

5. Project Description

5.1 Project History

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

In 1959 reconnaissance stream silt surveys were carried out by Kennco Explorations (Western) Limited (a subsidiary of Kennecott Copper Corporation) in the Stikine River area. Results from this work prompted Kennco to stake mineral claims the following year around the remaining 16 claims owned by Hudson Bay. Four of the original claims were subsequently optioned by Consolidated Mining and Smelting Company of Canada Limited from W. Bucholz. Late in 1962 the three companies agreed to participate jointly in future exploration work. As a result, Stikine Copper Limited was incorporated in 1963.

From 1960 to 1968, the property was explored by Kennco Exploration (Western) Ltd. Exploration work during this period included 53,164 metres of diamond drilling in 235 holes and 807 metres of tunnelling in two adits to extract a 50 ton bulk sample for pilot plant testing. Preliminary open pit design and metallurgical studies were done in 1967. The Central zone was the focus of most of this work. No work on the property was undertaken from 1968 to 1972.

In 1972 Hudson Bay Smelting became operator and in 1972 and 1973 an additional 25,352 metres of diamond drilling was completed in 111 holes. This work focused exclusively on delineating resources in the Central and North Junction zones. A further 5,310 metres of diamond drilling was completed in 24 holes in 1976. In 1981, Wright Engineers, for Hudson Bay Mining and Smelting and Stikine Copper, studied several access routes and transportation methods to develop the resource.

Mingold Resources, an affiliate of Hudson Bay, was operator of the property in 1989 and 1990. Mingold investigated the gold potential by drilling 18 core holes totalling 1,225 metres. Kennecott resumed operatorship of the property in 1991 and completed 18,380 metres of diamond drilling in 49 holes. A relatively limited geotechnical study was completed at this time. An Aerodat helicopter magnetic survey, electromagnetic and radiometric survey, ground magnetics/very low frequency (VLF) electromagnetic surveys and over 90 line kilometres of 60 metre pole-dipole induced polarization(IP)/resistivity surveys were also completed by Kennecott.

Mine Reserve Associates, Inc. completed a resource model in 1992 for Kennecott Exploration. Based on this model, Kennecott re-classified the mineral resource to comply with industry standards existing in 2002. Values used were \$10/tonne in-situ metal value as a cutoff grade based on US\$0.80/lb copper and US\$320/oz gold prices. Kennecott estimated an indicated resource of 243.2 million tonnes grading 0.75% copper and 0.45 g/t gold containing 3.6 million

ounces of gold and 920 million pounds of copper. Silver was not included in the 1992 resource model. This resource does not conform to National Instrument 43-101 standards and is reported for historical purposes only. Kennecott conducted more metallurgical investigations in 1992.

In August 2003 SpectrumGold Canada Inc. (now NovaGold) entered into an option agreement to acquire Stikine Copper from its shareholders. NovaGold carried out 2,950 metres of diamond drilling in 10 holes in 2003 to confirm copper and gold mineralization grades defined by previous drilling.

In 2004 NovaGold undertook 1,072 line kilometres of helicopter magnetic and radiometric surveys, 28 kilometres of large dipole IP/resistivity surveys combined with 2D IP/resistivity modeling, 10.5 kilometres of shallow seismic refraction surveys for engineering design and 25 kilometres of ground magnetic survey using five metre stations and 25 metre line spacings.

NovaGold drilled 25,976 metres in 79 holes in 2004 and a further 57,700 metres in 205 holes in 2005 to confirm grades for pre-feasibility level planning. Also in 2004 NovaGold initiated a review of potential access routes, environmental and socio-economic baseline studies and comprehensive engineering, geotechnical and metallurgical studies towards completion of a pre-feasibility study.

5.2 Location and Access

The Galore Creek Project is located between the Stikine and Iskut rivers and Highway 37 in northwestern British Columbia (Figure 5.2-1). The property is centred at latitude 57°07'30"N and longitude 131°27'W. It occurs within the Liard Mining Division and straddles the boundary between NTS map sheets 104G/3 and 104G/4.

The Galore Creek Valley is steep and encircled on three sides by high ice-covered mountains (Figure 5.2-2). Galore Creek flows northwards to the Scud River, a tributary of the Stikine River which in turn empties into the Pacific Ocean near Wrangell, Alaska.

The project is approximately 1,000 km northwest of Vancouver and 150 km northwest of the deepsea port located at Stewart, BC. Travel distance by existing and proposed roads from the project to Vancouver is about 1,600 km and to Stewart about 260 km.

The project is located on provincial Crown land in the area covered by the Cassiar Iskut-Stikine Land and Resource Management Plan, approved by the British Columbia government in 2000.

The project is also situated within the traditional territory claimed by the Tahltan Nation which includes the Tahltan Band Council and Iskut First Nation. The Tahltan Band Council has Indian reserves in: Dease Lake, approximately 280 kilometres from the project site by existing and proposed roads; and Telegraph Creek, approximately 380 kilometres from the project site by existing and proposed roads. The Iskut First Nation's reserve is in Iskut which is approximately 200 kilometres from the project site by existing and proposed roads.



**Galore Creek Project Location
Relative to British Columbia and Alaska**

FIGURE 5.2-1

The Bob Quinn airstrip on Highway 37 is about 70 km east of the Galore Creek Valley. Although two airstrips were constructed on the property in the 1960s, present access to the property is by helicopter only.

5.2.1 Location Plan with Site Layout

NovaGold proposes to develop the Galore Creek copper-gold-silver resources by using conventional truck and shovel mining methods and crushing/grinding/flotation processes to extract copper and precious metals to a bulk concentrate for shipment to an off-site smelter. A slurry pipeline will transport copper concentrate to a filter plant near Highway 37 where it will be de-watered for transport by truck to Stewart. A small diameter diesel pipeline will transport diesel fuel from the filter plant to the mine site, paralleling the concentrate pipeline.

The open pits, processing plant and related support facilities such as shops and employee accommodation will all be located in the Galore Creek Valley (Figure 5.2-3). These facilities will be discussed in detail in later sections of this report.

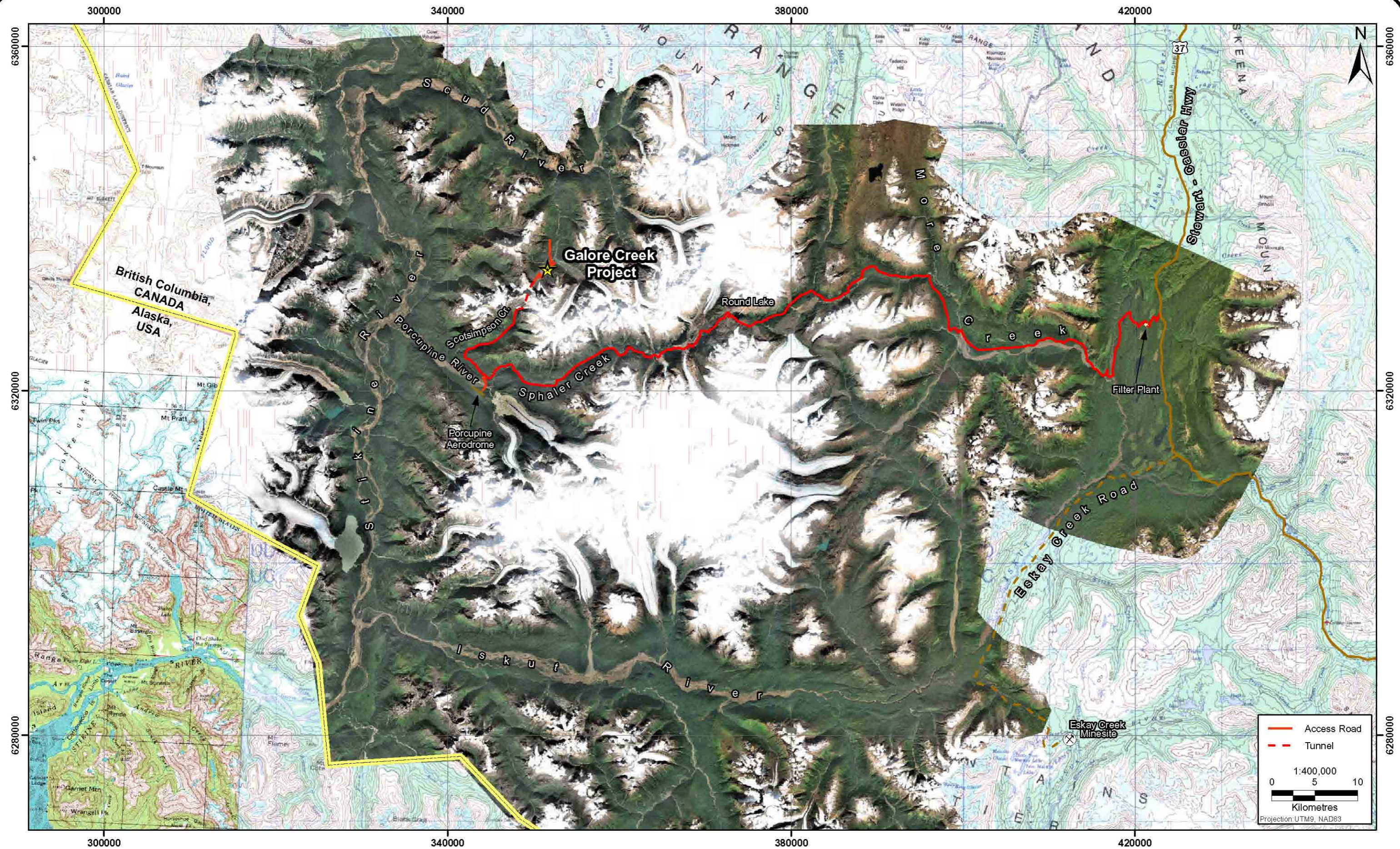
5.2.2 Access Description and Plan

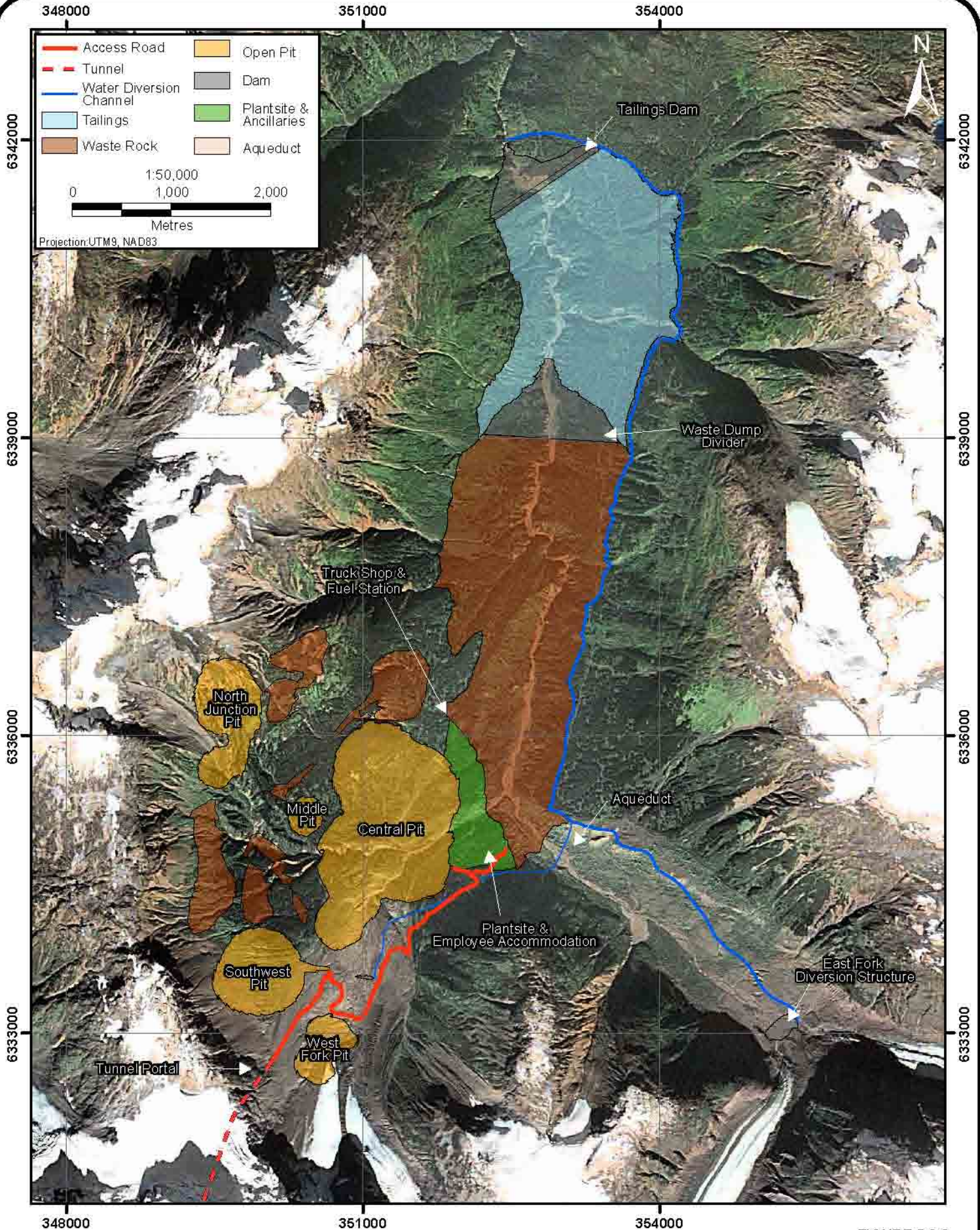
NovaGold investigated several alternatives for access to develop the Galore Creek Project. These alternatives are discussed in detail in Section 11.1 of this report. NovaGold's preferred access alternative, from environmental, socio-cultural and technical and economic feasibility perspectives, is the modified Northern route. This route was identified in consultation with government agencies and First Nations early in 2005. It will start from Highway 37 north of Bob Quinn Lake, follow More Creek upstream from its junction with the Iskut River to the pass at the head of More and Sphaler creeks, descend the Sphaler Creek Canyon to the Porcupine River, then ascend Scotsimpson Creek to a 3.8 km tunnel. The tunnel will allow access from Scotsimpson Creek Valley to the Galore Creek Valley. The distance from Highway 37 to the south tunnel portal will be 128 km, 118 km of which will require new construction. Section 5.11 presents a comprehensive discussion of the road. With relatively minor variations, the access road, concentrate slurry pipeline and electric power transmission line will all follow the same alignment (Figure 5.2-4).

Construction of the road will take about 12 months. The access road will be used for initial delivery of machinery and equipment for the construction of the mine and for ongoing delivery of grinding media, reagents and other operating supplies. Concentrate will be transported by slurry pipeline to a filter plant near the junction of the access road with Highway 37. Most heavy vehicles leaving the mine will be travelling empty, allowing relatively steep uphill grades for eastbound traffic.

An aerodrome, designed to accommodate fixed wing aircraft, will be established near the confluence of Sphaler Creek and Porcupine River, initially to support construction activities. The aerodrome will be maintained as a key feature of the operational infrastructure to permit timely all season transportation of personnel.

A temporary construction phase heavy lift heliport will be established at West More Camp east of Round Lake to support initial road construction.





Location Plan with Ultimate Mine Footprint

FIGURE 5.2-3

5.3 Geology and Mineralization

5.3.1 Regional Geology

The Galore Creek deposits lie in Stikinia Terrane, an accreted package of Mesozoic volcanic and sedimentary rocks intruded by Cretaceous to Eocene plutonic and volcanic rocks (Figure 5.3-1). The eastern boundary of the Coast Plutonic Complex lies about seven kilometres to the west of the claims. The property lies within a regional transcurrent structure known as the Stikine Arch.

5.3.1.1 Stratigraphy

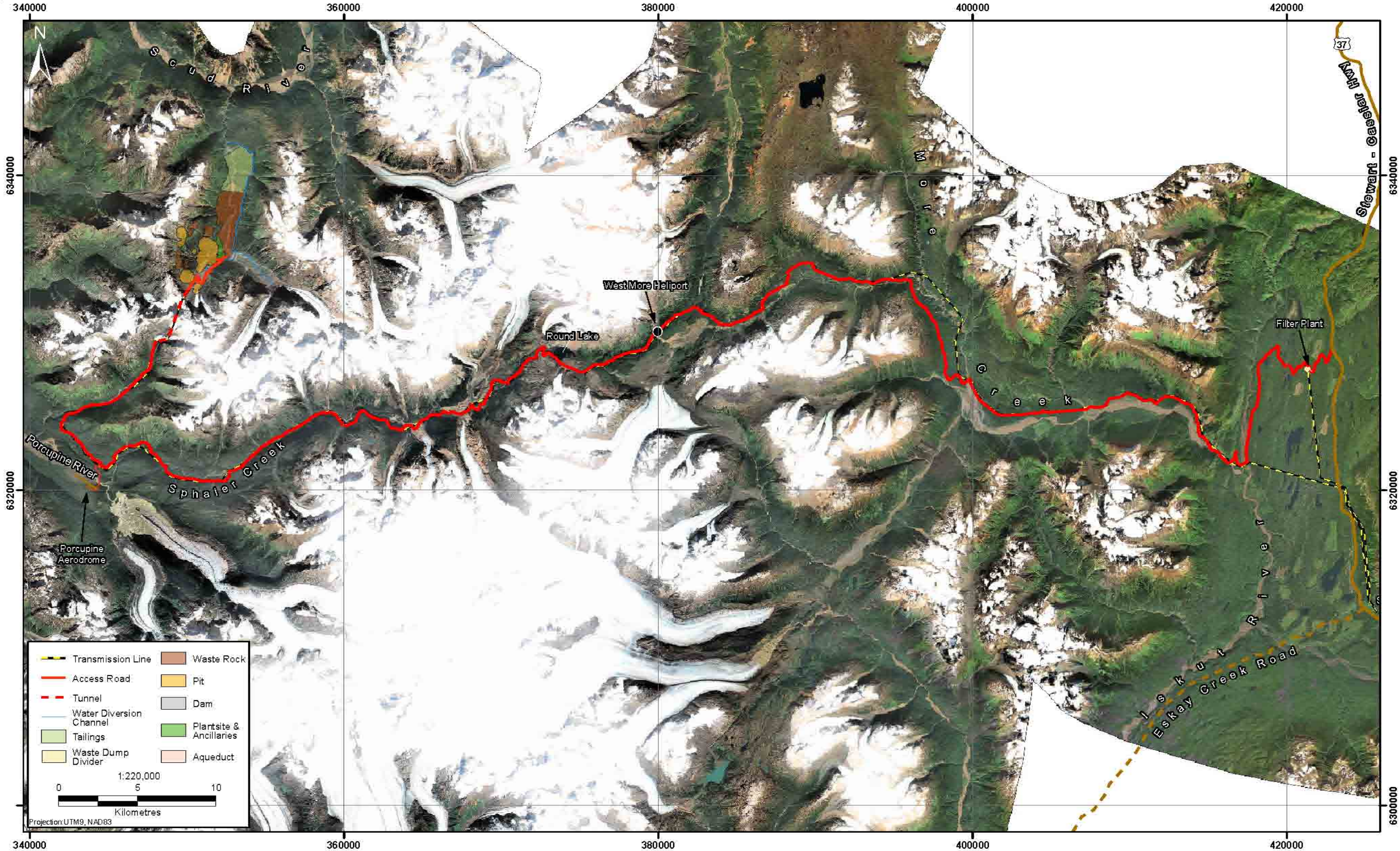
Stikinia Terrane at this latitude can be grouped into four tectonostratigraphic successions. The first and most important is a Late Paleozoic to Middle Jurassic island arc suite represented by the Stikine assemblage of Monger (1977), the Stuhini Group (Kerr, 1948) and Hazelton Group equivalent rocks. The other successions are: Middle Jurassic to early Late Cretaceous successor-basin sediments of the Bowser Lake Group (Tipper and Richards, 1976); Late Cretaceous to Tertiary transtensional continental volcanic-arc assemblages of the Sloko Group (Aiken, 1959); and Late Tertiary to Recent post-orogenic plateau basalt bimodal volcanic rocks of the Edziza and Spectrum ranges.

The oldest stratigraphy in the area is known as the Stikine assemblage and comprises Permian and older argillites, mafic to felsic flows and tuffs. These rocks grade upward into two distinctive Mississippian limestone members separated by intercalated volcanics and clastic sediments. The topmost stratigraphy consists of two regionally extensive Permian carbonate units that suggest a stable continental shelf depositional environment. These carbonate units crop out to the north of the proposed Galore Creek Valley tailings and waste rock impoundment and in the upper More Creek Valley. Karst topography associated with these units was a factor in decisions regarding the location of project facilities such as major dams. The Middle to Upper Triassic Stuhini Group unconformably overlies the Stikine assemblage. Stuhini Group rocks comprise a variety of flows, tuffs, volcanic breccias and sediments, and are important host rocks to the alkaline intrusive-related copper-gold-silver mineralization at Galore Creek. They define a volcanic edifice centered on Galore Creek and represent an emergent Upper Triassic island arc characterized by shoshonitic and leucitic volcanics (de Rosen-Spence, 1985), distal volcanoclastics and sedimentary turbidites. The succession at Galore Creek was divided by Panteleyev (1976) into a submarine basalt and andesite lower unit overlain by more differentiated, partly subaerial alkali-enriched flows and pyroclastic rocks.

A fault-bounded wedge of unnamed Jurassic sediments unconformably overlies the Stuhini Group rocks. Within this unnamed Jurassic succession is a basal purple to red polymictic boulder and cobble conglomerate with an arkosic matrix. It contains granitic clasts including distinctive K-feldspar porphyries that are Galore Creek syenite equivalents.

5.3.1.2 Intrusive Rocks

Three intrusive episodes have been recognized in the region. The earliest and most important is the Middle Triassic to Middle Jurassic Hickman plutonic suite that is coeval with Upper Triassic Stuhini Group volcanic flows. The Mount Hickman batholith comprises three plutons known as Hickman, Yehino and Nightout. The latter two are exposed north of the map area.

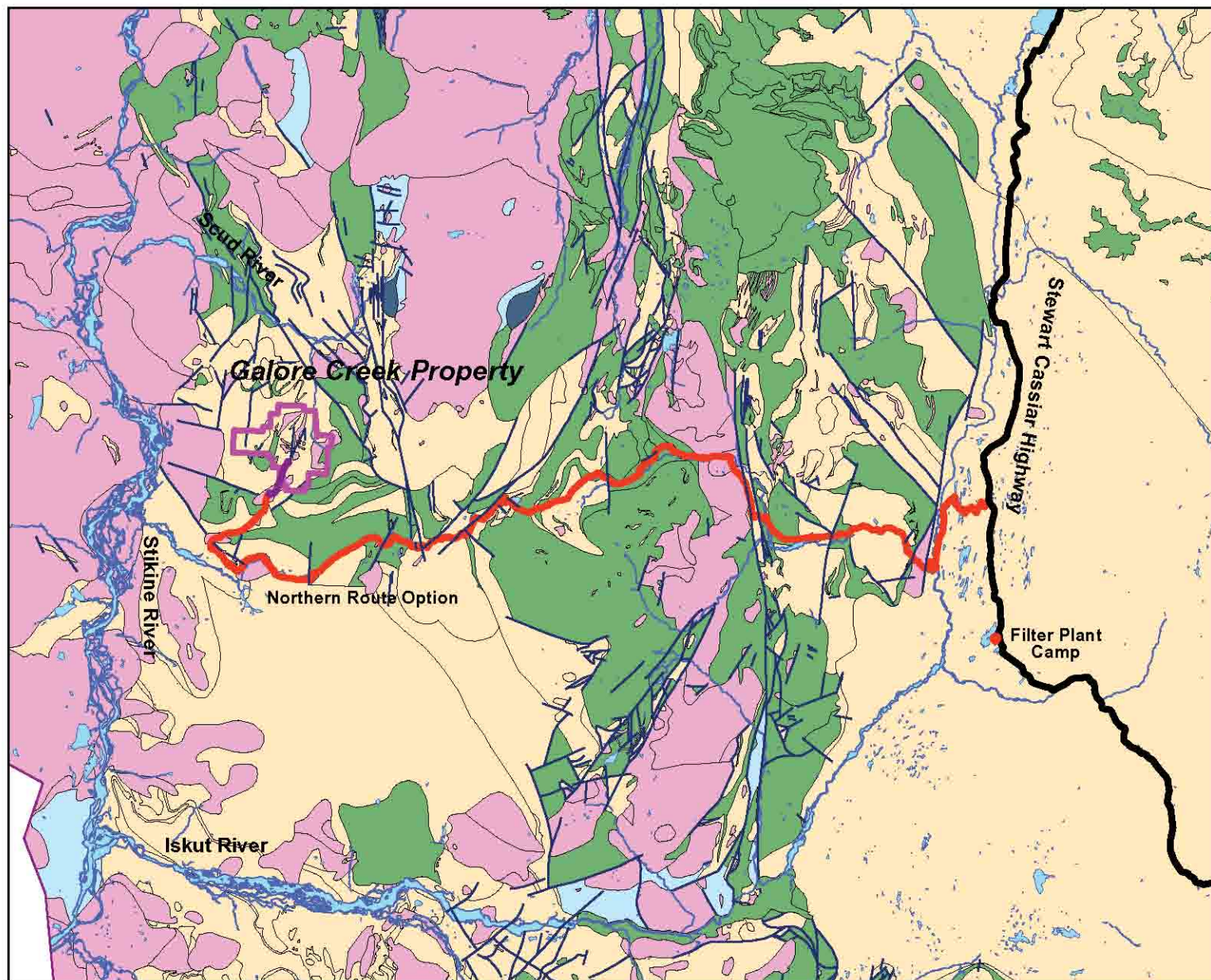


	Transmission Line		Waste Rock
	Access Road		Pit
	Tunnel		Dam
	Water Diversion Channel		Plantsite & Ancillaries
	Tailings		Aqueduct
	Waste Dump Divider		

0 5 10
Kilometres
Projection: UTM9, NAD83

Access Plan

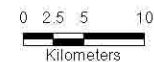
FIGURE 5.2-4



Legend

- volcanic rocks
- intrusive rocks
- ultramafic rocks
- sedimentary rocks
- metamorphic rocks
- Proposed Access Route
- Stewart Cassiar Highway
- Galore Creek Property
- Proposed tunnel

Galore Creek Project
Regional Geology
and
Proposed Access Route



Regional Geology Source: BC Geological Survey (2005)
North American Datum 1983; Projection UTM Zone 9 (m)

FIGURE 5.3-1

The Schaft Creek porphyry copper deposit is associated with the Hickman stock, and is located 39 kilometres northeast of Galore Creek. This stock is crudely zoned with a pyroxene diorite core and biotite granodiorite margin. Alkali syenites of the Galore complex like those found at the nearby Copper Canyon deposit and the pyroxene diorite bodies of the zoned Hickman pluton have been interpreted as differentiated end members of the Stuhini volcanic - Hickman plutonic suite by Souther (1972) and Barr (1966). The alkali syenites are associated with important copper-gold-silver mineralization at Galore Creek and at Copper Canyon. These rocks are believed to be at least as old as Early Jurassic in age, based on K-Ar dating of hydrothermal biotite in the syenites intruding the sequences (Allen, 1966). An Ar-Ar age of 212 Ma (Logan et al., 1989) in syenite may give the time of crystallization of the intrusive rocks at Copper Canyon. Uranium-lead ages (Mortensen et al., 1995) from intrusive rocks at Galore Creek range from 210 ± 1 Ma (Pb-Pb isochron, titanite) for an early intermineral syenite porphyry to 197.2 ± 1.2 Ma (Pb-Pb isochron, titanite) for a postmineral potassium feldspar porphyry. These span the Triassic-Jurassic boundary (~200 Ma), using the time scale of Palfy et al (2000), and suggest a protracted magmatic history for the Galore Creek magmatic complex (Logan, 2004).

Coast Range intrusions comprise the large plutonic mass west of the map area. Historically, three texturally and compositionally distinct intrusive phases have been mapped. From inferred oldest to youngest, they are K-feldspar megacrystic granite to monzonite; biotite hornblende diorite to granodiorite; and biotite granite. Small Tertiary intrusive stocks and dykes are structurally controlled in their distribution. At Galore Creek young post-mineral basalt and felsite dykes are abundant as a dyke swarm in the northwest part of the property. Elsewhere, Tertiary intrusions may be important in their association with small gold occurrences.

5.3.1.3 Regional Structure

Logan and Koyanagi (1988) recognized three phases of deformation in the Galore Creek area (Sphaler Creek (104G/3) and Flood Glacier (104G/4) map areas). The oldest Paleozoic rocks (pre-Upper Triassic) are affected in the form of 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.

Upper Triassic and younger strata are much less deformed, lack the penetrative planar fabric, and have been affected by D3 deformation shown by open box folding with west to northwest trending fold axes (Logan and Koyanagi, 1988), although two generations of folding were recognized in Triassic rocks at Galore Creek (Panteleyev, 1976). Mapping by Brown and Gunning (1988) in the Scud River area to the north also recognize this divide in structural styles, and propose four structural domains that are largely restricted to Paleozoic strata except for a post-Late Triassic contraction (Domain 4) that caused northeast-plunging tight folding in the Upper Triassic rocks. In regional studies from adjacent areas, such as Forrest Kerr- Mess Creek (Logan et al., 2000) and Western Telegraph Creek (Brown et al., 1996), two deformational phases have been proposed for Mesozoic strata: an Early Jurassic contraction that caused an angular unconformity between the Late Triassic Stuhini assemblages and Early Jurassic rocks; and a Middle or Late Jurassic to Tertiary contraction that could correlate to the D3 phase of Logan and Koyanagi (1988).

Logan and Koyanagi (1988) define four major fault sets in the region. The most prominent and longest lived fault structures strike north, and may be high angle to vertical or dipping to the east (thrust faulting in Copper Canyon). A set of northwest-striking faults dip to the northeast and often have normal displacement. East-striking shear zones may be localized along the eastern margin of the Coast Complex. The youngest faults strike north-northeast to northeast and have shown left-lateral and reverse motion. The four fault sets divide the area into a series of fault-bounded blocks.

Brittle extension is recorded in the Scud River area (Brown and Gunning, 1988). High angle felsic dykes define a Tertiary east-west extension, while north-south extension defined by east-striking mafic dyke swarms is post-Middle Jurassic but predates the Tertiary felsic dykes.

5.3.2 Local Geology

The Galore Creek intrusive-volcanic complex is composed of multiple intrusions emplaced into volcanic and sedimentary rocks of similar composition (Figure 5.3-2). Country rocks to the syenite intrusions are volcanic flows and volcanoclastic sediments, with subordinate greywacke, siltstone and local conglomerate (Enns et al, 1995). Augite-bearing volcanic flows and tuffs underlie and are interbedded with the pseudoleucite-bearing and orthoclase-bearing flows, tuffaceous and fragmental units, which are prominent in the south and southwest parts of the complex (Enns et al, 1995). Multiple alkali syenite intrusive phases occur in the complex and are divided into the pre- to syn-mineralization intrusives (I1 to I4), syn- to post-mineralization intrusives (I5 to I9) and post-mineralization intrusives (I10 to I12). The complex is centered in the west fork of Galore Creek and is approximately 5 kilometres in length and 2 kilometres in width. To date, twelve copper-gold-silver mineralized zones have been identified on the property. 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.

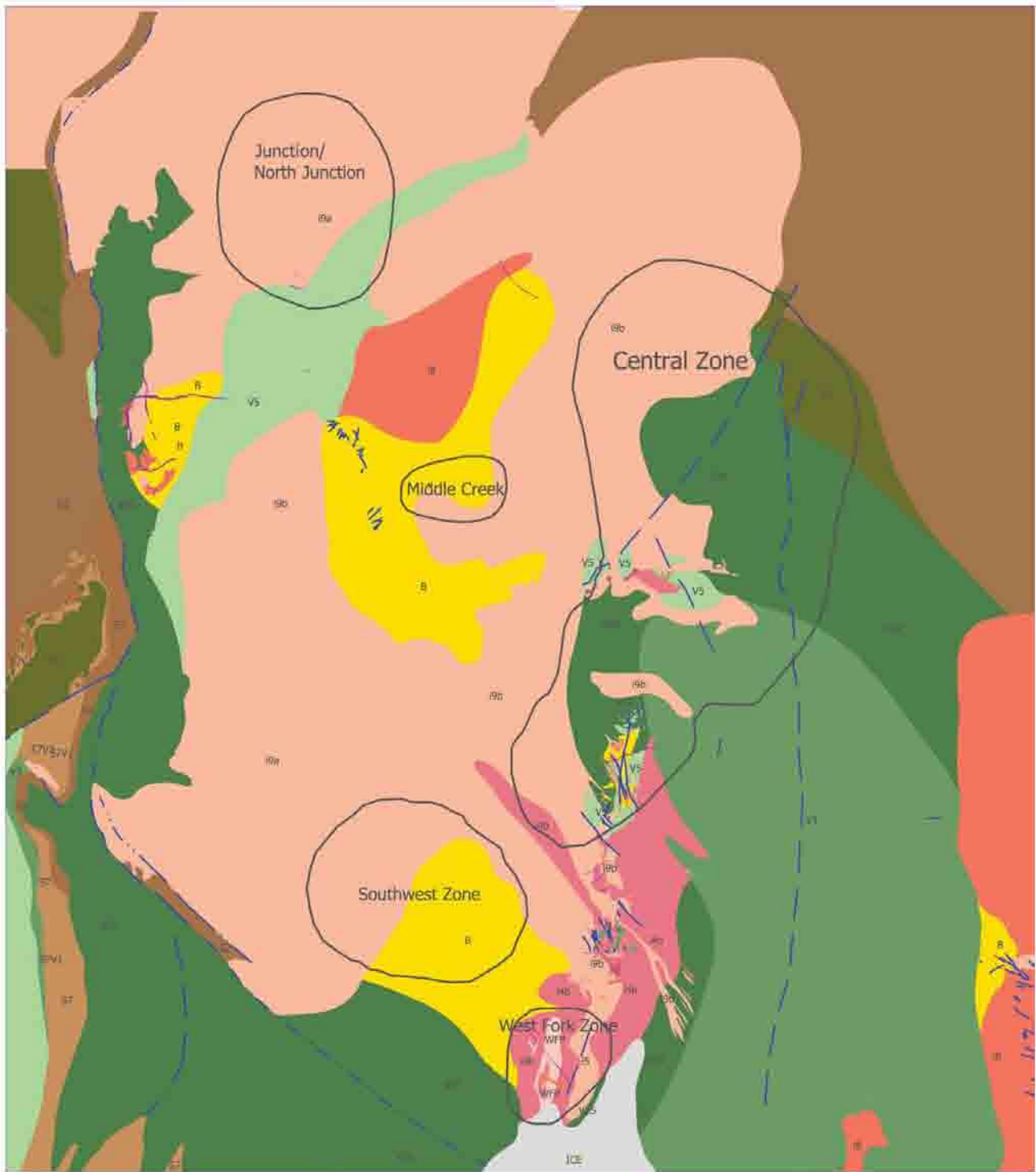
5.3.2.1 Volcanic Rocks

Volcanic rocks host the bulk of the mineralization at Galore Creek and have been subdivided into six units.

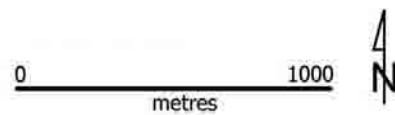
Augite-Bearing Volcanics (VI)

A heterogeneous sequence of augite-bearing mafic flows, flow breccias and volcanoclastics are interbedded with pseudoleucite volcanic rocks in the northern portion of the Central Zone. Rare bedding is preserved in lapilli tuffs of this unit and graded bedding is locally preserved. 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. Augite phenocrysts vary in size from 2-5 mm and are generally euhedral to subhedral, stubby and dark green to black. They comprise up to 30% of the rock and are supported in a medium to dark green, aphanitic groundmass. The augite phenocrysts are usually altered to biotite, epidote and chlorite. Locally, strong garnet-biotite-orthoclase alteration is also observed.



- | | | |
|--|---------------------------------|---|
| S2 | Greywacke | Post-Mineral Dykes |
| S7 | Diamictite | Faults |
| V1 | Augite-Bearing Volcanic | Mineralized Zone |
| V2 | Pseudoleucite-Bearing Volcanic | |
| V3 | Orthoclase-Bearing Volcanic | |
| V5 | Intermediate Volcanic | |
| I4 | Dark Orthoclase Syenite | |
| I5/I9 | Orthoclase Syenite Megaporphyry | |
| I8 | Medium Grained Syenite | |
| WFP | West Fork Porphyry | |
| B | Breccia | |



**Galore Creek Project
Generalized Geology Map**

FIGURE 5.3-2

Interbedded with the augite bearing flows are augite-bearing volcanoclastics 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 volcanoclastics are generally matrix supported.

Pseudoleucite-Bearing Volcanics (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/gold mineralization appears to occur preferentially in these rocks. In unaltered areas, euhedral and broken pseudoleucite phenocrysts up to 1.5 cm occur within a bluish grey to salmon pink groundmass. These phenocrysts often exhibit orthoclase-sericite altered cores. Rims are sometimes altered to sericite, magnetite and chlorite. In the Central Zone, fragments of pseudoleucite bearing volcanic rocks are present in mineralized hydrothermal breccias containing abundant garnet.

Orthoclase-Bearing Volcanics (V3)

Orthoclase-bearing volcanics 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 intrusives, which may be their subvolcanic equivalents. The crystal fragments in the tuffs are broken orthoclase shards up to 7 mm across supported by a highly altered biotite-orthoclase +/- garnet-anhydrite matrix. Rare bedding is preserved locally.

Undifferentiated Volcanics (V4, V5, V6)

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

Mafic volcanic rocks (V4) are dark green, chloritic flows and tuffs common in the north part of the Central Zone. These are interbedded, and may, in part, be correlated with unit V1 (Augite-Bearing Volcanics). 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 volcanoclastics and flows, and may be aphyric equivalents of the V2 (Pseudoleucite-Bearing Volcanic) unit. Included in this unit are possible trachy-andesites containing sub-rounded orthoclase aphyric fragments. Aphanitic volcanic clasts up to 3 cm across have also been observed within a fine-grained to aphanitic matrix. 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 volcanoclastics 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.

5.3.2.2 Sedimentary Rocks

Sedimentary rocks, such as diamictites and epiclastics, 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 right side up.

Conglomerate (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 are sub-rounded to rounded, matrix supported by sand and silt sized grains. 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 (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. This unit is locally very well bedded and graded. Fragments of argillite and volcanic material are sub-angular to sub-round.

Siltstone (S3)

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

Argillite (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. Local flame structures have been observed, particularly in North Rim Creek and in the 2070 Adit in the Central Zone, where it is interbedded with siltstone (S2).

Limestone (S5)

Micritic or crystalline limestone; includes all variations of grain size and bed thickness. Lithology is sedimentary in origin and should not be confused with overprinted carbonate alteration. Limestone is most commonly found in Copper Canyon.

Epiclastic Sediments (S6)

Composite lithology consisting primarily of reworked volcanic material; includes clay rich (lacustrine) beds, siltstone, fine-, to course-grained sandstone, and conglomerate. Lithology should show clear evidence of fluvial reworking such as planar or cross-bedding, sorting, normal or reverse-grading, etc.

Diamictite (S7)

Unsorted, mono- or poly lithic fragments that are matrix supported. The matrix consists of a mixture of clay, silt or sand. Lithology commonly shows either normal or reverse grading. Probably forms due to mass gravity flows such as lahars or debris flows.

5.3.2.3 Intrusive Rocks

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

Petrologic 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 (I1 through I5) consist of K-feldspar and pseudoleucite porphyritic dikes 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 fugacity's 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 (Units I1 to I3)

Pseudoleucite porphyries and megaporphyries (I1 & I2) are relatively rare, and occur most often as thin, steeply dipping dikes in the Central Zone. Distinct chill margins are often observed at contacts. Pseudoleucite porphyry (I1) is identified based on the presence of 10-30%, 4-10 mm and more rarely 10-20 mm euhedral pseudoleucite phenocrysts set in a pale-grey to pinkish-grey, aphanitic, orthoclase-rich groundmass. Pseudoleucite megaporphyry (I2) is identified based on the presence of 3-10%, 20-40 mm, subhedral to euhedral pseudoleucite megacrysts, as well as 3-5%, 1-3 mm, tabular orthoclase phenocrysts. All phenocrysts are set in a fine-grained, slate-grey groundmass.

Early work describes a Grey Syenite Porphyry (I3), also shown on historic drill sections as "Dark Syenite Porphyry". Work by NovaGold from 2003-2005 did not substantiate the existence of this unit.

Inter-Mineralization Intrusions (Units i4a & 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. The unit is easily identified based on the size, shape, and distribution of the orthoclase phenocrysts, and if present pseudoleucite megacrysts. Dark orthoclase syenite is commonly altered to biotite and

chlorite after orthoclase, which gives the groundmass a dark grey appearance. Orthoclase alteration after biotite can change the colour of the groundmass back to a pale grey colour.

Early dark orthoclase syenite (I4a) is a porphyritic unit with 3-7%, 2-5 mm and 10-20 mm, subhedral to rounded orthoclase phenocrysts set in a dark grey to pale brown/pink, fine-grained groundmass. Mineralization exists as disseminated and veined bornite and chalcopyrite. Early dark orthoclase syenite may be the subvolcanic equivalent of the V3 (Orthoclase-Bearing Volcanic) unit.

Late dark orthoclase syenite (I4b) is commonly observed in outcrop and drill core as irregular to tabular east-dipping dikes. It is dark grey-green, porphyritic, with infrequent large, zoned, euhedral pseudoleucite phenocrysts, 20-40 mm in diameter. Orthoclase phenocrysts 3-15 mm in diameter comprise 10-40% of the unit, and are supported by a fine-grained orthoclase matrix. The major alteration assemblage is biotite and chlorite after orthoclase.

Late-Mineralization Porphyries (Units I5 to I12)

Dikes and/or sills of late-mineralization porphyries intrude the Central and Junction/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), megaporphyry (I9a & I9b), grey medium grained syenite porphyry (I11a) and lavender porphyry (I12). Minor phases include dikes and small stocks of fine-grained syenite and syenite porphyry (I6 & I7), plagioclase syenite porphyry (I10) and medium-grained syenite porphyry (I11). Most of these late minor phases were defined from drill core in the Central Zone. Minor mineralization associated locally with these dikes suggests that they are either syn-mineral, or minor late mineralization was introduced with them. The two varieties of I9 (I9a and I9b) are very similar chemically and texturally, however the dominant occurrence of I9a is located in the Southwest Zone and is locally well mineralized. The I9b in the Central Zone is clearly post-mineral which suggests different intrusive timing for the two phases relative to mineralization which act as a basis for separate categorization. The lavender porphyry (I12) is a quartz bearing syenite that occurs as a small body near the centre of the complex. It intrudes a late-mineral breccia exposed in Dry Creek and is locally mineralized with chalcopyrite.

5.3.2.4 Breccias (B1, B2, & B3)

Diatreme (B1), Hydrothermal (B2), and Orthomagmatic (B3) Breccia's at Galore Creek are distinguished mainly by clast shape and lithology, matrix composition and packaging, alteration assemblage and mineralization presence. Diatreme clasts are rounded to sub-angular and form lapilli-sized fragments to fragments several tens of centimetres across. Clasts are generally orthoclase altered and sit in a matrix of sand and silt sized particles. Hydrothermal breccias are characterized by sub-angular, rotated clasts of pseudoleucite porphyry (I1 & I2), grey syenite porphyry (I3), and mafic to intermediate volcanics (V4 & V5). In most cases the breccias are matrix supported, with the primary composition being garnet, anhydrite, orthoclase, biotite ± diopside. Hydrothermal breccias are often well mineralized, with the primary copper mineral, chalcopyrite, occurring as disseminations and stringers. Orthomagmatic breccia clasts are multi-lithic, unsorted, and rounded to angular. They are set in a magmatic, often porphyritic, matrix.

The Southwest Zone is hosted by an unbedded, polyolithic, matrix-supported breccia. Clasts include a variety of altered and/or mineralized lithologies, which indicates that a mineralization event preceded brecciation. The breccia matrix is rich in rock flour but also appears to contain comminuted crystals of possible tuffaceous origin.

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. The breccias are mainly clast supported and characterized by a matrix of magnetite accompanied by one or more of epidote, garnet, pyroxene and biotite.

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.

Post Mineral Dikes

Fine-grained, mafic to felsic post mineral dikes occur mainly as steep to vertical, north-south and east-west trending bodies. Rarely, they have sub-horizontal attitude. The ages of these dikes are undetermined but thought to be related to the Coast intrusions of mid-Mesozoic to Tertiary age.

5.3.3 Mineralization

Mineralization at Galore Creek occurs in upper Triassic, felsic to intermediate, volcanic flows and fragmental rocks, as well as breccias and pre to syn-mineralization intrusives. Chalcopyrite and locally abundant bornite mineralization occur as replacements, disseminations, and weaker fracture-fillings associated with intense, pervasive potassic alteration. Higher gold values are normally associated with bornite mineralization.

A review of copper solubility above the oxide surface was conducted in the winter of 2005. Results from the study found that oxide mineralization, in the form of minor chalcocite, malachite and azurite, is present in several zones as fracture coatings, weak replacement and locally pervasive. Deposit wide, oxidation depth is variable and commonly occurs in conjunction with sulphides. It exists to an average depth of 20 m in the Central Zone and to greater depths in the Middle Creek area. Minor oxidation is present in the Southwest Zone whereas the West Fork and Junction areas display no oxide mineralization. Testwork has shown that the average solubility of all samples taken above the oxide surface from the Southwest, Middle Creek, and Central Zones, with grades $\geq 0.35\%$ Cu, is 27.7% soluble.

5.3.3.1 Central Zone

The Central Zone is the largest and most extensively explored of all the deposits and is characterized by complex geology (Figure 5.3-3). Mineralization is exposed in the southern end of the zone, but elsewhere it is covered by up to 75 m of glacial overburden. Between 80% and 90% of the copper-gold-silver occurs as sulphide replacement of the host volcanic rocks. The grade of the mineralization commonly exceeds 1% Cu, decreasing rapidly at the margin of the zone.

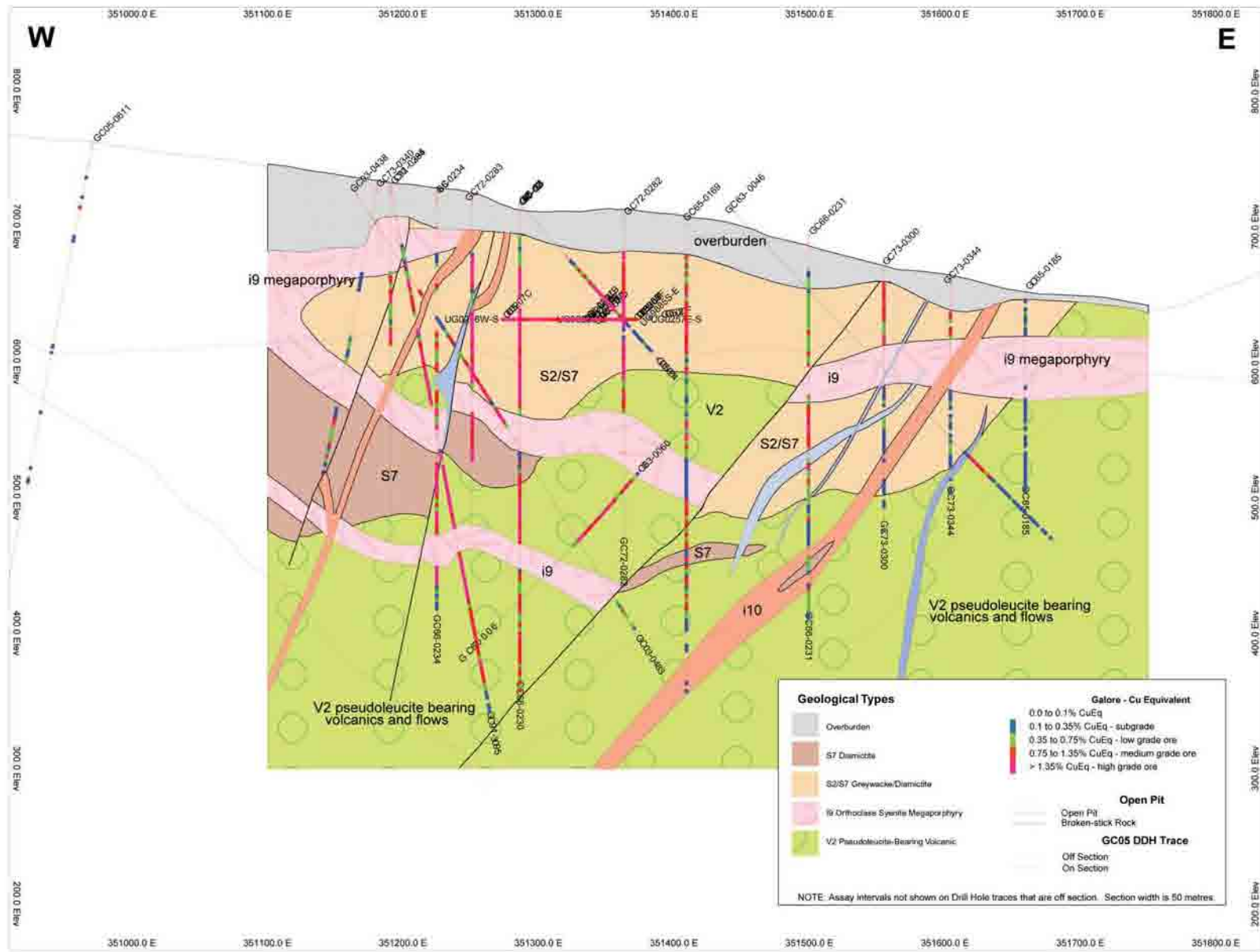


FIGURE 5.3-3

Cross Section of Central Replacement Zone

The Central Zone deposit has an orientation of 015 degrees and dips 25 - 30 degrees to the west. It is 1,700 m long, 200 to 500 m wide and has been traced to a depth of 450 m. Mineralization remains open at depth. The eastern boundary of Central Zone mineralization is associated, but not limited by, a steeply west dipping, brittle normal fault. Post-mineral megaporphyry dikes (I9b) locally truncate mineralization to the west and the south of the Central Zone. In the north, mineralized volcanic rocks terminate against a thick sequence of weakly to unmineralized epiclastic sedimentary and augite-bearing volcanic rocks. Either the East Fork or North Boundary Fault may displace mineralization to the northwest of its current trend.

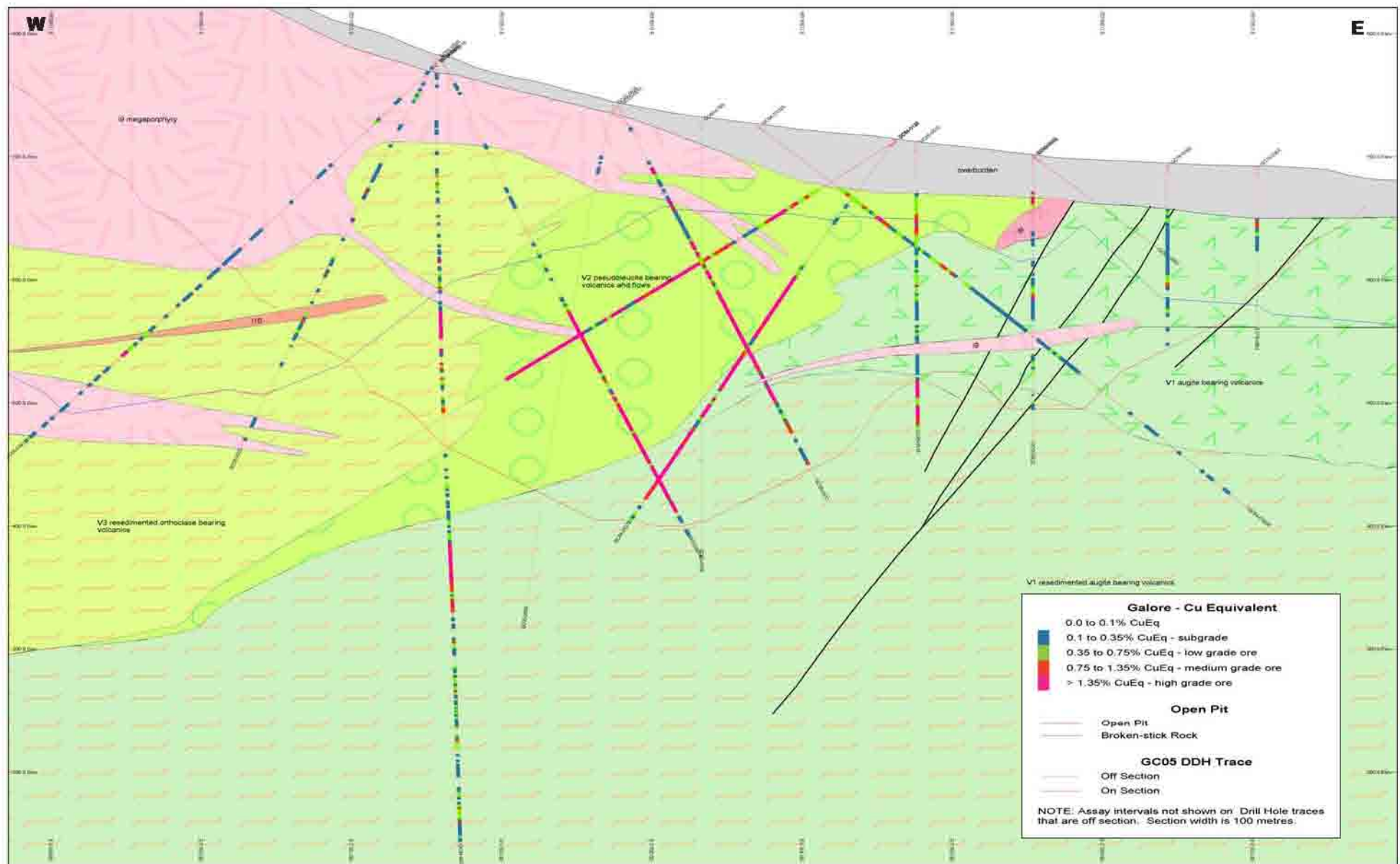
The Central Zone exhibits considerable internal variations in both mineralization and alteration. Hydrothermal alteration changes from Ca-K-silicate in the core region to intense K-silicate alteration toward the north and south ends of the zone. In terms of copper-gold-silver replacement mineralization, the most favourable volcanic lithologies are the pseudoleucite-bearing volcanic rocks in the north and the dark crystal tuffs in the south. Augite-bearing units in the north are low to moderate in copper content and the core of the deposit hosts a mineralized orthomagmatic breccia. Gold values are highest in the northern and southern portions of the Central Zone where significant disseminated bornite, magnetite and hematite are present (Figures 5.3-4 and 5.3-5). Lower gold grades correlate with the intense Ca-K-Silicate (garnet) altered core region. Chalcopyrite is the most important copper mineral and occurs as replacements, disseminations and fracture fillings throughout the zone. Supergene copper mineralization is minor and occurs primarily as malachite, azurite and chrysocolla on fractures within 20 metres of surface. Pyrite increases in abundance to the east of the Central Zone reaching concentrations of up to 5%.

5.3.3.2 Southwest Zone

The Southwest Zone is located about 600 metres southwest of the south end of the Central Zone. The Southwest Zone contains some of the highest grade near surface gold mineralization found on the Galore Creek property (Figure 5.3-6). Drilling has outlined an elongate tabular shaped body that trends roughly east-west and dips approximately 60 degrees to the south. The zone is up to 400 metres long and may be as wide as 140 metres; additionally, the 2005 drilling suggests that the zone remains open along strike and at depth. Primary hosts for the Southwest mineralization are breccia and the early mineralized variety of orthoclase syenite megaporphyry (i9a) different than that found as clearly post-mineral in the Central Zone. Located primarily on the footwall side of the Southwest Fault, the deposit is zoned from a central gold-copper core out to a gold halo.

5.3.3.3 Junction Zones

The Junction and North Junction deposits lie about two kilometres northwest of the Central Zone and about 460 metres higher in elevation. The Junction deposit is a tabular north-east striking, north-west dipping body and the North Junction deposit is podiform with the long axis plunging to the north-west. The mineralization, consisting of disseminated chalcopyrite and bornite, is hosted in both the Junction porphyry (JP) and Late Junction porphyry and orthoclase syenite megaporphyry (i5) (Figure 5.3-7). Higher gold and copper grades correlate with the presence of bornite in the North Junction Zone.



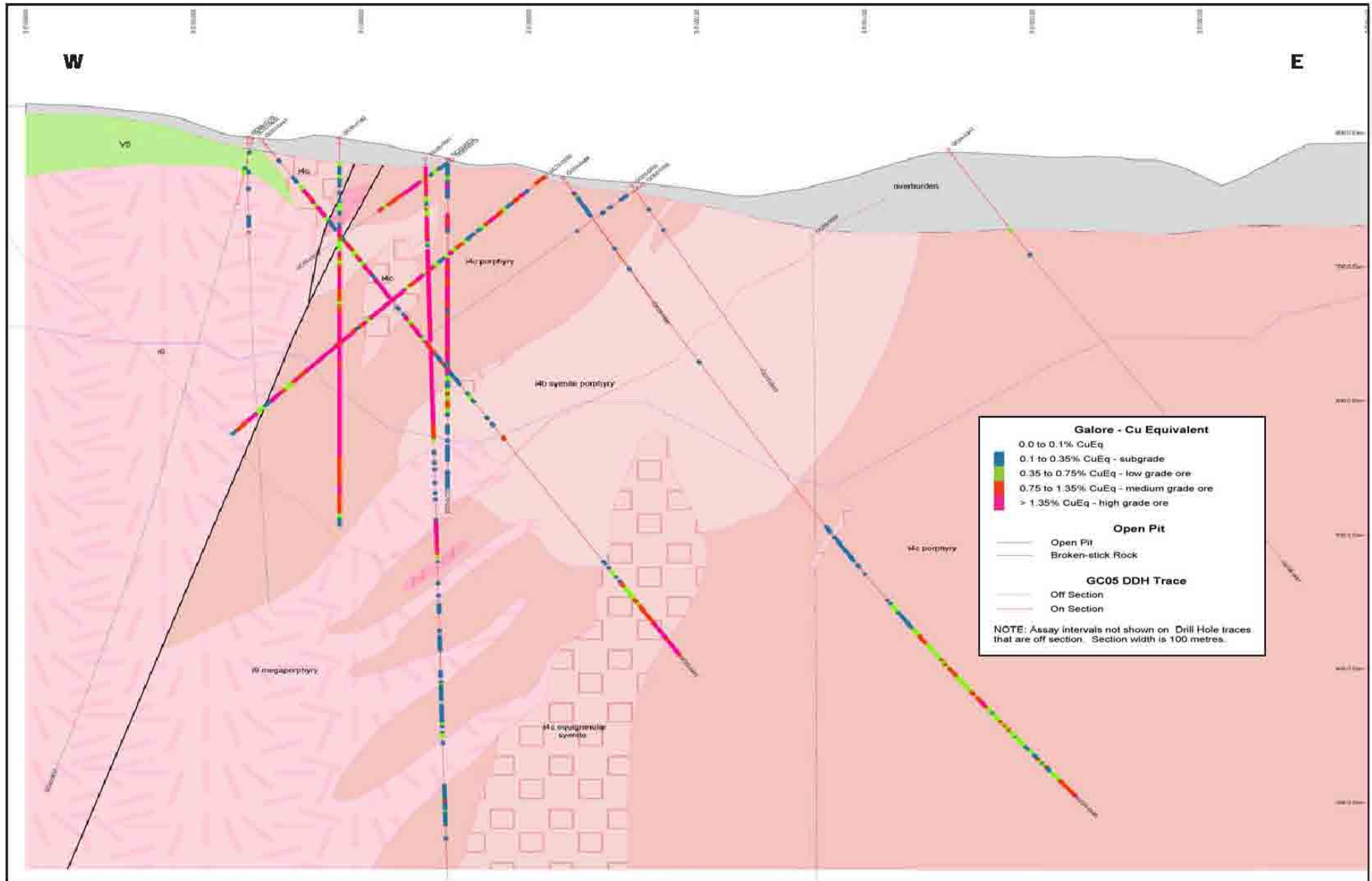
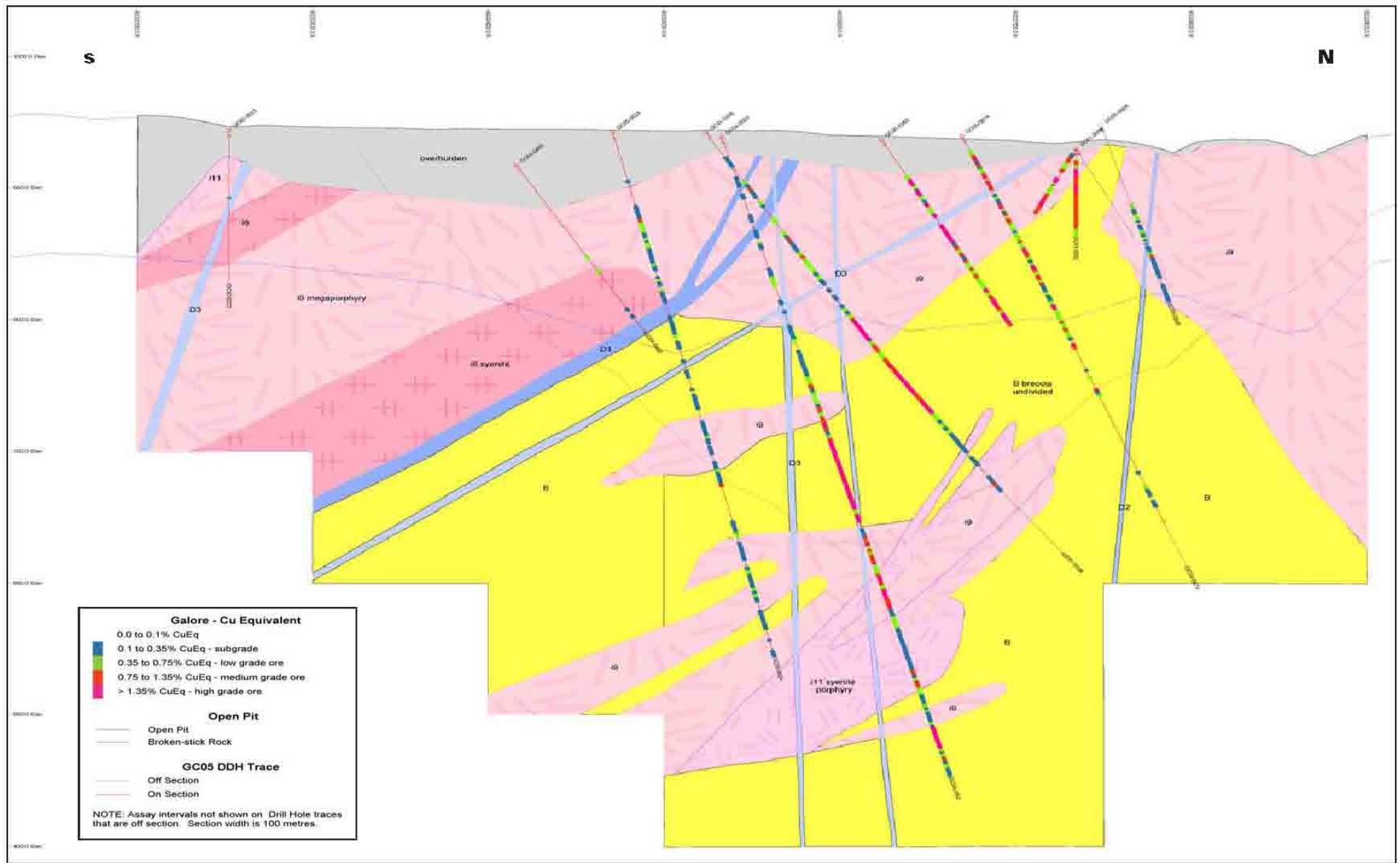
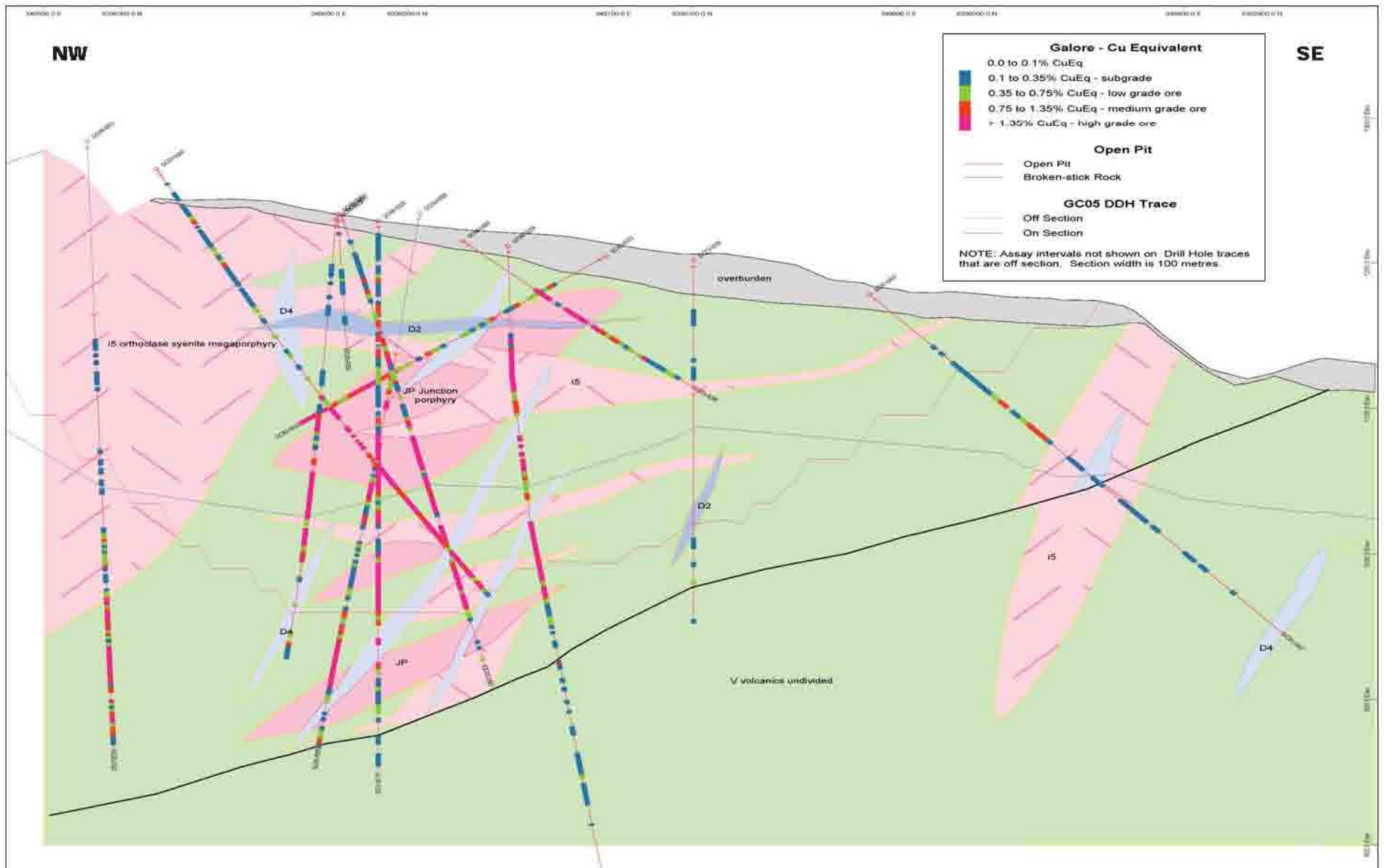


FIGURE 5.3-5

Cross Section of South Gold Lens, Central Zone





Cross Section of Junction Zone

FIGURE 5.3-7

5.3.3.4 West Fork Zone

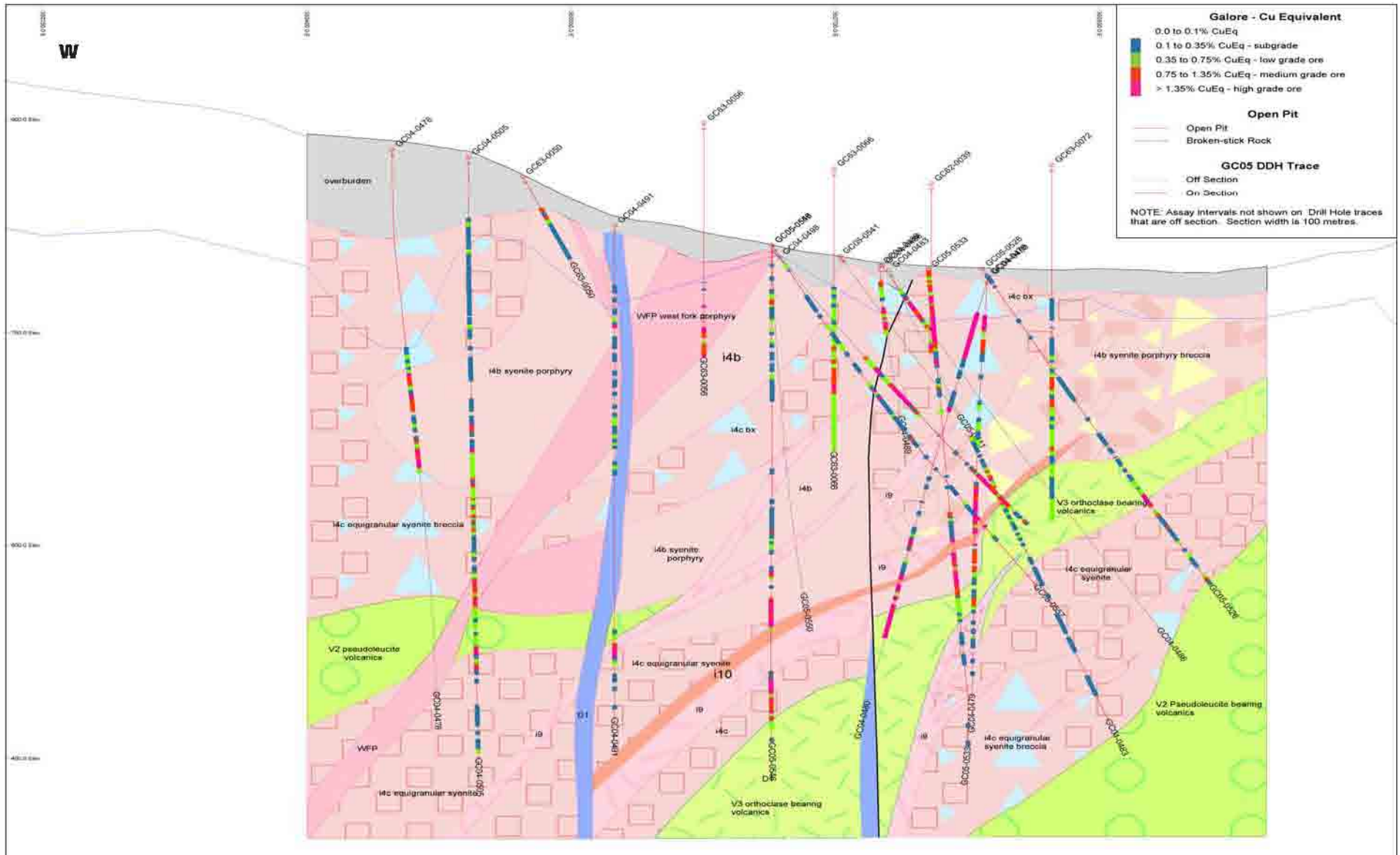
The West Fork deposit is located less than one kilometre south of the Central Zone. There are four distinct zones of mineralization, the Opulent Vein, the Upper Opulent Zone, the Lower Opulent Zone, and the Lower West Fork Zone. The Opulent Vein is north-south-trending and steeply west-dipping (Figure 5.3-8). The extremely high-grade mineralization is defined by massive magnetite-bornite-chalcopyrite 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. The Lower West Fork Zone is characterized by disseminated chalcopyrite and bornite replacement which cross-cuts multiple lithologies. This lower zone is east-west striking and moderately north-dipping.

5.3.3.5 Other Zones

Limited exploration work has occurred at a number of smaller copper-gold-silver showings in the claim group. The Butte and West Rim deposits lie along the western margin of the Galore Creek complex. Both host northeast-striking, west-dipping zones of disseminated chalcopyrite and bornite. At the Butte zone, mineralization is hosted by K-silicate altered pseudoleucite-bearing volcanics. At surface, mineralization grades up to 2.18% copper and 0.5 grams gold per tonne across 96.6 metres. The zone is truncated at depth by post-mineral (III) intrusions. Surface mapping in the 1960's revealed outcrop mineralization in the Lower Butte area. Drilling by NovaGold in 2005 intercepted a zone of 0.729% copper across 84 metres, making this new discovery a target for future exploration.

Drilling in 1964, on the West Rim showing, intersected 48.8 metres grading 0.8% copper. This zone is hosted by volcanic tuffs with intense K-silicate alteration and is truncated at depth by I9b megaporphyry intrusions. Between 1963 and 1991, ten holes were drilled in the North Rim Zone located about 1600 metres north-northwest from the Central Zone. Here, a broad area of widespread weak copper mineralization was discovered in outcrop and subsequent exploration revealed coincident copper-gold geochemical and chargeability anomalies. Two holes intersected zones of low grade copper (0.3-0.4%) over widths of 21-24 metres with accompanying gold grades up to 0.5 grams per tonne. The last two holes intersected many narrower (<10 m) intervals grading 0.3-1.0 grams gold per tonne and one as high as 6.34 grams per tonne over three metres. Spotty copper grades above 0.3% were also present. The area is underlain mainly by intrusive phases cut by small orthomagmatic breccia bodies with magnetite-rich matrix and Ca-K-Silicate alteration.

Middle Creek is located approximately one kilometre west of the Central Zone. In 1991, field mapping found mineralization reported by prospectors in the mid 1960's. Mineralization is characterized by finely disseminated bornite, chalcopyrite and magnetite associated with pervasive fine-grained biotite and garnet alteration, hosted in a breccia or volcanoclastic unit. The mineralized zone is open in all directions. In 2005, drilling in the Middle Creek area encountered a hole which showed significant grade (~1.1% Cu) over a 50 metre intercept. The grade was attributed to the presence of malachite and native copper and prompted a copper oxide and copper solubility study. The results of this study found Middle Creek to be the most oxidized zone discovered to date on the Galore Creek property.



The Saddle Zone lies 2.6 kilometres southeast of the Central Zone and is hosted by a breccia dominated by a matrix of magnetite. Four holes were drilled here between 1954 and 1963 and two more added in 1990. The later holes intersected two 12 metre zones of mineralization grading 0.55% copper and 2.19 grams gold per tonne, and 2.49% copper and 3.98 grams gold per tonne, respectively. Three of the older holes also intersected low to moderate copper and high gold grades (between 1.0 and 6.9 g/t Au) but gold analyses were incomplete. An adjacent showing named the South 110 zone is possibly a continuation of the Saddle Zone to the north.

5.3.4 Alteration

5.3.4.1 Potassic Alteration Associated with Cu-sulphide Introduction

Widespread potassic alteration is dominant at Galore Creek and is associated with bornite and chalcopyrite mineralization. K-feldspar flooding, most intense in the Central Zone, affects the volcanics and early intrusives 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 likely occurred when the potassic “K-silicate” altering fluids encountered more calcic mafic rocks, derived excess calcium and precipitated garnet. Garnet alteration decreases from the core region of the Central Zone to the north and south ends 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.

5.3.4.2 Later Potassic Alteration

In the later porphyries, such as I8 and I11a, potassic alteration is usually present, but is less intense than that described above. 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. Mafics 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).

5.3.4.3 Sericite-Anhydrite-Carbonate (SAC) Alteration

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 metres below surface. The gypsum has been leached out to depths ranging from 30 to 122 metres 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 sulphidized to chalcopyrite (Proffett, 2005).

5.3.4.4 Propylitic Alteration

Propylitic alteration, typical of prograde hydrothermal porphyry copper alteration, exists outside of the main deposit area. This epidote-chlorite-pyrite assemblage is strongly present in drill core from the Grace Claims property. The assemblage crosscuts all rock types. The highest percentages of pyrite occur in the propylitic halos that are distal to the main body of mineralization. Rare elevated gold values are associated with pyrite.

5.3.5 Structure

The Galore Creek intrusive complex is believed to represent an eroded volcanic centre. Syenite sheets are subvolcanic intrusions, the extrusive equivalents of which may be the orthoclase-bearing tuffs and pseudoleucite lavas. The two main structural elements are faults, which offset and segment intrusive bodies, and a subhorizontal fracture cleavage.

In general, faults in the Central Zone exhibit little displacement. An exception is the East Fault, a major post-mineralization west-dipping fault located east of the Central Zone. Historically the East Fault was thought to have a normal displacement of up to 300 metres; however, it is now believed to exhibit a left lateral displacement on the order of 10's of metres. Locally, late I9-megaporphyry intrusive units appear to cut the East Fault without significant displacement. Substantial displacement may occur on faults located in the northern end of the Central Zone, however correlating marker beds and dikes between drill holes proves difficult. A north-trending mylonite zone, at least 100 m in thickness, is exposed in volcanic rocks along the western margin of the Butte area intrusive complex. The zone dips 42 degrees to the west and predates mineralization and alteration in upper Butte Creek. A major east-west regional structure, marked by a prominent aeromagnetic linear, lies parallel to lower Dendritic Creek. This feature cuts through the Central Zone where it is occupied by a late-mineral I9b-megaporphyry dike. The long axis of the breccia in lower Dendritic Creek is aligned parallel to this trend, suggesting a common regional structural control.

Described above as SAC alteration, the hydration of anhydrite to gypsum by meteoric waters results in sub-horizontal fracture cleavage, also described as "sheet fracturing." The fracture

cleavage is well developed in the Central Zone where it reaches depths down to 213 metres below surface.

5.3.6 Metal Leaching and Acid Rock Drainage

The metal leaching and acid rock drainage (ML/ARD) potential of the rocks in the vicinity of the Galore Creek mine were assessed by SRK Consulting. The full SRK report is included in Appendix 5-A, *Galore Creek Project ML/ARD Characterization Report*. This section provides a summary of key information from that assessment.

5.3.6.1 Geological Summary

The Galore Creek deposit is hosted by late Triassic to Early Jurassic syenitic bodies intruded into volcanic flows, volcanoclastic sediments and lesser greywacke, siltstone and local conglomerate of the Upper Triassic Stuhini Group. The bulk of the host rock is made up of broadly silica under-saturated syenite-type volcanic and intrusive rocks with limited range in silicate mineralogy. Section 5.3 contains a detailed description of regional and property geology; the following discussion is limited to geological features that control the metal leaching and acid rock drainage (ML/ARD) characteristics of the Galore Creek deposits and surrounding rocks.

Twelve syenitic intrusive rock units have been identified in drill core, and these have been consolidated into three groups based on timing of emplacement relative to the main mineralizing event, as described in Section 5.3.2.3. These groups of rock units were used in the evaluation of the ML/ARD characteristics of the Galore Creek syenitic intrusives.

Six volcanic rock units (V1 through V6) have been identified in drill core, as described in Section 5.3.2.1. Evaluation of ML/ARD characteristics of the volcanic units was carried out by grouping units that are considered to be porphyritic and aphanitic phases of the same rock type. Unit V1 and V4 were considered as a group, as were Units V2 and V5. Units V3 and V6 appear to be unrelated, and each was evaluated independently.

Other rock units identified in outcrop and/ or drill core include a number of breccia units, unmineralized dykes, and several minor, laterally-discontinuous sedimentary units. These units were grouped by lithology and ML/ARD characteristics were evaluated for each of the breccia, dyke, and sedimentary groups.

Mineralogy

The sulphide mineralogy at Galore Creek is relatively simple. The dominant sulphide mineral is pyrite followed by chalcopyrite and bornite. Pyrite concentrations are greatest along the eastern edge of the Central Zone where concentrations rise to 5%. Sphalerite and galena are present but in trace amounts. The sulphide minerals occur primarily as disseminated replacements of the primary silicate and oxide mineralogy though veins and veinlets of all these minerals occur locally.

Carbonate minerals (primarily calcite) occur mainly as a component of the matrix of the rock as alterations of calc-silicate minerals, but also locally as veinlets.

Carbonate Mineralogy

Carbonate mineralogy of Central and Junction zones was evaluated through optical mineralogy and microprobe analyses. The vast majority of carbonate grains analyzed were found to be calcite containing up to 10% iron; several grains from both Junction and Central Zones were identified as dolomite, and isolated grains of rhodochrosite, ankerite, and siderite were also identified in Central Zone samples.

5.3.6.2 Static Geochemical Characteristics

Occurrence of Sulphur

The widespread distribution and high proportions of primary sulphate minerals in Galore Creek rocks cause total sulphur content to be a poor predictor of the acid-generating potential of the various rock units. Sulphate content determined during acid-base account testing provides a measure of the relative distribution of sulphur between the sulphate and sulphide forms, and allows estimates of acid potential to be calculated from sulphide sulphur alone. Acid-base accounting characteristics for the various rock units for the Central-Southwest-West Fork Zone and for the Junction Zone are discussed below.

Central, Southwest, and West Fork Zones

Pre-mineralization intrusives (Units I1, I2, and I3)

The pre-mineralization intrusions (units I1 to I3) had a weighted average total sulphur content of 1.9% (range 0.12 to 4.9%) and a weighted average sulphide-sulphur content of 0.69% (range 0.08 to 3.9%) (Table 5.3-1). Sulphur in chalcopyrite was calculated from the weighted average copper concentration to be 0.1%, and the remaining 0.59% of sulphide sulphur was attributed to pyrite. Both sphalerite and galena have been identified in core and in thin section. However, low zinc and lead concentrations in the pre-mineralization and syn-mineralization intrusions indicate that these phases are insignificant sulphur hosts although are potential sources of soluble zinc and lead.

Syn-mineralization intrusive (Unit I4)

The 88-sample weighted average sulphide sulphur of I4 is 0.72% (range 0.06 to 2.7%), and the weighted average sulphate content is 1.0% (range 0.04 to 2.5%), for a total sulphur weighted average of 1.7% (range 0.1 to 5.1%) (Table 5.3-1). Pyrite, chalcopyrite, and bornite have been observed in I4 drill core as disseminations and veins. Based on the weighted average copper concentration from assays, 0.11% of sulphide sulphur in Unit I4 occurs as chalcopyrite. On the basis of total sulphide sulphur less chalcopyrite sulphur, the weighted average concentration of pyritic sulphur in I4 is estimated to be 0.61%. Sulphate sulphur is present as gypsum and anhydrite, which are expected to have a combined weighted average abundance of about 5%, based on the weighted average of sulphate from ABA testing.

**Table 5.3-1
Summary of Acid-Base Accounting Characteristics for
Central-Southwest-West Fork Intrusives**

Rock Group	Summary Function	Paste pH	CO ₂ (Wt. %)	CO ₂ (Kg CaCO ₃ / Tonne)	Total Sulphur (Wt. %)	Sulphate Sulphur (Wt. %)	Sulphide Sulphur* (Wt. %)	IC _(Ca, Mg) (NP-10) NNP					
								AP (Kg CaCO ₃ /Tonne)	NP	NP/AP	IC _(Ca, Mg)	NNP	
I1+I2+I3	Number	28	28	28	28	28	28	28	28	28	28	28	28
	Weighted Average	8.5	1.46	33.12	1.90	1.21	0.69	21.53	40.8	30.78	19.2	4.56	3.56
	Min	7.7	0.05	1.14	0.12	0.03	0.08	2.50	16.5	6.50	-70.5	0.42	0.19
	5th Percentile	8.0	0.21	4.82	0.26	0.07	0.10	3.14	19.0	8.97	-32.8	0.50	0.27
	Median	8.4	1.35	30.57	1.61	1.10	0.49	15.16	37.7	27.70	16.0	2.07	1.55
	95th Percentile	9.2	2.93	66.57	3.50	2.56	2.41	75.17	65.5	55.46	56.0	17.52	14.04
	Max	9.3	3.32	75.45	4.88	2.72	3.92	122.50	100.1	90.10	94.8	19.40	16.96
I4	Number	18	18	18	18	18	18	18	18	18	18	18	18
	Weighted Average	8.5	1.23	27.86	1.61	0.90	0.71	22.10	35.4	25.42	13.3	6.13	4.75
	Min	8.1	0.11	2.50	0.10	0.04	0.06	1.88	8.0	-2.00	-59.5	0.29	-0.34
	5th Percentile	8.3	0.18	4.05	0.10	0.04	0.06	1.88	8.0	-2.00	-42.7	0.34	-0.20
	Median	8.5	0.90	20.45	1.34	0.64	0.48	14.84	33.0	23.00	10.2	1.51	1.00
	95th Percentile	9.1	2.79	63.40	4.31	2.42	2.17	67.77	67.9	57.88	64.7	29.78	24.44
	Max	9.2	3.41	77.50	5.13	2.46	2.67	83.44	87.0	77.00	76.7	34.40	29.07
I5 to I11	Number	96	96	96	96	96	96	96	96	96	96	96	96
	Weighted Average	8.4	1.67	37.97	1.06	0.49	0.57	17.68	45.5	35.50	27.8	6.00	4.56
	Min	7.5	0.22	5.00	0.07	0.01	0.04	1.25	15.3	5.30	-69.0	0.25	0.14
	5th Percentile	8.0	0.43	9.72	0.13	0.02	0.08	2.50	17.8	7.78	-19.4	0.62	0.39
	Median	8.5	1.51	34.20	0.80	0.22	0.35	10.94	41.6	31.60	30.2	3.79	3.13
	95th Percentile	9.0	3.55	80.57	2.91	2.22	1.79	55.86	82.2	72.15	65.5	18.76	14.25
	Max	9.2	4.39	99.77	6.22	3.11	3.21	100.31	171.4	161.40	133.3	26.20	22.65

* - Sulphide sulphur calculated as Total Sulphur – Sulphate Sulphur
 AP: acid potential, calculated from sulphide sulphur
 NP: neutralization potential (determined by the modified Sobek method)
 IC_(Ca, Mg) calculated as NP – 10 kg CaCO₃ equiv/tonne
 NNP: net neutralization potential (NP-AP)
 NP/AP: neutralization potential ratio
 IC_(Ca, Mg)/AP: carbonate neutralization potential ratio

Late-mineralization intrusives (Units I5 through I12)

The late-mineralization intrusions (units I5 to I12) had a weighted average sulphate sulphur content of 0.6% (range 0.01 to 3.1%) and a weighted average sulphide-sulphur content of 0.62% (range 0.04 to 3.2%), for a weighted average total sulphur content of 1.2% (range 0.07 to 6.2%) (Table 5.3-1).

The weighted average sulphur in chalcopyrite was estimated to be 0.14%, as calculated from copper assays on ABA intervals. The weighted average of pyritic sulphur is estimated to be 0.48%, based on the difference between total and sulphate sulphur, and sulphide sulphur in excess of stoichiometric quantities required to form chalcopyrite.

Volcanic Units

The volcanic units that host the deposit (variations of Units V1 through V6) had a weighted average sulphate sulphur content of 1.35% (range 0.02 to 4.1%) and a weighted average sulphide sulphur content of 1.33% (range 0.02 to 6.2%), which combined for a weighted average total sulphur content of 2.68% (range 0.09 to 8.4%) (Table 5.3-2). Anhydrite and gypsum were logged in drill core, and are considered to be the only significant sulphate phases. Sulphide phases observed in drill core were pyrite, chalcopyrite, and bornite. Zinc and lead concentrations

from assays suggest that trace sphalerite and galena are also present but that these are not proportionally significant sulphide sulphur phases.

**Table 5.3-2
Summary of Acid-Base Accounting Characteristics for
Central-Southwest-West Fork Volcanics**

Rock Group	Summary Function	Paste pH	CO ₂ (Wt. %)	CO ₂ (Kg CaCO ₃ / Tonne)	Total Sulphur (Wt. %)	Sulphate Sulphur (Wt. %)	Sulphide Sulphur* (Wt. %)	IC _(Ca, Mg)		NPP	NP/AP	IC _{(Ca, Mg)/AP}	
								AP	NP				
								(Kg CaCO ₃ /Tonne)					
V1 to V6	Number	131	131	131	131	131	131	131	131	131	131	131	131
	Weighted Average	8.4	1.26	28.53	2.59	1.30	1.29	37.4	37.4	27.37	-2.80	3.27	2.42
	Min	7.3	0.18	4.09	0.09	0.02	0.02	12.0	12.0	2.00	-154.59	0.08	0.02
	5th Percentile	7.9	0.26	5.80	0.19	0.04	0.12	15.6	15.6	5.60	-79.23	0.23	0.08
	Median	8.4	1.05	23.86	2.59	1.26	0.76	34.6	34.6	24.60	9.40	1.37	0.97
	95th Percentile	9.0	3.10	70.34	5.52	2.96	3.42	71.9	71.9	61.90	56.52	13.02	9.95
	Max	9.6	3.80	86.36	8.44	4.11	6.18	100.6	100.6	90.60	78.29	35.57	30.99
V1 + V4	Number	13	13	13	13	13	13	13	13	13	13	13	13
	Weighted Average	8.8	2.23	50.62	1.91	1.39	0.53	61.0	61.0	50.99	44.58	5.96	5.09
	Min	8.1	0.22	5.00	0.41	0.07	0.17	13.5	13.5	3.50	-40.25	0.25	0.07
	5th Percentile	8.2	0.24	5.41	0.53	0.15	0.21	15.3	15.3	5.30	-21.31	0.51	0.21
	Median	8.8	0.92	20.91	1.54	1.09	0.51	30.3	30.3	20.30	19.68	1.80	1.50
	95th Percentile	9.4	5.04	114.50	4.03	2.82	1.33	114.2	114.2	104.20	103.85	13.28	11.74
	Max	9.6	8.35	189.77	4.73	3.01	1.72	183.2	183.2	173.20	173.51	18.91	17.88
V2 + V5	Number	90	90	90	90	90	90	90	90	90	90	90	30
	Weighted Average	8.4	1.32	30.07	2.67	1.39	1.28	39.5	39.5	29.49	-0.46	3.38	2.60
	Min	7.7	0.20	4.55	0.09	0.02	0.06	12.4	12.4	2.40	-140.99	0.08	0.02
	5th Percentile	8.0	0.27	6.14	0.17	0.04	0.09	16.0	16.0	5.98	-68.18	0.24	0.11
	Median	8.4	1.07	24.32	2.60	1.53	0.71	37.0	37.0	26.95	11.14	1.43	1.06
	95th Percentile	9.0	3.36	76.27	5.79	2.90	3.30	78.3	78.3	68.30	62.38	14.88	10.92
	Max	9.3	3.80	86.36	7.30	3.31	6.18	100.6	100.6	90.60	78.29	35.57	30.99
V3	Number	30	30	30	30	30	30	30	30	30	30	30	30
	Weighted Average	8.26	1.03	23.43	2.58	1.20	1.38	31.6	31.6	21.58	-11.67	3.04	1.94
	Min	7.30	0.18	4.09	0.20	0.03	0.02	12.0	12.0	2.00	-154.59	0.12	0.02
	5th Percentile	7.75	0.23	5.16	0.38	0.06	0.16	16.2	16.2	6.21	-86.66	0.14	0.05
	Median	8.20	1.00	22.61	2.92	0.86	0.94	32.1	32.1	22.05	-0.84	0.96	0.72
	95th Percentile	8.81	1.96	44.50	4.73	2.92	3.64	50.3	50.3	40.25	36.72	8.88	7.02
	Max	9.00	2.05	46.59	8.44	4.11	5.59	61.9	61.9	51.90	43.60	35.00	18.33
V6	Number	6	6	6	6	6	6	6	6	6	6	6	6
	Weighted Average	8.1	1.7	39.4	1.70	0.46	1.25	43.5	43.5	33.5	4.5	4.04	3.09
	Min	7.8	1.1	25.7	0.17	0.03	0.12	28.6	28.6	18.6	-51.6	0.38	0.26
	5th Percentile	7.8	1.2	26.2	0.18	0.03	0.13	29.3	29.3	19.3	-45.9	0.43	0.31
	Median	8.0	0.8	40.6	0.46	0.06	0.41	48.5	48.5	38.5	32.3	4.64	3.85
	95th Percentile	9.3	2.8	62.7	3.77	1.19	2.57	62.2	62.2	52.2	52.4	11.06	8.79
	Max	9.6	2.9	65.5	4.21	1.55	2.66	63.9	63.9	53.9	52.5	12.20	10.07

* - Sulphide sulphur calculated as Total Sulphur – Sulphate Sulphur

AP: acid potential, calculated from sulphide sulphur

NP: neutralization potential

IC_(Ca, Mg) calculated as NP – 10 kg CaCO₃ equiv/tonne

NPP: net neutralization potential (NP-AP)

NP/AP: neutralization potential ratio

IC_{(Ca, Mg)/AP}: carbonate neutralization potential ratio

The weighted average sulphur in chalcopyrite was estimated from copper assays to be 0.19%. Pyritic sulphur was calculated from excess sulphide sulphur to have a weighted average of 1.14% for all volcanic units.

Junction Zone

The Junction Zone rocks had a weighted average sulphate sulphur content of 0.82% and a weighted average sulphide sulphur content of 0.36%, which combined for a weighted average total sulphur content of 1.18% (Table 5.3-3). These averages encompass a large number of distinct rock types, each represented by a relatively small number of samples.

**Table 5.3-3
Summary of Acid-Base Accounting Characteristics for
Junction Zone Rocks**

Rock Group	Summary Function	Paste pH	CO ₂ (Wt. %)	CO ₂ (Kg CaCO ₃ / Tonne)	Total Sulphur (Wt. %)	Sulphate Sulphur (Wt. %)	Sulphide Sulphur* (Wt. %)	IC _(Ca, Mg) (Kg CaCO ₃ /Tonne)				IC _(Ca, Mg) /AP	
								AP	NP	(NP-10)	NNP		
All	Number	75	75	75	75	75	75	75	75	75	75	75	75
	Weighted Average	8.62	1.13	25.63	1.18	0.82	0.36	11.34	34.63	24.63	23.30	11.00	7.87
	Min	8.10	0.07	1.59	0.03	0.01	0.01	0.31	7.00	-3.00	-35.06	0.28	-0.27
	5th Percentile	8.20	0.13	2.89	0.07	0.01	0.03	0.84	13.18	3.18	-11.50	0.63	0.27
	Median	8.70	0.92	20.91	0.35	0.05	0.22	6.88	30.20	20.20	22.69	4.47	3.20
	95th Percentile	9.00	2.06	46.75	3.61	3.05	1.22	38.00	52.95	42.95	51.20	36.66	26.28
	Max	9.20	7.97	181.14	4.94	3.36	1.81	56.56	143.30	133.30	132.94	111.36	79.36
All intrusives	Number	33	33	33	33	33	33	33	33	33	33	33	33
	Weighted Average	8.55	1.23	27.93	0.56	0.37	0.20	6.26	35.95	25.95	29.71	13.66	9.70
	Min	8.10	0.13	2.95	0.05	0.01	0.02	0.60	10.60	0.60	-27.84	0.28	0.02
	5th Percentile	8.16	0.15	3.45	0.07	0.01	0.03	0.81	13.14	3.14	4.16	1.39	0.41
	Median	8.50	1.02	23.18	0.23	0.04	0.14	4.38	33.50	23.50	27.40	7.26	5.40
	95th Percentile	9.08	2.14	48.55	2.59	2.55	0.44	13.69	54.40	44.40	50.96	37.83	26.67
	Max	9.20	7.67	174.32	3.39	3.11	1.23	38.44	139.50	129.50	132.94	45.67	34.24
All sediments and volcanics	Number	42	42	42	42	42	42	42	42	42	42	42	42
	Weighted Average	8.68	1.05	23.82	1.66	1.17	0.49	15.33	33.59	23.59	18.25	8.91	6.43
	Min	8.10	0.07	1.59	0.03	0.01	0.01	0.31	7.00	-3.00	-35.06	0.38	-0.27
	5th Percentile	8.40	0.12	2.73	0.11	0.02	0.04	1.34	13.92	3.92	-15.25	0.62	0.26
	Median	8.70	0.67	15.11	1.80	1.09	0.39	12.19	28.55	18.55	14.78	2.37	1.37
	95th Percentile	9.00	1.58	36.00	3.92	3.12	1.31	40.78	47.79	37.79	46.76	25.08	22.15
	Max	9.10	7.97	181.14	4.94	3.36	1.81	56.56	143.30	133.30	131.11	111.36	79.36

* - Sulphide sulphur calculated as Total Sulphur – Sulphate Sulphur

AP: acid potential, calculated from sulphide sulphur

NP: neutralization potential

IC_(Ca, Mg) calculated as NP – 10 kg CaCO₃ equiv/tonne

NNP: net neutralization potential (NP-AP)

NP/AP: neutralization potential ratio

IC_(Ca, Mg)/AP: carbonate neutralization potential ratio

Weighted average chalcopyrite sulphur concentration for all Junction rocks tested was 0.07%, with a calculated weighted average pyritic sulphur content of 0.29%.

Sulphate sulphur concentrations were typically lower than in Central Zone rocks, with a weighted average sulphate sulphur concentration of 0.82%. However, Units S6, CHL, V5D, V5H, and VJP all had elevated ABA sulphate sulphur contents, with weighted average values up to 3.06% (Unit V5D).

5.3.6.3 Occurrence of Metals

Copper in near-surface rock is hosted in sulphide minerals (bornite, chalcopyrite, rare chalcocite) as well as secondary carbonates (malachite and azurite). Exploration drill core logging has

shown that copper carbonates are insignificant below the near-surface weathering zone, where copper is present as copper sulphides exclusively.

Sphalerite and galena are present in Galore Creek rocks, generally in trace concentrations as noted in exploration drill logs and as confirmed in mineralogical investigations. Core logging has identified a sphalerite-rich zone at depth in the area of the Central deposit, and assay results have confirmed the elevated zinc content of this core.

The non-metal selenium is broadly correlated with both sulphide-sulphur and copper content, which suggests that pyrite and chalcopyrite in Galore Creek rocks carry trace, but variable, amounts of selenium.

5.3.6.4 Occurrence of Buffering Capacity

A conservative measure of the buffering capacity, or neutralization potential (NP), of the rocks hosting the Galore Creek deposit can be calculated from the calcium + magnesium carbonate content of individual samples. Measurement of inorganic carbon has been found to slightly overestimate the amount of calcium+magnesium carbonate present, due to the presence of minor proportions of iron and manganese carbonates in some samples. In addition, laboratory measurements of NP were found to exceed the neutralization potential provided by calcium+magnesium carbonate ($IC_{Ca,Mg}$). Mineralogical results suggest that several silicate phases (e.g. potassium feldspar, biotite and sericite) are present which contribute to NP measured in laboratory testing but which do not contribute to effective neutralization potential under field conditions.

Tables 5.3-1 to 5.3-3 summarize the measured NP and calculated $IC_{Ca,Mg}$ values for the Central-West Fork- Southwest zone and for the Junction zone. The summary statistics show that all broad rock groups have weighted average $IC_{Ca,Mg}$ values that range from slightly negative (indicating the presence of soluble acidity) up to 31 and 79 kg $CaCO_3$ / t, for the Central- West Fork- Southwest and Junction zones, respectively.

5.3.6.5 Potential for ARD

Geochemical testing has shown that some of the rocks at Galore Creek are potentially ARD generating (PAG) over long time scales. Evaluation of test results has led to the identification of a site-specific ratio of $IC_{Ca,Mg}$ to acid potential (AP) that defines PAG and not-PAG (not-acid generating) rock. This relationship is:

$$IC_{Ca,Mg} / AP = 1.3$$

Laboratory testing and field evaluation have shown that fresh PAG rock will ultimately generate acid, however there will be a delay between time of exposure and onset of ARD generation. This delay corresponds to a period in which buffering capacity is being consumed, and is conservatively estimated to be greater than 22 years for the large majority of rock at the site. The conservative nature of this estimate is supported by the lack of acidity observed in 1960s drill core that has been stored at the site under surface conditions for over 40 years.

5.3.6.6 Kinetic Geochemical Characteristics

Humidity Cell Results

Central Zone

Twenty-four humidity cell tests (HCTs) were carried out on samples from the Central, Southwest, and West Fork zones.

Of the twenty-four HCTs, pH remained neutral to slightly alkaline after the first week of testing in all but three cells. These three cells contain samples of Units D and I9. Leachate pH in these cells has varied from 5.5 to 7.5, and may have stabilized around pH 6.

Sulphate release rates to date have been directly related to sulphate content of the sample. Three groups of sulphate behaviour were observed. The highest leachate sulphate release rates, of around 800 mg/kg/week, were observed in a group of cells with high initial gypsum contents.

Another group of samples, which had either lower initial gypsum concentrations or had been preconditioned by gypsum removal, returned intermediate sulphate release rates ranging from about 50 to 200 mg/kg/week.

The lowest sulphate release rates occurred from samples that were either preconditioned by gypsum removal from samples with intermediate initial gypsum contents, or had low initial gypsum content; sulphate release rates were between 3.5 and 10 mg/kg/week for this group.

Copper release was initially elevated for two of twenty-four samples tested, however copper production from one of these four (a volcanic unit) declined rapidly following the onset of testing. The remaining sample with elevated copper loadings was collected from Unit I9 with low NP, and copper release rates appear to have stabilized at approximately 0.2 mg/kg/week. This sample was selected specifically to evaluate metal leaching prior to onset of ARD. The remaining cells are producing copper concentrations in cell effluent of less than 0.001 mg/kg/week.

Release rates for lead, zinc, molybdenum, fluoride and selenium are described in the SRK report in Appendix 5-A.

Junction Zone

Nine humidity cell tests (HCTs) were carried out on samples of different rock units from the Junction Zone.

Sulphate release rates were split into a group of four cells with release rates between 100 and 1000 mg/kg/week, and another group of five cells with lower release rates ranging from 1 to 5 mg/kg/week. The samples yielding lower sulphate releases had lower solid sulphate concentrations or were preconditioned by removing gypsum prior to onset of testing. Those samples which produced higher rates of sulphate release had high initial solid sulphate concentrations; two of these samples had been subjected to gypsum leaching prior to initiation of tests, but sufficient solid sulphate appears to have remained to yield high sulphate release rates.

Copper release from all nine HCTs ranged from 0.0004 to 0.003 mg/kg/week. Dyke, and volcanic and sedimentary rock release rates were at the lower end of this range, and intrusive porphyry and breccia rock had copper release rates at the high end of this range.

Release rates for lead, zinc, molybdenum, fluoride and selenium are described in the SRK report in Appendix 5-A.

5.3.6.7 Relationships between Weathering Rates and Bulk Characteristics of Rock

Sulphide Content

Sulphate release was compared with sulphide content of rock samples and showed an apparent wide distribution of sulphate release rates from as low as 2 mg/kg/week to 659 mg/kg/week. Sulphate release occurs in the following three groups.

1. The highest release rates are for samples containing high sulphate concentrations that were not leached to remove sulphate minerals prior to the test. The high sulphate release reflects dissolution of calcium sulphate.
2. The second group is samples containing high original sulphate concentrations (greater than 0.59%) and a range of sulphide concentrations (0.3 to 3.6%) that were leached to remove calcium sulphate before the test started. Although the leaching procedure removed at least 95% of the sulphate sulphur, sulphate release rates from the humidity cells (19 to 280 mg/kg/week) reflected continuing leaching of residual sulphate. In other words, the procedure to remove calcium sulphate was somewhat effective (compared to the unleached samples) but was not sufficient to allow sulphide oxidation to become the main source of sulphate in leachates.
3. The third group is samples containing low original sulphate concentrations (less than 0.1%) and a wide range of sulphide sulphur (0.13% to 2.9%), some of which were not leached to remove calcium sulphate. With the exception of one sample from Junction zone, these samples showed a linear relationship between sulphide content and sulphate release. The range of sulphate release was 0.3 to 10 mg/kg/week. Further examination of the sulphur speciation data for the Junction Zone sample (GC63-0054-57.91) showed discrepancies which may indicate uncertainties in the sulphate analysis result. The sample may have contained sufficient calcium sulphate to result in an elevated sulphate release for its low sulphide content (0.23%).

Using the linear trend for the third group, the regression relationship between the sulphide content and sulphate release was calculated to be:

$$\text{Sulphate Release (mg/kg/week)} = 2.33 \cdot \text{Sulphide (\%,S)} + 2.21.$$

The non-zero intercept (2.21 mg/kg/week) may be the effect of the detection limit for sulphate concentrations or leaching of residual calcium sulphate.

5.3.6.8 Oxidized Rocks

Oxidation of the Central Zone occurs to a depth of about 20 m on average and consists of limonite, chalcocite, malachite and azurite. Sulphides are commonly present in the oxide zone. Minor copper oxide is present in the Southwest Zone and is absent in the West Fork and Junction Zones.

Gypsum has been leached to a depth of 30 to 122 m by the action of groundwater. This leaching has resulted in a distinctive “gypsum line” that mimics topography and significantly affects rock quality. The sulphate content of rocks in the leached zone is generally low.

5.3.6.9 Static Geochemical Characteristics

Testing of fine (<2 mm) and coarse (>10 mm) fractions of gossan material provided insight into natural weathering characteristics. Fines were considered to represent mature weathering conditions, whereas the coarse fraction, which was relatively unweathered, provided an estimate of initial rock properties.

Figure 5.3-9 shows the rinse pH of gossan fines against Modified NP determined for the same material. The acidic rinse pH values observed for samples with NP contents less than 4.7 kg CaCO₃/t indicates that the Modified NP procedure measured a non-reactive NP component of the samples tested, which likely corresponded to dissolution of silicates during the analytical procedure. This result was incorporated into the development of the relationship between IC_{Ca,Mg} and Modified NP that was discussed in Section 5.3.6.5.

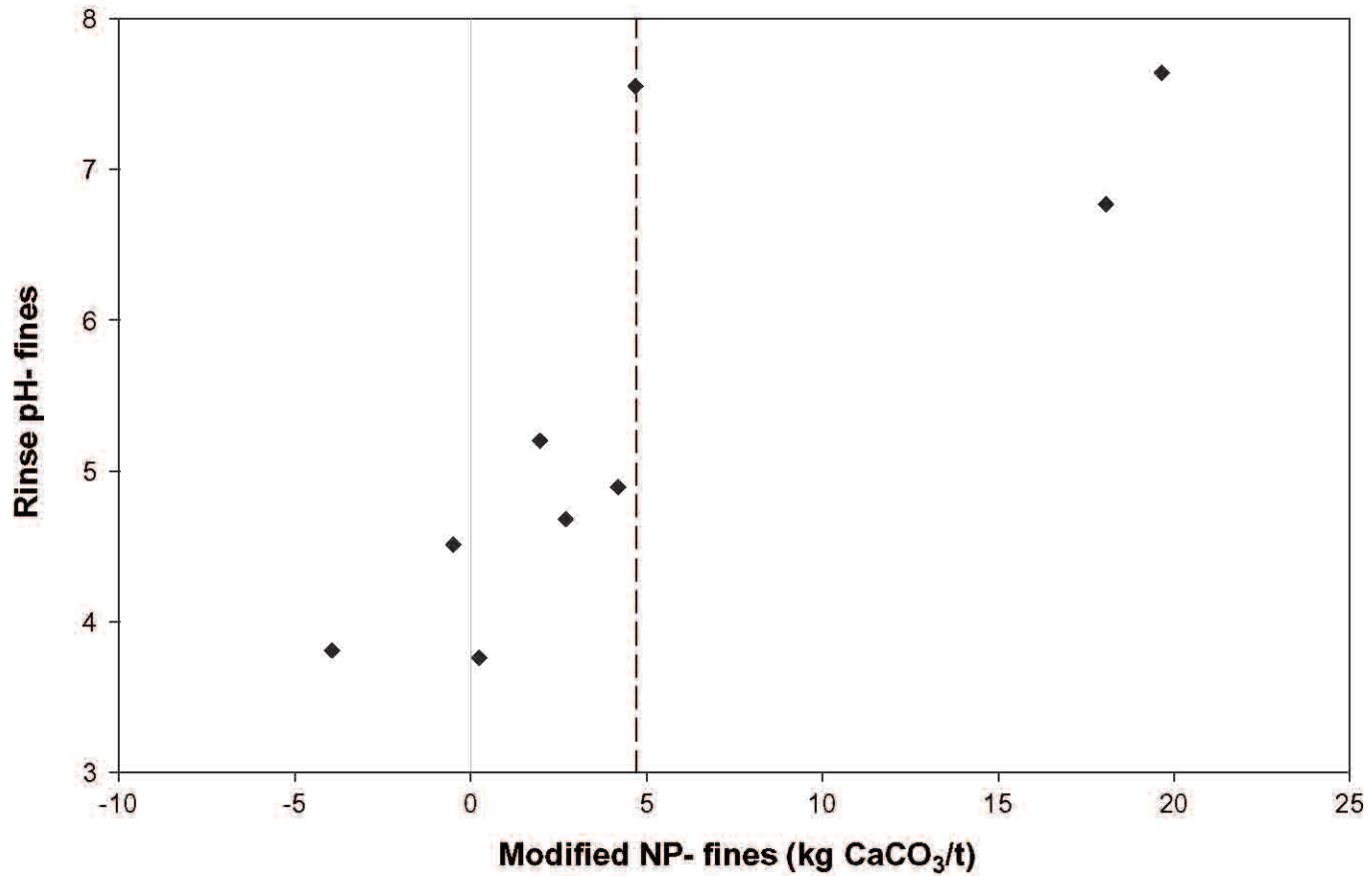
Most trace metals were enriched in the fine fraction, whereas calcium, magnesium, and potassium were present in greater concentrations in the coarse fraction.

5.3.6.10 Natural Leaching

Seepage from oxidized rock exposed in the Galore Creek gossan was found to be dominated by calcium and sulphate, with variable pH. Most seeps had neutral to slightly alkaline pH, but one seep was acidic (pH 3.7). Sulphate concentrations in gossan seeps varied between 100 and 1000 mg/L.

Concentrations of leachable elements in gossan seeps varied widely for several parameters including Al, Cd, Co, Cu, Fe, Mn, Mo, Ni, and Zn. Copper was highest (3.4 mg/L) in the acidic seep, but a seep with near-neutral pH (6.9) returned a dissolved copper concentration of 2.5 mg/L, indicating that copper leaching is occurring under neutral pH conditions. Zinc was measured at concentrations up to 1.7 mg/L, with the acidic seep returning a zinc concentration of 1.6 mg/L. Fluoride was elevated at concentrations of up to 2.3 mg/L.

Shake flask tests conducted on gossan fines (<2 mm) produced dilute leachates dominated by sulphate±alkalinity, and calcium. The range of leachate pH values was similar to the range of gossan seep pH values. Shake flask leachates contained detectable concentrations of Al, Cd, Ca, Co, Cu, Fe, Pb, Mg, Mn, Mo, Ni, and Zn.



Note: The dashed line is NP = 4.7 kg CaCO₃/t
Source: SRK Consulting

5.3.6.11 Overburden

Table 5.3-4 provides selected data from analysis of the -2 mm fraction of overburden in the Central Pit area. The samples indicate a narrow range of sulphur concentrations. Sulphide content of the material is approximately 0.5% (as sulphur) with low sulphate concentrations. The samples had NP/AP greater than 2.6 and are therefore classified as not-PAG (sometimes referred to in this report as NPAG). Copper concentrations were elevated (827 mg/kg to 1034 mg/kg) and consistent with the mineralized area as the source.

**Table 5.3-4
Acid-Base Accounting Data and Copper Concentration in
Overburden Samples**

Sample	Paste pH	Total S	Sulphate	Sulphide	AP	TIC	NP	NP/AP	Cu
		S, %	S, %	S, %	kg CaCO3/t	%CO2	kg CaCO3/t		mg/kg
TP-BGC05-08	8.15	0.54	0.03	0.51	16.1	3.63	85.4	5.3	901
TP-BGC05-09	8.21	0.45	0.01	0.44	13.8	3.70	87.9	6.4	827
TP-BGC05-10	8.15	0.59	0.01	0.58	18.3	3.43	80.8	4.4	816
TP-BGC05-11	8.12	0.50	0.01	0.49	15.3	2.78	66.4	4.4	1034
TP-BGC05-12	7.98	0.69	0.01	0.68	21.1	2.11	54.0	2.6	887

Leach tests performed on the same samples indicated generally non-detectable concentrations of most parameters. Leachable copper concentrations were between 0.009 and 0.015 mg/L. Leachable molybdenum was between 0.034 and 0.13 mg/L. Zinc and lead were not detected (<0.005 mg/L and <0.001 mg/L, respectively).

5.3.6.12 Summary

Geologic materials that would be disturbed by mining at Galore Creek include overburden, oxidized bedrock, and fresh bedrock. Overburden materials are considered unlikely to generate acid, however copper leaching may occur from this material if sourced from an area underlain by mineralized bedrock. Oxidized bedrock locally contains secondary copper mineralization, and ranges from neutral to acidic under present-day field conditions.

Fresh bedrock will be the largest component of the mine waste produced at Galore Creek. There are several rock types that have been identified during exploration, consisting of several intrusive, volcanic, and sedimentary units, along with dike and breccia lithologies. ML/ARD characteristics of the bedrock do not appear to be correlated with rock type, but instead are controlled by bulk material characteristics such as contained metal and distribution of buffering capacity and sulphide sulphur.

Field evaluation of exposed bedrock has shown that acidic conditions can develop from some of the Galore Creek bedrock. Further, this field evaluation and follow-up laboratory testing showed that conventional Modified NP tests overestimate the available buffering capacity. Carbonate buffering capacity, in the form of calcium and magnesium carbonates (referred to as IC_{Ca,Mg}), was found to be a better estimate of available NP. A relationship between measured NP and

$IC_{Ca,Mg}$ was identified that will allow $IC_{Ca,Mg}$ to be estimated from Modified NP test results, as follows:

$$IC_{Ca,Mg} = \text{Modified NP} - 10 \text{ (in kg CaCO}_3\text{/ tonne).}$$

Testing indicates this approximation will underestimate $IC_{Ca,Mg}$ where measured NP is high.

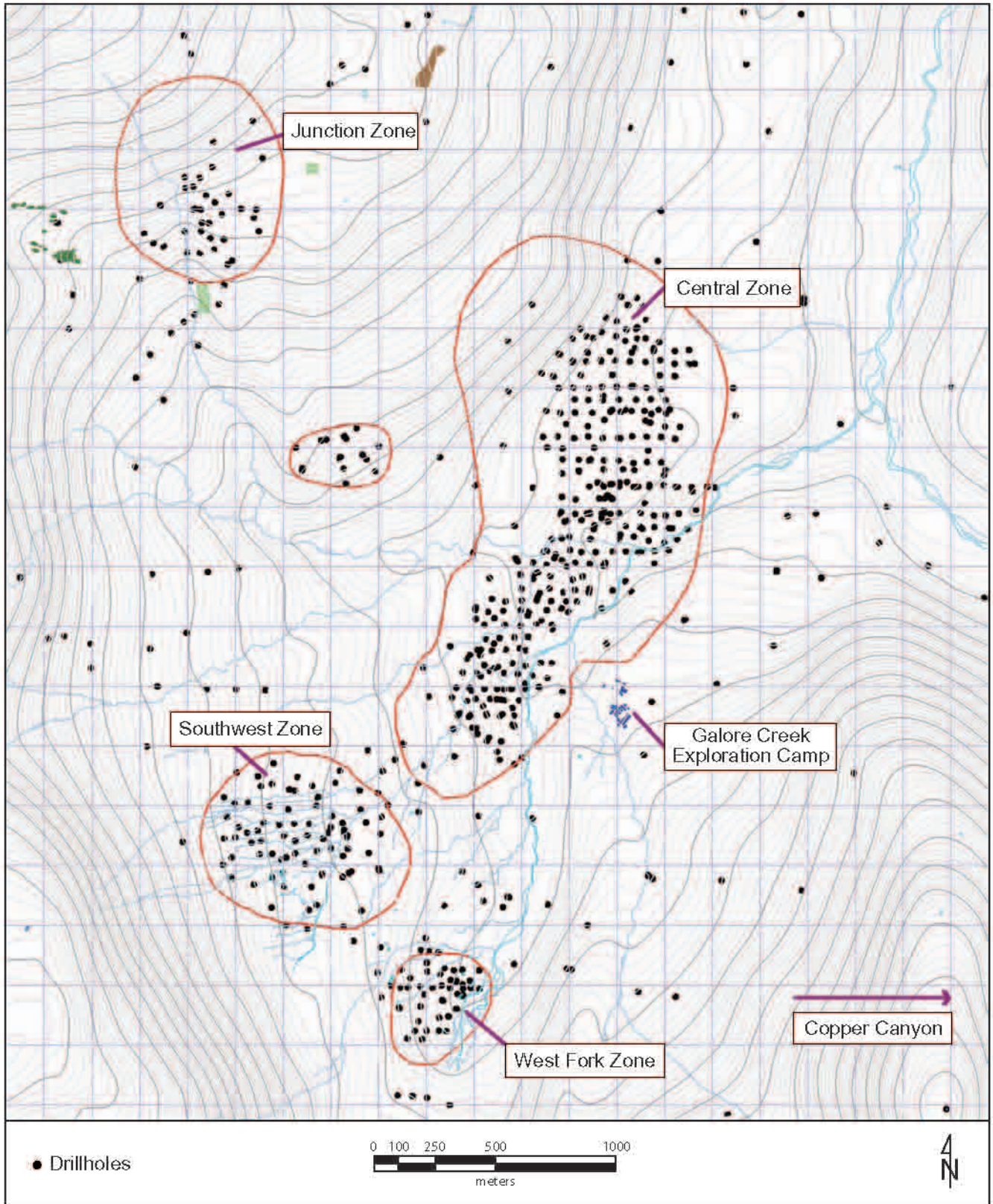
A provisional criterion for differentiating PAG from not-PAG waste rock was identified on the basis of ML/ARD characterization of a large number of samples, representing the full range of metal content, sulphur content, and rock types present in the vicinity of the deposits at Galore Creek. This evaluation indicated that rock material with $IC_{Ca,Mg} / AP > 1.3$ will not likely generate ARD over the long term. Furthermore, based on kinetic tests and the lack of acidic conditions in core from the 1960s, ARD generation has been estimated to take at least 22 years after rock is exposed.

5.4 Mineral Resource and Mineral Reserve Estimates

GR Technical Services Ltd. (GR Tech), Giroux Consultants Limited (Giroux) and Hatch were retained by NovaGold Resources Inc. (NovaGold) to complete a mineral resource estimate for the Galore Creek project, including the Central, Southwest, Junction, and West Fork Zones and complete a Technical Report summarizing the findings to meet the requirements of National Instrument 43-101 (the Instrument) and Form 43-101F1. The mineral resource study was a collaborative effort by NovaGold, GR Tech, and Giroux, with Mr. Robert J. Morris of GR Tech and Mr. Gary H. Giroux acting as independent Qualified Persons as defined by the Instrument. Much of the data, including the drill hole assay and geological database, upon which the estimate is based, has undergone thorough scrutiny by project staff as well as certain data verification procedures by Giroux and Morris. Mr. Morris conducted a site examination of the project area during the week of October 10, 2004.

The Galore Creek Project consists of 4 separate mineralized zones namely: Central, Southwest, West Fork and Junction. Each of these zones (Figure 5.4.-1) has been treated as a stand alone deposit for the purpose of resource estimation. This section will outline the procedures used to produce these four separate estimates.

The first step in calculating the mineral resource was to produce a detailed geologic interpretation to be converted into a three dimensional geologic solid model. The geologic solids have been built by NovaGold Senior geologists most familiar with the project, as this step often entails subjective decisions on boundaries and contacts of individual geologic units. This procedure was accomplished by interpreting geologic contacts from drill hole core logging, on cross sections cut through the deposit. The interpreted contacts were digitized into three dimensional solids, using *MineSight* 3D modeling software. The estimation process continues with a detailed statistical review (data analysis) of the various grade parameters as a function of its variables such as rock type and location. A block model consisting of a grid of blocks, sized appropriately to a dimension amenable to modern mining techniques, was superimposed over the geologic model for the purpose of interpolating a value for each variable into each block which is then tallied up for the resource tables. The same process, the same geologic model, and many of the same variables were used for the construction of the acid base accounting (ABA) model.



Galore Creek Valley Main Deposits

FIGURE 5.4-1

The acid-base accounting block model was subsequently used to calculate the relative volumes of potentially ARD generating (PAG) and non-PAG wastes and the proportion of waste exposure in the pit walls. The exposure of wastes in waste rock dumps was indicated by coupling the block model, pit designs and mining schedule. The block model was not used to evaluate exposure duration for the pit walls, but the effect of exposure times on the pit walls was assessed qualitatively based on the overall distribution of neutralization potential indicated by the acid-base accounting database.

5.4.1 Central Zone

5.4.1.1 Data Analysis

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

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

Simple statistical parameters for each group are shown below in Table 5.4-1.

**Table 5.4-1
Summary of Statistical Parameters for Geologic Domains**

Domain	Variable	Number	Mean	Standard Deviation	Minimum	Maximum	Coefficient of Variation
Group 1	Cu (%)	16,422	0.494	0.706	0.001	12.00	1.43
West (201)	Au (g/t)	16,140	0.253	0.803	0.001	35.31	3.16
	Ag (g/t)	16,156	3.88	8.98	0.001	886.0	2.31
Group 1	Cu (%)	4,136	0.344	0.458	0.001	3.65	1.33
East (202)	Au (g/t)	4,040	0.112	0.241	0.001	6.13	2.16
	Ag (g/t)	4,040	3.02	4.66	0.001	35.01	1.54
Group 2	Cu (%)	1,954	0.320	0.513	0.001	4.27	1.60
West (337)	Au (g/t)	1,931	0.195	0.687	0.001	21.60	3.52
	Ag (g/t)	1,934	2.61	4.05	0.001	27.80	6.55
Group 2	Cu (%)	96	0.311	0.414	0.001	1.76	1.33
East (338)	Au (g/t)	96	0.056	0.055	0.001	0.42	1.53
	Ag (g/t)	96	1.84	2.39	0.001	7.20	1.30
Group 3	Cu (%)	3,361	0.147	0.359	0.001	5.92	2.44
West (357)	Au (g/t)	3,346	0.105	0.357	0.001	10.25	3.38
	Ag (g/t)	3,348	1.47	6.89	0.001	362.0	4.69
Group 3	Cu (%)	319	0.065	0.137	0.001	1.03	2.12
East (358)	Au (g/t)	318	0.048	0.092	0.001	0.74	1.93
	Ag (g/t)	318	1.15	2.58	0.001	14.74	2.24

The coefficients of variation range between 1.33 and 6.55, indicating a fair degree of variability which is also evident in the maximum values present. Lognormal cumulative probability plots were produced for each variable within each of the six main geologic domains to examine the distributions of grade within each data set and determine if capping was required and if so at what level. In all cases the grade distributions were made up of a series of overlapping lognormal populations. Using the method of partitioning of lognormal cumulative probability plots, each population can be subdivided and evaluated.

Group 1

Group 1 is made up primarily of volcanic rocks, with minor sedimentary and intrusive units too small to model. The plot for copper shows five overlapping lognormal populations with parameters shown in Table 5.4-2. The upper population 1, representing 0.53 % of the data, has a mean of 4.14 % Cu. This population is not erratic and a reasonable cap would be at two standard deviations (SD) above the mean of population 1 or a value of 6.90 % Cu. A total of five samples were capped at 6.90 % Cu.

**Table 5.4-2
Summary of Cu Distribution in Group 1**

Population	Mean Cu (%)	Proportion	Number of Samples
1	4.135	0.53 %	109
2	0.983	30.39 %	6,247
3	0.183	42.13 %	8,660
4	0.036	14.01 %	2,880
5	0.010	12.95 %	2,662

The plot for gold shows six overlapping lognormal populations with parameters shown in Table 5.4-3. The upper population 1, representing 0.02 % of the data, has a mean of 28.39 g Au/t. This population can be considered erratic and should be minimized by capping at two standard deviations above the mean of population 2 or a value of 16.59 g Au/t. A total of five samples were capped at 16.59 g Au/t.

**Table 5.4-3
Summary of Au Distribution in Group 1**

Population	Mean Au (g/t)	Proportion	Number of Samples
1	28.39	0.02 %	4
2	10.78	0.12 %	24
3	4.47	0.66 %	133
4	1.28	4.74 %	957
5	0.17	44.45 %	8,970
6	0.018	50.00 %	10,090

The plot for silver shows three overlapping lognormal populations with parameters shown in Table 5.4-4. The upper population 1, representing 0.05 % of the data, has a mean of 53.4 g Ag/t. This population can be considered erratic and should be minimized by capping at the mean of population 1 or a value of 53.4 g Ag/t. A total of eight samples were capped at 53.4 g Ag/t.

Group 2

Group 2 is made up of I4 – Dark Orthoclase Syenite, I8 – Syenite and I3 – Grey Syenite Porphyry dykes and sills.

**Table 5.4-4
Summary of Ag Distribution in Group 1**

Population	Mean Ag (g/t)	Proportion	Number of Samples
1	53.40	0.05 %	10
2	4.83	53.91 %	10,888
3	0.02	46.03 %	9,296

The plot for copper shows five overlapping lognormal populations with parameters shown in Table 5.4-5. The upper population 1, representing 0.40 % of the data, has a mean of 3.8 % Cu. In this case the upper population does not appear to be erratic and as a result a reasonable capping level would be at two standard deviations above the mean of population 1 or a value of 4.5 % Cu. No samples required capping.

**Table 5.4-5
Summary of Cu Distribution in Group 2**

Population	Mean Cu (%)	Proportion	Number of Samples
1	3.833	0.40 %	8
2	0.964	20.78 %	426
3	0.122	54.30 %	1,113
4	0.016	18.40 %	377
5	0.004	6.12 %	126

The plot for gold shows six overlapping lognormal populations with parameters shown in Table 5.4-6. The upper two populations (1 & 2) representing a combined 0.39 % of the data, have mean grades of 14.48 and 4.75 g Au/t respectively. These two populations representing only a combined eight samples can be considered erratic and should be minimized by capping at two standard deviations above the mean of population 3 or a value of 3.85 g Au/t. A total of eight samples were capped at 3.85 g Au/t.

**Table 5.4-6
Summary of Au Distribution in Group 2**

Population	Mean Au (g/t)	Proportion	Number of Samples
1	14.48	0.14 %	3
2	4.752	0.25 %	5
3	1.801	1.66 %	34
4	0.311	27.70 %	560
5	0.034	60.76 %	1,227
6	0.003	9.48 %	191

The plot for silver shows four overlapping lognormal populations with parameters shown in Table 5.4-7. The upper population 1, representing 1.10 % of the data, has a mean of 31.48 g Ag/t. No capping is required for silver in Group 2.

**Table 5.4-7
Summary of Ag Distribution in Group 2**

Population	Mean Ag (g/t)	Proportion	Number of Samples
1	31.48	1.10 %	22
2	7.44	19.68 %	398
3	0.31	29.22 %	591
4	0.036	50.00 %	1,012

Group 3

Group 3 is made up of I5 – Fine Grained Orthoclase Syenite Megaporphyry, I9 – Medium Grained Orthoclase Syenite Megaporphyry, I10 – Plagioclase Syenite Porphyry and I11 – Syenite Porphyry.

The plot for copper shows five overlapping lognormal populations with parameters shown in Table 5.4-8. The upper population 1, representing 0.28 % of the data, has a mean of 3.85 % Cu. In this case the upper population does not appear to be erratic and as a result a reasonable capping level would be at two standard deviations above the mean of population 1 or a value of 6.1 % Cu. No samples required capping.

**Table 5.4-8
Summary of Cu Distribution in Group 3**

Population	Mean Cu (%)	Proportion	Number of Samples
1	3.853	0.28 %	10
2	1.242	4.21 %	155
3	0.226	18.33 %	675
4	0.038	64.00 %	2,355
5	0.002	13.19 %	485

The plot for gold shows five overlapping lognormal populations with parameters shown in Table 5.4-9. The upper population 1 representing 0.26 % of the data, has a mean grade of 3.6 g Au/t. This population can be considered erratic and should be minimized by capping at two standard deviations above the mean of population 2 or a value of 4.0 g Au/t. A total of five samples were capped at 4.0 g Au/t.

**Table 5.4-9
Summary of Au Distribution in Group 3**

Population	Mean Au (g/t)	Proportion	Number of Samples
1	3.595	0.26 %	10
2	2.300	0.54 %	20
3	0.694	1.61 %	59
4	0.194	17.45 %	639
5	0.018	80.14 %	2,936

The plot for silver shows four overlapping lognormal populations with parameters shown in Table 5.4-10. The upper population 1, representing 6.97 % of the data, has a mean of 6.95 g Ag/t. A reasonable capping level would be two standard deviations above the mean of population 1, a level of 27.72 g Ag/t. A total of four samples were capped at 27.72 g Ag/t.

**Table 5.4-10
Summary of Ag Distribution in Group 3**

Population	Mean Ag (g/t)	Proportion	Number of Samples
1	6.95	6.97 %	256
2	2.43	19.43 %	712
3	0.51	23.60 %	865
4	0.01	50.00 %	1,833

The capping information is summarized in Table 5.4-11.

**Table 5.4-11
Summary of Capping Levels**

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

Composites

Drill holes within the Central Zone were “passed through” the geologic solids with the point each entered and left each solid recorded. Uniform down-hole 5 metre (m) composites were produced that honoured the domain boundaries. Intervals at the boundary of solids were combined with adjoining composites if less than 2.5 m and left as is if greater or equal to 2.5 m. In this manner the composites formed a uniform support of 5 ± 2.5 m. The statistics for composites are summarized in Table 5.4-12. The combination of capping and smoothing over 5 m has reduced all coefficients of variation to below 2.88.

**Table 5.4-12
Summary of Statistical Parameters for 5 m Composites**

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

Variography

Pair wise relative semivariograms were produced for each variable in Group 1 east and west of the East Fault, in Group 2 west of the East Fault and in Group 3 west of the East Fault. There were insufficient data to model variography east of the East Fault in Groups 2 and 3 and the models generated west of the fault were used. Models were generated in the vertical direction and in 4 main directions within the horizontal plane (Az. 90, 0, 45 and 135). Anisotropic nested models were fit to the data in all but Group 3 silver where an omni-directional spherical nested model was fit. The parameters for all models are shown below in Table 5.4-13.

Geologic Block Model

A block model of 25 x 25 x 15 m blocks was superimposed on the geologic 3-D solids. The origin and parameters of the general Galore Creek block model, within which the Central deposit lies, were as follows:

Lower left corner Easting - 48000 E 25 m block 240 columns
 Northing - 32000 N 25 m block 240 rows
 Top Level Elevation - 160515 m block 107 levels
 No Rotation

**Table 5.4-13
 Parameters for Semivariogram Models at Galore Creek Central Zone**

Domain	Variable	Direction	C0	C1	C2	Range a1 (m)	Range a2 (m)
Group 1		Az. 30° Dip 0	0.20	0.25	0.45	30	200
Volcanics	Cu	Az. 300° Dip -45	0.20	0.25	0.45	40	100
West of Fault (201)		Az. 120° Dip -45	0.20	0.25	0.45	60	100
	Au	Az. 45° Dip 0	0.10	0.40	0.50	20	200
		Az. 315° Dip 0	0.10	0.40	0.50	20	120
		Az. 0° Dip -90	0.10	0.40	0.50	30	120
	Ag	Az. 45° Dip 0	0.10	0.45	0.65	20	120
		Az. 315° Dip 0	0.10	0.45	0.65	20	60
		Az. 0° Dip -90	0.10	0.45	0.65	30	200
Group 1		Az. 30° Dip 0	0.10	0.30	0.42	40	130
Volcanics	Cu	Az. 300° Dip -45	0.10	0.30	0.42	40	150
East of Fault (202)		Az. 120° Dip -45	0.10	0.30	0.42	40	90
	Au	Az. 90° Dip 0	0.15	0.25	0.40	50	70
		Az. 0° Dip 0	0.15	0.25	0.40	20	60
		Az. 0° Dip -90	0.15	0.25	0.40	40	100
	Ag	Az. 90° Dip 0	0.05	0.30	0.75	40	80
		Az. 0° Dip 0	0.05	0.30	0.75	10	20
		Az. 0° Dip -90	0.05	0.30	0.75	20	120
Group 2		Az. 0° Dip 0	0.10	0.30	0.60	15	100
Intrusives	Cu	Az. 270° Dip 0	0.10	0.30	0.60	20	50
I4, I8, I3		Az. 0° Dip -90	0.10	0.30	0.60	50	150
West of Fault (337)	Au	Az. 135° Dip 0	0.20	0.35	0.55	40	150
		Az. 45° Dip 0	0.20	0.35	0.55	35	100
		Az. 0° Dip -90	0.20	0.35	0.55	60	110
	Ag	Az. 135° Dip 0	0.10	0.60	0.70	20	100
		Az. 45° Dip 0	0.10	0.60	0.70	40	60
		Az. 0° Dip -90	0.10	0.60	0.70	90	200
Group 3		Az. 90° Dip 0	0.10	0.40	0.50	20	40
Intrusives	Cu	Az. 0° Dip 0	0.10	0.40	0.50	20	40
I5, I9, I10, I11		Az. 0° Dip -90	0.10	0.40	0.50	20	80
West of Fault (357)	Au	Az. 135° Dip 0	0.20	0.40	0.50	50	160
		Az. 45° Dip 0	0.20	0.40	0.50	30	120
		Az. 0° Dip -90	0.20	0.40	0.50	70	120
	Ag	Omnidirectional	0.05	0.30	0.85	40	60

For each block the following information was recorded:

- percentage of block below topography;
- percentage of block below bedrock;

- percentage of block below the broken/stick surface;
- the most prevalent solid percentage as OREPC1; and
- the second most prevalent solid percentage as OREPC2.

Bulk Density

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 amount of water it displaced. In April 1992 a total of 96 specific gravity determinations were made from drill core. During the 2003-2004 drill programs a further 30 specific gravity determinations were made. The SG determinations were made from small pieces of core at recorded distances down the hole. Using these distances the specific gravity values were joined to the assay from-to intervals that contained them. In a number of instances more than one SG determination was made on the same from-to interval. In these cases the SG values were averaged. As a result 631 assays had specific gravity determinations, across the total property. The specific gravity information can be sorted a number of ways. Table 5.4-14 shows average values for the various lithologies coded.

**Table 5.4-14
Summary of Specific Gravity Measurements – Central Zone**

	Volcanics	Intrusives	Breccia	Dyke	Sediments	Fault
Number	435	140	8	7	5	3
Mean S.G.	2.67	2.63	2.71	2.67	2.59	2.59
S.D.	0.182	0.156	0.095	0.077	0.066	0.114
Minimum	1.69	1.91	2.56	2.50	2.34	2.43
Maximum	3.40	3.30	2.86	2.76	3.04	2.69

For this resource estimation Group 1 was assigned the average specific gravity of volcanic rocks or 2.67 while Group 2 and Group 3 were made up of intrusive rocks with an average specific gravity of 2.63. These values were used for blocks below the broken/stick rock surface. Material above this boundary was severely broken and as a result a general 5% reduction in specific gravity was applied. Blocks completely above the surface were given a specific gravity of 2.54 in Group 1 domains and 2.50 in Group 2 and 3 domains. Blocks straddling the boundary were given a weighted average specific gravity based on the percentage of volume above and below the broken rock surface.

Block Interpolation

Ordinary kriging was completed in three passes for each variable in each domain. A minimum 4 and maximum 16 composites were used for each pass. The search ellipse was expanded for each pass, starting at ¼ the ranges in each direction for pass 1, to 2/3 in pass 2 and finally to the full ranges in pass 3. The kriging process was completed in the following order.

- All blocks with some proportion below the bedrock surface and proportion of primary solid (OREPC1) greater than 0.0 were estimated for each variable in separate runs using the appropriate composites for domains with 3 separate passes in each run. Results to Cu1, Au1 and Ag1 fields.
- All blocks with some proportion below bedrock and proportion of OREPC2 (BDRK – PCORE1) > 0.0 were estimated for each variable using appropriate composites with 3 separate passes. Results to Cu2, Au2 and Ag2 fields.
- The grades were then combined to form weighted averages for each block based on the proportion of each domain in the block.
- Specific gravity (SG) used was 2.67 for the volcanics (Group 1) for unbroken or stick rock and 5 % less or 2.54 for broken rock. Within the intrusives (Groups 2 and 3) a value of 2.63 was used for stick rock and 2.50 for broken rock. Blocks containing both were adjusted to form a weighted average based on proportions of domains and of stick and broken rock present.
- Copper equivalent was estimated using the following procedure:
 - copper price of US\$ 0.90/lb.;
 - copper recovery = (%Cu-0.06)/%Cu with a minimum of 50% and maximum of 95%;
 - gold price of US\$ 375/oz;
 - gold recovery = (Au g/t-0.14)/Au g/t with a minimum of 30% and maximum of 80%;
 - silver price of US\$ 5.50/oz; and
 - silver recovery = 80%.

Classification

Introduction

Based on the study herein reported, delineated mineralization of the Galore Creek Central Zone is classified as a resource according to the following definition from National Instrument 43-101.

“In this Instrument, the terms "mineral resource", "inferred mineral resource", "indicated mineral resource" and "measured mineral resource" have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by CIM Council on August 20, 2000, as those definitions may be amended from time to time by the Canadian Institute of Mining, Metallurgy, and Petroleum.”

*“A **Mineral Resource** is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.”*

The terms Measured, Indicated and Inferred are defined in NI 43-101 as follows:

*“A '**Measured Mineral Resource**' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.”*

*“An '**Indicated Mineral Resource**' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.”*

*“An '**Inferred Mineral Resource**' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.”*

Results

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

Measured	Blocks estimated in Pass 1 (using ¼ of the semivariogram ranges) for both Cu and Au.
Indicated	Blocks not classified and estimated in at least pass 2 (using 2/3 of the semivariogram ranges) for both Cu and Au.
Inferred	All other blocks estimated. In addition all blocks below the 200 m elevation were assigned Inferred status irregardless of when they were estimated.

The results are presented at a range of CuEq cutoffs in tables included as Appendix 5-B, *Grade-Tonnage Tables for Resource Estimates Central, Southwest, Junction, West Fork Zones*, as at this time no economic evaluation has been completed. Tables 5.4-15 and 5.4-16 shown below

summarize the results for three particular cutoffs. Copper equivalent was calculated on the basis of a copper price of US\$0.90/lb, a gold price of US\$375/oz and a silver price of US\$5.50/oz.

**Table 5.4-15
Grade – Tonnage for Central Zone Classes Measured Plus Indicated**

Cutoff (CuEq %)	Tonnes > Cutoff (tonnes)	Grade > Cutoff				Million lbs. of Cu	Million Ozs of Au	Million Ozs of Ag
		Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)			
0.35	423,900,000	0.614	0.302	4.681	0.780	5739.05	4.12	63.80
0.50	290,100,000	0.738	0.376	5.421	0.947	4720.77	3.51	50.56
1.00	90,100,000	1.100	0.724	7.257	1.511	2185.38	2.10	21.02

**Table 5.4-16
Grade – Tonnage for Central Zone Classed Inferred**

Cutoff (CuEq %)	Tonnes > Cutoff (tonnes)	Grade > Cutoff				Million lbs. of Cu	Million Ozs of Au	Million Ozs of Ag
		Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)			
0.35	173,600,000	0.467	0.283	3.425	0.623	1787.62	1.58	19.12
0.50	96,000,000	0.576	0.380	4.085	0.790	1219.28	1.17	12.61
1.00	18,200,000	0.781	0.929	5.193	1.312	313.42	0.54	3.04

Inverse Distance Squared Test

As a test of the interpolation procedure, a second run was made using inversed distance squared (IDS). The same elliptical search parameters used for kriging were used for the IDS run. The minimum 4 and maximum 16 composite limits were used. As with kriging, the various domains were estimated in a series of passes with expanding search ellipses. The results are presented as grade-tonnage tables in Appendix 5-C, *Grade-Tonnage Tables for Resource Estimates, Inverse Distance Squared Resource Estimates, Central and Junction Zones*, for the classification categories of measured plus indicated, and inferred.

The results are very similar with differences occurring due to the weighting techniques employed by the different interpolation methods.

5.4.2 Southwest Zone

5.4.2.1 Data Analysis

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

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

Simple statistical parameters for each group are shown below in Table 5.4-17.

**Table 5.4-17
Summary of Statistical Parameters for Geologic Domains**

Domain	Variable	Number	Mean	S.D.	Minimum	Maximum	Coef. of Var.
Group 352	Cu (%)	1,171	0.223	0.402	0.001	3.23	1.80
	Au (g/t)	1,171	0.322	0.730	0.001	12.00	2.27
	Ag (g/t)	1,171	1.417	2.615	0.001	65.20	1.84
Group 400	Cu (%)	1,792	0.310	0.621	0.001	7.02	2.00
	Au (g/t)	1,792	0.668	1.870	0.001	40.80	2.80
	Ag (g/t)	1,792	3.10	6.28	0.001	135.00	2.03

The coefficients of variation range between 1.80 and 2.8 indicating a fair degree of variability, which is also evident in the maximum values present.

Lognormal cumulative probability plots were produced for each variable within each of the two main geologic domains to examine the distributions of grade within each data set and determine if capping is required and if so at what level. In all cases the grade distributions were made up of a series of overlapping lognormal populations. Using the method of partitioning of lognormal cumulative probability plots, each population can be subdivided and evaluated.

Group 352

Group 352 is made up primarily of unit I9 - Medium Grained Orthoclase Syenite Megaporphyry. The plot for copper shows 5 overlapping lognormal populations with parameters shown in Table 5.4-18. The upper population 1, representing 2.67 % of the data, has a mean of 2.2 % Cu. This population is not erratic and a reasonable cap would be at two standard deviations above the mean of population 1 or a value of 3.57 % Cu. No copper samples required capping.

**Table 5.4-18
Summary of Copper Distribution in Group 352**

Population	Mean Cu (%)	Proportion	Number of Samples
1	2.197	2.67 %	31
2	0.867	5.93 %	69
3	0.231	33.74 %	396
4	0.050	42.62 %	499
5	0.005	15.04 %	176

The plot for gold shows five overlapping lognormal populations with parameters shown in Table 5.4-19. The upper population 1, representing 1.02 % of the data, has a mean of 5.01 g Au/t. This population can be considered erratic and can be minimized by capping at two standard deviations above the mean of population 2 or a value of 4.30 g Au/t. A total of eight samples were capped at 4.3 g Au/t.

**Table 5.4-19
Summary of Gold Distribution in Group 352**

Population	Mean Au (g/t)	Proportion	Number of Samples
1	5.01	1.02 %	12
2	2.15	2.20 %	26
3	0.93	4.15 %	49
4	0.22	52.97 %	620
5	0.01	39.66 %	464

The coefficients of variation range between 1.33 and 6.55 indicating a fair degree of variability which is also evident in the maximum values present. Lognormal cumulative probability plots were produced for each variable within each of the six main geologic domains to examine the distributions of grade within each data set and determine if capping is required and if so at what level. In all cases the grade distributions were made up of a series of overlapping lognormal populations. Using the method of partitioning of lognormal cumulative probability plots, each population can be subdivided and evaluated.

The plot for silver shows three overlapping lognormal populations with parameters shown in Table 5.4-20. The upper population 1, representing 4.6 % of the data, has a mean of 7.30 g Ag/t. This population is not considered erratic and a reasonable cap would be at two standard deviations above the mean of population 1 or a value of 11.85 g Ag/t. A total of two samples were capped at 11.85 g Ag/t.

**Table 5.4-20
Summary of Ag Distribution in Group 352**

Population	Mean Ag (g/t)	Proportion	Number of Samples
1	7.30	4.62 %	54
2	1.39	60.42 %	708
3	0.03	34.96 %	409

Group 400

Group 400 is made up of Breccia units.

The plot for copper shows five overlapping lognormal populations with parameters shown in Table 5.4-21. The upper population 1, representing 0.40 % of the data, has a mean of 2.6 % Cu. In this case the upper population does not appear to be erratic and as a result a reasonable capping level would be at two standard deviations above the mean of population 1 or a value of 4.8 % Cu. One sample required capping at 4.8 % Cu.

Table 5.4-21
Summary of Cu Distribution in Group 400

Population	Mean Cu (%)	Proportion	Number of Samples
1	2.59	3.87 %	69
2	1.07	12.16 %	218
3	0.09	59.89 %	1073
4	0.01	15.91 %	285
5	0.004	8.18 %	147

The plot for gold shows five overlapping lognormal populations with parameters shown in Table 5.4-22. The upper population (1) representing 0.33 % of the data has mean grades of 19.16 g Au/t. This population can be considered erratic and minimized by capping at two standard deviations above the mean of population 2 or a value of 15.7 g Au/t. A total of four samples were capped at 15.70 g Au/t.

Table 5.4-22
Summary of Au Distribution in Group 400

Population	Mean Au (g/t)	Proportion	Number of Samples
1	19.16	0.33 %	6
2	7.86	1.60 %	29
3	1.87	12.80 %	229
4	0.19	71.49 %	1281
5	0.02	13.79 %	247

The plot for silver shows four overlapping lognormal populations with parameters shown in Table 5.4-23. The upper population 1, representing 1.23 % of the data, has a mean of 33.74 g Ag/t. A reasonable cap would be at two standard deviations above the mean of population 1, a value of 79.42 g Ag/t. Two samples were capped at 79.42 g Ag/t.

Table 5.4-23
Summary of Ag Distribution in Group 400

Population	Mean Ag (g/t)	Proportion	Number of Samples
1	33.74	1.23 %	22
2	9.99	8.75 %	157
3	1.72	75.93 %	1361
4	0.13	14.09 %	252

Composites

Drill holes within the Southwest Zone were “passed through” the geologic solids with the point each entered and left each solid recorded. Uniform down hole 5 m composites were produced

that honoured the domain boundaries. Intervals at the boundary of solids were combined with adjoining composites if less than 2.5 m and left as is if greater or equal to 2.5 m. In this manner the composites formed a uniform support of 5 ± 2.5 m. The statistics for composites are summarized in Table 5.4-24. The combination of capping and smoothing over 5 m has reduced all coefficients of variation.

**Table 5.4-24
Summary of Statistical Parameters for 5 m Composites**

Domain	Variable	Number	Standard		Minimum	Maximum	Coefficient Of Variation
			Mean	Deviation			
Group 352 I9	Cu (%)	731	0.202	0.343	0.001	2.57	1.70
	Au (g/t)	731	0.276	0.476	0.001	4.22	1.73
	Ag (g/t)	731	1.23	1.67	0.001	8.93	1.36
Group 400 Breccia	Cu (%)	939	0.286	0.550	0.001	4.24	1.92
	Au (g/t)	939	0.615	1.357	0.001	12.05	2.20
	Ag (g/t)	939	2.93	4.24	0.001	42.16	1.45

Variography

Pair-wise relative semivariograms were produced for each variable in I9 and Southwest Breccia Zones. Models were generated in the vertical direction and in four main directions within the horizontal plane (Az. 90, 0, 45 and 135). Based on these results other directions in the horizontal plane were investigated until the direction of maximum continuity was established. The vertical plane perpendicular to the maximum horizontal direction was then tested. Anisotropic nested models were fit to the data in both domains for copper and gold. Silver data, in both domains, was very hard to model and an omnidirectional spherical nested model was used. The parameters for all models are shown below in Table 5.4-25.

Geologic Block Model

A block model of 25 x 25 x 15 m blocks was superimposed on the geologic 3-D solids. The origin and parameters of the general Galore Creek block model, within which the Southwest Zone lies, were as follows:

Lower left corner	Easting	-	48000 E	25 m block	240 columns
	Northing	-	32000 N	25 m block	240 rows
Top Level	Elevation	-	1605	15 m block	107 levels
No Rotation					

For each block the following information was recorded:

- Percentage of block below topography
- Percentage of block below bedrock
- Percentage of block below the broken/stick surface

**Table 5.4-25
Parameters for Semivariogram models at
Galore Creek Southwest Zone**

Domain	Variable	Direction	C0	C1	C2	Range a1 (m)	Range a2 (m)
Group 352 I9	Cu	Az. 55° Dip 0	0.20	0.10	0.40	40	90
		Az. 325° Dip -65	0.20	0.10	0.40	60	150
		Az. 145° Dip -25	0.20	0.10	0.40	20	40
	Au	Az. 70° Dip 0	0.10	0.45	0.45	25	180
		Az. 340° Dip -80	0.10	0.45	0.45	30	160
		Az. 160° Dip -10	0.10	0.45	0.45	20	40
Ag	Omnidirectional	0.10	0.30	0.40	30	120	
Group 400 Breccia	Cu	Az. 135° Dip 0	0.20	0.10	0.35	20	100
		Az. 45° Dip 0	0.20	0.10	0.35	10	30
		Az. 0° Dip -90	0.20	0.10	0.35	25	100
	Au	Az. 135° Dip 0	0.20	0.20	0.40	20	100
		Az. 45° Dip 0	0.20	0.20	0.40	20	40
		Az. 0° Dip -90	0.20	0.20	0.40	20	150
Ag	Omnidirectional	0.15	0.20	0.35	30	120	

- The breccia solid was coded first if any proportion was within a block to OREPC1
- Next the I9 solid proportion in empty blocks was stored as OREPC1
- Finally in blocks containing some, but less than 100 % Breccia, the percentage of Solid 352 was stored as OREPC2

Bulk Density

A total of 563 specific gravity 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 amount of water it displaced. In April 1992 a total of 96 specific gravity determinations were made from drill core. During the 2003-2004 drill programs a further 30 specific gravity determinations were made. The SG determinations were made from small pieces of core at recorded distances down the hole. Using these distances the specific gravity values were joined to the assay from-to intervals that contained them. In a number of instances more than one SG determination was made on the same from-to interval. In these cases the SG values were averaged. As a result 631 assays had specific gravity determinations, across the total property.

For the Southwest Deposit, I9 unit the average specific gravity of 129 measurements for I9 or 2.63 was used. On the entire Galore Creek property a total of 39 samples were measured in Breccia with an average specific gravity of 2.68. These values were used for blocks below the broken/stick rock surface. Material above this boundary was severely broken and as a result a

general 5% reduction in specific gravity was applied. Blocks completely above the surface were given a specific gravity of 2.50 in the I9 domain and 2.55 in Breccia. Blocks straddling the boundary were given a weighted average specific gravity based on the percentage of volume above and below the broken rock surface.

Block Interpolation

Ordinary kriging was completed in four passes for each variable in each domain. A minimum four and maximum 16 composites were used for each pass. The search ellipse was expanded for each pass, starting at 1/4 the ranges in each direction for pass 1, to 2/3 in pass 2, to the full ranges in pass 3 and finally twice the range in pass 4. As the transition of grades across the I9 – Breccia boundary was gradual this boundary was considered “soft” with composites allowed to influence blocks on either side.

The kriging process was completed in the following order.

- All blocks with some proportion below the bedrock surface and proportion of primary solid (OREPC1) greater than 0.0 were estimated for each variable in separate runs using 4 separate passes in each run. Results to Cu1, Au1 and Ag1 fields.
- All blocks with some proportion below bedrock and proportion of OREPC2 > 0.0 were estimated for each variable using appropriate composites with four separate passes. Results to Cu2, Au2 and Ag2 fields.
- Grades were then combined to form weighted averages for each block based on the proportion of each domain in the block.
- SG used was 2.63 in the I9 Domain for unbroken or stick rock and 5 % less or 2.50 for broken rock. Within the Breccia Domain a value of 2.68 was used for stick rock and 2.55 for broken rock. Blocks containing both were adjusted to form a weighted average based on proportions domains and of stick and broken rock present.
- Copper equivalent was estimated using the following procedure:
 - Copper price of US\$ 0.90/lb.
 - Copper Recovery = $(\%Cu - 0.06) / \%Cu$ with a minimum of 50 % and maximum of 95 %
 - Gold price of US\$ 375/oz
 - Gold Recovery = $(Au\ g/t - 0.14) / Au\ g/t$ with a minimum of 30 % and maximum of 80 %
 - Silver price of US\$ 5.50/oz
 - Silver Recovery = 80 %

Classification

Based on the study herein reported, delineated mineralization of the Galore Creek Southwest Zone is classified as a resource according to the National Instrument 43-101 definition.

Geologic continuity on the Southwest Zone has been established through surface mapping and diamond drilling over a number of years. Grade continuity has been measured through the use of

semivariograms. Using the ranges from the semivariograms for both copper and gold, a classification scheme was devised for the Southwest Zone.

- Measured Blocks estimated in Pass 1 (using ¼ of the semivariogram ranges) for both Cu and Au.
- Indicated Blocks not classified and estimated in at least pass 2 (using 2/3 of the semivariogram ranges) for both Cu and Au.
- Inferred All other blocks estimated.

The results have been presented at a range of CuEq cutoffs in Appendix 5-B as at this time no economic evaluation has been completed. A summary for three CuEq cutoffs is presented below (Tables 5.4-26 and 5.4-27).

**Table 5.4-26
Grade Tonnage for Southwest Zone
Classed Measured Plus Indicated**

Cutoff (CuEq %)	Tonnes > Cutoff (tonnes)	Grade > Cutoff				Million lbs. of Cu	Million Ozs of Au	Million Ozs of Ag
		Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)			
0.35	47,700,000	0.447	0.818	3.041	0.972	470.15	1.25	4.66
0.50	34,100,000	0.553	1.005	3.523	1.194	415.80	1.10	3.86
1.00	16,100,000	0.843	1.477	4.787	1.738	299.27	0.76	2.48

**Table 5.4-27
Grade Tonnage for Southwest Zone Classed Inferred**

Cutoff (CuEq %)	Tonnes > Cutoff (tonnes)	Grade > Cutoff				Million lbs. of Cu	Million Ozs of Au	Million Ozs of Ag
		Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)			
0.35	122,900,000	0.314	0.556	2.274	0.695	850.92	2.20	8.99
0.50	72,100,000	0.407	0.706	2.617	0.895	647.05	1.64	6.07
1.00	18,400,000	0.714	1.092	3.245	1.445	289.68	0.65	1.92

5.4.3 Junction Zone

5.4.3.1 Data Analysis

A total of 55 drill holes were provided for analysis containing 248 down hole survey measurements and 4,645 assays. Lithology codes were provided as shown in Appendix 5-D, *Lithologic Codes*. A total of 132 possible lithology codes have been devised for the Galore Creek Project. These codes for lithologies present at Junction have been simplified to the following list.

- Sediments (Code 160)
- Volcanics V3 (Code 230)
- Volcanics V4, V4F, V4H, V5, V5A, V5D, V5H V5h (Codes 240-258)
- Intrusives i, i5, i9, i9B, i6, i8 (Codes 300 – 362)
- Syenite (Codes 352-355)
- Junction Porphyry (Codes 367-368)
- Breccia (Codes 400-430)
- Dykes (Codes 510-540)

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

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

As a result of this audit a total of 4,510 samples had copper assays while 3,954 had gold assays. Statistics for Cu, Au and Ag within these groups are shown below in Table 5.4-28 for North Junction and in Table 5.4-29 for South Junction.

Geologic modelling did not allow for such a detailed lithological interpretation. Instead, lithologies were combined to form two mineralized zones separated by a sill in North Junction (called North Junction Upper and Lower) and a single mineralized zone in South Junction. The North and South junction solids are separated by a small septum of waste.

Individual drill holes were compared to these three solids with the point at which drill hole entered and left each solid recorded. Based on these intervals individual assays were tagged with the solid designation. Lognormal cumulative probability plots were produced for copper and gold within each of the three domains. Individual overlapping populations for each variable within each domain were determined and used to cap erratic high values if present (Table 5.4-30).

Table 5.4-28
Summary of Statistics for Cu, Au and Ag in North Junction Drill Hole
Samples Sorted by Lithology

Cu % Codes	Sediments 160	V3 Volcanics 230	Volcanics 240-258	Intrusives 300-351 361-362	Syenite 352-355	Junction Porphyry 367-368	Breccia 400-430	Dykes 510-540
Number	43	66	1443	528	305	298	25	288
Mean	0.041	1.54	0.567	0.234	0.183	0.515	0.303	0.218
S.D.	0.039	1.49	1.034	0.531	0.466	1.00	0.566	0.558
Min.	0.010	0.001	0.001	0.001	0.001	0.001	0.001	0.001
Max.	0.230	6.05	8.88	4.83	3.66	7.25	2.70	4.28
C.V.	0.93	0.97	1.82	2.27	2.55	1.95	1.87	2.56

Au (g/t) Codes	Sediments 160	V3 Volcanics 230	Volcanics 240-258	Intrusives 300-351 361-362	Syenite 352-355	Junction Porphyry 367-368	Breccia 400-430	Dykes 510-540
Number	43	66	1443	528	305	298	25	288
Mean	0.019	0.805	0.232	0.141	0.124	0.428	0.186	0.088
S.D.	0.017	1.039	0.569	0.428	0.193	1.231	0.309	0.195
Min.	0.001	0.001	0.001	0.001	0.001	0.001	0.010	0.001
Max.	0.230	5.290	7.16	6.71	37.82	7.37	1.12	1.03
C.V.	0.92	1.29	2.46	3.03	1.56	2.88	1.66	2.22

Ag (g/t) Codes	Sediments 160	V3 Volcanics 230	Volcanics 240-258	Intrusives 300-351 361-362	Syenite 352-355	Junction Porphyry 367-368	Breccia 400-430	Dykes 510-540
Number	43	66	1443	528	305	298	25	288
Mean	0.368	13.78	2.86	1.463	1.75	5.51	1.82	1.78
S.D.	0.484	14.70	6.41	3.571	5.18	10.72	2.04	5.05
Min.	0.001	0.001	0.001	0.001	0.001	0.001	0.03	0.001
Max.	2.500	59.20	45.63	32.67	37.82	30.86	8.80	31.00
C.V.	1.32	1.07	2.24	2.44	2.95	1.95	1.12	2.83

Table 5.4-29
Summary of Statistics for Cu, Au and Ag in
South Junction Drill Hole Samples Sorted by Lithology

Cu % Codes	Volcanics 240-258	Junction Porphyry 367	Intrusives 300-362 371-390	Breccia 400	Dykes 520-540
Number	659	147	489	72	60
Mean	0.284	0.161	0.188	0.263	0.106
S.D.	0.419	0.168	0.259	0.282	0.229
Min.	0.001	0.001	0.001	0.040	0.001
Max.	3.88	0.990	1.89	1.55	1.61
C.V.	1.47	1.04	1.37	1.07	2.17

(continued)

Table 5.4-29
Summary of Statistics for Cu, Au and Ag in
South Junction Drill Hole Samples Sorted by Lithology
(completed)

Au (g/t) Codes	Volcanics 240-258	Junction Porphyry 367	Intrusives 300-362 371-390	Breccia 400	Dykes 520-540
Number	659	147	489	72	60
Mean	0.151	0.056	0.129	0.251	0.081
S.D.	0.629	0.067	0.258	0.254	0.110
Min.	0.001	0.001	0.001	0.010	0.001
Max.	10.88	0.350	2.15	0.690	0.34
C.V.	4.16	1.19	1.99	1.01	1.36

Ag (g/t) Codes	Volcanics 240-258	Junction Porphyry 367	Intrusives 300-362 371-390	Breccia 400	Dykes 520-540
Number	659	147	489	72	60
Mean	1.54	0.537	0.947	0.575	1.03
S.D.	2.42	1.56	1.19	1.001	1.30
Min.	0.001	0.001	0.001	0.030	0.001
Max.	20.10	13.71	6.20	2.74	5.50
C.V.	1.57	2.89	1.26	1.74	1.26

Table 5.4-30
Capping Levels Sorted by Domain

Variable	North Junction Upper			North Junction Lower			South Junction		
	Cap	Level	# Capped	Cap	Level	# Capped	Cap	Level	# Capped
Cu %	2/2	1.29	3	2/1	6.24	0	2/2	2.50	1
Au (g/t)	2/1	2.23	0	2/1	8.90	0	2/1	1.07	2
Ag (g/t)	No capping required								

Where 2/1 – 2 Standard Deviations above the mean of population 1
 2/2 – 2 Standard Deviations above the mean of population 2

Composites

A reasonable mining bench at this stage of exploration would be 15 m. Assays within mineralized intervals were taken over a variety of sample lengths from 0.1 m to 12 m intervals with the most 54 % (2,518 out of the 4,645 samples with assays) taken at 3± .3 m intervals. For this estimate a 5 m composite length was chosen to best reflect mining selection and minimize excess smoothing down the hole.

Drill holes were “passed through” the 3D geologic solid models with the point the hole entered and left a solid recorded. Uniform down-hole 5 m composites were produced for Cu, Au and Ag that honoured the boundaries of the solid models (Table 5.4-31). Composites less than 2.5 m

found at the boundaries were joined to adjacent samples to produce a uniform support of 5 ± 2.5 m lengths.

**Table 5.4-31
Summary of Statistics for 5 m Composites Within Domains**

	South Junction Domain			North Upper Domain			North Lower Domain		
	Cu (%)	Au (g/t)	Ag (g/t)	Cu (%)	Au (g/t)	Ag (g/t)	Cu (%)	Au (g/t)	Ag (g/t)
Number of Composites	178	177	150	50	41	29	547	453	359
Mean	0.470	0.182	2.45	0.393	0.498	3.68	0.991	0.502	9.05
Standard Deviation	0.433	0.178	2.56	0.283	0.599	1.819	1.182	0.880	10.68
Minimum	0.001	0.001	0.001	0.010	0.010	0.030	0.001	0.001	0.001
Maximum	2.368	0.690	13.71	1.105	1.920	7.20	8.098	7.370	47.32
Coefficient of Variation	0.92	0.98	1.04	0.72	1.20	0.49	1.19	1.75	1.18

Semivariogram Analysis

Pair-wise relative semivariograms were produced for each variable in the Domains, Lower North Junction, South Junction and waste. There was insufficient data in the North Junction Upper domain to model so for this domain, the models developed for North Lower Junction were used. In each case the horizontal plane was examined first to find the direction of maximum continuity.

Once this direction was established the perpendicular vertical plane was evaluated. In all cases nested spherical models were fit to the data. The nugget to sill ratio, a measure of data variability, ranged from 10.5% to 18% for copper, from 19% to 28% for gold and from 3% to 9% for silver. All of these are quite reasonable ratios indicating low nugget effects and low sampling variability. A summary of semivariogram parameters is presented in Table 5.4-32.

**Table 5.4-32
Parameters for Semivariogram Models at Junction**

Domain	Variable	Direction	C0	C1	C2	Range	
						a1 (m)	a2 (m)
North Junction Lower	Cu	Az. 225° Dip -45	0.10	0.35	0.50	15	100
		Az. 135° Dip -45	0.10	0.35	0.50	15	25
		Az. 315° Dip -45	0.10	0.35	0.50	10	60
	Au	Az. 225° Dip -45	0.30	0.40	0.50	30	80
		Az. 135° Dip -45	0.30	0.40	0.50	25	60
		Az. 315° Dip -45	0.30	0.40	0.50	45	70
	Ag	Az. 225° Dip -45	0.05	0.50	0.90	20	40
		Az. 135° Dip -45	0.05	0.50	0.90	30	130
		Az. 315° Dip -45	0.05	0.50	0.90	40	80

(continued)

**Table 5.4-32
Parameters for Semivariogram Models at Junction (completed)**

Domain	Variable	Direction	C0	C1	C2	Range	
						a1 (m)	a2 (m)
South Junction	Cu	Az. 220° Dip -45	0.10	0.25	0.30	30	100
		Az. 130° Dip -45	0.10	0.25	0.30	30	60
		Az. 310° Dip -45	0.10	0.25	0.30	10	60
	Au	Az. 220° Dip -45	0.20	0.10	0.40	30	60
		Az. 130° Dip -45	0.20	0.10	0.40	30	60
		Az. 310° Dip -45	0.20	0.10	0.40	30	60
	Ag	Az. 220° Dip -45	0.10	0.40	0.60	30	50
		Az. 130° Dip -45	0.10	0.40	0.60	60	80
		Az. 310° Dip -45	0.10	0.40	0.60	40	50
Junction	Cu	Omnidirectional	0.2	0.4	0.31	48	150
Waste	Au	Omnidirectional	0.2	0.5	0.35	50	150
Rock	Ag	Omnidirectional	0.05	0.60	0.65	80	150

Bulk Density

There were no specific gravity measurements taken on Junction. On the adjacent Galore Creek property during the 1966-67 drill campaign a total of 563 specific gravity measurements were made by measuring the weight of the sample and dividing by the amount of water it displaced. In April 1992 a total of 96 specific gravity determinations were made from drill core. During the 2003-2004 drill programs a further 30 specific gravity determinations were made. The SG determinations were made from small pieces of core at recorded distances down the hole. Using these distances the specific gravity values were joined to the assay from-to intervals that contained them. In a number of instances more than one SG determination was made on the same from-to interval. In these cases the SG values were averaged. As a result 631 assays had specific gravity determinations, across the total property. The specific gravity information can be sorted a number of ways. Table 5.4-33 shows average values for the various lithologies coded.

**Table 5.4-33
Summary of Specific Gravity Measurements from
Galore Creek Property**

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

On Junction Zone the principal rock type was volcanics so a specific gravity of 2.67 was chosen. This value was used for blocks below the broken/stick rock surface. Material above this

boundary was severely broken and as a result a general 5 % reduction in specific gravity was applied. Blocks completely above the surface were given a specific gravity of 2.54. Blocks straddling the boundary were given a weighted average specific gravity based on the percentage of volume above and below the broken rock surface.

Block Model

A block model consisting of blocks 25 x 25 x 15 m in dimension was superimposed on the three dimensional solids model. Blocks were coded as inside or outside the solid model based on the centroids of the block. The block model had the following parameters:

Model Minimum	Model Maximum	Block Size	# of Blocks
48400 E	50400 E	25	80
34900 N	36800 N	25	76
600	1800	15	80

Note. Coordinates used were UTM East – 300000 and UTM North – 6300000 to make more workable units.

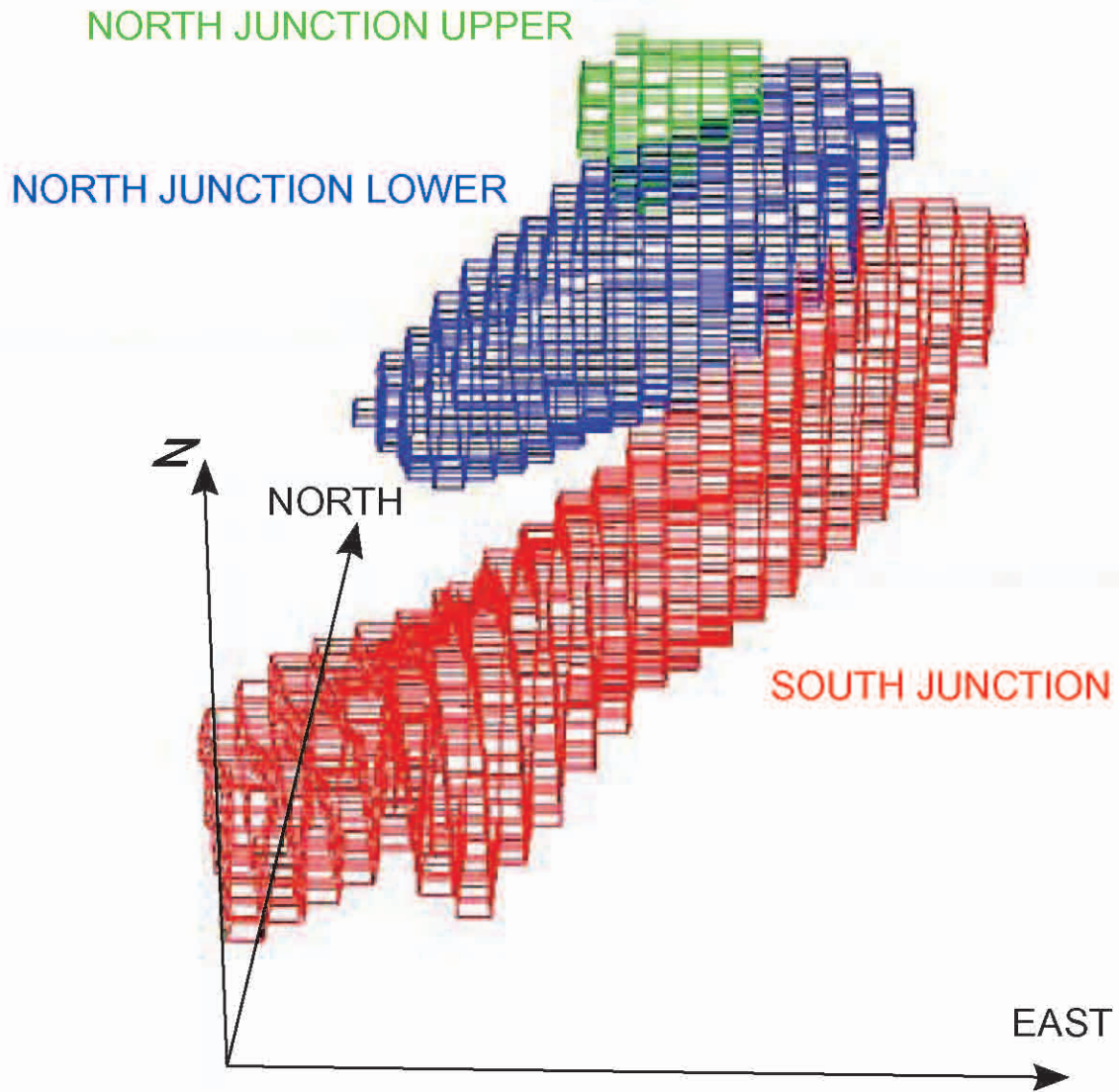
The model was not rotated.

Blocks were compared to the topographic surface and the overburden/bedrock surface and the percentage of each block in rock was recorded. Blocks were also evaluated relative to the stick/broken rock surface with the proportion of blocks below this surface recorded. Tonnage for each block was established by percentage of block below the bedrock surface multiplied by block volume and specific gravity. An isometric view showing the various domains as coloured blocks is shown below as Figure 5.4-2.

Block Interpolation

Ordinary kriging was completed in four passes for each variable in each domain. A minimum four and maximum 16 composites were used for each pass. The search ellipse was expanded for each pass, starting at ¼ the ranges in each direction for pass 1, to ½ in pass 2, to the full ranges in pass 3 and finally twice the range in pass 4. The kriging process was completed in the following order.

- All blocks with some proportion below the bedrock surface and proportion of primary solid (PCORE1) greater than 0.0 were estimated for each variable in separate runs using the appropriate composites for domains with four separate passes in each run. Results to Cu1, Au1 and Ag1 fields.
- All blocks with some proportion below bedrock and proportion of waste (BDRK – PCORE1) > 0.0 were estimated for each variable using composites (for waste) with four separate passes. Results to CuW, AuW and AgW fields.
- Grades were then combined to form weighted averages for each block based on the proportion of each domain in the block.



Isometric View of Block Model
Showing Three Domains

FIGURE 5.4-2

- SG used was 2.67 from the volcanics measured at Galore Creek Main zone for stick rock and 5 % less or 2.54 for broken rock. Blocks containing both were adjusted to a weighted average based on proportions of stick and broken rock present.
- Copper equivalent was estimated using the following procedure:
 - copper price of US\$ 0.90/lb.;
 - copper recovery = (%Cu-0.06)/%Cu with a minimum of 50 % and maximum of 95 %;
 - gold price of US\$ 375/oz;
 - gold recovery = (Au g/t-0.14)/Au g/t with a minimum of 30 % and maximum of 80 %;
 - silver price of US\$ 5.50/oz; and
 - silver recovery = 80 %.

Classification

Based on the study herein reported, delineated mineralization of the Galore Creek Junction Zone is classified as a resource according to the National Instrument 43-101 definition.

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

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

The results have been presented at a range of CuEq cutoffs in Appendix 5-B as at this time no economic evaluation has been completed. A summary at three of these cutoffs is presented below.

**Table 5.4-34
Grade – Tonnage for Junction Zone Classed Measured Plus Indicated**

Cutoff (CuEq %)	Tonnes > Cutoff (tonnes)	Grade > Cutoff				Million lbs. of Cu	Million Ozs of Au	Million Ozs of Ag
		Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)			
0.35	30,000,000	0.589	0.412	4.777	0.837	389.62	0.40	4.61
0.50	18,300,000	0.794	0.513	6.344	1.110	320.39	0.30	3.73
1.00	7,100,000	1.347	0.713	10.617	1.795	210.88	0.16	2.42

**Table 5.4-35
Grade – Tonnage for Junction Zone Classed Inferred**

Cutoff (CuEq %)	Tonnes > Cutoff (tonnes)	Grade > Cutoff				Million lbs. of Cu	Million Ozs of Au	Million Ozs of Ag
		Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)			
0.35	71,600,000	0.532	0.286	3.334	0.685	839.91	0.66	7.67
0.50	34,600,000	0.759	0.381	4.559	0.977	579.06	0.42	5.07
1.00	10,500,000	1.334	0.588	7.533	1.686	308.85	0.20	2.54

Inverse Distance Squared Test

As a test of the interpolation procedure, a second run was made using inversed distance squared (IDS). The same elliptical search parameters, used for kriging, were used for the IDS run. The minimum 4 and maximum 16 composite limits were used. As with kriging, the various domains were estimated in a series of passes with expanding search ellipses. The results are presented in Appendix 5-C, for the classification categories of measured plus indicated and inferred.

The results are very similar with differences occurring due to the weighting techniques employed by the different interpolation methods.

5.4.4 West Fork Zone

5.4.4.1 Data Analysis

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

Code	Description
200-250	Volcanics including Orthoclase-bearing (V3), crystal lithic tuff (V3h), and intermediate undivided (V5)
300-352/	Intrusives including:
371-380	Pseudoleucite Porphyry (I1), dark orthoclase syenite (I4), fine-grained orthoclase syenite megaporphyry (I5), medium-grained orthoclase syenite megaporphyry (I9), fine-grained syenite porphyry (I7), medium-grained syenite porphyry (I11) and plagioclase syenite porphyry (I10a)
362	Medium Grained Syenite (I8)
369	West Fork Porphyry (WFP)
400-435	Breccia including Diatreme (B1), Heterolithic Diatreme (B1b), Hydrothermal(B2), Hydrothermal Monolithic (B2b), Orthomagmatic (B3), Orthomagmatic Heterolithic (B3b)

500-540 Dikes including lamprophyre (D1), intermediate dikes (D3) and felsic dikes (D4)

Simple statistical parameters for each group are shown below in Tables 5.4-36, 5.4-37 and 5.4-38.

**Table 5.4-36
Statistics for Copper in West Fork Lithologies**

Lithology	Volcanics	Intrusives	I8 Intrusive	W.Fork Porphyry	Breccias	Dykes
Codes	(200-250)	(300-380)	(362)	(369)	(400-435)	(500-540)
Number	251	1,460	261	163	844	184
Mean Cu %	2.146	0.264	0.215	0.478	0.402	0.044
S.D. Cu %	6.147	0.436	0.343	0.612	1.537	0.149
Minimum Cu %	0.001	0.001	0.004	0.002	0.001	0.001
Maximum Cu %	36.000	2.78	1.98	2.37	35.30	1.26
Coef. of Variation	2.86	1.65	1.59	1.28	3.82	3.41

**Table 5.4-37
Statistics for Gold in West Fork Lithologies**

Lithology	Volcanics	Intrusives	I8 Intrusive	W.Fork Porphyry	Breccias	Dykes
Codes	(200-250)	(300-380)	(362)	(369)	(400-435)	(500-540)
Number	251	1,460	261	163	844	184
Mean Au (g/t)	0.363	0.264	0.235	0.363	0.287	0.057
S.D. Au (g/t)	1.051	0.436	0.474	0.499	0.558	0.155
Minimum Au (g/t)	0.002	0.001	0.002	0.014	0.002	0.001
Maximum Au (g/t)	9.69	2.78	5.94	2.77	6.70	0.69
Coef. of Variation	2.89	1.65	2.01	1.37	1.94	2.71

**Table 5.4-38
Statistics for Silver in West Fork Lithologies**

Lithology	Volcanics	Intrusives	I8 Intrusive	W.Fork Porphyry	Breccias	Dykes
Codes	(200-250)	(300-380)	(362)	(369)	(400-435)	(500-540)
Number	251	1,460	261	163	844	184
Mean Ag (g/t)	13.07	2.44	2.68	4.65	2.65	0.306
S.D. Ag (g/t)	38.82	4.47	7.30	8.33	9.13	0.661
Minimum Ag (g/t)	0.030	0.001	0.03	0.03	0.03	0.001
Maximum Ag (g/t)	222.0	65.3	63.7	38.4	206.0	6.8
Coef. of Variation	2.96	1.82	2.72	1.79	3.45	2.16

The Opulent Vein was not divided out in the above tabulations leading to the high copper values and averages in Volcanics and Breccias. NovaGold geologists next tried to model each lithology

in three dimensional space and found many could not be individually represented. It appears from the statistics that volcanics, intrusives and breccias are all mineralized in this area and could be lumped together if necessary. The four units that modelled well were the Opulent Vein, the I8 intrusive, the Lamprophyre dyke and the West Fork Porphyry. The remaining rock units northeast of the Opulent Vein were combined to form a geologic domain that appeared to dip steeply while those units SW of the Opulent Vein were combined into a more tabular geologic domain.

These geologic domains were then examined for copper, gold and silver using lognormal cumulative frequency plots to determine if capping was required and if so at what levels (Table 5.4-39). The Opulent Vein, which was highly mineralized and the Lamprophyre Dyke which was not mineralized were separated out while the remaining mineralized zones were combined.

**Table 5.4-39
Summary of Statistics for Geologic Domains**

Geologic Domain		Cu (%)	Au (g/t)	Ag (g/t)
West Fork Steeply Dipping Mineralized Zone NE of Opulent Boundary	Number	1,560	1,552	1,553
	Mean	0.275	0.224	2.23
	S.D.	0.429	0.357	3.69
	Minimum	0.001	0.002	0.03
	Maximum	2.780	2.960	46.80
	C.V.	1.56	1.59	1.66
West Fork Flat Lying Mineralized Zone SW of Opulent Boundary	Number	1,056	1,056	1,056
	Mean	0.219	0.64	2.59
	S.D.	0.377	0.309	5.97
	Minimum	0.001	0.002	0.03
	Maximum	2.36	3.54	65.3
	C.V.	1.72	1.88	2.31
Opulent Vein	Number	67	67	67
	Mean	9.490	1.874	53.64
	S.D.	9.937	1.887	66.07
	Minimum	0.06	0.114	0.03
	Maximum	36.0	9.69	222.0
	C.V.	1.05	1.01	1.23
Intrusive I8	Number	257	257	257
	Mean	0.182	0.172	1.50
	S.D.	0.291	0.397	4.21
	Minimum	0.003	0.002	0.10
	Maximum	1.980	5.940	63.7
	C.V.	1.59	2.30	2.80

(continued)

**Table 5.4-39
Summary of Statistics for Geologic Domains (completed)**

Geologic Domain	Cu (%)	Au (g/t)	Ag (g/t)
West Fork Porphyry	Number	185	185 185
	Mean	0.374	0.280 2.82
	S.D.	0.513	0.427 4.65
	Minimum	0.002	0.002 0.10
	Maximum	2.37	2.77 31.1
	C.V.	1.37	1.52 1.65
Lamprophyre Dyke	Number	67	67 67
	Mean	0.007	0.005 0.151
	S.D.	0.006	0.012 0.127
	Minimum	0.003	0.002 0.10
	Maximum	0.041	0.098 0.80
	C.V.	0.83	2.59 0.84

The mineralized zones of West Fork Steep, West Fork Flat, I8 and West Fork Porphyry were combined into one data set called West Fork Mineralized Zone (Table 5.4-40). For copper the highest population with a mean of 1.638 % represents 108 samples and cannot be considered erratic. A reasonable capping level for this population would be at two standard deviations past the mean of population 1 or a value of 2.58 % Cu. A single value at 2.78 was capped at 2.58 %. For gold the upper population 1 represented 0.41 % of the data or 13 samples. This population could be considered erratic and an effective cap would be at two standard deviations past the mean of population 2, a value of 2.56 g Au/t. A total of eight samples were capped at 2.56 g Au/t. The highest population for silver had a mean of 35.54 g Ag/t and represented 0.61 % of the data or 19 samples. A reasonable cap on silver would be at two standard deviations past the mean of population 2, a value of 39.4 g Ag/t. A total of five silver assays were capped at 39.4 g Ag/t.

**Table 5.4-40
Summary of Capping Parameters**

Zone	Variable	Cap Level	Number Capped
West Fork	Cu	2.58 %	1
Mineralized Zones	Au	2.56 g/t	8
	Ag	39.4 g/t	5
Opulent Vein	Cu	45 %	0
	Au	18.3 g/t	0
	Ag	232 g/t	0

Cap levels for the Opulent Vein were set at two standard deviations above the mean of the upper population and no samples required capping.

No samples in the Lamprophyre Dike required capping.

Composites

The drill hole assays were compared to the interpreted three dimensional solids (geologic domains) and the point at which each drill hole entered and left a particular solid was noted. Uniform down hole 5 m composites were then formed using the capped values for each geologic domain. Composites near the limits of the domain, less than 2.5 m in length, were combined with the adjacent composite to produce a uniform support of 5 ± 2.5 m. The composite statistics are shown below (Table 5.4-41).

**Table 5.4-41
Summary of Composite Statistics for Each Geologic Domain**

Geologic Domain		Composite		
		Cu (%)	Au (g/t)	Ag (g/t)
West Fork Steep	Number	666	666	666
	Mean	0.243	0.214	2.04
	S.D.	0.354	0.274	2.79
	Minimum	0.001	0.002	0.03
	Maximum	2.238	1.454	24.48
	C.V.	1.46	1.28	1.37
West Fork Flat	Number	523	523	523
	Mean	0.191	0.153	2.32
	S.D.	0.302	0.250	5.00
	Minimum	0.001	0.002	0.03
	Maximum	2.102	1.914	38.40
	C.V.	1.58	1.64	2.16
I8 Intrusive	Number	111	111	111
	Mean	0.164	0.155	1.30
	S.D.	0.239	0.160	1.76
	Minimum	0.003	0.002	0.10
	Maximum	1.748	1.045	15.22
	C.V.	1.46	1.03	1.35
West Fork Porphyry	Number	84	84	84
	Mean	0.302	0.257	2.35
	S.D.	0.399	0.364	3.61
	Minimum	0.002	0.002	0.10
	Maximum	1.819	1.919	21.63
	C.V.	1.32	1.42	1.54

(continued)

Table 5.4-41
Summary of Composite Statistics for Each Geologic Domain
(completed)

Geologic Domain	Composite		Composite		Composite	
	Cu (%)	Au (g/t)	Au (g/t)	Ag (g/t)	Ag (g/t)	Ag (g/t)
Opulent Vein	Number	21	21	21	21	21
	Mean	7.67	1.71	40.4		
	S.D.	8.56	0.99	57.0		
	Minimum	0.96	0.18	0.03		
	Maximum	27.46	4.45	164.2		
	C.V.	1.12	0.58	1.41		
Lamprophyre Dike	Number	37	37	37	37	37
	Mean	0.007	0.005	0.153		
	S.D.	0.005	0.011	0.108		
	Minimum	0.003	0.002	0.10		
	Maximum	0.029	0.064	0.56		
	C.V.	0.71	2.14	0.70		

Variography

Pair-wise relative semivariograms were produced for five of the six geologic domains. There were insufficient data points within the Lamprophyre Dike domain to model. In the case of West Fork Steep and West Fork Flat anisotropy was proven with nested spherical models fit to the three major directions of grade continuity. For the I8 intrusive, the West Fork Porphyry and the Opulent Vein there were insufficient data to disprove the assumption of isotropy so average omni-directional models were fit to the data. The parameters for all models are shown below in Table 5.4-42.

Table 5.4-42
Parameters for Semivariogram Models at West Fork

Domain	Variable	Direction	Range			Range	
			C0	C1	C2	a1 (m)	a2 (m)
West Fork Steep Domain	Cu	Az. 35° Dip 0	0.10	0.32	0.62	30	130
		Az. 305° Dip -65	0.10	0.32	0.62	45	180
		Az. 125° Dip -25	0.10	0.32	0.62	20	80
	Au	Az. 35° Dip 0	0.05	0.25	0.55	20	50
		Az. 305° Dip -65	0.05	0.25	0.55	20	150
		Az. 125° Dip -25	0.05	0.25	0.55	20	40
	Ag	Az. 35° Dip 0	0.10	0.25	0.55	15	100
		Az. 305° Dip -65	0.10	0.25	0.55	15	120
		Az. 125° Dip -25	0.10	0.25	0.55	20	40

(continued)

**Table 5.4-42
Parameters for Semivariogram Models at West Fork (completed)**

Domain	Variable	Direction	C0	C1	C2	Range	
						a1 (m)	a2 (m)
West Fork Flat Domain	Cu	Az. 45° Dip 0	0.05	0.08	0.60	20	80
		Az. 135° Dip 0	0.05	0.08	0.60	20	60
		Az. 0° Dip -90	0.05	0.08	0.60	5	85
	Au	Az. 45° Dip 0	0.15	0.25	0.40	20	60
		Az. 135° Dip 0	0.15	0.25	0.40	20	60
		Az. 0° Dip -90	0.15	0.25	0.40	20	120
	Ag	Az. 45° Dip 0	0.25	0.20	0.55	20	80
		Az. 135° Dip 0	0.25	0.20	0.55	20	80
		Az. 0° Dip -90	0.25	0.20	0.55	40	100
I8 Intrusive Domain	Cu	Omnidirectional	0.2	0.2	0.6	28	120
	Au	Omnidirectional	0.2	0.2	0.3	20	80
	Ag	Omnidirectional	0.2	0.3	0.15	25	120
West Fork Porphyry	Cu	Omnidirectional	0.2	0.1	0.55	10	100
	Au	Omnidirectional	0.25	0.10	0.30	20	80
	Ag	Omnidirectional	0.25	0.05	0.40	15	80
Opulent Vein	Cu	Omnidirectional	0.10	0.55		38	
	Au	Omnidirectional	0.10	0.35		35	
	Ag	Omnidirectional	0.40	1.00		35	

Geologic Block Model

A block model of 25 x 25 x 15 m blocks was superimposed on the geologic 3-D solids. The origin and parameters of the general Galore Creek block model, within which the West Fork deposit lies, were as follows:

Lower left corner	Easting	-	48000 E	25 m block	240 columns
	Northing	-	32000 N	25 m block	240 rows
Top Level	Elevation	-	1605	15 m block	107 levels

No Rotation

For each block the following information was recorded:

- percentage of block below topography;
- percentage of block below bedrock;
- percentage of block below the broken/stick surface;
- the most prevalent solid percentage was stored as OREPC1;
- the second most prevalent solid percentage was stored as OREPC2; and

- the percentage of OREPC1 plus OREPC2 was subtracted from the percentage of rock below the bedrock surface to give PCWESTFORK (the remaining material).

Bulk Density

Fifty-two measurements of specific gravity have been taken from the West Fork area as shown in Table 5.4-43. Blocks in the model were coded by geologic domain and each block had the proportion of broken and stick rock recorded. Stick rock or rock that is more or less in one piece was assigned the SG shown in Table 5.4-44 while broken rock was assigned the SG less 5 %. Blocks that straddled the broken/stick surface were assigned a weighted average based on relative volumes above and below this surface.

There were only three measurements from the Opulent Vein and all came from the same composite that averaged 23.6 % Cu. This was the second highest copper value in the vein and as a result was not considered representative. A reduced value for specific gravity of 3.5 was assigned to the stick portion of the Opulent Vein while 3.33 was assigned to the broken portion. Lamprophyre Dikes were assigned a value of 2.73 in stick rock and 2.59 in broken rock. All other units were assigned an SG of 2.66 in stick rock and 2.53 in broken rock.

**Table 5.4-43
Listing of Measured Specific Gravities for West Fork**

Hole	Sample Identification Number	SG	Lithology	Description
West Fork Porphyry				
GC04-0491	106379	2.60	WFP	From Chemex CertifID 667
GC04-0494	106072	2.74	WFP	From Chemex CertifID 667
	Average	2.67		
I8 Intrusives				
GC04-0474	110878	2.56	i8	From Chemex CertifID 667
GC04-0483	111374	2.74	i8	From Chemex CertifID 667
GC04-0486	106300	2.63	i8	From Chemex CertifID 667
GC04-0500	107636	2.64	i8	From Chemex CertifID 667
GC04-0500	107636	2.64	i8	From Chemex CertifID 667
GC64-0135	GC-6203	2.75	i8	From old logs - 10 ft BX = 24.7 lbs
	Average	2.66		
Volcanics, Intrusives and Breccias				
GC04-0477	111623	2.96	V3	From Chemex CertifID 667
GC04-0469	110033	2.63	i9	From Chemex CertifID 667
GC04-0472	110828	2.65	i9	From Chemex CertifID 667
GC04-0477	111649	2.65	i9	From Chemex CertifID 667
GC04-0486	106300	2.63	i9	From Chemex CertifID 667
GC04-0496	107583	2.61	i9	From Chemex CertifID 667
GC04-0496	107583	2.61	i9	From Chemex CertifID 667
GC63-0072	GC-1965	2.68	i7	From old logs – 10 ft BXW = 18.8 lbs
GC04-0498	106770	2.81	i5	From Chemex CertifID 667
GC04-0498	106770	2.81	i5	From Chemex CertifID 667

(continued)

**Table 5.4-43
Listing of Measured Specific Gravities for West Fork (completed)**

Hole	Sample Identification Number	SG	Lithology	Description
GC04-0464	109788	2.62	i4	From Chemex CertifID 667
GC04-0464	109788	2.62	i4	From Chemex CertifID 667
GC04-0472	110834	2.68	i4	From Chemex CertifID 667
GC04-0479	110990	2.67	i4	From Chemex CertifID 667
GC04-0480	111502	2.70	i4	From Chemex CertifID 667
GC04-0496	106906	2.65	i4	From Chemex CertifID 667
GC04-0500	107638	2.58	i4	From Chemex CertifID 667
GC63-0066	GC-1138	2.56	i4	From old logs – 10 ft BX = 18 lbs
GC63-0066	GC-1135	2.49	i4	From old logs – 10 ft BX = 22.4 lbs
GC63-0072	GC-1485	2.54	i4	From old logs – 10 ft BXW = 17.9 lbs
GC63-0072	GC-1955	2.85	i4	From old logs – 10 ft BXW = 20 lbs
GC63-0072	GC-1495	2.69	i4	From old logs – 10 ft BXW = 18.9 lbs
GC63-0050	GC-301	2.56	i11	From old logs – 10 ft NX = 44.2 lbs
GC63-0050	GC-313	2.48	i11	From old logs – 10 ft NX = 42.5 lbs
GC04-0496	106921	2.82	i10a	From Chemex CertifID 667
GC04-0476	106182	2.70	i1	From Chemex CertifID 667
GC64-0123	GC-5702	2.71	i	From old logs – 10 ft BX = 24.4 lbs
GC64-0127	GC-5805	2.56	i	From old logs – 10 ft BX = 23 lbs
GC04-0486	106235	2.67	B1	From Chemex CertifID 667
GC04-0464	109834	2.72	B	From Chemex CertifID 667
GC04-0464	109788	2.62	B	From Chemex CertifID 667
GC04-0464	109788	2.62	B	From Chemex CertifID 667
GC04-0483	111418	2.78	B	From Chemex CertifID 667
GC04-0494	106042	2.73	B	From Chemex CertifID 667
GC04-0496	106889	2.69	B	From Chemex CertifID 667
GC63-0066	GC-1282	2.55	B	From old logs – 10 ft BX = 17.92 lbs
GC63-0066	GC-1292	2.54	B	From old logs - 10 ft BX = 17.9 lbs
	Average	2.66		
Dykes				
GC04-0500	107675	2.75	D3	From Chemex CertifID 667
GC04-0491	106411	2.84	D1	From Chemex CertifID 667
GC04-0474	110885	2.62	D	From Chemex CertifID 667
GC04-0474	110893	2.71	D	From Chemex CertifID 667
	Average	2.73		
Opulent Vein				
GC04-0480		4.81		At 50.90
GC04-0480		4.74		At 51.54
GC04-0480		4.77		At 52.20
	Average	4.77		

**Table 5.4-44
Grade – Tonnage for West Fork Zone Classed
Measured Plus Indicated**

Cutoff (CuEq %)	Tonnes > Cutoff (tonnes)	Grade > Cutoff				Million lbs. of Cu	Million Ozs of Au	Million Ozs of Ag
		Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)			
0.35	15,100,000	0.580	0.379	4.793	0.798	193.11	0.18	2.33
0.50	10,800,000	0.694	0.438	5.675	0.949	165.27	0.15	1.97
1.00	3,300,000	1.118	0.631	9.930	1.514	81.35	0.07	1.05

Block Interpolation

Ordinary kriging was completed in four passes for each variable in each domain. A minimum four and maximum 16 composites were used for each pass. The search ellipse was expanded for each pass, starting at ¼ the ranges in each direction for pass 1, to ½ in pass 2, to the full ranges in pass 3 and finally twice the range in pass 4. The kriging process was completed in the following order.

- All blocks with some proportion below the bedrock surface and a proportion of primary solid (OREPC1) greater than 0.0 were estimated for each variable in separate runs using the appropriate composites for domains with four separate passes in each run.
- All blocks with some proportion below the bedrock surface and a proportion of secondary solid (OREPC2) greater than 0.0 were estimated for each variable in separate runs using the appropriate composites for domains with four separate passes in each run.
- All blocks with some proportion below the bedrock surface and a proportion of remaining rock (PCWESTFORK) greater than 0.0 were estimated for each variable in separate runs using the appropriate composites for domains with four separate passes in each run.
- Grades were then combined to form weighted averages for each block based on the proportion of each domain in the block.
- Copper equivalent was estimated using the following procedure:
 - copper price of US\$ 0.90/lb.;
 - copper recovery = (%Cu-0.06)/%Cu with a minimum of 50 % and maximum of 95 %;
 - gold price of US\$ 375/oz;
 - gold recovery = (Au g/t-0.14)/Au g/t with a minimum of 30 % and maximum of 80 %;
 - silver price of US\$ 5.50/oz; and
 - silver recovery = 80 %.

Classification

Based on the study herein reported, delineated mineralization of the Galore Creek West Fork Zone is classified as a resource according to National Instrument 43-101 the definition.

Geologic continuity on the West Fork property has been established through surface mapping and diamond drilling. Grade continuity has been measured through the use of semivariograms.

Using the ranges from the semivariograms for both copper and gold, a classification scheme was devised for the West Fork Deposit.

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

The results have been presented at a range of copper equivalent cutoffs in grade-tonnage tables shown in Appendix 5-C as at this time no economic evaluation has been completed. A summary for three cutoffs is shown below (Table 5.5-45).

**Table 5.4-45
Grade – Tonnage for West Fork Zone Classed Inferred**

Cutoff (CuEq %)	Tonnes > Cutoff (tonnes)	Grade > Cutoff				Million lbs. of Cu	Million Ozs of Au	Million Ozs of Ag
		Cu (%)	Au (g/t)	Ag (g/t)	CuEq (%)			
0.35	45,400,000	0.466	0.339	4.986	0.665	466.50	0.50	7.28
0.50	29,700,000	0.553	0.403	6.181	0.797	362.15	0.39	5.90
1.00	4,900,000	0.841	0.603	9.948	1.219	90.87	0.10	1.57

5.4.5 Total Resource

Overall, the known mineralized zones at Galore Creek have a measured and indicated resource, at a copper equivalent cutoff of 0.35%, of 516.7 million tonnes grading 0.596% copper, 0.358 g/t gold and 4.54 g/t silver. At a copper price of US\$0.90/lb, a gold price of US\$375/oz and a silver price of US\$5.50/oz this resource has an in situ contained value of 8.8 billion dollars. The inferred resource at a copper equivalent cutoff of 0.35% contains 413.5 million tonnes grading 0.433% copper, 0.371 g/t gold and 3.24 g/t silver.

The current mine plan is based on extracting 493.9 million tonnes of these resources.

5.5 Mining

5.5.1 Geotechnical Evaluation

BGC Engineering Inc. provided NovaGold with feasibility level open pit slope design parameters based on historical reports and field data collected in the 2004 and 2005 field seasons. The current data sources and resultant information are:

- NovaGold geology interpretation from the drill hole database used to create the mineralized zones and host rock types;
- structural mapping to model regional faults, local bedding, sills, intrusions, and contacts;

- geology logs and core samples for rock analysis and testing including petrographic analysis, point load and uniaxial compressive strength testing and direct shear testing;
- oriented core measurements (Ezy-mark, Ball mark and acoustic televiewer) for down hole structural mapping;
- 11 specific geotechnical boreholes as described in Table 5.5-1 and Figure 5.5-1;
- geophysical surveys including two shear wave lines and seven seismic refraction lines;
- airphoto interpretation / geohazard assessment of the mine site;

**Table 5.5-1
Geotechnical Boreholes Distribution**

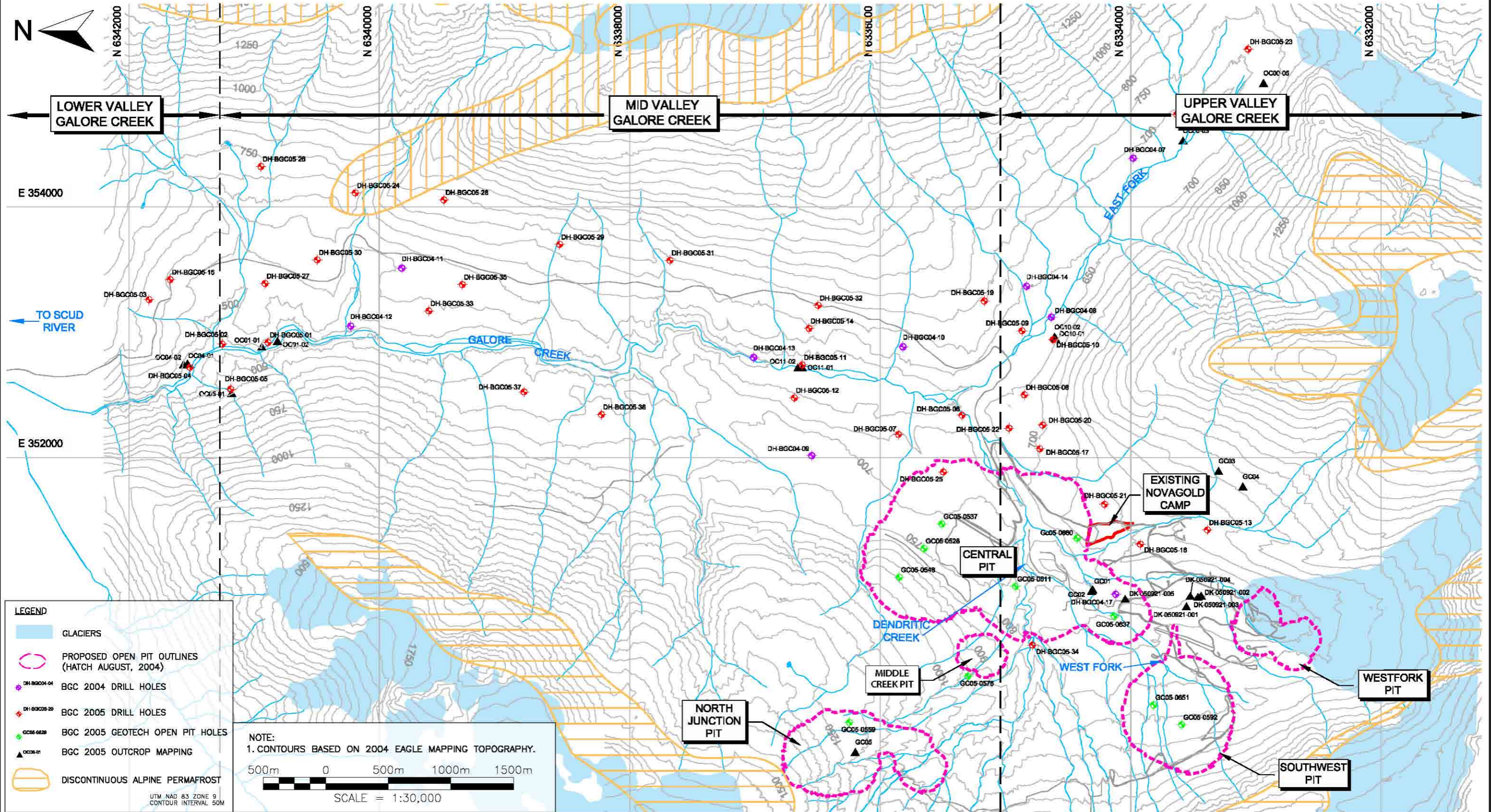
Structure (or Location)	No. of Geotechnical Boreholes
Main Tailings Dam	5
East Fork Diversion Structure	2
East Fork Aqueduct	4
Open Pit Diversion - Impervious Facing on Waste Dump	3
Main Diversion Channel	9
Dendritic Ck & West Fork Diversion	4
Tailings Pond Foundations	6
Waste Dump Foundations	8
Plant site Foundations	4
Open pits	11
Total	56

Note: Number of boreholes under each structure (or location) is approximate. Holes that do not lie directly under the footprint but are within the vicinity were considered.

- 10 piezometer installations for groundwater monitoring;
- 26 hydraulic tests (rising head, falling head, and constant flow packer tests for ground water flow; and estimates of hydraulic conductivity;

5.5.1.1 Pit Slope Stability

BGC provided design criteria for the proposed open pit slopes on the basis of structural discontinuity orientations, estimated discontinuity shear strengths, estimated rock mass strengths, and simulated achievable groundwater conditions. Bench face angles in the sheet-fractured (“broken rock”) and massive (“stick”) geotechnical units are controlled by steeply dipping discontinuities; multiple benching schemes are feasible in the sheet-fractured and massive rock. Interramp and overall slope angles are controlled by a combination of persistent structural fabric, faults, and rock mass quality.



Source: BGC Engineering Inc.

Current designs assume both vertical and horizontal dewatering wells and drains will be installed as part of mining operations. Detailed numerical groundwater simulations show that vertical perimeter wells advanced to the contact between the sheet-fractured and massive rock units and spaced evenly apart at a distance of approximately 150 m would be sufficient to dewater the sheet-fractured unit. Within the massive rock, 30 m long horizontal wells spaced approximately 20 m apart (installed on all pit benches) are predicted to be required. Horizontal wells will also be required within the sheet-fractured rock to control remaining groundwater seepage at the pit face as well as water infiltrated due to large storm events and annual spring runoff. The massive rock is assumed to have residual pore pressures which are not mitigated; the sheet-fractured rock is assumed to be dewaterable based on numerical groundwater simulations and hydraulic conductivities measured in during field tests.

Design criteria for bench (15 and 30 metre heights), interramp and overall open pit slopes have been developed for each of the proposed open pits. A reliability criterion of 80% has been adopted for the design bench face angle distribution. A design factor of safety of 1.3 has been used for the interramp and overall slope analyses. Bench face angles have been estimated for the sheet-fractured and massive rock to be 74° – 80° for good blasting practices. Design berm widths are 8 metres for the 15 metre bench height and 11 metres for the 30 m bench height. Design interramp angles can be as steep as 60° for specific wall orientations in the massive unit for 30 m bench height configuration. The contrasts between the geotechnical units of this site require a compound overall slope to maximize the design overall angle. Typically, the achievable interramp angles in the massive unit will be greater than those in the sheet-fractured unit (for similar slope heights). Overburden slopes will be cut to 2:1 slopes with a wide berm at the base to catch sloughing or rock fall.

MineSight pit design software was used for the detailed design of the pit phases, based on the slope design criteria provided by BGC. The ultimate pit walls will intersect several streams, which will have to be diverted. Diversions for this purpose are discussed in Section 5.10.

5.5.1.2 Hazards Evaluation

BGC Engineering Inc. conducted a comprehensive assessment of geohazards (Appendix 5-E, *Galore Creek Feasibility Study Geohazard Assessment*) over the proposed mine area and the access corridor (Figure 5.5-2). The eastern half of the proposed Central Pit will be located in terrain partially covered in thick till, glaciolacustrine and glaciofluvial material. These areas are considered to have low natural landslide hazard (Figure 5.5-3). However, development of the Central Pit will require removal of thick overburden deposits that may be subject to instability during or after excavation.

The area northwest of the end of the proposed Central pit contains steep rock slopes subject to sporadic shallow rock fall; however, this slope will be significantly modified or removed during pit construction. The western portion of the proposed Central Pit is intersected by Middle Creek, which is subject to debris flows with the potential to run out to the western boundary of the Central Pit. A debris flow basin or debris barrier will be constructed at this site to prevent debris flows from entering the pit. A drainage diversion is planned at this site to divert water away from the Central Pit. The design will accommodate potential debris flow activity in upper Middle Creek, to avoid debris flows damming the diversion channel and causing channel

avulsions. In addition, construction of the North Junction pit may partially excavate the debris flow source area in Middle creek, which will affect hazard levels at this location. Further geohazard assessment will be conducted at this location as part of the 2006 program to establish detailed information for permitting.

The proposed Southwest Pit will be located below steep rockslopes subject to frequent shallow rock fall (Figure 5.5-3). Approximately the western 200 m of the ultimate pit footprint will be located within the rock fall runout zone. A catch bench and earth berm will be constructed to block unexpected large rock fall from entering the pit. Design of these measures will also take into account the potential for blocking snow avalanches from entering the pit.

Figure 5.5-3 shows the location of a berm above the west side of the proposed Southwest pit area. During winter, berm height will need to be maintained by removing snow accumulations on the upslope side. An avalanche safety protocol will be established during the development stage to permit safe working conditions during snow removal.

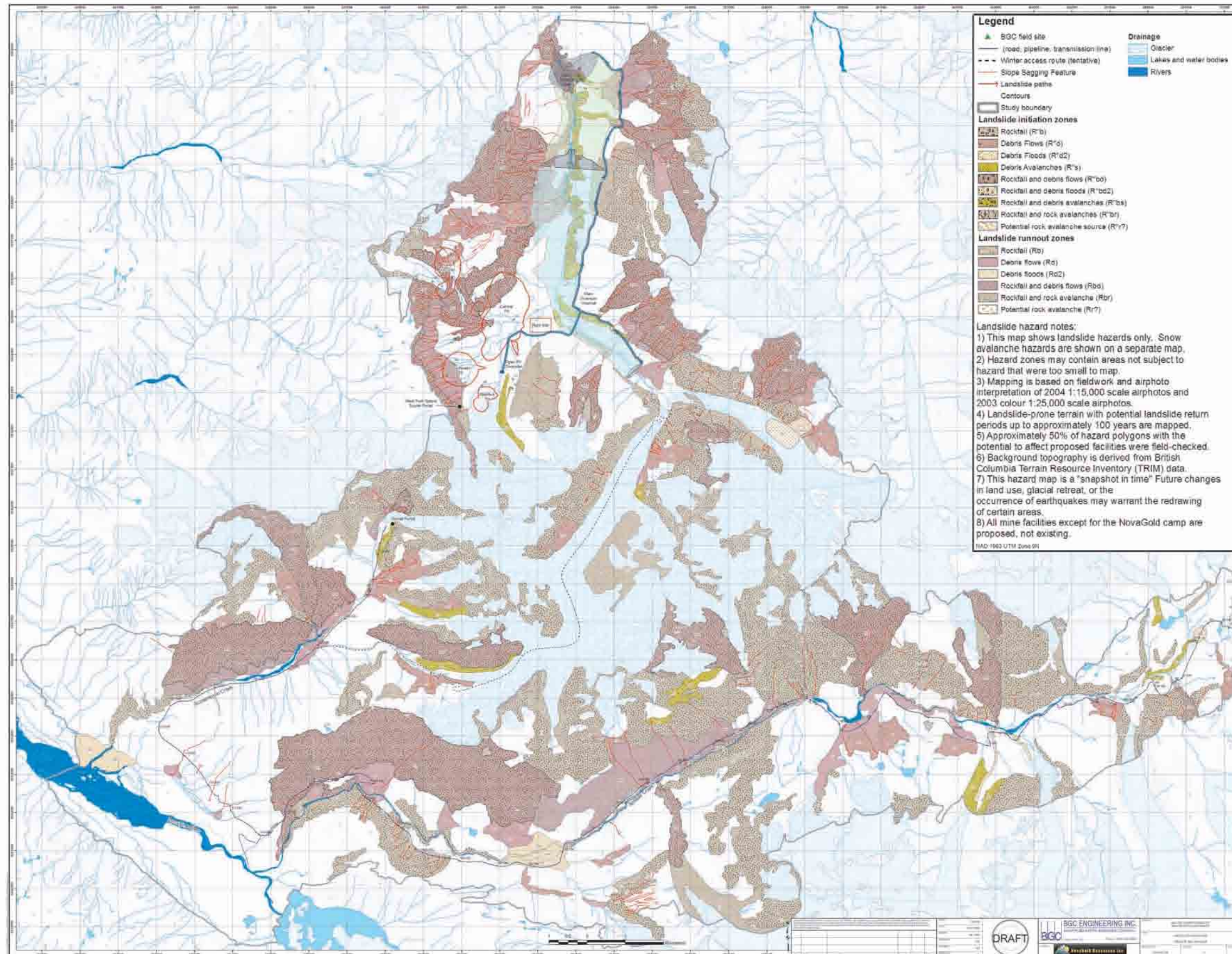
The north and west sides of the proposed North Junction Pit are located in highly fractured, gullied volcanic rock potentially subject to low magnitude rock fall and debris flows. However, excavation within the proposed footprint of the pit will significantly alter slope characteristics including removal of part of the landslide initiation zones. Further geohazard assessment will be completed at this location at the project design stage.

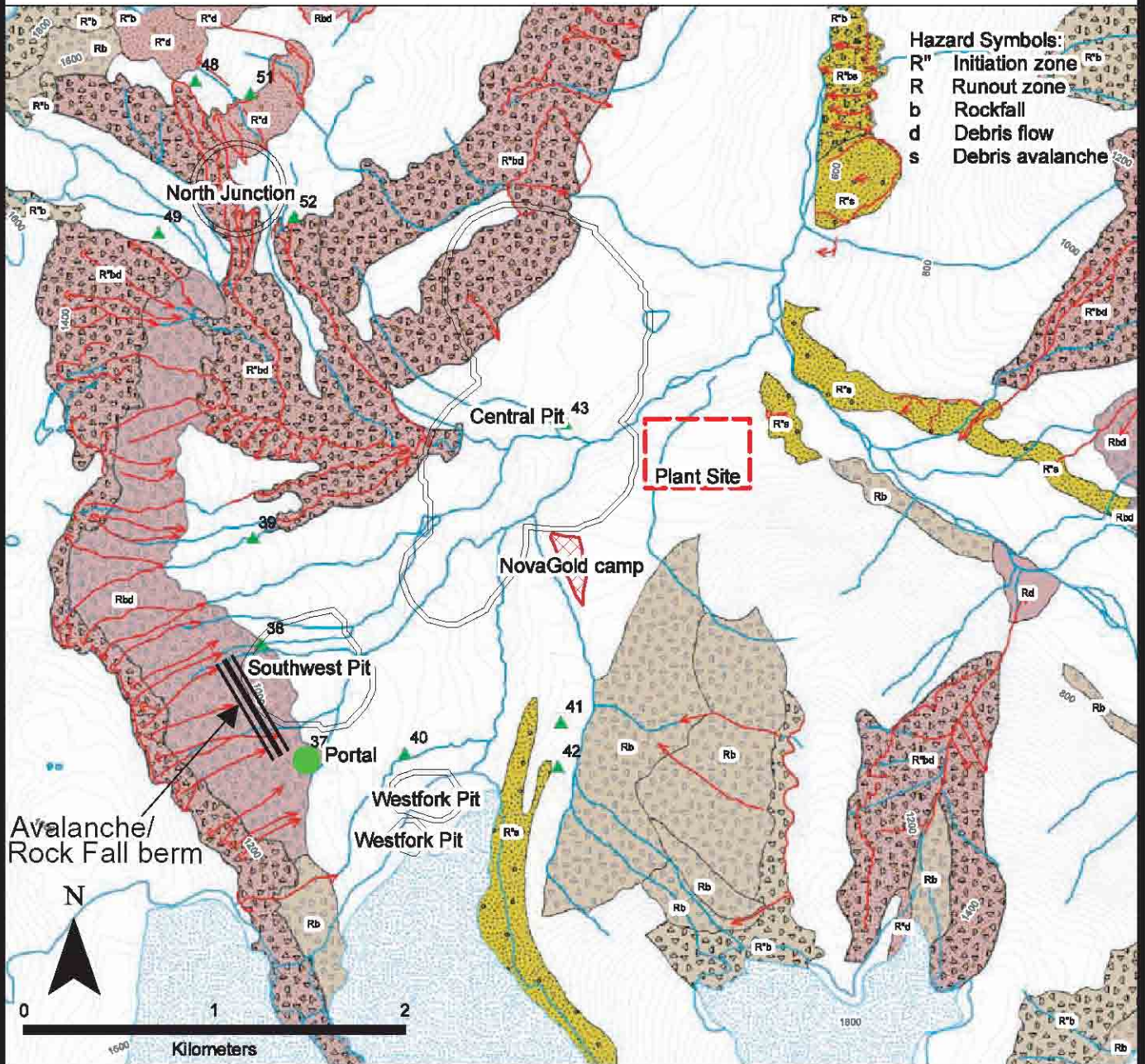
The main tailings dam will be in a steep canyon with densely jointed volcanic rock subject to shallow rock fall. The finished tailings dam will cover those slopes. The slopes above the northeast side of the main tailings dam are considered to have very low landslide and avalanche hazard. The slopes above the northwest side of the dam include a gully potentially subject to debris flows. Further investigation is required of this gully and scour protection measures may be required on the dam.

Portions of the slopes on either side of the tailings and waste impoundment are subject to rock fall and debris flows. Rock fall and debris flow events in these areas are thought to represent a low hazard for the tailings dam, but hazard mitigation will be required for roads and drainage channels.

Much of the Galore Creek Valley is subject to avalanche hazard. NovaGold engaged BGC Engineering Inc. and Chris Stethem and Associated Ltd. to assess this hazard. They created an Avalanche Atlas (Figure 5.5-4) that identified the avalanche paths in the Galore Mine area that could produce avalanches of Size 3 or greater.

These avalanches will be hazardous to facilities or vehicles (Table 5.5-2). A few smaller avalanche path crossings near known worksites (e.g. airport, tailings dam, north portal) were also identified.





Source: BGC Engineering

Landslide Hazard Map, West Fork Galore Creek

FIGURE 5.5-3

**Table 5.5-2
Avalanche Classification**

Size Code	Avalanche Destructive Potential	Typical Mass	Typical Path Length	Typical impact pressure (kPa)
1	Relatively harmless to people	< 10 t	10 m	1
2	Could bury, injure, or kill a person	10 ² t	100 m	10
3	Could bury and destroy a car, damage a truck, destroy a small building, or break a few trees	10 ³ t	1,000 m	100
4	Could destroy a railway car, large truck, several buildings, or a forest area up to 4 hectares (~10 acres)	10 ⁴ t	2,000 m	500
5	Largest snow avalanche known. Could destroy a village or a forest of 40 hectares (~100 acres)	10 ⁵ t	3,000 m	1000

Any one of the larger avalanche paths could also produce smaller avalanches (Size 2) that could be hazardous to persons on foot. These smaller avalanches will usually run-out higher on the slope than larger events.

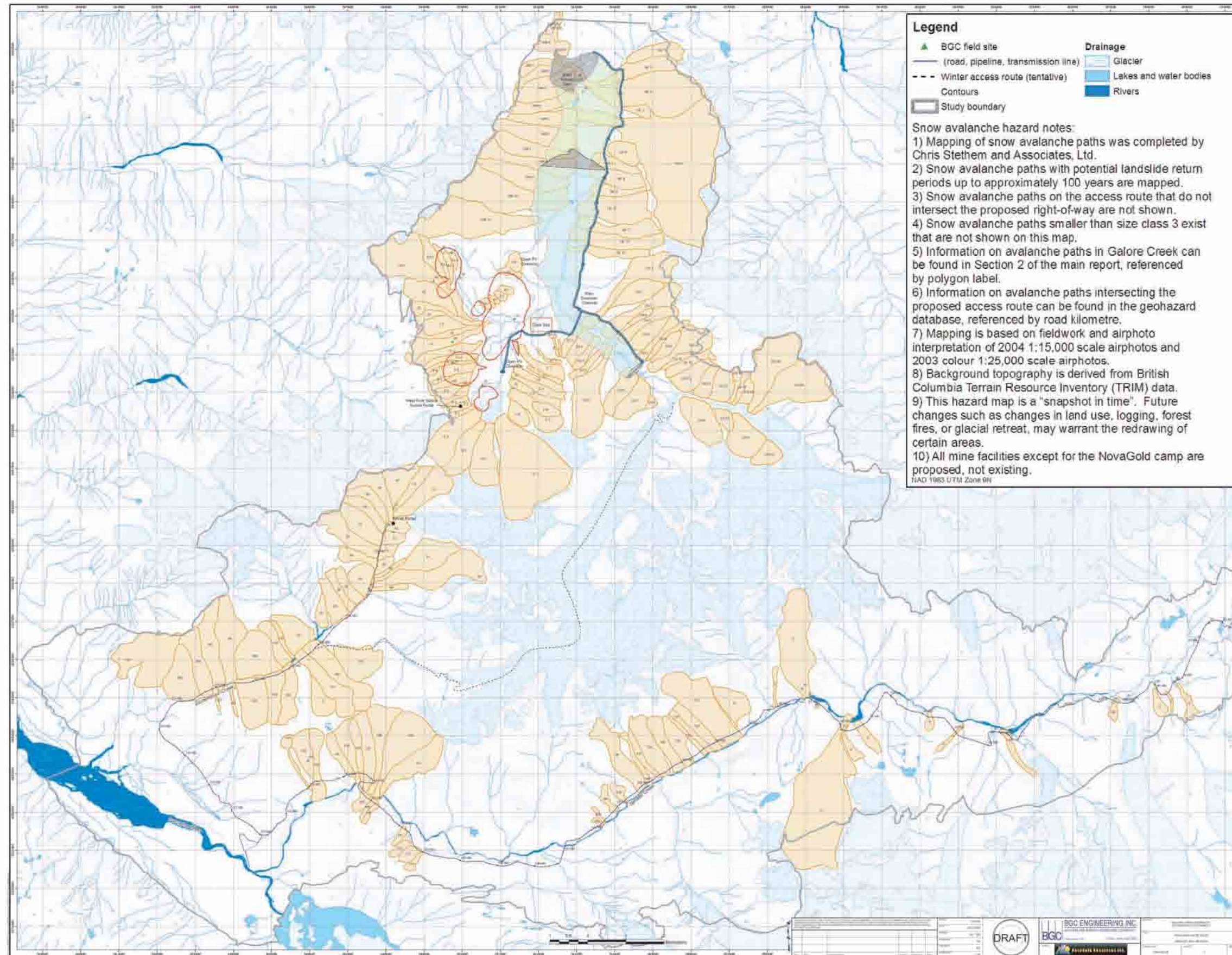
There are also numerous small terrain features in a mountain environment that could produce an avalanche hazardous to persons on foot and they might not be identified in the Mine Avalanche Atlas. Slopes with avalanche potential near worksites will be identified and managed as required during each phase of mine construction and operation.

Substantial forest borders that exist between a facility and an avalanche path should also be left intact where feasible. Forest borders near the runout zones of avalanches tend to reduce the potential runout distance of avalanches.

The electronic Galore Mine Avalanche Atlas (*BGC Galore Creek Feasibility Study Geohazard Assessment*) approximately locates and names individual avalanche paths by drainage using sequential numbering. The Atlas contains a summary of the avalanche path attributes for the avalanche paths and their estimated frequency and magnitude of effect at known facilities in the Galore Creek project area.

Many of these facilities have not been finally located and therefore the frequency and magnitude of effect can be expected to change. Generally speaking, a move up the slope will increase the frequency and a move down the slope will reduce the frequency.

Table 5.5-3 summarizes the avalanche paths and effects at known facilities planned in the Galore Creek project area. It is important to recognize that project planning continues to evolve and that the facilities affected could change with each new layout proposed. Uncertainty of location at this stage of planning means uncertainty with respect to frequency of effect, magnitude, elements at risk from avalanches and mitigation required.



**Table 5.5-3
Avalanche Hazard to Facilities at Galore Creek**

Facility	Avalanche Paths	Frequency (range events/y)	Magnitude Range	Elements at Risk	Mitigation Options
NovaGold Camp Plant Area	Camp 1 Plant 1	0.01 0	4 2	Buildings, personnel None	Diversion berm None
N Tunnel Portal	P 2, P3	10-0.1	2-3	Vehicles, Maintenance	Portal extension; road on fill; explosives
S Tunnel Portal	Km 128.8	6-0.1	2-3	Vehicles, Maintenance	Portal extension; explosives
Tailings Dam	GW 1, 2, 3	≥ 3-0.1	2-4	Construction operations	Temp. closure; explosives
Tailings Pond ¹	GW 3, 4, 6 and GE 3, 5	≥1-0.03	3-4	Wave generation	Dam design; explosives
Central Pit	A 1, 2, 3, 4, 5	1-0.5	2	Personnel	Avoidance Explosives
SW Pit	P 6, 7, 8, 9, 10, 11	>1-0.1	2-3	Personnel, equipment	Temp. Closure; explosives
N Junction Pit	NJ 2, 3, 4, 5	0.5-0.1	2-3	Personnel, equipment	Avoidance; temp. closure, explosives
West Fork Pits	G 4	0.03	4	Personnel, equipment	Avoidance; temp closure; explosives
Water Diversions	Unknown	≥1-0.03	2-4	Blockage, flooding of diversions	Cover in avalanche Paths
Galore East Fork (water, access or other)	CN 1-18 CS 1-12	≥1-0.03 ≥1-0.03	3-4 3-4	Roads, water facilities and operations	Y- Temp. closure; explosives

¹ Exposure of tailings pond will increase as fill level rises. GE 3 is closest to the dam and is the priority for explosive control to reduce avalanche size and the risk of wave generation.

Large ice avalanches might occur at the glacier 1, 2, 3 or 4 paths, which are above the West Fork Pit. The runout distance of large ice on snow avalanches is poorly understood.

Mitigation of avalanche hazards, besides avoidance of hazard areas, temporary shutdowns during periods of avalanche threat and construction of deflection barriers, include use of:

- case charging;
- helicopter bombing, assuming acceptance of long closures during storms;
- military surplus artillery; and
- remote systems such as Avalanche Guard or Gazex, or
- a combination of all approaches.

Case charging is the use of explosives set on the ground, usually on steep slopes immediately above a road, where the concussion from the blast can remove unstable snow over a distance of about 100 metres.

Helicopter bombing is accomplished by dropping explosive charges from helicopters into suspected avalanche initiation zones. It is an effective way of clearing avalanche threats, but cannot be used during blizzard conditions or at other times when it is not safe for helicopter to fly.

Military style artillery can accomplish the same effect as helicopter bombing without concerns about weather conditions. However, special training and licensing is required and the artillery is not as mobile as a helicopter.

Remote systems are permanent fixtures in avalanche initiation areas that can be loaded in advance and ignited remotely. An example is the Gazex system that uses a remotely ignited propane and oxygen mixture to produce directed explosions. The Avalanche Guard system uses a tower loaded with multiple mortar-like munitions that can be remotely launched. Remote systems have a higher initial capital cost but can be operated round the clock in any weather condition. Their use is restricted to fixed locations, although the fixtures can be moved as mining progresses.

5.5.2 Open Pit Optimization

An optimized 65,000 t/d mill feed schedule was developed for the Galore Creek mine. Detailed pit phases were engineered from the results of a Lerchs-Grossman (LG) sensitivity analysis, and yielded the phase delineated resources shown in Table 5.5-5. Phase delineated resources in Table 5.5-4 include a 10% dilution for blocks that contain both waste and mineralized material above cutoff grade (contact dilution). No mining losses were taken on the assumption that after the 2005 exploration data are included in the resource modeling, most blocks above cut-off grade will be 100% mill feed. Cut-off grade for the Phase delineated resources in Table 5.5-4 was US\$3.26/t Net Smelter Return (NSR).

**Table 5.5-4
Galore Creek Pit Resource Summary**

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

Mill feed was broken into grade bins for production schedule optimization for the pit delineated resource reports used in the preliminary economic assessment. These grade bins were based on the NSR value of each block, which was calculated from the metal grades in each block, the net smelter metal price, and the plant recovery for gold, copper and silver. The net smelter price was based on base case metal prices, offsite transportation, smelting, and refining charges, etc. Table 5.5-5 describes the NSR cutoffs used in the pit delineated resource reports.

**Table 5.5-5
NSR Grade Cutoff Bins for Pit Delineated Resources**

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

Pit resources for the detailed pit designs were calculated from the 3D block model using the *MineSight Pitres* routine.

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

5.5.2.1 Economic Pit Limits – Central Area

Three series of Lerchs-Grossman pits were developed. At the time of the analysis, exploration targets within an economic pit limit were still being drilled in the 2005 exploration program. Revenues from resource classes below the confidence level of Measured and Indicated were included in the study and consequently the engineering study is considered a preliminary economic assessment (Appendix 5-F, *Updated Preliminary Economic Assessment for the Galore Creek Project*). Resource classes used for the pits were as follows:

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

where:

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

- Class 4 = Inferred based on twice the full variogram range

Lerchs-Grossman pit shells were determined for each series by keeping mining costs constant and varying the estimated net smelter metal prices (NSP).

The economic pit limits were determined using Lerchs-Grossman optimization based on Measured, Indicated, and Inferred resources to give an overall scope for the current engineering and planning studies. The pit design parameters are pre-feasibility level estimates; key parameters have been varied to test the pit limit sensitivity.

The analysis demonstrated that Lerchs-Grossman economic pit limits are significantly sensitive to complex slopes, but not sensitive to incremental bench cost or bench time value discounting within the resource model limits.

5.5.2.2 Economic Pit Limits – Junction

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

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

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

5.5.3 Mine Design

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

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

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

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

5.5.3.1 Pit Phases

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

A comparison of the resources within the phase designs showed that each phase contains similar tonnes of mill feed (Table 5.5-6). Phase 1 has a low strip ratio (SR) for early ore release, and the NSR starts at higher values, decreasing to the lowest value for the last phase.

**Table 5.5-6
Comparison of Pit Resources, Central Pit Phases**

Phase	Pit	Bottom Elevation Description	Mill Feed (kt)	Waste (kt)	SR (t/t)	NSR (\$/t)
1	C615	600 L-G Pit 40* = 50% price shell – southern portion	56,881	91,396	1.6	16.00
2	C625	480 L-G Pit 40* = 50% price shell – northern portion	62,469	167,672	2.7	14.93
3	C635 B	435 L-G Pit 40* = 50% price shell – middle portion	58,397	71,534	1.2	12.80
4	C645	405 West wall to ultimate pit limit	79,298	226,941	2.9	11.87
5	C655	375 Incremental east wall phase	63,097	101,526	1.6	10.40
6	C695	270 L-G Pit 06** = 110% price shell (w/complex slopes)	55,138	63,850	1.2	8.67

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

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

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

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

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

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

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

Table 5.5-7 summarizes the pits not included in the main Central area. These auxiliary mining areas could be mined independently from the Central phases and could be scheduled as needed to improve the mill feed grade and optimize the waste stripping fleet. The only dependency for these outside pits will be that phase 2 must follow phase 1 for Junction and Southwest. The higher NSR values and lower strip ratio for Southwest Phase 1 (Table 5.5-7) indicate this pit should be mined early in the production schedule.

**Table 5.5-7
Comparison of Pit Resources, Auxiliary Pits**

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

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

5.5.3.2 Phase Designs

The resultant six cumulative pit phase designs for Central and the six auxiliary pits are illustrated in Figure 5.5-5 to Figure 5.5-16. These phases are shown relative to the truck shop/run of mine stockpile ahead of the crusher to the northeast of the drawings (at approximately 705 m elevation) and the plant site to the southeast. Topography contours are shown every 15 m at bench grade elevations.

5.5.4 Mining Method

Mining at Galore Creek will be by conventional truck and shovel operation with one main pit and several satellite pits. Ore extraction will be allocated between the pits as necessary to maintain an appropriate volume and grade of ore flow to the mill.

5.5.4.1 Pre-Stripping

Ore grade material is available near surface, and, from a mill supply perspective, requires very little pre-stripping to feed the plant at design rates. Mining activity in the start-up period at Galore Creek will instead be driven by infrastructure construction requirements and ultimately, will be further influenced by the design characteristics of the material required for these features. The main mining-related site features being constructed during start-up will be the coffer dams, main tailings dam, the freshwater diversion structure, aqueduct, diversion channels and spillway, and mining haul roads.

About 70 million tonnes of pre-stripping material will be used for construction of the main tailings starter dam. Much of the till will be used to construct the mill stockpile pad, discussed in Section 5.5.9. The remainder of the waste will be disposed of in a manner consistent with the project's waste rock storage strategy.

5.5.4.2 Operations

Mining will be performed top down by benches within multiple pits at the same time. The primary production will use a large electric cable shovel with a hydraulic shovel and large loader for support.

The haul truck fleet will use large capacity trucks for both waste and ore. Ore will be hauled to the plant crusher with some stockpiling in designated areas as required. A large portion of waste rock will be hauled to the waste rock disposal impoundment for sub-aqueous deposition because of acid rock drainage concerns. The remaining rock that is not required for construction of dams, roads and other facilities will be disposed in localized dumps and perhaps in mined out pits.

5.5.5 Ore Grade and Waste Control

5.5.5.1 Ore – Waste Separation

Grade control will be based on copper and gold grades. Samples of blasthole cuttings will be collected by trained drillers supervised by grade control engineers and geologists.

Phase 1 (P615)

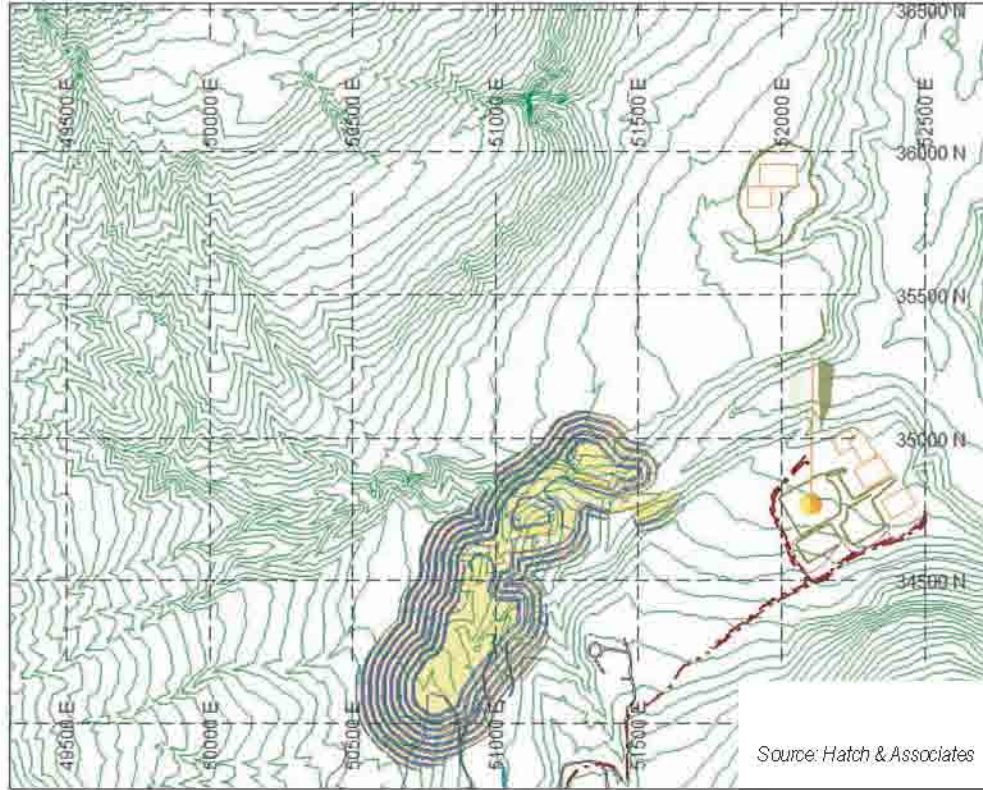


FIGURE 5.5-5

Phase 2 (P625)

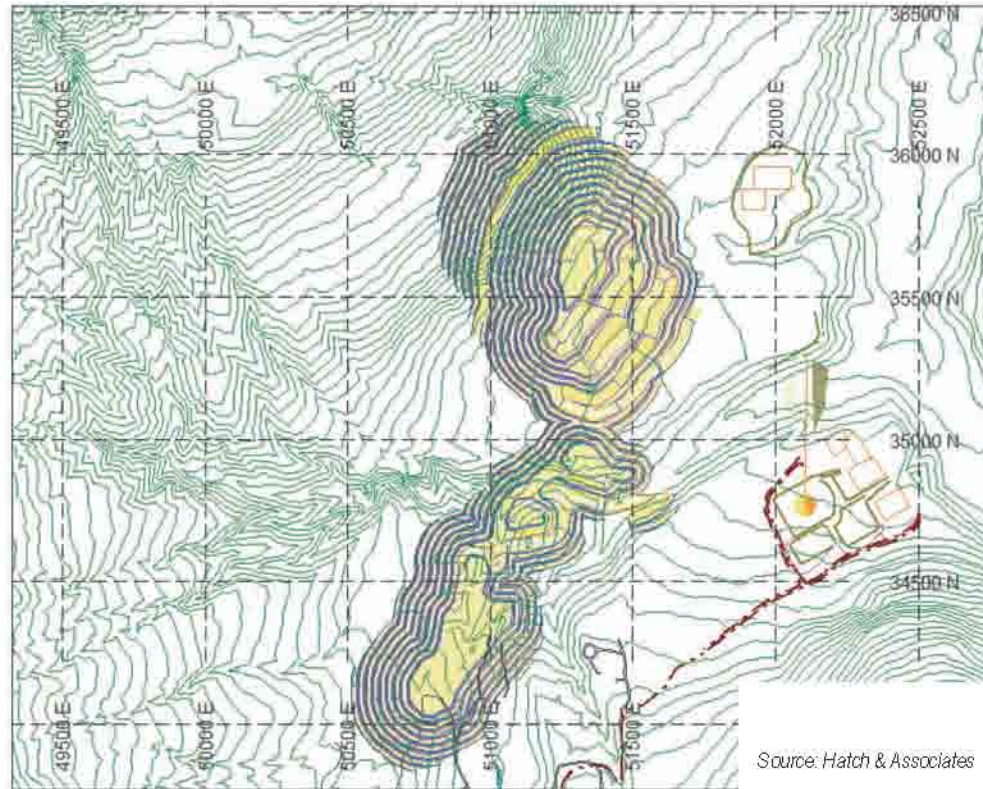


FIGURE 5.5-6

Phase 3 (P635B)

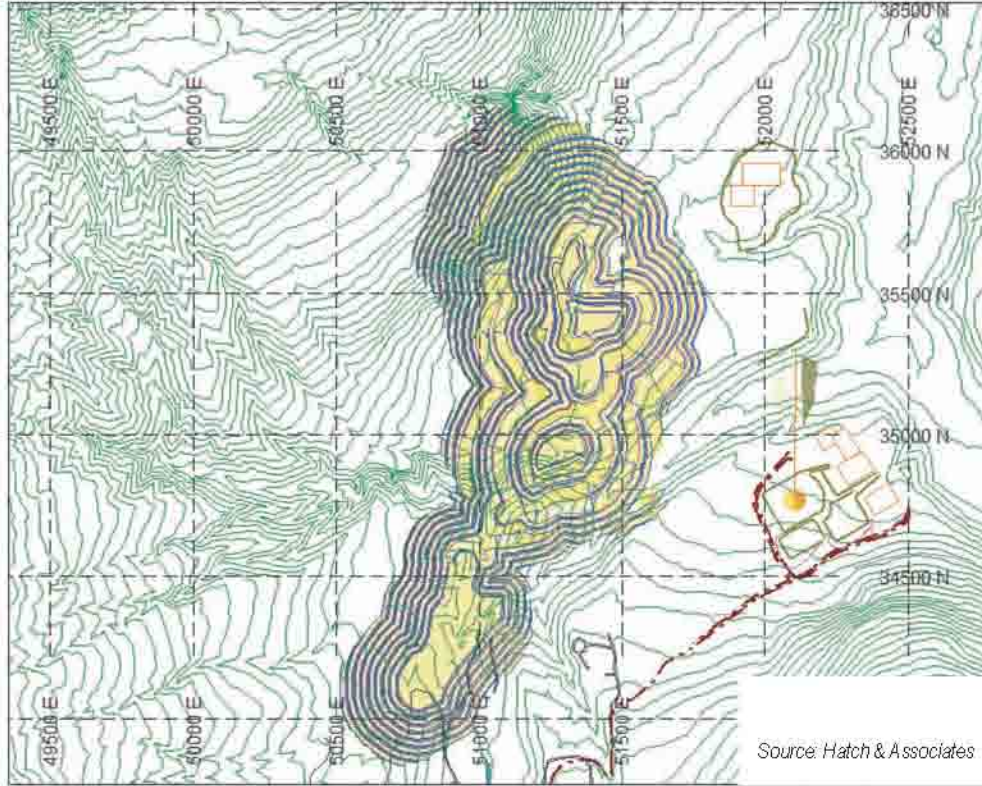


FIGURE 5.5-7

Phase 4 (P645)

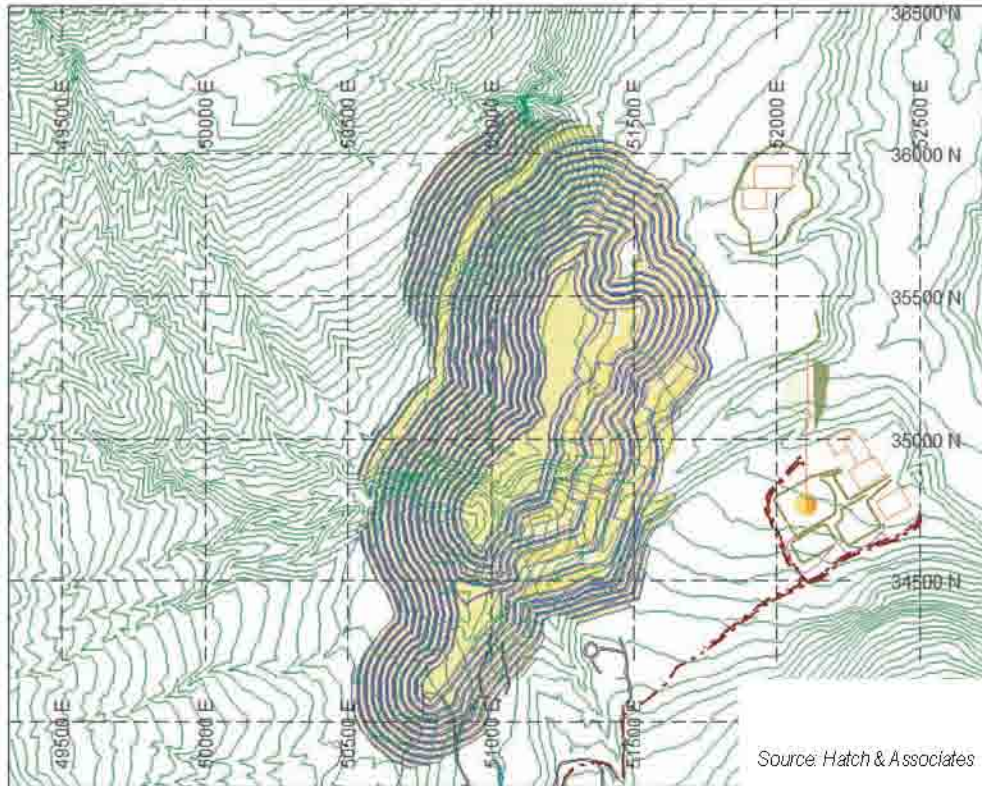


FIGURE 5.5-8

Phase 5 (P655)

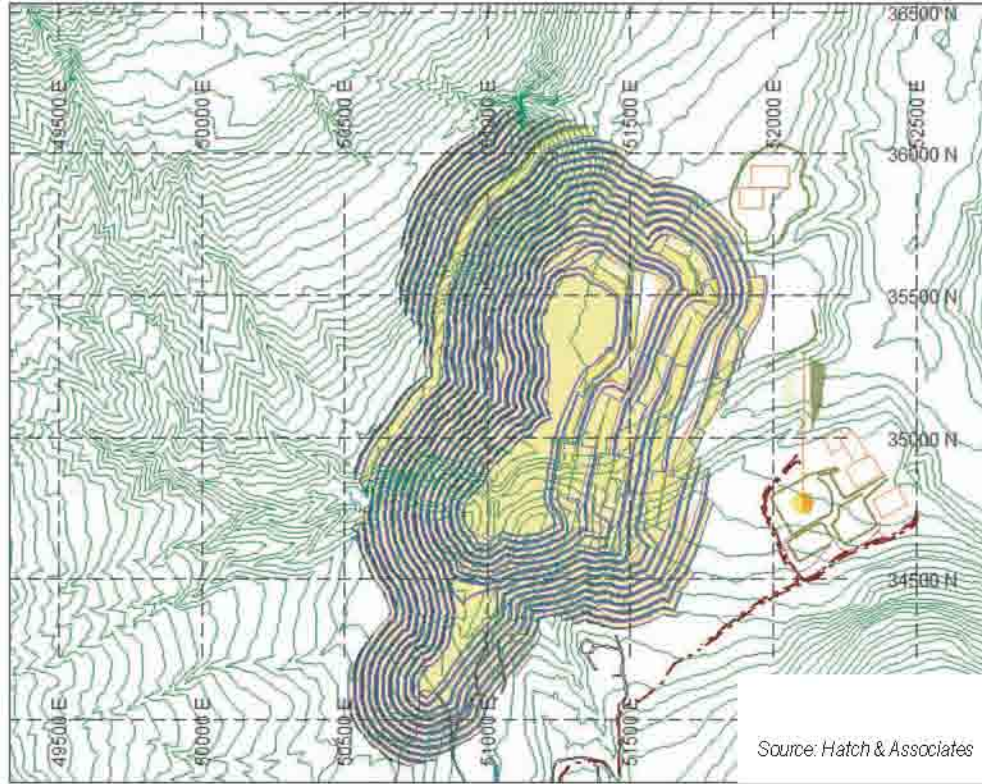


FIGURE 5.5-9

Phase 6 (P695)

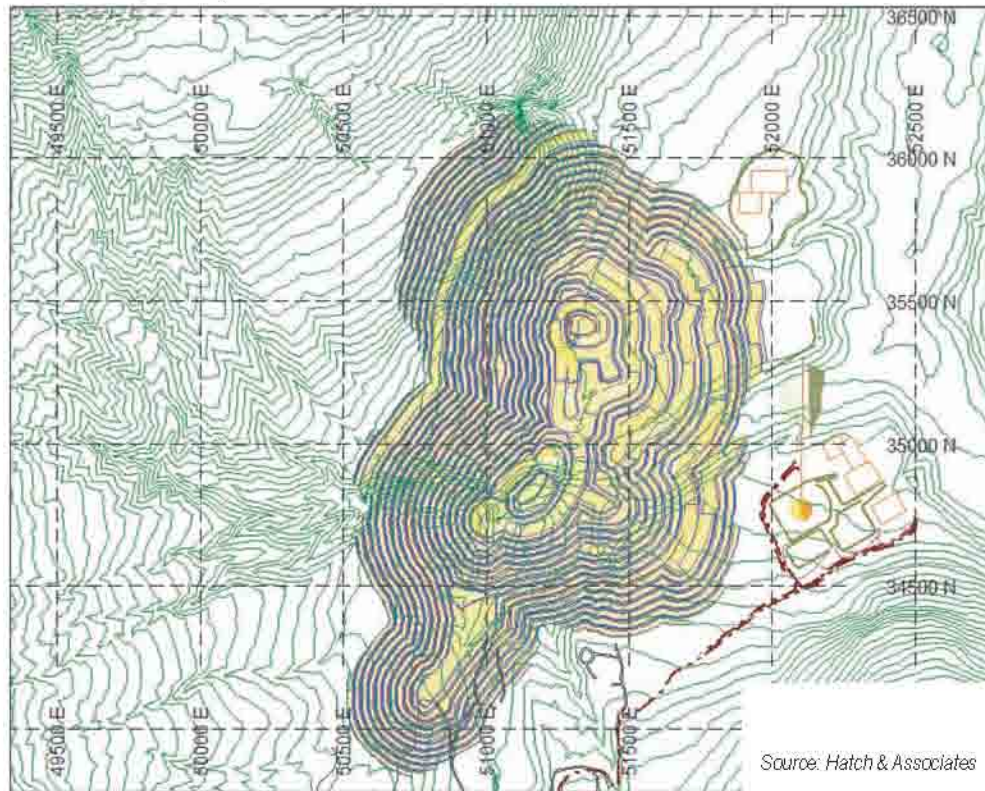


FIGURE 5.5-10

Phase 1 (SW601a)

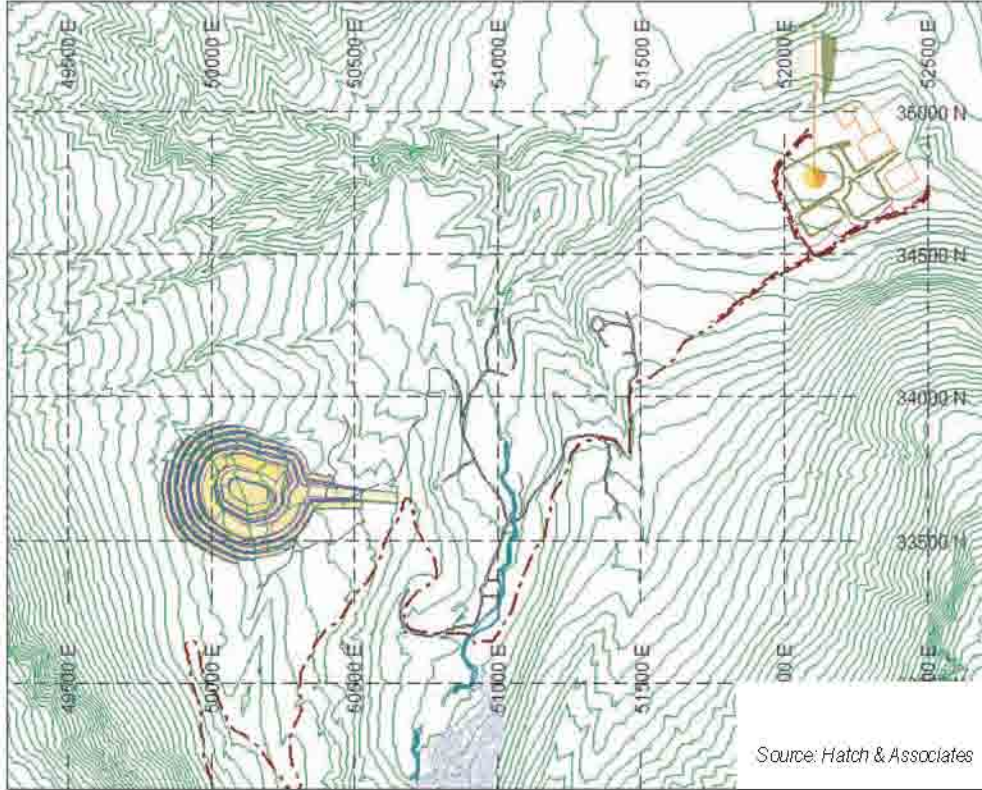


FIGURE 5.5-11

Phase 2 (SW611)

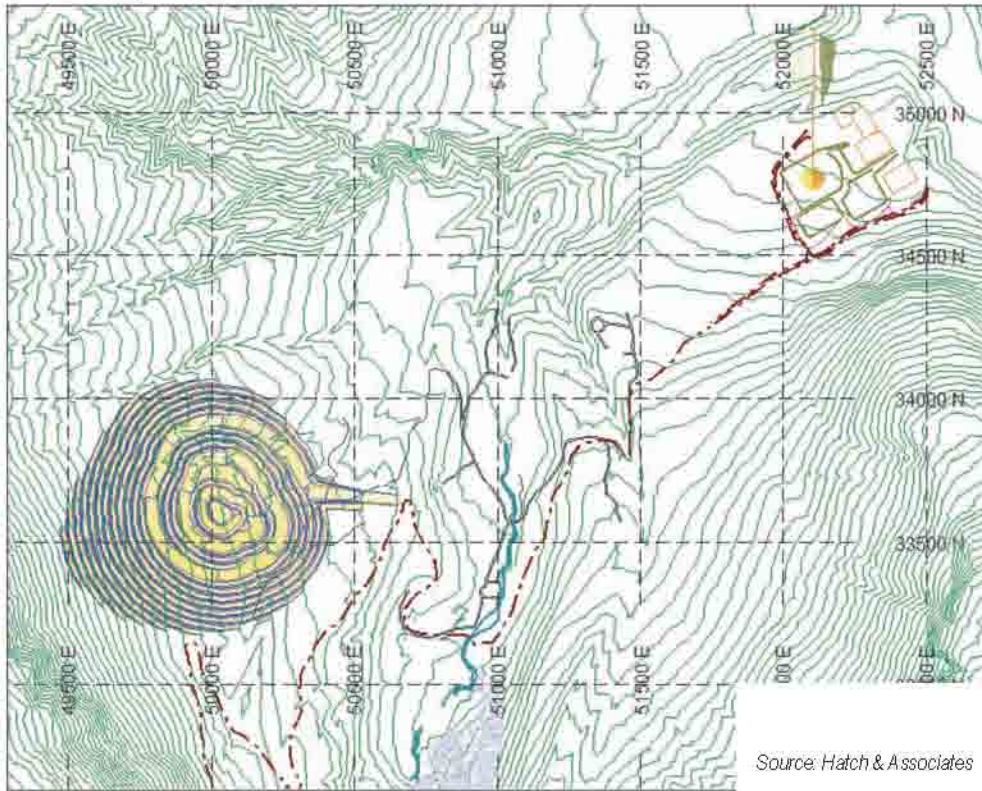


FIGURE 5.5-12

Middle Creek Pit (W611)

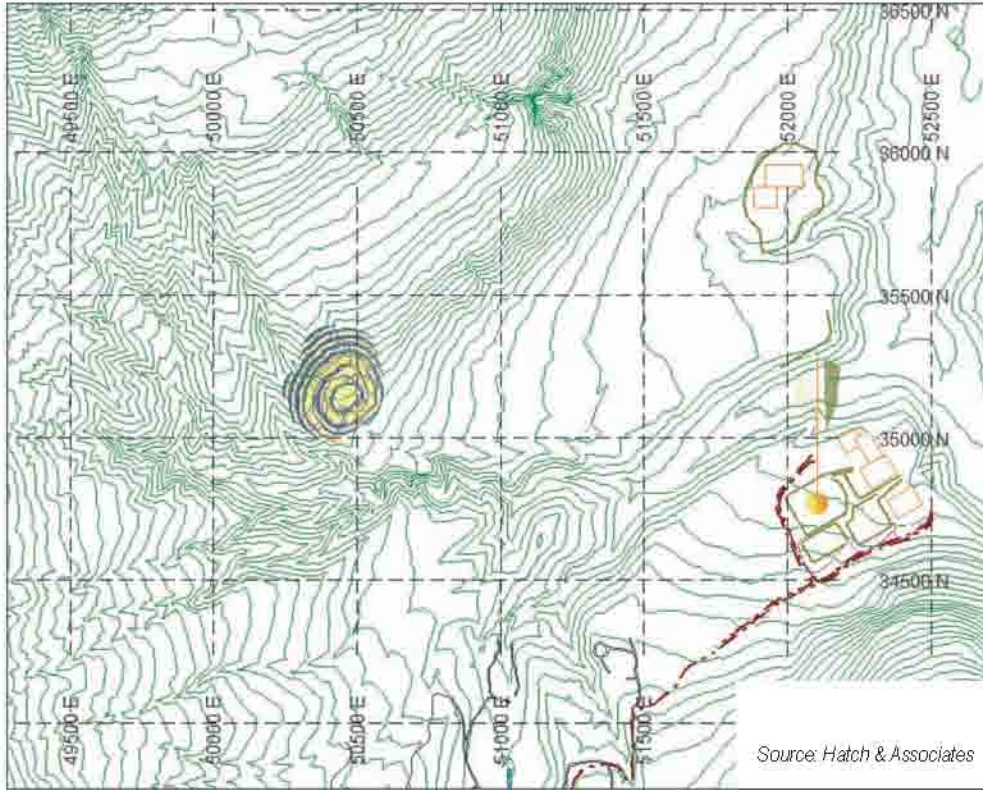


FIGURE 5.5-13

West Fork Pit (WF611)

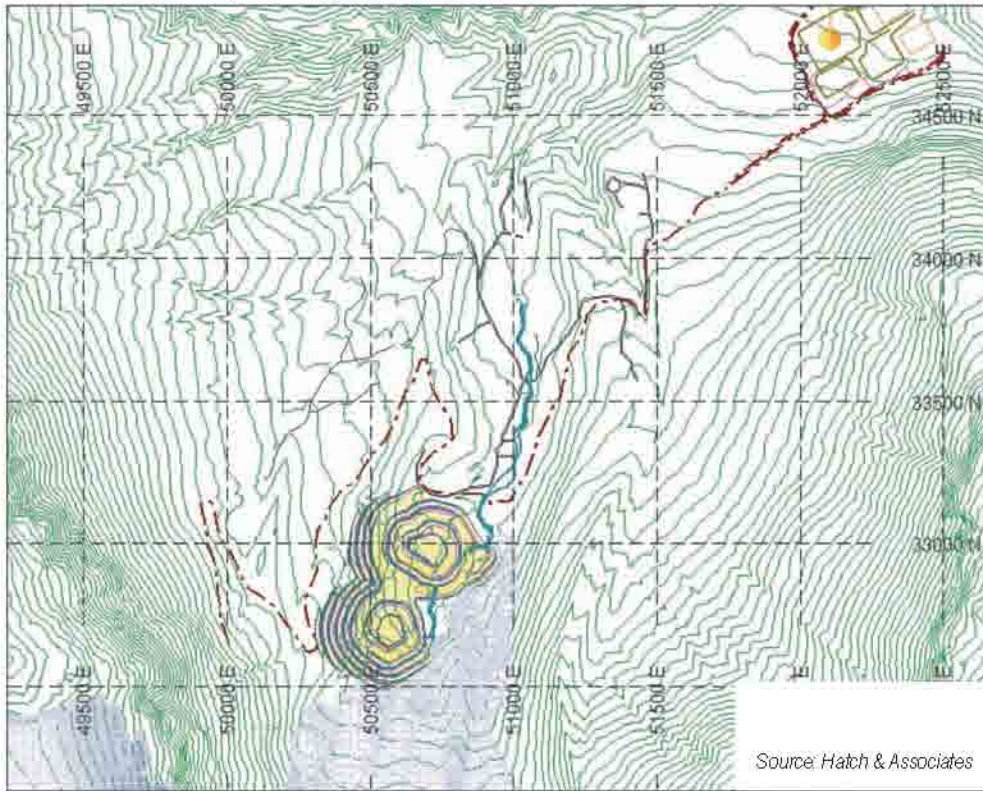


FIGURE 5.5-14

Phase 1 (JN6)

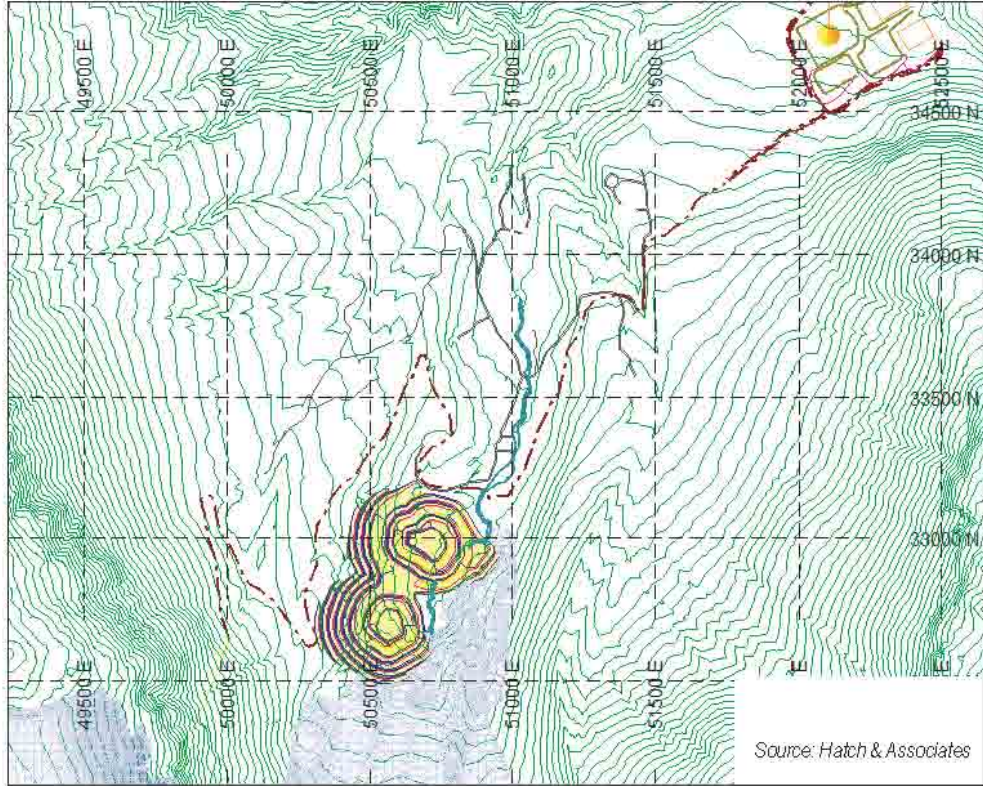


FIGURE 5.5-15

Phase 2 (JN5 & 6 Merged)

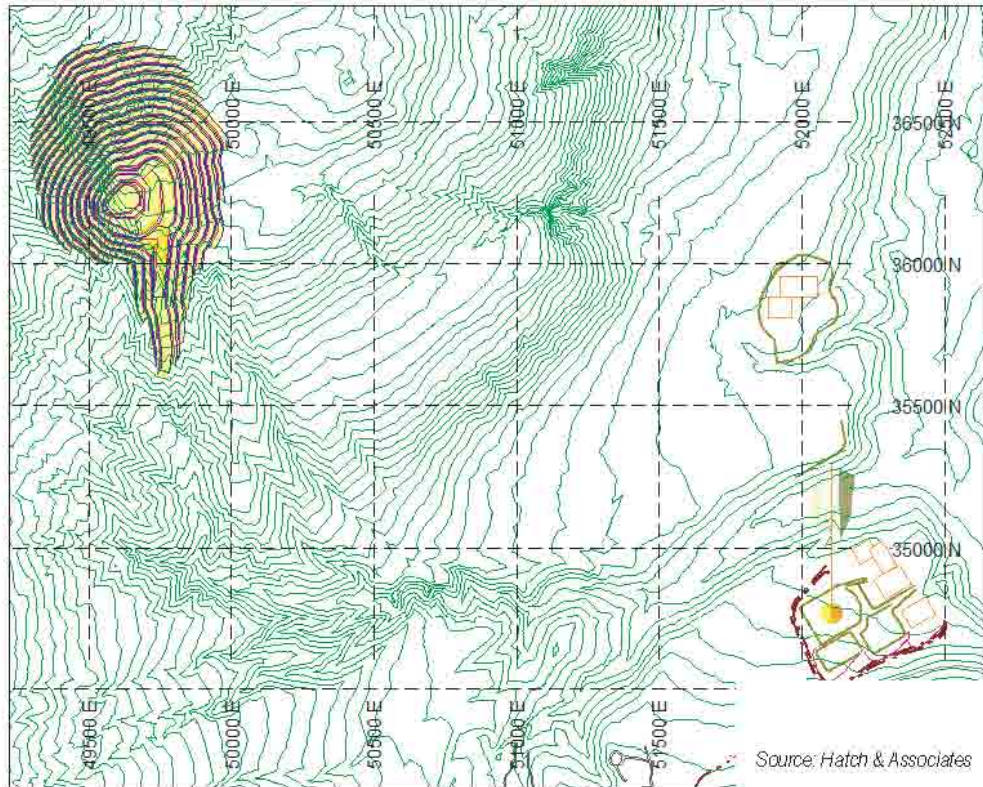


FIGURE 5.5-16

The blasthole locations and assays of cuttings will be entered by laboratory staff into a database. The survey and assay data will be merged and used in conjunction with geology and known ore controls to define mineable blocks.

An automated system using a Global Positioning System will define mineable blocks in the pits. Shovel operators and shift foremen will be provided with up to date digging plans to ensure proper grade selection to meet mill requirements.

5.5.5.2 PAG and Not-PAG Segregation

Using fairly conservative criteria, approximately half the designated waste rock is expected to be not potentially ARD generating (not-PAG or NPAG) and half will be PAG (potentially ARD generating). Work to date suggests that PAG waste rock occurs in large blocks in well-defined areas. It is therefore possible that waste rock can be segregated based on acid generation potential. Not-PAG and PAG rock will be separated to allow subaqueous disposal of the PAG and use of the not-PAG for tailings dam and other facilities construction. Substantial quantities of not-PAG rock will be available early in the life of the project for dam construction.

Due to the requirement for reliable classification of ARD and ML potential for construction rock, all blast holes in the pre-strip areas will be sampled to define dig limits until sufficient data are available to reduce sampling.

Blast hole chips from the pre-strip areas will be analyzed at the on-site laboratory in the following sequence:

- Copper;
- paste pH;
- sulphur as sulphide determined by Leco furnace on a sample first leached with hydrochloric acid to remove sulphate; and
- carbonate.

Carbonate is a surrogate for neutralization potential (NP) based on pre-mining testwork. The data obtained will be used to model dig limits.

The possibility of using total carbon as a surrogate for carbonate will be evaluated. Initial muck samples from blasting will be screened to obtain samples of blast rock size fractions. These samples will be analyzed for the same parameters to determine if partitioning of sulphide or carbonate minerals into the fines is occurring. This approach may lead to adjustment of management criteria.

PAG and not-PAG blocks will be defined in the pits. Shovel operators and shift foremen will be provided with up to date digging plans to ensure proper segregation of waste rock for long term disposal.

Management and disposal will be determined using the following criteria:

- Ore Stockpile – Cu > ore cut off level;

- permanent underwater disposal in impoundment – NP/Acid Potential (AP)<2 or paste pH<6.
- upland disposal within containment area above final flood level – NP/AP>2, and paste pH or Cu/S>x.
- dam construction rock – paste pH>6, NP/AP>3, Cu/S<x.

The value of x is currently being evaluated by leach column tests.

5.5.6 Mine Road Construction and Maintenance

Haul roads will be required between active pit faces, separate overburden, PAG and not-PAG dump and ore storage areas, the plant site crusher and dam sites. Roads will also be required parallel to diversion channels to permit access for maintenance. The length of required active haul roads on the site will vary over time from 20 to 50 km. Where possible, long hauls will be confined to the summer months, allowing shorter hauls and consolidation of equipment during the winter snow season.

Lighting for visibility in blizzard and fog conditions, application of crushed rock for traction in snow conditions, and removal of snow are discussed in Section 5.5.13.

Initial on-site road construction will access the top of the western Central pit phases at approximately 1,030 m elevation. Pioneer access for drill, dozer and service equipment to the Central pit area will be developed from above using the flatter terrain above the western limit. Table 5.5-8 summarizes major pre-development road segments.

**Table 5.5-8
Major Pre-Development Road Segments**

Pioneer Roads	Start Elev.	End Elev.	Length (m)	Grade
(West) Central Ph II to IV	1,050	1,030	1,000	2%
(South) Central Ph I	860	852	1,200	0.7%
Major Haul Roads				
West Central In-pit	1,030	748	5,261	5.4%
South Central In-pit	850	795	1,000	5.5%
Main North Haul Road	748	510	5,219	4.6%
Main East Haul Road	795	660	2,227	6.1%
Service / Construction Roads				
Upper Truck Shop Access	743	690	1,100	4.8%
Crusher Pad Access	660	690	800	-3.8%
Starter Dam Access	510	450	1,500	4.0%
Freshwater Dam Access	660	645	1,100	1.4%

Pit ramps will be designed for maximum grades of 8%.

Haul roads will be constructed to meet the requirements of the *Health, Safety and Reclamation Code for Mines in British Columbia*, including a travel width for single lane traffic of at least two times the width of the widest haulage vehicle used on the road and for dual lane traffic a travel

width of not less than three times the width of the widest haulage vehicle used on the road (Figures 5.5-17 and 5.5-18).

Shoulder barriers along the edge of the haulage road will be at least 3/4 of the height of the largest tire on any vehicle hauling on the road wherever a drop-off greater than 3 m exists. The width of the barrier is excluded from the travel width.

The design basis uses the following data:

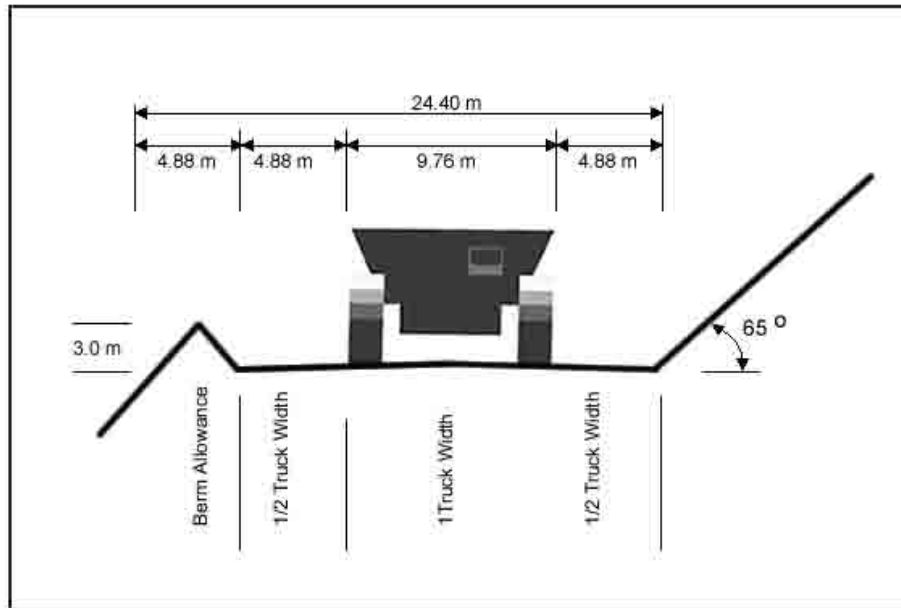
Largest Vehicle Overall Width (CAT 797B)	9.8 m
Maximum Tire Height (59/80R63)	4.0 m
Minimum Haul road outside berm height	3.0 m
Minimum Shoulder / Berm Width	4.9 m
Double lane highwall haul road allowance	34.2 m
Double lane external haul road allowance	39.0 m
Single lane highwall haul road allowance	24.4 m
Single lane external haul road allowance	29.3 m

5.5.7 Topsoil Salvage and Storage

For this project topsoil is defined as the organically enriched, moderately thick (approximately 0.5 m), root zone soil material common to the well developed soil profiles of the soils classified as podzols and brunisols, on suitable soil materials. They are primarily associated with the forested and shrub-covered areas on the lower slopes and valley bottom, within the areas below the alpine zones. Topsoil of significant thickness is generally not present on weakly developed soils, those classified as regosols, or areas of non-soils, and no separate distinction between topsoil and overburden will be made in these areas. Topsoil will be salvaged as a separate lift during surface pre-stripping, typically over the footprint of pit areas, dams, plant site, linear infrastructure (roads, diversion channels, buried pipelines), overburden storage area(s) and waste rock dumps outside the flooded area. No salvage is anticipated in areas to be flooded for tailings or underwater waste rock placement, unless soil balance requirements (salvage : replacement) show a need for additional salvage volume from these areas.

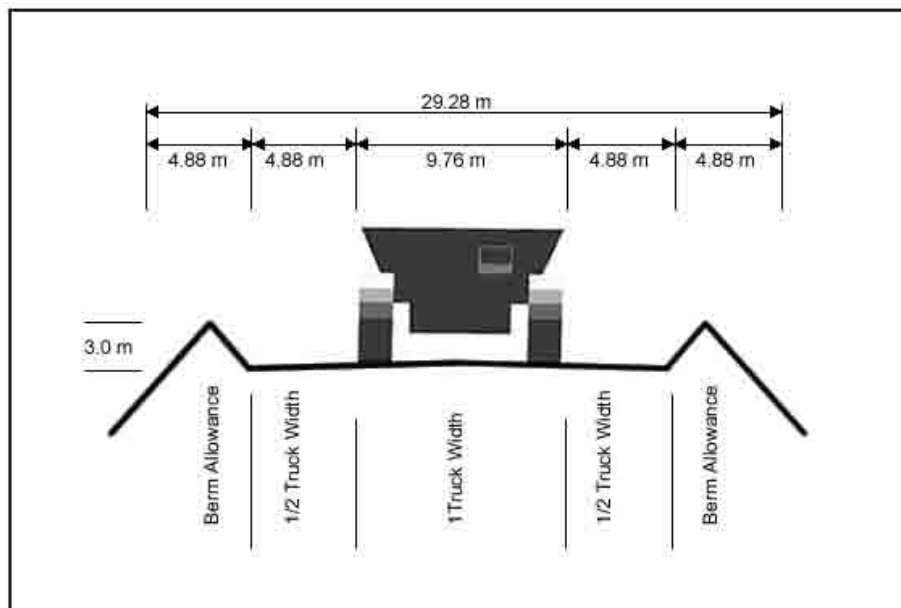
The separate salvage of topsoil from the overburden (soil parent material) is referred to as a two-lift soil salvage system. A one-lift system does not differentiate between the surface soil material and the overburden. The one-lift system will be common to areas of thick overburden with little or no soil development or where practical considerations, such as excess slope or wetness, make differential separation of topsoil from overburden impractical. The volume of salvaged topsoil will be relatively small compared to the volume of stockpiled overburden material.

Highwall



Note: Highwall face slopes will vary by geotechnical design criteria

External



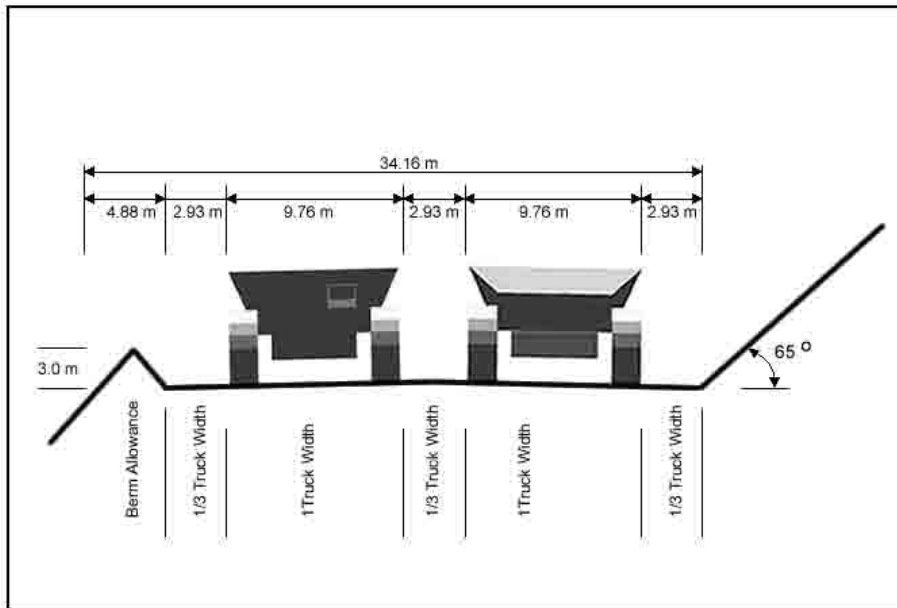
Source: GR Technical Services Ltd.

Single Lane Haul Road Cross Sections

FIGURE 5.5-17

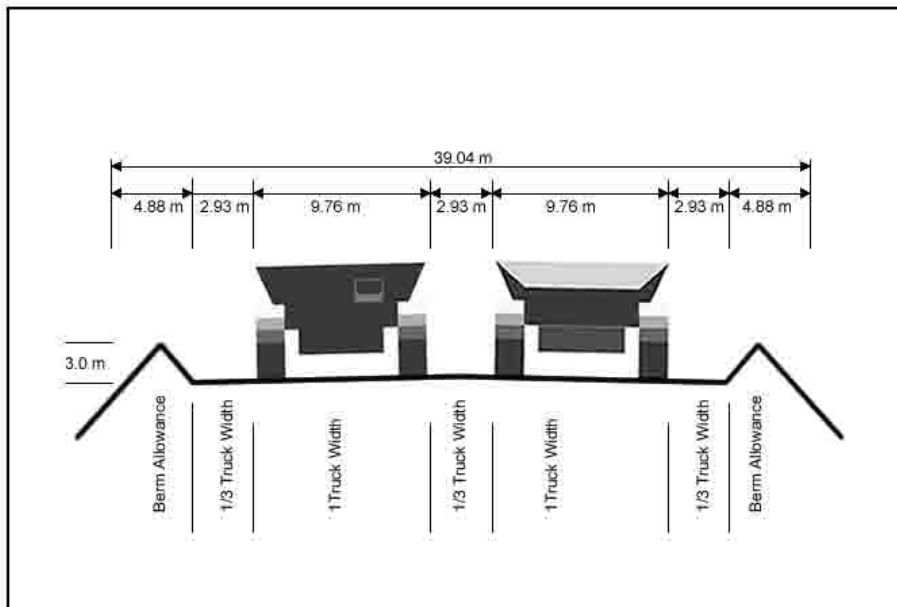


Highwall



Note: Highwall face slopes will vary by geotechnical design criteria

External



Source: GR Technical Services Ltd.

Dual Lane Haul Road Cross Sections

FIGURE 5.5-18



Additional root zone material, in the form of suitable overburden, will be available from the excavation of the pit areas. Thick overburden deposits are known to be present in the Central Pit. Selective handling of preferred overburden types will be possible if significantly thick, homogeneous stratigraphic units are identified in the pre-development investigations.

Topsoil will be stockpiled separately from other materials. It will be stored in stable areas and will not be disturbed until the soil material is required for replacement, for reclamation purposes, on the final (post-mine) recontoured surfaces. Topsoil may be windrowed adjacent to areas of linear development. Direct replacement (salvage and concurrent replacement) will be employed where permitted by scheduling. Direct replacement is anticipated to be a small portion of the total volume of soil handled as few surfaces will be available for reclamation in the early stages of development.

5.5.8 Marginal Ore Storage

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

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

5.5.9 Ore Stockpile

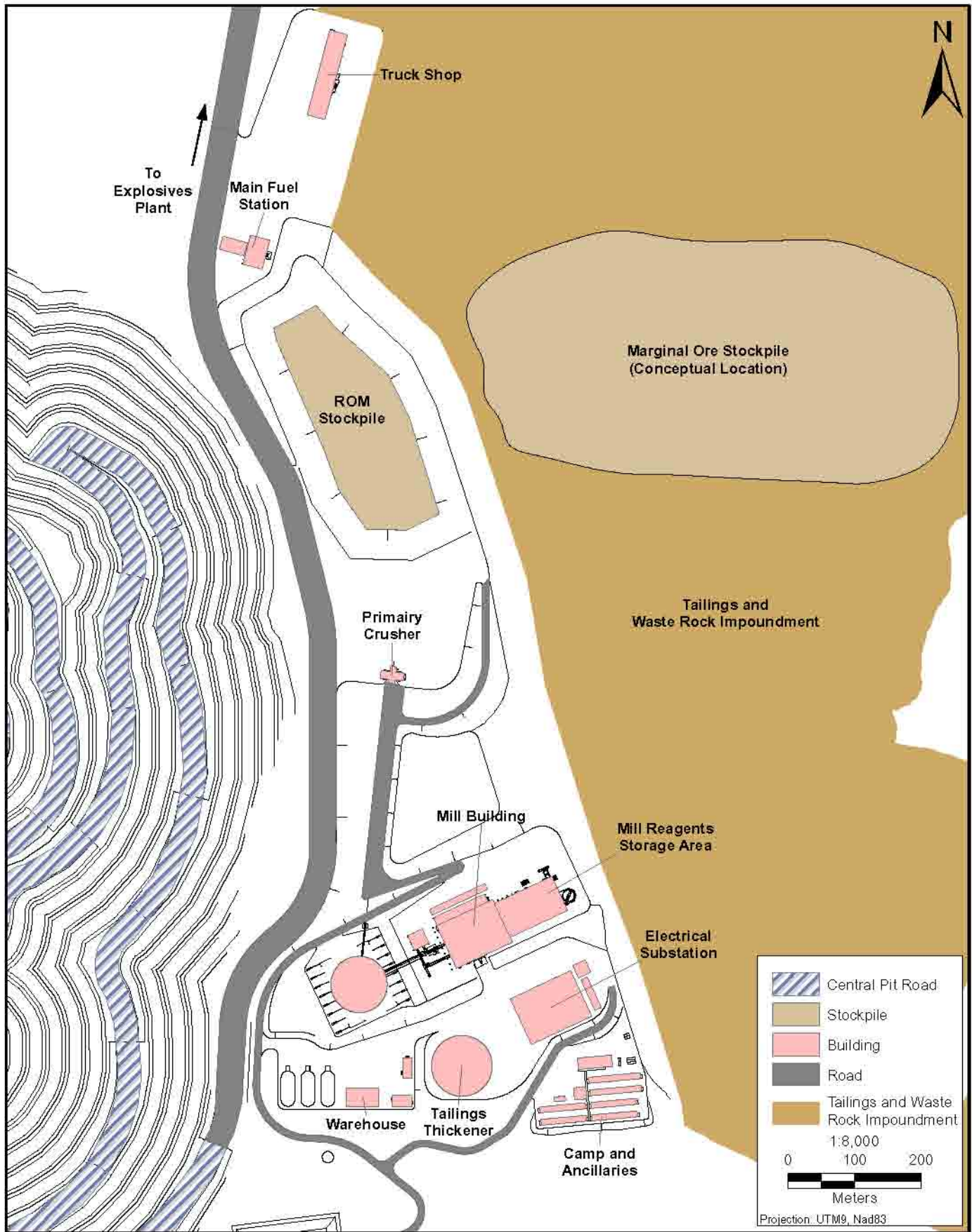
The mill stockpile pad will be located adjacent to the intermediate stockpile immediately north of the crusher area on waste placed early in the mine life (Figure 5.5-19).

During mine years 1 to 4 the run of mine stockpile will grow at a rate of about 2.5 million tonnes per year (Table 5.5-9). By the end of year 5 the stockpile will require a capacity of over 16 million tonnes. The stockpile will be built in a series of cells to allow access to multiple stockpile faces for grade control when reclaiming ore to the mill.

**Table 5.5-9
Run of Mine Stockpile Volume (tonnes)**

Project Year	Run of Mine Stockpile
-2	61,129
-1	122,258
1	2,074,669
2	4,699,296
3	7,479,367
4	13,244,050
5	16,037,021
6	18,949,764

(continued)



Run of Mine Ore and Marginal Ore Stockpile Locations

FIGURE 5.5-19

**Table 5.5-9
Run of Mine Stockpile Volume (tonnes) (completed)**

Project Year	Run of Mine Stockpile
7	19,375,529
8	9,990,194
9	4,819,915
10	3,522,900
11	7,598,418
12	8,061,061
13	7,087,986
14	11,399,850
15	13,185,069
16	2,345,247
17	8,385,327
18	12,636,498
19	10,258,894
20	1,097,732

5.5.10 Mine Production Schedule

The production schedule for Galore Creek has been developed to meet the input mill feed targets, while balancing waste stripping requirements with the given truck and shovel fleet. Various grade bins were evaluated to improve the relative project net present value (NPV). Pre-strip will begin in ‘Year -2’ and full production will begin at the beginning of ‘Year 1’. The project mill feed target is an average 65,000 t/d or 23,725,000 t/a.

Southwest Phase 1 (SW601a) has the highest grade and was selected as the startup phase. Most of the ore requirement in Year 1 will be mined from Southwest Starter Phase (SW601a). Sub-grade material will become uneconomical if stockpiled and will be wasted in the start-up period to prevent mill feed grade dilution. A pre-strip waste production volume of 75 Mt is required to ensure suitable material is available for start-up dam construction. Galore Creek production schedule is presented in Table 5.5-10 showing the first 5 years of production and life of mine (LOM) data.

5.5.11 Mining Equipment

The Galore Creek Project will require a substantial fleet of mining equipment to meet the production goal of 65,000 t/d on an all-season basis, as well as disposing of overburden and waste rock, moving materials for dam construction and removing snow during storms. The proposed mining fleet includes blast hole drills, shovels, loaders, excavators, haul trucks, dozers, graders and a variety of service and support vehicles (Table 5.5-11).

An electric blast hole drill was selected as the primary drill on the basis of discussions with manufacturers and suppliers with reference to experience in mines with similar rock conditions. A diesel blast hole drill is proposed for the satellite pits due to their mobility and flexibility. A track drill will be used for controlled blasting and development of initial upper benches.

**Table 5.5-10
Summarized Galore Creek Production Schedule**

PRODUCTION		Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	LOM
ROM Mill Feed									
ROM Mill Feed mined to crusher	tonnes	-	-	23,735,018	23,735,028	23,734,976	23,734,992	23,735,000	416,414,573
Cu	%	-	-	0.731	0.939	0.962	0.626	0.715	
Au	g/t	-	-	1.072	0.316	0.344	0.664	0.480	
Ag	g/t	-	-	3.976	6.270	6.756	3.690	4.536	
ROM Mill Feed mined to stockpiles	tonnes	61,129	61,129	1,952,410	2,624,628	2,780,071	5,764,682	2,792,971	58,227,732
Total ROM Mill Feed Mined	tonnes	61,129	61,129	25,687,428	26,359,656	26,515,047	29,499,675	26,527,971	474,642,306
ROM Mill Feed retrieved from stockpiles									
ROM Mill Feed retrieved from stockpiles	tonnes	-	-	-	-	-	-	-	57,130,000
Cu	%	-	-	-	-	-	-	-	
Au	g/t	-	-	-	-	-	-	-	
Ag	g/t	-	-	-	-	-	-	-	
Total Stockpile Inventory		61,129	122,258	2,074,669	4,699,296	7,479,367	13,244,050	16,037,021	
Total ROM Mill Feed to Mill	tonnes	0	0	23,735,018	23,735,028	23,734,976	23,734,992	23,735,000	473,544,573
Cu	%	-	-	0.731	0.939	0.962	0.626	0.715	
Au	g/t	-	-	1.072	0.316	0.344	0.664	0.480	
Ag	g/t	-	-	3.976	6.270	6.756	3.690	4.536	
Sub Grade to Waste	tonnes	48,211	48,211	2,005,984	2,651,625	3,203,052	4,861,323	2,815,308	19,215,394
Waste Mined	tonnes	37,389,525	37,389,525	41,041,618	59,722,766	84,011,896	44,372,848	49,390,511	1,015,937,962
Total Waste Mined	tonnes	37,437,736	37,437,736	43,047,602	62,374,391	87,214,947	49,234,170	52,205,819	1,035,153,356
Waste Types:									
Till	tonnes	10,654,989	10,654,989	17,193,520	16,875,780	31,066,715	5,191,076	9,111,231	166,117,378
Broken not-PAG	tonnes	258,836	258,836	12,733,734	9,138,005	10,277,417	7,042,877	16,099,452	180,306,378
Broken PAG	tonnes	25,671,525	25,671,525	10,769,786	31,200,173	32,287,318	13,648,787	11,509,365	227,328,043
Stick Not-PAG	tonnes	-	-	344,577	486,668	2,145,260	6,157,664	9,604,237	180,900,018
Stick PAG	tonnes	804,175	804,175	-	2,022,140	8,235,186	12,331,968	3,044,081	165,365,728
Junction (Un Zoned)	tonnes	-	-	-	-	-	476	22,146	95,920,417
Waste Hauled for Main Dam Construction	tonnes	13,289,391	13,289,391	11,430,428	12,519,762	12,519,762	12,519,762	12,519,762	227,876,181
Strip Ratio1 (Waste/Total ROM Mill Feed Mined)		612.4	612.4	1.7	2.4	3.3	1.7	2.0	2.2
Strip Ratio2 (Waste/ Total ROM Mill Feed to Mill)		-	-	1.8	2.6	3.7	2.1	2.2	2.2

(continued)

**Table 5.5-10
Summarized Galore Creek Production Schedule (completed)**

PRODUCTION	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	LOM
Strip Ratio ³ (Waste/ ROM Mill Feed Mined to Crusher)	-	-	1.8	2.6	3.7	2.1	2.2	2.5
Total Material Mined	tonnes 37,498,865	37,498,865	68,735,030	88,734,047	113,729,994	78,733,845	78,733,791	1,509,795,662
Total Material Moved	tonnes 37,498,865	37,498,865	68,735,030	88,734,047	113,729,994	78,733,845	78,733,791	1,566,925,662

**Table 5.5-11
Mining Equipment for Initial Five Year Operation**

Item	Number Required
Primary Blast Hole Drill	1
Satellite Pit Drill	1
Track Drill	1
Primary Shovel, Electric	1
Secondary Shovel, Hydraulic	1
Haul Trucks	15
Excavator, Diesel/Hydraulic	1
Excavator, Diesel/Hydraulic	1
Utility Hoe	1
Grader	3
Track Dozer	3
Track Dozer	3
Wheeled Dozer	2
Cable Handler	1
Loader	1
Water Truck	2
Float Tractor and Trailer	1
Fuel Truck	3
Fire Truck	2
Flat Deck Truck	2
Vans	3
Ambulance	1
Mine Rescue Truck	1
Hole Stemmer	1
Utility Loader	1
Fork Lift	2
T Truck	1
Service Truck	6
Pickups	20
Passenger Bus	1
Crane	1
Power Line Truck	1

(continued)

**Table 5.5-11
Mining Equipment for Initial Five Year Operation (completed)**

Item	Number Required
Welding Truck	2
Snow Clearing Trucks	2
Snow Clearing Loader	1
Scrapers	10
Mobile Crushing Plant	1
Grader	1

The primary shovel will be a large electric rope shovel suitable to meet the high throughputs requirements of the Central pit. A diesel hydraulic shovel will be used for the smaller pits. A wheel loader will be used primarily for stockpile retrieval, but will be available to assist the shovels as required. The wheel loader will be sized to be able to load the selected haul trucks.

Haul trucks with 345 tonne payload capacity were selected to haul all materials from the pits.

5.5.12 Explosives Storage, Handling and Use

Table 5.5-12 provides an estimate of the annual explosives use at the Galore Creek mine over mine years -2 to 20. Explosive type will likely be ammonium nitrate (AN) base emulsion. Ignition systems will likely include Cordtex AP detonating cord, Pentex AP 16 boosters, Exel MS detonators and Exel MS connectors.

Explosives will be manufactured on site to improve site efficiencies and reduce raw material transportation costs. The explosives manufacturing and storage facilities will be located away from other mine site facilities, about 800 m north of the truck shop, in a fenced compound with security provisions to limit access to only authorized personnel. The facilities will occupy an area of about 200 m by 200 m, although the exact configuration has not been determined at this time. The layout of the site will be designed to allow for a straight run into and out of the gate as well as ample turning radius for B-trains and bulk transport trucks of raw materials.

Alternative explosives manufacturing facility sites were investigated. This selected site was chosen to ensure the safety of other facilities in the event of a mishap, ease of use and access under extreme weather conditions and in consultation with potential explosives suppliers and plant operators.

A building (approximately 24 m x 18 m) will be constructed within the compound for explosives manufacture and truck loading. The building will meet the guidelines published by the Explosives Regulatory Division (ERD) of Natural Resources Canada and local and federal regulations. A sump will collect water for oil/water separation and disposal.

Prilled ammonium nitrate (ANP) will be delivered to the manufacturing facility in 40 tonne B-train truck and trailer combinations. The ANP will be pneumatically transferred into one of two 80 t overhead bins that will allow conveying of prill to the “melting tanks”. The prill will then be converted into AN solution (ANS) by melting the ANP in one of two 100 t stainless tanks with steam coils located in the manufacturing plant.

**Table 5.5-12
Estimate of Annual Explosives Consumption**

Project Year	Explosives (Kg)
Year -2	4,091,998
Year -1	4,091,998
Year 1	6,478,923
Year 2	9,810,075
Year 3	12,062,357
Year 4	11,775,036
Year 5	10,489,922
Year 6	13,602,326
Year 7	10,266,131
Year 8	10,758,528
Year 9	12,646,380
Year 10	15,240,221
Year 11	14,914,148
Year 12	9,039,014
Year 13	13,029,111
Year 14	9,926,086
Year 15	7,681,613
Year 16	4,229,189
Year 17	13,669,742
Year 18	9,342,216
Year 19	13,484,570
Year 20	4,740,228

Diesel fuel will be a key constituent of the oil phase component of the emulsion manufacturing process. Maximum expected consumption is 2,500 L/d. A diesel storage tank will be located near (at least 25 m away from) the manufacturing plant and will be re-supplied as required by tanker truck from the mine’s central fuel depot.

Surfactant will be shipped to the site in B-train tankers and pneumatically offloaded into a 35,000 L storage tank located inside the manufacturing plant.

The manufacturing site will use up to 10,000 L/d of clean water. Sodium nitrate and ethylene glycol for the emulsion process will be stored in a separate building.

In general terms, surfactant and diesel will be mixed to form an “oil phase” in a 10,000 L oil phase batch tank. This oil phase will then be mixed with ANS and processed through an emulsification unit. Several emulsification technologies are available. Once a final product mix is determined, the design for the plant can be finalized.

Also located within the explosives compound will be an office trailer, a truck storage/wash bay facility and a boiler and electrical room. The truck storage/wash bay facility will have a sump and evaporation system for collecting wash water and wastes and a mechanical area with toolboxes and bench for delivery unit and plant repairs.

The office trailer will be located at the main gate entrance to the compound to provide an office, change room, lunchroom and washroom for the employees of the contractor responsible for explosives manufacture and delivery.

A 2.4 m by 3.7 m (8' x 12') magazine for boosters and occasional case powder and a 1.8 m by 2.4 m (6' x 8') magazine for detonators will be required. The magazines will be bullet proof, fire resistant, theft-resistant, weather proof, well ventilated and conform to standards set by the department of Energy, Mines and Resources Canada. They will be located a safe distance away from both the main site and any structures or roads. (Final distances will be determined by volumes in the magazines with the table of distance formula.)

NovaGold will contract with a third party for the provision of explosives, including operation of the emulsion manufacturing facility and explosives storage magazines. Explosives will be manufactured on site and delivered by the contractor to the boreholes.

With the hole size and depth indicated by NovaGold, the explosives delivery units will be able to load approximately 20 - 25 holes per trip. An average pattern size of 90 - 100 holes will easily be completed and shot the same day. (The amount of snow at the Galore Creek site dictates loading and shooting the same day, at least in the winter months).

NovaGold employees will drill and dewater the boreholes and prime, stem, tie in and detonate the explosives. Boreholes will be 311 mm drilled on 9.8 m by 11.3 m patterns, with a five metre explosives collar and two metre subdrill. Major blasts will occur on average about once a day.

5.5.13 Winter Operations

The project falls within a region that experiences high snowfall that could be a challenge for mobile equipment due to loss of traction and loss of visibility during snow storms. Fog may also be a problem with respect to visibility at the mine site. A study of the most economic and practical methods of managing the mining operation in this climate was undertaken to provide a preliminary review of possible adverse winter mining conditions and solutions for the Galore Creek Project.

Reduced production can be expected when visibility is severely restricted. Reduced visibility can be addressed to some extent by roadway lighting, appropriate operations protocols and onboard navigation systems.

Significant snowfall at or near the freezing point will compress into an icy surface under the tires of a large mining truck. Traction could be enhanced by the application of $\frac{3}{4}$ inch (19 mm) crushed aggregate to road surfaces. A dedicated crusher, road rakes, scarifiers and strategically located stockpiles of crushed rock will be required.

Power cables will have to be suspended above the snow pack on pole stands to avoid the risk of freezing into the snow pack as snow melts and refreezes in the moderate local climate. Disposal of snow will have to be carefully planned to avoid creation of unstable slopes or sinkholes.

A strategically placed ore stockpile will allow continued safe supply of mill feed during the most severe storms and could supplement feed during more moderate storm conditions.

5.5.13.1 Operating Scenarios

Five operating scenarios were evaluated: a purely seasonal operation that would completely shut down during the winter; a scenario that would require shutdown during almost all snow storms; partial shutdown during every snow storm; partial shutdown during only severe storms; and full operations year round.

Cost Evaluation and Analysis for Winter Operation

Each of the scenarios 1 through 5 carries a capital and operational cost (Table 5.5-13).

**Table 5.5-13
Summary of Capital and Operating Costs for Snow Fleet Scenarios**

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

The higher capital costs of the seasonal and down for all storms scenarios is due to the need to increase the size of the mining fleet to maintain overall annual production levels over a shorter operating period. The most cost effective operating scenario is Scenario 4 - Partial Operations during severe storms.

5.5.13.2 Snow Fleet Proposal

NovaGold proposes to adopt a combination of Scenarios 3 and 4, a snow support fleet sized for Scenario 4 (15 lost days) and stockpile, fuel storage and additional production sized for Scenario 3 (30 lost days). The dedicated snow removal fleet will be a loader and two additional trucks; they will also be used to make up lost production. The snow support fleet will be made up of a crusher, ten scrapers, an additional grader and miscellaneous blades and rakes. Visibility issues will be addressed with road lighting and mobile equipment GPS. Other winter material needs such as pole stands, blast hole covers, outdoor storage covers, etc. will be covered by a 5% contingency.

5.6 Mill Operations

Galore Creek ore will be processed on site to produce a copper-gold-silver concentrate for shipment to an off site smelter. The nominal milling rate will be 65,000 tonnes per day.

5.6.1 Metallurgical Description of Ore

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

Two assessments of ore hardness, in terms of Bond Ball Mill Work Index, were conducted on a range of drill hole composites, as shown in Table 5.6-1.

**Table 5.6-1
Ore Hardness of Samples by Mineral Zones**

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

In the initial tests there was no distinction between “broken” and “stick” ores according to the work index measurements. A Bond Work Index of 16 kWh/t was selected as the criteria for mill sizing based on the data in Table 5.6-1.

Ore hardness was quite variable between intervals within the same drill hole and between drill holes. The Bond Work Index ranged from 8.8 kWh/t to 18.7 kWh/t. The average of the sample suite was 13.5 kWh/t. There was no correlation between ore hardness and feed copper grade which varied from 0.1% Cu to 1.6% Cu.

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

5.6.1.1 Ore Recoveries

A pre-feasibility level metallurgical test program was completed for NovaGold by G&T Metallurgical Services Ltd. The program included preliminary flotation tests to identify a common flowsheet for the mineralized groups in the various ore zones using coarse assay reject samples, followed by locked-cycle flotation tests on drill cores of the main ore types to confirm the flowsheet, metals recovery and concentrate grade. Tests were run on discrete drill hole intervals, composites of individual drill holes and composites of various drill holes, depending on the objectives in the program.

In the preliminary test phase, the tests were run on individual drill hole composites from the following zones: Central/Main, Junction, Bountiful, Southwest, West Fork, and Copper Canyon. Copper Canyon testwork is summarized in this section although no materials from the Copper

Canyon area are included in the current mine plan. A sample from South Gold Lens identified as copper oxide (oxides are infrequent in the overall deposit) was also tested for completeness.

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

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

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

The test work lead to the conclusion that at nominal production levels recoveries will be about 91% for copper, 71% for gold and 67% for silver. At maximum production levels the recoveries are estimated to be 93% for copper, 74% for gold and 71% for silver.

5.6.2 Plant Layout

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

The mill will consist of the following unit operations:

- ore storage;
- primary crushing;
- two-stage grinding;
- copper flotation;
- rougher concentrate regrinding;
- concentrate thickening and pumping;
- tailings thickening and pumping; and
- water reclaim.

The mill will operate 24 hours per day, 365 days per year with scheduled downtime for equipment maintenance. Figures 5.6-1 and 5.6-2 provide a conceptual plant layout and simplified mill flowsheet.

5.6.3 Mineral Processing

A standard flowsheet comprising single stage crushing, two-stage semi-autogenous-ball mill grinding and three-stage flotation with regrinding will be used for the Galore Creek plant. A semi-autogenous ball mill circuit (SABC) grinding circuit with pebble crushers has been selected in view of the relatively hard ore and the possibility of pebble generation. Membrane pressure filters will be used to achieve minimum concentrate cake moisture.

5.6.3.1 Crushing

Run of mine ore, or stockpiled ore as required to meet mill feed requirements, will be hauled by truck to the primary crusher facility.

Ore will be dumped directly from haul trucks into the dump pocket. A rock breaker will be installed over the dump pocket to break large pieces of ore and avoid plugging the crusher. The ore will be crushed in the gyratory crusher to nominally 80% passing 150 mm and conveyed to a 600 t coarse ore surge bin (Figure 5.6-3). From the bin, the ore will be withdrawn with an apron feeder onto a coarse ore conveyor and conveyed to a stockpile having a live capacity of 32,000 t. Water sprays, fog systems and dust collectors with pickups around the crusher, conveyors and ore transfer points will be installed to minimize fugitive dust in the facilities.

5.6.3.2 Grinding

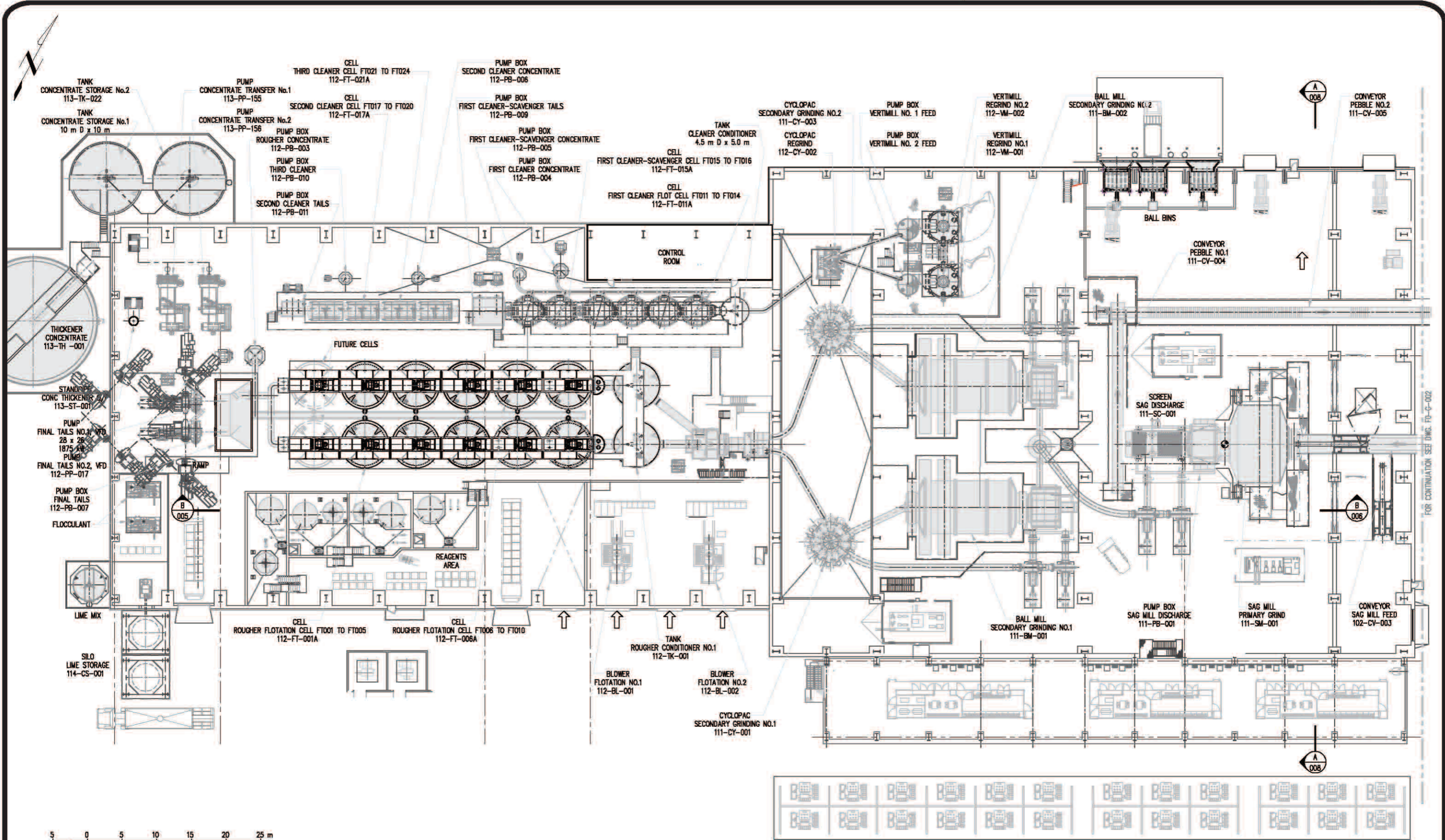
The grinding circuit will consist of: one semi-autogenous (SAG) mill with a trommel, vibrating screen, two ball mills with trommels, two cyclopacs, two pebble cone crushers and pebble recycle conveyors (Figure 5.6-4).

Ore will be reclaimed from the coarse ore stockpile via three apron feeders and conveyed to the two-stage grinding circuit. The apron feeders will be controlled by a weightometer on the SAG mill conveyor to achieve a nominal operating rate of 2,700 t/h. The ore will be ground to a final product size of 80% passing 170 μm (approximately) using a 11.58 m diameter by 6.10 m, 19,500 kW SAG mill and two 7.93 m diameter by 10.98 m, 11,190 kW ball mills operating in parallel.

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

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

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

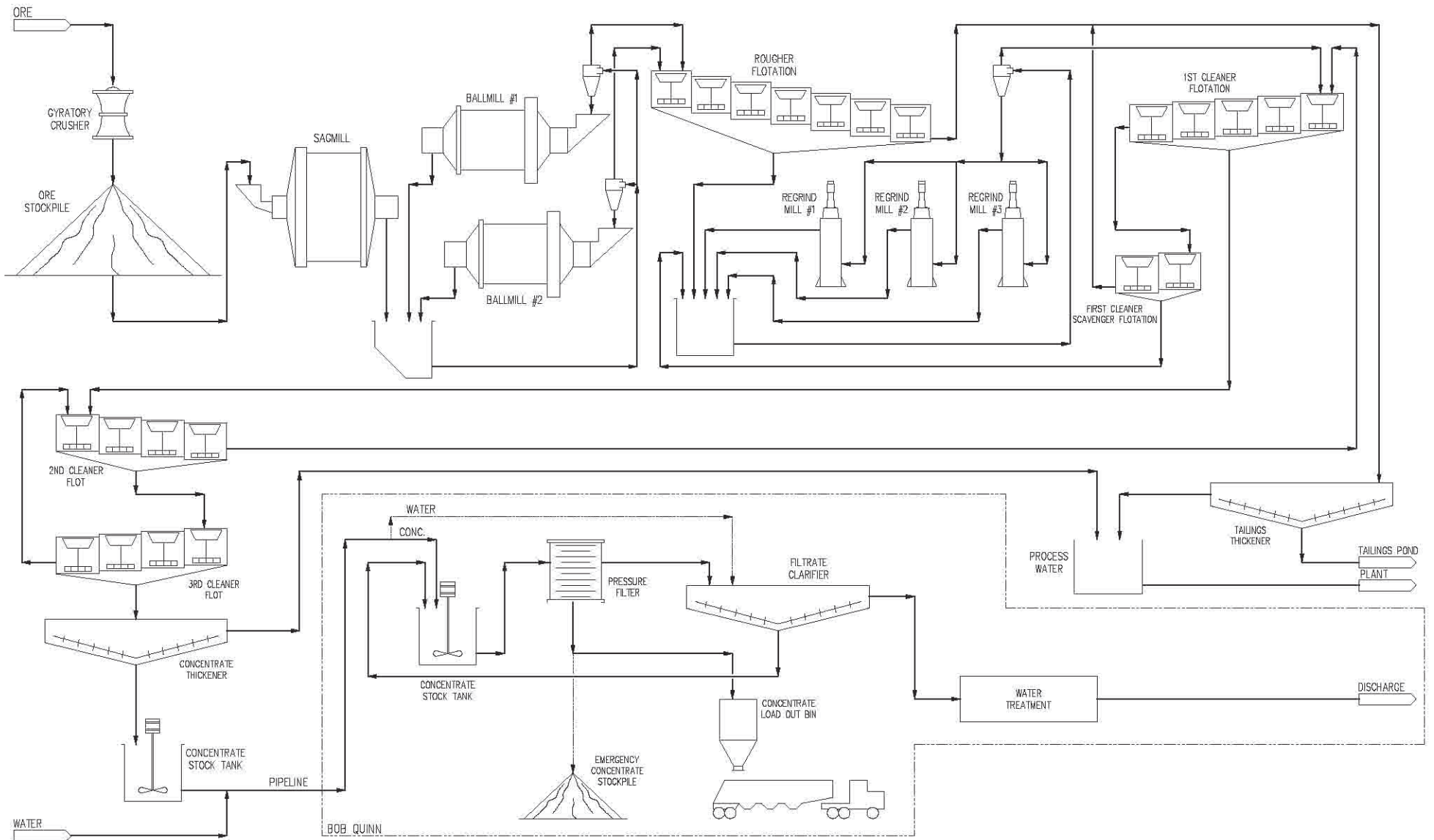


Source: Hatch & Associates

Conceptual Plant Layout

FIGURE 5.6-1





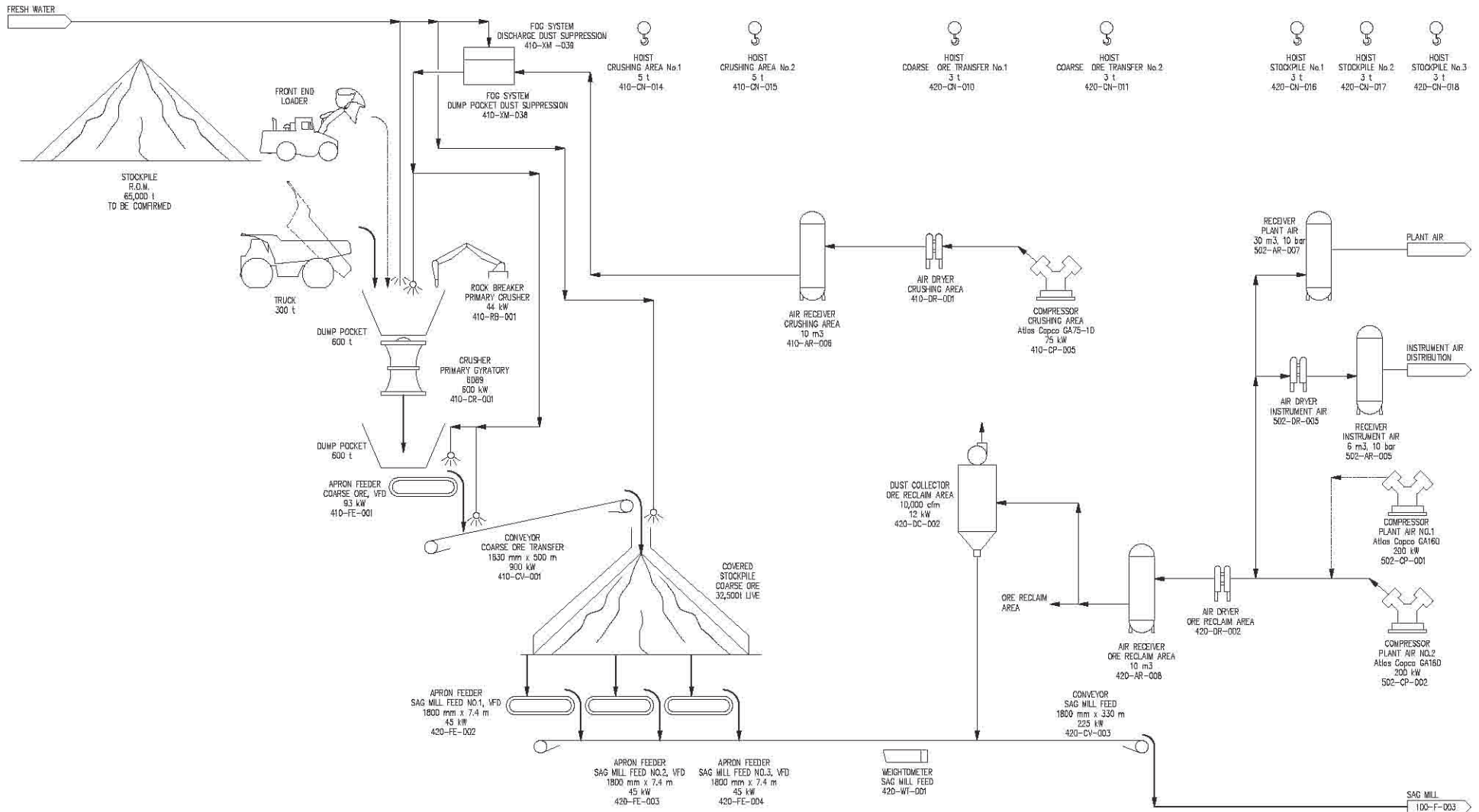
Source: Hatch & Associates



Simplified Mill Flowsheet

FIGURE 5.6-2





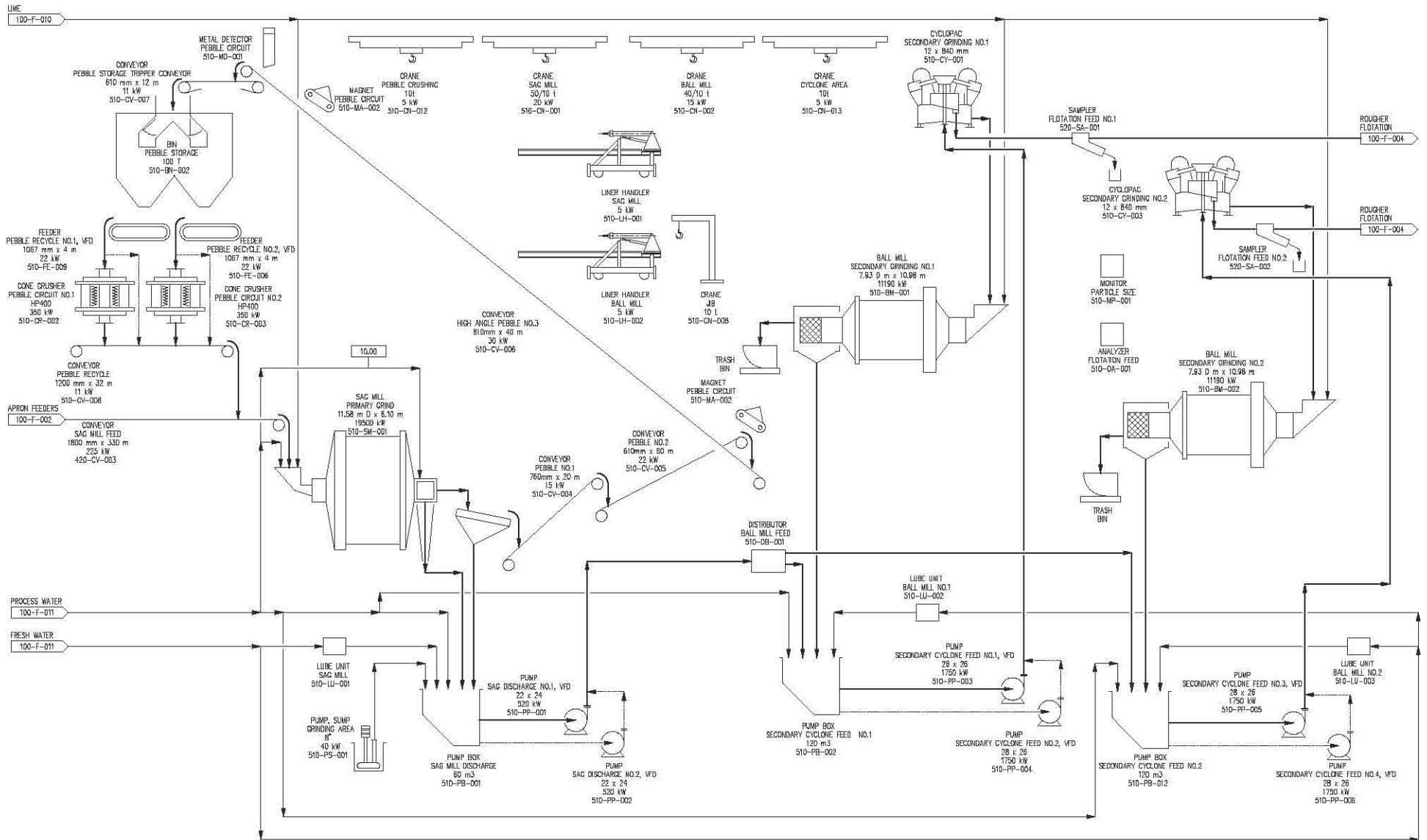
Source: Hatch & Associates



Crushing Flowsheet

FIGURE 5.6-3





Source: Hatch & Associates



Grinding Flowsheet

FIGURE 5.6-4



5.6.3.3 Flotation

Each of the ball mill cyclone overflows will be fed to a conditioner tank and a bank of seven 200 m³ rougher flotation cells. The concentrate from the two banks will be combined and pumped to the regrind circuit while the tailings will be pumped to the tailings thickener (Figure 5.6-5).

5.6.3.4 Rougher Concentrate Regrind

The rougher concentrate will be ground in three 1,119 kW vertimills operating in closed circuit with a cyclopac to achieve a product grind of 80% passing 45 µm (approximately) (Figure 5.6-6).

The cyclopac underflow will be split to the three vertimills while the overflow will feed the cleaner circuit. A sampling system and particle size monitor will be installed on the cyclone overflow to control the vertimill operation and achieve the target grind.

5.6.3.5 Cleaner Flotation

The cyclone overflow will be upgraded in a three-stage cleaner flotation circuit (Figure 5.6-7). The first cleaner will operate in open-circuit while the second and third cleaners will operate in closed circuit.

The first cleaner concentrate will be pumped to the second cleaner flotation cells while the tailings will flow into the first cleaner scavenger cells. The first cleaner scavenger concentrate will be reground and recycled to the first cleaner while the tailings will be pumped to the final tailings pump box then pumped to the tailings thickener together with the rougher tailings. The first cleaner will have five 50 m³ cells while the cleaner scavenger will have two 50 m³ cells.

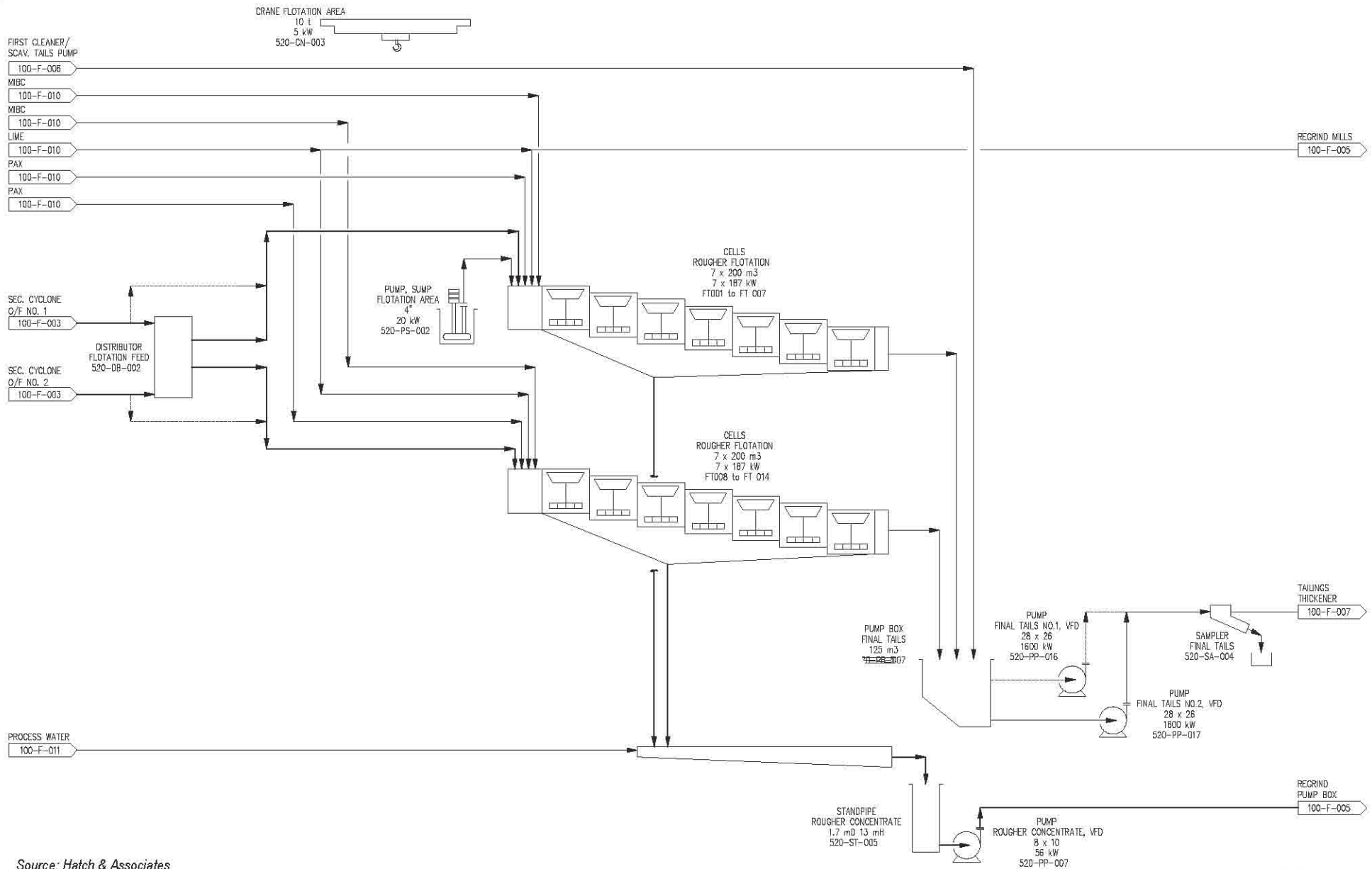
The second and third cleaner stages will be arranged as a single bank such that the third cleaner tailings will flow by gravity to the adjoining second cleaner cells. Each stage will have four 20 m³ cells. The second cleaner concentrate will be pumped to the third cleaner and the third cleaner concentrate will be the final concentrate feeding the concentrate dewatering circuit. The second cleaner tailings will be pumped back to the first cleaner circuit.

The flotation reagents, lime, PAX (potassium amyl xanthate) and MIBC (methyl-isobutyl-carbinol), will be added to the circuit to achieve the desired metals recovery and concentrate grade.

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

5.6.3.6 Concentrate Thickening and Storage Tanks

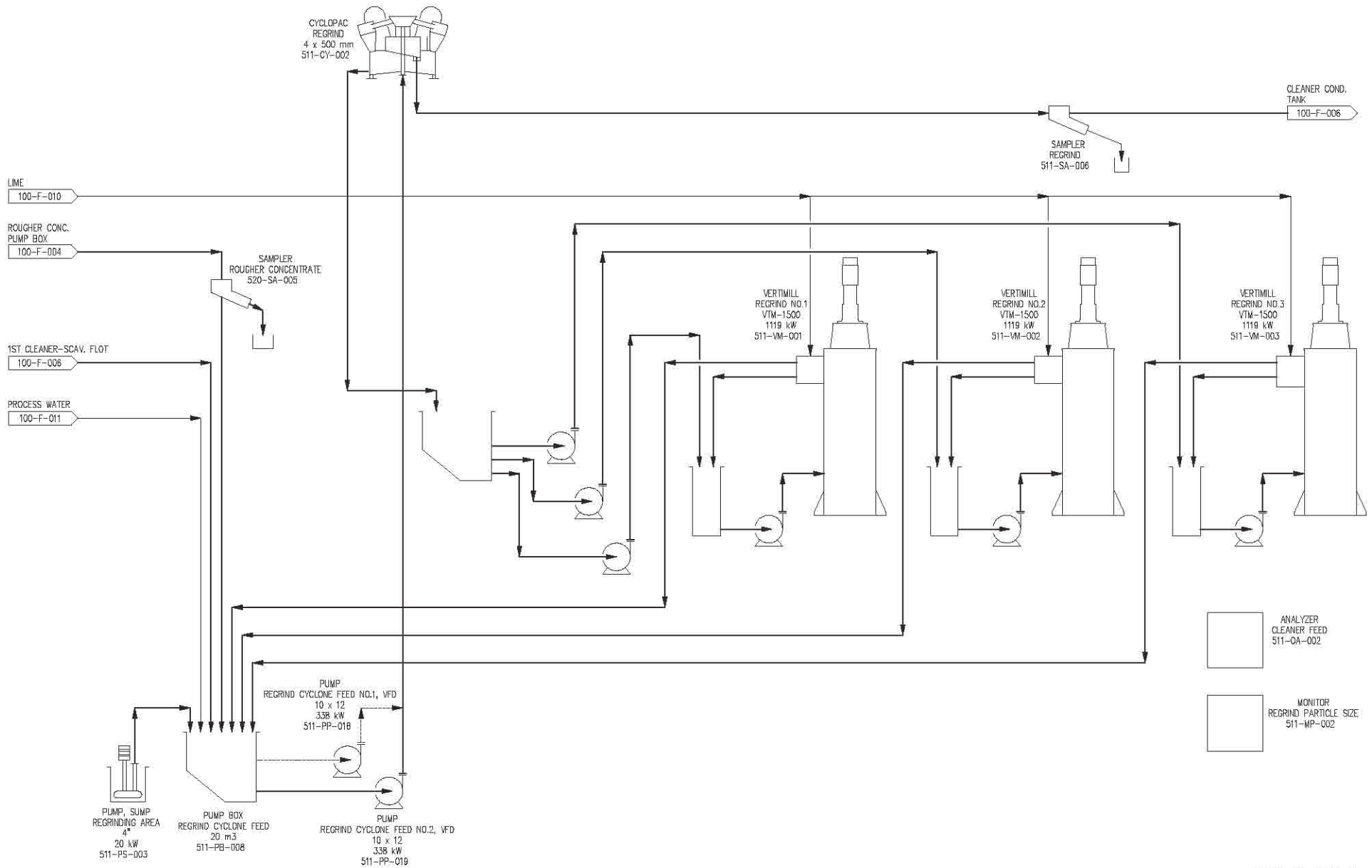
Concentrate will be sent to a 22 m diameter thickener tank where flocculant will be added to raise the underflow density to about 60% solids (Figure 5.6-8). Flocculants will be selected during the detailed design. Concentrate thickener overflow will be recycled within the plant as process water via the process water tank. The thickened concentrate will be pumped to two concentrate stock tanks, installed at the mine site to provide surge capacity for the concentrate



Source: Hatch & Associates

Rougher Flotation Flowsheet

FIGURE 5.6-5



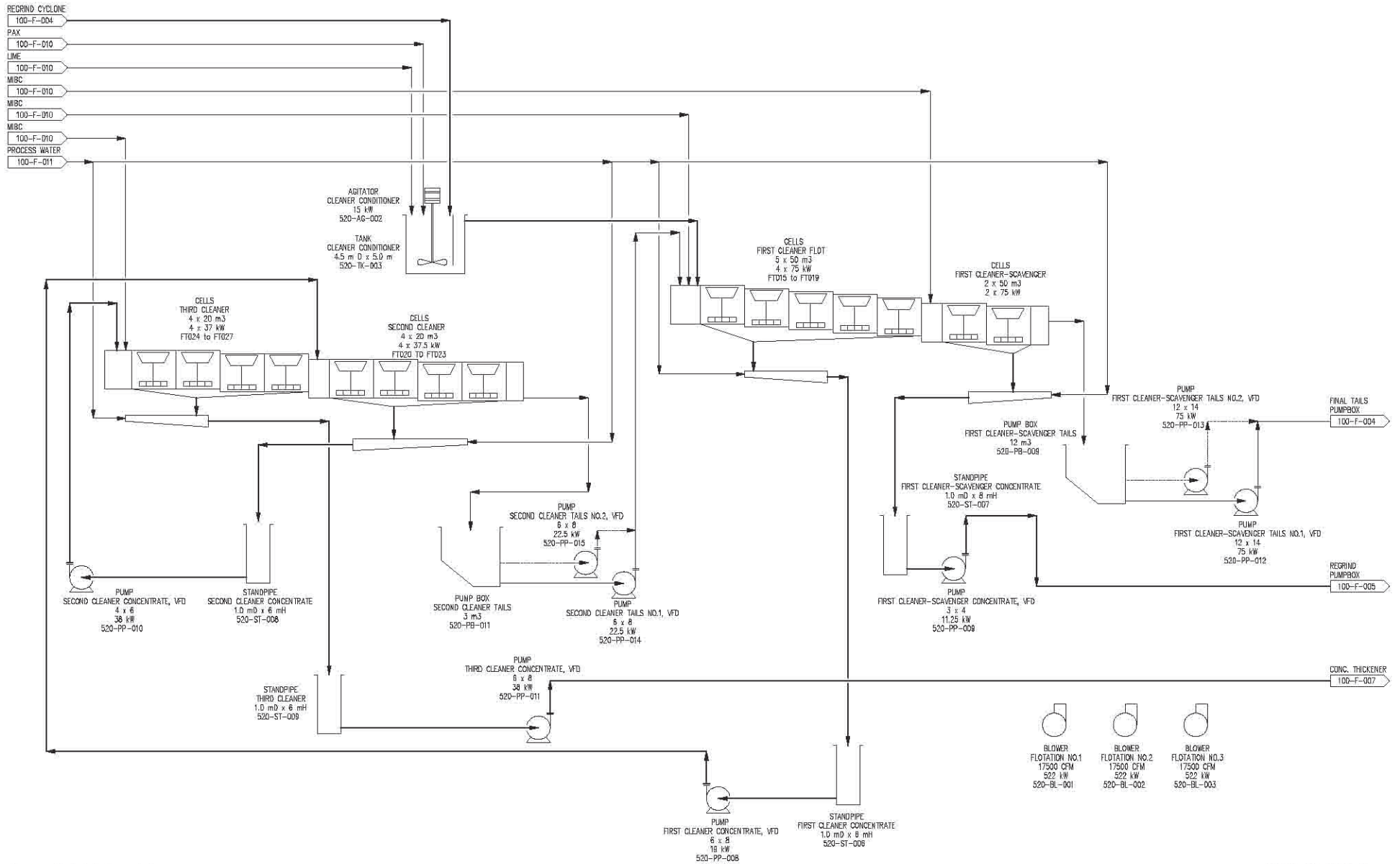
Source: Hatch & Associates



Regrind Flowsheet

FIGURE 5.6-6





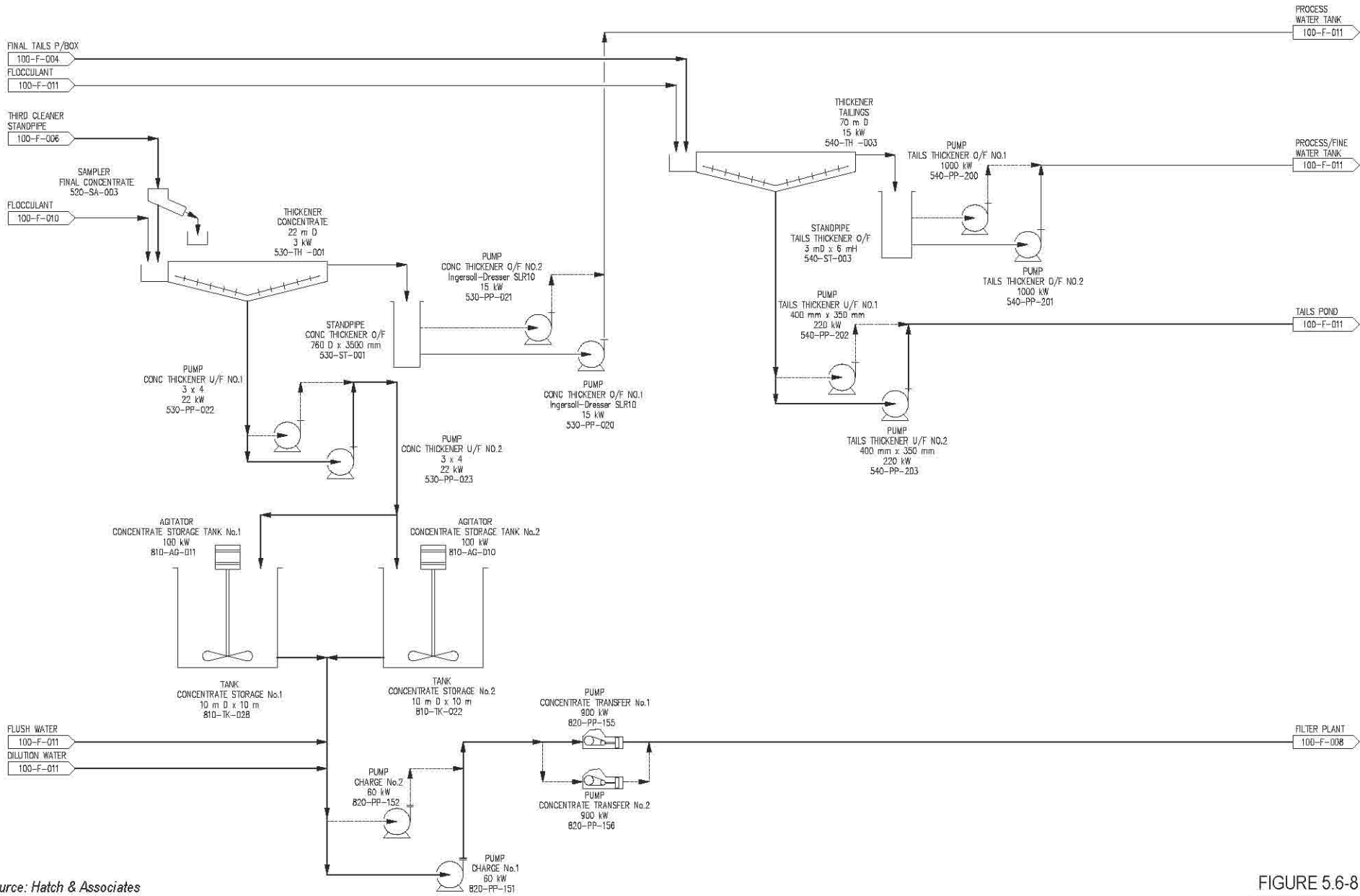
Source: Hatch & Associates



Cleaner Flotation Flowsheet

FIGURE 5.6-7





Source: Hatch & Associates



Concentrate and Tailings Dewatering Flowsheet

FIGURE 5.6-8



pumping system. The concentrate production rate will vary due to variable ore grades and dictates that the slurry will be batch-pumped in order to maintain the required line velocity. The use of two tanks will permit uninterrupted operation during tank maintenance. The concentrate slurry will be diluted to about 57% solids for pumping to the filter plant near Highway 37.

5.6.3.7 Tailings Disposal Pumping System

The final combined tailings from rougher flotation and first cleaner scavenger flotation will be pumped to a 70 m diameter tailings thickener where a flocculant will be added to achieve an underflow at about 60% solids content (Figure 5.6-8). Underflow will be pumped to the tailings pond. Overflow will be pumped to the process water tank. A pump and water line will permit reclaim of tailings water from the pond for process water.

Tailings water quality testwork and results are described in Appendix 5-G, *Process Pilot Plant Environmental Test Work*.

5.6.3.8 Reagent Handling System

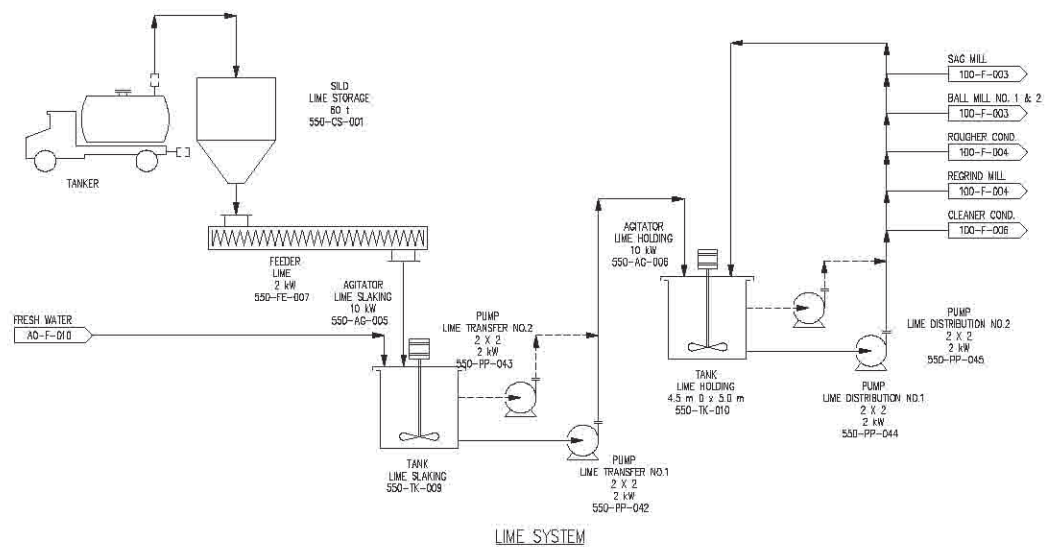
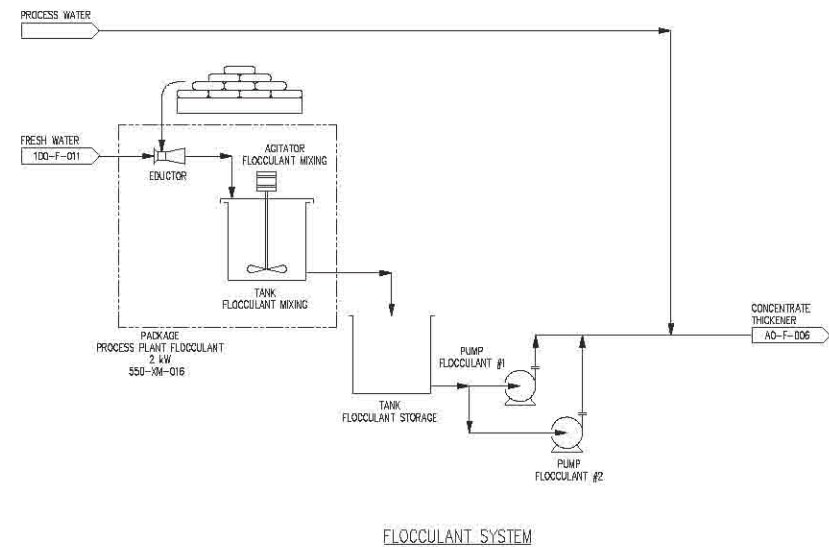
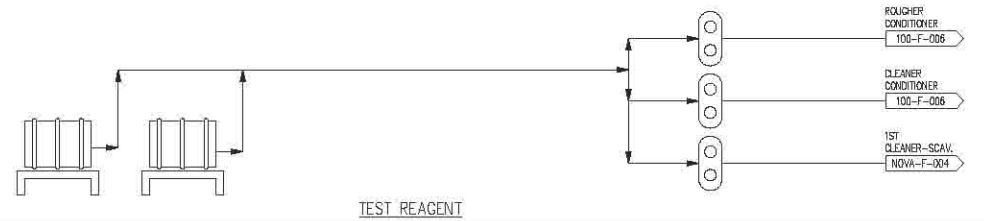
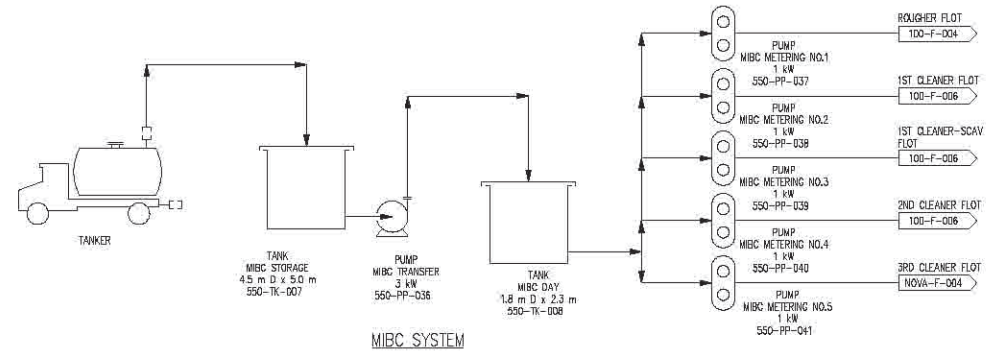
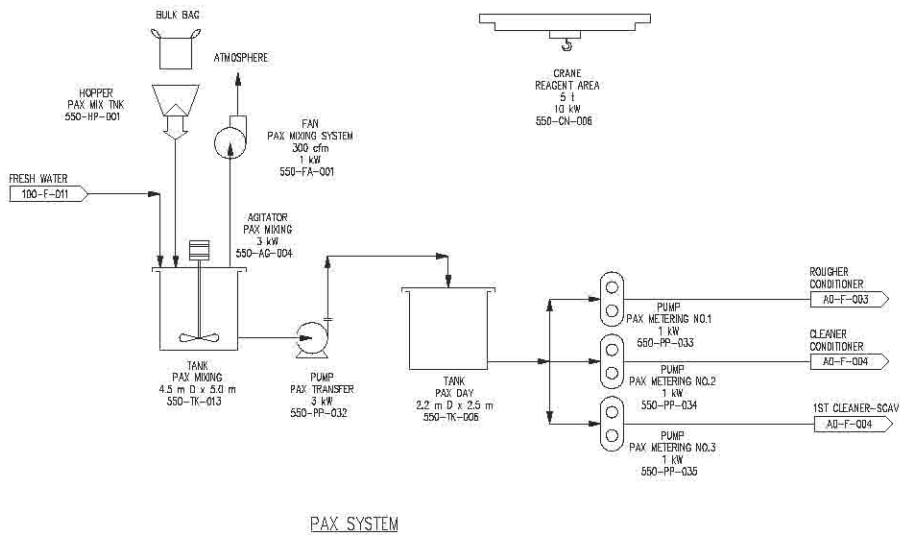
Potassium amyl xanthate (PAX), Methyl Isobutyl Carbinol (MIBC), lime (CaO) and flocculants will be used in the process. Figure 5.6-9 shows the reagent systems for PAX, MIBC and CaO and for flocculant. Bulk handling systems will be installed to suit the consumption and remote location of the plant.

PAX will be delivered as a powder in 1,100 lb super sacs. It will be dissolved in water on a batch basis, in an agitated 4.5 m diameter by 5 m high mix tank to a 10% w/w solution. Consumption will be one super sac per day. The mix tank will provide one week's supply of reagent. The PAX solution will be transferred on demand to a 2.2 m diameter by 2.5 m high day tank located in the flotation area. From the day tank, metering pumps will meter the PAX solution to the addition points in the flotation circuit.

MIBC will be delivered as a solution in 5,000 gallon tankers and transferred to a 4.5 m diameter by 5 m holding tank with a 28 day capacity. It will be pumped on demand to a 1.8 m by 2.3 m day tank located in the flotation area then metered via individual metering pumps to the addition points.

Lime will be supplied as quicklime in tankers and pneumatically transferred to a 100 tonne silo then slaked to a 20% slurry. The slaked lime slurry will be transferred to an agitated 4.5 m by 5 m holding tank. Lime slurry will be pumped continuously in a closed loop with the holding tank. Lime additions will be pulsed from the loop based on pH control at the addition points.

All reagents and process tankage will be located within containment areas, sized to accommodate 110% of the largest vessel therein, to ensure complete containment in the event of an accidental spill. Appropriate ventilation, fire and safety protection and MSDS stations will be provided at the necessary locations.



Source: Hatch & Associates



Process Flowsheet - Reagent Systems

FIGURE 5.6-9



5.7 Concentrate Handling and Transport

NovaGold investigated several options for transportation of concentrates, including trucking of dewatered concentrates directly from the mine to the Port of Stewart, transporting the concentrate in slurry form by pipeline from the mine to the port, and several combinations of pipeline and trucking. These alternatives are discussed in Chapter 11. NovaGold has chosen to construct a concentrate slurry pipeline from the mine to a filter (or dewatering) plant near the junction of the mine access road and Highway 37 and to haul dewatered concentrate by truck from that point to Stewart.

Although this alternative adds significant capital cost to the project, it has a lower life of mine operating cost than other alternatives, reduces dependency on the access road during difficult winter weather periods, reduces traffic on, and environmental impacts of, the access road and greatly reduces the risk of accidents on the road.

A second pipeline, to transport diesel fuel to the mine site from Highway 37, will be constructed parallel to the concentrate pipeline to further reduce the mine's dependence on daily road access. This small pipeline will eliminate the requirement for diesel tanker truck traffic on the access road, reduce the weekday traffic volume on the road by a further 10%, reduce the potential for diesel spills due to accidents and improve project safety.

5.7.1 Concentrate Pipeline

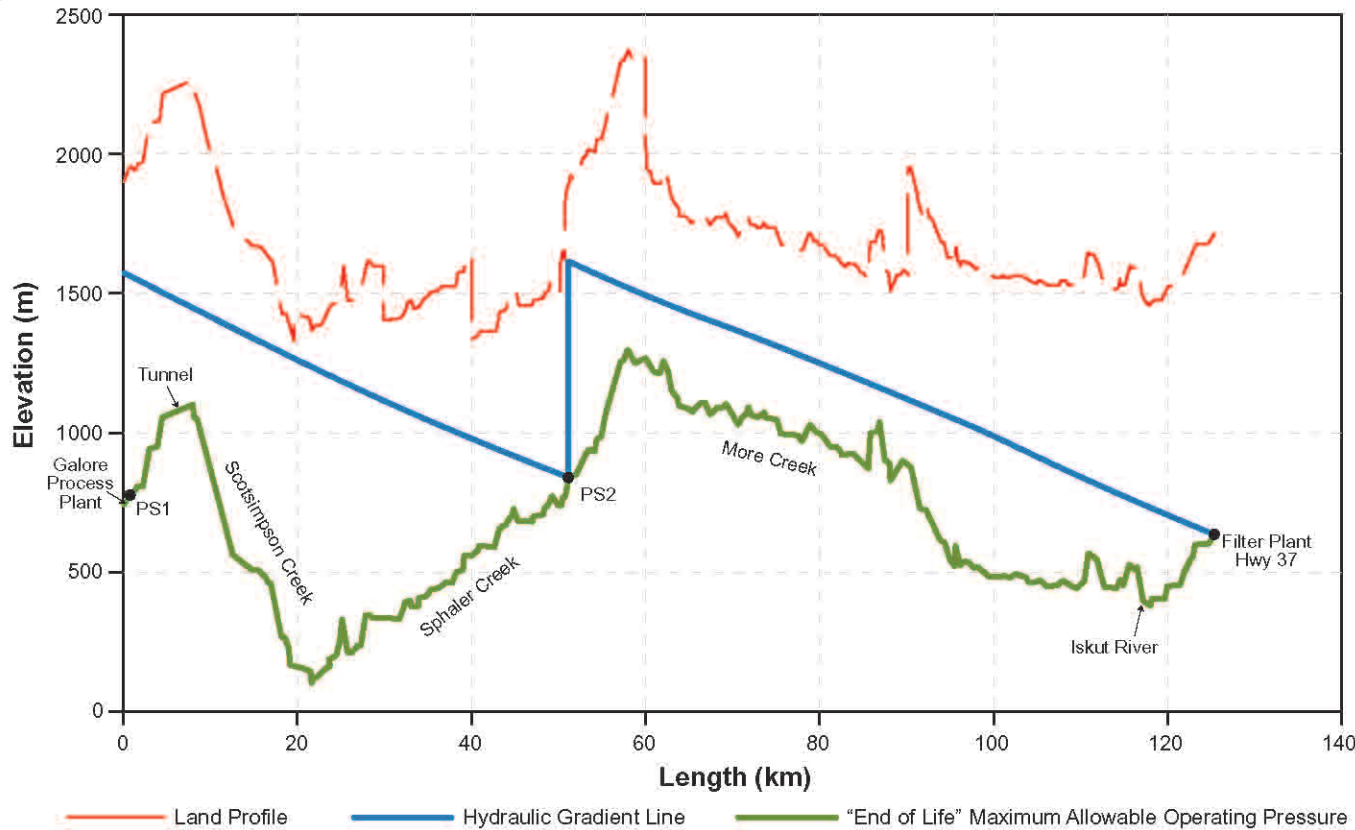
Pipeline Systems Inc. (PSI) was engaged to perform a conceptual study of the slurry pipeline transportation system to assess technical feasibility and develop order of magnitude cost estimates for the system.

The design criteria used by PSI for the study were:

- throughput of 1,000 to 2,000 t/d (design hourly throughput of 87.7 t/h at 95% availability);
- pipe length about 130 km.;
- specific gravity 4.2;
- solids concentration 52 to 58%;
- pH 10; and
- slurry temperature 15°C at the pipe inlet and minimum 5°C in the pipe.

5.7.1.1 Pumping System, Alignment, Specifications and Operations

The slurry will be pumped using large piston diaphragm pumps. There will be two pump stations; one at the plant site plus a booster station near the headwaters of Sphaler Creek. Two pumps will be installed at each site, one for operation and a second for standby. These pump stations are shown in Figure 5.7-1 as PS1 and PS2. This figure also illustrates the results of a hydraulic model showing land profile, hydraulic gradient line and operating pressure.



Slurry Characteristics

Solids S.G.	4.2
Temp (°C)	5
B'	2.92
Von Karman	0.9
Durand	150
Roughness (inches)	0.002
Slurry SG	1.74

Particle Size Distribution

P% 65	99.98
P% 100	99.91
P% 150	99.42
P% 200	96.74
P% 325	81.35
P% 400	76.29

Pipeline Characteristics

OD (inch)	6.625
Average Wall (inch)	0.242
Liner (inch)	0.000
ID (inch)	6.140
Wear Rate KP 0-20	8 (mpy)
Wear Rate KP 20-125	4 (mpy)

Length Pipe	125.18	km
Steel tonnage	3,229.98	
API5L X	65.00	

Pipeline Throughput		
Flowrate	89.79	m³/h
Cw%	56%	
Throughput	87.70	tph

Line Velocity

Minimum	1.31 m/s
Maximum	1.40 m/s

MAOP Clearance

Minimum	62.6 m
Maximum	1,076.0 m

Pumping Requirements

	PS 1	PS 2	
Location	0.0	50,677.2	KP
TDH	834.6	823.5	m
Number of pump	1op + 1stb	1op + 1stb	
Discharge Pressure	2,080.2	2,052.8	psig
	143.46	141.57	BAR
Pump Efficiency (%)	85%	85%	
Motor HP	597.5	589.6	HP
Motor kW	445.5	439.7	kW

Hydraulic Gradient

Minimum	12.37 m/km
Maximum	15.85 m/km

Profile Clearance

Minimum	0.0 m
Maximum	1,132.3 m

Choking

Head Loss	0.00 m
-----------	--------

Minimum Operating Safe Velocity	1.28
Safety on Flow	1.06
Length Factor	1.05

Source: PSI Engineering

The pressure in the line will vary between 14,340 KPa at the pump stations to 14,150 KPa. The pipeline system will incorporate concepts, equipment and operating conditions proven in currently operating slurry pipelines. Design of the system from a safety standpoint will be guided by the latest American Society of Mechanical Engineers (ASME) B31.11 Slurry Transportation Piping Systems code.

The pipeline will be buried over its entire length to a depth in compliance with ASME B31.11. Depths will range from 1.6 to three metres depending upon the geotechnical risks at each site, with deeper burial proposed for sites, such as potential debris flow channels, with higher risk. The alignment will generally be under the access road ditch, but minor diversions will be required to maintain appropriate grades to avoid the settlement of solids in low points during shut downs. Directional drilling will be used to install the pipeline under four streams and it will be attached to bridges for another 19 crossings (Table 5.7-1). Elsewhere it will be buried in a trench under the crossing. Where the pipeline is attached to bridges it will be insulated to prevent freezing.

**Table 5.7-1
Pipeline Stream Crossings**

Stream Name	Station	Length (m)	Crossing Type
Swamp Creek	25 + 640	400	Horizontal drilling
Muskwie Creek	30 + 340)	Horizontal drilling
Fish S1	30 + 485)Aggregate 550	Horizontal drilling
Fish S	30 + 550)	Horizontal drilling
Bridge (No Name)	3 + 740	30	Existing bridge
Moose Creek	15 + 510	107	Bridge
More Creek	39 + 100	91	Bridge
Blurry Creek	40 + 320	30	Bridge
Snow Creek	44 + 335	30	Bridge
More Creek (Canyon)	48 + 280	400	Bridge
Wealth Creek	54 + 650	90	Bridge
Artic Creek	63 + 770	34	Bridge
Bluff Creek	68 + 560	34	Bridge
Round Lake Creek	75 + 780	40	Bridge
Eros Creek	82 + 582	49	Bridge
Yuri Creek	83 + 985	88	Bridge
Saddle Creek	89 + 405	76	Bridge
Elliot Creek	94 + 075	27	Bridge
Sphaler Creek#1	96 + 893	67	Bridge
Parolin Creek	101 + 674	24	Bridge
Sphaler Creek #2	104 + 405	49	Bridge
O'Driscoll Creek	107 + 615	27	Bridge
Loptson Creek	113 + 30	49	Bridge

(continued)

**Table 5.7-1
Pipeline Stream Crossings (completed)**

Stream Name	Station	Length (m)	Crossing Type
Bridge (No Name)	42 + 320	30	Trench
Alexander Creek	44 + 210	18	Trench
Hankin Creek	45 + 200	300	Trench
Glove Creek	46 + 110	31	Trench
Natavas Creek	46 + 385	27	Trench
Schuss Creek	46 + 760	16	Trench
Mathew Creek	47 + 250	16	Trench
Debris Creek	48 + 820	31	Trench
Alignment Creek	51 + 110	24	Trench
Tahltan Creek	52 + 345	37	Trench
Run Out Creek	55 + 300	31	Trench
Nahata Creek	55 + 510	50	Trench
Bacchus Creek	56 + 055	27	Trench
Bent Tree Creek	57 + 420	16	Trench
Windfall Creek	59 + 530	37	Trench
Stump Creek	61 + 710	37	Trench
Spruce Creek	65 + 830	37	Trench
Roca Creek	66 + 670	37	Trench
Hickman Creek	70 + 570	37	Trench
Three Creek	73 + 715	76	Trench
Sam Creek	79 + 780	30	Trench
Holly Creek	81 + 475	30	Trench
Pheno Creek	85 + 683	34	Trench
Harold Creek	88 + 555	21	Trench
Mauer Creek	90 + 205	73	Trench
Glacier Creek	95 + 305	73	Trench
Adema Creek	97 + 764	34	Trench
Daniels Creek	100 + 355	15	Trench
Hardy Creek	100 + 650	13	Trench
Masson Creek	101 + 533	30	Trench
Reed Creek	104 + 960	37	Trench
Hiscock Creek	105 + 770	16	Trench
Sphaler Creek #3	111 + 147	37	Trench
Hayden Creek	117 + 50	26	Trench
Rioux Creek	118 + 35	23	Trench
Smit Creek	118 + 510	15	Trench
Ollenberger Creek	124 + 600	15	Trench
Moraine Creek	126 + 470	13	Trench
Icefield Creek	126 + 660	15	Trench
Last Creek	126 + 860	23	Trench

Control and monitoring of the concentrate pipeline will be from the mine site central control room. Control of the system will be automatic in the steady state mode with operator intervention required during process upsets.

The pipeline maximum operating point assumes a daily concentrate production of 2,000 tonnes. At times due to operating factors, ore quality, etc., the actual production rate will be lower, requiring batch operation.

Batch operation will start with full slurry tanks. The slurry will be delivered through the pipeline at the selected operating velocity. When the tanks reached the minimum operating level, the tank valves will be closed and water will be delivered into the pipeline at the same flow rate as selected for the slurry to ensure the slurry is moved safely through the pipeline. The mine site storage tanks will then be allowed to refill.

When the tanks are full, water delivery will be stopped and slurry flow resumed. This cycle will be repeated until production is high enough to support continuous pipeline throughput.

If the slurry batch in the pipeline is delivered to the terminal before the mine site storage tanks are full, the pipeline could be shut down filled with water.

The pipeline is designed to be shut down full of slurry, however operating experience suggests that multiple shutdowns in the pipeline of the same slurry batch can result in density gradients in the system (plugs of high concentration slurry) that have, in small diameter pipe, caused pipeline plugs that had to be mechanically removed.

At expected normal production levels slurry will be pumped through the pipe using large piston diaphragm pumps in batch mode with three batches in the pipeline at any time. Over 660,000 cubic metres of water will be used each year in the slurry and as batch water and will be discharged, after treatment, from the dewatering facility. The pipe will be high density polyethylene (HDPE) lined welded carbon steel with a nominal 7.625-inch (168.3 mm) outside diameter and varying wall thickness.

Under normal operating conditions, heat from the slurry and heat generated by pipeline friction will warm the surrounding soil. This soil will provide an insulating layer to mitigate heat losses during cold weather, preventing freezing.

A small choke station consisting of ceramic orifices in pipe loops will be installed at the dewatering plant to dissipate pipeline pressure if required. If head (pressure) dissipation is required, valves at the bottom of the loops will close, forcing the flow through the orifices. If no dissipation is needed, the valves will open and the line flow will be diverted directly to the agitated head tank at the filter plant.

In a planned short-term shutdown, or in an emergency shutdown, the pipeline will be assumed to be full or partially full of slurry. The pumps will be stopped and the terminal valve closed. The shutdown sequence will be programmed in the Supervisory Control and Data Acquisition (SCADA) system software, such that this action can be quickly and safely executed (emergency stop).

By maintaining a limitation on the installed pipeline slopes, the solids will settle in-place, leaving a clear fluid channel above the settled solids. Low points in the pipeline may accumulate a relatively full cross section of settled solids due to movement from a higher point in the profile after shutdown. However, commercial operating experience indicates that slopes less than 15% do not present a risk to pipeline plugging during shutdowns.

In a planned long-term shutdown, an orderly displacement of solids from the pipeline will occur by the introduction of flush water. It will take about 27 hours to fully flush the slurry from the line. If a long-term shut down is needed in winter, displacing the water with air or nitrogen on the pipeline may be required to eliminate the risk of pipeline freezing.

5.7.1.2 Protection System

The pipe would have five pressure monitoring stations, one at the mine pump station, three at intermediate monitoring stations and one at the terminal station to support pipeline operations and for leak detection. The monitoring stations will be powered by independent propane-fuelled generators (2 kW each). The leak detection system will issue a warning and predict the location of a leak within 2 to 15 minutes of an occurrence. The principles of leak detection are a comparison of the flow rates, pressures and densities along the pipeline based on the flow regime in the line and the position of the pipeline valves.

Isolation valves will not be installed along the line. Experience elsewhere has shown that isolation valves in concentrate slurry pipelines disturb the flow pattern of the slurry, causing extensive scouring and abrasion leading to pipe failures

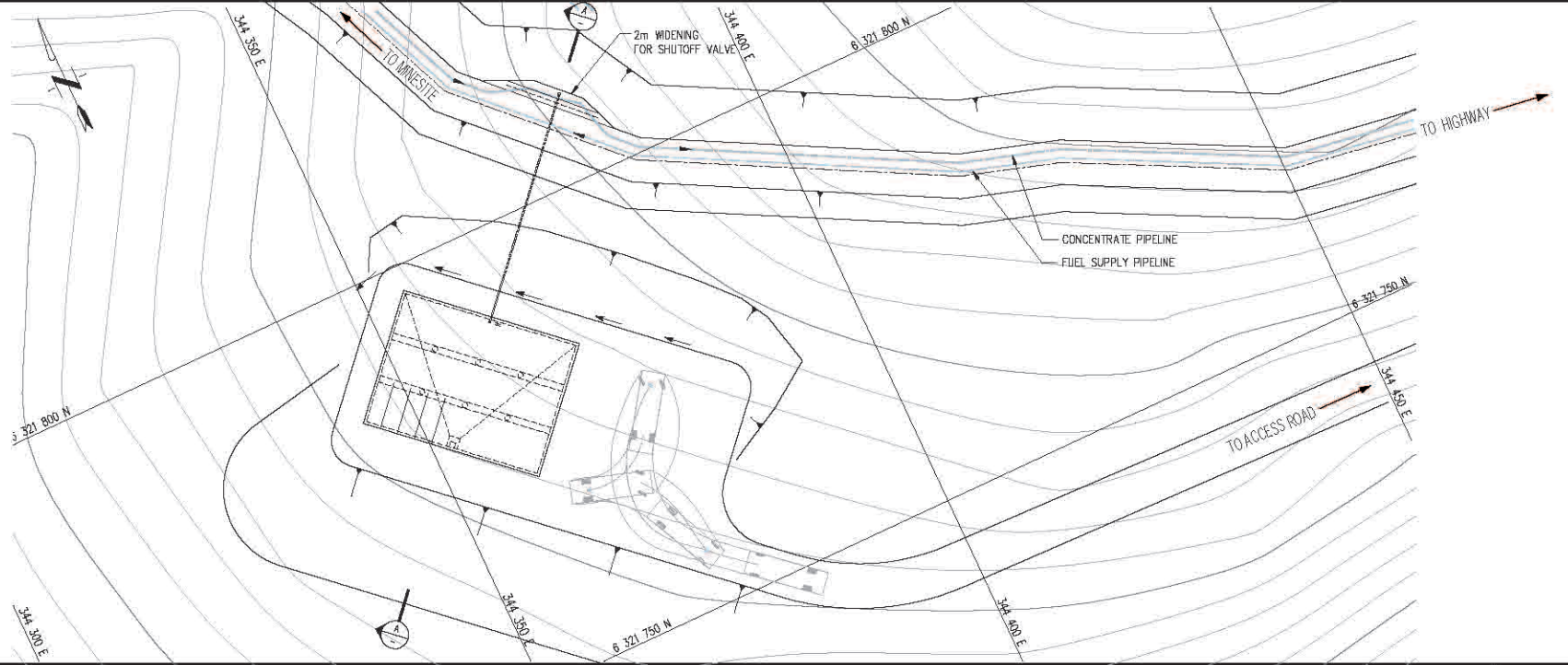
The pipeline will be equipped with a full cathode protection system supplementing the external corrosion protection pipe coating. The system will consist of rectifiers at each pump station and at the terminus of the line. Sacrificial anodes will be used to supplement the anti-corrosion system. The pipeline will be insulated at exposed creek and river crossings. The buried sections of the line will not require insulation.

A 1,200 m³ internally supported covered tank will be constructed near km 115 of the access road at the low point of the pipeline in the Lower Sphaler Creek area to provide an emergency drain point in case the pipeline has to be shut down (Figure 5.7-2). The tank will be capable of containing the drainable capacity of the pipeline between the Sphaler and Scotsimpson creeks divide and Round Lake. In an emergency NovaGold will tap into the pipeline via a 75 mm tie isolated by a high pressure ball valve to drain the pipeline into the tank. Any concentrate drained to the tank would be recovered by vacuum pumping at a later date.

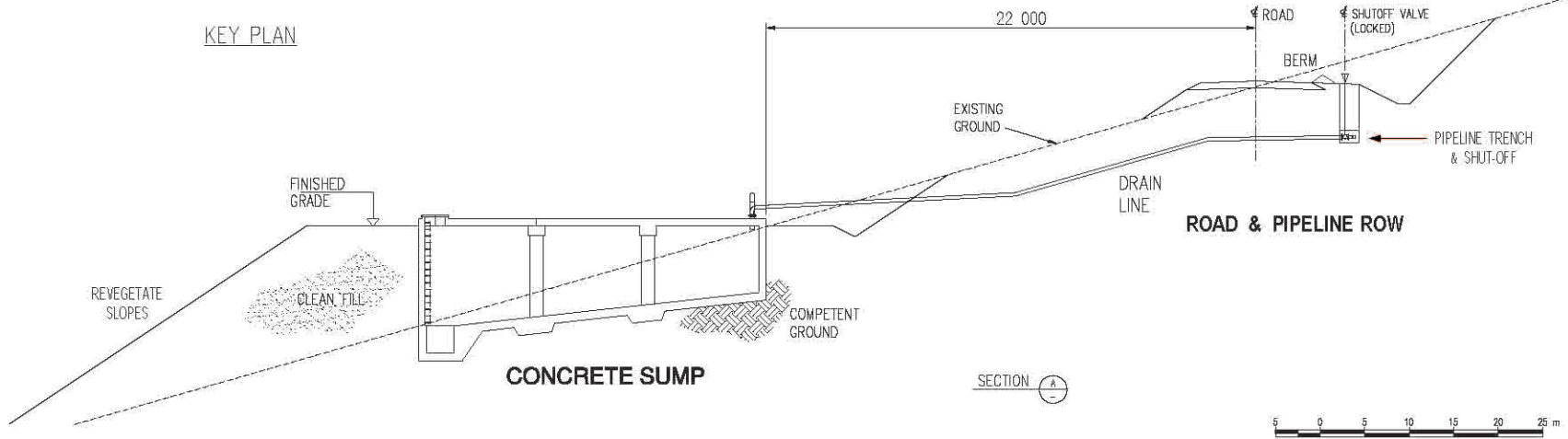
5.7.2 Filtration, Storage and Loadout Facility

5.7.2.1 Location

The filter plant will be located at the terminus of the concentrate slurry pipeline, adjacent to the access road approximately three kilometres west of Highway 37 (Figure 5.2-4).



KEY PLAN



Source: Hatch & Associates

Concentrate Pipeline Low Point Drain

FIGURE 5.7-2

5.7.2.2 Plant Layout and Process

The filter plant will be equipped to de-water the concentrate and treat all water received at the plant through the slurry pipeline prior to discharge. Consequently, the filter plant will include water treatment facilities to ensure that the discharge water is clean and readily meets discharge criteria. The plant will consist of the following: concentrate stock tank, two pressure filters, filtrate thickener, reactor clarifier, sand, cartridge and carbon filters and flocculant and sulphuric acid systems (Figure 5.7-3). The filter plant will also house the equipment for the necessary utilities such as power, compressed air, fire water and heating and will have an adjacent accommodation complex for about 30 plant workers, a covered concentrate storage area and truck loading and weighing facilities (Figure 5.7-4).

Concentrate slurry will be received in a 8 m diameter by 8 m high agitated stock tank at about 56% solids by weight, then filtered through the two 108 m² pressure filters to achieve 8% to 10% moisture. The filter cake will be conveyed to a 580 t loadout bin or a large concentrate storage shed until collected for transportation by truck to the tidewater port of Stewart, British Columbia. The peak design throughput will be about 2,000 t/d with normal operations being about 25,000 tonnes per month.

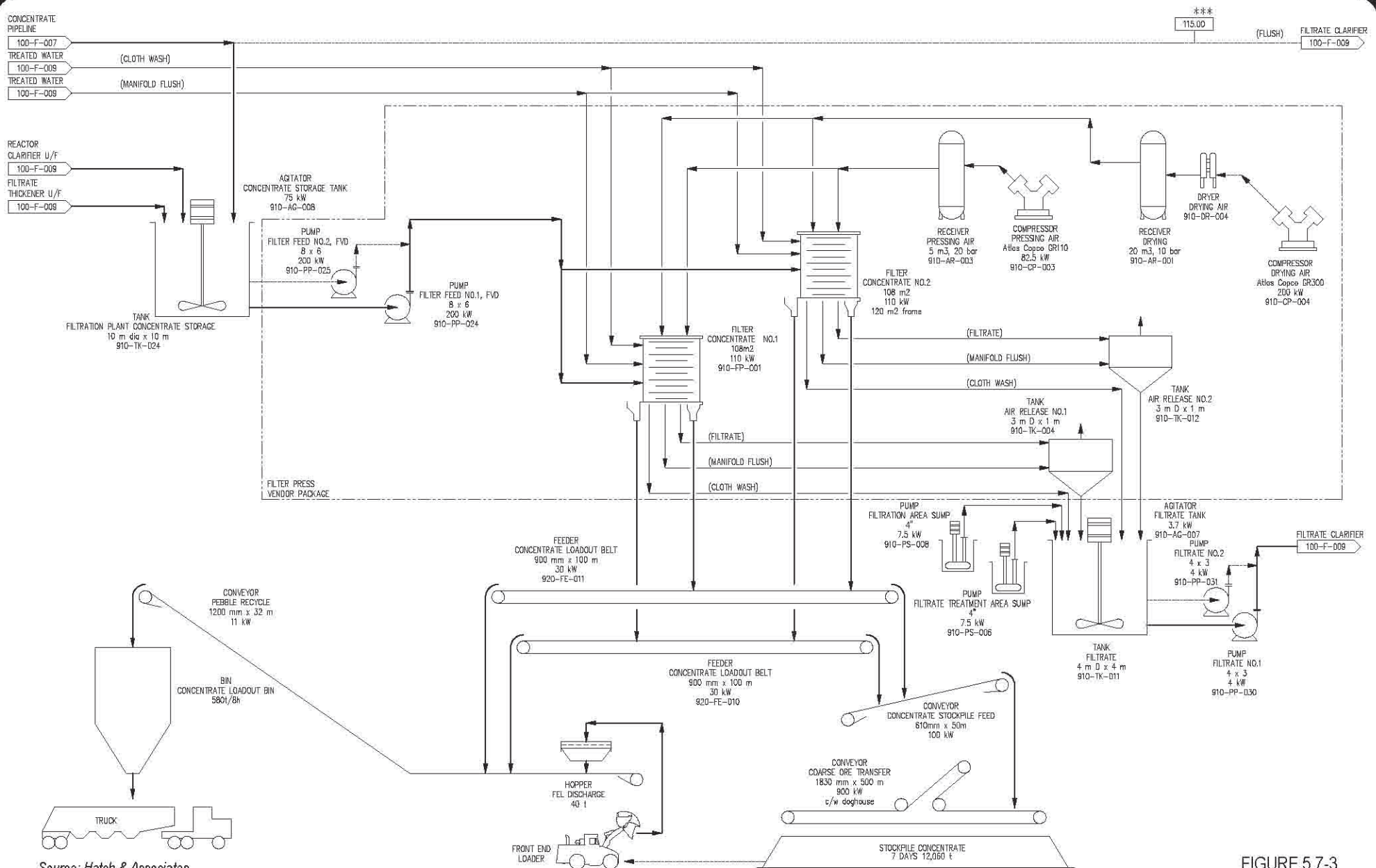
5.7.2.3 Water Treatment and Discharge

The filters used to produce the filter cake will remove most of the solids from the slurry water, but cloth washing and blowback (backwash) will produce a liquid with 10% suspended solids. This water will be contained in an agitated filtrate tank where lime and perhaps a surfactant will be added. The combined filtrate, manifold flush and cloth wash will be pumped from the filtrate tank to a filtrate clarifier thickener (Figure 5.7-5). Flocculant will be added at the thickener to settle the solids. The thickener underflow will be recycled to the concentrate stock tank and the clarified overflow with a suspended solids concentration of about 30 mg/L will be directed to a bank of sand filters that will reduce suspended solids to about 2 to 5 mg/L.

The final solids separation stage will be cartridge filters which will lower the suspended solids concentration to below 0.5 mg/L. This concentration represents approximately 0.15 mg/L copper. In addition to fine particulate copper, there will be 0.02 to 0.21 mg/L dissolved copper based on the pilot plant filtrate and other analyses. Total copper concentration will, therefore, be 0.17 to 0.36 mg/L.

The federal Metal Mine Effluent Regulations state that the maximum total copper concentration must not exceed 0.3 mg/L at the discharge point (end of pipe). The provincial objectives state that the dissolved copper concentration must not exceed 0.05 mg/L. Thus, additional treatment is necessary to depress the dissolved copper concentration from 0.21 mg/L to below 0.05 mg/L by raising pH.

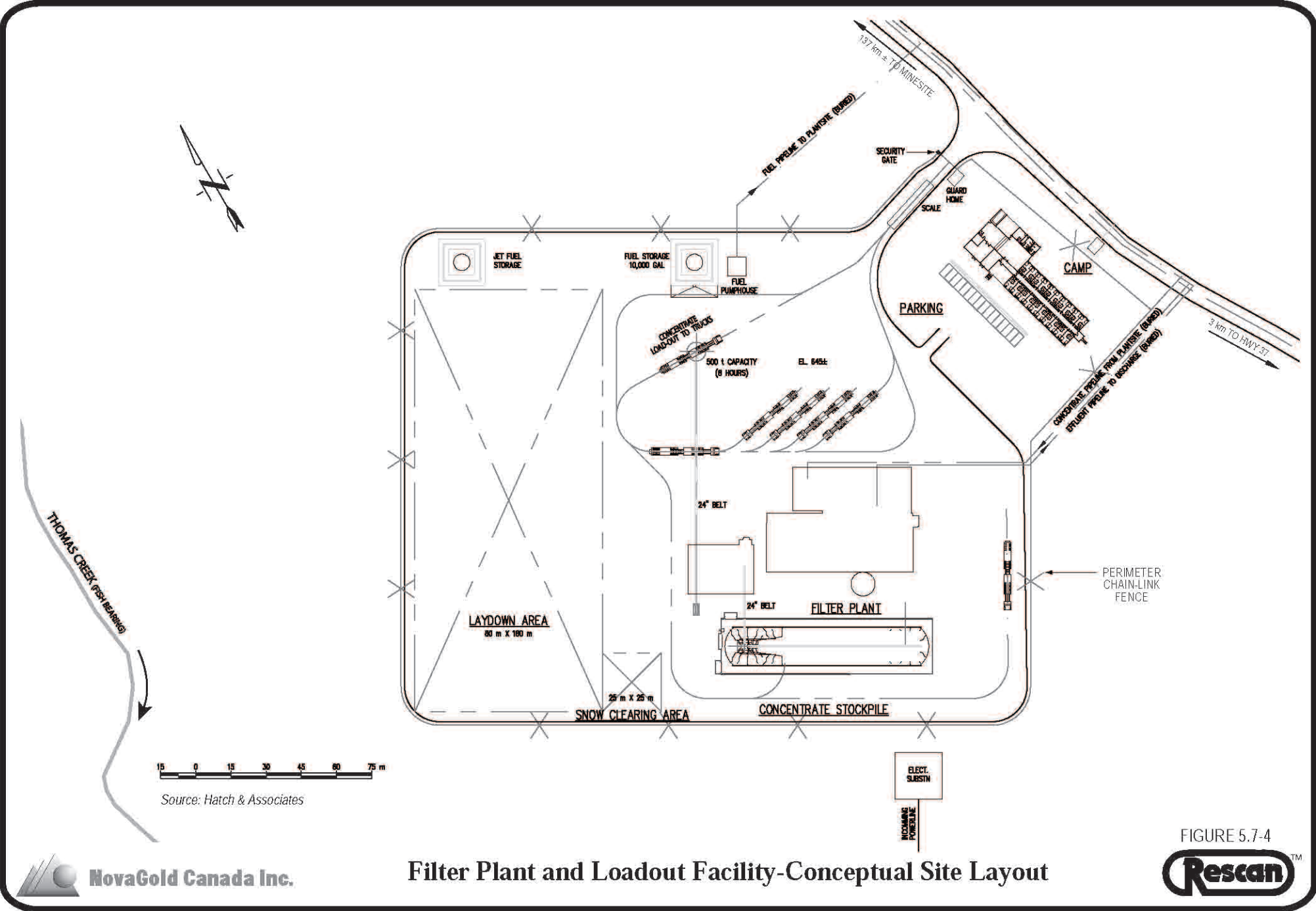
Laboratory tests were conducted to determine the effect of raising the pH and the quantity of lime required. For a design flow rate of 60 m³/h (16.7 L/s), it has been determined that approximately 113 kg/day of lime needs to be added to the agitated filtrate tank prior to the clarifier/thickener to raise pH from 7.5 to 10.8. If lime treatment is necessary a reaction time of 15 to 30 minutes will be required.



Source: Hatch & Associates

FIGURE 5.7-3

Concentrate Filtration Flowsheet

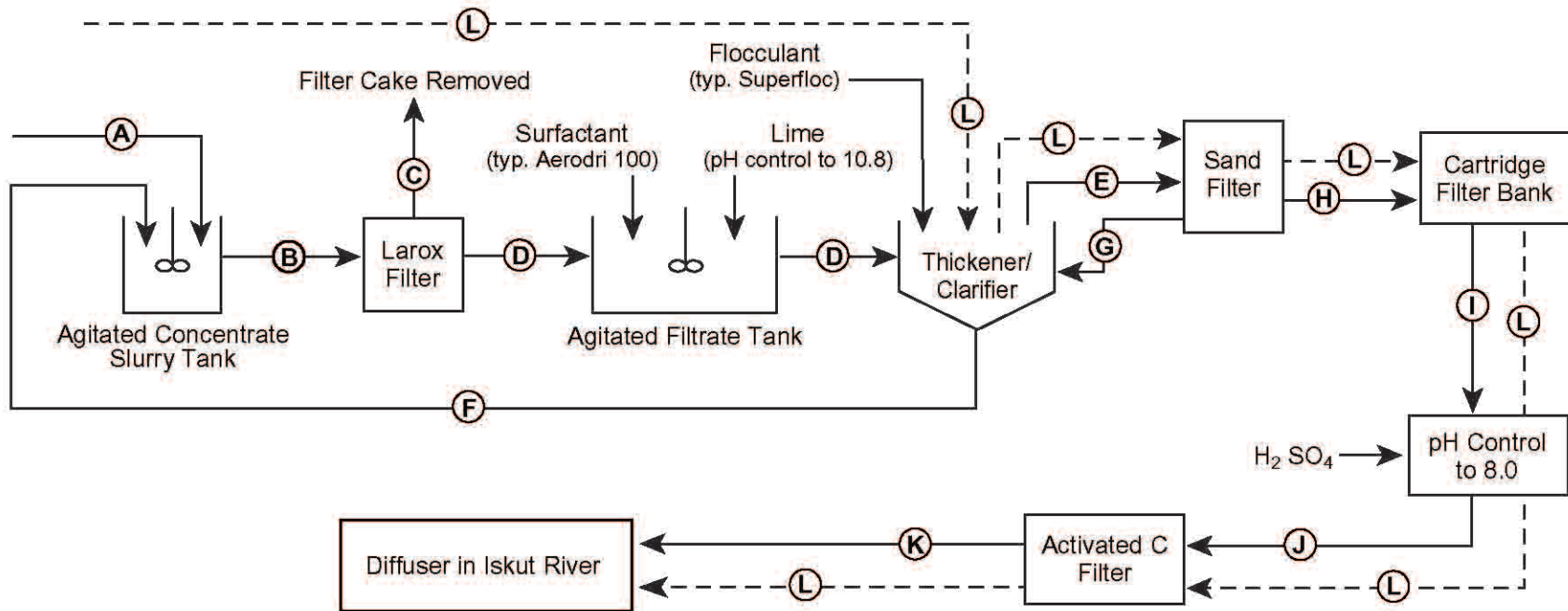


Source: Hatch & Associates

Filter Plant and Loadout Facility-Conceptual Site Layout

FIGURE 5.7-4





Stream	(A)	(B)	(C)	(D)	(E)	(F)	(G)	(H)	(I)	(J)	(K)	(L) Clear Water
Solids (tph)	87.69	94.89	87.69	7.20	0.00296	7.20	0.00296	0.000296	0	0	0	0
Liquid (m³/h)	68.97	74.63	9.75	64.88	59.24	5.66	0.03	59.22	59.22	59.22	59.22	89.85
Solids w/w	56%	56%	90%	10%	0.005%	56%	10%	0.0005%	0	0	0	0
Pulp tph	156.59	169.45	97.43	72.01	59.18	12.86	0.030	59.16	0	0	0	0
Pulp m³/h	89.85	97.22	30.63	66.59	59.24	7.38	0.027	59.22	0	0	0	0
Pulp SG	1.743	1.743	3.181	1.081	0.999	1.743	1.081	0.999	0.999			
Solids SG	4.2											

FIGURE 5.7-5

Filter Plant Water Treatment Flowsheet