# Kiggavik Project

# Environmental Impact Statement

Technical Appendix 5F

# Mine Rock Characterization, and Management

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# 1 INTRODUCTION

## 1.1 OVERVIEW

The Kiggavik Project is a proposed uranium ore mining and milling operation located in the Kivalliq region of Nunavut approximately 80 km west of the community of Baker Lake.

This document is a Technical Appendix to Volume 5, Aquatic Environment, of the Kiggavik Environment Impact Statement (EIS). Information relevant to mine rock characterization, assessment and management is presented in this Technical Appendix.

The Kiggavik Project includes four open pit mines and one underground mine. The Main Zone, Centre Zone and East Zone Pits are near each other and the Andrew Lake Pit and End Grid Underground Mine are located about 15 km to the southwest. With respect to mine rock management, which is the focus of this Technical Appendix, the main facilities will include permanent and temporary stockpiles at the Kiggavik and Sissons sites.

This document provides a detailed discussion of the proposed design, operation and decommissioning of the mine rock management system at Kiggavik and Sissons. Additionally, predicted operational and long-term effects, mitigative measures, and contingency measures related to mine rock management are presented. Detailed information relevant to tailings is presented in the Technical Appendix 5J “Tailings Characterization and Management”.

## 1.2 PURPOSE AND SCOPE

This purpose of this report is to document mine rock data and to describe the proposed mine rock management plan for the Kiggavik Project and the measures proposed to prevent and /or mitigate the potentially adverse effects of mine rock on the receiving environment.

The scope of work for description of the mine rock management plan included field and laboratory studies, geochemical interpretation synthesis to determine a conceptual design of the mine rock management facility.

The scope of this document is to:

- Characterize the physical and geochemical properties of the materials, as they relate to the long term management of mine rock;

- Assess the various options for long term management of mine rock;
- Choose the preferred option to propose in the mine plan submission;
- Describe the measures taken for design and construction of the preferred option; and
- Describe the measures to monitor and provide assurance that the management option performs as required to safeguard human and environment health and safety during operations and long term closure of the mine rock management facility.

This report is organized as follows:

- Section 1 - Introduction
- Section 2 – Review of Mine Rock Disposal Practices
- Section 3 – Geological Context
- Section 4 – Mine Rock Assessment Methodology
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- Section 10 – Andrew Lake Pit Water Quality
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- Section 12 – Conclusions

This report was prepared by AREVA and Ecometrix Inc.

# 2 REVIEW OF MINE ROCK DISPOSAL PRACTICES

Other than providing capacity and maintaining physical stability, the primary goal of mine rock management is to limit or prevent effects on quality of water in the downstream environment. Planning for mine rock management should consider the long-term physical stability of these facilities and the risks associated with long-term metal leaching and acidic drainage. In order for this to be completed effectively, the mine rock has to be well characterized and assessed with respect to the planned storage method. If the mine rock is potentially acid generating or is metal leaching, mitigation strategies will be required.

The Canadian Nuclear Safety Commission (CNSC) has issued a Draft document RD/GD-370 entitled “Management of Uranium Mine Waste Rock and Mill Tailings” (CNSC, 2011) that provides guidance for management practices for mine rock at new uranium mining projects throughout Canada. The CNSC document states that the use of mine workings such as open pits and underground developments should be maximized; however the use of natural fish-bearing water bodies shall be avoided. The guidance suggests the use of best management practices with due consideration of the characteristics of mine rock and tailings with controls that are designed to minimize release to the environment to ensure long-term protection of the environment and the safety of current and future generations. Wherever possible, clean materials, including overburden and rock should be used for construction or as a resource through effective segregation practices.

Mine rock management in a permafrost environment is not significantly different from that in non-permafrost areas. In some instances, engineering designs are established to take advantage of the sub-zero temperatures in permafrost terrain. For example, mine rock placement can be designed to accelerate and maintain frozen conditions to limit weathering or acid generation in reactive wastes. However, global warming suggests that permafrost conditions may be diminishing in Canada’s north over the next 100 years. Therefore, as a cautionary approach, it is prudent to develop designs that do not rely on permafrost conditions for successful mine rock or tailings management. Mine rock management in this project do not include designs that rely on permafrost for protection of the environment. The designs considered in this project are compatible with a permafrost environment and may result in enhanced performance from sub-zero conditions but do not include permafrost requirements for success.

Best practice for mine materials management is based on characterization of the rock to understand potential risks for acid generation and metal leaching. Materials can be classified according to master variables such as sulphur content, metal content and uranium grade to ensure that specific classes of material can be either stored safely with no controls or stored with appropriate natural or engineered barriers to prevent unacceptable releases of constituents to the environment.

In this project, mine materials were characterized and then, on the basis of characteristics and predicted behaviour in the environment, were classified into three categories. Clean rock material was classified into two categories; Type 1 for construction and unrestricted use as a resource and Type 2 that contained uranium levels above clearance criterion of 1 Bq/g (about 40mg-U/kg), was classified for long term storage in stockpiles on surface within the mine lease boundary. Material that was not ore and did not meet the "clean" criteria was classified as Type 3 and will be backfilled into pits and submerged in water for safe, long term disposal.

Management of mine waste at other uranium mining facilities in Northern Saskatchewan has included segregation of mine rock during operation into either "Clean Waste" being that with uranium contents below a critical value, or "Special Waste" being rock with elevated uranium and/or sulphide contents and requiring special management. Typically, the Special waste is placed back into the mined-out pit and flooded to limit oxidation and release of metals, acidity and other oxidation products. The process of segregating the different classes of materials is discussed in Section 6 of this report

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# 3 GEOLOGICAL CONTEXT

## 3.1 INTRODUCTION

The following sections summarize the geology of the Kiggavik Project deposits, as it is relevant to the development of a mine rock management plan. Additional information can be found in the Technical Appendix 5B (Geology and Hydrogeology Baseline).

From a regional perspective the Kiggavik Project is located in the Rae Province (Hoffman, 1990), at the southwest termination of the Woodburn Group. The Archean Woodburn Group is a typical “greenstone belt” or supracrustal sequence consisting of a lower (mafic/ultramafic) metavolcanic assemblage and an upper metasedimentary (metagreywacke and quartzite) assemblage (Ashton, 1982). The supracrustal rocks are in tectonic contact with, and structurally overlie, Archean basement gneisses which are predominantly granitic with a minor amphibolitic component.

## 3.2 LOCAL GEOLOGY

**Kiggavik area**

Basement host rocks are composed of metasediments, and to a lesser extent altered granite and intrusive rocks. Metasediments are sedimentary rocks that have been metamorphosed (altered by heat, pressure or chemically active fluids). Intrusive rocks are igneous rocks that intrude into pre-existing rocks, along some structure. Uranium mineralization in the Kiggavik area is hosted for the most part in altered metasedimentary rocks (mainly meta-arkose, metapelites and sericite schist), and to a much lesser extent in altered granite and intrusive rocks. There is no mineralization hosted in the Mackenzie diabase which cuts through Kiggavik.

**Andrew Lake area**

The Andrew Lake deposit is located in metasediments overlying granitic gneiss (banded metamorphic rocks) and granodiorite (an igneous rock). These formations have been strongly metamorphosed and altered, tectonized, and intruded. The rocks have gently dipping foliation, small scale recumbent folding, and low angle thrusting. The Andrew Lake deposit is located on a major east-northeast structure. This region has seen several episodes of hydraulic brecciation, mainly within the granite and syenite rocks, and to a lesser extent in the metasediment units. The subvertical faulting associated with the Andrew Lake deposit governs the extension of the mineralization.

Three main mineralized lenses have been identified at the Andrew Lake deposit. These are associated with strongly altered metasediments, altered paragneiss (gneiss metamorphosed from a sedimentary parent), and less altered metasediments. The zones overlie each other, and are separated by a quartz breccia (a poorly sorted rock commonly containing rock fragments), and paragneiss. Mineralization within the Andrew Lake area occurs between 70 and 300 m below ground surface.

Abundant interspersions of syenite and lamprophyre intrusive units are common throughout the Andrew Lake deposit area. The entire deposit area is overlain by approximately 10 to 20 m of unconsolidated overburden material.

### General

The End Grid deposit is located in an east-northeast sequence of metasediments, which are intruded by granite, porphyries, syenites and lamprophyres. It is related to the same major structure as the Andrew Lake deposit, but with a northeasterly trend. The mineralization within the deposit is controlled by horst and graben structures created by subvertical faulting.

Two main mineralized pods (North Pod and South Pod) have been identified at the End Grid deposit. These are located in strongly altered metasediment that are tectonized, with steeply dipping tension faults and hydraulic breccias. Mineralization at the End Grid deposit occurs between 75m and 475 m below ground surface.

The 2009 drilling program on the North Pod confirmed that a thick oxidised horizon is encountered from surface to 80-300m below the surface. It is traditionally interpreted as a paleoweathering profile emplaced before the sub-Thelon formations deposit. The mineralization envelopes are limited upwards by this oxidised horizon, which is interpreted to postdate the hydraulic silica breccias and granitic veins.

The primary low grade mineralization is disseminated within the foliation and in association with microfractures. Several examples show an association of granitic to quartzic small intrusions with the mineralization. The global trend of the primary mineralization is ENE/WSW. This trend is parallel to a former quartz breccias trend. No direct relation between breccias and mineralization has been observed. The secondary medium to high grade mineralization is associated with a late tectonic event. NNW/SSE faults are formed and the ENE/WSW silica breccias trend is reactivated by brittle faults cutting across the primary mineralization and oxidising fluids have remobilized it. Fractures with pitchblende are encountered around the main deformation fault zone and in association with Red-Ox fronts surrounding the late faults. Red-Ox fronts and associated mineralization are also common along foliation and former ductile shear zones parallel to foliation.

## 3.3 PETROGRAPHY AND MINERALOGY

From Farkas (1984), Fuchs et al. (1986), and unpublished AREVA petrographic determinations, the mineralogical compositions of the various rock types are typically:

1. meta-arkose (fine- to medium-grained, psammopelitic to psammitic gneiss; impure/“dirty” quartzite)

| *Mineral* | *Original* | | *altered* |
|---|---|---|---|
| Quartz | 30-50% | | 0-30% |
| Feldspar (plagioclase + K-feldspar) | 30-50% | | nil |
| Biotite/chlorite | 15-30% | chlorite | 5-40% |
| Muscovite/sericite | 5-40% | illite/sericite | 30-90% |
| Garnet | <5% | | nil |
| Pyrite | <5% | | <5% |
| Carbonate | <5% | | <5% |
| Tourmaline | accessory | | accessory |
| Zircon | accessory | | accessory |
| Fluorite | accessory | | accessory |
| Hematite | accessory | | 0-15% |
| Epidote | accessory | | nil |

2. Lone Gull granite (medium-grained, pinkish, non-foliated, holocrystalline-granular)

| *Mineral* | *Original* | | *altered* |
|---|---|---|---|
| Quartz | 35-55% | | 5-40% |
| K-Feldspar | 25-45% | | nil |
| Plagioclase | 15-25% | | nil |
| Biotite/chlorite | 15-25% | chlorite | 5-40% |
| Muscovite/sericite | 5-20% | illite/sericite | 30-60% |
| Fluorite | accessory | | accessory |
| Pyrite & Molybdenite | accessory | | <5% |
| Carbonate | accessory | | 0-40% |
| Zircon & Titanite | accessory | | accessory |
| Epidote | accessory | | nil |
| Apatite | accessory | | accessory |

3. Chlorite-sericite schist

| | |
|---|---|
| Quartz | 10-30% |
| Chlorite/sericite | 70-90% |

4. Metaquartzite (“orthoquartzite”)

| | |
|---|---|
| Quartz | >98% |
| Sericite | trace |
| Andalusite | accessory |

In general the mineralization is finely disseminated along foliation planes and/or in veinlets parallel to the foliation, but can also be found as fracture infill and coating along cross-cutting structures. The two major uranium minerals are pitchblende and coffinite. Secondary uranium minerals are not common. Fine-grained uranophane occurs in weathered rocks at surface but also occasionally at greater depth. Pitchblende and coffinite are often associated with marcasite and pyrite. Other sulphides or accessory metals are present only in minor amounts, indicating the single elemental composition characteristic of the Kiggavik ore zones. The uranium mineralization is associated with an intensive alteration halo. The alteration is characterized by desilification and by the conversion of feldspar and mica to clay minerals consisting mainly of illite and sericite which is somewhat typical of unconformity type deposits.

The Kiggavik mineralization is monometallic (U ± Au, Pt) and occurs in both metasediments and granite. The two major uranium minerals are pitchblende [$UO_2$] and coffinite [$USiO_4(OH)_4$]. This mineralization is present as finely-disseminated granules located along foliation planes and/or as colloform aggregates and rosettes in veinlets parallel to the foliation. It is also found as fracture fillings and coatings along cross-cutting structures (Fuchs et al., 1986) and rarely in brecciated metasediments (Farkas, 1984). Fracture-controlled mineralization is more frequently observed in granite than metasediment. The uranium minerals are often associated with brownish Fe-oxide/hydroxides (hematite/limonite). Coffinite surrounds and replaces pitchblende. Gangue minerals are illite, with minor dusty hematite, and variable amounts of chlorite.

Secondary uranium minerals are uncommon, although fine-grained uranophane occurs in weathered rocks at surface and also occasionally at greater depth. Pitchblende and coffinite are often associated with and replace marcasite and pyrite. Other sulphides or accessory metals are present only in minor amounts, resulting in the monometallic composition characteristic of the Kiggavik ore zones.

# 4 MINE ROCK ASSESSMENT METHODOLOGY

An investigation was initiated by AREVA and EcoMetrix to characterize the mine rock materials from the Kiggavik and Andrew Lake deposits, to assess the leaching behaviour and to compare the results with those from other uranium deposits in Northern Saskatchewan. The clean rock at Kiggavik and Andrew Lake was also investigated to ensure that water quality from potential construction material and on-land stockpiles will not be at risk.

The waste rock characterization study was conducted mainly in the laboratory and involved chemical analyses of the rock, including metal analysis, acid base accounting (ABA), humidity cell testing, customized leach tests to assess metal leaching, and column tests with submerged rock to evaluate the concentrations of metals in pore water for an in-pit disposal scenario. Four field cells were also constructed at the Kiggavik field camp to evaluate the leaching behaviour of the mine rock material under ambient site conditions with natural precipitation rates.

The mine rock assessment was completed in stages starting in 2006. Review of the waste rock assessment in the 1989 Environmental Assessment (Urangesellschaft, 1989) provided some basic information on “general” waste rock and ore conditions for a few samples. In addition to a review of existing reports, a site visit in July 2006 was completed, and samples of drill core segments from the selected core fragments from historic exploration activities in the 1980s and early 1990s were collected for preliminary screening analysis. The tests on those core samples were conducted by EcoMetrix.

Separate follow-up laboratory testing studies were completed by EcoMetrix and Saskatchewan Research Council (SRC). Testing was completed for both studies on drill core samples collected from AREVA’s 2007 drilling campaign from the Andrew Lake (ANDW), End Grid (END), and Main Zone (MZ) deposits. As additional exploration drilling was completed in 2008 and 2009, further characterization of the projected mine rock was completed. The methods and procedures used for the assessment are discussed below.

## 4.1 MINE ROCK SAMPLING

A total of 186 drill core samples from the 2007, 2008 and 2009 AREVA drilling campaigns were evaluated, including 51 samples from the Andrew Lake deposit, 42 samples from the End Grid deposit together with 55, 22 and 16 samples from the Main, Centre and East Zone deposits, respectively at the Kiggavik site (Table 4.1-1). Material used in the laboratory evaluation was provided by AREVA from 2007, 2008 and 2009 drilling campaigns and shipped to EcoMetrix. Additional rock core samples were collected by EcoMetrix from the core racks at the Kiggavik site in July 2009, and included samples from each of the 2007, 2008 and 2009 drilling campaigns.

The rock material was examined and assessed according to the drill core sample location, such that the Kiggavik rock encompassed all the Main, Centre, and East Zone samples, and the Andrew Lake rock was characterized by only Andrew Lake deposit samples. The samples collected from the End Grid deposit were characterized separately from the Main Zone and Andrew Lake waste rock. The End Grid rock will originate from an underground mine where much of the material will likely remain or be backfilled below ground, and therefore will not need to be managed on surface. Geochemical characteristics of the Kiggavik and Andrew Lake rock material are summarized and discussed separately to better understand the two mine site's geochemical characteristics. However, testing methods were the same for both the Kiggavik and Andrew Lake material.

## 4.2 TESTING METHODOLOGY

The mine rock characterization study involved chemical analyses of the mine rock, including bulk metal analysis and acid base accounting (ABA), humidity cell testing, customized leach tests, and column tests. The humidity cell tests were aimed at assessing the leaching properties of the mine rock materials for scenarios of use in construction and above ground stockpile disposal. The customized leach tests included shake flask testing and modified-Special Waste Extraction Procedure (SWEP) tests of the mine rock material to evaluate initial inventories of soluble constituents. The column tests were constructed to evaluate the geochemical properties and leaching behaviour of the material that requires special disposal by underwater submergence to simulate an in-pit disposal scenario.

In addition to the laboratory tests, four field cells were constructed at the Kiggavik field camp to evaluate the leaching behaviour of the mine rock material under ambient site conditions with natural precipitation rates. The four field cells had rock materials from either the Main Zone, Centre Zone, End Grid, or Andrew Lake deposits. The four cells were constructed such that any precipitation could enter the top of the cell and percolate down through the material, where the leachate was collected from a bottom drain and sampled for analysis.

### 4.2.1 Solids Characterization

Rock samples were submitted to either ALS Chemex in Vancouver, BC or SRC Analytical in Saskatoon, SK for analyses that included ABA and metals assays. The ABA analyses included determination of neutralization potential (NP), acid generation potential (AP), total sulphur ($S_T$), sulphate sulphur ($S_{SO4}$), sulphide sulphur ($S_{S2-}$) and inorganic carbon content. Metals content analyses included multi-element determination by four acid near-total digestion followed by ICP-MS quantification.

### 4.2.2 Leachability Testing

Several assessments and characterization studies of uranium deposits in the Athabasca region of Northern Saskatchewan have shown that some mine rock can be a source of acid and metals as a result of sulphide mineral oxidation if the rock is deposited on land. However, studies have also shown that the same rock can be a source of elevated arsenic concentrations when deposited under water, a mitigation measure intended to eliminate sulphide mineral oxidation and consequent acid generation. The effects and implications of arsenic leaching from mineralized mine rock associated with several deposits in the Athabasca basin have been evaluated and reported on in several studies (Cogema and CLMC, 2001; Cogema, 2004; Areva, 2007). Those studies identified a relationship between the soluble arsenic that can dissolve in pore water when the rock is flooded and the arsenic content of the solids. A relationship between the arsenic content of the rock and the "equilibrium" concentration of arsenic in the pore water was also demonstrated.

The initial rock samples from the Kiggavik and Andrew Lake deposits submitted to EcoMetrix in 2007 were therefore evaluated in a manner that was consistent with the leach test protocol for mine rock from the Athabasca Basin. Samples were crushed to less than 2.5 cm (1 inch) prior to testing. The leach tests generally used the standard Special Waste Extraction Procedure (SWEP), with a water:solids ratio of 20:1. The leach solutions were occasionally agitated, with sampling and leachate replacement taking place at 48 hour intervals, according to the following steps:

- Leach 1: Distilled Water (at t = 48 hrs)
- Leach 2: Distilled Water (at t = 96 hrs)
- Leach 3: Phosphoric Acid, pH 5 (at t = 144 hrs)
- Leach 4: Phosphoric Acid, pH 5 (at t = 192 hrs)

The concentration of phosphate in the solution of Leach steps 3 and 4 was approximately 200 mg/L. Sodium hydroxide was added to some samples during the phosphoric acid leaching step in order to maintain the solution pH between 5.0 and 5.5.

After each leaching step, leachate samples were filtered and preserved for analysis as appropriate, prior to laboratory submission. The pH and conductivity were measured immediately after sample collection. After completion of the leach tests, the remnant leached solids were analyzed.

The results for most constituents of potential concern in the leachate were below detection. Therefore, a second set of leach tests on selected core segments was conducted at a 3:1 water:solids ratio. The 3:1 water:solids ratio was used in order to limit dilution and provide better detection of COPCs in the leachate. The leach tests and sampling procedures were conducted using the same procedures as described above, with the exception that the remnant solids were not analyzed.

The results of the initial leaching tests suggested that the mine rock from the Kiggavik and Andrew Lake sites did not appear to have similar arsenic leaching issues as the other materials from the Athabasca region. This early finding prompted an adjustment in the leachability tests such that the remainder of the samples submitted to EcoMetrix for evaluation were subjected to a modified-SWEP test that consisted of one water leach only, following recommendations for mine rock evaluations (Price, 1997; 2009).

The findings from the initial SWEP tests provided, at an early stage, the overall investigation with information that was used to determine  the mine rock criteria for segregation, as well as a partial rational for subsequent geochemical tests.

## 4.3 KINETIC TESTS

Static testing provides an indication of the total and soluble constituent concentrations in the mine rock materials at a particular time, but does not provide an indication of potential changes with time during mining operations or after mine closure. Kinetic testing can be used to provide information on the relative rates of acid generation, neutralization, the approximate timing for the onset of acid generation, if it is predicted to occur, as well as potential drainage chemistry and downstream loadings from the mine rock material as a result of metal leaching in the presence or absence of acidic drainage.

### 4.3.1 Humidity Cell Testing

Kinetic testing was completed on composite rock samples for the Kiggavik and Andrew Lake deposits.  Composite samples, based on total sulphur and uranium content, were prepared from the drill core samples as shown in Table 4.3-1.  Each test cell contained approximately 1.0 kg of the material and was flushed weekly with 1.25 L of distilled water.  Samples of leachate were collected approximately two hours after initial inundation, and were then filtered and preserved accordingly prior to submission to the laboratory for analysis.  Measurements of pH and conductivity were completed on the leachate immediately after collection, but prior to filtering and preservation.

Leachate samples from the initial rinse, week 2 and week 20 from each of the humidity cells were also submitted for radionuclide analysis by Becquerel Laboratories, Inc. in Mississauga, ON, including thorium-230 (Th-230), radium-226 (Ra-226), lead-210 (Pb-210), and polonium-210 (Po-210). The Th-230, Ra-226, and Po-210 concentrations were quantified by alpha-particle spectrometry, and the Pb-210 concentration was quantified by gas-flow proportional counting.

### 4.3.2 Column Testing

Two column tests were also initiated to examine the geochemical characteristics of the potentially problematic Kiggavik and Andrew Lake rock material in an underwater storage scenario. Each column test was designed to assess the material's influence on pit water quality after the material is backfilled to the pit and submerged. The two columns were filled with approximately 6.8 kg and 5.5 kg of selected rock cores from the Kiggavik and Andrew Lake rock, respectively (Table 4.3-2).  Each column was then completed filled with distilled water. Water in each column was allowed to be circulated through the rock material prior to sampling to produce a sample that represents the pore water within the submerged material.  Leachate samples were collected on a weekly basis.   For the initial three weeks of testing, a 0.12L water sample was collected to verify that COPC concentrations would be quantifiable, after which a

1.0 L water sample was collected after it was determined that 1.0 L could be removed, and subsequently replenished with distilled water without diluting the sample to below laboratory detection limits. Samples were filtered (0.45 μm) and preserved (18% nitric acid) prior to submission to the laboratory for analysis. All water samples were submitted to ALS-Vancouver for analysis. The pH and conductivity of the samples were measured on the leachate immediately after collection.

One leachate sample from Day 17 of each of the two column tests was also submitted for radiological analysis, including Th-230, Ra-226, Pb-210, and Po-210. Two solids samples of the composite material that make up each of the columns were also submitted for radionuclide analysis, including Th-230, Ra-226, and Pb-210 by gamma-ray spectrometry. As part of the gamma-ray spectrometry analysis, Th-234, U-235, Th-227, Ra-223, Ra-228, and Th-228 were also quantified, where Ra-228 was estimated from actinium-228 (Ac-228) and Th-228 was estimated from Pb-212. Due to limitations at Becquerel Laboratories, and the high natural occurring activity of the samples, the samples could not be appropriately prepared for Po-210 analysis. However, the Po-210 concentration can be estimated based on equilibrium with Pb-210 after several months.

### 4.3.3 Field Cell Tests

Four field cells were constructed in July 2009 at the Kiggavik field camp. The cells were intended to represent larger scale humidity cells, which were exposed to the ambient environment and natural precipitation to assess *in situ* leaching rates. These cells were constructed using half of a clean 170 L barrel and were filled with whole drill core segments ranging from 10 to 50 cm in length (Figure 4.3-1). The top of each cell remained open so that rock samples were exposed to air and to precipitation falling as rain or snow. Each barrel has a bottom drain spout connected with tubing to pails in which the water collects between sampling events. Selected drill core segments were placed in each barrel to represent rock material from four of the five proposed deposits, namely Main Zone, Centre Zone, End Grid, and Andrew Lake (Attachment B). No field cell was constructed using East Zone deposit material due to the lack of availability of sufficient material. The total rock mass in each cell ranged from about 39 kg to 51 kg.

Sampling of the collected leachate from each field cells was been completed by AREVA personnel on three separate occasions; in August 2009 representing approximately 4 weeks after cell construction; in November 2009 just prior to closure of the field camp for the 2009 winter season and representing approximately 6 weeks after cell construction; and once more in July 2010. Water samples were submitted to SRC in Saskatoon, SK for laboratory analysis.

### 4.3.4 Loading Rate Calculations

Loading rates derived from humidity cell and column tests were transformed into field loading rates by taking into account conditional differences between the laboratory and field settings. In general, the specific surface area of rock particles, temperature and freezing conditions should be considered for modifying the loading rates observed in laboratory humidity cell tests. Therefore, the humidity cell test results were adjusted to account for temperature and grain-size (or surface area).

A temperature correction factor for the humidity cell test results is required because the testing was completed at room temperature (20 to 25°C) while temperatures under field conditions for the stockpile will be approximately equal to the average air temperature at the site. The average temperature at the Kiggavik and Andrew Lake sites was estimated to be 5°C for eight months of the year and 0°C for the remaining four months. Oxidation of sulphide minerals such as pyrite, are temperature dependent, as described by the Arrhenius equation:

$$\ln\left(\frac{k_1}{k_2}\right) = \frac{E_a\left(T_1 - T_2\right)}{RT_1T_2}$$

where $k_1$ and $k_2$ are reaction rates at temperatures $T_1$ and $T_2$ in Kelvins, $E_a$ is the activation energy of the reaction (J/mol), and $R$ is the universal gas constant (8.314 J/mol/K). The values of $E_a$ for pyrite oxidation reactions, that are likely responsible for releasing metals, have been shown to be about 80 kJ/mol (Nicholson *et al*., 1988). Therefore, a temperature difference from laboratory temperature (20°C) to field temperatures, where field temperatures were assumed to be 0°C for eight months and 5°C for four months would result in a difference factor of about 9 in the reaction rate. Estimated metal loads from the stockpile (field loads) therefore include a correction factor of 0.117 (or about 1/9) applied to the calculated laboratory loading rate.

The second correction factor applied to the loadings estimates accounts for the fine grain size of the rock samples used in the humidity cell tests. The fine-grained material should contribute essentially all of the leached constituents from the mine rock pile, based on a unit mass basis because of the large surface area in the fines. A qualitative grain size analysis was estimated on all 11 of the humidity cells. The findings from the seven Kiggavik humidity cells and the four Andrew Lake humidity cells were then averaged separately to get an estimated surface of the mine rock material from the two sites. The estimated surface area of the mine rock material from the two sites was used to establish a representative factor designed to account for the difference in material size from lab testing and actual field conditions. It was observed that the Kiggavik rock material is more competent and less friable than the Andrew Lake rock material. This observation has been noted in various laboratory investigations including the humidity cell tests, and correlates with geotechnical results for the site as discussed below. Therefore, based on these observations and to maintain a conservative approach to loading estimates, it was assumed that 5% of the material that makes up the Kiggavik rock stockpile will be of similar size to the material tested in the humidity cells (less than 1" to silt/clay size), whereas 20% of the material that makes up the Andrew Lake rock stockpile will be similar to the actual laboratory size distribution. Therefore, a factor of 0.05 was applied to the Kiggavik laboratory loading rates and a factor of 0.20 was applied to the Andrew Lake laboratory loading rates based on the assumption that only 5% and 20% of the Kiggavik and Andrew Lake stockpiles, respectively, are comprised of similar sized material as in the humidity cells.

# 5 MINE ROCK CHARACTERIZATION

Initial testing of the Kiggavik and Andrew Lake mine rock was completed based on characterization studies of uranium deposits in the Athabasca region of Northern Saskatchewan. Material was either classified as “Type 1 or 2” or “Type 3”. Results of preliminary testing of the Kiggavik and Andrew Lake material were used to better categorize the mine rock material.

A summary of the ABA analyses and metals contents for each deposit is provided in Table 5.1-1. The distributions of metals with depth are also provided in graphical format in Figure 5.1-1 for selected metals. Evaluation of the ABA analyses is provided in Figure 5.1-2 as NP vs AP and NP/AP vs depth plots.

The results of the SWEP tests for selected metals are presented in Table 5.1-2 as values of the equivalent mass of metal leached per mass of solids in units of mg/kg and percentage of the total inventory leached.

Complete results for both solids and SWEP analyses are provided in Attachment C.

## 5.1 SOLID CONTENT AND ACID GENERATION POTENTIAL

### 5.1.1 Kiggavik Mine Rock Material

**Main Zone Deposit**

Concentrations of most constituents of concern were below the screening criteria of ten times the average crustal abundance in the Main Zone Deposit samples. Arsenic concentrations were generally consistent, with a geometric mean value of about 0.7 mg/kg and only 14 of the 55 samples had arsenic concentrations above 1 mg/kg. Uranium concentrations ranged between 2.8 and 818 mg/kg, with a mean value of about 23 mg/kg. No discernable trends in uranium or arsenic contents were noted with depth. Copper concentrations ranged between 2 and 245 mg/kg. Mean concentrations of cobalt, lead, molybdenum and zinc were about 9.3 mg/kg, 29.2 mg/kg, 13.8 mg/kg and 25.3 mg/kg, respectively exhibiting no trends with depth. Nickel concentrations generally ranged from 1.3 to 116 mg/kg, with a mean value of about 20.1 mg/kg while the concentrations of selenium ranged from 1 to 7 mg/kg, with a mean concentration of 1.3 mg/kg. Cadmium concentrations were generally below or near detection. The detection limit for cadmium for some of the 2008 drill hole samples was elevated, with results reported at less than 1 mg/kg.

Results of ABA testing on the Main Zone deposit samples suggest that the majority of the material appears to be non-acid generating; however acid generation could be a risk in some materials. The mean NP and AP values for Main Zone samples were about 13.6 and 3.9 kg-CaCO3/t, respectively, with paste pH values between 7.2 and 9.3. The NP/AP ratios ranged from 0.2 to 240, with a geomean of 2.8. Of the 55 samples analyzed from the Main Zone deposit, 40 samples had NP/AP ratios greater than 1, with 20 samples having NP/AP ratios greater than 4 (Figure 5.1-2). Samples that exhibited NP/AP ratios of less than one generally also exhibited sulphur contents greater than 0.5%. The Main Zone samples, exhibited some of the highest sulphide contents of all samples, with 34 of 55 samples exceeding 0.1% sulphide and 4 samples exceeding 1% sulphide.

**Centre Zone Deposit**

The mean arsenic and uranium contents were 1.6 and 10.8 mg/kg, respectively and ranged from 0.6 to 12.5 mg/kg for arsenic and from 2.5 to 500 for uranium. For both arsenic and uranium, the higher concentrations were typically observed at shallower depths. The concentration of copper ranged from 1.7 to 182 mg/kg, with a geometric mean concentration of about 17 mg/kg. Geometric mean cadmium, cobalt, lead, molybdenum, nickel and zinc concentrations were about 0.03 mg/kg, 11 mg/kg, 18 mg/kg, 5.6 mg/kg, 33 mg/kg and 35 mg/kg, respectively.

The ABA results for the Centre Zone samples were generally had similar range in values as the Midwest Zone samples. Core samples from the Centre Zone had laboratory paste pH values ranging from 7.4 to 9 with a mean value of 8.4. Neutralization potential values averaged 12.2 kg-$CaCO_3$/t. Total sulphur contents ranged from 0.01% to 1.83%, with a geometric mean value of 0.12%, and were predominantly in the sulphide form. Half of the samples (11 of 22) had total and sulphide sulphur contents greater than 0.1%, with 8 samples (36%) having sulphide contents greater than 0.5%. The AP contents of the samples ranged from 0.3 to 57 kg-$CaCO_3$/t, with a geomean value of 3.7. The NP/AP ratio averaged 3.3, with 11 of 22 samples having ratios less than 3 resulting from the elevated sulphur contents (Figure 5.1-2).

**East Zone Deposit**

The uranium content of the East Zone mine rock samples were lower than those from the other deposits, ranging from 1.9 to 9.2 mg/kg with a mean content of 4.2 mg/kg (Figure 5.1-1). Arsenic concentrations ranged from 0.2 to 5.3 mg/kg with an average of 0.6 mg/kg. Cobalt concentrations generally ranged from 8.2 to 31 mg/kg, with a mean value of about 14 mg/kg while the concentration of copper ranged from 4.4 to 148 mg/kg, with a mean concentration of 30 mg/kg. The concentration of cadmium ranged from less than 0.02 to 0.09 mg/kg, with the exception of one core sub-sample taken from the depth interval 63 to 63.1 m that measured 2.54 mg/kg. This sample also exhibited elevated arsenic, lead and zinc contents in comparison to the other East Zone samples. Geometric mean concentrations of lead and molybdenum were about 10 mg/kg and 0.8 mg/kg, respectively, with concentrations of nickel ranging from 16.3 to 53 and concentrations of zinc ranging from 33 to 567 mg/kg.

Laboratory measured paste pH values were above 7.5 in all East Zone samples. The NP values ranged from about 6 to 19 kg $CaCO_3$/t with AP values between 0.3 to 18 mg/kg $CaCO_3$/t, resulting in NP/AP ratios between 0.8 and 43. The mean NP/AP ratio was 7.7, with 6 of the 16 samples (38%) having ratios less than 4. Sulphide contents ranged from 0.01 to 0.56%, with a geomean of 0.04% (Figure 5.1-2).

### 5.1.2 Sissons Mine Rock Material

#### Andrew Lake Deposit

The geomean concentrations of constituents of potential of concern in the Andrew Lake samples were below the screening criteria of ten times the average crustal abundance values, with the exception of arsenic, uranium and antimony. The range in arsenic concentration in the Andrew Lake core sample segments was between 0.3 and 56.9 mg/kg with a mean concentration of 3 mg/kg, with the elevated concentrations generally found at shallower depths. In general, the concentrations of uranium ranged between 0.9 and 452 mg/kg with a mean concentration of about 17 mg/kg. Mean concentrations of cobalt, copper, nickel and zinc were 3.6 mg/kg, 6.6 mg/kg, 28 mg/kg, and 12.9 mg/kg, respectively. The mean lead and molybdenum concentrations in the core segments were 9.3 and 0.7 mg/kg with a mean concentration of 0.3 for antimony.

The results of the ABA analysis of Andrew Lake samples indicate that most of the rock material is not likely acid generating, as 40 of the 51 samples (78%) had NP/AP ratios above 4, with only 2 samples (4%) having NP/AP ratios less than 1. The mean NP/AP ratio was 9.9. The neutralization potential (NP) values ranged from 1 to 30 kg-$CaCO_3$/t with a mean paste pH of the samples of 7.5. Sulphide contents were low and ranged from 0.01 to 0.2%, with a geomean of 0.01%. However, only 8 of the 51 samples had a sulphide content greater than 0.02%. Acid generation potentials (AP) values ranged between 0.3 and 6.3 kg-$CaCO_3$/t.

#### End Grid Deposit

The metal contents in the End Grid samples were similar to those in the Andrew Lake materials (Table 5.1-1). The geometric mean concentration of arsenic in the End Grid core segments was 2 mg/kg. Uranium concentrations ranged from 1 to 30 mg/kg, with the exception of one core sub-sample taken from the depth interval 341.5 to 341.76 m that measured 1,370 mg/kg. The concentration of cobalt ranged from 1.3 to 113 mg/kg, with an geometric mean concentration of 6.5 mg/kg. Geometric mean copper, lead and molybdenum concentrations were about 7.8 mg/kg, 5.9 mg/kg and 0.8 mg/kg, respectively. Nickel and zinc concentrations measured in the solids ranged between 4.4 and 209 mg/kg and 2 and 120 mg/kg, respectively, with geometric mean values of 25.3 mg/kg and 19.4 mg/kg, respectively.

The ABA results for the End Grid samples were on average similar to those in the Andrew Lake materials. Laboratory paste pH values for the End Grid samples ranged from 6.9 to 8.9, with a geometric mean NP value of about 8.0 kg-$CaCO_3$/t. Samples generally had low sulphur contents, with 36 of the 42 samples (86%) having total and sulphide contents below 0.05%. The resulting AP values ranged from 0.3 to 35 kg-$CaCO_3$/t, with a geomean value of 0.6 kg-$CaCO_3$/t. Neutralization potential values ranged from 1 to 82 kg-$CaCO_3$/t, with a geomean of 8 kg-$CaCO_3$/t. The NP/AP ratios ranged from 0.6 to over 200 with a geomean of 13. Over 80% of the samples (35 out of 42) had NP/AP ratios above 4.

## 5.2 METAL LEACHING (SWEP)

### 5.2.1 Preliminary Leachability Screening

Leach tests completed on the initial batch of rock samples submitted to EcoMetrix in 2007 were focused on metal leaching with an emphasis on arsenic release, similar to that observed for the uranium deposits in the Athabasca region.

The results of the preliminary leach tests for arsenic are presented in Table 5.2-1. Results for additional COPCs are provided in Attachment D. The values represent the equivalent mass of arsenic leached per mass of solids and are presented in units of mg/kg. The values for the total leached concentration of arsenic from the samples were determined as the sum of the leached arsenic concentration measured after each leaching step. The quantities of arsenic leached were also compared to their total concentrations in solids and the results are presented as percentages in Table 5.2-1. Where concentrations were below detection in the leach extract, calculations assumed that values that were reported as less than detection were equal to zero. Complete results are presented in Attachment E.

With the exception of a few leachate samples, arsenic concentrations were below detection in tests for both the 20:1 and 3:1 evaluations. The mean water and total leachable masses of arsenic were 0.02 and 0.04 mg/kg, respectively, representing a maximum value of about 2% of the total arsenic inventory of the samples.

The results of the initial leaching tests suggested that the mine rock from the Kiggavik site did not appear to have similar arsenic leaching issues as the materials from the Athabasca region. Therefore, the remainder of the samples submitted to EcoMetrix for evaluation were subjected to SWEP testing that consisted of one water leach only.

### 5.2.2 Kiggavik Mine Rock Material

#### Main Zone Deposit

Table 5.1-2 provides a summary of the SWEP testing for selected parameters. The pH of leachates from the SWEP testing on Main Zone samples ranged from 3.4 to 7.8 with a mean value of 6.0. Leachable concentrations of antimony, cobalt, nickel and lead were typically below detection. Arsenic and uranium leachability ranged from 0.003 to 0.05 mg/kg and 0.014 and 15 mg/kg, respectively with mean values of 0.007 mg/kg for arsenic and 0.043 mg/kg for uranium. The leachable concentrations represent between 0.1 and 17% of the total arsenic inventory and between 0.05 and 15% of the total uranium inventory. Mean leachable copper, molybdenum and zinc concentrations were 0.006 mg/kg, 0.07 mg/kg and 0.013 mg/kg, respectively, representing less than 0.1% of the total copper and zinc inventories and about 0.3% of the total molybdenum inventory. Cadmium was typically below detection in leachate samples with the exception of the samples from the 2008 program. Leachate concentrations ranged from less than 0.0003 to 0.06 mg/kg.

#### Centre Zone Deposit

The results of the SWEP analyses were similar to the Main Zone samples, with leachable concentrations of antimony, cadmium, cobalt, nickel, lead and zinc typically below detection. Leachate pH values were generally neutral with only 1 of the 22 samples exhibiting a leachate pH less than 5. Concentrations of uranium were detected in only 4 of the 22 samples, ranging from 0.05 to 0.44 mg/kg, representing 0.1 to 0.9% of the total uranium inventories. Arsenic leachability ranged from below detection to 0.06 mg/kg, or between less than 0.1% and 2.9% of the arsenic inventories. Mean leachable copper and molybdenum concentrations were 0.006 mg/kg and 0.014 mg/kg, respectively, representing about 0.04% and 0.3% of the total copper and molybdenum inventories, respectively.

#### East Zone Deposit

Leachate pH values from SWEP tests on East Zone material averaged 5.9. The mass of arsenic leached for the samples ranged from about 0.004 mg/kg to 0.04 mg/kg, with a mean leachable arsenic content of about 0.004 mg/kg. These masses represent a mean leachable fraction of less than 1% of the total arsenic inventory. The mean leachable mass for molybdenum was about 0.004 mg/kg representing about 0.5% of the total inventory. Leachable masses for other metals of concern, including antimony, cobalt, copper, lead, nickel, uranium and zinc were generally below detection.

### 5.2.3 Sissons Mine Rock Material

#### Andrew Lake Deposit

The pH of leachates from the SWEP testing on Andrew Lake core samples ranged from 3.6 to 7.3 with a mean value of 5.8. Arsenic and uranium leachability ranged from below detection to 0.014 mg/kg and below detection to 3.8 mg/kg, respectively with mean values of 0.005 mg/kg for arsenic and 0.03 mg/kg for uranium. The leachable concentrations represent less than 0.01% to 1.4% of the total arsenic inventory and less than 0.01% to 2.2% of the total uranium inventory. Mean leachable copper, molybdenum, nickel and zinc concentrations were 0.006 mg/kg, 0.009 mg/kg, 0.007 mg/kg and 0.015 mg/kg, respectively, representing less than 0.1% of the total copper and nickel inventories, 1.2% of the total molybdenum and about 0.13% of the total zinc inventories. Leachable concentrations of antimony, cadmium, cobalt, and lead were typically below detection.

**End Grid Deposit**

Leachate pH values from SWEP testing of End Grid rock cores exhibited some of the lowest values analysed for the Kiggavik project, ranging from 2.6 to 8.1 with a mean value of 5.5. The mass of arsenic leached for the samples ranged from below detection to 0.18 mg/kg, with a mean leachable arsenic content of about 0.01 mg/kg representing a geometric mean percentage of 0.5% of the total arsenic inventory. The mean leachable mass for copper was about 0.006 mg/kg representing about 0.08% of the total inventory. The mean leachable mass for molybdenum was about 0.01 mg/kg representing about 1.3% of the total inventory. Zinc concentrations averaged 0.014 mg/kg or 0.07% of the total zinc inventory. Leachable masses for other constituents of potential concern were generally below detection.

## 5.3 GEOTECHNICAL PROPERTIES

Geotechnical studies were conducted in order to develop geotechnical criteria for the purpose of mine design. A summary of the geotechnical properties for the Kiggavik and Andrew Lake zones, where the majority of the mine rock will originate from, is presented in Table 5.3-1 and Table 5.3-2.

**Table 5.3-1 Geotechnical Parameters for Kiggavik Main Zone & Centre Zones**

| Rock Type | Alteration (SRM) | UCS (MPa) | Youngs Modulus (GPa) | Density ($g/cm^3$) | Poissons Ratio | RMR (1976) |
|---|---|---|---|---|---|---|
| Metasediments | Highly altered (W5) | 21.8 | 6.6 | 2.37 | 0.01 | 46 to 66 (fair to good) |
| | Fresh (W1) | 98.4 +/-26.1 | 44.6+/-6.3 | 2.69+/-0.10 | 0.17+/-0.02 | 62 to 71 (good) |
| Granite | Slightly to Moderately altered (W2 to W3) | 55.9+/-27.0 | 23.9+/-18.2 | 2.42+/-0.22 | 0.10+/-0.03 | 62 to 76 (good) |
| | Fresh (W1) | 112.3+/-28.5 | 45.1+/-3.4 | 2.64+/-0.05 | 0.15+/0.01 | |

*MPa=mega pascal, GPa=giga pascal, $g/cm^3$=grams per cubic centimetre*

### Table 5.3-2 Geotechnical Parameters for Andrew Lake Zone

| Rock Type | Alteration | UCS (MPa) | Youngs Modulus(GPa) | Density (g/cm$^3$) | Poissons Ratio | RMR (1976) |
|---|---|---|---|---|---|---|
| Metasediments <200 m depth | Low to Mod Alt (Af=1 to2) | 29.9+/-13.2 | 11.3+/6.5 | 2.46+/-0.17 | 0.08+/-0.04 | 42 to 62 (fair) |
| | Mod. To High Alt (Af=3 to 6) | 15.9+/-1.7 | 2.8+/-1.5 | 2.39+/0.17 | 0.09+/0.05 | 52 to 74 (fair to good) |
| Granite | Low to Mod Alt (Af=1 to 2) | 66.5+/-20.4 | 22.4+/-2.5 | 2.53+/-0.10 | 0.15+/-0.02 | 52 to 65 (fair to good) |
| | Mod to High Alt (Af=3) | 24.7 | 6.1 | 2.45 | 0.24 | |
| Inferred Fault | Low to High Alt (Af=1 to 6) | 7.4+/-3.5 | 1.4+/-1.4 | 2.03+/-0.08 | 0.05+/-0.01 | 42 to 52 (poor to fair) |
| Quartz Breccia | Low Alt (Af=1) | 35.0 | 19.9 | 2.63 | 0.16 | |

MPa=mega pascal, GPa=giga pascal, g/cm$^3$=grams per cubic centimetre, Af=alteration factor

The ore is found predominantly in the moderate to highly altered rock while most of the mine rock is expected to be comprised of weakly to moderately altered material.

The average joint spacings assumed for the Kiggavik Zones range from 0.7 m to 4.3 m. The average joint spacings estimated for Andrew Lake range from 0.5 m to 0.9 m.

### Fragmentation

A fragmentation analysis was conducted to predict rock fragmentation in bench blasting (see Technical Appendix 2B “Drilling and Blasting Design and Related Regulatory Considerations”). The analysis suggests that proposed blast designs will produce greater than 95% passing for an upper limit of 1,000 mm diameter. Fragmentation is expected to be finer at Andrew Lake due to the decreased joint spacing and reduced rock strength.

# 6 MINE ROCK SEGREGATION

## 6.1 CRITERIA

The criteria used traditionally in the uranium mining industry to segregate clean waste from special waste and ore have been based on the uranium content. The definition of “Clean” mine rock for uranium deposits in Northern Saskatchewan has been primarily based on uranium concentrations less than 250 mg/kg (ppm), with arsenic and nickel serving as additional criteria for specific ore deposits that have elevated levels of those COPCs. Classification of Clean mine rock for the Kiggavik project was initially projected to be consistent with the Northern Saskatchewan deposits. However, results from the leachability (SWEP) testing were used to provide an assessment of additional criteria that would define mine rock that can be surface stockpiled, materials that can be used for construction and material that will require appropriate long term management.

The solids contents of most metals were generally very low. The leachable masses of most metals were also low. The low values and the lack of correlation between solid and leachable contents for metals such as arsenic, copper, molybdenum and nickel indicates that these parameters are not relevant to the classification of mine rock at the Kiggavik site (Figure 6.1-1). In addition, no correlations were present between uranium and the other constituents such as arsenic, copper, molybdenum or nickel (Figure 6.1-2).

Sulphur content however, appears to be a key parameter for the determination of problematic mine rock, particularly for the Main, Centre and East Zone materials. Mean total sulphur contents of material for these deposits were 0.15, 0.12 and 0.05% respectively, with a maximum value as high as 1.83% (Table 5.1-1). The mean and maximum total sulphur content for the Andrew Lake material was 0.02 and 0.21%, respectively.

There are no apparent correlations between uranium and total sulphur contents for either the Main Zone or Andrew Lake stockpile materials. However, comparison of total sulphur content and the ratio of neutralization and acid generation potentials (NP/AP) provides a preliminary screening criteria to evaluate problematic mine rock.

Out of the 87 samples evaluated during the laboratory investigation from the Main, Centre and East Zones that had uranium contents less than 250 mg/kg, 64 samples had a NP/AP value of 1 or greater, with 39 samples having a NP/AP value of 4 or greater (Table 6.1-1). Plots of NP/AP vs total Sulphur (Figure 6.1-3) suggest that all material with Total S less than 0.1% (36 samples) would have an NP/AP ratio greater than 1, and 34 of the 36 samples would have a NP/AP ratio of 4 or higher. If a total sulphur content of 0.2% is considered, then 37 of the 46 samples would have a NP/AP ratio of 4 or higher. A criteria of 0.3% total S results in 39 out of 57 samples having a NP/AP value above 4.

For Andrew Lake material, the majority of the samples with uranium contents less than 250 mg/kg had a NP/AP value of 1 or greater, with 36 out of 47 samples having a NP/AP value of 4 or greater (Table 6.1-1; Figure 6.1-3). Almost all of the samples with total S contents less than 0.2% exhibit a NP/AP ratio above 1. Ratios above 4 are noted for 36 of the 45, 46 and 47 samples with total S contents less than 0.1, 0.2 and 0.3%, respectively.

Results of the evaluation of total sulphur content as a preliminary screening criteria to evaluate problematic mine rock were similar if all samples are considered (i.e. samples with uranium contents greater than 250 mg/kg) because there were only a small number of samples with uranium contents above 250 mg/kg.

Based on static testing results, the proposed segregation criteria for mine rock at the Kiggavik Uranium Project are as follows:

- Type 1: for clean rock that can be used as construction material (i.e., haul road, pad, etc.), it is proposed that concentrations of uranium be less than exemption quantities as per the Nuclear Safety and Control Act (1 Bq/g that equates to approximately 40 mg/kg based on the specific activity for $U_{nat}$) and have total sulphur content less than 0.1%;

- Type 2: for mine rock that can be permanently stockpiled and managed above ground, it is proposed that uranium contents be less than 250 mg/kg and total sulphur content be less than 0.1%;

- Type 3: for mine rock that requires specific management (e.g. in-pit disposal), it is proposed that all material not described above, or considered to be ore, be included in this category.

The classification of the samples that were tested can be seen graphically in Figure 6.1-4. These segregation criteria are consistent with waste rock management practices developed in Northern Saskatchewan.

## 6.2 SEGREGATION

### 6.2.1 General

During open pit mining, mine rock will be segregated according to uranium grade as determined by radiometric scanning. Operational procedures will be used to ensure that Type 2 and Type 3 mine rock are effectively categorized, separated and transported to the appropriate disposal area. Segregation of mine rock will be based on the following:

- systematic radiometric scanning of blast hole cuttings in clean rock zones to detect anomalous radioactivity levels;

- systematic sampling of blast hole cuttings and if necessary analysis by the XRF method to detect anomalous metal content (e.g., uranium, arsenic)

- radiometric probing of blast holes in ore zones to define ore/waste boundaries;

- radiometric scanning of working faces during excavation to confirm blast hole scanning/probing results;

- overhead scanning of mine rock in the proximity of Type 3 or ore once loaded onto trucks;

- daily scanning of the Type 2 mine rock disposal area to ensure that no Type 3 mine rock or ore was inadvertently placed; and

- systematic sampling to assess acid generation potential of mine rock; and

- if uncertainty as to classification exits, all questionable Type 2 mine rock will be considered as Type 3 for further disposal in the Main Zone and Andrew Lake open pits as a mitigation method.

The uranium content of mine rock is estimated in the field by radiometric techniques and subsequently confirmed with drill cutting assay results. In addition, XRF technology has been identified as a reliable field evaluation tool that can be used to directly determine potentially problematic metals in mine rock during mining. The XRF technology has greatly advanced in recent years, and it is generally accepted as a quantitative screening tool for environmental investigations and industrial site clean up activities.

The above measures were successfully used for mine segregation at the McClean Lake Operation and it is proposed to apply them during mining of the Kiggavik and Andrew lake pits.

### 6.2.2 Grade Control

Grade control is the process of segregating mine rock and managing the quality and quantity of ore mined to ensure the delivery of the required ore tonnes and grade to the processing plant. Grade control is generally based on the detailed geological and grade information from blast holes or dedicated grade control drilling, interpreted to provide ore outlines for mining, production estimates, stockpile inventories and feed grades.

In Kiggavik, blast hole probing will be the main tool of grade control during the ore mining phase. Additional geological data will be collected as possible, to refine the geological model and improve the selectivity and resource estimation process.

### *Blast Hole Probing*

Probing is the process of collecting a radiometric log of the blast holes and converting the radioactivity in equivalent uranium content. Probing has several advantages over conventional blast hole sampling; high density sampling, immediate results and grade interpretation, low cost and minimal operator exposure to harsh climates, radiation and dust.

Ore is exposed on the first bench in all pits, and therefore a grade control system must in place for start of mining operations.

The basic equipment for blast hole probing is a gamma probe, cable winch, data logger and laptop PC. The equipment is mounted in a probing vehicle, a tracked vehicle will be the most appropriate for the conditions, a 4x4 pick up truck will be less well suitable to the anticipated conditions. A GPS system installed in the probe vehicle will be used for real time surveying during probing. The processing of probe data uses specialized software to download, compile and estimate ore, produce digline maps and transfer drill hole and block data to mine engineering and planners.

The probing will be conducted on a dedicated drilling grid, designed for grade control purposes and referred to as an ore grid. The grid spacing and orientation will be based on geological ore controls and mining equipment size. The ore grid will be located within the designed blast pattern from engineering development plans, however the blast pattern may extend beyond the area of the ore grid. The grid location and extent will be based on the resource block model and existing drill holes and the ore grid data for those benches below the first bench. The grid will be placed in the field using GPS survey equipment and drilled with ~5 inch diameter drill holes, depending on hole stability. Once completed, the holes will be probed, if hole collapse occurs, the probing must follow behind the drilling. After completion of all probing, the data is downloaded, converted to equivalent uranium grade, the blast is estimated, and digline maps and coordinates are loaded into GPS units. Once all data is verified, the blast hole pattern will be stacked out, drilled and blasted. After the blast is shot and levelled, diglines are staked out and mining started. Ore mining will be conducted on 3 meter benches and loaded into ore trucks by excavator. Geological technicians will monitor the ore mining, ensure that each load has consistent type ore as possible and trucks are loaded to a consistent height. Technicians will use hand held GPS to follow mining and map daily progress for reporting and grade verification.

All trucks from the ore zone are scanned for classification and grade determination, the scanner will indicate the stockpile destination. Material scanned as waste from the ore zone will be classified as Type 3 mine rock, all ore will be classified into 3 stockpiles based on the scanner readings. The cut-offs will be determined by analysis of the grade distribution of the resource model in order to generate 3 classes with sufficient tonnage for adequate stockpiles and moderate grade variability. A record of all scanned trucks and the stockpile assignment will be saved to a file in the scanner digital logger unit and downloaded on a daily basis to a database. Daily production reports are generated from the database for distribution. The database is an integral part of the reconciliation and grade control process.

### *Geological Mapping*

Geological mapping can contribute to the interpretation of the ore controls and improve the estimation and possibly indicate additional ore lenses. Mapping can also be important for improving geotechnical and hydrogeological interpretations and warn of potential stability problems or water issues. As part of the geological component of grade control, pit wall mapping by bench will be carried out and blast hole cuttings logging completed to interpret the geology by blast. The data will be compiled on bench plans and used for ongoing mine planning, ore grid placement and geological interpretation. Due to extreme conditions, cold temperature and short days, the amount of geological mapping conducted may be limited.

### *Equipment and Support Considerations*

Prior to start up, key equipment and systems must be in place to proceed to the mining stage.

- Grade correlation for probes, a separate correlation may be needed for each area.

- An ore truck scanner system must be designed, with appropriate considerations given to the conditions and truck size. The design must consider new technologies for radiometric detectors, weighing systems, truck positioning controls, data transfer and database development. A critical consideration is a sampling system for ore trucks to develop a grade correlation for the scanner.

- Extreme weather and extended darkness may be a significant motivation for GPS guided excavators in ore mining. Digline boundaries can be loaded into digital block model maps in the GPS system of the excavator where the bucket position is indicated in real time. The ore can be mined according to the diglines where visual control may be poor or absent. An added benefit is maintain even floor level, for reduced truck wear and more precise volume measurement for production. A consideration for this system is the GPS satellite availability at this high latitude location.

- Blast heave is often significant, may consider using blast movement indictors to measure and map the shift. This may be important in GPS guided systems where, maps can be generated with predicted blast movement for more accurate selectivity.

- Geological mapping of cuttings could be done to develop a more detailed geological plan of the pit area, especially outside the main ore zones. This information may be used to help with the waste rock classification and monitor blasting performance by rock type.

### *Personnel*

The grade control team will consist of geological technicians and geologists. The probing and sampling/analysis operations are time consuming and will require 2 technicians on a daily basis for the life of the pit. An additional technician will be required to monitor the ore mining operations, which is a more or less intermittent function. The requirements for technician involvement will however be dependant on the use of GPS machine guided systems for ore mining. With a GPS system the ore mining may require less direct monitoring, however would require more time for data processing and management. A geologist will be required for supervision, planning, data verification and interpretation, database maintenance, mapping and training of the on site geology personnel. As a minimum, the grade control functions could be performed by 2 technicians and one geologist per shift on a rotation type schedule. A senior geologist position is also required to oversee the operations, set and improve operating procedures, provide direction and additional coverage as required.

A geotechnical specialist will be retained during the summer field seasons to review and complete as required, the wall mapping data in order to interpret the information from a geotechnical and hydrological viewpoint and asses the potential for problems as the pit is advanced.

# 7 SOURCE TERM ASSESSMENT

Because the mine rock assessment was completed in stages, initial testing was completed based on characterization studies of uranium deposits in the Athabasca region of Northern Saskatchewan. Material was either classified as “Clean Waste” or “Special Waste”. Results of preliminary testing of the Kiggavik and Andrew Lake material were used to better categorize the mine rock material. Therefore, independent testing was not always possible for the different waste categories. Results discussed below have been separated based on the updated characterization criteria, but may be limited due to completion of the testing prior to definition of the Type 1, 2 and 3 waste categories.

## 7.1 TYPE 1 MINE ROCK

### 7.1.1 Kiggavik Rock Material

Kinetic testing on Type 1 Kiggavik rock material in the form of humidity cell tests was conducted on three composite samples with average uranium and total sulphur contents that categorize the material as Type 1 mine rock. The three humidity cells had a geomean bulk uranium content of 13.6, 35.1, and 8.2 mg/kg with corresponding total sulphur contents of about 0.01, 0.03, and 0.07 %S (Table 7.1-1). The leached COPC concentrations from the three humidity cell tests are shown as a function of time in Attachment E. It should be noted that dramatic drops in COPC concentrations illustrated at week 18, as shown in Attachment E**,** are a result of improved analytical detection limits that were implemented to enhance the sensitivity of the testing. Laboratory loading rates were calculated by using the steady-state COPC concentration demonstrated by the plateau in the COPC concentration as a function of time. The steady-state condition was considered to begin at week 20 for the various COPCs, where those leached concentrations were used in the loading rate calculations from week 20 to termination at week 36. The geomean of the loading rates calculated for each COPC for each of the three humidity cells was then calculated to represent the loading rate for the Type 1 Kiggavik rock. The calculated laboratory loading rates for each COPC are presented in Table 7.1-2.

There are several conservatisms inherent in the calculated laboratory loading rates, which will naturally translate to conservative values for the field loading rates. These conservatisms will impose influences on many calculations, such as pore water concentrations, assessments for amounts of construction material, and constituent loadings in the various management strategies. These conservatisms were considered in further detail in order to better understand the conservative nature of the calculated values. First, it is well known that some metals are pH sensitive and that concentrations of those metals will be controlled to low concentrations at neutral pH. The pH sensitive metals include Al and Cu. The loading rates of many other COPCs in the laboratory humidity cell tests represent maximum values because the concentrations in the leachate were reported as less than the analytical detection limits. These COPCs include sulphate, As, Cd, Co, Ni, Pb, Sb, Se, V and Zn. Only Mn, Mo, Sr and U exhibited concentrations in leachate that were consistently above detection limits.

Initial flushes of COPCs were also calculated from the initial five weeks of testing, according to the assumption that oxidation products present on each of the rock sample's surfaces were washed off during the initial five weeks. This flushing phenomenon is depicted for sulphate in Attachment F, which provided a preliminary basis for making this assumption. The initial flush calculations were used to further enhance the understanding of the mine rock material characteristics in laboratory settings compared to the actual field setting.

The leachate collected from each of the three Type 1 humidity cells generally had circum-neutral pH values during the steady-state period with overall average pH values ranging from 5.5 to 5.9. The lowest individual pH was 4.5 and the highest was 6.6 during the steady-state period. The relatively lower pH values during the initial 12 weeks of testing, as well as variability from week to week, was determined to be caused by acid contamination during sample processing. Once this laboratory issue was noticed it was quickly remedied such that pH values after week 12 were not adversely affected.

There were no field cells constructed that contained material that would have been solely representative of the Type 1 Kiggavik rock because at the time of the initiation of field cells the segregation criteria had yet to be defined. Nonetheless, the two field cells that contain the Kiggavik rock material, namely Main Zone and Centre Zone material, are considered to be Type 3 Kiggavik rock according to their weighted total sulphur contents, and as such are discussed below for the Type 3 Kiggavik rock material.

### 7.1.2 Andrew Lake Rock Material

Two humidity cells contained material classified as Type 1 from the Andrew Lake deposit, with mean U-contents of about 12 mg/kg and 16 mg/kg and total sulphur contents of 0.01 and 0.03 %S, respectively. The loading rate for each COPC is presented in Table 7.1-2, where the loading rate was calculated from the geomean of the individual loading rates from the two humidity cells. Initial flushes of COPCs were also calculated from the initial five weeks of testing, according to the assumption that oxidation products present on each of the rock sample's surfaces were washed off during the initial five weeks. The initial flush calculations were used to further enhance the investigations understanding of the mine rock material characteristics in laboratory settings compared to the actual field setting.

The leachate collected from the two Type 1 Andrew Lake rock humidity cells had overall average pH values slightly more acidic than the circum-neutral range with overall averages of 5.45 and 5.50 during the steady-state period, as defined and discussed previously. The lowest individual pH value was 4.5 and the highest value was 6.5.

There was one field cell constructed with Andrew Lake rock material defined as Type 1 based on the weighted uranium and total sulphur average of about 17.9 mg/kg and 0.017 %S, respectively. The field cell was sampled five different times for which all COPC concentrations were quantified as above detection, with the exception of Cd in three of the five sample events and Cr in one sampling event. The results of the dissolved COPC concentrations were converted to mass of constituent leached per mass of material present in order enhance the understanding of the leaching behavior of the mine rock material. The total mass leached per total amount of material is presented in Table 7.1-3. The field cell COPC leached amounts were used to draw comparisons with the leaching rates calculated from shake flasks, modified-SWEP tests, and initial laboratory flushed quantities in order to better quantify the amount of leaching that will actually take place from the Type 1 Andrew Lake rock. The calculated leaching of the material in the field cells was used to compare to laboratory leaching results by taking into account different flushing rates and sizes (or surface area) of the material. Further analysis of the field cells, shake flasks, modified-SWEP tests, and initial laboratory flushes indicates that insufficient precipitation had entered the field cell material to remove oxidation products present on the rock surfaces. However, when differences in total amount of precipitation collected and surface area of the material is compared to laboratory tests the field cell constituent concentrations are generally consistent with predictions from adjusted laboratory rates (field rates).

## 7.2 TYPE 2 MINE ROCK

### 7.2.1 Kiggavik Rock Material

Loading rates for the Type 2 Kiggavik rock mine materials were the same as the loading rates presented in Table 7.1-1 for the Type 1 mine material, as previously discussed. Furthermore, the field loading rates are summarized in Table 7.2-1 with the appropriate adjustments for temperature and surface area differences between laboratory and field conditions. The field loading rates were used to predict the total mass loadings from the mine rock pile as a whole, but also were used to predict pore water concentrations expected at the bottom of the mine rock stockpile. The pore water calculations and evaluation are completed in Section 9.2 as part of the long-term effects assessment of the Type 2 material with comparisons made to CCME aquatic freshwater guidelines.

### 7.2.2 Andrew Lake Rock Material

Loading rates for the Type 2 Andrew Lake mine rock materials were the same as the loading rates presented in Table 7.1-2 for the Type 1 mine rock material. The adjusted field loading rates are summarized in Table 7.2-1 based on correction factors for temperature and surface area differences between laboratory and field conditions. The field loading rates were used to predict the total mass loadings from the permanently stored mine rock pile as a whole, but also were used to predict pore water concentrations expected at the bottom of the mine rock stockpile (Section 9.2).

There were no field cells constructed to specifically test the Type 2 Andrew Lake mine rock material, but because of the geochemical characteristics and leaching behavior previously discussed regarding the Type 1 and 2 similarities, the Type 1 Andrew Lake field cell findings were used to further expand of the Type 2 mine rock long-term effects. The long-term effects from the disposal of the Type 2 Andrew Lake mine rock material are evaluated and discussed in Section 9.2.

## 7.3 TYPE 3 MINE ROCK

### 7.3.1 Kiggavik Rock Material

There were four humidity cells tested that contained the Type 3 Kiggavik mine rock material. The composite material in each of the four humidity cells had geomean U-contents of about 407, 15.8, 10.4, and 5.1 mg/kg with corresponding geomean total sulphur contents of 0.07, 0.18, 0.36, and 0.89 %S, respectively (Table 7.1-1). The loading rates for each COPC for the Type 3 Kiggavik mine rock materials were calculated from the humidity cell results during the steady-state period. The calculated loading rates from the four humidity cells were then geometrically averaged to produce a loading rate term for all expected Type 3 Kiggavik rock (Table 7.1-2) and adjusted for field conditions (Table 7.2-1**)**. The field loading rates were then used to calculate the predicted pore water concentration that will be produced by the temporary stockpile during operation, as well as the long-term loads from the in-pit disposal. The predicted pore water concentrations for all the COPCs, as they relate to the temporary stockpiles, as well as long-term evaluations are conducted in Section 9.3. Initial flushes of COPCs were also calculated from the initial five weeks of humidity cell testing, according to the assumption that oxidation products present on each of the rock sample's surfaces were washed off during the initial five weeks. The initial flush calculations were used to further enhance the investigation's understanding of the mine rock materials characteristics in laboratory settings compared to the actual field setting. The initial flushes from the humidity cell tests from the four Type 3 Kiggavik rock humidity cells are presented in Table 7.3-1.

The leachate collected from the four Type 3 Kiggavik rock humidity cells during the steady-state period had overall average pH values within the circumneutral pH range with overall averages ranging from 6.03 to 7.06. The lowest individual pH value was 5.48 and the highest value was 7.55 during the steady-state period.

The results from the radionuclide analysis on the leachate from the four Type 3 Kiggavik mine rock humidity cells from the three sampling events (Day 0, 14, and 140) are summarized in Table 7.3-2. The results from the radionuclide analysis on the leachate collected from the Type 3 Kiggavik column from the sampling event on Day 17 are also provided in Table 7.3-2.

There were two field cells constructed with material from the Kiggavik site that was classified as Type 3 mine rock material. One of the field cells that contained Type 3 mine rock material had rock core samples from the Main Zone deposit, which had a weighted U-content and total sulphur content of 17.6 mg/kg and 0.46 %S, respectively (Table 7.3-3). The other field cell with

Type 3 mine rock material contained rock core samples from the Centre Zone deposit, which had a weighted U-content and total sulphur content of 30.1 mg/kg and 0.50 %S, respectively (Table 7.3-3). The two field cells were sampled five different times for which all COPC concentrations were quantified as above detection, with the exception of As, Cd, and Cr, which were quantified as less than detection for either the Main Zone or Centre Zone field cells during at least one sampling event. The results of the dissolved COPC concentrations were converted to mass of constituent leached per mass of material present in order enhance the understanding of the leaching behavior of the mine rock material. The total mass leached per total amount of material is presented in Table 7.1-3. The COPC masses leached from the two field cells were then averaged to get a value representative of the Kiggavik mine rock material as a whole, which are also presented in Table 7.1-3. These values were used to draw comparisons with the leaching rates calculated from leachability tests and initial laboratory flushed quantities in order to better quantify the amount of leaching that will actually take place from the Type 3 Kiggavik rock in the field. The calculated leaching of the material in the field cells was used to compare to laboratory leaching results by taking into account different flushing rates and sizes (or surface area) of the material. Further analysis of the field cells, leachability tests, and initial laboratory flushes indicates that the field cell material was probably not flushed enough to remove all oxidation products on the rock surfaces. However, when differences in total amount of precipitation collected and surface area of the material is compared to laboratory tests the field cell constituent concentrations are generally consistent with our predictions from adjusted laboratory rates (field rates).

Plots of dissolved COPC concentrations in time for the underwater column experiment containing the Type 3 Kiggavik mine rock material are presented in Attachment G. The pH of the leachate collected from the column, generally on a weekly basis, has remained relatively consistent throughout the testing ranging from pH values between 6.6 and 7.7. For the most part, the COPC dissolved concentrations have slowly decreased during the duration of the testing, where it appears that steady-state conditions may have been reached for most COPCs after 20 to 30 weeks. However, dissolved concentrations of Al, Cr, Fe, and Pb continue to remain variable relative to each sampling event, and Al and Cr have actually demonstrated slight increases in concentrations since the initiation of the testing. Loadings from the solid to the pore water were calculated from the column experiment results and are summarized in Table 7.3-4. In general the loadings exhibit decreases with time or are steady with small positive values. These results will be incorporated into the source term for the backfilled pit that will contain tailings overlain by special waste and covered to surface by clean rock.

### 7.3.2 Andrew Lake Rock Material

There were two humidity cells constructed that contained the Type 3 Andrew Lake mine rock material. The composite material in the two humidity cells had geomean U-contents of about 297 and 18 mg/kg with corresponding geomean total sulphur contents of 0.02 and 0.16 %S, respectively (Table 7.1-1). The loading rates for each COPC for the Type 3 Andrew Lake mine rock materials were calculated from the humidity cell results during the steady-state period. The calculated loading rates from the four humidity cells were then geometrically averaged to produce a loading rate term for all expected Type 3 Andrew Lake mine rock (Table 7.1-2) and adjusted for field conditions (Table 7.2-1). The field loading rates were then used to calculate the predicted pore water concentration that will be produced by the temporary stockpile during operation, as well as the long-term loads from the in-pit disposal. The predicted pore water concentrations for all the COPCs, as they relate to the temporary stockpiles, as well as long-term evaluations are conducted in Section 9.3. Initial flushes of COPCs were also calculated from the initial five weeks of humidity cell testing, according to the assumption that oxidation products present on each of the rock sample's surfaces were washed off during the initial five weeks. The initial flush calculations were used to further enhance the investigation's understanding of the mine rock material characteristics in laboratory settings compared to the actual field setting. The initial flushes from the humidity cell tests from the two Type 3 Andrew Lake mine rock humidity cells are presented in Table 7.3-1.

The leachate collected from the two Type 3 Andrew Lake mine rock humidity cells during the steady-state period had overall average pH values of about 6.1 and 4.6. The pH range during the steady-state period for the humidity cell with the geomean total sulphur content of 0.02 %S was 5.7 to 6.8, whereas the pH range for the humidity cell with a total sulphur content of 0.16 %S was 3.9 to 5.9.

The results from the radionuclide analysis on the leachate from the two Type 3 Andrew Lake rock humidity cells from the three sampling events (Day 0, 14, and 140) are summarized in Table 7.3-2. The results from the radionuclide analysis on the leachate collected from the Type 3 Andrew Lake column from the sampling event on Day 17 are summarized in Table 7.3-2.

Plots of dissolved COPC concentrations in time for the underwater column experiment containing the Type 3 Kiggavik mine rock material are presented in Attachment G. The pH of the leachate collected from the column has remained relatively consistent throughout the testing ranging from pH values of about 7 to 8. The lowest pH was measured on the first day of sampling. For the most part, the COPC dissolved concentrations have slowly decreased during the duration of the testing, where it appears that steady-state conditions may have been reached for many COPCs by week 20. Loadings values were calculated from the column experiment data and are summarized in Table 7.3-4.

# 8 MINE ROCK MANAGEMENT FACILITIES

## 8.1 VOLUMES AND TONNAGES

A production rate for mine rock material has been estimated based on the proposed mine plan. Table 8.1-1 provides an estimate of material volumes. The “split” between Type 2 and Type 3 is considered a conservative estimate, such that the volume of Type 3 material is likely over-estimated. The overburden will be comprised mainly of frozen sand and gravel, and granite boulders with some zones of silt and glacial till. The clean overburden will be stockpiled for use in reclamation while the contaminated overburden, if any, will be disposed of within Main Zone Tailings Management Facility during decommissioning.

**Table 8.1-1 Estimated Quantities of Mine Rock and Overburden Materials**

| | In situ overburden (Mbcm) | In situ mine rock (Mbcm) | | Mine rock broken estimates ($Mm^3$) | | |
|---|---|---|---|---|---|---|
| | | Type 1 + Type 2 | Type 3 | Type 1 | Type 2 | Type 3 |
| **Kiggavik** | | | | | | |
| Purpose Built Pit | 0.3 | 0.35 | - | 0.3 | 0.1 | - |
| East Zone | 0.6 | 2.0 | 0.06 | 0.5 | 2.1 | 0.1 |
| Centre Zone | 1.8 | 4.6 | 0.1 | 1.2 | 4.8 | 0.2 |
| Main Zone | 3.0 | 21.3 | 0.3 | 5.5 | 22.1 | 0.5 |
| **Sissons** | | | | | | |
| Andrew Lake | 3.3 | 33.1 | 0.5 | 8.6 | 34.4 | 0.8 |
| End Grid | | 0.3 | 0.04 | 0.1 | 0.3 | 0.1 |

*Notes:*

*Open pits: the volume/tonnages are at a cut off grade of 900ppm and post mining recovery*

*End Grid: the volume/tonnage are at a cut off grade of 2100 ppm and are post mining*

*In situ mine rock*

*Volumes are in million bank cubic meters (Mbcm), in situ volume prior to excavation*

*Mine rock broken estimates*

*Volumes are in million loose cubic meters ($Mm^3$)*

*A swelling factor of 1.30 is considered*

*A Type 1/ Type 2 ratio of 20% is considered for all deposits*

*A Type 1/ Type 2 ratio of 75% is considered for the Purpose Built Pit*

*Type 3 estimates conservatively include a 30% contingency*

## 8.2 TEMPORARY STOCKPILES

### 8.2.1 Design

**Kiggavik Site**

Type 3 mine rock from the perspective of acid rock drainage and metal leaching will be segregated and temporarily stored during operation in a stockpile adjacent to the Main Zone pit. Runoff and water percolating through the temporary stockpile will be collected using ditches and a holding pond, such that the water can be recycled for use in the mill and/or treated before release. During decommissioning of the site, all Type 3 mine rock from the Kiggavik pits will be hauled and disposed of within the Main Zone TMF.

It is estimated that the volume of Type 3 mine rock from the Kiggavik Pits will fall within the range of 700,000 to 1,400,000 $m^3$ (broken). The corresponding temporary stockpile will be located along the north perimeter of the pit. The planned footprint of the temporary stockpile at Kiggavik is approximately 12 ha.

The design for the Type 3 waste rock pad for Kiggavik includes a single liner system and the corresponding sedimentation pond includes a double liner system. Due to the permafrost foundation conditions, the liners are required to be constructed on a rockfill pad. The pad footprints will be stripped of organic materials and ice rich soils (soils containing ice lenses or high ice contents) which may be prone to creep when loaded. Excavations in permafrost ground will be covered immediately using Type 1 mine rock fill to prevent permafrost degradation. Fill material will include non-potentially acid generating free draining rock materials. The minimum thickness of the rock pad will be 1.5 m. The pad grading will be designed to drain into the sedimentation ponds.

Typically the pad will include the following layers (from top to bottom):

1. Transition layer of 100 mm minus rockfill;
2. Cover sand over liner;
3. Non-woven filter cloth as cushion between sand cover and liner.
4. 80 mil HDPE or LLDPE liner;
5. Non-woven filter cloth as cushion between sand bedding and liner;
6. Base sand/liner bedding-as above;
7. Transition layer of 100 mm minus rockfill; and,
8. Minimum 1.5 m thick non-potentially acid generating Type 1 mine rockfill pad.

The rockfill pad will include perimeter berms to 1 m height. The pad liner system will run up the perimeter berms and will be anchored in a trench at the berm crest to provide control for runoff. More detailed information regarding pads is included in Technical Appendix 2D (Conceptual Design for Ore and Special Waste Pads and Ponds)

**Sissons Site**

It is proposed to manage Type 3 mine rock from the Andrew Lake pit during operation in a temporary surface stockpile with drainage collection and treatment. This material will then be placed at the bottom of the Andrew Lake pit during the decommissioning phase. The design concept for the Sissons temporary stockpile will be similar to that of the Kiggavik temporary stockpiles. The planned footprint of the temporary stockpile at Sissons is approximately 13 ha.

### 8.2.2 Stability Analysis

Stability analyses were performed using the limit equilibrium slope stability program SLOPE/W (Geoslope, 2010), applying the Morgenstern-Price Method to compute factors of safety for selected failure surface. The critical section for stability examined was the Type 3 mine rock temporary pile and its sedimentation pond with a conservative maximum design height of 40 m. Material properties used in the analysis are presented in Figure 8.2-1.

Materials are expected to be frozen, but thawed parameters were used for analyses. Generally, freezing of materials will add strength and increase factors of safety for stability. Static and seismic loadings conditions were examined. Pseudo-static analyses considered the 1/1000 year return period event including a peak ground acceleration (PGA) of 0.035 g (2005 National Building Code Seismic Hazard Calculation). One-half of the PGA, or 0.02 g was applied in stability analyses as the horizontal seismic coefficient.

A friction angle of the liner system of 17 degrees was assumed based on experience and testing for soil – liner interactions at other projects in the north.  However laboratory testing of liner system friction is recommended for future design stages.

The results of stability analysis can be summarized as follows:

- Static conditions
    - Design criteria/Minimum Factor of Safety 1.0
    - Calculated Factor of Safety 1.3
- Pseudo-static conditions
    - Design criteria/Minimum Factor of Safety 1.0
    - Calculated Factor of Safety 1.2

The analysis shows that the calculated factors of safety meet or exceed the minimum design criteria values (see also Technical Appendix 2D ("Conceptual Design for Ore and Special Waste Pads and Ponds")

# 8.3 PERMANENT STOCKPILES

## 8.3.1 Design

### Kiggavik Site

Mine rock not utilized for construction and deemed acceptable from the perspective of acid rock drainage and metal leaching will be permanently stockpiled at the Kiggavik site. It is proposed to construct two stockpiles to accommodate the Type 2 material excavated from the Kiggavik pits.

Approximately 22 million $m^3$ (unbroken) of Type 2 mine rock material will be generated. The cumulative volume of the two stockpiles is estimated to be on the order of 29 million $m^3$, based on a swelling factor of 1.3. Excavated mine rock will be loaded on trucks and hauled to the stockpile.

Stockpile North will be located to the north of the Kiggavik pits (Figures 8.3-1). The planned footprint of stockpile North is approximately 52 ha. Stockpile South will be located to the south of the pits. The planned footprint of stockpile South is approximately 77 ha. Both stockpiles will be surrounded by perimeter ditches designed to collect runoff water from the stockpiles. Drainage water from the Kiggavik stockpiles will largely consist of direct precipitation (i.e., rainfall and snowfall), with minor amounts of blowing snow and surrounding catchment runoff, impinging the pile perimeter. Berms will be constructed along the outer edge of each ditch. These berms will prevent surface runoff from surrounding undeveloped areas from flowing into the perimeter ditch and mixing with runoff water from the stockpiles.

The stockpiles will be constructed in order to meet appropriate physical stability criteria. A layered approach to stockpile construction is proposed to increase the overall stockpile stability. The layered placement creates a high uniform density while minimizing segregation to create a stockpile with minimal permeability to air and water penetration. The method also reduces settlement and therefore further enhances overall stockpile stability. It is expected that the stockpiles will be 30 m to 40 m high and constructed in approximately 10 m lifts with catchments remaining at the completion of each lift. An angle of repose of 37° and an overall dump slope of 26.5° are considered at this stage. The catchments will also act as a slope break and minimize erosion caused by surface runoff.

### Sissons Site

At Andrew Lake it is proposed to manage the Type 2 mine rock in one stockpile (Figure 8.3-2). This stockpile is designed in a manner similar to that of the Kiggavik stockpiles. Based on the open pit mine plan, the volume of the Sissons mine rock stockpile is estimated to be on the order 35 million $m^3$, based on a swelling factor of 1.3. Excavated mine rock will be loaded on trucks and hauled to the waste rock pile(s).

It is proposed to stockpile End Grid mine rock from the ramp extraction with the Andrew Lake mine rock. Some mine rock from End Grid will be stockpiled separately and crushed to be used in mine backfill. The planned footprint of the permanent stockpile at Sissons is approximately 132 ha

## 8.3.2 Thermal Conditions

Thermal modeling was carried out using the finite element model SVHEAT (SoilVision Systems, 2009) to simulate thermal conditions during the construction of the permanent piles as well as long term conditions. The objective of the modeling assessment was to evaluate the rate of permafrost development within the pile given different construction scenarios. This was achieved by varying the lift thickness, the sequence and season of deposition. The piles were modeled by placing two or five lifts with thickness of 5, 10 or 25 m. The lifts were placed at intervals of 6, 12 or 18 months; seasonal deposition varied between summer deposition (warmest) and winter deposition (coldest). Additional information regarding the thermal modeling is included in the Technical Appendix 5G (Thermal and Water Transport Modeling for the Waste Rock Piles and TMF).

Three material types were used in the simulations, namely coarse waste rock, waste rock subject to traffic, and bedrock. The thermal properties used in the simulations are summarized in Table 8.3-1. The properties of the bedrock are based on the values presented in the Technical Appendix 5B (Geology and Hydrogeology Baseline). The properties of the waste rock material were deducted from the properties of the bedrock as a granular material and on the observations reported by Neuner et al. (2009). Traffic zone waste rock has higher water content as a result of the higher fraction of fines.

**Table 8.3-1 Thermal Properties of Mine Rock and Bedrock**

| Material | Thermal conductivity [kJ/(day m C)] | | Volumetric heat capacity [kJ/($m^3$ C)] | | Porosity | Volumetric water content |
|---|---|---|---|---|---|---|
| | Unfrozen | Frozen | Unfrozen | Frozen | | |
| Waste rock traffic | 135 | 161 | 2099 | 1722 | 0.30 | 0.18 |
| Waste rock | 71 | 74 | 1597 | 1471 | 0.30 | 0.06 |
| Bedrock | 271.9 | 275.5 | 2314 | 2293 | 0.01 | 0.01 |

Two climate scenarios were considered; a current climatic condition scenario and a climate change scenario. The climate change scenario was simulated by introducing a warming trend of 5°C over a 100 year period (see Technical Appendix 4D “Baker Lake Long Term Climate Scenario”).

The results of the thermal modelling indicate that:

- Permafrost would eventually develop in the permanent stockpiles under the current climate conditions;
- Undisturbed ground surfaces (natural ground) will likely retain permafrost after a warming of 5°C;
- Permafrost could become marginal or disappear at the base of the permanent pile when certain surface conditions are considered and when the 5°C warming from climate change is considered;
- Winter deposition of mine rock in the pile will promote development of frozen conditions, while summer deposition will delay it. The time difference could be in the order of several decades before reaching similar thermal conditions;
- Frozen conditions could be promoted by depositing alternating lifts of mine rock in the summer and winter; and
- Traffic zones have negligible effects on the thermal regime, but their presence will promote the development of water barriers once frozen, thus restricting percolation through the pile.

### 8.3.3 Water Balance

**Climatic Parameters**

The mean climatic parameters and upper and lower ranges suggested for the Kiggavik Project area are presented in Table 8.3-2. These parameters have been estimated by Golder Associates Ltd. (Golder) using a number of sources and methods. The ranges indicated in Table 8.3-2 serve to address the upper and lower boundary that may occur depending on climatic conditions. Depending on the application, a higher or lower value may be used to bring a degree of conservatism to a particular assessment or for a design basis.

**Table 8.3-2 Suggested Climatic Parameters to be used for Water Balance Calculations at Kiggavik**

| Parameter (mm/year) | Mean | Lower value | Upper Value |
|---|---|---|---|
| Rainfall | 169.8 | 121.9 | 217.7 |
| Snowfall | 174.8 | 118.1 | 231.4 |
| Evaporation | 182.8 | 127.9 | 253.9 |
| Evapotranspiration | 122.5 | 85.7 | 170.1 |
| Sublimation | 48.9 | 33.1 | 64.8 |
| Blowing Snow | 31.5 | 21.3 | 41.7 |

**Snow Distribution**

A summary of recent studies characterizing the hydrology of waste rock piles at mines sites located in cold climates (Diavik, Ekati, Red Dog and Svea) is included in the Technical Appendix 5I (Hydrology of Waste Rock Piles in Cold Climates). This summary provides a background for predictions of infiltration from the proposed permanent piles at the Kiggavik and Sissons sites. It also suggests that wind plays a large role in the redistribution of the snow at the top and slopes of the piles.

Quantitative results from a comprehensive research program at Diavik have indicated, based on results from 2006 and 2007, that snow depth prior to melt had been from 0 cm to 5 cm at top of pile while the piles slopes were accumulating approximately 150 cm (Neuner 2009). This study does not highlight differences between the windward and leeward slopes of the piles. Taking into account that the annual snowfall in millimetres of snow water equivalent at Diavik is approximately 187 mm, these results are an indication that the snowfall at the top of the piles is transported downwind likely to the leeward slope of the pile while the windward slope trap snow until a drift in equilibrium is formed. Similarly, the leeward slope will also trap snow until a drift in equilibrium is formed.

Climate variables at Diavik are comparable with climate variables at Baker Lake (Table 8.3-3). Consequently the findings from the hydrology studies at Diavik are reasonably representative for the Project, i.e., it will be expected that future mine rock piles at the Kiggavik Project will have large proportion of snow transported from the top to the leeward slopes of the piles, but there will not likely be large differences between the snow deposited in the windward and leeward slopes. The explanation of this could be the topographic characteristic surrounding the piles, which is basically a flat, tundra non-vegetated land surface where the only major topography feature is the pile. It means that most of the blowing snow from the land on the site of the windward slope will deposit on that slope forming a drift that eventually will reach equilibrium while the snow from the top will be transported to the leeward side and farther where the drift formed on that slope reaches equilibrium. The fetch for the blowing snow at the top is the average width of the pile top perpendicular to the dominant wind direction.

**Table 8.3-3 Relevant Climate variables for Diavik and Baker Lake**

| Parameter | Baker Lake | Diavik |
|---|---|---|
| Temperature ($^oC$) | - 12 (1946 – 2008) | - 11 (1959 – 2006) |
| Rainfall (mm) | 169 | 184 |
| Snowfall water equivalent (mm) | 175 | 177 |

An estimate of the amount of snow remaining on the permanent mine rock pile surface prior to the spring snowmelt period is included in the Technical Appendix 5H (Waste Rock Water Balance). In this assessment the height of the piles was assumed to be 50 m and the length of the pile is assumed to be 34 times the height or larger. Results of the estimate can be summarized as follows:

- The expected amount of snow on the top of the permanent mine rock piles prior to the snowmelt at the Project site, which typically occurs in late May or early June will be close to zero for average winters, zero for dry winters and less than 5 mm for wet winters.

- The expected amount of snow on the slope of the mine rock piles before snowmelt at the end of May will be less than 307 mm for average winters, less than 257 mm for dry winters and less than 401 mm for wet winters. If these amounts exceed the maximum equilibrium volume, the amount trapped will be the maximum volume.

### Net Percolation

The water balance of the permanent stockpiles was assessed using a water transport model that incorporated the climatic conditions at the Kiggavik site. The objective was to estimate the net percolation through the entire profile. The simulations were applied to uncovered and covered mine rock profiles. The soil water modelling was performed using Hydrus-1D software (Simunek et al., 2009)

The geometry that was modelled consisted of a 25 m thick vertical profile that included 0.5 m zones to represent traffic zones. This profile was to represent a pile constructed in five lifts of 5m each, with the top 0.5 m of each lift being composed of finer material to represent traffic zones. A soil cover was incorporated in some of the simulations, and consisted of adjusting the hydraulic properties of the top traffic zones to represent the cover material.

Three material types were used in the simulations: coarse mine rock, fine mine rock and silty clay. The silty clay was used in some simulations to represent the inclusion of a soil cover over the stockpile.

The properties of the waste rock material (fine and coarse) are based on the values presented in Neuner et al. (2009) for the Diavik Mine. The hydraulic properties of the silty clay were selected from Hydrus-1D (Simunek et al. 2009). The hydraulic properties were modelled according to the van Genuchten-Mualem model with the air entry value limited to 0.02 m of suction. The parameters used to define the hydraulic properties of the materials are listed in Table 8.3-4 (the parameters α and η are for the soil water retention function while l is the tortuosity parameter in the hydraulic conductivity function).

**Table 8.3-4 Hydraulic Properties used in the Infiltration Model**

| | Volumetric water content | | Saturated hydraulic conductivity | α | η | l |
|---|---|---|---|---|---|---|
| | Residual | Saturated | (m/s) | (1/m) | | |
| Fine waste rock | 0.01 | 0.25 | 9.0E-06 | 5.9 | 1.45 | 0.5 |
| Coarse waste rock | 0.01 | 0.25 | 1.0E-02 | 1.6 | 4.00 | 0.5 |
| Silty clay | 0.07 | 0.36 | 5.6E-08 | 0.5 | 1.09 | 0.5 |

More detailed information regarding the infiltration model is included in Technical Appendix 5G (Thermal and Water Transport Modeling for the Waste Rock Piles and TMF).

The results of the water transport modelling indicate that:

- Net percolation through mine rock material will likely be in the order of 30 to 50 percent of the annual total precipitation under unfrozen conditions. For frozen permanent mine rock piles, the percolating water will also freeze and eventually block the flow paths. The bulk of the mine rock material would consequently be isolated from percolating water and restrict the transport of contaminants that could be present in the mine rock material. The percolation values mentioned above would eventually be limited to the freeze-thaw zone near the surface (active zone).

- Fine grained soil covers have the potential of reducing the net percolation to less than 5 percent of the annual total precipitation;

- The traffic zones (finer particles) will retain more water, thus acting as barriers to water transport once the pile is frozen. The zones could encapsulate and maintain large dry zones within the pile that would be deprived from percolating water. It could also act as a barrier to oxygen transport and consequently limit the oxidation of the mine rock material; and

- The water storage of the permanent mine rock pile profile could take years to stabilise as the mine rock material will be deficient in pore water when placed in the pile.

# 9 POTENTIAL POST-DECOMMISSIONING EFFECTS

## 9.1 CONSTRUCTION MATERIAL – TYPE 1 MINE ROCK

Based on the segregation criteria for the mine rock material that has been predicted to be produced during mining operation, the Type 1 mine rock material was assessed for its use as construction material. The characterization of the Type 1 mine rock material presented in Section 5.0 provided an initial understanding of the static and kinetic properties of the material. Further evaluation of the mine rock characterization, and calculations with results from the various kinetic tests, has provided a guideline for the Type 1 material's use for construction purposes.

The field loading rates for the Type 1 material (Table 7.2-1) were used to calculate an estimated constituent concentration that would be produced from the Type 1 material that may be used in construction at either the Kiggavik or Andrew Lake sites. The constituent concentrations were estimated based on a unit area and mass of mine rock material that could be used in construction in accordance with the annual water balance. The amount of mine rock material that could safely be used in any type of site construction was determined to insure CCME guidelines for the protection of freshwater aquatic life would not be exceeded.

### 9.1.1 Kiggavik Type 1 Mine Rock

The summary of the estimated constituent concentrations that would originate from the Type 1 Kiggavik mine rock material used in construction according to mass are shown in Table 9.1-1. Using the water balance values provided in Table 8.3-3 and assuming a total infiltration of 80%, the net inflow of water into the rock pile was calculated to be 0.275 $m^3$/yr based on a unit surface area. Therefore, only an estimated 225 kg of Type 1 Kiggavik material could be used per square meter in order to stay within the CCME guidelines for all COPCs. However, due to the conservatism incorporated into the field loading rates due to detection limit sensitivity, as well as pH controls on the some of the actual COPC concentrations, a greater amount of Type 1 material could potentially be used for construction. When considering the pH dependency of Al and Cu, as well as screening index associated with Cd, Ni, Pb, Sb, Se, V, and Zn, a more realistic amount of material that could be used in construction would be 1,800 kg of material per square meter, above which the predicted dissolved U-concentration begins to exceed CCME guidelines. It is considered that this estimate is still conservative because unfrozen conditions are assumed. In reality the rock is frozen during several months of the year and no dissolution occurs.

### 9.1.2 Andrew Lake Type 1 Mine Rock

The summary of the estimated constituent concentrations that would originate from the Type 1 Andrew Lake mine rock material used in construction based on mass are shown in Table 9.1-1. Similar to results for the Kiggavik Type 1 rock material, an estimated 60 kg of material could be used per square meter in order to stay within the CCME guidelines for all COPCs. However, accounting for conservatism in the results, up to 525 kg of material per square meter could potentially be used for construction purposes and remain below CCME guidelines, after which predicted Cu concentrations exceed CCME guidelines assuming a pH of about 7. Additionally, if the loading rates for the COPCs that are influenced by detection limit issues are considered to be no more than ten times conservative, as for Cd, then 3,600 kg of material could be used for construction purposes before U concentration begin to exceed CCME guidelines.

## 9.2 PERMANENTLY STOCKPILED TYPE 2 MINE ROCK

The influence on the site's water quality was assessed for the Type 2 mine rock material that will be permanently stored during operation and at closure at both the Kiggavik and Andrew Lake site. The field loading rates calculated and discussed previously in Section 7 were used to make estimates of the quality of water that will discharge from Pointer and Andrew Lakes. The discharge concentrations were conservatively calculated from the expected loadings from the rock piles and the annual average discharge rates from the lakes that were considered to be the first receiving water body at each site. The loadings were calculated from the volumes of mine rock to be produced (Table 8.1-1) with an assumed bulk density of 1500 kg/$m^3$. The mean annual discharges for Pointer Lake and Andrew Lake were estimated to be 15.9 and 6.63 $Mm^3$/a,, respectively (see Technical Appendix 6A "Hydrology Baseline"). The resulting concentrations were then conservatively calculated as the loading rate divided by the flow rate as a simple mixing process with no chemical attenuation for any constituents.

For example, the field loading rate for sulphate from the Kiggavik stockpile is 0.011 mg-$SO_4$/kg-rock/wk or 0.57 mg /kg/year. Therefore, based on an estimated mass of rock in the stockpile of 54.9 Mt (36.6 $Mm^3$ x 1500 kg/$m^3$), the annual mass loading to Pointer Lake would be 31,500 kg/a. Using a mean annual discharge of 15.9 $Mm^3$/a, the average annual concentration of sulphate at the Pointer Lake discharge is about 1.98 mg/L. The predicted discharge concentrations for all COPCs are presented in Table 9.2-1 and are compared to CCME Canadian water quality guidelines for the protection of freshwater aquatic life.

There are a number of conservative assumptions incorporated into these calculations, such that these values likely represent maximum concentrations, and under actual field conditions many COPC concentrations will likely be much lower due to natural pH control (and solubility limitations) for some analytes, and natural attenuation or chemical reactions that have not been accounted for in the calculations. In addition, the calculations assume that all drainage from the stockpile would report directly to either Pointer or Andrew Lakes, and do not account for attenuation or dilution. The stockpiles were also assumed to fully contribute to the loadings and to not contain a frozen core.

Based on these assumptions the predicted concentrations for Pointer Lake, which will receive drainage from the Kiggavik rock stockpile, exceed the CCME guidelines for Al while the predicted concentrations for Andrew Lake, which will receive drainage from the Andrew Lake rock stockpile, exceed the guideline values for Al, Cd, Cu, Se, and Zn based on mass balance and mixing predictions (Table 9.2-1).

The aluminum concentration estimated for Pointer Lake is based only on mass balance for aluminum release from the clean rock. Under neutral pH conditions, aluminum will be controlled to low concentrations in the pore water of the rock and the predicted concentration in Pointer Lake is therefore greatly exaggerated over the more realistic value that will occur in the field. Also, concentrations of Cd, Cu, Se and Zn, that are predicted to exceed CCME values in Andrew Lake are based on detection limit values from the humidity cell results and therefore are biased high and are likely to be lower in the field than the values predicted in Table 9.2-1. Therefore, at a screening level, the loadings from the clean rock piles are considered to remain protective of downstream water quality at the Kiggavik and Sissons sites for the long term.

## 9.3 TYPE 3 ROCK MATERIAL

As part of the mine rock characterization, management, and effects assessment for the Type 3 Kiggavik and Andrew Lake mine rock material, the long-term effects on the sites water quality were evaluated from laboratory and field tests. It has been proposed that the Type 3 material will be temporarily stored on lined pads during operation and all drainage from the rock will be collected and treated as required. At closure, the rock will be relocated to the open pits, where it will be submerged by water as the pits flood. The rock will be permanently stored underwater in the respective mined out pits at each of the two primary sites, namely Kiggavik and Andrew Lake. However, the in-pit disposal at closure of the mine materials produced during operation will be partitioned differently and contain different materials at each of the two primary sites. At the Kiggavik site, there are a few different proposals to manage both the tailings and Type 3 mine rock material. Although a few different management strategies have been proposed, they all essentially consist of an initial deposition of tailings followed by the deposition of the Type 3 mine rock, which would then be overlain by a portion of the Type 2 mine rock with a final layer of till cover. Whereas, at the Andrew Lake site, only the Type 3 mine rock material will be first placed in the open pit followed by a thin layer (~4 m) of Type 2 mine rock, which would finally be covered with a thin layer of till (~1 m). Once all of the solid materials are place in the Andrew Lake pit, the pit will either fill by natural precipitation or under accelerated conditions to eventually submerge the placed materials.

The results from the humidity cell tests on the Type 3 waste rock material provided a basis to calculate the initial soluble load to the pore water in the flooded rock in the Kiggavik pit and to the pit water in the Andrew Lake pit. The pore water concentrations originating from the temporarily stored material were calculated using the field loading rates and by estimating the sizes of the temporary stockpiles.

Other sources of loadings to the Andrew Lake pit water include the rubble left on benches and the walls of the mine out pit. A thorough analysis and assessment of the immediate and long-term influences on the water quality of the Andrew Lake pit are discussed in Section 10 below.

**Temporary Rock Piles - Pore Water Concentrations**

The pore water concentrations expected to be produced in the temporarily stored Type 3 mine rock material were calculated using the field loading rates, an estimate of the mine rock piles dimensions, and the expected annual water balance. The expected loose volume and surface areas of Type 3 mine rock are about 2.0 $Mm^3$ and 15.1ha for the Kiggavik temporary mine rock storage pile and 1.52$Mm^3$ and 14.2$Mm^2$ for the Andrew Lake stock pile. Based on the expected loose volume and surface area available for storage it was calculated that the final height of the piles would be about 13 and 11m for the Kiggavik and Andrew Lake piles, respectively. Using a net inflow of water into the rock pile of 0.275 $m^3$/yr based on a unit surface area and assumed density of 1500 kg/$m^3$, the pore water concentrations in the rock piles were calculated and are summarized in Table 9.2-1.

Runoff and water percolating through the temporary stockpile will be collected using ditches and a holding pond underlain by liners, such that the water can be recycled for use in the mill and/or treated before release.

**In-pit Disposal of Type 3 Material**

After the Type 3 mine rock material is placed in the mined out pits, it will continue to contribute loads as the pit fills, either naturally or under accelerated conditions, and after it is submerged. Although leaching behavior of mine rock is greatly reduced once materials become submerged and thus placed in a relatively anoxic environment, the materials can continue to contribute small loadings. Underwater leaching tests were established to assess what loads could be expected from the submerged Type 3 mine rock material.

At closure, the Type 3 material will be relocated to the mined-out open pits, where it will be submerged by water as the pits flood. Although a few different management strategies have been proposed for the Kiggavik site, all options essentially consist of an initial deposition of tailings followed by the deposition of the Type 3 mine rock, which would then be overlain by a portion of the Type 2 or clean mine rock with a final layer of till cover. A laboratory assessment has been initiated to assess the water quality of this layered system. The objective of the testing is to simulate the release of COPCs from the tailings porewater to the water within the pore spaces of the overlying rock. These data will be used to support transport calculations and to calculate expected water quality at the top of the final water table within the rock backfill.

The Type 3 Kiggavik Mine rock testing involves the placement of Type 3 mine rock above tailings within a column. Type 2 mine rock material will be placed above the Type 3 rock. The water level in the column is maintained within the special waste to mimic the proposed disposal strategy. Within the pit, the water near the water table surface will represent the active flow system through which water will migrate out of the closed pit. Therefore, fresh water is injected into the special waste layer weekly, and the overflow collected and analyzed. The test will run for a minimum of 20 weeks and the results will be compiled to provide a source term for inclusion into the water quality model.

At the Andrew Lake site, backfilling of the pit will consist of the Type 3 mine rock material with a thin layer of Type 2 mine rock, which would finally be covered with a thin layer of till (~1 m). No tailings deposition is planned for the Andrew Lake pit. Once all of the solid materials are backfilled, the pit will either fill by natural precipitation or under accelerated conditions to eventually submerge the relocated materials. An assessment of post-closure water quality in the Andrew Lake pit that incorporates the relocation of Type 3 rock to the pit is provided in Section 10 below.

# 10 ANDREW LAKE PIT WATER QUALITY

The Andrew Lake open pit will have a capacity of approximately 40 $Mm^3$ after mining is completed and will fill with water over time to become a flooded pit. The pit will be used for disposal of Type 3 Andrew Lake rock material after closure that has been temporarily stored near the pit during operation. The water quality in the flooded pit was assessed to determine conditions after flooding is complete and before discharge from the pit will occur. There are two possible scenarios for flooding after closure. One scenario is to allow flooding that will occur naturally as a result of the accumulation of rain and snow melt and the small amount of seepage that may be expected to occur in the active layer near ground surface. At the expected natural filling rate, complete flooding will require approximately 480 years.

Alternatively, the natural filling of the pit can be complemented by flow from a larger water body, such as Andrew Lake, during periods of high flow in order to shorten the flooding period. Experience has shown that while leaching of metals and other COPCs can occur while rock, including pit walls, is exposed to the atmosphere and natural weathering processes, such leaching tends to be insignificant to non-measurable when the same rock is submerged below water. This difference in behaviour suggests that rapid flooding may have some advantages for maintaining good water quality at some sites. However, water quality will depend on site-specific conditions, including expected leaching rates for pit rock. Therefore, the water quality in the flooded Andrew Lake pit was evaluated at a conservative screening level in order to determine whether or not natural filling would result in acceptable water quality after flooding was complete.

There are several potential sources of loadings of COPCs to the pit water after closure while the pit is filling with water and after filling is complete and flow out of the pit occurs. The Type 3 rock that will be stored temporarily on a pad near the pit will be exposed to weathering during the mining operation and loadings will result in pore water or moisture in the rock to have elevated concentrations. When this rock is relocated to the pit at the end of mining, the COPCs in the pore water with the rock will mix with the flooding water as it fills the pore spaces in the rock. Although not all of the dissolved mass in the pore water will be released to the water above the flooded rock, it can be conservatively assumed that all of the mass will mix in the pit water for this screening assessment. After the rock is relocated to the pit, it will require about 19 years to completely flood the rock in the pit. During the flooding period, the rock that is above water will continue to leach and to be a source of loadings to the pore water within the rock. The contribution of COPCs can be estimated with the loading rates and exposure time before complete flooding. An important potential source of loadings to the pit water while filling is occurring is the leaching of the rock material that occurs as rubble on the pit floor and that remaining on the benches after mining is complete. The rubble will be exposed to weathering before it is submerged in water. The loadings for the unflooded rubble can be estimated. The final potential source of loadings to the pit water is the leaching from the pit walls that are above the rising water level. The walls are exposed and subject to leaching while above water and the loadings report to the pit water. All of these potential contributors to mass loadings to the pit water were quantified in order to estimate the concentrations of COPCs in the flooded pit at approximately 500 years post closure.

At a screening level, the mass loadings to the pit water were calculated for potential contributions of COPCs from the rubble on the pit floor, the pit wall loadings during flooding, pore water from the temporarily stored material that will be relocated to the pit, and the short-term leaching of the re-placed Type 3 material during flooding. The total loads for each COPC was then assumed to be fully mixed in the pit water to estimate the concentrations when filled.

After filling, the natural input of water to the pit will continue until overflow or to a level controlled by an outlet or structure. The only rock that can contribute important loadings to the natural inflow will be the pit walls remaining above the final water level. The resulting concentrations in the pit water were calculated for steady-state conditions by estimating the annual loadings from the pit walls mixed with the annual outflow volume. At a screening level, this provides a limit on the long term concentrations that may develop in the pit water.

## 10.1 METHODS

The prediction of water quality was based on mass balance principles and mixing with no consideration of geochemical reactions and therefore is highly conservative in nature. The leaching rates for COPCs were based on the results from humidity cell testing on Andrew Lake rock that are summarized in Tables 10.1-1, 10.1-2 and 10.1-3. The leaching or loading rates for the rubble and wall material are shown in Table 10.1-3. The loading rates used for the Type 3 material that will be temporarily stored on a lined pad and the pore water concentrations that will develop in the rock before it is relocated to the pit are shown in Table 10.1-1 and Table 10.1-2, respectively. All loading rates used in the Andrew Lake pit water quality predictions have conservative adjustments for temperature and grain size. Additionally, the loading rates used in the calculated loading contributions from the rubble material and the pit walls were those of the Type 3 material in order to remain conservative in the assessment, as it is assumed that the rubble and walls will have compositions and chemical properties similar to all three criteria types of material.

Leaching of the rock in the pit was assumed to occur while materials were exposed to the atmosphere prior to submergence. After flooding, the leaching rate of the rock was assumed to be zero. The flooding rate of the pit and the stage curve for volume as a function of elevation were used to calculate the area of the pit that was exposed as a function of time. Figure 10.1-1 shows the pit volume as a function of elevation. The natural filling rate was then used to calculate the water elevation as a function of time that is shown graphically in Figure 10.1-2 . The unflooded area of the pit floor was calculated as a function of filling time and the results are presented graphically in Figure 10.1-3.

The cumulative loads for COPCs were calculated for the pit filling period. While the deepest areas of the pit floor will be exposed for a relatively short period of a few tens of years, the topmost bench will be exposed for the entire approximately 500 year filling period. Therefore, the four components, including walls, rubble, pore water, and replaced Type 3 material, that contributed to constituent loadings, either at the beginning or during pit filling, were summed to get the total constituent load once the pit has filled. At the end of the pit flooding, it was assumed that the cumulative loads of COPCs were mixed with 43 $Mm^3$ of pit water and the resulting concentrations were calculated. The value of 43 $Mm^3$ of water represents a conservative volume of water that could be in the pit with 15 m of exposed pit walls above the water.

After filling, it was assumed the water will flow out of the pit as a result of net natural inflow from precipitation and runoff to the pit. The annual flow rate through the pit was estimated to be about 89,500 $m^3$/a. The total loads of COPCs to the pit from the exposed walls once the pit has completely flooded were then mixed with the annual flow to estimate steady-state concentrations in the outflow water.

The calculated concentrations in the pit water immediately after flooding and at steady-state were then compared to guideline values for the protection of aquatic life to determine if the estimated water quality is acceptable for release to the environment.

## 10.2 RESULTS

The calculated well-mixed concentrations of COPCs after approximately 480 years of pit filling are summarized in Table 10.2-2. The calculated concentrations are compared to guideline values for the protection of freshwater aquatic life (CCME, 2007). The results show that the pit water will have concentrations that are less than the guideline values at the end of natural pit filling with the exception of aluminum (0.069 mg/L) and cadmium (0.000017 mg/L) concentrations. The calculated value for aluminum is very conservative and is based on mass balance and mixing only. In reality, aluminum concentrations are pH sensitive and the final aluminum values will likely be much lower than the calculated one at neutral pH expected in the pit. The calculated concentration for cadmium was biased on the high side by using the detection limit values for concentrations in the humidity cell leachate samples. The cadmium concentrations in the leachate were consistently less than the detection limit and therefore the calculated concentration in the pit represents a maximum value and the actual value is likely to be much less.

The pit water quality may also be improved with accelerated pit filling. The enhanced pit filling would produce water quality better than that for the natural filling scenario because more rapid filling would result in shorter exposure times for the same rock mass or surface area creating less cumulative loadings of COPCs that will mix with the same total volume of water. The mass loads originating from the four loadings sources are shown in Table 10.2-1 under three different filling conditions. The filling rate of 89,553 $m^3$/a is representative of the natural filling rate, whereas the filling rates of 428,000 $m^3$/a and 4,280,000 $m^3$/a are representative of accelerated fill rates to achieve complete pit filling in 100 and 10 years, respectively. Figure 10.2-1 shows the resulting Andrew Lake pit water concentrations for sulphate and uranium under the three pit filling scenarios. Therefore, if natural long-term filling results produce unacceptable water quality for only two constituents, more rapid filling can produce acceptable water quality.

The results for long term steady-state concentrations in the pit water from the exposed pit walls are also summarized in Table 10.2-2 with surface water quality guidelines for the protection of freshwater aquatic life. The calculated long term steady-state concentrations for all COPCs are less than the applicable guideline values. The long term steady-state concentrations are also less than the calculated concentrations at the end of the natural filling period. That means that the concentrations of COPCs in the pit water will decrease slowly from the values at the end of pit filling period toward the long term steady-state values. In other words, because of the small loading rates associated with the pit walls, the maximum predicted concentrations in the pit will be those at the end of pit filling and the values will decrease over the long term after pit outflow occurs. Therefore, with accelerated pit filling the pit water quality will remain acceptable without risk to freshwater aquatic life or to humans.

# 11 MONITORING AND FOLLOW-UP PROGRAM

Contingency plans are intended to address unforeseen circumstances which could result in a significant increase in predicted environmental impacts. Extensive investigations into the chemical properties of mine rock at Kiggavik and Sissons will continue to be undertaken during operation as part of a Mine Rock Optimization and Validation Program (MROVP). This program will focus on Type 3 mine rock and will provide further insight into the geochemical characteristics of the materials to validate our proposed long-term management plan

A laboratory investigation has been initiated to evaluate the water quality implications for the tailings and mine rock co-disposal scenario for the Kiggavik Pit. Results are pending, and will be provided once sufficient data are obtained.

It is proposed to install monitoring wells within the backfilled rock to monitor and compare to predicted water quality within the pit. Monitoring wells would also be installed in the active zone downgradient of the permanent Type 2 mine rock stockpiles to monitor any seepage.

It is also proposed to install shallow active layer monitoring wells down gradient of the Type 3 temporary stockpiles and ponds.

# 12 CONCLUSION

An assessment was completed to characterize the physical and geochemical properties of the potential mine rock materials expected from the development of the Kiggavik Project as they relate to the long-term management. The proposed mine rock management plan for the Project was developed and evaluated in relation to potentially adverse effects of mine rock on the receiving environment. The assessment included field and laboratory studies and geochemical interpretation synthesis to determine the management strategy.

Unlike the mine rock from uranium deposits in the Athabasca region of Northern Saskatchewan that have been shown to be a source of elevated arsenic concentrations when deposited under water, the results of the initial leaching tests of the Kiggavik and Andrew Lake material suggest that the mine rock from these sites does not appear to have similar arsenic leaching issues as the other materials from the Athabasca region.

The proposed segregation criteria for mine rock at the Kiggavik Uranium Project includes Type 1 material as clean rock that can be used for construction purposes and contains concentrations of uranium less than exemption quantities as per the Nuclear Safety and Control Act (1 Bq/g or 40 mg/kg) and a total sulphur content less than 0.1%. Type 2 material is defined as mine rock that can be permanently stockpiled and managed above ground and contains uranium contents less than 250 mg/kg and total sulphur content be less than 0.1%. Type 3 material is defined as mine rock that requires specific management (e.g. in-pit disposal), and includes all rock that is not classified as Type 1 or 2 material or considered to be ore.

Type 2 mine rock from the Kiggavik Site will be placed in two permanent stockpiles. One permanent stockpile will be present at the Sissons Site to accommodate the Type 2 material from the Andrew Lake Pit. Stockpiles will be surrounded by perimeter ditches designed to collect runoff water from the stockpiles. It is proposed that Type 3 mine rock will be segregated and temporarily stored during operation in a stockpile adjacent to either the Main Zone or Andrew Lake pits. Runoff and water percolating through the temporary stockpiles will be collected using ditches and a holding pond. During decommissioning of the site, all Type 3 mine rock will be disposed of within the Main Zone TMF or Andrew Lake Pit. The Type 3 mine rock pad will include a single liner system with a double liner system in the sedimentation ponds.

At the Kiggavik site, there are a few different proposals to manage both the tailings and Type 3 mine rock material, however they all essentially consist of an initial deposition of tailings followed by the deposition of the Type 3 mine rock, which would then be overlain by a portion of the Type 2 mine rock with a final layer of till cover. At the Andrew Lake site, only the Type 3 mine rock material will be first placed in the open pit followed by a thin layer (~4 m) of Type 2 mine rock, which would finally be covered with a thin layer of till (~1 m). Once all of the solid materials are place in the Andrew Lake pit, the pit will either fill by natural precipitation or under accelerated conditions to eventually submerge the placed materials.

The results of static and humidity cell testing of the mine waste materials were used to assess potential post-decommissioning effects on water quality. The three types of waste rock material were assessed for potential effects on site water quality in accordance with each of the proposed management strategies. Humidity cell test results were used to calculate anticipated loading rates in the field, according to the waste rock type, by applying adjustments for grain size and temperature. The field loading rates were used to estimate COPC loads and pore water concentrations expected from the permanently stored Kiggavik and Andrew Lake rock in the proposed stockpiles.

Based on conservative estimations, between 225 and 1,800 kg of Type 1 Kiggavik material per square meter and between 525 and 3,600 kg of Type 1 Andrew Lake material per square meter could be used for construction purposes before the predicted dissolved uranium concentration in leachate from the rock begins to exceed CCME guidelines for the protection of freshwater aquatic life.

The influence on the site's water quality was assessed for the Type 2 mine rock material that will be permanently stored during operation and at closure at both the Kiggavik and Andrew Lake site. The calculations are considered conservative, such that the values likely represent maximum concentrations, and under actual field conditions many COPC concentrations will likely be much lower due to natural pH control and solubility limitations for some analytes, and natural attenuation or chemical reactions that have not been accounted for in the calculations. The predicted discharge concentrations for Pointer Lake, which will receive drainage from the Kiggavik rock stockpile, exceed the CCME guidelines for aluminum while the predicted concentrations for Andrew Lake, which will receive drainage from the Andrew Lake rock stockpile, exceed the guideline values for aluminum, cadmium, copper, selenium and zinc based on mass balance and mixing predictions. However, predicted concentrations of cadmium, copper, selenium and zinc in Andrew Lake are based on detection limit values from the humidity cell test results and therefore are biased high and are likely to be lower in the field than the values predicted.

A laboratory assessment has been initiated to assess the water quality of the tailings-Type 3 mine rock co-disposal for the Kiggavik Site. The objective of the testing is to simulate the release of COPCs from the tailings porewater to the water within the pore spaces of the overlying rock. These data will be used to support transport calculations and to calculate expected water quality at the top of the final water table within the rock backfill.

The calculated Andrew Lake pit water concentrations after backfilling with Type 3 mine rock is estimated to be less than the CCME guideline values for all COPCs, except for aluminum and cadmium. However, the calculated aluminum concentration is considered to be very conservative as aluminum is pH sensitive, such that the expected pit water pH will likely result in lower aluminum concentrations. In addition, the estimated cadmium pit water concentration is also extremely conservative as the field loading rates used in the calculations were based on detection limit values from the humidity cell testing. Furthermore, the pit water quality can be improved with accelerated pit filling, which can effectively reduce concentrations of all COPCs to below the CCME guidelines by limiting the time for oxidation of the Type 3 mine rock.

This extensive assessment of the influence of the mine rock materials produced during operation and post-closure on the site`s water quality has suggested that, if properly managed, the mine rock materials will not pose an environmental risk. The use of the Type 1 waste rock material for construction purposes will not present a risk of acid generation or substantial metal leaching. The permanently stored Type 2 waste rock material will not present an acid generating issue, although the initial contact water may require management prior to discharge in order to reduce any COPC concentrations that may exceed water quality guidelines. The temporarily stored Type 3 waste rock material will need to be managed through the appropriate containment and collection of all contact water, which will require treatment before discharge. At closure the Type 3 Kiggavik and Andrew Lake rock materials will be placed in their respective mined out pits and managed underwater after the pits fill.

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## Tables

**Table 4.1-1: List of Samples Evaluated for the Kiggavik Mine Rock and Andrew Lake Rock Geochemical Assessment**

| Andrew Lake | End Grid | Main Zone | Centre Zone | East Zone |
|---|---|---|---|---|
| ANDW09-01 (135-136) | ENDG-09-12 (361.6-362.2) | MZ09-01A (200-201) | CZ09-01 (108-109) | EZ08-01 (39.15-39.21) |
| ANDW09-01 (175-176) | ENDG-09-03 (372-372.5) | MZ09-01A (25-26) | CZ09-01 (159-160) | EZ08-01 (42-42.5) |
| ANDW09-01 (20-21) | ENDG-09-01 (269.5-270) | MZ09-01A (250.05-251.06) | CZ09-01 (19-20) | EZ08-01 (44.9-45) |
| ANDW09-01 (222-222.9) | ENDG-09-07 (114.42-114.92) | MZ09-01A (99.73-100.65) | CZ09-01 (198-199) | EZ08-01 (46.3-46.4) |
| ANDW09-01 (299-300) | ENDG-09-05 (297-297.5) | MZ09-02 (101-102) | CZ09-01 (239-240) | EZ08-01 (48-48.1) |
| ANDW09-01 (323-324) | ENDG-09-02 (238-238.45) | MZ09-02 (150-151) | CZ09-01 (267-268) | EZ08-01 (51-51.11) |
| ANDW09-01 (75-76) | ENDG-09-02 (320.5-321) | MZ09-02 (50.91-51.94) | CZ09-01 (60-61) | EZ08-01 (53-53.1) |
| ANDW-08-01 (25.0-25.5) | ENDG-09-02 (442.1-442.6) | MZ09-03 (114.22-174.85) | CZ08-01 (138-139) | EZ08-01 (54.8-54.9) |
| ANDW-08-01 (50.0-50.5) | ENDG-09-03 (195-195.5) | MZ09-03 (124.15-124.68) | CZ08-01 (171-172) | EZ08-01 (57-57.53) |
| ANDW-08-01 (75.0-75.5) | ENDG-09-04 (205.1-205.6) | MZ09-03 (74.5-75) | CZ08-01 (183-183.9) | EZ08-01 (59-59.1) |
| ANDW-08-01 (215.0-215.5) | ENDG-09-04 (368.4-369) | MZ-08-02 (29.5-30.0) | CZ08-01 (193.92-195) | EZ08-01 (61.04-61.14) |
| ANDW-08-01-007 (214.75-215.0) | ENDG-09-06 (233.45-234) | MZ-08-02 (85.0-85.5) | CZ08-01 (99-100) | EZ08-01 (63-63.1) |
| ANDW-08-01-008 (249.0-249.2) | ENDG-09-08A (261-261.5) | MZ-08-04 (110.0-110.5) | CZ07-01 (106-107) | EZ08-01 (64.96-65.06) |
| ANDW-08-03 (30.0-30.5) | ENDG-09-13 (172.5-173) | MZ-08-04-005 (136.26-136.50) | CZ07-01 (13-14) | EZ08-01 (66.95-67.1) |
| ANDW-08-03 (60.0-60.5) | END-08-02-008 (240.64-240.86) | MZ-08-04-006 (166.23-166.50) | CZ07-01 (20-21) | EZ08-01 (68.5-69) |
| ANDW-08-03 (80.0-80.5) | END-08-02-014 (418.22-418.43) | MZ-08-04 (189.06-189.32) | CZ07-01 (49-50) | EZ08-01 (71.5-71.6) |
| ANDW-08-03 (110.0-110.5) | END-08-03A-002 (60.0-60.25) | MZ-08-04-008 (216.18-216.41) | CZ07-01 (71.16-72.04) | |
| ANDW-08-03 (190.0-190.5) | END-08-03A-004 (104.50-104.72) | MZ-08-05 (19.5-20.5) | CZ07-02 (101-102.05) | |
| ANDW-08-03-002 (61.0-61.2) | END-08-03A-006 (159.70-159.90) | MZ-08-05 (39.5-40.0) | CZ07-02 (26-29) | |
| ANDW-08-03-004 (124.60-124.80) | END-08-03A-008 (220.88-221.10) | MZ-08-05-004 (123.56-123.84) | CZ07-02 (44-44.95) | |
| ANDW-08-04-005 (203.3-203.5) | END-08-03A-0012 (341.50-341.76) | MZ-08-06 (65.0-65.5) | CZ07-02 (48.95-49.95) | |
| ANDW-08-04-006 (227.50-227.78) | END08-01 (11.38-12) | MZ-08-06 (80.0-80.5) | CZ07-02 (95-95.85) | |
| ANDW-08-03-007 (202.46-202.71) | END08-01 (141-141.55) | MZ-08-07 (29.5-30.0) | | |
| ANDW-08-04 (40.0-40.5) | END08-01 (176.4-177) | MZ-08-07-001 (30.0-30.32) | | |
| ANDW-08-04 (60.0-60.5) | END08-01 (59.4-60) | MZ-08-16 (29.5-30.0) | | |
| ANDW-08-04-007 (265.6-265.80) | END08-01 (86.23-87) | MZ08-03 (10.48-11) | | |
| ANDW-08-05-010 (318.64-318.90) | END08-02 (131.5-132) | MZ08-03 (24-24.25) | | |
| ANDW08-01 (17.5-18) | END08-02 (15-15.47) | MZ08-04 (151-151.5) | | |
| ANDW08-01 (28.6-29.1) | END08-02 (190.9-192) | MZ08-04 (166.5-167.02) | | |
| ANDW08-01 (49.1-49.6) | END08-02 (249-249.5) | MZ08-04 (211.5-211.84) | | |
| ANDW08-01 (72-72.5) | END08-02 (35.5-36) | MZ08-04 (251.05-251.55) | | |
| ANDW08-04 (36.5-37) | END08-03 (16.5-17) | MZ08-05 (129-129.5) | | |
| ANDW08-04 (69-69.5) | END08-03 (21-21.5) | MZ08-05 (15-15.5) | | |
| ANDW08-04 (95-95.75) | END08-03 (23.5-24) | MZ08-06 (72-72.3) | | |
| ANDW08-05 (116.55-117) | END08-03 (29.5-30) | MZ08-06 (9-9.5) | | |
| ANDW08-05 (13-13.5) | END08-03 (46.5-47) | MZ-07-01 (20-20.5) | | |
| ANDW08-05 (29.5-30) | ENDG-07-01 (20-20.5) | MZ-07-01 (50-50.5) | | |
| ANDW08-05 (69-69.4) | ENDG-07-01 (50-50.5) | MZ-07-01 (94-94.5) | | |
| ANDW-07-01 (50-50.5) | ENDG-07-01 (140-140.5) | MZ-07-01 (105-105.5) | | |
| ANDW-07-01 (80-80.5) | ENDG-07-01 (215-215.5) | MZ-07-03 (55-55.5) | | |
| ANDW-07-01 (185-185.5) | END07-01 (253.45-254) | MZ-07-03 (100-100.5) | | |
| ANDW-07-01 (200-200.5) | END07-01 (98-98.5) | MZ-07-03 (170-170.5) | | |
| ANDW-07-01 (255-255.5) | | MZ-07-03 (195-195.5) | | |
| ANDW-07-02 (26-26.5) | | MZ-07-04 (20-20.5) | | |
| ANDW-07-02 (51-51.5) | | MZ-07-04 (30-30.5) | | |
| ANDW07-01 (17-18.16) | | MZ-07-04 (40-40.5) | | |
| ANDW07-01 (51.5-53) | | MZ-07-05 (31-31.5) | | |
| ANDW07-01 (75.9-76.9) | | MZ-07-05 (38-38.5) | | |
| ANDW07-01 (91-92) | | MZ-07-05 (55-55.5) | | |
| ANDW07-02 (31-31.96) | | MZ07-01 (13-13.5) | | |
| ANDW07-02 (44-44.7) | | MZ07-01 (164-164.43) | | |
| | | MZ07-01 (209-209.56) | | |
| | | MZ07-01 (26-26.6) | | |
| | | MZ07-04 (7-7.5) | | |
| | | MZ07-04 (93.72-94.2) | | |
| Total Number of Samples | Total Number of Samples | Total Number of Samples | Total Number of Samples | Total Number of Samples |
| 51 | 42 | 55 | 22 | 16 |

**NOTES:**

ANDW = Andrew Lake; ENDG and END = End Grid; MZ = Main Zone; CZ = Centre Zone; EZ = East Zone

07; 08; 09 indicates year hole was drilled

"01" indicates drill hole number

Values in brackets represent sample depth in metres

**Table 4.3-1: Summary of Material used in Humidity Cell Tests with Uranium and Sulphur Contents**

**Kiggavik Rock Humidity Cell Material**

| Humidity Cell Name: | Drill Core Sample ID: | Uranium (U) mg/kg | Total S % |
|---|---|---|---|
| MZ > 250 (Type 3) | MZ-08-04-005 136.26-136.50 | 261 | 0.18 |
| | MZ-07-01 (105-105.5) | 264 | <0.01 |
| | MZ-07-05(31-31.5) | 294 | 0.01 |
| | CZ07-01 (71.16-72.04) | 500 | 0.16 |
| | MZ-07-03(195-195.5) | 550 | 0.09 |
| | MZ-08-16 29.5-30.0 | 818 | 0.61 |
| | **Geomean** | 407.17 | 0.07 |
| MZ <0.02 (Type 1) | EZ08-01 (54.8-54.9) | 1.9 | <0.01 |
| | CZ09-01 (19-20) | 3.2 | 0.01 |
| | EZ08-01 (53-53.1) | 3.4 | <0.01 |
| | EZ08-01 (51-51.11) | 3.7 | <0.01 |
| | EZ08-01 (42-42.5) | 5.7 | 0.01 |
| | CZ07-01 (13-14) | 16.9 | <0.01 |
| | MZ-08-05 19.5-20.5 | 17 | 0.01 |
| | CZ07-02 (48.95-49.95) | 18.5 | <0.01 |
| | MZ-08-04-006 166.23-166.50 | 22 | 0.01 |
| | MZ-07-03(170-170.5) | 24.8 | 0.01 |
| | CZ07-02 (26-29) | 29.7 | 0.01 |
| | MZ-08-02 85.0-85.5 | 33 | 0.01 |
| | CZ07-01 (20-21) | 62 | 0.01 |
| | MZ-07-01 (50-50.5) | 97.2 | 0.01 |
| | **Geomean** | 13.61 | 0.01 |
| MZ 0.02 - 0.05 (Type 1) | CZ08-01 (99-100) | 23.5 | 0.02 |
| | CZ07-02 (44-44.95) | 37.1 | 0.02 |
| | MZ08-04 (166.5-167.02) | 44.1 | 0.02 |
| | MZ-07-04(20-20.5) | 80.6 | 0.02 |
| | MZ-08-04 110.0-110.5 | 121 | 0.02 |
| | MZ07-01 (13-13.5) | 13.5 | 0.03 |
| | CZ07-01 (106-107) | 18.9 | 0.03 |
| | MZ07-04 (93.72-94.2) | 29.9 | 0.03 |
| | CZ07-01 (49-50) | 50 | 0.03 |
| | MZ-07-05(38-38.5) | 10.9 | 0.04 |
| | CZ07-02 (95-95.85) | 33.7 | 0.04 |
| | MZ-07-01 (94-94.5) | 136.5 | 0.04 |
| | MZ-08-02 29.5-30.0 | 3 | 0.05 |
| | MZ-07-04(40-40.5) | 32 | 0.05 |
| | MZ-07-04(30-30.5) | 215 | 0.05 |
| | **Geomean** | 35.05 | 0.03 |
| MZ 0.05 - 0.1 (Type 1) | CZ07-02 (101-102.05) | 8.7 | 0.06 |
| | MZ07-01 (209-209.56) | 22.9 | 0.07 |
| | MZ09-03 (124.15-124.68) | 2.8 | 0.09 |
| | **Geomean** | 8.23 | 0.07 |

| Humidity Cell Name: | Drill Core Sample ID: | Uranium (U) mg/kg | Total S % |
|---|---|---|---|
| MZ 0.1 - 0.3 (Type 3) | EZ08-01 (68.5-69) | 3.6 | 0.11 |
| | MZ07-01 (164-164.43) | 21.2 | 0.13 |
| | EZ08-01 (57-57.53) | 3.1 | 0.14 |
| | MZ08-06 (9-9.5) | 6.7 | 0.14 |
| | MZ07-04 (7-7.5) | 36.7 | 0.14 |
| | MZ08-04 (211.5-211.84) | 26.5 | 0.15 |
| | MZ-08-04 189.06-189.32 | 85 | 0.15 |
| | EZ08-01 (61.04-61.14) | 3 | 0.16 |
| | EZ08-01 (59-59.1) | 3.2 | 0.22 |
| | EZ08-01 (66.95-67.1) | 3.2 | 0.22 |
| | MZ-07-03(100-100.5) | 7 | 0.24 |
| | MZ-08-05 39.5-40.0 | 186 | 0.24 |
| | MZ-07-05(55-55.5) | 164.5 | 0.26 |
| | MZ08-04 (251.05-251.55) | 30.2 | 0.28 |
| | MZ08-05 (129-129.5) | 35.4 | 0.28 |
| | **Geomean** | 15.76 | 0.18 |
| MZ 0.3 - 0.5 (Type 3) | MZ09-01A (250.05-251.06) | 9.4 | 0.3 |
| | MZ-08-06 65.0-65.5 | 28 | 0.3 |
| | MZ08-04 (151-151.5) | 33.7 | 0.3 |
| | MZ-08-04-008 216.18-216.41 | 29 | 0.31 |
| | MZ-08-05-004 123.56-123.84 | 57 | 0.31 |
| | MZ09-01A (200-201) | 6 | 0.33 |
| | MZ-09 03 (174.22-174.85) | 3.9 | 0.37 |
| | MZ08-03 (24-24.25) | 3.1 | 0.44 |
| | CZ08-01 (138-139) | 3.7 | 0.46 |
| | MZ09-02 (150-151) | 3.2 | 0.47 |
| | CZ09-01 (239-240) | 11.6 | 0.48 |
| | **Geomean** | 10.35 | 0.36 |
| MZ >0.5 (Type 3) | MZ07-01 (26-26.6) | 30.9 | 0.53 |
| | EZ08-01 (63-63.1) | 3.5 | 0.58 |
| | MZ09-03 (74.5-75) | 3 | 0.59 |
| | MZ-07-01 (20-20.5) | 3.5 | 0.61 |
| | MZ08-05 (15-15.5) | 8.8 | 0.65 |
| | MZ-08-06 80.0-80.5 | 13 | 0.65 |
| | MZ08-06 (72-72.3) | 4 | 0.66 |
| | MZ-08-07 29.5-30.0 | 15 | 0.7 |
| | MZ-08-07-001 30.0-30.32 | 15 | 0.72 |
| | CZ09-01 (108-109) | 2.8 | 0.78 |
| | CZ09-01 (198-199) | 2.5 | 0.81 |
| | CZ09-01 (159-160) | 3.3 | 0.84 |
| | MZ09-01A (25-26) | 3.6 | 0.85 |
| | CZ09-01 (60-61) | 4.8 | 0.91 |
| | CZ09-01 (267-268) | 2.8 | 0.98 |
| | CZ08-01 (171-172) | 2.6 | 1.08 |
| | MZ09-02 (50.91-51.94) | 3.6 | 1.16 |
| | MZ-07-03(55-55.5) | 6.5 | 1.21 |
| | MZ09-02 (101-102) | 22.5 | 1.61 |
| | CZ08-01 (193.92-195) | 2.5 | 1.64 |
| | MZ09-01A (99.73-100.65) | 3.3 | 1.65 |
| | CZ08-01 (183-183.9) | 2.8 | 1.83 |
| | **Geomean** | 5.13 | 0.89 |

**Andrew Lake Rock Humidity Cell Material**

| Humidity Cell Name: | Drill Core Sample ID: | Uranium (U) mg/kg | Total S % |
|---|---|---|---|
| ANDW >250 (Type 3) | ANDW-07-01(200-200.5) | 253 | 0.05 |
| | ANDW08-05 (116.55-117) | 254 | <0.01 |
| | ANDW-07-01(185-185.5) | 268 | 0.04 |
| | ANDW-08-04-006 227.50-227.78 | 452 | 0.01 |
| | **Geomean** | 297.03 | 0.02 |
| ANDW <0.02 (Type 1) | ANDW07-02 (31-31.96) | 2.8 | 0.01 |
| | ANDW07-01 (75.9-76.9) | 4.1 | <0.01 |
| | ANDW09-01 (323-324) | 4.2 | <0.01 |
| | ANDW09-01 (75-76) | 5.1 | 0.01 |
| | ANDW08-01 (17.5-18) | 5.3 | <0.01 |
| | ANDW08-05 (29.5-30) | 6.2 | <0.01 |
| | ANDW09-01 (299-300) | 7.3 | 0.01 |
| | ANDW08-04 (69-69.5) | 7.8 | <0.01 |
| | ANDW09-01 (175-176) | 8.8 | 0.01 |
| | ANDW-08-03 190.0-190.5 | 9 | 0.01 |
| | ANDW-08-03-002 61.0-61.2 | 9 | 0.01 |
| | ANDW08-01 (72-72.5) | 10 | <0.01 |
| | ANDW-08-03 80.0-80.5 | 10 | 0.01 |
| | ANDW08-04 (36.5-37) | 12.4 | 0.01 |
| | ANDW08-05 (13-13.5) | 13 | 0.01 |
| | ANDW-08-01 25.0-25.5 | 15 | 0.01 |
| | ANDW-08-01-007 214.75-215.0 | 18 | 0.01 |
| | ANDW-08-01 75.0-75.5 | 19 | 0.01 |
| | ANDW09-01 (20-21) | 21.4 | <0.01 |
| | ANDW08-05 (69-69.4) | 23.3 | <0.01 |
| | ANDW-08-03 60.0-60.5 | 27 | 0.01 |
| | ANDW-08-04 40.0-40.5 | 27 | 0.01 |
| | ANDW07-02 (44-44.7) | 37.5 | <0.01 |
| | ANDW-08-03 30.0-30.5 | 55 | 0.01 |
| | ANDW08-04 (95-95.75) | 58.9 | <0.01 |
| | **Geomean** | 12.11 | 0.01 |
| ANDW 0.02 - 0.1 (Type 1) | ANDW-08-01-008 249.0-249.2 | <2 | 0.02 |
| | ANDW08-01 (28.6-29.1) | 3.2 | 0.02 |
| | ANDW08-01 (49.1-49.6) | 3.8 | 0.02 |
| | ANDW07-01 (51.5-53) | 5.2 | 0.02 |
| | ANDW09-01 (222-222.9) | 7 | 0.02 |
| | ANDW-07-02(26-26.5) | 7.2 | 0.02 |
| | ANDW07-01 (17-18.16) | 3.9 | 0.03 |
| | ANDW07-01 (91-92) | 7.5 | 0.03 |
| | ANDW-08-01 50.0-50.5 | 19 | 0.03 |
| | ANDW-08-03 110.0-110.5 | 34 | 0.03 |
| | ANDW-07-02(51-51.5) | 43.9 | 0.03 |
| | ANDW-08-05-010 318.64-318.90 | 87 | 0.03 |
| | ANDW-08-03-007 202.46-202.71 | 208 | 0.03 |
| | ANDW-07-01(50-50.5) | 3.9 | 0.04 |
| | ANDW-08-01 215.0-215.5 | 19 | 0.04 |
| | ANDW-08-03-004 124.60-124.80 | 33 | 0.04 |
| | ANDW-07-01(255-255.5) | 33.4 | 0.04 |
| | ANDW-07-01(80-80.5) | 8.5 | 0.05 |
| | ANDW-08-04-005 203.3-203.5 | 231 | 0.05 |
| | **Geomean** | 14.71 | 0.03 |
| ANDW >0.1 (Type 3) | ANDW09-01 (135-136) | 15.5 | 0.1 |
| | ANDW-08-04 60.0-60.5 | 25 | 0.18 |
| | ANDW-08-04-007 265.6-265.80 | 15 | 0.21 |
| | **Geomean** | 17.98 | 0.16 |

**Table 4.3-2: Summary of Special Waste Material from Kiggavik Rock and Andrew Lake Rock Used for the Column Testing**

| Kiggavik Rock Special Waste Material | |
|---|---|
| Drill Core Sample ID | Mass Used (kg) |
| MZ-08-04 (47.0-47.5) | 0.66 |
| MZ-08-04 (47.5-48.0) | 0.65 |
| MZ-08-04 (48.0-48.5) | 0.60 |
| MZ-08-04 (48.5-49.0) | 0.64 |
| MZ-08-04 (49.0-49.5) | 0.57 |
| MZ-08-04 (49.5-50.0) | 0.64 |
| MZ-08-04 (50.0-50.5) | 0.24 |
| MZ-08-04 (50.5-51.0) | 0.62 |
| MZ-08-04 (51.0-51.5) | 0.26 |
| MZ-07-01 (60.7-61.2) | 0.23 |
| MZ-07-01 (66.5-66.6) | 0.38 |
| MZ-07-04 (35.0-35.5) | 0.12 |
| MZ-07-04 (35.5-36.0) | 0.16 |
| MZ-07-04 (43.0-43.5) | 0.23 |
| MZ-07-04 (43.5-44.0) | 0.17 |
| MZ-07-04 (44.5-45.0) | 0.10 |
| MZ-07-04 (45.5-46.0) | 0.13 |
| MZ-07-04 (49.5-50.0) | 0.23 |
| MZ-07-04 (50.0-50.5) | 0.09 |
| MZ-07-04 (50.5-51.0) | 0.22 |
| MZ-07-04 (51.0-51.5) | 0.14 |
| MZ-07-04 | 0.17 |
| MZ-07-05 (18.4-18.9) | 0.35 |
| Total Mass Used | 7.61 |

| Andrew Lake Rock Special Waste Material | |
|---|---|
| Drill Core Sample ID | Mass Used |
| ANDW-07-01 (122.0-122.5) | 1.04 |
| ANDW-07-01 (124.0-124.5) | 0.97 |
| ANDW-07-01 (124.5-124.8) | 0.34 |
| ANDW-07-01 (222.0-222.5) | 0.54 |
| ANDW-07-01 (226.5-227.0) | 0.55 |
| ANDW-07-01 (238.5-239.0) | 0.51 |
| ANDW-07-01 (239.0-239.5) | 0.26 |
| ANDW-07-01 (238.5-239.0) | 0.35 |
| ANDW-07-02 (80.5-80.7) | 0.24 |
| ANDW-07-02 (92.5-93.0) | 0.64 |
| ANDW-07-02 (97.0-97.3) | 0.34 |
| Total Mass Used | 5.78 |

**TABLE 5.1-1: Summary of Metal Content and Acid Base Accounting for Kiggavik and Andrew Lake Drill Core Samples**

| Parameter | Units | 10x Average Crustal Abundance[1] | Andrew Lake 51 Samples | | | End Grid 42 Samples | | | Main Zone 55 Samples | | | Centre Zone 22 Samples | | | East Zone 16 Samples | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Geomean | Minimum | Maximum | Geomean | Minimum | Maximum | Geomean | Minimum | Maximum | Geomean | Minimum | Maximum | Geomean | Minimum | Maximum |
| Arsenic (As) | mg/kg | 10 | 3.0 | 0.3 | 57 | 2.0 | <0.2 | 14 | 0.7 | <0.2 | 34 | 1.6 | 0.6 | 13 | 0.6 | <0.2 | 5.3 |
| Cadmium (Cd)[2] | mg/kg | 1 | 0.09 | <0.02 | <1.0 | 0.04 | <0.02 | <1.0 | 0.06 | <0.02 | <1.0 | 0.03 | <0.02 | 0.43 | 0.05 | <0.02 | 2.54 |
| Cobalt (Co) | mg/kg | 290 | 3.6 | 0.30 | 31 | 6.5 | 1.3 | 113 | 9.3 | 1.1 | 177 | 11 | 2.3 | 51 | 14 | 8.2 | 31 |
| Copper (Cu) | mg/kg | 750 | 6.6 | 1.0 | 116 | 7.8 | 1.7 | 932 | 32 | 2.0 | 245 | 17 | 1.7 | 182 | 30 | 4.4 | 148 |
| Molybdenum (Mo) | mg/kg | 10 | 0.73 | 0.20 | 2.5 | 0.82 | 0.15 | 12 | 14 | 0.95 | 1940 | 5.6 | 0.46 | 218 | 0.78 | 0.18 | 2.6 |
| Nickel (Ni) | mg/kg | 1050 | 28 | 4.0 | 418 | 25 | 4.4 | 209 | 20 | 1.3 | 116 | 33 | 10.4 | 362 | 26 | 16 | 53 |
| Lead (Pb) | mg/kg | 80 | 9.3 | 2.6 | 50 | 5.9 | 2.1 | 38 | 29 | 3.6 | 169 | 17 | 3.3 | 288 | 9.7 | 2.6 | 117 |
| Antimony (Sb) | mg/kg | 2 | 0.34 | 0.05 | 3.0 | 0.20 | <0.05 | 2.4 | 0.18 | <0.05 | 1.0 | 0.10 | 0.05 | 0.43 | 0.13 | 0.07 | 0.22 |
| Selenium (Se) | mg/kg | 0.5 | 1.3 | <1.0 | 6.0 | 1.1 | <1.0 | 3.0 | 1.3 | <1.0 | 7.0 | 1.6 | 1.0 | 3.0 | 1.1 | 1.0 | 2.0 |
| Uranium (U) | mg/kg | 9.1 | 17 | <2.0 | 452 | 4.0 | 1.1 | 1370 | 23 | 2.8 | 818 | 11 | 2.5 | 500 | 4.2 | 1.9 | 9.2 |
| Zinc (Zn) | mg/kg | 800 | 13 | <2.0 | 151 | 19 | <2.0 | 120 | 25 | 5.0 | 83 | 35 | 3.0 | 122 | 79 | 33 | 567 |
| | | | | | | | | | | | | | | | | | |
| Paste pH | --- | --- | 7.5 | 7.0 | 8.3 | 7.9 | 6.9 | 8.9 | 8.4 | 7.2 | 9.3 | 8.4 | 7.4 | 9.0 | 8.5 | 7.5 | 9.5 |
| Neutralization Potential (NP) | kg-CaCO3/t | --- | 6.0 | 1.0 | 30 | 8.0 | 1.0 | 82 | 13 | 3.8 | 72 | 12 | 2.0 | 45 | 11 | 6.0 | 19 |
| Acid Generating Potential (AP) | kg-CaCO3/t | --- | 0.6 | 0.3 | 6.3 | 0.6 | 0.3 | 35 | 4.6 | 0.3 | 52 | 3.7 | 0.3 | 57 | 1.4 | 0.3 | 18 |
| NP/AP | --- | --- | 9.9 | 0.62 | 61 | 13 | 0.6 | 237 | 2.8 | 0.2 | 240 | 3.3 | 0.1 | 43 | 7.7 | 0.8 | 43 |
| Total Sulphur | % | --- | 0.02 | <0.01 | 0.21 | 0.02 | <0.01 | 1.13 | 0.15 | <0.01 | 1.65 | 0.12 | <0.01 | 1.83 | 0.05 | <0.01 | 0.58 |
| Sulphate Sulphur | % | --- | 0.01 | <0.01 | 0.05 | 0.01 | <0.01 | 0.05 | 0.02 | <0.01 | 0.05 | 0.02 | <0.01 | 0.05 | 0.01 | <0.01 | 0.02 |
| Sulphide Sulphur | % | --- | 0.01 | <0.01 | 0.20 | 0.02 | <0.01 | 1.12 | 0.12 | <0.01 | 1.61 | 0.10 | <0.01 | 1.79 | 0.04 | <0.01 | 0.56 |
| Inorganic Carbon (C) | % | --- | 0.05 | <0.05 | 0.09 | 0.09 | <0.05 | 0.91 | 0.09 | <0.05 | 0.52 | 0.08 | <0.05 | 0.33 | 0.06 | <0.05 | 0.13 |
| CO2 | % | --- | 0.21 | <0.20 | 0.30 | 0.36 | <0.20 | 3.3 | 0.34 | <0.20 | 1.9 | 0.30 | <0.20 | 1.20 | 0.24 | <0.20 | 0.50 |

**NOTES:**

1 - From Faure, Gunter. 1998. Principles and Applications of Geochemistry. Prentice Hall. New Jersey.

2 - Detection limit varied for Cadmium

When value is reported as below detection, detection limit used for calculation of geometric mean.

**TABLE 5.1-2: Summary of Leachable Metal Concentrations from SWEP Testing**

| Parameter | Units | Andrew Lake 51 Samples | | | End Grid 42 Samples | | | Main Zone 55 Samples | | | Centre Zone 22 Samples | | | East Zone 16 Samples | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Geomean | Minimum | Maximum | Geomean | Minimum | Maximum | Geomean | Minimum | Maximum | Geomean | Minimum | Maximum | Geomean | Minimum | Maximum |
| pH | --- | 5.81 | 3.56 | 7.25 | 5.48 | 2.63 | 8.13 | 6.02 | 3.56 | 7.84 | 6.48 | 4.74 | 8.10 | 5.94 | 5.14 | 7.56 |
| Arsenic (As) | mg/kg | 0.005 | <0.003 | 0.014 | 0.010 | <0.003 | 0.175 | 0.007 | <0.003 | 0.055 | 0.007 | <0.003 | 0.060 | 0.004 | 0.003 | 0.037 |
| | % | 0.134 | <0.007 | 1.40 | 0.456 | <0.022 | 19.5 | 0.869 | <0.096 | 16.6 | 0.440 | <0.081 | 2.87 | 0.625 | 0.058 | 4.13 |
| Cadmium (Cd) | mg/kg | 0.0003 | <0.0003 | 0.0005 | 0.0003 | <0.0003 | 0.0009 | 0.0007 | <0.0003 | 0.06 | 0.0003 | <0.0003 | 0.0004 | 0.0003 | <0.0003 | 0.0003 |
| | % | 0.3 | <0.03 | 1.54 | 0.8 | <0.03 | 3.34 | 0.8 | <0.04 | 64 | 1.1 | <0.07 | 2.00 | 0.6 | <0.01 | 1.59 |
| Cobalt (Co) | mg/kg | 0.002 | <0.001 | 0.004 | 0.002 | <0.001 | 0.005 | 0.003 | <0.001 | 0.814 | 0.002 | <0.002 | 0.002 | 0.002 | <0.001 | 0.002 |
| | % | 0.048 | <0.005 | 0.514 | 0.024 | <0.001 | 0.089 | 0.033 | <0.006 | 8.14 | 0.014 | <0.003 | 0.067 | 0.011 | <0.005 | 0.019 |
| Copper (Cu) | mg/kg | 0.006 | <0.003 | 0.054 | 0.006 | <0.003 | 0.091 | 0.006 | <0.003 | 2.01 | 0.006 | <0.003 | 0.095 | 0.003 | <0.003 | 0.003 |
| | % | 0.089 | 0.004 | 2.37 | 0.079 | <0.0003 | 4.56 | 0.014 | <0.001 | 6.56 | 0.036 | 0.002 | 0.831 | 0.010 | <0.002 | 0.070 |
| Molybdenum (Mo) | mg/kg | 0.009 | <0.003 | 0.105 | 0.011 | <0.003 | 0.254 | 0.065 | <0.003 | 22.2 | 0.014 | 0.003 | 0.158 | 0.004 | <0.003 | 0.012 |
| | % | 1.19 | 0.207 | 10.5 | 1.26 | <0.119 | 23.8 | 0.294 | <0.006 | 42.8 | 0.255 | 0.003 | 5.17 | 0.518 | <0.115 | 5.48 |
| Nickel (Ni) | mg/kg | 0.007 | <0.006 | 0.021 | 0.006 | <0.006 | 0.015 | 0.009 | <0.006 | 0.694 | 0.006 | <0.006 | 0.008 | 0.006 | <0.006 | 0.006 |
| | % | 0.028 | 0.001 | 0.195 | 0.025 | <0.003 | 0.139 | 0.052 | <0.010 | 4.34 | 0.019 | <0.002 | 0.077 | 0.023 | <0.012 | 0.038 |
| Lead (Pb) | mg/kg | 0.003 | <0.003 | 0.005 | 0.004 | <0.003 | 0.064 | 0.004 | <0.003 | 0.029 | 0.003 | <0.003 | 0.009 | 0.003 | <0.003 | 0.003 |
| | % | 0.034 | <0.007 | 0.117 | 0.066 | <0.016 | 0.464 | 0.011 | <0.002 | 0.083 | 0.019 | 0.003 | 0.093 | 0.031 | <0.003 | 0.119 |
| Antimony (Sb) | mg/kg | 0.015 | <0.014 | 0.015 | 0.015 | <0.015 | 0.015 | 0.017 | <0.014 | 0.147 | 0.015 | <0.015 | 0.015 | 0.015 | <0.015 | 0.016 |
| | % | 3.72 | <0.490 | 30.4 | 6.78 | <0.637 | 30.8 | 6.77 | <1.44 | 30.3 | 15.1 | <3.57 | 30.4 | 11.7 | <6.90 | 21.5 |
| Selenium (Se) | mg/kg | 0.015 | <0.014 | 0.017 | 0.015 | <0.015 | 0.015 | 0.019 | <0.014 | 0.147 | 0.015 | <0.015 | 0.015 | 0.015 | <0.015 | 0.016 |
| | % | 1.16 | <0.245 | 1.72 | 1.32 | <0.499 | 1.55 | 1.40 | <0.217 | 14.7 | 0.963 | <0.512 | 1.54 | 1.40 | <0.744 | 1.59 |
| Uranium (U) | mg/kg | 0.030 | <0.014 | 3.84 | 0.018 | <0.015 | 3.03 | 0.043 | <0.014 | 14.9 | 0.024 | <0.015 | 0.441 | 0.015 | <0.015 | 0.016 |
| | % | 0.194 | <0.006 | 2.15 | 0.475 | <0.051 | 5.49 | 0.249 | <0.045 | 5.62 | 0.218 | <0.024 | 0.882 | 0.360 | <0.167 | 0.793 |
| Zinc (Zn) | mg/kg | 0.015 | <0.009 | 0.135 | 0.014 | <0.009 | 0.070 | 0.013 | <0.009 | 0.343 | 0.009 | <0.009 | 0.011 | 0.009 | <0.009 | 0.010 |
| | % | 0.127 | <0.006 | 0.470 | 0.074 | <0.008 | 0.454 | 0.051 | <0.011 | 1.52 | 0.027 | <0.008 | 0.306 | 0.012 | <0.002 | 0.028 |

**NOTES:**

% = percentage of total constituent inventory leached during SWEP testing

**TABLE 5.2-1: Summary of the Mass of Arsenic Leached From Kiggavik Rock Samples Drilled in 2007**

| Sample ID | Water:Solids Ratio | Residual Solids Content [1] (mg/kg) | Water Leachable[2] (mg/kg) | Total Leachable[2] (mg/kg) | Water Leachable (% of total)[3] | Total Leachable (% of total)[3] |
|---|---|---|---|---|---|---|
| ANDW-07-01 (185-185.5) | 20:1 | 0.4 | < | < | < | < |
| ANDW-07-01 (200-200.5) | 20:1 | 0.9 | < | < | < | < |
| ANDW-07-01 (200-200.5) | 3:1 | 0.9 | < | < | < | < |
| ANDW-07-01 (255-255.5) | 20:1 | 0.3 | < | < | < | < |
| ANDW-07-01 (50-50.5) | 20:1 | 1.3 | < | < | < | < |
| ANDW-07-01 (80-80.5) | 20:1 | 4.2 | 0.09 | 0.09 | 2.1% | 2.1% |
| ANDW-07-01 (80-80.5) | 3:1 | 4.2 | 0.005 | 0.061 | 0.1% | 1.4% |
| ANDW-07-02 (26-26.5) | 20:1 | 3.6 | < | < | < | < |
| ANDW-07-02 (51-51.5) | 20:1 | 6.9 | < | < | < | < |
| ANDW-07-02 (51-51.5) | 3:1 | 6.9 | < | < | < | < |
| ENDG-07-01 (140-140.5) | 20:1 | 2.5 | < | < | < | < |
| ENDG-07-01 (20-20.5) | 20:1 | 1.6 | < | < | < | < |
| ENDG-07-01 (215-215.5) | 20:1 | 0.2 | < | < | < | < |
| ENDG-07-01 (215-215.5) | 3:1 | 0.2 | < | < | < | < |
| ENDG-07-01 (50-50.5) | 20:1 | 0.6 | < | < | < | < |
| MZ-07-01 (20-20.5) | 20:1 | 0.6 | < | < | < | < |
| MZ-07-01 (50-50.5) | 20:1 | 0.6 | < | < | < | < |
| MZ-07-01 (94-94.5) | 20:1 | 0.6 | < | < | < | < |
| MZ-07-01 (94-94.5) | 3:1 | 0.6 | 0.005 | 0.005 | 0.8% | 0.8% |
| MZ-07-01 (105-105.5) | 20:1 | 0.3 | < | < | < | < |
| MZ-07-03 (55-55.5) | 20:1 | 0.7 | < | < | < | < |
| MZ-07-03 (100-100.5) | 20:1 | 0.2 | < | < | < | < |
| MZ-07-03 (170-170.5) | 20:1 | 0.2 | < | < | < | < |
| MZ-07-03 (170-170.5) | 3:1 | 0.2 | < | < | < | < |
| MZ-07-03 (195-195.5) | 20:1 | 1.8 | < | < | < | < |
| MZ-07-04 (20-20.5) | 20:1 | 2.6 | < | < | < | < |
| MZ-07-04 (20-20.5) | 3:1 | 2.6 | 0.017 | 0.017 | 0.6% | 0.6% |
| MZ-07-04 (30-30.5) | 20:1 | 0.6 | < | < | < | < |
| MZ-07-04 (40-40.5) | 20:1 | 0.3 | < | < | < | < |
| MZ-07-05 (31-31.5) | 20:1 | 0.2 | < | < | < | < |
| MZ-07-05 (31-31.5) | 3:1 | 0.2 | 0.003 | 0.003 | 1.5% | 1.5% |
| MZ-07-05 (38-38.5) | 20:1 | 0.2 | < | < | < | < |
| MZ-07-05 (55-55.5) | 20:1 | 0.3 | < | < | < | < |

**Notes:**

1 - Indicates the residual As Content of the sample after leaching based on analysis of remnant solids for 20:1 leaches and difference between original and amount leached for 3:1 leaches.

2 - Values with "less than" symbols (<) indicate that concentrations in the leachate were below detection limits. All values less than detection limits were assigned a value of zero.

**TABLE 6.1-1:** Preliminary Classification of Mine Rock

| Total S Content | NP/AP Ratio | Number of Samples | Percentage of Rock Meeting Criteria |
|---|---|---|---|
| **Main, Centre and East Zone** | | | |
| All Samples | >1 | 64 | 74% |
| (87 Samples) | >2 | 52 | 60% |
| | >4 | 39 | 45% |
| <0.1 | >1 | 36 | 41% |
| (36 Samples) | >2 | 36 | 41% |
| | >4 | 35 | 40% |
| <0.2 | >1 | 46 | 53% |
| (46 Samples) | >2 | 44 | 51% |
| | >4 | 37 | 43% |
| <0.3 | >1 | 55 | 63% |
| (57 Samples) | >2 | 50 | 57% |
| | >4 | 39 | 45% |
| **Andrew Lake** | | | |
| All Samples | >1 | 45 | 96% |
| (47 Samples) | >2 | 44 | 94% |
| | >4 | 36 | 77% |
| <0.1 | >1 | 44 | 94% |
| (45 Samples) | >2 | 43 | 91% |
| | >4 | 36 | 77% |
| <0.2 | >1 | 45 | 96% |
| (46 Samples) | >2 | 44 | 94% |
| | >4 | 36 | 77% |
| <0.3 | >1 | 45 | 96% |
| (47 Samples) | >2 | 44 | 94% |
| | >4 | 36 | 77% |

**Table 7.1-1: Summary of Metal Contents and Acid Base Accounting from the Type 1 and 3 Kiggavik Rock and Andrew Lake Rock Humidity Cell Composite Samples**

| Parameter | Units | 10x Average Crustal Abundance1 | Kiggavik Rock | | | | | | | Andrew Lake Rock | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Type 1 | | | Type 3 | | | | Type 1 | | Type 3 | |
| | | | MZ < 0.02 | MZ 0.02-0.05 | MZ 0.05-0.1 | MZ > 250 | MZ 0.1-0.3 | MZ 0.3-0.5 | MZ > 0.5 | ANDW < 0.02 | ANDW 0.02-0.1 | ANDW > 250 | ANDW > 0.1 |
| | | | Geomean | Geomean | Geomean | Geomean | Geomean | Geomean | Geomean | Geomean | Geomean | Geomean | Geomean |
| Aluminum (Al) | % | 83 | 8.63 | 8.70 | 7.05 | 8.35 | 7.60 | 8.22 | 7.93 | 9.63 | 10.38 | 9.11 | 10.05 |
| Arsenic (As) | mg/kg | 10 | 0.94 | 1.07 | 0.36 | 1.42 | 0.42 | 0.67 | 1.04 | 3.33 | 3.21 | 1.26 | 3.36 |
| Cadmium (Cd) | mg/kg | 1 | 0.05 | 0.04 | 0.04 | 0.07 | 0.04 | 0.07 | 0.05 | 0.08 | 0.10 | 0.05 | 0.27 |
| Chromium (Cr) | mg/kg | 1850 | 48.71 | 47.35 | 22.74 | 55.15 | 28.08 | 37.70 | 57.80 | 96.90 | 70.56 | 42.02 | 172.91 |
| Cobalt (Co) | mg/kg | 290 | 8.82 | 9.14 | 6.32 | 9.27 | 9.72 | 8.63 | 14.18 | 3.76 | 3.68 | 2.40 | 2.98 |
| Copper (Cu) | mg/kg | 750 | 8.47 | 10.56 | 14.43 | 8.60 | 37.42 | 39.59 | 88.15 | 6.78 | 6.64 | 4.88 | 6.85 |
| Manganese (Mn) | mg/kg | 14000 | 137.60 | 117.43 | 292.89 | 184.07 | 227.89 | 272.21 | 331.36 | 70.42 | 84.86 | 81.08 | 74.89 |
| Molybdenum (Mo) | mg/kg | 10 | 1.19 | 2.77 | 25.08 | 14.34 | 9.80 | 20.00 | 19.45 | 0.73 | 0.64 | 0.67 | 1.12 |
| Nickel (Ni) | mg/kg | 1050 | 30.35 | 24.87 | 9.41 | 39.55 | 14.57 | 19.03 | 28.26 | 31.26 | 24.30 | 43.94 | 15.22 |
| Lead (Pb) | mg/kg | 80 | 8.41 | 18.73 | 21.13 | 46.40 | 28.80 | 0.16 | 0.15 | 8.99 | 8.33 | 20.90 | 8.36 |
| Antimony (Sb) | mg/kg | 2 | 0.19 | 0.12 | 0.07 | 0.24 | 0.16 | 30.79 | 33.65 | 0.34 | 0.36 | 0.15 | 0.63 |
| Selenium (Se) | mg/kg | 0.5 | 1.16 | 1.15 | 1.00 | 1.35 | 1.25 | 1.37 | 1.72 | 1.38 | 1.16 | 1.00 | 1.26 |
| Strontium (Sr) | mg/kg | 2600 | 32.88 | 56.93 | 123.09 | 26.65 | 123.68 | 123.89 | 208.07 | 111.37 | 98.41 | 46.66 | 198.44 |
| Uranium (U) | mg/kg | 9.1 | 13.61 | 35.05 | 8.23 | 407.17 | 15.76 | 10.35 | 5.13 | 12.11 | 16.44 | 297.03 | 17.98 |
| Vanadium (V) | mg/kg | 2300 | 56.97 | 62.07 | 27.64 | 124.04 | 30.02 | 30.86 | 60.46 | 43.19 | 64.83 | 106.66 | 31.40 |
| Zinc (Zn) | mg/kg | 800 | 23.91 | 25.69 | 36.49 | 20.80 | 29.56 | 26.34 | 51.87 | 11.87 | 15.80 | 12.32 | 7.49 |
| | | | | | | | | | | | | | |
| Paste pH | --- | --- | 8.01 | 8.01 | 8.55 | 7.95 | 8.61 | 8.69 | 8.83 | 7.42 | 7.60 | 7.40 | 7.69 |
| Neutralization Potential (NP) | kg-CaCO3/t | --- | 7.88 | 7.42 | 15.52 | 9.07 | 10.50 | 17.10 | 17.48 | 5.77 | 6.30 | 6.57 | 4.85 |
| Acid Generating Potential (AP) | kg-CaCO3/t | --- | 0.34 | 0.92 | 2.27 | 2.27 | 5.69 | 11.25 | 27.73 | 0.36 | 0.85 | 0.74 | 4.78 |
| NP/AP | --- | --- | 38.67 | 11.40 | 6.87 | 1.00 | 2.11 | 1.75 | 0.63 | 15.83 | 3.36 | 19.20 | 0.64 |
| Total Sulphur | % | --- | 0.01 | 0.03 | 0.07 | 0.07 | 0.18 | 0.36 | 0.89 | 0.01 | 0.03 | 0.02 | 0.16 |
| Sulphate Sulphur | % | --- | 0.01 | 0.02 | 0.01 | 0.02 | 0.01 | 0.02 | 0.02 | 0.01 | 0.02 | 0.02 | 0.01 |
| Sulphide Sulphur | % | --- | 0.01 | 0.02 | 0.06 | 0.24 | 0.16 | 0.35 | 0.87 | 0.01 | 0.02 | 0.01 | 0.15 |
| Inorganic Carbon (C) | % | --- | 0.05 | 0.05 | 0.09 | 0.08 | 0.07 | 0.16 | 0.11 | 0.05 | 0.06 | 0.06 | 0.05 |
| CO2 | % | --- | 0.20 | 0.21 | 0.33 | 0.33 | 0.28 | 0.57 | 0.41 | 0.20 | 0.23 | 0.23 | 0.20 |

**NOTES:**

1 - From Faure, Gunter. 1998. Principles and Applications of Geochemistry. Prentice Hall. New Jersey.

Reported values are weighted averages based on individual sample results.

**TABLE 7.1-2: Summary of Calculated Laboratory Loading Rates for Kiggavik Rock and Andrew Lake Rock Material according to the Type 1, 2, 3 Criteria**

| Site Source: | | Kiggavik Rock Waste Material | | Andrew Lake Rock Waste Material | |
|---|---|---|---|---|---|
| **COPC** | **Units** | **Type 1 and Type 2 Lab Loading Rate** | **Type 3 Lab Loading Rate** | **Type 1 and Type 2 Lab Loading Rate** | **Type 3 Lab Loading Rate** |
| Sulphate ($SO_4^{2-}$) | mg/kg/wk | 1.89 | 1.93 | 1.98 | 2.71 |
| Aluminum (Al) | mg/kg/wk | 0.018 | 0.038 | 0.019 | 0.049 |
| Arsenic (As) | mg/kg/wk | 0.00021 | 0.00026 | 0.00023 | 0.00025 |
| Cadmium (Cd) | mg/kg/wk | 0.000012 | 0.000018 | 0.000012 | 0.000012 |
| Chromium (Cr) | mg/kg/wk | 0.00015 | 0.00016 | 0.00016 | 0.00019 |
| Cobalt (Co) | mg/kg/wk | 0.00017 | 0.00069 | 0.00012 | 0.00013 |
| Copper (Cu) | mg/kg/wk | 0.0010 | 0.0010 | 0.00066 | 0.00066 |
| Manganese (Mn) | mg/kg/wk | 0.0022 | 0.0047 | 0.00030 | 0.0011 |
| Molybdenum (Mo) | mg/kg/wk | 0.0012 | 0.0249 | 0.00014 | 0.00021 |
| Nickel (Ni) | mg/kg/wk | 0.00077 | 0.00095 | 0.00063 | 0.00065 |
| Lead (Pb) | mg/kg/wk | 0.000073 | 0.00014 | 0.000070 | 0.000087 |
| Antimony (Sb) | mg/kg/wk | 0.00012 | 0.00012 | 0.00012 | 0.00012 |
| Selenium (Se) | mg/kg/wk | 0.00016 | 0.00025 | 0.00012 | 0.00012 |
| Strontium (Sr) | mg/kg/wk | 0.0082 | 0.0261 | 0.0065 | 0.0094 |
| Uranium (U) | mg/kg/wk | 0.0075 | 0.0319 | 0.0010 | 0.0045 |
| Vanadium (V) | mg/kg/wk | 0.0012 | 0.0015 | 0.0012 | 0.0012 |
| Zinc (Zn) | mg/kg/wk | 0.0040 | 0.0038 | 0.0041 | 0.0041 |

**Notes:**

Calculated loading rates are the geomean averages of the loading rate for each humidity cell, according to the rock type criteria

When analytical data is reported as less than detection, the detection limit was used in the average calculation

**Table 7.1-3: Summary of Mass Leached from Field Cells**

| Site Source | | Kiggavik Rock Material | | | | Andrew Lake Rock Material | |
|---|---|---|---|---|---|---|---|
| | | Main Zone Field Cell | | Centre Zone Field Cell | | Andrew Lake Field Cell | |
| Parameter | Units | Average of Sample Events | Cumulative Mass Leached | Average of Sample Events | Cumulative Mass Leached | Average of Sample Events | Cumulative Mass Leached |
| Sulphate ($SO_4$) | mg/kg | 4.45E-01 | 1.34E+00 | 9.02E-01 | 2.71E+00 | 3.85E+00 | 1.15E+01 |
| Aluminum (Al) | mg/kg | 3.36E-03 | 1.68E-02 | 3.79E-02 | 1.89E-01 | 1.70E-01 | 8.51E-01 |
| Arsenic (As) | mg/kg | 1.44E-04 | 7.21E-04 | 2.37E-04 | 1.19E-03 | 4.72E-04 | 2.36E-03 |
| Cadmium (Cd) | mg/kg | 1.28E-05 | 6.41E-05 | 1.10E-04 | 5.52E-04 | 1.60E-04 | 7.99E-04 |
| Chromium (Cr) | mg/kg | 9.50E-06 | 4.75E-05 | 2.22E-05 | 1.11E-04 | 8.12E-06 | 4.06E-05 |
| Cobalt (Co) | mg/kg | 6.11E-05 | 3.06E-04 | 3.53E-04 | 1.76E-03 | 7.64E-04 | 3.82E-03 |
| Copper (Cu) | mg/kg | 4.28E-03 | 2.14E-02 | 7.98E-05 | 3.99E-04 | 1.12E-04 | 5.58E-04 |
| Manganese (Mn) | mg/kg | 2.73E-04 | 1.37E-03 | 9.79E-04 | 4.89E-03 | 4.54E-04 | 2.27E-03 |
| Molybdenum (Mo) | mg/kg | 9.15E-05 | 4.58E-04 | 7.29E-04 | 3.64E-03 | 4.31E-04 | 2.16E-03 |
| Nickel (Ni) | mg/kg | 2.71E-03 | 1.36E-02 | 1.86E-03 | 9.30E-03 | 3.11E-03 | 1.56E-02 |
| Lead (Pb) | mg/kg | 2.81E-02 | 1.41E-01 | 3.57E-03 | 1.78E-02 | 8.13E-04 | 4.07E-03 |
| Antimony (Sb) | mg/kg | 4.23E-04 | 2.12E-03 | 2.72E-04 | 1.36E-03 | 8.84E-04 | 4.42E-03 |
| Selenium (Se) | mg/kg | 6.21E-05 | 3.10E-04 | 1.16E-04 | 5.81E-04 | 1.24E-04 | 6.19E-04 |
| Strontium (Sr) | mg/kg | 1.11E-03 | 5.56E-03 | 6.00E-03 | 3.00E-02 | 3.85E-02 | 1.92E-01 |
| Uranium (U) | mg/kg | 1.37E-04 | 6.85E-04 | 7.94E-03 | 3.97E-02 | 1.06E-02 | 5.32E-02 |
| Vanadium (V) | mg/kg | 1.55E-05 | 7.77E-05 | 2.64E-04 | 1.32E-03 | 2.29E-04 | 1.15E-03 |
| Zinc (Zn) | mg/kg | 2.15E-03 | 1.08E-02 | 3.45E-03 | 1.73E-02 | 5.45E-03 | 2.72E-02 |

**Table 7.2-1: Summary of Calculated Field Loading Rates for Type 1, 2 and 3 Kiggavik Rock and Andrew Lake Rock Material**

| Site Source: | | Type 1 and 2 Kiggavik Rock Material | | Type 3 Kiggavik Rock Material | | Type 1 and 2 Andrew Lake Rock Material | | Type 3 Andrew Lake Rock Material | |
|---|---|---|---|---|---|---|---|---|---|
| COPC | Units | Laboratory Loading Rate | Field Loading Rate[1,3] | Laboratory Loading Rate | Field Loading Rate[1,3] | Laboratory Loading Rate | Field Loading Rate[2,3] | Laboratory Loading Rate | Field Loading Rate[2,3] |
| Sulphate ($SO_4^{2-}$) | mg/kg/wk | 1.89 | 0.011 | 1.93 | 0.011 | 1.98 | 0.046 | 2.71 | 0.063 |
| Aluminum (Al) | mg/kg/wk | 0.018 | 0.000106 | 0.038 | 0.00022 | 0.019 | 0.00043 | 0.049 | 0.00115 |
| Arsenic (As) | mg/kg/wk | 0.00021 | 0.0000012 | 0.00026 | 0.0000015 | 0.00023 | 0.000005 | 0.00025 | 0.00001 |
| Cadmium (Cd) | mg/kg/wk | 0.000012 | 0.00000007 | 0.00002 | 0.0000001 | 0.000012 | 0.0000003 | 0.00001 | 0.0000003 |
| Chromium (Cr) | mg/kg/wk | 0.00015 | 0.0000009 | 0.00016 | 0.0000009 | 0.00016 | 0.000004 | 0.00019 | 0.000004 |
| Cobalt (Co) | mg/kg/wk | 0.00017 | 0.0000010 | 0.00069 | 0.000004 | 0.00012 | 0.000003 | 0.00013 | 0.000003 |
| Copper (Cu) | mg/kg/wk | 0.0010 | 0.000006 | 0.0010 | 0.000006 | 0.00066 | 0.000015 | 0.00066 | 0.00002 |
| Manganese (Mn) | mg/kg/wk | 0.0022 | 0.000013 | 0.0047 | 0.00003 | 0.00030 | 0.000007 | 0.00111 | 0.00003 |
| Molybdenum (Mo) | mg/kg/wk | 0.0012 | 0.000007 | 0.0249 | 0.00015 | 0.00014 | 0.000003 | 0.00021 | 0.000005 |
| Nickel (Ni) | mg/kg/wk | 0.00077 | 0.000004 | 0.00095 | 0.0000056 | 0.00063 | 0.000015 | 0.00065 | 0.00002 |
| Lead (Pb) | mg/kg/wk | 0.000073 | 0.0000004 | 0.00014 | 0.0000008 | 0.000070 | 0.000002 | 0.00009 | 0.000002 |
| Antimony (Sb) | mg/kg/wk | 0.00012 | 0.000001 | 0.00012 | 0.0000007 | 0.00012 | 0.000003 | 0.00012 | 0.000003 |
| Selenium (Se) | mg/kg/wk | 0.00016 | 0.000001 | 0.00025 | 0.0000015 | 0.00012 | 0.000003 | 0.00012 | 0.000003 |
| Strontium (Sr) | mg/kg/wk | 0.0082 | 0.000048 | 0.0261 | 0.00015 | 0.0065 | 0.000153 | 0.0094 | 0.00022 |
| Uranium (U) | mg/kg/wk | 0.0075 | 0.000044 | 0.0319 | 0.0002 | 0.0010 | 0.000022 | 0.0045 | 0.00011 |
| Vanadium (V) | mg/kg/wk | 0.0012 | 0.000007 | 0.0015 | 0.000009 | 0.0012 | 0.000029 | 0.0012 | 0.00003 |
| Zinc (Zn) | mg/kg/wk | 0.0040 | 0.000023 | 0.0038 | 0.000022 | 0.0041 | 0.000095 | 0.0041 | 0.00010 |

**Notes:**

1 - Adjusted for surface area (particle size), such that 5% of stockpile material is similar size to Humidity Cell material

2 - Adjusted for surface area (particle size), such that 20% of stockpile material is similar size to Humidity Cell material

3 - Adjusted for field temperature, such that the rock pile is at 0oC for 8 months and 5oC for 4 months

**Table 7.3-1: Summary of Results for the Initial Flush from the Type 1 and 3 Kiggaivk Rock and Andrew Lake Rock Humidity Cells**

| Parameter | Units | Kiggavik Rock | | | | | | | Andrew Lake Rock | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | Type 1 | | | Type 3 | | | | Type 1 | | Type 3 | |
| | | MZ < 0.02 | MZ 0.02-0.05 | MZ 0.05-0.1 | MZ > 250 | MZ 0.1-0.3 | MZ 0.3-0.5 | MZ > 0.5 | ANDW < 0.02 | ANDW 0.02-0.1 | ANDW > 250 | ANDW > 0.1 |
| Sulphate ($SO_4$) | mg/kg | 8.2 | 35.5 | 7.4 | 125.2 | 24.5 | 72.9 | 38.6 | 17.1 | 68.7 | 7.9 | 22.3 |
| Aluminum (Al) | mg/kg | 0.34 | 0.34 | 0.37 | 0.50 | 0.44 | 0.31 | 0.68 | 0.25 | 0.15 | 0.30 | 0.12 |
| Arsenic (As) | mg/kg | 0.01 | 0.01 | 0.01 | 0.02 | 0.01 | 0.01 | 0.01 | 0.01 | 0.02 | 0.01 | 0.01 |
| Cadmium (Cd) | mg/kg | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.001 | 0.02 | 0.001 | 0.001 | 0.001 | 0.001 |
| Chromium (Cr) | mg/kg | 0.01 | 0.01 | 0.01 | 0.04 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 |
| Cobalt (Co) | mg/kg | 0.005 | 0.01 | 0.005 | 0.07 | 0.03 | 0.06 | 0.02 | 0.005 | 0.01 | 0.005 | 0.005 |
| Copper (Cu) | mg/kg | 0.02 | 0.03 | 0.02 | 0.03 | 0.02 | 0.02 | 0.01 | 0.02 | 0.02 | 0.03 | 0.02 |
| Manganese (Mn) | mg/kg | 0.02 | 0.04 | 0.02 | 0.09 | 0.06 | 0.19 | 0.10 | 0.02 | 0.03 | 0.07 | 0.03 |
| Molybdenum (Mo) | mg/kg | 0.10 | 0.35 | 0.06 | 5.69 | 0.13 | 0.32 | 5.60 | 0.02 | 0.01 | 0.04 | 0.03 |
| Nickel (Ni) | mg/kg | 0.02 | 0.02 | 0.02 | 0.06 | 0.02 | 0.05 | 0.02 | 0.02 | 0.02 | 0.02 | 0.02 |
| Lead (Pb) | mg/kg | 0.01 | 0.01 | 0.01 | 0.02 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.03 | 0.01 |
| Antimony (Sb) | mg/kg | 0.05 | 0.05 | 0.05 | 0.004 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 |
| Selenium (Se) | mg/kg | 0.05 | 0.05 | 0.05 | 0.04 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 | 0.05 |
| Strontium (Sr) | mg/kg | 0.08 | 0.21 | 0.16 | 0.84 | 0.28 | 0.75 | 0.57 | 0.42 | 0.95 | 4.55 | 0.27 |
| Uranium (U) | mg/kg | 0.14 | 0.50 | 0.06 | 18.18 | 0.59 | 0.44 | 0.05 | 0.16 | 0.15 | 0.68 | 0.05 |
| Vanadium (V) | mg/kg | 0.03 | 0.01 | 0.01 | 0.05 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 | 0.01 |
| Zinc (Zn) | mg/kg | 0.05 | 0.07 | 0.04 | 0.05 | 0.04 | 0.05 | 0.04 | 0.05 | 0.08 | 0.14 | 0.09 |

## Table 7.3-2: Radiological Results for the Laboratory Humidity Cell and Column Tests

| Parameter | Radium-226 (Ra-226) | Thorium-230 (Th-230) | Polunium-210 (Po-210) | Lead-210 (Pb-210) |
|---|---|---|---|---|
| Units | Bq/L | Bq/L | Bq/L | Bq/L |
| **Column Tests** | | | | |
| Column ANDW-SW | 24 | 0.014 | 0.07 | 0.82 |
| Column MZ-SW | 2.3 | 0.055 | 0.14 | 0.86 |
| **Laboratory Humidty Cells** | | | | |
| MZ > 250 (Day 0) | 1.07 | 0.03 | 0.04 | <0.1 |
| MZ > 250 (Day 14) | 0.63 | 0.042 | 0.011 | <0.1 |
| MZ > 250 (Day 140) | 0.11 | 0.016 | 0.019 | <0.1 |
| MZ < 0.02 (Day 0) | 0.04 | 0.1 | 0.015 | <0.1 |
| MZ < 0.02 (Day 14) | 0.022 | 0.043 | <0.01 | <0.1 |
| MZ < 0.02 (Day 140) | <0.01 | <0.01 | <0.01 | <0.1 |
| MZ 0.02-0.05 (Day 0) | 0.19 | <0.01 | 0.02 | <0.1 |
| MZ 0.02-0.05 (Day 14) | 0.088 | 0.01 | <0.02 | <0.1 |
| MZ 0.02-0.05 (Day 140) | <0.01 | <0.01 | 0.018 | <0.1 |
| MZ 0.05-0.1 (Day 0) | <0.01 | <0.01 | <0.01 | <0.1 |
| MZ 0.05-0.1 (Day 14) | <0.01 | <0.01 | <0.01 | <0.1 |
| MZ 0.05-0.1 (Day 140) | <0.01 | <0.01 | <0.01 | <0.1 |
| MZ 0.1-0.3 (Day 0) | 0.034 | <0.01 | <0.01 | <0.1 |
| MZ 0.1-0.3 (Day 14) | 0.032 | <0.01 | <0.01 | <0.1 |
| MZ 0.1-0.3 (Day 140) | 0.011 | <0.01 | <0.01 | <0.1 |
| MZ 0.3-0.5 (Day 0) | 0.07 | <0.01 | <0.01 | <0.1 |
| MZ 0.3-0.5 (Day 14) | 0.033 | <0.01 | 0.013 | <0.1 |
| MZ 0.3-0.5 (Day 140) | <0.01 | <0.01 | 0.03 | <0.1 |
| MZ > 0.5 (Day 0) | 0.017 | <0.01 | <0.01 | <0.1 |
| MZ > 0.5 (Day 14) | <0.01 | <0.01 | <0.01 | <0.1 |
| MZ > 0.5 (Day 140) | <0.01 | <0.01 | 0.011 | <0.1 |
| ANDW >250 (Day 0) | 6 | 0.034 | 0.046 | <0.1 |
| ANDW >250 (Day 14) | 1.66 | <0.01 | <0.01 | 0.2 |
| ANDW >250 (Day 140) | 0.028 | 0.011 | 0.011 | <0.1 |
| ANDW < 0.02 (Day 0) | 0.22 | <0.01 | <0.01 | <0.1 |
| ANDW < 0.02 (Day 14) | <0.01 | <0.01 | <0.01 | <0.1 |
| ANDW < 0.02 (Day 140) | <0.01 | <0.01 | <0.01 | <0.1 |
| ANDW 0.02-0.1 (Day 0) | <0.01 | <0.01 | <0.01 | <0.1 |
| ANDW 0.02-0.1 (Day 14) | 0.06 | <0.01 | <0.01 | <0.1 |
| ANDW 0.02-0.1 (Day 140) | <0.01 | <0.01 | <0.01 | <0.1 |
| ANDW > 0.1 (Day 0) | <0.01 | <0.01 | <0.01 | <0.1 |
| ANDW > 0.1 (Day 14) | <0.014 | <0.01 | <0.01 | <0.1 |
| ANDW > 0.1 (Day 140) | <0.01 | <0.01 | <0.01 | <0.1 |

**Table 7.3-3: Summary of Weighted Uranium and Total Sulphur Leached from Field Cells**

| Parameter | Units | Kiggavik Rock Material | | Andrew Lake Rock Material |
|---|---|---|---|---|
| | | MZHC | CZHC | ANDHC |
| Weighted Uranium Content | mg/kg | 17.6 | 30.1 | 17.9 |
| Weighted Total Sulphur Content | % | 0.5 | 0.5 | 0.02 |

## TABLE 7.3-4: Summary of Calculated Loading Rates for Kiggavik Rock and Andrew Lake Rock Column Testing

| Site Source: | | Kiggavik Rock Column | Andrew Lake Rock Column |
|---|---|---|---|
| **COPC** | **Units** | **MZ-SW** | **ANDW-SW** |
| Sulphate ($SO_4^{2-}$) | mg/kg/wk | 11.3 | 10.1 |
| Aluminum (Al) | mg/kg/wk | 0.03 | 0.008 |
| Arsenic (As) | mg/kg/wk | 0.002 | 0.001 |
| Cadmium (Cd) | mg/kg/wk | 0.00003 | 0.00002 |
| Chromium (Cr) | mg/kg/wk | 0.0007 | 0.001 |
| Cobalt (Co) | mg/kg/wk | 0.0001 | 0.0001 |
| Copper (Cu) | mg/kg/wk | 0.0007 | 0.0003 |
| Manganese (Mn) | mg/kg/wk | 0.004 | 0.027 |
| Molybdenum (Mo) | mg/kg/wk | 0.13 | 0.023 |
| Nickel (Ni) | mg/kg/wk | 0.002 | 0.002 |
| Lead (Pb) | mg/kg/wk | 0.0001 | 0.0001 |
| Antimony (Sb) | mg/kg/wk | 0.0001 | 0.0002 |
| Selenium (Se) | mg/kg/wk | 0.006 | 0.002 |
| Strontium (Sr) | mg/kg/wk | 0.04 | 0.67 |
| Uranium (U) | mg/kg/wk | 2.6 | 0.26 |
| Vanadium (V) | mg/kg/wk | 0.001 | 0.003 |
| Zinc (Zn) | mg/kg/wk | 0.001 | 0.001 |

**Notes:**

Calculated loading rates are the averages of the loading rate for each column

When analytical data is reported as less than detection, the detection limit was used in the average calculation

**Table 9.1-1: Summary of the Estimated Concentrations Originating from Type 1 Kiggavik Rock and Andrew Lake Rock Used in Construction**

| Contaminant of Potential Concern | CCME Guideline for Freshwater Aquatic Life (mg/L) | Kiggavik Rock Material | Andrew Lake Rock Material |
|---|---|---|---|
| | | MZ Type 1 Construction Rock Material (mg/L) | ANDW Type 1 Construction Rock Material (mg/L) |
| Sulphate ($SO_4^{2-}$) | NV | 3.76 | 31.42 |
| Aluminum (Al) | 0.005 | 0.036 | 0.296 |
| Arsenic (As) | 0.005 | 0.00042 | 0.00369 |
| Cadmium (Cd) | 0.000017 | 0.000025 | 0.00020 |
| Chromium (Cr) | 0.0089 | 0.00031 | 0.00258 |
| Cobalt (Co) | NV | 0.00034 | 0.00198 |
| Copper (Cu) | 0.002 | 0.0020 | 0.0134 |
| Manganese (Mn) | NV | 0.0043 | 0.0048 |
| Molybdenum (Mo) | 0.073 | 0.0023 | 0.0022 |
| Nickel (Ni) | 0.025 | 0.0015 | 0.0100 |
| Lead (Pb) | 0.001 | 0.00015 | 0.00112 |
| Antimony (Sb) | NV | 0.00025 | 0.00198 |
| Selenium (Se) | 0.001 | 0.00032 | 0.00198 |
| Strontium (Sr) | NV | 0.016 | 0.104 |
| Uranium (U) | 0.015 | 0.015 | 0.015 |
| Vanadium (V) | NV | 0.0025 | 0.0198 |
| Zinc (Zn) | 0.030 | 0.008 | 0.065 |

**Notes:**

NV = No Value

Highlighted values exceed the CCME guideline for the Freshwater Aquatic Life

Table 9.2-1. Predicted Discharge Concentrations from Pointer Lake and Andrew Lake

| Constituent of Potential Concern | CCME Guideline for Freshwater Aquatic Life (mg/L) | Kiggavik[1] | | Sissons[2] | |
|---|---|---|---|---|---|
| | | Field Loading rate (mg/kg/wk) | Pointer lake Discharge (mg/L) | Field Loading rate (mg/kg/wk) | Andrew lake Discharge (mg/L) |
| Sulphate ($SO_4^{2-}$) | NR | 0.011 | 1.98 | 0.046 | 23.59 |
| Aluminum (Al) | 0.005 | 0.00011 | **0.019** | 0.00043 | **0.22** |
| Arsenic (As) | 0.005 | 1.24E-06 | 0.00022 | 5.42E-06 | 0.0028 |
| Cadmium (Cd) | 0.000017 | 7.25E-08 | 0.000013 | 2.91E-07 | **0.00015** |
| Chromium (Cr) | 0.0089 | 8.98E-07 | 0.00016 | 3.80E-06 | 0.0019 |
| Cobalt (Co) | NR | 9.85E-07 | 0.00018 | 2.91E-06 | 0.0015 |
| Copper (Cu) | 0.002 | 5.86E-06 | 0.0011 | 1.54E-05 | **0.0079** |
| Manganese (Mn) | NR | 1.27E-05 | 0.0023 | 7.03E-06 | 0.0036 |
| Molybdenum (Mo) | 0.073 | 6.80E-06 | 0.0012 | 3.23E-06 | 0.0017 |
| Nickel (Ni) | 0.025 | 4.48E-06 | 0.00080 | 1.47E-05 | 0.0075 |
| Lead (Pb) | 0.001 | 4.28E-07 | 0.000077 | 1.64E-06 | 0.0008 |
| Antimony (Sb) | NR | 7.25E-07 | 0.00013 | 2.91E-06 | 0.0015 |
| Selenium (Se) | 0.001 | 9.43E-07 | 0.00017 | 2.91E-06 | **0.0015** |
| Strontium (Sr) | NR | 4.77E-05 | 0.0086 | 1.53E-04 | 0.078 |
| Uranium (U) | 0.015 | 4.37E-05 | 0.0078 | 2.24E-05 | 0.011 |
| Vanadium (V) | NR | 7.25E-06 | 0.0013 | 2.91E-05 | 0.015 |
| Zinc (Zn) | 0.030 | 2.32E-05 | 0.0042 | 9.53E-05 | **0.049** |

**Notes:**

NR - Not Reported: No CCME guideline value for constituent

1 - Estimated quantities of broken Mine Rock include Type 1 and Type 2 materials from the Purpose Built Pit, East Zone, Centre Zone, and Main Zone sources

2 - Estimated quatities of broken Mine Rock include Type 1 and Type 2 materials from the Andrew Lake and End Grid sources

**0.0005** Predicted Concentration exceeds CCME Guideline Value

## Table 10.1-1: Summary of Field Loading Rates for Type 3 Material at Andrew Lake

| Site Source: | | Type 3 Andrew Lake Rock Material | |
|---|---|---|---|
| **COPC** | **Units** | **Laboratory Loading Rate** | **Field Loading Rate[1,2]** |
| Sulphate ($SO_4^{2-}$) | mg/kg/wk | 2.71 | 0.063 |
| Aluminum (Al) | mg/kg/wk | 0.049 | 0.00115 |
| Arsenic (As) | mg/kg/wk | 0.00025 | 5.73E-06 |
| Cadmium (Cd) | mg/kg/wk | 1.25E-05 | 2.91E-07 |
| Chromium (Cr) | mg/kg/wk | 0.00019 | 4.36E-06 |
| Cobalt (Co) | mg/kg/wk | 0.00013 | 3.04E-06 |
| Copper (Cu) | mg/kg/wk | 0.00066 | 1.53E-05 |
| Manganese (Mn) | mg/kg/wk | 0.00111 | 2.60E-05 |
| Molybdenum (Mo) | mg/kg/wk | 0.00021 | 4.80E-06 |
| Nickel (Ni) | mg/kg/wk | 0.00065 | 1.53E-05 |
| Lead (Pb) | mg/kg/wk | 0.00009 | 2.04E-06 |
| Antimony (Sb) | mg/kg/wk | 0.00012 | 2.91E-06 |
| Selenium (Se) | mg/kg/wk | 0.00012 | 2.91E-06 |
| Strontium (Sr) | mg/kg/wk | 0.0094 | 0.00022 |
| Uranium (U) | mg/kg/wk | 0.0045 | 1.05E-04 |
| Vanadium (V) | mg/kg/wk | 0.0012 | 2.91E-05 |
| Zinc (Zn) | mg/kg/wk | 0.0041 | 9.63E-05 |

**Notes:**

1 - Adjusted for surface area (particle size), such that 20% of stockpile material is similar size to Humidity Cell material

2 - Adjusted for field temperature, such that the rock pile is at $0^oC$ for 8 months and $5^oC$ for 4 months

**Table 10.1-2: Summary of Calculated Pore Water Concentrations for Andrew Lake Rock Type 3 Material**

| COPC | Andrew Lake Waste Rock Material | | |
|---|---|---|---|
| | Field Adjusted Loading rates (mg/kg/wk) | Pore Water Concentration (mg/L) | |
| | | Upper Bound Infiltration (150 mm/yr) | Lower Bound Infiltration (50 mm/yr) |
| Sulphate ($SO_4^{2-}$) | 0.063 | 1315 | 3946 |
| Aluminum (Al) | 0.00115 | 24.0 | 71.9 |
| Arsenic (As) | 5.73E-06 | 0.12 | 0.36 |
| Cadmium (Cd) | 2.91E-07 | 0.0061 | 0.0182 |
| Chromium (Cr) | 4.36E-06 | 0.091 | 0.272 |
| Cobalt (Co) | 3.04E-06 | 0.063 | 0.190 |
| Copper (Cu) | 1.53E-05 | 0.32 | 0.96 |
| Manganese (Mn) | 2.60E-05 | 0.54 | 1.62 |
| Molybdenum (Mo) | 4.80E-06 | 0.10 | 0.30 |
| Nickel (Ni) | 1.53E-05 | 0.32 | 0.95 |
| Lead (Pb) | 2.04E-06 | 0.042 | 0.127 |
| Antimony (Sb) | 2.91E-06 | 0.061 | 0.182 |
| Selenium (Se) | 2.91E-06 | 0.061 | 0.182 |
| Strontium (Sr) | 0.00022 | 4.6 | 13.7 |
| Uranium (U) | 1.05E-04 | 2.2 | 6.6 |
| Vanadium (V) | 2.91E-05 | 0.61 | 1.82 |
| Zinc (Zn) | 9.63E-05 | 2.00 | 6.01 |

**Table 10.1-3: Summary of Field Loading Rates for Pit Rubble Material and Pit Walls**

| Site Source: | Andrew Lake Pit Walls and Rubble | | | |
|---|---|---|---|---|
| COPC | Laboratory Loading Rate (mg/kg/wk) | Field Loading Rate for Rubble [1] (mg/kg/wk) | Laboratory Loading Rate [2] ($mg/m^2/wk$) | Field Loading Rate for Walls [1] ($mg/m^2/wk$) |
| Sulphate ($SO_4^{2-}$) | 2.71 | 0.32 | 2.20 | 0.26 |
| Aluminum (Al) | 0.049 | 0.0058 | 0.040 | 0.0047 |
| Arsenic (As) | 0.00025 | 2.86E-05 | 0.00020 | 2.32E-05 |
| Cadmium (Cd) | 1.25E-05 | 1.46E-06 | 1.01E-05 | 1.18E-06 |
| Chromium (Cr) | 0.00019 | 2.18E-05 | 0.00015 | 1.77E-05 |
| Cobalt (Co) | 0.00013 | 1.52E-05 | 0.00011 | 1.23E-05 |
| Copper (Cu) | 0.00066 | 7.66E-05 | 0.00053 | 6.21E-05 |
| Manganese (Mn) | 0.00111 | 1.30E-04 | 0.00090 | 1.06E-04 |
| Molybdenum (Mo) | 0.00021 | 2.40E-05 | 0.00017 | 1.95E-05 |
| Nickel (Ni) | 0.00065 | 7.64E-05 | 0.00053 | 6.20E-05 |
| Lead (Pb) | 0.00009 | 1.02E-05 | 7.09E-05 | 8.28E-06 |
| Antimony (Sb) | 0.00012 | 1.46E-05 | 0.00010 | 1.18E-05 |
| Selenium (Se) | 0.00012 | 1.46E-05 | 0.00010 | 1.18E-05 |
| Strontium (Sr) | 0.0094 | 0.0011 | 0.0076 | 0.00089 |
| Uranium (U) | 0.0045 | 0.00053 | 0.0037 | 0.00043 |
| Vanadium (V) | 0.0012 | 0.00015 | 0.0010 | 0.00012 |
| Zinc (Zn) | 0.0041 | 0.00048 | 0.0033 | 0.00039 |

**Notes:**

1 - Adjusted for field temperature, such that the rock pile is at 0°C for 8 months and 5°C for 4 months

2 - Converted from mg/kg/wk rate to $mg/m^2/wk$ using estimated surface area of material in humidity cells

**Table 10.2-1: Summary of Mass Loads from each Source for Sulphate and Uranium with Different Filling Rates**

| Andrew Lake Loads for Sulphate and Uranium | | | | | | | |
|---|---|---|---|---|---|---|---|
| COPC | Filling Time (a) | Filling Rate ($m^3$/a) | Pit Wall Load (mg) | Pit Rubble Load (mg) | Pore Water Load[1] (mg) | Re-Placed Material Load (mg) | **Total Load (mg)** |
| Sulphate | 10.0 | 4.28E+06 | 1.06E+08 | 8.86E+08 | 2.99E+11 | 0 | 3.00E+11 |
| | 100.1 | 4.28E+05 | 1.06E+09 | 8.86E+09 | 2.99E+11 | 1.49E+10 | 3.24E+11 |
| | 478.3 | 8.96E+04 | 5.06E+09 | 4.23E+10 | 2.99E+11 | 7.10E+10 | 4.17E+11 |
| Uranium | 10.0 | 4.28E+06 | 1.76E+05 | 1.47E+06 | 4.97E+08 | 0 | 4.99E+08 |
| | 100.1 | 4.28E+05 | 1.76E+06 | 1.47E+07 | 4.97E+08 | 2.49E+07 | 5.38E+08 |
| | 478.3 | 8.96E+04 | 8.42E+06 | 7.04E+07 | 4.97E+08 | 1.18E+08 | 6.94E+08 |

Notes:
1 - Pore Water Load calculated using the upper bound infiltration of 150 mm/yr

**Table 10.2-2: Summary of Calculated Andrew Lake Pit Water Concentrations after Flooding**

| **COPC** | Units | CCME Guideline for Protection of Aquatic Life | Andrew Lake Pit Water Concentrations at Filling | Steady-State Pit Water Concentrations from Exposed Pit Walls |
|---|---|---|---|---|
| Sulphate ($SO_4^{2-}$) | mg/L | NR | 9.74 | 0.026 |
| Aluminum (Al) | mg/L | 0.005 | **0.178** | 0.00047 |
| Arsenic (As) | mg/L | 0.005 | 0.00088 | 2.36E-06 |
| Cadmium (Cd) | mg/L | 0.000017 | **0.000045** | 1.20E-07 |
| Chromium (Cr) | mg/L | 0.0089 | 0.00067 | 1.80E-06 |
| Cobalt (Co) | mg/L | NR | 0.00047 | 1.25E-06 |
| Copper (Cu) | mg/L | 0.002 | **0.00236** | 6.31E-06 |
| Manganese (Mn) | mg/L | NR | 0.0040 | 1.07E-05 |
| Molybdenum (Mo) | mg/L | 0.073 | 0.00074 | 1.98E-06 |
| Nickel (Ni) | mg/L | 0.025 | 0.00235 | 6.30E-06 |
| Lead (Pb) | mg/L | 0.001 | 0.00031 | 8.41E-07 |
| Antimony (Sb) | mg/L | NR | 0.00045 | 1.20E-06 |
| Selenium (Se) | mg/L | 0.001 | 0.00045 | 1.20E-06 |
| Strontium (Sr) | mg/L | NR | 0.034 | 9.02E-05 |
| Uranium (U) | mg/L | 0.015 | **0.0162** | 4.33E-05 |
| Vanadium (V) | mg/L | NR | 0.0045 | 1.20E-05 |
| Zinc (Zn) | mg/L | 0.03 | 0.0148 | 3.97E-05 |

**Notes:**

**Highlighted and bold** concentrations exceed the CCME guideline

NR - Not Reported

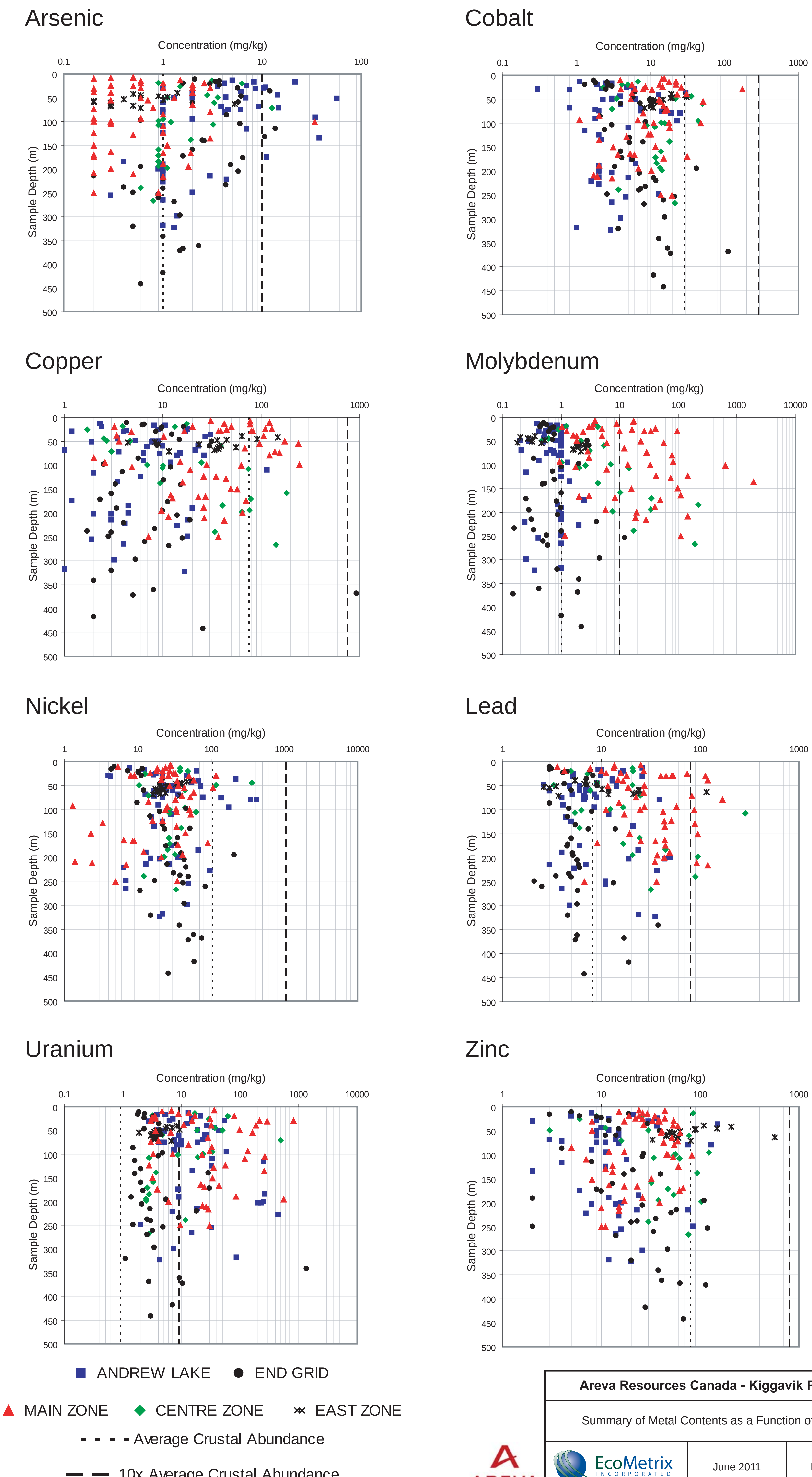
## Figures



**Areva Resources Canada - Kiggavik Project**

Kiggavik Mine Rock Field Humidity Cells

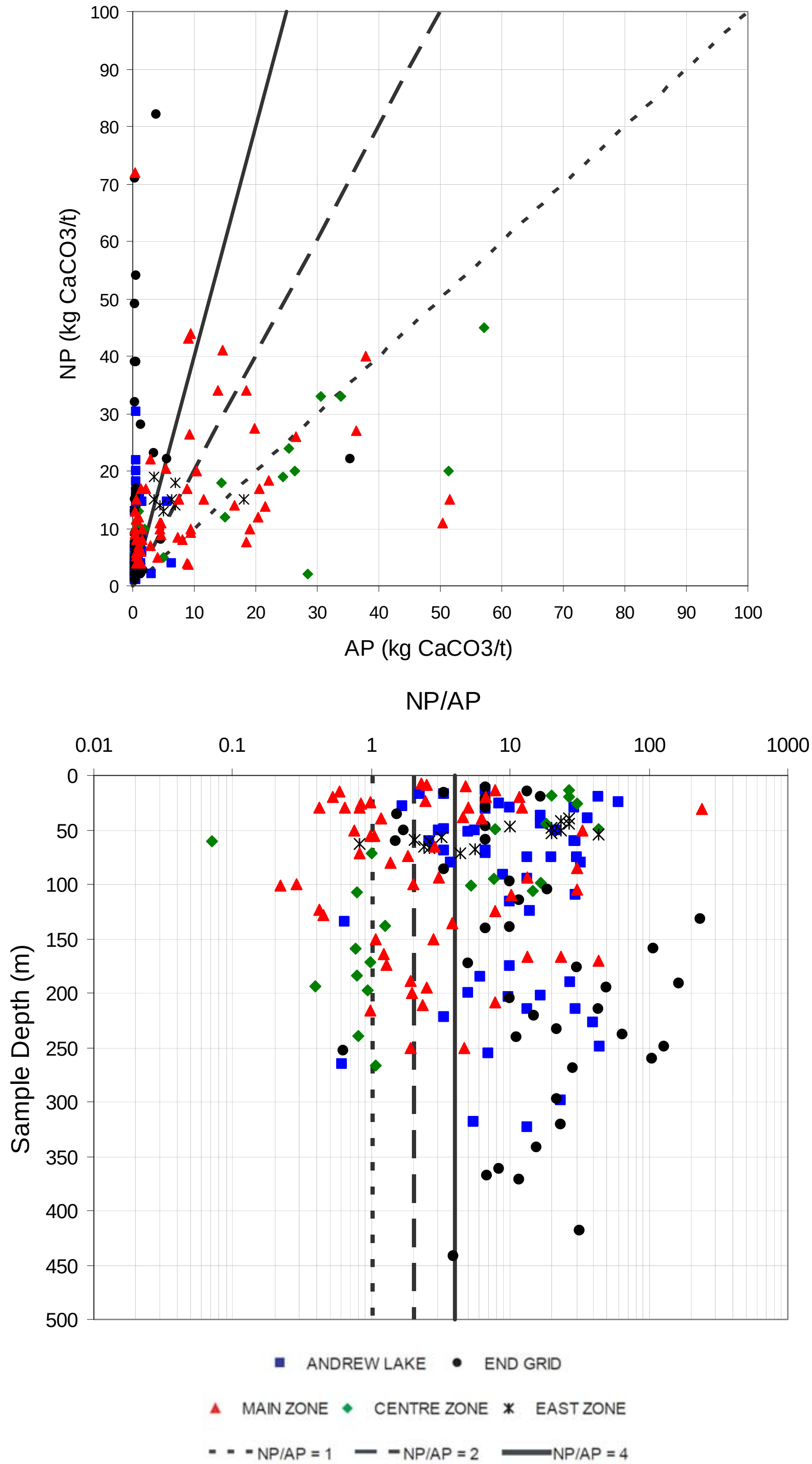
AREVA | EcoMetrix INCORPORATED | June 2011 | **Figure 4.3-1**




**Areva Resources Canada - Kiggavik Project**

Summary of Metal Contents as a Function of Depth

| | |
|---|---|
| June 2011 | **Figure 5.1-1** |

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Mine Rock Characterization and Management
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**Figure 5.1-2**
Summary of ABA Results – Neutralization Potential Ratio

**Kiggavik Project**

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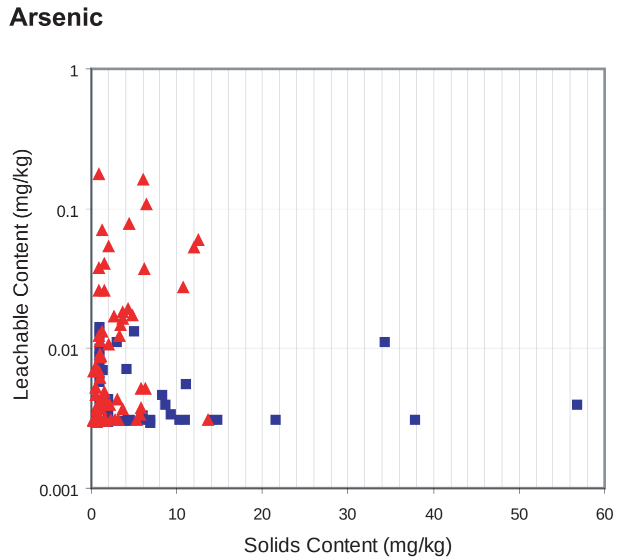
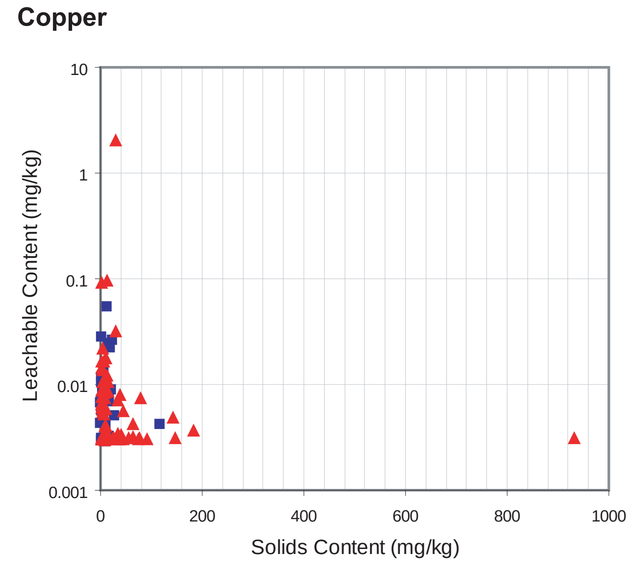
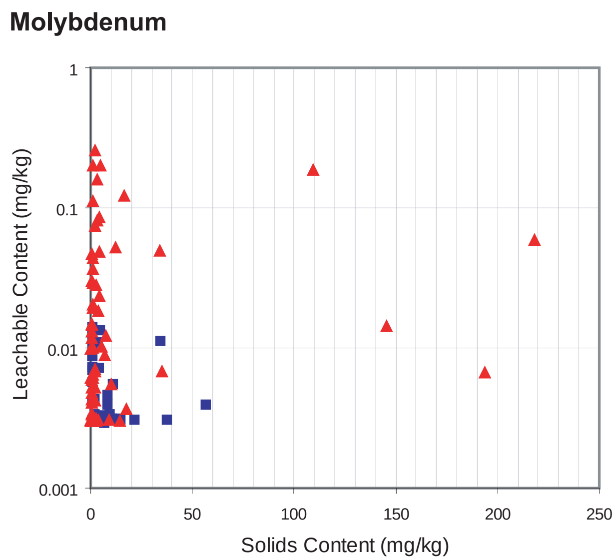
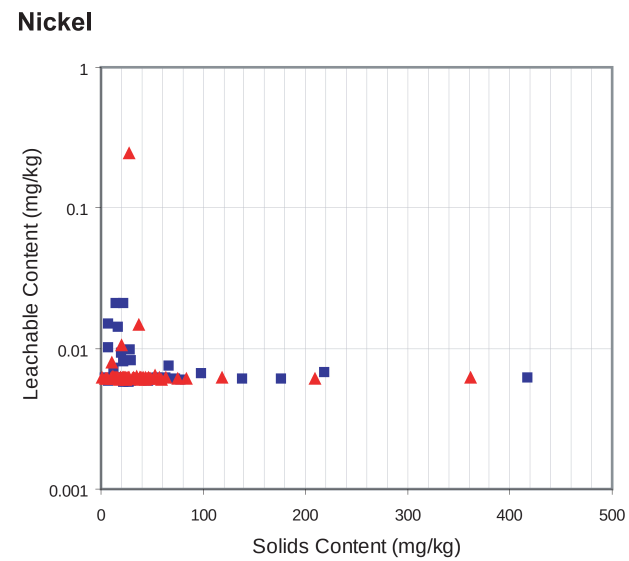


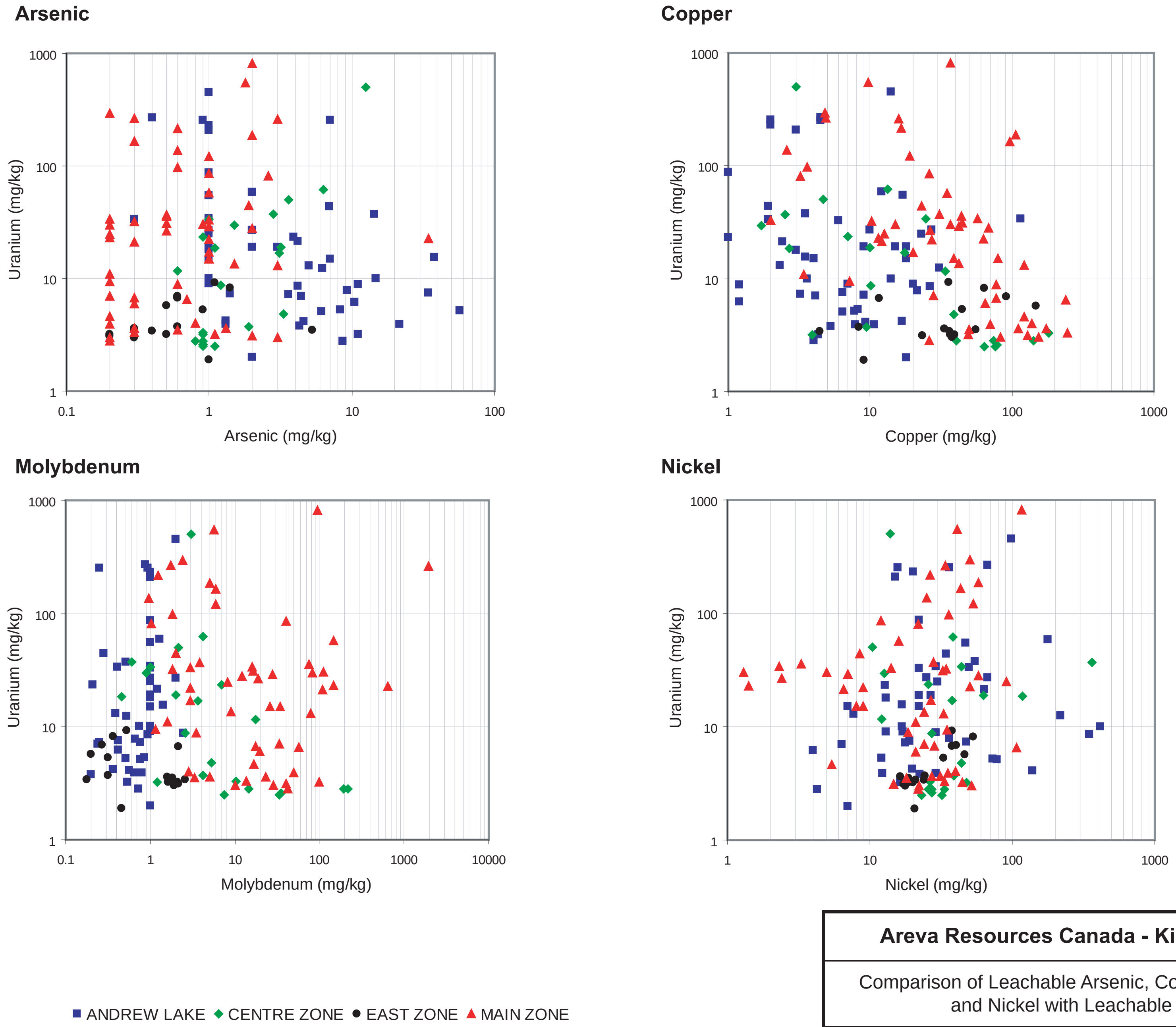




■ Andrew Lake ▲ Main Zone

**Areva Resources Canada - Kiggavik Project**

Comparison of Leachable Arsenic, Copper, Molybdenum and Nickel with Solid Content

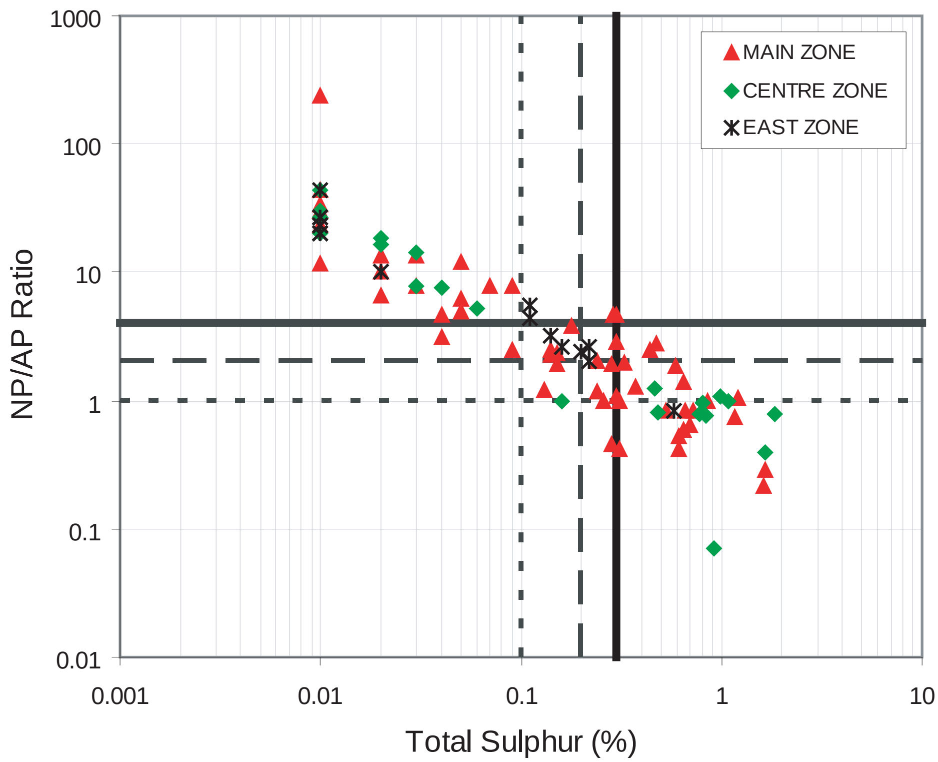
| | | |
|---|---|---|
| EcoMetrix INCORPORATED | June 2011 | **Figure 6.1-1** |

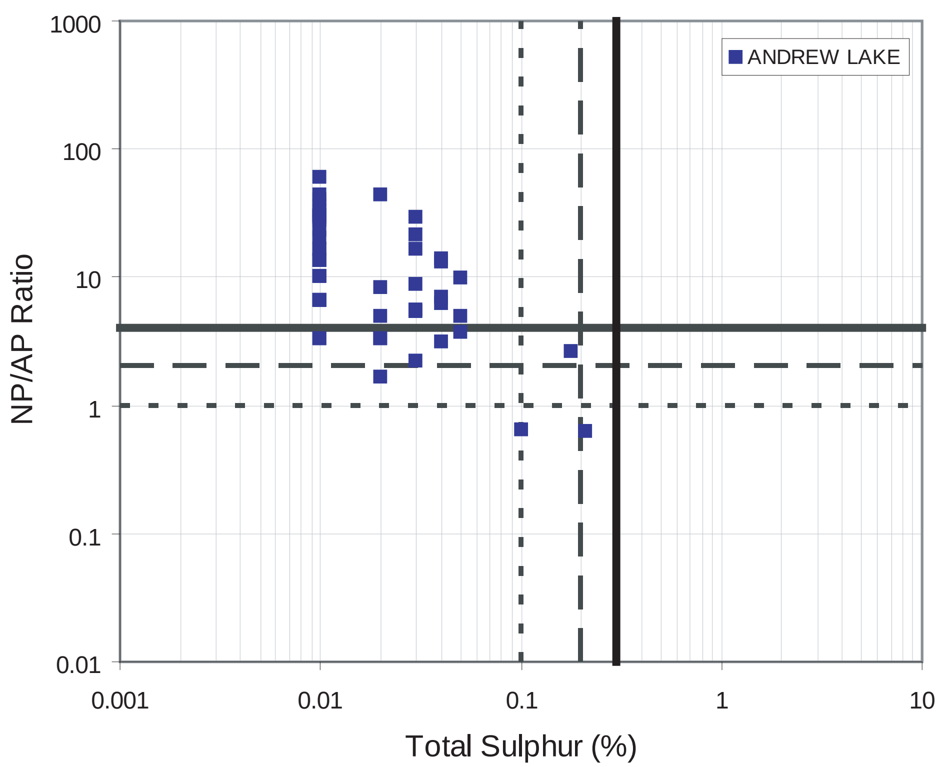



**Areva Resources Canada - Kiggavik Project**

Comparison of Leachable Arsenic, Copper, Molybdenum and Nickel with Leachable Uranium

EcoMetrix INCORPORATED | June 2011 | **Figure 6.1-2**




| Areva Resources Canada - Kiggavik Project | | |
|---|---|---|
| Correlation Between NP/AP Ratio and Total Sulphur Content For Kiggavik Mine Rock | | |
| EcoMetrix INCORPORATED | June 2011 | **Figure 6.1-3** |

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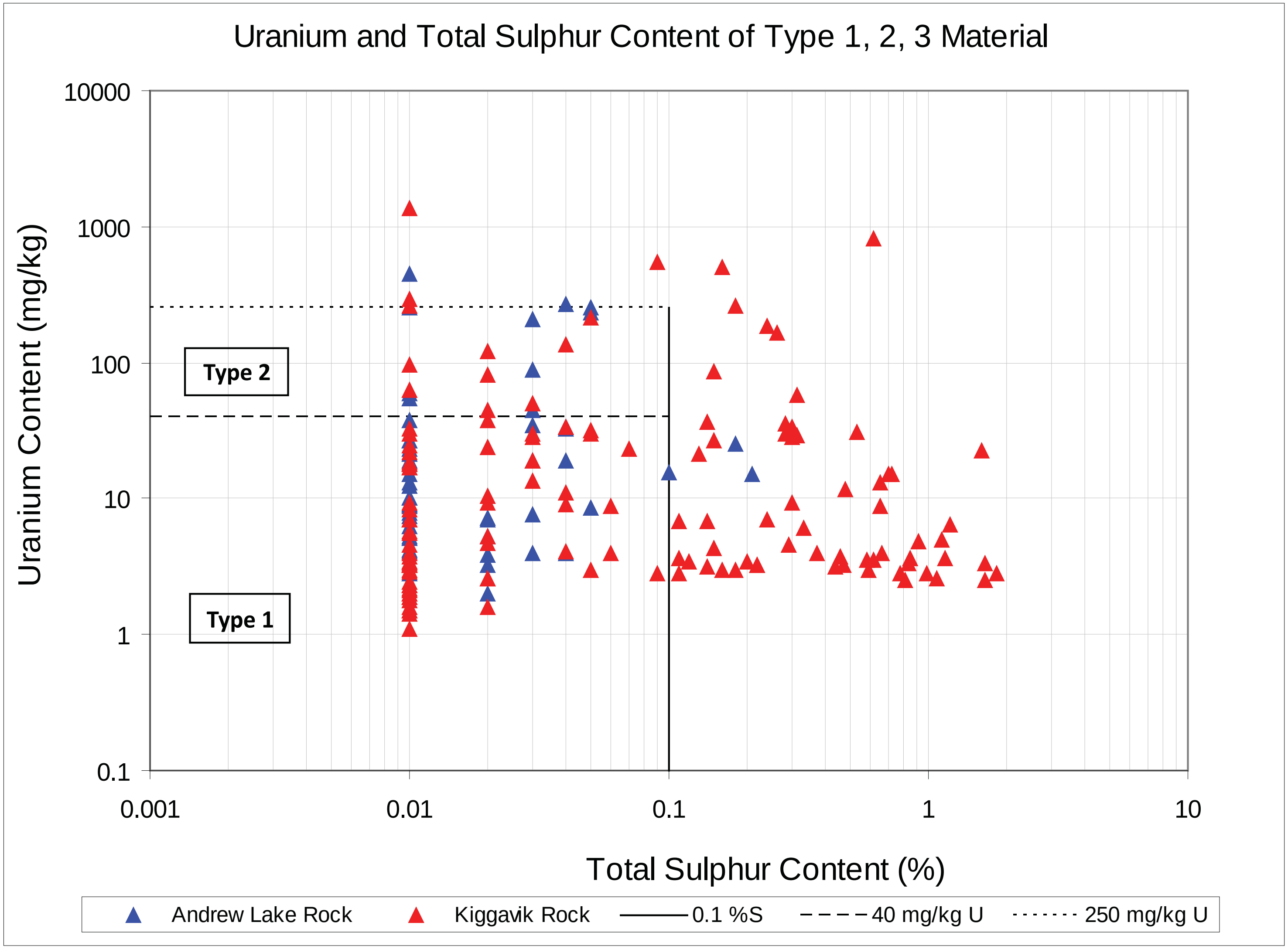


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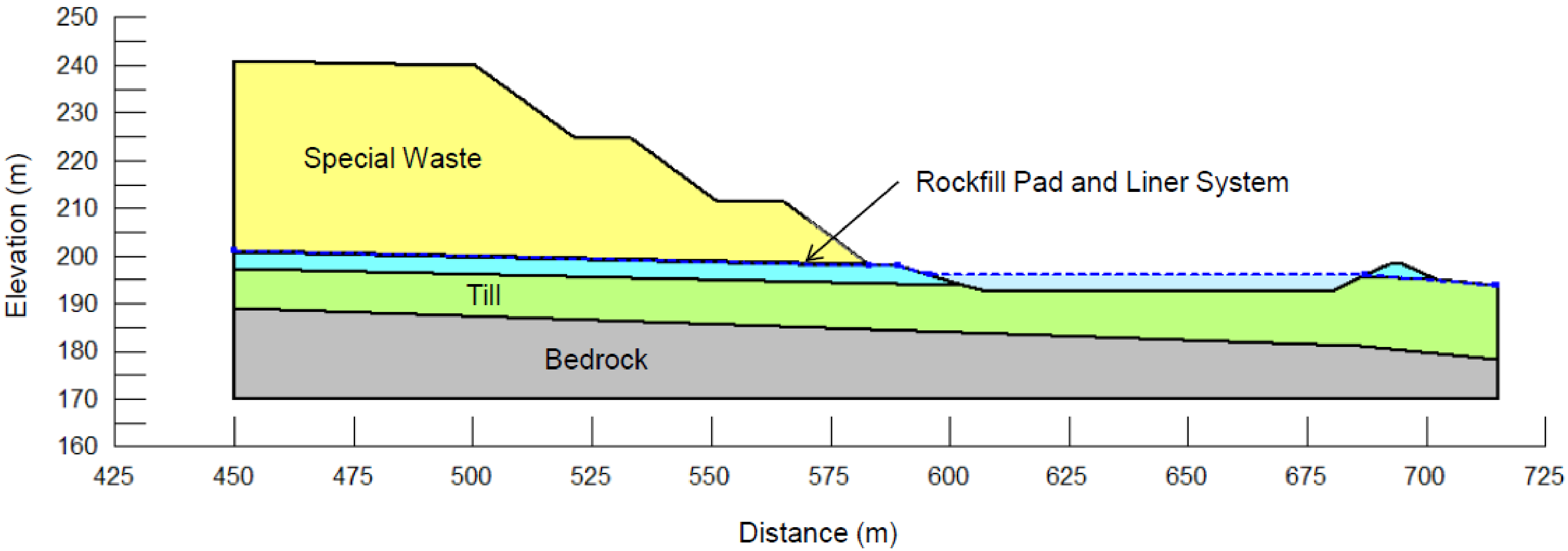
| Areva Resources Canada - Kiggavik Project | | |
|---|---|---|
| Classification of Rock Type | | |
| EcoMetrix INCORPORATED | June 2011 | **Figure 6.1-4** |




| Material | Condition | Unit Weight (kN/m³) | Friction Angle (degrees) | Cohesion (kPa) | |
|---|---|---|---|---|---|
| Ore / Special Waste | unsaturated | 20 | 38 | 0 | |
| Rockfill Pad | saturated | 23 | 38 | 0 | |
| Rockfill Pad | unsaturated | 20 | 38 | 0 | |
| Liner System | | 15 | 17 | 0 | |
| Till | saturated | 22 | 32 | 0 | |
| Bedrock | Impenetrable | | | | |

Golder Associates



**Figure 8.2-1**
Stability analysis section

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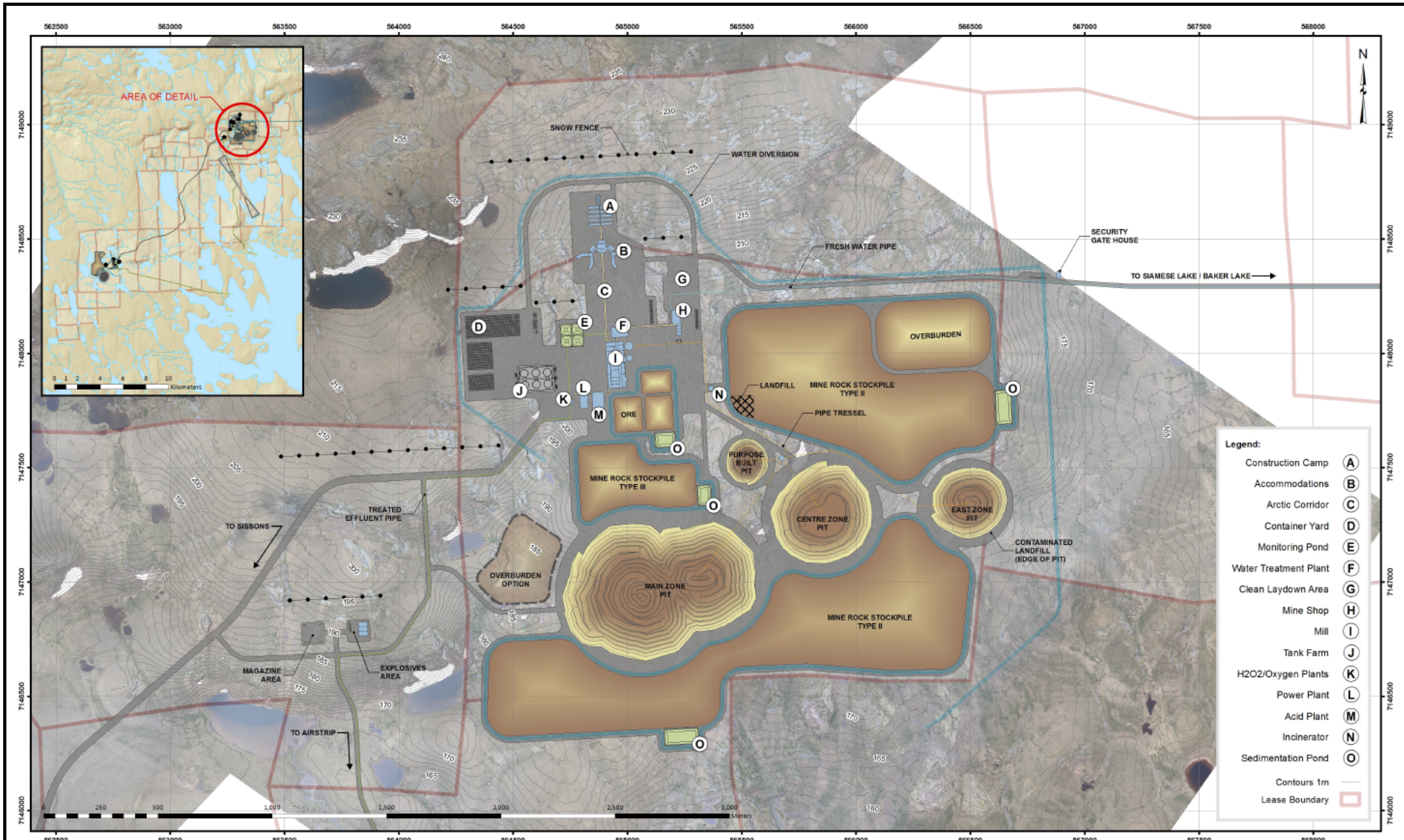


Technical Appendix 5F
Mine Rock Characterization and Assessment
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**Figure 8.3-1**
Kiggavik general site layout

**Kiggavik Project**

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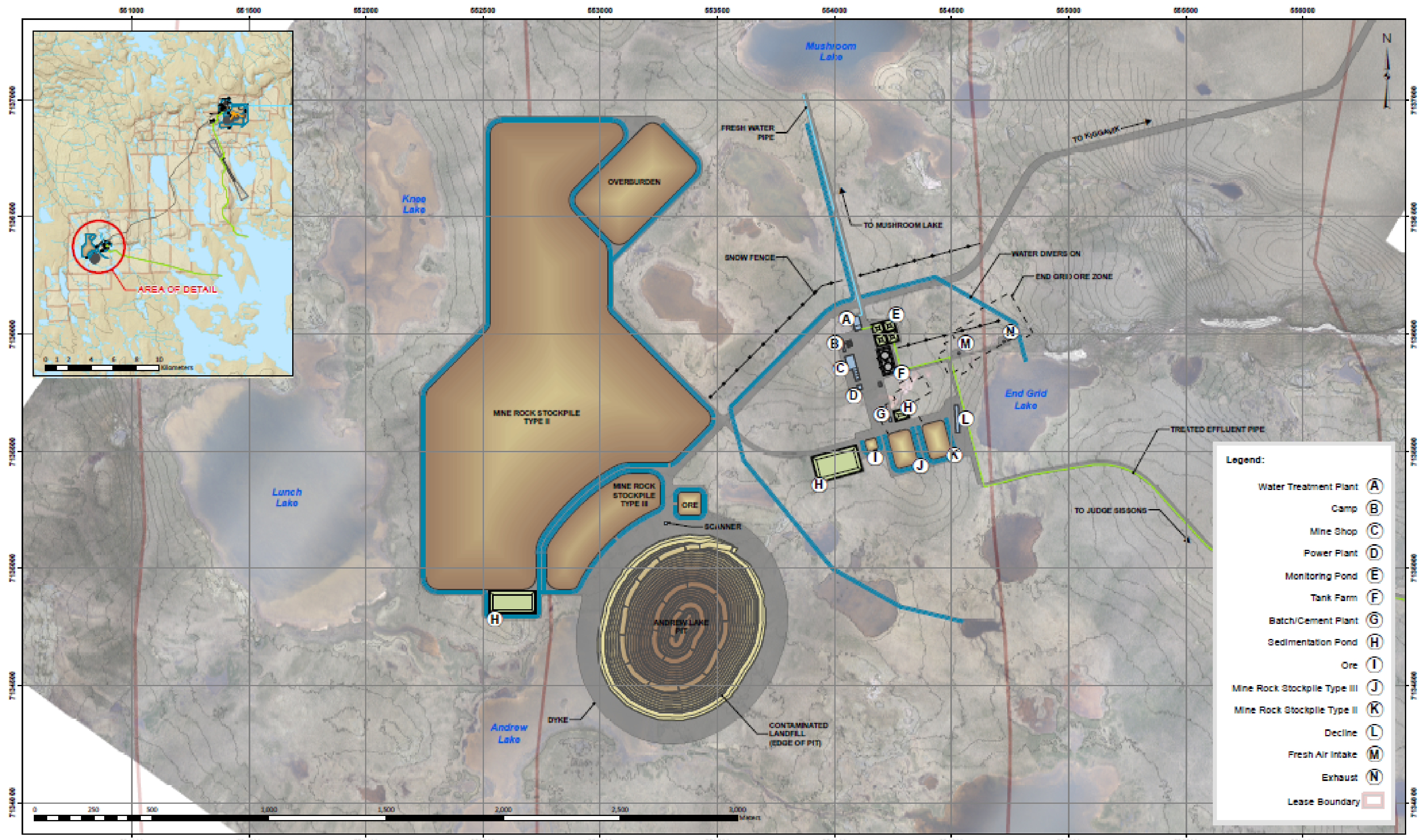


Technical Appendix 5F
Mine Rock Characterization and Assessment
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**Figure 8.3-2**
Sissons general site layout

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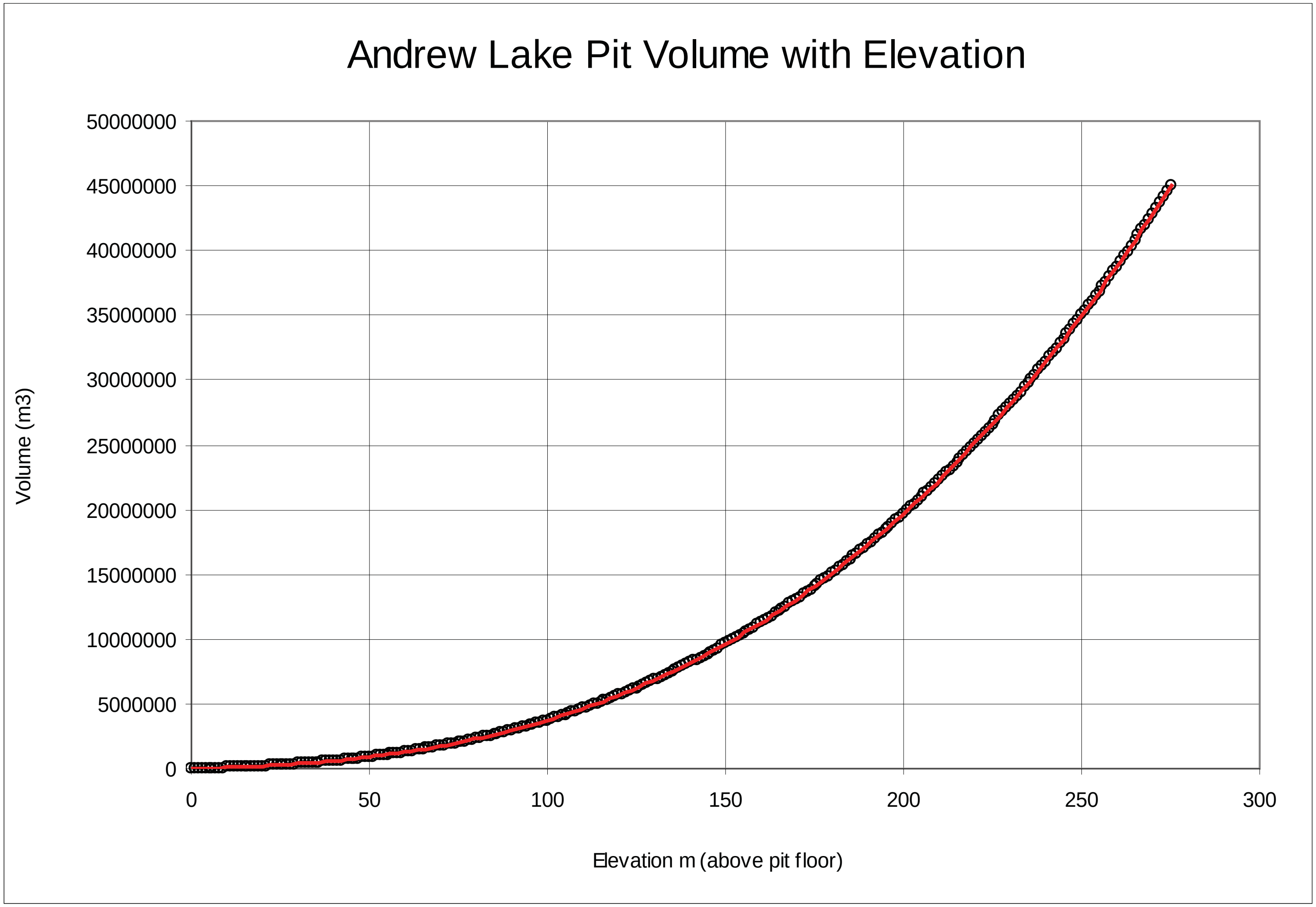
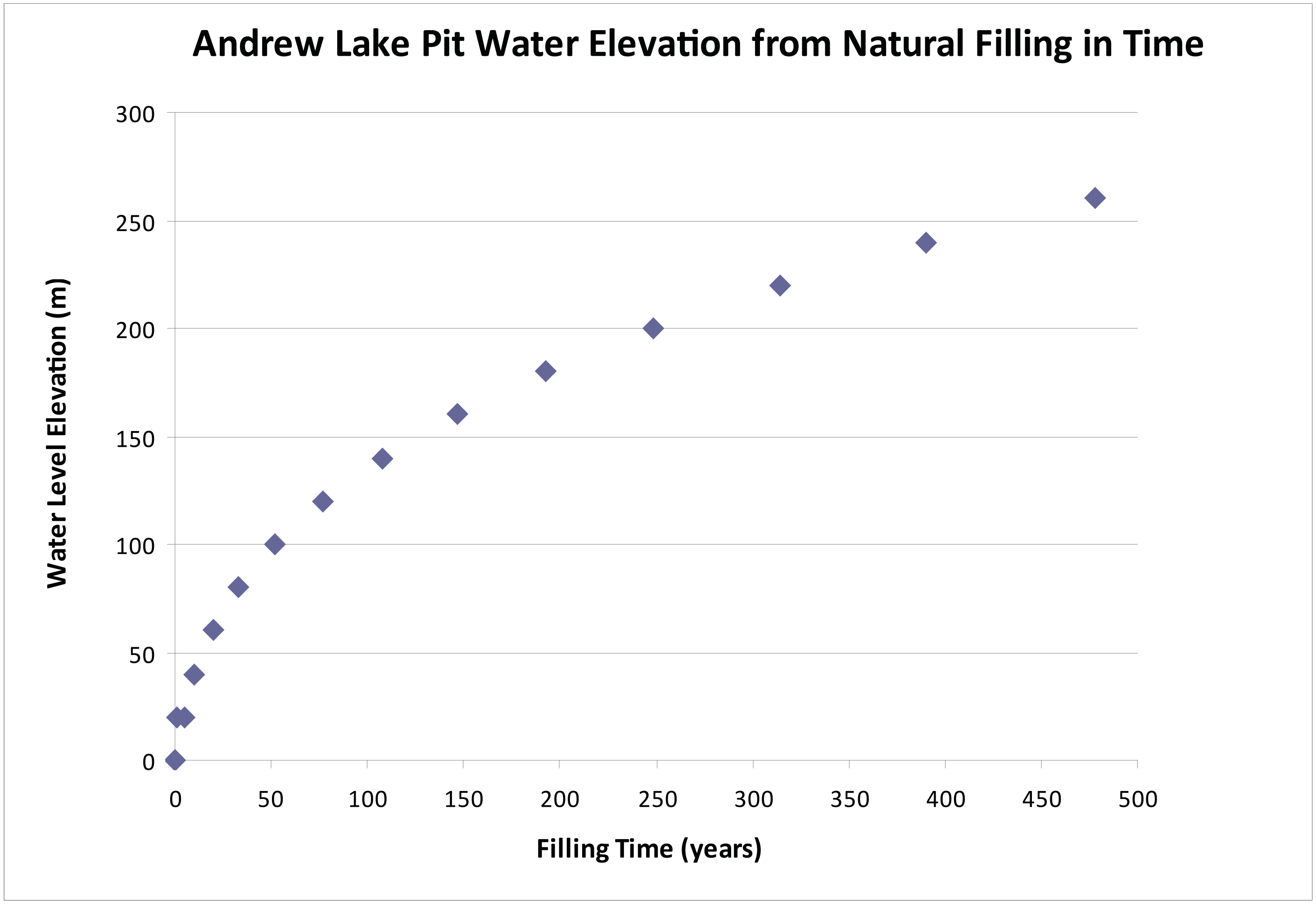



| Areva Resources Canada - Kiggavik Project | | |
|---|---|---|
| Andrew Lake Pit Volume with Elevation | | |
| EcoMetrix INCORPORATED | June 2011 | **Figure 10.1-1** |




**Areva Resources Canada - Kiggavik Project**

Andrew Lake Pit Water Elevation from Natural Filling

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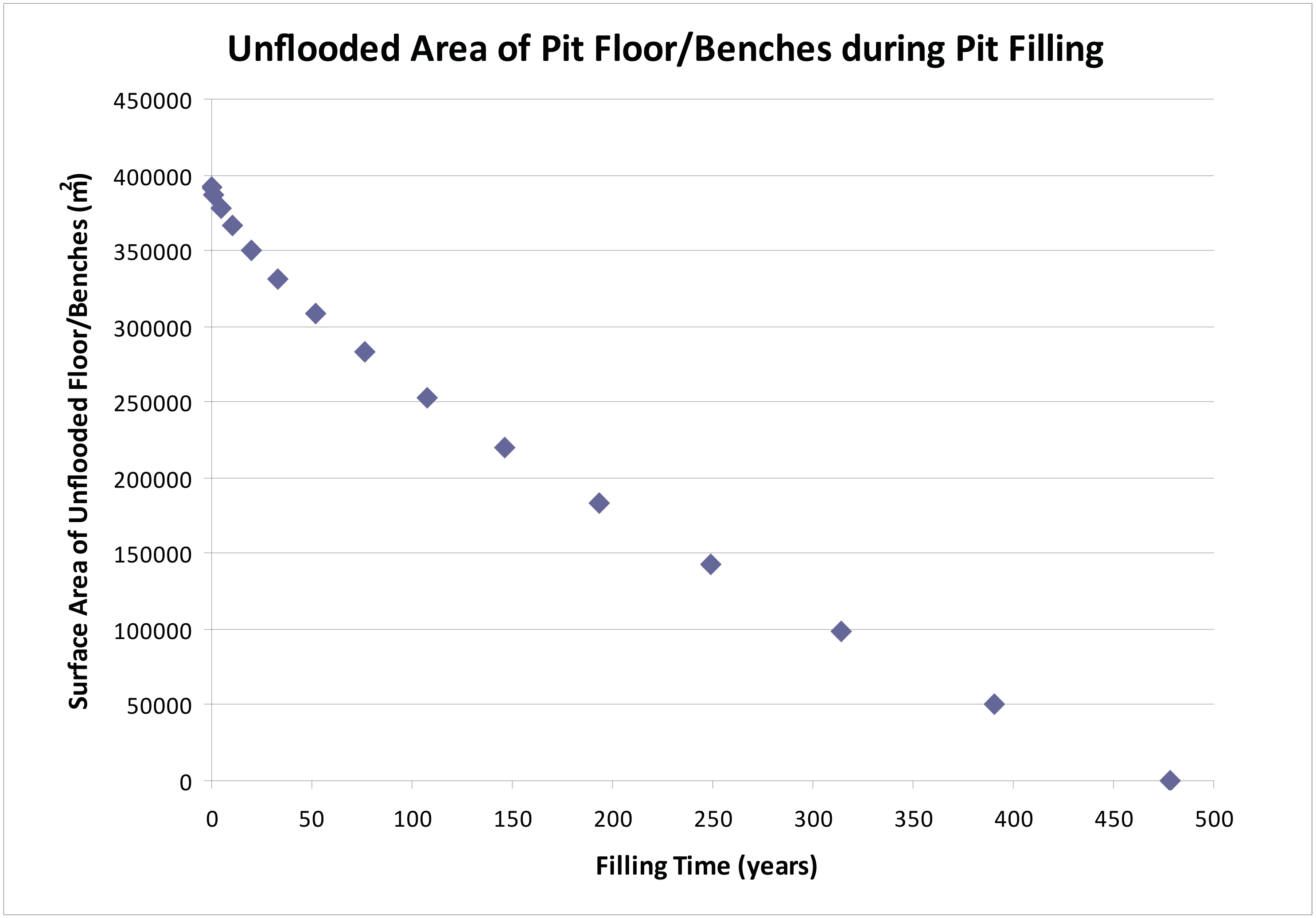
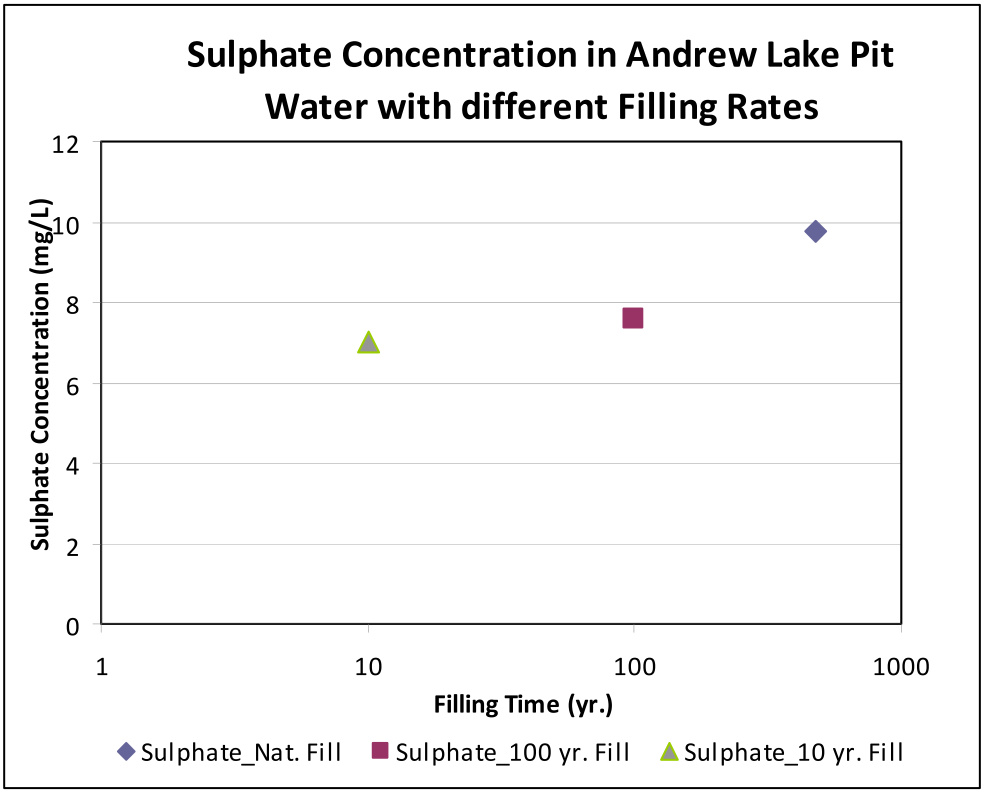
**Figure 10.1-2**




**Areva Resources Canada - Kiggavik Project**

Unflooded Areas of Pit Floor and Benches During Filling

| | | |
|---|---|---|
| EcoMetrix INCORPORATED | June 2011 | **Figure 10.1-3** |



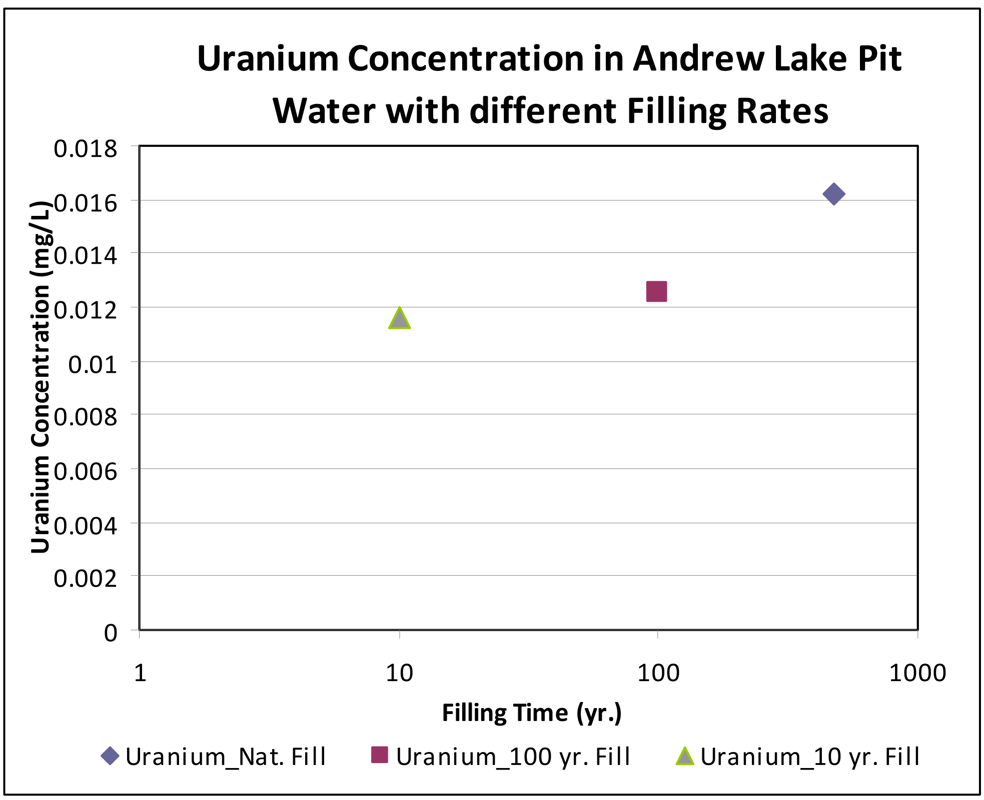






**Figure 10.2-1**
Sulphate and uranium content in Andrew Lake pit waters with different filling rate



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