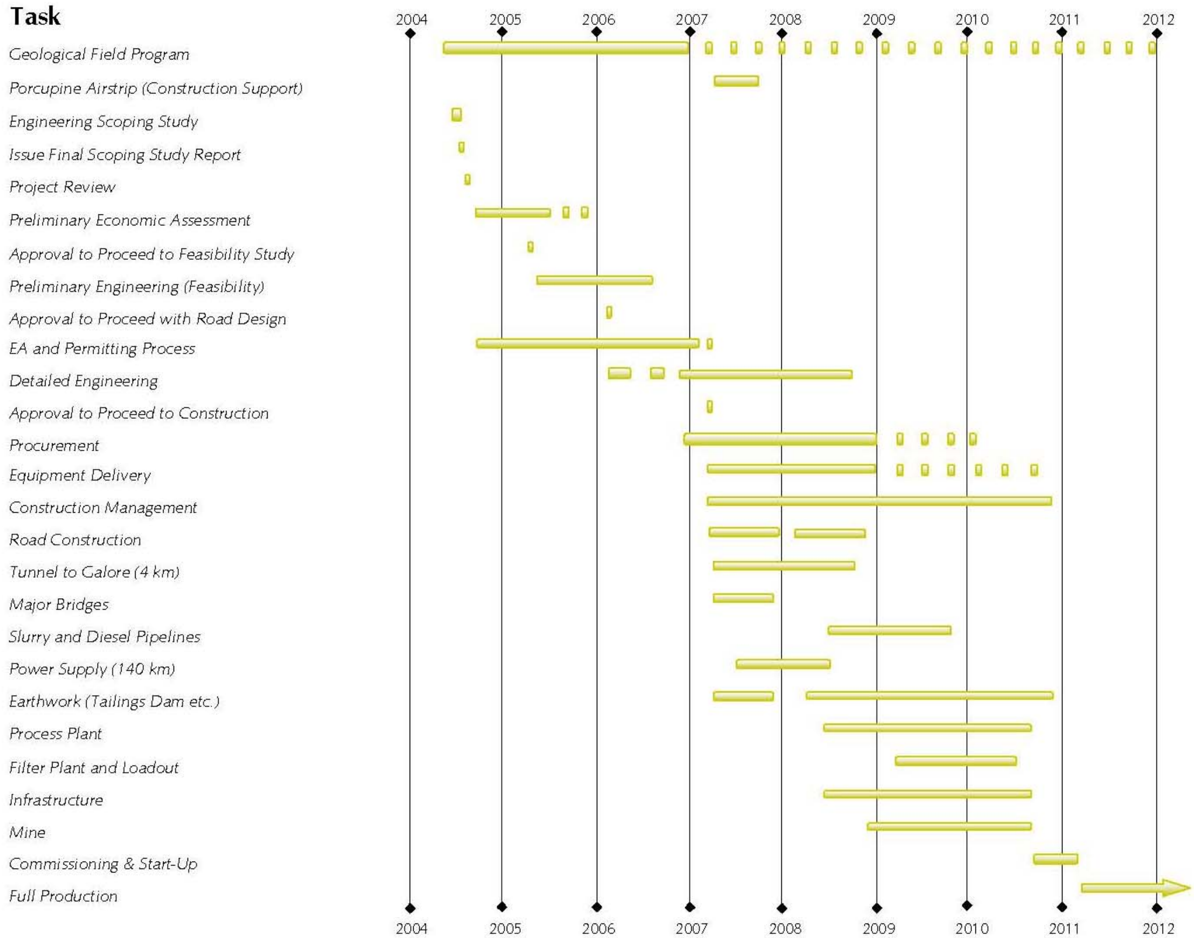
Section 3.8 was prepared by Mr. Bruce Rustad, P.Eng., Hatch

A construction schedule has been developed indicating a four year construction period commencing April 2007. The critical path for the schedule flows through construction of the access road and tunnel, followed by construction of the water diversion channels and core till placement associated with construction of the dam structures. Initially, construction operations will be supported by helicopter from the Bob Quinn Filter Plant Laydown Area. Access to the Galore Valley is expected to be gained after the completion of the 4.3 km Road Access Tunnel in the 3rd quarter of 2008. An additional 28 months will be required to complete the mine facilities. Start up is scheduled for the 1st Quarter of 2011. A summary of the schedule milestones is provided below.

3.8.1 *Project Milestones:*

- **2006**
  - Feasibility Study Completion and successful project financing.
  - Successful completion of the EA and permitting process.
  - Early procurement of the mining equipment.
  - Early engineering activities on road, tunnel and water diversion
- **2007**
  - Engineering of Process Facilities
  - Receive construction permits.
  - **Phase 1** - Construction start: April, 2007.
  - Set up construction laydown area near highway 37.
  - Launch helicopter mobilization of road, tunnel and water management contracts.
  - Construction of the access road, bridges and tunnels to gain access to the minesite.
  - Powerline construction will lag road construction and be completed shortly after tunnel breakthrough.
- **2008**
  - **Phase 2** - Construction
  - Access to the Galore Valley by September 2008.
  - Construction of mine truck shop to support construction mine fleet.
  - Construction of plant substation to support construction activities.
  - Pipeline construction will follow road and power to level construction camp capacity.
  - Completion of construction water diversion system.
- **2009**
  - 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.
  - Season 1 of tailings dam clay till core Placement.
  - Filter Plant construction
  - Primary Crusher construction
  - Placement of concentrator SAG and Ball Mills
  - Concentrator building closed in for winter.
- **2010**
  - Construction of the plant and infrastructure.
  - Season 2 of tailings dam clay till core Placement.
  - Commissioning
- **2011**
  - Plant Handover to operations (first Quarter).

3.8.2 Project Schedule



3.9 Capital Costs

Section 3.9 was prepared by Mr. Bruce Rustad, P.Eng., Hatch

The capital costs for the Galore Creek Project including process plant, infrastructure and mining were estimated using 2nd quarter of 2006 Canadian dollars. This estimate includes no allowance for escalations in prices or fluctuations in exchange rates. The summary capital cost estimate for the Galore Creek Project is presented in Table 3-3 and reflects an intended level of accuracy of $^{+}15\%$ / $^{-}10\%$.

Table 3-3: Capital Cost Summary - Base Case, 65,000 tpd Mill

Description		Responsible Party	Estimate (CA$M)
Mine		GR Tech	457
Concentrator		Hatch	410
Water / Waste Management	Structure Water Recovery	Ledcor Hatch	247
Infrastructure	Tunnel Pipeline Road Power Line	HMM PSI Ledcor IHI	330
Total Directs			1,444
Indirect Costs		All	295
EPCM Costs		Hatch	151
Total Direct and Indirect			**1,890**
Contingency		Hatch	268
Owners Costs		Hatch	70
Total Project Estimate			**2,228**

Other qualifications, assumptions and exclusions that are relevant to the capital cost estimate are set out in Section 25.6 and detailed in the referenced feasibility study.

3.10 Operating Costs

Section 3.10 was prepared by Mr. Bruce Rustad, P.Eng., Hatch

The base case operating cost for the Galore Creek Project including mining, general and administrative and process costs have been estimated using 2nd Quarter of 2006 Canadian dollars. These include no allowances for escalation or exchange rate fluctuations. The summary operating cost estimate is presented in Table 3-4 and reflects an intended level of accuracy of +15%/-10%.

Table 3-4: Operating Cost Summary - 65,000 tpd

Summary	Area	Responsible Party	CA$/t ore
G&A	G&A Labour	Hatch	0.20
	Fixed		0.79
	Total G&A		**0.99**
Mining	**Life of Mine Average**	**GR Tech**	**3.61**
Plant & Infrastructure	Labour	**Hatch**	0.41
	Consumables		1.62
	Maintenance		0.27
	Fuel		0.03
	* Power – Rest of plant site		0.39
	* Power - Mills		0.87
	Total Plant & Infrastructure		**3.59**
Concentrate Filter Plant (Incl. Concentrate/Diesel Pipelines & Utilities)	Labour	**Hatch**	0.09
	Consumables		0.03
	Maintenance		0.02
	Pipeline Maintenance	**PSI**	0.02
	Total Concentrate Filter Plant		**0.17**
Total Mine-site	**Total Mine-site**		**8.36**
Concentrate Transport and Storage	Concentrate Transport	**Hatch**	0.57
	Port loading and storage		0.19
	Total Concentrate Handling		**0.76**
Total Project			**9.12**

* Note: Study power rate at CA$ 0.0524/kWh (including wheeling charges) was provided by W.N. Brazier Associates Inc. (July 2006)

Other qualifications, assumptions and exclusions that are relevant to the operating cost estimate are set out in Section 25.7 and the referenced feasibility study documents.

3.11 Project Economics

Section 3.11 was prepared by Mr. Bruce Rustad, P.Eng., Hatch

Capital and operating costs were developed for a nominal 65,000 tpd operation and cashflow forecasts generated to evaluate the economic performance of the Galore Creek Project, testing the sensitivity to metal prices, exchange rates and input prices. The base case price assumptions are US$1.50/lb Cu, US$ 525/oz Au, US$8.00/oz Ag and a US$ / CA$ exchange rate of 0.81/1.00. An analysis has also been performed using spot prices, a three year trailing price and a low metals price case.

Table 3-5: Metal Recovery

	Copper	Gold	Silver
First Five Years of Production:			
Average Grade	0.82%	0.55 g/t	6.78 g/t
Annual Average Metal Production	432 M lbs	341,000 oz	4.00 M oz
Life of Mine:			
Average Grade	0.57%	0.31 g/t	5.42 g/t
Annual Average Metal Production	262 M lbs	165,000 oz	2.65 M oz
Total Recovered Metal	5.8 B lbs	3.6 M oz	58.50 M oz

A economic analysis using the base case parameters indicates that, the Galore Creek project could generate an after-tax rate of return (IRR) of approximately 10.6% and have an undiscounted after-tax net present value (NPV) of US$1,736 million. Key project economic results are summarized in Table 3-6 below:

Table 3-6: Galore Creek – Summary Economic Results

Economic Results	Units	Base Case	Three Yr. Trailing Ave.	Spot Case (Sept 1, 06)	Low Case
Mine Basis					
Mine Life	years	22			
Ore Tonnage milled	Mt	522			
Strip Ratio		1.64			
Mill throughput (nominal)	tpd	65,000			
Initial capital cost	CA$ (millions)	2,228			
Sustaining capital cost	CA$ (millions)	151			
Unit Operating costs:					
Mining cost per tonne mined	CA$/t	1.50			
Milling / Process cost per tonne ore	CA$/t	3.76			
G&A cost per tonne ore	CA$/t	0.99			
Total Cash Cost (Copper) First 5 Years**	US$ / lb Cu	0.378			
Total Cash Cost (Copper) Life of Mine**	US$ / lb Cu	0.616			
Total Cash Cost (Gold) First 5 Years***	US$ / oz Au	(889)			
Total Cash Cost (Gold) Life of Mine***	US$ / oz Au	(874)			
Total Co-Product (Copper) First 5 Years	US$ / lb Cu	0.67			
Total Co-Product (Copper) Life of Mine	US$ / lb Cu	0.82			
Total Co-Product (Gold) First 5 Years	US$ / oz Au	150			
Total Co-Product (Gold) Life of Mine	US$ / loz Au	200			
Metal price assumptions					
Copper	US$/lb	1.50	1.70	3.50	* 1.27
Gold	US$/oz	525	461	626	* 495
Silver	US$/oz	8.00	7.72	12.87	* 6.70
US$/CA$ exchange Rate		0.81	0.81	0.89	0.75
Cashflow					
Annual Ave. After-tax Net Cashflow (years 1-5)	US$ (millions)	414	445	936	384
Cumulative After-tax Net Cashflow (years 1-5)	US$ (millions)	2,069	2,227	4,678	1,921
Economic Results					
Project IRR (pre-tax)	(%)	14.1	16.6	39.0	12.9
Project IRR (after-tax)	(%)	10.6	12.7	30.7	9.5
NPV 0% discount (pre-tax)	US$ (millions)	2,935	3,689	13,822	2,101
NPV 0% discount (after-tax)	US$ (millions)	1,736	2,189	8,287	1,235
NPV 5% discount (pre-tax)	US$ (millions)	1,187	1,604	7,224	833
NPV 5% discount (after-tax)	US$ (millions)	599	856	4,254	395
Payback	years	4.0	3.7	1.5	3.9

* Average metal price -based on N.Seldon Marketing Report (July 2006) with staggered metal prices.
** Copper cash cost includes gold and silver credits.

*** Gold cash costs include copper and silver credits.

The Feasibility Study evaluated the capital costs, operating and processing costs, taxes and treatment charge for the project. A Metal Prices Sensitivity Matrix is presented in Table 3-7 below. Additional information can be found in Section 25.8.

Table 3-7: Galore Creek Metal Price Sensitivity (US$M)

Cu Price (US$/lb)		Au/Ag Price (US$/oz)						
		450/7.00	**500/7.50**	**525/8.00**	**550/8.25**	**600/8.50**	**650/9.00**	**700/9.50**
1.25	NPV @ 0% (US$M)	807	926	993	1,052	1,164	1,284	1,404
	NPV @ 5% (US$M)	56	128	169	205	272	343	415
	Pre-tax IRR (%)	7.8%	8.7%	9.2%	9.7%	10.5%	11.3%	12.2%
	After-tax IRR (%)	5.6%	6.3%	6.7%	7.1%	7.7%	8.4%	9.1%
	Payback (years)	5.7	5.5	5.3	5.1	4.9	4.6	4.4
1.50	NPV @ 0% (US$M)	1,548	1,669	1,736	1,796	1,908	2,028	2,149
	NPV @ 5% (US$M)	488	559	599	634	700	771	841
	Pre-tax IRR (%)	12.9%	13.7%	14.1%	14.5%	15.2%	15.9%	16.6%
	After-tax IRR (%)	9.6%	10.3%	10.6%	10.9%	11.5%	12.1%	12.6%
	Payback (years)	4.4	4.2	4.0	4.0	3.8	3.7	3.6
1.75	NPV @ 0% (US$M)	2,293	2,413	2,481	2,541	2,654	2,774	2,894
	NPV @ 5% (US$M)	915	985	1,024	1,059	1,125	1,195	1,265
	Pre-tax IRR (%)	17.2%	17.9%	18.3%	18.6%	19.2%	19.9%	20.5%
	After-tax IRR (%)	13.1%	13.7%	14.0%	14.3%	14.8%	15.3%	15.8%
	Payback (years)	3.6	3.5	3.4	3.4	3.2	3.1	3.0
2.00	NPV @ 0% (US$M)	3,104	3,224	3,292	3,352	3,465	3,585	3,705
	NPV @ 5% (US$M)	1,376	1,445	1,485	1,519	1,585	1,654	1723
	Pre-tax IRR (%)	21.4%	22.0%	22.3%	22.6%	23.2%	23.8%	24.4%
	After-tax IRR (%)	16.5%	17.0%	17.3%	17.5%	18.0%	18.5%	18.9%
	Payback (years)	2.9	2.8	2.8	2.8	2.7	2.6	2.6
2.50	NPV @ 0% (US$M)	4,760	4,880	4,948	5,008	5,120	5,240	5,360
	NPV @ 5% (US$M)	2,310	2,379	2,418	2,452	2,517	2,586	2,654
	Pre-tax IRR (%)	28.7%	29.2%	29.5%	29.8%	30.3%	30.8%	31.3%
	After-tax IRR (%)	22.4%	22.9%	23.1%	23.3%	23.7%	24.1%	24.5%
	Payback (years)	2.0	2.0	2.0	1.9	1.9	1.9	1.9
3.50	NPV @ 0% (US$M)	8,070	8,190	8,257	8,317	8,430	8,550	8,670
	NPV @ 5% (US$M)	4,163	4,232	4,271	4,305	4,370	4,439	4,507
	Pre-tax IRR (%)	40.6%	41.0%	41.2%	41.4%	41.8%	42.2%	42.6%
	After-tax IRR (%)	32.0%	32.3%	32.5%	32.6%	32.9%	33.3%	33.6%
	Payback (years)	1.4	1.4	1.4	1.4	1.4	1.4	1.4

Note:

- NPV = net present value using a discounted cash flow analysis;
- IRR = internal rate of return.
- Base case is shown in Green
- Assumes 100% equity
- All NPV and payback figures are after tax

3.12 Conclusions and Recommendations

The Interpretations, Conclusions and Recommendations are provided in Sections 21 and 22.

4. Introduction

Section 4 was prepared by Mr. Bruce Rustad, P.Eng., Hatch

NovaGold commissioned a Feasibility Study for its Galore Creek Project in 2005. The purpose of the Feasibility Study was to assist management of NovaGold in making decisions with respect to the potential development of the Galore Creek Project. In particular, the Feasibility Study was prepared to define the scope of the Galore Creek Project, estimate capital and operating costs and determine the overall economics to develop the property as an open pit mine and mill facility. The Feasibility Study follows three years of site and investigative work by numerous companies and consultants. This Feasibility Report is a compilation of the results of the Galore Creek Feasibility Study which was approved by the NovaGold Board of Directors and is being filed in support of the October 25, 2006 news release of NovaGold disclosing the results of the Feasibility Study.

Table 4-1 lists the Consultants Technical Reports specifically produced for this study.

Table 4-1: Feasibility Study - Consultant Technical Reports

Author	Report Title	Date
BGC Engineering Inc.	"Geohazard Assessment"	February 16th, 2006
	"Feasibility Geotechnical Report - Plant Site Design"	September 22nd, 2006
	"Feasibility Geotechnical Report – Waste and Water Management"	April 21st, 2006.
	"Feasibility Geotechnical Report – Open Pit Slope Design"	July 31st, 2006
	"Feasibility Study – 2005 Laboratory Testing Results"	April 28th, 2006
G & T Metallurgical Services Ltd.	"The Metallurgical Performance of Galore Creek Ores"	April, 2006
Hatch Mott MacDonald	"Mine Access Tunnel"	June 28th,2006
Ian Hayward International Ltd.	"Galore Creek Project Power Supply Feasibility Study"	August 2006
JKTech Pty Ltd.	"SMC Test Report on Fourteen Samples"	February 2006
LPS Aviation Inc.	"Aerodromes Concept"	September 15th, 2006
Ledcor CMI Ltd.	"Galore Creek Mine Project – Mine Access Road and Site Facilities"	April/May 2006.
	"Galore Creek Mine Project – Mine Site Dam and Diversion Channel Construction"	July 24th,2006
McElhanney Consulting Services Ltd.	"Galore Creek Mine Access Road – 2005 Study Report - Final"	April 2006
Neil S. Seldon & Associated Limited	"Marketing and commercial Input Into A Feasibility Study For The Galore Creek Project"	July 2006
Pipeline Systems Incorporated	"Overland Pipeline Detailed Feasibility Study and Cost Estimate"	July, 2006
SGS written by Robert Sloan	"Grinding Circuit Design for the NovaGold – Galore Creek Project"	April 2006
SGS Minerals	"Standard Bond Abrasion Test"	March 15th, 2006
SGS written by Catherine McInnes	"Report on Flotation testwork and FLEET Circuit Design"	June 2006
W.N. Brazier Associates Inc.	" Galore Wheeling Charges & Line Losses"	July 27th 2006

4.1 Sub Consultants Responsibility

This report is the product of technical contributions from the Consultants listed below. Consultants' contributions are also identified in the text. Consultants were retained by NovaGold Canada Inc. and Hatch. Hatch compiled all contributions to the feasibility report but did not supervise the preparation of, or verify, the information provided by other contributors to this report and takes no responsibility for any sections of this report that were prepared by persons other than Hatch:

- Bruce Rustad, P. Eng. – IQP, Study Manager, Hatch
- Jim Gray, P. Eng. – IQP, Mining Section, GR Technical Services Ltd.
- Michael J. Lechner, RPG. – IQP, Resource Estimate, Resource Modeling Inc.
- Hoe Teh, P. Eng. – IQP, Study Metallurgist and Process Engineer, Hatch
- Iain Bruce, P. Eng , P.Geo– IQP, Tailings Design, BGC Engineering Inc.
- Bob Parolin, P.Eng. – IQP, Access Road, McElhanney Consulting Services Inc.
- Dean Brox, P.Eng. – IQP, Access Tunnel, Hatch Mott MacDonald (HMM)
- Allan Guy, P.Eng – IQP, Power Transmission, Ian Hayward International Ltd.
- Kelly Boychuk, P.Eng – IQP, Earthworks Construction, Estimating and Scheduling, Ledcor CMI Ltd.
- Don Hallbom, P.Eng, – IQP, Pipeline Systems Incorporated (PSI)
- Clem Pelletier President – Rescan Environmental Services Ltd. / RTEC

Note:

- Don Hallbom did not visit the site, however, his head of construction on the project performed the site visit and advised on the site issues. Don did, however, visit and review the testwork lab for the pipeline product behaviour.
- Hoe Teh did not visit the site, however, he did perform several site visits to the testing facilities in Kamloops and Toronto.
- Numerous additional professional staff from the named organisations performed various functions including site and test-work and specific design-work in the preparation of the report.

The persons taking responsibility for certain sections of this Report, and the extent of their responsibility for each section, for the purposes of NI 43-101 are set out in the table below.

Table 4-2: Consultant Companies Commissioned for the Galore Creek Project

Responsible Person	Ind. QP	Consultant	Primary Areas of Responsibilities	Relevant Sections
Bruce Rustad	Yes	Hatch *	Study Compilation, Economic Analysis	Section 3 subsections 1,8,9,10,11 Section 4 Section 5 Section 20.3.1, 6, 7 Section 20.4 Section 21.1 Section 22.10 Section 25.3 Section 25.4 Section 25.5 Section 25.6 Section 25.7 Section 25.8
Michael J. Lechner	Yes	Resource Modeling Incorporated	Geological Investigation Land status and History Drill Plan Core Analysis / classification Block Model Resource Estimate	Section 3.2,3.3 Section 6 Section 7 Section 8 Section 9 Section 10 Section 11 Section 12 Section 13 Section 14 Section 15 Section 16 Section 17 Section 19.1 Section 21.2 Section 22.1
Jim Gray	Yes	GR Technical Services Ltd.	Reserve estimation Economic pit limits Pit Design Mine load / haul equipment selection Mine production schedule Pit period mapping Pre-production construction considerations Mine closure / reclamation Mine capital costs Mine operating costs	Section 3 subsections 4,8,9,10 Section 19.2 Section 21.3 Section 22.2 Section 25.1 Section 25.6 Section 25.7
Iain Bruce	Yes	Bruce Geotechnical Consultants	Geotechnical assessment and design for tailings and waste rock storage Open pit slope stability Water management plan Geohazard evaluation for road, powerline routes and site	Section 3.7 Section 3.8 Section 3.9 Section 20.2 Section 21.5 Section 22.4

Responsible Person	Ind. QP	Consultant	Primary Areas of Responsibilities	Relevant Sections
			Characterization of the foundation conditions for: the tailings and water retaining dams footprints; the waste dump footprint; the freshwater diversion channels and diversion structure footprints; Identification of potential sources of impervious (i.e. clay) borrow and cohesionless (i.e. sand and gravel) borrow. Design of the tailings and water retaining dams, diversion channels and construction facilities; Quantification of earthworks volumes	Section 25.6 Section 25.7
Hoe Teh	Yes	Hatch	Metallurgical Interpretation Process Plant	Section 3.5 Section 3.6 Section 18 Section 20.1 Section 21.4 Section 22.3
Bob Parolin	Yes	McElhanney Engineering	Complete the final route selection Locate and survey the preliminary road centerline Complete site surveys for all fish bearing streams and major creek crossings Perform hydrology analysis for bridge and culvert installations Arrange for geophysical survey of major bridge crossings Design the road horizontal and vertical alignment to avoid or mitigate impact of geohazards, Incorporate mitigation measures in the road design Generate earthwork quantities and prepare a construction cost estimate and schedule of activities.	Section 3.7 Section 3.8 Section 3.9 Section 3.10 Section 20.3.2 Section 22.7 Section 25.6 Section 25.7
Dean Brox	Yes	Hatch Mott MacDonald	Review of geotechnical data Locate north and south portal Finalise tunnel cross section and alignment Identify schedule and construction risks Prepare a capital estimate Prepare a construction schedule	Section 3.7 Section 3.8 Section 3.9 Section 3.10 Section 20.3.4 Section 22.8 Section 25.6 Section 25.7
Allan Guy	Yes	Ian Hayward International Ltd.	Load stability analysis Structure and line design Substation design Cost estimate Construction schedule System design criteria	Section 3.7 Section 3.8 Section 3.9 Section 3.10 Section 20.3.3 Section 22.9 Section 25.6 Section 25.7
Don Hallbom	Yes	Pipeline Systems	Concentrate pumping system design Concentrate corrosion analysis	Section 3.7 Section 3.8

Responsible Person	Ind. QP	Consultant	Primary Areas of Responsibilities	Relevant Sections
		Incorporated	Diesel pumping system design Diesel corrosion analysis Construction schedule Pipeline capital costs Operating estimate Supervisory control and emergency response plan	Section 3.9 Section 3.10 Section 20.3.5 Section 22.6 Section 25.6 Section 25.7
Kelly Boychuk	Yes	Ledcor CMI Ltd.	Construction cost estimate and schedule access road Construction cost estimate and schedule water diversion and structures	Section 3.8 Section 3.9 Section 22.5 Section 25.6
Clem Pelletier	No	Rescan Tahltan Environmental Consultants	Stikine River bathymetric survey and environmental permitting	Section 25.4
Neil Seldon	No	Neil Seldon & Associates	Assessment of metal prices, concentrate sales and treatment charge forecasts	Section 25.3
Tom Shouldice	No	G&T Metallurgical	Metallurgical testwork	Section 18.3 Section 18.4 Section 18.5
Catherine McKinnes	No	MinnovEX Technologies	Grinding circuit design	Section 18.7, 18.8
LPS Aviation	No	LPS Aviation	Aerodrome assessment, costing and requirements	Section 20.3.7
Neil Brazier	No	W.N. Brazier Associates Inc	System planning and performance assessment Assessment of transmission wheeling charges	Section 25.7

* Hatch has prepared Sections 3.1, 21.6 and 22.10, however, these sections summarize information provided by other consultants on behalf of the owner and Hatch assumes no responsibility or liability for information that is not within Hatch's specific areas of responsibility.

* Hatch has prepared Section 3.8, however, this section summarizes information provided by other consultants on behalf of the owner and Hatch assumes no responsibility or liability for information that is not within Hatch's specific areas of responsibility.

* Hatch has compiled the cost estimates in Sections 25.6 and 25.7 and the economic analysis in Section 25.8, but, does not assume any liability or responsibility for information which has been incorporated or used in these estimates or analysis and which were supplied other consultants (e.g. see tables 25.19 and 25.21).

* The mineral resource estimate in this Feasibility study was completed by Mr. Michael J. Lechner,RPG from Resource Modeling Incorporated an independent Qualified Person as defined by NI 43-101 and is the subject of a technical report entitled "Updated Galore Creek Mineral Resources, Northwestern British Columbia", dated September 7, 2006 and posted to SEDAR on September 12, 2006.

DISCLAIMER

This Report is directed solely for the development and presentation of data with recommendations to allow for NovaGold Canada Inc. to reach informed decisions. Except for the purposes legislated under provincial securities law, (a) any use of this report by any third party is at that party's sole risk, and none of the contributors to this Report nor any of their respective directors, officers, or employees shall have any liability to any third party for any such use for any reason whatsoever, including negligence, and (b) the contributors to this Report disclaim responsibility for any indirect or consequential loss arising from any use of this report or the information contained herein.

This report is intended to be read as a whole, and sections should not be read or relied upon out of context. This Report contains the expression of the professional opinions of the contributors to this Report and other consultants, based upon information available at the time of preparation. The quality of the information, conclusions and estimates contained herein is consistent wit the intended level of accuracy as set out in this report, as well as the circumstances and constraints under which the report was prepared which are also set out herein.

As permitted by Item 5 of Form 43-101F1, the contributors to this Report have, in the preparation of this Report, relied upon certain reports, opinions and statements of certain experts. These reports, opinions and statements, the makes of each such report, opinion or statement and the extent of reliance is described in Section 5 of this Report. Each of the contributors to this Report hereby disclaim liability for such reports, opinions and statements to the extent that they have been relied upon in the preparation of this Report, as described in Section 5.

As permitted by Item 16 of Form 43-101F1, the contributors to this Report have, in the preparation of this Report, relied upon certain data provided to them by NovaGold Canada Inc. and certain other parties.

None of the contributors to this Report accept any responsibility or liability for the information in this Report that was prepared by other contributors.

5. Reliance on Other Experts

Section 5 was prepared by Mr. Bruce Rustad, P.Eng., Hatch

In preparing its sections of this Report, Hatch has relied upon certain reports, opinions and statements of other experts. These reports, opinions and statements, the makers of each such report, opinion or statement and the extent of reliance is described below. Hatch hereby disclaims liability for such reports, opinions and statement to the extent that have been relied upon in the preparation of this Report, as described below.

5.1 Previous technical report

Information on the items set out in Sections 6 to 11 of this Report can be found in the previous Galore Creek Mineral Resources Independent Technical Report dated September 7, 2006 and posted to SEDAR on September 12, 2006. To Hatch's knowledge, there has not been any material change in the information since that date.

5.2 Environmental and Legal

NovaGold Canada Inc, as part of the permitting process for a new mine in the British Columbia, has recently submitted a full environmental impact assessment (EIA) report to the government to facilitate the issuance of relevant permits and Licenses required for mining operations. RTEC (Rescan –Tahltan JV) has provided environmental data and information to Hatch and other Consultants in connection with Section 25.4.

5.3 Marketing

A marketing study was performed by Neil S. Seldon & Associates Limited. This study provides current smelter terms for copper, gold and silver concentrates, possible buyer issues and various viewpoints. This technical report has relied upon this study as well as other market related information in the preparation of Section 25.3 of this Report.

5.4 Tax

The tax component of the economic description and the cashflow model was provided by Ernst and Young. This model was utilized to complete the economic analysis in Section 25.8.

5.5 Other Information

In addition to relying on information provided by other QPs (as detailed in Section 4), Hatch has relied on certain data and information provided by other independent consultants in preparation of the capital cost estimate, operating cost estimate and economic analysis are set out in Sections 25.6, 25.7 and 25.8. Hatch has relied on this information without independent verification.

6. Property Description and Location

Refer to "Updated Galore Creek Mineral Resources, Northwestern British Columbia", dated September 7, 2006 and posted to SEDAR on September 12, 2006. This report was prepared by Mr. Michael J. Lechner,RPG from Resource Modeling Incorporated.

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

Refer to "Updated Galore Creek Mineral Resources, Northwestern British Columbia", dated September 7, 2006 and posted to SEDAR on September 12, 2006. This report was prepared by Mr. Michael J. Lechner,RPG from Resource Modeling Incorporated.

8. History

Refer to "Updated Galore Creek Mineral Resources, Northwestern British Columbia", dated September 7, 2006 and posted to SEDAR on September 12, 2006. This report was prepared by Mr. Michael J. Lechner,RPG from Resource Modeling Incorporated.

9. Geological Setting

Refer to "Updated Galore Creek Mineral Resources, Northwestern British Columbia", dated September 7, 2006 and posted to SEDAR on September 12, 2006. This report was prepared by Mr. Michael J. Lechner,RPG from Resource Modeling Incorporated.

10. Deposit Type

Refer to "Updated Galore Creek Mineral Resources, Northwestern British Columbia", dated September 7, 2006 and posted to SEDAR on September 12, 2006. This report was prepared by Mr. Michael J. Lechner,RPG from Resource Modeling Incorporated.

11. Mineralization

Refer to “Updated Galore Creek Mineral Resources, Northwestern British Columbia”, dated September 7, 2006 and posted to SEDAR on September 12, 2006. This report was prepared by Mr. Michael J. Lechner,RPG from Resource Modeling Incorporated.

12. Exploration

Section 12 was prepared by Mr. Michael J. Lechner,RPG. Resource Modeling Incorporated

The aims of the 2005 exploration drill program were to upgrade resource blocks within the main deposits, to test for extensions of known mineralization, and to explore for new targets within the Galore Creek valley. Additional drilling was utilized for engineering and environmental testing. Mapping focused on defining drill targets, major structures, and alteration assemblages, as well as recognizing sedimentary facies transitions. The geophysical exploration program included a wide-spaced Vector IP reconnaissance program and Induced Polarization surveys both south of the Central Zone and along the East Fork of Galore Creek.

The 2005 Galore Creek diamond drill program included 63,189.89 metres of HQ and NQ sized core recovered from 260 diamond drill holes. 213 holes, totaling 61,320.88 metres, were part of the geology exploration drill program with the remaining 37 holes drilled for geotechnical purposes.

12.1 Extent of All Relevant Exploration

Geological mapping, along with results from surface geochemical and geophysical studies, have added considerable value to the project. Table 12-1 lists the relevant exploration work that has been completed at Galore Creek along with the contractor name and supervisor.

Table 12-1: Relevant Exploration Work

Job Function / Year	Operator / Owner	Contractors	Work Performed
Geology			
1957-91	Hudson Bay / Kennco	unknown	
2003	NovaGold	Vancouver Petrographics	Petrography
2004	NovaGold	John Proffett, Vancouver Petrographics	Mapping, Petrography
2005	NovaGold	John Proffett, Vancouver Petrographics	Mapping, Petrography
Laboratory			
1957-89	Kennco	unknown	
1990	Mingold	TSL Laboratories	
1991	Kennco	Min-En	
2003	NovaGold	ALS Chemex Laboratories	
2004	NovaGold	ALS Chemex Laboratories	Geochemical Analysis, Specific Gravity determinations
2005	NovaGold	ALS Chemex Laboratories	Geochemical Analysis, Specific Gravity determinations
Geophysics			
1962	Kennco		Airborne Mag
1964	Kennco		IP
1966	Kennco		IP, Ground Mag
1989	Kennco	Aerodat	Airborne EM
2004	NovaGold	Fugro, Zonge, Frontier, Aurora	Airborne Mag, Radiometrics, Ground IP, Seismic, Ground Mag
2005	NovaGold	Frontier	IP
Drilling			
1957-90	Kennco, Hudson Bay, Mingold		AQ,BQ,BTW and NQ diamond drilling
2003	NovaGold	Britton Brothers	HQ and NQ diamond drilling
2004	NovaGold	Britton Brothers	HQ,NQ and BQ diamond drilling
2005	NovaGold	Hy –Tech, Cyr	HQ,NQ and BQ diamond drilling

12.2 Results of Surveys, Procedures and Parameters

Regional stream silt geochemistry was instrumental in the discovery of the mineralization at Galore Creek and more detailed silt sampling programs were carried out in 1960-61 and 1989.

A significant area of the property lacks sufficient soil development for soil geochemistry to be of any practical use. Soil grids were established in the areas around the North Rim and Southwest zones. A few reconnaissance traverse soil lines were also sampled along topographic contours between the Saddle Zone and the Central Zone.

In 1991, 600 soil samples were collected from a grid established in the North Junction / North Rim area. Samples were taken on 20-metre stations along lines spaced 100 metres apart. A coincident Cu-Au soil anomaly with peak values of 9060 ppm copper and 550 ppb gold was located over the North Rim area. A total of 63 surface rock chip samples were also collected from various outcrops on the property.

At least two previous geophysical surveys, dating back to 1961, have been conducted on the Galore Creek property. In the period of 1961 to 1967, major surveys included aeromagnetics, dipole-dipole IP/Resistivity, ground magnetics and AMT. Between 1989 and 1991 a second episode of geophysical exploration occurred. Surveys included an Aerodat helicopter magnetic survey, EM and radiometric survey, ground magnetics/VLF and 60-metre pole-dipole IP/Resistivity surveys.

Geophysical surveys completed at Galore Creek during 2004 included the following:

- A helicopter flown by Fugro Airborne Surveys supported magnetic and radiometric surveys to the north and east covering the Grace and Copper Canyon claims and covered 1,072 line kilometres.
- Wide-spaced, large dipole IP/Resistivity lines combined with 2D IP/Resistivity modeling were used to extend the depth of mineral exploration. The work was conducted by Zonge Engineering, and covered approximately 28 linear kilometres on 17 lines using a 100 or 150-metre dipole-dipole array.
- Shallow seismic refraction surveys for engineering design were run by Frontier Geosciences. The survey covered a total of 10.5 kilometres on 11 lines using 10-metre spaced geophones.
- A ground magnetics survey was completed by Aurora Geophysics that used a 25 metre line spacing with 5 metre stations across the Opulent vein in the West Fork area.

The 2005 geophysical exploration program included a wide-spaced, Vector IP reconnaissance program, a 2 kilometre line of 100-metre pole-dipole IP/resistivity (Line 18) along the south bank of the East Fork of Galore Creek and a 1.5 kilometre by 1.5 kilometre 3D IP survey south of the Central Zone. The surveys were conducted by Frontier Geosciences using their PC based full waveform time domain IP receiver. The reconnaissance survey recorded 55 wide-spaced IP and resistivity stations in an area of 40 square kilometres. This survey produced anomalous IP responses over the entire survey area with relatively lower IP responses associated with the Central Zone and north along the Galore Creek drainage. Survey results are consistent with a large sulphide alteration system associated with a porphyry copper system. Line 18 was run to get detailed information across an area of high reconnaissance IP responses in the East Fork of Galore Creek drainage. Drilling showed that the IP responses were associated with barren pyrite. The 3D IP survey used a pole transmitter and 100 and 200-metre receiving dipoles located north and south of the transmitter lines to build a 3D mesh of readings. Inverted models showed a surface resistivity low associated with overburden and fractured rock and surprisingly low IP effects compared to previous 2D model data.

12.3 Underground Development

This section has been adapted from Hatch et al. (2004).

In order to extract a 50-ton bulk sample for pilot plant testing, an adit was driven into the Central Zone and samples were collected from four crosscuts. The work was carried out by Haste Mine Development between August 1966 and January 1967 and totaled 799 metres of underground drifting (2m x 2m). The rock quality in this part of the Central Zone was found to be generally weak and intensely fractured in gypsum-free areas, but tough and competent in zones of gypsum cementation.

Seven underground diamond drill holes were collared from the 2070 adit. Severe recovery problems were encountered because the holes were of small diameter and drilled sub parallel to the flat-lying, sheet fractures. No assay data were located for these holes.

Sampling of the adit and drift ribs was carried out over continuous horizontal 3- metre intervals plus vertical channels alongside the traces of diamond drill holes. Although commonly referred to as "channel" samples, one internal memo described them as "contentious (sic) chip samples". The vertical samples taken adjacent to the drill hole traces correlated within 0.1% copper. There was considerable variability between rib samples that were collected from opposite sides of the drift, particularly in higher-grade areas above 1.5% Cu where massive blebs of chalcopyrite were randomly distributed. In these areas variations often exceeded 0.4% copper for opposing walls. Subsequent check sampling along some of the same channels confirmed this variation.

At the North Junction Zone, a smaller adit (1.2m x 2.1m) was collared in badly fractured and altered tuff. After driving through 26 metres of material grading about 0.5% copper, a low-grade dike was encountered. Total length of the adit was 51 metres.

12.4 Interpretation of Exploration Results

Based upon the historical and NovaGold exploration information, a significan copper porphyry deposit was interpreted within the Galore Creek claim boundaries.

13. Drilling

Section 13 was prepared by Mr. Michael J. Lechner,RPG. Resource Modeling Incorporated

Refer to Simpson, 2003; Hatch et al., 2004; and Morris 2005 for more information on drilling prior to the 2003 programs. There has been no material change to this information.

An area exceeding 0.5 km by nearly 1 km has been tested by exploration drilling. The nominal sample length of 2.5 m is appropriate given the porphyry-style of mineralization and the proposed bulk mining method. The drill hole spacing varies across the drilled extent but is nominally 75 meters. In general, samples are of adequate quality for use in resource estimation. A discussion of sample quality and potential biases are discussed in Section 16-3.

13.1 Pre-2003 Drilling

Prior to NovaGold's involvement with Galore Creek in 2003, there were about 439 diamond core drill holes completed totaling 99,637 metres. Most of these holes were located in the Central Zone, with lesser amounts of work conducted on eleven other areas. Some of the mineralized zones only received reconnaissance level drilling.

During the 1970's drilling was principally confined to the Central Zone with nine holes drilled into the North Junction Zone. In the Central Zone the average core recovery was between 75% and 85% with the poorest recovery at depths between the surface and 90 metres where open sheet fractures were encountered. At depths below 90 metres, core recovery typically approached 100%. In the North Junction Zone core recovery averaged about 60% due to shattered and sheared sections that were encountered both near the surface and at various intervals throughout the holes.

The summary of drilling campaigns by period is shown in Table 13-1

Table 13-1: Summary of Drilling

Period	Hole Prefix	Purpose of Drilling	Holes	Metres	Core Size	Assayed Samples
1961	GC61	Exploration	5	363	BQ	108
1962	GC62	Exploration	39	4,681	AQ, BQ	1,084
1963	GC63	Exploration	49	11,708	AQ, BQ, NQ, BTW	3,444
1964	GC64	Exploration	52	11,271	Unknown, NQ, BQ	3,102
1965	GC65	Exploration	84	18,585	Unknown, NQ	5,281
1966	GC66	Exploration	18	5,990	NQ, BQ,	1,622
1966	UG	Exploration, Bulk Sample	12	1,491	Underground Channel Samples	455
1972	GC72	Exploration	50	10,416	NQ, BQ	2,553
1973	GC73	Exploration	61	14,689	NQ, BQ, HQ	4,096
1974	PC74	Exploration	4	430	Unknown	0
1976	GC76	Exploration	25	5,317	NQ, BQ, HW, NW	1,356
1990	GC90	Exploration	17	1,893	BDBGM, Unknown	597
1990	PC90	Exploration	4	785	Unknown	0
1991	GC91	Exploration	49	13,820	NQ, NW, BQ, Unknown	3,566
2003	GC03	Exploration	10	2,950	HQ, NQ, BQ	1,243
2004	GC04	Exploration	63	20,866	HQ, NQ2, HW	7,599
2004	PC04	Exploration	6	1,445	Unknown	465
2005	GC05	Exploration	200	58,887	HQ, NQ2, HW, HWT, HQ3	22,267
2005	PC05	Exploration	10	1,679	HQ, HQ2, HW, NW	577
Total			**758**	**187,267**		**59,415**

13.2 2003 NovaGold Drilling

During 2003, NovaGold completed an eight hole 3,000-metre drill program to verify previous results and to better understand grade variability, mineral zonation and potential controls of mineralization. Particular emphasis was directed at understanding the variability of gold content in the deposit. All eight diamond core holes were angle drilled in order to intersect the mineralized structures as close to right angles as possible. All six holes within the Central Zone were drilled towards the east and focused on the gold-rich lenses at the north and south ends of the zone as well as the central, higher-grade copper replacement zone. The two remaining holes were drilled in the Southwest Zone. These holes were angled towards the north. In addition, two holes were lost in overburden and not completed. All of the 2003 drilling was completed at NQ size. NovaGold's core storage and handling methods that were developed in 2003 and subsequently used for their 2004–2006 campaigns are described in more detail in Sections 13.4 and 13.5.

Drill hole GC03-441, drilled in 2003, was targeted to test both an upper mineralized horizon and the potential for a lower mineralized zone. The hole successfully intersected the upper horizon and encountered a new mineralized horizon that had not been tested in previous drilling and is now known as the Bountiful Zone. The intercept totaled 65 metres in width and did not exit the mineralized section before the hole was terminated.

13.3 2004 NovaGold Drilling

Diamond drilling in 2004 targeted eight different mineralized areas: the Central Zone (which includes the North Gold Lens, Central Replacement Zone, and South Gold Lens), the Copper Canyon property, the Gap Zone, the Grace claims, the North Junction Zone, the Saddle Zone, the Southwest Zone, and the West Fork Zone. Drill hole collar locations were selected to test surface mineralization and geophysical targets, confirm results from past drilling, and to extend the limits of known mineralization. Britton Brothers Diamond Drilling out of Smithers B.C. completed the 2004 work using both skid and helicopter portable drill rigs. All of the 2004 drilling on the property has been continuous core diamond drilling, using HQ, NQ and BQ size core. Exploration drilling in 2004 resulted in the discovery of the West Fork and Opulent Zones.

13.4 2005 NovaGold Drilling

Diamond drilling in 2005 primarily focused on (1) infill drilling and upgrading of Inferred and Indicated Mineral Resource blocks to Indicated and Measured status within proposed pits; (2) delineation of pit boundaries, which were based on the 2004 resource model; (3) expansion and/or extension of known mineralization through step-out drilling; (4) targeting of known surface mineralization and structures based on previous mapping; (5) targeting of previously identified geophysical anomalies; and (6) geotechnical and environmental drilling for future assessment. Due to the large scale nature of the 2005 program, NovaGold subdivided the Galore Creek Project into four discrete areas with geologic teams assigned to each area. Team 1 was responsible for the Central Zone (which includes the North Gold Lens, Central Replacement Zone, and South Gold Lens). Team 2 was responsible for Exploration/Reconnaissance targets (which includes Middle Creek, Butte, North Rim, the Grace Claims, and IP and Airborne Magnetic Drilling). Team 3 was responsible for the Southwest and West Fork Zones, as well as the "Gap" between these two zones. Team 4 was responsible for the North Junction Zone and the Copper Canyon Property.

The 2005 drilling work was completed by two companies: Cyr Drilling International Ltd. of Winnipeg, Manitoba who provided three Longyear 38 skid-mounted drill rigs, and Hy-Tech Drilling Ltd. of Smithers, British Columbia who provided five custom-built S-5, S-10 and B-15, helicopter supported fly rigs. HQ-sized rods were used in drilling through the "broken" rock and NQ-sized rods through "stick" rock. The holes were surveyed using Reflex down-hole cameras on 50-metre intervals. Oriented core measurements were taken on roughly 20% of the exploration drill holes. Most geotechnical holes and all water-monitoring holes were drilled with two HT-750 top drive rotary drill rigs, provided by Foundex Explorations Ltd. of Surrey, British Columbia. Triple tube core barrels were used for all geotechnical drill holes and whenever possible, each drill run was oriented using an Ezy-Mark core orientation tool.

13.5 2003-2005 Drilling Procedures

The procedures used to locate exploration drill holes between 2003 and 2005 were generally as follows: the proposed drill site was located in the field by a geologist using a hand-held GPS unit; a pad was then built and the drill rig placed on the site by helicopter or dragged into position using a bulldozer. The orientation of the drill hole was set by the geologist with a set of pickets to provide the azimuth for the angle hole. The inclination (dip) of the drill hole was also noted on the alignment pickets. Typically most drills were checked by a geologist before drilling began to verify azimuth and inclination. Upon completion, drill hole collars were surveyed using a differential GPS with an Ashtech receiver. Down-hole surveys were completed using a Reflex E-Z-shot tool on 50-metre intervals. In most cases the drill pipe was removed from the hole with surface casing occasionally left to mark the hole location. When casing was not left in the hole a cement plug and wooden stake were used to identify hole locations. Artesian holes were plugged and capped to minimize surface water flow in the area.

All drill core was transported by helicopter or truck in secure core "baskets" to the Galore camp for logging and sampling. Although the rocks are complicated and their genetic interpretations may vary slightly from those determined historically, a policy of "correlateable units" was used and rocks were coded according to the historic nomenclature. Additionally, a few new rock codes were created to accommodate lithologies found in 2005 that were not present in the rock type dictionary.

Logging included coded and textural descriptions of lithology, alteration, mineralization, structure, core recovery and rock quality designation (RQD). Geotechnical measurements, including rock strength, specific gravity, fracture density, and fracture filling were also recorded. Data were entered in an Access database using DDH Tool, an in-house front-end data entry program constructed in Visual Basic. After geologic and geotechnical logging, the core was photographed and one sample for approximately every ten metres of core was selected for point load testing and specific gravity measurements. Once the core was sawed, half was sent to ALS Chemex Labs for analysis and the other half stored at the Galore Creek camp for future inspection. In addition to the core, control samples were inserted into the shipments at the approximate rate of one standard, one blank and one duplicate per 17 core samples. Petrographic analyses in 2005 were completed on 17 samples by Vancouver Petrographics; these include Galore Creek and historic drill core samples.

13.6 Sample Length/True Thickness

Sample intervals were determined by the geological relationships observed in the core and limited to a 3-metre maximum length and 1-metre minimum length. An attempt was made to terminate sample intervals at lithological and mineralization boundaries.

The term "true thickness" is not generally applicable to porphyry-like deposits as the entire rock mass is potentially ore grade material and there is often no preferred orientation to the mineralization. Because of the potential of ore grade material through the entire length of the hole, sampling was generally continuous from the top to the bottom of the drill hole. The mineralization is generally confined to three main lithologies: volcanic rocks, intrusive rocks, and breccias. These lithologies form large massive bodies within the Galore Creek deposit.

13.7 Interpretation of Results

The exploration drilling has defined a large copper-Gold porphyry deposit. The orientation and continuity is variable across the area currently defined by drilling. In general the mineralized zones are several hundred meters wide with reasonable continuity of grade along strike and down dip.

In general drilling has not closed off potential mineralization for several of the mineralized zones and NovaGold plans to continue exploration drilling for the foreseeable future. To the extent presently drilled, the drilling has adequately tested the target. The drilling database is inclusive of the 2005 drilling campaign. The 2006 drilling information is not included in the September 2006 Resourse estimate.

Michael J. Lechner, is not aware of any drilling, sampling, or recovery factors, other than the comments in section 16-3 regarding apparent low biases associated with several pre NovaGold drilling campaigns.

14. Sampling Method and Approach

Section 14 was prepared by Mr. Michael J. Lechner,RPG. Resource Modeling Incorporated

Samples in the Galore Creek project database come only from drill core; there are no trench or grab samples in the database. Drill hole sample intervals in 2005 were determined by a geologist and averaged 2.4 metres in length. Due to the style of mineralization and difficulty in determining potential ore from non-ore material, all of the cored material from each drill hole was sampled. When the hole was in unmineralized rock the sample length was generally 3.0 metres, which provides a representative sample weight for NQ core. In mineralized units, the sample length was shortened to 2.0 metres. The core recovery was very high with an average of ~88% in 2005.

Unsampled intervals (from older drilling programs) were mainly late stage, post mineral dykes. Core was split using a diamond saw (older programs used a mechanical splitter). One half of the core was returned to the core box and the other half shipped to an outside laboratory for analysis. The core returned to the boxes remains on site as a record of the hole. Pulps and coarse rejects were stored either on site, in a warehouse in Smithers or in Vancouver. Those remaining on site are in degraded sample bags and are not considered worth salvaging.

Core has been stored in either plastic, galvanized steel or wooden boxes. All have been marked with metal tags inscribed with the hole number and interval. An estimated 1,500 metres of core was spilled in 1972 due to the collapse of a core storage rack. In the winter of 1976 one core shed collapsed and although most of the core was rescued, a number of intervals were not salvageable. Core from the Central Zone was largely re-logged as part of the 1991 exploration program. It is now stacked on pallets exposed to the elements and the top layers have suffered deterioration from weathering. Several intervals have also been removed in the past for the purposes of metallurgical testing. Other intervals have been ¼ split for check assaying.

No site-specific standards, blanks or field duplicate samples were used in any of the previous exploration programs. During the 1991 program, every twentieth sample was re-assayed by an umpire laboratory and internal checks were performed by the main assay laboratory. A QA/QC program was instituted and consisted of assaying duplicate samples along with the insertion of standard reference materials (SRM's) and blanks. A list of all drill hole samples that were used to estimate Mineral Resources is shown in Table 14-1 by mineralized zone.

Table 14-1: Summary of Drill Hole Data

Mineral Zone	No. of Holes	Metres Drilled
Butte	6	1,703.20
Central Replacement Zone	406	104,333.70
Gap	10	3,164.60
Grace	27	4,695.32
Junction	15	2,751.25
Middle Creek	21	4,993.21
North Junction	41	10,432.22
North Rim	15	2,865.64
Proffitt	5	799.48
Reconnaissance	46	9,909.65
Saddle	10	1,360.52
Southwest Zone	86	21,710.49
West Fork	66	17,708.38
West Rim	4	838.84
Grand Total	**758**	**187,266.50**

Notes: ***1 - The "Gap" is the area between the Central and Southwest Zones. 2 - "Reconnaissance" holes were drilled in areas outside of defined zones.***

All significant drill hole composite assays at a 1% total copper cutoff grade are summarize in Table 14-2

Table 14-2: Summary of Significant Drill Hole Composite Assays at 1% Cu Cutoff Grade

Hole ID	From	To	Length	%Cu	Au g/t	Ag g/t	Mineral Zone
GC03-0436	45	50	5	2.67	4.25	11.2	SWZ
GC03-0437	20	25	5	2.77	0.70	12.9	SWZ
GC03-0437	25	30	5	5.06	2.21	25.7	SWZ
GC03-0437	30	35	5	3.22	2.86	51.2	SWZ
GC03-0437	35	40	5	3.04	2.40	52.3	SWZ
GC03-0437	40	45	5	2.32	2.15	14.5	SWZ
GC03-0437	70	75	5	2.16	3.50	8.8	SWZ
GC03-0438	265	270	5	2.31	0.57	13.9	220
GC03-0438	270	275	5	5.11	0.77	30.7	372
GC03-0438	275	280	5	2.76	0.51	18.1	372
GC03-0439	190	195	5	2.52	5.19	39.0	210
GC03-0439	225	230	5	2.38	4.66	11.5	350
GC03-0439	230	235	5	3.84	3.68	22.8	210
GC03-0441	60	65	5	2.43	2.19	14.4	362
GC03-0441	115	120	5	3.49	2.10	16.7	362
GC03-0441	130	135	5	3.62	1.39	17.6	362
GC03-0441	135	140	5	3.87	1.64	16.5	362

Hole ID	From	To	Length	%Cu	Au g/t	Ag g/t	Mineral Zone
GC03-0441	435	440	5	2.01	0.44	17.0	230
GC03-0441	450	455	5	2.07	0.24	13.5	230
GC03-0445	30	35	5	2.81	0.32	22.6	340
GC03-0445	105	110	5	2.51	0.41	12.6	230
GC03-0445	110	115	5	3.70	0.38	28.5	230
GC03-0445	115	120	5	3.30	0.42	26.2	230
GC03-0445	120	125	5	6.30	0.60	29.9	380
GC03-0445	125	130	5	4.04	0.58	23.2	230
GC03-0445	130	135	5	3.89	0.41	29.4	230
GC04-0450	50	55	5	2.06	0.18	19.0	230
GC04-0450	55	60	5	3.07	0.31	26.2	230
GC04-0450	60	65	5	2.43	0.20	17.5	230
GC04-0450	115	120	5	2.70	0.28	22.6	340
GC04-0452	60	65	5	2.85	0.82	6.9	362
GC04-0453	300	305	5	4.73	2.70	31.8	350
GC04-0453	305	310	5	3.39	2.85	17.9	350
GC04-0455	225	230	5	2.06	0.80	19.7	350
GC04-0465	70	75	5	2.75	0.89	21.3	NJ
GC04-0465	75	80	5	2.03	0.70	16.0	NJ
GC04-0465	80	85	5	2.28	0.54	22.0	NJ
GC04-0465	110	115	5	2.20	0.43	16.2	NJ
GC04-0465	115	120	5	2.24	0.81	13.0	NJ
GC04-0465	155	160	5	2.64	0.86	31.6	NJ
GC04-0465	160	165	5	3.28	1.27	28.6	NJ
GC04-0465	175	180	5	4.35	1.03	34.2	NJ
GC04-0465	180	185	5	5.32	1.76	49.7	NJ
GC04-0465	185	190	5	2.78	3.74	32.3	NJ
GC04-0469	180	185	5	2.17	1.89	24.8	WF
GC04-0475	200	205	5	2.20	5.58	17.7	220
GC04-0476	260	265	5	2.04	1.49	18.4	WF
GC04-0479	25	30	2.57	33.01	5.85	193.6	OP
GC04-0479	30	35	5	15.90	0.91	71.2	OP
GC04-0479	35	40	5	5.30	0.14	1.9	OP
GC04-0480	25	30	3.61	4.61	3.40	1.2	OP
GC04-0480	30	35	5	4.13	2.98	7.3	OP
GC04-0480	35	40	5	14.01	2.02	85.2	OP
GC04-0480	40	45	5	23.66	1.44	148.9	OP
GC04-0480	45	50	5	18.26	1.93	116.1	OP
GC04-0480	50	55	5	23.34	1.42	160.4	OP
GC04-0480	55	60	5	12.87	1.53	63.6	OP
GC04-0480	255	260	5	2.24	0.88	16.1	WF
GC04-0483	75	80	5	2.20	1.68	7.1	OP
GC04-0484	30	35	5	2.44	0.37	23.1	340
GC04-0484	65	70	5	2.12	0.24	16.3	340
GC04-0488	60	65	5	2.02	0.22	15.0	340
GC04-0488	75	80	4.25	2.28	0.27	19.7	380

Hole ID	From	To	Length	%Cu	Au g/t	Ag g/t	Mineral Zone
GC04-0488	80	85	2.7	2.47	0.21	24.9	380
GC04-0488	165	170	5	3.70	0.49	30.0	220
GC04-0488	170	175	5	2.82	0.58	25.9	220
GC04-0496	255	260	5	2.06	0.46	14.4	WF
GC04-0498	130	135	5	2.81	2.16	15.1	OP
GC04-0501	480	485	5	2.04	6.06	17.2	210
GC04-0502	245	250	5	2.07	0.52	7.6	SWZ
GC04-0502	280	285	5	2.41	1.64	11.6	SWZ
GC04-0503	5	10	1.68	4.53	1.53	20.6	900
GC04-0503	10	15	5	3.04	1.14	14.2	230
GC04-0503	15	20	5	2.12	0.72	11.1	230
GC04-0503	20	25	5	2.24	0.60	12.8	230
GC04-0503	30	35	5	2.40	1.21	17.7	230
GC04-0503	35	40	5	2.40	1.70	18.5	230
GC04-0503	40	45	5	2.54	1.67	17.1	230
GC04-0503	45	50	5	2.16	1.37	15.2	230
GC04-0503	55	60	5	2.10	0.48	14.4	230
GC04-0508	35	40	5	2.91	0.93	5.9	OP
GC04-0508	40	45	4.81	3.35	1.07	6.8	OP
GC05-0512	145	150	5	2.57	4.49	9.2	220
GC05-0513	120	125	5	2.13	7.35	11.6	MC
GC05-0513	160	165	5	2.28	6.49	19.6	MC
GC05-0513	165	170	5	2.28	5.34	18.5	MC
GC05-0514	225	230	5	2.78	6.74	17.6	220
GC05-0514	240	245	5	2.72	6.70	20.5	372
GC05-0514	250	255	5	3.60	8.33	34.3	220
GC05-0514	255	260	5	4.97	8.86	43.4	220
GC05-0514	260	265	5	3.60	6.44	28.3	210
GC05-0514	360	365	5	2.88	1.54	13.4	210
GC05-0514	365	370	5	3.30	2.31	18.3	210
GC05-0514	375	380	5	2.11	0.57	10.5	210
GC05-0521	325	330	5	3.74	1.93	57.5	NJ
GC05-0521	350	355	5	2.52	1.65	24.0	NJ
GC05-0521	355	360	5	2.43	2.59	7.8	NJ
GC05-0521	360	365	5	2.37	2.98	7.1	NJ
GC05-0525	275	280	5	2.03	2.69	15.4	220
GC05-0525	280	285	5	2.53	2.91	14.4	220
GC05-0533	20	25	5	6.04	1.81	10.8	OP
GC05-0533	30	35	5	2.17	2.72	0.6	OP
GC05-0533	35	40	5	8.72	14.57	2.3	OP
GC05-0533	40	45	5	6.60	4.05	20.7	OP
GC05-0533	45	50	5	6.98	3.29	27.0	OP
GC05-0533	50	55	5	2.07	0.26	0.9	OP
GC05-0537	45	50	5	2.12	1.88	7.3	220
GC05-0537	195	200	5	2.18	1.63	13.9	210
GC05-0537	200	205	5	2.31	1.49	15.1	210

Hole ID	From	To	Length	%Cu	Au g/t	Ag g/t	Mineral Zone
GC05-0546	310	315	5	2.31	0.39	33.5	WF
GC05-0555	130	135	1.2	2.56	0.75	14.2	NJ
GC05-0555	135	140	4.9	2.56	0.75	14.2	NJ
GC05-0555	160	165	0.72	2.77	2.20	228.0	NJ
GC05-0555	165	170	5	2.77	2.20	228.0	NJ
GC05-0555	170	175	0.38	2.77	2.20	228.0	NJ
GC05-0555	190	195	5	4.34	0.56	15.7	NJ
GC05-0555	195	200	5	6.77	2.16	42.7	NJ
GC05-0555	200	205	5	2.20	0.61	20.5	NJ
GC05-0555	315	320	5	2.88	0.99	35.1	NJ
GC05-0555	320	325	5	2.73	1.58	27.9	NJ
GC05-0555	325	330	5	2.17	0.76	16.1	NJ
GC05-0558	275	280	5	2.29	2.08	10.6	220
GC05-0558	280	285	5	2.36	1.86	10.3	220
GC05-0562	110	115	5	2.01	1.38	9.4	220
GC05-0562	230	235	5	2.13	0.80	15.3	210
GC05-0562	235	240	5	2.17	0.96	15.1	372
GC05-0562	240	245	5	2.39	0.81	14.2	210
GC05-0562	250	255	5	2.19	1.13	21.5	210
GC05-0562	260	265	5	2.65	1.06	25.9	210
GC05-0562	265	270	5	2.21	1.11	20.6	210
GC05-0562	270	275	5	2.08	0.85	20.7	210
GC05-0567	160	165	5	4.44	2.37	55.5	NJ
GC05-0567	165	170	5	7.65	2.75	115.7	NJ
GC05-0567	170	175	5	4.42	1.97	35.1	NJ
GC05-0567	175	180	5	2.74	1.44	21.6	NJ
GC05-0567	190	195	5	2.26	0.49	5.9	NJ
GC05-0567	200	205	5	2.63	0.56	11.6	NJ
GC05-0567	205	210	5	2.19	0.57	13.6	NJ
GC05-0567	250	255	5	2.66	0.49	22.5	NJ
GC05-0567	255	260	5	2.31	0.75	31.0	NJ
GC05-0568	40	45	2.64	2.59	0.47	26.2	230
GC05-0568	115	120	5	2.18	0.54	27.4	230
GC05-0568	120	125	5	2.64	0.80	27.5	230
GC05-0575	45	50	4.15	2.46	3.25	24.5	210
GC05-0580	60	65	5	3.61	4.50	23.5	MC
GC05-0580	65	70	5	2.17	4.81	13.5	MC
GC05-0581	45	50	5	2.89	1.34	9.8	362
GC05-0581	85	90	5	2.32	2.19	9.4	230
GC05-0581	90	95	5	3.16	1.89	12.3	372
GC05-0581	95	100	5	4.02	3.00	15.9	372
GC05-0581	100	105	5	3.66	2.28	21.3	230
GC05-0581	105	110	5	2.86	1.67	14.7	230
GC05-0581	110	115	5	3.37	1.99	20.0	362
GC05-0581	115	120	5	3.52	1.05	16.4	362
GC05-0581	120	125	5	3.63	0.70	27.2	362

Hole ID	From	To	Length	%Cu	Au g/t	Ag g/t	Mineral Zone
GC05-0581	130	135	5	2.11	0.19	8.8	362
GC05-0581	135	140	1.85	2.74	0.19	12.0	230
GC05-0581	140	145	2.05	2.39	0.35	15.7	230
GC05-0581	145	150	5	3.00	0.77	24.3	230
GC05-0581	150	155	5	2.78	1.31	25.1	230
GC05-0581	155	160	5	2.98	0.90	17.1	230
GC05-0581	160	165	5	2.94	1.04	15.1	230
GC05-0581	165	170	5	2.45	0.66	18.5	230
GC05-0581	185	190	5	2.25	0.61	19.0	230
GC05-0581	190	195	5	2.16	0.76	20.2	230
GC05-0585	215	220	5	2.36	0.29	8.4	WF
GC05-0586	90	95	5	3.08	6.62	16.7	220
GC05-0586	110	115	5	2.27	3.33	8.4	220
GC05-0595	135	140	5	2.28	3.54	7.6	210
GC05-0596	100	105	5	2.70	4.56	12.8	220
GC05-0596	105	110	5	3.68	3.50	13.9	220
GC05-0596	110	115	5	3.00	4.08	13.2	220
GC05-0596	115	120	5	2.07	4.64	11.4	220
GC05-0597	60	65	5	2.01	1.42	14.9	WF
GC05-0597	65	70	5	3.14	2.20	30.2	WF
GC05-0599	55	60	5	2.15	0.40	18.8	230
GC05-0599	60	65	5	2.22	0.60	17.9	230
GC05-0599	65	70	5	2.10	0.64	16.0	230
GC05-0599	75	80	5	2.29	0.86	11.0	230
GC05-0599	115	120	5	2.76	0.68	12.4	230
GC05-0619	210	215	5	2.03	0.65	5.1	220
GC05-0621	15	20	5	2.34	1.48	6.1	SWZ
GC05-0621	20	25	5	2.26	1.31	4.2	SWZ
GC05-0625	80	85	5	2.24	1.20	18.2	210
GC05-0625	120	125	5	2.30	1.55	19.6	210
GC05-0625	160	165	5	2.03	1.20	11.7	210
GC05-0625	165	170	5	2.13	1.24	16.5	210
GC05-0625	170	175	5	2.40	1.35	16.6	210
GC05-0625	175	180	5	2.48	1.78	15.1	210
GC05-0625	180	185	5	2.36	1.57	15.8	210
GC05-0628	160	165	5	2.29	6.30	10.6	SWZ
GC05-0634	180	185	5	2.26	0.78	20.4	220
GC05-0637	285	290	5	2.04	0.73	17.1	340
GC05-0637	290	295	5	2.99	2.49	22.5	340
GC05-0637	295	300	5	2.40	2.34	11.7	340
GC05-0647	260	265	5	2.09	1.20	6.2	SWZ
GC05-0647	265	270	5	2.32	1.46	11.6	SWZ
GC05-0647	295	300	5	2.23	2.33	15.6	SWZ
GC05-0647	300	305	5	2.42	3.36	18.5	SWZ
GC05-0655	20	25	3.97	4.15	1.99	12.2	SWZ
GC05-0662	35	40	0.84	2.70	0.38	28.3	900

Hole ID	From	To	Length	%Cu	Au g/t	Ag g/t	Mineral Zone
GC05-0665	105	110	5	3.15	3.13	28.9	SWZ
GC05-0665	110	115	5	2.97	2.22	11.6	SWZ
GC05-0665	115	120	5	2.57	1.64	10.6	SWZ
GC05-0665	120	125	5	2.02	1.91	10.5	SWZ
GC05-0665	125	130	5	2.75	1.55	13.2	SWZ
GC05-0665	130	135	5	2.65	2.15	19.7	SWZ
GC05-0669	70	75	5	2.03	0.33	10.1	230
GC05-0669	75	80	5	3.15	0.36	14.0	230
GC05-0669	80	85	5	3.03	0.45	15.2	230
GC05-0685	330	335	5	5.28	3.13	55.9	WF
GC05-0685	335	340	5	4.05	3.52	127.2	WF
GC05-0698	305	310	5	2.41	1.07	16.1	210
GC05-0698	320	325	5	2.09	1.16	12.7	210
GC05-0698	335	340	5	3.21	1.81	18.2	210
GC05-0698	340	345	5	2.43	1.07	13.8	210
GC05-0701	130	135	5	2.14	0.31	17.6	220
GC05-0710	15	20	4.9	2.79	3.25	11.2	230
GC05-0710	20	25	5	2.43	2.88	13.9	230
GC05-0710	25	30	5	2.12	2.36	12.7	230
GC05-0710	50	55	5	2.45	0.79	12.6	230
GC05-0710	55	60	5	2.23	0.44	10.7	230
GC05-0710	60	65	5	3.31	0.61	13.0	230
GC05-0710	65	70	5	2.27	0.26	9.4	230
GC05-0710	75	80	5	2.38	0.47	10.8	230
GC05-0710	80	85	5	2.81	0.47	12.9	230
GC05-0710	85	90	5	2.56	0.29	14.5	230
GC05-0710	90	95	5	2.26	0.38	16.2	230
GC05-0711	20	25	5	3.03	0.53	18.9	230
GC05-0711	25	30	5	2.55	1.10	29.5	230
GC05-0711	40	45	3	2.06	1.14	11.4	230
GC05-0711	45	50	5	2.20	1.50	11.7	230
GC05-0711	50	55	1	2.20	1.50	11.7	230
GC05-0711	75	80	5	2.18	0.38	10.1	230
GC05-0711	80	85	5	2.48	0.41	13.8	230
GC05-0711	95	100	5	2.29	0.29	16.9	230
GC61-0004	30	35	5	2.29	2.19	0.0	SWZ
GC61-0004	35	40	5	2.59	3.20	2.1	SWZ
GC62-0007	120	125	5	2.34	1.28	28.1	230
GC62-0007	135	140	5	2.35	0.17	13.2	230
GC62-0013	40	45	5	3.81	0.14	23.5	230
GC62-0013	45	50	5	2.89	0.11	13.1	230
GC62-0018	110	115	5	2.32	1.27	11.2	362
GC62-0026	40	45	2.06	2.27	0.03	13.7	220
GC62-0033	145	150	5	3.66	1.03	15.8	NJ
GC62-0033	170	175	5	2.17	0.28	9.7	NJ
GC62-0033	175	180	5	2.43	0.62	15.6	NJ

Hole ID	From	To	Length	%Cu	Au g/t	Ag g/t	Mineral Zone
GC62-0033	180	185	5	2.37	0.65	22.1	NJ
GC62-0033	185	190	5	2.86	0.63	18.4	NJ
GC62-0033	190	191.11	1.11	3.63	0.62	15.6	NJ
GC62-0036	95	100	1.52	2.64	0.07	1.4	350
GC63-0046	290	295	5	2.11	0.34	3.4	220
GC63-0048	95	100	5	2.18	0.31	19.8	230
GC63-0048	100	105	5	2.06	0.31	19.8	230
GC63-0048	110	115	5	2.14	0.14	17.4	230
GC63-0048	125	130	5	2.60	0.24	10.7	230
GC63-0048	130	135	5	2.36	0.24	10.7	230
GC63-0048	155	160	5	2.16	0.21	8.4	230
GC63-0048	190	195	2.37	2.39	0.17	12.4	230
GC63-0049	80	85	5	2.11	0.34	13.2	230
GC63-0049	85	90	5	3.28	0.34	13.2	230
GC63-0049	90	95	5	4.46	0.34	13.4	230
GC63-0049	95	100	5	3.44	0.31	15.1	230
GC63-0049	100	105	5	3.08	0.31	15.1	230
GC63-0049	105	110	5	2.83	0.29	15.1	230
GC63-0053	95	100	5	2.44	0.21	13.7	JUNC
GC63-0053	100	105	5	2.21	0.21	13.7	JUNC
GC63-0060	90	95	5	3.05	0.16	14.9	340
GC63-0060	95	100	2.54	3.92	0.27	9.2	230
GC63-0060	120	125	5	2.10	0.34	11.0	230
GC63-0060	125	130	5	3.39	0.34	11.0	350
GC63-0060	290	295	5	2.08	0.21	12.6	220
GC63-0063	155	160	5	2.14	0.69	10.3	372
GC63-0063	170	175	5	2.81	1.37	11.0	372
GC63-0063	185	190	5	2.64	1.32	11.0	372
GC63-0071	10	15	5	2.37	0.23	13.7	NJ
GC63-0071	15	20	3.29	2.50	0.28	13.7	NJ
GC63-0071	20	25	4.27	5.14	1.33	37.6	NJ
GC63-0071	25	30	3.65	8.05	1.51	41.6	NJ
GC63-0074	60	65	5	2.42	0.23	13.2	230
GC63-0074	65	70	5	2.17	0.38	12.6	230
GC63-0074	70	75	5	2.67	0.38	12.6	230
GC63-0074	75	80	5	3.31	1.73	13.8	230
GC63-0074	80	85	5	2.94	1.52	14.1	230
GC63-0077	80	85	5	2.00	0.27	13.5	230
GC63-0077	90	95	5	2.09	0.21	13.0	230
GC63-0077	220	225	5	2.00	0.45	16.0	350
GC63-0080	5	10	5	2.35	0.23	11.1	NJ
GC63-0080	10	15	5	4.81	0.10	7.9	NJ
GC63-0080	15	20	5	5.32	0.10	7.9	NJ
GC63-0080	20	25	5	3.43	0.34	7.6	NJ
GC63-0084	205	210	1.96	2.17	3.75	8.6	NJ
GC63-0084	225	230	3.94	2.62	0.86	30.9	NJ

Hole ID	From	To	Length	%Cu	Au g/t	Ag g/t	Mineral Zone
GC63-0084	230	235	5	6.59	0.86	30.9	NJ
GC63-0084	235	240	5	3.31	0.86	30.9	NJ
GC63-0084	240	245	5	3.60	0.86	30.9	NJ
GC63-0084	245	250	5	2.16	0.63	30.9	NJ
GC63-0084	260	265	5	2.00	0.77	30.9	NJ
GC63-0084	265	270	5	2.08	0.78	30.9	NJ
GC63-0084	310	315	5	2.18	0.72	30.9	NJ
GC63-0085	50	55	5	2.00	0.29	-1.0	NJ
GC63-0085	55	60	5	3.37	1.15	-1.0	NJ
GC63-0085	70	75	5	2.68	0.42	-1.0	NJ
GC63-0085	75	80	5	2.80	0.32	-1.0	NJ
GC63-0085	80	85	5	2.10	0.55	-1.0	NJ
GC63-0085	90	95	5	2.60	0.37	-1.0	NJ
GC63-0086	45	50	5	2.27	0.34	9.3	230
GC63-0089	305	310	5	2.82	0.98	14.3	220
GC63-0089	325	330	5	2.11	0.69	16.2	220
GC63-0089	475	480	5	2.39	0.39	12.4	372
GC64-0097	55	60	5	2.10	0.22	9.6	230
GC64-0097	60	65	5	2.12	0.20	9.6	230
GC64-0097	65	70	5	3.39	0.64	11.6	230
GC64-0097	70	75	5	2.81	0.63	13.0	230
GC64-0097	105	110	5	2.57	0.74	14.1	230
GC64-0097	110	115	5	2.07	0.57	12.0	230
GC64-0097	180	185	5	2.07	0.52	9.7	340
GC64-0104	35	40	4.95	4.14	4.61	17.4	230
GC64-0104	40	45	5	3.58	3.45	17.4	230
GC64-0104	45	50	5	4.85	3.30	17.4	230
GC64-0104	50	55	5	3.98	2.29	17.3	230
GC64-0107	315	320	5	2.21	0.58	11.3	230
GC64-0110	15	20	4.46	3.02	7.50	7.9	SWZ
GC64-0118	20	25	5	2.10	0.71	13.5	220
GC64-0118	45	50	5	4.58	6.40	13.7	220
GC64-0118	50	55	5	4.89	5.10	12.7	220
GC64-0118	150	155	5	2.36	0.68	10.7	220
GC64-0118	165	170	5	2.23	0.56	8.9	220
GC64-0120	180	185	5	3.26	5.55	17.1	220
GC64-0120	190	195	5	3.36	7.10	20.0	220
GC64-0120	225	230	5	2.27	1.86	10.9	220
GC64-0120	280	285	5	4.30	5.20	32.5	220
GC64-0120	285	290	5	3.73	7.11	32.5	220
GC64-0120	290	295	5	2.57	6.94	27.2	220
GC64-0120	295	300	5	3.08	7.75	21.5	220
GC64-0120	300	305	5	3.73	10.45	21.5	210
GC64-0120	305	310	5	2.81	4.19	21.5	210
GC64-0120	315	320	5	2.35	4.26	14.5	210
GC64-0120	320	325	5	3.11	11.39	14.5	210

Hole ID	From	To	Length	%Cu	Au g/t	Ag g/t	Mineral Zone
GC64-0120	325	330	5	2.43	2.63	12.4	210
GC64-0124	225	230	1.1	3.05	0.34	3.1	NJ
GC64-0134	75	80	5	2.20	0.60	15.0	230
GC64-0134	85	90	5	2.29	0.29	15.1	230
GC64-0134	90	95	5	2.78	0.40	15.1	230
GC64-0134	95	100	5	2.95	0.56	15.1	230
GC64-0134	100	105	5	3.53	0.94	13.7	230
GC64-0134	105	110	5	2.55	0.37	13.5	230
GC64-0134	110	115	5	2.91	0.44	13.5	230
GC64-0138	120	125	5	2.00	0.36	10.5	372
GC64-0138	145	150	5	3.04	0.79	13.3	372
GC64-0138	155	160	5	3.53	0.39	17.2	220
GC64-0138	160	165	5	2.47	0.28	17.6	220
GC64-0138	165	170	5	2.84	0.34	17.6	220
GC64-0138	175	180	5	2.24	0.33	10.0	220
GC64-0138	180	185	5	2.46	0.36	10.0	220
GC65-0142	145	150	5	2.51	1.16	11.6	230
GC65-0142	150	155	5	2.51	0.39	14.6	230
GC65-0142	155	160	5	2.13	1.19	17.4	230
GC65-0142	160	165	5	2.02	2.04	17.4	230
GC65-0142	165	170	5	2.69	1.48	15.9	230
GC65-0149	210	215	5	3.10	4.01	12.0	220
GC65-0149	215	220	5	2.19	3.11	12.0	220
GC65-0150	190	195	5	2.03	3.27	11.9	210
GC65-0150	200	205	5	2.14	3.57	12.5	210
GC65-0151	400	405	5	2.62	0.17	3.4	350
GC65-0154	25	30	5	2.43	4.95	10.0	362
GC65-0154	30	35	5	2.42	3.09	9.0	362
GC65-0156	20	25	1.96	3.32	0.64	10.3	900
GC65-0156	25	30	5	2.75	0.97	10.3	230
GC65-0156	115	120	5	2.20	0.70	19.8	230
GC65-0157	85	90	5	2.38	0.85	13.7	230
GC65-0157	110	115	5	2.17	0.28	13.0	220
GC65-0157	115	120	5	2.32	0.45	13.0	220
GC65-0158	85	90	5	2.24	0.30	11.9	230
GC65-0158	90	95	5	2.44	0.30	13.0	230
GC65-0161	180	185	5	2.28	0.38	24.7	220
GC65-0161	200	205	5	2.23	0.60	16.6	220
GC65-0163	175	180	5	2.09	0.34	10.9	372
GC65-0172	265	270	5	2.05	1.34	13.7	220
GC65-0173	225	230	5	2.13	0.42	10.3	372
GC65-0173	235	240	5	2.02	0.54	13.0	220
GC65-0173	240	245	5	2.62	1.21	13.0	220
GC65-0174	235	240	5	2.07	2.05	10.0	210
GC65-0177	175	180	5	2.23	1.31	8.3	210
GC65-0178	160	165	5	4.87	0.46	14.4	230

Hole ID	From	To	Length	%Cu	Au g/t	Ag g/t	Mineral Zone
GC65-0178	165	170	5	2.16	0.21	14.4	230
GC65-0178	170	175	5	3.14	0.31	14.4	230
GC65-0178	175	180	5	4.39	0.39	18.3	230
GC65-0178	180	185	5	2.90	0.36	20.4	230
GC65-0180	20	25	4.88	2.76	0.57	14.6	230
GC65-0180	25	30	5	2.73	0.59	14.6	230
GC65-0180	30	35	5	2.51	0.45	14.6	230
GC65-0180	45	50	5	2.46	0.62	19.8	230
GC65-0180	50	55	5	2.39	0.74	20.2	230
GC65-0180	55	60	5	3.48	1.45	20.4	230
GC65-0180	60	65	5	2.80	0.46	20.4	230
GC65-0180	65	70	5	2.30	0.34	12.9	230
GC65-0180	80	85	5	2.23	0.37	6.6	230
GC65-0181	75	80	5	2.60	0.79	19.4	372
GC65-0181	80	85	5	2.14	0.35	16.8	372
GC65-0181	85	90	5	2.56	0.63	16.8	372
GC65-0181	90	95	1.95	2.61	0.55	15.9	372
GC65-0181	95	100	5	2.52	0.39	8.4	372
GC65-0182	105	110	5	2.34	0.90	7.5	340
GC65-0182	140	145	5	2.33	1.36	10.5	230
GC65-0182	145	150	5	4.09	1.66	26.4	230
GC65-0182	150	155	5	3.91	1.51	32.0	230
GC65-0182	155	160	5	5.73	2.94	32.0	230
GC65-0182	160	165	5	5.68	2.70	27.3	230
GC65-0182	165	170	5	2.93	1.28	25.2	230
GC65-0182	170	175	5	2.51	0.57	25.2	230
GC65-0182	175	180	5	2.99	1.14	15.6	230
GC65-0182	180	185	5	2.23	0.67	10.3	230
GC65-0182	185	190	5	2.15	0.55	10.3	230
GC65-0182	190	195	5	2.52	0.49	11.2	230
GC65-0182	195	200	5	2.27	0.52	11.9	230
GC65-0182	200	205	5	3.24	0.99	11.9	230
GC65-0182	205	210	5	4.84	1.34	21.2	230
GC65-0182	210	215	5	5.26	0.80	28.8	230
GC65-0182	215	220	5	4.03	0.60	28.8	230
GC65-0182	220	225	5	3.29	0.55	24.9	230
GC65-0182	225	230	5	2.31	0.45	21.0	230
GC65-0182	230	235	5	2.01	0.36	21.0	230
GC65-0186	255	260	5	2.09	0.72	15.3	210
GC65-0186	270	275	5	2.70	0.96	10.0	210
GC65-0187	70	75	5	4.70	0.13	2.2	230
GC65-0187	75	80	5	2.61	0.21	2.1	230
GC65-0189A	195	200	5	2.08	4.64	15.7	220
GC65-0191	40	45	5	2.33	0.24	21.6	230
GC65-0191	55	60	5	3.90	0.59	21.0	340
GC65-0191	60	65	5	2.28	0.66	21.0	340

Hole ID	From	To	Length	%Cu	Au g/t	Ag g/t	Mineral Zone
GC65-0191	70	75	5	2.68	0.28	14.5	230
GC65-0193	95	100	2.16	2.59	1.51	16.2	372
GC65-0193	100	105	5	2.59	0.84	16.2	372
GC65-0193	105	110	5	3.31	1.00	16.2	372
GC65-0193	110	115	5	2.98	1.08	14.5	372
GC65-0195	0	5	1.95	2.12	0.17	11.4	900
GC65-0195	80	85	5	3.32	1.01	23.5	230
GC65-0195	90	95	5	2.15	0.16	14.7	230
GC65-0195	105	110	5	2.17	0.12	8.6	230
GC65-0196A	130	135	5	2.57	4.04	9.5	380
GC65-0196A	135	140	5	2.33	3.04	6.2	380
GC65-0197	180	185	5	2.16	0.83	15.1	220
GC65-0199	10	15	5	2.20	0.61	11.6	230
GC65-0204	160	165	5	2.69	1.55	10.8	220
GC65-0204	165	170	5	2.50	0.59	10.8	220
GC65-0206	55	60	5	2.62	0.22	9.1	230
GC65-0206	60	65	5	2.83	0.30	9.1	230
GC65-0206	65	70	5	2.03	0.31	8.7	230
GC65-0206	125	130	5	2.16	0.22	5.8	230
GC65-0210	40	45	5	2.59	0.51	14.0	350
GC65-0210	65	70	5	3.48	1.16	24.0	230
GC65-0210	70	75	5	3.01	0.31	24.0	230
GC65-0210	75	80	5	4.07	1.73	23.2	230
GC65-0210	80	85	5	3.93	1.44	18.6	230
GC65-0210	85	90	5	2.57	1.76	18.6	230
GC65-0210	90	95	5	3.08	0.89	17.3	230
GC65-0210	95	100	5	3.41	0.60	16.8	230
GC66-0220	135	140	5	2.26	0.58	16.2	220
GC66-0220	140	145	5	2.28	0.43	16.2	220
GC66-0220	160	165	5	2.01	0.54	11.8	220
GC66-0220	170	175	5	2.47	0.44	14.0	220
GC66-0223	10	15	5	2.24	0.49	17.6	230
GC66-0223	15	20	5	2.28	0.28	17.6	230
GC66-0223	95	100	5	2.25	0.47	13.3	230
GC66-0224	320	325	5	2.22	1.27	12.5	372
GC66-0225	95	100	5	2.18	1.95	10.1	220
GC66-0225	100	105	5	2.78	2.89	11.7	220
GC66-0225	115	120	5	2.19	1.73	8.5	220
GC66-0226	150	155	5	3.50	0.39	25.6	NJ
GC66-0226	155	160	5	6.19	0.49	25.6	NJ
GC66-0226	160	165	5	4.93	0.59	27.2	NJ
GC66-0226	165	170	5	6.45	1.54	45.6	NJ
GC66-0226	170	175	5	4.49	1.54	45.6	NJ
GC66-0226	175	180	5	6.08	1.84	45.3	NJ
GC66-0226	180	185	5	4.58	0.89	35.0	NJ
GC66-0226	250	255	5	2.04	0.42	12.2	NJ

Hole ID	From	To	Length	%Cu	Au g/t	Ag g/t	Mineral Zone
GC66-0226	255	260	5	2.19	0.53	12.2	NJ
GC66-0228	75	80	5	2.84	2.27	17.5	NJ
GC66-0228	80	85	5	2.98	2.43	17.5	NJ
GC66-0228	85	90	5	2.93	1.06	16.0	NJ
GC66-0228	90	95	5	2.66	0.86	12.8	NJ
GC66-0228	190	195	5	2.93	0.28	12.6	NJ
GC66-0228	200	205	5	2.21	0.38	12.8	NJ
GC66-0228	205	210	5	4.03	0.59	26.5	NJ
GC66-0228	210	215	5	5.31	0.73	37.8	NJ
GC66-0228	220	225	5	2.31	0.65	14.4	NJ
GC66-0229	50	55	5	2.34	1.09	9.7	NJ
GC66-0229	55	60	5	3.12	0.69	9.7	NJ
GC66-0229	60	65	5	3.03	1.10	11.9	NJ
GC66-0229	75	80	5	2.84	0.47	19.5	NJ
GC66-0230	95	100	5	2.29	0.64	12.2	230
GC66-0230	125	130	5	2.29	0.88	16.6	220
GC66-0233	135	140	5	2.97	2.53	14.6	NJ
GC66-0233	140	145	5	2.35	1.22	15.5	NJ
GC66-0233	165	170	5	2.02	0.94	9.7	NJ
GC66-0233	190	195	5	3.72	3.14	33.8	NJ
GC72-0236	80	85	5	2.94	1.06	24.7	340
GC72-0236	165	170	5	2.05	0.15	11.3	340
GC72-0238	25	30	5	2.20	0.19	26.7	340
GC72-0238	30	35	5	2.05	0.31	25.5	340
GC72-0238	35	40	5	4.16	0.48	22.6	230
GC72-0238	40	45	5	2.99	0.18	22.6	230
GC72-0238	45	50	5	2.72	0.21	21.3	230
GC72-0238	50	55	5	2.48	0.18	17.1	230
GC72-0242	40	45	5	2.02	0.93	7.9	340
GC72-0242	125	130	5	2.85	1.10	16.1	362
GC72-0242	130	135	5	2.08	0.70	18.7	372
GC72-0242	165	170	5	2.26	3.57	11.3	230
GC72-0242	180	185	5	6.43	3.38	49.4	230
GC72-0242	185	190	5	6.78	3.06	49.4	230
GC72-0242	190	195	5	5.31	2.20	36.9	230
GC72-0242	195	200	5	4.96	1.94	28.5	230
GC72-0242	200	205	5	3.28	1.41	24.9	230
GC72-0242	210	215	5	2.03	0.82	5.5	230
GC72-0242	230	235	5	3.28	0.51	17.3	230
GC72-0243	5	10	3.9	4.67	0.54	13.0	230
GC72-0243	10	15	3.41	3.10	0.40	13.0	230
GC72-0243	15	20	3.84	3.00	1.59	13.0	230
GC72-0244	75	80	1.5	2.20	0.51	15.1	230
GC72-0244	85	90	5	3.17	1.58	22.6	230
GC72-0244	90	95	5	3.17	1.44	22.6	230
GC72-0244	95	100	5	2.05	1.00	17.4	230

Hole ID	From	To	Length	%Cu	Au g/t	Ag g/t	Mineral Zone
GC72-0244	100	105	5	2.91	0.77	12.0	230
GC72-0244	105	110	5	3.35	1.02	12.0	230
GC72-0244	115	120	5	3.20	0.73	25.4	230
GC72-0244	120	125	5	2.64	0.73	25.4	230
GC72-0244	125	130	5	3.89	1.06	25.8	230
GC72-0244	130	135	5	3.55	0.69	26.4	230
GC72-0244	135	140	5	2.84	0.56	26.4	230
GC72-0244	145	150	5	3.02	0.29	17.1	230
GC72-0244	150	155	5	3.18	0.25	17.1	230
GC72-0244	155	160	5	3.31	0.23	17.3	230
GC72-0244	160	165	5	2.69	0.14	17.8	230
GC72-0244	165	170	5	2.12	0.21	17.8	230
GC72-0244	170	175	5	2.16	0.19	19.1	230
GC72-0244	175	180	5	2.53	0.20	23.0	230
GC72-0244	180	185	5	2.67	0.21	23.0	230
GC72-0245	115	120	4.94	2.10	0.91	8.2	340
GC72-0248	90	95	5	2.04	0.14	15.0	230
GC72-0248	110	115	5	2.14	0.13	13.0	230
GC72-0248	125	130	5	2.62	0.26	18.5	230
GC72-0249	100	105	3.02	2.47	0.03	7.2	230
GC72-0252	40	45	5	2.32	0.04	9.3	230
GC72-0252	45	50	5	2.75	0.11	9.3	230
GC72-0253	60	65	2.82	2.45	0.62	13.0	900
GC72-0253	70	75	5	2.53	0.26	14.4	230
GC72-0253	75	80	5	2.98	0.48	16.8	230
GC72-0254	40	45	5	2.47	1.87	11.5	230
GC72-0254	45	50	5	2.01	1.19	11.3	230
GC72-0254	50	55	5	2.20	1.53	11.3	230
GC72-0254	70	75	5	2.30	1.71	11.4	230
GC72-0254	75	80	5	3.63	1.82	15.1	230
GC72-0254	80	85	5	3.93	1.24	15.1	230
GC72-0257	30	35	2.08	2.15	0.42	15.1	900
GC72-0257	35	40	5	2.10	0.58	15.1	230
GC72-0257	40	45	5	3.05	0.88	15.1	230
GC72-0257	45	50	5	3.69	0.90	18.3	230
GC72-0257	50	55	5	3.98	1.89	18.9	230
GC72-0257	55	60	5	4.32	1.89	18.9	230
GC72-0257	60	65	5	4.63	0.69	16.6	230
GC72-0257	65	70	5	3.04	0.41	16.1	230
GC72-0259	45	50	1.23	2.60	0.17	9.9	900
GC72-0259	60	65	5	2.94	0.21	13.5	350
GC72-0259	65	70	5	4.00	0.40	28.1	350
GC72-0259	70	75	5	3.42	0.64	28.1	230
GC72-0259	75	80	5	2.63	0.40	22.9	230
GC72-0259	80	85	5	2.48	0.31	21.3	230
GC72-0259	85	90	5	2.16	0.26	21.3	230

Hole ID	From	To	Length	%Cu	Au g/t	Ag g/t	Mineral Zone
GC72-0259	90	95	5	2.16	0.23	19.3	230
GC72-0259	95	100	5	2.33	0.32	18.5	230
GC72-0259	100	105	5	2.47	0.30	18.5	230
GC72-0260	50	55	2.88	2.07	0.40	11.3	350
GC72-0260	55	60	5	3.64	0.63	11.3	350
GC72-0260	200	205	5	2.42	0.21	18.9	350
GC72-0263	70	75	5	2.97	2.39	16.8	230
GC72-0263	75	80	5	2.76	1.23	16.8	230
GC72-0263	80	85	5	2.86	2.02	16.4	230
GC72-0264	50	55	3.49	2.45	0.41	18.9	350
GC72-0264	55	60	5	2.34	0.35	11.3	350
GC72-0264	95	100	0.33	2.45	1.08	26.1	350
GC72-0264	100	105	5	2.51	1.07	26.1	230
GC72-0266	55	60	0.87	3.30	0.07	23.7	900
GC72-0266	60	65	4.01	2.27	0.33	23.7	230
GC72-0269	85	90	5	2.26	0.19	18.9	230
GC72-0271	135	140	2.56	2.44	0.27	11.3	NJ
GC72-0272	40	45	3.85	2.80	6.98	10.3	230
GC72-0272	45	50	0.72	2.80	6.98	10.3	230
GC72-0273	10	15	0.98	2.25	1.14	14.1	900
GC72-0273	15	20	5	2.13	0.94	14.1	230
GC72-0273	20	25	5	2.10	0.76	13.8	230
GC72-0273	25	30	5	2.02	0.73	11.7	230
GC72-0273	65	70	5	2.07	0.25	20.6	230
GC72-0277	30	35	2.69	2.70	-1.00	-1.0	230
GC72-0277	35	40	0.36	2.70	-1.00	-1.0	230
GC72-0277	90	95	5	3.21	1.26	15.1	230
GC72-0277	195	200	5	2.03	0.18	22.3	220
GC72-0280	120	125	5	2.04	0.50	15.7	380
GC72-0280	175	180	5	2.31	0.39	12.5	220
GC72-0280	220	225	5	2.33	0.63	11.3	220
GC72-0282	25	30	4.09	2.14	0.36	12.0	230
GC72-0282	65	70	5	2.61	0.57	14.4	230
GC72-0283	50	55	5	2.56	0.65	18.5	380
GC72-0283	55	60	5	2.18	0.81	15.6	380
GC72-0283	90	95	5	2.00	0.48	11.7	350
GC72-0283	235	240	5	2.25	1.06	12.0	220
GC72-0284	220	225	5	2.43	1.26	-1.0	220
GC72-0284	225	230	5	2.20	1.13	-1.0	220
GC73-0289	100	105	5	2.07	0.15	28.1	230
GC73-0290	195	200	5	2.07	5.85	-1.0	220
GC73-0290	200	205	5	2.27	4.60	-1.0	220
GC73-0292	290	295	5	2.17	0.70	18.9	210
GC73-0293	40	45	5	2.42	0.07	14.8	230
GC73-0293	45	50	5	2.30	0.16	28.8	230
GC73-0307	175	180	5	2.23	0.77	9.3	220

Hole ID	From	To	Length	%Cu	Au g/t	Ag g/t	Mineral Zone
GC73-0309	150	155	5	2.23	0.14	13.0	230
GC73-0309	155	160	5	2.82	0.20	13.0	230
GC73-0313	225	230	5	2.37	0.79	-1.0	NJ
GC73-0313	235	240	5	2.85	0.60	-1.0	NJ
GC73-0313	240	245	5	3.34	1.09	-1.0	NJ
GC73-0313	245	250	5	3.41	2.17	-1.0	NJ
GC73-0313	250	255	5	3.32	1.31	-1.0	NJ
GC73-0313	255	260	5	2.21	0.82	-1.0	NJ
GC73-0317	150	155	5	2.09	-1.00	-1.0	NJ
GC73-0322	140	145	5	3.17	-1.00	-1.0	NJ
GC73-0322	170	175	1.26	4.15	-1.00	-1.0	NJ
GC73-0322	175	180	5	3.46	-1.00	-1.0	NJ
GC73-0322	180	185	5	4.32	-1.00	-1.0	NJ
GC73-0322	185	190	5	3.06	-1.00	-1.0	NJ
GC73-0322	190	195	5	3.77	-1.00	-1.0	NJ
GC73-0322	195	200	5	3.59	-1.00	-1.0	NJ
GC73-0322	200	205	5	2.88	-1.00	-1.0	NJ
GC73-0322	215	220	5	2.73	-1.00	-1.0	NJ
GC73-0322	235	240	1.22	2.00	-1.00	-1.0	NJ
GC73-0328	65	70	5	2.48	0.30	12.3	230
GC73-0332	45	50	5	3.37	20.22	-1.0	372
GC73-0332	50	55	5	2.49	17.52	11.3	372
GC73-0332	55	60	5	2.20	4.80	11.3	372
GC73-0333	140	145	5	2.37	1.01	-1.0	NJ
GC76-0346	245	250	5	2.22	0.52	11.0	220
GC76-0347A	220	225	5	2.02	1.07	11.0	210
GC76-0347A	235	240	5	2.33	1.15	14.1	210
GC76-0350	185	190	5	2.20	1.07	14.7	210
GC76-0350	190	195	5	2.50	1.15	14.4	210
GC76-0350	195	200	5	2.64	1.38	14.4	210
GC76-0353	70	75	5	2.72	7.36	16.5	210
GC76-0353	75	80	5	2.80	5.00	16.5	210
GC76-0355	195	200	5	2.32	1.16	10.3	210
GC76-0357	250	255	5	2.90	0.56	12.3	220
GC76-0365	295	300	5	2.12	0.31	5.5	210
GC90-0379	25	30	5	2.41	4.36	18.7	SWZ
GC90-0379	30	35	5	2.78	3.88	18.9	SWZ
GC90-0379	65	70	5	2.47	12.01	13.0	SWZ
GC90-0379	70	75	5	2.81	10.25	12.5	SWZ
GC90-0381	20	25	0.92	2.55	2.61	6.2	SWZ
GC90-0381	75	80	5	2.08	5.26	11.2	SWZ
GC90-0381	95	100	5	3.34	10.85	20.9	SWZ
GC90-0381	100	105	5	2.11	7.73	24.1	SWZ
GC90-0382	80	85	5	3.62	3.53	25.4	SWZ
GC90-0382	85	90	5	4.09	4.21	27.3	SWZ
GC90-0382	90	95	5	2.69	9.90	15.5	SWZ

Hole ID	From	To	Length	%Cu	Au g/t	Ag g/t	Mineral Zone
GC90-0382	170	175	5	2.24	2.85	19.8	SWZ
GC90-0382	175	180	5	2.02	2.70	21.4	SWZ
GC90-0383	135	140	5	2.07	1.79	8.1	SWZ
GC90-0383	150	155	5	2.10	2.05	9.0	SWZ
GC90-0386	40	45	5	2.67	4.03	16.8	SWZ
GC91-0395	200	205	5	2.23	0.75	14.5	220
GC91-0395	210	215	5	2.47	1.34	24.6	220
GC91-0395	215	220	5	2.13	1.21	20.2	220
GC91-0395	280	285	5	2.40	1.00	25.1	220
GC91-0419	125	130	5	2.63	7.10	15.9	MC
GC91-0419	130	135	5	2.07	2.72	16.0	MC
GC91-0431	35	40	5	3.07	0.54	12.3	230
GC91-0431	40	45	5	2.89	0.80	9.5	230
GC91-0431	45	50	5	2.14	0.42	11.2	230
GC91-0431	50	55	5	4.04	0.36	20.9	230
GC91-0431	55	60	5	3.00	0.19	16.3	230
GC91-0431	60	65	5	2.50	0.22	15.2	230
GC91-0431	275	280	5	2.25	0.09	4.3	230
UG0256W-N	0	5	5	3.08	-1.00	-1.0	230
UG0256W-N	5	10	5	2.29	-1.00	-1.0	230
UG0256W-N	10	15	5	3.30	-1.00	-1.0	230
UG0256W-N	15	20	5	2.11	-1.00	-1.0	230
UG0256W-N	20	25	5	2.03	-1.00	-1.0	230
UG0256W-N	60	65	0.62	3.50	-1.00	-1.0	500
UG0256W-N	65	69.91	4.91	2.45	-1.00	-1.0	500
UG0256W-S	35	40	5	2.05	-1.00	-1.0	500
UG0256W-S	50	55	5	3.38	-1.00	-1.0	230
UG0256W-S	55	60	5	2.77	-1.00	-1.0	230
UG0256W-S	60	63.53	3.53	4.71	-1.00	-1.0	230
UG0257E-N	15	20	5	2.13	-1.00	-1.0	230
UG0257E-N	20	25	5	2.23	-1.00	-1.0	230
UG0257E-N	25	30	5	2.38	-1.00	-1.0	230
UG0257E-S	10	15	5	2.39	-1.00	-1.0	230
UG0257E-S	15	20	5	2.19	-1.00	-1.0	230
UG9885S-W	115	120	5	3.94	-1.00	-1.0	230
UG9885S-W	120	125	4.13	2.90	-1.00	-1.0	230

15. Sample Preparation, Analysis and Security

Section 15 was prepared by Mr. Michael J. Lechner,RPG. Resource Modeling Incorporated

The Galore Creek project has seen many different sampling campaigns. The first drilling was completed in 1961 and very little is known about the sample preparation, analysis or security from the first three years of drilling.

15.1 Pre-2003 Samples

This section has been adapted from Simpson (2003).

Sample preparation has gone through several transitions since the early drilling in the 1960's. Prior to 1964, drill core was split in 3-metre lengths then half of the core was shipped to Coast Edridge laboratory in Vancouver for copper assay. Apparently some 30- metre-long composites were assayed for gold during this period.

In 1964, a small assay laboratory was constructed on site and during the first season of operation, processed 3,747 samples. Half of the split core was crushed on site to ¼ inch then a 340 gram split was separated using a Jones riffle splitter. At the lab the sample was split and crushed to –10 mesh then 95% of the crushed material was pulverized to -100 mesh and assayed for copper using a double digestion with titration and colorimetric determinations. Intervals that reached or exceeded a minimum copper grade of 0.4% over intervals of 12 to 18 metres were composited and shipped to Coast Edridge for gold and silver assaying. It is not known if these pulps were re-homogenized before compositing. Within distinctly anomalous gold intervals, there are gaps in the original sampling. Early gold assays performed on 30.5 metre composites contradict strongly with later 12-18 metre long composites assayed. The later composites were assayed by a commercial lab then checked by the Kennecott Research Centre Lab (located in Salt Lake City, Utah) and are considered to be reasonably accurate. The author does not know whether the Kennecott lab was certified.

During 1964, cross checking of Galore Creek laboratory copper analyses was carried out on a routine basis by Kennco Explorations laboratory in North Vancouver and at Coast Eldridge Laboratory. The author is unaware of where the now defunct Coast Eldridge Lab was located or if it was certified. Several samples were also checked at Hawley and Hawley assayers of Tucson and by Bear Creek Laboratory in Denver.

In 1966, composites from the Central Zone were reportedly re-assayed for gold and silver after discrepancies were found between the values obtained in 1964 and 1965. The re-assay was carried out at the Kennco Laboratory in North Vancouver. The results showed marginal increases in gold and silver content. Assay certificates could not be located for either the original or re-assayed material.

In 1967, the pulps from 140 samples were split and a portion analyzed by five separate laboratories; Coast Edridge, Sudbury and three of Kennecott's labs. A standard was included with the samples in order to check the reproducibility of the method being used. Comparison of the standards showed that the Coast Eldridge laboratory (using the titration method) was the least reliable. The field lab assays compared well with atomic absorption analyses at other labs. The author is unaware of what lab in

Sudbury, Ontario was used. Similarly the author is unaware of the location of the three Kennecott labs or if they were certified.

In the 1970's programs, the split half of the core samples were still crushed on site to ½ inch and split to obtain a ¾ lb sample. This was further crushed in a cone crusher then placed in Kraft paper bags and shipped by air in locked metal boxes to either the Kennco Exploration Lab in North Vancouver (1972/73), or Chemex Lab (1974) for assay. Assaying for Au and Ag was only performed on composite samples (up to 15m) which averaged over 0.4% Cu. No information on check assays or quality control from the 1970's drill programs could be located. All coarse rejects from the 1970's were stored on the property. The author is unaware of the location of the Chemex Lab that was used in 1974 or if the lab was certified.

During the 1990 drill program carried out by Mingold, half of the split core was crushed on site to ¼ inch (6.35 mm) and a 300-325 gram split was taken and shipped to Min-En Laboratory in Smithers for further processing and assaying. For gold analysis, a 30 gram sample split underwent fire assay pre-concentration with an A.A. finish. Samples in excess of 1000 ppb Au were re-assayed. If high copper content was noted in drill logs the sample was directly fire assayed. If gold content reached or exceeded 3.11 g/t (0.1 oz/ton) then the reject portion of the sample was shipped to the Min-En lab in Vancouver for metallic screen fire assaying. For this process, the entire reject was pulverized to -120 mesh, recombined with the previous pulp portion and metallic screened for +120 mesh gold. Two 30-gram assays were then completed on the -120 mesh fraction and the results averaged. The values from both fractions (+120/-120) were then mathematically combined to produce a net gold value. Copper and silver analyses were done on a 2-gram sample split from the initial pulp. No check assays were documented and rejects were stored on site.

In 1991 sample preparation was modified on recommendations from Min-En after they undertook a number of tests on coarse reject core samples. The raw core was crushed to 3mm and a 500 gram split taken, pulverized to 95% passing -120 mesh then rolled and bagged for analysis. The remaining reject was bagged and stored in Smithers. Samples were fire assayed using a one assay ton sample weight. For each batch of 24 samples a blank and a standard were submitted. When the value of the standard fell outside of a 95% confidence limit the entire batch was re-run.

Internal monitoring of copper assays was routinely conducted on 50 sample batches. The top 10% of all gold assays per page were rechecked and reported in duplicate along with the standard and blank. Every 20th sample was shipped to Eco-Tech laboratories of Kamloops for check assay. The check assays showed reasonable correlation for copper and fairly good correlation with gold at grades exceeding 0.25 g/t, although Eco-Tech assays tended to be marginally higher. Gold grades below 0.25 g/t showed considerable variation.

During 1991, metallic screening of high-grade gold samples was not routinely carried out. Min-En laboratories tested three high-grade gold samples (+3 g/t) for metallic gold content. Based on this preliminary work they concluded metallic or coarse particles could have influenced high gold assays at Galore Creek. Min-En recommended that metallic gold assays be done on composite samples from high grade zones prior to further resource estimation. A comprehensive re-assay program was undertaken in 1991 to reliably establish the distribution and grade of gold mineralization in the Central Zone. This was mainly due to the absence of continuous gold assays from drill holes completed before 1990. Thirty-one holes drilled in the Central Zone during the 1960's and 70's had no gold assays and the remainder only had gold and silver assays performed on composited mineralized zones (+0.4 % Cu). A total of 100 tonnes of samples were shipped from the property to Min-En laboratories in Smithers, B.C. for assay. This total encompassed 18,784 samples from the Central, Southwest and North Junction Zones with 95% of the samples from the Central Zone. The sample total included 12,786 coarse reject samples from earlier drilling and underground sampling and 5,990 core samples from pre-1991 Central Zone drilling. Results from about 600 of the reject samples could not be used due to problems with duplicate sample numbers.

15.2 2003-2004 NovaGold Samples

NovaGold's work during the 2003-2004 campaigns included core logging, sample layout, and sample splitting. A professionally registered geologist employed by NovaGold oversaw all of the work including core logging, sample splitting, and the shipment to the labs.

Shipment of the samples between 2003 and 2004 from the Galore Creek camp occurred on a by-hole basis. Rice bags, containing 4 poly-bagged core samples each, were marked and labeled with the sample numbers and the Vancouver lab address. The rice bags were then assembled into sling loads for transport by helicopter to the Bob Quinn airstrip. The samples were stored in a secure metal structure at the Bob Quinn airstrip until they were transported by Banstra, a commercial trucking company, who delivered the samples directly to ALS Chemex in Vancouver.

All assay analysis for the 2003-2004 program, as in the 2005 program, were carried out by ALS Chemex Labs of Vancouver B.C., which is widely used by the mining and exploration industry. ALS Chemex carries the highest certification for registered assayers, including ISO 9002 and ISO: 9001:2000. According to NovaGold, ALS Chemex is working towards ISO 17025 certification.

Upon arrival at the lab the samples were logged into a tracking system and each individual sample weight was recorded. The samples were then prepped by drying and the entire sample crushed. A 250 gram split was pulverized to >85% passing 75 microns. Sample analysis for gold content was completed using 30 gram fire assay charges with an atomic absorption spectrometry (AA) finish. Accurate results were provided between 0.05 ppm and 1000 ppm gold. Additional ICP analysis was conducted for 34 elements by aqua regia acid digestion and ICP-AES. The copper analyses were completed by AA, following a triple acid digestion. In total during 2003-04, excluding quality control samples, 38,866 samples from the four project areas were submitted to ALS Chemex for analysis.

A comprehensive quality assurance/quality control (QA/QC) program was followed during the 2003-04 seasons. Duplicate samples were used to monitor and measure precision (reproducibility), blank samples representing material with very low concentrations of copper and gold and were used to test for contamination of the samples, while standard samples and assay checks were used to test the degree of accuracy. In total, 1,005 samples were sent for quality control purposes, as blind duplicates, blanks or standards, representing approximately one in every 10 samples or 10.7 % of the samples collected during 2003-04.

15.3 2005 NovaGold Samples

In 2005, as in the prior NovaGold programs, the drill core was logged by a team of geologists and split using a rock saw. A professionally registered geologist continued to supervise all of the work. Half of the drill core has been retained in core boxes at the Galore Camp for future reference and sampling. The other half of the core was sampled and shipped to ALS Chemex Laboratory in Vancouver. The half-core samples were placed in a plastic bag and tagged with a sample number. Groups of samples were placed in larger rice bags and shipped by helicopter to the Bob Quinn airstrip. From the Bob Quinn storage area the samples were trucked directly to the lab in Vancouver. A submission sheet was sent along with each batch of samples so the lab could confirm receiving the samples. In 2005, security tags were strapped onto the rice bags to ensure that the bags were not opened prior to their arrival to ALS Chemex Labs.

NovaGold has organized the core storage facility, where much of the historic drill core is still in core boxes and available for review as well as assay checks. Minor amounts of the old drill core are difficult to identify by hole number and depth due to degradation of the boxes.

ALS Chemex Labs continued to carry out all of the assay work in 2005. In total, excluding quality control samples, 57,180 samples from the Galore Creek property have been submitted for analyses since 2003. In 2005, as in 2003-2004, the copper analyses were completed by atomic absorption spectrometry (AA), following a triple acid digestion. Gold analyses were completed by standard one-assay-ton fire assay with AA finish. All samples submitted from 2003 to 2005 were also analyzed for 31 elements by ICP-MS following an aqua regia digestion.

Essentially the same QA/QC program was followed during the 2005 season. Duplicate samples were used to monitor and measure precision (reproducibility), blank samples representing material with very low concentrations of copper and gold were used to test for contamination of the samples, while standard samples and assay checks were used to test the degree of accuracy. In 2005, about 2,900 samples were sent for quality control purposes, representing approximately 15% of the samples collected during 2005.

The results as reported by ALS Chemex are within acceptable error limits with respect to accuracy and precision, while the contamination was deemed to be minimal. The sample preparation security and analytical procedures used for the 2005 drilling campaign are adequate.

16. Data Verification

Section 16 was prepared by Mr. Michael J. Lechner,RPG. Resource Modeling Inc.

16.1 Electronic Database Verification

Data verification has been an ongoing process since the property was acquired by NovaGold in 2003. The drill hole database has been audited and checked by both in-house personnel and several independent consulting companies. Prior to 1990, drilling data were manually entered into a computer database. After 1990, assay data were transferred digitally from data files supplied by the assay laboratory, which minimized the potential for generating data errors.

In 1992 Kennecott conducted an assay database check of 375 assay files representing approximately 7,500 samples. The most common mistakes that were found consisted of typographical errors and missing assay data. There was also some confusion because of missing prefixes in check samples from Eco-Tech Lab. Consequently, all previous data were merged into a single database, audited and converted from imperial to metric units prior to the final resource estimation.

Following NovaGold's involvement with the project, at least three separate data verification programs have been completed. These consist of the 2003 Simpson, 2004 G.R. Tech, and 2006 RMI audits.

The author requested original ALS Chemex assay certificates for ten 2005 drill holes totaling about 3,863 metres, which represents about 7% of the 2005 drilling program. ALS Chemex assay certificate results for 1,720 copper, gold, and silver assay records were then compared with the assay records stored in NovaGold's electronic database. The author found four actual errors and nine records in the official database that could not be verified by assay certificates. According to NovaGold personnel, ALS Chemex has no record of having received or assayed these samples. Table 16-1 summarizes the results for the last major independent database reviews.

Table 16-1: Summary of Assay Data Verification Results

Review Program	Number Checked	Errors Found	Error Frequency
Simpson, 2003	1,329	23	1.70%
Hatch et al., 2004	15% of database	minimal	minimal
G.R. Tech, 2005	3,368 drill hole location and orientation	19	0.60%
G.R. Tech, 2005	2,307 gold assays	92	4.00%
G.R. Tech, 2005	3,368 copper assays	50	1.50%
Resource Modeling Inc., 2006	1,720 copper assays from 2005 drilling	5	0.29%
Resource Modeling Inc., 2006	1,720 gold assays from 2005 drilling	4	0.23%
Resource Modeling Inc., 2006	1,720 silver assays from 2005 drilling	4	0.23%

Based on the results from previous independent database audits and the author's own audit of the 2005 drilling data, it is the author's opinion that the Galore Creek assay database is well within the generally acceptable limits for North American drill hole databases and can be used to estimate Mineral Resources.

16.2 Drill Hole Collar Check

The author compared drill hole collar elevations from the electronic database with the NovaGold supplied topographic surface. Seventeen drill holes were found to have a difference in collar elevations of ± 7.5 metres. All but one of these differences was explained by the fact that some of the older holes were collared on top of a glacier which has since retreated giving the appearance that the drill hole collar was too high. Drill hole GC04-490 was found to be 20.6 metres too high in the database that was supplied to the author. This hole was only weakly mineralized and was used in the estimate of Mineral Resources that are subject to this report. The collar elevation for this hole has since been corrected by NovaGold.

16.3 Historical Drilling Comparisons

The Galore Creek Project has been explored by core drilling since 1961. Nearly 56% of the core that has been drilled predates NovaGold and the implementation of NI 43-101. The majority of this drilling is located in the Central Zone of the Galore Creek Project. Information regarding QA/QC programs and their enforcement on historic assay analysis is incomplete. Table 16-2 shows the distribution of NovaGold drilling data by area versus historically derived data.

Table 16-2: Distribution of NovaGold Drilling by Area

Mineralized Zone	% Metres Drilled by NovaGold
Butte	53%
Central Area	30%
Gap	72%
Grace	92%
Junction	0%
Middle Creek	78%
North Junction	35%
North Rim	57%
Profit	100%
Reconn	27%
Saddle	52%
Southwest	72%
West Fork	96%
West Rim	0%

NovaGold has indirectly assessed the quality of the historic assays by pairing recent NovaGold drilling, which have been subjected to rigorous QA/QC protocols, to the historic data, and then comparing the distributions using quantile–quantile plots (QQ plots).

Copper and gold assay grades were composited into 5-metre lengths and then grouped by year, which was deemed as the smallest unit in which to investigate. A block model with 5m x 5m x 5m blocks was constructed with copper and gold composite grades from each annual campaign assigned by the nearest neighbor method to a variable in the block model. The search limit for composite assignment was 30 metres.

Blocks which had pairs of historic composite grades and NovaGold composite grades within 10 metres of each other were exported to an ASCII file and were used in subsequent QQ plot analyses. The average copper grades for the historical drilling by year are compared with copper grades derived from NovaGold drilling in Table 16-3. These averages are for blocks that are within 10 metres of both a historical and a NovaGold drill hole composite. Similar statistics are shown by year for gold composites.

Table 16-3: Historical vs. NovaGold Copper Grades

Year	No of Pairs	Historical Cu Grade (%)	NovaGold Cu Grade (%)	% Difference
1962	30	0.34	0.34	0
1963	513	0.53	0.48	11
1964	477	0.43	0.6	-28
1965	742	0.47	0.48	-2
1966	271	0.4	0.21	88
1972	195	0.81	1.1	-26
1973	240	0.35	0.22	58
1976	41	0.19	0.24	-23
1991	63	0.13	0.06	126
Total	2,572	0.47	0.48	-3

Table 16-4: Historical vs. NovaGold Gold Grades

Year	No of Pairs	Historical Au Grade (g/t)	NovaGold Au Grade (g/t)	% Difference
1962	32	0.28	0.11	154
1963	513	0.28	0.24	14
1964	477	0.62	0.94	-34
1965	742	0.36	0.41	-12
1966	271	0.14	0.10	39
1972	190	0.24	0.24	1
1973	193	0.19	0.14	33
1976	41	0.03	0.06	-51
1991	63	0.14	0.06	146
Total	2,520	0.33	0.39	-14

Annually, there are indications of both low and high bias for both copper and gold grades when compared to the NovaGold composites. Overall, the QQ plots indicate that lower grade ranges correlate reasonably well; however, for both copper and gold, historic assays appear to be biased low relative to NovaGold drilling. The historic composites averaged about 3% lower than NovaGold for copper and about 14% lower for gold composites. The historic gold composites were much lower in grade than the NovaGold composites at higher cutoff grades. Since NovaGold assay results have been demonstrated to be reasonable and reproducible based on QA/QC results, it suggests that the historical copper and gold analyses may have been biased low. No bias adjustment has been incorporated by NovaGold.

QQ plots are shown in Figure 16-1 and Figure 16-2 which compare all historical assay data with NovaGold assay data for copper and gold, respectively. In these QQ plots, the historical data is shown along the X-axis while NovaGold data is shown along the Y-axis. Copper grades from the historical and NovaGold programs compare reasonably well up to one percent copper. Above a one percent copper cutoff grade there is an increasing bias between the two data sets. A similar relationship exists between the historical and NovaGold gold assays as shown in Figure 16-2 above a 0.5 g/t cutoff grade.

Figure 16-1: Historical vs. NovaGold Assays - Copper

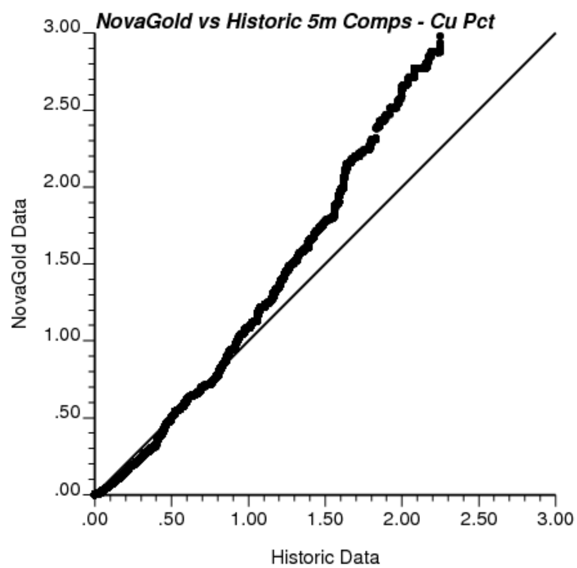
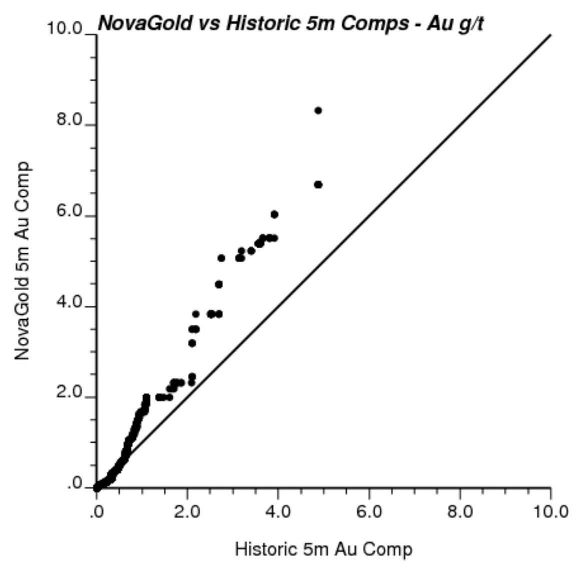


Figure 16-2: Historical vs. NovaGold Assays - Gold



Similar QQ plots are shown for copper and gold in the feasibility study documents. These plots compare nine annual drilling campaigns with NovaGold drilling campaigns for the years 1962-66, 1972-73, 1976, and 1991.

Based on this analysis the author recommends that NovaGold complete some twin holes to further investigate and possibly quantify the indicated bias. The author believes that as a result of this analysis the use of the unadjusted historic assay data for resource estimation is reasonable and at worst may result in an underestimation of copper and gold grades in areas largely supported by historic assays.

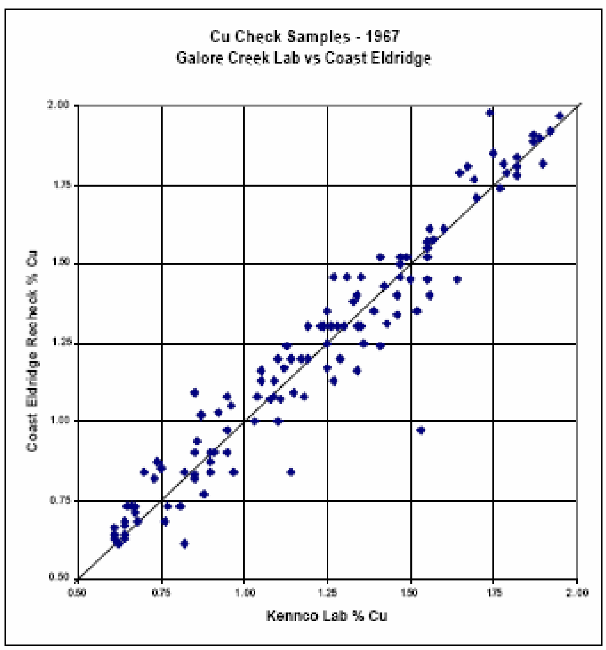
16.4 QA/QC Results

Quality Control/Quality Assurance data are used to monitor the precision and accuracy of assay sample data. For example, the routine submission of blank or barren material is commonly used to monitor possible contamination in the lab and to provide a measure of data accuracy. Similarly, submitting control or standard reference materials (SRM's) of known values into the sample stream are used to monitor data accuracy. Collectively all of the QA/QC results are used to monitor assay results to ensure that the data are repeatable and appropriate to be used to estimate Mineral Resources.

16.4.1 Pre-NovaGold

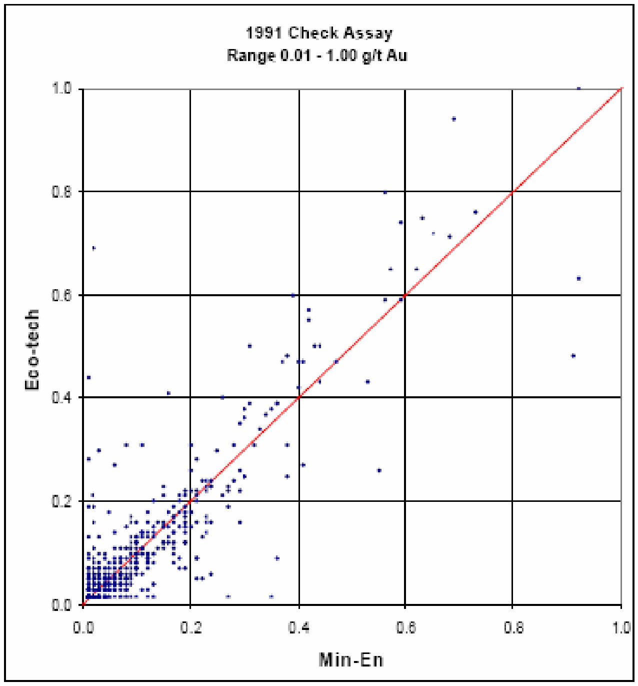
The majority of the pre-NovaGold drilling is located in the Central Zone of the Galore Creek Project as shown in Table 16-2. Very little information regarding QA/QC programs or check assay results are available. As described in Section 16.1, Kennecott investigated assay reproducibility by sending about 140 pulps to five separate labs. Figure 16-3 (taken from Simpson, 2003) is a scatter graph that compares copper assays from the Kennco Lab (X-axis) with copper assays from the Coast Eldridge Lab (Y-axis).

Figure 16-3: 1967 Kennco Same Pulp Assay Comparison



In 1991, Mingold assayed their samples at Min-En Lab in Smithers, B.C. They shipped every 20th sample to the Eco-tech Lab in Kamloops, B.C. for check assay. According to Simpson, 2003, there was a "fairly good" correlation between the Min-En and Eco-Tech labs for gold exceeding 0.25 g/t, although the Eco-Tech assays tended to be marginally higher than the Min-En lab results. Figure 16-4 is a scatter graph that compares 571 check assays that were analyzed in 1991.

Figure 16-4: 1991 Gold Check Assays



16.4.2 2003 – 2005 NovaGold

NovaGold instituted a rigorous QA/QC program with the initiation of their first limited drilling campaign in 2003. That program was continued and improved with ramped up drilling activities in 2004 and especially 2005. The basic elements of the NovaGold QA/QC program consisted of submitting standard reference materials (SRM), blanks (barren material), and requesting that a duplicate sample be prepared at a frequency of one sample each per every 20 samples that were submitted to ALS Chemex. A variety of commercial copper and gold standards were purchased from WGC, which is located in Vancouver, B.C. Table 16-5 summarizes the number of QA/QC samples by year that were submitted by NovaGold.

Table 16-5: 2003-2005 NovaGold QA/QC Samples

Year	Total No. of Assayed Samples	No. of Duplicate Samples	No. of Blanks	No. of Standards	Total No. of QA/QC Samples	Total No. of Samples	Percentage of QA/QC Samples
2003	1,243	72	58	67	197	1,440	14%
2004	7,599	397	408	382	1,187	8,786	14%
2005	22,267	1,356	1,347	1,342	4,045	26,312	15%
Total	31,109	1,825	1,813	1,791	5,429	36,538	15%

The performance of blanks, standards, and duplicates are shown in the Feasibility Study for the 2003 to 2005 NovaGold QA/QC samples. Figure 16-5 and Figure 16-6 show box plots, QQ plots, scatter graphs, and relative percent difference graphs for the 2005 NovaGold duplicate copper and gold analyses.

Figure 16-5: 2005 NovaGold Duplicate Copper Assays

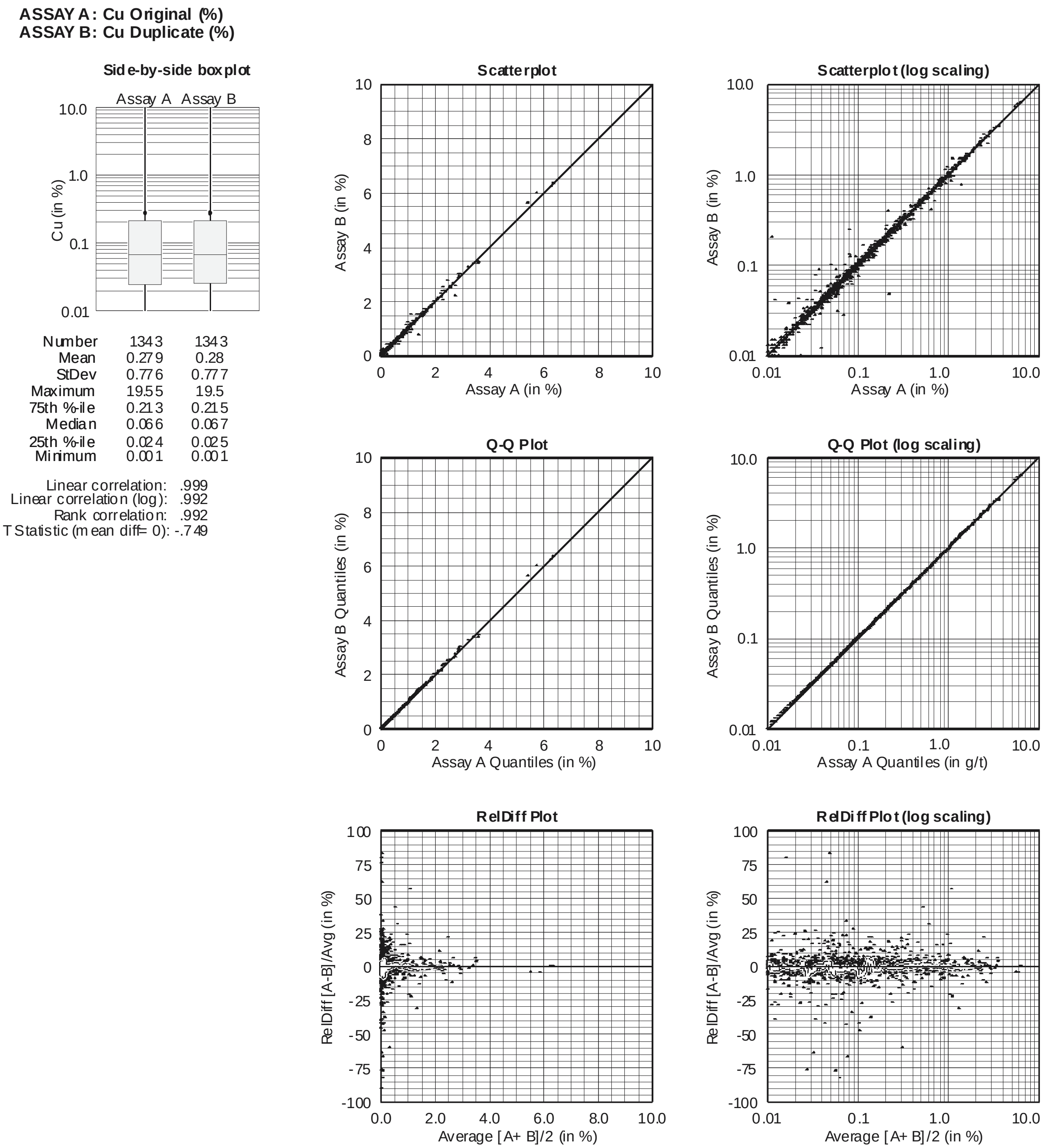
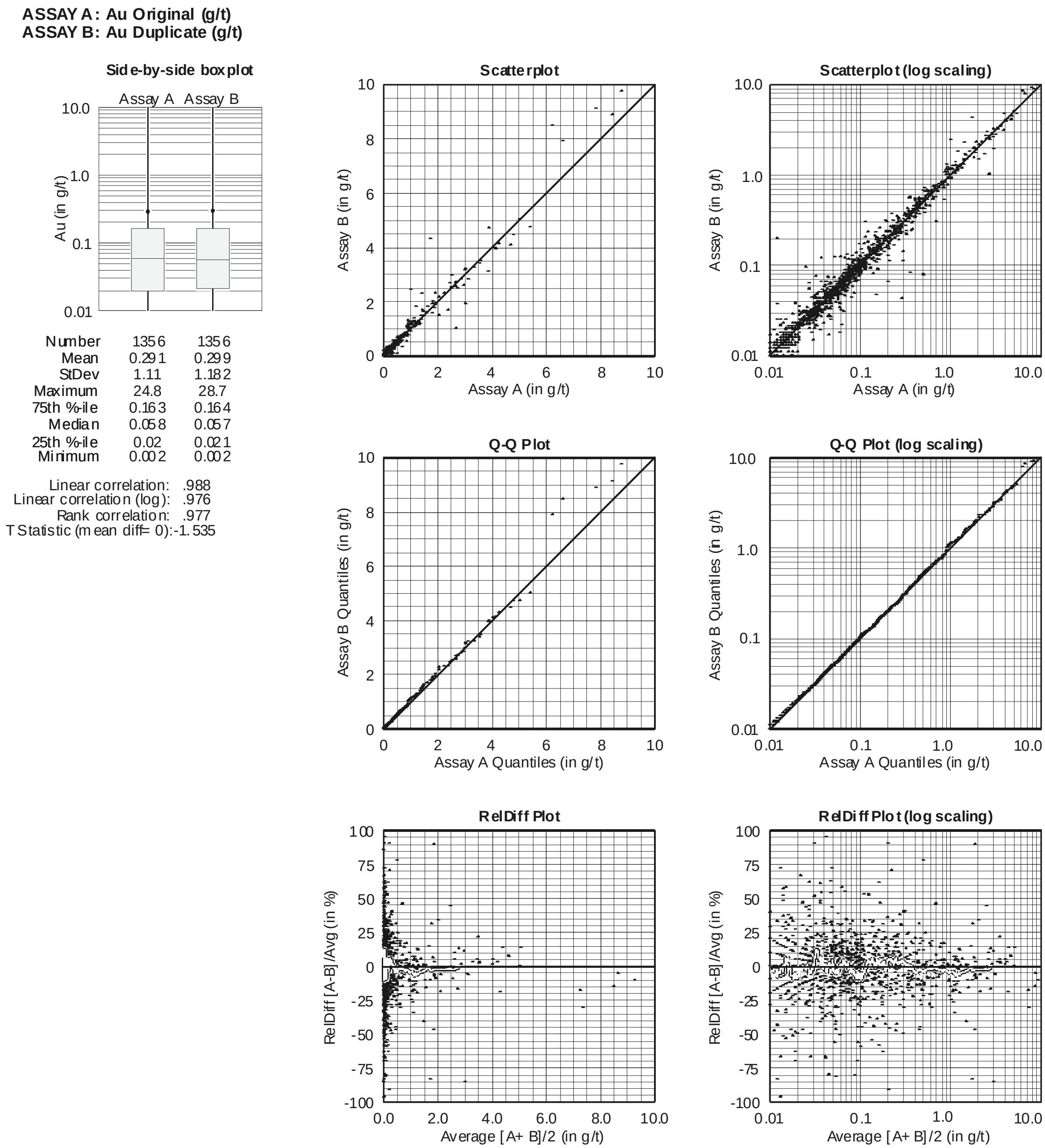


Figure 16-6: 2005 NovaGold Duplicate Gold Assays



NovaGold monitored the performance of the standard reference (SRM) samples that were assayed by ALS Chemex. During 2005, NovaGold requested that approximately 350 copper and 140 gold assays be re-assayed due to SRM's associated with those samples returning assay values that were outside of expected tolerances. The author reviewed those samples and agrees with NovaGold's decision to have those samples re-assayed. NovaGold did not request samples to be re-assayed based on the performance of blanks and duplicates. The author has recommended that in the future, NovaGold request that all samples be re-assayed that are associated with blanks that are out of tolerance (i.e. return values 3-5 times that of the lab's detection limit). In 2005 there were about six batches of samples (3 copper and 3 gold) where the returned values for the barren material was significantly above the detection limit for copper or gold. The author does not consider this to be material, but urges NovaGold to more closely track QA/QC results and to re-assay all sample batches that are associated with any control samples that are out of tolerance.

17. Adjacent Properties

Section 17 was prepared by Mr. Michael J. Lechner,RPG. Resource Modeling Incorporated

There are two other significant copper/gold properties in the immediate Galore Creek area, Copper Canyon and Grace. The Copper Canyon claims are adjacent and immediately to the east of Galore Creek, and cover 1,574 hectares.

In 2004 SpectumGold Inc. entered into an option agreement with Eagle Plains Resources Ltd. giving NovaGold the exclusive right to earn up to an 80% interest in the Copper Canyon Property comprised of 4 located claims adjoining the Galore Creek property. The effective date of the agreement is October 1, 2003. During the first option period, NovaGold must issue 296,296 shares to Eagle Plains on or before February 26, 2007 and incur property expenditures of CA$3 million on or before October 1, 2013 to earn a 60% interest. To earn another 20% interest, NovaGold must make a payment of CA$1 million within 90 days of exercising the first option and complete a feasibility study by no later than September 2011. In addition, NovaGold assumed the commitments of the underlying Eagle Plains option agreement dated May 28, 2002 with Bernard Kreft that included payments totalling CA$250,000 and a 2% NSR.

The Copper Canyon property has seen intermittent exploration work from the mid 1950's up through 1991. This historic exploration work at Copper Canyon indicated the presence of widespread gold, silver and copper mineralization similar to the Galore Creek deposit. To date mineralization has been identified in three separate areas on the property. A resource estimate for Copper Canyon was completed in February 2005 by Hatch for NovaGold and outlined an inferred resource, at a cut-off grade of 0.35% copper equivalent, totaling 164.8 million tonnes grading 0.35% Cu, 0.54g/t Au, and 7.15g/t Ag (Hatch, 2005).

The author did not verify information regarding adjacent properties and does not know whether that mineralization is indicative of the mineralization on the property that is subject to this technical report.

18. Mineral Processing and Metallurgical Testing

Section 18 was prepared by Mr. Hoe Teh, P.Eng., Hatch

18.1 Summary

A comprehensive metallurgical program has been completed on fresh drill core samples from 2005 drilling to further 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 using CEET and JKSimMet methodologies, mineralogy, and minerals recovery by batch and locked cycle flotation. Models have been developed to project copper, gold and silver recoveries in mining blocks for each pit. Pilot plant campaigns have also been completed, primarily to generate concentrate samples for dewatering tests and marketing purposes, and tailings samples for dewatering tests and environmental purposes.

The 2005 drill cores again show that copper occurs predominantly as chalcopyrite and chalcopyrite-bornite in a mixed silicate host. Pyrite occurrence is variable, with pyrite-copper sulphide mass ratio averaging less than the 3:1 ratio observed in the samples in the PEA study. At a grind of 80% passing 150 microns, 50% to 60% of copper sulphides are liberated resulting in good overall copper recovery by flotation. The gold particles are fine at nominally 8 to 12 microns and are contained largely within the sulphide matrix. Although unlikely to be recoverable by gravity concentration they exhibit a high recovery by flotation into the rougher copper concentrate. A primary grind of 80% passing 200 microns could be employed to achieve the same metals recovery. The metallurgical response deteriorates as the grind approached 300 microns.

Ore hardness, in terms of Bond Ball Mill Work Index, varied between 13 kWh/t and 21 kWh/t over the various pits. The average hardness in the dominant Central Pit was 16.5 kWh/t, similar to that determined in the PEA work. The hardness, measured as SAG Power Index (SPI), ranged from 20 minutes to 141 minutes across the deposit. The MinnovEX CEET model indicated that the proposed mill circuit would be SAG mill limiting when treating ores with SPI greater than 115 minutes. The “stick” rock is generally harder and more abrasive than the “broken” rock.

For 65,000 tpd ore throughput, a SABC grinding circuit comprising a 40 ft (12.2 m) x 24 ft (7.3 m) SAG mill, two 26 ft (7.9 m) by 36 ft (11 m) ball mills and two 600kW pebble crushers has been proposed based on the CEET and JKSimMet models and optimal equipment design.

The current feasibility study program has further validated the flowsheet developed in the previous work. The flowsheet will comprise of rougher flotation, regrind of rougher concentrate and three stages of cleaner flotation using a simple reagent scheme that utilizes PAX as the primary collector and MIBC as the frother. The use of 3418A, a more selective dithiophosphinate collector, instead of PAX, might produce slightly higher concentrate grade at similar recovery. Occasionally, a guar gum carboxymethyl cellulose reagent will be required to disperse talc-like materials and minimize their adverse impact on flotation responses. Variable amounts and occurrences of these talc-like materials have been observed in the drill cores from certain areas of the deposit. The talc-like mineral species have not been identified.

The feasibility study program also confirmed that chalcopyrite and bornite ores from various pits have similar metallurgical responses.

Models have been developed for each pit to project copper recovery from variable head grades at constant concentrate grade and to project gold and silver recoveries from copper recovery. Using a head grade of 0.7% copper for each pit, the projected recoveries are as follows:

- Central Pit – 92% Cu, 76% Au, 71% Ag at 28% Cu concentrate grade
- Southwest Pit – 88% Cu, 68% Au, 57% Ag at 26% Cu concentrate grade
- North Junction Pit – 88% Cu, 70% Au, 62% Ag at 28% Cu concentrate grade
- West Fork Pit – 91% Cu, 70% Au, 68% Ag at 28% Cu concentrate grade

A model has also been developed for projecting copper recovery from ores containing non-sulphide copper. Copper recovery will be lower and will vary with the proportion of non-sulphide copper content while the gold and silver recoveries will correlate with copper recovery. Using a head grade of 0.7% copper and assuming 20% of the total copper occurring as a non-sulphide, the model projects recoveries of 71% copper, 55% gold and 51% silver at a 28% Cu concentrate grade.

Since gold and silver recoveries largely follow copper recovery, the gold and silver in ores with very low copper, and largely occurring within pyrite grains, would not be recovered.

The concentrates are relatively clean, with the exception of variable fluorine content, as indicated in the Neil Seldon marketing report (July 2006). This may attract some penalty although the dollar amount is not likely to be significant.

18.2 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. A process flowsheet was developed based on batch and locked cycle tests then validated in a pilot plant program. The work showed that a high copper recovery into a marketable concentrate could be achieved.

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 metallurgical program was run to determine the amenability of the ore in the new Southwest Zone ore to the flowsheet developed in the 1960's.

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 its 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 the PEA. It expanded the understanding of the variability of mineralization in the Central and Southwest Zones and identified additional ore zones designated as Junction, West Fork and Copper Canyon. As a result, a comprehensive metallurgical program was conducted to determine the metallurgy associated with the variable mineralization. All the samples tested were drill cores from discrete intervals, drill hole composites and composites of drill holes by ore zones or pits. They represented ore at various head grades, mineralogy and geological classification of "broken" or "stick" ore. The program also included a limited grinding test program on drill core samples for preliminary grinding circuit design.

During 2005, NovaGold conducted in-fill drilling to define the ore reserves and metallurgical drilling to provide samples for further detailed testing on the variability of mineralogy towards this Feasibility Study.

The 2005 - 2006 metallurgical test program has been conducted by G&T Metallurgical in three phases:

- Phase I program was run on four composites of mineralized zones to assess their mineralogy, grindability and metallurgical performance using locked cycle techniques based on the flowsheet developed in the earlier studies. The gold occurrence and deportment was also investigated.
- Phase II program investigated the variability of metallurgical responses of ores from the various ore zones, geographically and spatially.
- Phase III program involved pilot plant campaigns on the same composites as in Phase II to generate environmental samples and concentrate samples for marketing purposes.

Grindability studies by SGS MinnovEX and SGS Lakefield were also conducted to design the grinding circuit. A brief flotation study and simulation was also conducted by SGS MinnovEX.

18.3 2005 - 2006 Metallurgical Testwork Summary

G&T Metallurgical Services Ltd. (Kamloops, BC, Canada) conducted all the flotation tests and the Bond ball mill work index of the composites used in the flotation program, while SGS Lakefield and MinnovEX (Toronto, ON, Canada) ran the additional grindability and the flotation simulation tests.

18.3.1 Phase I Program

Laboratory locked-cycle flotation tests and grindability tests have been conducted on four composites of drill cores. The composites have been made up according to geological mineralization groupings, ore zones, copper head grade, and spatial distributions in the deposit, with input from NovaGold geologists. The drill hole maps showing the origins of the four composites are shown in Figure 18-1 to

Figure 18-4.

The locked-cycle flotation simulated the flowsheet comprising roughers, regrind of the rougher concentrate followed by three cleaner stages.

Bond ball mill work indices were obtained for the four composites to assess their grindability.

Figure 18-1: Drill Hole Maps

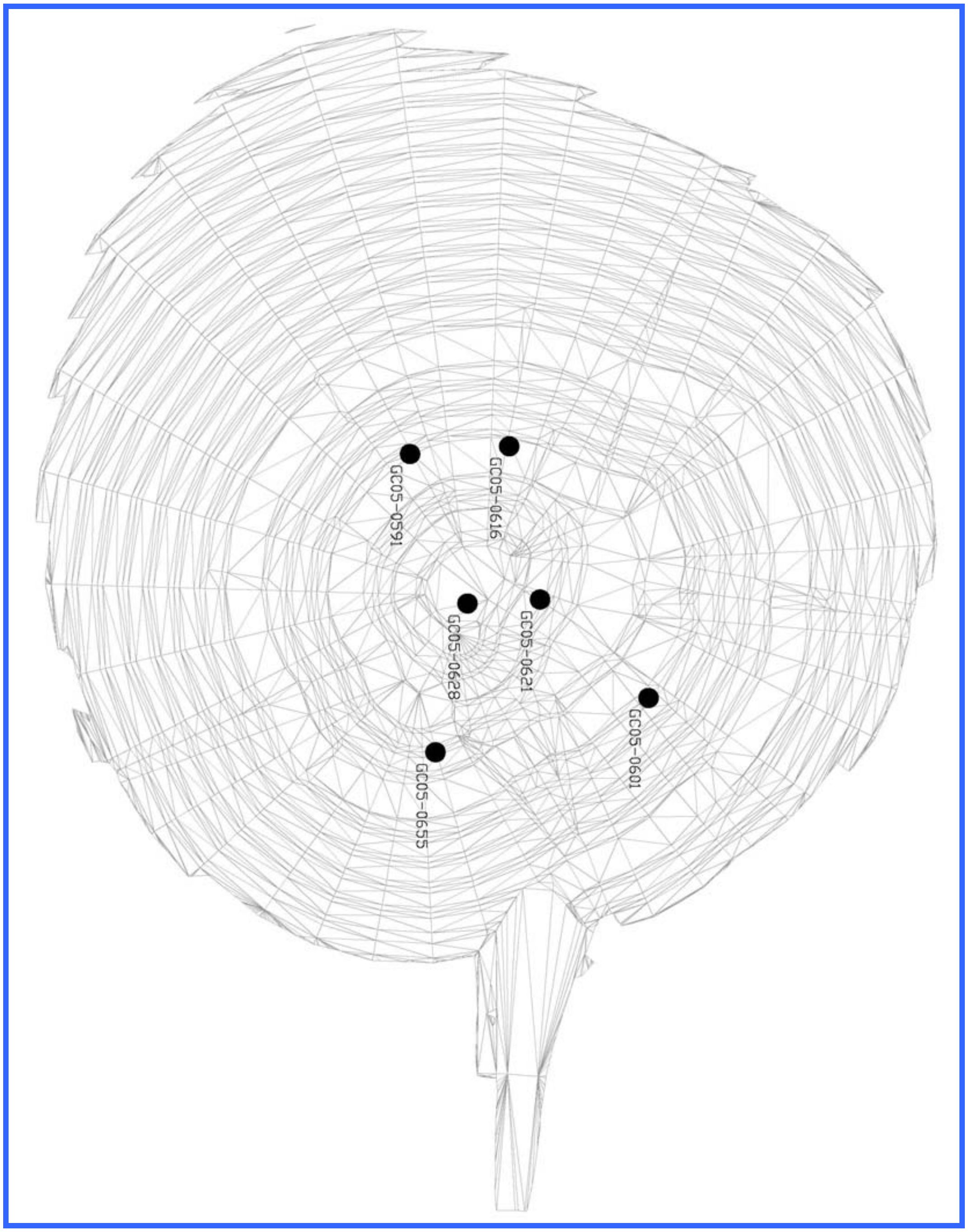


Figure 18-2: Drill Hole Maps

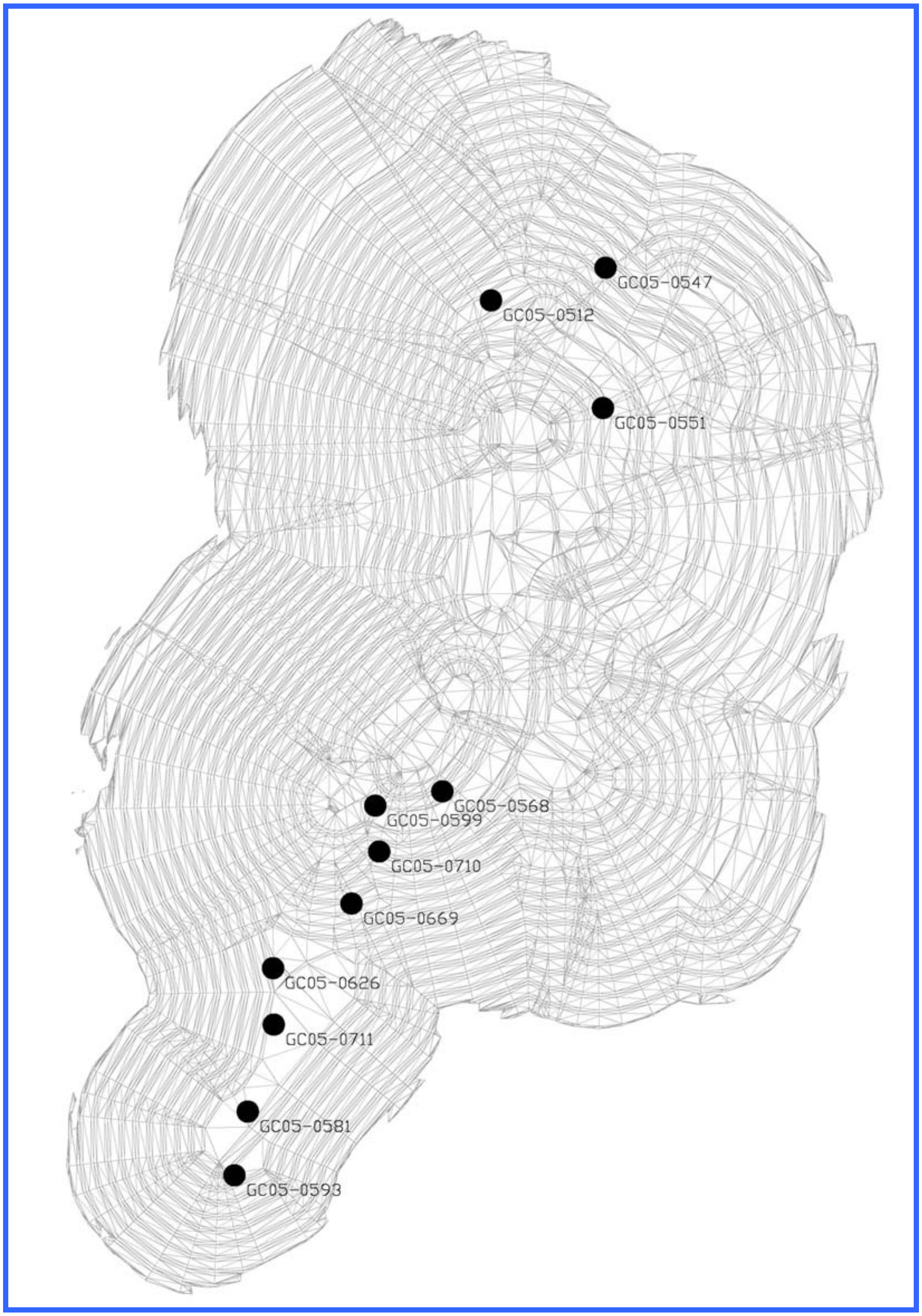


Figure 18-3: Drill Hole Maps

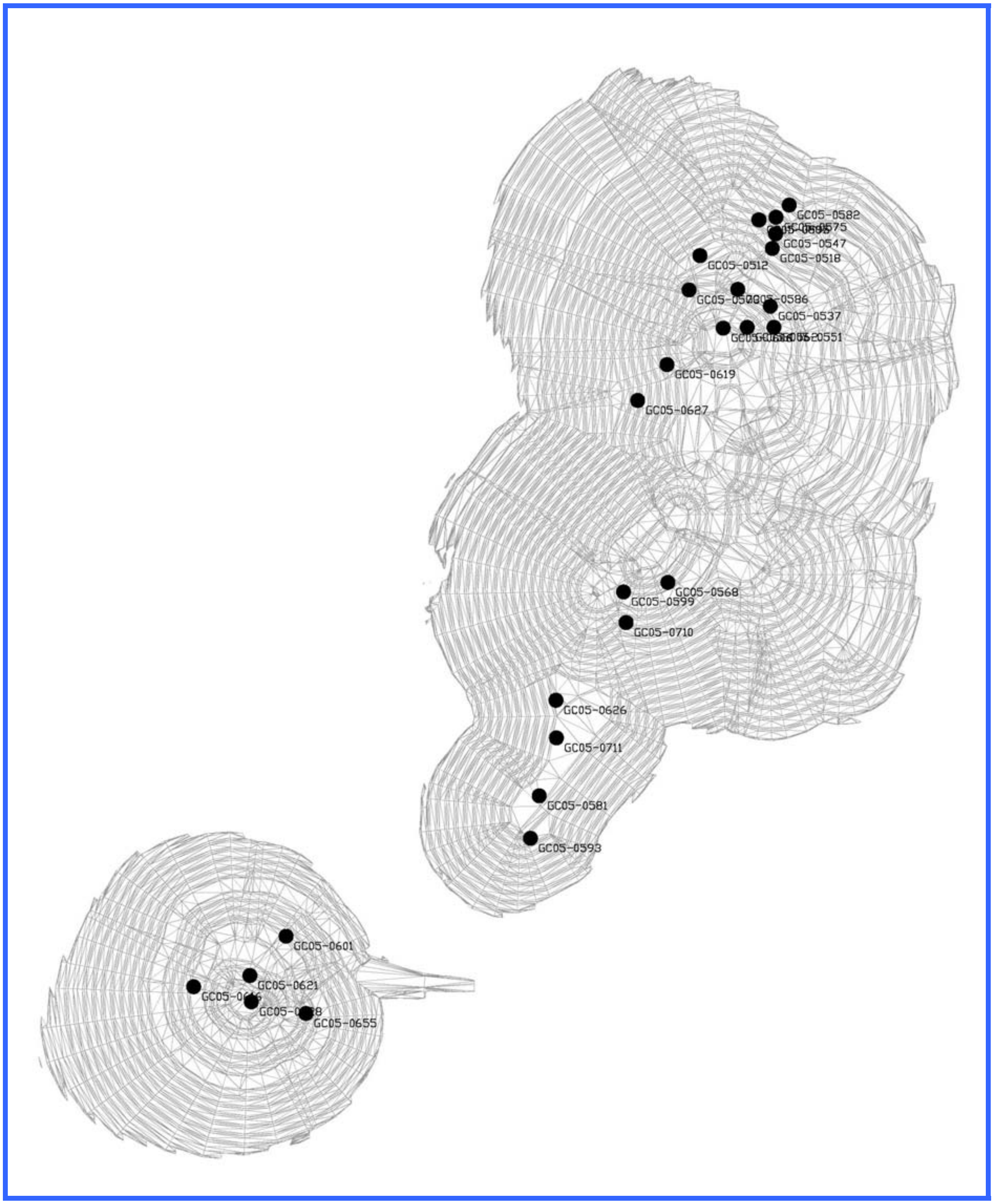
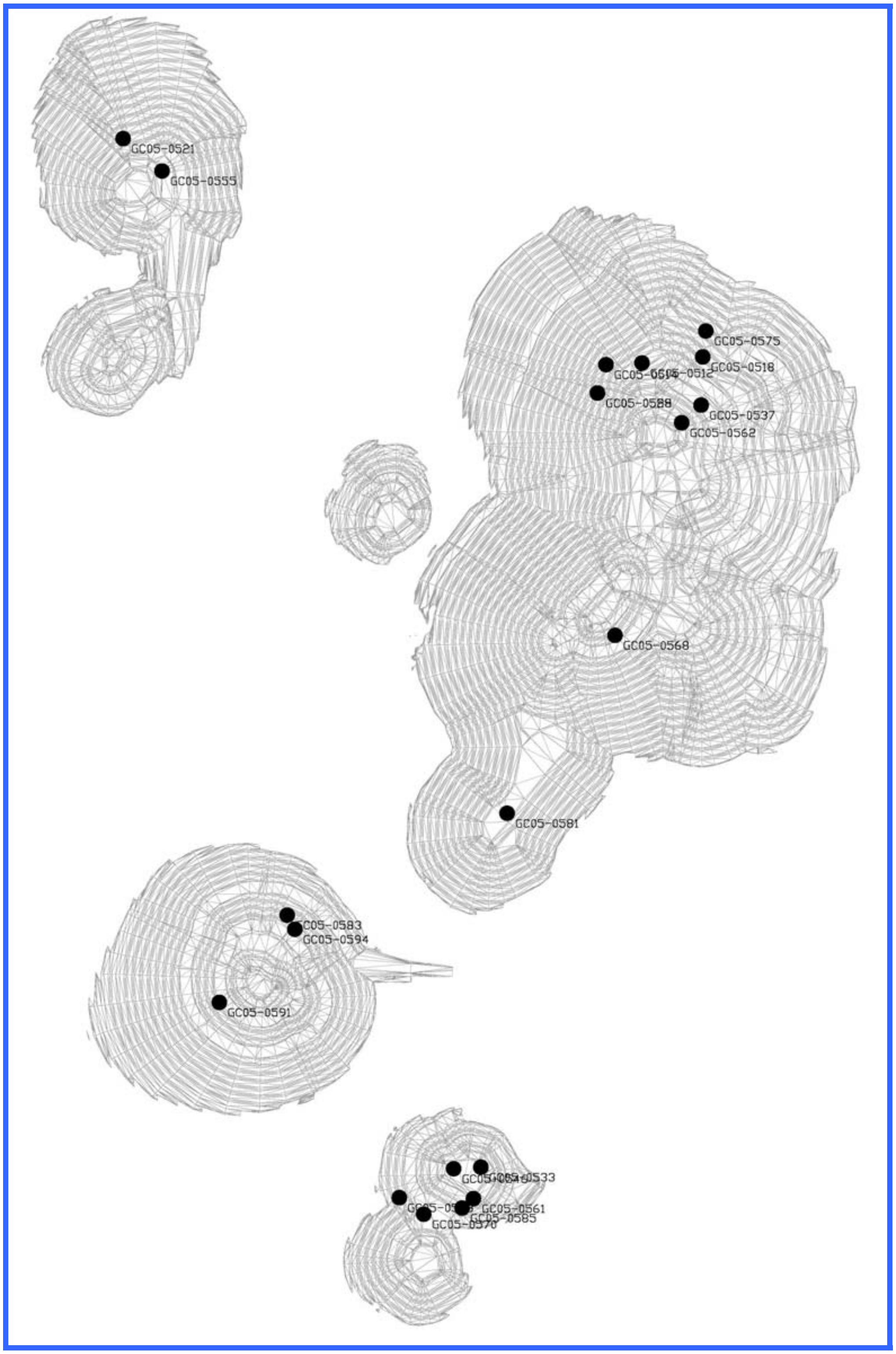


Figure 18-4: Drill Hole Maps

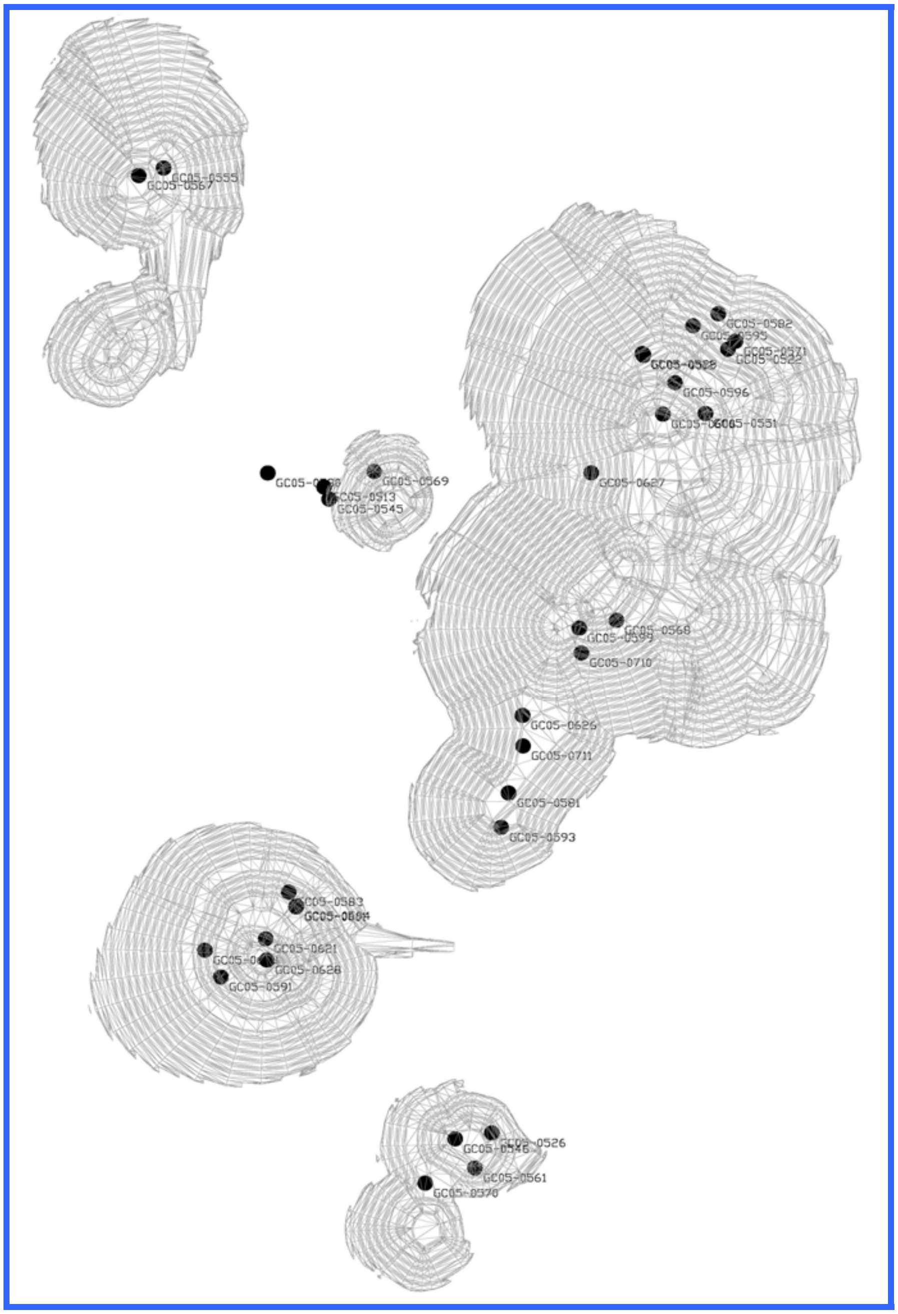


18.3.2 *Phase II Program*

In the Phase II program, 66 drill core samples with a range of head Cu grades from the seven mineralized zones were tested to investigate grade-recovery relationships. Some samples with significant non-sulphide content were also tested. Comparative Bond Ball Mill work Index and single open cleaner flotation tests were run on each of the samples.

The locations of drill cores used in this phase are shown in Figure 18-5.

Figure 18-5: Variability Sample Drill Hole Locations



18.3.3 Phase III Program

In Phase III, three composites were run through the G&T pilot plant to generate environmental data, concentrate and tailings samples for dewatering tests, and concentrate for marketing purposes.

18.3.4 Program Development

The initial work in the 1960's, conducted by Kennecott Corporation, developed a flowsheet for the Central Pit ore comprising grinding, rougher flotation, regrinding and three-stage cleaner flotation. The work included batch open-circuit, locked cycle and pilot plant tests.

Further work was conducted in 1991 to validate the flowsheet as a result of additional drilling in the Central Pit and Southwest Pit. The metallurgical performance was not as good as that obtained in the 1960's. Finer primary grind, increased reagent additions and longer residence times were used to achieve the results that are comparable to the 1960's.

In 2003, NovaGold continued with the exploration of Central and Southwest Pits and selected high grade drill cores to determine their amenability to the flowsheet developed previously using the current standard reagents for sulphide flotation. The work validated the flowsheet and demonstrated that copper floated readily using the current industry standard reagent scheme.

Further exploration in 2004 produced additional drill core samples for metallurgical testwork to investigate the impact of mineralization variability on the flowsheet and metallurgy. The testwork also investigated the amenability of the newly identified Junction, West Fork and Copper Canyon ores to the flowsheet.

Infill drilling and drilling for metallurgical samples continued in 2005 to confirm the mineral reserves in the various pits and their metallurgy for the Feasibility Study.

The study metallurgical test program began in mid 2005 and was completed in February 2006. The test program utilized fresh quarter and half-core samples from the 2005 drilling program.

In addition, 70 drill core samples were selected for SAG mill power index measurements while thirteen drill core samples were selected for grindability testing by the JKSimmet method to compare mill circuit designs by the two methodologies.

18.4 2005 Phase I Program

The samples from the different ore zones and pits were characterized in terms of mineralogy, hardness and mineral fragmentation then floated in batch open-circuit and locked cycle runs to determine their metallurgy. Four composites covering the following pits were tested:

- Central Pit - North Gold Lens (NGL), South Gold Lens (SGL), Central Replacement Zone (CRZ)
- North Junction Pit (NJ)
- Southwest Pit (SW)
- West Fork Pit (WF)
- Middle Creek Pit (MC)

The four composites were made up as follows:

- Composite 1 – 100% Southwest (Figure 18-1)

- Composite 2 – 100% Central comprising 61% SGL, 23% CRZ, 16% NGL (Figure 18-2)
- Composite 3 – 19% Southwest, 81% Central comprising 22% SGL, 20% CRZ and 39% NGL (Figure 18-3)
- Composite 4 – 2% Southwest, 7% North Junction, 14% West Fork, 77% Central comprising 3% SGL, 10% CRZ and 64% NGL (Figure 18-4)

18.4.1 Ore Characterization

The assays and mineralogy of the composites are shown in Table 18-1.

The principal copper mineral in the composites was chalcopyrite. Some of the samples showed chalcopyrite-bornite mineralization with traces of chalcocite. Non-sulphide copper accounted for approximately 1% of the total copper content in the composites. Pyrite was the other dominant sulphide mineral occurring at variable mass ratios with copper sulphide. Magnetite accounted for the remaining iron in the composites.

Table 18-1: Assays and Mineralogy of Composites

		Composites			
	units	1	2	3	4
Assay					
Copper (total)	%	0.67	0.98	0.78	0.71
Copper (non-sulphide)	%	0.01	0.12	0.01	0.01
Gold	g/t	1.23	0.35	0.74	0.93
Silver	g/t	3	8	7	7
Iron	%	4.2	2.8	4.2	6
Mineralogy					
Chalcopyrite	%	1.9	2	1.9	1.5
Bornite	%	0.1	0.3	0.2	0.3
Chalcocite	%	<0.1	0.1	<0.1	<0.1
Pyrite	%	4	1.2	0.6	1
Gangue	%	94	96.4	97.3	97.2

A mineral fragmentation analysis has shown that 50% to 60% of the copper sulphides are liberated at a grind of 80% passing 150 μm as shown in Table 18-2.

This is consistent with the analysis of many of the large copper-gold flotation plants currently in operation. Subsequent flotation tests have shown that such degree of liberation was sufficient for high copper recovery in rougher flotation. The majority of the unliberated copper sulphides are locked primarily with silicate gangue but may be floated with sufficient collector additions. Only approximately 1% of the unliberated copper sulphides was locked with pyrite.

Table 18-2: Mineral Fragmentation Analysis

Composite	P80	Mineral Liberation (%)		
	Microns	Cu Sulphides	Pyrite	Gangue
1	147	48	73	95
2	138	62	86	97
3	152	58	89	95
4	160	59	85	97

Gold occurrences in the composites were located by the Automated Digital Imaging System. For Composites 1 and 2, up to 66% of the gold occurred as fine liberated grains of 8 to 12 microns. For Composites 1 and 4, its occurrence suggested that the majority of it would follow copper in flotation, whereas in Composite 2, the gold would appear to follow pyrite. The gold occurrence in Composite 3 suggested that it would report equally with pyrite and copper during flotation.

The ore hardness, in terms of Bond Ball Mill Work Index, for each of the composites is shown in Table 18-3. These are similar to that obtained in the previous study.

Table 18-3: Ball Mill Bond Work Index of Phase 1 Composites

Composite	Bond Work Index
	kWh/tonne
1	16
2	15.7
3	15.5
4	16.3

Ore hardness measurements were also made during the investigation into ore variability in Phase II and described in Section 18.5.2.

18.4.2 Flowsheet Validation

Batch and locked cycle rougher and cleaner flotation tests confirmed that the flowsheet and reagent scheme developed in the previous work apply to all four composites which represented ores from across the deposit. Tests were conducted at the nominal grind of 80% passing 150 microns and nominal regrind size of 80% passing 25 microns.

18.4.3 Batch Open Circuit Flotation

Batch rougher and cleaner tests have been run to investigate copper and gold recoveries at the standard grind of 80% passing 150 microns and the effect of coarser primary grind and regrind sizes on metal recoveries and concentrate grade.

18.4.3.1 Copper and Gold Recovery

Batch flotation tests showed that gold recovery correlated well with copper recovery in both rougher and cleaner flotation as shown in Figure 18-6 and Figure 18-7. Hence, process conditions that maximize copper recovery should also maximize gold recovery.

Figure 18-6: Copper-Gold Rougher Recovery Correlations

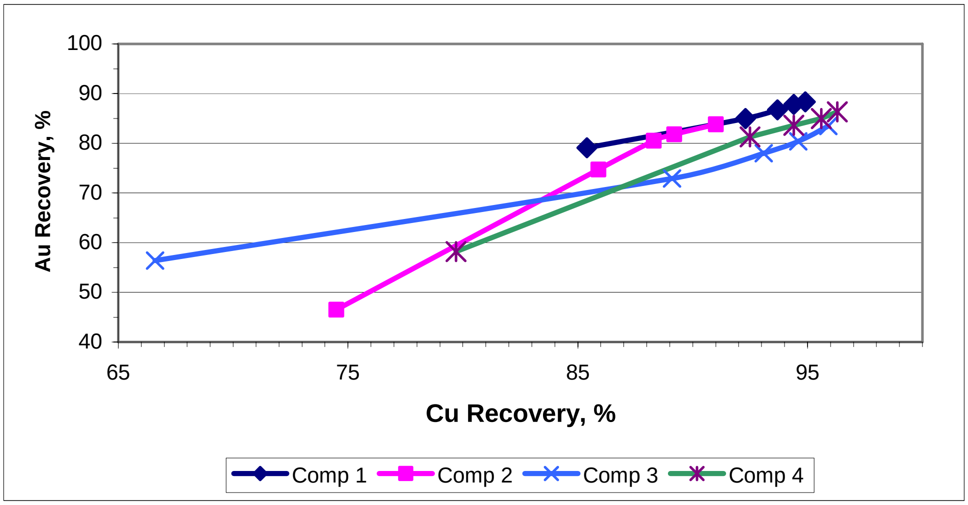
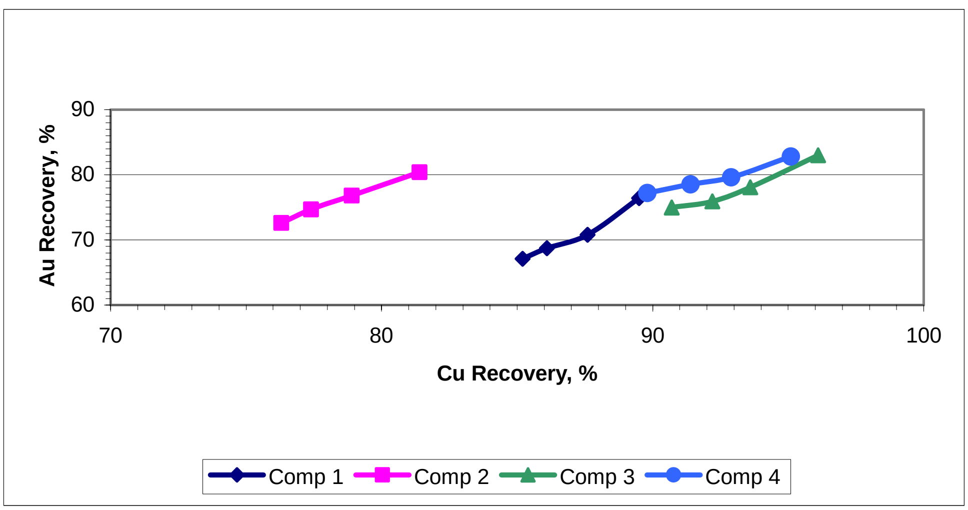


Figure 18-7: Copper-Gold Cleaner Recovery Correlations



18.4.3.2 Effect of Primary Grind Size

Tests were run Composites 3 and 4 to investigate the effect of primary grind size on metals recovery in rougher flotation based on the mineralogical analysis which indicated that a coarser than 150 micron grind might provide sufficient liberation for recovery. The results have been tabulated in Table 18-4

The results from the single tests showed that a coarser primary grind of around 80% passing 200 microns maintained copper and gold recoveries. The metal recoveries reduced significantly as the grind approached 80% passing 300 microns.

Table 18-4: Effect of Primary Grind in Rougher Flotation

Composite	Grind (micron)	Rougher Recovery (%)	
		Cu Recovery	Au Recovery
3	152	96.3	86.3
	200	95.9	83.6
	246	93.6	78.9
4	111	96.3	86.3
	212	96.8	89.4
	302	92.9	80.8

18.4.3.3 Effect of Regrind Size

The effect of 25 microns and 40 microns regrind size on metallurgical response was investigated using Composites 3 and 4. The results in Table 18-5 show that a slightly higher concentrate grade could be achieved at 25 microns regrind but the recovery might be slightly lower. The 40 microns size appears to be a reasonable regrind target. This was confirmed in the locked cycle program discussed in Section 18.4.4.

Table 18-5: Effect of Regrind Size

Composite	Regrind P80	Concentrate	Cu Recovery
	Microns	% Cu	%
3	25	32	90.7
	35	29.4	91.6
4	28	32.9	91.4
	38	29.1	92.4

18.4.4 Locked Cycle Flotation

Locked cycle flotation tests were carried out on the four composites to determine the stability of the circuit in a continuous operation, and the effects of primary grind, regrind size and collector type, PAX versus 3418A, on the metallurgical performance. One test was also run using site water to compare its effect on metallurgy with that using Kamloops tap water.

In general, the locked cycle results confirmed the batch test observations that a relatively coarse primary grind of 200 microns was adequate to produce a similar metallurgical response as at a finer grind. A regrind to 40 microns would be sufficient to achieve marketable concentrate grade. A finer regrind to 25 microns might improve the concentrate grade slightly at similar recovery, but its benefit would be offset by the higher grinding energy requirements.

The 3418A collector appeared to give better selectivity over PAX but at approximately 3 to 5 times higher consumption. Only single comparative tests have been run and more tests are recommended to confirm the observations. PAX has been selected for plant design and 3418A is an option for evaluation during plant operation.

Site water and Kamloops tap water gave similar metallurgical responses. This demonstrated that the use of tap water in all the tests and their metallurgical results were valid.

18.4.4.1 Composite 1

The results for Composite 1 have been tabulated in Table 18-6.

Table 18-6: Metallurgical Performance of Composite 1

Parameter	Product Head	Assay (% or g/t)			Recovery (%)		
		Cu 0.73	Au 1.19	Ag 6	Cu	Au	Ag
Effect of Primary Grind							
147 microns	Rougher Concentrate	5.1	7.46	30	94	82	75
218 microns	Rougher Concentrate	5.0	7.37	27	93	82	70
Effect of Regrind							
25 microns	Cleaner Concentrate	22.4	30.2	115	92	73	64
49 microns	Cleaner Concentrate	21.7	27.3	102	90	72	56
76 microns	Cleaner Concentrate	19.5	25.4	94	86	70	50
Effect of Collector							
3418A	Cleaner Concentrate	27.8	36.4	143	92	72	57
PAX	Cleaner Concentrate	22.4	30.2	115	92	73	64

The results showed that similar rougher performance was achieved at the primary grind of 147 microns and 218 microns. In the regrind tests, the performance at 49 microns was similar to that at 25 microns and slightly better than at 76 microns.

The 3418A collector appeared to be more selective producing a higher grade concentrate than with PAX at similar copper and gold recoveries. It is noted, however, that 3418A is more expensive than PAX at the same concentration and that its consumption was approximately five times more than PAX to achieve the observed results.

18.4.4.2 Composite 2

The results for Composite 2 have been tabulated in Table 18-7.

Table 18-7: Metallurgical Performance of Composite 2

Parameter	Product	Assay (% or g/t)			Recovery (%)		
		Cu	Au	Ag	Cu	Au	Ag
	Head	**0.99**	**0.38**	**8**			
Effect of Primary Grind							
127 microns	Rougher Concentrate	7.6	2.94	57	89	84	88
213 microns	Rougher Concentrate	9.9	3.54	63	86	75	72
Effect of Regrind							
23 microns	Cleaner Concentrate	35.1	10.2	225	81	68	64
41 microns	Cleaner Concentrate	30.7	9.5	208	82	66	64
Effect of Collector							
3418A	Cleaner Concentrate	30.0	9.3	213	86	66	70
PAX	Cleaner Concentrate	30.7	11.2	213	83	74	76

A primary grind of 213 microns gave slightly lower metal recoveries than at 127 microns. The concentrate grade, however, was higher. This would suggest that similar recoveries could be obtained at the same concentrate grade.

Based on the single tests, for this composite, the concentrate grade at the finer regrind size of 22 microns was higher than at 41 microns at similar metal recoveries.

There were no significant differences in metallurgical performance from using 3418A or PAX as the collector.

18.4.4.3 Composite 3

The results for Composite 3 have been listed in Table 18-8.

Table 18-8: Metallurgical Performance of Composite 3

Parameter	**Product**	**Assay (% or g/t)**			**Recovery (%)**		
		Cu	**Au**	**Ag**	**Cu**	**Au**	**Ag**
	Head	**0.79**	**0.79**	**7**			
Effect of Primary Grind							
152 microns	Rougher Concentrate	7.9	8.64	56	96	89	84
225 microns	Rougher Concentrate	7.5	8.01	43	96	87	68
Effect of Regrind							
28 microns	Cleaner Concentrate	28.2	22	159	93	81	61
39 microns	Cleaner Concentrate	26.3	21	151	94	71	66
48 microns	Cleaner Concentrate	25.9	18.9	150	92	75	64
Effect of Collector							
3418A	Cleaner Concentrate	32.0	23.1	183	94	76	66
PAX	Cleaner Concentrate	28.8	30.7	195	94	85	79

As shown in Table 18-8, the coarser primary grind at 225 microns produced the same recovery and concentrate grade as at 152 microns.

For this Composite, it appears that the finer regrind size of 28 microns improved the concentrate grade slightly at the same metals recovery over that at 48 microns.

Compared with PAX, the 3418A collector gave a higher grade concentrate. Copper recovery was similar but gold recovery was higher with PAX.

Table 18-9 shows the results from comparative tests using site water and Kamloops tap water. Similar flotation response was obtained using both types of water. Consequently, all remaining tests were completed using tap water.

Table 18-9: Effect of Site Water on Flotation - Composite 3

	Assay (% or g/t)			Recovery (%)		
	Cu	Au	Ag	Cu	Au	Ag
Tap Water						
Rougher Concentrate	7.5	7.7	47	95	83	73
Cleaner Concentrate	25.9	18.9	150	92	75	64
Site Water						
Rougher Concentrate	6.8	5.8	45	95	84	74
Cleaner Concentrate	24.8	19.8	147	93	77	65

18.4.4.4 Composite 4

The results for Composite 4 have been tabulated in Table 18-10.

Table 18-10: Metallurgical Performance of Composite 4

Parameter	Product	Assay (% or g/t)			Recovery (%)		
	Head	Cu 0.76	Au 0.96	Ag 7	Cu	Au	Ag
Effect of Primary Grind							
160 microns	Rougher Concentrate	8.3	7.34	52	97	86	84
213 microns	Rougher Concentrate	7.7	7.13	43	97	89	75
Effect of Regrind							
28 microns	Cleaner Concentrate	27.6	34.3	183	95	86	68
36 microns	Cleaner Concentrate	28.5	34.4	197	92	81	67
47 microns	Cleaner Concentrate	26.9	29.3	183	93	80	67
91 microns	Cleaner Concentrate	22.4	26.2	149	92	81	62
Effect of Collector							
3418A	Cleaner Concentrate	33.0	42.8	224	96	88	66
PAX	Cleaner Concentrate	28.4	24.3	168	95	81	78

As with all the other composites, the coarser primary grind at approximately 210 microns gave the same metallurgical performance as at the original design grind of 150 microns.

In the comparative regrind tests, ranging from 28 microns to 91 microns, the finer regrind will produce a higher grade concentrate at comparable recovery. It appears that regrinding to approximately 40 microns would result in the minimum copper losses to the cleaner scavenger tailings and maximum concentrate grade.

The 3418A collector appeared to be more selective than PAX, resulting in a higher grade concentrate. Its consumption, though, was approximately 3 times higher than PAX to achieve similar metal recoveries.

18.4.4.5 Concentrate Quality

A concentrate generated from each Composite was selected for a full minor elements assay to assess the quality of the concentrate that may be expected for that blend. The full analysis of the concentrates has been tabulated in Table 18-11.

The following are noted.

- Composite 4 appears to have the highest levels of a number of impurities
- Arsenic, tellerium, antimony, mercury and bismuth are generally low
- Selenium is relatively high in all the concentrates
- Chlorine is low in all the concentrates
- Fluorine is the highest in Composite 4

Table 18-11: Chemical Analysis of Copper Concentrates

Element	Unit	Composite 1	Composite 2	Composite 3	Composite 4
Cu	%	27.8	30.8	28.2	25.8
Au	g/t	36.4	9.2	22.0	34.4
Ag	g/t	143	202	159	197
Fe	%	30.0	26.5	30.1	28.8
Sb	%	0.014	0.010	0.012	0.015
As	g/t	<10	<10	<10	58
Bi	g/t	17	8	25	92
Cl	g/t	<100	<100	<100	<100
Co	%	0.009	0.015	0.018	0.022
F	g/t	330	230	240	1100
Pb	%	0.017	0.050	0.042	0.063
MgO	%	0.38	0.28	0.19	0.23
Hg	g/t	<1	<1	<1	<1
Ni	g/t	48	58	70	84
Pd	g/t	2.46	0.02	1.24	2.50
Pt	g/t	0.15	0.03	0.20	0.10
Rh	g/t	0.04	<0.01	0.01	0.05
Se	g/t	158	164	185	208
SiO_2	%	4.15	3.04	2.57	3.19
Te	g/t	20	<20	<20	<20
Zn	%	0.087	0.63	0.41	0.46

18.5 2005 Phase II Program

A total of 66 composites of drill core intercepts were selected from all the mineralized zones and pits. The samples listed below were selected to reflect geographical and spatial distributions, relative abundance in the deposit, and to provide sufficient data for developing recovery models for each pit.

- Central – 27 samples

- Southwest – 14 samples
- West Fork – 9 samples
- North Junction – 9 samples
- Middle Creek – 7 samples

This program phase investigated the variability in ore hardness and metallurgical response across the deposit. Seven samples, primarily from Middle Creek, that contained significant amounts of non-sulphide copper were also tested to determine their flotation response. The results were also used to develop models to project copper recovery against head grades and correlations between copper, gold and silver recoveries for use in the mining blocks of each pit.

18.5.1 Ore Composition

Except for the samples from Middle Creek, the rest of the samples had low levels of non-sulphide copper as determined by a dilute sulphuric acid leach. In the Middle Creek samples, 15% to 65% of the copper was in the non-sulphide form. The pyrite content was generally low at an average ratio of 1:1 with copper sulphides. This is lower than in the samples tested in the previous studies. Its occurrences across the deposit should be better defined as it impacts concentrate grade.

Note: the material form Middle Creek (high levels of non-sulphide copper) has been eliminated from the mine-plan.

18.5.2 Ore Hardness

Comparative Bond Ball Mill Work Index was estimated using the standard Bond work index determined for four samples in Phase I. The average comparative Work Index for each pit has been tabulated in Table 18-12.

The ore hardness varied over a narrow range for most of the pits, except West Fork where the ore was relatively harder.

Table 18-12: Average Comparative Work Index

Pit (Zone)	Comparative Bond Work Index
	kWh/tonne
Central (CRZ)	15
Central (NGL)	16
Central (SGL)	17
North Junction	17
Southwest	16
West Fork	20
Middle Creek	15

18.5.3 Metallurgical Response

Single open circuit cleaner test was run on each sample using the standard conditions established in the Phase 1 program to assess the variability in metallurgical response of ores across the deposit. The standard conditions were a target primary grind of 220 microns, regrind of 40 microns and the use of PAX as collector.

As expected, there is variation in metal recoveries depending on the head grade. The data indicated that copper recovery declined significantly when the head grade has less than 0.5% copper. The concentrate grade also varied depending on the copper mineralogy as ores containing bornite tended to produce higher grade concentrate.

18.5.4 Metals Recovery Models

The flotation results for each pit have been analyzed to develop the equations for projecting copper, gold and silver recoveries. The batch test data obtained during the previous Preliminary Economic Assessment (PEA) have also been incorporated to develop the equations. The equations are reasonable for projecting recoveries at constant concentrate grade in the resource and mining models. Locked cycle flotation data has shown some upside potential in the metal recoveries which typically may be achieved in the plant when operations have been optimized and, therefore, no de-rate or scale-up factor has been applied for the projection. As in the previous PEA study, copper recovery has been correlated with head grade (% Cu), while gold and silver recoveries have been correlated with copper recovery. The recommended maximum copper recovery for each pit has also been noted.

18.5.4.1 Central Pit – for Constant 28% Cu Concentrate Grade

The Cu equation is based on constant tails and applies to feed grades of 0.06% to 1.15% Cu.
Cu recovery is capped at 95% for feed grades above 1.15% Cu.

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

  Where F = % Cu in the feed

Recommended max recovery = 95%

- $\% \text{ Au recovery} = 18.123e^{0.0157x}$
- $\% \text{ Ag recovery} = 8.7262e^{0.0229x}$

  where x = % Cu recovery

18.5.4.2 South West Pit – for Constant 26% Cu Concentrate Grade

Two Cu equations are used – one for lower feed grade (0.062% to 0.45% Cu) and one for higher feed grade (0.45% Cu to 1.57% Cu). Cu recovery is capped at 92% for feed grades above 1.57% Cu.

For higher feed grades (0.45% to 1.57% Cu):

- $\% \text{ Cu recovery} = 4.8073\text{Ln}(F) + 89.835$

  where F = % Cu in feed

For lower feed grades (0.062% to 0.45% Cu):

- $\% \text{ Cu recovery} = 100.24\left(1 - \frac{0.062}{F}\right)$

  where F = % Cu in feed

The Au and Ag equations are applied to the entire range of Cu recovery.

- $\% \text{ Au recovery} = 15.766e^{0.0166x}$
- $\% \text{ Ag recovery} = 0.5059e^{0.0537x}$

  where x = % Cu recovery

18.5.4.3 North Junction – for Constant 28% Cu Concentrate Grade

The copper recovery equation and the applicable feed grade ranges are the same as for South West but is applied to a higher grade concentrate.

The same cap of 92% Cu recovery as for SW is used for 1.57% Cu and higher feed grades in this pit.

The gold and silver correlations are as follows.

- % Au recovery = $16.935e^{0.0161x}$
- % Ag recovery = $6.4242e^{0.0258x}$

where x = % Cu recovery

18.5.4.4 West Fork – for constant 28% Cu concentrate grade

For this pit, copper recovery has been capped at 95% for 1.25% Cu and higher feed grade.

For feed grades of 0.062% to 1.25% Cu:

- % Cu recovery = $100.21\left(1 - \frac{0.065}{F}\right)$

where F = % Cu in feed

- % Au recovery = $15.169e^{0.0167x}$
- % Ag recovery = $6.4242e^{0.0258x}$

where x = % Cu recovery

18.5.4.5 Oxide Cu Zones – for Constant 28% Cu Concentrate Grade

As expected, the recovery was low for ores with significant non-sulphide copper. The overall copper recovery was found to be dependent on the proportion of non-sulphide to sulphide copper.

- % Cu recovery = $100.234\frac{(F - S - 0.07)}{F(1 - 0.0334S)}$

where F = % Cu(tot) in feed

S = % Cu(soluble) in feed

- % Au recovery = $18.123e^{0.0157x}$
- % Ag recovery = $13.134e^{0.0193x}$

where x = % Cu recovery

18.5.5 Data Analysis

The variability tests produced a range of concentrate grades between the various pits and also within each pit. However, for mining and resource modelling purposes, metal recovery projections should be based on a fixed concentrate grade. Consequently, a normalized concentrate grade for each pit was obtained by interpolating the recovery-concentrate grade relationship in the tests. Correlations for Cu, Au and Ag recoveries were then developed for each pit at the following normalized concentrate grades:

- 28% Cu in Central, West Fork and North Junction
- 26% Cu in South West (due to higher pyrite occurrences)

18.5.5.1 Copper Recovery

All the Cu correlations are shown in Figures 18-8 to 18-12. The relative results from all the pits are shown in Figure 18-8 and the correlations for the four pits are shown in Figure 18-12.

The data confirmed the PEA results that there is no significant metallurgical differentiation between bornite-gold and chalcopyrite–gold domains in the Central pit. A common set of Cu, Au and Ag recovery equations may be used to cover NGL, CRZ and SGL zones within the pit.

For the Central pit, the copper recovery equation is similar to that used in the PEA. In Figure 18-9, the recoveries obtained for a composite of the Central pit in this FS were significantly lower. This was due to the presence of non-sulphide copper, particularly from two of the drill holes in SGL that were part of the composite. The recovery of the sulphide portion was high at over 90%. It is recommended that the copper recovery be capped at 95% in the resource and mining models, considering the results that were obtained for head grades of 1.2% Cu and higher (Figure 18-9).

Figure 18-10 shows the correlations for South West and North Junction. The equation for South West provides the best fit for North Junction. It is noted that the South West equation will be applied to a 26% Cu concentrate while North Junction equation will be applied to a 28% Cu concentrate.

shows the correlation for West Fork. The equation has been developed considering its results against those from the other pits.

Figure 18-8: Copper Recovery vs. Head Grade – All Pits 28% Cu Concentrate Grade

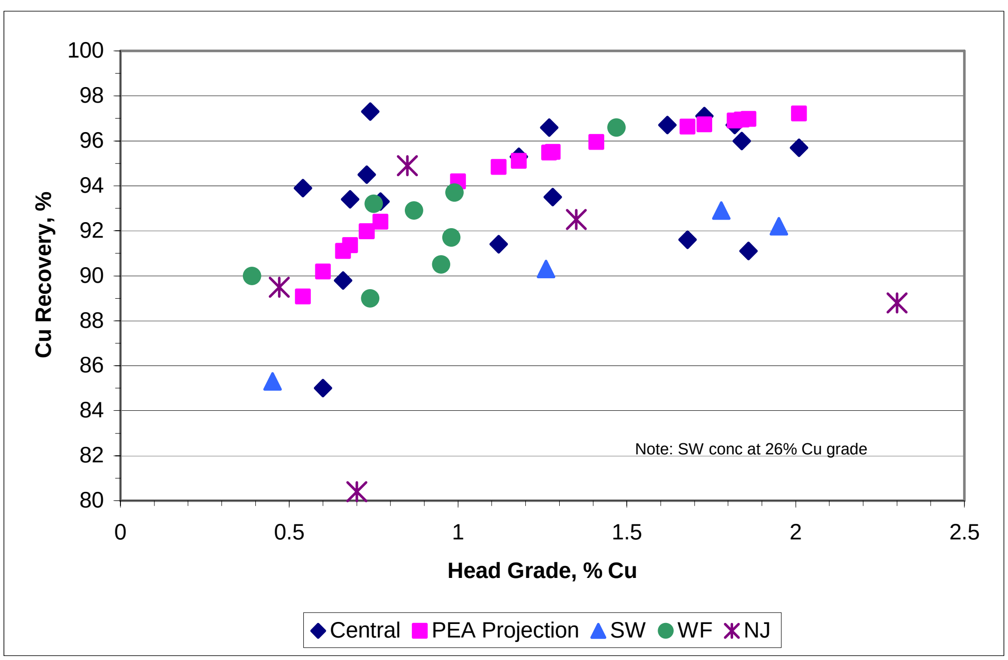


Figure 18-9: Copper Recovery vs. Head Grade – FS/PEA (Central) 28% Cu Concentrate Grade

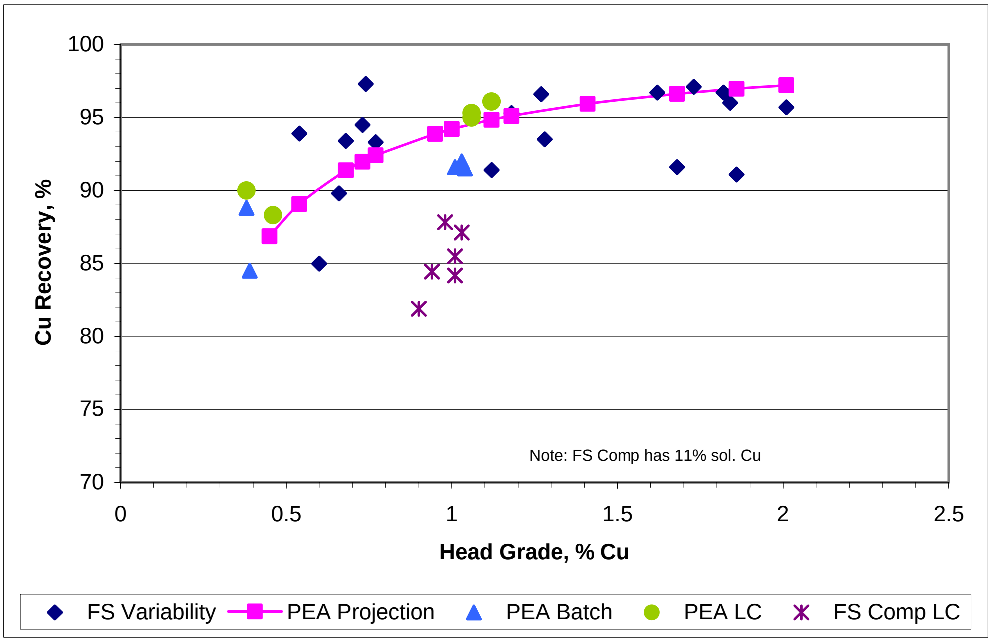


Figure 18-10: Copper Recovery vs. Head Grade – SW/NJ 26% Cu Concentrate Grade

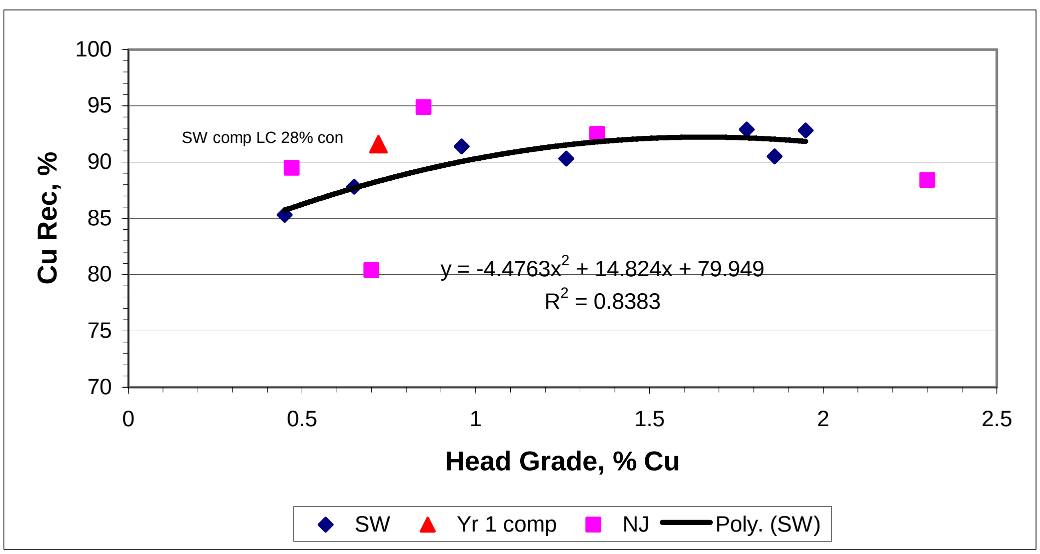


Figure 18-11: Copper Recovery vs. Head Grade – FS (WF) 28% Cu Concentrate Grade

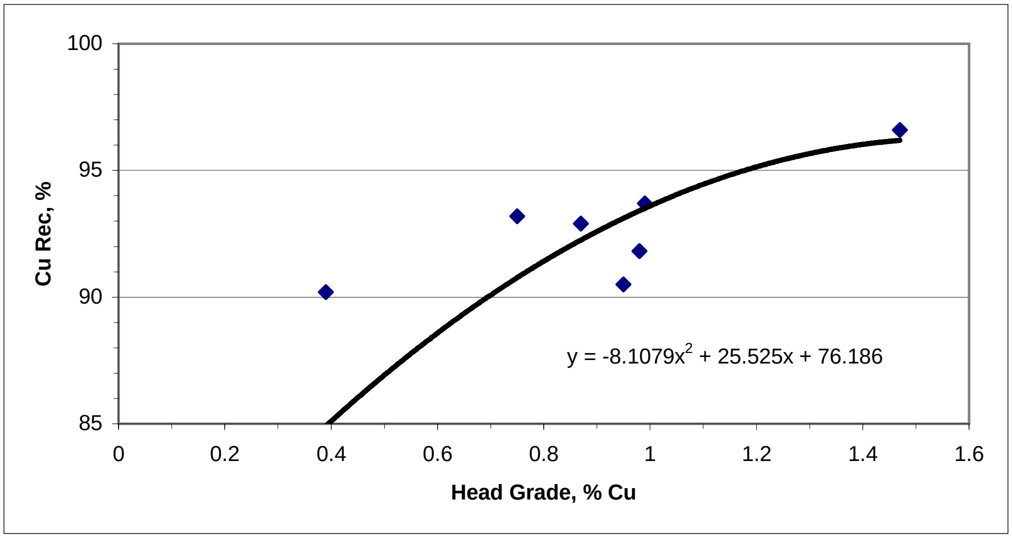
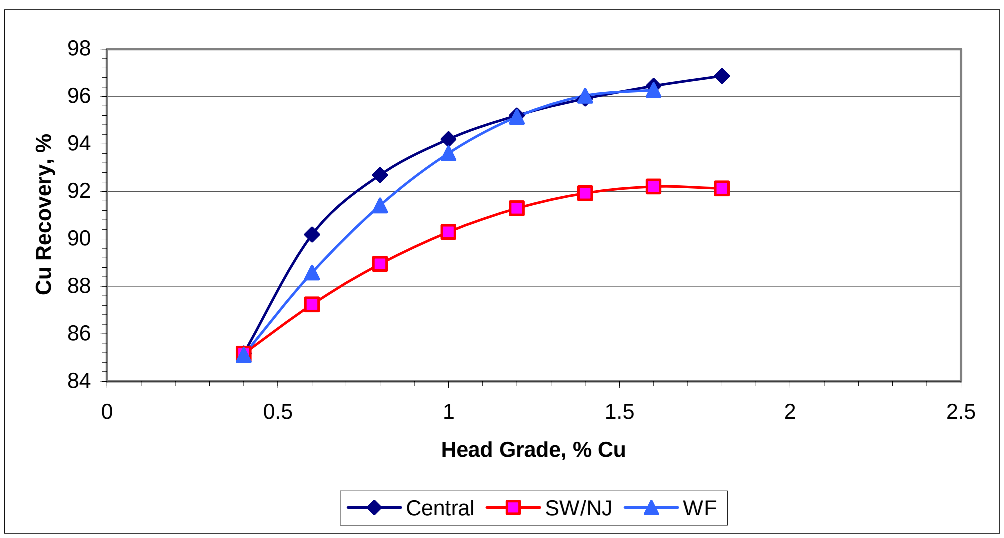


Figure 18-12: Copper Recovery vs. Head Grade Projection (All Pits)



18.5.5.2 Gold Recovery

In the PEA, Au was observed to follow Cu and its recovery was dependent on Cu recovery. Similar correlations have been adopted in the Feasibility Study.

The results for all the pits are shown in Figure 18-13. The correlation for the Central pit is shown in Figure 18-14. The recovery equation has been adjusted upwards from that in PEA based on the combined results from PEA and the current Feasibility Study. The equation for South West is similar to that used in the PEA. The equations for West Fork and North Junction have been developed based on the best fit of their results relative to those in the other pits.

Figure 18-13: Gold Recovery vs. Copper Recovery - All Pits (FS Batch)

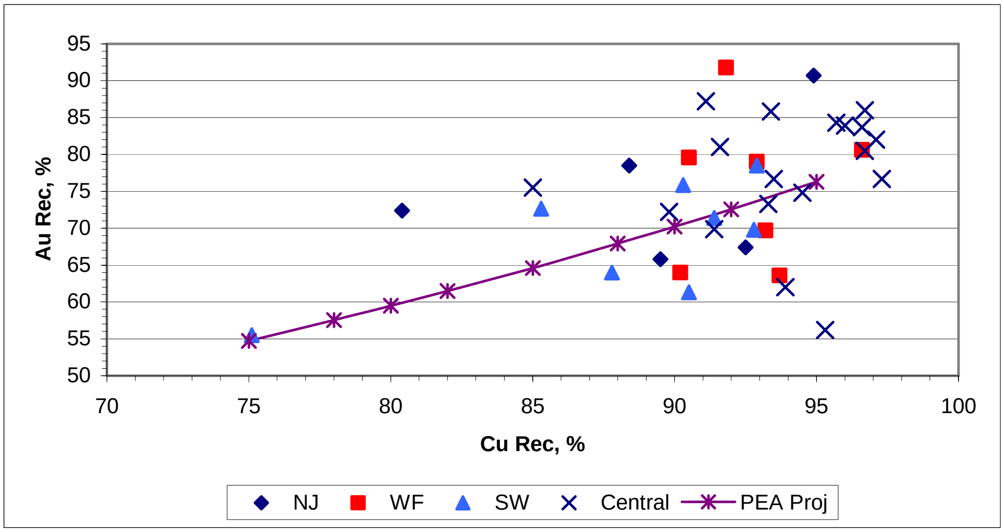
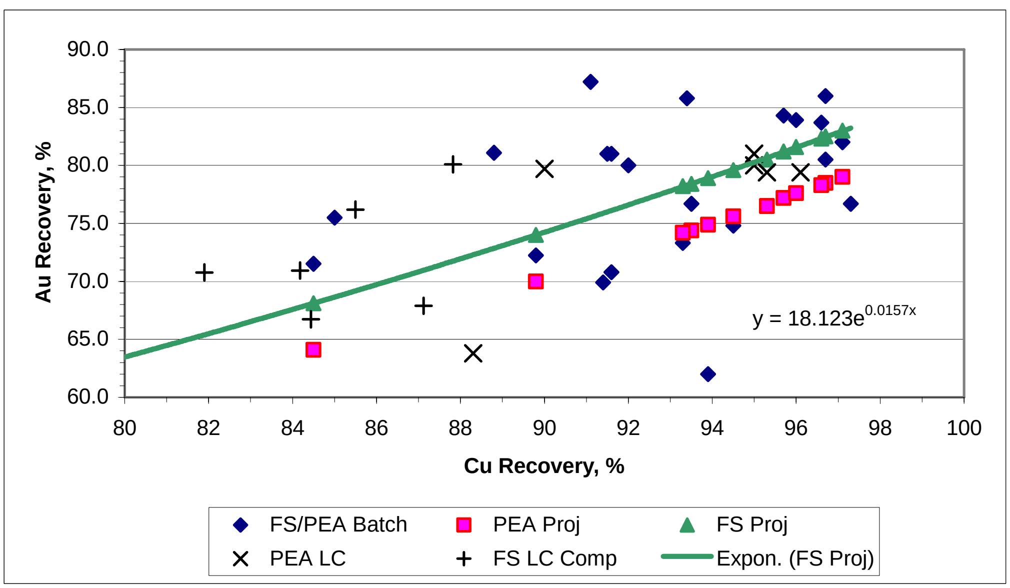
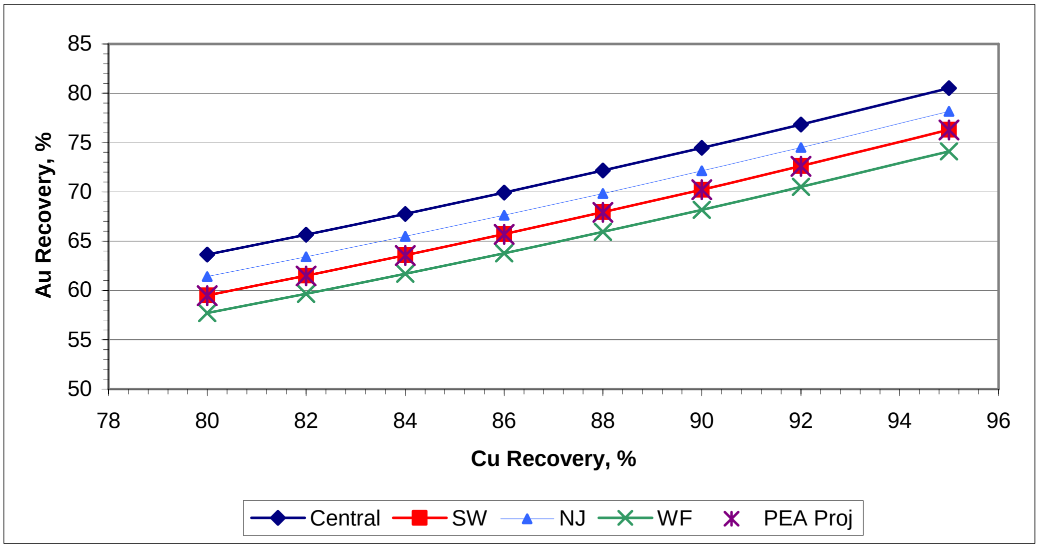


Figure 18-14: Gold Recovery vs. Copper Recovery – FS/PEA (Central) 28% Cu Concentrate Grade



The relative recovery correlations between the four pits are shown in Figure 18-15.

Figure 18-15: Gold Recovery vs. Copper Recovery Projection (All Pits)



18.5.5.3 Silver Recovery

As in the PEA, Ag recovery has been correlated with Cu recovery.

The results for all the pits are shown in Figure 18-16. The correlation for the Central pit is shown in Figure 18-17. It is slightly higher than in PEA based on the combined results from PEA and Feasibility Study.

The correlation for South West is shown in Figure 18-18. There is some scatter of results for North Junction but an equation, comparable to that used in the PEA, provides the best fit when the results are compared with those for the other pits. Some of the results for West Fork appear to be anomalous and have been excluded from the analysis. A comparative analysis of West Fork against the other pits suggests that an equation, comparable to that in PEA, would approximate the correlation.

Figure 18-16: Silver Recovery vs. Copper Recovery – All Pits (FS Batch)

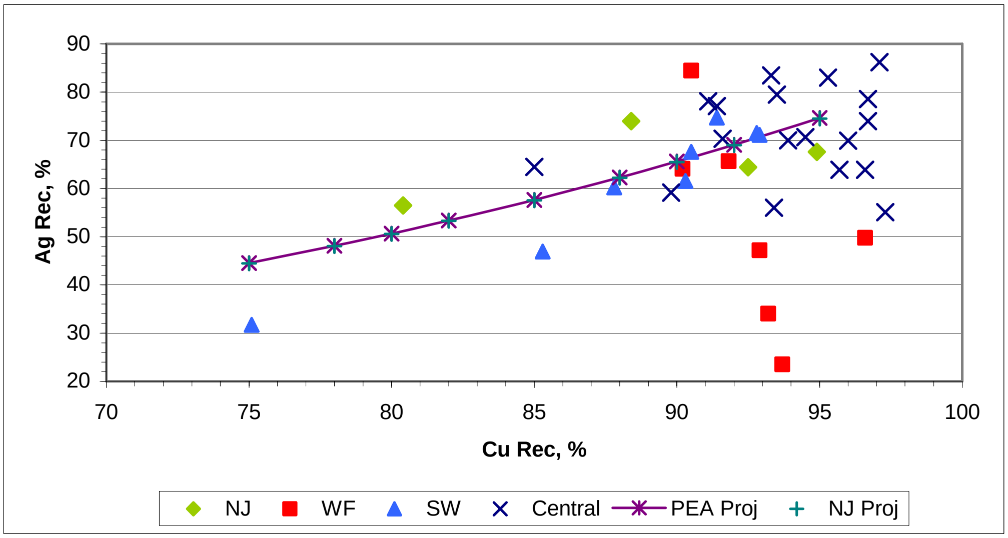


Figure 18-17: Silver Recovery versus Copper Recovery – Central

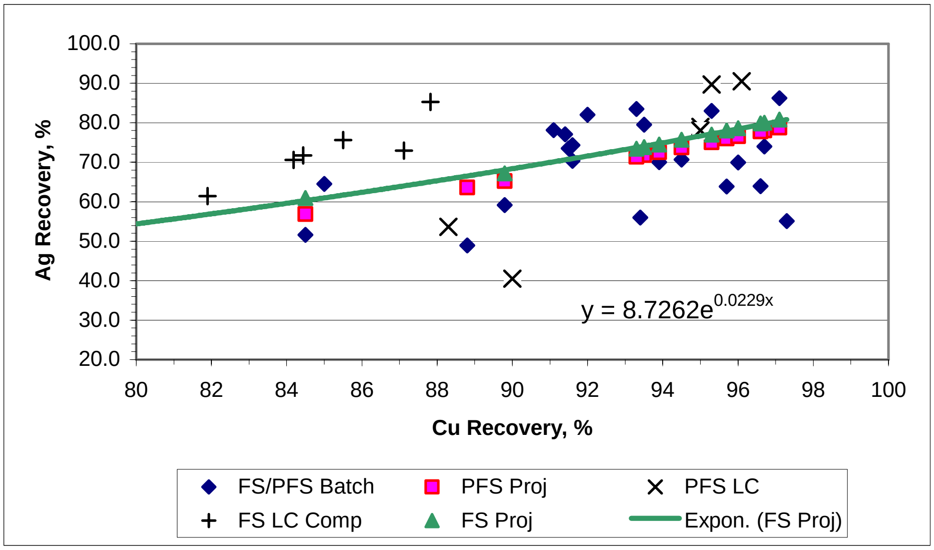
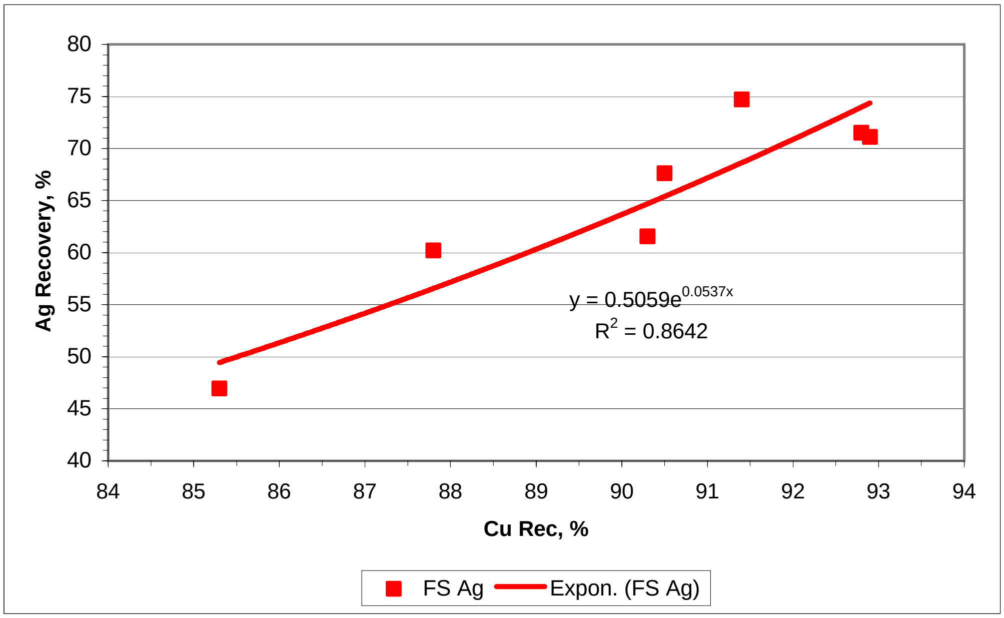
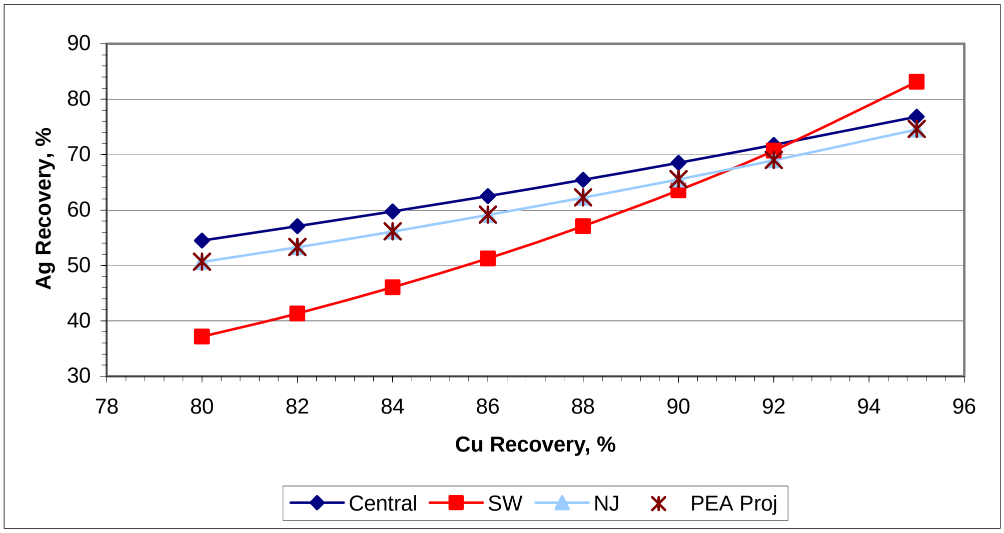


Figure 18-18: Silver Recovery versus Copper Recovery – SW



The relative correlations between the four pits are shown in Figure 18-19.

Figure 18-19: Silver Recovery versus Copper Recovery Projection - All Pits



18.5.6 Ancillary Studies

Limited ancillary studies have also been conducted to investigate the variability in magnetics content in the mineralized zones and the lead and zinc contents in the concentrates.

18.5.6.1 Magnetic Separation

Scoping magnetic separation tests using a Davis Tube have been made on four variability samples from the Central Pit (NGL zone), Southwest Pit and North Junction Pit to determine the amount of magnetics in the ore. The results have been tabulated in Table 18-13.

The results showed a range of magnetics content from trace amount to almost 10%. Further investigation over the deposit would be needed in the future to determine if opportunities exist for the addition of a magnetite recovery circuit.

Table 18-13: Magnetic Separation

Pit	Drill Hole	Magnetics
Central - NGL	GC05-0512	4.6
Central - NGL	GC05-0595	3.8
Southwest	GC05-0628	9.0
N. Junction	GC05-0567	0.1

18.5.6.2 Lead and Zinc Deportment

Lead and zinc have been identified in various parts of the deposit. The presence of these metals in the concentrate could impact the marketability of the concentrate. Consequently, selected concentrate and flotation tailings samples were analysed to scope the extent of occurrences and deportment of these metals. The results have been tabulated in Table 18-14.

The results indicated relatively low levels of these metals in the concentrates, except for the samples from the central replacement zone in the Central Pit that contained 1% to 2% zinc. Analysis of more samples is required to assess the occurrence of zinc, particularly in the Central Pit.

As shown in Table 18-14, a rough metals balance indicates that the recovery of lead and zinc in the concentrate would be relatively high at over 85%. Further work is required to determine if their recovery is due to their direct flotation, locking with copper sulphides, or by entrainment.

Table 18-14: Lead and Zinc Deportment in Concentrate

Pit	Drill Hole	Assay (%)				Recovery to Concentrate (%)	
		Concentrate		Tailings			
		Pb	Zn	Pb	Zn	Pb	Zn
Central - NGL	GC05-0571	0.05	0.19	0.003	0.007	94	96
Central - NGL	GC05-0571	0.17	0.43	0.004	0.008	98	98
Central - CRZ	GC05-0568	0.11	1.92	0.003	0.030	97	98
Central - CRZ	GC05-0568	0.18	0.95	0.007	0.031	96	97
Central - SGL	GC05-0711	0.02	0.12	0.003	0.019	86	84
West Fork	GC05-0526	0.04	0.18	0.005	0.018	89	90
Middle Creek	GC05-0516	0.05	0.37	0.004	0.021	92	94
Middle Creek	GC05-0545	NA	NA	0.008	0.020	93	94

18.6 2005 Phase III Program

The objectives of the Phase III pilot plant program were to generate sufficient quantities of tailings and water samples for environmental studies, concentrate and tailings samples for thickening and filtration tests, and concentrate for marketing purposes. The pilot plant operation was not optimized for metallurgy to support any laboratory results but data was collected to provide some indications of metallurgical performance in a continuous operation.

The following three bulk composites were run through the pilot plant.

- PP Composite 1 – 2004 assay rejects and drill cores
- PP Composite 2 – 2005 drill cores (similar to Composite 4 in Phase I program)
- PP Composite 3 – 2005 drill cores (similar to Composite 3 in Phase I program)

PP Composite 1 was used to calibrate the pilot plant while PP Composites 2 and 3 were used to generate the samples for the various studies.

The environmental test results are contained in a separate report by Rescan Environmental.

18.6.1 Metallurgical Data

The pilot plant was run ahead of laboratory tests due to the urgency to generate samples for the auxiliary studies. As such, it was not run to confirm any laboratory data. Sampling and analysis, however, were carried out to provide indications of validity of the operation and samples for the studies. The pilot plant performance has been summarised in Table 18-15. The results from the subsequent locked cycle tests on the same samples have also been included for comparison.

The pilot plant copper recovery was reasonable compared to the locked cycle results. The gold recovery, however, was significantly lower than the locked cycle tests. It is not known if this was due to the significantly lower gold contents in the pilot plant feeds and slight differences in grinds and reagent usages.

Table 18-15: Pilot Plant Metallurgical Performance

	Sizing P80	Assay (% or g/t)			Recovery (%)		
	Microns	Cu	Au	Ag	Cu	Au	Ag
Pilot Plant							
PP Composite 3							
Flotation Feed	129	0.79	0.58	7	100	100	100
Concentrate	38	25.4	13	142	87	62	52
PP Composite 2							
Flotation Feed	121	0.67	0.61	6	100	100	100
Concentrate	38	30.4	20.6	197	91	67	64
Locked Cycle							
Composite 3							
Flotation Feed	195	0.8	0.86	7	100	100	100
Concentrate	33	28.1	22.8	137	93	78	67
Composite 4							
Flotation Feed	192	0.76	0.97	7	100	100	100
Concentrate	42	27.4	31.2	182	94	83	68

The analysis of the concentrate composite produced for marketing purposes from ore Composite 2 and 3 is shown in Table 18-16.

Table 18-16: Pilot Plant Concentrate Composite for Marketing

Element	Unit	Assay
Copper	%	29.9
Gold	g/t	15.9
Silver	g/t	130
Iron	%	25.4
Antimony	%	0.011
Arsenic	g/t	<10
Bismuth	g/t	35
Chlorine	g/t	<100
Cobalt	%	0.010
Fluorine	g/t	533
Lead	%	0.059

Element	Unit	Assay
Magnesium Oxide	%	0.24
Mercury	g/t	<1
Nickel	g/t	30
Palladium	g/t	1.1
Platinum	g/t	0.05
Rhodium	g/t	2.2
Selenium	g/t	153
Silica	%	2.70
Sulphur	%	32.5
Tellurium	g/t	<20
Zinc	%	0.29

18.6.2 Concentrate and Tailings Thickening

Dynamic settling tests on concentrate and tailings were conducted by Outokumpu personnel on fresh samples at G&T Laboratory to determine flocculant requirements and for thickener sizing. Tests included flocculant screening and simulation of a high rate thickener. The optimum conditions indicated by the test work are summarized in Table 18-17.

It was observed in the tests that the concentrate tended to form a froth layer due to air entrainment from the flotation process. This would impact overflow clarity (quality of recycled process water) and capacity of the thickener. It is recommended that a de-aeration system be considered for plant operations to avoid these potential restrictions.

Table 18-17: Concentrate and Tailings Thickening

	Concentrate	Tailings
Feed Density, % solids	11	15
Flocculant Dosage, g/t	17	7
U/flow Density, %	68	68
Rise Rate, m/h	2.1	5.2
Unit Area, m^2/tpd	0.167	0.047

18.6.3 Concentrate Filtration

Concentrate samples were sent to two filter vendors, Larox and Metso, for filtration tests and filter sizing. The projected filter performance and sizing have been summarized in Table 18-18.

In both filter technologies, extended air dry time was required to achieve the target 8% cake moisture due to the fineness of the concentrate.

Table 18-18: Concentrate Filtration

	Units	Larox	Metso
Sizing P80	**Microns**	**40**	**39**
Feed Density	% solids	60	59
Cycle Time:			
Feeding	min	1.5	1.5
Pressing	min	1.5	0.5
Air Blow	min	5.0	6.0
Misc	min	4.5	4.5
Total	min	12.5	12.5
Cake Thickness	mm	40	33
Cake Moisture	%	8	8

18.7 Grindability Studies

Drill core samples were provided to SGS MinnovEX and SGS Lakefield to assess the grinding characteristics of the ore from the various pits and between the "broken" and "stick" ores, and to design the grinding circuit by two different technologies. SGS MinnovEX used their SPI/CEET technology while SGS Lakefield used the JKSimMet technology for circuit design.

During the PEA study, 14 samples were tested by SGS MinnovEX for a preliminary grinding circuit design. A further 92 drill core samples have been tested during the Feasibility Study. The results from both studies have been incorporated to produce the final mill design. As part of the feasibility study, 16 drill core samples were also tested by SGS Lakefield to provide a comparative mill design by the JKSimMet technology.

Mill designs were generated for the target grinds of 80% passing 150 microns and 200 microns for a blend of the overall deposit. The blend, based on the relative abundance from the various pits, consisted of Central (78%), West Fork (2%), North Junction (7%), and Southwest (13%). The key mill design parameters from both technologies are summarized in Table 18-19.

The SAG and ball mill sizing and the power requirements are comparable between the two technologies.

Table 18-19: Mill Design Summary

		Target P80 150 Microns		Target P80 200 Microns	
		CEET	JKSimMet	CEET	JKSimMet
One SAG Mill					
SAG Nominal Size	Ft	40 ft x 20 ft	40 ft x 20 ft	40 ft x 20 ft	40 ft x 20 ft
Design Ball Charge	%	14	13	14	13
Mill Charge	%	25	25	25	25
Mill Speed	% critical	75	78	75	87
Power Draw	KW	19,300	19,300	19,300	19,300
Two Ball Mills, Each					
Ball Mill Nominal Size	Ft	26 ft x 40.5 ft	26 ft x 39.5 ft	25 ft x 39.5 ft	25 ft x 36 ft
Design Ball Charge	%	35	30	35	30
Mill Speed	% critical	75	75	75	75
Power Draw	KW	15,005	14,200	12,521	11,325

Based on the mill simulation, a SABC circuit consisting of a 40 ft (12.2 m) x 24 ft (7.3 m) SAG mill, two 26 ft (7.9 m)x 36 ft (11 m) ball mills and two 600kW pebble crushers has been designed to process an average of 65,000 tpd of ore. Mill power draws would be 19,300 kW at the SAG mill and 15,000 kW at each ball mill. The mills have been sized to be able to grind to a P80 of 150 microns.

The West Fork ore was the hardest as measured by SPI and Bond Work Index. Similar observations were obtained by G&T on a different set of West Fork core samples.

In terms of the Bond Ball Mill Work Index, SGS MinnovEX reported higher values than G&T. The simple average of the overall deposit was determined as 15.9 kWh/t by G&T and 17.5 kWh/t by SGS MinnovEX. For the dominant Central Pit (approximately 80% of total ore body), the average hardness was determined to be 16.0 kWh/t by G&T and 17.1 kWh/t by SGS MinnovEX. The reasons for the difference are not known. An average of 16.5 kWh/t between the two measurements have been selected for this study.

The SGS MinnovEX data showed that the “stick” ore was generally harder than the “broken” ore based on the SPI. The “stick” ore was also generally more abrasive than the “broken” ore. Their model indicated that the SAG mill would be limiting when processing ores with SPI greater than 115 minutes for both 150 microns and 200 microns target grinds. There were “stick” and “broken” samples that exhibited SAG mill limiting hardness.

18.8 SGS MinnovEX Flotation Simulation

Seven drill core samples from Central, West Fork, North Junction and Southwest pits have been tested by SGS MinnovEX using open circuit rougher-cleaners. The tests measured the flotation kinetics of the mineral species which were then used to model the flotation circuit.

The details of this program are contained in the SGS MinnovEX report.

Compared to the SGS MinnovEX Database of porphyry deposits, the Galore copper sulphide flotation kinetics were faster than the median, and Central and West Fork were in the fastest 5% of the database.

Gold in all the pits floated as fast as the top 5% in the gold database. The size-selectivity-recovery correlations indicated that a large portion of the gold occurs with copper sulphide minerals and that the gold losses are in the fine sizes.

Pyrite flotation kinetics was variable. The pyrite in the Southwest and North Junction pits were highly activated and floated fast with high recovery whereas the West Fork pyrite appeared depressed.

SGS MinnovEX also ran simulations to compare the performance of the design flowsheet using all mechanical cleaner cells with their proposed flowsheet which incorporates columns in place of the second and third stages of mechanical cells.

The simulation of the design flowsheet indicated that the proposed circuit would provide the required metallurgical performance and that additional first cleaner capacity might improve copper recovery in the cleaner circuit. The simulation of their proposed circuit with the column cells indicated that it would improve the concentrate grade slightly at the same copper recovery.

The simulations indicated similar gold recovery from both flowsheets.

19. Mineral Resource and Mineral Reserve Estimates

Section 19.1 is extracted from the technical report "Updated Galore Creek Mineral Resources, Northwestern British Columbia", posted to SEDAR on 12 Sept. 2006. This report was prepared by Mr. Michael J. Lechner, RPG from Resource Modeling Incorporated. To Hatch's knowledge there have been no changes since this issue.

19.1 Mineral Resources

Section 19.1 was prepared by Mr. Michael J. Lechner,RPG. Resource Modeling Incorporated

19.1.1 Summary and Conclusions

Copper, gold and silver are the principal product metals for Galore Creek. Metal concentrations are broadly controlled by lithology and vary over the extent of the drilled area.

Estimation of copper, gold and silver grade requires deterministic grade or rock type zones. Copper, gold and silver were modeled using a three-pass ordinary kriging approach with outlier restriction. One-pass nearest neighbor estimation was also implemented for validation purposes. Acid soluble copper grades were estimated with a subset of the assay database using inverse distance cubed methods. Grade shell models were constructed by project geologists at a nominal 0.35 copper equivalent (CuEQ) cutoff in order to constrain mineralization in areas (North Junction, Junction, Middle Creek, Southwest Zone and West Fork) where the relationship between mineralization, structure and lithology has been elusive. In the Central Zone (South Gold Lens, Bountiful, Central Replacement Zone, and North Gold Lens) rock types were grouped based on statistical and genetic criteria into three groups: volcanic, mineralized intrusive, and intrusive.

Validations proved the model to be acceptable for resource reporting and mine planning. Copper, gold and silver grades appear to be well-behaved spatially, thus increasing the predictability of the estimation model.

19.1.2 Recommendations

Geologic study should continue to isolate the controls to mineralization in the West Fork, Southwest Zone, Middle Creek, Junction and North Junction areas. The cutoff grade used to design grade envelopes that are used to constrain the estimate of block grades should be lowered to reflect economic cutoff grades.

19.1.3 Introduction

The 2006 Resource Model workflow and activities for Cu, Au and Ag grade estimation were:

- Exploratory Data Analysis
  - Histograms
  - Boxplots
  - Contact plots

  - Grade variography
- Modeling:
  - 5 m down-hole assay compositing
  - Three-pass kriging
  - One-pass nearest neighbor estimation
  - Swath plots
  - Histograms
  - Volume variance adjustment

19.1.4 Coordinate System

The coordinate system used for resource modeling is truncated UTM. The project coordinates were calculated as follows:

Easting = UTM Easting – 6,300,000

Northing = UTM Northing – 300,000

19.1.5 Topographic Data

The initial digital elevation model (DEM) for the project was generated by Eagle Mapping as contracted by Kennecott Minerals in 1991 from government issued aerial photos flown in the 1950's. The survey control for these photos was based on an historical iron pin located 800 metres west of the Central Zone by traditional transit and plumb-bob survey methods and was tied into pre-existing control points in Telegraph Creek and Dease Lake.

In August of 2003, NovaGold contracted Eagle Mapping of Vancouver, B.C. to acquire new aerial photography and to generate a more accurate DEM file for the project. Survey control for the aerial photography was placed as visible crosses by project personnel using an Ashtech DGPS system. The aerial photography was taken at a resolution of two metres using a single frequency DGPS for control. The resulting DEM surface was different in elevation and accuracy from the historically generated topography.

The 2003 version of the topography was higher in elevation than the previous topographic surface. When the historic drill hole collar elevations were compared to the 2003 topographic surface it was seen that the difference in elevation was found to be somewhat random. The average difference between the drill hole collars and the 2003 topographic surface was 24.2 metres. To account for this in the 2003 scoping study analysis, the new topographic surface was lowered by the average difference (24.2 m). Drill hole collars were then left either sticking above the topographic surface or "collared" below the surface. Since the construction of three-dimensional geologic shapes for resource modeling had already been initiated, this issue was handled by appropriate coding of the data. Drill hole composites were assigned a zone code based on the location of the composite midpoint located. To insure that the drill hole composite codes matched the lithological units in the drill logs, the composites were coded for the bedrock and surface interfaces according to the original surveys.

Before updating the resource model in 2004, the drill holes were registered to the topographic surface and the data were moved back up 24.2 metres to the elevation of the original 2003 photo and DEM survey results which are thought to be more accurate. At this point, all of the geologic shapes were started from scratch with the newly registered drill holes and upwardly adjusted topography.

On October 3rd, 2004 a higher resolution, 1-metre aerial photo set with dual-channel DGPS was flown for Rescan Environmental Services by Eagle Mapping for the Galore Creek project. The control point used for the aerial photography was set by Peter Walcott of Walcott and Associates. Walcott, a registered land surveyor, noted that the 2003 in-house surveying had not accounted for a provincial datum correction related to the NAD 83 conversion. The 2004 DEM showed a -15 metre difference from the increased accuracy of the control work that generated it.

The modeling and resource estimation for the 2005 Preliminary Economic Assessment did not use the new surface as it was not received from Eagle Mapping until late January 2005, after the estimation process and pit modeling had already begun.

To align the project objectives, insert the correct elevations, and use the best DEM surface, a final elevation adjustment was made to the NovaGold MineSight projects on May 8th, 2005 to lower the 2004 topography by 15 metres. Notification to all parties was made and a final datum was distributed to all Feasibility Study participants.

In October of 2005 a registered professional land surveyor, Peter Thomson BCLS CLS, confirmed the accurate locations of the control points used to provide survey control of the DEM and air photos and that no significant differences were found in the XY coordinates; however, a difference of about one-metre was determined in the elevations. No adjustments were made to the digital elevation model or base station control points based on his findings.

Mr. Thomson utilized a dual frequency survey grade Trimble 4700 GPS receiver. Both static observation techniques and RTK (real time kinematic) techniques were employed. Processing of the results was performed using Trimble Geomatics Office software.

The primary control for this confirmatory survey was provided by the Geodetic Survey of Canada station 75C134, which is approximately 7 kilometres south of the Bob Quinn airstrip along Highway 37, which was used as the origin of coordinates and elevations. The primary control was extended into the vicinity of Galore Creek using static observation techniques. This was confirmed by looping the control survey back to Highway 37 by another route, again using static observation techniques. The closure obtained was 0.02 metres horizontal and 0.04 metres vertical. An additional check was performed by processing 8 hours of data on Station 268 with the Precise Point Positioning service of the Geodetic Survey of Canada.

19.1.6 Specific Gravity Data

19.1.6.1 Sample Data

The current Galore Creek database contains specific gravity (SG) determinations for a total of 8,855 samples. These determinations were made between 1963 and 2005 by a variety of companies. The majority of the determinations were made using the water displacement method (i.e. samples weighed in air and in water). In addition to the "conventional" SG determinations several other methods were used by NovaGold including samples collected from triple-tube core barrels (355) and samples submitted for waxed analysis at ALS Chemex (50) in 2005. Elimination of suspect or erroneous SG determinations reduced the database to a total of 8,331 records. Table 19-1 summarizes the number of samples that have been collected by year, by company, method and the arithmetic average SG value.

Table 19-1: Specific Gravity Determinations by Year

Year	Company	Number	SG	Location	Method - Notes
1961	Kennecott	3	2.41	On site	Water displacement method - DDH core
1962	Kennecott	65	2.62	On site	Water displacement method - DDH core
1963	Kennecott	222	2.63	On site	Water displacement method - DDH core
1963	Stikine	7	2.57	On site	Water displacement method - DDH core
1964	Kennecott	205	2.68	On site	Water displacement method - DDH core
1964	Stikine	3	2.65	On site	Water displacement method - DDH core
1965	Kennecott	30	2.64	On site	Water displacement method - DDH core
1965	Stikine	279	2.64	On site	Water displacement method - DDH core
1966	Stikine	70	2.67	On site	Water displacement method - DDH core
1972	Stikine	13	2.66	On site	Water displacement method - DDH core
1976	Stikine	5	2.53	On site	Water displacement method - DDH core
1991	Kennecott	50	2.72	On site	All '91 samples from 1 hole + some older samples
2003	NovaGold	2	2.77	ALS Chemex	Only two samples
2004	NovaGold	76	2.78	ALS Chemex	Includes West Fork area - 355 split-tube samples
2005	NovaGold	7,301	2.62	On site	Water displacement method - DDH core
Grand Total		**8,331**	**2.62**		

Prior to NovaGold's involvement in the project there were approximately 957 SG determinations made from samples collected from the Galore Valley. These samples averaged 2.658 g/cm^3. Samples collected from the 2005 field season have greatly increased the number and distribution of samples taken across the deposit. With the inclusion of many more samples, the overall SG was reduced by 1.4% to 2.62 g/cm^3. The reduction may be due simply to the increased density of sampling as it is unlikely that there are large data quality problems with the pre-2005 data, excluding the random data entry issues mentioned above. Sampling methods during the 2005 program were recommended by 2004 Qualified Person Bob Morris and carried out with the oversight of senior field staff. The weight of unbroken pieces of core less than 15 cms long was determined both in air (dry) and in water. Hard tap water was used for the measurement. Samples were taken at the rate of one sample for approximately every 10 metres of drilling by the geotechnical staff during the core photography process. Results were written on data entry sheets and entered by a data entry clerk into the SG-Point load Access database. Table 19-2 summarizes the number of samples and mean SG value by mineralized area.

Table 19-2: Specific Gravity Determinations by Area

Area	Number	Mean SG
Butte	93	2.58
Central Zone	3,847	2.64
Gap	11	2.63
Grace	247	2.78
Junction	33	2.64
Middle Creek Zone	405	2.61
North Junction Zone	307	2.63
North Rim Zone	151	2.64
Profit Zone	111	2.58
Reconn	341	2.70
Saddle	101	2.61
Southwest Zone	1,753	2.56
West Fork Zone	918	2.62
West Rim	13	2.67
Grand Total	**8,331**	**2.62**

There is some variability in the SG data ranging from 2.78 in the Grace area to 2.56 at the Southwest Zone, an 8.6% difference. Interestingly, the data shows a general increase in SG values from south to north across the deposit. The low value of the SW Zone is a bit surprising considering increased pyrite associated with the mineralization but its value is confirmed by the low average values encountered in the Profit area and the gap which are located just east and NE of the Southwest Zone. The lack of significant oxide in these areas may contribute to the low value. These areas are dominated by intrusive rocks and lesser breccia with strong K-spar alteration. The high values in the Grace area are likely due to increases in mafic content of the volcanic/sedimentary rock mass.

19.1.6.2 In-situ Rock Disaggregation

Core logging indicates that the all unoxidized rocks contain common veinlets of anhydrite. These veinlets are open spaces in rocks within the zone of oxidation. NovaGold has studied the SG data to determine the impact of disaggregation resulting from the removal of anhydrite, commonly referred to by NovaGold as “broken rock”, on bulk density. Rock below the level of disaggregation is referred to as “stick rock” as the core typically comes out of the core barrel in long, competent sticks.

A comparison of the SG data both above and below the disaggregation surface is shown in Table 19-3.

Table 19-3: Specific Gravity Determinations vs. Disaggregation Surface

Location	Number	Mean SG
Above	4,501	2.60
Below	3,830	2.64
Grand Total	**8,331**	**2.62**

As shown in Table 19-3, material above the “broken” – “stickrock” surface is about 1.5% lighter than the material below the contact. Although this clean break of the data has less resolution than in the review of SG data by depth from surface, the results are still correlative with those observations, in that SG increases with depth. However, this does account for the expected difference in values due to the extreme fractured/broken nature of the rocks across this surface. SG samples taken above the broken rock surface were limited to whole rock pieces within broken material and cannot adequately characterize the bulk density of these rocks.

For the 2005 PEA a factor of 5% was used to further reduce the density of material above the stick rock surface to more accurately reflect the differences in the bulk density of the broken material. This factor was applied based on the experience of other properties. NovaGold staff could not find any specific gravity data that may have been derived from the bulk sample that was collected from the shallow adits or from a test bench that is believed to have existed in the South Gold Lens.

In 2005 NovaGold examined specific gravity values in the disaggregation zone by collecting data by two separate methods: test pits and split tube core measurements. Small test pits were constructed across the exposed and backhoe accessible portions of the deposit in the West Fork and the South Gold Lens areas for the express purpose of collecting SG data. Procedures used for the test pitting were as follows:

- An area of ground was leveled by the backhoe and a 1/2m by 1/2m by 1/2m pit dug.
- Where possible the pits were located near drill hole collars for comparative purposes.
- Rock excavated from the pit was placed on a tarp and weighed.
- The pit was lined with plastic and filled with water to the original height of the rock as the water volume equals the volume of the rock and the void space.
- Specific gravity was then computed by dividing the weight of the excavated rock by the weight of the equivalent volume of water.

Specific gravity determinations based on the pit method are summarized in Table 19-4.

Table 19-4: Specific Gravity Values From Test Pits

Test site	Easting	Northing	Wt (kg)	Vol water(litres)	SG (g/cm^3)	Comment
SG-001	350,953	6,333,551	850	400	2.13	
SG-002	350,589	6,332,788	102	45	2.27	
SG-003	350,550	6,332,806	132	95	1.39	
SG-004	350,694	6,332,952	146	65	2.25	
SG-005	350,903	6,333,436	119	65	1.83	
SG-006	350,896	6,333,644	94	50	1.88	
SG-007	350,810	6,333,722	65	50	1.30	Rock very broken and dirty
SG-008	350,849	6,333,993	63	35	1.80	
SG-009	350,855	6,334,062	59	35	1.69	
SG-010	350,867	6,334,039	39	15	2.60	Rock quite competent
Mean					**2.01**	

The SG values generated from the test pits are significantly lower than other determinations. Results from the test pit program are likely biased low due to several reasons:

- The inability to hold the integrity of the pit wall caused sloughing not attributable to the original volume of the excavation.
- Determination of what the exact water level should be in the pit was difficult.
- The plastic used to line the pit did not conform to the sides as required to accurately reflect the void created during excavation.
- The methods that were used in obtaining bulk density measurements are considered too crude to be accurate enough, and the sample sites too few to apply these data for consideration as a factor in reduction of the SG for the broken rock material.

A second method of sampling the bulk density of the broken rock consisted of measuring the weight of the split-tube run from the triple-tube geotechnical drilling. The method of collecting the samples was as follows:

- Split tube is hydraulically removed from the casing at the completion of a drill run.
- A precise core recovery is measured and excess water is drained.
- The weight of the split tube with core intact is measured by a hanging spring scale.
- The known weight of the split tube is subtracted from the measured weight with core.
- The volume of the rock contained in the run is calculated using an average core diameter and the length of the recovered core.
- The adjusted weight is divided by the calculated volume for the determination of specific gravity.

Due to the late initiation of this measurement method, only three holes were measured in 2005. The results from these determinations are summarized in Table 19-5.

Table 19-5: Specific Gravity Values From Split Tube Core Barrel

Drill Hole	Location	GT Measured Interval (m)	Mean SG (g/cm3)	% Rec	Relative % diff SG
GC05-637	SGL	23-124	2.28	95.7	
GC05-637	SGL	124-292	2.49	100.0	8.80%
GC05-651	SW	8-83	2.33	94.8	
GC05-651	SW	83-252	2.57	100.0	9.80%
GC05-660	East Cent	19-199	2.28	88.7	
GC05-660	East Cent	199-401	2.58	100.0	12.30%
Average					**10.30%**

The method used does not account for what has not been recovered in the drill run. This could be void space or clay/sulfate/gravel infilling that has been washed away during the drilling process. The recovered core does represent what is thought to be typical broken rock and is therefore a reasonable representation. Additionally, it was not possible in the course of drilling to dry the sample being measured and as such the method used does not account for un-drained water that may have existed in pore space before the weight measurement.

Despite that the measurements taken may be questioned as to the actual representation of bulk density of the broken rocks, the results are consistent with what would be expected geologically. However, the sampling demonstrated excellent value in a comparative sense as the entire length of each of the three tested holes was measured. The relative percent difference SG between the broken and stick rock intervals proxies as a factor to be analyzed for use in the 2006 feasibility tonnage calculations.

Using data from each of the three holes as a factor to reduce the SG values in the broken zone would result in a 10.3% reduction, more than double the 5% reduction that was used in the 2005 PEA. Restricting the data to the conceptual pit shapes brings the factor to the recommended 9.3% SG reduction above the broken zone – stick rock surface.

19.1.6.3 Waxed vs. Un-waxed Core Determinations

All historical SG measurements have been made without a wax coating which could result in slightly heavier SG readings due to entrapped moisture in voids or fractures. During the 2005 season a set of 50 randomly distributed samples were submitted to the ALS Chemex Laboratory in Vancouver, B.C. in order to compare those results with the previous data. The average SG of the 50 waxed core samples was 2.65 g/cm^3, which is about 1.1% higher than the average of the un-waxed SG determinations. In the opinion of the RMI, 50 samples may be too few to be statistically representative. However, there is a close comparison between the waxed and unwaxed determinations.

19.1.6.4 Moisture Content

Historically all SG measurements at Galore Creek have been completed with core samples that have not been dried in an oven and therefore the question of latent moisture in pore space affecting the bulk density must be accounted for. The moisture content of the deposit has been measured in rock samples submitted for metallurgical study at G&T Metallurgical Labs. A variety of near surface “broken rock” samples (48 samples) were collected from representative rock types throughout the deposit and analyzed for moisture content. The moisture content for these samples ranged from 0.8% to 4.29%. The average moisture was 0.89%. No statistical trends were found relative to association with deposit area or rock type.

19.1.6.5 Block Model SG Assignment

Specific gravity values were assigned to the block model by area, mineral zone, and lithology. Table 19-6 and Table 19-7 summarize the bulk density (SG) values that were assigned to the block model.

Table 19-6: Specific Gravity Assigned by Mineral Zone

Area	Specific Gravity (g/cm^3)	
	Outside Minzone	Inside Minzone
Junction	2.66	2.67
Middle Creek	2.61	2.62
North Junction	2.65	2.61
Southwest Zone	2.56	2.57
West Fork	2.57	2.59
Upper Opulent	N/A	2.59
Opulent	N/A	3.61

Table 19-7: Specific Gravity Assigned by Lithology Group

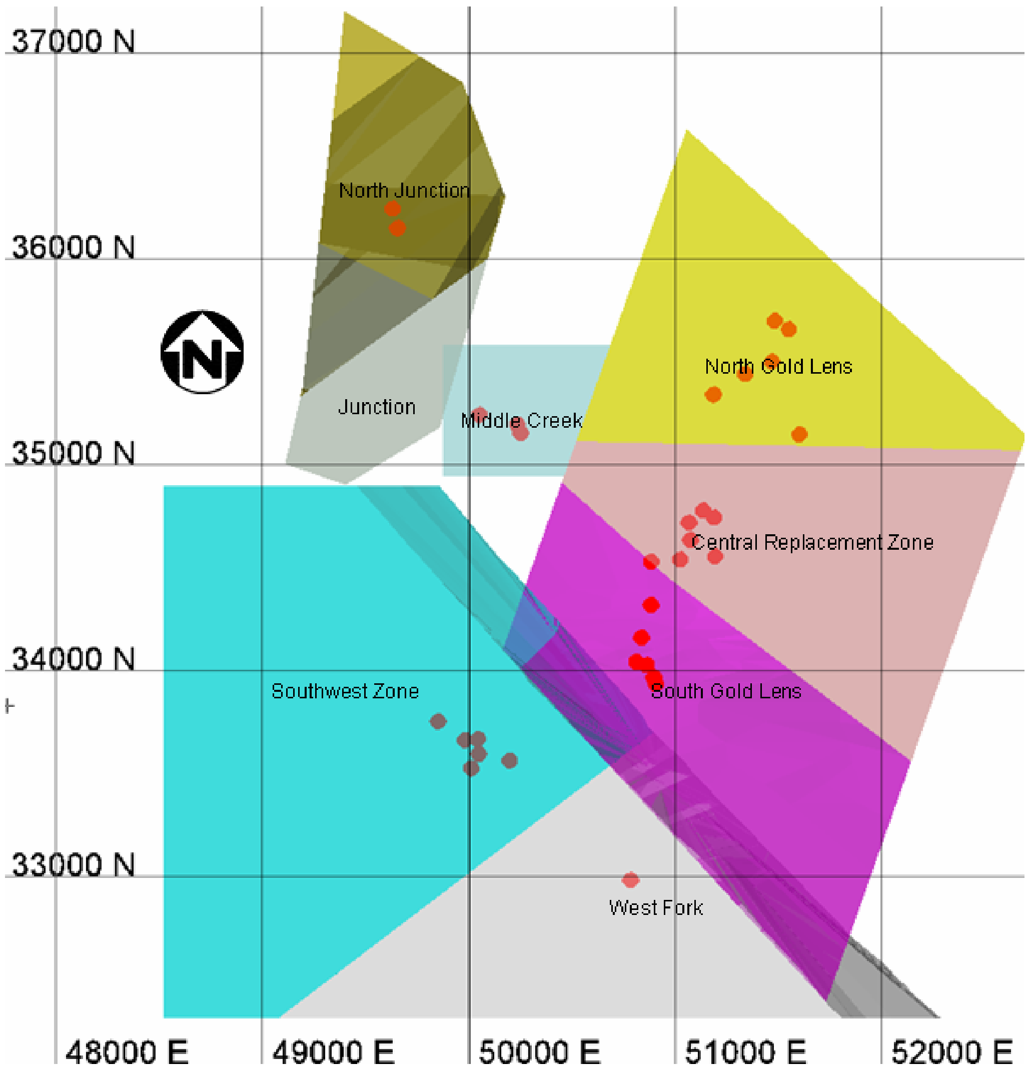
Lithologic Unit	SG (g/cm^3)
Volcanic Group	2.66
Mineralized Intrusive Group	2.57
Intrusive Group	2.59

The SG values that were loaded to the block model were adjusted for disaggregation and moisture content to arrive at the final bulk density values. SG values above the broken rock – stick rock surface were reduced by 9.3%. SG values below the broken rock – stick rock were reduced 0.5% to account for moisture content.

19.1.7 Acid Soluble Copper Data

Based on the current drill hole assay data, acid soluble copper is irregularly distributed in the near surface environment of Galore Creek. As of the date of this resource estimate, 916 acid soluble assays have been obtained from 31 drill holes. All near surface mineral zones have had samples analyzed for soluble copper except the Junction Zone. Figure 19-1 shows the distribution and location of the 31 drill holes that were analyzed for acid soluble copper relative to the mineral zones.

Figure 19-1: Distribution of Acid Soluble Copper Assays



There is a very strong correlation between total copper, acid soluble copper and their depth below the topographic surface. These relations are shown in Table 19-8, which shows that acid soluble copper grades decrease with increasing depth.

Table 19-8: Acid Soluble Copper (%) vs. Depth

Depth (m)	Central Replacement Zone	Middle Creek Zone	North Gold Lens	North Junction	South Gold Lens	Southwest Zone	West Fork Zone
0-15	0.11	0.61	0.19	0.21	0.27	0.22	0.03
15-30	0.03	0.65	0.14	0.10	0.22	0.07	0.05
30-45	0.03	0.45	0.11	0.03	0.11	0.05	0.04
45-60	0.10	0.37	0.05	0.03	0.08	0.05	0.04
60-75	0.06	0.21	-	0.01	0.03	0.02	0.06
75-90	0.02	0.23	-	0.03	0.03	0.47	0.06
90-105	0.01	0.25	-	0.01	0.03	-	0.03
>105	0.04	0.05	-	0.01	-	-	0.03

Acid soluble copper grades are particularly high in the Middle Creek area. The low acid soluble copper grades in the West Fork area may be related to having been covered by glacial ice until recent times.

19.1.8 Acid Soluble Copper Estimation

Acid soluble copper grades were estimated so that an "available" copper content (i.e. total copper less soluble copper) could be determined. Both total and acid soluble copper grades were estimated using only those drill holes that contained both total and acid soluble analyses. Grades were estimated using inverse distance weighting methods (cubed) using 5-metre-long assay composites. To preserve the trend of decreasing acid soluble copper grade with depth, the block model and composites were coded into 15 metre bins below bedrock, accomplished by repeatedly translating the bedrock surface in 15 metre increments. The total and acid soluble copper grade estimates were constrained to blocks where the block and composites were located in the same elevation ranges.

Available copper grades were then calculated using the following equation:

Available Copper = total copper (exhaustive) x (acid soluble copper / total copper)

where total copper (exhaustive) is the original estimated total copper block grade that was estimated using all available copper data as discussed in Section 19.15.

19.1.9 Evaluation of Extreme Values

Lognormal probability plots of assay grades inside and outside of grade shells, and by rock type in the Central Zone were examined. Most populations exhibit a lognormal grade distribution and caps on Cu, Au, and Ag assays were placed where significant deviation occurred. Grade capping levels are summarized in Table 19-9.

Table 19-9: Assay Capping Thresholds

Area	Grade Shell or Rock Type	Cu (%)	Au (g/t)	Ag (g/t)
North Junction	Inside of Grade Shell	7.0	7.0	50.0
	Outside of Grade Shell	1.2	1.5	9.0
Junction	Inside of Grade Shell	2.5	0.7	8.0
	Outside of Grade Shell	0.3	0.2	4.0
Middle Creek	Inside of Grade Shell	2.5	7.0	20.0
	Outside of Grade Shell	0.5	1.5	7.0
Southwest	Inside of Grade Shell Breccia	5.0	10.0	45.0
	Inside of Grade Shell Non-Breccia	3.0	5.0	11.0
	Outside of Grade Shell	1.8	7.0	30.0
West Fork	Inside of Grade Shell	3.6	6.5	50.0
	Inside of Opulent Shell	35.0	10.0	150.0
	Inside Upper Opulent Shell	2.0	2.0	15.0
	Outside of Grade Shell	1.0	2.6	25.0
Bountiful	V2 Rock Type	0.8	0.3	15.0
	V3 Rock Type	2.1	0.7	20.0
	I4 Rock Type	0.7	0.4	9.0
	I5 & I9 Rock Types	0.2	0.3	none
	I11 Rock Type	0.7	none	10.0
South Gold Lens	V3 Rock Type	3.0	0.8	30.0
	I4 Rock Type	3.0	2.0	9.0
	I8 Rock Type	0.5	0.3	10.0
	I5 & I9 Rock Types	0.6	0.7	6.0
	I11 Rock Type	0.5	0.8	6.0
Central Replacement	V1 Rock Type	0.1	0.1	1.0
	V2 Rock Type	3.5	2.5	30.0
	V3 Rock Type	5.5	3.0	40.0
	I4 Rock Type	3.0	0.3	20.0
	I5 & I9 Rock Types	4.0	1.5	20.0
	I10 Rock Type	2.5	0.9	17.0
	I11 Rock Type	3.5	1.0	20.0
	D2 & D3 Rock Type	2.0	1.0	13.0
North Gold Lens	V1 Rock Type	3.0	2.5	29.0
	V2 Rock Type	3.0	2.0	28.0
	V3 Rock Type	2.0	0.5	18.0
	I4 Rock Type	0.6	0.1	none
	I5 & I9 Rock Types	2.0	1.5	18.0
	I10 Rock Type	0.3	0.4	4.0
	I11 Rock Type	2.5	3.0	18.0
	D2 & D3 Rock Type	2.0	0.8	15.0
North Au-Bornite Domain	V1 Rock Type	3.5	20.0	40.0
	V2 Rock Type	4.5	10.0	30.0
	I5 & I9 Rock Types	2.0	4.0	15.0
	I10 Rock Type	1.0	7.0	15.0
	I11 Rock Type	3.0	20.0	12.0
South Au-Bornite Domain	V3 Rock Type	7.0	7.0	30.0
	I4 Rock Type	3.0	3.0	25.0
	I8 Rock Type	4.0	4.5	25.0
	I5 & I9 Rock Types	3.0	3.0	50.0
	I11 Rock Type	3	4	20.0

Michael Lechner,RPG of RMI has reviewed the grade capping strategy that NovaGold developed for each of the Galore Creek mineralized zones. In the RMI's opinion, NovaGold's capping thresholds for copper and gold are reasonable and have helped to minimize local over-estimation of block grades.

19.1.10 Composite Sample Grade Exploratory Data Analysis

Composite samples were generated down hole in nominal 5 metre lengths, generating 31,703 Cu composite samples within the resource estimation areas. The fewer Au and Ag composite samples result from not every historic rock sample having each metal assayed. Assay grades were capped as specified in Section 19.1.9 prior to compositing. Composites were not broken at geologic or grade shell boundaries. This is reasonable given the broad nature of mineralization and the proposed large-scale open-pit mining operation. Each composite was tagged with the majority rock type of the geological triangulated solid or grade shell. The composite file was inspected to ensure proper capping and composite calculation

19.1.11 Description of Composite Fields

Two composite files were used for grade estimation: galofin.cmp.isis was used to estimate all areas outside of the Central Areas and galofin.skm.isis was used in the Central Areas. Composite fields are listed in Table 19-10. Rock type codes and abbreviations used in modeling are listed in Table 19-11 and Table 19-17. Estimation areas are illustrated in Figure 19-2.

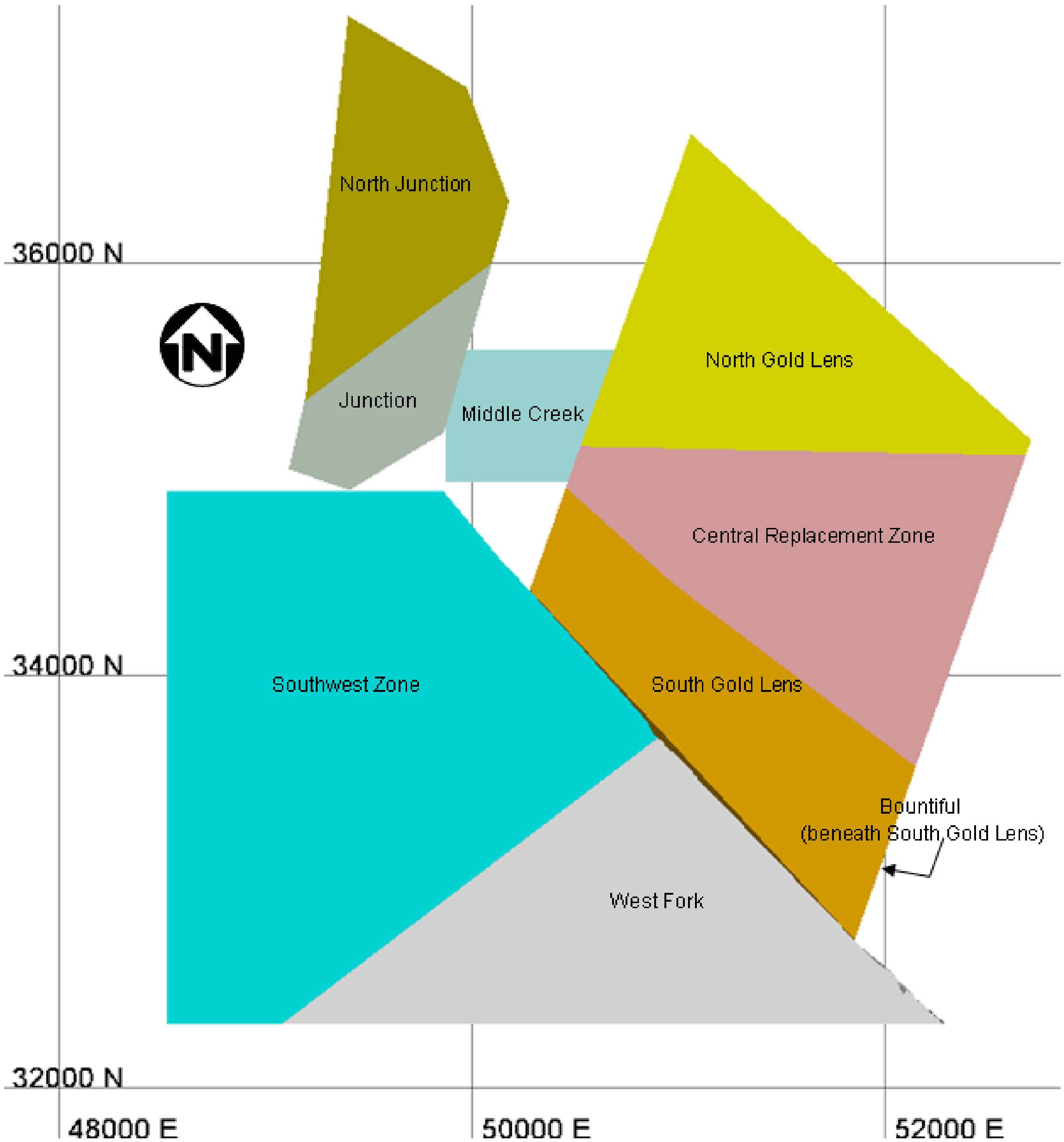
Table 19-10: Principle Composite Fields used for Modeling

Field Name	Description
MIDX	Mid-point point Easting
MIDY	Mid-point point Northing
MIDZ.	Mid-point point elevation
CUCAP	Copper grade
AUCAP	Back tagged rock type code
AGCAP	Variogram domain code
MINZON	Grade shell flag
MZ2	Interpolation pass code 2
GEOCOD	Interpolation pass code 3
TEXT	Area code
MZ3	Rock Group code used in Central areas

Table 19-11: Area Codes and Abbreviations Used in Modeling

Field Name	Abbreviation	Composite and Block Code
Bountiful	BOUN	1
Central Replacement	CRZ	2
Junction	JUNC	3
Middle Creek	MC	4
North Gold Lens	NGL	5
North Junction	NJ	6
South Gold Lens	SGL	7
Southwest Zone	SWZ	8
West Fork	WF	9

Figure 19-2: Area Domain Names



19.1.12 *Histograms and Probability Plots*

A series of histograms and probability plots were generated using 5-metre copper, gold and silver composites to characterize the grade distribution of each grade shell and rock type. General histogram and probability plots, regrouped Central Zone rock types, and grade shells are shown in Figures 19-3 to 19-5.

Figure 19-3: Histogram and Probability Plot for All Capped 5m Cu Composites

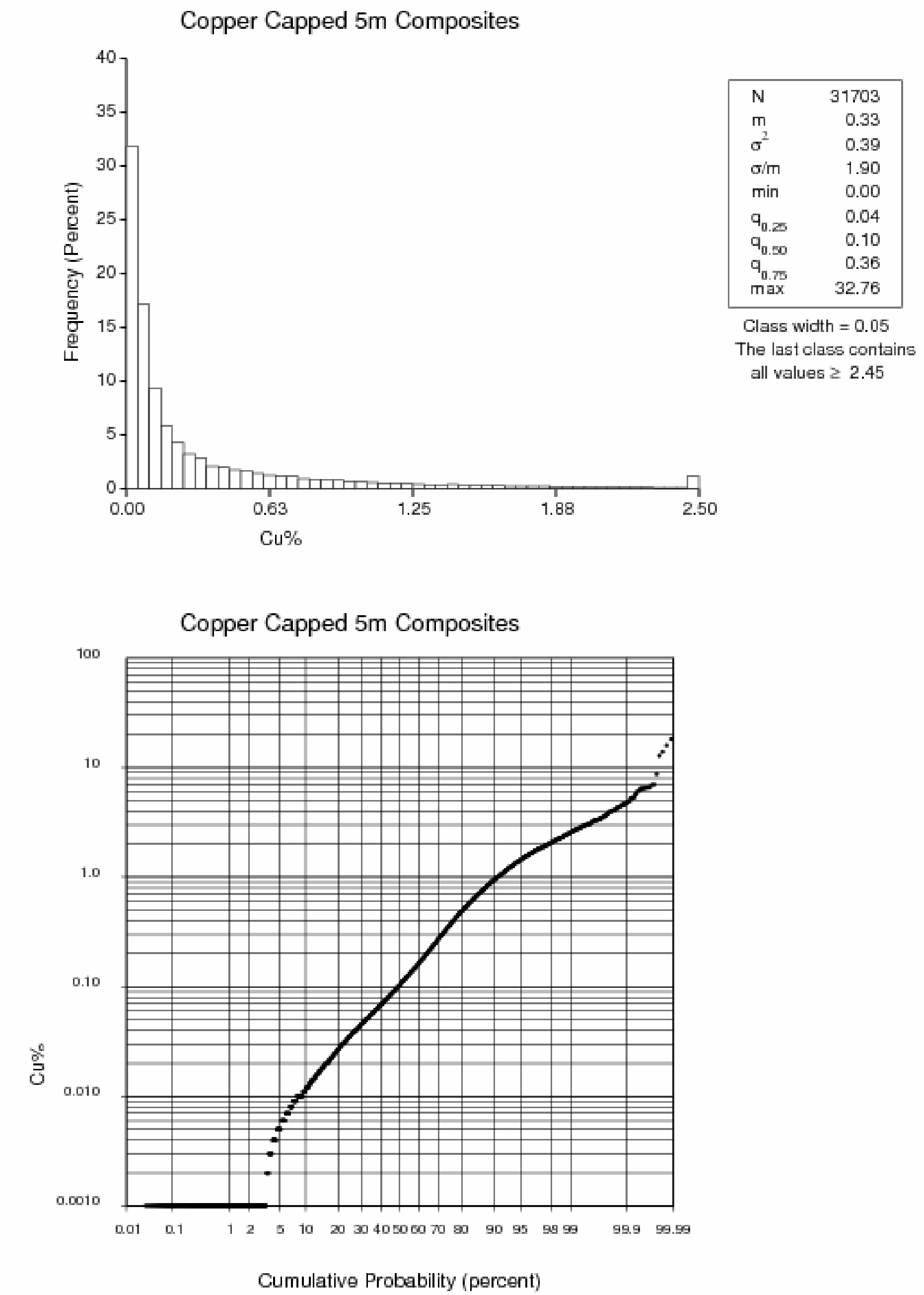
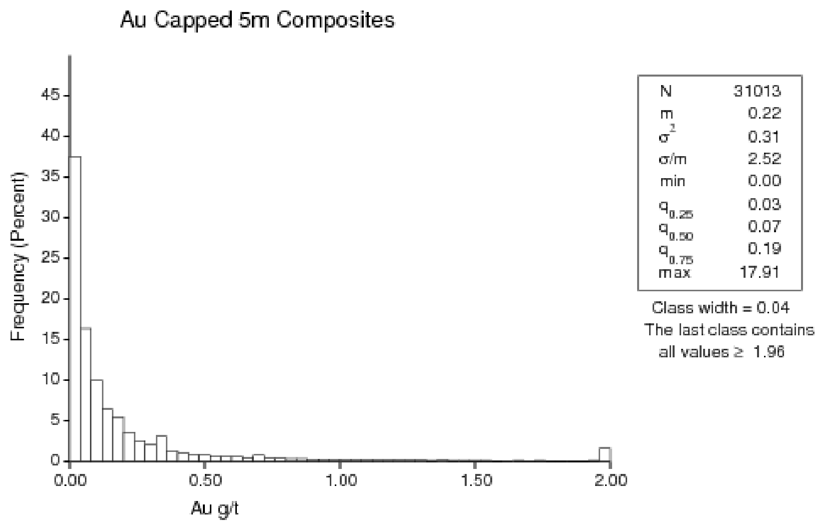


Figure 19-4: Histogram and Probability Plot for All Capped 5m Au Composites



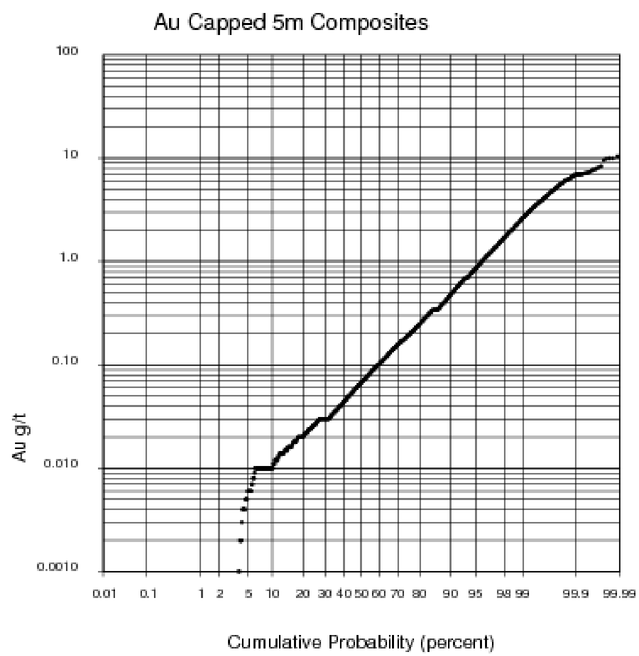
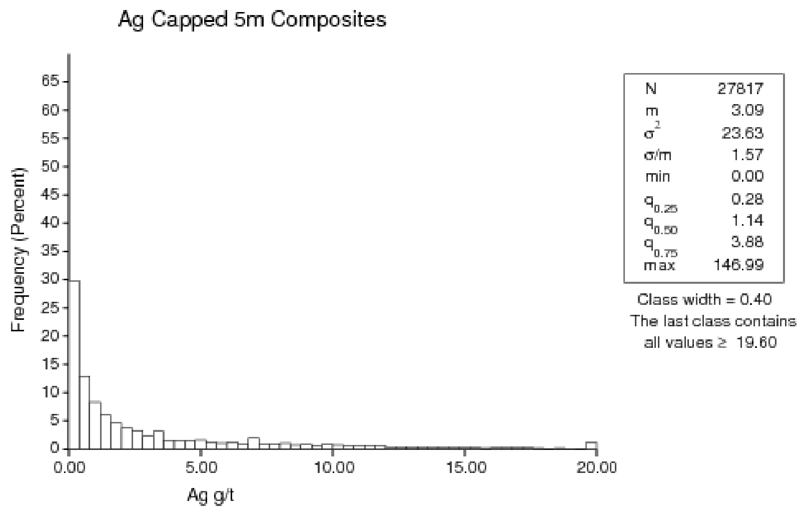
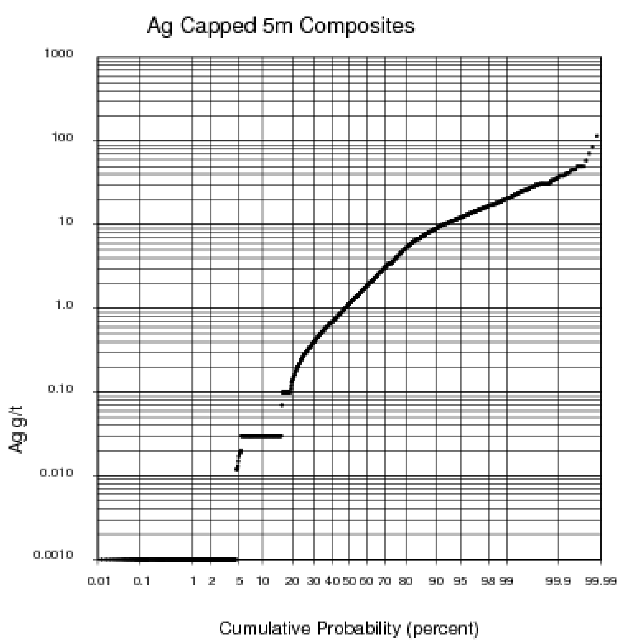


Figure 19-5: Histogram and Probability Plot for All Capped 5m Ag Composites





Tables 19-12 to 19-14 summarize average copper, gold, and silver composite grades and coefficients of variation (CV) calculated for each rock group and grade shell.

Table 19-12: 5m Cu Composite Grades by Area

Area	Grade Shell or Rock Group	Number of Composites	Mean Cu (%)	CV
North Junction	Inside of Grade Shell	850	0.86	1.27
	Outside of Grade Shell	1,030	0.09	1.64
Junction	Inside of Grade Shell	247	0.40	0.94
	Outside of Grade Shell	356	0.08	0.89
Middle Creek	Inside of Grade Shell	235	0.37	1.24
	Outside of Grade Shell	594	0.08	0.87
Southwest	Inside of Grade Shell	1,474	0.46	1.25
	Inside Low Grade Shell	2,038	0.10	1.04
	Outside of Grade Shell	1,588	0.09	1.87
West Fork	Inside of Grade Shell	1,054	0.40	1.13
	Opulent Grade Shell	45	5.00	1.45
	Upper Opulent Grade Shell	131	0.36	0.90
	Outside of Grade Shell	2,181	0.06	1.62
South Gold Lens	Volcanic Group	1,710	0.59	1.40
	Mineralized Intrusive Group	1,042	0.38	1.48
	Intrusive Group	782	0.22	2.05
Central Replacement Zone	Volcanic Group	4,277	0.51	1.29
	Mineralized Intrusive Group	124	0.74	0.90
	Intrusive Group	2,441	0.17	2.35
Bountiful	Volcanic Group	649	0.18	1.58
	Mineralized Intrusive Group	119	0.09	1.22
	Intrusive Group	70	0.09	1.65
North Gold Lens	Volcanic Group	6,504	0.41	1.21
	Mineralized Intrusive Group	6	0.47	0.16
	Intrusive Group	1,823	0.19	1.77

Table 19-13: 5m Au Composite Grades by Area

Area	Grade Shell or Rock Group	Number of Composites	Mean Au (g/t)	CV
North Junction	Grade Shell	744	0.47	1.63
	Outside Grade Shell	905	0.07	1.89
Junction	Grade Shell	246	0.16	0.99
	Outside Grade Shell	341	0.04	1.07
Middle Creek	Grade Shell	235	0.52	2.04
	Outside Grade Shell	594	0.08	1.59
Southwest	Grade Shell	1,474	0.80	1.43
	Low Grade Shell	2,038	0.16	1.79
	Outside Grade Shell	1,574	0.19	2.75
West Fork	Grade Shell	1,054	0.25	1.42
	Opulent Grade Shell	45	1.73	0.95
	Upper Opulent Grade Shell	131	0.35	0.86
	Outside Grade Shell	2,183	0.09	1.81
South Gold Lens	Volcanic Group	1,692	0.22	2.02
	Mineralized Intrusive Group	1,030	0.22	1.91
	Intrusive Group	773	0.17	1.94
Central Replacement Zone	Volcanic Group	4,093	0.14	1.66
	Mineralized Intrusive Group	117	0.12	0.73
	Intrusive Group	2,403	0.06	1.96
Bountiful	Volcanic Group	649	0.07	1.41
	Mineralized Intrusive Group	119	0.05	1.29
	Intrusive Group	70	0.06	1.49
North Gold Lens	Volcanic Group	6,364	0.30	2.39
	Mineralized Intrusive Group	3	0.06	0.44
	Intrusive Group	1,812	0.18	4.08

Table 19-14: 5m Ag Composite Grades by Area

Area	Grade Shell or Rock Group	Number of Composites	Mean Ag (g/t)	CV
North Junction	Grade Shell	626	7.61	1.32
	Outside Grade Shell	828	0.90	1.73
Junction	Grade Shell	206	1.90	1.06
	Outside Grade Shell	299	0.85	1.19
Middle Creek	Grade Shell	235	2.34	1.46
	Outside Grade Shell	594	0.88	0.94
Southwest	Grade Shell	1,474	3.03	1.27
	Low Grade Shell	2,038	0.99	1.34
	Outside Grade Shell	1,574	1.49	1.77
West Fork	Grade Shell	1,054	3.32	1.53
	Opulent Grade Shell	45	21.51	1.84
	Upper Opulent Grade Shell	131	2.28	0.91
	Outside Grade Shell	2,183	0.96	1.71
South Gold Lens	Volcanic Group	1,403	5.54	1.12
	Mineralized Intrusive Group	907	3.45	1.24
	Intrusive Group	684	2.22	1.74
Central Replacement Zone	Volcanic Group	3,472	5.60	0.99
	Mineralized Intrusive Group	112	7.22	0.77
	Intrusive Group	2,110	2.01	1.87
Bountiful	Volcanic Group	602	1.99	1.45
	Mineralized Intrusive Group	113	1.06	1.21
	Intrusive Group	56	1.65	1.40
North Gold Lens	Volcanic Group	5,182	4.01	1.08
	Mineralized Intrusive Group	3	5.87	0.01
	Intrusive Group	1,620	1.88	1.59

19.1.13 Grade Variography

The Cu, Au, and Ag grade variograms were computed with Sage 2001 constrained by grade shell or rock type group using the correlogram method.

Down-hole and experimental variograms (in 37 directions) were computed and variogram modeling was completed with Sage 2001. The nugget effect was measured using down-hole variograms and manually set in the directional variogram models. Two spherical structures were automatically fitted in Sage 2001: the structure sills C1 and C2, and their ranges. If necessary, revisions were made to these based on geological knowledge. Vulcan® rotation conventions were specified. An inadequate number of composites prevented reasonable variogram determination in the Bountiful area. The variogram from the geologically similar and spatially adjacent Central Replacement Zone was used for the Bountiful area. Resulting variograms are summarized in the Feasibility Study.

19.1.14 Block Model Setup

Block dimensions are 25m x 25m in plan and 15m vertically. The model extends a total of 6,000 metres in both the north-south and east-west directions and a total of 1,605 metres in the vertical direction. Block model fields are listed in Table 19-15.

Table 19-15: Principal Block Model Fields

Field Name	Description
CU	Weight-averaged calculation of Cu grade in bedrock portion
AU	Weight-averaged calculation of Au grade in bedrock portion
AG	Weight-averaged calculation of Ag grade in bedrock portion
CU1	Kriged estimate of Cu grade in volcanic rock group and in grade shells
CU2	kriged estimate of Cu grade in mineralized intrusive rock group and SWZ low grade shell
CU3	Kriged estimate of Cu grade in intrusive rock group
AU1	Kriged estimate of Au grade in volcanic rock group and in grade shells
AU2	Kriged estimate of Au grade in mineralized intrusive rock group and SWZ low grade shell
AU3	Kriged estimate of Au grade in intrusive rock group
AG1	Kriged estimate of Ag grade in volcanic rock group and in grade shells
AG2	Kriged estimate of Ag grade in mineralized intrusive rock group and SWZ low grade shell
AG3	Kriged estimate of Ag grade in intrusive rock group
CU1_NN	Nearest neighbor estimate of Cu in volcanic rocktype and SWZ low grade shell
CU2_NN	Nearest neighbor estimate of Cu in mineralized intrusive rocktype
CU3_NN	Nearest neighbor estimate of Cu in intrusive rocktype
AU1_NN	Nearest neighbor estimate of Au in volcanic rocktype and SWZ low grade shell
AU2_NN	Nearest neighbor estimate of Au in mineralized intrusive rocktype
AU3_NN	Nearest neighbor estimate of Au in intrusive rocktype
AG1_NN	Nearest neighbor estimate of Ag in volcanic rocktype and SWZ low grade shell
AG2_NN	Nearest neighbor estimate of Ag in mineralized intrusive rocktype
AG3_NN	Nearest neighbor estimate of Ag in intrusive rocktype
CU1_NN	Nearest neighbor estimate of Cu in volcanic rocktype and SWZ low grade shell
CU2_NN	Nearest neighbor estimate of Cu in mineralized intrusive rocktype
CU3_NN	Nearest neighbor estimate of Cu in intrusive rocktype
CUOUT	Nearest neighbor estimate of Cu outside of grade shell
AUOUT	Nearest neighbor estimate of Au outside of grade shell
AGOUT	Nearest neighbor estimate of Ag outside of grade shell
SOLCU	Inverse distance cubed acid soluble copper - near surface environment
CUPCT_SOL	Inverse distance cubed total copper - near surface environment
SOL_RATIO	Copper solubility ratio - SOLCU/CUPCT_SOL
VFLAG, DFLAG,	Estimation pass flags
AREA	Estimation Area Code
JUNC1	Proportion of block within Junction grade shell (sj1_minshell.00t)
JUNCOUT	Proportion of block outside of Junction grade shell
MC1	Proportion of block within Middle Creek grade shell (mc_mz1.00t)
MC2	Proportion of block within Middle Creek grade shell (mc_mz2cut.00t)
MCOUT	Proportion of block outside of Junction grade shell
NJ1	Proportion of block within North Junction grade shell (sj1_minshell.00t)
NJOUT	Proportion of block outside of North Junction grade shell
SWZ1	Proportion of block within Southwest Zone grade shell (swz_minshell.00t)
SWZ2	Proportion of block within Southwest Zone low grade shell (sw_subore_trrimmed.00t)
SWZOUT	Proportion of block outside of Southwest Zone grade shell
WF1	Proportion of block within West Fork grade shell (wf_minshell.00t)
OP1	Proportion of block within Opulent and Upper Opulent grade shell (wf_opulent.00t and wf_upr_opulent.00t)
WFOUT	Proportion of block outside of West Fork and Opulent grade shells
VOLPCT	Proportion of block within the volcanic rock group
DILPCT	Proportion of block within the mineralized intrusive rock group
INTPCT	Proportion i of block within the intrusive rock group
DIST (1 to 3)	Distance to the nearest composite used by kriging stored by kriging pass
NCOMP (1 to 3)	Number of composites used by kriging, stored by kriging pass
NHOLE (1 to 3)	Number of drillholes used by kriging, stored by kriging pass
TOPO	Proportion of block below topographic surface
BEDROCK	Proportion of block below overburden
CLASS	Resource classification 1=measured, 2=indicated, 3=inferred
STICKROCK	Proportion of block below broken rock/stick rock surface
CU_BLK	Fully diluted block Cu estimated grade
AU_BLK	Fully diluted block Au estimated grade
AG_BLK	Fully diluted block Ag estimated grade
SGDRY	Bulk density for bedrock proportion of partial block
SG_ADJUSTED	Bulk density for whole block adjusted for topography proportion
SG	Block bulk density unadjusted by topography proportion

19.1.15 Grade Estimation Plan

Three estimation methods were used during the interpolation process:

- Ordinary kriging
- Nearest neighbor
- Inverse distance to a power

Estimates were constrained by grade shell or, in the Central Area, by rock group. Estimation of soluble copper (as described in Section 19.1.8) was constrained by depth below bedrock. Hard contacts were used to constrain the use of composite samples to their respective shell or rock group. Blocks with multiple estimation domains (e.g. blocks straddling lithologic or grade shell contacts) contain the estimated grade for each proportion stored in individual block variables.

The minimum and maximum number of composites used in block interpolation and the maximum number of selected composites per drill hole is unchanged from Pass 1 to 3. Kriging plans are detailed in the feasibility study and a summary is provided in Table 19-16.

The nearest neighbor interpolation was completed for the capped composites using the 5 metre composite samples.

Search ellipsoid ranges were determined using the following methodology:

- Pass 1: Ranges selected after inspection of variograms and sample spacing.
- Pass 2: Ranges from Pass 1 are extended by 50%.
- Pass 3: 100% increase in search distances from pass 2.

Details on the capping policy applied for each grade shell and rock type are summarized in Table 19-9.

In the South and North Gold Lens, Bountiful, and Central Replacement areas, the geologic model was simplified for grade estimation by grouping genetically and statistically similar rock types into three lithology groups: volcanic, mineralized intrusive, and intrusive material. Mineralization is predominantly hosted by the volcanic group (V1, V2 and V3). Two intrusive units (I4 and I8) may be mineralized when they crosscut volcanic rocks as relatively narrow dykes and sills. Intrusive units primarily contain weak, discontinuous mineralization. The remaining intrusives within the extent of drilling are post mineral. Each model block contains the proportion of volcanic, mineralized intrusive and intrusive groups in the block. The block model was constructed to accept up to three grades per block: volcanic, mineralized intrusive, and intrusive.

Table 19-16: Summary of Kriging Search Parameters

Estimation Run		Estimation Domain									
		Middle	Junction	North	SW	Opulent	West	CRZ	North	South	Bountiful
Pass 1	Range X	75	75	75	75	75	75	75	75	75	75
	Range Y	75	75	75	75	75	75	75	75	75	75
	Range Z	20	20	20	20	20	20	30	30	30	30
	1st Rot	82	33	34	114	0	90	30	33	33	30
	2nd Rot	0	0	0	0	0	0	0	0	0	0
	3rd Rot	63	57	48	-48	60	34	30	40	60	30
Pass 2	Range X	120	120	120	120	120	120	120	120	120	120
	Range Y	120	120	120	120	120	120	120	120	120	120
	Range Z	30	30	30	30	30	30	45	45	45	45
	1st Rot	82	33	34	114	0	90	30	33	33	30
	2nd Rot	0	0	0	0	0	0	0	0	0	0
	3rd Rot	63	57	48	-48	60	34	30	40	60	30
Pass 3	Range X	240	240	240	240	240	240	240	240	240	240
	Range Y	240	240	240	240	240	240	240	240	240	240
	Range Z	60	60	60	60	60	60	60	60	60	60
	1st Rot	82	33	34	114	0	90	30	33	33	30
	2nd Rot	0	0	0	0	0	0	0	0	0	0
	3rd Rot	63	57	48	-48	60	34	30	40	60	30

Table 19-17: Central Area Rock Groups

Rock Group	Rock Type Assignment	Composite Codes
Volcanic	V1, V2, V3	210, 220, 230
Mineralized	I4, I8	362, 340
Post Mineral	D2, D3, I5, I9, I10, I11	500, 350, 380, 372

The East fault which bisects the North Gold Lens and Central Replacement Zone was treated as a hard boundary. It is a northerly-trending, 50 degree west dipping structure. No significant displacement has been observed in drilling; however, contact analysis indicates a weak discontinuity in grade at the hangingwall / footwall contact.

19.1.15.1 Ordinary Kriging

Kriging was completed using Vulcan® estimation batch files galoboun.bef, galocrz.bef, galojunc.bef, galomc.bef, galongl.bef, galonj.bef, galosgl.bef, galoswz.bef and galowf.bef. Composites of nominal 5 metre length were used in the interpolation.

Variogram parameters are grade shell or rock group specific and are described in the kriging plan. During kriging, blocks were selected based on their grade shell or rock group domain. Copper, gold and silver estimates within grade shells or rock groups were stored in the block model CU1, CU2 and CU3, AU1, AU2 and AU3, and AG1, AG2 and AG3 depending upon the grade shell or rock group being estimated. CUOUT, AUOUT and AGOUT contain grade estimates for blocks outside of grade shells. Other kriging variables were also stored in each block during interpolation:

- Average distance to composites in DIST1, DIST2, and DIST3
- Number of composites used in NCOMP1, NCOMP2, AND NCOMP3
- Number of drill holes used in NHOLE1, NHOLE2, AND NHOLE3

19.1.15.2 Nearest Neighbor Estimation

Nearest neighbor estimation was done using 5 metre composites. Block selection and block model-composite matching were the same as implemented for ordinary kriging described in Section 19.1.15.1. Estimation was performed in one pass using kriging Pass 3 search parameters.

19.1.15.3 Inverse Distance to a Power Estimation

Total and acid soluble copper grades were estimated by inverse distance cubed weighting methods to calculate the copper solubility ratio. The copper solubility ratio was used to calculate the available copper content as described in Section 19.1.8. Blocks and composites were matched relative to depth below the bedrock interface using an isotropic search of 150 metres.

19.1.15.4 Calculation of Whole Block Grades

The kriged and nearest neighbor grades were estimated for up to three proportions individually. The three Cu, Au and Ag grades per block were combined, weighted by density and rock type proportion.

19.1.15.5 Block Model Visual Inspection

Visual inspection confirmed that the block model honors the drill hole data. Screen-capture plots in Figures 19-6, 19-7 and 19-8 present a representative bench of the copper, gold and silver resource models.

Figure 19-6: 502.5m Bench – Block Model Cu Values (%)

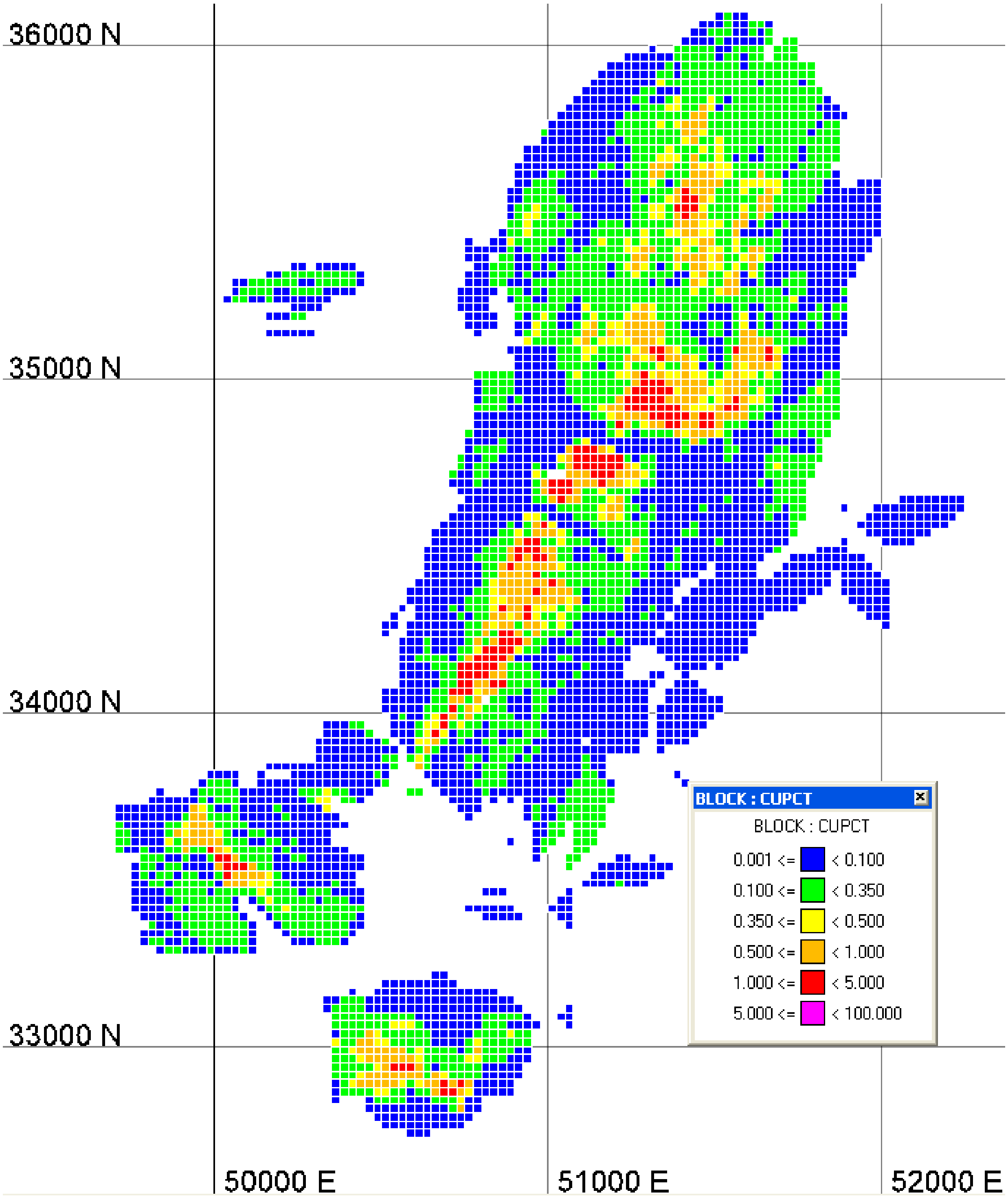


Figure 19-7: 502.5m Bench – Block Model Au Values (g/t)

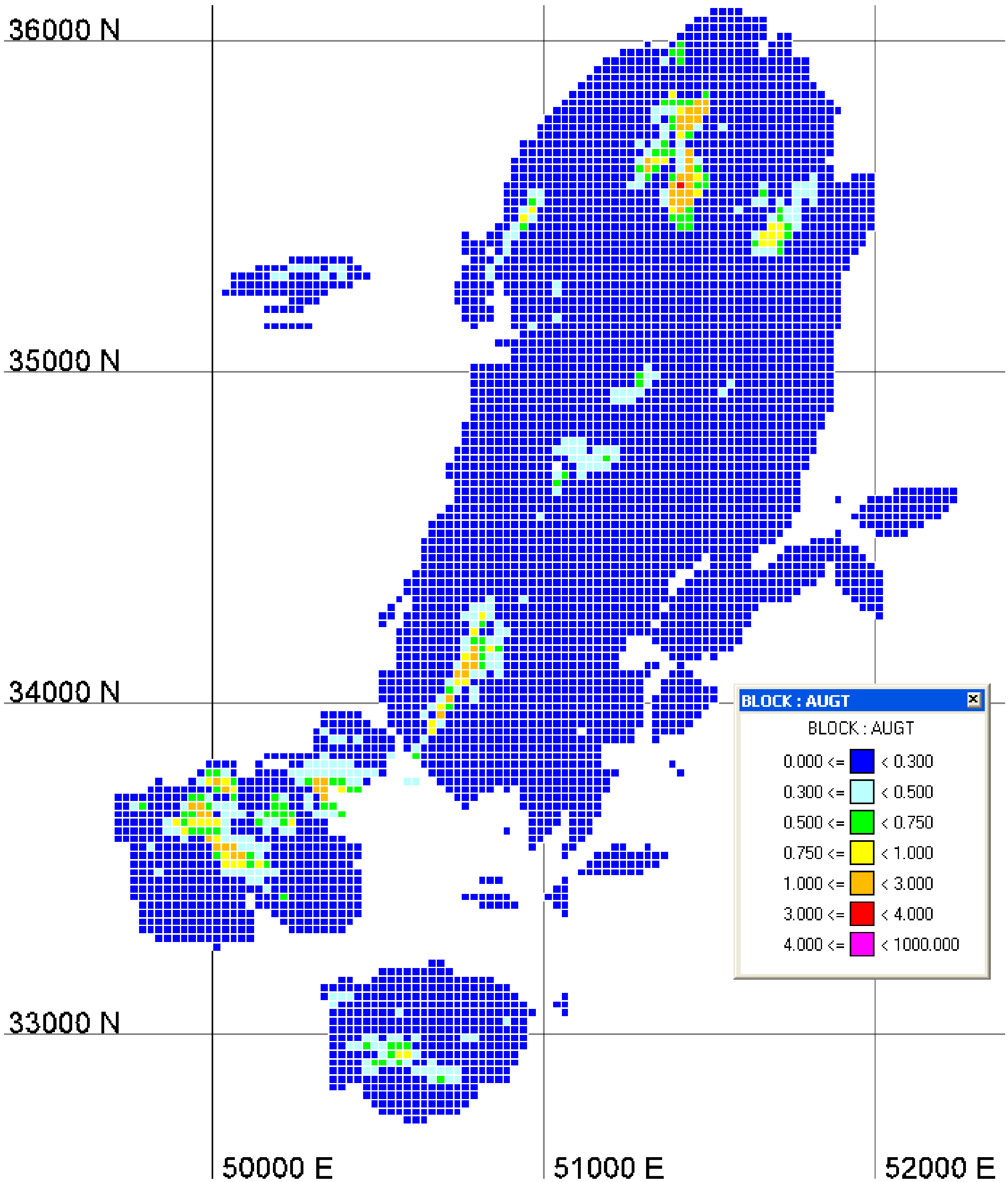
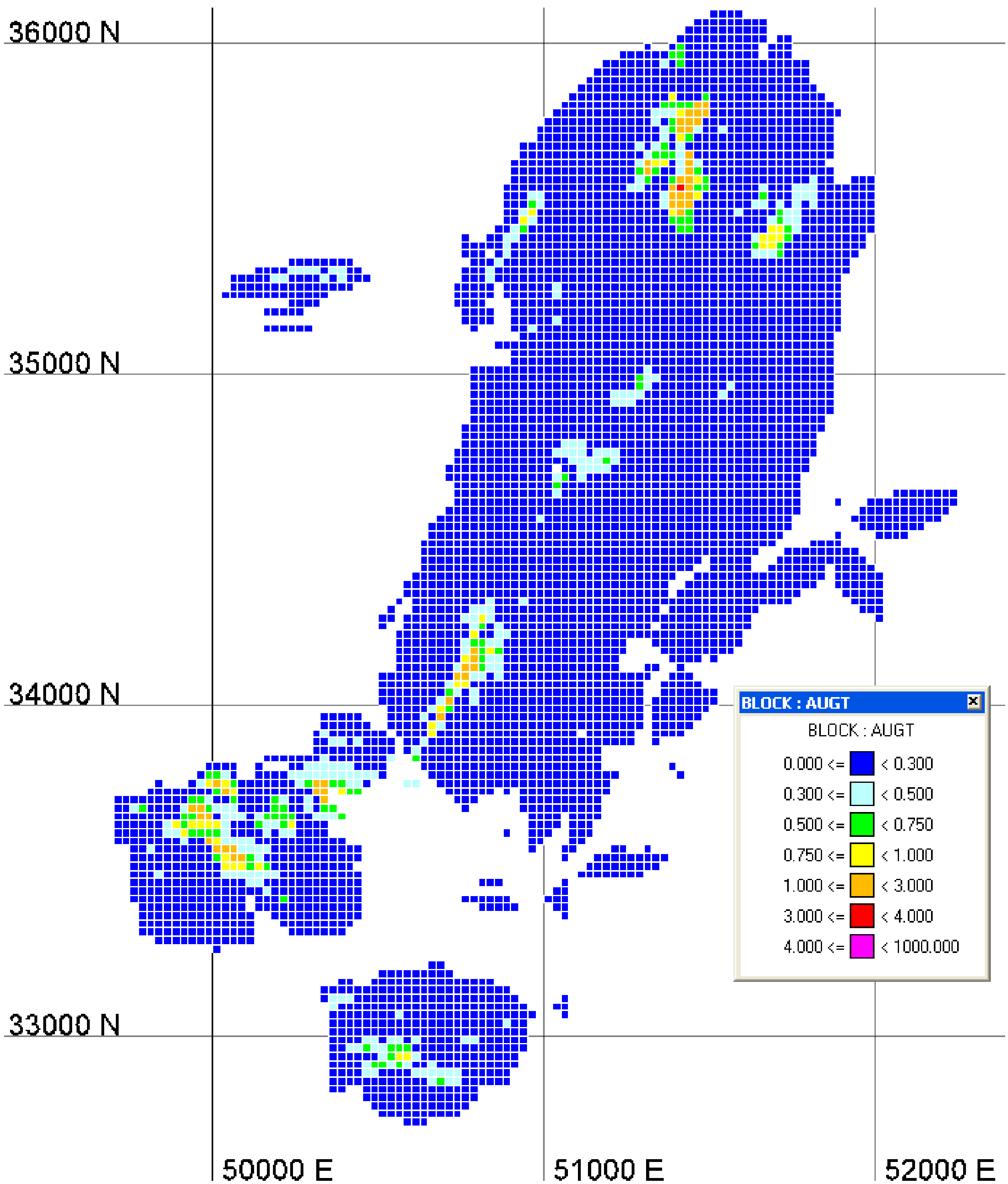


Figure 19-8: 502.5m Bench – Block Model Ag Values (g/t)



19.1.16 Model Validation

Validation was performed on blocks constrained by grade shells or rock group and estimated in passes 1 and 2. This encompasses the section of the deposit which contains the highest concentration of drill holes and is an important dataset on which to perform validation.

19.1.16.1 Swath Plots

Swath plots comparing kriged Cu, Au and Ag and nearest neighbor estimates were plotted in East-West, North-South and Vertical directions. They are presented in the Feasibility Study, and examples are found in Figure 19-9 and Figure 19-10. Variables agree well in general, and no major spatial bias was observed. Differences do occur between the raw composites and the nearest neighbor and kriged grade estimates. The composites are not declustered or constrained by outlier restrictions, and differences between them, the kriged and nearest neighbor estimates are not unreasonable.

19.1.16.2 Comparison of Composites and Estimated Grades

Histograms of the validation set of blocks and composites were plotted for Cu, Au and Ag. They are presented in the Feasibility Study. Table 19-18 and Table 19-19 compare the validation set of block model grades with the 5-metre-long composite grades. The estimated grades compare reasonably well with the nearest neighbor model, indicating the process did not introduce bias.

As expected the estimated grades produce distributions that are smoother than the nearest neighbor model. The amount of smoothing has been adjusted to match the selective mining unit (SMU) and produce a model that is appropriate for mine planning.

Figure 19-9: Cu Swath Plots

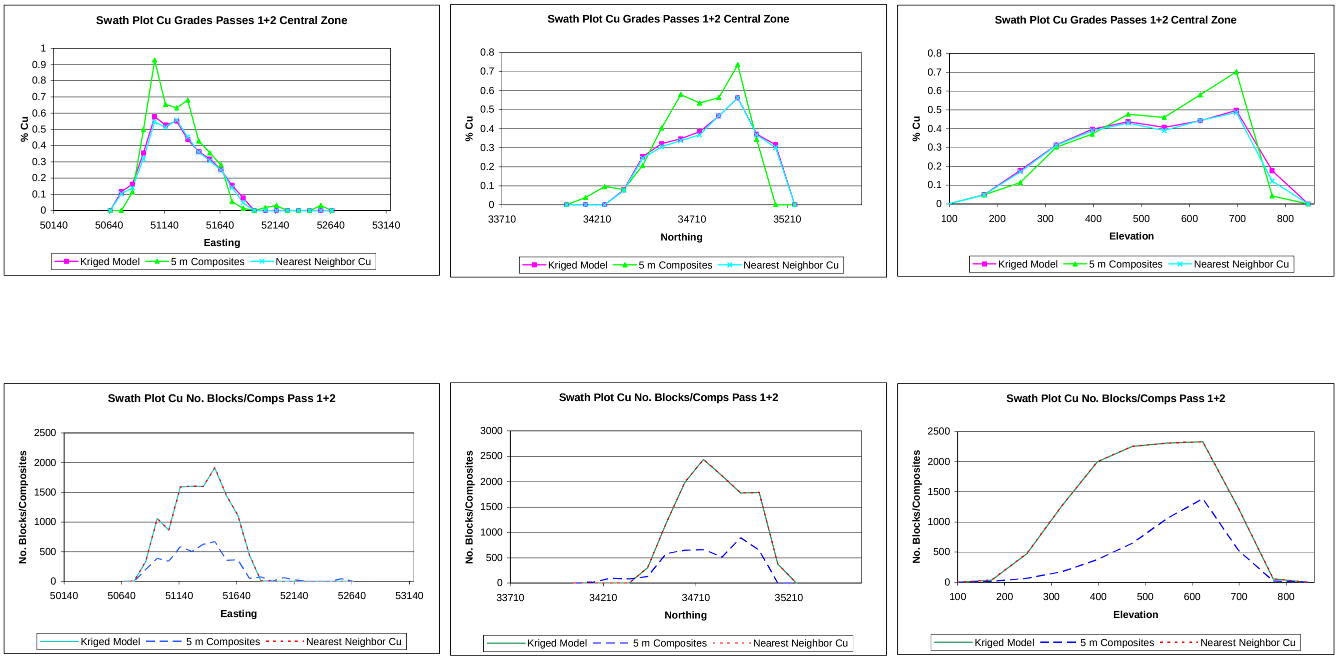


Figure 19-10: Au Swath Plots

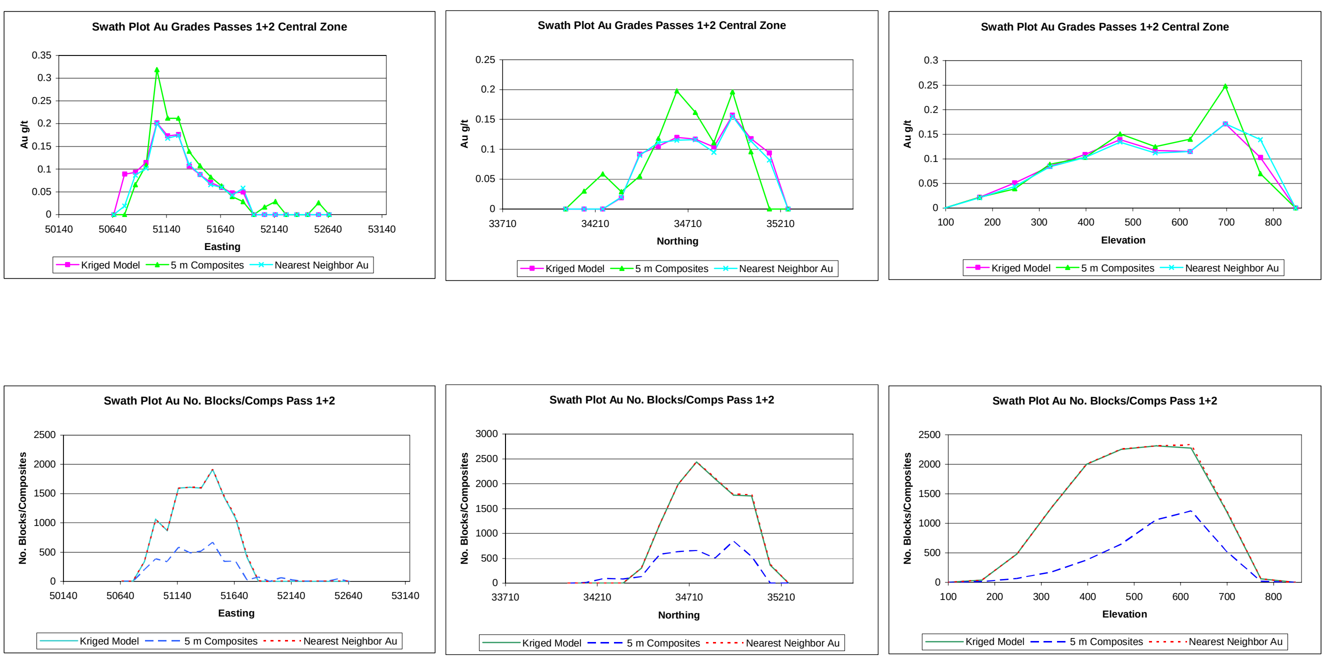


Table 19-18: Comparison of Cu Composites and Model Block Grades

Mineral Zone Area	Rock Group or Grade Shell	Number of Block Pairs	5 m Composites			Kriged Model		Nearest Neighbor		% Difference (Krige-NN)/NN
			No.	Mean Cu	CV	Mean Cu	CV	Mean Cu	CV	
Bountiful	Volcanic	2,882	649	0.18	1.58	0.305	0.77	0.302	1.19	1
	Intrusive	247	70	0.09	1.65	0.050	0.59	0.053	1.17	-6
Central Replacement Zone	Volcanic	11,991	4,277	0.51	1.29	0.400	0.99	0.392	1.30	2
	Intrusive	4,877	2,441	0.17	2.35	0.131	1.60	0.126	2.49	4
Junction	Grade Shell	906	247	0.40	0.94	0.474	0.55	0.486	0.92	-2
Middle Creek	Grade Shell	540	235	0.37	1.24	0.321	0.65	0.314	1.26	2
North Gold Lens	Volcanic	18,881	6,504	0.41	1.21	0.325	0.93	0.319	1.32	2
	Intrusive	2,603	1,823	0.19	1.77	0.134	1.29	0.124	1.94	8
North Junction	Grade Shell	1,537	850	0.86	1.27	0.811	0.78	0.779	1.27	4
South Gold Lens	Volcanic	3,674	1,710	0.59	1.40	0.429	1.23	0.420	1.61	2
	Intrusive	2,037	782	0.22	2.05	0.183	1.32	0.188	2.04	-3
Southwest Zone	Grade Shell	3,655	1,474	0.46	1.25	0.417	0.82	0.420	1.25	-1
	Low Grade Shell	6,909	1,588	0.09	1.87	0.106	0.68	0.106	1.03	0
West Fork	Grade Shell	2,727	1,054	0.40	1.13	0.389	0.63	0.399	1.13	-3
	Opulent	91	45	5.00	1.45	0.615	1.02	0.594	1.44	4

Table 19-19: Comparison of Au Composites and Model Blocks

Mineral Zone Area	Rock Group or Grade Shell	Number of Block Pairs	5 m Composites			Kriged Model		Nearest Neighbor		% Difference (Krige-NN)/NN
			No.	Mean Au	CV	Mean Au	CV	Mean Au	CV	
Bountiful	Volcanic	2,882	649	0.07	1.41	0.100	0.77	0.100	1.14	1
	Intrusive	247	70	0.06	1.49	0.040	0.90	0.050	1.58	-6
Central Replacement Zone	Volcanic	11,907	4,093	0.14	1.66	0.120	1.08	0.110	1.62	3
	Intrusive	4,872	2,403	0.06	1.96	0.060	1.03	0.050	1.82	2
Junction	Grade Shell	906	246	0.16	0.99	0.150	0.47	0.160	0.88	-2
Middle Creek	Grade Shell	540	235	0.52	2.04	0.410	1.11	0.420	2.15	-1
North Gold Lens	Volcanic	18,724	6,364	0.30	2.39	0.230	1.64	0.230	2.52	0
	Intrusive	2,592	1,812	0.18	4.08	0.130	2.79	0.130	4.67	2
North Junction	Grade Shell	1,308	744	0.47	1.63	0.500	0.84	0.470	1.77	6
South Gold Lens	Volcanic	3,670	1,692	0.22	2.02	0.180	1.54	0.170	2.75	5
	Intrusive	2,036	773	0.17	1.94	0.170	1.18	0.190	1.94	-9
Southwest Zone	Grade Shell	3,655	1,474	0.80	1.43	0.690	0.82	0.670	1.42	3
	Low Grade Shell	6,909	1,574	0.19	2.75	0.160	0.88	0.150	1.70	1
West Fork	Grade Shell	2,727	1,054	0.25	1.42	0.250	0.74	0.260	1.29	-2
	Opulent	91	45	1.73	0.95	0.530	0.86	0.530	1.48	-1

As expected, the variance and CV of kriged model is much lower than the original 5-metre composites.

For the validation set, the average estimated grades compare well with the average nearest neighbor grades. The distribution's upper tail disappears, however, inducing a variance reduction. The CV for the kriged estimates is significantly lower than the CV for the 5-metre composites or nearest neighbor models. This is expected given the grade estimation method (ordinary kriging).

The mean grades for the kriged estimates are comparable with the nearest neighbor model and are reasonable.

19.1.16.3 Change of Support Checks

An independent check on the smoothing in the estimates was made using the Discrete Gaussian or Hermitian polynomial change-of-support method (Herco) described by Journel and Huijbregts (Mining Geostatistics, Academic Press, 1978). The distribution of hypothetical block grades derived by this method is compared to the estimated model grade distribution by means of grade-tonnage curves. The grade-tonnage curves allow comparison of the histograms of the two grade distributions in a format familiar to mining. If the estimation procedure has adequately predicted grades for the selected block size, then the grade-tonnage curves should match fairly closely. If the curves diverge significantly, then there is a problem with the estimated resource. The grade-tonnage predictions produced for the model show that grade and tonnage estimates are validated by the change-of-support calculations.

Block Dispersion Variances (BDV) are required to perform Herco validation and were calculated for interpolation groups, such as:

- Grade shells by area
- Volcanic and intrusive rock groups in the Central areas

BDV using a unit sill variogram model was calculated after modeling 3D variograms for each interpolation group. The selective mining unit (SMU) size used was 20m x 20m x 15m.

Resulting BDVs and CVs are tabulated in Table 19-20 and Table 19-21.

Table 19-20: Cu Block Dispersion Variances

Area	Domain or Rocktype	Block Dispersion Variance	CV 5m Composites
Bountiful	Volcanic	0.7149	1.19
	Intrusive	0.5273	1.17
Central Replacement Zone	Volcanic	0.7149	1.30
	Intrusive	0.5273	2.49
Junction	Grade Shell	0.2662	0.92
Middle Creek	Grade Shell	0.3709	1.26
North Gold Lens	Volcanic	0.6559	1.32
	Intrusive	0.5165	1.94
North Junction	Grade Shell	0.4917	1.27
South Gold Lens	Volcanic	0.6512	1.61
	Intrusive	0.6202	2.04
Southwest Zone	Grade Shell	0.4347	1.25
	Low Grade Shell	0.3808	1.03
West Fork	Grade Shell	0.3592	1.13

Table 19-21: Au Block Dispersion Variances

Area	Domain or Rocktype	Block Dispersion Variance	CV 5m Composites
Bountiful	Volcanic	0.617	1.14
	Intrusive	0.4585	1.58
Central Replacement Zone	Volcanic	0.617	1.62
	Intrusive	0.4585	1.82
Junction	Grade Shell	0.178	0.88
Middle Creek	Grade Shell	0.413	2.15
North Gold Lens	Volcanic	0.6124	2.52
	Intrusive	0.4181	4.67
North Junction	Grade Shell	0.3226	1.77
South Gold Lens	Volcanic	0.4221	2.75
	Intrusive	0.469	1.94
Southwest Zone	Grade Shell	0.3319	1.42
	Low Grade Shell	0.2553	1.70
West Fork	Grade Shell	0.3409	1.29

Herco validation was performed for blocks that were estimated by passes one and two for both copper and gold. An SMU block size of 20m x 20m x 15m was specified by NovaGold, reflecting the selectivity of the Feasibility Study. Resulting grade-tonnage and grade-above-cutoff curves (Herco adjusted and kriged) are presented in the Feasibility Study. A representative grade-tonnage graph is in Figure 19-11 and Figure 19-12.

In general, the Herco-adjusted grade-proportion curves fit the kriged grade-proportion curves reasonably well, indicating the appropriate amount of smoothing has been achieved via kriging.

Figure 19-11: Herco Cu Grade-Tonnage Curves – Central Replacement Zone

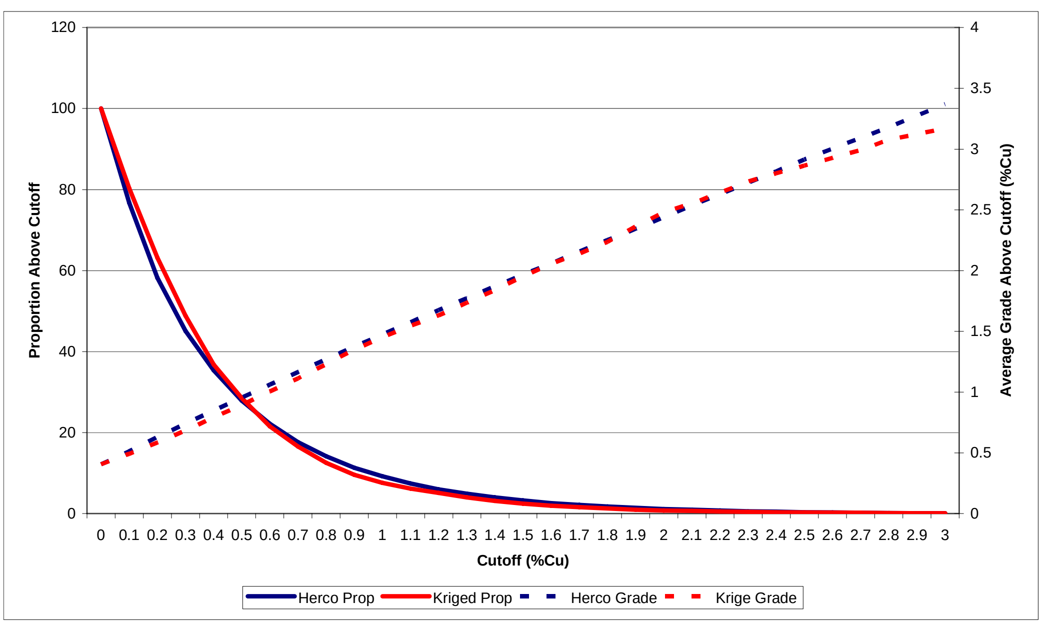
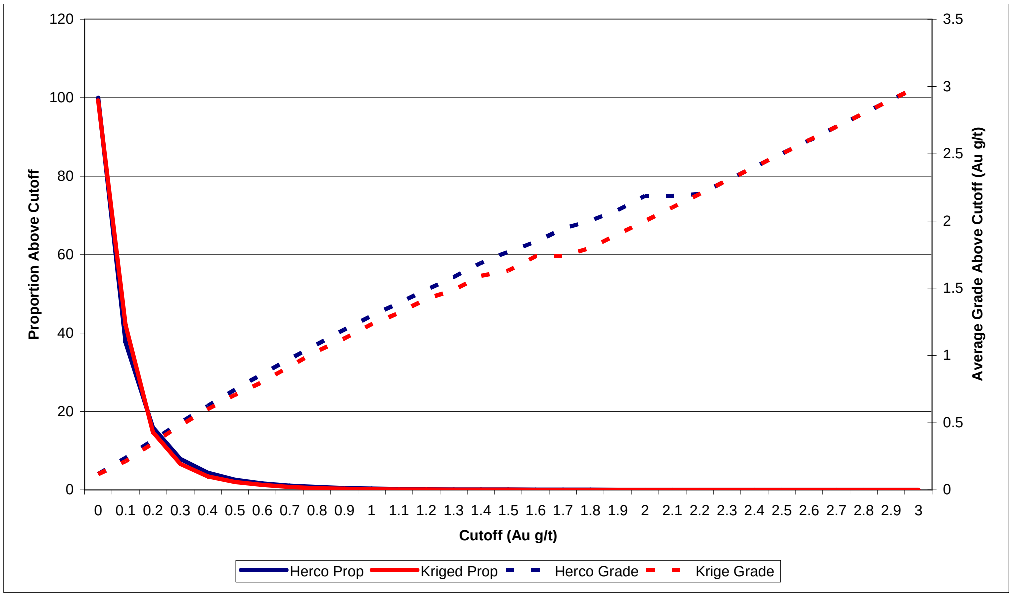


Figure 19-12: Herco Au Grade-Tonnage Curves – Central Replacement Zone



19.1.17 Resource Classification

The Galore Creek mineral resource has been classified using logic consistent with the CIM Standard Definitions as dictated by National Instrument 43-101. Resource classification is based on various block model parameters together with the demonstrated confidence in the assayed values with a well-functioning QA/QC program. The estimated Galore Creek resources have been classified into Measured, Indicated and Inferred Resources categories and are summarized in Table 19-23.

In general, the limit of the Mineral Resources is based on the following criteria: blocks within specified distances of an assay composite, a minimum of two drill holes, are required to estimate the Cu grade and resources are contained within a conceptual pit using metal prices of US$1.25/lb Cu, US$450/oz Au and US$7/oz Ag. Additionally, Measured or Indicated Resources must be within a grade shell or lithology group. Composite spacing criteria were determined using the confidence limit on the grade estimate and its relationship to the projected production rate of 65,000 tonnes per day of ore. Confidence limits were calculated for Cu and Au grades. In general, distances are comparable with Au tending to require closer spaced composite spacing. Copper grade is proposed to be the primary criteria for ore/waste selection and constitutes the major saleable product, and it is reasonable to set classification criteria on the basis of copper grades.

19.1.17.1 Measured Resource Criteria

The criteria for defining Galore Creek Measured Resources was that there were a minimum of two drill holes within 80 metres horizontally and 55 metres vertically of the estimated block. Furthermore, the block had to be constrained within a grade shell or lithology group.

19.1.17.2 Indicated Resource Criteria

The criteria for defining Indicated Resources was that the block was within 80 metres horizontally and 55 metres vertically of the estimated block and that a second drill hole had to be within the distance as specified in Table 19-22. Confidence limits were determined and classification designation assigned by grade shell and lithology group.

Table 19-22: Drill Hole Spacing Based on Confidence Limits

Resource Area	Estimation Group	Drill Hole Spacing for ± 15% Annual Confidence Limit (metres)	
		Cu	Au
Bountiful	Volcanic	160	150
	Intrusive	150	100
Central Replacement Zone	Volcanic	150	100
	Intrusive	100	100
Junction	Grade Shell	150	100
Middle Creek	Grade Shell	150	100
North Gold Lens	Volcanic	160	100
	Intrusive	125	35
North Junction	Grade Shell	150	100
South Gold Lens	Volcanic	125	75
	Intrusive	110	100
Southwest	Grade Shell	150	150
Southwest LG	Grade Shell	150	150
West Fork	Grade Shell	150	150

That portion of the Galore Creek deposit designated in this manner as the Measured and Indicated Mineral Resource has been estimated with a "level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit" (NI 43-101).

19.1.17.3 Inferred Resource Criteria

That portion of the Galore Creek deposit for which an estimate has been made but is not deemed part of the Measured and Indicated Mineral Resource has been classified as the Inferred Mineral Resource. This portion is based on more limited drill information, and will require additional drilling to support detailed mine planning and evaluation.

19.1.17.4 Galore Creek Mineral Resources

Measured, Indicated, and Inferred Mineral Resources were tabulated for the Galore Creek Project. In lieu of summarizing resources within the total block model at a particular cutoff grade, only those resources located inside of a conceptual pit that were 50% greater than the following metals prices of US$1.25/lb of copper, US$450/oz of gold, and US$7/oz of silver were used to generate the conceptual pit. Feasibility Study mining costs, processing costs, metal recoveries, and pit slope angles were also used to generate the conceptual pit. The Galore Creek Mineral Resources within the conceptual pit are summarized in Table 19-23 using a 0.25% copper equivalent (CuEq) cutoff grade.

Table 19-23: Galore Creek Mineral Resources @ 0.25% CuEq Cutoff

Resource Category	Tonnes (millions)	Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)	Cu Pounds (billions)	Au Ounces (millions)	Ag Ounces (millions)
Measured	263.6	0.62	0.35	5.9	0.81	3.6	3.0	50.0
Indicated	485.3	0.46	0.28	4.3	0.63	4.9	4.4	67.1
Measured + Indicated	748.9	0.52	0.30	4.9	0.69	8.5	7.4	117.1
Inferred	300.1	0.37	0.21	3.7	0.51	2.4	2.0	35.7

The copper equivalent grade was calculated as follows:

CuEq = Recoverable Revenue / 2204.62 * 100 / 1.25 / Cu Recovery / 100

Where:

CuEq = Copper equivalent grade

Recoverable Revenue = Revenue in US dollars for recoverable copper, recoverable gold, and recoverable silver using metal prices of US$1.25/lb, US$450/oz, and US$7/oz for copper, gold, and silver, respectively

Cu Recovery = Recovery for copper based on mineral zone and total copper grade

Michael Lechner,RPG. has performed several independent reviews of NovaGold's Galore Creek resource model. These checks included a visual comparison of drill hole composite and block grades in section and plan, a global grade bias check (independent nearest neighbor models), and a set of swath plots that compare the author's nearest neighbor grades with NovaGold's kriged copper and gold block grades. Based on these reviews, it is the author's opinion that the NovaGold resource model is globally unbiased and is suitable to be used for subsequent pit optimization and mine planning activities.

19.2 Mineral Reserve

Section 19.2 was prepared by Mr. Jim Gray,P.Eng. GR Technical Services.

19.2.1 Summary

An optimized feasibility level 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 produced the Phase reserves in the table below. The Phase reserves include a 3.6% dilution for all material above cutoff grade, and mining losses of 2.4%.

Cut-off grade for the Phase reserves in the table below is CA$3.82/t Net Smelter Return (NSR).

Table 19-24: Summarized Proven And Probable Pit Reserves for Galore Creek

Phase	Insitu	Insitu	Run of	Waste	Rom	Diluted Grades			
	Ore	Ore	Mine	Total	S/R	NSR	CU	AU	AG
	(kBCMS)	(kt)	(kt)	(kt)	(t/t)	CA$/t	%	g/t	g/t
PreStrip	2,272	5,417	5,483	109,618	20.0	19.0	0.702	0.374	5.45
C626	44,349	114,245	115,616	89,949	0.8	16.6	0.528	0.380	4.95
C636	13,020	32,257	32,644	22,757	0.7	20.6	0.746	0.193	7.29
C646	18,715	45,595	46,142	40,178	0.9	19.6	0.682	0.291	6.31
C656i	49,895	130,179	131,742	143,363	1.1	13.7	0.508	0.146	6.04
C666i	45,100	118,573	119,997	210,100	1.8	14.2	0.497	0.229	5.16
Total Central	**173,351**	**446,266**	**451,624**	**615,965**	**1.4**	**15.7**	**0.548**	**0.249**	**5.64**
J606	7,883	19,150	19,380	74,740	3.9	25.3	0.888	0.428	5.93
J616i	2,689	6,944	7,029	38,681	5.5	21.4	0.753	0.398	3.05
J626i	1,971	5,079	5,139	13,462	2.6	16.3	0.664	0.124	2.41
Total Junction	**12,543**	**31,173**	**31,548**	**126,883**	**4.0**	**23.0**	**0.822**	**0.372**	**4.71**
S616	9,464	22,481	22,751	23,648	1.0	22.4	0.604	0.944	3.31
S626i	11,939	29,270	29,621	62,312	2.1	12.6	0.378	0.539	2.67
Total South West	**21,403**	**51,751**	**52,372**	**85,960**	**1.6**	**16.9**	**0.476**	**0.715**	**2.95**
WF616	2,024	5,129	5,191	10,095	1.9	17.5	0.558	0.473	5.40
Total West Fork	**2,024**	**5,129**	**5,191**	**10,095**	**1.9**	**17.5**	**0.558**	**0.473**	**5.40**
Total Reserve	**209,321**	**534,319**	**540,735**	**838,903**	**1.6**	**16.3**	**0.557**	**0.303**	**5.32**

CIM Standards On Mineral Resources And Reserves Definitions And Guidelines define 'Probable Mineral Reserve' as *" …the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified."*

CIM Standards on Mineral Resources and Reserves Definitions and Guidelines [3] defines A 'Proven Mineral Reserve as *"…the economically mineable part of a Measured Mineral Resource demonstrated by* *at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified."*

Proven and Probable Reserves at Galore Creek are summarized in the table below.

Table 19-25: Proven and Probable Reserves at Galore Creek

Classification	Ore (Mt)	Diluted Grades Cu (%)	Au (g/t)	Ag (g/t)	Contained Metal Cu Pounds (billions)	Au Ounces (millions)	Ag Ounces (millions)
Proven	239.5	0.625	0.343	6.01	3.30	2.64	46.28
Probable	301.3	0.503	0.271	4.78	3.34	2.63	46.30
Total	540.7	0.557	0.303	5.32	6.64	5.27	92.58

19.2.2 Mine Planning 3D Block Model and MineSight Project

GR Tech has collated data from NovaGold, Hatch and BGC to form a common MineSight project, which forms the basis of the mine planning for the 2006 Feasibility Study.

The 3D Block model supplied by NovaGold was developed using compositing and interpolation of composite grades. Composite samples have been generated downhole in nominal 5 m lengths and assay grades were capped prior to compositing. Composites were not broken at geologic or grade shell boundaries. Each composite was tagged with the majority rock type of the geological triangulated solid or grade shell.

Two estimation methods were used during the interpolation process:

- Ordinary Kriging
- Nearest Neighbour.

Estimates are constrained by grade shell or, in the Central Area, by rock group. Hard contacts were used to constrain the use of composite samples to their respective shell or rock group. Blocks with multiple estimation domains (e.g. blocks straddling lithologic or grade shell contacts) contain the estimated grade for each proportion stored in individual block variables for up to 3 zones per Block. The resulting 3D Block Model supplied by NovaGold contains whole block grades.

A MineSight project called "galore06r1" has been initialized using drill hole data and the 3D Block Model supplied by NovaGold.

The project and model dimensions are:

Model Limits

	Minimum	Maximum	Size	Number
X	48000	54000	25	240
Y	32000	38000	25	240
Z	-15	1590	15	106

19.2.2.1 Mine Planning 3 D Block Model Set Up.

Another 3D Block Model was set up in "galore06r1" by GR Tech as "gc0615.da1" for mine planning. This 3D Block Model is identical to gc0615.dat from Nova Gold, but includes added items that are used for mine planning.

19.2.2.1.1 Model Check

Model grade items are supplied by NovaGold and imported to MineSight. A total model resource was estimated for gc0615.dat and compared to the result obtained by NovaGold's Vulcan model to check the accuracy of the MineSight import. The total 3D Block Model model resource estimate checks are compared in the table below:

Table 19-26: 3D Block Model Resource Estimate Checks

Estimate	ROM Ore MTonnes	CuEq %	Cu %	Au g/t	Ag g/t
NovaGold (Vulcan)	903.6	0.724	0.528	0.342	4.916
gc0615.dat	904.4	0.723	0.528	0.342	4.913
Variance %	+ 0.09%	-0.14%	0.00%	0.00%	0.06%

NOTE: The above resource estimate used:

- all classes
- cut off grade of 0.35% CuEq

Individual block comparisons confirmed that the insignificantly small differences were a result of the differences in precision between the two block models. Where the Vulcan block model values for Cu, Au and Ag grades, as well as SG carried a precision of 0.000001, the imported MineSight contains values rounded to 0.001. The TOPO item in gc0615.dat was visually checked and confirmed that it was aligned with GC06MG model topography surface. The precision of the original topography model is rounded from 0.000001 to 0.0001. The import is therefore deemed complete and correct.

19.2.2.1.2 Model Areas

The Area item in the 3D Block Model has an integer value representing geological zones in pit areas. Codes are defined as:

Code	Area
1	Bountiful
2	Central replacement zone
3	Junction
4	Middle Creek
5	North Gold Lens
6	North Junction
7	South Gold Lens
8	Southwest Zone
9	West Fork

19.2.2.1.3 Rock Type Classification

ABA2 is an integer code where 1 = PAG and 2 = NPAG where PAG = potentially acid generating and NPAG = None Potentially Acid Generating. The ABA2 values are NPAG only for blocks where the ratio of neutralizing potential to acid producing potential is >2.

The rock mass has been grouped into three major rock types primarily distinguished by degree of fragmentation. Theses are till rock (overburden, glacial moraines), broken (sheet-fractured rock) and stick rock (massive rock). Rock types are coded in the 3D Block Model (WSTZN item) for mine planning purposes as:

Code	Rock Type
1	Till
2	Broken NPAG
3	Broken PAG
4	Stick NPAG
5	Stick PAG

19.2.2.1.4 Metallurgical Recovered Grades

Metallurgical recovery relationships defined by 2005/2006 test work are applied to the interpolated copper, gold, and silver grades to obtain metallurgical recovered grades. The recovered grades items (RCU, RAU, and RAG) are used in the economic pit limit studies and long range strategic planning, where revenue calculations are required and a single recovery factor by metal is not applicable. Individual blocks are checked in all the zones at various grade bins to verify the recovery calculations are carried out correctly.

19.2.2.1.5 Net Smelter Return (NSR)

Cutoff grades are determined using the Net Smelter Return (NSR) in CA$/tonne which is calculated using Net Smelter Prices (NSP). The NSR (Net of offsite concentrate and smelter charges and onsite plant recovery) is used as a cutoff item for break-even ore/waste selection and for the grade bins for cashflow optimization. The net smelter price is based on metal prices, US$ exchange rate, and offsite transportation, smelting, and refining charges, etc. The metal prices used are:

- Cu CA$/lb = 1.130
- Au CA$/gm = 13.87
- Ag CA$/gm = 0.191

The NSR formula is:

NSR = RCU(%) x 22.046 x $1.13/lb + RAU(g/t) x $13.87/gm + RAG (g/t) x$ 0.191/gm

19.2.2.1.6 Mining Loss And Dilution

The Galore Creek deposits are to be mined with large truck/shovel operations, at an nominal ore mining rate of 65,000 tpd feeding a conventional copper concentrator. The mining is described as typical hard rock bulk mining method. Large equipment will be used and high mining rates are planned to ensure the lowest possible unit costs for mine operations. Selective mining methods will not be used. The waste and ore will require blasting and typical ore grade control methods using blasthole sampling and possibly blasthole Kriging will be used to determine cut-off grades and digging control limits for the mining shovels. Blast heave, the lack of loading selectivity, haul back in the trucks, and stockpile reclaim will create some ore loss (mining recovery) and dilution as the material moves from Insitu modeled resource to ROM mill feed. Since the ROM mill feed determines the production schedule and revenue stream for the project, proper evaluation of the mining loss and dilution is required. The definition of the mining parameters used in the reserves calculations are also a NI 43-101 reporting requirement.

Mining Recovery and Dilution Parameters are required to account for the following:

- Dilution of waste into ore where separate ore/waste blasts are not possible
- Loss of ore into waste where separate ore/waste blasts are not possible
- General mining losses during hauling and repeated handling.

Calculations are made in the mine planning 3DBM (gc0615.da1) to account for isolated waste and ore blocks in the mine planning for the Galore Creek feasibility study.

The MineSight routine "gndiln.dat" has been used to calculate the number of waste blocks which touch an ore block (measured and indicated) in the 2006 3D Block Model. Ore blocks are restricted to measured and indicated blocks, and a cutoff NSR grade of CA$3.82 /tonne is used. This routine only assesses neighbour blocks on a bench and does not consider the bench above or below. Whole block dilution should cover most of the effects of material above and below whereas this evaluation is considering the recovery and dilution effects as the shovel extracts ore while it advances across a bench. Therefore the "gndiln.dat" routine should create a reasonable evaluation across the bench. The program checks for waste blocks east, west, north and south of the ore block.

Waste	Waste	Waste	Waste	Waste
Waste	Waste	Waste	Waste	Waste
ORE	ORE	ORE	ORE	ORE
ORE	ORE	ORE	ORE	ORE

Figure 19-13: Single Edge Contacts

Some ore blocks as shown in the figure above have only one contact. Contact like this will be blasted as separate ore and waste blocks with very little dilution from neighbouring blocks.

Waste	Waste	Waste	Waste	Waste
Waste	ORE	ORE	ORE	Waste
Waste	ORE	ORE	ORE	Waste
Waste	Waste	Waste	Waste	Waste

Figure 19-14: Single and two Edge Contacts

More dilution will happen on the ends of the blasts where there are two or more contact blocks as shown above. The illustrations below show instances where 2, 3 and 4 contact blocks occur.

Waste	Waste	Waste
Waste	ORE	Waste
Waste	Waste	Waste

ORE1 block has 4 waste edges

Waste	Waste	Waste	Waste	Waste
Waste	ORE	ORE	ORE	Waste
Waste	Waste	Waste	Waste	Waste

ORE2 and ORE4 block have 3 waste edges
ORE 4 has 2 waste edges

Figure 19-15: Four and Three Edge Contacts

It is assumed that blocks with a DLNE1 of 1 will have minimum mining dilution and almost 100% mining recovery because they will be mined in separate blasts. The other contact blocks will be diluted along their edges since they will be mixed in the blast throw along their edges and the shovels will not be able to separate the material as defined in the pre-blasted block. More edges will result in more dilution.

The 80 or 100 tonne class dippers on the proposed shovels will be attempting to separate the ore and waste after the blast (in mixed blasts). These buckets are then 30 m^3 and 55 m^3 respectively, with an approximate lip width of 6 m, therefore approximately 2.5 m deep X 2 m high. Assuming the operator can dig into a block $^{\pm}$1 dipper depth, the selectively in the blasted muck pile is approximately 10% for each contact edge. For each edge of the DLNE1 blocks the following then applies

Each block of ore has an estimated (15 m x 25 m x 25 m x 2.67 t/m^3) = 25,031 tonnes

Therefore for each edge of a block:

(2.5 m x 25 m x 15 m x 2.67 t/m^3) = 2,503 tonnes

This will result in the following dilution:

- 2 edge blocks. 2 X 2,503 = 5,006 tonne or 5,006/25,031 = 20%
- 3 edge blocks. 3 X 2,503 = 7,509 tonne or 7,509/25,031 = 30%
- 4 edge blocks. 4 X 2,503 = 10,013 tonne or 10,013/25,031 = 40%

The amount of edge dilution is dependant on how isolated individual blocks are on the bench. At higher cut off grades, presumably 'ore' blocks are less contiguous, whereas at lower cutoff grades, the 'ore' boundary should merge into groups of blocks (with fewer contact edges).

The total dilution within the PEA pit area at the design basis cutoff grade is estimated to be 3.6% and is not sensitive to cutoff grade.

Since the dilution material on the contact edge of the blocks described above is mineralized, it will have some grade value. The dilution grades are estimated by determining the grades of the envelope of waste in contact with ore blocks inside the PEA pit delineated area. This is estimated by statistical analysis of grades in Measured and Indicated blocks below the design basis cutoff of CA$3.82/t. The number of waste blocks in the cutoff grade bin of CA$2.50/t to CA$3.82/t is 40%, and it is reasonable to assume that waste in contact with 'ore' blocks will be this material. The weighted dilution grades are therefore:

- NSR 3.032 CA$/t
- RCU 0.089 %
- RAU 0.048 g/t
- RAG 0.574 g/t
- CU 0.154 %
- AU 0.122 g/t
- AG 1.875 g/t

19.2.2.1.7 Isolated Waste Blocks

In the blocks defined by blast patterns (approx 6mX6m or 8mX8m) isolated waste blocks surrounded by 'ore' blocks with 3 or 4 ore block edges will be likely be mined as ore since on a blast pattern scale of selectivity these blocks would not be separable. However on the scale of the 3DBM isolated waste blocks are large enough and are deemed to be separable. The whole blocks will not report to ore but the edges will be included in the neighbouring ore blocks as dilution. The previous description of ore dilution has covered this occurrence.

19.2.2.1.8 Mining Recovery

Some loss of 'ore' from the resource model occurs from dilution on a whole block basis, where the whole block grade drops below the specified cut-off grade. In future operations, when grade control from closer spaced blast holes is used, presumably less tonnes than the full resource model blocks will drop below the cut-off grade. Therefore whole block dilution on the large blocks is already causing some ore loss. In addition to this, additional ore loss is applied to account for carry back, stockpile reclaim, and misdirected loads. Total Mining losses are included at 2.4% as an allowance.

19.2.2.1.9 Diluted Grades For LG economic pit limits

The economic pit limit routine (MSEP) does not have parameter inputs for loss and dilution. Therefore to account for the costs and loss of revenue in the economic pit limit calculations due to dilution and loss, a separate set of items has been created in the mine planning 3DBM for the Lerchs Grossman pit expansions with the MSEP routine:

LORE% = ORE% - 2.4% Mining loss

$$\text{LG Diluted Grades} = \frac{\text{Recovered Grade} + \%\ \text{Dilution}/100 \times \text{Dilution Grade}}{1 + \%\ \text{Dilution}/100}$$

19.2.3 Economic Pit Limits, Pit Designs

19.2.3.1 Introduction

The economic pit limit is determined using the MSEP optimization routines in MineSight which are based on the Lerchs Grossman (LG) algorithm. The LG algorithm runs against the 3D Block model, evaluating the costs and revenues of the blocks within potential pit shells. The routine uses input costs, net smelter prices, plant recoveries, and overall slope angles, and expands downwards and outwards from previous interim economic 3d surfaces, until the last increment is at break-even economics.

Additional cases are included in the analysis to evaluate the sensitivities of prices, and slope angles. Block discounting for time value is also evaluated to determine the NPV effect of the delay between earlier stripping costs to the revenue released from deeper ore.

The economic pit limit is determined for each pit area using data from the mine planning 3D Block model described above, and MineSight Gridded Surface Files (GSF) created for each pit area.

19.2.3.2 Pit Slopes

A Geotechnical Feasibility report Open Pit Slope Design by BGC gives details for interramp slopes and overall open pit slopes for each of the PEA proposed open pits. A design factor of safety of 1.3 has been used for the interramp and overall slope analyses. BGC have estimated bench face angles to be 74º – 80º for the sheet-fractured and massive rock. Design berm widths are 11 m for the 30 m bench height. Design interramp angles can be as steep as 60º for specific wall orientations in the massive unit where 30 m bench heights are used. The contrasts between the geotechnical units of this site require a compound overall slope to maximize the design overall angle. Typically, the achievable interramp angles in the massive unit will be greater than those in the sheet-fractured unit (for similar slope heights).

To determine the overall design angles for the proposed pits, walls representative of a group of design sectors and specific critical walls were selected for two dimension limit equilibrium stability analyses. Critical walls in the proposed pits were identified based on:

- Major structures (faults) within or near the proposed walls
- Wall height

The design overall pit slope angles can be controlled by:

- Overall slope stability
- Achievable interramp geometry
- Achievable bench geometry

In the situations where the design overall slope angles are controlled by bench or interramp geometry or where interramp slopes are controlled by bench geometry, the factors of safety for the whole slope may exceed the design factor of safety (1.3).

The BGC recommended slope angles are adjusted for the economic pit limit study to include estimated highwall haul road widths. Highwall haul road locations are estimated using the PEA pit designs as a basis.

19.2.3.3 Incremental Mining Costs

Mining unit costs are made up of fixed and variable components where the major variable component is haulage cost due to the different cycle times and resultant truck productivities from the different pit benches. To facilitate how the MSEP software handles variable unit costs, in this study the fixed component of the haul costs are also determined as equivalent to mining benches at the elevation of the pit rim and this ‘fixed’ amount is added to the other fixed cost to make up the ‘Base Cost’. The remaining variable cost is the additional mining cost (or extra incremental cost) for haulage from mining benches above or below the elevation of the pit rim.

19.2.3.4 Sensitivity Cases

The economic pit limits are based on the current cost and metal price assumptions, but are applied to ~20 years of mine life. Since these economic parameters are estimates, the sensitivity of the ultimate economic pit limits need to be evaluated. This is done by varying the economic parameters in series of cases. The pit shells from these cases are also used to select pit pushbacks or phases. For each case being tested the series of LG pit shells are determined by keeping mining costs constant and varying the estimated net smelter metal prices (NSP).

Five cases of economic pit limit sensitivities are tested for each pit area. These are:

- Pit Design Criteria
- $^{+}5^{0}$ Pit Slope Angle.
- $^{-}5^{0}$ Pit Slope Angle
- 6% Time value Discounting
- 10% Time value Discounting

19.2.3.5 Economic Pit Limits – Central Area

The economic pit limit for the Central pit area has been chosen at the previously described metal prices. There is opportunity to significantly increase the economic pit limit at higher prices. Any future opportunities for expansion of the pit limits are left for future pit phases beyond the scope of this study. Future expansions will not affect the current planned plant location. The slope sensitivity results indicate that there is insignificant sensitivity to increase the economic pit limit at higher slope angles, but that there is a significant sensitivity to reducing the economic pit if flatter slopes are required. The ultimate pit limit is not sensitive to time value discounting.

19.2.3.6 Economic Pit Limits – Junction Area

The economic pit limit for the Junction pit area has been chosen at the previously described metal prices. There is limited opportunity to increase the economic pit limit at higher prices. The slope sensitivity and time value discounting sensitivity results indicate that there is insignificant economic pit limit sensitivity due to slope angles or time value discounting.

19.2.3.7 Economic Pit Limits – South West Area

The economic pit limit for the South West pit area has been chosen at the previously described metal prices. There is opportunity to increase the economic pit limit at higher prices. The slope sensitivity results indicate that there is insignificant sensitivity to increase the economic pit limit at higher slope angles, but that there is a significant sensitivity to reducing the economic pit if flatter slopes are required. The ultimate pit limit is not sensitive to time value discounting.

19.2.3.8 Economic Pit Limits – West Fork Area

The economic pit limit for West Fork Pit area has been chosen at the previously described metal prices. The economic pit limit is sensitive to slope angles and time value discounting.

The economic pit limit mines out the shallow high grade zone. The grade interpolation has left a waste layer between the upper and lower zones. The lower zone requires metal prices greater than the design criteria before it becomes economically mineable.

19.2.4 Detailed Pit Designs

GR Tech has completed feasibility level pit designs demonstrating the viability of accessing and mining economically mineable resources at the Galore Creek site. The designs are developed using MineSight® software, geotechnical recommendations by BGC Engineering Inc, regulated standards for road widths, and minimum mining widths based on efficient operation for the size of mining equipment chosen for the project.

BGC's Open Pit Report forms the basis of geotechnical guidance for the pit designs. The following description of the pit design process summarizes germane points from the BGC Open Pit Report with respect to variable wall angle coding and describes the strategies used for pit development.

19.2.4.1 LG Phase Selection

The LG pits previously discussed are used to evaluate alternatives for determining the economic pit limit and the best pushbacks or phases to begin detailed design work on. LG pits provide a geometrical guide to detailed pit designs. Among the details will be the addition of roads and bench access, removal of impractical mining areas with a width less than the minimum, and insuring the pit slopes meet the detailed geo-technical recommendations.

The 100% price cases LG pits discussed above are the economic pit limits for Central Pit, Junction Pit and South West Pit. Small pit phases exist within the economic pit limits that are economically mineable at lower metal prices. When considering pit economics, these lower price case pits have higher NSR values due to lower strip ratios or better grades than the full economic pit limit. Mining these pits as phases, from higher NSR to lower NSR, maximizes revenue and minimizes mining cost at the start of mining operations and thereby shortens the project capital payback and improves the project cash flow.

The first phases or starter pits requires some practical mining constraints. The starter pits must:

- Provide sufficient ore to sustain the mill operations for at least 1 year (23.7 mtpa).
- The pit should not be too narrow to avoid excessive vertical bench mining rate.

The pit areas are examined to find the lowest LG Price Case that can sustain mining operations.

Waste from the starter pits must also be pre-stripped to expose ore for plant start up. For the Galore Creek project, pre-stripping must also provide significant quantities of construction material, especially for the starter tailings dam. Costs are reduced and the project NPV is improved, if the construction material is mined from the earlier pit phases, rather than an outside borrow pit. This also has a significant influence on the size of the pits selected for the starter pits.

19.2.4.1.1 Central Pit Phase Selection

Examination of the Central LG pits show three potentially separate pit areas. Keeping the mining areas separated enables the scheduling of higher grade pit areas independently within Central Pit allowing further improvements to project NPV.

The LG price cases, after restricting the pit limit search to the three pit sub areas for the Central Pit area are illustrated in the section view below.

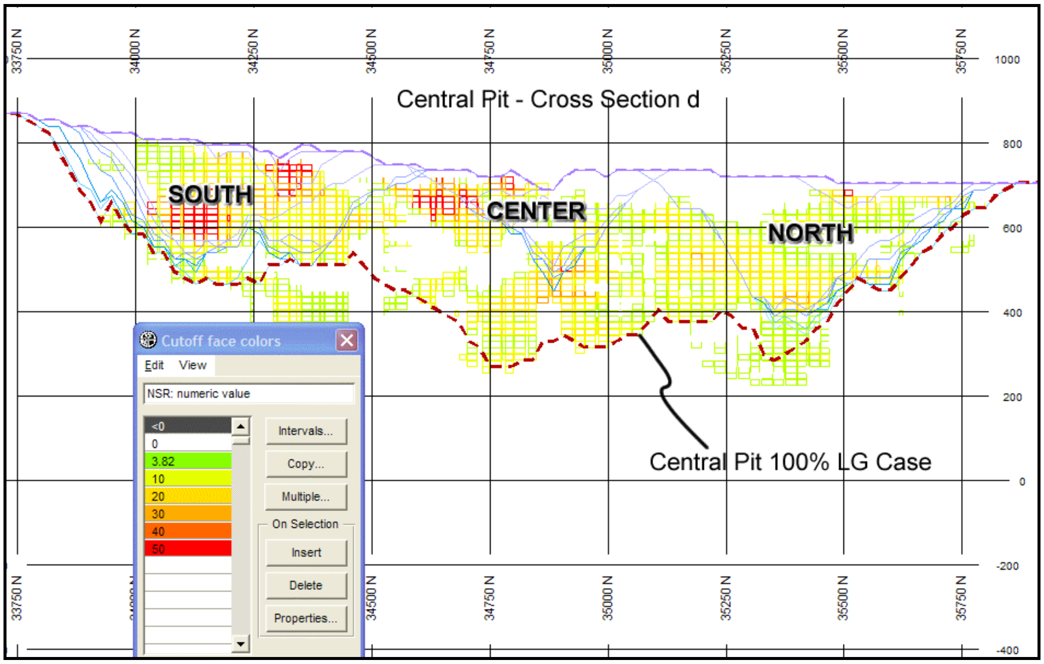


Figure 19-16: Central Pit LG Cases After Restricting the Pit Limit Search to the Sub Pit Areas

Reserve tables for the sub areas are analysed to find the lowest LG price case that meets practical mining requirements for each central sub-area. Waste rock for tailings dam construction will be mined inside the starter pit perimeters and so the estimated pre-strip waste and ore is removed prior to evaluating the pit suitability as a starter pit.

The amount of material required for the pre-strip/construction is larger than can be supplied by any single starter pit alone. Therefore the pre-strip pits are designed based on the amount of waste material required, initial bench access during start-up, access to dump/construction sites, and minimized mining costs. This pre-production material should all be mined within the boundary of the earlier pit phases. To follow the sequence of optimized NPV phases, the most economical starter pit is selected after pre-strip material is removed. Profit per tonne ore is then calculated for the material inside the pit shells after Prestrip removal. The profit per tonne, remaining ore reserve, and estimated vertical advance rate of the mining benches are then examined to find the lowest LG price case that meets practical mining requirements.

The resultant pre-production/starter pits are summarized below.

Table 19-27: Pre-Production/Starter Pit Summary

Prestrip Mining Source *				
	Waste (Mt)	Ore (Mt)	LG Price Case	Bottom Bench (m)
Central Pit - North	81	5	40%	675
Central Pit - Centre	13	1	60%	690
Central Pit - South	13	2	60%	765
TOTAL	106	8		

* These quantities are deducted from the LG Shells

Starter Pit Values After Prestrip				
	Profit * CA$/tonne Ore	Ore (Mt)	Years	Vertical Rate (benches/yr)
Central Pit - North	13.3	88	3.7	5.9
Central Pit - Centre	16.1	46	1.9	8.3
Central Pit - South	14.9	46	1.9	7.3

* Net revenue minus cost for pit shell

It may not be possible to use all of the mining equipment in a single starter phase (as illustrated by the high vertical advance rate). The high vertical mining rate shown in the south and center phases will be reduced by scheduling mine production from multiple phases in any given mining period.

After the starter phases, the phase sequence is generally to the eastern ultimate limit, then to the western ultimate pit limit. The eastward push back has less strip ratio and establishes the final pit access earlier in the pit sequencing. The final westward pushback is deeper and at a higher strip ratio. The final phase is based on the remaining material to complete the Central Pit 100% LG Price Case. The east and west pushbacks are further subdivided to provide better ore release in the project schedule. Experience gained in the PEA study influenced the size and placement of these final pit phases.

The selected phases also provide sufficient mining minimum width between the starter phases and the east and west final walls.

The sources of the Central Pit phases are summarized below:

Table 19-28: LG Price Cases Selected for Central Pit Phases Detailed Design

Pit	LG Case
Central Pit - South Starter	South 60 % Price Case
Central Pit - Center Starter	Center 60 % Price Case
Central Pit - North Starter	North 40 % Price Case
Central Pit - East Intermediate	Total 60% Price Case
Central Pit - Ultimate	Total 100% Price Case

19.2.4.1.2 Junction Pit

The Junction LG cases show separate North and South Pits that can be mined independently. The 50% price case for Junction is a suitable starter phase, leaving sufficient mining width on the high wall in Junction North for an incremental pushback. There is 20 Mt of ore in the 50% LG with a strip ratio of 2.4

compared to a strip ratio of 3.1 in the economic pit limit. The starter pit is in North Junction and does not extend to South Junction. South Junction is therefore considered a single phase.

19.2.4.1.3 South West Pit

The 70% price case for South West is a suitable starter phase, leaving sufficient mining width on the high wall for an incremental pushback. There is 26Mt of ore in the 70% LG with a strip ratio of 0.8 compared to a strip ratio of 1.3 in the South West economic pit limit.

19.2.4.2 Dynamic Variable Wall Angles

Pit designs for Galore Creek are constructed honouring inter-ramp wall angles, a fixed excavation face angle (80 degrees) and variable safety berm widths (minimum 11 metre width).

The MineSight ® 3D block model item SLOPE for Galore Creek is coded to conform to the maximum inter-ramp angles recommended by BGC for defined geotechnical design sectors. These sector maximums are set assuming drained conditions and a wall height less than 390 metres.

Geotechnical design sectors are developed on the basis of rock type, quality, and orientation characteristics (geotechnical units), hydrological features, and anticipated orientation of the pit walls within geotechnical units.

19.2.4.3 Pit Expansions

For the feasibility study, pit shells of variable size and value were generated in the MineSight ® MSEP routine using the Lerchs Grossman (LG) technique and variable economic conditions. Ultimate pit size is based on the modeled resource and agreed economic and mining assumptions for ore recovery and costs. Similar to the detailed pit expansions, the LG pit shells for the feasibility study also use variable wall angles based on design sectors, but overall wall angle values have been modified to account for access roads. To approximate the final pit size and orientation including pit access, assumptions have been made concerning the general location, length, and number of ramps required for access to the ultimate pit bottom.

Detailed pit expansions are then founded on the economic pit bottoms identified by the LG shells, the design standards for minimum mining widths, single lane and dual lane road widths, access strategy, and inter-ramp angles coded in the model item SLOPE based on the description in the preceding section.

19.2.4.4 Design Results

The following detailed pit designs are developed from the LG pit limits, potential pit phases, and design considerations reviewed above.

19.2.4.4.1 Central Pits

The resultant six cumulative pit phase designs for Central and the 6 auxiliary pits are illustrated in the figures below. These are shown relative to the crusher and plant site. Topography contours are shown every 15 m. at bench grade elevations.

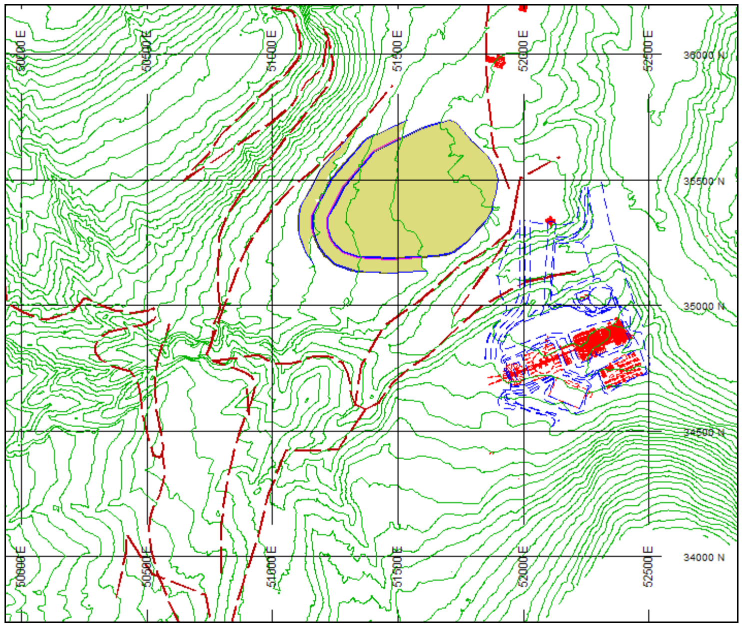


Figure 19-17: Central Pit - North Starter Phase - C616

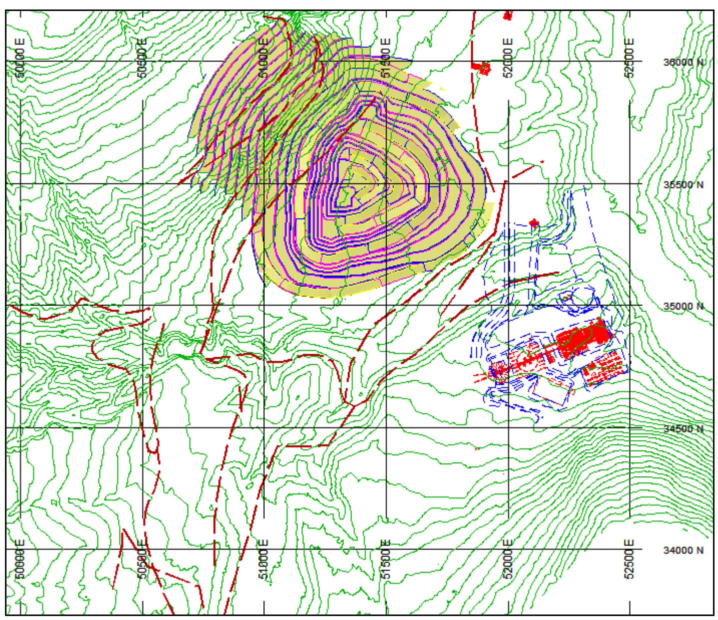


Figure 19-18: Central Pit - North Phase - C626

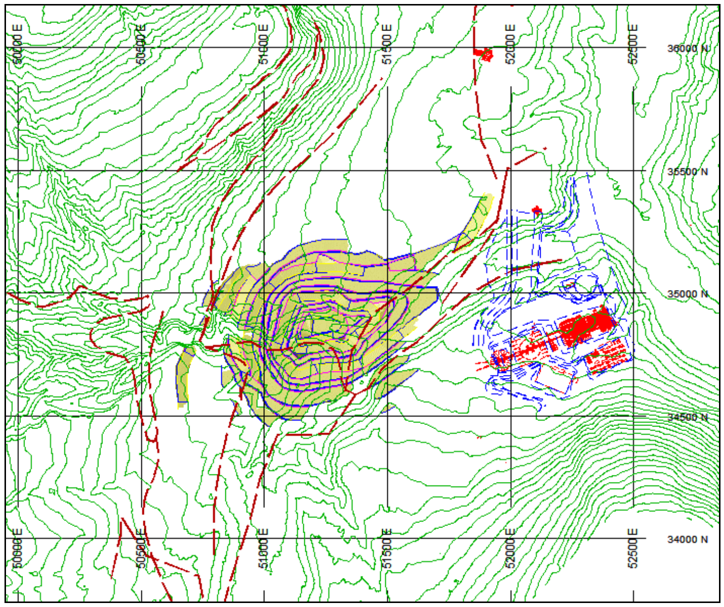


Figure 19-19: Central Pit - Center Phase - C636

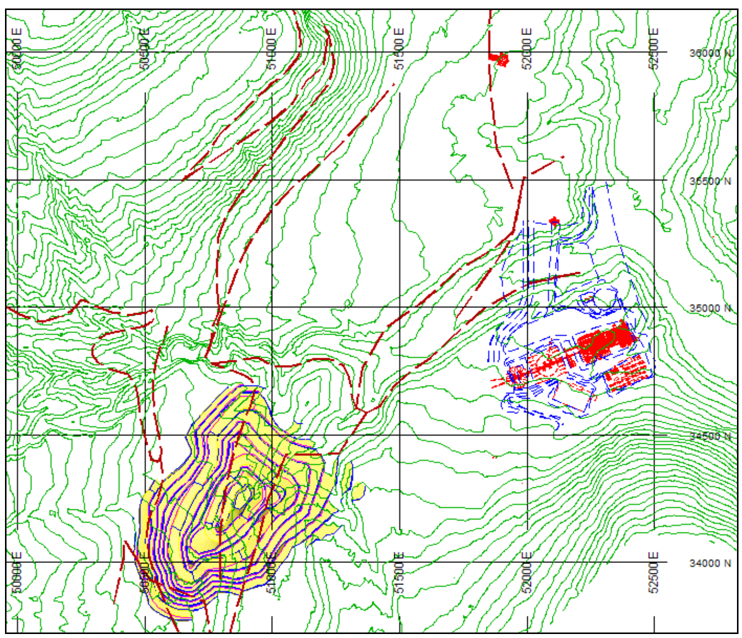


Figure 19-20: Central Pit - South Phase - C646

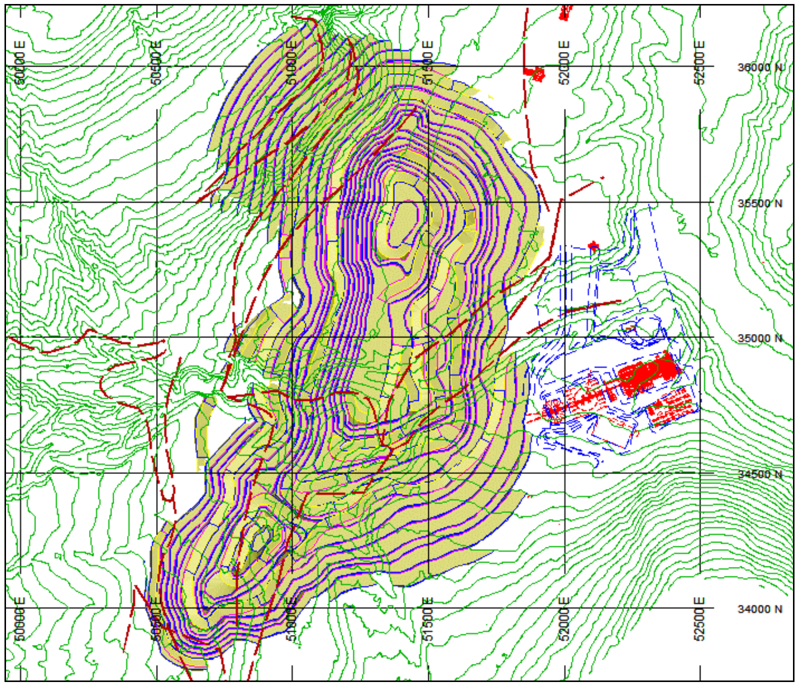


Figure 19-21: Central Intermediate Phase - C656

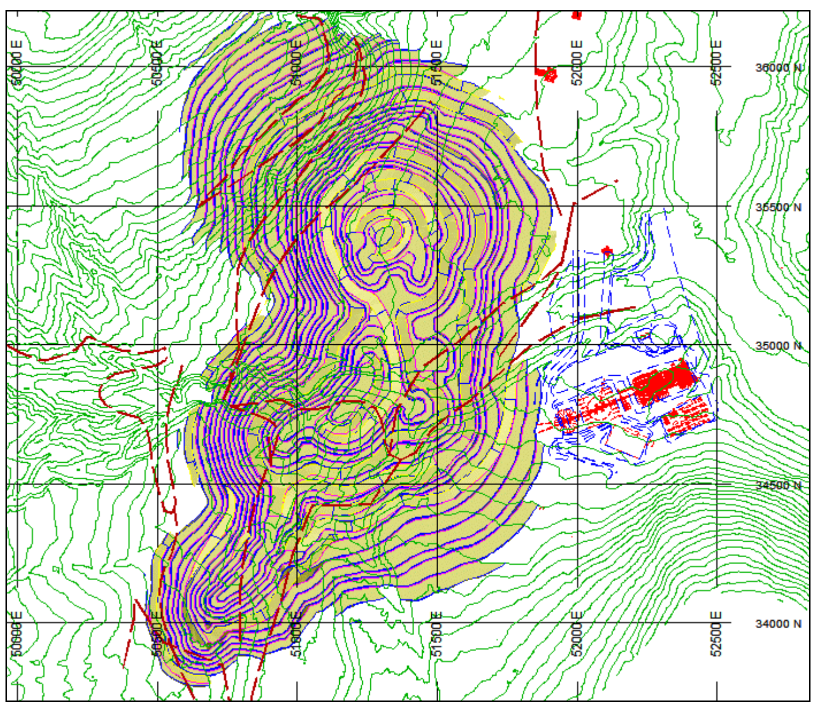


Figure 19-22: Central Pit - Ultimate Phase - C666

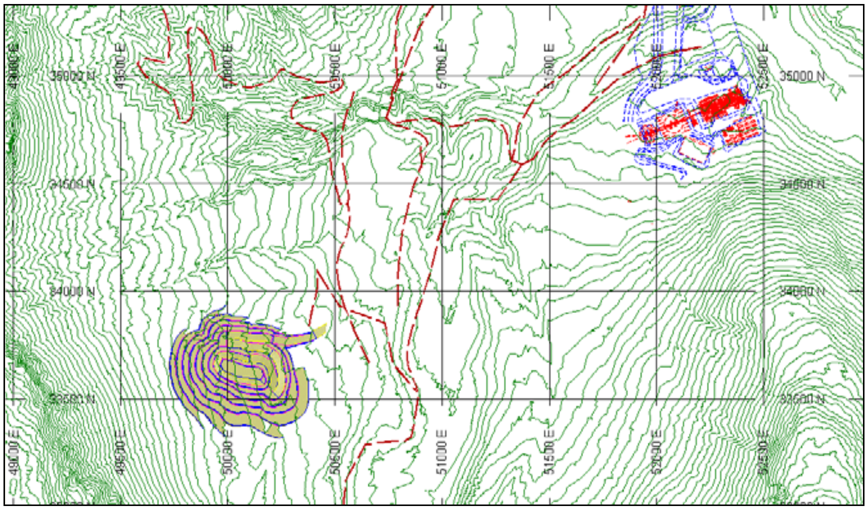


Figure 19-23: South West Pit – Phase 1 (S616)

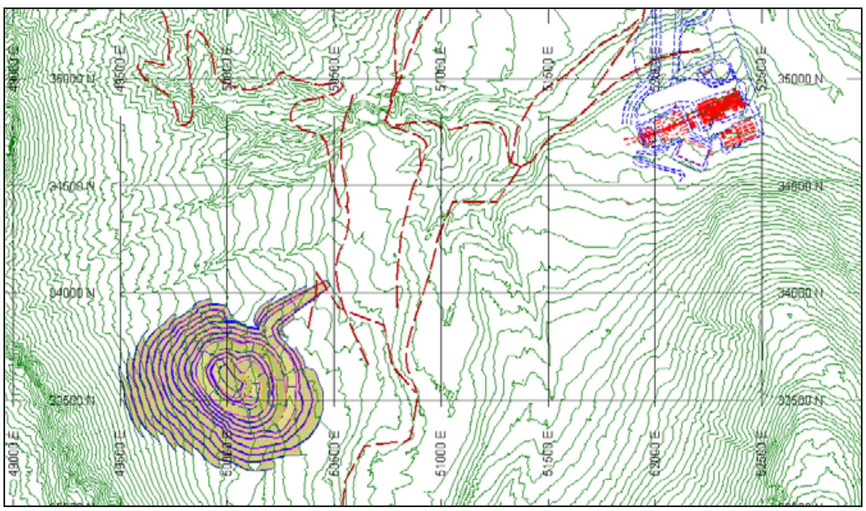


Figure 19-24: South West Pit – Phase 2 (S626)

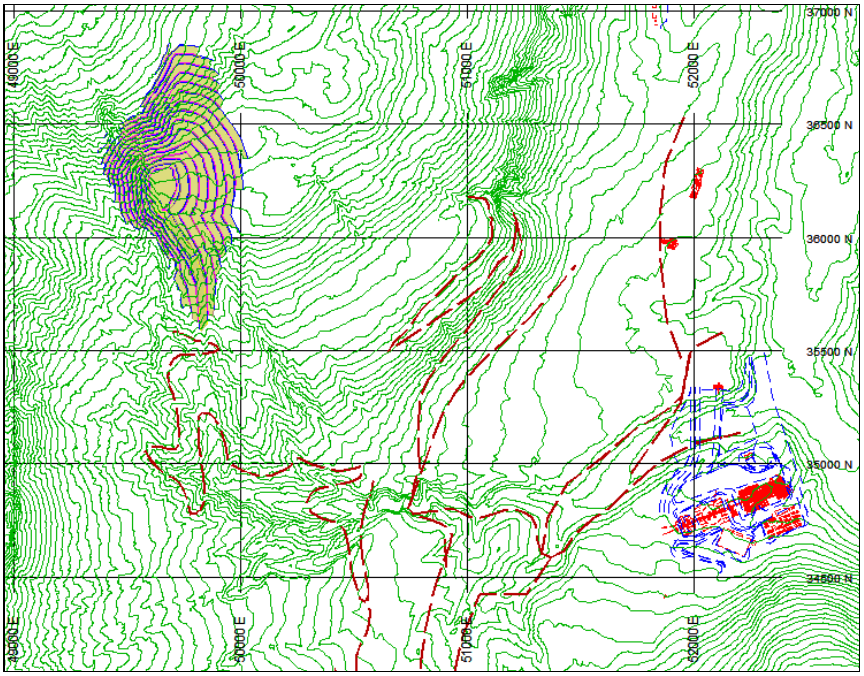


Figure 19-25: Junction North Starter Pit – J606

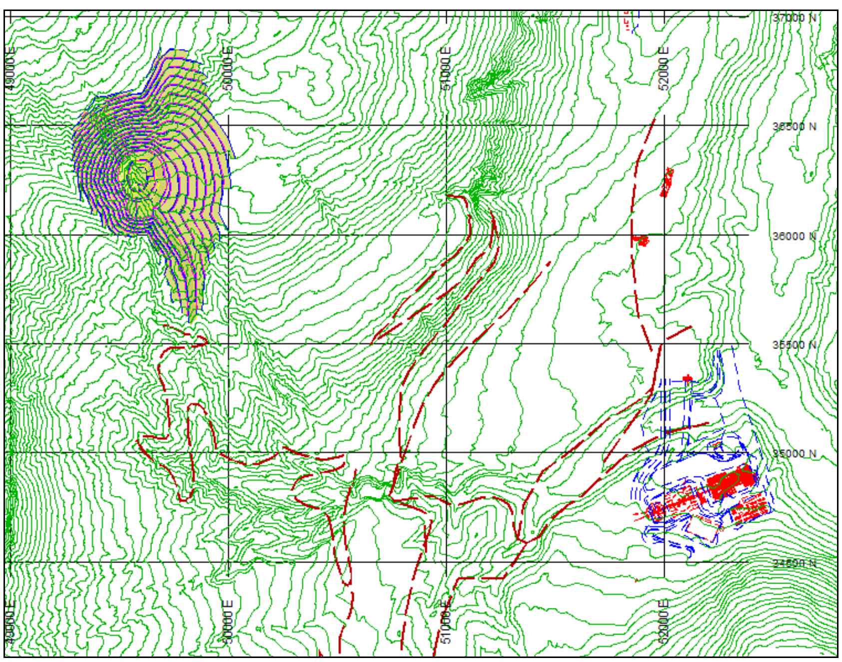


Figure 19-26: Junction North Ultimate Pit – J616

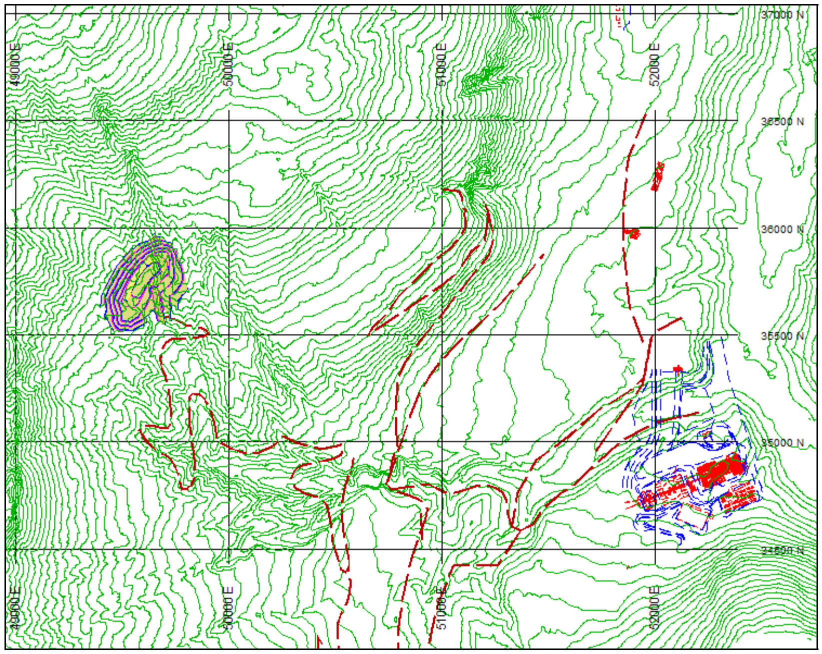


Figure 19-27: Junction South Ultimate Pit – J626

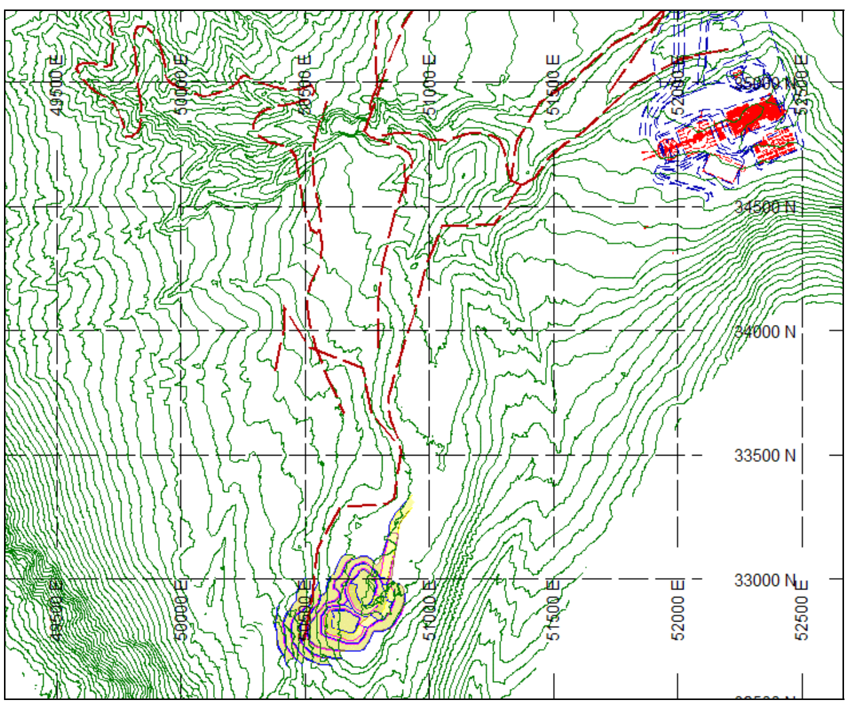


Figure 19-28: West Fork Pit Pit – WF616

19.2.4.5 Geotechnical Checks

The above pit designs have been reviewed by BGC with respect to the recommended slope design parameters and subsequent slope stability. BGC has confirmed that the pits meet or are within, the geotechnical design criteria.

19.2.4.6 Pit Reserves

The construction needs of the Galore Creek Project require pre-production waste rock mining in excess of 100 Mt. All of the tailings dam ballast rock and core is planned from within the starter pit perimeters. This has a positive contribution to the project value by avoiding the high incremental cost of mining the required rock from borrow sources.

C616 is completely mined out by the end of pre-production, C626 is partially mined to 780 m, C636 is partially mined to 690 m and C646 is partially mined down to 765 m. to meet the construction needs of the project. The Topography surface at the end of preproduction is used as the start surface for the production schedule. The reserves for the production schedule are derived from this surface.

Table 19-24 lists the waste and ore reserves for each incremental pit phase. The pre-production material is listed in a separate table and has been subtracted from all the subsequent phase reserves listed. Note any ore listed in the pre-production areas is considered stockpiled until the crusher starts up. The pit reserves are estimated in Table 19-24 use the MineSight ® PITRES routine with the following parameters:

- Ore waste NSR cutoff grade of CA$ 3.82 / t
- Dilution = 3.6 %
- Dilution grades:
    - CU = 0.154 %
    - AU = 0.122 g/t
    - AG = 1.875 g/t
- Mining Loss of 2.4 %
- Ore% item is used
- SG from the 3DBM value and defaults of 2.67 t/m^3 for ore and waste respectively
- Topo item was not used as the pit partials are clipped to the pre-production topography.
- Dilution is applied on all blocks - Contact dilution is not used.

20. Other Relevant Data and Information

20.1 Processing

Section 20.1 was prepared by Mr. Hoe Teh,P.Eng. Hatch.

20.1.1 Summary

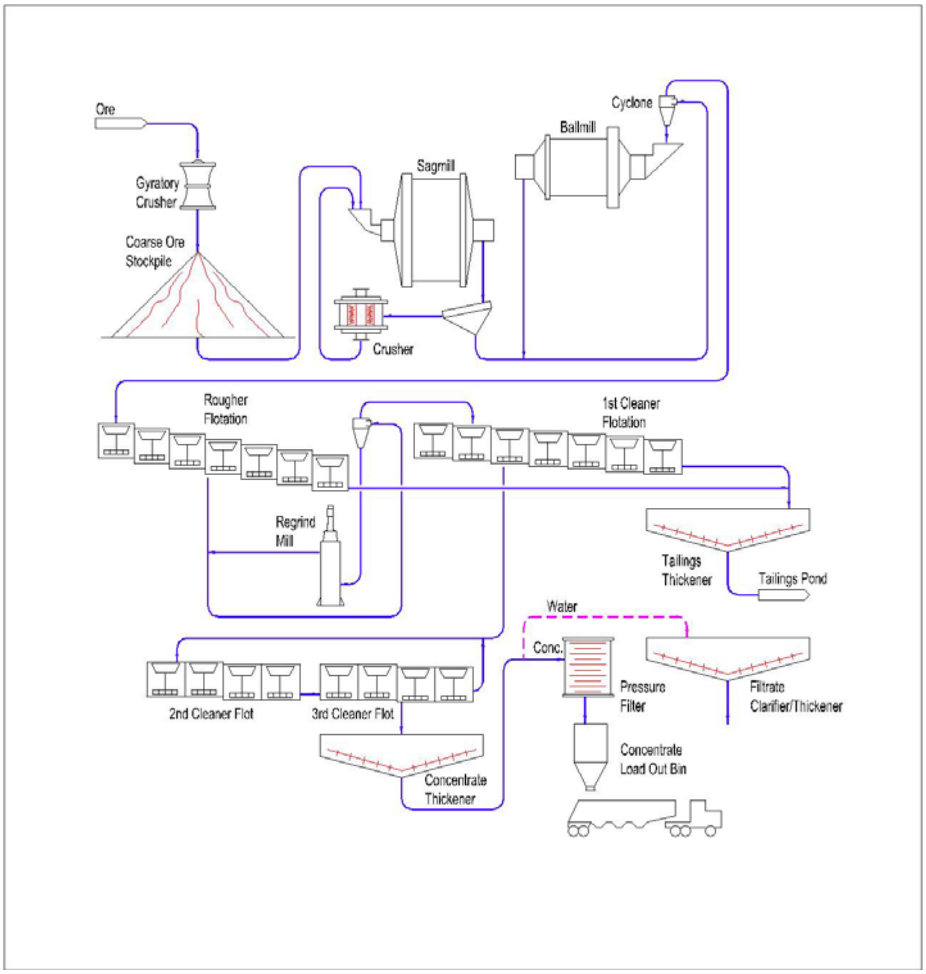
A conventional crushing, grinding and flotation process plant is proposed for the project, utilizing standard unit processes and equipment. The process design criteria is based on a combination of testwork results on Galore Creek ores, experience from other similar operations and industry standards.

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 process can be used for all ore types.

Figure 20-1 illustrates a simplified flowsheet for the process which will have a nominal throughput of 65,000 dry tpd. The concentrator is able to process 71,500 dry tpd with an ore hardness of 16.5 kWh/t or less. Complete flowsheets and mass balance are available in the Feasibility Study.

The concentrator will be located adjacent to Central pit and will be built solely on cut. Ore will be dumped into the primary crusher located 500 m north of the concentrator. The crushed ore will then be conveyed to a coarse stockpile where it will subsequently be fed to the grinding circuit which consists of one SAG mill, two ball mills and two pebble crushers. Once the ore has been ground to 200 microns, it will be fed to the rougher flotation cells via the overflow of a cyclopac. All of the rougher concentrate will be further reduced in size to 40 microns using three vertimills and then pumped to three cleaner flotation stages while the rougher tails will report to the tailings thickener. The third cleaner concentrate will be thickened, stored and then pumped via a 135 km pipeline to the Bob Quinn site where it will be dewatered. Copper concentrate at a nominal tonnage of 1,722 tpd and 28% copper grade and 8% moisture will be the final product. The plant will operate 24 hours per day, 365 days per year with scheduled downtime for equipment maintenance.

Figure 20-1: Simplified Flowsheet



20.1.2 Design Criteria

Table 20-1 lists an abbreviated version of the design criteria listing the key parameters for the project. The full design criteria can be found in the Feasibility Study. The legend at the top of the table designates the source of information in each line. This ranges from client input to benchmarking with other similar operations which was used in conjunction with experience to ensure that the data obtained through testwork was reasonable.

Source:

1. Client
2. Vendor
3. Experience
4. Calculation
5. Mass Balance
6. Testwork
7. Other
8. Benchmark

Table 20-1: Indicate Sources Design Criteria Document

PROCESS DESIGN CRITERIA

PROJECT	: GALORE CREEK FEASIBILITY STUDY	
PROJECT NO	: 317882	
CLIENT	: NOVAGOLD	
DATE	: 18 September, 2006	
REV	: A	

SOURCE
1 - CLIENT
2 - VENDOR
3 - EXPERIENCE
4 - CALCULATION
5 - MASS BALANCE
6 - TESTWORK
7 - OTHER
8 - BENCHMARK

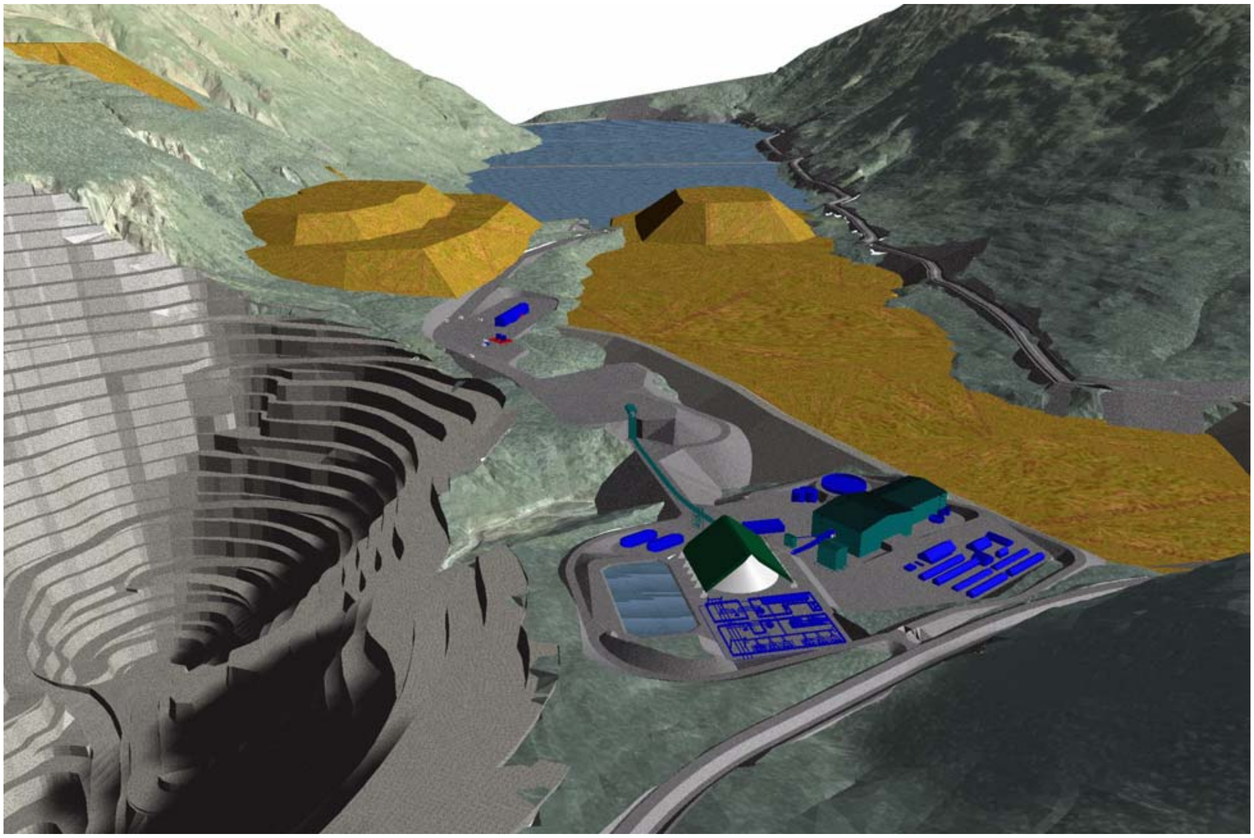
			SOURCE	COMMENT
GENERAL				
Operating Time	d/a	**365**		
Operating Time	h/d	**24**		
Ore Production	t/d	**65,000**	1	
Overall Plant Availability, nominal	%	**92**	1	

			SOURCE	COMMENT
ORE CHARACTERISTICS				
Ore Solids Density	t/m^3	**2.67**	6	JK Tech - SMC Test Report, February 2006
Ore Bulk Density	t/m^3	**1.60**	7	Jim Gray - email Feb 9th
Ore Moisture	%	**5**	1	Includes slimes around the ore
Ball Mill Bond Work Index		**16.5**	6	Average of G&T and SGS Testwork
Abrasion Index		**0.173**	6	SGS Standard bond Abrasion Test
GENERAL PROCESS PLANT DATA				
Overall Plant Availability	%	**92**	1	
Concentrate Mass Recovery	%	2.65	6	
Concentrate Production	dry tpd	1722	5	
Concentrate Moisture	%	**8**	3, 6	
Concentrate Copper Grade	%	**28.0**	1	
Tailings Copper Grade	%	**0.06**	6	

20.1.3 Ore Storage and Primary Crushing

Figure 20-2 illustrates the entire process from primary crushing through to concentrate pumping and tailings thickening and also the topography on site. The crusher dump grade is at an elevation (above sea level) of 686.5 m while the coarse ore stockpile is at 703.5 m and the concentrator is at 682 m.

Figure 20-2: Galore Creek Plantsite



20.1.3.1 Run of Mine Stockpile

A R.O.M stockpile, which is located just north of the primary crusher, will have a nine day storage capacity and can be transferred via front end loader to the primary crusher dump pocket at a reduced rate. This stockpile will only be used during periods in which there are no haul trucks reporting from the mine such as during times of limited visibility.

20.1.3.2 Primary Crushing

Ore will be hauled from the open pit mining operation to the primary crushing facilities and dumped directly into the dump pocket with a capacity of two haul trucks or 700 t. There will be two dumping locations directly opposite one another and barricades will be installed to ensure safety for the haul truck drivers when approaching the precipice on which the dump pocket lies. A rock breaker will be installed over the dump pocket to break large pieces and avoid plugging. An atomizing spray fog system utilizing compressed air and fresh water will be activated during truck dump to control dust emissions.

The 60″ x 89″, 600 kW gyratory crusher will operate with an OSS of 240 mm and a CSS of 165 mm in order to produce a product size of 80% passing 150 mm. The product will be discharged into a 600 t surge bin. From the bin, the ore will be withdrawn with an apron feeder onto the coarse ore conveyor. A fog system will also be employed at the transfer points below the gyratory crusher to reduce fugitive dust emissions and improve working conditions.

The crushing building will utilize the on site 250 t crane which has sufficient capacity to lift the spider & mainshaft at the required reach. There will also be a lift well which can be used to access equipment which the crane cannot access directly. A dedicated 56 kW compressor and 23 m^3 air receiver will be situated in the building to provide instrument air as well as air for the dust suppression system..

Crushing will operate at a reduced utilization of around 82% and overall availability of 75%. The resulting increased throughput will ensure that the daily production is achieved when the crusher requires maintenance.

20.1.3.3 Coarse Ore Transfer, Storage and Reclaim

The coarse ore conveyor is 568 m long and is driven by two 800 kW motors. The max slope of the conveyor will be 14 degrees and there is single discharge onto the coarse ore stockpile with a live and total capacity of half a day and two days respectively. In order to reduce dust emissions and keep the concentrator area more clean, the coarse ore stockpile will be completely enclosed by an A-frame type structure with dimensions 120 m x 90 m. One half a day was chosen for this operation as it was deemed that a storage capacity any larger than this would not provide a considerable higher availability or ore going to the concentrator.

Reclaim from the coarse ore stockpile will be provided by three inline 1800 mm x 8.5m, 93 kW apron feeders which can handle the full tonnage of 2944 tph using only two of the three feeders. This overcapacity allows full tonnage to be achieved even if one of the belt feeders is undergoing maintenance. The apron feeders will feed onto a single SAG mill feed conveyor. Dust collectors with pickups around the crusher, conveyors and ore transfer points will be installed to minimize fugitive dust in this area.

20.1.4 Concentrator

Figure 20-3 is a 3-D model which depicts the concentrator building looking south.

The Galore Creek concentrator has been designed to process 65,000 tpd of ore containing copper, gold and silver to produce a copper concentrate which is then pumped 135 km by pipeline to the filter plant site at Bob Quinn Lake.

The concentrator building (13,600 m^2) is a pre-engineered structure divided into two main sections: the grinding section, which houses the SAG mill, two ball mills and three vertimills, and the beneficiation section which houses the flotation cells, the reagent storage and concentrate and tailings handling. In addition, the pebble crushing circuit is housed in two smaller buildings adjacent to the concentrator. Testwork shows that by including a pebble crushing circuit in primary grinding, the power requirements can be reduced significantly indicating a portion of mill feed which is resistant to grinding while being more susceptible to crushing. Tailings thickening is done 150 m north of the concentrator building. Reclaim water is pumped from the tailings dam located 8 km north of the concentrator and stored on site in process water tanks.

20.1.4.1 SAG Mill Circuit

Figure 20-4 shows a picture of the SAG Mill circuit looking southwest.

Ore will be reclaimed from the coarse ore stockpile via three apron feeders which will in turn feed a 1,800 mm x 183 m SAG mill feed conveyor. The apron feeders will be controlled by a weightometre on the SAG mill conveyor to achieve a nominal operating rate of 2,944 tph dry ore. The 40 ft (12.2 m) x 24 ft (7.3 m) SAG mill will be driven by a 21.2 MW gearless, variable speed drive which will be cooled using fresh water. Process water will be added to the feed chute to achieve a 70% solids concentration in the mill. The steel ball size used will be 5″ and the steel charge will be 14%. Nominally, the SAG mill will draw 18.0 MW with an ore hardness of 16.5 kWh/t. However, the motor has been designed to handle ore up to a hardness of 19.4 kWh/t while maintaining full tonnage and drawing the full power of 21.2 MW.

SAG mill grates of 75 mm will be used to maximize pebble extraction/crushing. A discharge SAG mill trommel, with 12.7 mm apertures, will be used to minimize the load on the double deck vibrating screen which will also have apertures of 12.7 mm on the lower deck. Undersize from both the trommel and vibrating screen will report to a common SAG mill discharge pumpbox which in turn will feed the ball mill circuit. The oversize ore in the trommel will be fed to the vibrating double deck screen.

The oversize from the screen will be conveyed via three recycle conveyors to the 100 t pebble crusher surge bin. Two magnets and one metal detector will be installed over the pebble recycle conveyors to remove ball chips and other metallic objects to protect the cone crushers from damage. From the surge bin, two belt feeders will feed two cone crushers (HP 800) operating in parallel. The ore feeding the two crushers can be bypassed if maintenance is being done on the crushers.

The 5″ SAG mill steel balls will be stored in a bunker with a capacity of two weeks usage. The balls will be fed via conveyor and discharged onto the SAG mill feed conveyor. A dedicated SAG mill liner handler will obtain liner sections from a jib which in turn will be supplied the liner sections from a dedicated SAG mill liner handler crane located between bays 26 and 27 on the South end of the building. Milk of lime slurry will be added at a nominal rate into the SAG mill while its addition will be controlled into each of the ball mills to achieve the desired pH of 10.

20.1.4.2 Ball Mill Circuit

Figure 20-5 shows a picture of the Ball mill circuit looking north-east.

The product from the SAG mill circuit will be discharged in to a pumpbox and diluted to 55% solids using process water. The transfer size from the SAG mill circuit will be approximately 3,500 microns. The diluted product will then be pumped to a distributor that will equally split the flow to two ball mill discharge sumps – the two ball mill circuits are identical and will operate independently. From the discharge sumps, the slurry will be pumped directly to a dedicated cyclopac in order that any undersize product from the SAG mill circuit is processed directly to flotation. As mentioned, the two ball mills {each 26 ft (7.9 m) by 36 ft (11 m)} will operate independently, each in a closed circuit with a cyclopac (12 x 840 mm) with a circulating load of 300%. The underflow from the cyclopac will report to the ball mill to be further ground. The overflow from the cyclone will have a P80 of 200 microns. Testwork was done at a grind size of both 150 and 200 microns. Sufficient liberation was achieved at 200 micron (see Section 18.4.1) and the power requirements are 2,500 kW less for grinding to 200 microns vs. 150 microns.

Both cyclopac overflow streams will have a dedicated double stage sampling station which will feed a reduced metallurgically accurate sample to a single analyzing station which will measure both the metal content and particle size distribution of the stream.

There will be one crane and one liner handler for both ball mills. Two bunkers will store two weeks usage worth of 3″ ball mill balls. The balls will be fed to the ball mill circuit via two conveyors that will discharge the balls onto the two cyclopac underflow launders which feed both ball mills.

20.1.4.3 Flotation and Regrind

Figure 20-6 shows a picture of the Ball mill circuit looking north-west.

Each of the ball mill cyclopac overflows will be fed to a bank of seven 200 m^3 rougher flotation cells providing 23 minutes of residence time and giving a mass pull of about 9.1%. The rougher concentrate from the two banks will be pumped separately to the regrind circuit while the tailings will be gravity fed to the tailings thickener.

The rougher concentrate will be ground in three 1,119 kW vertimills operating in closed circuit with a single cyclopac to achieve a product grind of 80% passing 40 μm. The recirculating load will be 250%. The underflow from the cyclopac will feed a distributor that will split the flow into three streams – each gravity fed to their respective vertimills.

The vertimill power requirement was estimated using the bond ball work index of the ore (average of 16.5 kWh/t) and using 70% of the energy requirement that a ball mill would require for the same duty. The 70% factor accounts for the higher efficiency of vertimills over ball mills for fine grinding. The use of work index of the ore is reasonable for this estimate considering the small upgrading in the rougher concentrate. Specific grinding energy measurements should be conducted on rougher concentrate to confirm the regrind power requirements.

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 ball mill crane will be used to service the three vertimills. One bunker will provide storage for 1″ balls of a months capacity. A kibble will be used in conjunction with a crane to ensure each of the vertimills maintains the desired ball charge.

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 five 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.

There is space provision at the north end of the concentrator building for two 200 m^3, one 50 m^3 and one 20 m^3 spare flotation mechanisms. A dedicated 10 t crane will service the flotation area.

It was determined that it would be economical to have an in-line spare for each of the five concentrate pumps. The plant downtime due to unscheduled maintenance of these pumps would be more costly than the extra capital to purchase and install the spares.

As common with every flotation process, chemical reagents will be used to aid in achieving the optimal conditions for the recovery of the desired minerals. Testwork was performed by G&T Metallurgical Services..

The primary collector used will be PAX and it will be added at a rate of 9 g/t ore to the rougher cells and 5 g/t ore to the cleaner conditioning tank. In addition, there is provision to add a secondary collector (3418A – a diisobutyldithiophosphinate) which is more selective in recovering gold. However, this reagent is extremely expensive and is required in higher dosages and should only be used when economically feasible.

MIBC will be used as the frother in the flotation process and will be added a rate of 16 g/t ore of the rougher cells and 22 g/t to the cleaner cells.

A dispersant (guargum carboxymethyl cellulose) will occasionally be required to disperse talc-like materials and minimize their adverse impact on flotation responses. It will be screw fed to the rougher concentrate feedbox. This location was chosen as it provides a high dilution factor which is paramount when attempting to dissolve a guargum based product. Further testwork is required to estimate the addition rates of dispersant required as the current testwork is inconclusive. In addition, the best addition method (dry or wet) should be confirmed. Studies have shown that dry addition of guargum carboxymethyl cellulose is as effective if not more effective at dispersing talc-like material as wet addition (1-3%).[1]

Refer to Section 20.1.4.7 for details on reagent handling.

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

[1] CMC Addition Methods in Ultramafic Ore Flotation, Steve Wilson, CMP 2000.

Figure 20-3: Overall Concentrator Layout

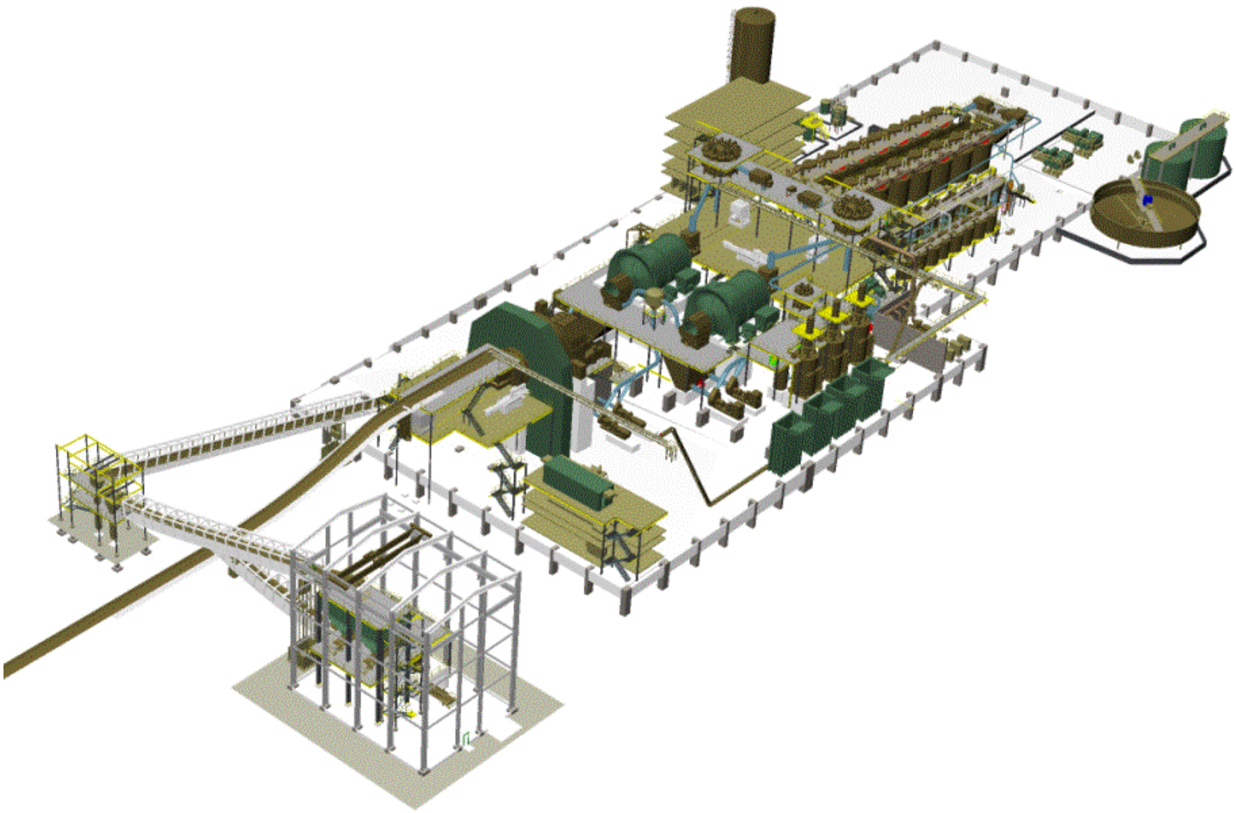


Figure 20-4: SAG Mill Circuit

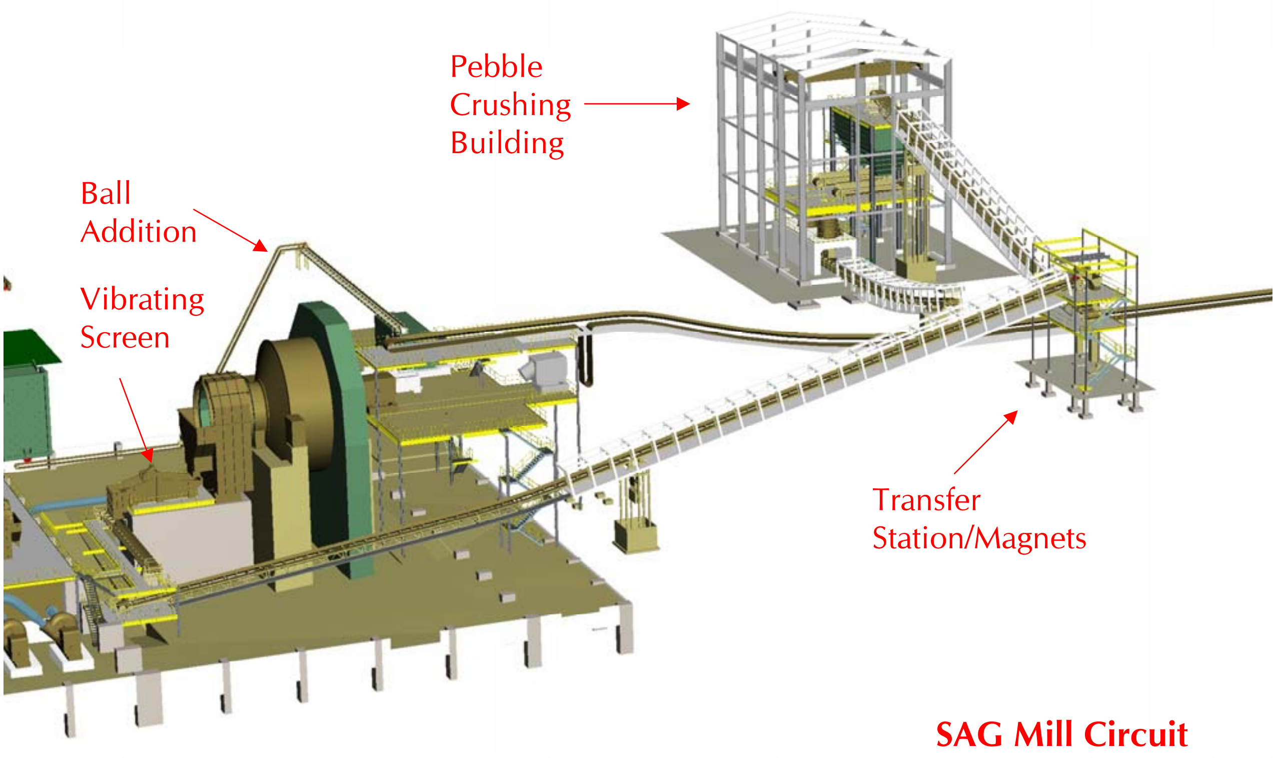


Figure 20-5: Ball and SAG Mill Layout

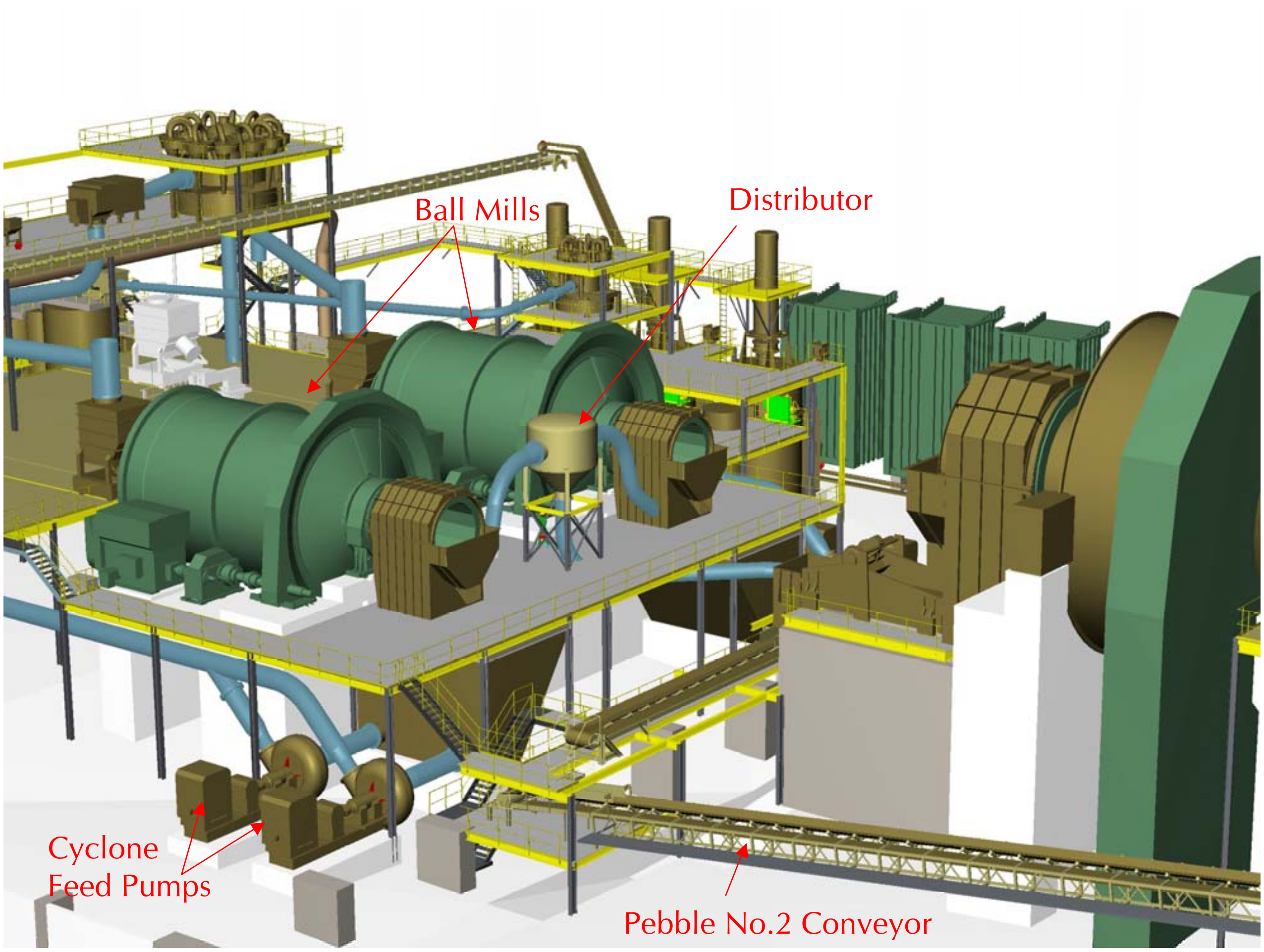
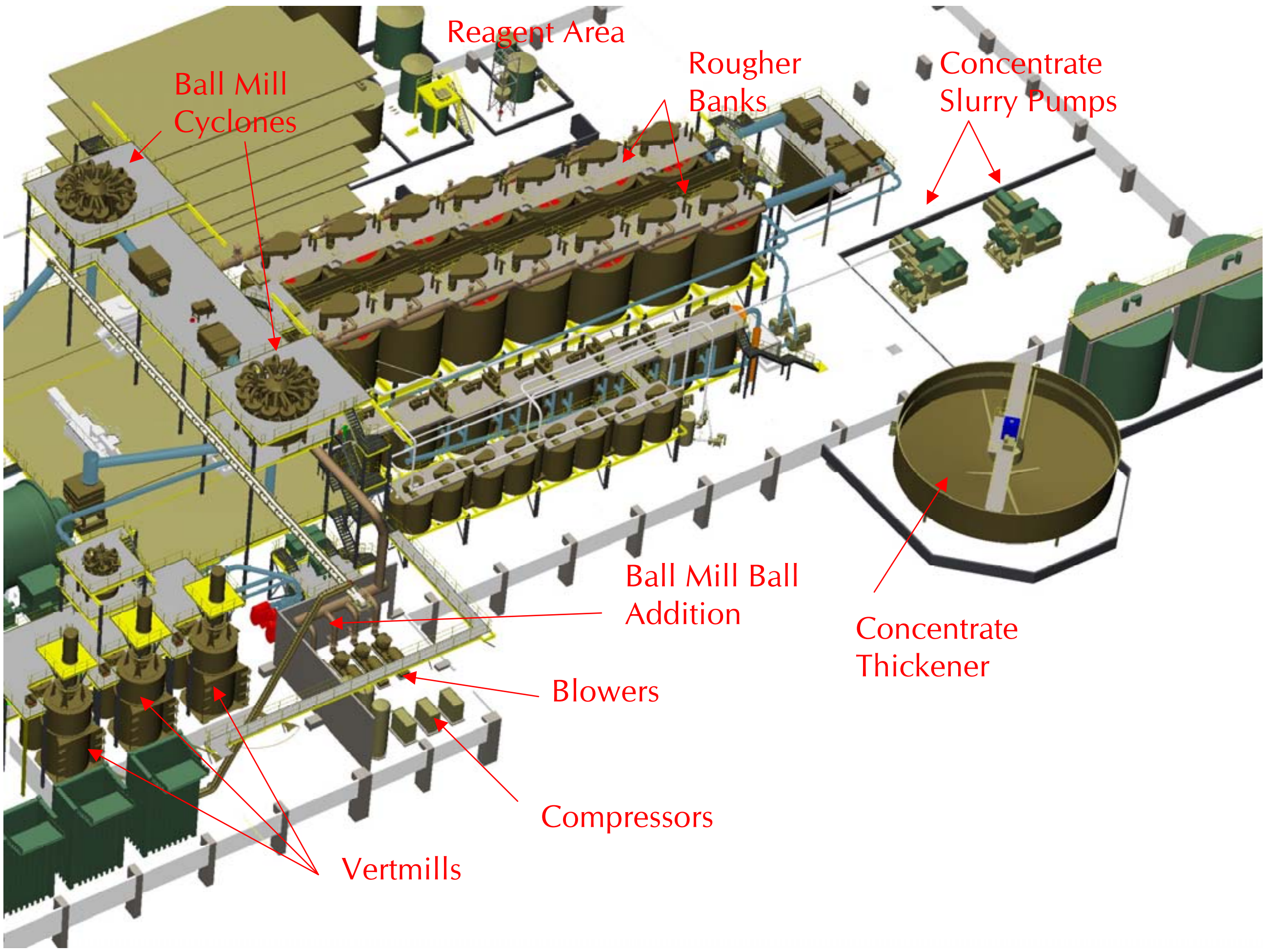


Figure 20-6: Flotation and Regrind Layout



20.1.4.4 Concentrate Thickening

Figure 20-6 shows a picture of the concentrator looking north-west.

Third cleaner concentrate will be pumped to a 22 m diameter high-rate thickener based on Outokumpu testwork. [2] The testwork also found that the addition of 17 g/t of dry concentrate was the optimum flocculant dosage. The thickener underflow, at a concentration of 60% solids, will be pumped to two 9.0 m diameter x 9.0 m high storage tanks which will provide a surge capacity (9 to 17 hours variable) to allow for possible differences in availability between the concentrator and the pipeline. Two mainline positive displacement pumps (1044 kW each), each with a centrifugal charge pump) are connected to the storage tank, one functioning as an operating unit and the other as a standby. These pumps are equipped with variable frequency drives. A piston diaphragm positive displacement pump was selected, as they are well suited for moderate flow and high pressure (in this case up to 300 Bar)

A study for evaluation of two pumps stations versus one pump station (as was in the PEA) was carried out and the study found that one pump station located at concentrator with a higher pump discharge pressure could be used if the line was oversized. Concentrate slurry will be pumped 135 km to the Bob Quinn filter plant.

20.1.4.5 Tailings Thickening

The bulk rougher tailings and the cleaner scavenger tailings will flow by gravity to a thickener which thickens the tailings from approximately 31% solids to 60% solids. The tailings thickener reduces the pumping volumes of the tailings and reclaim water, results in a smaller pump installation and reduction in capital cost.

The tailings thickener size was determined based on the results of the testwork conducted by Outokumpu. To achieve design underflow density of 60% solids at nominal dry tonnage of 2,866 t/hr for the final tailings thickener, a 64 m diameter Supaflo High Rate thickener was recommended for the solids loading rate of 0.88 tonnes/m^2/hr.

The optimum feed density was found to be 14-15% and will be achieved by auto-dilution within the thickener. Recommended Cytec's A-130 flocculant provide best flocculation and settlings at lowest addition rates. The optimum flocculant addition rate was determined to be 7 g/tonne to obtain clear overflow with TSS of 170 mg/l.

The site layout has been configured so that the tailings can gravity feed to the tailings thickener via concrete channel. The tailings streams from each rougher bank and cleaner/scavenger will be individually sampled and will flow to the final tailings collection box located at the northern end of the flotation building. The concrete channel 1 m wide and 1 m deep will maintain consistent slope of 2% to ensure stream velocity at approximately 3 m/s. A drop box outside concentrator building will provide a transition point between the concrete channel and HDPE in which the tailings slurry will be transported to the thickener.

[2] Outokumpu testwork TH-0364, Novemeber 23rd-25th, 2005.

The thickener overflow will gravity flow in two interconnected process water tanks 13 m diameter and 14 m high. Three horizontal centrifugal water pumps (2 operating, one stand-by) will supply process water to primary and secondary grinding and to the pressurized process water loop.

The thickener underflow will contain 60% solids by weight with P80 of approximately 200 microns. Four horizontal slurry pumps, two pumps operating in series, will pump thickened tailings via 32" HDPE SDR11 pipeline to tailings disposal. The initial 1,500 m of tailings pipeline will be CS HDPE lined in order to meet pressure requirements. The tailings delivery pipeline will be laid on in a shallow ditch alongside road. Pipeline bedding material will be free draining sand or gravel. Adequate clearance and appropriate anchoring will be provided for the thermal expansion and contraction.

20.1.4.6 Process and Freshwater Distribution

20.1.4.6.1 Process Water

Process water is required for the purposes of ore dilution for milling as well as to flush the concentrate down the launders. The main source of process water will be recycled water from the tailings and concentrator thickener overflows. This will constitute about 71% of the process water requirements. The remaining water will be obtained from mine dewatering and reclaim water from the tailings dam.

Dewatering wells will be installed around the perimeter of the operating pits. The main purpose of these wells is to depressurize the pit walls. However, since this water will be of good quality, it can be used as both process, fresh and fire water if required. This water will first be pumped to the process water storage reservoir which is a common design for a plant of this size. This reservoir will provide a readily available source of clean water and act as a large buffer for the concentrator. It will hold eight hours of process water. The reservoir will be located adjacent to the covered coarse ore stockpile and will gravity feed the process water tanks. The two process water tanks will provide sufficient water to run the process for 30 minutes thus handling surges in plant water demand.

Supernatant water from the tailings pond will be recycled to the plant. During the normal operation 519 m^3/h of water will be required. However, reclaim water pumps are sized to provide full process water requirement of 1,779 m^3/h in case of decreased yield of mine dewatering wells The reclaim water pumps will be vertical turbine type pumps installed on a floating barge. Three pumps will operate in parallel, which allow for head and flow variation during the life of the mine.

At the beginning of operations, the pumps will overcome static head of approximately 260 m, to reach the aqueduct elevation of 709 m. At this point three pumps will operate in parallel. As tailings level raises, two pump operating in parallel may satisfy process water demand.

The water will be pumped to a process water storage reservoir located at the process plant via 20 inches diameter pipeline. The pipeline will be divided in two sections in order to meet pressure requirements. The initial section of carbon steel standard schedule pipe will be provided with floats and ball joints creating flexible floating pipeline system from the reclaim barge to shore. As the pipeline reaches elevation of diversion ditch, HDPE SDR17 pipe rated to 100 psi will be used. Reclaim water line will be buried 1.5 m below the grade to provide freezing protection, no heat tracing will be required.

20.1.4.6.2 Fresh Water

The fresh water will be used as gland seal water and fire water. The initial fill of fresh water will be obtained from two 203 mm diameter groundwater wells in the vicinity of the mill. The fresh water for ore processing will be replenished from the process water storage reservoir. The water in process water reservoir will be obtained from mine dewatering wells or reclaimed from tailings and is believed to be suitable for gland seal services without additional filtration due to its low turbidity.

Two vertical turbine pumps installed at process water reservoir will supply water into the common manifold. The fresh/fire water tank will be equipped with level and temperature probes which will feed information to a controlled immersion heater to ensure the water does not freeze. The 4" diameter buried HDPE water supply pipeline will be installed below freezing level and will be terminated at the fresh/fire water tank located southwest of the mill building.

20.1.4.6.3 Compressed Air

Figure 20-6 shows where the compressed air equipment is located in the concentrator. This equipment, which includes compressors and blowers which provide high and low pressure air respectively, will be located in a separate enclosed room for noise abatement.

High Pressure Air

Compressed air will be required in the concentrator for pulsating the dust collector, for instrument air and for plant air to run pneumatic tools. This air will be provided by two compressors (1 operating, 1 stand-by) each providing 450 Nm^3/h at 8.6 bar.

Lower Pressure Air

Low pressure air (0.63 bar) will be required by all of flotation cells to create the froth needed for the flotation process. This air will be provided by 4 blowers, (3 operating, 1 stand-by), each to provide 12,265 SCFM.

20.1.4.7 Reagent Handling

Refer to Figure 20-6 for the location of the reagent storage area within the concentrator.

The reagent area consists of both storage and mixing tanks for preparing the reagents for use. Storage on site for each of the reagents will be at least seven days which will ensure that there is availability for the process even if there is difficulty reaching the site due to road conditions.

Pulverized quicklime will be delivered by a bulk tanker (42 mt) approximately once per day to and outdoor storage silo. The lime silo will have an integrated slaking system and the resulting slurry will be pumped to a lime holding tank. Lime slurry (MOL) will be pumped through a return ring main feeding the various points of use.

The MIBC will be delivered by a bulk tanker (20 m^3) once per week. MIBC will be stored in an outdoor storage tank. Five metering pumps, including one spare, will distribute the product to the various points of use from a header located inside the concentrator building.

PAX, dispersant will be delivered in bulk bags and stored in a dedicated area adjacent to the rougher flotation cells. The PAX will be delivered in a 800 kg bulk bag which will be broken in a hopper, fed to a mixing tank and then diluted to the corrected consistency (15% w/w) with fresh water.

Flocculant will be delivered in bulk bags and stored in a dedicated area adjacent to the rougher flotation cells. Flocculant will be delivered in 25 kg sacs and will be manually added to the package system which includes an eductor for carefully wetting the flocculant flakes ensuring they maintain their integrity, and a storage tank which further mixes the floc-water mixture to a 0.5% w/w consistency. This product is then gravity fed to a storage tank from where is pumped to the thickeners using peristaltic metering pumps. Further water dilution is added after pumping to bring to delivery consistency to 0.05% w/w flocculant.

The dispersant (guargum carboxymethyl cellulose) will occasionally be required and will be delivered in 1000 kg bulk bags and will be added to two 1 t hoppers above the first cell of both rougher banks. This will be accomplished by using the flotation crane and each hopper will need a new bulk bag every eight hours (based on 100g/t ore). The dispersant will be screw fed to the feedbox of both rougher banks at a controlled rate.

The liquid reagent 3418A, as discussed in Section 20.1.4.3, will not initially be used. If it is deemed economical to use, it will be delivered by bulk truck and stored and subsequently pumped by metering pumps to the flotation circuit.

20.1.4.8 Layout Considerations

20.1.4.8.1 Crushing

The crushing plant is situated adjacent to the Central pit approximately 500 metres from the concentrator. This location provides the shortest distance for hauling ore while also keeping these large trucks away from the concentrator for safety purposes. Both berms and bollards will be located at the truck dump for safety purposes.

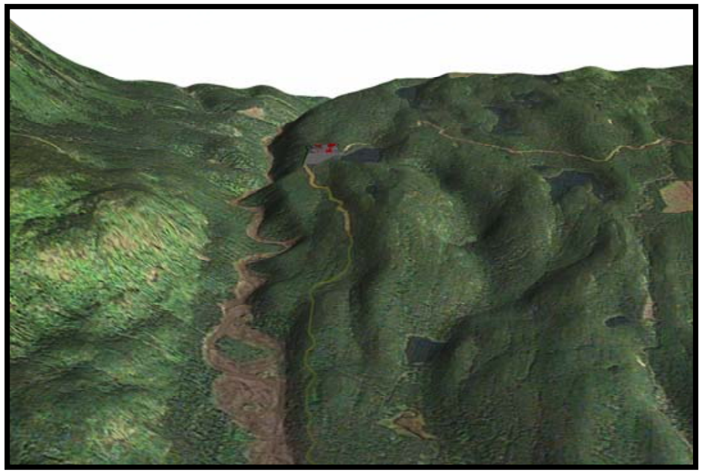
20.1.4.8.2 Concentrator

The concentrator has been located on the most flat area in the vicinity of the central pit. However, with the terrain still having a significant slope, it will be situated entirely on cut. While this will require significant earthworks, there are advantages to the resulting elevation change which have been used in the design. Steel ball delivery, used for comminution, can be made on a terrace above the concentrator allowing for easy storage. With the inherent slope, tailings from both the rougher and first cleaner scavenger cells will be gravity fed to the tailings thickener. In addition, the overflow water from both the tailings and concentrate thickener can be gravity fed to the process water tanks which are located at a lower elevation. The use of gravity flow both reduces the number of pumps required in the process as well as the power requirements which are significant at the high tonnage seen in the plant.

The control room will be located in a central elevated location to allow for a good observation point for the operators to be able to see both the grinding and flotation operations. The metallurgical laboratory will be located within the process building on the main floor thus reducing the amount of sample transport which would be required if it was in the assay building.

There will be two main MCC areas, one located in the grinding area and the other located below the control room adjacent to the rougher flotation cells. By having these two areas, the amount of cabling to the motors will be minimized.

20.1.5 Bob Quinn Filter Plant



Copper concentrate slurry will be pumped overland 135 km to a filter plant located approximately 8.6 km by vehicle from the junction with highway 37. Copper concentrate will be filtered and loaded as a cake into b-train concentrate hauling trucks and transported approximately 235 km to the port terminal of Stewart B.C.

The Bob Quinn Filter Plant will be comprised of two buildings. The filter building will house two concentrate pressure filters while a separate building will be dedicated for the treatment and disposal of filtrate water and ancillaries. The maintenance shop and offices will be located in lean-to structure adjacent to water treatment building.

The slurry pipeline transport system comprises of a single pump station at the Galore Creek concentrator, an overland pipeline, and terminal facilities at the filter plant to receive the slurry prior to filtration and water treatment. The concentrate slurry will be pumped via centrifugal charge pumps into either of two Geho high pressure piston pumps, which will provide sufficient pressure at the design flowrate to transport the slurry to the filter plant without booster pumping. The slurry will pass through a choke station located at the filter plant and will report to the 9 m diameter by 9 m high stock tank, capable of storing 4.5 hours of concentrate slurry at pipeline design transfer rate of 120 m^3/hr.

Concentrate will be pumped through the pipeline in batch mode, with intermediate batches of process water. The low density slurry, immediately before and after a batch of concentrate (termed "tail out"), will be directed to the lime reactor tanks following by clarifier-thickener for further thickening. The thickened slurry will be pumped back to the stock tank.

20.1.5.1 Concentrate Filtration

The filter plant will be comprised of two 120 m^2 pressure filters, each configured to run independently of one another.

Concentrate slurry will arrive at an agitated concentrate storage tank at nominal flowrate of 75 t/hr dry solids and approximately 57% of solids by weight. Solids will have size distribution of 80% passing 39 microns, 50% passing 21 micron and a specific gravity of 4.15. The targeted moisture content in a filter cake will be approximately 8%. The filtration rate is estimated at 388 kg./m^2h.

Concentrate slurry will be fed by a horizontal centrifugal slurry pump to one of two filters. The feed pump will run until a maximum pressure is attained, followed by pressing of the moist cake by inflation of the interplate diaphragms to squeeze out excess moisture. Dry air will be blown through the cake to further remove moisture and the filter cake will be finally discharged at a nominal 8%

moisture into the concentrate loadout bin. The filter cycle time consisting of pumping, press time, blow time and discharge time will be 13.5 minutes.

Filter cake from both filters will be discharged to 1,800 mm wide belt feeder. To prevent overloading of the belt feeder and concentrate loadout conveyor, operation of two filters will be staggered in time, so cake discharge will not take place simultaneously for both filters.

Filtrate water and flush water, with low solids concentrations (about 6% solids in combined flow), will report to a filtrate tank on the ground floor of filter building and will be pumped to lime reactors in the water treatment facility.

20.1.5.2 Concentrate Handling and Storage

Concentrate cake will be transported by 610 mm wide belt conveyor to a concentrate loadout silo for the final loading in haul trucks. The silo will be 10 m diameter and 20 m high steel construction and will provide capacity for approximately eight hours of concentrate production. The 60 degree cone bottom and reclaim auger will ensure consistent discharge at the rate approximately 240 t/hr or 114.3 m^3/hr without posing a bridging problem. The retractable telescoping chute will minimize spillage and dust generation.

The load out area will be enclosed in a steel clad building. A dust collection system will be provided around the loading point. A truck scale will be installed to measure vehicle weight empty and full and to carry totalizing function of concentrate loadout.

The diverter valve installed at the conveyor head chute will allow by passing of the concentrate bin if required. In this case, a horizontal concentrate stockpile feed conveyor will transport material to the covered emergency 13,100 t stockpile. The stockpile will provide capacity for seven days of concentrate production. Material can be reclaimed from the emergency stockpile by loading the concentrate into a haul truck by front end loader.

20.1.5.3 Water Treatment Plant

Filtrate water from the pressure filters and flush water from the pipeline will partially recycle within the plant. The excess water at 56.74 m^3/hr will be treated and discharged. To meet the regulatory requirements prior to discharge to the Iskuit river, effluent will be treated to reduce TSS content and dissolved metals concentration, particularly Cu.

The testwork conducted by Rescan Environmental Services indicates, that the drop in pH caused by the presence of Thiosalts in the filtrate water, will result in elevated dissolved Cu concentration. To meet the BC permit level for dissolved copper at 0.05 mg/l, effluent must be treated with lime to raise pH from 7.5 to 10.8 prior to entering the clarifier/thickener.

Two agitated lime reactor tanks 5 m diameter and 5.5 m high will be installed in the water treatment building to treat the combined flow of flush water and filtrate water with lime. Lime will be added in first tank to maintain pH at 10.8. Mixed slurry will overflow to the second tank via an upcomer. This arrangement will ensure proper mixing and provide the required reaction time of 15-30 min.

The residual solids and precipitated metal in a form of sludge will be removed in an E-CAT type clarifier /thickener. Flocculant will be added to promote settling. The underflow from the clarifier/thickener will be returned to the concentrate storage tank to be fed to the concentrate filters.

Lime will not pose a problem for filtration because of low addition rate. The overflow will be pumped through a series of filters for final solids removal.

The effluent filtration will be comprised of sand filters and cartridge filters to reduce TSS to 0.5 mg/l, and carbon filters to reduce total organic carbon (TOC) concentration to meet the 10 mg/l Provincial objective. Each type of filter will have operating and stand-by units for continuous operation. Sand filters will be periodically backflushed, while activated carbon will be exchanged as required. Spent carbon will be regenerated off site.

Treated effluent will report to a treated water tank. Provincial regulation states that maximum allowable pH of an effluent shall be 8.5. Sulphuric acid will be added to adjust the pH of the final effluent. Water will be sampled before the discharge. Off-spec water will report to 7.5 m diameter and 8.5 m high tank, from where it will be pumped to clarifier/thickener for additional treatment.

The on-spec treated effluent will be discharged into the Iskut River through a 9 km long pipeline and diffuser system.

20.1.5.4 Site Utilities

20.1.5.4.1 Compressed Air

The plant air system will include one pressing air compressor (20 bar) and two drying air compressors (10 bar) for the pressure filters, and one plant air compressor that will produce compressed air for instrumentation and small tools.

20.1.5.4.2 Potable Water

Two wells were identified as a source of potable water. The potable water demand for the Bob Quinn filtration plant and camp was established based on camp capacity of 30 workers and average daily water consumption of 0.19 m^3 (50 US gallons) per person. Potable water treatment units will comprise of 1-micron and 10-micron cartridge filters, UV disinfection unit, a hypochlorite addition systems, raw water tank, small mix tank, metering pumps and booster pumps.

20.1.5.4.3 Process and Gland Water

The process and gland water will be obtained from the on-spec treated water tank.

20.1.5.4.4 Fire Water

Water will be available to the firewater main from the fire water tank at a nominal flow rate of 272 m^3/h (1,000 US gal/min). The storage tank has been designed to have a firewater reserve that will supply two hours of firewater in the event of an emergency. The fire pumps will be housed in 40' long shipping container. The fire water distribution system will comprise of 200 mm diameter fire water main, 150 mm diameter branches to all main facilities including pressure filter plant, water treatment plant, camp and fuel station. All pipes shall be HDPE SDR9 rated for 200 psi buried below frost depth, which is 1.5 m below grade.

20.1.5.4.5 Reagents

The reagent mixing area will be located in the water treatment building. The quick lime and polymer will be delivered in 25 kg bags and manually loaded into mixing systems. Sulphuric acid will be delivered in drums.

20.1.5.4.6 Fuel Unloading

A fuel unloading and transfer station will be provided at Bob Quinn facility. Diesel fuel will be unloaded from the trucks to the fuel storage tank (160,000 litre capacity). From this tank fuel will be transferred via 135 km long pipeline to the mine at the maximum rate of 6.20 m^3/h. A small dispensing station provided with 50,000 litre dispensing tank will also be located in the vicinity of the storage tank. A small fuel transfer pump will be used to transfer fuel from the storage tank to the dispensing tank.

20.1.5.5 Layout Consideration

The plant site has been carefully selected to minimize amount of civil works. Also it provides easy access from the road to concentrate loadout and fuel loadout facility. The plant layout takes advantage of site topography. The process facility will be situated at a higher elevation than the loadout bin thus reducing the length required for the concentrate loadout conveyor. Space between plant roads and buildings is provided to allow for traffic turning and to eliminate any blind corners or intersections.

20.1.6 Process Control

20.1.6.1 Scope

The Mill and Process Plant Control System will control all equipment from the primary gyratory crusher to the tailings dam, including the Bob Quinn Filtration Facilities. It will also control fresh and fire water supply, water treatment, and mine dewatering pumps.

20.1.6.2 Design Criteria

Instrumentation will be used to automate many process operations. The aim of the process control system is to:

- Ensure plant personnel safety
- Ensure equipment availability and operability
- Ensure process efficiency at required throughput
- Operate most of the equipment in automatic mode
- Reduce operator interventions

20.1.6.3 Control System Overview

All concentrator plant process operations will be controlled from a central control room which will be manned 24 hours a day 7 days a week. It will also be possible to monitor operation of the filter plant from this control room. Remote control of the filter plant will only be possible if the necessary control permissives are activated by the filter plant operator. Operation of the filter plant will be controlled from the filter plant control room (also manned 24 hours a day, 7 days per week). Monitoring, but not control, of the concentrator process plant will be possible from the filter plant control room.

A secondary control room at the concentrator site will be dedicated to the control of the gyratory crusher operation. The central control room will have full remote control of the crusher operation.

The pipeline between the concentrator plant and the filter plant will be controlled from either the concentrator plant or the filter plant control rooms. Each operator will have access to a fully operational control station.

The control system will be fully automated within the context of a sound, modern and cost effective approach. Sensors and actuators will be connected to the Programmable Logic Controllers (PLC) to maximize automated process operations. All tank levels will be automatically controlled. Whenever possible, a VFD on the exit pump will be used to maintain the level. Throughput and flowrates will be controlled at an operator chosen set-point. “On/Off” valves on pump suction and discharge lines will be automatically switched from the control room.

All vendor PLC’s will be connected to the Mill PLC to allow for remote monitoring of the equipment. When necessary, the connection will also allow for remote start and stop of the vendor’s equipment. The communication link between the two PLC’s will also allow for the vendor PLC to be interlocked to equipment wired to and controlled by the Mill PLC.

The control system is comprised of the following:

- PLC - Programmable Logic Controllers
- EWS - Engineering Workstation
- PRN - Network Printer
- UPS - Uninterruptible Power Supplies
- I/O Network (communication between the PLC and its I/O, dependent on the processor selected)
- PCS - Process Control System network LAN’s (Communication between PLC’s, HMI and EWS)
- DHS - Data Historian Server

20.1.6.4 Operation

Each control room will be equipped with HMI workstations and will require an operator to be present 24 hours a day. The central control room will be located in the Concentrator Plant Building overlooking the flotation cells and the ball mills.

The PLC’s control the process and perform motor and equipment logic, interlocking and monitoring, analog signal conditioning and instrumentation PID loop control.

The HMI software, runs on office grade desktop PC’s, allows the operator to view equipment status, process parameters, device values, alarms and trends. The operator controls the plant by having the ability to start/stop equipment, adjust process set-points, and acknowledge alarms.

The PLC's, a minimum of one HMI per control room and instrumentation associated with emergency equipment will be connected to the emergency generator. During power failure, plant control will be maintained. UPS units will be used as necessary to provide smooth transition between mainline power and emergency power.

An engineering workstation will be located in the concentration plant and filter plant control rooms. The engineering workstation will contain software for PLC programming and HMI configuration. A laptop will also be available to program the PLC and HMI.

A network printer can be used to print alarms, trends and snap shots of operator screens.

The data historian logs and archives plant data, which is used for trending, reports and trouble shooting. The data historian can also provide plant data to the office LAN via an isolated gateway.

The PCS LAN's (local area networks) will use the TCP/IP protocol to communicate. The LAN's are comprised of network switches, media converters, Cat.5e copper cables, fibre-optic cable and fibre patch panels. The control LAN is completely independent and cannot be used for outside communication.

20.2 Tailings and Waste Rock Storage and Water Management Plan

Section 20.2 was prepared by Mr. Iain Bruce,P.Eng, P.Geo, BGCEngineering Inc.

20.2.1 Summary

BGC Engineering Inc. (BGC) has been retained to evaluate the options for storing tailings and waste rock in Galore Creek Valley north of the proposed mine, as well as assess open pit wall stability and plant site foundations BGC's feasibility estimates are based on 3 seasons of detailed field study and have been reviewed and endorsed by an independent expert panel (through a peer review process) to ensure that all designs are to the highest industry standard.

Figure 20-7 on the next page shows the proposed tailings and waste rock storage areas as well as the water diversion structures.

20.2.2 Site Characterization

20.2.2.1 Weather

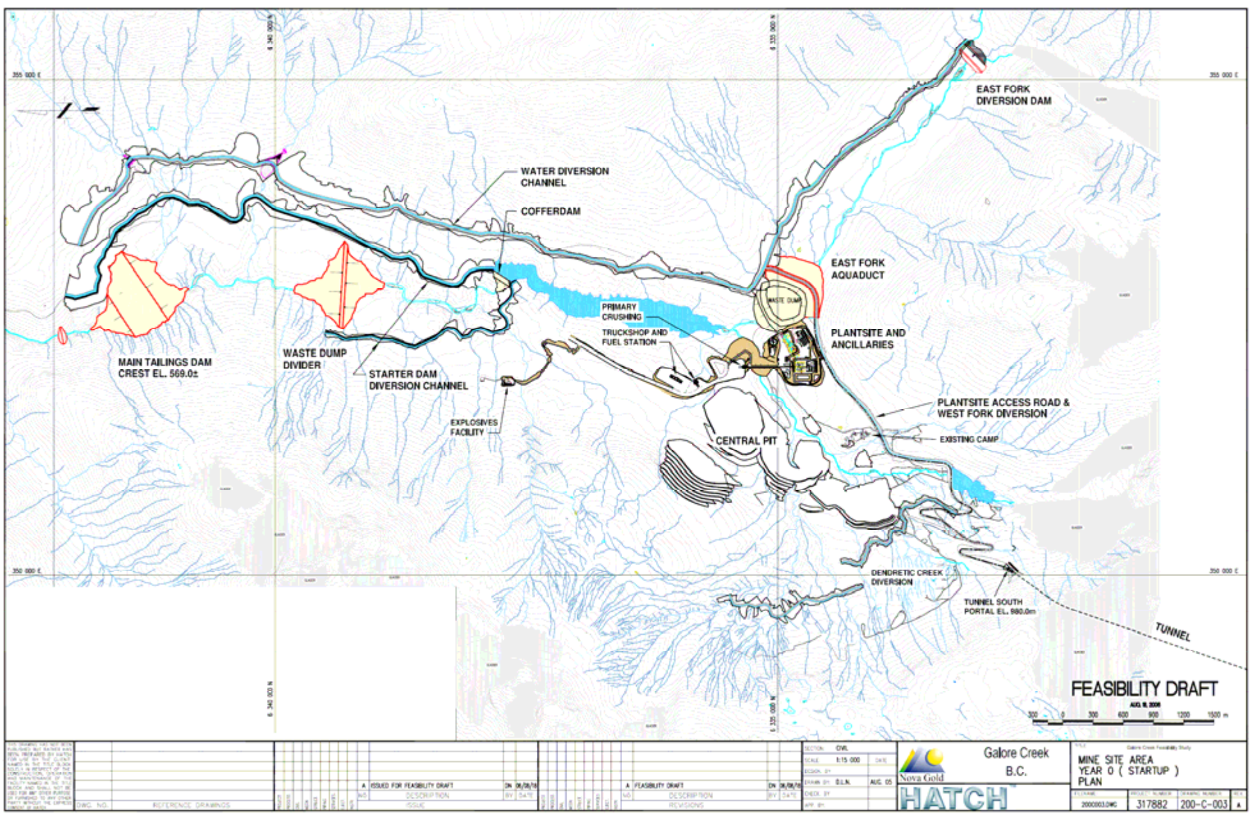
The average annual precipitation for Galore Creek is approximately 2.2 m (Rescan 2005). Approximately 35% of this falls as rain between May and October and 65% as snow between November and April. Average monthly temperatures at the Galore Creek Mine site will likely range from -13°C in January to 11°C in July. The lowest flow months for Galore Creek will be November through April with higher flows from May through October. The driest warm weather period for construction may be approximately four months from May through August.

Discontinuous permafrost is expected in the upper reaches of Galore Creek Valley, but ground temperatures collected from a thermistor in the East Fork of the Galore Creek project confirm that permafrost will not have a major impact on the main structures located at lower elevations.

20.2.2.2 Site Investigations

Site investigation for the tailings and waste were undertaken by BGC within the Galore Creek Valley during the summer field seasons of 2004, 2005 and 2006. These investigations included geotechnical drilling, surficial mapping, test pit excavation, seismic and shear wave surveying, piezometer installation and permeability testing, and laboratory testing. A total of 56 geotechnical boreholes have been drilled in mid to upper Galore Valley: 45 holes in the vicinity of the tailings and waste impoundment, diversions, plant site, and potential borrow areas, and eleven holes within the proposed open pits. Thirteen test pits were excavated in the vicinity of the proposed plant site and crusher areas. Frontier Geophysics conducted geophysical surveys during both field seasons. Approximately 12.8 km of seismic refraction survey were undertaken throughout the property. Laboratory testing of the physical properties of the natural soils, rocks and tailings streams has been undertaken. Borehole and test pit logs, laboratory test results, and seismic refraction survey results have been completed in the Feasibility Study.

Figure 20-7: Tailings and Waste Rock Storage Areas and Water Diversion Structures



20.2.2.3 Ground Conditions

The surficial, bedrock and groundwater conditions within the Galore Creek valley were characterized based on the results of these investigations. In general, the surficial geology within the study area is extremely complex due to multiple advances and retreats of the Galore West Fork and East Fork glaciers during the Pleistocene and Holocene Epochs. Overburden soils typically vary between 0 to

80 m deep and include glacial till (basal and ablation), glaciolacustrine, fluvial, glaciofluvial, colluvium, and organic deposits in various mixtures.

The glacial tills are typically well-graded, unstratified, dense sandy gravels with trace to some silt and trace to some clay. The glaciolacustrine deposits are typically stratified clayey silts with fine sand layers. These soils are typically soft to hard depending on the degree of desiccation. Fluvial and glaciofluvial deposits consist of loose to dense sandy gravels with trace silt/clay, trace to some cobbles and trace boulders. Colluvial deposits are typically sands and gravels with some silt, loose, with sub-angular and angular clasts, poorly graded, and unstratified. Colluvial fans at the outlet of debris flow gullies contain well-graded, stratified deposits ranging from fine silts to boulders. Organic deposits vary from a peat to traces of rootlets intruding the underlying soils. Organic depths in the valley range from 0.1 to 2.7 m below ground surface.

The bedrock geology is also complex due to faulting and folding within the area. In general, there are five primary rock units at the site: sedimentary, volcanics, intrusives, breccias and dykes. However, most of the rocks in the waste containment area are volcanics with some intrusives. Limestone exists at the northern limit of the waste containment area, but all structures have been sited to ensure the karstic limestone will not impact the tailings containment facility.

Faults have been identified within the waste containment area. A single steeply dipping normal fault runs almost perpendicular to the Main Tailings Dam. The fault is narrow, appears relatively impervious and not considered a significant detriment to the dam integrity. Similarly, a healed mylonite zone is located high on the right abutment.

Throughout the waste containment area, the upper 100 m of bedrock is highly fractured due to a well-developed set of closely spaced sub-horizontal joints or ′sheet-fractures. Below the broken rock, the less fractured bedrock is referred to as ‘stick rock’ and is recovered typically in long segments (i.e. joint spacing greater than one metre).

The groundwater surface mimics the existing ground topography. The general groundwater flow direction is from higher elevations in the east, west, and south ends of the central valley towards the lower elevations occurring at the northern end of the valley. The steep topographic relief results in steep hydraulic gradients and artesian pressures in the valley bottom and recharge of the ground water is primarily from precipitation, snow melt and surface runoff.

The Galore Creek project is situated in a seismically active area. A Maximum Design Earthquake characterized by a peak horizontal bedrock acceleration of 0.25g, from a Magnitude 7.0 event on a strike slip fault (at an epicentral distance of ~20 km from the project area) has been adopted for feasibility level design of the tailings dam. The waste dump slopes have been designed to resist 1:475 year return period with a peak ground acceleration of 0.097g.

20.2.2.4 Natural Hazards

The proposed waste containment area lies in steep terrain in an area prone to landslides and avalanches. An assessment of the areas prone to natural hazards was completed from air photograph interpretation and is reported in detail under separate cover (BGC, 2006a).

20.2.3 Tailings and Waste Storage System

20.2.3.1 General

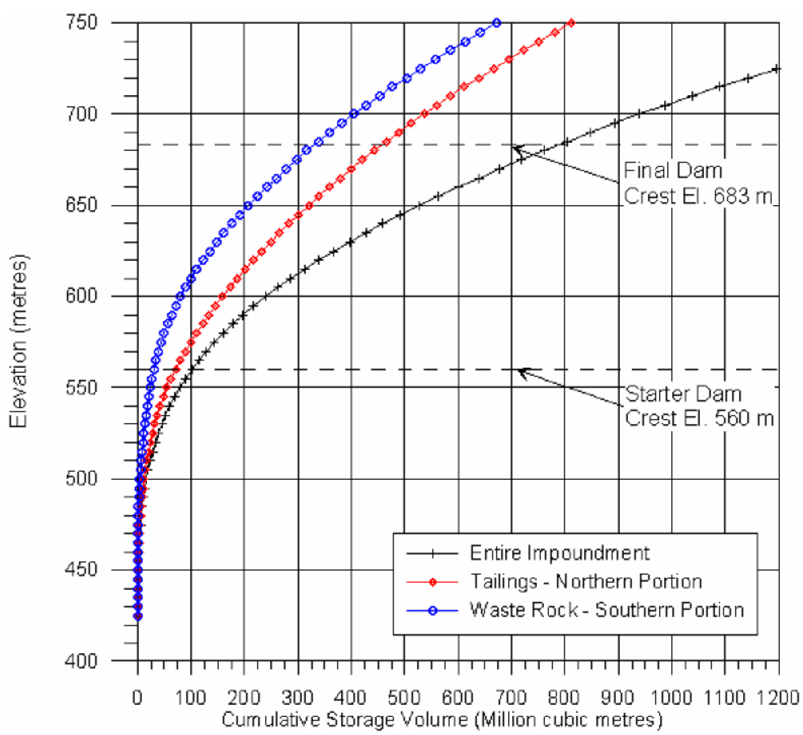
For feasibility level design, a resource of 540 million tonnes was assessed for a 22 year mine life, based on a mill production of 65,000 tonnes per day. During this period, ore will be extracted from approximately four open pits. The ore will be processed, generating approximately 540 million tonnes of tailings and 840 million tonnes of waste rock.

The proposed mill process will involve grinding the ore so that 80% will pass –200 microns. The ore will be processed by flotation without the use of cyanidation. The tailings are expected to settle to dry densities varying from 1.1-1.2 t/m^3 at shallow depths to 1.6 t/m^3 at depths in excess of 75 m. An average settled dry density of 1.35 t/m^3 was selected for feasibility design to represent a conservative density for the entire mine life.

The solid mine wastes, including tailings, waste rock, and pre-stripped overburden from the pits will be entirely contained within the Galore Creek Valley north of the proposed open pits. All the tailings will be deposited in the northern portion of the impoundment and retained by a cross-valley tailings dam. The Main Tailings Dam, located approximately 7 km north of the Central Pit, will be constructed in stages to an ultimate crest elevation of approximately 683 m above sea level with an ultimate dam height of approximately 275 m. Waste rock will be placed in the same impoundment area, immediately south of the tailings solids, but not co-mingled.

Volume elevation curves for the tailings containment and waste dump areas are provided in Fig 20-8.

Figure 20-8: **Galore Volume Elevation Curves (above sea level)**



Tailings will be transported hydraulically to the designated tailings deposition area where it will be spigotted off the crest of the dam and/or nearby valley slopes. During operations, an operating pond (and settling basin) will be created to allow water to be reclaimed to the plant. This pond will facilitate settling of suspended solids. At the end of the mine life, the tailings impoundment will be approximately 2.6 km long, 1.5 km wide and 200 m above the existing valley bottom. The tailings will be flooded during operations and for perpetuity at closure.

Waste rock will be hauled by mine fleet to the middle of Galore Valley for deposition. A total of 840 million tonnes of waste rock will be produced over the mine life. The majority of the waste rock will be placed in lifts in the southern portion of the designated waste area adjacent to the pit. Approximately 41% of the waste rock has been conservatively characterized as potentially acid generating (PAG) and this will be placed in lifts at lower elevations in the valley. Throughout the mine life, the elevation of the PAG must remain higher than the tailings solids elevation to allow controlled placement and must be kept above pond level for a short period to facilitate waste rock placement. At the end of the life mine, the impoundment has been designed such that PAG waste rock will be flooded in perpetuity in accordance with best industry practice.

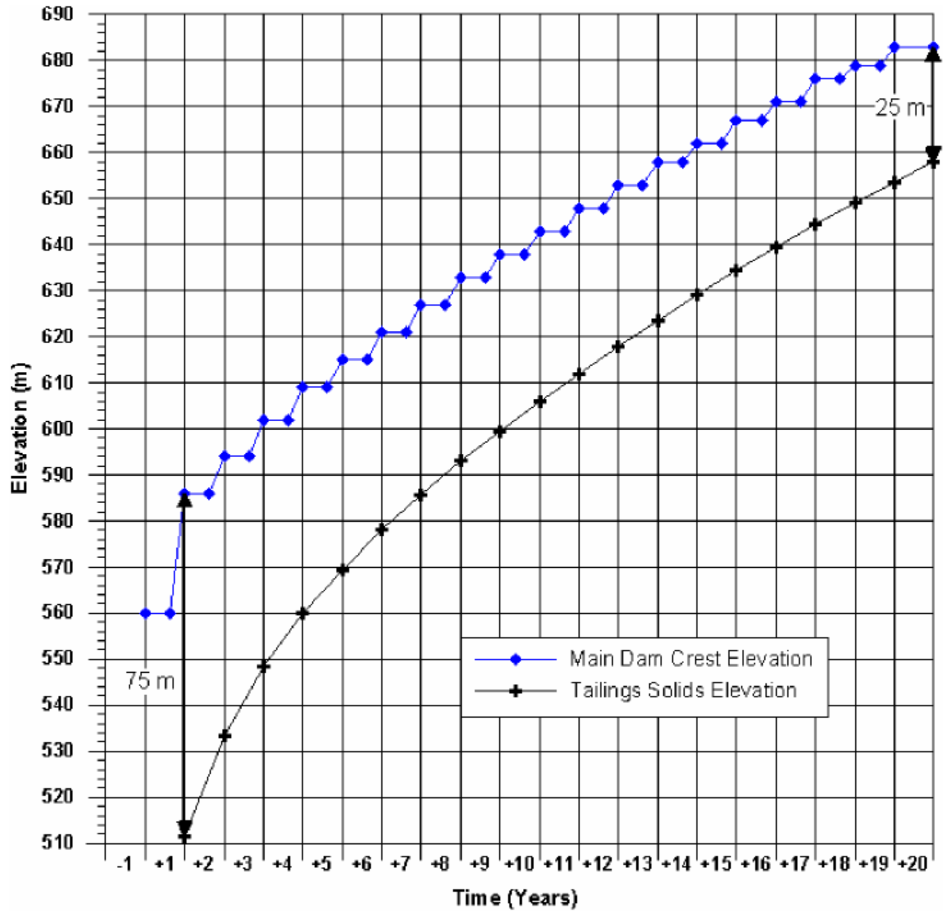
Approximately 59% of the waste rock will be of non-potentially acid generating material (NPAG). The majority of the NPAG waste rock will be placed above the PAG waste. The NPAG waste dumps on the western and eastern slopes will likely vary in elevation (above sea level) from 659 m to 900 m at the end of the mine life. NPAG waste from the upper pits (i.e. Junction Pit) may also be deposited in small dumps around the pits. At the end of the mine life, the waste dump will be approximately 4.2 km long, 1.4 km wide and range from 100 to 200 m above the existing valley bottom.

Waste (both rock and overburden pre-stripping) will be used for construction of the Main Tailings Dam and other auxiliary facilities such as roads, and deep fills. NPAG will be used in all areas which will not be flooded. Areas of fill which will be flooded may be built using PAG, if NPAG is not available.

To prevent tailings from migrating into the designated waste dump area, a waste dump divider will be constructed before the initiation of tailings deposition. The waste dump divider is located approximately 2.6 km upstream of the Main Tailings Dam. The divider will be constructed of either PAG or NPAG waste rock up to elevation 674 m above sea level and with NPAG waste rock above this elevation. Throughout the mine life, the divider will be raised such that the divider crest is always higher than the tailings solids elevation to provide separation of the two mining waste products.

The tailings starter dam must be constructed to a crest elevation (above sea level) of 560 m (or dam height of 152 m) prior to tailings deposition, assumed to be early November 2010. A dam raise of 26 m is required in the first 12 months of production (or Year +1) so that the dam can store to an elevation of 586 m above sea level by the end of Year $^{+}$1 (Nov 2011). Every year the tailings dam is scheduled to be raised in a downstream direction as the tailings are deposited in the basin. At the end of the mine life, the dam reaches its ultimate height of 275 m, with a crest elevation of 683 m above sea level. Figure 20-9 shows the Main Tailings Dam crest elevation and tailings solids elevation over the life of the mine.

Figure 20-9: **Main Tailings Dam Crest and Tailings Solids Elevation (above sea level)**



20.2.4 Tailings Dam Design

20.2.4.1 General

The Main Tailings Dam is a rockfill structure with an impervious (i.e. clay till) central core. The starter dam will be constructed to an elevation 560 m above sea level. The ultimate dam will be constructed to elevation 683 m above sea level and will have a 30 m wide crest. The rockfill shells will be constructed of compacted waste rock. The downstream shell must be constructed entirely of NPAG waste rock. PAG waste rock can be used in the upstream rockfill shell as it will flooded in the short term. Side slopes of 1.7 horizontal to 1 vertical (1.7H:1V) have been selected for the downstream face and 2.5H:1V have been selected for the upstream face of the dam.

The tailings dam is located approximately mid-valley. The contact between the limestone to the north and intrusive breccia to the south is located under a portion of the ultimate downstream dam shell. The dam footprint is currently set so the impervious core is south of the contact, but the dam location can be optimized once the contact is better established.

The internal core of clay till is an impermeable barrier to control seepage through the dam. The core will be built of compacted local glacial deposits such as a clay rich till and glaciolacustrine clays and silts. The proposed slopes of the core are 1H:4V on both sides. The transition from the fine grained impervious core to the downstream rockfill shell will be through two filter zones of sand and sand and gravel. The two filter zones are proposed to be 4 m thick for ease of construction and to allow for potential deformation and offset during seismic events.

Foundation preparation prior to fill placement will consist of a shear key and grouting under the footprint of the impervious core to minimize the potential for seepage, piping and core cracking. A shear key is required on the left abutment under the upstream shell of the starter dam to meet the recommended CDA guidelines for the starter dam upstream slope at 2.5H:1V. The key is approximately 130 m wide by 104 m long by 14 m thick.

Immediately downstream of the ultimate toe, a seepage collection system will be constructed to intercept seepage out of the pond. This system comprises a surface water diversion berm, a seepage recovery ditch and interceptor wells.

To protect the integrity of the tailings dam, a series of emergency spillways will be constructed on the right abutment during the mine life. All spillways have been designed to pass the routed flow from a Probable Maximum Flood (PMF). All spillways will have a 20 m wide channel invert. The spillway control section will either be excavated into rock or riprap and slush grout will be used to armour the control section of the spillway.

New spillway channels will be excavated approximately every two to three years as the tailings dam is raised. In order to use a spillway for more than one year, the spillway invert must be raised incrementally until the next (higher elevation) channel is constructed.

A snow avalanche hazard exists on both sides of the valley, with the potential to run out into the footprint of the proposed tailings and waste impoundment, and intersect and possibly block the Main Diversion Channel. A snow avalanche management program will be instituted during construction and operations as outlined in (BGC, 2006a).

The potential of snow avalanches to impact the tailings pond and any waves generated by avalanche would be contained within the 12 m of freeboard above the spillway invert.

20.2.4.2 Tailings Dam Stability Analyses

Limit equilibrium static stability analyses have been conducted for the Main Tailings Dam with these recommended side slopes and were found to meet recommended Canadian guidelines for dam safety. Static stability of the embankment slopes was analysed for three critical loading conditions:

- End of construction
- Steady state seepage, and
- Partial rapid drawdown.

For each loading condition, the minimum factor of safety was determined for the downstream and/or upstream slope of the Main Tailings Dam.

Pseudostatic stability analyses were conducted for the ultimate tailings dam. The estimated permanent displacements and crest settlements using these various methods range from 0.1 to 0.3 m. These estimated dam deformations are significantly less than the 3 m of emergency freeboard. Based on these results, the dam is considered safe against the loss of freeboard during a seismic event.

20.2.4.3 Tailings Dam Seepage Analyses

Two dimensional seepage analyses have been conducted for the Main Tailings Dam using the same cross-sections used in the stability analyses. Foundations under the dam were modeled assuming the upper 100 m of bedrock was highly fractured (‘broken’). This broken bedrock unit was further divided into two hydrogeologic units: the upper 50 m was assigned conductivity of 10^{-6} m/s and the lower 50 m was assigned a conductivity of 10^{-7}m/s.

Seepage out of the impoundment through the valley walls was not modelled because hydrodynamic containment of the reservoir is expected to be preserved by high groundwater levels in the surrounding slopes and ridges.

The base case (best) estimate of seepage out of the tailings starter dam is 18 L/s, assuming the designated tailings impoundment area is filled only with water. Once the tailings dam reaches its ultimate height, seepage out of the facility may increase to 74 L/s. Sensitivity analyses were conducted and the upper estimates of seepage out of the starter and ultimate dams are 180 L/s and 740 L/s respectively.

To reduce seepage out of the tailings dam impoundment, a grout curtain will be constructed under the impervious core of the dam. A 10 m wide, 50 m deep grout curtain was modeled assuming the curtain extends along the dam alignment. The results of these analyses showed that this cut-off reduced seepage from 74 to 63 L/s, or by 15%.

Sensitivity analysis with the grout curtain indicate that if the overburden and upper broken bedrock units are an order of magnitude more permeable than our base case estimate, seepage can be reduced by as much as 72% with a 50 m grout curtain. If both the upper and lower broken bedrock and overburden are an order of magnitude more permeable than our base case estimate, seepage can be reduced by as much as 67% with a 100 m deep grout curtain.

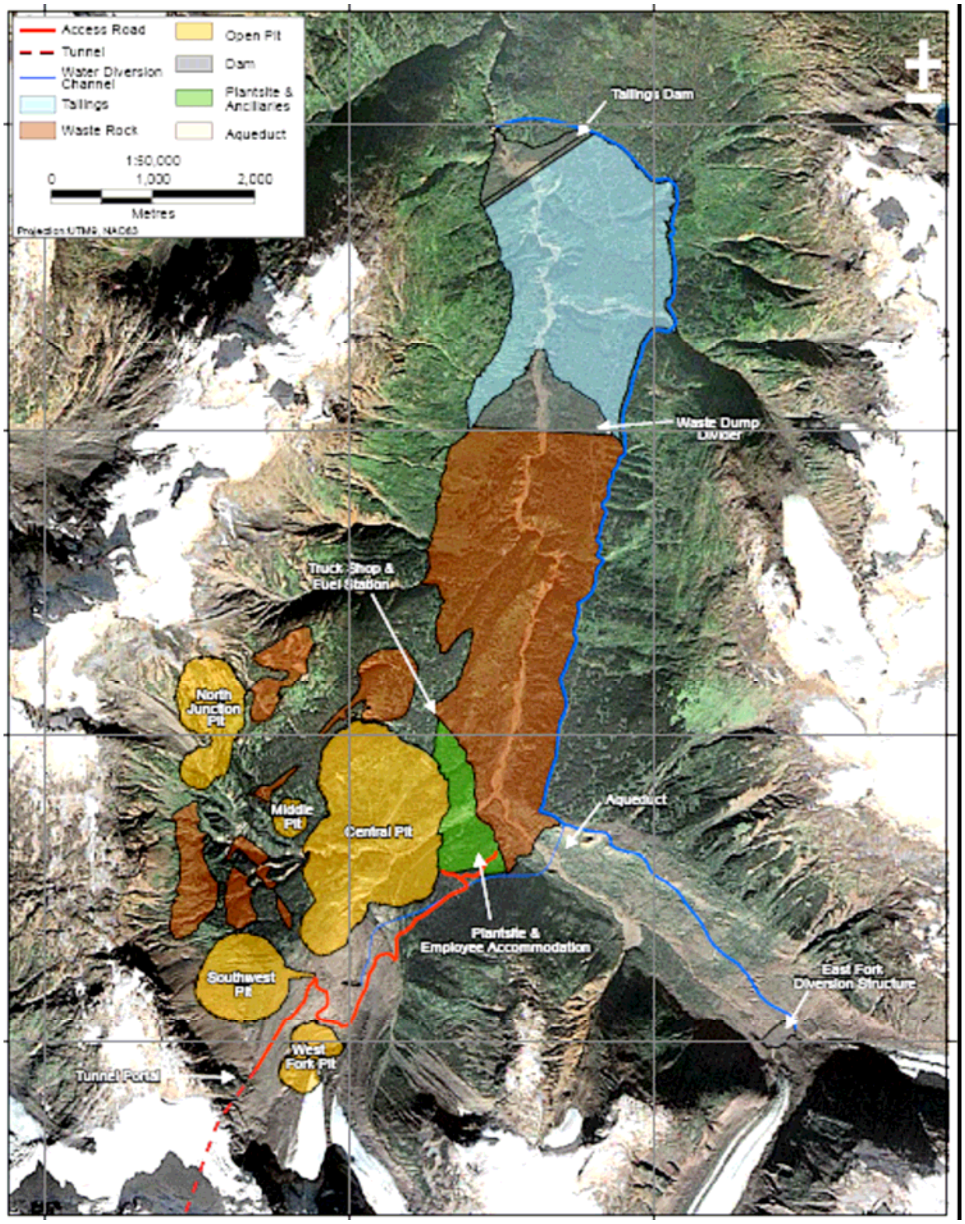
Any seepage out of the tailings impoundment (from the dam and foundations) will be captured by the seepage collection system located immediately downstream of the Main Tailings Dam. All collected water will be pumped back to the pond, or released to the environment if water quality is met.

20.2.5 Freshwater Diversions

20.2.5.1 General

The total potential undiverted catchment area of the proposed tailings/waste dump facility and open pits is estimated to be 125 km^2. Freshwater diversions will be constructed to divert as much runoff as possible around the tailings and waste facility, main plant site, and pits. These diversions mainly consist of open channels with various structures (i.e. diversion dam, aqueduct) required to divert flows into the channels. During operations, diversions will reduce the total catchment area to 38 km^2.

Figure 20-10: Catchment Areas



The open channels will have the following typical design features:

- Channels are designed for peak flows from a 200-year return period 24 hour event with some allowance for snow melt.
- Unlined channel longitudinal slopes will be about 0.3% to minimize erosion potential. Steeper slopes will require erosion resistant linings depending on the erosion resistance of the local materials.
- In areas of impervious soils and sound rock the channel will not be lined. In areas of pervious soil and broken rock the channel will be lined with a geomembrane then covered with riprap to protect the liner.
- Cut slopes are assumed to be 27° (2H:1V) in overburden and 65° (0.5H:1V) in rock. Channel depths will typically range from 3 m to 5 m.
- Channel bends shall typically have a minimum radius of curvature of five times the channel width.
- For purposes of channel maintenance (i.e. snow clearing), a 10 m wide running surface or access road was assumed alongside each channel. This surface will be constructed of fill from the channel excavation.
- All trees, organic material and any soft or unsuitable soils will be cleared from the footprint of the 10 m wide road.
- Bed protection/stabilization and energy dissipation structures will be required in areas where diversion channels discharge into the downstream creek.
- A perforated pipe sub-drain system will lower groundwater uphill of the channel to prevent liner uplift.

Blockage of the channels could occur due to soil and rock falls from the upper slopes as well as snow avalanches leading to periodic overflows to the tailings/waste basin. A total of five designated emergency overflow areas are planned along the main diversion channel so that flow will pass into the waste catchment area upstream of the blockage. Regular channel inspections and maintenance to keep the channels operational year round have been scheduled and included in the normal operations of the minesite.

Some of the major channel crossings (e.g. Bear and Friendly creeks), exhibit signs of past debris flow activity. Potential debris flows will be mitigated upstream of the channel to minimize blockages that would render the channel inoperable. Maintenance of these structures will be undertaken to remove accumulated debris.

20.2.5.2 Construction Diversions

For the construction period, two starter dam diversion channels are required. These are required only for construction and have been allowed for in the capital estimate:

- Starter Dam Diversion – West Channel, which is approximately 2.4 km long and will divert water along a portion of the western slope between the cofferdam (for the Main Tailings Dam) and the toe sump; and,
- Starter Dam Diversion – East Channel, which is approximately 5.3 km long and will transport water north from behind the cofferdam around the starter dam construction area.

A 71 m high cofferdam will be constructed to divert Galore Creek into the eastern Starter Dam Diversion Channel. The Primary Cofferdam will have a 20 m wide crest and will be constructed in a narrow part of the valley with bedrock foundations. The dam will be constructed of waste rock with a geomembrane on the upstream face, which will be mechanically anchored to the bedrock. PAG waste rock can be used for construction material as the cofferdam crest is below the ultimate pond elevation of 674 m above sea level.

To facilitate construction of the cofferdam out of the valley bottom, a 10 to 15 m high secondary earthfill dam (or Temporary Cofferdam) will be constructed 1.5 km upstream of the primary cofferdam. Water collected behind this small structure will be pumped up to the eastern starter dam diversion channel. The primary cofferdam will be constructed during a low flow winter period so that the winter flow of 1 to 2 m^3/sec can be pumped.

20.2.5.3 West Fork Diversion Channel

The West Fork Diversion Channel, is a 2.0 km long channel constructed immediately south of the central pit, across Camp Creek, behind the plant site to divert runoff from the West Fork area.

The 10 m high West Fork Diversion Dam will be constructed of waste rock structure with an impervious clay upstream face. This dam will divert water into the West Fork Diversion Channel. The West Fork Diversion Dam will be constructed during the first twelve months of construction, after the West Fork diversion channel is complete. The channel will be 5 m wide and 5 m deep with an elevation (above sea level) varying from 700 to 750 m and will likely be lined.

20.2.5.4 Main Diversion Channel

The 12.3 km long Main Diversion Channel will be constructed along the eastern slopes above the tailings/waste impoundment extending from the East Fork of Galore Creek to the Main Tailings Dam. The main diversion channel will divert runoff around the proposed waste facility from an 87 km^2 catchment area.

The channel is 10 m wide at its base and the flow section deepens from 4.5 to 5.5 m near the Main Tailings Dam. Along most of the channel section in the East Fork Valley, the natural slopes are steep and will be buttressed with NPAG waste rock or quarried rock.

Downstream of the Main Tailings Dam, the channel will connect with the emergency spillway outlet channel down to Galore Creek. This outlet channel section is approximately 0.9 km long. The channel invert varies from elevation (above sea level) 678 m at the tailings dam alignment to 390 m (approximately) at the valley bottom and will be excavated into bedrock. If overburden is encountered in the outlet channel during construction, an erosion resistant lining of large riprap with appropriate filter layers will be required. If the bedrock quality is poor, fibre-reinforced shotcrete may be required.

Excavation of the Main and Starter Dam Diversion Channels will result in large cut and fill slopes. With the average natural slope angle of eighteen degrees along the proposed alignment, a 30 m high soil cut slope will be produced along the majority of the main channel length. A shallow perched water table is expected within portions of the overburden along the channel alignment, while water levels in the bedrock range from deep static levels to flowing artesian conditions at the ground surface. Perimeter drains excavated into a trench on the uphill base of the slope will be used to intercept and reduce water flows and water pressures. The fill slope will be approximately 4 to 5 m high and a footprint approximately 30 m horizontally, assuming a 10 m wide access/maintenance road.

An access road will be built downslope of the diversion ditch to allow inspection and maintenance of the channel. Water lines returning supernatant from the pond will be buried below the 10 m wide road surface. The tailings slurry line(s) will be placed on a bench on the downhill side of the road to provide protection from avalanches while allowing access for maintenance. Any leaks from the line will report to the tailings pond or waste rock dump.

A 17 m high earthfill dam (or East Fork Diversion Dam) is required at the upstream end of the Galore Creek east fork drainage to divert water into the Main Diversion Channel.

An 84 m high waste rock aqueduct (or East Fork Aqueduct) is required across the East Fork near the confluence of the East and West forks to eventually convey flow from the West Fork Diversion Channel into the Main Diversion Channel. It will be constructed entirely of waste rock with the exception of a lined channel along the crest.

20.2.5.5 Major Crossings

Where the diversion channels cross other streams or gullies, energy dissipation structures and sediment traps are planned to capture and divert flows into the main channel. Three major crossings have been identified: Bear Creek, Friendly Creek and Camp Creek. In these areas, the creeks have eroded steep incised gullies that are susceptible to rockfall, debris flows and snow avalanches.

The crossing structures have been designed to pass regular water flows while retaining solids that may pass down the creek during high flows or even debris floods. Debris flow barriers consisting of a gabion rockfill structure with a spillway overlying a base of slush-grouted riprap with culverts are proposed. Excess flow will pass through the upper spillway on the crest. During debris flow events, it is expected that the culverts will plug and debris will fill the upper basin. Water will then flow through the spillway on the crest of the debris flow barrier. The debris will have to be removed following these events. The structure will pond only minor amounts of water.

20.2.5.6 Dendritic Creek Channels

For the north and south Dendritic Creek channels, a water pickup and diversion structure will be required across the Dendritic Creek drainage. For the northern channel a single crossing at elevation 750 m above sea level can be constructed. For the southern channel, several smaller crossings between elevations (above sea level) 800 to 1000 m will be required. Debris flow mitigation will be required at specific locations within the Dendritic Creek drainage.

Energy dissipation structures and coarse sediment traps will be required at various locations where there is a large change in channel slope.

20.2.5.7 Central Pit Seepage Control Structure

Seepage from the tailings pond is expected to report through the waste dump and into the eastern wall of the Central pit where is meets the West Fork Valley. All seepage reporting to the toe of the dump will be monitored; however, if the seepage is higher than predicted, an impervious soil face can be constructed on the downstream slope of the waste dump when required.

In this same area, a small winter stockpile will be located near the proposed truck shop. Downstream of the confluence, there will be a separate cell for over-wet overburden. A snow dump will be located upstream of the confluence, in the East Fork Valley, separate from the overburden cell to ensure melting of the snow pack.

20.2.6 Construction Material and Volumes

Construction material for the Main Tailings Dam, diversion dams, and cofferdams will be primarily waste rock derived from mining operations. The mine fleet will be used to excavate, transport and place as much of the construction material as possible. Impervious soils are also required for these dams. Cohesive glacial deposits may be present in the pit overburden that may be used for core construction of the Main Tailings Dam. However, given the possibility these soils may be excessively moist and/or heterogeneous, and the likelihood that the core will be placed by a smaller truck fleet for traffic-ability, the potential to obtain impervious soils from a local borrow source has also been assessed. At present, four local areas of potential impervious borrow have been identified within the valley.

For the tailings dam, processed granular materials for the filters and geomembrane cushion/filter will also be required. It is presently envisaged that a process plant will be constructed near the mine to manufacture road-surfacing material for haul roads. This plant will initially be used to manufacture required granular material quantities for dam construction. In addition, suitable granular borrow materials appear have been identified at several locations within the valley. Generally all boreholes in the area indicate the presence of granular resources.

Table 20-2 summarizes the earthworks volumes for the Main Tailings Dam (starter and ultimate) and other diversion structures.

Table 20-2: **Total Cumulative Earthworks Volumes For Each Diversion Structure**

Structure Name	Total Cumulative Earthworks Volume (Million m^3)
Main Tailings Dam - Starter	29.8
Main Tailings Dam - Ultimate	66.1
East Fork Aqueduct	9.0
East Fork Div	0.4
Central Pit Seepage Control Structure	1.7
West Fork Diversion Dam	0.03
Primary Cofferdam	1.4
Temporary Cofferdam	0.02
Bear Creek Crossing	0.2
Friendly Creek Crossing	0.4
Camp Creek Crossing	0.5

Based on the design assumptions of the diversion channels, the excavation volumes have been calculated and summarized in Table 20-3.

Table 20-3: **Diversion Channel Excavation Volumes**

Diversion Channel	Excavation Volumes (Million m^3)		
	Soil	**Rock**	**Total**
Main			
Main Dam to E Fk Div Structure	6.23	0.24	6.47
Main Dam to Galore Ck outlet	0.68	0.09	0.77
West Fork	0.88	0.02	0.90
Starter Dam – East	1.37	0.14	1.51
Starter Dam – West	0.56	0.06	0.62
Dendritic Creek – North	0.44	0.08	0.53
Dendritic Creek – South			
Lower section	0.36	0.00	0.36
Upper section	0.22	0.18	0.40

20.2.7 Waste Dumps

Detailed planning and waste dump layout are being undertaken by NovaGold and Hatch. BGC are supporting this effort by providing geotechnical design constraints and considerations.

Waste material from the open pits can be subdivided into three types: overburden (i.e. glacial till), broken waste rock, and stick waste rock. Table 20-4 summarizes the estimated total tonnages of each material type, including the amount of PAG and NPAG waste expected based on ABA testing (Hatch, PEA 2005).

Table 20-4: Waste Rock Types and Tonnages

Waste Types	Tonnages (Mt)
Overburden	166
Broken NPAG	180
Broken PAG	227
Stick NPAG	181
Stick PAG	165
Junction (Un Zoned)	96
Total	1,016

The proposed in-valley rock dump is to be located in the middle and upper valley of Galore Creek, or immediately south of the tailings solids. The dump will be used to contain material originating from the rock stripping required prior to and during mine operation.

The PAG waste rock will be placed in the valley bottom to an elevation below 674 m above sea level, the ultimate spillway invert elevation or ultimate water level, so all PAG waste will be flooded at the end of the mine life. The NPAG dumps will rise up the valley walls as required.

The overburden waste will be placed in upper Galore Creek Valley in a variety of locations, so it can be reclaimed later as part of revegetation.

The waste rock dump foundations vary as follows:

- The valley bottom is a steep walled, deeply incised canyon. The sides of this canyon consist of exposed volcanic rock. A layer of fluvial sand and gravel overlies the volcanic rock in the active floodplain in the valley bottom;
- Above and outside the canyon, the ground flattens with natural slopes angles ranging from 0 to 30 %. The area comprises glacial deposits varying from 0 to 50 m thick overlying bedrock. These deposits are mixture of gravels, sands, silts and clays;
- Glaciolacustrine silts and clays are widespread and could be continuous throughout the valley. At some locations, this clay is soft and exhibits an undrained strength of 25 kPa.

The key elements of the waste dump design are outlined below:

- A haul road will initially be built using waste rock from the Central Pit, to the plant site, and down the valley to the Main Tailings Dam area.
- Once the road infrastructure is built, pre-stripped material will be placed in the existing valley without additional soil stripping.
- The slope of the waste dump divider facing the tailings pond will be built on glaciolacustrine deposits and will be built as an engineered structure.
- The waste dump will generally be built from the valley bottom up.
- The first dump lift is contained within the steeper, incised portion of the valley and may be built up to 550 m above sea level by end dumping in one lift (if desired).
- In areas of soft foundation, the waste dump can be built in lifts by one of two methods using an observational approach:
  - *1) Thin lift controlled approach* - The dump is built from the valley bottom up, using an initial thin lift to bridge over the soft soils. The remaining lift thickness will be modified during construction following monitoring of the initial placement to ensure safety is adequately addressed.
  - *2) Thick lift approach* - Thick lifts could be constructed providing that during operations, the dump would be closely monitored to ensure the safety and reliability of the operation.
- Waste rock may be placed in the dump year round, however all snow must be removed from the previous lift prior to placement of the next lift.

A series of NPAG waste dumps have been proposed on the natural side slopes below the North Junction Pit. The dumps will be built as side hill structures on slopes varying between 0 to 45°. The dumps will be built as wrap around dumps from the bottom up.

Dumps founded on soft glaciolacustrine clays will be started using a low level dump with lifts similar to those noted above. This will act as a stabilizing toe or wrap around. Once the toe is stabilized additional side hill dumping will be undertaken with monitoring to ensure safety.

During construction of the waste dumps, visual and instrumentation monitoring is planned to monitor the deformation of the dump slopes. The monitoring is expected to consist of simple crest extensometers which measure and provide deformation rates of the dump crest and upper slopes. This monitoring will be initiated when the dumps are built on the glaciolacustrine deposits. Piezometers placed in the foundation soils will be used to monitor foundation pore pressures and the response to loading.

20.2.8 Closure

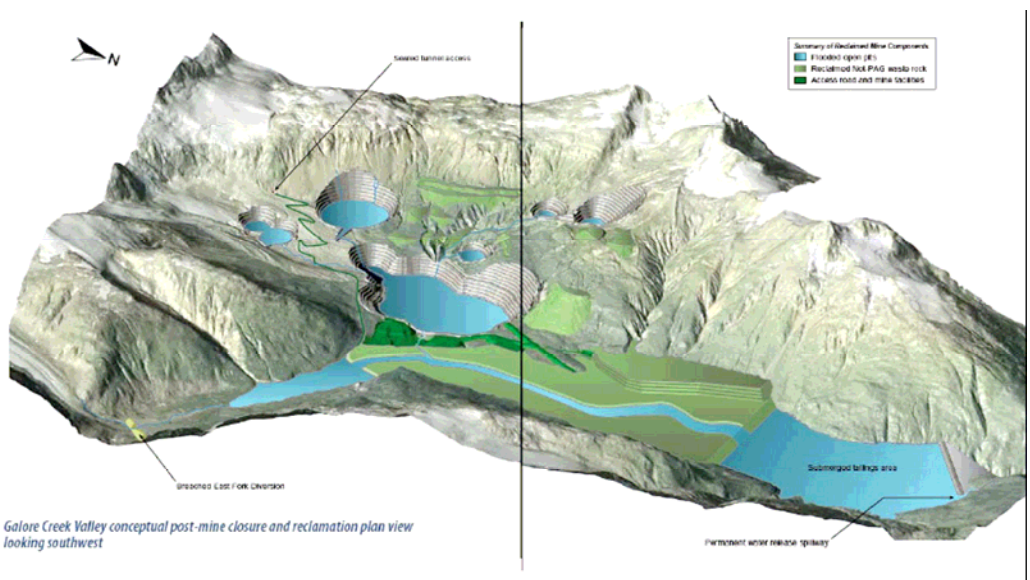
In accordance with best industry practice all the tailings and PAG waste rock within the impoundment as well as the open pits will be flooded at closure. The approximate elevation of the flooded surface is 674 m above sea level, equivalent to the spillway invert elevation at closure. All NPAG waste rock, located above the PAG waste or as higher elevation waste dumps in the Junction area, will be reclaimed. Approximately 1.7 km^2 of the pit slopes (i.e. high wall of the central pit and high walls in the satellite pits) will be exposed.

All freshwater diversions (including pit dewatering wells) will be decommissioned. The East Fork and West Fork Diversion Dams as well as any major energy dissipater structures along the diversion channels will be breached.

To protect the integrity of the Main Tailings Dam (by controlling the pond level), an emergency spillway on the right abutment will be excavated into rock. This spillway will be designed to handle the PMP and must remain in operation for as long as the dam exists.

For the exposed NPAG waste rock above the PAG waste rock, a cover system would include revegetating the entire area above water. The exposed surface area is approximately 5.7 km^2. All dumps will be recontoured to an angle of approximately 26 degrees followed by revegetation of the surface. The exposed surface area of these proposed high elevation dumps is approximately 1.35 km^2.

Figure 20-11: Galore Creek Valley Conceptual Post Mine Closure View



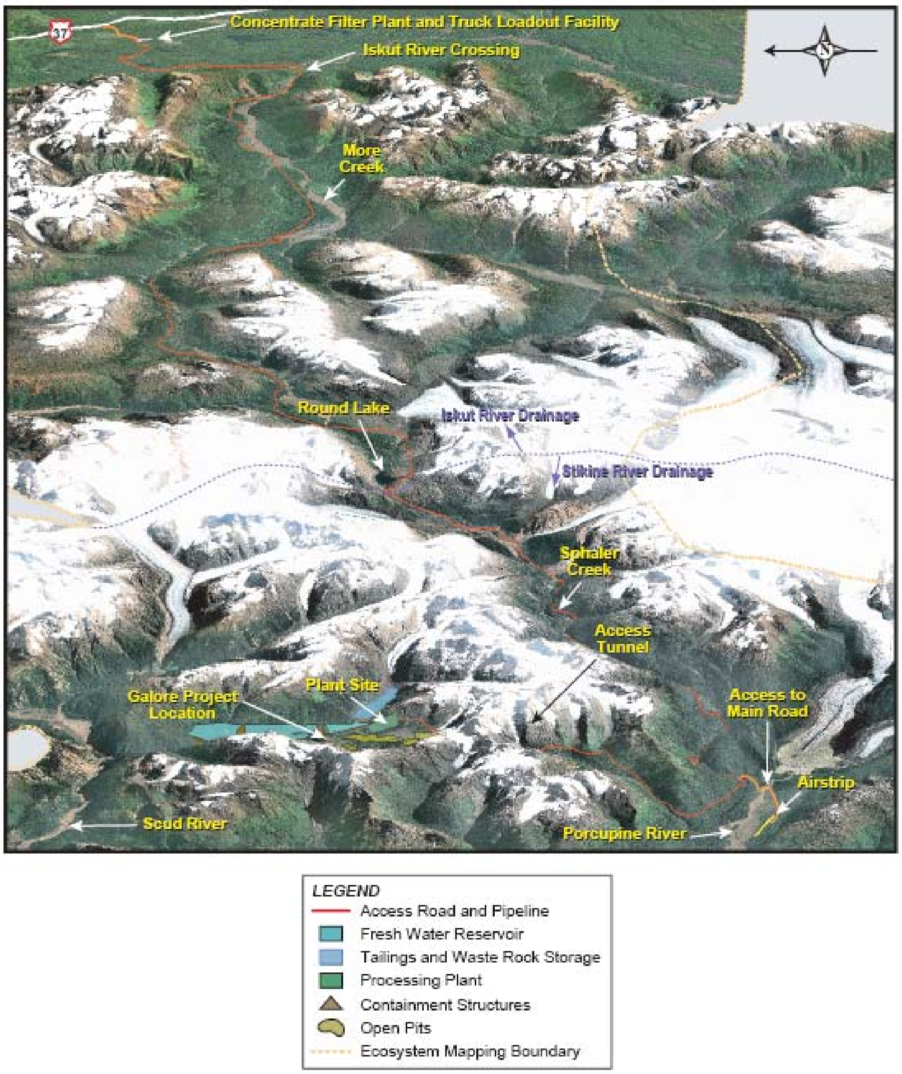
Galore Creek Valley conceptual post-mine closure and reclamation plan view looking southwest

20.3 Infrastructure – Off Site

20.3.1 Summary

The off site infrastructure for the Galore Creek project includes the infrastructure associated with providing an access supply corridor for goods and services. The access corridor 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.

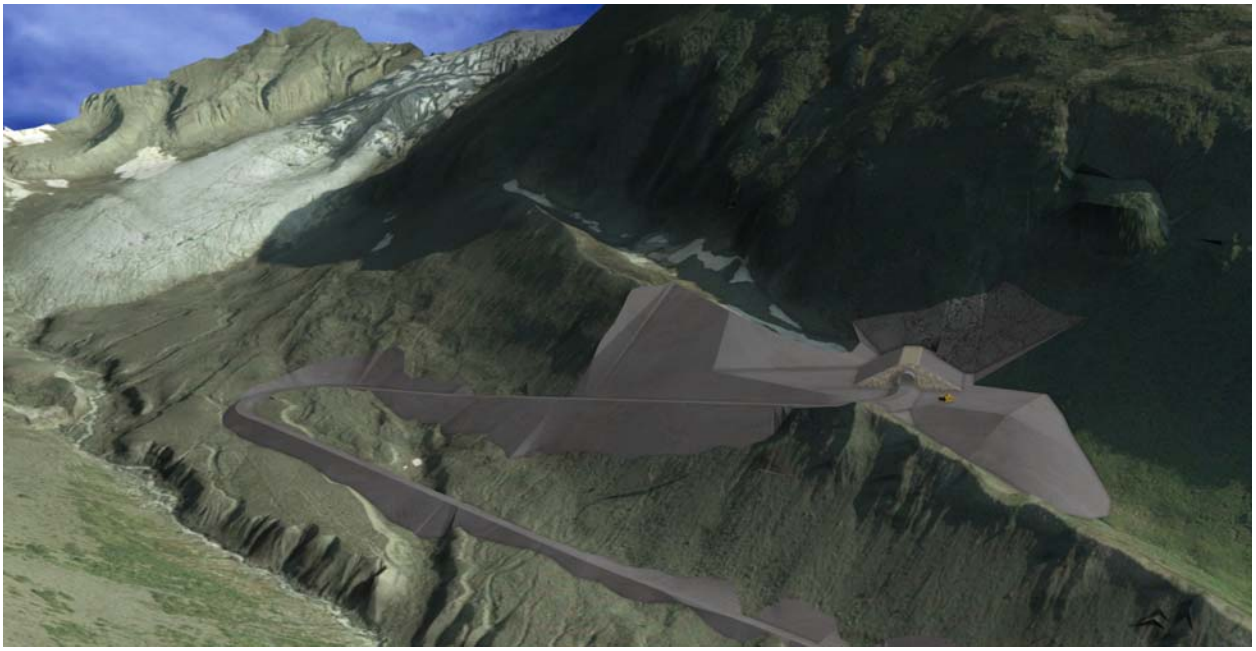
Figure 20-12: Access Alignment



Elements of the access corridor include:

- Mine Access Road
- Power Transmission Line
- Mine Access Road Tunnel
- Concentrate / Diesel Pipeline

Figure 20-13: Tunnel South Portal Scotsimpson Creek



20.3.2 Mine Access Road

Section 20.3.2 was prepared by Mr. Robert W. Parolin,P.Eng. McElhanney Consulting Services ltd.

20.3.2.1 Road Engineering

The access road with 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. The section of road through the Sphaler Creek is the most difficult with respect to gradient, curves and geohazard risk.

20.3.2.1.1 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. Representatives of the Tahltan First Nation familiar with the local climate, wildlife and topography were instrumental in selecting the final route alignment. 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 which involved walking the 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.

20.3.2.2 Detailed Route Description

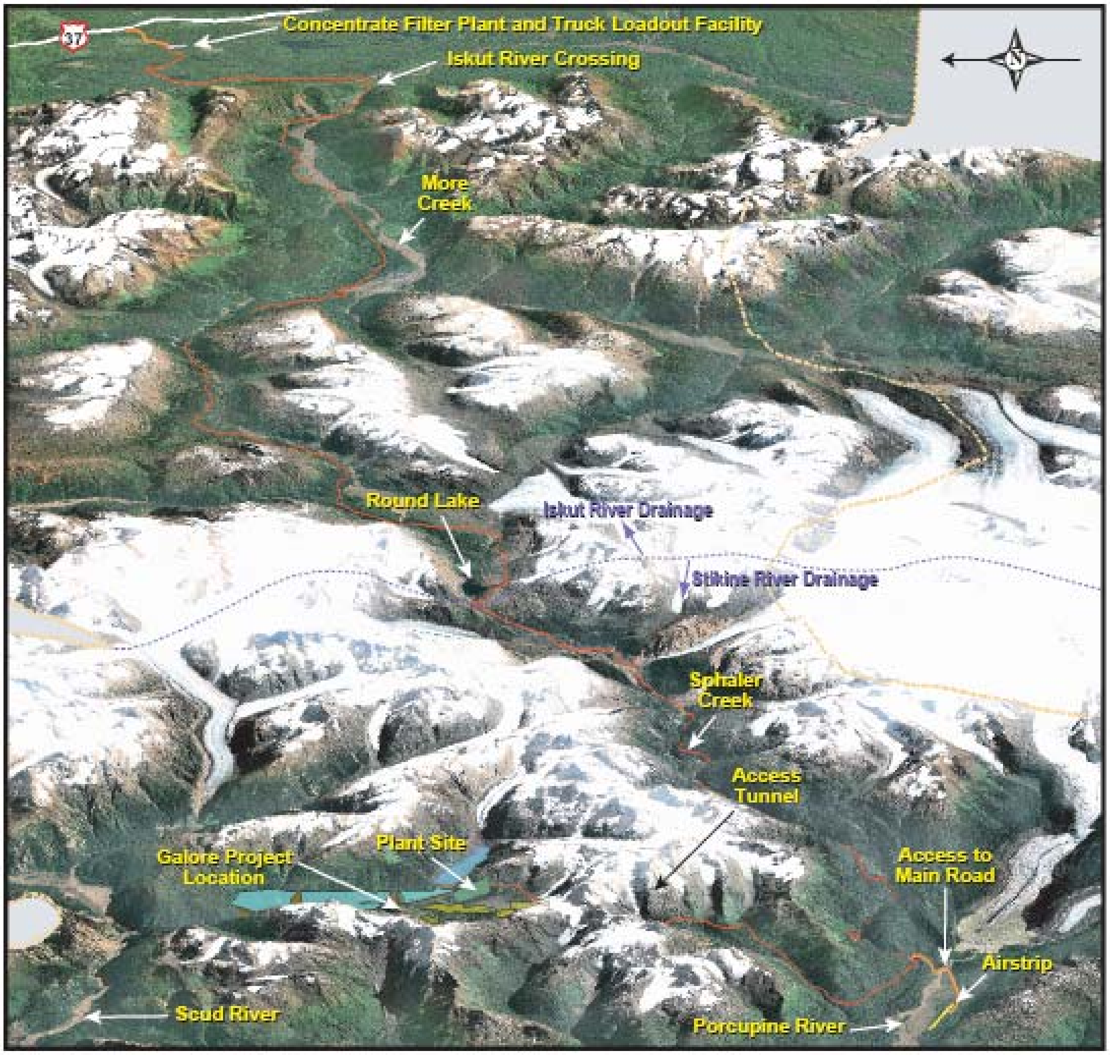
The primary objective was to locate and GPS the most appropriate road alignment. The road was to be established from Highway 37 near Bob Quinn Lake into the Galore Creek deposit. The road alignment is based on standards established by the project team and on information from field based environmental and geotechnical studies and the traditional knowledge of the Tahltan First Nation. Numerous cost/risk/benefit trade-off studies around design standards, environmental impact and geohazards had to be made to determine the optimal alignment.

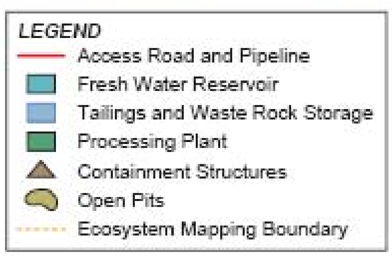
The selected route is described below and shown on Figure 20-14:

- The Lower More section (km 0-48.0) commences at Hwy 37, 15 km north of Bob Quinn Lake and follows the existing Thomas Forest Service Road for 7.5 km. New construction begins here and the route crosses the Iskut River immediately upstream of the More Creek confluence at km 15. From here the proposed road location does a switchback and climbs up and over a rock nose above the lower More Creek canyon. Before leaving the canyon, the route passes under potentially unstable rock falls and along the toe of an active slide.
- More Creek valley eventually widens beyond km 20 and the route traverses the north side of the flood plain up to the More Creek Bridge Crossing (km 38). The P-line then climbs steadily at a 10% grade to above the upper More Creek canyon.
- At this point there are two route options; to either traverse the steep unstable south slope to a bridge crossing at km 48, or install a major bridge across the canyon (approximately 250 m) at km 46.3, then construct the road along the gently sloping north side of More Creek. The feasibility design includes the cost for the bridge option.
- The Upper More section (km 48 – 76) follows along the north side of the valley to the headwaters of More Creek above the tree line at km 66. In the area of the Roca Minerals exploration camp the route swings south, crossing More Creek at km 68 then climbs to an elevation of 1200 m near the toe of Andrei Glacier. This section ends at Round Lake.
- Beyond Round Lake, the road will be located down the Sphaler Creek drainage (km 76 – 113). The first section is primarily alpine and sub-alpine terrain with several unnamed creek crossings at the bottom of deep wide gullies and outflows from the glacier between large medial and lateral moraines.
- The Sphaler Creek canyon commences at km 90 and the road climbs steadily to reach a bench at km 91 before starting to lose elevation. Numerous snow slides and rock falls are encountered along the south slope before a major bridge crossing at km 95.4. The road then stays on the north side of Sphaler Creek until approximately km 105 where another bridge crossing is necessary.
- On the south side of Sphaler Creek to km 109, the road crosses again to the north side of Sphaler Creek and climbs up and out of the Sphaler drainage onto the north slope of the Porcupine River drainage.
- The final section of the North Route is the Porcupine section (km 113 – 125.5) which follows the north side of the Porcupine River to km 116 before climbing into the Scotsimpson Creek valley and reaching a proposed tunnel portal near the headwaters at km 125. The upper Scotsimpson Creek is open alpine terrain with numerous avalanche chutes extending to the valley bottom.

- A separate access road from km 114 south across the Porcupine River to an airstrip is included to allow operations to have access / air support local to the facilities.
- Winter road access for construction equipment from Scotsimpson Creek over the glacier and down into Copper Canyon was also investigated and found to be feasible although current construction schedules do not require this approach. Access would require steeper grades (max. 20%) and the use of snow and ice fills combined with matting to bridge the numerous crevices over a 500 m section of the ice field. The access road alignment is shown on Figure 20-14.

Figure 20-14: Access Alignment





20.3.2.3 Estimated Road Traffic

20.3.2.3.1 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, food and water.

20.3.2.3.2 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, as well as operations and maintenance personnel, food and water. The traffic frequency including the maximum daily trips is summarized below in Table 20-5.

Table 20-5:Traffic Frequency Estimate

Truck Type	Load	Delivery Frequency		Total Annual Trips	Maximum Daily Trips
20,000 L tanker	MIBC	1	One week	52	1
40 tonne Flat Deck Trucks	PAX, Flocculant, Maintenance Supplies	1	Two weeks	26	1
40 tonne Tandem Trucks	Lime	6	Week	310	1
	Grinding Balls	10	Week	520	2
	Explosive	2	Day	500	2
	Concentrate (Hwy 37)	40	Day	13,870	50
40 ft Trailers	Food	3	Week	156	1
Super B Train	Diesel (Hwy 37)	3	Day	1,100	2
50-person Buses	People	2.5	Week	130	1
Small Vehicles	People	7	Day	2,500	10
Total Highway 37				**19,164**	**71**
Total Mine Access Road				**4,620**	**21**

20.3.2.4 Road Design

McElhanney, a Northwestern BC Engineering firm has extensive experience in road design in the region and provided continuity to the project as they were also involved in the preparation of the Scoping Level Study and Pre-feasibility Study. The road design considered the following major objectives:

- Complete the final route selection
- Locate and survey the preliminary road centerline
- Complete site surveys for all fish bearing streams and major creek crossings
- Perform hydrology analysis for bridge and culvert installations
- Arrange for geophysical survey of major bridge crossings
- Design the road horizontal and vertical alignment to avoid or mitigate impact of geohazards

- Design road crossing bridge and culvert structures
- Incorporate mitigation measures in the road design
- Generate earthwork quantities
- Prepare a construction cost estimate
- Prepare a road construction schedule of activities.

The final route selected starts at the junction of Hwy 37 and the Thomas Creek Forest Service Road and proceeds west across the Iskut River, up More Creek to Round Lake, then down Sphaler Creek to the Porcupine River before climbing up to the south tunnel portal at the headwaters of Scotsimpson Creek.

Figure 20-15: Mine Access Road

A single lane (6m wide) resource access road is planned to support construction of the slurry pipeline, and the power transmission line and provide supplies, equipment, and crew transport during construction and operation of the mine. With 15% grades and an average design speed of 40 km/hr this is intended to be a low impact road within the utilities corridor.

Bridges and culverts are designed for the 200 year and 100 year instantaneous flood, respectively, with a minimum 1.5m clearance to the underside of the bridge girders unless additional clearance is required for navigable waters or geotechnical requirements. Seismic refraction surveys were conducted by Frontier Geosciences Inc. on the following major bridge crossings:

Table 20-6: Major Bridge Crossings

Iskut River	km 15.5	107 m span
Muskwie Creek	km 30.4	97.5 m span
Lower More Creek	km 39.2	91 .4 m span
Eros Creek	km 82.6	48.8 m span
Yurie Creek	km 84.0	88.4 m span
Saddle Creek	km 89.4	76.2m span
Maurer Creek	km 90.2	73m span
Sphaler #1	km 96.9	67m span
Porcupine River	km 115.0	84m span (Porcupine Connector)

In total, there are 130.6 km of road construction required to access the south tunnel portal. Included in this are 11 km of upgrade to the Thomas Creek Forestry Service Road and 2.1 km of new construction of the Porcupine connector road. There are 50 bridge crossings, and 11 open bottom arches plus a range of round elliptical, corrugated steel culverts.

The road design criteria is summarized in Table 20-7.

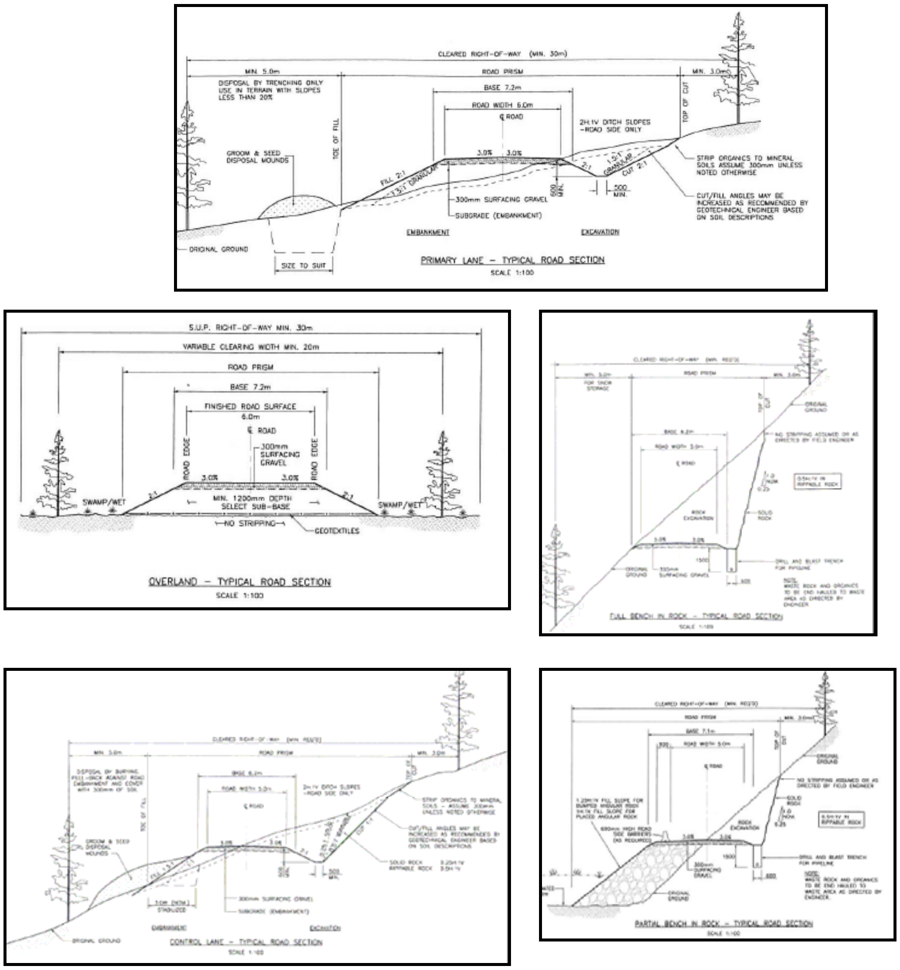
Table 20-7: Design Criteria:

Classifications	Single Lane
Average Daily Traffic (ADT)	≥ 50
Design Speed (km/hr)	≤ 30 ≥ 60
Min. SSD (m)	65
Min. Radius (m)	35
Switch-Backs – Min. Radius (m)	15
Switch-Backs – Max. Grade %	10
Min. K. Factor – Sag	5.1
Min. K. Factor – Crest	3.1
Max. Grades	15%
Road Width (m)	≤ 5.0 ≥ 6.0
Pull-out Width	Add 4.0m
Right-of-way (m)	≤ 30

The design and drafting is in accordance with Ministry of Forests Standards as described in the Forest Road Engineering Guidebook.

Figure 20-16 below depicts typical road sections of the Mine Access road.

Figure 20-16: Typical Road Sections:



In order to meet the proposed construction schedule, three construction camps are planned:

- Bob Quinn Camp at 3 km on the Thomas Creek Forest Service Road.
- Roca Camp near 73 km of the access road
- Porcupine Camp on the south side of the Porcupine River

Both Roca Camp and Porcupine Camp are remote locations requiring air support for mobilization and operation.

The cost to monitor and control geohazards is outlined in BGC's Geohazard Assessment Report is included in the operational cost estimate. In addition, there are special mitigation structures such as no-post barriers, lock block walls, earth berms and snow sheds, required to protect the road and pipeline. An allowance has been included in the cost estimate and construction plan for these structures.

The construction of the Galore Creek Access Road is a major undertaking requiring the falling of 80,000 m^3 of timber; clearing 400 hectares of right-of way; moving 3.0 million m^3 of rock and earthwork, and placing over 300,000 m^3 of surfacing material.

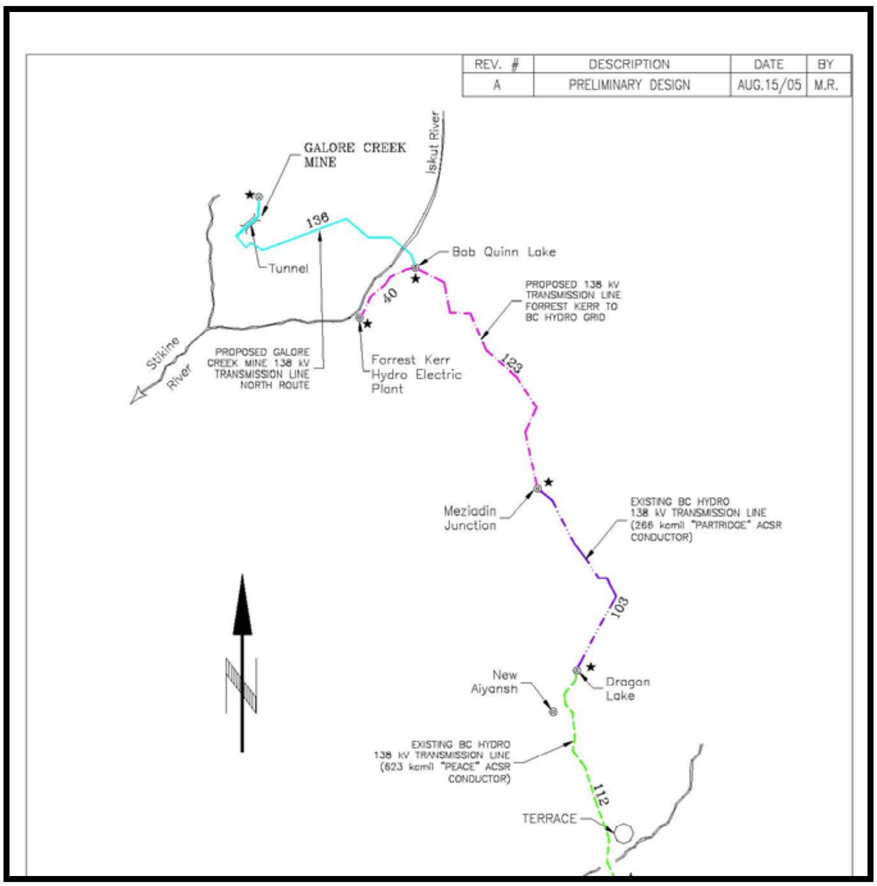
20.3.3 Power Supply

Section 20.3.3 was prepared by Mr. Allan Guy,P.Eng. Ian Hayward International Ltd.

20.3.3.1 Summary

The project is currently isolated from power and other public infrastructure. A 138 kV transmission line has been proposed to connect the minesite with a maximum recommended peak load capability of 92 MW of electrical power (with an ultimate capacity of 98 MW). It has been determined that the estimated load at the mine site will have a maximum peak load of 87 MW. Electrical power will be supplied from the BC Hydro (BCH) grid from a point near Coast Mountain Power Corp's (a Division of NovaGold) (CMPC) Forrest Kerr run-of-river hydropower plant (Forrest Kerr). A map of the existing grid and the proposed connection configuration is shown in Figure 20-17.

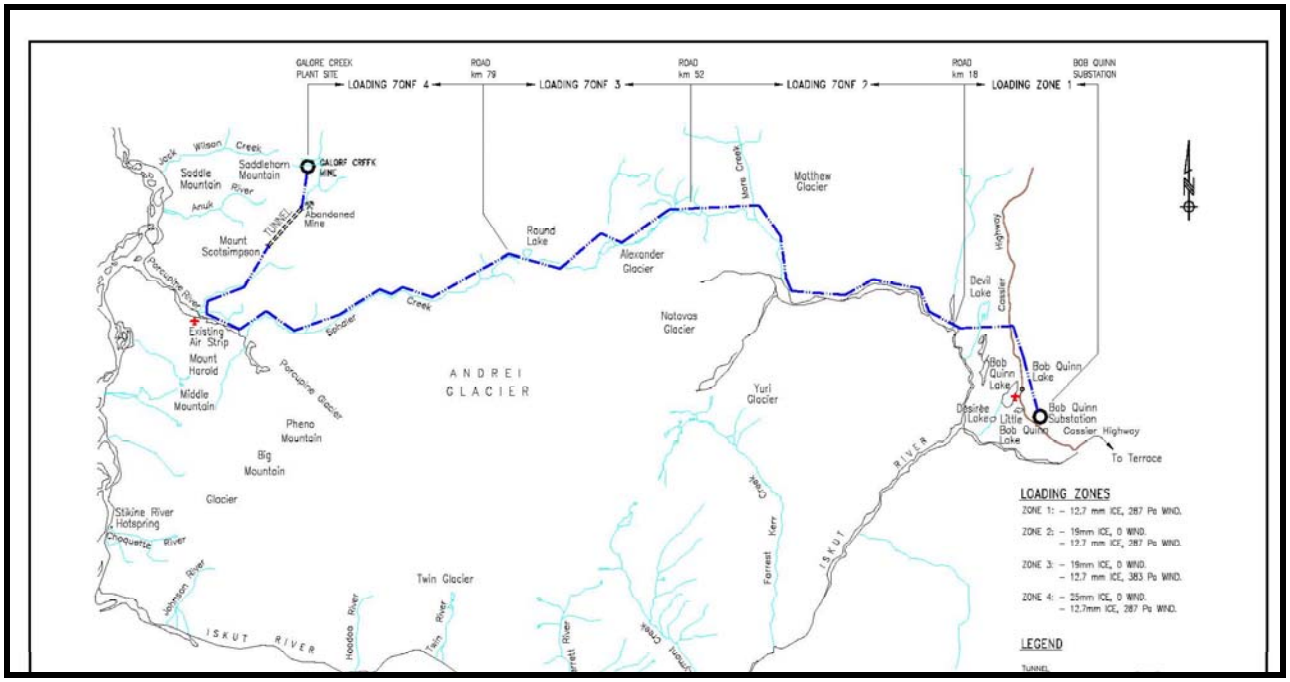
Figure 20-17: Existing Grid Map



20.3.3.2 Transmission Line Route Selection

The Feasibility 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 20-18.

Figure 20-18: 138 kV Transmission Line



Segments of a transmission line are categorized in terms of ease of construction:

Easy	Flat gentle rolling terrain
Medium	Rolling terrain with gullies and / or some sidehill
Medium Rough	Along valley sidehills with intervening creeks and rivers and well defined slide paths
Rough	Along steep sidehills with intervening creeks and rivers, numerous slide areas
Very Rough	Similar to rough but in winding valleys with heavy snow-pack, or high exposed routes through high mountain passes

Access is also considered in assigning terrain categories. For example, otherwise terrain be designated Medium Rough or Rough if there are few or no access roads into the area.

The majority of the route description is described based on the access road kilometre.

- Bob Quinn Lake to Iskut River

This section of transmission line follows the east side of Highway 37 north from the Eskay Creek road junction for 8 km and then turns northwesterly for another 8 km crossing the Iskut River. The transmission line does not follow the access road in this section from Highway 37 to the Iskut River. The Iskut River crossing is a relatively short span and will not require special structures. The terrain is classified as Medium.

- Lower More Creek Section Road km 16 to 50

From Road km 16 to 30, narrow rock canyons are encountered as the route follows up More Creek in a westerly direction. This section of transmission line is Very Rough. Some H-Frame woodpole structures and special conductors may be needed in this section to increase the span lengths. A large slide (hazard) area exists in the Lower More Creek canyon. Some deliberation must be made in the design stage, whether the hazard can be spanned across with H-frame woodpole structures, or constructed as a sacrificial section of standard woodpole spans.

From the Road km 30 to 39, the More Creek valley sides are not as steep as the previous section and the transmission line can closely follow the road through most of this section. There will be a mix of rock and soil foundation conditions. The terrain is classified as Rough.

From the Road km 39 to 50, the transmission line will not follow the road and will be located on the north side of More Creek. Some H-frame woodpole structures will be used to span numerous narrow rock creek canyons. Heavy tree clearing will be required throughout this section. The terrain is classified as Very Rough due in part to lack of conventional access.

In this Lower More Creek section the maximum height of hemlock, balsam, spruce and other kind of trees will be approximately 30 m.

- Upper More Creek Section Road km 50 to 65

The terrain moderates in this section and the transmission line will closely follow the road through most of this section. The terrain is classified as Rough. Some tree clearing will be required.

- Mess Creek Section Road km 65 to 78

The terrain moderates in this section and is designated Rough. It is anticipated that this section may be susceptible to heavy snowfalls and heavy ice loading conditions particularly around the Round Lake area.

- Upper Sphaler Creek Section Road km 78 to 103

The terrain in this section is classified as Very Rough. The side slopes are extremely steep and it is heavily treed throughout most of this section. Given the extreme level of terrain difficulty, the transmission line might be forced further away from the road. Some avalanche areas will require steel pole structures. This section is subject to heavy snowfalls and ice loading conditions.

- Lower Sphaler Creek Section Road km 103 to 118

From the Road km 103 to 111, the terrain is classified as Medium. The transmission line will closely follow the road in most of this section.

From the Road km 111 to 118, the terrain is very similar as in the Upper Sphaler Creek section and is classified as Very Rough. The side slopes are extremely steep and covered by trees throughout most of this section. Many structures will be in close proximity to the road but substantially above it. The access to these towers will be possible only by helicopter. Some avalanche areas, which cannot be avoided, will require steel pole structures.

This whole section, from the Road km 103 to 118, is subject to heavy snow falls and ice loading conditions.

- Scotsimpson Creek Section Road km 118 to 129

This section is approximately 11 km long from the Porcupine River, terminating at the access tunnel to the mine and plant sites. From the Porcupine River, the transmission line turns in a northerly direction towards the access tunnel. Generally the terrain is Very Rough. The risk from avalanches is extremely high over a significant portion of this section. Snowfalls and heavy ice loading are significant factors. Underground XLPE cable from km 121 to 129 will be used to mitigate operational issues associated with potential avalanches causing damage and outages on overhead lines.

- Tunnel Section

The overhead transmission line will convert to XLPE cables through the access tunnel. A suitable pothead site will be selected on either side of the tunnel to locate the structure and equipment to convert the transmission line from overhead to trench/cable tray supported XLPE cables.

- Mine Section

Beyond the access tunnel, a further 5-km of overhead line will be required to reach the plant site.

20.3.4 Access Road Tunnel

Section 20.3.4 was prepared by Mr. Dean Brox, P.Eng. Hatch Mott MacDonald (HMM)

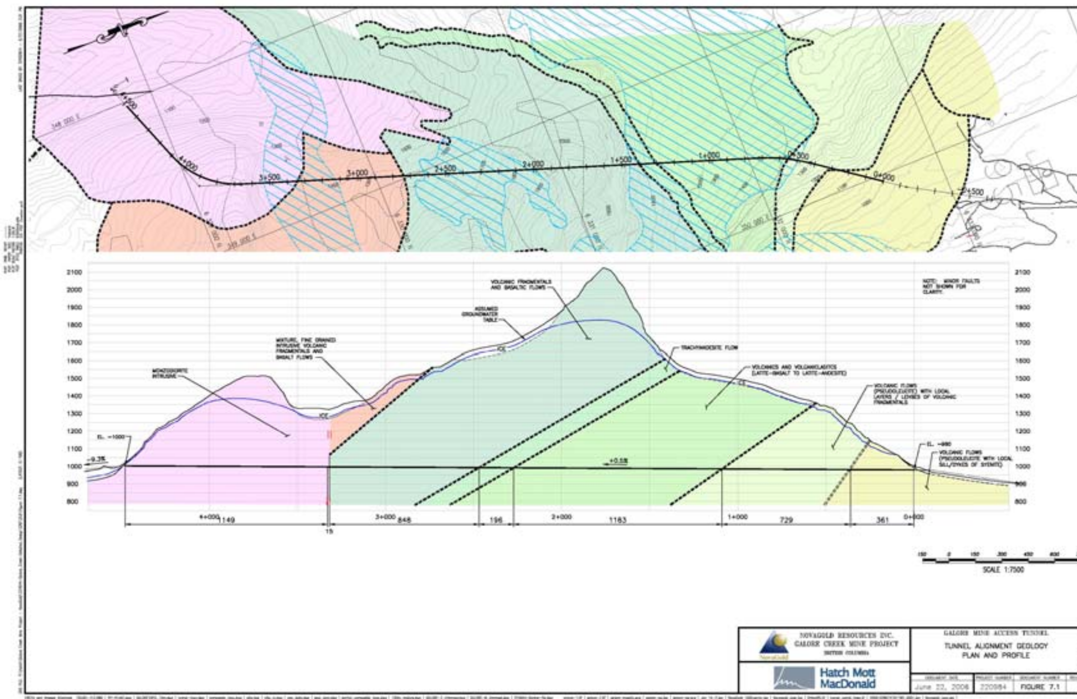
20.3.4.1 Summary

Access to the Galore Creek Valley and minesite will require construction of a 4.3 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 6.9 m wide x 6.5 m high (excavated dimensions approximately 8 m wide x 8 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.

20.3.4.2 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 20-19 shows the horizontal and vertical alignments of the tunnel, as well as geological information. The south portal elevation is estimated at approximately El. 1,100 m, while the north portal is anticipated at approximately El. 1,020m. 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 20-19: Mine Access Tunnel Plan and Profile

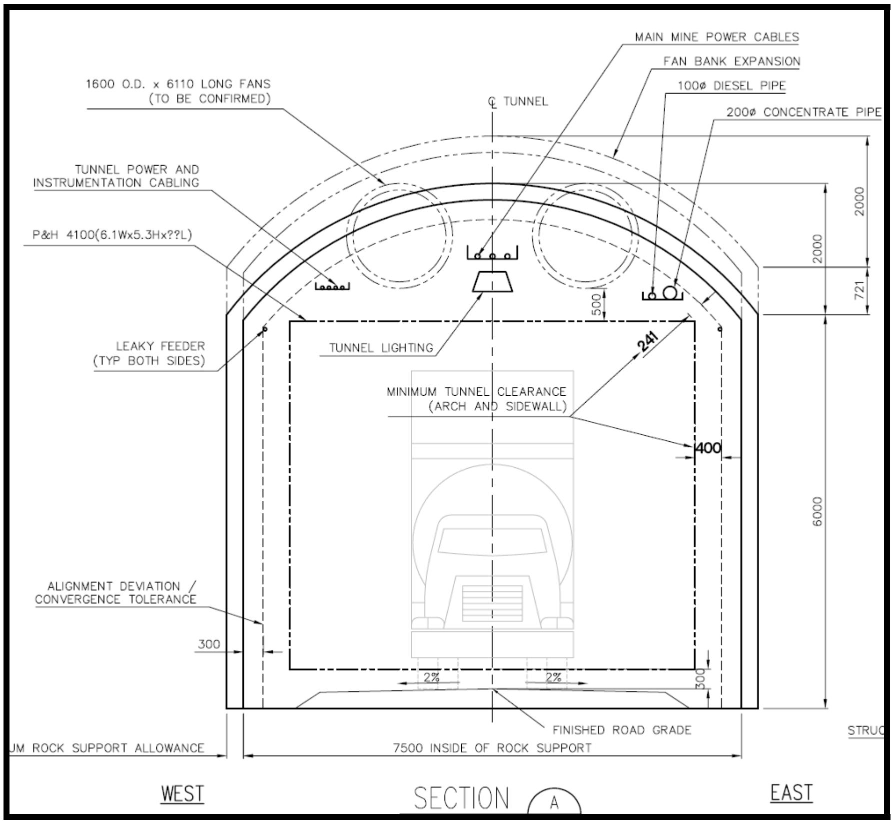


20.3.4.3 Access Tunnel Sizing

The access tunnel size was designed to allow passage for all of the equipment and supplies for initial construction and on-going operations. The equipment and supplies are expected to be broken down into the smallest practical size for transportation, but size reductions are limited by concerns over the effectiveness of site welding and consequent warrantee/liability issues and to significant non-standard engineering and manufacturing practices.

A tunnel cross-section is shown in Figure 20-20.

Figure 20-20: Road Access Tunnel – Clearances Sketch – Base Case

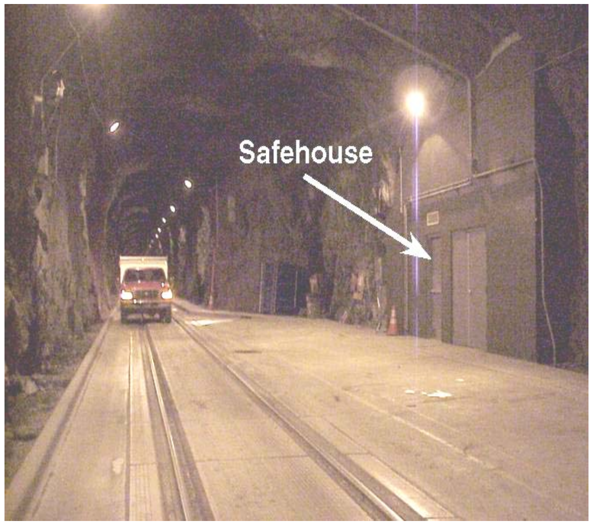


20.3.4.4 Expected Geological Conditions

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.

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.

Figure 20-21: Fire Life and Safety – Example of Tunnel Refuge Bay Station



20.3.4.5 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 are 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. 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 faces of the Stuhini Group. Overstressing of bedrock is manageable but 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.

20.3.5 Concentrate Pumping

Section 20.3.5 was prepared by Mr. Don Hallbom,P.Eng / P.E. Pipeline Systems Incorporated (PSI)

20.3.5.1 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. Feasibility estimates are 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. Filtrate will be treated and clean water will be discharged into the Iskut River. This option results in the lowest operating cost, at the expense of increased capital cost for the pumping system.

PSI was commissioned to carry out a feasibility level study for the concentrate piping system, and to develop scoping level cost estimates for the system. The design criteria of the system are summarized below in Table 20-8. A site visit was performed by a PSI pipeline construction specialist, in April 2005 and feasibility designs reflect additional testwork and information gathered in the field since that time .

Table 20-8: Pipeline Systems Summary

Design Parameter	
Design Throughput	87.7 tph
Concentrator Availability	95%
Concentration	57 wt%
Pipe Length	135 km
Outside Diameter	194 mm
Average Wall Thickness	6.4 mm
Design Pressure	350 Bar
Pipe Steel Tonnage	3,229 t
Design Flowrate	108 m^3/h
Number of pump Stations	1
Piston Diaphragm Pumps per station	1 Op + 1 Std by

20.3.5.2 Cold Weather Engineering

A primary consideration in developing the feasibility design for the pipeline operation is the cold weather experienced during winter months. Minimum winter weather temperatures of minus 20°C are typical of the area.

Depending on the depth of snow on the ground (snow actually insulates the soil from harsh air conditions), the frost penetration could be over a metre below ground. To protect the pipeline from cold weather, it will be buried below the frost line wherever possible. An allowance has been included in the cost estimate for deeper burial of the pipeline.

Experience on similar projects indicates that the risk of a pipeline freezing 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 in future project phases and is expected to show that the pipeline should be initially commissioned when the ground is not frozen (e.g. on water during the fall of 2010 prior to hot commissioning). During normal operation, friction will provide an internal heat generation source that will keep the exposed areas of the pipeline from freezing. Additionally, exposed areas will be insulated and heat traced as required.

In case of extended shut down, the pipeline will be flushed with water and a jockey pump will be run during extreme cold conditions. This will maintain flow and eliminate a risk of freezing. The thermal model will calculate the allowable duration of the shutdown, which will be compared to power reliability to determine the final design.

20.3.5.3 General Operating and Control Philosophy

The concentrate slurry pipeline is designed to operate continuously with a design throughput of 2,000 tpd (2,600 tpd maximum). The pipeline must also accommodate a 60% turndown (i.e. 1,200 tpd) through batch mode operation.

The design of the pipeline is such that capacity-control shutdowns will not occur as a normal operating practice. Short-term shutdowns with slurry in the line are possible 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.

20.3.6 Concentrate Truck Haulage and Port Handling

Section 20.3.6 was prepared by Mr. Bruce Rustad ,P.Eng. Hatch

20.3.6.1 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. McElhanney 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 highway haul, without significant difficult grades. The round trip time for this route is approximately 6 to 8 hours, and will enable the drivers to be based in Stewart.

Table 20-9 lists travel times for trucks on each section of roadway, as well as the estimated overall cycle times.

Table 20-9: Heavy Vehicle Travel Times

Direction	Section	Distance	Time
Inbound	Hwy 37A – Stewart to Meziadin junction	60.4 km	49 min.
	Hwy 7 – Meziadin junction 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			**346 min.**

Note: Loading/unloading times and other delays at Port in Stewart or at Bob Quinn Filter Plant are not included.

20.3.6.2 Concentrate Port Handling

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.

Production, as indicated earlier is expected to be about 600,000 t per year (50,000 dmt per month) for the first six years and about 500,000 t (40,000 dmt per month) thereafter. 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 say 60,000 to 100,000 t for larger sales down to say 30,000 to 40,000.

Generally the intent will be to spread deliveries to a smelter evenly over the year. Given these volumes two three ships per month will likely have to be scheduled in order to move production.

Parcel size for the shipments could vary from about 6,000 to 8,000 wet tonnes or about 12,000 to 14,000 wet tonnes or 18,000 to 26,000 wet tonnes depending on the number of holds utilized. It is expected that actual vessel sizes will be either around 25,000 / 30,000 Dead Weight Tonnage (DWT) with about 9 metres draft or 45,000 / 55,000 DWT with about 13, metres draft. The existing dock facility can handle ocean going vessels up to 50,000 DWT

At Stewart, there is an existing concentrate loader and at present, two 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 mine lives remaining. It is probable that such storage will become available about the time of Galore start-up.

Terminal operations and costs were discussed in 2006 with the owners of Stewart Bulk Terminals. They were confident that a project of this size could be handled with little difficulty. Terminal costs have risen considerably in the last few years especially fuel, taxes and insurance. The estimated CA$10.00 per tonne rate is up from present charges in the CA$7.50 - CA$8.50 range.

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 present environmental state at the present facilities.

If an additional warehouse was necessary, Stewart Bulk Terminals would likely finance construction of the shed at its cost and amortize against throughput. Space and plans for additional 50,000 tonnes of storage are in place at the facility.

The Stewart facility is non-union operation and this keeps costs competitive but more importantly allows greater operational flexibility. Ships currently are loaded 24 hours a day with no extra overtime cost and with very little notice.

Figure 20-22: Stewart Bulk Terminals Concentrates Load-out Facility with Deep Sea Vessel

20.3.7 Airstrip

Air support at two airstrip locations (Round Lake and Porcupine River) are considered for the construction and operation of the Galore Creek mine, access roads and the tunnel. Note: the Round Lake site is temporary and only considered during the road construction. The Porcupine River strip will be primarily constructed for operations.

20.4 Infrastructure – On Site

Section 20.4 was prepared by Mr. Bruce Rustad ,P.Eng. Hatch

20.4.1 Mill Building

The mill building has a total area of 13,630 m^2 and is divided into two sections - one that houses the grinding process (~8,000 m^2) and the other that accommodates the flotation process (~5,600 m^2). This sectioning is done to reduce the building size as the flotation area doesn't require the same building height as the grinding section where the 40 foot (12.2 m) diameter SAG and 26 foot (7.9 m) diameter ball mills are located. The grinding section has four cranes, ranging from 10 t to 50 t, located across its span supported by three internal lines of steel columns. The cranes will be used for maintenance of the equipment as well as providing make-up steel grinding balls to the three vertimills.

There will be a machine and welding shop located at the north end of the building to be used for minor maintenance. Above these two areas will be a shower facility for the mill operators which will include six showers and 50 lockers. Access to the camp will be from this end of the building.

Offices for the mill staff will be located above the two floors of MCC rooms which are located along the North end of the building. The control room will be located above these offices and will provide a good observation point for most of the process.

20.4.2 Truckshop Complex, Warehouse and Administration Building

The truckshop, warehouse and administration building will be contract engineered steel buildings and are shown in detail in the Feasibility Study.

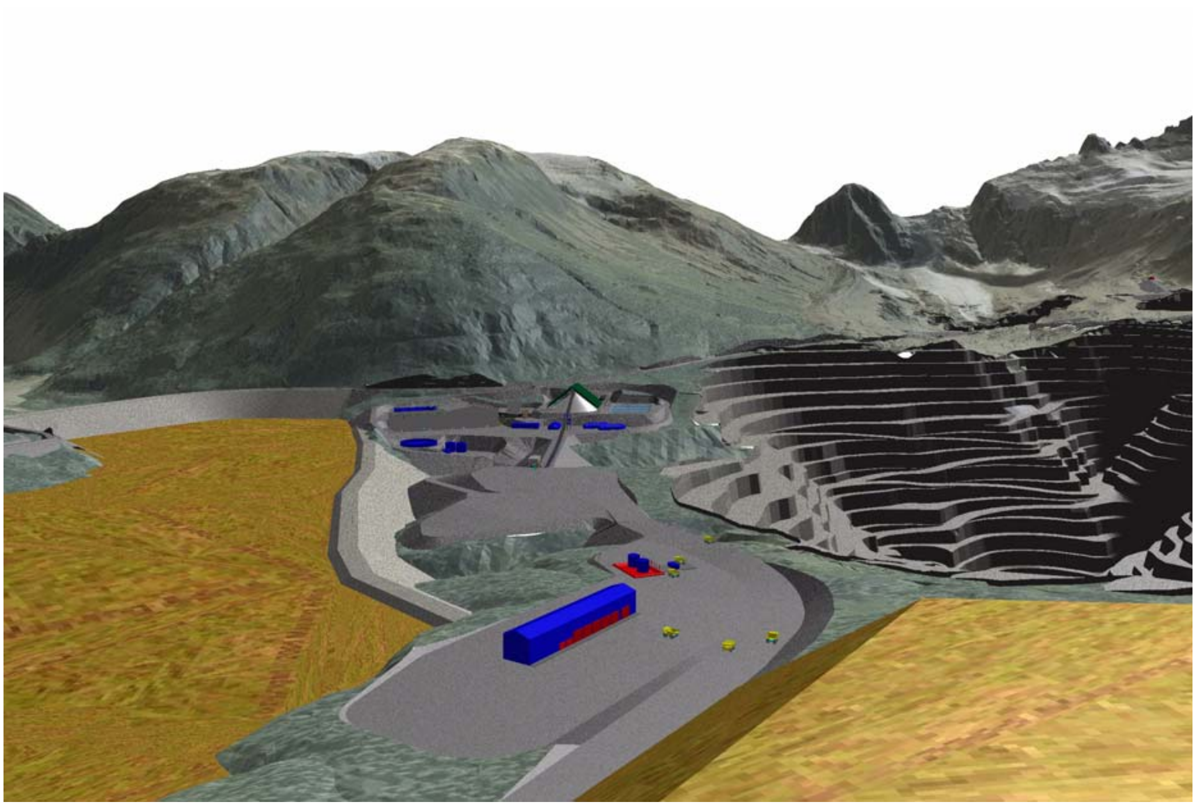
20.4.2.1 Truckshop

The truck shop will be a large complex which serves multiple purposes. In addition to providing an area for maintenance bays, tire shops and a wash bay (all of these on the south end of the building), the truck shop complex will also house the following: welding bay, electrical shop, ambulance, first aid room, first aid office, machine shop area, mine dry, warehouse, offices for mining and engineering staff, lunch room and the foreman's office.

There will be four truck maintenance bays, which will be suitable for the 345 t haul trucks. The workshop/warehouse/office complex will be located on the north end of the building, together with two small truck service bays, a welding bay, electrical shop, machine shop, ambulance and first aid room

Figure 20-23 shows a 3-D picture looking south towards the truckshop and main mine fuelling station in the foreground and concentrator in the background.

Figure 20-23: Truckshop



20.4.3 Fuel and Lubricant Storage and Distribution

Diesel fuel will be delivered to the site via a 135 km, 3.5″ pipeline and will follow the same route as the concentrate pipeline alongside the access road to the east side of the concentrator. Prior to reaching the concentrator, the diesel pipeline will be branched to supply the emergency generator fuel storage tank located adjacent to the electrical substation. The diesel pipeline is also branched on the north side of the concentrator, next to the coarse ore transfer conveyor, to feed the surface vehicle diesel storage tank. This storage tank will also provide fuel to the explosives tanker which transports fuel to the explosives factory. The main diesel pipeline is then supported across the coarse ore transfer conveyor and then buried several hundred metres to storage tanks located on the same platform as the truck shop.

Due to the high probability of intermittent temporary road closure during the winter months, two 1,244 m^3 (330,000 gal) storage tanks are included in the cost estimate, each containing approximately 10 days of fuel requirements. The diesel fuel storage tanks will each be 12 m in diameter x 11 m high, installed within a lined containment area sized to contain 110% of the capacity of the storage tank. The two large storage tanks are pumped to a 50,000 litre fuel dispensing tank which supplies fuel to four fuel dispensing pumps for the haul trucks.

Lubricants will be delivered to the site in containers. The containers will be stored in two locations: next to the truck shop for general truck maintenance, and also next to the fuel storage tanks so that the haul truck drivers can add oil during refuelling.

20.4.4 Waste Water

There will be two domestic waste water treatment units on site. One unit will serve main industrial facilities and camp and the other will be dedicated to the truckshop and explosive facilities due to their remote locations.

20.4.5 Potable Water

The potable water demand for the industrial complex and camp was established based on camp capacity of 460 workers and average daily water consumption per person of 0.19 m^3 (50 US gallons). The number of workers will vary during the day and night shift. The peak flow will take place during the shift change and will have short duration. Peak flow was estimated at 72 m^3/hr based on 30 min duration and average daily potable water requirement at 3.6 m^3/h. The potable water tank 5 m diameter and 5.5 m high will provide one day storage capacity.

20.4.6 Fire Water

Water will be available to the firewater main from the fresh/fire water tank at a nominal flow rate of 376 m^3/h, based on combined requirements for the truck shop sprinkler system (two bays only) at approximately 261 m^3/h and one hydrant capacity at 115 m^3/h. The storage tank has been designed to have a firewater reserve that will supply two hours of firewater in the event of an emergency. The firewater reserve will not be accessible to the mill fresh water system. A two hour firewater reserve will be ensured by piping the firewater from the bottom of the storage tank, and the fresh/potable water systems from approximately 8.5 metres higher in the tank in order to guaranteed water availability. Level and temperature probes will be connected to the fire alarm panel to monitor the conditions of the water in the fresh/firewater tank.

20.4.7 Site Storage Facilities & Shop Warehouse

A 48 m long x 22 m wide, heated, pre-engineered steel building adjacent to the concentrator will house maintenance shops, offices, storage and working space required for maintaining the equipment in the concentrator.

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.

20.4.8 Administration Building

A 29 m long x 17 m wide, single level pre-fabricated building will house the main administrative functions for the operation. It is located adjacent to the permanent camp next to the concentrator. Fifteen offices and nine open plan work areas are provided for senior management and administration. There is also a small lunch room, mud and storage room, meeting rooms and an electrical/mechanical room.

20.4.9 Assay Laboratory

The assay laboratory will be formed from two ATCO type structures side by side with an overall dimension of 17 m long x 7.2 m wide and will be located adjacent to the Administration building. The assay lab will provide equipment for crushing, pulverizing for sample preparation. The laboratory is designed to handle 200 samples per day.

20.4.10 Permanent Camp

The permanent camp facility will be used during construction and converted and put in like new condition for operations. The permanent camp consists of:

- 450 single occupancy rooms for hourly paid workers with ganged amenities constructed from transportable pre-fabricated modules stacked to reduce footprint area,
- 120 single occupancy rooms for salaried staff and visitors with amenities shared per pair of rooms constructed from transportable pre-fabricated modules stacked to reduce footprint area,
- Kitchen and dining facilities sized for the larger construction force
- Rest and recreation facilities.
- Reception area and administrative offices

With due economy in mind, manufacturers of modular camp facilities were requested to provide budget level quotations reflecting their most economical designs and configurations. This included provision for double room occupancy for the hourly paid workers during construction. British Columbia Regulation 427/83 "*Industrial Camps Health Regulation*" is the governing document.

All manufacturers recommended that three level sleeping quarters and single level common areas provide the most economical mix. The proviso is that three level sleeping quarters require a well compacted base to support foundations. Their consensus opinion is that dictated, customized designs add significant cost with questionable benefit to tried and proven standards by interfering with their established and cost effective design, manufacture and construction processes.

The camp is scheduled for construction immediately following tunnel break-through. This enables the earliest start to the main construction effort in Galore Creek Valley.

20.4.11 Communication System

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

The fibre optic cable between the plant's buildings will be strung along the electrical transmission lines. The fibre optic cable is primarily used by the control system to link its components. Thus, the fibre optic cost is included with the instrumentation and control system estimate.

In the plant's buildings, one hundred phones will be available for internal site communications. Phone communication will be carried over the Ethernet network (voice-over-IP technology). Phones will be distributed over the site. Both camps will offer a phone, an ethernet plug, and satellite TV in each room.

A satellite dish will be installed on-site to provide external voice and data links. A maximum of 50 simultaneous phone communications will be possible.

Data links for computers allow an upload rate (from site to the satellite) of 768 kbps and a download rate (from satellite to site) of 1,000 kbps. This bandwidth should provide a reasonable response time for approximately fifteen computers to be simultaneously connected to the Internet (e-mail and web browsing). This bandwidth cannot support video-conferencing.

20.4.12 Site Power Distribution and Motor Control

20.4.12.1 Power Distribution

20.4.12.1.1 Substation

The 138 kV transmission line from Meziadin will terminate at the Galore Creek Substation, approximately 500 km from the grid supply at Skeena Substation. The estimated total operating load is 85-100 MW. A single line diagram for the main substation was completed in connection with the Feasibility Study.

At the Galore Creek 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.

20.4.12.1.2 34.5 kV System

For a plant running load of 65MW (normal) to 87MW (peak), 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. The switchgear comprises a split bus with one incomer (from one step-down transformer) supplying each bus. During normal operating conditions the switchgear tie-breaker will be open with Bus A supplying power to the SAG mill area gearless drive (GMD). The GMD constitute approximately 26% of plant running load. Bus B supplies power to all remaining plant loads.

All feeder circuit breakers are 1200 A rated. There are also 600 A fused load break switches feeding the small excitation transformers for the GMD. These switches are a cost and space effective alternative to circuit breakers, providing both short circuit protection and a full load isolating means for the small transformers.

20.4.12.1.3 4.16 kV system

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. This transformer size provides approx. 1400 A current at 4.16 kV, suitable for standard switchgear and bus ratings of 1,600 A. The connecting loads are grouped according to area and load demand.

All site distribution transformers typically have a connected load ranging from 50 - 60% of naturally cooled capacity, and there is the option to increase each transformer rating by 33% with forced air-cooling.

20.4.12.1.4 600 V system

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. The fused disconnect switches (non-load break) are selected as a cost effective solution to achieve a point of isolation and short circuit protection for each transformer, while intelligent trip operation can be achieved either upstream or downstream by circuit breakers. In other words, this configuration allows for a generous level of segregation within the 600 V system while rationalizing the main 34.5 kV switchgear line-up. The use of several smaller transformers also allows for a less labour intensive cable installation for secondary side cables.

Each transformer supplies a 600 V 2000 A distribution board comprising 1200 A 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.

20.4.12.1.5 34.5 kV overhead lines

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, with each pit having a continuous transmission line along half its perimeter. This will allow for full pit coverage of connected mining equipment via trailing cables.

20.4.12.1.6 Standby Generation

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.

20.4.12.2 Ancillaries

20.4.12.2.1 Camp

The camp will be supplied from a dedicated 1.5/2 MVA transformer feeding a PDC. As mentioned previously, there will be standby power generation for the camp site (1 MW capacity), which will be connected to the camp PDC via a tie-breaker.

20.4.12.2.2 Other infrastructure and utilities

Smaller pad-mounted transformers (<1 MVA) will supply the ancillaries such as truck shop, fuel station, concrete batch plant, mechanical workshop, assay lab etc. Smaller MCCs and PDCs will distribute this power within each building.

20.4.12.2.3 Tunnel

One containerized switchgear set has been allocated for tunnel distribution. This switchgear will feed ventilation fans, lighting and other auxiliary tunnel services.

20.4.12.3 Electrical Equipment Layout

20.4.12.3.1 34.5 kV Switchroom

The main 34.5 kV switchgear line-up is to be located in a separate building adjacent to the main outdoor substation. Standby generation equipment connecting to the 34.5 kV system will be nearby.

20.4.12.3.2 Transformer bay

The main transformer bay will contain all transformers feeding Concentrator building equipment, and will be located between the 34.5 kV switchroom and the Concentrator building. The bay will consist of two back-to-back rows, with each row able to accommodate up to twelve transformers. All GMD and 600 V transformers will be allocated an optimal bay location to reduce outgoing cable length.

20.4.12.3.3 Concentrator Building Switchrooms

The Concentrator Power distribution is subdivided into the following areas:

- *Concentrator building*: entire building facility comprising grinding, flotation, concentrate dewatering and reagent handling equipment.
- *Grinding area*: area of concentrator building containing grinding equipment
- *Processing area*: area of concentrator building containing flotation, concentrate dewatering and reagent handling equipment

An annex to the grinding area will house the greater part of all remaining switchgear. The annex will contain all 4.16 kV distribution switchgear and 4.16 kV VSDs (including VSD transformers). It will also include 600 V switchgear and MCCs for grinding area equipment only, due to the close proximity to the grinding area. The GMD E-House will be located as close as practical to the mill drive.

A two level building within the processing area will comprise a switchroom on ground level and a control room facility on the upper level. This central and elevated location provides a useful vantage point to equipment located within the concentrator building. The ground level switchroom will contain 600 V switchgear and MCCs for the processing area equipment.

20.4.12.3.4 Site Switchrooms

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 and borehole dewatering, and a third type for borehole dewatering only. 'Fixed' equipment locations, including the crushing area and several remote pumping stations along the dam, will have power distributed by local containerized switchgear.

21. Interpretations and Conclusions

21.1 Project Economics

Section 21.1 was prepared by Mr. Bruce Rustad ,P.Eng. Hatch

Hatch can offer no comment as to future metal prices, exchange rates or costs, as these all depend on many factors, beyond the control of the project participants. These all have significant impact on the project economics. Nonetheless, subject to the relevant base case conditions, qualifications, assumptions and exclusions set out in this Report, the Galore Creek project is economically viable.

Under the “Base Case” metal price assumptions, the Galore Creek project achieves an after tax internal rate of return (IRR) of 10.6%. With the “Spot Case” scenario (Sept 1, 2006 prices), the project has IRR 30.7%, a three year Trailing Average scenario of 12.7% and a low case scenario of 9.5%.

The Study shows that the Galore Creek project has the potential to produce:

Average Annual Production -First Five Years: Recovered Metal 432 M lbs of copper, 341,000 ozs gold and 4.0 M ozs silver, at an estimated 5 year cash cost of US$0.38 per pound of copper inclusive of credits.

Total Production Life of Mine: Total Recovered Metal 5.8 B lbs of copper, 3.6 M ozs gold and 58.5 M ozs silver, at an estimated LOM cash cost of US$0.62 per pound of copper inclusive of credits.

Base Case Economic Summary:

- NPV After tax undiscounted 1,736 US$M
- NPV After tax 5% 599 US$M
- Payback 4.0 Years
- Pre tax IRR 14.1%
- After tax IRR 10.6%
- Life of Mine 22 years

This study considers the resources defined in the Central, Southwest, West Fork and Junction deposits and does not include any previously reported upside from any inferred resources specifically extensions of the Southwest and Central pits, the Bountiful zone or from Copper Canyon.

21.2 Geology

Section 21.2 was prepared by Mr. Michael J. Lechner, RPG from Resource Modeling Incorporated.

The Galore Creek property is characterized as a large copper-gold porphyry system consisting of a number of mineralized zones including the Central Replacement Zone, the Southwest Zone, the Junction Zone, Middle Creek, North and South Gold Lenses, Bountiful, and the West Fork Zone. In total, the property has been tested with 758 diamond drill holes totaling about 187,267 metres. The assay database for the property contains about 60,000 assay records. In addition to the diamond drilling data, a tremendous amount of data has been collected from the property since the early 1960’s. Some of this data includes soil, stream sediment, and rock geochemistry programs, helicopter airborne magnetic and radiometric surveys, ground based IP/resistivity surveys, and seismic refraction surveys.

Using past results and the diamond core data that were collected from the 2005 program, an updated resource model has been constructed. In the opinion of the author the underlying data are adequate to define the Mineral Resources that are the subject of this report.

21.3 Mining

Section 21.3 was prepared by Mr. Jim Gray,P.Eng. GR Technical Services.

The mine design, operations planning, and costing for the Galore Creek property is based on a large open pit, bulk mining operation typical in western North America and particularly typical to similar mines in British Columbia. Large equipment and high through puts have been used to achieve economies of scale. The equipment fleet and onboard technologies specified in the Feasibility Report have operating histories at other properties, proto-type equipment has not been used. In other words the mining equipment and operating practices used in this evaluation and report are tried and true in this general location. The capital costs for the mining equipment and for the mine area construction items have been verified by local suppliers of goods and services with experience in the area.

The technical and operating experience in the area is more than adequate to run the operations and the infrastructure and support services for mining equipment and other operations support are available in British Columbia. The productivities and operating costs used in the evaluation reflect the actual operating experience at similar operations in the western Canada. Allowance for local conditions especially weather and high precipitation as snow fall has been included in the estimate and are deemed to be representative of the operating costs that the operations will achieve. The weather effects and derates that are included are difficult to predict, especially visibility effects, and will be variable from year to year but the long term average effect included in the operating plans, is deemed to be suitable.

The high activity for mining operations and new projects in western Canada and around the world has created concern related to the availability of equipment and operating supplies particularly large shovels and truck, and truck tires. Negotiations have been started to reserve supply positions on long delivery items but these issues need to be tracked. Commitments will need to be made to ensure supply positions are maintained to meet the project schedule.

The availability of experienced technical and operating personnel is also a concern. It is assumed that sufficient experienced personnel will be hired to provide training for people new to the industry.

Based on these parameters, an economic mine plan for a 22 year mine life has been developed as a Feasibility Study for the Galore Creek property. It is the opinion of GR Technical Services Ltd. that the mining estimate from the Galore Creek October 17, 2006 Feasibility Study and described in this Technical Report is suitable for this Reserve estimate.

21.4 Metallurgy and Process

Section 21.4 was prepared by Mr. Hoe Teh ,P.Eng. Hatch

The Galore Creek deposit has variable mineralogy and rock types. It is the opinion of Hatch that the metallurgical samples tested are a reasonable representation of the mineralogy, metallurgical domains and rock types of the Galore Creek deposit.

The comprehensive metallurgical program completed for the study has satisfied the requirements for process and flowsheet design. The metallurgy, design flowsheet and reagent suite are consistent with current operations on other porphyry copper deposits.

The process flowsheet can recover copper at a marketable concentrate grade from the various metallurgical domains in the Galore Creek deposit. Gold is primarily associated with copper minerals and can be recovered with the copper by flotation.

Given the magnitude of the deposit, it is possible that certain ores mined during the life of the operation might exhibit metallurgical responses that have not been observed during this study which might impact metals recovery and grade. The plant operating experiences gained by then might help with the processing of such ores.

The Galore Creek plant has been designed based on proven technologies and equipment, including the large grinding mills and flotation cells, to minimize risks. Equipment sizing has been based on test data or experience with operating mines. Process and equipment have been scaled up based on generally accepted principles, experience and vendor recommendations.

Although the concentrates generated from the ore composites of various zones and pits are relatively clean, there appears to be significant variability in the levels of several elements that may impact the value of the concentrate. It is possible, that concentrates from certain pit areas and production periods may contain levels of minor elements that will incur penalties. This may be mitigated by ore or concentrate blending as experience is gained during operations.

21.5 Geotechnical

Section 21.5 was prepared by Mr. Iain Bruce,P.Eng., P.Geo, BGC Engineering Inc.

Mining waste, including tailings, waste rock and pre-stripped overburden from pits will be contained within the Galore Creek Valley, north of the proposed Central pit. A cross-valley tailings dam, located approximately mid-valley, has been designed to retain a minimum of 540 million tonnes of tailings. The tailings dam is designed as a rockfill structure with an impervious central core. This structure will be raised in a downstream direction from a starter height of 152 m to an ultimate height of 275 m from valley bottom. Tailings will be deposited in the northern portion of the impoundment and spigotted off the dam crest and/or nearby valley slopes.

Approximately 840 million tonnes of waste rock will be placed in lifts in the southern portion of the impoundment, adjacent to the tailings, but not co-mingled. Potentially acid-generating rock (PAG) will be placed in lifts at lower elevations. At the end of the mine life, the impoundment has been designed such that the PAG waste rock and tailings will be flooded in perpetuity.

Freshwater diversions have been designed for construction and operations to reduce the total catchment area of the impoundment to a manageable size. These diversions mainly consist of open channels and diversion dams to divert as much run-off as possible around the tailings and waste facility.

The proposed plant site is located near the confluence of the East and West forks of Galore Creek. Shallow foundations consisting of spread or mat footings can be used to create the bulk of the plantsite foundations.

Geotechnical investigations were undertaken by BGC within the Galore Creek Valley during the summer/fall of 2004 and 2005. These investigations included geotechnical drilling, surficial mapping, test pit excavations, seismic refraction and shear wave surveying, peizometer installation, permeability testing, and laboratory testing. A total of 56 geotechnical boreholes have been drilled in mid to upper Galore Valley. Thirteen test pits have been excavated in the vicinity of the plantsite and primary crusher areas. Thirteen kilometres of seismic refraction traverses have been undertaken throughout the property. From these investigations, the sub-surface soil and rock conditions, as well as groundwater conditions, have been characterized throughout the valley. The results of these investigations were used to design the tailings impoundment and plantsite foundations.

Additional geotechnical investigations for the tailings and waste facilities, plantsite, and open pit will be required for final design. In fact, the majority of these investigations have been completed by BGC during the summer/fall of 2006.

21.6 Project Risks

General – All

The Project is located in the Province of British Columbia, Canada. Accordingly, it is expected that there will be a relatively low degree of political, legal or regulatory risk (e.g., changes to laws or agreements reached with regulatory authorities, expropriation, changes in taxation or royalty regimes or non-issuance, cancellation or revocation of permits or licenses required to develop and operate the Project) associated with the Project. Nonetheless, an assessment of this risk is beyond the scope of expertise of the authors of this report; and, accordingly, no allowance for such risk has been included in the cost estimates or economic analysis set out in Sections 25.6, 25.7 and 25.8 of this report.

In addition, a project of this nature is also sensitive to several project risk factors that would be expected to potentially impact any major project of a similar scale and scope (e.g., adverse weather conditions, acts of god and force majeure events, delays due to unforeseen factors such as late delivery, or unavailability of equipment or materials or unavailability of labour resources, poor performance by EPCM contractors or construction contractors, disputes with local residents, etc.) All of these risks are, to a greater or lesser extent, outside of the Owner's control. Although operating and capital costs have been developed to reflect expected climatic and hazard conditions typical in this area based on field observations, abnormally adverse weather patterns could increase costs during any given period. No allowance for such risk factors has been included in the cost estimates or economic analysis set out in Sections 25.6, 25.7 and 25.8 of this report.

Project risk factors that have been specifically identified as potentially impacting this Project are detailed below.

Design Risks

The main areas of design risk for the project are as follows:

- The submitted environmental permit is currently under review this includes the June 21, 2006, NovaGold filed application with the British Columbia government to obtain a surface lease over the Grace property. NovaGold intends to build a tailings and waste rock storage facility over a portion of the Grace property to facilitate operations at Galore Creek.If any major issues arise from this process the anticipated start of construction date may be impacted. Additionally, any issues raised may impact on selected equipment, layout and project capital or operational costs.

- Open pit slope design – Bench face angles have been estimated for the sheet fractured rock and massive rock assuming good blasting practices. Over blasting could add waste to the optimal pit design.
- For detailed design, field checking of geohazard events, as well as, determination of the section of the central pit potentially impacted by periodic overflow of the Dendritic creek diversion will be required to minimize operational risks.
- Geotechnical – The availability of adequate bulk waste rock of appropriate size may change the allowable slope angles for the earth structures. This will be further investigated during the construction phase.
- Road limits on the Hwy 37 river crossings may result in a change in shipping plans during construction. This will be evaluated further during detailed engineering when a more comprehensive bridge and culvert survey is conducted as part of the logistics planning.
- Discharge from the tailings/waste dam has to meet an acceptable water quality. Although substantial residence time is available inside the containment area, water quality issues may arise that require additional capital to treat at source.

Construction Risks

There are a fairly large number of risks associated with the construction of this project, typical of a remote mining operation, which can impact capital cost and schedule. The main risks are as follows:

- Adverse weather. This probably constitutes the greatest area of risk, 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 hoarding.
  - 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 readily available of NPAG waste rock.

- Schedule delays may result from a slow tunnel progression. This has been mitigated by constructing from both ends, however, a delay may put pressure on the critical core till placement schedule at the tailings dam.
- Schedule delay – there are several factors including the waste and water management concept that can adversely effect the construction schedule:
  - The missing of the winter –time low flow window in Galore Creek for installation of the coffer dam could add up to a year to the construction schedule.
  - The placement of the till core in the tailings dam assumes summer season placement over two seasons. A major disruption in this activity could also add to the construction schedule.
  - If seepage under and around the dam is higher than expected there could be increased operating costs associated with seepage pump back.
  - This is a complex project that has several items, both on-site and off-site, on the critical path. The failure to complete any of these items in accordance with the project schedule will likely have a consequential impact on the remainder of the project, both with respect to schedule and cost.
- 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.
- The construction of the tunnel is on the critical path. The tunnel design has been based on subjective evaluation of rock quality, any items that are outside the anticipated could result in a schedule improvement or slippage. The construction of the tunnel could be impacted where:
  - extent of poor ground is different than interpreted.
  - groundwater inflows are different than interpreted.
  - quantity of grouting is different than anticipated which could result in costs and duration differences.
  - the quantity of PAG rock is different than expected which could require additonal temporary storage and re-handling through tunnel resulting in higher cost.
- Construction labour availability and cost – The current construction market in BC and Alberta has lead to significant shortage of skilled construction labour in BC. Much of this activity is focussed on the Oil Sands projects in Alberta and civil infrastructure projects in BC associated with the 2010 Olympics. These large users of resources and materials may impact availability and costs associated with construction services, construction equipment and materials.
- Construction material cost – Current prices for steel, mining and processing equipment are on an upward trend driven by supply and demand. Although the project budget has been updated to reflect the prevailing market conditions, these trends may continue.

- 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.

Operations Risks

The key risks for operations are largely associated with the consequence of adverse weather conditions.

- Open pit slope design – Dewatering is required to maximize the pit slope angles. Dewatering flow rates are predicted to be approximately 23,000 to 32,000 m^3/d from vertical wells and horizontal drains during the life of mine. Should the predicted flow rates vary operating costs for the dewatering could be affected.
- Dewatering - Careful implementation of the proposed dewatering plan will be required as residual pore pressures will result in a severe (approx 10 deg) reduction in achievable inter-ramp angles in the sheet fractured unit.
- Reduced mine production – The frequency and severity of winter storms could result in reduced production from the mine.
- Water management in the mine – Water Management may prove more difficult than anticipated, resulting in higher costs and loss of production time.
- Terrain 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.
- Seismic Activity - The Galore creek project is situated in a seismically active area. The tailings dam has been designed to withstand an earthquake characterized by a peak horizontal bedrock acceleration of 0.25g, from a magnitude 7.0 event (on a strike slip fault at an epicentre distance of 20 km from the project area). A seismic event of magnitude 7.0 or even less would cause damage of un-quantified impact to the other less critical structures and facilities.
- Impurities - The final concentrates had relatively low penalty elements. Fluorine, selenium, lead and zinc concentrations were variable and might be of concern.
- Supply - The current short supply with long delivery times for tires mean that supply contracts are required from tire manufactures before the end of 2006 to ensure that tires will be available in time for project start up. Tire supply remains a project risk.

22. Recommendations

22.1 Geology

Section 22.1 was prepared by Mr. Michael J. Lechner, RPG from Resource Modeling Incorporated.

Michael Lechner has reviewed the Galore Creek database and NovaGold's resource modeling methods. Based on these reviews the he has the following recommendations:

- Drill a reasonable number of twin holes to investigate the apparent bias between the pre-2003 and the NovaGold data. Approximately 56% of the diamond core data were derived from pre-NovaGold drilling and these data are approximately 3% and 14% lower than the NovaGold data for copper and gold, respectively. The author estimates that drilling ten twin holes totaling about 2,500 metres should cost approximately US$850,000.
- Monitor the performance of blanks and duplicates more closely. All sample batches associated with blanks that fail should be re-assayed. The costs associated with this activity are considered by the author to be a fixed cost to NovaGold as all of the monitoring work would be routinely done in house.
- Lower the copper equivalent (CuEq) cutoff grade used to design grade envelopes that are used to constrain the estimate of block grades. The resource model subject to this report used a 0.35% CuEq cutoff grade, yet the breakeven cutoff grade used to summarize Mineral Resources was 0.25% CuEq. The costs associated with this activity are considered to be a fixed cost to NovaGold as all work would be done in house.
- Obtain more moisture content data from representative rock types. The author estimates that obtaining a reasonable number of moisture determinations would cost several thousand to ten thousand dollars.

22.2 Mining

Section 22.2 was prepared by Mr. Jim Gray,P.Eng. GR Technical Services.

Based on GR Tech's work on the Feasibility Study for the Galore Creek Property the following recommendations are made to advance the project into detailed design and operations planning:

- Ongoing weather data should continue to be collected to verify the storm and snow operating contingencies and allowance used in this estimate. Fog affects and visibility issues are the hardest to predict and local data is needed. More information will be useful to adapt plans for short term effects and perhaps reduce some of the contingencies included in this plan. The costs associated with this activity are considered by the author to be a fixed cost to NovaGold as all of the monitoring work would be routinely done in house.
- The pre-production mining is based on meeting the construction requirement for the tailings dam, diversion ditches, and initial fill sites. The initial excavation in the mine is in glacial tills and the 'broken rock' zone. The glacial tills are very variable in nature and particularly difficult to predict. The long term average material characteristics will be suitable but regional variations need to be forecast so the construction material requirements can be planned on a daily and weekly basis

during the construction period. This will require a high degree of field testing prior to excavation on a short term planning basis. Since much of this material is not going to be blasted, typical open pit methods using blast holes will not be applicable and a test hole and/or test pit program should be initiated.

- A fleet management system and a high degree of GPS location/navigation capabilities has been included in the estimate but a detailed specification has not been developed. This technology is currently changing quickly with increased utility and reducing costs. The technology for site wireless communications in particular is undergoing significant changes. A dialogue with suitable technology providers should be initiated and maintained to develop an eventual specification document for eventual commercial negotiations. The specified system needs to be integrated into other management systems, reporting systems and general enterprise systems for the minesite.
- A conservative approach has been taken with respect to ARD issues and disposal of PAG rock from the mine. The mine plan is based on submersion of PAG rock in the tailings pond, within 5 years. This adds significant cost to the mine plan. More sample data and evaluation may reduce the amount of material designated as PAG and a mine plan that doesn't have to place the material for submersion within 5 years may be able to reduce the mining costs.
- Avalanche control and geo-hazard avoidance has been included in the operation planning and costing. Consideration of avalanche control structure can be considered as well to reduce some of the contingencies included in the plan for down time and avoidance.
- Backfilling of mined out pits has not been optimized in the mine plan. Backfill planning requires iterations of the mine plan and schedule to determine when mining areas are completed and available for backfilling. The feasibility planning is based on a positive placement of all material in defined waste dump areas and a subsequent reclamation plan is developed and costed. Future planning should consider backfilling to reduce operating costs as well as reclamation costs. It may also reduce the area of land disturbance of the project.
- Commercial commitments need to be made to ensure supply of the long delivery items, including the shovels and trucks. A supply of truck tires also needs to be secured.

22.3 Metallurgy and Process

Section 22.3 was prepared by Mr. Hoe Teh ,P.Eng. Hatch

It is the opinion of Hatch that the metallurgical samples tested are a reasonable representation of the mineralogy, metallurgical domains and rock types of the Galore Creek deposit. The comprehensive metallurgical program completed for the study has satisfied the requirements for process and flowsheet design. As a result no additional recommendations are considered necessary to advance the project. It is believed that the metallurgical flowsheet can proceed to the next phase of Engineering.

22.4 Geotechnical

Section 22.4 was prepared by Mr. Iain Bruce,P.Eng., P.Geo, BGC Engineering Inc.

It is the opinion of BGC that sufficient geotechnical investigation has been performed to advance the project to the next phase of detailed engineering. During construction, monitoring of geotechnical activities (e.g. earthworks placement) will be required by qualified engineers and or technicians. To ensure that specified engineering standards are maintained, ongoing analysis of materials of construction will be performed as a standard procedure and quality control measure. These costs have been included in the overall capital cost estimate.

22.5 Civil Construction

Section 22.5 was prepared by Mr. Kelly Boychuk,P.Eng, Ledcor CMI Ltd.

Mine Site Dams and Diversion Channels

It is the opinion of Ledcor CMI that the various dam and diversion channel structure design information from Bruce Geotechnical was sufficient to estimate quantities and understand the concept of the design. This information has allowed Ledcore CMI to provide a construction cost estimate to the required level of accuracy.

Mine Site Access Road

It is the opinion of Ledcor CMI that the survey information and geological interpretations provided by McElhanney were sufficient to carry out the constructability estimate and understand the concept of the design. The proposed road alignment appears to best utilize the natural terrain and local borrow materials and optimize cuts and fills. This information has allowed Ledcore CMI to provide a construction cost estimate to the required level of accuracy.

22.6 Pipeline Systems

Section 22.6 was prepared by Mr. Don Hallbom,P.Eng., PE, Pipeline Systems Incorporated (PSI)

It is the opinion of PSI that the Galore Creek copper concentrate pipeline is considered to be technically feasible. Existing mineral slurry pipelines have operated successfully and have demonstrated that, with qualified personnel and adherence to operating procedures and maintenance programs, high reliabilities can be achieved in comparison with other transportation methods such as railway or trucking. The selected systems are adequately conservative, such that they should be able to withstand normal design changes as the project advances.

Rheology and corrosion testwork were performed on concentrate samples provided from pilot plant runs of drill core and provided sufficient data to design the pumping and piping system. The corrosion testwork revealed that the concentrate had a pH of 10 but had corrosive properties. As a result a HDPE liner was added to the capital cost estimate.

22.7 Road Design

Section 22.7 was prepared by Mr. Robert W. Parolin,P.Eng., McElhanney Consulting Services Ltd.

After two years of field reconnaissance, survey and design we are confident that the best route has been selected that meets the road design criteria and minimizes the environmental impact. The level 3 survey is adequate to complete the class IV road design and generate earthwork quantities which are necessary to estimate both the Capital and Operating costs. This information meets the objectives of preparing a "Feasibility Study".

22.8 Tunnel Design

Section 22.8 was prepared by Mr. Dean Brox, P.Eng, Hatch Mott MacDonald

A comprehensive evaluation of the geology of the project site and in particular the tunnel alignment has been undertaken by reviewing existing information from past exploration and detailed research activities/information provided by Dr. Jim Logan of the BC Geological Survey who has undertaken field work in the project area. In addition, detailed surface mapping along the tunnel alignment was carried out to verify work by Logan and has allowed for an interpretative profile of the tunnel geology given the fairly simple volcanic geology present in the project area. From the evaluation of all existing information and the detailed field mapping, there are no indications or plausible expectations of any adverse geological conditions to be present along the tunnel alignment.

Many rock tunnels have been constructed around the world and the extent and scope of pre-construction investigations for most tunnel projects typically varies based on the complexity of the geological environments.

The tunnel portals have been sited in terms of constructability issues regarding road access grade and to facilitate early tunnel excavation without having to complete extensive excavation of glacial overburden along the slopes where the portals are located and the selected portal locations have therefore defined the overall tunnel alignment.

The groundwater inflows into the tunnel have been calculated based on a theoretical analytical approach by Heuer (1985, 2005) assuming typical rock mass permeability along the tunnel alignment that vary with overburden as testing data from past projects suggests. This approach has been verified to be in good agreement with observed findings from constructed tunnels.

Provisions have been made in the contract documents that require probe drilling ahead of the advancing tunnel faces and equipment and materials for pre-excavation grouting in the event that unmanageable groundwater inflows may be encountered.

It is the opinion of HMM that sufficient investigation has been performed to advance the project to the next phase of detailed engineering.

22.9 Power Transmission Design

Section 22.9 was prepared by Mr. Allan Guy,P.Eng., Ian Hayward International Ltd. (IHI)

It is the opinion of IHI that the transmission line is technically feasible. Hazard protection is a concern on this project and will result in a cost premium, principally from either special steel structures or the inclusion of an underground cable section. The capital cost estimate has included 8.6 km of protection via underground cable.

22.10 Opportunities

Section 22.10 was prepared by Mr. Bruce Rustad, P.Eng., Hatch

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

Increased Production Rate

Increasing the rate of metal production could possibly be achieved by:

- NovaGold has continued with ongoing exploration of the property and currently there are targeted areas not yet drilled off to level of measured and indicated resource that is required for incorporation into a feasibility. In addition to extending the mine life, the project could be improved if higher grade ore could be mined in operating year seven, when grades in the present mine schedule start to decline.
- The mill circuit is capable of processing 71,500 tpd when the ore hardness is of 16.5 kWh/t or less. The feasibility study considers this rate only during the first five years of operation where as the ore hardness is expected to be at these levels for at least half of the mine life.

Reduction in Mining Costs

- The mine plan will utilize a number of pit stages that will eventually be developed by a series of push backs to the final pit wall. The pit staging will provide an opportunity to test / verify the pit slope design by making use of the interior (temporary) pit stages.
- Waste hauls can be significantly reduced early in the production schedule if the time required for PAG material submersion in water is increased from the current assumption of five years, with a significant improvement in cash flow and economics.
- Backfill opportunities have been ignored in this study to avoid placing waste on areas where ongoing exploration may expand the future pit limit. Backfilling will significantly reduce haul costs and should be investigated further during detailed design.
- A smaller North Central Pit starter may avoid low grade areas and improve mill feed grade in the start-up period. Alternative pits in the North Central Pit should be evaluated.
- Input costs (consumables, fuel, etc...) during the operating period may be significantly lower. The current commodity price environment is incorporated into long term operating cost projections. If commodity prices return to long term average levels, operating costs will be lower.

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$23 M to US$30 M. This system could feed power back into the BCH grid at the cessation of mining operations.

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.

23. References

23.1 List of References

A significant body of work is referenced in and forms the basis for the study. Key external and internal reports are listed as follows:

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2. “Galore Creek Project Power Supply Feasibility Study” August 2006, Ian Hayward International Ltd.
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5. “Galore Creek Concentrate Pipeline,” Final Report, July 2006, PSI.
6. “The Metallurgical Response of Galore Creek Ores,” Vol. 1 & 2, Project KM1547, G&T Metallurgical Services Ltd., 4 April 2005.
7. “Lead in the Galore Creek Copper Concentrates”, Project KM1641, G&T Metallurgical Services Ltd., 8 April 2005.
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11. “Galore Creek PFS, Geohazard Assessment”, BGC, February 2006.
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14. “Feasibility Geotechnical Report – Plant Site Design, BGC, September 2006.
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23. “Verification Ore Characterization Metallurgical Results – KM1659”, G&T

24. "Galore Creek – Pre-Feasibility Study – Power Supply", IHI, May 2005
25. "Assessment of Barge Transport on Stikine River – Final Report", Sandwell, November 2004
26. "Preliminary Grinding Circuit Design for the NovaGold – Galore Creek Mine", MinnovEX, February 2005
27. "Geohazard Assessment – Draft", BGC, May 15, 2005
28. "Open Pit Design – Draft", BGC, February 25, 2005
29. "Waste and Water Management – Draft - Pre-Feasibility Geotechnical Report", BGC February 9, 2005
30. "Aerodromes Concept", LPS Aviation, September 15, 2005
31. "Marketing & Commercial Input into a Scoping Study for the Galore Creek Project", NSA, April 2005
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34. MSRM. 2000. Cassiar Iskut – Stikine Land and Resource Management Plan. Prepared by the British Columbia Ministry of Sustainable Resource Management, Skeena Region. Online resource at: http://srmwww.gov.bc.ca/skw/lrmp/cassiar/approved_lrmp/plan/1.htm.
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43. "SMC Test Report on Fourteen Samples", February, 2006, JKTech Pty Ltd.
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45. "Galore Creek Mine Project – Mine Access Road and Site Facilities", May 2006. Ledcor CMI Ltd.
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23.2 Abbreviations, Acronyms and Units of Measure

23.2.1 Units of Measure

Table 23-1: Units of Measurement

Unit	Abbreviation
American Dollar	US$
Canadian Dollar	CA$
Centigrade	°C
centimetre	cm
cubic metre	m^3
day	d
dry metric tonne	dmt
foot/feet	ft
gram	g
gram/litre	g/L
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
litre 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
millimetre	mm
million	M
million tonnes	Mt
part per million	ppm
percent	%
second	s
square metres	m^2
troy ounce	oz
US gallon per minute	usgpm
wet metric tonne	wmt
Work Index	WI
year	yr

23.2.2 Acronyms and Abbreviations

Table 23-2: Acronyms and Abbreviations

Abbreviation	Acronyms	Abbreviation	Acronyms
ADFG	Alaska Department of Fish and Game	ME	Measured, Indicated & Inferred Resources
ADIS	Automated Digital Imaging System	MinnovEX	MinnovEX Technologies
Ag	Silver	ML/ARD	Metal Leaching/Acid Rock Drainage
ASC	Aluminum Standard	NBCC	National Building Code of Canada
Au	Gold	NMFS	US National Marine Fisheries Service
BCH	British Columbia Hydro Corporation	NPAG	Non-potentially Acid Generating
BCTC	British Columbia Transmission Corporation	NPC	Net Present Cost
BFA	Bench Face Angle	NPV	Net Present Value
BGC	Bruce Geotechnical Consultants	NSA	Neil Seldon & Associates
CCA	Capital Cost Allowances	NSP	Net Smelter Metal Prices
CDE	Canadian Development Expenses	NSR	Net Smelter Return
CEA	Cumulative Expenditure Account	PAG	Potentially Acid Generating
CEE	Canadian Exploration Expenses	PEA	Preliminary Economic Assessment
Cu	Copper	PFS	Pre-Feasibility Study
DFO	Department of Fisheries and Oceans	PGA	Peak Ground Acceleration
EIA	Environmental Impact Assessment	PMF	Probable Maximum Flood
EMPs	Environmental Management Plans	PSA	Pit Slope Angle
G&T	G&T Metallurgical	PSC	Pacific Salmon Commission
Giroux	Giroux Consultants Ltd.	PSI	Pipeline Systems Incorporated
GRTech	GR Technical Services Ltd.	RIC	British Columbia Provincial Resource Inventory Committee
HMM	Hatch Mott MacDonald	ROM	Run of Mine
IHI	Ian Hayward & Associates	RTEC	Rescan Tahltan Environmental Consultants
IPP	Independent Power Producer	SABC	SAG, Ball Mill and Pebble Crushing circuit
IRA	Inter-Ramp Angle	SAG	Semi Autogenous Grinding
IRR	Internal Rate of Return	Sandwell	Sandwell Engineering Inc.
ISO	International Organization of Standardization	SCADA	Supervisory Control and Data Acquisition System
ITC	Investment Tax Credit	SPI	SAG Power Index
LCT	Large Corporation Tax	SPT	Standard Penetration Testing
LG	Lerchs-Grossman	TNDC	Tahltan Nation Development Corporation
LOM	Life of Mine	UCS	Uniaxial Compressive Strength
LPS	LPS Aviation	VECs	Valued Ecosystem Components
LRMP	Cassiar Iskut-Stikine Land and Resource Management Plan	vpd	Vehicles per day
MCE	Maximum Credible Earthquake	vph	Vehicles per hour
MDE	Maximum Design Earthquake		

24. Not Used

Refer to Section 26 for Date and Signature Section.

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

25.1 Mining Operations

Section 25.1 was prepared by Mr. Jim Gray,P.Eng. GR Technical Services.

25.1.1 Introduction

A Preliminary Economic Assessment (PEA) study on Galore Creek has been completed on October 21, 2005. The PEA used resource classes less than Measured and Indicated; however the subsequent exploration drilling has upgraded much of the PEA pit delineated resources from the Inferred class to Measured and Indicated. This 2006 feasibility study uses Measured and Indicated ore classes only.

The entire mine planning for the Galore Creek mineral property is based on work done with MineSight® a suite of software well proven in the Industry. This includes the resource model, pit optimization (Minesight Economic Planner, MSEP), detailed pit design, and optimized production scheduling (Minesight Strategic Planner, MSSP).

In addition to the geological information used for the block model, other data used for the mine planning includes the economic parameters, mining cost data derived from supplier estimates and historical data, geotechnical slope design parameters, metallurgical recoveries, and project design plant costs and through put 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 precipitation, including snow accumulations, and the mountainous terrain at the mine site will require proactive snow control and extra road maintenance/traction control for roads and mining benches. Stream diversions or water handling facilities and structures are also included in the mine and infrastructure designs.

25.1.2 Mining Datum

The historical drill hole information and topography are based on various surveys with different sets of control. Effort has been made to ensure the Mine design is using the most up to date topography, in conjunction with the Infrastructure planning and that the drill hole data base is congruent with the topography surface.

Historical geology and topographic data for Galore Creek has used various survey grids which have been converted to a single basis. Geology data and Mine design in this study are based on an earlier simplified grid which is similar to the coordinate system for the larger mine site area. Of particular importance, topography and drill hole collars are now consistent.

NovaGold has chosen to use the Eagle Mapping 2004 survey (revised 15 Feb 2005) as the basis of the 2006 Galore Creek Feasibility Study. This coordinate system is called “Galore Creek 2006 Mine Grid” (GC06MG). GC06MG uses the same coordinate truncation as the PEA:

- Northing: - 300,000
- Easting: - 6,300,000

BGC & Hatch are using the un-truncated Eagle Mapping (Feb05) grid for other sections of this Feasibility Study.

An additional survey control conducted by Peter Thompson BCLS CLS in October 2005 showed that the control coordinates used by NovaGold staff during the 2005 survey are correct and that the survey methods employed by NovaGold staff are valid.

GR Technical Services (GR Tech) created both the resource model topography surface and a construction topography surface based on GC06MG from original (Eagle Mapping Feb05) digital elevation model (DEM) data set. The surface triangulation methodology is described in the mining technical report. A chronology of topography adjustments between 2003 and 2005 is described in the Feasibility Study.

25.1.3 Project Production Rate Consideration

A number of factors are considered in establishing an appropriate mining and processing rate, the key ones are discussed below in relation 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. Mine life is set at 12.5 to 20 years; as for anything beyond this, time value discounting shows insignificant contribution to Net Present Value (NPV) of the project, and capital investment typically is targeted at projects with payback of 3 to 5 years.
- Unit Capacity: Generally, unit operating costs are lower using the largest possible equipment for a single train. In the case of the proposed SAG mills, at least two 40 foot (12.2 m) 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 Economic 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.
- Higher production rates generally pay back sunk capital at a faster rate, thereby improving project NPV.

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 higher throughput scenarios and after testing different throughputs, the PEA study was subsequently based on a mill throughput of 65,000 tpd ore. Evaluation of grind size and metallurgical recovery, and an analysis of the effect of throughput on project payback undertaken in the feasibility study showed an optimum economic production rate of 71,500 tpd for the first five years of mill feed, and 65,000 tpd for the remaining life of mine (LOM).

25.1.4 Mine Production Schedule

25.1.4.1 Mine Load and Haul Fleet Selection

Results from a fleet selection study showed that the lowest cost/tonne fleet of cable shovels and haul trucks is the P&H4100XPB or equivalent cable shovel matched with CAT 797B or equivalent haulers. This is also the loader/truck combination that provides the lowest cost/tonne method for moving material at Galore Creek. This load and haul combination is used for production scheduling in the Central Pit.

A hydraulic shovel is required for operation in satellite pits at Galore Creek. The study also showed that the lowest cost/tonne fleet of hydraulic shovels and CAT 797 or equivalent trucks is the Terex O&K RH400 or equivalent hydraulic shovel.

A front wheel loader is required for stockpile reclaim and for making up shortfalls from the electric and hydraulic loaders. The lowest cost/tonne fleet of wheel loaders and CAT 797 or equivalent trucks is the Letourneau L-2350 or equivalent wheel loader.

25.1.4.2 Extreme Weather Effects on Production

Severe snow storms will impair the traction of the trucks on the haul roads which will negatively affect production. The loss of production will manifest itself by shovels stopping operation when the truck cannot return to the shovel sites, the number of shovels out of operation depending on the duration of the severe storm. In extreme cases, all three shovels may shut down. It will take time to return the roads to passable conditions so that in addition to the actual storm duration, there is a recovery period.

A simulation study has been carried out that used climate and weather data together with a loss of production and recovery model to estimate the expected number of hours of low production during any one year.

The snow study simulated storm conditions and their affect on production loss and recovery. The storm and recovery times have been simulated using random storm durations to determine the average time spent in each storm condition.

The results of the above simulation are used in the Galore Creek mine schedule as follows:

Total lost days from storms = 4

The truck and shovel operating efficiencies are also de-rated

Weighted Annual Shovel De-rate = 3.0%

Weighted Annual Truck De-rate = 1.8%

25.1.4.3 LOM Production Schedule

The mine production schedule after pre-stripping was developed with MineSight Strategic Planner (MSSP), a comprehensive long range scheduling tool for open pit mines. It is typically used to produce a life-of-mine schedule that will maximize the Net Present Value of a property subject to user specified conditions and constraints. Annual production requirements, mine operating considerations, product prices, recoveries, destination capacities, equipment performance and operating costs are used to determine the optimal production schedule.

Full production mill feed is expected to commence in January 2011. The production schedule uses 'Year 1' as 2011, 'Year 2' as 2012 etc.

25.1.4.3.1 Pre Production Mining

Pre-production mining is modeled by matching the Tailings Dam material requirements with material available in the starter pits on a monthly basis. All of the tailings dam rock and core is planned from within the starter pit perimeters. This has a positive contribution to the project value by avoiding the high incremental cost of mining the required rock from borrow sources.

Preliminary site investigations and borehole data indicate that material for construction of the tailings starter dam will be available from pre-stripping the Central Pit areas. Geotechnical exploration programs within the pit limits commencing in the summer of 2006 will further delineate suitable material sources for the starter dam core, dam ballasts, road construction, coffer dam, and aqueduct. Borrow pits outside the mine pit areas will be alternate sources if the pit supply is insufficient or untimely. The initial starter dam platform will be constructed with materials from a nearby borrow pit.

For the purpose of the Feasibility Study, assumptions have been made for the availability of suitable construction materials, specifically overburden tills. These assumptions are summarized in the table below, and will be re-evaluated when results of the geotechnical exploration program are studied.

Table 25-1: Assumptions on the Availability of Suitable Construction Materials

Till Material Type	C646 Pit	C616 & C636 Pits	C626 Pit
Suitable Clay Tills for Dam Core (>30% fines, <3% moist.)	5%	40%	5%
Suitable for Dam Ballast, Road Construction (<20% fines)	50%	50%	50%
Suitable for Road Construction (20 -30% fines)	40%	10%	40%
Non Suitable Tills (>30% fines, >3% moisture)	5%		5%

The starter dam construction has been sequenced by month using waste reserves inside the Central Starter Pits, and the constrains of the available mining equipment.

A conceptual study showed that there is no significant advantage to using conveyors with a mobile stacker instead of a truck fleet to haul the tailings dam construction material to the dam area.

25.1.4.3.2 Waste and Ore Schedule

Full production mill feed is expected to commence in January 2011 . The production schedule uses ‘Year 1’ as 2011, ‘Year 2’ as 2012 etc. The logic used in MSSP is described in the Feasibility Study.

Schedule Criteria

Schedule assumptions for mining equipment are listed in the design basis in the Feasibility Study. The Galore Creek schedule setup included:

Table 25-2: Material Types Defined For MSSP

NSR Grade Bin	Reserve Class
Sub (CA$3.82/t – CA$4.72/t)	Sub Grade- direct to crusher only *
Low (CA$4.72/t – CA$5.58/t)	Low Grade
Mid (CA$5.58/t – CA$7.79/t)	Mid Grade
Hi (CA$7.79/t – CA$10.0/t)	High Grade Bin
Hi1 (CA$10.0/t – CA$12.5/t)	High Grade Bin 1
Hi2 (CA$12.5/t – CA$15.0/t)	High Grade Bin 2
Hi3 (>CA$15.0/t)	High Grade Bin 3

* not reclaimed from stockpile due to recovery loss and re-handle cost.

Table 25-3: Destination Definitions Are Definitions for MS-SP:

Destination ID	Destination
Waste1-1	Waste Tills
Waste1-2	Waste Tills to tailings dam
Waste2-1	Broken NPAG
Waste2-2	Broken NPAG to tailings dam
Waste2-3	Broken NPAG to Aqueduct
Waste3-1	Broken PAG
Waste3-2	Broken PAG to tailings dam
Waste4-1	Stick NPAG
Waste4-2	Stick NPAG to tailings dam
Waste5-1	Stick PAG
Waste5-2	Stick PAG to tailings dam
Waste6-1	Undefined Waste
Waste6-2	Undefined Waste to tailings dam

- A secondary stockpile is used to allow separate calculation of the loading and hauling hours used in retrieving 580 ktpa of ore during mine shut downs.
- The primary program objective in each period is to maximize NPV.
- There are 356 operating days scheduled and 21 hours per day.
- South West and Junction pits are limited to using the RH400 shovel, and daily mining rates are restricted in these phases to prevent exceeding a feasible vertical mining rate.
- Dump rates by period have been set for the tailings dam, aqueduct and waste divider. Waste destination precedence is set so that MSSP does not choose the shortest haul, ensuring that the volume requirements by period for Tailings Dam, Aqueduct and Waste Divider are met.
- Haul and Return Times are estimated from all phases to all destinations using simulations from CAT's FPC program. For all benches in all pits, the haul times, return times and fuel burn are linearly interpolated.
- The optimum production schedule is obtained by maximizing the use of the mining fleet after the peak equipment requirement is reached, whilst minimizing stockpile reclaim.

Cut off Grade Optimization

The mill feed grade can be increased by sending low and mid grade classes to stockpiles whilst simultaneously preventing stockpile reclaim. The mill feed rate is maximized and this effectively increases the revenue per tonne milled of ore. Stockpiling ore also results in increased total mined rock and the mine cost per tonne milled ore also increases. At some point the cost of mining more material will exceed the incremental revenue from the higher grade ore milled.

Cutoff grade optimization analysis indicates that:

- Stockpiling to create an effective ore/waste cut off is CA$12.5/t NSR in year 1 – 5 provides optimum cut off grade optimization and stockpile utilization.
- Subgrade ore should be waste in years 1 – 7.

Schedule Results

The summarized production schedule results are shown in the table below.

Table 25-4: Summary of Production Schedule

		Year-3	Year-2	Year-1	Year1	Year2	Year3	Year4	Year5	LOM
ORE Mined										
Ore mined to crusher	kt	-	-	-	26,087	26,061	26,061	26,062	26,062	372,912
Cu	%	-	-	-	0.956	0.897	0.709	0.754	0.781	0.684
Au	g/t	-	-	-	0.400	0.722	0.516	0.687	0.415	0.369
Ag	g/t	-	-	-	7.949	6.808	6.189	6.263	6.687	6.189
ROM Mill Feed mined to stockpiles	kt	-	-	-	10,507	17,708	16,114	26,570	11,731	149,104
Total ORE Mined	kt	-	-	-	36,595	43,769	42,175	52,632	37,792	522,016
ROM reclaim from stockpiles	kt	-	-	-	-	26	26	26	26	149,104
Cu	%	-	-	-	-	0.330	0.314	0.318	0.322	0.283
Au	g/t	-	-	-	-	0.221	0.208	0.195	0.189	0.162
Ag	g/t	-	-	-	-	3.678	3.638	3.746	3.692	3.490
Total Stockpile Inventory	kt	-	-	-	10,507	28,189	44,277	70,821	82,526	-
Total ROM Mill Feed to Mill	kt	-	-	-	26,087	26,087	26,087	26,088	26,088	522,016
Cu	%	-	-	-	0.956	0.897	0.709	0.754	0.781	0.569
Au	g/t	-	-	-	0.400	0.722	0.515	0.687	0.414	0.310
Ag	g/t	-	-	-	7.949	6.805	6.186	6.260	6.684	5.418
Waste										
Mined Sub Grade to Waste	kt	-	-	-	1,801	2,702	4,182	3,807	976	18,715
Waste Mined	kt	2,370	44,682	59,187	50,530	48,038	61,702	47,621	62,291	838,906
Total Waste Mined	kt	2,370	44,682	59,187	52,331	50,740	65,884	51,428	63,266	857,621
Waste Types:										
TILL	kt	2,059	21,904	25,921	13,114	10,976	17,183	9,947	21,743	156,702
Broken NPAG	kt	299	14,407	26,464	21,485	24,125	30,012	9,054	23,312	288,495
Broken PAG	kt	12	8,371	6,802	12,405	11,239	9,318	17,307	13,918	201,312
Stick NPAG	kt	-	-	-	481	1,536	1,576	4,039	1,292	59,758
Stick PAG	kt	-	-	-	3,045	162	3,613	7,274	2,027	132,639
SR (Total Waste/ Ore Mined to Crusher)		-	-	-	2.0	1.9	2.5	2.0	2.4	1.6
Total Material *Mined*	kt	2,370	44,682	59,187	88,925	94,509	108,059	104,060	101,059	1,379,638
Total Material *Moved*	kt	2,370	44,682	59,187	88,925	94,535	108,085	104,086	101,085	1,528,742

Note: Schedule ore tonnage based on 522 Mt and does not include 18.7 Mt of low grade material displaced by higher grade during the early mine life.

25.1.4.3.3 Pit End Of Period Maps

End of period maps are produced to show the progress of benching in each pit phase yearly. They also show the progression of the waste dumps. The end of period maps indicate a reasonable mining progression with bench, crusher and dump accesses maintained.

25.1.4.3.4 Waste Rock Storage

Design Parameters

PAG waste rock will be covered by water within five years of being placed. This is a conservative design criteria considering that recommendations from a kinetic geochemical characterization from core samples at Galore Creek indicated that a delay to ARD onset in any rock is conservatively predicted to be 22 years.

In each period of waste material placement, the maximum PAG dump elevation is kept 3 m below the final water elevation five years in the future.

The distance and elevation differences between the mining area and the waste dumps are minimized to minimize truck hours and reduce costs.

In advance of the water level rising, a 20 m thick lift of PAG material will be placed on the very bottom of the Galore creek valley south of the Waste Divider. This will take place over the first two years of mining production. The lift down the middle of the valley will establish stable dump foundations for future dump lifts that will be dumped into water. When dumps are being built that have to toe out in the water, they are to be built from the middle of the valley outwards. This will ensure that the first part of the dump toe is based on a stable foundation, the 20 m lift on the valley floor. As the dumping progresses from the middle out towards each side, the dump toes are reinforced by the natural ground surface. The general dump face progression is laterally down the centre of the valley and then working out towards the sides. This method of dumping in the Galore Creek valley is a mixture of the thin and thick lift approach as described in the BGC Waste and Water Management Report (April 2006).

All of the crests of the dumps have been kept a minimum of 500 m from the explosives plant and the magazines.

The backfill potential has been considered but not modelled at this stage. Backfilling is very dependent on mining sequence and advance of the pits. Backfilling should be considered in further detail as the mining progresses. Any realized backfill opportunities will reduce the haul distance required and will lead to savings in the mining costs.

Dump Monitoring and Planning

The most economical high lift dumping approach will be used with due diligence and control. Experience in the Elk Valley mining area of British Columbia has shown that high lift dumps can be built in mountainous terrain with minimal risk if proper practices and procedures are followed. Operating experience with dump stability will also be used to predict and control dumps in the Galore Creek mining operations. With careful dump planning and a formal monitoring program minimal dump failures can be expected.

Dumping during the initial stages of mining will be done with low lifts in areas that are non-critical. As experience is gained and stable foundations have been placed, dumping can proceed with higher lifts as required.

Dump Reclaim

The dump design include terraces from wrap around stages. These reduce the uphill haul requirements as the pit benching mines down ward and also reduce the amount of material to be moved to re-slope the final reclaimed dump faces upon completion. As dumps are completed to the final stage, they will be reclaimed as specified in the Mine Closure and Reclamation section.

25.1.5 Mine Operations

The mining operations will be typical of Open pit operations in mountainous terrain in western Canada and will employ tried and true mining methods and equipment. There is a wealth of operating and technical expertise, services and support in western Canada, British Columbia, and in the local area for the proposed operations. The Mine Operations is organized into three areas, Direct Mining, Mine Maintenance, and General Mine Expense (GME).

Direct mining includes the equipment operating costs, operating labour, and distributed mine maintenance costs for the drilling, blasting, loading, hauling, and pit mine operation support activities in the mine. The distributed mine maintenance includes items such as maintenance labour and repair parts which contribute to the hourly operating cost of the equipment.

Mine maintenance accounts for the supervision, planning, general shop and any undistributed costs of the Mine Maintenance activities. The cost in these items are not distributed to the equipment fleets.

GME includes the Supervision and training for the direct mining activities as well as technical support from mine engineering and geology functions.

In the Feasibility Study the direct mining and mine maintenance is planned as an owner's fleet with the equipment ownership and manpower being directly under operations. It may be possible to contract out some of the mining activities under typical mine stripping and maintenance and repair contracts (MARC) as has been done at other operations. The viability and cost effectiveness of contracting can be determined in future detailed and commercial planning. The exception is in blasting where the mine will employee the blasters but due to the specialty expertise required, the supply and onsite manufacturing of blasting materials is assumed to contracted out.

25.1.5.1 Direct Mining Unit Operations

Insitu rock will require drilling and blasting to create suitable fragmentation for efficient loading and hauling of both waste and ore material. For the Feasibility Study the drilling and blasting design provides a particle size distribution and a dig-ability (Looseness in the muck pile) suitable for high productivity from the selected shovel and truck fleet. Some operations also increase the blasting energy in ore to enhance ore comminution (Crushing and grinding). Blasting for improved mine to mill performance can be optimized in future studies.

Ore and waste will be defined in the blasted muck pile and a fleet management system will keep track of each load to ensure material is hauled to the correct destination as well as to provide production statistics for management and reconciliation of the mine operations with respect to the mine plan. Descriptions of the unit mining operations follows.

25.1.5.1.1 Drilling

Much of the material on the initial mining benches may be free digging weathered rock or glacial tills. This will be followed by the more friable broken zone then intact porphryitic rocks both requiring drilling and blasting. Dozers will be used to establish initial flat working areas for the shovels and trucks and as blasting is required, areas will be created on the bench floor elevation to provide rows of holes on the spacing and burden specified for the broken and stick rock material. On hill-side benches, ramps will be cut on the slope where the flat bench doesn't provide drill access close enough to the edge to meet the burden and spacing requirement of the pattern for the next bench below.

The blasthole drills will be fitted with GPS navigation and drill control systems to optimize drilling. The GPS navigation will allow for stake-less drilling and is considered a necessity due to high snow levels at the Galore site. It is a proven technology utilized at most mines in Western Canada. The drills will also be fitted with samplers to provide grade control samples from the drill cuttings in the ore zones. The drillers will take the cuttings samples (two to three samples per hole may be required) and bag and tag the samples for the ore control technician to collect each day. These samples will be used for blast hole Kriging to define the ore waste boundaries on the bench as well as stockpile grade bins for the grade control system to the mill.

Two types of drills have been specified, heavy electric rotary drills for the majority of the drilling in Central pit where in-pit electric power is available, and a more mobile, diesel hydraulic drill with a similar hole size, for use in the other satellite pits which don't have electric power and where seasonal operations will require more drill moves. The diesel drill can also act as a supplemental drill for Central pit.

A 150 mm diesel highwall drill is also specified to operate in all pits for controlled blasting and development of initial upper benches. The highwall drill and the development drilling requirements have not been detailed in this study. An allowance of 15% of the production drill hours has been used for costing purposes.

25.1.5.1.2 Blasting

Explosives manufacturer, Orica has assessed the effect of powder factor on fragmentation and provided data for blasting at the Galore Creek operation using the SABREX blast model. This model is a proprietary computer program of Orica. A review of the SABREX fragmentation analysis indicates that a powder factor of 0.32 kg/t for stick rock will achieve a fragmentation of 90% passing 150 cm which is adequate for the size shovels to be used at Galore Creek. Informal discussion with other mines and explosive suppliers in British Columbia confirm a power factor of 0.32 kg/t is suitable in this area.

The broken rock is highly fractured and will not require much fragmentation. Lower powder factor (0.15 kg/t) will be required to achieve sufficient heave (looseness or dig-ability) to enable efficient shovel productivity in broken rock.

Till rock is predominantly loosely packed glacial moraine material. Blasting will be unnecessary in till rock.

A contract explosives supplier will provide the blasting materials and technology for the mine. Because of the remote nature of the operation, an explosives plant will be built on site. The exact location of this plant will be affected by the table of distances that govern the storage of explosives and blasting agents such as ANFO (Ammonium Nitrate Fuel Oil). The current planned explosives plant location is 1.6 km away from any operating areas where men or equipment will be operating on a continual basis.

The type of explosive that will be used is a 70/30 ANFO and emulsion mix. ANFO alone has no water resistance. The inclusion of an emulsion provides water resistance but it may be necessary to use borehole liners to prevent incomplete detonations, however in this Feasibility Study, it has been assumed they will not be required.

Loading of the explosives will be done with bulk explosives loading trucks provided by the explosives supplier. The trucks should be equipped with GPS guidance and be able to receive automatic loading instructions for each hole from the engineering office. The GPS guidance will be a necessity since the high snow loads will require stakeless patterns. The explosives product that is being used is a mix of ANFO and emulsion; therefore, the container on the truck will have two separate compartments. The separation will be set at the proper ratio so that both compartments will be emptied at the same time. This will minimize trips back and forth from the blast pattern to the explosives storage site.

The holes will also have to be stemmed to avoid fly-rock and excessive air blasts. Crushed rock will be provided for stemming material and will be dumped adjacent to the blast pattern. A Cat IT28 loader can then tram the crush to each hole and use the side-tilt feature on the bucket to dump into the hole.

From time to time during the high snow fall period, it is expected that some of the mining areas will be shutdown and may lose regular road access. If a pattern is partially loaded, it will be necessary to tie in the loaded holes and blast before the snow accumulation gets too high to find the surface lines for tie-in. To blast a partial blast it will be necessary to 'square-off' the pattern by loading some holes to complete some rows in the pattern. To this end a specialty loading unit will be required during months of high snow fall. The unit (called a goat by one explosives supplier) is similar to a skidder used in the logging industry (a large tired 4-wheel drive tractor) or alternatively a highway truck fitted with tracks similar to 'Nodwell' units used in the oil and gas industry. This will be fitted with a bulk explosives tank and explosives pumping/mixing capabilities.

The blasting crew will be mine employees and will be on day shift only. Based on existing mines of similar size and previous experience, the estimated crew size will be 4 people. The main duties of the blasting crew will include setting up guard fences around the loading area, guiding and directing the explosives loading truck, preparing the boosters and primers ahead of the actual loading of the holes, stemming the blast holes after they are loaded, tie-in of the blast patterns and detonating of the blasts.

The blasting crew will coordinate the drilling and blasting activities to ensure a minimum two weeks of broken material inventory is maintained for each shovel. The drilling areas and ramps for the hillside holes, will be prepared in suitable time for the next pattern and ramps will be surveyed if required. In winter, the pattern preparation will also include snow removal.

Due to the high snow load, the drilled holes will need to be covered. Also, the blast patterns will not be staked therefore the blasting activities will also have to have GPS control. The blasters will require hand held GPS to identify the holes for the pattern tie-in.

A B-line, Nonel Caps, and Detonating Cord detonation system is used. The blasting crew will place the down hole boosters and down lines and will tie-in the pattern with surface delays. The pattern size may be limited by the rate of snow fall in some months. As the snow depth gets too high it will be problematic to find the holes and the down line making it difficult to tie-in the blast. This may require smaller more frequent blasts to complete smaller patterns before the snow gets too deep.

The explosives contractor will supply and manufacture bulk explosives on site and deliver them to the holes using a digital controlled 'Smart' truck as is common in Western Canadian surface mines.

A 1.5 m subgrade is assumed to ensure there are minimum high spots between holes on the resultant bench floor. The height of the explosives column is calculated from the explosives density and hole diameter to give the required powder factor. The remainder of the hole is backfilled with drill cuttings or crushed rock.

25.1.5.1.3 Loading

The design basis assumes three shovels as an optimum fleet size to ensure minimum risk to availability along with minimum capital equipment. Two 100 tonnes dipper class electric cable shovels have been selected as the primary digging units plus one 85 tonne bucket class diesel hydraulic shovel. The hydraulic shovel will have a higher operating cost but it is more mobile and so will facilitate operating in the seasonal satellite pits which won't be electrified.

A rubber tired front end loader is also specified to rehandle stockpiled material and as a pit clean up, snow removal, or as an alternate to load trucks in the pit if periodic low shovel availability requires it. The use of the FEL should be minimized due to its higher hourly operating cost and longer truck loading times.

The loading units will also be fitted with a GPS based digging monitor which will enable digital dig boundaries from the ore control system to define the ore types and waste on the shovel operators graphics screen in the cab. This will be definite requirement since grade control stakes will not be visible during high snow fall periods.

25.1.5.1.4 Hauling

Ore and waste haulage will be handled by large off highway haul trucks with a 345 tonne payload. Haulage profiles have been estimated from pit centroids at each bench to designated dumping points for each time period. These haul profiles are inputs to the truck simulation program and the resulting cycle times are used in MineSight® schedule optimization routine (MSSP) which is set to maximize project NPV by using the shortest haul to a feasible destination. The payload, loading time and haul cycle then determines the truck productivity. In the production schedule, the truck productivities range from 415 tph to 2,800 tph, depending on the loading equipment and haul distance.

A GPS based Fleet Management/Dispatch system is specified for the trucks, shovels, and the ancillary equipment fleets to ensure coordination and proper management of the fleet over multiple pits in a large mining area. This will be particularly important with the high snow fall at the site. State-of–the art wireless communication and location systems for management and potential navigation assistance should be considered during the detailed planning and specifications for the project. The capacities, and capabilities of these systems have improved greatly in the last few years and the costs are decreasing.

25.1.5.1.5 Mine Operations Support

Mine operations support including haul road maintenance, dewatering, transporting operating supplies, and the snow removal crew will be directed by the General Foreman. Manpower and equipment costs are included for these activities. The snow fleet will be manned by mine operations staff in normal winter conditions with operators taken from reduced activities such as dust control and summer field programs. During severe storms additional crew to man the snow fleet will be drawn from truck and shovel operations as the long-haul fleets shutdown. During adverse conditions some Mine Maintenance personnel from the shovel crew etc. may be required to man additional snow fleet equipment.

25.1.5.2 Mine Maintenance

Mine maintenance activities will be directed by the Maintenance General Foreman who will assume overall responsibility for mine maintenance and will report to the Mine Superintendent. (In an alternate organization plan, this position may be filled at a Superintendent level reporting to the General Manager.) Maintenance planners will co-ordinate planned maintenance schedules. The daily maintenance shift co-ordination will be carried out by Mine Maintenance Foremen and Electrical Foremen.

The Mine Maintenance department will perform break-down and field maintenance and repairs, regular PM maintenance, component change-outs, and in-field fuel and lube and tire change-outs. Major component re-builds are done by specialty shops off-site and are costed as sustainable capital repairs.

25.1.5.3 GME and Technical

Mine GME and technical departments will include Mine operation supervision down to the Foreman level, training, mine engineering and geology.

A mine superintendent will assume responsibility for overall supervision for the mining operation. The General Mine Foreman will be responsible for overall open pit supervision and equipment coordination. Supervision will also be required for drilling and blasting, training, and dewatering. A mine shift foreman is required on each 12-hour shift, with overall responsibility for the shift operation. Security/First aid staff and mine clerks will also report to the mine superintendent.

Initial training and equipment operation will be provided by experienced operators. As performance reaches adequate levels, the number of experienced operators can be decreased to an optimal level.

A Chief Mine Engineer will direct the mine engineering department. The mine engineer will co-ordinate the senior and junior engineers, the mine planning group, geotechnical monitoring, and the surveyors. A senior surveyor will assume responsibility for surveying for the entire property who will supervise surveyors. Surveying will use GPS based systems. The surveyors will also provide mine technical services. The Geotechnical Engineer will assume responsibility for all mine geotechnical issues including pit slope stability and tailings dam construction quality control.

The Geology department will include a Senior Geologist, Junior Geologists and Ore Grade Technicians. This department will be responsible for local step out and infill drill programs for onsite exploration activities and updating the long range mine ore body models. The Geology department will also provide grade control support to mine operation, managing and executing the blast hole sampling and blast hole Kriging of the short range blast hole models for operations planning and ore grade definition.

25.1.6 Mine Fleet Details

The mining equipment descriptions below illustrate general specifications so that, dimensions and capacities can be determined from the manufactures, specification documents. Where the list includes specific make and brands, this does not mean that final equipment has been selected. The make and model listed is meant to be representative of equipment of that size class. Final brand selection will be subject to a commercial tender process.

25.1.6.1 Major Mining Equipment

A summary of the major mining equipment fleet schedule until Year 10 is tabled below.

Table 25-5: Major Mining Equipment Schedule

Year	-3	-2	-1	1	2	3	4	5	6	7	8	9	10
Drilling													
Primary Drill - P&H 120A - Electric Drill	0	1	1	1	1	1	1	1	1	2	2	2	2
Secondary Drill - P&H 250XP Diesel Drill	0	1	1	1	1	1	1	1	1	1	1	1	1
Highwall Drill - Sandvik D245S 150 mm Diesel	0	1	1	1	1	1	1	1	1	1	1	1	1
Loading													
P&H4100XPC Cable Shovel - 104 t	0	1	2	2	2	2	2	2	2	2	2	2	2
O&K RH400 Hydraulic Shovel - 85 t	1	1	1	1	1	1	1	1	1	1	1	1	1
Le Tourneau L-2350 Wheel Loader - 73 t	0	1	1	1	1	1	1	1	1	1	1	1	1
Cat 3516 GENSET	5	5	9	2	2	2	2	2	2	2	2	2	2
Hauling													
Cat 797B Haul Truck - 345 tonnes	5	14	18	23	23	23	23	23	23	23	23	23	23

25.1.6.2 Drilling

Drilling will be carried out with electric (primary drill) and diesel (secondary drill) 12¼ " drills. Central pit area will mainly use electric drills, while Junction and South West can only use a diesel drill. The cost data used in the Feasibility Study is based on the P&H 120A drill and a P&H 250 XP drill.

Mining will begin in overburden and broken rock, and finish off in stick rock. No blasting is planned in overburden and the broken rock assumes a relatively low powder factor with high penetration rates. The results is one electric drill and one diesel drill required at start up and a second electric drill required in Year 7.

All highwalls in stick rock will be drilled for wall control and none of the walls in broken or till rock will require the use of a high wall drill. The total length of pit crests in stick rock have been calculated in MineSight 3D. The average length of crest exposed per year was estimated by dividing the total crest length by the number of years in the life of mine. The cost model assumes a 3m pre-shear drill spacing and a charge equal to 30% of normal production drill holes charge.

25.1.6.3 Blasting

The blasting activities described above will require an onsite storage and explosives manufacturing plant as well as a management and maintenance facilities for the explosives contractor. The location of the plant and the magazines are determine by the table of distance as specified in the Canadian Federal Explosives Act. The Galore operation will own and build the facilities for use by the contractor. This will include serviced buildings and power and communication links. The facilities have been specified by the explosives contractor. The contractor will also provide specialty equipment such as the computer controlled bulk loading trucks and any other site specific equipment. The capital and operating costs of these facilities and equipment is part of the contractor's unit rate for supply of the explosives.

25.1.6.4 Loading and Hauling

A fleet matching study indicates the optimum truck shovel fleet for Galore Creek is the 345 tonne truck (Cat 797 or Equivalent) and 105 t dipper shovel (P&H 4100 XPC or equivalent).

The lowest cost/tonne fleet of hydraulic shovels and trucks is the Terex O&K RH400 hydraulic shovel matched with the Caterpillar 797B haulers. A hydraulic shovel will be required for operation in non electrified satellite pits on the property.

The lowest cost/tonne fleet of wheel loaders and trucks is the Letourneau L-2350 wheel loader matched with the Caterpillar 797B haulers. A wheel loader will be required as back up for production loaders that are mechanically unavailable, as well as for loading material from Run Of Mine (ROM) stockpiles. The wheel loader will also be used for road cleanup especially during snowfall events.

The surge in demand for large mining truck tires over the past three years has led to a tire supply shortage. Manufacturers indicate that this shortage has been sustained for the last two years and that new tire production due to come online in 2007 will not alleviate the shortage. The tire shortage applies to all large mine haul trucks and therefore remains a project risk. Tire supply commitments from tire suppliers and manufacturers for all tire types are imperative to the success of this project. Tire suppliers have suggested that NovaGold enter a fixed contract to ensure site tire requirements are met.

25.1.6.5 Pit De-Watering

It is important to control the water that is in the active mining areas so that continuous operations are un-impeded. Water greatly increases the sensitivity of tires to rock cuts and the presence of water in the shovel digging area can greatly decrease the average tire life of the trucks. Rocks can easily be hidden in puddles that the haul trucks have to drive through and this can lead to tire failure (puncture). Wet muck that the shovel is digging can easily get frozen to the sides of the truck boxes in the wintertime and this "carry back" results in less actual payload (ie lower productivities). Water also affects the stability of walls and dumps.

Horizontal drain holes must be established in the final walls as they are exposed. On the active bench floor, the water that is collected from the horizontal drain holes will be directed to the sump where it can be removed from the pit. Ditches will be put into the berms to collect the water and direct it to a berm sump in an area where the berm is sufficiently wide. These ditches must be lined and kept clean to avoid water seeping back into the wall. The water from the sump on one berm can be drained down to the next berm below and collect into another berm sump. Actual operating conditions and detailed engineering will determine how many berm sumps can be collected together before putting a pump in place to remove the collected water. It is estimated that as the mining progresses down, the horizontal drains in the wall higher up will be mostly dry except during seasonal times when the water in the wall is recharged by rainfall or snowfall.

Sloped bench floors will also aid in keeping the digging face dry. A gradient of 1% is usually sufficient to collect the water to one area where sump pumps can then be used to pump the water out of the pit. The direction of the slope will have to be determined individually for each pit but generally the floors should slope downwards to the initial starting point of each bench. That way as the shovels dig outwards and away from this starting point, the water will drain back away from the shovel digging area. A sump can be dug into the bench floor to collect this water and a pump put in to remove the water from the pit. The sloping of the floors will also cause the berms to be sloped and the ditches that are established in the berms will naturally drain to one side of the pit. This side of the pit is where the berm sumps should be established.

All surface water and precipitation in the pit will be handled by submersible sump pumps installed in each active pit bottom as part of the flexible and moveable bench scale pumping system. The sump pump will be connected to semi-permanent and permanent piping systems to convey the sump water out of the pits. The sump will be installed with each box cut as the benching is advanced. With the high amount of precipitation it is assumed that the box cuts will have to be made wide enough to facilitate the sump pump and piping as required, as the face advances and until a bench sump can be established on each new bench. The peak monthly rainfall over the area of the pits is estimated at 9,000 Gallons / minute. A series of 6 submersible pumps, capable of handling 1,500 Gallons/minute of water at 75 metres of head, will be required for pit sump dewatering. The excavation of the sumps is therefore included in the direct mining costs but the pump handling and piping is included in mining support costs. These sump pumps will run the water through HDPE pipe to a tank located on a mined out pit bench at the side of the pit not currently being advanced. This tank is sized to handle approximately 1 minute of water residence, with a series of centrifugal pumps, sized to handle 9000 Gallons/minute flow at 200m of head. Permanent piping will then take the water to the pit rim either up the highwall, along service berms or on the side of the highwall ramp, as required, using HDPE piping. The piping will run from the pit rim into the Galore Creek Valley, where it will be collected into the waste dump/tailings dump pond water. The costs for operating this system are included in the mine power consumption cost section of the cost model.

25.1.6.6 Mine Operations Support

The following is a summary of the support equipment requirements at Galore Creek. There are a number of pieces of equipment described below that are identical to equipment chosen for the fly-in/contractor fleet of equipment required for preproduction construction tasks. That equipment is considered completely separate from the support fleets described below, but there are opportunities for synergy that have a potential to save money in upfront capital expenditure that should be examined in the detailed design phase of the project.

The following equipment is chosen specifically for support of the mining operations. All equipment is chosen to start operations in the last two months of Year -3 and continues to the end of mine life, unless otherwise noted. The equipment is replaced as required and costs for this equipment are applied according to the details included in the cost model.

The mine operations support fleet size in Year 5 is listed in the table below.

Table 25-6: Mine Operations Support Fleet or Equivalent (Year 5)

Mine Operations Support Fleet	Fleet Size
Cat 385C Hydraulic Excavator - 12 tonne	1
Cat 24H Grader - 7.32 m Blade	3
Cat D11R - 634 kW	3
Cat 988H Multipurpose - Cable Reeler, Forks, Brushes, Bucket	1
Cat 345 Hydraulic Excavator - 5-7 tonnes	1
Cat 844H Rubber Tired Dozer - 463 KW	2
CAT IT28G Hole Stemmer - 3 tonnes	1
Cat 988H Tire Manipulator	1
Cat 789 C Water Truck - 48,000 Gallons	2
Cat 740 Fuel/Lube Truck	3
Kenworth T800 FireTruck	1
Kenworth C500 Picker Truck	2
Chevrolet G3500 Passenger Vans	2
Ambulance	1
Hyster 620F Forklift - 30 tonne	1
Hyster 210HD Forklift - 10 tonne	1
Ford F550 Maintenance Truck - 1 Tonne	2
Kenworth T300 Service Truck	3
Ford 1/2 Ton Pickups	20
Cat D10T - 433kW	3
Cat 789 Float Tractor/Trailer - 189 tonnes	1
Mine Rescue Truck	1
GMC Guide XL Crew Bus	1
Fintec 570 Screening Plant - 12" max	1

25.1.6.7 Mine Maintenance Support

Mine Maintenance Support equipment is chosen specifically for support of the duties of the mine maintenance department. The mine maintenance fleet size in Year 5 is listed in the table below.

Table 25-7: Mine Maintenance Fleet in Year 5

Pit Maintenance Fleet	Fleet Size
LTM1250-6.1 - 250 tonne crane	1
Kenworth T300 Welding Truck	1
PowerLine Truck	1
LTM1100 - 100 tonne crane	1
Tsurumi LH8110-60 Water Pump - 1,400 Gal/min	5

25.1.6.8 Snow Fleet

A dedicated snow fleet has been designed for Galore Creek based on a snow fleet study conducted for the PEA. All equipment is chosen to start operation in the last two months of period -3, according to the schedule, and continues to the end of mine life, unless otherwise noted. The equipment is replaced as required and the costs for this equipment are applied according to the details included in the cost model.

The Snow Fleet has a low utilization and is only required in wintertime. Other than the use of the crusher to produce road gravel, operating this equipment outside of wintertime is optional and not necessary. The size of the Snow Fleet in Year 5 is listed in the table below.

Table 25-8: Snow Removal Fleet in Year 5

Snow Removal Fleet	Fleet Size
Cat 637G Scraper - 37 tonnes	10
Cat 988H Wheel Loader - 14 tonnes	1
Cat 16H Grader - 4.88m blade	1
Fintec 1080 cone crusher - 7.5" max.	1
LMC 1500 Snowcat	2

25.1.6.9 Personnel

Operations and maintenance and Hourly labour are estimated in the mine cost model based on labour factor estimates for each operating equipment in the mine fleet. Labour costs are estimated and allocated to each unit of equipment.

25.1.7 Mine Start-up and Construction

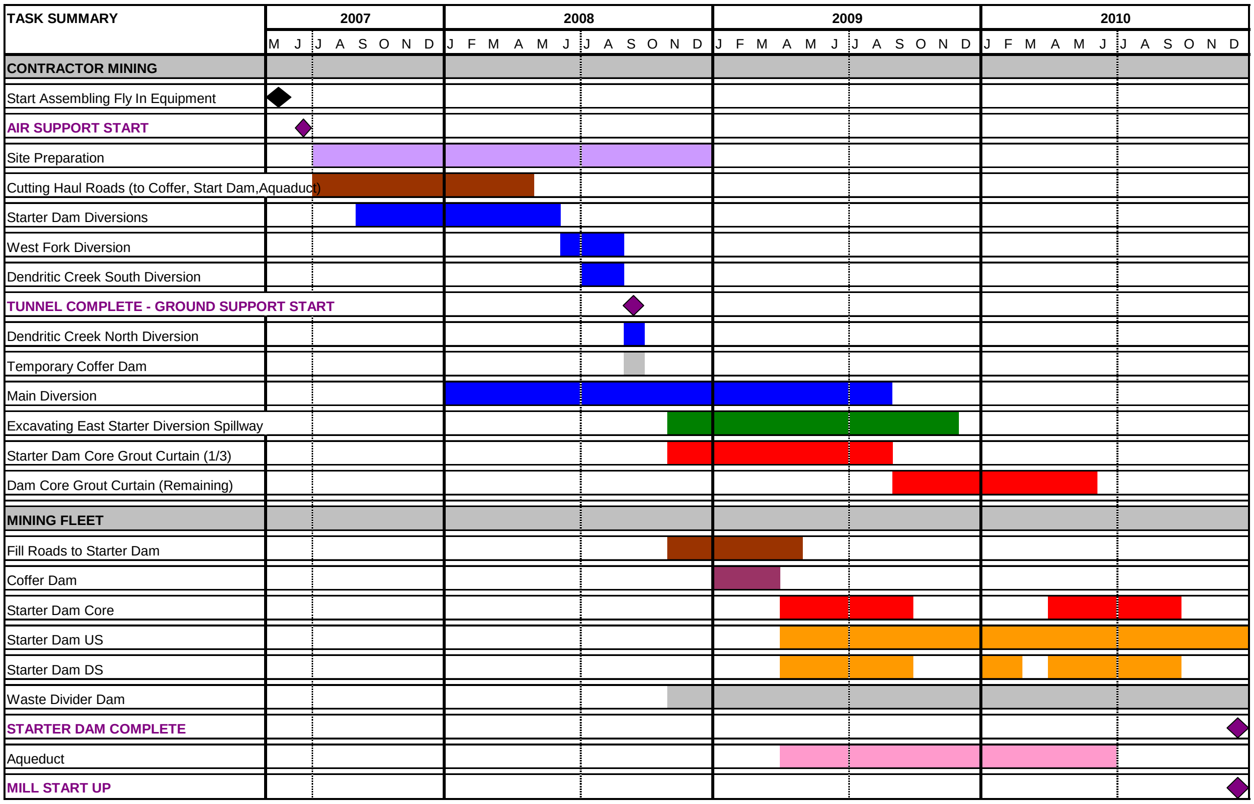
25.1.7.1 Introduction

Substantial earthworks are required prior to the Galore Creek mill startup. The preproduction earthworks activities recommended by BGC have been engineered to provide schedules, materials, equipment, labour and infrastructure requirements and their associated cost estimations.

25.1.7.2 Construction Schedule

The tasks described below are summarized in the following project schedule:

Table 25-9: Summarized Galore Creek Project Schedule (Mining)



25.1.7.3 Diversion Channels Optimization

BGC Engineering has designed layouts for the diversion channels that allow for the building of various structures during the pre-production period and also allow for mining of the pits. These diversion channels divert water away from the working areas such as dam construction areas and pit production areas. There are six main diversion channel structures: Main, East Starter Dam, West Starter Dam, West Fork, Dendritic Creek – North and Dendritic Creek – South. The design parameters for the diversion channels are given in the BGC Waste and Water Management Report. All of these diversion channels are shown in the end of period maps and in the BGC Waste and Water Management Report.

Cut and fill for the diversion channel layouts have been optimized by GR Tech to minimize construction costs. The cut and fill volumes are balanced along 500m sections. A summary of all the cut and fill volumes is shown in the table below:

Table 25-10: Cut and Fill Volumes of Diversion Channels

Name	Cut Vol kBCM	Fill Vol kBCM
Main Channel	6,225	4,699
East Starter Dam Channel	6,333	1,243
West Fork Channel	767	182
Dendritic South Channel	540	613
Dendritic North Channel	145	153
West Starter Dam Channel	414	492
TOTAL	**14,424**	**7,382**

Note: KBCM = Thousands of Bank Cubic Meters

25.1.7.4 Contractor Construction Tasks

25.1.7.4.1 Introduction

The site preparation at Galore Creek is broken into tasks. The major material required for the dam construction has been assigned to the owners mine fleet as described above, and general site preparation tasks and water structure construction tasks have been estimated as contractor tasks. These contractor task estimates include: the equipment to perform the tasks; the people required to operate and maintain the equipment, as well as manage the project; the materials required to support the operation and transportation of materials to site. The detailed contractor construction tasks are described in the Feasibility Study.

25.1.7.4.2 Summary

The early construction period refers to the period of 2007 to 2010 (Period -3 to Period – 1) in the production schedule), and includes all on site jobs related to mining, but not being accomplished with the mining fleet. The early construction tasks are outlined in the table below.

Table 25-11: Required Early Construction Tasks at Galore Creek

Job #	Water Diversion Work	Amount	Equipment Required	Timeline Required (months):
1	Building Starter Dam East Diversion (m^3)	5,600,000	6 D10T's	9.99
2	Building Starter Dam West Diversion (m^3)	415,000	3 D10T's	1.48
3	Building West Fork Diversion (m^3)	700,000	2 637G's, 3 D10T's	2.55
4	Building Dendritic Creek South Diversion (m^3)	540,000	3 D10T's	1.93
5	Building Main Diversion to Friendly Creek (m^3)	1,550,000	6 D10T's	2.76
6	Building Main Diversion to Aqueduct (m^3)	3,125,000	4 D10T's	7.43
7	Building Main Diversion to End (m^3)	1,550,000	3 D10T's	5.53
8	Building Dendritic Creek North Diversion (m^3)	0	3 D10T's	0.00
9	Placing Rip Rap Material Along Lined Diversions (m^3)	334,200	1 345C, 2 730's	4.25
10	Compacting Water Diversion Channel Material (m^3)	6,740,000	1 825H	31.89

Job #	Water Diversion Work	Amount	Equipment Required	Timeline Required (months):
11	Building Temporary Coffer Dam (m^3)	20,000	1 345C, 2 730's, 1 D10T	0.25
12	Building West Fork Diversion Dam (m^3)	30,000	1 345C, 2 730's, 1 D10T	0.38
13	East Fork Diversion Dam (m^3)	364,500	1 345C, 2 730's, 1 D10T	4.64
14	Starter Dam Fine and Coarse Filter Material (m^3)	321,000	1 345C, 3 730's, 1 D10T	4.08
15	Placing the Initial Lifts of the Starter Dam (m^3)	93,000	1 345C, 3 730's, 1 D10T	1.48
16	Clearing Overburden from Starter Dam Area (m^3)	1,573,500	2 345C, 4 730's	10.01
17	Preparing Primary Coffer Dam Area (m^3)	100,000	1 345C, 2 730's, 1 D10T	1.27
18	Starter Dam Toe Sump (m^3)	210,000	2 345C, 4 730's	1.34
19	Drilling Starter Dam Core Grout Curtain (m)	98,338	3 Grout Hole Drills, Hoists	10.47
20	Drilling Remaining Dam Core Grout Curtain (m)	80,117	3 Grout Hole Drills, Hoists	8.53
21	Grouting Main Dam Core and Toe (kg)	7,826,952	Grout Plants and Loader	10.47
22	Grouting Remaining Main Dam Core and Toe (kg)	1,728,833	Grout Plants and Loader	8.53
23	Compacting Tailings Dam Core Material (m^3)	6,095,430	2 825H	5.66
24	Spreading Tailings Dam Truck Loads (m^3)	25,529,374	6 D10T	20.84
25	Tree Clearing and Grubbing (m^2)	4,000,000	1 D10T	17.69
26	General Site Tasks	-	Various Pieces	42.00
27	Drilling and Blasting Spillway Outlet Channel (m)	36300	1 Airtrak Drill, Blasthole Loader	4.19
28	Drilling and Blasting Site Bedrock (m^3)	2,600,000	2 Airtrak Drills, Blasthole Loader	36.29
29	Clearing and Leveling Plant Area (m^3)	2,500,000	6 D10T's	2.97
30	Excavating Dewatering Trenches in Plant Area (m^3)	36,000	1 345C, 2 730's	0.46
31	Excavating Spillway Outlet Channel (m^3)	168,000	1 345C, 2 730's	2.14
32	Excavating East Starter Diversion Spillway (m^3)	2,065,000	2 345C, 6 730's, 1 D10T	13.14
33	Building Bear Creek Crossing (m^3)	200,000	2 345C, 6 730's, 1 D10T	1.27
34	Building Camp Creek Crossing (m^3)	500,000	2 345C, 6 730's, 1 D10T	3.18
35	Building Friendly Creek Crossing (m^3)	400,000	2 345C, 6 730's, 1 D10T	2.54
36	Cutting Haul Route to Coffer Dam (m^3)	1,097,244	3 D10T's	2.61
37	Continue Cutting Haul Routes to Tailings Dam (m^3)	2,543,768	2 D10T's	9.07
38	Cutting Haul Route to Aqueduct (m^3)	602,869	1 D10T	4.30
39	Placing Piping along Diversions	-	Picker Truck	31.67
40	Placing Geotextile along Diversions and dams (m^2)	600,000	Picker Truck	42.00

As part of the Feasibility Study, detailed task descriptions were completed for each of the early construction tasks, the complete fleet size for each month was determined.

The figure below shows the manpower on site at one time and the total manpower size for all fleets.

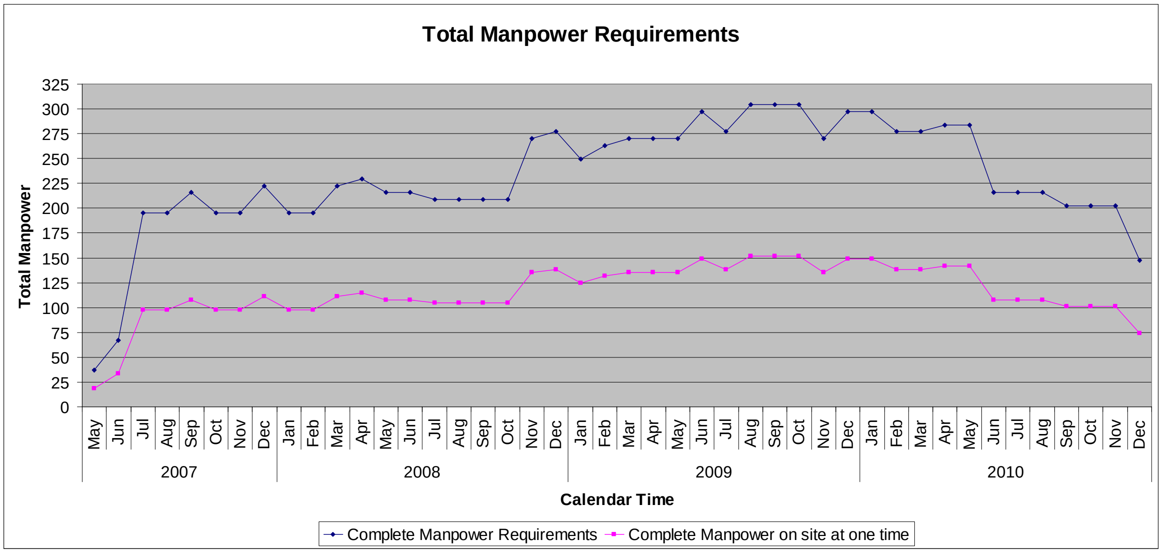


Figure 25-1: Total Early Construction Manpower Requirements

25.1.7.5 Construction Fly-in Requirements

For the Early Construction phases of the Galore Creek project, all materials and people will be flown to site by helicopter. A study estimate of the required helicopter support is summarized below.

Three types of transportation are available for flying into the Galore Creek work areas:

- 20,000 kg payload Mi-26 helicopter at an estimated CA$26,000 per hour, 45 persons per trip
- 10,000 kg payload Boeing 234 (Chinook) helicopter at CA$13,500 per hour, 24 persons per trip.
- 3,400 kg payload Sikorsky S-61 L helicopter at CA$3,750 per hour, 5 persons per trip.

For each transportation method, the number of loads to site is determined. The cost per load is determined, and the total cost for each month is determined. Manpower loads are considered separate and on top of material loads.

The table below shows the costs of each method of transport, as well as a per kg cost for each method

Table 25-12: Total Costs of Fly-In

	20,000 kg loads	10,000 kg loads	3,400 kg loads
Total Fly-In Cost	CA$47,256,259	CA$48,880,638	CA$41,431,277
$/kg Hauled (includes costs for manpower on top)	CA$2.08	CA$2.15	CA$1.82

The following tables are summaries of the loads to support early construction operations before the tunnel to site is open. They do not include any site services, camp services, etc. They also do not include the construction needs for the access tunnel. The number of loads shown includes both material loads and manpower loads.

Table 25-13: Summary of 20,000kg Helicopter Fly-In Loads

Summary:	20,000 kg loads	# of Loads	Days to Haul in Monthly Loads	Months Required	Transport Costs (CA$)
Total Loads Before May 1st, 2007		**94**	16	0.8	$3,652,350
Total Monthly Loads					
2007	May	**6**	1	0.0	$217,490
	Jun	**6**	1	0.1	$243,490
	Jul	**64**	11	0.5	$2,479,782
	Aug	**64**	11	0.5	$2,498,638
	Sep	**67**	11	0.6	$2,607,081
	Oct	**63**	11	0.5	$2,470,853
	Nov	**63**	11	0.5	$2,470,853
	Dec	**68**	11	0.6	$2,658,258
2008	Jan	**63**	11	0.5	$2,470,853
	Feb	**63**	11	0.5	$2,470,853
	Mar	**68**	11	0.6	$2,658,258
	Apr	**69**	11	0.6	$2,686,536
	May	**67**	11	0.6	$2,606,371
	Jun	**67**	11	0.6	$2,606,371
	Jul	**65**	11	0.5	$2,527,409
	Aug	**65**	11	0.5	$2,527,409
	Sep	**65**	11	0.5	$2,527,409
	Oct	**65**	11	0.5	$2,527,409
	Nov	**60**	10	0.5	$2,348,587
	Dec	**0**	0	0.0	$0

Table 25-14: Summary of 10,000kg Helicopter Fly-In Loads

Summary:	10,000 kg loads	# of Loads	Days to Haul in Monthly Loads	Months Required	Transport Costs (CA$)
Total Loads Before May 1st, 2007		**187**	31	1.6	$3,792,825
Total Monthly Loads					
2007	May	**11**	2	0.1	$223,763
	Jun	**12**	2	0.1	$249,075
	Jul	**127**	21	1.1	$2,564,171
	Aug	**128**	21	1.1	$2,583,753
	Sep	**133**	22	1.1	$2,695,216
	Oct	**126**	21	1.1	$2,554,899
	Nov	**126**	21	1.1	$2,554,899
	Dec	**136**	23	1.1	$2,747,978

Summary:	10,000 kg loads	# of Loads	Days to Haul in Monthly Loads	Months Required	Transport Costs (CA$)
Total Loads Before May 1st, 2007		187	31	1.6	$3,792,825
Total Monthly Loads					
2008	Jan	126	21	1.1	$2,554,899
	Feb	126	21	1.1	$2,554,899
	Mar	136	23	1.1	$2,747,978
	Apr	137	23	1.1	$2,776,960
	May	133	22	1.1	$2,694,479
	Jun	133	22	1.1	$2,694,479
	Jul	129	22	1.1	$2,612,863
	Aug	129	22	1.1	$2,612,863
	Sep	129	22	1.1	$2,612,863
	Oct	129	22	1.1	$2,612,863
	Nov	120	20	1.0	$2,438,917
	Dec	0	0	0.0	$0

Table 25-15: Summary of 3,400kg Helicopter Fly-In Loads

Summary:	3,400 kg loads	# of Loads	Days to Haul in Monthly Loads	Months Required	Transport Costs (CA$)
Total Loads Before May 1st, 2007		551	92	4.6	$3,098,713
Total Monthly Loads					
2007	May	35	6	0.3	$199,019
	Jun	41	7	0.3	$232,769
	Jul	388	65	3.2	$2,180,004
	Aug	390	65	3.3	$2,196,003
	Sep	408	68	3.4	$2,295,981
	Oct	386	64	3.2	$2,172,429
	Nov	386	64	3.2	$2,172,429
	Dec	416	69	3.5	$2,342,058
2008	Jan	386	64	3.2	$2,172,429
	Feb	386	64	3.2	$2,172,429
	Mar	416	69	3.5	$2,342,058
	Apr	421	70	3.5	$2,368,708
	May	408	68	3.4	$2,295,379
	Jun	408	68	3.4	$2,295,379
	Jul	396	66	3.3	$2,225,728
	Aug	396	66	3.3	$2,225,728
	Sep	396	66	3.3	$2,225,728
	Oct	396	66	3.3	$2,225,728
	Nov	354	59	3.0	$1,992,579
	Dec	0	0	0.0	$0

- The Sikorsky S-61L Helicopter is the lowest cost/kg helicopter available, but it is not productive enough to handle all loads to site. Therefore, for feasibility all loads are assumed to be done by the Mi-26 helicopter.
- The type of equipment chosen to work during the early construction phase at Galore Creek is based on its ability to breakdown to 10,000kg maximum weight loads. If the smallest helicopter is used, smaller construction equipment must be examined. If the larger helicopter is used, larger equipment may be possible.

25.2 Recoverability

Section 25.2 was prepared by Mr. Hoe Teh, P.Eng., Hatch

(Refer to Section 18 Mineral Processing and Metallurgical Testing)

25.3 Markets

A marketing study was performed by Neil S. Seldon & Associates Limited. This study provides current smelter terms for copper, gold and silver concentrates, possible buyer issues and various viewpoints. As stipulated in Section 5.3, this technical report has relied upon this study as well as other market related information in its preparation.

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.

25.3.1 Copper

Commodity prices in general have risen in face of a falling dollar value, rising costs, expanding demand in face of tight supply and over the last two years or so considerable flow of investment funds. Metal prices are no exception and it is apparent that a higher plateau of long-term levels has been established.

The copper price on September 01, 2006 was at the US$3.50/lb level, at the same time, the 3-year trailing average copper price was US$1.70/lb. As a base case, this Feasibility Study has selected a copper price of US$1.50/lb. As a low case, the study has used the staggered copper prices forecast forwarded by Neil Seldon & Associates during July 2006. For the period up to 2015 US$1.44, for the period 2015 to 2019 US$1.28, and for the period 2020 to 2030 US$1.17.

Economic cycles based on historical patterns may suggest a period of future lower prices as the current cycle peaks, but rising Asian and other demand will fuel the need for more production in the face of rising costs of production which may continue to support higher price levels over the medium to long term.

To calculate the requirement for mine expansions and Greenfield projects, one must net out the proportion that will be satisfied by secondary production, but one must also add in the amount of new capacity needed to offset ore grade declines at continuing mining operations and offset mine closures through ore reserve exhaustion. The amount of new mine capacity each year required from Greenfield mining projects and expansions will be around 1,025 ktpa copper in 2010, 1,250 ktpa by 2020, and

1,500 ktpa by 2030. This demand requirement and the dearth of new copper mine developments to an advanced stage of production supports a continued market for new copper projects.

Table 25-16: Annual Mine Project Requirements (ktpa copper)

Year	2010	2020	2030
To Satisfy demand growth	775	1,025	1,425
To offset grade declines	225	300	400
To offset closures	125	175	225
Less extra secondary production	-100	-250	-500
Overall project requirement	1,025	1,250	1,500

Source: BME

25.3.2 Silver and Gold

25.3.2.1 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 has boomed in price over 2004. The silver price on September 01, 2006 was at the US$12.87/oz level, at the same time, the 3-year trailing average silver price was US$7.72/oz. As a base case, the Feasibility Study has selected a silver price of US$8.00/oz. During July 2006, Neil Seldon & Associates were forecasting silver prices forward at US$6.70 for the period up to 2030 ; these assumptions are incorporated into the low case scenario. In US dollar terms along with other commodities, a higher price plateau seems assured. Silver has found favour with investors and speculators and many analysts are forecasting higher long-term prices.

25.3.2.2 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.

The gold price on September 01, 2006 was at the US$626.40/oz level, at the same time, the 3-year trailing average gold price was US$461.67/oz. As a base case, the Feasibility Study has selected a gold price of US$525/oz. During July 2006, Neil Seldon & Associates were forecasting gold prices forward at US$495.19 for the period up to 2030. In US dollar terms along with other commodities, a higher price plateau seems reasonable. Production costs are rising and gold has found favour with many investors. While the US dollar still remains to some extent a haven for value and wealth, gold has regained favour and many analysts are forecasting prices well in excess of the study base price.

25.3.3 US Dollar Exchange Rate

One key feature of recent 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. 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.

The value of the Canadian dollar is susceptible to commodity prices, particularly oil and metals. The current US/Can dollar exchange rate is 0.89/1.00. As of September 01, 2006, the 3-year trailing average exchange rate was 0.81/1; the 4-year trailing average exchange rate was 0.75/1. It does not seem unreasonable to link predictions of the US/Can dollar exchange rate with metals prices (i.e., higher metal price in US$ corresponding to a higher Canadian dollar). As a base case, the Feasibility Study has selected 0.81/1.00.

25.3.4 Smelter Terms

25.3.4.1 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. For 2007 a downward trend in annual charges is likely. Indeed, by April 2006 a drop in spot charges appeared and set the tone for midyear 2006 levels. However, more important is a move to reduce and cap price participation.

***Note the following**:*

According to: London (Mineweb.com) It now appears confirmed that BHP Billiton has achieved a substancial reduction in smelter charges from at least one Japanese copper smelter, which follows on from similar charge benefits gained in negotiations with a major Chinese smelter earlier in September. For the period July 2006 to June 2007, the toll charges agreed withJapan's Pan Pacific Copper Co. will be US$60 per tonne for treatment and 6.0 cents per pound for refining.

According to: The American Metal market has indicated that Minera Escondid, the worlds largest copper mine has settled a mid-year copper treatment and refining charge (TCs&RCs) contract at a rate of US$73 per tonne and US$7.3 cents per pound with at leas one Chinese smelter. … It has also been reported tht the Highland Valley Copper mine (97.5%-owned by Teck Cominco Ltd.) has negotiated a mid-year TCs and RCs contract of US$60 per tonne and US$6 cents per pound with a Japanese smelter.

In any assessment of charges today, the emergence of China as a smelter "super-power" cannot be over-looked as indeed is the case for India. For long-term valuation, a treatment charge of US$80 to US$85 per dmt and a refining charge of US$0.080 to US$0.085 per pound of payable copper is generally assumed by many, with price participation of $^{+}$10% from a base price of US$1.20 per pound with a price participation cap at US$1.80.

25.3.4.2 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 Feasibility Study preparation, the only likely penalty will be for fluorine which is expected to be minor, but for the record, some typical penalties are included. It should also be noted that penalties vary with the market and the capability of particular plants.

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:

25.3.4.3 Payable Metals

Copper: Deduct 1 unit and pay for balance of content with refining charges of US$0.08/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

25.3.4.4 Deductions

Treatment Charge (TC):
TC CIFFO main Asian port parity, US$80/dmt

Price Participation (PP):
PP +10% over US$1.20/lb with a cap at US$1.80/lb
-10% under US$1.20/lb with a floor at 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

25.3.5 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

25.3.6 Marketability

Based on assays received and shown in the Table below the concentrates are relatively clean except for the fluorine content. This may attract some penalty from time to time although the dollar amount is not likely to be significant. However, while some buyers may well consider this a concern from time to time, Galore Creek will have a major advantage in terms of location and political stability. Therefore, a healthy market for Galore Creek concentrate is expected. The expected target market area for the concentrate is Asia given the west coast location and the accessibility of west coast ports.

Table 25-17: Concentrate Assays

Element	Units	Grade
Antimony	%	0.011
Arsenic	g/t	<10
Bismuth	g/t	35
Chlorine	g/t	<100
Cobalt	%	0.010
Copper	%	29.9
Fluorine	g/t	533
Gold	g/t	15.9
Iron	%	25.4
Lead	%	0.059
Magnesium Oxide	%	0.24
Mercury	g/t	<1
Nickel	g/t	30
Palladium	g/t	1.1
Platinum	g/t	0.05
Rhodium	g/t	2.2
Selenium	g/t	153
Silica	%	2.70
Silver	g/t	130
Sulphur	%	32.5
Tellurium	g/t	<20
Zinc	%	0.29

25.3.7 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.

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 while 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 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.

25.3.8 Other Offsite Costs

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

25.3.9 Contracts

No mining, concentrating, smelting, refining, transportation, handling, sales and hedging and forward sales contracts or arrangements have been negotiated to date. Rates and assumptions used within the Feasibility Study Economic analysis are within industry norms.

A Participation Agreement was signed on February 10, 2006 with the Tahltan First Nation. The agreement outlines how NovaGold will work together with the Tahltan during construction, operations and closure phases of the project and addresses environmental, employment and business opportunities. An annual cash payment of a minimum of $1M CA/yr is guaranteed during the operations phase of the project. Upon reaching certain threshold levels related to Cumulative Net Smelter Return, this payment can increase and these payments then become related to annual Net Smelter Return with certain provisions to provide for a minimum $1M/yr payment. These payments have been incorporated into the cashflow and economic analysis/sensitivities sections of this report.

25.4 Environmental Consideration

The Environmental Consideration section has been written under the supervision of Clem Pelletier, President –Rescan / RTEC

25.4.1 Introduction

NovaGold recognizes that local community involvement and support of mineral development projects is key to the success of the project, and acknowledges the value of Traditional Knowledge (information gained about an area through observation over an extended period of time, rather than through scientific study). The geographic location of the proposed mine has dictated that much of the focus of consultation efforts has been with the Tahltan Nation.

In early meetings with the Tahltan people, NovaGold agreed to support the formation of several joint ventures, one of which was the May 2004 formation of Rescan Tahltan Environmental Consultants (RTEC), a 50/50 joint venture between Rescan Environmental Services Ltd. and the TNDC. RTEC combines the environmental expertise of Rescan with the regional savvy of the TNDC. The ethos of the union is perhaps clearest in its logo, which includes the phrase “Keepers of the Land,” representing a long-standing moral obligation assumed by the Tahltan people to maintain the sanctity of the natural landscape. Both companies see the RTEC venture as an opportunity to meet their business goals while contributing to sustainable development practices in the resource sector.

During the 2004 and 2005 field seasons, RTEC took the lead in collecting and assembling all baseline data and information on environmental components such as climate, water quality, fish, and wildlife, for the development of the environmental assessment. NovaGold also collaborated with RTEC and Kwantlen University College to develop a learning/training program for environmental field work geared toward the Tahltan community.

NovaGold initiated discussions with the Tahltan Nation in November 2004, within two months of signing the deal to explore Galore Creek. Several open houses were held in the predominantly Tahltan communities of Dease Lake, Iskut and Telegraph Creek as well as in Stewart, in 2004 and 2005. NovaGold prepared and distributed the *Galore Creek Newsletter*, providing details and updates about the project, in these communities, and a Tahltan Elder and members of the TCC and Iskut First Nation participated in a series of site visits. Nova-Gold also funded local researchers to conduct projects that incorporated and documented Traditional Knowledge about the region. A highlight of the consultation process was a Special Assembly held in January 2005 in Dease Lake. A series of workshops and information sessions were held, and community members provided extensive feedback on the project.

Much knowledge was gained from the consultation process. NovaGold’s decision to select the modified northern access route over to the southern access route was based heavily on information provided by Tahltan Traditional Knowledge, including the importance of the Iskut and Stikine rivers, and the fish and wildlife habitats, wetlands and vegetation found along the southern route.

Tahltan Elders expressed concern over the toxic impacts the Galore Creek Project could have on wildlife if concentrate were to spill into the environment. The project team addressed these concerns and incorporated pipelines rather than truck transport to pump the concentrate from the process plant to Highway 37 and to supply diesel to site, in order to decrease the number of trucks on the access road.

Water management options, such as the timing of discharges from the tailings storage facility, were guided by the needs of aquatic life (e.g., pacing discharges from the storage facility to match natural flow conditions). Traditional Knowledge interviews emphasized the economic importance of fish and the importance of preserving the integrity of aquatic resources.

A number of the wildlife and aquatic VECs, such as marmot and marten, were identified or confirmed through interviews with Tahltan members. Interviews with Tahltan Elders about seasonal rainfalls helped to guide water management strategies for discharges from the tailings storage facility. Tahltan Elders and members confirmed the importance of protecting hunting grounds and wildlife stocks in the localized study area. The Participation Agreement between NovaGold and the Tahltan provides for the development of a road protocol to address concerns about use of the road. Traditional Knowledge also provided NovaGold with an understanding of traditional land use in both the broad Cassiar Iskut-Stikine region and the local project area. This information was used during numerous baseline studies.

Ongoing discussion with the Tahltan community resulted in the signing of the Participation Agreement on February 10, 2006 (following a vote which endorsed this agreement with 85% Tahltan support). The agreement supports the Tahltan Nation's principles of environmental stewardship, economic sustainability and self-determination. It also commits both parties to working collaboratively throughout the environmental assessment review and the permitting process for Galore Creek. The agreement provides certainty for local communities and investors alike that Galore Creek will be developed with the support and involvement of the Tahltan Nation.

NovaGold initiated meetings with government regulators shortly after the project officially entered the pre-application phase of the B.C. environmental assessment process in February 2004. Overview meetings were held with Canadian regulators from the provincial and federal governments in Smithers in April 2004, and with American regulators in Juneau in May 2004. A joint meeting with close to 30 regulators held in June 2004 helped define and clarify the similarities and differences between the B.C., Canadian, Alaskan and U.S. regulatory system. Tahltan leadership participated in all of these meetings as well. These general meetings and the technical working groups that were formed to address specific issues provided NovaGold with valuable input on the baseline program.

Regulators became more familiar with the project by participating in two site tours in the summer of 2004. Additional overview meetings were held in Smithers in 2005 to provide an update of the project and present the 2005 baseline program for comments and input. The technical working groups continued to meet throughout 2004 and 2005 and held more than twenty meetings. Tours to site were conducted for smaller groups of regulators, as well as representatives from the provincial and regional governments.

The early engagement and participation of the Tahltan Nation and various government regulators and the Tahltan has enabled NovaGold to incorporate valuable input into the design and planning for the Galore Creek Project.

25.4.2 Regulatory Framework

The federal environmental process, governed by the CEAA, is the federal measure by which the project's integrity is tested. In a similar mandate to the BCEAA, the CEAA also ensures that the environmental effects of projects are carefully reviewed before federal authorities take action in connection with them so that projects do not cause significant adverse environmental effects. Under CEAA, projects again receive a level of environmental assessment tailored to their impact potential. There are four environmental assessment review options under CEAA: screening, comprehensive study, mediation and panel review. The Galore Creek Project has initiated the CEAA process, and a comprehensive study report is required. This environmental assessment is meant to satisfy both the provincial and federal approval requirements.

On the matter of public consultation for First Nation groups, government and the community at large, NovaGold began an extensive outreach to interested parties who will become involved in the Galore Creek Project.

Table 25-18: Environmental Assessment Process Under the BC Environmental Assessment Act

Step 1: Determining if the *British Columbia Environmental Assessment Act* Applies
1. Project is included in Reviewable Projects Regulation: • provide brief project description to the Environmental Assessment Office (EAO), including information related to Reviewable Projects Regulation threshold criteria 2. Minister designates project reviewable: • government determines designation 3. EAO designates project reviewable (proponent requested): • apply to EAO to designate project reviewable, providing brief project description and reasons for seeking designation
Step 2: Determining the Review Path
Step 3: Determining How the Assessment will be Conducted
• respond to EAO requests for information needed to establish framework for asssessment, including scope of assessment and methods and procedures to be used • provide project information to interested parties
Step 4: Developing and Approving Application Terms of Reference
• in accordance with Procedural Order, undertake issue identification/scoping and consult with government agencies, First Nations and public • prepare draft Terms of Reference (TOR) and revise as required based on review comments • provide final TOR to EAO
Step 5: Preparing and Submitting the Application
• conduct studies as specified in terms of reference • consult as appropriate/required as studies proceed • prepare application in accordance with TOR and submit application to EAO for screening • revise application if required and resubmit • provide required copies of acceptable application
Step 6: Reviewing the Application and Referring to Ministers
• provide notice of application review • conduct consultation in accordance with approved consultation plan and any additional required measures • respond to issues raised during comment period • provide additional information as required
Step 7: Preparing the Assessment Report
• continue any ongoing consultation activities • continue to respond to information requests
Step 8: Deciding to Issue/Not Issue a Certificate
Ministers make decision (within 45 days) and: • issue environmental assessment certificate; or • refuse to issue environmental assessment certificate; or • require further assessment

25.4.3 Description

25.4.3.1 Mining

Mining of the open pits will employ conventional truck and shovel methods over a twenty-year period. Mining will be performed top-down, in bench-like steps, within multiple pits simultaneously. The slopes of the pit sides have been designed specifically for stability under local geological and groundwater conditions.

The ore deposits are near the surface, so little pre-stripping will be necessary. Both electric and diesel equipment will be used to extract ore and load a fleet of large-capacity trucks. Mining operations are scheduled to allow an even flow of materials to the mill using a consistent number of haul vehicles.

Overburden and topsoil stockpiled during mining operations will be used later for reclamation purposes. Waste rock will be hauled to several dump areas. Waste rock that has been determined to be not potentially-acid-generating will be used to construct a tailings dam and other water management structures, while acid-generating waste will be hauled to a designated engineered dump for long term management under water.

25.4.3.2 Processing

Ore will be processed at a rate of approximately 65,000 tonnes per day in a conventional mill. The mill will operate 24 hours a day, 365 days a year, with scheduled downtime for equipment maintenance.

Trucks will deliver ore from the open pits to either an ore stockpile or the primary crusher, which will be fitted with water sprays and other measures to minimize dust. Crushed ore will be conveyed to a stockpile with a 32,000 tonne capacity. From there, it will enter the grinding circuit, where it will be pulverized to an even finer texture. The ground ore will be further refined to a concentrate using conventional flotation processes. Residual materials will be removed and placed in a tailings pond. The concentrate will be pumped to the filter plant via the concentrate slurry pipeline.

25.4.3.3 Pipeline

A 135 kilometre concentrate slurry pipeline will be constructed between the mill and the filter plant located near Highway 37. The pipeline system will incorporate concepts, equipment and operating conditions proven in currently operating slurry pipelines. A large pump will move the concentrate slurry through the pipeline at a rate of about 90 tonnes per hour. Five monitoring stations will be installed along the pipeline to monitor its operation and detect leaks within a few minutes of their occurrence.

The pipeline will be buried at depths ranging from 1.6 to 3 metres. It will generally follow the access road ditch, but minor diversions will be constructed to maintain appropriate grades and to avoid the settlement of solids in low points during shut-downs. Heat from friction and the slurry will be sufficient to prevent the pipeline from freezing in most areas. Where the pipeline crosses watercourses on bridges, it will be insulated to prevent freezing.

The pipeline will also feature a protection system and external coating to prevent corrosion. A covered tank will be constructed at the low point of the pipeline in the Lower Sphaler Creek area to serve as an emergency drain point.

A smaller pipeline will be installed next to the concentrate pipeline to transport diesel fuel to the mine. The diesel pipeline will have similar leak detection and protection systems as the concentrate pipeline.

25.4.3.4 Filter Plant

A filter plant will be constructed at the terminus of the concentrate slurry pipeline, next to the access road and 8.5 kilometres west of Highway 37. At this plant, water will be removed from the concentrate slurry and treated before it is discharged. The clean water from the filter plant will be pumped through a small underground pipeline to the Iskut River. The concentrate will be conveyed by truck to the Port of Stewart, via Highways 37 and 37A. In addition to treatment and storage facilities for water and slurry, the filter plant will include accommodations for 30 plant workers and truck loading and weighing facilities. It will also include the storage and pumping systems for the diesel pipeline.

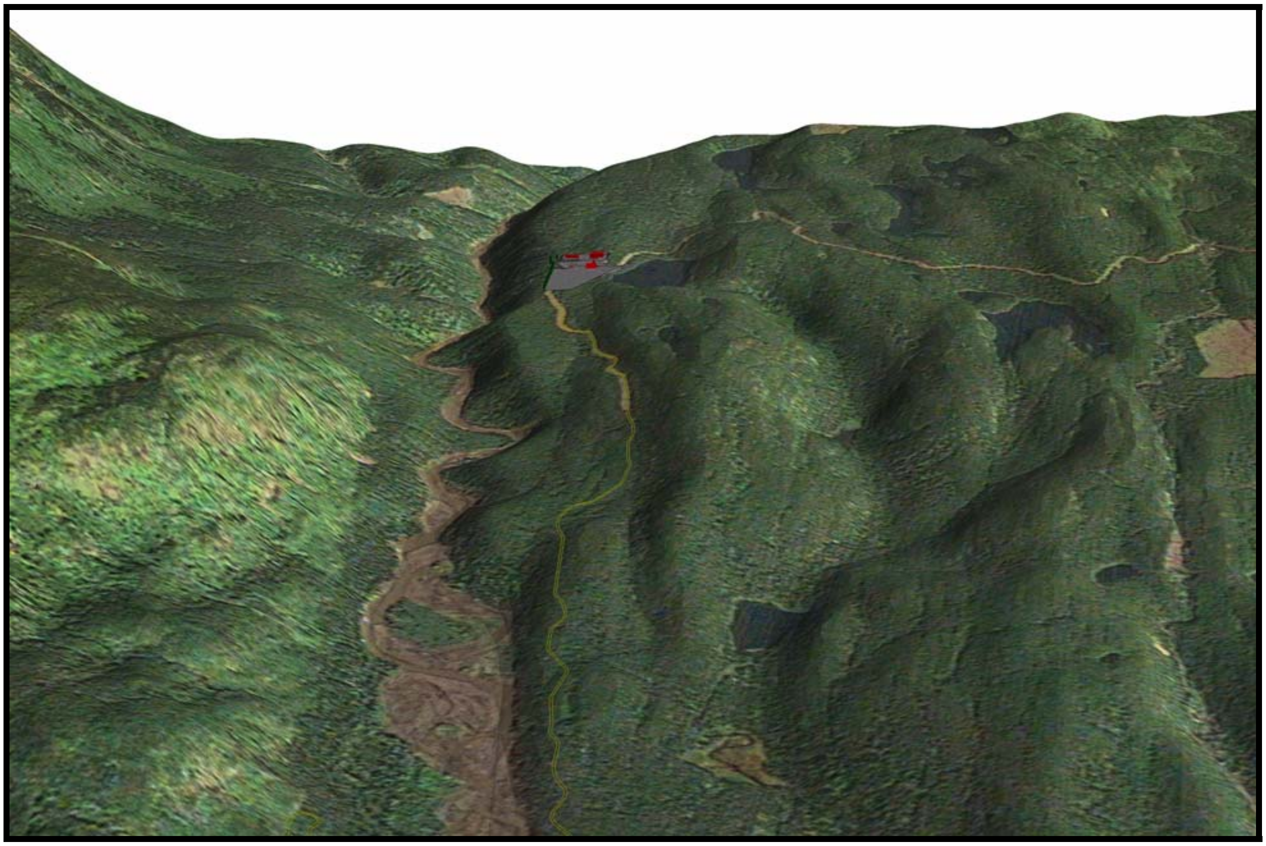


Figure 25-2: Filter Plant Location Aerial View

25.4.3.5 Water Supply

Potable water for the construction, mine and filter plant camps will be supplied from wells. This water will be treated by ozonation or chlorination, and will be stored at dedicated tanks at each camp. Process water will be pumped from wells around and within the pits, as well as from the concentrate and tailings thickener overflows. If required, the water will be combined with water recycled from the tailings pond. The process water will be stored in a steel tank or in a small process water reservoir to be constructed near the mill, and pumped to the various distribution points.

25.4.3.6 Tailings and Waste

The main tailings dam will be constructed with an impervious central core. Initially, the height of the dam will be 152 metres, but it will be raised every year to accommodate the anticipated annual volume of tailings. The completed dam will have a 30 metre wide crest and a 1.1 kilometre wide base. A seepage collection system will be constructed below the dam to collect seepage from the tailings impoundment, and an emergency spillway will provide a route for excess water in case of flooding.

Tailings will be pumped to the tailings impoundment. During operations, a pond and settling basin in the impoundment will be used to collect and reclaim water for use at the plant.

The Galore Creek mine is expected to generate a total of 1,016 million tonnes of waste rock. This waste rock will be hauled by the mine fleet to a designated dump in the middle of the Galore Creek valley. To prevent tailings from mixing with the waste rock dump, a waste dump divider will be constructed 2.6 kilometres upstream of the tailings dam, and will be built up throughout the life of the mine so that its crest remains above the top of the tailings.

25.4.3.7 Freshwater Diversions

Freshwater runoff will be diverted around the proposed plant, pits, tailings, and waste dump facilities using several open channels. These channels will also feature diversion dams, an aqueduct, and other structures necessary to divert and control water flow. Some of the major drainages that the channels will cross, such as Bear Creek and Friendly Creek, exhibit signs of past debris flow activity. Debris flows will be managed upstream of the diversion channels, to minimize blockages. The diversion channels will be inspected and maintained year-round.

The main diversion channel will be 12.3 kilometres long, excavated into bedrock or surficial materials along the eastern slopes above the tailings and waste impoundment from the East Fork of Galore Creek to the main tailings dam. It will be up to 5.5 m deep and 10 m wide at the base. A total of five emergency overflow areas are planned along the main diversion channel so that if a blockage occurs, water will pass into the waste catchment area upstream of the blockage.

25.4.3.8 Access Road

The construction and operation of the remote Galore Creek mine will require the development of a new road. After considering several alternative routes, and consulting with the TCC, NovaGold chose a route beginning at the Devil Creek forest road north of the Bob Quinn airstrip near Highway 37, crossing the Iskut River just upstream of the mouth of More Creek, following More Creek to the headwaters of its western fork, descending Sphaler Creek to near its mouth, and then crossing over the divide to Scotsimpson Creek. From there, it will follow the course of Scotsimpson Creek to its headwaters and enter a 4.2 kilometre tunnel to the western fork of Galore Creek. The road will be about 128 kilometres from Highway 37 to the tunnel, 118 kilometres of which will require new construction. A three kilometre spur road will also be built to the proposed Porcupine Aerodrome.

This route is more difficult to engineer compared with other potential routes, but was determined to be the least harmful to sensitive wildlife habitats. Due to steep grades and other factors affecting the access road, NovaGold has opted not to use the road for regular shipments of concentrate, which will instead be transported via the concentrate slurry pipeline. This pipeline, as well as an electrical power transmission line and a fuel pipeline, will follow the access road alignment.

25.4.3.9 Aerodrome

An aerodrome designed to accommodate fixed wing aircraft will be established near the confluence of Sphaler Creek and Porcupine River. Its initial function will be to support construction activities. The aerodrome will be maintained as a key feature of the operational infrastructure to permit timely all-season transportation of personnel.

25.4.3.10 Other Facilities

Employees at the mine will live at camp on a two-week-in, two-week-out work rotation. The camp will house 570 people, accommodating all direct and contract employees for the site during mine operation. The camp will include kitchen, dining, recreation and fitness facilities. The mine will also be equipped with a full laboratory to support mining and environmental functions.

Three storage buildings will provide covered storage and a venue for maintenance and other activities during the winter. A truck shop, warehouse and office complex constructed 1.3 kilometres north of the plant site will house truck maintenance bays, a truck wash bay, a tire shop, repair areas, office space, a first aid room, and other amenities.

The on-site explosives manufacturing and storage facilities will be constructed in a fenced, secured compound approximately 1,200 metres north of the truck shop. Access will be limited to authorized personnel.

25.4.3.11 Mine Schedule

Once the main access road and the tunnel are established, major production and construction equipment will be mobilized to the site. The timeline of construction will begin with the development of the access road from the mine site to the north tunnel portal. Next, diversion channels on the east and west sides of the valley will be installed. This construction will take place during the spring of 2007 to facilitate summer start for the tailings dam development.

The key cut for the tailings dam installation is slated for 2007, to be completed in 2008, at which time the plant site development will begin. The pipeline pumping station will be constructed in 2008/2009. The power transmission lines will be installed shortly after the access road is completed, and the main plant substation will be completed in late 2008, coinciding with the completion of the tunnel construction.

The main infrastructure buildings will be constructed beginning in early 2009, and equipment will be shipped and installed at the end of that year. The remaining construction and installation will be completed by the late fall of 2010.

25.4.3.12 Galore Creek Project Implementation Schedule

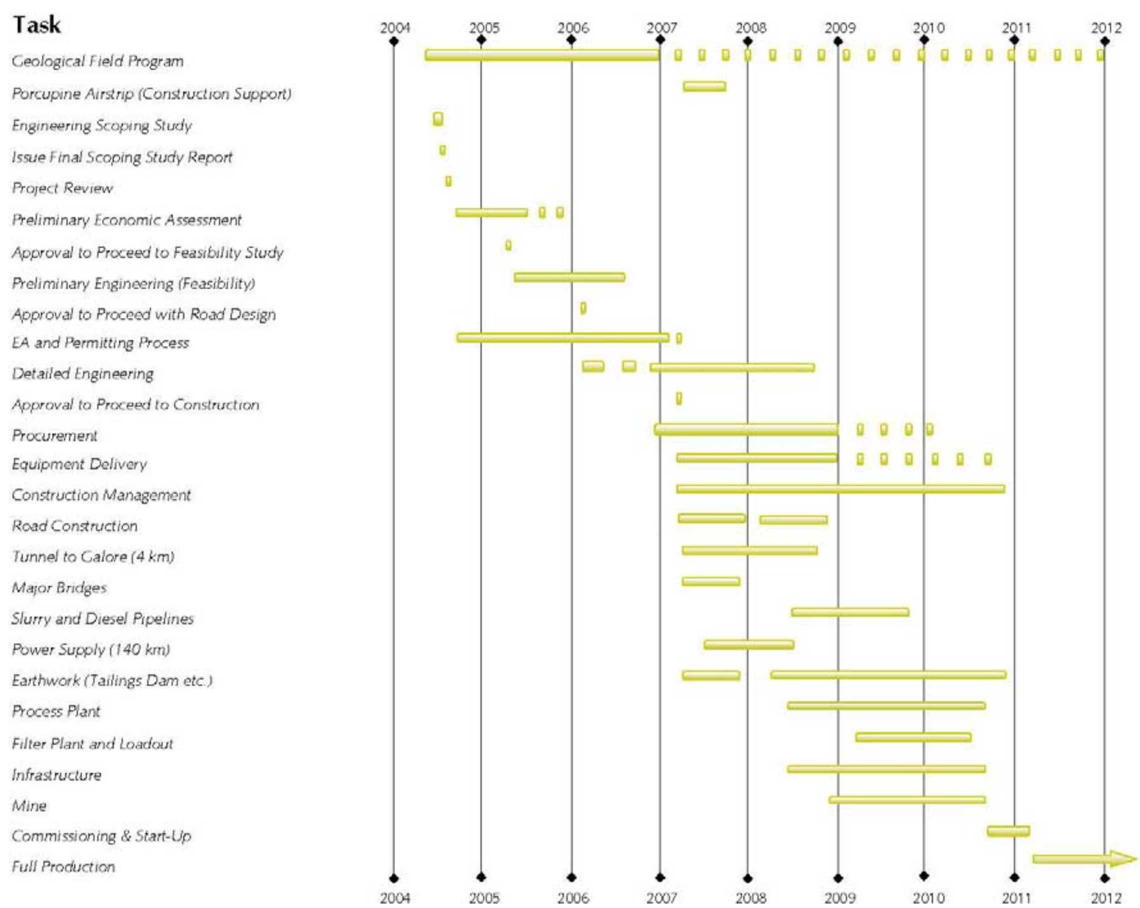


Figure 25-3: Galore Creek Project Implementation Program

25.4.4 Environmental Setting

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 LRMP identifies fifteen geographic resource management zones, covering 31% of the plan area. One of these, the Lower Stikine-Iskut Grizzly Salmon Management zone, includes the valley of the Stikine River from the Chutine confluence to the US border, and the lower Iskut River west of the Craig River. It also includes the Scud River into which Galore Creek drains. Mineral exploration and development are accepted activities within the Coastal Grizzly/Salmon Management zone, including road access where needed. Adjacent areas include an operating mine (Eskay Creek), a number of past producers including the recently closed Snip mine, two projects in the Environmental Assessment

Process (mine development review phase), and a number of other exploration prospects. Logging is only allowed for the purposes of mineral exploration and/or mine development and for localized use.

The Galore Creek Project property is located in northwestern British Columbia, adjacent to the Alaska panhandle, some 1,000 kilometres north of Vancouver and approximately 90 kilometres northeast of Wrangell, Alaska. The area is a transitional landscape between Coast and Mountain, Sub-Boreal Interior and Northern Boreal Mountains ecosystems. Typical biogeoclimatic zones (geographic areas having similar patterns of vegetation and soils as a result of a homogenous climate) range from Coastal Western Hemlock and Mountain Hemlock zones to the west of the Galore Creek property and the Interior Cedar Hemlock and Engelmann Spruce-Subalpine Fir zones to the east. Alpine tundra is present at higher elevations.

The property lies within a regional structure known as the Stikine Arch. Medium to steep slopes characterize the local terrain in the central and northern parts of the Galore Creek property. The surrounding topography is mountainous. Elevations on the property range from 500 metres to over 2,000 metres above sea level. The elevation of the tree line is variable, but alpine vegetation predominates above 1,100 metres. The forests below consist of Balsam fir, Sitka spruce and cedar. A variety of unique habitat types exist within the larger regional project area, including extensive floodplain habitat and wetlands, moist alpine meadows and mature and old growth forest.

The Galore Creek project area includes major watersheds of both the Stikine and Iskut river drainages. The Stikine watershed is recognized as a major wilderness area of significant ecological value to both Canada and the United States and the feasibility design has been carefully considered to mitigate potential impacts to this ecosystem.

The climate of the Stikine-Iskut basin is strongly influenced by the Pacific Ocean to the west and continental Arctic regions to the northeast. As a result, the Galore Creek Project area has typically mild, wet coastal climatic conditions at lower elevations and cold, dry northern interior at higher ones.

The meteorology in the valley is typical for a humid continental climate zone. Summers are cool and winters are cold, humid and wet with several months of snow cover. Average stream flows are high and annual freshet runoff rates very high, as remnant glaciers are present within the headwaters of many watersheds in the area. Recent glacial retreat within the Cordillera of North America has exposed large areas previously covered by millions of tonnes of snow and ice. These areas are now subject to precipitation and runoff erosion that results in large quantities of rock and sediments into the aquatic environment.

The Stikine, Iskut, More, Sphaler and Porcupine valleys themselves are relatively pristine areas with road access currently limited to the upper reaches of the Iskut Valley. The Stikine and Iskut rivers and their tributaries provide important habitat for all five species of Pacific salmon as well as other resident fish species such as Dolly Varden. The area is also one of the more important remaining grizzly bear habitats in British Columbia. Wetlands along the Porcupine and Stikine rivers provide breeding habitat and migration staging areas for waterfowl. The valleys and associated floodplains provide important moose winter range and the rugged Coast Range supports high densities of mountain goats. There are resident populations of black bears, wolves, foxes, martens and other mammals.

Important Tahltan communities in the vicinity include Dease Lake, Iskut and Telegraph Creek, which are respectively 166, 118 and 90 kilometres by air from the proposed mine site. Neighbouring First Nations include the Nisga'a, Kaska and Tlingit. Traditionally, the Tahltans travelled through their territory in small family groups following the available food supply. They hunted animals, fished and gathered plants for food. Fish harvesting from the Stikine River was an important activity each spring and summer, when the Tahltan would gather in large social groups for feasting, visiting and trading.

The Stikine River also provided the Tahltan with a link to the ocean. Situated between coastal and interior native groups, the Tahltan became a crucial trading link for the region. Each year, the Tlingit from Wrangell, Alaska, made their way up the river to dry fish and trade goods with the Tahltans.

25.4.5 Environmental Impacts

NovaGold recognizes that different components of the environment require particular consideration during the planning and design of the Galore Creek Project. Local communities and others have recognized these components as matters of scientific concern, or as relevant to cultural values. These components are widely termed Valued Ecosystem Components (VECs) and were formally identified through a comprehensive consultation with the Tahltan Nation, federal and provincial regulatory bodies, and other interested parties. Following the consultation, further screening was done to determine if the identified natural and human environmental components qualified as VECs.

Evaluation of the potential environmental effects of the Galore Creek Project begins with the VECs. They are the most representative aspects of the natural environment deemed susceptible to influence by the wide-reaching scope of the project. As the environmental footprint of the project is large, the determinations about the future integrity of each VEC over the life of the mine are crucial to the development of initiatives designed to lessen environmental impacts whenever it is reasonable to do so.

25.4.5.1 Air Quality

Air quality is a fundamental component of the natural ecosystem and an important health and safety issue; it is of the utmost biological importance to vegetation and wildlife. Air emissions from the project will consist primarily of diesel emissions from the mobile mining equipment and, to a lesser extent, fugitive dust from drilling, blasting and traffic along the unpaved haul and access roads during the two driest summer months.

The effects assessment for air quality was designed to explicitly address the potential air quality issues and concerns, and issues included potential effects on air quality associated with all phases of the project. The project will meet both federal and provincial government ambient air quality objectives intended to ensure long-term protection of public health and the environment. The magnitude of the effects to air quality for all project activities was deemed to be moderate, since the geographic extent of the air quality effects was confined to the local ambient air quality study area.

25.4.5.2 Climate

Climate describes the predominant weather patterns of an area and has been selected as a VEC for the project because it is a fundamental aspect of the natural environment, and changes to the climate will affect many other ecosystem components.

The Galore Creek Project will marginally affect the climate by changing the level of greenhouse gases (GHGs) emissions in the atmosphere. The primary GHGs emissions from man made sources are CO_2,

CH_4 and N_2O. All three gases will be emitted as a consequence of diesel emissions from mobile equipment and, to a lesser extent, small amounts of N_2O from blasting. The amounts are relatively small and the impacts are negligible.

25.4.5.3 Site Noise

Noise has been selected as a VEC because it has intrinsic value for employees and wildlife. High noise levels from mining activities have the capacity to negatively affect employee performance, as well as cause wildlife to leave their preferred foraging, resting and breeding habitats.

Due to the remote location of the Galore Creek Project site, with no residences in close proximity, noise from the project site will have a minimal effect on communities. Noise may disperse wildlife from their preferred foraging and breeding areas, with mountain goat and grizzly gear being particularly sensitive to disturbance. Detailed evaluation of various noise sources at the project site and along the access road indicate that, overall, noise will have a minimal impact in these areas. There are, however, concerns that elevated noise levels from concentrate haul trucks in Stewart may have the capacity to irritate local residents. Stewart's Official Community Plan includes a by-pass around residential and commercial districts, and NovaGold will support Stewart's initiative to develop this road.

25.4.5.4 Surface Water

Surface water is a VEC and a critical component of the biological and physical environment. Water is protected under the *British Columbia Water Act* and the *Canada Water Act*. Surface water quantity is the first of two water-based considerations. The construction of mine site infrastructure for the Galore Creek Project will alter the natural flow paths within Galore Creek Valley. During mine operations much of the mainstem of Galore Creek Valley will be flooded under the waste rock and tailings storage facility. At closure, the diversion channels within Galore Creek Valley will be decommissioned and all runoff will enter the flooded tailings and waste rock storage facility. The storage facility will operate as a free-flowing reservoir or lake. In the West Fork area, the pits will be allowed to fill and will overflow into the storage facility.

Surface water quality is the second consideration, and is linked to other key ecosystem components, including fish and fish habitat, aquatic resources and wildlife. It acts as an indicator of environmental health. Surface water quality for the project has international implications as well, as the Stikine River flows into Alaska.

The assessment of the potential effects of the project on water quality are focused around Galore Creek, where liquid discharge from the tailings and waste rock impoundment in Galore Valley in the receiving environment has the potential to cause low magnitude effects. These low level effects may also occur in a localized area of the Scud River. Discharge of a similar nature from the filter plant to the receiving environment has the potential for similar low magnitude impacts, restricted to a localized area of the Iskut River immediately downstream of the diffuser. Neither of these effects will occur as far downstream as the Stikine River, downstream of the Scud and Iskut rivers.

25.4.5.5 Groundwater

Groundwater has been included as a VEC because the groundwater system is a connected and inseparable component of the surrounding hydrological (creeks, rivers, wetlands) and terrestrial environment of the Galore Creek watershed.

Mine development and operation, including the construction of water diversions, groundwater extraction, open pits and waste management facilities all have the potential to affect the quantity and quality of groundwater in the aquifer contained in Galore Creek basin.

Appropriate mitigation measures, such as materials handling procedures, lined containment structures for potential contaminants, and spill contingency plans will protect groundwater quality. Impacts on the groundwater regime are not expected to be material.

25.4.5.6 Aquatic Resources

The term aquatic resources refers to organisms such as periphyton and benthic invertebrates in streams and river systems, as well as phytoplankton, zooplankton and benthic invertebrates in slow-moving water habitats, which include wetlands and lakes. These biotic components are fundamental to aquatic ecosystems because they serve important functions in cycling nutrients, photosynthesis and the production and processing of organic matter. They also provide food resources to fish, birds and other organisms. In addition to forming the base of the aquatic food chain, aquatic resources are often useful as indicators of environmental quality and can be used to assess various impacts related to degraded water and/or sediment quality.

The aquatic resources of streams, rivers, wetlands and lakes may experience various effects related to the development of the project; effects related to this VEC may vary depending on the phase of mine development and activity of concern. The various physical or chemical stressors introduced to the aquatic habitats may be related to different effects and recoveries for the various groups of organisms present in each system. This is related to biological sensitivity to various stressors, life histories, rates of adaptation, as well as stressor factors such as chemical fate (movement to water or sediment), timing (exposure of critical life stages), duration and magnitude of exposure. Effects on aquatic resources will be observed in the Galore Valley within the mine site area due to the alteration of streams. This will have minimal effects on the fisheries resources located at the mouth of the Galore Creek, and negligible effects in the Scud River.

25.4.5.7 Sediment Quality

Environment Canada specifically requested that sediment quality be characterized as an independent VEC for this project. Federal metal mining effluent regulations require sediment quality monitoring. Sediment quality refers to the physical and chemical nature of the upper sediment layers of aquatic systems in relation to the sensitivities of aquatic organisms found in these areas. Federal and provincial guidelines have been created to assess sediment quality on a parameter-by-parameter basis, as sediment quality is strongly tied to surface water quality with regard to the potential for contamination. Degraded sediment quality can affect aquatic organisms (such as fish), with the potential for contaminants to be passed along through consumption, a potential human health risk. The inclusion of sediment quality as a VEC is thus justified because of its linkage to several VECs including surface water quality, aquatic resources and various fish and wildlife species.

Activities associated with the Galore Creek Project have the potential to cause adverse effects to sediment quality in the immediate and downstream aquatic environments. These issues include physical and chemical changes to sediment quality, such as siltation or altered chemical loadings of metals or nutrients. Siltation is the unintentional transport and deposition of soil particles into the aquatic environment due to surface runoff. The project will have an Erosion Control and Sediment Management Plan in place which will mitigate effects to sediment quality. There will be negligible effects on sediment quality in the downstream aquatic environment.

25.4.5.8 Fish and Fish Habitat

Many fish species serve an important role in the ecological, economic and cultural health of British Columbia and within neighbouring Alaska. Salmon species in particular support local economies and cultures, while other species may be used as indicators of environmental health and water quality. The project has the potential to affect many fish species and their habitats both directly and indirectly over the lifespan of the mine. As a result, individual species and groups of species were isolated for further study as VECs due to their conservation status, commercial value, cultural importance and ecological significance.

The Stikine River was also identified as a VEC because of the importance of its habitat for many life forms, including fish, and because of its cultural and commercial significance. The Stikine River supports a modest commercial salmon fishery as well as subsistence food fisheries for local First Nations communities. It is an important river for all five species of Pacific salmon and supports the highest fish species diversity of any body of water in the region. The most relevant effects of the project on fish and fish habitat are the potential loss of productive capacity at the mouth of Galore Creek and direct loss of habitat in one wetland and a few stream crossings due to construction of access corridors. Fish habitat compensation, primarily construction of off-channel wetlands, will ensure a net regional increase in aquatic biological production.

25.4.5.9 Wetlands

Wetlands were selected as a VEC for the environmental assessment because of their importance in ecosystem functions and in respect of wetland-related policy and legislation. NovaGold has recognized the value of wetlands in the project area and has committed to following the federal wetland mitigation process.

Where logistically possible, project engineers have adhered to the first step in the federal wetland mitigation process and have designed mine infrastructure so as to avoid wetland areas. However, there are locations where portions of mine infrastructure are confined logistically, making it unfeasible to avoid wetlands. Other sections of the project area are also susceptible to wetland loss or alteration due to indirect effects of infrastructure development. As wetlands are defined by their hydrology, subtle changes to this hydrology as a consequence of development could alter wetland quality and quantity.

Project infrastructure has been designed to minimize effects on the hydrology and quality of wetlands, including installation of culverts and the use of concentrate and diesel pipelines to minimize truck haulage along the access road. The loss of wetland habitat will be compensated by construction of off-channel wetlands.

25.4.5.10 Terrestrial Ecosystems, Vegetation, and Soil Landscapes

Terrestrial ecosystem, vegetation and soil landscape VECs primarily include ecosystems and plant species that are sensitive to development, provide habitat for wildlife (with emphasis on wildlife species identified as VECs in the project area) and are of conservation concern to regulatory agencies and the public. Ecosystems selected as VECs were identified as important biodiversity components to multiple stakeholders. The ecosystems include rare ecological communities that would be considered potential candidates for conservation. Soil landscapes supporting these systems are included within each VEC.

Development associated with the Galore Creek Project will directly and indirectly affect terrestrial ecosystems, vegetation, and soils. Direct impacts will include the alteration of ecosystems through the disturbance of vegetation and soils during construction, operations, and decommissioning. Indirect impacts could be in the form of dust from pits or roads, microclimate changes (as with increased sunlight to previously shaded environments) and a decrease in ecological integrity through the potential introduction of invasive plant species. The potential impacts of invasive plant species on ecosystem integrity are thought to have the largest effect. Focussed mitigation measures including progressive reclamation where feasible and revegetation of disturbed sites will avoid or mitigate adverse affects to vegetation and terrestrial ecosystems. The effects on these VECs are not considered to be significant relative to the socio-economic benefits of the project.

25.4.5.11 Wildlife and Wildlife Habitat

Wildlife VECs were recognized as those species or groups of species that are most likely to generate concern from regulatory agencies and the public. They include species of conservation concern, keystone species and species of economic or cultural significance to humans.

A number of species were selected as wildlife VECs for the project, including species at risk or of conservation concern, requested to receive enhanced consideration by regulatory agencies, or identified through traditional knowledge interviews with Tahltan elders as being culturally and/or economically significant to the Tahltan Nation.

The most likely effect to wildlife is sensory disturbance due to noise from industrial activity, aircraft activity, avalanche control and blasting. In many cases wildlife will habituate to these disturbances or will move to adjacent less disrupted areas. Mountain goat, however, are sensitive to disturbance during their breeding season, and grizzly bear may be affected by disturbance when feeding on salmon spawning along the Porcupine River Valley during late summer/early fall. It is uncertain to what extent these two VECs will habituate to disturbance.

Project construction and operation activities will be planned to minimize disturbance of mountain goat natal habitat and grizzly bear salmon feeding habitat during sensitive times of the year. A wildlife management plan developed for the project, which includes access restrictions, a no-firearms policy, and an employee education program, will mitigate many of the other potential adverse effects. Although there may be localized adverse impacts, the overall effects on wildlife will not affect the sustainability of populations.

25.4.5.12 Archaeology

Archaeological resources are non-renewable and are considered valuable provincial resources, similar to mineral deposits, forests, fish and wildlife. In British Columbia, these resources are protected under the *Heritage Conservation Act*, established to encourage and facilitate the protection and conservation of the heritage of British Columbia. In the case of the Galore Creek Project, the provincial Archaeology Branch is directly involved in the referral process through the environmental assessment, making approvals contingent on satisfactory assurance that archaeological sites will not be destroyed, excavated or altered without a permit issued by the Minister or designate.

All documentary and archaeological evidence pertaining to the archaeological resources of the Stikine region suggest that the project area was peripheral to the intensive use areas of Mount Edziza, the upper Stikine drainage system and the Klappan Plateau. Archaeological surveys conducted along the proposed access road route and within the Galore Creek Valley resulted in the discovery of six sites. The access route has been re-aligned to avoid direct impacts to all but one of these sites; a recent helicopter crash site from which most of the remains have been removed. The Galore Creek Project will therefore have minimal effects on archaeological and heritage resources.

25.4.5.13 Navigable Waters

At the majority of the stream and rivers that the access corridor will cross there is a limited human requirement for navigation. Galore Creek, More Creek and Sphaler Creek are steep sided mountain streams with limited access for river craft. However, there are some reaches along the Iskut and the Porcupine rivers that are either currently or potentially capable of being navigated. All bridge crossings for this project are designed to accommodate human navigational requirements.

25.4.6 Socio-economic Setting

With the exception of Stewart, the residents of northwestern B.C. are largely members of the Tahltan Nation living in the communities of Dease Lake, Iskut and Telegraph Creek. Historical and archaeological data suggests that the Tahltan are Athapaskan-speaking people of Dene descent. The Tahltan people migrated to northwestern B.C. from the Athabasca region thousands of years ago and claim extensive territorial hunting and fishing grounds. These long-inhabited Tahltan communities are considered to lie within the primary area of socio-economic impact of the Galore Creek Project.

Northwestern B.C. is relatively remote from the rest of the province and supports a small population generally dependent upon the region’s resource base, making land-based economies important for Tahltan people. The nearest large communities to the project site are Terrace and Smithers to the south and southeast; these communities lie within the secondary area of impact of the Galore Creek Project. The Tahltan people represent two-thirds of the residents in the Galore Creek area. Annually, families gather during the summer months in fishing villages such as Tahltan Village, located at the confluence of the Stikine and Tahltan rivers, and during the fall and winter families head into known hunting areas to acquire meat provisions.

The Tahltan people have identified the long-term viability of the regional mining sector as an essential driver for the economic, cultural and political advancement of the Tahltan Nation. Northwestern B.C. is a rich geological environment where mining has been a characteristic of economic activity since the 1850s, but reliance on mining has led to a boom-and-bust pattern of population and economic growth throughout the region. Economic development is hampered by limited infrastructure, the long distance to markets, long and cold winters, and a small and scattered resident population that to date has been able to provide only a limited range of goods and services to the mining industry.

The total population resident along Highway 37 is approximately 1,000, two-thirds of whom are Tahltan. The remoteness of Dease Lake, Iskut, Telegraph Creek and Stewart and the limited availability of employment opportunities have contributed to extensive out-migration of residents. Some of those remaining in northwestern B.C. are employed in public service, forestry or the resurgent mining industry, but until recently unemployment rates have been high. Employment remains subject to the vagaries of outside economic influences and decisions. These negative economic circumstances are exacerbated locally by a range of social issues including low education attainment, substance abuse, lack of or inadequate local infrastructure and services, family stress arising from spousal absences and lack of money management skills.

Many Tahltan see a long-term sustainable mining industry as providing the means to encourage and sustain their culture; they welcome economic development that will benefit the Tahltan people and culture, and provide an incentive to former residents to return to their home communities.

The TNDC was created through the collaborative efforts of the predominantly Iskut and Tahltan populations of Dease Lake, Iskut and Telegraph Creek. Representing the Iskut First Nation, the Tahltan Band and the TCC, the TNDC has evolved into a major local and regional employer and a force for Tahltan economic development through its own activities and through joint-venture relationships with other companies. It has established a range of long-term initiatives geared to increasing Tahltan employment, enhancing skill levels and ensuring sustainable economic livelihoods for greater numbers of Tahltan people.

Community-based issues are most likely to generate concern from government and the public, giving them a special place among the VECs. The TCC, elders, leaders and community members of the Tahltan and Iskut Bands, the District of Stewart, local governments of Smithers and Terrace as well as representatives of regional, provincial and federal governments were all consulted during the environmental assessment. Their feedback rendered a scope of issues ranging from employment and business development opportunities, substance abuse, highway traffic and accidents to the cultural implications of development and the ability of local jurisdictions to be opportunistic. A number of issues were considered as potential socio-economic VECs for the project, including broadly based community concerns, implications to the Tahltan Nation and delivery of services to people, as well as potential cause-effect relationships.

The Galore Creek Project will take place in the context of the imminent closure of two mines, a situation that requires significant consideration when weighing the project's socio-economic impact. The Eskay Creek and Huckleberry mines are scheduled to cease operations in 2007 and 2008 respectively, which will mean a loss of direct employment for 138 residents of Dease Lake, Iskut, Telegraph Creek and Stewart. The Red Chris mine is scheduled for start-up in 2007, Mt. Klappan mine in 2008 and Galore Creek mine in 2010. This activity prompted a July 2005 socio-economic overview assessment of prospective mine developments in northwestern B.C. The report concluded that:

- Most of the social concerns related to the new developments in the most affected communities exist presently and are not specific to mining development. Rather, they are characteristic of economic development problems experienced in many communities in the north.
- Iskut and Tahltan communities are having difficulty coping with the problems that bear some association with existing mine operations. There is evidence to suggest that existing education,

social and health programs and capacities are not adequate to meet present demands. The closure of Eskay mine and the opening of one or more of the proposed mines may increase the stress on the present communities.

- The potential socio-economic impacts of a large mine such as Galore Creek could provide stable employment, training and apprenticeship openings, and business supplier opportunities that would provide economic stability.
- The Tahltan and Iskut communities are small and their ability to participate in the mines is limited. Hence, the effect of one mining development may be significant, while the impact of a second or third mining development would be less so.

In the specific case of Galore Creek, there are several other pivotal socio-economic factors that will evolve in the region due to the presence of such a significant, long-term project. Many of these have been addressed through the Participation Agreement between Nova-Gold and the TCC – a document designed to minimize potential adverse impacts from the Galore Creek project while enhancing opportunities for the Tahltan.

25.4.6.1 Socio-economic Impacts

The physical, mental and social health of the region's residents is unavoidably intertwined with economic issues of development, employment and income. Healthy individuals are a prerequisite for healthy communities, but while the Galore Creek Project contributes to health by offering economic development, identifiable health issues, most notably substance abuse and related spousal and family violence, existed before NovaGold arrived. Health impacts associated with the Galore Creek Project give recognition to these issues while illustrating how best to manage prosperity.

Overall, the Galore Creek Project is expected to have a positive impact on the education of locally based employees through preparation programs, on-the-job training, apprenticeship programs and opportunities for career advancement. Training and apprenticeships leading to trade credentials are an investment in the future, contributing over the long term to develop education and expand the range of goods and services available from the region.

The preservation of the Tahltan culture is a stated priority of the TCC and is a top priority of NovaGold. It has been deemed integral to the individual and community health of the Tahltan, and both the Tahltan and NovaGold acknowledge the opportunity to reassert this culture through the project.

The construction and operation of the Galore Creek Project represents an opportunity for local businesses to acquire contracts to provide goods and services to NovaGold. NovaGold has established practices with local contractors relating to the notification of upcoming contracts, access to contracts through bidding assistance, bonding waivers and other practices.

Figure 25-4 illustrate the expected long term employment for the Galore Creek Project, classified in accordance with the type of positions available.

25.4.6.2 Long-term Employment at the Galore Creek Project

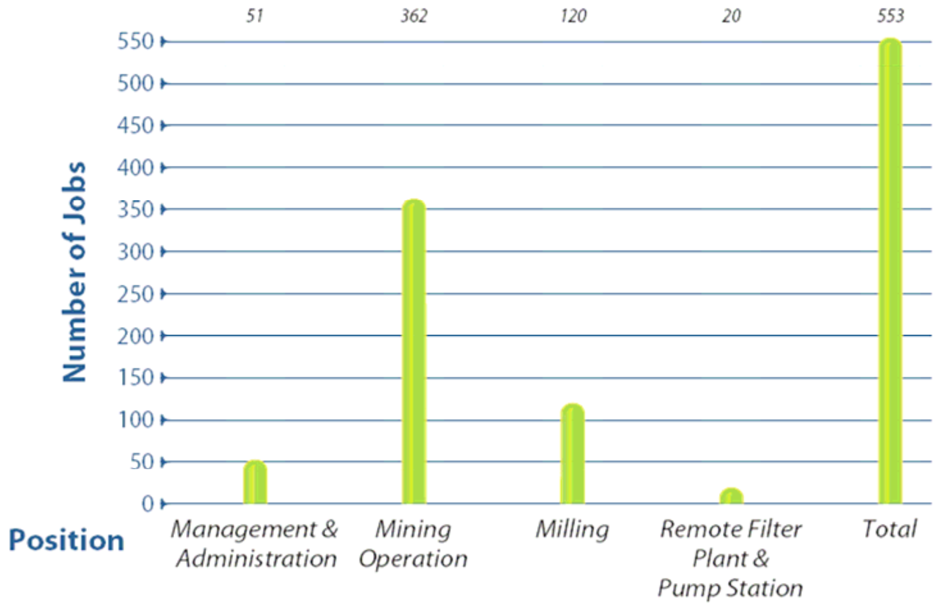


Figure 25-4: Long-Term Employment at the Galore Creek Project

25.4.7 Environmental Management, Monitoring and Follow-up

NovaGold has developed, and will continue to develop and adapt, comprehensive environmental management plans to avoid or mitigate adverse impacts of the Galore Creek Project facilities and activities during the construction, operation and closure of the mine. These plans have been developed using best management practices and scientific and engineering studies focused on site-specific conditions.

The Galore Creek Project has proposed a number of different monitoring systems, each of which has been designed to monitor sentinel environmental components to ensure the Environmental Management System (EMS) is functioning effectively. As part of the adaptive management process, the EMS will be updated and associated training programs enhanced to improve the level of environmental protection based on the results of these monitoring programs. Final details of these plans are being developed in anticipation of the start of construction in the second quarter of 2007.

25.4.7.1 Climate Change and Glacier Monitoring

The sensitivity of the Galore Creek Project to changing aspects of the environment and vice versa makes it important to monitor climate change within the Galore Creek Valley. Climate change monitoring will include collecting meteorological and hydrological data and comparison to regional norms, monitoring of changes to glaciated areas, as well as documenting greenhouse gas emissions from the project.

Figure 25-5 details the process whereby the Environmental Health and Safety Management System is designed.

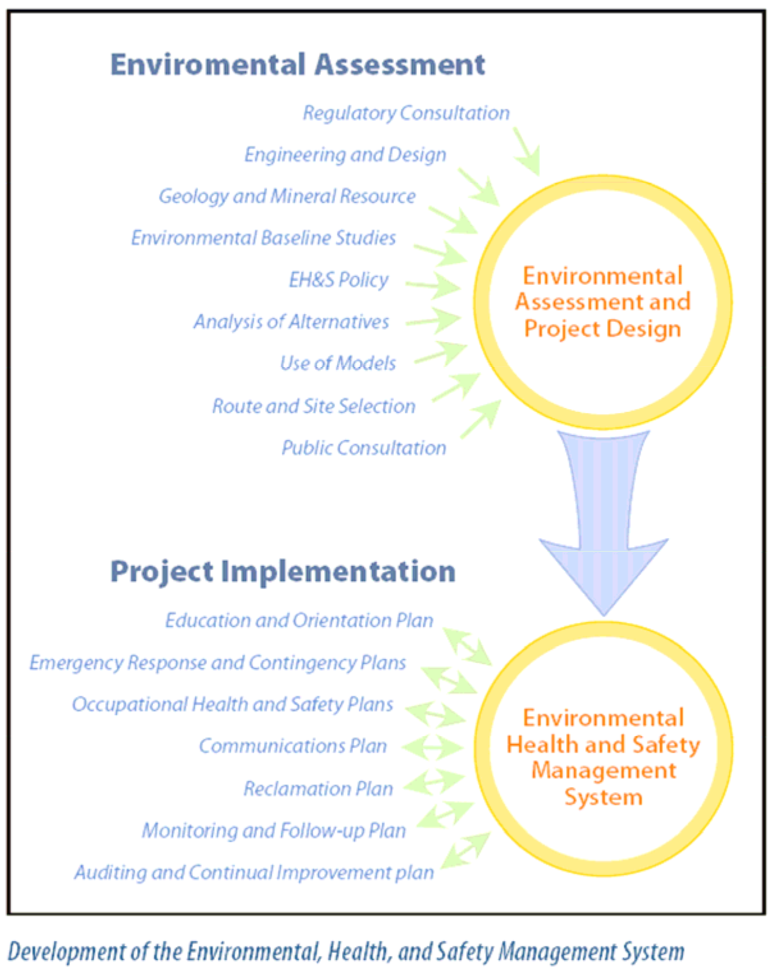


Figure 25-5: Development of the Environmental, Health, and Safety Management System

25.4.7.2 Air Quality Monitoring

Air quality monitoring is planned for both the ambient and workplace environments, although the air quality guidelines are different for each. The air quality monitoring for the ambient environment will consist of dustfall monitoring, whereas the air quality monitoring for the workplace will consist of control measures in the process plant and open pit mining areas.

25.4.7.3 Noise Monitoring

Noise monitoring in the workplace will consist of applying the best practical hearing protection equipment for the process plant and open pit mining work areas based on the maximum equivalent noise levels associated with each work area. The Galore Creek mine health and safety personnel will implement the noise monitoring program and conduct periodic spot checks to see that adequate hearing protection is being used.

25.4.7.4 Wildlife and Wildlife Habitat

A Wildlife Effects Monitoring Program (WEMP) will be implemented to document changes in wildlife abundance, behaviour, health and habitat resulting from project development, operation and closure.

Changes to wildlife will be documented to assess the success of mitigation measures proposed in the Wildlife Management Plan, identify opportunities for adaptive management and enable the actual wildlife effects to be compared with those predicted in the environmental assessment. This will allow the Wildlife Management Plan to be refined to foster management strategies that continue to minimize wildlife effects in the context of ongoing environmental change.

25.4.7.5 Project Components

The project components selected for monitoring are those with the greatest potential for interactions between project activities and wildlife. Monitoring of access roads, aerodrome and air traffic, transmission line, pit walls, waste, wildlife attractants, and the tailings facility will occur through all project phases (construction, operation and decommissioning).

Additional project components may be added to the monitoring program under a process of adaptive management if evidence emerges that the components pose a potential hazard to wildlife. Monitoring of mine components will be done through regular surveys conducted by environmental staff and by incidental reports from other mine employees and contractors.

25.4.7.6 Aquatic Effects Monitoring Program

A rigorous regimen of collection and testing procedures under the Aquatic Effects Monitoring Program (AEMP) will ensure regulatory compliance with respect to discharge limits and other criteria, as well as verification of predictions made in the environmental assessment. This testing will also serve to detect any unforeseen impacts as measured against the baseline established as part of the initial environmental assessment and identify cause-effect relationships between project activities and environmental impacts.

Cumulative effects are defined as changes to the environment that are caused by human projects and activities in combination with other past, present and future human actions. While the effects of an individual action may be relatively small, the effects of two or more actions may combine to produce cumulative effects that could be considered significant. For cumulative effects to occur, the action under review must have a residual effect on a Valued Ecosystem Component (VEC), and that VEC must also be affected by one or more other actions. When assessing cumulative effects, other projects that were considered included the Schaft Creek, Red Chris and Kutcho Creek metal mines, the Mount Klappan coal mine and the Forrest Kerr hydroelectric facility.

The residual effects of the Galore Creek Project on surface water quantity are limited to the Galore Creek and Scud River watersheds. There are no other existing or foreseeable human actions that could also affect water quantity in these watersheds.

On the matter of surface water quality, residual effects of the Galore Creek Project are limited to Galore Creek, a short section of the Scud River downstream of Galore Creek, and a confined section of the Iskut River downstream of the filter plant. There is potential for future mineral exploration and timber harvesting to also affect these water courses. Any mineral exploration will, however, likely have minimal influence on surface water quality, and any project to harvest timber will be required to implement sediment control measures. No residual cumulative effects are anticipated.

In addition to the effects discussed for surface water quality, the Galore Creek Project has the potential to affect aquatic resources and fish and fish habitat along the access road corridor. Aquatic resources and Dolly Varden char within the More Creek watershed could be affected by increased sedimentation as a result of future timber harvesting or development of access to the proposed Schaft Creek Project. Any future activity would, however, be required to implement erosion and sediment control plans. With this mitigation measure in place residual cumulative effects are not expected to be significant.

The Galore Creek project will have residual effects on forested, parkland and alpine ecosystems, occurring through the permanent loss of terrestrial ecosystems in the Galore Creek Valley and along the access road. There is also the possibility that invasive vegetation species will be introduced to the project area. Additional loss and disturbance of terrestrial ecosystems could occur as a consequence of future timber harvesting, mineral exploration, recreational activity, or development of access to the proposed Schaft Creek Project. With appropriate access controls and vegetation management procedures in place, residual cumulative effects will not affect the sustainability of terrestrial ecosystems.

Residual effects of the Galore Creek Project on wildlife will be limited to sensory disturbance of grizzly bear salmon feeding habitat and mountain goat feeding and natal habitat.

25.4.8 Cumulative Effects

The potential effects of fishing activities on salmon abundance could act in combination with the residual effects of the project to produce cumulative effects. Although the probability of occurrence and magnitude of these potential cumulative effects are unknown, the contribution of the Galore Creek Project to any cumulative effects are expected to be negligible.

Mountain goats in the project area may be affected by guide-outfitting, resident and First Nations harvest, and by disturbance from mineral exploration, recreational activities, and road access to Eskay Creek Mine, Forrest Kerr hydroelectric project and the proposed Schaft Creek Project. Any cumulative effects will, however, be restricted to localized areas and the sustainability of the mountain goat population will not be affected.

It is within the realm of socio-economic impacts where the Galore Creek Project will render the most tangible and visible positive results. Other projects that may be developed in a similar time-frame to Galore Creek include Red Chris, Mount Klappan Coal Project, Kutcho Creek and Schaft Creek. Together, these projects provide an unsurpassed opportunity for First Nations peoples to acquire the skills, education and training to enter sustainable employment, while facilitating negotiations with mining companies to provide a range of socio-economic mitigation measures that address issues beyond employment and business opportunities and that extend to broader community, capacity building, health and infrastructure.

By recognizing the employment policies presented in the NovaGold-Tahltan Participation Agreement, resident Tahltan and then non-resident Tahltan will be sought as priority employees for the Galore Creek Project before opportunities are opened to the general public. This competition for employment will cause a ripple in the labour market, which will likely to lead to the full employment of locally available Tahltan. Some abandonment of current employers will occur in favour of permanent employment at higher wages, and an influx of non-resident Tahltan could potentially be constrained by the lack of housing infrastructure and other services to accommodate them. This could also occur if a substantial number of non-Tahltan decide to return to their homeland to work.

The simultaneous development of a number of mining projects would lead to cumulative truck traffic and noise along Highways 37 and 37A and in the port of Stewart. This may result in adverse cumulative effects on non-development traffic, highway maintenance and the quality of life of residents in Stewart and along haul routes

The complex, and often unpredictable interactions and relationships between various socioeconomic factors make the significance of cumulative socio-economic effects challenging to assess. However, the successful management of both benefits and adverse impacts over the life cycles of projects could result in overall effects that are both positive and significant.

25.4.9 Alternatives

NovaGold's decisions regarding its proposed operational practices have been based on a considerable amount of evaluation of alternative scenarios. In each of the following cases, all viable options were explored to the depth that allowed for a reasonable decision about how best to approach further planning.

25.4.9.1 Access Roads

In early 2004, NovaGold commissioned a complete evaluation of seven access alternatives as part of the Galore Creek Project Scoping Study. The key factors considered during this short-listing process include safety and operability, environmental constraints, timeline on permits, trans-boundary issues, costs, power transmission and proximity to a deep-sea port. After careful consideration, the original northern route, modified to create efficiencies and satisfy environmental concerns raised by other access road options, was chosen.

25.4.9.2 Location of Site Infrastructure

The Galore Creek ore deposits are located in a very rugged and remote area. Siting of mine infrastructure must consider availability of sufficient flat ground for buildings, foundation conditions, safety from geohazards, proximity to ore zones and tailings, waste rock and low grade ore storage areas, and access for construction materials, personnel, operations consumables and for shipment of concentrates. Three primary site configurations were considered, with the final decision being made to locate principal project facilities in the Galore Creek Valley, with concentrate storage and pipeline water treatment facilities located nearer Highway 37.

25.4.9.3 Mining Method

The determination of mining method to be used on the Galore Creek Project came as a result of scrutiny of both open pit and underground approaches. It was deemed that although underground mining would produce less waste rock, the higher cost did not make it a viable choice; the preferred method was determined to be open pit.

25.4.9.4 Tailings Storage

The Galore Creek Project is expected to produce approximately 540 million tonnes of tailings over the 22-year life of the mine. NovaGold assessed eleven potential tailings storage sites, considering potential adverse environmental effects, geotechnical stability, volume capacity, distance from the mine and overall cost. Among the various alternative locations for the storage facilities, a tailings impoundment site located behind a cross valley dam in the Galore Creek Valley is the preferred alternative based on superior economics, smaller overall footprint, simpler monitoring and maintenance on closure and less dispersed receiving environments. NovaGold believes that it is feasible and reasonable to manage any and all potential environmental, economic and safety risks associated with this alternative.

25.4.9.5 Waste Rock Storage

The Galore Creek Project will require the disposal of about a billion tonnes of waste rock, produced mainly from five open pits with lesser volumes from numerous rock cuts for road and diversion channel construction and from the access tunnel. NovaGold has determined that the most effective alternative for disposal of waste rock from the Galore Creek Project is to store all reactive and potentially reactive waste rock under water and the rest in subaerial dumps adjacent to the tailings impoundment. The dam constructed to create the tailings impoundment will flood a large enough area to accommodate all of the reactive and potentially reactive waste rock.

25.4.9.6 Effluent Discharge Alternatives

There are two primary types of liquid waste from the Galore Creek Project mill: water discharged with the tailings slurry, and water discharged with the concentrate slurry. Tailings slurry water will be stored in the tailings impoundment and water that is not recycled to the mill will be released to Galore Creek downstream of the dam during natural high flow periods. It was decided that the preferred discharge location for effluent from the filter plant is a narrow reach of the Iskut River upstream of the mouth of the More Creek. This location will provide ample year-round dilution and good mixing with minimal in-stream construction. Fisheries impacts will be minimal and the proximity to the filter plant will keep the length of the necessary pipeline relatively short.

25.4.9.7 Water Supply

The Galore Creek mill will require over 6,600 cubic metres of water per hour for the grinding and flotation processes. Much of the water could be reclaimed and recycled to the plant process water system, but a significant volume of water will be lost to the tailings and concentrate slurries. Additional water will be required on an ongoing basis to make up process requirements. NovaGold investigated both groundwater and surface water sources of auxiliary water.

Water from pit area dewatering wells has become the preferred alternative for additional water for the mill. Surface water has better quality parameters and should be diverted around disturbed areas and the tailings impoundment so that it is available to maintain Galore Creek flows and support natural processes. Surface freshwater would be used only rarely, if at all, for tertiary make-up water for the mill.

25.4.9.8 Concentrate Management and Transportation Alternatives

The Galore Creek mine will produce up to 730,000 (average 480,000) tonnes of copper/ gold/silver concentrate per year, operating 24 hours per day and 365 days per year. It is expected that this concentrate will be marketed to offshore smelters, probably in Asia.

NovaGold has investigated several options to transport this concentrate to a port, including trucking of dewatered concentrate directly from the processing plant, from the terminus of a slurry pipeline located near Highway 37, or to the railhead at Kitwanga and then by train to Prince Rupert. The possibility of using a slurry pipeline to transport the concentrate from the processing plant directly to a port was also explored.

NovaGold proposes to construct a concentrate slurry pipeline along the alignment of the modified Northern route to a filter plant near Highway 37, to then be trucked to the existing port at Stewart.

25.4.9.9 Power Supply Alternatives

The Galore Creek Project will require a consistent and reliable source of electrical power of about 80 MW. NovaGold has examined a range of electrical power supply options to identify the most reliable and cost effective source with the least environmental impact. After careful consideration, it was determined that the best option with respect to economy, reliability, environmental impact and feasibility would be a connection to the provincial electricity grid at a point near Forrest Kerr power plant or Bob Quinn Lake.

25.4.10 Effects of the Environment on the Project

Just as a project can have effects on the environment, the environment can have effects on a project. Environmental impacts can occur as a result of minor events such as small windstorms, or from catastrophic events such as volcanic eruptions. The effects of the environment on a project are generally not simple or direct, such as trees blown down by a windstorm. They tend to be more complex in their effect – depending on the type of environmental occurrence, one or more components of the project could be affected, on a scale from minimal to extreme.

The discussion of possible consequences of environmental occurrences on the Galore Creek Project is rife with variables, yet every effort has been made to devise a plan to minimize their potential impact relative to the likelihood of their occurrence, their severity and the complexity of mitigation measures.

25.4.11 Extreme Weather

Extreme weather events could include droughts, storms, heat waves and cold snaps. As the Galore Creek Project area typically experiences high annual precipitation levels, a significant reduction in the accumulated annual rain and snowfall could produce several adverse scenarios. The most important of these is the effective management of tailings; the process requires a predictable and significant amount of water, and the absorption of discharged water is predicated on consistent water levels. In the event of such a drought, absorption can be managed by strategically pumping water at times of the year when the water quality and supply can accommodate the discharge.

25.4.11.1 Storms

Storm events can include rainstorms, thunderstorms, hailstorms, damaging winds, tornadoes and blizzards. In the case of rainstorms, severe precipitation could result in several million cubic metres of water being rapidly added to the Galore Creek catchment. The mine site water management strategy includes a design that can accommodate a massive influx of unpredicted water, and the diversion channel design includes five emergency overflow structures.

Thunderstorms, sometimes accompanied by hail and damaging winds, can cause flash flooding, damage building infrastructure, cause temporary blockages in the diversion channels and create unsafe working conditions. Lightning could cause forest fires under dry conditions, or damage infrastructure such as buildings and power lines.

Mitigation measures will rely heavily on the monitoring capabilities of the project. Weather forecasts will be monitored for advanced warning of incoming weather patterns to allow time for extreme storm preparation, such as securing buildings and equipment, mobilizing equipment, initiating maintenance procedures (snow or debris removal, repairing damage), and even shutting down the mill if necessary. To help mitigate the effects on all mine infrastructure (buildings, power poles, bridges, etc.) relevant emergency supplies will be stored at the site to facilitate timely repairs and reconstruction. Design and construction of the project's infrastructure are highly resistant to damage in these situations of inclement weather.

25.4.11.2 High and Low Temperature Extremes

Extended periods of higher temperatures could bring on heat waves and fewer frosts, decrease the amount of discontinuous permafrost and possibly trigger a wetter climate, which could in turn reduce the return period of severe flooding. Extended cold spells could result in more precipitation falling as snow than rain, thus increasing the amount of snow and ice to be managed at the mine site and along the access corridor. Both scenarios generate many issues related to water level, but also create challenges in maintaining the quality of life at times of extreme cold. Mitigation of water-related issues is consistent with other instances when water poses a problem for tailings management. Overall, extreme temperatures should not pose significant challenges for equipment operation because all equipment will be designed for these conditions.

25.4.11.3 Seismic Activity

The project site is located in a moderately high seismic zone. Although all project components could be affected by a seismic event, the stability of the tailings management structures poses the greatest concern. The downstream consequences of the failure of a tailings dam in the Galore Creek Valley are considered to be very high because of potential socio-economic, financial and environmental losses. This consequence remains for all stages of the life of the tailings dam: construction, operations and closure. Given this very high consequence, the main tailings dam and seepage recovery dams have been designed for the maximum credible earthquake.

25.4.11.4 Flooding

In the Coast Mountains, which includes the Galore Creek area, the largest floods are typically caused by rain falling on melting snow during freshet conditions in June or July, or during early winter in November and December, or because of heavy rainfall during September or October. Floods occurring along the access corridor could result in access road closures due to excess water on the road surface, erosion of the road surface or damage to stream crossings. Under the most extreme flood conditions there is the potential for drainage structure washouts (bridges, culverts and cross-drains) and pipeline ruptures.

The location of the access road was strategically selected so that less than five kilometres of the road runs through an active floodplain. The road maintenance program will anticipate and repair the consequences of flood events, and an extensive leak detection system will minimize loss of materials from any pipeline rupture.

25.4.11.5 Forest Fires

The number and size of forest fires in a region each year vary with annual weather, natural disturbance type (which reflects climate) and suppression effort. A safety plan will be developed for the Galore Creek Project, which will outline and describe appropriate procedures and protocols to effectively deal with hazards such as forest fires. The plan will address hazard evaluation, appropriate control procedures and protocols (including action levels), personal protective equipment to be used, air and water monitoring protocols and specifications, and detailed fire-fighting procedures.

25.4.11.6 Climate Change

Global observations suggest a number of climate trends during the 20th century, including increased average surface temperature, precipitation, frequency of heavy precipitation events and cloud cover, together with reductions in the length of the freeze season, the frequency of extreme low temperatures, and the extent of snow cover and mountain glaciers. These types of trends will affect the Galore Creek Project in a manner similar to many of the previously explained scenarios. It is good practice to exercise the same stringent observational practices to be able to anticipate climate change events, and plan the evolution of the project accordingly.

25.4.12 Accidents and Malfunctions

Planning stages of the Galore Creek Project required a detailed look at the potential for accidents and malfunctions to occur during the life of the mine across both construction and operation phases. This assessment included many possibilities, ranging from the most probable (a water treatment failure or a concentrate spill) to the least-likely (major damage from landslides, avalanches or volcanic activity). Each hypothesis began with the question of "what if?" Each hypothetical situation was then used as a starting point to work out all conceivable mitigation measures. The result was a bank of probability analyses to be used as a risk assessment approach that identifies the probability, potential magnitude and likelihood of accidents and/or malfunctions associated with various components of the project.

Three risk evaluations were completed to identify failure modes and effects. The first was a risk evaluation that included the identification of all potential technical, environmental, health and safety, cost, schedule, legal, regulatory and corporate reputation issues. The second focused on failures and possible environmental consequences associated with geotechnical failures of major mine structures during construction, operations and closure. The third risk evaluation addressed a catastrophic failure of the tailings dam. This analysis was consistent with typical practices for modeling catastrophic failures of large hydropower and water supply reservoir dams.

The Galore Creek Project entails many large-scale, interrelated components that must be built and operated in isolated, uncompromising terrain. By identifying and understanding the risks, controls can be established to ensure that, if they cannot be eliminated, the risks are at least managed. NovaGold's strategy with respect to risk is to identify risks of concern and, where elimination, avoidance or transfer is not possible, reduce the risk to as low as reasonably practicable. Then, it will apply due diligence by identifying and fully assessing all the material risks, taking appropriate measures to control them and ensuring that the justification for accepting the risk that remains is acceptable, leading to the development of reduction plans for identified major risks of concern. This includes making monetary provisions for remaining major risks, which are generally external risks beyond NovaGold's control, as a component of the project contingency.

To identify all risks associated with the construction and operation of the project, all potential technical performance, health and safety hazards, project delays and environmental hazards associated with performing the work, including any existing safeguards, were considered. It was also necessary to quantify the likelihood of occurrence and the severity of the consequences for each of the identified risks, and develop a risk response plan for each major risk.

25.4.13 Closure and Reclamation

Once the life of the Galore Creek Project is complete, the primary focus of management will be towards the reclamation and closure of the mine site. The key objective of this process will be to restore the equivalent capability of the site land so that it can once again be of significant use to wildlife, natural vegetation and human inhabitants of the area. These activities will be directed toward the development of appropriate and functional ecosystems, supported by appropriate soil handling procedures and strategies to reintroduce indigenous vegetation. The present timeline for operation of the mine will see closure activities begin in the year 2031 and extend over a two-year period, with annual monitoring conducted thereafter.

The scope of the site closure is dramatic; the government permitting process requires Nova-Gold to outline an exit strategy that details its plan, according to permit conditions, for the closure and reclamation of the Galore Creek Project. This plan has three main objectives. The first is the disassembly and disposal of the mine infrastructure, where all buildings not relevant to the closure process are disposed of in a manner consistent with government regulations. This includes a strict mandate to recycle where possible, and to remove facilities in a manner that enables the re-establishment of vegetation cover.

The second objective involves the contouring of the natural terrain. By landscaping the areas in and around the open pit – including access roads and other similar environments the Galore mine will be blended to more closely match its original aesthetic.

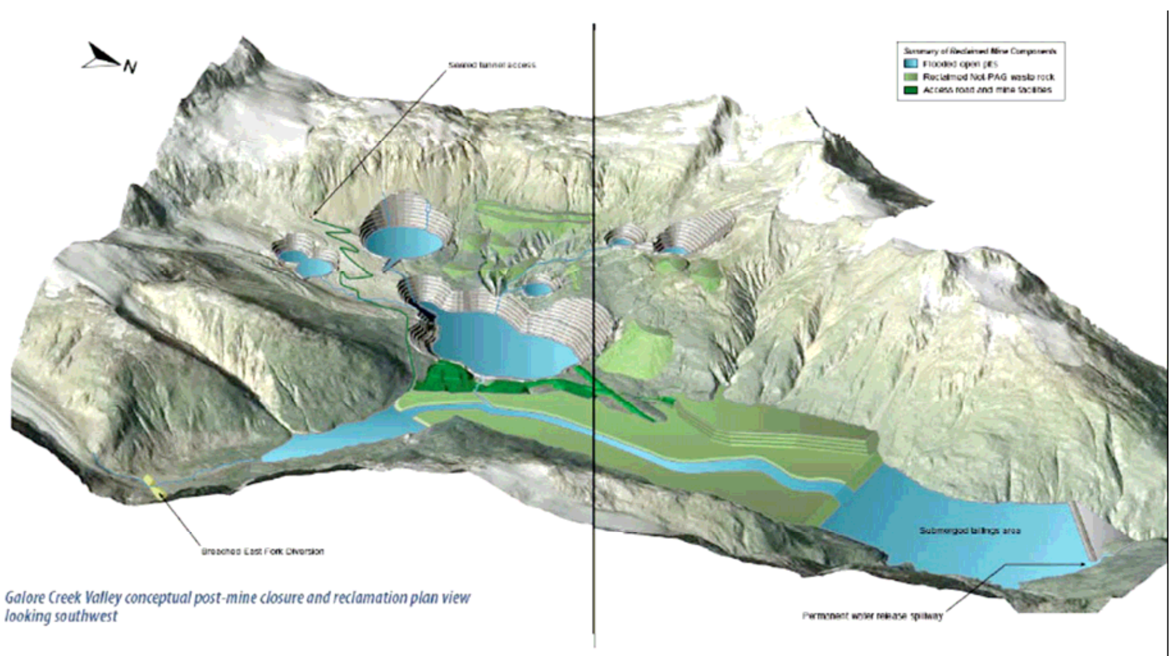


Figure 25-6: Conceptual Post-Mine Closure and Reclamation Plan View Looking Southwest

The final objective of this process pertains to the regional waterways that had been diverted for the purposes of the mine. These diversions will be decommissioned with the goal of re-establishing natural drainage patterns. The reintroduction of these natural patterns will then provide the area with long-term geotechnical stability. Open pits will be allowed to flood and overflow sequentially into one another with the re-establishment of a surface connection from the West Fork of Galore Creek to allow water movement into the main portion of the Galore Creek Valley.

NovaGold's commitment to the community is also considered in this plan. To address the impacts of mine closure on the people and communities of northwestern B.C., NovaGold and the Tahltan Nation will provide advance notification of impending closure and termination of employment, assist former employees as they pursue alternative employment, and, together with the Tahltan Nation, implement an employee assistance plan oriented toward developing successive employment.

NovaGold has also committed to the education, skill development and apprenticeship training of its employees, which will significantly enhance the employability and labour mobility of members of the Tahltan Nation beyond the life of the Galore Creek Project. Similarly, the capacity-building projects initiated during the life of the mine will allow alternative sources of employment to be generated locally, thus fulfilling the TNDC's objective to build a sustainable economic base for its people.

25.4.14 Commitments

During the short history of NovaGold, the company has demonstrated corporate commitment to community investment, good citizenship, environmental responsibility and economic development. This comes despite NovaGold's position as a relatively new company; the management team has extensive experience gained through former corporate employers. NovaGold is committed to operating and doing business in British Columbia in a socially and environmentally responsible manner, evident throughout all phases of NovaGold's various projects in western Canada and Alaska. The company is committed to the underlying principles of integrated environmental management and sustainable development.

NovaGold has made numerous corporate commitments for the Galore Creek Project, and the context of these commitments is evident throughout the environmental assessment document. They range from building relationships with the Tahltan Nation to how the company will manage the construction, operation, and closure of the mine. Many measures that will help to mitigate the effects of the Galore Creek Project on the environment are the direct result of decisions made by NovaGold early in the conceptual and design stages of the project.

NovaGold is committed to building long-term relationships with the communities in which it operates and recognizes and respects cultural and regional diversity. The successful development of a precedent-setting Participation Agreement with the Tahltan Nation is an example of NovaGold's commitment to the people in the Tahltan Traditional Territory. The inputs of the Tahltan Nation and other communities are critical to the success of the Galore Creek Project.

The fruits of this mutually beneficial relationship are many for the Tahltan. NovaGold is committed to hiring as many Tahltan and local northwestern British Columbians as possible as a first priority, with the goal to provide succession planning to help prepare and train them for a long-term career with NovaGold at the Galore Creek Project.

NovaGold is also committed to a strict program of risk reduction. This will provide a safe and healthy environment for all personnel employed by the Galore Creek Project. NovaGold will establish a Loss Control Policy, which will guide the company in health and safety matters through all phases of the project. Accidental losses will be controlled through best management practices including a drug and alcohol policy (dry camp) and systems, combined with the active participation of the workforce.

NovaGold's commitment to the concept of sustainable development, which requires balancing good environmental stewardship with economic growth, is consistent in every aspect of project planning. NovaGold has established an Environmental Policy to help guide the company in all phases of the Galore Creek Project. To help meet the Environmental Policy, NovaGold will continue to examine areas where the project can be improved. This includes a commitment to the continued application of traditional knowledge, to searching for alternative energy sources and further power reduction opportunities and to the use of an adaptive and constantly evolving management approach in developing the project's final closure plans.

Throughout the preliminary consultation process, the Tahltan Nation and other communities repeatedly stated their concerns that NovaGold respect the environment and work diligently to ensure that the air, water and land are protected. NovaGold has listened to these concerns, and has fully committed itself to construct and operate the Galore Creek Project in an environmentally and socially responsible manner.

NovaGold will develop a formal EMS that will meet the highest international standard: ISO 14001 (an International Organization for Standardization standard that represents an international consensus on the "state of the art" in a variety of areas). NovaGold will also abide by its own Environmental Policy, which requires it to operate in a manner that meets the expectations of the Tahltan Nation and other northwestern British Columbia communities, providing a consistent level of assurance that NovaGold is indeed "doing what it said it would do."

Once operations have begun, remediation and reclamation of the site will be required in accordance with federal and provincial regulations. Required bonding amounts generally increase with disturbed area during the course of mining operations and eventually bonding amounts are released as reclamation is successfully completed.

25.4.15 Conclusions

As an extremely important initiative for the province of British Columbia, the various complexities of the Galore Creek Project, a world-class deposit of copper, gold and silver, have been investigated in great detail over the past two years. The environmental baseline study, through its extensive consultation and wide-ranging consideration of all aspects of the natural environment, was second to none in British Columbia. NovaGold consulted with residents of the local communities and listened to their concerns. This research changed planning approaches in a number of different instances.

Through the project evaluation process, it became evident that all mining activity should be focused in the Galore Valley to reduce the size of the footprint and focus waste management in one area. This decision meant contending with other variables, such as building a 4.2 kilometre tunnel south to the Scotsimpson Valley, in addition to the special considerations made to manage waste and waste storage. Also, because the area is exposed to very high levels of precipitation that falls mostly as snow, a significant effort has been expended to divert as much natural runoff away from the waste rock/tailings facility as possible.

All the various components of the project have been subjected to risk evaluations to address and minimize the potential for accidents and malfunctions. A large number of construction and operations management plans have been developed to manage and minimize the effects of the project on the environment.

During construction, NovaGold plans to respect all identified fisheries and wildlife issues in order to minimize impacts to the environment. NovaGold has also maintained an excellent relationship with the Tahltan, having not only negotiated a precedent-setting Participation Agreement, but cooperated with the Tahltan community on preliminary research as well.

Given the level of scrutiny in the design and the regulatory review of the Galore Creek Project, the predicted impacts to the receiving environment should be negligible. Nova-Gold is making commitments to provide a further level of comfort to the authorities, regulators and indigenous people by keeping them at the highest level of involvement.

Although the mere presence of the project will have some negative effects, they are outweighed by the project's capacity to benefit the community. The group of professional consultants who have worked diligently on this project are collectively of the opinion that the Galore Creek Project can be built and operated in a safe manner, and can contribute to society while maintaining NovaGold's commitment to sustainability.

25.5 Description of Taxes

Refer to Project Economic Analysis found in Section 25.8 for a description of the tax bases for the project. The project Economic Analysis and Cashflow Analysis are based on these criteria.

25.6 Capital Cost

Section 25.6 was prepared by Mr. Bruce Rustad, P.Eng., Hatch

25.6.1 Summary

The capital costs for the Galore Creek Project including process plant, infrastructure and mining were estimated using 2nd quarter of 2006 Canadian dollars. This estimate includes no allowance for escalations in prices or fluctuations in exchange rates. The summary capital cost estimate for the Galore Creek Project is presented in Table 25-19 and reflects an intended level of accuracy of $^{+}$15% / $^{-}$10%.

Table 25-19: **Capital Cost Summary - Base Case, 65,000 tpd mill**

Description		Responsible Party	Estimate (CA$M)
Mine		GR Tech	457
Concentrator		Hatch	410
Water / Waste Management	Structure Water Recovery	Ledcor Hatch	247
Infrastructure	Tunnel Pipeline Road Power Line	HMM PSI Ledcor IHI	330
Total Directs			1,444
Indirect Costs		All	295
EPCM Costs		Hatch	151
Total Direct and Indirect			**1,890**
Contingency		Hatch	268
Owners Costs		Hatch	70
Total Project Estimate			**2,228**

Exclusions from the capital cost estimate are set out in Sections 24.6.5.

25.6.1.1 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 mine into production, as defined by a 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 (Hwy 37).
- 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-station and remote filter plant with concentrate load-out facilities.

- The construction of the 4.3 km long access tunnel to the minesite, including north and south portals, ventilation systems and refuge bays.
- No allowance for the refurbishment of a concentrate storage shed at Stewart as costs associated with this activity, if required, are assumed to be included in the unit cost charged by the owner of this facility.
- Engineering, procurement and construction management (EPCM) implementation strategy for the project.
- Construction indirects including the provision and operation of construction camps for the various sites, construction equipment, crew rotations, communications, supplies etc.
- Early-works (pre-access road) helicopter mobilization and support.
- Owner's costs to include the implementation team, training, etc.
- First fill of steel, reagents and supplies etc.
- Freight and initial spares.
- Contingency, assessed by Monte Carlo simulation, reflecting a P90 (10% chance of overrun).

25.6.2 Basis of Estimate

25.6.2.1 General

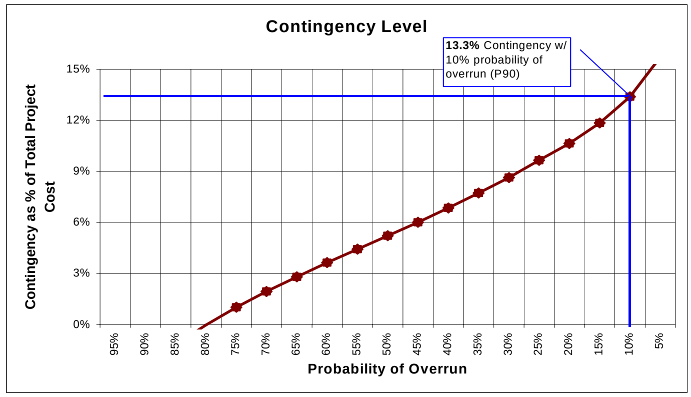
The capital cost estimate is based on the following project data:

- Process design criteria, developed from extensive metallurgical testwork conducted in 2005 and 2004 as well as review of previous testwork carried out in the period 1960 – 1992 and experience.
- Process flowsheets identifying all major unit operations.
- Sizing of all major equipment items.
- Plantsite layout and development of general arrangement drawings for the major facilities and buildings.
- Detailed mine design, mine scheduling and mine fleet sizing and selection.
- Three years of geotechnical study has been carried out in the Galore Valley. Areas of tailings/waste and water dams and structures have been investigated to a feasibility level. A conservative approach has been included in the estimate.
- 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 feasibility engineering design, field inspections, digital topographic maps and preliminary geotechnical investigations.
- Feasibility level engineering and cost study for the concentrate pipeline, based on site inspection, digital topographic maps and extensive previous experience from PSI.

This estimate has been prepared using a combination of quoted, estimated and factored costs to provide a level of accuracy consistent with feasibility level of engineering. All costs herein are shown in 2Q, 2006 Canadian dollars. The estimate includes no allowance for escalations in prices or fluctuations in exchange rate.

25.6.3 Contingency

Contingency included in the capital cost estimate is an allowance for normal and expected items of work which have to be performed within the defined scope of work covered by the estimate, but which could not be explicitly foreseen or described at the time the estimate was completed. The contingency amount is an intergral part of the cost estimate. It does not cover potential scope changes, price escalation, currency flucturations, allowances for force majeure or other project risk factors or any of the other items that are excluded from the capital cost estimate (see section 25.6.5). It should be assumed that the contingency will be spent.



The recommended contingency resulted in 13.3% of directs plus indirects for a P90, using the @Risk software.

The contingency level for a Feasibility Study will depend on the risk exposure that NovaGold considers appropriate. In order to graphically observe the trade-off between the contingency as a percentage of the total project costs and the project's probability of experiencing a cost overrun, the graph below has been included. In most cases a contingency of 80-90% probability of underrun (or 20-10% of overrun) is appropriate, Hatch recommends for this stand alone project a contingency of 90% probability of underrun, which in the case amounts to a CA$ 268 million in contingency (13.3% of directs plus indirects cost). See the following graph.

25.6.3.1 Contingency Calculation

The risk model was performed using the @Risk software, with 10,000 iterations of the model; on each of the model's iterations, the software obtains values for the subtotals by discipline and risk code. Some of the model's results are explained below:

Base Estimated Cost: This amount comes directly from the Project's Capital Cost Estimate, which in this case CA$M 1,890. (Direct + Indirect Costs).

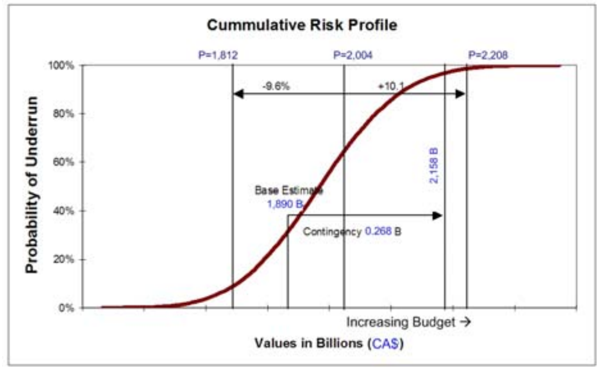
This base cost does not include owner's costs. Owner's contingency was considered separately and included within the owner's costs itself.

P5: This is the cost associated to a 5% probability of the project's cost being lower than the Base Estimated Cost plus Contingency (under run), in this case CA$M 1,812

P50: This is the cost associated to a 50% probability of cost under run/over run, in this case CA$M 2,004.

P90: This is the cost associated to a 90% probability of cost under run, in this case CA$M 2,158.

P95: This is the cost associated to a 95% probability of cost under run, in this case CA$M 2,208. The graph below shows the risk distribution of the Capital Cost Estimate for the Project as an ascending cumulative curve, it's used to graphically analyze the precision level of the estimate.



In this case the confidence level of 90% from the P50 can be observed as the interval between the values P5 and P95. For this estimate and based on the level of engineering and the quality of information used, we can conclude that with a 90% confidence level for the estimate, the precision falls in between a $^{-}$9.6% / $^{+}$10.1%, and that the contingency represents 13.3% of directs plus indirect cost.

25.6.4 Capital Cost Assumptions

The following assumptions have been made in the preparation of the capital cost estimate.

- The permanent camp for both the mine and the plant will be modifications to part of the construction camps. These modification costs have been included in the direct portion.
- Sufficient earthworks material can be mined out of the optimal pit design to meet the earth structure specifications associated with the water diversion requirements inside the valley.

25.6.5 Capital Cost Exclusions

Costs related to the following are excluded from the capital cost estimate:

- All facilities not identified in the Summary Description of the Project (as set out in Section 3).
- Any environmental, archaeological or ecological considerations, other than those expressly included in the current design.
- Any costs incurred in connection with upgrades to BC Hydro transmission lines up to the location of the Galore Creek tie-in at Bob Quinn.
- Costs incurred to accelerate the work (e.g., overtime charges, expediting charges, etc.).
- Sunk costs – being any costs incurred in connection with the project prior to the commencement of the construction phase of the Project. (e.g., site acquisition costs, costs associated with the preparation of this study and any prior studies, licensing and royalty charges already incurred).
- Any licensing or permitting costs, including any environmental permitting costs.
- Any costs incurred to acquire real property or rights-of-way that are required to build the off-site infrastructure.
- Project financing costs, whether done by way of debt or equity and including any interest costs.
- Any currency exchange fluctuations relative to the exchange rates used for the purposes of this estimate.
- Any price escalation after second quarter 2006.

- Costs incurred to dispose of existing hazardous substances, including any toxic or contaminated materials.
- Costs that may be incurred as a result of the failure of the Owner to follow an EPCM project implementation approach.
- Costs of the work that has been recommended but which shall be conducted at the option of the Owner.
- Fees or royalties relating to use of certain technologies or processes
- Value added tax.
- Credits for salvage value of any demolition, modification work, residual construction materials, vehicles and temporary buildings.

In addition, no allowance has been made in the capital cost estimate for any of the following risk factors:

- the project risk factors that would be expected to potentially impact any project such as this Project (e.g., adverse weather conditions beyond those described in this report, acts of god and other force majeure events, delays due unforeseen factors such late delivery or unavailability of equipment or materials or unavailability of labour resources, poor performance by EPCM contractors or construction contractors, disputes with local residents, etc.);
- the specific project risk factors identified in Section 21.6; or
- political, legal or regulatory risk factors (e.g., changes to laws, expropriation, changes in taxation or royalty regimes or non-issuance, cancellation or revocation of permits or licenses required to develop and operate the Project).

The occurrence of any such risk factors may have a material impact on the accuracy of the capital cost estimate.

25.6.6 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 below.

Table 25-20: Sustaining Capital Schedule (CA$K)

Year	Mill Sustaining Capital CA$K	Mining Sustaining Capital CA$K	Working Capital CA$K	Mine Replacement Capital CA$K	Ongoing Dam Building (Haulage in Mine Costs) CA$K	Total CA$K
1	-	30,247	61,728	-	3,704	
2	2,469	-	-	-	3,704	
3	-	3,086	-	617	1,235	
4	-	-	-	1,235	1,235	
5	4,938	617	-	617	-	
6	-	-	-	5,556	-	
7	-	4,938	-	617	1,235	
8	6,173	-	-	617	-	
9	6,173	-	-	1,852	-	
10	-	-	-	617	3,704	
11	6,173	-	-	617	-	
12	-	-	-	15,432	-	
13	6,173	-	-	4,321	3,704	
14	-	-	-	617	-	
15	6,173	-	-	3,086	-	
16	-	-	-	-	-	
17	-	-	-	617	-	
18	6,173	-	-	-	-	
19	-	-	-	-	-	
20	6,173	-	-	-	-	
21	-	-	-	-	-	
22	-	-	61,728	-	6,173	
Total	**50,617**	**38,889**	**-**	**36,420**	**24,691**	**150.617**

25.7 Operating Cost

Section 25.7 was prepared by Mr. Bruce Rustad, P.Eng., Hatch

25.7.1 Summary

The operating cost for the Galore Creek Project including mining, general and administrative and process costs have been estimated using 2nd Quarter of 2006 Canadian dollars. These include no allowances for escalation in prices or exchange rate fluctuations. The summary operating cost estimate is presented in Table 25-21 and reflect an intended level of accuracy of +15%/-10%.

GR Technical estimated the mining operating costs, PSI estimated the operating costs associated with the concentrate/diesel pipelines while Hatch estimated the operating costs of the remaining facilities including the concentrator, tailings and reclaim pumping, site services and the facilities at Bob Quinn. The experienced operators assisted with the establishment of the G&A Labour and Fixed costs.

Table 25-21: Operating Cost Summary

Summary	Area	Responsible Party	CA$/t ore
G&A	G&A Labour	**Hatch**	0.20
	Fixed		0.79
	Total G&A		**0.99**
Mining	**Life of Mine Average**	**GR Tech**	**3.61**
Plant & Infrastructure	Labour	**Hatch**	0.41
	Consumables		1.62
	Maintenance		0.27
	Fuel		0.03
	* Power – Rest of plant site		0.39
	* Power - Mills		0.87
	Total Plant & Infrastructure		**3.59**
Concentrate Filter Plant (Incl. Concentrate/Diesel Pipelines & Utilities)	Labour	**Hatch**	0.09
	Consumables		0.03
	Maintenance		0.02
	Pipeline Maintenance	**PSI**	0.02
	Total Concentrate Filter Plant		**0.17**
Total Mine-site	**Total Mine-site**		**8.36**
Concentrate Transport and Storage	Concentrate Transport	**Hatch**	0.57
	Port loading and storage		0.19
	Total Concentrate Handling		**0.76**
Total Project			**9.12**

* Note: Study power rate at CA$ 0.0524/kWh (including wheeling charges) provided by W.N. Brazier Associates Inc. (July 2006)

25.7.2 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:

- General and administration costs for the operations including road, tunnel and powerline maintenance, crew rotation by air and property taxes.
- Mining labour and consumables for operations, maintenance and technical support services
- Fuel costs for mining, milling and concentrate dewatering activities
- Process labour and consumables for operations and maintenance for crushing, grinding, flotation and concentrate dewatering
- Concentrate/fuel pumping and pipeline operations
- Site water management, including effluent pumping from the tailings dam to discharge
- Filtrate water treatment costs, prior to discharge at the concentrate dewatering plant
- Supply of electrical power, including demand charges, an estimate for line losses and an estimate for wheeling charges applicable to the use of CMPC's transmission line to BCH's point of service at Meziadin.

The scope of the estimate does not include:

- Escalation beyond the 2nd Quarter of 2006 for labour or material prices
- Exchange rate fluctuations

In addition, each of the exclusions to the capital cost estimate (as set out in Section 25.6.5) are generally also excluded from the operating cost estimate. Any costs incurred in connection with any such items could have a material impact on the accuracy of the operating cost estimate.

25.7.3 Basis of Estimate

Unit costs for consumable and labour rates were estimated from sources listed below while the magnitude of consumables and labour required are determined for each specific activity as outlined in Section 25.7 from experience and first principles.

The unit costs are based on the following data and were considered in connection with the Feasibility Study:

- Salaries 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. For hourly employees, the labour rates for 2006 from other mines in Northwestern BC were used.
- Budgetary quotations for chartered air services for crew rotation, camp costs, concentrate transportation costs and concentrate storage costs at the port of Stewart.
- Mine designs to determine the size and makeup of the mine fleet as well as fuel requirements which is affected by distance and topography
- Budgetary quotations, including freight, for all consumables, including grinding media, reagents, tires and fuel. Fuel is estimated at a delivered cost to site of CA$ 0.80 /L.
- Power costs were estimated as the sum of consumption charges, demand charges and wheeling charges and estimated at CA$ 0.0524/kWh by W.N. Brazier Associates.
- Capital maintenance costs were considered to be a percentage (5%) of the purchased equipment cost.
- Daily camp costs were obtained from the projected 2007 rates for the current Galore Creek camp.
- Travel costs for shift changes are based on Air charter travel assuming flights are from Vancouver to Bob Quinn via Terrace and Smithers using a Beach 1900 aircraft, which can carry a maximum of eighteen passengers.

25.7.4 Operating Requirements

25.7.4.1 Labour

The G&A costs cover the labour requirements including senior technical positions as well as human resources, safety and environment positions. The G&A costs also include fixed costs which are not allocated to a specific operation and include: personnel transport, property taxes, business travel, road maintenance, avalanche control, snow removal, environmental monitoring, insurance , warehouse, security etc.

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.

The plant labour schedule is based on two twelve hour shifts per day and a two week in/two week out operation.

Labour requirements for the mill are based on experience from other similar size operations in Canada and include a total of 30 salaried staff as well as 112 hourly workers which include operators, maintenance, site services and material management personnel.

Labour requirements were estimated from experience of other similar size operations in Canada. Table 25-22 lists the labour requirements.

Table 25-22: Labour Requirements

Area	Function	Number
G&A	Administration	14
	Information Systems	3
	Material Management	10
	Human Resource	5
	Safety	14
	Environmental	7
	S/Total G&A	**53**
Process	Staff	30
	Operations	48
	Maintenance Services	46
	Site Services	8
	Warehouse	6
	Security	4
	S/Total Process	**142**
Mine	Operations	175
	Maintenance	99
	Technical	31
	S/Total Mining	**305**
Total Direct Mine Operations		**500**

25.7.4.2 G&A Fixed Costs

Table 25-23 outlines the G&A fixed costs broken down by specific cost areas for the Galore Creek Project.

Table 25-23: G & A Fixed Cost Summary (CA$/t ore)

Cost Area	Annual CA$	CA$/t
Property Taxes	1,450,000	0.061
Business Travel	100,000	0.004
Road Maintenance & Repairs including snow and avalanche debris removal	$1,761,100	0.074
Avalanche Control Costs	$650,000	0.084
Tunnel Operations and maintenance	160,000	0.007
Powerline Maintenance	215,100	0.009
Galore Valley Camp costs	3,892,725	0.164
Bob Quinn Camp costs	213,525	0.009
Crew Air Transport Composite Cost	5,187,000	0.219
Crew Bus Travel	487,500	0.021
Small Vehicles	70,000	0.003
Mobile Equipment Rentals	60,000	0.003
Safety Training Supplies	112,000	0.005
First Aid Supplies	50,000	0.002
Janitorial Supplies	40,000	0.002
Communications	56,000	0.002
Consultants - Allowance	300,000	0.013
Regulatory compliance - Allowance	150,000	0.006
Legal fees - Allowance	160,000	0.007
Recruiting/Relocation	250,000	0.011
Insurance	350,000	0.015
Environmental monitoring	3,000,000	0.051
Community Relations	100,000	0.004
Total	**18,814,950**	**0.79**

Property taxes and other cost elements are commensurate with those for other central BC mines. Environmental monitoring costs have been estimated by Rescan and include Federal MMER monitoring requirements, fisheries compensation monitoring, BC permit compliance monitoring and environmental effects, reclamation research and monitoring as well as the cost of labour and transportation to site Road maintenance costs were estimated by McElhanney, the company that is engineering the access the road.

The majority of the G&A fixed costs are associated with camp costs including transportation of crews to site. Camp levels are based on housing the combined labour requirements for all of the areas as shown above with people being housed in main camp in the Galore Valley and thirteen at the Bob Quinn facility. Other general costs such as first aid supplies and legal fees were estimated by the client NovaGold Resources Inc.

Communications includes the cost of maintaining satellite system for television use and for the telephone and internet systems.

25.7.4.3 Mining Operating Costs

Mine capital and operating costs are derived from a combination of supplier quotes and historical data collected by GR Tech. 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 for Sustaining and Replacement Capital.

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 using CAT's FPC simulation program. 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 consumables consumption rates and unit operating costs to calculate the total equipment operating costs for each time period.

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 SABREX blasting study conducted by ORICA for Galore Creek and historical blasting cost. Geotechnical costs are based on historical data collected by GR Tech.

The mine operating costs have been broken down into three components in the Production and cost schedules. These are: G&A Labour, Operating Labour, and Equipment Operating costs.

Labour factors in Man Hours/operating hour were also assigned to each of the equipment. Labour costs were calculated by multiplying the labour factor by the equipment operating hours, and labour costs are allocated to the equipment where labour has been assigned. The total hours required for each job type on all the equipment are summated and any additional labour required to completed a crew is assigned to unallocated labour. Some trades in Mine Operations (Grader Operator, Track Dozer Operator, Scraper Operator, Crusher Operator, Water Truck Operator, and Fuel Truck Operator) and Mine Maintenance (Crane Operator, Welder, Tireman, Labourer and Service man) are treated as shared labour during the unallocated labour assignment and labour contingents of these are therefore not rounded off in the tables below. The mine hourly and salaried labour schedules are listed in the tables below.

Unit operating cost for the mine up to year 5 and LOM are listed in the tables below. Complete mine cost tables including mine capital and operating cost schedules were considered in the Feasibility Study.

Table 25-24: Mining Costs per Tonne Moved (CA$/t moved)

Cost / Tonne Material *Moved*	Year-3	Year-2	Year-1	Year1	Year2	Year3	Year4	Year5	LOM
	Preproduction			Operations					
Drilling	$-	$0.02	$0.02	$0.04	$0.04	$0.04	$0.04	$0.04	$0.04
Blasting	$0.28	$0.17	$0.17	$0.15	$0.14	$0.15	$0.16	$0.16	$0.15
Loading	$0.50	$0.18	$0.18	$0.12	$0.11	$0.15	$0.15	$0.12	$0.15
Hauling	$0.93	$0.91	$0.87	$0.72	$0.73	$0.56	$0.66	$0.69	$0.70
Mine Maintenance	$0.03	$0.02	$0.01	$0.01	$0.01	$0.01	$0.01	$0.01	$0.01
Mine Ops - Support	$-	$0.29	$0.22	$0.14	$0.14	$0.14	$0.13	$0.14	$0.16
Snow Removal	$-	$0.05	$0.04	$0.02	$0.02	$0.02	$0.02	$0.02	$0.02
Geotech	$0.07	$0.02	$0.01	$0.01	$0.01	$0.01	$0.01	$0.01	$0.01
Unallocated Labour	$0.62	$0.03	$0.03	$0.02	$0.01	$0.01	$0.01	$0.01	$0.02
Direct Costs - Subtotals	$2.43	$1.68	$1.55	$1.23	$1.21	$1.09	$1.19	$1.19	$1.27
Mine Ops G&A - $'s	$-	$0.04	$0.03	$0.02	$0.02	$0.02	$0.02	$0.02	$0.03
Mine Maintenance G&A - $'s	$-	$0.04	$0.03	$0.03	$0.03	$0.02	$0.02	$0.02	$0.03
Mine Engineering G&A	$-	$0.03	$0.02	$0.02	$0.02	$0.01	$0.01	$0.01	$0.02
Technical Services G&A	$-	$0.03	$0.02	$0.01	$0.01	$0.01	$0.01	$0.01	$0.01
Total GME Costs	$-	$0.14	$0.10	$0.08	$0.08	$0.07	$0.07	$0.07	$0.09
Total Operating Cost	**$2.43**	**$1.82**	**$1.65**	**$1.32**	**$1.28**	**$1.16**	**$1.26**	**$1.26**	**$1.35**

25.7.4.4 Fuel Consumption

Fuel consumption rates are estimated in the mine schedule for each equipment type. These consumption rates are applied to the operating hours of the equipment to estimate the total fuel consumption. Fuel costs have been included in the unit operating costs estimated above.

Explosive factory fuel consumption is estimated based on the quantity of explosives used, and an estimated 40 litres diesel fuel consumed per tonne of explosives.

Fuel quantities scheduled for until first five years of milling are shown in the table below.

Table 25-25: Mine Fuel Consumption Schedule

Fuel Consumption (m^3)	Year-3	Year-2	Year-1	Year1	Year2	Year3	Year4	Year5
Drilling	-	13	13	492	558	700	751	624
Blasting (Explosives Factory)	31	464	591	814	818	1,016	1,042	981
Loading	571	4,532	4,418	2,080	2,914	3,962	3,711	3,616
Hauling	1,156	21,566	27,314	34,071	33,953	29,868	34,351	33,775
Mine Maintenance	-	139	139	139	139	148	139	139
Mine Operations - Support	-	5,270	5,270	5,225	5,225	5,715	5,341	5,317
Snow Removal	-	1,245	1,245	904	904	954	904	904
Total	**1,759**	**33,229**	**38,990**	**43,724**	**44,511**	**42,365**	**46,238**	**45,356**

25.7.4.5 Plant & Infrastructure

The plant and infrastructure operations are associated with the crushing plant, concentrator, tailings pumping, water reclaim, they do not include operations associated with the Bob Quinn filter plant, concentrate transportation nor concentrate pipeline maintenance (these are included in filter plant operations). The plant and infrastructure costs comprise of labour, consumables such as grinding media and reagents, maintenance (labour and material) and power and fuel costs for service vehicles.

25.7.4.6 Consumables

The quantity of consumables required for the operation of the mill is based on testwork, historical data and vendor information.

A study was done on grinding media consumption which comprises of about 50% of the overall consumables costs. Historical grinding media consumption data from several mines in BC was obtained from literature for operations which mined porphyry copper ore. Grinding media consumption was also estimated from testwork done by SGS Minnovex this year to determine the abrasion index. Estimates from Historical data gave consumptions of 0.09 lb/kWh for the SAG mill and 0.121 lb/kWh for the ball mill, while the testwork estimated 0.200 lb/kWh for both SAG and Ball mills. Details of this can be found in the design criteria in Section 20.1.2. In general, the grinding media consumption based on the abrasion gives a rate much higher than observed in industry. It was decided based on experience that the consumption based on historical data is more accurate and will be carried for this estimate.

Liner consumption was based on experience for other similar operations and it is assumed that the SAG will under go relining once per year while the two ball mills will be relined every two years. Details for the costs of relining the crushers, both gyratory and cone, were considered in the Feasibility Study.

Reagent consumption for the mill is based on locked cycle testwork done by G&T Metallurgical Inc. reported in their 2006 report. The consumption rates of PAX, lime, MIBC and flocculant will be consistent during the life of mine.

The consumption rate of the carboxymethyl cellulose, a guargum based reagent (which is used to depress talc-like materials which can have adverse impact on flotation responses) is expected to fluctuate significantly throughout the life of mine. An average consumption rate of 10g/t mill feed is carried in this estimate based on the minor amount of testwork done. Further testwork is required to more accurately predict the consumption rates of the guargum based reagent over the project life.

Table 25-26: Consumables Cost Summary

Consumables	Consumption			Supply Cost, CA$/t	Freight Cost, CA$/t	Total Cost CA$/y	Operating Cost, CA$/t ore	Remarks
	Units	Rate	tpy					
Steel			**20,192**			**29,286,448**	**1.23**	
Primary crusher liners	each			250,824		250,824	0.01	2.5 lower mantles,1 upper mantle, 1 set of concaves/year
SAG mill liners	kg/t ore	0.033	N/A	3,480,000		3,480,000	0.15	2 liner changes/year.Based on FFE quote
Cone crusher liners	each			97,600		97,600	0.004	Based on Metso schedule of liner changes
Ball mill liners - both mills	kg/t ore	0.043	N/A	1,838,000		1,838,000	0.08	1 liner change/year/ball mill
Regrind mill liners (3 units)				662,484		662,484	0.03	Based on Metso schedule of liner changes
SAG mill balls, 125 mm	kg/kWh	0.040	6,307	1,030	150	7,442,496	0.31	based on average power draw of 18000 kW at shell
Ball mill balls, 76.2 mm - both mills	kg/kWh	0.055	12,267	950	150	13,493,291	0.57	based on average power draw of 12200kW at shell
Regrind mill balls, 25 mm	kg/kWh	0.055	1,617	1,100	150	2,021,753	0.09	unit consumption based on kg/kWh of Primary Grind

Consumables	Consumption			Supply Cost, CA$/t	Freight Cost, CA$/t	Total Cost CA$/y	Operating Cost, CA$/t ore	Remarks
	Units	Rate	tpy					
Reagents								
Collectors -PAX	g/t ore	14	332	2,950	250	1,062,880	0.04	based on quadra chemicals - 1000kg bulk bag
Collectors -3418A	g/t ore	0	-	6,750	250	-	-	X550 is offset - 200kg drum - Will no initially be used
Frother - MIBC	g/t ore	38	902	2,800	150	2,659,573	0.11	bulk delivery - 20,000kg/tanker
Dispersant - Omnicel 106	g/t ore	10	237	1,900	250	510,088	0.02	offset of PE 29 - 1000kg bulk bags; used 10% of time
Flocculant-Cytec A-130	g/t ore	7.5	177	5,000	250	929,522	0.04	25kg bags - quadra chemicals
Quick lime - pulverized	g/t conc.	540	12,812	168	150	4,074,057	0.17	$1680/t - FOB ~ Kamploops - Graymont
Reagents - Filter plant			**75**			**388,828**	**0.02**	
Flocculant-Cytec A-130	g/t conc.	14	8	5,000	250	40,672	0.002	25kg bags - quadra chemicals
Hydrated Lime	g/t conc.	82.9	45	400	250	18,155	0.001	750kg bulk bag - univar canada
Sulphuric acid	g/t conc.	40.2	22	400	250	250,000	0.01	1740kg totes - univar canada quote
Assay/Met consumables+spares for 1 year						80,000	0.003	From Anachemia quote for Wolverine - 2006

25.7.4.6.1 Maintenance Supplies

The maintenance supplies cost is estimated at 5% of the purchased equipment cost. The labour costs associated with maintenance have been captured in the labour portion of the estimate.

25.7.4.7 Power

Table 25-27 contains a summary of the power cost estimates for the Galore Creek project.

Table 25-27: Power Cost Summary

Area	Description	Operating Power, kW (at shaft)		Energy Consumption, MWh/y	Energy cost CA$/y
		Normal Consumed	Peak Consumed		
200	Mining (including dewatering)	2,473	4,971	23,047	1,207,653
300	Ancillary Buildings/Facilities/ Fuel/Truckshop	3,339	3,608	31,116	1,630,493
400	Primary Crushing - crusher to SAG Mill	1,676	2,220	15,620	818,506
500	Process (excluding milling)	11,029	12,206	102,784	5,385,883
510	SAG	16,560	21,200	154,325	8,086,636
510	BALL - 2 Mills	22,448	30,000	209,196	10,961,884
512	VERTIMILL - 3 Mills	3,088	3,088	28,782	1,508,158
600	Tailings/Diversions/Water Supply	2,211	7,703	20,607	1,079,818
800	Concentrate/Fuel Pipelines/Road/Tunnel/Airstrip	853	964	7,946	416,353
900	Filtration Plant (Bob Quinn)	759	911	7,074	370,702
Total Project		**64,437**	**86,872**	**600,498**	**31,466,086**

The power requirements are based on the average power consumed by each piece of equipment and is based on a consideration of the load. This average power takes into account the availability (for example in the case of the concentrator is 92%), a demand factor (what percentage of the installed motor power is drawn during normal operation) as well as diversity factor which examines the loads on a seasonal basis.

The major consumer of power is the concentrator which accounts for 53 MW (75%) of the average consumed power of the entire operation. Of this 53 MW, 42 MW is consumed by the six mills – the SAG mill, two ball mills and three vertimills. The average consumed power for the SAG and ball mills was determined by testwork to be that power which is required to grind the full tonnage at the average ore hardness of 16.5 kWh/t and estimated to be 18 MW at the shell of the SAG mill and 12.2 MW at the shell of both ball mills. The installed mill power has been designed to handle the full tonnage of 2944 t/h (92% availability) on all but 5% of the hardest ore. It has been assumed that the vertimills will run at 100% of their rated capacity.

Pump power requirements (accounting for ~13% of average consumed power) have been estimated by Hatch by taking into account elevation differences and friction losses in the pipeline. Pump vendors were asked to calculate the power requirements based on this data and the values were compared for correctness.

25.7.4.8 Concentrate Handling

Concentrate handling includes all activities from pumping of the concentrate slurry from the concentrator to the filter plant at Bob Quinn to the storage of the concentrate cake at the port of Stewart.

25.7.4.8.1 Concentrate Filter Plant

The concentrate filter plant site at Bob Quinn includes the processes for concentrate dewatering and water treatment. Manpower requirements have been estimated from experience, consumable consumption from testwork and power requirements are based on vendor data and first principles.

25.7.4.8.2 Concentrate/Diesel Pipeline & Utility

The power requirements for pumping the concentrate slurry were estimated by PSI to be 825 kW based on the rheology of the slurry determined from testwork and the profile of the pipeline. The manpower was estimated by PSI .

25.7.4.8.3 Concentrate Transport & Storage

Estimated concentrate production over the life of mine has been calculated producing a variable production from 2000 tpd in year three to 850 tpd in year 12. This production rate is dependant mainly on the copper grade being mined and thus varies significantly. No backhaul credits were included at this time, although this represents an opportunity. Unit rates for concentrate haulage by truck were obtained from a number of haulage contractors with this experience. An average haulage rate of CA$30/wmt was used, and applied to the total haulage distance from the minesite to the port of Stewart. Loading and storage costs of CA$10.00/wmt of concentrate have been included in the cashflow forecast model.

25.8 Economic Analysis

Section 25.8 was prepared by Mr. Bruce Rustad, P.Eng., Hatch

25.8.1 Summary

The results indicate that the Galore Creek project is most sensitive to changes in the US$/CA$ exchange rate and copper prices, followed by, operating and capital costs.

Capital and operating costs were developed for a nominal 65,000 tpd operation and cashflow forecasts generated to evaluate the economic performance of the Project, testing the sensitivity to metal prices, exchange rates and input prices. The base case price assumptions are US$1.50/lb Cu, US$ 525/oz Au, US$8.00/oz Ag and a US$ / CA$ exchange rate of 0.81/1.00. An analysis has also been performed using spot prices, a three year trailing price and a low metals price case.

Table 25-28: Economic Model Case Summary

Economic Results	Units	Base Case	Three Yr. Trailing Ave.	Spot Case (Sept 1, 06)	Low Case
Mine Basis					
Mine Life	years	22			
Ore Tonnage milled	Mt	522			
Strip Ratio		1.64			
Mill throughput (nominal)	tpd	65,000			
Initial capital cost	CA$ (millions)	2,228			
Sustaining capital cost	CA$ (millions)	151			
Unit Operating costs:					
Mining cost per tonne mined	CA$/t	1.50			
Milling / Process cost per tonne ore	CA$/t	3.76			
G&A cost per tonne ore	CA$/t	0.99			
Total Cash Cost (Copper) First 5 Years**	US$ / lb Cu	0.378			
Total Cash Cost (Copper) Life of Mine**	US$ / lb Cu	0.616			
Total Cash Cost (Gold) First 5 Years***	US$ / oz Au	(889)			
Total Cash Cost (Gold) Life of Mine***	US$ / oz Au	(874)			
Total Co-Product (Copper) First 5 Years	US$ / lb Cu	0.67			
Total Co-Product (Copper) Life of Mine	US$ / lb Cu	0.82			
Total Co-Product (Gold) First 5 Years	US$ / oz Au	150			
Total Co-Product (Gold) Life of Mine	US$ / loz Au	200			
Metal price assumptions					
Copper	US$/lb	1.50	1.70	3.50	* 1.27
Gold	US$/oz	525	461	626	* 495
Silver	US$/oz	8.00	7.72	12.87	* 6.70
US$/CA$ exchange Rate		0.81	0.81	0.89	0.75
Cashflow					
Annual Ave. After-tax Net Cashflow (years 1-5)	US$ (millions)	414	445	936	384
Cumulative After-tax Net Cashflow (years 1-5)	US$ (millions)	2,069	2,227	4,678	1,921
Economic Results					
Project IRR (pre-tax)	(%)	14.1	16.6	39.0	12.9
Project IRR (after-tax)	(%)	10.6	12.7	30.7	9.5
NPV 0% discount (pre-tax)	US$ (millions)	2,935	3,689	13,822	2,101
NPV 0% discount (after-tax)	US$ (millions)	1,736	2,189	8,287	1,235
NPV 5% discount (pre-tax)	US$ (millions)	1,187	1,604	7,224	833
NPV 5% discount (after-tax)	US$ (millions)	599	856	4,254	395
Payback	years	4.0	3.7	1.5	3.9

* Average metal price -based on N.Seldon Marketing Report (July 2006) with staggered metal prices.
** Copper cash cost includes gold and silver credits.
*** Gold cash costs include copper and silver credits.

The economic analysis set out in this section has been developed based on the capital cost and operating cost estimates set out in Sections 25.6 and 25.7 and is therefore generally subject to the same qualifications, assumptions and exclusions. For example, the occurrence of any of the risk factors that have been excluded from the capital and operating cost estimates would likely have a material impact on the economic analysis set out in this Section.

25.8.2 Base Case Model Inputs

The base case model inputs are summarized below.

25.8.2.1 Reserves

Table 25-29: Summarized Proven And Probable Pit Reserves for Galore Creek

Phase	Ore Total	Ore Total	Waste Total	Strip Ratio	Diluted Grades			
					NSR	CU	AU	AG
	(kBCMS)	(kt)	(kt)	(t/t)	CA$/t	%	g/t	g/t
Total Reserve	**209,321**	**540,735**	**838,903**	**1.6**	**16.3**	**0.557**	**0.303**	**5.32**

25.8.2.2 Production Rate

Table 25-30: Summarised Production Schedule

		Year-3	Year-2	Year-1	Year1	Year2	Year3	Year4	Year5	LOM
ORE Mined										
Total ORE Mined	kt	-	-	-	36,595	43,769	42,175	52,632	37,792	522,016
Ore mined to crusher	kt	-	-	-	26,087	26,061	26,061	26,062	26,062	372,912
Cu	%	-	-	-	0.956	0.897	0.709	0.754	0.781	0.684
Au	g/t	-	-	-	0.400	0.722	0.516	0.687	0.415	0.369
Ag	g/t	-	-	-	7.949	6.808	6.189	6.263	6.687	6.189
ROM reclaim from stockpiles	kt	-	-	-	-	26	26	26	26	149,104
Cu	%	-	-	-	-	0.330	0.314	0.318	0.322	0.283
Au	g/t	-	-	-	-	0.221	0.208	0.195	0.189	0.162
Ag	g/t	-	-	-	-	3.678	3.638	3.746	3.692	3.490
Total Stockpile Inventory	kt	-	-	-	10,507	28,189	44,277	70,821	82,526	-
Total ROM Mill Feed to Mill	kt	-	-	-	26,087	26,087	26,087	26,088	26,088	522,016
Cu	%	-	-	-	0.956	0.897	0.709	0.754	0.781	0.569
Au	g/t	-	-	-	0.400	0.722	0.515	0.687	0.414	0.310
Ag	g/t	-	-	-	7.949	6.805	6.186	6.260	6.684	5.418
Waste										
Mined Sub Grade to Waste	kt	-	-	-	1,801	2,702	4,182	3,807	976	18,715
Waste Mined	kt	2,370	44,682	59,187	50,530	48,038	61,702	47,621	62,291	838,906
Total Waste Mined	kt	2,370	44,682	59,187	52,331	50,740	65,884	51,428	63,266	857,621
Waste Types:										
TILL	kt	2,059	21,904	25,921	13,114	10,976	17,183	9,947	21,743	156,702
Broken NPAG	kt	299	14,407	26,464	21,485	24,125	30,012	9,054	23,312	288,495
Broken PAG	kt	12	8,371	6,802	12,405	11,239	9,318	17,307	13,918	201,312
Stick NPAG	kt	-	-	-	481	1,536	1,576	4,039	1,292	59,758
Stick PAG	kt	-	-	-	3,045	162	3,613	7,274	2,027	132,639
SR		-	-	-	2.0	1.9	2.5	2.0	2.4	1.6
Total Material *Mined*	kt	2,370	44,682	59,187	88,925	94,509	108,059	104,060	101,059	1,379,638

Note: Schedule Total ORE Mined tonnage based on 522 Mt and does not include 18.7 Mt of low grade material displaced by higher grade during the early mine life.

25.8.2.3 Capital Costs

The capital costs for the Galore Creek Project including process plant, infrastructure and mining were estimated in the 2nd quarter of 2006 Canadian dollars. This estimate includes no allowance for escalations in prices or fluctuations in exchange rates. The summary capital cost estimate for the Galore Creek Project detailed in Section 25.6 is presented in Table 25-31 and reflects an intended level of accuracy of $^{+}15\%$ / $^{-}10\%$.

Table 25-31: Capital Cost Estimate Summary

Description		Responsible Party	Estimate (CA$M)
Mine		GR Tech	457
Concentrator		Hatch	410
Water / Waste Management	Structure Water Recovery	Ledcor Hatch	247
Infrastructure	Tunnel Pipeline Road Power Line	HMM PSI Ledcor IHI	330
Total Directs			1,444
Indirect Costs		All	295
EPCM Costs		Hatch	151
Total Direct and Indirect			**1,890**
Contingency		Hatch	268
Owners Costs		Hatch	70
Total Project Estimate			**2,228**

25.8.2.4 Operating Costs

Includes Mine / Concentrate / Transportation to Port and Port Handling Costs.

- Overall Cost of Production 9.12 CA$/t milled

25.8.2.5 Metal Prices

The study metal price assumptions are as follows.

- Copper US$1.50 /lb
- Gold US$ 525 /oz
- Silver US$ 8.00 oz

25.8.2.6 Exchange Rate

The study cashflow exchange rate assumption is as follows.

- US$/CS$ 0.81

25.8.2.7 Taxes

Income earned from the Galore Creek is generally subject to Federal income tax, provincial income tax and BC mining tax. It is assumed that the Galore Creek project will commence commercial production in 2011.

Federal income tax rates are legislated to decline. Based on current legislation, in 2006 and 2007 the federal income rate will be 22.12%, 2008 will be 20.50%, 2009 will be 20%, and 2010 and onward will be 19%. The provincial income tax rate is assumed to remain the same as it is currently at 12% The BC mining tax rate is 13%, with a minimum tax of 2% of proceeds less defined current costs.

Capital expenditures incurred prior to 2011 are assumed to be 90% tangible and 10% intangible. For 2011, the capital expenditures incurred are assumed to be 90% tangible and 10% intangible. For years after 2011, the capital expenditures incurred are assumed to be 80% tangible and 20% intangible. For income tax purposes, tangible capital expenditures are generally treated as Class 41(b) but can be treated as Class 41(a), depending on whether the tangible assets were acquired prior to the mine coming into production. For Class 41(b), up to 25% of undepreciated capital cost ("UCC") is deductible as capital cost allowance ("CCA") in a taxation year, subject to half-year rule for new tangible asset acquisitions during the taxation year. For Class 41(a), CCA is deductible up to the income from the mine for that year. Depending on facts and circumstances, intangible expenditures can be treated as Canadian exploration expense ("CEE"), Canadian development expense ("CDE") or operating costs. CEE is generally deductible up to 100% of unclaimed pool balance but cannot be deducted to create or increase non-capital loss of a principal business corporation. CDE is deductible at 30% on a declining balance basis.

For BC mining tax purposes, the tangible and intangible capital costs are generally accumulated in a cumulative expenditure account ("CEA") and can be used to reduce income for BC mining tax purposes. Any unclaimed CEA can be carried forward indefinitely.

For income tax purposes, non-capital losses incurred in the taxation year ended November 30, 2004 and 2005 can be carried forward for 10 years, and thereafter can be carried forward for 20 years.

25.8.2.8 Smelter Terms

- Treatment Charge — US$ 80/dmt
- Copper Accountability — 96.5 Deduct 1 unit pay balance
- Gold Accountability — 97%
- Silver Accountability — 90%
- Copper Refining Charge — US$ 0.08/lb
- Price Participation — $^{+}$10% over US$1.20/lb with a cap at US$1.80/lb
  $^{-}$10% under US$1.20/lb with a floor at US$0.90/lb
- Gold Refining Charge — US$ 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$ 40 per dmt
- Other / Marketing — US$ 6 per dmt

25.8.3 Basis of the Cashflow Analysis

For the project economic analysis the following final results have been calculated.

Table 25-32: Base Case Economic Results

	Before Tax	After Tax
IRR, %	14.1%	10.6%
0% NPV, $US M	2,935	1,736
5% NPV, $US M	1,186	599
8%NPV, $US, M	623	223
Payback (yrs)		4.0

25.8.4 Payback Calculation

Payback has been calculated to be 4.0 years, Table 25-33 is an excerpt from the cash flow model.

Table 25-33: Payback Summary

Financial Year	**-1**	**1**	**2**	**3**	**4**	**5**
	US$M	**US$M**	**US$M**	**US$M**	**US$M**	**US$M**
Gross Revenue		885,817	935,515	723,922	830,587	765,535
Processing Costs		(117,121)	(111,294)	(90,340)	(97,731)	(97,014)
Operating Costs		(259,052)	(258,818)	(252,182)	(260,800)	(256,703)
Sustaining Capital		(78,310)	(5,810)	(4,810)	(2,810)	(5,810)
Cash Flow (Pre-Tax, Pre-Royalties)		**431,334**	**559,593**	**376,589**	**469,247**	**406,008**
Income Tax		(10,177)	(11,292)	(7,612)	(13,872)	(131,160)
Royalties						
Cash Flow (After-Tax, After-Royalties)		**421,157**	**548,301**	**368,977**	**455,375**	**274,849**
Accumulated Cash Flow (Pre-Tax, Pre-Royalties)	(1,806,242)	(1,374,909)	(815,315)	(438,726)	30,520	**436,529**
Accumulated Cash Flow (After-Tax, After-Royalties)	(1,806,242)	(1,385,085)	(836,784)	(467,807)	(12,432)	**262,417**

Payback occurs in the 4rth year of production - 4.0 yrs.

25.8.5 *Base Case Sensitivity Analysis*

A Sensitivity analysis is presented in the following tables and graphs. As can be seen from these results the project appears to be more sensitive to Copper price and USD exchange than other variables.

Table 25-34: Galore Creek Metal Price Sensitivity (US$M)

Cu Price (US$/lb)		Au/Ag Price (US$/oz)						
		450/7.00	500/7.50	525/8.00	550/8.25	600/8.50	650/9.00	700/9.50
1.25	NPV @ 0% (US$M)	807	926	993	1,052	1,164	1,284	1,404
	NPV @ 5% (US$M)	56	128	169	205	272	343	415
	Pre-tax IRR (%)	7.8%	8.7%	9.2%	9.7%	10.5%	11.3%	12.2%
	After-tax IRR (%)	5.6%	6.3%	6.7%	7.1%	7.7%	8.4%	9.1%
	Payback (years)	5.7	5.5	5.3	5.1	4.9	4.6	4.4
1.50	NPV @ 0% (US$M)	1,548	1,669	1,736	1,796	1,908	2,028	2,149
	NPV @ 5% (US$M)	488	559	599	634	700	771	841
	Pre-tax IRR (%)	12.9%	13.7%	14.1%	14.5%	15.2%	15.9%	16.6%
	After-tax IRR (%)	9.6%	10.3%	10.6%	10.9%	11.5%	12.1%	12.6%
	Payback (years)	4.4	4.2	4.0	4.0	3.8	3.7	3.6
1.75	NPV @ 0% (US$M)	2,293	2,413	2,481	2,541	2,654	2,774	2,894
	NPV @ 5% (US$M)	915	985	1,024	1,059	1,125	1,195	1,265
	Pre-tax IRR (%)	17.2%	17.9%	18.3%	18.6%	19.2%	19.9%	20.5%
	After-tax IRR (%)	13.1%	13.7%	14.0%	14.3%	14.8%	15.3%	15.8%
	Payback (years)	3.6	3.5	3.4	3.4	3.2	3.1	3.0
2.00	NPV @ 0% (US$M)	3,104	3,224	3,292	3,352	3,465	3,585	3,705
	NPV @ 5% (US$M)	1,376	1,445	1,485	1,519	1,585	1,654	1723
	Pre-tax IRR (%)	21.4%	22.0%	22.3%	22.6%	23.2%	23.8%	24.4%
	After-tax IRR (%)	16.5%	17.0%	17.3%	17.5%	18.0%	18.5%	18.9%
	Payback (years)	2.9	2.8	2.8	2.8	2.7	2.6	2.6
2.50	NPV @ 0% (US$M)	4,760	4,880	4,948	5,008	5,120	5,240	5,360
	NPV @ 5% (US$M)	2,310	2,379	2,418	2,452	2,517	2,586	2,654
	Pre-tax IRR (%)	28.7%	29.2%	29.5%	29.8%	30.3%	30.8%	31.3%
	After-tax IRR (%)	22.4%	22.9%	23.1%	23.3%	23.7%	24.1%	24.5%
	Payback (years)	2.0	2.0	2.0	1.9	1.9	1.9	1.9
3.50	NPV @ 0% (US$M)	8,070	8,190	8,257	8,317	8,430	8,550	8,670
	NPV @ 5% (US$M)	4,163	4,232	4,271	4,305	4,370	4,439	4,507
	Pre-tax IRR (%)	40.6%	41.0%	41.2%	41.4%	41.8%	42.2%	42.6%
	After-tax IRR (%)	32.0%	32.3%	32.5%	32.6%	32.9%	33.3%	33.6%
	Payback (years)	1.4	1.4	1.4	1.4	1.4	1.4	1.4

Note:

- NPV = net present value using a discounted cash flow analysis;
- IRR = internal rate of return.
- Base case is shown in Green
- Assumes 100% equity
- All NPV and payback figures are after tax

Sensitivity analysis to exchange rate, capital and operating cost variations are presented in Table 25-35. Sensitivity to Metal Prices are shown in Table 25-36. Economic sensitivity to grade variations will mirror those resulting from metal price sensitivity.

Table 25-35: Base Case Sensitivity After-Tax IRR

IRR% (After Tax)	Variation from Base Case				
	-20%	-10	0%	10%	20%
Exchange Rate / Operations	16.2%	13.2%	10.6%	8.2%	6.1%
Capital Cost	14.2%	12.2%	10.6%	9.2%	8.0%
Operating Cost	12.7%	11.7%	10.6%	9.4%	8.2%
Copper	5.9%	8.4%	10.6%	12.7%	14.6%
Gold	9.5%	10.0%	10.6%	11.2%	11.7%
Silver	10.4%	10.5%	10.6%	10.7%	10.8%
Diesel Price	10.9%	10.7%	10.6%	10.5%	10.4%
TC/RC	11.3%	10.9%	10.6%	10.3%	9.9%
Ocean Freight, Concentrate	10.8%	10.7%	10.6%	10.5%	10.4%

Table 25-36: Base Case NPV Price Sensitivity

Sensitivity Factors	Change %	After Tax NPV @ 0%, US$M	After Tax NPV @ 5%, US$M	After Tax NPV @ 8%, US$M	Pre-Tax IRR, %	After-Tax IRR %	Payback (Yrs)
Base Case							
	0%	$1,736	$599	$223	14.1%	10.6%	4.0
Copper Price							
	20%	$2,630	$1,109	$605	19.0%	14.6%	3.3
	10%	$2,183	$854	$415	16.7%	12.7%	3.7
	-10%	$1,289	$342	$29	11.3%	8.4%	4.7
	-20%	$846	$82	-168	8.1%	5.9%	5.6
Gold Price							
	20%	$1,955	$729	$322	15.5%	11.7%	3.8
	10%	$1,845	$664	$273	14.8%	11.2%	3.9
	-10%	$1,627	$534	$173	13.4%	10.0%	4.3
	-20%	$1,517	$469	$123	12.6%	9.5%	4.5
Silver Price							
	20%	$1,786	$627	$244	14.4%	10.8%	4.0
	10%	$1,761	$613	$233	14.2%	10.7%	4.0
	-10%	$1,711	$585	$213	13.9%	10.5%	4.1
	-20%	$1,686	$571	$202	13.8%	10.4%	4.1
Operating Costs							
	20%	$1,221	$318	$17	11.1%	8.2%	4.7
	10%	$1,476	$458	$120	12.7%	9.4%	4.4
	-10%	$1,996	$740	$325	15.4%	11.7%	3.9
	-20%	$2,256	$880	$428	16.7%	12.7%	3.7
Capital Costs							
	20%	$1,521	$377	$3	10.8%	8.0%	5.0
	10%	$1,629	$488	$113	12.3%	9.2%	4.6
	-10%	$1,843	$708	$331	16.1%	12.2%	3.7
	-20%	$1,951	$817	$438	18.6%	14.2%	3.3

25.8.5.1 Graphical Representations

Figure 25-7: Sensitivity Analysis of Metal Prices on After-Tax NPV

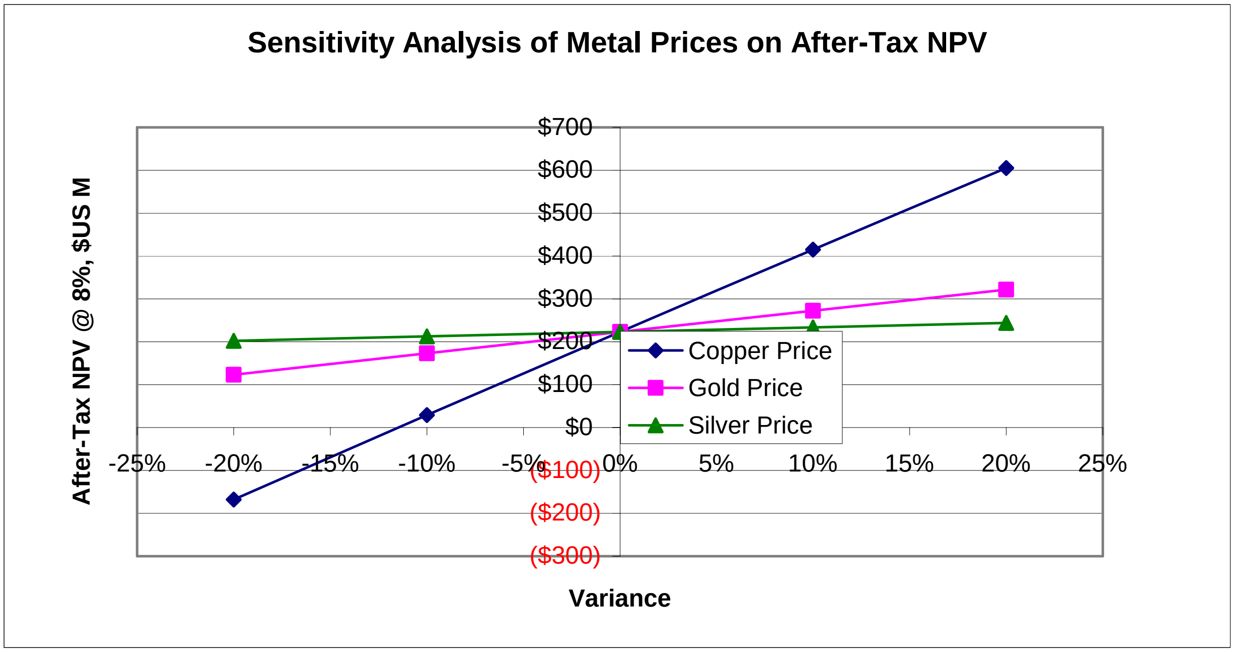


Figure 25-8: Sensitivity Values of OPEX and CAPEX on After-Tax NPV

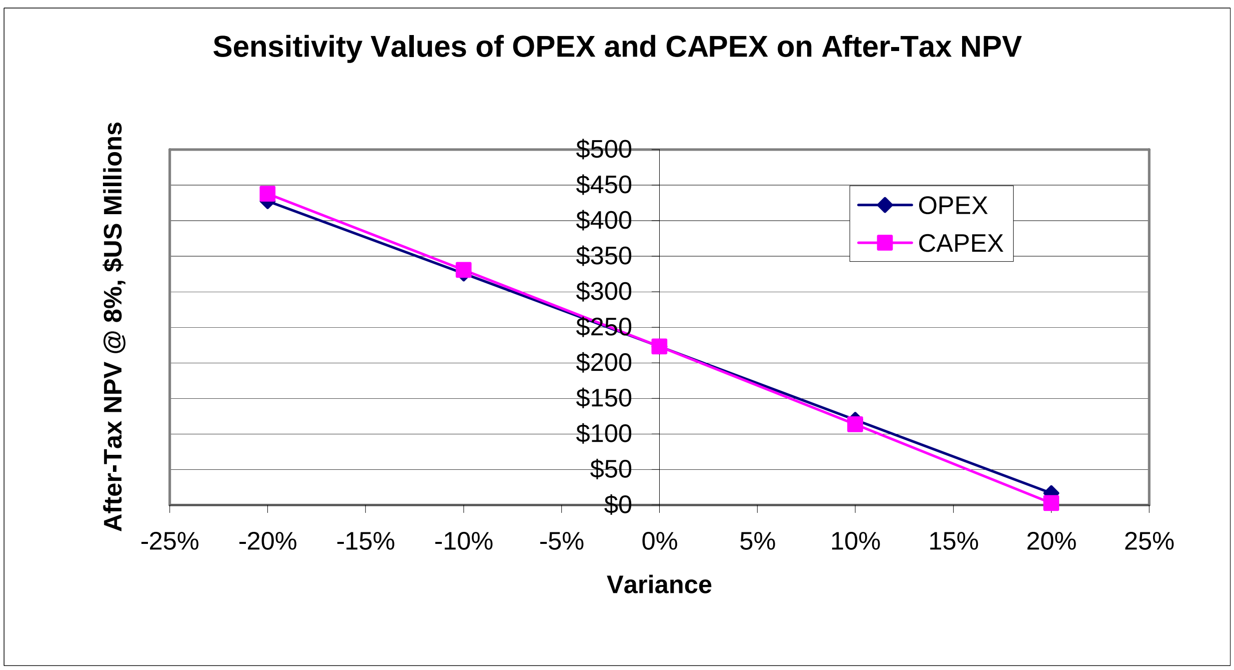


Figure 25-9: Sensitivity Analysis of Metal Prices on After-Tax IRR

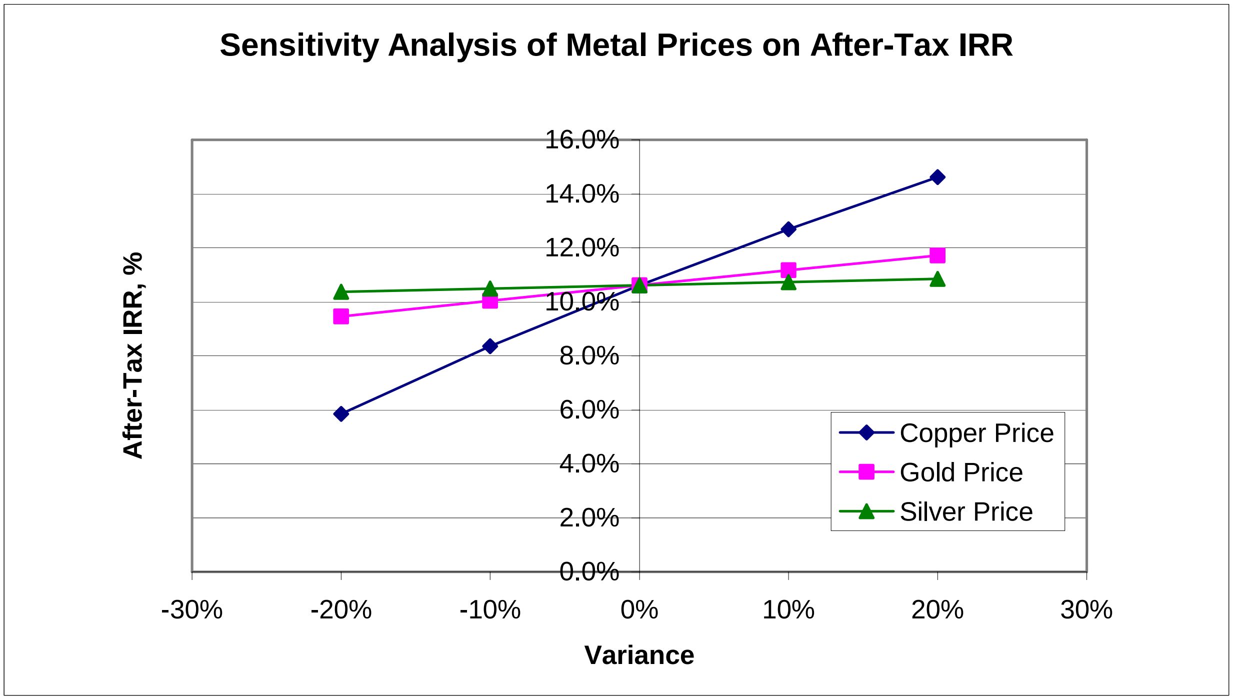
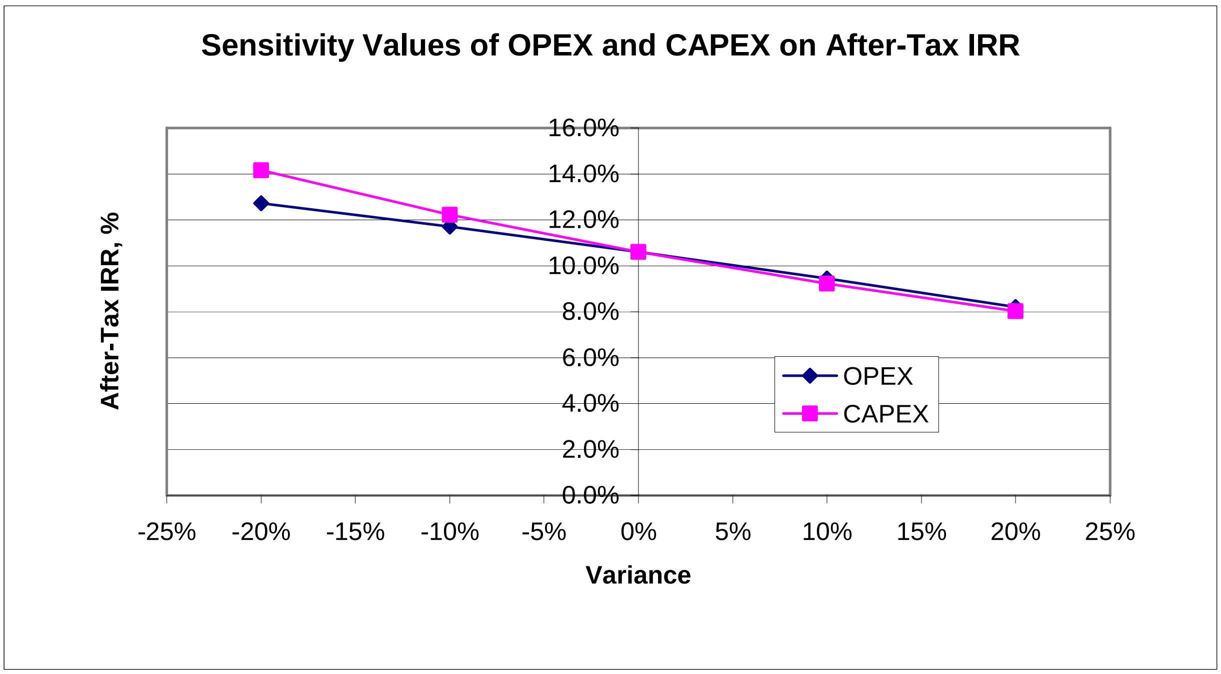


Figure 25-10: Sensitivity Values of OPEX and CAPEX on After-Tax IRR



25.9 Payback

Payback is approximately 4.0 years on a zero return basis. Refer to Section 25.8.4 and Table 25-33.

25.10 Life of Mine

Based on the currently identified reserves and a nominal 65,000 tpd throughput the mine life is expected to be 22 years.

NovaGold is nearing completion of its 2006 expansion drilling program at Galore Creek. Additional results from this 35,000 meter program are expected over the coming weeks. Drilling to date has shown potential to expand the resource.

26. Date And Signature Page

26.1 Hatch Associates

CERTIFICATE Bruce Rustad, P.Eng.

I, Bruce Rustad, P.Eng., 2200-1066 W. Hastings Street, Vancouver, BC V6E 3X2 do hereby certify that:

- I am a Professional Engineer employee of Hatch Ltd.
- I am an Associate with Hatch.
- I am a graduate of the University of Calgary.
- I am member in good standing of Professional Engineers Ontario.(#40155509)
- I have practiced my profession since 1989.
- I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in National Instrument 43-101.
- I am responsible for coordinating the study and the author of the following Sections 3.1, 3.8, 3.9, 3.10, 3.11, 4, 5, 20.3.1, 20.3.6, 20.3.7, 20.4, 21.1, 22.10, 25.3, 25.4, 25.5, 25.6, 25.7, 25.8 of the Independent Technical Report; Dated - October 31, 2006 for the Galore Creek Feasibility Study.
- I have visited the site on several occasions in 2005 and 2006 the first visit was 13-15 September 2005.
- 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.
- I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
- 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.
- I consent to the use of this Technical Report dated, October 31, 2006, by NovaGold for making representations about the subject property and to the public filing of the Technical Report.
- I consent to the filing of the Technical Report dated October 31,2006, with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 31st day of October.

[redacted]

[redacted]

Bruce Rustad, P.Eng.
Director of P&CM / Project Manager
Hatch Vancouver

CERTIFICATE Hoe Teh, P.Eng.

I, Hoe Teh, a Senior Process Engineer of Hatch Ltd. a Canadian corporation with a business address of Suite 2200 – 1066 West Hastings Street, Vancouver, B.C., Canada V6E 3X2, HEREBY CERTIFY THAT

- I am a graduate of the University of British Columbia in Vancouver, Canada, with a Bachelor's degree in Applied Science (1973).
- From 1973 to the present I have been actively employed in the mineral processing industry in research and development, operations, process and flowsheet development and process engineering.
- I am a Registered Professional Engineer in the province of British Columbia (#10452).
- As a result of my education, experience and professional associations, I am a "Qualified Person" as defined by National Instrument 43-101 (the "Instrument").
- I am responsible for the metallurgy and process for the study and the author of the following Sections 3.5,3.6,18 and 20.1 of the Independent Technical Report; Dated - October 31, 2006 for the Galore Creek Feasibility Study.
- My work on the Galore Creek Project consisted of management of the metallurgical program, interpretation of the results, and process and flowsheet development.
- The sources of all information are noted and referenced in the Technical Report.
- I am independent of the issuer as defined in the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make this report not misleading.
- I have read and understand the terms of the Instrument its companion documents and the Technical Report has been prepared in compliance with the Instrument.
- I have not had any prior involvement with the Galore Creek property that is the subject of this Technical Report.
- I consent to the use of this Technical Report dated, October 31, 2006, by NovaGold for making representations about the subject property and to the public filing of the Technical Report.
- I consent to the filing of the Technical Report dated October 31,2006, with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 31st day of October.

[redacted]

Hoe Teh, P.Eng.
Senior Process Engineer
Hatch

CERTIFICATE Dean Brox, P.Eng.

I, Dean R. Brox, a consulting tunnel engineer and senior tunnel engineer of Hatch Mott MacDonald Ltd., (HMM) a Canadian corporation with a business address of Suite 1010 – 1066 West Hastings Street, Vancouver, B.C., Canada V6E 3X2, HEREBY CERTIFY THAT

- I am a graduate of the University of British Columbia in Vancouver, Canada, with a Bachelor's degree in Geological Engineering (1985) and of Imperial College, London, United Kingdom, and a Master's degree with Distinction in Engineering Rock Mechanics (1990).
- From 1985 to the present I have been actively employed in various capacities of the mining industry in numerous locations throughout the world. I have worked as a geotechnical/tunnel engineer for the planning, design, and construction of mine infrastructure tunnels for mining projects located throughout the world.
- I am a Registered Professional Engineer in the province of British Columbia (#23480).
- As a result of my education, experience and professional associations, I am a "Qualified Person" as defined by National Instrument 43-101 (the "Instrument").
- My work on the Galore Creek Project consisted of several site visits between 2004 to 2006, to inspect the geological, geotechnical, and hydrogeological conditions related to NovaGold's mine access tunnel.
- I am responsible for the access tunnel for the study and the author of the following Sections 3.5, 3.6, 18, 20.1, 21.4 and 22.3 of the Independent Technical Report; Dated - October 31, 2006 for the Galore Creek Feasibility Study.
- I am independent of the issuer as defined in the Instrument.
- The sources of all information are noted and referenced in the Technical Report.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make this report not misleading.
- I have read and understand the terms of the Instrument its companion documents and the Technical Report has been prepared in compliance with the Instrument.
- I have not had any prior involvement with the Galore Creek property that is the subject of this Technical Report.
- I consent to the use of this Technical Report dated, October 31, 2006, by NovaGold for making representations about the subject property and to the public filing of the Technical Report.
- I consent to the filing of the Technical Report dated October 31,2006, with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 31st day of October.

[redacted]

[redacted]

Dean Brox, P.Eng.
Senior Mining Engineer
Hatch Mott MacDonald

26.2 GR Technical Services

CERTIFICATE James Gray, P.Eng.

I, James H Gray. P.Eng. do hereby certify that:

- I am a Principal of GR Technical Services Ltd., 1584 Evergreen Hill SW Calgary, Alberta Canada T2Y 3A9.
- I graduated with a Bachelor of Applied Science in Mining Engineering from the University of British Columbia in 1975.
- I am registered by The Association of Professional and Geoscientists of the Province of British Columbia, registration number 11,919, and the Association of Professional Engineers, Geologists and Geophysicists of Alberta (M47177).
- I have worked as a Professional Engineer for over 25 years since my graduation from university.
- 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.
- I am responsible for the reserves and mine plan for the study and the author of the following Sections 3.4, 3.8, 3.9, 3.10, 19.2, 21.3, 22.2, 25.1, 25.6 and 25.7 of the Independent Technical Report; Dated - October 31, 2006 for the Galore Creek Feasibility Study.
- I visited the Galore Creek properties during the period 13-15 September 2005 and June 28 & 29, 2006.
- I have not had prior involvement with the property that is the subject of the Technical Report.
- 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.
- I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.
- 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.
- I consent to the use of this Technical Report dated, October 31, 2006, by NovaGold for making representations about the subject property and to the public filing of the Technical Report.
- I consent to the filing of the Technical Report dated October 31,2006, with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 31st day of October , 2006.

[redacted]

James H. Gray PEng.

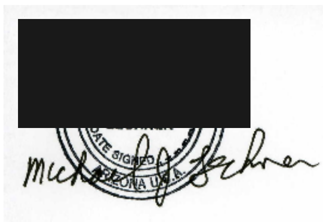
26.3 Resource Modeling Incorporated

CERTIFICATE Michael J. Lechner, RPG.

I, Michael J. Lechner, a consulting geologist and President of Resource Modeling Incorporated, (RMI) an Arizona corporation with a business address of 1960 West Muirhead Loop, Tucson, AZ 85737, HEREBY CERTIFY THAT

- I am the author of the previously filed technical report titled "Updated Galore Creek Resources, Northwestern British Columbia" dated September 7, 2006 and filed on SEDAR September 12, 2006.
- I am a graduate of the University of Montana with a B.A. degree in Geology (1979).
- From 1979 to the present I have been actively employed in various capacities of the mining industry in numerous locations throughout the world. I have worked as an exploration geologist exploring for precious and base metals throughout western North America, a mine geologist working at precious metal mines in California and Nevada, and have estimated Mineral Resources for numerous precious and base metal deposits located throughout the world.
- I am responsible for the resource and geology for the study and the author of the following Sections 3.2, 3.3, 6, 7, 8, 9, 10, 11, 12, 13, 14, 15, 16, 17, 19.1, 21.2 and 22.1 of the Independent Technical Report; Dated - October 31, 2006 for the Galore Creek Feasibility Study.
- I am a Registered Professional Geologist in the State of Arizona (#37753), a Certified Professional Geologist with the American Institute of Professional Geologists (#10690) and a Registered Member of the Society of Mining Engineers (# 4124987RM).
- As a result of my education, experience and professional associations, I am a "Qualified Person" as defined by National Instrument 43-101 (the "Instrument").
- My work on the Galore Creek Project consisted of a site visit on October 17-18 2005, to observe drilling and sampling procedures, review drill core and a review of NovaGold's resource model.
- I am independent of the issuer as defined in the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make this report not misleading.
- I have read and understand the terms of the Instrument its companion documents and the Technical Report has been prepared in compliance with the Instrument.
- I have not had any prior involvement with the Galore Creek property that is the subject of this Technical Report.
- I consent to the use of this Technical Report dated, October 31, 2006, by NovaGold for making representations about the subject property and to the public filing of the Technical Report.
- I consent to the filing of the Technical Report dated October 31,2006, with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated in Tucson, Arizona, this 31st day of October, 2006.


[redacted]

Michael J. Lechner

26.4 BGC Engineering Inc.

CERTIFICATE Iain Bruce, P.Eng.

I, Iain Bruce, P.Eng, do hereby certify that:

I am currently employed as a senior engineer by:

BGC Engineering Inc
500 1045 Howe St Vancouver BC
V6L 2S8

- I graduated with the degree of BSC Eng. from the University Queen's in 1973. In addition, I have obtained a PhD from University of Alberta in 1978.
- I am a member of the Professional Engineers of Alberta, British Columbia, Ontario and Manitoba as well as both Yukon and Northwest territory.
- I have worked as a geotechnical engineer for a total of 33 years since my graduation from university.
- 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.
- I visited the Galore Creek property on July 25, 2005 for five days and on August 23, 2006 for six days.
- I am responsible for the Waste and Water Management Design, Plant Site Design, Geohazard Assessment, and Open Pit Slope and other geotechnical issues and the author of the following Sections 3.7, 3.8, 3.9, 20.2, 21.5, 22.4, 25.6 and 25.7 of the Independent Technical Report; Dated - October 31, 2006 for the Galore Creek Feasibility Study.
- I have not had prior involvement with the property that is the subject of the Technical Reports.
- I am not aware of any material fact or material change with respect to the subject matter of the Technical Reports that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
- I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.
- I have read National Instrument 43-101 and Form 43-101F1, and the sections of the Technical Reports for which I am responsible have been prepared in compliance with that instrument and form.
- I consent to the use of this Technical Report dated, October 31, 2006, by NovaGold for making representations about the subject property and to the public filing of the Technical Report.
- I consent to the filing of the Technical Report dated October 31,2006, with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 31st day of October 2006.

Signature of Qualified Person [redacted]

Iain Bruce

Print Name of Qualified Person

26.5 Pipeline Systems Incorporated

CERTIFICATE Donald Hallbom, P.Eng.

I, Donald J. Hallbom, a registered professional engineer acting as a technical specialist to Pipeline Systems Incorporated, a California corporation with a business address of 5099 Commercial Circle, Suite 102, Concord, CA, USA, 94520, HEREBY CERTIFY THAT

- I am a graduate of the University of British Columbia with a B.A.Sc. degree in Mechanical Engineering (1989)
- I will be defending my thesis (on the flow of mineral slurries) at the University of British Columbia for a PhD degree in Mining Engineering in the next 12 months.
- From 1989 to present I have been actively employed as an engineer in primary extractive industries throughout the world, with a focus on mining since 1995. My area of specialty is the hydraulics of non-Newtonian suspensions (e.g., mineral slurry, wood pulp) and process design.
- I am currently employed as a Project Engineer by MG Engineering Incorporated, a British Columbia corporation with a business address of 1016 Seamount Way, Gibsons, BC, V0N 1V7.
- MG Engineering has seconded me to Pipeline Systems Incorporated – my current job assignments, training, work direction, work product, etc. are overseen by Pipeline Systems Incorporated and resident expert staff therein.
- I am a Registered Professional Engineer in the Province of British Columbia (#18910). I am also a Registered Professional Engineer with the State of Washington (#30819)
- As a result of my education, experience and professional associations, I am a "Qualified Person" as defined by National Instrument 43-101 (the "Instrument")
- I am responsible for the overland pipelines etailed design and cost estimate and the author of the following Sections 3.7, 3.8, 3.9 ,3.10, 20.35, 22.6, 25.6 and 25.7 of the Independent Technical Report; Dated - October 31, 2006 for the Galore Creek Feasibility Study.
- I am independent of the issuer as defined in the Instrument.
- I have not had any prior involvement with the Galore Creek property that is the subject of this Technical Report.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed in order to make this report not misleading.
- I have read and understand the terms of the Instrument and this Technical Report has been prepared in compliance with the Instrument.
- I consent to the use of this Technical Report dated, October 31, 2006, by NovaGold for making representations about the subject property and to the public filing of the Technical Report.
- I consent to the filing of the Technical Report dated October 31,2006, with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 31st day of October.

[redacted]

[redacted]

Don Hallbom, P.Eng.
Process Engineer
Pipeline Systems Incorporated

26.6 Ian Hayward International Ltd.

CERTIFICATE Allan Guy, P.Eng.

Ian Hayward International Ltd., an Electrical Consultant located in Surrey, B.C., certify that the Feasibility Study performed was based on the following scope of work:

- Selection of the optimum transmission line alignment for the North Route was from maps, aerial photo study, terrain typing and helicopter reconnaissance.
- Provide preliminary design of structures, foundations, conductors, insulators, hardware and overhead shield wires. Substation design was limited to creation of station one-line diagrams with the resulting equipment requirements. This was to provide a cost estimate of the transmission line and substation construction.
- Design criteria was formulated in the process of performing this study covering environmental conditions, power system conditions, transmission line insulation levels, lightning protection, conductor, ice and wind loadings, loading and limiting conditions, ground clearance, structures, foundations and terminations.

Allan Guy is a Registered Profession Engineer in the Province of British Columbia and Province of Alberta.

David Brown is a Registered Profession Engineer in the Province of British Columbia.

The source of all information are noted and referenced in the Feasibility Study of August 2006.

- I am responsible for the power transmission line and the author of the following Sections 3.7, 3.8, 3.9 ,3.10, 22.3.3, 22.9, 25.6 and 25.7 of the Independent Technical Report; Dated - October 31, 2006 for the Galore Creek Feasibility Study.
- I am independent of the issuer as defined in the Instrument.
- As a result of my education, experience and professional associations, I am a "Qualified Person" as defined by National Instrument 43-101 (the "Instrument").
- As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make this report not misleading.
- I have read and understand the terms of the Instrument its companion documents and the Technical Report has been prepared in compliance with the Instrument.
- I have not had any prior involvement with the Galore Creek property that is the subject of this Technical Report.
- I consent to the use of this Technical Report dated, October 31, 2006, by NovaGold for making representations about the subject property and to the public filing of the Technical Report.
- I consent to the filing of the Technical Report dated October 31,2006, with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated in Surrey, British Columbia this 31st day of October.

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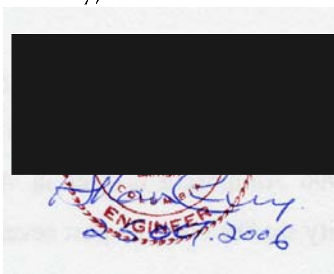


[redacted]

Allan Guy, P.Eng.

David Brown, P.Eng.

26.7 McElhanney Consulting Services Ltd.

CERTIFICATE Robert Parolin, P.Eng.

I, Robert W. Parolin, a consulting engineer with McElhanney Consulting Services Ltd. (MCSL), a British Columbia corporation with a business address of 1633 First Avenue, Prince George, BC, V2L 2Y8, HEREBY CERTIFY THAT

- I am a graduate of the University of New Brunswick with a Bachelor of Science degree in Forest Engineering (1974).
- From 1974 to the present, I have been actively employed in various capacities of the resource-based industries in British Columbia. I have worked as an engineering consultant for the past 25 years, providing road access solutions to the mining, forestry and oil & gas industries. Projects have included route selection, road location, survey and design, construction, cost estimating and project supervision.
- I am a Registered Professional Engineer in the Province of British Columbia (#11134), and also registered in the Provinces of Alberta and Manitoba.
- As a result of my education, experience and professional associations, I am a "Qualified Person" as defined by National Instrument 43-101 (herein known as the "Instrument").
- My most recent work on the Galore Creek Project consisted of a site visit on September 27-28, 2006, to conduct a viewing of the road corridor and bridge crossings with qualified contractors.
- I am responsible for the access road and the author of the following Sections 3.7, 3.8, 3.9 ,3.10, 20.3.2, 22.7 and 25.7 of the Independent Technical Report; Dated - October 31, 2006 for the Galore Creek Feasibility Study.
- The sources of all information are noted and referenced in the Technical Report.
- I am independent of the issuer as defined in the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make this report not misleading.
- I have read and understand the terms of the Instrument, its companion documents, and the Technical Report has been prepared in compliance with the Instrument.
- I have not had any prior involvement with the Galore Creek property that is the subject of this Technical Report.
- I consent to the use of this Technical Report dated, October 31, 2006, by NovaGold for making representations about the subject property and to the public filing of the Technical Report.
- I consent to the filing of the Technical Report dated October 31,2006, with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated in Prince George, BC, this 31st day of October 2006-10-25

[redacted]

Robert W. Parolin P.Eng.

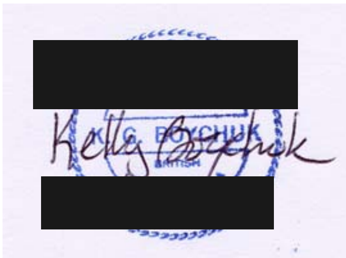
26.8 Ledcor CMI Ltd.

CERTIFICATE Kelly Boychuk, P.Eng.

I, Kelly Boydhuk, P.Eng., 1200-1067 W. Cordova Street, Vancouver, BC V6C 1C7 do hereby certify that:

- I am a Professional Engineer and a consultant for Ledcor CMI Ltd.
- I am a graduate of the University of British Columbia.
- I am member in good standing of Professional Engineers and Geoscientists of British Columbia.
- I have practiced my profession since 1990.
- I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in National Instrument 43-101.
- I am responsible for the for cost estimates and schedule analysis of the road and water diversion structures and the author of the following Sections 3.8, 3.9 22.5 and 25.6 of the Independent Technical Report; Dated - October 31, 2006 for the Galore Creek Feasibility Study.
- This report titled, "Independent Technical Report; June 2006" is based on a study of the data available on the Galore Creek Project from NovaGold Canada Inc.. I am responsible for cost estimates and schedule analysis; that is I have contributed to Sections 3.8, 3.9, 22.5, 25.6
- I have visited the site.
- 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.
- I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
- 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.
- I consent to the use of this Technical Report dated, October 31, 2006, by NovaGold for making representations about the subject property and to the public filing of the Technical Report.
- I consent to the filing of the Technical Report dated October 31,2006, with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 31st day of October.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Kelly Boychuk, P.Eng.
Project Manager
Ledcor CMI Ltd.