APPENDIX 15
PREDICTION OF METAL LEACHING AND
ACID ROCK DRAINAGE, PHASE 1 (PART 1)

Rescan™

Morrison Copper/Gold Project: Metal Leaching and Acid Rock Drainage Characterization Report

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List of Acronyms and Abbreviations

3D	3-dimensional
ABA	Acid Base Accounting
ANFO	Ammonia nitrate fuel oil
AP	Acid potential
ARCB	Argillic Carbonate
ARSE	Argillic Sericite
BC	British Columbia
BFP	Biotite feldspar porphyry
CaNP	Carbonate-equivalent neutralization potential
(Ca) CaNP	Calculated Calcium Carbonate-based Neutralization Potential
CD	Condemnation drilling
CMRs	Carbonate molar ratios
DDH	Diamond drill hole
DYK	Dyke
EDCM	Empirical drainage chemistry model
FLT	Fault
HPGR	High Pressure Grinding Rolls
Inorg CaNP	Inorganic carbon carbonate-equivalent neutralization potential
KH	Potassic
KL	Kaolinitic
LCT	Locked cycle test
LTP	Leach test pad

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MDL	Method detection limit
ML/ARD	Metal leaching and acid rock drainage
not-PAG	Not potentially acid-generating
NP	Neutralization potential
NPR	Net Potential Ratio
PAG	Potentially acid-generating
the Project	Morrison Copper/Gold Project
QZSE	Quartz Sericite
Rescan	Rescan Environmental Services Ltd.
SAP	Sulphide sulphur acid potential
SED	Sedimentary
SFE tests	Shake Flask Extraction Tests
SGS	SGS Lakefield Research Limited
SNPR	Sulphide Net Potential Ratio
TAP	Total sulphur acid potential
TSF	Tailings storage facility
TDS	Total dissolved solids
Total CaNP	Total carbon carbonate-equivalent neutralization potential
TSS	Total suspended solids
WRD	Waste rock dump

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1.   Introduction

The accurate prediction and control of metal leaching and acid rock drainage (ML/ARD) at mine sites in British Columbia (BC) are high priorities as outlined by the provincial government in its formal Policy, Guidelines and draft Prediction Manual (Price 1997; Price and Errington 1998b, 1998a). This report follows the recommendations set out in those documents.

The Morrison Copper/Gold deposit, in north-central BC, has been classified as a calc-alkaline porphyry type. Mineralization of economic or environmental interest includes major occurrences of chalcopyrite ($CuFeS_2$) and pyrite (FeS_2) with minor bornite (Cu_5FeS_4), marcasite (FeS_2), pyrrhotite ($Fe_{(1-x)}S$), galena (PbS), molybdenite (MoS), and sphalerite (ZnS). Mining activities over the 21-year operation mine life will generate substantial volumes of waste rock (169 Mt), overburden (15 Mt), and low grade ore (51 Mt) from the open pit outline. Waste rock will be permanently stored in a waste rock dump (WRD), ore material will be stored in a low-grade ore stockpile for mill feed and overburden (OVB) will be stockpiled for construction and/or reclamation purposes. Annual ore mill throughput is estimated at 11.0 Mt and will generate an estimated 10.7 Mt of tailings.

Because of the ML/ARD potential of these mined materials, a phased approach was taken to the ML/ARD prediction and geochemical characterization of all materials that will be excavated, exposed, or disturbed by mining activities during construction, operations, and closure phases of the proposed Morrison Copper/Gold Project (the Project) mine plan.

Four phases of ML/ARD prediction have been completed to date for the Project and are summarized below:

- Phase 1 ML/ARD prediction program: developed to determine the geochemical characterization of initial tailings and deposit rock drill core samples (Appendix A).

- Phase 2 ML/ARD prediction program: developed to determine the geochemical characterization of more representative tailings samples and cycloned tailings sand to be used for tailings dam construction (Appendix C).

- Phase 3 ML/ARD prediction program: developed to determine and/or update existing geochemical characterization of ore, low-grade ore, waste rock, overburden, condemnation drilling (CD) samples, monitoring well samples, pit wall rock, and representative tailings.

- Phase 4 ML/ARD prediction program: currently in progress to determine aqueous mine drainage chemistry at near equilibrium conditions using field leaching barrels of PBM deposit drill core samples representing "fresh" waste rock material and Noranda deposit drill core samples representing "aged" waste rock materials.

For the first three phases, a total of 588 samples were submitted and tested for ML/ARD prediction and geochemical characterization. The geochemical data is typically used to classify mine materials depending on their potential (static tests), leachate quality (leachate extraction

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tests), reactivity and leachate quality evolution over time (laboratory and field kinetic tests). ML/ARD Prediction and Prevention and Waste management strategies are developed based on the geochemical behaviour and classification assigned to these various mine materials. This geochemical information also provides input for 3-dimensional geochemical block modeling, which is used as a Waste Management Planning tool at the feasibility study stage and subsequent phases of mine construction, operation and post closure. This geochemical information also provides input data for predictive water quality modelling, used to predict changes in drainage / water quality and assess the potential impacts to surface water and groundwater during construction, operation, closure and post closure mining phases. Note that cconservative assumptions were employed (i.e., assumptions that produce reasonable worst case water quality predictions).

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2.   Geology

The geologic descriptions following are primarily sourced from MacIntyre, Webster, and Villeneuve (1997).

2.1   Regional Geology

The Morrison copper-gold porphyry deposit is on the northern edge of the Skeena Arch in a region underlain by volcanic, clastic, and epiclastic rocks ranging in age from the Lower Jurassic to Lower Cretaceous, including the Takla Group, Hazelton Group, Bowser Lake Group, Skeena Group, and Sustut Group.   Intrusive rocks in the area include the Early Jurassic diorite and granodiorite Topley Intrusions, Eocene rhyolite and rhyodacite intrusions, and most importantly from an economic viewpoint, the Eocene Babine Igneous Suite which consists of quartz, hornblende, biotite, and plagioclase phyric intrusions.

The rock units are disrupted by a series of dominantly north to northwesterly trending faults into uplifted blocks, downfaulted grabens, and tilted fault blocks.   This has resulted in older lithologic units being juxtaposed and locally truncated against younger rock units (Figure 2.1-1).   Figure 2.1-2 provides a summary of the known regional stratigraphy of the Project area.

2.2   Structure

The most prominent regional structure is the north-northwesterly trending Morrison fault, which bisects the Morrison copper-gold porphyry deposit.   The Morrison deposit composes the central part of a major regional graben 1.5- to 2.5-km wide and is believed to be formed as a result of the movement of two subparallel dextral faults (Figure 2.2-1).   Geological evidence indicates that the development of this north-northwesterly structure is late-Eocene or younger, as rock units such as the Babine Intrusions have been truncated and offset by the bounding faults of the graben.   The fault that bounds the eastern margin of the graben is correlated with the Morrison fault and is approximately 800 m east of the property.   The western bounding fault is believed to be along Morrison Lake.   These faults have displaced the Morrison deposit with dextral en echelon-oriented offsets.   These dextral offsets have been accentuated, to some degree, by a family of subsidiary faults that are subparallel and related to the east and west Faults.   Ogryzlo et al. (1995) presented a geologic concept whereby the Morrison fault dextrally disrupted the Morrison/Hearne Hill copper-gold porphyry system into two displaced bodies, 2.0 km apart, with the Morrison deposit representing the downward extension of the higher level Hearne Hill deposit.

In the north, Lower to Upper Jurassic formations, including Telkwa, Nikitkawa, Saddle Hill, and Smithers (Hazelton Group), form the foundation of the proposed tailings storage facility (TSF) area.   Sedimentary lithologic units, which belong to continental shelf facies, consist of sandstone, siltstone, conglomerate, and limestone.

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Morrison Copper/Gold Project
Regional Geologic Setting

FIGURE 2.1-1

Age	Unit	Description
Eocene	EBc	Endako Gp., Buck Creek Fm. - vesicular basalt
	ENv	Newman Volcanics - lahar, debris flows, volc. breccia, hb-bt-fd porph. flows, sills
	EB	Babine intrusions - bt-fd porph. (EBp), rhyo-dacite (Ebr), qz-bt-fd porph (EBq)
	PEcg	conglomerate, sandstone
Cretaceous	uKK	Kasalka Gp. - hb-bt-fd porph. flows & breccia
	LKBp	Bulkley intrusions - bt-hb-qz porph. (LKBp)
	KTg	diorite, granodiorite, quartz diorite
	lKS	Skeena Gp. - sandstone, shale, siltstone
	lKv	trachytic basaltic andesite flows
U. Jur.	muJA	Ashman Fm. - fossiliferous argillaceous siltstone, wacke
M. Jur.	mJS	Smithers Fm. - fossiliferous feldspathic sandstone, siltstone, conglomerate
	lmJv	basaltic flows and related pyroclastic and volcaniclastic rocks; ash flow tuffs, lapilli tuffs and flow banded rhyolite (r).
L. Jur.	lJN	Nilkitkwa Fm. - siltstone, sandstone, conglomerate, minor limestone
	lJT	Telkwa Fm. - andesitic pyroclastics, andesite, basalt and rhyolite flows, felsic pyroclastics
U.Triassic	uTrcg	maroon to red conglom. with clasts of mega-crystic porphyry, px-fd porph., siltstone megacrystic feldspar porph. flows (uTrmp)
		banded siltstone, feldspathic sandstone
	LTrJbx	Nose Bay intrusive breccia - breccia with Topley and fd-px porphyry clasts
	LTrJT	Topley intrusions - pink granite, qz. monzonite, monzonite, aplite, rhyolite (EJT)
	uTrp	pyroxene-feldspar porphyry flows, breccia
M.Trias.	mTrs	calcareous siltstone, chert, tuff (M. Triassic radiolarian)
Perm.	Pc	massive, thick-bedded limestone (Middle Penn. Early Permian conodonts)

Labels in columns: Eocene — Ootsa Lk. Gp.; EBp, EBr; Ar-Ar 50.5-52.8 Ma; Ar-Ar 51.4 Ma. Cretaceous — K-Ar 77 Ma; LKBp; KTg. Hazelton Gp. — F; U-Pb 184.5 Ma; rhy; Pleinsbachian-Sinemurian. U.Triassic/M.Trias. — uTrmp; W. Takla Gp. (uTrs); LTrJT; U-Pb 217 Ma; Ar-Ar 210 Ma; LTrJbx; F.

Source: Macintyre et al., (1997)

Morrison Copper/Gold Project
Diagrammatic Stratigraphy of the Regional Geology

FIGURE 2.1-2

Source: MacIntyre et al., (1997) and references therein
Note: Refer to figure 2.1-2 for legend

**Morrison Copper/Gold Project
Regional Structural Geology
of the Morrison Study Area**

FIGURE 2.2-1

2.3   Local and Deposit Geology

The Morrison deposit is a zoned annular calc-alkaline copper-gold porphyry deposit. It is largely within a multi-phased Eocene "Babine Type" biotite feldspar porphyry (BFP) body that intruded the Middle to Upper Jurassic Ashman Formation (Bowser Lake Group). The lower part of this formation is made of marine pebble conglomerate interbedded with greenish grey sandstone and siltstone grading in the upper parts to argillaceous siltstone and greywacke. The subvolcanic intrusion, with nearly vertical contacts, occupies a northwest-oriented elliptical area of 900 m by 150 m to 300 m. Based on the regional geology map, the Niktiwa formation, which consists of shallow to deep fossiliferous marine feldspathic wackes as well as siltstone and conglomerate, is placed by the Ashman formation with a fault contact (Figure 2.3-1).

2.4   Mineralization and Alteration

Hydrothermal alteration at the Morrison copper-gold porphyry deposit is similar to that at other Babine porphyry copper deposits (e.g., Bell and Granisle mines). Alteration is concentrically zoned and characterized by central potassic-chloritic zoning with chlorite-carbonate zoning mostly dominant in the peripheral zones. A third alteration facies, clay (argillic)-carbonate alteration, is considered retrograde and associated with major faults and shears and subsidiary fracture zones. No well-developed phyllic zone has been identified.

The central core of the intrusive contains the bulk of the copper-gold mineralization, which corresponds with the potassically altered BFP plug with intercalations of older siltstone. The BFP at the Morrison deposit is typically a fine- to medium-grained, crowded biotite-hornblende-feldspar porphyry of quartz diorite composition. There are abundant 1.0 to 5.0 mm-size plagioclase phenocrysts that impart a distinctive speckled texture. This deposit was initially intruded into the siltstone unit as a near-vertical subcircular intrusion approximately 700 m in diameter. It was subsequently disrupted by the east and west faults and now forms an elongated body extending some 1,500 m in a northwestern direction.

Chalcopyrite ($CuFeS_2$) is the main copper-bearing sulphide mineral and is distributed as fine-grained disseminations in the BFP and in siltstone as fracture coatings or as stockworks of quartz veinlets in which the chalcopyrite occurs as coarse grains (1 to 3 mm) within veinlets that range from 1.0 mm to approximately 15 mm width. Other minor sulphide minerals such as bornite (Cu_5FeS_4), marcasite (FeS_2), pyrrhotite ($Fe_{(1-x)}S$), galena (PbS), and molybdenite (MoS) occur in spatially restricted amounts, particularly in the southeastern portion of the deposit. Arsenopyrite (FeAsS) and sphalerite (ZnS) are hosted in carbonate-rich rock or sulphide veins. Sulphide mineralization generally occurs as fracture filling and/or disseminated textures in the rock matrix (Appendix A; D. MacIntyre, Webster, and Desjardins 1997; D. G. MacIntyre, Webster, and Villeneuve 1997).

Diamond drilling, geological mapping, and detailed polished-section studies performed by Carson and Jambor (1976) indicate that pyrite and chalcopyrite have a well-defined zonal relationship. Although pyrite predominates in the pyrite halo, the 0.2% copper isopleth precisely marks a change in pyrite-to-chalcopyrite ratios; chalcopyrite consistently exceeds pyrite in

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Morrison Copper/Gold Project
Local Geologic Setting

FIGURE 2.3-1

samples only from the inside of this boundary. Although the absolute abundance of pyrite decreases toward the centre of the Morrison deposit, disseminated grains of pyrite persist throughout the copper zone and in the low-grade core.

The following is from Geosim (2007) based on studies by Carson and Jambor (1976). A pyrite halo is developed in the chlorite-carbonate altered wall rock surrounding the copper zone. The pyrite is mineralization characteristically occurs as thin (0.1- 5.0 cm) fracture-fillings and quartz-pyrite-minor chalcopyrite stringers in the form of stockwork within the halo. There is a crude zonation to the pyrite development with coarse (0.5 to 5.0 mm) disseminated crystals within the inner parts of the halo where pyrite content ranges from 5 to 15% by volume. Pyrite in the outer zone is predominantly developed as a stockwork and averages 1 to 2% by volume accompanied by weak copper mineralization (<0.1%). The pyrite halo is developed as a more extensive zone around the eastern and southeastern segment of the Morrison deposit. Drilling and geophysical surveys indicate that the halo at this position attains widths up to 500m with up to 14% pyrite for the inner margin and decreasing abruptly to 1 to 2% in the outer two thirds of the halo. The pyrite halo is more restricted at the western and northwestern segments of the deposit where pyrite abundances decrease more gradually to the 3 to 5% range. The siltstone host rock at this location is intruded by large northerly-trending BFP and rhyodacite dykes.

Polished thin-section studies have also shown that, in addition to chalcopyrite and pyrite, magnetite and minor bornite are present in the low-grade core of the deposit. Magnetite is a finely disseminated original constituent of the BFP and siltstones, and is most abundant in the western segment of the copper zone. Many magnetite grains are partly altered to hematite, which seems to be most abundant at the outer 0.2% Cu boundary. No iron oxides have been observed in the pyrite halo.

Carbonate-rich rock units and alteration are common within the Morrison deposit, with calcite ($CaCO_3$), siderite ($FeCO_3$) and ankerite ($Ca(Fe,Mg,Mn)(CO_3)_2$) present. Neutralizing aluminosilicate minerals such as plagioclase (An_{25-35}), biotite ($K(Mg,Fe)_3AlSi_3O_{10}(OH,F)_2$), hornblende ($Ca_2[Fe_4(Al,Fe)]Si_7AlO_{22}(OH)_2$) or ($Ca_2[Mg_4(Al,Fe)]Si_7AlO_{22}(OH)_2$), epidote ($Ca_2(Fe,Al)_3(SiO_4)_3(OH)$), and chlorite ($Ca_{0.5}(Al,Mg)_6(Si,Al)_8O_{18}(OH)_{12}\cdot5(H_2O)$), are abundant within host rocks of the Morrison deposit (Appendix A).

Logged pyrite content from the Morrison master diamond drillhole database indicates that approximately 70 logged occurrences have pyrite greater than 5% (Figure 2.4-1 and Figure 2.4.2). In general, rock with pyrite greater than 2% will be milled where a pyrite flotation circuit will operate. More than 99% of waste rock contains 1-2% pyrite according the available data.

Figure 2.4-3 shows the extent of the documented pyrite halo and the 0.3% and 0.1% copper isopleths with respect to the current Morrison pit limits. This shows the 5-15% pyrite halo will intersect the pit walls (at closure) and potentially in waste if grades are confirmed to be low enough in the southeast according to the available data to date.

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~65 of 8367 (0.8%)
samples have Logged Pyrite > 5%

Logged Pyrite (%)

Sample Count

Morrison Copper/Gold Project
Logged Pyrite in Master Diamond Drillhole Database

FIGURE 2.4-1

**Morrison Copper/Gold Project: Logged Pyrite
>5% in Master Diamond Drillhole Database**

FIGURE 2.4-2

GEOLOGY OF MORRISON

L E G E N D

QUATERNARY
GLACIAL DRIFT, ALLUVIUM

EOCENE (BABINE INTRUSIONS)
BIOTITE PLAGIOCLASE HORNBLENDE PORPHYRY (BFP)
(Only Major Bodies Shown)

EOCENE ?
RHYODACITE DYKES

JURASSIC (HAZELTON GROUP)
SILTSTONE, ARGILLITE, MINOR CONGLOMERATE

0.3% COPPER CONTOUR

PYRITE HALO (PYRITE = 5% - 15%)

BEDDING

DOMINANT ATTITUDE OF MINERALIZED FRACTURES

FAULT

0.1 ISOPLETH

CURRENT PIT OUTLINE

Source: Carson and Jambor (1976); Ogryzlo et al. (1995) and Wardrop (2009)

Note: Map above represents a two dimensional plan view and may not accurately reflect the distribution of the pyrite halo at depth.

FIGURE 2.4-3

Pyrite Halo with Current Pit Shell

3. Mine Plan

The following mine plan information is referenced from Klohn Crippen Berger (2008) and Wardrop (2009). Figure 3-1 shows the mine site layout of the main mine components at maximum extent. An on-land waste rock dump to the immediate east of the open pit will store all potentially acid-generating (PAG) waste rock generated during the Project's mine life. Overburden will be stripped before the Phase I starter open pit and during the subsequent Phase II, III, and IV open pit expansions and will be stockpiled on the peninsula to the west of the open pit. Several sediment and organic stockpiles will be stockpiled to the north and south of the open pit.

3.1 Materials Excavation

The Project is a proposed conventional open pit development at an approximate rate of 30,000 t/d (11 Mt/yr) ore throughput. Stripping ratios range between 0.44, in the early years of the mine life, and 1.99 in the latter years with an overall average ratio of 0.82. The total mine material destined for the WRD, overburden stockpile and low-grade ore stockpile are approximately 169 Mt, 15 Mt, and 51 Mt, respectively. Based on optimization of the mine plan, the low-grade ore stockpile has been designed for a maximum tonnage of 36 Mt during the Project's operations phase.

Table 3.1-1 summarizes the mining schedule and mine materials destination to stockpiles and dumps over the mine life. The mining schedule indicates that the quantity of not potentially acid-generating (not-PAG) material available early in the mine life for construction purposes will be limited. Figure 3.1-1 show the WRD, overburden stockpile, and low-grade ore stockpile balances throughout the mine life. Note that the balance of low-grade ore materials in the low-grade ore stockpile will be processed during the last two years of operations. At closure, the WRD (150-ha total footprint) is scheduled to be reclaimed with 0.3 m organic soil cover underlain by a 1-m low permeability glacial till layer. Although the overburden stockpile will also be reclaimed with a cover, conceptual engineering design is unavailable at this time and will be completed at a later date.

3.2 Milling, Tailings deposition and Tailings Storage Facility

Annual mill throughput of 11 Mt ore will be processed through a conventional milling circuit consisting of a primary crusher, secondary cone crusher followed by high pressure grinding rolls (HPGR), primary ball mills, and flotation circuit including regrinding tower mills. Copper will be concentrated by flotation in large tank cells then cleaned and filtered to achieve acceptable shipping moistures without thermal drying. A molybdenum concentrate will be produced from the deposit ore zones containing significant quantity and grades of molybdenum.

Tailings will be produced at rate of approximately 10.95 Mt/a. Tailings solids and process waters (slurry) will be pumped to the TSF via a 760-mm diameter tailings slurry pipeline where the tailings slurry stream will be cycloned between March and October of each year to produce coarse sand to construction the main, north and west TSF dams. Although the TSF will operate as a "zero" discharge facility, seepage recovery ponds will be constructed downstream of the

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main and north dams. Total seepage losses are estimated by Rescan to be 112 m^3/hr in the early stages of mine operations (Phase I, Year 7) up to 208 m^3/hr at maximum pond volume and into post-closure (see Rescan 2009b for further details). During operations, a portion of the tailings will be submerged by the tailings pond, determined by the annual TSF water balance. At closure, the TSF pond water level will be allowed to rise to an elevation of 1,013 m, the ultimate TSF spillway elevation.

3.3   Waste Rock

Waste rock will be excavated from the pit and stored in the WRD, where it will be exposed throughout the life mine, with progressively greater surface area from the start of Phase I open pit pre-stripping during the construction phase to post-closure. For determining relative geochemical reactivity of waste rock and water quality modelling (Rescan 2009b), see Section 3.4 below.

3.4   Pit Wall Rock

Pit wall rock will be exposed throughout the mine life. For determining relative reactivity of wall rock, and for water quality modelling purposes (Rescan 2009b), a similar approach was used as per waste rock outlined above in Section 3.3. The pit wall rock was divided into four adjusted SNPR intervals according to Table 3.4-1. The SNPR values for the pit wall were derived using the total sulphur (assumed to equal sulphide) and adjusted NP values from pit rock ABA data (Figure 3.4-1). SNPR values were then calculated for each block (20 m x 20 m x 12 m) according to:

$$Adjusted\ SNPR = (NP\text{-}13)/(Total\ Sulphur*31.25) \hspace{2cm} Eqn\ 3.4\text{-}1$$

Thirteen was previously determined to represent the unavailable NP for the Morrison rock (Appendix A). The sulphur estimation used an anisotropic search ellipsoid the same as that used for copper grade estimation (Wardrop 2009). The NP interpolation used an isotropic search. The maximum search distance was initially 200 m but expanded later to 250 m after the pit limits were increased in 2007.

Although, in most areas of the pit walls the confidence of the estimation is low as there are few nearby data points to rely on, the majority of the entire pit wall rock (~56%) is classified as having SNPRs below 1.5. This proportion increases to 80% for the exposed pit wall on closure after the final pit lake elevation has been established at 728 masl (4 m of freeboard).

3.5   Lake Sediments

Lakes near the proposed open pit footprint (i.e., Booker Lake and Ore Pond) are scheduled for draining before sediment dredging and disposal (Klohn Crippen Berger 2008). Preliminary estimates of sediment volumes from Booker Lake and Ore Pond for disposal purposes are 600,000 m^3 and 50,000 m^3, respectively (Klohn Crippen Berger 2008), although detailed investigations into actual sediment depth have not been completed to date. An assumed dry bulk density of 1.4 t/m^3 yields an estimated 840,000 tonnes and 70,000 tonnes of Booker Lake and Ore Pond sediments, respectively, that will be placed in sediment and organic stockpiles at the Project.

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**Main Mine Site Infrastructure
at Morrison Copper/Gold Project**

FIGURE 3-1

Legend:

- Gravel Road
- Access Road
- 100m Contour
- 20m Contour
- Water Diversion
- Water Pipeline
- Tailings Pipeline
- Preferred Transmission Line Alignment
- Alternative Transmission Line Alignment
- Lake
- Wetland
- Stockpile and Borrow Area
- Access Road/ Haul Road
- Other Infrastructure
- Dam
- Waste Rock Dump
- Pond
- Sanitary Landfill
- Tailings Impoundment at 1015 m
- Ultimate Pit Extent

1:50,000

0   1,500   3,000

Metres

Projection: UTM9, NAD83

Morrison Lake

Hearne Hill

Babine Lake

Table 3.1-1
Morrison Copper/Gold Project: Mine Schedule

Materials	Units	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
Phase I													
Mill Feed	t	2,299,000	14,366,000	21,340,000	19,104,000	8,943,000	10,328,000	7,533,000	3,188,000	0	0	0	0
Waste Rock	t	7,701,000	14,834,000	7,860,000	4,621,000	1,793,000	1,301,000	75,000	0	0	0	0	0
Phase II													
Potential Mill Feed	t	0	0	0	0	2,671,000	5,143,000	6,816,000	10,378,000	9,244,000	6,962,000	9,496,000	9,195,000
Waste Rock	t	0	0	0	0	10,318,000	6,953,000	7,476,000	8,334,000	3,732,000	1,111,000	393,000	0
Phase III													
Potential Mill Feed	t	0	0	0	0	0	0	0	0	673,000	1,910,000	1,832,000	3,856,000
Waste Rock	t	0	0	0	0	0	0	0	0	8,251,000	11,918,000	10,179,000	8,850,000
Phase IV													
Potential Mill Feed	t	0	0	0	0	0	0	0	0	0	0	0	0
Waste Rock	t	0	0	0	0	0	0	0	0	0	0	0	0
Total Potential Mill Feed	t	2,299,000	15,803,000	21,340,000	19,104,000	13,371,000	15,471,000	15,022,000	14,747,000	12,388,000	12,850,000	12,547,000	13,051,000
Total Waste Rock	t	7,701,000	14,834,000	7,860,000	4,621,000	12,111,000	8,254,000	7,551,000	8,334,000	11,983,000	13,029,000	10,572,000	8,850,000
Total	t	10,000,000	30,637,000	29,200,000	23,725,000	25,482,000	23,725,000	22,573,000	23,081,000	24,371,000	25,879,000	23,119,000	21,901,000
Cumulative Potential Mill Feed	t	2,299,000	18,102,000	39,442,000	58,546,000	71,917,000	87,388,000	102,410,000	117,157,000	129,545,000	142,395,000	154,942,000	167,993,000
Cumulative Waste Rock	t	7,701,000	22,535,000	30,395,000	35,016,000	47,127,000	55,381,000	62,932,000	71,266,000	83,249,000	96,278,000	106,850,000	115,700,000
Cumulative Total	t	10,000,000	40,637,000	69,837,000	93,562,000	119,044,000	142,769,000	165,342,000	188,423,000	212,794,000	238,673,000	261,792,000	283,693,000
Stockpile Recovery	t	0	1,437,000	0	0	1,757,000	0	673,000	1,181,000	2,471,000	3,978,000	1,219,000	0
Cumulative Stockpile Recovery	t	0	1,437,000	1,437,000	1,437,000	3,194,000	3,194,000	3,867,000	5,048,000	7,519,000	11,497,000	12,716,000	12,716,000
Stockpile Balance	t	2,299,000	6,810,000	17,200,000	25,354,000	26,018,000	30,539,000	33,938,000	36,554,000	35,521,000	33,443,000	33,821,000	35,922,000
Potential Total Stockpile (Recovery+Balance)	t		8,247,000	17,200,000	25,354,000	27,775,000	30,539,000	34,611,000	37,735,000	37,992,000	37,421,000	35,040,000	35,922,000
Actual To Mill	t		9,855,000	10,950,000	10,950,000	10,950,000	10,950,000	10,950,000	10,950,000	10,950,000	10,950,000	10,950,000	10,950,000
Actual to LGO		2,299,000	5,948,000	10,390,000	8,154,000	2,421,000	4,521,000	4,072,000	3,797,000	1,438,000	1,900,000	1,597,000	2,101,000
Mill Feed	t	2,299,000	15,803,000	21,340,000	19,104,000	13,371,000	15,471,000	15,022,000	14,747,000	12,388,000	12,850,000	12,547,000	13,051,000
Overburden	t	2,551,000	5,468,000	343,000	0	973,000	329,000	482,000	36,000	1,699,000	2,120,000	66,000	0
Unknown	t	8,000	0	0	0	0	0	0	0	3,000	0	0	0
Potentially Acid Generating Waste	t	4,936,000	8,932,000	7,186,000	4,182,000	11,029,000	7,858,000	6,395,000	6,858,000	9,254,000	8,839,000	9,172,000	7,655,000
Not -Potentially Acid Generating Waste	t	206,000	433,000	330,000	439,000	110,000	67,000	674,000	1,440,000	1,027,000	2,069,000	1,334,000	1,195,000
Total Waste	t	7,701,000	14,833,000	7,859,000	4,621,000	12,112,000	8,254,000	7,551,000	8,334,000	11,983,000	13,028,000	10,572,000	8,850,000

Table 3.1-1
Morrison Copper/Gold Project: Mine Schedule (completed)

Materials	Units	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Total
Phase I												
Mill Feed	t	0	0	0	0	0	0	0	0	0	0	87,101,000
Waste Rock	t	0	0	0	0	0	0	0	0	0	0	38,185,000
Phase II												
Potential Mill Feed	t	0	0	0	0	0	0	0	0	0	0	59,905,000
Waste Rock	t	0	0	0	0	0	0	0	0	0	0	38,317,000
Phase III												
Potential Mill Feed	t	7,969,000	9,709,000	8,030,000	7,839,000	5,120,000	1,867,000	0	0	0	0	48,805,000
Waste Rock	t	5,981,000	3,647,000	1,457,000	394,000	229,000	128,000	0	0	0	0	51,034,000
Phase IV												
Potential Mill Feed	t	0	0	34,000	811,000	5,420,000	9,625,000	10,884,000	1,607,000	0	0	28,381,000
Waste Rock	t	7,950,000	8,544,000	12,379,000	12,856,000	11,071,000	2,880,000	616,000	289,000	0	0	56,585,000
Total Potential Mill Feed	t	12,562,000	10,951,000	11,335,000	11,075,000	10,966,000	11,492,000	11,109,097	10,950,000	10,950,000	6,345,000	275,728,097
Total Waste Rock	t	13,931,000	12,191,000	13,836,000	13,250,000	11,300,000	3,008,000	616,000	289,000	0	0	184,121,000
Total	t	26,493,000	23,142,000	25,171,000	24,325,000	22,266,000	14,500,000	11,725,097	11,239,000	10,950,000	6,345,000	459,849,097
Cumulative Potential Mill Feed	t	180,555,000	191,506,000	202,841,000	213,916,000	224,882,000	236,374,000	247,483,097	258,433,097	269,383,097	275,728,097	
Cumulative Waste Rock	t	129,631,000	141,822,000	155,658,000	168,908,000	180,208,000	183,216,000	183,832,000	184,121,000	184,121,000	184,121,000	
Cumulative Total	t	310,186,000	333,328,000	358,499,000	382,824,000	405,090,000	419,590,000	431,315,097	442,554,097	453,504,097	459,849,097	
Stockpile Recovery	t	4,593,000	1,242,000	3,271,000	2,425,000	426,000	0	225,097	9,343,000	10,950,000	6,345,000	51,536,097
Cumulative Stockpile Recovery	t	17,309,000	18,551,000	21,822,000	24,247,000	24,673,000	24,673,000	24,898,097	34,241,097	45,191,097	51,536,097	
Stockpile Balance	t	32,941,000	31,700,000	28,814,000	26,514,000	26,164,000	26,706,000	26,640,000	17,297,000	6,347,000		
Potential Total Stockpile (Recovery+Balance)	t	37,534,000	32,942,000	32,085,000	28,939,000	26,590,000	26,706,000	26,865,097	26,640,000	17,297,000	6,345,000	
Actual To Mill	t	10,950,000	10,950,000	10,950,000	10,950,000	10,890,000	10,950,000	10,950,000	10,950,000	10,950,000	6,347,000	224,192,000
Actual to LGO		1,612,000	1,000	385,000	125,000	76,000	542,000	159,097	0	0	-2,000	51,536,097
Mill Feed	t	12,562,000	10,951,000	11,335,000	11,075,000	10,966,000	11,492,000	11,109,097	10,950,000	10,950,000	6,345,000	275,728,097
Overburden	t	720,000	164,000	192,000	0	0	0	0	0	0	0	15,143,000
Unknown	t	0	0	0	0	0	43,000	0	0	0	0	54,000
Potentially Acid Generating Waste	t	12,662,000	11,564,000	12,922,000	10,907,000	8,751,000	2,427,000	119,000	31,000	0	0	151,679,000
Not -Potentially Acid Generating Waste	t	549,000	462,000	723,000	2,343,000	2,549,000	537,000	497,000	258,000	0	0	17,242,000
Total Waste	t	13,931,000	12,190,000	13,837,000	13,250,000	11,300,000	3,007,000	616,000	289,000	0	0	184,118,000

Morrison Copper/Gold Project
Low Grade Ore, Waste Rock Dump and Overburden
Stockpile Balances throughout the Mine Life

FIGURE 3.1-1
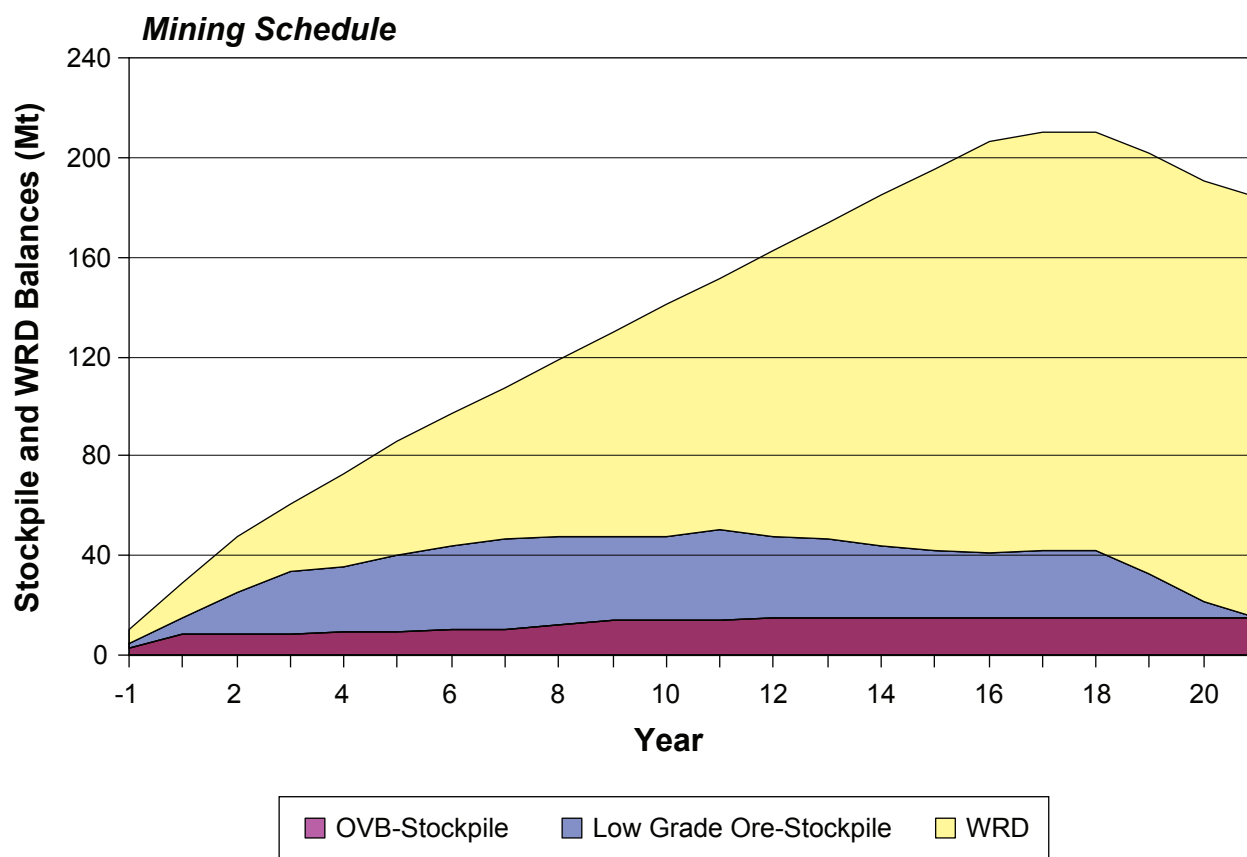
Table 3.3-1

Morrison Copper/Gold Project: Mill Feed and Mine Waste Production Schedule Showing Weighted Waste Rock Adjusted SNPR Tonnages

End of Year	Potential Mill Feed (tonnes)	Waste + Overburden (tonnes)	Total (tonnes)	Overburden (tonnes)	Waste Adj-SNPR 0 to 0.5 (tonnes)	Waste Adj-SNPR 0.5 to 1.5 (tonnes)	Waste Adj-SNPR 1.5 to 2.5 (tonnes)	Waste Adj-SNPR > 2.5 (tonnes)	Waste Subtotal (tonnes)
Year 1	2,299,000	7,701,000	10,000,000	2,551,000	69,000	365,000	2,786,000	1,930,000	5,150,000
Year 1	14,366,000	14,834,000	29,200,000	5,469,000	146,000	1,014,000	4,599,000	3,606,000	9,365,000
Year 2	21,340,000	7,860,000	29,200,000	344,000	130,000	650,000	2,997,000	3,739,000	7,516,000
Year 3	19,104,000	4,621,000	23,725,000	0	377,000	210,000	796,000	3,238,000	4,621,000
Year 4	11,614,000	12,111,000	23,725,000	973,000	73,000	290,000	7,721,000	3,054,000	11,138,000
Year 5	15,471,000	8,254,000	23,725,000	329,000	18,000	419,000	5,967,000	1,521,000	7,925,000
Year 6	14,349,000	7,551,000	21,900,000	481,000	154,000	1,536,000	2,822,000	2,558,000	7,070,000
Year 7	13,566,000	8,334,000	21,900,000	36,000	552,000	1,648,000	1,724,000	4,374,000	8,298,000
Year 8	9,917,000	11,983,000	21,900,000	1,700,000	595,000	1,336,000	2,228,000	6,124,000	10,283,000
Year 9	8,872,000	13,029,000	21,900,000	2,122,000	1,525,000	1,596,000	4,560,000	3,226,000	10,907,000
Year 10	11,328,000	10,572,000	21,900,000	66,000	852,000	1,876,000	5,914,000	1,864,000	10,506,000
Year 11	13,051,000	8,850,000	21,900,000	0	817,000	2,164,000	5,108,000	761,000	8,850,000
Year 12	7,969,000	13,931,000	21,900,000	720,000	429,000	1,591,000	10,372,000	819,000	13,211,000
Year 13	9,709,000	12,191,000	21,900,000	164,000	291,000	1,267,000	9,291,000	1,178,000	12,027,000
Year 14	8,064,000	13,836,000	21,900,000	191,000	442,000	1,672,000	8,893,000	2,638,000	13,645,000
Year 15	8,650,000	13,250,000	21,900,000	0	1,866,000	2,206,000	6,708,000	2,470,000	13,250,000
Year 16	10,600,000	11,300,000	21,900,000	0	2,314,000	1,330,000	5,586,000	2,071,000	11,301,000
Year 17	11,492,000	3,008,000	14,500,000	0	366,000	508,000	1,690,000	444,000	3,008,000
Year 18	10,884,000	616,000	11,500,000	0	484,000	66,000	45,000	21,000	616,000
Year 19	1,607,000	289,000	1,896,000	0	189,000	92,000	8,000	0	289,000
Year 20	0	0	0	0	0	0	0	0	0
Year 21	0	0	0	0	0	0	0	0	0
Total	224,252,000	184,121,000	408,371,000	15,146,000	11,689,000	21,836,000	89,815,000	45,636,000	168,976,000

Table 3.4-1
Morrison Copper/Gold Project: Weighted Adjusted SNPR values for Pit Wall Rock by bench Elevation

Elevation	Units	Adjusted SNPR (n)					Areas by Adjusted SNPR (m²)					Cumulative Areas by Adjusted SNPR (m²)				
(masl)		0 to 0.5	0.5 to 1.5	1.5 to 2.5	2.5+	Total	0 to 0.5	0.5 to 1.5	1.5 to 2.5	2.5+	Total	0 to 0.5	0.5 to 1.5	1.5 to 2.5	2.5+	Total
486		0	18	42	36	96	0	4,320	10,080	8,640	23,040	0	4,320	10,080	8,640	23,040
498		0	28	53	39	120	0	6,720	12,720	9,360	28,800	0	11,040	22,800	18,000	51,840
510		1	61	44	71	177	240	14,640	10,560	17,040	42,480	240	25,680	33,360	35,040	94,320
522		0	67	27	50	144	0	16,080	6,480	12,000	34,560	240	41,760	39,840	47,040	128,880
534		0	96	30	67	193	0	23,040	7,200	16,080	46,320	240	64,800	47,040	63,120	175,200
546		0	78	30	62	170	0	18,720	7,200	14,880	40,800	240	83,520	54,240	78,000	216,000
558		3	94	51	70	218	720	22,560	12,240	16,800	52,320	960	106,080	66,480	94,800	268,320
570		7	95	38	42	182	1,680	22,800	9,120	10,080	43,680	2,640	128,880	75,600	104,880	312,000
582		16	123	52	49	240	3,840	29,520	12,480	11,760	57,600	6,480	158,400	88,080	116,640	369,600
594		15	106	44	34	199	3,600	25,440	10,560	8,160	47,760	10,080	183,840	98,640	124,800	417,360
606		24	130	65	56	275	5,760	31,200	15,600	13,440	66,000	15,840	215,040	114,240	138,240	483,360
618		18	106	62	28	214	4,320	25,440	14,880	6,720	51,360	20,160	240,480	129,120	144,960	534,720
630		23	153	71	44	291	5,520	36,720	17,040	10,560	69,840	25,680	277,200	146,160	155,520	604,560
642		23	129	44	39	235	5,520	30,960	10,560	9,360	56,400	31,200	308,160	156,720	164,880	660,960
654		32	170	50	68	320	7,680	40,800	12,000	16,320	76,800	38,880	348,960	168,720	181,200	737,760
666		28	120	61	56	265	6,720	28,800	14,640	13,440	63,600	45,600	377,760	183,360	194,640	801,360
678		35	171	72	49	327	8,400	41,040	17,280	11,760	78,480	54,000	418,800	200,640	206,400	879,840
690		30	135	65	34	264	7,200	32,400	15,600	8,160	63,360	61,200	451,200	216,240	214,560	943,200
702		47	192	73	46	358	11,280	46,080	17,520	11,040	85,920	72,480	497,280	233,760	225,600	1,029,120
714		46	153	54	30	283	11,040	36,720	12,960	7,200	67,920	83,520	534,000	246,720	232,800	1,097,040
726		72	202	65	34	373	17,280	48,480	15,600	8,160	89,520	100,800	582,480	262,320	240,960	1,186,560
738		60	156	42	26	284	14,400	37,440	10,080	6,240	68,160	115,200	619,920	272,400	247,200	1,254,720
750		105	209	51	34	399	25,200	50,160	12,240	8,160	95,760	140,400	670,080	284,640	255,360	1,350,480
762		78	154	35	29	296	18,720	36,960	8,400	6,960	71,040	159,120	707,040	293,040	262,320	1,421,520
774		93	186	51	45	375	22,320	44,640	12,240	10,800	90,000	181,440	751,680	305,280	273,120	1,511,520
786		86	112	27	56	281	20,640	26,880	6,480	13,440	67,440	202,080	778,560	311,760	286,560	1,578,960
798		119	121	16	71	327	28,560	29,040	3,840	17,040	78,480	230,640	807,600	315,600	303,600	1,657,440
810		92	114	8	29	243	22,080	27,360	1,920	6,960	58,320	252,720	834,960	317,520	310,560	1,715,760
822		104	80	6	10	200	24,960	19,200	1,440	2,400	48,000	277,680	854,160	318,960	312,960	1,763,760
834		66	74	3	1	144	15,840	17,760	720	240	34,560	293,520	871,920	319,680	313,200	1,798,320
846		49	37	1	0	87	11,760	8,880	240	0	20,880	305,280	880,800	319,920	313,200	1,819,200
858		16	7	0	0	23	3,840	1,680	0	0	5,520	309,120	882,480	319,920	313,200	1,824,720
870		0	1	0	0	1	0	240	0	0	240	309,120	882,720	319,920	313,200	1,824,960
Blocks with Attribute	(n)	1,288	3,678	1,333	1,305	7,604										
Pit Wall Block Area	(m²)						309,120	882,720	319,920	313,200	1,824,960					
Pit Wall Block Percentage	(%)						17	48	18	17	100					
Area Below Morrison Lake Level	(m²)											108,000	601,200	267,360	244,080	1,220,640
Percentage Below Morrison Lake Level	(%)											9	49	22	20	100
Area Above Morrison Lake Level	(m²)											201,120	281,520	52,560	69,120	604,320
Percentage Above Morrison Lake Level	(%)											33	47	9	11	100

Notes:
Morrison Lake Level ~ 732 masl.

FIGURE 3.4-1

Morrison Copper/Gold Project: Predicted Adjusted SNPR Categories at the Pit Shell
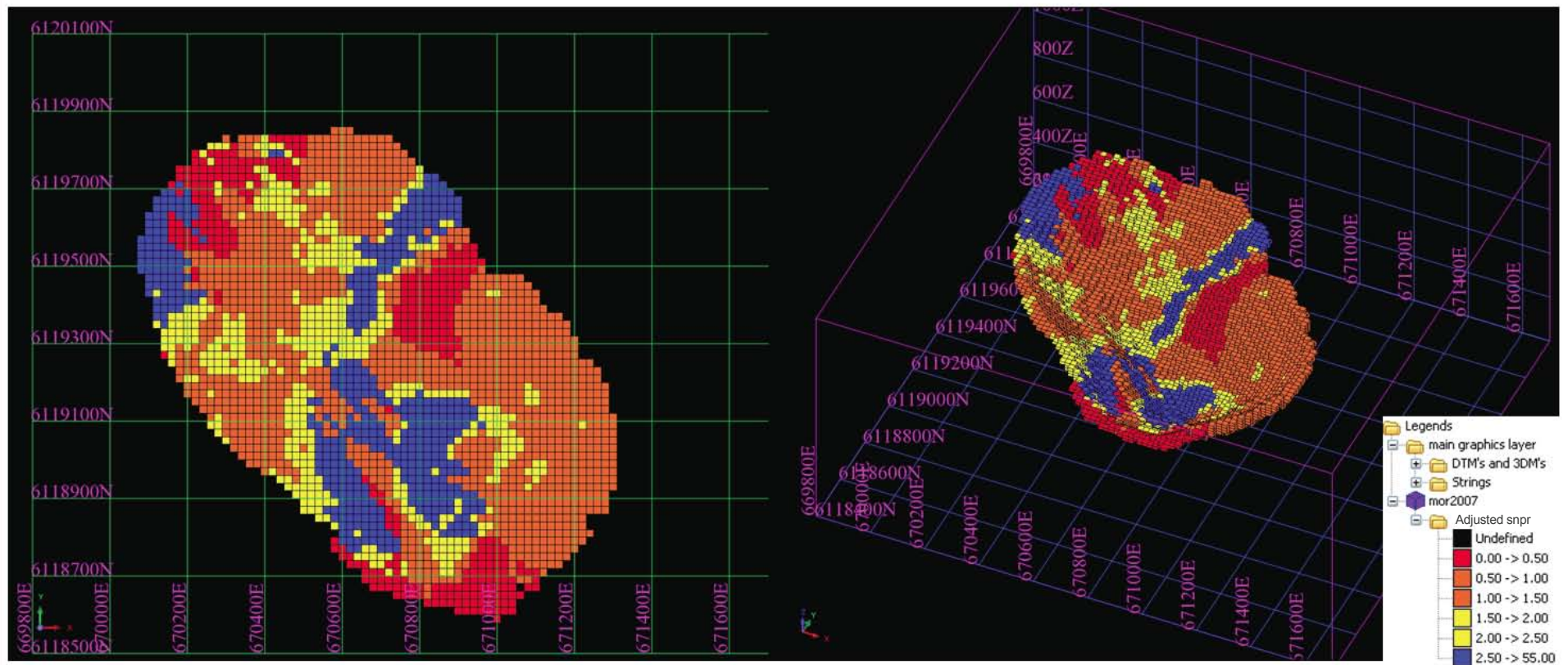
4.   Geochemical Characterization Methodologies

Previous ML/ARD prediction and geochemical characterization sample descriptions and analytical methodologies for static and kinetic testing of representative pit rock types and tailings are discussed in detail in Morin and Hutt (Appendix A).   The following sections describe additional static, leachate extraction, and kinetic testing completed to date.   The reader is encouraged to read Morin and Hutt (Appendix A) before reading the following sections as this phase of ML/ARD prediction and geochemical characterization is a fundamental component of the overall program that is not repeated here.

4.1   Sample Description

4.1.1   Overburden

Approximately 15 Mt of overburden will be excavated from within the pit outline throughout the mine life (Wardrop 2009).   Overburden thickness in the Project area ranges from <1 m to over 50 m and is typically between 2-m and 10-m deep in the proposed pit area (Klohn Crippen Berger 2008).   The static test data and interpretation provided in this section consists of 36 overburden till samples (Table 4.1-1 and Figure 4.1-1).   The overburden samples were screened with a 2-mm mesh, indicated in the sample identification by an A(+) for the larger particle size fraction and a B(-) for the smaller particle size fraction for a total sample dataset of 36.

4.1.2   Monitoring Well and Condemnation Samples

Samples were retrieved from diamond drill hole core samples (Table 4.1-2) from southwest and northeast of the Pit as well as from the plant site for ML/ARD testing.   These samples have been grouped together as representative overburden samples from outside of the ultimate open pit limits. Note that condemnation drilling and monitoring well samples are abbreviated as CD and MW, respectively in the following sections.

4.1.3   Tailings

SGS Lakefield Research Limited (SGS) produced two tailings samples from metallurgical locked cycle tests (LCTs; SGS Project Reference 11474-001) to determine if tailings material was suitable for TSF dam construction and environmental testing (Appendix C).   The rougher (F25) and cleaner (F26) tailings from LCTs were composited and screened using a 270 mesh (0.053 mm; Klohn Crippen Berger 2008).   The pyrite was removed from the coarse fraction (representing the cycloned tailings sand for dam construction) by an additional floatation circuit. The pyrite was then combined with the fine tailings (representing the tailings overflow).   The combined whole tailings are estimated to comprise 65% coarse and 35% fines (Klohn Crippen Berger 2008).   Static and kinetic tests were conducted by SGS Lakefield (Appendix C) and all data reported in this section have been obtained from this source.

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Table 4.1-1
Summary of Overburden Samples Submitted for ML/ARD
Prediction Testing

Sample Type	Primary Sample Code	Secondary Sample Code	Depth (m)	Coordinates Northing	Easting
TP	TP08-RS1	RS1-1A(+)	1.0	6119463	670856
TP	TP08-RS2	RS2-1A(+)	1.0	6119463	670656
TP	TP08-RS2	RS2-2A(+)	2.0	6119463	670656
TP	TP08-RS3	RS3-0.8A(+)	0.80	6118960	670639
TP	TP08-RS3	RS3-2A(+)	2.0	6118960	670639
TP	TP08-RS3	RS3-3A(+)	3.0	6118960	670639
TP	TP08-RS4	RS4-1A(+)	1.0	6119645	670839
TP	TP08-RS4	RS4-2A(+)	2.0	6119645	670839
TP	TP08-RS4	RS4-3A(+)	3.0	6119645	670839
TP	TP08-RS5	RS5-1A(+)	1.0	6118965	671078
TP	TP08-RS5	RS5-2A(+)	2.0	6118965	671078
TP	TP08-RS6	RS6-1A(+)	1.0	6119219	670391
TP	TP08-RS6	RS6-2A(+)	2.0	6119219	670391
TP	TP08-RS7	RS7-1A(+)	1.0	6119510	670282
TP	TP08-RS7	RS7-2A(+)	2.0	6119510	670282
TP	TP08-RS1	RS1-1B(-)	1.0	6119463	670856
TP	TP08-RS2	RS2-1B(-)	1.0	6119463	670656
TP	TP08-RS2	RS2-2B(-)	2.0	6119463	670656
TP	TP08-RS3	RS3-0.8B(-)	0.80	6118960	670639
TP	TP08-RS3	RS3-2B(-)	2.0	6118960	670639
TP	TP08-RS3	RS3-3B(-)	3.0	6118960	670639
TP	TP08-RS4	RS4-1B(-)	1.0	6119645	670839
TP	TP08-RS4	RS4-2B(-)	2.0	6119645	670839
TP	TP08-RS4	RS4-3B(-)	3.0	6119645	670839
TP	TP08-RS5	RS5-1B(-)	1.0	6118965	671078
TP	TP08-RS5	RS5-2B(-)	2.0	6118965	671078
TP	TP08-RS6	RS6-1B(-)	1.0	6119219	670391
TP	TP08-RS6	RS6-2B(-)	2.0	6119219	670391
TP	TP08-RS7	RS7-1B(-)	1.0	6119510	670282
TP	TP08-RS7	RS7-2B(-)	2.0	6119510	670282
DH	DH-07-1	SPT 5	7.3	6120054	670234
DH	DH08-1	SPT 1	1.2	6120064	670403
DH	DH08-1	Grab 3	5.2	6120064	670403
DH	DH08-2	SPT 1	0.061	6120472	669743
DH	DH08-2	SPT 4	4.0	6120472	669743
TP	TP08-L	NA	4.0	6121305	670015
TP	TP08-N	NA	1.0	6121117	669994
TP	TP08-J	NA	1.0	6120999	669939
TP	TP08-J	NA	2.0	6120999	669939

(continued)

Table 4.1-1
Summary of Overburden Samples Submitted for ML/ARD Prediction Testing (completed)

Sample Type	Primary Sample Code	Secondary Sample Code	Depth (m)	Coordinates	
				Northing	Easting
TP	TP08-M	NA	1.0	6123188	672197
MW	MW08-1	SPT 1	0.61-1.22	6123524	672499
MW	MW08-1	LPT 3	4.27-4.87	6123524	672499
DH	DH-07-02	SPT 6	8.8	6125496	671104
DH	DH-07-02	SPT 7	10	6125496	671104
DH	DH-07-03	SPT 4	5.8	6123345	671450
DH	DH-07-03	SPT 7	10	6123345	671450
DH	DH-07-04	SPT 4	5.8	6123637	671060
DH	DH-07-04	SPT 7	10	6123637	671060
DH	DH-07-05	SPT 3	4.3	6123951	670477
DH	DH-07-05	SPT 7	10	6123951	670477

Table Legend/footnote
Datum is NAD 83
TP = Test Pit
A(+) denotes sieved material greater than 2 mm.
B(-) denotes sieved material less than 2 mm
NA = Not Available

Table 4.1-2
Summary of Tailings Samples Submitted for ML/ARD Prediction Testing

Year	#Samples	Sample Description	Test Description
2005	10	Rough-scavenger and cleaner-scavenger tailings of lock cycle test work for cycles 3+4+5 in tests F46, F47, F48, F51 and F52 plus F46 and F47 rougher-scavenger composite.	Limited Acid-base accounting (ABA)
2006	2	Low S Tailings (F21) and total tailings (F17)	Limited modified ABA Aging Tests
2006	10	Rough-scavenger and cleaner-scavenger tailings of lock cycle test work for tests F53 C1 to C5	Static tests: ABA, solids total metals and oxides, mineralogical (optical and XRD).
2007	2	Composites of 90% rougher-scavenger and 10% cleaner-scavenger tailings for Tests F46 and F47	Static tests: ABA, solids total metals and oxides, mineralogical (optical and XRD). Kinetic Tests: humidity cells
2007	2	Rougher (F25) and cleaner (F26) tailings	Static tests: ABA, solids total metals and oxides, Kinetic Tests: humidity cells and ageing tests

Source: Modified from Morin and Hutt (Appendix A).

Table 4.1-2
Morrison Copper/Gold Project: Summary of Monitoring Well and Condemnation Drill Core Samples Submitted for ML-ARD Prediction Testing

Hole-ID	Type	Sample Number	From (m)	To (m)	Width	Depth to Bedrock	Lithology	Location	Northing	Easting	Elevation	Date
DH08-03	Condemnation	4101	4.42	6.00	1.58	4.42	BFP	Plant Site	6120228.23	669834.11	820.73	16/10/2008
DH08-03	Condemnation	4102	6.00	8.00	2.00	4.42	BFP	Plant Site	6120227.96	669833.12	819.27	16/10/2008
DH08-03	Condemnation	4103	8.00	10.00	2.00	4.42	BFP	Plant Site	6120227.66	669832.01	817.63	16/10/2008
DH08-03	Condemnation	4104	10.00	12.00	2.00	4.42	BFP	Plant Site	6120227.37	669830.91	815.99	16/10/2008
MW08-02A	Monitoring Well	4105	4.48	7.00	2.52	75.59	BFP	SW of Pit	6118990.00	670305.00	746.26	16/10/2008
MW08-02A	Monitoring Well	4106	7.00	9.00	2.00	75.59	Mafic Dyke	SW of Pit	6118990.00	670305.00	744.00	16/10/2008
MW08-02A	Monitoring Well	4107	9.00	11.00	2.00	75.59	Mafic Dyke	SW of Pit	6118990.00	670305.00	742.00	16/10/2008
MW08-02A	Monitoring Well	4108	11.00	13.00	2.00	75.59	BFP	SW of Pit	6118990.00	670305.00	740.00	16/10/2008
MW08-01A	Monitoring Well	4109	55.79	58.00	2.21	55.79	Siltstone	NE of Pit	6119626.00	671032.00	775.11	16/10/2008
MW08-01A	Monitoring Well	4110	58.00	61.00	3.00	55.79	Tuff	NE of Pit	6119626.00	671032.00	772.50	16/10/2008
MW08-01A	Monitoring Well	4111	61.00	64.00	3.00	55.79	Tuff	NE of Pit	6119626.00	671032.00	769.50	16/10/2008
MW08-01A	Monitoring Well	4112	64.00	67.00	3.00	55.79	Tuff	NE of Pit	6119626.00	671032.00	766.50	16/10/2008
MW08-01A	Monitoring Well	4113	67.00	70.00	3.00	55.79	Tuff	NE of Pit	6119626.00	671032.00	763.50	16/10/2008

Notes:
BFP = Biotite Feldspar Porphyry

**Morrison Copper/Gold Project Overburden,
Monitoring Well and Condemnation Drilling Sample Locations**

FIGURE 4.1-1

Legend:
- Condemnation Drilling
- Monitoring Well
- Overburden Sample Location
- Water Diversion
- Water Pipeline
- Tailings Pipeline
- Preferred Transmission Line Alignment
- Access Road
- 20m Contour
- 100m Contour
- Depression Contour
- Wetlands
- Stockpile and Borrow Area
- Access Road; Haul Road
- Other Infrastructure
- Dam
- Waste Rock Dump
- Pond
- Sanitary Landfill
- Tailings Impoundment at 1015 m
- Ultimate Pit Extent

1:25,000

0   500   1,000

Metres

Projection: UTM9, NAD83

Other metallurgical testing and tailings production is summarized in Morin and Hutt (Appendix A) and listed in Table 4.1-3 below. However, the SGS (Appendix C) test programs described above are considered the most representative tailings samples based on the current proposed mine plan. The other tailings test program is not discussed in this section but is detailed in Morin and Hutt (Appendix A) and Klohn Crippen Berger (2008).

Table 4.1-3
Cut-off Copper Grades Separating Ore, Low Grade Ore and Waste Rock Samples Used in the ML/ARD Prediction Testing

Waste Rock (< 0.2 CuEq%)		Low Grade Ore (0.2-0.3 CuEq%)		Ore (>0.3 CuEq%)	
Sample Numbers	%	Sample Numbers	%	Sample Numbers	%
207	40	97	19	213	41

Total ML/ARD prediction samples from within the pit outline = (excluding overburden samples).

4.1.4   Deposit Rock

4.1.4.1   Static Testing

A total of 516 deposit rock samples from diamond drill holes (DDHs) were collected between 2003 and 2006 for ML/ARD prediction and geochemical characterization). The ML/ARD sample distribution of lithology and alterations is similar to the larger Morrison DDH database (Compare Figures 4.1-2 to 4.1-5). Note that alteration codes were not available for all ML/ARD samples. The lithology of the samples is composed of BFP, sedimentary (SED), fault (FLT), dyke (DYK), argillic carbonate (ARCB), argillic sericite (ARSE), potassic (KH), kaolinitic (KL), quartz sericite (QZSE), and overburden (overburden).

To differentiate between potential waste rock, low-grade ore, and ore, the deposit rock ML/ARD prediction dataset was subdivided using estimated cut-off grades summarized in Table 4.1-4 and calculated as per Wardrop (2009).[1] Note a low-grade ore range of 0.2 to 0.3 CuEq% was selected. Wardrop (2009) states previous 3D model domains used for limiting block model estimations were based on a 0.2% Cu grade envelope and mineral resources were reported at a cut-off grade of 0.3% Cu (Snowden 2003; Beacon Hill 2004). For the 2007 resource update, it was decided to extend the grade envelope to a 0.1% Cu cut-off grade. This was considered justified for the following reasons: (1) metal prices have increased significantly since 2004; (2) copper and gold mineralization are not directly correlated; and (3) significant molybdenum grades occur in the southeastern portion of the deposit. Despite expanding the constraining grade envelope, the ultimate extends did not change significantly in many areas as there tends to be a sharp decrease in the outer limits of copper mineralization grade between a grade of 0.2% and <0.1%. However, a number of areas previously treated as internal waste (<0.2% copper) were above the 0.1% copper cut-off as well as a significant portion to the northwest.

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[1] Cu Eq% = Cu% + Au g/t * 0.23 + Mo% * 7.794

ML/ARD db Sample Lithology (Lith, n, %)

Legend:
- BFP
- FLT
- SED

SED, 183, 35%

BFP, 322, 63%

FLT, 11, 2%

Morrison Copper/Gold Project
Pit Rock Sample Lithology for ML/ARD Testing

FIGURE 4.1-2

ML/ARD db Sample Alteration (Alt, n, %)

Morrison Copper/Gold Project
Pit Rock Sample Alteration for ML/ARD Testing

FIGURE 4.1-3
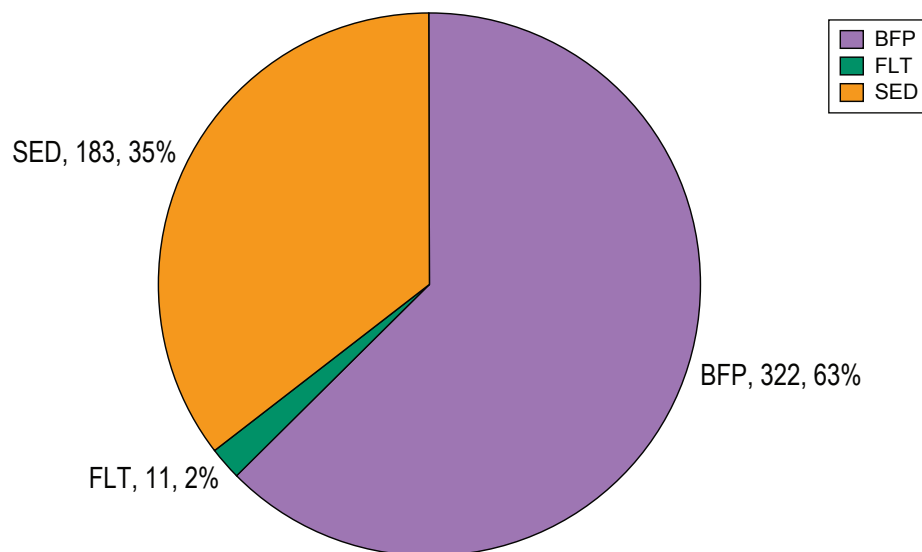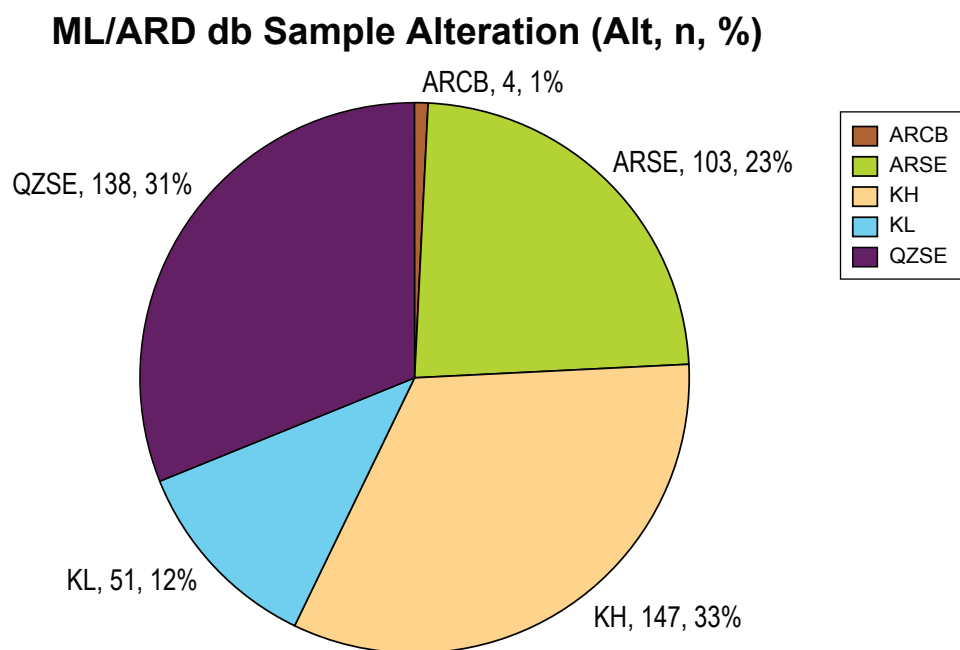
Geology Model Lithology: Based on 29,800 m of DDH
(Lithology; Total Lithology Core Length; Proportion of Total Core Length)

SED; 9,715 m; 32%

BFP; 17,785 m; 60%

OVB; 1,376 m; 5%

FLT; 703 m; 2%

DYK; 323 m; 1%

Legend:
- BFP
- DYK
- FLT
- OVB
- SED

Morrison Copper/Gold Project
Morrison Deposit Lithology Distribution

FIGURE 4.1-4

Alteration: Based on 21,700 m of DDH
(Alteration; Total Core Length; Proportion of Total Core Drilled)

ARCB; 977 m; 4%

QZSE; 3,124 m; 14%

KL; 3,124 m; 14%

ARSE; 4,967 m; 22%

KH; 10,834 m; 46%

Legend:
- ARCB
- ARSE
- KH
- KL
- QZSE

Morrison Copper/Gold Project
Morrison Deposit Alteration Distribution

FIGURE 4.1-5
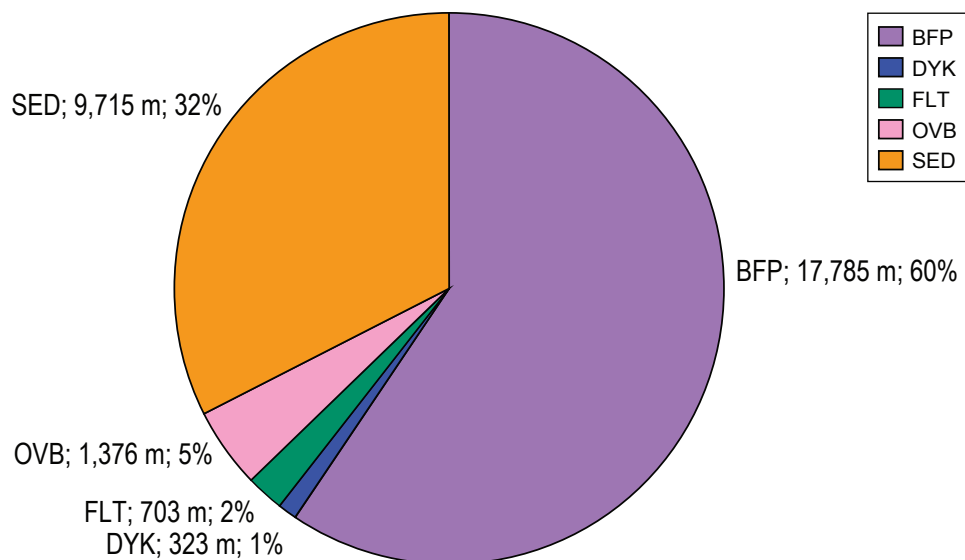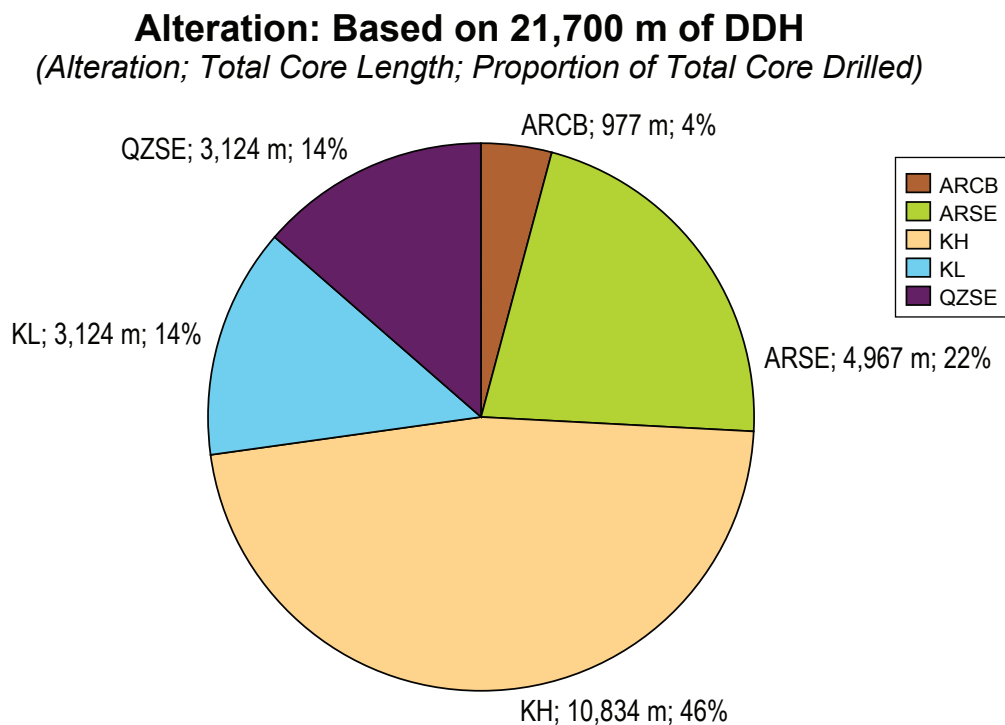
Table 4.1-4
Leach Test Pad One

Drill Hole	From (m)	To (m)	CuEq (%) Low	CuEq (%) High	Chalcopyrite (%) Low	Chalcopyrite (%) High	Pyrite (%) Low	Pyrite (%) High	Alteration	Lithology
MO-00-10	50	60	0.11	0.40	0	0.50	0	0.50	ARCB	BFP
MO-00-14	100	110	0.059	0.16	0	0.50	1.0	10	ARCB	BFP
MO-00-14	269	279	0.17	0.35	0	0.25	0.010	2.0	ARCB	BFP
MO-00-15	85	95	0.061	0.12	0.010	0.30	1.5	3.0	ARCB	BFP
MO-00-17	20	30	0.21	0.26	0.25	0.80	2.0	4.0	ARCB	BFP
MO-00-19	87	97	0.012	0.034	0	0	1.5	3.0	ARCB	BFP
MO-00-21	100	110	0.060	0.12	0.10	0.40	0.15	0.50	ARCB	BFP
MO-00-23	40	50	0.12	0.17	0.30	0.65	1.0	1.1	ARCB	BFP
MO-01-31	165	175	0.12	0.19	0.25	0.70	2.0	2.5	ARCB	BFP
MO-01-33	125	135	0.040	0.055	0	0	0.60	0.75	ARCB	BFP
MO-01-34	90	100	0.090	0.13	0	0.20	0.30	4.0	ARCB	BFP
MO-01-36	150	160	0.16	0.28	0.40	1.0	0.50	1.0	ARCB	BFP
MO-01-44	130	140	0.11	0.34	0	4.5	1.0	50	ARCB	BFP
MO-01-47	100	110	0.084	0.22	0	2.0	1.0	7.0	ARCB	BFP
MO-01-49	15	25	0.22	0.58	0.10	1.2	1.0	2.0	ARCB	BFP
MO-01-51	5.0	15	0.12	0.21	0.10	0.40	1.0	2.0	ARCB	BFP
MO-01-54	25	35	0.067	0.14	0.40	0.60	0.50	1.0	ARCB	BFP
MO-01-55	110	120	0.044	0.075	0	0.50	0.60	0.80	ARCB	BFP
MO-01-60	22	32	0.14	0.24	0	0.10	0.40	1.0	ARCB	BFP
MO-02-67	70	80	0.15	0.27	0.60	1.0	1.0	1.5	ARCB	BFP
MO-02-68	75	85	0.10	0.22	0	0.40	2.0	3.0	ARCB	BFP
MO-02-69	37	47	0.098	0.20	0	0.10	0.50	3.0	ARCB	BFP
MO-02-70	183	200	0.11	0.29	0	0.80	0.15	1.1	ARCB	BFP
MO-02-72	102	112	0.16	0.43	0.40	2.0	1.5	5.0	ARCB	BFP
MO-02-73	169	179	0.18	0.23	0	0.40	1.0	1.5	ARCB	BFP
MO-02-76	385	395	0.11	0.26	0	0.40	0.30	2.0	ARCB	BFP
MO-02-77	105	115	0.075	0.23	0	0.80	0.90	1.0	ARCB	BFP
MO-02-78	40	50	0.17	0.26	0.40	0.50	1.0	1.0	ARCB	BFP
MO-03-86	70	80	0.090	0.28	NA	NA	NA	NA	ARCB	BFP
MO-03-88	229	234	0	0	NA	NA	NA	NA	ARCB	BFP

CuEq (%) >0.3	0.2 ≤CuEq (%) ≤ 0.3	CuEq (%) <0.2

CuEq% = Cu%+Au g/t*0.23+Mo%*7.794 from Wardrop Feasibility, (2009)
ARCB = Argillic Carbonate
BFP = Biotite Feldspar Porphyry
NA = Not Available

Therefore, as a conservative approach, the copper equivalent % (Cu Eq%) cut-off for waste rock was considered 0.2% and low grade ore was set between 0.2 and 0.3%. This provides a conservative estimate (i.e., more tonnage of waste rock and low grade ore). Note that the

CuEq% calculation primary used assayed copper, gold and molybdenum from the master Morrison DDH database where available and used solid-phase ICP data from the ML/ARD database in lieu of.

4.1.4.2   Laboratory Scale Kinetic Testing

Individual drill core samples were selected for laboratory scale humidity cell testing by Morin and Hutt (Appendix A). Details of sampling and ML/ARD prediction results are provided in Appendix A.

4.1.4.3   Field Scale Kinetic Testing

Approximately one tonne of broken core was placed in each of four on-site leach test pads (LTPs) (Plate 4.1-1). Available information to date was used to compile descriptions of the core intervals, generally 10 m long, placed in each LTP (see Tables 4.1-5, 4.1-6, 4.1-7, and 4.1-8). Figure 4.1-6 shows the LTP sample interval locations and indicates good spatial distribution according to available Pacific Booker drill core.

Plate 4.1-1. Previous (Before July 14, 2009) Leach Test Pad One (Top Left), Leach Test Pad Two (Top Right), Leach Test Pad Three (Bottom Left), and Leach Test Pad Four (Bottom Right).

According to LTP composition indicated in Tables 4.1-5 to 4.1-8 and sample distribution shown in Figure 4.1-7, the LTPs are representative of the following mine materials to be excavated from the pit at the Project:

- LTP1 - Waste to Ore grade BFP lithology of ARCB alteration with an average pyrite content of approximately 2.5%.

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Report Version D.1   4–13   Rescan™ Environmental Services Ltd. (Proj. #0793-001-02)

- LTP2 - Waste to Ore grade SED lithology of QZSE alteration with an average pyrite content of approximately 1.3%.

- LTP3 – Waste to Ore grade BFP lithology of KH and KL alteration with an average pyrite content of approximately 1.1%.

Table 4.1-5
Leach Test Pad Two

Drill Hole	From (m)	To (m)	CuEq (%) Low	CuEq (%) High	Chalcopyrite (%) Low	Chalcopyrite (%) High	Pyrite (%) Low	Pyrite (%) High	Alteration	Lithology
MO-00-12	160	170	0.10	0.33	0.50	1.0	0.010	0.50	QZSE	SED
MO-00-13	4.0	14	0.10	0.22	0.25	0.50	1.0	1.5	QZSE	SED
MO-00-14	30	40	0.058	0.10	0	0.010	1.0	2.0	QZSE	SED
MO-00-20	20	30	0.098	0.17	0.40	0.85	1.0	1.5	QZSE	SED
MO-00-21	120	130	0.075	0.20	0.15	0.40	0.30	0.70	QZSE	SED
MO-01-25	135	145	0.16	0.28	0	0.30	0.010	0.30	QZSE	SED
MO-01-26	50	60	0.20	0.28	0.30	0.50	0.50	0.50	QZSE	SED
MO-01-27	55	65	0.15	0.35	0.010	0.50	1.0	1.5	QZSE	SED
MO-01-28	230	240	0.17	0.46	0.010	0.75	0.75	1.5	QZSE	SED
MO-01-29	71	79	0.16	0.31	0	1.2	0	1.0	QZSE	SED
MO-01-30	50	60	0.034	0.12	0	0.010	1.5	2.0	QZSE	SED
MO-01-31	70	80	0.070	0.15	0	0.010	0.50	1.5	QZSE	SED
MO-01-36	120	130	0.10	0.30	0.10	0.80	0.90	1.5	QZSE	SED
MO-01-39	51	61	0.18	0.25	0.10	0.70	0.40	1.0	QZSE	SED
MO-01-42	315	325	0.24	0.36	0.10	0.40	0.80	1.5	QZSE	SED
MO-01-44	40	50	0.17	0.38	0	0	1.5	2.0	QZSE	SED
MO-01-46	93	103	0.21	0.31	0.10	0.40	1.5	2.0	QZSE	SED
MO-01-47	60	73	0.14	0.30	0.50	2.0	1.0	4.0	QZSE	SED
MO-01-53	14	24	0.060	0.090	0	0.10	2.0	2.5	QZSE	SED
MO-01-55	78	88	0.25	0.35	0	1.0	0.80	3.5	QZSE	SED
MO-01-57	154	164	0.10	0.25	0	0.10	1.0	1.5	QZSE	SED
MO-02-64	41	51	0.012	0.054	0.50	0.60	1.5	2.0	QZSE	SED
MO-02-72	40	50	0.12	0.27	0.50	0.70	1.0	1.0	QZSE	SED
MO-02-73	40	50	0.057	0.10	0.10	0.50	0.40	3.5	QZSE	SED
MO-02-76	20	30	0.15	0.23	0.10	0.70	1.5	2.0	QZSE	SED
MO-03-83	240	250	0.18	0.25	NA	NA	NA	NA	QZSE	SED
MO-03-84	40	50	0.043	0.12	NA	NA	NA	NA	QZSE	SED
MO-03-84	180	190	0.10	0.25	NA	NA	NA	NA	QZSE	SED
MO-03-86	190	200	0.073	0.13	NA	NA	NA	NA	QZSE	SED
MO-03-86	40	50	0.057	0.083	NA	NA	NA	NA	QZSE	SED

CuEq (%) >0.3	0.2 ≤CuEq (%) ≤ 0.3	CuEq (%) <0.2

CuEq% = Cu%+Au g/t*0.23+Mo%*7.794 from Wardrop Feasibility, 2009
QZSE = Quartz Sericite
SED = Sedimentary
NA = Not Available

Table 4.1-6
Leach Test Pad Three

Drill Hole	From (m)	To (m)	CuEq (%) Low	CuEq (%) High	Chalcopyrite (%) Low	Chalcopyrite (%) High	Pyrite (%) Low	Pyrite (%) High	Alteration	Lithology
MO-00-13	50	60	0.049	0.18	0.25	0.75	0.50	2.0	KL	BFP
MO-00-18	78	88	0.097	0.14	0.20	1.6	0.20	0.60	KH	BFP
MO-00-21	66	76	0.032	0.056	0.50	0.65	0.70	0.85	KL	BFP
MO-01-24	230	242	0.14	0.26	0	0.80	0	0.25	KL	BFP
MO-01-25	21	31	0.10	0.30	0.010	0.60	0.50	1.5	KL	BFP
MO-01-27	200	210	0.41	0.59	1.0	1.5	0.25	0.50	KH	BFP
MO-01-29	390	400	0.19	0.23	0.50	0.80	0.010	0.75	KH	BFP
MO-01-30	75	85	0.083	0.19	0	0.50	1.5	3.0	KL	BFP
MO-01-31	208	222	0.17	0.26	0.010	1.2	1.0	3.0	KL	BFP
MO-01-33	66	76	0.19	0.23	0.50	1.0	0.50	1.0	KL	BFP
MO-01-36	60	70	0.079	0.18	0.10	0.30	0.10	1.0	KL	BFP
MO-01-38	255	265	0.14	0.26	0.20	0.80	1.0	1.5	KH	BFP
MO-01-39	100	110	0.28	0.38	0.80	1.0	0.80	2.5	KH	BFP
MO-01-42	50	60	0.15	0.24	0.10	0.95	0.40	0.40	KH	BFP
MO-01-43	75	85	0.13	0.25	0.20	0.80	0.50	2.0	KL	BFP
MO-01-51	85	95	0.068	0.46	0.50	0.50	1.0	1.5	KL	BFP
MO-01-60	97	107	0.13	0.28	0.30	0.60	0.60	1.1	KL	BFP
MO-01-61	94	104	0.080	0.11	0	0.40	0.90	2.1	KL	BFP
MO-02-65	220	230	0.27	0.36	0.40	1.5	0.90	2.0	KH	BFP
MO-02-69	221	231	0.11	0.58	0.15	0.70	0.25	1.2	KH	BFP
MO-02-71	85	95	0.12	0.27	0.30	0.40	3.5	4.0	KH	BFP
MO-02-75	136	146	0.15	0.23	0.30	1.2	0.20	0.80	KL	BFP
MO-02-76	117	127	0.078	0.38	0.10	1.5	0.70	3.0	KL	BFP
MO-02-77	40	50	0.14	0.27	0.40	0.50	0.30	2.5	KH	BFP
MO-02-78	71	81	0.15	0.23	0.30	0.50	0.20	0.40	KH	BFP
MO-02-80	170	180	0.18	0.26	0.50	0.90	0.50	0.60	KL	BFP
MO-03-83	280	290	0.0060	0.015	NA	NA	NA	NA	KL	BFP
MO-03-84	245	255	0.16	0.21	NA	NA	NA	NA	KH	BFP
MO-03-85	19	28	0.049	0.11	NA	NA	NA	NA	KH	BFP
MO-03-86	214	224	0.18	0.27	NA	NA	NA	NA	KH	BFP

CuEq (%) >0.3	0.2 ≤CuEq (%) ≤ 0.3	CuEq (%) <0.2

CuEq% = Cu%+Au g/t*0.23+Mo%*7.794 from Wardrop Feasibility, (2009)
KL = Kaolinitic
KH = Potassic
BFP = Biotite Feldspar Porphyry
NA = Not Available

Table 4.1-7
Leach Test Pad Four

Drill Hole	From (m)	To (m)	CuEq (%) Low	CuEq (%) High	Chalcopyrite (%) Low	Chalcopyrite (%) High	Pyrite (%) Low	Pyrite (%) High	Alteration	Lithology
MO-00-10	155	164	0.087	0.25	0.50	1.0	0.50	1.3	QZSE	SED
MO-00-12	70	78	0.15	0.35	0.50	0.50	0.50	0.50	QZSE	SED
MO-00-13	87	95		0.23	0.75	1.0	0.50	1.0	KH	BFP
MO-00-14	90	99	0.046	0.10	0.010	0.010	0.50	1.0	ARCB	BFP
MO-00-16	109	117	0.26	0.30	0.80	1.0	0.30	0.50	QZSE	SED
MO-00-20	64	73	0.16	0.22	0.60	1.0	0.50	0.60	QZSE	SED
MO-01-30	103	111	0.100	0.24	0.010	0.25	1.0	2.0	QZSE	SED
MO-01-31	14	23	0.073	0.10	0	0	1.3	2.0	QZSE	SED
MO-01-37	144	152	0.18	0.28	0.45	0.45	0.10	0.50	QZSE	SED
MO-01-41	288	297	0.070	0.20	0.10	0.20	0.80	1.0	ARCB	BFP
MO-01-41	28	37	0.25	0.89	0.40	1.5	0.40	0.50	KL	BFP
MO-01-45	103	108	0.19	0.22	0	0	0.50	1.0	QZSE	SED
MO-01-46	40	47	0.19	0.44	0	0.50	0.50	2.0	QZSE	SED
MO-01-48	195	203	0.16	0.27	0	0.50	0.50	1.0	QZSE	SED
MO-01-56	86	94	0.38	0.44	0.50	0.80	0.50	1.5	QZSE	SED
MO-01-56	145	153	0.19	0.32	0.10	0.40	1.5	2.0	ARCB	BFP
MO-01-58	210	219	0.17	0.27	0	0.50	0.80	1.5	QZSE	SED
MO-01-73	97	105			NA	NA	NA	NA	QZSE	SED
MO-02-50	154	162			NA	NA	NA	NA	KL	BFP
MO-02-70	78	89	0.16	0.27	0.60	0.80	0	2.0	ARCB	BFP
MO-02-76	69	77	0.13	0.16	0.40	0.60	0.50	0.70	QZSE	SED
MO-02-77	77	86	0.056	0.29	0	0.10	0	1.1	ARCB	BFP
MO-02-79	46	53	0.054	0.38	0.10	0.90	0.50	2.5	ARCB	BFP
MO-02-81	236	244	0.074	0.16	0.10	0.30	0.40	0.60	QZSE	SED
MO-02-82	52	61	0.14	0.18	0.40	0.60	0.45	2.0	KH	BFP
MO-03-84	78	86	0.0080	0.031	NA	NA	NA	NA	QZSE	SED
MO-03-85	182	191	0.086	0.17	NA	NA	NA	NA	KH	BFP
MO-03-86	133	145	0.13	0.46	NA	NA	NA	NA	QZSE	SED
MO-03-87	188	196	0	0.010	NA	NA	NA	NA	0	SED
MO-03-88	164	170	0	0	NA	NA	NA	NA	0	SED
MO-03-89	84	94	0.10	0.28	NA	NA	NA	NA	ARCB	BFP
MO-03-90	133	145	0.15	0.32	NA	NA	NA	NA	KH	BFP
MO-03-90	277	288	0.010	0.030	NA	NA	NA	NA	0	SED

CuEq (%) >0.3	0.2 ≤CuEq (%) ≤ 0.3	CuEq (%) <0.2

CuEq% = Cu%+Au g/t*0.23+Mo%*7.794 from Wardrop Feasibility, (2009)
QZSE = Quartz Sericite
ARCB = Argillic Carbonate
KH = Potassic
KL = Kaolinitic
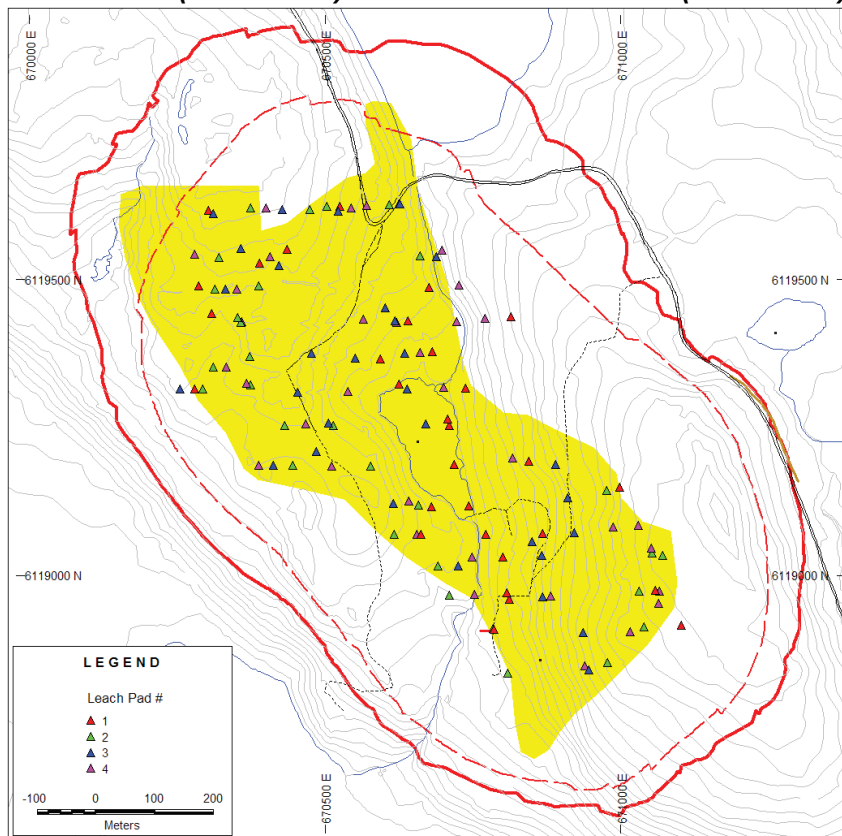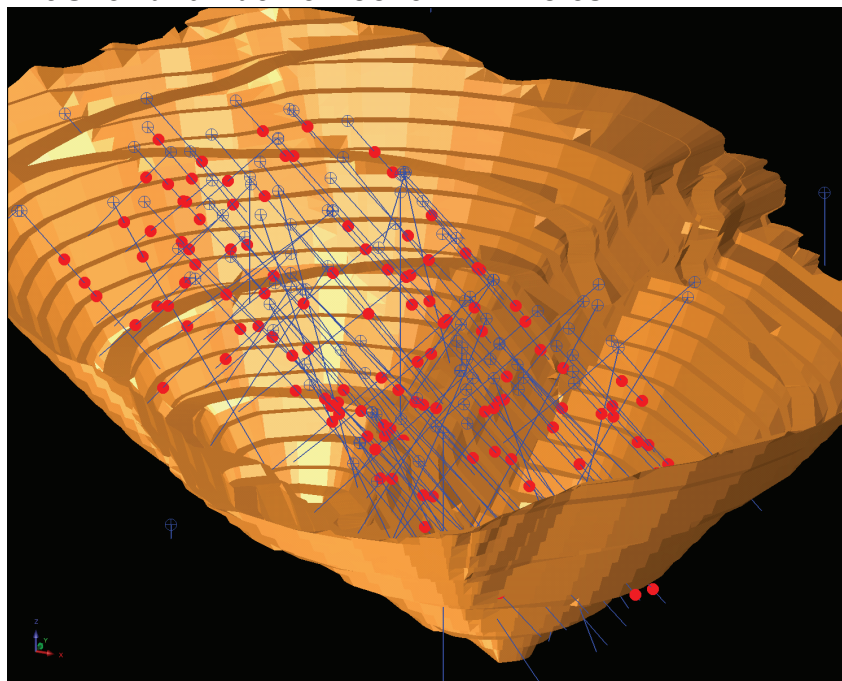**BFP = Biotite Feldspar Porphyry**
SED = Sedimentary
NA = Not Available

**Plan View of Leach Test Pad Distribution with Previous
Pit outline (Red Dash) and Current Pit Shell (Solid Red)**

**Oblique View Looking North of Current
Pit Shell and Pacific Booker Drill Holes**

**Morrison Copper/Gold Project
Distribution of Leach Test Pad Samples**

FIGURE 4.1-6

- LTP4 – Waste to Ore grade mixtures of SED and BFP lithology of QZSE, KH, KL, and ARCB alteration with an average pyrite content of approximately 0.89%.

The four on-site LTPs were reconstructed in 2009 with the addition of barrel type LTPs for historical Noranda core. This is discussed further in Section 5.2.2.

4.1.5   Lake Sediments

Because of the likely reducing conditions in sediments of Booker Lake and Ore Pond (i.e., authigenic sulphide precipitation) and potential metal enrichment (i.e., proximity to the deposit), a preliminary geochemical program was initiated to assist in developing material management strategies. Several submerged sediment samples were retrieved with an Ekman grab sampler from waterbodies within and outside the ultimate pit footprint as well as from the TSF footprint (see Rescan 2008, 2009a).

4.2   Metal Leaching and Acid Rock Drainage Testing Program

4.2.1   Overburden, Monitoring Well, and Condemnation Drill Core

The ML/ARD prediction program developed the overburden, monitoring wells, and condemnation DDH core samples as outlined in sections 4.1.1 and 4.1.2 above included the following static test (i.e., data which are one-time only and NOT a function of time) analyses:

- Four acid "near total" digestion followed by a 48 element suite (Ag, Al, As, Ba, Be, Bi, Ca, Cd, Ce, Co, Cr, Cs, Cu, Fe, Ga, Ge, Hf, Hg, In, K, La, Li, Mg, Mn, Mo, Na, Nb, Ni, P, Pb, Rb, Re, S, Sb, Sc, Se, Sn, Sr, Ta, Te, Th, Ti, Tl, U, V, W, Zn, Zr) Inductively Coupled Plasma-Mass Spectrometry (ICP-MS) scan.

- X-Ray Fluorescence for the oxides Al_2O_3, BaO, CaO, Cr_2O_3, Fe_2O_3, K_2O, MgO, MnO, NaO_2, SiO_2, SrO, TiO_2, P_2O_5 and loss on ignition.

- ABA:
  - paste pH
  - sulphide speciation (Total Sulphur, Sulphide-Sulphur, Sulphate-Sulphur (carbonate and HCl leach))
  - fizz Rating
  - Sobek NP
  - carbon speciation (Total Carbon, Total Inorganic Carbon)

- In addition, overburden samples collected from outside the pit outline were submitted for BC MEM or modified SWEP shake flask leachate extraction procedure (end-over-end 24 hr agitation of 250 grams of the <9.5mm fraction in 750 mL Reagent Water Type IV extraction fluid). Analyses on leachate included:

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- pH, EC, acidity, alkalinity, sulphate, chloride, fluoride, dissolved metals Al, Sb, As, Ba, Be, Bi, Cd, Ca, Cr, Co, Cu, Fe, Pb, Mg, Mn, Hg, Mo, Ni, P, K, Se, Si, Ag, Na, Sr, Tl, Sn, Ti, U, V, Zn.

4.2.2   Tailings

The ML/ARD prediction program carried out by SGS for the tailings samples outlined in Section 4.1.3 includes the following static and kinetic (i.e., data which IS a function of time) test analyses:

4.2.2.1   Static Testing

- Four acid "near total" digestion followed by a 29 element suite (Ag, Al, As, B, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, Li, Mg, Mn, Mo, Na, Ni, Pb, Sb, Se, Si, Sn, Ti, V, Zn) Inductively Coupled Plasma-Optical Emission Spectrometry (OES) or Mass Spectrometry (MS) scan.

- X-Ray Fluorescence for the oxides Al_2O_3, CaO, Cr_2O_3, Fe_2O_3, K_2O, MgO, SiO_2, P_2O_5, V_2O_5 and loss on ignition.

- ABA:

  - pH Rinse

  - paste pH

  - sulphide speciation (Total Sulphur, Sulphide-Sulphur, Sulphate-Sulphur (assumed to be HCl leach))

  - fizz Rating

  - Sobek NP

  - carbon speciation (Total Inorganic Carbon)

4.2.2.2   Kinetic Testing

- Tailings supernatant 61 day (day 0, 7, 29 and 61) ageing tests for analyses of pH, acidity, alkalinity, conductivity, TDS, TSS, EMF, F, Cl, SO4, NO2, NO3, NH3+NH4, Ag, Al, As, B, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, Li, Mg, Mn, Mo, Na, Ni, Pb, Sb, Se, Si, Sn, Ti, V, and Zn.

- Humidity cell testing[2] and leachate analyses of pH, conductivity, alkalinity, acidity, SO4, Ag, Al, As, B, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, Li, Mg, Mn, Mo, Na, Ni, Pb, Sb, Se, Si, Sn, Ti, V, and Zn.

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2 The data set discussed in this chapter includes 80 weeks-worth as of January, 2008. Humidity cells are still operational at the time of document writing.

4.2.3   Pit Rock

The ML/ARD prediction program completed by Morin and Hutt (Appendix A) included the following static and kinetic test analyses:

4.2.3.1   Static Testing

- The standard-Sobek (EPA 600) expanded ABA as per Section 4.2.1 above.

- A four acid "near total" digestion followed by a 48 element ICP-MS scan as per Section 4.2.1 above.

- X-Ray Fluorescence for 13 elements including LOI as per Section 4.2.1 above.

4.2.3.2   Kinetic Testing

- Five rock samples were submitted for humidity cell testing with aqueous determinations for pH, acidity, alkalinity, sulphate, conductivity, and dissolved metals.

- Approximately one tonne of broken core was selected and placed on each of four on-site field LTPs in 2005. Collection of LTP leachate began in 2006.

In addition, all kinetic test samples were submitted for Petrographic (PET) and quantitative X-Ray Diffraction analyses for mineralogical determinations.

4.2.4   Lake Sediments

Detailed testing methodology and data can be found in Rescan's aquatic baseline reports (2008; 2009a). A summary of the testing is listed below and includes:

- particle size analyses (sieve and pipette)

- pH by 1:2 solid to water rinse

- moisture content

- organic carbon by combustion

- total nitrogen by Leco

- available phosphate

- total solid–phase cyanide

- 30 element suite for total metals by ICP-OES or MS finish

5. Results and Discussion

Previous interpretations of geochemical results are detailed in Morin and Hutt (Appendix A). The following sections describes:

- additional ML/ARD prediction data gathered to date;
- additional ML/ARD prediction data gathered to supplements interpretations found in Morin and Hutt (Appendix A), such as sample refinement into the different lithologies, alteration types, and grade according to CuEq%[3] ranges previously indicated (Table 4.1-4);
- additional ML/ARD prediction data that provides new interpretations (primarily on overburden and tailings).

The reader is encouraged to read Morin and Hutt (Appendix A) before the following sections. Waste rock, low grade ore, ore, overburden, monitoring well, condemnation drilling, and representative tailings data can be found in Appendices A, B, and C.

5.1 Static Testing

5.1.1 Acid-Base Accounting

ABA provides a guide to determining the acid producing and acid consuming components of mine waste materials. Brief descriptions of the ABA parameters tested and calculated are included in the following sections after Sobek et al. (1978), Price (1997), and Morin and Hutt (1997).

5.1.1.1 Paste pH

Alone, paste pH does not provide an indication of the ARD potential of a sample. However, it is a principal determinant of both mineral reaction rates and mineral solubility. Therefore, it can provide insight into drainage chemistry. For example, acidic paste pH can be indicative of the onset of sulphide oxidation and ARD, whereas near-neutral to higher paste pH values suggest that available NP in a sample provide local micro- to meso-scale buffering of acidity generated from sulphide oxidation.

Figure 5.1-1 shows the range of measured paste pH against measured sulphide sulphur from all sampled materials types. The paste pH ranged from acidic to alkaline (3.9 to 9.3). In general, lower paste pH values are a function of the flushing of accumulated oxidation products as well as the pH of the paste pH water used in this type of analysis (generally in the pH range of 5 to 6). Sulphide sulphur and other sulphur species are discussed in Section 5.1.1.2.

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[3] CuEq% = Cu%+Au g/t*0.23+Mo%*7.794 from Wardrop 2009.

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Paste pH vs. Measured Sulphide

Any value less than detection
is shown at one-half detection

Measured Sulphide (%S)

Legend: Ore, LGO, WR, OVB, CD, MW, Coarse Tails, Fine Tails

Morrison Copper/Gold Project
Paste pH vs. Measured Sulphide

FIGURE 5.1-1

The two tailings samples had alkaline paste pH values of 9.3 for the coarse fraction and 8.8 for the fine fraction. Note that the coarse fraction tailings sample is considered the most representative tailings material for cycloned sand dam construction. Paste pH values for these two samples indicate sulphide oxidation is buffered by NP, at least over short time scales (i.e., in this case, the time elapsed between metallurgical testing and paste pH determinations, unknown at this time but likely on the order of days). An additional noteworthy observation is the consistent near-neutral to slightly alkaline paste pH values of the condemnation and monitoring well samples. This indicates short-term NP is available, at least for these samples. Waste rock, low grade ore, and ore paste pH values span a large range from acidic (defined as paste pH less than 5) to alkaline (defined as paste pH greater than 8).

5.1.1.2 Sulphur Species and Acid Potentials

Possible sulphur species that could be found in Morrison rock and tailings are: sulphide minerals including pyrite, leachable sulphate like gypsum or anhydrite, and non-leachable sulphate like barite. The sum of these species theoretically equals total sulphur, although analytical inaccuracy and the existence of other sulphur species rarely yield an exact balance (Appendix A). Minerals containing sulphide sulphur are the main source of acidity and trace metal contaminants from mining disturbed rock materials including waste rock, low grade ore, pit walls, and tailings. Therefore, the determination of all sulphur species is a fundamental component in predicting material reactivity and mine drainage chemistry.

Sulphide sulphur (Figure 5.1-1) ranges between less than the method detection limit (MDL; 0.01 %) and 8.8%. The coarse- and fine-tailings samples had 0.11% and 0.46% sulphide sulphur, respectively. Interestingly, overburden samples showed very little sulphide except for four samples collected from DDHs within the pit outline, which have measured sulphide sulphur greater than 0.3% (Table 5.1-1). The lithology of these samples should be confirmed through visual inspection by a qualified professional geoscientist (i.e., exploration geologist).

Table 5.1-1
Overburden Sample Requiring Confirmation of Lithology

DDH	From (m)	To (m)
MO-00-10	2.0	5.0
9240-3	14.0	17.0
MO-02-66	6.0	9.0
9240-1	5.0	8.0

The high correlation between total sulphur and measured sulphide sulphur in the DDH samples indicates the majority of the total sulphur is present as sulphides (Figure 5.1-2). The overburden samples are the exception, which show a deviation from the 1:1 total sulphur to sulphide sulphur ratio, especially near the MDL. In general, overburden samples show some leachable (HCl and carbonate) sulphate sulphur. However, for the remainder of the sample types, total sulphur is a good indication of sulphide sulphur and could be used as a surrogate for sulphide sulphur.

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Morrison Copper/Gold Project
Total Sulphur vs. Measured Sulphide-Sulphur

FIGURE 5.1-2
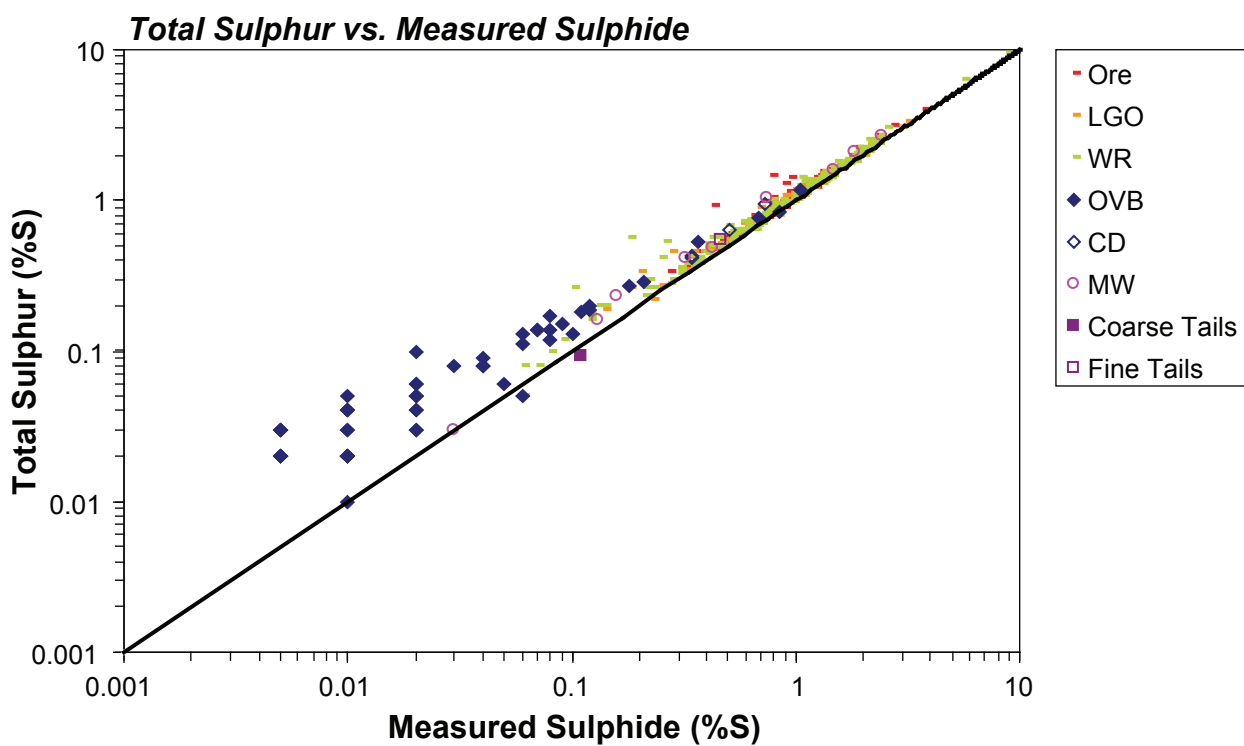
Some minerals containing sulphate, such as barite ($BaSO_4$), can be relatively insoluble or non-leachable. To determine if non-leachable barite was a significant source of sulphur species it was assumed that all measured solid-phase barium by near-total digestion and ICP-MS analyses occurred as barite. This showed that maximum barite ranged from 0.0012 to 0.16%S (Figure 5.1-3). In general, most sample types have very little relative sulphate sulphur as barite, except for the overburden, where samples approached the total sulphur to sulphate sulphur due to barite 1:1 ratio. In addition, there are also some waste rock, monitoring wells, and low-grade ore samples with a high non-leachable sulphate sulphur content.

To account for any missing sulphur caused by analytical error and as a conservative approach to determining the sulphide sulphur acid potential (SAP) of mine materials, a QA/QC mass-balance equation (Equation 5.1-1) for sulphur species is used (Morin and Hutt 1997).

$$\%S(del\ actual) = \%S(Total) - \%S(Sulphide) - \%S(HCl\text{-}leachable\ sulphate) - \%S(BaSO4) \qquad Eqn\ 5.1\text{-}1$$

Large negative values of $\%S(del_{actual})$ indicate the sum of sulphur species exceeds the measured total sulphur, caused by analytical inaccuracy and detection limits. Large positive values indicate either (1) total sulphur was overestimated, or (2) one or more sulphur species were underestimated. In this case, positive values (missing sulphur) can be added to acid-generating sulphide for safer SAP calculations. Morin and Hutt (Appendix A) found acceptable balances for the Morrison samples.

Total sulphur acid potential (TAP) can provide a safer estimate of ARD potential when there is a significant amount of error when comparing TAP and SAP values. However, TAP and SAP (including any missing sulphur) showed good correlation (Figure 5.1-4) with the exception of overburden samples as previously noted in Figure 5.1-2.

5.1.1.3   Neutralization Potentials

There were three types of NPs tested in these rock samples, all of which are expressed in units of kg $CaCO_3$ equivalent/tonne (kg/t) and discussed further in Morin and Hutt (Appendx A). Briefly, the different types of NP include:

1. Sobek "Bulk Neutralization Potential." This is based on placing a sample in an acid bath for an hour to determine how much acid was neutralized in the short term (Sobek et al. 1978).

2. Carbonate-equivalent neutralization potential (CaNP). This is calculated from measured solid phase concentrations of inorganic carbon (Inorg CaNP) or total carbon (Total CaNP).

3. Calculated Calcium CaNP. This assumes that all calcium from ICP occurs as calcite (Ca CaNP) or all calcium + magnesium occurs as calcite and dolomite (Ca + Mg CaNP).

Morin and Hutt (Appendix A) set the unavailable NP in Morrison pit rock samples at 13 kg/t. At the time of the analyses, the subdivision of samples (i.e., overburden, waste rock, low-grade ore, ore, CD, and monitoring wells) was not available to determine if the unavailable NP varied with each type of material. Figure 5.1-5 shows the bulk Sobek NP potential of all samples versus paste pH. Note that Morin and Hutt (Appendix A) identified samples with uncertain Sobek NP based on

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Total Sulphur vs. Maximum Calculated Non- Leachable Sulphate as Barite

Legend:
- Ore
- LGO
- WR
- OVB
- CD
- MW
- Coarse Tails
- Fine Tails

Y-axis: **Total Sulphur (%S)**

X-axis: **Maximum Calculated Non-Leachable Sulphate as Barite (%S)**

Note: Any value less than detection is shown at one-half detection

Morrison Copper/Gold Project
Total Sulphur vs Maximum Calculated
Non-Leachable Sulphate-Sulphur as Barite

FIGURE 5.1-3

Total-Sulphur-Based Acid Potential vs. Sulphide-Based Acid Potential

Note: Any value less than detection is shown at one-half detection

Morrison Copper/Gold Project
Total Sulphur-Based Acid Potential
vs Sulphide-Based Acid Potential

FIGURE 5.1-4

Paste pH vs. Bulk Sobek NP

Morrison Copper/Gold Project
Paste pH vs Bulk Sobek Neutralisation Potential

FIGURE 5.1-5

the disagreement of the fizz rating, which determines the amount of acid added during the test. The relevant samples have been identified in Figure 5.1-5 and have been omitted from Table 5.1-2, which shows the potential adjustments to NP (i.e., a NP value subtracted from each material type that is considered as unavailable NP) based on criteria of paste pH less than 6 or less than 5. As a conservative approach, the unavailable NP of 13 kg/t derived from the pit ore material type was retained for further calculations of adjusted NP.

Table 5.1-2
Potential Adjustments to NP for Different Material Types

Type	Paste pH	Potential Unavailable NP Adjustment
Ore	5.8	13
	4.8	2
Low-grade ore	5.3	9
	4.4	0.5
Waste rock	5.3	12
	4.7	11
Overburden	5.9	9
	4.9	4
CD	NA	NA
MW	NA	NA
Tailings	NA	NA

Morin and Hutt (Appendix A) found a comparison of bulk Sobek NP to Inorg CaNP (representing total carbonate) showed a good correlation (Figure 5.1-6). Based on a combined analytical error for both bulk Sobek NP and inorganic carbon of a factor of 1.5 (up to 50% inaccuracy above the 1:1 line and up to 33% inaccuracy below), most samples had about the same levels of bulk Sobek NP and Inorg CaNP. However, a portion of all material types, except CD samples (condemnation drilling at the plant site) showed more Inorg CaNP than bulk NP indicating the presence of net-neutral carbonate minerals, such as siderite. Some samples showed less bulk Sobek NP than Inorg CaNP, suggesting large analytical errors or large non-carbonate, aluminosilicate sources of NP. Available mineralogical analyses of Morrison rock in humidity cells indicate the presence of both siderite and aluminosilicates, supporting these interpretations (Section 5.1.4, Table 5.1-3). Note that PET analyses are a volume % determination, whole XRD is a weight-% determination, and thus the two colums for each sample should not necessarily be the same value.

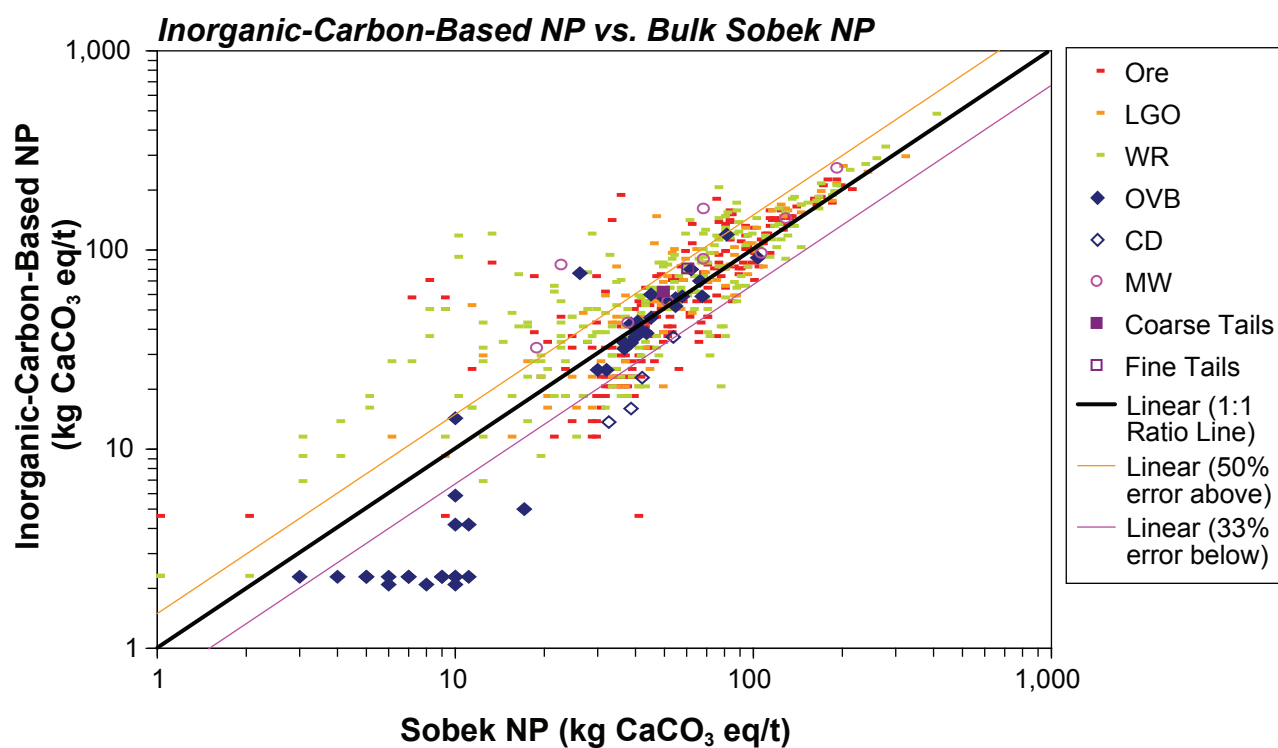Morin and Hutt (Appendix A) showed good correlations between Ca+Mg NP and bulk Sobek NP greater than approximately 125 kg/t (Figure 5.1-7), indicating dolomite and calcite account for the majority of NP. At bulk Sobek NP values of less than 125 kg/t, Ca+Mg NP can overestimate NP, specifically for overburden, CD, and tailings samples as well as waste rock samples below bulk Sobek NP values of approximately 12.

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Inorganic-Carbon-Based NP vs. Bulk Sobek NP

Morrison Copper/Gold Project
Bulk Sobek NP vs Inorganic-Carbon-Based NP

FIGURE 5.1-6

Table 5.1-3
Humidity Cell Mineralogy

Sample		MO-01-55		MO-00-19		MO-01-24		Leach Pad 2		MET-02	
From-To (m)		80.4-80.8		83-83.4		86.33-86.87		NA		92.72-96.08	
Grade (CuEq%)		0.28		0.038		0.51		0.17		0.081	
Material Type		Low Grade Ore		Waste Rock		Ore		Waste Rock		Waste Rock	
Lithology		SED		BFP		BFP		SED		BFP	
Alteration		QZSE		ARSE		ARSE		QZSE		KL	
Mineral	**Ideal Formula**	**PET**	**XRD**	**PET**	**XRD**	**PET**	**XRD**	**PET**	**XRD**	**PET**	**XRD**
Quartz	SiO_2	40	36.6	45	46.9	45	47.3	40	39.4	25	25.8
Biotite	$K(Mg,Fe^{2+})_3AlSi_3O_{10}(OH)_2$									8	5.2
Muscovite-sericite	$KAl_2AlSi_3O_{10}(OH)_2$	30	35.6	2	1.6	1	2	30	28.6		
Kaolinite	$Al_2Si_2O_5(OH)_2$	10	10.7	35	42.8	20	30.8	10	9.1	5	6.7
Feldspars	General							5			
K-feldspar	$KAlSi_3O_8$		3.5		1.5				1.7		
Plagioclase	$NaAlSi_3O_8$-$CaAl_2Si_2O_8$		0.5						10.3	45	49.9
Carbonates	General	10		5		30		8		3	
Calcite	$CaCO_3$		0.7				0.2		1		0.9
Siderite	$FeCO_3$		6.3		3.6		2.5		2.3		
Dolomite	$CaMg(CO_3)_2$		3.3				16		1.6		4.8
Sulphides	General										
Pyrite	FeS_2	2	2	1	1.4	tr		2	1.2	tr	
Marcasite	FeS_2	tr		2	1.1	tr		tr	0.2	tr	
Chalcopyrite	$CuFeS_2$	tr		tr		2	0.3	1	0.1	1	0.3
Sphalerite	$(Zn,Fe)S$	1	0.8	tr				tr	0.2		
Molybdenite	MoS_2	tr	0.1			tr?	0.1				
Bornite	Cu_5FeS_4	tr									
Galena	PbS	tr									
Anhydrite	$CaSO_4$				0.4						
Chlorite		tr		tr				1		2	
Clinochlore	$(Mg,Fe^{2+})_5Al(Si_3Al)O_{10}(OH)_8$								0.9		2.2
Actinolite	$Ca_2(Mg,Fe^{2+})_5Si_8O_{22}(OH)_2$	tr									2.7
Andalusite	Al_2SiO_5								2.4		
Magnetite	Fe_3O_4				0.4		0.3		0.4	2	1.5
Rutile	TiO_2	1		2		1	0.6	1	0.7	1	
Gorceixite	$BaAl_3(PO_4)(PO_3OH)(OH)_6$				0.3						
Fe-oxides				5							
Calci-amphibole										3	
Ilmenite										tr?	

PET = volume% from petrography.
XRD = weight % from X-ray Diffraction.
tr = trace <1%.
NA = not available.

Calcium-and-Magnesium-Based NP vs. Bulk Sobek NP

Note: Any value less than detection is shown at one-half detection

Morrison Copper/Gold Project
Calcium and Magnesium-Based NP vs Bulk Sobek NP

FIGURE 5.1-7
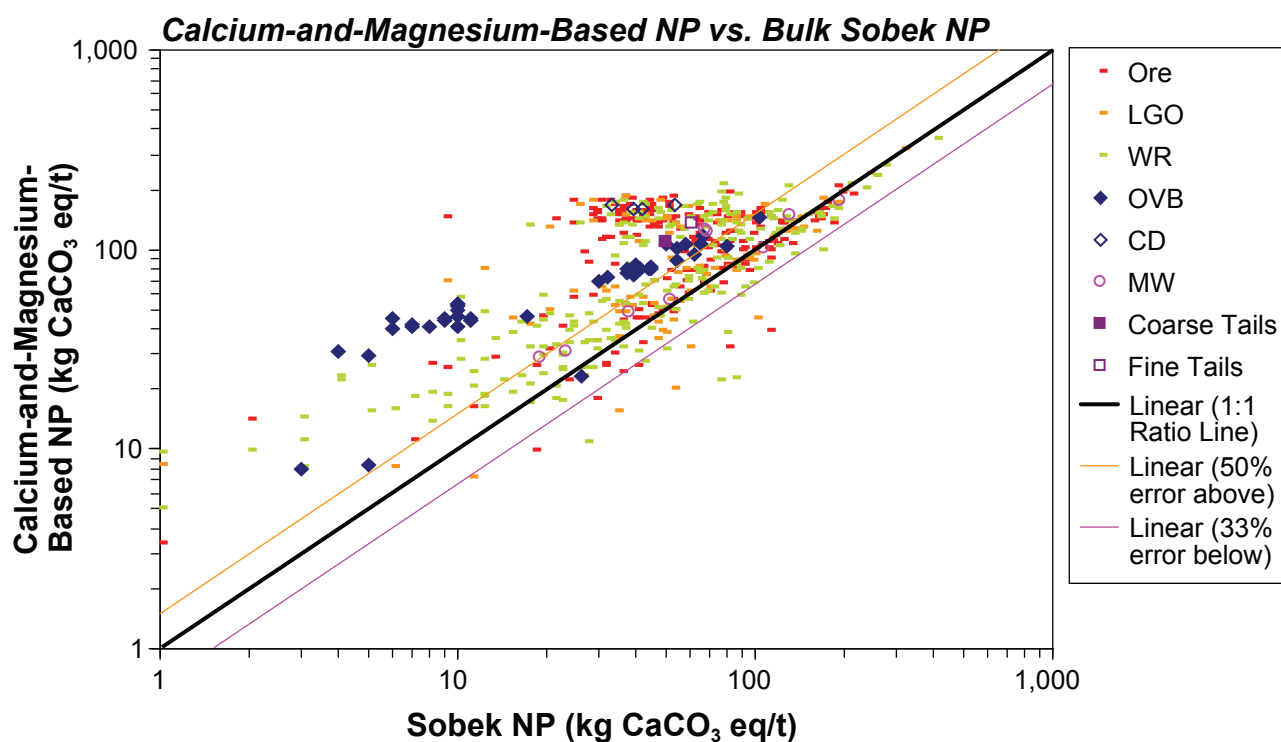
As a conservative approach to NP designation, bulk Sobek NP should be used for materials NP designation, with the understanding that Inorg CaNP estimates may not provide immediate buffering in mine drainage caused by the series of siderite weathering reactions (Equation 5.1-2, Equation 5.1-3 and Equation 5.1-4), which results in an overall net-neutral reaction (Equation 5.1-5).

$$FeCO_{3(s)} + 2H^+_{(aq)} = Fe^{2+}_{(aq)} + CO2_{(g)} + H2O_{(l)} \qquad\qquad Eqn\ 5.1\text{-}2$$

$$Fe^{2+}_{(aq)} + \tfrac{1}{4}O_{2(aq)} + H^+_{(aq)} = Fe^{3+}_{(aq)} + \tfrac{1}{2}H_2O_{(l)} \qquad\qquad Eqn\ 5.1\text{-}3$$

$$Fe^{3+}_{(aq)} + 2H_2O_{(l)} = \alpha FeOOH_{(s)} + 3H^+_{(aq)} \qquad\qquad Eqn\ 5.1\text{-}4$$

$$FeCO_{3(s)} + \tfrac{1}{2}H_2O_{(l)} + \tfrac{1}{4}O_{2(l)} = \alpha FeOOH_{(s)} + CO_{2(g)} \qquad\qquad Eqn\ 5.1\text{-}5$$
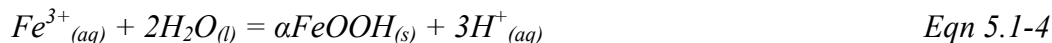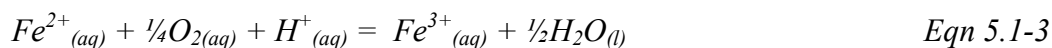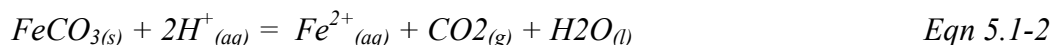
5.1.1.4   Net Balances of Acid-Generating and Acid Neutralizing Potentials

As identified by Morin and Hutt (Appendix A) and explained in 5.1.1.2, total sulphur and sulphide sulphur are closely correlated. As a conservative approach the SAP plus any unaccounted for sulphur is used in determining acid potential (AP). NPs were also previously discussed in 5.1.1.3, and the unavailable NP was determined to be 13 kg/t for Morrison pit rock. This value should be subtracted from the measured value for bulk Sobek NP.

Net balances of these two potentials were calculated by using division (sulphide-based NPR, SNPR = adjusted NP/SAP). Net balances were used to predict if a sample would be net acid-generating after a long near-neutral lag time or if the rock is net acid-neutralizing indefinitely.

SNPR is the preferred net balance provincially; as such it was used in this case. The adjusted SNPR values were calculated by subtracting 13 kg/t of unavailable NP from the measured NP:

$$Adj\ SNPR = \frac{[NP - 13]}{[\%S(Sulphide + Positive\ Del\ S\ Values) * 31.25]} \qquad Eqn\ 5.1\text{-}6$$

The non-site specific ABA screening criteria within BC are: SNPR>2 is not-PAG, 1≤SNPR≤2 uncertain and requires further testing, and SNPR <1 is PAG. For the purpose of this report, any adjusted SNPR value less than 2 was considered PAG. Figure 5.1-8 shows the adjusted SNPRs versus paste pH for the different types of materials with respect to the BC SNPR. Results show adjusted SNPR values ranged from the default of 0.001, meaning NP was less than the unavailable value of 13 kg/t, to a maximum default of 200, meaning sulphide was less than the detection limit of 0.01%S. All materials span one or more provincial criteria except for tailings samples, which are classified as PAG. The overburden samples with adjusted SNPRs greater than the default value of 0.001 and less than 2 are samples from DDH cores, previously identified in Section 5.1.1.2 and Table 5.1-1 requiring lithology confirmation. Figure 5.1-8 shows an overburden sample plotting in the theoretically impossible lower right quadrant. Further investigations into this one sample show that the NP is 5 kg/t, sulphide sulphur is below
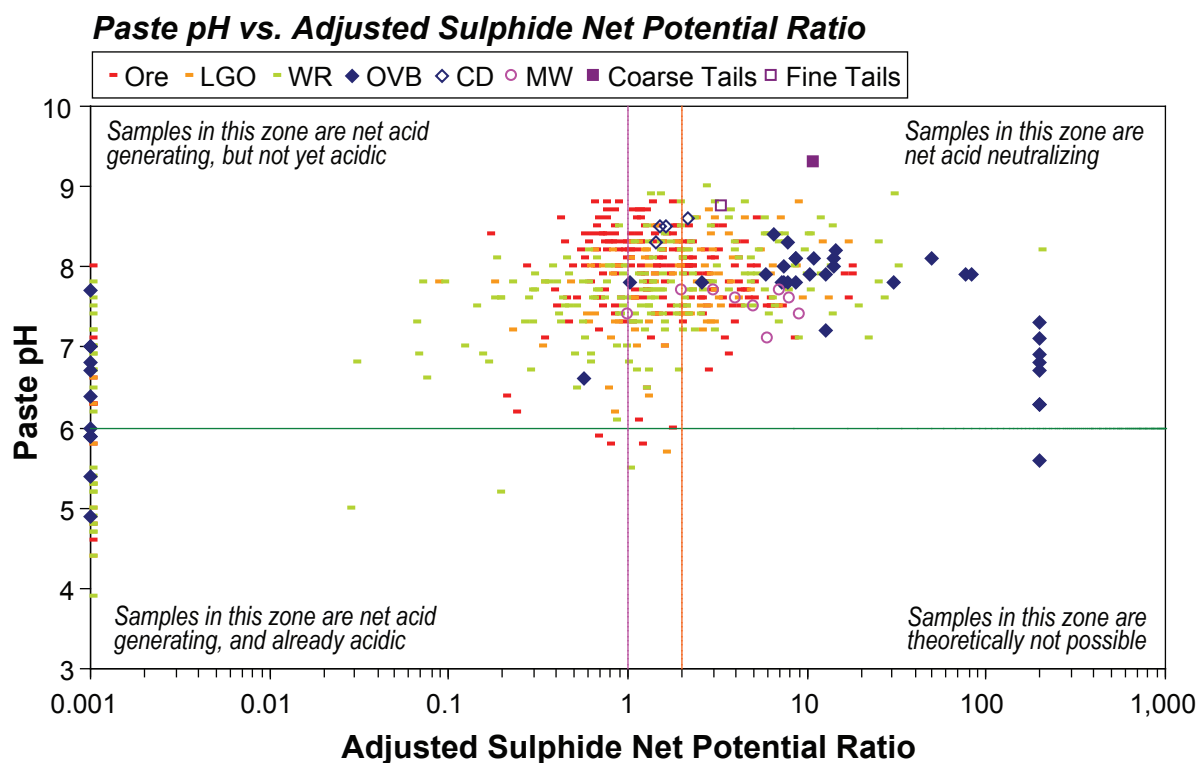
Morrison Copper/Gold Project
Adjusted Sulphide-based
Net Potential Ratios vs Paste pH

FIGURE 5.1-8

the MDL and that the pH of water used in the paste pH test was 5.6, which indicates the placement in this sample in this quadrant is caused by analytical error. Four samples plotted in the lower left quadrant indicating samples are PAG and already producing acidity indicating some sulphide oxidation had occurred by the time laboratory testing was initiated.

Figures 5.1-9 and 5.1-10 show the distribution and proportion of provincial SNPR criteria with respect to ore grade in the ML/ARD prediction sample set collected to date. Results show that although waste rock samples appear to have more PAG samples, the overall proportion of PAG and uncertain samples, here treated together as PAG until further work on the uncertain portion of waste rock samples confirms otherwise, are approximately the same for waste rock and ore samples (i.e., 70% of waste rock and ore samples have adjusted SNPRs less than 2). Approximately 55% of low-grade ore samples have adjusted SNPR values less than 2, although the sample set is smaller by a factor of two when compared to the waste rock and ore sample sets.
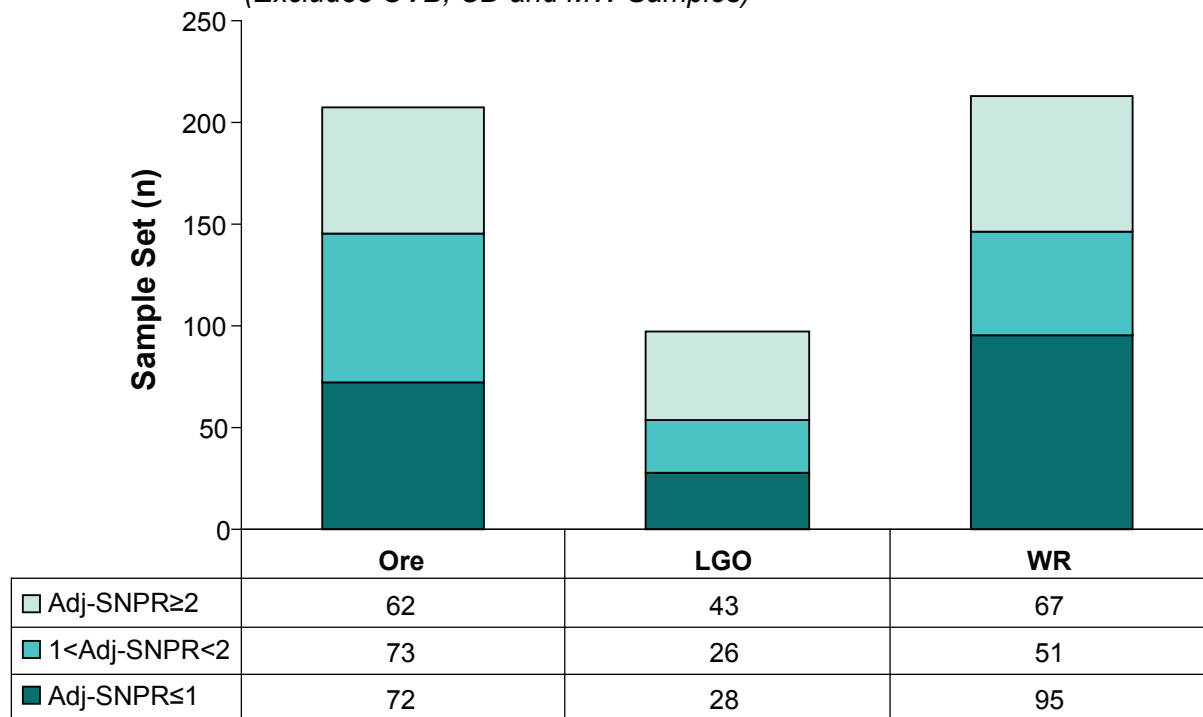
Similar analyses were performed for the different lithologies and alterations (Figures 5.1-11 to 5.1-14). In view of the conservative approach to classifying mine materials discussed above and based on the ML/ARD prediction sample set collected to date, the SED lithology (sedimentary units) is comprised of approximately 80% PAG rock while the BFP lithology is comprised of approximately 60% PAG rock. Note that the overburden samples show a significant PAG percentage, but most of these samples have very low sulphide sulphur values at or near the MDL (0.01%). Similar comparisons among the different alterations indicate the proportion of PAG materials ranges between 50% and 80%.

5.1.2   Total Element Analysis

Morin and Hutt (Appendix A) found that the Morrison deposit rock samples had two dominant elements overall, silicon and aluminum, which reflects the presence of quartz and aluminosilicate minerals. In most rock samples iron, calcium, magnesium, sodium, and potassium represented up to a few percent of the sample.

Given the deposit sulphide mineralogy described in Section 2.4 and in consideration of the toxicity of other metal(loids) at low concentrations in the aqueous phase, elements of interest include copper, molybdenum, zinc, lead, arsenic, antimony, cadmium, and selenium and are shown in Figures 5.1-15 to 5.1-22. Metal(loids) are compared against adjusted SNPRs and 10 times whole crustal abundance (Price 1997) to appreciate the magnitude of elemental reservoirs potentially available for leaching during material weathering under different drainage pH conditions. The overburden samples have been identified as having low sulphide content, but high copper concentrations were measured in overburden samples collected from within the pit outline comparable to tailings, waste rock, and low-grade ore copper concentrations. Note that these overburden samples include shallow test pitting, as well as DDH samples from resource drilling identified in Section 5.1.1.2 Table 5.1-1, which indicates incorporation of underlying copper sulphide minerals from bedrock is unlikely, but has not been confirmed to date. No clear copper correlations with other elements in the overburden were noted. Identifying the host phase, which is likely a secondary oxide as overburden has been weathered during transport, would be valuable in determining the stability of phase and has implications for suitability of

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Adjusted (-13 NP) SNPR for Pit Rock Samples
(Excludes OVB, CD and MW Samples)

	Ore	LGO	WR
☐ Adj-SNPR≥2	62	43	67
☐ 1<Adj-SNPR<2	73	26	51
☐ Adj-SNPR≤1	72	28	95

Morrison Copper/Gold Project
Distribution of Provincial Adjusted SNPR
Categories Based on Grade

FIGURE 5.1-9

pbm

Rescan™

Adjusted (-13 NP) SNPR for Pit Rock Samples
(Excludes OVB, CD and MW Samples)

	Ore	LGO	WR
☐ Adj-SNPR≥2	62	43	67
☐ 1<Adj-SNPR<2	73	26	51
☐ Adj-SNPR≤1	72	28	95
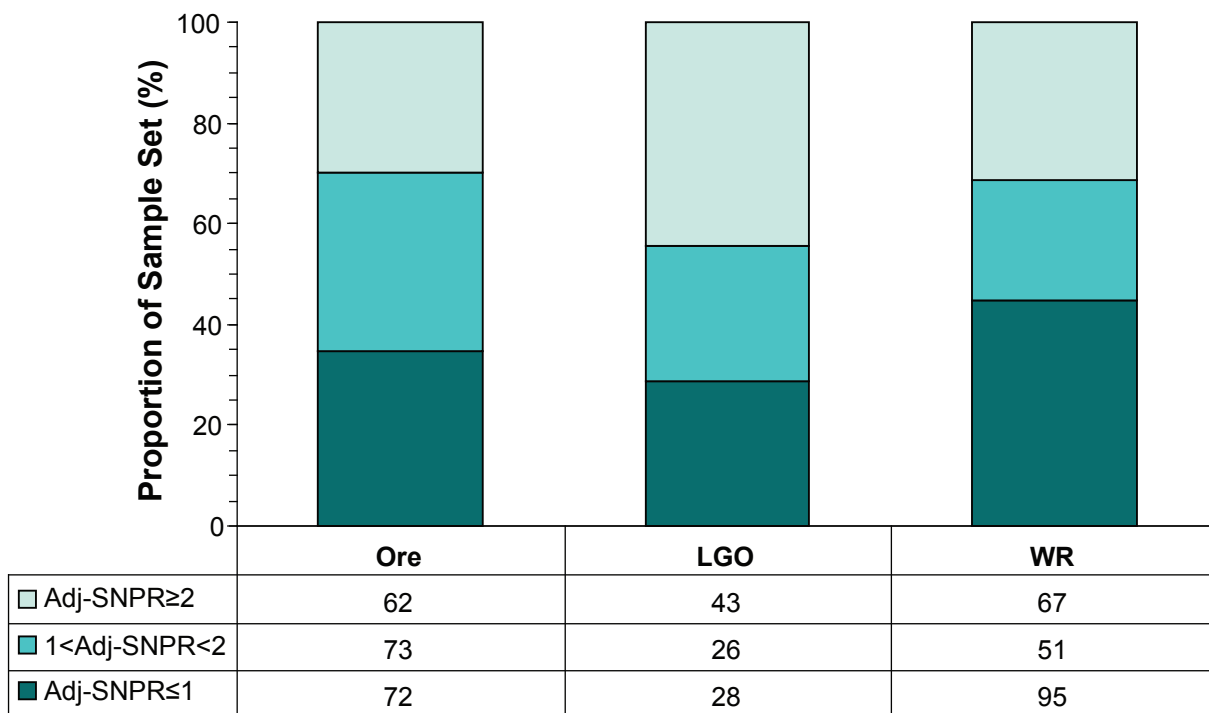
Morrison Copper/Gold Project
Proportions of Provincial Adjusted SNPR
Categories Based on Grade

FIGURE 5.1-10

Adjusted (-13 NP) SNPR for Lithology

	BFP	FLT	SED	OVB
☐ Adj-SNPR≥2	132	4	36	30
☐ 1<Adj-SNPR<2	101	1	48	1
■ Adj-SNPR≤1	89	7	99	12

Morrison Copper/Gold Project
Distribution of Provincial Adjusted SNPR
Categories Based on Lithology

FIGURE 5.1-11

Adjusted (-13 NP) SNPR for Lithology

	BFP	FLT	SED	OVB
☐ Adj-SNPR≥2	132	4	36	30
☐ 1<Adj-SNPR<2	101	1	48	1
■ Adj-SNPR≤1	89	7	99	12

Morrison Copper/Gold Project
Proportion of Provincial Adjusted SNPR
Categories Based on Lithology

FIGURE 5.1-12

Adjusted (-13 NP) SNPR for Alteration

	ARCB	ARSE	KH	KL	QZSE
Adj-SNPR≥2	1	54	52	27	25
1<Adj-SNPR<2	2	25	53	14	35
Adj-SNPR≤1	1	24	42	11	78

Morrison Copper/Gold Project
Distribution of Provincial Adjusted SNPR
Categories Based on Alteration

FIGURE 5.1-13

Adjusted (-13 NP) SNPR for Alteration

	ARCB	ARSE	KH	KL	QZSE
☐ Adj-SNPR≥2	1	54	52	27	25
☐ 1<Adj-SNPR<2	2	25	53	14	35
■ Adj-SNPR≤1	1	24	42	11	78

Morrison Copper/Gold Project
Proportion of Provincial Adjusted SNPR
Categories Based on Alteration

FIGURE 5.1-14

Rescan™

Note: Any value less than detection is shown at one-half detection

FIGURE 5.1-15

Morrison Copper/Gold Project
Copper Concentrations vs Adjusted SNPR

Solid-Phase Molybdenum vs. Adjusted SNPRs

Legend:
- Ore
- LGO
- WR
- OVB
- CD
- MW
- Coarse Tails
- Fine Tails
- SNPR = 1
- SNPR = 2
- 10x Whole Crustal Abundance

X-axis: **Adjusted SNPRs**
Y-axis: **Molybdenum (ppm)**

Note: Any value less than detection is shown at one-half detection

Morrison Copper/Gold Project
Molybdenum Concentrations vs Adjusted SNPR

FIGURE 5.1-16

Solid-Phase Zinc vs. Adjusted SNPRs

Note: Any value less than detection is shown at one-half detection

Morrison Copper/Gold Project
Zinc Concentrations vs Adjusted SNPR

FIGURE 5.1-17

Solid-Phase Lead vs. Adjusted SNPRs

Lead (ppm) vs Adjusted SNPRs

Legend:
- Ore
- LGO
- WR
- OVB
- CD
- MW
- Coarse Tails
- Fine Tails
- SNPR = 1
- SNPR = 2
- 10x Whole Crustal Abundance

Note: Any value less than detection is shown at one-half detection

FIGURE 5.1-18

Morrison Copper/Gold Project
Lead Concentrations vs Adjusted SNPR

Solid-Phase Arsenic vs. Adjusted SNPRs

Note: Any value less than detection is shown at one-half detection

FIGURE 5.1-19

Morrison Copper/Gold Project
Arsenic Concentrations vs Adjusted SNPR

**Morrison Copper/Gold Project
Antimony Concentrations vs Adjusted SNPR**

FIGURE 5.1-20

Solid-Phase Cadmium vs. Adjusted SNPRs

Note: Any value less than detection is shown at one-half detection

Morrison Copper/Gold Project
Cadmium Concentrations vs Adjusted SNPR

FIGURE 5.1-21

Morrison Copper/Gold Project
Selenium Concentrations vs Adjusted SNPR

FIGURE 5.1-22

construction materials and mine waste management. Note that Figure 5.1.15 shows the ML/ARD solid-phase ICP copper data on the y-axis. Morrison rock waste, low-grade ore and ore was classified using the master DDH database assay data to be consistent with previous resource estimates. Figure 5.1-15 shows the heterogeneity in the solid-phase distribution of elements in core samples as some waste and low-grade ore samples plot above the 0.2% and 0.3% copper values.

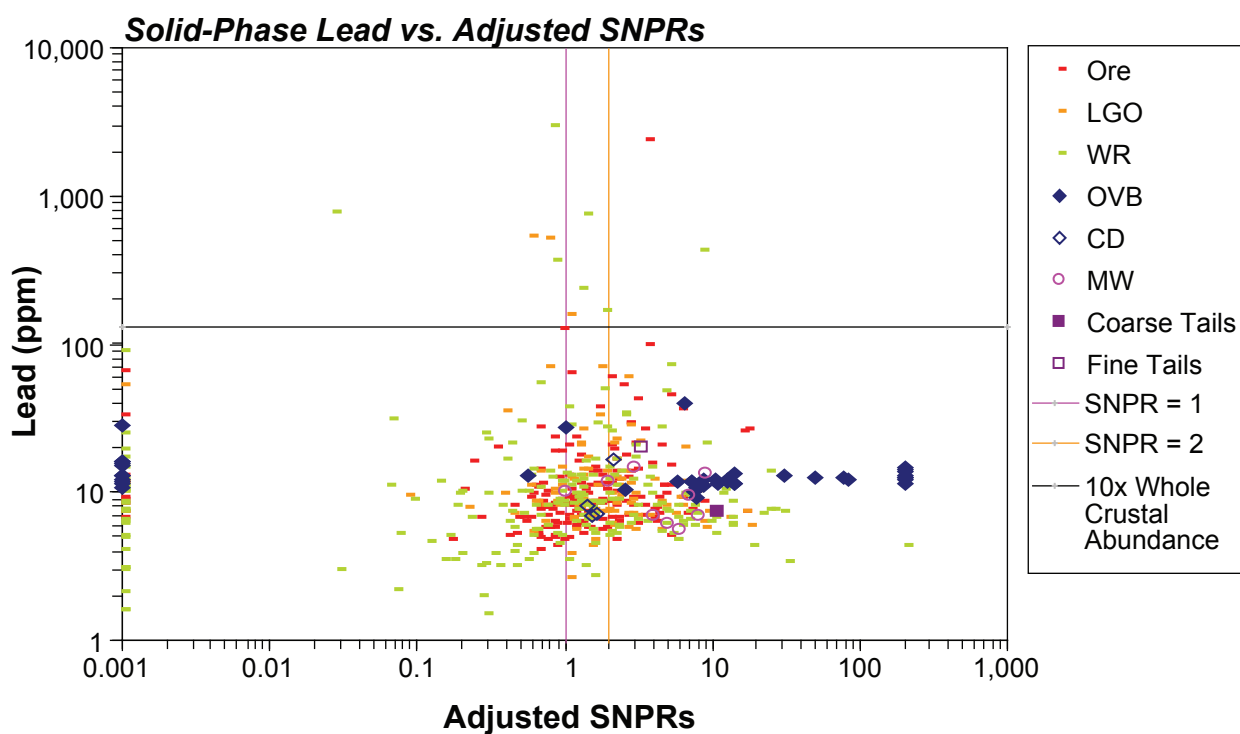Preliminary estimates of sediment volumes from Booker Lake and Ore Pond for relocation and storage purposes are 600,000 m^3 and 50,000 m^3, respectively (Klohn Crippen Berger 2008). Figure 5.1-23 shows the measured metal(loid) values of sediment samples collected to date. Comparisons against available sediment quality guidelines indicate several elements exceeding including arsenic, cadmium, chromium, copper, iron, mercury, nickel, and zinc, not unexpected given the vicinity of the Morrison deposit. The *in situ* redox conditions of lake sediments, and hence the likelihood of authigenic metal(loid)-bearing sulphides, is unknown at this time. However, it is likely that the sediments are reducing in nature, except for the possible presence of an oxic surficial veneer, as they have been decoupled from atmosphere. Draining these lakes and ponds, excavation, and relocation may oxid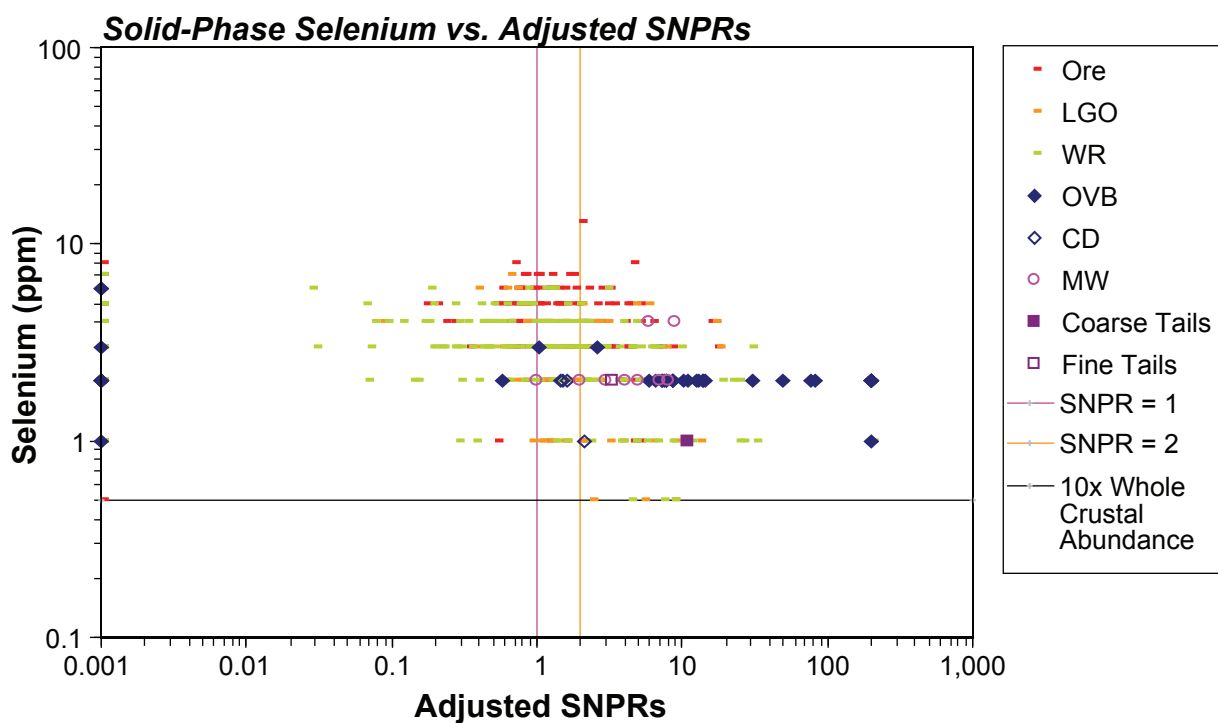ize these sediments if they are at reducing conditions *in situ*, much like the sulphide at depth in the Morrison deposit, and may cause ML under either oxic neutral conditions initially and possibly oxic acidic conditions thereafter, if neutralization is inadequate.

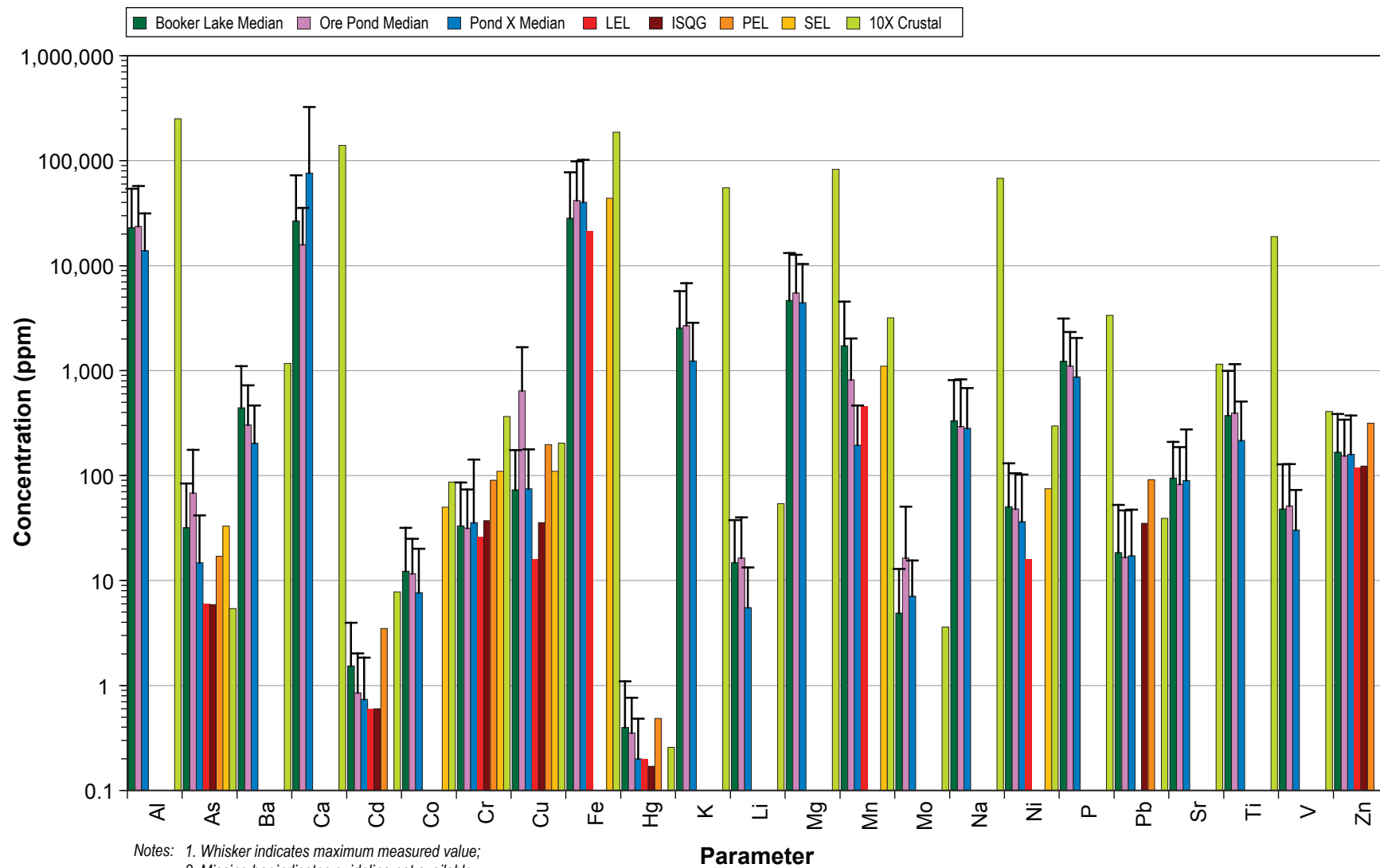## 5.1.3   Shake Flask Extraction Tests

The shake flask extraction (SFE) tests were conducted to determine the composition of the leachate that may flush from overburden samples when exposed to rain or groundwater discharge. This procedure, a BC MEM or modified SWEP leachate extraction, is used to determine the presence of easily soluble mineral components.

Note that the aggressive nature of the SFE test (24-hr end-over-end agitation) does not mimic typical field weathering and oxidation conditions. Instead, the SFE test is designed to maximize sample exposure to the extraction fluid, therefore maximizing particle surface coverage and reactivity. The SFE test also decreases solubility limits by using a 3:1 liquid-to-solid extraction ratio. Generally, the test results are considered to represent possible leachate concentrations for freshly exposed rock under the measured pH conditions. If samples had undergone previous weathering before testing, then testing would generally indicate the presence of soluble secondary weathering products under the measured pH conditions. Therefore, the SFE test is more appropriately used as a screening tool to determine which parameters may be of potential concern and which solid samples should undergo further testing.

SFEs were conducted on overburden samples collected outside the pit outline (Figures 5.1-24 and 5.1-25). Results show that overburden sample leachate pH values ranged from near-neutral (pH = 6.1) to slightly alkaline (pH = 8.5). The pH values are directly correlated to released alkalinity and inversely correlated to acidity, as would be expected. Dissolved calcium, magnesium, and sulphate concentrations all correlate directly with alkalinity, indicative of calcium- and magnesium-bearing carbonate minerals (i.e., calcite and dolomite) and dissolution of gypsum or anhydrite. Note that if

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Lake Sediment Metals Content

Legend: Booker Lake Median, Ore Pond Median, Pond X Median, LEL, ISQG, PEL, SEL, 10X Crustal

Y-axis: Concentration (ppm)

X-axis: Parameter — Al, As, Ba, Ca, Cd, Co, Cr, Cu, Fe, Hg, K, Li, Mg, Mn, Mo, Na, Ni, P, Pb, Sr, Ti, V, Zn

Notes: 1. Whisker indicates maximum measured value;
        2. Missing bar indicates guideline not available
        3. LEL = Lowest Effects Level;
        4. IQSG = Interim Quality Sediment Guideline;
        5. PEL = Probable Effects Level; and
        6. SEL = Severe Effects Level.

FIGURE 5.1-23

Morrison Copper/Gold Project
Booker Lake, Ore Pond and Pond X Elemental Abundance

Overburden Samples Outside Pit Area

Note: Any value less than detection is shown at one-half detection
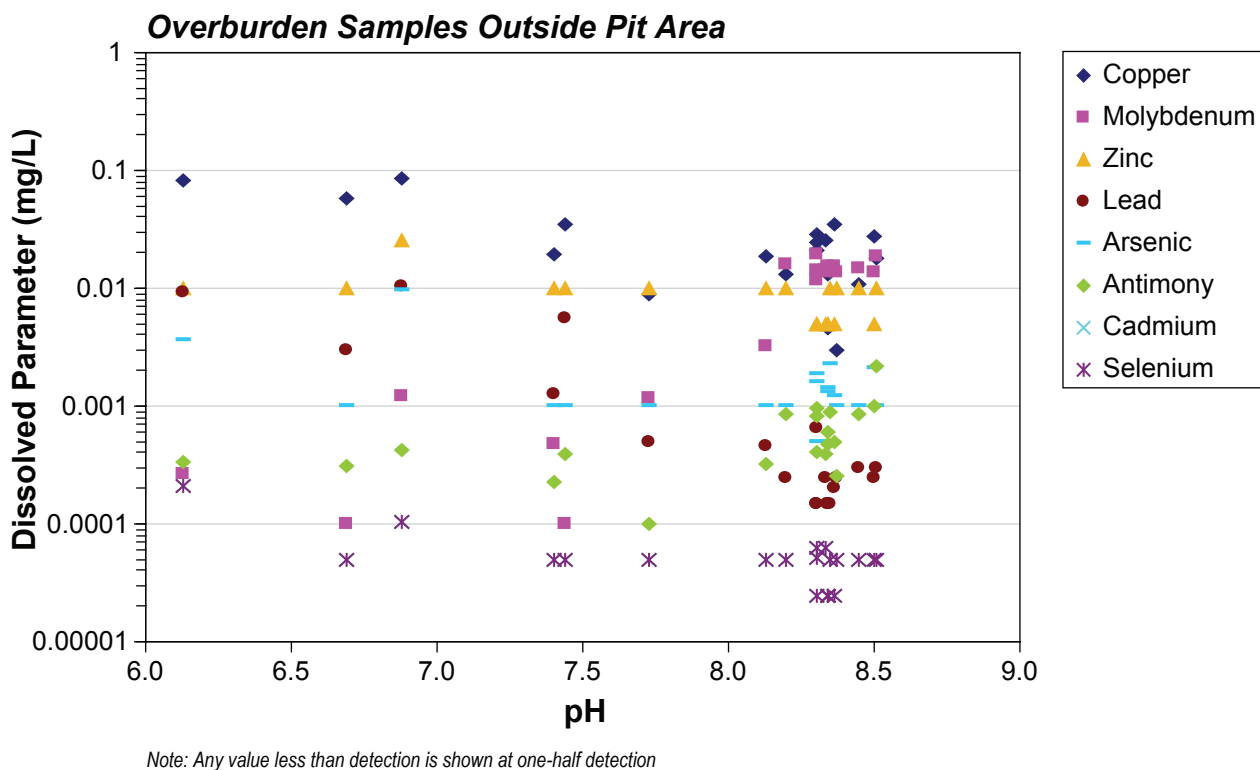
Morrison Copper/Gold Project
Overburden Shake Flask Extraction Results
for General Parameters and Major Ions

FIGURE 5.1-24

Overburden Samples Outside Pit Area

Note: Any value less than detection is shown at one-half detection

Morrison Copper/Gold Project
Overburden Shake Flask Extraction Results
for Dissolved Metal(loids)

FIGURE 5.1-25

sulphide oxidation in overburden samples was occurring, sulphate trends would be inversely related to pH. Other notable major ion trends include increasing aluminum and iron with decreasing pH, well understood solubility behaviour of hydroxides, the likely host phases in weathered overburden mineral phases (Stumm and Morgan 1996).

The leachability of other trace metal(loid) were low but show trends of interest (Figure 5.1-25) and include inverse correlations between copper-lead and pH and direct correlations between molybdenum and pH. The mineralogy of the overburden samples is not well known at this time, which limits a comprehensive and detailed interpretation of host-phase behaviour under the SFE conditions. However, the SFE results do identify potential elements of interest that are readily leachable under neutral pH conditions and therefore should be evaluated further by future kinetic test work.

5.1.4   Mineralogy

Mineralogy (petrographic and X-Ray Diffraction) data collected to date have concentrated on the characterization of humidity cell materials and has been described by Morin and Hutt (Appendix A). Table 5.1-3 summarizes the mineral phases documented in humidity cells currently in operation and discussed in Section 5.2.1. Interestingly, no arsenic-bearing minerals were detected even though the total metal content shows relatively high arsenic concentrations in pit rock (Figure 5.1-19) and literature reports of arsenopyrite (Section 2.4)
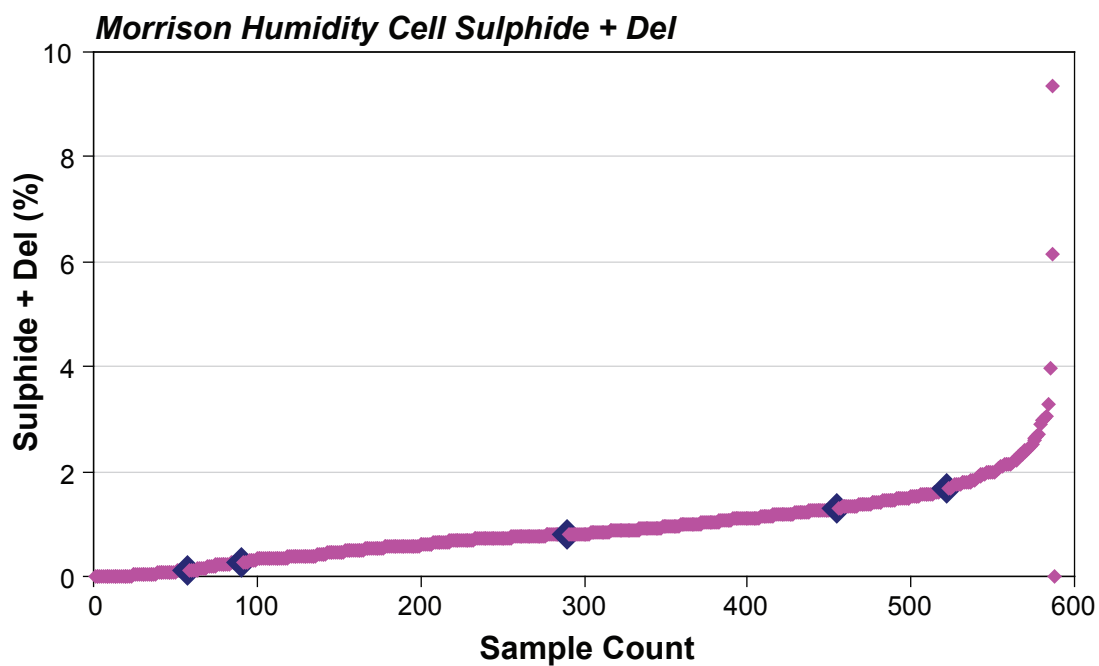
5.2   Kinetic Testing

5.2.1   Laboratory Scale Testing

5.2.1.1   Morrison Rock Humidity Cells

Five humidity cells were set up with discrete samples of Morrison deposit rock drillcore. Figures 5.2-1 and 5.2-2 show the distribution of sulphide sulphur + del and copper equivalent among the ML/ARD database. Indicated humidity cells cover the ranges for waste rock (< CuEq < 0.2%) and low grade ore (0.2 < CuEq < 0.3). Additional distributions of NP and SNPR are presented in Morin and Hutt (Appendix A). These also show a good breadth of representation within the ML/ARD prediction database.

An initial assessment of Morrison deposit rock humidity cell leachate results up to week 50 was provided by Morin and Hutt (Appendix A). It was determined by Morin and Hutt that by week 40 sulphate production levels, representing sulphide oxidation and total-acidity generation, had stabilized. A comparison done by Morin and Hutt (Appendix A) between the nearby Bell Mine and Morrison rock humidity cell data showed that sulphate production rates and trends were similar (i.e., ~5 to 100 mg sulphate per kg).

The Morrison deposit rock humidity cell has remained in operation but the humidity cell leachate has been sampled less frequently (i.e., approximately 8 weeks for general parameters and approximately 12 weeks for metals). Additional data up to week 168 are presented below. Note that loading rates are not presented here but can be visually estimated as the leachate flushed is

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Morrison Humidity Cell Sulphide + Del

Morrison Copper/Gold Project
Morrison Rock Humidity Cell Sulphide +
Del Distribution in ML/ARD Database

FIGURE 5.2-1
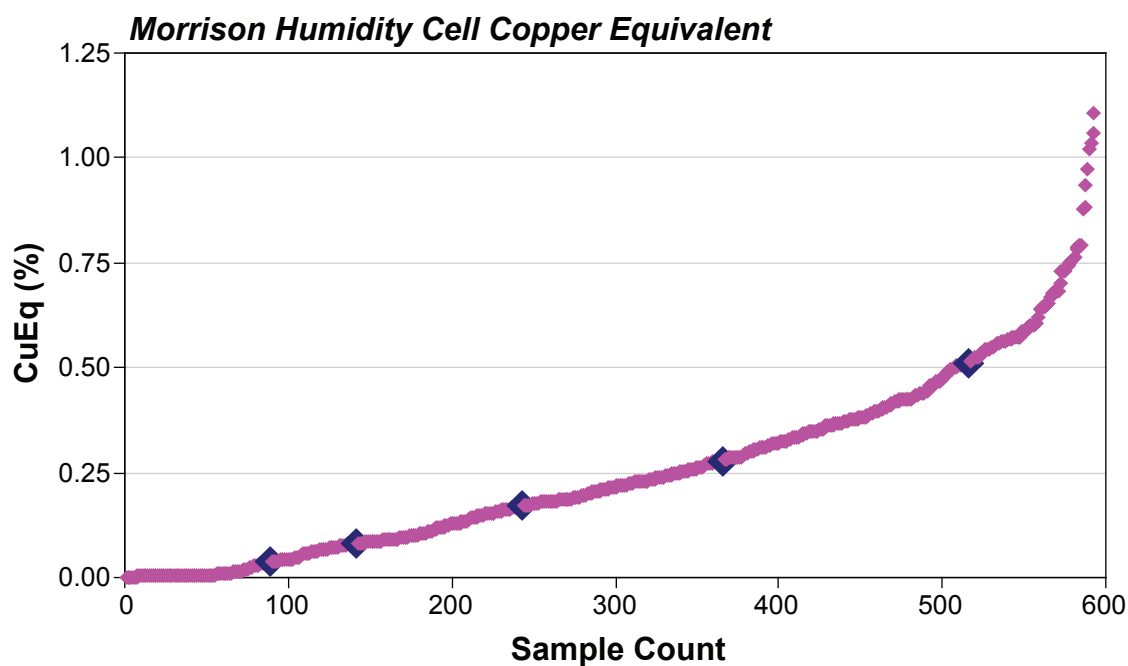
Morrison Copper/Gold Project
Morrison Rock Humidity Cell Copper
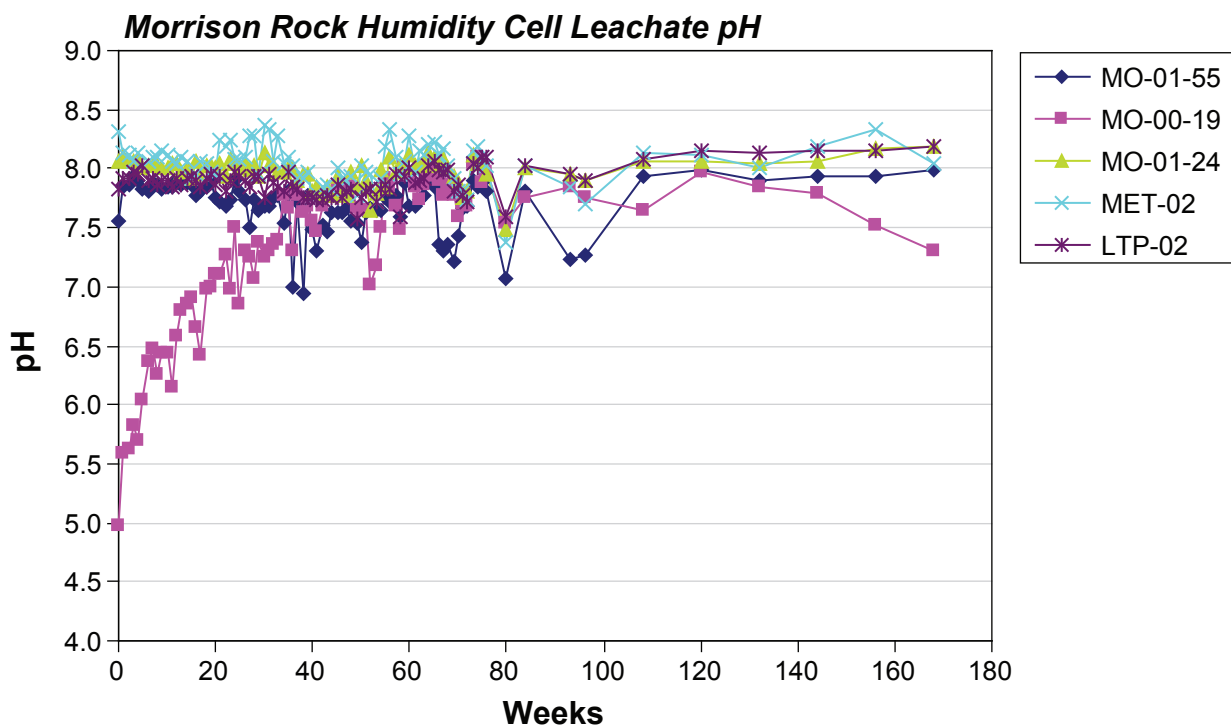**Equivalent Distribution in ML/ARD Database**

FIGURE 5.2-2

consistently approximately 0.5 L per flushing event and humidity cell were constructed with 1 kg of pit rock resulting in loading rates in mg/kg of approximately one half the measured concentrations presented below. The entire results are presented in Appendix B.

Humidity cell leachate pH has remained near-neutral to slightly alkaline and shown little temporal variability (Figure 5.2-3). A notable trend is the recent decrease in pH of humidity cell sample MO-00-19. This pH decrease is concomitant with an increase in sulphate (Figure 5.2-4), but without a notable release of acidity (Figure 5.2-5). Additional parameter trends show a decrease in alkalinity (Figure 5.2-6), an increase in dissolved calcium (Figure 5.2-7) with dissolved iron below the MDLs (not shown). These results support the conclusion that the pH decrease is caused by a lack of alkalinity production and the increase in sulphate production is caused by anhydrite dissolution (see Section 5.1.4, Table 5.1-3). This indicates that initial humidity cell flushing of sample MO-00-19 did not remove all leachable sulphate, but that sulphate-bearing minerals persisted until humidity cell flow paths shifted or specific minerals were exposed to new flow paths within the humidity cell. Laboratory methodology indicates humidity cells were gently agitated during weekly flushing events (Appendix A), however a comparison of water added and recovered typically showed samples retained some water each week indicating samples were not saturated during weekly cycles. This also indicates that relatively soluble minerals, such as anhydrite, could persist. Alternatively, the decrease in pH, increase in sulphate, increase in dissolved calcium and decrease in alkalinity as it is used to buffer the system could be due to the onset of acidic conditions via sulphide oxidation as soluble minerals like anhydrite are expected to be flushed out early in the testing period. Nevertheless, additional humidity cell data is required to confirm that an increase in sulphate is due to anhydrite dissolution versus sulphide oxidation.

Additional parameters of interest in humidity cell leachate include variable but low copper-leaching, recent releases of molybdenum with an increase in trend, moderate, and variable zinc releases concomitant with low cadmium releases, initial flushing of arsenic, antimony, and lead, and increasing barium release. Of particular interest is the near-identical leaching trends for zinc and cadmium, indicating cadmium substitution in zinc-bearing minerals (e.g., sphalerite), which is not uncommon. Figures 5.2-8 to 5.2-15 show dissolved metal results for the humidity cell leachate. Based on laboratory-scale humidity cell testing at the indicated neutral to alkaline pH, the drainage chemistry from Morrison waste rock, low grade ore, and ore is expected to show initial increases in sulphate, copper, zinc, cadmium, arsenic, antimony, and lead with possible later stage releases of molybdenum and barium. However, humidity cell concentrations cannot be used reliably to predict the magnitude of concentrations in drainage from full scale mine components.

5.2.1.2   Morrison Rock ARD Lag Times

Morin and Hutt (Appendix A) interpreted NP depletion and sulphate production rates to determine lag times to the onset of ARD from 50 weeks of Morrison deposit rock humidity cell data. Since then, pH, NP depletion, and sulphate production rates have not substantially changed. Nevertheless, figures 5.2-16 and 5.2-17 are presented below for an update.

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Morrison Copper/Gold Project
Morrison Rock Humidity Cell
Leachate Results for pH

FIGURE 5.2-3

Morrison Copper/Gold Project
Morrison Rock Humidity Cell
Leachate Results for Sulphate

FIGURE 5.2-4

**Morrison Copper/Gold Project
Morrison Rock Humidity Cell
Leachate Results for Acidity**

FIGURE 5.2-5
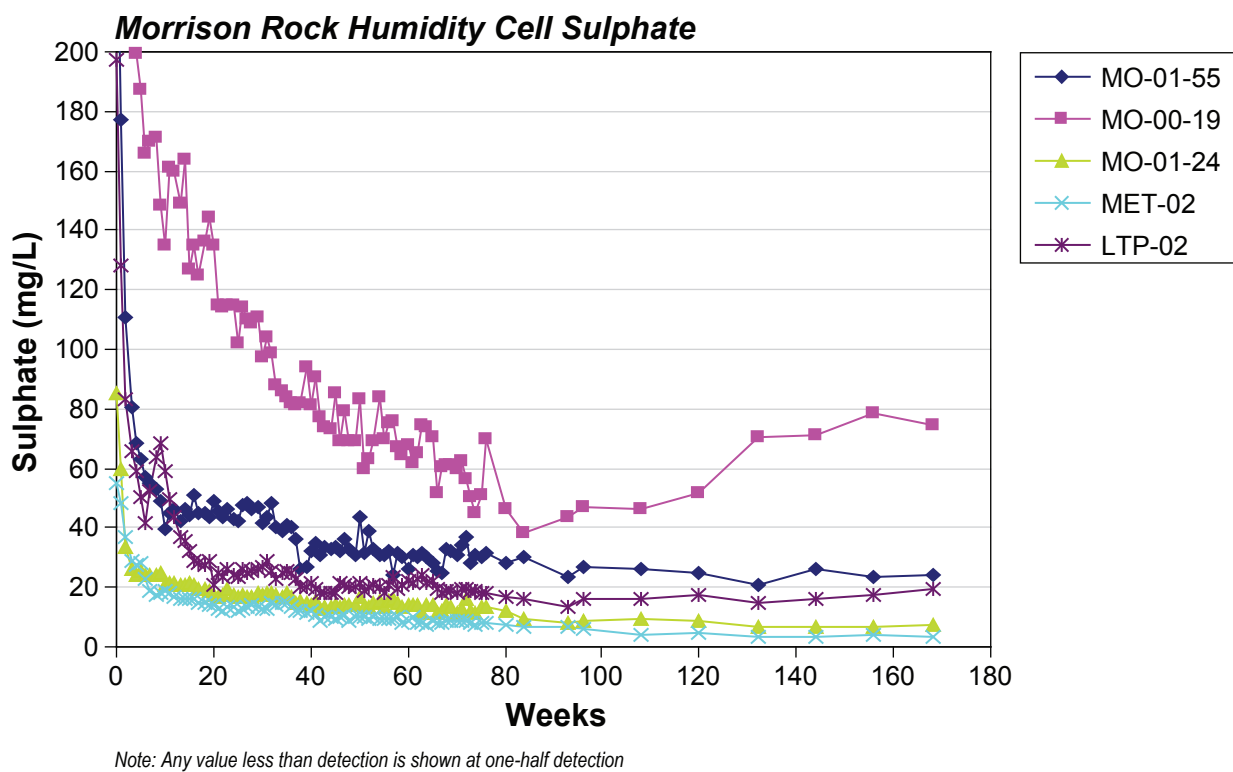
Morrison Copper/Gold Project
Morrison Rock Humidity Cell
Leachate Results for Alkalinity

FIGURE 5.2-6

Morrison Copper/Gold Project
Morrison Rock Humidity Cell Leachate
Results for Dissolved Calcium

FIGURE 5.2-7

Morrison Copper/Gold Project
Morrison Rock Humidity Cell
Leachate Results for Dissolved Copper

FIGURE 5.2-8

Morrison Copper/Gold Project
Morrison Rock Humidity Cell Leachate
Results for Dissolved Molybdenum

FIGURE 5.2-9

Morrison Rock Humidity Cell Leachate Zinc

Note: Any value less than detection is shown at one-half detection

**Morrison Copper/Gold Project
Morrison Rock Humidity Cell
Leachate Results for Dissolved Zinc**

FIGURE 5.2-10

Morrison Copper/Gold Project
Morrison Rock Humidity Cell
Leachate Results for Dissolved Cadmium

FIGURE 5.2-11

Morrison Copper/Gold Project
Morrison Rock Humidity Cell
Leachate Results for Dissolved Lead

FIGURE 5.2-12

Morrison Rock Humidity Cell Leachate Antimony

Note: Any value less than detection is shown at one-half detection

Morrison Copper/Gold Project
Morrison Rock Humidity Cell Leachate
Results for Dissolved Antimony

FIGURE 5.2-13

Morrison Rock Humidity Cell Leachate Arsenic

Legend:
- MO-01-55
- MO-00-19
- MO-01-24
- MET-02
- LTP-02

Y-axis: Dissolved Arsenic (mg/L)
X-axis: Weeks

Note: Any value less than detection is shown at one-half detection

Morrison Copper/Gold Project
Morrison Rock Humidity Cell Leachate
Results for Dissolved Arsenic

FIGURE 5.2-14

pbm

Rescan™

Note: Any value less than detection is shown at one-half detection
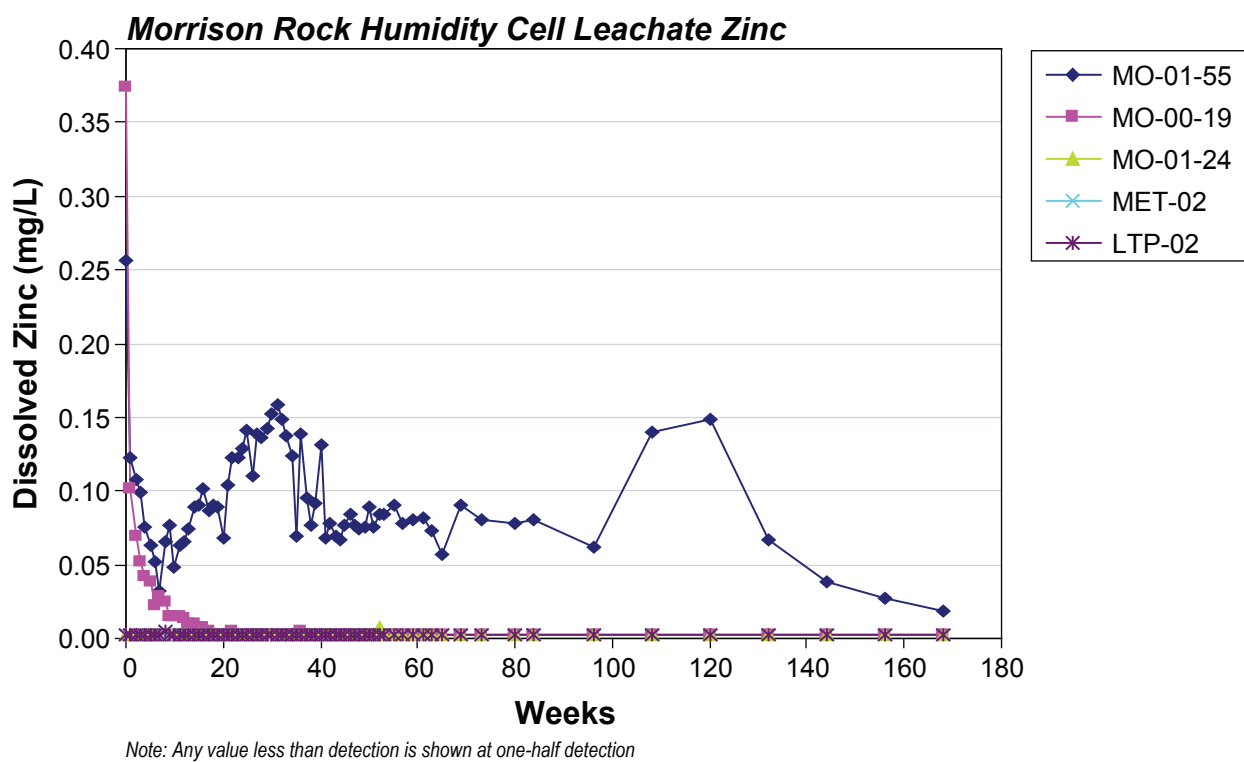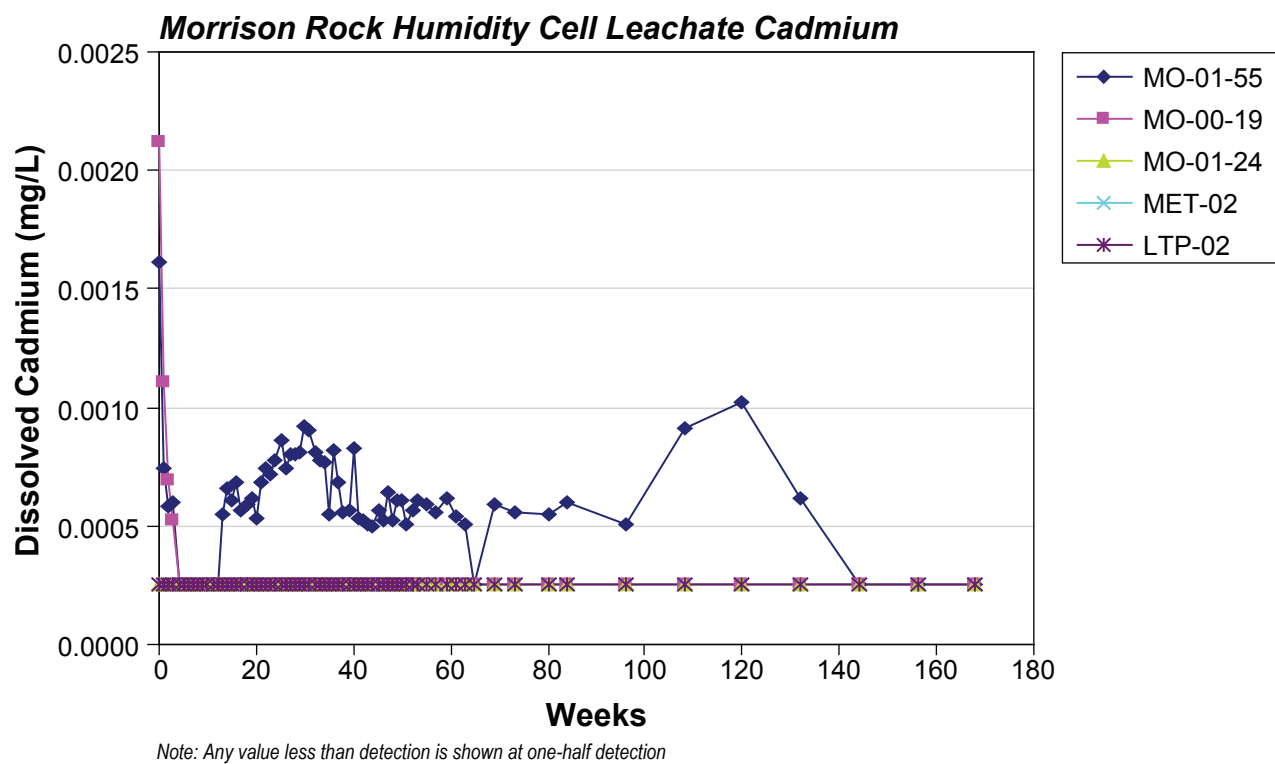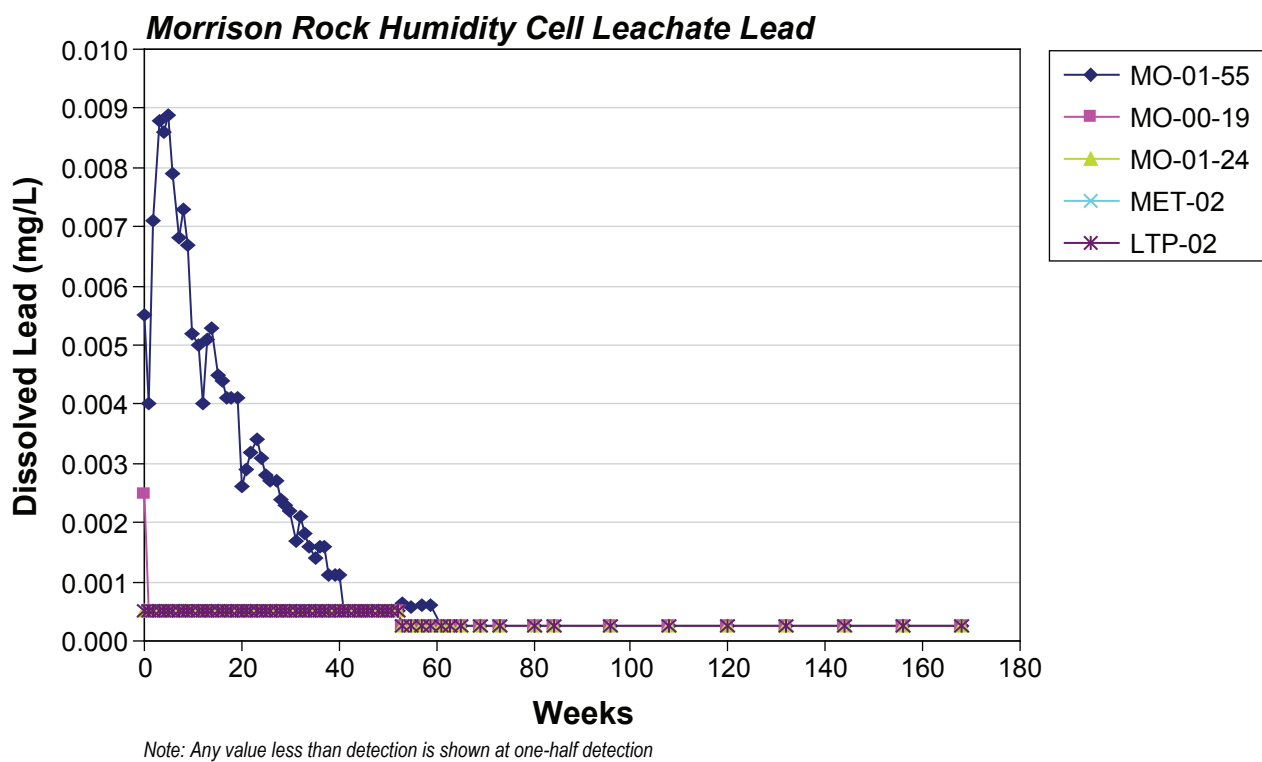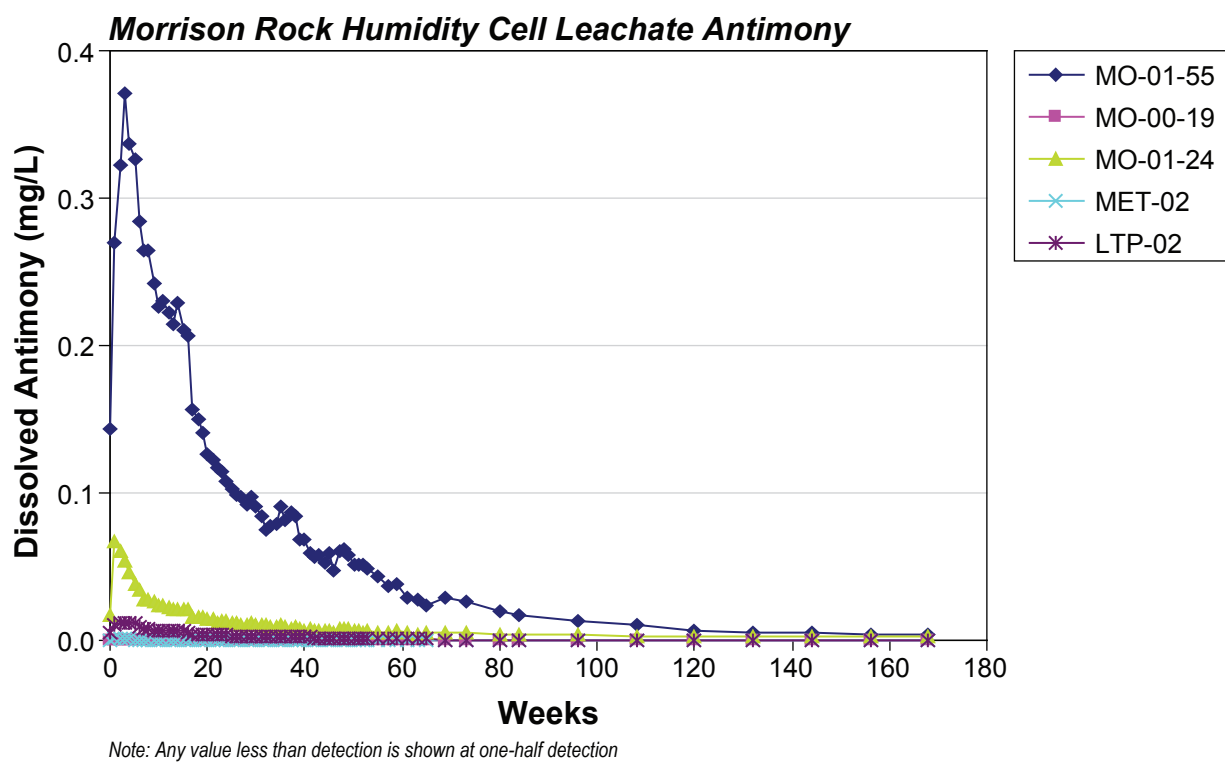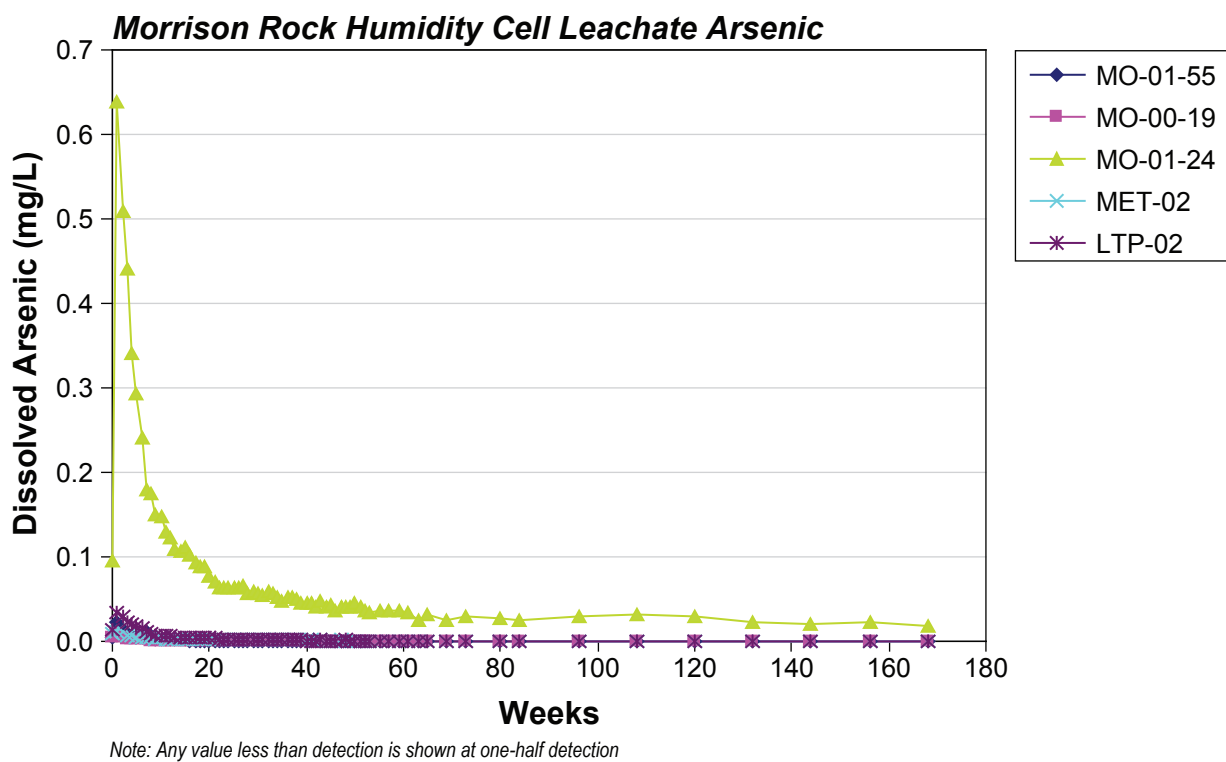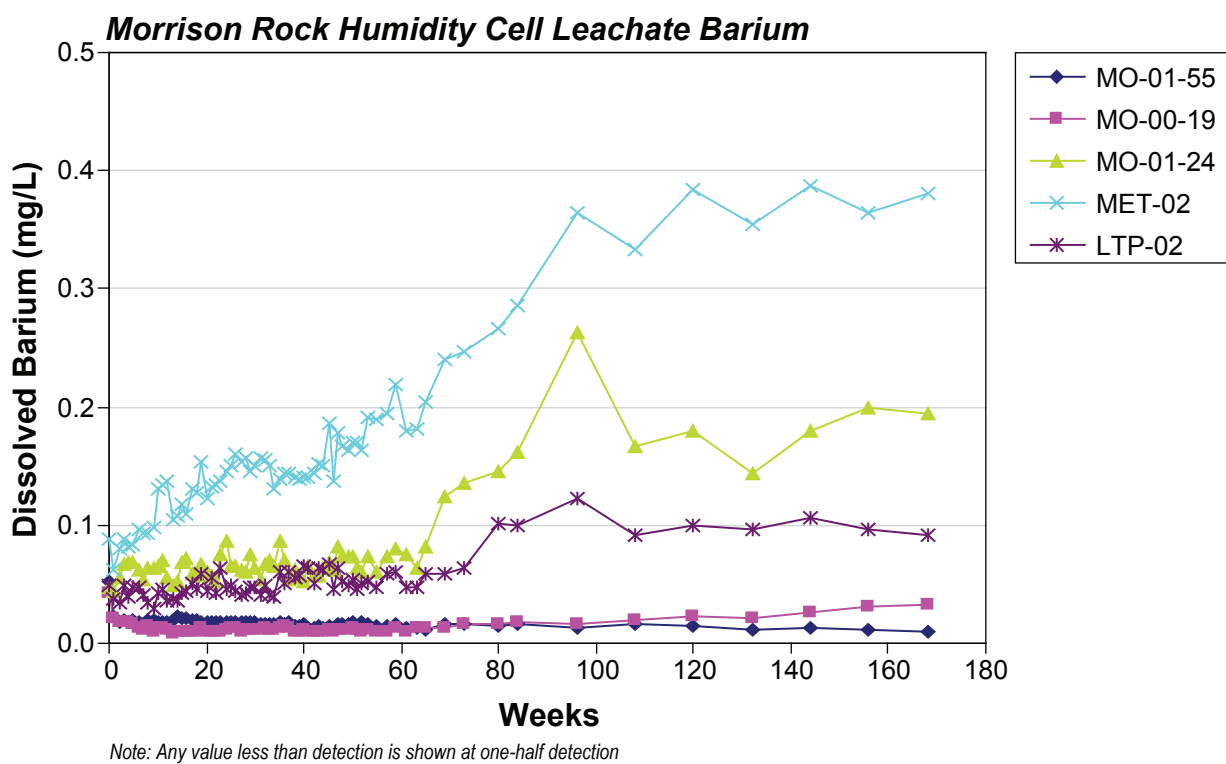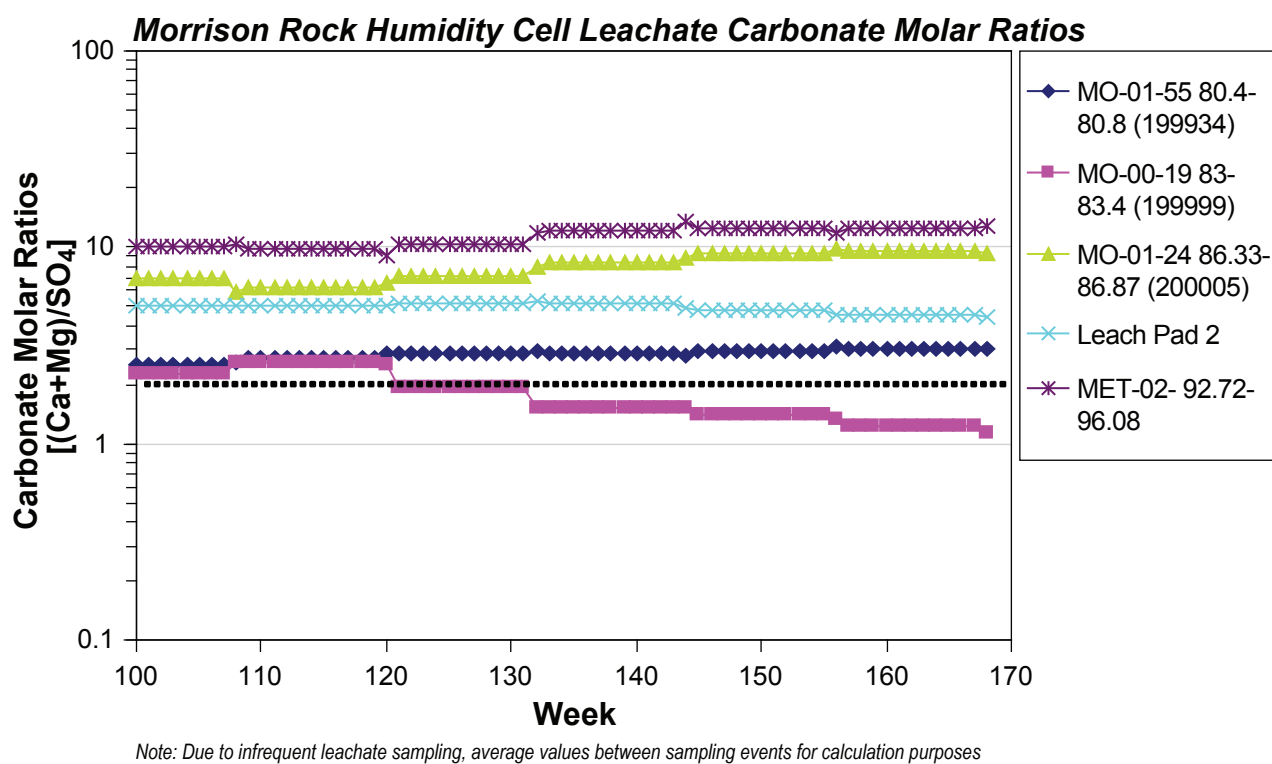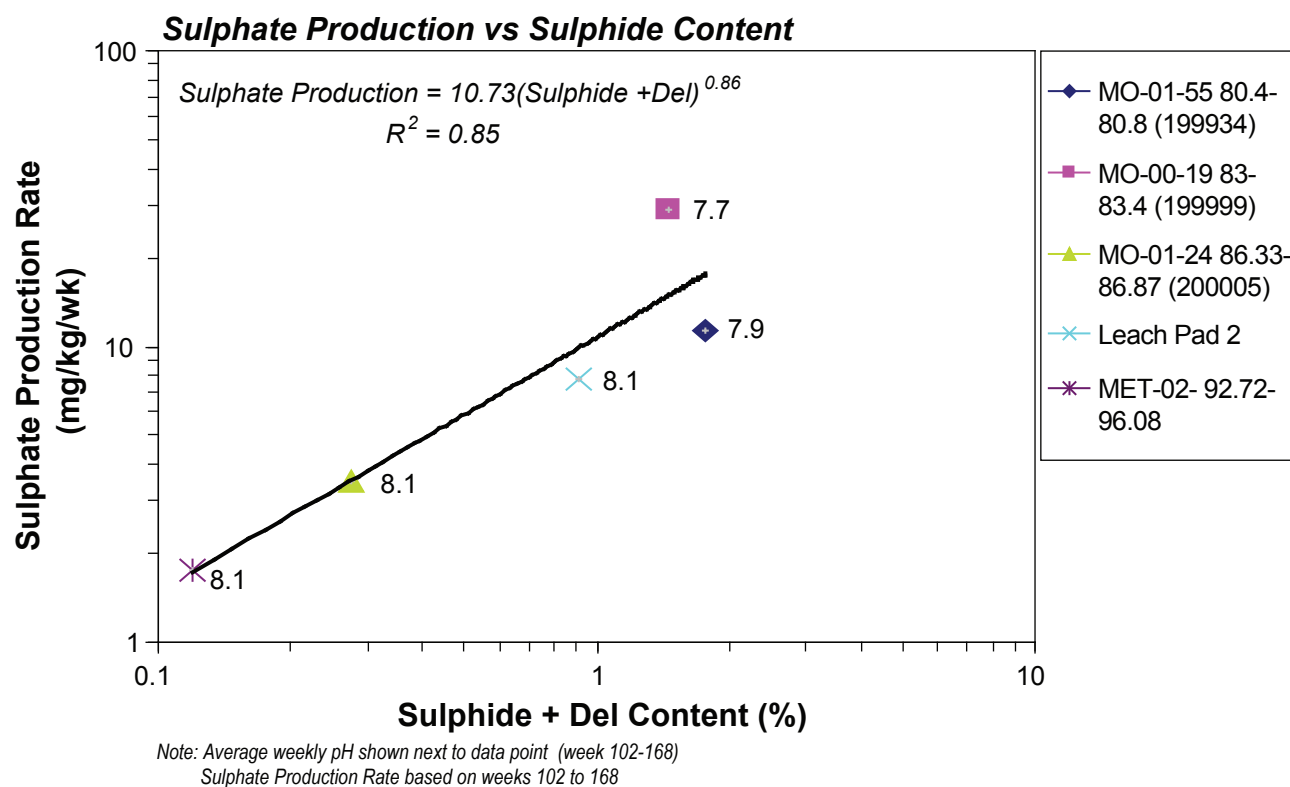
Morrison Copper/Gold Project
Morrison Rock Humidity Cell Leachate
Results for Dissolved Barium

FIGURE 5.2-15

Morrison Copper/Gold Project
Carbonate Molar Ratios for
Morrison Humidity Cell Rock

FIGURE 5.2-16

Morrison Copper/Gold Project
Sulphate Productions Rates vs Sulphide + Del
Content for Morrison Humidity Cell Rock

FIGURE 5.2-17

Carbonate molar ratios (CMRs; the molar ratio of calcium plus magnesium production rates divided by the sulphate production rate) provide insight into the rate of NP consumption relative to the sulphate production rate and are theoretically between 1.0 and 2.0 when sulphate oxidation drives carbonate NP consumption (Morin and Hutt 2006). Figure 5.2-16 shows that for the last 66[4] weeks of humidity cell leachate data, CMRs have remained steady with a slight decrease for MO-00-19 and that CMRs in general have remained above 2.0. This indicates that humidity cell flushing results in excess carbonate NP dissolution not attributed to sulphide oxidation. The presence of carbonates in all humidity cells except for MO-00-19 is confirmed by mineralogical data, therefore SNPR criteria are capped at the theoretical value of 2.0. Under full-scale on-site conditions, exposure to such high water to solid ratios would not be expected for Morrison rock. This implies that more NP will be available for sulphide oxidation buffering in the field at full mine scale, but this needs to be confirmed with the larger scale LTP kinetic data.

A relationship can be developed for average sulphate production rates versus sulphide + del (Figure 5.2-17), that predicts the rate of sulphate production according to the sulphide content of materials for the leachate pHs observed in the last 66 weeks of testing (Equation 5.2-1). Results are consistent with higher sulphide materials having higher sulphate production rates.

$$Sulphate\ Production = 10.73(Sulphide + Del)^{0.86} \qquad Eqn\ 5.2\text{-}1$$

A similar relationship was developed for average calcium plus magnesium production rates versus sulphide + del (Figure 5.2-18), that predicts the rate of calcium plus magnesium production according to the sulphide content of materials for the leachate pHs observed in the last 66 weeks of testing (Equation 5.2-2). Results are consistent with higher sulphide materials having higher calcium plus magnesium production rates.

$$Calcium + Magnesium\ Production = 12.38(Sulphide + Del)^{0.32} \qquad Eqn\ 5.2\text{-}2$$

Equations 5.2-1 and 5.2-2 have notably different slopes, which make for unreliable predictions when extrapolated to lower and higher sulphide levels, both of which are seen at Morrison. These equations apply only the specific range shown, and should not be used for predictions outside this range at Morrison.

Mineralogical analyses of humidity cell deposit rock samples indicate the presence of calcite and dolomite, and therefore NP from carbonate sources is assumed to be dissolving over the humidity cell testing period. The sulphate production and carbonate NP depletion rates for the last 66 weeks of testing are assumed to remain constant in the calculation of ARD lag times.

Table 5.2-1 shows the results of calculations to sulphide and carbonate NP depletion and time to ARD onset for Morrison rock. Calculations indicate three of the five humidity cells are predicted to generate ARD, while there is enough NP in the two remaining humidity cell

---

4 Note that due to the sampling frequency, average values among the six leachate sampling events are presented for calculation purposes. This accounts for the step–like graphical pattern of the data in Figure 5.2-16.

Calcium + Magnesium Production vs Sulphide Content

$$Calcium + Magnesium\ Production = 12.38(Sulphide + Del)^{0.32}$$
$$R^2 = 0.68$$

Note: Average weekly pH shown next to data point (week 102-168)
Ca + Mg Production Rate based on weeks 102 to 168

Legend:
- MO-01-55 80.4-80.8 (199934)
- MO-00-19 83-83.4 (199999)
- MO-01-24 86.33-86.87 (200005)
- Leach Pad 2
- MET-02- 92.72-96.08

Morrison Copper/Gold Project
Calcium + Magnesium Production Rates vs
Sulphide + Del for Morrison Humidity Cell Rock

FIGURE 5.2-18

materials to maintain neutral conditions until sulphides are exhausted. This assumes NP depletion rates and sulphate production rates remain constant. These results are consistent with adjusted SNPRs of less than two and greater than two for PAG and not-PAG materials, respectively. As a conservative approach, sulphate production for the last 66 weeks was assumed to be caused by sulphide oxidation for ARD lag time calculations. These results are similar and consistent with Morin and Hutt (Appendix A). Note that laboratory humidity cell reaction rates may be as much as three to twenty times greater than site-specific kinetic testing according to ASTM (2007). This leads to possible overestimations of estimated lag times to ARD. Morrison field scale LTP reaction rates, when enough data is made available, should provide a comparison with humidity cell reaction rates to confirm or update lag times to ARD estimates to date.

Predicted lag times shown in Table 5.2-1 apply specifically to rock samples in operating humidity cells. However, for illustrative purposes only, an assessment of the Sobek NP from all the rock samples from the ML/ARD database versus predicted lag times for the humidity cell rock samples was done. Figure 5.2-19 shows that possible interpretations of predicted lag times include:

- 4 to 19% of rock materials, if excavated, may go acidic within the operational time frame (less than 21 years);

- 46% of rock materials, if excavated, may go acidic within 57 years after excavation; and

- some proportion greater than 54% of rock materials, if excavated, may never go acidic (Note that 1,000 years has been assigned the "never/infinity" time frame listed in Table 5.2-1 for illustrative purposes only).

Note that rock materials with different NPs will be excavated according to the proposed mine schedule which implies that materials that may never go acidic will be mixed in with materials that may go acidic and placed into storage (Table 5.2-2. Since segregation of excavated rock materials is not proposed for the Morrison Project, investigations using the Geochemical Block Model in conjunction with the proposed mine schedule to understand the sequence of mined blocks and associated NPs do not apply. Therefore, lag times to ARD for full scale mine components should be estimated with a conservative approach, as is the case for predictive water quality modeling where mine components are assumed to have acidic to neutral drainage quality as soon mine materials are exposed (See Section 6 and Appendix 23).

5.2.1.3 Morrison Tailings Humidity Cells

Metallurgical testing was done by SGS in 2007 to improve pyrite recovery for cycloned sand dam construction from the coarse fraction of the combined rougher and cleaner locked cycle test tailings. Two tailings samples were prepared from this metallurgical test program. The rougher (F25) and cleaner (F26) tailings from LCTs were composited and screened using a 270-mesh (0.053 mm) screen (Klohn Crippen Berger 2008). The pyrite was removed from the coarse fraction (representing the cyclone tailings sand for dam construction) by an additional floatation circuit. The pyrite was then combined with the fine tailings (representing the fine tailings overflow to the TSF). Whole tailings are estimated to be a combination of 65% coarse and 35%

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fines (Klohn Crippen Berger 2008). Kinetic testing was carried out by SGS Lakefield (Appendix C) and all data reported in this section have been taken from this source.

Note that loading rates are not presented here but can be visually estimated as the leachate flushed is consistently approximately 1 L per flushing event and humidity cells are charged with 1 kg of tailings resulting in loading rates in mg/kg of approximately the measured concentrations presented below. The complete results are presented in Appendix C.

Humidity cell leachate pH has remained near-neutral and steady throughout 80 weeks of testing (Figure 5.2-20). Figures 5.2-21 and 5.2-22 illustrate alkalinity and acidity, respectively. These results indicate excess alkalinity, particularity in the coarse fraction, is responsible for observed pH values. Sulphate concentrations have remained low, with the fine tailings producing consistently two times more with the exception of initial cell flushing (Figure 5.2-6).

Additional parameters of interest in the tailings humidity cell leachate include:

- steady and variable, but low copper leaching for the coarse- and fine-tailings leachate, respectively;
- initial releases of molybdenum with a steady low concentration trend afterwards;
- moderate and variable zinc releases;
- variable but low releases of lead;
- initial flushing of arsenic and antimony and decreasing concentration trends for both tailings fractions;
- variable but very low cadmium releases;
- selenium below MDLs.

See Figures 5.2-23 to 5.2-31.

In general, the coarse tailings fraction shows higher elemental leaching than the fine tailings fraction leachate. The tailings mineralogy was not characterized, and therefore it is not possible to provide additional insight into solid-phase reactions responsible for humidity cell leachate chemistry trends.

5.2.1.4   Morrison Tailings ARD Lag Times

As discussed in Section 5.2.1.2, CMRs can provide important insight into the rate of NP consumption relative to the sulphate production rate. Figure 5.2-32 shows that for the last 25 weeks of data, CMRs have remained steady and that the carbonate NP in the coarse tailings fractions is, in part, being depleted because of excessive dissolution and flushing, while the fine tailings fraction carbonate NP dissolution is occurring primarily because of sulphide oxidation. However, the dissolved iron (data not shown) is below MDLs and the pH of the fine-fraction leachate has remained near -neutral. It is likely that the high flushing rate used (i.e., ~1 L per

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Table 5.2-1
Morrison Copper/Gold Project: Morrison Deposit Rock Humidity Cell Lag Times to Onset of Acid Rock Drainage

| Humidity Cell ID | Grade | Designation | Lithology | Alteration | ABA Results | | | | | | | Average Sulphide-Depletion Rate [1] | Time to Sulphide Depletion [1] | Average Carb NP Depletion Rate [1] | Time to Carb NP Depletion [1] | Average Carbonate Molar Ratio [1] | Time to ARD Onset | Anticipated Humidity Cell Results |
					Sulphide + del (%)	SAP kg CaCO3/t	TIC (%)	Carb NP kg CaCO3/t	Adj NP kg CaCO3/t	SNPR nu	Adj SNPR nu	(mmol S /kg/wk)	(years)	(mmol/kg/wk)	(years)	(Ca+Mg/SO4)	(years)	
MO-01-55 80.4-80.8 (199934)	0.28	Low Grade Ore	SED	QZSE	1.7	56	1.28	45	32	0.81	0.58	0.12	85	0.34	57	7	57	Delayed acidic pH and onset of ML/ARD
MO-00-19 83-83.4 (199999)	0.038	Waste Rock	BFP	ARSE	1.3	46	0.60	8.0	-5.0	0.18	0.0010	0.30	24	0.52	15	1.78	15	Delayed acidic pH and onset of ML/ARD
MO-01-24 86.33-86.87 (200005)	0.51	Ore	BFP	ARSE	0.27	8.6	2.07	161	148	19	17	0.036	40	0.29	112	8.0	never	Neutral drainage and ML
Leach Pad 2	0.17	Waste Rock	SED	QZSE	0.80	28	0.54	24	11	0.85	0.39	0.081	63	0.40	19	4.9	19	Delayed acidic pH and onset of ML/ARD
MET-02-92.72-96.08	0.081	Waste Rock	BFP	KL	0.12	3.8	0.76	56	43	15	11	0.018	32	0.21	56	11.3	never	Neutral drainage and ML

Notes:
1. Calculations based on humidity cell data from weeks 102-168
nu = no units

sampling event) results in excessive carbonate dissolution, in addition to that carbonate dissolution consumed directly by sulphide oxidation, resulting in low CMRs for the fine tailings fraction leachate.

Table 5.2-2
Morrison Copper/Gold Project:
Waste Rock Storage Adjusted SNPR Areas

End of Year	Subset Waste Adj-SNPR 0-0.5	Subset Waste Adj-SNPR 0.5-1.5	Subset Waste Adj-SNPR 1.5-2.5	Subset Waste Adj-SNPR > 2.5	Waste Subtotal
	tonnes				
Year -1	69,000	365,000	2,786,000	1,930,000	5,150,000
Year 1	146,000	1,014,000	4,599,000	3,606,000	9,365,000
Year 2	130,000	650,000	2,997,000	3,739,000	7,516,000
Year 3	377,000	210,000	796,000	3,238,000	4,621,000
Year 4	73,000	290,000	7,721,000	3,054,000	11,138,000
Year 5	18,000	419,000	5,967,000	1,521,000	7,925,000
Year 6	154,000	1,536,000	2,822,000	2,558,000	7,070,000
Year 7	552,000	1,648,000	1,724,000	4,374,000	8,298,000
Year 8	595,000	1,336,000	2,228,000	6,124,000	10,283,000
Year 9	1,525,000	1,596,000	4,560,000	3,226,000	10,907,000
Year 10	852,000	1,876,000	5,914,000	1,864,000	10,506,000
Year 11	817,000	2,164,000	5,108,000	761,000	8,850,000
Year 12	429,000	1,591,000	10,372,000	819,000	13,211,000
Year 13	291,000	1,267,000	9,291,000	1,178,000	12,027,000
Year 14	442,000	1,672,000	8,893,000	2,638,000	13,645,000
Year 15	1,866,000	2,206,000	6,708,000	2,470,000	13,250,000
Year 16	2,314,000	1,330,000	5,586,000	2,071,000	11,301,000
Year 17	366,000	508,000	1,690,000	444,000	3,008,000
Year 18	484,000	66,000	45,000	21,000	616,000
Year 19	189,000	92,000	8,000	0	289,000
Total	**11,689,000**	**21,836,000**	**89,815,000**	**45,636,000**	**168,976,000**

A relationship can be developed for average sulphate production rates versus sulphide + del (Figure 5.2-33), that predicts the rate of sulphate production according to the sulphide content of materials for the leachate pHs observed in the last 25 weeks of humidity cell testing (Equation 5.2-3). Results are consistent with higher sulphide tailings having higher sulphate production rates, but leachate data are presented for only two tailings samples.

$$Sulphate\ Production = 25.33(Sulphide + Del)^{0.79} \qquad Eqn\ 5.2\text{-}3$$
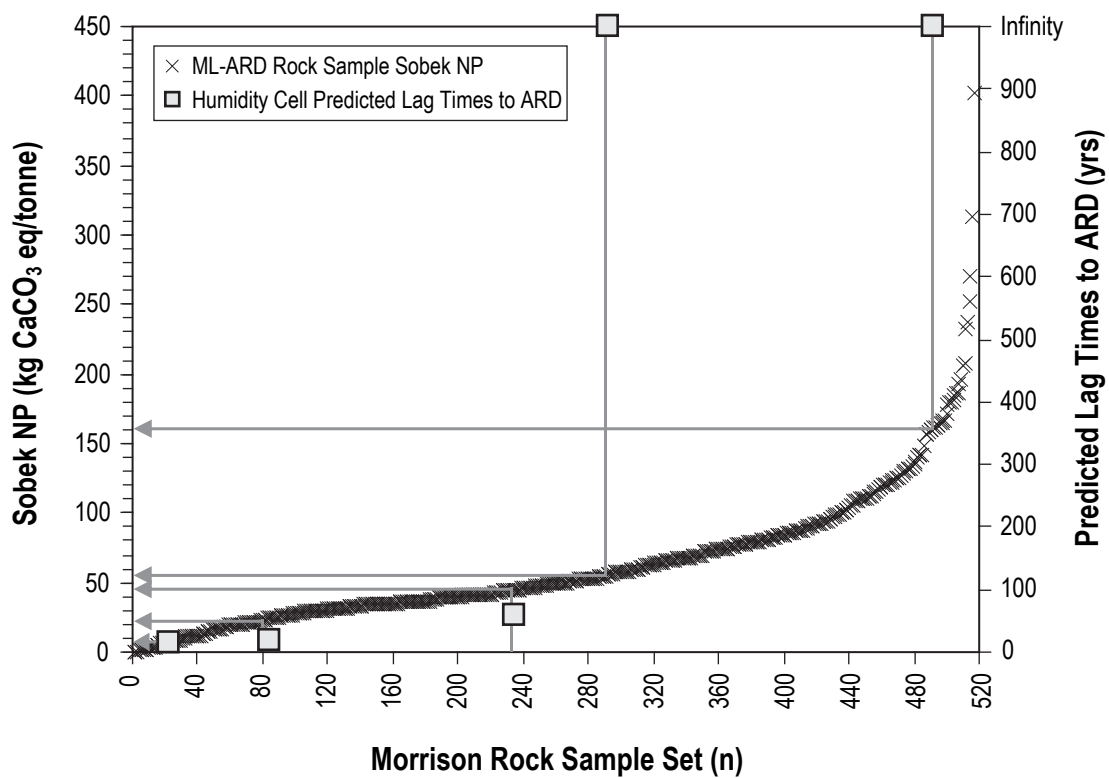
Morrison Copper/Gold Project
Morrison Rock Cumulative Sobek NP and
Predicted Lag Times for Humidity Cell Samples

FIGURE 5.2-19
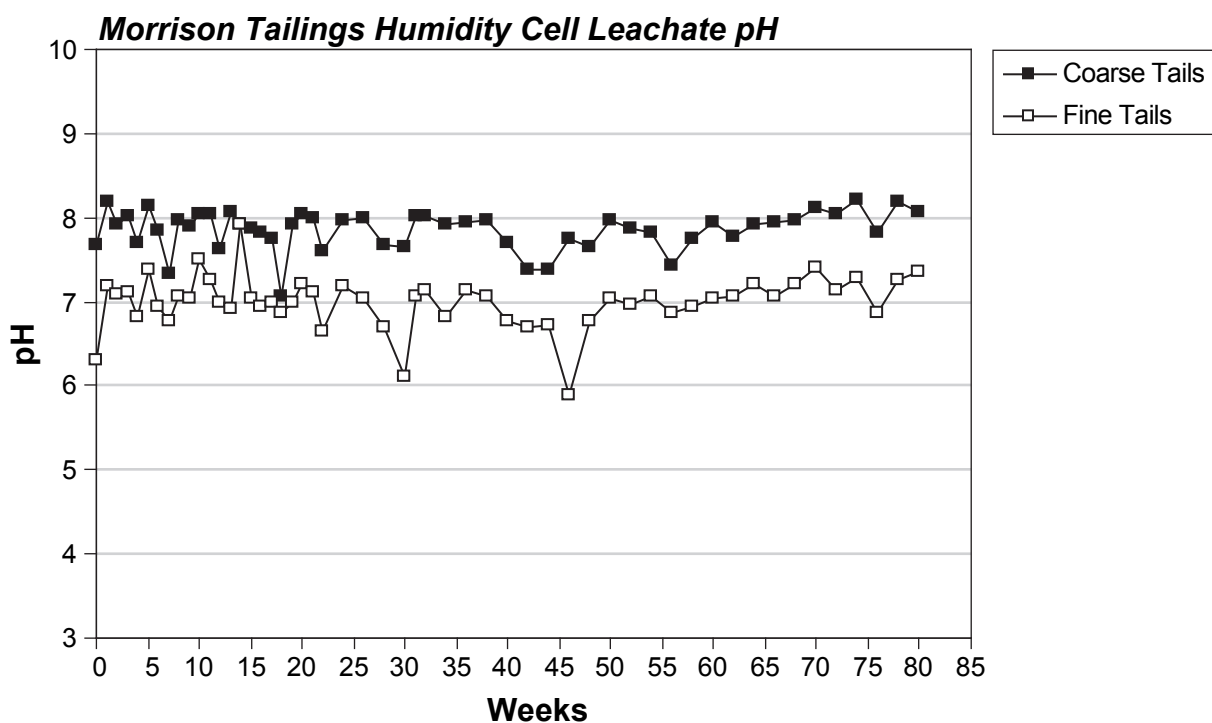
Morrison Copper/Gold Project
Morrison Tailings Humidity
Cell Leachate Results for pH

FIGURE 5.2-20
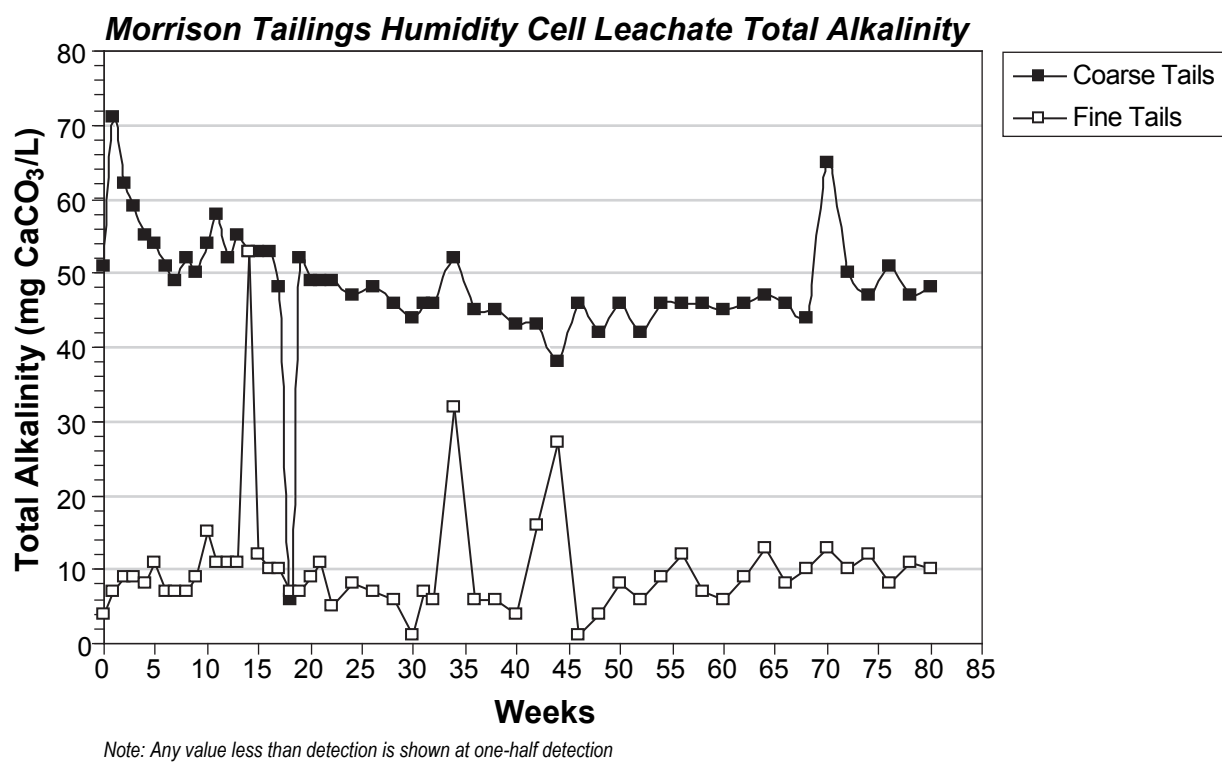
**Morrison Copper/Gold Project
Morrison Tailings Humidity Cell
Leachate Results for Total Alkalinity**
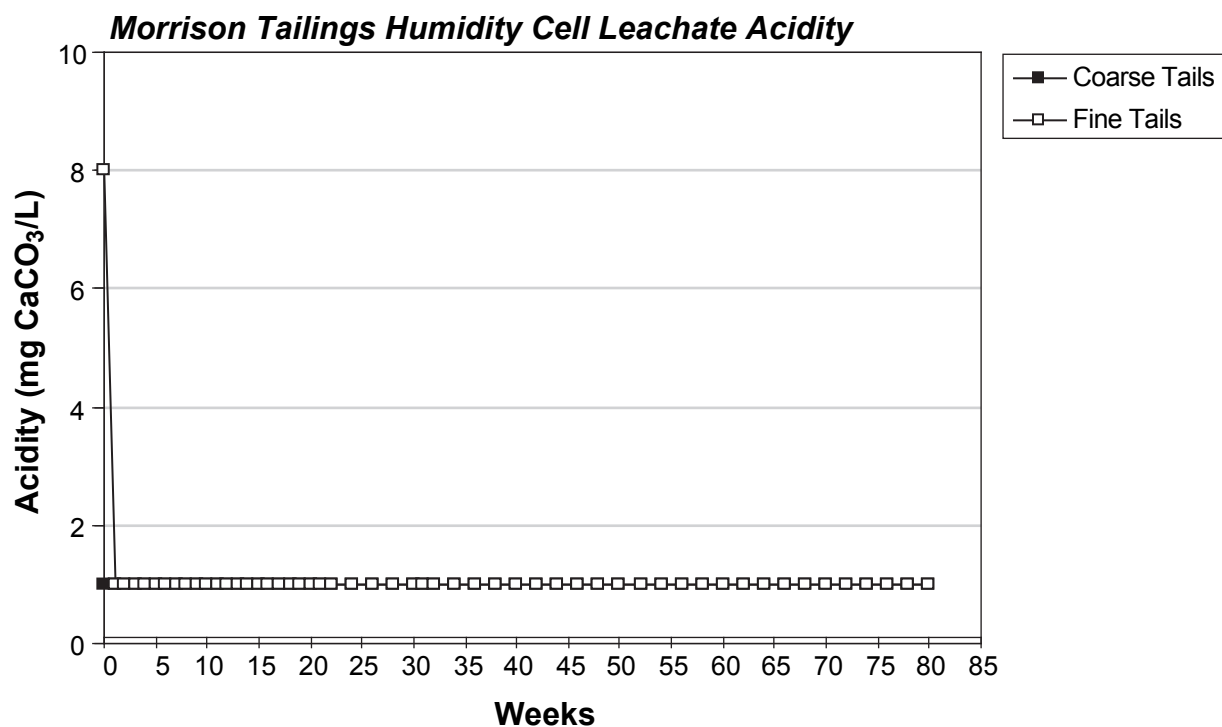
FIGURE 5.2-21

**Morrison Copper/Gold Project
Morrison Tailings Humidity Cell
Leachate Results for Acidity**

FIGURE 5.2-22

Morrison Tailings Humidity Cell Leachate Sulphate

Legend: Coarse Tails, Fine Tails

Y-axis: Sulphate (mg/L)
X-axis: Weeks

Note: Any value less than detection is shown at one-half detection

Morrison Copper/Gold Project
Morrison Tailings Humidity Cell
Leachate Results for Sulphate

FIGURE 5.2-23

Morrison Tailings Humidity Cell Leachate Dissolved Copper

Note: Any value less than detection is shown at one-half detection

Morrison Copper/Gold Project
Morrison Tailings Humidity Cell
Leachate Results for Dissolved Copper

FIGURE 5.2-24

Morrison Tailings Humidity Cell Leachate Dissolved Molybdenum

Note: Any value less than detection is shown at one-half detection

Morrison Copper/Gold Project
Morrison Tailings Humidity Cell
Leachate Results for Dissolved Molybdenum

FIGURE 5.2-25

Morrison Copper/Gold Project
Morrison Tailings Humidity Cell
Leachate Results for Dissolved Zinc

FIGURE 5.2-26

Morrison Tailings Humidity Cell Leachate Dissolved Lead

Note: Any value less than detection is shown at one-half detection

Morrison Copper/Gold Project
Morrison Tailings Humidity Cell
Leachate Results for Dissolved Lead

FIGURE 5.2-27

**Morrison Copper/Gold Project
Morrison Tailings Humidity Cell
Leachate Results for Dissolved Arsenic**

FIGURE 5.2-28

Morrison Copper/Gold Project
Morrison Tailings Humidity Cell
Leachate Results for Dissolved Antimony

FIGURE 5.2-29

Morrison Tailings Humidity Cell Leachate Dissolved Cadmium

Note: Any value less than detection is shown at one-half detection

Morrison Copper/Gold Project
Morrison Tailings Humidity Cell
Leachate Results for Dissolved Cadmium

FIGURE 5.2-30

Morrison Tailings Humidity Cell Leachate Dissolved Selenium

Note: Any value less than detection is shown at one-half detection

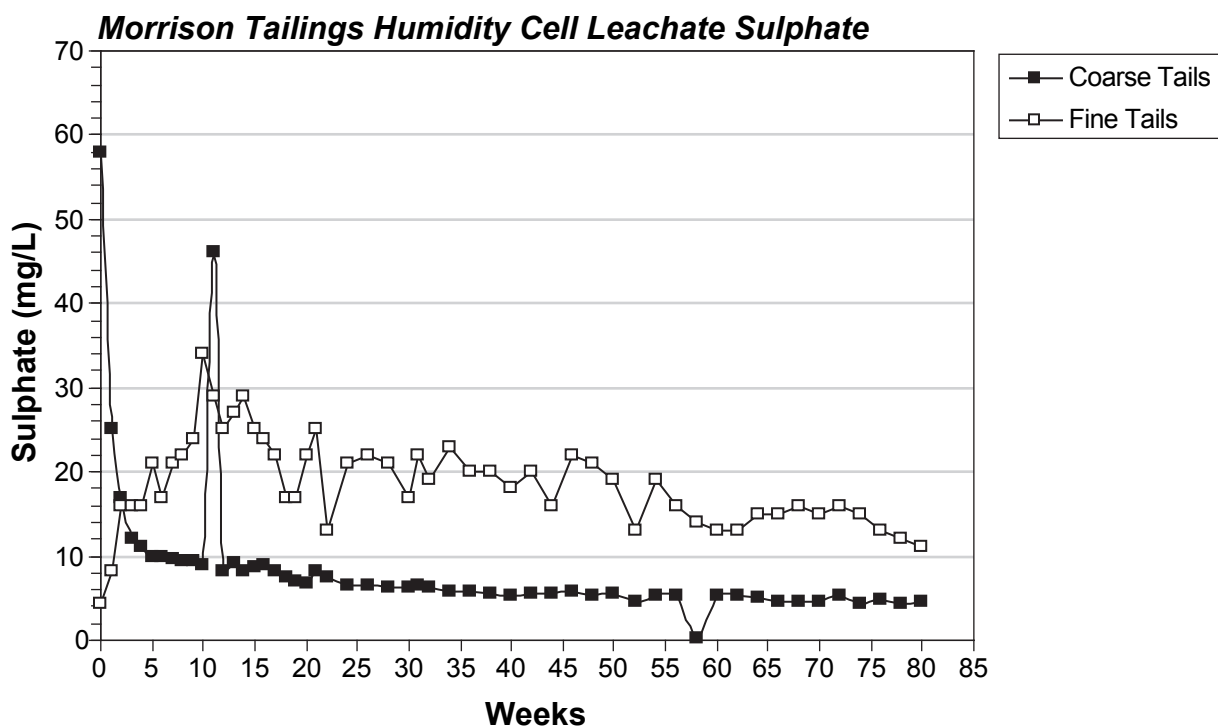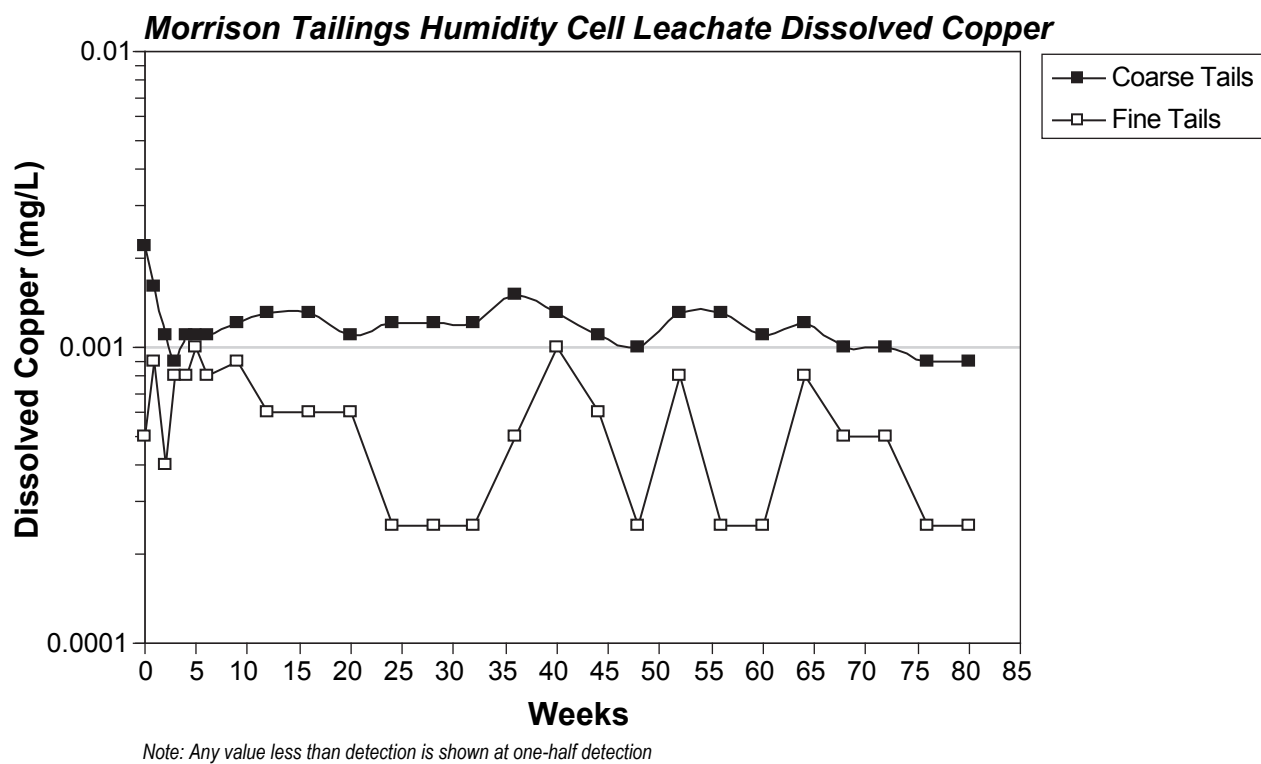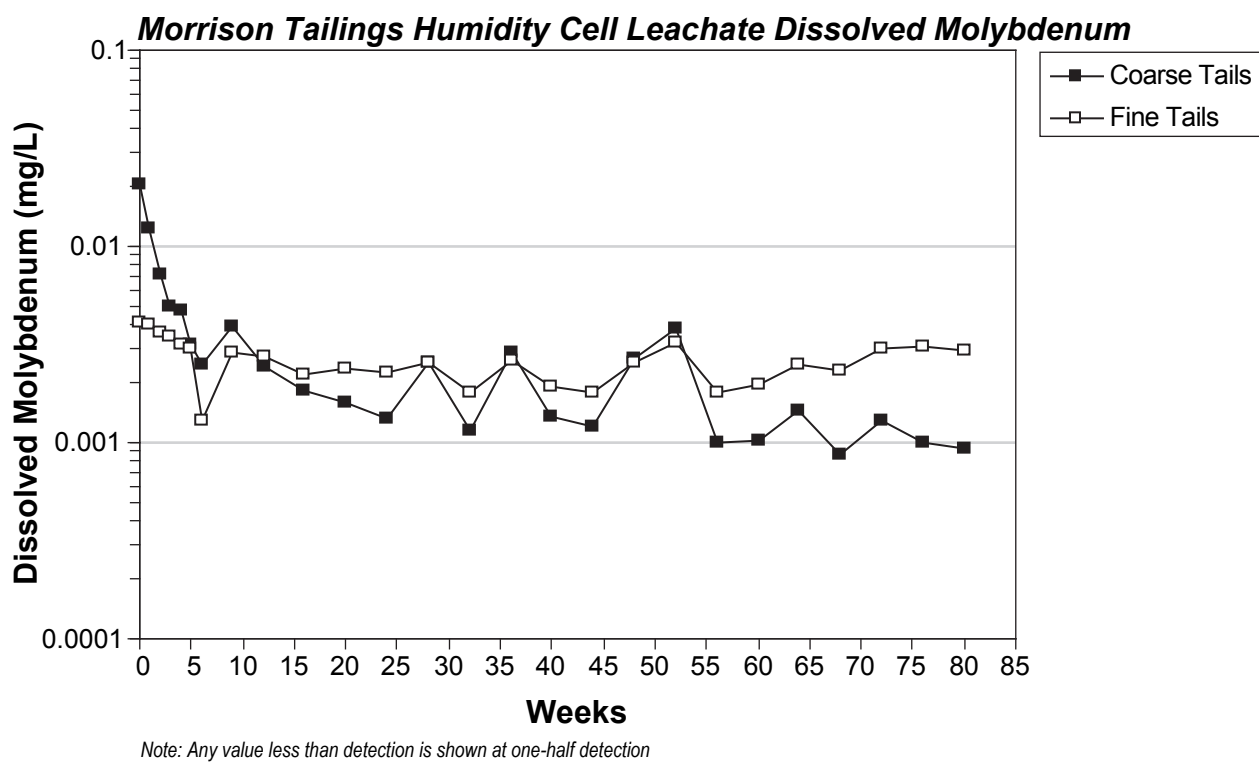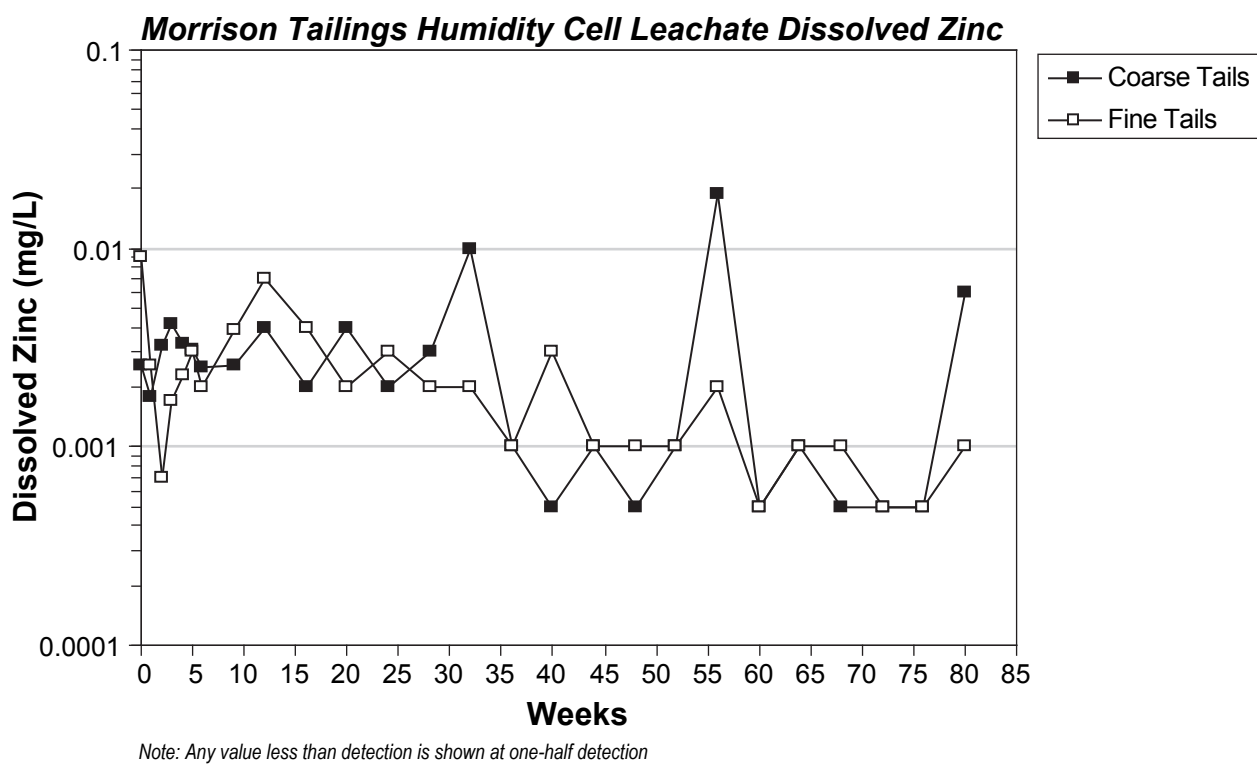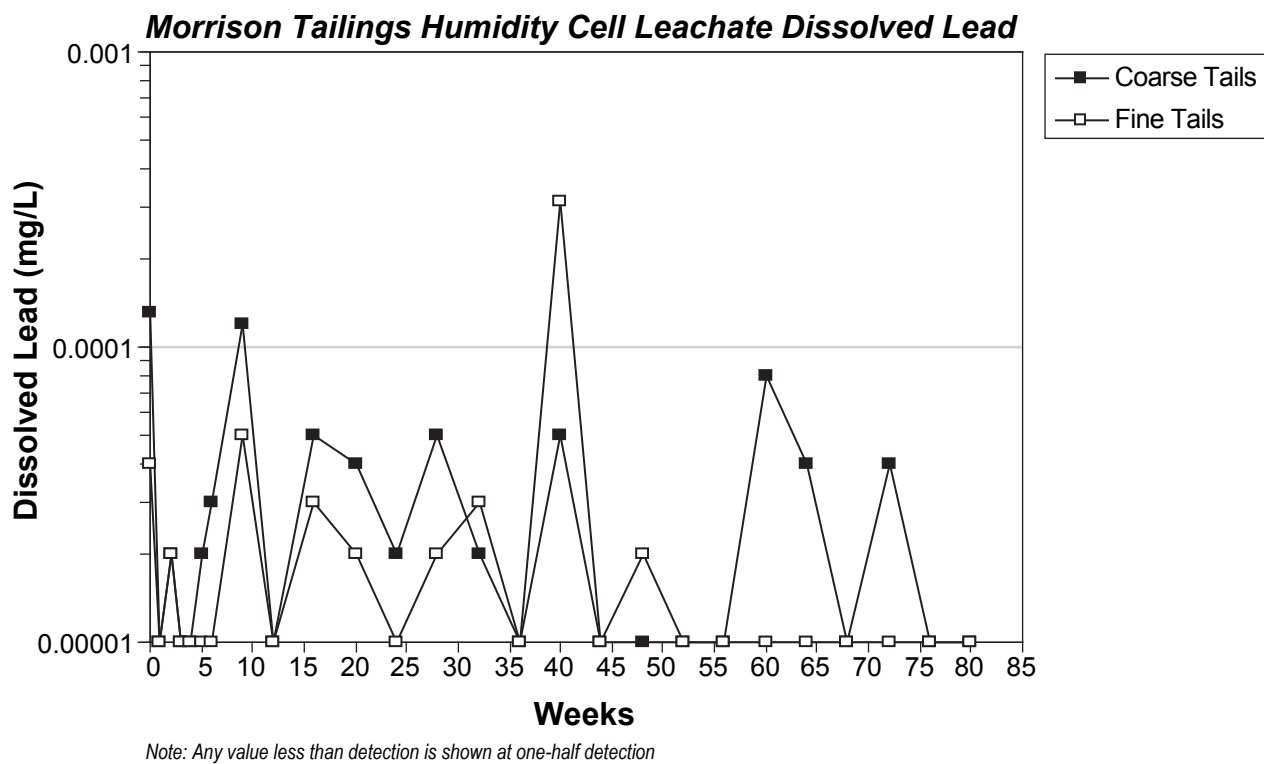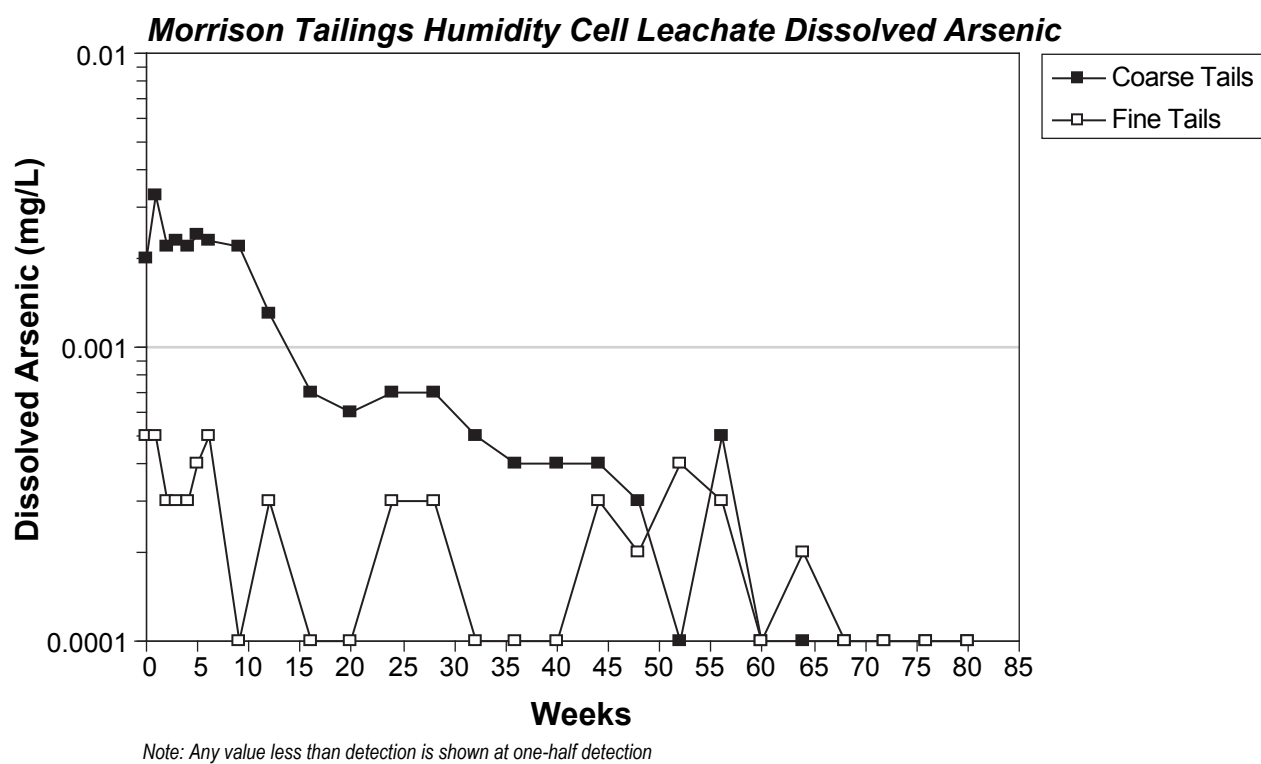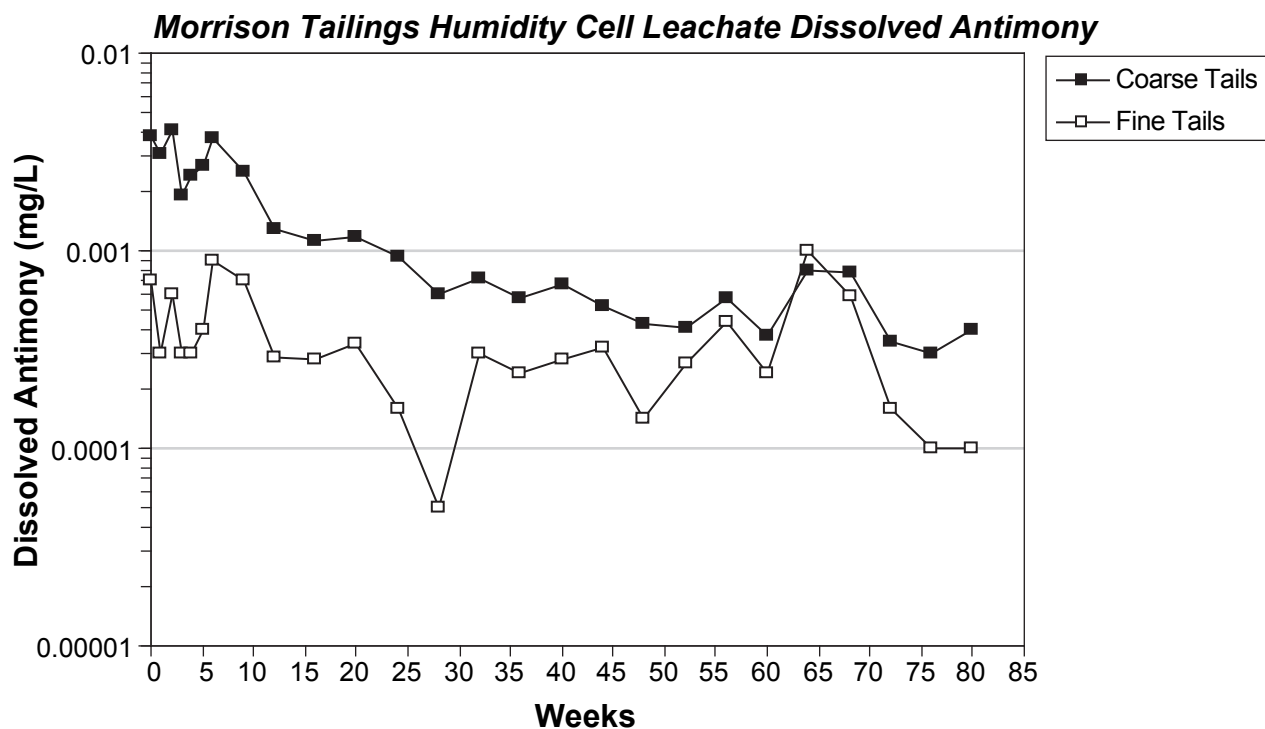**Morrison Copper/Gold Project**
Morrison Tailings Humidity Cell
Leachate Results for Dissolved Selenium

FIGURE 5.2-31

Morrison Copper/Gold Project
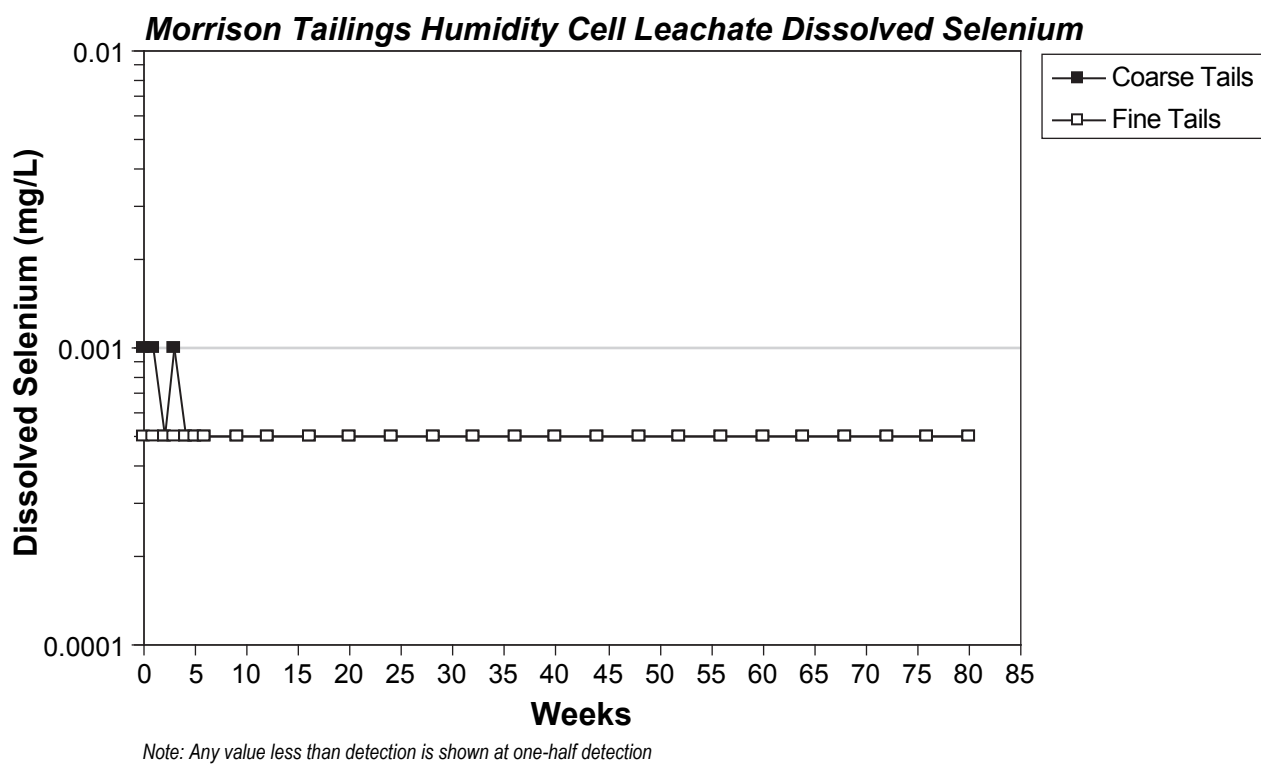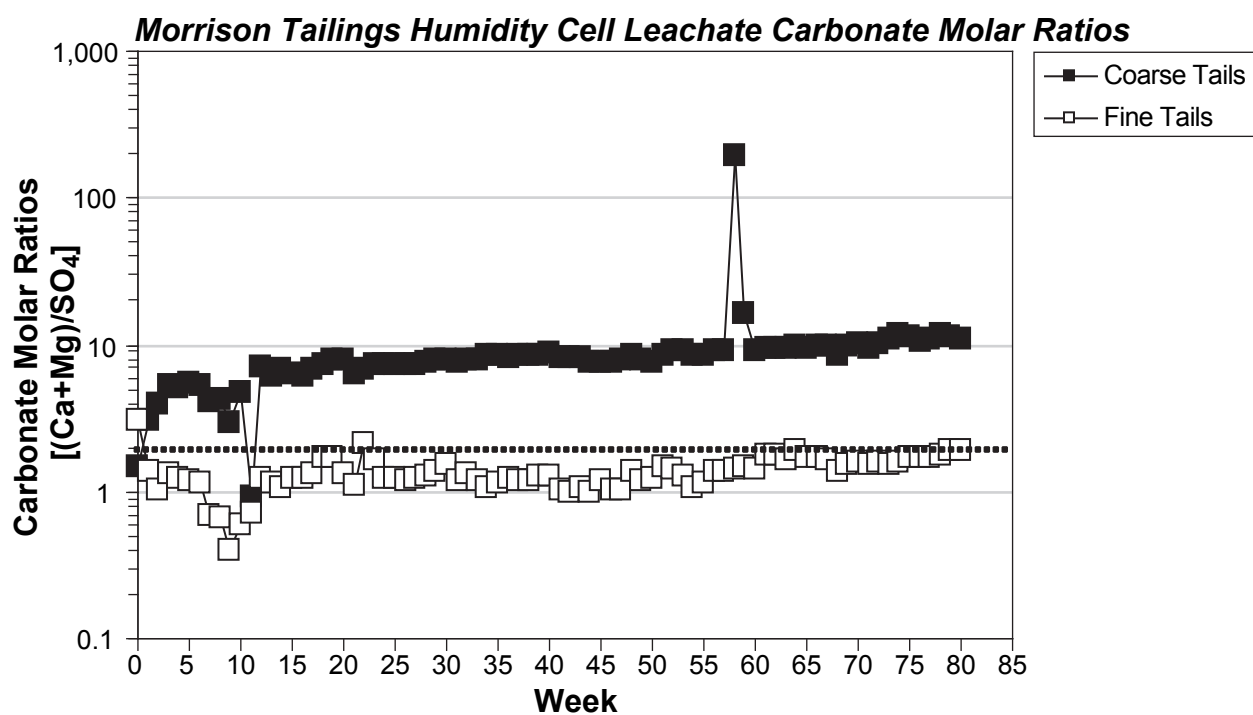**Morrison Tailings Humidity Cell**
Leachate Carbonate Molar Ratios

FIGURE 5.2-32

A similar relationship was developed for average calcium plus magnesium production rates versus sulphide + del (Figure 5.2-33), that predicts the rate of calcium plus magnesium production (Equation 5.2-4). However, the equation suggests more calcium plus magnesium production with less sulphide, which is counterintuitive and re-illustrates the effect of excessive carbonate depletion seen in CMRs in Figure 5.2-32 for the coarse tailings fraction.

$$Calcium + Magnesium\ Production = 4.98(Sulphide + Del)^{-0.46} \qquad Eqn\ 5.2\text{-}4$$

The sulphate production and carbonate NP depletion rates for the last 25 weeks of testing are assumed to remain constant in calculating the onset of ARD lag times. As a conservative approach, excess carbonate depletion in the coarse tailings fraction is included in lag time to ARD onset calculations.

Table 5.2-3 shows the results of calculations to sulphide and NP depletion and time to ARD onset for Morrison tailings. Note that carbonate NP is greater than bulk Sobek NP indicating the presence of net neutral carbonates, such as siderite. It is likely that this mineral is present in tailings because Morrison deposit rock mineralogical analyses confirm its presence (Section 5.1.4). Therefore, the bulk Sobek NP was used for ARD lag-time calculations as a reasonable and conservative approach. Calculations predict that leachate from both coarse and fine tailings will remain near-neutral as there is sufficient NP to maintain neutral conditions until sulphides are exhausted. This assumes NP depletion rates and sulphate production rates remain constant.

Specifications for the cycloned tailings sand used in dam constructions are <20% fines (<75 micron) (Klohn Crippen Berger 2008). The particle size analyses on the coarse tailings shows 25% fines (Appendix C) suggesting, from a geochemical perspective, that the data gathered to date on the coarse tailings fraction (given the fine tailings fraction has higher sulphide sulphur and higher surface area) is a conservative and a representative estimate of the ML/ARD characteristics of cycloned tailings sand that will be used for TSF dam construction. In addition, because whole tailings are to be a 65-to-35%, coarse-to-fine tailings mixture, the ML/ARD data gathered to date indicate that neither the coarse nor fine fraction will generate ARD. Therefore, a 65:35 coarse-to-fine mixture is also predicted to be not-PAG.

5.2.1.5   Tailings Aging Tests

As part of the ML/ARD prediction program for the Morrison tailings, aging tests of the Morrison tailings supernatant (i.e., process waters) were conducted for 61 days by SGS (Appendix C). The test methodology and full analytical results can be found in Appendix C.

The aging test data indicate tailings process waters remained slightly alkaline over the 61-day test period caused by the moderate concentrations of alkalinity, presumably because of the metallurgical process and carbonate dissolution (figures 5.2-35 and 5.2-36). The latter presumption is inferred from ore mineralogy (Section 5.1.1). An increase in sulphate with time is detected, but significant sulphide oxidation is unlikely because of the submergence of tailings (i.e., supernatant over tailings is decanted at aged intervals) and the lack of concomitant increases in iron (figures 5.2-37 and 5.2-38). Total suspended solids (TSS) show initially high
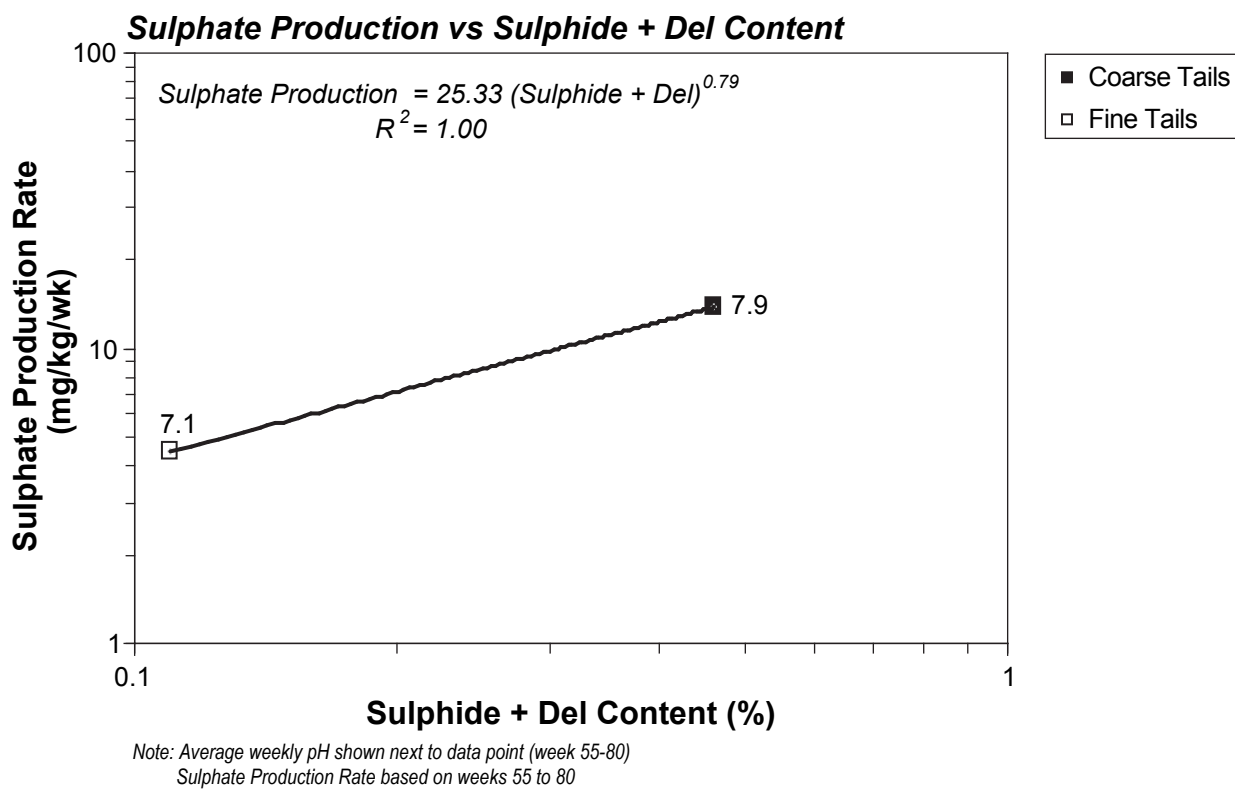
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Morrison Copper/Gold Project
Sulphate Productions Rates vs Sulphide + Del
Content for Morrison Humidity Cell Tailings

FIGURE 5.2-33

Table 5.2-3
Morrison Copper/Gold Project:
Morrison Tailings Humidity Cell Lag Times to Acid Rock Drainage

Sample Description	Sample Type	ABA Results						Average Sulphide Depletion Rate [1] (mmol/ kg/wk)	Time to Sulphide Depletion[1] (years)	Average NP Depletion Rate[1] (mmol/kg/wk)	Time to NP Depletion[1] (years)	Average Carbonate Molar Ratio[1] (Ca+Mg)/SO$_4$	Time to ARD Onset (years)	Anticipated Humidity Cell Results
		%S	SAP	TIC (%)	Carb NP	NP	SNPR							
Coarse fraction leachate (>53 micron)	Tailings [2]	0.11	3.4	0.74	62	51	15	0.046	11.7	0.49	18	10.6	-7	near neutral pH and no onset of ML/ARD
Fine fraction leachate (<53 micron)	Tailings [2]	0.46	14	0.96	80	61	4.2	0.14	17	0.23	50	1.6	-32	near neutral pH and no onset of ML/ARD

Notes:
1. **Based on last 25 weeks of leachate data**
2. **Generated from Locked cycle tests, pyrite flotation and screening**
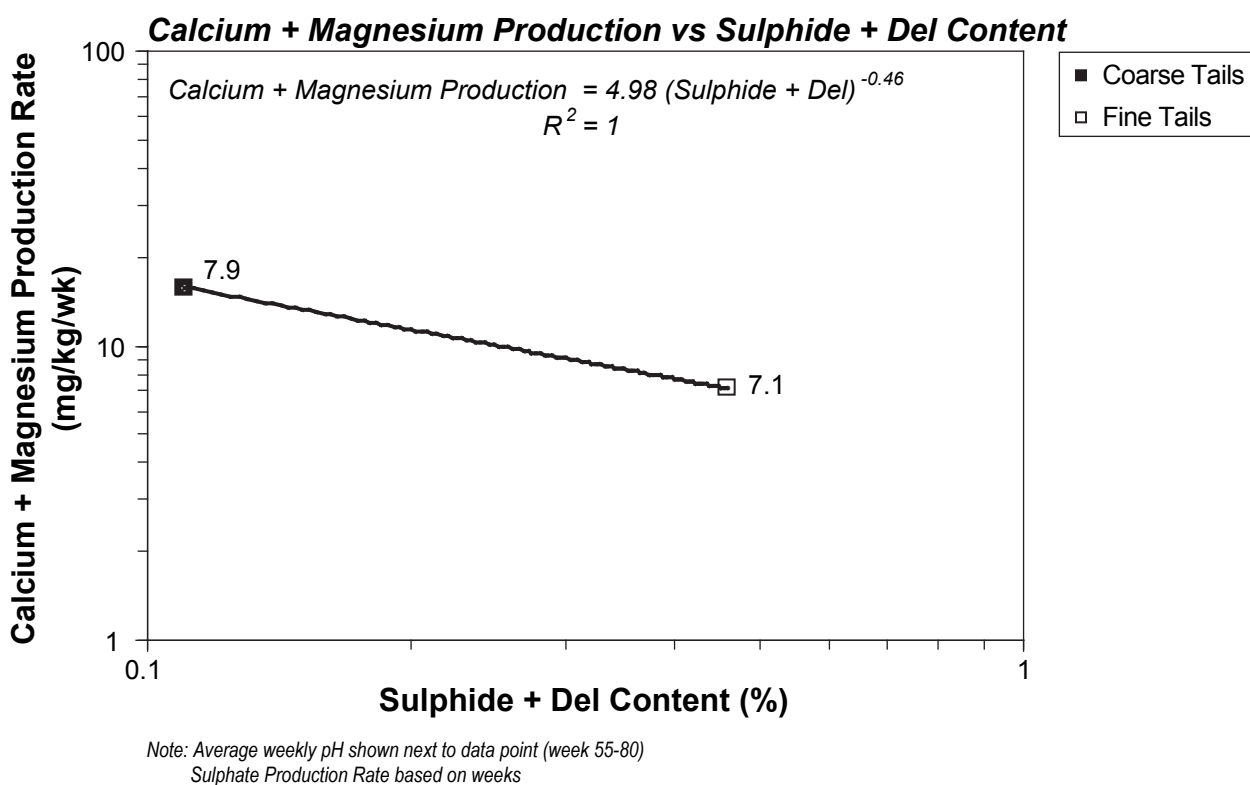
Morrison Copper/Gold Project
Calcium + Magnesium Productions Rates vs Sulphide +
Del Content for Morrison Humidity Cell Tailings

FIGURE 5.2-34

Morrison Copper/Gold Project
Morrison Tailings Ageing Test pH

FIGURE 5.2-35

Ageing Test Alkalinity and Acidity

Note: Any value less than detection is shown at one-half detection limit
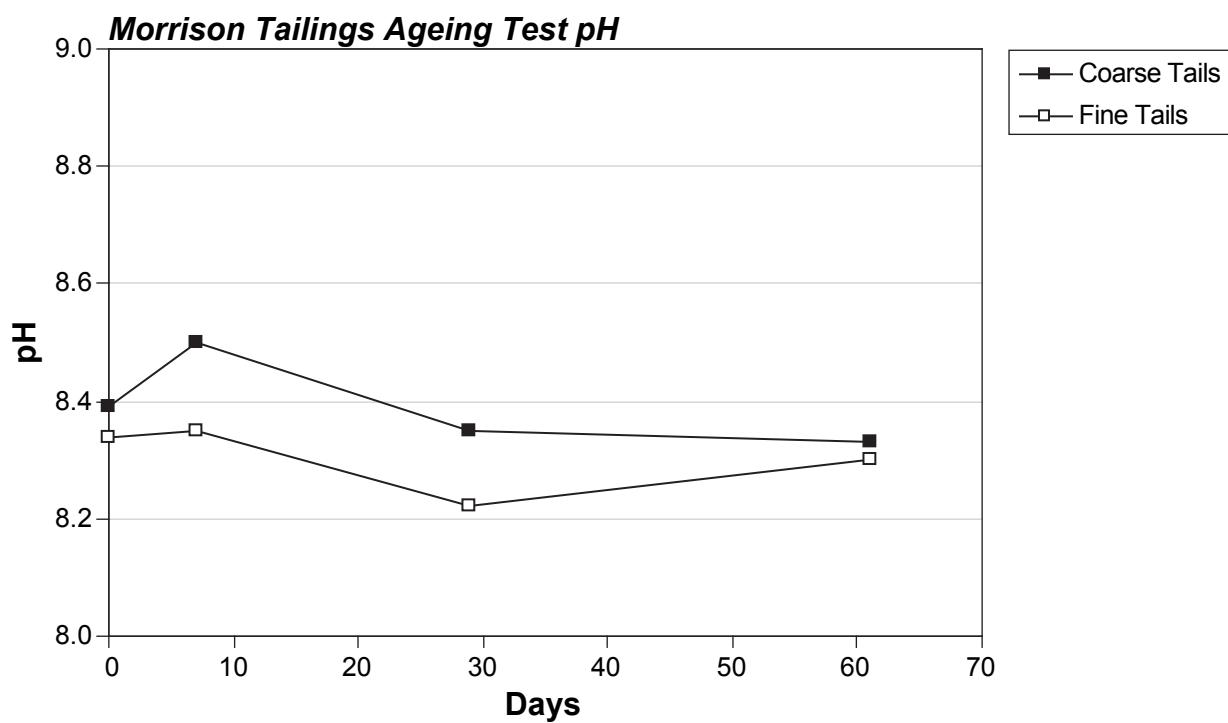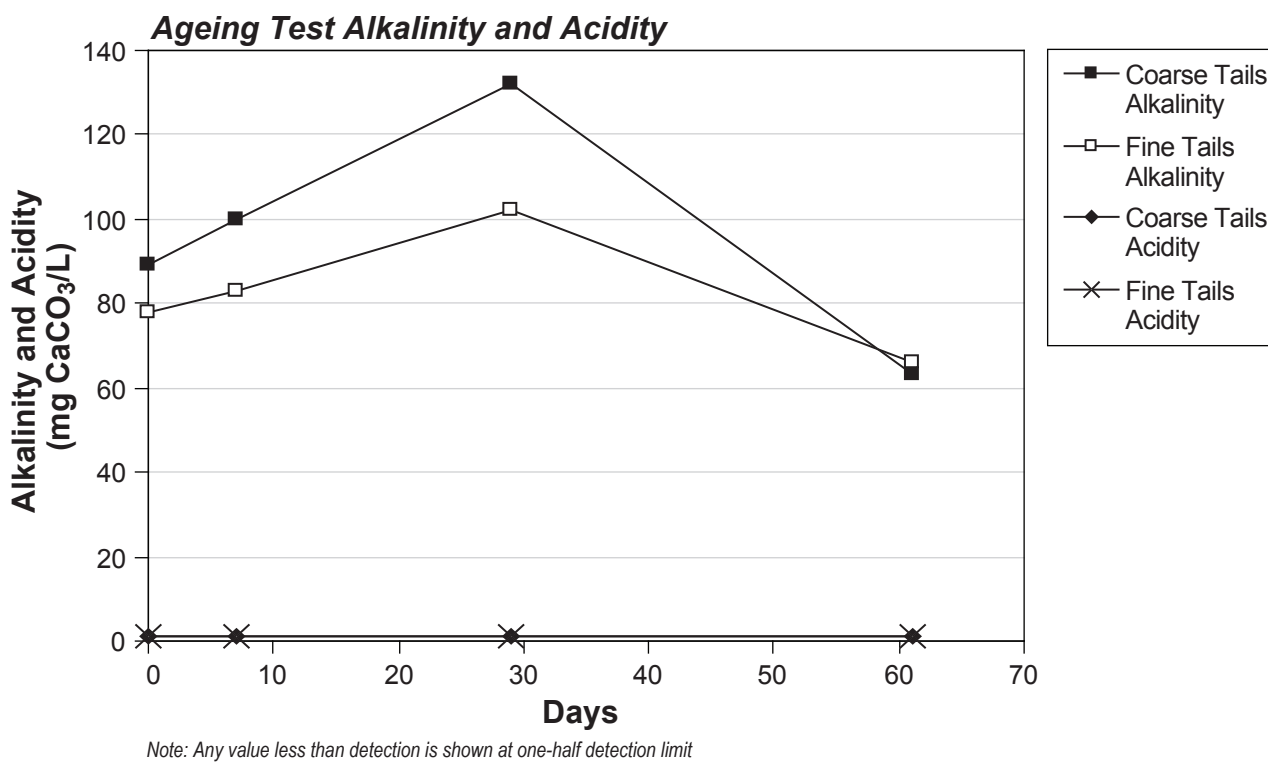
Morrison Copper/Gold Project
Morrison Tailings Ageing
Test Acidity and Alkalinity

FIGURE 5.2-36

Morrison Copper/Gold Project
Morrison Tailings Ageing Test Sulphate
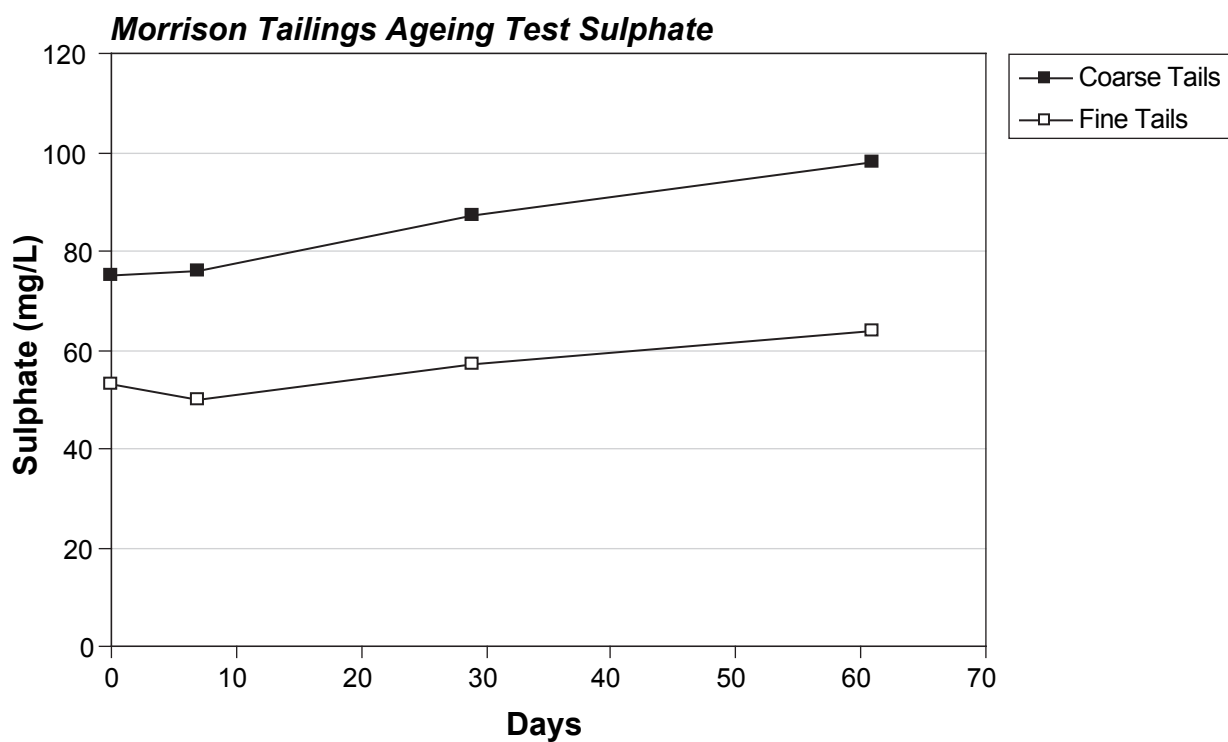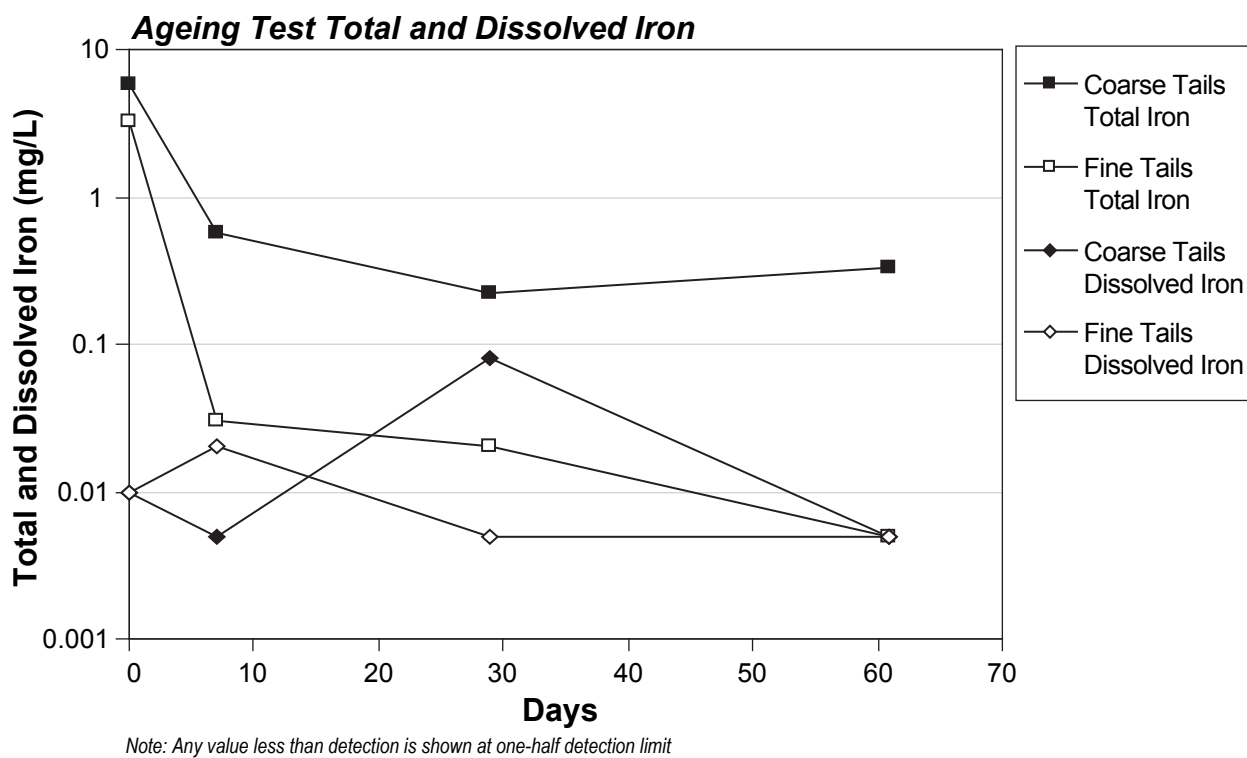
FIGURE 5.2-37

Morrison Copper/Gold Project
Morrison Tailings Ageing
Test Total and Dissolved Iron

FIGURE 5.2-38

concentrations but decrease to below the MDL for the duration of the test because of settling (Figure 5.2-39). Interestingly, total dissolved solids (TDS; Figure 5.2-39) increased with time, and are the result of major ion releases of potassium, silicon, and sodium to solution (data not shown). Additional parameters of interest generally show concentration changes with time are less than an order of magnitude, are low because of the alkaline pH and are initially higher for copper, lead, zinc, arsenic, cadmium, and selenium, but variable or somewhat steady for molybdenum and antimony (figures 5.2-40 to 5.2-47). Figures 5.2-48 and 5.2-49 show elemental concentrations are typically dominated by the total fraction initially during the aging test, but decrease and are dominated by the dissolved fraction in progressively aged tailings supernatant samples. This is a result of the settling of the TSS throughout the 61-day test period. Note that ratios below one are theoretically not possible (i.e., more dissolved element than total element), but values near the MDL can produce ratios less than one.

5.2.2   Field Scale Testing

Four on-site LTPs were constructed in 2005 by Pacific Booker Mineral's (PBM) staff. Each LTP contained approximately one tonne of broken deposit rock drill core. However, these leach pads were open to adjacent runoff and did not have adequate hydrologic control and leachate containment to properly determine the water balance and therefore leachate quality for each LTP. In addition, leachate water quality showed anomalously low concentrations compared with laboratory-based humidity cells and the nearby Bell Mine mine drainage chemistry (Appendix A). Therefore, the four LTPs were moved into above-ground 1-m^3 plastic containers called cubes in July, 2009. In addition, older Noranda exploration drill core from the early 1970s had weathered in place for more than 35 years. Some of this core was placed in smaller scale on-site barrels during larger LTP reconstruction.

Static test data were collected for LTP 1, 2, and 3 in 2005 but was considered insufficient to adequately characterize the approximate one tonne of rock in each LTP. During LTP and barrel re-construction, additional samples were collected for ML/ARD prediction testing to increase confidence in the characterization of the LTP materials. Additional information on LTP reconstruction, LTP materials description, sampling, and testing is detailed in Appendix D.

The first leachate sampling event for reconstructed and Noranda core on-site field barrel is planned for early fall of 2009.

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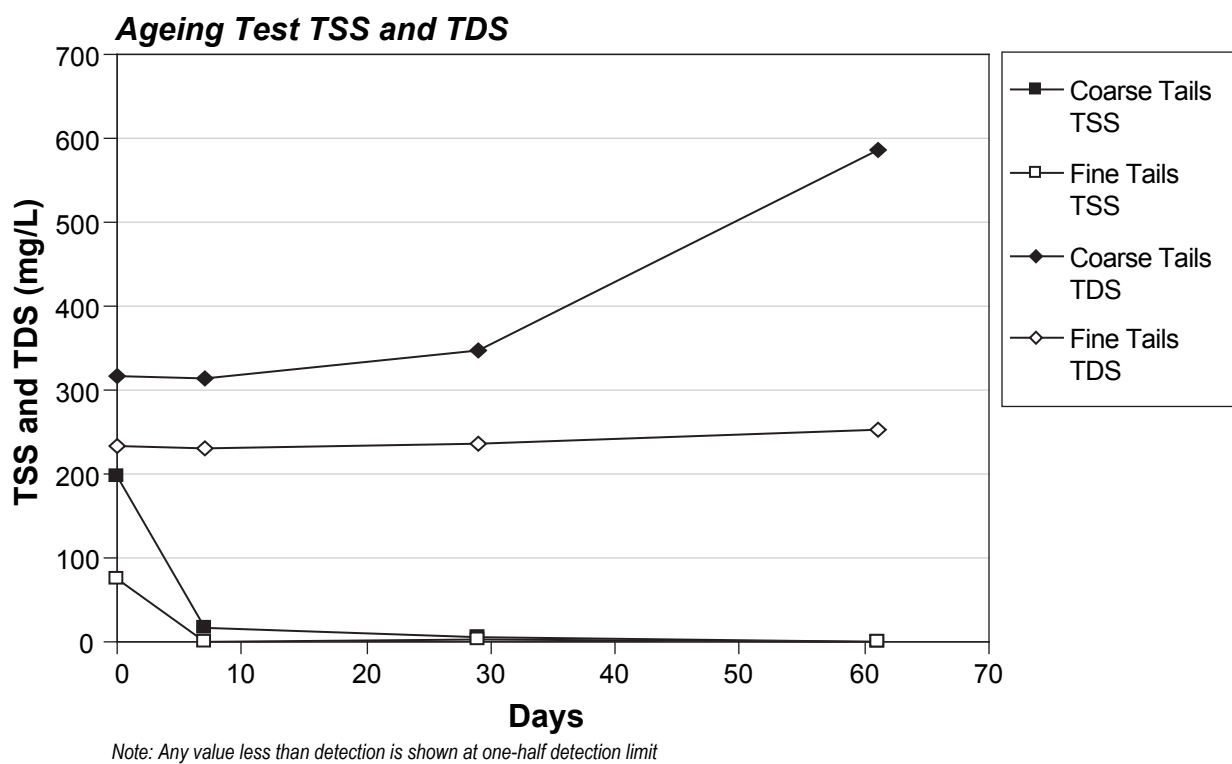
Morrison Copper/Gold Project
Morrison Tailings Ageing Test Total
Suspended Solids and Total Dissolved Solids
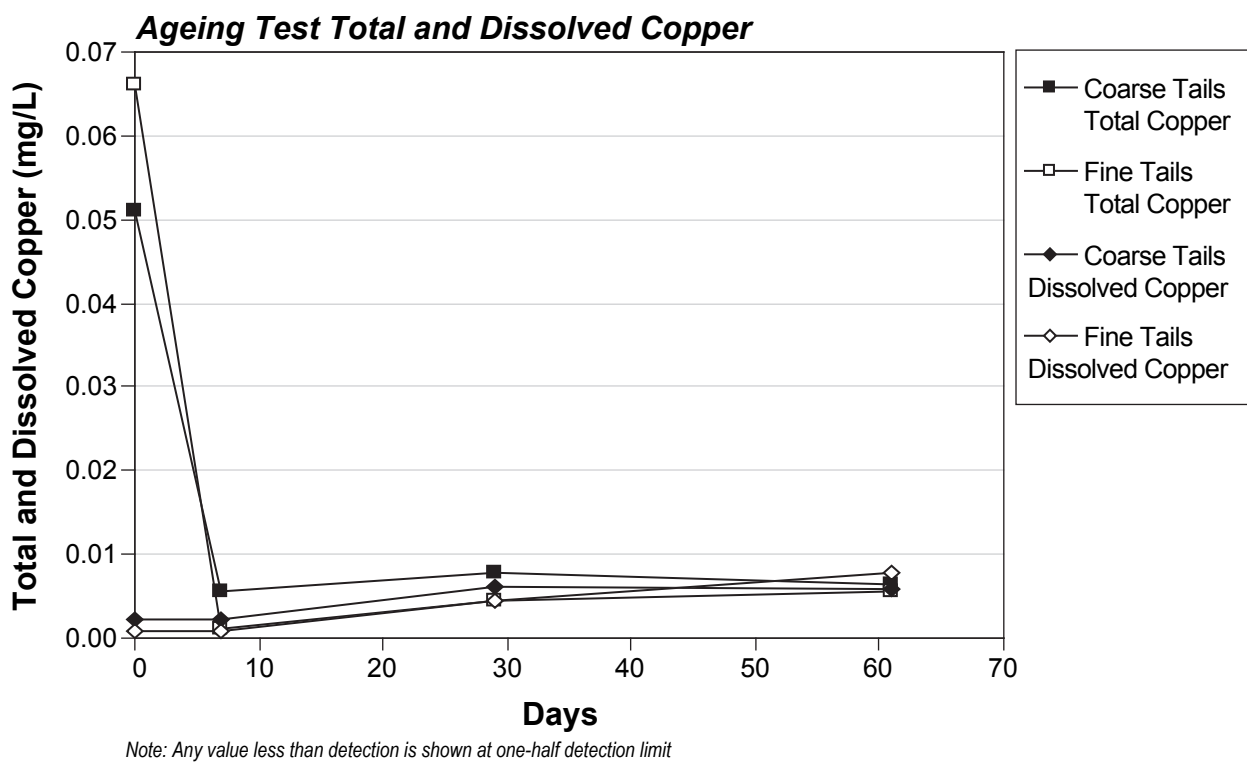
FIGURE 5.2-39

Ageing Test Total and Dissolved Copper

Note: Any value less than detection is shown at one-half detection limit

**Morrison Copper/Gold Project
Morrison Tailings Ageing Test
Total and Dissolved Copper**

FIGURE 5.2-40

Ageing Test Total and Dissolved Molybdenum

Note: Any value less than detection is shown at one-half detection limit

Morrison Copper/Gold Project
Morrison Tailings Ageing Test
Total and Dissolved Molybdenum

FIGURE 5.2-41

Morrison Copper/Gold Project
Morrison Tailings Ageing Test
Total and Dissolved Zinc

FIGURE 5.2-42

Ageing Test Total and Dissolved Lead

Note: Any value less than detection is shown at one-half detection limit

Morrison Copper/Gold Project
Morrison Tailings Ageing Test
Total and Dissolved Lead

FIGURE 5.2-43
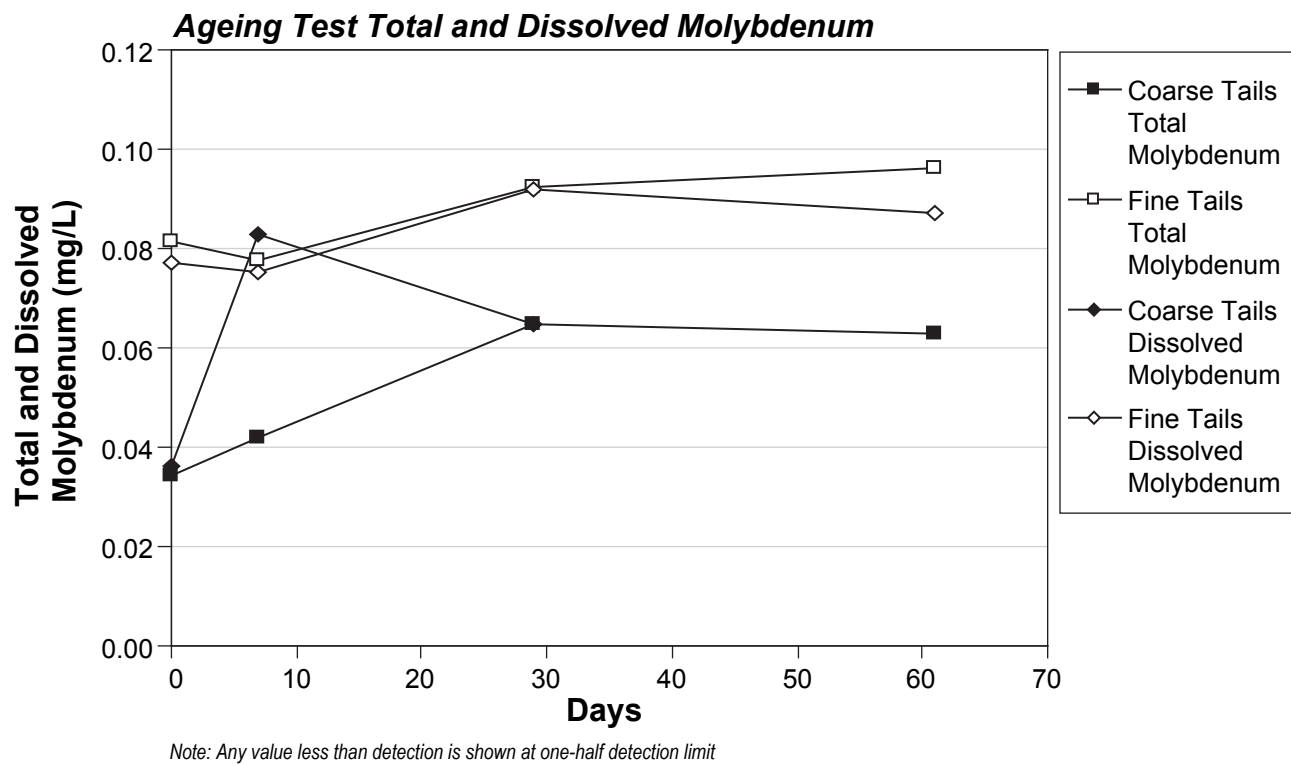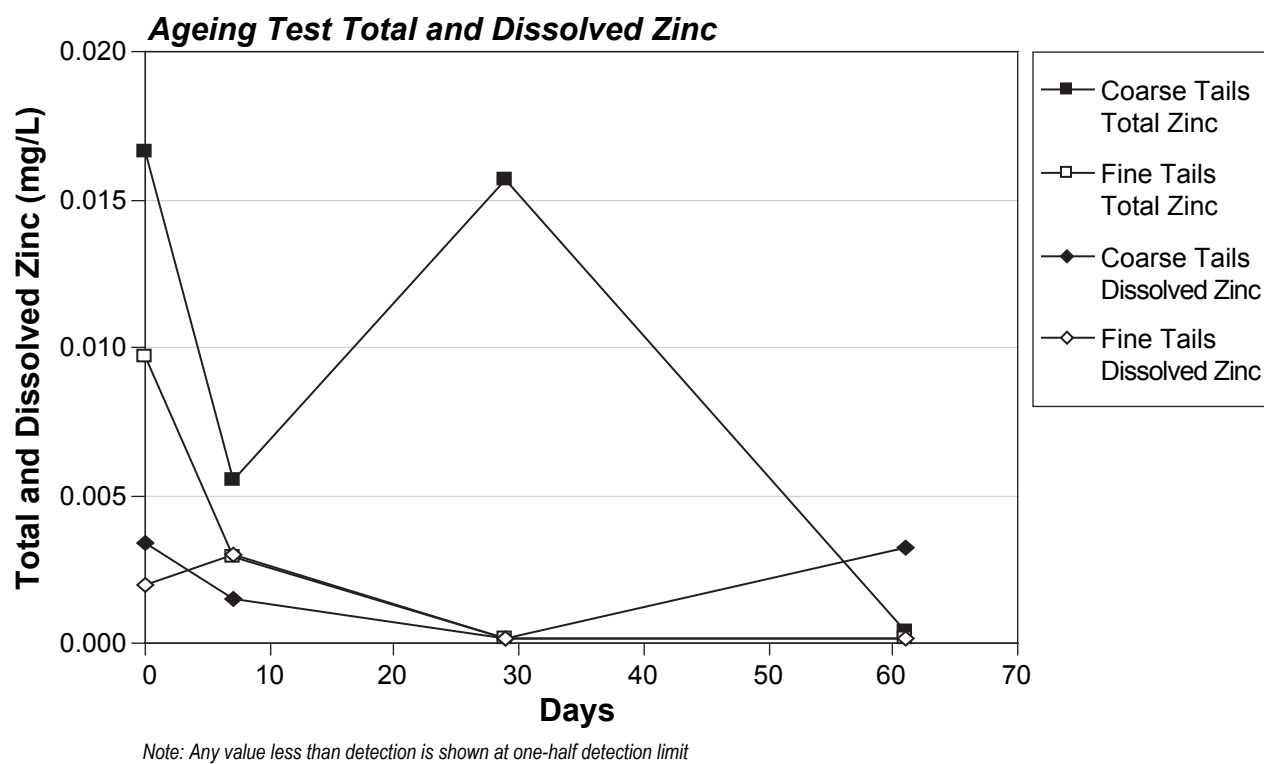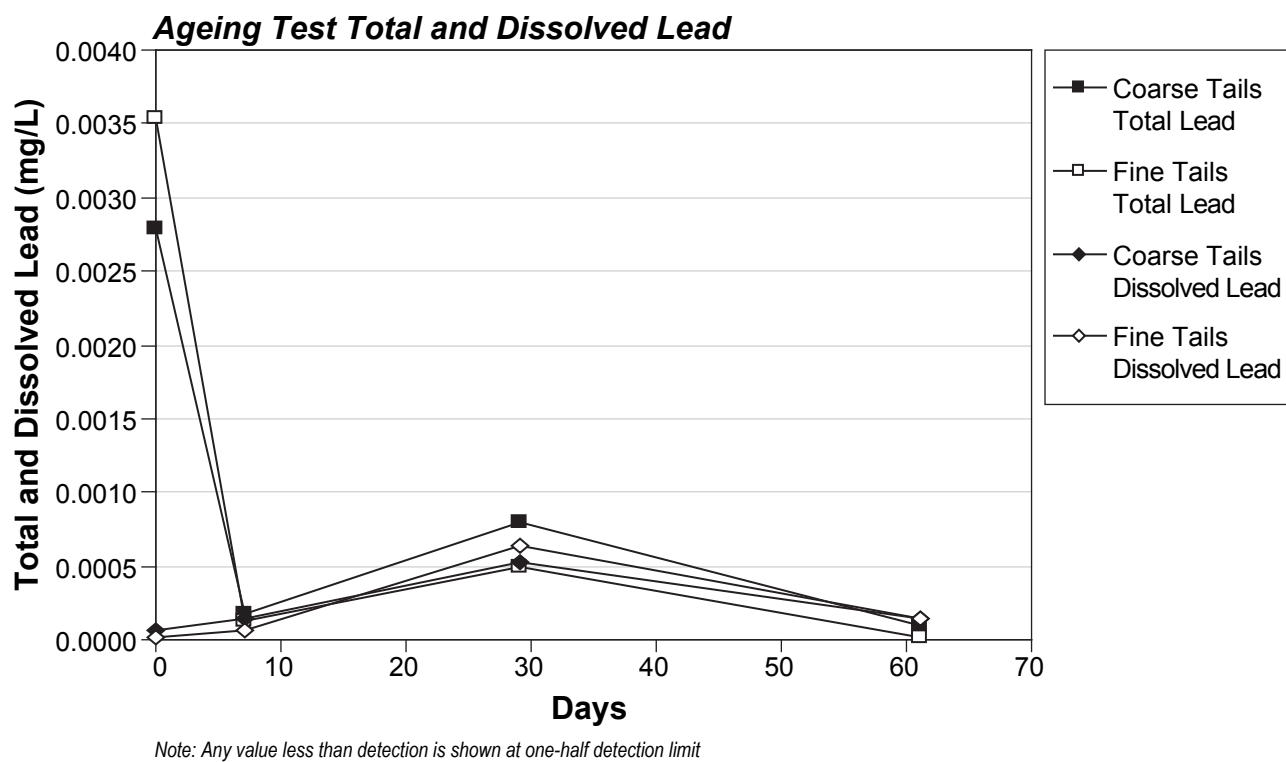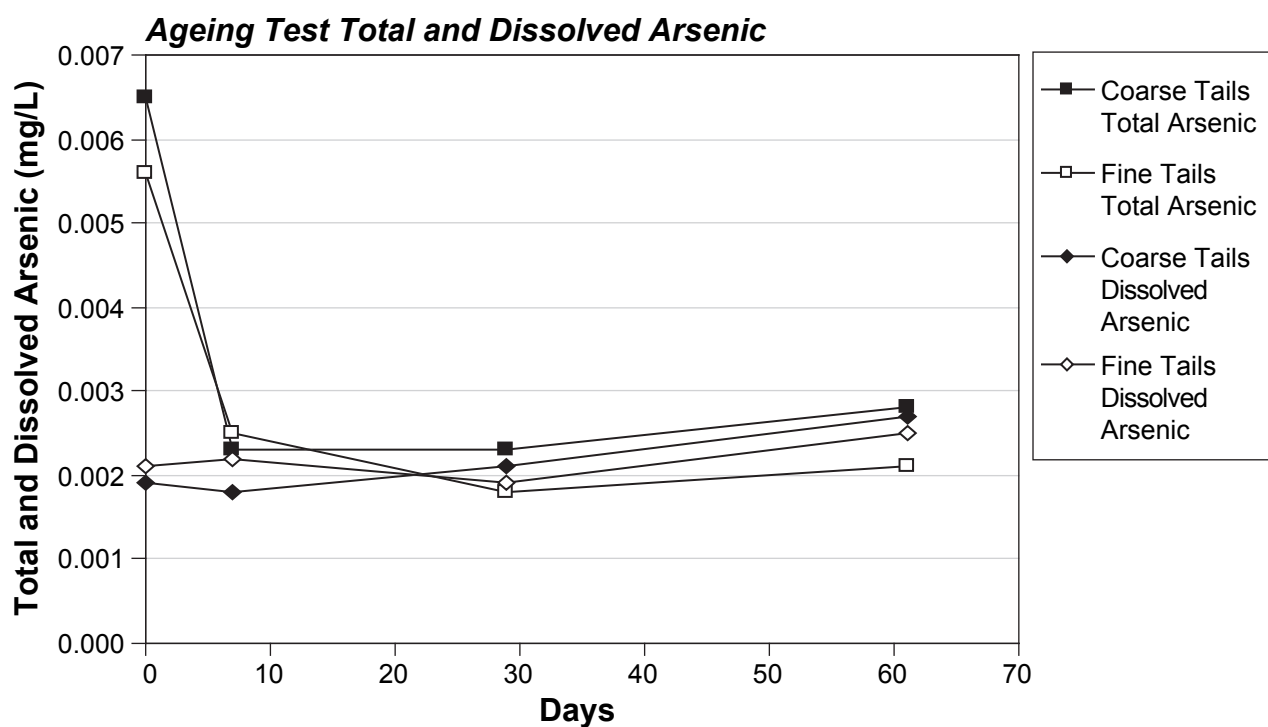
Ageing Test Total and Dissolved Arsenic

Note: Any value less than detection is shown at one-half detection limit

Morrison Copper/Gold Project
Morrison Tailings Ageing Test
Total and Dissolved Arsenic

FIGURE 5.2-44

Ageing Test Total and Dissolved Antimony

Note: Any value less than detection is shown at one-half detection limit

Legend:
- Coarse Tails Total Antimony
- Fine Tails Total Antimony
- Coarse Tails Dissolved Antimony
- Fine Tails Dissolved Antimony

Morrison Copper/Gold Project
Morrison Tailings Ageing Test
Total and Dissolved Antimony

FIGURE 5.2-45

Ageing Test Total and Dissolved Cadmium

Note: Any value less than detection is shown at one-half detection limit

Morrison Copper/Gold Project
Morrison Tailings Ageing Test
Total and Dissolved Cadmium

FIGURE 5.2-46

Ageing Test Total and Dissolved Selenium

Note: Any value less than detection is shown at one-half detection limit

Morrison Copper/Gold Project
Morrison Tailings Ageing Test
Total and Dissolved Selenium

FIGURE 5.2-47

Coarse Tailings Total:Dissolved Ratios in Ageing Tests

Legend: Hg, Ag, Al, As, Ba, Be, B, Bi, Ca, Cd, Co, Cr, Cu, Fe, K, Li, Mg, Mn, Mo, Na, Ni, Pb, Sb, Se, Si, Sn, Ti, V, Zn

Y-axis: Total:Dissolved Ratios
X-axis: Time (Days)
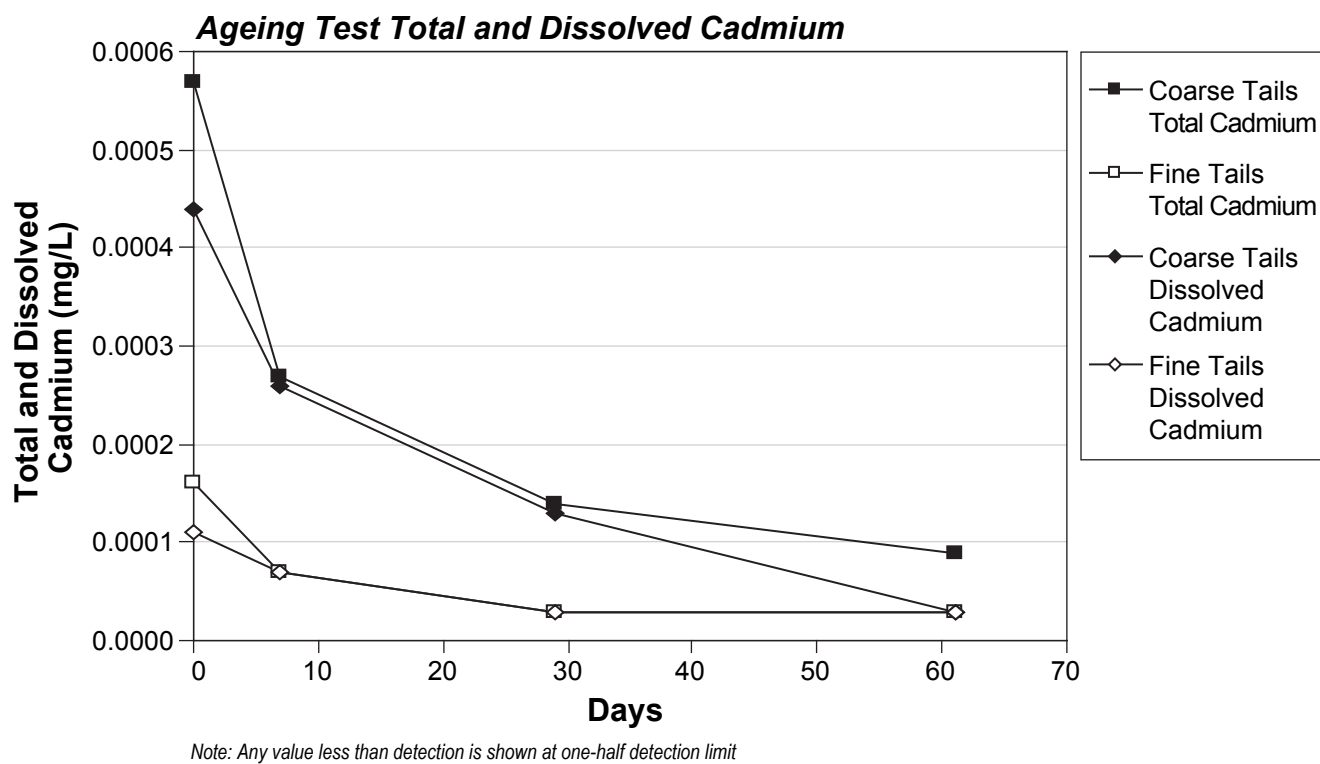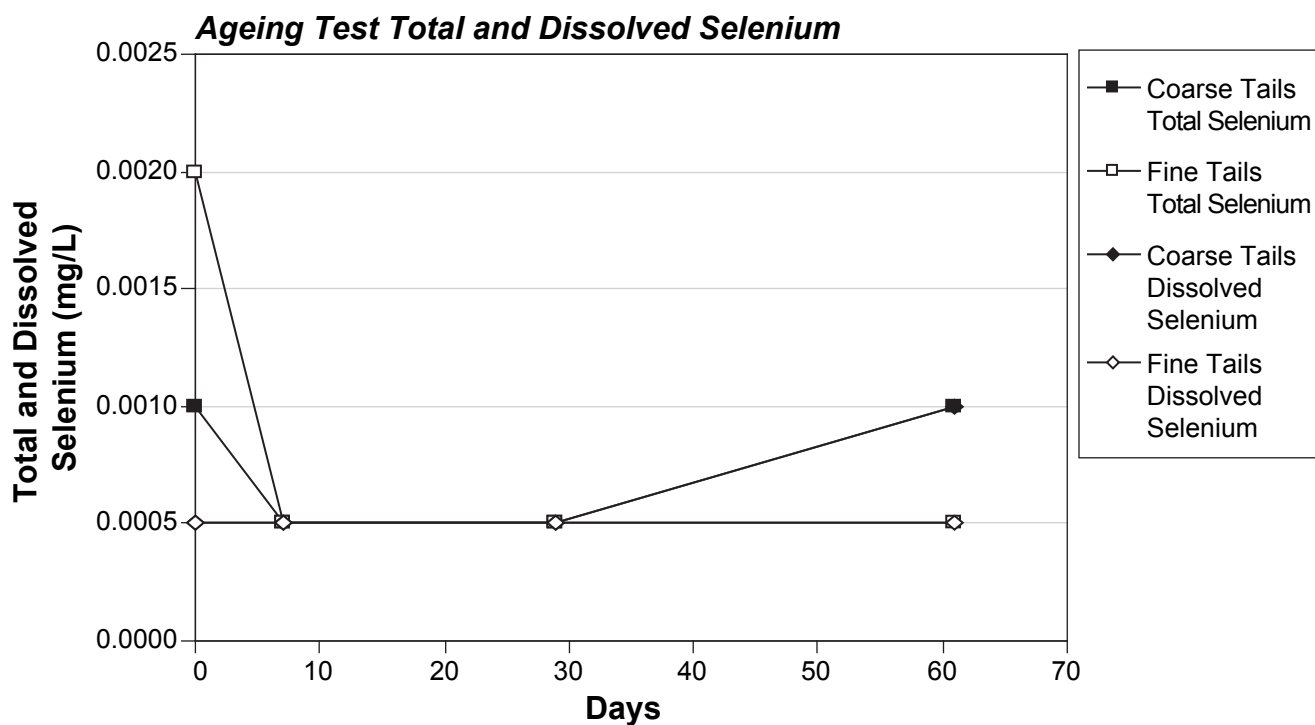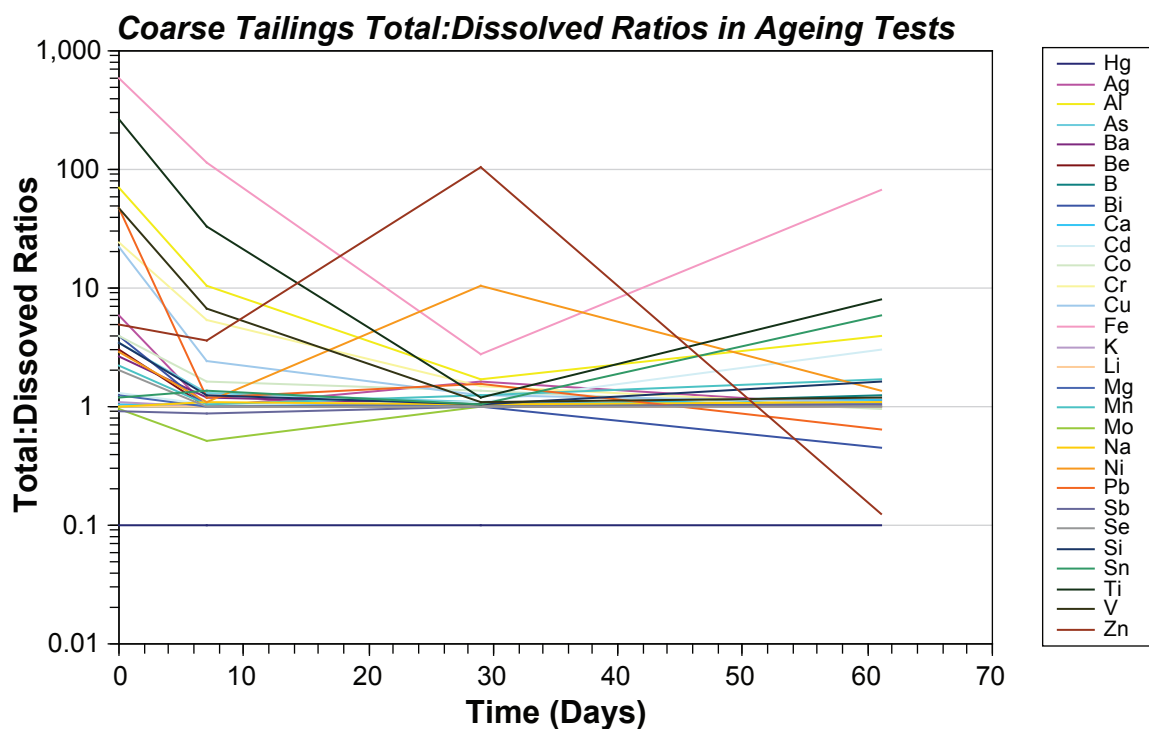
Morrison Copper/Gold Project
Morrison Coarse Tailings Fraction Ageing
Test Supernatant Total: Dissolved Ratios

FIGURE 5.2-48
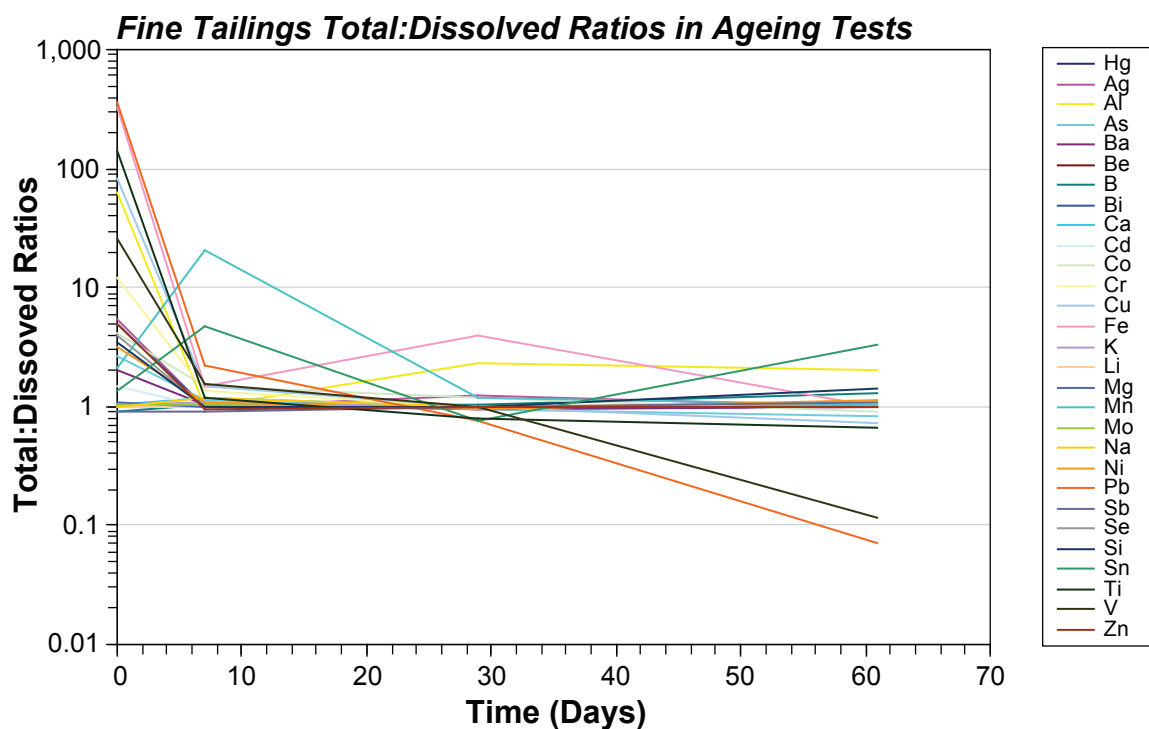
Morrison Copper/Gold Project
Morrison Fine Tailings Fraction Ageing
Test Supernatant Total: Dissolved Ratios

FIGURE 5.2-49

6.   Mine Component Drainage Quality

Site-wide semi-empirical water quality modelling was undertaken to predict the water quality during operations and post closure for the following mine components:

1. TSF pond water;

2. downstream surface waters from the TSF at stations MCS-7, MCS-8, and MCS-10 affected by TSF seepage;

3. overburden leachate;

4. WRD leachate;

5. Pit Lake.

As no reliable site-specific kinetic data are available at this time for empirical water quality modelling, analogue data from nearby Granisle mine and the empirical drainage chemistry model (EDCM) from Morin and Hutt (Appendix A) were used to develop source term water quality inputs for predicting water quality modeling at a range of pH values. Further details of Morrison water quality methodology and modelling results can be found in Rescan's water quality/water balance model (Rescan 2009c). Rationale for using Granisle mine site water quality as source terms for predictions are discussed below.

6.1   Granisle Mine Site Geology and Mineralogy

One of the reasons Granisle mine site water quality was chosen was due to similarities in geology and mineralogy sourced from MINFILE (1985).

Copper mineralization of the Granisle Mine site deposit is associated with a series of Eocene Babine Intrusions that occur in the central part of the island. The oldest is an elliptical plug of dark grey quartz diorite approximately 300 by 500 m in plan. The most important intrusions are BFPs of several distinct phases that overlap the period of mineralization. The largest and oldest is a wide northeastly trending dike that is intrusive into the western edge of the quartz diorite pluton.

An oval zone of potassic alteration is coincident with the ore zone. The main alteration product is secondary biotite. This potassic alteration zone is gradational outward to a quartz-sericite-carbonate-pyrite zone that is roughly coaxial with the ore zone. Within this zone, the intrusive and volcanic rocks are weathered to a uniform buff colour with abundant fine-grained quartz. Mafic minerals are altered to sericite and carbonate with plagioclase clouded by sericite. Pyrite occurs as disseminations or as fracture-fillings. Beyond the pyrite halo, varying degrees of propylitic alteration occurs in the volcanics with chlorite, carbonate, and epidote in the matrix and carbonate-pyrite in fractured zones. Clay mineral alteration is confined to narrow gouge in the fault zones.

The principal sulphide minerals within the ore zone are chalcopyrite, bornite, and pyrite. Coarse-grained chalcopyrite is widespread, occurring principally in quartz-filled fractures with preferred orientations of 035 to 060 degrees and 300 to 330 degrees with near vertical dips. Bornite is

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widespread in the southern half of the ore zone with veins up to 0.3 m wide hosting coarse-grained bornite, chalcopyrite, quartz, biotite, and apatite.

Molybdenite occurs within the ore zone, most commonly in drusy quartz veinlets that appear to be later than the main stage of mineralization. Magnetite and specularite are common in the north half of the ore zone where they occur in fractures with chalcopyrite and pyrite. Pyrite occurs in greatest concentrations peripheral to the ore body as blebs, stringers, and disseminations.

Mining at Granisle was suspended in mid-1982. Production from 1966 to 1982 totalled 52,273,151 t yielding 69,752,525 grams of silver, 6,832,716 grams of gold, 214,299,455 kilograms of copper, and 6,582 kilograms molybdenum.

6.2 Granisle Mine site Water Quality as an Analogue for Morrison Rock Drainage

The Granisle Mine site EDCM (Appendix A) was developed using approximately 1,200 water analyses from February 1980 to May 1999. The EDCM uses a least-linear fitting methodology, which is equivalent to calculating average annual values from 12 monthly values. The mine drainage database and analyses included ditches, ponds, the pit, the tailings pond, and groundwater monitoring wells around the mine site. Because Granisle closed in 1982, most analyses in the mine drainage dataset were during post-closure years.

Morin and Hutt (Appendix A) compared Morrison deposit rock humidity cell and LTP leachate data to the Granisle EDCM and found near-comparable (within an order of magnitude) concentrations at a low pH of 4.98 indicating equilibrium was attained in the Morrison deposit rock testing at least during the initial flushing of humidity cell MO-00-19 (pH 4.98). This assumes Granisle Mine site-wide water quality has reached or is approaching large-scale equilibrium as well, a reasonable assumption given the 16 years of weathering during operations and an additional 27 years of weathering after closure. Note that Morin and Hutt (2007) compared both Bell Mine EDCM data and Granisle EDCM data to available Morrison kinetic data at equilibrium (i.e., for pH 4.98 as discussed above) and concluded that both the Bell and Granisle EDCM can be used as predictors of Morrison drainage chemistry for acidic and alkaline pH. Justification for the use of the Granisle EDCM as an analogue for the Morrison project include:

- similar geology and mineralogy between Granisle and Morrison rock;
- the majority of Granisle data supporting the EDCM is from post-closure which assists in modelling the Morrison Pit Lake water quality (WQ);
- a larger list of predictable parameters is available in the Granisle EDCM versus the Bell EDCM as presented in Appendix A;
- the Granisle EDCM provides a conservative (i.e., worst case higher concentration) prediction of rock drainage chemistry at a range of pHs versus the Bell EDCM.

The advantage of the Granisle Mine site drainage quality dataset is the wide-ranging pH values found on-site (pH ~2.5 ~10). The Granisle EDCM requires pH inputs to predict a particular

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parameter. Note that the modeling used average annual concentrations from the Granisle EDCM (Rescan 9c).

ABA data are material balances, however in the absence of site-specific kinetic data which can provide insight into reactivity, adjusted SNPRs were considered reasonable estimations of relative reactivity for the Morrison project water quality modeling at this time. The relative reactivity of Morrison deposit rock was assumed to be related to weighted adjusted SNPRs (see Section 5.1.1.4) and was categorised using best professional judgment. This relative reactivity assumption allows for the mass balance approach to be used in the absence of site-specific kinetic data at equilibrium conditions. Water quality of a particular pH (i.e., pH 3, pH 4.5, pH 5.5, and pH 7.0) as determined by the Granisle EDCM was then assigned to the different adjusted SNPR categories. These selected pH values, and thus rock drainage water quality, were chosen with the objectives of assigning the worst (acidic) water quality (pH 3) to those materials that had very little NP (more reactive PAG) relative to sulphide content (i.e., adjusted SNPR <0.5). Less reactive rock was then assigned a progressively better water quality. Complete assignments are listed below and include

- assigning the worst (acidic) water quality (pH 3) to those materials that had very little NP (more reactive PAG) relative to sulphide content (i.e., adjusted SNPR <0.5);

- assigning moderately acidic water quality (pH 4.5) to less reactive PAG rock (0.5 < weighted adjusted SNPR <1.5);

- assigning weakly acidic water quality (pH 5.5) to lesser reactive rock (1.5 < weighted adjusted SNPR <2.5);

- assigning neutral water quality (pH 7.0) to neutral ML rock (weighted adjusted SNPR >2.5).

Note that the modeling approach was conservative in assigning adjusted SNPR categories and associated water quality. This conservatism results in modeling additional rock (i.e., adjusted SNPR of 2.0-2.5) generating mildly acidic pHs with its associated water quality.

Some parameters, like copper, show markedly different aqueous solubility at lower pH values while others, like molybdenum, do not show relationships with pH. In the latter cases, a maximum value was selected from available Granisle Mine site drainage quality dataset (Morin and Hutt 2003). Although calculated ARD lag times are as short as approximately 15 years (see Section 5.2.1.2 Table 5.2-1) and apply directly to the rock sample within the humidity cell, a conservative approach to water quality modelling (Rescan 2009c) of Morrison deposit rock was used and assumes mine materials exhibit water quality shown in Table 6.2-1 as soon as it is excavated and placed in the waste storage facility or pit walls are exposed. At a mine scale, each block of Morrison rock will have a distinct lag time that ranges from years to decades to centuries or longer. Justification for selecting a lag time of zero is due to the lack of site-specific kinetic data to the contrary and the possible excavation of materials with near zero lag times in the early phases of mining. In addition, in order to avoid being overly conservative, a range of pHs and associated water quality, derived from the Granisle EDCM, was selected for the different adjusted SNPR categories.

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Table 6.2-1
Granisle Minesite EDCM Results as Analogues for Morrison Deposit
Rock Water Quality Dissolved Metal(loids)

Parameter	Units	Adj-SNPR <0.5	Adj-SNPR 0.5 to 1.5	Adj-SNPR 1.5 to 2.5	Adj-SNPR >2.5
		pH 3	pH 4.5	pH 5.5	pH 7
Sulphate	mg/L	10,814	2,111	1,834	1,486
TDS	mg/L	12,995	3,040	2,661	2,221
Fluoride	mg/L	3	3	3	3
Chloride	mg/L	30	30	30	30
Nitrite	mg/L	0.62	0.62	0.62	0.62
Nitrate	mg/L	217	217	217	217
Ammonia	mg/L	0.0072	0.0072	0.0072	0.0072
Aluminum	mg/L	738	5.9	1.3	0.14
Antimony	mg/L	0.1	0.1	0.1	0.1
Arsenic	mg/L	2.7	0.1	0.1	0.1
Barium	mg/L	0.032	0.032	0.032	0.032
Beryllium	mg/L	0.014	0.0070	0.0044	0.0022
Bismuth	mg/L	0.05	0.05	0.05	0.05
Cadmium	mg/L	0.024	0.012	0.0077	0.0038
Calcium	mg/L	275	275	275	275
Chromium	mg/L	0.078	0.030	0.016	0.0060
Cobalt	mg/L	7.2	0.16	0.058	0.013
Copper	mg/L	162	1.2	0.40	0.081
Iron	mg/L	83	1.2	0.34	0.053
Lead	mg/L	0.28	0.099	0.049	0.017
Lithium	mg/L	0.058	0.029	0.018	0.0093
Magnesium	mg/L	522	298	204	116
Manganese	mg/L	14	4.9	2.4	0.83
Mercury	mg/L	0.00001	0.00001	0.00001	0.00001
Molybdenum	mg/L	0.1	0.1	0.1	0.0275
Nickel	mg/L	2.7	0.13	0.064	0.022
Potassium	mg/L	6.3	6.3	6.3	6.3
Selenium	mg/L	0.1	0.1	0.1	0.1
Silicon	mg/L	18.7	3.7	3.7	3.7
Silver	mg/L	0.1	0.1	0.1	0.1
Sodium	mg/L	80	80	80	80
Tin	mg/L	0.15	0.15	0.15	0.15
Titanium	mg/L	1	1	1	1
Thalium	mg/L	<0.3	<0.3	<0.3	<0.3

(continued)

Table 6.2-1
Granisle Minesite EDCM Results as Analogues for Morrison Deposit
Rock Water Quality Dissolved Metal(loids) (completed)

Parameter	Units	Adj-SNPR <0.5	Adj-SNPR 0.5 to 1.5	Adj-SNPR 1.5 to 2.5	Adj-SNPR >2.5
		pH 3	pH 4.5	pH 5.5	pH 7
Vanadium	mg/L	<0.7	<0.7	<0.7	<0.7
Zinc	mg/L	16	0.22	0.12	0.046
Acidity	mg CaCO$_3$/L	3,443	128	62	21
Alkalinity	mg CaCO$_3$/L	0	2.9	11	85
TSS	mg/L	6.4	6.4	6.4	6.4

Notes:
1. Metal(loid) values are for dissolved.
2. Source of EDCM (Appendix A).
3. TDS calculated from the sum of all listed ions.
4. Estimated concentrations of nitrogen species based on another BC mine site.
5. Values in orange indicate maximum concentrations from Granisle site wide water quality (Morin and Hutt 2003)

A 15 Mt overburden stockpile is scheduled to be built on the peninsula west of the open pit. Drainage chemistry (surface runoff and seepage) originating from this mine component is assumed to be of neutral pH. Data supporting this assumption include low sulphide content (Section 5.1.1.2). As noted previously, SFE testing on copper-rich overburden from within the pit outline (Section 5.1.2) has not been done. However, SFE testing of overburden from outside the pit outline indicates near-neutral to slightly alkaline leachate and low ML (Section 5.1.3). Table 6.2-2 shows the overburden SFE data to date compared to the EDCM pH 7 SNPR >2.5 category and indicates that the water quality modelling approach is conservative with respect to parameters such as sulphate, chloride, antimony, arsenic, beryllium, bismuth, cadmium, calcium, cobalt, magnesium, molybdenum, selenium, silver, tin, and acidity by at least an order of magnitude or more. Therefore, the Granisle EDCM analogue data was used as a source-term for overburden water quality in predictive modeling (Rescan 2009c).

6.3 Nitrogen Species

Approximately 0.24 kg of ammonia nitrate fuel oil (ANFO) per tonne of ore and waste rock will be used in pit rock blasting throughout the mine's operations phase when conditions are dry or if the hole can be pumped and lined (Wardrop 2009). It is anticipated that approximately 5% of the holes will be wet where emulsion explosives will be used. An alternative explosive is a 70/30 ANFO-emulsion blend that can be used in wet or dry holes.

The generation of nitrogen species at a Morrison mine site is generally governed by the detonation of ANFO during blasting activities, producing oxides of nitrogen as unwanted by-products caused by incomplete blasting reactions. Ferguson and Leask (1998) outline a method for apportioning nitrogen species from blasting residue as determined by the ANFO mass used to blast one tonne of rock.

Table 6.2-2
Overburden Shake Flask Extraction Compared with the
Granisle EDCM pH 7 Water Quality

Parameter	Overburden Shake Flask Extraction Mean	Overburden Shake Flask Extraction Maximum	Granisle EDCM
pH	8.0	8.5	7
Sulphate	65	168	1,486
TDS	182	399	2,221
Fluoride	0.49	1.2	3
Chloride	0.37	1.3	30
Nitrite	0.034	0.075	0.62
Nitrate	0.13	1.3	217
Ammonia	NA	NA	0.0072
Aluminum	0.63	6.5	0.14
Antimony	0.00062	0.0022	0.1
Arsenic	0.0018	0.0098	0.1
Barium	0.048	0.076	0.032
Beryllium	0.00039	0.00050	0.0022
Bismuth	0.00039	0.00050	0.05
Cadmium	0.000057	0.00021	0.0038
Calcium	26	48	275
Chromium	0.0012	0.0092	0.0060
Cobalt	0.00049	0.0050	0.013
Copper	0.027	0.087	0.081
Iron	0.99	6.0	0.053
Lead	0.0017	0.010	0.017
Lithium	0.0042	0.0055	0.0093
Magnesium	7.5	17	116
Manganese	0.064	0.30	0.83
Mercury	0.000027	0.000062	0.00001
Molybdenum	0.010	0.019	0.0275
Nickel	0.0020	0.011	0.022
Potassium	2.8	5.3	6.3
Selenium	0.0036	0.0088	0.1
Silicon	6.0	16	3.7
Silver	0.000042	0.000085	0.1
Sodium	10	25	80
Tin	0.00059	0.0046	0.15
Titanium	0.036	0.21	1
Thallium			<0.3

(continued)

Table 6.2-2
Overburden Shake Flask Extraction Compared with the
Granisle EDCM pH 7 Water Quality (completed)

Parameter	Overburden Shake Flask Extraction Mean	Overburden Shake Flask Extraction Maximum	Granisle EDCM
Vanadium	0.0020	0.016	<0.7
Zinc	0.0088	0.026	0.046
Acidity	1.9	8.0	21
Alkalinity	60	94	85
TSS	NA	NA	6.4

Notes:
1. **Source of EDCM (Appendix A).**
2. **TDS in calculated from the sum of all listed ions.**
3. **Estimated concentrations of nitrogen species in EDCM column based on another BC minesite.**
4. **Values in orange indicate maximum concentrations from Granisle Mine site wide water quality (Morin and Hutt 2003).**

This approach was used to determine the mass of nitrogen species remaining in ore material after blasting to illustrate the amount of nitrogen species that can potentially end up in the tailings transport water and by extension in the TSF pond water and tailings pore water. Assumptions included the conservation of nitrogen mass through the process plant metallurgical circuit as ore is processed into concentrate and tailings are generated. In addition, as the actual material of economic interest is typically 10% of the tonnage of ore, nitrogen associated with the concentrate stream and potentially leaving the system was ignored and assumed to report to the tailings (typically 90% of the ore tonnage throughput). A water balance of 2,500 m^3/hr (i.e., average tailings transport water rate in year 10 of operations) and a 30,000 tonne process plant throughput rate were used in calculating the nitrogen species of interest (Table 6.3-1). Preliminary estimations indicate very high nitrate-nitrogen and nitrite-nitrogen , which seems high compared to water quality from actual mining operations, but reasonable ammonia-nitrogen estimates. However, further considerations of nitrogen species geochemistry at a BC mine site are provided below that preclude the use of the mass balance approach in estimating nitrogen species. In addition, investigations of actual mine site data was undertaken to provide a comparison to the mass balance approach and showed that the mass balance approach may overestimate predicted nitrate and nitrite nitrogen species estimates.

Table 6.3-1
Estimated Nitrogen Species in Tailings Transport Water

Tailings Process Waters (mg/L)		
Nitrate-N	Ammonia-N	Nitrite-N
453	57	10

Although a mass-water balance can be developed for nitrogen species for the tonnage of pit rock being placed in a WRD or low-grade ore stockpile, there are large uncertainties in estimating the hydrologic response of a large-scale constructed mine component and the loadings actually available that would contact either infiltration or runoff water. Additionally, the proportion of the dump/stockpile that is active in terms of contributing loadings can be considered as the shell initially but becomes more complicated as the dump/stockpile grows and flow pathways are established, disturbed, isolated or re-established. Geochemical processes regarding nitrogen species (e.g., reduction-oxidation transformations, volatilization, etc) also reduce the confidence in predicting drainage chemistry. As mentioned earlier, the assumption of mass conservation through the process plant metallurgical process for predicting tailings transport water is also a considerable assumption.

Instead, nitrogen species (i.e., nitrate, ammonia, and nitrite) concentrations for Morrison rock and tailings contact water used for predictive water quality modeling (Rescan 2009c) were assigned analogue water quality data from a BC mine site (Table 6.3-2). Note that the analogue mine site nitrogen species differ dramatically with respect to ammonia. This is likely caused by the reducing conditions maintained after rapid burial of tailings while surface drainage ammonia is allowed to oxidize to nitrate under aerobic conditions or simply volatize. Nitrogen reduction in tailings pore water is also a likely mechanism contributing to the ammonia budget. It is reasonable to expect Morrison drainage from WRDs, low-grade ore stockpiles, pit walls, and tailings pond water will be well mixed and aerobic. For the purposes of water quality predictions, solution chemistry of Morrison mine components are assumed to have the average concentrations seen in analogue Mine A water quality. As a conservative approach, a rate of decay for ammonia, nitrate, or nitrite was not considered in the water quality modelling.

Table 6.3-2
Mine A Nitrogen Species Water Quality as an Analogue for Morrison Mine Components

Morrison Component	Mine A Analogue	Average			Maximum		
		Nitrate-N (mg/L)	Ammonia-N (mg/L)	Nitrite-N (mg/L)	Nitrate-N (mg/L)	Ammonia-N (mg/L)	Nitrite-N (mg/L)
Tailings Porewaters	Tailings Pore Waters	22	54	1.0	45	113	3.5
Tailings Pond Water, Cycloned Sand Runoff and Waste Rock Dump, Overburden Stockpile and Pit Wall Drainage	Creek Draining Pit and Waste Rock Dump Catchment	49	0.0059	0.14	90	0.020	0.25

6.4   Morrison Rock Solid-phase Elemental Reservoirs

To recognize that predictive water quality modeling can not assume an infinite source of elements for leaching, finite elemental reservoirs in mine components available for leaching were estimated for the WRD, exposed pit wall at final lake elevation, and overburden stockpile. Cumulative mass releases for these mine components were checked at selected time steps in water quality modelling to determine if elements were indeed available for potential leaching contributing to predicted water quality.

Note that waste rock and overburden average elemental concentrations from the ML/ARD database (Section 5.1.2) and the tonnage of designated weighted adjusted SNPR categories (Section 3.3, Table 3.3-1, and Section 5.1.1.4) were used to develop an estimate of the total elemental reservoir available for potential ML and sulphide oxidation (Table 6.4-1).

Similarly, an average pit wall rock density of 2.7 tonne/m^3, a blast influence of 20 m and the approximate area of each designated adj-SNPR category (see Sections 3.4 and 5.1.1.4) above the final pit lake elevation (~728 m), a tonnage of rock can be estimated for potential leaching. Average elemental abundance from the ML/ARD pit rock waste rock (i.e., CuEq <0.2%) database (Section 5.1.2) were then used to estimate the potential elemental reservoir available for leaching in the 20-m thick pit shell (Table 6.4-2).

Also included (Tables 6.4-1 and 6.4-2) is a comparison against the same mass of overburden, waste rock, and pit wall but assuming average whole crustal abundance according to Price (1997) and indicates reservoir estimates are not unreasonable.

6.5   Tailings Pond and Pore-water

Tailings slurry water (which contributes to the tailings pond and is trapped as tailings porewater) source terms for water quality modelling (Table 6.5.1) used a combination of aging tests results and estimates of pond water residence time. Klohn Crippen Berger (2008) indicate the TSF pond has a volume of approximately 1.25 Mm^3 at end of Year 10 of operations with a reclaim rate of 2,031 m^3/hr. This results in a residence time of 26 days. At the end of Year 20 the pond is 2.34 Mm^3 with an annual reclaim rate of 1,983 m^3/hr. The larger pond volume increases the residence time to 49 days. Nevertheless, as a conservative approach, ageing test data from Day 1 was used for the mass loading contributions from tailings slurry water. Note that nitrogen species concentrations for water quality modelling are discussed in Section 6.3 and listed in Table 6.3-2.

The use of small scale lab ageing test data as a source term for tailings pond water (i.e., tailings transport water aka process water) and tailings pore-water in predictive water quality modeling (Rescan 2009c) instead of analogue Granisle EDCM data was used due to the different metallurgical processes proposed for the Morrison Project (e.g., High Pressure Grinding). It is believed that the ageing test data is more representative of the predicted tailings pond and pore-water chemistry during operations and post closure. It is interesting to note that the High Pressure Grinding process, presumably an aggressive process resulting in the exposure ofhigh surface area produces relatively good quality slurry waters (Section 5.2.1.5) in agreement with the not-PAG classification of tailings.

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Metal Leaching and Acid Rock Drainage Characterization Report

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Pacific Booker Minerals Inc.

Rescan™ Environmental Services Ltd. (Proj. #0793-001-02)

Table 6.4-1

Morrison Copper/Gold Project: Estimated Elemental Reservoirs of Morrison Overburden and Waste Rock Dump Pit Wall Available for Metal Leaching and Sulphide Oxidation

Parameter	Whole Crustal Abundance (ppm)	Overburden (ppm)	Waste								Total Waste Rock (tonnes)	Total Waste Rock at Whole Crustal Abundance (tonnes)	Overburden (tonnes)	Overburden at Whole Crustal Abundance (tonnes)
			Adj SNPR 0 to 0.5 (ppm)	Adj SNPR 0.5 to 1.5 (ppm)	Adj SNPR 1.5 to 2.5 (ppm)	Adj SNPR 2.5+ (ppm)	Adj SNPR 0 to 0.5 (tonnes)	Adj SNPR 0.5 to 1.5 (tonnes)	Adj SNPR 1.5 to 2.5 (tonnes)	Adj SNPR 2.5+ (tonnes)				
Ag	0.080	0.42	0.82	1.20	1.24	0.77	9.6	26	111	35	182	14	6.3	1.2
Al	83,600	69,214	79,554	81,076	80,821	88,144	929,911	1,770,373	7,258,966	4,022,537	13,981,787	14,126,394	1,048,312	1,266,206
As	1.8	29	60	212.5	154.6	201	702	4,641	13,888	9,191	28,422	304	443	27
Ba	390	878	1,374	734	677	779	16,063	16,038	60,839	35,555	128,495	65,901	13,302	5,907
Be	2.0	1.1	1.1	1.2	1.31	1.49	13	27	117	68	225	338	16	30
Bi	0.0082	0.43	0.41	0.67	0.54	0.55	4.7	15	49	25	93	1	6.6	0.12
Ca	46,600	12,659	31,661	20,941	16,303	4,721	370,090	457,276	1,464,279	215,457	2,507,102	7,874,282	191,727	705,804
Cd	0.16	0.33	0.391	0.46	1.29	0.245	4.6	10.0	116	11	142	27	5.0	2.4
Co	29	17	16	18	22	23	184	396	1,948	1,041	3,569	4,900	251	439
Cr	122	64	83	91	104	99	973	1,992	9,383	4,531	16,880	20,615	968	1,848
Cu	68	469	990	1,201	1,343	681	11,577	26,220	120,616	31,071	189,483	11,490	7,104	1,030
Fe	62,200	40,026	38,796	39,203	39,375	40,355	453,492	856,046	3,536,502	1,841,620	6,687,661	10,510,307	606,232	942,081
Hg	0.086	0.23	0.402	1.241	0.509	0.671	4.7	27	46	31	108	15	3.4	1.3
K	18,400	10,493	11,886	14,648	13,067	15,412	138,935	319,860	1,173,632	703,348	2,335,774	3,109,158	158,929	278,686
Li	18	29	27	25	29	38	317	538	2,566	1,756	5,177	3,042	442	273
Mg	27,640	6,686	13,039	12,466	10,377	7,500	152,408	272,197	932,015	342,270	1,698,890	4,670,497	101,269	418,635
Mn	1,060	823	678	719	472	664	7,929	15,698	42,354	30,324	96,305	179,115	12,460	16,055
Mo	1.2	12	26.5	11.7	17.5	4.6	310	255	1,570	212	2,347	203	177	18
Na	22,700	13,845	14,896	13,286	10,377	6,441	174,125	290,118	932,015	293,937	1,690,195	3,835,755	209,694	343,814
Ni	99	41	57	74	66	73	670	1,607	5,900	3,338	11,515	16,729	619	1,499
P	1,120	782	1,182	973	1,321	704	13,818	21,249	118,629	32,132	185,828	189,253	11,848	16,964
Pb	13	14	34.0	113.1	16.9	24.6	397	2,470	1,521	1,122	5,510	2,197	213	197
S	340	1,241	11,081	12,007	11,943	12,561	129,522	262,183	1,072,627	573,216	2,037,547	57,452	18,802	5,150
Sb	0.20	1.7	13.9	8.31	6.41	3.89	162	181	575	177	1,096	34	26	3.0
Se	0.050	2.2	3.1	3.1	3.0	2.8	36	69	265	125	495	8	33	0.76
Sn	2.1	1.2	1.9	1.9	1.8	1.8	22	41	162	80	305	355	18	32
Sr	384	273	532	679	498	408	6,220	14,821	44,688	18,613	84,342	64,887	4,131	5,816
Ti	6,320	3,679	3,864	3,788	3,786	3,807	45,167	82,713	340,045	173,721	641,646	1,067,928	55,719	95,723
Tl	0.72	0.40	0.56	0.62	0.78	1.16	6.6	14	70	53	143	122	6.1	11
U	2.3	1.7	1.8	1.8	2.2	2.1	21	40	193	95	350	389	25	35
V	136	112	130	134	137	140	1,514	2,930	12,315	6,381	23,140	22,981	1,691	2,060
Zn	76	103	159	630	123	142	1,856	13,749	11,035	6,496	33,137	12,842	1,553	1,151

Notes:
1. Source of average whole crustal abundance from Price (1997).
2. Elemental values listed are means according to the ML-ARD database.

Table 6.4-2
Estimated Elemental Reservoirs of the Morrison Pit Wall Available for Metal Leaching and Sulphide Oxidation

Parameter	Whole Crustal Abundance ppm	Adj-SNPR 0 to 0.5 ppm	Adj-SNPR 0.5 to 1.5 ppm	Adj-SNPR 1.5 to 2.5 ppm	Adj-SNPR 2.5+ ppm	Adj-SNPR 0 to 0.5 tonnes	Adj-SNPR 0.5 to 1.5 tonnes	Adj-SNPR 1.5 to 2.5 tonnes	Adj-SNPR 2.5+ tonnes	Pit Wall Total tonnes	Average Whole Crustal tonnes
Ag	0.080	0.82	1.2	1.2	0.77	8.4	13	13	8.9	48	2.6
Al	83,600	79,554	81,076	80,821	88,144	957,285	877,758	880,523	863,999	3,909,161	2,728,142
As	1.8	60	213	155	201	2,187	1,679	2,308	653	7,456	59
Ba	390	1,374	734	677	779	8,461	7,357	7,977	14,925	42,285	12,727
Be	2.0	1.1	1.2	1.3	1.5	16	14	13	12	61	65
Bi	0.0082	0.41	0.67	0.54	0.55	6.0	5.9	7.3	4.4	26	0.27
Ca	46,600	31,661	20,941	16,303	4,721	51,275	177,061	227,433	343,858	873,255	1,520,711
Cd	0.16	0.39	0.46	1.3	0.25	2.7	14	5.0	4.2	28	5.2
Co	29	16	18	22	23	248	236	197	171	930	946
Cr	122	83	91	104	99	1,078	1,135	991	904	4,486	3,981
Cu	68	990	1,201	1,343	681	7,394	14,585	13,041	10,756	49,991	2,219
Fe	62,200	38,796	39,203	39,375	40,355	438,270	427,636	425,768	421,349	1,870,752	2,029,790
Hg	0.086	0.40	1.2	0.51	0.67	7.3	5.5	13	4.4	33	2.8
K	18,400	11,886	14,648	13,067	15,412	167,383	141,916	159,087	129,087	652,487	600,452
Li	18	27	25	29	38	418	310	268	295	1,409	587
Mg	27,640	13,039	12,466	10,377	7,500	81,454	112,700	135,382	141,605	514,521	901,984
Mn	1,060	678	719	472	664	7,216	5,122	7,808	7,367	30,046	34,591
Mo	1.2	27	12	17	4.6	50	190	127	288	715	39
Na	22,700	14,896	13,286	10,377	6,441	69,951	112,700	144,295	161,783	533,729	740,775
Ni	99	57	74	66	73	794	713	799	622	3,199	3,231
P	1,120	1,182	973	1,321	704	7,647	14,345	10,568	12,838	49,578	36,549
Pb	13	34	113	17	25	267	184	1,229	369	2,237	424
S	340	11,081	12,007	11,943	12,561	136,414	129,703	130,401	120,342	564,450	11,095
Sb	0.20	14	8.3	6.4	3.9	42	70	90	151	385	6.5
Se	0.050	3.1	3.1	3.0	2.8	30	32	34	33	141	1.6
Sn	2.1	1.9	1.9	1.8	1.8	19	20	20	21	87	69
Sr	384	532	679	498	408	4,430	5,404	7,371	5,779	25,100	12,531
Ti	6,320	3,864	3,788	3,786	3,807	41,342	41,118	41,139	41,965	180,809	206,242
Tl	0.72	0.56	0.62	0.78	1.2	13	8.4	6.7	6.1	37	23
U	2.3	1.8	1.8	2.2	2.1	23	23	20	20	94	75
V	136	130	134	137	140	1,518	1,489	1,457	1,407	6,413	4,438
Zn	76	159	630	123	142	1,546	1,334	6,838	1,725	12,497	2,480

Notes:
1. Source of average whole crustal abundance from Price (1997).
2. Elemental values listed are means according to the ML-ARD database.

Table 6.5-1
Tailings Slurry Water and Tailings Porewater Quality for
Water Quality Modeling

Parameter	Units	Ageing Test Day 1
		65% Coarse : 35% Fine
Acidity	mg CaCO$_3$/L	1
Alkalinity	mg CaCO$_3$/L	85
TDS	mg/L	288
TSS	mg/L	154
F	mg/L	0.36
Cl	mg/L	35
SO$_4$	mg/L	67
Dissolved		
Hg	mg/L	0.00034
Ag	mg/L	0.000015
Al	mg/L	0.037
As	mg/L	0.0020
Ba	mg/L	0.15
Be	mg/L	0.000020
B	mg/L	0.027
Bi	mg/L	0.000010
Ca	mg/L	37
Cd	mg/L	0.00032
Co	mg/L	0.00040
Cr	mg/L	0.00050
Cu	mg/L	0.0018
Fe	mg/L	0.010
K	mg/L	13
Li	mg/L	0.0010
Mg	mg/L	9.0
Mn	mg/L	0.037
Mo	mg/L	0.051
Na	mg/L	31
Ni	mg/L	0.0050
Pb	mg/L	0.000043
Sb	mg/L	0.0047
Se	mg/L	0.00050
Si	mg/L	1.8
Sn	mg/L	0.0035
Ti	mg/L	0.00080
V	mg/L	0.00027
Zn	mg/L	0.0029

Notes:
1.Source SGS (2007)

6.6   Tailings Beach Contact Water

Throughout the mine life, a whole tailings beach will be present. Contact water through precipitation, runoff, and infiltration will report to the tailings pond. Although humidity cell data are available for coarse and fine fractions, which can then be weighted according to the whole tailings estimates (65% coarse and 35 % fine) according to Klohn Crippen Berger (2008), comparisons with the Granisle EDCM indicate laboratory data for leachate maximums are at least an order of magnitude lower and as much as three orders at near neutral pH (Table 6.6-1). As a conservative approach, the EDCM data for pH 7 was used in site-wide water quality modelling (Rescan 2009c).

Table 6.6-1
Humidity Cell Leachate Results for Whole Tailings
(65% Coarse Tailings Fraction and 35% Tailings Fraction)

Parameter	Units	Humidity Cell		Adj-SNPR >2.5
		Last 25 Weeks of Testing	Maximum	pH 7[1]
Conductivity	microS/cm	65	129	
pH	no units	7.1	7.9[2]	7.0
Acidity (as CaCO3)	mgCaCO$_3$/L	1.0	8.0	21
Alkalinity, Total (as CaCO3)	mgCaCO$_3$/L	8.9	53	85
Sulfate (SO4)	mg/L	15	34	1,486
Fluoride	mg/L			3
Chloride	mg/L			30
Dissolved Elements				
Aluminum (Al)-Dissolved	mg/L	0.022	0.093	0.14
Antimony (Sb)-Dissolved	mg/L	0.00045	0.0030	0.10
Arsenic (As)-Dissolved	mg/L	0.00016	0.0023	0.10
Barium (Ba)-Dissolved	mg/L	0.21	0.30	0.032
Beryllium (Be)-Dissolved	mg/L	0.000011	0.000020	0.0022
Bismuth (Bi)-Dissolved	mg/L	0.0000050	0.000031	0.050
Cadmium (Cd)-Dissolved	mg/L	0.0000036	0.000030	0.0038
Calcium (Ca)-Dissolved	mg/L	9.6	18	275
Chromium (Cr)-Dissolved	mg/L	0.00025	0.00064	0.0060
Cobalt (Co)-Dissolved	mg/L	0.00011	0.0025	0.013
Copper (Cu)-Dissolved	mg/L	0.00085	0.0018	0.081
Iron (Fe)-Dissolved	mg/L	0.0050	0.10	0.053
Lead (Pb)-Dissolved	mg/L	0.000023	0.00019	0.017
Lithium (Li)-Dissolved	mg/L	0.00083	0.0021	0.0093
Magnesium (Mg)-Dissolved	mg/L	2.8	8.5	116
Manganese (Mn)-Dissolved	mg/L	3.1	4.3	0.83
Mercury (Hg)-Dissolved	mg/L	0.000052	0.000068	0.000010

(continued)

Table 6.6-1
Humidity Cell Leachate Results for Whole Tailings
(65% Coarse Tailings Fraction and 35% Tailings Fraction) (completed)

Parameter	Units	Humidity Cell		Adj-SNPR >2.5
		Last 25 Weeks of Testing	Maximum	pH 7[1]
Molybdenum (Mo)-Dissolved	mg/L	0.0016	0.015	0.10
Nickel (Ni)-Dissolved	mg/L	0.00083	0.0045	0.022
Potassium (K)-Dissolved	mg/L	2.2	8.1	6.3
Selenium (Se)-Dissolved	mg/L	0.00050	0.00083	0.10
Silicon (Si)-Dissolved	mg/L	0.28	1.1	5.0
Silver (Ag)-Dissolved	mg/L	0.0000083	0.000037	0.0060
Sodium (Na)-Dissolved	mg/L	0.22	10	80
Tin (Sn)-Dissolved	mg/L	0.000043	0.0013	0.15
Titanium (Ti)-Dissolved	mg/L	0.000054	0.00030	0.01
Vanadium (V)-Dissolved	mg/L	0.00010	0.00045	0.015
Zinc (Zn)-Dissolved	mg/L	0.0024	0.016	0.046

Notes:
1. Source of EDCM (Appendix A).
2. Minimum measured pH at 5.89 for Fine Tailings Fraction
3. Values in orange indicate maximum concentrations from Granisle site wide water quality (Morin and Hutt 2003)

6.7   Cycloned Tailings Sand Dam Contact Water

The TSF dams are scheduled for cycloned tailings sand construction after the start-up dam has been placed and ore begins to be processed through the process plant. Section 5.2.1.3 described the leachate characteristics of the coarse tailings fraction, which is considered representative of cycloned tailings sand construction material. However, the Granisle Mine site did not employ cycloned tailings sand construction methods. For comparison, the cycloned tailings sand humidity cell leachate data and the Granisle Mine site EDCM predictions for pH 7 are shown in Table 6.7-1. Note that the minimum pH in cycloned tailings sand leachate (i.e., coarse tailings fraction) was measured at 7.1 and generally corresponds with maximum reported concentrations. Comparisons indicate the EDCM model overestimates almost all parameters by several factors to orders of magnitude. Although cycloned tailings are classified as not-PAG (SNPR = 15) and drainage is predicted to remain near neutral, small-scale humidity cell test results may not be at equilibrium concentrations. In addition, scaling effects such as warmer laboratory conditions (i.e., 20°C) versus colder project site-specific temperatures, high liquid to solid flushing ratios, and the general increased reactivity of laboratory-scale experiments (ASTM 2007) introduces uncertainty into estimating equilibrium concentrations from small-scale humidity cell results. Therefore, as a conservative approach for water quality modelling purposes, the EDCM water quality at pH 7 was used for tailings dam face runoff and infiltration, until a further understanding of the equilibrium drainage characteristics of the coarse tailings is obtained.

Table 6.7-1
Humidity Cell Leachate Results for Cycloned Tailings Sand
(Coarse Fraction Tailings)

Parameter	Units	Humidity Cell		Adj-SNPR >2.5
		Last 25 Weeks of Testing	Maximum	pH 7[1]
Conductivity	microS/cm	118	308	
pH	no units	7.8	8.2	7.0
Acidity (as CaCO3)	mgCaCO$_3$/L	1.0	1.0	21
Alkalinity, Total (as CaCO3)	mgCaCO$_3$/L	48	71	85
Sulfate (SO4)	mg/L	4.6	58	1,486
Fluoride	mg/L			3
Chloride	mg/L			30
Dissolved Elements				
Aluminum (Al)-Dissolved	mg/L	0.031	0.127	0.14
Antimony (Sb)-Dissolved	mg/L	0.00051	0.00410	0.10
Arsenic (As)-Dissolved	mg/L	0.00015	0.00330	0.10
Barium (Ba)-Dissolved	mg/L	0.32	0.40	0.032
Beryllium (Be)-Dissolved	mg/L	0.000011	0.000020	0.0022
Bismuth (Bi)-Dissolved	mg/L	0.0000050	0.0000200	0.05
Cadmium (Cd)-Dissolved	mg/L	0.0000034	0.0000300	0.0038
Calcium (Ca)-Dissolved	mg/L	12	24	275
Chromium (Cr)-Dissolved	mg/L	0.00025	0.00050	0.0060
Cobalt (Co)-Dissolved	mg/L	0.000122	0.002420	0.013
Copper (Cu)-Dissolved	mg/L	0.0011	0.0022	0.081
Iron (Fe)-Dissolved	mg/L	0.0050	0.0050	0.053
Lead (Pb)-Dissolved	mg/L	0.000030	0.000130	0.017
Lithium (Li)-Dissolved	mg/L	0.0010	0.0030	0.0093
Magnesium (Mg)-Dissolved	mg/L	2.6	11	116
Manganese (Mn)-Dissolved	mg/L	4.8	6.6	0.83
Mercury (Hg)-Dissolved	mg/L	0.000050	0.000050	0.00001
Molybdenum (Mo)-Dissolved	mg/L	0.0011	0.021	0.10
Nickel (Ni)-Dissolved	mg/L	0.0010	0.0056	0.022
Potassium (K)-Dissolved	mg/L	2.6	11	6.3
Selenium (Se)-Dissolved	mg/L	0.00050	0.0010	0.10
Silicon (Si)-Dissolved	mg/L	0.36	1.4	5.0
Silver (Ag)-Dissolved	mg/L	0.0000073	0.0000300	0.006
Sodium (Na)-Dissolved	mg/L	0.14	15	80

(continued)

Table 6.7-1
Humidity Cell Leachate Results for Cycloned Tailings Sand
(Coarse Fraction Tailings) (completed)

Parameter	Units	Humidity Cell		Adj-SNPR >2.5
		Last 25 Weeks of Testing	Maximum	pH 7[1]
Tin (Sn)-Dissolved	mg/L	0.000039	0.0011	0.15
Titanium (Ti)-Dissolved	mg/L	0.000050	0.00040	0.01
Vanadium (V)-Dissolved	mg/L	0.00013	0.00051	0.015
Zinc (Zn)-Dissolved	mg/L	0.0033	0.019	0.046

Notes:
1. Source of EDCM (Appendix A).
2. Minimum measured pH at 7.1
3. Values in orange indicate maximum concentrations from Granisle site wide water quality (Morin and Hutt 2003)

6.8   Total Metals Estimations

The Granisle Mine site EDCM also makes provisions for estimating total elemental concentrations, generally a function of the dissolved concentrations (Table 6.8-1). In the absence of EDCM estimates for total metals for some parameters, measured Granisle Mine site pit lake water quality was checked (Morin and Hutt 2003) and an upper bound was assigned. Tailings pond water total metals estimates were derived from the mean of aging test data (Section 5.2.1.5) at the appropriate ratio (i.e., 65% coarse to 35% fine) and are listed in Table 6.8-2.

Table 6.8-1
Granisle Minesite EDCM Results as Analogues for Morrison Deposit
Rock Water Quality Total Metal(loids)

Parameter	Units	pH 3	pH 4.5	pH 5.5	pH 7
Aluminum	mg/L	D-Al for D-Al >2.0	D-Al for D-Al >2.0	D-Al	D-Al
Antimony	mg/L	<0.2	<0.2	<0.2	<0.2
Arsenic	mg/L	D-As	D-As	D-As	D-As
Barium	mg/L	D-Ba for D-Ba > 0.03	D-Ba for D-Ba > 0.03	D-Ba for D-Ba > 0.03	D-Ba for D-Ba > 0.03
Beryllium	mg/L	D-Be for D-Be >0.02 mg/L	D-Be for D-Be >0.02 mg/L	D-Be for D-Be >0.02 mg/L	D-Be for D-Be >0.02 mg/L
Bismuth	mg/L	D-Bi	D-Bi	D-Bi	D-Bi
Cadmium	mg/L	D-Cd	D-Cd	D-Cd	D-Cd
Calcium	mg/L	D-Ca	D-Ca	D-Ca	D-Ca
Chromium	mg/L	D-Cr	D-Cr	D-Cr	D-Cr

(continued)

Table 6.8-1
Granisle Minesite EDCM Results as Analogues for Morrison Deposit Rock Water Quality Total Metal(loids) (completed)

Parameter	Units	pH 3	pH 4.5	pH 5.5	pH 7
Cobalt	mg/L	D-Co for D-Co > 0.01 mg/L	D-Co for D-Co > 0.01 mg/L	D-Co for D-Co > 0.01 mg/L	D-Co for D-Co > 0.01 mg/L
Copper	mg/L	D-Cu for D-Cu > 10 mg/L	D-Cu for D-Cu > 10 mg/L	D-Cu for D-Cu > 10 mg/L	D-Cu for D-Cu > 10 mg/L
Iron	mg/L	D-Fe for D-Fe > 7 mg/L	D-Fe for D-Fe > 7 mg/L	D-Fe for D-Fe > 7 mg/L	D-Fe for D-Fe > 7 mg/L
Lead	mg/L	D-Pb	D-Pb	D-Pb	D-Pb
Lithium	mg/L	D-Li for D-Li >0.03 mg/L	D-Li for D-Li >0.03 mg/L	D-Li for D-Li >0.03 mg/L	D-Li for D-Li >0.03 mg/L
Magnesium	mg/L	D-Mg	D-Mg	D-Mg	D-Mg
Manganese	mg/L	D-Mn for D-Mn > 2 mg/L	D-Mn for D-Mn > 2 mg/L	D-Mn for D-Mn > 2 mg/L	D-Mn for D-Mn > 2 mg/L
Mercury	mg/L	D-Hg	D-Hg	D-Hg	D-Hg
Molybdenum	mg/L	D-Mo for D-Mo > 0.02 mg/L	D-Mo for D-Mo > 0.02 mg/L	D-Mo for D-Mo > 0.02 mg/L	D-Mo for D-Mo > 0.02 mg/L
Nickel	mg/L	0.37	D-Ni for D-Ni > 0.1	D-Ni for D-Ni > 0.1	D-Ni for D-Ni > 0.1
Potassium	mg/L	D-K for D-K > 5 mg/L	D-K for D-K > 5 mg/L	D-K for D-K > 5 mg/L	D-K for D-K > 5 mg/L
Selenium	mg/L	D-Se	D-Se	D-Se	D-Se
Silicon	mg/L	D-Si	D-Si	D-Si	D-Si
Silver	mg/L	D-Ag	D-Ag	D-Ag	D-Ag
Sodium	mg/L	D-Na for D-Na > 4 mg/L	D-Na for D-Na > 4 mg/L	D-Na for D-Na > 4 mg/L	D-Na for D-Na > 4 mg/L
Tin	mg/L	D-Sn	D-Sn	D-Sn	D-Sn
Titanium	mg/L	D-Ti	D-Ti	D-Ti	D-Ti
Thalium	mg/L	D-Tl	D-Tl	D-Tl	D-Tl
Vanadium	mg/L	D-V	D-V	D-V	D-V
Zinc	mg/L	D-Zn for D-Zn > 0.3 mg/L	D-Zn for D-Zn > 0.3 mg/L	D-Zn for D-Zn > 0.3 mg/L	D-Zn for D-Zn > 0.3 mg/L

Notes:
1. Metal(loid) values are for dissolved.
2. Source of EDCM (Appendix A).
3. TDS calculated from the sum of all listed ions.
4. Estimated concentrations of nitrogen species based on another BC minesite.
5. Values in orange indicate maximum concentrations from Granisle minesite wide water quality (Morin and Hutt 2003).

Table 6.8-2
Ageing Test Dating Mean Total to Dissolved Ratios

Parameter	Coarse Tails T:D Mean	Fine Tails T:D Mean	Whole Tailings T:D Mean
Hg	0.10	1.0	0.42
Ag	2.4	2.1	2.3
Al	22	17	20
As	1.7	1.4	1.6
Ba	1.5	1.3	1.4
Be	1.5	1.0	1.3
B	1.1	1.1	1.1
Ca	1.1	1.1	1.1
Cd	1.6	1.1	1.4
Co	2.0	1.9	2.0
Cr	7.9	3.8	6.5
Cu	6.7	21	12
Fe	191	82	153
K	1.0	1.0	1.0
Li	1.0	1.0	1.0
Mg	1.1	1.0	1.1
Mn	1.6	6.2	3.2
Mo	0.82	1.0	0.88
Na	1.1	1.1	1.1
Ni	3.9	1.6	3.1
Pb	12	89	39
Sb	0.95	0.98	0.96
Se	1.3	1.8	1.5
Si	1.8	1.7	1.8
Sn	2.4	2.5	2.4
V	14	7.0	12
Zn	28	2.0	19

6.9   Additional Modeling Considerations

6.9.1   General Assumptions

The semi-empirical mass balance accounting and dilution approach developed for predictive water quality modelling using Goldsim (Rescan 2009c) over the operational mine-life and into closure assumes:

1. The leachate solutions originating from the different mine components are at equilibrium and do not change with time (i.e., constant loading rates with no kinetic constraints) and are removed quantitatively from the sources.

2. No solid-aqueous interaction regarding buffering or cation exchange occur.

3. Waterbodies were assumed to be well-mixed with moderate redox conditions (i.e., pe ~5 to 8).

6.9.2   pH

The mass-balance accounting approach is not well-suited for evaluating pH directly in reactive systems. However, as alkalinity and acidity are inputs to the mass-balance modelling, the net result of the balance of these two parameters from the mass-balance modelling was checked against the Granisle Mine site EDCM to back out a pH value for further solubility modelling Table 6.9-1. For example, when the balance of these two parameters is calculated for the Morrison Pit Lake at final elevation (i.e., 363 mg $CaCO_3$/L Acidity - 269 mg $CaCO_3$/L Total Alkalinity = 94 mg $CaCO_3$/L Acidity) produces a pH of 4.92, which was then used as input pH in solubility modelling to determine appropriate thermodynamic constraints. This assumes the acidity is the negative of alkalinity. Note that the TSF pond pH input in solubility modelling was pH 7 because tailings aging test and background surface water quality (flowing into the pond at closure) show excess alkalinity and near-neutral to slightly alkaline pH values.

Conservatism was built into assigning weighted adjusted SNPR values to the deposit rock and the subsequently assigned leachate quality. Predictive modeling (Rescan 2009c) resulted in acidity and pH predictions comparable to measured Gransile Mine site pit lake water quality (Table 6.9-1). This therefore supports the conservative approach to metal(loid) predictions, a function of pH.

Table 6.9-1
Mass Balance Modeling Acidity and Alkalinity Results Compared to the Granisle Minesite Empirical Drainage Chemistry Model and Measured Granisle Minesite Pit Lake Water Quality

		Acidity	Total Alkalinity	pH
		mg CaCO3/L		pH units
Granisle Mine EDCM[1]	All except North Dump with pH >3.0	182	na	4.01
	All except North Dump with pH <3.0	2.7	na	4.01
	North Dump	824	na	4.01
	All except North Dump with pH >3.1	93	na	4.92
	All except North Dump with pH <3.1	0.03	na	4.92
	North Dump	221	na	4.92
	All except North Dump with pH >3.2	21	89	7.04
	All except North Dump with pH <3.2	0	na	7.04
	North Dump	12	na	7.04

(continued)

Table 6.9-1
**Mass Balance Modeling Acidity and Alkalinity Results Compared to
the Granisle Minesite Empirical Drainage Chemistry Model and
Measured Granisle Minesite Pit Lake Water Quality (completed)**

		Acidity	Total Alkalinity	pH
		mg CaCO3/L		no units
Morrison Predictions	All except North Dump with pH >3.2	24	69	6.85
	All except North Dump with pH <3.2	0	na	6.85
	North Dump	15	na	6.85
	TSF Pond Spill at 1013masl (year 24 after start of mining)	2.3	72	7.00
	TSF Pond Steady State at 1,013 masl (year 99 after start of mining)	2.6	47	7.00
	Pit Lake "Spill" at Final Elevation 728 masl (year 42 after start of mining)	363	269	4.92
	Pit Lake Steady State at Final Elevation 728 masl (year 99 after start of mining)	428	248	4.00
Granisle Mine[2]	Pit Lake Water Quality	~2-100	~5-100	~5.0-8.5

[1].Source: Appendix A.
[2] Source: (Morin and Hutt 2003).
na = not applicable.

6.9.3 Solubility

Reactive geochemical modelling (i.e., solubility limits and adsorption onto iron (hydr)oxides) was modelled to determine upper-bound solute concentrations for mass balance dilution water quality predictions important times after closure and include:

1. TSF pond spill (1,013 masl) estimated to occur 24 years after mining starts.

2. TSF pond steady state chosen at 99 years after start of mining.

3. Pit Lake final elevation (~728 masl) estimated to occur 42 years after mining start.

4. Pit Lake steady state chosen at 99 years after start of mining.

The software Phreeqc (Parkhurst and Appelo 1999) was employed for reactive geochemical modelling after an initial estimate of solution chemistry was provided from the mass-balance modelling. Credible minerals at equilibrium (Eary 1999; Agbenin and Felix-Henningsen 2004) were included to limit concentrations of sulphate, aluminum, barium, calcium, and copper only if saturation was achieved (i.e., SI>0) and included:

- Barite ($BaSO_4$)
- Calcite ($CaCO_3$)
- Geothite ($FeOOH$)

- Gibbsite [$Al(OH)_3$]

- Gypsum ($CaSO_4$)

- Cupricferrite ($CuFe_2O_4$)

6.9.4   Adsorption

Modelling included metal(loid) adsorption onto iron (hydr)oxides, in this case the amount of precipitated goethite caused by saturation. Goethite was the iron (hydr)oxides of choice because of an order of magnitude fewer adsorption sites compared to other phases, such as ferrihydrite (Appelo and Postma 2007) and it has been documented as a common secondary precipitate at mine sites (Eary, 1999). Note that the application of adsorption in water quality prediction modeling is applied after mass balance results are produced for time periods of interest (i.e., TSF spill, ultimate pit lake elevation and far into closure).

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7. Conclusions and Recommendations

7.1 Conclusions

Based on the static test results presented in Section 5.1, the following conclusions are provided.

1. The main minerals of ML/ARD interest at the Morrison deposit include chalcopyrite ($CuFeS_2$), distributed as fine-grained disseminations in the BFP and siltstone as fracture coatings or as stockworks of quartz veinlets. Minor bornite (Cu_5FeS_4), marcasite (FeS_2), pyrrhotite ($Fe_{(1-x)}S$), galena (PbS), molybdenite (MoS); arsenopyrite (FeAsS) and sphalerite (ZnS). Sulphide mineralization generally occurs in the form of fracture filling and/or disseminated textures in the rock matrix.

   A pyrite halo is developed in the chlorite-carbonate altered wall rock surrounding the copper zone. The pyrite is mineralization characteristically occurs as thin (0.1- 5.0 cm) fracture-fillings and quartz-pyrite-minor chalcopyrite stringers in the form of stockwork within the halo. There is a crude zonation to the pyrite development with coarse (0.5 to 5.0 mm) disseminated crystals within the inner parts of the halo where pyrite content ranges from 5 to 15% by volume. Pyrite in the outer zone is predominantly developed as a stockwork and averages 1 to 2% by volume accompanied by weak copper mineralization (<0.1%). The pyrite halo is developed as a more extensive zone around the eastern and southeastern segment of the Morrison deposit. Drilling and geophysical surveys indicate that he halo at this position attains widths up to 500m with up to 14% pyrite for the inner margin and decreasing abruptly to 1 to 2% in the outer two thirds of the halo. The pyrite halo is more restricted at the western and northwestern segments of the deposit where pyrite abundances decrease more gradually to the 3 to 5% range. The siltstone host rock at this location is intruded by large northerly-trending BFP and rhyodacite dykes.

2. The main minerals important for the assessment of NP include calcite ($CaCO_3$), siderite ($FeCO_3$) and ankerite ($Ca(Fe,Mg,Mn)(CO_3)_2$) and aluminosilicate minerals like plagioclase (An_{25-35}), biotite ($K(Mg,Fe)_3AlSi_3O_{10}(OH,F)_2$), hornblende ($Ca_2[Fe_4(Al,Fe)]Si_7AlO_{22}(OH)_2$) or ($Ca_2[Mg_4(Al,Fe)]Si_7AlO_{22}(OH)_2$), epidote ($Ca_2(Fe,Al)_3(SiO_4)_3(OH)$), and chlorite ($Ca_{0.5}(Al,Mg)_6(Si,Al)_8O_{18}(OH)_{12}\cdot5(H_2O)$).

3. Ore throughput at Project will be approximately 30,000 t/d (11 Mt/yr). Stripping ratios range between 0.44, in the early years of the mine life, and 1.99 in the latter years, with an overall average of 0.82. Total waste destined for the WRD, overburden stockpile, and low-grade ore stockpile are approximately 151 Mt, 15 Mt, and 51 Mt, respectively. Stockpile recovery will result in a low-grade ore-stockpile peaking at a maximum of 36 Mt.

4. Tailings at the Project will be produced at an annual rate of approximately 10.95 Mt. Tailings solids and process waters will be pumped as slurry to the TSF, where the tailings slurry stream will be cycloned to produce coarse sand for construction purposes of the main, north, and west dams. The TSF will operate as a "zero" discharge facility, meaning that tailings pond water from the TSF will be reclaimed to the process plant for further ore processing and there is no planned discharge from the TSF to the receiving environment.

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5. A total of 571 samples consisting of 496 deposit drill core, 36 overburden, 9 monitoring well, 4 condemnation drilling, and 26 tailings samples were collected and characterized as part of the ML/ARD prediction program to determine the ML/ARD potential of overburden, waste rock, low grade ore, ore , pit wall rock, and tailings.

6. Static testing is summarized in Table 7.1-1 and shows that total sulphur ranges between less than detection (0.01%) up to 9.4 % while sulphide sulphur ranges between below detection (0.01%) up to 8.8 %. Bulk Sobek NP ranged between below the MDL (1 kg CaCO$_3$/tonne) up to 403 kg CaCO$_3$/tonne.

Table 7.1-1
Static Testing Summary of Morrison Mine Materials

Morrison Mine Materials	Total Sulphur (%)			Measured Sulphide -Sulphur (%)			Bulk Sobek Neutralization Potential (kg CaCO$_3$/tonne)		
	Max	Min	Mean	Max	Min	Mean	Max	Min	Mean
Ore	4.0	0.31	1.0	3.7	0.27	1.00	403	0.50	62
Low Grade Ore	3.4	0.060	0.98	3.1	0.14	0.91	208	1.0	63
Waste Rock	9.4	0.010	1.1	8.8	0.060	1.0	314	0.50	67
Overburden	1.2	0.010	0.12	1.1	0.0050	0.087	104	3.0	30
Condemnation Drilling	0.94	0.42	0.61	0.73	0.34	0.48	54	33	42
Monitoring Wells	2.7	0.030	0.97	2.4	0.030	0.84	193	19	78
Tailings[1]	0.55	0.094	0.32	0.46	0.11	0.29	61	51	56

Notes:
1. Representative tailings only.
2. Values below method detection listed as one half the method detection limit.

7. Various NP adjustments (i.e., decrease for unavailable NP) were determined for the different mine components based on paste pH and bulk Sobek NP (Table 7.1-2). However, as a conservative approach, the highest NP adjustment of 13 kg CaCO$_3$/tonne was applied for subsequent sulphide sulphur net potential balances of PAG and not-PAG.

Table 7.1-2
Determined Neutralisation Potential Adjustments for Different Morrison Mine Materials

Type	Paste pH	Potential Unavailable NP Adjustment (Decrease)
ore	5.8	13
	4.8	2
low-grade ore	5.3	9
	4.4	0.5

(continued)

Table 7.1-2
Determined Neutralisation Potential Adjustments for Different
Morrison Mine Materials (completed)

Type	Paste pH	Potential Unavailable NP Adjustment (Decrease)
WR	5.3	12
	4.7	11
overburden	5.9	9
	4.9	4
CD	NA	NA
MW	NA	NA
Tailings	NA	NA

8. Adjusted SNPR calculations indicate ore, low grade ore, waste rock, condemnation drilling, and monitoring well samples are both PAG and not-PAG, if an SNPR ratio of less than two is considered PAG. An estimated 73% of waste rock is classified as PAG and therefore as a conservative approach, all waste rock has been designated PAG and will be placed and managed in one WRD to the east of the open pit. Note that two overburden samples from within the pit limits have adjusted SNPR <2, but the lithology of these two samples requires confirmation as they were sampled from DDH core. Some overburden adjusted SNPR values are calculated as 0.001. This is an artefact of very low measured sulphide sulphur and Sobek NPs <13, which are assigned default values of adjusted SPNRs of 0.001. Tailings are classified as not-PAG, but are considered neutral pH metal leaching according to available laboratory kinetic data .

9. Overburden samples collected from outside the pit outline were low in sulphide content, high in Sobek NP, were classified as not-PAG and had low solid-phase metal(loid) concentrations. Overburden samples from within the pit outline had high metal(loid) contents. Based on this information, overburden from outside the pit outline is more suitable for reclamation or construction. However, the long-term leaching potential of overburden from within and outside the pit outline has not been quantified at this time.

10. Preliminary estimates of sediment volumes from Booker Lake and Ore Pond for relocation and storage purposes are 600,000 m^3 and 50,000 m^3, respectively. Comparisons against available sediment quality guidelines indicate exceedances for arsenic, cadmium, chromium, copper, iron, mercury, nickel, and zinc, which are not unexpected given the vicinity of the Morrison deposit. The *in situ* redox conditions of lake sediments, and hence the likelihood of authigenic metal(loid) sulphides, is unknown at this time. However, it is likely that the majority of sediments are reducing in nature and draining these lakes and ponds, excavation, and relocation may result in sediment oxidation. The acid-generating potential and NP of these sediments are unknown at this time.

11. Morrison deposit rock humidity cell leachate pH values from five cells containing ore, low grade ore, and waste grade materials of varying lithologies and alteration types has remained

near-neutral to slightly alkaline and steady up to week 80. The exception is one humidity cell, which showed initial acidic pH values, presumably due to previous weathering and release of stored acidity during the initial flushing. According to laboratory-scale humidity cell testing at the indicated neutral to alkaline pH, Morrison waste rock, low grade ore, and ore drainage chemistry is expected to show initial increases in sulphate, copper, zinc, cadmium, arsenic, antimony, and lead with possible later stage releases of molybdenum and barium. Note that humidity cell data interpretations do not take into account site-specific differences in temperature, grain size, preferential flow, or equilibrium and their effects on large-scale mine drainage chemistry.

12. Calculations based on sulphate generation and neutralization depletion rates indicate three of the five humidity cell rock samples are predicted to generate ARD in 15 to 57 years, while there is enough carbonate NP in the two remaining humidity cell rock samples to maintain neutral conditions until sulphides are fully exhausted. This assumes carbonate depletion rates and sulphate production rates remain constant. Note that these estimates of lag times apply to the humidity cell rock samples and should not be applied directly to full scale mine components. Greater confidence in lag time estimates require additional site-specific larger scale kinetic testing data.

13. Representative Morrison tailings humidity cell leachate pH for coarse and fine fractions has remained near neutral (i.e., pH ~6-8) and steady throughout 80 weeks of testing. Additional parameters of interest and trends in humidity cell leachate include:

    a. steady and variable, but low copper leaching for the coarse and fine tailings leachate respectively;

    b. initial releases of molybdenum with a subsequent steady low concentration trend;

    c. moderate and variable zinc releases;

    d. variable but low releases of lead;

    e. initial flushing of arsenic and antimony with decreasing concentration trends for both tailings fractions;

    f. variable but very low cadmium releases;

    g. selenium below MDLs;

    h. in general, the coarse tailings fraction shows higher elemental leaching than the fine tailings fraction leachate.

14. Representative Morrison tailings mineralogy was not characterized, therefore the evaluation of solid-phase reactions responsible for humidity cell leachate chemistry trends is limited.

15. Representative Morrison tailings are classified as not-PAG but are considered neutral pH metal leaching.

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16. Morrison tailings supernatant (i.e., process waters) 61-day aging tests indicate tailings process waters remain slightly alkaline over the testing period. TSS shows initial high concentrations but decrease to below detection for the duration of the test due to settling. Parameters of interest generally show concentration changes with time are less than an order of magnitude, are low due to the alkaline pH and are initially higher for copper, lead, zinc, arsenic, cadmium, and selenium, but variable or somewhat steady for molybdenum and variable for antimony. Concentrations are typically dominated by the total fraction initially (high TSS) and by the dissolved fraction in progressively aged samples.

17. Four on-site LTPs were constructed in 2005 by PBM staff, each containing about one tonne of broken core. However, these leach pads were open to adjacent runoff and did not have adequate hydrologic control or leachate collection to determine the water balance and leachate quality for each LTP. In 2009, the LTPs were reconstructed into field cubes. In addition, field barrels charged with historical Noranda core were added. During reconstruction, several samples were collected for static testing. The first leachate sampling events for the reconstructed LTP field cubes and Noranda core barrels are planned for sampling in early fall, 2009.

18. A semi-empirical mass-water balance accounting and dilution approach was developed for predictive water quality modelling using the software GoldSim for the operational mine-life and after closure. The Granisle Mine site EDCM (Appendix A) was developed using approximately 1,200 water analyses from February 1980 to May 1999 (Granisle closed in 1982). Drainage chemistry analyses included ditches, ponds, the pit, the tailings pond, and groundwater monitor wells around the mine site. In the absence of Morrison deposit rock long-term acidic pH leachate data (i.e., ARD), a conservative approach for Morrison site-wide water quality modelling (Rescan 2009c) was used. Morin and Hutt (2007) compared both Bell Mine EDCM data and Granisle EDCM data to available Morrison laboratory kinetic data at equilibrium (i.e., for pH 4.98 as discussed in Section 6.2) and concluded that both the Bell and Gransile EDCM can be used as predictors of Morrison drainage chemistry for acidic and alkaline pH. Justification for the use of the Gransile EDCM as analogue for the Morrison project include:

    a. similar geology and mineralogy between Granisle and Morrison rock;

    b. the majority of Granisle data supporting the EDCM is from post-closure which assists in modelling the Morrison Pit Lake water quality (WQ) into post closure;

    c. a larger list of predictable parameters is available in the Granisle EDCM versus the Bell EDCM as presented in Appendix A;

    d. the Granisle EDCM provides a conservative (i.e., worst case higher concentration) prediction of rock drainage chemistry at a range of pHs versus the Bell EDCM.

ABA data are material balances; however in the absence of site-specific kinetic data which can provide insight into reactivity, adjusted SNPRs were considered reasonable estimations of relative reactivity for the Morrison project water quality modeling at this time. The relative reactivity of Morrison deposit rock was assumed to be related to weighted adjusted

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SNPRs (see Section 5.1.1.4) and was categorised using best professional judgment. This relative reactivity assumption allows for the mass balance approach to be used in the absence of site-specific kinetic data at equilibrium conditions. Water quality of a particular pH (i.e., pH 3, pH 4.5, pH 5.5, and pH 7.0) as determined by the Granisle EDCM was then assigned to the different adjusted SNPR categories. These selected pH values, and thus rock drainage water quality, were chosen with the objectives of assigning the worst (acidic) water quality (pH 3) to those materials that had very little NP (more reactive PAG) relative to sulphide content (i.e., adjusted SNPR <0.5). Less reactive rock was then assigned a progressively better water quality. Although calculated lag time to ARD onset were as low as 15 years and apply to the humidity cell rock sample, water quality modeling of Morrison rock drainage is assumed to exhibit Granisle EDCM equilibrium water quality as soon as excavated material is placed in the waste storage facility or pit walls are exposed for a conservative approach.

19. Approximately 0.24 kg of ANFO per tonne of ore and waste rock will be used in materials blasting throughout the mine life. Ferguson and Leask (1998) outline a method for apportioning nitrogen species from blasting residue as determined by the ANFO mass used to blast one tonne of rock. Instead, analogue aqueous nitrogen species from a BC mine were used in site-wide water quality modelling due to large uncertainties in estimating nitrogen leaching caused by the hydrologic response of a large-scale constructed mine components, the loadings actually available to contact water either as infiltration or as runoff, geochemical processes (redox transformations and volatilization), and effects of the process plant metallurgical process on the nitrogen mass balance.

20. Tailings pond water and pore water source terms for site-wide water quality modelling assumed results from aging tests. As a conservative approach, aging test data from Day 1 were used. The use of small scale lab ageing test data as a source term for tailings pond water (i.e., tailings transport water aka process water) and tailings pore-water in predictive water quality modeling (Rescan 2009c) instead of analogue Granisle EDCM data was used due to the different metallurgical processes proposed for the Morrison Project (e.g., High Pressure Grinding). It is believed that the ageing test data is more representative of the predicted tailings pond and pore-water chemistry during operations and on closure.

21. Tailings beach and cycloned tailings sand dam runoff and infiltration source terms for site-wide water quality modelling assumed results from the Granisle EDCM water quality set at pH 7 in the absence of site-specific large scale kinetic data for these mine components.

22. The Granisle EDCM also makes provisions for estimating total elemental concentrations, generally a function of the dissolved concentrations. In the absence of EDCM estimates for total metals for some parameters, measured Granisle mine site pit lake water quality was checked and an upper bound was assigned. Tailings pond water and pore water total metal estimates were derived from the mean of aging test data at the appropriate ratio, (i.e., 65% coarse to 35% fine).

23. Additional water quality modelling assumptions included:

    a. the leachate solutions originating from the different mine components are at equilibrium and do not change with time (i.e., constant concentrations with no kinetic constraints) and are removed quantitatively from the sources;

b. no solid-aqueous interaction regarding buffering or cation exchange was considered;

c. waterbodies were assumed to be well-mixed with moderate redox conditions (i.e., pe ~5-8);

d. the Granisle EDCM was used to back out the pH values using the net balance of acidity and alkalinity results form the mass balance water quality modelling;

e. the software Phreeqc (Parkhurst and Appelo 1999)was employed for reactive geochemical modelling and included credible solid phases for metal attenuation (Barite ($BaSO_4$), Calcite ($CaCO_3$), Geothite (FeOOH), Gibbsite [$Al(OH)_3$], Gypsum ($CaSO_4$) and Cupricferrite ($CuFe_2O_4$)], only if saturation was achieved (i.e., SI>0);

f. metal(loid) adsorption onto iron (hydr)oxides, in this case the amount of precipitated goethite caused by saturation, was used for metal attenuation.

24. Conservatism built into assigning adjusted SNPR values to the deposit rock and subsequent assigned leachate quality results in conservative (i.e., worse case) acidity and pH predictions when compared to measured Granisle mine site pit lake and tailings pond water quality. For results of predictive water quality modeling see Rescan (2009c).

7.2 Recommendations

Based on the available data and interpretation to date, the following recommendations are provided.

1. Static testing of lake sediments from Booker Lake and Ore Pond.

2. Laboratory and field scale kinetic testing of:

    a. overburden from within the pit outline;

    b. lake sediments from Booker Lake and Ore Pond.

3. Static testing and site-specific kinetic testing of at least one tonne of representative tailings and cycloned sand, when available, to confirm acid-generating and acid-consuming balances and not-PAG designation for closure water management strategies. Note that sufficient material is typically available after pilot plant testing or as material becomes available during the early stages of process plant operations.

4. Overburden sampling at borrow areas outside the pit outline, including within the TSF, should be sampled to an appropriate degree that provides an acceptable level of confidence for the characterization of ML/ARD. Sampling numbers should be appropriate for the total tonnage of materials to be used in the construction of mine infrastructure. Guidance on the number of samples per tonnage is provided in Price (1997).

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5. Confirming the geological and geochemical nature and hence representativeness of the LTP materials (in progress).

6. Confirming the lithology of high sulphide overburden samples indicated in Table 7.2-1.

7. The use of Morrison rock field LTP data as source terms for site-wide water quality modelling.

8. Additional diamond drilling at the pit shell to confirm pyrite halo presence and magnitude of pyrite content.

9. ABA analyses of additional diamond drill hole samples collected in the future and inclusion into the geochemical block model, especially samples near the current pit shell for use in updating adjusted SNPRs, assigned leachate quality and pit lake water quality predictions.

Table 7.2-1
Overburden Samples Requiring Lithology Confirmation

DDH	From (m)	To (m)
MO-00-10	2.0	5.0
9240-3	14.0	17.0
MO-02-66	6.0	9.0
9240-1	5.0	8.0

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