Spatial and Temporal Variations in the Ruttan Mine Tailings, Leaf Rapids, Manitoba, Canada.

By

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Abstract

The Ruttan Cu-Zn mine produced approximately 30 million tons of fine-grained tailings over 30 years. Since the closure of the mine in 2002, the tailings have been systematically dewatered through trenches which drain into Ruttan Lake and into the open pit and underground mine workings. This study evaluated the evolution of the tailings of Cell 3, which was underwater in 2002, and Cell 2, which was already dry and oxidized.

The mining process at the Ruttan mill removed most of the Zn and Cu sulphides during beneficiation, leaving the tailings dominated by pyrite and pyrrhotite. Iron oxides and oxyhydroxides are formed during oxidation of the Ruttan Tailings. Sequential extraction results show that there is a considerable fraction of metals precipitated in secondary Fe-oxide/hydroxide minerals: Up to 50 wt. % of the total Fe at the top of Cell 2. In Cell 3, the amount of Fe in crystalline oxyhydroxide phases was lower than Cell 2, but there was a notable increase in Fe in this fraction from 11 wt. % in 2004 to 23 wt. % in 2005. Ni and Zn dissolved in stagnant pools on the tailings surface, were found to precipitate with schwertmannite, and Cu in jarosite. Additionally, Fe was found to

Very fine grained material from the milling process, found in fresh, reduced tailings was absent after one year of oxidation and may be the initial cause of mobile metals. During the summer months, evaporative conditions favour the formation of evaporite materials such as melanterite, which temporarily attenuate metals in a thin crust on the surface of the tailings. These evaporites are quickly dissolved during precipitation events

and their metals are re-released into surface and groundwaters. To track the movement of metals in the tailings, water samples were collected from the tailings pore water, groundwater, and adjacent surface waters. In 2004, pore water in Cell 3 had a pH of 4.4 at < 10 cm depth, increasing to 7 by 30 cm. Dissolved constituents decrease between 10 and 30 cm depth, sulphate (36,000 to 1,500 ppm) Fe and Zn from 18,000 and 9,550 ppm respectively to below detection limits. By 2005, pH had dropped to 3.2 above 10 cm. Sulphate concentration was unchanged but Fe had increased to 47,000 ppm. Pore water from the oxidized upper 20 cm of Cell 2 had the lowest pH (1.8) in 2004 and the highest sulphate (230,000 ppm), 65,000 ppm Fe, 1,100 ppm Cu, and 9,800 ppm Zn. By 2005, compositions were unchanged but the acidic front had moved from 120 cm depth to 250 cm. In shallow groundwater in Cell 3, there was a progressive decrease in pH (7 to 2) and an increase in Fe (14 to 5,870 ppm), Zn (5 to 327 ppm) and sulphate (1860 to 32,600 ppm) from 2004 to 2005. Surface pools on the tailings had a constant pH of 3±0.5 and high sulphate (<13,000 ppm), Fe (<2,700 ppm) and Zn (<420 ppm). Ruttan Lake is acidic (pH 2.9) but with a constant composition despite Tailings Cell 3 draining into the lake. Sampling in 2007 comprised surface and groundwater samples that were compositionally similar to those of 2005.

In conclusion, dewatering the submerged tailings in Cell 3 resulted in measureable changes in acidification in pore and shallow groundwater from rapid oxidation of very fine grained sulphides. Depth profiles of the concentration of metals in dissolved and solid fractions, when compared to other studies, suggest that the tailings in Cell 2 and Cell 3 are in an early stage of oxidation and that these tailings will produce low pH, metal laden water for years to come.

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1 Introduction

Mining is the second largest primary resource industry in Manitoba averaging \$2.5 billion in annual production and directly employing approximately 3,500 people (Manitoba Industry, Economic Development, and Mines, 2007). Although mining is a necessary cornerstone of Manitoba's economy, it often has a stressful impact on the surrounding environment. To help lessen the environmental impact that mines have in Manitoba, companies are required to file a closure plan which outlines the aspects of environmental remediation, prior to the start-up of their mine. Depending on the conditions agreed upon between the company and the government, and the year in which the closure plan was drafted, the amount of remediation varies. Often the largest hurdle to overcome when cleaning up a former mine-site, is the problem of the waste created during the beneficiation process.

All mining processes require the beneficiation of millions of tons of 'raw' rock to extract useful materials (e.g. gold, nickel, copper, and iron). Often the processing involves the grinding and crushing of rock into fine-grained material so that the target minerals or elements can be separated from the waste material (mostly silicates and oxides) (e.g. Blowes *et al.*, 2003 b). The waste material is present in two forms: waste rock, and tailings. Waste rock is generated when mining activities require the removal of rock that does not contain the target substance. Waste rock is not processed, instead it is crushed to a size (usually on the decimeter scale) that makes it easy to transport to a waste-rock-dump where it is discarded. The other gangue material, tailings, are produced as part of the ore beneficiation process. Tailings comprise material that has

been crushed and ground to a fine grain size (usually clay to silt size) and has been separated from target substances.

In many mines, the target substances are metals associated with or incorporated within sulfide minerals. One of the most common sulfides associated with ore deposits is pyrite (FeS₂), other common sulfides include chalcopyrite (CuFeS₂), pyrrhotite (Fe_{1-x}S), galena (PbS), arsenopyrite (FeAsS), and sphalerite (ZnS). All of these sulfides are relatively unstable under Earth's oxygenated atmospheric conditions where they combine with water and oxygen to form a suite of other, more stable minerals, namely oxides and hydroxides. The transformation of sulfides to oxides involves a number of chemical reactions and is the process by which acid mine drainage is generated.

Oxidation reactions occur on the surface of the sulphide minerals where they are exposed to the surrounding environment. Thus the amount of exposed surface of the mineral will greatly impact the rate of oxidation of a mineral deposit. This becomes very important when looking at the oxidation of waste rock and even more importantly, tailings. Stromberg and Banwert (1999) suggest that there is a large difference in weathering rates between fine materials (diameters (d) <0.25 mm) and larger waste rock particles (d>0.25 mm) for sulfides, resulting in particles with d<0.25 mm contributing to 80% of mineral dissolution, for a given area. Therefore, although oxidation can take place on the actual ore deposit within the mine itself, particularly in open-pit mining, oxidation of fine-grain tailings deposits is of particular concern worldwide.

In addition to surface area, the relative exposure of the grains to an oxygenated atmosphere also affects oxidation rates. The setting in which the rocks or tailings are placed can have a large impact on oxidation, particularly with respect to water and

hydration. Water affects the rate of oxygen diffusion to the mineral surfaces, and also provides a conduit for the removal and mobility of oxidation products.

1.1 Study Site

The study location is centered at the former Ruttan mine-site. The area is located approximately 950 km north of Winnipeg, 23 km east of the community of Leaf Rapids, Manitoba. The site is accessible on PR 391 from PTH 6 (Figure 1-1).

Figure 1-1. Location plan for the Ruttan Mine site.



From 1973 until the mid 1980's, Sherritt Gordon Mines Limited (SGML) extracted copper and zinc ore from volcaniclastic and siliciclastic sequences of the Rusty Lake Greenstone Belt at the Ruttan Mine. During the mid 1980's, the mine was shut-down due to low grade and low metal prices, however, it was acquired and reopened in 1987 by Hudson Bay Mining and Smelting (HBM&S). HBM&S operated the Ruttan Mine until its final shut-down in 2002 (Stantec Consulting Ltd., 2003).

The ore from the Ruttan Mine is hosted within kilometer scale lenses of a sequence of bimodal volcanic, volcaniclastic and siliciclastic rocks (Barrie *et al.*, 2005). The massive-sulfide lenses are hosted by, and intercalated with tholeitic, felsic, and intermediate volcanic/volcaniclastic rocks (Barrie *et al.*, 2005). Metamorphism of the local formations to upper greenschist and lower amphibolite facies is characterized by the presence of cordierite, almandine, andalusite, sillimanite, biotite, staurolite, anthophylite, and talc (Barrie *et al.*, 2005). The footwall rocks are strongly chloritized with local silicification (Barrie *et al.*, 2005). In the hanging wall, the alteration assemblage is sericite +/- gahnite (Barrie *et al.*, 2005). Other gangue minerals include muscovite, chlorite, pyroxene and quartz as well as minor anhydrite and the carbonate minerals, siderite, dolomite, ankerite, and calcite (Barrie *et al.*, 2005). The primary sulfide phases within the deposit are pyrite, pyrrhotite, chalcopyrite, sphalerite, and galena, with minor tetrahedrite

Flotation refinement of the ore produced over 30 million tons of fine-grained sulfiderich tailings, some of which were submerged in retention ponds. Since mine closure, there has been a systematic dewatering of the tailings ponds to allow for a more stable tailings pile. At the time of this study, 2004 and 2005, much of the surface tailings

remained in a reduced state due to relatively wet conditions. However, parts of the tailings have been exposed to the atmosphere and have been oxidizing for many years (Frank Bloodworth, pers. Comm.).

The Ruttan Mine has operated under numerous regulations that put limits on contaminant levels in discharge from the site into the surrounding watershed. Currently discharge waters in Manitoba must fall within guidelines set by the Canadian Federal Government's Fisheries Act and the Metal Mining Liquid Effluent Regulations.

A number of remedial measures have been taken by SGML and HBM&S to keep the mine-site effluent within regulations. Various combinations of dry lime and lime slurries, created at the onsite mill, were fed into Ruttan Lake and the Brehaut lake inlet to raise the pH of these waters. At the Ruttan Lake outlet, a wooden weir was constructed to control the flow out of Ruttan Lake and aid in the liming process. This was later replaced by a more robust cement weir (Stantec Consulting Ltd., 2005).

In the final stages of remedial activity, during mine-shutdown, HBM&S carried out a number of tasks to help mitigate future environmental concerns: The cement weir at the Ruttan Lake outlet was destroyed and the outlet sealed so that Ruttan Lake now drains into the onsite open pit. The entire onsite watershed has been redirected and now drains into the open pit (Figure 1-2). The watershed surrounding the mine has been altered so that natural waters skirt the site. To reduce the potential for failure of the dams surrounding the tailings there has been a systematic dewatering of the tailings ponds through a network of drainage trenches (Figure 1-2). These trenches were excavated with a backhoe approximately 3 - 4 m into the tailings and 3 m across. The excavated

material was piled along-side the trench (Figure 1-3). As the tailings de-water via the drainage trenches, the amount of material exposed to oxidizing conditions increases.

All onsite infrastructure has been removed and the footprint of all mine buildings has been bulldozed, covered with sand and backfill material, and been re-vegetated.

Surface water drainage at the Ruttan mine site has been engineered to flow towards the on-site open-pit and underground mine works. The mill-pond and part of the waste rock surface water flows directly into the open pit. Water from the tailings ponds is directed into Ruttan Lake, which subsequently flows into the open-pit. There are locations with the tailings and waste rock areas where there are stagnant ponds with very little surface flow.

Since the shutdown of the mine in 2001, HBM&S has carried out all of its environmental obligations and the mine now belongs to the Government of Manitoba, classified as an orphaned/abandoned mine-site. There are no immediate plans for this site in terms of remediation, however, once the open pit fills with onsite water, a treatment plant will have to be erected to treat water as it will then be directed offsite. The open pit was expected to be full by approximately 2040 (Ben Edirmanasinghe, pers. Comm.), however, as of 2007 the pit had already half-filled.

Figure 1-2. Spot elevations within the tailings and waste rock, relative to the elevation of Ruttan Lake. Red arrows indicate onsite surface water flow direction. Note location of drainage trenches within tailings.



Figure 1-3. Drainage trench excavated in tailings Cell 1, 2004.



1.2 Objectives

The tailings at the Ruttan Mine in Manitoba, Canada, offer a unique setting to study the processes of acid mine drainage during the onset of tailings oxidation. The subarcticclimate of this region, coupled with the exposure of reduced tailings to oxidizing conditions set this project apart from similar studies. The dewatering of the Ruttan tailings offered the opportunity to observe potentially acid generating tailings as they enter oxidizing conditions in a natural setting. The objective of this thesis are:

- To observe changes in levels of oxidation within the tailings of Cell 3 over the course of 2004-2005.
- To characterize the partially oxidized tailings in Cell 2 to allow for predictions in the evolution of the tailings in Cell 3.
- To obtain the biomass of microbes in the tailings to assess the level of microbial activity.
- To follow metals as they are liberated from primary sulphide minerals within the tailings and move through tailings pore water, groundwater, and surface waters to reveal buffering and metal attenuation processes that effect the chemical evolution of the tailings.
- To identify the speciation and coordination of metals in ground and surface waters.
- To understand the contribution of the tailings to the chemical composition of the Ruttan Lake, groundwater modeling of the flow from the tailings to trenches will be performed, and surface flow from trenches to Ruttan Lake will be measured.

2 Methods

2.1 Field Methods

Various forms of solid material and water were collected in and adjacent to the tailings impoundment at the Ruttan Mine. The tailings themselves, as well as evaporates and precipitates directly associated with the tailings, were collected as solid samples for various analyses. Water samples were collected from surface ponds within the tailings impoundment, from groundwater collected from tailings piezometers, from various locations around Ruttan lake, and from pore water squeezed directly from solid tailings samples.

2.1.1 Solid Sampling

Sampling locations for solid tailings were chosen as representatives of tailings that had been oxidizing over a long period of time (~ 20 years) versus tailings that are water saturated and thus not oxidized. Tailings Cell 2 was chosen for the location of oxidized representatives as it is the Cell with the lowest groundwater levels and deepest zones of oxidation. Tailings Cell 3 was chosen as the representative of un-oxidized tailings because groundwater levels are close to the surface and the tailings remain virtually un-oxidized. Solid sampling locations are presented in Figure 2-1.

Evaporite material was collected from overhangs in erosional cracks on the tailings surface. Precipitates were collected from the bottom of standing ponds on the tailings surface as well as from samples that were left to precipitate in the lab. Figure 2-1. Ruttan Mine solid sampling locations. The R-TP-# symbols represent testpit locations and R-TH-# symbols represent test-hole locations.



Solid tailings were collected primarily from test-pits (Figure 2-2) and test-holes dug into the tailings. Each test pit was approximately 1 m³, depending on variability between sampling locations. One face of each test pit was 'cleaned' with a shovel to expose a surface in which the stratigraphy could be described, photographed, and sampled. Sampling locations in each test-pit were based on changes in stratigraphy as well as at 10 cm intervals.

Test-holes, were holes in the tailings, completed by hand-auger, next to test-pits, to extract samples at depths below the bottom of each test-pit. Samples from test-holes were collected at 0.5 m intervals below the bottom of the test-pits, to a depth where significant sloughing prevented further collection. The auger used for test-hole sampling

had a 20 cm sampling head, this, coupled with occasional sloughing while auguring, meant that at least 0.5 m between samples was required to collect representative samples. The saturated nature of the tailings in Cell 3 did not allow for the completion of test-holes. Solid samples were collected from Tailings Cells 2 and 3 to represent oxidized and reduced tailings, respectively.

Figure 2-2. Collecting samples for protein analysis from a test-pit in Tailings Cell 3. PVC tubes in the background were pushed into the tailings to collect *in situ* solid samples.



Each of the grab samples obtained from test-pit walls or from the sampling auger, were collected using a new pair of nitrile gloves and placed into a labeled Ziplock[™] bag (Figure 2-2). Samples were immediately put into an ice pack-filled cooler and then frozen at the end of the day. In the case of the microbiological samples, each sample was wrapped in aluminum foil before being placed into the Ziplock[™] bag (Figure 2-2).

Samples for thin section preparation were collected in 4x6x1 cm aluminum boxes by pressing the box into the desired tailings (Figure 2-3). Once in the tailings, the box and the surrounding tailings were scooped out by hand and the tailings in the box were leveled. These boxes were then tightly wrapped in plastic wrap, labeled, and frozen at the end of the day.



Figure 2-3. Aluminum boxes inserted into the tailings to be used for thin section analyses (scale bar is in inches).

Where the tailings in the test-pits allowed, i.e. non-indurated, PVC pipes were tapped into the tailings at different depths. The pipes were 30 cm long with a 7.62 cm I.D., and schedule 40 wall thickness (Figure 2-2). Each pipe was stored up-right to represent *in-situ* intervals within the tailings. Samples were sealed with Ziplock[™] bags and duct tape, or with 7.62 cm end-caps on the PVC pipes. Once collected, samples were placed up-right in icepack-containing coolers and then frozen at the end of the day.

Evaporite material was collected from cracks in the surface of the tailings, placed into 50 ml tubes and frozen at the end of the day.

2.1.2 Water Sampling

Water samples were collected from the surface and ground-water of the Ruttan Mine tailings, as well as a number of locations around Ruttan Lake (Figure 2-4 and Figure

2-5). Surface water was collected from standing pools and flowing trenches along the flow path from Cell 1 through Cell 3 to Ruttan Lake. These samples included colourless and dark red water. Descriptions of sample locations are listed in Appendix I.



Figure 2-4. Surface water sampling locations.

Figure 2-5. Monitoring well locations.



Groundwater samples were obtained from monitoring wells installed within the tailings (refer to Section 2.1.3, for piezometer construction). Groundwater surface elevations were measured. Each well was purged three times its volume with a Solinst® Model 428 Disposable Bailer to allow for a representative groundwater sample. Samples were then extracted using the same dedicated bailer.

Ruttan Lake samples were taken to see the effect of the lake on the attenuation of metals and the dilution of low pH influent from the tailings. Samples from the lake were obtained from locations around its perimeter (Figure 2-4). The specific locations of these samples were based primarily on access.

Water samples were filtered through Nalgene 0.22 µm pore filters using a hand pump and stored in HDPE bottles. Each sample was split into 3 aliquots: One for cation analyses, acidified with 0.01 ml of 5.6 M HNO₃ per 1 ml of sample to prevent the precipitation of dissolved metals; one for iron-speciation determination, acidified with 0.08 ml of 6 M HCL per 1 ml of sample; and one for anion analyses, left unacidified. Dissolved oxygen, Eh, and pH were measured immediately upon sample collection, *insitu* if possible with an AP-62 pH/Eh meter using a Thermo Orion OPR-97-87 Ag / AgCl electrode (for Eh measurements) and an Accumet 13-620-AP50 Ag / AgCl electrode (for pH measurements). Dissolved oxygen was measured with an AP-64 Accumet Portable probe.

A set of groundwater samples were also collected from RBH-2, -4, -5, -6, -7, -10, -11 in 2007.

2.1.3 Geotechnical Field Methods

The *in-situ* hydraulic conductivity of the tailings in Cell 1 and 3 as well as the volume of surface-water flow from Cell 3 to Ruttan Lake was measured. Eight monitoring wells, RBH-2, RBH-3, RBH-4, RBH-5, RBH-6, RBH-7, RBH-8, and RBH-9 were completed to various depths within all three of the tailings cells (Figure 2-5). The wells were constructed with solid 50.8-mm diameter PVC casing, followed by No. 10 factory slot well screen to the borehole base (Figure 2-6). Threaded sections on the PVC, or 50.8 mm diameter PVC connectors, were used to join the solid and screened portions. The annulus was backfilled with clean silica sand from the borehole base to just above the solid/screen join (Figure 2-7), and with granular bentonite from just above the solid/screen join to the surface. Table 2-1 gives the construction parameters for each well. The elevation of each well was surveyed relative to the other wells to determine variations in the level of tailings groundwater.

Figure 2-6. Bottom end of a screened PVC piezometer.



Figure 2-7. Pouring of silica sand used to back-fill annulus of piezometer.



Table 2-1. Monitoring well construction.

Well Number	Solid Length	Screen Length	Height Above Grade
RBH-1	1.63	3.05	1.53
RBH-2	1.52	1.52	1.52
RBH-3	1.36	1.52	0.51
RBH-4	2.70	0.40	0.57
RBH-5	1.09	2.54	0.28
RBH-6	1.47	1.98	0.17
RBH-7	0.76	2.13	0.10
RBH-8	0.13	1.35	0.61
RBH-9	2.13	3.05	0.05

All lengths measured in meters.

In-situ hydraulic conductivity was measured by removing the water from a specific well, measuring the rate of groundwater recharge into the well, and then relating those results with the well dimensions through the Hvorslev calculation (Section 3.4). To obtain an average value of the flow of surface water from the Ruttan tailings into Ruttan Lake, flow measurements were taken at the Tailings Cell 3 outlet. These measurements were taken by calculating the speed of match-sticks in a relatively straight portion of the stream along a measured distance. Measurements to establish cross-sections of the stream depth were taken at 10 cm intervals along lines at the beginning, middle, and end

of the flow path. The cross-sections, coupled with the velocity of the stream, were used to estimate the amount of water flowing through the outlet.

2.2 Lab Methods

Solid samples collected from the Ruttan mine tailings included the tailings and evaporate material associated with the tailings. Upon return from the field each day, these samples were placed in a freezer.

Samples collected for thin section analyses were unwrapped and impregnated with epoxy, before being sent to Vancouver Petrographics to be made into thin sections. To preserve highly soluble mineral phases, water was not used in the creation of the thin sections. Once returned from Vancouver Petrographics, the samples were analyzed under transmitted and reflected light with a Nikon[™] Eclipse E400 microscope, and with a Cambridge Instruments[™] Scanning Electron Microscope (SEM) operated in secondary emission and backscatter emission modes. The SEM was equipped with an Energy Dispersive Spectrometer (EDS), which was used to obtain relative elemental composition of specific mineral phases.

A selection of solid samples was subject to geotechnical analyses including: Water content, specific gravity, and grain-size. Water content was measured by oven drying following ASTM D 2216-90 and also by weighing samples for protein analysis before and after lyophilization. Specific gravity was obtained through procedures outlined in ASTM D 854. Grain-size analyses were attempted using a combination of the sieve and hydrometer techniques from ASTM D 421 and ASTM D 422. A laser particle counter, with 6% NaPO₃ as a defloculent, was also used in an attempt to classify grainsize. The

porosity of the tailings was obtained from the bulk density and particle density using the following relationship:

Porosity = 1- bulk density particle density

Bulk density was calculated based on the weight of a sample of know volume. Particle density was calculated using the method outlined in ASTM D854: Specific gravity of soils. Samples collected in PVC tubes for pore water analyses were split in half while frozen using a table saw equipped with a diamond tipped blade. One half of the core was placed back in the freezer, the remaining half was cut into 10 cm lengths and thawed in a nitrogen-filled glove bag. Once thawed inside the glove bag the sample was placed into the sample chamber of a squeezing apparatus (Figure 2-8) and compressed with a 20 ton, manual hydraulic shop press. To avoid damage to the squeezing apparatus the press was operated at less than 5 tons of pressure. Water collected from the squeezing apparatus was filtered through a 0.2 µm syringe filter and separated into three aliquots, one for anion analysis, one for iron-speciation determination, and one for cation analyses. Samples for cation analyses were acidified with 0.01 ml of 5.6 M HNO₃ per 1 ml of sample. Iron speciation samples were acidified with 0.08 ml of 6 M HCL per 1 ml of sample. Pore water collected for anion analyses were left unacidified.

Crushed bulk samples of Ruttan Mine tailings obtained from test-pits and test-holes were analyzed at the University of Manitoba's Department of Geological Science's geochemistry laboratory. A Varian Liberty 200 Inductively Coupled Plasma Optical Emission Spectrometer (ICP-OES) was used to analyze total concentration of a suite of cations. ICP-OES samples were referenced to ICP standard solutions (SCP Science®).

Bulk samples from specific locations within the tailings underwent a sequential extraction process as outlined in Table 2-2. The sequence of phases for extraction from tailings samples was: Water soluble fraction, adsorbed/exchangeable/carbonate fraction, amorphous Fe oxyhydroxide fraction, and crystalline Fe oxide fraction. The residual sulfide/silicate phase was inferred from comparison of sequential extraction results with the bulk chemistry of that sample.

Evaporite material found on the tailings was primarily made up of yellow and white mineral phases. These phases were manually separated under a binocular microscope. Portions of the evaporite material were ground by hand, placed on a zero-background glass slide and examined from 10° to 60° at 0.1° intervals with a dwell time of 5.0 sec by X-ray diffraction (XRD). XRD analyses were performed using a Philips PW3830/40 X-ray generator with PW1710 X-ray powder diffraction (XRPD) system using MDI Datascan/Jade data collection/processing software. Evaporites were also mounted on aluminum posts and coated with C to be examined by SEM.

All unpreserved water samples in the lab began to form precipitates at the bottom of their containers over the period of a couple months. This material was filtered out, dried under a fume hood, and examined by XRD and SEM.



Figure 2-8. Diagram illustrating the squeezing apparatus used to extract pore water from solid tailings.

s.

Target Phase	Reagent	Method	Reference
Water soluble fraction	Deionized H ₂ O	1.0 g of sample into 20 ml deionized H_2O , vortex, shakeDold 2003	
		for 1 hr. Centrifuge, remove supernatant, preserve with	
Adsorbed/exchangeab	1.0 M CH ₃ COONa	Take the residue from the previous leach and add 20 mL	Hall <i>et al.</i> , 1996
le/carbonate	0.25 M HCI	1.0 M CH ₃ COONa; to pH 5.0 with HCl. Vortex for $5 - 10$ s	
		and place in shaker for 6 h. Centrifuge for 10 mins at	
		2800 rpm and decant supernatant into a labeled test-tube	
		and preserve with 5.6 M HNO ₃ . Rinse residue with 5 ml of	
		water, vortex and centrifuge again: add supernatant rinses	
		to the test-tube. Carry out a second 20 ml 1.0 M	
		CH ₃ COONa leach of the residue, repeating steps 2 and 3	
		and supernatant to the test-tube.	
Amorphous Fe	0.2 M NH ₄ -oxalate	Take the residue from the previous leach and add 20 ml	Dold 2003
oxyhydroxide	0.1 M (NH ₂) ₂ C ₂ O ₄	NH ₄ -oxalate; to pH 3 with 0.1 M oxalic acid. Vortex, shake	
		on horizontal shaker for 1 hr in dark at room temp.	
		Centrifuge the sample for 10 mins, and decant the	
		supernatant into a storage container. Acidify the	
		supernatant with HNO ₃ . Wash the sample with deionized	
		water, shake to mix and centrifuge for 30 mins. Add rinse	
		water to sample.	
Cystalline Fe oxide	1.0 M	Take the residue from the previous leach and add 20 ml of	Hall <i>et al</i> ., 1996
	NH2OH·HCL 25%	1.0 M NH2OH·HCL in 25% CH ₃ COOH, vortex from $5 - 10$	
	CH₃COOH	s. Place in water oven at 90°C for 3 h (tightly capped),	
		vortex every 20 mins. Centrifuge for 10 min. and decant	
		supernatant fluid into a labeled test-tube. Carry out a	
		second 1.0 M NH2OH HCI leach of the residue but heat	
		for only 1.5 h, add supernatant from this step to that of the	
		last, preserve with 5.6 M HNO ₃ and analyse.	

All water samples collected for cation analyses were acidified with 0.01 ml of 5.6 M HNO₃ per 1 ml of sample. Samples collected for iron-speciation were acidified with 0.08 ml of 6 M HCL per 1 ml of sample. Eh and pH measurements of each sample were taken directly after collection just prior to acidification. Cations were measured using the ICP-OES at the University of Manitoba's Department of Geological Science's or at Envirotest[™] Laboratories by Inductively Coupled Plasma Mass Spectrometry (ICP-MS). Anions were analyzed at the University of Manitoba's Department of Microbiology or at Envirotest[™] Laboratories. Analyses at the University of Manitoba used a Dionex[™] ionchromatographer (IC) equipped with an As11 ion pac, using 100 mM NaOH eluent, detecting at a rate of 5 Hz. The anion data from the IC was collected and analyzed using the PeakNet 5.11[™] software package.

A large number of water samples were collected over a period of 6 years from the Ruttan mine-site. These samples represent water from the tailings' surface and Ruttan Lake, as well as tailings ground- and pore-water. In order to show the variations in these data in a meaningful way with time and space, all water data are plotted as a normalized values relative to the average of all surface, ground, and lake water samples. Pore water samples are presented graphically with actual values. Tailings pore water composition differs greatly from all other types of water sampled onsite, therefore, pore water data were not used in the site average calculation. See Table 2-3 and Figure 2-9 for an example of the site normalization calculation.

	Ruttan Average	Ruttan Average	RW-1-03	RW-1-03
	(ppm) ¹	Normalized ²	(September) (ppm) 3	Normalized ⁴
Na	45	1	86	1.91
K	31	1	51	1.65
Mg	230	1	76	0.33
Ca	360	1	570	1.58
Sr	1.7	1	2.5	1.47
Mn	6	1	2.6	0.43
Fe	640	1	200	0.31
Cu	5.6	1	0.46	0.08
Zn	110	1	15	0.14
Cd	0.17	1	0.016	0.09
Al	170	1	40	0.24
CI	17	1	21	1.24
SO4	4200	1	3200	0.76
рН	4.1	1	3.1	0.76

Table 2-3. Calculation of Sile Normalized Value	Table 2-3.	Calculation	of Site	Normalized	Values
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¹Average value of analyte from all tailings surface, groundwater, and Ruttan Lake water samples.

 ² Site average value divided by site average value.
³ Measured analytical value of analyte from sample RW-1-03 September.
⁴ Measured analytical value of RW-1-03 September divided by the site average value.




Iron speciation of water samples was based on the technique described by To *et al.*, 1999. Using this technique, Fe(III) was isolated from the samples preserved with HCl by adding acetohydroxamic acid, an Fe(III) complexing agent. Ferrous iron was removed from the sample by first adding FerroZine ($C_{20}H_{13}N_4S_2O_6Na\cdot H_2O$), an Fe(II) complexing agent, and then filtering out the Fe(II)-FerroZine complex with a C18 cartridge. Samples preserved with HNO₃ were analyzed before iron species separation so that samples preserved with HCl could be diluted to total iron concentrations below 15 mg/L. At these concentrations the Fe(II) and Fe(III) can fully complex with the amount and concentration of the complexing agents used. Once the Fe(III) was isolated and the Fe(II) removed, the samples were sent to the University of Manitoba's Department of Geological Science's geochemistry lab for total iron analyses. The Fe(III) results were then subtracted from the total iron measured in the HNO₃ samples to determine Fe(II).

Speciation of ions within water samples were calculated using the WATEQ4F computer modeling program. The pH of the solution and the redox state, determined from the field Eh and measured species of iron were used in these calculations.

Protein assays were completed at the University of Manitoba's Microbiology Department on tailings material using a BCA-200 Protein Assay Kit from Pierce™ Chemical Company. For this procedure 6 ml of TRIS buffer were added to approximately 5 grams of lyophilized tailings. The percentage of moisture was later used to correctthis value. Each sample was sonicated for 3 min at 195 W, 20 kHz. The samples were then centrifuged for 10 min at 8°C and 10 000 RPM. A reagent was added to the supernatant of each centrifuged sample and a set of standards was made with bovine serum albumin. A colourimetric assay was then used to measure

absorbance at 592 nm. The measured absorbance of the standards was used to create a standard to curve to which the samples were compared and protein concentrations calculated.

3 Results

3.1 Site Observations

As dewatering of Ruttan Tailings Cells 1 and 3 progressed, the area of reduced tailings exposed to the atmosphere increased. The purpose for visiting the site in 2004 and 2005 was to observe changes in the tailings as they were exposed and progressive oxidation took place. However, rainfall in this region between April and October 2005 was over 100% greater than average (Environment Canada; Figure 3-1) which resulted in very wet conditions in 2005 and minimized the effects of dewatering.

Figure 3-1. Average monthly precipitation for 2003, 2004, and 2005 measured at the Thompson airport, Thompson, Manitoba (data obtained from Environment Canada). The overall average is calculated from historic Environment Canada data, obtained at the Thompson.



In June 2004, a sample was taken from the trench in Tailings Cell 3 during a precipitation event (RW-2 Rain). A second sample was then taken at the same location 24 hours after the precipitation ceased, to determine the effect of precipitation (RW-2 After Rain). Transition metal and Mg concentrations were higher during the precipitation event, while alkali elements (Na, K, Ca, and Sr), were higher after precipitation ceased. Sulphate concentrations decreased slightly after precipitation and pH remained constant around 3.4 (Figure 3-2). Without a change in pH, calculated element speciation was the same during and after precipitation (Appendix XII).

Figure 3-2. Water composition of trench location RW-2-04 during and 24 hours after a rain event: Normalized to the site average.



RW-2-04 Precipitation Effect

Despite the wet conditions encountered in 2005, increases in the level of oxidation in Tailings Cells 1 and 3 were observed. The most notable visible changes in oxidation were observed between the surface and 15 cm depth (Figure 3-3). In a preliminary visit in 2003 and fieldwork of 2004, oxidation in Cell 1 was limited to just the top centimeter of the tailings and was observed to be patchy, usually constrained to areas of higher

elevation (Figure 3-3 A). In 2005, the entire surface of Cell 1 was oxidized (Figure 3-3 B); with, the oxidized layer being up to 10 cm thick in areas of higher elevation, compared with the 1cm thick oxidized layer in the zones at lower elevations. A layer of white evaporite minerals, that covered the surface of the tailings in 2004 (Figure 3-4), were not present in 2005.

Figure 3-3. A. A freshly dug drainage trench in Tailings Cell 1, July, 2003. The surface has patchy oxidation alteration and abundant evaporite formation. B. The same trench in August, 2005. Note the lack of surface evaporites due to frequent precipitation events, a uniformly oxidized surface, and increased water levels.



In 2004, oxidation in Cell 3 was restricted to 1 to 2 cm halos along vertical and horizontal cracks up to 20 cm depth (Figure 3-5 A). Below this, the tailings were un-oxidized. In 2005, the halos of oxidation went much deeper into the tailings, almost completely oxidizing the first 10 to 15 cm (Figure 3-5 B). Below this depth, the tailings was as unoxidized as in 2004.

Water outflow from the tailings into Ruttan Lake greatly increased between 2004 and 2005 as large standing ponds on the tailings surface overflowed and poured into the tailings trenches. The outflow from the tailings into Ruttan Lake was estimated at 30 Ls⁻¹ in 2004. Outflow from the tailings was not measured in 2005, due to the lack of a

suitable stable measuring location because of much higher water volumes leaving the tailings.

Figure 3-4. White evaporites located on the dry surface of Tailings Cell 1 in July 2003. Note the Ruttan mill in the distance for scale.



Figure 3-5.

(A) A depth profile of the upper 12 cm of the tailings in Cell 3, 2004. Oxidation is taking place along cracks and coarser grained layers within the tailings.

(B) The upper 25 cm of the tailings in Cell 3, 2005. Oxidation is more pervasive extending out from cracks and coarser grained layers. The white scale-bars represent 3 cm.



3.2 Mineralogy

Mineralogy is the starting point of any mine waste investigation. The mineralogy of the ore deposit determines the composition of the tailings and the chemistry of waters in contact within the mine site. In this chapter there is a description of primary minerals in the ore deposit, secondary minerals from the weathering of primary minerals, evaporite minerals from the evaporation of mine-waste water, and precipitates from saturated mine-waste water.

Thin section samples of reduced tailings were obtained from Cell 3 in 2004. Mineral estimations were based on five samples taken in one test pit. It would be expected that some variation in mineralogy within the tailings would occur as the mill processed different areas of the ore deposit. Through optical microscopy of these thin sections, the reduced tailings in Cell 3 were found to contain 20 to 30 % sulfides and 70 to 80 % gangue and very fine-grained unidentifiable material (Figure 3-6). In these tailings, the sulfides were found to comprise 50 to 75 % pyrite, 30 to 40 % pyrrhotite, 1 to 5 % chalcopyrite, and 1 to 5 % sphalerite. The average grain diameter for the sulfides is approximately 0.02 mm. Sulfide grains are angular showing slight signs of erosion (0.1 μ m deep) and have thin (0.2 μ m), dark brown rims (Figure 3-7).

Figure 3-6. Composition of the reduced tailings as estimated from optical microscopy.



Reduced Tailings - Optical Microscopy

Figure 3-7. Photomicrograph of reduced tailings showing a very thin rim of Feoxyhydroxides around a pyrite grain (grain boundary outlined in grey). Viewed under plane polarized transmitted light.



Of the gangue material visible in thin section, 70 to 75 % is dark-gray to black, opaque, and very fine grained creating a 'groundmass' or matrix for the sulfides and gangue minerals (Figure 3-8). From EDS analysis this material was found to comprise a mixture of fine-grained iron sulfides and silicate material (Appendix II). Other gangue material includes 15 to 20 %, equant, 0.1 mm, quartz grains, 5 to 10 % elongate 1:5, 0.2 mm long, muscovite and biotite grains, trace to 5 % equant, 0.1 mm, chlorite grains, and trace to 5 % equant, 0.1 mm, pyroxene grains. Gangue minerals do not show signs of secondary tailings alteration, however, they are discontinuously coated with dark-brown precipitates. Carbonate minerals were not observed in thin sections of the tailings by optical or electron microscopy despite being listed in the original ore mineralogy.

Alteration within the oxidized zone of the tailings in Cell 2, obtained in 2004, appears to be limited primarily to pyrrhotite with minor alteration of sphalerite. Pyrrhotite alteration occurs as thick zoned-rims of Fe-oxy-hydroxides completely surrounding pyrrhotite grains (Figure 3-9), with the original grain boundaries of pyrrhotite becoming indiscernible. Alteration of sphalerite grains is less pronounced, characterized by thin (1 or 2 μ m), often discontinuous, rims of Fe-oxy-hydroxide (Figure 3-10). Zn was found by EDS to be absent from the sphalerite rims (Appendix II).

Figure 3-8. Reflected light photomicrograph showing fine grained groundmass in grey.



Figure 3-9. SEM backscatter image of a corroded pyrrhotite (po) grain with a zoned rim of precipitated secondary minerals (Fe oxy-hydroxides). Pyrite (py) is noticeably less altered than pyrrhotite.



Pyrite remains relatively unaltered, with rims of Fe-oxyhydroxide precipitates. Chalcopyrite was only found disseminated within silicate material, and therefore sheltered from oxidation processes (Figure 3-11). The fine-grained matrix or 'groundmass' that was observed in the reduced tailings was not observed in these tailings.

The tailings from a hardpan, found at the north end of Cell 2, are very similar to the non-indurated oxidized tailings at the south end of Cell 2 in terms of alteration characteristics. The grain-size of the tailings in the hardpan area (0.3 mm average) is greater than that of the south end of Cell 2 and the whole of Cell 3, as this was very near the discharge point of the tailings for many years.

Figure 3-10. SEM backscatter image of corroded sphalerite (sph) in oxidized tailings. The rim around the sphalerite grain does not contain Zn. Compared to the sphalerite, the alteration of pyrite (py) is much less pronounced.





Figure 3-11. SEM backscatter image in oxidized tailings of chalcopyrite (cp) disseminated within a quartz grain.

Pyrrhotite is highly altered, with thick zoned rims (Figure 3-12). In places, the pyrrhotite is almost entirely dissolved with only a small amount of the original grain left at the center of the zoned alteration. Rims of alteration around sphalerite grains are thicker (5-10 µm) and usually continuous around the grain. EDS line scans across a number of sphalerite grains did not detect Zn in these alteration zones. Pyrite grain boundaries are weakly eroded (similar to the non-oxidized tailings), however the grains are coated with secondary Fe-oxy-hydroxides that completely surround the pyrite grains, acting as a cementing matrix (Figure 3-13). Again, all observed chalcopyrite was limited to disseminations within quartz. The fine-grained matrix that was observed in the reduced tailings was not observed in these tailings.

Figure 3-12. SEM backscatter image of highly altered pyrrhotite (po) grains. Pyrite (py) remains relatively unaltered.



Figure 3-13. A and B Polished thin section of the tailings hardpan.

A) Slight corrosion of biotite (bt) along with Fe-oxyhydroxide precipitation around the grain, viewed in plane polarized light.

B) The same thin section viewed with a combination of plane-polarized-transmitted- and reflected-light. Notice the slight corrosion and rim of Fe-oxyhydroxides surrounding pyrite (py). The Fe-oxyhydroxide minerals are coating all grains acting as a cementing matrix.



White and yellow evaporite material form as thin crusts on the surface of the tailings, or as 1 to 2 cm long stalactites within fissures and cracks in the tailings (Figure 3-14). Iron, zinc, and aluminum were identified by SEM EDS in evaporite minerals. The SEM also revealed the presence of sulphur associated with this evaporite material, which caused the yellow colour. XRD analyses proved these evaporites to be hydrated iron sulphates, primarily melanterite (FeSO₄·7H₂O) and rozenite (FeSO₄·4H₂O) (Figure 3-15). Zinc partially replaces iron in melanterite forming zincmelanterite ((Zn,Fe)SO₄·7H₂O), and AI is taken up primarily in halotrichite (FeAl₂(SO₄)₄·22(H₂O)) (Alpers *et al.*, 2000).

Dark, orange-brown material precipitated from groundwater samples after storage at 21°C for two months in the lab. The final pH of the solution from which these samples precipitated was pH 2.6. XRD analysis of these samples showed that lepidocrocite (Fe³⁺O(OH)) was the principal precipitate (Figure 3-15). Metals were not detected in two of the solutions (RBH-5 and RBH-7), however, RBH-1 water contained 0.14 ppm Ni, 2.2 ppm Cu, and 41 ppm Zn.

Figure 3-14. Photograph of an evaporite mineral crust, found along the edge of a fissure near the surface of the tailings in Cell 1.





Figure 3-15. XRD traces for tailings surface evaporite minerals.

Figure 3-16. XRD trace of material precipitated from groundwater. The broadening of the lepidocrocite peaks indicates the presence of amorphous material.



3.3 Sequential Extraction

Total metal concentrations in the reduced tailings varied for iron (18 to 25 wt. %), zinc (0.2 to 0.5 wt. %), and copper (0.03 to 0.09 wt. %). Similar variations in total metal concentration were also measured within the oxidized tailings:17 to 28 wt. % iron, 0.04 to 0.44 wt. % zinc, and 0.03 to 0.10 wt. % copper. Variation in total metal concentration with depth and between years is depicted in Figure 3-17, -18, and -19.

Several factors can lead to the variation observed in total metal concentrations. The most important may be mineralogical heterogeneity within the tailings. Heterogeneity would be caused by variations in the composition of the original ore deposit and also by mechanical separation, with more dense and larger mineral grains being deposited closer to the point of discharge. Oxidation processes within the tailings may also cause variations in total metal concentrations. Sulphides break down during oxidation and the metals are released from solids to pore and ground water. Errors in values were calculated from variations in total metal concentrations from similar sampling locations between 2004 and 2005 within the reduced portions of the tailings (Table 3-1).

	Depth (cm)	Fe Error (%)	Zn Error (%)	Cu Error (%)
Cell 2	50	2	28	17
	100	24	45	14
	400	33	19	17
Cell 3	20 – 30	34	84	25
	50 - 60	13	3	35

Table 3-1. Values of errors were based on variations in total metal concentrations from similar sampling locations between 2004 and 2005 in the reduced tailings using a percent difference calculation.



Figure 3-17. Total metal concentrations for Fe, Zn, and Cu in the reduced tailings for **Cell 3** in **2004** and **2005**. Error bars calculated from a comparison of results between similar sampling locations in 2004 and 2005.



Figure 3-18. Total concentrations of Fe, Zn, and Cu with depth in the oxidized tailings of **Cell 2** in **2004**. Error bars were calculated based on a comparison of samples taken from similar sampling location in 2004 and 2005.



Figure 3-19. Total concentrations of Fe, Zn, and Cu with depth in the oxidized tailings of **Cell 2** in **2005**. Error bars were calculated based on a comparison of samples taken from similar sampling location in 2004 and 2005.

Sequential extraction showed that Iron is primarily found in the residual (average 72 % of total iron in reduced and in oxidized tailings) and, to a lesser extent, the crystalline iron-hydroxide phases (17 % of total iron in reduced and 22 % of total iron in oxidized tailings, on average). Small portions of the total iron are found in the amorphous iron-hydroxide (average 5 to 6 % total iron in reduced and oxidized tailings) and adsorbed/exchangeable/carbonate and water soluble phases (< 1 % total iron in both reduced and oxidized tailings).

Zinc is primarily split between the residual (52 % total zinc in reduced and 37 % zinc in oxidized tailings, on average), water soluble (24 % total zinc in reduced and 29 % in oxidized tailings, on average) and crystalline iron-hydroxide phases (18 % total zinc in reduced, and 25 % in oxidized tailings, on average). Lesser amounts of zinc are in the amorphous iron-hydroxide (average 2 to 3 % total zinc in both reduced and oxidized tailings) and adsorbed/exchangeable/carbonate phases (4 % total zinc in reduced, and 8 % in oxidized tailings, on average).

Similar to iron, copper is found primarily in the residual (33 % total copper in reduced, and 22 % in oxidized tailings, on average) and crystalline iron-hydroxide phases (31 % total copper in reduced, and 40 % in oxidized tailings, on average). The remaining copper is split between the amorphous iron-hydroxide (average 11 % total copper in reduced and oxidized tailings), adsorbed/exchangeable/carbonate (average 8 % total copper in reduced and oxidized tailings) and water soluble phases (17 % total copper in reduced, and 21 % in oxidized tailings, on average).

Comparing concentrations of metals with depth and between 2004 and 2005 is inconclusive in the reduced tailings. This is because the error is equal to, or larger than,

the amount of change. When error is taken into account there does not appear to be any change with depth or between 2004 and 2005, in the reduced tailings.

In the oxidized tailings, the error in results is low enough that variations in total metal concentration with depth can be observed with confidence. Total iron in the oxidized tailings is depleted near the surface, as much as 3 wt. % within the first 30 cm. Zinc is depleted by as much as 0.3 wt. % within the first 100 cm, and copper by as much as 0.07 wt. % within the first 50 cm. Below 100 cm depth there is little variation in total metal concentrations within the oxidized tailings. Changes in the distribution of metals between different phases measured in the sequential extraction analyses were not greater than the error of the method. Temporal variations between 2004 and 2005 were also below the error in the method.

3.4 Water Chemistry

3.4.1 Pore water

Pore water was extracted from tailings samples collected from test pits and boreholes in Tailings Cells 2 and 3 in September 2004 and 2005. It is to be expected that porewater concentrations would vary with mineralogical variations within the tailings, but the depth profiles due to the effects of oxidation would be similar.

Variation with depth of the composition of pore water in both Cells 2 and 3 are similar, with low pH (down to 1.8 in Cell 3) and high total dissolved solids (~300 g/L) near the tailings surface where the tailings are oxidized and more neutral pH and lower dissolved solids with depth. In general, pH values were lower and dissolved metal concentrations higher in 2005 than 2004.

In 2004, the pH of pore water in Cell 3 was 4.4 within the top 10 cm of the tailings, increasing to near 7 by 30 cm depth. Dissolved constituents follow pH, with concentrations of 35,600 ppm SO_4^{2-} , 259 ppm Al, 18,000 ppm Fe, and 9,550 ppm Zn, 1,340 ppm Na, 1,100 ppm K, and 8,530 ppm Mg in the top 10 cm of Cell 3. These concentrations decrease steadily down to a depth of 30 cm, below which most heavy metals drop to below detection limits (Figure 3-20 and Appendix VIII).

Due to a higher groundwater table in 2005, the test-pit in Tailings Cell 3 could only be dug to 60 cm below grade. Pore water extracted from samples between 0 and 60 cm were noticeably different from that taken in 2004. There were increased concentrations of metals, with the highest concentrations measured between 10 and 20 cm below grade: 47,000 ppm Fe, and 9,100 ppm Al (Figure 3-20). Sulfate concentrations also

peaked at this depth (290,000 ppm SO_4). The maximum concentrations of Cu (1.9 ppm) corresponded with the lowest pH, between the surface and 10 cm below grade, in both 2004 (pH 4.4) and 2005 (pH 3.2), however, Cu concentrations were always below 2 ppm.

Pore water extracted from Tailings Cell 2 in September 2004 was characterized by high concentrations of dissolved metals and low values of pH down to approximately 0.7 m (Figure 3-21). The top 17 cm of this section contained the lowest pH at 1.8 with the next 10 cm (17 to 27 cm below grade) containing the highest concentrations of dissolved metals (230,000 ppm TDS) (Appendix VIII) in a sample that was dark blood-red after filtering. In the five samples from 1.2 m below grade to the bottom of the borehole at 3.7 m, concentrations of heavy metals decreased but remained above detection limits. The pH below 1.2 m was 6.6 +/- 0.2.

The 2004/2005 time variation seen in Cell 2 was different to that of Cell 3. Maximum concentrations of all dissolved ions in Cell 2 were primarily higher in 2004 (64,800 ppm Fe, 48.5 ppm Ni, 9,800 ppm Zn, 1,130 ppm Cu, 33.2 ppm Cd,7.93 ppm Pb, and 29,400 ppm Al) compared to than in 2005 (40,900 ppm Fe, 60.5 ppm Ni, 6,950 ppm Zn, 156 ppm Cu, 105 ppm Cd, 175 ppm Pb, and 3,720 ppm Al). In 2005 the maximum concentrations were located at lower depths in (Figure 3-21). There is a higher value of dissolved ions (6,970 ppm Fe,31.8 ppm Ni,3,000 ppm Zn,156 ppm Cu,105 ppm Cd, 32.0 ppm Pb,and 2,570 ppm Al)and lower pH (3.5) near the groundwater table at approximately 4m in 2005.

Figure 3-20. Cell 3 pore water analyses. All concentrations are in ppm. The solid blue line represents concentrations measured in 2004 and the solid red line represents concentrations measured in 2005. The blue inverted triangle and dashed blue line represents the water table in 2004. The red inverted triangle and dashed red line represents the water table in 2005.



Figure 3-21. Cell 2 pore water analyses. All concentrations are in ppm. The redshaded zone near the surface represents the zone of observed oxidation. The solid blue line represents concentrations measured in 2004 and the solid red line represents concentrations measured in 2005. The inverted triangle and dashed red line represents the water table in 2005 (the water table was below the depth of sampling in 2004).



Calculated metal speciation from the surface of the tailings down to between 10 and 20 cm depth in Cell 2 and 3 in 2004, was split between complete dissociation and complexation with SO_4 . Below this depth metals are primarily present as cations without complexation. In 2005, there was a similar trend, however, metals are complexed with SO_4 to a greater depth, that is, down to 1 m below grade in Cell 2 and to the bottom of the test-pit in Cell 3 (Appendix IX).

3.4.2 Groundwater

With pH at or near neutral (pH 6.0 to 7.0) in 2004, the alkali and alkali-earth metals (Ca, Mg, K, and Na) were the main elements in the groundwater, with sulphate as the primary anion. Minor amounts of Fe, Cu, Zn, and Al were present in the 2004 groundwater (Appendix X). In Tailings Cell 3, the least oxidized of the tailings, groundwater from R-BH-1 changed composition during summer 2004. Between June and October 2004, groundwater composition at sampling location R-BH-1 increased by two orders of magnitude for metals such as Fe (from 1.23 to 131 ppm), Al (from 0.81 to 79 ppm), and Zn (from 1.70 to 41.6 ppm). During this same time period, alkali metal concentrations decreased: Na (92.8 to 18.5 ppm), K (49.3 to 6.70 ppm), and Ca (465 to 329 ppm) but Mg increased from 19.3 to 152 ppm.

The trend of increasing metal concentrations continued from 2004 to 2005 at R-BH-1, where a sample taken in September 2005 contained 13,400 ppm Fe, 2,040 ppm Al, and 874 ppm Zn, which were two orders of magnitude higher than that in October 2004. From 2004 to 2005 pH decreased to 2.4. Alkali metal concentrations continued to decrease, except for Mg which increased to 991 ppm.

Similar, although less pronounced, changes were also noted in Cell 1 from September 2004 to September 2005 at sampling locations R-BH-5 (Fe from 1.09 to 803 pmm; Al from <0.005 to 1.76 ppm; and Zn from 0.18 to 1.26 ppm) and R-BH-7 (Fe from 2.60 to 3770 ppm; Al from <0.005 to 8.60 ppm; and Zn from 1.26 to 106 ppm) (Figure 3-22). The pH values decreased over the same time period from 6.9 to 3.5 at R-BH-5 and from 7.5 to 3.5 at R-BH-7.

The composition of ground water from Cell 1 is similar in 2007 to 2005 at RBH-5 (840 ppm Fe, 0.65 ppm Al, and 0.527 Zn). The well at RBH-7 had lower dissolved metal contents in 2007 (1670 ppm Fe, 1.95 ppm Al, and 0.784 ppm Zn) than it did in 2005.

Metal speciation in groundwater samples, calculated through geochemical modeling, is primarily split between complexes with SO₄ and complete dissociation, with a tendency towards SO₄ complexation with decreasing pH. Figure 3-23 shows the percentage of SO₄ complexed with metals relative to pH. In samples with pH values >7.0, Fe would complex with OH⁻ as well as SO₄²⁻, however, at these pH values Fe concentrations are low: <0.02 ppm. Al also shows complexation with OH for samples with pH >6.0. Aluminum concentrations for these samples are between 0.07 and 79 ppm.

Figure 3-22. 2005 chemical data from Cell 1 sampling locations R-BH-5 and R-BH-7, compared to the average data for those locations in 2004.







3.4.3 Tailings Surface Water

Surface water samples represent waters collected from either pools on, or drainage trenches in the surface of the tailings. Samples were primarily collected in 2004 with only two samples taken in 2003, one in 2005, and two in 2007.

All mine surface water in direct contact with the tailings is characterized by low pH (2.0 to 3.5) and a large amount of dissolved solids (1,400 to 17,000 ppm TDS). Average concentrations of elements in solution include 7,100 ppm SO₄, 326 ppm Ca, 358 ppm Mg, 1,170 ppm Fe, and 254 ppm Al with 21.8 ppm K, 33.9 ppm Na, 0.0150 ppm As, 0.356 ppm Cd, 11.4 ppm Cu, and 184 ppm Zn (Appendix XII). The highest levels of total dissolved solids were measured in standing pools associated with the least oxidized tailings (RW-12: 17,000 ppm TDS; and RW-13: 14,000 ppm TDS) and the trenches draining these pools (RW-2: 12,000 ppm TDS). In an area where standing pools were associated with more oxidized tailings, dissolved constituents were lower (RW-6: 1,400 ppm TDS in 2004), despite the pH being lower (2.7). Surface water samples obtained in 2007 had similar chemical compositions to that in 2004 and 2005 (Appendix XII).

Ion speciation calculations suggests that metals in the surface water are divided between complete dissolution as ions and as complexes with SO_4^{-2} (Appendix XIII). More than 90% of Na and K in surface waters do not form complexes. As the pH decreases and the concentration of SO_4^{-2} and major cations increase, Mg, Ca, and the major transition metals in solution become increasingly complexed with SO_4^{-2} , with up to $50\% SO_4^{-2}$ complexation. Additionally, Al increasingly forms complexes with $(SO_4)_2^{-4}$ as SO_4^{-2} concentration increases in the surface water.

3.4.4 Lake Water

Since the inception of the Ruttan Mine, Ruttan Lake has been influenced by contaminated surface water run-off from the mine site. Water entering the lake from the tailings impoundment (sampling location RW-4) has an average pH of 2.9 and high concentrations of Ca (153 to 381 ppm), Fe (80.1 to 1510 ppm), Zn (20.3 to 169 ppm), Mn (1.83 to 6.48 ppm), Cu (1.37 to 21.7 ppm), Cd (0.0360 to 0.340 ppm), Al (47.8 to 529 ppm) and SO₄ (1470 to 11,000 ppm) (Appendix XIV). Metal and sulphate concentrations become more dilute on the shore opposite the tailings impoundment (e.g. RW-8 and RW-9) but remain at high levels (Appendix XV). At the outlet from Ruttan Lake toward the open pit (e.g. RW-3), the concentration of total dissolved constituents increases (maximum TDS at RW-3 of 14,000) as this stream also contains run-off from nearby tailings and waste rock. On average, Ruttan lake is more dilute than tailings ground- and surface-water.

Samples of lake water were obtained in 2004 and 2005 to observe the effect of changes in oxidation of the tailings. Unfortunately the high rainfall in 2005 diluted the lake water (Figure 3-24). However, a comparison of lake water from June to October 2004 shows that progressive oxidation of the tailings during the summer months increased the average concentration of dissolved constituents to 37.2 ppm Na, 21.7 ppm K,382 ppm Mg,343 ppm Ca,1.36 ppm Sr, 1,320 ppm Fe, 12.4 ppm Cu, 12.4 ppm Zn, 0.225 ppm Cd, 345 ppm Al, 24.8 ppm Cl, and 4,510 ppm SO₄⁻² (Figure 3-24).

Geochemical speciation modeling showed that, on average, ions in the lake water are either completely dissociated or attached to SO_4 in solution. Al and Zn also form SO_4^{-2} complexes, and show increased dissociation at lower pH values.

Figure 3-24. Temporal variations in Lake Chemistry. Average composition of all lake waters at different points in time, normalized to the average of all lake samples.



Average Lake Chemistry Temporal Variations

3.5 Hydrogeology

As the Ruttan tailings oxidize, low pH, metal laden waters will drain from them into Ruttan Lake. Understanding how the oxidation of the tailings will affect Ruttan Lake, requires an assessment of the flow of heavy metals liberated by sulfide oxidation from the tailings. This can be partially achieved through modeling the flow of groundwater and surface water within the tailings impoundment.

Depth measurements across the outlet stream of the tailings into Ruttan Lake, presented in Figure 3-25, were used to calculate an average cross-sectional area for the stream (786 cm²) along the 2.3 m portion used for flow calculations. With a calculated volume of 181 L and an average time for water to flow from start to finish of 5.8 seconds, a flow rate of 31 L/s was calculated.





Modeling of groundwater flow through tailings requires a number of properties of the tailings to be known. Grainsize distribution, water content, and specific gravity

measurements obtained from laboratory experiments were used to determine bulk density, particle density, porosity, and saturation of the tailings (Table 3-2). These laboratory analyses coupled with field data allowed for confident modeling.

Although rough grainsize estimates were obtained for use in groundwater flow modeling, proper grainsize distribution curves were not prepared and most grainsize data was not incorporated into this study. Grainsize measurements for these tailings were very difficult due to the large amount of very fine grained material, as noted in observation of thin sections (Section 3.2). During sieving this material would stick to coarser grains creating 'balls' of even larger material. Washing each sieve was necessary to make sure all appropriate material passed through to the finer screened sieves. When it came to the hydrometer tests to find the distribution of the fine grained material a problem with flocculation arose. When ore is processed it is mixed with chemicals that encourage this material to flocculate in solution. De-flocculating chemicals were added in the lab to the hydrometer samples to prevent this, however, flocculation continued to occur. Similar problems occurred when trying to define the distribution of the fines with a laser particle counter (Appendix XVII).

Groundwater monitoring wells were installed at various locations within the Ruttan tailings. The purpose of these wells was to measure groundwater levels, hydraulic conductivity, and water chemistry. Measurements taken from three wells in Cell 1 of the tailings were used to model groundwater flow with the Seep/W[™] groundwater flow modeling program. Groundwater flow is used to assess the effect of the trench on subsurface tailings flow, the rate of flow into the trench in relation to solute transport, and the effect of precipitation on the groundwater table.

Hydraulic conductivities (k) of the saturated tailings measured in the field yielded values ranging from $4x10^{-7}$ to $8x10^{-7}$ m/s (typical values for silts and clays); an average of these values ($5x10^{-7}$ m/s) was used in the model (Table 3-2).

Hydraulic conductivities of saturated tailings can be measured in the field through pumping tests or in the lab using draw-down tests on a small amount of material. However, lab tests don't account for vertical changes in hydraulic conductivity that may arise through the deposition of tailings in layers. By installing piezometers that are screened through multiple layers in the tailings an average hydraulic conductivity can be assumed.

Although layering within the tailings can be accounted for by installing piezometers with long screen lengths, they do not account for vertical hydraulic flow from the surface of the tailings down to the water table. Hydraulic conductivities in unsaturated media are less than that of saturated media, with a dependence on the water content. A function relating pore-water pressure and hydraulic conductivity with volumetric water content for the specified media is required to define unsaturated flow. This function can be found using the van Genutchen (1980) predictive method:

$$k_{w} = \frac{k_{s}^{*} [1 - (\alpha P_{c}^{(n-1)})^{*} (1 + (\alpha P_{c})^{n})^{-m}]^{2}}{((1 + \alpha P_{c})^{n})^{m/2}}$$
3-1

Table 3-2. Porous media properties for tailings and adjacent materials.

Unit	Hydraulic Conductivity (cm/s)	Grainsize Distribution	Water Content (%)	Specific Gravity	Bulk Density (g/cm³)	Solids Density (g/cm ³)	Porosity (%)	Saturation (%)
¹ Tailings Saturated	5 x10 ⁻⁰⁵	15% m.g. sand 85% silt	-	3.5	2.02	3.5	42	-
^{2,3} Tailings Unsaturated	-	15% m.g. sand 85% silt	30	3.5	2.02	3.5	42	80
⁴ Bedrock	3 x10⁻¹⁰	-	-	-	-	-	Low	-
⁵Dykes	1 x10 ⁻⁰⁹	-	-	-	-	-	33	-
¹ Onsite and laboratory testing								

Onsite and laboratory testing. K in unsaturated flow is less than saturated flow and depends on volumetric water content.

Saturation in the unsaturated tailings is at field capacity. Unfractured metamorphic rock: $K = 3 \times 10^{-12}$ to 3×10^{-08} cm/s (Freeze and Cherry, 1979). Clay: $K = 10^{-09}$ to 10^{-06} cm/s; Porosity = 33 - 60% (Fetter, 2001)
In equation 3-1, α , n, and m are fitting parameters, Pc is the capillary water pressure, and ks is the saturated hydraulic conductivity. The values used for the equation in the model ($\alpha = 0.016 \text{ cm}^{-1}$, n = 1.37, and m = 0.270) represent a silt of similar saturated hydraulic conductivity and porosity as the Ruttan tailings (Fetter, 1999). Using the values in Table 3-2, and function linking volumetric water content and pressure (Figure 3-26), k was plotted, for capillary water pressures ranging from -100 to 0 kPa, with the Seep/W program (Figure 3-27).





Figure 3-27. Hydraulic conductivity (m/s) versus pressure plot based on silt fitting parameters and a ks of $5x10^{-7}$ m/s.



At the time these measurements were taken, the tailings in Cell 1 were not oxidized at the surface and appeared to be fairly isotropic and homogenous with little to no variation in grainsize. Therefore it could be assumed that the porous media properties would be the same for all the tailings represented in the model.

3.5.1 Boundary Conditions for Flow Modeling

Historic mine documents indicate that the tailings are underlain by unfractured meta-volcaniclastic bedrock, which would have a hydraulic conductivity between 3x10⁻¹⁴- and 3x10⁻¹⁰-m/s (Freeze and Cherry 1979). Porosity in the bedrock would be negligible. From survey data acquired in 1992 by Hudson Bay Mining and Smelting, depth to the bedrock in Cell 1 was estimated to be 10 m.

Because of the large difference in k (at least 3 orders of magnitude) between the tailings and the underlying bedrock, the boundary between the two is assumed to have no-flow (zero flux).

Many types of dykes have been constructed at Ruttan to contain the tailings (Bloodworth, pers. comm.). Plans for dykes designed by Sherritt Gordon Mines Ltd. in 1973 suggest the presence of a clay core that would aid in slowing water movement through the dykes. Fetter (2001) suggests k values of approximately 10⁻¹¹ to 10⁻⁰⁸ m/s, and porosities of 33 to 60% for tight clays. There is at least one order of magnitude difference in k between the tailings and the clay of the dyke. The nearest dyke is at least 100 m from the drainage trench. These factors allow the assumption of a no-flow boundary between the dykes and the tailings and allow for a first approximation of flux of fluids entering the trench.

In September 2004, the water table within the tailings was found to be about 8 m above the underlying bedrock (i.e. 2 m below grade) (Figure 3-28). Water levels in the trench are relatively constant at about 7 m above the bedrock due to drainage of the trench to the Ruttan Lake at its southeast end. Water levels between BH-5 (7.5 m above bedrock) and BH-6 (8 m above bedrock) prove that the groundwater table slopes within the tailings towards the trench. The sloping groundwater table could be accounted for by adding an initial groundwater table into the model. For no-flux steady state analysis, head boundaries of 7 m within the trench, and 8 m within the tailings were used. The head boundaries were not entered where the groundwater table was sloped as the groundwater table would be interpolated during the modeling.

To model precipitation effects a unit flux value (q), representing Environment Canada's designation of moderate to light-rainfall (2.8×10^{-06} m/s), was applied to the surface of the tailings. The initial water table within the tailings was kept the same as for the no-flux scenario; however, the head boundaries within the tailings were removed. A head value of 7 m within the trench was used to simulate drainage out of the trench.

The Seep/W program needs a user defined grid to perform a series of partial differential equations that predict the response of a physical system subjected to external influences. For the Ruttan study, a finite element mesh size of 100 cm² was used in the tailings, up to the edge of the trench. The element size was reduced to 50 cm² below the trench so that more detailed flow calculations could be made within the model.



Figure 3-28. Relation of drainage trench to monitoring wells BH-5 and BH-6, water table (blue line and triangles), dykes, and underlying bedrock.

3.5.2 Modeling Results

The Seep/W TM model, run at a steady state with head boundaries of 8 m within the tailings and 7m within the trench, indicated that the trench influences groundwater movement within an area of approximately 25 m, with the fastest flows closest to the trench (Figure 3-29). Flow rates within the tailings increase towards the trench with a maximum flow of 1.22×10^{-7} m/s across the tailings/trench interface below the water table. Over a 10 m section this flow rate relates to about 10^{-5} m³/s of groundwater, entering the trench.

As the primary source of water into the tailings is precipitation (headwaters are redirected around site), transient conditions were modeled in an attempt to view the effects of precipitation over time. Transient analyses were also performed to model the effects of groundwater table draw-down due to drainage of the tailings trench. To model transient conditions, the Seep/W[™] program performed a number of steady state analyses with input values of the length of time for each steady state time step and the number of steps. Time increments of 3600 s were used over a period of 1 week (336 steps).

In the transient mode the Seep/W[™] program also allows boundary conditions to be defined for a set period of time. To simulate an episodic rain event the boundary flux was set at the surface of the tailings to 10 mm/hr for 6 hr and then to zero flux after 6 hours.

Adding a flux value to the surface of the tailings to simulate rainfall, under steady state conditions, raised the water table to the surface of the tailings, increased the flow within the tailings, and expanded the area of influence of the trench for flow within the subsurface (Figure 3-30). Placing a flux boundary at the surface of the tailings appeared to force water into and through the tailings at unrealistic rates (i.e. rates at least an order of magnitude higher than the hydraulic conductivity of the tailings). Placing the head at the surface of the tailings would simulate the increase in head within the tailings that would occur during a rainfall, without creating the unrealistic flow rates observed during the flux boundary tests. By changing the head values in the tailings so that they equal the surface elevation, and removing the flux boundary, water moved through the tailings at more reasonable rates (i.e. rates the same as the hydraulic conductivity) (Figure 3-31).

During transient analyses, the groundwater table was shown to react instantly to changes in flux or head. To simulate the drop in water table following a rain event, a surface flux of 10 mm/hr was introduced to the surface of the tailings for a period of 2 hrs and then set to zero to allow the tailings to return to equilibrium (In Figure 3-32). There was a change in groundwater table height after only 1 min. Again, to remove the effect of water being forced into the tailings, the flux boundary was removed, the head in the model was set to the surface of the tailings to represent a rain even, and then returned to 8 m after 2 hrs. Stopping the rain in both models, flux rates returned to zero or head values returned to 8 m, resulted in the water table returning to static conditions but at much slower rates (Figure 3-33) than was seen during the water table rise. This displays the drawdown mechanism in the tailings: The drainage trench. The change in height of the groundwater table becomes exponentially smaller as it approaches static conditions (Figure 3-34).



Figure 3-29. Steady state, zero flux model of groundwater flow within the tailings. The length of the arrows within the tailings represents larger flow.

Figure 3-30. Steady state, constant flux of 2.8x10-6 m/s, groundwater flow model.





Figure 3-31. Steady state, tailings head raised to 10 m, groundwater flow model.

Figure 3-32. Instantaneous groundwater reaction to surface flux.

Ruttan Tailings Cell 1 - Drainage Trench 2-D Transient 10 mm/hr - 2hr Surface Flux



Figure 3-33. Graph of boundary flux (m/s) versus time (s) for flux across the tailings surface. The spike represents the increased flux during the rain event.



Figure 3-34. Graph of total head (m) versus time (s) at a point at 9 m above datum. Note the slowing rate of subsidence of total head back to steady.



3.6 Microbiology

3.6.1 Background

The low pH solutions laden with heavy metals that characterize acidic tailings make these environments toxic to most organisms including plants, animals, and humans (Nixdorf *et al.* 2001). Not all organisms are excluded from these environments. By interacting with their surroundings, acidophilic microbes may alter oxygen conditions, redox potentials, pH and chemistry of surrounding waters in order to survive and thrive in low pH conditions.

Microbes that live in low pH conditions are called acidophiles. Acidophiles preferentially grow at a pH below 6, or, more specifically below pH 5.4, and extreme acidophiles optimally grow at pH values below 3 (e.g. Perry *et al.*, 2002). Microorganisms in ARD are significantly diverse given the extreme nature of their environment.

Microorganisms exert important controls on the rates of weathering of minerals, because they derive metabolic energy from redox reactions and can promote the dissolution of minerals by both reductive and oxidative pathways (Baker and Banfield, 2003). The presence of microbes in ARD solutions as well as attached to sulfide surfaces suggests that there may be direct and indirect mechanisms by which microorganisms breakdown sulfides.

Microbes need electrons as a source of energy. Inside their Cells, microbes move the electrons through an electron transport chain to create an electrochemical membrane potential that can be used to produce adenosine triphosphate or ATP (a multifunctional

nucleotide that transports chemical energy within a Cell for metabolism). By oxidizing certain compounds, for instance Fe^{2+} to Fe^{3+} , microorganisms can acquire the needed electrons. Not only does this reaction produce energy, but it can also keep the internal pH of microorganisms more neutral. Some microbes not only oxidize iron, but under certain conditions they may also reduce it. Under anaerobic conditions, they can use elemental sulfur as an electron donor and ferrous iron as an electron acceptor (González-Toril *et al.*, 2003).

Many ARD sites are located in areas that experience fluctuations in environmental conditions due to seasonal changes in light, temperature, and precipitation, which could have a large impact on their microbial communities. Additionally, microbes live in a wide range of environments and ecosystems with each having a different community of microorganisms. This variation in microbial communities is the result of different chemical and physical conditions. The most important of conditions in the ARD environment are the temperature, pH, redox state, and water availability. Changes in these parameters will change the types of organisms and the reactions that take place with the ARD system.

In this study, the biomass of the microbes within the Ruttan tailings was measured and compared to levels cited in the literature to assess the microbial activity. In order to estimate microbial biomass, proteins were assayed of the most oxidized tailings (Cell 2) and the least oxidized (Cell 3). Additionally, samples were collected in 2004 and in 2005 to see if there were significant changes in microbial concentrations as oxidation of the tailings proceeds.

3.6.2 Tailings Cell 3 Protein Assays

In general, protein concentrations increase from the surface (337 μ g/g) down to the water table at approximately 90 cm depth (1120 μ g/g) (Figure 3-35 and Appendix XIX). There is slightly more protein at the surface than just below (161 μ g/g at 15 cm depth). There is also a spike (827 μ g/g) at approximately 45 cm depth. In the zone where oxidation is only taking place along cracks and fissures within the tailings, protein concentrations are below 337 μ g/g. Below this zone, where the tailings have not been oxidized, the values are all above 561 μ g/g.

In 2005, protein concentrations were higher than that of 2004, however, the profile of the near surface concentrations was similar: In general, protein increased from the surface (2830 μ g/g) down to the water table (4380 μ g/g) at approximately 20 cm depth (Figure 3-35 and Appendix XX). Also similar to 2004, the 2005 protein concentrations at the surface were higher than just below (110 μ g/g at 5 cm depth). Samples from below the water table at 30 cm (867 μ g/g) (the bottom of the zone of oxidation) and 40 cm (617 μ g/g) depth had lower protein concentrations than at the water table.

Figure 3-35. Cell 3 biomass. Protein concentration (μ g/g) with depth (cm), in Tailings Cell 3. The 2004 protein concentrations are depicted with a solid blue line. The 2005 protein concentrations are depicted with a dashed red line. The 2004 and 2005 groundwater levels are marked with a black line and a solid blue and red triangle, respectively. The lighter (yellow) shaded region marks the extent of minor oxidation in 2004 and the darker (pink) shaded region marks the extent of major oxidation in 2005.



Biomass Cell 3

3.6.3 Tailings Cell 2 Protein Assays

In 2004, protein concentrations ranged from 637 μ g/g to 5550 μ g/g (Figure 3-36 and Appendix XIX). Protein concentrations were higher at the surface (2570 μ g/g) than at the near surface (2040 μ g/g at 10 cm depth). Within the first 50 cm of the surface, the highest protein concentrations were at the interface between the highly oxidized tailings and the weakly oxidized transition to reduced tailings. In the reduced zone of Cell 2, protein concentrations ranged from 1120 μ g/g to 5550 μ g/g, however, most of the variation is within the values of precision of measurement. Protein values increase from 50 cm (637 μ g/g) to 90 cm (5550 μ g/g) depth then decrease to 2060 μ g/g by 120 cm depth. Between 170 and 270 cm depth, protein concentrations increase to between 4000 and 6000 μ g/g. Below 270 cm depth, protein concentration decreases down to 370 cm (approximately 1500 μ g/g).

Unlike Cell 3, Cell 2 had an overall reduction in protein concentrations from 2004 to 2005 (Figure 3-36 and Appendix XX). However, the Cell 2 protein depth profile is very similar from 2004 to 2005, with surface values (412 μ g/g) higher than the near surface (201 μ g/g at 10 cm depth) and a peak at the bottom of the oxidized zone (1150 μ g/g at 20 cm depth). Values are slightly lower in the transition zone between oxidized and reduced tailings (696 μ g/g to 947 μ g/g between 30 and 50 cm depth). At the bottom of the transition zone, protein concentrations increase to 2180 μ g/g at 60 cm depth), but increase to 2980 μ g/g by 100 cm depth. Below this, protein decreases to 1990 μ g/g by

200 cm depth. At 250 cm depth the highest protein concentration in this test hole was

measured: 3110 μ g/g. The sample, at the water table assayed at 1600 μ g/g protein.

Figure 3-36. Cell 2 biomass with depth. Protein concentration (μ g/g) with depth (cm), in Tailings Cell 2. 2004 protein concentrations are depicted with a solid blue line. 2005 protein concentrations are depicted a dashed red line. 2004 and 2005 groundwater levels are marked with a black line and a solid blue and red triangle, respectively. The lighter yellow shaded region marks the extent of minor oxidation and the red shaded region marks the extent of minor oxidation and the red shaded region marks the extent of major oxidation in 2004 and 2005.



4 Discussion

4.1 Acid Generation and Metal Release

When sulphide minerals are exposed to oxygen and water they become unstable and, through oxidation reactions, begin to break down. There are a number of reactions that take place during oxidation, but, in general when oxygen is the oxidant, sulphides with metal to sulfur ratio of less than one generate acid, and those with the ratio equal to one generally do not (Nordstrom, 1999). The oxidation of pyrite by oxygen is one example of a reaction where acid is generated by these processes (Equation 4-1).

$$FeS_2 + 3.5O_2 + H_2O = Fe^{2+} + SO_4^{2-} + 2H^+$$
 4-1

Ferric iron produced through the oxidation of ferrous iron (Equation 4-2), reacts with sulphide minerals to produce sulphate and acid (Equation 4-3).

Generation of
$$Fe^{3^+}$$
: $Fe^{2^+} + 0.5O_2 + H^+ = Fe^{3^+} + 0.5H_2O$ 4-2Oxidation by Fe^{3^+} : $FeS_2 + 14Fe^{3^+} + 8H_2O = 15Fe^{2^+} + 2SO_4^{2^-} + 16H^+$ 4-3

In the Ruttan tailings, there are abundant sulphides, up to 30 wt. % of the tailings material. Of these sulphides, 90 wt. % are pyrite and pyrrhotite with metal to sulphide ratios less than one with the remaining 10 wt. % of the sulphides being sphalerite, chalcopyrite, and galena. Therefore, within the Ruttan tailings the conditions are met for the generation of acid through the oxidation of sulphides by oxygen. However, the total amount of acid produced not only depends on acid generating reactions, but acid consuming reactions must also be considered.

In all mineral deposits, gangue minerals with little economic value are processed along with the ore and pumped out as waste. These gangue minerals often comprise the bulk of the mine waste: 70 wt. % of the tailings at Ruttan. Commonly, within this gangue material are minerals that have the potential to neutralize acidic solutions by consuming protons. At some mine-sites there are enough acid consuming minerals to neutralize all the acid produced by the oxidation of sulphides.

The most efficient acid consuming minerals are carbonates, which can buffer solutions to a pH between 5.0 and 7.0 (Blowes *et. al.*, 2003 a) (Equation 4-4). These minerals have not been observed in the Ruttan tailings, although reported in the ore zone by Barrie *et al.* (2005), and, therefore, do not play a role in acid consumption at this site. After primary carbonate minerals, the next set of neutralizing reactions take place between pH 4.0 and 4.5 through the dissolution of secondary alumino-hydroxides (Blowes and Jambor, 1990; Johnson *et. al.* 2000; Blowes *et. al.*, 2003a; Sherriff *et al. in press*) (Equation 4-5). Hydroxides are part of a suite of secondary minerals precipitated within the tailings at Ruttan (Section 3).

$$CaCO_3 + H^+ = Ca^{2+} + HCO_3^-$$
 4-4
Al(OH)₃ + 3 H⁺ = Al³⁺ + 3 H₂O 4-5

Silicates, more specifically aluminosilicates, are much less reactive than carbonates and hydroxides, at the same surface area, but they can also react to consume protons from solution in pore water (Blowes *et. al.* 2003). There are silicate minerals present in the Ruttan tailings that are able to go through acid consuming reactions. The rate at which these reactions occur is relatively slow compared to the rate of acid production from sulphides (Blowes *et. al.* 2003). Table 4-1 compares the relative reactivities of acid consuming minerals common to mine tailings and lists the ones found at Ruttan. The

rate of dissolution of silicates does increase with a decrease in pH, however, as pH

lowers the solubility of a powerful sulphide oxidant, ferric iron, increases.

Mineral Group	Typical Minerals	Observed at Ruttan	Relative Reactivity at pH5
Dissolving	Calcite, aragonite, dolomite, magnesite, and brucite.	None.	1.00
Fast Weathering	Anorthite, nepheline, forsterite, olivine, garnet, jadeite, leucite, spodumene, diopside, and wollastonite.	Almandine (garnet).	0.40
Intermediate Weathering	Sorosilicates, pyroxenes, amphiboles, and phyllosilicates.	Pyroxenes, amphiboles, phyllosilicates (biotite and muscovite)	0.02
Slow Weathering	Plagioclase feldspars and clays.	Plagioclase feldspars.	0.01
Very Slow Weathering	K-feldspars and muscovite.	K-feldspars.	0.01
Almost Inert	Quartz, rutile, and zircon.	Quartz.	0.0004

Table 4-1. Relative reactivities, at pH 5, of acid consuming minerals often found in mine tailings, with the minerals observed in the Ruttan tailings.

(after Sverdrup, 1990; and Kwong, 1993)

As proton producing reactions continue, the pH of the system decreases creating conditions suitable for further oxidation of ferrous iron. Even the sulphides with metal to sulphur rations equal to one produce acid when oxidized by ferric iron (Equation 4-6).

$$ZnS + 8Fe^{3+} + 4H_2O = Zn^{2+} + 8Fe^{2+} + SO_4^{2-} + 8H^+$$
4-6

Given that there are approximately 30 million tons of tailings spread over a 2 km area at the Ruttan Mine, with an abundance of sulphides and a paucity of acid consuming gangue minerals, there is the potential to continuously generate acid at this site. In addition to acid production, the reactions outlined in Equations 4-1 to 4-4 show that metals are released into solution.

Assays of the reduced tailings at the Ruttan mine show average concentrations for zinc of 2200 ppm and for copper of 527 ppm (Appendix III). From historical metallurgical data and thin section analysis (Figure 3-6), it is apparent that the primary source for these metals is sphalerite (ZnS) and chalcopyrite (CuFeS₂). There are numerous studies regarding the reactivity of sulphide minerals which are summarized in Jambor, (1994). Although the results somewhat vary from study to study depending on the conditions of the study, a general order of increasing resistance to reactivity is:

pyrrhotite > sphalerite/chalcopyrite > pyrite

At the Ruttan site a similar order of alteration of sulphides is present in the oxidized zones of the tailings. Pyrrhotite has thick rims of Fe oxyhydroxide secondary material with the grain boundaries of the original mineral no longer visible; sphalerite shows some dissolution textures; and pyrite is relatively unaltered. The chalcopyrite as disseminated within silicate minerals and therefore sheltered from oxidation. However, there must be grains of chalcopyrite and secondary minerals containing copper dissolving because Cu is present in the tailings pore water.

As described in Section 3.5, microbes play an active role in the oxidation of sulfides in mine waste. With an average protein concentration of approximately 2100 μ g/g, protein biomass in the tailings at Ruttan is higher than that measured in gold mine tailings in Manitoba (Londry 2005). This is probably due to the higher sulphide content of tailings at Ruttan.

In Tailings Cell 3, we see an increase in oxidation of the tailings near the surface both visually and chemically from 2004 to 2005. Protein values also increase showing that certain microbes in this environment are amenable to lower pH conditions (Section 3.6, and Appendix XIX and Appendix XX).

The protein concentrations in Cell 2, are lower in 2005 than they were in 2004. This is opposite to what is observed in Cell 3. It would be expected that as oxidation proceeds, the microbial population that interacts with this process would either increase, or remain constant. The pore water concentrations of metals derived from sulphides (Fe, Cu, Zn) also have a greater concentration in 2004. So it appears that in Cell 2, oxidation is either slowing down or the major increase in meteoric precipitation between 2004 and 2005 diluted metal and protein concentrations.

In both Tailings Cells, the protein concentrations at the surface are higher than just below the surface in both years. This is possibly due to the increased availability of oxygen and water at the surface.

Another phenomenon that is observed in the tailings is an increase in the abundance of protein at the interface between oxidized and reduced tailings. This may be due to the increase in the availability of non-rimmed sulphides (i.e. not armored), compared to the

oxidized zone, as observed in thin section. This would allow for direct contact between the sulphide surface and the microbe. In this same zone there is also an increase in total Fe in the pore water, which can be utilized as an energy source by Fe oxidizing microbes in solution. So regardless of whether or not microbes actually attach to mineral surfaces, there is an abundant energy source in this zone and the bacteria are releasing Fe to the pore water.

Below the zone of oxidation the tailings show different concentrations and profiles over geographic distance and time. In Tailings Cell 3, 2004, the protein concentrations remained relatively constant below the surface with a slight increase at 45 cm depth. At this depth there is a 1 cm thick lens of coarser-grained tailings (Figure 4-1). The water content at this depth is also slightly lower than the rest of the tailings, so it is possible that groundwater is draining faster through this layer and that due to this drainage there is an increased amount of oxygen available for oxidation and microbial utilization (more on this in the discussion of hydrology). Unfortunately, due to high groundwater levels in 2005, the tailings were not sampled for protein content below 40 cm depth in Cell 3.

In Tailings Cell 2, below the zone of active oxidation, protein concentrations change with depth, suggesting changes in the availability of energy sources. As seen in Tailings Cell 3, a relatively thin layer of coarser material creates a noticeable impact on protein concentrations. As these layers are most likely the result of coarser discharge from the tailings spigot, it is probable, that similar layers exist in Cell 2.

Figure 4-1. Course grained layer at approximately 45 cm depth in Tailings Cell 2004.



4.2 Transport and Attenuation of Metals

The pore water in tailings is in direct contact with sulphide grains and acts as the source of water for oxidation reactions. Furthermore, the metals and protons released from these oxidation reactions accumulate in this pore water. The tailings near the surface of Cell 2 at Ruttan have been exposed to oxidizing conditions longer (approximately 20 years) than any other tailings on the site; therefore the pore water in this zone has the highest acidity and concentrations of metals. Table 4-2 summarizes the highest metal concentrations measured in the pore water of Cell 2 and compares it to the highest levels measured in the pore water of the Sherridon mine tailings. The mine at Sherridon, Manitoba produced Cu, Zn, and Au from about 70 years ago and the

tailings from this operation have been exposed to oxidizing conditions since that time (Moncur *et. al.*, 2005). The sulphide content of the Sherridon tailings (up to 60 wt. %) is twice that of the Ruttan tailings, and pH levels are lower and metal concentrations are higher.

Table 4-2. Comparison of the most acidic and highest metal concentrations (g/L) in tailings pore water between the Sherridon Heath Steele and Ruttan Mines.

	рН	Cu	Zn	Cd	Fe	SO ₄	
Sherridon ¹	0.67	1.6	55	0.1	129	280	
Ruttan Cell 2	1.8	1.1	9.8	0.03	41	230	

Dissolved metal and SO₄ concentrations are all presented in ppm.

¹ (in Moncur *et. al.*, 2005, summarized from Moncur *et. al.*, 2003)

Pore water metal concentrations and acidity of the Cell 3 tailings at Ruttan are not as extreme as that of Cell 2, but it is evident that Cell 3 is beginning to oxidize and approaching Cell 2 conditions. In 2004, maximum concentrations of Fe (18 gL⁻¹), Cu (0.94 mgL⁻¹), Zn (10 gL⁻¹), and SO₄ (36 gL⁻¹) were measured in the upper 10 cm of the tailings and rapidly decreased over the next 20 cm depth, however, pH did not drop as low as in Cell 2. In 2005, the maximum concentrations of Fe (47 gL⁻¹), Cu (1.0 mgL⁻¹), and SO₄ (290 gL⁻¹) were all higher than in 2004 and occurred between 20 and 30 cm depth, however pH did not drop as low as in Cell 2. These values correspond to observations of test-pits in the field: Oxidation was observed to be starting along crack and fissures in the first 10 cm of Cell 2 in 2004, whereas it was more pervasive and deeper (down to 20 cm) in 2005.

As described in Section 1 the processes of ore beneficiation creates fine grained tailings as waste. Usually this material is clay to silt size, however the reduced tailings at the Ruttan Mine site contain a fraction that is very fine grained (Section 3.1). This very

fine fraction was not observed in the oxidized tailings. Sulphides with this very fine size fraction would oxidize very quickly.

For oxidation to be observed in the field, visible secondary mineral products must develop within the tailings. These are usually in the form of yellow-orange to brown ochre deposits comprised of oxide, hydroxide, and sulphate minerals that are precipitated from the tailings pore water. Precipitates form from mine-waters when saturation products are exceeded in weathering solutions (Alpers *et al.*, 2000). Precipitation can occur as a mineral's surface undergoes incongruent dissolution, producing a rim of altered material (Alpers *et al.*, 2000). Precipitation can be the result of changing water chemistry due to a number of factors including oxidation, concentration, mixing, evaporation, and neutralization (Alpers *et al.*, 2000).

Sequential extraction results from the Ruttan tailings show that there is a considerable fraction of metals precipitated in secondary Fe-oxide/hydroxide minerals: Up to 50 wt.% of the total Fe at the top of Cell 2. In Cell 3, the amount of Fe in crystalline oxyhydroxide phases is lower than Cell 2. Iron precipitation can also be seen in thin section as rims of very fine grained to amorphous material around all sulphide grains from Cell 2. From sequential extraction data it is also clear that Cu, Zn, and other metals are being attenuated in these phases, although there is a considerable amount of Zn located in the water soluble fraction; up to 50 wt. % total near the surface of Cell 2.

The majority of secondary phases are crystalline and amorphous Fe-oxides and hydroxides. Sidenko and Sherriff (2005) identified precipitates from Ruttan surface water which showed that a high proportion of Fe was in a poorly crystalline form, indicative of schwertmannite ($Fe^{3+}_{16}O_{16}(OH)_{12}(SO_4)_2$). XRD analyses showed a range of minerals

from schwertmannite, to a jarosite-schwertmannite. The total concentration of metals in surface water precipitates varied by up to two orders of magnitude, with values ranging from 1.3 to 38 ppm for Ni, 42 to 439 ppm for Zn, and 7.4 to 976 ppm for Cu. From this study, it was found that the affinity of the metals to the precipitate rather than the solution is in the order Cu > Zn > Ni. Sequential extraction indicated that Ni and Zn accumulate primarily in poorly crystalline schwertmannite, while Cu is concentrated in crystalline phases (jarosite) as well as in amorphous (schwertmannite) material.

The final pH of the solution of the precipitate samples in this study (pH 6.3) was much higher than that of the Sidenko and Sherriff (2005) samples (pH 2.6). Under these pH conditions it is to be expected that minerals like the lepidocrocite found in this study would form (Table 4-3) (Jönsson *et al.*, 2006). Adsorption of any metals in solution to the surface of lepidocrocite would be expected in the order of Zn>Cu>Ni, as prescribed by the water chemistry (Section 4). All of the secondary precipitates observed in the Ruttan tailings have stable crystal structures. These minerals will not dissolve and re-release metals back into the tailings pore water.

Metals that are not attenuated by precipitation of secondary minerals within the tailings may also be removed from the pore water at the surface of the tailings through the precipitation of evaporite minerals, primarily sulphates. Melanterite (FeSO₄·7H₂O), rozenite (FeSO₄·4H₂O), and halotrichite (FeAl₂(SO₄)₄·22H₂O) were found in cracks and fissures on the surface of the Ruttan tailings. Zincmelanterite ((Zn,Fe)SO₄·7H₂O) and copper-melanterite ((Fe,Cu)SO₄·7H₂O) can also form by Zn and Cu partially replacing Fe in melanterite (Peterson 2003).

Sample	Location Description	Analyses	Final pH	Precipitate
RBH-1	Groundwater sample from well R-BH-1	XRD	6.3	Lepidocrocite
RBH-5	Groundwater sample from well R-BH-5	XRD	6.3	Lepidocrocite
RBH-7	Groundwater sample from well R-BH-7	XRD	6.3	Lepidocrocite
RW-1 ¹	Red stream draining a stand of trees, west end of Cell 3.	XRD, SEM EDS, EMP	3.0	Schwertmannite, (Goethite)
RW-2 ¹	Clear stream draining Cell's 1 and 3, west end of Cell 3.	XRD, SEM EDS, EMP	2.4	Schwertmannite, Jarosite
RW-4 ¹	The inflow point where the tailings drain into Ruttan Lake.	XRD, SEM EDS, EMP	2.7	Schwertmannite
RW-6 ¹	Pond in tailings that have exposed to oxidation the longest.	XRD, SEM EDS, EMP	2.3	Schwertmannite, Jarosite

Table 4-3. Precipitate samples collected from the Ruttan Mine Site water.

¹ Work completed by Sidenko and Sherriff (2005).

Metals not precipitated within the tailings or formed as evaporites on the surface will eventually be mobilized and transported by surface and groundwater. Figure 4-2 shows the flow path for zinc through the tailings.

The rate of metal movement through groundwater flow in the tailings is dependent on environmental stresses and hydrogeologic setting. The fine grained nature of tailings, with typical grainsize between <25 μ m and 1 mm (Robertson 1994), results in hydraulic conductivities between 1 x 10⁻⁶ and 1 x 10⁻².(Robertson 1994, Blair 1981, Blowes 1990, and Coggans *et al.* 1999) Porosity measurements at a number of tailings impoundments indicate mean values at approximately 0.5 (Blowes 1990, Coggans *et al.* 1999, Johnson *et al.* 2000). Similar properties were observed in the Ruttan tailings (Table 3-2).

During precipitation events, it was noted on-site that water was quick to pond on the surface of the tailings. It is thought that the rate of rainfall onto the tailings usually exceeds the saturated hydraulic conductivity of the tailings thus allowing the ponding. In addition, the tailings themselves are very moist close to the surface so there may not be a pronounced matrix suction drawing the water down into the tailings. In models of the tailings under transient conditions, the groundwater table reacted instantly to increases in flux and head. This quick reaction may be due to the 'unsaturated' tailings being very close to saturation and thus very close to the saturated hydraulic conductivity. Because the tailings near the surface are almost saturated it takes very little influx for them to reach saturated conditions.

To understand the effect of groundwater on the composition of water in the trenches, the amount of metals exiting the tailings over a given area was calculated from the metal concentrations in a well near a drainage trench, and the modeled groundwater flow conditions,. It is estimated that in 2004, 0.5 g/day Fe and 0.1 g/day Zn entered the drainage trench from Tailings Cell 1 over a 10 m section. Multiplying this by the approximately 1.9 km of trenching in the Ruttan tailings it was calculated that 95 g/day Fe and 19 g/day Zn were added from groundwater to the trench water. Prediction of the amount of metals that flow from the tailings groundwater into the trenches is complex. The draw-down of the water table will slow with time as the tailings reach equilibrium with the trench water level. This occurs because the head difference between the water table and the trench water decreases and therefore the rate of flow towards the trench will

become slower exponentially, thus contributing less metals to the trench. However, the tailings are also oxidizing continuously and releasing more metals which means that even though the rate of flow of groundwater from the tailings into the trenches is slowing the amount of metals in that groundwater is increasing.

Comparing groundwater conditions between 2004 and 2005 is difficult because all but three of the monitoring wells installed in 2004 were flooded in 2005. Water in the three wells that were measured in both years showed a increase in dissolved metal concentrations and an equally remarkable decrease in pH. Based on pore water profiles within the tailings that show metal concentrations lowering and pH levels increasing towards the groundwater table it seems improbable that the tailings groundwater quality decreased so significantly naturally. It is possible that surface pooling of low quality water around the wells drained into the well annulus affecting the chemistry of water measured in the well, creating a false change in measured groundwater chemistry. Groundwater samples taken in 2007 were similar to that measured in 2005. This can either be due to an overall change in the groundwater or it may be due to oxidation of the tailings in zone adjacent to the well.

Surface water at Ruttan is located in several different regimes: Standing ponds with no lateral flow, trenches of slow to moderate flow, and the water of Ruttan Lake. Standing ponds are generally have the lowest pH values, as low as 2.5. Metal concentrations may not be as high as in drainage trenches because the standing water is buffered by iron-oxyhydroxides and precipitate some metals as secondary minerals (Section 3).

Drainage trenches are constantly removing metals and acid produced by the tailings. Acid concentrations are not able to build up in the trenches to the level of buffering attainable in the tailings, therefore, metals do not precipitate out as secondary products. Based on a stream flow calculation (Section 3.5) and metal concentrations measured at the discharge point of the trenches (Appendix XII), an estimated 27 kg/day Fe and 10 kg/day Zn can enter Ruttan Lake from the tailings drainage trenches. When compared with the amount of metals introduced to the trenches from the tailings groundwater, it appears that most of the metals introduced to the trenches are from surface waters.

The immediate ponding of water on the surface of the tailings during rain events allows pools of low pH, metal laden water that were once stagnant on the surface of these tailings to join together, and eventually flow into the drainage trenches (Figure 4-3). Thin layers of evaporite material on the surface of the tailings disappear during rain events which would account for the increase in dissolved metals and sulphate, but not the alkali elements (Na, K). Alkali concentrations would be more dilute because they are mixed with rain water and behave as conservative components. The overall effect of rain on this site would be an initial pulse of metals and sulphate as evaporite materials become reincorporated into the tailings water, followed by an overall dilution as the rain continued.

Figure 4-2. Conceptual model of metal release, attenuation, and transport for the Ruttan Mine Site. Yellow arrows show movement during dry weather.



The quality of the water in Ruttan Lake is somewhat better than the water entering it from the tailings, however, the pH levels are much lower and metal concentrations much higher than meteoric conditions. The volume of the Lake as well as natural sources of fresh water help to dilute the water coming off the Ruttan waste, however, it does not appear that there are neutralization processes taking place as secondary minerals were not observed along the edges of the lake and has a pH of 2.7 to 3.2.

Figure 4-3. Pools of standing surface water overflowed in 2005 due to high precipitation. At this location, water from Cell 1 overflowed into Cell 3 and hence into the Cell 3 drainage trenches.



During mining operations at Ruttan, lime was added at a discharge point to buffer the lake water and precipitate dissolved metals, this no longer occurs with Ruttan Lake being re-directed into the open pit. A study conducted during the closure planning of Ruttan had concluded that the underground workings beneath the pit would take approximately 40 years to fill, however, as of 2007 the pit was already half full.

4.3 Evolution of the Tailings

The aim of this study was to observe changes in the level of oxidation of the tailings during one year of exposure to oxidizing conditions. There are two main factors, both mineralogical, that appear to be the primary factors affecting the rate of tailings oxidation. First, the tailings comprise approximately 5 to 10 wt. % pyrrhotite, a highly reactive sulphide mineral that releases protons and indirectly Fe^{3+} into adjacent pore waters. Secondly, the rate of oxidation is affected by the buffering capacity of the tailings. Since the amount of carbonate minerals in the tailings are able to oxidize until they reach pH of the Al(OH)₃ (pH 4 to 4.5) and Fe(OH)₃ (pH 2.5 to 3.5) buffering. This produced a noticeable increase in the depth and level of oxidation in Cell 3 from 2004 to 2005.

As the tailings in Cell 3 continue to de-water and the water table drops, it is expected that oxygen will continue to diffuse deeper into the tailings creating a deepening zone of oxidation. Modelling shows that sulphide-rich tailings with a deep water table can oxidize for centuries (Blowes and Jambor, 1990). Blowes *et al.* (2003) summarized and outlined a model of the geochemical evolution of tailings. It illustrates the location and movement of metals as the tailings oxidize. Metals are leached from solid material and enter the pore water, in the early stages of oxidation they are quickly precipitated near the surface of the tailings as secondary metals at the base of the zone of oxidation. As oxidation progresses, metal laden low-pH waters migrate down through the tailings and the zone of oxidation and metal precipitation deepens. In the final stages of oxidation, secondary metals are re-dissolved by low pH waters and primary metal sulphides are either completely dissolved or coated in a rim of secondary products that shield the sulphides from further oxidation. Pore water composition, near the surface of the tailings, return to

more meteoric conditions because there is no longer oxidation of sulphides taking place in this zone.

A comparison of the conditions at Ruttan with Blowes' conceptual model suggests that much of Ruttan in 2005 was still in the early to moderate stage of oxidation (Figure 4-4 and Figure 4-5). In both Cells 2 and 3, there is at least a minor depletion in solid metal concentrations near the surface and a peak in dissolved metal concentrations just below the top of the tailings. Although it appears that there is a zone of lower metal content of the pore water near the surface of Cell 2 in 2005, it is probably due to dilution from heavy rainfall and less likely that primary sulphides have been totally oxidized. Pore water metal concentrations do not increase with depth, rather, they peak within the first meter of the tailings, further suggesting a juvenile oxidation stage. This early stage oxidation is not surprising for Cell 3, as these tailings were only very recently introduced to oxidizing conditions. As a comparison, porewater metals concentrations within the Sherridon tailings remain high (for example, up to 80 g/L Fe) down to the water table at 4 m depth.

Figure 4-4. Total metal concentrations in solid tailings and in tailings pore water with depth in Cell 2. Blue diamonds represent data from 2004 and red squares represent data from 2005. The solid concentration curve is a total of metals in sulphides and secondary metal oxides. The dissolved concentration represents total metals in tailings pore water.



Figure 4-5. Metal concentrations in solid tailings and in tailings pore water with depth in Cell 3. Blue diamonds represent data from 2004 and red squares represent data from 2005. The solid concentration curve is a total of metals in sulphides and secondary metal oxides. The dissolved concentration represents total metals in tailings pore water.



Cell 3

Due to the noticeable change in level of oxidation in Cell 3 from 2004 to 2005, one would expect the tailings in Cell 2 to have oxidized to a greater extent given the amount of time they have been exposed. This may be due to the fine-grained tailings maintaining a level of saturation that does not allow for the diffusion of oxygen to great depths. Under these conditions initial oxidation at the tailings surface would proceed quickly, however, oxidation below this zone would be limited by the speed at which the tailings are able to drain. In years of heavy rain events, such as that observed in 2005, the rate of oxidation would be slowed. Given that the tailings are greater than 10 m thick and are only just beginning to oxidize, it can be predicted that these tailings will be oxidizing for many years.

Ruttan Lake is affected by oxidation changes in the Ruttan tailings (Figure 3-24). Given that a large amount of tailings have yet to be oxidized it should be expected that the amount of metals in Ruttan Lake will increase and pH levels will lower as oxidation in the tailings proceeds. However, due to its constant flow into the Ruttan open pit and partial recharge by metoric waters it is doubtful that conditions in the lake will became as acidic as the ponds on the tailings surface. Lake waters associated with the Sherridon Mine, unlike Ruttan Lake, are part of an 'open system' whereby headwaters are allowed to flow through the site. However, similar to Ruttan Lake, lake water adjacent to the Sherridon tailings also have low pH values (down to pH 2.8) (Saquet *et al.* 2002).

5 Conclusions and Suggestions for Future Research

The purpose of this study was to record the state of the Ruttan tailings during its early stages of oxidation and to observe changes in the tailings from one year to the next. Further, the presence of already oxidizing tailings at the site allowed a comparison between tailings at different stages of oxidation. Unfortunately anomalous precipitation events in 2005 may have affected the data, however a number observations remain clear:

- The depth of oxidation in Tailings Cell 3 increased and became more pervasive over the course of one year.
- A very fine grained fraction in the reduced tailings, not described in other studies, contributed to the rapid oxidation of the tailings in Cell 3.
- Mineralogically respective weathering rates of sulphide minerals is similar to that observed in other studies.
- The buffering capacity of the tailings is low, allowing the onset of low pH conditions to occur rapidly.
- Large volumes of tailings and slow drainage conditions will lead to acid mine drainage conditions at this site for many years.

Metals are temporarily attenuated from waters associated with the tailings either as evaporite minerals at the surface of the tailings, or as buffering by-product precipitates in pore water and surface waters. These secondary minerals are only temporary metal sinks as they can re-dissolve. Evaporites will re-dissolve in wet weather conditions, such as that seen in 2005. Amorphous precipitates will re-dissolve if the buffering regime of adjacent solutions changes to a lower pH buffering state.
High dissolved metal concentrations and low pH conditions are associated with pore-, ground-, and surface-waters that are in contact with the tailings and mine waste. Any attempt to re-direct water off this site within the near future will require treatment of the discharge water to remove the dissolved metals and return pH levels to more neutral conditions.

Remediation activities at this site would have to be on a major scale. To stop the tailings from generating acidic conditions would require the application of a cover material that would halt the diffusion of oxygen and meteoric waters into the tailings. Considering that the tailings were previously under a wet cover and subsequently drained due to stability issues with containment dykes, dry cover of clay, lime, and organics would be the preferred cover. Due to the size of the tailings, the cost of materials and labour needed to set up an organic dry cover would be very large.

Once the on-site open pit fills completely, treatment of discharge water will need to occur for the discharge water to be within provincial guidelines.

Further studies of oxygen diffusion into the tailings and humidity Cell leach tests, would help to define the rate of oxidation within the tailings and possibly allow for calculations to determine a timeline for the progression of oxidation (Lapakko, 2003).

A drill program would be beneficial in determining the depth of the tailings material and to determine the level of homogeneity within the tailings. It is expected that as the mill-discharge spigot was moved around, lenses of varying grainsize would be present. A comprehensive drill program would be able to define these zones and be used to create a more detailed groundwater flow regime. This flow model would allow for a more

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accurate depiction of groundwater drawdown near drainage trenches and could also be used for oxidation timeline estimations.

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Appendices

Appendix I Water Sampling Location Descriptions

Water sampl	Nater samples collected at the Ruttan Mine site.								
Sample ID	Site ID	Sample Type	Dates Sampled	Description					
RW-1	Cell 3	Surface – Creek	September 2003 June 2004 July 2007	Red stream draining a stand of trees, west end of Cell 3.					
RW-2	Cell 3	Surface – Creek	September 2003 June 2004 June 2004 (after Rain) October 2004 July 2007	Clear stream draining Cell's 1 and 3, west end of Cell 3.					
RW-3	Lake Outflow	Downstream – Creek	June 2004 October 2004 August 2005 July 2007	The outflow point where Ruttan Lake empties into the open-pit.					
RW-4	Lake Inflow	Downstream – Creek	September 2003 June 2004 September 2004 August 2005	The inflow point where the tailings drain into Ruttan Lake.					
RW-5	Cell 1	Surface – Creek	June 2004 August 2005	East end of drainage trench in Cell 1.					
RW-6	Oldest Tailings	Surface – Pond	September 2003 June 2004	Pond in tailings that have exposed to oxidation the longest.					
RW-7	Mill Creek	Creek	June 2004	Creek draining the mill-pond into the open-pit.					
RW-8	South Lake	Surface – Lake	June 2004 August 2005	Southwest corner of Ruttan Lake, opposite RW-4.					
RW-9	North Lake	Surface – Lake	June 2004 August 2005	Northwest corner of Ruttan Lake at the former drainage point.					

Water samples collected at the Ruttan Mine site cont'd.						
Sample ID	Site ID	Sample Type	Dates Sampled	Description		
RW-12	Cell 3	Surface – Pond	June 2004	Pond at the east of Cell 3, green streamers located in pond.		
RW-13	Cell 3	Surface – Pond	June 2004	Pond at the east end of Cell 3.		
R-BH-1	Cell 3	Well	June 2004 September 2004 October 2004 August 2005	Groundwater sample from R-BH-1.		
R-BH-2	Cell 3	Well	July 2007	Groundwater Sample from R-BH-2		
R-BH-3	Cell 3	Well	September 2004	Groundwater sample from R-BH-3		
R-BH-4	Cell 3	Well	September 2004 July 2007	Groundwater sample from R-BH-4		
R-BH-5	Cell 1	Well	September 2004 August 2005 July 2007	Groundwater sample from R-BH-5		
R-BH-6	Cell 1	Well	July 2007	Groundwater sample from R-BH-6		
R-BH-7	Cell 1	Well	September 2004 August 2005 July 2007	Groundwater sample from R-BH-7		
R-BH-8	Cell 2	Well	September 2004	Groundwater sample from R-BH-8		
R-BH-9	Cell 3	Well	Not Sampled			
R-BH-10	Cell 3	Well	September 2004 July 2007	Groundwater sample from R-BH-10		
R-BH-11	Cell 3	Well	September 2004 July 2007	Groundwater sample from R-BH-11		
R-BH-12	Cell 3	Well	Not Sampled			

Appendix II SEM/EDS Analyses





Spc. 1 - Grain surrounded by alteration.

Elem W	/t. %	At %	K-Ratio
--------	-------	------	---------

MgK	4.17 6.46	5.47	0.0145
SiK	34.45	39.12	0.1562
SK	33.64	33.46	0.1197
K K Cak	2.81	2.29	0.0119
TiK	0.97	0.65	0.0057
FeK	11.38	6.50	0.0909
Total	100.00	100.0)0



Spc. 2 - Alteration

lem Wt. %	At % K-Ratio
-----------	--------------

S K 27.75 40.08 0.1284 FeK 72.25 59.92 0.6669 Total 100.00 100.00

R-4/03 - East End of Cell 3, Reduced



Spc. 4 – 'Calc-Silicate'



0.90

1.80

2.70

109

7.20

4.50

5.40

6.30

3.60

Spc. 6 - Titanium

Elem Wt. % At % K-Ratio

TiK 98.32 98.56 0.9821 FeK 1.68 1.44 0.0105 Total 100.00 100.00



Spc. 7 – Quartz

Elem Wt. % At % K-Ratio

---SiK 98.12 99.04 0.9242 FeK 1.88 0.96 0.0155 Total 100.00 100.00

Spc. 8 – Groundmass

Elem Wt. % At % K-Ratio

SiK 3.72 5.49 0.0120 S K 38.41 49.66 0.1941 CaK 6.48 6.70 0.0395 FeK 51.40 38.15 0.4426 Total 100.00 100.00



Spc. 9 – Mica

Elem Wt. % At % K-Ratio

-			
	MgK	11.50 15.19 0.0357	
	AIK	14.40 17.13 0.0438	
	SiK	35.04 40.06 0.1060	
	ΚK	20.09 16.50 0.1124	
	TiK	2.21 1.48 0.0139	
	FeK	16.76 9.64 0.1376	
٦	Total	100.00 100.00	



Spc. 10 – Groundmass

SiK	11.17	15.81	0.0400
SK	24.97	30.97	0.1194
ΚK	4.67	4.75 0	.0267
CaK	22.60	22.42	0.1419
FeK	36.58	26.04	0.2929
Total	100.00	100.00	

Elem Wt. % At % K-Ratio

Spc. 11 – 'Calc-Silicate' (Pyrite Interferrence)		Ca	
Elem Wt. % At % K-Ratio			
AlK 23.59 27.36 0.1190 SiK 39.23 43.70 0.1338 S K 2.60 2.53 0.0076 CaK 31.92 24.92 0.2052 FeK 2.66 1.49 0.0203 Total 100.00 100.00			
	S 1.00 2.00	3.00 4.00 5.	Fe Fe
Spc. 12 – 'Calc-Silicate' with Mg, Fe, Al			
	Si		
MgK 12.75 16.48 0.0423 AlK 4.46 5.19 0.0139 SiK 47.37 53.00 0.1800		Ca	
CaK 24.37 19.11 0.1571 FeK 11.06 6.22 0.0872			
I otal 100.00 100.00			
			Fe

i in a

1.00

2.00

3.00

4.00

5.00

6.00

Spc. 13 – Metal Concentration

Elem Wt. % At % K-Ratio

---AlK 46.26 66.94 0.0868 FeK 9.45 6.60 0.0939 ZnK 44.29 26.45 0.4077 Total 100.00 100.00



Spc. 15 – Calc-Sulphale	Spc.	spc	DC. 15	– Ca	ic-Su	pnate	Э
--------------------------------	------	-----	--------	------	-------	-------	---

				Ca					
		5							
	2	2.00		4	.00			6.00	
 s									
						Fe	e		
			2.00	S				Ca Ca S S Ca A S Ca A Ca Ca Ca Ca Ca Ca Ca Ca Ca Ca	Ca Ca S S Ca Ca Ca Ca Ca Ca Ca Ca Ca Ca

2.00

4.00

6.00

8.00

Spc. 17 – Pyrrhotite

Elem Wt. % At % K-Ratio

---S K 43.84 57.62 0.2317 FeK 56.16 42.38 0.4953 Total 100.00 100.00



Spc. 18 – Groundmass

Elem Wt. % At % K-Ratio

AIK 12.73 15.11 0.0549 SiK 46.77 53.33 0.1871 S K 14.20 14.19 0.0404 K K 4.84 3.97 0.0224 CaK 4.89 3.91 0.0260 FeK 16.56 9.50 0.1362 Total 100.00 100.00 **Spc. 19** – RTP-2 Box 1 – Alteration

Elem Wt. % At % K-Ratio

-			
	MgK	9.20	13.26 0.0211
	AIK	4.35	5.65 0.0110
	SiK	28.76	35.89 0.0914
	SΚ	13.57	14.84 0.0446
	ΚK	1.09	0.97 0.0057
	CaK	9.66	8.45 0.0600
	FeK	33.37	20.94 0.2823
٦	Fotal	100.00	100.00



HP Ex 2 – North End of Cell 2, Hardpan



Spc. 20 – HP Ex 2 – Chalcopyrite inside Quartz

Elem	Wt.	%	At %	K-Ratio
------	-----	---	------	---------

S K 39.84 55.20 0.1794 FeK 28.46 22.64 0.2646 CuK 31.70 22.16 0.2589 Total 100.00 100.00



Spc. 21 – HP Ex 2 – Amphibole?

Elem Wt. % At % K-Ratio

MgK	7.99	10.74	0.0213
AIK	17.55	21.25	0.0511
SiK	40.39	46.99	0.1117
ΚK	4.32	3.61 0).0223
FeK	29.76	17.41	0.2604
Total	100.00	100.00)



HP Ex 2 – North End of Cell 2, Hardpan



Spc. 22 – HP Ex 2 – Pyrrhotite

Elem Wt. % At % K-Ratio

S K 43.62 57.40 0.2302

FeK 56.38 42.60 0.4975

Total 100.00 100.00

Image: Spc in the image in the ima

Spc. 23 – HP Ex 2 – Rim

Elem Wt. % At % K-Ratio

S K 18.44 28.25 0.0796 FeK 81.56 71.75 0.7731 Total 100.00 100.00



2.00

3.00

4.00

5.00

6.00

7.00

HP Ex 2 – North End of Cell 2, Hardpan



Spc. 26 – HP Ex 2 Sphalerite

Elem Wt. % At % K-Ratio

S K 38.09 55.23 0.1441 FeK 6.12 5.09 0.0581 ZnK 55.79 39.68 0.5096 Total 100.00 100.00



Spc. 27 -	– HP Ex 2 S	phalerite Rim
-----------	-------------	---------------

Elem Wt. % At % K-Ratio

SK	39.10	51.09	0.2137
ΚK	10.05	10.76	0.0510
FeK	50.85	38.15	0.4368
Total	100.00	100.00)



HP Ex 2 – North End of Cell 2, Hardpan



Spc. 28 HP Ex 2 Sphalerite 2

Elem Wt. % At % K-Ratio

S K 38.05 55.16 0.1442 FeK 6.64 5.53 0.0631 ZnK 55.30 39.32 0.5042 Total 100.00 100.00



Spc. 29 HP Ex 2 Rutile

|--|

TiK 100.00 100.00 1.0000 Total 100.00 100.00



Spc. 30 HP Ex 2 – Chalcopyrite in Mica

Elem Wt. % At % K-Ratio

Cur 20.94 12.70 0.1711	AIK	8.15	11.72	0.0195
	SiK	10.72	14.80	0.0305
	S K	32.48	39.28	0.1302
	CaK	7.96	7.71	0.0462
	FeK	19.76	13.72	0.1723
Total 100.00 100.00	FeK	19.76	13.72	0.1723
	CuK	20.94	12.78	0.1711
	Total	100.00	100.00)



Spc. 31 HP Ex 2 – Mica surrounding Chalco.

Elem	Wt.	%	At %	K-Ratio
	ννι.	/0	Πι /0	I VII VALIO

AIK	20.38	22.67	0.1234
SiK	55.41	59.20	0.2214
CaK	24.21	18.13	0.1461
Total	100.00	100.00)







Spc. 32 HP Ex 2 – Pyrrhotite (zoned rim)

Elem Wt. % At % K-Rati	0
------------------------	---

S K 44.08 57.86 0.2335 FeK 55.92 42.14 0.4928 Total 100.00 100.00



HP Ex 2 – North End of Cell 3, Hardpan



Spc. 33 HP Ex 2 – Chalcopyrite 2 in grain

Elem Wt. % At % K-Ratio

S K 39.65 55.01 0.1782 FeK 28.50 22.70 0.2652 CuK 31.85 22.30 0.2601 Total 100.00 100.00



Spc. 34 HP Ex 2 – Grain surrounding said Chalco

Elem Wt. % At % K-Ratio

MgK 8.75 11.96 0.0234 AlK 9.07 11.17 0.0261 SiK 39.52 46.78 0.1276 CaK 20.09 16.66 0.1335 FeK 22.58 13.44 0.1844 Total 100.00 100.00



Spc. 35 HP Ex 2 – More grain surrounding said Chalco

Elem Wt. % At % K-Ratio

NaK5.747.060.0238AIK16.2917.080.1008SiK69.1269.610.3077CaK8.856.250.0491Total100.00100.00

HP Ex 2 – North End of Cell 2, Hardpan



Spc. 36 HP Ex 2 – Pyrite and gypsum?

Elem	Wt.	%	At %	6 K-Ratio
			/ /	

S K 35.33 44.04 0.2178 CaK 34.39 34.29 0.2265 FeK 30.28 21.67 0.2321 Total 100.00 100.00



HP Ex 2 – North End of Cell 2, Hardpan



Spc 37 HP Ex 2 - Plag

Elem	Wt. % At % K-Ratio	
ZnL AlK SiK CaK	7.46 3.52 0.0466 20.62 23.57 0.0956 53.47 58.72 0.1845 18.45 14.20 0.1096	



HP Ex 2 – North End of Cell 2, Hardpan



Spc 38 HP Ex 2 – Sphalerite Alteration

Elem Wt. % At % K-Ratio

S K 26.77 38.90 0.1229 FeK 73.23 61.10 0.6778 Total 100.00 100.00



Spc. 39 HP Ex 2 – Alteration near sphalerite grain

Elem Wt. % At % K-Ratio

SΚ	38.45	54.27	0.1606
FeK	26.42	21.41	0.2435
ZnK	35.13	24.32	0.2991
Total	100.00	100.00)



Spc. 40 HP Ex 2 – Sphalerite Grain

Elem Wt. % At % K-Ratio

S K 38.40 55.49 0.1463 FeK 7.01 5.82 0.0664 ZnK 54.59 38.69 0.4968 Total 100.00 100.00


Spc. 41 TP-2 Box 1 – Zn, Fe, Al Oxide?

Elem Wt. % At % K-Ratio

AIK	45.82	66.56 0.0854
FeK	9.36	6.57 0.0931
ZnK	44.82	26.87 0.4129
Total	100.00	100.00



R-TP-2 Cell 3, Reduced Tailings



2.00

3.00

4.00 5.00 6.00 7.00 8.00

Spc 43. –	TP-2 Box 1	Calc silicate
-----------	------------	---------------

Elem	Wt.	%	At %	K-Ratio
------	-----	---	------	---------

AIK	19.14	21.39	0.1138
SiK	55.37	59.44	0.2265
CaK	25.49	19.18	0.1548
Total	100.00	100.00)



2.00

3.00

4.00

5.00

6.00

S K 43.55 57.33 0.2296 FeK 56.45 42.67 0.4983 Total 100.00 100.00

Elem Wt. % At % K-Ratio

Spc 44. - TP-3 Surface Phyrrhotite

7.00

HP Ex 2 – North End of Cell 2, Hardpan



Spc. 45 - Chalcopyrite

Elem	Wt. % At % K-Ratio
SK	33.76 48.65 0.1439
FeK	31.74 26.26 0.3012
CuK	34.51 25.09 0.2813
Total	100.00 100.00



Spc. 46

Elem	Wt. % At % K-Ratio)
SK	30.10 42.91 0.1412	
FeK	68.76 56.27 0.6311	
CuK	1.14 0.82 0.0081	
Total	100.00 100.00	



2.00

3.00

4.00

5.00

6.00

7.00

Spc. 47

Elem Wt. % A	t % K-Ratio
--------------	-------------

S K10.6017.140.0432FeK88.3081.960.8570CuK1.100.900.0075Total100.00100.00

HP Ex 2 – North End of Cell 2, Hardpan



Spc. 48

Elem	Wt. % At % K-Ratio	
MgK AlK SiK FeK Total	16.64 22.28 0.0421 19.55 23.60 0.0460 29.34 34.02 0.0677 34.47 20.10 0.3096 100.00 100.00	



Appendix III Solid Tailings Chemical Analyses - 2004

2004 0			10.20 om	20 20 om	50 60 om
		0-10 Cm	10-20 Cm	20-30 Cm	50-60 Cm
Al	ppm	29000	33500	10000	28600
As	ppm	0.0	0.00	0.00	0.00
В	ppm	0.0	0.00	0.00	0.00
Ba	ppm	134.0	150	87.3	152
Be	ppm	0.0	0.00	0.00	0.00
Ca	ppm	19200	27200	14900	18000
Cd	ppm	5.40	8.6	9.40	0.60
Co	ppm	106	81.1	54.5	56.7
Cr	ppm	24.8	40.7	41.9	41.1
Cu	ppm	537	863	506	661
Fe	ppm	230000	182000	248000	201000
K	ppm	4210	5430	3520	6360
Mg	ppm	15300	23100	5830	18500
Mn	ppm	460	495	339	524
Мо	ppm	0.00	0.00	0.00	0.00
Na	ppm	4210	4700	2600	3910
Ni	ppm	29.1	30.0	12.5	29.2
Р	ppm	0.00			
Pb	ppm	930	1060	938	746
Si	ppm	90900	102000	114000	107000
Sr	ppm	113	114	70.4	80.4
Ti	ppm	989	895	509	883
U	ppm	0.00	0.00	0.00	0.00
V	ppm	22.9	23.8	33.2	26.9
Zn	ppm	2330	3090	1920	1370

2004 C	2004 Cell 2 Solid Tailings Chemical Analyses – Sampling Sites: RTP-3 and RTH-1							
		0-17 cm	17-27 cm	45-56 cm	120 cm	370 cm		
Al	ppm	14300	9160	26900	10900	10400		
As	ppm	0.00	0.00	0.00	123	114		
Ba	ppm	138	92.2	117	112	110		
Ca	ppm	18500	16100	20300	15900	12800		
Cd	ppm	6.70	3.00	6.50	10.9	8.30		
Со	ppm	0.00	66.0	88.8	67.6	91.4		
Cr	ppm	22.8	23.7	35.6	49.7	17.5		
Cu	ppm	350	220	1040	762	940		
Fe	ppm	202000	224000	211000	259000	281000		
K	ppm	4830	4100	4870	4540	4370		
Mg	ppm	5310	6270	17700	6190	7200		
Mn	ppm	373	364	449	370	301		
Na	ppm	3680	3070	3710	3050	2830		
Ni	ppm	9.00	9.40	34.7	14.6	10.3		
Pb	ppm	1080	752	631	1110	847		
Si	ppm	126000	137000	88500	112000	84400		
Sr	ppm	99.3	93.5	83.9	78.6	59.3		
Ti	ppm	966	864	757	697	638		
V	ppm	34.5	34.3	29.0	41.1	27.9		
Zn	ppm	1140	572	2740	2920	1610		

Appendix IV Solid Tailings Chemical Analyses – 2005

2005 Ce	ell 3 Solid T	ailings Chemi	cal Analyses	- Sampling S	Site: RTP-7
		0-10 cm	10-20 cm	20-30 cm	50-60 cm
Al	ppm	25200	29300	29300	29000
As	ppm	0.00	0.00	0.00	0.00
В	ppm	0.00	0.00	0.00	0.00
Ba	ppm	123	122	150	105
Be	ppm	0.00	0.00	0.00	0.00
Ca	ppm	15900	14400	20000	14300
Cd	ppm	0.00	0.00	0.00	3.60
Со	ppm	115	81.7	631	88.9
Cr	ppm	24.6	77.4	36.4	61.0
Cu	ppm	286	511	393	466
Fe	ppm	201000	189000	176000	228000
K	ppm	5650	5340	5510	4070
Mg	ppm	18500	18400	17500	14100
Mn	ppm	404	431	460	438
Мо	ppm	0.00	0.00	0.00	0.00
Na	ppm	4020	4210	4770	3270
Ni	ppm	24.4	30.6	20.6	26.0
Pb	ppm	676	558	641	697
Si	ppm	82000	99700	103000	89400
Sr	ppm	94.1	92.7	116	73.7
Ti	ppm	989	870	877	780
U	ppm	0.00	0.00	0.00	0.00
V	ppm	26.1	26.8	27.4	22.3
Zn	ppm	405	637	786	1410

2005 Cell 2 Solid Tailings Chemical Analyses – Sampling Sites: RTP-5 and RTH-3									
		0-10 ppm	10-20 ppm	50 cm	100 cm	250 cm	300 cm	400 cm	
Al	ppm	26100	25100	32200	31900	33800	27100	31400	
Ва	ppm	167	192	108	142	124	132	133	
Ca	ppm	24000	20400	18200	16600	20000	14600	20100	
Cd	ppm	0.00	0.00	21.4	6.80	6.80	2.20	14.3	
Со	ppm	299	63.5	119	71.9	121	120	281	
Cr	ppm	47.8	45.7	57.4	58.8	35.5	28.7	28.1	
Cu	ppm	532	328	882	659	934	630	791	
Fe	ppm	166000	166000	207000	204000	203000	214000	201000	
K	ppm	6490	6800	4120	5180	5140	5770	4720	
Mg	ppm	17600	16500	18800	22400	27500	20000	19200	
Mn	ppm	421	453	404	465	548	425	433	
Na	ppm	5150	5240	3240	4150	3810	3920	4570	
Ni	ppm	21.5	24.9	61.6	27.7	21.5	25.8	27.4	
Pb	ppm	1590	619	682	788	575	567	911	
Si	ppm	113000	122000	83600	91400	95300	102000	101000	
Sr	ppm	131	123	78.8	95.6	84.3	88.5	108	
Ti	ppm	964	970	769	736	801	847	885	
V	ppm	28.7	29.1	26.6	27.0	22.0	24.1	25.3	
Zn	ppm	575	315	2060	4620	2750	1390	1950	

Appendix V Sequential Extraction Results For Iron

1 tooulto of boquor			10111-00	lia bain			, 0.					
			W	ater	Ex	ch. +	Am	. Fe-	Cry.	Fe-		
Sample	Thickness	Total	sol	uble	Ca	rbon.	hyo	drox.	hyd	rox.	Res	idual
				% of		% of		% of		% of		% of
	cm	wt.%	wt.%	tot.	wt.%	tot.	wt.%	tot.	wt.%	tot.	wt.%	tot.
RTP-1 (Cell 3 – 2	004)											
0-10	10	23	0.08	0.4	0.07	0.3	1.2	5	3.1	14	18	80
10-20	10	25	0.09	0.4	0.12	0.5	1.3	5	2.3	9	21	85
20-30	10	25	0.09	0.4	0.12	0.5	1.3	5	2.3	9	21	85
50-60	10	20	0.13	0.7	0.09	0.4	1.3	7	2.1	10	16	82
Weighted mean*		23	0.10	0.5	0.10	0.4	1.3	6	2.5	11	19	83
RTP-3 (Cell 2 – 2	004)											
0-17	17	20	0.19	0.9	0.02	0.1	0.8	4	5.7	28	13	67
17-27	10	22	0.04	0.2	0.02	0.1	0.6	3	4.9	22	17	75
45-56	11	21	0.09	0.4	0.09	0.4	1.0	5	2.8	13	17	81
Weighted mean*		21	0.12	0.6	0.04	0.2	0.8	4	4.7	22	15	73
RTH-1 (Cell 2 – 2	004)											
120	10	26	0.12	0.5	0.06	0.2	1.3	5	2.9	11	22	83
370	10	28	0.13	0.5	0.05	0.2	1.5	5	5.2	19	21	76
Weighted mean*		27	0.12	0.5	0.05	0.2	1.4	5	4.1	15	21	79

Results of sequential extraction of Fe from solid samples of the tailings.

							35 (55.15	- /				
			W	ater	Ex	ch. +	Am	. Fe-	Cry	. Fe-		
Sample	Thickness	Total	so	luble	Ca	rbon.	hyo	drox.	hyd	lrox.	Res	idual
				% of		% of		% of		% of		% of
	cm	wt.%	wt.%	tot.	wt.%	tot.	wt.%	tot.	wt.%	tot.	wt.%	tot.
RTP-7 (Cell 3 – 2	005)											
0-10	10	20	0.19	0.9	0.02	0.1	0.9	4	5.2	26	14	69
10-20	10	19	0.04	0.2	0.04	0.2	0.7	4	4.1	22	14	74
20-30	10	18	0.12	0.7	0.03	0.2	0.9	5	5.4	31	11	63
50-60	10	23	0.02	0.1	0.07	0.3	1.4	6	2.9	13	18	81
Weighted mean*		20	0.10	0.5	0.04	0.2	1.0	5	4.4	23	14	72
RTP-5 (Cell 2 – 2	005)											
0-10	10	17	0.18	1.1	0.01	0.1	1.1	6	7.2	43	8	49
10-20	10	17	0.14	0.8	0.00	0.0	0.8	5	6.2	37	9	57
50	5	21	0.05	0.3	0.05	0.2	1.6	8	3.1	15	16	77
Weighted mean*		17	0.14	0.8	0.01	0.1	1.1	6	6.0	35	10	58
RTH-3 (Cell 2 – 2	2005)											
100	10	20	0.02	0.1	0.04	0.2	1.3	6	3.2	15	16	78
250	10	20	0.01	0.1	0.07	0.3	1.2	6	4.0	20	15	74
300	10	21	0.10	0.5	0.18	0.8	1.0	5	1.1	5	19	89
400	10	20	0.03	0.2	0.07	0.3	1.5	7	3.5	17	15	75
Weighted mean*		21	0.04	0.2	0.09	0.4	1.2	6	2.9	14	16	79

Results of sequential extraction of **Fe** from solid samples of the tailings (cont'd).





Appendix VI Sequential Extraction Results For Zinc

									Cry.	Fe-		
Sample	Thickness	Total	Water s	oluble	Exch. +	Carbon.	Am. Fe-ł	nydrox.	hydr	ΌΧ.	Resid	dual
				% of		% of		% of		% of		% of
	cm	wt.%	wt.%	tot.	wt.%	tot.	wt.%	tot.	wt.%	tot.	wt.%	tot.
RTP-1 (C	ell 3 – 2004)											
0-10	10	0.350	0.061	18	0.016	4.6	0.007	1.9	0.064	18	0.201	58
10-20	10	0.464	0.127	27	0.000	0.0	0.007	1.4	0.053	11	0.244	53
20-30	10	0.288	0.026	9	0.017	5.9	0.008	2.9	0.036	13	0.201	70
50-60	10	0.205	0.079	38	0.007	3.4	0.005	2.6	0.022	11	0.092	45
Weighted	mean*	0.327	0.073	23	0.010	3.5	0.007	2.2	0.044	13	0.185	56
RTP-3 (C	ell 2 – 2004)											
0-17	17	0.170	0.206	56	0.012	6.8	0.005	3.0	0.049	29	0.009	5
17-27	10	0.086	0.024	28	0.007	7.9	0.003	3.8	0.029	34	0.022	26
45-56	11	0.411	0.107	26	0.024	5.8	0.006	1.5	0.052	13	0.222	54
Weighted	mean*	0.218	0.129	40	0.014	6.8	0.005	2.7	0.045	26	0.074	25
RTH-1 (C	ell 2 – 2004)											
120	10	0.438	0.159	36	0.034	7.8	0.009	2.0	0.049	11	0.188	43
370	10	0.241	0.092	38	0.022	9.1	0.008	3.4	0.053	22	0.066	27
Weighted	mean*	0.339	0.125	37	0.028	8.4	0.008	2.7	0.051	17	0.127	35

Results of sequential extraction of **Zn** from solid samples of the tailings.

									Cry.	Fe-		
Sample	Thickness	Total	Water s	oluble	Exch. +	Carbon.	Am. Fe-ł	nydrox.	hydr	οx.	Resid	dual
-				% of		% of		% of	-	% of		% of
	cm	wt.%	wt.%	tot.	wt.%	tot.	wt.%	tot.	wt.%	tot.	wt.%	tot.
RTP-7 (C	ell 3 – 2005)											
0-10	10	0.061	0.010	16	0.001	2.4	0.001	1.4	0.018	29	0.031	51
10-20	10	0.096	0.019	20	0.003	3.3	0.002	2.3	0.018	19	0.053	55
20-30	10	0.118	0.029	25	0.004	3.5	0.003	2.5	0.035	29	0.047	40
50-60	10	0.211	0.078	37	0.011	5.2	0.005	0.0	0.021	10	0.097	46
Weight	ed mean*	0.121	0.034	25	0.005	3.6	0.003	1.6	0.023	22	0.057	48
RTP-5 (C	ell 2 – 2005)											
0-10	10	0.086	0.007	8	0.003	3.3	0.003	3.5	0.048	56	0.025	29
10-20	10	0.047	0.006	13	0.001	2.1	0.002	3.2	0.021	45	0.017	37
50	5	0.310	0.128	41	0.018	5.8	0.003	0.9	0.037	12	0.124	40
Weight	ed mean*	0.115	0.031	17	0.005	3.3	0.002	2.8	0.035	43	0.042	34
RTH-3 (C	ell 2 – 2005)											
100	10	0.693	0.254	37	0.050	7.2	0.009	1.3	0.059	8	0.322	46
250	10	0.412	0.053	13	0.030	7.3	0.017	4.0	0.105	25	0.208	50
300	10	0.208	0.021	10	0.014	6.9	0.009	4.3	0.023	11	0.142	68
400	10	0.292	0.079	27	0.015	5.2	0.004	1.5	0.045	15	0.148	51
Weight	ed mean*	0.401	0.102	22	0.027	6.7	0.010	2.8	0.058	15	0.205	54

Results of sequential extraction of **Zn** from solid samples of the tailings (cont'd).



Figure 0-1. Sequential extraction results for Zn (ppm) in Tailings Cell 2, 2004 (A), Tailings Cell 2, 2005 (B), Tailings Cell 3, 2004 (C), and Tailings Cell 3, 2005 (D). Depths are measured in cm below the surface of the tailings. Sequential Extraction

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Appendix VII Sequential Extraction Results For Copper

Results o	f sequential ex	traction o	of Cu fro	m solid s	amples of	the tailin	gs.					
Sample	Thickness	Total	Water	soluble	Exch. +	Carbon.	Am. Fe-ł	nydrox.	Cry. Fe-h	ydrox.	Residua	I
-				% of		% of		% of	-	% of		% of
	cm	wt.%	wt.%	tot.	wt.%	tot.	wt.%	tot.	wt.%	tot.	wt.%	tot.
RTP-1 (C	ell 3 – 2004)											
0-10	10	0.054	0.004	7	0.004	7.6	0.006	11.3	0.016	29	0.024	44
10-20	10	0.086	0.018	21	0.009	10.5	0.010	12.0	0.023	26	0.026	30
20-30	10	0.051	0.004	9	0.009	17.0	0.009	17.8	0.009	18	0.019	38
50-60	10	0.066	0.010	16	0.004	6.0	0.007	10.1	0.012	18	0.033	50
Weighted	mean*	0.064	0.009	13	0.006	10.2	0.008	12.8	0.015	23	0.026	41
RTP-3 (C	ell 2 – 2004)											
0-17	17	0.035	0.009	24	0.002	6.2	0.005	13.3	0.022	63	0.000	0
17-27	10	0.035	0.009	24	0.002	6.2	0.005	13.3	0.022	63	0.000	0
45-56	11	0.104	0.019	18	0.008	7.9	0.009	8.5	0.172	16	0.052	49
Weighted	mean*	0.055	0.011	23	0.004	6.7	0.006	11.9	0.065	49	0.015	14
RTH-1 (C	cell 2 – 2004)											
1.2	10	0.076	0.031	40	0.009	12.3	0.010	13.1	0.023	30	0.003	4
3.7	10	0.094	0.024	25	0.010	10.4	0.011	12.0	0.029	31	0.021	22
Weighted	mean*	0.085	0.027	33	0.010	11.4	0.011	12.6	0.026	30	0.012	13

Deculto	f and upptial av	traction	f Cu fro	m aalid a	omploo of	the tailin	ao (oont'd	\				
Results 0	sequential ex				amples of		gs (cont a).	<u> </u>			
Sample	Thickness	Total	Water	soluble	Exch. + (Carbon.	Am. Fe-ł	nydrox.	Cry. Fe-hy	ydrox.	Residua	l
				% of		% of		% of		% of		% of
	cm	wt.%	wt.%	tot.	wt.%	tot.	wt.%	tot.	wt.%	tot.	wt.%	tot.
RTP-7 (C	ell 3 – 2005)											
0-10	10	0.029	0.007	23	0.001	4.8	0.003	10.6	0.011	38	0.007	24
10-20	10	0.051	0.016	32	0.004	7.7	0.004	7.9	0.015	30	0.011	22
20-30	10	0.039	0.006	16	0.001	3.3	0.002	6.3	0.020	50	0.010	25
50-60	10	0.047	0.006	13	0.004	7.8	0.006	13.8	0.018	39	0.013	27
RTP-5 (C	ell 2 – 2005)											
0-10	10	0.053	0.002	4	0.000	0.8	0.002	3.7	0.031	59	0.174	33
10-20	10	0.033	0.002	5	0.000	1.0	0.001	4.3	0.023	69	0.007	21
50	5	0.088	0.041	47	0.008	8.6	0.006	7.2	0.024	27	0.009	10
Weighted	l mean*	0.052	0.010	13	0.002	2.4	0.003	4.6	0.026	57	0.074	23
RTH-3 (C	cell 2 – 2005)											
100	10	0.066	0.013	19	0.006	9.5	0.008	12.1	0.020	30	0.019	29
250	10	0.093	0.007	8	0.015	16.2	0.014	15.4	0.013	13	0.044	47
300	10	0.063	0.001	2	0.008	13.4	0.009	15.0	0.004	6	0.040	63
400	10	0.079	0.018	22	0.008	9.8	0.008	10.3	0.030	38	0.015	19
Weighted	l mean*	0.075	0.010	13	0.009	12.3	0.010	13.2	0.017	22	0.029	39
Weighted	l mean*	0.041	0.009	21	0.003	5.9	0.004	9.6	0.016	39	0.010	24

Appendix VIII Pore water Geochemical Analyses - 2004

2004 (ilingo Doro ur	tor Chamical		malina Cita, D				
2004 (allings Pore wa	aler Chemical /	Analyses – Sal	mpling Sile: R	-12-1			
		0-10 cm	10-20 cm	20-30 cm	30-40 cm	40-50 cm	50-60 cm	60-70 cm	70-80 cm
AI	ppm	259	203	0.99	2.13	0.00	0.00	0.00	259
Ca	ppm	2360	2700	3380	4120	3750	3510	4170	5270
Cd	ppm	12.57	1.70	0.00	0.00	0.00	0.00	0.00	0.00
Со	ppm	2.58	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Cu	ppm	0.94	0.78	0.00	0.00	0.00	0.00	0.00	0.62
Fe	ppm	18000	16700	2260	4.90	0.98	0.00	1.30	195
K	ppm	1100	1280	747	687	439	648	724	673
Mg	ppm	8530	6890	1780	31.5	402	1200	477	483
Mn	ppm	555	403	72.1	0.21	1.14	0.89	0.31	6.82
Na	ppm	1340	1220	700	975	302	224	437	567
Ni	ppm	72.8	44.6	1.98	0.00	0.00	0.00	0.00	0.00
Pb	ppm	12.1	14.9	0.00	0.00	0.00	0.00	0.00	0.00
Si	ppm	127	159	17.6	3.04	4.15	3.52	3.86	9.46
Sr	ppm	11.5	17.3	22.1	25.7	31.0	41.5	45.1	35.8
S	ppm	44800	35600	9420	5210	4860	6820	9070	170
Zn	ppm	9550	5560	55.6	0.00	0.00	0.00	0.00	7.46
SO4	ppm	35600	25800	4200	1390	1550	1520	1330	1090
Eh		112	66.0	-28.0	45.0	4.00	-3.00	36.0	-10.0
pН		4.35	4.59	6.05	7.62	6.89	7.25	7.65	7.00

2004 (Cell 2 Ta	ailings Pore wa	ater Chemical A	nalvses – Sa	ampling Site:	R-TP-3			
		0-17 cm	17-27 cm	27-37 cm	37-46 cm	45-56 cm	56-71 cm	71-81 cm	81-90 cm
Al	ppm	1340	29400	0.00	8710	62.5	1.86	8.49	861
Са	ppm	512	2500	648	342	470	3100	3490	361
Cd	ppm	5.44	33.2	0.00	19.1	0	0.00	0.00	3.19
Со	ppm	4.27	25.8	0.00	23.9	0	0.00	0.00	2.83
Cu	ppm	88.0	1130	0.74	369	0	0.00	0.00	26.0
Fe	ppm	6950	64800	53.6	29900	5460	3470	37.7	3150
K	ppm	0.00	0.00	177	0.00	131	1190	1040	39.5
Mg	ppm	1820	10800	1140	4740	1330	7210	5800	1280
Mn	ppm	67.5	451	10.4	136	71.5	314	24.8	56.5
Na	ppm	18.5	41.7	74.1	4.88	104	551	507	81.3
Ni	ppm	15.0	48.5	0.00	20.3	3.50	7.50	0.00	8.09
Pb	ppm	0.00	0.00	0.00	7.93	1.40	0.00	0.00	0.00
Si	ppm	36.4	616	0.00	310	33.2	18.7	10.7	54.4
Sr	ppm	0.93	2.69	3.12	0.55	2.30	16.3	23.2	0.86
S	ppm	14000	169000	7600	61000	7300	25300	18700	8650
Zn	ppm	2200	9800	3.03	4730	1240	352	2.65	1500
SO4	ppm	230000	106000	35200	21200	21200	9310	2950	4200
Eh		-	402	-	-	95.0	-25.0	120	-
рН		1.82	2.18	4.01	2.96	4.11	5.90	7.10	3.38

2004	Cell 2 Ta	ailings Pore v	vater Chemi	cal Analyses	- Sampling	Site: R-TH-
		120 cm	170 cm	220 cm	270 cm	370 cm
Al	ppm	0.00	1.69	0.00	5.30	3.61
Ca	ppm	3060	4390	4560	4100	6700
Cd	ppm	0.00	0.00	0.00	0.00	0.00
Со	ppm	0.00	0.00	0.00	0.00	0.00
Cu	ppm	0.00	0.00	1.46	0.00	0.00
Fe	ppm	1.08	0.00	50.7	30.8	0.00
K	ppm	764	505	727	402	465
Mg	ppm	4780	52.4	1590	54.8	7.66
Mn	ppm	2.06	0.00	113	1.10	0.00
Na	ppm	409	422	391	292	350
Ni	ppm	0.00	0.00	5.72	0.00	0.00
Pb	ppm	0.00	0.00	0.00	0.00	0.00
Si	ppm	10.3	2.65	11.3	3.39	1.51
Sr	ppm	23.6	12.9	12.5	18.0	32.6
S	ppm	16700	6380	12700	5490	12900
Zn	ppm	0.00	0.00	151	9.64	0.00
SO4	ppm	5010	1150	1900	1230	925
Eh		80.0	47.0	40.0	85.0	165
рН		6.84	6.62	6.35	6.65	6.40

Appendix IX Pore water Chemical Analyses – 2005

2005	Cell 3 Ta	ailings Pore	water Chemic	al Analyses -	- Sampling S	Site: R-TP-7	
		0-10 cm	10-20 cm	20-30 cm	30-40 cm	40-50 cm	50-60 cm
Al	ppm	5740	9100	496	1620	3.60	37.6
Ca	ppm	1180	1580	1280	1360	1290	1110
Cd	ppm	2.00	2.80	0.40	3.00	0.00	0.00
Со	ppm	13.1	17.3	0.00	1.10	0.00	0.10
Cu	ppm	1.90	0.00	1.00	0.00	0.00	0.00
Fe	ppm	37600	47000	28500	31600	16500	14500
K	ppm	143	122	419	415	613	508
Mg	ppm	6280	8030	4550	4310	3860	3080
Mn	ppm	96.9	129	115	136	120	109
Na	ppm	81.8	92.9	196	177	158	93.6
Ni	ppm	11.3	14.3	5.50	4.90	0.90	0.00
Pb	ppm	163	201	115	126	68.4	48.8
Si	ppm	43.2	120	22.2	24.2	6.80	8.40
Sr	ppm	4.0	4.90	7.40	8.60	9.10	6.90
S	ppm	36800	47000	20300	23800	1320	11500
Zn	ppm	839	999	524	2600	75.0	8.00
SO4	ppm	78800	290000	37800	46200	35700	57800
Eh		200	212	66.0	181	-50.0	70.0
рН		3.16	3.47	4.62	4.00	5.56	4.95

2005	Cell 2 Ta	ailings Pore v	vater Chemica	al Analyses –	Sampling Si	ite: R-TP-5
		0-10 cm	10-20 cm	20-30 cm	50 cm	60 cm
Al	ppm	3720	3650	527	291	409
Ca	ppm	1450	1640	1000	1220	1440
Cd	ppm	5.10	2.50	2.60	15.6	45.3
Co	ppm	17.2	15.0	2.70	6.50	28.7
Cu	ppm	94.1	115	0.10	1.00	1.50
Fe	ppm	30700	38800	9720	25900	36600
K	ppm	0.00	0.00	29.6	357	637
Mg	ppm	2950	3020	663	3170	3750
Mn	ppm	40.4	37.3	10.0	90.4	76.7
Na	ppm	42.1	58.3	66.4	144	120
Ni	ppm	4.70	4.90	2.80	30.2	60.5
Pb	ppm	120	155	33.4	105	157
Si	ppm	255	310	219	41.9	43.2
Sr	ppm	4.30	4.80	2.60	4.10	3.20
S	ppm	29600	33700	7190	18300	29100
Zn	ppm	1620	725	445	2630	6950
SO4	ppm	74000	69600	46000	71500	28300
Eh		428	446	233	50.0	136
рН		2.07	1.94	3.67	4.37	4.42

2005	Cell 2 Ta	ailings Pore v	vater Chemio	cal Analyse	s – Samplin	g Site: R-TH-3
		100 cm	150 cm	250 cm	300 cm	400 cm
Al	ppm	63.4	1.30	0.00	2.30	2570
Са	ppm	1310	1080	2800	2600	1380
Cd	ppm	0.50	0.80	0.00	0.00	105
Со	ppm	0.00	0.00	0.00	0.00	38.9
Cu	ppm	0.90	1.80	0.00	0.00	156
Fe	ppm	26900	40900	13.0	63.0	6970
K	ppm	1060	1350	292	265	77.2
Mg	ppm	5510	10600	11.0	408	2020
Mn	ppm	312	450	0.00	5.60	50.2
Na	ppm	474	353	247.3	176	112
Ni	ppm	0.00	0.00	0.00	0.00	31.8
Pb	ppm	106	175	0.00	0.00	32.0
Si	ppm	38.1	23.4	0.40	1.10	219
Sr	ppm	5.20	8.80	7.70	8.90	2.00
S	ppm	23000	37600	7910	4680	12400
Zn	ppm	3440	383	0.00	4.00	3000
SO4	ppm	29500	3030	6050	4680	1450
Eh		38.0	-80.0	75.0	-55.0	285
рН		5.09	5.50	9.51	6.85	3.45

Appendix X Groundwater Chemical Analyses

Groundwater Sample Geochemical Analyses							
		R-BH-1	R-BH-1	R-BH-1	R-BH-3	R-BH-4	R-BH-5
0-000		04-Jun		04-Sep	04-Sep	04-Sep	04-Sep
	mg/L	-	19.0	-	-	-	-
	mg/L	-	2500	-	-	-	-
	mg/L	- 15 0	24.0	- 18.8	- 173	16 5	- 28.1
SO4	mg/L	1/70	1520	1/10	16500	2/30	1770
90 - Р	mg/L	-	0.37	1410	10500	2430	-
AI	ma/l	0.80	0.07	0.07	0.09	0.22	0 00
Sb	ma/l	-	<0.001	-	-	-	-
As	ma/L	-	< 0.0005	-	_	-	-
Ba	ma/L	-	0.026	-	_	-	-
Be	ma/L	_	< 0.001	-	-	-	_
Bi	ma/L	-	< 0.0001	-	-	-	-
В	mg/L	-	0.029	-	-	-	-
Cs	mg/L	-	0.0002	-	-	-	-
Cd	mg/L	0.00	0.0005	<0.50	<0.50	<0.50	<0.50
Са	mg/L	465	510	506	526	523	570
Cr	mg/L	-	0.002	-	-	-	-
Со	mg/L	0.00	0.004	<0.05	<0.05	<0.05	<0.05
Cu	mg/L	0.06	<0.0004	<0.05	<0.05	<0.05	<0.05
Fe	mg/L	1.00	<0.01	3.66	0.01	0.02	1.09
Pb	mg/L	0.00	0.0043	<0.10	<0.10	<0.10	<0.10
Li	mg/L	-	0.018	-	-	-	-
Mg	mg/L	19.3	19.7	10.8	13.2	2.63	10.4
Mn	mg/L	0.36	0.388	0.35	0.08	-	0.43
Мо	mg/L	-	0.0189	-	-	-	-
NI	mg/L	0.00	0.0305	< 0.10	<0.10	< 0.10	< 0.10
K	mg/L	49.3	52.0	54.7	74.0	43.8	45.6
RD Q-	mg/L	-	0.0497	-	-	-	-
Se	mg/L	-	0.015	-	-	-	-
51 A a	mg/L	0.40		0.00	0	0	0
Ay Na	mg/L	-		- 112	83 O	- -	-
INd Sr	mg/L	92.0	99.3	2.61	03.0	03.9 2.01	00.3
S	mg/L	2.00	2.71	508	637	752	958
J To	mg/L	_	<0.0005	500	-	152	-
TI	ma/l	_	<0.0000	_	_	_	_
Sn	ma/l	_	<0.0001	-	_	_	-
Ti	ma/l	-	0.0085	-	-	-	-
Ü	ma/l	-	0.0002	-	-	-	-
V	ma/L	-	< 0.001	-	-	-	_
Zn	ma/L	1.70	1.71	0.10	<0.10	-	0.18
Zr	mg/L	-	0.0021	-	-	-	-
pН	5	6.33	6.33	6.01	7.00	6.90	6.90
Lab	-	U of M	ETL	U of M	U of M	U of M	U of M

Groundwater Sample Geochemical Analyses (cont'd)						
		R-BH-7	R-BH-8	R-BH-10	R-BH-11	R-BH-1
		04-Sep	04-Sep	04-Sep	04-Sep	04-Oct
CI	mg/L	23.1	23.4	18.7	24.9	22.4
SO4	mg/L	1690	1770	1850	1970	2480
Al	mg/L	0.00	0.00	0.00	0.29	78.7
Cd	mg/L	<0.50	<0.50	<0.50	<0.50	0.0
Ca	mg/L	540	521	556	530	329
Co	mg/L	<0.05	<0.05	<0.05	<0.05	0.1
Cu	mg/L	<0.05	<0.05	<0.05	<0.05	2.2
Fe	mg/L	2.60	0.01	1.12	0.01	131
Pb	mg/L	<0.10	<0.10	<0.10	<0.10	0.0
Mg	mg/L	8.18	39.7	13.4	9.99	152
Mn	mg/L	0.57	0.45	0.46	0.20	-
Ni	mg/L	<0.10	<0.10	<0.10	<0.10	0.1
PO4	mg/L	-	-	-	-	
K	mg/L	38.4	58.1	67.3	55.3	6.7
Si	mg/L	0.00	0.00	0.00	0.00	12.3
Na	mg/L	57.5	76.4	89.6	120	18.5
Sr	mg/L	3.29	3.50	2.44	2.48	0.7
S	mg/L	934	682	735	1010	854
Zn	mg/L	1.26	<0.10	<0.10	<0.10	41.6
рН	pH units	7.50	7.10	7.15	8.20	6.2
Lab	-	U of M	U of M	U of M	U of M	U of N

Groundwater Sample Geochemical Analyses (cont'd)					
	-	R-BH-1	R-BH-5	R-BH-7	
		05-Sep	05-Sep	05-Sep	
CaCO3	mg/L	4380	1840	3200	
TDS	mg/L	-	5900	19000	
CI	mg/L	18.0	25.0	25.0	
SO4	mg/L	82700	3790	11300	
Soluble P	mg/L as P	0.57	0.13	<0.02	
Al	mg/L	2040	1.76	8.60	
Sb	mg/L	0.003	<0.001	<0.001	
As	mg/L	0.261	0.0008	0.002	
Ва	mg/L	0.0009	0.05	0.0194	
Be	mg/L	0.015	<0.001	<0.001	
Bi	mg/L	0.0072	<0.0003	< 0.0003	
В	mg/L	0.05	0.05	0.07	
Cs	ma/L	0.0012	0.0003	0.0007	
Cd	ma/L	1.16	0.0006	0.0049	
Ca	ma/L	122	447	269	
Cr	ma/L	0.249	<0.001	< 0.001	
Co	ma/L	1.13	0.0009	0.0057	
Cu	ma/L	48.1	0.0086	0.0119	
Fe	ma/L	13400	803	3770	
Pb	ma/L	0.0275	0.0048	0.005	
Li	ma/L	0.844	0.097	0.133	
Ма	ma/L	991	176	614	
Mn	mg/L	10.4	3.79	15.1	
Мо	mg/L	0.0013	0.0019	0.0009	
Ni	mg/L	0.796	0.0123	0.0221	
К	mg/L	0.47	124	97.0	
Rb	mg/L	0.043	0.095	0.205	
Se	mg/L	0.075	0.003	0.013	
Si	mg/L	72.3	7.80	9.90	
Ag	mg/L	0.0007	<0.0005	<0.0005	
Na	mg/L	11.4	67.5	46.9	
Sr	mg/L	0.623	3.55	2.87	
Те	mg/L	0.014	<0.0005	<0.0005	
TI	mg/L	0.0008	0.0001	0.0002	
Sn	mg/L	0.003	<0.0003	<0.0003	
Ti	ma/L	0.312	0.0238	0.0476	
U	mg/L	0.139	0.0012	0.0061	
V	mg/L	0.082	<0.001	<0.001	
Zn	mg/L	874	1.26	106	
Zr	mg/L	0.002	<0.0004	< 0.0004	
pН	pH units	2.43	3.46	3.52	
Cond	uS/cm	27300	4720	10700	

Groundwater Sample Geochemical Analyses (cont'd)									
		RBH-2	RBH-4	RBH-5	RBH-6	RBH-7			
		07-Jul	07-Jul	07-Jul	07-Jul	07-Jul			
CaCO3	mg/L	1350	1160	2690	2150	2460			
SO4	mg/L	2850	1310	4460	8080	5710			
Р	mg/L	<0.02	0.52	0.06	<0.02	<0.02			
Al	mg/L	10.1	<0.02	0.65	1.04	1.95			
Sb	mg/L	<0.001	<0.001	<0.001	<0.001	<0.001			
As	mg/L	0.0023	<0.0005	0.0008	0.0025	0.0008			
Ba	mg/L	0.0241	0.0161	0.0303	0.0329	0.0053			
Be	mg/L	<0.001	<0.001	<0.001	<0.001	<0.001			
Bi	mg/L	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003			
В	mg/L	<0.02	0.02	0.02	0.03	0.03			
Cs	mg/L	0.0010	0.0001	0.0002	0.0011	0.0010			
Cd	mg/L	0.0825	<0.0002	<0.0002	0.0005	0.0062			
Ca	mg/L	428	457	375	285	320			
Cr	mg/L	0.005	<0.001	<0.001	0.005	<0.001			
Co	mg/L	0.0911	0.0033	0.0021	0.0012	0.0044			
Cu	mg/L	0.0134	0.0055	0.0125	0.0190	0.0293			
Fe	mg/L	571	1.36	840	3140	1670			
Pb	mg/L	0.298	0.0002	0.0051	0.0231	0.0018			
Li	mg/L	0.020	<0.005	0.090	0.139	0.070			
Mg	mg/L	68.5	5.16	427	350	404			
Mn	mg/L	2.55	0.138	10.1	18.0	12.0			
Мо	mg/L	<0.0001	0.0165	0.0017	0.0015	<0.0001			
Ni	mg/L	0.0915	0.0050	0.0016	0.0084	0.0027			
K	mg/L	70.7	43.0	98.1	125	115			
Rb	mg/L	0.0895	0.0293	0.0975	0.185	0.140			
Se	mg/L	0.006	0.002	0.003	0.005	0.004			
Si	mg/L	5.40	0.20	2.10	2.80	4.70			
Ag	mg/L	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005			
Na	mg/L	68.2	74.1	53.1	33.0	37.5			
Sr	mg/L	3.63	3.73	4.61	4.84	4.89			
Те	mg/L	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005			
TI	mg/L	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001			
Sn	mg/L	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003			
Ti	mg/L	0.0129	0.0067	0.0258	0.0326	0.0293			
U	mg/L	0.0040	<0.0001	0.0006	0.0041	0.0013			
V	mg/L	<0.001	<0.001	<0.001	<0.001	<0.001			
Zn	mg/L	23.7	0.078	0.527	1.17	0.784			
Zr	mg/L	<0.0004	<0.0004	<0.0004	<0.0004	<0.0004			
рН		4.75	7.05	4.05	8.47	4.27			
Lab		ALS	ALS	ALS	ALS	ALS			
Groundwat	er Sample	e Geochemical Ana	alyses (cont'd)						
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	-	RBH-10	RBH-11						
		07-Jul	07-Jul						
CaCO3	mg/L	1580	2800						
SO4	mg/L	6070	20400						
Р	mg/L	0.04	<0.02						
Al	mg/L	180	379						
Sb	mg/L	<0.001	<0.001						
As	mg/L	0.0048	0.0055						
Ва	mg/L	<0.0003	<0.0003						
Be	mg/L	0.001	0.006						
Bi	mg/L	<0.0003	<0.0003						
В	mg/L	<0.02	0.04						
Cs	mg/L	0.0008	0.0003						
Cd	mg/L	0.0619	0.125						
Ca	mg/L	328	214						
Cr	mg/L	0.107	0.060						
Co	mg/L	0.675	0.0698						
Cu	mg/L	7.71	3.68						
Fe	mg/L	2300	8540						
Pb	mg/L	0.0276	0.0259						
Li	mg/L	0.046	0.107						
Mg	mg/L	184	551						
Mn	mg/L	3.64	40.8						
Мо	mg/L	<0.0001	0.0002						
Ni	mg/L	0.216	0.110						
K	mg/L	<0.05	29.1						
Rb	mg/L	0.0087	0.153						
Se	mg/L	0.016	0.019						
Si	mg/L	8.80	8.30						
Ag	mg/L	<0.0005	<0.0005						
Na	mg/L	3.85	21.3						
Sr	mg/L	0.512	2.48						
Те	mg/L	<0.0005	<0.0005						
TI	mg/L	0.0011	0.0006						
Sn	mg/L	<0.0003	<0.0003						
Ti	mg/L	0.150	0.0636						
U	mg/L	0.0069	0.0520						
V	mg/L	0.098	0.014						
Zn	mg/L	33.7	303						
Zr	mg/L	<0.0004	0.0010						
рН		2.78	3.91						
Lab		ALS	ALS						

Appendix XI Groundwater Ion Speciation

Modeled Using Wateq4f

Groundwater Sample Calculated Ion Speciation								
	RBH-1	RBH-1	RBH-3	RBH-4	RBH-5	RBH-7	RBH-8	
	04-Jun	04- Sep	04-Sep	04-Sep	04-Sep	04-Sep	04-Sep	
Na	93.0	110	83.0	84.0	68.0	57.0	76.0	
Na	98.0	98.0	97.0	96.0	97.0	97.0	97.0	
NaSO4	2.40	2.30	2.60	3.90	2.70	2.70	2.80	
K	49.0	55.0	74.0	44.0	46.0	38.0	58.0	
K	98.0	97.0	97.0	96.0	97.0	97.0	97.0	
KSO4	2.50	2.60	2.90	4.40	3.00	2.90	3.50	
Mg	19.0	11.0	13.0	2.60	10.0	8.20	40.0	
Mg	74.0	72.0	70.0	61.0	70.0	70.0	65.0	
MgOH	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
MgSO4	26.0	28.0	30.0	39.0	30.0	30.0	35.0	
Ca	460	510	530	520	570	540	520	
Ca	69.0	70.0	67.0	59.0	67.0	67.0	65.0	
CaSO4	31.0	30.0	33.0	41.0	33.0	33.0	35.0	
Fe(total)	1.20	3.70	0.010	0.020	1.10	2.60	0.01	
Fe	77.0	76.0	74.0	0.00	74.0	74.0	70.0	
FeSO4 aq	23.0	24.0	26.0	0.00	26.0	26.0	29.0	
Fe(OH)3	0.00	0.00	0.07	30.0	0.00	0.00	0.00	
Fe(OH)4	0.00	0.00	0.00	69.0	0.00	0.00	0.00	
Ni	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
Ni	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
NiSO4	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
Cu	0.063	0.00	0.00	0.00	0.00	0.00	0.00	
Cu	6.00	0.00	0.00	0.00	0.00	0.00	0.00	
Cu	62.0	0.00	0.00	0.00	0.00	0.00	0.00	
CuSO4	29.0	0.00	0.00	0.00	0.00	0.00	0.00	
Zn	1.70	0.100	0.00	0.00	0.18	1.30	0.00	
Zn	63.0	64.0	0.00	0.00	60.0	60.0	0.00	
ZnSO4	33.0	33.0	0.00	0.00	35.0	35.0	0.00	
Zn(SO4)2	3.80	3.30	0.00	0.00	4.60	4.40	0.00	
AI	0.810	0.07	0.09	0.22	0.00	0.00	0.00	
AI	16.0	20.0	0.00	0.00	0.00	0.00	0.00	
AIOH	29.0	26.0	2.20	0.00	0.00	0.00	0.00	
AI(OH)2	26.0	24.0	16.0	0.00	0.00	0.00	0.00	
AI(OH)4	3.80	1.50	78.0	100	0.00	0.00	0.00	
AISO4	17.0	20.0	0.00	0.00	0.00	0.00	0.00	
AI(SO4)2	6.90	7.80	0.00	0.00	0.00	0.00	0.00	
SO4	1500	1400	1700	2400	1800	1700	1800	
SO4	74.0	72.0	73.0	78.0	73.0	73.0	71.0	
MSO4	26.0	28.0	27.0	22.0	27.0	27.0	29.0	
pH	6.30	6.00	7.00	9.60	6.90	7.50	7.10	
	2100	2100	2400	3100	2500	2200	2500	

Groundwater Sample Calculated Ion Speciation (cont'd)							
	RBH-10	RBH-11	RBH-1	RBH-1	RBH-5	RBH-7	
	04-Sep	04-Sep	04-Oct	05-Sep	05-Sep	05-Sep	
Na	90.0	120	19.0	11.0	68.0	47.0	
Na	97.0	97.0	96.0	78.0	96.0	92.0	
NaSO4	2.90	3.20	3.50	22.0	4.40	8.00	
К	67.0	55.0	6.70	0.47	120	97.0	
К	97.0	96.0	96.0	80.0	95.0	92.0	
KSO4	3.40	3.70	4.00	20.0	4.70	8.30	
Mg	13.0	10.0	150	990	180	610	
Mg	66.0	64.0	65.0	29.0	64.0	54.0	
MgOH	0.00	35.0	0.00	0.00	0.00	0.00	
MgSO4	34.0	0.00	35.0	71.0	36.0	46.0	
Са	560	530	330	120	450	270	
Са	65.0	63.0	62.0	29.0	60.0	51.0	
CaSO4	35.0	37.0	38.0	69.0	40.0	48.0	
Fe(total)	1.10	0.01	130	13000	800	3800	
Fe	71.0	60.0	69.0	22.0	68.0	59.0	
FeSO4 aq	29.0	27.0	31.0	75.0	32.0	41.0	
Fe(OH)3	0.00	10.0	0.00	0.00	0.00	0.00	
Fe(OH)4	0.00	1.00	0.00	0.00	0.00	0.00	
Ni	0.00	0.00	0.14	0.80	0.012	0.022	
Ni	0.00	0.00	63.0	18.0	61.0	52.0	
NiSO4	0.00	0.00	37.0	81.0	39.0	48.0	
Cu	0.00	0.00	2.20	48.0	0.0086	0.012	
Cu	0.00	0.00	3.40	17.0	59.0	50.0	
Cu	0.00	0.00	58.0	0.00	0.00	0.00	
CuSO4	0.00	0.00	36.0	83.0	41.0	50.0	
Zn	0.00	0.00	42.0	870	1.30	110	
Zn	0.00	0.00	54.0	5.70	50.0	35.0	
ZnSO4	0.00	0.00	39.0	32.0	39.0	39.0	
Zn(SO4)2	0.00	0.00	6.90	62.0	10.0	26.0	
Al	0.00	0.29	79.0	2000	1.80	8.60	
Al	0.00	0.00	11.0	0.00	32.0	21.0	
AIOH	0.00	0.00	23.0	0.00	0.00	0.00	
AI(OH)2	0.00	0.05	34.0	0.00	0.00	0.00	
Al(OH)4	0.00	100	7.00	0.00	0.00	0.00	
AISO4	0.00	0.00	14.0	15.0	39.0	32.0	
Al(SO4)2	0.00	0.00	8.60	84.0	30.0	47.0	
SO4	1800	2000	2500	83000	3800	11000	
SO4	73.0	74.0	72.0	54.0	69.0	62.0	
MSO4	27.0	26.0	28.0	46.0	31.0	38.0	
рН	7.20	8.20	6.30	2.40	3.50	3.50	
TDS	2500	2700	3300	100000	5400	16000	

Appendix XII Surface Water Chemical Analyses

Surface	Water Sa	mple Chemical	Analyses			
		RW-1-03	RW-2-03	RW-6-03	RW-1-04	RW-2-04
		03-Sep	03-Sep	03-Sep	04-Jun	04-Jun
CaCO3	mg/L	1740	2940	1270	-	-
TDS	mg/L	3960	13600	7950	-	-
F	mg/L	1.10	6.90	7.10	-	-
CI	mg/L	21.3	7.30	3.30	0.00	17.7
SO4	mg/L	3190	9900	5660	6880	4800
Р	mg/L	< 0.05	< 0.05	< 0.05	-	-
Al	mg/L	39.9	641	92.9	491	145
Sb	mg/L	0.0005	0.0008	0.0005	-	-
As	mg/L	0.0020	0.0029	0.0130	-	-
Ba	mg/L	0.0230	0.0120	0.0012	-	-
Be	mg/L	0.0003	0.0040	0.0005	-	-
В	mg/L	0.0200	0.0200	0.0200	-	-
Cd	mg/L	0.0160	0.311	0.145	0.514	0.306
Ca	mg/L	573	297	324	236	335
Cr	mg/L	0.0100	0.034	0.0280	-	-
Co	mg/L	0.0660	1.68	0.3480	0.865	0.075
Cu	mg/L	0.4570	23.0	3.39	27.3	4.69
Fe	mg/L	200	598	908	1170	1220
Pb	mg/L	0.0910	0.086	0.0140	0.00	1.10
Li	mg/L	0.0130	0.08	0.0350	-	-
Mg	mg/L	75.8	535	183	443	393
Mn	mg/L	2.58	15.8	2.53	12.3	13.8
Hg	ug/L	< 0.0200	< 0.02	0.0200	-	-
Ni	mg/L	0.0830	0.81	0.4730	0.640	0.460
PO4	mg/L	2.28	0.68	0.650	-	-
K	mg/L	50.6	10.6	5.53	8.60	24.3
Se	mg/L	0.00350	0.035	0.0250	-	-
Si	mg/L	17.7	39.1	43.2	36.7	25.2
SiO2	mg/L	17.7	39.1	43.2	-	-
Na	mg/L	86.2	7.63	16.0	8.80	44.3
Sr	mg/L	2.54	0.698	0.7100	0.440	1.42
	mg/L	0.0001	0.0010	0.0036	-	-
Th	mg/L	0.0016	0.0084	0.0048	-	-
	mg/L	0.0027	0.0058	0.0210	-	-
U	mg/L	0.0032	0.0350	0.0053	-	-
Zn	mg/L	14.9	228	64.4	299.7	247
∠r	mg/L	< 0.0020	0.0030	< 0.0020	-	-
рн	0/	3.11	2.75	2.02	2.85	3.38
Cond	uS/cm	5/30	14000	11300		-
Lab	-	ETL	ETL	ETL	U of M	U of M

Surface	Surface Water Sample Chemical Analyses (cont'd)								
		RW-2-04	RW-5-04	RW-6-04	RW-12-04				
		June - Rain	04-Jun	04-Jun	04-Jun				
CI	mg/L	16.9	15.5	0.00	25.4				
SO4	mg/L	3960	5730	972	11800				
Al	mg/L	90.9	94.8	17.2	500				
Cd	mg/L	0.164	1.36	0.036	0.750				
Ca	mg/L	414	368	119	406				
Со	mg/L	0.0730	0.628	0.000	0.603				
Cu	mg/L	1.67	12.0	0.68	25.7				
Fe	mg/L	844	523	162	2530				
Pb	mg/L	0.240	0.650	0.00	0.00				
Mg	mg/L	261	531	32.5	728				
Mn	mg/L	9.64	21.1	0.610	23.1				
Ni	mg/L	0.31	2.00	0.00	1.64				
K	mg/L	39.6	26.3	6.60	1.10				
Si	mg/L	21.8	18.8	20.1	72.0				
Na	mg/L	66.4	44.2	4.70	32.0				
Sr	mg/L	2.06	1.40	0.230	1.04				
Zn	mg/L	139	420	14.6	341				
рН		3.48	3.52	2.65	2.57				
Lab		U of M	U of M	U of M	U of M				

Surfac	Surface Water Chemical Analyses (cont'd)						
		RW-12-04	RW-13-04	RW-2-04			
		Dup.	04-Jun	04-Oct			
TDS	mg/L	21000	-	-			
F	mg/L	<0.100	-	-			
CI	mg/L	41.0	22.7	21.6			
SO4	mg/L	13000	10100	5030			
Р	mg/L	0.0200	-	-			
Al	mg/L	494	244	0.500			
As	ma/L	0.0230	-	-			
Ва	mg/L	0.0008	-	-			
Ве	mg/L	0.0050	-	-			
Bi	ma/L	0.0005	-	-			
В	ma/L	0.0820	-	-			
Cs	ma/L	0.0014	-	-			
Cd	ma/l	0.985	0 0960	0.00			
Ca	ma/l	299	334	514			
Cr	ma/l	0 132	-	-			
Co	ma/l	1 09	0 011	0.00			
Cu	ma/l	27.1	0.490	0.00			
Fe	ma/l	2520	2330	415			
Ph	ma/l	0 0109	0.00	0 00			
l i	ma/l	0.0100	0.00	-			
Ma	ma/l	/08	555	106			
Mn	mg/L	26.4	10.6	100			
Mo	ma/l	0.0006	15.0	_			
Ni	mg/L	1 50	- 0 470	-			
K	mg/L		20.4	60.8			
Dh	mg/L	<0.0300 0.0112	23.4	00.0			
So	mg/L	0.0112	-	-			
Se Si	mg/L	40.4	- 29.7	- 3 40			
01 4 a	mg/L	49.4	30.7	5.40			
Ay Na	mg/L	<0.0005 22 5	-	- 01 7			
na Sr	mg/L	22.0	44.0	2 02			
0	mg/L	1.00	1.00	5.05			
3 To	mg/L	-	-	950			
	mg/L	0.0014	-	-			
11 Cm	mg/L	0.0003	-	-			
50 T:	mg/L	0.0003	-	-			
	mg/L	0.0593	-	-			
U	mg/L	0.0584	-	-			
V	mg/L	0.0260	-	-			
Zn	mg/L	361	261	2.48			
∠r	mg/L	0.0013	-	-			
	рн	0.50	0.00				
рн	units	2.58	3.36				
Lab	-	EIL	U of M	U of M			

Surface Water Sample Chemical Analyses (cont'd)						
		RW-5	RW-1	RW-3		
		05-Sep	07-Jul	07-Jul		
CaCO3	mg/L	2610	2240	1050		
TDS	mg/L	17000	-	-		
CI	mg/L	14.0	-	-		
SO4	mg/L	12600	9030	3890		
Р	mg/L	0.05	0.05	<0.02		
Al	mg/L	288	552	115		
Sb	mg/L	<0.001	<0.0010	<0.0010		
As	mg/L	0.0358	0.0194	0.0059		
Ba	mg/L	0.0054	<0.0003	<0.0003		
Be	mg/L	0.0030	0.0080	0.0020		
Bi	mg/L	0.0017	<0.0003	<0.0003		
В	mg/L	0.0500	0.0300	0.0300		
Cs	mg/L	0.0017	0.0002	<0.0001		
Cd	mg/L	0.288	0.309	0.0629		
Са	mg/L	287	207	180		
Cr	mg/L	0.102	0.0610	0.031		
Со	mg/L	0.456	1.80	0.246		
Cu	mg/L	6.24	34.4	3.24		
Fe	mg/L	2720	1240	245		
Pb	mg/L	0.0328	0.0251	0.0385		
Li	mg/L	0.219	0.186	0.0490		
Mg	mg/L	459	419	147		
Mn	mg/L	10.0	11.4	3.82		
Мо	mg/L	0.0002	0.0002	< 0.0001		
Ni	mg/L	0.313	0.650	0.1540		
K	mg/L	35.1	3.16	4.18		
Rb	mg/L	0.109	0.0299	0.0204		
Se	mg/L	0.0250	0.0190	0.0060		
Si	mg/L	46.1	35.6	13.9		
Ag	mg/L	< 0.0005	< 0.0005	< 0.0005		
Na	mg/L	31.4	6.07	12.2		
Sr	mg/L	2.06	0.399	0.544		
le	mg/L	0.0017	< 0.0005	< 0.0005		
	mg/L	0.0020	0.0008	8000.0		
Sn T	mg/L	0.0004	< 0.0003	< 0.0003		
	mg/L	0.0772	0.0582	0.0134		
U	mg/L	0.0309	0.0424	0.0105		
V	mg/L	0.0240	0.0240	<0.0010		
∠n 才	mg/L	164	169	38.0		
∠r	mg/L	0.0007	0.0020	<0.0004		
الم	pH unita	0.05	2.05	0.00		
pH Const	units	2.65	2.65	2.66		
Cond.	uS/cm	8360	-	-		
Lab	-	EIL	ALS	ALS		

Appendix XIII Surface Water Ion Speciation

Modeled Using Wateq4f

Surface Water Sample Calculated Ion Speciation								
	RW-1	RW-2	RW-6	RW-1	RW-2	RW-2	RW-5	
	03-Sep	03-Sep	03-Sep	04-Jun	04-Jun	04-Jun	04-Jun	
Na	86	7.6	16	8.8	44	66	44	
Na	96	93	95	95	96	96	95	
NaSO4	4.1	6.8	4.6	4.7	4.1	4	5.2	
К	51	11	5.5	8.6	24	40	26	
К	95	92	95	95	96	95	94	
KSO4	4.8	7.7	5.4	5.1	4.4	4.9	5.7	
Mg	76	540	180	440	390	260	530	
Mg	60	53	60	64	67	60	60	
MgSO4 aq	40	47	40	36	33	40	40	
Ca	570	300	320	240	340	410	370	
Ca	59	53	57	62	64	62	58	
CaSO4 aq	41	47	39	38	36	38	42	
Mn	2.6	16	2.5	12	14	9.6	21	
Mn	66	60	66	69	72	67	66	
MnSO4 aq	34	40	34	31	28	33	34	
Fe	200	600	910	1200	1200	840	520	
Fe	65	59	63	69	72	67	65	
FeSO4 aq	34	40	33	31	28	33	35	
Ni	0.083	0.81	0.47	0.64	0.46	0.31	2	
Ni	61	54	61	63	65	63	59	
NiSO4 aq	39	46	39	37	35	37	41	
Cu	0.46	23	3.4	27	4.7	1.7	12	
Cu	59	53	59	61	63	62	57	
CuSO4 aq	41	47	41	39	37	38	43	
Zn	15	230	64	300	250	140	420	
Zn	51	40	50	51	54	54	47	
ZnSO4 aq	40	41	39	37	36	38	40	
Zn(SO4)2	8.4	19	11	12	9.8	7.9	13	
Al	40	640	93	490	150	91	95	
Al	29	24	28	35	38	34	29	
AISO4	40	34	34	36	36	39	37	
Al(SO4)2	28	41	27	29	26	26	33	
SO4	3200	9900	5700	6900	4800	4000	5700	
SO4	68	50	50	48	56	57	59	
MSO4	32	50	50	52	44	43	41	
рН	3.1	2.8	2.6	2.9	3.4	3.5	3.5	
TDS	4200	12000	7300	9600	7200	5900	7700	

Surface Water Sample Calculated Ion Speciation (cont'd)							
	RW-6	RW-12	RW-13	RW-2	RW-5		
	04-Jun	04-Jun	04-Jun	04-Oct	05-Sep		
Na	4.7	32	44	82	31		
Na	98	93	93	94	91		
NaSO4	1.7	6.7	7.2	6.1	8.7		
К	6.6	1.1	29	61	35		
K	98	92	92	94	90		
KSO4	1.9	8.2	8	6.3	10		
Mg	33	730	550	110	460		
Mg	76	52	53	58	46		
MgSO4 aq	24	48	47	42	54		
Ca	120	410	330	510	290		
Са	73	54	53	52	48		
CaSO4 aq	26	45	47	47	52		
Mn	0.61	23	20	0	10		
Mn	80	60	60	0	54		
MnSO4 aq	20	40	40	0	46		
Fe	160	2500	2300	420	2700		
Fe	79	59	59	62	53		
FeSO4 aq	21	40	40	38	46		
Ni	0	1.6	0.47	0	0.31		
Ni	0	56	53	0	49		
NiSO4 aq	0	44	46	0	51		
Cu	0.68	26	0.49	0	6.2		
Cu	73	54	52	0	47		
CuSO4 aq	27	46	48	0	53		
Zn	15	340	260	2.5	160		
Zn	69	42	38	40	33		
ZnSO4 aq	29	40	40	43	41		
Zn(SO4)2	2	18	22	17	26		
Al	93	500	240	0.5	290		
Al	45	26	23	23	18		
AISO4	42	34	34	37	32		
AI(SO4)2	12	40	43	40	50		
SO4	970	12000	10000	5000	13000		
SO4	71	45	56	78	56		
MSO4	29	55	44	22	44		
pН	2.7	2.6	3.4	3.4	2.7		
TDS	1400	17000	14000	6200	17000		

Appendix XIV Lake Water Chemical Analyses

Lake Water S	Sample Che	mical Analyses	s (cont'd)		
		RW-4-03	RW-3-04	RW-3-04	RW-4-04
		03-Sep	04-Jun	04-Oct	04-Jun
CaCO3	mg/L	1460	-	-	-
TDS	mg/L	3710	-	-	-
F	mg/L	5.30	-	-	-
CI	mg/L	34.2	13.8	28.6	10.9
SO4	mg/L	4970	2520	11000	1660
Al	mg/L	65.1	103.5	529	60.1
Sb	mg/L	0.0006	-	-	-
As	mg/L	0.0095	-	-	-
Ba	mg/L	0.0051	-	-	-
Be	mg/L	0.0008	-	-	-
Bi	mg/L	0.0005	-	-	-
В	mg/L	0.0300	-	-	-
Cd	mg/L	0.0469	0.0820	0.340	0.0440
Ca	mg/L	381	294	312	286
Cr	mg/L	0.018	-	-	-
Со	mg/L	0.169	0.0410	1.12	0.0940
Cu	mg/L	1.56	3.21	21.7	1.66
Fe	mg/L	125	878	1509	129
Pb	mg/L	0.110	0.00	0.00	0.00
Li	mg/L	0.0340	-	-	-
Mg	mg/L	123	286	492	122.1
Mn	mg/L	3.34	6.48	-	3.00
Ni	mg/L	0.155	0.170	0.570	0.120
PO4	mg/L	0.330	-		-
K	mg/L	16.5	9.30	15.4	10.2
Se	mg/L	0.012	-	-	-
Si	mg/L	17.7	21.2	33.5	14.7
SiO2	mg/L	17.7	-	-	-
Na	mg/L	20.5	23.8	26.1	15.9
Sr	mg/L	1.11	0.860	0.990	0.760
S	mg/L	-	-	3120	-
TI	mg/L	0.00069	-	-	-
Th	mg/L	0.00250	-	-	-
Ti	mg/L	0.00470	-	-	-
U	mg/L	0.00710	-	-	-
Zn	mg/L	34.6	63.2	169	29.3
рН	pH units	2.83	2.87	3.22	2.87
Conductivity	uS/cm	6140	-	-	-
Lab		ETL	U of M	U of M	U of M

Lake W	ater Sample (Chemical Analy	yses (cont'd)		
		RW-8-04	RW-9-04	RW-7-04 (mill pond)	RW-4-04
		04-Jun	04-Jun	04-Jun	04-Oct
CI	mg/L	13.04	11.2	8.50	21.0
SO4	mg/L	1700	1990	2970	7920
Al	mg/L	52.7	55.3	72.5	162
Cd	mg/L	0.0380	0.0420	0.145	0.110
Ca	mg/L	273	286	167	374
Со	mg/L	0.0930	0.1040	0.335	0.290
Cu	mg/L	1.42	1.55	5.97	3.06
Fe	mg/L	93.0	108	97.0	1140
Pb	mg/L	0.00	0.00	0.00	0.00
Mg	mg/L	109	114	164	273
Mn	mg/L	2.95	2.94	4.81	0.200
Ni	mg/L	0.00	0.00	0.250	27.9
K	mg/L	9.40	9.50	6.30	22.4
Si	mg/L	14.1	14.6	11.4	48.2
Na	mg/L	17.1	17.3	8.50	1.73
Sr	mg/L	0.700	0.760	0.360	1820
Zn	mg/L	28.4	28.4	67.0	81.8
рН	pH units	2.70	2.91	3.30	3.10
Lab	-	U of M	U of M	U of M	U of M

Lake Wat	er Sample Ch	nemical Analy	ses (cont'd)			
	·	RW-3	RW-4	RW-8	RW-9	RW-3
		05-Sep	05-Sep	05-Sep	05-Sep	07-Jul
CaCO3	mg/L	1140	1280	744	1070	1050
TDS	mg/L	3200	4600	1800	2700	-
CI	mg/L	10.0	10.0	<9.00	9.00	-
SO4	mg/L	2460	3290	1470	2140	3890
Р	mg/L as P	<0.02	0.0800	0.0300	0.0300	<0.0200
Al	mg/L	101	127	47.8	70.2	115
Sb	mg/L	<0.0010	<0.0010	<0.0010	<0.001	<0.0010
As	mg/L	0.0104	0.0150	0.0030	0.0045	0.0059
Ва	mg/L	0.0074	0.0073	0.0153	0.0047	<0.0003
Be	mg/L	0.0020	0.0020	<0.0010	0.001	0.0020
Bi	mg/L	<0.0003	0.0004	<0.0003	<0.0003	<0.0003
В	mg/L	0.0400	0.0400	0.0300	0.0300	0.0300
Cs	mg/L	0.0004	0.0004	0.0002	0.0003	<0.0001
Cd	mg/L	0.105	0.0916	0.0362	0.0561	0.0629
Ca	mg/L	205	210	153	216	180
Cr	mg/L	0.026	0.054	0.0170	0.027	0.0310
Со	mg/L	0.260	0.258	0.108	0.165	0.246
Cu	mg/L	4.44	3.4	1.37	2.31	3.24
Fe	mg/L	161	470	80.1	131	245
Pb	mg/L	0.0520	0.0612	0.0278	0.0437	0.0385
Li	mg/L	0.0840	0.104	0.0790	0.0900	0.0490
Mg	mg/L	152	185	87.9	128	147
Mn	mg/L	3.44	3.71	1.83	2.63	3.82
Мо	mg/L	0.0001	0.0001	<0.0001	<0.0001	<0.0001
Ni	mg/L	0.184	0.159	0.0800	0.121	0.154
Κ	mg/L	4.96	4.88	3.69	4.29	4.18
Rb	mg/L	0.0188	0.0199	0.0128	0.0185	0.0204
Se	mg/L	0.0100	0.0110	0.0050	0.0080	0.0060
Si	mg/L	13.4	17.6	10.5	13.8	13.9
Ag	mg/L	0.0006	<0.0005	<0.0005	<0.0005	<0.0005
Na	mg/L	13.2	14.3	11.5	15.2	12.2
Sr	mg/L	0.512	0.538	0.376	0.513	0.544
Те	mg/L	<0.0005	0.0007	<0.0005	<0.0005	<0.0005
TI	mg/L	0.0007	0.0008	0.0004	0.0006	0.0008
Sn	mg/L	0.0003	<0.0003	<0.0003	< 0.0003	<0.0003
Ti	mg/L	0.0191	0.0384	0.0098	0.0148	0.0134
U	mg/L	0.0100	0.0117	0.0049	0.0070	0.0105
V	mg/L	<0.0010	0.0130	<0.0010	<0.0010	<0.001
Zn	mg/L	50.1	50.4	20.3	29.6	38.0
Zr	mg/L	0.0012	0.0007	<0.0004	<0.0004	<0.00040
pН	pH units	2.84	2.82	2.91	2.78	2.66
Cond.	uS/cm	3120	3770	2120	2820	-
Lah	-	FTI	ETL	ETL	ETL	ALS

Appendix XV Lake Water Ion Speciation Modeled Using Wateq4f

Lake Water Sample Calculated Ion Speciation (cont'd)								
	RW-4	RW-3	RW-4	RW-8	RW-9	RW-3		
	03-Sep	04-Jun	04-Jun	04-Jun	04-Jun	04-Oct		
Na	21	24	16	17	17	26		
Na	94	98	98	98	97	92		
NaSO4	6.4	2.4	2.2	2.4	2.8	7.6		
K	17	9.3	10	9.4	9.5	15		
K	92	97	98	97	97	92		
KSO4	7.7	2.6	2.4	2.6	3	8.4		
Mg	120	290	120	110	110	490		
Mg	49	76	75	73	71	52		
MgSO4	51	24	26	27	29	48		
Ca	380	290	290	270	290	310		
Ca	49	73	71	70	67	51		
CaSO4	50	27	29	30	33	49		
Fe	130	880	130	93	110	1500		
Fe	56	79	77	76	74	58		
FeSO4 aq	44	21	22	23	26	42		
Ni	0.16	0.17	0.12	0	0	0.57		
Ni	51	74	72	0	0	52		
NiSO4	49	26	28	0	0	48		
Cu	1.6	3.2	1.7	1.4	1.6	22		
Cu	50	72	70	69	66	50		
CuSO4	50	28	30	31	34	50		
Zn	35	63	29	28	28	170		
Zn	39	67	66	64	60	36		
ZnSO4	45	29	31	32	35	41		
Zn(SO4)2	16	4.2	3.4	3.7	4.8	23		
Al	65	100	60	53	55	530		
Al	18	50	44	42	38	22		
AISO4	34	35	40	41	42	33		
AI(SO4)2	37	15	15	16	20	45		
SÒ4	5000	2500	1700	1700	2000	11000		
SO4	74	56	64	65	67	54		
HSO4	4.1	2.4	3.2	4.9	3	0		
MSO4	22	42	33	30	30	45		
pН	2.8	2.9	2.9	2.7	2.9	3.2		
TDS	5700	4200	2300	2300	2600	14000		

Lake Water	Sample Ca	Iculated Io	n Speciatio	n (conťd)	
	RW-4	RW-3	RW-4	RW-8	RW-9
	04-Oct	05-Sep	05-Sep	05-Sep	05-Sep
Na	48	13	14	12	15
Na	93	97	96	98	97
NaSO4	7.3	3.2	3.8	2.4	3
К	28	5	4.9	3.7	4.3
К	92	96	95	97	96
KSO4	8.1	3.9	4.6	2.9	3.7
Mg	270	150	190	88	130
Mg	52	63	61	67	64
MgSO4	48	37	39	33	36
Ca	370	210	210	150	220
Ca	50	63	61	67	64
CaSO4	50	36	38	32	35
Fe	1100	160	470	80	130
Fe	58	69	67	72	70
FeSO4 aq	42	31	33	27	30
Ni	0.2	0.18	0.16	0.08	0.12
Ni	51	65	63	69	66
NiSO4	48	35	37	31	34
Cu	3.1	4.4	3.4	1.4	2.3
Cu	50	63	61	67	64
CuSO4	50	37	39	33	36
Zn	82	50	50	20	30
Zn	37	57	54	62	58
ZnSO4	42	38	39	34	37
Zn(SO4)2	21	5.4	7.1	3.2	4.8
Al	160	100	130	48	70
Al	21	34	33	38	35
AISO4	34	43	41	45	43
Al(SO4)2	45	23	26	18	22
SO4	7900	2500	3300	1500	2100
SO4	65	62	59	67	64
HSO4	1.5	3.8	3.6	3.9	4.7
MSO4	33	34	37	29	32
рН	3.1	2.8	2.8	2.9	2.8
TDS	10000	3200	4400	1900	2700

Appendix XVI Stream Flow Calculation

		Sta	art	Mide	dle	Fini	sh
Reading	Time ¹ (s)	Distance (cm)	Depth (cm)	Distance (cm)	Depth (cm)	Distance (cm)	Depth (cm)
1	6.69	0	0	0	0	0	0
2	6.75	10	6.5	10	2.8	10	3.8
3	5.19	20	4.5	20	4.1	20	5.9
4	6.69	30	5	30	4.9	30	5.8
5	6.97	40	11.4	40	7.6	40	7.8
6	6.6	50	14.8	50	10.4	50	9.4
7	5.38	60	14.2	60	7.7	60	8.8
8	6.16	70	15.8	70	8.6	70	8.2
9	5.16	80	11.8	80	9.2	80	7.3
10	4.97	90	11.4	90	6.3	90	5.9
11	5.16	100	4.3	100	3.6	100	4.8
12	4.97	105	0	102	0	110	3.4
13	5.47					114	0
14	5.22						
15	6.03						
16	5.34						
17	5.00						
18	5.91						
19	6.37						
20	6.22						
Average	5.81						

¹ Time it took for a match stick to travel from start to finish. ² Depth measurement taken every 10 cm across stream at the start, middle, and end of the 2.3 m flow measurement zone.

Appendix XVII Geotechnical Analyses

Tailings	Grain Size Analyses	- Sieve					
-				Sieve +	Mass		
Sieve #	Sieve Diam. (µm)	Sieve Diam. (mm)	Empty (g)	Retained (g)	Retained (g)	% Retained	% Passing
10	2000	2	485	500.9	15.9	3.2	96.82
20	850	0.85	374.9	455.3	80.4	16.1	80.75
30	595	0.595	417.2	458.3	41.1	8.2	72.53
40	420	0.42	509.1	531.9	22.8	4.6	67.97
60	250	0.25	370.4	395.3	24.9	5.0	62.99
80	180	0.18	314	328.8	14.8	3.0	60.04
100	147	0.147	434.3	441.7	7.4	1.5	58.56
140	106	0.106	315.5	331.5	16	3.2	55.36
200	75	0.075	302	329.3	27.3	5.5	49.90
Pan	-	-	346.5	593.2	246.7	49.3	

Elapsed Time (Minutes)	0	2	4	8	16	60	30	120	240
Temperature (°C)	28.00	28.04	28.08	28.13	28.17	29.00	29.00	28.29	28.33
Hydrometer (Ra in Gs)	1010.0	1004.5	1004.2	1004.1	1003.5	1003.1	1003.0	1002.8	1002.7
Hydrometer (Ra in g/L)	10.0	4.5	4.2	4.1	3.5	3.1	3.0	2.8	2.7
Control (Gs)	999.0	999.0	999.1	999.5	999.1	999.1	998.9	999.0	998.8
Control (g/L)	-1.0	-1.0	-0.9	-0.5	-0.9	-0.9	-1.1	-1.0	-1.2
Temp. Corr. Factor (CT)	2.50	2.50	2.50	2.50	2.50	3.05	3.05	2.50	2.50
Corrected Hydrometer									
Reading (Rc)	13.50	8.00	7.60	7.10	6.90	7.05	7.15	6.30	6.40
% Finer	22.02	13.05	12.40	11.58	11.26	11.50	11.66	10.28	10.44
Adjusted % Finer	10.86	6.43	6.11	5.71	5.55	5.67	5.75	5.07	5.15
Hyd. Corr. Only For									
Meniscus (Rc)	11.0	5.5	5.2	5.1	4.5	4.1	4.0	3.8	3.7
L	14.50	15.40	15.48	15.52	15.55	15.59	15.60	15.64	15.66
L/t		7.700	3.870	1.940	0.972	0.520	0.260	0.130	0.065
К	0.009	0.009	0.009	0.009	0.009	0.009	0.009	0.009	0.009
D, mm	0.025	0.018	0.013	0.009	0.007	0.005	0.003	0.002	0.002
Dispersing Agent: 125 mL	of Na ₂ SiO ₃	α:		0.81735					

Gs of Solids:	3.51
Zero Correction:	-1
Meniscus:	1
Mass Soil:	50.1 g
% Finer:	49.3

0	Control Siev	/e: 200
	K at 28°:	0.009162
	K at 29º:	0.009213
	K at 26º:	0.009311

Tailings Grain Size Analys	sis – Hydrom	eter (cont'd)	
Elapsed Time (Minutes)	480	Ì344 ´	2001
Temperature (°C)	28.38	26.00	28.46
Hydrometer (Ra in Gs)	1002.6	1002.3	1001.9
Hydrometer (Ra in g/L)	2.6	2.3	1.9
Control (Gs)	999.0	999.2	999.2
Control (g/L)	-1.0	-0.8	-0.8
Temp. Corr. Factor (CT)	2.50	1.65	2.50
Corrected Hydrometer			
Reading (Rc)	6.10	4.75	5.20
% Finer	9.95	7.75	8.48
Adjusted % Finer	4.91	3.82	4.18
Hyd. Corr. Only For			
Meniscus (Rc)	3.6	3.3	2.9
L	15.68	15.77	15.82
L/t	0.033	0.012	0.008
К	0.009	0.009	0.009
D, mm	0.001	0.001	0.001



Grain Size A	nalyses - I	Laser Part	icle Counter Test				
	Sample	H2O	Defloculent	Mixing		Mean Paticle	
Test ID	Weight	Volume	Volume	Time	Total Counts	Size	Comments
	(g)	(mL)	(6% NaPO3)(mL)	(mins)		(µm)	
Jared 1	0.1000	100.0	-	2	1058000.00	9.26	- particles clumpin and settling,
							particles stuck to surface tension.
Jared 2	0.0999	100.0	-	2	263750.00	9.47	
Jared 3	0.0999	100.0	-	10	10261666.00	9.60	- "
6% NaPO3	-	-	-	2	1107.00	2.93	- Tested particle size of 6%
							NaPO3 before using in test.
Jared 4	0.1006	87.5	12.5	10	273.00	10.89	- Did not change dilution
			•				factor.
Jared 5	0.1006	87.5	12.5	20	1101333.00	10.19	- Still some clumping and
			-	-			settling, no particles stuck to
							surface tension
Jared 6	0.0999	80.0	20.0	10	1077000.00	10.70	- less settling, no particles stuck
							to surface tension
Jared 7	0.0999	80.0	20.0	Overniaht	1065500.00	13.95	- all particles in suspension.
				<u>s</u>			some particles stuck to surface
							tension
Jared 8	0.1001	80.0	20.0	5	271250.00	11.46	- broke sample up with morter
				-			and pestle and razor blade.
							Some particles still settling out of
							solution. No paticles stuck to
							surface tension.
Jared 9	0 1001	80.0	20.0	10	1101000 00	11 79	_ "

Conclusions

- Longer mixing times and the addition of a defloculent allowed larger particles to be measured.

- There is a fraction of particles that cannot be measured by this method because they settle too fast. It is hard to tell if this fraction is made up of floculated particles or single grains.

Specific Gravit	y Analyses							
Sample No	1	1b	2	2b	3	3b	4	4b
Ms	101.6	102.1	101.2	102.1	101.7	100.2	101.1	100.5
Mbws	725.8	738.2	724.7	738.3	725.7	737.9	726.3	735.1
Temperature	24	24	24	24	24	24	27	27
Flask #	Н	Q	Н	Q	Н	Q	Н	Q
Flask Vol.	500	500	500	500	500	500	500	500
Mbw								
(@temp)	653.0	665.3	653.0	665.3	653.0	665.3	652.6	664.9
Mw	28.8	29.2	29.5	29.1	29.0	27.6	27.4	30.3
α	0.9991	0.9991	0.9991	0.9991	0.9991	0.9991	0.9983	0.9983
Gs	3.52	3.50	3.43	3.51	3.50	3.63	3.68	3.31
Diff.	1.0	1	0.9	8	0.9	6	1.1	1
Avg Gs	3.5	1	3.4	7	3.5	7	3.5	0
Standard								
Dev.	0.0	17806294	0.0	59428639	0.0	91929833	0.2	6171417

- Fine-grained sieving would need to be done to remove settling fraction.

 $\alpha = \rho T / \rho 20^{\circ}C$ Gs = α Ms / Mw

ρw (24°C): 0.99732

ρw (27°C): 0.99651 ρw (20°C): 0.99823

Sample No	5	5h [′]	6	6h
Ma	100.2	100 1	100.9	100 1
IVIS	100.3	100.1	100.8	100.1
Mbws	732.5	749.8	725.6	737
Temperature	27	27	27	27
Flask #	BE	Μ	Н	Q
Flask Vol.	500	500	500	500
Mbw				
(@temp)	662.0	679.6	652.6	664.9
Mw	29.8	29.9	27.8	28.0
α	0.9983	0.9983	0.9983	0.9983
Gs	3.36	3.34	3.62	3.57
Diff.	1.0	00	1.01	1
Avg Gs	3.3	5	3.60)
Standard				
Dev.	0.0	10676179	0.03	34125387

Tray Woight (g)	Frozen	Dry Woight (g)	Water	Saturation	Bulk Donaity	Density of	•	n	Particle	Spe
weight (g)	weight (g)	weight (g)	Content (%)	Saturation	Density	water	е	Π	Density	Gia
369.4	2206.5	1791.5	29.18	82.48	2.02	1	1.24	0.55	3.50	3.50
356	2127.1	1722.8	29.58	83.72	2.02	1	1.23	0.55	3.47	3.47
360.8	2513.7	2009.8	30.56	84.55	2.02	1	1.27	0.56	3.51	3.5
206.5	2643.1	2158	24.86	77.78	2.02	1	1.07	0.52	3.35	3.35
168.1	2440.9	1921	29.66	81.99	2.02	1	1.29	0.56	3.57	3.57
284.1	2403.6	1920.5	29.52	81.23	2.02	1	1.31	0.57	3.60	3.60

Appendix XVIII Rising Head Tests

2004 RBH-1 Rising Head Test									
Water	Instrument	Well Height							
Level	Correction	Correction (H)	Time	Elapsed Time	Depth - Initial Depth	Head Ratio	Log(Head Ratio)		
			Pre-						
3.485	3.479	2.209	Pumping						
<0.100	<0.100	4.473	09:49:00	00:00:00	2.264	1.000	0.000		
<0.100	<0.100	4.473	09:49:30	00:00:30	2.264	1.000	0.000		
4.460	4.454	3.184	09:51:00	00:02:00	0.975	0.431	-0.366		
4.449	4.443	3.173	09:53:00	00:04:00	0.964	0.426	-0.371		
4.437	4.431	3.161	09:54:00	00:05:00	0.952	0.420	-0.376		
4.428	4.422	3.152	09:55:00	00:06:00	0.943	0.417	-0.380		
4.421	4.415	3.145	09:56:00	00:07:00	0.936	0.413	-0.384		
4.418	4.412	3.142	09:57:00	00:08:00	0.933	0.412	-0.385		
4.414	4.408	3.138	09:58:00	00:09:00	0.929	0.410	-0.387		
4.396	4.390	3.120	09:59:00	00:10:00	0.911	0.402	-0.395		
4.351	4.345	3.075	10:04:00	00:15:00	0.866	0.383	-0.417		
4.271	4.265	2.995	10:15:00	00:26:00	0.786	0.347	-0.459		
4.035	4.029	2.759	10:50:00	01:01:00	0.550	0.243	-0.615		
3.972	3.966	2.696	11:01:00	01:12:00	0.487	0.215	-0.667		
3.542	3.536	2.266	13:19:00	03:30:00	0.057	0.025	-1.599		
T _000 0									

 $T_o=990.000$ seconds $L_e=304.8$ cm

 $\frac{\text{Hvorslev}}{\text{K= }r^2\text{ln}(\text{L}_{e}/\text{R})\,/\,2\text{L}_{e}\text{T}_{o}}$

r=2.54 cm R=3.81 cm

K=4.68E-05 cm/s

2004 RBH-3 Rising Head Test									
Water	Instrument	Well Height							
Level	Correction	Correction (H)	Time	Elapsed Time	Depth - Initial Depth	Head Ratio	Log(Head Ratio)		
			Before						
1.397	1.391	1.111	Pumping						
3.182	3.176	2.896	10:37:00	00:00:00	1.785	1.000	0.000		
3.145	3.139	2.859	10:37:45	00:00:45	1.748	0.979	-0.009		
3.122	3.116	2.836	10:38:20	00:01:20	1.725	0.966	-0.015		
3.092	3.086	2.806	10:38:45	00:01:45	1.695	0.950	-0.022		
3.065	3.059	2.779	10:39:30	00:02:30	1.668	0.934	-0.029		
3.03	3.024	2.744	10:40:00	00:03:00	1.633	0.915	-0.039		
3.009	3.003	2.723	10:40:30	00:03:30	1.612	0.903	-0.044		
2.982	2.976	2.696	10:41:00	00:04:00	1.585	0.888	-0.052		
2.955	2.949	2.669	10:41:30	00:04:30	1.558	0.873	-0.059		
2.937	2.931	2.651	10:42:00	00:05:00	1.540	0.863	-0.064		
2.908	2.902	2.622	10:42:30	00:05:30	1.511	0.846	-0.072		
2.89	2.884	2.604	10:43:00	00:06:00	1.493	0.836	-0.078		
2.849	2.843	2.563	10:44:00	00:07:00	1.452	0.813	-0.090		
2.798	2.792	2.512	10:45:00	00:08:00	1.401	0.785	-0.105		
2.748	2.742	2.462	10:46:00	00:09:00	1.351	0.757	-0.121		
2.715	2.709	2.429	10:47:00	00:10:00	1.318	0.738	-0.132		
2.469	2.463	2.183	10:53:00	00:16:00	1.072	0.601	-0.221		
2.235	2.229	1.949	10:58:00	00:21:00	0.838	0.469	-0.328		
2.038	2.032	1.752	11:03:00	00:26:00	0.641	0.359	-0.445		
1.405	1.399	1.119	13:17:00	02:40:00	0.008	0.004	-2.349		

 T_o =1869.5 seconds L_e=152.4 cm r=2.54 cm

 $\frac{\text{Hvorslev}}{\text{K= }r^2\text{ln}(\text{L}_{e}/\text{R})\,/\,2\text{L}_{e}\text{T}_{o}}$

R=3.81 cm

K=4.18E-05 cm/s

200

2004 RBH-5 Rising Head Test								
Water	Instrument	Well Height						
Level	Correction	Correction (H)	Time	Elapsed Time	Depth - Initial Depth	Head Ratio	Log(Head Ratio)	
			Before					
2.309	2.303	2.024	Pumping		0			
3.801	3.795	3.516	12:00:30	00:00:00	1.492	1.000	0.000	
3.755	3.749	3.47	12:01:00	00:00:30	1.446	0.969	-0.014	
3.706	3.7	3.421	12:01:30	00:01:00	1.397	0.936	-0.029	
3.653	3.647	3.368	12:02:00	00:01:30	1.344	0.901	-0.045	
3.602	3.596	3.317	12:02:30	00:02:00	1.293	0.867	-0.062	
3.556	3.55	3.271	12:03:00	00:02:30	1.247	0.836	-0.078	
3.507	3.501	3.222	12:03:30	00:03:00	1.198	0.803	-0.095	
3.448	3.442	3.163	12:04:00	00:03:30	1.139	0.763	-0.117	
3.401	3.395	3.116	12:04:30	00:04:00	1.092	0.732	-0.136	
3.357	3.351	3.072	12:05:00	00:04:30	1.048	0.702	-0.153	
3.302	3.296	3.017	12:05:30	00:05:00	0.993	0.666	-0.177	
3.261	3.255	2.976	12:06:00	00:05:30	0.952	0.638	-0.195	
3.208	3.202	2.923	12:06:30	00:06:00	0.899	0.603	-0.220	
3.162	3.156	2.877	12:07:00	00:06:30	0.853	0.572	-0.243	
3.127	3.121	2.842	12:07:30	00:07:00	0.818	0.548	-0.261	
3.074	3.068	2.789	12:08:00	00:07:30	0.765	0.513	-0.290	
3.033	3.027	2.748	12:08:30	00:08:00	0.724	0.485	-0.314	
3.007	3.001	2.722	12:09:00	00:08:30	0.698	0.468	-0.330	
2.968	2.962	2.683	12:09:30	00:09:00	0.659	0.442	-0.355	
2.937	2.931	2.652	12:10:00	00:09:30	0.628	0.421	-0.376	
2.876	2.87	2.591	12:11:00	00:10:30	0.567	0.380	-0.420	
2.82	2.814	2.535	12:12:00	00:11:30	0.511	0.342	-0.465	

Water	Instrument	Well Height						
Level	Correction	Correction (H)	Time	Elapsed Time	Depth - Initial Depth	Head Ratio	Log(Head Ratio)	
2.773	2.767	2.488	12:13:00	00:12:30	0.464	0.311	-0.507	
2.723	2.717	2.438	12:14:00	00:13:30	0.414	0.277	-0.557	
2.688	2.682	2.403	12:15:00	00:14:30	0.379	0.254	-0.595	
2.552	2.546	2.267	12:20:00	00:19:30	0.243	0.163	-0.788	
2.442	2.436	2.157	12:29:00	00:28:30	0.133	0.089	-1.050	
2.345	2.339	2.060	12:54:00	00:53:30	0.036	0.024	-1.617	
2.310	2.304	2.025	13:53:00	01:52:30	0.001	0.001	-3.174	
T _o =662.6 seconds		Hvorslev						
1 - 254 or		$12 - \frac{2}{12} + \frac{2}{12}$						

2004 RBH-5 Rising Head Test (cont'd)

L_e=254 cm

 $K = r^{2} ln(L_{e}/R) / 2L_{e}T_{o}$

r=2.54 cm R=3.81 cm

K=8.05E-05 cm/s

2004 RBH-7 Rising Head Test								
Water	Instrument	Well Height						
Level	Correction	Correction (H)	Time	Elapsed Time	Depth - Initial Depth	Head Ratio	Log(Head Ratio)	
			Before					
1.870	1.864	1.762	Pumping		0			
2.78	2.774	2.672	12:36:30	00:00:00	0.91	1.000	0.000	
2.703	2.697	2.595	12:37:00	00:00:30	0.833	0.915	-0.038	
2.626	2.62	2.518	12:37:30	00:01:00	0.756	0.831	-0.081	
2.569	2.563	2.461	12:38:00	00:01:30	0.699	0.768	-0.115	
2.512	2.506	2.404	12:38:30	00:02:00	0.642	0.705	-0.152	
2.458	2.452	2.350	12:39:00	00:02:30	0.588	0.646	-0.190	
2.401	2.395	2.293	12:39:30	00:03:00	0.531	0.584	-0.234	
2.361	2.355	2.253	12:40:00	00:03:30	0.491	0.540	-0.268	
2.309	2.303	2.201	12:40:30	00:04:00	0.439	0.482	-0.317	
2.276	2.270	2.168	12:41:00	00:04:30	0.406	0.446	-0.351	
2.225	2.219	2.117	12:41:30	00:05:00	0.355	0.390	-0.409	
2.197	2.191	2.089	12:42:00	00:05:30	0.327	0.359	-0.444	
2.172	2.166	2.064	12:42:30	00:06:00	0.302	0.332	-0.479	
2.154	2.148	2.046	12:43:00	00:06:30	0.284	0.312	-0.506	
2.132	2.126	2.024	12:43:30	00:07:00	0.262	0.288	-0.541	
2.109	2.103	2.001	12:44:00	00:07:30	0.239	0.263	-0.581	
2.098	2.092	1.990	12:44:30	00:08:00	0.228	0.251	-0.601	
2.079	2.073	1.971	12:45:00	00:08:30	0.209	0.230	-0.639	

Water	Instrument	Well Height								
Level	Correction	Correction (H)	Time	Elapsed Time	Depth - Initial Depth	Head Ratio	Log(Head Ratio)			
2.062	2.056	1.954	12:45:30	00:09:00	0.192	0.211	-0.676			
2.051	2.045	1.943	12:46:00	00:09:30	0.181	0.199	-0.701			
1.962	1.956	1.854	12:51:00	00:14:30	0.092	0.101	-0.995			
1.922	1.916	1.814	12:56:00	00:19:30	0.052	0.057	-1.243			
1.871	1.865	1.763	13:56:00	01:19:30	0.001	0.001	-2.959			
T -266 (

2004 RBH-7 Rising Head Test (cont'd)

 T_o =266.33 seconds L_e =213.36 cm r=2.54 cm R=3.81 cm

 $\frac{\text{Hvorslev}}{\text{K= }r^2\text{ln}(\text{L}_{e}/\text{R})\,/\,2\text{L}_{e}\text{T}_{o}}$

K=2.29E-04 cm/s

204
2004 RBH-8 Rising Head Test									
Water	Instrument	Well Height							
Level	Correction	Correction (H)	Time	Elapsed Time	Depth - Initial Depth	Head Ratio	Log(Head Ratio)		
			Before	•	· · · · ·		i		
1.523	1.517	0.907	Pumping		0				
2.005	1.999	1.389	14:47:00	00:00:00	0.482	1.000	0.000		
2.005	1.999	1.389	14:47:30	00:00:30	0.482	1.000	0.000		
2.005	1.999	1.389	14:48:00	00:01:00	0.482	1.000	0.000		
2.005	1.999	1.389	14:48:30	00:01:30	0.482	1.000	0.000		
2.005	1.999	1.389	14:49:00	00:02:00	0.482	1.000	0.000		
2.004	1.998	1.388	14:51:30	00:04:30	0.481	0.998	-0.001		
2.001	1.995	1.385	14:52:30	00:05:30	0.478	0.992	-0.004		
1.997	1.991	1.381	14:53:00	00:06:00	0.474	0.983	-0.007		
1.993	1.987	1.377	14:53:30	00:06:30	0.47	0.975	-0.011		
1.988	1.982	1.372	14:54:00	00:07:00	0.465	0.965	-0.016		
1.986	1.98	1.37	14:54:30	00:07:30	0.463	0.961	-0.017		
1.981	1.975	1.365	14:55:00	00:08:00	0.458	0.950	-0.022		
1.978	1.972	1.362	14:56:00	00:09:00	0.455	0.944	-0.025		
1.967	1.961	1.351	14:57:00	00:10:00	0.444	0.921	-0.036		
1.95	1.944	1.334	15:00:00	00:13:00	0.427	0.886	-0.053		
1.922	1.916	1.306	15:05:00	00:18:00	0.399	0.828	-0.082		
1.862	1.856	1.246	15:15:00	00:28:00	0.339	0.703	-0.153		
1.838	1.832	1.222	15:20:00	00:33:00	0.315	0.654	-0.185		
1.716	1.71	1.1	15:47:00	01:00:00	0.193	0.400	-0.397		
1.589	1.583	0.973	16:49:00	02:02:00	0.066	0.137	-0.864		

 T_0 =3993 seconds L_e=134.62 cm r=2.54 cm

 $\frac{\text{Hvorslev}}{\text{K= }r^2\text{ln}(\text{L}_{e}/\text{R})\,/\,2\text{L}_{e}\text{T}_{o}}$

R=3.81 cm

K=2.14E-05 cm/s

Appendix XIX Protein Analyses - 2004

Protein As	ssay Results	5								
Ruttan Ta	ailings Cell 2	, Septem	ber 2004							
Sample	Sample	Depth	A562 ¹	Protein	Tris	Sample	Protein ²	Moisture ³	Standard Deviation ⁴	Average Protein
Number	Location	(cm)		(µg/ml)	(ml)	(g)	(µg/g)	(%)	(µg/g)	⁵ (μg/g)
0a	RTP-3	0	0.476	1640	6	5.003	1970	11.40	539	2570
0b		0	0.453	2350	6	5.128	2750	11.52		
0c		0	0.473	2450	6	4.898	3000	11.42		
10a	RTP-3	10	0.313	1620	6	5.082	1910	15.68	230	2040
10b		10	0.369	1910	6	4.966	2310	17.93		
10c		10	0.304	1580	6	4.959	1910	16.63		
20a	RTP-3	20	0.980	5080	6	5.096	5980	23.52	895	5470
20b		20	0.989	5120	6	5.129	5990	23.85		
20c		20	0.741	3840	6	5.194	4430	23.11		
30a	RTP-3	30	0.193	664	6	4.960	803	17.64	61.2	713
30b		30	0.167	575	6	5.064	681	17.96		
30c		30	0.187	644	6	5.183	745	17.08		
40a	RTP-3	40	0.446	1540	6	5.310	1730	10.87	256	1440
40b		40	0.322	1110	6	5.020	1330	11.31		
40c		40	0.312	1070	6	5.107	1260	11.24		
50a	RTP-3	50	0.142	489	6	4.903	598	37.39	93.3	637
50b		50	0.177	609	6	4.920	743	36.49		
50c		50	0.143	492	6	5.193	569	37.82		
1 Δł	nsorhance m	heasured	at 562 nm							

2 Protein in µg/g was calculated by multiplying protein in µg/ml by the amount of tris (ml) and dividing by the amount of dry solid sample used (g).

3 Percent moisture was calculated from samples weighed before and after lyophilization. Standard deviation calculated from triplicate protein analyses.

4

Protein Assay Results											
Ruttan Ta	ilings Cell 2	, Septem	ber 2004 ((conťd)							
Sample	Sample	Depth	A562 ¹	Protein	Tris	Sample	Protein ²	Moisture ³	Standard Deviation ⁴	Average Protein	
Number	Location	(cm)		(µg/ml)	(ml)	(g)	(µg/g)	(%)	(μ g/g)	⁵ (μg/g)	
60a	RTP-3	60	0.268	922	6	5.042	1100	14.08	78.1	1120	
60b		60	0.256	881	6	4.986	1060	13.98			
60c		60	0.299	1030	6	5.102	1210	13.52			
75a	RTP-3	75	0.720	3730	6	5.052	4430	11.15	60.8	4450	
75b		75	0.744	3850	6	5.122	4520	32.26			
75c		75	0.748	3880	6	5.287	4400	16.17			
90a	RTP-3	90	0.932	4830	6	5.143	5630	7.75	326	5550	
90b		90	0.936	4850	6	4.995	5820	8.94			
90c		90	0.846	4380	6	5.069	5190	9.12			
105a	RTH-1	105	0.645	3340	6	5.107	3930	31.47	291	3610	
105b		105	0.534	2770	6	4.950	3350	27.89			
105c		105	0.582	3020	6	5.110	3540	30.28			
1.2a	RTH-1	120	0.210	723	6	5.014	865	30.97	1242	2060	
1.2b		120	0.517	2800	6	5.020	3340	32.32			
1.2c		120	0.306	1660	6	5.076	1960	36.00			
1.7a	RTH-1	170	0.749	4050	6	5.109	4760	27.26	1552	5490	
1.7b		170	1.140	6170	6	5.091	7270	26.41			
1.7c		170	0.672	3640	6	4.922	4430	24.87			
⁻¹ Δh	1 Absorbance measured at 562 pm										

2 Protein in µg/g was calculated by multiplying protein in µg/ml by the amount of tris (ml) and dividing by the amount of dry solid sample used (g).

3 Percent moisture was calculated from samples weighed before and after lyophilization. Standard deviation calculated from triplicate protein analyses.

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Protein Assay Results											
Ruttan Ta	ailings Cell 2	, Septerr	ber 2004 ((conťd)							
Sample	Sample	Depth	A562 ¹	Protein	Tris	Sample	Protein ²	Moisture ³	Standard Deviation ⁴	Average Protein	
Number	Location	(cm)		(µg/ml)	(ml)	(g)	(µg/g)	(%)	(μ g/g)	⁵ (μg/g)	
2.2a	RTH-1	220	0.629	3400	6	5.015	4070	13.39	783	4750	
2.2b		220	0.873	4720	6	5.053	5610	14.79			
2.2c		220	0.703	3800	6	4.987	4580	13.32			
2.7a	RTH-1	270	2.308	7940	6	5.143	9270	34.05	2435	5460	
2.7b		270	1.630	5610	6	5.180	6500	32.43			
2.7c		270	1.078	3710	6	5.046	4410	32.23			
3.2a	RTH-1	320	0.972	3350	6	5.061	3970	11.21	1504	2610	
3.2b		320	0.702	2420	6	5.042	2880	12.16			
3.2c		320	0.243	836	6	5.056	992	11.64			
3.7a	RTH-1	370	0.300	1030	6	4.926	1260	34.73	787	1540	
3.7b		370	0.610	2100	6	5.175	2430	34.39			
3.7c		370	0.227	781	6	4.988	940	34.95			
4.2a	RTH-1	420	0.990	3407	6	5.096	4010	23.69	1060	4280	
4.2b		420	1.313	4519	6	4.980	5450	25.12			
4.2c		420	0.820	2822	6	5.010	3380	24.99			
4.7a	RTH-1	470	0.900	4870	6	4.997	5850	31.50	1800	4660	
4.7b		470	0.405	2192	6	5.076	2590	31.46			
4.7c		470	0.864	4675	6	5.068	5540	30.48			
1 ΔΙ	heorhanco m	haseurad	at 562 nm								

2 Protein in µg/g was calculated by multiplying protein in µg/ml by the amount of tris (ml) and dividing by the amount of dry solid sample used (g).

Percent moisture was calculated from samples weighed before and after lyophilization. Standard deviation calculated from triplicate protein analyses. Average calculated from triplicate protein analyses. 3

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Protein Assay Results												
Ruttan Tai	ilings Cell 3	, Septem	ber 2004									
Sample	Sample	Depth	A562 ¹	Protein	Tris	Sample	Protein ²	Moisture ³	Standard Deviation ⁴	Average Protein		
Number	Location	(cm)		(µg/ml)	(ml)	(g)	(µg/g)	(%)	(μ g/g)	⁵ (μg/g)		
0a	RTP-1	0	0.318	479	6	5.083	479	N.M.	123	337		
0b		0	0.242	268	6	5.184	268					
0c		0	0.230	264	6	5.007	264					
6a	RTP-1	15.2	0.164	183	6	5.153	183	N.M.	22.1	161		
6b		15.2	0.136	139	6	5.637	139					
6c		15.2	0.140	161	6	4.993	161					
12a	RTP-1	30.5	0.229	249	6	5.288	249	N.M.	38.3	229		
12b		30.5	0.161	185	6	4.999	185					
12c		30.5	0.223	254	6	5.048	254					
18a	RTP-1	45.7	0.664	998	6	5.097	998	N.M.	172	827		
18b		45.7	0.542	831	6	4.997	831					
18c		45.7	0.434	652	6	5.095	652					
24a	RTP-1	61.0	0.389	576	6	5.173	576	N.M	14.5	561		
24b		61.0	0.380	561	6	5.185	561					
24c		61.0	0.366	547	6	5.126	547					

¹ Absorbance measured at 562 nm.

² Protein in μ g/g was calculated by multiplying protein in μ g/ml by the amount of tris (ml) and dividing by the amount of dry solid sample used (g).

³ Percent moisture was calculated from samples weighed before and after lyophilization.

⁴ Standard deviation calculated from triplicate protein analyses.

⁵ Average calculated from triplicate protein analyses.

N.M. = Not Measured

Protein Assay Results											
Ruttan Tailings Cell 3, September 2004 (cont'd)											
Sample	Depth	A562 ¹	Protein	Tris	Sample	Protein ²	Moisture ³	Standard Deviation ⁴	Average Protein		
Location	(cm)		(µg/ml)	(ml)	(g)	(µg/g)	(%)	(μ g/g)	⁵ (μg/g)		
RTP-1	76.2	0.531	814	6	4.996	814	N.M	58.5	818		
	76.2	0.591	878	6	5.154	878					
	76.2	0.500	761	6	5.030	761					
RTP-1	91.4	1.080	1631	6	5.071	1630	N.M.	469	1120		
	91.4	0.476	711	6	5.125	711					
	91.4	0.670	1004	6	5.110	1000					
	3ay Results ings Cell 3, Sample Location RTP-1 RTP-1	say Results ings Cell 3, Septem Sample Depth Location (cm) RTP-1 76.2 76.2 RTP-1 91.4 91.4 91.4	say Results ings Cell 3, September 2004 (Sample Depth Location (cm) RTP-1 76.2 0.531 76.2 0.591 76.2 0.500 RTP-1 91.4 1.080 91.4 0.670	say Resultsings Cell 3, September 2004 (cont'd)SampleDepthA562 1ProteinLocation(cm)(µg/ml)RTP-176.20.59187876.20.59187876.20.500761RTP-191.491.40.47691.40.6701004	$\begin{array}{c c c c c c c c c c c c c c c c c c c $	$\begin{array}{c c c c c c c c c c c c c c c c c c c $	$\begin{array}{c c c c c c c c c c c c c c c c c c c $	$\begin{array}{c c c c c c c c c c c c c c c c c c c $	$\begin{array}{c c c c c c c c c c c c c c c c c c c $		

¹ Absorbance measured at 562 nm.

² Protein in μ g/g was calculated by multiplying protein in μ g/ml by the amount of tris (ml) and dividing by the amount of dry solid sample used (g).

³ Percent moisture was calculated from samples weighed before and after lyophilization.

⁴ Standard deviation calculated from triplicate protein analyses.

⁵ Average calculated from triplicate protein analyses.

N.M. = Not Measured

Appendix XX Protein Analyses - 2005

Protein A	ssay Results	5								
Ruttan Ta	ailings Cell 2	, Septerr	ber 2005							
Sample	Sample	Depth	A562 ¹	Protein	Tris	Sample	Protein ²	Moisture ³	Standard Deviation ⁴	Average Protein
Number	Location	(cm)		(µg/ml)	(ml)	(g)	(µg/g)	(%)	(μg/g)	⁵ (μg/g)
7a	RTP-5	0	0.100	288	6	4.833	358	22.17	48.3	412
7b		0	0.126	363	6	4.831	451			
7c		0	0.119	343	6	4.828	426			
8a	RTP-5	10	0.000	0.00	6	4.703	0.00	23.81	180	201
8b		10	0.072	207	6	4.884	255			
8c		10	0.102	294	6	5.078	347			
9a	RTP-5	20	0.300	865	6	4.864	1070	17.73	390	1150
9b		20	0.233	672	6	5.051	798			
9c		20	0.437	1260	6	4.825	1570			
10a	RTP-5	30	0.194	559	6	4.607	728	11.70	121	783
10b		30	0.192	553	6	4.767	697			
10c		30	0.275	793	6	5.166	921			
11a	RTP-5	40	0.177	510	6	4.452	688	9.34	225	947
11b		40	0.303	874	6	5.014	1050			
11c		40	0.319	920	6	4.994	1100			
12a	RTP-5	50	0.244	704	6	4.967	850	11.94	134	696
12b		50	0.173	499	6	4.959	603			
12c		50	0.186	536	6	5.084	633			
1 <u>Ak</u>	sorbance m	hasurad	at 562 nm							

2 Protein in µg/g was calculated by multiplying protein in µg/ml by the amount of tris (ml) and dividing by the amount of dry solid sample used (g).

Percent moisture was calculated from samples weighed before and after lyophilization. Standard deviation calculated from triplicate protein analyses. Average calculated from triplicate protein analyses. 3

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Protein Assay Results											
Ruttan Ta	ailings Cell 2	, Septerr	ber 2005 ((conťd)							
Sample	Sample	Depth	A562 ¹	Protein	Tris	Sample	Protein ²	Moisture ³	Standard Deviation ⁴	Average Protein	
Number	Location	(cm)		(µg/ml)	(ml)	(g)	(µg/g)	(%)	(µg/g)	⁵ (μg/g)	
13a	RTP-5	60	0.574	1660	6	5.226	1900	23.80	499	2180	
13b		60	0.806	2330	6	5.068	2750				
13c		60	0.550	1590	6	5.082	1870				
14a	RTP-5	70	0.024	69.3	6	5.087	81.7	13.50	13.2	80.0	
14b		70	0.026	75.0	6	4.890	92.0				
14c		70	0.019	54.8	6	4.997	65.8				
15a	RTH-3	100	0.845	2440	6	4.972	2940	21.46	241	2980	
15b		100	0.878	2530	6	4.699	3230				
15c		100	0.840	2420	6	5.276	2760				
16a	RTH-3	150	0.895	2580	6	5.335	2900	14.74	581	2480	
16b		150	0.806	2330	6	5.132	2720				
16c		150	0.556	1600	6	5.298	1820				
17a	RTH-3	200	0.659	1900	6	4.935	2310	8.56	323	1990	
17b		200	0.582	1680	6	5.074	1990				
17c		200	0.477	1380	6	4.962	1660				
18a	RTH-3	250	0.834	2410	6	5.117	2820	19.87	254	3110	
18b		250	0.951	2740	6	5.157	3190				
18c		250	0.948	2740	6	4.957	3310				
1 <u>Ak</u>	sorbance m	hasurad	at 562 nm								

2 Protein in µg/g was calculated by multiplying protein in µg/ml by the amount of tris (ml) and dividing by the amount of dry solid sample used (g).

3 Percent moisture was calculated from samples weighed before and after lyophilization. Standard deviation calculated from triplicate protein analyses.

4

Protein Assay Results											
Ruttan Tailings Cell 2, September 2005 (cont'd)											
Sample	Sample	Depth	A562 ¹	Protein	Tris	Sample	Protein ²	Moisture ³	Standard Deviation ⁴	Average Protein	
Number	Location	(cm)		(µg/ml)	(ml)	(g)	(µg/g)	(%)	(µg/g)	⁵ (μg/g)	
19a	RTH-3	300	0.531	1530	6	5.237	1760	27.45	230	1600	
19b		300	0.371	1070	6	4.803	1340				
19c		300	0.479	1380	6	4.837	1710				

2 Protein in µg/g was calculated by multiplying protein in µg/ml by the amount of tris (ml) and dividing by the amount of dry solid sample used (g).

3 Percent moisture was calculated from samples weighed before and after lyophilization.

4 Standard deviation calculated from triplicate protein analyses. Average calculated from triplicate protein analyses.

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Protein As	ssay Results	3								
Ruttan Ta	ilings Cell 3	, Septerr	ber 2005							
Sample	Sample	Depth	A562 ¹	Protein	Tris	Sample	Protein ²	Moisture ³	Standard Deviation ⁴	Average Protein
Number	Location	(cm)		(µg/ml)	(ml)	(g)	(µg/g)	(%)	(µg/g)	⁵ (μg/g)
1a	RTP-7	0	0.683	1970	6	5.182	2280	27.30	495	2830
1b		0	0.800	2310	6	4.678	2960			
1c		0	0.933	2690	6	4.977	3250			
2a	RTP-7	5	0.030	86.6	6	4.990	104	12.25	16.2	110
2b		5	0.028	80.8	6	4.983	97.3			
2c		5	0.036	104	6	4.866	128			
3a	RTP-7	10	0.542	1560	6	4.899	1920	28.77	222	1780
3b		10	0.524	1510	6	4.761	1910			
3c		10	0.451	1300	6	5.121	1530			
4a	RTP-7	20	1.28	3700	6	4.959	4480	28.99	96.6	4380
4b		20	1.24	3580	6	5.012	4280			
4c		20	1.18	3410	6	4.665	4390			
5a	RTP-7	30	0.219	632	6	4.642	817	35.74	94.4	867
5b		30	0.246	710	6	5.264	809			
5c		30	0.284	819	6	5.036	976			
6a	RTP-7	40	0.185	534	6	5.286	606	36.12	83.6	617
6b		40	0.197	568	6	4.835	705			
6c		40	0.160	462	6	5.138	539			2830
1 Δh	sorhance m	haseurad	at 562 nm							

2 Protein in μ g/g was calculated by multiplying protein in μ g/ml by the amount of tris (ml) and dividing by the amount of dry solid sample used (g).

Percent moisture was calculated from samples weighed before and after lyophilization. Standard deviation calculated from triplicate protein analyses. 3

4