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Date Sept 7, 1999
ABSTRACT

The Eskay Creek Mine, owned by Prime Resources Group Inc. is located about 80 kilometres north of Stewart, British Columbia. The mine is an unusual polymetallic, precious-metal rich volcanogenic massive sulfide and sulfosalt deposit which has generated much interest over the past decade. The Eskay Creek Mine is one of the highest grades gold and silver deposits in the world.

The objective of this study is to conduct a baseline study on the potential for ARD generation from the two types of mill tailings derived from two different ore zones (109 zone and NEX zone) at the Eskay Creek Mine in order to assess the maximum possible ARD potential under the worst disposal conditions among the several tailings disposal options. The study involved Sobek ABA tests, Modified ABA tests, shake flask tests and kinetic tests.

The results from ABA tests show that the acid potential is much higher than the neutralization potential in both 109 and NEX tailings. This means that both 109 and NEX tailings have the probability for acid generation. Shake Flask test results show that generation of ARD from both tailings is likely and temperature is an important factor in acid generation. Submarine tailings disposal may be the best way to dispose tailings materials in Eskay Creek Mine.
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1. INTRODUCTION

The Eskay Creek Mine is an unusual polymetallic, precious-metal rich volcanogenic massive sulfide and sulfosalt deposit which has generated much interest over the past decade. Located in the metallogenetically-rich Iskut River (Roth et al., 1997), 80 kilometers north of Stewart, British Columbia, the deposit is distinct in style and mineralogy from other known deposits in the area and from other volcanic-associated massive sulfide deposits.

The deposit is hosted by a Middle Jurassic bimodal volcanic sequence on the west flank of an anticlinal structure, at the contact between felsic volcanic rocks and a mudstone-basalt package. Economic concentrations of precious and base metals are contained in the 21 Zone, which is subdivided into a number of distinct sub-zones. The bulk of the reserves are contained in the stratiform 21B zone, which is characterized by clastic sulfide-sulfosalt beds composed of sphalerite, tetrahedrite, lead sulfosalts, galena, pyrite, electrum-amalgam, and variable amounts of mudstone, chloritized and sericitized rhyolite, and barite fragments. In January 1995, Homestake Canada Inc./Prime Resources Group Inc. commenced production from the 21B zone as direct-shipped ore with a proven and probable mining reserve of 1.08 million metric tonnes grading 65.5 grams Au per tonne and 2,930 grams Ag per tonne (Roth et al., 1997).

The high grade gold mine had indicated that an extremely complex hydro-metallurgical and chemical process flowsheet would be required to recovery the economic metals (eg.
gold, silver, copper and zinc). This process was complex requiring a large capital outlay with high operating costs. The original operating plan was to provide the mining infrastructure at the mine site and transport the ore to the process facility that would be located near Houston, B.C. A suitable facility for the tailings storage, essentially encapsulating the filtered tailings cake had been designed. The lower grade ore zones, which are less economic or uneconomic to mine and ship directly to the smelters (Anand et al., 1998).

Continued exploration in 1995 and 1996 identified the NEX zone and expanded the 109 zone. These zones are relatively lower grade than the ore being directly shipped and also lower in deleterious minerals, such as arsenic and antimony. With the high cost of direct shipping and smelting, a large portion of this material would be uneconomic to be mined. It was recognized that some type of on site or local processing was needed to improve the economics of the NEX and 109 zone. A 150 tonne per day flotation plant was constructed to concentrate the gold and silver values for upgrading ore from the NEX and 109 zones into marketable concentrates. The mine will then, in addition to the direct shipping of the shippable ore, ship concentrate from the millable ore (Anand et al., 1998).

For a long time, the serious environmental consequences of acid rock drainage (ARD) have been one of the most important challenges for the mining industry. ARD occurs as a result of natural oxidation of sulfide minerals contained in rock when exposed to air and water. The oxidation reactions are often accelerated by biological reactions which can yield low pH water with the potential to mobilize heavy metals that may be contained in
the waste rock or elsewhere. Even though the pH of the resultant drainage might be close to neutral, it can contain products of the acid generation process, typically elevated metal levels and sulfate, and can cause a detrimental impact on water quality in the receiving environment.

There is also acid rock drainage potential from Eskay Creek mill tailings. The purpose of this study is to conduct a baseline study on the potential for Acid Rock Drainage (ARD) generation from the two types of mill tailings derived from two different ore zones (109 zone and NEX zone) at the Eskay Creek Mine in order to assess the maximum possible ARD potential under the worst disposal conditions among the several tailing disposal options available.

Sobek acid-base accounting and Modified acid-base accounting methods (Lawrence and Wang, 1997) have been used to assess the neutralization potential (NP) in a series of tailings samples, ore, and other material related to the tailing disposal environment. Maximum acid potential has been calculated from the sulphide sulphur content of the tailings. Leachability of heavy metals was assessed through a series of shake flask tests under different temperature, pH and with or without bacteria conditions. Also, kinetic tests on tailings and agglomerated tailings were conducted to predict the ARD potential over time. At the same time, a mineralogical examination of the ore and tailings samples was conducted using Scanning Electron Microscope (SEM) and X-ray Diffraction (XRD) to cast light on the mineral associations and the texture of different constituents that contribute to acid and neutralization potential. Finally, based on the data on Standard
Acid Base Accounting (ABA) and leachability tests, the tailing disposal options are discussed.
2. BACKGROUND

2.1 Eskay Creek Mine

2.1.1 Introduction

The Eskay Creek Mine, owned by Prime Resources Group Inc., is located about 80 kilometres north of Stewart, British Columbia in the Skeena Mining Division (Figure 2.1). The Property lies 4 kilometres east of Tom MacKay Lake on the Prout Plateau between the Unuk and Iskut rivers. Primary access to the mine is via a 61 kilometres access road between the mine site and Highway 37 (Stewart-Cassiar Highway). Alternate access to the Eskay Creek Mine can be gained by fixed-wing aircraft to the Johnny Mountain or Bronson Creek airstrips, then helicopter to the Property or a company helicopter based at Bell-Irving River on Highway 37.

The Eskay Creek Mine is one of the highest grade gold and silver deposits in the world. Production from the Eskay Creek 21 Zone, from startup in January 1995 until the end of December 1997, totaled 720,000 ounces of gold and 35.0 million ounces of silver since commercial production commenced in January 1995. Proven and probable ore reserves at the beginning of 1998 were 1.5 million tons containing 2.69 million ounces of gold and 117 million ounces of silver. The bulk of the ore is contained in the 21B zone and is shipped directly to smelters in Japan and Eastern Canada. The other reserves are contained in subzones with different mineralogical and metallurgical characteristics which are amenable to concentration through a gravity and flotation circuit. Construction
of an on-site mill to process these ores was completed in December 1997 and production began in January 1998 (Roth and Rogers, 1997).

Figure 2.1 General location map for the Eskay Creek Mine, northwestern British Columbia.
2.1.2 Exploration History

Exploration began in 1932 when prospectors were drawn to the area by prominent gossanous bluffs formed by the altered felsic intrusive rocks which feed the rhyolite. Two adits were driven on precious metal-rich stockwork veins in the felsic intrusive and footwall rhyolite. Numerous companies explored the property in the following years, mainly concentrating on stockwork mineralization. In the mid 1980's a junior company, Kerrisdale Resources, drilled four holes near the 21 zone, including one that encountered stratiform mineralization in the 21A zone. Follow up drilling in 1988 by joint venture partners Stikine Resources Limited and Calpine Resource Inc. confirmed stratiform mineralization in the 21A zone and subsequently encountered high grade mineralization in the 21B zone about 200m to the north of the discovery hole. Step-out drilling intersected the 109 zone, which has subsequently been delineated from underground drill-stations. In 1995, drill testing of the productive rhyolite contact horizon, to the north of the 21B zone, delineated the North Extension (NEX) zone. In contrast to the 21B zone, portions of the 109 and NEX zones can be concentrated through gravity and flotation because the ores from the 109 and NEX zones have a relatively lower grade than the ores from the 21B zone so that the ore concentration from 21B zone is much higher that those from 109 and NEX zones.

2.1.3 Geological Setting

The Eskay Creek ore bodies are hosted by Middle Jurassic volcanic and sedimentary rocks in the upper part of the Hazelton Group. The mine sequence comprises an upright stratigraphic succession of andesite, marine sediments, intermediate to felsic
volcaniclastic rocks, rhyolite, mudstone hosting stratiform mineralization (termed the contact mudstone), and basaltic sills and flows which are intercalated with turbiditic mudstones. The lower part of the sequence is intruded by porphyritic monzodiorite to diorite and by younger felsic dikes and sills which are feeders to the rhyolite package. Basaltic dikes and sills feeding the hanging wall basaltic package crosscut all strata in the mine sequence. Regional metamorphic grade in the area is lower greenschist facies.

The mine stratigraphy is folded into a northeast trending upright open fold, plunging gently to the north. Stratiform mineralization in the 21 Zone occurs on the western limb of the fold, near the fold closure, and generally dips 30 to 45 degrees to the west. Stratigraphy is disrupted by a series of northwest trending faults, as well as north to northwest axial planar faults.

2.1.4 Mineralization

The 21 Zone contains a number of subzones distinguished by varying mineralogy, textures, grades and metallurgical characteristics (Table 2.1) (Roth and Rogers, 1997). Stratiform mineralization is hosted in marine mudstone at the contact between underlying rhyolite and overlying basalt packages. The 21A, 21B, 21C and NEX zones occur at this stratigraphic contact (Figure 2.2) (Roth et al., 1997). The HW zone is stratigraphically higher in the sequence, usually above the first basaltic sill. Stockwork vein and disseminated mineralization are present in the rhyolite, most notably in the 109 and Pumphouse zones.
The 21B zone contains the bulk of the ore and consists of clastic sulphide-sulphosalt beds. Sedimentary facies variations are well preserved locally, although commonly overprinted by later sulphosalts and locally stibnite ± cinnabar in the core of the deposit. The thickest beds and coarsest clasts occur at the core of the orebody. Clast size and bed thickness typically decrease stratigraphically upwards and laterally outwards, progressively thinning to fine laminations and disseminated sulphides and sulphosalts in mudstone.

Surface drilling in 1995 led to the discovery of the North Extension (NEX) zone. This zone occurs at the nose of the Eskay Anticline; its geometry is complicated by a number of faults associated with the fold closure. Textures, mineralogy and precious metal grades in the NEX zone are similar to those in the 21B zone, although pyrite and chalcopyrite are more prevalent and Sb-Hg rich minerals are scarcer. Chalcopyrite occurs mainly in stringers which overprint earlier clastic mineralization and which are probably related to the formation of the HW zone. The formation of the NEX zone is probably synchronous with the formation of the 21B zone.

The 109 zone is named after its discovery hole which intersected 61 metres of 99 g/t gold and 29 g/t silver. This zone is characterized by a distinctive, siliceous stockwork of crustiform quartz veins with coarse-grained, zoned sphalerite, galena, minor pyrite and chalcopyrite. The zone occurs entirely within the rhyolite underlying the north end of the 21B zone and the HW zone. The 109 zone is associated with abundant black
carbonaceous material disseminated throughout the rhyolite and locally remobilized into the quartz veins. Gold and silver occur in electrum and sulfosalts.

Presently, the mining activity is focused on NEX and 109 zones with respect to mill feed. Although the depositional environment, geometry and associated alteration of the Eskay Creek deposit most closely resemble a VMS model, the high grades of precious metals, association with an Sb-Hg±As element suite, and the importance of bedded clastic sulphide-sulphosalt ore indicate that processes more commonly associated with epithermal environments were active at the time of formation (Roth et al., 1997).

Figure 2.2 The distribution of the deposit in the Eskay Creek 21 zone.
Table 2.1 Summary of mineralization styles in the Eskay Creek 21 zone.

<table>
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<tr>
<th>Zone</th>
<th>Characteristics</th>
<th>Stratigraphic Position</th>
</tr>
</thead>
<tbody>
<tr>
<td>21A</td>
<td>Stratiform lens of massive to semi-massive sulphide (realgar, stibnite, cinnabar, arsenopyrite) underlain by disseminated stibnite, arsenopyrite, tetrahedrite and veinlets of pyrite, sphalerite, galena, tetrahedrite, ±chalcopyrite. Exploration target.</td>
<td>Same horizon as 21B zone, at base of contact mudstone, overlies disseminated mineralization within rhyolite</td>
</tr>
<tr>
<td>21B</td>
<td>Stratiform, bedded clastic sulphides and sulphosalts including: sphalerite, jamesonite, stibnite, galena, pyrite, electrum, amalgam. High grade Au-Ag. Ore is shipped directly to smelter.</td>
<td>At or near base of contact mudstone</td>
</tr>
<tr>
<td>East Block of 21B</td>
<td>Fine-grained massive to locally clastic sulphides and sulphosalts. Massive pyrite-flooding in rhyolite grading upwards into massive sulphides and sulphosalts. Ore is shipped directly to smelter; some millable ore.</td>
<td>within fault-bounded block, at contact between rhyolite and mudstone</td>
</tr>
<tr>
<td>21C</td>
<td>Massive to bladed barite associated with very fine-grained disseminated sulfides including pyrite, tetrahedrite, sphalerite and galena. Exploration target.</td>
<td>Same horizon as 21B zone, at base of contact mudstone</td>
</tr>
<tr>
<td>Hanging Wall (HW)</td>
<td>Massive, fine-grained stratabound sulphide lens dominated by: pyrite, sphalerite, galena, &amp; chalcopyrite (mainly as stringers). This zone has generally lower gold-silver grades and higher base metals relative to the 21 zones. Portions shipped directly to smelter.</td>
<td>Within mudstone; at a higher stratigraphic level than the 21 zones; generally above first basalt sill</td>
</tr>
<tr>
<td>Pumphouse</td>
<td>Very fine disseminations and veinlets of pyrite, sphalerite, galena and tetrahedrite in rhyolite. Geological resource under evaluation.</td>
<td>Discordant, within rhyolite</td>
</tr>
<tr>
<td>NEX</td>
<td>Massive to locally clastic bedded sulphides with fewer sulphosalts than in the 21 B zone and local overprint of chalcopyrite stringer. Millable ore.</td>
<td>Same horizon as 21B zone, at base of contact mudstone</td>
</tr>
<tr>
<td>109</td>
<td>Stockwork veins of quartz, sphalerite, galena, pyrite and visible gold associated with silica flooding and fine-grained amorphous carbon alteration. Millable ore.</td>
<td>Discordant, within rhyolite</td>
</tr>
</tbody>
</table>

2.1.5 Mill Flowsheet and Tailing Production on Eskay Creek Mine

A two-stage crushing plant which include the primary jaw crusher, vibrating feeder, and take-away conveyor was used to produce a 50 mm product suitable for direct shipment of the ore from the mine to the smelters. In order to deliver ore to the flotation plant, the tertiary crushing circuit has been used to produce a 10mm product suitable for flotation
A cone crusher which has a design capacity of 62 tonnes per hour at 10 mm closed side setting (CSS). The crusher is operated in closed circuit with the screen, where the crusher discharge is conveyed back to the vibrating screen.

Figure 2.3 is a simplified flowsheet of the grinding and gravity circuit. The grinding circuit consists of a single stage ball mill operated in closed circuit with a cyclone. The plant flowsheet has been designed for a nominal throughput of 150 tonnes per day, which may increase to 200 tonnes per depending on ore type. To accommodate a gravity circuit in the Eskay Creek flotation plant, the cyclone under-flow launder was modified so that a partial stream of cyclone under-flow could be diverted to a 508 mm Knelson concentrator. The concentrate is discharged to a holding tank from which it feeds to a Gemeni shaking table. Both the Knelson and Gemeni table tails are returned to the mill discharge pumpbox. Table concentrate is collected in drums for shipment to a refinery operation.

The flotation circuit is designed to operate at natural pH to maximize the recovery of all the sulphides, which tends to maximize the recovery of the precious metals. Staged additions of standard flotation reagents, PAX, 3418A and MIBC, are used to promote metal recovery. Figure 2.4 is a simplified flowsheet of the flotation circuit. The cells are configured as one re-cleaner, two cleaners, two rougher and four scavengers. The flotation concentrate and tailings are directed to their respective de-watering circuits.
The location of the Eskay Creek minesite does not provide sufficient or suitable space for a tailing storage pond facility. The tailings from scavengers is processed by thickening and then the thickener under-flow at 55% solids are sent to the tailings storage tank which provides storage for 7 hours of tailings production at nominal plant feed rates. The tailings from the storage tank are pumped to a PF60 Larox pressure filter. The filter operates in cycles, including the stages of feeding, air blowing, pressing and discharge. The filtered tailing cake is discharged directly to the floor below the filter area in a storage. Finally, the tailings have been sent to Albino Lake located about 8 km from the mine for subaqueous disposal. The simplified tailings de-watering flowsheet is illustrated in Figure 2.5.
Figure 2.3  Grinding and gravity concentration flowsheet on Eskay Creek Mine (Anand et al., 1998).
Figure 2.4 Flotation flowsheet on Eskay Creek Mine (Anand et al., 1998).
Figure 2.5  Tailings dewatering flowsheet on Eskay Creek Mine (Anand et al., 1998).
2.2 Acid Rock Drainage

2.2.1 Introduction

Exposure of rock or soil containing reactive sulfide minerals to oxidants (typically dissolved oxygen) in the presence of water can result in the production of acidic and metal-contaminated surface and ground waters. The run-off from the oxidation and leaching is commonly referred to as acid mine drainage (AMD) or, acid rock drainage (ARD).

ARD is a major obstacle to successful reclamation for mining operations throughout the world. Mined rock can contain abnormally high concentrations of heavy metals, which can be released in the sulfide weathering process. If water is available as a transport medium, the resultant drainage can contain products of the acid generation process, typically elevated concentrations of metals and sulfate. This can cause a detrimental impact on water quality in the receiving environment. Uncontrolled ARD emissions can result in significant ecological disruption in sensitive and productive receiving waters.

The capability of a particular waste to generate net acidity and produce contaminated drainage is a function of the balance between the potentially acid-producing minerals and the potentially acid-consuming minerals contained in the material. This balance is further influenced by physical, chemical and biological controls on the relative rates of acid generation and acid consumption. The process by which acid is consumed is known as neutralization. Acidic waters generated by sulfide oxidation of a waste may be neutralized
upon contact with acid-consuming minerals. As a result, the water draining from the waste may have a neutral pH and negligible acidity despite on-going sulfide oxidation. However, if the acid-consuming minerals are dissolved or coated by other minerals through encapsulation as a result of secondary mineralization, then as acid generation continues, acidic water may eventually drain from the wastes.

2.2.2 Source of ARD

The primary sources of acidic drainage at a mine site are areas in which mine rock has been disturbed and the exposed surface area increased. These areas include:

- underground mine workings;
- open pits;
- mine rock piles (waste rock dumps);
- ore and low grade stockpiles;
- tailings impoundments;
- roads, dams etc. constructed from mine rock; and,
- exposed rock faces in rock cuts for roads, etc.

In addition, acid rock drainage can occur from other areas of the mine site which usually contain a lesser volume of rock but should be considered in the evaluation of the site, such as:

- exposed rock faces in rock cuts for roads, etc.;
- concentrate load-out facilities;
- residual coarse and fine ore stockpiles;
• roads along which concentrate trucks travel;
• emergency spill ponds; and
• treatment plant sludge storage ponds.

It is important in developing a prediction test program to understand the difference in the physical and chemical controls on ARD from each of these sources.

Tailings and mine waste rock are probably the major sources of ARD from most sites, and therefore are the emphasis of most prediction programs (Steffen and Kirsten Inc., 1992).

2.2.3 Acid Generation

The oxidation of sulfur and hydrolysis of iron from sulfur- and iron-bearing minerals is the source of acidity. Minerals or other substances in rocks containing sulfur can be divided into four main types. The first group, sulfide minerals, is regarded as the most important in generation of acidity. Among them, pyrite and pyrrhotite are the most important acid producers due to their abundance. The second group, sulfate-containing minerals, are generally not regarded as acid generators because the sulfur present is already in its highest oxidation state. However, dissolution of limonite containing jarosite \( \text{KFe(SO}_4\text{)}_2(\text{OH})_6 \) or other iron sulfates (e.g. melanterite) in heavily oxidized rock will generate acidity because some iron (II) will oxidize and precipitate as iron (III) hydroxide. The third and the fourth groups, organically-bound sulfur and native sulfur,
are oxidized slowly and generally are not sufficiently abundant to be a source of acidity in hard rock ore deposits.

The primary ingredients for acid generation are reactive sulfide minerals, water or a humid atmosphere and an oxidant, particularly oxygen from the atmosphere. Generally, acid generation refers to the reaction in the microenvironment and around a sulfide mineral grain. Pyrite is the most commonly used mineral to illustrate the acid generation process although other sulfide minerals have different reaction pathways, stoichiometries, and rates.

The “standard” reaction for pyrite oxidation in neutral or alkaline conditions can be represented as follows:

\[
\text{FeS}_2 + \frac{7}{2}\text{O}_2 + \text{H}_2\text{O} \rightarrow \text{Fe}^{2+} + 2\text{SO}_4^{2-} + 2\text{H}^+ \quad (2-1)
\]

The dissolved Fe\(^{2+}\), \(\text{SO}_4^{2-}\), and \(\text{H}^+\) represent an increase in the total dissolved solids and acidity of the water. Unless neutralized, the increasing acidity is often associated with a decrease in pH. If the surrounding environment is sufficiently oxidizing, much of the ferrous iron will oxidize to ferric iron:

\[
\text{Fe}^{2+} + \frac{1}{4}\text{O}_2 + \text{H}^+ \rightarrow \text{Fe}^{3+} + \frac{1}{2}\text{H}_2\text{O} \quad (2-2)
\]
At pH values above 2.3 to 3.5, the ferric iron will precipitate as Fe(OH)$_3$, leaving little Fe$^{3+}$ in solution while lowering the pH at the same time due to the release of hydrogen ions:

$$Fe^{3+} + 3H_2O \rightarrow Fe(OH)_3 + 3H^+ \quad (2-3)$$

Any Fe$^{3+}$ from Reaction (2-2) that does not precipitate from solution through Reaction (2-3) may oxidize additional pyrite:

$$FeS_2 + 14 Fe^{3+} + 8H_2O \rightarrow 15Fe^{2+} + 2SO_4^{2-} + 16H^+ \quad (2-4)$$

The net reaction for Reaction (2-1), (2-2), (2-3) and (2-4) is:

$$FeS_2 + 15/42O_2 + 7/2H_2O \rightarrow Fe(OH)_3 + 2SO_4^{2-} + 4H^+ \quad (2-5)$$

Based on these reactions, 1 mole of pyrite will produce 4 moles of $H^+$. Certain bacteria may participate in the oxidation of sulfide minerals by deriving their energy for cell reproduction from the chemical reaction energy released during the oxidation process and by accelerating the rate at which some of the above reactions proceed, thereby increasing the rate of acid generation.
2.2.4 Acid Neutralization

The presence of neutralization material in the tailings plays an important role in minimizing the impact of acid generation by decreasing levels of acidity, increasing pH towards neutral values, and causing metals in solution to precipitate. There are many minerals that can neutralize acid. Some are more effective than others depending on their relative reactivity and the pH at which they buffer the reaction. These minerals include: carbonates, hydroxides and oxides, and silicates.

**Carbonates**

Carbonate minerals are regarded as the most effective natural acid neutralizing agents. They are relatively reactive and can buffer pH in the circum-neutral range. The most common example of carbonate is calcite (CaCO$_3$) which buffers pH near 7. Other common carbonates are dolomite (CaMg(CO$_3$)$_2$) and magnesite (MgCO$_3$) which are capable of buffering pH to 6 to 7. Another common carbonate is siderite (FeCO$_3$). Siderite is an effective neutralizing agent under non-oxidizing conditions because ferrous iron does not oxidize to ferric iron. It is not a useful buffering agent, however, under well-oxidized conditions because the oxidation of ferrous to ferric iron and subsequent precipitation of ferric iron hydroxide produces as much acidity as is consumed by the mineral.

The primary carbonate species are related by the following pH-dependent reactions.

\[
H_2O + CO_2 \leftrightarrow H_2CO_3^0 \leftrightarrow HCO_3^- + H^+ \tag{2-6}
\]
At any particular pH, one of these species will be dominant in terms of concentration with $\text{H}_2\text{CO}_3^0$ dominant below a pH of approximately 6.3 to 6.4 (depending on temperature and pressure) and $\text{CO}_3^{2-}$ dominant above a pH of approximately 10.3. At pH values between 6.3 and 10.3, the two species will be similar in concentration and significant pH buffering can occur which would allow pH to remain nearly constant despite the addition of acidity or alkalinity.

These reactions and the capacity for pH buffering have major implications for neutralization of acid drainage. For example, if $\text{CO}_3^{2-}$ from the dissolution of a carbonate mineral is continuously added to acidic water (e.g., at pH 2), the pH will slowly increase towards 6 through the reverse reactions Equations (2-6) and (2-7) forming $\text{H}_2\text{CO}_3^0$ by consuming 2 moles of acidity ($\text{H}^+$) for each mole of $\text{CO}_3^{2-}$. Around pH 6.3, $\text{HCO}_3^-$ begins forming in preference to $\text{H}_2\text{CO}_3^0$. Because $\text{HCO}_3^-$ represents a consumption of only 1 mole of acidity for each mole of carbonate, neutralization beyond this point becomes less efficient. As $\text{CO}_3^{2-}$ continues to be added to the water, the pH begins to rise above 6.4, however, this is resisted by $\text{H}_2\text{CO}_3^0$ which begins to break down to $\text{HCO}_3^-$ and releasing acidity according to Equation (2-6). This release of acidity requires additional carbonate to raise the pH above 6.4. That means neutralization with carbonate to pH 7 requires significantly more carbonate than for neutralization to pH 6. Because a pH near 6 may not be sufficiently neutral to ensure protection of freshwater life and the precipitation of
metals to acceptable levels, carbonate consumption to reach a pH of 7 to 8 may often be a more reliable target for neutralization, in general requiring significantly more carbonate.

**Hydroxides and oxides**

Hydroxides and oxides can also neutralize acid and buffer solutions, although at a lower pH than carbonate minerals. Examples are iron hydroxide, which buffers pH between 3.0 and 3.5, and aluminum hydroxide that buffers pH between 4.5 and 5.0. The reactions are as follows:

Iron hydroxide buffers pH between 3.0 and 3.5

$$\text{Fe(OH)}_3 + 3\text{H}^+ \rightarrow \text{Fe}^{3+} + 3\text{H}_2\text{O}$$

(2-8)

Aluminum hydroxide buffers pH between 4.5 and 5.0

$$\text{Al(OH)}_3 + 3\text{H}^+ \rightarrow \text{Al}^{3+} + 3\text{H}_2\text{O}$$

(2-9)

**Silicates**

Silicates are the largest mineral group and comprise an extensive variety of minerals composed of silicon combined with other common elements such as iron, aluminum, potassium, sodium, calcium and magnesium. They are the most important rock forming minerals and thus make up the bulk of the earth’s outer shell. Silicate minerals were divided into more than ten groups. They have different reactivities under weathering.

The buffering pH of silicate minerals depends on the solubility of the metals released by interaction between the mineral and acidity conditions. For example, the buffering
capacity of a relatively simple calcium silicate such as wollastonite (CaSiO₃) would be controlled by the solubility of calcium minerals, assuming buffering occurs in equilibrium with the atmosphere (Reaction 2-10). The pH would be expected to be buffered at near neutral conditions.

\[
\text{CaSiO}_3 + \text{H}_2\text{O} + 2\text{H}^+ \rightarrow \text{Ca}^{2+} + \text{H}_4\text{SiO}_4 \quad (2-10)
\]

A similar conclusion would be expected for magnesium silicate minerals. Alumino-silicates release aluminum when interacting with acid. For example, for kyanite:

\[
\text{Al}_2\text{Si}_2\text{O}_5(\text{OH})_4 + 6\text{H}^+ \rightarrow 2\text{Al}^{3+} + 2\text{H}_4\text{SiO}_4 + \text{H}_2\text{O} \quad (2-11)
\]

However, aluminum solubility is controlled by the solubility of aluminum hydroxide which buffers pH around 4.5 (Equation 2-12). So, the importance of mineralogy is evident for understanding the buffering capacity of minerals.

\[
\text{Al}^{3+} + 3\text{H}_2\text{O} \rightarrow \text{Al(OH)}_3 + 3\text{H}^+ \quad (2-12)
\]

Silicate neutralization in ARD systems has been identified by elevated concentrations of silica and aluminum in tailings water and waste rock drainage (Morin et al., 1991; and Blowes and Ptacek, 1994). Morin et al. (1991) suggests that ARD has elevated silica and aluminum concentrations due to alumino-silicate mineral dissolution. Blowes and Ptacek (1994) also suggest that the source for aluminum ions for gibbsite (Al(OH)₃) precipitation, during the step-wise pH decrease, is alumino-silicate dissolution.
2.2.5 Development of ARD

The development of ARD and its fate in the receiving environment is made more complex by a series of chemical, physical and biological oxidation reactions along the flow-path (Figure 2.6). Therefore, the ultimate drainage water quality depends on characteristics controlling contaminant release, rate of mineral oxidation and acid generation, rate of metal leaching, rate acid neutralization, solubility of metal species, accumulation of oxidation products and the reactions along the flow path.

Figure 2.6 Simplified ARD development and the reactions along the flow-path (Lawrence and Day, 1997).
2.2.6 ARD Prediction

The purpose of ARD prediction testing is to provide sufficient data to satisfy the operators, owners and the regulatory authorities that the potential for ARD generation can be predicted and controlled. Mining companies are required to provide evidence that ARD generation from waste materials will be controlled during operation and long after mine closure. On the other hand, regulators are required to develop and apply regulations that are both fair to the mine operator and protective of the environment.

ARD predictions are very important from both environmental and economic perspectives. Information derived from properly conducted and interpreted static tests, in conjunction with experience, can form the basis for preliminary estimates of metal leaching and ARD. The early identification of "problem" wastes and development of an appropriate waste management plan can significantly reduce long term environmental problems and remediation costs. Early identification and incorporation of control measures can also reduce the financial liabilities of maintaining long term collection and treatment facilities. It is important to the mining company as well as to the regulatory authorities to do sufficient, appropriate testing to ensure that all potential problems have been identified and addressed.

Several ARD prediction tests and methods are in use or have been proposed which differ in complexity of procedure, complexity of data interpretation, time required to achieve a predictive result, and the cost of carrying out the test. Broadly, ARD prediction test procedures can be divided into two categories, static tests and kinetic tests. Static tests are
designed primarily to examine the balance between the acid producing components and the acid consuming components of a mine waste sample. The term static is used since the tests do not consider the relative rates of acid production and consumption. The Standard Acid-Base Accounting test (Sobek et al., 1978) and the Modified Acid-Base Accounting test (Lawrence and Wang, 1997) are two major methods among a number of static test procedures. The objective of kinetic tests is to predict the long term weathering characteristics of a mine waste material as a function of time.

From a geochemical perspective, acidic drainage can be viewed as the result of competition between acid-generating and acid-neutralizing minerals, with the dominant minerals regulating the chemistry of water passing over the material. Acid-producing minerals are generally reactive sulfide minerals. Acid-consuming minerals are primarily carbonate minerals although hydroxides and silicates can also provide neutralization potential. Static tests involve laboratory methods to determine the acid-consuming capacity of the sample, termed the Neutralization Potential or NP, and the acid-producing potential, termed the Acid Potential or AP. The difference between the two values is termed the net neutralization potential or NNP, where \( NNP = NP - AP \). The ratio between the two values is termed the NP/AP Ratio or NPR. NP, AP and NNP are expressed in consistent units (e.g., kg CaCO\(_3\) equivalent/tonne of sample) to facilitate comparison of the values. Theoretically, a sample is net acid-generating if the NNP is less than zero, the values of NNP between -20 and +20 tonnes CaCO\(_3\)/1000 tonnes of sample (-2% to +2% CaCO\(_3\)) may be acid-generating. This range of uncertainty can be attributed to errors inherent in the test procedure, conversions to total acidity, analytical error and small
sample size (Figure 2.7). In addition to the NNP value, the ratio between the NP and AP should also be considered. Samples with 3 times the neutralization potential to the acid potential can be considered acid-consuming. Samples with less than a 1:1 ratio are probably acid-generating. Between these ratios however samples are neither clearly acid-generating or acid-consuming due to other factors which influence acid generation potential (Figure 2.8) (Steffen and Kirsten Inc., 1992).

Figure 2.7 Interpretation of Acid-Base Account results based on NNP values (Steffen and Kirsten Inc., 1992).

Figure 2.8 Interpretation of Acid-Base Account results based on NP:AP ratios (Steffen and Kirsten Inc., 1992).
2.3 Tailings Disposal Problems at Eskay Creek Mine

The location of the Eskay Creek minesite does not provide sufficient or suitable space for a tailing storage pond facility. The tailings is currently being trucked and discharged into a mountain lake 8 kilometers from the mill - Albino Lake. In designing the mill, a decision was made to filter the flotation plant tailings for disposal with the waste rock using tracks for transportation. There is the possibility of using the tailing material as backfill in the underground mine.

In either case, it is important to establish the possible generation of acid and alkaline drainage from this stored material. The sulphidic material discharged from concentrators becomes reactive upon exposure to air and/or water. Pyrrhotite (Fe$_n$S$_m$) and pyrite tailings in particular which are discharged from beneficiation process in mining and mineral processing activities consist of finely ground particles. Upon exposure to oxygen and moisture, the tailings become reactive due to the instability of the metal sulphide minerals present which undergo spontaneous oxidation, thereby producing a weak sulphuric acid. Therefore, long-term behaviour of the tailings can have a significant impact on the environment, due to acid rock drainage (ARD).
2.4 Options for Tailings Disposal

Strategies to manage mine tailings in an environmentally benign way are invariably a key hurdle for both the approval process and long-term operation of mining projects throughout the world. Various strategies have been employed including surface (land-based) tailings disposal, backfill, discharge to freshwater bodies, disposal with organic cover and, in some coastal operations, disposal through submarine tailings discharge.

2.4.1 Submarine Tailings Disposal

It is important to establish a working definition of the term *submarine tailings discharge* (STD) since this terminology has previously been applied to a wide range of tailings disposal practices. *Bona fide* STD operations are considered to be those whereby tailings solids are placed in deep compartments of the ocean through discharge of a tailings slurry via a submerged outfall located in water depths typically exceeding 50m. Following discharge, a turbidity currently facilitates transport of tailings solids to deeper waters. Examples of such operations include the former BHP Island Copper Mine, Rupert Inlet, B.C. (Ellis et al., 1995), Kitsault Molybdenum Mine, Alice Arm, B.C. (Pederson et al., 1995) and Misima Gold-Silver mine, Papua New Guinea (Jones and Ellis, 1995). STDs have been proposed in circumstances where (i) it provides an economic advantage to the proponent, and/or (ii) is believed to be a more ecologically acceptable alternative to land-based tailings disposal.
The objective in using a STD system is to place tailings in a “semi-contained” deep marine environment where ambient dissolved oxygen concentrations are minimal. The lack of dissolved oxygen significantly reduces the potential for metals to be oxidized, mobilized into the water column and rendered bioavailable or transported away from the site of tailing deposition (Pederson, 1985). A further objective is to place tailings in a stable, depositionary environment, and to prevent tailings fines from entering the shallow, biologically productive euphotic zone (Rankin et al., 1997).

Several considerations are warranted in determining if STD is an appropriate strategy for a proposed coastal mine operation. These include the potential for the ore body to leach metals upon placement of tailings in the adjacent marine environment (i.e., geochemical stability), the dispersive nature of the receiving environment, the rate and duration of the proposed tailings discharge and the resultant potential for adverse ecological impacts. In addition to the potential for adverse environmental impacts during mine operation, the potential for post-operational recovery of regions affected by tailings discharge must be considered.

Table 2.2 provides a summary of potential environmental concerns associated with discharge of metal mine tailings to the marine environment. In some cases there simply has not been adequate data collected from any site to confirm or negate some of the listed items. A further consideration in weighing the importance of these items is whether a statistically significant deviation from the norm is environmentally significant. Large-
scale ecosystem parameters can often be quite variable and may hamper the ability to statistically resolve temporal and spatial trends.

Table 2.2 Effects potentially associated with submarine tailings discharge.

1) Physical Disturbance to Sea Bed
   - acute and chronic smothering of benthic habitat.
   - severe dilution of detrital organic carbon.
   - catastrophic effects of sediment instability: turbidity currents, slope failure and sediment resuspension.
   - changes in sediment particle size distribution, leading to packing or delayed consolidation following deposition; enhanced nepheloid layer.

2) Physical Disturbance to Water Column
   - increased turbidity in the water column.
   - changes in ocean circulation patterns due to changes in bathymetry and density effects.

3) Geochemical Effects
   - increased concentrations of dissolved metals/metalloids in water, sediment, and sediment interstitial water.
   - possible enhancement of upward pore water diffusion to the water column in association with changes in sediment compaction.
   - changes in oxidation-reduction potential of near-surface sediments.
   - alterations in benthic-pelagic fluxes of metals and nutrients.

4) Chemical Effects
   - increased bioavailability of metals.
   - alterations in chemical speciation (e.g., methylation/demethylation) and fate.
5) Organism-level Effects
- effects on microbial community of water column or sea bed.
- avoidance reactions by mobile species, or other behaviour responses such as valve closure in bivalves or changes in burrowing behaviour.
- acute and/or chronic toxicity.
- clogging of fish or invertebrate gills and invertebrate filtering appendages.
- sub-organismic effects (adaptive and maladaptive biochemical, physiological, ultrastructural and/or histological alterations). Changes in energy allocation with effects on growth and reproduction.
- impaired fitness of transient species such as highly mobile epifauna and pelagic fish species.

6) Ecological/Ecosystem-level Responses
- smothering or impairment of hard-substrate communities.
- deleterious or impairment of hard-substrate communities.
- deleterious effects on planktonic communities.
- effects on primary productivity (of macrophytes, phytoplankton, or autotrophic microplankton).
- effects on heterotrophs (secondary production).
- effects on predation or competition, leading to shifts in community structure.
- loss of rare organisms; reduced biodiversity.
- reductions in microhabitat diversity and ecosystem redistribution of uniform sediment type.

7) Human Resource Use
- food-source contamination.
- interference with harvesting, tourism, or recreation.
- aesthetic effects.
2.4.2 Dam Construction Tailings Disposal

Dam construction is a conventional tailing disposal method which would result in unsaturated tailings exposed to atmospheric oxygen and tailings disposal under a water cover which would result in saturation of the entire tailings mass. Exposed unsaturated tailings will reach a higher temperature during the summer. This would allow for high production of acid and possible establishment of Thiobacillus Ferroxidans. This would result in contamination of ground water.

2.4.3 In-mine Tailings Disposal

Only a faction of the tailings, usually the coarse fraction, has been utilized underground as backfill. A high density tailings fill which accommodates a larger volume of tailings than traditional hydraulic fill, with the inclusion of fine tailings and a relatively low moisture content, is known as paste fill. The use of paste fill is becoming increasingly popular around the world, however, paste fills prepared entirely of tailings, devoid of aggregates, have limited strength and thus have had limited use. Hence, supplementary, strength enhancing aggregate material, such as sand, gravel and rock chips are added to reinforce the paste fill. Therefore, the paste fills, as practiced today, are far from being a total tailings fill.

Amaratunga and Annor (1989) first reported the Cold Bond Tailings Agglomeration (CBTA) process developed at the Tailings Agglomeration Research Laboratory at Laurentian University. It involves subjecting dewatered fine tailings to a rolling action in a disc or drum pelletizer, in the presence of alkaline or non alkaline binders and water.
Unlike in the iron ore pelletization industry, tailings are agglomerated using cementitious binders and for strength enhancement, no heat induration process such as sintering is employed. Cold-bonding takes place due to curing, and a curing chamber is preferably used to develop the pellet strength. Hence, the process is called Cold-Bond Tailing Agglomeration (CBTA) process. The concept of CBTA process is shown in Figure 2.9.

Figure 2.9  The Concept of Cold-Bond Tailings Agglomeration (CBTA) for Paste Backfill (Amaratunga and Hein, 1997).
The CBTA process offers a novel concept for management of reactive fine tailings to develop a high strength, high modulus total tailings paste fill. This fill, known as the Agglomerated Tailings Paste Fill (ATPF) consists of aggregates made up of agglomerated fine tailings in a tailings matrix. Furthermore, the ATPF increases the volume of fine tailings (<45 m) sent underground, and may significantly reduce surface impoundment of tailings. The technique also provides an environmentally sound mill waste disposal method with several economical and operational advantages. Environmentally, the ATPF technique has great merit in the disposal of reactive sulphide mill tailings which greatly reduces the potential for acid mine drainage (AMD) from surface impoundment ponds (Amaratunga and Hmidi, 1998).

It appears that long-term stability of the cured pellets is highly dependent upon the initial dosage of the additives provided to the pellet. The primary purpose of the chemical additives is to enhance binding properties. However, a secondary purpose is to act as an inhibitor for acid generation. The chemical additives used in the laboratory study were selected according to their toxicity, environmental friendliness and cost. The majority of these chemicals have been reported in the literature as effective chemical agents in retarding ARD. Detergents [sodium lauryl sulphate (SLS)], potassium sorbate, sodium benzoate and sodium chloride can be used at low dosages without any environmental problem. Previous studies showed that (SLS) is more attractive in treating ARD from mine rock than potassium sorbate and sodium benzoate (Siwik et al., 1989). Sodium silicate was also studied according to previous research work done in treating pyrite tailings (Misra, 1993).
Two commercially available surfactants “ENVIROSEAL” and “IMPC” were among the additives used in the study. Enviroseal is the general name for a “family of coatings” which are employed in several industrial and commercial sectors. The product known as Entac in Canada in fact has been renamed as Enviroseal DC. It is understood that the manufacturer can adjust the process to regulate the physical and chemical properties of the surfactant once applied and cured. Enviroseal DC is an organic emulsion which is produced form naturally occurring resins (tree sap). It is produced in Hamilton, Ontario, Canada and generally used by INCO for dust control of its tailings disposal site. The Enviroseal products are environmentally friendly while a previous study showed promising results when the material was used as a retardant in the prevention of acid mine drainage for mine waste rock (Moskalyk et al., 1995).

2.4.4 Organic Cover Tailings Disposal

Different types of covers are being used for limiting the oxygen infiltration and thus to prevent or reduce the acid drainage production in waste rocks or tailings. Some of the covers commonly used in industry today are as follows: water cover, organic and inorganic covers, soil covers, clay slurry seal and sand-bentonite mixtures. Presently, large quantities of organic material are stockpiled, or may be available in the near future, from urban and industrial sources. Cities in Ontario are capable of producing approximately 680,000 tonnes of municipal solid waste (MSW) compost annually and create comparable amounts of sewage sludge, which is currently landfilled (Elliott et al, 1995). Peat from bogs in the Canadian shield region, although not a waste material,
represents a vast renewable source matter. Peat bogs are often found near base metal and precious metal mines.

These materials may provide effective and affordable solutions to the reclamation of acidic mine tailings. Sulfate and iron reduction rates may be limited in a tailings system by a lack of organic substrates for bacterial metabolism. Consequently, an organic cover layer on the tailings may provide an important source of carbon compounds for bacteria as decomposition and leaching proceeds. The biological oxygen demand of an aerobic, actively-decomposing organic layer also constitutes a strong sink for atmospheric oxygen, preventing it from moving down the oxygen concentration gradient toward the ARD oxidation processes in the tailings.

Oxidation of an organic cover layer material will eventually decline as the remaining material becomes more humidified and more resistant to further decomposition. Therefore, the biological oxygen demand of the organic cover layer will eventually reach a lower level based on a lower input rate of natural carbon compounds and other nutrients into the cover layer. The resistance that the cover layer will still be able to offer to the downward diffusion of atmospheric oxygen will then be determined by its physical properties, especially its depth and gas-filled porosity. Gas-filled porosity, in turn, will be mainly determined by the structural composition of the material and the degree that the pore spaces are filled with water (i.e. water content).
Several different organic materials, such as peat, lime stabilized sewage sludge (LSSS) and municipal solid waste compost have been used in tailings disposal. Of the covers that have been tested before, the LSSS appear to offer the greatest potential for reducing metal loading in water migrating from oxidized tailings to the environment. The reason for this effect is that the LSSS is actively changing the underlying tailing environment by reversing the ARD processes with an increase in pH, decrease in dissolved metals concentrations and formation of a reducing environmental at the tailings - cover interface.

The main problem using organic cover tailings disposal is that organic materials can lead to methylation and cause even more dangerous bioaccumulates.
3. METHODS AND PROCEDURES

3.1 Sample Description and Preparation

The tailings come from two sources, the 109 zone and the NEX zone. Both the 109 and the NEX zone tailing materials were provided in two different batches at different times. The first batch (June, 1998) consisted of 109 filtered tailings, ball mill feed and flotation feed. The second batch (August, 1998) consisted of 109 and NEX tailings. The flotation tailings from Eskay Creek Mine has particle size distribution between 70%-85% -200 mesh, a moisture content of 11-15 % H$_2$O and may contain mainly rock material (some limestone). The particle size analysis is shown in Table 4.1.

Samples were prepared as required for testing as follows:

- The samples used for assay, acid-base accounting and shake flask testing were ground, crushed, and pulverized with a typical size range of 100% minus 60 mesh to 80% minus 200 mesh. In these cases, no further sample preparation was required.

- Tailing samples were air dried and used for leaching testing in the as-received size.

- Pellets samples from tailings agglomeration tests were used to conduct kinetic tests with a size range of 3/4" – 1/2" (100g), 1/2" - 1/4" (400g) and 1/4" - 0.1" (500g).
• Homogenized samples were taken from all tailing samples for screen analysis. Representative samples were taken from both tailings, ball mill feed and flotation feed for acid-base accounting and leach tests.

3.2 Test Methods

3.2.1 Static Prediction Tests

3.2.1.1 Determination of Acid-Generating Potential

In order to determine the Acid Potential (AP) of a sample, a sulfur analysis must be performed. A sample may contain several forms of sulfur. Among them, sulfide minerals are regarded as the most important in generation of acidity. In the standard acid-base accounting test method, the AP is calculated on the basis of total sulfur. In the modified acid-base accounting test method, the AP is calculated on the basis of sulfide-sulfur which normally is determined from sulfide sulfur analysis or calculated by the difference of total sulfur and sulfate sulfur. The purpose of using sulfide sulfur is to overcome AP overestimation in the standard acid-base accounting.

In these studies, for prediction of maximum Acid Potential (AP), the samples were analyzed for total sulphur, sulphate sulphur. The acid potential was calculated from the sulphide sulphur using the procedure shown in Appendix A. Sulphate was analyzed by the gravimetric method, total sulphur by gravimetric and LECO-IR Detector, and the sulphide was calculated by difference.
3.2.1.2 Determination of Neutralization Potential

There are two ways used to determine the neutralization potential in these studies: the standard acid-base accounting test method and the modified acid-base accounting test method.

Standard Acid-Base Accounting (Sobek ABA)

Standard acid-base accounting (ABA) developed by Sobek in 1978 (Sobek et al., 1978) is one of the most widely used static prediction tests in British Columbia. The objective of this part of the standard ABA is to determine the neutralization potential of the sample.

The analysis consists of two steps: a fizz test and NP determination. The fizz test is used to determine the volume and concentration of acid that is to be added to the sample during the NP determination procedure. One or two drops of 25 percent HCl are added to 0.5g of ground sample (minus 60 mesh) and the resulting reaction is rated as none, slight, moderate, or strong. The neutralization potential (NP) is determined by adding excessive hydrochloric acid to the sample, heating the pulp to ensure complete reaction, and then titrating the solution with sodium hydroxide to an end-point of pH 7. The NP is calculated from the calcium carbonate equivalent of the acid that was consumed during the reaction. The detailed procedure and calculations are shown in Appendix A.
Modified Acid-Base Accounting (Modified ABA)

The other method used to determine the neutralization potential is referred to as the modified NP determination procedure. This test was developed to overcome the overestimation of the neutralization potential due to the exaggerated low-pH and high temperature (near-boiling) of the acid reaction of the standard Sobek method.

The Acid Rock Drainage Prediction Manual provides a description of the equipment required and the method followed to perform a modified NP test. A revised procedure has been developed by Lawerence and Wang (1996) and it is commonly used to simulate the more realistic conditions at which NP minerals become reactive. In this procedure, the reaction temperature is kept at laboratory conditions (25°C) for a relatively long period of 24 hours. The solution is then titrated back to pH 8.3. The detailed procedure is described in Appendix A.

Both standard acid-base accounting and modified acid-base accounting NP tests were performed on several specimens from each tailing sample. In addition, the total inorganic and organic carbon (using LECO-IR detector and LECO-Gasometric) were analyzed to cast light on the carbonate contribution to NP.

Prior to conducting these tests, a paste pH is typically measured to determine if acid generation has already occurred in the sample. A paste pH above 7 indicates that reactive carbonate is still present in the sample. A paste pH is below 5, it indicates that the sample
contains acidity from previous acid generation. The detailed procedure for paste pH is presented in Appendix A.

3.2.2 Shake Flask Tests

In order to understand the time-dependent oxidation and metal leaching reactions in aqueous environments, a series of shake flask tests were designed and carried out on both NEX and 109 tailings material.

These tests were conducted at different temperatures (26°C and 33°C), initial pH, and bacterial conditions. The chemistry and pH of the leachate was monitored during the three weeks duration of each test. In one set of samples, acid was added to keep the pH at a controlled level of 4.5 to 5. In a second set, no acid was added. Acid-generating bacteria (Thiobacillus ferroxidans) were added to a third set of samples to cast light on the influence of bacteria on oxidation of sulphides in aqueous environments.

The leachate samples were collected after 1, 2 and 3 weeks. After removal of suspended solids, the solutions were sent to ASL Laboratories for metal scan and sulphate analysis. The test procedure is presented in Appendix A.

3.2.3 Kinetic Prediction Test

Kinetic testing has become one of the most common methods of assessing the impact of acid rock drainage in Canada. The purpose of conducting kinetic tests on flotation tailings
from Eskay Creek Mine is to determine the rate of oxidation and metal leaching in a simulated environment. The most commonly used kinetic tests utilize humidity cells and leaching cells. The kinetic tests conducted as part of this research employed humidity cells for backfill materials (pellets from tailing agglomeration process) and leaching cells for tailings samples. The procedure followed to perform the kinetic tests is outlined in Appendix A.

**Humidity cells**

Regular cylindrical humidity cells (4” x 8”) were used to conduct the kinetic test for 109 pellets and NEX pellets. The cells were exposed to weekly cycles of dry air, humid air and washing rinsing. Leachates were tested weekly for pH and conductivity. The cells have been run for 6 months. At the end of the experiment, the content of the cells will be chemically and mineralogically examined to assess the effect of weathering.

**Leaching cells**

A total of two leaching cells (for 109 tailings and NEX tailings) have been conducted at normal temperatures for 6 months. Short cylindrical containers (4” x 2.5”) have been used in this test. The containers were exposed to the air from the top and leached weekly with distilled water. Leachates were tested weekly for pH and conductivity.
3.2.4 Mineralogy

The X-Ray Diffraction (XRD) and the Scanning Electron Microscope analyses were conducted on the tailings and ball mill feed samples to analyze the chemical composition and to study the mineralogical context of the ore.

3.2.5 Analyses

An ICP scan was used to analyzed for heavy and precious metals on tailings samples. Assays were conducted both at UBC (MMPE Assay Laboratory and Chemex Labs. UBC). Analyses include assays of the whole sample and specimens of different size ranges. The analytical data are included in Appendix B.
4. RESULTS

4.1 Sample Characterization

Samples were characterized by the particle size analysis, the x-ray diffraction (XRD) analysis and the Scanning Electron Microscope (SEM) analysis. The results of these various analyses are presented below.

4.1.1 Particle Size Analysis

Particle size analyses were performed on all the samples that were used for static prediction tests and kinetic tests. Table 4.1 shows the results from all tailings samples for screen analysis.

Table 4.1 Particle size analysis for all tailings samples in Eskay Creek ARD studies.

<table>
<thead>
<tr>
<th>Screen Size (mesh)</th>
<th>wt (g)</th>
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<th>cum % retain</th>
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Sample: NEX Filtered Tails-Test #2

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<td>wt (g)</td>
<td>% retain</td>
<td>cum % retain</td>
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Sample 109, 2nd batch

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<th>cum % passing</th>
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Sample 109, Flotation Feed

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<th>% retain</th>
<th>cum % retain</th>
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<td>99.1</td>
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</tr>
<tr>
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<td>110</td>
<td>72.1</td>
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<td>24.3</td>
<td>75.7</td>
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<td>74</td>
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Sample 109, 2nd batch

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<th>Screen Size (mesh)</th>
<th>(microns)</th>
<th>wt (g)</th>
<th>% retain</th>
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<td>99.3</td>
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<tr>
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<td>149</td>
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<td>4.8</td>
<td>5.5</td>
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<td>110</td>
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<td>81.9</td>
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</tr>
<tr>
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<td>53</td>
<td>91.5</td>
<td>27.1</td>
<td>74.1</td>
<td>25.9</td>
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<tr>
<td>400</td>
<td>37</td>
<td>36.8</td>
<td>10.9</td>
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<td>15.0</td>
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</table>
4.1.2 Mineralogy Analysis

The computer printouts of the X-Ray Diffraction (XRD) analysis can be found in Appendix B. The Scanning Electron Microscope (SEM) inspection is also presented in Appendix B. The analysis results have shown that the dominant sulphide minerals are pyrite, sphalerite and galena.

4.2 Static Prediction Tests

The static prediction test that were performed can be divided into two main categories: determining the acid-generating potential and determining the neutralization potential. The acid-generating potential was determined using the Sobek method and the Modified method. The neutralization potential was determined also using both of the above methods.

4.2.1 Acid-Generating Potential

To calculate the acid-generating potential of a sample, a sulphur analysis was performed. In the case of Sobek method (Sobek ABA), the total sulphur content of the sample must be determined. For Modified method (Modified ABA), both the total sulphur and the sulphate sulphur must be determined. Table 4.2 shows the acid-generating potential results calculated from the procedure in Appendix A for all tailing samples.
Table 4.2  Acid-generating potential for all tailings samples in Eskay Creek ARD studies.

<table>
<thead>
<tr>
<th>Sample Type</th>
<th>% Sulfate (Total)</th>
<th>S(%)</th>
<th>AP Sobek</th>
<th>AP Modified</th>
<th>AP Sobek/AP Modified</th>
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<tr>
<td>109#1 Filtered Tailings</td>
<td>0.07</td>
<td>3.88</td>
<td>3.84</td>
<td>120</td>
<td>119</td>
</tr>
<tr>
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<td>0.07</td>
<td>5.46</td>
<td>5.37</td>
<td>168</td>
<td>168</td>
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<td>0.09</td>
<td>5.36</td>
<td>5.26</td>
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<td>165</td>
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<tr>
<td>NEX Tailings</td>
<td>0.11</td>
<td>1.86</td>
<td>1.9</td>
<td>59</td>
<td>55</td>
</tr>
</tbody>
</table>

Note:  For Sobek ABA, AP = Percent sulphur * 31.25 kg CaCO₃ equivalent per tonne;
For Modifed ABA, AP = Percent sulphide sulphur * 31.25 kg CaCO₃ equivalent per tonne; Where, Sulphide sulphur = Total sulphur – Sulphate sulphur.

4.2.2 Neutralization Potential

Before the neutralization potential of the samples could be calculated, a paste pH analysis had to be conducted to determine if the samples had already generated acidity. After the paste pH was measured, the NP of the samples could be determined using acid-base accounting method. The results of these analyses are presented below.

4.2.2.1 Paste pH

The results of the paste pH analyses are shown in Table 4.3.
Table 4.3 Paste pH results for the tailings sample in Esaky Creek ARD studies.

<table>
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<tr>
<th>Sample Type</th>
<th>Paste pH</th>
<th>Av. Paste pH</th>
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<td>11.08</td>
</tr>
<tr>
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<td>11.08</td>
<td></td>
</tr>
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<td>8.53</td>
</tr>
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</tr>
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<td>109#2 Tailings</td>
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</tr>
<tr>
<td>NEX Tailings</td>
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<td>8.20</td>
</tr>
<tr>
<td>NEX Tailings</td>
<td>8.17</td>
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</tr>
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</table>

4.2.2.2 Neutralization Potential

The NP results from all tailing samples that were determined by Sobek method and Modified method are presented in Table 4.4. The completed test data for the Sobek and Modified tests are shown in Appendix B.

4.2.3 Summary of Static Prediction Tests

The results of the various analyses used to conduct the static prediction tests are summarized in Table 4.5. Figure 4.1 shows that the neutralization potential correlation between the standard acid-base accounting method and the modified acid-base accounting method.
Table 4.4 Neutralization Potential for tailings samples in Eskay Creek ARD studies.

<table>
<thead>
<tr>
<th>Sample Type</th>
<th>NP Sobek</th>
<th>NP Modified</th>
<th>Av. NP Sobek</th>
<th>Av. NP Modified</th>
<th>NP Sobek/NP Modified</th>
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<td>41</td>
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</table>

Table 4.5 Summary of Static Prediction Tests.

<table>
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<tr>
<th>Sample Type</th>
<th>Av. NP Sobek</th>
<th>Av. NP Modified</th>
<th>AP Sobek</th>
<th>AP Modified</th>
<th>NP/AP Sobek</th>
<th>NP/AP Modified</th>
<th>Net NP Sobek</th>
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<td>-144</td>
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<td>0.7</td>
<td>-18</td>
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53
Figure 4.1 The neutralization potential correlation between the standard ABA test and the modified test in static prediction test.
4.3 Shake Flask Tests

The results of shake flask tests on metal scan and sulphate analysis for both 109 and NEX tailings samples are shown in Table 4.6. Note that when collecting a leachate sample, the pulp was diluted with 50 ml of distilled water prior to collection of the sample for 21 days, this was not done for 7 and 14 days. The data presented in Table 4.6 was corrected to account for this dilution. Complete analytical data are shown in Appendix B. Figure 4.2 shows the relationship between pH and time at different conditions for 109 tailings and NEX tailings. Figure 4.3 shows the relationship among Ca, Mg, and SO₄ under different conditions for 109 tailings and NEX tailings. The relationship between Zn and time at different conditions for 109 tailings and NEX tailings are shown in Figure 4.4.
Figure 4.2 The relationship between pH and time under different conditions (no bacteria and using bacteria) for 109 tailings and NEX tailings.
Table 4.6  Metal scan and sulphate analysis for both 109 and NEX tailings samples.

<table>
<thead>
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<th>Metal</th>
<th>Sample Type</th>
<th>Time (Day)</th>
<th>NA, NB, 26°C</th>
<th>NA, NB, 33°C</th>
<th>A, NB, 26°C</th>
<th>A, NB, 33°C</th>
<th>A, B, 26°C</th>
<th>A, B, 33°C</th>
</tr>
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<tbody>
<tr>
<td>Ca (mg/L)</td>
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<td>279</td>
<td>338</td>
<td>674</td>
<td>686</td>
<td>528</td>
<td>792</td>
</tr>
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<td></td>
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</table>

Note: NA = without acid, A = with acid, NB = without bacteria, B = with bacteria.
Figure 4.3 The relationship among Ca, Mg, and SO$_4$ under different conditions for 109 tailings and NEX tailings. (Note: NA = without acid, A = with acid, NA = without bacteria, B = with bacteria)
Figure 4.4 The relationship between Zn and time at different conditions for 109 tailings and NEX tailings.
4.4 Kinetic Tests

Two types of kinetic tests were conducted: humidity cells for 109 pellets and NEX pellets; Tailings leaching cells for 109 tails and NEX tails. The kinetic test results (pH and conductivity) have been presented below. The raw data from which the following figures were generated can be found in Appendix A.

4.4.1 pH Value

The pH of the sample leachates for pellets and tailings is presented in Figure 4.5 and Figure 4.6 respectively.

![Figure 4.5](image)

Figure 4.5 The relationship between pH and cycle time in kinetic test for 109 and NEX pellets.
Figure 4.6 The relationship between pH and cycle time in leaching test for 109 and NEX tailings.
4.4.2 Conductivity

Figure 4.7 and Figure 4.8 present the conductivity measured during the two types of kinetic tests.

Figure 4.7  The relationship between conductivity and cycle time in kinetic test for 109 and NEX pellets.
Figure 4.8  The relationship between conductivity and cycle time in leaching test for 109 and NEX tailings.
5. DISCUSSION

This section will provide an explanation of the results presented in previous section.

5.1 Sample Characteristic

5.1.1 Particle Size Analysis

Particle size analysis results have shown that the particle size of NEX tailings samples is much finer than the particle size of 109 tailings samples in that the average particle size of 109 tailings samples is 74% -200 mesh and 60% -270 mesh and the NEX tailings samples have an average particle size of ~85% -200 mesh and 72-73% -270 mesh, however, the tailings samples from both 109 and NEX zone have a fine particle size distribution. This means that the tailing samples were composed of small particles and therefore had a much greater surface area per unit mass than the tailings contained coarse particles. This higher relative surface area means that the oxidation and leaching rates are probably much higher than that of lower relative surface area.

5.1.2 Mineralogy Analysis

From the XRD analysis, it is apparent that the dominant sulphide minerals are pyrite, sphalerite and galena. Rock-forming minerals include quartz and micas (phlogopite and muscovite). Sulphides contribute to acid potential while the micas can contribute to neutralization potential at low pH conditions. Small amounts of carbonates that were observed in fizz tests were not traced by XRD.
Small amounts of chalcopyrite, associated with sphalerite and pyrite, were detected in reflection optical microscopy.

SEM inspection indicates that pyrite is the dominant sulphide. Galena is commonly found around pyrite grains (often square-shaped) or as inclusions. Galena inclusions are also associated with sphalerite (found within microfractures of sphalerite). These associations can be important with regard to accelerating or diminishing sulphide reactivity.

5.2 Static Prediction test

5.2.1 Acid-Generating Potential

The higher the sulphur content of a sample, the higher the acid-generating potential (AP) because AP is calculated by multiplying the sulphur content (in %) by a conversion factor of 31.25. Therefore the samples with the highest sulphur contents will be the samples with highest AP.

From the results of AP for all the tailing samples, the highest AP was measured in the ball mill feed and the lowest measured in NEX. The AP value for the flotation feed is close to that of ball mill feed.

Sulphate sulphur values are negligible (about 0.1%) and are apparently related to trace amounts of barite (Appendix B) in the ore (Roth et al., 1997). Therefore there is no
significant difference between Sobek method and Modified method to calculate AP. So the total sulphur (the standard acid-base accounting method) can be used adequately in these Eskay Creek samples for calculation of acid potential.

5.2.2 Neutralization Potential

5.2.2.1 Paste pH

The paste pH analysis is a standard step when conducting an acid-base accounting. It is used to determine if acid generation has already occurred in the sample. From the paste pH results, since all the paste pH are above 7, the samples, especially the 109 tails (pH 7.58) and NEX tails (pH 8.20), must contain some reactive carbonate. The above neutral paste pH values also suggest the presence of calcic NP (most probably lime) in the tailings.

5.2.2.2 Neutralization potential

In all samples analyzed for NP, acid-base accounting results show that both 109 and NEX tailings samples have low values (Low NP value means that the NP:AP ratio is less than 1:1 or the difference between NP and AP is less than –20 kg CaCO3 equivalent per tonne (Mine Rock Guidelines, 1992)) in neutralization potential. NEX tailings samples have the maximum NP values of all tailing samples. NP values for the flotation feed are the lowest NP values measured. The two 109 tailings samples are slightly different in their NP values (109#1 being about 30% higher than 109#2). The NP of NEX tailing samples is almost two times that of the 109 tailing samples. The NP value for ball mill feed samples is falling between the NEX and 109 tailings.
From the results of standard acid-base accounting and modified acid-base accounting, the Sobek and Modified NP values appear to be similar (the ratio between NP Sobek /NP modified values for all tailings samples are almost equal to 1). This indicates that the NP values are related to carbonates and are not overestimated. It is therefore suggested that for future NP qualification of Eskay Creek samples, Sobek test is adequate and there is no need for the Modified test.

5.2.3 Summary of Static Prediction Tests

According to the AP and NP results for all the tailings samples, it appears that the AP value of 109 tailings samples is significantly higher than that of NEX tailings samples, but the NP value of the NEX tailings samples is the highest NP value of the all tailings samples. So with a higher NP and a lower AP, the NEX tailings is slightly more environmentally benign in comparison to 109 tailings.

In both 109 and NEX tailings samples, the acid potential is significantly higher than the neutralization potential (the Net NP values for all tailing samples are all negative). This means that both 109 tailings and NEX tailings have a good probability for acid generation.
5.3 Shake Flask Tests

From the results of the relationship between pH and time (Figure 4.2), it is shown that the pH values for the 109 tailings and NEX tailings decrease with the time. There is little effect on pH by adding bacteria into the solution (the pH values with bacteria are a little bit lower than that without bacteria).

From the results of the sulphate, Ca and Mg concentration in the leachate samples (Figure 4.3), it is clear that temperature plays an important role in acid generation (indicated by the increased sulphate at higher temperatures). Calcium increases in the solution as a response to acid generation.

The high levels of zinc (Figure 4.4), manganese, lead and mercury (Table 4.6) also indicate the acid potential problems on both tailings samples. Especially, mercury can cause long term problems on ARD potential and environmental pollution because of its significant toxicity.

With the acid-base accounting results, the shake flask tests suggest that generation of acidic effluent (ARD) from both tailings is likely.

5.4 Kinetic Tests

The single biggest concern regarding the kinetic prediction tests was the length of time for which these tests should be conducted. Due to time constraints, the kinetic tests
carried out as part of this research could only be conducted for 20 weeks. This is quite a short period of time relative to most current kinetic tests and the analysis for metals has not been conducted because of lack of funds.

5.4.1 pH

Figure 4.5 and Figure 4.6 shows that the pH values obtained from humidify cells and leaching cells. From the results of the humidify cells, the pH values for both 109 and NEX pellets decreased at the beginning of the test and then kept on an average pH value of around 7.0. There is no significant change in pH values for the tailing leaching cells.

5.4.2 Conductivity

The conductivity measured from the humidify cells and leaching cells are displayed in Figure 4.7 and Figure 4.8. The conductivity for humidify cells decreased over time while the conductivity for leaching cells fluctuated severely during the first 10 weeks and then kept in a low average conductivity (0.6ms/cm).

In summary, the 109 and NEX pellets in humidify cell tests do not show a large acid potential. The 109 and NEX tailings in leaching cell tests also do not appear to pose a problem but these tests require continued time and analysis of metals. The testwork should also be done with bacteria-inoculated tests and under flooded conditions.
5.5 Implication of Tailings Disposal Alternatives

There are many methods that can be used to dispose the tailings from mining and mineral process activity: submarine tailings disposal, conventional dam construction tailings disposal, in-mine tailings disposal and organic cover tailings disposal. The following paragraph will discuss the possible tailings disposal methods used in Eskay Creek Mine.

Conventional dam construction tailings disposal

Dam construction tailings disposal is a conventional tailings disposal method. At Eskay Creek Mine, it is not possible to use this method because of the topographic feature of Eskay Creek minesite. The most important problem is that this method can result in the issues of ARD control because the unsaturated tailings have been exposed to atmospheric oxygen, it would produce high rates of acid and result in contamination of ground water.

In-mine tailings disposal

In-mine tailing disposal method is divided into three categories: conventional backfill tailings disposal, paste backfill tailings disposal and agglomeration tailings disposal.

Conventional backfill is that the tailings are mixed with rock fill/cement. In Eskay Creek Mine, the tailings from the mineral process activities were very fine (<200mesh) and had an average moisture content at 15 wt% so that tailings materials were very sticky and difficult to mix and homogenize with rock fill/cement.
Paste backfill tailings disposal that uses tailings materials are below 200 mesh is becoming increasingly popular around the world, however, paste fills prepared entirely of tailings have limited strength and the paste need to be pumped so that it can result in the mechanization transportation problems.

Agglomeration tailings disposal is that the fine tailing materials are agglomerated and then mixed with rock fill/cement. The agglomerated tailings with sufficient strength are used in the mine backfill in order to remove the need for surface disposal of the potentially acid-producing tailings, however, the agglomeration tailings disposal need to use 25-50% more cement than conventional backfill method and does not guarantee elimination of ARD because of the long term weathering effects on tailing materials.

Submarine tailings disposal

The tailings are currently being discharge into a mountain lake 8 kilometers from the mill – Albino Lake to reduce the oxidizing potential of tailings by limiting exposure to dissolved oxygen and solve the space problem for tailings storage. In submarine environments, some acid potential may still exist, but none of our kinetic tests indicate a problem so far. Submarine tailings disposal may be the best way to dispose tailings materials in Eskay Creek Mine.
6. CONCLUSION

In both 109 and NEX tailings, the acid potential is significantly higher than the neutralization potential. This means that both 109 tailings and NEX tailings have the probability for acid generation. Due to the higher NP and lower AP, the NEX tailings is slightly more environmentally benign compared to the 109 tailings.

Shake flask tests suggest that generation of acidic effluent (ARD) from both tailings is likely. Temperature is an important factor in acid generation. Lead, zinc, manganese, magnesium, arsenate (Appendix B) and mercury are main toxic metals associated with possible acid rock drainage from Eskay Creek tailings.

Kinetic tests show that the agglomerated tailings have a lower acid potential than the unagglomerated tailings. The long-term acid potential has not been determined until the whole kinetic tests have been finished. At this stage, simulation of tailings disposal on the surface with cycled waterflows does not show any acid-generating potential, however, these tests need to be continued for longer periods of time to be certain of this result and metal assays should be done to see if metal release is significant. Test work should also be done with bacteria-inoculated tests.
7. RECOMMENDATION

In the continuation of the baseline ARD study on Eskay Creek mill tailings, kinetic tests including standard tailings cells and columns (for simulation of sub-aqueous environments) should be continued for more information on pH, conductivity, sulfur concentration and metal release to predict the long-term acid potential of both 109 and NEX tailings.

Although the agglomeration testwork indicate that acid-generation is low, the long term uncertainty of this issue is high. Once cement is leached out of the pellets, acid-generation may begin. This process may take place very far in the future so its study really requires accelerated testing methods.
REFERENCES


Murphy, F.M. and Napier, W., 1996, "Eskay Creek Access Road: a Case Study", pp. 142-152.


APPENDIX  A

Procedures and Protocols
Standard Acid Base Accounting

Objectives

- To determine the balance between acid producing and acid consuming components of a mine waste.

Principles

- Determination of the neutralization potential (NP) of a sample;
- Calculation of the acid potential (AP) of the sample.

Procedure

- Sample preparation (minus 60 mesh);
- Fizz test: add two drops of 25% HCl to the 0.5g sample;
- Add HCl to the 2.00g of the sample in the flask;
- Heat pulp nearly to boiling and swirling the flask;
- Add distilled water to make a total volume of 125 mL and boil for 1 minute;
- Cool to room temperature;
- Titrate the contents of the flask using NaOH to pH 7.0;

Formula

- The neutralization potential (NP)

\[ NP = \frac{50a(x-(b/a)y)}{c} \]

where: NP = neutralization potential in kg CaCO₃ equivalent per tonne of material
  a = normality of HCl
  b = normality of NaOH
  c = sample weight in grams
  x = volume of HCl added in mL
  y = volume of NaOH added to pH 7.0 in mL

- The acid potential (AP), in kg CaCO₃ equivalent per tonne

\[ AP = \text{Percent sulphur} \times 31.25 \]

- The net neutralization potential (Net NP), in kg CaCO₃ equivalent per tonne of material

\[ \text{Net NP} = \text{NP} - \text{AP} \]
Modified Acid Base Accounting

Objectives

- To determine the balance between acid producing and acid consuming components of a mine waste.

Principles

- Determination of the neutralization potential (NP) of a sample;
- Calculation of the acid potential (AP) of the sample.

Procedures

- Sample preparation (80 percent minus 200 mesh);
- Fizz test-add two drops of 25 percent HCl to 0.5g sample;
- Add HCl to 2.00g sample in 250 mL flask;
- Agitate the contents of the flask for 24 hours;
- Check the pH of the pulp after 6 hours and 24 hours of reaction;
- Titrate the contents of the flask using NaOH to pH 8.3.

Formula

- The neutralization potential (NP)

\[
NP = 50a[x-(b/a)y]/c
\]

where: NP = neutralization potential in kg CaCO₃ equivalent per tonne of material
\(a\) = normality of HCl
\(b\) = normality of NaOH
\(c\) = sample weight in grams
\(x\) = volume of HCl added in mL
\(y\) = volume of NaOH added to pH 8.3 in mL

- The acid potential (AP), in kg CaCO₃ equivalent per tonne

\[
AP = \text{Percent sulphide sulphur} \times 31.25
\]

(Sulphide sulphur = Total sulphur - Sulphate sulphur)

- The net neutralization potential (Net NP), in kg CaCO₃ equivalent per tonne of material

\[
\text{Net NP} = NP - AP
\]
Shake Flask Test

Objectives

- To confirm the results of static prediction tests;
- To determine the rate of and temporal variation in acid generation and leachate water quality.

Principles

- The test can be performed in a number of different ways to determine the effects of time, temperature, pH and bacteria.

Procedures

- Sample preparation (80 percent minus 200 mesh);
- Weigh out 50g of sample for each flask;
- Add 150 mL distilled water to each flask;
- Add acid for pH adjustment or bacterial nutrients to the flasks;
- Place flasks on shaker (at different temperatures);
- Collect 50mL leachates for analysis at different times (one week, two weeks and three weeks);
- Analysis might include pH, sulphate, dissolved metals and other species.
Kinetic Test

Objectives

- To determine the rate of oxidation and metal leading in a simulated environment.

Procedures

- Sample preparation (1000g pellets for 109 and NEX tailings);
- Put the sample into the column and weigh them;
- Leach the sample using 250 mL distilled water;
- Collect the leacheate and check the pH and conductivity of leacheate;
- Weigh the leacheate and the sample;
- Dry the sample for 3 days and weigh the sample;
- Humidify the sample for 3 days and weigh the sample;
- Then carry out next cycle of the test.
APPENDIX B

Raw Data and Analysis Data
### Eskay Creek ARD Project
#### Sobek Results

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NP = 50a(x-(b/a)y)/c

Where:
- NP = Neutralization potential kg CaCO₃ equivalent per tonne
- a = Normality of HCL added in digestion
- b = normality of NaOH used in titration
- c = mass of sample in grams
- x = volume of HCL added in mL
- y = volume of NaOH added in titration
**Eskay Creek ARD Project**  
**Modified ABA**

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\[ \text{NP} = 50a\{(x-(b/a)y)/c \} \]

Where:  
- \( \text{NP} \): Neutralization potential kg CaCO₃ equivalent per tonne  
- \( a \): Normality of HCL added in digestion  
- \( b \): normality of NaOH used in titration  
- \( c \): mass of sample in grams  
- \( x \): volume of HCL added in mL  
- \( y \): volume of NaOH added in titration
### UNIVERSITY OF BRITISH COLUMBIA
DEPARTMENT OF MINING & MINERAL PROCESS ENGINEERING
DATE: JUNE 30/98

TO: JOHN MEECH

**CERTIFICATE OF ANALYSIS**

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**ESKAY CREEK**

**BALL MILL FEED**

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**REF:** ESKAY.DOC

LARRY WONG
(PROVINCIAL ASSAYER)