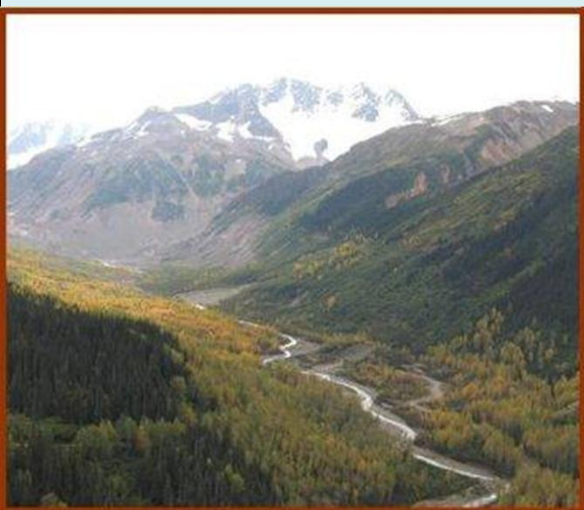
Galore Creek Project
British Columbia
NI 43-101 Technical Report on Pre-Feasibility Study



Submitted by:

Robert Gill, P.Eng.
Greg Kulla, P.Geo
Greg Wortman, P.Eng
Jay Melnyk, P.Eng
Dana Rogers, P.Eng

Effective Date: 27 July 2011
Project Number: 166824

CERTIFICATE OF QUALIFIED PERSON

Robert Gill P.Eng.
AMEC Americas Limited
111 Dunsmuir Street, Suite 400
Vancouver, BC, Canada V6B 5W3
Tel: (604) 664-4204
Fax: (604) 669-9516
robert.gill@amec.com

I, Robert Gill P.Eng., am employed as a Principal Consultant and Study Manager with AMEC Americas Limited.

This certificate applies to the technical report entitled "Galore Creek Project British Columbia NI 43-101 Technical Report on Pre-Feasibility Study" that has an effective date of 27 July 2011 (the "Technical Report").

I graduated with a Bachelor of Applied Science (Hons) in Geological Engineering from the University of British Columbia in 1985. I am a Member of the Association of Professional Engineers and Geoscientists of British Columbia.

I have practiced my profession since 1985 and have been involved in design, construction, mining operations, health, safety and environmental management, and consulting. I have been involved in Environmental, Geotechnical, and Mining consulting practice for 7 years, including working on both gold deposits and copper deposits for at least 5 years.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I did not visit the Galore Creek Project (the "Project").

I am responsible for Sections 1, 2, 3, 4, 5, 6, 18 (except Sections 18.8 and 18.9), 19, 20, 21, 22, 23, 24, 25, 26 and 27 of the Technical Report.

I am independent of NovaGold Resources Inc. as independence is described by Section 1.5 of NI 43–101.

Prior to completion of this technical report I have had no other involvement with this project.

I have read NI 43–101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the Sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"Signed and sealed"

Robert Gill, P.Eng.

Dated: 12 September 2011

AMEC Americas Limited
111 Dunsmuir Street, Suite 400
Vancouver, B.C. V6B 5W3
Tel (604) 664-3030
Fax (604) 664-3057

CERTIFICATE OF QUALIFIED PERSON

Greg Kulla P.Geo.
Associate, AMEC Americas Limited
111 Dunsmuir Street, Suite 400
Vancouver, BC, Canada V6B 5W3
Tel: (604) 664-3229
Fax: (604) 664-3057
greg.kulla@amec.com

I, Gregory Kenneth Kulla, P.Geo., am employed as a Principal Geologist with AMEC Americas Ltd.

This certificate applies to the technical report entitled "Galore Creek Project British Columbia NI 43-101 Technical Report on Pre-Feasibility Study" that has an effective date of 27 July 2011 (the "Technical Report").

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC, member #23492), and of the Association of Professional Geoscientists of Ontario (APOG #1752). I graduated from the University of British Columbia with a Bachelor of Science in Geology degree in 1988.

I have practiced my profession continuously since 1988 and have been involved in precious and base metal disseminated sulphide deposit assessments in Canada, United States, Australia, Mexico, Chile, Peru, and India.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I visited the Galore Creek Project (the "Project") between 21 to 24 September 2010.

I am responsible for Sections 7, 8, 9, 10, 11, 12, 14, and those portions of the Summary, Interpretations and Conclusions and Recommendations that pertain to those Sections. of the Technical Report.

I am independent of NovaGold Resources Inc. as independence is described by Section 1.5 of NI 43–101.

Prior to completion of this technical report I have had no other involvement with this project.

I have read NI 43–101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the Sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"Signed and sealed"

Greg Kulla, P.Geo.

Dated: 12 September 2011

AMEC Americas Limited
111 Dunsmuir Street, Suite 400
Vancouver, B.C. V6B 5W3
Tel (604) 664-3030
Fax (604) 664-3057

CERTIFICATE OF QUALIFIED PERSON

Gregory Wortman P.Eng.
AMEC Americas Limited
2020 Winston Park Drive, Suite 700
Oakville, Ontario L6H 6X7
Tel: (905) 403-3539
Fax: (905) 829 5401
Gregory.Wortman @amec.com

I, Gregory Wortman P.Eng., am employed as a Technical Director Process with AMEC Americas Ltd.

This certificate applies to the technical report entitled "Galore Creek Project British Columbia NI 43-101 Technical Report on Pre-Feasibility Study" that has an effective date of 27 July 2011 (the "Technical Report").

I am a member of Professional Engineers, Ontario. I graduated from the Technical University of Nova Scotia in 1968.

I have practiced my profession continuously since 1968 and have been involved in precious and base metal disseminated sulphide deposit assessments in Canada, United States, Chile, Peru, and Russia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not visited the Galore Creek Project (the "Project").

I am responsible for Sections 13 and 17, and those portions of the Summary, Interpretations and Conclusions and Recommendations that pertain to those Sections. of the Technical Report.

I am independent of NovaGold Resources Inc. as independence is described by Section 1.5 of NI 43–101.

Prior to completion of this technical report I have had no other involvement with this project.

I have read NI 43–101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the Sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"Signed and sealed"

Gregory Wortman, P.Eng.

Dated: 12 September 2011

CERTIFICATE OF QUALIFIED PERSON

Jay Melnyk, P.Eng.
Associate, AMEC Americas Limited
111 Dunsmuir Street, Suite 400
Vancouver, BC, Canada V6B 5W3
Tel: (604) 317-6170
Fax: (604) 664-3057

I, Jay Melnyk, P.Eng., am employed as an Associate with AMEC Americas Limited.

This certificate applies to the technical report entitled "Galore Creek Project British Columbia NI 43-101 Technical Report on Pre-Feasibility Study" that has an effective date of 27 July 2011 (the "Technical Report").

I am a member of a Professional Engineer in the province of British Columbia (P.Eng #25975). I graduated from the Montana Tech of the University of Montana with a Bachelor of Mining Engineering degree in 1988 and from the British Columbia Institute of Technology with a Diploma in Mining Technology in 1984.

I have practiced my profession for 22 years. I have been directly involved in open pit mining operations, and design of open pit mining operations in Argentina, Eritrea, Indonesia, Canada, the United States, Chile, Peru and Mexico.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I visited the Galore Creek Project (the "Project") between 27 and 28 September, 2007.

I am responsible for Sections 15 and 16, and those portions of the Summary, Interpretations and Conclusions and Recommendations that pertain to those Sections of the Technical Report.

I am independent of NovaGold Resources Inc. as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Project intermittently since 2007, performing mine planning work and reviewing mine planning work performed by others.

I have read NI 43–101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the Sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

“Signed and sealed”

Jay Melnyk, P.Eng.

Dated: 12 September 2011

CERTIFICATE OF QUALIFIED PERSON

Dana J. Rogers, P.E.
Lemley International
604 N. 16th Street
Boise, ID 83702
Tel: (208) 345-5226
Fax: (208) 345-5254
djrogers@lemleyinternational.com

I, Dana J. Rogers, P.E., am employed as a Principal Tunnelling Engineer, with Lemley International.

This certificate applies to the technical report entitled "Galore Creek Project British Columbia NI 43-101 Technical Report on Pre-Feasibility Study" that has an effective date of 27 July 2011 (the "Technical Report").

I am a licensed Professional Engineer, Mining, in the State of Idaho, USA. I graduated from the University of Utah at Salt Lake City, Utah in 1979 with Bachelor of Science Degrees in both Mining Engineering and in Geology.

I have practiced my profession continuously since 1979 and have been involved in the design, construction and construction management of underground mines and tunnels throughout my career.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I visited the Galore Creek Project (the "Project") from 15 to 17 June 2011.

I am responsible for Sections 18.8 and 18.9, and those portions of the Summary, Interpretations and Conclusions and Recommendations that pertain to those Sections of the Technical Report.

I am independent of NovaGold Resources Inc. as independence is described by Section 1.5 of NI 43–101.

Prior to preparation of this technical report I have had no other involvement with this project.

I have read NI 43–101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the Sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"Signed and sealed"

Dana J. Rogers, P.E.

Dated: 12 September 2011

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1.0 SUMMARY

Galore Creek Mining Corporation (GCMC), NovaGold Resources Inc. (NovaGold) and Teck Resources Limited (Teck) requested AMEC Americas Limited (AMEC) to prepare a Technical Report (the Report) on the results of a pre-feasibility study (the GCMC 2011 pre-feasibility study) for the Galore Creek Copper–Gold–Silver Project (the Project) in British Columbia, Canada.

NovaGold is using the Report in support of a press release dated 27 July 2011, entitled “NovaGold Announces Prefeasibility Study Results for Galore Creek Project”.

The Project is a 50:50 partnership between NovaGold Canada Inc. (a wholly-owned subsidiary of NovaGold) and Teck Metals Ltd (a wholly-owned subsidiary of Teck). The partners use an operating company, GCMC, to manage the Project. For the purposes of this Report, GCMC is used as a synonym for the partnership.

GCMC commissioned a team of engineering firms and consultants to determine the engineering and environmental requirements, and financial viability of the Project, and collated these data, together with contributions from GCMC, into a pre-feasibility document (the GCMC 2011 pre-feasibility study report). AMEC completed this Report as an independent assessment of the the GCMC 2011 pre-feasibility study report, with the assistance of Lemley International (Lemley) who reviewed the proposed access tunnel design. All monetary units in the Report are in Canadian dollars unless otherwise specified.

1.1 Key Outcomes

The GCMC 2011 pre-feasibility study report incorporates an increase in scale and redesign of the Project from previous configurations. AMEC reviewed the supporting data in the GCMC 2011 pre-feasibility study report, and has restated the results of the financial analysis as a consequence of this review (Table 1-1).

Additional AMEC outcomes include:

- Proven and Probable Mineral Reserves of 528 Mt grading 0.58% Cu, 0.32 g/t Au and 6.02 g/t Ag
- Measured and Indicated Mineral Resources exclusive of Mineral Reserves of 286.7 Mt grading 0.33% Cu, 0.27 g/t Au and 3.64 g/t Ag
- Inferred Mineral Resources of 346.6 Mt grading 0.42% Cu, 0.24 g/t Au and 4.28 g/t Ag

Table 1-1: Key Outcomes Table

Summary of Financial Results	Units	Life of Mine
Copper payable	klb	5,950,000
Gold payable	koz	3,850
Silver payable	koz	56,1005
Total cash costs	$/lb	1.83
Secondary metal credit	$/lb	(1.04)
Cash costs net of credits (C1 Net Direct Cash Cost)	$/lb	0.79
Cumulative net after-tax cash flow	$M	5,120
After-tax internal rate of return	%	7.4%
After-tax net present value @ 7%	$M	137
Mine life (including one year of pre-production)	Years	18.5
After-tax payback period	Years	7.8
Total start-up capital	$M	5,160
Total LOM capital (inc.$88.7 M closure cost)	$M	5,840

- Assumed production rate of 34.6 Mt/a, and a mine life of 17.6 years (not including a year of pre-production, or 18.5 years including a pre-production year
- Average annual metal production over the life-of-mine (LOM) of 322,000 thousand pounds of copper, 208 thousand ounces of gold and 3,040 thousand ounces of silver
- Total capital cost of $5,840 M, including start-up capital costs of 5,160 M, sustaining capital costs of $552 M and closure costs of $88.7 M
- LOM cash cost per pound of payable copper of $0.79
- After-tax Project net present value (NPV) at a discount rate of 7% is $137 M using Base Case metal pricing of US$2.65/lb Cu, US$1,100/oz Au and US$18.50/oz Ag
- After-tax internal rate of return (IRR) of 7.4%
- Cumulative, undiscounted, after-tax cash flow for the Project of $5,120 M
- After-tax payback period of 7.8 years.

1.2 Location, Climate, and Access

The Galore Creek Project is located approximately 70 km west of the Bob Quinn airstrip on Highway 37, 150 km northeast of the Port of Stewart, and 370 km northwest of the town of Smithers, British Columbia, Canada, within the Tahltan Nation Traditional Territory.

The Project area is characterized by cold winters and short, cool, summers. Precipitation begins to fall as snow in early October and continues until the end of May. A basin average precipitation for the whole Galore Creek Valley watershed was estimated to be in the order of 3,000 mm.

The Galore Creek Project is currently not accessible by road. The closest Provincial road to the proposed mine site is Highway 37, from which a mine access road will be constructed. The access road will be used to transport employees to and from the mine and plant sites, and to deliver mine capital equipment and mine operating consumables.

The mining operation and associated waste rock facilities will be located in the Galore Creek Valley, whereas the plant and tailings facilities will be in the adjacent West More Valley. A tunnel and a short section of the mine access road will connect the two facility areas.

Smithers is the nearest major supply centre to Galore Creek. At present, most personnel, supplies, and equipment are staged from the Bob Quinn airstrip, on the Stewart-Cassiar Highway (Highway 37) and transported via helicopter to the Galore Creek camp.

1.3 Agreements and Royalties

On May 23, 2007, NovaGold and Teck announced a 50:50 partnership to develop the Galore Creek property. On August 1, 2007 the Galore Creek Partnership was established to develop the Galore Creek mine and created the jointly-controlled operating company, GCMC. The agreement was amended in 2007, and again in 2009. As Teck has expended the required moneys under the agreement, from June 2011 forward, all costs will be met as equal shares by the Partnership participants.

Upon reaching certain agreed financial targets, and subject to positive mine operating cash flow, the Tahltan Heritage Trust Fund will receive the greater of $1 M or a 0.5% to 1.0% net smelter royalty (NSR) each year. The agreement will remain in effect throughout the life of the Galore Creek Project and will be binding on any future operator of the mine. This NSR payment is incorporated into the Project financial analysis.

1.4 Mineral Tenure and Surface Rights

The Project consists of 264 mineral claims, totalling 118,911.88 ha, held in the name of GCMC. Contiguous claims within the Galore Creek property have had assessment work filed on them. Assessment work was not filed for claims that were not contiguous.

Information from a land-management expert retained by GCMC supports that the mining tenure held is valid and is sufficient to support declaration of Mineral Resources and Mineral Reserves. Claims are a combination of map-staked and ground-staked.

The Project falls within the boundaries of the Cassiar Iskut-Stikine Land and Resource Management Plan (LRMP) which was finalized in May 2000. Mineral exploration and development are accepted activities within the Coastal Grizzly Salmon Management Zone, including road access where needed.

Two guide outfitter territories and seven registered trap lines overlie the Galore Creek deposits and planned access road.

1.5 Environment, Permitting and Socio-Economics

The Galore Creek Project received its Environmental Assessment (EA) approval in February 2007. The Project's first permits were obtained in May 2007, and in June 2007, GCMC received final Federal approval for the Project as envisaged in the 2007 EA. However, the new Project design and configuration is different from that which was permitted under the original EA Certificate and received Federal approval.

It is anticipated that a new EA process will be requested by the regulators. This is likely to require parallel and harmonized reviews by both the BC Environmental Assessment Office (BCEAO) and the Canadian Environmental Assessment Agency (CEAA). A comprehensive study report is also likely to be required through CEAA. It is anticipated that the entire EA review process will require approximately two years from submission of a project description to issuance of a new EA Certificate (by the BC government) and a decision by the federal Minister of Environment.

One significant potential addition to the scope of the EA will be the inclusion of a port facility. This would also require assessing the transportation of the concentrate from the plant site to the port by road, as well as the alternative of using a pipeline. The new EA may include assessing the cumulative effects of the Galore Creek Project in connection with other projects that have been developed or are proposed for development since the previous EA was completed. This will include, at a minimum, the Northwest Transmission Line, the Forrest Kerr hydroelectric project, the concentrate transport from Yukon Zinc's Wolverine mine, and the Red Chris project.

The existing Special Use Permit (SUP) for construction of the access road remains valid as long as there are no proposed changes to the SUP, thereby permitting GCMC to continue to build the access road. Existing permits associated with the existing construction camps, including water use and waste discharge, will continue to be maintained. All other Project permits will have to be applied for following completion of the EA process, although the time-critical permits, such as those needed for starting the tunnelling can be prepared concurrent with the EA such that there should be little lag time following EA certification before tunnelling could begin.

The Galore Creek Project is located within the territory of the Tahltan Nation. The Project access route via Highway 37 will be through the Skii Km Lax Ha Traditional Territory and the Gitanyow Traditional Territory to the south. The proposed port facility is in the District of Stewart located in the Nisga'a Nation's Traditional Territory. . Further along the coast, shipping routes pass through the Gitxaala and Haida Nations Traditional Territories. It will be critical in the period leading up to the EA review process that GCMC meaningfully engages with all communities of interest, including the First Nations, within whose traditional territory Project facilities may be located. Ongoing discussion with the Tahltan community resulted in the signing of a Participation Agreement on February 10, 2006. The presence of additional First Nations interests will necessitate an expanded communications and consultation program.

GCMC have identified the key Provincial and Federal permits that will be required for construction of a mine.

The estimated total reclamation liability for the Project is estimated at $88.7 M at the end of the mine life. This estimate includes a contingency of 35%, because many elements of mine design are conceptual; a placeholder cost has been included for those components that have not been detailed in the GCMC 2011 pre-feasibility study.

1.6 Geology and Mineralization

The Galore Creek property is interpreted to be an example of an alkalic porphyry copper–gold–silver system.

The Project is situated within the Stikine Terrane, an exotic terrane accreted to the ancestral North American craton. A sequence of Permian, Mississippian and Devonian age calc-alkaline and bimodal flows and volcaniclastic rocks, interbedded carbonate and minor shale and chert, termed the Stikine assemblage, form the basement of the terrane. Unconformably overlying the Stikine assemblage is a succession of Lower to Middle Triassic sedimentary and upper Triassic volcanic rocks.

The Galore Creek Syenite Complex, of Tertiary age, is centered in the west fork of Galore Creek and is 5 km in length and 2 km in width. The deposits are hosted by potassium-enriched volcanic rocks and pipe-like breccias adjacent to syenite stocks and dykes. They are manto-shaped, and trend north to northeast, following either, or both, syenite contacts and structural breaks. To date, 14 deposits and prospects have been identified, five of which, the Central Zone, Southwest Zone, Junction Zone West Fork Zone and Middle Creek Zone are of economic interest.

The Galore Creek property has undergone at least three temporally different mineralizing events. These include the early formation of the nearby Copper Canyon eruptive centre and its associated mineralization; deposition of the Central Zone mineralization at the Central and Junction deposits, and the Butte prospect; and emplacement of the West Fork mineralization at the Southwest and West Fork deposits.

Bornite and generally higher-grade gold are developed in the intense potassic alteration zone, and are associated with magnetite and sparse pyrite. Within the propylitic zone, zones of moderate potassic alteration have developed, and have associated chalcopyrite and pyrite mineralization. External to these potassic zones, but still within the propylitic zone, replacement lodes of gold, silver and base metals have formed.

In the opinion of the QPs, knowledge of the deposit settings, lithologies, and structural and alteration controls on mineralization is sufficient to support Mineral Resource and Mineral Reserve estimation. The mineralization style and setting of the Project deposit is also sufficiently well understood to support Mineral Resource and Mineral Reserve estimation. Prospects and targets are at an earlier stage of exploration, and the lithologies, structural, and alteration controls on mineralization are currently insufficiently understood to support estimation of Mineral Resources.

1.7 Exploration

Work completed prior to NovaGold acquiring the Project in 2003 consisted of geological mapping, reconnaissance stream sediment surveys, soil sampling, pole-dipole resistivity/IP, magnetics, electromagnetics (EM), radiometrics, very low frequency (VLF) and audio frequency magnetics (AFMAG) airborne geophysical surveys, diamond (core) drilling, underground development work in two adits, access road construction, metallurgical testwork, Mineral Resource estimation and mining studies. Work was undertaken by Kennco Explorations (Western) Limited (Kenncott), Stikine Copper Limited (Stikine), Hudson Bay Mining and Smelting Company Limited (Hudson Bay), and Mingold Resources Inc. (Mingold). Collectively information from these programs is termed the “legacy” data.

SpectrumGold Inc. (now NovaGold Canada Inc.), NovaGold and GCMC have completed all work on the Galore Creek deposits since 2003. Exploration activities have included core drilling, ground and airborne geophysical surveys, metallurgical testwork, thesis studies, and Mineral Resource estimation. Mining studies were performed in 2004, 2006, and 2008.

In 2010–2011, GCMC commissioned a pre-feasibility study on the Project, which included multiple trade-off option considerations. The remainder of this Report discusses the results of the GCMC 2011 pre-feasibility study and the review completed by AMEC and Lemley on the data and results.

In the opinion of the QPs, the exploration programs completed to date are appropriate to the style of the deposits and prospects within the Project. The exploration and research work supports the genetic and affinity interpretations.

1.8 Exploration Potential

The Project retains excellent exploration potential. The Project area is host to five defined Mineral Resource areas, seven under-explored copper–gold prospects, and numerous showings and conceptual target areas.

Additional mineralization that supports estimation of Inferred Mineral Resources, but not drilled to sufficient confidence to be included in the GCMC 2011 pre-feasibility study report, exists below the area that hosts the Central Zone Mineral Reserves, and in the Southwest area of the Bountiful Zone.

Alteration and mineralization vectors, together with the lack of precursor intrusions driving the systems at both the Central and West Fork systems, have major exploration significance. GCMC considers that the potential to make a major discovery at depth or even laterally, as with the case of the West Fork deposit, is high.

1.9 Drilling

Approximately 255,601 m has been drilled in 1,078 core holes on the Project since 1961. Over the Project history, a number of different drill companies have been used. Core drilling has been performed at BQ, NQ, HQ or PQ size (36.5 mm, 47.6 mm, 63.5 mm and 85 mm, respectively).

Limited information is available on the legacy drilling programs. Geological, alteration, and mineralization data, together with some geotechnical data appear to have been collected. Some drill programs have existing collar data, and the 1991 drill program has down-hole survey data. Drilling protocols used in the Stikine and Kennecott legacy programs are assumed to have been in line with industry standards at the time; however, this has not been confirmed.

Core recovery has been evaluated by campaign and generally improves throughout the exploration history of the Project. Recovery is typically poor in the near surface environment where gypsum and anhydrite veinlets have been dissolved and the rock is “broken”.

Standardized logging forms and geological legends were developed for the SpectrumGold, NovaGold, and GCMC drill programs. Geotechnical logs were completed in sequence for these programs prior to the geological logging. SpectrumGold, NovaGold, and GCMC geological logging used standard procedures and collected information on mineralization, lithic breaks, alteration boundaries, and major structures.

Upon completion, drill hole collars from the SpectrumGold, NovaGold, and GCMC were surveyed using a differential GPS. Down-hole surveys were carried out for the SpectrumGold, NovaGold, and GCMC drill programs for dip and deviation using a number of different instruments, including Sperry Sun, IceField, Reflex Easy Shot, and Gyroscope tools. Surveys were performed by a contractor. Magnetic declination factors were applied.

SpectrumGold, NovaGold, and GCMC sample intervals were determined by the geological relationships observed in the core and limited to a 3 m maximum length and a 1 m minimum length. An attempt was made to terminate sample intervals at lithological and mineralization boundaries.

In the opinion of the QPs, the quantity and quality of the lithological, geotechnical, collar survey and downhole survey data collected in the SpectrumGold, NovaGold, and GCMC exploration and infill drill programs completed on the Project are sufficient to support Mineral Resource and Mineral Reserve estimation.

1.10 Sample Analysis and Security

Kennecott/Stikine, SpectrumGold, NovaGold and GCMC exploration and infill core samples were analysed by independent laboratories using industry-standard methods for gold, copper, and silver analysis. A number of different laboratories have been used on the Project. Since 2004, ALS Chemex Laboratories (ALS Chemex) in Vancouver, BC, has been the primary laboratory. ALS Chemex holds ISO 9001-certification.

Metallurgical testwork has been completed at a number of laboratories, but primarily by G&T Metallurgical (G&T) laboratories in BC. Metallurgical laboratories typically do not hold accreditations.

Sample preparation for pre-2003 drill programs are assumed to be in line with industry-standard methods at that time although this has not been confirmed. Information on the QA/QC protocols for legacy programs is currently limited.

Sample preparation for SpectrumGold, NovaGold and GCMC drill programs are in line with industry-standard methods for porphyry gold–copper–silver deposits.

For the SpectrumGold, NovaGold, and GCMC programs, samples were crushed, dried, and a 250 g split pulverized to greater than 85% passing 75 µm. Gold assays were determined using fire analysis followed by an atomic absorption spectroscopy (AAS) finish. The lower detection limit was 0.005 ppm Au; the upper limit was 1,000 ppm Au. An additional 34-element suite was assayed by inductively-coupled plasma optical emission spectroscopy (ICP_AES) methodology, following nitric acid aqua regia digestion. The copper analyses were completed by AAS, following a triple-acid digest. Values over the detection limits were rechecked using nitric acid aqua regia digestion of a 0.4–2.0 g sample followed by AAS finish.

In 2005, NovaGold obtained a total of 916 acid-soluble copper assays from 31 drill holes. There have been no additional acid-soluble copper assays performed since that date.

SpectrumGold, NovaGold, and GCMC maintained a quality assurance and quality control (QA/QC) program for the Project. This comprised submission of analytical certified reference materials (CRMs), duplicate and blank samples. QA/QC submission rates meet industry-accepted standards of insertion rates. The QA/QC program results do not indicate any problems with the analytical programs, therefore the gold, copper, and silver analyses from the SpectrumGold, NovaGold, and GCMC core drilling programs are suitable for inclusion in Mineral Resource and Mineral Reserve estimation. Gold, copper, and silver analytical data from the pre-SpectrumGold, NovaGold and GCMC drill programs are sufficiently reliable to support Mineral Resource and Mineral reserve estimation, but due to the lack of appropriate supporting QA/QC results, the data should not be used to support classification of Measured blocks

No information is currently available for sample security and chain-of-custody protocols used in the legacy drill programs. Sample security for the SpectrumGold, NovaGold, and GCMC drill programs has relied upon the fact that the samples were always attended or locked in the logging facility. Chain-of-custody procedures for the

SpectrumGold, NovaGold, and GCMC drill programs consisted of filling out sample submittal forms that were sent to the laboratory with sample shipments to make certain that all samples were received by the laboratory.

1.11 Data Verification

A number of data verification programs and audits have been performed over the Project history, primarily by independent consultants in support of compilation of technical reports on the Project and in support of mining studies. Data verification checks were performed in 2003, 2004, and 2006 by third-party consultants and in 2008 and 2011 by AMEC.

The 2008 AMEC audit identified some minor errors in the Project database which were considered to require fixing, but the identified errors were typically considered to be non-material for Mineral Resource estimation purposes. Limitations were placed by AMEC on the use of pre-2000 samples due to the lack of QA/QC support and possible grade biases between pre and post 2000 samples. In addition, the data entry error rate in the specific gravity (SG) database was above that considered acceptable for a pre-feasibility or feasibility-level study.

AMEC performed a second audit in 2011. GCMC were found to have reviewed and corrected the SG database. No additional QA/QC data was available to support pre-2000 samples. A review of possible grade biases for legacy assay results noted in the 2008 audit indicated that no correction should currently be applied for legacy copper, gold, and silver results. The biases interpreted from the AMEC review may, in part, be due to spatial variability (distances >10 m) and lithological variability (composite pairs across lithological boundaries). The copper and gold biases are generally expected to cause an overall underestimation of grade in the Mineral Resource estimate. The high silver bias could result in an overestimation of the containd silver but as silver is considered a minor economic contributor to the project this impact is considered minor. Additional work is warranted to quantify the biases and to confirm the impact on the estimated Mineral Resources.

Conclusions of the 2011 audit were that the Galore Greek drill collar, down-hole survey, and assay data were of sufficient quality to support Mineral Resource and Mineral Reserve estimation.

Pre-2000 samples were considered suitable to support Mineral Resource estimations but with limitations. Estimated blocks supported primarily by pre-2000 samples are limited to Indicated classification.

Sample data collected adequately reflect deposit dimensions, true widths of mineralization, and the style of the deposits. Drill data are typically verified prior to Mineral Resource and Mineral Reserve estimation, by running a software program check to ensure the estimation data are free from errors such as overlapping intervals, and drill hole depths that are greater than the survey depth.

AMEC considers that a reasonable level of verification has been completed, and that no material issues would have been left unidentified from the programs undertaken. The QPs, who rely upon this work, have reviewed the appropriate reports, and are of the opinion that the data verification programs undertaken on the data collected from the Project adequately support the geological interpretations, the analytical and database quality, and therefore support the use of the data in Mineral Resource and Mineral Reserve estimation.

1.12 Metallurgical Testwork

Since 2003, the majority of metallurgical testwork was completed at G&T Metallurgical Services laboratories in Kamloops, British Columbia. GCMC representatives were involved in all aspects of the testwork, including sample selection, test design, and data review and interpretation. Periodic laboratory visits were completed in 2010 to verify that tests were being carried out at an acceptable standard, and no issues were identified. All tests have been documented according to standard G&T Metallurgical standard practices. Standard tests completed included open circuit rougher-cleaner flotation tests, locked cycle tests, Bond ball mill work index determinations, SMC hardness determinations, and JK drop-weight test index determinations. All tests were completed using industry standard methods.

The design for the process plant is based on processing the ore through a conventional crushing, grinding and flotation plant using standard proven processes and equipment. The plant is designed to handle a blend of ore from the various zones of the Galore Creek deposit.

Using results of flotation tests conducted during three campaigns in 2005/6, 2008/9 and 2010, empirical relationships to estimate recoveries for copper, silver, and gold were derived as a function of head grade. Separate models were prepared for material types defined as Standard or Oxidized/Near Surface material consistent with the geological block model.

The recovery relationships are shown in Table 1-2. The recovery relationships in Table 1-2 are different to those used by GCMC in the GCMC 2011 pre-feasibility study report. AMEC adjusted the average recovery estimates of GCMC down by 1% for copper, 3% for gold and 7% for silver. AMEC recommends that additional metallurgical tests be completed to establish recovery curves to support more detailed studies.

Table 1-2: Process Recovery Relationships

Recovery (%)	Standard Material	Oxidized/Near Surface Material
Copper	7.66*Ln([Head Cu(%)])+94.34 *(cap at 95%)*	([OxRConc]*([Head Cu(%)]-0.18)/([Head Cu(%)]*([OxRConc]-0.18))*94.8 where: ([OxRConc] =7.2*[Head Cu(%)]+1.6 *(cap at 95%)*
Gold	8.1*Ln [HeadAu(g/t)]+78 *(cap at 90%)*	8.1*Ln [HeadAu(g/t)]+78 *(cap at 90%)*
Silver	19.7* Ln([HeadAg(g/t)])+26 *(cap at 90%)*	14.5* Ln([HeadAg(g/t)])+28 *(cap at 75%)*

1.13 Mineral Resource Estimate

Five zones were modelled: the Central Zone (including the Bountiful deposit), Southwest Zone, Junction/North Junction Zone, Southwest Zone, and West Fork Zone.

Grade estimations for copper, gold and silver were completed utilizing ordinary kriging (OK) methods. An inverse distance to the second power (ID2) and nearest-neighbour (NN) models were constructed as checks.

After review of the Mineral Resource estimate, AMEC revised some criteria supporting the conceptual pit shell, including drill spacing for classification purposes, the commodity prices, and NSR value; metallurgical recoveries were also revised downward. Commodity prices used by AMEC to state the Mineral Resources were US$2.50/lb copper, US$1,050/oz gold, and US$16.85/oz silver, and the NSR cut-off was $10.08/t milled. As a consequence, AMEC updated the Mineral Resources which were used in the GCMC 2011 final pre-feasibility study report.

Mineral Resources take into account geological, mining, processing and economic constraints, and have been confined within appropriate Lerchs–Grossmann (LG) pit shells, and therefore are classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves.

Mineral Resources, exclusive of Mineral Reserves, are stated in Table 1-3 using an NSR cut-off grade of $10.08/t milled.

The Qualified Person for the Mineral Resource estimate is Greg Kulla, P.Geo., an AMEC employee. Mineral Resources have an effective date of 11 July 2011. Mineral

Resources that are not Mineral Reserves do not have demonstrated economic viability.

Factors which may affect the Mineral Resource estimates include: commodity price and exchange rate assumptions, assumptions used to estimate metallurgical recoveries, pit slope angles, and SG values assumed for the broken rock.

A legacy pulp re-assay program intended to quantify negative assay biases for copper and gold, and positive biases for silver was initiated in July 2011 and may result in elimination of the biases or support development and application of a correction factor for the legacy assay results. This may result in local changes to the classification assigned to some Mineral Resource blocks.

Table 1-3: Galore Creek Mineral Resource Table, Effective Date 11 July 2011, G. Kulla, P.Geo.

Category	Tonnage (Million tonnes)	Cu Grade (%)	Au Grade (g/t)	Ag Grade (g/t)	Contained Cu (Billion pounds)	Contained Au (Million ounces)	Contained Ag (Million ounces)
Measured	39.5	0.25	0.39	2.58	0.22	0.50	3.27
Indicated	247.2	0.34	0.26	3.81	1.85	2.04	30.26
Total Measured and Indicated	286.7	0.33	0.27	3.64	2.07	2.53	33.54
Inferred	346.6	0.42	0.24	4.28	3.23	2.70	47.73

Notes to Accompany Mineral Resources Table

1. Mineral Resources are exclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
2. Mineral resources are contained within a conceptual Measured, Indicated and Inferred optimized pit shell using the same economic and technical parameters as used for Mineral Reserves. Tonnages are assigned based on proportion of the block below topography. The overburden/bedrock boundary has been assigned on a whole block basis. Commodity prices used to constrain the Mineral Resources are US$2.50/lb copper, US$1,050/oz gold, and US$16.85/oz silver
3. Mineral resources have been estimated using a constant NSR cut-off of C$10.08/t milled. The Net Smelter Return (NSR) was calculated as follows: NSR = Recoverable Revenue – TCRC (on a per tonne basis), where: NSR = Diluted Net Smelter Return; TCRC = Transportation and Refining Costs; Recoverable Revenue = Revenue in Canadian dollars for recoverable copper, recoverable gold, and recoverable silver using silver using the economic and technical parameters used for mineral reserves.
4. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content
5. Tonnage and grade measurements are in metric units. Contained gold and silver ounces are reported as troy ounces, contained copper pounds as imperial pounds.

1.14 Mineral Reserve Estimate

Mineral Reserves were modified from Measured and Indicated Mineral Resources by taking into account geologic, mining, processing, and economic parameters and therefore are classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves.

AMEC restated the Mineral Reserves which were used in the GCMC 2011 final pre-feasibility study. Changes included resource model classification revisions, different commodity prices, NSR cut-off value, and a downward revision of metallurgical recoveries.

Mineral Reserves are reported at commodity prices of US$2.50/lb copper, US$1,050/oz gold, and US$16.85/oz silver, and have an effective date of July 11, 2011.

The Qualified Person for the Mineral Reserve estimate is Jay Melnyk, an AMEC Associate. Mineral Reserves are summarized in Table 1-4.

Factors which may affect the Mineral Reserve estimates include commodity price and exchange rate assumptions, mill throughput of the identified ore types may prove to be higher or lower than modelled, and variations from the 2011 GCMC pre-feasibility study infrastructure design, construction schedules, and budget estimates.

Table 1-4: Mineral Reserve Statement, Effective Date 11 July 2011, Jay Melnyk, P.Eng.

	Tonnes	Diluted Grade					
	Mt	Cu (%)	Au (g/t)	Ag (g/t)	Contained Cu (Billion pounds)	Contained Au (Million ounces)	Contained Ag (Million ounces)
Proven	69.0	0.606	0.520	4.94	0.9	1.15	11.0
Probable	459.1	0.582	0.291	6.18	5.9	4.30	91.2
Total Proven and Probable	*528.0*	*0.585*	*0.321*	*6.02*	*6.8*	*5.45*	*102.1*

Notes to Accompany Mineral Reserves Table

1. Mineral Reserves are contained within Measured and Indicated pit designs using metal prices for copper, gold and silver of US$2.50/lb, US$1,050/oz, and US$16.85/oz, respectively
2. Appropriate mining costs, processing costs, metal recoveries and inter ramp pit slope angles varying from 42º to 55º were used to generate the pit phase designs
3. Mineral Reserves have been calculated using a 'cashflow grade' ($NSR/SAG mill hr) cut-off which was varied from year to year to optimize NPV. The net smelter return (NSR) was calculated as follows: NSR = Recoverable Revenue – TCRC (on a per tonne basis), where: NSR = Net Smelter Return; TCRC = Transportation and Refining Costs; Recoverable Revenue = Revenue in Canadian dollars for recoverable copper, recoverable gold, and recoverable silver using metal prices of US$2.50/lb, US$1,050/oz, and US$16.85/oz for copper, gold, and silver, respectively, at an exchange rate of CAD$1.1 to US$1.0; Cu Recovery = Recovery for copper based on mineral zone and total copper grade; for Mineral Reserves this NSR calculation includes mining dilution. SAG throughputs were modeled by correlation with alteration types. Cashflow grades were calculated as the product of NSR value in $/t and throughput in t/hr
4. The life of mine strip ratio is 2.16
5. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content
6. Tonnage and grade measurements are in metric units. Contained gold and silver ounces are reported as troy ounces, contained copper pounds as imperial pounds.

1.15 Proposed Mine Plan

The proposed Project will be a conventional, large-tonnage, open-pit operation with approximately 528 Mt of ore processed over the life-of-mine, at a nominal daily throughput of 95,000 t/d.

Because of the revised Mineral Resource estimate, and changes to the metallurgical recovery assumptions and different commodity price assumptions, AMEC redeveloped the mine plan for the Project.

The mine plan for Galore Creek deposits was based on mining six separate phases that were developed from detailed designs based on optimized Whittle^{TM} pit shells for four open pits, at Central, Junction, West Fork, and Southwest.

The production schedule contains one year of pre-production and envisages a mine life of 17.6 years, exclusive of that pre-production year. Annual mine production of ore and waste will peak at 136 Mt/a with a LOM waste/ore stripping ratio of 2.16-to-1 (refer to Table 1-5).

Table 1-5: Life-of-Mine Planned Production

Mine Life	Average LOM Cut-off Grade ($/hr)	Total Moved over LOM (kt)	Total LOM Waste (kt)	LOM PAG Waste (kt)	LOM NPAG Waste (kt)	LOM Overburden Waste (kt)
17.62	30,726	1,666,811	1,138,842	283,547	719,691	135,605
LOM Ore to mill (kt)	**LOM Ore to Stockpile (kt)**	**LOM Reclaimed from Stockpile (kt)**	**LOM Strip Ratio**	**LOM Copper Grade (%)**	**LOM Gold Grade (g/t Au)**	**LOM Silver Grade (g/t Ag)**
527,969	24,988	24,988	2.16	0.59	0.35	6.02

The waste/ore split is defined within the scheduling process to maximize NPV. Broken ore represents approximately 35% of the total ore feed. Lower-grade ore that must be released at the same time as higher-grade ore will be sent to a coarse ore stockpile near the crusher and will be milled later in the schedule or during weather and operations delays.

A variable cut-off grade strategy was used to design the life-of-mine. The cut-off grade strategy incorporates a stockpile capacity of 20 Mt. In order to properly reflect the high variability in mill throughput for the different rock types, a cash flow grade item was calculated, which is a function of both the NSR and the mill throughput by rock type, and has units of $/SAG mill hour.

The marginal cut-off grade equation is:

Marginal cut-off grade = $0/hr (NSR = $3.37/t)

and is the theoretical minimum grade of material that can be processed as it ensures that variable costs are covered. The variable cut-off grade strategy ensures that the mill is processing material in a manner that maximizes the NPV of all future cash flows.

Accompanying the six mining phases are two non-potentially acid-generating (NPAG) dumps and three potentially acid-generating (PAG) dumps, all of which will be located in close vicinity to the Central pit at Galore Creek. The NPAG waste dumps will initially be situated out of the valley floor until PAG rock is mined then PAG waste will be deposited in the valley bottom which will be flooded at mine closure.

Mining equipment selection was based on the mine production schedule and equipment productivities, as well as consideration of workforce and operating hours. The operation will use a conventional truck-and-shovel fleet. In the opinion of the AMEC QPs, the fleet is appropriate to the planned production schedule.

Water management of the Galore Creek watershed will be a major design challenge. A number of water control structures are planned, including diversion channels, and closure and sedimentation dams. Based on preliminary modelling, water quality will be suitable for direct discharge with no requirement for water treatment.

Geohazards are present in the Galore Creek Valley and will require careful consideration in waste and water management throughout the life of the mine and during the reclamation and closure period.

1.16 Process Design

The design criteria for the process plant were based on processing ore through a conventional crushing, grinding and flotation plant using conventional processes and equipment. The plant is proposed to handle a blend of ore from the various zones of the Galore Creek deposits. A no-stockpile blending strategy will be utilized to deliver consistent copper grades to optimize process plant performance.

A production summary for the process plant is included as Table 1-6 and a payable metal summary for the Project is included in Table 1-7.

Run-of-mine (ROM) ore will feed a single gyratory crusher in the Galore Creek Valley. Crushed ore will then be transported via three overland conveyors from Galore Creek Valley, through the access tunnel to the West More Valley, to a single coarse ore stockpile near the mill site that will be adjacent to the proposed West More tailings facility.

Apron feeders will reclaim ore from the coarse ore stockpile to feed a semi-autogenous grinding (SAG) mill. SAG discharge material will be screened, with coarse pebbles reporting to two cone crushers before reporting back to the SAG feed stream. SAG discharge undersize material will be split between three ball mills in closed circuit with hydrocyclones. The hydrocyclone overflow, with a target 80% passing size of 200 µm, will report to flotation for further processing.

Standard process reagents will be added prior to flotation to allow efficient separation of valuable minerals. Some ores will require the addition of a talc depressant to maximize recovery while achieving acceptable concentrate grades.

The flotation circuit will consist of two parallel rougher banks, with the rougher concentrate reporting to regrinding. Regrind will occur in four vertical tower mills, with a gravity concentrator installed parallel to the tower mills to remove any build up of high-density material in the regrinding circuit. The gravity concentrate will report to the final copper concentrate stream.

Table 1-6: Life-of-Mine Process Plant Summary

Parameter	Unit	Copper Concentrate
Concentrate produced	kt (dry)	10,002
Moisture content	%	8.0
Copper recovery	% Cu	91
Gold recovery	% Au	73
Silver recovery	% Ag	64
Concentrate grade of copper	% Cu	28
Concentrate grade of gold	g/t Au	12.4
Concentrate grade of silver	g/t Ag	205
Recovered copper	M lbs Cu	6,174
Recovered gold	000 oz Au	3,983
Recovered silver	000 oz Ag	65,829

Table 1-7: Payable Metal Summary

	Copper (000 lb Cu)	Gold (000 oz Au)	Silver (000 oz Ag)
First Five Years Average	383,199	262	3,427
Life of Mine Average	343,767	223	3,233
Life of Mine Total	5,947,946	3,849	56,125

The remainder of the flotation circuit will consist of three stages of cleaning utilizing mechanical tank-type flotation cells with forced air. Third-cleaner concentrate will report to a concentrate thickener for dewatering.

Rougher tailings will report by gravity to the tailings storage facility, either directly or through a hydrocyclone system that will produce a coarse sand product for tailings dam construction. Cleaner tailings will be deposited sub-aqueously as a separate stream in the tailings storage facility.

Process water for the mill facility will be reclaimed from the tailings pond, with minimal fresh makeup water being supplied by wells located in the vicinity. The wells will also be used for the production of potable water, mixing of reagents, and other uses. Mill reagents, grinding steel, and maintenance supplies will be delivered to the site by transport truck and stored within the mill as required.

Thickened concentrate will be pumped approximately 71 km to a remote filter plant and truck-loading facility to be located near the junction of the mine access road and Highway 37. The filter plant facility will use recessed plate pressure filters to remove water from the concentrate to reach moisture levels below transportable limits. Water produced from the process will be treated in a water treatment plant, complete with multiple stages of filtration for solids removal, to meet discharge water quality standards. Treated water will be discharged to the Iskut River. Filtered concentrate will be loaded onto trucks for transportation to a port facility to be constructed in the town of Stewart, BC for shipment to various international destinations.

1.17 Tailings Impoundment Management

The West More tailings facility will be located at the upper limits of the More Creek watershed at elevations above 1,100 m. The proposed configuration of the West More tailings facility includes three dams, a Main Dam and two saddle dams, the East and West Saddle dams. Submergence of the cleaner tailings is all that will be required to mitigate acid-rock drainage (ARD) risk for the West More tailings impoundment.

The dams and impoundment will accommodate up to 678 Mt of tailings, although storage for only 510 Mt is required for the current mine plan.

1.18 Planned Project Infrastructure

The Project will require construction of significant infrastructure to support the planned producing facilities.

The Project is currently not accessible by road. The closest Provincial road to the mine site is Highway 37. A controlled-access road is planned from this highway to the proposed mine site, a road distance of 69 km. Controlled access is required in order to protect the health and safety of company personnel and the public, and to protect the environment.

GCMC has an existing Special Use Permit for the construction of the access road following the route permitted under the existing EA. A section of the access road from Highway 37 (Km 0) to approximately Km 40 was constructed during a previous Project phase and is currently in service. The entire road would require upgrading to meet final design criteria; the first 8 km would be upgraded to a dual-lane route to access the filter plant at Kilometre 8 and fuel off-loading facility. The balance of the access road would be single-lane, with occasional pullouts.

The requirement for the mine access and ore conveyor tunnel to provide both permanent access for large components of mining equipment for start-up and ongoing operations, as well as conveyor haulage during mining operations, has necessitated the need for a large tunnel. The GCMC 2011 pre-feasibility study design consists of a 13.6 km-long tunnel from a South Portal located in a limestone cliff face on the side of a bluff at the upper end of the Sphaler Creek valley, to a North Portal located in a sloping volcanic rock outcrop in the upper reaches of the East Fork of Galore Creek. The proposed tunnel is aligned under high rock cover of more than 600 m over a significant portion (75%) and with a maximum rock cover of 1,250 m.

The approach GCMC describes for constructing the tunnel would be to drive headings from both ends, using an open gripper, high-performance, main beam tunnel boring machine (TBM) starting from a 165 m-long, conventionally-mined starter tunnel at the South Portal (access road end) and a helicopter-supported, drill and blast heading from the North Portal.

Lemley reviewed the tunnel construction plans. Lemley considered that while the risks identified point to a very challenging tunnelling project, there is nothing inherent in these risks that has not been dealt with successfully on other projects, using either drill/blast and/or TBM methodologies, or which would cause Lemley to render an opinion that the Galore Creek tunnel is not constructible using the approach described by GCMC.

Lemley’s opinion is that additional time should be added to the original schedule outlined in the GCMC 2011 pre-feasibility study report; this results in an approximate four-year long construction period. Lemley estimates a total of 37 months should be allocated for boring/excavating the tunnel and that a 49-month overall tunnel construction duration is appropriate. Lemley advises that the changes adding time to the tunnel construction schedule resulted in a corresponding increase in the cost estimates for the tunnel.

Lemley also judges that there are greater cost risks and schedule risks associated with the use of a TBM on this Project than there would be with two opposing drill and blast operations. Future studies should evaluate replacing the TBM drive with a second high-speed drill and blast heading as a way to reduce risks and better plan a predictable and successful completion for the tunnel. Lemley concluded that if a TBM is used, then that section of the tunnel should be constructed up-grade rather than using the GCMC 2011 pre-feasibility study down-grade design.

Logistically speaking, the most complex and challenging construction phase will be related to the work required in the Galore Creek Valley during tunnel excavation as only air support is available until tunnel break-through. The previous construction activities carried out in the Galore Creek Valley in 2007 utilized construction equipment flown in by helicopter to support the earlier tunnel drill and blast operations which followed a different tunnel alignment that has since been superceded. Helicopter transport will be needed to fly manpower, additional equipment, fuel, and construction materials into the Galore Creek Valley. This will enable activities other than the tunnel drill-and-blast operation to proceed immediately upon receipt of the construction permits.

GCMC will construct a new 287 kV transmission line to supply the power demand at the proposed Galore Creek development. Power for the Project will be provided from the Northwest Transmission Line (NTL) currently being constructed by the Provincial electrical authority, BC Hydro. The 69 km-long power distribution line will run adjacent to the access road.

The transmission/distribution lines will have sufficient capacity to service the power demand for mining and process equipment throughout the life of the mine. The tunnelling operation is a high consumer of power and initially diesel. The tunnel boring machine (TBM) will require 10 to 15 MVA of power, which will be provided from the 287 kV power source. The development of the 287 kV line from Bob Quinn and the 287 kV substation at West More are both activities that require early completion to help reduce the overall diesel consumption. Diesel generators will be required to provide power in the initial stages of construction. Early engineering of the 287 kV substation at West More will enable early procurement of the long delivery substation electrical equipment (i.e., transformers and switchgear in the substations), which will allow power to be connected to the south portal at the earliest opportunity. However, construction of the West More substation will require EA approval.

There will be three permanent camps at the Galore Creek Project. West More will support operations and maintenance personnel associated with the main concentrator facilities and administration building; the Galore Creek Valley camp will support mine operations, and the Km 8 camp will support the filter and dewatering plant.

A diesel storage and pumping facility will be located at Km 8. Diesel will be delivered to the facility by trucks, and then pumped to fuel storage tanks at West More, then delivered by pipeline to the mine site. The supply of diesel to support early construction is critical to both the Galore Creek Valley and West More areas. Prior to any construction work commencing, a secure supply of diesel will need to be established via a long-term supply contract.

Freshwater will be provided from wells. Process water is projected to be sourced from tailings reclaim.

The port site proposed in the GCMC 2011 pre-feasibility is the former Arrow Dock facility, a causeway made of reclaimed land to the southeast of Stewart. The port site will include a concentrate storage and shiploading system. Habitat compensation is considered a key environmental consideration for the development of the port.

It is envisaged that the construction time for the Project will be approximately four years, from the commencement of tunnel boring until the beginning of pre-operational commissioning activities for the mill and associated facilities.

1.19 Markets

GCMC requested a market opinion on the copper concentrate market balance and demand outlook from Teck's internal marketing experts. Teck is of the opinion that the copper concentrate market will remain tight to 2020 due to an increase in smelting capacity ahead of mine production growth. With copper demand projected to grow at a rate of 3.1% per annum out to 2020, Teck predicts that demand will exceed refined production by close to 6.5 Mt in 2020, and therefore that additional mine production will be required to satisfy projected demand.

The conceptual production level will be an average of over 600,000 dmt of copper concentrates produced annually over the Project life. The only element that is of concern as an impurity is fluorine, a low-level deduction for fluorine has been assumed to apply.

The sales plan is to establish long-term contracts for approximately 75% of its minimum long-term production quantity in order to provide stable and reliable sales. GCMC plans that contract durations may extend for as long as 10 years, with most terms fixed in the contract. The marketing strategy will focus on the major custom smelting companies in the world that are logistically practical for the delivery of concentrates. GCMC has not sought expressions of interest or letters of intent from smelters. This will be required to support feasibility-level studies.

1.20 Capital Costs

The capital cost estimate for the Project was developed by GCMC, with input from consultants for specific areas. The capital cost estimates are based on a combination of quotes, vendor pricing, and experiences with similar-sized operations. Capital cost estimates in the GCMC 2011 pre-feasibility study report were noted by GCMC to be reported at a pre-feasibility level where the estimate accuracy range is defined as +25%/-20% (including contingency) and are consistent with an AACE Class 4 estimate. The GCMC 2011 pre-feasibility study estimate includes an 18% contingency allocation.

AMEC considered that the earthworks and tunnelling costs were underestimated in the GCMC 2011 pre-feasibility study and made an upward adjustment of $140 M to cover these areas. This increased the capital cost estimate to $5,115.2 M. Due to changes in the mine plan, AMEC also re-estimated the sustaining capital costs.

When sustaining capital ($552 M) and closure costs ($89 M) are incorporated, the total Project capital cost estimate as restated by AMEC is $5,840 M.

Capital costs, as endorsed by AMEC, are summarized in Table 1-8. Sustaining capital costs are included as Table 1-9.

Table 1-8: Galore Creek Construction Capital Cost Estimate

Description	Initial Capital Estimate ($ millions)
Mine	357
Plant	835
Tunnel	580
Infrastructure	697
Total Direct Costs	**2,470**
Mine and Pre-production Costs	582
Indirect Costs	1,320
Owner's Costs	111
Contingency	678.
Total Capital	**5,160**

Note: Numbers may not sum due to rounding.

Table 1-9: Galore Creek Sustaining Capital Estimate

Description	Sustaining Capital Estimate ($ millions)
Mine	163
Plant	66
Tailings	212
Infrastructure	110
Total	552

Note: Numbers may not sum due to rounding.

1.21 Operating Costs

Operating costs were based on estimates performed by GCMC from first principles for major items, and included allowances or estimates for minor costs. The assumed power cost in the GCMC 2011 pre-feasibility study report is $50/MW-hr and the assumed diesel fuel cost is $1.04/L. Manpower requirements were based by GCMC on industry experience with similar-scaled operations.

AMEC reviewed these estimates, and as a consequence of changes to the mine plan, revised sections of the estimate. AMEC has restated the estimated life-of-mine operating cost as $15.10/t milled.

Operating costs, as endorsed by AMEC, are summarized over the life-of-mine in Table 1-10.

Table 1-10: Average Annual Operating Cost

Area	$/tonne milled
Mine	6.70
Process	5.76
Port	0.16
Site G&A	1.56
Other	0.89
Total	**15.10**

Note: Numbers may not sum due to rounding.

1.22 Financial Analysis

The results of the economic analysis represent forward-looking information that are subject to a number of known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here.

Forward-looking information includes Mineral Reserve estimates, commodity prices and exchange rates, the proposed mine production plan, projected recovery rates, uncertainties and risks regarding the estimated capital and operating costs, uncertainties and risks regarding the cost estimates and completion schedule for the proposed Project infrastructure, in particular the proposed access tunnel, and the need to obtain permits and governmental approvals.

Financial analysis of the Galore Creek Project was carried out using a discounted cash flow (DCF) approach. This method of valuation requires projecting yearly cash inflows (or revenues) and subtracting yearly cash outflows (such as operating costs, capital costs, royalties, and taxes). The resulting net annual cash flows are discounted back to the date of valuation and totalled in order to determine the Net Present Value (NPV) of the Project at selected discount rates. The internal rate of return (IRR) is expressed as the discount rate that yields an NPV of zero.

The payback period is the time calculated from the start of Project cash flows until all initial capital expenditures have been recovered.

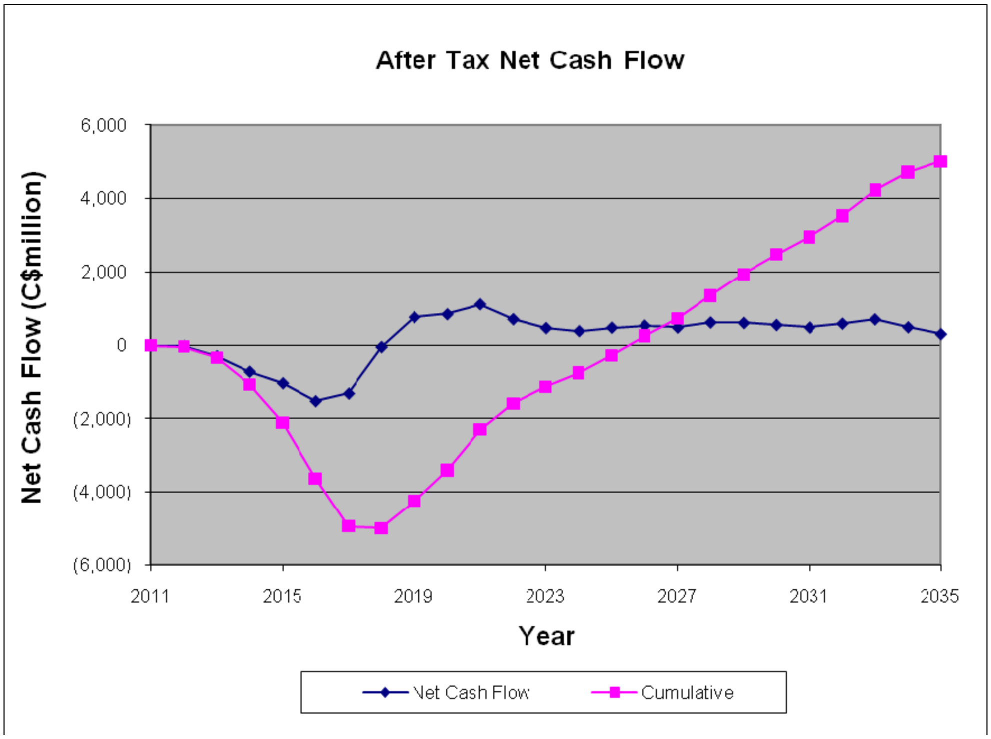
The financial analysis for the Galore Creek Project, using a discount rate of 7%, indicates that the after-tax Project NPV is $137 M and the IRR is 7.4%. The cumulative undiscounted after-tax cash flow value for the Project is $5,120 M and the payback period is 7.8 years.

The financial results are summarized in Table 1-11 for the life-of-mine (LOM). C1 cash costs are as defined by Brook Hunt and are shown as costs per pound of payable copper. The after-tax annual cash flows and cumulative cash flow are depicted in Figure 1-1.

Table 1-11: Summary of Financial Results

Summary of Financial Results	Unit	LOM
Copper payable	klb	5,950,000
Gold payable	koz	3,850
Silver payable	koz	56,100
Total cash costs	$/lb	1.83
Secondary metal credit	$/lb	(1.04)
Cash costs net of credits (C1 Net Direct Cash Cost)	$/lb	0.79
Cumulative net after-tax cash flow	$M	5,120
After-tax internal rate of return	%	7.4%
After-tax net present value @ 7%	$M	137
Mine life (including one year of pre-production)	Years	18.5
After-tax payback period	Years	7.8
Total start-up capital	$M	5,160
Total LOM capital (inc. $88.7 M closure cost)	$M	5,840

Figure 1-1: After-Tax Net Cash Flow (Undiscounted)



The Project is most sensitive to changes in metal price, secondly to changes in exchange rate, less so to changes in operating cost and least sensitive to capital cost changes.

1.22.1 Real Option Sensitivity Analysis

An alternative sensitivity case for the economic analysis using a method called "Real Options" was assessed. Real Options is an alternative method of calculating the NPV of a proposed project. The Real Option method calculates a NPV (Real Option NPV) value using the same inputs as the more common method of discounted cash flow, but with adjustments for uncertainty and time.

Ernst & Young LLP (Ernst & Young) was retained by NovaGold to develop an evaluation model that calculated a Real Option NPV for the Project using the same Project capital and operating inputs as used in the AMEC financial model, which was created using discounted cash flow assumptions.

Table 1-12 presents after-tax cumulative net cash flow and the after-tax conventional discounted cash flow and Real Option NPVs estimated for the Galore Creek Project by the Ernst & Young evaluation model. The Ernst and Young Real Option after-tax NPV is $811 M, as compared to the $137 M AMEC Base Case after-tax NPV.

The Ernst & Young after-tax cumulative net cash flow is higher than the AMEC cumulative after-tax net cash flow, primarily because the Ernst & Young Real Option NPV model uses a higher, January 01, 2011 copper spot price as a starting price for the simulation and is modeled to revert back to the AMEC model's long-term forecast price. For the initial years of production, expected copper prices in the Ernst & Young model are higher than the AMEC long-term forecast price assumptions.

AMEC notes that discounted cash flow NPV and real option NPV are different methods for calculating NPV and the reader needs an understanding of their differences to compare them between projects.

Table 1-12: Ernst and Young Real Option NPV vs AMEC Discounted Cash flow NPV

Item	Ernst and Young Real Options NPV(C$ million)	AMEC Discounted Cashflow NPV(C$ million)
Cumulative After Tax Net Cash Flow	5,755	5,118
After Tax Net Present Value	811	137

1.23 Preliminary Development Schedule

A preliminary Project development schedule has been generated. The schedule includes consideration of early work requirements, the EA assessment process, EPCM and construction activities. Critical items identified in the construction schedule are the tunnel, supply of power, and diesel usage. An efficient and well-executed construction strategy will be integral to the fiscal approval of the Project. A specific constraint on maintaining the schedule will be completion of activities in the Galore Creek Valley prior to tunnel completion.

The development schedule planned indicates that Project success will dependent in part on developing a practical and efficient logistics plan for the movement of manpower and materials to site during the construction and the subsequent operating phases of the Project.

A statement of work for the planned feasibility study on the Project has been issued, with requests for proposals from sub-consultants expected to be issued in October, 2011. GCMC will be preparing the Project description for the prior to the end of 2011 for submission of the EA document to the relevant authorities.

1.24 Work Plans

The GCMC 2011 pre-feasibility study identified opportunities that could be evaluated to potentially expand the mine life, improve the production profile, and reduce likely capital costs. GCMC has reviewed sections of these opportunities in order to outline an "Enhanced Plan" for consideration during future work. The Enhanced Plan incorporates considerations of Inferred Mineral Resources in the mining plan and capital and operating costs estimates that have a lower confidence than required for a prefeasibility level study.

Mine plan studies indicate that estimated Mineral Resources in the Bountiful area could become part of the mine plan using long-term price assumptions. The Mineral Resources lie approximately under the current exploration camp, and were excluded from consideration in the GCMC 2011 pre-feasibility study due to perceived high strip ratio requirements. If the currently estimated Inferred Mineral Resources can be successfully converted to Measured and Indicated Mineral Resources so that they may then be used as part of an appropriately engineered Mineral Reserve pit, then there is potential for the Bountiful area to be added to the mine plan.

Inclusion of this material in the Enhanced Plan would result in generation of additional waste tonnes, requiring some modification to the waste rock management plan that is currently envisaged for the Galore Creek Valley.

In order to provide sufficient space for waste rock storage, the access causeway and ore conveying systems would need to be relocated from the East Fork area. The Enhanced Plan includes the extension of the access tunnel by an additional 4 km to allow the entire East Fork area to be utilized as waste rock storage. The tunnel extension would increase the initial capital required for the Project. In addition, a substantial amount of additional PAG waste rock would need to be re-handled during the mine closure period in order to submerge the PAG waste rock during future mine rehabilitation and reclamation programs.

The Enhanced Plan lengthens the mine life sufficiently that mill expansion would be undertaken mid-mine life. The mill expansion would require some additional initial capital in order to design the coarse ore stockpile to eventually feed two SAG mills, and would require additional sustaining capital to install a second SAG mill. The addition of a SAG mill would increase the total site power requirements and annual operating costs, but would reduce the overall operating costs on a unit basis.

The Enhanced Plan also seeks to reduce capital costs, particularly in the areas of the proposed Stewart port facility and the concentrate transportation and filtration system, by reviewing potential design and assumption re-configurations. Options include relocating the filter plant to a location adjacent to the mill at the West More site, and using trucking transport for the dry concentrate. Effects of such changes include upgrading the current road access design, incorporation of a water treatment plant, and increased accommodation requirements at the West More camp.

The relocation of the filter plant to the mill site could result in a net reduction in capital cost, partially offset by increased operating costs, primarily as a result of the additional costs of trucking fuel and concentrate along the access road.

Additional work is required to better define this scenario such that it is an appropriate approach to be taken during future more detailed Project studies.

1.25 Conclusions

AMEC considers that the scientific and technical information available on the Project can support proceeding with additional data collection, trade-off and engineering work and preparation of more detailed studies. However, the decision to proceed with a Feasibility Study on the Project is at the discretion of GCMC and the partners.

1.26 Recommendations

AMEC recommends that GCMC consider the recommendations in this section as activities which may support Project advancement should the partners and GCMC determine that a Feasibility Study is warranted. The Project is located in a remote area, with significant logistics considerations. These factors indicate that completion of any Feasibility Study will require significant expenditure.

As part of the recommended work program, the following areas of work should be considered: additional drilling, topographic surveys, geotechnical studies, engineering and metallurgical studies, land management, including applications for mining leases where appropriate, additional baseline studies, and environmental and permitting activities. Additional areas for work are also likely to be identified as activities progress. AMEC’s recommendations do not include provision for pre-construcion and construction activities for site and access infrastructure such as the road and tunnel.

The program is envisaged as a two-phase program, with all elements of the first phase of the program to be conducted concurrently. The outcome of the work will be included in Phase 2, which will consist of completion of a Feasibility Study.

The Phase 1 activities include data collection, trade-off studies and investigations and studies and activities to support EA and public consultation processes. Some more specific recommendations for work focus have also been included for mineral resource estimation, tunnel design, and plant design purposes. The total cost of these activities is estimated to be between about $31 M and $39 M.

The Phase 2 activity comprises completion of a Feasibility Study, estimated at between about $11 M and $13 M, and including a contingency provision.

Total program costs for Phases 1 and 2 are likely to range between approximately $42 M and $52 M.

AMEC notes that GCMC has already commenced some initial work, which includes geotechnical drilling for both the tunnel and the open pits, sample collection and re-assaying, discussions relating to port usage, and review of information and recommendations arising from the 2011 GCMC pre-feasibility study report.

2.0 INTRODUCTION

Galore Creek Mining Corporation (GCMC), NovaGold Resources Inc. (NovaGold) and Teck Resources Limited (Teck) requested AMEC Americas Limited (AMEC) to prepare a Technical Report (the Report) on the results of a pre-feasibility study (GCMC 2011 pre-feasibility study) for the Galore Creek Copper–Gold-Silver Project (the Project) in British Columbia, Canada (Figure 2-1).

The Project is a 50:50 partnership between NovaGold Canada Inc. (a wholly-owned subsidiary of NovaGold) and Teck Metals Ltd (a wholly-owned subsidiary of Teck). The partners use an operating company, Galore Creek Mining Corporation (GCMC) to manage the Project. For the purposes of this Report, GCMC is used as a synonym for the partnership.

NovaGold is using the Report in support of a press release dated 28 July 2011, entitled “NovaGold Announces Prefeasibility Study Results for Galore Creek Project”.

2.1 Terms of Reference

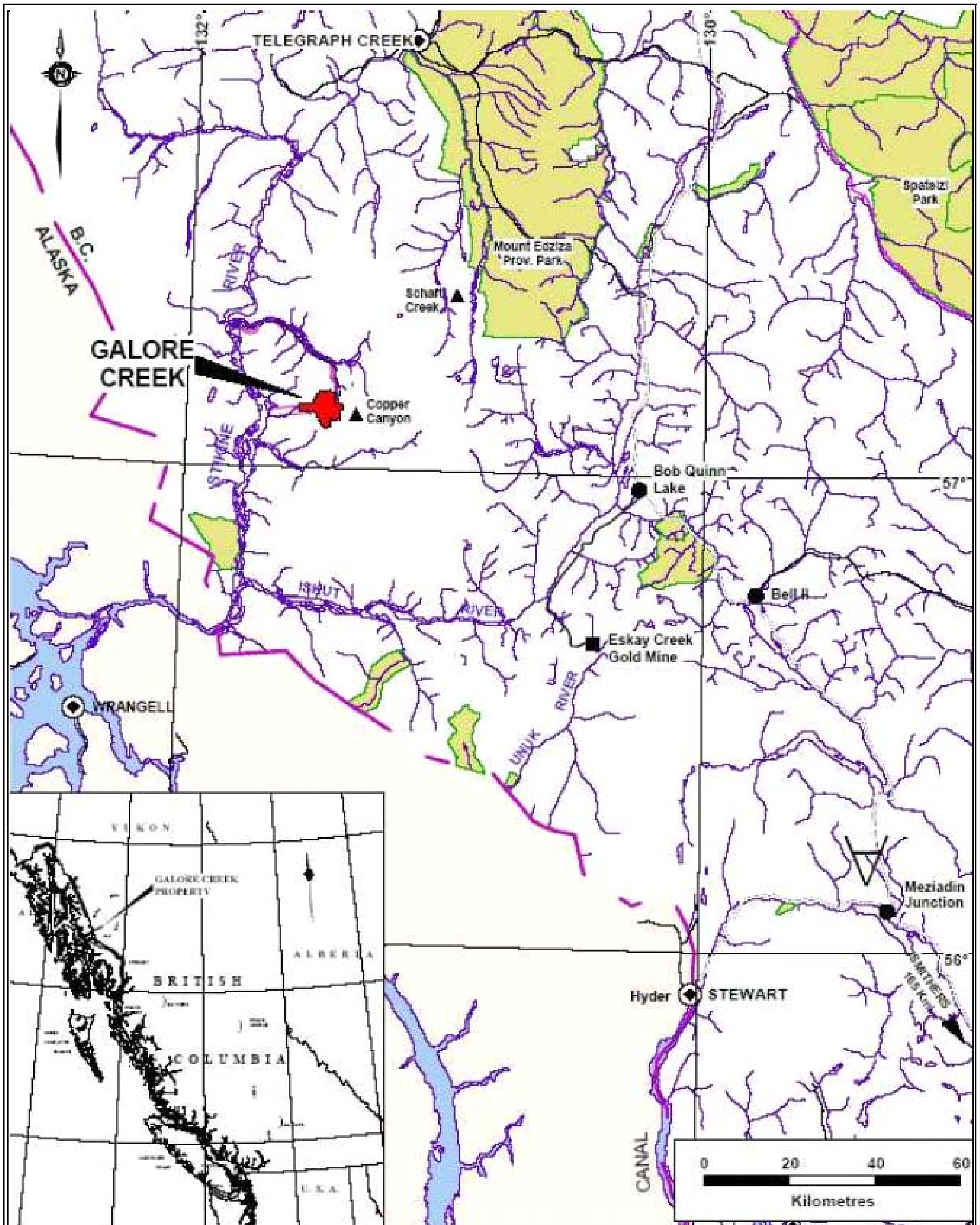
The GCMC 2011 pre-feasibility study was completed in June, 2011, and was a compendium of different studies by a number of companies, as indicated in Table 2-1.

The overall study was collated by GCMC personnel. The geology and mining sections of the study were completed by GCMC personnel, with contributions from both Teck and NovaGold personnel.

AMEC used the information completed by these contributors to support information in the current Report. AMEC’s QPs performed or commissioned independent due diligence reviews on the information supplied by GCMC and made adjustments to the results of the GCMC 2011 pre-feasibility study report based on the outcome of those reviews.

Lemley International Ltd (Lemley) was retained by GCMC at AMEC’s request to perform an endorsement-level review of all pertinent Project technical information for the tunnelling section of the GCMC 2011 pre-feasibility study report and provide their professional judgment as to the suitability of that work to meet the requirements of a pre-feasibility study. Lemley’s expertise includes tunnel design, tunnel construction and program/construction management of tunnels and other large infrastructure projects. Jack Lemley was the CEO of Transmanche-Link, the tunnel contractor consortium that successfully built the Channel Tunnel between England and France. Lemley suggested adjustments to the tunnelling section of the 2011 GCMC 2011 pre-feasibility study report based on the outcome of their review.

Figure 2-1: Project Location Map



Note: Figure courtesy GCMC, NovaGold and Teck. Note that the Copper Canyon deposit is currently not part of the Galore Creek Project, and is illustrated on this figure for reference purposes only.

Table 2-1: Study Contributors

Consulting Firm or Entity	Area of Responsibility in GCMC 2011 Pre-Feasibility Study Document
GCMC Third-party Consultant	Plant infrastructure, capital and operating cost estimates, execution plan and schedule
GCMC Third-party Consultant	Tunnel
Ausenco PSI	Concentrate pipeline
WorleyParsons Canada Services Ltd	Port
Knight Piésold Ltd	Transmission line
Tahltan-Allnorth Limited Partnership	Access road
AMEC E&E Services Inc.	Tailings, waste rock and water management, geochemistry
Brodie Consulting Ltd	Closure plan
Lorax Environmental Services Ltd.	Water quality modeling
G &T Metallurgical Services Ltd.	Metallurgical services and metallurgical testwork
Rescan Tahltan Environmental Consultants	Ongoing baseline studies
Consulting Firm or Entity	**Area of Responsibility in Review of GCMC 2011 Pre-feasibility Study Document**
Lemley International	Tunnel
AMEC Americas Ltd.	All other areas

Note: the consulting firm or entity noted as a “GCMC Third-party Consultant” is unable to be identified under the terms of their contracts with GCMC

The Report uses Canadian English. Unless specified in the text, monetary amounts are in Canadian dollars (C$) and units are metric.

2.2 Qualified Persons

The following people served as the Qualified Persons (QPs) as defined in National Instrument 43-101, *Standards of Disclosure for Mineral Projects*, and in compliance with Form 43-101F1:

- Robert Gill, P.Eng, Principal Consultant and Study Manager, AMEC Vancouver
- Jay Melnyk, P.Eng., AMEC Associate Engineer, Vancouver
- Greg Wortman, P.Eng., Technical Director, Process, North America, AMEC Oakville
- Greg Kulla, P.Geo., Principal Geologist, AMEC Vancouver
- Dana Rogers, P.Eng., Principal Tunnelling Engineer, Lemley International.

2.3 Site Visits

QPs conducted site visits to the Project as shown in Table 2-2.

Table 2-2: QPs, Areas of Report Responsibility, and Site Visits

Qualified Person	Site Visits	Report Sections of Responsibility (or Shared Responsibility)
Robert Gill	No site visit	Sections 1, 2, 3, 4, 5, 6, 18 (except Sections 18.6 and 18.7), 19, 20, 21, 22, 23, 24, 25, 26 and 27.
Jay Melnyk	27 to 28 September, 2007	Sections 15 and 16, and those portions of the Summary, Interpretations and Conclusions and Recommendations that pertain to those Sections.
Greg Kulla	21 to 24 September 2010	Sections 7, 8, 9, 10, 11, 12, 14, and those portions of the Summary, Interpretations and Conclusions and Recommendations that pertain to those Sections.
Greg Wortman	No site visit	Sections 13 and 17, and those portions of the Summary, Interpretations and Conclusions and Recommendations that pertain to those Sections.
Dana Rogers	15 to 17 June 2011	Sections 18.6 and 18.7, and those portions of the Summary, Interpretations and Conclusions and Recommendations that pertain to those Sections.

2.4 Scope of Personal Inspections

Mr Kulla, during a September 2010 site visit, undertook a helicopter inspection of the proposed road and power transmission routes, the Espaw exploration camp, and the Galore Greek valley. A total of 47 drill sites were inspected in the field, during a traverse through the West Fork, Southwest, and Central Replacement zones. Collar locations were checked using a hand-held GPS, and within the limits of such instrumentation, collar locations matched those of the 2007 drill database. Several zones of outcrop were inspected, including a zone of “broken rock”. At AMEC’s request, Erin Workman, a resource geologist with GCMC, pre-selected several drill holes representing the various mineralized areas of the deposit for review; some drill holes selected by AMEC were not able to be located.

There was no active drilling, logging or sampling in progress during the site visit. Logging and sampling facilities were shut down and locked. Mr Kulla also held meetings with GCMC Project staff including Clair Chamberlain (Senior Geologist), Barry Duff (Logistics Manager) and Peter Wells (Design Manager).

Mr Melnyk visited site during September 2007. During the site visit, he undertook a high-level review of the Project geology, inspected drill core, viewed the Project topography and the locations of existing infrastructure, including road cuts and borrow pits, and the locations and outlines of the surface drainages.

Mr Rogers during a June 2011 site visit performed a visual, on-the-ground inspection of surface conditions at both portal sites. He also conducted a visual inspection of core recovered from the three tunnel borings, and undertook a helicopter traverse along tunnel alignment, including landing and on-the-ground visual inspection of surface conditions near the location of borehole GCT 10-2, which is the boring located near the midpoint of the alignment. A general reconnaissance of the Project area via helicopter was also undertaken. Mr Rogers also held meetings with GCMC Project management including Henri Letient (Project Director), Peter Wells (Project Manager) and Paul Cocklin (Construction Manager/ Mine Manager Designate).

In addition to these visits, other AMEC personnel have visited site, and have provided input to the AMEC QPs in the areas of core splitting and sample preparation, tailings, waste rock, and water management, and geochemistry.

2.5 Effective Dates

The Report has a number of effective dates, as follows:

- Effective date of the Mineral Resources: 11 July 2011
- Effective date of the Mineral Reserves: 11 July 2011
- Effective date of the tenure and surface rights data: 27 July 2011
- Effective date of the financial analysis: 27 July 2011.

The overall effective date of the Report, based on the date of the financial analysis and provision of information on mineral tenure and surface rights, is 27 July 2011.

GCMC have commenced an infill drill program in the Bountiful area that is designed to support potential upgrades in the confidence categories of the Mineral Resources in this area. This drilling was ongoing at the effective date of the Report.

There has been no material change to the scientific and technical information on the Project between the effective date of the Report, and the signature date.

2.6 Previous Technical Reports

NovaGold has previously filed the following technical reports on the Project:

Francis, K., 2008: Galore Creek Property NI 43-101 Technical Report British Columbia – Canada: unpublished technical report to NovaGold Canada Inc., effective date 25 January 2008.

Rustad, B., Gray, J., Lechner, M., Teh, H., Bruce, I., Parolin, B., Guy, A., Boychuck, K., Brox, B., and Holborn, D., 2006: Galore Creek Project Feasibility Study Northwestern British Columbia: unpublished technical report to NovaGold Canada Inc. by Hatch Ltd., effective date 31 October 2006.

Giroux, G.H. and Morris, R.J., 2005: Geology and Resource Potential of the Galore Creek Property: unpublished technical report to NovaGold Canada Inc by Hatch Ltd., GRTechnical Services Ltd. and Giroux Consultants Ltd., effective date 18 May 2005.

Hosford, P., 2004: NovaGold Resources Inc. & NovaGold Canada Inc. Preliminary Economic Assessment for The Galore Creek Gold - Silver – Copper Project: unpublished technical report prepared by Hatch Limited for NovaGold, effective date 5 August 2004.

Lacroix, P.A., 2004: Update on Resources Galore Creek Project, British Columbia: unpublished technical report to NovaGold Resources Inc. and SpectrumGold Inc. by Associated Mining Consultants Ltd., effective date June 3, 2004.

Prior to NovaGold's interest in the Project, SpectrumGold filed the following technical report:

Simpson, R.G., 2003: Independent Technical Report For The Galore Creek Property Liard Mining Division British Columbia: unpublished technical report to SpectrumGold Inc., effective date 11 August 2003.

GCMC is not a listed entity.

Teck has not filed a technical report on the Project.

2.7 Information Sources

The primary data source for this Report is the GCMC 2011 pre-feasibility study, entitled:

Galore Creek Mining Company, 2011: Galore Creek Project Prefeasibility Study Report, Including Appendices A to J: unpublished internal report prepared by GCMC, dated June 2011, 3,643 p.

Reports and documents listed in the Section 3, Reliance on Other Experts and Section 27, References sections of this Report were also used to support preparation of the Report. Additional information was sought from NovaGold, Teck, and GCMC personnel where required.

3.0 RELIANCE ON OTHER EXPERTS

The QPs have relied upon the following other expert reports, which provided information regarding mineral rights, surface rights, property agreements, and marketing sections of this Report as noted below.

3.1 Mineral Tenure and Mining Rights Permits

The QPs have not reviewed the mineral tenure, nor independently verified the legal status, ownership of the Project area, underlying property agreements or permits. AMEC has fully relied upon, and disclaims responsibility for, information derived from GCMC experts and experts retained by GCMC for this information through the following documents:

Letient, H., 2011: Galore Creek Project: letter from Henri Letient, Project Director Galore Creek Project, GCMC, to Robert Gill, AMEC, regarding mineral tenure, surface and water rights, agreements and proposed marketing strategy options, 9 September 2011.

This information is used in Section 4.3 of the Report and was also used to support considerations of reasonable prospectsof economic extraction and declaration of Mineral Resources in Section 14.3 and 14.4, and for consideration of appropriate modifying factors for declaration of Mineral Reserves in Section 15.3.

3.2 Surface Rights

The QPs have fully relied upon and disclaim responsibility for information supplied by GCMC staff and experts retained by GCMC for information relating to the status of the current surface rights as follows:

Letient, H., 2011: Galore Creek Project: letter from Henri Letient, Project Director Galore Creek Project, GCMC, to Robert Gill, AMEC, regarding mineral tenure, surface and water rights, agreements and proposed marketing strategy options, 9 September 2011.

This information is used in Section 4.5 of the report and for consideration of appropriate modifying factors for declaration of Mineral Reserves in Section 15.3.

3.3 Agreements

The QPs have fully relied upon and disclaim responsibility for information supplied by GCMC staff and experts retained by GCMC or NovaGold for information relating to the status of the current Property Agreements as follows:

Letient, H., 2011: Galore Creek Project: letter from Henri Letient, Project Director Galore Creek Project, GCMC, to Robert Gill, AMEC, regarding mineral tenure, surface and water rights, agreements and proposed marketing strategy options, 9 September 2011.

This information is used in Section 4.4 of the Report and was also used to support considerations of reasonable prospectsof economic extraction and declaration of Mineral Resources in Section 14.3 and 14.4, and for consideration of appropriate modifying factors for declaration of Mineral Reserves in Section 15.3..

3.4 Royalties

The QPs have fully relied upon and disclaim responsibility for information supplied by GCMC staff and experts retained by GCMC for information relating to the status of the current royalties payable as follows:

Letient, H., 2011: Galore Creek Project: letter from Henri Letient, Project Director Galore Creek Project, GCMC, to Robert Gill, AMEC, regarding mineral tenure, surface and water rights, agreements and proposed marketing strategy options, 9 September 2011.

This information is used in Section 4.7 of the report and was also used to support considerations of reasonable prospectsof economic extraction and declaration of Mineral Resources in Section 14.3 and 14.4, and for consideration of appropriate modifying factors for declaration of Mineral Reserves in Section 15.3.

3.5 Marketing

The QPs have fully relied upon and disclaim responsibility for information supplied by GCMC staff and experts retained by GCMC for information relating to the status of the potential Projecgt marketing regime as follows:

Okamura, H., 2010: Galore Creek Prefeasibility Marketing Study: unpublished marketing study prepared by H. Okamura, Director, Concentrate Sales at Teck, for GCMC, 16 March 2010, 15 p.

This information is used in Section 19, and was used to to support considerations of reasonable prospectsof economic extraction and declaration of Mineral Resources in Section 14.3 and 14.4, declaration of Mineral Reserves in Section 15.3, and the cashflow analysis in Section 22.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Galore Creek Project is located approximately 70 km west of the Bob Quinn airstrip on Highway 37 and 150 km northeast of the port of Stewart, and 370 km northwest of the town of Smithers, British Columbia, Canada, at approximate latitude 57° 07'30"N and longitude 131°27'W (UTM NAD83, Zone 9 (m) coordinates 6334850N, 351200E).

Smithers is the nearest major supply centre and has an airport with regularly scheduled flights to and from Vancouver, BC. In Alaska, the closest community is Wrangell.

4.2 Mineral Tenure History

The Galore Creek Property was discovered in 1955, and originally staked by Stikine Copper Ltd., a joint venture between Kennecott Canada Inc. (Kennecott), Hudson Bay Mining and Smelting Co. Ltd. (Hudson Bay) and Cominco Ltd. Until 1968, Kennecott was operator. From 1972 to 1976, Hudson Bay was operator. Mingold Resources Inc. (an affiliated company of Hudson Bay) explored during 1989. In 1991, Kennecott resumed as operator, but work was only completed on the Project by Kennecott during that year.

SpectrumGold Inc. (a separately-listed subsidiary of NovaGold Resources Inc., which in turn is a wholly-owned subsidiary of NovaGold) signed an option on July 31, 2003 with Stikine Copper Limited, QIT-Fer et Titane Inc. (a Kennecott subsidiary), and Hudson Bay to acquire a 100% interest in the Galore Creek Property. In mid-2004, NovaGold acquired all outstanding shares in SpectrumGold Inc. and transferred all Project rights to NovaGold Canada Inc.

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 BC Mineral Tenure Act. 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.

On December 18, 2007 GCMC applied drilling expenditures incurred on the Galore Creek property as assessment work to advance all claims contiguous with the Galore Creek property to an expiry year of 2017 or 2018; different expiry years were applicable to different claims. This was the maximum allowed under the Mineral Tenure Act. Assessment work was not applied to claims that were legally surveyed and being considered for mining lease application.

On March 28, 2007, NovaGold exercised the Stikine option, and acquired 100% of the Project as at June 1, 2007. Teck Cominco Ltd became a 50:50 partner in the entire Galore Creek Project with NovaGold on August 1, 2007; under the agreement Teck has certain funding and other obligations to retain its interest. The remaining 50% is held by NovaGold. The joint venture partners created the jointly-controlled operating company, the Galore Creek Mining Corporation (GCMC).

In October 2007, all Galore Creek mineral claims held by NovaGold Canada Inc. were transferred to GCMC.

In November 2007, NovaGold and Barrick Gold Corporation (Barrick) as the successor company to Pioneer Metals Ltd (Pioneer) reached an agreement regarding a group of claims known as the Grace Claims, which are situated immediately to the north of hte current Mineral Resources, and the five claims were sold to GCMC on 3 December 2007. These claims are now part of the Galore Creek Project.

During March 2008, GCMC acquired additional mineral claims in the Scud River area, Stikine River area and north of the West More area to support Project infrastructure development, in particular, the access road.

4.3 Mineral Tenure

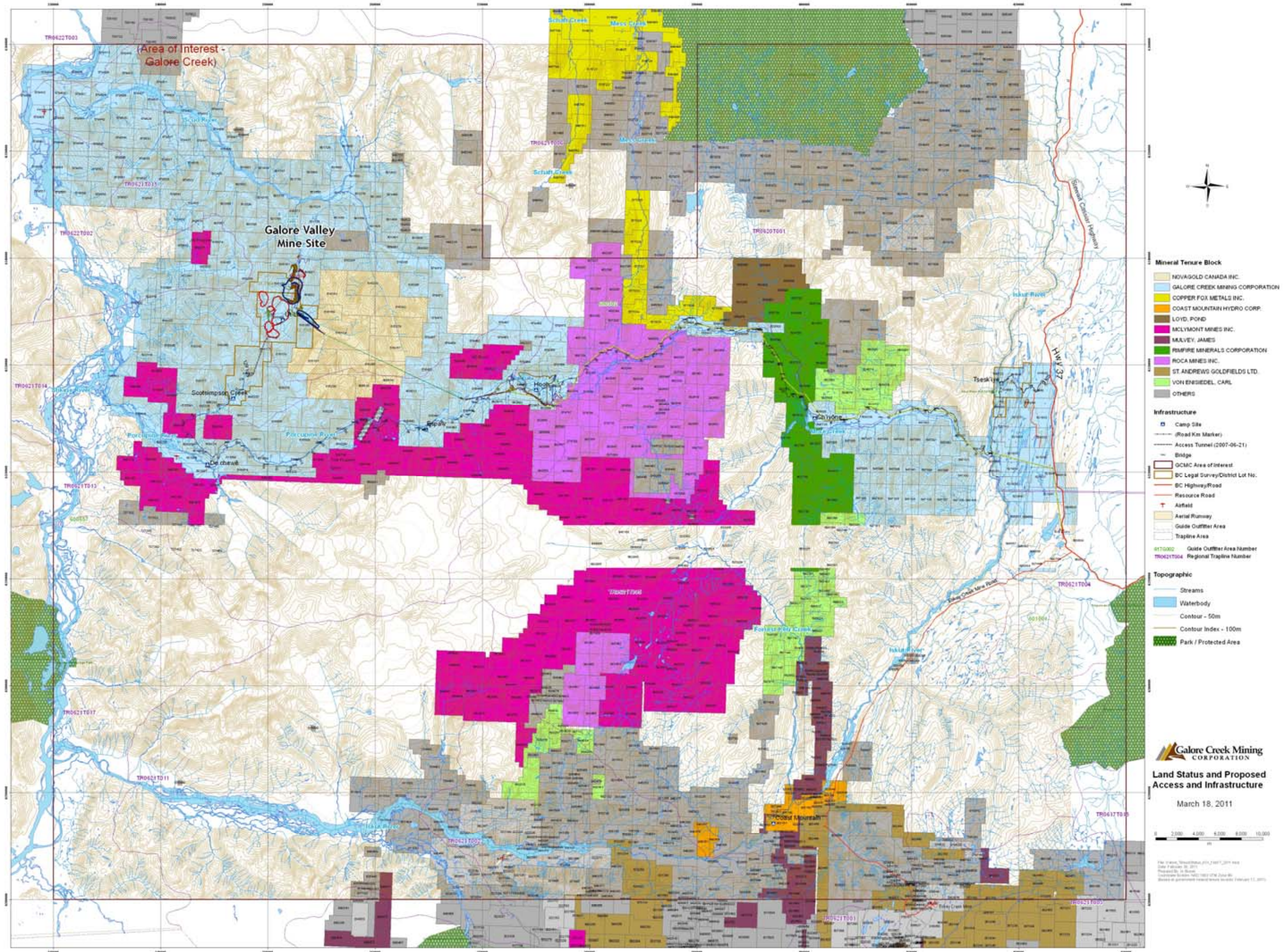
4.3.1 Galore Creek Project

The Project consists of 264 mineral claims, totalling 118,911.88 ha, held in the name of GCMC. A list of the claims with their expiry dates is included as Appendix A. A tenure location plan is included as Figure 4-1. Figure 4-2 shows the location of the mineralized centres in relation to the claims boundaries that host mineralization. Mineralization is almost all contained within Claim 546459.

Contiguous claims within the Galore Creek property have had assessment work filed on them. Assessment work was not filed for claims that were not contiguous. The dates marked in Appendix A against the claims as “good to” represent the dates for which assessments have been filed and the assessment reports approved by the Mineral Titles division of the BC Government.

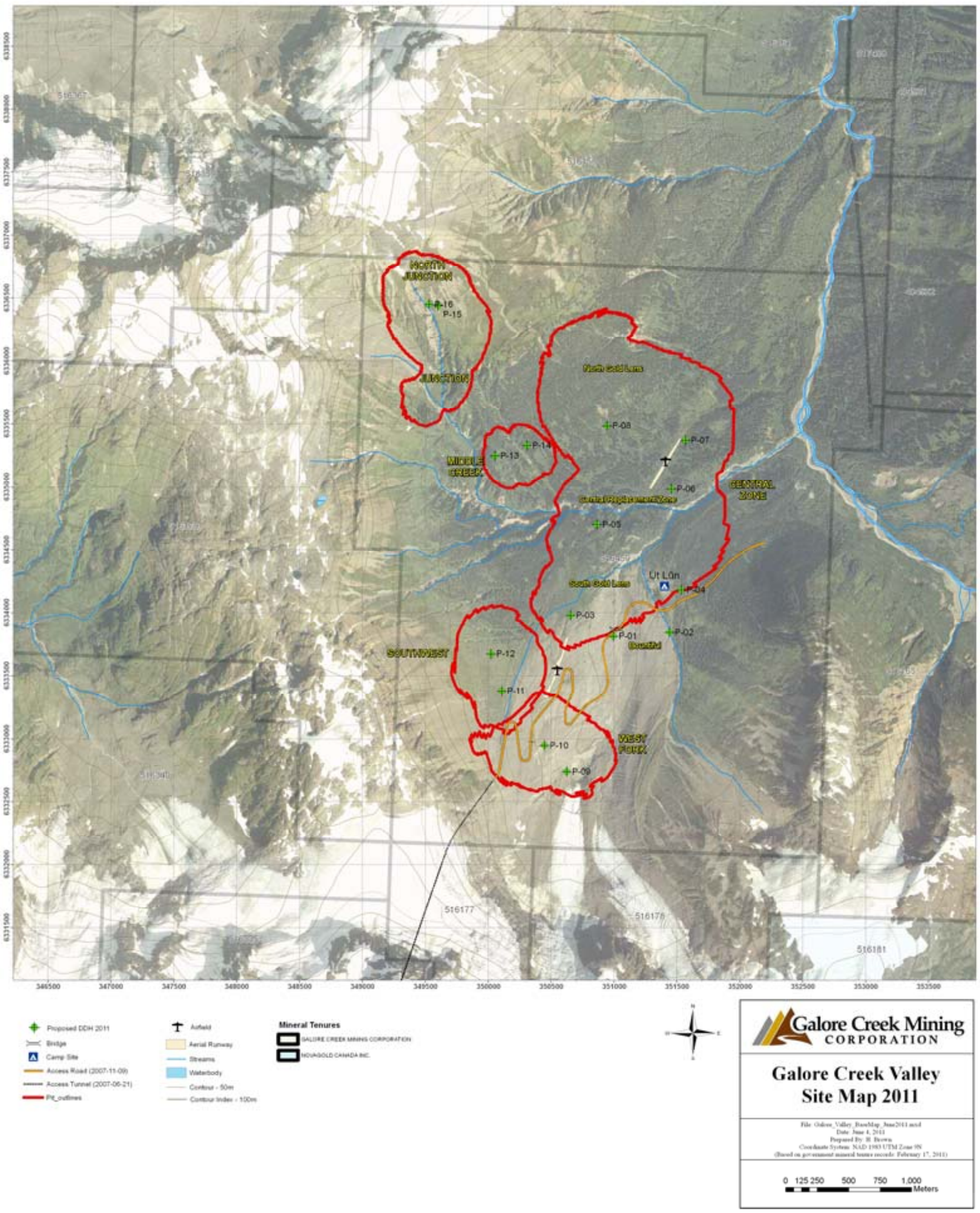
Claims are a combination of map-staked and ground located.

Figure 4-1: Mineral Tenure Plan



Note: Figure courtesy GCMC, NovaGold and Teck. Yellow line on plan is the trace of the proposed access road; the green line is the proposed tunnel alignment. Red polygons on plan are the proposed open pit outlines.

Figure 4-2: Plan Showing Main Mineralization Centres in Relation to Claims Boundaries



Note: Figure courtesy GCMC, NovaGold and Teck

4.3.1 Other Regional Projects

The Inferred Mineral Resource contained within the Copper Canyon property is owned 70% by NovaGold and 30% by Teck.. The property is surrounded on three sides by land owned by GCMC but the property concessions are not currently consolidated into the partnership or the Galore Creek Project.

NovaGold has offered the Copper Canyon property to Teck to vend 100% of the property into the partnership, but until a decision on the offer is made by Teck, the Copper Canyon deposit is not considered part of the Galore Creek Project.

4.4 Property Agreements

4.4.1 Pre-NovaGold Agreements

The claims that cover the core of the Galore Creek property were owned by Stikine Copper Ltd (Stikine). Stikine was incorporated in 1963, and consolidated the regional holdings of Kennecott Canada Inc. (76% and operator), Hudson Bay Mining and Smelting Company Limited (19%), and Consolidated Mining and Smelting Company of Canada Limited (5%; Barr, 2004). Stikine was controlled by QIT-Fer et Titane Inc. (55%; a wholly-owned subsidiary of Rio Tinto Ltd.) and Hudson Bay (45%).

The Galore Creek property consisted of 292 two-post claims, of which 39 were fractions, all of which were held in the name of Stikine.

4.4.2 SpectrumGold and Stikine

In August 2003, SpectrumGold Inc. entered into an option agreement to acquire Stikine. The agreement included completion of a pre-feasibility study on the project and making payments to the parties totalling US$20.3 M within a period of eight years. Payments of US$0.3 M in aggregate were required over the first three years of the option, with the remaining US$20 M to be paid over the following five years. There was to be no retained interests, royalties or back-in rights on the project.

On 28 March, 2007, NovaGold acquired the Galore Creek property by exercising its option to purchase 100% of Stikine; full acquisition was completed on July 1, 2007. In June 2007, six mineral claims held by Stikine Copper Limited were transferred to NovaGold Canada Inc.

4.4.3 SpectrumGold and NovaGold

In July 2004, NovaGold Resources Inc. acquired the balance of SpectrumGold Inc. that it did not own, and SpectrumGold Inc. was renamed NovaGold Canada Inc. Agreements with Stikine, Pioneer Metals Corporation and Eagle Plains Resources, (now Copper Canyon Resources Ltd.) were transferred to NovaGold.

4.4.4 NovaGold and Teck

On May 23, 2007, NovaGold and Teck Cominco Ltd. (now Teck Resources) announced a 50:50 partnership to develop the Galore Creek property. On August 1, 2007 the Galore Creek Partnership was established to develop the Galore Creek mine and created the jointly controlled operating company called the Galore Creek Mining Corporation. In October 2007, all Galore Creek claims held by NovaGold Canada Inc. were transferred to the Galore Creek Mining Corporation.

To earn its 50% interest in the Galore Creek partnership, Teck was to fund approximately $520 M in construction costs, with each company responsible for its pro rata share of funding thereafter. In addition, NovaGold was to receive up to US$50 M of preferential distributions when Galore Creek was fully operational, if the Project exceeded certain agreed upon minimum revenues in the first year of commercial production.

On 26 November, 2007 NovaGold and Teck suspended construction of the Galore Creek Project. In light of this development, NovaGold and Teck agreed to amend the terms of Teck's earn-in obligations in connection with the Project. Under the amended arrangements, Teck's total earn-in was $430 M. Teck agreed to invest an additional $72 M in the partnership to be used over the following five years principally to reassess the Project and evaluate alternative development strategies. In addition, NovaGold and Teck agreed to share the next $100 M of Project costs 33% and 67%, respectively, and proportionately thereafter.

On 11 February 2009, NovaGold and Teck further amended certain provisions of the Partnership Agreement. Under the agreement, Teck agreed to fund 100% of all costs incurred by the Partnership from 1 November 2008 until the aggregate additional amount contributed by Teck, including certain amounts previously spent to fund optimization studies, equalled $60 M. If any portion of the $60 M was not contributed by 31 December 2012, Teck agreed to contribute in cash any shortfall on that date to the partnership.

In June 2011, the $60.0 M commitment was met, and from June 2011 forwards, the partners are obligated to contribute equally to any share of costs associated with Project development.

4.4.5 GCMC and the Tahltan Nation

On 13 February 2006, NovaGold announced that it had entered into a comprehensive agreement with the Tahltan Nation for their participation in, and support of, the development of the Galore Creek Project. When GCMC was formed, the Participation Agreement with the Tahltan Nation was transferred from NovaGold to GCMC.

The agreement covers all claims, infrastructure, equipment, plants and facilities held, controlled or acquired by GCMC to explore, develop, construct, operate and reclaim gold-silver-copper within the Galore Creek Valley, including any ancillary or related activities or operations that support the mine processing facilities such as the access road. The agreement does not include material changes to the Project or any proposed additional development by GCMC which required an additional or separate Environmental Approval to that envisaged in 2006.

Financial contributions will be made by GCMC to the Tahltan Heritage Trust Fund, which is set up so that funds in the trust will be used to mitigate any adverse social and cultural impacts of mine development. During any mining operations, Trust Fund payments are guaranteed to be no less than $1 M annually. Upon reaching certain agreed financial targets, and subject to positive mine operating cash flow, the trust will receive the greater of $1 M or a 0.5% to 1.0% net smelter royalty (NSR) each year. The agreement will remain in effect throughout the life of the Galore Creek Project and will be binding on any future operator of the mine.

4.5 Surface Rights

The mineral claims are on Crown land. Access, power, and pipeline facilities cross mineral claims held by third parties. Mineral claims do not confirm exclusive surface rights to a mineral claims holder.

GCMC presently has a Special Use Permit (SUP) which entitles the partnership to the right-of-way for the access road corridor, in accordance with the original road layout design permitted under the current Environmental Assessment approval. A small portion of the new road layout and the new tunnel alignment fall outside of that SUP (refer to Sections 5.1, 18.2 and 18.3).

Because the new Project design and configuration is different from that previously permitted, new Environmental Assessment approval is likely to be required. This is likely to include the area of the Galore Creek Valley access tunnel and road. When Environmental Assessment approval is forthcoming, an amendment to the SUP will be applied for.

A portion of the Galore Creek Valley access tunnel passes through mine claims held by NovaGold Canada Inc., a wholly-owned subsidiary of NovaGold.

Except for the access corridor which is covered by the SUP, all other infrastructure, including the processing plant and tailings area in the West More Valley and the proposed filter plant area near Km 8 are located within GCMC's mineral claims.

GCMC intends to file for mining leases to secure the surface rights for these areas.

4.5.1 Cassiar Iskut-Stikine Land and Resource Management Plan

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 areas mineral resources by providing information to be considered during the permitting and impact assessment processes. The LRMP is primarily in territory claimed by the Tahltan First Nation. The Tahltan Joint Councils, representing the Tahltan Band from Telegraph Creek and the Iskut Band, were full table members throughout the process and endorse the LRMP. Neighbouring First Nations include the Nisga'a, Kaska, and Tlingit Nations.

The LRMP identifies 15 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.

Logging is only allowed for the purposes of mineral exploration and/or mine development and for localized use.

4.5.2 Guide and Trapping Surface Rights

Two guide outfitter territories and seven registered trap lines overlie the Galore Creek deposits and planned access road.

4.6 Water Rights

Currently, GCMC has short-term water approvals for exploration, access road, and camp use, granted through Section 8 of the BC Water Act. GCMC intends to apply for water licences under the Water Act as required to support Project development and operational activities.

4.7 Royalties

There are no third-party or Government royalty obligations associated with the Galore Creek Project other than a net smelter return royalty payable to the Tahltan nation, under the Tahltan Agreement, the details of which are discussed in Section 4.4.5 and Section 20 of this Report.

4.8 Encumbrances

There are currently no encumbrances associated with the Project. On December 21, 2007 two Builders Liens/Encumbrances (Event Numbers 4186086 and 4186084) were filed on selected GCMC claims. Both of these liens were discharged on January 31, 2008 under Event Numbers 4193255 and 4193252.

4.9 Permits

Permits required to support Project development are discussed in Section 20.

4.10 Environment

Environmental studies, closure plans and costs, and environmental liabilities and issues are discussed in Section 20.

4.11 Social and Community Impact

The potential social and community impact assessments of the Project are discussed in Section 20.

4.12 Comment on Section 4

In the opinion of the AMEC QPs, the following conclusions are appropriate:

- Information from GCMC experts supports that the mining tenure held is valid and is sufficient to support declaration of Mineral Resources and Mineral Reserves

- The mineral concessions have been surveyed in accordance with relevant BC regulations in the case of the 17 remaining legacy claims, and are otherwise map-staked
- Annual claim-holding fees have been paid to the relevant regulatory authority where exploration work on the claims was insufficient to have met the required assessments. All of the claims that host mineralization have appropriate work assessments filed and are current until 2018. Other claims within the land package are current to 2017
- The Copper Canyon property has been offered to the partnership; but until a decision on the offer is made by Teck, the Copper Canyon deposit is not considered part of the Galore Creek Project
- Mineral claims are on Crown land. GCMC presently has a SUP which entitles the partnership to the right-of-way for the access road corridor. Modifications to the road design may require application for modification to the SUP
- Because the new Project design and configuration is significantly different from that previously permitted, new Environmental Assessment approval is likely to be required from both Provincial and Federal authorities
- Water rights are granted under short-term approvals. Additional water licences will be applied for to support Project development
- There are no Government royalty obligations
- Financial contributions will be made by the Galore Creek Partnership to the Tahltan Heritage Trust Fund. During mine operations, Trust Fund payments are guaranteed to be no less than $1 M annually. Upon reaching certain agreed financial targets, and subject to positive mine operating cash flow, the trust will receive the greater of $1 M or a 0.5% to 1.0% net smelter royalty each year. The agreement will remain in effect throughout the life of the Project and will be binding on any future operator of the mine
- Exploration activities to date have been conducted within the regulatory framework required by the BC Government
- Additional permits will be required for Project development.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Current Accessibility

5.1.1 Air

The town of Smithers, located 370 km to the southeast, is the nearest major supply centre to Galore Creek. Most personnel, supplies, and equipment are staged from the Bob Quinn airstrip, on the Stewart-Cassiar Highway (Highway 37) and transported via helicopter to the Galore Creek camp.

Bob Quinn is serviced by contract flights from Smithers and Terrace, each of which has daily flights from Vancouver. Flight time from Vancouver to Smithers/Terrace is about 90 minutes, then an additional 45 minutes to Bob Quinn. The helicopter flight from Bob Quinn to the Galore Creek camp is about 30 minutes.

The main helicopter landing pad for the Galore Creek Valley is constructed about 500 m southwest of the Galore Creek exploration camp.

5.1.2 Water

The Stikine area was accessed by shallow draft barges and riverboats, in particular during the Stikine–Cassiar and Klondike gold rushes of the late 19th century, but continuing to the late 1960s. These boats were used to transport goods from Wrangell, Alaska to Telegraph Creek, British Columbia, a distance of 302 km. The Stikine River remains navigable for this type of watercraft from about mid-May to October. The nearest point on the Stikine River to the Project is the mouth of the Anuk River, about 16 km west of the camp.

5.1.3 Road

Proposed Access Road

A report for the Galore Creek mine access road was prepared by Allnorth (2010). Allnorth utilized a report prepared in 2005 by McElhanney Consulting Services Ltd as a basis for the study, but updated the current as-built road conditions in the road status report and utilized TNR Bridge Construction Limited Partnership (TNR) to update the report and cost estimates for the bridges and culverts.

The route selected starts at the junction of Highway 37 and proceeds west across the Iskut River, up More Creek, over the More Canyon bridge, around the new tailings location at Round Lake and then down Sphaler Creek to the South Portal (Figure 5-1).

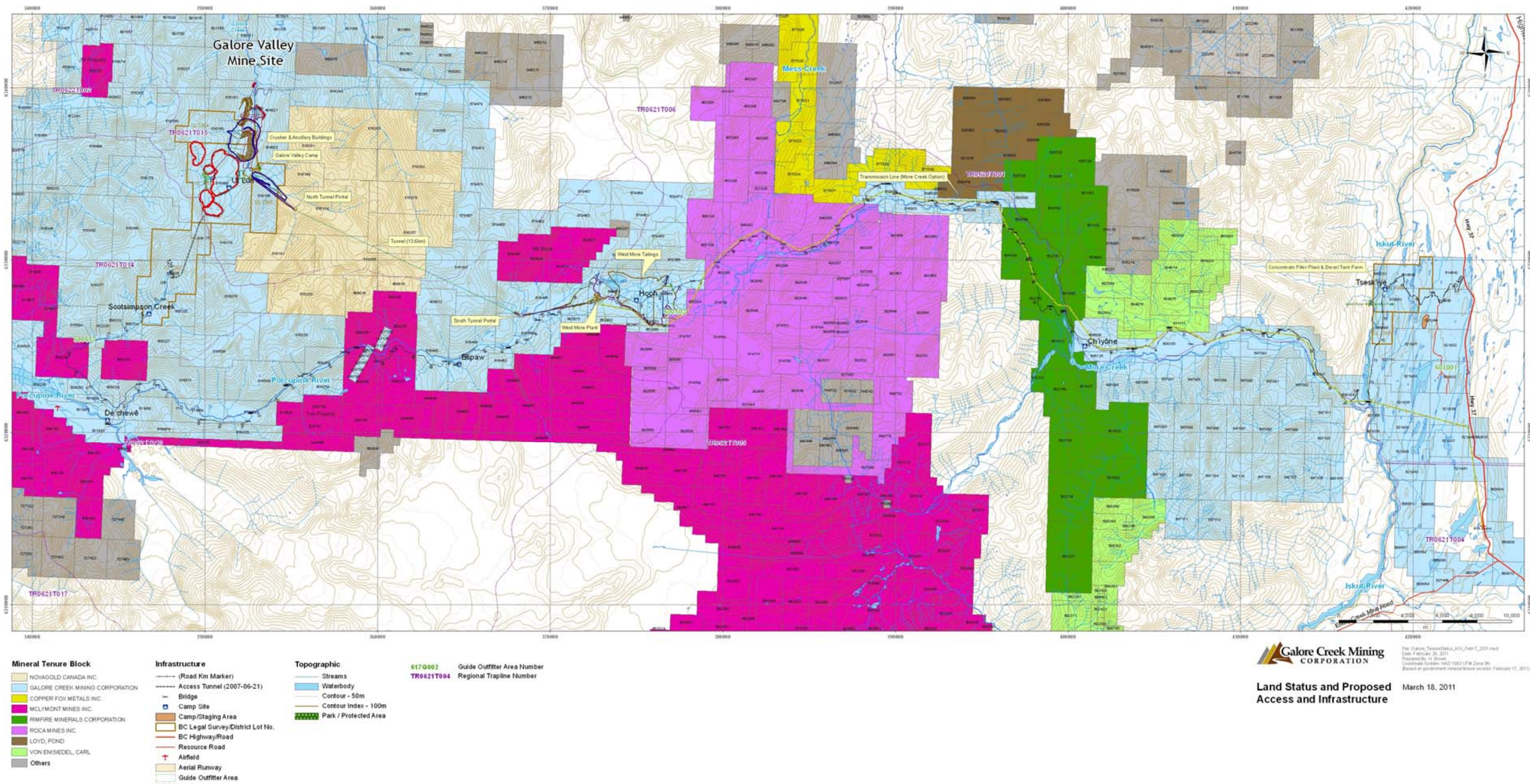
The initial 8 km of the road will be double lane, narrowing to a single-lane (6 m wide) resource access road. The road is planned to support construction of the diesel supply line, concentrate pipeline, and the power transmission line and provide supplies, equipment, and crew transport during construction and operation of the mine. The road will be constructed with less than 15% grades and an average design speed of 40 km/h. The road is intended to be a low-impact road within the utilities corridor.

Bridges and culverts are designed for a 200-year and 100-year instantaneous flood, respectively, with a minimum 1.5 m clearance to the underside of the bridge girders unless additional clearance is required for navigable waters or geotechnical requirements. The bridges and culverts are all rated for a maximum load of 100 t.

A summary of the current access road status is:

- Over the section of road from the Highway 37 (Km 0 to Km 40), preliminary construction has been completed and the road is currently in service. Even though this section of the road is in service, it will require upgrading to the final design and the first 8 km will be upgraded to dual lane
- The section of road from Km 40 to Km 48 has been cleared, but construction has not commenced
- The More Canyon bridge is located at Km 48. The basic design of the bridge has been completed; however, detailed engineering, design procurement and construction is required prior to the access road linking the north and south sections of the road, thus enabling road access from Highway 37 to the proposed plant, tunnel and mine
- The section of the road from Km 48 to Km 71.5 has been cleared, but construction has not commenced
- The section of road from Km 71.5 to Km 78 is currently completed and in service as per the original design. Due to the relocation of the tailings storage facility, however, this section will eventually be flooded and a tailings perimeter road will need to be constructed. The new tailings perimeter road will need to be completed prior to the commissioning of the cofferdam required for construction of the main tailings dam. The tailings perimeter road will also incorporate the East and West Saddle dams. This new road has not been permitted

Figure 5-1: Proposed Access Road Layout Plan



Note: Figure courtesy GCMC, NovaGold, and Teck

- The section of road from Km 78 to Km 80 has been partially completed
- The final section of the access road from where it meets the stockpile feed conveyor to the South Portal has not commenced and is not permitted.

Kennecott Road

During early exploration efforts in the 1960s, Kennecott constructed 48 km of road from the mouth of the Scud River to the exploration camp in the Galore Creek Valley. The road is in very poor condition and would require repair along the Scud River and portions of the Galore Creek Valley before it could be used. No plans exist to conduct this work.

5.2 Climate

The Project area is characterized by cold winters and short, cool, summers.

Within the Galore Creek Valley, mean monthly temperatures range from -8.2ºC during the winter to 12.4ºC during the summer, with January and July typically being the coolest and warmest months, respectively. In the Upper West More Valley area, monthly average temperatures range from -8.9ºC in the winter to 7.9ºC in the summer.

Precipitation begins to fall as snow in early October and continues until the end of May. A basinal average precipitation for the whole Galore Creek Valley watershed was estimated to be in the order of 3,000 mm (Rescan, 2006a). June and July tend to receive the least amount of precipitation on an annual basis (typically 40 to 60 mm of rain per month).

Surface run-off across the Project area is relatively high compared to other regions of BC and Canada. The hydrological regime of the region is very dynamic and temporally and spatially variable. The estimated annual average run-off for the Galore Creek Valley watershed (drainage area is about 145 km^2) is 2,340 mm (Rescan 2006a). Based on site-specific data, annual run-off in Galore Creek has ranged from 1,760 to 3,830 mm. The More Creek Valley, which is located further east than Galore Creek Valley, experiences lower annual run-off, which has been observed to range from 1,410 to 2,773 mm.

Glacial melt processes are dominant in Galore Creek, while snowmelt processes dominate in More Creek. This is reflected in the run-off data, with Galore Creek experiencing more consistent run-off from July through to September. More Creek, on the other hand, produces a relatively higher proportion of run-off earlier in the season during the months of June and July. Across the Project area, annual low flows generally occur in late March when the majority of available water is stored within the snowpack. The onset of spring freshet typically takes place by early May. Peak flows are largely dependent on significant rainfall storm events, which can take place at any time during the summer months.

Strong winds generally occur in all seasons at high elevations.

Any future mining operations will be conducted year-round.

5.3 Infrastructure

The nearest large communities to the Project site are Terrace and Smithers.

The current exploration and construction camps can host 810 persons. Power is currently supplied to the exploration camp via diesel generators. The Galore Creek Project communicates to the outside world using voice-over-internet protocols (VOIP) for telephones) and Internet protocols (for regular computer business) over a satellite link. The satellite link terminates in Langley, BC, where it connects to regular land lines.

Infrastructure requirements for Project development as detailed within the GCMC 2011 pre-feasibility study are discussed in Section 18 of this Report.

5.4 Physiography

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

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 Project 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. The elevation of the tree line is variable, but alpine vegetation predominates above 1,100 m. 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 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.

The Stikine, Iskut, More, Sphaler, and Porcupine valleys 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.

5.5 Seismicity

As part of geotechnical studies performed on waste and water management in 2006, BGC assessed the seismic risk for the Project (BGC Engineering Inc, 2006b). The Galore Creek area is located in a moderately high seismic zone.

The national seismic hazard map produced by the Geological Survey of Canada for use in the 1995 National Building Code of Canada, indicates that the Project is located in acceleration zone 2, characterized by a peak horizontal ground acceleration (PGA) of 0.8 *g*[1] to 0.11 *g* with a 10% chance of exceedance in 50 years (1 in 475).

Revised seismic hazard maps for incorporation into the 2005 National Building Code of Canada show that the site has a PGA of approximately 0.1 *g* to 0.2 *g* with a 2% chance of exceedance in 50 years (1 in 2,475).

5.6 Comment on Section 5

In the opinion of the AMEC QPs, the existing and planned infrastructure, availability of staff, the existing power, water, and communications facilities, the methods whereby goods could be transported to any proposed mine, and any planned modifications or supporting studies are reasonably well-established, or the requirements to establish such, are reasonably well understood by GCMC, NovaGold and Teck, and can support the declaration of Mineral Resources and Mineral Reserves.

Within the ground holdings of GCMC, there is sufficient area to allow construction of all required Project infrastructure. Except for the access corridor which is covered by the SUP, all other infrastructure, including the processing plant and tailings area in West More and for the Filter Plant Area near Km 8 are located within GCMC's mineral claims. GCMC intends to file for mining leases to secure the surface rights for these areas.

Surface rights are held by the Crown. GCMC considers it a reasonable expectation that surface rights usages will be granted the Project.

It is expected that any future mining operations will be able to be conducted year-round.

1 *g* is the the acceleration due to Earth's gravity

6.0 HISTORY

6.1 Early Exploration of the Galore Creek Claims

Mineralization was first discovered in the upper Galore Creek Valley in 1955 by M. Monson and W. Buchholz while prospecting for a subsidiary of Hudson Bay. Staking and sampling were completed in the area in 1955. Work in 1956 included mapping, trenching and diamond drilling. No further work was undertaken and most of the claims were allowed to expire.

In 1959, reconnaissance stream sediment surveys were carried out by Kennco Explorations (Western) Limited (the Canadian subsidiary of Kennecott Copper, now Rio Tinto Ltd.) in the Stikine River area. Results prompted Kennco to stake mineral claims around the remaining 16 Hudson Bay claims the following year. Four of the original claims were subsequently optioned by Consolidated Mining and Smelting Company of Canada Limited (Cominco) from W. Buchholz. Late In 1962, the three companies agreed to participate jointly in future exploration work. As a result, Stikine Copper Limited was incorporated in 1963, on the basis of the following interests: Kennco Explorations, (Western) Limited (Kennecott; 59%), Hudson Bay Mining and Smelting Company Limited (Hudson Bay; 34%) and Consolidated Mining and Smelting Company of Canada, Limited (Consolidated; 5%).

Work conducted since discovery in 1955 outlined a significant gold–silver–copper mineralized zone in the Central Zone and identified several satellite mineralized zones, most importantly the Southwest, West Fork, North Junction and Junction Zones. This work included soil sampling, pole-dipole resistivity/IP, magnetics, electromagnetics (EM), radiometrics, very low frequency (VLF) and audio frequency magnetotellurics (AFMAG) airborne geophysical surveys, and drilling.

From 1960 to 1968, the exploration on the property was performed by Kennco. Exploration work during this period included 53,164 m of diamond drilling in 235 holes and 807 m of underground development work in two adits. The Central Zone was the focus of most of this work. During the same period, a road was constructed from an airstrip at the confluence of the Stikine and Scud rivers along the Scud River and up Galore Creek to the then exploration camp.

No work was done between 1968 and 1972. In 1972, Hudson Bay became operator and in 1972 and 1973 an additional 25,352 m of diamond drilling was completed in 111 holes. This work concentrated on the mineralization in the Central and North Junction Zones. A further 5,310 m of diamond drilling was completed in 24 holes in 1976.

In 1989, Mingold Resources Inc. (an affiliated company of Hudson Bay) operated the property in order to investigate its gold potential. In 1990, Mingold completed 1,225 m of diamond drilling in 18 holes.

Kennecott resumed as operator of the Project in 1991 and completed 13,830 m of diamond drilling in 49 holes. An airborne geophysics survey and over 90 line km of induced polarization (IP) survey were also completed. At the end of this initial exploration phase, a total of 12 prospects and deposits had been identified: Central, Junction, North Junction, West Rim, Butte, Southwest, Saddle, West Fork, South Butte, South 110, Middle Creek, and North Rim.

6.2 Exploration of the Grace Claims

The Grace Claims are a unit of five claims to the north of the area where Mineral Resources have been estimated, and became part of the Galore Creek Project in 2007. NovaGold acquired an interest in the claims in 2004, and the claims became part of the Project in 2007. Prior to that date, the claims had been subject to a different set of exploration programs to those conducted on the main Project claims.

In 2006, NovaGold completed six NQ-size diamond holes for 1,785 m. Holes were drilled as condemnation holes to verify there was no economically significant copper or gold mineralization in the area proposed to be covered by a tailings site for Galore Creek (Petsel, 2006). Additional geotechnical and hydrological work has been undertaken on the claims in support of development of potential tailings dams for the Galore Creek deposits (Petsel, 2006).

In 2007, 21 holes totalling 6,840 m were drilled on the Grace claims. Drilling on the property was designed to meet initial earn-in requirements for 60% interest in the property and to further condemn the low level targets of the tailings impoundment area. No significant mineralization was encountered.

6.3 SpectrumGold/NovaGold Exploration at Galore Creek

In August 2003, SpectrumGold Inc. (now NovaGold Canada Inc., and a wholly-owned subsidiary of NovaGold) entered into an option agreement to acquire a 100% interest in the Galore Creek property from Stikine Copper Limited, a company owned by QIT-FER et Titane Inc. (a wholly-owned subsidiary of Rio Tinto Ltd.) and Hudson Bay. From September–October 2003, SpectrumGold carried out a 10-hole, 2,950 m diamond drill program on the property. The work program was directed toward verifying grades of copper and gold mineralization defined by previous drilling in the Central and Southwest Zones.

In 2004, NovaGold carried out a 79 hole, 25,976 m diamond drill program to upgrade and expand the existing resource. Drilling was also conducted on exploration targets to test several peripheral occurrences and nearby properties in which NovaGold has an interest. Extensive geophysical surveys were conducted to assist the exploratory drilling. The results of the 2004 drilling program provided the basis for geological modeling, resource estimation, preliminary mine planning and economic evaluation at Preliminary Assessment (PA) level.

The aim of the 2005 Galore Creek exploration program was 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 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.

During 2006, 33,574.70 m of NQ and HQ-sized diamond drilling, in 57 holes, was completed. The 2006 drilling tested new exploration targets based on geophysical anomalies and new geologic interpretations, and included step-out drilling by expansion and/or extension of known mineralization, delineation drilling of proposed pit boundaries, and infill drilling of areas of known mineralization in an attempt to upgrade the resource estimation categories.

Drilling on the main Galore Creek property during 2007 totalled 4,547 m in 17 drill holes. The drilling was distributed among many areas, including the Southwest zone, the Central Zone, the Lower Butte Zone and in some reconnaissance areas. Initially discovered in 2005, work at Lower Butte was a follow-up to the minimal drilling on the zone done in prior years and was focused to help expand the extent of the deposit. Further efforts were directed toward using the results in the completion of a 3D model to be used in preliminary resource estimates. Additional holes were drilled in and around the Central Zone to spot test areas of potential expansion of the resource and to identify controls on mineralization, particularly in the Dendritic Creek area of the Central Replacement Zone of the Main deposit. Additional, but minor, geotechnical drilling was completed around the Southwest and Central Zones.

6.4 GCMC Exploration at Galore Creek

In 2008, GCMC carried out a nine hole, 2,050 m diamond drill program on the main Galore Creek property to obtain acid–base accounting (ABA) data in the area of the proposed Central, Southwest, North Junction and Junction pits. Grades of the legacy assays in the area of the proposed Junction pit were verified and metallurgical data were collected from the Central Replacement Zone and North Gold Lens.

In 2010, Galore Creek Mining Corporation (GCMC) carried out a nine hole, 2,803 m diamond drill program on the main Galore Creek property to obtain metallurgical and in-fill data in the Central deposit. Drill holes targeted mineralization that is likely to support the first five years of planned production within the South Gold Lens and Central Replacement Zone (named subdomains of the Central Zone).

6.5 Development Studies

An assessment of the development potential of the deposits was completed by Mine Reserve Associates, Inc. for Kennecott in 1992. A number of pit shells and pit designs were completed. In 2002, Kennecott reclassified the Project Mineral Resources in accordance with NI43-101 guidelines. As this work has been superseded by the GCMC 2011 pre-feasibility study, no further information on the Kennecott work is included in this Report.

A PA study was completed for NovaGold on the Galore Creek Project by partners Associated Mining Consultants and Hatch Limited (Hatch) in 2004 (Hosford, 2004), returning positive economics for the Project, and supporting ongoing studies.

A feasibility study on the Galore Creek Project was compiled by Hatch in October 2006 (Rustad et al, 2006), and incorporated a first-time declaration of Mineral Reserves for the Project.

In April 2007, AMEC was contracted to review capital and operating costs, and work completed on the Project. The review covered the entire Project with a focus on construction of the mine facilities and tailings and water management structures. In October 2007, AMEC's preliminary work indicated that capital costs would be significantly higher than originally estimated in the feasibility study. The Project was also impacted by the rapidly escalating capital costs affecting major construction projects worldwide. This, combined with reduced operating margins because of the stronger Canadian dollar, made the Project uneconomic as conceived and permitted from the October 2006 feasibility study. The feasibility study update was discontinued. In light of the economic status of the Project at the time, NovaGold and Teck agreed to reclassify Mineral Reserves as Mineral Resources, in compliance with NI 43-101 requirements.

In January 2008, NovaGold updated the Galore Creek Mineral Resource estimate to include drilling completed during 2007 and to support the reclassification of the Mineral Reserves as Mineral Resources.

GCMC asked AMEC to undertake a two-phase optimization study during 2008, in an effort to reduce capital, mitigate the Project's risk, and improve the Project schedule, constructability and financial profile. The November 2007 basic engineering report served as the basis of this 2008 optimization study. At the end of Phase 2, two options were identified as having the most potential for a path forward:

- Case 35 – Mining, crushing and grinding in Galore Creek Valley, 9.2 km tunnel, flotation near Km 81 of the access road, tailings storage facility (TSF) at West More Area (Km 77), concentrate pipeline to Stewart, and filtration and new load-out facilities at Stewart
- Case 37 – Mining and crushing in Galore Creek Valley, grinding and flotation near Km 91 of the access road, 11.6 km tunnel, TSF at West More Area, concentrate pipeline to Stewart, and filtration and new load-out facilities at Stewart.

In 2010–2011, GCMC commissioned a number of third-party consultants to assist GCMC in preparing a pre-feasibility study on the Project, which included multiple trade-off option considerations. The remainder of this Report summarizes the results of the GCMC 2011 pre-feasibility study and discusses modifications to the GCMC 2011 pre-feasibility study made by AMEC and Lemley.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Setting

During the Late Palaeozoic to Late Mesozoic, the Canadian Cordillera was formed as an assemblage of oceanic and near-continental terranes accreted onto the western margin of the North American craton. The accreted terranes form five morphogeological belts, namely the Foreland, Omineca, Intermontane, Coast, and Insular belts. The Intermontane belt consists of the Stikinia, Cache Creek, Slide Mountain and parts of Quesnellia and Yukon-Tanana terranes (McMillan, 1991).

Similarities in rock type and geological history between the Stikinia and Quesnellia terranes have led a number of researchers to consider that the two terranes are segments of the same Triassic arc (e.g. Wernicke and Klepacki, 1988; Nelson and Mihalynuk, 1993; Mihalynuk et al., 1994). The Galore Creek district is one of seven major mineralized alkalic porphyry systems in the Stikinia–Quesnellia arc.

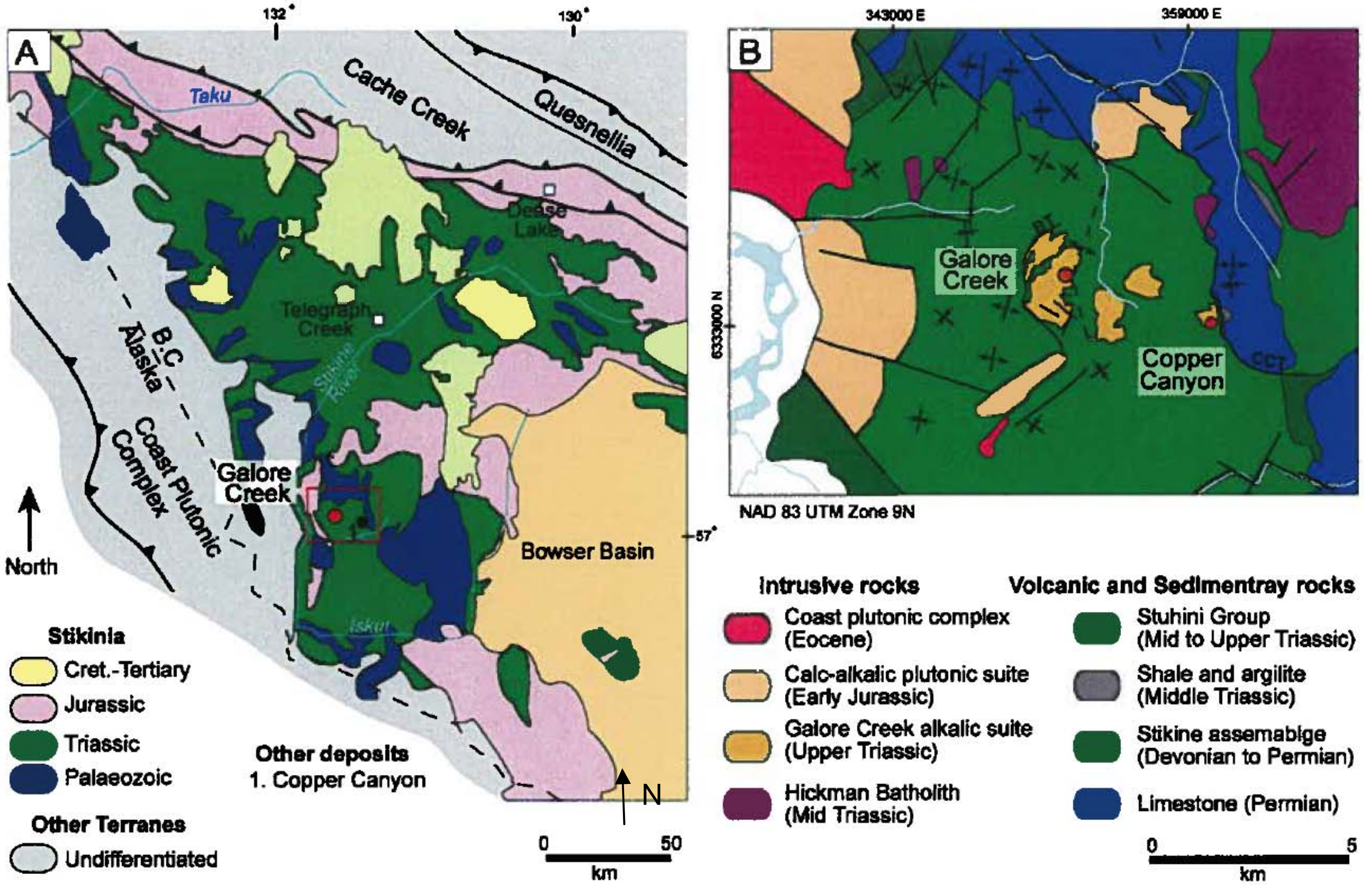
The Stikinia terrane consists of four early Devonian to middle Jurassic arc-related mafic to felsic volcanic rocks, co-eval plutons, and sedimentary rock sequences that are separated by unconformities (Figure 7-1) as follows:

- Late Palaeozoic to Middle Jurassic island arc volcano-plutonic and sedimentary rocks of the Stikine assemblage, the Stuhini Group and Hazelton Group
- Middle Jurassic to early Upper Cretaceous basin sedimentary rocks of the Bowser Lake Group
- Upper Cretaceous to Tertiary continental arc volcanic rocks of the Sloko Group
- Late Tertiary to Recent post-orogenic plateau basalts of Edziza and Spectrum Ranges.

The terrane has been affected by three major periods of intrusive activity, which in the Galore Creek area include:

- Upper Triassic to Lower Jurassic intrusion that include the calc-alkaline Hickman pluton and the Galore Creek alkalic suite
- Upper Cretaceous to Palaeocene Coast plutonic complex that occurs as several granite bodies west of Galore Creek
- Tertiary quartz monzonite, diorite stocks and mafic to felsic dykes that occur within west- and north-striking extensional structures.

Figure 7-1: Regional Geological Setting



Note: Figure from Byrne (2009), and modified after Wheeler and McFeely 1991; Gabrielse et al. 1991; Logan and Koyanagi, 1994; and Enns et al., 1995

7.2 Galore Creek Area Geology

Syenite, monzonite and monzodiorite dykes and stocks of the multiphase Galore Creek alkalic complex are hosted within Stuhini Group rocks. The highest concentration of intrusions occurs in the west fork of the Galore Creek Valley, where the intrusive complex, consisting of composite dykes and stocks. The intrusions are considered to be sub-volcanic, and co-eval and co-magmatic with the younger units of the Stuhini Group. Figure 7-2 shows the local geology of the mineralized portion of the Galore Creek Valley. Timing relationships of mineralization to the various intrusive units suggests at least two distinct periods of hydrothermal activity punctuated by intrusion of voluminous, megacrystic, orthoclase-phyric syenite and monzonite dykes.

In the Galore Creek Valley, the Stuhini Group comprises a lower unit of submarine basaltic to andesitic volcanic rocks interspersed with locally derived sandstones and siltstones (Allen et al., 1976) and an upper unit of partially subaerial, compositionally distinct, alkali-enriched, volcanic and volcanogenic sedimentary rocks.

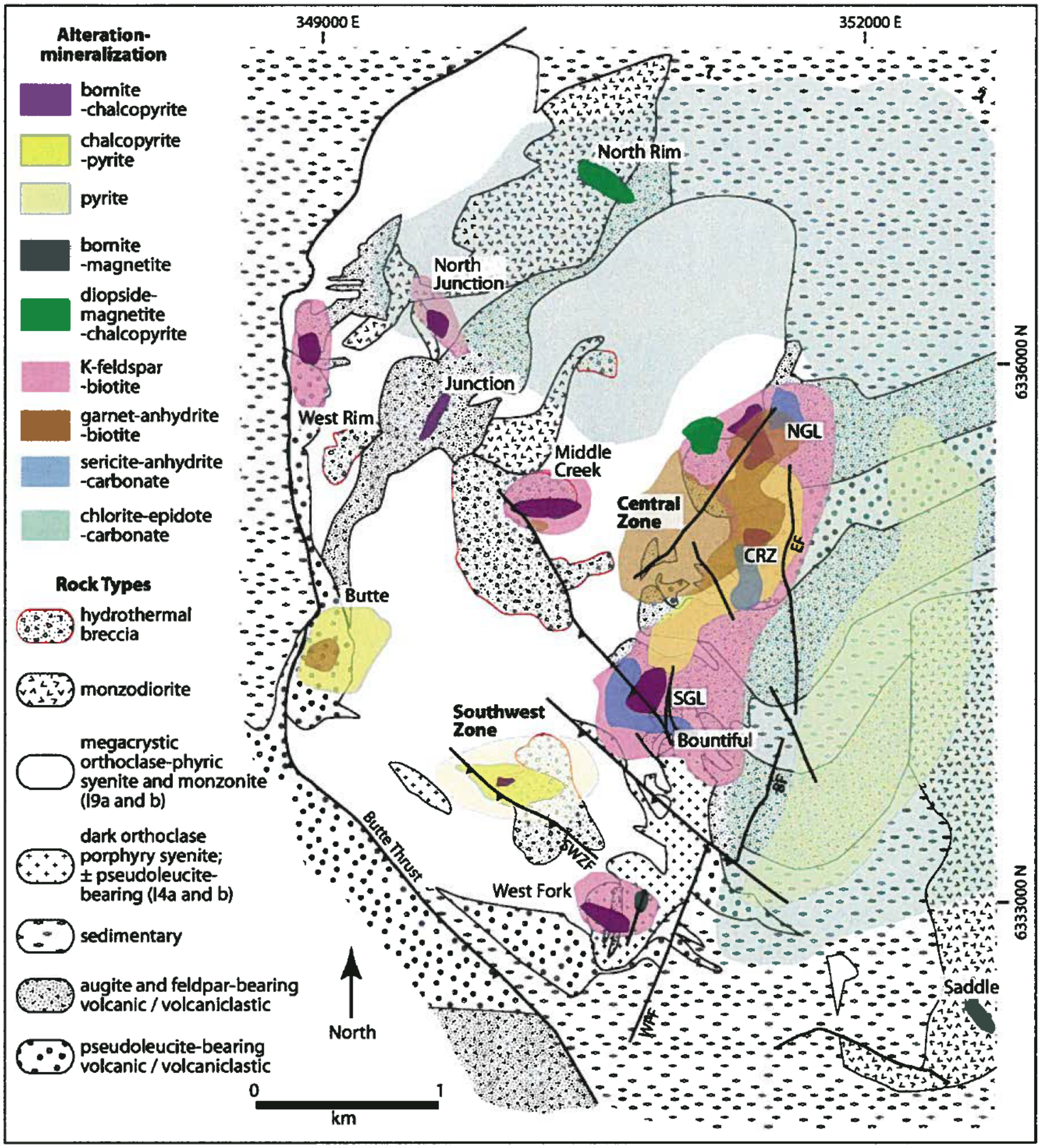
The Galore Creek district is interpreted to have undergone early and broad-scale Triassic north–south compression followed by post-early Jurassic development of northerly-trending folds and thrust faults (Logan and Koyanagi, 1994). Small displacement reverse faults cut the Stuhini Group rocks, the intruding syenite and monzonite intrusions, and the mineralized zones.

Mineralization is developed in potassium-enriched volcanic rocks and pipe-like breccias adjacent to syenite stocks and dykes. Deposits are manto-shaped, and trend north to northeast, following either, or both, syenite contacts and structural breaks.

The largest deposit is the northerly-elongated Central Zone that is divided into the North Gold Lens, Central Replacement Zone and the South Gold Lens, and has the Bountiful zone partially superimposed. Smaller deposits peripheral to the Central Zone are also known, these include: Southwest Zone, Junction, Butte, West Rim, West Fork and the Saddle zones (Figure 7-3).

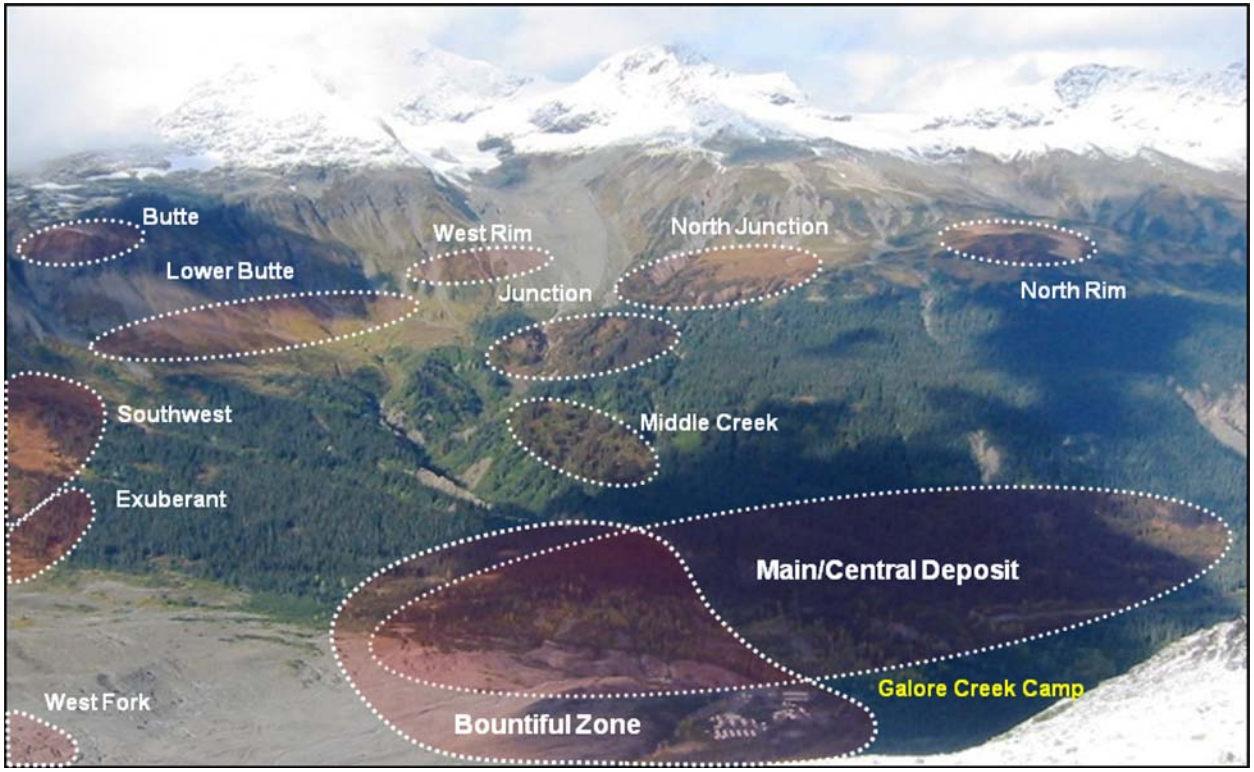
Most zones, including the Central, North Junction, Junction, Middle Creek, West Rim, Butte and South 110, occur in highly-altered volcanic rocks and to a lesser degree in syenite intrusions. The Southwest, Opulent Vein, and Saddle zones are hosted by breccias and the North Rim and West Fork zones occur within syenite intrusions.

Figure 7-2: Local Geology Plan



Note: NGL = North Gold Lens; CRZ = Central Replacement Zone; SGL = South Gold Lens; SWZF = Southwest Zone Fault; WFF = West Fork Fault; BF = Bountiful Fault; EF = East Fault. Figure from Byrne (2009).

Figure 7-3: Deposit Location Plan



Note: Photograph looks west. As the photograph is a perspective view to show the relative locations of the various prospects and deposits to the Galore Creek Camp and Valley, no scale is included. Figure courtesy NovaGold.

7.3 Lithological Descriptions

A total of 107 geological codes are used to describe the geology of the Galore Creek area during drill hole logging. About 30 different primary rock types have been determined; subdivisions of these primary types, based on textural differences or temporal (cross-cutting) relationships, account for the remaining codes. Stikine Copper Limited delineated the first 100 codes in 1991. Seven additional codes were created in 2004 by NovaGold.

7.3.1 Volcanic Rocks

At Galore Creek the volcanic rocks are defined by four main classifications and numerous sub-types based on texture and mineral content. From youngest to oldest they include the following:

- V5/S6-7 – Intermediate volcanic rocks and sediments characterized by re-sedimented pyroclastic rocks and flysch turbidites
- V3 – Orthoclase-bearing volcanic rocks characterized by orthoclase-rich pyroclastic rocks and reworked sediments
- V2 – Pseudoleucite-bearing volcanic rocks characterized by trachytic pseudoleucite-rich lavas
- V1/V4 – Augite-bearing mafic volcanic rocks characterized by mafic lavas and reworked equivalents.

Augite-Bearing Volcanic Rocks (Rock Code V1)

A heterogeneous sequence of augite-bearing mafic flows, flow breccias and volcaniclastic rocks are interbedded with pseudoleucite volcanic rocks in the northern portion of the Central Zone. These rocks generally host only weak to moderate mineralization in comparison to the pseudoleucite-bearing rocks.

Augite-bearing flows contain porphyritic and, infrequently, amygdaloidal textures. Interbedded with the augite-bearing flows are augite-bearing volcaniclastic rocks in the form of fine and coarse lapilli tuffs, tuff breccias and flow breccias containing sub-angular to sub-rounded fragments of augite porphyry. These volcaniclastic rocks are generally matrix-supported.

Pseudoleucite-Bearing Volcanic Rocks (Rock Code V2)

Pseudoleucite-bearing trachytes occur as moderately west-dipping sequences interbedded with augite-bearing units, intermediate and lesser mafic volcanic rocks. The original textures are often obliterated by intense orthoclase and sericite alteration. Copper and gold mineralization appears to occur preferentially in these rocks. In the Central Zone, fragments of pseudoleucite-bearing volcanic rocks are present in mineralized hydrothermal breccias that also contain abundant garnet.

Orthoclase-Bearing Volcanic Rocks (Rock Code V3)

Orthoclase-bearing volcanic rocks are predominantly fine and coarse crystal lithic tuffs with possible subordinate flows, and are common in the southern part of the Central Zone, where they crop out on surface and are often seen in drill core. In this area, they are often strongly mineralized with disseminated bornite, chalcopyrite and gold. They appear to be cogenetic and coeval with dark syenite porphyry intrusive bodies, which may be their subvolcanic equivalents.

Undifferentiated Volcanic Rocks (Rock Codes V4, V5, V6)

In some areas, intense alteration has obliterated original textures resulting in the more generic classification of “undifferentiated volcanics”. Such rocks have been classified on the basis of colour and association.

Mafic volcanic rocks (*V4*) are dark green, chlorite-rich flows and tuffs, and are common in the north part of the Central Zone. These are interbedded, and may, in part, be correlated with unit *V1*. Porphyritic and amygdaloidal flow textures have been preserved locally, and volcanic clasts are sometimes preserved in pyroclastic rocks.

Intermediate volcanic rocks (*V5*) are very common in the Central Zone. These rocks are medium-greenish-grey volcaniclastic rocks and flows, and may be aphyric equivalents of the *V2* unit. Included in this unit are possible trachy-andesites containing sub-rounded orthoclase phyric fragments. Secondary biotite occurs both as a spotted to patchy alteration and as coarse aggregates and veins.

Intense orthoclase flooding has resulted in pale grey, felsic volcanic rocks (*V6*) which are fine- to medium-grained volcaniclastic rocks and flows. *V6* rocks are present in the north and central part of the Central Zone, often interbedded with pseudoleucite volcanic rocks that may be their equivalent.

7.3.2 Sedimentary Rocks

Sedimentary rocks, such as diamictites and epiclastic rocks, derived from re-worked volcanic material are often found interbedded with their volcanic equivalents. Minor siltstone, argillite, greywacke and conglomerate are common immediately to the north of the Central Zone. Sedimentary structures such as graded bedding, flame structures and channel scour features have been observed in drill core north of the Central Zone, from outcrop in North Rim Creek and more rarely in drill core from the Central Zone. Where observed in outcrop, tops indicators show the sequence is younging upward.

Conglomerate (Rock Code S1)

Conglomerates are common north of the Central Zone, in North Rim Creek and North Rim Zone, and in the North Junction Zone. The unit is heterolithic and unsorted. Fragments of volcanic and syenitic rocks are present and comprise up to 30% of the rock. Conglomerate contains local intercalations of argillite and greywacke. Channel scours and load casts are common.

Greywacke (Rock Code S2)

Grey-green, poorly sorted, medium- to coarse-grained greywackes are common north of the Central Zone, in North Rim Creek. They also appear rarely in drill core within the Central Zone as intercalations with lapilli tuffs.

Siltstone (Rock Code S3)

Siltstone is fine-, to medium-grained, grey, massive- to well-bedded and locally contains graded bedding.

Argillite (Rock Code S4)

Argillite occurs as alternating medium- to dark-grey and black, aphanitic, well-bedded sequences. Beds vary in thickness from 0.5 to 1 cm.

Limestone (Rock Code S5)

Micritic or crystalline limestone is primarily sedimentary in origin, and includes many variations of grain size and bed thickness. It is most commonly found in Copper Canyon (see Section 7.3).

Epiclastic Sediments (Rock Code S6)

This unit is a composite lithology consisting primarily of reworked volcanic material. It includes clay-rich (lacustrine) beds, siltstone, fine-, to coarse-grained sandstone, and conglomerate.

Diamictite (Rock Code S7)

The diamictite consists of unsorted, mono- or polylithic fragments that are matrix-supported. The matrix consists of a mixture of clay, silt or sand. It commonly shows either normal or reverse grading, and may have formed as a result of mass gravity flows such as lahars or debris flows.

7.3.3 Intrusive Rocks

The Galore alkaline intrusive complex underlies the district and controls the known Cu-Au mineralization. The intrusions occur primarily as sills but also form dikes and stocks. The suite of intrusions can be broadly characterized into five distinct magmatic pulses, from oldest to youngest, as follows:

- Early pseudoleucite-bearing porphyries of which *I4* is the first major magmatic pulse
- A subsequent suite of more equigranular to weakly porphyritic syenites
- A relatively late voluminous event characterized by very distinctive megaporphyries
- A late trachytic syenite porphyries as thin dykes and sills
- A series of thin aphantic felsic to mafic dykes.

The Galore Creek intrusive rock classification scheme including over 30 intrusive rock types was built by Kennecott in the 1990s, based on observed field relationships particularly temporal relationships implied by pre- and post-mineral cross-cutting relationships in the various intrusions. Recent recognition by GCMC of overlapping mineralized systems along the end of the Central deposit will simplify the previous magmatic complexities arising from the Kennecott framework, and is expected to result in a more coherent interpretation.

Multiple intrusive phases are present in the complex and divided into pre-, inter-, late- and post-mineralization phases. The classification is based on crosscutting relationships, together with the degree of alteration and mineralization characteristics.

Petrological examination (Enns et al, 1995) has shown that the Galore Creek intrusive rocks contain variable proportions of orthoclase, plagioclase (oligoclase or albite), pseudoleucite, melanite, clinopyroxene, biotite and hornblende phenocrysts in a matrix of pilotaxitic K-feldspar, disseminated magnetite, apatite and titanite.

Early intrusive units (*rock units I1 through I5*) consist of K-feldspar and pseudoleucite porphyritic dykes and sills. These are followed by relatively equigranular intrusions (*I6 and I8*), K-feldspar porphyritic and megaporphyritic units (*I9–I11*), and a relatively equigranular intrusion (*I12*). This apparent oscillation between porphyritic and equigranular textures may reflect variations in the volatile fugacities of the melts. The modal change in the primary mineral assemblage of the Galore Creek intrusions, from syenitic to monzonitic, back to syenite and finally to quartz syenite, suggests differences in the compositions of the parent melts.

Pre-Mineralization Intrusions (Rock Codes I1 to I3)

Pseudoleucite porphyries and mega-porphyries (*I1 and I2*) are relatively rare, and occur most often as thin, steeply-dipping dykes in the Central Zone. Distinct chill margins are often observed at contacts.

Early mapping and logging includes a Grey Syenite Porphyry (*I3*), also shown on historic drill sections as “Dark Syenite Porphyry”. Work by NovaGold from 2003 to 2005 did not substantiate the existence of this unit.

Inter-Mineralization Intrusions (Rock Codes I4a and I4b)

Dark orthoclase syenite, both early (*I4a*) and late (*I4b*), is the most common syn-, to late-mineral intrusive in the southern part of the Central Zone. The unit is also common in the West Fork Zone, where it occurs unmineralized near surface, and as a mineralized, flat-lying, tabular body at depth.

Late-Mineralization Porphyries (Rock Codes I5 to I12)

Dykes and/or sills of late-mineralization porphyries intrude the Central and Junction and North Junction zones and may also constitute a large stock between these zones. These phases are easily recognized because of the lack of mineralization and the propylitic alteration assemblage. The most common phases are grey equigranular to porphyritic, medium-grained syenite (*I8*), mega-porphyry (*I9a and I9b*), grey medium-grained syenite porphyry (*I11a*) and lavender porphyry (*I12*). Minor phases include dykes and small stocks of fine-grained syenite and syenite porphyry (*I6 and I7*), plagioclase syenite porphyry (*I10*) and medium-grained syenite porphyry (*I11*).

Minor mineralization associated locally with these dykes suggests that they are either syn-mineral, or that minor late mineralization was introduced with them.

Breccias (Rock Codes B1 to B3)

Diatreme, hydrothermal, and orthomagmatic breccias (*B1, B2 and B3* rock codes, respectively) are distinguished at Galore Creek mainly by clast shape and lithology, matrix composition, alteration assemblage and the presence of mineralization.

The mineralization in the Southwest Zone is hosted by an unbedded, polylithic, matrix-supported breccia. Several relatively small bodies of orthomagmatic breccia carrying copper–gold–silver mineralization were emplaced during the latter intrusive phases of the system. Breccia bodies south and west of the Central Zone carry clasts of propylitized, late-mineralization mega-syenite porphyry.

Other orthomagmatic breccias include the mineralized Saddle Zone breccia and the West Fork breccia. The latter was found to contain clasts of unaltered intrusive phases including *I11*, suggesting that it formed late in the intrusive sequence.

7.4 Structure

The oldest Palaeozoic rocks (pre-Upper Triassic) have widespread penetrative planar fabrics, north to northwest-trending isoclinal folding (*D1*), northwest-trending upright open folding (*D2*), and west to northwest-trending chevron folds and kink bands (*D3*). *D1* and *D2* are characterized by regional metamorphism to greenschist facies.

The earliest structures are syn-metamorphic, pre-Triassic, potentially Carboniferous age. These structures, and related northeast-striking penetrative foliations, are deformed by west-trending folds that are interpreted to be of post-Triassic age. Two post-Early Jurassic events are recognized, one is characterized by north-trending southwest-verging folds and reverse faults; and the younger is characterized by northeast-verging kink folds.

7.4.1 Faults

The Galore Creek area is a mosaic of fault-bound blocks, controlled by five major fault sets (Logan and Koyanagi, 1994). From oldest to youngest these include: north-trending vertical faults; east and west-dipping reverse faults, northwest-striking vertical faults in part coeval with, but also truncated by the north-trending structures; west-trending vertical shear zones and normal, generally north-side down, extensional faults and northeast striking sinistral shear zones and vertical normal faults (Logan and Koyanagi, 1994).

Cataclasite zones are associated with all but the youngest northeast-trending faults. These have channelled fluids and alteration has obliterated protolith rocks in zones up to tens of metres wide (Logan and Koyanagi, 1994).

7.4.2 Folds

Upper Triassic strata are folded about two virtually orthogonal axes. The older folds are generally west-trending, the younger are north to northwest-trending. However, no folded cleavages or refolded folds have been noted in mapping (Logan and Koyanagi, 1994).

7.5 Alteration

Alteration and mineralization are contemporaneous, and spatially overlap. Four main alteration facies have been described at Galore Creek. The extent of alteration was indicated in Figure 7-2.

Potassic alteration associated with the introduction of copper sulphides is the most widespread, and dominant, alteration type. K-feldspar flooding, most intense in the Central Zone, affects the volcanic and early intrusive rocks in all areas of the deposit. Biotite alteration is present and is closely associated with copper mineralization and sheeted gypsum fracturing. The combined occurrence of gypsum fractures and fine-grained biotite cause mineralized outcrops to weather recessively.

Within the core region of the Central Zone a “Ca–K-silicate” assemblage exists which is characterized by the presence of dark brown garnet with locally occurring diopside, epidote and plagioclase. This alteration most probably occurred when the potassic “K-silicate” altering fluids encountered more calcic mafic rocks, derived excess Ca and precipitated garnet. Garnet alteration decreases from the core region of the Central Zone toward the north and south, and is accompanied by increasing magnetite and early hematite.

Magnetite is an abundant accessory in the syenite porphyries. When present, the copper sulphide most commonly associated with it is chalcopyrite (Proffett, 2005). Magnetite is also present as disseminations and veinlets throughout altered volcanic rocks at Galore Creek. The greatest concentrations of magnetite occur in magnetite breccias at the Saddle Zone and the West Fork Zone.

The second alteration phase is a later potassic phase. In the later porphyries, such as *I8* and *I11*, plagioclase that is only partly altered to K-feldspar is usually present, and because the relict plagioclase is a source of calcium, epidote may be common. Mafic minerals are usually altered to secondary biotite that is distinctly green compared to the biotite of the more intense potassic alteration, which is black under a hand lens (Proffett, 2005).

Sericite–anhydrite–carbonate (SAC) alteration overprints the dominant early alteration phases and is locally extensive. In the southern part of the Central Zone it is accompanied by late hematite. In the northern portion it is marked by increasing pyrite ± hematite and decreasing bornite-chalcopyrite, suggesting that copper was remobilized during SAC alteration. In the core of the Central Zone, SAC alteration is patchy and more common on the periphery. The later hydration of anhydrite to gypsum by groundwater action and consequent volume increase has resulted in intense sheet fracture development in parts of the Central Zone to depths of as much as 213 m below surface. The gypsum has been leached out to depths ranging from 30 to 122 m, leaving loose crumbly sheets of rock. The broken rock boundary is defined by an abrupt change in rock quality designation (RQD) values and generally mimics the topography. The sheet fractures are best developed in volcanic rocks.

Salmon-coloured alteration with abundant carbonate is common in several zones up to several metres wide, especially along fault zones. The presence along late faults indicates that this alteration is late. Other minerals that may be present are sericite, specular hematite, pyrite and chlorite. K-feldspar and chalcopyrite may also be present, but these minerals may be relicts from earlier assemblages. Bornite is rarely, if ever present, and any bornite that may have been present in the rock before this alteration occurred was apparently sulphidised to chalcopyrite (Proffett, 2005).

7.6 Genesis

The current genetic model for the Galore Creek Project is being refined, based on student research commissioned by NovaGold and ongoing review of drill core and geophysical data. The following presents the current interpretations by GCMC on deposit genesis.

Multiple stages of intrusive activity have resulted in nested porphyries and overlapping mineralized systems in the Galore Creek district.

The earliest intrusive system and mineralizing event in the district is interpreted to be the emplacement of the Copper Canyon porphyries which is interpreted as an eruptive volcanic centre along the Triassic magmatic arc. This shed debris laterally into the Galore Creek basin.

Mineralization related to the Central system including Junction and the Butte zone is characteristically hosted in intensely-altered volcanic and volcaniclastic rocks filling the active Galore Creek basin to the west of Copper Canyon. The causative magmatic pulse driving this mineralizing event is as yet unrecognized. The moderately-voluminous *I4* porphyry which is weakly mineralized on its margins was emplaced just after the Central Zone mineralizing event on the southwest margin of the deposit.

The Southwest and West Fork deposits are related to a third mineralizing event which post dates the earlier events and is centered to the south of those deposits in the headwall of the Galore Creek Valley. Voluminous sills, dykes and breccias crosscut the Central deposit mineralization but in turn are mineralized by fluids of the younger West Fork system. The causative magmatic phase responsible for the West Fork system has not yet been identified, but a unique mineralized phase informally termed the "West Fork porphyry" has been encountered in drill core, and could be the progenitor of the mineralized system.

The Central and West Fork systems overlap along the southern margin of the Central deposit and have greatly complicated interpretation of the district but represent exploration potential for the Project.

7.7 Mineralization

Disseminated pyrite is the most abundant sulphide mineral. Chalcopyrite and bornite in the ratio 10:1 are the main copper minerals. Sphalerite and galena are associated with garnet-rich areas and trace amounts of molybdenite, native silver, native gold and tetrahedrite have been noted (Allen, 1966). Magnetite occurs in veinlets with or without chalcopyrite, and often cements breccias. Secondary minerals identified include chalcocite, cuprite, native copper and tenorite.

Bornite and generally higher-grade gold are developed in the intense potassic alteration zone, and are associated with magnetite and sparse pyrite. Within the propylitic zone, zones of moderate potassic alteration have developed, and have associated chalcopyrite and pyrite mineralization. External to these potassic zones, but still within the propylitic zone, replacement lodes of gold, silver and base metals have formed.

In general, mineralization shows a progression from bornite laterally to chalcopyrite with increasing pyrite peripheral in the system. Isolated intervals of anomalous gold occasionally >1 g/t Au have been encountered.

7.8 Oxidation

Where surface exposures have been mapped in the Central deposit, malachite, azurite, hematite, and limonite, occurring both as locally pervasive disseminations and as widespread fracture-controlled infills, have been noted. Rare chalcocite and native copper were also noted. Typically, chalcopyrite and bornite are also found with the copper oxides. Drill core logs from Central have identified an oxide zone with average depths of 20 m below the overburden surface; however, weak fracture-controlled oxidation has been logged to depths of ≥ 200 m. The oxide zone mimics the topography, weakens down-hole, and rarely reaches the broken–stick rock surface.

Oxidation is variable outside of the Central Zone. The Junction and West Fork deposits show trace to minimal oxidation. The Southwest Zone shows minimal oxidation and the Middle Creek deposit shows significant oxidation.

7.9 Deposits and Prospects

A map of the deposits and prospects identified within the Galore Creek Property as at the effective date of this Report was included as Figure 7-2.

7.9.1 Central Zone

The Central Zone consists of the North Gold Lens, the Central Replacement Zone, and the South Gold Lens, which are differentiated based on gross differences in mineralogy.

The long axis of the Central Zone deposit has an orientation of 015° and dips steeply to the west. It is 1,700 m long, 200 to 500 m wide and has been traced to a depth of 450 m and remains open at depth. Mineralization crops out in the southern part of the zone, but elsewhere is covered by up to 75 m of glacially-derived material (Workman, 2006a).

The deposit plunges gently to the north and at its maximum thickness is 335 m. It comprises several parallel, en-echelon copper zones centred on a steeply-dipping breccia pipe. Mineral zoning consists of an intense potassic core zone, which hosts the mineralization, and a spotty propylitic alteration zone which occurs mainly along the eastern edge of the deposit.

The mineralization in the Central Zone is primarily disseminated and fracture-controlled chalcopyrite with subordinate bornite. The main hosts for copper mineralization are volcanic and volcaniclastic rocks with some mineralization occurring in the early intrusive rocks. Chalcopyrite mineralization occurs throughout the Central Zone accompanied by locally abundant bornite in the north and south parts of the deposit. Gold occurs in association with bornite. Elevated disseminated pyrite, reaching concentrations up to 5%, occurs in the footwall and is associated with a transition boundary between the Central Zone potassic alteration and the propylitic alteration halo. Pyrite mineralization in the Central Zone is not constrained by rock type. Magnetite occurs disseminated throughout the volcanic rocks; however, it is most abundant in the north part of the deposit. Secondary copper mineralization (malachite, azurite, and chrysocolla) is relatively minor and occurs primarily on fractures within 20 m of the surface.

The intensity of copper mineralization is mainly influenced by lithology. The pseudoleucite-bearing volcanic rocks in the north and the dark crystal tuffs in the south are the most favourable hosts to mineralization. Augite-bearing volcanic rocks appear to have been less receptive to mineralization. The copper mineralization is broadly conformable to volcanic stratigraphy in the northern half of the deposit. In the south, pod-shaped zones of disseminated chalcopyrite and bornite generally reflect the distribution of volcaniclastic host rocks (Workman, 2006a).

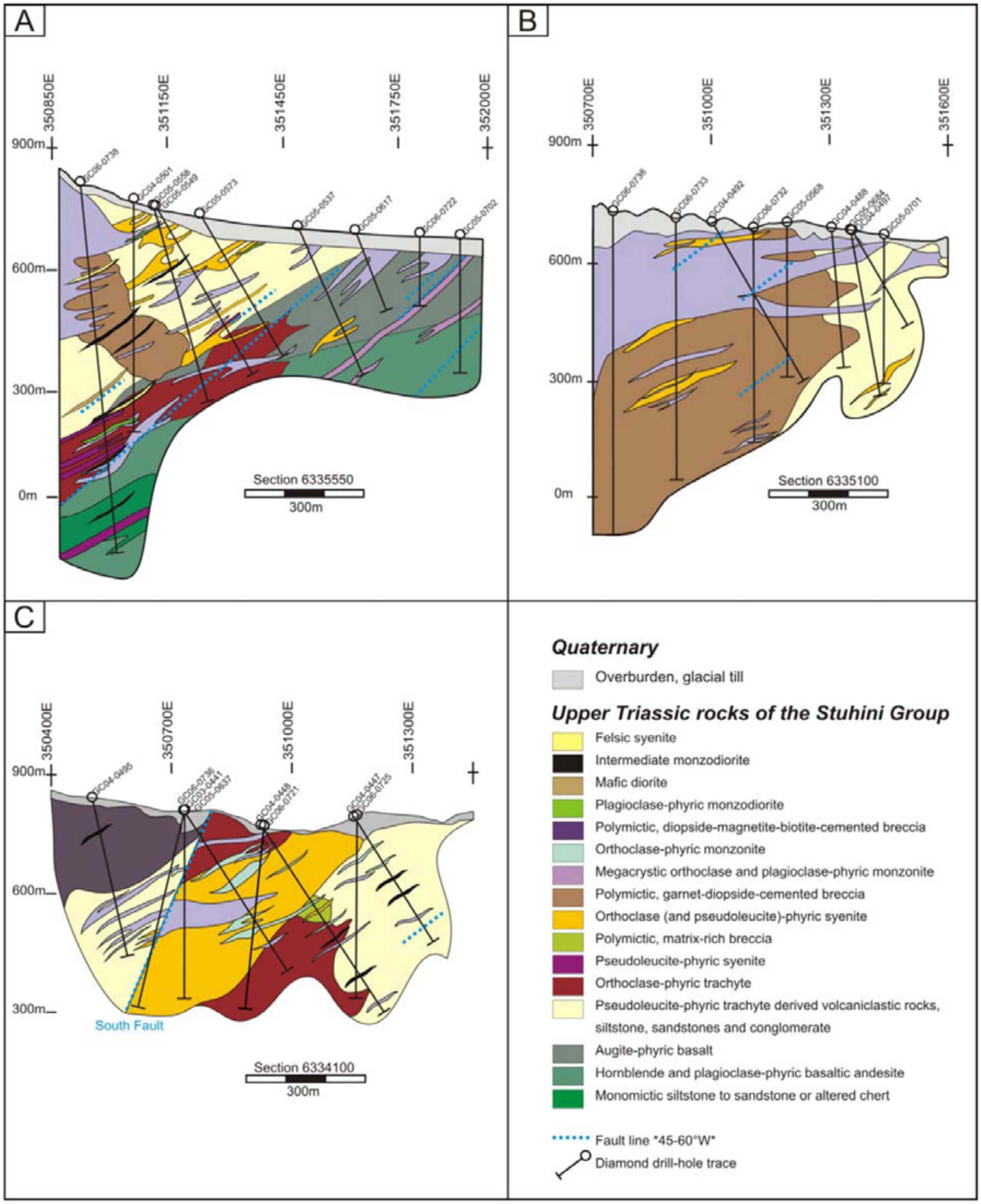
The North Gold Lens is characterized by K-spar, biotite and magnetite alteration with elevated gold and bornite. The Central Replacement Zone is characterized by lesser K-spar and biotite and the presence of abundant garnet. Mineralization is dominated by chalcopyrite and pyrite and lower gold values. Mineralization is interpreted to be more distal and gives way to strong propylitic alteration with epidote and pyrite further to the east. The alteration and mineralogy of the South Gold Lens is similar to that of the North Gold Lens with elevated gold and bornite and is interpreted as a proximal higher temperature assemblage.

An early suite of syenite porphyries (*i1–i4a*) are moderately to intensely altered and mineralized. The younger dark syenite porphyry (*i4b*), so abundant in the southern part of the Central Zone, is generally, weakly mineralized. Late mega-porphyry intrusive bodies (*i8, i9b*) are weakly mineralized to barren. They cut mineralized bodies, and are considered a late-mineral intrusive phase (Workman, 2006a).

The eastern boundary of the Central Zone mineralization lies near the surface projection of a major, steeply west-dipping, brittle, normal fault. In the west and south, mineralization is partially truncated by post-mineral mega-porphyry dykes. In the north, mineralized volcanic rocks end abruptly against a thick sequence of weakly to unmineralized epiclastic sedimentary and volcanic rocks as a result of a west–northwest-oriented post mineral fault.

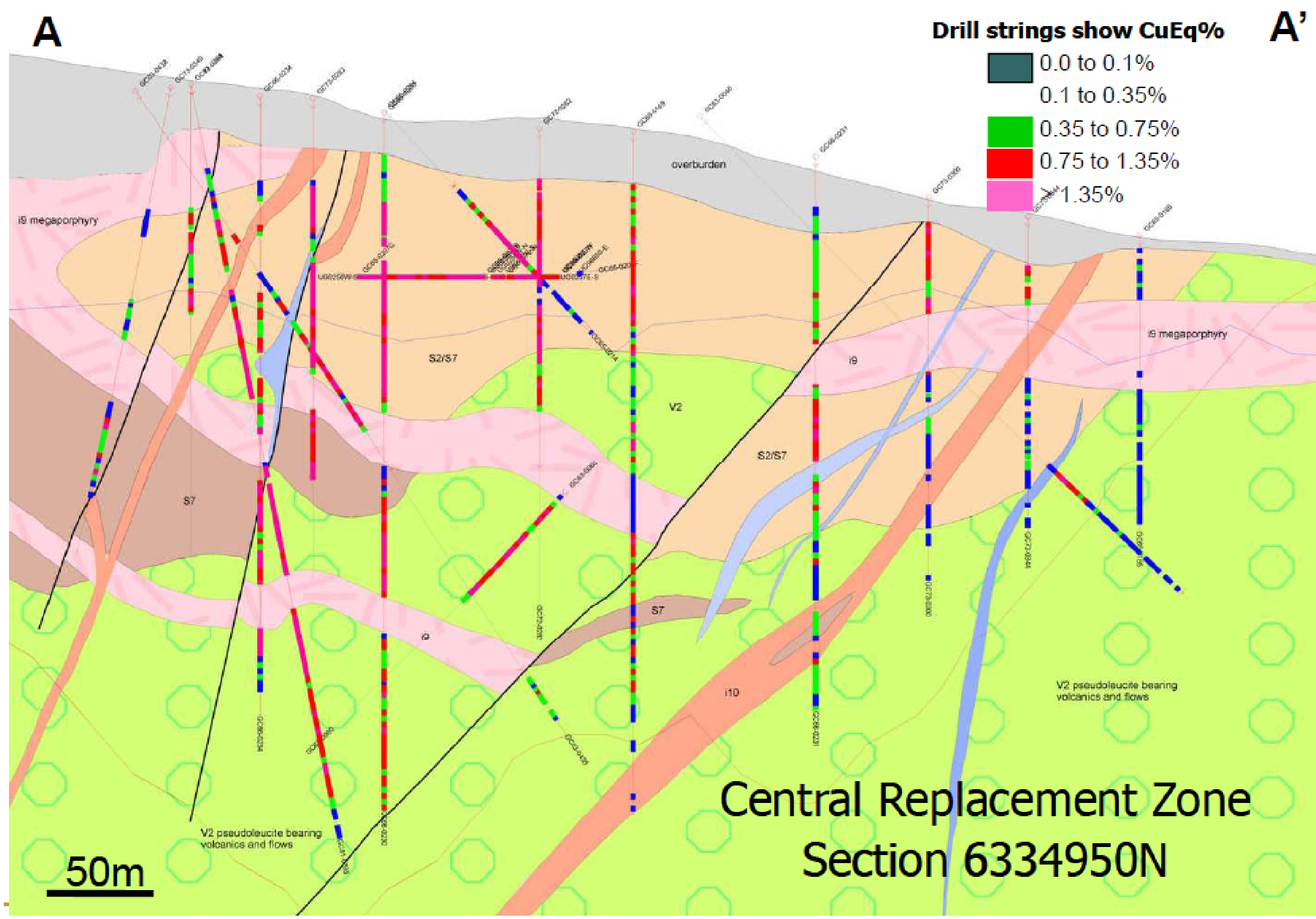
Geological sections through the Central Zone are shown in Figure 7-4 and Figure 7-5.

Figure 7-4: Geological Cross-section, Central Zone



Note: A) North Gold Lens (section 6335547), B) Central Replacement Zone (section 6335100); and C) the South Gold Lens (section 6334100). Figure from Micko (2010).

Figure 7-5: Geological Cross-section, Central Replacement Zone



Note: Figure courtesy GCMC, NovaGold, and Teck

7.9.2 Bountiful Zone

Discovered by deeper than average drilling beneath the eastern margin of the South Gold Lens of the Central Zone in 2003, the Bountiful Zone consists of chalcopyrite and pyrite mineralization hosted by pseudoleucite-bearing volcaniclastic rocks, overlying siltstone and sandstones, and a hydrothermally-cemented polylithic breccia. The mineralization is generally pod shaped with the long dimension in a north–northeast–south–southwest direction.

The deposit is rather large, approaching 500 m x 500 m x 1,000 m with the top of mineralization typically intersected at 350 m to 400 m from surface. Gold assays are typically low. A pod-shaped hydrothermal breccia occurs within the center of best copper mineralization. Clast size in the breccia decreases towards the center of the breccia body.

The relationship between the un-mineralized rock and the breccia has not been established, as unequivocal intrusive contact relations have not been identified. The spatial relationship and the presence of pseudoleucite megacrystic syenite clasts suggest that the latter intrusive may be associated with the breccia as the candidate causative intrusion for Bountiful Zone mineralization.

It is unclear whether the South Gold Lens and the Bountiful mineralized zones are related genetically or whether one is older or younger. Mineralization at Bountiful may represent a slightly later event or a lower temperature expression of the chalcopyrite–pyrite mineralization of the South Gold Lens.

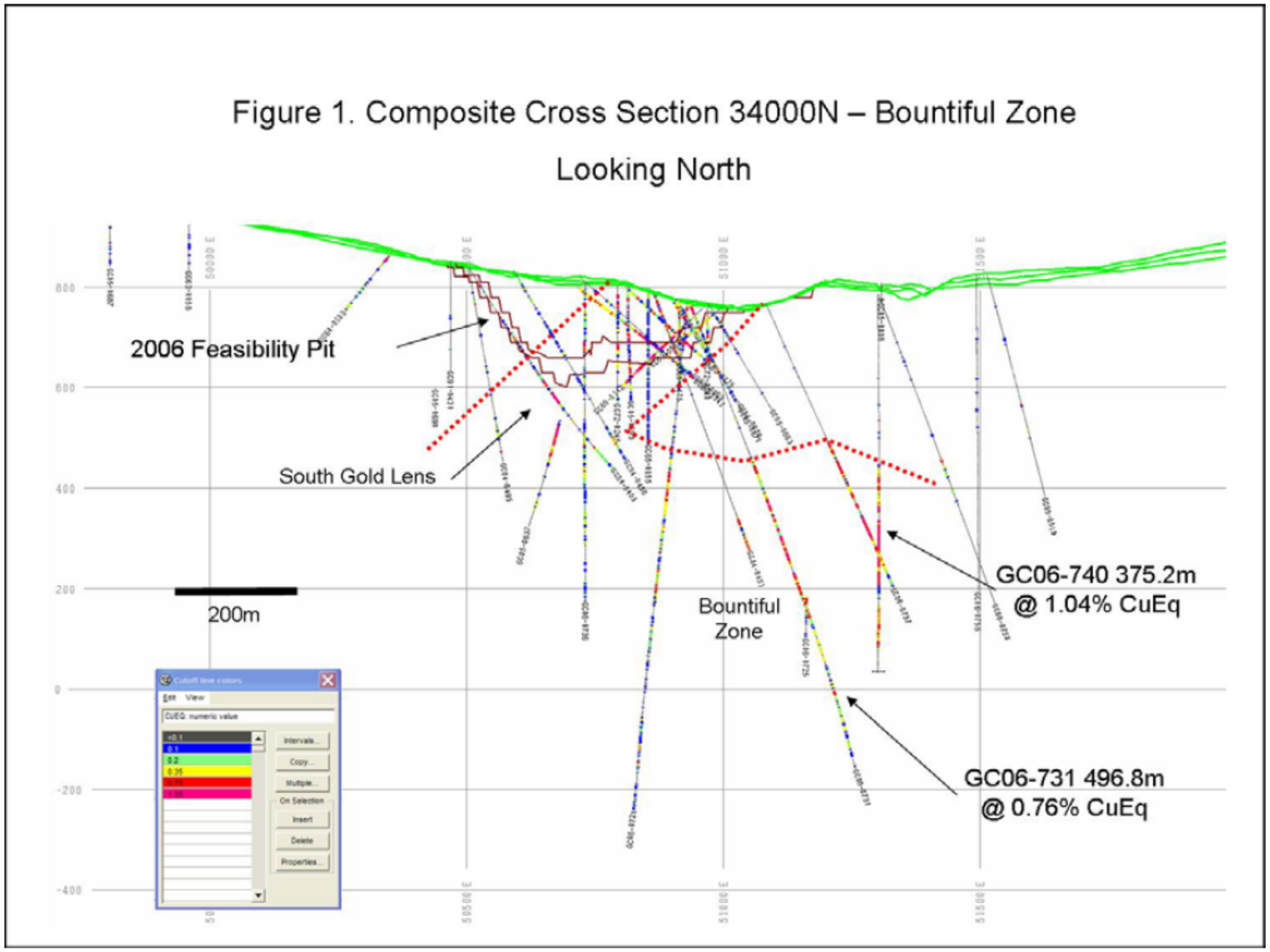
The East fault cuts through the middle of the Bountiful Zone but does not appear to offset it.

Figure 7-6 is a section through the Bountiful Zone.

7.9.3 Southwest Zone

The Southwest Zone is located about 600 m southwest of the south end of the Central Zone and contains some of the highest grade near-surface gold mineralization. Drilling has outlined an elongate pod-shaped body that trends roughly east–west and dips approximately 60° to the south. The zone is up to 400 m long and may be as wide as 140 m. It is still open at depth, and from 2005 drilling, may also be open along strike.

Figure 7-6: Geological Cross-section, Bountiful Zone



Note: pit outline shown on the figure is superceded. Figure courtesy NovaGold

Primary hosts for the Southwest mineralization are a diatreme breccia and an early-phase syenite intrusion. The main zone of mineralization strikes east to southeast and dips steeply south. Sulphides and magnetite occur as disseminations, fracture fillings and replacements. Localization of high-grade copper–gold–silver mineralization within the breccia appears to be related to a combination of structural traps.

Located primarily on the footwall side of the Southwest Fault, the deposit is zoned from a central copper–gold core out to a gold halo. Pyrite occurs in the hanging wall as a disseminated halo adjacent to the chalcopyrite and bornite rich core. Copper mineralization in the Southwest Zone occurs mainly as fine-grained, disseminated and blebby chalcopyrite within the breccia matrix, or as narrow fracture fillings within the orthoclase syenite megaporphyry (*i9*) country rock. Bornite is rare, and unlike the Central Zone, gold mineralization is not necessarily associated with it.

Pyrite occurs in the hanging wall as a disseminated halo adjacent to the chalcopyrite and bornite rich core. It is more abundant in the Southwest Zone (4–6% Cu) than in the Central Zone, and is the cause of a strong IP anomaly in the Southwest area. The majority of pyrite mineralization is hosted in the i9 country rock, and is associated with near surface grades of >0.35 g/tonne Au and <0.35% Cu; suggesting a local gold–pyrite relationship.

Examination of the Southwest drill core suggests a structural control to mineralization; however, the source of the mineralizing fluids is still unknown (Workman, 2006a).

Figure 7-7 is a geological section through the Southwest Zone.

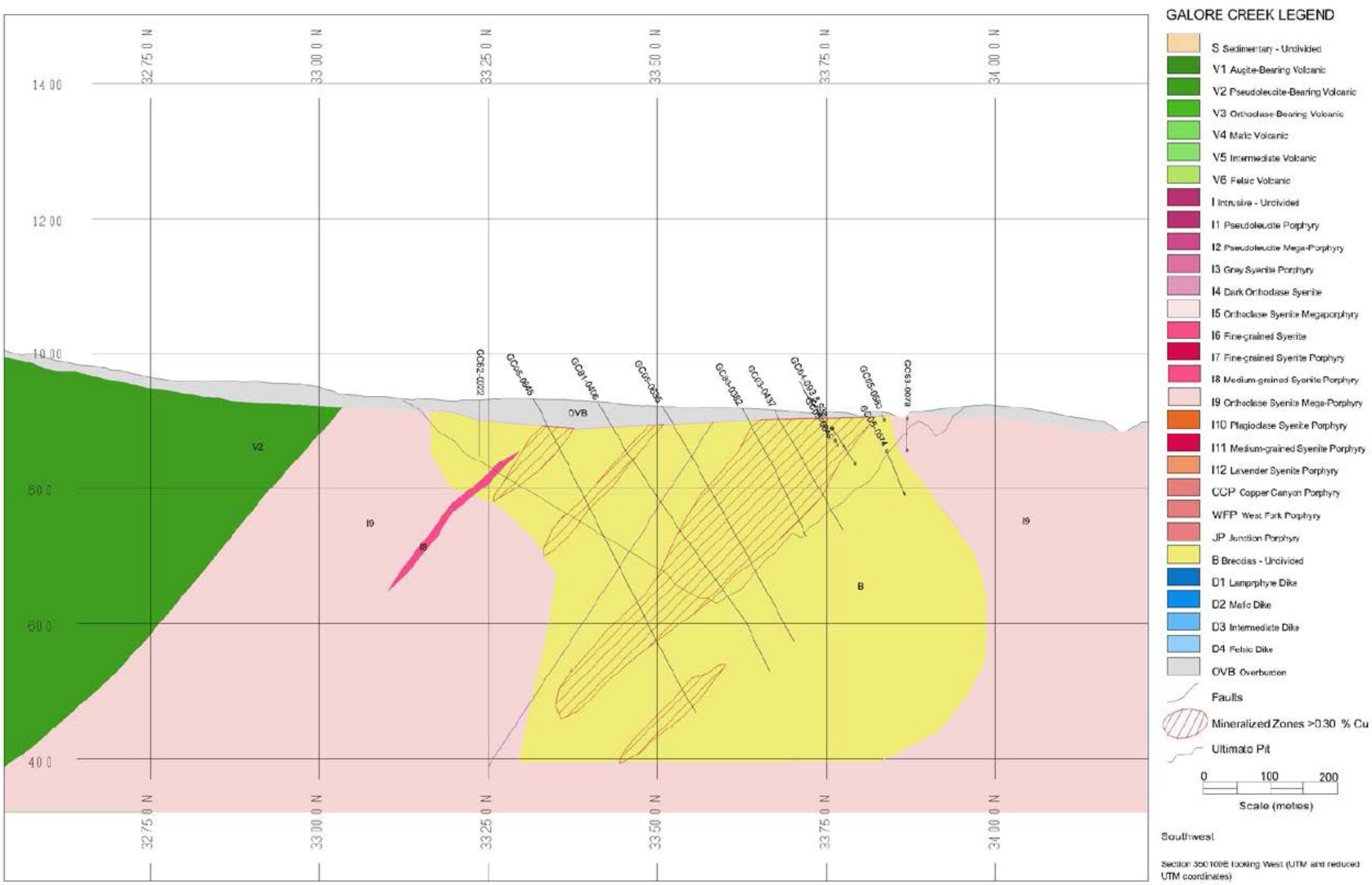
7.9.4 Junction and Junction North Zones

The Junction and North Junction Zones lie about 2 km northwest of the Central Zone and are about 460 m higher in elevation than Central. Both deposits have similar geological characteristics to the Central Zone.

The Junction deposit is a tabular northeast-striking, northwest-dipping body. It can be traced from surface exposures and drill holes 700 m along strike and 200 m down dip. The North Junction deposit, 350 m to the east, is podiform with the long axis plunging to the northwest.

Copper mineralization is well exposed on the slopes of Junction Creek, where the stream cuts through the Junction deposit. Mineralization is partially controlled or bound by Junction Porphyry and syenite dykes that parallel the deposit. On the south end of the deposit, the Junction Porphyry marks the hanging wall of the mineralization.

Figure 7-7: Geological Section, Southwest Zone



Note: Figure courtesy GCMC, NovaGold, and Teck

The footwall is limited by either the Junction Porphyry, syenite porphyry dykes, or assay limits.

The mineralization, consisting of disseminated chalcopyrite and bornite, is hosted in both the Junction porphyry (JP) and the Late Junction porphyry and orthoclase syenite mega-porphyry. K-silicate alteration consisting of pervasive hydrothermal biotite and K-feldspar flooding is associated with the mineralization. A large mass of late-mineral mega-porphyry truncates the zone on the west. A brittle–ductile fault marks the southern limit of North Junction mineralization.

Mineralization at the North Junction Zone is primarily disseminated, and to a lesser degree vein-like, chalcopyrite and bornite mineralization. Mineralization is associated with zones of texturally destructive alteration. Petrography indicates mineralization resides within cores of relict mafic crystals.

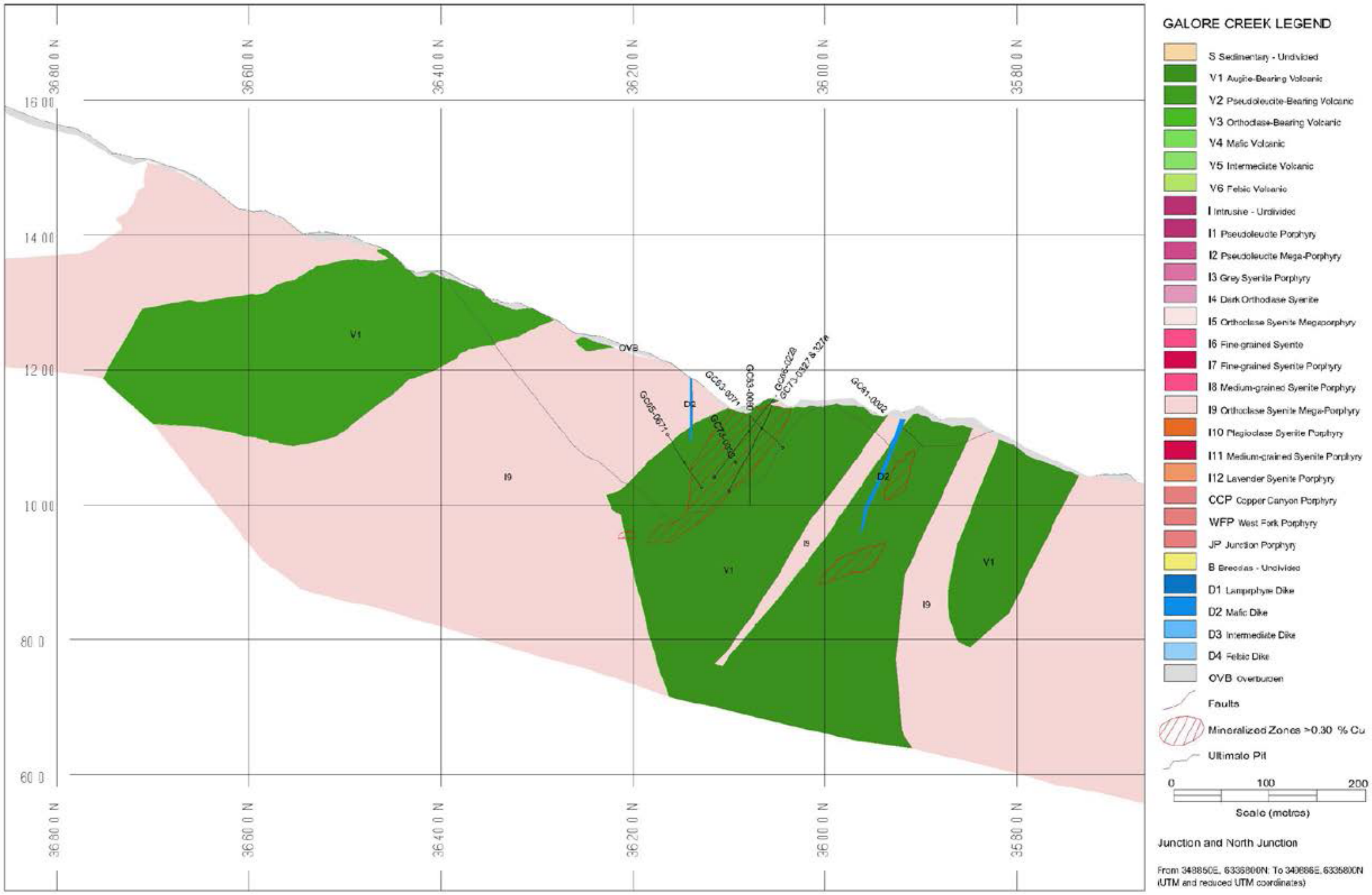
Figure 7-8 shows a cross-section through the Junction and North Junction Zones.

7.9.5 West Fork Zone

The West Fork Zone (also known as the West Fork Glacier Zone) lies in the valley floor less than 1 km south of the Central Zone and less than 50 m higher in elevation than Central. West Fork contains two adjacent but distinctly different styles of mineralization (Figure 7-9): disseminated sulphide replacements similar to portions of the Central Zone, and massive veining (Opulent Vein). Higher grade disseminated zones appear to be controlled by structures, though distinct veining is absent.

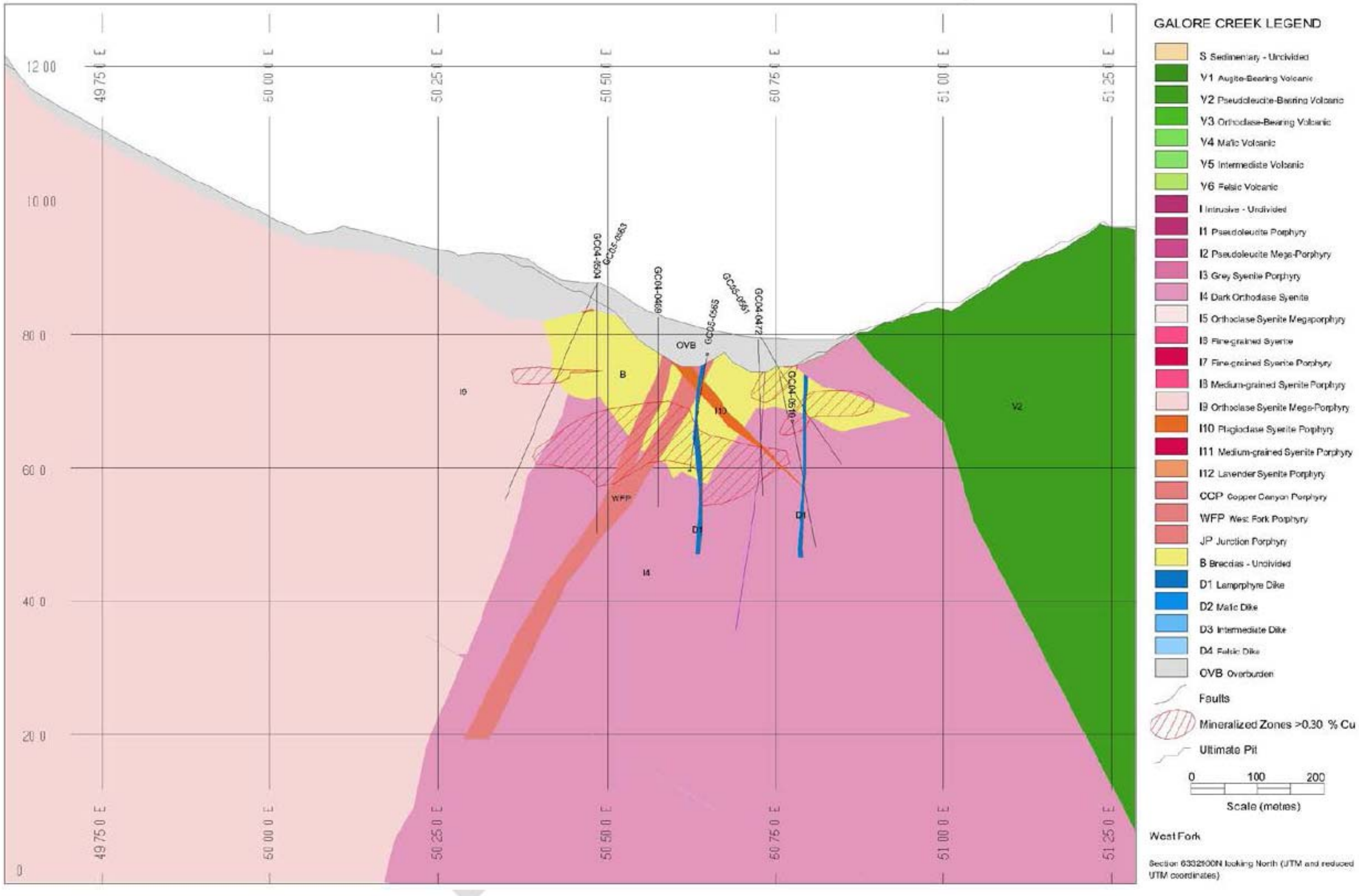
There are four distinct zones of mineralization, the Opulent Vein, the Upper Opulent Zone, the Lower Opulent Zone, and the Lower West Fork Zone. The first three mineralized zones are hosted in a breccia unit. The Opulent Vein is north–south-trending and steeply west-dipping. The high-grade mineralization is defined by a massive magnetite–bornite–chalcopyrite assemblage which is hosted in a near surface magnetite breccia. The Upper and Lower Opulent Zones parallel the Opulent Vein and are characterized by less intense bornite and chalcopyrite mineralization. Sulphide textures indicate fissure-style fillings of open space, but associated calc-silicate gangue minerals, possibly tremolite, indicate replacement. The known extent of the Opulent vein is limited within a breccia mass and strikes approximately 355° with a steep westerly dip. The extent of the zone is 150 m in length and 100 m in depth.

Figure 7-8: Geological Section, Junction and North Junction



Note: Figure courtesy GCMC, NovaGold, and Teck

Figure 7-9: Geological Section, West Fork



Note: Figure courtesy GCMC, NovaGold, and Teck

The Lower West Fork Zone is east–west striking and moderately north-dipping and is characterized by disseminated chalcopyrite and bornite replacement. The Lower West Fork Zone is unusual, because it has not been constrained by any lithological boundaries. Pyrite concentrations average between 0.2–1% on the hanging wall side of the Southwest Fault, and appear to halo the late syenite porphyries and breccia.

7.9.6 Middle Creek

Limited exploration has been completed over this prospect. Middle Creek is located approximately 1 km west of the Central Zone. In 1991, field mapping found mineralization reported by prospectors in the mid 1960s.

Mineralization is characterized by finely-disseminated bornite, chalcopyrite and magnetite associated with pervasive fine-grained biotite and garnet alteration, hosted in a breccia or volcaniclastic unit. Middle Creek is the most oxidized zone discovered to date on the Galore Creek property. Drilling during 2005 encountered malachite and native copper mineralization (Workman, 2006a).

7.9.7 West Rim

The West Rim Zone is about 700 m due west of the Junction Zone. It lies in the west margin of the Galore Creek intrusive complex. Mineralization occurs in pervasive, intense biotite-altered volcanic tuffs. The outer limits of the West Rim zone are based on exposed mineralization along the creek gullies.

The zone is 250 m long, has a northeast orientation and appears to be wider at the north, perhaps due to offset by faulting.

7.9.8 North Rim

The North Rim Zone lies in the northeast corner of the North Junction grid. It is an early-stage exploration target, defined by the presence of widespread scattered malachite in poorly-exposed outcrops, a large, coincident copper and gold soil anomaly, and a large, strong, chargeability anomaly that is broadly coincident with the soil anomaly. One large and two smaller breccia bodies were mapped in the area. Two clast types are present: late megaporphyry (*i9b*) and equigranular syenite (*i8*). Chalcopyrite accompanied by biotite and sometimes bornite are locally present either as veins or more frequently as an accessory in the breccia matrix.

Mineralization at an old cross-shaped trench, within the North Rim Zone, consists of disseminations and veins of chalcopyrite and bornite. The mineralized veins and fractures are locally 1 to 2 cm thick and occur with veins consisting of dark green diopside, biotite, and magnetite.

Chalcopyrite accompanied by biotite and sometimes bornite are locally present either as veins or more frequently as an accessory in the breccia matrix, in two of the mapped breccia bodies in the prospect area.

7.9.9 Butte and South Butte

The Butte deposit crops out on the west edge of the syenite complex, 2 km west of the Central Zone. It is localized along a west-dipping faulted contact between altered volcanic rocks and syenite intrusions.

The South Butte deposit crops out on a nunatak in the West Fork Glacier, 4 km south of the Central Zone. North-trending dykes and mineralized shear zones cut the altered host volcanics, but most of the chalcopyrite and pyrite mineralization is fracture-controlled.

Copper mineralization occurs as disseminated, fine-grained bornite, subordinate chalcopyrite and minor chalcocite in three northeast oriented bands. Mineralization is sometimes difficult to see due to its fine-grained nature in a mottled, dark, pseudoleucite-bearing tuff host. This peculiar, altered, pseudoleucite-bearing tuff is the preferred host to bornite mineralization at Butte.

7.9.10 Saddle

The Saddle deposit crops out on a steep slope, 2.6 km southeast of the Central Zone. It trends easterly and dips northerly at 50° along the contact between buckshot syenite and green syenite porphyry. Primary mineralization is contained within a magnetite-cemented breccia body; secondary copper minerals extend beyond the breccia. The breccia contains angular fragments of buckshot syenite and green syenite porphyry and metavolcanic rocks. Graded beds and flow textures suggest that some of the green syenite represents potassium-metasomatized volcanic extrusive rather than intrusive rocks.

A strong quartz–sericite–pyrite alteration zone extends down the north-facing flank of Saddle Ridge to Jack Wilson Creek. Discontinuous mineralized shears and narrow sigmoidal chalcopyrite–pyrite–quartz veins are exposed at upper elevations of the Saddle Ridge area.

The Spire zone is an east-trending zone of propylitic alteration and disseminated chalcopyrite cropping out at 1,065 m to 1,370 m elevation on Saddle Horn Ridge. In the creek valley, gold values are associated with sericitized, pyritized and silicified zones in andesites.

Although localized chalcopyrite mineralization is recorded in the northern portion of the Saddle ridge, copper mineralization is largely restricted to the southern end of the Saddle Zone. Here, the strongest zones of mineralization are found in the oldest rocks, the epiclastic sediments (*S6*), which are replaced by extensive secondary biotite and a localized texturally-destructive assemblage of secondary orthoclase-magnetite. The main stage of mineralization correlates well with the orthoclase-magnetite ± biotite ± magnetite assemblage.

Near 352779mE, 6331946mN and 352885mE, 6331882mN, outcrops of epiclastic sediments, near the contact with *i8*, contain noticeable copper mineralization. Here, orthoclase–chalcopyrite veins have a marked east–west trend and contain 3–4% vein-hosted and disseminated chalcopyrite. Although the younger *i8* is generally less well mineralized, it contains a local zone of well mineralized magnetite breccia (*B2a*) with the best mineralization (2–5% chalcopyrite) occurring at 352770mE, 6332336mN. The breccia infill or matrix mineralogy consists of massive euhedral magnetite ± chalcopyrite.

7.9.11 South 110 Creek

The South 110 Creek deposit is exposed between 1,220 m and 1,300 m elevation on the west slope of 110 Creek, 500 m north of the Saddle deposit. Disseminated mineralization trends north, along the fractured contact between buckshot syenite, green syenite porphyry and metavolcanic rocks. Magnetite and sulphides are intimately associated as at the Saddle deposit.

7.9.12 Exuberant Zone

The Exuberant zone lies on the northern margin of the Southwest deposit. The prospect comprises a late silica-rich flooded zone with high gold and low copper values.

7.10 Comment on Section 7

In the opinion of the AMEC QPs:

- Knowledge of the deposit settings, lithologies, and structural and alteration controls on mineralization is sufficient to support Mineral Resource and Mineral Reserve estimation
- The mineralization style and setting of the Project deposit is sufficiently well understood to support Mineral Resource and Mineral Reserve estimation
- Prospects and targets are at an earlier stage of exploration, and the lithologies, structural, and alteration controls on mineralization are currently insufficiently understood to support estimation of Mineral Resources.

8.0 DEPOSIT TYPES

Alkalic porphyry copper deposits tend to form in orogenic belts at convergent plate boundaries, commonly oceanic volcanic island arcs overlying oceanic crust. In British Columbia, the deposits are restricted to the Late Triassic/Early Jurassic (215–180 Ma), and have only been identified to date in the Stikinia and Quesnellia Terranes.

Host rocks range from fine- to coarse-grained, equigranular to coarsely porphyritic and occasionally pegmatitic, high-level stocks and dyke complexes of gabbro to syenite composition. The intrusive rocks frequently display multi-phase emplacement, and produce a wide range of breccias. Coeval volcanic rocks are basic to intermediate alkalic varieties of the high-K basalt and shoshonite series.

Deposits can form as stockworks and veinlets, minor disseminations and replacements throughout large areas of hydrothermally-altered rock. They are frequently co-incident either wholly, or partly, with hydrothermal or intrusion breccias. Deposit boundaries are normally based on economic criteria, as large areas of low-grade, laterally-zoned mineralization are common.

Typical alteration minerals include biotite, K-feldspar, sericite, anhydrite/gypsum, magnetite, hematite, actinolite, chlorite, epidote and carbonate. Garnets, if present, are typically Ti-enriched. Mineralization typically forms with early-stage potassic alteration. This central potassic zone commonly contains K-feldspar and generally abundant secondary biotite and anhydrite. Zones with relatively high-temperature calc-silicate minerals such as diopside and garnet can form within the potassic zone. Outwards from the potassic zone can be flanking zones in basic volcanic rocks that contain abundant biotite which grades into extensive, marginal propylitic zones. Older alteration assemblages can be overprinted by phyllic sericite–pyrite alteration and, less commonly, sericite–clay–carbonate–pyrite alteration.

Mineralization consists primarily of chalcopyrite, pyrite and magnetite, with lesser bornite, and chalcocite. Pyrite is less abundant than chalcopyrite in most mineralized zones. Rare mineral assemblages include galena, sphalerite, tellurides, tetrahderite, gold and silver. The deposits are characteristically enriched in silver and gold, and are particularly silver-rich in comparison with calc-alkaline porphyry deposits (Sinclair et al., 1982).

Gangue minerals associated with the sulphides include biotite, K-feldspar and sericite, garnet, clinopyroxene (diopsidic) and anhydrite.

Examples of economically-significant alkalic porphyry copper deposits include the deposits of the Iron Mask batholith in British Columbia, such as the Afton, Ajax, Mount Polley, and Mount Milligan deposits.

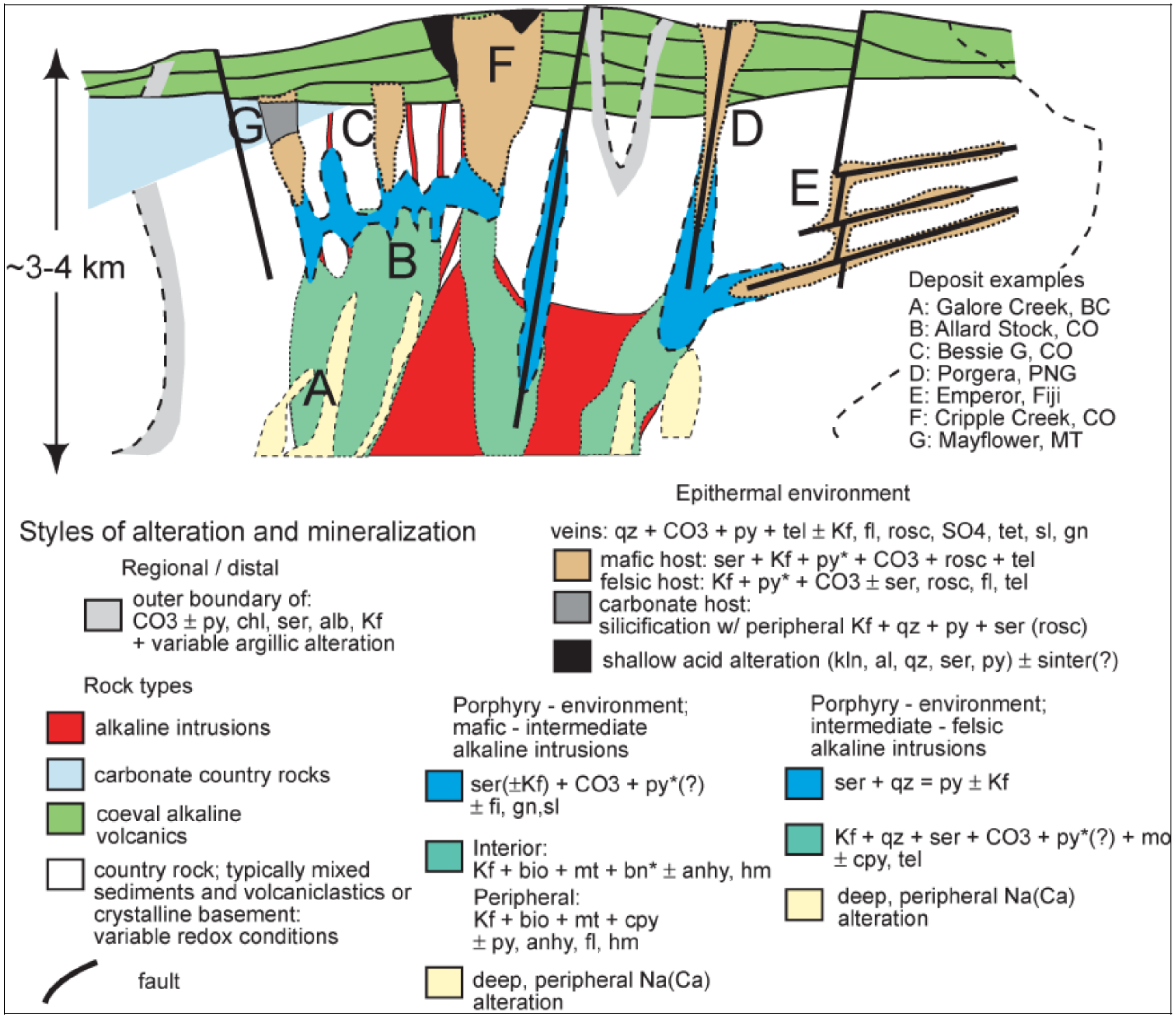
8.1 Comment on Section 8

In the opinion of the AMEC QPs, the deposits of the Project area are considered to be examples of alkalic porphyry copper deposits, based on the following:

- Associated with orthoclase–porphyritic syenite intrusions within the Stikine assemblage; associated with four distinct intrusive pulses of syenite
- Hosted by potassium-enriched volcanic rocks and pipe-like breccias adjacent to the syenite stocks and dykes
- Alteration assemblages are zoned, and include potassic alteration (K-feldspar flooding), Ca–K-silicate assemblages, sericite–anhydrite–carbonate (SAC) alteration, carbonate alteration, and propylitic alteration
- Higher-grade mineralization is associated with the zone of potassic alteration
- Mineralization primarily consists of disseminated pyrite, chalcopyrite, and bornite. Lesser sulphides include sphalerite, galena, molybdenite, native silver, native gold and tetrahedrite
- Replacement lodes of gold, silver and base metals have formed within the propylitic zone.

Figure 8-1 shows the projected genetic setting of the Galore Creek deposit in relation to global examples of calc-alkalic porphyry deposits.

Figure 8-1: Schematic Model of Alkalic Igneous Complexes and Associated Mineralized Systems



Note: Figure from Micko, 2010, and redrafted after Jensen and Barton (2000).

9.0 EXPLORATION

Exploration commenced on the Galore Creek Project in 1955. Table 9-1 shows a summary of the exploration work completed at Galore Creek to May 2011. Exploration has been undertaken by GCMC, NovaGold, predecessor companies such as Kennecott, or by contractors (e.g. geophysical surveys).

9.1 Grids and Surveys

Historical geology and topographic data for Galore Creek have used various survey grids which have been converted to a single grid for consistency.

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 1950s. The survey control for these photos was based on an historical iron pin located 800 m 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 2003, NovaGold contracted Eagle Mapping of Vancouver, BC 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 NovaGold personnel using an Ashtech DGPS system. The aerial photography was taken at a resolution of 2 m using a single-frequency digital global positioning system instrument (DGPS) for control. The resulting DEM surface was different in elevation and accuracy from the historically-generated topography.

On 3 October 2004, a higher resolution, 1 m aerial photo set with dual-channel DGPS was flown for Rescan Environmental services by Eagle Mapping. The control point used for the aerial photography was set by Peter Walcott of Peter E. Walcott and Associates. Walcott 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 m difference from the increased accuracy of the control work that generated it.

In October 2005, a registered professional land surveyor, Peter Thomson BCLS CLS, verified 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 X and Y coordinates; however, a difference of about 1 m was determined in the elevations. No adjustments were made to the digital elevation model or base station control points based on his findings.

Table 9-1: Exploration Summary Table

Work Completed	1960	1961	1962	1963	1964	1965	1966 to 1967	1972	1974	1976	1988	1989	1990	1991	1997	1999	2003	2004	2005	2006	2007	2008	2010
Geologic Mapping (sq km)																							
	196.8	51.8	15.5	5.2	5.2								x	31.1	x			5.2	10.4, x	3.4	4.6		
Geophysical Surveys (line km)																							
Dip Needle	4																						
Airborne Geophysics		270												459				1,552			14.9		
Ground magnetics		55					x						18	85									
Ground VLF-EM												11	11	70									
Induced polarization		43	42	30			x											28	2				
Induced polarization (sq km)																			42				
Remote sensing																x							
Geochemistry (No. of Samples)																							
Stream sediment	47	45										157											
Soil		700		250								729	37	600									
Rock			149				x					210	13	63						4	127		
Reassay of old core											459	219	232	18,000									
Underground Drilling (m)																							
Underground Drilling							163																
Underground Drifting (m)																							
Underground Drifting							850																
Surveys and Boundaries																							
Linecutting (line km)		53	21	32														28	2			2	
Post Location					267		14																
Boundary Surveys					21	47	3																
Airstrip construction																							
Galore Creek (520 m x 30 m)					1	1																	
Scud River (1,500 m x 45 m)					1																		
Project Evaluations									Wright								Hatch	Hatch	Hatch	Hatch	AMEC	AMEC	GCMC and third-party consultants

Note: An "x" in the table denotes that work was performed, but program details such as numbers of samples or areas covered are not available

The primary control for this confirmatory survey was provided by the Geodetic Survey of Canada station 75C134, which is approximately 7 km south of the Bob Quinn airstrip along Highway 37. This station 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 validated by looping the control survey back to Highway 37 by another route, again using static observation techniques. The closure obtained was 0.02 m horizontal and 0.04 m vertical. An additional check was performed by processing eight hours of data on Station 268 with the Precise Point Positioning service of the Geodetic Survey of Canada.

Terra Remote Sensing Inc. completed a light detection and ranging (LIDAR) survey in September 2006 around the then-proposed Galore Creek tailings dam area, More Canyon crossing, and the filter plant area. DEMs and 1 m-contour maps were created for each area based on the results of this survey.

Mineral Resource models use the 2004 topography lowered by 15 m.

9.2 Geochemistry

A summary of the geochemical sampling completed on the Project is included in Table 9-1. Information generated from the sampling programs were used to vector into copper–gold anomalies. However, due to a lack of sufficient soil development to allow for soil sampling to act as a mineralization vectoring tool, sampling was discontinued in the early 1990s in favour of drill data.

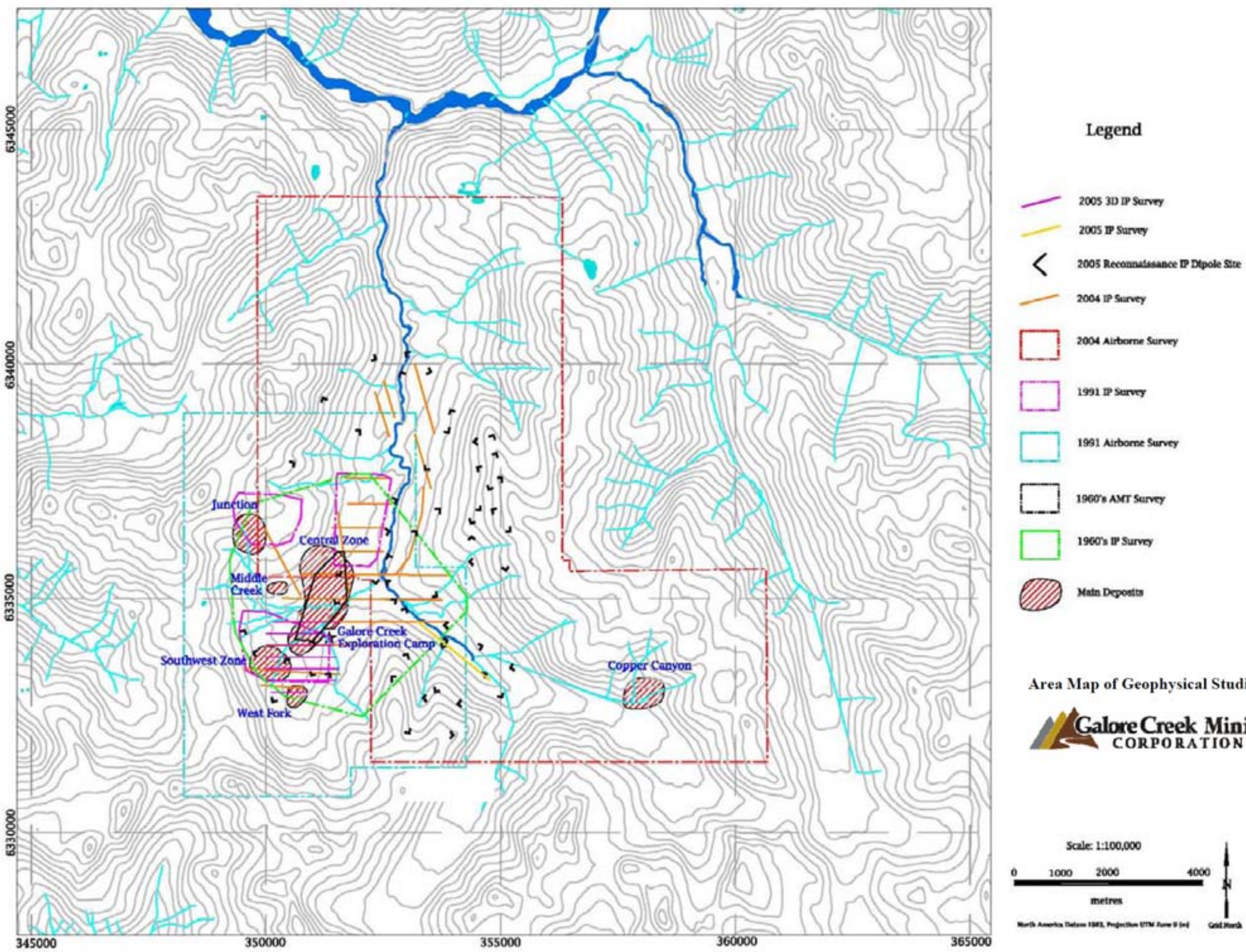
9.3 Geophysics

A number of different geophysical survey methods have been utilized at Galore Creek. Areas covered by the surveys, where known, are indicated on Figure 9-1 and are summarized in Table 9-2. Geophysical surveys were used as vectors for exploration programs, and provided drill targets that were tested using core drilling.

9.4 Underground Sampling

Haste Mine Development drove an adit into the Central Zone during August 1966 to January 1967. A total of 799 m were driven in a 2 x 2 m underground drift. Samples were collected from four cross-cuts to make up a 50-ton bulk sample.

Figure 9-1: Geophysical Survey Locations



Note: Figure courtesy GCMC, NovaGold, and Teck. Copper Canyon is included for reference purposes only, and is not currently part of the Galore Creek Project

Table 9-2: Geophysical Surveys Summary Table

Year	Company	Survey		Comments
1961	H.W. Fleming of Toronto	vertical magnetic field survey	area of 22 km^2 on a line spacing of 800 feet (244 m)	
	Aero Surveys Limited	airborne magnetic survey	area of 64 km^2 using 270 m-spaced lines	
1961 and 1963	McPhar Geophysics Ltd.	pole-dipole resistivity/IP, VLF and AFMAG surveys	area of 20 km^2 with line spacings of 122 and 244 m for the VLF and AFMAG surveys respectively. Dipole lengths used were 30, 60 and 120 m. A total of 71.80 miles (115 km) of IP were run. Lines were generally oriented east–west	
1966	Asarco	Ground magnetic and IP surveys		
1960s	Kennecott		in-house, natural-source, scalar AMT receiver. Electric fields were measured with a 100 t (30 m) dipole	
1990	Gigi Resources Ltd	Airborne magnetic survey; ground geophysical surveys		
1991	Aerodat Limited	airborne geophysical survey including magnetics, EM, radiometrics and VLF	area of 35 km^2 using a line spacing of 100 m east–west and 150 m north–south	
1991	Lloyd Geophysics	combination of pole-dipole, resistivity/IP, VLF and total magnetic surveys	area of 5.5 km^2 on lines spaced 100 m apart. Dipole spacing was 60 m	
1999	Earth Resource Surveys Inc	Remote sensing study		
2004	Fugro Airborne Surveys	magnetic and radiometric surveys	1,072 line km	Three magnetic targets: north of the Central Zone, an area 2–3 km northwest of the Copper Canyon deposit, and magnetic high/low boundaries along the margins of the East Fork of Galore Creek, northwest of the Copper Canyon deposit
	Zonge Engineering	dipole IP/Resistivity	28 linear km on 17 lines using a 100 or 150 m dipole–dipole array	combined with 2D IP/Resistivity modeling and used to extend the depth of mineral exploration
	Frontier Geosciences	seismic refraction surveys	10.5 km on 11 lines using 10 m-spaced geophones	
	Aurora Geophysics	ground magnetic survey	25 m line spacing with 5 m stations	Opulent Vein area
	NovaGold	High-resolution helicopter-borne magnetometer and radiometric survey	480 line km; dipole-dipole IP survey (14 line km)	
2005	Frontier Geoscience	Vector IP	55 wide-spaced IP and resistivity stations in an area of 40 km2	
		pole-dipole IP/resistivity	2 km line of 100-metres spacing	East Fork of Galore Creek. Identified three resistivity lows: a zone 500 m northwest of North Rim, a zone 1 km southeast of West Fork, and the South 110 Creek Zone, which only has a single drill hole located in the western margin of the anomaly
		3D IP	pole transmitter and 100 and 200 m receiving dipoles	1.5 x 1.5 km survey south of the Central Zone

At the North Junction Zone, a smaller, 4 ft x 7 ft (1.2 m x 2.1 m) adit was collared in badly fractured and altered tuff. After driving through 26 m of material grading about 0.5% Cu, a low-grade dyke was encountered. The total length of the adit was 51 m.

Sampling of the adit and drift walls was carried out over continuous horizontal 10 ft (3 m) intervals plus vertical channels alongside the traces of diamond drill holes. Although commonly referred to as “channel” samples, the sampling was more typically chip sampling. The vertical samples taken adjacent to the drill hole traces correlated within 0.1% Cu. When compared to horizontal samples on the opposite side of the drift, significant variation was found in higher-grade areas (>1.5% Cu) where massive blebs of chalcopyrite were encountered. In these areas variations often exceeded 0.4% Cu for opposing walls. Subsequent check sampling along some of the same channels verified this variation.

9.5 Drilling

Drilling completed on the Galore Creek Property is discussed Section 10 of this Report.

9.6 Bulk Density

Bulk density determinations are discussed in Section 11 of this Report.

9.7 Petrology, Mineralogy, and Research Studies

9.7.1 Theses

Four theses have been completed on aspects of the geology and mineralization of the Galore Creek Property:

Allen, D.G., 1966: Mineralogy of Stikine Copper’s Galore Creek Deposits: unpublished M.Sc. thesis, University of British Columbia, 38 p.

Holbek, P.M., 1988: Geology and Mineralization of the Stikine Assemblage, Mess Creek Area, Northwestern British Columbia: unpublished M.Sc. thesis, University of British Columbia, 174 p.

Byrne, K., 2009, The Southwest Zone Breccia-Centered Silica-Undersaturated Alkalic Porphyry Cu- Au Deposit, Galore Creek, B.C: Magmatic-Hydrothermal Evolution and Zonation, and a Hydrothermal Biotite Perspective: Unpublished M.Sc. thesis, The University of British Columbia, 169 p.

Micko, J., 2010. The Geology and Genesis Of The Central Zone Alkalic Copper-Gold Porphyry Deposit, Galore Creek District, Northwestern British Columbia, Canada: Unpublished Ph.D thesis, University of British Columbia, 387 p.

9.7.2 Mineralogy

Petrographical analyses were completed in 2004 on 45 samples by Vancouver Petrographics; these include Galore Creek and Copper Canyon 2004 drill core and historic drill core samples. During 2005, 17 drill core samples were subject to petrographical description by Vancouver Petrographics.

Thesis studies have included electron-microprobe analysis, fluid halogen fugacity estimates, geochronological analysis (SHRIMP-RG), and evaluation of S-isotopic data and whole rock geochemical data.

9.8 Geotechnical and Hydrological Studies

During 1967, a series of geotechnical studies on overburden, bedrock, massive (intact) rock and reservoir slope stability were completed by Golder, Brawner and Associates.

A geotechnical study was carried out in 1991 (Heah, 1991). A total of 381 structural measurements were collected from cliff faces in the northwestern part of the Central Zone. Particular attention was paid to fractures steeper than 30° (the assumed dry friction angle), which dip easterly. In addition, exposed, easterly dipping fractures were also noted. Lastly, the presence of faults, possible release surfaces, and major groundwater seeps were recorded. Rock hardness was also tested.

In 2004, BGC Engineering Inc. (BGC) completed a scoping level study on the location of a proposed tailings facility and access road to the Galore mine site (BGC, 2005). Later the same year, BGC completed site investigations for two proposed tailings impoundment sites: Galore Creek Valley and West More Creek headwaters (BGC 2006a). A total of 17 boreholes were completed during this period – nine boreholes in the Galore Creek Valley and eight boreholes at the headwaters of West More Creek. Geological outcrop mapping and 11 seismic refraction traverses were also undertaken comprising 4.3 kilometres of traverse at West More Creek and 6.1 kilometres within the Galore Creek Valley.

Between June and October 2005, BGC completed site investigations for a proposed tailings facility, waste dumps, open pits and plant site foundations in the Galore Creek Valley (BGC, 2006b). A total of 47 geotechnical boreholes were completed during this period - 37 boreholes to investigate tailings/plant sites and 10 boreholes to investigate the area of proposed open pits. Geological mapping, seven additional seismic refraction traverses (total of 6.7 kilometres of survey line), and 13 test pit excavations near the camp were also conducted.

During 2006, BGC conducted geotechnical and hydrogeological field investigations that included geotechnical drilling, test pit excavations, hydrogeological data collection, surficial (soil and rock) mapping, photogrammetric mapping, a grout injection test under the proposed tailings dam site, and pump testing for a potable groundwater evaluation (BGC, 2007).

A total of 39 geotechnical boreholes were drilled in the vicinity of a proposed tailings dam and other freshwater diversion dams, plant site, filter plant, and various potential impervious borrow areas (BGC, 2007). A total of 110 test pits were also completed, primarily in areas that might be required for borrow material, and water diversion channels (BGC, 2007).

Hydrogeological investigations in 2006 comprised packer testing during drilling, piezometer installation, falling/rising head testing, water level measurements, and pump testing (BGC, 2007). Surficial (soil and rock) mapping and photogrammetry were conducted.

A two day field grout injection test was completed by Eco Grouting Specialists Ltd. in July 2006 (BGC, 2007). The test was conducted in borehole DH-BGC06-20.

During 2008, AMEC investigated the foundation conditions at the proposed tailings dams alignments, evaluated potential borrow material for construction of the proposed dams, and identified and evaluated potential karst features at West More Creek area where underlain by limestone (AMEC, 2008).

Ten vertical and inclined boreholes were drilled with a total of 315 m of drilling at West More Creek area. Hydrogeological testing (packer tests) was conducted to estimate hydraulic conductivity at selected intervals within the boreholes. Three test pits were excavated to evaluate borrow material for the construction of proposed tailing dams (AMEC, 2008).

Geophysical surveys comprising seismic refraction, resistivity and ground penetrating radar (GPR) surveys were performed to establish the bedrock profile, characterise subsurface soil stratigraphy and to evaluate potential karst features within limestone at the West More Creek area (AMEC, 2008).

Surface geologic mapping was undertaken in the West More Creek area. Rock mass fabric data collected during large scale mapping was processed by employing “Dips” software to characterize bedrock structures and to identify main orientations (AMEC, 2008).

Index laboratory tests were performed to classify soil stratigraphy and to estimate geotechnical characteristics of the soils encountered during drilling (AMEC, 2008).

During 2010, six boreholes were completed in the West More Creek area to better define the persistence and depth of potential karst features in the west ridge area. Downhole geophysical surveys were conducted by Frontier Geoscience Inc. in the West More boreholes.

One additional geotechnical borehole was drilled in the Galore Creek Valley a location under consideration for construction of a water-retaining dam. Three hydrogeological boreholes were sited in the area of the proposed Central Open Pit in Galore Creek Valley, for the purposes of installing standpipe piezometers to be used in the monitoring of pump tests undertaken in the summer of 2010.

9.9 Exploration Potential

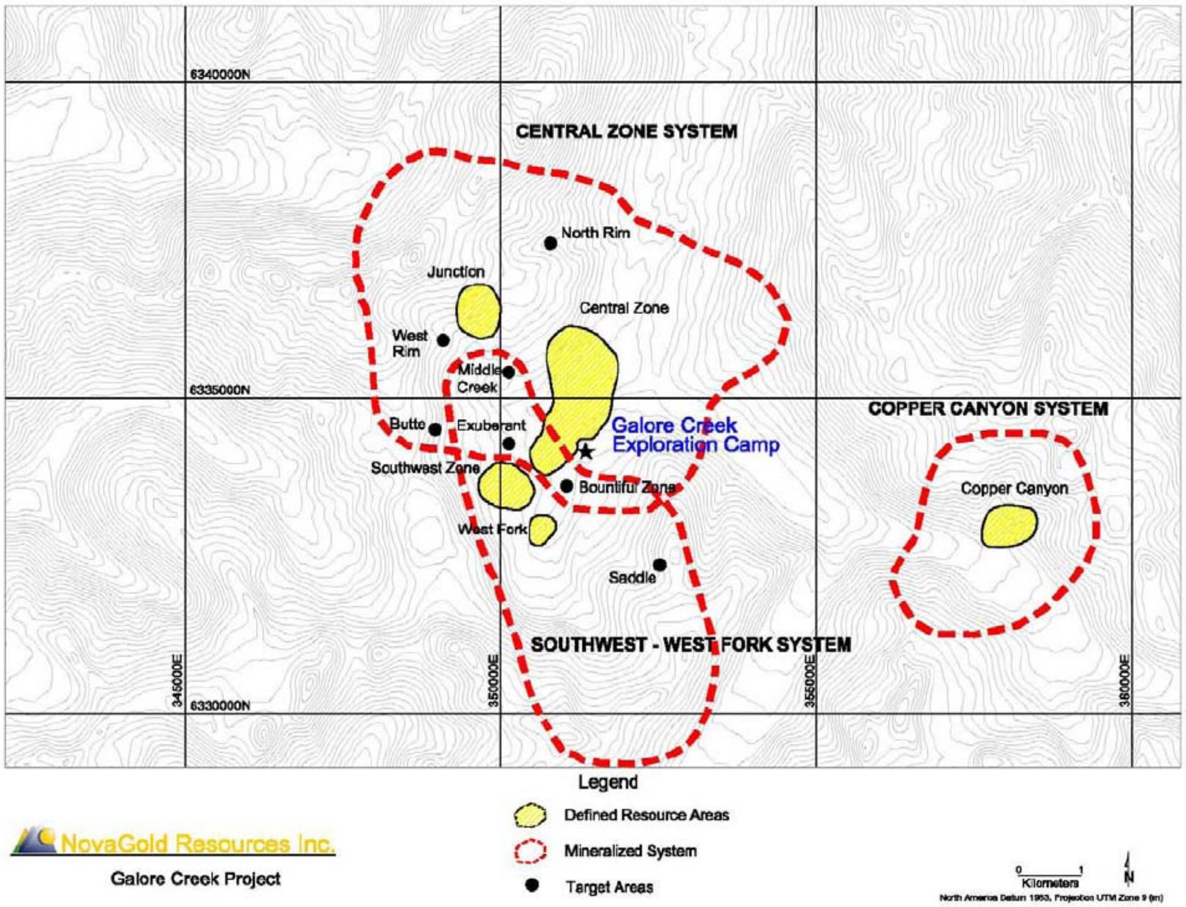
The Galore Creek Project is host to seven under-explored copper-gold prospects, five defined Mineral Resource areas, and numerous showings and conceptual target areas. The discussion on the exploration potential of the Project which follows is based on the latest GCMC interpretations.

GCMC considers that the Galore Creek property has undergone at least three temporally different mineralizing events (Figure 9-2). These include the early formation of the Copper Canyon eruptive centre and its associated mineralization; deposition of the Central Zone mineralization at the Central and Junction deposits and Butte prospect; and emplacement of the West Fork mineralization at the Southwest and West Fork deposits.

Micko (2010) concluded that erosional levels of the Galore mineralizing system are minimal and have only exposed high level magmatism characterized by widespread sill and dyke emplacement in coeval volcanic and sedimentary rocks.

GCMC consider that alteration and mineralization vectors and the underlying thermo-chemical gradients controlling those vectors are typical of porphyry-style deposits.

Figure 9-2: Mineralizing Systems of the Galore Creek Area



Note: Figure courtesy NovaGold: Copper Canyon is not part of the Project area, but is shown on this figure to indicate the different mineralizing systems in the vicinity of Galore Creek.

Proximal mineralization in the Galore porphyry systems is dominated by intense potassic or K-silicate alteration with higher-grade gold and copper (as bornite) related to strong K-spar, biotite and magnetite alteration.

Marginal to the potassic zones are typical calcic and sodic alteration haloes characterized by the K–Ca–silicate zones with elevated garnet and albite. Outboard the systems zone into typical marginal propylitic alteration assemblages.

GCMC is of the opinion that these vectors along with the lack of progenitor intrusions driving the systems at both the Central and West Fork systems has major exploration significance. GCMC considers that the potential to make a major discovery at depth or even laterally as with the case of the West Fork deposit is high. GCMC notes that some of the most significant discoveries of the last two decades such as Grasberg, Oyu Tolgoi and Pebble all discovered higher-grade progenitor porphyries at depth late in each deposit's exploration history.

Potential exists to increase the known extent of the mineralizing systems within the Central Zone area at the North Gold Lens and within the Bountiful deposit. The Bountiful deposit remains open, although there are some indications that the mineralization is becoming lower-grade at depth. There is also potential to identify strike extensions at Butte and on the Southwestern–Junction Trend, and in the vicinity of the West Fork–South West–Middle Creek system.

A conceptual exploration target identified by GCMC is to undertake alteration modeling, which could be used to vector in on higher-temperature gold and bornite mineralization that may be developed beneath the *I9* high-wall sill. The Lower *V1/V2* contact is as yet undrilled to the south under the Central Replacement Zone, and GCMC considers that it also represents a conceptual exploration target.

Devonian carbonates in the basement are exposed in over-thrusts on the east and west margins of the district, where they lie beneath the V1 stratigraphy. In GCMC's opinion, there is potential that deep AMT modeling might identify a conceptual, deep, carbonate-hosted (skarn) target.

9.10 Comment on Section 9

In the opinion of the AMEC QPs:

- The exploration programs completed to date are appropriate to the style of the deposits and prospects within the Project

- The exploration and research work supports the interpretations of the of the deposits
- The Project retains significant exploration potential, and additional work is planned.

10.0 DRILLING

Approximately 255,601 m has been drilled in 1,078 core holes on the Project since 1961. Details of the various drilling programs are summarized in Table 10-1, and drill hole locations are shown in Figure 10-1. In this table, drill holes completed for metallurgical purposes are classed as exploration drill holes. Table 10-2 shows the drill holes which are used to support the Mineral Resource and Mineral Reserve estimates.

Figure 10-2 shows geotechnical, hydrogeological, and condemnation drill holes. Figure 10-3 shows the location of the metallurgical drill holes. Drill holes completed in the area of the Mineral Resource estimate since the last technical report filed on the Project in 2008 are shown in Figure 10-4.

The drilling between 1961 and 1976 was for early-stage, exploration-focused programs and for initial resource estimates. From 1990, drilling was designed primarily to support Mineral Resource estimation, and define deposit limits. In 2006, a minor amount of prospect and exploration drilling occurred. Drilling at the Grace Claims has either been for exploration or condemnation purposes; to date, no mineralization of significance has been outlined in drilling on the claims.

10.1 Drill Methods

Over the Project history, a number of drill companies have been used. Where these are known, they are summarized in Table 10-2.

Drill holes completed from 1963 to 1965 during the Stikine drill campaigns included AQ (27 mm), BQ (36.5 mm), and BTW (42 mm) core sizes. From 1966, holes were typically BQ or NQ (47.6 mm) size. From the 1970s to the 1990s, HW (77.8 mm), NQ and BQ core was drilled.

Drill holes completed from 1961 to 1965 by Kennecott included AQ, BQ, and BTW core sizes. From 1966, holes were typically BQ or NQ size. From the 1970s to the 1990s, HW, NQ and BQ core was drilled.

Core drilling has been performed at BQ, NQ, HQ (63.5 mm) or PQ size (85 mm) during the SpectrumGold, NovaGold and GCMC campaigns.

Table 10-1: Summary, All Drill Programs

Program by Company and Year	Number of Holes	Drilled Metres
Kennco 1961	5	363
Kennco 1962	40	4,697
Kennco 1963	49	11,261
Stikine 1963	2	470
Kennco 1964	54	11,117
Stikine 1964	1	245
Kennco 1965	8	1,525
Stikine 1965	80	17,174
Stikine 1966	30	7,482
Stikine 1972	50	10,416
Stikine 1973	61	14,689
Silver Standard Mines 1974	4	430
Stikine 1976	25	5,317
Stikine 1990	20	1,925
Trophy Gold 1990	4	829
Kennecott 1991	49	13,820
SpectrumGold 2003	10	2,950
NovaGold 2004		
exploration	70	22,311
geotechnical	17	488
well monitoring	4	50
NovaGold 2005		
exploration	211	60,590
geotechnical	37	1,628
well monitoring	10	242
NovaGold 2006		
exploration	59	34,322
geotechnical	58	2,856
condemnation	2	495
NovaGold 2007		
exploration	36	12,517
geotechnical	25	2,258
Barrick 2007	13	5,207
GCMC 2008		
exploration	9	2,050
geotechnical	14	1,345
GCMC 2010		
exploration	9	2,803
geotechnical	12	1,729
Total	*1,078*	*255,601*

Figure 10-1: Drill Hole Location Plan, Galore Creek Deposits

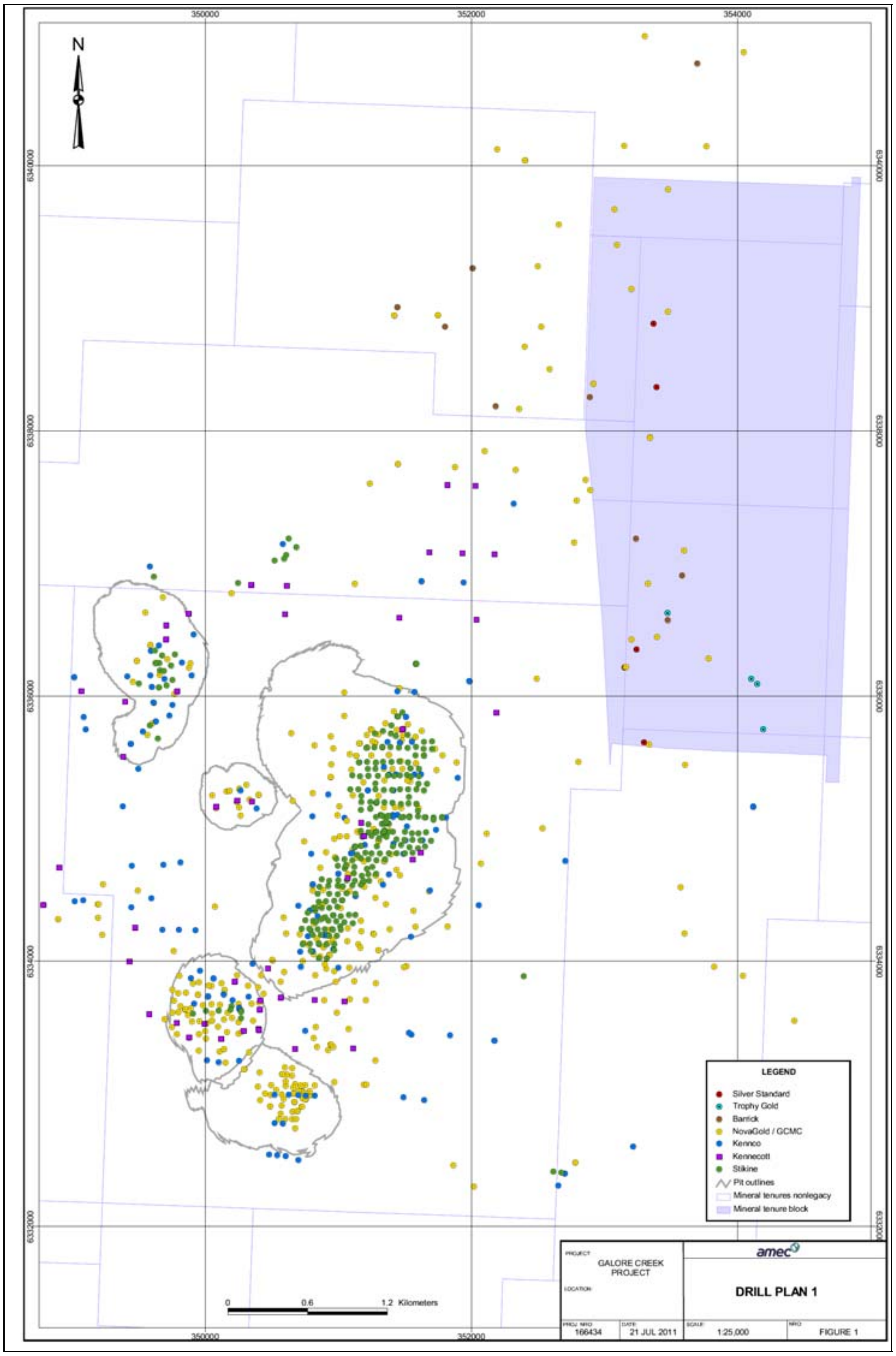


Figure 10-2: Drill Hole Location Plan, Geotechnical, Hydrological and Condemnation Drilling

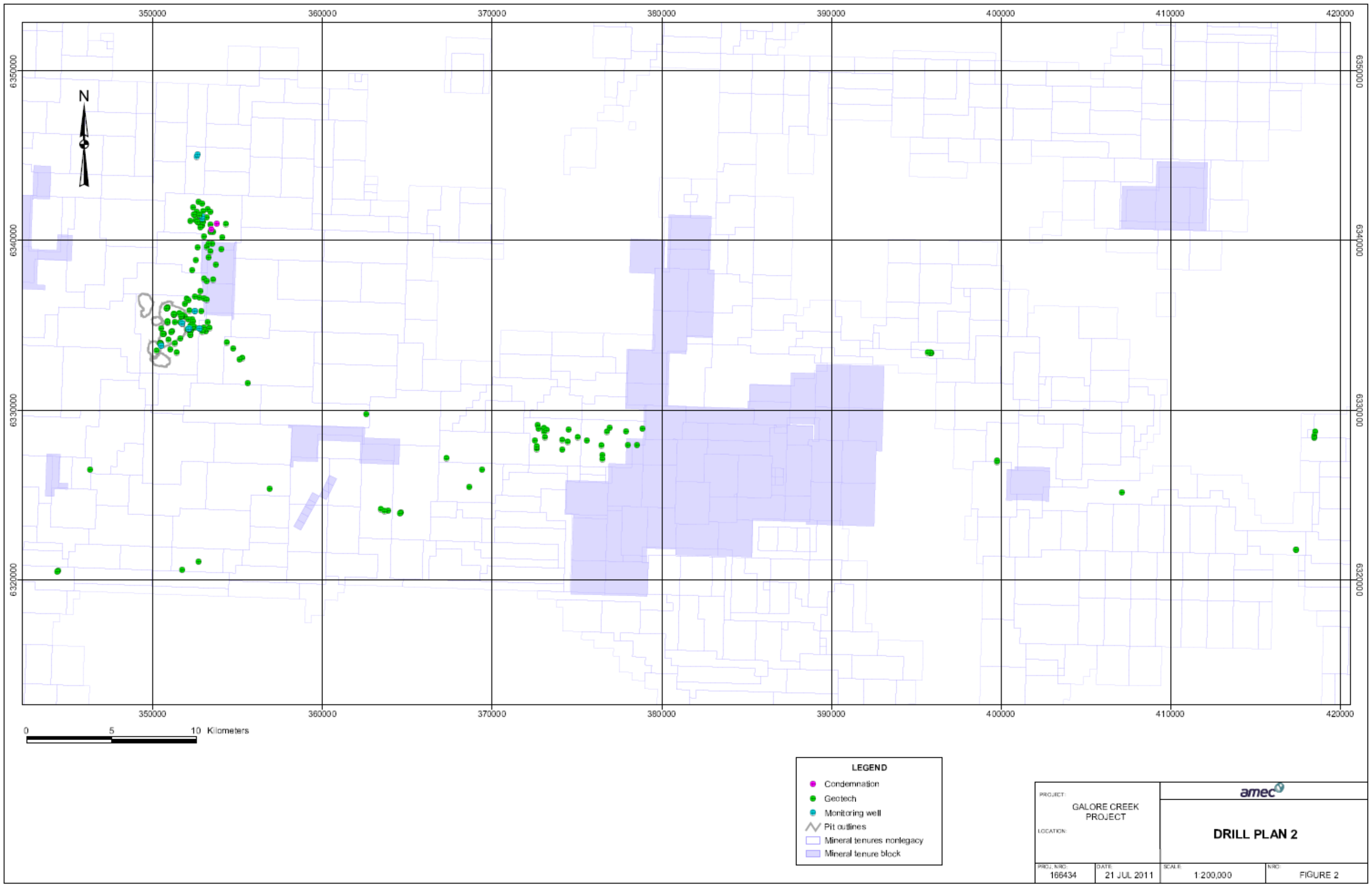


Figure 10-3: Drill Hole Location Plan, Metallurgical Drilling

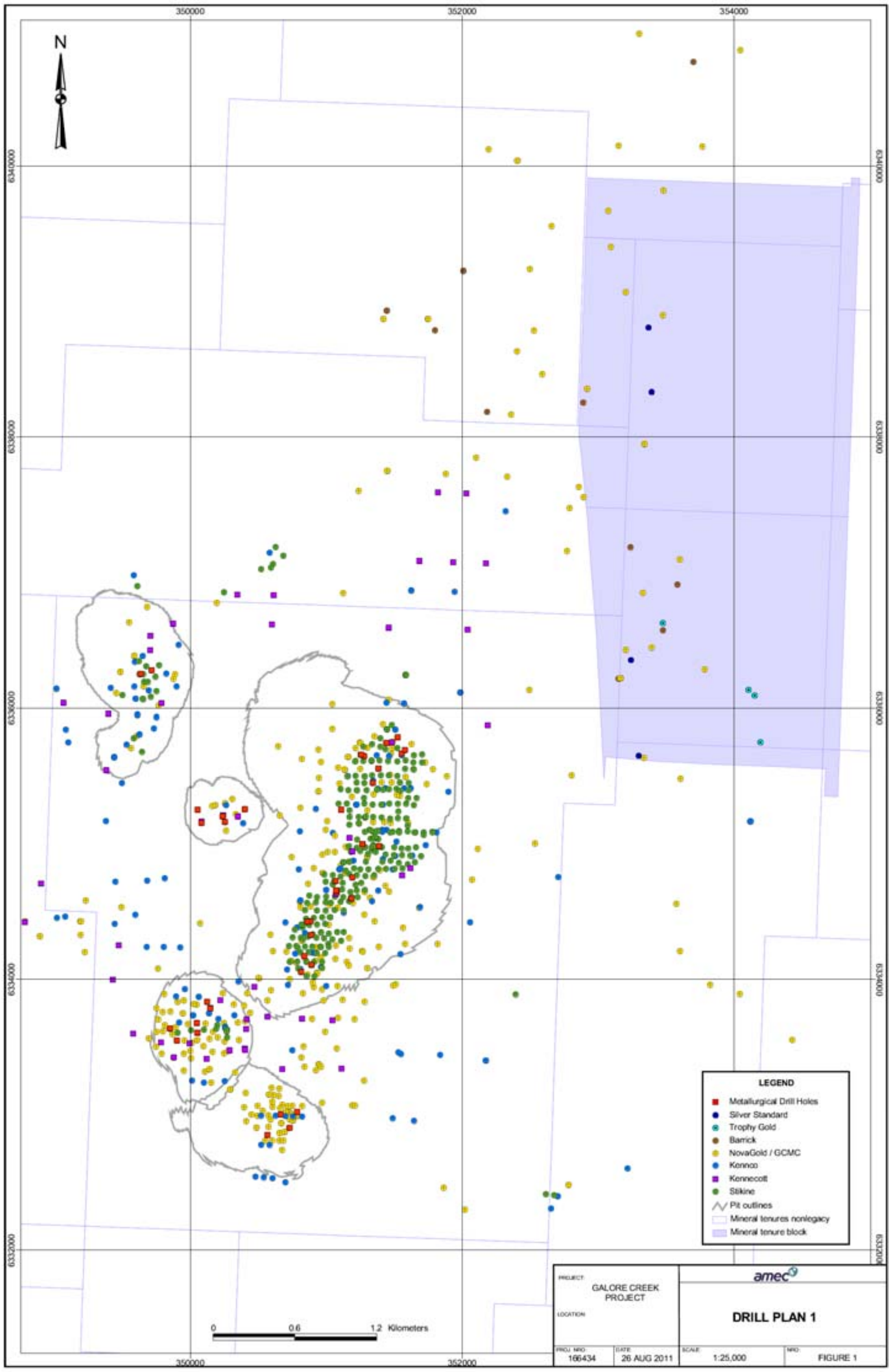
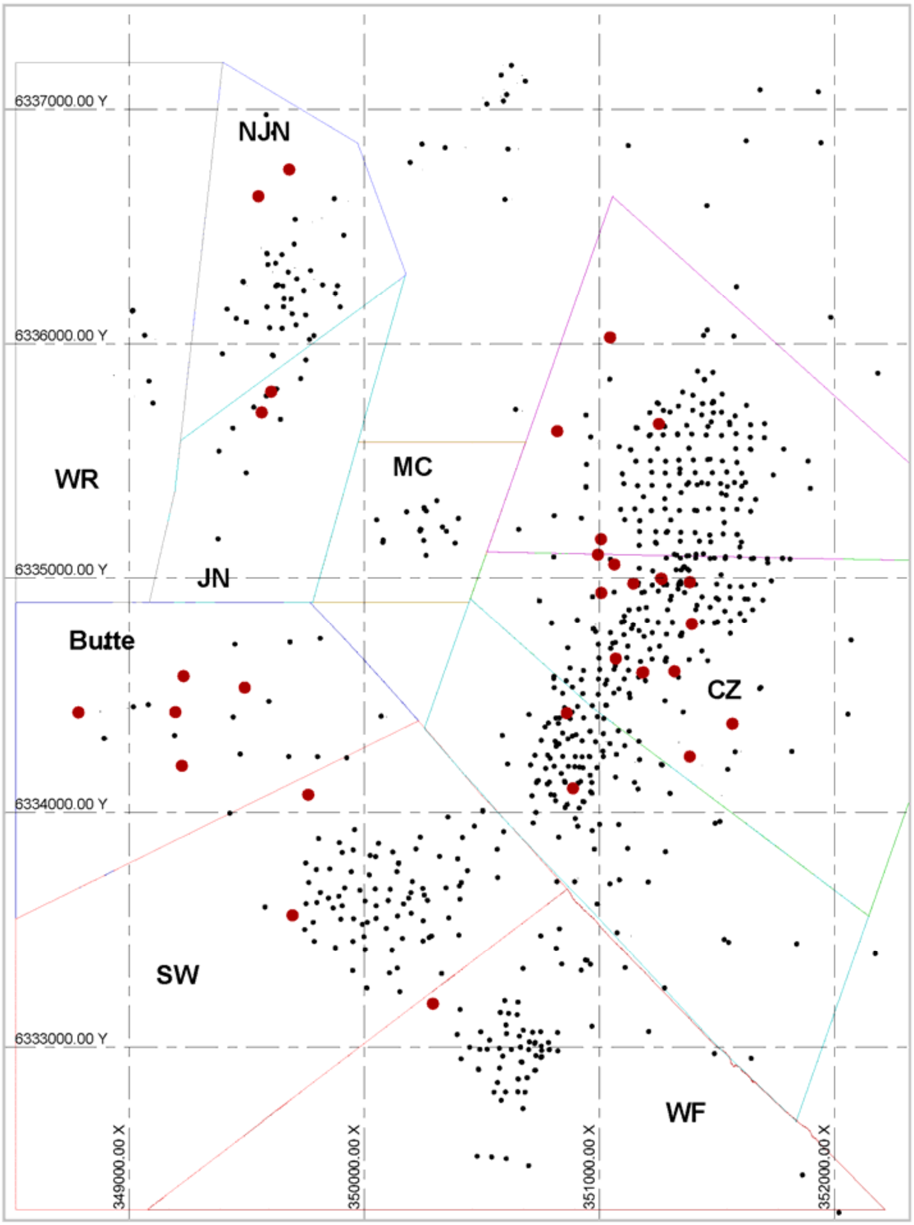


Figure 10-4: Drill Plan Showing New Holes Drilled Inside the Area of the Mineral Resource Estimate Since the 2008 Mineral Resource Estimate Update



Note: new drill holes shown as red circles. WR= West Rim, NJN = North Junction, JN = Junction, MC = Middle Creek, CZ = Central Zone, SW = South West, WF = West Fork. Map north is to top of plan.

Table 10-2: Drill Companies Used 2003–2010

Drilling Company	Year	Drill Rigs
T. Connors Diamond Drilling Company	1961	Rig type unspecified
Midwest Diamond Drilling Company	1962–1965	BBS 14 and BBS 20 diesel drill rigs
unspecified	Latter part of 1962–1965	Rotary overburden drill to facilitate drilling in areas of deep overburden
Boyles Brothers	1963	Boyles wire-line drill
Quest Canada Drilling	1991	Rig type unspecified
Britton Bros. Diamond Drilling Ltd	2003	Longyear 38; Britton Bros. 2500
Britton Bros. Diamond Drilling Ltd	2004	Skid-mounted and helicopter-portable drill rigs
Cyr Drilling International Ltd	2005	Skid-mounted Longyear 38s
Hy-Tech Drilling Ltd	2005	Custom-built S-5, S-10 and B-15, helicopter-supported fly rigs
Cyr Drilling International Ltd	2006	Boyles Brothers model 56
Hy-Tech Drilling Ltd	2006	Custom-built S-5 helicopter-supported fly rigs
Hy-Tech Drilling Ltd	2007	Custom-built S-5 helicopter-supported fly rigs
Foundex	2008	Custom-built fly rig
Black Hawk Drilling Ltd	2008	Custom-built fly rig
Black Hawk Drilling Ltd	2010	Custom-built fly rig

10.2 Legacy Drill Data

Yarrow and Enns (1992) note that all drill data collected during the Kennecott programs at Galore Creek were logged on paper drill logs. No records are available as to the methods of data collected prior to 1991 for any other operator, or for operators other than SpectrumGold, NovaGold, and GCMC prior to 2003.

Marr (1992) noted that information on legacy drilling during the Kennecott programs was entered into computer databases during four separate periods:

- Drill holes GC1 to GC235 were entered by Kennecott's Computing Centre in 1966
- Drill holes GC236 to GC369 were entered by Rocky Mountain Data Control in Salt Lake City on behalf of Kennecott during 1990
- Data for drill holes GC370 to GC387 were obtained on-line from MinEn Laboratories, and data-entered into the database by Kennecott personnel during 1990
- Data for drill holes GC388 to GC435 were entered by Galore Creek project staff during 1991.

Kennecott noted that for drill holes GC001 to GC369, drill hole identifiers, from, to, copper, silver, and composite gold data were directly entered from the original drill logs, and subject to data entry validation. Sample numbers were apparently omitted from the original data entry, and subsequently added.

All of these data were merged into a single database, audited, and converted from Imperial units to metric units during 1991–1992 (Marr, 1992).

During the 1991 Kennecott drill program, logging data collected included geological information such as lithologies, alteration, mineralization, preparation of a graphic log at 1:200 scale, and geotechnical data such as percent core recovery, rock quality designation (RQD), hardness and degree of breakage (Yarrow and Enns, 1992). AMEC has assumed that standard logging practices were employed during the original Stikine and Kennecott drill programs.

Much of the drill core that still existed in 1991 was relogged by Kennecott staff based at Galore Creek. Lithological, alteration and fault-block data were entered into separate databases, which were subsequently provided to Kennecott and all data converted to numeric codes for modelling purposes (Yarrow and Enns, 1992).

During the Kennecott drill programs, drill holes averaged depths of about 209 m; the deepest hole was 598 m, the shallowest 11 m. Azimuths were variable, although again a southerly direction predominated. Dips ranged from vertical to -36º.

Very limited information is available on original downhole and collar survey data collected prior to 1991 for any operator other than Kennecott, or for operators other than SpectrumGold, NovaGold, and GCMC prior to 2003. Drill collars for the 1966 Stikine program were surveyed by personnel from Underhill and Underhill, who were professional surveyors. During the 1991 Kennecott drill program, all but eight of the drill collars were located using a TDM Total Station 20 instrument. For the remaining legacy data, the original collar survey method is unknown.

A Sperry Sun single-shot down-hole survey unit was used for all holes drilled in the Central and Southwest Zones, and acid-etch dip tests were completed on the reconnaissance drill holes (Yarrow and Enns, 1992). Approximately 150 legacy drill holes were not surveyed down-hole. Most of these holes are vertical in orientation. Database records for inclined holes drilled prior to 1991 show no change in azimuth. AMEC has assumed that these drill holes were surveyed for dip only using a method such as the acid etched tube test. For the remaining legacy data, the original down-hole survey method is unknown.

AMEC has no information as to core photography, or QA/QC for the legacy drill programs.

10.3 Geological Logging

Workman (2005) codified the logging procedures used in all SpectrumGold, NovaGold, and GCMC drilling. A reference core library is maintained of all lithology types encountered in drilling. Geotechnical data are recorded according to procedures documented in a site-specific geotechnical manual (BGC, 2005).

The core boxes are initially checked for driller errors, run-block positions are recorded, and blocks are converted from feet to metres, if needed. Box “from–to” lengths are determined and boxes are labelled.

A geotechnician records geotechnical information such as recovery, rock quality designation (RQD), number of fractures, joint condition, and joint alteration. These data are written up on a specifically-designed geotechnical log sheet.

The core-logging geologist records geological information about the core, commencing with looking over the core for mineralization, lithic breaks, alteration boundaries, and major structures. Sample intervals are determined and alteration, mineralization, rock units, and structures are described and recorded. The complete log includes a graphic log, descriptive section, and coded alteration and mineralization information.

Core is moved into the core photography tent where specific gravity and rock strength is tested. Wet core is then digitally photographed, three boxes at a time, and subsequently moved to the saw shack lay-down area. Core photographs are uploaded to a computer, and filed under folders that are named with the appropriate drill hole identifier.

The completed logs are given to the data entry clerks who enter the information into an Access database using the in-house front-end data entry program DDH-Tool. Once the data are in the database, each geologist signs a “data verification” form to validate that data collection/entry for the appropriate hole is complete and checked.

Working cross-sections are maintained for each drill hole, where the drill hole trace, lithological contacts, major structures, and mineralized zones are plotted on the appropriate cross section at the completion of the drill hole log. As completed, each drill hole is correlated on these sections with adjacent drill holes.

The original geological log, geotechnical log, and downhole survey for each drill hole are filed in a designated filing cabinet in the geology office onsite at Galore Creek. Digital back-up copies of the geological logs are maintained.

10.4 Recovery

Core recovery has been evaluated by campaign and generally improves throughout the exploration history of the property.

Recovery is typically poor in the near surface environment where gypsum and anhydrite veinlets have been dissolved and the rock is broken (“broken rock”). Below this interface, where the core is more competent ("stick rock"), better recoveries are returned.

A weak relationship between copper assays and recovery has been identified by GCMC but has not been shown to date to be material and all assays, regardless of recovery percentage, were accepted for Mineral Resource estimation.

10.5 Broken Rock/Stick Rock Boundary

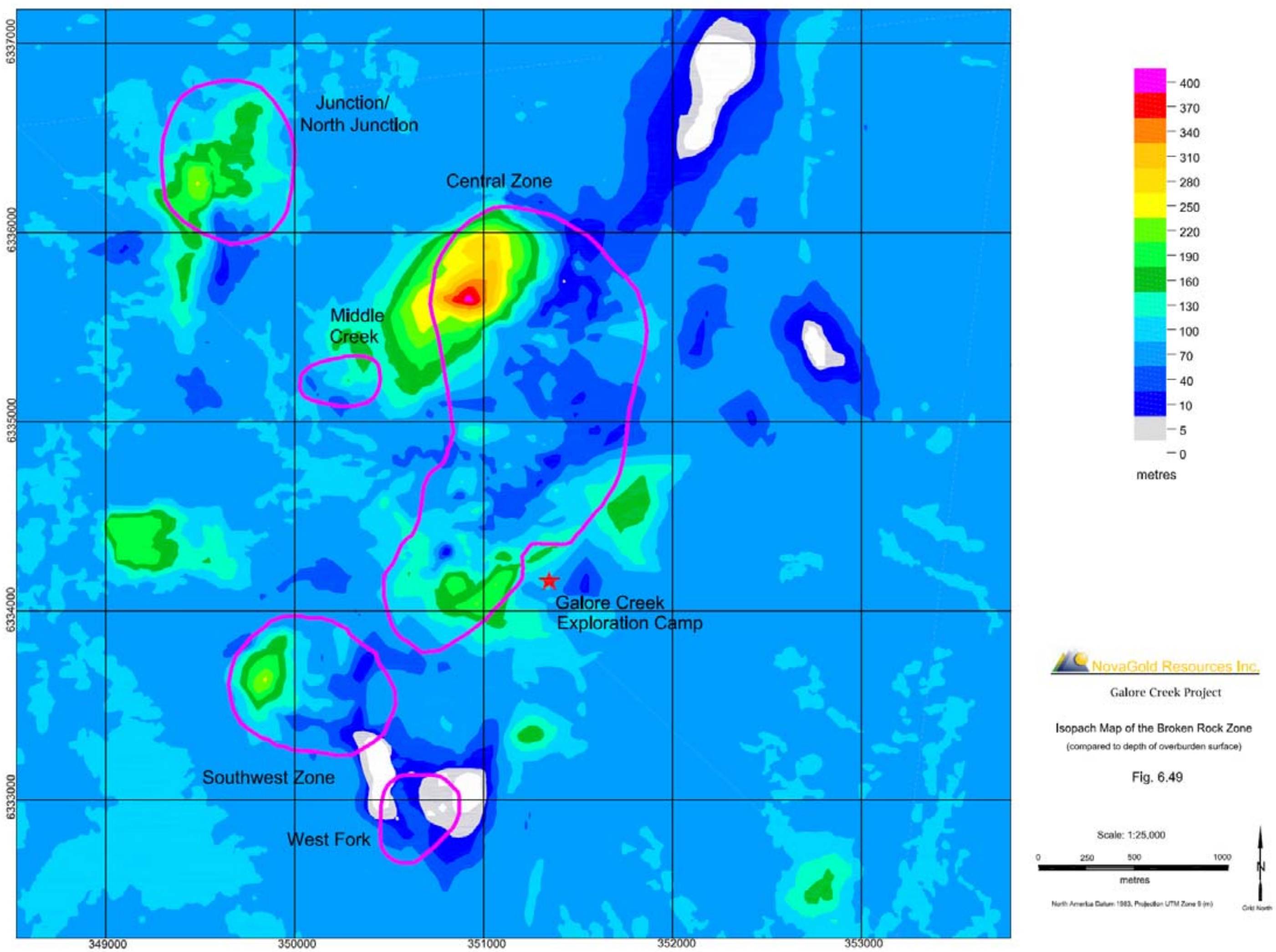
The transition from broken rock to stick rock is commonly abrupt and can usually be identified accurately from a change in RQD from less than 25% RQD (broken rock) to greater than 75% RQD (stick rock). The broken rock/stick rock boundary surface roughly parallels the topography and occurs at depths commonly in the range of 80 m to 150 m, with rare depths as deep as 410 m (Workman, 2006a).

Figure 10-5 is an isopach map of the deposit areas that shows the depth of the broken rock/stick rock boundary in the areas of the Mineral Resource and Mineral Reserve estimates. Figure 10-6 is a long-section through the area between the West Fork and Central Replacement Zone deposits, and provides an example of the variations of depths in the boundary zone.

Workman (2006a) reports that much of the broken rock zone can be attributed to persistent, closely-spaced fracture cleavage, ubiquitous within near surface bedrock of the Galore Creek deposits. This sub-horizontal fracturing is characterized by millimetre-scale spacing and is commonly filled with gypsum; however, the gypsum is commonly leached by meteoric waters. Fracture cleavage is pervasive through volcanics, early intrusives, and brecciated lithologies but is not well developed in late-mineral to postmineral dikes (*D1–D4*) and intrusions (*i9* and *i9b*). It is currently interpreted as a late stage feature localized in the Galore Creek Valley.

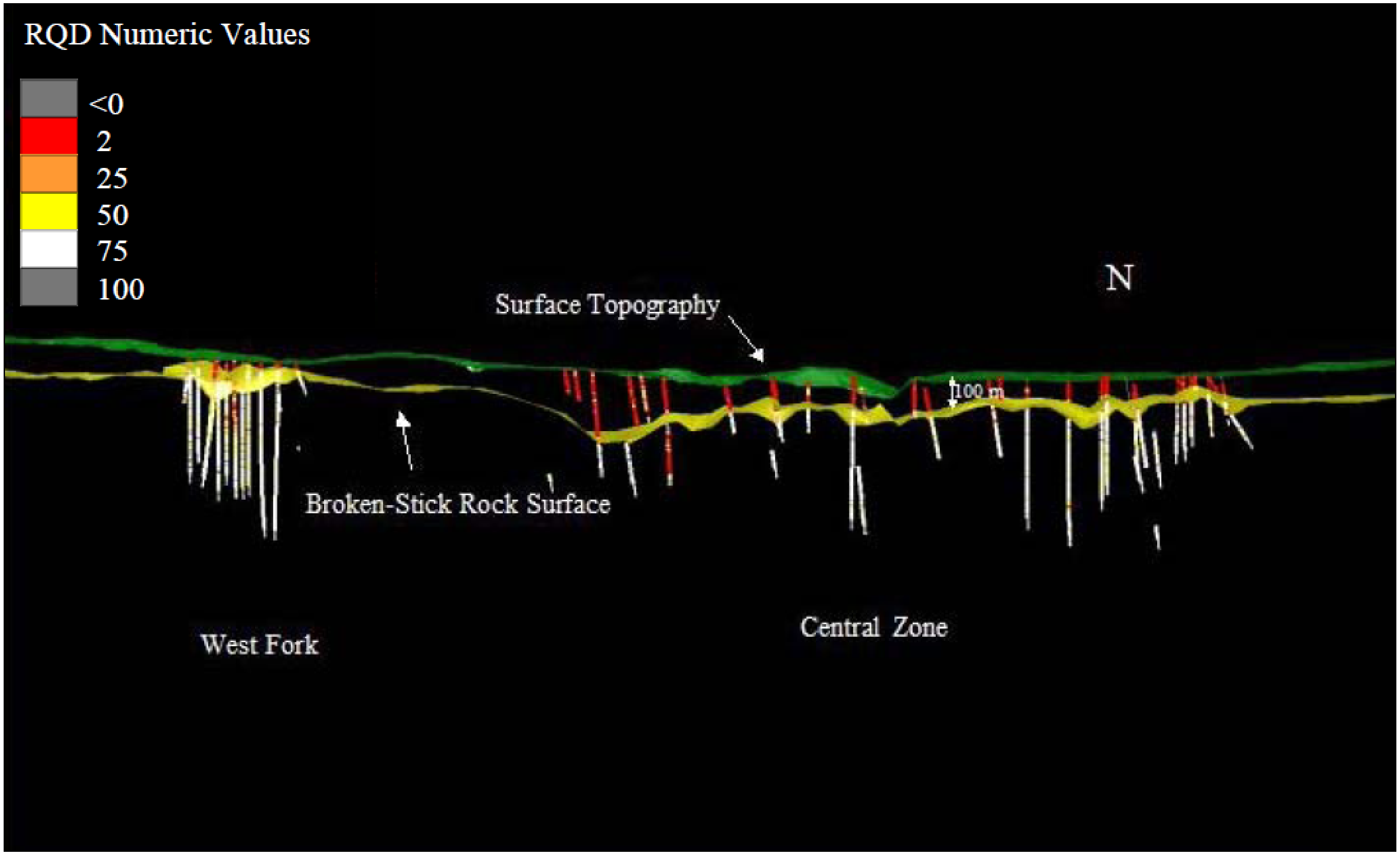
Some of the modeled broken rock zone is attributed by GCMC to a blockier fracture style. In these zones, the fracture style differs, with greater fracture spacing occurring at a number of orientations, and fractures are generally not associated with gypsum infill Workman (2006a).

Figure 10-5: Stick Rock/Broken Rock Boundary, Isopach Map



Note: Figure from Workman (2006a).

Figure 10-6: Stick Rock/Broken Rock Boundary, Example Long Section



Note: Long section through the Central Zone and West Fork Zone looking West showing the broken-stick rock surface, and topography. The red drill hole intervals represent RQD <25%, and the white intervals represent RQD >75%. Figure from Workman, 2006a.

The origin of the fracture cleavage (termed sheet fractures for mining engineering purposes) was described by Allen (1971), who recognized a depth zonation from open sheet fractures near surface, to gypsum-filled fractures, to a deeper zone characterized by the presence of anhydrite veining and the absence of sheet fracturing. Allen attributed the formation of these fractures to the hydration of anhydrite by meteoric waters, which caused a volume increase of up to 67% that allowed fractures to develop. The fractures filled with gypsum which was re-deposited from anhydrite dissolution Allen (1971).

A contributing factor may have been that fracture cleavage orientation was enhanced by, or possibly as a result of, glacial unloading Workman (2006a).

10.6 Collar Surveys

Proposed drill sites were initially located in the field by a geologist using a hand-held GPS unit; a pad was then built for the drill, 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. Nominal accuracy of these positions is “capable of delivering centimetre level static post produced point reconnaissance” (Workman, 2006a).

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.

A total of 544 drill holes have been surveyed by SpectrumGold, NovaGold and GCMC, representing the 2003 to 2010 field seasons.

10.7 Downhole Surveys

NovaGold collected down-hole survey data using various methods and instrumentation from 2003 to 2006. The methods are as follows:

- 2003: Sperry Sun - entered to Excel spreadsheets
- 2004: IceField Tool – data stored on electronic data files (only a subset of data could be located from archives)

- 2005–2010: Reflex EZ Shot and/or Gyroscope – data entered from original downhole survey records included with scanned drill logs.

Magnetic declination correction factors were applied for all drilling between 2003 and 2010; corrections were between 22°E and 24°E, depending on the year.

10.8 Geotechnical and Hydrological Drilling

Most geotechnical holes and all water-monitoring holes completed on behalf of NovaGold and GCMC were drilled with two HT-750 top drive rotary drill rigs, provided by Foundex Explorations Ltd. of Surrey, BC. Artesian holes were plugged and capped to minimize surface water flow in the area.

To increase sample recovery of the soils, overburden coring was conducted in the majority of the holes. Standard penetration testing (SPT) was carried out in select holes where required. When bedrock was encountered, triple tube coring with either an HQ3 core barrel (61 mm diameter core) or PQ3 core barrel (83 mm diameter core) was used. Representative soil and rock samples were collected during drilling for laboratory index and strength testing.

Holes were primarily drilled vertically, ranging from 26 to 200 m in total length (average length of 67 m). Three deep inclined boreholes were also drilled: boreholes PC06-019 and PC06-021 at the proposed tailings dam; and GC06-0726 in the planned Central pit highwall. These holes ranged from 240 to 402 m (average of 314 m) in length and were drilled 50º to 65º from the horizontal. To better characterize the orientation of major and minor discontinuities under the tailings dam, boreholes PC06-019 and PC06-021 were oriented where possible using the Ezy-mark system.

10.9 Metallurgical Drilling

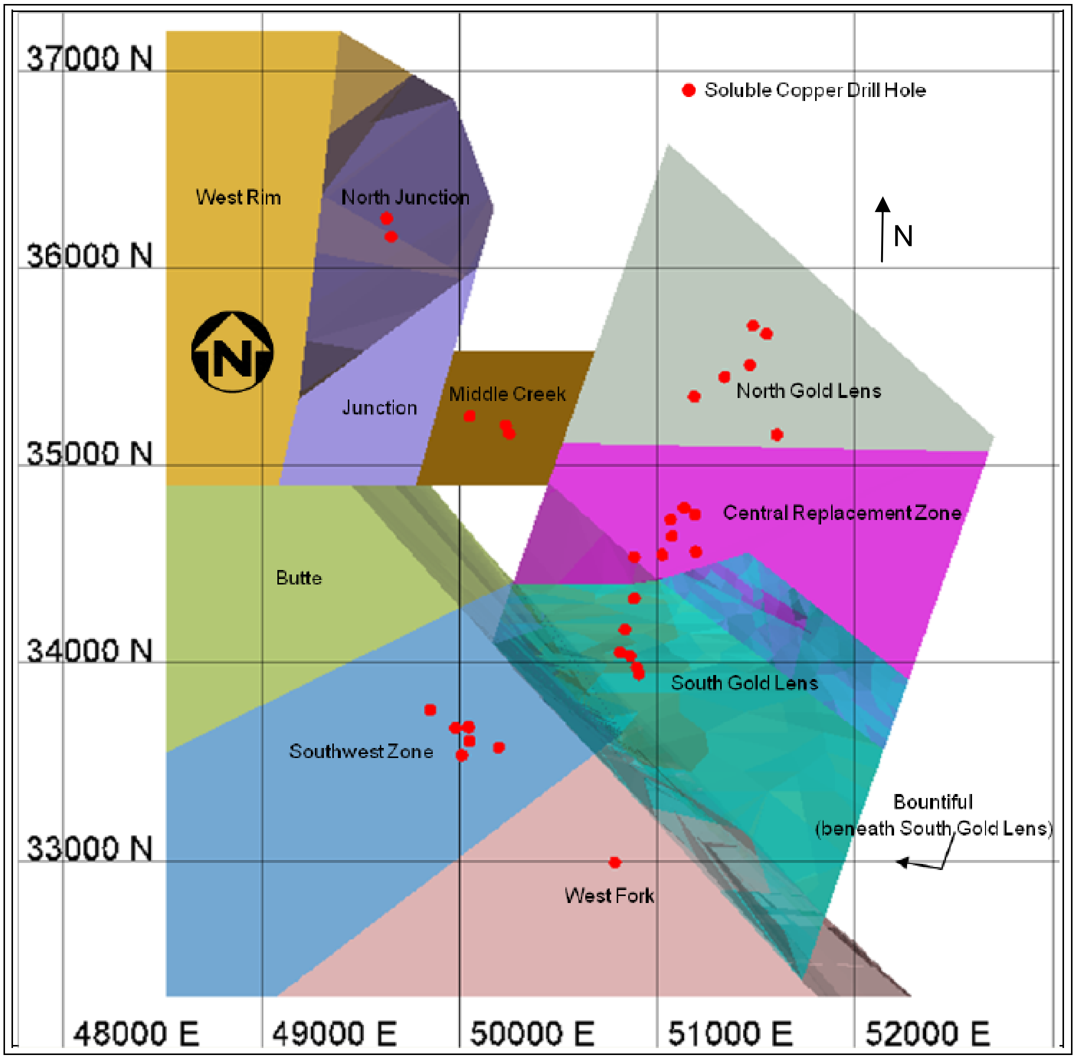
The metallurgical drilling, on which the testwork described in Section 13 was performed, comprises selected drill samples from the 2005 and 2008 drill programs. Drill hole locations were indicated in Figure 10-3.

The locations of the core samples that were used to generate the soluble copper assays are shown in Figure 10-7.

10.10 Sample Length/True Thickness

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

Figure 10-7: Distribution of Drill Holes with Acid-Soluble Copper Assays



Note: Figure courtesy GCMC, NovaGold and Teck.

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.

10.11 Drill Intercepts

Table 10-3 presents an example of the types of drill intercepts that have been returned for the Galore Creek deposit areas. Drill hole orientations are indicated on the cross-sections included in Section 7 of this Report.

10.12 Comment on Section 10

In the opinion of the AMEC QPs, the quantity and quality of the lithological, geotechnical, collar and downhole survey data collected in the exploration and infill drill programs completed by NovaGold and GCMC are sufficient to support Mineral Resource and Mineral Reserve estimation as follows:

- Core logging meets industry standards for copper, gold, and silver exploration within a porphyry setting
- Collar surveys have been performed using industry-standard instrumentation
- Downhole surveys were performed using industry-standard instrumentation
- Recovery data from core drill programs are acceptable
- Geotechnical logging of drill core meets industry standards for planned open pit operations
- Drill orientations are generally appropriate for the mineralization style, and have been drilled at orientations that are optimal for the orientation of mineralization for the bulk of the deposit area
- Drill orientations are shown in the example cross-sections included in Section 7, and can be seen to appropriately test the mineralization
- Drill hole intercepts as summarized in Table 10-3 appropriately reflect the nature of the copper, gold, and silver mineralization. The table demonstrates that sampling is representative of the copper, gold, and silver grades in the deposits, reflecting areas of higher and lower grades
- No material factors were identified with the data collection from the drill programs that could affect Mineral Resource or Mineral Reserve estimation.

Table 10-3: Drill Intercept Summary Table

Hole ID	Easting	Northing	Elevation	Azimuth	Dip	From	To	Length (m)	Cu (%)	Au (g/t)	Ag (g/t)	Comment
Bountiful												
GC06-0740	351300.181	6334083.447	785.985	0	-90	345	797.27	452.27	0.67	0.25	6.2	
GC06-0741	350978.882	6333844.579	767.636	0	-90	350.66	668.42	317.76	0.72	0.21	7.2	
GC06-0743	351120.186	6333848.213	771.284	0	-90	398.07	658.7	260.63	0.59	0.26	4.9	
GC06-0748	350827.194	6333895.793	796.496	0	-90	522	555	33	0.72	0.32	5.9	
GC06-0749	351281.051	6333833.907	822.135	0	-90	346.05	580.2	234.15	0.55	0.18	3.7	
GC06-0753	351168.356	6334303.927	739.703	90	-70	438	681	243	0.25	0.05	4.3	
GC06-0754	351086	6333713.554	771.091	0	-90	414	629	215	0.73	0.30	5.1	
North Gold Lens												
GC66-0225	351399.61	6335503.99	714.361	0	-90	68.9	131.7	62.8	1.31	1.26	9.2	
GC65-0204	351282.15	6335498.55	725.632	90	-58	82.3	188.7	106.4	0.81	0.43	5.1	
GC06-0734	350962.704	6335602.372	862.801	0	-90	608.3	697.7	89.4	0.60	1.38	4.5	
GC06-0727	351626.125	6335736.835	694.193	0	-90	48.0	82.0	34.0	0.32	0.23	1.8	
GC05-0625	351585.081	6335493.495	697.803	0	-90	44	187.9	143.9	1.12	0.64	9.9	
GC05-0606	351341.695	6335447.857	717.131	0	-90	102	336.5	234.5	0.84	0.42	5.7	
Central Replacement Zone												
GC91-0431	351067.99	6334625.36	735.24	90	-75	18.3	75.0	56.7	2.04	0.34	11.5	
GC66-0224	351372.83	6335399.56	712.662	0	-90	94.5	484.6	390.1	0.65	0.22	6.2	
GC66-0221	351340.73	6335289.4	717.867	0	-90	137.2	317.0	179.8	0.76	0.17	9.0	
GC66-0220	351289.8	6334865.93	670.47	0	-90	216.8	381.0	164.2	0.72	0.15	9.6	
GC65-0217	351714.87	6335083.29	660.865	272	-35	21.3	101.7	80.3	0.46	0.12	4.9	
GC06-0733	350906.515	6334820.659	709.071	0	-90	310.3	359.3	49.0	0.30	0.08	2.6	
GC06-0732	351106.249	6334838.273	689.569	0	-90	192.1	317.0	124.9	1.01	0.37	12.0	
South Gold Lens												
GC06-0746	351267.252	6334203.39	747.421	90	-70	299.0	706.0	407.0	0.64	0.26	6.6	
GC06-0737	351082.322	6333949.708	767	90	-66	297.0	588.0	291.0	0.73	0.26	6.2	
GC65-0215	350854.24	6334128.9	779.593	265	-41	9.1	91.4	82.3	0.98	0.56	4.6	
GC65-0206	350959.67	6334346.88	758.916	275	-44	48.8	173.0	124.2	1.18	0.20	6.8	
GC06-0731	350934.228	6334046.261	767.168	90	-67	333.5	708.9	375.4	0.60	0.17	5.9	
GC06-0725	351164.697	6334100.706	795.137	0	-90	344.5	641.1	296.6	0.74	0.26	6.9	
GC05-0660	351356.563	6334429.013	745.165	139	-60	394.0	459.0	65.0	0.50	0.25	7.0	
Junction												
GC91-0408	349396.57	6335956.15	1145.642	100	-55	305.0	395.0	90.0	0.36	0.23	2.7	
GC91-0407	349785.93	6336033.94	1178.188	130	-50	92.0	134.0	42.0	0.55	0.19	4.6	
GC73-0313	349622.97	6336344	1262.246	0	-90	151.9	347.5	195.5	1.03	0.68	not assayed	Au assay interval of 140.21 m of Cu interval 207.26 to 347.47
GC66-0229	349650.31	6336064.98	1162.234	282	-65	33.5	136.8	101.7	1.53	1.17	10.7	1.53 m interval no recovery
GC05-0567	349632.285	6336251.134	1233.112	132	-75	82.9	282.7	199.8	1.41	0.61	12.7	
GC05-0558	351132.801	6335547.417	758.344	250	-75	273	357.0	84.0	1.15	0.96	7.0	

Hole ID	Easting	Northing	Elevation	Azimuth	Dip	From	To	Length (m)	Cu (%)	Au (g/t)	Ag (g/t)	Comment
West Fork												
GC06-0751	350578.919	6333148.243	833.618	250	-67	270.4	460.6	190.3	0.68	0.50	4.3	
GC05-0707	350670.805	6332935.96	804.554	70	-70	191.0	250.0	59.0	0.62	0.32	2.9	
GC05-0703	350602.688	6333094.59	822.039	270	-82	201.3	242.0	40.7	0.53	0.38	2.9	
GC05-0685	350715.824	6333063.786	801.234	0	-90	287.0	341.0	54.0	1.26	0.86	22.2	
GC05-0648	350659.511	6333047.602	813.699	0	-90	229.7	275.6	45.9	0.79	0.49	6.9	
GC05-0648				0	-90	340.5	414.4	73.9	0.64	0.28	5.0	
GC05-0638	350533.061	6333030.498	852.298	0	-88	268.5	334.0	65.5	0.64	0.35	3.4	
GC04-0480	350787.278	6332990.032	789.518	255	-77	26.4	60.0	33.6	14.33	1.62	86.5	Opulent
Southwest												
GC04-0502	350012.617	6333532.971	938.363	0	-75	212.2	370.8	158.6	0.96	0.78	5.5	
GC91-0406	350117.71	6333413.52	925.49	0	-60	283.0	319.0	36.0	0.75	0.80	7.0	
GC91-0403	349872.58	6336618.65	1298.461	0	-61.5	187.0	244.0	57.0	0.51	0.25	3.9	
GC91-0398	349992.18	6333524.15	941.706	0	-61.5	160.8	256.8	96.0	1.05	0.82	5.2	
GC91-0395	351190.9	6334941.94	730.408	0	-61.5	47.1	380.3	333.2	0.89	0.46	9.4	
GC90-0383	350003.41	6333623.88	936.385	0	-61.5	30.2	159.7	129.5	0.85	1.12	3.7	
Middle Creek												
GC91-0419	350238.72	6335208.89	939.57	270	-65	124	141.8	17.8	1.77	3.68	12.4	
GC05-0580	350053.015	6335249.139	983.195	145	-70	55.5	105.0	49.5	1.11	1.69	6.8	
GC05-0569	350400.223	6335254.679	918.309	160	-60	78	108.0	30.0	0.72	0.71	3.0	
GC05-0545	350253.164	6335161.173	923.76	0	-88	28.5	87.0	51.0	0.63	1.02	3.8	7.5 metres unassayed within interval
GC05-0516	350079.001	6335153.197	958.628	120	-60	147	160.8	13.8	0.61	0.11	3.0	
GC05-0513	350236.599	6335201.804	942.41	0	-90	110	175.5	50.3	1.32	2985.38	12.5	15.15 metres unassayed within interval

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sampling Methods

11.1.1 Geochemical Sampling

There is no information that was made available to AMEC for the various geochemical sampling programs. As the data has been superseded by underground and surface drill hole sampling, it is not considered further in this Report.

11.1.2 Underground Sampling

Underground sampling in 1966 and 1967 used two methods, and had two different purposes, which included:

- Provide sufficient material that a metallurgical composite could be prepared from drift round samples
- Completion of channel sampling for determination of grade.

The following notes on the underground sampling program have been compiled from McAusland (1967).

Drift Sampling

Approximately 56 tons (50.8 t) of metallurgical sample were taken from four crosscuts in the west section of the Central Zone, plus two locations in the adit. These samples were taken from ore that had been dumped off a trestle, then hauled and stockpiled by the round in cones containing about 30 tons (27.2 t) of ore. The metallurgical sample was bagged from a 2-foot (0.61 m) deep channel cut up the side of the cone. A sample splitter was tried but found to be unsatisfactory for several reasons: the available loader was unsuitable, split muck had to be moved three times, the splitter was easily plugged in snowy weather, and sample material stuck to the splitter in cold weather. All bags were marked as to round location and were flown to Terrace prior to shipment by railway to the Kennecott Research Centre's pilot plant in Salt Lake City. This composite is not used to support Mineral Resource estimation.

Channel Sampling

Continuous 10 ft (3 m) channel samples were taken on all drift walls, plus vertical channels alongside the traces of diamond drill holes. Correlation across the drifts to within 0.10% Cu was maintained in low-grade (less than 1% Cu) zones, but where massive chalcopyrite occurred and assays exceeded 1.50% Cu, there were often variations exceeding 0.40% Cu for opposing walls. Samples taken alongside core drill holes were found to agree to within 0.10% Cu. The channel samples have been converted into "pseudo drill holes" and are used in the resource estimation database.

11.1.3 Drill Sampling

Historic Sampling

1960s

Prior to 1964, drill core was halved and then split in 10 ft (3 m) lengths. Samples were despatched to the now closed Coast Eldridge laboratory in Vancouver for copper analysis. Gold analysis was completed on some intervals.

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 (6.3 mm) then a 0.75 lb (340 g) split was separated using a Jones splitter.

1970s

During the 1970s, the onsite laboratory at Galore Creek was still in use. Half core samples were crushed to ½ inch (12.7 mm) and split to obtain a 0.75 lb (340 g) 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 Laboratory in North Vancouver or Chemex Laboratory, also in North Vancouver, for assay. Kennco Exploration Laboratory was used during 1972–1973, whereas the Chemex laboratory was used in 1974.

Mingold

During the 1990 Mingold program, half of the split core was crushed on site at the Galore Creek Laboratory to ¼ inch (6.35 mm) and a 300–325 g split was taken and shipped to the former Mineral Environments Laboratories (Min-en Laboratories) in Smithers, BC for further processing and assaying.

SpectrumGold

Drill core sampling occurred within a minimum of 3.2 feet (1 m) and a maximum of 10 ft (3 m) intervals. Drill core in mineralized intervals was generally sampled on approximately 6.5 ft (2 m) intervals. Where core was considered to be unmineralized, sample intervals were increased to 3 m. Sampling also honoured lithological, mineralogical and major structural changes in the drill core, leading to sampling lengths that were longer or shorter than the average.

All core samples were tagged by the geologist that logged the hole. All the drill core samples were split using a rock saw. One half of the core was returned to its original box (5 ft or 1.5 m long wooden box) for long-term storage. The remaining half was sealed in a polyethylene bag for direct shipment to the ALS Chemex laboratory in Vancouver for analysis.

Sampling protocol called for the geologist to insert three control samples; a blank, a standard and a duplicate for every 20 samples to be submitted to the lab for analysis. The 20 sample size equates with the size of the sample batch grouped for analysis at the laboratory. The placement of all control samples was essentially random within the 20-sample batch. Blanks, which consisted of non-metalliferous marble, were inserted as determined by the geologist and bagged during the core splitting and sampling collection. Duplicate sample locations were marked by the geologist. Empty bags with duplicate tags were submitted to the laboratory to indicate the sample was to be split for duplicate analysis.

Shipment of core samples from the Galore Creek camp occurred on a hole by hole basis. Rice bags, containing four poly-bagged core samples each, were marked and labelled with the sample numbers and the ALS Chemex address. Rice bags were assembled into sling loads for transport by helicopter to the Bob Quinn airstrip, where they were stored in a secure metal container. Subsequently, the samples were transported by truck by Banstra Freight Forwarders, and delivered directly to the laboratory.

NovaGold

The NovaGold programs used protocols developed during the SpectrumGold work.

All drill core was transported by helicopter or truck in secure core “baskets” to the Galore camp for logging and sampling. Sample intervals were determined by the geologist during the geological logging process. Sample intervals were labelled with white paper tags and butter (aluminum) tags which were stapled to the core box. Each tag had a unique number which corresponded to that sample interval. Core was brought into the saw shack where it was split in half by the rock saw, divided into sample intervals, and bagged by the core cutters. Not all core was oriented; however, core that had been oriented was identified to samplers by a line drawn down the core stick. If core was not competent, it was split by using a spoon to transfer half of the core into the sample bag.

Sample intervals were determined by the geological relationships observed in the core and limited to a 3 m maximum length and 1 m minimum length. An attempt was made to terminate sample intervals at lithological and mineralization boundaries. Sampling was generally continuous from the top to the bottom of the drill hole. When the hole was in unmineralized rock, the sample length was generally 3 m, whereas in mineralized units, the sample length was shortened to 2 m.

One sample for approximately every 10 m of core was selected for point load testing and specific gravity measurements. Once the core was sawed, half was sent to ALS Chemex Laboratories (Vancouver) for analysis and the other half was stored at the Galore Creek camp.

Shipment of core samples from the Galore Creek camp occurred on a drill hole by drill hole basis. Rice bags, containing four poly-bagged core samples each, were marked and labelled with the sample numbers and the ALS Chemex address. Rice bags were assembled into sling loads for transport by helicopter to the Bob Quinn airstrip, where they were stored in a secure metal container. Subsequently, the samples were transported by truck by Banstra freight forwarders, and delivered directly to the laboratory. In 2005, security tags were strapped onto the rice bags as a means of verifying that the bags were not opened prior to their arrival at ALS Chemex.

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 20 core samples:

- Standards: 10 standards were used at Galore Creek. The core cutter inserted a sachet of the appropriate standard, as well as the sample tag, into the sample bag
- Blanks: were composed of an unmineralized landscape aggregate. The core cutter inserted about 150 grams of blank, as well as the sample tag, into the sample bag
- Duplicates: the assay laboratory split the sample and ran both splits. The core cutter inserted a sample tag into an empty sample bag.

GCMC

The GCMC programs used the NovaGold protocols. During the 2010 campaign, logging and sampling was executed from the Espaw camp; core was flown to the Galore Creek Valley core-yard for long-term storage.

11.2 Metallurgical Sampling

No information on the sampling methods used for metallurgical samples is available for the legacy metallurgical testwork, with the exception of the information in Section 11.1.2 on the drift sampling.

Depending on the date of the program, NovaGold and GCMC samples could be either half-core splits, or complete core. For the 2010 metallurgical program, the following metallurgical compositing was undertaken at G&T Laboratories (Kamloops):

- A list of 138 sample intervals was provided that included samples from six drill holes, holes 794 through to 799. The intervals averaged 4 m in drill depth, but varied between 2 m and 12 m
- Each interval was hand split into two equal portions, one for ore hardness testing and the other for assaying and metallurgical testing. The intervals were divided by selecting approximately 10 cm of drill core for ore hardness testing and the next 10 cm for metallurgical testing, until the interval was completed. Competent sections of core were split using a chisel and a hammer when there were no natural breaks that satisfied the 10 cm division target
- The interval portions for assaying and metallurgical testing were first crushed to -6 mesh and a sub-sample split out for head assay.

11.3 Density Determinations

11.3.1 Historic Specific Gravity Work

A total of 563 specific gravity (SG) measurements were made on the total Galore Creek property during the 1966–67 drill campaign by measuring the weight of the sample and dividing by the volume of water it displaced.

11.3.2 Mingold

A total of 1,337 specific gravity determinations were collected using the water displacement method:

(weight of sample in air) ÷ (volume of water displaced by sample when immersed)

Documentation of SG determinations can be verified against scans of the original drill logs. An additional 164 specific gravity determinations were collected from the 1960s, 1970s and 1990s drill campaigns; however, the methodology and name of who conducted the work is unknown.

11.3.3 SpectrumGold

During the 2003 and 2004 drill programs, 80 specific gravity determinations were made by ALS Chemex using Specialty Assay Procedure OA-GRA08.

11.3.4 NovaGold

A total of 12,599 SG determinations were collected by water immersion methods.

Specific gravity determinations were collected at a rate of one sample per approximately every 10 m of drilling. The weight of unbroken pieces of core less than 15 cm long was determined both in air (dry) and in water (wet) by the geotechnical staff during the core photography process. Hard tap water was used for the measurement. Samples were not wax-coated; this can result in a slight increase in final specific gravity readings due to water being retained in microfractures, voids and pores within the drill core. Results were written on data entry sheets, which are located with the scanned logs on the GCMC ftp, and were entered by a data entry clerk in the NovaGold SG-Point Load Access database.

In 2005, NovaGold collected specific gravity values for materials above and below the disaggregation zone (the point at which anhydrite first becomes visible in core) by collecting data from test pits and split tube core measurements.

Ten small test pits were constructed across the exposed and backhoe accessible portions of the West Fork and the South Gold Lens areas. The average specific gravity for the pits was 2.01, but ranged from 1.39 in Pit 3, to 2.6 in Pit 10 (Lechner, 2006). Pit values were considered to bias low on the specific gravities, due to difficulties in determining appropriate water-fill levels, collapsing of test pit walls. In addition, the ten determinations do not provide a statistically meaningful sample.

Split tube measurements were completed on the entire diamond core of three drill holes, and comprised measuring the weight of an empty tube barrel, and subtracting this from the weight of a tube barrel with core. Corrections were made for recovery; the volume of rock in the core tube was estimated based on the core tube length, and core recovery. Excess water was drained from the tube prior to weighing. The weight was then divided by the estimated volume to produce a density value. Specific density values ranged from 2.28 to 2.57.

11.3.5 GCMC

During 2010, GCMC collected 255 specific gravity determinations from drill core. Sampling protocols were identical to NovaGold sampling procedures.

11.3.6 Density Data Verification

During the fall of 2008, Mr. Rex Turna, a geologist and NovaGold employee seconded to GCMC, conducted a 100% audit of the NovaGold era (2003–2007) specific gravity measurements in response to concerns raised during an audit conducted by AMEC (2008).

Mr. Turna compared original SG logs to the database and made corrections where data entry errors were apparent, as well as entered data that had originally been omitted.

Mr. Turna conducted his audit using the software DataLogger, as the Galore Creek database had been converted to DataShed™ format during the spring/summer of 2008.

Elimination of suspect or erroneous specific gravity determinations (SG values <1 and/or >6) reduced the database to 14,722 records that could be used to support Mineral Resource estimation.

11.3.7 Disaggregation Zone Adjustment Factor

Split tube measurements were completed on the entire diamond core of three 2005-era drill holes. Only two of the three drill holes pierced the 2005 pit shells, thus only these measurements were used to calculate the disaggregation adjustment factor of 9.3%.

Because the 9.3% reduction factor is significantly greater than the deposit average reduction factor of 4.65%, and because this factor is based on only two drill holes located in the Central Replacement Zone and South Gold Lens, as opposed to >5,000 samples scattered throughout the deposit, it is suggested that additional measurements be collected of specific gravity values within the broken rock.

11.3.8 Moisture Content

Historically all specific gravity 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 Laboratories. 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.

Bulk density values were determined by adjusting specific gravity values below the broken rock–non-broken rock surface by -0.5% to account for moisture content.

11.4 Analytical and Test Laboratories

The laboratories used during the various exploration, infill and step-out drill analytical programs completed on the Galore Creek Project are summarized in Table 11-1.

Metallurgical testwork has been completed at a number of laboratories, but primarily by G&T Metallurgical (G&T) laboratories in British Columbia. Laboratories used are summarized in Table 11-2.

Metallurgical laboratories are not typically accredited or certified.

11.5 Sample Preparation and Analysis

11.5.1 1960s

From 1961 to 1963, core samples were assayed for copper at the Coast Eldridge laboratory in Vancouver. Gold was assayed on 100 ft (30 m) composites for select drill hole intervals.

In 1964, the Galore Creek assay laboratory was constructed on site. Drill core samples were split in half and one half was crushed to nominal ¼ inch.

A 340 g split of this material was then crushed to -10 mesh, pulverized to -100 mesh, and assayed for copper using a double digestion with titration and colorimetric determinations. Samples reporting assays greater than 0.4% copper over intervals of 40 to 60 ft (12 to 18 m) were composited and shipped to Coast Eldridge to be assayed for gold and silver. The assay methods employed for gold and silver at Coast Eldridge are not known by GCMC. It is not known if these pulps were rehomogenized before compositing. Security measures taken during this program are also unknown.

Table 11-1: Analytical Laboratories

Laboratory Name	Location	Years Used	Accreditation	Comment
Coast Eldridge	Vancouver, BC	1961 to 1963	Accreditations are not known	Primary laboratory. Also performed verification checks in 1964
Galore Creek	Onsite preparation facility	1964	Accreditations are not known	
Kennecott Explorations	North Vancouver	1964	Accreditations are not known	Verification checks
Hawley and Hawley Assayers & Chemists	Tucson, Arizona	1964	Accreditations are not known	Verification checks
Kennecott Bear Creek laboratory	Denver, Colorado	1964	Accreditations are not known	Verification checks
Coast Eldridge	Sudbury	1967	Accreditations are not known	Verification checks
Kennco Exploration	Vancouver	1972–1973	Accreditations are not known	Primary laboratory
Chemex Laboratories	Vancouver	1974	Accreditations are not known	Primary laboratory
Mineral Environments Laboratories	Smithers, BC	1990–1991	An ISO 17025-certified laboratory, though GCMC does not know whether it was certified at the time the assays were performed	Primary laboratory
Eco Tech Laboratory Ltd	Kamloops	1991	An ISO 9001-registered assay laboratory, though GCMC does not know whether it was registered in 1991	Verification checks
ALS Chemex	Vancouver	1994	An ISO 9001-certified laboratory; though GCMC does not know whether ALS Chemex was certified at the time the assays were completed	Primary laboratory
ALS Chemex	Vancouver	2004–2010	In 2004, ALS Chemex held ISO 9002 accreditation, this changes to ISO 9001 accreditations from late 2004; ISO/IEC 17025 accreditation was obtained in 2005	Primary laboratory

Table 11-2: Metallurgical Laboratories

Laboratory Name	Location	Years Used	Comment
Hazen Research Inc.	Unknown, but likely to be the Denver/Golden Colorado facility	1960–1967	Comminution, flotation, pilot plant on bulk sample
Britton Research Laboratories	Vancouver, BC	1960s	Grinding and flotation on low grade samples
Dawson Metallurgical Laboratories	Salt Lake City, Utah	1992	bench flotation tests
G&T Metallurgical Services Ltd	Kamloops, British Columbia	2003–2011 and ongoing	Primary testwork facility
SGS Lakefield	Toronto, Ontario	2006	Comminution and flotation simulation tests, CEET modelling (grinding)
SGS MinnovEX	Toronto, Ontario	2006	Comminution and flotation simulation tests, CEET modelling (grinding)

Significant differences were noted between the pre-1964, 100 ft (30 m) composite gold assays and the later 40 to 60 ft (12 to 18 m) composite gold assays. These long composites are predominantly outside of the resource in unmineralized intrusive rocks; therefore, they are not material to the resource, but nevertheless, long composites have not been used to support the Mineral Resource estimates.

In 1991, Kennecott re-assayed approximately 64% of the sample intervals from 1960s drilling using acceptable QA/QC protocols.

In 1964, checks of Galore Creek laboratory copper assays were reportedly carried out by the in-house Kennecott Explorations laboratory in North Vancouver, Coast Eldridge, Hawley and Hawley Assayers & Chemists and the in-house Kennecott Bear Creek laboratory. These data were not made available to NovaGold or GCMC.

In 1967, 140 samples originally assayed by the on-site laboratory were assayed for copper at a number of other laboratories: Coast Eldridge, Sudbury, and three in-house Kennecott laboratories. Galore Creek laboratory copper assays were found to agree well with the copper assays at the other laboratories.

Security measures taken during these programs are unknown to NovaGold or GCMC. NovaGold is not aware of any reason to suspect that any of these samples have been tampered with.

11.5.2 1970s

Core from the 1970s drill campaigns was split in half, and one half was crushed to nominal ½ inch (13 mm) on site and split to obtain a ¾ lb (0.3 kg) sub-sample. This material was then further crushed in a cone crusher, placed in Kraft paper bags, and shipped in locked metal boxes for assay. The in-house Kennco Exploration laboratory served as the primary laboratory for 1972 to 1973, and Chemex Laboratories served as the primary laboratory in 1974.

The assay methods employed for copper and gold during this time period are not known by GCMC. Gold and silver were assayed on composited intervals where copper assayed greater than 0.4%. Quality control procedures and security measures employed during these programs are unknown to GCMC.

In 1991, Kennecott re-assayed approximately 95% of the sample intervals from 1970s drilling using QA/QC protocols that GCMC considers to be acceptable.

11.5.3 Mingold

During the 1990 drill program, drill core was split in half and one half was crushed to nominal ¼ inch (6 mm) on site, split to generate a 320 g to 325 g sub-sample, and sent to Mineral Environments Laboratories (Min-En) for assaying. Gold was assayed by fire assay pre-concentration and atomic absorption finish on a 30 g sub-sample. Samples reporting greater than 1.0 g/t Au were assayed a second time. Metallic screen assays were performed on samples reporting greater than 0.1 oz/ton Au (3.11 g/t Au) by fire assay. Metallic screen assays at Min-En were performed by pulverizing the coarse reject for the interval to -102 mesh, recombining this material with the previous pulp portion, and sieving the recombined sample with a 120 mesh screen. The assay from the +120 mesh fraction and two assays from the -120 mesh fraction were then weight-averaged to produce a net gold value.

Copper and silver assays were performed on a 2 g sub-sample split from the initial pulp. The assay methods employed are unknown to GCMC. Quality control procedures and security measures employed during this program are unknown to GCMC.

In 2006, the accuracy of the 1990s copper assays were checked by NovaGold and found to be biased high.

11.5.4 Kennecott

In 1991, Min-En was again used as the primary laboratory, but the sample preparation procedures were improved over the methods used in previous years. Core was split in half and one half was crushed to nominal 1/8" (3 mm) before a 500 g split was taken, pulverized to 95% passing -120 mesh, and rolled and bagged for analysis.

Gold was assayed by standard fire assay on a one assay ton sub-sample (29.166 g). Internal QA/QC procedures at Min-En included one blank and one standard in each assay batch of 24 samples. Where the value of the standard fell outside the 95% confidence limit, the entire batch was re-run. The top 10% of gold assays on each assay page were rechecked and reported in duplicate along with the standard and blank results.

Check assays were performed on every 20th sample by Eco Tech Laboratory Ltd. (Eco Tech). Comparison by NovaGold of 571 check assays against original assay values in NovaGold's opinion showed reasonable correlation for copper and fairly good correlation for gold greater than 0.25 g/t Au, although Eco Tech assays tended to be marginally higher. Gold grades less than 0.25 g/t Au showed considerable variation.

Kennecott also undertook a major resampling program in 1991 to replace gold assays from large composite intervals from the 1960s and 1970s drill campaigns. A total of 100 t (18,784 samples) of drill core and coarse reject samples were shipped from the property to Min-En for gold assay. Approximately 64% of 1960s sample intervals and 95% of 1970s sample intervals were re-assayed as part of this program. This re-assay campaign included quality control procedures and the resulting assays replaced the original gold assays in the Galore Creek resource database.

11.5.5 SpectrumGold

Samples were logged into a tracking system on arrival at ALS Chemex, and weighed. Samples were then crushed, dried, and a 250 g split pulverized to greater than 85% passing 75 µm.

Gold analysis was undertaken on a 30 g sample, using fire analysis, followed by atomic absorption spectroscopy (AAS). Lower and upper detection limits were 0.005 ppm Au and 10 ppm Au, respectively. Values over the detection limits were rechecked using nitric acid aqua regia digestion of a 0.4–2.0 g sample followed by AAS finish.

An additional 34-element suite was assayed by inductively-coupled plasma optical emission spectroscopy (ICP_AES) methodology, following nitric acid aqua regia digestion. Analytical results were corrected for inter-element spectral interferences.

Pulp and reject samples from the 2003 program are stored at the Main Staging area.

11.5.6 NovaGold

Sample preparation methods during the NovaGold 2004–2007 programs were similar to those developed by SpectrumGold.

Samples were logged into a tracking system on arrival at ALS Chemex, and weighed. Samples were then crushed, dried, and a 250 g split pulverized to greater than 85% passing 75 µm.

Gold assays were determined using fire analysis followed by an AAS finish. The lower detection limit was 0.005 ppm Au; the upper limit was 1,000 ppm Au. An additional 34-element suite was assayed by ICP_AES methodology, following nitric acid aqua regia digestion. The copper analyses were completed by atomic absorption (AA), following a triple acid digest.

11.5.7 GCMC

Sample preparation methods during the GCMC 2008 and 2010 programs were similar to those developed by NovaGold.

Samples were logged into a tracking system on arrival at ALS Chemex, and weighed. Samples were then crushed, dried, and a 250 g split pulverized to greater than 85% passing 75 µm.

Gold assays were determined using fire analysis followed by an AAS finish. The lower detection limit was 0.005 ppm Au; the upper limit was 1,000 ppm Au. An additional 34-element suite was assayed by ICP_AES methodology, following nitric acid aqua regia digestion. The copper analyses were completed by AA, following a triple acid digest.

11.6 Acid-Soluble Copper Determinations

Acid-soluble (oxide) copper is irregularly distributed in the near-surface environment of Galore Creek. In 2004–2005, NovaGold obtained a total of 916 acid-soluble assays from 31 drill holes. There have been no additional acid-soluble assays performed since that date. With the exception of the Junction and Butte Zones, every zone has at least one drill sample analyzed for soluble copper. 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.

Acid-soluble copper assays were performed by ALS Chemex, using an ore-grade Cu preparation method (non-sulphide) by sulphuric acid leach, with an AAS finish.

Workman (2006a) notes that the average solubility of all samples taken from the Central Zone, Middle Creek area, and Southwest Zone, above the oxide surface and with grades ≥ 0.35% total Cu is 27.7% soluble. The result was determined by:

% soluble = (soluble Cu) ÷ (total Cu) x 100

Sulphide samples, even at depth, showed a consistent solubility of about 5–6% (Workman 2006a).

11.7 Quality Assurance and Quality Control

The quality assurance and quality control (QA/QC) programs for the Project are discussed in the sub-section on sample preparation (Section 11.1).

11.8 Databases

All drilling related data are stored in a Microsoft Access database. There are currently three Access databases for Galore Creek:

- GaloreCC DDH2: This database has 20 tables: Alteration, Assay Composites, Certificate Data, Certificate Header, Collar, Core Photos, Corrections, Descriptions, Geotech, Grids, Litho, Minerals, Pima, Quicklog, Remarks, SG, Soluble Cu, Structs, Survey, Units, Various
- SG_PointLoad: This database has one table: Point Load
- GaloreDrillStatus: Project and Rig Geologists use this database to monitor drill site status.

All data collected in the field is transferred into the database via a set of prescribed steps, outlined in detail within the Galore Creek Procedures Manual (Workman, 2005).

The following methodologies were used for the NovaGold and GCMC dataflows:

- All data collected in the field is transferred into the database via a set of prescribed steps, outlined in detail within the Galore Creek Procedures Manual (Workman, 2005)
- Geological (including lithology, mineralization, alteration, structure etc.) and geotechnical data (RQD, recovery, fracture, weathering, hardness, etc.) is collected and recorded on paper logging sheets by on-site geologists and geotechnicians
- For the NovaGold programs, these sheets were transferred to two data entry personnel who input the data into the Access database via a Visual Basic interface, DDH-Tool, a proprietary internal software program developed in 1995. DDH-Tool produces an entry log which is saved along with each zipped and date tagged version of the database. Data entry was overseen by the Database Manager, to verify that proper procedures were utilized for data entry
- For the GCMC programs, drill data were input into DataLogger, a user-friendly data-entry interface. DataLogger generates an export file which is uploaded to the Database Manager who updates the central DataShed™ database. DataShed™ is a commercial data management software package
- Survey data are entered in the same manner to the geological data, although the original data are produced and recorded by the drillers, and is transferred to the data entry personnel via the drill foreman and geologists

- At the end of each field season, a 100% line-by-line check of all database tables is conducted, comparing values in the database to the those recorded on the original documents (which were scanned and filed at camp), to verify that the data transfer was accurate and that no errors had been introduced during data entry and upload
- Assay data is received from the laboratories via comma-separated value (CSV) data files. These files are compiled and imported by the Database Manager using Excel importers, text files and another Visual Basic interface called Import Edit Log. For the GCMC programs, assay data are input individually by certificate. After data are imported, visual checks are done to verify that data placement was correct within the various database fields. After each update, assay data together with all geologic data is loaded and visually validated in MineSight©, a commercial 3D mine planning software package.

All drilling-related data are currently stored in a DataShed™ database which resides on the NovaGold DataShed™ server in Vancouver B.C. and copies are stored on both GCMC and Teck servers.

11.9 Security

11.9.1 Sample Storage

Historic drill core has been stored in either plastic, galvanized steel or wooden boxes. All have been marked with metal tags inscribed with the drill hole number and interval. An estimated 1,500 m 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 was stacked on pallets from 1991 to 2005, and exposed to the elements. 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 quarter-split for check assaying.

During 2004 to 2007, NovaGold expended considerable effort to recover as much historic core as possible. Stacks of unorganized core trays and boxes from Galore Creek were pulled out of overgrown areas in the camp and reorganized in an orderly manner. Some drill core was not recaptured as the original boxes were spilled or tipped over by animals. The rest were restacked and stored in a temporary location in camp until 2005 when all core was transported to a designated storage area.

During the 2010 site visit, AMEC noted that the core is in an unprotected area and may be at risk of being destroyed due to weather, exploration or construction work.

Rejects from the Mingold drilling program do not exist. The drill core from this program is stored at Galore Creek, and is basically intact, apart from those intervals used in sampling.

Drill core from the SpectrumGold, NovaGold and GCMC programs are stored onsite at Galore Creek. Core is stored in an orderly, catalogued manner in the core yard. Post-assaying, all pulps from the SpectrumGold, NovaGold and GCMC programs are stored onsite at the Main Staging area.

All reject material was disposed of during the fall of 2010.

11.9.2 Sample Security

For all SpectrumGold, NovaGold and GCMC drill programs, the core cutters and the Saw Shack Manager verified in the Galore Creek saw shack that samples were properly cut and bagged and that any relevant information was recorded. Samples are placed into plastic bags, numbered with the sample tag inserted in the bag. Four of these sample bags were placed into one larger white rice bag, along with an Assay Instruction sheet. The outside of the bag had the sample numbers, hole number, and shipping address printed on the side. Beginning in 2005, the rice bag was secured using a red tamper-proof, numbered security tag. That tag number was recorded by the Saw Shack Manager, along with the sample numbers and hole number for that white rice bag.

There were typically as many as 40 rice bags per drill hole, depending on the drill hole depth. Typically up 20 white rice bags were bundled onto a pallet, depending on the weight of the core, and wrapped in polyurethane. The batch of core was labelled with an arbitrary batch number (which was also recorded along side the security tag number, hole, and sample numbers), and the address to ALS Chemex for assay. The batches were then strapped with metal banding.

2003 to 2006

Rice bags were assembled into sling loads for transport by helicopter to the Bob Quinn airstrip, where they were stored in a secure metal container. The samples were then transported by truck by Banstra freight forwarders, and delivered directly to the laboratory.

2007 to 2010

The batches were shipped via helicopter to the Staging Area and Filter Camp, and placed out of the way in a lay-down area, usually near the propane tanks until shipment. The shipment agent from 2005 to 2010 has been Canadian Freightways. During 2003 to 2004, and on one occasion in 2007, Bandstra was the shipping agent.

Core was not stored in a secured area; however, access to the area is limited to authorized employees. Upon arrival of the core at Staging, the warehouse attendants would record the batch and drill hole numbers of the core, as well as the condition in which it arrived and the date it was shipped out. The attendants would take the corresponding receipt with a way-bill number from Canadian Freightways and send a copy of that along with their other recorded info to the Saw Shack Manager at Galore Creek camp, where it is filed for tracking purposes. Using the way-bill number, GCMC could track the core via their website, or by calling Canadian Freightways if need be.

ALS Chemex and Canadian Freightways were instructed to contact GCMC if there was a problem with a broken security tag or bag; to date there have been no reported problems.

At Terrace the samples were stored at the ALS Chemex receiving facility. Unless specifically checking on a sample shipment with ALS Chemex or Canadian Freightways, GCMC would not receive notification of the sample arriving in Terrace until ALS Chemex started preparing the samples for assay.

11.10 Comment on Section 11

Sample collection, preparation, analysis and security for all SpectrumGold, NovaGold and GCMC drill programs are in line with industry-standard methods for porphyry gold–copper–silver deposits:

- SpectrumGold, NovaGold and GCMC drill programs included insertion of blank, duplicate and standard reference material samples
- SpectrumGold, NovaGold and GCMC QA/QC program results do not indicate any problems with the analytical programs
- SpectrumGold, NovaGold and GCMC data is subject to validation, which includes checks on surveys, collar co-ordinates, lithology data, and assay data. The checks are appropriate, and consistent with industry standards

- Independent data audits have been conducted, and indicate that the sample collection and database entry procedures are acceptable
- All core has been catalogued and stored in designated areas but is not being appropriately safeguarded against damage by weather or machines.

Sample collection, preparation, analysis and security for pre- SpectrumGold, NovaGold and GCMC drill programs are in assumed line with industry-standard methods for porphyry gold–copper–silver deposits but have not been verified with appropriate supporting QAQC results.

The AMEC QPs are of the opinion that the quality of the gold, copper, and silver analytical data from the SpectrumGold, NovaGold and GCMC drill programs are sufficiently reliable to support Mineral Resource and Mineral reserve estimation without limitation.

The AMEC QPs are also of the opinion that the quality of the gold, copper, and silver analytical data from the pre- SpectrumGold, NovaGold and GCMC drill programs are sufficiently reliable to support Mineral Resource and Mineral reserve estimation, but due to the lack of appropriate supporting QA/QC results, the data should not be used to support classification of Measured blocks.

Section 12 discusses findings associated with the pre- SpectrumGold, NovaGold and GCMC data, where a potential positive bias for low-grade copper and gold assays, and a potential negative bias for silver values may exist.

12.0 DATA VERIFICATION

A number of data verification programs have been undertaken on the Project data by independent consultants other than AMEC, and by NovaGold and GCMC personnel. This work is summarized in Table 12-1.

AMEC performed a data audit in 2007–2008 in support of Mineral Resource estimation and a second audit in 2011. The 2011 audit examined data collected or amended since the 2008 review.

12.1 2007–2008 AMEC Project Audit

An audit was completed by AMEC during 2007–2008 on the Project database, and consisted of a review of the drill data collected by Kennecott, examination of drill collar and down-hole survey data, a QA/QC review, and review of density data. Check sample results were also examined.

12.1.1 Database Audit – Legacy Data

AMEC randomly selected and checked a minimum of 10% of collar surveys, down-hole surveys, drill logs, and copper and gold assays from all legacy drilling campaigns against source documentation. These checks were completed to verify that sample data used in Mineral Resource estimation accurately represented the original logs, surveys, and assay certificates. AMEC found the Galore Creek resource database to be acceptably error-free. Error rates for the surveys, logs, and assays were found to be below 1.0%, the threshold commonly used by AMEC to indicate an acceptably error-free database.

The error rate for historic assays was determined to be acceptable at 0.9%. Errors found were typically typographic errors, resulting in small discrepancies in copper and gold assays. In AMEC's opinion these errors are not likely to significantly affect resource estimation. Though assays for historic drill holes were typically recorded in the margin of drill logs, as was commonly the case during this time period, original certificates were also present in the files for most drill holes.

The error rate for collar locations was also acceptable at <0.9%. Collar elevations on drill logs did not match values in the resource database; however, a check of collar elevations against the digital topographic surface shows that the elevations used in the database are acceptably correct. AMEC understands that several corrections have been applied to the drill hole elevations over the Project history.

Table 12-1: Data Verification Programs

Year	Review Party	Work Conducted	Findings
1992	Kennecott	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	All previous data were merged into a single database, audited and converted from imperial to metric units
2003	Ron Simpson	Spot checks on the 1991 re-assay program by comparing values from the original assay certificates and digital assay files to the digital database used in the 2003 resource model	A significant number of discrepancies were discovered, however, none of them were in intervals containing significant mineralization. The majority of issues related to approximate drill collar co-ordinates. Data were considered acceptable for use in Mineral Resource estimation.
2004	Peter Lacroix	Review of assay, collar and downhole survey data, with about 15% of all data spot-checked. All of the data were screened automatically by the modeling software (Medsystem®/Minesight®) for missing intervals and values outside the normal range of data.	Some errors and discrepancies noted with analyses, collars, and downhole surveys. Data were considered acceptable for use in Mineral Resource estimation.
2006	Mike Lechner	Review of assay data for 10 core holes; comparisons of collar elevations from the electronic database with the NovaGold supplied topographic surface;	Collar elevation discrepancies were primarily due to older drill holes having been collared on top of a glacier which has since retreated giving the appearance that the drill hole collar was too high. Data were considered acceptable for use in Mineral Resource estimation.
2008	GCMC	Response to AMEC 2008 audit. All NovaGold era drill logs were visually verified against the database which resolved some differences between scanned logs and the database that were identified by AMEC. The Galore Creek database was migrated into Datashed database software to screen for errors in drill hole depths, gaps between intervals, overlapping intervals, etc. A 100% review of the 2003–2007-era SG data was completed (refer to Section 11.3.6).	Data were considered acceptable for use in Mineral Resource estimation.
2011	GCMC	Review of pre-1990 gold data collected from 100 ft composites	A gold field that contained all gold data (10 ft and remnant 100 ft) was modelled. A gold field that had the ≥0.01 oz/t Au historic data factored down by 37% was also modelled to analyze potential bias effects. Neither of these models showed a material difference from the model constructed using only new data.
		Detailed review of the copper and gold information collected prior to 2007	The check led to the correction of 93 copper records from a total of 90,082. The errors were predominantly typographical errors from the 1960s and 1970s drill holes. From a total of 89,965 gold records, 684 were corrected.
		Full review of the pre-2010 drill hole survey data used in the 2010 Resource Update including 1,017 pre-2008 drill hole records from the 2007 Access database, 11 of the 2008 drill hole records from the NovaGold Datashed database and nine metallurgical/resource definition holes drilled in 2010	No changes were made to the drill hole collar locations; however, it appears the 2007 and 2008 drill hole collars average ~12 m higher in elevation than the topographic surface. Check of the drill hole lengths identified edits to 13 records where the maximum value in the Lithology or Assay field was greater than the collar data. Review of the down hole surveys located 88 records identified with negative codes (-9, -19, -39) and

Year	Review Party	Work Conducted	Findings
			described as "unreliable records"; these records were excluded from the mineral resource estimate database. Drill hole azimuth and down hole survey checks for holes dipping shallower than -80º identified 24 records from nine drill holes which had extreme and erratic (i.e. extreme one direction then returning the other) azimuth or dip deviation of >10º per 100 m. These records were also excluded. A simple validation of the lithology table determined records at the end of drill hole GC72-0271 contained an overlap due to relogging. Duplicate records were removed

Historic logs did not exactly match logged intervals in the resource database, mainly because Kennecott relogged most historic drill holes in 1991 and these re-logs were loaded into the resource database. These Kennecott re-logs were not made available to AMEC. Therefore, the AMEC audit of historic lithological logs was limited to checking for obvious interpretations errors. AMEC found no obvious interpretation errors in the historic logs checked and thus finds the lithology information acceptable for modeling and resource estimation.

Down-hole survey measurements from historic drill holes were loaded into the Galore Creek resource database from digital files acquired from Kennecott. Down-hole surveys were not recorded on most historic drill logs from the 1960s and 1970s. AMEC was therefore not able to audit the down-hole surveys from this time period.

Down-hole surveys from the 1990 campaign matched values in the resource database. Down-hole surveys from the 1991 campaign were found to have discrepancies in azimuth of 1.5 degrees.

AMEC performed an analysis of drill hole deviation based upon vertical drill holes that had been surveyed down-hole. Drill holes were found to deviate, on average, 2.2 m per 100 m of down-hole advance.

12.1.2 Geology and Geological Interpretation Review, NovaGold/SpectrumGold Programs 2003–2006

AMEC checked approximately 7%, or 502 of a total geological 6,747 entries in the drill hole database using scanned versions of the original geology logs as source files. The entire lithology table was checked for contiguous intervals; the error rate, at 0.06% interval data entry errors, is acceptable.

AMEC audited approximately 7% of the entries using 20 drill hole logs from within the resource area. Minor differences between the scanned logs and the database records were noted for one hole. The total error for this review was 1%, which is considered borderline acceptable for advanced studies.

The lithology table was examined for interval errors, overlapping intervals and missing intervals. A total of 12 holes were identified with this error type.

Recommendations arising from the geological review of the database are that NovaGold correct all identified errors, and resolve differences between scanned logs and the database for the identified drill hole.

12.1.3 Drill Collar Review, NovaGold/SpectrumGold Programs 2003–2006

A total of 42 drill holes (10% of the drill holes used in resource estimation) from the 2003–2007 NovaGold/SpectrumGold drilling programs were reviewed. No data errors were found in the transcription of data from original drill logs to the database collar table that would materially affect Mineral Resource estimates.

A comparison of the database with Ashtech survey data noted a 15 m difference in elevation for the 2003 and 2004 drill holes. The difference was attributed to a correction applied by NovaGold to the 2003 and 2004 and pre 1991 collar elevations (Reid, 2007).

12.1.4 Downhole Survey Review, NovaGold/SpectrumGold Programs 2003–2006

Downhole survey records were assessed for 40 drill holes. There were no zero depth survey record errors noted by AMEC.

The entire survey table was checked for interval errors and 34 cases identified where the survey depth exceeds the total length of hole. Survey depths for these holes were recommended to be adjusted to match the hole length.

Review of transcription or manipulation errors between the downhole survey records and the database survey table indicated an error rate of 0.48% for depth entries, 0.16% for azimuth and 0.95% for dip. The error rate, at under 1%, is acceptable.

AMEC audited the downhole survey table for consistent application of the correction factor to account for magnetic declination. No significant issues were noted.

12.1.5 Density Review

Of the 13,638 SG determinations in the database that pertained to the resource areas, 95 are based on laboratory measurements, 1,307 are based on water displacement methods (the sample weight divided by the volume of displaced water), 84 are based on core weight divided by the core volume which was derived from the core length and core diameter and 12,147 values are based on water immersion methods (the sample weight in air divided by the difference between the weight in air and the weight in water). Eight measurements were discarded because of missing weights.

AMEC recalculated 13,538 SG values, representing 99% of the density table records, from source record data and compared the values to the database. Minor differences in calculated SG were noted in 532 records (3.9%). Some unusual density values, ranging from -367.68 to 774.55 were noted, and may be due to transcription errors when recording the weight in air or weight in water values on the original field logs or when entering the data into the database.

A total of 711 values from 2005–2006 drill program source records, or 5.2% of the density table, were checked for typographical errors. AMEC noted 10 (1.4% error rate) depth data entry errors, 13 (1.8%) weight-in-air data entry errors and 15 (2.1%) weight-in-water entry errors. A total of 13 (1.8%) of these errors were regarded as significant errors in the determination of the SG value.

AMEC concluded that the SG error rate was above that considered acceptable for a prefeasibility- or feasibility-level study.

12.1.6 Assay Audit, NovaGold/SpectrumGold Programs 2003–2006

Original electronic CSV (comma separated value) files from ALS Chemex were converted to Excel spreadsheets and then imported into an Access database. About 99% of the samples in the assay table for holes drilled within the defined resource areas were audited. The copper, gold, and silver values from ALS Chemex were matched against database entries, and the following numbers of errors noted: Cu: seven errors (0.01% error rate); Au: five errors (0.01% error rate) and Ag: four errors (0.01% error rate). The results were considered acceptable.

12.1.7 Sample Intervals Audit, NovaGold/SpectrumGold Programs 2003–2006

AMEC checked approximately 40,606 of a total 52,391 records in the sample database against data entry files received from NovaGold. Fields in the data entry files were compared to the values found in the database. A total of 22 records were noted with errors. This resulted in an error rate of 0.05%, and was considered acceptable.

The sample interval table was checked for negative intervals, overlapping intervals and the occurrence of duplicate sample identification numbers. One error was noted.

12.1.8 Standard Reference Materials (SRMs), NovaGold/SpectrumGold Programs 2003–2006

NovaGold used several SRMs during 2004–2006 to monitor analytical results from ALS Chemex. There are a total of 2,709 results from standards found in the database, which corresponded to the one in 20 insertion rate. There were no significant biases noted in any of the copper SRM results; that is, all biases for copper are less than 5%. A potential bias was noted for SRM Std-Pm152 during 2004 and 2005 where the biases for gold are 6.6% and 7.5% respectively. All other gold SRM biases are less than 5%.

12.1.9 Blanks, NovaGold/SpectrumGold Programs 2003–2006

NovaGold inserted a blank sample that tests for contamination of samples, on a rate of approximately one blank for every 20 samples.

NovaGold submitted a total of 2,777 blanks from 2003 through 2006. The blank material performance with respect to copper was considered acceptable. In all years, an average of 31.6% of the copper results were below or equal to 3 ppm, and 95.5% of the results that were above this level were below 100 ppm or 0.01% copper. The average grade at Galore Creek is around 0.52% copper. Blank sample copper values of 2,470 ppm and 7,640 ppm respectively are reported for 2004 and 2005.

The performance of gold contamination in blanks was also considered acceptable. Overall, 98% of the samples returned results below 0.015 ppm, but most years showed some values above this limit. Generally these values were all below 0.09 ppm Au, except during 2005 when three values reported above 0.09 ppm Au, including one sample which reported a value of 0.304 ppm Au.

The silver blank results were more erratic than either the copper or the gold results, except for 2003 where all the results fell below 0.6 ppm Ag. A total of 87% of the silver results fell below the 0.6 ppm limit; 2004, 2005 and 2006 showed a significant number of results above this limit. The maximum values returned for 2004, 2005 and 2006 were 2.3 ppm Ag, 7.9 ppm Ag and 4.8 ppm respectively. In 2006, 5% of the results were above 1 ppm Ag, in 2005, 12% of the results were above 1 ppm Ag and in 2004, 8% of the results were above 1 ppm Ag. The average silver grade reported for Galore Creek is 4.9 g/t (4.9 ppm).

12.1.10 Duplicate Performance, NovaGold/SpectrumGold Programs 2003–2006

Duplicates at Galore were “preparation duplicates”; splits were taken from crushed reject material at a rate of about one in 20 samples.

The precision noted for copper analysis was acceptable. Copper is analysed by two methods, depending on concentration; ME-ICP41 for values less than 10,000 ppm (or <1% Cu) and Cu-AA46 for values greater than 1% Cu.

The precision for gold analysis is low, but is considered acceptable. The precision for silver is low, but is also considered acceptable.

12.1.11 Check Assays NovaGold/SpectrumGold Programs 2003–2006

NovaGold conducted two check assay programs during 2003–2006; results of the programs were reviewed.

The 2006 program consisted of a random selection of approximately 5% of all Galore Creek pulp samples as check assay samples. A total of 699 pulps (4.2% of assays for 2006) were sent for re-assay at Assayers Canada as umpire laboratory, for direct comparison with the results from ALS Chemex, the primary laboratory.

No significant bias was noted for the copper and gold check assays. ALS Chemex silver results were biased 5.4% low relative to Assayers Canada check assays. As silver is of minor economic importance, this bias is not considered to be of material significance.

No background information regarding the sample selection for the 2004 check assay program was provided to AMEC. Check assays were submitted to both Assayers Canada and to ACME Laboratories in Vancouver. ACME provided check assays for copper and gold, while Assayers Canada only provided check assays for copper.

ALS Chemex results were biased low (7.6%) compared to the ACME check assays for gold (595 samples). The ICP copper results (copper results below 1%) indicated that ALS Chemex was biased low (11%) compared to the ACME results for the 473 samples submitted. The copper values greater than 1% copper showed no significant bias for 111 samples submitted. No bias was noted for the 472 copper values below 1% copper, but ALS Chemex was biased 7.7% lower than Assayers Canada for the 123 copper check assays greater than 1% copper.

As these biases were marginal, and as ALS Chemex was biased low with respect to the check laboratory, AMEC considered the biases to be not material to the resource. The results obtained from ALS Chemex were considered conservative, as the bias was consistently low.

12.1.12 Conclusions of 2008 Audit

The QA/QC programs adequately addressed issues of precision, accuracy and contamination. The QA/QC work completed was sufficient to support resource estimation. Although there was a consistent low assay bias in the ALS Chemex data for gold and for copper values under 1% Cu, the biases were considered to be conservative, and would not affect resource estimates.

AMEC noted a number of minor database errors which required rectification, including:

- Resolve differences between scanned logs and the database for drill holes identified in the audit that had discrepancies
- NovaGold have the drill hole collars professionally surveyed by a registered land surveyor, and fix the minor identified errors in the drill collar tables
- Correct the noted transcription or manipulation errors in the downhole survey tables
- Apply the proper magnetic declination correction to the four identified holes
- Adjust the 34 identified drill holes which have survey depths extending past the length of hole
- Review the 35 drill holes that are lacking surveys, and determine if any can be re-entered and surveyed
- Identified assay table errors be rectified
- Identified errors in the sample interval table, although minor, be fixed
- NovaGold should review the supporting data for SRM Std-Pm152 and determine whether the calculated "best value" for this standard is correct, or if any of the laboratories originally used to analyze this standard also reported values similar to the bias noted
- AMEC concurred with the recommendation made in Lechner, (2006) that *"NovaGold more closely track QA/QC results and re-assay all sample batches that are associated with any control samples that are out of tolerance."*

AMEC concluded that the SG error rate was above that considered acceptable for a prefeasibility or feasibility level study. Recommendations were that NovaGold review the SG table in greater detail and make modifications as required, and additionally, develop a more rigorous error checking protocol during data entry. NovaGold was requested to recalculate the SG values in the density table, as this exercise would remove minor errors noted in 532 of the SG values.

12.2 2011 AMEC Audit

12.2.1 Database Review

AMEC was provided with a database for review by GCMC. The database is divided into 14 areas; however, AMEC restricted its audit to those deposits and zones which support the Mineral Resource estimate in the GCMC 2011 pre-feasibility study. These are the Central Zone, South West, West Fork, Junction, North Junction, and Middle Creek areas, and are collectively termed the mineral resource estimate area.

Database verification comprised checking of the drilling prior to 2006, by comparing the data available to the database audited by AMEC in 2008. Drilling completed after 2006 was compared against supporting documents provided by GCMC.

AMEC notes that there were a few unexpected discrepancies identified in the database, which is an indication that the database is changing and that changes to the database are not fully documented.

Clear documentation of the Datashed database should be completed prior to initiation of a feasibility study on the Galore Creek Project. Documentation should include details of a 5% double data entry check all data in the database.

Collar Review

A comparison by AMEC of the 2010 and 2008 collar record tables showed, apart from drill holes added since the previous AMEC audit, all drill holes in the 2010 table had a matching drill hole in the 2008 drill hole table. Of these, only seven holes have different collar positions. Only one of the seven drill holes was within the area of the mineral resource estimate. One drill hole was noted to have a significant collar shift; this change in position is likely a result of modifications made by Teck during their internal audit.

Drill holes completed since the 2008 resource update were compared against the original drill logs. Although not all recent collar positions could be verified with the drill logs, those that could be compared did match. The collar positions reported in the 2010 drill hole database are considered reasonably free of errors and based on field inspection by AMEC in September 2010, are considered reasonably accurate.

Down Hole Survey Review

The 2010 survey table was filtered to show only the holes drilled in the area of the mineral resource estimate prior to 2006 and was then compared to the 2008 survey table. There are 50 records present in the 2008 which are no longer reported in the 2010 table. These intervals may have been removed as part of an ongoing audit process but no documentation is available to verify this. The absence of these surveys is not considered to have a material impact on the estimate.

No down-hole survey records for drill holes added to the database since the previous AMEC audit were available for review.

In the absence of original records for verification the entire down hole survey database was checked for unusual kinks and bends using AMEC-proprietary software. No anomalous deviations were noted that could materially affect Mineral Resource estimation.

Assays

A total of 25 intervals from the pre-2006 drilling in the 2010 assay table do not have matching intervals in the 2008 assay table. There are 83 intervals that have different copper values. There are 441 intervals that have different gold values than previously reported. Of these, only 30 have differences in the gold value that are more than 0.02 ppm. Apart from 90 legacy silver values which have been reset to zero, there is only one silver interval that does not match.

Assay certificates for 725 samples from the drill holes in the 2010 database were compared to the 2010 assay table. There were no differences. Assays for the 2007 and 2008 drilling were not checked by AMEC during the 2011 review, as they were reviewed in detail during the earlier AMEC audit.

The discrepancies in the assay data noted by AMEC would not cause a material impact on the mineral resource estimate.

Specific Gravity

Verification of the 2010 SG database was based on comparing a 2010 Gems SG table with the 2008 SG table. The 2008 and 2010 legacy databases were compared taking into account repeat intervals and after removal from the 2008 SG database of one legacy sample where the SG value was repeated.

The comparison of the 2008 and 2010 tables identified 196 records in 73 holes in the 2008 SG table that had no match in the 2010 table. Several of the 2008 records with no 2010 match have negative or unlikely large or small SG measurements. There are also 136 records matched on Hole ID and Depth that have different SG values. These differences are likely due to changes made during a 100% data entry check of non-legacy SG completed by NovaGold in 2008. No documentation is available to validate this.

The comparison of the 2008 and 2010 tables identified 1,372 records in the 2010 table with no matching depth record in the 2008 table. These differences are likely a result of additional SG measurements from holes completed since the 2008 audit.

In order to assess the quality of the 2010 SG database AMEC repeated the 5% data entry check of the 2010 non-legacy data using the same records used in the 2008 data entry check. The combined data entry error rate for the SG database is 1% which is considered to indicate the database is sufficiently free of errors that there would be no material impact on the resources using the 2010 SG data.

Bulk Density Determinations and Block Model Assignment

AMEC reviewed the application of the adjustment that was used to account for disaggregation and moisture content to arrive at the final bulk density values used in mineral resource estimation.

The SG correction factor applied to account for disaggregation and moisture content is considered by AMEC to be reasonable.

12.2.2 Legacy Data Review

A comprehensive QA/QC program is not evident for much of the pre-2003 data, referred to as “legacy data”. A twin hole drill program or re-sampling program of existing pulps, coarse rejects or drill core, has not been completed.

In order to assess for grade biases in the legacy data, a “near-twin” composite comparison was performed.

Scatter plots, Q-Q plots, and relative difference plots were used to examine for biases in copper, gold, and silver grades between non-legacy and legacy composites. Non-legacy and legacy composite pairs were prepared at separation distances of 100 m, 25 m and 10 m for all composites within the Central Zone and South West, West Fork, Junction, North Junction, and Middle Creek. Non-legacy-legacy composite pairs were also examined in groups based on drill campaign time frames of 1990s, 1970s, and 1960s.

As a result of this review AMEC concluded that:

- Legacy copper and gold assays appear biased low at grades above 0.15% Cu and 0.1 ppm Au
- Legacy copper and gold assays appear biased high at grades below 0.15% Cu and 0.1 ppm Au
- Legacy silver assays show a positive bias at grades above 1 ppm Ag
- The biases interpreted from the AMEC review may, in part, be due to spatial variability (distances > 10 m) and lithological variability (composite pairs across lithological boundaries)
- The copper and gold biases are generally low and are expected to cause an overall underestimation of grade in the Mineral Resource estimate

- The legacy silver assay bias could be quantified and corrections could be developed using regression analysis
- There are some unusually long-length silver assays in the database that should be removed during future studies
- The apparent high bias for legacy silver is a concern but is mitigated by the low overall economic value that silver is likely to contribute to the Project relative to the contributions from copper and gold values.

12.2.3 Site Visit

No issues were identified during the September 2010 site visit that were considered to negatively impact AMEC's ability to support the Mineral Resource estimate.

12.3 Comments on Section 12

AMEC considers that a reasonable level of verification has been completed during the 2008 and 2011 audits, and that no material issues would have been left unidentified from the programs undertaken.

The AMEC QPs, who rely upon this work, have reviewed the appropriate reports, and are of the opinion that the data verification programs undertaken on the data collected from the Project adequately support the geological interpretations, the analytical and database quality, and therefore support the use of the data in Mineral Resource and Mineral Reserve estimation:

- Sample data collected adequately reflect deposit dimensions, true widths of mineralization, and the style of the deposits
- AMEC completed a database audit in 2008. The audit identified some minor errors in the Project database that AMEC recommended correcting, but they were not considered material to the Mineral Resource estimation. However, the SG error rate at the time was above that considered acceptable
- A second audit was completed by AMEC in 2011; results were:
  - The database is sufficiently error free to cause no restrictions on the confidence categories of the resource estimate and are therefore suitable to support resource estimates used in prefeasibility or more advanced studies
  - No correction should currently be applied for legacy copper, gold, and silver results, but a cautionary note explaining the potential low bias for copper and gold and high bias for legacy silver assays and the potential

impact on the Mineral Resources will be presented as a footnote to disclosure of Mineral Resources

- Limitations on the use of legacy assays to support classification of Measured blocks are still warranted due to the lack of supporting quality control samples; estimated blocks supported primarily by legacy assays should be limited to Indicated classification
- There is more than one source for data and original supporting documentation. In some cases the supporting documentation provided to AMEC was not the finalized documentation. The drill hole database has changed since the 2008 AMEC audit. Documentation of changes made to the database since then was not adequately documented at the time of the AMEC review. These should be rectified so that the documentation can support a clear audit trail.
- The data for the Galore Creek Project are in the process of being migrated to a Datashed database. Documentation of this migration is limited, and should be developed.

AMEC recommends:

- Clear documentation of the verification of the Datashed database is needed to support the efficient execution of any future audit. Documentation should include all details of a 5% data entry check for all components of the Datashed database
- Clear documentation identifying the location of, and the type of original supporting documentation should be prepared prior to initiation of more advanced studies
- Estimated blocks supported primarily by legacy assays should be limited to Indicated classification
- No correction should be applied for legacy copper, gold, and silver results
- A cautionary note explaining the potential low bias for copper and gold and high bias for legacy silver assays and the potential impact on the resources should be presented with any disclosure of resources
- Long-length silver assays be excluded from the database that supports Mineral Resource estimation and excluded from use in future model updates
- A legacy pulp or archived core check re-assay program be undertaken that is supported by appropriate quality control measures
- Re-assessment of grade bias be undertaken after completion of the re-assay program

- If no significant bias is evident after the reassay check program is completed GCMC should consider removing limitations on the use of legacy data to support Measured blocks
- A twin-hole drill program could be considered for areas where re-assaying may not be possible due to absence of archived pulps or core.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Over the Project history, a number of metallurgical testwork campaigns have been undertaken. These are summarized in Table 13-1.

Since 2003, the majority of metallurgical testwork was completed at G&T Metallurgical Services laboratories in Kamloops, British Columbia. GCMC representatives were involved in all aspects of the testwork, including sample selection, test design, and data review and interpretation. Periodic laboratory visits were completed in 2010 to verify that tests were being carried out at an acceptable standard, and no issues were identified.

All tests have been documented according to G&T Metallurgical standard practices. Standard tests completed included open circuit rougher-cleaner flotation tests, locked-cycle tests, Bond ball mill work index determinations, SMC hardness determinations, and JK drop-weight test index determinations. All tests were completed using industry-standard methods.

Due to the history of the Galore Creek Project, large amounts of test data were available and were reviewed to determine potential gaps in the metallurgical understanding of the Galore Creek orebody.

The following subsections present additional details of the various test programs conducted.

Table 13-1: Metallurgical Testwork Summary Table

Year	Laboratory	Testwork Performed
1960–1967	Hazen Research, Golden, CO	Pilot plant on bulk sample
1960s	Britton Research Laboratories, Vancouver, BC	Grinding and flotation on low grade samples
1992	Dawson Metallurgical, Salt Lake City, UT	Bench flotation tests
2003	G&T Metallurgical Services, Kamloops, BC	Grinding, flotation, gold gravity recovery
2005	G&T Metallurgical Services, Kamloops, BC	Grinding Bond Work Indices
2006	G&T Metallurgical Services, Kamloops, BC	Flotation and pilot plant
2006	SGS Lakefield and SGS MinnovEX	CEET modelling (grinding)
2008	G&T Metallurgical Services, Kamloops, BC	Aging tests on slurry
2009	G&T Metallurgical Services, Kamloops, BC	Locked-cycle flotation
2010	G&T Metallurgical Services, Kamloops, BC	Grinding and locked-cycle flotation

13.1 Hazen Research and Britton Research Ltd, 1960s

The initial Galore Creek metallurgical work was at the request of Kennecott Corporation in the 1960s and focused on the Central Zone. The testwork was carried out by Hazen Research and commenced with initial bench tests on drill core samples.

The testing started in 1962, and culminated in 1967 with a 50 ton pilot plant milling test using a bulk sample taken from an adit in the Central Zone. The bulk head sample assayed 1.28% Cu. The optimum grind for rougher flotation was established at 25% +100 mesh (50% -200 mesh). The optimum grind for concentrates was established at 95% -325 mesh.

Britton Research Ltd. conducted testing of lower-grade material and estimated recoveries of 83% and 81% for grades of 0.54% Cu and 0.31% Cu respectively. Kennecott developed a flowsheet, equipment recommendations, and operating cost estimates for a 20,000 t/d concentrator. The flowsheet anticipated a relatively coarse primary grind (25% plus 100 mesh) followed by regrinding the rougher concentrate to 97% passing -200 mesh to obtain suitable concentrate grades.

13.2 Dawson Metallurgical Laboratories, 1992

In 1992, bench flotation tests were carried out on drill core from five 1991 holes by Dawson Metallurgical Laboratories in Salt Lake City. The object of this study was to determine the amenability of the composites to a standard flowsheet developed for Kennecott and to determine if gold recovery could be significantly improved. The study used four composites from the Southwest Zone and two from the Central Zone. It was found that both gold recoveries and copper concentrate grades for the Southwest Zone were lower than those indicated for the Central Zone. This was attributed to the higher pyrite content in the Southwest Zone and the association of at least part of the gold with pyrite.

Overall copper and gold recoveries in a copper concentrate grading 25% Cu were estimated to average approximately 90.3% and 58%, respectively based on constant tail grades of 0.065% Cu. Concentrator tail grades for Au also tended to remain fairly constant at 0.137 g/t Au for the Central Zone and 0.274 g/t Au for the Southwest Zone. Gold recovery was projected based on head assay and rougher tail residue. A nugget effect was observed in tests from many of the higher-grade composites. Gold recoveries were not optimized as part of these studies. It was also reported that several composites were not upgraded to 25% Cu in concentrate after two stages of cleaning with regrinding.

Copper recovery was slightly lower than the 1965–1967 test results; however, recovery at a modestly finer grind was in line with the earlier work. Several composites were not upgraded to the 25% copper grade obtained in the pilot plant; this was attributed to the presence of “talc” which was not observed in the earlier samples.

13.3 G&T Metallurgical Services, 2003

The 2003 testwork was undertaken to assess the metallurgy of one Southwest zone and three Central Zone samples in a standard rougher and cleaner circuit. The program included a preliminary evaluation of the effect of grind on metallurgical performance and the potential for gold recovery by gravity concentration.

Grinding tests indicated that approximately 60% of the copper sulphide was liberated at a P_{80} of 150 µm. The majority of the non-liberated copper sulphides were locked with non-sulphide gangue. The data suggested that good rougher copper recovery should be achieved at a relatively coarse grind and regrinding would be required to maximize the concentrate grade during cleaner flotation. Gold occurred as liberated fine-grained particles and its co-recovery in copper concentrate was typical of porphyry copper deposits.

The preliminary batch flotation results were consistent with the 1966 pilot plant studies undertaken by Kennecott 96% copper recovery and a concentrate grade of 25% copper were achieved.

Conclusions from the 2003 testwork included:

- A primary grind at a P_{80} of 150 µm nominal is sufficient for copper mineral and gold liberation. Pyrite liberation is high at this grind
- Rougher flotation of copper and gold was fast and with high recovery from all four samples. Copper recovery ranged from 95% to 99% and gold recovery ranged from 81% to 88% within 5 minutes using simple flotation schemes and standard reagents
- A fraction of the floatable gold is fine-grained and free, despite the relatively coarse grind, and floats with the copper sulphides
- The cleaner concentrates are relatively clean. Selenium appeared to be the only impurity of concern.

The potential for gravity concentration of gold was assessed on each sample in a laboratory-scale Knelson concentrator. Some of the gold appeared to be recoverable by gravity. More work was recommended to determine the merits of this operation compared with direct flotation after primary grinding since gold is readily recovered by flotation with the copper sulphides.

13.4 G&T Metallurgical Services, 2005

Four Bond Work Index determinations were conducted by G&T on Minnovex reject samples completed earlier in 2005. The resulting BWis ranged from 14.3 to 17.3 kWh/t.

13.5 G&T Metallurgical Services, SGS Lakefield, SGS MinnovEX, 2006

The 2005–2006 metallurgical test program was managed by Hatch and carried out by G&T Metallurgical Services Ltd (Kamloops, BC). G&T Metallurgical Services determined the Bond Ball Mill Work Index and conducted the flotation testwork on the composites used in the flotation program, while SGS Lakefield and SGS Minnovex (Toronto, ON) ran additional grindability and flotation simulation tests.

A comprehensive metallurgical program was 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 were developed to project copper, gold and silver recoveries in mining blocks for each pit. Pilot plant campaigns were also completed, primarily to generate concentrate samples for dewatering tests and marketing purposes, and tailings samples for dewatering tests and environmental purposes.

At a grind of 80% passing 150 µm, 50% to 60% of copper sulphides and the majority of gold particles were liberated and recoverable by flotation. The gold particles were fine at nominally 8 to 12 µm and would be unlikely to be recovered by gravity concentration. A primary grind of 80% passing 200 µm was suggested to achieve the same metals recovery. The metallurgical response deteriorated as the grind approached 300 µm.

Mineralization hardness, in terms of Bond Ball Mill Work Index, varied between 13 kWh/t and 21 kWh/t over the various proposed pits. The average hardness in the dominant Central Pit was 16.5 kWh/t, similar to that determined from the 2003 metallurgical testwork.

The hardness, measured as SAG power index (SPI), ranged from 20 minutes to 141 minutes across the deposit. The MinnovEX CEET model indicated that any proposed mill circuit would be SAG mill-limiting when treating mineralization with SPI greater than 115 minutes. The “stick” rock was found to be generally harder and more abrasive than the "broken" rock (refer to Section 10.4 for a description of these rock types).

The proposed flowsheet design consisted of rougher flotation, regrind of rougher concentrate, and three stages of cleaner flotation using a simple reagent scheme that utilized PAX as the primary collector and MIBC as the frother. The use of 3418A, a more selective dithiophosphinate collector, instead of PAX, was suggested to produce slightly higher concentrate grade at similar recovery. A guar gum carboxymethyl cellulose reagent was noted to be required to disperse talc-like materials and minimize their adverse impact on flotation responses. Variable amounts and occurrences of these talc-like materials were observed in the drill cores from across the deposit. The talc-like materials were not identified. The program also verified that chalcopyrite and bornite materials from various mineralization zones have similar metallurgical responses.

Models were developed for each deposit to project copper recovery from head grades at constant concentrate grade and to project gold and silver recoveries from copper recovery for use in mining blocks. Using a head grade of 0.7% copper for each deposit, the projected recoveries were as follows:

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

A model was also developed for projecting copper recovery from mineralization containing non-sulphide copper. Copper recovery was expected to be lower and to vary with the proportion of non-sulphide copper content, whereas the gold and silver recoveries were expected to correlate with copper recovery. Using a 0.7% total copper head and assuming 20% of the total copper occurring as a non-sulphide, the model projected recoveries of 71% copper, 55% gold and 51% silver at a 28% Cu concentrate grade. Since gold and silver recoveries largely followed copper recovery, the gold and silver in mineralization with very low copper grades, and largely occurring within pyrite grains, may not be recovered.

A preliminary flotation model indicated that the concentrate grade might improve at the same recovery if flotation columns were used for final cleaning in place of mechanical cells. Further work was recommended on this option.

The final concentrates had relatively low penalty elements. Fluorine, selenium, lead and zinc concentrations were variable and might have the potential to be of concern. It was recommended that further work be conducted to address a number of key issues and increase confidence in the projected metallurgical performance of the mineralization from each pit given the variable mineralization, head grades and observed metallurgy. The work should be conducted on fresh drill core samples, in particular, to better define and quantify the occurrences and spatial distributions of talc-like minerals and pyrite, non-sulphide copper, the penalty elements and the extent of their impact on metallurgy, and to determine how the recovery of lead and zinc into the concentrate may be minimized.

13.6 G&T Metallurgical Services, 2008

In 2008, G&T conducted testwork to investigate the effect of aging on metallurgical performance. The principal objective of this study was to simulate the effect of transporting ground slurry in a pipeline for seven hours prior to flotation processing in the rougher bank. The test procedure was to be conducted on two composites identified as CRZ Zone Stick and CRZ Zone Broken from the Galore Creek deposits. These composites were prepared from samples stored at the laboratory since mid-2006. The testing process involved grinding the samples to 140 µm K_{80} for the CRZ Stick and 185 µm K_{80} for the CRZ Broken composites. The mill discharge slurry was allowed to age with occasional stirring for a period of 7 hours. Following this aging period, flotation proceeded to produce four timed rougher concentrates.

The results of these tests were then compared to baseline tests conducted on each sample under near identical conditions. The results indicated under the condition tested no perceived metallurgical disadvantage in copper flotation kinetics or gold recovery.

13.7 G&T Metallurgical Services, 2009

Locked-cycle testing carried out on six samples from the CRZ and NGL zones validated that metallurgical performance was achieved on all the samples tested. Copper recoveries ranged from 84% to 94% at grades ranging between 27% and 33% copper in the concentrates. Associated gold recoveries to the concentrate ranged from 46% to 78% with gold content in the copper concentrate ranging between 4.2 and 58 g/t.

In order to achieve these results, non-standard conditions were required for 3 met samples from the CRZ zone. The concentrates produced using the standard flowsheet and test conditions contained less than 20% copper.

Modal analyses carried out on these low grade concentrates revealed that they were contaminated with either liberated non-sulphide gangue or pyrite. To reject these diluents the test conditions were modified. These modifications resulted in production of acceptable copper grades and recoveries.

A three-day pilot plant campaign was carried out on two pilot plant feed samples: Chalcopyrite and chalcopyrite-bornite material. The main purpose of this work was to generate flotation products (mainly tailings) for environmental testing. Metallurgical performance was also measured during each pilot plant run.

The average metallurgical performance for the chalcopyrite only feed sample was about 90% copper recovery into a copper concentrate grading 30% by weight copper. On average, about 77% of the gold in the feed was recovered into the copper concentrate. The average gold content in the copper concentrate was about 24 g/t.

For the chalcopyrite-bornite feed sample, the average copper recovery was 95 percent into a copper concentrate assaying about 41% copper. About 85% of the feed gold was recovered to the copper concentrate. The average gold content in the copper concentrate was also about 24 g/t gold.

The samples tested in this program did not explore the effect on metallurgical performance resulting from processing material containing less than 0.4% copper.

13.8 G&T Metallurgical Services, 2010

Grinding and flotation testwork was conducted by G&T Metallurgical Services in 2010. Full core from six diamond drill holes were used as feed stock to this test program.

Fifty-five discrete samples were generated for material hardness testing and 59 samples for flotation testing. The samples that were generated for flotation testing ranged in copper feed grade form about 0.15% up to greater than 2.0%. The gold feed grades in the flotation composites ranged from near zero to about 1.25 g/t.

Material hardness testing included JKTech Drop Weight and SMC tests, along with Bond ball mill work index testing. The Axb parameter value, a measure of resistance to impact breakage in the SAG mill ranged from about 28 to 236. The lower the Axb value the more resistance to impact breakage in the SAG mill. The samples tested in this program ranged from very hard to very soft but, on average, were moderately soft.

The Bond ball mill work index, a measure of resistance to breakage in the ball mill, ranged from about 13 to 20 kWh/t and averaged 15 kWh/t. This range of values of the

Bond ball mill work index indicates that these material samples range from moderate hardness to hard with respect to breakage in a ball mill.

A single open circuit batch cleaner test was carried out on each of the 59 flotation samples. Feed copper recovery, to the final concentrate, ranged from about 25% to 98%. The copper grade in the copper concentrate ranged from about 10% up to 40%.

Locked-cycle tests were carried out on 4 composite samples. The copper feed grades in these samples ranged from 0.13% to 0.80% copper. Metallurgical performance was variable across the four composites with copper recoveries ranging from about 77% to 92%. The copper grades in the final concentrate ranged from 17% to 37% copper.

Additional open circuit flotation tests, using modified conditions were carried out on 2 composites. In these tests the rougher circuit pH was increased and PE26, a non-sulphide gangue depressant, were utilized. Under the modified conditions, these samples had acceptable metallurgical performance and were comparable to typical response for Galore Creek materials.

Minor element determinations were carried out on the final copper concentrate produced from one test. The zinc and cadmium levels were elevated in the concentrate produced from this sample. There was not enough concentrate to carry out minor element determinations on the other three locked-cycle test concentrates.

Almost all the samples from drill hole 799 produced lower copper grades in the final concentrate, averaging 16% copper in batch open circuit cleaning tests. The reason for lower concentrate grade, for Composites F799-50, was identified as contamination with liberated pyrite and non-sulphide gangue. It is not known if this is the common cause of lower final copper concentrate grades for the remaining samples in that drill hole.

Samples of the solid and liquid phase from the exit streams from four locked-cycle tests were submitted for environmental testing.

13.9 G&T Metallurgical Services, 2011

A set of 11 locked-cycle tests was performed using intervals from the remaining stock of Galore Creek samples. GCMC requested that AMEC include the results of the tests in the recovery estimation for the Project.

Upon examination, AMEC concluded that the samples used to generate the new results are unrepresentative of any category of ore type, and are very likely to be biased toward higher recoveries. The new results were not used to estimate metal recoveries.

13.10 Throughput Calculations

Hardness values generated from samples obtained from a metallurgical test program completed in 2010 were combined with historical hardness values to define hardness variability. Throughput estimations were generated using an Excel-based engineering throughput model developed from multiple JKSimMet grinding circuit simulations. This model can be used to calculate the mill throughput of the grinding circuit at various ore hardnesses and feed sizes.

Two ore fracture models were used to estimate SAG mill feed size. For broken ore (near surface sheet fractured ore), an 80% passing feed size of 57 mm was assumed. Although finer than normal SAG feed sizes, this value was deemed conservative based on on-site estimates of the broken ore. Due to the extremely fractured and friable nature of this ore, it was expected that the ROM ore size will be similar to SAG feed size. For the more competent stick ore, a typical 60” crusher discharge size distribution was chosen to represent the SAG mill feed size. The 80% passing size for stick ore SAG feed was 150 mm.

Multiple mineral alteration shells were generated in order to delineate the most appropriate geometallurgical category for the resource blocks within the geological block model. These shells, coupled with a defined boundary between competent and sheet fractured ore, resulted in each model block being assigned one of eight throughput values (Table 13-2).

The values in Table 13-2 were utilized as part of the mine production optimization analysis. In this analysis, physical mill capacity was limited to 110,000 t/d to represent expected non-power-related limitations within the processing circuit, such as pumping capacity and conveyor capacity.

Each block was assigned a defined number of mill operating hours to process: these hours were inversely proportional to the throughput capacity of the ore. As the mine plan was developed, cumulative mill hours were tallied until a full year of mill operational time was reached. The sum mass of these blocks equalled the calculated mill throughput for the year.

Table 13-2: Throughput Rates Assumed Based on Geometallurgical Types

Structure	Geometallurgical Ore Type, Mineral Alteration-Based	dmt/h	dmt/h at 92% availability	dmt/day at 92% availability
Competent (stick) F80 = 150 mm	Low garnet, low orthoclase	3321	3055	73328
	High garnet, low orthoclase	3536	3253	78075
	High garnet, high orthoclase	3438	3163	75911
	Low garnet, low orthoclase	3162	2909	69817
Sheet-fractured (broken) F80 = 57 mm	Low garnet, low orthoclase	5517	5076	121815
	High garnet, low orthoclase	5871	5401	129632
	High garnet, high orthoclase	5709	5252	126055
	Low garnet, low orthoclase	5101	4693	112630

13.11 Deleterious Elements

During the various metallurgical testwork programs, the presence of potential deleterious elements to the process route was noted. These are summarized in Table 13-3. The only element that is considered to be above penalty levels in the final concentrates is fluorine.

Table 13-3: Deleterious Elements

Year of Testwork	Laboratory	Comments
1992	Dawson Metallurgical Laboratories	Several composites were not upgraded to the 25% copper grade obtained in the pilot plant; this was attributed to the presence of "talc" which was not observed in the earlier samples
2003	G&T Metallurgical Services	The cleaner concentrates are relatively clean. Selenium appeared to be the only impurity of concern
2006	G&T Metallurgical Services, SGS Lakefield, SGS MinnovEX	A guar gum carboxymethyl cellulose reagent was noted to be required to disperse talc-like materials and minimize their adverse impact on flotation responses. Variable amounts and occurrences of these talc-like materials were observed in the drill cores from across the deposit. The talc-like materials were not identified. The final concentrates had relatively low penalty elements. Fluorine, selenium, lead and zinc concentrations were variable and might have the potential to be of concern. Further work should be conducted on fresh drill core samples, in particular, to better define and quantify the occurrences and spatial distributions of talc-like minerals and pyrite, non-sulphide copper, the penalty elements and the extent of their impact on metallurgy
2007	AMEC review of testwork conducted to 2007	In 2006, four concentrate batches were subject to multi-element analysis. Within three batches, fluorine returned low-level values, and one sample had a high fluorine analysis, which may be an analytical error. The fluorine level in the three batches noted was only just at a typical penalty level, and well below the reject level.
2010	G&T Metallurgical Services,	Minor element determinations were carried out on the final copper concentrate produced from one test. The zinc and cadmium levels were elevated in the concentrate produced from this sample. There was not enough concentrate to carry out minor element determinations on the other three locked-cycle test concentrates

13.12 Recovery

Using results of flotation tests conducted during three campaigns in 2005–2006, 2008–2009 and 2010, empirical relationships to estimate recoveries for copper, silver, and gold were derived as a function of head grade. Separate models were prepared for material types defined as Standard or Oxidized/Near Surface material consistent with the geological block model.

The final models for the recovery relationships are shown in Table 13-4.

The subsections which follow discuss how the models were generated.

Table 13-4: Process Recovery Relationship Models

Recovery (%)	Standard Material	Oxidized/Near Surface Material
Copper	7.66*Ln([Head Cu(%)])+94.34 *(cap at 95%)*	([OxRConc]*([Head Cu(%)]-0.18)/([Head Cu(%)]*([OxRConc]-0.18))*94.8 where: ([OxRConc] =7.2*[Head Cu(%)]+1.6 *(cap at 95%)*
Gold	8.1*Ln([HeadAu(g/t)])+78 *(cap at 90%)*	8.1*Ln [HeadAu(g/t)]+78 *(cap at 90%)*
Silver	19.7* Ln([HeadAg(g/t)])+26 *(cap at 90%)*	14.5* Ln([HeadAg(g/t)])+28 *(cap at 75%)*

13.12.1 Copper Recovery Estimate

Standard and Oxidized Material

Results of flotation testwork indicated a significant difference between the metallurgical response of standard and oxidized material. As such, separate recovery estimates were developed for each type.

Standard Material Estimate

The recovery was estimated by regression analysis of locked-cycle test results. The resulting relationship was:

*Recovery = 7.66*Ln([Head Cu(%)])+94.34*

The recovery estimate was reduced by 1% to accommodate for scale-up from laboratory work to plant scale.

The recovery was capped at 95%, equivalent to a flotation test recovery of 96%, since the uncapped relationship rapidly exceeds 100% at higher head grades and the maximum test recovery achieved was in the order of 96%.

Oxidized/Near-Surface Material Estimate

The copper recovery relationship for oxidized material is based on a combination of utilization of a median tailings grade value of all available tests, average of available cleaner recovery tests and rougher flotation recovery relationship to head grade. Correlation using open-circuit and two locked-cycle flotation results was poor.

The algorithm for copper recovery from oxide material is:

Recovery = ([OxRConc]([Head Cu(%)]-0.18)/([Head Cu(%)]*([OxRConc]-0.18))*94.8*

where: ([OxRConc] =7.2*[Head Cu(%)]+1.6

13.12.2 Gold Recovery Estimate

Standard and Oxidized Material

A single recovery estimate was developed as the difference between the metallurgical flotation response of standard and oxidized material appeared to be insignificant in light of the general variability of the results.

Estimate

Gold recovery shows general trends with copper head grade, silver head grade, copper recovery and gold head grade. Since gold head grade also has trends with the other parameters, gold head grade was taken to be the performance indicator; the other relationships were assumed to be spurious. Regression analysis generated the following model:

*Gold Recovery = 8.1*Ln([HeadAu(g/t)])+78*

A cap of 90% is applied since no test result exceeded that value.

13.12.3 Silver Recovery Estimate

Standard and Oxidized Material

Analysis of the flotation results indicated a significant difference between the metallurgical response of standard and oxidized material. Separate recovery estimates were developed for each type.

Standard Material Estimate

As with gold recovery, there were presumably spurious relationships with other parameters, but the silver head grade was taken as the performance indicator. The model was derived from regression analysis of the locked-cycle test results, and is:

*Silver Recovery = 19.7*Ln([Head Ag (g/t)])+26*

A cap of 90% is applied.

Oxidized Material Estimate

There were only two locked-cycle tests for oxidized material. However, there was a reasonable trend for the open-circuit tests and the silver recovery was estimated by regression analysis of the entire oxidized material suite. It was:

*Silver Recovery = 14.5*Ln ([HeadAg(g/t)])+28*

A cap of 75% is applied.

13.13 Comment on Section 13

In the opinion of the AMEC QPs, the following conclusions are appropriate:

- Metallurgical testwork and associated analytical procedures were performed by recognized testing facilities, and the tests performed were appropriate to the mineralization type
- Samples selected for testing were representative of the various types and styles of mineralization at Galore Creek. Samples were selected from a range of depths within the deposit. Sufficient samples were taken so that tests were performed on sufficient sample mass
- Only two locked-cycle tests were performed on oxide ore; however the oxide ore model was based on these plus another 15 open-circuit tests. The samples were from the oxidized ore found in all of the drill holes used for metallurgical testing
- Testwork has established the most appropriate grind size for plant design. An 80% passing feed size of 57 mm is planned for broken ore, whereas for stick ore, an 80% passing size of 150 mm is assumed
- Assumed life-of-mine copper, gold, and silver recovery assumptions are based on appropriate testwork, and the copper, gold and silver recoveries average 90.6%, 73.1% and 64.5% respectively over the life-of-mine

- Various elements have been mentioned throughout the testwork that might have the potential for concern for concentrate quality including selenium in 2003, fluorine, selenium, lead, and zinc in 2006, and zinc and cadmium in 2010. In 2006, four concentrate batches were subject to multi-element analysis. Within three batches, fluorine returned low-level values, and one sample had a high fluorine analysis, which may be an analytical error. The fluorine level in the three batches noted was only just at a typical penalty level, and well below the reject level. Fluorine has been included as a penalty element in the financial analysis considerations. There were no other elements noted in the sampling that would cause penalties to be levied against Galore Creek concentrates
- No other processing factors were identified from the metallurgical testwork that would have a significant effect on extraction.

AMEC comments that:

- The variability in results was high, perhaps in part due to the shortness of the core intervals from which the variability samples were taken (mostly 2 m, but variable and up to 10 m)
- Over the three test campaigns, the primary grind size, the regrind product size, and flotation times varied more than would be achieved in a single, controlled, experimental campaign. The test suites included material from zones no longer expected to be delivered to the mill. However, in general, the tests indicate that on the whole very good metallurgical response is to be expected
- A simple effective flowsheet has been developed, and the zone of mining activity has been more closely defined
- It is recommended that at the next stage of development of the Project a metallurgical drilling and test program be undertaken to define the metallurgical response more accurately. The economic shell should be evenly represented spatially, the sample interval length should be consistent (preferably bench height, approximately 15 m) and tight control of laboratory procedure should be followed. The sample intervals should be logged geologically while in the core boxes. It is reasonable to expect that better overall flotation results would be achieved with such a program
- The occurrence and potential mitigation of fluorine and hydrophobic gangue minerals in particular should be further defined.

14.0 MINERAL RESOURCE ESTIMATES

14.1 GCMC Mineral Resource Estimate

14.1.1 Basis of Estimate

Mineral Resources are based on a total of 673 drill holes, and 62,141 assay results, collected between 1961 and 2011. The database supporting estimation was closed as at September 5, 2010.

The mineral estimate was prepared by GCMC and audited by AMEC.

Composites and 3D solid models were constructed utilizing GEMCOM GEMS™ commercial mine modelling software. The models extend a total of 6,000 m in both the north–south and east–west directions and a total of 1,905 m in the vertical direction. Six zones were modelled: the Central, Southwest, Junction, North Junction, Southwest, and West Fork.

Grade estimations for copper, gold and silver were completed utilizing ordinary kriging (OK) methods. An inverse distance to the second power (ID2) and nearest-neighbour (NN) models were constructed as checks. The block size was selected at 25 m x 25 m in plan and 15 m vertically, and used for both the kriged and ID2 models. For the NN model, the block size was 25 m x 25 m in plan and 5 m vertically.

The coordinate system used for resource modelling is UTM NAD 83, Zone 9. Resource estimation models use the 2004 topography lowered by 15 m.

The Mineral Resource estimates were prepared with reference to the Canadian Institute of Mining Metallurgy and Petroleum (CIM) Definition Standards (2010) and CIM Best Practice Guidelines.

14.1.2 Geological Models

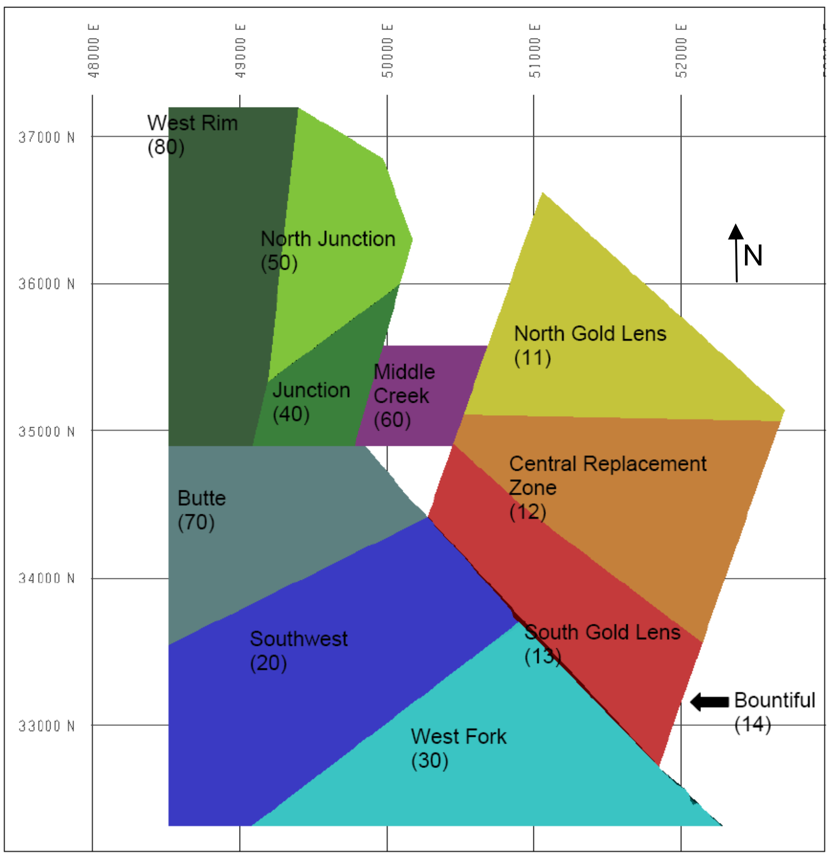
The Project data are grouped in 11 areas. Mineral Resources are estimated for nine of these areas. Grade domains were prepared in five of the areas and lithology domains in four areas.

Grade shell models were constructed in order to constrain mineralization. A nominal 0.2% Cu cut-off was used for Middle Creek and North Junction, 0.15% Cu cut-off at Junction and 0.60% Cu cut-off in Southwest, and at a nominal 0.25% CuEq in West Fork. The Opulent domain in West Fork was constructed based on grade, magnetite abundance and a surficial ground magnetics anomaly.

In the Central Zone (North Gold Lens, Central Replacement Zone, South Gold Lens, and Bountiful Zone), lithology domains were prepared by grouping genetically and statistically similar rock types into four lithology groups: volcanic, mineralized intrusive, intrusive material, and breccia. No grade shells were used in the Central Zone.

Figure 14-1 shows the locations of the various areas estimated in 2010. No mineral resources were estimated in areas 70 and 80.

Figure 14-1: Galore Creek Mineral Resource Estimation Areas



Note: Figure courtesy GCMC, NovaGold and Teck

14.1.3 Exploratory Data Analysis

Exploratory data analysis (EDA), in the form of histograms, probability plots and contact analysis) was performed on uncapped composites for copper, gold and silver to determine suitable geological constraints to mineralization.

14.1.4 Density Assignment

Average stick rock-specific gravity values for each estimation domain adjusted to account for disaggregation and moisture content were loaded to the block model. Average specific gravity values for the stick rock were reduced by 9.3% to account for disaggregation in the broken material. Average specific gravity values for the stick rock were reduced by 0.5% to account for moisture content in the non-broken material.

The SG block model was assigned weighted bulk density "SG" values based on the average adjusted specific gravity values, as well as the percentage of the block in air (AIR%), overburden (OVB%), broken bedrock (BRK%) and non-broken bedrock (STK%). For each Area + Rock Type ± Minzone ± Lithology, the block model calculation was:

*SG = ((0 * AIR%) + (2 * OVB%) + (SG_Broken * BRK%) + (SG_Stick * STK%)) / XTRA1*

Where: XTRA1 = AIR% + OVB% + BRK% + STK%

14.1.5 Acid-Soluble Copper Assignment

During the 2010, 2008, and 2007 drilling programs no additional acid-soluble copper grades were analysed. The most recent acid-soluble copper grades were assayed in 2006, and were used to construct a 3D block model (Francis, 2006), described below. Since no additional information was available in 2010, the copper solubility ratio from the 2006 model was used to tag blocks as either "oxidized ore" or "non-oxidized ore". An oxidized ore recovery model was applied to the blocks tagged as "oxidized ore"; these blocks had a soluble copper ratio of >0.01% acid-soluble copper (>1% of available copper was soluble).

During 2006, Francis (2006) estimated acid-soluble copper grades so that an acid-soluble copper ratio (i.e., acid-soluble copper/total copper (paired data)) 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 to the third power weighting methods (ID3) using 5 m-long assay composites.

To preserve the trend of decreasing acid-soluble copper grade with depth, the block model and composites were coded into 15 m bins below bedrock, accomplished by repeatedly translating the bedrock surface in 15 m 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.

14.1.6 Composites

Composite samples were generated down hole in nominal 5 m lengths, generating 37,944 copper, 34,186, gold, and 34,086 silver composite samples within the resource estimation areas. Small intervals (<1 m) were merged to the previous interval of the same material at the end of drill holes. The fewer gold and silver composite samples result from not every historic rock sample having each metal assayed.

Composites were not broken at geological or grade shell boundaries. This is considered reasonable given the broad nature of mineralization and the proposed large-scale open-pit mining operation. Each composite was tagged with lithology or grade domain identifier.

14.1.7 Grade Capping

Lognormal probability plots of composite grades inside and outside of grade shells, and by lithology type were examined visually by GCMC. Most populations exhibit a lognormal grade distribution, and caps on Cu, Au, and Ag composites were placed where significant deviation occurred. Capping is summarized in Table 14-1. In addition to the capping, outlier restrictions are applied during Pass 2 to restrict the range of influence of some copper and gold values (Table 14-2).

14.1.8 Variography

The copper, gold, and silver 5 m composite grade variograms were computed using Snowden Supervisor software, and were constrained by grade shell or rock type group.

Variograms were computed and variogram modelling was completed using traditional variogram models. The nugget effect was measured using down-hole variograms and set in the directional variogram models. Two spherical structures were fitted.

14.1.9 Estimation

A two pass modelling approach was adopted to estimate blocks inside and outside of grade domains in Junction, Middle Creek, North Junction, Southwest, and West Fork. Within the Central Zone, estimation was constrained by lithology domain.

Hard contacts were used to constrain the use of composite samples to their respective domains. Blocks with multiple estimation domains (e.g., blocks straddling lithology or grade domain contacts) were assigned by majority rock type.

Table 14-1: 2010 Outlier Restrictions

Area	Element	Capping Restriction	Range (m) X,Y,Z	Domain
Butte	Cu	>1.10%	100, 100, 45	Volcanic
Central Zone	Au	>3.0 g/t	85, 85, 35	Volcanic
Central Zone	Au	>1.5 g/t	50, 50, 25	I9 Intrusive
Middle Creek	Au	>0.25 g/t	100, 60, 30	Outside Mineralized Zone
Middle Creek	Cu	>0.25%	100, 80, 45	Outside Mineralized Zone
Southwest	Au	>1.5 g/t	50, 50, 25	Outside Mineralized Zone

Table 14-2: 2010 Capping Thresholds

Area	Grade Shell or Rock Type	Cu (%)	Au (g/t)	Ag (g/t)
Butte	Volcanic (200)	None	None	23.0
	Intrusive (340+350)	0.5	None	10.0
Central	Volcanic (200)	None	12.0	50.0
	Mineralized intrusive (340)	3.0	1.7	26.0
	Mineralized intrusive (361)	None	None	None
	Intrusive (350)	2.6	2.7	26.0
	Intrusive (362,372,380,500)	None	12.0	50.0
	Breccia (400)	None	None	None
Junction	Inside of grade shell (53)	2.5	0.7	8.0
	Outside of grade shell	0.3	0.2	4.0
Middle Creek	Inside of grade shell (61)	2.3	None	24.0
	Outside of grade shell	0.5	1.0	5.0
North Junction	Inside of grade shell (51)	7.0	None	50.0
	Outside of grade shell	1.2	1.5	9.0
Southwest	Inside of grade shell (22)	None	10.0	21.0
	Outside of grade shell	2.3	7.5	21.0
West Fork	Hangingwall (31)	2.3	2.5	None
	Footwall (32)	2.3	2.5	25.0
	Upper Minzone (33)	3.0	2.0	None
	Opulent (34)	23.4	6.0	150.0
	Outside of grade shell	0.4	1.5	20.0

A minimum number of six composites and maximum number of 12 composites with a maximum number of selected composites per drill hole of five was used during Pass 1. A minimum number of three composites and maximum number of 12 composites with a maximum number of selected composites per drill hole of five was used during Pass 2.

Search ellipsoid ranges were determined using the following methodology:

- Pass 1: Ranges selected based on approximately 85% of the variogram sill
- Pass 2: Ranges selected based on approximately 95% of the variogram sill and/or inspection of sample spacing.

Grades were kriged using batch files.

14.1.10 Block Model Validation

Swath plots, comparing kriged copper, gold, and silver to inverse distance squared and nearest neighbour estimates, indicate that in general the variables were in agreement, and no major spatial bias was observed. Differences do occur between the raw composites and the nearest neighbour and kriged grade estimates. The composites are not declustered or constrained by outlier restrictions, and differences between them, as well as the kriged and nearest neighbour estimates, were considered to be acceptable by GCMC.

Histograms for the OK, ID2 and NN models copper, gold and silver. indicate the OK estimate compares well with the inverse distance to a power model and reasonably well with the nearest neighbour model. The OK estimated grades produced distributions that are smoother than the nearest neighbour model. The amount of smoothing was adjusted to match the selective mining unit (SMU) and produce a model that was appropriate for mine planning. The average OK estimated grades compared well with the average nearest neighbour grades, except in domains with small numbers of blocks such as the Opulent Zone or low-grade domains such as the Central Zone breccia domain. The relatively small tonnage associated with the Opulent Zone was considered to minimize the impact of the bias.

The co-efficient of variation (CV) for the kriged estimates was significantly lower than the CV for the 5 m composites or nearest neighbour models. In GCMC's opinion, this was expected given the grade estimation method (ordinary kriging) and the amount of change-of-support reduction that was found to be appropriate for the proposed SMU size.

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 (1978). Herco validation was performed for blocks that were estimated by passes one and two for both copper and gold. In general, in GCMC's opinion, the Herco-adjusted grade-proportion curves matched the kriged grade-proportion curves reasonably well, indicating to GCMC that the appropriate amount of smoothing was achieved via kriging.

14.2 AMEC Review of GCMC 2011 PFS Estimate

14.2.1 EDA Checks

Descriptive statistics, histograms, cumulative probability plots and contact plots were completed by AMEC for Cu, Au, and Ag by area and grade or lithology domain. Results obtained were used to assess the construction of the block model and the estimation plans. Data analyses were conducted on composited assay data (5 m down-hole composites).

As a result of the EDA review AMEC concluded:

- Copper and gold co-efficients of variation (CVs) are higher than expected for a typical porphyry. This may reflect the complex nature of an alkalic porphyry or may indicate a need for more carefully-prepared domains
- The composites used in the estimation of grades in the 2010 Mineral Resource estimate are appropriately prepared and suitable for use in the resource estimate. An exception is the inclusion of long-length pre-1980 silver assays
- The capping thresholds applied by GCMC to the composites is reasonable although somewhat more optimistic when compared with AMEC's results. The differences in the capping thresholds generally are not expected to cause any material impact on the mineral resource estimate. An exception is in the Central Zone post-mineral intrusive rocks where the omission of a 1% Cu cap previously applied in 2008 allowed for a 20% increase in the contained copper in composites in 2010. This could have and impact on local grade estimation but is unlikely to impact the estimation on a global basis. The Central Zone intrusive rocks are generally a low-grade domain
- Contact plots indicate at least some form of composite sharing is warranted across the majority of domain boundaries. Absence of composite sharing in the current model is not anticipated to result in any material difference to the global estimate
- Grade and lithology domains are essentially unchanged since the 2008 resource update

  - The grade domain wireframes are irregular and angular and generally do not appear to reflect reasonable geologic shapes
  - There are local gaps and overlaps between some adjacent lithology wireframes that are causing inadvertent incorrect tagging of blocks and composites possibly resulting in local incorrect estimate of grades
- The grade shells are treated as a hard boundary during estimation, preventing sharing of composites across this boundary. Composite histogram and cumulative probability plots do not show any obvious indication of population boundaries. Contact plots between low and high grade zones generally show distinctly different average grades for the two zones but the contact relationship suggests a firm rather than hard contact. Given the disseminated style of mineralization and, in the absence of any known lithological controls, composite sharing may be warranted. Alternatively, the grade threshold for the high grade shell may need to be reduced.

14.2.2 Block Model Visual Inspection

A visual inspection of the block model and composites was completed for the Central, West Fork, South West, Junction, Junction North, and Middle Creek zones. Composites and blocks tagged from lithology wireframes in the Central Zone and from grade shells in all other zones compare well with the wireframes. OK estimated copper, gold and silver block grades compare well with composite grades. There are local areas, typically at the deposit margins, where economic grades have been extrapolated over large distances (blow-outs). Classification constraints are applied to prevent design pits and resource-constraining pits from inadvertently including blocks that are extrapolated over large distances. In future studies grade domains and search distances could be modified to limit over extrapolation of grades.

14.2.3 Comparison of the OK and NN Estimates

A comparison of paired OK and NN estimated blocks shows that in general the global copper and gold means for the various estimated zones are within ±5% of each other. This is considered an acceptable level. Exceptions where the Kriged mean grade is greater than 5% of the NN mean are the gold for the South West 200 and Junction 53 and Junction 200 domains and for the copper for Central Zone 400, the West Fork Zone 34, and Junction 200 domains. West Fork Zone 34 Kriged gold mean is 21.7% less than the NN mean. The West Fork Zone 34 is a very small high grade zone. The Central Zone 400 is also small. The impact on the resource estimate of the copper and gold biases for these zones is considered small. The Junction Zone 200 and South West Zone 200 are unconstrained low grade zones. The Junction 200 zone is predominantly classified as Inferred mitigating the impact that this bias could have on the resource estimate. The South West 200 zone is predominantly classified as Measured or Indicated. The copper bias in this zone could impact the recovered metal or mill throughput during production.

14.2.4 Comparison of the AMEC and GCMC 2011 Pre-feasibility Study Estimates

The GCMC 2011 pre-feasibility study OK estimate was prepared using capped composites. AMEC prepared an independent OK estimate using uncapped composites as a check, and compared copper, gold, and silver global capped and uncapped estimated mean grades of the blocks in the block model. The reduction in the estimated mean grades due to capping is generally in agreement with GCMC's capping thresholds.

A comparison of paired OK and NN estimated blocks shows that in general the global copper and gold means for the various estimated zones are within ±5% of each other. This is considered an acceptable level.

Copper and gold swath plots were prepared comparing composites, nearest-neighbour and kriged blocks plotted in east–west, north–south and vertical directions. These plots show that the kriged copper estimates generally match very closely to the nearest neighbour model in most zones where there is sufficient number of composites and estimated blocks. There are some zones showing a weak to moderate high bias for the kriged model.

AMEC notes that the poorest correlations are consistently in the Junction and North Junction zones, particularly where the composites are compared to the NN model. This may indicate a declustering problem or zone mismatch has occurred. This area should be verified during the next model update.

A comparison of the 2008 and 2010 block models shows that the number of blocks, before classification, with estimated copper grades in 2010 is significantly more than the number estimated in 2008. The copper content of the 2010 model is generally underestimated and the gold and silver content of the 2010 model are generally overestimated relative to the 2008 model.

14.2.5 Grade–Tonnage Curve Checks

AMEC conducted a discrete Gaussian or Hermitian polynomial change-of-support (Herco) check for the 2010 estimated zones. Checks were completed using 5 m composites, assuming a selective mining unit (SMU) block with dimensions of 25 m x 25 m x 15 m and using blocks within 35 m of a composite. Copper and gold grade–tonnage graphs were prepared for the volcanic, mineralized intrusive, and post mineral lithology domains in the Central Zone. In the South West, West Fork, Junction, and North Junction, where grade shell domain are used, grade–tonnage graphs were prepared for the low-grade and high-grade zones, and for a combined low- and high-grade zone.

The Central Zone copper estimate for volcanic rocks shows an acceptable degree of smoothing (less than 5%) at grades between 0.1 and 0.2% Cu. The Central Zone late-mineral intrusive shows moderate smoothing (the grade is underestimated and the tonnes overestimated) at copper cut-off grades above 0.1%. The Central Zone gold estimates for volcanic, mineralize intrusive, and late-post mineral intrusive all show moderate smoothing at cut-off grades between 0.1 g/t Au and 0.2 g/t Au.

The West Fork high-grade copper domain appears too smooth; the low grade copper domain curve is difficult to interpret with confidence. The combined low- and high-grade copper grade–tonnage curves curves indicate the estimate may be too selective (the grade is overestimated and the tonnes underestimated).

The South West combined low- and high-grade copper curves indicate the estimate may be too smooth at grades between 0.1 and 0.2% Cu.

The Junction high grade copper domain shows an acceptable level of smoothing at cut-off grades above 0.1% copper. The low grade domain is difficult to interpret with confidence. The combined low and high grade copper curves indicate the estimate may be too selective.

The North Junction high-grade copper domain estimate appears overly selective and the low grade copper domain estimate appears too smooth, although the plot is difficult to interpret with confidence. The combined low-grade and high-grade copper curves.

An estimate which is too selective or too smooth introduces a risk that the estimated metal may not be recovered during production or mill throughput may need to be adjusted to achieve recovered metal expectations for a given production period.

The flip from an acceptably smooth estimate in the high-grade zone to an excessively selective estimate when both the high-grade zone and the low-grade zone are assessed together may indicate the grade domains and/or estimation plan needs modification.

This risk is considered minor but should be examined in the next estimate prior to commencing Feasibility Study work.

14.2.6 Drill Hole Spacing Study

AMEC completed a drill hole spacing study as part of an audit of the 2008 Galore Creek mineral resource estimate. In 2010 AMEC completed a revised drill hole spacing study for each of the Central Zone (Areas 11 to 14), South West (Area 20), West Fork (Area 30), Junction (Area 40), North Junction (Area 50), and Middle Creek (Area 60). Results of the drill spacing study were used as part of the classification criteria for the Mineral Resources.

14.2.7 Review of Classification Criteria

AMEC recommended and GCMC agreed that the 2010 Mineral Resources classification use the criteria outlined in Table 14-3. These criteria account for recommended limitations in the use of legacy data, results of the drill hole spacing study, and concerns of local over extrapolation of grade.

Table 14-3: AMEC Recommended Classification Criteria for the Mineral Resource Estimate

	Measured			Indicated			Inferred
Area	Drill Hole Spacing (m)	Closest Composite (m)	Non-legacy Kriging Weight	Drill Hole Spacing (m)	Closest Composite (m)	or	Closest Composite (m)
Central Zone (CZ)	3 holes at 70 m	55	>0.67	3 holes at 100 m	70	2 holes at 55 m	120
West Fork (WF)	3 holes at 50 m	35	>0.67	3 holes at 100 m	70	2 holes at 55 m	120
South West (SW)	3 holes at 70 m	55	>0.67	3 holes at 100 m	70	2 holes at 55 m	120
Junction (JN)	3 holes at 70 m	55	>0.67	3 holes at 100 m	70	2 holes at 55 m	120
North Junction (NJ)	3 holes at 70 m	55	>0.67	3 holes at 100 m	70	2 holes at 55 m	120
Middle Creek (MC)	3 holes at 70 m	55	>0.67	3 holes at 100 m	70	2 holes at 55 m	120

14.3 Consideration of Reasonable Prospects of Economic Extraction Criteria

The extent of the classified material that might have reasonable expectation for economic extraction was assessed by applying a Lerchs–Grossmann (LG) pit outline to the Mineral Resources. The pit shells were then run using the following assumptions:

- A constant NSR cut-off of $10.08/t milled. The Net Smelter Return (NSR) was calculated as follows:

  NSR = Recoverable Revenue – TCRC (on a per tonne basis),

  where: NSR = Diluted Net Smelter Return; TCRC = Transportation and Refining Costs; Recoverable Revenue = Revenue in Canadian dollars for recoverable copper, recoverable gold, and recoverable silver

- Metal price assumptions of US$2.50/lb copper, US$1,050/oz gold, and US$16.85/oz silver
- Exchange rate assumptions of C$1.10 to US$1.00
- Cu Recovery = Recovery for copper based on mineral zone and total copper grade.

14.4 Mineral Resource Statement

After review of the GCMC 2011 pre-feasibility study Mineral Resource estimate, AMEC revised some criteria supporting the conceptual pit shell, including drill spacing for classification purposes, the commodity prices, and NSR value; metallurgical recoveries were also revised downward. As a consequence, AMEC has restated the Mineral Resources from those used in the GCMC 2011 pre-feasibility study report.

Mineral Resources take into account geologic, mining, processing and economic constraints, and have been confined within appropriate LG pit shells, and therefore are classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves.

The Qualified Person for the Mineral Resource estimate is Greg Kulla, P.Geo., an employee of AMEC.

Mineral Resources are reported at commodity prices of US$2.50/lb copper, US$1,050/oz gold, and US$16.85/oz silver, and have an effective date of 11 July 2011. Mineral Resources are stated in Table 14-4 using an NSR cut-off grade of $10.08/t milled.

Table 14-5 is a restatement of the Mineral Resources in Table 14-4, by deposit zone.

Table 14-4: Galore Creek Mineral Resource Table, Effective Date 11 July 2011, G. Kulla, P.Geo.

Category	Tonnage (Million tonnes)	Cu Grade (%)	Au Grade (g/t)	Ag Grade (g/t)	Contained Cu (Billion pounds)	Contained Au (Million ounces)	Contained Ag (Million ounces)
Measured	39.5	0.25	0.39	2.58	0.22	0.50	3.27
Indicated	247.2	0.34	0.26	3.81	1.85	2.04	30.26
Total Measured and Indicated	286.7	0.33	0.27	3.64	2.07	2.53	33.54
Inferred	346.6	0.42	0.24	4.28	3.23	2.70	47.73

Notes to Accompany Mineral Resources Table

1. Mineral Resources are exclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
2. Mineral Resources are contained within a conceptual Measured, Indicated and Inferred optimized pit shell using the same economic and technical parameters as used for Mineral Reserves. Tonnages are assigned based on proportion of the block below topography. The overburden/bedrock boundary has been assigned on a whole block basis.
3. Mineral resources have been estimated using a constant NSR cut-off of $10.08/t milled. The Net Smelter Return (NSR) was calculated as follows: NSR = Recoverable Revenue – TCRC (on a per tonne basis), where: NSR = Diluted Net Smelter Return; TCRC = Transportation and Refining Costs; Recoverable Revenue = Revenue in Canadian dollars for recoverable copper, recoverable gold, and recoverable silver using silver using the economic and technical parameters used for mineral reserves.
4. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content
5. Tonnage and grade measurements are in metric units. Contained gold and silver ounces are reported as troy ounces, contained copper pounds as imperial pounds.

Table 14-5: Galore Creek Mineral Resource Table Restated by Deposit Zone, Effective Date 11 July 2011, G. Kulla, P.Geo.

Area	Class	Tonnes (Mt)	Cu Grade (%)	Au Grade (g/t)	Ag Grade (g/t)	Contained Cu (Million lb)	Contained Au (Million oz)	Contained Ag (Million oz)
Central	Measured	65.2	0.51	0.39	4.68	73,000	0.81	9.81
Central	Indicated	568.5	0.52	0.23	5.75	645,000	4.24	105.08
Central	Inferred	293.7	0.46	0.21	4.54	295,000	2.01	42.83
SouthWest	Measured	32.17	0.33	0.66	2.45	23,000	0.68	2.53
South West	Indicated	60.16	0.28	0.66	2.68	37,000	1.27	5.18
South West	Inferred	33.13	0.17	0.46	2.27	12,000	0.49	2.42
West Fork	Measured	6.56	0.74	0.41	4.68	11,000	0.09	0.99
West Fork	Indicated	23.33	0.38	0.29	4.27	20,000	0.22	3.20
West Fork	Inferred	7.22	0.31	0.43	7.16	5,000	0.10	1.66
Junction	Measured	0.48	0.43	0.15	2.31	0	0.00	0.04
Junction	Indicated	13.20	0.54	0.16	2.45	16,000	0.07	1.04
Junction	Inferred	6.45	0.49	0.17	1.89	7,000	0.03	0.39
North Junction	Measured	2.23	1.17	0.57	10.21	6,000	0.04	0.73
North Junction	Indicated	34.05	0.74	0.41	6.43	55,000	0.45	7.04
North Junction	Inferred	3.66	0.22	0.26	2.20	2,000	0.03	0.26
Middle Creek	Measured	1.79	0.39	0.71	3.23	2,000	0.04	0.19
Middle Creek	Indicated	7.11	0.37	0.50	2.30	6,000	0.12	0.53
Middle Creek	Inferred	2.43	0.37	0.47	2.07	2,000	0.04	0.16

Notes to Accompany Mineral Resources Table

1. Mineral Resources are exclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
2. Mineral Resources are contained within a conceptual Measured, Indicated and Inferred optimized pit shell using the same economic and technical parameters as used for Mineral Reserves. Tonnages are assigned based on proportion of the block below topography. The overburden/bedrock boundary has been assigned on a whole block basis.
3. Mineral resources have been estimated using a constant NSR cut-off of $10.08/t milled. The Net Smelter Return (NSR) was calculated as follows: NSR = Recoverable Revenue – TCRC (on a per tonne basis), where: NSR = Diluted Net Smelter Return; TCRC = Transportation and Refining Costs; Recoverable Revenue = Revenue in Canadian dollars for recoverable copper, recoverable gold, and recoverable silver using silver using the economic and technical parameters used for mineral reserves.
4. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content
5. Tonnage and grade measurements are in metric units. Contained gold and silver ounces are reported as troy ounces, contained copper pounds as imperial pounds.

14.5 Comment on Section 14

The AMEC QPs are of the opinion that the Mineral Resources for the Project, which have been estimated using core drill data, have been performed to industry best practices, and conform to the requirements of CIM (2010).

Factors which may affect the conceptual pit shells used to constrain the mineral resources, and therefore the Mineral Resource estimates include:

- Commodity price assumptions

- The NSR value used to constrain the Mineral Resources is based on technical and economic parameters used to support Mineral Reserve estimation. Should these assumptions change, then the pit constraining the Mineral Resources will also change
- Metallurgical recovery assumptions
- Pit slope angles used to constrain the estimates
- Assignment of acid-soluble copper values
- SG values assumed for broken rock.

AMEC notes that the following should be considered during more detailed Project studies:

- Re-assay of legacy (Pre-1990) samples in order to assess for biases and possibly the removal of limitations on the use of legacy data
- A review of logged lithology with the goal of preparing better defined lithology domains
- A review of grade distribution with the goal of preparing better defined grade domains
- Grade capping in future studies be based on the results of assessment from several different capping methods to determine the most appropriate capping methodology. An assessment of the risk associated with high grade composites should be undertaken in future studies using AMEC's metal-at-risk approach or similar assessment method
- Composite sharing across domain boundaries, where supported by EDA data, should be considered
- Long-length silver assays from pre-1980s drill holes should be excluded from use in future studies
- A modified estimation plan to include a three hole minimum estimation pass.

15.0 MINERAL RESERVE ESTIMATES

15.1 GCMC 2011 Pre-feasibility Study Mineral Reserve Estimate

AMEC restated the Mineral Reserve estimate from that in the GCMC 2011 pre-feasibility study. Changes between the Mineral Reserve estimates are due to different Mineral Resources to those reported in the GCMC 2011 pre-feasibility study, commodity prices, metallurgical recovery assumptions, cut-off grade (NSR) assumptions, and operating costs.

15.2 AMEC Mineral Reserve Estimate

The Mineral Reserves for the Galore Creek property were prepared by Rudy Zdravlje, a Senior Mining Engineer with Teck Resources Ltd., on behalf of GCMC. This work was performed under the supervision of Jay Melnyk, Associate Mining Engineer, AMEC Americas Limited. The effective date of the Mineral Reserves estimate is 11 July 2011.

15.2.1 Key Assumptions, Parameters and Methods

Pit Slopes

AMEC Earth and Environmental reviewed and updated pit slope design criteria for the Project in 2008–2009. These geotechnical design criteria were incorporated into the GCMC 2011 pre-feasibility study pit slope designs. Data compilation for the AMEC designs included oriented core data from 19 boreholes drilled by BGC between 2004 and 2007 as well as mapping of several surface outcrops. The investigations also consisted of point load testing, downhole packer permeability testing and laboratory testing of selected rock core samples for unconfined compression, direct shear and triaxial tests.

AMEC's methodology for pit slope design consisted of identifying the bench face and inter-ramp angles using primarily a kinematic analysis procedure supplemented by probabilistic analysis for the various domains and pit wall orientations. A kinematic analysis was used to define the achievable bench face angles (BFAs) by identifying if toppling, wedge or planar failure geometries may be present for various wall orientations since there was insufficient data on fracture continuity to support a probabilistic approach.

Inter-ramp angles (IRAs) were calculated with the design BFAs using a minimum 8 m wide berm width as per BC Mining Regulations requirements and single or double benches. Faulting and shearing identified in each of the domains were checked against the proposed IRAs to verify whether daylighting or complex wedges would occur.

Inter-ramp pit slope angles are shown in Figure 15-1. For pit optimization purposes, the inter-ramp angles were flattened to account for ramp access as required.

Dilution

Prior to pit optimization, adjustments were made to the resource model to account for dilution and ore loss on ore/waste boundaries. The diluted grade for each block was calculated as a weighted average of the original block grade and horizontally adjacent block grade values.

The assigned weighting was 90% of the original block grade and 2.5% of each adjacent block grades. The assigned weighting distribution is considered appropriate for the selective mining unit (block sizes of 25 m x 25 m x 15 m), spatial nature of the mineralization and proposed scale of mining activities. For dilution estimates of Measured and Indicated blocks, Inferred material was treated as barren. This process effectively smears block grades across block edges, independent of a cut-off grade, such that all blocks are diluted and a variable cut-off methodology can be used during mine scheduling.

Metallurgical Recoveries

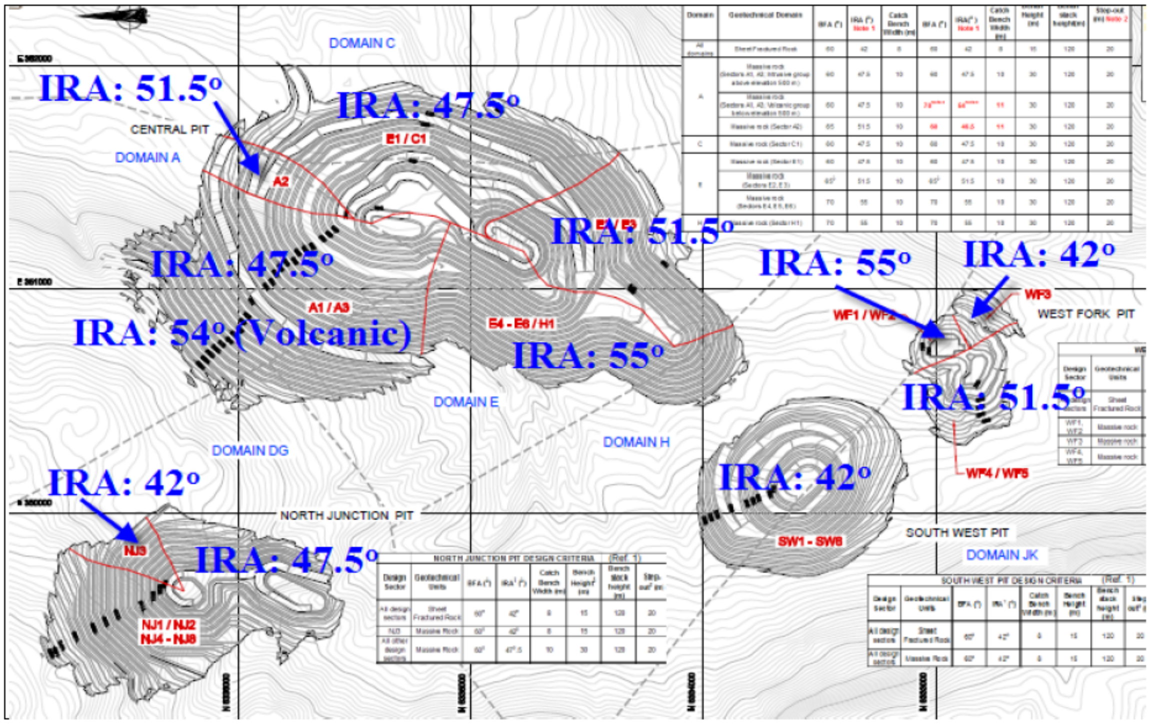
Diluted block grades were used to estimate recoverable block grades by applying the metallurgical recovery formulas shown in Table 15-1, in conjunction with an oxidized/non-oxidized flag in the block model.

NSR Calculations

Revenue will be generated from the sale of copper concentrates, which contain payable co-products of gold and silver. To capture the multi-element and variable recovery complexity, NSR values were calculated for block valuation during pit optimization. The NSR grade determination considers the diluted, recovered block grades and rock types, and applies the price and cost parameters shown in Table 15-2 resulting in a net value per tonne of ore, inclusive of all costs outside the mine gate.

Gold and silver refinery payables are variable by concentrate grade, and are shown in Table 15-3.

Figure 15-1: Pit Slope Design Angles



Note: Figure courtesy GCMC, NovaGold and Teck

Table 15-1: Metallurgical Recovery Equations Used to Support Mineral Reserve Estimation

Recovery (%)	Standard Ore	Oxidised/Near Surface Ore
Copper	7.66*Ln([Head Cu(%)])+ 94.34 *(cap at 95 %)*	([OxRConc]*([Head Cu(%)]-0.18)/([Head Cu(%)]*([OxRConc]-0.18)) * 94.8 where : ([OxRConc] =7.2*[Head Cu(%)]+1.6 *(cap at 95%)*
Gold	8.1*Ln [HeadAu(g/t)]+78 *(Cap at 90%)*	8.1*Ln [HeadAu(g/t)]+78 *(Cap at 90%)*
Silver	19.7* Ln([HeadAg(g/t)])+ 26 *(Cap at 90%)*	14.5* Ln([HeadAg(g/t)]) + 28 *(Cap at 75%)*

Table 15-2: NSR Parameters

Section	Value Used
Metal Prices ($US)	
- Cu	2.5/lb
- Au	1,050/oz
- Ag	16.85/oz
Exchange Rate (CAD$/US$)	1.1:1.0
Cu Metal Contract Terms	
- Cu Concentrate Grade (%) – variable depending on rock type	26 to 28%
- Treatment Charge ($/dmt)	70
- Cu Price Participation ± 5% ($2.00/lb basis) US$/dmt conc.	
- Copper Refining ($/lb)	0.07
Au Metal Contract Terms	
- Concentrate Grade (g/t)	variable
- Payable Au	variable
- Refining Charge ($/t oz)	6
Ag Metal Contract Terms	
- Concentrate Grade (g/t)	variable
- Payable Au	variable
- Refining Charge ($/t oz)	0.4
Transport	
- Ocean freight (US$/wmt)	53
- Port Handling (US$/wmt)	5
- Moisture (%)	8%
- Other Offsite Costs (Losses, Insurance, Sell, Supervision, Assay) (US$/wmt)	6.00

Table 15-3: Gold and Silver Payable Guidelines

Concentrate Grade (g/dmt)	Total Payable
Au ≤ 1	0% Au
1 < Au ≤ 3	90% Au
3 < Au ≤ 5	92% Au
5 < Au ≤ 8	95% Au
8 < Au ≤ 10	96% Au
10 < Au ≤ 15	97% Au
15 < Au ≤ 50	97.5% Au
Au > 50	98% Au
Ag ≤ 30	0% Ag
Ag > 30	90% Ag

15.2.2 Pit Optimization

Lerchs-Grossmann (LG) pit optimization was performed to determine the economic limits of the deposits. The NSR grade item (refer to Section 15.1.4) was used to assign block values. The previously mentioned pit slope recommendations were applied according to spatial area. The waste and ore based costs applied were calculated from first principles during a previous planning iteration, and are shown in

Table 15-4. Mine general costs were grouped with ore based costs as they are fixed on an annual basis.

A combined ore-based cost of $10.08/t milled was used for pit optimization. Because the mineralization-waste delineation was performed using an NSR block value, the combined mill ore based cost of $10.08/t milled represents the marginal breakeven cut-off grade for pit optimization purposes. Only Measured and Indicated Mineral Resources were considered as being processed. Inferred Mineral Resources were treated as waste.

Nested shells were generated over a range of revenue factors. Shell selection for guiding pit phase designs was performed with the criteria of maximizing NPV rather than selecting the revenue factor 1.0 shell which would maximize contained metal. The selected shells were 0.75, 0.69 and 0.55 for Central, Southwest–West Fork, and Junction respectively. Pit phase designs where then generated using 100 m minimum mining widths and 40 m-wide ramps. The Central pit was designed in three phases, while the Southwest–West Fork, and Junction pits were designed as single phase pits.

Table 15-4: Operating Costs Used for Pit Optimization and Scheduling

Cost Centre	Cost ($/t milled)
Mine General	1.75
Process	5.69
G&A	2.64
Total Ore Based Cost	10.08
Cost Centre	**Cost ($/t mined)**
Drilling	0.08
Blasting	0.16
Loading	0.17
Hauling	1.19
Total Direct Mining	1.60

15.2.3 Production Schedule

In order to properly reflect the high variability in mill throughput for the different rock types, a cash flow grade (CFG) item was calculated, which is a function of both the NSR and the mill throughput by rock type, and has units of $/SAG mill hour. A mining schedule was developed that honours operating constraints such as practical phase sequencing, descent rate limitations and equipment productivities. Cut-off optimization was applied, resulting in an optimized schedule. Ore/waste delineation for mineral reserves declaration is performed by the optimized scheduling process, with a variable cut-off applied to the CFG item.

The life of mine average cut-off grade of $30,726 per SAG hour (variable by year), which at a nominal throughput of approximately 95,000 t/d, is approximately equivalent to an NSR cut-off of $11.96/t.

At a long-term forecast copper price of US$2.50/lb, the life-of-mine average cut-off grade equates to a copper-equivalent (CuEq) grade of approximately 0.27%.

15.3 Mineral Reserves Statement

Mineral Reserves have been modified from Mineral Resources by taking into account geologic, mining, processing, and economic parameters and therefore are classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves.

The Qualified Person for the Mineral Reserve estimate is Jay Melnyk, P.Eng., an associate of AMEC Americas Limited.

Mineral Reserves are reported at commodity prices of US$2.50/lb copper, US$1,050/oz gold, and US$16.85/oz silver, and have an effective date of 11 July 2011.

Mineral Reserves are summarized in Table 15-5.

15.4 Comment on Section 15

The AMEC QPs are of the opinion that the Mineral Reserves for the Project, which have been estimated using core drill data, appropriately consider modifying factors, have been estimated using industry best practices, and conform to the requirements of CIM (2010).

Factors which may affect the Mineral Reserve estimates include: dilution, metal prices, smelter, refining and shipping terms, metallurgical recoveries, geotechnical characteristics of the rock mass, capital and operating cost estimates, effectiveness of surface and ground water management, and likelihood of obtaining required permits and social licenses.

Table 15-5: Mineral Reserve Statement, Effective Date 11 July 2011, Jay Melnyk, P.Eng.

	Tonnes	Diluted Grade					
	Mt	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**Contained Cu (Billion pounds)**	**Contained Au (Million ounces)**	**Contained Ag (Million ounces)**
Proven	69.0	0.61	0.52	4.94	0.9	1.15	11.0
Probable	459.1	0.58	0.29	6.18	5.9	4.30	91.2
Total Proven and Probable	*528.0*	*0.59*	*0.32*	*6.02*	*6.8*	*5.45*	*102.1*

Notes to Accompany Mineral Reserves Table

1. Mineral Reserves are contained within Measured and Indicated pit designs, and supported by a mine plan, featuring variable throughput rates, stockpiling and cut-off optimization. The pit designs and mine plan were optimized on diluted grades using the following economic and technical parameters: Metal prices for copper, gold and silver of US$2.50/lb, US$1,050/oz, and US$16.85/oz, respectively. Mining and ore based costs (process, G&A and mine general) of $1.60/t mined and $10.08/t milled respectively; an exchange rate of CAD$1.1 to US$1.0; variable recovery versus head grade relationships for both oxidized and non-oxidized material; appropriate smelting, refining and transportation costs; and inter ramp pit slope angles varying from 42º to 55º
2. Mineral Reserves are reported using a 'cash flow grade' ($NSR/SAG mill hr) cut-off which was varied from year to year in the scheduling process to optimize NPV. The cash flow grade is a function of the NSR ($/t) and SAG mill throughput (t/hr). The net smelter return (NSR) was calculated as follows: NSR = Recoverable Revenue – TCRC (on a per tonne basis), where: NSR = Net Smelter Return; TCRC = Transportation and Refining Costs; Recoverable Revenue = Revenue in Canadian dollars for recoverable copper, recoverable gold, and recoverable silver using the economic and technical parameters mentioned above. SAG throughputs were modeled by correlation with alteration types.
3. The life of mine strip ratio is 2.16
4. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content
5. Tonnage and grade measurements are in metric units. Contained gold and silver ounces are reported as troy ounces, contained copper pounds as imperial pounds.

The AMEC QPs are of the opinion that these potential modifying factors have been adequately accounted for using the assumptions in this Report, at this pre-feasibility level of study, and therefore the Mineral Resources within the mine plan may be converted to Mineral Reserves using the appropriate confidence categories.

Factors which may affect the assumptions in this Report include:

- Commodity price and exchange rate assumptions
- Mill throughput of the identified ore types may prove to be higher or lower than modelled. If certain rock types or delivered blends of rock types have lower throughputs than currently modelled, this would increase the processing cost, which would in turn increase the mill cut-off grade. All other things held constant, this would tend to reduce the tonnage of the Mineral Reserve and the amount of contained metal. If throughput reductions are significant, this could reduce the size of the economic pit limits, further reducing the Mineral Reserve. Furthermore, a reduction in throughput would delay cashflow, resulting in a negative impact on Project economics

- The construction schedule, including pre-production mining, may take longer than currently estimated. Longer construction duration would increase the construction cost and delay cashflow, resulting in a negative impact on Project economics and a potential reduction in the Mineral Reserves
- The tunnel construction schedule and budget may be underestimated. A longer tunnel construction duration could significantly increase helicopter supported construction costs in the Galore Creek Valley, or an underestimate of the cost of tunnel construction could significantly increase capital costs, which can have a negative impact on Project economics and a potential reduction in the Mineral Reserves
- Effective surface and ground water management will be important to the safety and productivity of the mining operation. If the currently-planned water management methods prove to be ineffective, additional measures such as underground drainage galleries may be required, which would significantly add to the capital and operating costs, resulting in a negative impact on Project economics and a potential reduction in the Mineral Reserves
- Declaration of Mineral Reserves assumed that granting of appropriate environmental and construction permits would be forthcoming from the relevant authorities. While AMEC believes there is a reasonable basis for this assumption, if any permit was denied or permit grant was significantly delayed, there could be an impact on the Mineral Reserves.

16.0 MINING METHODS

16.1 GCMC 2011 Pre-feasibility Study Mine Plan

As AMEC restated the Project's Mineral Reserves, the mine plan in the GCMC 2011 pre-feasibility study was also restated by AMEC.

16.2 AMEC Mine Plan

16.2.1 Optimization

The economic parameters used for pit and mine schedule optimization are shown in Table 16-1.

Mining costs, direct and general, which were used to optimize pit shells, were determined from first principle estimates from a previous mine planning iteration, based on a mixed electric and hydraulic shovel fleet and 345 t-class haul trucks, averaged over the mine life. Mining general costs were treated as ore-based costs.

The total ore-based cost (process, general and administrative (G&A) and direct mining) of $10.08/t milled was used as the NSR-based cut-off grade in Whittle^{TM} for optimized pit shell creation. Table 16-2 summarizes the mining and ore-based costs used for pit and mine schedule optimization.

Metallurgical recoveries of the Galore Creek deposits will vary with copper feed grade ranges and degree of oxidization. The metal recoveries and concentrate grades are based on metallurgical testwork, and the equations are shown in Table 13-2 and discussed in Section 13.10.

Ocean freight and port handling costs were estimated to be US$58/wmt. An additional US$6/wmt was included to account for miscellaneous costs such as losses, supervision, and assays, among others. Payable copper is based on a one-unit deduction. Gold and silver payables are based on the guidelines shown in Table 16-3.

Treatment and refining charges are US$70/dmt of concentrate and US$0.070/lb of copper, respectively. The refining charge for gold and silver are US$6/oz and US$0.4/oz, respectively. Presently, no infrastructure is affecting the reserve estimate, as infrastructure has been placed outside of the pit design limits.

Table 16-1: Galore Creek Mine Plan Optimization Parameters

Parameter	Section
Metal Prices ($US)	
- Cu	2.5/lb
- Au	1,050/oz
- Ag	16.85/oz
Process Rate (t/d)	95,000
Process Rate (Mt/a)	34.68
Discount Rate (%)	8
Exchange Rate (CAD$/US$)	1.1:1.0
Diesel Fuel (CAD$/L)	1.04
Power (CAD$/kWh)	0.05
Cu Metal Contract Terms	
- Treatment Charge ($/dmt)	70
- Cu Price Participation ± 5% ($2.00/lb basis) US$/dmt conc.	
- Copper Refining ($/lb)	0.07
Au Metal Contract Terms	
- Concentrate Grade (g/t)	variable
- Payable Au	variable
- Refining Charge ($/t oz)	6
Ag Metal Contract Terms	
- Concentrate Grade (g/t)	variable
- Payable Au	variable
- Refining Charge ($/t oz)	0.4
Transport	
- Ocean freight (US$/wmt)	53
- Port Handling (US$/wmt)	5
- Moisture (%)	8%
- Other Offsite Costs (Losses, Insurance, Sell, Supervision, Assay) (US$/wmt)	6.00
Operating Costs	
- Direct mine cost (CAD$/t mined)	1.60
- Mine General (CAD$/t milled)	1.75
- Process (CAD$/t milled)	5.69
- G&A (CAD$/t milled)	2.64
Marginal Grade Costs (CAD$/t milled)	10.08
Marginal Grade Costs (US$/t milled)	9.16
Pit Slopes	42° to 55°

Table 16-2: Operating Costs used for Pit Optimization and Scheduling

Cost Centre	Cost ($/t milled)
Mine General	1.75
Process	5.69
G&A	2.64
Total Ore Based Cost	10.08
	Cost ($/t mined)
Drilling	0.08
Blasting	0.16
Loading	0.17
Hauling	1.19
Total Direct Mining	1.60

Table 16-3: Gold and Silver Payable Guidelines

Concentrate Grade (g/dmt)	Total Payable
Au ≤ 1	0% Au
1 < Au ≤ 3	90% Au
3 < Au ≤ 5	92% Au
5 < Au ≤ 8	95% Au
8 < Au ≤ 10	96% Au
10 < Au ≤ 15	97% Au
15 < Au ≤ 50	97.5% Au
Au > 50	98% Au
Ag ≤ 30	0% Ag
Ag > 30	90% Ag

Optimized pit shells for the Galore Creek Project were developed utilizing the Whittle™ (Lerchs–Grossmann) software. Separate optimizations were run for the Central, Southwest/West Fork, and Junction areas using Measured and Indicated Mineral Resources. A suite of nested pit shells were generated at varying revenue factors (RF), and selected for design guidance by ranking by NPV of preliminary schedules at an 8% discount rate (note the financial analysis discussed in Section 23 uses a discount rate of 7%).

The Central ultimate optimized pit shell (RF = 0.75), was selected using a series of phases that provided the highest NPV. Following selection of the ultimate pit shell, a directional mining approach was used to phase the Central pit. To take advantage of higher value ore with low stripping requirements in the south of the Central Zone, the Central pit will be mined in a three-phase, south to north sequence. This sequence allows for 1.5 years of mine production to occur from Central phase 1, which lies completely south of the dendritic water channel, during which time the channel can be diverted, allowing production in Central pit phase 2 to begin.

The Southwest/West Fork and Junction ultimate pit shells (RF 0.69 and 0.55 respectively) were similarly selected to maximize NPV. However, these pits were not sufficiently large to benefit from phasing, and will therefore be mined in single phases. The Southwest/West Fork pit optimization was constrained by a geotechnical limit to keep the pits away from the steep talus slope to the south west.

For pit shell optimization purposes, a fixed processing rate of 95,000 t/d was assumed. However, for scheduling purposes, a variable throughput was used based on the correlations to rock type and alteration.

The selected optimized pit shells were used to guide the designs of mineable pit phases which adhere to geotechnical and operational constraints (e.g., 40 m wide ramps, 10% ramp gradients and 100 m minimum mining widths). Phasing of the Central pit enabled the deferral of waste stripping and resulted in a schedule with balanced stripping requirements from year to year. In addition, phasing allowed higher-grade ore to be mined early in the schedule.

16.2.2 Marginal Cut-off Grade Considerations

For schedule optimization purposes, the marginal cut-off grade is the hurdle rate that must be met to cover the variable portion of the cost of processing. The other ore-based costs (the fixed portion of processing, mine general and G&A) are treated as fixed annual costs by the scheduling program. Due to the high variability in mill throughput rates, a cash flow grade was used for the marginal cut-off grade to optimize the order in which material is sent to the mill. The cash-flow grade represents cash-flow-per-SAG-mill-hour, and is calculated by subtracting the ore value ($/t) by the waste value ($/t) and multiplying by ore processing rate (t/hr). The marginal cut-off grade by definition is $0/hr.

Waste and ore tonnages are based on scheduled material flows from the optimized mine schedule and the variable cash-flow grade cut-off. Due to the variable cut-off grade, some material above a breakeven economic cut-off is sent to the waste dumps, as higher-grade material is available to fill the concentrator at the same time this lower grade material is being released in the mining sequence. For the purposes of the GCMC 2011 pre-feasibility study, a nominal 20 Mt stockpile was assumed, so some of this marginal material was scheduled to be wasted and not reclaimed. During more detailed studies, the benefits utilizing a much larger low-grade stockpile should be analyzed.

The marginal cut-off grade of CFG = $0/h (NSR = $3.68/t for average throughput material) is the theoretical minimum grade of material that can be processed as this value covers variable costs. In practice, this minimum grade is never met as the annual fixed costs must also be covered. The lowest grade block processed (in the last production year) has a CFG of $6,641/hr, which equated to an NSR value of $5.25/t milled. A total of 97% of the LOM mill feed has NSR grades exceeding $10.08/t milled, with the remainder being soft, high throughput material. The annual CFG cut-offs are included in the discussions in Section 16.2.

A variable cut-off grade strategy was used during the life of mine, based on the Kenneth Lane cut-off grade equations (Lane, 1988). The variable cut-off grade strategy was selected so that the mill is processing material in a manner that maximizes the NPV of all future cash flows.

The benefit in terms of NPV optimization, by using cash flow grades, is the optimal ordering of material to be sent to the mill. More simplistic approaches would treat material with the same NSR values but drastically different throughput rates as identical; whereas by using the cash flow grade cut-off strategy, material with a higher throughput is treated preferentially as the same revenue is recovered in a shorter period of time.

16.3 Proposed Production Schedule

The mine sequence for Galore Creek deposits is based on mining the six separate phases. The three-phase Central pit is scheduled to produce ore throughout the mine life with the satellite, Southwest, West Fork, and Junction pits supplementing ore production at various stages of the mine life. The Southwest pit will begin production at the start of the schedule to take advantage of lower waste stripping requirements and higher grades. The Southwest pit is also located south of the dendritic water flow allowing additional time for water diversion measures to be implemented. Mining of the Junction pit will begin later in the mine life, as it is located in a higher, more operationally challenging, area. Mining in the Junction pit will occur after mine operations have had several years of experience mining in the Galore Creek environment. Table 16-4 shows the contributions from the various pits over the life-of-mine.

After taking into consideration operational constraints, the scheduling of phases was based on maximizing NPV using Comet, an industry-standard optimization software package, and equipment fleet constraints (how many phases can be mined in a single time to not spread the mine equipment fleet over too great an area).

The production schedule contains one year of pre-production and envisages a mine life of 17.6 production years (not including one year of pre-production). The waste/ore delineation is defined within the scheduling process to maximize NPV. Minimum thresholds are set as to what can be processed profitably, but what is sent to the mill in any given period as ore is dependent on what material is available, remaining stockpile capacity, and haulage and processing constraints. Ore feed to the processing plant will be composed of broken and stick rock types (refer to Section 10.4 for a description of these types). The processing rates for broken and stick ore vary greatly based on the amount of garnet and orthoclase alteration. Table 16-5 shows the material throughputs based on 92% mill utilization. Broken ore represents approximately 35% of the total ore feed.

To select the final production schedule, many production schedules were developed. Each alternative incorporated different methodologies to examine a wide range of strategies. The final schedule was selected based on its operability and high NPV and is included as Table 16-6.

Table 16-4: Planned Pit Schedule

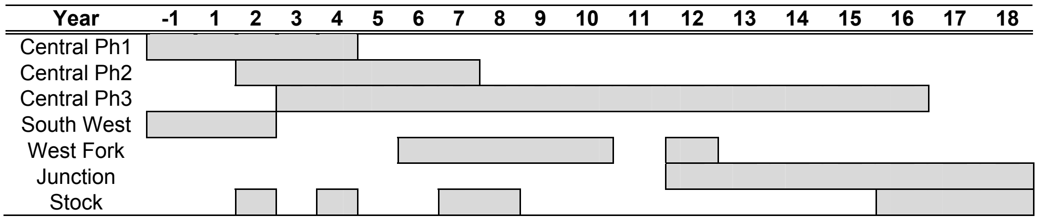


Table 16-5: Material Throughputs based on 92% Mill Utilization

Rock Type	Alteration (Garnet/Orthoclase)	Mill Throughput (t/d)
Stick	low/low	73,328
	high/low	78,075
	high/high	75,911
	low/high	69,817
Broken	low/low	121,815
	high/low	129,632
	high/high	126,055
	low/high	112,630

Table 16-6: Life of Mine Production Plan

Period Name	Start Time	Duration	Cut-off Grade ($/Hr)	Total Rock Mined (kt)	Total Waste (kt)	PAG Waste (kt)	NAG Waste (kt)	Overburden (kt)	Ore Direct to Mill (kt)	Ore to Stockpile (kt)	Ore from Stockpile (kt)	Stockpile Inventory (kt)	Strip Ratio
Totals		17.62	30,726	1,666,811	1,138,842	283,547	719,691	135,605	527,969	24,988	24,988		2.16
Prestrip	1	1		28,078	25,712	10,351	4,025	11,336		2,365		2,365	
2	2	1	64,882	84,524	50,793	16,066	18,383	16,343	24,887	8,844		11,210	2.04
3	3	1	14,411	136,344	98,484	18,300	54,026	26,157	37,860			11,210	2.60
4	4	1	30,719	126,912	89,099	28,329	51,054	9,716	37,813			11,210	2.36
5	5	1	16,724	133,177	98,780	22,474	73,496	2,810	34,397			11,210	2.87
6	6	1	21,683	135,089	106,675	25,099	78,464	3,112	28,414			11,210	3.75
7	7	1	18,088	135,547	109,129	22,064	78,083	8,983	26,417			11,210	4.13
8	8	1	27,207	127,311	105,821	26,424	59,427	19,971	30,334		8,844	2,365	3.49
9	9	1	26,775	132,683	100,917	20,938	51,791	28,188	32,949		1,183	1,183	3.06
10	10	1	45,928	120,453	82,225	26,544	53,500	2,181	33,932	4,296		5,479	2.42
11	11	1	53,300	82,328	48,852	15,020	33,819	13	27,831	5,645		11,124	1.76
12	12	1	31,334	59,269	32,936	7,205	25,586	145	26,333			11,124	1.25
13	13	1	33,514	68,207	41,762	5,655	35,938	170	26,445			11,124	1.58
14	14	1	37,292	76,240	48,983	4,910	40,642	3,431	27,256			11,124	1.80
15	15	1	50,255	76,766	43,308	8,849	33,575	884	29,620	3,838		14,961	1.46
16	16	1	27,274	64,407	35,726	15,880	17,683	2,164	30,600		1,919	13,042	1.17
17	17	1	26,425	37,841	10,606	5,202	5,404		27,235			13,042	0.39
18	18	1	23,147	35,759	8,395	3,906	4,488		27,364			13,042	0.31
19	19	0.62	6,641	5,879	638	332	306		18,284		13,042		0.03

Note: table is reported as periods, and not as production years. Period 1 = Year -1.

Figure 16-1 shows the configuration of the planned mine site at the end of operations, but prior to implementation of reclamation and closure.

16.4 Mining Equipment

Mining equipment selection was based on the mine production schedule and equipment productivities, as well as including consideration of workforce and operating hours.

The major equipment selected includes:

- 40 haul trucks, 345 tonne class
- 3 electric cable shovel, 59 m^2 capacity
- 2 hydraulic shovel, 42 m^2 capacity
- 1 front end loader, 38 m^2 capacity
- 5 blast hole rotary drills, 311 mm size.

Shift rotation will consist of two 12-hour shifts per day. The schedule will require four crews working 14 days on, followed by 14 days off. With each 14-day rotation, the crews will alternate between dayshift and nightshift. A vacation and sick time relief burden rate of 18% was incorporated into the hourly employee estimates.

Equipment was assumed to operate 356 days per year. Nine days without pit production were estimated as weather days due to the heavy snowfall in the Galore Creek Valley during the winter season. Equipment productivities were based on benchmarks from AMEC (2008) report and in-house mining expertise. The projected mine life exceeds the economic life of the hauling and support equipment. As a result, haul trucks, dozers, and graders will require replacement when their operating hours reach the economic limit. Major loading and drilling equipment are not considered to require replacement.

16.4.1 Loading Fleet

The primary loading fleet will consist of three electric cable shovels, two hydraulic shovels, and one front-end loader. Electric shovel, hydraulic shovel and loader productivities are based on an annual maximum mining rate of 30 Mt, 26 Mt, and 18 Mt, respectively. These rates are based on first-principle calculations by GCMC.

Figure 16-1: Pit Layout Plan (Year 18)

Note: Figure courtesy GCMC, NovaGold, and Teck.

Shovels will generate the majority of ROM production. The loader will be required in the pits during years when the production rate exceeds the capacity of the shovel fleet. The loader will be responsible for re-handling ore from the stockpile. In addition, the loader will be responsible for pioneering and snow removal work. Operating hours have been allocated to the loader for these auxiliary duties. Annual operating hours for pioneering are based on 5% of ROM production. Annual operating hours for snow removal are based on assumptions of surface area, compaction, equipment performance, and snowfall (10 m).

No unit in the loading fleet is expected to require replacement during the mine life.

16.4.2 Hauling Fleet

The haul truck fleet size was determined using Talpac™ industry standard software, and was based on the number of operating hours required to meet the proposed production schedule and haul cycles. Speed restrictions were set for truck type, as well as a 15 km/h loaded downhill limit. Cycle times were converted to truck "workhour" productivities through an operations factor of 85%. This workhour productivity, accompanied with the bench production schedule and stockpile production schedule, allowed for workhours to be calculated for hauling all material over the mine life. Applying the operations efficiency factor, "operating hours" were calculated for the life of mine schedule. In addition, a provision for snow removal work was added to the total production operating hours. The provision for snow removal was based on assumptions of surface area, compaction, equipment performance and annual snowfall (10 m).

Projected haulage fleet requirements over the life-of-mine are included as Table 16-7.

16.4.3 Support Equipment

The main support equipment fleet will consist of track dozers, rubber-tire dozers, graders, wheel loaders, excavators, and water trucks. Criteria for the numbers in the fleet are based on haul lengths, haul numbers, and active waste dumps:

- 1.3 dozers per shovel + loader
- 1 rubber-tire dozer during pre-production, and 2 when crusher is in operation
- 1 grader per 7 trucks
- 2 excavators, 2 wheel loaders, and 2 water trucks during full production.

Table 16-8 details the projected support equipment requirements.

Table 16-7: Haulage Fleet Requirements over the Life of Mine

Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Total Req	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**
New Req	14	36	36	36	36	40	40	38	31	34	26	22	27	29	29	22	15	17	6

Table 16-8: Support Fleet Requirements

Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Track Dozers																			
Total Required	3	5	8	8	8	8	8	8	8	8	5	4	4	5	5	5	4	2	2
New Required	3	2	3	0	0	0	0	0	0	0	0	0	0	0	4	0	0	0	0
Rubber-tire Dozers																			
Total Required	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
New Required	1	1	0	0	0	0	0	0	0	0	0	1	1	0	0	0	0	0	0
Graders																			
Total Required	2	6	5	5	5	6	6	5	6	5	4	3	4	4	4	3	2	3	1
New Required	2	4	0	0	0	0	0	0	0	0	0	0	0	0	2	1	0	0	0
Wheel Loaders																			
Total Required	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
New Required	1	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Excavators																			
Total Required	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
New Required	1	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Water Trucks																			
Total Required	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
New Required	1	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0

16.4.4 Drilling

The drilling fleet requirement consists of four large diameter production drills, one highwall drill, and one small diameter tank drill. These requirements should be further defined during more detailed Project studies.

Production drilling will be carried out by a fleet of electric rotary drills producing 12 ¼” diameter blast holes. A penetration rate of 25.2 m/h was selected during earlier Project studies (Hatch, 2006). After reductions in set-up and move, the total cycle time estimated was 43.3 minutes per 16.5 m hole.

A 150 mm diameter highwall drill is planned to be utilized to drill starter benches (pioneering) for the larger production drills, as well as wall control blasting requirements. Wall control blasting typically has pre-shear and buffer holes; these are drilled at closer spacing than production holes to control highwall exposure to the amount of explosives. The small diameter tank drill (nominal 90 mm) will be used for secondary blasting and pioneering requirements.

16.5 Blasting and Explosives

Due to the wet ground conditions, production blasting will consist of a heavy ANFO combination, making the explosive mixture resistant to water. The blasting agent will be a mixture of 30% ANFO to 70% emulsion to match the wet conditions. No allowances have been incorporated in the blasting methodology for wall control.

16.6 Comment on Section 16

In the opinion of the AMEC QPs, the following conclusions are appropriate:

- The proposed Project will be a conventional, large-tonnage, open-pit operation with approximately 528 Mt of ore processed over the life-of-mine
- The mine plan developed for Galore Creek envisages mining six separate phases from four open pits at Central, Junction, West Fork, and Southwest
- The SMU block size of 25 m x 25 m x 15 m reflects the selectivity of the proposed open pit mine milling rate. The bench height in all pits will be 15 m, with 40 m wide ramps and a designed minimum mining width of 100 m allowing for double-sided loading in the narrowest mining areas
- The GCMC Project will feature a large-scale conventional open pit mine that will provide process plant feed at a nominal rate of 95,000 t/d or 34.6 Mt/a. Annual mine production of ore and waste will peak at 136 Mt/a with a LOM waste/ore stripping ratio of 2.16-to-1
- A marginal cut-off grade of CFG = $0/hr (NSR = $3.37/t) is the minimum grade of material that can be processed. In practice, the lowest grade block processed has a CFG of $6,641/hr, which equated to an NSR value of $5.25/t milled
- Ore feed to the processing plant will consist of broken and stick rock types. The processing rates for broken and stick ore vary greatly based on the amount of garnet and orthoclase alteration. Broken ore represents approximately 35% of the total ore feed
- Lower-grade ore that must be released at the same time as higher-grade ore will be sent to a coarse ore stockpile near the crusher and will be milled later in the

schedule or during weather and operations delays. No blending of stockpiles is planned

- Mining in the Galore Creek Valley will be carried out using a mixed shovel fleet and trucks. Mining equipment requirements were based on the mine production schedule and equipment productivities, and included consideration of workforce and operating hours. The fleet is appropriate to the planned production schedule.

17.0 RECOVERY METHODS

The design for the process plant is based on processing the mined material through a conventional crushing, grinding and flotation plant using standard proven processes and equipment. The plant will handle a blend of material from the various zones of the Galore Creek deposit with approximately 80% coming from the Central Zone. Although the characteristic of materials from the various zones are different, the process applied is the same. Only the process conditions, mainly slurry densities and reagent dosages change for the different materials. A simplified flowsheet is shown in Figure 17-1.

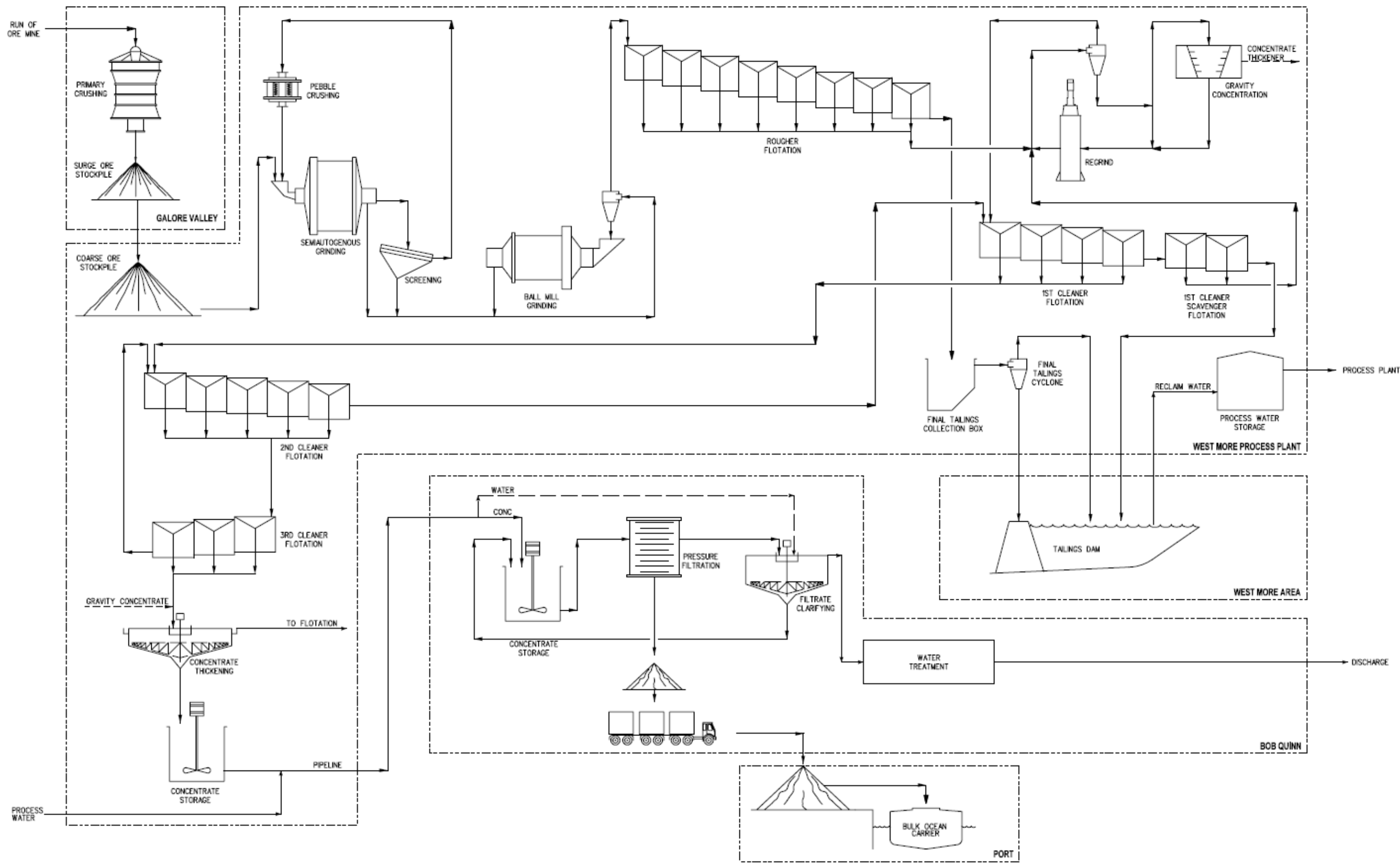
17.1 Proposed Process Plant Design

Run-of-mine (ROM) material will be delivered by 345 t- capacity mine haul trucks to the primary crushing plant, located in Galore Creek Valley. The crushing plant is designed to operated at 365 d/a with an average utilization of 70%. Crushed material will be conveyed to a surge pile in the East Fork of the Galore Creek Valley, then by overland conveyors to the coarse material stockpile adjacent to the grinding and flotation plants in the West More Valley. The average utilization of the overland conveyor system is planned at 87%.

Material reclaimed from the coarse ore stockpile will be fed to the mill where it will be ground to approximately 80% passing 200 µm in an SABC grinding circuit will consist of one SAG mill, three ball mills, two pebble crushers and three cyclone packs. The SAG mill will be in closed-circuit with the pebble crushers, while the ball mills will be in close-circuit with the cyclones. The cyclone overflow will be gravity fed to a distributor, which directs the slurry equally into two banks of rougher flotation cells. The rougher concentrate will be pumped to the cleaning circuit, while the rougher tailing will be cycloned to produce sand for tailings dam and slime for deposition in a tailings storage facility (TSF) to be located in the West More Valley.

The rougher concentrate will be reground by four tower mills to approximately 40 µm and will be cleaned in three stages to produce marketable-grade copper concentrate containing copper, gold and silver. The cleaner tailing flows by gravity to a designated area in the TSF where it will be kept underwater at all times. Grinding, flotation, thickening, and tailings disposal facilities are designed to operate on a continuous basis, i.e. 365 d/a, with a utilization of 92%.

Figure 17-1: Proposed Process Flowsheet



Note: Figure courtesy GCMC, NovaGold, and Teck

The final concentrate will be thickened to approximately 60% solids and pumped via a pipeline to a filter plant where pressure filters will reduce the moisture content to approximately 8%. The concentrate pipeline will be designed to operate at 1.32 times the nominal rate to allow for fluctuations in both head grade and tonnage. From the filter plant, the concentrate will be trucked to and stockpiled in a storage shed at the port of Stewart prior for transfer to ocean-going vessels.

17.1.1 Primary Crusher

ROM ore will be transported to the primary crusher station near the final pit rim by 363 t mine haul trucks. The crusher station will be an in-ground type with a building over the dump hopper and maintenance area. The station will comprise a dump hopper, gyratory crusher, surge bin, discharge apron feeder, maintenance crane, rock breaker, hydra-set maintenance trolley compressor and other associated ancillary equipment.

The truck dump hopper will have a live capacity of approximately 700 t. Two truck dumping aprons are provided in the design, one on either side of the dump hopper. ROM ore will be gravity-fed to a 60 x 110 class gyratory crusher with an open side setting of 175 mm and driven by a 750 kW motor. Crushed ore will fall directly from the crusher into a 700 t capacity surge bin. The apron feeder will draw the crushed ore from the bin and transfer it to the discharge conveyor for transportation to the coarse ore surge pile.

A dry-type dust collection system will be installed at the transfer point from the apron feeder to the discharge conveyor.

The discharge belt conveyor will be 1,829 mm wide, approximately 500 m long and will be designed to carry 7,000 t/h at 3.75 m/s, driven by a 630 kW motor. The first section of the conveyor will run in a tunnel and then in an enclosed gallery to the coarse ore surge pile.

The crusher station operator will be located in a cabin overlooking the truck dump hopper. From this location, the operator can control the complete primary crushing system from the traffic lights at the truck-dumping aprons to the discharge of coarse ore into the surge pile.

17.1.2 Coarse Ore Surge Pile

The coarse ore surge pile is designed to absorb short-term fluctuations in the output from the primary crusher and provide a uniform feed rate to the overland conveyor system.

The conical-shaped pile will have a total capacity of 40,000 t and a live capacity of approximately 8,000 t or 1.5 hours of feed at design capacity to the overland conveyor system.

Two apron feeders beneath the surge pile will transfer ore to the first of a series of overland conveyors. Each feeder is designed to deliver the full design capacity of the overland conveyor system. Under normal operation, each feeder will deliver 50% of the feed to the conveyor and the control system will adjust the feeder's speed so that the combined output from the two feeders does not exceed the conveyor design capacity.

The surge pile will be 70 m in diameter x 27 m high. A geodesic dome over the pile will protect the ore from the elements and will reduce the risk of dust emissions from the surge pile.

17.1.3 Overland Conveyor System

The system will comprise three conventional belt conveyors in series. Each conveyor will include the provision of belt turnover on the return to reduce the effects of carry-back and spillage under the conveyor.

The crushed ore discharge conveyor will be the first conveyor in a series. The crushed ore discharge conveyor will convey primary crushed ore from the surge pile at the west end of Galore Creek Valley's East Fork to a transfer station adjacent to the north portal of the tunnel at the head of the East Fork. The conveyor raises 89 m over its 3,152 m length and will be driven at the head end by three 1,400 kW variable speed shaft-mounted drive units. The conveyor will be supported on conventional tables at grade, except where it approaches the drive and transfer station at the head end, where it will be supported in a gallery. The conveyor will be located alongside the main access road from the surge pile to the north portal of the tunnel. It will be enclosed along its entire length with access on both sides for maintenance. The section at grade will have sufficient space inside the cover to permit access with an ATV or small pickup truck for maintenance.

The overland tunnel conveyor will be the second in the series, conveying ore from the transfer tower adjacent to the tunnel north portal to the transfer tower at the south portal where it will discharge to the stockpile feed conveyor. The tunnel conveyor raises 200 m over its 14,577 m length and will be driven by six 2,500 kW variable speed shaft-mounted drive units. Two of these units will be located at the tail of the conveyor and four at the head.

In the tunnel, the conveyor will be hung from the back with clearance underneath to allow access with a maintenance vehicle for inspection and idler change-out. Sections of the conveyor outside of the tunnel will be supported on table or in galleries as required for the specific location.

The stockpile feed conveyor will be the third in the system and transport the ore from the tunnel conveyor transfer tower at the south portal to the coarse ore stockpile at the West More plant site. The conveyor raises 460 m over its 4,036 m length and will be driven by four 2,800 kW shaft-mounted drive units located in a drive station at grade near the base of the coarse ore stockpile. The conveyor will be located on tables at grade for the majority of its length, then in an elevated gallery from the drive station to the head pulley at the coarse ore stockpile. It will be totally enclosed along its entire length with access on both sides for maintenance. The section at grade will have sufficient space inside the cover to permit access with an ATV or small pickup truck for maintenance.

17.1.4 Coarse Ore Stockpile

The coarse ore stockpile will receive primary crushed ore from the overland conveyor system and will provide surge capacity between the conveyor system and the mill. The conical-shaped pile will have a total capacity of 240,000 t and a live capacity of approximately 47,500 t or 12 hours of feed at mill design capacity. The stockpile will be 125 m in diameter x 47 m high. A geodesic dome over the pile will protect the ore from the elements and will reduce the risk of dust emissions from the stockpile.

Four apron feeders beneath the pile will transfer ore to the SAG mill feed conveyor. Each feeder is designed to deliver one-third of the mill design capacity; this will allow the operator to adjust the feed from different parts of the stockpile. Under normal operation, each feeder will deliver 25% of the feed to the conveyor. The control system will adjust the feeder’s speeds to match the required mill feed rate (nominally 4,300 dmt/h) that will be monitored by the belt scale on the SAG feed conveyor.

17.1.5 Grinding

The grinding circuit will be rated at a nominal 95,000 t/d. The grinding circuit will be designed to handle competent ore, which will constitute the mill feed for the majority of the mine life. In the early years of the operation, the mill will receive less competent, sheet-fractured rock from near the surface part of the deposits. The operating conditions described here, particularly the sizes of the grate and screen opening, will be modified to process this material that will be potentially easier to mill. Consequently, the maximum mill throughput with this ore is projected to be

110,000 t/d, which will be within the design capacity for conveyors, pumps and other process equipment.

SAG Milling

The 12.2 m diameter x 7.9 m EGL (40' x 26') SAG mill will be driven by a 26 MW gearless, variable speed drive. Process water will be added to the feed chute to achieve 60% to 70% solids in the mill feed. The steel ball size will be 5” (125 mm) diameter and the steel charge will be nominally 15%.

Lime slurry will be added at a constant rate into the SAG mill to raise its pH to just below 10. Smaller, controlled quantities of lime will be added to each ball mill to trim the flotation feed to pH 10.

Ore will leave the SAG mill through the discharge grates, with 75 mm ports for pebble relief and the trommel screen. Oversize pebbles will be discharged from the trommel screen onto a double-deck vibrating screen equipped with water sprays to wash the pebbles before they will be conveyed to the pebble crushers. The undersize product from both screens will report to a common pump box, where it will be subsequently pumped to the ball mill circuit. Trommel and deck screen openings will vary throughout the life of the mine, depending on the characteristics of the ore being processed.

Pebble Crushing

Oversize material from the SAG mill discharge vibrating screen will be discharged onto SAG mill oversize conveyor #1. This conveyor will be equipped with a cross-belt self-cleaning magnet to remove ball chips from the ore and prevent damage to the cone crushers.

SAG mill oversize conveyor #1 feeds SAG mill oversize conveyor #2, which also has a second cross-belt self-cleaning magnet and a metal detector so that all ball chips are removed prior to feeding the pebble bin. The pebble bin capacity will be 100 t. SAG mill oversize conveyor #2 will be equipped with a shuttle head so that the feed to the surge bin and crushers can be bypassed directly to the pebble crusher discharge conveyor.

Two belt feeders will extract the pebbles from the surge bin at a controlled rate to feed two 750 kW cone crushers operating in parallel. The cone crushers will reduce the pebbles to 13 mm, and discharge them onto the pebble crusher discharge conveyor. This conveyor will discharge crushed pebbles onto the SAG mill feed conveyor.

A dust control system will be installed with dust pickups at key points in the pebble crusher system.

Ball Milling

The undersize products from the SAG mill trommel and vibrating screen, at an 80% passing size of approximately 3,000 µm, will discharge to the pump box that will also collect used mill cooling water and floor sump discharges. Process water will be added to dilute the resulting slurry product to 55% solids. A single pump will elevate the slurry to a three-way gravity splitter that will distribute the slurry between the three ball mill circuits. The splitter will be designed to split flow evenly between one, two, or all three ball mills, depending on the ball mill operating mode. An installed, spare, SAG discharge pump will be provided.

The three ball mill circuits are identical and will operate independently. Slurry from the gravity splitter will flow to the cyclone feed pump box associated with each of the ball mill circuits, where it will be combined with ball mill discharge and makeup water. From the pump box, the slurry will be pumped by a single pump to a cluster of ten 840 mm (53") cyclones. Cyclone overflow, at 80% passing 200 µm, will flow by gravity to the rougher flotation circuit, and cyclone underflow will be piped into the feed spout of the respective ball mill. The cyclone underflow density, maintained at about 70% solids, will enter the ball mill through a feed spout, through which grinding balls and reagents will also be added. The three ball mill circuits will share an uninstalled spare cyclone feed pump.

Each ball mill will be 7.9 m diameter x 11 m EGL (26' x 36') and will be driven by 15 MW dual-pinion low-speed synchronous drives. The ball mills will operate at approximately 32.5% ball load, and generally will draw approximately 14.4 MW, although the ball mill will be structurally designed for a ball load of up to 40%. Ball mill product will leave through the discharge trunnion equipped with a reverse spiral to retain the grinding balls.

Ball Handling and Grinding Media Addition

Approximately two weeks' supply of grinding media will be kept in three bins: one bin will contain 5" (127 mm) balls for the SAG mill, and two bins will contain 2.5" (63 mm) and 3" (76 mm) balls for the ball mills. The balls will be direct-dumped from haul trucks into the bins.

The SAG mill grinding media will be added directly onto the SAG mill feed conveyor at a targeted addition rate. Ball mill grinding media will be transferred from the storage bin to each individual ball bin via a series of small conveyors.

Layout and Maintenance

The SAG and ball mills will be arranged in two grinding bays, and each will be serviced by a bridge crane, sized for both construction and maintenance activities. Other than lift wells above the pumps and screens, the operating floor will be of concrete construction to allow for forklift access for maintenance. The SAG mill will be re-lined through the feed trunnion, and the ball mills will be re-lined through the discharge trunnions. There will be one dedicated liner handler for the SAG mill, and one for the three ball mills. Jib cranes, located adjacent to each mill, will serve the liner handlers. New and used liners and other maintenance items will be moved around the concrete operating floor using a forklift. The hydraulic bolt removal tools will be transported by their own mobile gantries.

The SAG grinding bay crane will also service a large maintenance shop adjacent to the SAG mill. A smaller crane will service all three ball mill cyclopacs.

17.1.6 Rougher Flotation and Rougher Tailings Handling

The cyclone overflows from the three ball mills, at 34% solids, will be collected in a common discharge distributor that will subsequently split the slurry into two streams feeding the two banks of rougher flotation cells. Each bank will consist of eight 300 m^3 rougher flotation cells, providing 23 minutes of residence time. The rougher concentrate from the two banks will be combined in the regrind cyclone feed pump box. Rougher tailings will report to a final tails collection box, where they will be joined by floor sump discharges and other discharge streams from the mill. The combined streams from the final tails collection box will be pumped to either the two-stage tailings cyclones or directly to the TSF. During summer months the underflow from the second stage cyclone will be used for tailings dam construction, while the overflows from both cyclone stages will be piped into the TSF.

Each ball mill cyclone overflow stream will have a proportional sampler, which will feed a single triplex particle size analyzer. From the analyzers, samples will go to the on-stream analyzer system that will provide a control assay for each stream and a filtered shift sample for laboratory analysis.

17.1.7 Regrinding

The four 1,125 kW tower regrind mills will operate in close-circuit with a single cluster of fourteen 380 mm (15") cyclones to achieve a product grind of 80% passing 40 µm. A portion of the cyclone underflow will be split off to an enhanced-gravity concentrator that will capture any coarse metallic gold that otherwise could build up in the circuit.

Gravity concentrate from this concentrator will flow by gravity to the third cleaner concentrate pumps for transport to the final concentrate thickener. The remaining cyclone underflow will be split four ways to feed the individual tower mills by gravity. A sampling system and particle size monitor will be installed on the cyclone overflow to provide a rougher concentrate sample and control of the regrind mills. The cleaner flotation bay crane will be used to charge the regrind mills with grinding media via a ball bucket and magnet.

17.1.8 Cleaner Flotation and Cleaner Tailings Handling

The reground rougher concentrate in the cyclone will overflow by gravity to the first cleaner flotation circuit, which has a bank of six 100 m^3 tank cells. The first four cells in this circuit will be cleaners. They will be pulled carefully to begin the upgrading process, and their concentrate will be collected and forwarded to the second cleaners. The last two cells will be operated as scavenger cells to float low-grade particles. The scavenger concentrate will be recycled to the regrind mill feed for further size reduction.

Tailings from the first cleaner-scavenger cells will flow by gravity through a dedicated line to the TSF for sub-aqueous deposition. This mode of cleaner tailings deposition allows for total saturation with or covering by water upon closure to minimize the risk of acid generation in the TSF.

The second and third cleaners will consist of a total of eight 40 m^3 tank cells – five in the second cleaners and three in the third cleaners. The first-cleaner concentrate will feed the second cleaners, the second-cleaner concentrate will feed the third cleaners, and third-cleaner concentrate will be collected as final concentrate product. Each cleaning stage will produce a progressively higher-grade concentrate with the third cleaners generating a concentrate averaging 26% copper. Third-cleaner tails will flow by gravity to the second cleaners, and second-cleaner tails will be pumped to the first-cleaner feed.

Concentrates from the cleaner stages will be transferred using double-suction vertical pumps. Spray bars will be installed to wash the froth on the second and third cleaners to minimize non-sulphide slimes reporting to the concentrate.

Layout and Maintenance

There will be two parallel bays within the flotation area, each serviced by an overhead crane. The cleaner area crane will be sized to service the regrind mills and the concentrate pumping system. There will be a lay down area in the middle of the plant for maintenance work on flotation machines and regrind mill screws. In the centre core of the building, electrical equipment will occupy the lower two floors while offices, washrooms and the central control room will occupy the top floor. Other electrical equipment will be installed to the south of the reagent area.

Reagent Mixing and Receiving

Reagents will be received, stored, and mixed in a separate building adjacent to the flotation area of the plant. An enclosed utilidor will connect the reagent building and flotation area of the plant. The reagent facility will include equipment for mixing and distributing potassium amyl xanthate (primary collector), a secondary collector if required, lime, flocculant, methyl isobutyl carbinol (frother), and a fine particle dispersant. Lime slurry will be circulated through the plant via a pressurized loop, while each of the remaining reagents will be circulated to its head tank in the plant and distributed from that central point.

17.1.9 Concentrate Thickening and Pumping

Third-cleaner concentrate will be pumped to a high-rate thickener, where it will be flocculated and thickened to 60% solids. Thickener overflow will be recycled to the concentrator for use within the flotation circuit, and thickener underflow will be pumped to two agitated storage tanks. The tanks will provide surge capacity to smooth out fluctuations in feed rate to the pipeline, which will have a limited range of operating flow rates.

17.1.10 Ancillary Services

The assay laboratory and metallurgical laboratory will be located near the maintenance shop adjacent to the SAG mill circuit. Offices for mill operations and maintenance staff will be located in the upper portion of the plant between the flotation and grinding modules.

17.1.11 16.3.12 Utilities

Process, fire, and freshwater systems will be installed at the West More plant site. Water will be reclaimed from the tailings storage facility and pumped back to the flotation and the grinding areas for reuse in the process. Fresh water will be supplied by a series of wells in the vicinity of the mill.

17.2 Product Handling

Two concentrate pipeline options were considered:

- From the West More concentrate storage tanks to a filter plant at Stewart
- From the West More concentrate storage tanks to a filter plant at Bob Quinn (Km 8).

The Bob Quinn/Km 8 option was selected for the GCMC 2011 pre-feasibility study design. Concentrate from the West More concentrate storage tanks will be pumped into a premix tank, where it will be mixed with water to the density and viscosity required by the pipeline system (in the range of 52% to 57% solids by weight). The concentrate slurry will be pumped from the pre-mix tank by a centrifugal pump system into a PD pump that will transfer material approximately 70 km through the concentrate pipeline to the Km 8 filter plant.

The pipeline was designed to transport 150 t/h (3,600 t/d) of copper concentrate (dry basis) at 55%. With a solids specific gravity of 4.15, the resulting design flow rate is 159 m^3/h. At this flow rate, the operating velocity is approximately 1.5 m/s, which is above the minimum allowable operating velocity. Ausenco PSI has recommended an operating range between 50% to 55%, which is typical for a copper concentrate slurry pipeline.

The pipeline will be 219 mm (8.625 inch) diameter API 5L Grade X-65 carbon steel and lined with HDPE. It will be buried with a minimum of 1.6 m cover from the mine site to the filter plant with the exception of short above-ground sections at bridge crossings. The depth of the pipeline will provide protection from freezing during short-term shutdowns (72-hour minimum) during periods of extreme cold temperatures.

Copper concentrate slurry will be pumped overland to a filter plant located at Km 8. Concentrate slurry, at approximately 55%, will arrive at the agitated concentrate receiving tank in the filter plant at an average flow rate of 117 t/h solids (dry). A choke station at the end of the pipeline will dissipate most of the pressure in the concentrate before it is directed to the receiving tank. If there is insufficient concentrate available to maintain the minimum velocity in the pipeline, batches of water will be pumped through the pipeline, and routed to the water treatment plant (WTP) instead of the concentrate receiving tank.

The filter plant will consist of four 120 m^2 pressure filters, configured to run independently. The filtration rate is estimated at 388 kg/h per square meter of filter area.

Concentrate slurry solids will have a size distribution of 80% passing 40 µm, 50% passing 20 µm, and a specific gravity of 4.28. Each filter will have its own dedicated horizontal centrifugal slurry feed pump. The filters will operate in a batch mode. The feed pump will run to fill each filter until the maximum packing pressure is attained. This will be followed by pressing the moist cake by inflation of the filter diaphragm to squeeze out excess moisture. Dry air will then be blown through the cake to further remove the moisture. Finally, the filter cake will be discharged at nominal 8% moisture.

The filtration cycle time will be approximately 13.5 minutes, comprising slurry feeding, cake pressing, air blowing, cake discharge and cloth washing. Filter cake from all four filters will be discharged to an 1800 mm-wide belt feeder, and then to a transfer conveyor that will transport it to the covered concentrate storage shed. To prevent overloading of the belt feeder and concentrate transfer conveyor, operation of the filters will be staggered in cycle time, so that only one filter will discharge at any time. Filtrate and flush water, with low solids concentrations, will be pumped to a filtrate thickener via a 3,000 m^3 filtrate surge pond. Underflow from the thickener will report to the concentrate storage tanks and the overflow will report to the water treatment plant lime reactor.

The four concentrate filters will discharge the de-watered concentrate onto a belt feeder that will meter the concentrate onto a transfer conveyor. The concentrate will be then transferred to the 10,000 t capacity concentrate storage shed. The concentrate will be reclaimed by front-end loader and discharged into 50 t side-dump B-train trucks.

To meet regulatory requirements prior to discharging to the environment, filtrate water and flush water from the pressure filters will be treated for discharge into the Iskut River. The water will be treated to reduce total suspended solids, total dissolved solids and dissolved metals.

Facilities located at Bob Quinn will receive process water from the water treatment plant, and fresh water will be sourced from wells in the vicinity.

17.3 Energy, Water and Process Materials Requirements

The process requirements for energy, water, and process materials are discussed in Section 21.2.4.

17.4 Comment on Section 17

In the opinion of the AMEC QPs, the following conclusions are appropriate:

- The Galore Creek Project will use conventional mineral processing equipment to produce a marketable copper concentrate
- Run-of-mine (ROM) ore will feed a single gyratory crusher in the Galore Creek Valley. The crusher size selected would be close to its operating limit but can meet the design capacity of 95,000 t/d
- Crushed ore will then be transported via three overland conveyors from Galore Creek Valley through the access tunnel to a single coarse ore stockpile near the mill site. The three x 1400 kW drives mentioned in the GCMC 2011 pre-feasibility study are potentially marginally sized. Dynamic analysis will play an important role in the design of the crushed ore conveyor, and potentially could change the required operating parameter
- Apron feeders will reclaim ore from the coarse ore stockpile to feed the SAG mill. Having the take-up at the tail end of the conveyor may be more suitable than the current design
- SAG discharge material will be screened. SAG discharge will be split between three ball mill circuits in closed circuit with hydrocyclones. The hydrocyclone overflow, with a target 80% passing size of 200 µm, will report to flotation for further processing
- The flotation circuit will consist of two parallel rougher banks, with the rougher concentrate reporting to regrinding. The gravity concentrate will report to the final copper concentrate stream. The remainder of the flotation circuit will consist of three stages of cleaning utilizing mechanical tank-type flotation cells with forced air. Third-cleaner concentrate will report to a concentrate thickener for dewatering. Rougher tailings will report by gravity to the tailings storage facility, either directly or through a hydrocyclone system that will produce a coarse sand product for tailings dam construction. Cleaner tailings will be deposited sub-aqueously as a separate stream in the tailings storage facility
- Thickened concentrate will be pumped approximately 71 km to a filter plant located near the junction of the mine access road and Highway 37. Filtered concentrate will be loaded onto trucks for transportation to the port facility at Stewart, BC
- Mill reagents, grinding steel, and maintenance supplies will be delivered to the site by transport truck and stored within the mill as required
- The filter plant will be remote from the process plant at Km 8. Copper concentrate will be dewatered using four pressure filters to produce a filter cake with

approximately 8% moisture; the filter cake will be transported to a port facility at Stewart

- Concentrate from the storage facility at Stewart will be reclaimed and loaded onto ships using a dedicated shiploading facility
- Process water for the mill facility will be reclaimed from the tailings pond, with minimal fresh makeup water being supplied by wells located in the vicinity. The wells will also be used for the production of potable water, mixing of reagents, and other uses
- Process tailings will be stored in the West More Valley tailings facility
- The process design is based on the metallurgical testwork and is appropriate to the grind, flotation and recovery characteristics defined for the different ore types.

18.0 PROJECT INFRASTRUCTURE

Unless otherwise noted, the information in this section was prepared by GCMC and its third-party contractors. AMEC has reviewed the information as noted in the text. Lemley International (Lemley) reviewed the tunnel construction details.

18.1 Infrastructure and Design Considerations

The logistics of moving material, equipment and personnel to the site are influenced by several limiting factors. Factors which will affect Project logistics include:

- Weather conditions prevailing during the seasonal construction schedule, including late thaw or early freeze-up
- Tunnel construction schedule
- Availability of transport and equipment
- Geohazards, including flooding, landslides and avalanches
- Availability of port facilities.

The capacity and size limitations of road bridges along the route to site have been considered in the selection of equipment and components incorporated in the Project.

The size of the tunnel accessing the Galore Creek Valley has been based on the size of components that have to be transported into the valley during construction and operations. Early works required in the valley prior to tunnel breakthrough are based on the use of helicopter support to move equipment and personnel into the valley.

The civil scope of work for the Project includes site preparation and earthworks, overland piping, and underground utilities for Area A (mine site in the Galore Creek Valley), Area C (West More Valley process plant site), Area D (tailings area), and Area F (filter plant site at Bob Quinn), along with the temporary facilities required during the construction period.

The underground utilities include freshwater pipelines, firewater pipelines, sanitary collection pipelines and potable water pipelines. These pipelines will be buried 3 m underground, as frost penetration has an average depth of 1.5 m. Where possible, pipelines will be buried in a common trench to minimize excavation and backfill.

Where required in each of the areas, stormwater diversion channels will be excavated around the various facilities to redirect runoff to the sedimentation control systems.

Gradients on all normal access roads in all areas will not exceed 8%. The gradient on the conveyor maintenance access will not exceed 24%; where a 24% gradient occurs, the area will be restricted to specific operating and maintenance personnel.

Design of the fire protection system is in accordance with the National Fire Protection Association (NFPA). A buried firewater loop will serve yard hydrants, as well as buildings and structures with supplies for automatic sprinkler systems and standpipe systems. The sewage from the camps and recreational centres, truckshop, process plant and office buildings will be collected by gravity and sent to nearby packaged sanitary sewage treatment plants.

18.2 Waste Rock Facilities

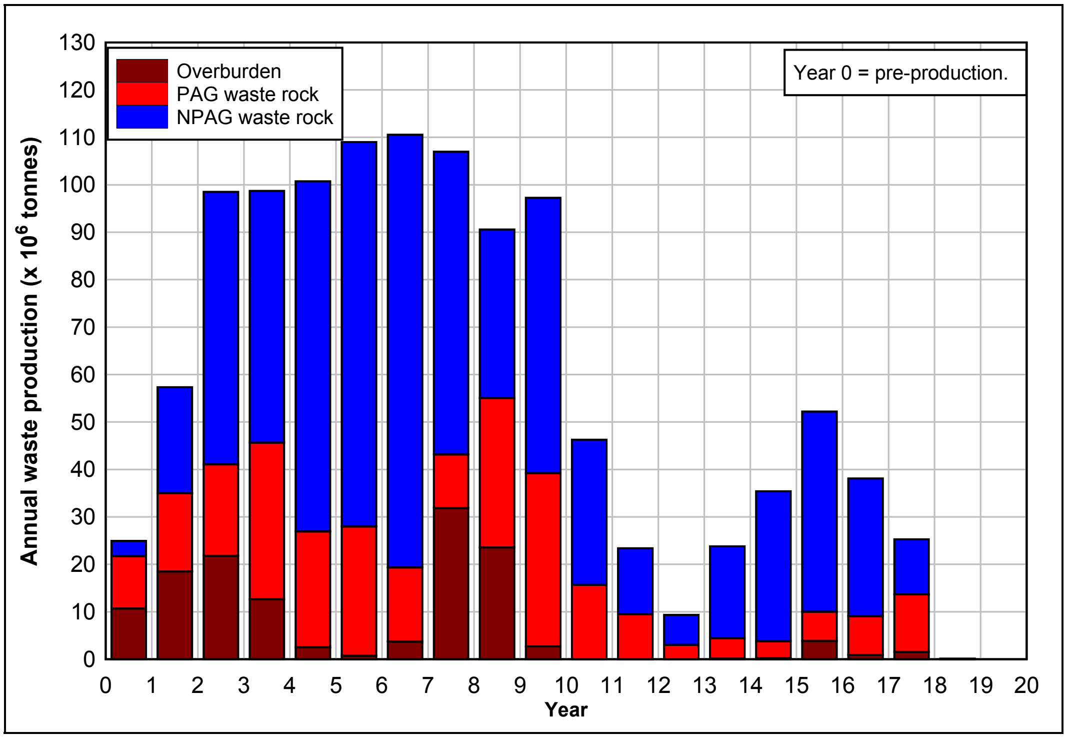
Mine waste will comprise the following:

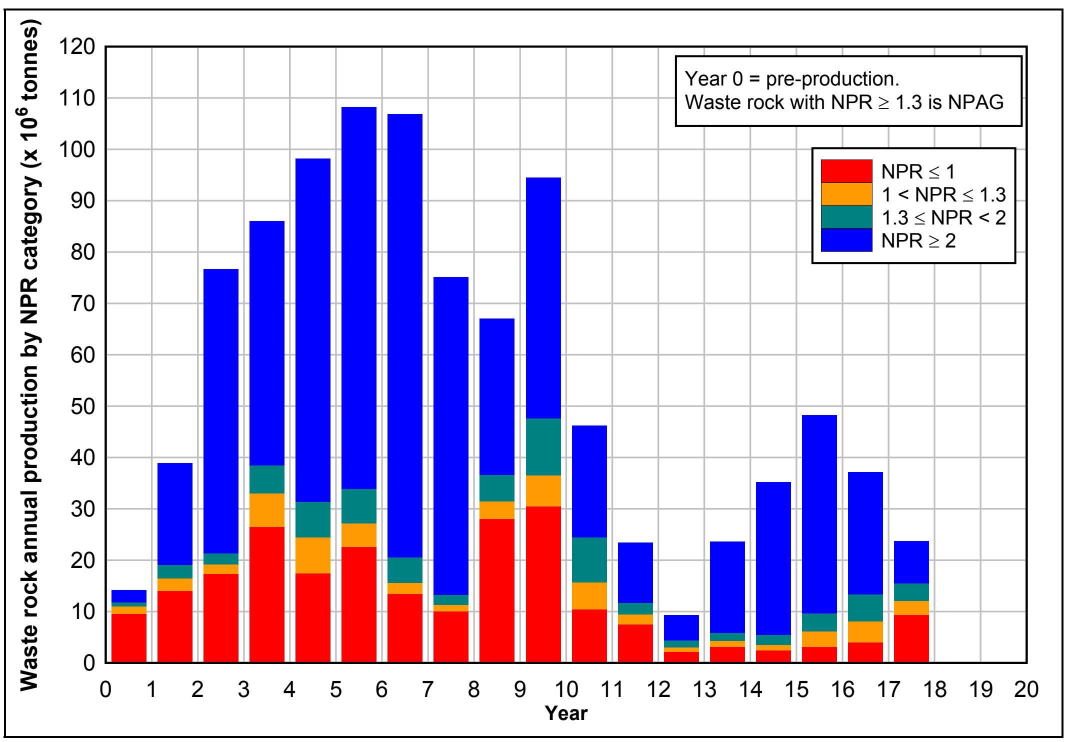
- Overburden soils – primarily till, but also including glaciolacustrine soils, colluvium, fluvial and glacio-fluvial deposits and, in some areas minor organics
- Waste rock, which is categorized in terms of its physical (rock quality) and geochemical characteristics as follows:
  - Geochemical
    - Non-potentially acid-generating (NPAG)
    - Potentially acid-generating (PAG)
  - Rock quality
    - Broken rock
    - Stick rock.

The distinction between broken and stick rock (refer to Section 10.3.2 for a definition of the terms stick rock and broken rock) is of relevance to waste rock dump design and construction, and use of waste rock as a construction material, because the broken rock comprises smaller rock sizes, and is of lower shear strength, and of lower hydraulic conductivity.

The mine plan waste rock and overburden produced over the projected 18.5-year mine life (including one year of pre-production) is summarized in Figure 18-1. The waste schedule is plotted in terms of annual tonnages by waste type (overburden, NPAG waste rock, and PAG waste rock) and by waste rock NPR category. Figure 16-2 shows the waste by type. Of the 723.9 Mt of NPAG waste rock, 75.8 Mt (about 10.5%) lie within the range of $1.3 \leq NPR < 2$.

Figure 18-1: Waste Tonnes by Type





Waste rock that is classified as PAG will be maintained permanently submerged upon closure, behind the closure dam, to limit oxidation and thus prevents generation of acid rock drainage (ARD). Waste rock that is NPAG need not be, and will not be, submerged upon closure.

Five waste storage facilities have been designed as indicated in Table 18-1; the location of these facilities was indicated in Figure 16-1.

A waste dump classification scheme was assessed to provide a numerical dump stability rating, and indicated that for the dumps with moderate to high stability issues (PAG – moderate; valley NPAG – high).

Table 18-1: Waste Storage Facilities

Facility Name	Storage	Capacity
East Fork NPAG Phase 1	87 Mt	design capacity
Valley NPAG.	720.5 Mt	design capacity
PAG dump to El. 655.5 m	227.2 Mt	114% capacity
Closure dam	52 Mt	per design
Southwest pit (PAG rock)	27 Mt	full capacity
West Fork pit (PAG rock)	34.8 Mt	92% capacity

18.2.1 East Fork NPAG Facility

The East Fork NPAG dump is planned to accommodate waste rock and overburden soils and have a waste volume of about 43.5 Mm^3 (87 Mt).

The extent of the dump is constrained by the proposed conveyor causeway embankment and the main valley access road to the west, the tunnel portal to the south, and by the main diversion channel to the east. GCMC has evaluated the potential for a second phase for this dump that would involve a relocation of the conveyor, access road, and other infrastructure and facilities following that corridor. However, at present, GCMC considers that mine waste storage requirements for the Project can be met without requiring an expansion of this dump. The additional capacity of the second phase of the East Fork Dump corresponds to a waste volume of about 75 Mm^3 (150 Mt). GCMC has identified a total East Fork NPAG dump capacity of 237 Mt, of which only 87 Mt is required under the mine plan and waste rock disposition scheme envisaged in the GCMC 2011 pre-feasibility study.

The East Fork Dump will be constructed to a maximum height of about 90 m.

18.2.2 PAG Dump

The PAG rock dump will be located at the base of the main north-south portion of Galore Creek Valley, allowing the dump to be flooded behind the closure dam to achieve permanent submergence of the PAG waste rock. This will be the means of mitigating acid rock drainage (ARD) concerns associated with that rock. Geochemical testing indicates there to be a lag time of decades between exposure (mining) of PAG waste rock and the onset of ARD meaning that such rock need not be flooded until closure.

To the crest elevation of 655.5 m the dump would provide a storage capacity of about 230 Mt. The PAG rock will be capped with a 5 m lift of NPAG rock up to 660.5 m elevation with the portion against the east valley slope left somewhat lower, to form the new channel for Galore Creek upon closure. The level of saturation of the PAG rock will be controlled by the spillway invert of the closure dam.

Portions of the PAG rock dump will be covered by the Valley NPAG dump. The PAG dump will be advanced from south to north, although it may be necessary, from a stability perspective, to advance the dump in lesser lifts, staggered so as to provide a flatter overall north-facing slope.

Large sizes of NPAG stick waste rock will be used for armouring of the final channel for Galore Creek once the main diversion channel has been decommissioned.

18.2.3 Valley NPAG Dump

The Valley NPAG dump will be situated atop the PAG dump, will tie into high ground on the west valley slope, and has a design capacity of 706 Mt, raised to crest at 870 m elevation.

The dump cannot be advanced across the valley to tie into high ground on the east side of the valley as it is necessary to leave a low area there for the reconfigured Galore Creek at closure, once the main diversion channel has been decommissioned, and the main diversion channel, on the east valley slope, must be operational until site closure and reclamation has been achieved. The Valley NPAG dump will progress by lagging the south to north advance of the PAG dump. The Valley NPAG dump will have to be constructed in controlled lifts, with foundation piezometers to monitor foundation pore pressure response to evaluate rate loading effects, and additional investigations are required to better define the presence of glaciolacustrine silt and clay soils in the foundation. Stability issues around the Valley NPAG dump can be managed by appropriately staging the progression of the dump with the valley-bottom PAG dump, which provides buttressing.

The Valley NPAG dump will cross a significant gully (and debris flow hazard) along its northwest perimeter. A coarse rock flow-through drain will be required to infill this gully. Further, select large waste rock will be required along the northwest toe of the dump in order to provide a secondary drainage course, in the event of large storms or debris flow events, to prevent scour of the dump toe and subsequent undermining of the dump.

18.2.4 Closure Dam

The closure dam will accommodate about 40 Mt of NPAG waste rock and 12 Mt of waste till from the pit stripping operations.

18.2.5 In-Pit Storage

In addition to the 200 Mt capacity of the PAG dump, there will also be the storage for PAG waste rock within mined-out portions of the open pits. The Southwest pit will be mined out by the end of Year 2, while completion of the West Fork pit is scheduled for the end of Year 12. Based on the spill elevations for these pits (elevation at which water from the pit lake will spill), the volumes of the Southwest and West Fork pits would be 18.9 Mm^3 and 24.6 Mm^3, respectively.

It is assumed that the maximum level of PAG waste rock within the pits would be 20 m below the pit rims. This would yield the following, likely conservative, waste rock storage capacities for these two pits:

- Southwest pit – 27 Mt (available beginning in Year 3)
- West Fork pit – 34.8 Mt (available beginning in Year 13).

When mined-out, the West Fork pit is expected to be able to accommodate all PAG waste rock scheduled from Year 13 through to the end of the mine life.

GCMC has identified the potential opportunity to place up to about 70 Mt of waste rock within the mined-out portion of the Central pit following completion of Central pit phase 2 mining. However, such placement would affect plans for use of that portion of the pit for dewatering sumps and pump stations. At present, there are no plans for PAG rock backfilling of the Central pit, and this represents an opportunity to be explored in a feasibility-level study.

The total tonnage of waste rock identified as PAG is about 289 Mt (144.5 Mm^3). As such, a combination of 200 Mt of PAG rock within the valley-bottom PAG dump (to crest at 650 m elevation), and 61.8 Mt within the mined-out Southwest and West Fork pits, would result in a PAG rock storage shortfall of 27.2 Mt. This could be made up by raising the crest elevation of the valley-bottom PAG dump from 650 m elevation to about 655.5 m elevation, with less of a raise if some in-pit storage can be achieved within the Central pit.

18.3 Tailings Impoundment Management

The West More tailings facility will be located at the upper limits of the More Creek watershed at elevations above 1,100 m.

The proposed configuration of the West More tailings facility includes three dams: a Main Dam and two saddle dams, termed the East and West Saddle dams. The dams were designed by AMEC (2011a) and will accommodate up to 678 Mt of tailings, although storage for only 509.3 Mt is required for the mine plan incorporated within the GCMC 2011 pre-feasibility study. A starter dam (for the Main Dam) approximately 55 m in height will be required to provide a two-year starter impoundment. As the starter dam blocks the course of West More Creek, a cofferdam will be required for construction.

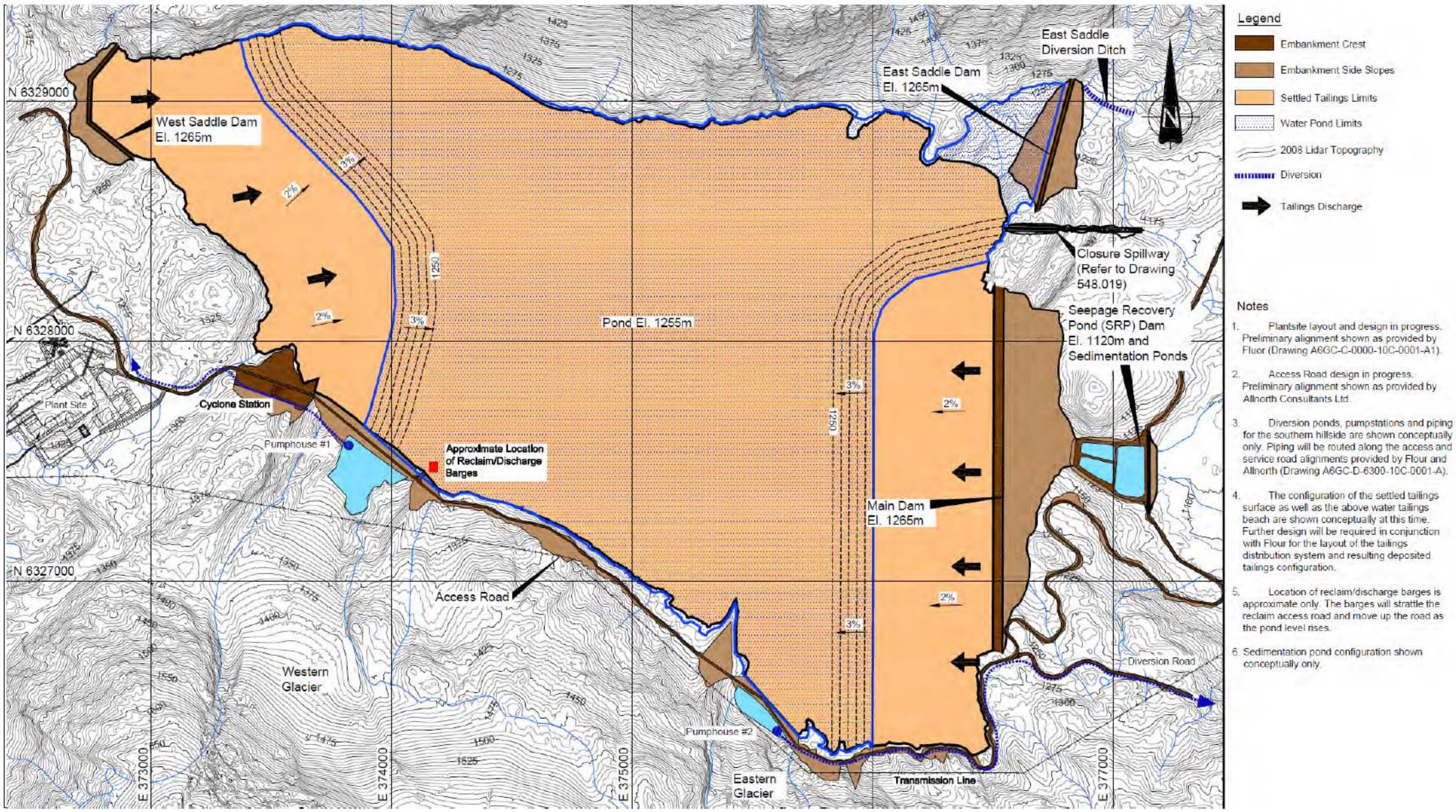
Figure 18-2 shows the proposed facility layout.

The Main Dam is planned to be raised in annual increments (initially in the downstream direction and then using centerline construction) to provide sufficient storage volume for the tailings and process water produced via milling processes. The starter dam will be constructed using locally-borrowed materials for the various embankment zones, which will then change to include cyclone sand construction, using the rougher tailings feed, of the downstream shell throughout operations. Precedents for similar dam configurations, including centerline construction using cyclone sand, are the tailings dams at the Kemess and Highland Valley Copper mines.

Construction of the East and West Saddle dams will not be required until later in the life of the facility and will be accomplished in two construction phases each. These dams will not require cofferdams due to their location on topographical divides.

A seepage recycle pond including a sedimentation area to manage the decant water associated with cyclone sand construction will be constructed downstream of the Main Dam. Seepage and run-off from the East Saddle Dam will be directed to the seepage recycle pond via ditching (or via pipeline) where it will be pumped back into the tailings impoundment, or potentially discharged if the water quality is suitable for direct discharge.

Figure 18-2: Proposed Tailings Impoundment Layout Plan



The main site access road is planned to traverse the southern perimeter of the tailings impoundment passing the termini of two glaciers. The current construction scheme involves construction of the main access road above the final impoundment limits during preproduction with accommodation for a lifeline corridor (power, diesel, and concentrate lines).

The process water system has been designed such that the reclaim system is to be at least 99% efficient, meaning that 99% of the volume of process water leaving the plant with the tailings slurry will be returned to the plant via reclaim, with only 1% freshwater makeup required. Process water reclaim will be via a floating reclaim barge located along the south side of the impoundment adjacent to the toe of the western glacier, readily accessible by the access road. The reclaim pipeline will extend to the plant site along the access road.

A discharge schedule was developed for the discharge of surplus water. The object of the discharge schedule will be to maintain a relatively constant annual volume of free water within the impoundment of approximately 20 Mm^3 to 30 Mm^3 to facilitate the yearly dam raising schedule and maintain above-water tailings beaches in front of the Main and West Saddle dams. Tailings discharge operations must be optimized to generate acceptable water quality around the reclaim and discharge barges at all times, while maintaining significant above water beaches at both the Main and West Saddle dams. In order to achieve the required beach configurations for construction it is anticipated that at least 50% of the total tailings tonnage will need to be discharged from the east side of the impoundment along the main tailings dam.

Submergence of the cleaner tailings is all that will be required to mitigate ARD risk for the West More tailings impoundment.

18.4 Tailings Facility Closure Aspects

Following the cessation of active mining activities, the site will be reclaimed to return the site to as natural a state as is practicable. The primary objective for reclamation will be to achieve self sustaining landforms that enhance the local ground stability and reduce erosion potential on and adjacent to the tailings impoundment while creating a permanent water cover over the impounded tailings to elevation 1,255 m.

It is envisioned that construction of the downstream (cyclone sand) shell of the main tailings dam will be completed about two years in advance of active mining in order to facilitate reclamation activities while operating revenue and equipment are still available.

The closure work on the downstream side of the dam will include resloping and contouring of the downstream sand shell to facilitate drainage. The final (design) slope for dam stability is 2.5H:1V.

A key element of the designs of the tailings dams is that at closure, they will be separated from the closure water pond via an above-water tailings beach. This separation is a significant advantage in terms of long-term dam safety.

At closure, an open-channel spillway will be required to discharge water from the impoundment to West More Creek. The spillway will be designed to route, without overtopping of the perimeter dams, the inflow from a probable maximum flood (PMF) of critical duration.

Ongoing monitoring over the long term will be required following closure and reclamation of the facility. The ongoing stewardship of the facility will include monitoring of critical dam instrumentation (to be selected for operations during the feasibility or detailed design studies), performance of annual inspections and periodic dam safety reviews as per CDA Guidelines (2007). Instrumentation will be maintained, and periodically replaced as required, over the long-term to satisfy the monitoring requirements for the safety and performance of the three dams.

At mine closure, the flow regime will return to a configuration that more closely reflects the pre-mining conditions of West More Creek. All the diversions will be breached with the exception of the East Saddle diversion ditch that will be retained as a permanent closure feature. All run-off will be routed through the impoundment and back to West More Creek through the closure spillway.

18.5 Water Management

The Project site is an area of high precipitation, averaging 3,000 mm/year over the elevation range of the watershed, with average annual run-off estimated at 2,340 mm per year, depending on elevation, glaciation of the watershed and summer air temperature.

More than 60% of the annual precipitation falls as snow during the winter months, while approximately 80% of the run-off occurs between May and September. Mean monthly flows range from about 1 m^3/s in January to about 30 m^3/s in the peak run-off month of July, during which about 23% of total annual run-off typically occurs.

Estimated peak flow in Galore Creek with a one-in-two-year recurrence interval is 110 m^3/s.

Much of the Galore Creek watershed lies up-gradient of the proposed area that will be affected by mining operations, and diversion of this water represents a major challenge for the construction and operation of the Project.

The layout of the key water management facilities is presented on Figure 18-3.

18.5.1 Main Diversion Channel

The main diversion channel will extend from the upper end of the East Fork drainage to Friendly Creek and will divert the runoff generated by a large fraction of the overall Galore Creek watershed around the proposed mine area. The runoff from the areas marked as East Fork, East Slope, and East Slope North on Figure 18-3 will be diverted by this channel.

The Main Diversion Channel is designed to convey the peak flows resulting from a one-in-50-year return period, 24-hour duration storm event from the 65 km^2 catchment area, meaning that the channel must accommodate peak flows of up to 135 m^3/s.

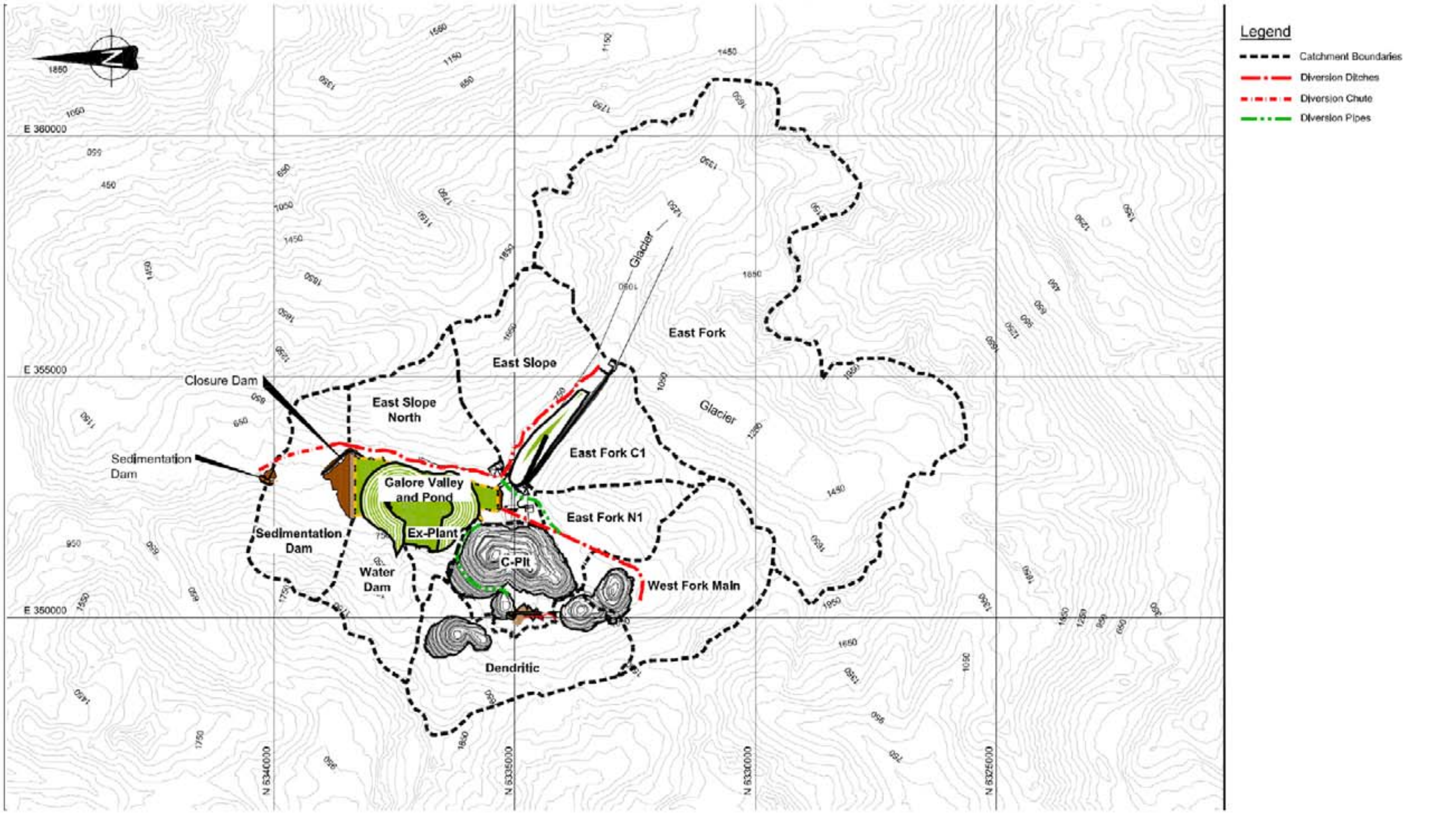
An access road will be built downslope of the diversion ditch to allow inspection and maintenance of the channel and for access to the closure dam and sedimentation dam. The main diversion channel will cross a number of drainage courses. Within many of those, construction of the channel in cuts will be impractical, and thus fills will have to be constructed to ford those drainages. Overflow protection will be required.

The main diversion channel must be constructed at a sufficient grade to bypass the right abutment and spillway of the final configuration of the closure dam. From that point northwards, a chute will be required, with a drop from approximately 700 m elevation (abutment of the closure dam) to about 440 m elevation, at the base of Galore Creek Valley downstream of the sedimentation dam. A robust lined chute, and extensive energy dissipation structures, will be required.

A detailed reconnaissance of Friendly Creek will be required at the feasibility study stage to determine if there is the potential for the main diversion channel to discharge into Friendly Creek and avoid the need for a large (approximately 2 km long, at 13% grade, with a channel base width of about 20 m) chute structure.

The main diversion channel is planned to be decommissioned at closure, once the closure dam and its spillway are complete to their final design configurations.

Figure 18-3: Water Management Structures Layout Plan



Note: Figure courtesy GCMC, NovaGold and Teck

18.5.2 Sedimentation Dam

The planned sedimentation dam will be located on Galore Creek downstream of the proposed pits (refer to Figure 18-3). The purpose of the sedimentation pond impounded by this dam is to reduce total suspended solids (TSS) in the contact water discharge from this reservoir (via a low level outlet and/or the emergency spillway on its left abutment) to Galore Creek to levels that achieve compliance with TSS criterion specified in the Metal Mining Effluent Regulations (MMER) and BC water quality guidelines.

Background TSS levels in Galore Creek are frequently an order of magnitude or more higher than the MMER regulations, and thus conformance with those regulations, being incongruent with baseline conditions, will prove challenging.

The design storm for a sedimentation pond is a one-in-10-year return period, 24-hour event.

The sedimentation dam will have a low level outlet, which initially will be the diversion tunnel used to divert Galore Creek, along with a cofferdam, to facilitate construction of the sedimentation dam in the pre-production period. The low level outlet will incorporate the diversion tunnel and will be used to maintain operational control of the reservoir level, which will generally be below the spillway invert elevation. Flow through the low level outlet will be controlled through a decant tower structure on the upstream end, which will enable pond level and pond outflow control.

An overflow spillway will be constructed in bedrock on the left abutment of the sedimentation dam. There will be an access road to the sedimentation dam on both abutments.

The GCMC 2011 pre-feasibility study assumes that the sedimentation dam will only be required for operations and the site reclamation phases. Once site reclamation is well advanced, and sediment sources are reduced, the sedimentation dam is planned to be decommissioned.

18.5.3 Closure Dam

The closure dam will be to form the downstream (north) limit of the valley bottom PAG dump, to contain PAG waste rock and keep it submerged to prevent generation of acidic drainage at closure.

The closure dam will be constructed in stages through the duration of the mine life. Stage 1 will be constructed up to elevation 600 m, while Stage 2 will be constructed up to elevation 660 m. The ultimate dam crest will be 115 m wide.

The closure dam will be constructed of an upstream compacted till, filter layer, transition zone and downstream compacted NPAG. Most of the materials for the construction of this dam (NPAG waste rock for the downstream shell, and till for the core) will be obtained from stripping operations at the open pits.

A cofferdam and 5 m diameter, horse-shoe shaped diversion tunnel will be required to initiate construction of the closure dam. Upon completion of the Stage 1 closure dam, the diversion tunnel gate will be closed, and the permanent tunnel plug will be constructed.

Water level control upstream of the closure dam during operations will be achieved via a spillway, a siphon system passing over the spillway, and a floating pump barge that will allow the water level to be drawn down below the elevation at which the siphon system becomes non-functional. The spillway location for the closure dam is a critical issue and will require significant focus and resolution during more detailed studies.

The closure dam and its final spillway will be required in perpetuity, and thus will require ongoing monitoring and maintenance.

18.5.4 Diversion Structures

West Fork Diversion

The purpose of the West Fork diversion system is to divert run-off from the upstream catchments of West Fork Main and East Fork N1 away from the planned Central and Southwest–West Fork pits (refer to Figure 18-3). This system will be constructed in phases commensurate with the advancement of the Southwest and West Fork pits, and will incorporate diversion channels where it is practical to divert run-off upslope of the pits, and in-pit sumps and pump-out facilities where such diversion is not practical.

The diversion channel is designed to divert a one-in-50-year return period, 24-hour duration storm from approximately 12.8 km^2 of catchment area. An access road will be built downslope of the diversion channel to allow for inspection and maintenance of the channel.

The West Fork diversion channel will be decommissioned upon closure.

Dendritic Diversions

The dendritic diversion system will divert run-off from the dendritic drainage area upslope and to the west of the Central pit. Given the terrain and the geometry of the Central pit highwall, a long diversion channel is not viable. Instead, the dendritic diversion system will be formed via a waste rock embankment blocking off the gullies upstream of the pit perimeter.

Low spots upstream of this embankment will be filled in with waste rock and till (from pit stripping), and a channel formed flowing from south to north. This channel will drain into the Central pit lobe sump. Water from the sump will be discharged via outlet pipelines which will be routed onto one of the pit benches (several relocations of the pipelines will be required), then between the Central pit and the NPAG waste dump, before discharging into the valley bottom. Alternatively, if this flow is of suitable water quality, and the pipelines can be extended across the valley to discharge into the main diversion channel, the diverted flow would no longer represent a contact water contribution reporting to the sedimentation dam reservoir prior to discharge.

Water collection structures, with pipeline outlets will also be installed downstream of the embankment to collect leakage from the embankment.

The dendritic diversion system has been designed to divert flows up to a one-in-10-year return period, 24-hour storm event.

The diversion system will be decommissioned and reclaimed upon closure.

18.5.5 Run-off Diversions

Run-off diversions will be created for the Central and West Fork pits. Up-slope of the Southwest and Junction pits, however, runoff diversion channels are not feasible due to the difficult terrain. Run-off must therefore be handled within these pits. This is considered a practical approach, since the catchments upslope of these pits are not large.

The in-pit dewatering system will consist of diversion ditches, event ponds, in-pit collection sumps and pumps for controlled removal of event water. A series of pumps will be installed to transfer the water from active mining bench sumps to surface for discharge of the water into the valley bottom area; flow that will eventually report to the sedimentation dam reservoir.

The in-pit pumping system will be designed to dewater the pit during a one-in-five-year storm event, which could see 4,277 L/s of run-off into the pit.

The in-pit pumping requirements will vary annually throughout each season, and will increase as the catchment area increases with successive push-backs to ultimate walls. As the pit deepens, additional banks of pumps will be added to enable the pumping of event water in stages.

18.5.6 Water Balance

A water balance model was developed for the Galore Creek Valley to quantify water volumes from upstream catchment areas under average, dry and wet year conditions on a monthly basis.

The water balance was evaluated for three scenarios: average, one-in-100-year return period dry year, and one-in-100-year return period wet year.

A monthly water balance was carried out for the proposed Central open pit to ascertain the required design pump-out capacity to maintain the pit acceptably dewatered during mining operations. The West Fork and dendritic diversion systems are planned to limit runoff into the pit, but there will still be undiverted run-off, and direct precipitation and snowmelt, to be managed. The diversion systems were assumed to be have diversion efficiencies ranging from 70% (low flow months) to 95% (peak flow). An inflow to the Central pit of about 481,000 m^3 is predicted to result from the one-in-10-year, 24-hour storm event within the pit and its undiverted catchment area.

18.5.7 Water Quality Considerations

GCMC retained Lorax Environmental Services Ltd. (Lorax) to conduct a preliminary loading model to evaluate the effect of Project reconfiguration on downstream water quality for the Galore watershed.

A mass loading model was developed for the proposed mine site.

For all watersheds, including the East Fork that is upstream from the mineralization, sulphate exceeds the BC water quality guideline in winter, and dissolved Al, As, Cd, Co, Cu, Fe, Mn, and Zn exceed guidelines during the summer. Consequently, site-specific water quality objectives will need to be established during permitting. Upstream background water quality on the Scud River is also naturally elevated above guidelines for a similar list of parameters, again indicating the site-specific guidelines will need to be established.

Various scenarios (or cases) were evaluated, examining the expected water quality during the first 40 years of post-closure and for long-term post-closure (assuming potentially acid-generating rock in the unflooded portions of the pit wall would go acid.

To help identify which source terms warrant closer examination during the feasibility study/environmental assessment work, contribution ratios were determined for the key parameters. For sulphate it was determined that the NPAG dumps and the pit walls were the biggest contributors. For the metals Cd, Cu and Zn, both the NPAG and PAG were seen to be major contributors, until the pit walls acidify, after which the pit walls dominate.

Further refinements of the model are required to validate these expected increases over background. Development of a "Best Professional Judgement" model in addition to the "worst case" model needs to be undertaken to provide a clearer understanding of the expected values, rather than focusing solely on the extreme values.

18.6 Geohazards

Landslide hazards exist along most of the main diversion channel alignment within the East Fork, in the area above the planned Central pit, in particular the dendritic drainage area, and along either side of the north–south portion of Galore Creek Valley, downstream (north) of the confluence of the East Fork and West Fork. The proposed sites for the Project infrastructure avoid the worst of the debris flow hazards in the main north–south portion of the valley. There is also an identified debris flow hazard on the west valley slope of East Fork, immediately above the planned conveyor corridor and main valley access road.

Snow avalanche hazard is ubiquitous throughout the valley, including along most of the alignment of the main diversion channel (BGC, 2006a). Avalanches along the main diversion channel alignment, be they deliberately triggered, or occur in an uncontrolled manner, will result in frequent and near-total blockages of the diversion channel. An active avalanche monitoring and control program will be required in support of the mining operation, including crews and equipment dedicated to monitoring, operation, and maintenance of the main diversion channel and all significant water management facilities within the valley.

18.7 Road

A report for the Galore Creek mine access road was prepared by Allnorth (2010). Allnorth utilized a 2005 report by McElhanney Consulting Services Ltd as a basis for the study, but updated the current as-built road conditions in the road status report and utilized TNR Bridge Construction Limited Partnership to update the report and estimate for the bridges and culverts.

The More Canyon bridge revised budget cost estimate prepared by Buckland and Taylor (2008) was utilized as the base document for the More Canyon bridge design and costing for the GCMC 2011 pre-feasibility study.

The route selected for the GCMC 2011 pre-feasibility study starts at the junction of Highway 37 and will proceed west across the Iskut River, up More Creek, over the More Canyon bridge, around the proposed new tailings location at Round Lake and then down Sphaler Creek to the South Portal. The road plan was included in Figure 5-1.

The initial 8 km of the road will be double lane, narrowing to a single-lane (6 m wide) resource access road. The road is planned to support construction of the diesel supply line, concentrate pipeline, and the power transmission line and provide supplies, equipment, and crew transport during construction and operation of the mine. The road will be constructed with less than 15% grades and an average design speed of 40 km/h. The road is intended to be a low impact road within the utilities corridor.

Bridges and culverts were designed for the 200-year and 100-year instantaneous flood, respectively, with a minimum 1.5 m clearance to the underside of the bridge girders unless additional clearance was required for navigable waters or geotechnical requirements. The bridges and culverts were all rated for a maximum load of 100 t.

Details of the current status of road construction are discussed in Section 5.1.3, together with a location plan showing the proposed access route.

18.8 GCMC 2011 Pre-feasibility Study Tunnel Design

The proposed mine access and ore conveyor tunnel represents a major tunnel project in terms of international tunnelling practice. The proposed tunnel is aligned under high rock cover of more than 600 m over a significant portion (75%) and with a maximum rock cover of 1,250 m. The proposed mine access and ore conveyor tunnel represents a private tunnel where a large volume of traffic is anticipated immediately after breakthrough to facilitate construction of the mine, but where a limited volume of traffic is expected during the normal mining operations. The estimated time to excavate the tunnel in the GCMC 2011 pre-feasibility study is approximately 30 months, but this duration is considered optimistic by Lemley as will be discussed in Section 18.9.

The tunnel design and cost estimate were performed on behalf of GCMC by third-party consultants to GCMC.

18.8.1 Proposed Layout

The requirement for the mine access and ore conveyor tunnel to provide both permanent access for large components of mining equipment for start-up and ongoing operations, as well as conveyor haulage during mining operations, has necessitated the need for a large diameter tunnel.

Given the extensive length of the identified tunnel alignment, for purposes of GCMC's 2011 prefeasibility study it was considered prudent to assume that the most cost-effective approach for the construction of the majority of the mine access and ore conveyor tunnel would be by means of a tunnel-boring machine (TBM). As road access and power is expected to be available at the South Portal, the TBM would commence excavation from that point.

Excavation utilizing conventional drill-and-blast equipment would commence simultaneously from the North Portal in order to reduce the overall construction schedule.

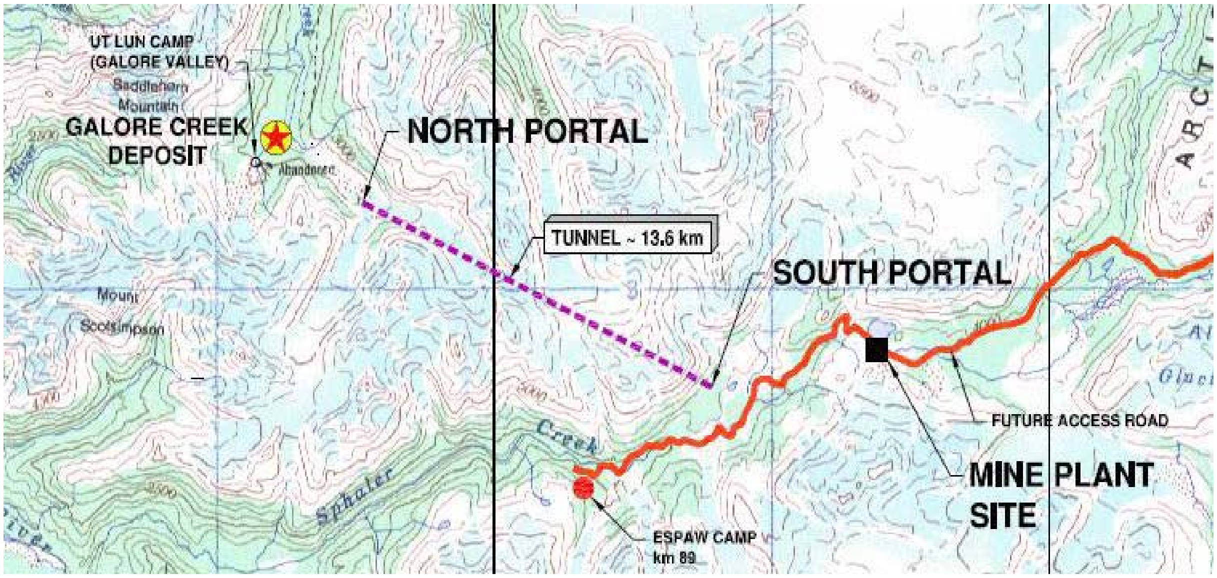
In the prefeasibility study, the tunnel alignment for the mine access and ore conveyor tunnel was based on a layout by GCMC as the most direct route between the Galore Creek Valley and the desired location for the mine processing plant site in the upper reaches of the Sphaler Valley. The proposed route in relation to the access road was shown in Figure 5-1, and is illustrated in more detail in Figure 18-4.

The alignment has resulted in a location for the South Portal along a sub-vertical rock bluff in the upper reaches of the Sphaler Valley, and a location for the North Portal along a rock outcrop in the southeast corner of the East Fork Valley. Both portal locations are deemed to be suitable for construction of the proposed tunnel. The total length of the proposed mine access and ore conveyor tunnel is about 13.6 km. The azimuth of the tunnel alignment is 120º.

The elevations of the South and North portals are 915 m and 720 m, respectively, resulting in a downward grade of 1.5% towards the North Portal.

The proposed tunnel is aligned with a significant portion under a high rock cover that rises sharply at both portals. There are no practical possibilities for intermediate access adits along the tunnel alignment due to the very thick cover.

Figure 18-4: Proposed Tunnel



Note: Map north is to top of figure. Figure is approximately 35 km across, and 20 km top to base. Figure courtesy GCMC, NovaGold, and Teck

18.8.2 Excavation

Tunnel excavation is expected to proceed from both portals in order to minimize the duration of construction. For GCMC's 2011 prefeasibility study, GCMC's third-party consultant estimated typical overall average sustained excavation advance rates of TBM excavation from the South Portal to be about 12 m/d. This is expected to vary from 3 m/d with very poor ground conditions (associated with major faults zones requiring the installation of high capacity support measures) to 20 m/d with very good rock conditions.

The base case option for the GCMC 2011 pre-feasibility study assumed an overall average sustained excavation advance rate for a high-speed drill-and-blast excavation approach from the North Portal of about 6.6 m/d.

Based on the expected rock conditions along the tunnel alignment, the assessments of tunnel stability have indicated strong potential for appreciable overstressing, and in accordance with standard industry practice, an appropriate level of tunnel support will be required for worker safety; to maintain the long-term stability of the tunnel for the expected operating requirements; and to minimize maintenance disruptions during operations.

It is envisaged that initial tunnel support systems comprising rock bolts, mesh, shotcrete, and steel ribs will be installed concurrently with tunnel excavation and that final tunnel support comprising additional rock bolts and any necessary shotcrete lining will be installed after breakthrough and removal of the ventilation duct and all services. This tunnel excavation and support approach is considered to result in the shortest overall construction schedule without jeopardizing the integrity of the tunnel and will maintain worker safety.

A series of tunnel support classes will be designed to cater for the expected variable rock conditions to be encountered along the tunnel.

18.8.3 Camp and Utilities

It is envisaged that a tunnel construction camp would be prepared and located in the Hooly Creek area near the South Portal and the existing Galore Creek Valley camp would be used for tunnel construction works at the North Portal. Laydown areas for shops and offices will be established near each of the portals.

It is envisaged that the mine access and ore conveyor tunnel will be excavated using conventional drill-and-blast methods from the North Portal where four 1000 kW generator sets already owned by GCMC will be available. TBM excavation is planned from the South Portal and for the envisaged size of TBM in conjunction with ventilation and lighting the electrical demand is approximately 8,000 kW (10,000 kVA). For this large power demand, it will be cost-effective to provide an early installation of the power supply to the mine site and allow a feed to be established from Bob Quinn to the South Portal. The potential exists that a grid power supply may not be available in time for commencing TBM excavation. The current cost and schedule estimates assume that grid power for the TBM will be available beginning in the 7th month of TBM operation and that diesel generators will be used up until that time.

18.8.4 Water Considerations

The mine access and ore conveyor tunnel will act as a drain during excavation and operations and will cause groundwater to flow towards the tunnel from the surrounding rock mass along the tunnel alignment. Treatment of tunnel construction water will be required at both the north and south portals during tunnel excavation prior to release into any natural stream courses to meet the requirements of the Department of Fisheries and Oceans (DFO). Large settling / containment ponds will be constructed near each of the portals. These ponds will be maintained, which will include cleaning of sludge materials and disposal to designated spoil sites. Routine environmental monitoring of all treated construction water will be required to verify that DFO standards can be met prior to release. In the GCMC 2011 pre-feasibility study design, at the completion of tunnel excavation, all groundwater inflows will channel via the mine access and ore conveyor tunnel to the North Portal. The long-term estimated sustained groundwater inflows during operations after pre-excavation grouting of the major fault zones and fracture zones during tunnel construction is approximately 235 L/s (20,304 m^3/d).

18.8.5 Acid-Base Accounting

A significant portion of the mine access and ore conveyor tunnel is to be excavated through volcanic rocks and limestones in which no trace amounts of pyrite have been identified based on acid base accounting (ABA) testing undertaken in 2008. The disposal of the spoil created during tunnel excavation, will however, need to be tested and controlled/disposed of at a designated site if the presence of sulphides is established. Some additional ABA testing should be completed as part of any future studies to further assess the low potential for sulphides within the expected tunnel spoil. The total volume of spoil at the South and North Portals is estimated to be about 1.0 Mm^3 and 0.70 Mm^3, respectively. For purposes of the GCMC 2011 pre-feasibility study, it was assumed that all tunnel spoil would be placed in an engineered disposal site near the tunnel portals and that there were no special disposal requirements.

18.8.6 Ventilation

During normal operations, the mine access and ore conveyor tunnel may be partially self-ventilating due to the aerodynamic drag of the operating conveyor. This passively induced ventilation will be affected by weather conditions at the portals which will also induce airflow, or “natural ventilation”. A mechanical ventilation system will be installed to provide positive means of controlling the airflow when the natural ventilation is inadequate, or in the event of a fire. The results of a ventilation analysis for a selected 30 MW fire event indicated that it will be necessary to include 36 to 56 kW jet fans in the tunnel for emergency ventilation purposes. The GCMC 2011 pre-feasibility design includes refuge bays constructed at intervals of 500 m along the tunnel to provide emergency shelter in the event of a fire within the tunnel.

18.8.7 Communications

The tunnel will be equipped with a cellular communications fibre optic line to allow radio communications at all locations within the tunnel to the mine and mine plant dispatch offices. Air quality monitoring will be completed either by means of sensors installed at three locations along the tunnel or by manual testing to detect unacceptable levels of air in order to activate the ventilation system and purge the tunnel.

18.8.8 Traffic Controls

A nominal traffic speed of 30 km/h is foreseen for safe operations, which represents a travel time of about 30 minutes. Traffic will be strictly controlled from mine dispatch, with the requirement of communication call-ins at each 1 km station through the tunnel.

A motor temperature and braking check will be required at the South Portal prior to approval to proceed through the tunnel.

18.9 Lemley International Review of GCMC 2011 Pre-feasibility Study Tunnel Design

Because the access tunnel to the Galore Creek Valley was viewed as a critical path item for the development of the Galore Creek deposit, AMEC requested that a suitably qualified expert firm undertake an endorsement-level review of all pertinent Project technical information for the tunnelling section of the GCMC 2011 pre-feasibility study and provide their professional judgment as to the suitability of that work to meet the requirements of a pre-feasibility study.

Lemley International (Lemley) was contracted by GCMC to perform this review. Lemley's expertise includes tunnel design, tunnel construction and program/construction management of tunnels and other large infrastructure projects. Jack Lemley was the CEO of Transmanche-Link, the tunnel contractor consortium that successfully built the Channel Tunnel between England and France.

Lemley's findings are discussed in the following sub-sections.

18.9.1 Review of Design Considerations

The GCMC 2011 pre-feasibility study design consists of a 13.6 km-long tunnel extending from a South Portal located in a limestone cliff face on the side of a bluff at the upper end of the Sphaler Creek valley, to a North Portal located in a sloping volcanic rock outcrop in the upper reaches of the East Fork of Galore Creek.

The tunnel is designed entirely on tangent (straight), with a bored diameter of 9.5 m, which GCMC's third-party consultant considered the minimum necessary to pass the largest component pieces of open pit mining equipment envisioned over the life of the mine, while providing minimal clearance alongside a structure supporting a 1,500 mm wide belt conveyor and various utilities, and including a 0.55 m radial allowance for lining and tunnel convergence and/or alignment issues.

As currently designed, the alignment would slope downward from the South portal at elevation 915 m to the North Portal at elevation 720 m; a rate of -1.5%. The tunnel passes under steep mountain ridges and peaks surrounding the Galore Creek Valley, resulting in relatively deep cover of 600 m or more over approximately 75% of the alignment, with a maximum depth of 1,250 m.

Based on limited geotechnical investigations performed on behalf of GCMC to date, the tunnel will be in rock throughout the alignment with unconfined compressive strengths (UCS) ranging from a low of around 20 MPa (inferred from testing performed on chipped samples) in a relatively weak shale formation, to 435 MPa in extremely high-strength volcanic rocks. Limestone and silicified, dolomized limestone are also present along the alignment. A total of 19 mapped and inferred faults are predicted.

According to GCMC's third-party consultant, approximately 3 km of the tunnel drive at the southern end of the alignment will be in rock with an average UCS of 150 MPa, with the remaining 10.6 km in rock exhibiting an average UCS of 275 MPa. Included within these sections are a short, 145 m long reach of shale (average UCS = 40 MPa) and a short, 125 m long reach of limestone (average UCS = 60 MPa). Significant groundwater inflows may occur along the various faults, in fracture zones, and through fractures and/or potential karst features in the limestone.

The approach GCMC describes in the GCMC 2011 pre-feasibility study for constructing the tunnel would be to drive headings from both ends, using an open gripper, high-performance, main beam tunnel boring machine (TBM) starting from a 165 m long, conventionally-mined starter tunnel at the South Portal (access road end) and a helicopter-supported, drill and blast heading from the North Portal.

Given that the Project is in the pre-feasibility stage, there are many unknowns with regard to the actual geotechnical and hydrological conditions that may be encountered along the tunnel alignment and the GCMC 2011 pre-feasibility study has mentioned many of those that could affect the viability of the tunnel project including:

- Potential for encountering karstic conditions with associated high groundwater inflows in limestone sections of the alignment
- Potential for encountering hydrogen sulphide gas
- A total of 19 inferred faults, classified as "moderate" to "major", a few of which appear to intersect at the location of the tunnel. The faults have the potential to

exhibit weak ground conditions and high water inflows, which, when combined with high rock cover, can create extremely difficult tunnelling conditions

- Potential for encountering severe overstressing and possible squeezing ground conditions due to tunnel reaches exhibiting low rock strength under high cover (up to 1,250 m)
- Potential for encountering long reaches of very hard and abrasive volcanic rock that can significantly slow TBM advance rates and cause high rates of cutter ring and cutterhead wear with significant associated downtime and expense.

Assessment of whether these or other conditions will be realized in the course of driving a tunnel is normally made by obtaining a sufficient number of borings and conducting a sufficient amount of field mapping beforehand to provide an adequate level of confidence that the ground conditions to be encountered are reasonably well understood before tunnelling begins. However, for the Galore Creek tunnel, the same surface constraints that preclude building a road into the site, which include severe topography, snowpack, glaciers, and weather, will also limit the amount of borehole information and geological mapping data that can be obtained.

To date, only three boreholes have been drilled on, or close, to the 13.6 km-long tunnel alignment; one near each end and one located near the midpoint of the tunnel. At least two separate geological mapping programs, one in the summer of 2008 and another in the summer of 2010, have also been completed for the tunnel, but those efforts were also constrained by year-round snow cover and steep topography. Drilling of an additional three to four geotechnical drill holes is planned for the 2011 summer season which will help considerably toward improving the understanding of the ground conditions affecting tunnel construction.

18.9.2 Review of Timeframes and Productivities for Tunnel Construction

Lemley reviewed the construction schedule as included in the GCMC 2011 pre-feasibility study.

Lemley's opinion is that the GCMC 2011 pre-feasibility schedule is likely to be optimistic and that time should be added to the schedule. Lemley estimates a total of 37 months should be allocated for boring/excavating the tunnel and that a 49-month overall tunnel construction duration is appropriate based on the information available at the time the Lemley review was performed. Lemley is of the opinion that their estimate represents a "most probable" construction duration, such as would result from a formal, quantitative risk assessment based on experienced and unbiased input to the process furnished by seasoned, independent, tunnel professionals. This kind of risk assessment involves a Monte Carlo type of statistical evaluation that provides a most probable overall schedule duration based on the estimated likelihood and estimated schedule impact of a number of individually identified risk scenarios.

Key factors considered in the formulation of Lemley's opinion include the following:

- GCMC's third-party consultant performed a contingency analysis to assess the impact associated with mining a bypass around, and then conventionally mining through, a 135 m-thick shale bed indicated to be crossing the alignment near the location of borehole GCT 10-2. This was because structural analyses they performed indicated that the shale bed will be overstressed beyond failure throughout a region encompassing two diameters outside the tunnel perimeter due to the low strength of the shale relative to the high depth of cover at that location. Assuming inputs into this analysis are valid, it would not be prudent to try to mine through this reach with a TBM when overstress conditions of this magnitude are predicted beforehand. Based on the information available at this time, it would be prudent to schedule for a bypass around the shale bed.
- An approximate 200 m-wide limestone unit has been identified crossing the alignment in the vicinity of chainage 3+000 and another limestone unit with a contact aligned sub-parallel to the tunnel between chainages 6+000 and 9+000. There are serious concerns expressed by GCMC's third-party consultant that the limestone unit aligned sub-parallel to the tunnel could actually be intercepted over long distances if their interpretation of how the unit lies relative to the alignment is only slightly in error. The presence of limestone in the alignment poses a couple of potentially serious problems for TBM tunnel excavation, including:
  - The limestone will be severely overstressed at the depths present over much of the tunnel alignment and could easily trap a TBM during a breakdown or other downtime event
  - Small openings in the limestone can contribute significant water inflows. Grouting may be required. GCMC's third-party consultant allowed 15 days in their schedule for this kind of grouting. Lemley assumed an additional month, or 1½ months total, would be required
- Review of the geological logs for drilling completed in the area of the proposed tunnel indicates fairly thick beds of limestone and dolomite and wide shear zones were intercepted. If any of these weaker units lie in the same, southward-dipping attitude drawn for the 7.8 km of volcanics north of this borehole, bringing them under even deeper cover south of drill hole GCT 10-2, then chances are a large diameter TBM would have a great deal of difficulty negotiating much of that reach

and would suffer even greater schedule impacts than those considered probable in Lemley's analysis

- For the TBM, instantaneous penetration rate (IPR) estimates were developed by a leading TBM manufacturer based on a couple of different scenarios related primarily to the frequency of in-situ fracturing already present in the rock mass. This company calculated a low average IPR of 0.52 m/hr and a high average of 1.67 m/hr for the Galore Creek rock strengths and other conditions. In the very strong volcanic rocks indicated to be present at Galore Creek, the frequency of existing, in-situ fracturing of the rock mass, and the orientation of that fracturing relative to the alignment of the tunnel, can affect the IPR by a factor of three to four
- A literature review was performed on published case histories from similar-diameter tunnels mined with open gripper, main beam TBMs and constructed under high cover, in high strength rock and/or in squeezing ground conditions. There were a considerable number of tunnels described that suffered major delays due to squeezing ground and/or high water inflows.

Lemley's schedule estimate considered that an average IPR of 1.32 m/hr and an average TBM utilization of 35% is appropriate. Lemley assumed that tunnel boring is scheduled on a 24 hr/day basis, six days per week, with the seventh day used for heavy maintenance. It was also assumed that two weeks per year would be lost to holidays and weather. Using these assumptions, Lemley's estimated duration for TBM excavation of 7,686 m of tunnel beyond the starter tunnel is 28 months. A nine-month period was then added to provide for expected major lost-time events, including mining a bypass tunnel around, and conventionally mining through, the overstressed shale bed; remining of squeezing ground; additional grouting; and potential slower penetration rates through some of the stronger reaches of volcanic rocks. Combined, these result in the 37 month-long TBM tunnel excavation duration estimated by Lemley.

A productivity forecast of 6.6 m/day for a high-speed drill and blast operation advancing a 8.9 m-wide by 7.3 m-high, arched roof heading southward from the North Portal was assumed by GCMC's third-party consultant in the GCMC 2011 pre-feasibility study design. Lemley is of the opinion that this average rate is achievable so long as adequate supplies, spare parts and spare equipment are stockpiled and staged at the north portal during good weather to carry the operation through extended periods of poor, or otherwise unflyable weather, and that the supporting camp location is close enough to the tunnel for crew transport to be unaffected by the weather. Lemley also considers that specific requirements for relatively fast drill/blast production rates consistent with safe working practices will be required during the bidding process and excavation.

Lemley applied the 6.6 m/day average drill/blast advance rate over a distance of 5,750 m driven from the North Portal on the same work schedule as the TBM operation. A total of 2½ months was included in the schedule as a provision for expected major downtime events and for enlarging the end of the drill/blast tunnel and making other preparations so that the TBM can be moved into a cavern on the side, out of the way of traffic, immediately following hole through.

18.9.3 Review of Budget Estimates

Lemley advised that the changes adding time to the tunnel construction schedule will result in an increase in the budget that should be carried for the tunnel. Lemley reviewed the budget estimates developed by GCMC's third-party consultant, and made adjustments to account for the anticipated longer revised excavation duration, costs for expected major downtime events, contingency allocations, and owner-furnished insurance. Lemley's opinion of the most probable total cost for the Galore Creek Mine Access Tunnel is shown in Table 18-2.

18.9.4 Conclusions

The following conclusions are appropriate from Lemley's review:

- While the risks identified point to a very challenging tunnelling project, there is nothing inherent in these risks that has not been dealt with successfully on other projects, using both drill/blast and TBM methodologies, or which would cause Lemley to render an opinion that the Galore Creek tunnel is not constructible using the approach described in the GCMC 2011 pre-feasibility study
- GCMC's third-party consultant did a fair job of preparing a realistic design and identified key risks to the Project
- GCMC's third-party consultant did a good job of analyzing the tunnel construction costs and provided a sound basis from which realistic cost scenarios can be derived.

Table 18-2: Tunnel Cost Estimate

Section Description	GCMC December 2010 Estimate	Lemley Estimate
TBM Tunnel Excavation	$114,365,726	$139,050,838
D&B Tunnel Excavation	$63,829,359	$81,695,619
Equipment Capital	$49,891,133	$46,676,833
Field Indirect Costs	$78,862,019	$98,370,398
Sub-Total Direct + Field Indirect Costs	*$306,948,237*	*$365,793,687*
Mark-up on Tunnel Works (15% Mark-up)	$46,042,236	$54,869,053
Subtotal	**$352,990,473**	**$420,662,740**
Tunnel Construction Contingency	$47,117,011	$21,033,137
Total Value of Tunnel Contract (excluding taxes, escalation and GCMC site preparation and support costs)	**$400,107,484**	**$441,695,877**
Portal Surface Works - Site Prep (GCMC)	$6,047,275	$8,634,775
GCMC's Support Costs (camps, helicopter, etc.)	$116,836,801	$82,779,625
GCMC's Site Preparation and Support Costs Subtotal	**$122,884,076**	**$91,414,400**
Contingency on GCMC costs =	N/A	*by AMEC*
Tunnel Works Total (excluding taxes, GCMC contingency and escalation)	**$522,991,559**	**$533,110,277**
Escalation to start of construction	$9,619,118	*by AMEC*
Escalation during construction	$16,601,802	*by AMEC*
GRAND TOTAL	**$549,212,480**	**$533,110,277 + escalation & GCMC contingency**

It is Lemley's opinion that modifications to several design features and construction strategy assumptions made by GCMC's third-party consultant in their approach to the tunnel project warrant serious consideration and would, if adopted, enhance the constructability of the tunnel. These include:

- Risks associated with catastrophic flooding of the TBM as a result of a higher than anticipated groundwater inrush event, or related to evacuation of the tunnel due to unsafe levels of hydrogen sulphide, could best be mitigated by driving the TBM reach at a slight uphill grade. This will also enhance the productivity of the operation by making it easier to get rid of nuisance water in the ground support installation working area near the cutterhead support
- Constructing the roadbed in the TBM reach should await completion of the TBM drive and the movement of critical mining equipment through the tunnel so that the work is completed off the schedule critical path
- Excavation of half of the refuge bays during tunnel excavation should only be performed to the extent they are needed for installation of booster drives for the continuous conveyor and/or for similar purposes that support the TBM drive. Some

work could also be completed during major downtime events. The balance should wait until the roadbed construction is underway; using the waste rock removed from the enlargement excavations as subgrade fill in that operation

- Conventional excavation equipment should be available for rapid deployment on the TBM side should it become necessary to mine around the machine and through sections of the most adverse ground
- The drill and blast operation at the opposite end of the tunnel should be prepared to extend the length of their drive if necessary due to problems encountered on the TBM drive
- Consideration should be given to beginning the drill/blast operation from the north portal as early as possible in order to gain critical time on the schedule
- The Lemley schedule assumes the North and South Portal site preparation work and the access roads to each portal will be nearly complete when the tunnel contractor mobilizes. The schedule for the overall mine-development program should include permitting and construction of these facilities to support mobilization of the tunnel contractor with appropriate considerations for weather restrictions
- Geotechnical investigations underway now and in the future should collect as much information as is possible related to joint and fracture spacing and orientation. Additional tests for drillability (DRI) and cutter life (CLI) should also be made to verify the earlier results
- There are greater cost risks and schedule risks associated with the use of a TBM on this Project than there would be with two opposing drill and blast operations. Future studies should evaluate replacing the TBM drive with a second high-speed drill and blast heading as a way to reduce risks and better allow for a predictable and successful completion for the tunnel.

18.10 Camps

18.10.1 Construction Camps

There are a number of existing camps at the site (refer to Figure 5-1), as follows:

- Ch'yione (Km 37) – 100 persons
- Hooh (Km 73) – 125 persons
- Espaw (Km 89) – 125 persons
- Dechewe (Km 117) – 70 persons
- Ut Lun (Galore Creek Valley mine site) – 200 persons.

The Ch'yione camp is currently operational and is planned to support the early activity at the site, for road and bridge construction.

The Espaw camp is planned to be reopened to support the south portal construction work in the early phases of portal development, while the Dechewe camp from Km 117 will be relocated to the south portal. Once in place, the Dechewe camp will support ongoing work at the south portal and the Espaw camp will be relocated into the Galore Creek Valley to support the Ut Lun camp. The Espaw camp will be located such that it will not be affected by pre-stripping and mining operations.

The Hooh camp will be reopened to support the ongoing site access road construction and site preparation at West More. As the level of construction work increases, the Hooh camp will be expanded by 400 beds. Upon completion of the West More Valley site preparation, a new 1,000 bed camp will be constructed on the West More Valley site. The 400 bed expansion at Hooh camp will then be relocated to the West More Valley, with the remaining 125 bed Hooh camp supporting tailings construction.

Further into the construction schedule, another new 150 bed camp will be constructed at Km 8 to support the filter and dewatering plant construction.

A 200 bed camp will be constructed at the port of Stewart to support the construction of the concentrate storage and shiploader. However, it may be possible to accommodate construction workers in the town and reduce the camp size. Further investigation will be required during more detailed studies to provide support for this.

18.10.2 Permanent Camps

There will be three permanent camps for the Project. A 200 bed camp in the West More Valley will support operations and maintenance personnel associated with the main concentrator facilities and administration building, and a 300 bed camp in the valley will support mine operations. These two camps will be connected by a bus that will travel through the tunnel. A smaller 50 bed camp will be located at Km 8 to support the filter and dewatering plant.

During normal operational turnaround times, both mill and mine personnel will travel by road from Bob Quinn to their respective permanent camps.

Temporary construction offices will be established at the West More Valley plant site and in the Galore Creek Valley to accommodate initial construction management staff. These offices will be upgraded or moved to suit peak forces after site preparation is completed. A construction management office will also be located at the south portal for use by the tunnel construction management staff. A site office will also be located at the port site.

18.11 Power and Electrical

Power to the plant site will be supplied by an 87 km-long, 287 kV single circuit overhead pole line from the BC Hydro Bob Quinn substation. The pole line will terminate at a 287 kV switchyard, where it will be fed into a 287 kV bus system. Each main transformer will be fed from a HV circuit breaker and HV isolating switch. Each main transformer will be rated 287-34.5 kV, 100/133/167 MVA. The secondaries of the two main substation transformers will feed a 34.5 kV sectionalized switchgear line-up in the main substation that will be connected by an open bus tie circuit breaker. The bus tie will only close if a section of the 34.5 kV switchgear has lost power due to a failure of the transformer that feeds the switchgear section. The bus tie will then provide power to this section. The two mains and the tie can all be closed for a maximum of three seconds to allow for a closed transition. The main substation transformers will be equipped with fans to provide the additional capacity required in the event of a single main transformer failure.

The system will have a steady state voltage regulation of ±5%. It will be able to start the largest connected motor within the permitted voltage drop from the feeder bus to the motor terminals, while simultaneously supplying the remainder of the connected loads.

Knight Piésold completed a prefeasibility level evaluation for a proposed power supply transmission line. The Project will interconnect at the BC Hydro Bob Quinn substation for the supply of an average 152 MW of power for the planned mine.

From BC Hydro's Bob Quinn substation, the line will head north along Highway 37 for approximately 6 km where it will then head due east to meet up with the Galore Creek mine access road. From this location, the alignment generally follows the More Creek Valley along the Galore road to the West More Valley plant site. At 33.8 km along this alignment, two options were assessed. The first option follows the More Creek Valley (More Creek alignment), while the alternative follows the Boulder Creek Valley (Boulder Creek alignment), rejoining with the More Creek alignment at 64.5 km along its proposed routing. The selected preferred alternative, the More Creek alignment, was selected because the Galore Creek mine access road will follow approximately the same route.

The proposed overhead power line for the interconnection of the Project will be designed and constructed in accordance with sound engineering practices for satisfactory operation and to avoid adverse impacts on the safety and security of the transmission system. The line will be approximately 69 km in length with a rated voltage of 287 kV. The major electrical load requirements are as follows:

- The interconnection nominal bus voltage at the BC Hydro Bob Quinn substation, is 287 kV
- The total anticipated connected load on the main substation is 210 MVA/205 MW
- The total peak operating load at the mine and process site is estimated at 162 MVA/160 MW
- The total average operating load is estimated at 154 MVA/152 MW.

A nominal transmission line voltage study, conductor optimisation study, and power flow analysis were undertaken to determine key parameters of the single line diagram (i.e., transmission line voltage, transmission line conductor, and voltage compensation). The nominal voltage was selected as 287 kV for the interconnection of the Project to the BC Hydro transmission system. Both 230 and 287 kV nominal voltages are technically viable. However, 287 kV provides cost savings in terms of capital cost, plus slightly lower transmission line losses.

Two step-down transformers will step down the voltage from 287 kV to 34.5 kV for use by the plant and pit equipment in the Galore Creek Valley. The total average load for the Galore Creek Valley plant and pit is about 30 MW; a double circuit overhead line was recommended for reliability.

18.11.1 Site Electrical Reticulation

The backbone for the power to the Galore Creek Valley and West More Valley sites will be a 34.5 kV, 60 Hz, three-phase distribution system.

The mine area requires approximately 30 MW for pit operations, crushing, conveyor drives, the camp and the truckshop. Two 34.5 kV cables, each capable of carrying the full load, will transit the tunnel on a rack located above the conveyor.

There are a number of substations where 34.5 kV will be stepped down to 4.16 kV for motor loads greater than 187 kW and 600 V for items such as smaller motor loads, lighting, and heat. Large substations will be located adjacent to the following locations with power demands:

- Near the West More Valley stockpile for the conveyor drives
- At the South Portal for four of the tunnel conveyor drives

- At the North Portal for two of the tunnel conveyor drives and three drives at the head of the crushed ore discharge conveyor from the surge pile at the primary crusher
- In the tailings area to supply the reclaim barge pumps and associated equipment.

These large substations will be built in modularized E-houses that require a minimum of on-site assembly and wiring. The remaining substations will be located in electrical rooms in the process plant area.

18.11.2 Review of the Proposed Electrical Systems

AMEC reviewed the electrical design, and noted the following areas that will need to be addressed:

- Information provided in the text of the GCMC 2011 pre-feasibility study in relation to electrical loads does not agree with the load list data. Electrical loads should be reviewed to verify that they meet peak load operability during more detailed studies
- The list of pre-fabricated electrical rooms does not include the process area electrical rooms. This raises the question of whether or not the process area electrical rooms are to be prefabricated or stick-built on site
- Quantification of costs for the electrical line should be substantiated, given guidance from BC Hydro that the first project using power from the proposed new power line would pay a disproportionately higher share of the cost until more projects using the line and power supply are brought on line. BC Hydro are currently reviewing their rate schedule for this power line
- The process control communications system on site is based on a fibre optic “collapsed ring” backbone which means there is a ring system but instead of routing the fibre optic cable ring in separate paths, the cable fibres are in a common cable jacket. In the event that the common cable is damaged, the whole system is compromised. This assumption should be reviewed during more detailed studies
- The control system motor control is based on using DeviceNet™ communications. This should be reviewed and justified, because the choice eliminates equipment from at least two major manufacturers.

18.12 Fuel

During construction, tankers will deliver diesel to site via the mine access road. During the initial phase of construction, prior to completion of the access road, tanker traffic will only travel as far as the Ch’yione camp at Km 36 where a temporary diesel storage facility will be constructed. The Ch'yione camp will be the hub for all helicopter support until the site access road to the West More Valley is completed.

Completion of the site access road to West More and the Hooh camp sites will allow the fuel tankers unload into storage tanks in the West More area to support the camps and site-preparation activities. Completion of the connecting road to the south portal will allow the tankers to deliver fuel to temporary storage tanks supporting the tunnelling and Galore Creek Valley helicopter-support operations.

Diesel fuel flown into the Galore Creek Valley by helicopter will support the camps, mobile equipment used in the construction of the dams and diversions, and the drill-and-blast operation at the north portal. The existing fuel storage and dispensing facility in the valley will need to be overhauled prior to the start of construction work. The equipment will be used to set up a temporary centrally located tank farm, from where distribution will be controlled. A second tanker will be flown in to supplement the existing fuel tanker in the valley. The two tankers will be used to distribute fuel to the mobile equipment and generators.

Upon completion of the construction phase of the Project, a diesel storage and pumping facility will be located at Km 8. Diesel will be delivered to the facility by trucks, and then pumped to the West More fuel storage tanks via a buried pipeline along the site access road. From the West More storage tanks, the fuel will be pumped through the tunnel to another fuel storage facility located near the crusher and truckshop in the Galore Creek Valley.

18.13 Fresh Water Supply

In the Galore Creek Valley, freshwater for facilities will be provided by two wells, one of which will be operational, and the second will be on standby. Each well will be approximately 100 m deep. A freshwater tank will be located near the permanent mine camp. The freshwater tank will supply water to the buildings via 3″ HDPE pipelines with a combined length of approximately 800 m.

In the West More area, freshwater required for the process plant, camp and other facilities will be provided by three wells (two operating and one standby). Each well will be 450 m deep, and be located in a triangular shape outside of the plant site, two on the north and one on the south. The freshwater pipeline will be buried under the 5 m wide access roads to the freshwater wells. The freshwater tank will be inside the process plant and will supply water to the buildings by gravity via a 3″ HDPE pipeline.

At the Km 8 filter plant, freshwater will be provided by two wells (one standby and one operating). Each well will be approximately 100 m deep. A freshwater tank will be located near the plant building and supplies water to the buildings by gravity via a 3″ HDPE. The freshwater pipeline will be buried under the 5 m wide access roads to the freshwater wells.

Further hydrogeological investigation is required to support the availability and quality of the water in the different areas.

At each site, some of the freshwater will be treated and stored in potable water tanks located adjacent the freshwater tanks.

The freshwater tanks will also provide firewater to automatic sprinklers, standpipe systems and yard hydrants at each site.

18.14 Sewage Systems

The sewage from the facilities in the Galore Creek Valley, West More, and Bob Quinn will be collected and sent to packaged sanitary sewage treatment plants to be located in each of the three areas. The sewage piping system will be designed to accommodate estimated peak flows.

18.15 Process Control and Instrumentation

The Galore Creek mining operation will be split into four control areas:

- Crushing and conveying facility in the Galore Creek Valley
- Concentrator and tailings facility in the West More Valley
- Filtration and water treatment plant at Km 8
- Port of Stewart.

The distance between each area is significant:

- Approximately 20 km between the crusher control area in the Galore Creek Valley and the concentrator in the West More Valley
- Approximately 80 km between the concentrator in the West More Valley and the filtration plant at Km 8
- Approximately 320 km between the concentrator in the West More Valley and the shiploader at Stewart.

The plant control system proposed is a distributed control system (DCS) communicating over fibre optic (collapsed ring) backbone and a satellite network. The fibre optic backbone will connect the crusher area, tunnel and concentrator areas. The satellite network will connect the concentrator area to the filtration plant at Km 8 and the shiploading facility at Stewart.

18.16 Port

The proposed port site is the former Arrow Dock facility, a causeway made of reclaimed land to the southeast of Stewart at the head of the Portland Canal bounded on the east by the Bear River and the west by estuarine marsh and mudflats. The Arrow Dock causeway was previously used as a port facility for the Cassiar mine for asbestos packaging, storage, and transfer to barge and for fuel transfer from barge to tank to truck. The Cassiar operations ended in 1992, although the four fuel tanks associated underground pipeline, storage building and the derelict barge ramp remain on the site.

18.16.1 Design Considerations

Habitat compensation and remediation will be required on site. Habitat compensation is considered a key environmental consideration for the development of the port. Remediation will include areas impacted by releases of petroleum hydrocarbons (areas adjacent to the fuel tanks and underground fuel lines), and localized areas potentially impacted by asbestos from former asbestos packaging operations (areas adjacent the existing storage building).

Four above-ground fuel tanks and associated underground piping, the storage shed, the fuelling platform, the barge ramp and associated equipment, the existing overhead power line, culvert, and, potentially, an underground septic tank will require demolition.

The preliminary geotechnical investigation has predicted significant settlement over the entire site due to ongoing geological processes and significant differential settlement at the large facilities, particularly the storage shed. Ground densification at locations where new structures will be placed has been recommended to limit settlement and provide site stability during seismic events.

The possibility of the causeway flooding was recognized as a very serious concern over the scheduled life of the port facility. The GCMC 2011 pre-feasibility study is based on raising the level of the site to the minimum elevation to protect against the worst case flood scenario (a combination of aggradation of the Bear River Channel, fluvial flow in the Bear River, storm surge, high tide and the expected one metre overall settlement of the site over the design life of the port).

18.16.2 Facility Design

The planned facility flowsheet is included as Figure 18-5.

Geared side dump haul trucks and trailers (B-trains) with a capacity of 50 tonnes will be used to transport material to the port site. The trucks will discharge into the truck dump hopper inside an enclosed building. The building will contain a truck wash prior to the truck exit and a water treatment facility for the truck wash water. The truck dump hopper will discharge onto a truck dump transfer conveyor, which will discharge onto the concentrate storage tripper conveyor.

Material transfer to the concentrate storage tripper conveyor will occur in transfer tower #1. The tripper conveyor will discharge the concentrate in a continuous stockpile inside the storage building. Two front-end loaders (FELs) will reclaim the concentrate in two above-ground hoppers located in the storage building. The peak reclaim rate for the reclaim and shiploading system will be 1,500 t/h. Each reclaim hopper will be equipped with a screen to prevent oversize material from entering the conveyor stream.

The reclaim conveyor will transfer onto the trestle conveyor, which will deliver the material to the shiploader. The shiploader will be a totally enclosed fixed radial slewing machine with a luffing shuttle boom, sized to accommodate a 55,000 dwt vessel with single hold coverage. The vessels will need to be warped (moved along the berth face) during shiploading to access to all of the ship's holds.

Mobile equipment purchased for the site will include two FELs for material reclaim, one D6 Caterpillar dozer or equivalent for hold trimming, one forklift, one bobcat with a snow blower or snowplow attachment, and one pickup for use as a maintenance vehicle.

The port construction is expected to take two years, with the marine construction starting six months prior to the land facilities. This includes the soil treatment (densification) and site fill for flood protection; estimated to take four months but depends on the number of densification rigs that are available. The long-lead items to be considered in the construction planning include the shiploader.

The port facilities will be designed for maximum pre-assembly where economically feasible. The port also has limited laydown area given the space restrictions on the causeway.

Figure 18-5: Planned Loading/Port Facility Design

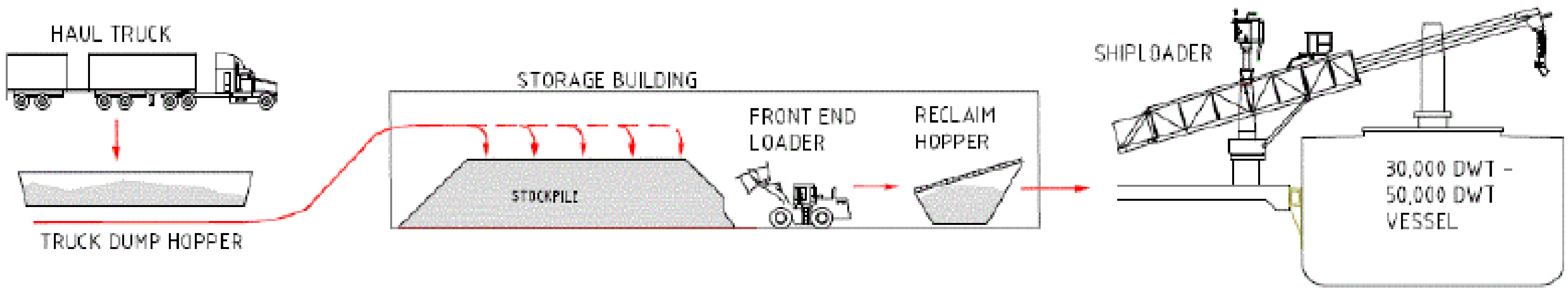


Figure courtesy GCMC, NovaGold and Teck

18.16.3 Access Considerations

The existing gravel access road that extends south along the causeway from Stewart will be upgraded to an 8 m-wide, two-lane paved road extending to the main vehicle entrance gate. South of the site gate, the road will continue outside the security fence as a single lane gravel road to the public boat launch parking lot. The existing boat launch ramp will require replacement due to the flood protection fill provided across the site, but will be replaced in kind (i.e., only accessible during high tide).

Security fencing will be provided at the northern and eastern perimeter of the site. The main parking lots will be located outside of the fencing, immediately north of the administration facilities. There will be insufficient parking in the main lot for shift change during shiploading; this can be accommodated by staggering shift starts or creating a second lot at the north end of the causeway and shuttling people to the site.

There will be additional Marine Security (MARSEC) requirements as the terminal will be used to export copper concentrate to overseas markets, and as such, will be served by foreign-flagged vessels or Canadian-flagged vessels bound for foreign ports. If overall site security does not meet MARSEC requirements, jetty access will be need to be restricted with security fencing, a gate to the wharf access trestle roadway and a barrier at the conveyor gallery walkway.

18.16.4 Storm Water Considerations

The drainage design will permit storm water to be collected and stored in a sedimentation pond for a minimum retention time of 24 hours prior to release to the receiving environment. Melting areas for ploughed up piles of snow will be designed to permit meltwater to be captured and sent to the sedimentation pond. The storm water from the sedimentation pond will be directed to the east side of the site where it will be discharged through a perforated pipe along the riverbank. A storm water discharge typically does not require permitting.

Wash water from the truck wash facility and other potential concentrate containing areas will be directed to a small water treatment plant. Accumulated solids from the water treatment process will be shipped back to the filtration plant site.

18.16.5 Environmental Considerations

Habitat compensation is considered a key environmental consideration for the development of the port. Other environmental aspects applicable to the port site include: storm water management, disposal of dredging arisings, culvert replacement, a re-fuelling station, site raising to alleviate flood potential, potential river aggregate dredging in the Bear River to alleviate flood potential, potential contamination from historical site activities, and densification works in the estuary and in the Bear River.

18.16.6 Utilities and Facilities

The District of Stewart has confirmed available capacity in the city water and sanitary systems.

The fire water system will be a dry system, activated with seawater once a fire is detected.

Where required, compressed air for the site will be provided by portable compressors. The power supply to the terminal will come from an existing overhead 25 kV BC Hydro distribution line, which originates from the main BCH substation in Stewart. BCH will ascertain whether upgrades to the line are needed, and will require GCMC to bear the associated costs. The GCMC 2011 pre-feasibility study assumed that replacement of the powerline down the causeway was required.

The estimated maximum overall electrical connected load for the GCMC terminal is 2 MW.

A diesel generator and associated fuel tank will be provided in case of power outage.

The peak number of personnel will be 25 during one shift and 45 across shifts.

The required facilities will include offices, an administration area, washrooms, lunchroom, and change/shower rooms. A maintenance facility for small repairs and minor maintenance will be required. Major maintenance will be contracted out. The plant control room will also be located in the main facilities. The facilities will be located in a separate building exterior to the truck dump and storage shed.

18.16.7 Loading Design

The recommended range of vessel sizes that can be accommodated by the marine facilities is from 30,000 to 55,000 dwt with respective laden draft range of 10.5 to 13.0 m. The berth arrangement is designed based on the use of the fixed radial slewing shiploader and the recommended vessel sizes. The Pilot Authority restrictions limit berthing and de-berthing at Stewart to daylight hours only.

Provision has been made in the layout of the marine facilities for a temporary barge facility, if a barge facility is required during the initial construction phase of the Project.

Capital and maintenance dredging works are proposed to counteract the westward advancement of the Bear River delta over time towards the eastern end of the berth. An initial capital dredge of 50,000 m^3 is proposed, with a subsequent maintenance dredge of 50,000 m^3 to be removed after 10 years of operation. The GCMC 2011 pre-feasibility study assumed the dredgeate would be non-contaminated and disposed of at sea at an approved disposal site.

18.17 Consideration of Logistics

Previous studies by McElhanney in 2008 and Transera International Logistics in 2006 (updated 2010), identified three viable alternative transportation routes for moving materials to the Project site. These routes were reviewed taking into account the information on the size of equipment currently proposed for the Project.

The proposed logistics plan includes the following considerations:

- Nearest air access is at the Bob Quinn airstrip, which is capable of landing Beach 1900, Dash 8s and Hercules aircraft
- Nearest road access is at Bob Quinn Lake on Highway 37, approximately 90 km east of the site
- Site road access from Highway 37 to the south portal is partially constructed. The road, once completed, will be suitable for highway tractors and trailers and be able to accommodate the heaviest loads to meet the early construction of the tunnel. This section forms part of the early construction requirements identified by GCMC
- The existing road section through the tailings impoundment area will be kept until tailings construction prevents its use
- Helicopter services will support the early site activities until construction of the access road and tunnel is completed
- Nearest rail link is at Kitwanga
- Nearest port access is at Stewart, which can accommodate a shiploading facility and has an airstrip with similar capabilities as Bob Quinn.

Three transport routes were assessed, and the from route Stewart–Highway 37A–37 route appears to have the least constraints and is proposed for all major shipments (e.g., tunnel boring machine, transformers, mill components, mine haul trucks and primary crusher), with some smaller loads travelling by rail and/or road. The current envelope size constraints on road transportation that apply to all routes to the site are the Bell Irving #2 and Devil Creek bridges on Highway 37.

The shipment of major equipment required in the Galore Creek Valley will be scheduled to match completion of the access tunnel. Prior to tunnel completion, essential equipment, supplies and personnel will be transported by helicopter to the Galore Creek Valley from heliports at Ch'yione or an area adjacent to the tunnel south portal.

Large components will be shipped via ocean freight to Vancouver or Prince Rupert where they will be transferred to barges for shipment to Stewart. At Stewart, the equipment will be transferred to trucks for transportation to site. Where practical, smaller components will be consolidated at ports of loading for transportation in containers. Staging areas will need to be established during feasibility-level studies to optimize transportation logistics.

A load-on/load-off (lo-lo) barge ramp has been included in the overall preliminary Project construction execution plan. However, the permanent ramp will not be completed in time to meet the early delivery of the TBM. The use of a temporary barge ramp and shallow bottom barges has been assumed. Currently there are no permanent cranes at the Kitwanga rail sidings for transferring cargo from rail cars to trucks and it is planned that mobile cranes from Prince George or other centres could be used for this duty.

18.18 Comment on Section 18

In the opinion of the AMEC QPs, the following conclusions are appropriate:

- The Project will require construction of significant infrastructure to support the planned producing facilities
- Accompanying the six mining phases will be two NPAG waste dumps and three PAG waste dumps. The NPAG waste dumps will initially be situated out of the valley floor until PAG rock is mined then PAG waste will be deposited in the valley bottom which will be flooded at mine closure. The West Fork, and Southwest satellite pits will be partially backfilled with PAG rock and flooded at closure
- Process tailings will be stored in the West More tailings facility, which will consist of three dams, a Main Dam and two saddle dams. The dams and impoundment will accommodate up to 678 Mt of tailings, although storage for only 510 Mt is required for the current mine plan
- Tailings design is based on appropriate geotechnical testwork, and acid-base accounting tests. The design incorporates considerations of geohazards

- Water management of the Galore Creek watershed will be a major design challenge. A number of water control structures are planned, including diversion channels, and closure and sedimentation dams
- Based on preliminary modelling, water quality will be suitable for direct discharge with no requirement for water treatment
- Geohazards are present in the Galore Creek Valley and require careful consideration in waste and water management
- The Galore Creek Project is currently not accessible by road. The closest provincial road to the mine site is Highway 37. An access road is planned from this highway to the proposed mine site. A section of the access road from Highway 37 (Km 0) to approximately Km 40 was constructed during a previous Project phase and is currently in service
- GCMC will construct a new 287 kV transmission line to supply the power demand at the proposed Galore Creek mine site. The transmission/distribution lines will have sufficient capacity to service the power demand for mining and process equipment throughout the life of the mine. Electricial loads should be reviewed to verify that they meet peak load operability during more detailed studies
- There will be three permanent camps at the Galore Creek Project. West More will support operations and maintenance personnel associated with the main concentrator facilities and administration building; the valley camp will support mine operations, and the Km 8 camp will support the filter and dewatering plant
- A diesel storage and pumping facility will be located at Km 8. Diesel will be delivered to the facility by trucks, and then pumped to fuel storage tanks at West More, then delivered by pipeline to the mine site
- Freshwater will be provided from wells
- The proposed port site is the former Arrow Dock facility, a causeway made of reclaimed land to the southeast of Stewart. The port site will include a concentrate storage and shiploading system. Habitat compensation is considered a key environmental consideration for the development of the port.

In the opinion of the Lemley QP, the following conclusions are appropriate in relation to the proposed tunnel:

- The 13.6 km access tunnel represents a major tunnel project in terms of international tunnelling practice. The proposed tunnel is aligned under high rock cover of more than 600 m over a significant portion (75%) and with a maximum rock cover of 1,250 m

- The tunnel will take about 37 months to excavate once TBM boring commences, and an overall tunnel construction time of 49 months is considered appropriate. This timeframe is longer than that envisaged in the GCMC 2011 pre-feasibility study. AMEC has modified the schedule and adjusted the cost to reflect this longer timeframe
- Lemley has made specific recommendations for changes to the tunnel design, including up-grade TBM excavation, and consideration of two opposing drill-and-blast operations rather than use of a TBM; these recommendations should be reviewed during more detailed studies.

AMEC notes that the most complex and challenging construction phase will be related to the work required in the Galore Creek Valley. Only air support is available until tunnel break-through, creating a very difficult and expensive operation. The previous construction activities carried out in the Galore Creek Valley in 2007 utilized construction equipment flown in by helicopter to support the previous tunnel drill and blast operations. Helicopter transport will be needed to fly manpower, additional equipment, fuel and construction materials into the Galore Creek Valley. This will enable activities other than the tunnel drill-and-blast operation to proceed immediately upon receipt of the construction permits.

The work in the valley will comprise the following:

- Water diversion channels
- The east branch conveyor right-of-way (ROW) with the elevated causeway (including the filling of the conveyor corridor in the flood plain)
- Galore Creek Valley water diversion sedimentation dam and temporary bypass tunnel, Decant tower
- Spillway
- Access roads
- Earth works for the primary crusher area.

Earthworks quantities will need to be reviewed and optimized during feasibility-level studies, as there are large quantities of cut-and-fill involved.

To minimize capital costs, the early work in the valley, before tunnel breakthrough, should be restricted to that which can be done to conform to the EA permits and to maintain the construction schedule at a reasonably low cost.

The supply of diesel to support early construction is critical to both the Galore Creek Valley and West More areas. Prior to any construction work commencing, a secure supply of diesel will need to be established via a long-term supply contract.

The tunnelling operation is a high consumer of power and, initially, diesel. The TBM will require 10 to 15 MVA of power, which will be provided from the 287 kV power source. The development of the 287 kV line from Bob Quinn and the 287 kV substation at West More are both activities that require early completion to help reduce the overall diesel consumption. Diesel generators will be required to provide power during on-site assembly of the TBM and during the initial six months of tunnel boring.

It will be important to get power to the site as early as possible to reduce the fuel requirement supporting the diesel generators at the south portal. Early engineering of the 287 kV substation at West More will enable early procurement of the long delivery substation electrical equipment (i.e., transformers and switchgear in the substations), which will allow BC Hydro NTL power (287 kV) to be connected to the south portal at the earliest opportunity. However, construction of the West More substation will require EA approval.

19.0 MARKET STUDIES AND CONTRACTS

The market studies and contracts completed by GCMC as part of the GCMC 2011 pre-feasibility study were reviewed by AMEC. AMEC did not perform an independent market assessment.

19.1 Market Review

GCMC requested a market opinion on the copper concentrate market balance and demand outlook from Teck's internal marketing experts. Teck is of the opinion that the copper concentrate market will remain tight to 2020 due to an increase in smelting capacity ahead of mine production growth. With copper demand projected to grow at a rate of 3.1% per annum out to 2020, Teck predicts that demand will exceed refined production by close to 6.5 Mt in 2020, and therefore that additional mine production will be required to satisfy projected demand.

Teck notes that if all of the Highly Probable and the Probable mine projects listed by Brook Hunt (2011) were to come into production on time the refined copper market could move into surplus by 2013. Teck considers that this would suggest the market would need to deliver the already committed 3.5 Mt of contained copper increases in the base case assumption made by Teck, plus an additional 5.0 Mt of copper production by 2020. This would represent a 67% increase over current global mine production levels.

19.2 Galore Creek Concentrate

The conceptual production level will be an average of over 600,000 dmt of copper concentrates produced annually over the Project life. The only element that is of concern as an impurity is fluorine, a low-level deduction for fluorine content has been assumed to apply. The preliminary concentrate specifications are shown in Table 19-1.

Table 19-1: GCMC Preliminary Specification Expectations for Concentrate

Element	Range
Cu	25–32 %
Au	5–40 ppm
Ag	100–200 ppm
F	200–500 ppm

19.2.1 Sales Strategy and Sales Plan

The sales plan is to establish long-term contracts for approximately 75% of its minimum long-term production quantity in order to provide stable and reliable sales. The preliminary geographic sales plan is shown in Table 19-2.

Table 19-2: GCMC Preliminary Geographic Sales Plan

	% of Production	Freight, US$/wmt
China/Korea/SEA	50%	75
Japan	30%	70
India	10%	90
Europe	10%	95

19.3 Contracts

GCMC plans that contracts may be as long as 10 years in duration, with most terms fixed in the contract. Contracts would be expected to address quantity terms, in particular. Treatment and refining charges would be likely to be negotiated periodically, typically on an annual basis. Considerations for contracts and marketing are summarized in Table 19-3.

The marketing strategy will focus on the major custom smelting companies in the world that are logistically practical for the delivery of concentrates.

An estimated time line for development of contract terms is included as Table 19-4.

From an organization perspective Teck Metals Ltd. will be the sole marketing agent for GCMC. The Teck marketing organization will provide administrative support from the Toronto marketing office and logistic and corporate support from the Vancouver head office. Personnel to provide the marketing and transportation support will come from the existing groups and any additional personnel required will be considered as part of the overall requirement for the Teck marketing organization.

Table 19-3: GCMC Concentrate Assumptions

	Asia	Europe
Treatment Charge	70 $/dmt	same
Cu Payable	<22% (min ded 1.1%) < 32% 96.50% (min ded 1%) >=32% 96.65% >=38% 96.75%	same
Cu Refining Charge	7.0 c/lb	same
Price Participation	None	same
Au Payable	0% <= 1 g/dmt 90% 1<Au<=3 92% 3<Au<=5 95% 5<Au<=8 96% 8<Au<=10 97% 10<Au<=15 97.5% 15<Au<=50 98% Au>50	ded 1 g/dmt, pay 100%
Au Refining Charge	6 $/toz	6 $/toz
Ag Payable	0% Ag<=30 g/dmt 90% Ag>30	ded 30 g/dmt, pay 100%
Ag Refining Charge	0.40 $/toz	0.40 $/toz
Deductions for F content	F, $1.00/100 ppm > 300 ppm	same
Payment	90% provisional 3 days after arrival	90% provisional 30 days after arrival

Table 19-4: Marketing Contract Development Milestones

Contract	Timing	Activity
Pre Marketing	Ongoing	Making potential customers aware of the Galore Creek Project and the provisional parameters of quality, quantity and timing for concentrate supply; gauge how much interest there is in the concentrate and what constraints may exist
Letter of Intent	Anytime	A non-binding indication of a quantity range based on some quality and timing assumptions for the Project
Memorandum of Agreement	Around Project commitment	Negotiation of major terms and conditions for a Sales Agreement
Sales Agreement	After MOA, until Start-Up	Final legal contract that includes all terms and conditions. May not be signed prior to mine start-up
Production	Start-Up	

19.4 Comment on Section 19

In the opinion of the AMEC QPs, the following conclusions can be drawn from the marketing strategy used to support the GCMC 2011 pre-feasibility study estimates:

- The concentrate will be of average copper grade, which will be acceptable to most smelters; if a concentrate on the upper end of the range indicated in Table 19-1 can be achieved, the concentrate will be more attractive to smelters
- It will be considered a gold-bearing concentrate and may be less attractive to those smelters which do not have efficient precious metal recovery
- Based on the currently-available, and limited, multi-element analyses of concentrates, the concentrate may contain one significant impurity, fluorine, and

that will be at levels such that deductions will apply, but the expected levels are below rejection concentrations. There may be occasions when the level of fluorine could attract a penalty structure from a smelter. The presence of fluorine in the concentrates may limit the quantity any one smelter may wish to purchase

- The Project as envisaged in the GCMC 2011 pre-feasibility study is of medium size and therefore will not have a significant market impact
- The planned sales strategy for GCMC is to be to establish long-term contracts for approximately 75% of the minimum long-term production quantity from the Galore Creek deposit
- Geographically the most economical markets from a transportation perspective are China, Japan and Korea. India and Europe are relatively longer shipping destinations and therefore more expensive
- The long-lead time for start-up could correspond with supply from several other projects becoming available, and therefore the concentrates could be marketed under different market conditions than the assumptions and forecasts that support the GCMC 2011 pre-feasibility study
- GCMC is currently in the pre-marketing phase documented in Table 19-4. The company has not sought expressions of interest or letters of intent from smelters. This will be required to support more detailed Project studies
- Contracts that will be negotiated are expected to be within industry norms.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

The environmental studies, permitting, and social or community impact assessments completed by GCMC and Rescan as part of the GCMC 2011 pre-feasibility study were reviewed by AMEC. AMEC did not perform an independent evaluation of the Project environmental status, permitting activities required to support development, or socio-economic factors.

20.1 Change of Project Scope

The Galore Creek Project received its Environmental Assessment (EA) approval in February 2007. The Project's first permits were obtained in May 2007, and in June 2007, GCMC received final Federal approval.

The new Project design and configuration is different from the design that was permitted under the original EA Certificate and that received Federal approval. Some of the most significant changes are:

- Better understanding of geochemistry, resulting in a different approach to waste rock and tailings management
- Simplified waste and water management strategy in the Galore Creek Valley plant site and tailings relocated outside of the Galore Creek Valley, in a new previously unaffected watershed (West More)
- Deletion of 30 km section of access road down the Sphaler Valley to Porcupine and the Scott Simpson Valley, significantly reducing potential environmental impacts and geohazards
- Deletion of the airstrip that was to be constructed in the Porcupine Valley
- Addition of new loadout facilities at the Port of Stewart.

While the new configuration is considered an improvement, with reduced overall environmental impacts, it is anticipated that a new EA process will be requested by the regulators. This will involve parallel and harmonized reviews by both the BC Environmental Assessment Office (BCEAO) and the Canadian Environmental Assessment Agency (CEAA). A comprehensive study report will be required through CEAA. It is anticipated that the entire EA review process will require two full years from submission of a Project description to issuance of a new EA Certificate (by the BC government) and a decision by the federal Minister of Environment.

One significant potential addition to the scope of the EA will be the inclusion of a port facility. This would also require assessing the transportation of the concentrate from the plant site to the port by road, as well as the alternative of using a pipeline.

The new EA will include assessing the cumulative effects of the Galore Creek Project in connection with other projects that have been developed or are proposed for development since the previous EA was completed. This will include, at a minimum, the Northwest Transmission Line, the Forrest Kerr hydroelectric project, the concentrate transport from Yukon Zinc's Wolverine mine, and the Red Chris project.

The existing Special Use Permit (SUP) for construction of the access road remains valid as long as there are no proposed changes to the SUP, thereby permitting GCMC to continue to build the access road. While there will eventually be changes to the SUP to accommodate the new mine design configuration, that relevant portion of the access road can be excluded from a new EA as long as any SUP mitigation plan measures (e.g., fish compensation and PAG rock management plans) are implemented. Changes to the current SUP will ultimately be required around the new TSF, plus a branch to the south portal of the tunnel to the Galore Creek Valley. An amendment to make these changes will be applied for once the EA process has been completed.

Existing permits associated with the existing construction camps, including water use and waste discharge, will continue to be maintained. All other Project permits will have to be applied for following completion of the EA process, although the time-critical permits, such as those needed for starting the tunnelling can be prepared concurrent with the EA such that there should be little lag time following EA certification before tunnelling could begin.

20.2 Baseline Environmental Studies

The original baseline studies incorporated into the 2006 EA Process provided a characterization of the existing environment of the Project area considering the original Project development plan. The original development plan had the majority of the mine site infrastructure in the Galore Creek Valley, including the open pit, plant site and a combined waste rock and tailings management facility. The More Creek Valley was used as an access corridor that included alignments of an access road, transmission line, and pipeline. The baseline studies have been ongoing since the EA Certificate was received.

Studies completed collected baseline information on:

- Air quality
- Climate
- Site noise
- Surface water
- Ground water
- Aquatic resources
- Sediment quality
- Fish and fish habitat
- Wildlife and wildlife habitat
- Wetlands
- Terrestrial ecosystems, vegetation and soil landscapes
- Archaeology
- Navigable waters.

The revised mine development plan in the GCMC 2011 pre-feasibility study includes an increased footprint in the More Creek Valley. The increase is due to the relocation of the tailings management facility and associated infrastructure to the Upper West More area, near the watershed boundary with Sphaler Creek. Additional changes to the original mine plan include a reduced footprint in the Galore Creek Valley due to relocation of the tailings facility and a reduction in the total length of the access corridor.

Although the More Creek Valley was included within the original baseline study area, study intensity was comparatively less than the Galore Creek Valley, because the Project footprint within the More Creek Valley was to be limited to an access corridor rather than a mine area-receiving environment.

A gap analysis was conducted to identify additional environmental data requirements in the Upper West More and downstream receiving environments to support the Project through another environmental effects assessment (RTEC, 2010). An additional objective of this gap analysis is to identify whether more baseline information was required within, or downstream of, the Galore Creek Valley considering the elapsed period since the original baseline study programs were largely completed in 2005. For ecosystem components that have relatively large natural variation, additional baseline studies are also recommended for West More Creek, as well as for key locations within the Galore Creek Valley and downstream receiving environments. The recommended programs are included in Table 20-1.

Table 20-1: Additional Recommended Environmental Data Collection Areas

Discipline	Recommended Studies
Archaeology	Assessment of areas of known archaeological potential in the Upper West More area
Aquatics	Water quality, soil quality, toxicology, and aquatic biology at several new sites within West More and North More creeks. Water quality, soil quality, toxicology, and aquatic biology in upper Sphaler Creek. Additional aquatic monitoring at selected locations in lower More Creek, Iskut River, Galore Creek, Scud River, and Stikine River.
Ecosystem and Soils	Additional terrain, soils, and ecosystem mapping and field verifications in the Upper West More area outside of the original Local Study Area mapped along the Galore Creek Access Road.
Fisheries	Additional physical measurements of fish in More Creek at the upper end of their range below West More Creek. Fish tissue metal sampling in lower Sphaler Creek. Detailed habitat assessments may be required, if alterations to flow are expected in lower More Creek.
Hydrogeology	Installation and monitoring of several groundwater wells in the Upper West More and upper Sphaler areas near the site of the proposed tailings facility.
Hydrology and Glaciers	Continued operation of sites in the Galore Creek and More Creek watersheds. Hydrometric stations on upper Sphaler Creek and North More Creek, and installation of a new station on North More Creek above West More Creek. Glacier dynamic studies of glaciers in the Upper West More and Galore East Fork areas.
Meteorology, Air Quality, and Noise	Continued operation of the five active meteorological stations in the Project area. Relocation of the Porcupine River station to the Upper West More area. Passive air quality monitoring in the Upper West More area. Noise monitoring in the Upper West More area.
Wildlife	Winter and summer goat aerial surveys in the Upper West More area. Western toad inventories along the active portion of the Galore Creek Access Road.

As the port was not included in previous studies, the following baseline technical studies, covering a variety of environmental and social topics, would be required to support the EA process, if the port is included:

- Coastal processes and flood risk assessment
- Water quality, sediment quality and benthic invertebrates
- Marine and freshwater environment
- Fish and fish habitat
- Coastal birds
- Terrestrial wildlife and vegetation
- Hydrological
- Geomorphology
- Air quality
- Noise
- Traffic (including truck traffic)
- Visual landscape
- Lighting

- Socioeconomics
- Archaeological and cultural heritage.

These studies would need to be completed in sufficient depth to cover all reasonably foreseeable baseline work that may be requested by the regulatory agencies. Baseline studies will be conducted as necessary to support an EA process.

20.3 Environmental Liabilities

Environmental liabilities on the Project site are those that would be expected from an exploration project with over 30 years of exploration activity. Over this time period, liabilities have included the exploration camp, drill pads and pad access roads constructed in support of ongoing drill programs, construction camps built to support access road construction, and the partially-completed access road itself.

GCMC reclaims disturbances associated with exploration programs, such as drill pads, and temporary access roads on an annual basis. Besides activities planned for 2011, GCMC has advised AMEC that there is no outstanding reclamation work within the Project area related to exploration activities. Annual reclamation reports are submitted to MEM to substantiate reclamation activities completed.

When the Project was placed on temporary care-and-maintenance in 2007, GCMC requested an assessment of costs that would be required to rehabilitate the access road and associated road construction infrastructure, including camps, from Highway 37 to the Galore Creek exploration camp.

GCMC maintains bonds to cover all environmental activity, and has advised AMEC that the monetary amounts lodged are sufficient to cover all current estimated environmental liabilities, including the 2011 planned programs and rehabilitation of the access road if required.

20.4 Closure Plan

Mine development and operation at Galore Creek will incorporate techniques that are designed to prevent or minimize future reclamation liabilities and to progressively reclaim areas affected by mining during the operations phase. Measures intended to minimize future liabilities include limiting the footprint of the disturbed areas to the smallest practical extent, underwater disposal of PAG materials, and construction that facilitates low risk reclamation design such as terraced non-PAG dumps.

Mine development will be planned with a focus on those aspects of development that have the potential for adverse impacts after the mine has ceased operating. To the extent practical, those aspects are minimized or avoided during planning and operation, rather than leaving them to be addressed during closure. GCMC has applied the "Design for Closure" concept to all aspects of the Project, including the management of PAG rock and the design of the tailings dam.

An allowance of 10 years of water quality monitoring is included; this assumption is subject to discussions with Provincial and Federal regulatory agencies. In addition, a provisional amount for infrequent inspection/maintenance of the spillway is included. The actual scope of this work cannot be determined at this stage; however, a fund based upon an average annual cost over 500 years is included. The work is not anticipated to be required annually.

The estimated total reclamation liability for the Galore Creek mine, including the filter facility, is $88.7 M at the end of the mine life. The estimate includes a contingency of 35%.

The most significant assumption underlying the reclamation cost estimate is that post-closure water quality predictions are found to be valid for all mine components. Some mine components, such as the PAG rock which is not immediately submerged or areas of pit walls could be sources of ARD and/or metal leaching. Additional geochemical assessment and water quality modeling, building on the work completed for the previous EA, is underway to inform further characterization of these risks. In the event that these problems arise, then potentially significant costs for post-closure water management/treatment may be incurred.

20.5 Tahltan Nation and Traditional Knowledge

With the exception of Stewart, the residents of northwestern BC are largely members of the Tahltan Nation living in the communities of Dease Lake, Iskut and Telegraph Creek.

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.

NovaGold initiated discussions with the Tahltan Nation in November 2004, within two months of signing the deal to explore Galore Creek. In early meetings with the Tahltan people, NovaGold agreed to support the formation of several joint ventures, NovaGold also funded local researchers to conduct studies that incorporated and documented traditional knowledge about the region.

Ongoing discussion with the Tahltan community resulted in the signing of the Participation Agreement on 10 February 2006. 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 the Project. The agreement allows for Project development with the support and involvement of the Tahltan Nation.

Much knowledge was gained from the consultation process. NovaGold's decision to select the modified northern access route over 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 impacts the Project could have on wildlife if concentrate were to spill into the environment. Pipelines to pump the concentrate from the process plant to Highway 37 and to supply diesel to site were incorporated in order to decrease the number of trucks on the access road.

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 valued-ecosystem components (VECs) that were assessed during environmental review, such as marmot and marten, were identified or selected through interviews with Tahltan members. Tahltan Elders and members confirmed the importance of protecting hunting grounds and wildlife stocks in the localized study area. The Participation Agreement between GCMC (previously with NovaGold) and the Tahltan provides for the development of a road protocol to address concerns about use of the access road. Traditional knowledge also provided GCMC 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.

20.6 Other First Nations

While the Project is located within the territory of the Tahltan Nation in northwestern British Columbia, access to the Project will be via Highway 37 through the Skii Km Lax Ha Traditional Territory and the Gitanyow Tradition Territory to the south. The proposed port facility is in the District of Stewart located in the Nisga'a Nation's Traditional Territory. An expanded communications and consultation program, the scope of which will be partly identified during the initial stages of the EA process, will be required to elicit the interests of the other First Nations within whose traditional territory project facilities are planned to be located. . Such a program will be needed to identify topics of interest to the various communities.

20.7 Cassiar Iskut–Stikine Land and Resource Management Plan

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 15 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.

20.8 Permitting

20.8.1 Exploration

Exploration work was carried out on the Project under Mines Act permit M-230 and mineral exploration permit numbers MX-1-621 (West More) and MX-1-622 (Copper Canyon).

20.8.2 Map Reserves

In 1969, two map reserve permits were granted to Stikine Copper by the B.C. government. Notation of Interest Number 85742 covers an area of approximately 1,060 hectares in the vicinity of the Anuk and Stikine Rivers and was established due to its importance as potential mill site, tailings disposal and town site. A second Notation of interest (Number 886067) was established to protect the most viable access route from Galore Creek to the Stikine River (Scud Airstrip). These reserves do not give the owner any exclusive rights to the areas but simply allowed for Stikine Copper to be informed of any other applications for land alienation. Both these permits have been renewed at various times and are presently extended until further notice by the Skeena Regional Office of Land and Water Management.

A large no-staking reserve was granted by the Ministry of Energy, Mines and Petroleum Resources in 1968 to cover a tract of land extending from the Galore Creek claims to the Stikine River. This was requested by Kennco (Stikine) Mining Ltd. for prevention of nuisance staking over a proposed tunnel access route to the property. In March 1989, this was amended to a "Conditional Reserve" where staking was permitted but claim holders could not interfere with any tunnel or underground workings created by previous mineral title holders.

Three Conditional Reserves[2] exist in the More Creek and Bob Quinn areas for hydrological purposes.

20.8.3 Project Development Permitting

Permitting and licensing was started in 2006. A list of the Provincial permits that will be required for Project development is presented in Table 20-2; the Federal permits that are likely to be required are shown in Table 20-3.

20.9 Comment on Section 20

In the opinion of the QPs, the following conclusions are appropriate:

- The new design and Project configuration is significantly different from the operation permitted in 2006
- A new Environmental Assessment process is likely to be requested by the regulators. This could involve parallel, harmonized, reviews by both the BCEAO and the CEAA. A Comprehensive Study Report may be required through the CEAA
- It is anticipated that the entire EA review process will require on the order of two full years from submission of a Project Description to issuance of a new EA Certificate by the BC government and a Decision by the federal Minister of Environment
- A port facility will now be required to be included in the EA. An assessment of the transportation of concentrate from the plant site to the port by road will need to be undertaken

2 In British Columbia, conditional reserves allow exploration and mining to proceed subject to these activities being compatible with a specified land use.

Table 20-2: Provincial Permit Requirements for Project Development

Permit or Licence	Legislation and Issuing Authority	Status
Environmental Assessment Certificate	*Environmental Assessment Act* - Environmental Assessment Office	Was Issued February 2007, but new EA to be prepared in 2012.
Permit Approving Work System and Reclamation	*Mines Act* - Ministry of Energy, Mines and Petroleum Resources (MEMPR)	Initial permit issued for exploration, tunnel development and early phase earthworks such as on site road construction. Will require amendments over time to accommodate additional works as the engineering and management plans are completed and approved. Once new EA process begins, this permit will likely be suspended.
Explosives Storage and Use Permit	Mines Act – MEMPR	Issued with federal *Explosives Act* Licence for storage and use of explosives for tunnel construction. Will likely be suspended late in 2011.
Water Licence – Storage and Diversion	*Water Act* - Ministry of Environment (MOE)	Not issued – required for storage of water behind a dam and diversion of streams
Water Licence – Use	Water Act – MOE	Not issued - required for water use for the mill and other uses.
Authorization for Changes in and About a Stream	Water Act – MOE	Issued for some works in and about streams along the access road. Additional approvals will be required for road construction and activities in the Galore Creek Valley
Water Licence for Land Improvement	Water Act – MOE	Not issued – required for stream diversions in Galore Creek Valley
Potable Water Permit	*Water Act* - Northern Health Authority (NHA)	Issued for some construction camps for use of water
Water Supply System Construction Permit	Drinking Water Protection Act (NHA)	Issued for some camps for construction of a water supply system
Short Term Water Use Approval	Water Act – MOE	Issued for some construction camps for use of water
Special Use Permit	*Forest Act* - Ministry of Forests and Range	Issued for access road. Will require amendments over time to accommodate evolution of alignment. May accommodate much of the transmission line as well. Additional Special Use Permit or amendment to existing Permit may accommodate aerodrome
Occupant License to Cut	*Forest Act* - Ministry of Forests and Range	Issued for initial works in the Galore Creek Valley and for construction of the access road. Additional licences will be required for the transmission line and aerodrome. Amendments may be required to existing licences as the footprint of development evolves
Road Use Permit	*Forest Act* - Ministry of Forests and Range	Issued – provides authority for use of existing forest service road
Forestry Licence to Cut	*Forest Act* - Ministry of Forests and Range	Issued for cutting of trees required to upgrade the existing forest service road
Mining Lease – Mine Site Facilities	*Mineral Tenure Act* – MEMPR	Not issued -required for Galore Creek Valley mine operation and facilities
Mining Lease – West More Site Facilities	*Mineral Tenure Act - MEMPR*	Not issued - required for West More area plant site, tailings area, conveyor and tunnel
Mining Lease – Km 8.5	*Mineral Tenure Act - MEMPR*	Not issued -required for filter plant site
Highway Access Permit	*Highway Act* - Ministry of Transportation	Issued – permits connection to Highway 37. May require amendment over time as Project proceeds
Geotechnical Investigation Permit	*Highway Act* - Ministry of Transportation	Issued for assessment of intersection of access road with Highway 37
Highway Upgrade Permit	*Highway Act* - Ministry of Transportation	Issued for upgrade of intersection of access road with Highway 37
Permit to Remove Pit Run	*Highway Act* - Ministry of Transportation	Issued for use of gravel to upgrade intersection with Highway 37
Pipeline Permit	*Pipeline Act* - Oil and Gas Commission	Application not complete. Approves construction and use of pipelines
Licence of Occupation –	*Pipeline Act* - Oil and Gas	Not issued – provides tenure for pipelines

Permit or Licence	Legislation and Issuing Authority	Status
Pipeline	Commission	
Licence of Occupation, and later, Statutory Right-of-way – Transmission line	*Land Act* - Land Management Bureau, Ministry of Agriculture and Lands (MAL)	Not issued
Licence of Occupation – Communication Towers	Land Act – MAL	Not issued – will be required communication towers
Effluent Discharge Authorization	Environmental Management Act – MOE	Temporary authorization issued for tunnel discharge. No longer valid.
Waste Management Permit – Effluent Discharge	Environmental Management Act – MOE	Not issued – will be required for long term approval of discharge of tunnel water, tailings, water from filter plant, etc.
Waste Management Permit – Air	Environmental Management Act – MOE	Not issued – will be required for crushers, concentrator, incinerators, etc.
Waste Management Permit – Refuse	Environmental Management Act – MOE	Not issued – will be required for landfills
Special Waste Generator Permit (Waste Oil)	Environmental Management Act – MOE	Not issued
Municipal Sewage Regulation Registration	Environmental Management Act – MOE	Issued for some camps to authorize use of sewage system and related discharge
Incinerator Permit	Environmental Management Act – MOE	Issued for some camps
Hunt/Trap/Kill Nuisance Bears	*Wildlife Act* - Ministry of Environment	Issued for control of nuisance bears

Table 20-3: Federal Permit Requirements for Project Development

Permit or Licence	Legislation and Issuing Authority	Status
Canadian Environmental Assessment Act Approval	*Canadian Environmental Assessment Act* - Canadian Environmental Assessment Agency	Issued in June 2007. Will be replaced following new EA process.
Fish Habitat Compensation Agreement	*Fisheries Act* - Department of Fisheries and Oceans	Issued - Will require expansion and revision. Compensation for road construction completed.
Section 35 (2) Authorization	*Fisheries Act* - Department of Fisheries and Oceans	Issued to approve some harmful alteration, disruption or destruction of fish habitat
Metal Mining Effluent Regulation	*Fisheries Act* – Environment Canada	Not issued, no application yet made. Establishes allowable discharge chemistry.
Approval for Navigable Water Work(Bridge Crossings)	Navigable Waters Protection Act - Transport Canada	Issued for construction of some bridges. Renewals applied for regarding three bridges.
International River Improvements Permit	International River Improvements Act – Environment Canada	Applied for, not issued. Process takes about a year. Application may require revision if dam design changes.
Explosives Factory Licence	*Explosives Act* - Natural Resources Canada	Not issued
Explosives Magazine Licence	*Explosives Act* - Natural Resources Canada	Issued with Provincial Explosives Storage and Use Permit
Ammonium Nitrate Storage Facilities Permit	Canada Transportation Act - Transport Canada	Not issued
Radio Licences	Radio Communication Act -	Issued for existing radio equipment
Radioisotope Licence	Atomic Energy Control Act	Not issued

- A key feature of the new EA will be the consideration of cumulative effects of the Galore Project with other projects that have been developed or are proposed for development since the previous EA was completed. This may include the Northwest Transmission Line, the Forrest Kerr hydroelectric project, the concentrate transport from Yukon Zinc's Wolverine mine, and the Red Chris project
- There will be numerous commitments for the implementation of Environmental Management Plans, environmental monitoring and follow-up. The commitments made during the previous EA will be revisited and adopted, amended or discarded as appropriate
- In the period leading up to the EA review process, GCMC should meaningfully engage with all communities of interest, including the First Nations, within whose traditional territory the proposed project facilities will be located
- Ongoing discussion with the Tahltan community resulted in the signing of a Participation Agreement on 10 February 2006
- The Project is located within the territory of the Tahltan Nation. The Project access route via Highway 37 will be through the Skii Km Lax Ha Traditional Territory and the Gitanyow Traditional Territory to the south. The proposed port facility is in the District of Stewart located in the Nisga'a Nation's Traditional Territory
- An expanded communications and consultation program, the scope of which will be partly identified during the initial stages of the EA process, will be required to elicit the interests of the other First Nations within whose traditional territory project facilities will be located
- The Project falls within the boundaries of the Cassiar Iskut–Stikine LRMP. Mineral exploration and development are accepted activities within the Coastal Grizzly/Salmon Management zone, including road access where needed
- GCMC has identified the key Provincial and Federal permits that will be required for construction of a mine under the assumptions in the GCMC 2011 pre-feasibility study.

21.0 CAPITAL AND OPERATING COSTS

21.1 GCMC 2011 Pre-feasibility Study Capital Cost Estimate

The capital cost estimate for the Project was developed by GCMC, with input from consultants for specific areas as indicated in Table 21-1.

Under the direction and supervision of GCMC, where applicable, third-party consultants acted as the principal engineers for their specific areas of expertise. GCMC and one of the third-party consultants compiled the estimate from detailed sets of quantities provided by all third-party consultants and in some cases, specialized costs (i.e., tunnelling, pipeline, powerline and marine works at the Stewart port facility) were provided by each respective consultant. All third-party consultants participated in the scheduling risk and overall execution strategies. Regular meetings were held with the third-party consultants to establish a clear understanding of the scope and how the direct costs relate to their reported indirect costs.

The costs were reported by GCMC as at a prefeasibility level of confidence where the estimate accuracy range is defined as +25%/-20% including contingency and are consistent with an AACE Class 4 Estimate. Vendors' pricing was obtained for long-lead items, and the consultants actively participated in obtaining appropriate level of budget quotations.

Due to the specific nature of this Project, where sections (i.e., portions of the access road) are under construction or complete, the accuracy of this estimate in some cases exceeded the expected accuracy range of a prefeasibility estimate. GCMC considers that the level of detail applied to develop Project execution strategies, schedule, and costs is in some areas more detailed than is typical for a pre-feasibility level study.

Historical cost data were applied to the GCMC 2011 pre-feasibility study estimate where applicable.

Costs in the GCMC 2011 pre-feasibility study were reflective of Q4 2010 market conditions. GCMC and its third-party consultant assessed overall construction personnel requirements, material availability and logistics, work methods, and risks. Escalation was excluded for anticipated future costs during the construction schedule up to and including pre-commissioning.

Table 21-1: Corporations Contributing to GCMC Capital Cost Estimates

Area	Corporation Responsible
Access Roads	All North
Tunnel	GCMC Third-party Consultant
Mine	GCMC
Civil Works in the Galore Creek Valley	AMEC Earth and Environmental
Tailings Dam	AMEC Earth and Environmental
Crushing / Conveying / Concentrator	GCMC Third-party Consultant
Concentrate pipeline	PSI
Transmission Line	Knight Piésold
Port Facility	WorleyParsons Canada Services Ltd

Note: Companies identified in this table as “GCMC Third-party Consultant” are unable to be identified under the terms of their contract with GCMC.

21.1.1 Labour Assumptions

Approximately 13 M direct and indirect manhours were projected to be associated with construction. Personnel requirements on the Project will peak at 1,800 people (including Galore Creek site, Smithers and Stewart) based on total manhours (direct + indirect manhours). Work rotation assumptions for on-site personnel were a three weeks on/one week off rotation with a standard workday comprising 10 hours per day, and seven days per week. Crew rotations were staggered to such that work can be performed on a continuous basis.

Craft labour rates were established based on a representative set of current CLAC union labour rates agreement in 2010, considered typical for contractors likely to be engaged to perform the work.

Camp and catering costs were included separately in the indirect cost portion of the estimate.

Travel costs were based on labour being sourced principally from British Columbia (80%), with the balance from other western Canadian provinces (10% Alberta, 10% Saskatchewan). Personnel will be flown in to Bob Quinn airport, and then transported by bus to the various camps and work areas.

Productivity factors were developed for each discipline and applied to the base manhour units where applicable.

21.1.2 Material Costs

Average all-inclusive construction equipment rates (Canadian dollars per manhour) were estimated by discipline.

All permanent material, including bulk costs (i.e., concrete, steel, cables etc.), is covered under material costs. Bulk material costs (where applicable) were based on escalated prices from other projects. The cost of temporary and consumable materials used during construction of direct works (i.e., drill steels, formwork, welding consumables, grout, temporary supports, etc.) were also included in the material costs.

For all major equipment, budget quotations were obtained. These vendor quotations were reviewed for completeness and technical adequacy, and where vendors' quotations were incomplete an appropriate factor was applied before the final costs were incorporated into the estimate.

An independent logistics plan was prepared by a third-party consultant to GCMC to determine the overall costs associated with freight forwarding (including barge transportation) to Stewart and trucking to site.

All sales taxes, including Harmonized Sales Tax (HST), were excluded.

21.1.3 Contingency

The GCMC 2011 pre-feasibility study estimate included an 18% contingency at P85 with an expected accuracy range of ±19%.

21.1.4 Mine Capital Costs

Mine capital costs were prepared using a combination of benchmarks and internal pricing information. Benchmarks from AMEC (2008) were used when specific quotes were unavailable for the GCMC 2011 pre-feasibility study estimate. The total mine capital was estimated at $561 M.

Mine equipment capital costs reflected the equipment requirements for pre-production and the first year of production. Equipment unit costs were based on internal budget quotes. The unit costs included all costs required to prepare the equipment for production, excluding sales taxes.

The pre-stripping capital cost covered all mine operating costs required to move the scheduled tonnage during pre-production. It was estimated that 5 km of haul roads will need to be constructed in preparation for mining. The cost of building the initial haul roads and site preparation was benchmarked from the 2009 Mining Cost Service at $16 M.

21.1.5 Owner's Costs

Owner's costs were developed by GCMC and totalled $111 M. Costs included provision for corporate costs, Project management costs, and commissioning and ramp-up costs. Ramp-up and working capital costs were calculated from the operating cost budget based on a fixed portion and variable portion for a six-month period. The assumption took into consideration the increasing usage of variable costs such as power, diesel, and reagents.

21.1.6 Sustaining Capital

Sustaining capital costs were estimated by GCMC and its third-party consultants, and were partly based on MTO information provided by specialists in tailings.

The sustaining capital costs were assumed to be executed by both independent contractors and the Owner. Cost areas considered in the estimate included:

- Replacements of the mining fleet and major components over the life-of-mine
- Capitalization of mining costs for the haul and replacement of waste rock in the tailings storage facility
- Provisions for an annual raise of the tailings dam
- Dredging costs associated with the port facility in Stewart. These costs were assumed to be incurred during Year 10
- Construction of a by-pass road around the tailings storage facility in Year 10
- Upgrades to the process plant, including installation of a pebble crusher in Year 3.

21.2 AMEC Review of Capital Cost Estimate

AMEC performed a detailed estimate review of the capital cost estimate developed by a GCMC Third-party Consultant as at March 16, 2011. Some changes were subsequently made by GCMC to the estimate; these later adjustments were generally reviewed by AMEC.

Late adjustments to the GCMC 2011 pre-feasibility study estimate that were performed by GCMC included modifications to crusher costs, management and Owner costs, moving some items from capital costs to sustaining capital, changes in allocations of contingency costs, and adjustments to civil rate estimates. These late adjustments reduced the capital cost estimate from the $5,135.40 M provided to AMEC in March 2011, to $5,060.08 M reported in the GCMC 2011 pre-feasibility study to $5,014.22 provided to AMEC on 6 July 2011. The total late adjustments amounted to a decrease of $121.18 M in the estimate.

The capital cost estimate that was provided to AMEC on March 16, 2011 was in Excel format. High-level cost analyses were performed on major cost centres to determine if the costs were reasonable when compared to AMEC in-house data. No detailed line-by-line estimate review was performed.

AMEC concluded from the review:

- The work breakdown structure (WBS) was sufficient to break down the costs into reasonable portions; however, there was some overlap in the facility codes; this did not present any issues in the estimate structure for review purposes, but the issue should be addressed in future studies
- In areas where work sites overlap with third-party consultants such as the access road, concentrate pipeline and diesel pipeline, coordination of design and construction schedule was limited. A potential cost impact exists with regard to aligning design and construction expectations, but is difficult to assess at this point in the Project
- The common bulk fuel rate, as described in the Basis of Estimate document, appeared not be used consistently throughout the estimate by GCMC and its third-party consultants. The potential cost impact may not be significant, but consideration should be given in future estimates such that all parties adhere to common bulk rates for the Project
- The construction labour rates used for the GCMC and its third-party consultant portion of the estimate have been built-up in a reasonable manner for a pre-feasibility study and are in an acceptable range of cost per hour
- The productivity loss adjustment may be excessive, in the order of 5–10%, which could relate to a potential approximate cost reduction impact of $25–50 M
- The application of some design development allowance is reasonable for a pre-feasibility study
- The review of the earthworks cost estimate indicated to AMEC that earthworks costs were underestimated. The late adjustment to the capital cost estimate was partly to address the earthworks discrepancies noted by AMEC. The contingency on earthworks is approximately 24%, which is reasonable for the level of study
- The concrete supply unit rate was based on a Project average price and not site-specific for each area of the Project. Depending on the quantities required at each physical location and number of batch plants required, there may be a cost impact
- The aggregate supply unit rate was based on a Project average from in-house data. The rate used would imply that aggregate supply is close to the work site. Although the methodology is not unreasonable for a pre-feasibility study, the

potential cost impact of developing the costs relative to each work location may be significant

- The material pricing methodology is sufficient for a pre-feasibility study, if there is no requirement to obtain materials from a specific country. The installation unit manhours are reasonable as compared to AMEC in-house standards
- Building costs were developed mainly with third-party consultant in-house data, with the exception of obtaining a quotation on the permanent camp facilities. That approach is acceptable for a pre-feasibility study
- The estimating methodology for mechanical equipment, piping, and instrumentation is reasonable for a pre-feasibility study
- AMEC reviewed the basis of estimate and production rates for the access road estimates and found the estimating methodology to be reasonable for a pre-feasibility study. AMEC notes concerns regarding schedule coordination with PSI activities related to the construction of the concentrate pipeline along the road and More Canyon bridge
- The estimating methodology for the concentrate pipeline is considered reasonable for a pre-feasibility study, although AMEC has general concerns regarding coordination of the provided estimate in relation to the Basis of Estimate document in terms of common rates, the access road engineering, diesel pipeline, and overall Project schedule
- The percentage of indirect costs to direct costs is within an expected range for a complex project in Northern British Columbia. The review found the indirect estimate methodology generally to be very good for a prefeasibility-level study
- As presented in the Basis of Estimate document, the Owner’s costs include typical cost centres such as personnel, corporate costs, environmental programs and insurance
- The indirect cost was estimated generally by percentage, with consideration given for GCMC management and the operating mine providing some level of construction support and services. The methodology is reasonable for a pre-feasibility study
- As AMEC identified potential undercalls in some areas of the capital cost estimate, AMEC has re-stated the capital cost estimate as shown in Table 21-2. The potential cost impact of $140 M for additional earthworks and the tunnel area would increase the capital cost estimate to a rounded $5,160 M.

Table 21-2: Restated Capital Cost Estimate

Description	$ Millions
Mine	**357**
Plant	**835**
Tunnel	**580**
Infrastructure	**697**
Total Direct Costs	2,470
Mine and Pre-production Costs	**582**
Indirect Costs	**1,320**
Owner's Costs	**111**
Contingency	**678.**
Total Capital	5,160

Note: Numbers may not sum due to rounding.

21.3 AMEC Review of Sustaining Capital Cost Estimate

AMEC re-estimated the mining sustaining capital based on the mine plan in Section 16.

Total sustaining capital requirements are estimated at $552 M, and include, over the planned life-of-mine, the following rounded sustaining capital estimates:

- Mine: $163 M
- Plant: $66 M
- Tailings: $212 M
- Common Infrastructure: $100 M
- Port and Loadout: $10 M.

Sustaining capital costs are summarized in Table 21-3.

21.4 GCMC 2011 Pre-feasibility Study Operating Costs

The operating costs were estimated in Q4 2010 Canadian dollars and did not include allowances for escalation or exchange rate fluctuations. Operating costs were built from first principles where costs were expected to be material, such as for power, fuel, and labour. Costs were divided into six departments, mining, processing, port, infrastructure, site G&A, and corporate.

Table 21-3: Sustaining Capital Cost Estimates

Year		Mine ($M)	Plant ($M)	Tailings ($M)	Common Infrastructure ($M)	Port/ Loadout ($M)	Total Costs ($M)	% of Total
01	2018	$57	$5	$19	$0	$0	$81	16%
02	2019	$6	$26	$14	$0	$0	$46	13%
03	2020	$0	$34	$18	$0	$0	$52	8%
04	2021	$33	$0	$17	$0	$0	$50	7%
05	2022	$0	$0	$24	$0	$0	$24	4%
06	2023	$0	$0	$6	$0	$0	$6	2%
07	2024	$0	$0	$6	$0	$0	$6	1%
08	2025	$0	$0	$6	$0	$0	$6	3%
09	2026	$0	$0	$6	$100	$0	$106	17%
10	2027	$2	$0	$44	$0	$10	$56	14%
11	2028	$2	$0	$6	$0	$0	$8	1%
12	2029	$49	$0	$6	$0	$0	$55	1%
13	2030	$12	$0	$6	$0	$0	$18	3%
14	2031	$2	$0	$6	$0	$0	$8	2%
15	2032	0	$0	$6	$0	$0	$6	1%
16	2033	$0	$0	$6	$0	$0	$6	1%
17	2034	$0	$0	$6	$0	$0	$6	2%
18	2035	$0	$0	$6	$0	$0	$6	3%
19	2036	$0	$0	$6	$0	$0	$6	1%
20	2037	$0	$0	$0	$0	$0	$0	2%
	TOTAL	**$163**	**$66**	**$212**	**$100**	**$10**	**$552**	**100%**

Note: Totals may not add due to rounding

The operating cost estimates included all the costs associated with the mining, processing, and infrastructure activities for a large-scale mining operation in northern BC.

21.4.1 Basis of Estimate

Major unit costs were estimated from first principles, vendor quotations, site experience, or other methods as outlined in this sub-section. Quantities of materials and equipment were estimated based on the Project design documents.

Salaries and hourly wages for each job category were based on input from NovaGold's and Teck's experiences of similar functions in BC mines. An average burden rate of 50% was applied to base salaries to include all statutory Canadian and BC social insurance, medical and insurance costs, pension costs, and vacation costs.

Typical bonus and long-term incentive plans estimates were also included on a position basis. All positions were categorized for one of five work locations (Galore Creek Valley camp, West More camp, Bob Quinn camp, Smithers or Stewart) to facilitate camp requirement estimates and to reduce overall camp costs. Labour costs did not vary by work-site location. All site personnel were scheduled to work a standard two-week-in/two-week-out schedule. It was assumed that the workforce would be non-union.

Property taxes were based on estimates of extrapolation of existing tax agreements, and non-labour environmental and insurance costs were estimated based on Teck's experience with operating similar-sized mines in BC.

Non-mining mobile equipment requirements were based on the experience of similar large mines in BC, with hourly usage costs calculated based on hourly fuel consumption and maintenance factors provided by GCMC and its third-party consultant.

Major operating supplies, including process reagents, lime, grinding media, and mill/crusher liners were estimated based on budget quotations from suppliers. Freight for these items was included in the unit cost calculation as different unit freight rates are projected to be applied to these bulk commodities.

Fuel was estimated at a delivered cost to site of $1.04/L and was based on Teck's internal fuel supply models for use in supporting other large mines in BC. The difference between this price and the $1.15/L used elsewhere in the study was not considered by AMEC to have a material impact on the cost estimate.

Power costs were estimated based on current industrial power rates within BC. Power costs, including both demand and consumption charges, were estimated to be $50/MWh ($0.05/kWh). Power usage was based on average calculated power draw of all equipment identified in the Project equipment list, including transmission line losses.

Maintenance supply costs were estimated as 5% of the purchased mechanical equipment cost. Freight was estimated as 5% of the maintenance supply costs.

21.4.2 Mining Operating Costs

Mine operating costs were separated into the cost activities of drilling; blasting; loading; hauling; and mine general. The operating costs for equipment were obtained from the Teck's strategic resourcing group, and were also benchmarked from operating mines. Maintenance labour rates were taken from estimates provided by Teck's strategic sourcing group. Where information was not available, a benchmarked ratio of 0.44 mine maintenance personnel to mine operations personnel was used. This ratio was benchmarked from the AMEC (2008) review. Similarly, parts, materials, and supplies were benchmarked from AMEC (2008).

The cost estimates were based on hourly cost and productivity benchmarks for unit mining activities.

Staff salaries were based on GCMC forecasts. Hourly labour rates were benchmarked from the Highland Valley Copper Collective Bargaining Agreement. The mine general cost center captured the operating expenditures associated with areas such as support equipment, pit dewatering, slope stability, engineering, geology and supervision.

Drilling

For the purposes of the GCMC 2011 pre-feasibility study, the highwall drill operating cost was estimated to be 15% of the total production hours. Operating hours for the tank drill were assumed to be 20% utilization for a total of 1,269 working hours per year out of a maximum 6,344 operating hours per year.

For production drills a penetration rate of 25.2 m/h was selected. After reductions in set-up and move, the total cycle time was 43.3 minutes per 16.5 m hole. Drill operating costs included operating and maintenance labour; parts/materials/supplies; and fuel/energy. Maintenance labour was developed using a ratio of 0.44 mine maintenance personnel to mine operations personnel.

Total drilling costs are estimated at $321 per operating hour, and the life-of-mine average drilling cost is projected to be $0.05/t.

Blasting

The powder factor and other parameters were the same as used in AMEC (2008). The blasting component of mine operating costs was assumed as a down-the-hole service.

With this type of arrangement, the explosive supplier assumes the capital portion of the explosives plant and related magazines as well as the costs associated with putting the product in the ground. A cost of $0.73/kg was the "all-in" down-the-hole service. In addition to the all-in service, there were two blasters scheduled per crew (dayshift only) for tie-in and blast detonation, including stemming material, boosters, downhole and surface delays, and surface detonation cord.

Further work is needed at the next phase of study to fully capture additional accessories that would be required in a high snowfall environment. These additional accessories could include hole markers and blast hole buckets.

The blasting cost estimate as an average over the life-of-mine is $0.19/t.

Loading

The energy consumption rate for the shovel was assumed to be 937 kWh per operating hour, based on benchmarking from a similar-size operation. The loader's fuel consumption was rated at 195 L per operating hour and was benchmarked from AMEC (2008).

The hydraulic shovel fuel consumption was provided through manufacturer estimates and is rated at 542 L per operating hour. The total loading cost varies annually to reflect different production allocations between the shovels and loader.

Operating hours were allocated to the loader for its auxiliary duties. Annual operating hours for pioneering were based on 5% of run-of-mine production. Annual operating hours for snow removal were based on assumptions of surface area, compaction, equipment performance, and snowfall (10 m). No unit in the loading fleet is expected to require replacement during the mine life.

The loading/operating cost estimate was $648 per operating hour for the front-end loader, $866.9 per operating hour for the hydraulic shovel, and $451 per operating hour for the electric rope shovel.

Hauling

Hauling costs were based on operating hours necessary for a 345 t haul truck fleet to achieve the production schedule, inclusive of run-of-mine and snow removal hours.

The hauling cost estimate was $523.45 per operating hour, and the hauling cost estimate averaged $1.02/t over the life-of-mine.

21.4.3 Mine General

The mine general cost was intended to account for costs associated with supporting the direct mining operations; these costs include operation and maintenance of support equipment; slope stability; pit dewatering; engineering; geology; supervision; and other miscellaneous cost items. The miscellaneous costs were benchmarked from Highland Valley Copper.

The main support equipment fleet will consist of track dozers, rubber-tire dozers, graders, wheel loaders, excavators, and water trucks. The number of equipment operators varies directly with the number of units operating each year. The organizational structure for supervisory personnel was benchmarked to, and slightly modified from, AMEC (2008).

21.4.4 Processing Costs

Process operating costs included all costs associated with crushing, conveying, grinding, flotation, dewatering, tailings disposal, and water treatment unit operations. Operating costs associated with the concentrate slurry pipeline between the mill site and the Bob Quinn area were excluded from the process costs as they were included in the infrastructure costs.

People-on-role (POR) required to operate, maintain, and manage the process unit operations were estimated to be 52 salary and 144 hourly employees.

Processing will consume the majority of the electrical power on the site. Of the total average site power draw of 156 MW, 127 MW of power will be consumed by the process operations. Of this total, the two major consumers of energy will be the grinding circuit (73 MW) and the ore conveyance system (30 MW). Power costs for crushing and conveying of ore vary year-by-year based on mill throughput; as the grinding circuit is expected to be fully utilized throughput the mine life, grinding power costs were assumed to be constant on a year-by-year basis. In a typical year, it was estimated that process electrical power costs will be $60 M/a. Diesel fuel consumption was calculated based on estimates of mobile equipment used to support process operations.

Operating supplies included process reagents, grinding mill liners, and grinding media. Process reagent consumption was based on laboratory-scale testwork, and was established during a pilot plant run in 2008. Reagent costs were based on current quotations from established suppliers capable of supplying the required quantities to the site. Grinding media consumption rates were based on calculated wear rates modified by actual wear rates at similar mines. Media unit rates were estimated based on Q4 2010 steel prices, with quotations supplied by an established supplier in British Columbia. Relining rates were based on estimates of reline frequency, for all grinding mills and crushers.

Liner unit rates were estimated based on Q4 2010 steel prices, with quotations supplied by an established supplier in British Columbia.

Maintenance supplies were estimated to be 5% of the original capital cost of process equipment at $167 M/a.

Contracts and other miscellaneous costs supporting process operations were estimated to be $0.55 M/a.

21.4.5 Port Costs

The port operating costs included all costs associated with the receiving and storage of concentrates at the port facility, as well as costs associated with loading bulk carriers. Transportation (trucking) costs from the Bob Quinn site to the port were not included in the port costs.

People-on-role required to operate, maintain, and manage port functions were estimated to be three salaried and 20 hourly employees.

Port power consumption was a relatively small portion of the overall Project power requirement, and was estimated to average 2 MW.

Maintenance supplies were estimated to be 5% of the original capital cost of port equipment at $4.7 M/a. Fuel consumption was estimated based on predicted mobile equipment requirements for port operations.

21.4.6 Infrastructure Costs

Costs associated with the four major infrastructure items, the mine access road from Highway 37 to the south tunnel portal, the high voltage power transmission line from the Bob Quinn substation to the West More substation, the slurry concentrate pipeline from the mill site to Bob Quinn, and the access tunnel from the West More area to the Galore Creek Valley were included within this cost category.

No Owner labour is associated with these activities, as the majority of work focused on these items will be contract-based.

Electrical requirements for infrastructure were limited to the power required to pump concentrate from the mill to the filter plant at Km 8. Minor power usage for items such as tunnel ventilation fans and lighting were included within other categories based on power source location. Power losses associated with transmission line losses were distributed amongst the site power consumers on a pro-rated basis.

Infrastructure contract costs included three major contracts. The largest of these three was the road maintenance contract, estimated to cost $3.6 M/a due to the heavy snowfall and extended winter season. A tunnel maintenance and power line maintenance contract will be required, and were estimated to be $0.3 M/a and $0.1 M/a, respectively.

21.4.7 Site G&A Costs

G&A people-on-role requirements were estimated to be 68 salary and 46 hourly employees. This included multiple smaller functional groups; site services, environmental, human resources, safety, warehousing and supply chain management, and accounting. Labour requirements within these groups were estimated from experience of other similar-size operations in Canada.

Non-labour environmental and insurance costs were estimated as $5.0 M/a and $5.3 M/a, respectively, based on Teck's experience with operating similar-sized mines in BC. Communications included the cost of maintaining satellite system for television use and for the telephone and internet systems.

The site services group were expected to complete tasks such as site road maintenance, material transport (supplies/earthworks) in the West More area, and provide support for the tailings construction dam contractor. Fuel and maintenance supplies for the site services group were estimated based on projected equipment requirements and usage. Freight for all items in inventory was included within this category (with the exclusion of process liners, grinding media, and process reagents) and was estimated to be 5% of the value of the inventory, or $1.4 M/a.

The majority of the G&A fixed costs were associated with camp costs including transportation of crews to site. Air service to Bob Quinn was estimated to be an average of $500 per round trip, per person. It was assumed that small charter aircraft reporting to multiple destinations will be utilized to access both local and remote labour markets. A round trip, per person cost of $75 was applied for bus transport from the Bob Quinn airstrip to the employee's assigned camp. Daily camp costs were based on escalated 2007 camp quotations from a local camp catering company currently under contract with GCMC. The West More and Bob Quinn (Km 8) permanent camps were assumed to be 90% full at all times. The Galore Creek Valley camp occupancy was calculated on a year-by-year basis based on mine workforce plus allowances for contractors. Total camp costs were estimated to be $11 M/a, and total travel costs were estimated to be $4.6 M/a.

Avalanche control contract costs were estimated at $0.6 M/a based on quotations received from local vendors. Other minor contracts, such as donations, legal support, business travel, and recruiting costs were estimated based on operating experience.

21.4.8 Corporate Costs

Corporate costs include property taxes, community relations cost, and concentrate transportation costs.

Concentrate transportation costs were based on contracted truck haulage from Bob Quinn to Stewart in 50 t B-train haul trucks. Estimates were received from multiple haulage contractors operating in the vicinity, and a logistics specialist reviewed the data to determine a per tonne estimate for haulage costs. The estimated unit cost, including fuel, was $22.89 per wet metric tonne of concentrate. The total shipping cost will vary from year to year depending on the total amount of concentrate produced.

It is expected that a small GCMC corporate office, providing workspace for 17 employees, will be maintained in Smithers.

The total estimated corporate costs were $21 M/a.

21.5 AMEC Review of Operating Cost Estimate

AMEC reviewed the operating cost estimate and made adjustments to the GCMC 2001 pre-feasibility study operating cost data to accommodate the review findings. These included additional allocations to operating costs in the mining area to accommodate a changed mining schedule from the GCMC 2001 pre-feasibility study. The operating cost estimates are re-stated in Table 21-4.

Table 21-4: Restated Operating Cost Estimate

Area	$ per tonne milled
Mining	6.70
Process	5.76
Port	0.16
Corporate	1.56
Infrastructure	0.16
G&A	0.73
Total	**15.07**

21.6 Comment on Section 21

In the opinion of the AMEC QPs:

- The GCMC capital cost estimates are based on a combination of quotes, vendor pricing, and experiences with similar-sized operations. Capital cost estimates were reported by GCMC at a pre-feasibility level where the estimate accuracy range is defined as +25%/-20% (including contingency) and are consistent with an AACE Class 4 estimate. The GCMC 2011 pre-feasibility study estimate was considered to have an 18% contingency at P85 with an expected accuracy range of -4% to +20%. The GCMC 2011 pre-feasibility study capital cost estimates include direct and indirect costs
- AMEC reviewed the estimate and considered that the requirement for earthworks and tunnelling was low, and made an adjustment of $141 M to cover these areas. AMEC re-stated the capital cost estimate to $5,160M
- AMEC re-stated the sustaining capital cost estimate to $552 M
- When sustaining capital ($551.7 M) and closure costs ($88.7 M) are incorporated into the capital cost estimate, the total Project capital cost estimate is $5,840 M
- In the GCMC 2001 pre-feasibility study estimate, operating costs were based on estimates from first principles for major items, and included allowances or estimates for minor costs. The assumed power cost was $50/MW-hr and the assumed diesel fuel cost was $1.04/L; not all parties quoted using these pre-set assumptions. Manpower requirements were based on industry experience with similar-scaled operations
- AMEC reviewed the operating cost estimate and made adjustments to cover a revised mine plan. Operating costs are restated as $15.07/t milled.

22.0 ECONOMIC ANALYSIS

The results of the economic analysis represent forward-looking information that are subject to a number of known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Forward-looking information includes Mineral Reserve estimates, commodity prices and exchange rates, the proposed mine production plan, projected recovery rates, tunnel construction costs and schedule, other infrastructure construction costs and schedules, use of a port facility at Stewart, and assumptions that a revised and amended EA will be approved by Provincial and Federal authorities.

22.1 Valuation Methodology

Financial analysis of the Galore Creek Project was carried out using a discounted cash flow (DCF) approach. This method of valuation requires projecting yearly cash inflows (or revenues) and subtracting yearly cash outflows (such as operating costs, capital costs, royalties, and taxes). The resulting net annual cash flows are discounted back to the date of valuation and totalled in order to determine the Net Present Value (NPV) of the Project at selected discount rates.

The internal rate of return (IRR) is expressed as the discount rate that yields an NPV of zero.

The payback period is the time calculated from the start of significant Project cash flows until all initial capital expenditures have been recovered.

This economic analysis includes sensitivities to variations in operating costs, capital costs, and metal prices.

All monetary amounts are presented in Canadian dollars (CAD$).

For discounting, cash flows are assumed to occur at the end of each period. Net present value calculations were adjusted so as to make the Project start align with the beginning of 2013.

22.2 Financial Model Parameters

22.2.1 Resource and Mine Life

The Proven and Probable Mineral Reserves will be processed at an average rate of 34.6 Mt/a over a planned mine life of approximately 18.5 years (including one year of pre-production).

22.2.2 Metallurgical Recoveries

Copper, gold and silver recoveries are estimated for each year of production and average 90.6%, 73.1% and 64.5% respectively over the life-of-mine.

22.2.3 Smelting and Refining Terms

The base case analysis incorporates the copper treatment and refining terms shown in Table 22-1.

22.2.4 Metal Prices

A range of metal prices shown in Table 22-2 were used to assess the Project. The base case is highlighted for the Project. Case 6 represents metal spot prices as of July 26, 2011.

22.2.5 Operating Costs

Operating costs used for the financial analysis are averaged as follows:

- Mining: $6.70/t milled
- Process: $5.76/t milled
- Port: $0.16/t milled
- Corporate: $1.56/t milled
- Infrastructure: $0.16/t milled
- G&A: $0.73/t milled.

This resulted in an overall average rounded operating cost for the Project of $15.10/t milled.

Table 22-1: Smelting and Refining Terms

Term	Unit	Amount	Pay Factor
Copper pay			
Moisture content	%	8.0	
Concentrate losses	%	0.10	
Ocean freight	USD/wmt	53.00	
Treatment charge	USD/dmt	70.00	
Refining charge	USD/pay lb	0.070	
Pay factor	%	96.5	
Unit deduction	%	1%	
Gold pay			
Refining charge	USD/oz	6.00	
		Ceiling (g/t)	% Pay
Pay factor		0	0.0%
		1	90.0%
		3	92.0%
		5	95.0%
		8	96.0%
		10	97.0%
		15	97.5%
Silver pay			
Pay factor	%	90.0%	
Unit deduction	g/t	30.0	
Refining charge	USD/oz	0.40	

Table 22-2: Metal Price Ranges (base case is highlighted)

		Base Case	Low Case	Current Case (spot metal prices as at 26 July 2011
Copper	US$/lb	2.65	2.00	4.44
Gold	US$/oz	1,100	900	1,613
Silver	US$/oz	18.50	15.00	40.34

22.2.6 Capital Costs

The distribution of the estimated Project capital costs are as follows:

- Construction capital: $5,160 M
- Sustaining capital: $552 M
- Closure costs $89 M
- Total Project capital cost: $5,840 M.

22.2.7 Royalties

Royalty payments are due to the Tahltan nation and are calculated as a percentage of the Net Smelter Return on 100% of the production, on a moving scale. There is a minimum annual payment and higher payments when the cumulative NSR (measured in US$) passes certain thresholds. There is no life-of-mine cap to these payments. Using Base Case prices the current financial model estimates the total value of the royalty payment at $166 M.

22.2.8 Working Capital

A working capital allocation of three months operating cost was included in the cash flow model. The allocation varies throughout the Project life and peaks at $142 M. The assumption is made that all of the working capital can be recovered at Project termination. Thus, the sum of all working capital over life of mine is zero.

22.2.9 Taxes

AMEC does not provide expert advice on taxation matters. The simplified tax calculations are made based on information provided by GCMC together with documentation that is publicly available.

It is assumed that the Galore Creek Project will be subject to income and/or revenue taxes as shown in Table 22-3. No municipal taxes or other levies were considered.

The amount of income taxes likely to be payable over the duration of the Project is summarized in Table 22-4.

22.2.10 Closure Costs and Salvage Value

Closure costs of $88.7 M have been included in the model.

22.2.11 Financing

The Base Case economic analysis is based on 100% equity financing.

22.2.12 Inflation

The Base Case economic analysis included no inflation. Capital and operating costs are expressed in fourth-quarter 2010 Canadian dollars.

Table 22-3: Tax Assumptions

Tax Rate	Percentage
Federal tax rate	15
Provincial tax rate	10
BC mining tax rate	2

Table 22-4: Taxation Payable over the Life of Mine

Tax Rate	Amount Payable ($ millions)
Federal tax	993
Provincial tax	662
BC mining tax	712
Total	2,370

Note: Numbers may not sum due to rounding.

22.3 Financial Results

Financial analysis of the Base Case (discount rate of 7%) showed the after tax Project NPV to be $137.3 M and the internal rate of return (IRR) to be 7.4%. The cumulative, undiscounted, after-tax cash flow value for the Project is $5,120 M and the after-tax payback period is 7.8 years.

Table 22-5 shows a summary of the financial evaluation. Figure 22-1 shows the projected after-tax net cash flow for the Project. Table 22-6 presents the cash flow for the Project on an annualized basis.

C1 cash costs, as defined by Brook Hunt are displayed in Table 22-7. The life-of-mine cash cost per pound of payable copper is $0.79 after secondary metal credits.

22.4 Sensitivity Analysis

Sensitivity analysis was performed on the Base Case, taking into account variations in metal prices, exchange rates, operating costs, and capital costs. Metal recoveries and metal grades mirror the metal prices. The results are shown graphically for NPV in Figures 22-2 and 22-3 and for IRR in Figures 22-4 and 22-5. The results from the analysis showed that the Project sensitivity was (in order from highest to lowest) metal price, exchange rate, operating expenditure, and capital expenditure.

Table 22-8 shows the post-tax IRRs and NPVs for a range of metal prices and exchange rates.

Table 22-5: Cashflow Summary Table

Summary of Financial Results	Units	Life-of-Mine
Copper payable	klb	5,950,000
Gold payable	koz	3,850
Silver payable	koz	56,100
Total cash costs	$/lb	1.83
Secondary metal credit	$/lb	(1.04)
Cash costs net of credits (C1 Net Direct Cash Cost)	$/lb	0.79
Cumulative net after-tax cash flow	$M	5,120
After-tax internal rate of return	%	7.4%
After-tax net present value @ 7%	$M	137
Mine life (including one year of pre-production	Years	18.5
After-tax payback period	Years	7.8
Total start-up capital	$M	5,160
Total LOM capital (inc. $88.7 M closure cost)	$M	5,840

Figure 22-1: After-Tax Net Cash Flow (Undiscounted)

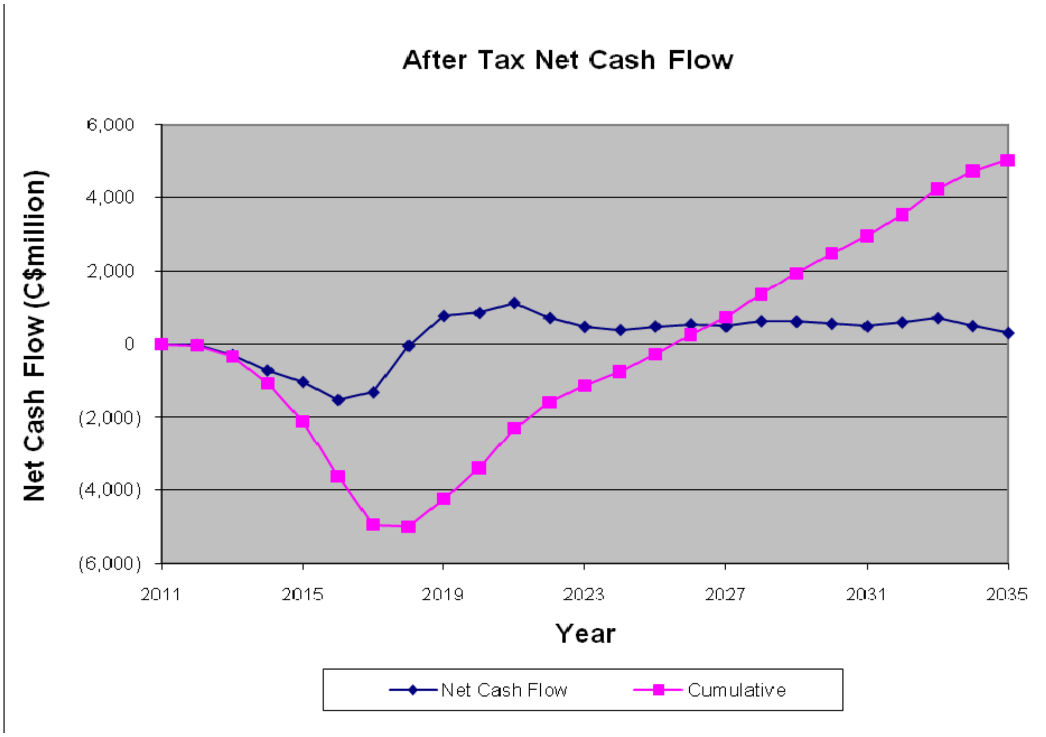


Table 22-6: Cashflow Analysis

Project year			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26
Production year			-8	-7	-6	-5	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Metal prices																												
Copper	US$/lb		2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65	2.65
Gold	US$/oz		1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100
Silver	US$/oz		18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50	18.50
Exchange rate																												
US$/CAD$	US$/CAD		0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90
		Life of Mine																										
Recovered metal value																												
Copper	CAD000	18,162,041								205,758	872,424	1,188,635	1,595,123	1,229,665	972,052	841,358	851,906	974,423	958,877	995,330	964,635	1,019,016	1,035,378	1,197,191	1,326,020	996,756	699,552	237,942
Gold	CAD000	4,863,173								186,561	655,798	357,802	245,086	136,455	103,114	114,322	198,121	307,781	376,215	386,495	352,002	313,930	240,145	249,630	266,621	186,204	127,676	59,216
Silver	CAD000	1,351,798								8,188	41,472	81,285	117,458	99,890	88,711	80,598	73,746	77,554	64,648	65,145	67,657	71,255	69,995	83,453	95,867	82,046	62,409	20,422
Total	CAD000	24,377,012								400,507	1,569,693	1,627,721	1,957,666	1,466,009	1,163,877	1,036,278	1,123,773	1,359,759	1,399,740	1,446,970	1,384,294	1,404,201	1,345,518	1,530,274	1,688,508	1,265,006	889,637	317,579
Smelter deductions																												
Copper	CAD000	(648,644)								(7,348)	(31,158)	(42,451)	(56,969)	(43,917)	(34,716)	(30,049)	(30,425)	(34,801)	(34,246)	(35,548)	(34,451)	(36,393)	(36,978)	(42,757)	(47,358)	(35,598)	(24,984)	(8,498)
Gold	CAD000	(159,484)								(4,664)	(16,395)	(10,734)	(12,254)	(6,823)	(8,249)	(5,716)	(5,944)	(9,233)	(9,405)	(9,662)	(8,800)	(9,418)	(7,204)	(9,985)	(10,665)	(7,448)	(5,107)	(1,776)
Silver	CAD000	(198,112)								(2,244)	(9,516)	(12,966)	(17,400)	(13,413)	(10,603)	(9,178)	(9,293)	(10,629)	(10,459)	(10,857)	(10,522)	(11,115)	(11,294)	(13,059)	(14,464)	(10,873)	(7,631)	(2,595)
Total	CAD000	(1,006,240)								(14,257)	(57,069)	(66,151)	(86,623)	(64,153)	(53,568)	(44,942)	(45,661)	(54,663)	(54,110)	(56,067)	(53,774)	(56,927)	(55,476)	(65,801)	(72,487)	(53,919)	(37,722)	(12,870)
Price participation																												
Copper	CAD000																											
Treatment charge																												
Concentrate	CAD000	(777,185)								(8,805)	(37,333)	(50,864)	(68,258)	(52,619)	(41,596)	(36,003)	(36,455)	(41,697)	(41,032)	(42,592)	(41,278)	(43,605)	(44,306)	(51,230)	(56,743)	(42,653)	(29,935)	(10,182)
Refining charges																												
Copper	CAD000	(462,618)								(5,241)	(22,222)	(30,277)	(40,630)	(31,322)	(24,760)	(21,431)	(21,699)	(24,820)	(24,424)	(25,353)	(24,571)	(25,956)	(26,373)	(30,494)	(33,776)	(25,389)	(17,819)	(6,061)
Gold	CAD000	(25,656)								(992)	(3,488)	(1,893)	(1,270)	(707)	(517)	(592)	(1,048)	(1,628)	(2,001)	(2,055)	(1,872)	(1,661)	(1,271)	(1,307)	(1,396)	(975)	(669)	(313)
Silver	CAD000	(24,945)								(129)	(691)	(1,477)	(2,163)	(1,870)	(1,689)	(1,544)	(1,394)	(1,447)	(1,172)	(1,174)	(1,235)	(1,300)	(1,269)	(1,522)	(1,760)	(1,539)	(1,184)	(385)
Total	CAD000	(513,219)								(6,362)	(26,401)	(33,647)	(44,064)	(33,898)	(26,966)	(23,567)	(24,141)	(27,896)	(27,597)	(28,582)	(27,678)	(28,917)	(28,913)	(33,324)	(36,932)	(27,903)	(19,672)	(6,760)
Penalty charges																												
Lead	CAD000																											
Concentrate transport																												
Land freight	CAD000																											
Port storage & handling	CAD000																											
Ocean freight	CAD000	(640,249)								(7,253)	(30,755)	(41,902)	(56,231)	(43,348)	(34,267)	(29,660)	(30,031)	(34,350)	(33,802)	(35,087)	(34,005)	(35,922)	(36,499)	(42,203)	(46,745)	(35,138)	(24,661)	(8,388)
Marketing & other	CAD000																											
Insurance charges	CAD000																											
Total	CAD000	(640,249)								(7,253)	(30,755)	(41,902)	(56,231)	(43,348)	(34,267)	(29,660)	(30,031)	(34,350)	(33,802)	(35,087)	(34,005)	(35,922)	(36,499)	(42,203)	(46,745)	(35,138)	(24,661)	(8,388)
Net smelter return																												
Net smelter return	CAD000	21,440,118								363,831	1,418,136	1,435,158	1,702,490	1,271,991	1,007,480	902,106	987,484	1,201,152	1,243,198	1,284,642	1,227,559	1,238,829	1,180,325	1,337,716	1,475,601	1,105,393	777,648	279,380
Net smelter return	USD000	19,296,106								327,448	1,276,322	1,291,642	1,532,241	1,144,792	906,732	811,895	888,735	1,081,037	1,118,879	1,156,177	1,104,803	1,114,946	1,062,292	1,203,945	1,328,041	994,854	699,883	251,442
Cumulative NSR	USD000									327,448	1,603,770	2,895,412	4,427,653	5,572,445	6,479,177	7,291,072	8,179,807	9,260,844	10,379,723	11,535,901	12,640,704	13,755,649	14,817,941	16,021,886	17,349,927	18,344,781	19,044,664	19,296,106
Tahltan royalty																												
Minimum payment	CAD000									1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000
CNSR threshold A	USD000												4,043	6,360														
CNSR threshold B	USD000														9,769	9,021	9,875	12,012	12,432	12,846	12,276	12,388	11,803	13,377	14,756	11,054	7,776	2,794
Applied royalty	CAD000	(165,581)								(1,000)	(1,000)	(1,000)	(4,043)	(6,360)	(9,769)	(9,021)	(9,875)	(12,012)	(12,432)	(12,846)	(12,276)	(12,388)	(11,803)	(13,377)	(14,756)	(11,054)	(7,776)	(2,794)
Production costs																												
Mining	CAD000	(3,536,192)								(53,997)	(221,200)	(237,066)	(239,302)	(248,849)	(264,017)	(260,549)	(241,184)	(226,521)	(223,821)	(176,443)	(160,336)	(174,338)	(185,859)	(182,986)	(151,536)	(125,021)	(112,376)	(50,790)
Process	CAD000	(3,041,102)								(39,960)	(168,036)	(192,596)	(190,348)	(180,094)	(167,492)	(166,182)	(175,257)	(180,835)	(178,844)	(166,334)	(163,564)	(164,290)	(167,321)	(172,420)	(172,151)	(165,856)	(150,812)	(78,709)
Port	CAD000	(82,158)								(1,166)	(4,662)	(4,662)	(4,662)	(4,662)	(4,662)	(4,662)	(4,662)	(4,662)	(4,662)	(4,662)	(4,662)	(4,662)	(4,662)	(4,662)	(4,662)	(4,662)	(4,222)	(2,175)
Corporate	CAD000	(824,510)								(11,957)	(47,868)	(47,965)	(48,009)	(48,240)	(47,933)	(47,705)	(47,843)	(47,705)	(46,317)	(45,377)	(45,078)	(45,641)	(46,083)	(46,271)	(45,670)	(45,101)	(41,663)	(22,084)
Infrastructure	CAD000	(85,284)								(1,208)	(4,834)	(4,834)	(4,834)	(4,834)	(4,834)	(4,834)	(4,834)	(4,834)	(4,834)	(4,834)	(4,834)	(4,834)	(4,834)	(4,834)	(4,834)	(4,834)	(4,403)	(2,334)
G&A	CAD000	(385,401)								(6,096)	(26,194)	(29,721)	(23,222)	(20,522)	(19,647)	(19,540)	(18,552)	(22,333)	(21,607)	(22,538)	(20,461)	(20,630)	(20,492)	(19,701)	(21,054)	(23,453)	(19,371)	(10,268)
Total	CAD000	(7,954,647)								(114,384)	(472,794)	(516,844)	(510,378)	(507,200)	(508,585)	(503,473)	(492,332)	(486,891)	(480,085)	(420,187)	(398,936)	(414,394)	(429,251)	(430,874)	(399,908)	(368,927)	(332,846)	(166,359)
Operating cash flow		13,319,890								248,447	944,342	917,314	1,188,070	758,430	489,126	389,612	485,277	702,250	750,682	851,608	816,347	812,046	739,271	893,465	1,060,937	725,413	437,026	110,227
Capital expenditure																												
Construction capital	CAD000	(5,155,261)	(21,000)	(24,000)	(428,708)	(710,970)	(1,011,959)	(1,491,511)	(1,273,886)	(193,227)																		
Sustaining capital	CAD000	(551,682)								(20,362)	(72,494)	(47,226)	(51,607)	(43,822)	(19,500)	(6,000)	(6,000)	(31,000)	(93,490)	(43,959)	(19,880)	(46,362)	(16,165)	(8,567)	(7,000)	(7,000)	(6,750)	(4,500)
Working capital	CAD000		[redacted]							(30,409)	(95,478)	(13,799)	(1,966)	4,015	1,924	2,430	2,692	281	1,838	14,653	5,583	(4,344)	(3,858)	(1,832)	6,606	10,647	11,639	89,377
Reclamation & closure		(88,674)																										(88,674)
Total	CAD000	(5,795,617)	(21,000)	(24,000)	(428,708)	(710,970)	(1,011,959)	(1,491,511)	(1,273,886)	(243,998)	(167,971)	(61,025)	(53,573)	(39,807)	(17,576)	(3,570)	(3,308)	(30,719)	(91,651)	(29,306)	(14,296)	(50,706)	(20,024)	(10,398)	(394)	3,647	4,889	(3,798)
Pre-tax net cash flow	CAD000	7,524,273	(21,000)	(24,000)	(428,708)	(710,970)	(1,011,959)	(1,491,511)	(1,273,886)	4,449	776,370	856,289	1,134,497	718,624	471,551	386,042	481,969	671,531	659,031	822,302	802,051	761,340	719,247	883,067	1,060,544	729,060	441,915	106,429
Taxation																												
BC mining taxes		(716,087)								(4,989)	(18,907)	(18,366)	(23,842)	(15,296)	(9,978)	(7,973)	(9,903)	(14,285)	(15,262)	(17,289)	(16,572)	(39,916)	(95,538)	(116,776)	(138,930)	(94,831)	(56,947)	(487)
Federal income tax		(998,216)																(78,663)	(101,871)	(116,239)	(112,088)	(108,669)	(90,024)	(111,138)	(133,985)	(91,088)	(54,129)	(320)
Provincial income tax		(665,477)																(52,442)	(67,914)	(77,493)	(74,725)	(72,446)	(60,016)	(74,092)	(89,324)	(60,725)	(36,086)	(213)
Total		(2,379,781)								(4,989)	(18,907)	(18,366)	(23,842)	(15,296)	(9,978)	(7,973)	(9,903)	(145,391)	(185,048)	(211,022)	(203,386)	(221,031)	(245,579)	(302,006)	(362,239)	(246,644)	(147,163)	(1,020)
Affter tax net cash flow	CAD000	5,144,492	(21,000)	(24,000)	(428,708)	(710,970)	(1,011,959)	(1,491,511)	(1,273,886)	(540)	757,463	837,922	1,110,655	703,328	461,573	378,069	472,066	526,140	473,983	611,280	598,665	540,310	473,668	581,061	698,305	482,416	294,753	105,409

Table 22-7: Summary of Cash Costs

Cash Costs	Unit	LOM Total	Cost per tonne milled (C$/t)	Cost per pound Cu payable (C$/lb)
Cash costs				
Mining	C$000	3,536,192	6.70	0.59
Process	C$000	3,041,102	5.76	0.51
Port	C$000	82,158	0.16	0.01
Corporate	C$000	824,510	1.56	0.14
Infrastructure	C$000	85,284	0.16	0.01
G&A	C$000	385,401	0.73	0.06
Smelting costs	C$000	2,296,645	4.35	0.39
Concentrate transport	C$000	640,249	1.21	0.11
Sub-total	C$000	10,891,541	20.63	1.83
Credits				
Gold	C$000	(4,863,173)	(9.21)	(0.82)
Silver	C$000	(1,351,798)	(2.56)	(0.23)
Sub-total	C$000	(6,214,971)	(11.77)	(1.04)
Adjusted cash costs				
Total	C$000	4,676,569	8.86	0.79

Figure 22-2: Sensitivity of After-Tax NPV Discounted at 7%

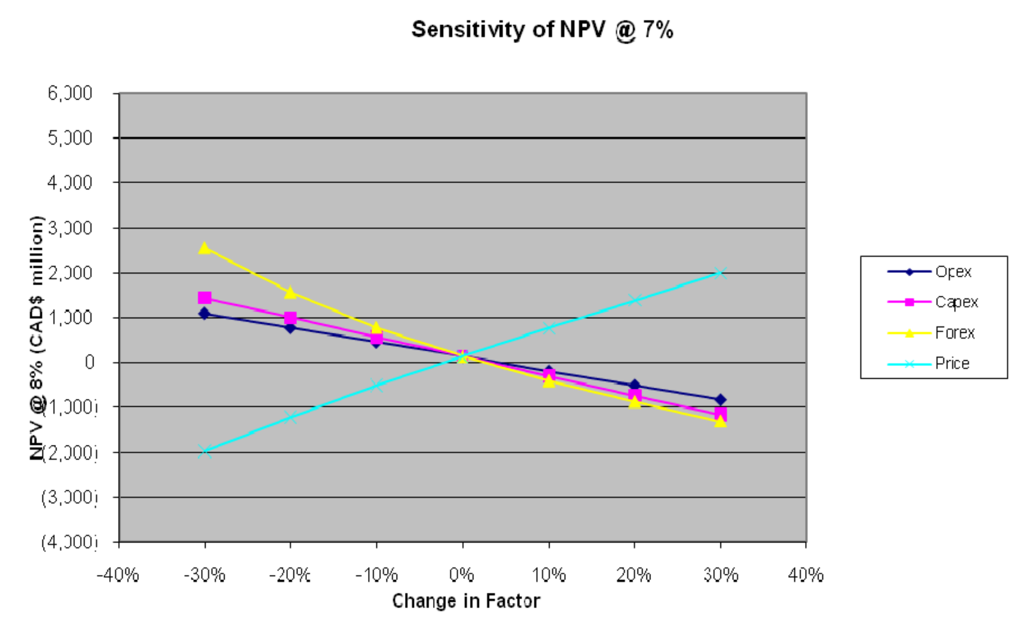


Figure 22-3: Sensitivity of After-Tax NPV Discounted at 7%

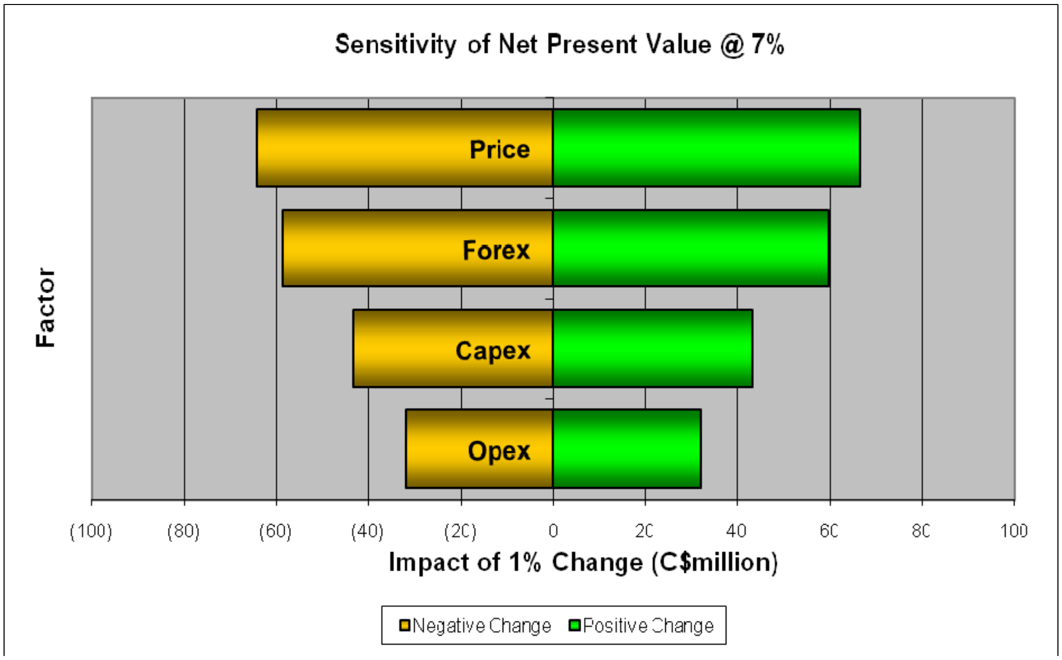


Figure 22-4: Sensitivity of After-Tax IRR

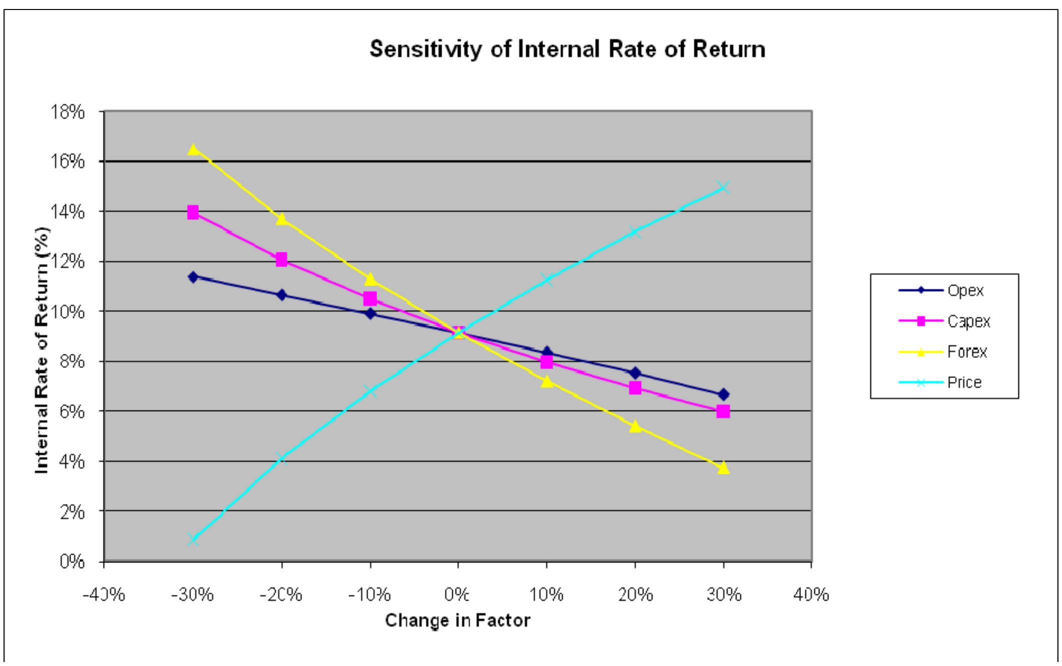


Figure 22-5: Sensitivity of After-Tax IRR

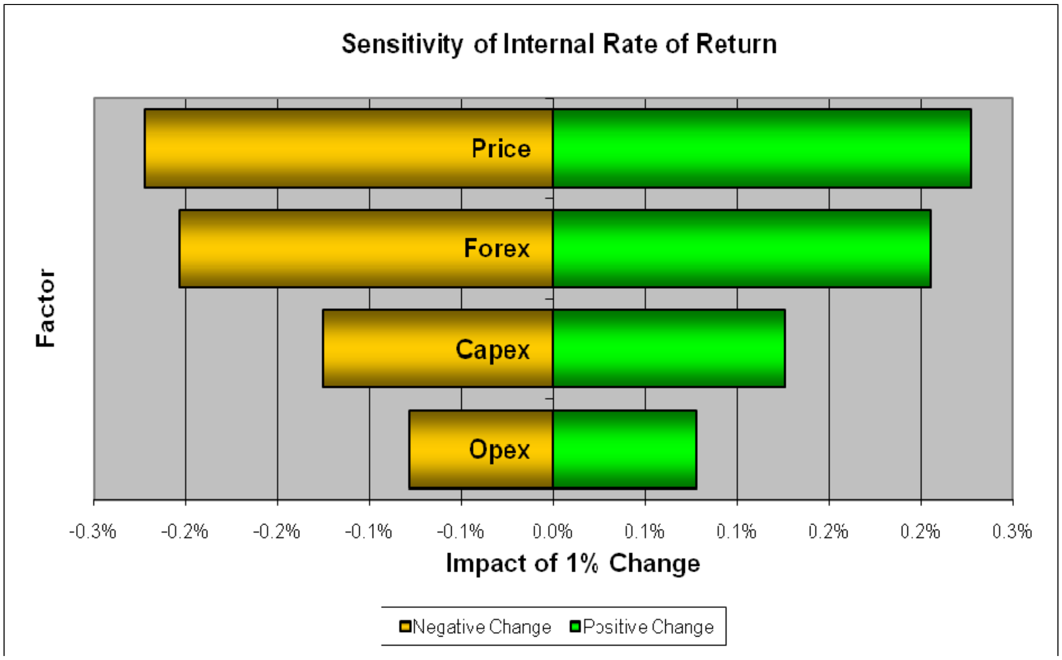


Table 22-8: Sensitivity to Metal Price Changes and Exchange Rates (Base Case is highlighted)

Metal	Base Case	Case 2	Case 3	Case 4	Case 5	Case 6
Copper (US$/lb)	**2.65**	**2.00**	**3.00**	**3.50**	**4.00**	**4.44**
Gold (US$/oz)	**1,100**	**900**	**1,100**	**1,200**	**1,300**	**1,613**
Silver (US$/oz)	**18.50**	**15.00**	**20.00**	**25.00**	**30.00**	**40.34**
Exchange Rate (US$/C$)	**Internal Rate of Return (%)**					
0.80	9.4%	4.6%	11.3%	14.0%	16.4%	19.0%
0.85	8.4%	3.5%	10.2%	12.9%	15.3%	17.9%
0.90	**7.4%**	2.4%	9.2%	11.9%	14.3%	16.8%
0.94	6.7%	1.6%	8.5%	11.1%	13.5%	16.1%
1.00	5.6%	0.5%	7.4%	10.1%	12.4%	14.9%
1.05	4.8%	N/A	6.5%	9.2%	11.5%	14.0%
Exchange Rate (US$/C$)	**Cumulative Net Cash Flow ($million)**					
0.80	6,815	2,996	8,543	11,460	14,382	17,918
0.85	5,914	2,212	7,534	10,274	13,021	16,348
0.90	**5,118**	1,514	6,641	9,223	11,812	14,952
0.94	4,542	1,004	5,997	8,463	10,938	13,942
1.00	3,771	304	5,131	7,439	9,761	12,580
1.05	3,152	(279)	4,439	6,620	8,818	11,488
Exchange Rate (US$/C$)	**Net Present Value 5% ($million)**					
0.80	1,895	(175)	2,789	4,284	5,765	7,567
0.85	1,416	(593)	2,263	3,675	5,073	6,769
0.90	**988**	(969)	1,794	3,134	4,458	6,060
0.94	676	(1,246)	1,453	2,741	4,010	5,547
1.00	250	(1,628)	988	2,208	3,405	4,854
1.05	(98)	(1,944)	612	1,778	2,920	4,297
Exchange Rate (US$/C$)	**Net Present Value 7% ($million)**					
0.80	862	(798)	1,567	2,743	3,902	5,315
0.85	480	(1,131)	1,151	2,263	3,360	4,690
0.90	**137**	(1,431)	778	1,837	2,877	4,134
0.94	(114)	(1,653)	507	1,528	2,525	3,732
1.00	(457)	(1,958)	135	1,105	2,049	3,189
1.05	(738)	(2,210)	(167)	764	1,667	2,753
Exchange Rate	**Net Present Value 10% ($million)**					
0.80	**(153)**	(1,370)	353	1,194	2,017	3,023
0.85	**(432)**	(1,613)	53	851	1,631	2,579
0.90	**(683)**	(1,831)	(217)	545	1,288	2,184
0.94	**(867)**	(1,993)	(415)	323	1,036	1,898
1.00	**(1,121)**	(2,215)	(687)	18	695	1,512
1.05	**(1,329)**	(2,398)	(909)	(230)	421	1,201
Exchange Rate (US$/C$)	**Payback Period (Years)**					
0.80	5.9	10.5	4.4	3.4	3.0	2.6
0.85	6.9	11.8	5.2	3.8	3.1	2.7
0.90	**7.8**	13.2	6.1	4.1	3.3	2.9
0.94	8.4	14.3	6.8	4.5	3.6	3.0
1.00	9.5	15.8	7.8	5.2	4.0	3.2
1.05	10.4	42.3	8.6	6.1	4.3	3.4

22.5 Real Option Sensitivity Case

In the conjunction with the preparation of the Galore Creek NI43-101 Report, NovaGold supplemented the AMEC Base Case analysis of Project NPV (AMEC DCF) by retaining Ernst & Young LLP (Ernst & Young) to develop an alternative evaluation model calculating a Real Option (RO) NPV for the Project. In this context, Ernst &

Young has delivered a report to NovaGold for NovaGold's private and confidential use. The report sets out Ernst & Young's scope of work, assumption, methodology and analysis, conclusions, and restrictions and limiting conditions. The summary contained in this report is also subject to the assumptions, limitations and qualifications set out in the Ernst & Young report. AMEC has prepared this summary to provide an overview of Ernst & Young's mandate and conclusions after having reviewed and considered the Ernst & Young report in its entirety.

The objective of the Ernst & Young analysis was to investigate whether the use of stochastic Monte Carlo simulation to model the effects of metal price uncertainty and the application of the RO method could provide additional insights into characteristics of the Project that influence long-term cash flow uncertainty and Project NPV. The analysis considered the difference between the RO NPV and DCF NPV methods and how well each method recognizes the unique cash flow uncertainty characteristics of the project within their respective risk-adjustment process.

The following should be noted regarding the Ernst & Young evaluation model and analysis:

- Ernst & Young has relied upon the completeness, accuracy and fair presentation of all financial and other information obtained from public sources and information from NovaGold and its consultants and advisors for purposes of developing the analysis. Ernst & Young did not and has not attempted to verify for the completeness or accuracy of the information. As such, the analysis is conditional upon the completeness and accuracy of such information.
- The RO method is a NPV calculation method that is being investigated by some participants in the natural resource industries. It is not asserted that either the RO method or the Dynamic DCF method provides more insightful evaluation results than the other.
- The Ernst & Young analysis does not consider the possible effects that management flexibility may have on the RO analysis and its possible impact on project NPV.
- The RO NPV and Dynamic DCF NPV calculations provided in this section cannot be considered fair market value or fair value estimates under the guidelines of the Canadian Institute of Chartered Business Valuators as there may be other factors and risks that should be considered and other analyses to be performed when estimating such values.
- The Ernst & Young evaluation model uses input data describing conditions in the overall market environment and metal price behaviour. The market environment is described by the inflation rate, risk-free rate, and a measure of

investor risk aversion. Modelling of metal price behaviour reflects the long-price assumptions, short-term metal price uncertainty, strength of reversion to a long-term equilibrium level in the case of copper, and the correlation between metal price movements and general financial market movements.

AMEC reviewed the work completed by Ernst & Young and has summarized the information to present a sensitivity case for the economic analysis. AMEC highlights that the AMEC DCF NPV and Ernst & Young RO NPV have been calculated with different cash flows models and risk adjustment methods so that a comparison of the two results requires an understanding of both NPV calculation methods.

22.5.1 Comparison of RO and DCF NPV Methods

A key step in the calculation of NPV is adjusting or discounting the cash flows for the value effects of project uncertainty and time. Under the DCF approach, this is done by incorporating provisions for these effects into a single aggregate risk-adjusted (or risk impounding) discount rate. The choice of discount rate may be based on a financial market model of asset returns such as the Capital Asset Pricing Model or an estimate of the Weighted Average Cost of Capital of the investor.

The RO method is an alternative method of calculating project NPV. The primary difference between the RO NPV and the DCF NPV methods is how the uncertainties in project cash flows are processed into the NPV estimate.

The RO method starts with the same quantitative cash flows as are considered in the DCF method. However, it augments that model by identifying and explicitly modelling the primary sources of cash-flow uncertainty, such as input and output prices. Measures of the estimated risk associated with these uncertainties are developed based on finance theory and capital market information to convert initial project net cash flows into risk-adjusted (reduced) cash flow amounts. In an analysis in which this process has dealt with all systemic uncertainties, the risk-adjusted cash flow would be present valued by discounting it for the time value of money only (i.e. at the risk-free rate). Where aspects of project uncertainty is not explicitly recognized using the RO method (as is the case with the Ernst & Young analysis), a residual project risk factor may also be applied to discount the risk-adjusted cash flow. The primary advantage of the RO method is its ability to adapt the pattern of risk adjustment across time in response to the annual variation of cash flow uncertainty over the life of the Project.

An important feature of the RO approach is its ability to recognize the annual variation in cash flow uncertainty characteristics that is produced by the complex interaction of reverting and non-reverting metal price uncertainty, variable costs and mill feed grades, and a detailed tax, royalty and financing structure. The RO method reflects this variation in the pattern of implied net cash flow risk adjustments.

When management flexibility or non-linear cash flow effects are an important value influence, the basic RO process can be combined with numerical techniques to estimate project NPV. These numerical techniques include lattice methods, finite difference techniques or flexible Monte Carlo simulation (the Ernst & Young analysis does not consider the possible effects that management flexibility).

22.5.2 Steps in Ernst & Young's RO DCF analysis

As first step in its analysis, Ernst & Young prepared a Dynamic DCF NPV evaluation model. The Dynamic DCF analysis used the same discounting methodology and real discount rates as the AMEC DCF model; however, it differed from the AMEC DCF model in that:

- The AMEC DCF model calculates NPV as at 1 January 2013, which aligns with the expected start date for Project development. The Project evaluation date selected by Ernst & Young is 01 January 2011.
- Ernst & Young used Stochastic Monte Carlo simulation to explicitly recognize copper and gold, price uncertainty and the effect of this uncertainty on project cash flows. Metal prices are simulated in constant monetary terms and then escalated into nominal terms.
- The Ernst & Young analysis uses the 01 January 2011 spot copper price as the initial price for the simulation after which these prices revert over time to the AMEC DCF model long-term forecast prices.
- The EY evaluation model calculates cash flow in nominal (inflation adjusted) terms with the production, operating cost, and capital cost schedules from the NovaGold cash flow models as its foundation. A nominal dollar analysis is adopted to eliminate tax calculation anomalies that arise in real dollar models. Once calculated, nominal dollar results are deflated to allow presentation of results in real dollar terms.

As a second step in its analysis, Ernst & Young applied the RO method to risk adjust (reduce) gold, copper and silver cash flows to recognise their specific risk and uncertainty characteristics. These risk-adjusted amounts were used to calculate risk-adjusted net Project cash flows, which were then discounted at a risk free rate plus a residual Project risk premium to determine RO NPV.

22.5.3 Metal Price Uncertainty Models

The Ernst & Young evaluation model uses input data describing conditions in the overall market environment and metal price behaviour. The market environment is described by the inflation rate, risk-free rate, and a measure of investor risk aversion. Metal price behaviour is modelled by a one-factor stochastic process which reflects the long- price assumptions, short-term metal price uncertainty, strength of reversion to a long-term equilibrium level, and the correlation between metal price movements and general financial market movements.

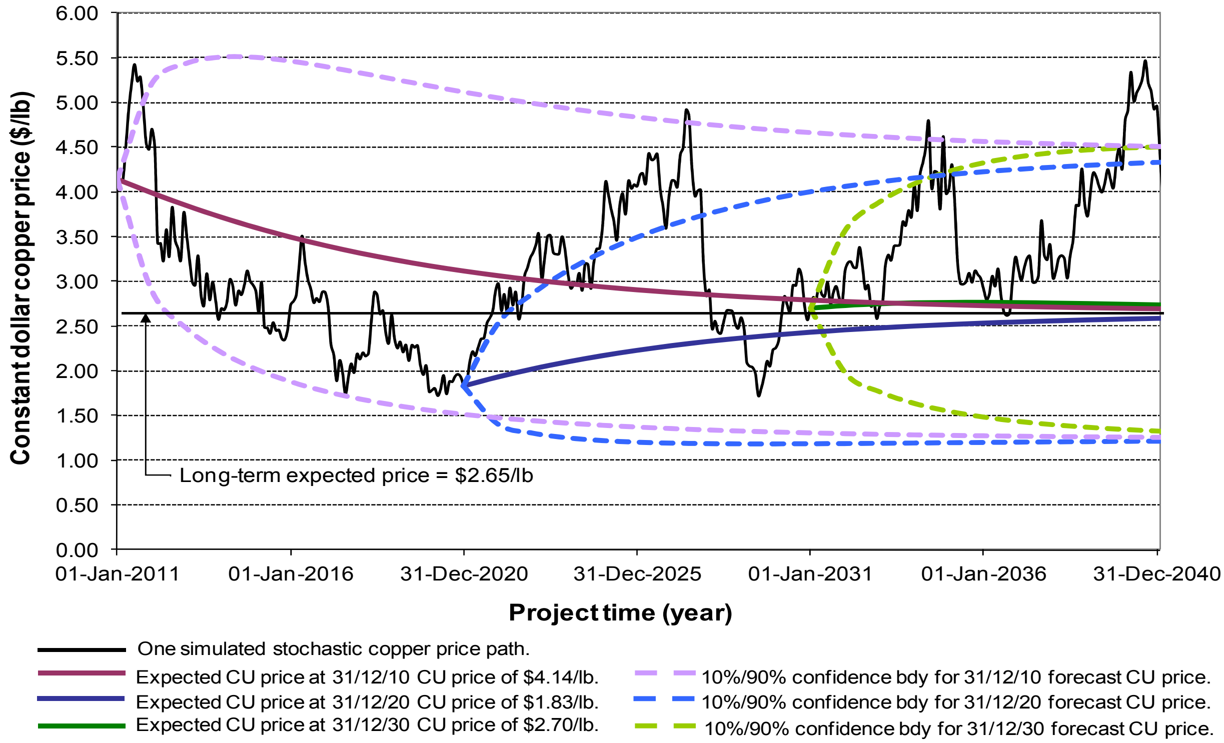
Model parameters were estimated from market data using econometric methods and real long term metal prices of $2.65 per pound for copper, $1,100 per ounce for gold and $18.50 per pound for silver (based on the long term price assumptions in the AMEC Model).

Figure 22-6 graphically summarizes the uncertainty characteristics of the copper price during the operating life of the Project. Uncertainty in the copper price is modelled as a one-factor lognormal stochastic process that exhibits price reversion. Reversion is the tendency for metal spot prices to fluctuate randomly around a long-term equilibrium level. When a metal exhibits reversion, prices that are much higher or lower than the long-term equilibrium price are hard to sustain as it is assumed that supply and demand economic forces move metal prices back into alignment. The econometric analysis obtained or prepared by Ernst & Young for copper price supports the assumption of price reversion.

The solid Purple line outlines initial price expectations at the start of the Project. The copper price is initially set at $4.14/lb, the copper spot price on January 1, 2011. Copper price expectations then revert back towards the long-term expected copper price of $2.65/lb. The dashed Purple lines in Figure 22-6 outline the 90% and 10% confidence boundaries between which 80% of metal prices are expected to fall at a particular future time at the start of Project development. The confidence boundaries for the copper price stabilize at $4.44 / $1.23 per pound in real terms. This is reflective of copper price reversion where copper price uncertainty initially grows with term but then stabilizes as economic supply / demand forces result in the tendency for copper prices to move back towards a long-term equilibrium level.

The copper price model also recognizes that market participants can update their price forecasts in response to new price information. This characteristic is illustrated in Figure 22-6 where a single price scenario is simulated for copper in constant monetary terms which is represented by the solid Black line. This is just one price scenario among many that is generated during the Monte Carlo simulation process.

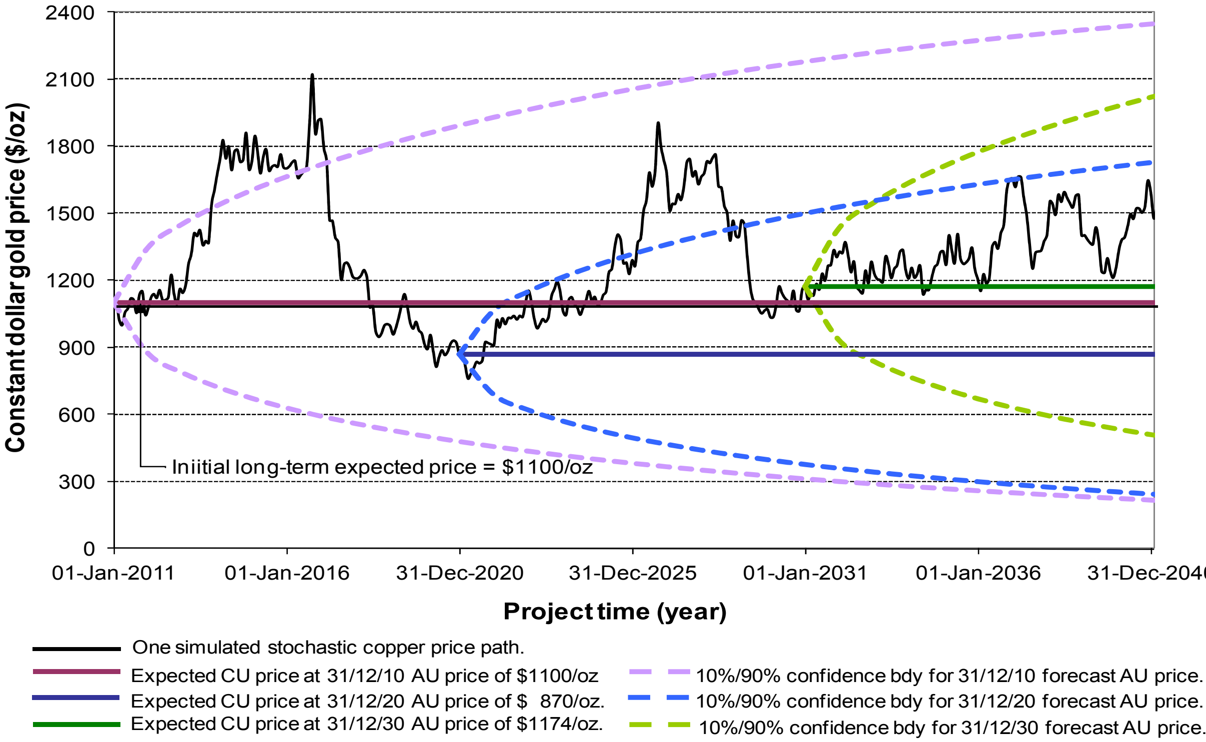
Figure 22-6: One simulated constant dollar copper price path with expectation and confidence boundary updating.



At select points in time, price expectations and associated confidence boundaries are re-drawn for the remaining project time based on the simulated spot price at that time. For example, the solid Blue line and its accompanying light Blue dashed lines delineate the revised metal price expectations and confidence boundaries at December 31, 2020 when the simulated spot price is $1.83/pound. In a similar manner, the solid Green and dashed light Green lines present revised price expectations and confidence boundaries based on simulated prices for copper at December 31, 2030. At both these price points, the copper price expectations still revert back towards the long-term equilibrium level of $2.65/lb. The long-term copper confidence boundary after 2040 is mostly unaffected by the price change because of price reversion.

Figure 22-7 graphically presents the characteristics of the gold price uncertainty model. Uncertainty in the gold price is modelled as a non-reverting one-factor stochastic process whereby the gold price moves in a manner similar to stock prices in the financial markets. Gold price confidence boundaries (dashed lines) are different in comparison to the copper price boundaries as they do not revert to an equilibrium level. These boundaries continue to move apart over most of the Project time horizon which reflects the lack of economic forces pulling the gold price back to an equilibrium level. Once again, the non-reverting characteristics of the gold price were confirmed by econometric analysis.

Figure 22-7: One simulated constant dollar gold price path with expectation and confidence boundary updating.



The solid Purple, Blue, and Green lines in Figure 22-7 have a similar interpretation as those in Figure 22-6. These solid lines delineate gold price expectations as of December 31, 2010 at a gold spot price of $1100/oz, December 31, 2020 at a gold spot price of $870/oz and December 31, 2030 at a gold price of $1174/oz. The associated dashed lines represent the confidence boundaries for each expectation.

The metal price uncertainty models depicted in Figures 22-6 and 22-7 are used in cash flow simulation for the Dynamic DCF NPV calculation. However, the RO NPV calculation requires uncertainty models that are risk-adjusted. The RO models have a similar structure as the metal price uncertainty models in the DCF calculation except that they have been subject to a risk-adjustment based on the Capital Asset Pricing Model and statistical analysis of financial market risk information. The difference between the expected metal prices in a DCF model and the risk-adjusted metal prices in a RO model represents the compensation an investor holding the refined metal requires for direct exposure to metal price uncertainty.

The RO risk adjustment applied to the gold price model has the same structure as standard discounting formula but the discount rate used in this formula is very small (0.02%) which reflects the low correlation between gold prices and financial market returns. Because the model in the Ernst & Young report is structured to reflect the constant dollar price assumptions adopted in the AMEC Model, the risk-adjusted gold price does not increase like the gold forward curve in the financial markets. The structure of forecast prices adopted in the Ernst & Young evaluation is not equivalent to deflating the gold forward curve for inflation. To adopt forward curve pricing, the forecast price would need to be inflated into nominal terms based on the gold forward curve contango structure and then deflated into real terms at the inflation rate.

The RO risk adjustment for copper price is more complicated in that it must recognize the effect of copper price reversion on long-term price uncertainty. Reversion causes copper price uncertainty to stabilize in the long-term and so causes the copper price risk adjustment to also stabilize. In this case, the copper risk adjustment causes a downward revision in long-term equilibrium price such that in the RO cash flow simulation the copper price varies around a long-term risk-adjusted expected price of US$2.21/lb.

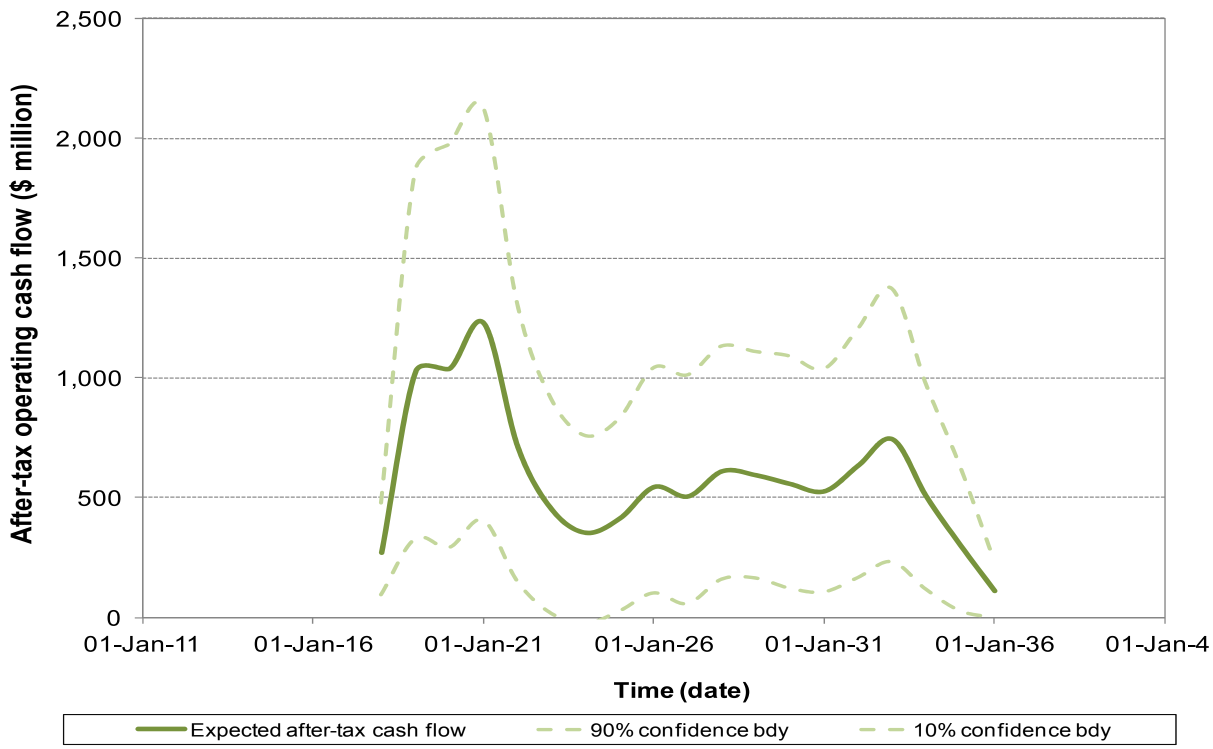
A non-reverting static stochastic price models is used in the RO and Dynamic DCF valuation models for silver. The price models for silver combine the AMEC Model's long-term price assumptions for these metals with a risk adjustment based on financial market information from the Bloomberg data service. The Dynamic DCF model estimates secondary metal revenues with price expectations based on the true price distribution while the RO model estimates secondary metal revenues with expectations from a risk-adjusted distribution.

22.5.4 After-tax operating cash flow uncertainty characteristics

The cash flow characteristics of the Galore Creek project do not change with the choice of NPV evaluation method. The following comments about expected cash flows hold regardless of evaluation approach since the analysis does not consider flexibility and since both RO and Dynamic DCF evaluation methods use the same input data and metal price models.

Expected real after-tax operating cash flows and associated 10% and 90% confidence boundaries for the Project are presented in Figure 22-8. These amounts are equal to Project revenues less direct and indirect mining and processing costs, smelting and refining charges, sustaining capital, government taxes and royalties and the Tahltan Royalty but do not include development capital expenditure or working capital. Average expected operating cash flow over the life of the Project is $611 million with a 90% confidence boundary of $1159 million and a 10% confidence boundary of $144 million. Expected operating cash flows of more than $1 billion in 2019 to 2021 are the result of higher initial metal grades and the impact of tax shields created by initial development capital.

Figure 22-8: One simulated constant dollar gold price path with expectation and confidence boundary updating.



22.5.5 RO Evaluation Results - Cumulative net cash flow and net present value

The assumptions supporting the Ernst & Young evaluation model are taken from the AMEC Model. Ernst & Young relies on the same cost information, long-term metal price forecasts, production schedules, DCF discount rate, and calculations for tax, royalty, and working capital as are reflected in the AMEC Model. However, in addition to differences in methodology and discount factors, the Ernst & Young evaluation model contains the following differences as compared to the AMEC Model:

- The AMEC Report calculates NPV as at 1 January 2013, which aligns with the expected start date for Project development. The Ernst & Young analysis is based on a copper price model that reconciles a copper spot price observable in financial markets on a particular day with a long-term forecast price level.

This price modelling approach cannot be used with a future evaluation date since there is no observable copper spot price. For these reasons, the Project evaluation date selected by Ernst & Young is January 1, 2011.

- Stochastic Monte Carlo simulation is used to explicitly recognize copper and gold, price uncertainty and the effect of this uncertainty on project cash flows. Metal prices are simulated in constant monetary terms and then escalated into nominal terms using a 2.45% escalation rate.
- The spot copper price in global markets for 1 January 2011 are used as the initial price for the simulation after which these prices revert over time to the AMEC Model long-term forecast prices. Initial gold and silver prices are the same as the AMEC Model.

The RO model generates risk-adjusted net cash flows using risk-adjusted stochastic price processes for copper and gold. Risk-adjusted net cash flows in the RO model are then discounted to determine a net cash flow RO present value using a nominal risk-free interest rate of 4.3% and a residual risk premium of 2.2%.

The residual risk premium is calculated based on the 5.1% project risk premium implied by the AMEC DCF discount rate. This premium is divided into price and non-price risk using an adjusted present value approach to generate the split in risk premium between price and non-price components. The residual risk premium used in the Ernst & Young evaluation model is ultimately determined by the risk premium implied by the AMEC DCF discount rate. Under this approach, the residual risk premium would be different if the AMEC Model assumed a different discount rate.

Table 22-9 presents cumulative net cash flow and the AMEC DCF and RO NPVs calculated for the Galore Creek project by the Ernst & Young evaluation model. Ernst and Young's RO NPV is $811 million, as compared to the $137 million AMEC Base Case NPV.

Table 22-9: AMEC DCF and Ernst & Young RO NPVs

	(C$ million)	
Evaluation Model	**EY RO NPV**	**AMEC DCF NPV**
Cumulative Net Cash Flow	**5,755**	**5,118**
Net Present Value	**811**	**137**

Viewed in the two steps, the Ernst & Young RO NPV reconciles to the AMEC DCF NPV as follows:

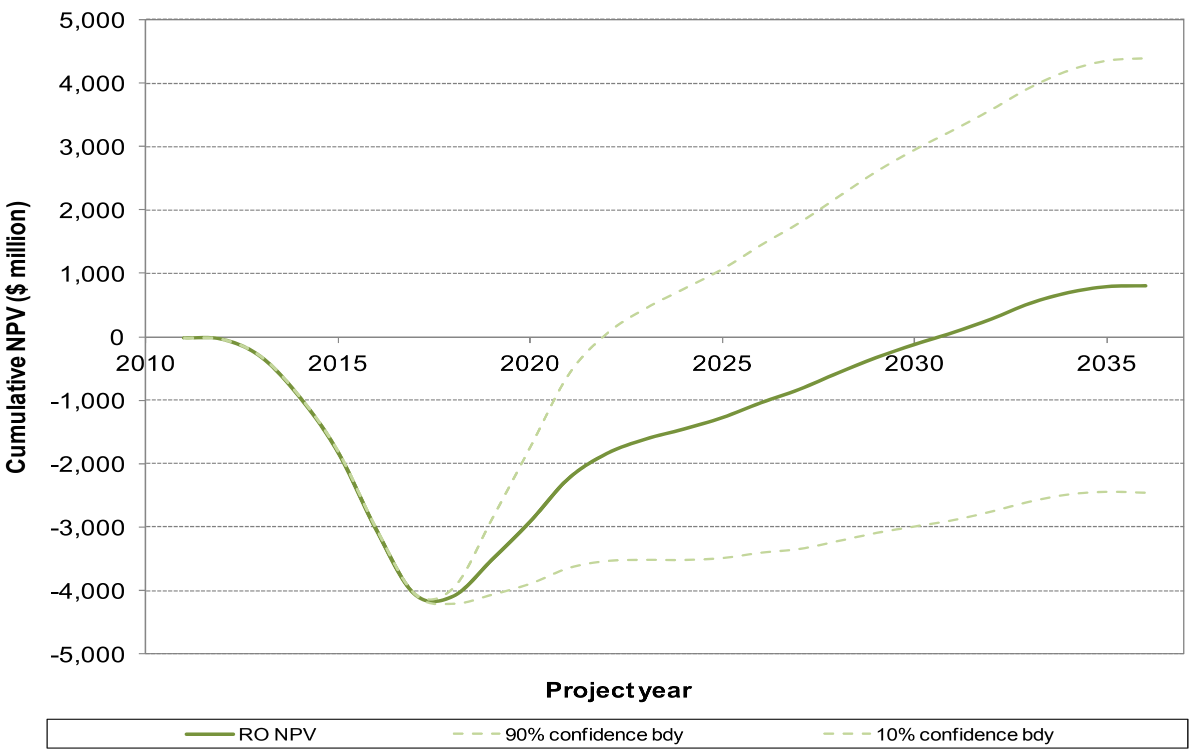
- The use of a reverting copper price model and stochastic simulation, nominal cash flow analysis, and an earlier discount date results in an NPV increase of $269 million over the AMEC DCF model to $406 million.

- Risk-adjusting project cash flows using the RO method by Ernst & Young results in Project NPV increasing by a further $405 million to $811 million.

It should be understood that the RO method will not always generate a higher NPV than the DCF method. The relative ranking of RO or DCF NPVs depends on the unique risk and uncertainty characteristics of the Project which may result in a RO NPV that is lower than a DCF NPV.

Figure 22-9 presents the build up of RO NPV over the life of the Project. The 90% and 10% confidence boundaries are also plotted to show the spread of possible RO NPV outcomes. These confidence boundaries only appear after 2017 as metal price uncertainty only begins to affect project cash flow once production starts in 2018. The RO NPV boundaries range from <u>negative</u> $2,453 million to positive $4,390 million.

Figure 22-9: Cumulative RO NPV and confidence boundaries for the Project.



The choice between the DCF NPV and RO NPV methods is a matter of professional preference and judgement. One factor to consider in making this choice is which NPV method recognizes the dynamic nature of cash flow uncertainty in its risk adjustments over the life of the Project. The Ernst & Young report investigated the relationship between project cash flow uncertainty and risk adjustments. The analysis demonstrated that the RO method adapted its pattern of risk adjustment to reflect annual variations in cash flow uncertainty due to the complex interaction of Project structure such as metal grade and costs and the characteristics of metal price uncertainty such as price reversion.

In contrast, the DCF risk adjustments derived from a single discount rate reflected a trend of increasing cash flow uncertainty and did not recognize the uneven increases and decreases in annual cash flow uncertainty that are likely to occur over the life of the Project.

22.6 Comment on Section 22

The financial analysis performed by AMEC for the Galore Creek Project, using a discount rate of 7%, indicates that the after tax Project NPV is $137.3 M and the IRR is 7.4%. The cumulative undiscounted after tax cash flow value for the Project is $5,117.8 M and the after tax payback period is 7.8 years.

The Project is most sensitive to changes in metal price, secondly to changes in exchange rate, less so to changes in capital cost and least sensitive to operating cost changes.

Completion of a RO evaluation for the Project using essentially the same production and long-term metal price forecast assumptions as the Base Case AMEC financial model indicates that there is potential for a higher after-tax Project NPV for the Project if the financial model employed is able to take into account the NPV impact of price reversion, low gold price correlation, and the annual variation of cash flow uncertainty.

23.0 ADJACENT PROPERTIES

There are no adjacent properties that are relevant to this Report.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Preliminary Development Schedule

A preliminary Project construction schedule was developed as part of the GCMC 2011 pre-feasibility study.

After EA approval and construction permits are issued, the work will start on the preparation of the north and south portals. Tunnel construction is projected to be completed over a 2–3 year period in the GCMC 2011 prefeasibility study schedule but, as discussed earlier, Lemley expects the tunnelling to require about 43 months from the time the tunnel contractor mobilizes until the first mine equipment can be transported through the tunnel. Upon holing through of the tunnel, the main sections of the primary crusher and large mining equipment will be transported into the valley. This will be followed by construction of a permanent roadbed in the tunnel and completion of the tunnel mechanical installations and connection of the conveying system through the tunnel to the processing plant.

The main pit pre-stripping activities will not commence until opening of the tunnel allows large mining equipment to access the proposed pit areas. However, water diversions, water sediment ponds, and preparation of haul roads and dumping areas will need to be initiated utilizing the equipment already in the valley and additional small equipment that can be airlifted as required.

Work in the tailings area will be seasonal starting in the spring and continuing through the summer. Work will shut down for the winter and recommence in the spring due to the temperature critical nature of dam building. This sequence will take three summer seasons to complete once the EA approval has been received.

Site preparation at West More will commence on receipt of the construction permit and will start with the clearing of the main benches associated with the concentrator and then move on to the coarse ore stockpile area and conveyor corridor to the south portal. Work on the West More overland conveyor and coarse ore stockpile/ reclaim area will not be carried out during the winter months.

Work will continue with the construction of the foundations for the process plant through the fall and winter to allow for steel erection/cladding to commence in the following spring/summer. The installation of equipment and electrical/instrumentation will then proceed in covered conditions.

Construction of the main electrical substation at West More, the substations at the south portal and the tailings area and the transmission lines connecting them will be installed after receipt of construction permits. This will allow for the use of permanent power for construction activities when BC Hydro connects to the Project power lines at Bob Quinn.

Permanent infrastructure (camps, fuel, water systems, warehousing, admin offices etc) will be scheduled to follow weather windows for concrete and steel with mechanical and electrical finishing installations taking place under cover.

Initially work will be monitored from offices set up in the tailings and north and south portal areas. Refurbished camp facilities in the Galore Creek Valley and the tailings area will be utilized to accommodate the initial workforce. The Hooh camp at the tailings area will be augmented to accommodate the bulk of the workforce.

Construction site offices will be constructed in the tailings area, at both portals, at the primary crusher site and near the concentrator. The office at the concentrator will be the main site office and hold the core group of staff. A first aid/safety/orientation complex will be erected close to the concentrator with a secondary first aid post in the Galore Creek Valley.

Construction of the load out facilities at Stewart will be monitored from a construction office at Stewart. A warehouse and laydown area will also be established in this area along with a camp to accommodate the construction workforce and staff. As work in this area is not scheduled to start until later in the construction sequence it is considered that there is adequate time after permit issue to establish the temporary facilities and construct the permanent facilities to meet the first concentrate milestone.

24.2 Project Opportunities

The GCMC 2011 pre-feasibility study identified opportunities to expand the mine life, improve the production profile, and reduce the capital requirements of the Galore Creek base case scenario. These opportunities are collectively termed the “Enhanced Plan” by GCMC.

24.2.1 Capital Cost Reduction Opportunities

A significant portion of the capital cost of the prefeasibility base case is estimated for the construction of a Galore Creek port facility at Stewart. Another major capital component is the concentrate transportation and filtration system, which includes a 70 km slurry and fuel pipeline, filter plant, water treatment plant, 50 person full-service camp, and fuel tank farm. Reconfiguration of this portion of the process could reduce the overall capital requirement, and GCMC has made plans to investigate the reconfiguration options.

Initial study indicated that the existing (third-party owned and operated) port facility in Stewart can be upgraded to meet the Project’s concentrate transportation requirements. This alternative approach to port facilities could reduce overall capital requirements. Under the Enhanced Plan, a capital estimate to complete facility upgrades would be included, as well as an estimate of unit-based concentrate terminal fees. This component of the Enhanced Plan would reduce the initial capital required, which would be partially off-set by an increase in operating costs (terminal charges).

The GCMC 2011 pre-feasibility study includes pipelines to transport concentrate and diesel fuel along the access road between the West More mill site and the Kilometre 8 site, located at highway 37. The Kilometre 8 site would have necessary support facilities, including a camp. Piping concentrate in slurry form requires a water treatment plant to treat filtrate prior to its release. Filtered concentrate would then be trucked from the Kilometre 8 site to Stewart via Highway 37.

The Enhanced Plan considers relocating the filter plant to a location adjacent to the mill at the West More site which would eliminate the need for a slurry pipeline and instead utilize truck transport for dry concentrate. Under this scenario, filtrate would be recycled to the processing plant, removing the requirement for a standalone water treatment plant. Filtered concentrate would be trucked from the West More mill site to Stewart, along the Galore Creek access road to Highway 37 and on to Stewart. Fuel would be trucked along the access road to the storage tanks at West More. The access road would need to be substantially upgraded from the current design in order to accommodate the increased truck traffic. The Kilometre 8 camp would not be required, and the number of beds at the West More camp would need to be increased to accommodate the additional personnel associated with concentrate filtration and haulage.

The relocation of the filter plant to the mill site could result in a net reduction in capital cost, partially offset by increased operating costs. The additional costs of trucking fuel and concentrate along the access road would be the primary drivers of the increase in operating costs.

24.2.2 Additional Mineral Resources

Under the GCMC 2011 pre-feasibility study approximately 528 Mt of Proven and Probable Mineral Reserves would be mined over the life of the Project. In the GCMC 2011 pre-feasibility study, the ore is processed through a single 95,000 t/d nominal SAG mill grinding circuit during the full life-of-mine. As the ore is expected to become harder as the mine life progresses, throughput is reduced in the latter years of the mine life due to grinding circuit limitations and result in an average life-of-mine milling rate of 84,000 t/d over the 17.6 year mine life (not including a year of pre-production).

Mine plan studies indicate that estimated Mineral Resources in the Bountiful area could become part of the mine plan using long-term price assumptions. The Mineral Resources lie approximately under the current exploration camp (refer to Figure 7-2), and were excluded from consideration in the GCMC 2011 pre-feasibility study due to perceived high strip ratio requirements. If the currently estimated Inferred Mineral Resources can be successfully converted to Measured and Indicated Mineral Resources so that they may then be used as part of an appropriately engineered Mineral Reserve pit, then there is potential for the Bountiful area to be added to the mine plan.

In-fill drilling and geotechnical drilling to support pit slope design in this area are currently on-going.

Additional mineralization that can be identified through step-out and infill drilling in the existing deposits and prospects also has significant potential to support estimation of additional Mineral Resources, and may support conversion to Mineral Reserves, and therefore consideration in future mine plans. Additional reported Mineral Resources occur at depth in the Central Zone and in four adjacent satellite deposits on the Project. Mineralization has also been identified on other prospects, including the Butte, North Rim, West Rim and Saddle areas, but there is currently insufficient drill data to support Mineral Resource estimates.

The Copper Canyon deposit, now owned by NovaGold, is located mid-way up the east fork of Galore Creek, approximately 2 km from the Central Zone. NovaGold has offered the Copper Canyon lands to Teck for inclusion in the Galore Creek partnership agreement. The mineralization at Copper Canyon, with additional drilling to upgrade the confidence categories on the existing Mineral Resource estimate, may also have potential to add to any future mine plan.

24.2.3 Waste Considerations

Should the Bountiful area be added to the mine plan, there will be an increase in the amount of waste rock that would need to be handled, and therefore some modification to the waste rock management plan proposed for the Galore Creek Valley would be required. This could involve relocating the access causeway and ore conveying systems would need to be relocated from the East Fork area. A substantial amount of additional PAG waste rock would also need to be re-handled during the mine closure period in order to submerge the additional PAG waste rock for reclamation.

A further consideration could be the use of the entire East Fork area for waste storage. In this instance, the tunnel would no longer discharge into the East Fork area, but would have to be extended past East Fork for approximately 4 km. Such a tunnel extension would increase the initial capital required for the Project.

24.2.4 Mill and Plant Considerations

Inclusion of the Bountiful area in any mine plan is likely to sufficiently lengthen the mine life that a mill expansion could be contemplated during the earlier years of operations. The mill expansion would require some additional initial capital in order to design the coarse ore stockpile to eventually feed two SAG mills, and would require additional sustaining capital to install a second SAG mill. The addition of a second SAG mill would increase the total site power requirements and annual operating costs, but may have the potential to reduce the overall unit operating costs.

25.0 INTERPRETATION AND CONCLUSIONS

The AMEC QPs, who reviewed the GCMC 2011 pre-feasibility study, have made the following interpretations and conclusions:

25.1 Agreements, Mineral Tenure, Surface Rights, and Royalties

- The Project is operated under a 50/50 partnership agreement between Teck and NovaGold. Teck agreed to fund 100% of all costs incurred by the Partnership from 1 November 2008 until the aggregate additional amount contributed by Teck equals $60.0 M. If any portion of the $60.0 M has not been contributed by 31 December 2012, Teck has agreed to contribute in cash any shortfall on that date to the partnership
- Information from legal and GCMC experts support that the mining tenure held is valid and is sufficient to support declaration of Mineral Resources and Mineral Reserves. Tenures have been surveyed in accordance with appropriate regulatory requirements. Annual claim-holding fees have been paid to the relevant regulatory authority
- Information from GCMC experts support that surface rights are held by the Crown
- Upon reaching certain agreed financial targets, and subject to positive mine operating cash flow, the Tahltan Heritage Trust Fund will receive the greater of $1 M or a 0.5% to 1.0% net smelter royalty each year. The agreement will remain in effect throughout the life of the Galore Creek Project and will be binding on any future operator of the mine. This NSR payment is incorporated into the Project financial analysis.

25.2 Geology and Mineralization

- The deposits of the Project area are considered to be examples of alkalic porphyry copper deposits
- Knowledge of the deposit settings, lithologies, and structural and alteration controls on mineralization, and the mineralization style and setting is sufficient to support Mineral Resource and Mineral Reserve estimation.

25.3 Exploration, Drilling, and Data Analysis

- The exploration programs completed to date are appropriate to the style of the deposits and prospects within the Project. The exploration and research work supports the orogenesis interpretations

- The quantity and quality of the lithological, geotechnical, and collar and down hole survey data collected in the exploration and delineation drill programs are sufficient to support Mineral Resource and Mineral Reserve estimation
- Sampling methods are acceptable, meet industry-standard practice, and are acceptable for Mineral Resource and Mineral Reserve estimation
- The quality of the gold, copper, and silver analytical data are sufficiently reliable to support Mineral Resource and Mineral Reserve estimation and sample preparation, analysis, and security are generally performed in accordance with exploration best practices and industry standards. The QA/QC programs adequately address issues of precision, accuracy and contamination. The copper and gold biases identified in legacy assay data are generally low and are expected to cause an overall underestimation of grade in the Mineral Resource estimate. The legacy silver assay bias could be quantified and corrections could be developed using regression analysis. Limitations on the use of legacy assays to support classification of Measured blocks are still warranted due to the lack of supporting quality control samples
- The data verification programs undertaken on the data collected from the Project adequately support the geological interpretations, the analytical and database quality, and therefore support the use of the data in Mineral Resource and Mineral Reserve estimation.

25.4 Metallurgical Testwork

- Metallurgical testwork and associated analytical procedures were performed by recognized testing facilities, and the tests performed were appropriate to the mineralization type
- Samples selected for testing were representative of the various types and styles of mineralization at Galore Creek. Samples were selected from a range of depths within the deposit. Sufficient samples were taken such that tests were performed on sufficient sample mass
- Samples used to generate the locked cycle test results in May 2011 are considered by AMEC to be unrepresentative of any category of ore type. The new results were not used to estimate metal recoveries
- Testwork has established the most appropriate grind size for plant design. An 80% passing feed size of 57 mm is planned for broken ore, whereas for stick ore, an 80% passing size of 150 mm is assumed.

- Assumed life-of-mine copper, gold, and silver recovery assumptions are based on appropriate testwork, and copper, gold and silver recoveries average 90.6%, 73.1% and 64.5% respectively over the life-of-mine
- Various elements have been mentioned throughout the testwork that might have the potential for concern for concentrate quality including selenium in 2003, fluorine, selenium, lead, and zinc in 2006, and zinc and cadmium in 2010. In 2006, four concentrate batches were subject to multi-element analysis. Within three batches, fluorine returned low-level values, and one sample had a high fluorine analysis, which may be an analytical error. The fluorine level in the three batches noted was only just at a typical penalty level, and well below the reject level. Fluorine has been included as a penalty element in the financial analysis considerations. There were no other elements noted in the sampling that would cause penalties to be levied against Galore Creek concentrates.

25.5 Mineral Resource and Mineral Reserve Estimation

- Estimations of Mineral Resources and Mineral Reserves for the Project conform to industry best practices, and meet the requirements of CIM (2010). An open pit extraction scenario is appropriate to the style of mineralization and LG shells have been used to constrain the estimates. Assumptions used in the shells are appropriate to the envisaged process route and mine plan
- Measured and Indicated Mineral Resources exclusive of Mineral Reserves of 286.7 Mt grading 0.33% Cu, 0.27 g/t Au and 3.64 g/t Ag and Inferred Mineral Resources of 346.6 Mt grading 0.42% Cu, 0.24 g/t Au and 4.28 g/t Ag. Mineral Resources are reported on an NSR cut-off of $10.08/t milled
- Factors which may affect the Mineral Resource estimate include commodity price and exchange rate assumptions, assumptions used to estimate metallurgical recoveries, pit slope angles, and SG values assumed for the broken rock
- Proven and Probable Mineral Reserves total 528 Mt grading 0.585% Cu, 0.32 g/t Au and 6.02 g/t Ag. Mineral Reserves are reported within mineable pit designs using metal prices for copper, gold and silver of US$2.50/lb, US$1,050/oz, and US$16.85/oz, respectively. The exchange rate used was $1.10 to US$1.00. Mineral Reserves have been calculated using a 'cashflow grade' ($NSR/SAG mill hr) cut-off which was varied from year to year to optimize NPV. SAG throughputs were modeled by correlation with alteration types. Cashflow grades were calculated as product function of NSR value in $/t and throughput in t/hr. The life of mine strip ratio is 2.16
- Factors which may affect the Mineral Reserve estimate include commodity price and exchange rate assumptions, variations in throughput rates, variations in the

planned tunnel construction schedule and budget, variations in the planned construction schedule, including pre-production mining, and assumptions in relation to water management requirements.

25.6 Mine Plan

- Mining will use a conventional truck-and-shovel fleet
- The three-phase Central pit is scheduled to produce ore throughout the mine life with the satellite Southwest, West Fork, and Junction pits supplementing ore production at various stages of the mine life
- The proposed open pit operations will provide process plant feed at a nominal rate of 95,000 t/d or 34.6 Mt/a
- The production schedule contains one year of pre-production and a mine life of 17.6 years (not including one year of pre-production)
- The bench height in all pits will be 15 m, with 40 m wide ramps and a designed minimum mining width of 100 m allowing for double-sided loading in the narrowest mining areas
- The mine and fleet design is appropriate for the Mineral Reserves defined. Mine design has included consideration of the high rainfall and snowfall conditions and expected geohazards in the Galore Creek Valley. Additional geohazard assessments will be required
- It is expected that any future mining operations will be able to be conducted year-round, and will include appropriate provisions for winter operating conditions
- Five waste rock storage facilities are planned. The two NPAG waste dumps will initially be situated out of the valley floor until PAG rock is mined then PAG wasted will be deposited in the valley bottom in three PAG waste facilities, which will be flooded at mine closure
- Water management of the Galore Creek watershed will be a major design challenge. A number of water control structures are planned, including diversion channels, and closure and sedimentation dams. Water control structures were designed by AMEC. Based on preliminary modelling by Lorax, water quality will be suitable for direct discharge with no requirement for water treatment.

25.7 Process Design

- The Project will use conventional mineral processing equipment to produce a marketable copper concentrate

- The process plant will be a conventional grinding–flotation concentrator with a pipeline transferring concentrate to a remote filter plant and concentrate truckloading facility located at Kilometre 8 near Highway 37. From the filter plant, the concentrate will be transported by truck to the Port of Stewart for shipment to various destinations. Plant design was undertaken by a third-party consultant to GCMC, who cannot be identified under terms of contract with GCMC
- Process tailings will be stored in the West More tailings facility, which will consist of three dams, a Main Dam and two saddle dams. The dams and impoundment will accommodate up to 678 Mt of tailings, although storage for only 510 Mt is required for the current mine plan.

25.8 Infrastructure Considerations

- The Project will require construction of significant infrastructure to support the planned producing facilities
- The existing and planned infrastructure, availability of staff, the existing power, water, and communications facilities, the methods whereby goods are transported to the mine, and any planned modifications or supporting studies are well-established, or the requirements to establish such, are well understood by GCMC, and can support the declaration of Mineral Resources and Mineral Reserves
- The Project is currently not accessible by road. The closest provincial road to the mine site is Highway 37. An access road is planned from this highway to the proposed mine site. A section of the access road from Highway 37 (Km 0) to approximately Km 40 was constructed during a previous Project phase and is currently in service
- The most complex and challenging construction phase will be related to the work required in the Galore Creek Valley. Only air support is available until tunnel break-through, creating a very difficult and expensive operation
- The supply of diesel to support early construction is critical to both the Galore Creek Valley and West More areas. The development of the 287 kV line from Bob Quinn and the 287 kV substation at West More are both activities that require early completion to help reduce the overall diesel consumption
- GCMC will construct a new 287 kV transmission line to supply the power demand at the proposed Galore Creek mine site. It is important to get power to the site as early as possible to reduce the fuel requirement supporting the diesel generators at the south portal. However, construction of the West More substation will require EA approval

- Fresh and potable water will be obtained from wells. Process water is planned to be obtained through reclaim from the tailings pond
- The proposed port site is the former Arrow Dock facility, a causeway made of reclaimed land to the southeast of Stewart. The port site will include a concentrate storage and shiploading system. Habitat compensation is considered a key environmental consideration for the development of the port.

25.9 Tunnel

- The 13.6 km access tunnel represents a major tunnel project in terms of international tunnelling practice. The proposed tunnel is aligned under high rock cover of more than 600 m over a significant portion (75%) and with a maximum rock cover of 1,250 m
- While the risks identified point to a very challenging tunnelling project, there is nothing inherent in these risks that has not been dealt with successfully on other projects, using both drill/blast and TBM methodologies, or which would cause Lemley to render an opinion that the Galore Creek tunnel is not constructible using the approach described by GCMC's third-party consultants
- A total of 37 months should be allocated for boring/excavating the tunnel and a 49-month overall tunnel construction duration is appropriate
- Driving the TBM reach at a slight uphill grade rather than the currently-envisaged negative grade would enhance the ability to deal with water inflows or hydrogen sulphide gases
- There are greater cost risks and schedule risks associated with the use of a TBM than there would be with two opposing drill and blast operations.

25.10 Markets and Contracts

- The concentrate will be of average copper grade, which will be acceptable to most smelters; if a concentrate on the upper end of the range projected can be achieved, the concentrate will be more attractive to smelters. It will be considered a gold-bearing concentrate and may be less attractive to those smelters which do not have efficient precious metal recovery
- Based on the currently-available, and limited, multi-element analyses of concentrates, the concentrate may contain one impurity, fluorine, and that will be at levels such that deductions will apply, but the expected levels are below rejection concentrations. There may be occasions when the level of fluorine could attract a penalty structure from a smelter. The presence of fluorine in the concentrates may limit the quantity any one smelter may wish to purchase

- The Project as envisaged in the GCMC 2011 pre-feasibility study is of medium size and therefore will not have a significant market impact
- The planned sales strategy for GCMC is to be to establish long-term contracts for approximately 75% of the minimum long-term production quantity from the Project.

25.11 Environmental, Social Issues and Permitting

- The Project received its original EA approval in February 2007. The Project's first permits were obtained in May 2007. In June 2007, GCMC received final federal approval. The new design and Project configuration is significantly different from what was permitted under the original EA Certificate. It is anticipated that a new EA process will be requested by BCEAO and CEAA. This is expected to take at least two years
- GCMC will need to apply for appropriate mining operations-related permits under Provincial and Federal laws to allow proposed mining operations. Exploration and development activities completed to date have been conducted under the relevant permits
- GCMC has informed AMEC that current environmental liabilities are covered with the value of bonds that have been lodged for the Project
- A preliminary closure and remediation plan has been prepared. The estimated total reclamation liability for the proposed Galore Creek mine, including the Bob Quinn facility, is $88.7 M at the end of the mine life. The estimate includes a contingency of 35%
- The Project is located in an area where consultation with First Nations groups is critical for Project support. An agreement is in force with the Tahltan Nation, in whose territory the mining and processing facilities are planned to be located. Additional First Nations groups that may be impacted by Project development include Skii Km Lax Ha Traditional Territory and Gitanyow Tradition Territory (usage of Highway 37), Nisga'a Nation Traditional Territory (port of Stewart), Metlakatla, Kitselas, Kitsumkalum, and Lax Kw'alaams First Nations Traditional Territories (passage of the Portland Canal), Gitxaala and Haida Nations Traditional Territories (shipping lanes).

25.12 Capital and Operating Cost Estimates

- The capital cost estimate for the GCMC 2011 pre-feasibility study was developed by GCMC and its third-party consultant, with input from consultants for specific areas. Estimates were based on a combination of quotes, vendor pricing, and experiences with similar-sized operations. Capital cost estimates in the GCMC

2011 pre-feasibility study were reported at a prefeasibility level where the estimate accuracy range is defined as +25%/-20% (including contingency) and are consistent with an AACE Class 4 estimate. GCMC considered that the pre-feasibility estimate had an 18% contingency at P85 with an expected accuracy range of -4% to +20%

- AMEC reviewed the capital costs and considered that the estimate for earthworks and tunnelling did not reflect likely actual costs, and additional capital expenditure allocations should be added to those areas. The addition of $140.94 M for extra earthworks and tunnelling increased the capital cost estimate to approximately $5,155 M
- The operating cost estimate for the Project was developed by GCMC and its third-party consultant. Operating costs were built from first principles where costs were expected to be material and included allowances or estimates for minor costs. The assumed power cost for the purposes of the GCMC 2011 pre-feasibility study were $50/MW/hr and the assumed diesel fuel cost was $1.04/L; however, not all quotes used those figures. Manpower requirements were based on industry experience with similar-scaled operations
- AMEC reviewed the operating costs, and restated costs. The estimated average annual operating cost is $418.7 M/a or $15.07/t milled.

25.13 Financial Analysis

- The financial analysis for the Project, using a discount rate of 7%, indicates that the after-tax Project NPV is $137.3 M and the IRR is 7.4%. The cumulative, undiscounted, after-tax cash flow value for the Project is $5,117.8 M and the payback period is 7.8 years
- The Project sensitivity was (in order from highest to lowest) metal price, exchange rate, operating expenditure, and finally capital expenditure.

25.14 Preliminary Development Schedule

- A preliminary Project development schedule was generated in the GCMC 2011 pre-feasibility study. The schedule includes consideration of early work requirements, the EA assessment process, and EPCM and construction activities. Critical items identified in the construction schedule are the tunnel, supply of power, and diesel usage. An efficient and well-executed construction strategy will be integral to the fiscal approval of the Project. A specific constraint on maintaining the schedule will be completion of activities in the Galore Creek Valley prior to tunnel completion

- The development schedule planned indicates that Project success will dependent in part on developing a practical and efficient logistics plan for the movement of manpower and materials to site during the construction and the subsequent operating phases of the Project.

25.15 Conclusions

AMEC considers that the scientific and technical information available on the Project can support proceeding with additional data collection, trade-off and engineering work and preparation of more detailed studies. However, the decision to proceed with a Feasibility Study on the Project is at the discretion of GCMC and the partners.

AMEC recommends that GCMC considers the recommendations in Section 26 as activities which may support Project advancement should the partners and GCMC determine that a Feasibility Study is warranted.

26.0 RECOMMENDATIONS

AMEC recommends that GCMC considers the recommendations in this section as activities which may support Project advancement should the partners and GCMC determine that a Feasibility Study is warranted. The Project is located in a remote area, with significant logistics considerations. These factors indicate that completion of any Feasibility Study will require significant expenditure.

As part of the recommended work program, the following areas of work should be considered: additional drilling, topographic surveys, geotechnical studies, engineering and metallurgical studies, land management, including applications for mining leases where appropriate, additional baseline studies, and environmental and permitting activities. Additional areas for work are also likely to be identified as activities progress.

AMEC's recommendations do not include provision for pre-construction and construction activities for site and access infrastructure such as the road and tunnel.

The program is envisaged as a two-phase program, with all elements of the first phase of the program to be conducted concurrently. The outcome of the work will be included in Phase 2, which will consist of completion of a Feasibility Study.

The Phase 1 activities include data collection, trade-off studies and investigations and studies and activities to support EA and public consultation processes. Some more specific recommendations for work focus have also been included for mineral resource estimation, tunnel design, and plant design purposes. The total cost of these activities is estimated to be between about $31 M and $39 M.

The Phase 2 activity comprises completion of a Feasibility Study, estimated at between about $11 M and $13 M, and including a contingency provision.

Total program costs are likely to range between approximately $42 M and $52 M.

AMEC notes that GCMC has already commenced some initial work, which includes geotechnical drilling for both the tunnel and the open pits, sample collection and re-assaying, discussions relating to port usage, and review of information and recommendations arising from the 2011 GCMC pre-feasibility study report.

26.1 Phase 1

The Phase 1 activities include data collection, trade-off studies and investigations and studies and activities to support EA and public consultation processes. The total cost of these activities is estimated to be between about $31 M and $39 M. The most significant cost contributors will be the drill programs and logistical support required for this work.

26.1.1 Land Management

- Land management activities to support Project advancement will be required and will include land surveys, and applications to convert existing tenure to mining leases as appropriate (estimated cost of $40,000 to $50,000).

26.1.2 Drill Programs

- Geotechnical drilling to support initial mine and tunnel design is currently underway. Additional geotechnical drilling is likely to be required for the sites selected for areas of major infrastructure. AMEC has assumed the program will comprise approximately 12,000 m of drilling, and including consultant support for programs (estimated cost of $5 M to $5.5 M)
- Infill drilling is planned in the Bountiful area, and other areas may be drilled as determined by resource estimators. The program is estimated to comprise approximately 10,000 m of drilling (estimated cost of $3.5 M to $4 M)
- It is recommended that at the next stage of development of the Project a metallurgical drilling and test program be undertaken to define the metallurgical response more accurately. The economic shell should be evenly represented spatially, the sample interval length should be consistent (preferably bench height, approximately 15 m) and tight control of laboratory procedure should be followed. The sample intervals should be logged geologically while in the core boxes. Additional metallurgical test holes are also likely to be required to support feasibility-level testwork. AMEC has allocated an additional 5,000 m of drilling and consultant time for the metallurgical drilling (estimated cost of $2 M to $2.5 M).

26.1.3 Mineral Resource Estimation

- There is risk associated with only using two drill holes to determine the SG reduction factor for the broken rock. It is recommended that additional data are collected to support the broken rock reduction factor (estimated cost of $10,000 to $20,000 depending on the number of samples collected. AMEC recommends that a minimum of 30 determinations is taken per rock type containing broken rock)
- There is a lack of QA/QC documentation regarding legacy drilling (pre-2003). There may be an opportunity to support potential upgrade of some mineralization that is currently classified as Indicated to Measured through select reassaying of legacy drill hole pulps, or potentially, skeleton core. Consideration should be made

during the programs to collecting soluble copper and fluorine assay data. At a minimum, AMEC recommends that 10% per deposit per legacy drill campaign is undertaken. Additional sampling may be required if the legacy data and the new assays show significant deviations (estimated cost of $500,000)

- Lithology, grade, and structural wireframes should be updated to include GCMC drilling and any pertinent information from the two theses recently completed on the Project (estimated cost of $40,000 to $50,000)
- Three-dimensional alteration/mineralization wireframes should be constructed for use in future resource models. This task may require re-logging of select drill holes at site (estimated cost of $40,000 to $50,000).

26.1.4 Metallurgy and Process Design

- Various elements have been mentioned throughout the testwork that might have the potential for concern for concentrate quality including “talc” in 1992, selenium in 2003, fluorine, selenium, lead, zinc, and talc-like materials in 2006, and zinc and cadmium in 2010. The occurrence and potential mitigation of fluorine and hydrophobic gangue minerals in particular should be further defined. During the next phase of metallurgical testwork, a program should be developed that addresses whether these elements could impact concentrate quality (estimated cost of $100,000 to $125,000)
- Complete additional loop tests for slurry underflow from second stage cyclone to define viscosity at different densities such as at 55%, 60%, 65% (estimated cost of estimated cost of $20,000 to $30,000)
- A detailed metallurgical simulation on grinding and flotation circuits to support metallurgical performance assumptions needs to be performed (estimated cost of $75,000 to $100,000)
- An additional allocation over these specific recommendations for additional metallurgical testwork on the planned core holes that may be required to support Feasibility-level designs is included (estimated cost $1 M to $1.2 M).

26.1.5 Field Investigations

- Investigations are required to support powerline, pipeline and port site areas and alignments, reviews on the tunnel alignment in view of the Enhanced Plan assumption of additional tunnelling distances, and continued work on the access road alignment. Additional geohazards reviews will also be required (estimated cost of $3 M to $3.5 M).

26.1.6 Tunnel

- The previous risk assessment should be updated to recognize any new or relevant information that may have an impact on tunnel alignment and construction, such as from the planned geotechnical program (estimated cost of $80,000 to $100,000).

26.1.7 Engineering Studies

- A number of trade-off and design updates will be required, including mine phasing and production planning, mill design, updates and review of tailings design and water and waste management plans, and port design and assumptions review (estimated cost of $1 M to $1.4 M).

26.1.8 Environmental Baseline Studies

- Additional baseline data collection is warranted in support of the proposed mine plan and infrastructure as outlined in the GCMC 2011 pre-feasibility study, and should include areas such as hydrological, air quality, meteorology, aquatics, vegetation and fauna studies, ecosystem mapping, soil surveys, and archaeological surveys. Baseline studies applicable to the proposed port area should also be conducted and may include oceanographic, sediment, and water quality evaluations (estimated total cost of $2.7 to $3 M).

26.1.9 Environmental and Social Assessment

- A number of studies will be required in support of the Project EA. These are likely to include acid rock drainage and metals leaching studies, water quality modelling, public and First Nations consultations (estimated cost of $2 M to $2.2 M).

26.1.10 Support Costs

- Due to the remote location of the Project, and the requirement, ahead of tunnel breakthrough, for all on-site activities to have air support, significant provision is required for major logistics including camp support, and helicopter and helicopter fuel costs to support data collection activities and any Feasibility Study activities. In addition, provision for site personnel and overheads is included (estimated cost of $10 M to $15 M).

26.2 Phase 2

The information collected as part of the Phase 1 work program will be used to complete an advanced engineering study that will incorporate an updated Mineral Resource and Mineral Reserve estimate, mine plan, infrastructure layout and economic analysis. The results of this study will position the partners to make a decision as to any future mine development. This Phase 2 work is estimated to cost at a minimum of between $11 M and $13 M.

The estimated cost of the advanced engineering study, assuming the study is performed by third-party consultants with support throughout the phases of work from GCMC personnel, is expected to be approximately in the range of $8 M to $10 M, and is based on analogy with similar-scale projects in remote locations. A contingency cost of $3 M is recommended to cover the partners in the event that additional supporting engineering studies are required as the Feasibility Study advances.

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Appendix A

Claims List

Tenure No.	Claim Name	Owner	Tenure Type	Map No.	Issue Date	Good To Date	Area (ha)
226786	SPHAL #25 M.C.	211373 (100%)	Mineral	104G004	1968/Oct/18	2011/Dec/01	25
226787	SPHAL #27 M.C.	211373 (100%)	Mineral	104G004	1968/Oct/18	2011/Dec/01	25
226788	SPHAL #29 M.C.	211373 (100%)	Mineral	104G004	1968/Oct/18	2011/Dec/01	25
226789	SPHAL #31 M.C.	211373 (100%)	Mineral	104G004	1968/Oct/18	2011/Dec/01	25
226790	SPHAL #33 M.C.	211373 (100%)	Mineral	104G004	1968/Oct/18	2011/Dec/01	25
227134	KIM #38	211373 (100%)	Mineral	104G004	1970/Aug/10	2011/Dec/01	25
227135	KIM #40	211373 (100%)	Mineral	104G004	1970/Aug/10	2011/Dec/01	25
227136	KIM #42	211373 (100%)	Mineral	104G004	1970/Aug/10	2011/Dec/01	25
404921	GRACE 4	211373 (100%)	Mineral	104G013	2003/Sep/07	2018/Dec/01	500
404922	GRACE 5	211373 (100%)	Mineral	104G013	2003/Sep/07	2018/Dec/01	500
408606	VIA 17	211373 (100%)	Mineral	104G007	2004/Mar/06	2013/Dec/01	500
408613	VIA 32	211373 (100%)	Mineral	104G004	2004/Feb/29	2018/Dec/01	450
410802	J3	211373 (100%)	Mineral	104G013	2004/May/26	2018/Dec/01	300
410810	CONTACT 5	211373 (100%)	Mineral	104G023	2004/May/26	2018/Dec/01	200
410812	CONTACT 7	211373 (100%)	Mineral	104G012	2004/May/26	2018/Dec/01	450
412228	GL 16	211373 (100%)	Mineral	104G003	2004/Jul/04	2018/Dec/01	500
412241	GL 29	211373 (100%)	Mineral	104G012	2004/Jul/06	2018/Dec/01	500
501126	SPC11	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	368.042
501150	SPC01	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	438.094
501166	SPC02	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	438.096
501212	SPC03	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	437.848
501276	SPC04	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	437.851
501341	SPC06	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	315.279
501401	SPC07	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	210.367
501428	SPC05	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	315.486
501454	SPC09	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	438.097
501496	SPC10	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	437.858
501524	SPC12	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	367.917
501560	SPC13	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	367.793
501583	SPC14	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	420.171
501603	SPC15	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	420.137
501634	SPC16	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	280.043
501660	SPC17	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	420.095

Tenure No.	Claim Name	Owner	Tenure Type	Map No.	Issue Date	Good To Date	Area (ha)
501669	SPC18	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	437.659
501685	SPC20	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	419.889
501726	SPC19	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	437.421
501738	SPC21	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	420.221
501755	SPC22	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	385.557
501775	SPC23	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	437.899
501787	SPC24	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	437.661
501798	SPC25	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	420.67
501815	SPC26	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	420.408
501829	SPC27	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	210.068
501839	SPC29	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	438.001
501857	SPC28	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	420.672
501865	SPC30	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	438.002
501882	SPC31	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	420.291
501891	SPC32	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	420.136
501905	SPC08	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	210.366
501931	PORC01	211373 (100%)	Mineral	104B	2005/Jan/12	2017/Jan/12	405.39
501965	PORC02	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	440.514
501999	PORC03	211373 (100%)	Mineral	104G	2005/Jan/12	2017/Jan/12	105.708
508124	CV 1	211373 (100%)	Mineral	104G	2005/Mar/01	2013/Dec/01	440.17
508337	CV 2	211373 (100%)	Mineral	104G	2005/Mar/07	2013/Dec/01	985.4798
508338	CV 3	211373 (100%)	Mineral	104G	2005/Mar/07	2013/Dec/01	1354.832
509232	tunnel	211373 (100%)	Mineral	104G	2005/Mar/18	2018/Dec/01	333.757
509234	porc 04	211373 (100%)	Mineral	104G	2005/Mar/18	2017/Mar/18	440.357
509235	porc 05	211373 (100%)	Mineral	104G	2005/Mar/18	2017/Mar/18	405.158
509250	porc 06	211373 (100%)	Mineral	104G	2005/Mar/18	2017/Mar/18	123.308
509253	sphaler 01	211373 (100%)	Mineral	104G	2005/Mar/18	2017/Mar/18	422.571
509259	sphaler 02	211373 (100%)	Mineral	104G	2005/Mar/18	2017/Mar/18	211.356
509261	ng 01	211373 (100%)	Mineral	104G	2005/Mar/18	2017/Mar/18	420.826
509262	ng 02	211373 (100%)	Mineral	104G	2005/Mar/18	2017/Mar/18	105.208
509886	NR 1	211373 (100%)	Mineral	104G	2005/Mar/30	2018/Sep/30	421.565
509889	NR 2	211373 (100%)	Mineral	104G	2005/Mar/30	2018/Sep/30	351.223
509893	NR 3	211373 (100%)	Mineral	104G	2005/Mar/30	2018/Dec/01	70.379
511868	SPHCR 01	211373 (100%)	Mineral	104G	2005/Apr/30	2017/Apr/30	405.262

Tenure No.	Claim Name	Owner	Tenure Type	Map No.	Issue Date	Good To Date	Area (ha)
511869	SPHCR02	211373 (100%)	Mineral	104G	2005/Apr/30	2017/Apr/30	422.876
511870	SPHCR03	211373 (100%)	Mineral	104G	2005/Apr/30	2017/Apr/30	422.878
512425		211373 (100%)	Mineral	104G	2005/May/11	2018/Dec/01	700.818
512426		211373 (100%)	Mineral	104G	2005/May/11	2018/Dec/01	473.235
512478	CONT 1	211373 (100%)	Mineral	104G	2005/May/12	2017/May/26	770.372
514542	THOMAS 1	211373 (100%)	Mineral	104G	2005/Jun/15	2013/Dec/01	421.877
514545	THOMAS 2	211373 (100%)	Mineral	104G	2005/Jun/15	2013/Dec/01	422.043
514548	THOMAS 3	211373 (100%)	Mineral	104G	2005/Jun/15	2013/Dec/01	421.895
514551	THOMAS 4	211373 (100%)	Mineral	104G	2005/Jun/15	2013/Dec/01	369.29
515244	ISKUT 1	211373 (100%)	Mineral	104G	2005/Jun/24	2013/Dec/01	422.157
516158		211373 (100%)	Mineral	104G	2005/Jul/06	2016/Dec/01	772.237
516161		211373 (100%)	Mineral	104G	2005/Jul/06	2018/Dec/01	543.835
516163		211373 (100%)	Mineral	104G	2005/Jul/06	2018/Dec/01	1244.967
516165		211373 (100%)	Mineral	104G	2005/Jul/06	2016/Dec/01	667.543
516177		211373 (100%)	Mineral	104G	2005/Jul/06	2016/Dec/01	175.777
516178		211373 (100%)	Mineral	104G	2005/Jul/06	2018/Dec/01	457.053
516179		211373 (100%)	Mineral	104G	2005/Jul/06	2018/Dec/01	1317.27
516235		211373 (100%)	Mineral	104G	2005/Jul/07	2018/Dec/01	1161.63
516271		211373 (100%)	Mineral	104G	2005/Jul/07	2018/Dec/01	315.411
516275		211373 (100%)	Mineral	104G	2005/Jul/07	2018/Dec/01	1407.331
516284		211373 (100%)	Mineral	104G	2005/Jul/07	2018/Dec/01	947.189
516285		211373 (100%)	Mineral	104G	2005/Jul/07	2018/Dec/01	614.229
516286		211373 (100%)	Mineral	104G	2005/Jul/07	2018/Dec/01	912.089
516327		211373 (100%)	Mineral	104G	2005/Jul/08	2018/Dec/01	999.585
516335		211373 (100%)	Mineral	104G	2005/Jul/08	2015/Dec/01	1354.185
516340		211373 (100%)	Mineral	104G	2005/Jul/08	2018/Dec/01	1195.156
516342		211373 (100%)	Mineral	104G	2005/Jul/08	2018/Dec/01	1107.372
516345		211373 (100%)	Mineral	104G	2005/Jul/08	2018/Dec/01	949.18
516359		211373 (100%)	Mineral	104G	2005/Jul/08	2018/Dec/01	789.736
516367		211373 (100%)	Mineral	104G	2005/Jul/08	2018/Dec/01	1052.596
516377		211373 (100%)	Mineral	104G	2005/Jul/08	2018/Dec/01	1143.352
516433		211373 (100%)	Mineral	104G	2005/Jul/08	2018/Dec/01	1318.728
516441		211373 (100%)	Mineral	104G	2005/Jul/08	2018/Dec/01	1390.457
516443		211373 (100%)	Mineral	104G	2005/Jul/08	2018/Dec/01	880.157

Tenure No.	Claim Name	Owner	Tenure Type	Map No.	Issue Date	Good To Date	Area (ha)
516445		211373 (100%)	Mineral	104G	2005/Jul/08	2018/Dec/01	985.011
516448		211373 (100%)	Mineral	104G	2005/Jul/08	2018/Dec/01	862.311
516452		211373 (100%)	Mineral	104G	2005/Jul/08	2018/Dec/01	879.374
516458		211373 (100%)	Mineral	104G	2005/Jul/08	2018/Dec/01	949.726
516459	GALORE 1 CELL CLAIM	211373 (100%)	Mineral	104G	2005/Jul/08	2016/Dec/01	1721.252
516463	NR 4	211373 (100%)	Mineral	104G	2005/Jul/08	2018/Dec/01	140.84
516474	SPHCR 04	211373 (100%)	Mineral	104B	2005/Jul/08	2017/Jul/08	422.996
516475	SPHCR 05	211373 (100%)	Mineral	104B	2005/Jul/08	2017/Jul/08	422.996
516496		211373 (100%)	Mineral	104G	2005/Jul/09	2018/Dec/01	1299.197
516498		211373 (100%)	Mineral	104G	2005/Jul/09	2018/Dec/01	1105.922
516500		211373 (100%)	Mineral	104G	2005/Jul/09	2018/Dec/01	1527.806
516503		211373 (100%)	Mineral	104G	2005/Jul/09	2018/Dec/01	1178.494
516505		211373 (100%)	Mineral	104G	2005/Jul/09	2018/Dec/01	1126.672
516508		211373 (100%)	Mineral	104G	2005/Jul/09	2018/Dec/01	1020.993
516509		211373 (100%)	Mineral	104G	2005/Jul/09	2018/Dec/01	1039.113
516511		211373 (100%)	Mineral	104G	2005/Jul/09	2018/Dec/01	968.695
516674		211373 (100%)	Mineral	104G	2005/Jul/11	2018/Dec/01	157.819
516691		211373 (100%)	Mineral	104G	2005/Jul/11	2018/Dec/01	563.2
516839	NR 4	211373 (100%)	Mineral	104G	2005/Jul/11	2018/Sep/30	35.123
516900	NR 05	211373 (100%)	Mineral	104G	2005/Jul/11	2018/Sep/30	87.817
516903	NR 06	211373 (100%)	Mineral	104G	2005/Jul/11	2018/Sep/30	175.648
517018	NR 06	211373 (100%)	Mineral	104G	2005/Jul/12	2018/Sep/30	105.381
517480	GRACE G	211373 (100%)	Mineral	104G	2005/Jul/12	2017/Jul/12	52.637
520000	MORE CK	211373 (100%)	Mineral	104G	2005/Sep/15	2018/Sep/30	228.307
521931	BQ 1	211373 (100%)	Mineral	104G	2005/Nov/04	2012/Dec/01	422.299
521932	BQ 2	211373 (100%)	Mineral	104G	2005/Nov/04	2012/Dec/01	422.295
521933	BQ 3	211373 (100%)	Mineral	104G	2005/Nov/04	2012/Dec/01	422.482
521934	BQ 4	211373 (100%)	Mineral	104G	2005/Nov/04	2012/Dec/01	440.088
521935	BQ 5	211373 (100%)	Mineral	104G	2005/Nov/04	2012/Dec/01	440.241
521936	BQ 6	211373 (100%)	Mineral	104G	2005/Nov/04	2012/Dec/01	422.668
521937	BQ 7	211373 (100%)	Mineral	104G	2005/Nov/04	2012/Dec/01	422.855
521938	BQ 8	211373 (100%)	Mineral	104G	2005/Nov/04	2013/Dec/01	440.388
521939	BQ 9	211373 (100%)	Mineral	104G	2005/Nov/04	2013/Dec/01	422.915

Tenure No.	Claim Name	Owner	Tenure Type	Map No.	Issue Date	Good To Date	Area (ha)
521941	BQ 10	211373 (100%)	Mineral	104B	2005/Nov/04	2012/Dec/01	440.652
521943	BQ 11	211373 (100%)	Mineral	104B	2005/Nov/04	2013/Dec/01	246.771
521945	BQ 12	211373 (100%)	Mineral	104G	2005/Nov/04	2012/Dec/01	88.089
522111	BQ 13	211373 (100%)	Mineral	104G	2005/Nov/07	2013/Dec/01	70.397
522318	CONT 2	211373 (100%)	Mineral	104G	2005/Nov/15	2018/Dec/01	386.718
522319	CONT 3	211373 (100%)	Mineral	104G	2005/Nov/15	2018/Dec/01	245.815
537446	ECR	211373 (100%)	Mineral	104B	2006/Jul/20	2013/Dec/20	158.873
545723	THOMAS 5	211373 (100%)	Mineral	104G	2006/Nov/22	2013/Dec/01	87.8953
545725	CV 4	211373 (100%)	Mineral	104G	2006/Nov/22	2013/Dec/01	175.9993
547085	BTO 01	211373 (100%)	Mineral	104G	2006/Dec/09	2012/Dec/09	440.1768
547086	BTO 02	211373 (100%)	Mineral	104G	2006/Dec/09	2012/Dec/09	352.1113
547087	BTO 03	211373 (100%)	Mineral	104G	2006/Dec/09	2012/Dec/09	422.5071
547088	BTO 04	211373 (100%)	Mineral	104G	2006/Dec/09	2012/Dec/09	422.5069
547089	BTO 05	211373 (100%)	Mineral	104G	2006/Dec/09	2012/Dec/09	422.5117
547090	BTO 06	211373 (100%)	Mineral	104G	2006/Dec/09	2012/Dec/09	422.5141
547091	BTO 07	211373 (100%)	Mineral	104G	2006/Dec/09	2012/Dec/09	422.5103
547092	BTO 08	211373 (100%)	Mineral	104G	2006/Dec/09	2012/Dec/09	422.5198
547093	BTO 09	211373 (100%)	Mineral	104G	2006/Dec/09	2012/Dec/09	299.1494
547094	BTO 10	211373 (100%)	Mineral	104G	2006/Dec/09	2012/Dec/09	422.7855
547095	BTO 11	211373 (100%)	Mineral	104G	2006/Dec/09	2012/Dec/09	422.7853
547096	BTO 12	211373 (100%)	Mineral	104G	2006/Dec/09	2012/Dec/09	422.791
547097	BTO 13	211373 (100%)	Mineral	104G	2006/Dec/09	2012/Dec/09	422.8057
547098	BTO 14	211373 (100%)	Mineral	104G	2006/Dec/09	2012/Dec/09	422.8018
547099	BTO 15	211373 (100%)	Mineral	104G	2006/Dec/09	2012/Dec/09	422.7944
547100	BTO 16	211373 (100%)	Mineral	104G	2006/Dec/09	2012/Dec/09	422.8065
547101	BTO 17	211373 (100%)	Mineral	104G	2006/Dec/09	2012/Dec/09	246.5536
547102	BTO 18	211373 (100%)	Mineral	104B	2006/Dec/09	2012/Dec/09	423.0759
547103	BTO 19	211373 (100%)	Mineral	104B	2006/Dec/09	2012/Dec/09	423.0746
547104	BTO 20	211373 (100%)	Mineral	104B	2006/Dec/09	2012/Dec/09	423.0805
547105	BTO 21	211373 (100%)	Mineral	104B	2006/Dec/09	2012/Dec/09	423.102
547106	BTO 22	211373 (100%)	Mineral	104B	2006/Dec/09	2012/Dec/09	423.1018
547107	BTO 23	211373 (100%)	Mineral	104B	2006/Dec/09	2012/Dec/09	423.097
547108	BTO 024	211373 (100%)	Mineral	104B	2006/Dec/10	2012/Dec/10	423.0539
547111	BTO 25	211373 (100%)	Mineral	104B	2006/Dec/10	2012/Dec/10	440.9764

Tenure No.	Claim Name	Owner	Tenure Type	Map No.	Issue Date	Good To Date	Area (ha)
547113	BTO 26	211373 (100%)	Mineral	104B	2006/Dec/10	2012/Dec/10	440.9843
547115	BTO 27	211373 (100%)	Mineral	104B	2006/Dec/10	2012/Dec/10	211.5211
556327		211373 (100%)	Mineral	104G	2007/Apr/13	2018/Dec/01	387.2667
556330		211373 (100%)	Mineral	104G	2007/Apr/13	2018/Dec/01	281.5297
556331		211373 (100%)	Mineral	104G	2007/Apr/13	2018/Dec/01	140.7942
556334		211373 (100%)	Mineral	104G	2007/Apr/13	2018/Dec/01	211.1915
560604	BQ 14	211373 (100%)	Mineral	104B	2007/Jun/13	2012/Dec/01	423.1134
560608	BQ 15	211373 (100%)	Mineral	104B	2007/Jun/13	2012/Dec/01	405.7895
560612	BQ 16	211373 (100%)	Mineral	104B	2007/Jun/13	2012/Dec/01	317.7117
560615	BQ 17	211373 (100%)	Mineral	104G	2007/Jun/13	2012/Dec/01	176.1681
566898	THOMAS 6	211373 (100%)	Mineral	104G	2007/Sep/28	2013/Dec/01	211.1012
579405	SCU 1	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	437.2202
579406	SCUD 1	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	419.9753
579407	SCUD 2	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	122.4604
579408	SCU 2	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	437.2223
579409	SCUD 3	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	349.8247
579410	SCU 3	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	436.9756
579411	SCUD 4	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	419.9061
579412	SCUD 5	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	349.7099
579413	SCU 3	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	419.0939
579414	SCUD 6	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	157.3518
579416	SCU 4	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	401.6306
579417	SCUD 7	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	419.9056
579418	SCU 5	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	436.9768
579420	SCUD 8	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	419.6281
579421	SCU 6	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	436.9789
579423	SCUD 9	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	437.1346
579424	SCU 7	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	436.9808
579426	SCU 8	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	436.9835
579428	SCUD 10	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	244.6974
579429	SCU 9	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	419.2886
579431	SCUD 11	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	366.949
579432	SCU 10	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	419.2913
579434	SCU 11	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	419.3084

Tenure No.	Claim Name	Owner	Tenure Type	Map No.	Issue Date	Good To Date	Area (ha)
579435	SCUD 12	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	209.7657
579436	SCU 12	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	436.7655
579437	SCUD 13	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	419.4795
579439	SCU 13	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	437.0121
579441	SCU 14	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	437.2245
79443	SCU 15	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	437.2253
579454	RDL 1	211373 (100%)	Mineral	104G	2008/Mar/28	2018/Dec/01	421.8799
579456	RDL 2	211373 (100%)	Mineral	104G	2008/Mar/28	2018/Dec/01	439.4831
579457	LIN 1	211373 (100%)	Mineral	104G	2008/Mar/28	2015/Dec/01	421.6811
579458	RDL 3	211373 (100%)	Mineral	104G	2008/Mar/28	2015/Dec/01	439.34
579459	LIN 2	211373 (100%)	Mineral	104G	2008/Mar/28	2015/Dec/01	421.7224
579461	RDL 4	211373 (100%)	Mineral	104G	2008/Mar/28	2015/Dec/01	421.6429
579462	LIN 3	211373 (100%)	Mineral	104G	2008/Mar/28	2015/Dec/01	298.7028
579463	RDL 5	211373 (100%)	Mineral	104G	2008/Mar/28	2015/Dec/01	421.6515
579467	RDL 6	211373 (100%)	Mineral	104G	2008/Mar/28	2015/Dec/01	421.5126
579469	RDL 7	211373 (100%)	Mineral	104G	2008/Mar/28	2015/Dec/01	421.512
579470	LIN 6	211373 (100%)	Mineral	104G	2008/Mar/28	2015/Dec/01	333.6831
579472	LIN 7	211373 (100%)	Mineral	104G	2008/Mar/28	2015/Dec/01	438.8378
579473	RDL 8	211373 (100%)	Mineral	104G	2008/Mar/28	2015/Dec/01	421.5266
579479	LIN 10	211373 (100%)	Mineral	104G	2008/Mar/28	2015/Dec/01	421.016
579517	SCUD S1	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	437.3757
579519	SCUD S2	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	420.114
579521	SCUD S3	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	350.0739
579523	SCUD S4	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	420.2729
579526	SCUD S5	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	420.2704
579528	SCUD S6	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	419.7174
579530	SCUD S7	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	419.7149
579532	SCUD S8	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	419.9041
579535	SCUD S9	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	420.0905
579537	SCUD S10	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	350.2287
579541	SCUD S11	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	385.4026
579542	SCUD S12	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	420.4623
579544	SCUD S13	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	419.9021
579545	SCUD S14	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	420.0891

Tenure No.	Claim Name	Owner	Tenure Type	Map No.	Issue Date	Good To Date	Area (ha)
579547	SCUD S15	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	437.4696
579548	SCUD S16	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	437.4701
579549	SCUD S17	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	437.4678
579550	SCUD S18	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	437.4649
579551	SCUD S19	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	420.2738
579552	SCUD S20	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	437.7128
579553	SCUD S21	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	437.7161
579554	SCUD S22	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	437.7156
579556	SCUD S22	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	437.7135
579557	SCUD S23	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	420.4638
579558	SCUD S24	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	420.4437
579559	SCUD S25	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	437.964
579560	SCUD S26	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	437.9651
579561	SCUD S27	211373 (100%)	Mineral	104G	2008/Mar/28	2014/Mar/28	437.9638
585412	RDL 21	211373 (100%)	Mineral	104G	2008/May/29	2018/Dec/01	35.1912
586551	BQ 18	211373 (100%)	Mineral	104B	2008/Jun/19	2013/Dec/01	423.215
586552	BQ 19	211373 (100%)	Mineral	104B	2008/Jun/19	2013/Dec/01	423.492
586554	BQ 20	211373 (100%)	Mineral	104B	2008/Jun/19	2013/Dec/01	105.918
586555	BQ 21	211373 (100%)	Mineral	104B	2008/Jun/19	2012/Dec/01	211.6139
586556	BQ 22	211373 (100%)	Mineral	104B	2008/Jun/19	2013/Dec/01	70.5912
586557	BQ 23	211373 (100%)	Mineral	104B	2008/Jun/19	2013/Dec/01	176.4411
590765	BQ 24	211373 (100%)	Mineral	104B	2008/Sep/03	2013/Dec/09	441.2752
601066	BQ 25	211373 (100%)	Mineral	104G	2009/Mar/14	2012/Dec/09	35.2236
601070	BQ 26	211373 (100%)	Mineral	104G	2009/Mar/14	2012/Dec/09	35.231
662956	RLS 1	211373 (100%)	Mineral	104G	2009/Oct/31	2018/Dec/01	70.3864
662967	RLS 2	211373 (100%)	Mineral	104G	2009/Oct/31	2018/Dec/01	70.3828
662975	R 1	211373 (100%)	Mineral	104G	2009/Oct/31	2018/Dec/01	87.9738
662982	RLS 3	211373 (100%)	Mineral	104G	2009/Oct/31	2018/Dec/01	105.567
	264	**Mineral Claims**				**Hectares:**	**118,911.88**
						Acres:	**293,831.26**