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**GEOLOGICAL SURVEY OF CANADA
OPEN FILE 8753**

**Geoenvironmental characterization of carbonatite tailings,
Saint Lawrence Columbium Mine, Oka, Québec**



**A.J. Desbarats, J.B. Percival, P. Pelchat, J. Sekerka, I. Bilot,
I. Girard, and P. Gammon**

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Cover photograph: View looking southwest across the SLC tailings pond toward the decant tower. Photograph by A.J. Desbarats, September 2018. NRCan Photo Database number 2020-634.

FOREWORD

The emergence of new green technologies is driving increasing demand for advanced materials containing metals termed “critical” due to their scarcity or risk of supply disruption. However, potential environmental issues associated with the mining of many critical metals are not yet well understood. In order to address these knowledge gaps and Natural Resources Canada (NRCan) strategic priorities with respect to sustainable resource development, the Geological Survey of Canada (GSC) conducted a project on “Geoenvironmental Characteristics of Canadian Critical Metal Deposits” during the 2014-2019 cycle of the Environmental Geosciences Program. The objectives of the project were to characterize the chemistry of potentially hazardous trace elements in mine wastes and waters, and to understand their mobilization, transport and fate in the environment. The project focused on carbonatite intrusions, an important class of mineral deposit hosting critical metals including niobium (Nb) and rare earth elements (REE). Because carbonatite deposits exhibit many common geological and mineralogical features, their mine wastes will share similar environmental characteristics as well. Studying these wastes at historical mine sites provides benchmark information for improving environmental assessment predictions and the planning of mitigation measures for new mines hosted in carbonatite deposits elsewhere. The past-producing Saint Lawrence Columbium (Nb) mine near Oka, Québec, is an example of carbonatite-hosted critical metal deposit that serves here as an analogue for other carbonatite deposits currently under consideration for development in Canada and abroad. This report is specifically concerned with the environmental signature of mine tailings generated by the processing of carbonatite ores. It describes the physical properties of the tailings, their mineralogy, their bulk chemistry, and the chemistry of seepage draining from the tailings impoundment. The results of this investigation promote the sustainable development of critical metal resources by making new open data and knowledge available to stakeholders. This information will help better predict and mitigate environmental risks associated with resource extraction in carbonatites, and thereby to improve the efficiency of the regulatory approval process for future mining projects.

SUMMARY

Between 1961 and 1976, the St. Lawrence Columbium (SLC) mine produced pyrochlore concentrate and ferroniobium alloy from a carbonatite deposit near Oka, Québec. This report is concerned with the environmental characteristics of tailings generated by the processing of carbonatite ore at the SLC mine. It describes their physical properties, their mineralogy, their bulk chemistry and the chemistry of seepage draining from the tailings impoundment. Tailings solids were sampled at closely-spaced intervals in a borehole intersecting the full thickness of the impoundment. Tailings pore-waters were sampled in a nest of eight piezometers screened at different depths near the decant pond. Effluent was sampled at seepage points on the toes of the embankments of the tailings impoundment.

Grain-size distribution data for the tailings show that they range in texture from medium sand to clayey silt. The average water content, density, and porosity of the tailings are 30 %, 2.82 g/cm³, and 47 %, respectively. The geometric mean hydraulic conductivity of the tailings is estimated at 1.4 E-07 m/s using an empirical formula based on grain-size data.

The tailings are composed mainly of calcite (64-89 %) containing significant Sr, Mn, Mg, Ba, and REE. Other identified minerals include biotite (6-17 %), fluorapatite (2-22 %), chlorite (0-5 %), nepheline (0-2 %), magnetite, pyrite, sphalerite, and unrecovered pyrochlore. The fluorapatite contains high concentrations of REE although most of the REE inventory in the tailings is hosted by calcite. Most of the fluorine is hosted in biotite. The average total S content (mainly as sulfide) of the tailings is 0.53 %. In addition to Nb, the pyrochlore hosts U, Th, Ta, and Zr. Average U and Th concentrations in the tailings are 20 and 65 ppm, respectively.

Depth profiles of major ion pore-water chemistry show significant variations across the thickness of the impoundment as a result of several processes. In the shallow tailings, chemistry is controlled by calcite dissolution, sulfide oxidation and acid neutralization, which releases sulfate, Fe, Zn and Mo. At greater depths, oxygen is depleted and sulfate reduction occurs. Throughout the tailings, Ca, Mg, and Sr concentrations decrease with depth whereas Na and K concentrations increase steadily. This may be the combined result of nepheline hydrolysis and cation exchange. Fluoride concentrations increase with depth, approaching 25 mg/L. Aqueous concentrations of trace elements (U, Nb) in pore-water are very low and do not show any clear depth trend because of the low solubility of their mineral hosts. Total (aqueous and sorbed particulate) concentrations of REE reach 1 mg/L. The average pH and specific conductance of effluent from the tailings impoundment are approximately 8 and 1400 µs/cm, respectively. Effluent from the SE embankment contains 500 mg/L of sulfate, and fluoride concentrations of 8 mg/L, which exceeds the 0.12 mg/L guideline for the protection of freshwater aquatic life. Concentrations of U and other trace elements in tailings effluent are low.

This investigation of tailings at the former St. Lawrence Columbium mine, Oka, Québec, provides new geoscience data and knowledge useful for the environmental assessment of mining projects in carbonatite settings. This information will allow proponents and other stakeholders to anticipate potential environmental impacts associated with future resource development and to plan appropriate mitigation measures accordingly.

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1. INTRODUCTION

1.1 Background

With the emergence of new green technologies, there is increasing demand for advanced materials containing metals termed “critical” due to their scarcity or risk of supply disruption. Niobium (Nb) and rare earth elements (REE) are critical metals often mined in carbonatite deposits (Richardson and Birkett, 1996; Berger et al., 2009; Simandl and Paradis, 2018). Carbonatites are defined as igneous rocks composed of at least 50% carbonate minerals such as calcite, dolomite, and ankerite. Typically, they form the central core of alkaline intrusive complexes (Simandl and Paradis, 2018). The main Nb ore mineral is pyrochlore whereas bastnaesite and monazite are the main REE economic host minerals (Verplanck et al., 2014; Mitchell, 2015). The magmatic differentiation processes that concentrate Nb and REE in these minerals also concentrate other incompatible elements, such as uranium (U) and thorium (Th), in the same hosts (Verplanck and Van Gosen, 2011). Thus, anomalous values of U and Th are common in carbonatites and represent an environmental concern (Weng et al., 2013; Huang et al., 2016). However, there have been few published investigations of environmental issues associated with the development of carbonatite deposits (Verplanck et al., 2014; Edahbi et al., 2019). Because a number of carbonatites in Canada are being considered as sources of Nb or REE (or both), greater understanding of potential environmental effects is needed in order to better inform the regulatory approval process for these and similar projects. Such knowledge could reduce uncertainty associated with their environmental impact assessments.

Carbonatite-hosted Nb and REE deposits (Simandl and Paradis, 2018) form a class of mineral deposits defined by shared geological, mineralogical, and geochemical characteristics. Wastes produced by the mining of different carbonatite deposits, therefore, are likely to share similar environmental characteristics. Studying these wastes at historical mine sites provides transferable benchmark information that can be used to improve predictions and mitigation measures for environmental assessments of new mines in carbonatite deposits elsewhere. The former St. Lawrence Columbium (SLC) mine near Oka, Québec, is an example of carbonatite-hosted critical metal deposit and it was studied here as an analogue for other carbonatite deposits currently under consideration for development in Canada and abroad.

1.2 The Saint Lawrence Columbium Mine

Spurred by the 1950s uranium boom, the discovery of radioactive minerals near Oka prompted the St. Lawrence River Mining syndicate to stake a large portion of the surrounding area in early 1954. While the original exploration target was uranium, the greater economic significance of the niobium (columbium) content of the Oka carbonatite soon became apparent after drilling revealed the presence of several mineralized lenses and large bodies of low-grade pyrochlore-bearing rocks (Carbonneau and Caron, 1965). However, the challenge was to develop a viable pyrochlore concentration process. Research carried out at the École Polytechnique in Montréal and the Department of Mines and Technical Surveys in Ottawa eventually led to the design and construction of a pilot plant in 1959. The plant was able to achieve a 50 % Nb₂O₅ concentrate



Figure 1: View of the mine site in 1967, looking south toward the Lac des Deux Montagnes. The photograph shows the newly completed headframe and the tailings disposal area in the foreground. Photograph from SLC (1967).

from a head grade of 0.4 % Nb_2O_5 . After the original mining group merged with another company, the newly formed St. Lawrence Columbian and Metals Corporation (SLC) started construction of a 500 tons/day mill on the site and negotiated a two-year contract for 1 million pounds of Nb_2O_5 concentrate (Vallières, 1989). Mining and concentrate production started in October of 1961. At that time, SLC's Oka mine was the world's first primary producer of Nb_2O_5 as well as the single largest producer (Carbonneau and Caron, 1965).

Initially, ore was obtained from two open pits. Underground mining started in 1965 with access to the workings from an adit in the wall of the deeper pit. Following the completion of a 410 m (1345 ft.) shaft in late 1966, production shifted completely underground in 1968 (Figure 1). In 1971, the shaft was deepened to 586 m (1922 ft.) in order to access higher-grade reserves at depth. Throughout the mine life, mill capacity was gradually increased to 1700 tons/d in 1967, 2200 tons/d in 1972, and then 3000 tons/d in 1975. At the same time, modifications to the milling process improved pyrochlore recovery (Carbonneau and Caron, 1965, SLC, 1967). With research funding from the federal government, a new lower cost and higher efficiency concentrator flow-sheet was developed and plans were made to reprocess mine tailings containing an estimated 12 million pounds of Nb_2O_5 (SLC, 1971). Starting in 1972, the original milling process was phased out in favour of the new process, which was implemented in a parallel milling circuit (SLC, 1972). Meanwhile, in 1969, the company constructed a small-scale plant for custom production of ferroniobium alloy (FeNb), the form of niobium consumed by most end users (SLC, 1969). In order to meet increasing demand from the steel industry, the company began building a new full-scale FeNb conversion facility in 1971 (SLC, 1971). Thereafter, the company marketed both pyrochlore concentrate and ferroniobium. The company also explored opportunities to market

useful milling by-products including high-purity calcite, magnetite, and apatite as a source of phosphate fertilizer and rare earth elements (SLC, 1971).

In latter years, despite all the measures aimed at increasing the profitability of its operation, the company was struggling financially because of competition from new lower-cost and higher-grade producers in Brazil. In 1976, after a labour dispute closed the mine, SLC went into liquidation (Vallières, 1989).

During its 15-year life, SLC's Oka mine produced a total of 6.35 million tons of ore at an average grade of 0.46 % Nb₂O₅ (NIOCAN, 2010). Annual production data are summarized in Table 1. At the time of closure, reserves (noncompliant with NI 41-103) were estimated at 21.7 million tonnes of ore grading 0.44 % Nb₂O₅ (NIOCAN, 2010). Currently (in 2020), the mine site (Figure 2) is owned by the municipality of Oka while the mining rights belong to Niocan Inc. (<http://www.niocan.com>).

1.3 Potential Environmental Risks from Mine Wastes

During public environmental hearings for Niocan's proposed mine development of an extension of the SLC niobium deposit, radioactive mine wastes and impacts to groundwater quality emerged as the greatest public concerns (BAPE, 2002; 2005). Groundwater is the sole source of potable supply for local inhabitants and is tapped extensively for irrigation of market gardens and orchards in the surrounding area.

Trace metals associated with carbonatite-hosted Nb-REE deposits that could potentially impact groundwater include U, Th, and their radioactive decay products, as well as Nb and REE (USEPA, 2012; Verplanck et al., 2014; Schultz et al., 2017). Fluorine, found in apatite and biotite, may be another potential contaminant of concern (Verplanck et al., 2014). Usually, however, these elements are hosted by relatively stable, insoluble minerals, and are released slowly from carbonatite deposits in their undisturbed state. However, in mine wastes such as fine-grained tailings, geochemical reactions rates and the mobilization of contaminants are significantly enhanced because of the much larger surface contact area between groundwater and contaminant-bearing minerals. Seepage of contact waters from these wastes may then affect the quality of receiving surface waters downstream. To date, however, there has been little published information on the geochemical signature of mine tailings associated with carbonatite-hosted Nb-REE deposits.

Table 1: Annual production statistics for the St. Lawrence Columbium and Metals Corporation mine at Oka, Québec. Beginning in 1971, the company marketed its niobium production as Nb₂O₅ concentrate or as ferroniobium (FeNb).

Year	Ore Milled (tons)	Nb ₂ O ₅ Produced in concentrate (lbs)	Ore Grade % Nb ₂ O ₅	Reference
1961	n.a.*	134 006	n.a.	C&C, 1965 ³
1962	172 767	775 313	0.43	SLC, 1962
1963	291 403	1 393 536	0.40	SLC, 1963
1964	321 585	2 091 725	n.a.	SLC, 1964
1965	383 553	2 203 985	0.43	SLC, 1965
1966	406 698	2 647 667	0.47	SLC, 1966
1967	369 642	2 368 225	n.a.	SLC, 1967
1968	360 194	2 005 989	n.a.	SLC, 1968
1969	475 201	3 059 052	n.a.	SLC, 1969
1970	724 345	4 886 957	n.a.	SLC, 1970
1971	408 500	n.a.	n.a.	SLC, 1971
1972	589 147	n.a.	n.a.	SLC, 1972
1973	461 741 ¹	n.a.	n.a.	
1974	461 741	n.a.	n.a.	
1975	461 741	n.a.	n.a.	
1976	461 742	n.a.	n.a.	
Total	6 350 000	39 725 600 ²	0.46	NIOCAN, 2010

*n.a. Data not available

¹ Estimates for 1973-1976

² Estimate of equivalent Nb₂O₅ in concentrate assuming a mill recovery of 68%

³ Carbonneau and Caron (1965), Table VI

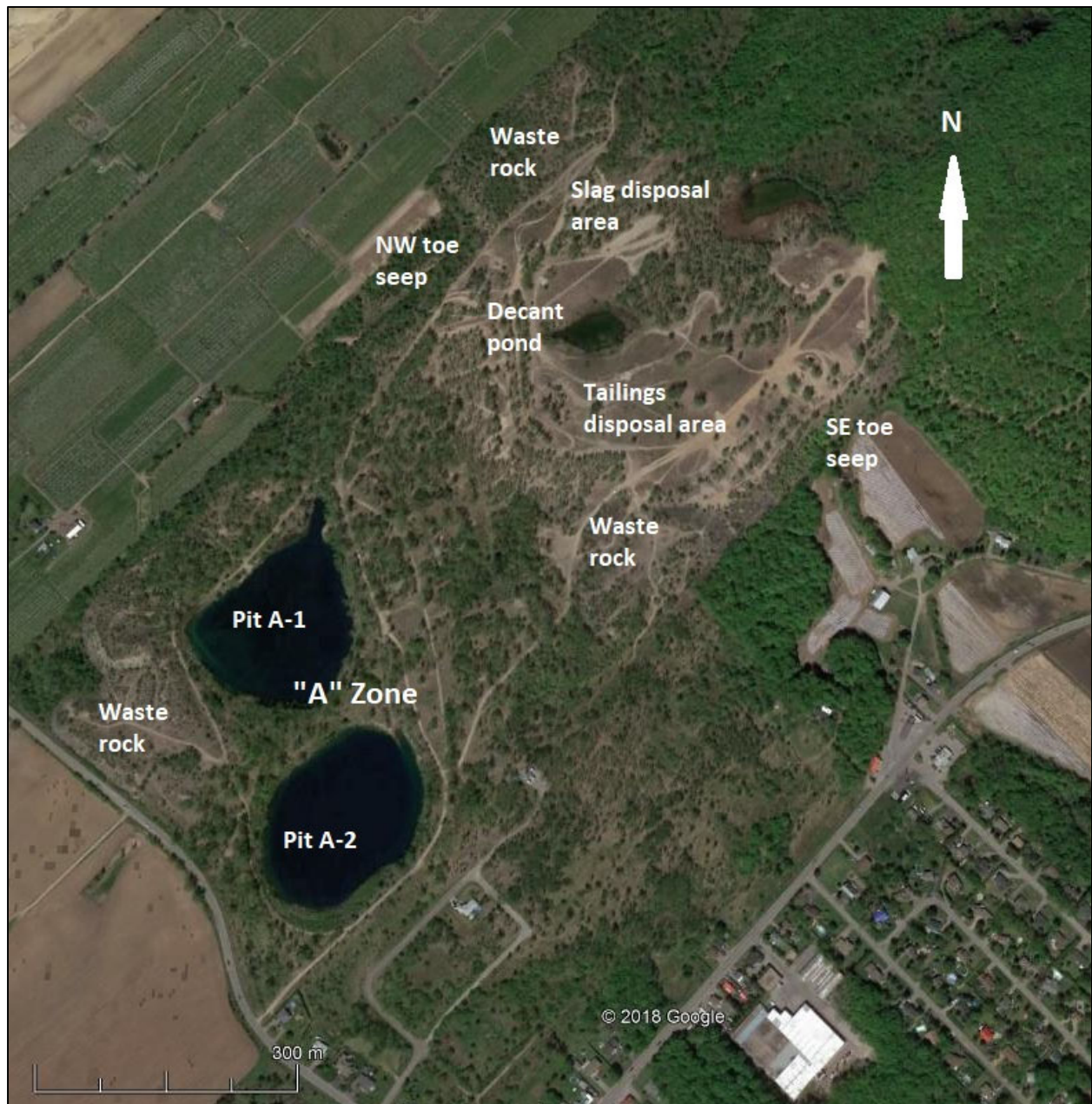


Figure 2: Satellite image with annotations showing the main features of the former SLC Oka mine site in 2018. Google Earth Image used with permission.

1.4 Previous and Related Studies

In response to public concerns related to the abandoned SLC site and to possible geogenic (natural) contamination associated with the broader Oka carbonatite complex, the GSC conducted an airborne radiometric survey of the area (GSC, 1996; Ford et al., 2001) while local health authorities performed surveys of residential radon (RRSSSL, 1998) and uranium in domestic wells (RRSSSL, 1999). Limited investigations of the SLC site were carried out in support of the environmental impact assessment for the proposed Niocan niobium mine located approximately 1.5 km to the west (BAPE, 2002, 2003, 2005; and supporting documents). As part of the GSC project on the geoenvironmental characteristics of the Oka carbonatite-hosted Nb-REE deposit, a companion open file report has been prepared on the geochemistry of ferroniobium slag and its leachate (Desbarats et al., 2020). Reports on the geochemistry of the pit lakes (M. B. Parsons, personal communication) and on the mineralogy of the different types of mine waste (J. B. Percival, personal communication) are in preparation. Master's theses on Oka slag mineralogy and leaching (Balkwill-Tweedie, 2020) and on mine waste mineralogy and leaching (Des Roches, 2020) have been completed recently. In the summer of 2017, the Ministère de l'Énergie et des Ressources Naturelles du Québec (MERNQ) carried out site characterization studies as the first phase of its remediation plans for the former SLC mine (WSP, 2018).

1.5 Objectives and Scope

To supplement sparse information on the environmental characteristics of tailings from carbonatite-hosted Nb-REE deposits, this study investigated tailings at the past-producing SLC mine near Oka, Québec. Its objectives were to address the following questions:

1. What are the physical, mineralogical, and geochemical properties of the mine tailings?
2. How does the chemistry of tailings pore-water evolve along the flow path between recharge and discharge points?
3. What are the potential contaminants of concern in tailings seepage and how do their concentrations compare to regulatory standards?
4. What are the geochemical processes that control the release, transport, and fate of species of interest in the mine tailings environment?

Tailings were sampled at closely-spaced intervals in a borehole intersecting the full thickness of the impoundment. Pore-water sampling was conducted in a multilevel piezometer nest installed adjacent to the tailings decant pond. No groundwater sampling was conducted beyond the limits of the SLC Oka mine site. Tailings effluent was sampled at the toes of the southeast and northwest embankments of the impoundment. All surface and groundwater samples were analyzed for a full suite of inorganic chemical parameters. Samples were not analyzed for radioactive decay products of ^{238}U and ^{232}Th , including radon, nor were they analyzed for microbiological parameters.

1.6 Accompanying Data Files

This report is accompanied by three data files in Microsoft Excel® format. The files are described as follows:

- of_8753_tailings properties.xlsx: file containing tailings grain-size analyses, gravimetric water contents, and densities.
- of_8753_tailings chemistry.xlsx: file containing tailings chemical analyses.
- of_8753_tailings water chemistry.xlsx: file containing chemical analyses of tailings pore-water and drainage water.

1.7 Acknowledgments

This project was supported by the Environmental Geosciences Program of the Geological Survey of Canada. Water chemistry analyses were performed by Pierre Pelchat, Isabelle Girard, and John Sekerka under the supervision of Paul Gammon. Shauna Madore and Miriam Wygergangs performed the physical measurements on tailings cores. Mineralogical work was carried out by Katherine Venance, Igor Bilot, and Thomas McLoughlin-Coleman under the supervision of Jeanne Percival. The authors would like to thank Bruce Kjarsgaard for providing helpful comments on a draft version of this report. The authors would also like to thank Malek Zetchi of the MERNQ for giving them access to their monitoring wells on the SLC site.

2. STUDY AREA

2.1 Climatic and Physiographic Setting

The former St. Lawrence Columbium mine (lat. 45.502°; long. 74.029° W) is located near Oka, Québec, approximately 36 km west of downtown Montréal. The village of Oka is part of the Deux Montagnes Municipalité Régionale de Comté (MRC), in the Basses Laurentides administrative region of Québec. The mine site is readily accessed from Montréal via highways 640 and 344.

The study area falls within the “Humid Mid-Cool Temperate” (MCTh) ecoclimatic region, which is characterized by warm summers and relatively mild winters (Strong et al., 1989). Between April and November, mean daily temperatures are above 0°C. At Environment Canada’s Oka weather station, 3.2 km from the mine site, the mean annual temperature is approximately 6.0°C. The mean summer (July) temperature is 20.3°C and the mean winter (January) temperature is -10.9°C. Monthly precipitation, usually in excess of 70 mm, is fairly evenly distributed throughout the year. Mean annual precipitation is approximately 1107 mm including 871 mm of rainfall and 236 mm of snowfall equivalent. Climate Normals for the Oka weather station are presented in Table 2.

The mine site lies within the Saint Lawrence Lowlands ecoregion, which includes the valleys of the Ottawa and Saint Lawrence rivers. The ecoregion extends from the Frontenac Axis, near Brockville in the west, to Québec City in the east. It is bounded by the Laurentian Highlands to the north and by the Eastern Québec uplands to the south (Ecozones, 2014).

Most of the lowlands are underlain by flat-lying Paleozoic strata resting on Precambrian bedrock. In the study area, however, Precambrian inliers form the Oka, Saint-André, and Rigaud Hills. Elevations range from 230 m on the highest of the Oka hills to 22 m on nearby Lac des Deux Montagnes. To the east of the study area, the Monteregian Hills form an elongated east-west trend of isolated features rising sharply above the surrounding plain and reaching elevations sometimes exceeding 400 m. They represent resistant erosional cores of early Cretaceous alkaline igneous intrusions related to the Oka and Saint-André carbonatites.

The landscape of the Saint Lawrence lowlands has been shaped by the last continental glaciation, followed by the Champlain Sea incursion and retreat, and subsequent river erosion and deposition. Most of the lowlands are blanketed by glacio-marine clay of the Champlain Sea, which can reach a thickness of over 60 m. More elevated areas, such as the Oka Hills, are covered by glacial till resting on bedrock. Reworking of the till by wave action during the marine retreat has left numerous sandy terraces representing former strand lines. Gleysols are developed on the level, poorly drained, Champlain Sea deposits whereas humo-ferric podzols and dystic brunisols are found on morainal uplands and beach terraces (Ecozones, 2014).

The broader study area is drained by the Rivière du Nord, the Rivière Rouge and the Rivière du Chêne. On the west, the mine area is drained by a ditch, which eventually joins the Ruisseau Rousse. On the east, the site is drained by a small creek known as the “effluent Saint-Pierre” (WSP, 2018). Both these water courses flow to the Grande Baie on the Lac des Deux Montagnes. Groundwater flow patterns in the study area (Nastev et al., 2005) are controlled by bedrock topography and permeability. Most recharge occurs in the Oka and Saint-André hills from which

groundwater flows radially outward before discharging in local streams or in the Lac des Deux Montagnes.

Table 2: Climate Normals (1981-2010) for the weather station at Oka, Québec (lat 45° 30.00', lon 74° 04.00' W, elevation 91.4 m asl). Data from ECCC (2018).

Month	Daily Average (°C)	Daily Maximum (°C)	Daily Minimum (°C)	Rainfall (mm)	Snowfall (cm)	Precipitation (mm)
Jan	-10.9	-6.1	-15.7	30.2	59.9	90.2
Feb	-8.9	-3.6	-14.1	21.4	50.9	72.3
Mar	-2.9	2.1	-7.9	33.9	37.7	71.6
Apr	5.7	11.0	0.4	78.1	10.9	88.7
May	12.7	18.7	6.7	93.3	0	93.3
Jun	17.8	23.6	12.0	105.4	0	105.4
Jul	20.3	26.0	14.6	97.5	0	97.5
Aug	19.0	24.9	13.2	99.9	0	99.9
Sep	14.5	20.3	8.8	95.3	0	95.3
Oct	7.7	12.6	2.8	102.6	2.5	105.1
Nov	1.3	5.1	-2.5	78.0	20.0	97.9
Dec	-6.4	-2.2	-10.4	35.3	54.3	89.6
Year	5.8	11.0	0.6	871	236	1107

The dominant vegetation in the ecoregion is mixed-wood forest consisting of sugar maple, yellow birch, eastern hemlock, and eastern white pine, with beech appearing on warmer sites. Dry sites are dominated by red and eastern white pine, and red oak. Wetter sites support red maple, black ash, white spruce, tamarack, and eastern white cedar. Characteristic wildlife includes white-tailed deer, black bear, raccoon, skunk, squirrel, and chipmunk (Ecozones, 2014).

In the study area, agriculture is the most important land use with numerous apple orchards, market gardens, and vineyards. Maple sugar bushes are found on the wooded hills. Because of its proximity to the Greater Montréal Area, farmland around Oka is threatened by urban expansion.

2.2 Geological setting

The Oka carbonatite complex has been studied extensively because of its rare and unusual rock types and minerals. Gold et al. (1967) and Gold and Vallée (1969) provide an overview of the geology of the complex and of the St. Lawrence Columbian mine, which is summarized here.

The Cretaceous (113-127 Ma, Chen and Simonetti, 2013) Oka carbonatite complex belongs to the suite of alkaline intrusions that form the Monteregian Hills. It was emplaced within a Precambrian inlier of Grenville gneisses and anorthosites metasomatically altered (finitized) locally in contact with the intrusion. The inlier is surrounded by gently dipping basal Paleozoic strata represented by Potsdam conglomerates and sandstones of the Cambrian Covey Hill and Cairnside Formations. The carbonatite complex is clearly revealed by airborne radiometric surveys as an area of high equivalent thorium concentrations (Figure 3).

The Oka complex consists of two ring-shaped structures forming a distorted “figure 8” 7.2 km long by 2.4 km wide and aligned in a NW-SE direction (Figure 4). The carbonatites are intercalated with ring-dykes and cone-sheets of silica-undersaturated alkaline and ultramafic rock types. In the larger northern ring, dips of the units are outward on the margins of the complex and inward toward the centre. In the smaller southern ring, dips of the units are outward but steepen toward the margins. The main geological units within the complex belong to four distinct series:

- Carbonatites. Most of the carbonatite, including the core of the complex, consists of a coarse-grained calcite-carbonatite (sövite) although dolomite-carbonatite (rauhaugite) is found in the northwest of the complex. The sövites typically contain 70-90 % calcite and variable amounts of accessory biotite, sodian augite, richterite, apatite, nepheline, monticellite, melilite, magnetite, perovskite, pyrochlore, niocalite, pyrite, and pyrrhotite.
- Melteigite-Ijolite-Urtite series. Rocks of this series are composed of nepheline and variable amounts of mafic minerals such as alkali pyroxene and amphiboles, biotite, and melilite. Urtite contains over 70% nepheline whereas melteigite, the mafic end-member, consists mostly of pyroxenes. Most rocks are ijolites containing up to 50 % nepheline.
- Okaite-Jacupirangite series. These ultramafic rocks are composed essentially of melilite, hauyne and pyroxene with accessory biotite, perovskite, apatite, magnetite, and calcite. Within the complex, the series exhibits considerable compositional variation.
- Alnoites. Alnoite and breccias with an alnoite matrix form late-stage dykes and plugs around the complex and in the surrounding gneisses. Alnoites are porphyritic ultramafic rocks containing phenocrysts of biotite or phlogopite, augite, olivine and magnetite in a matrix of melilite, phlogopite, perovskite, calcite, and apatite. Here, the melilite has been replaced by calcite and perovskite is largely absent.

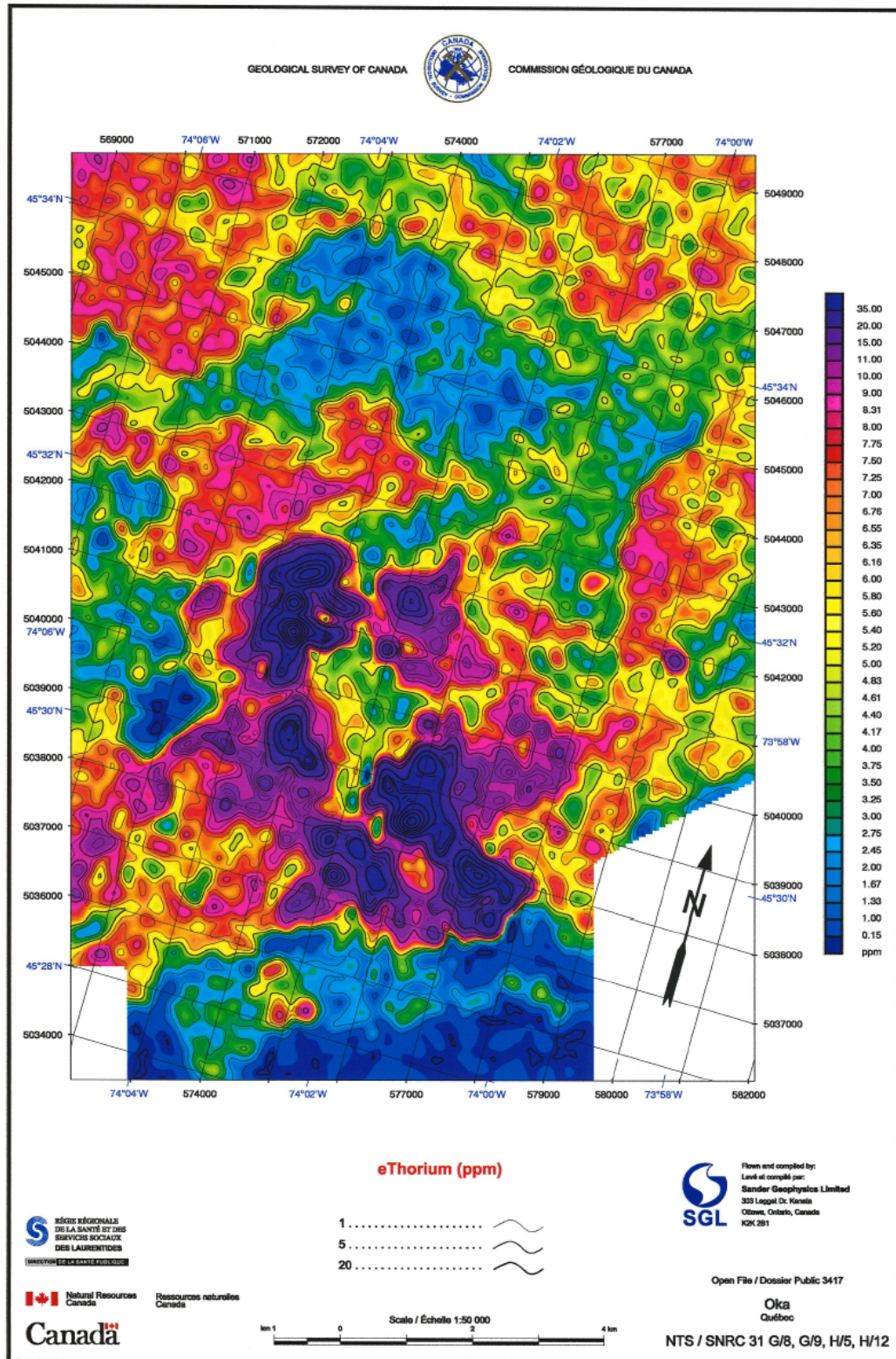


Figure 3: Airborne radiometric map of equivalent thorium concentrations in the study area. The Oka Carbonatite Complex is clearly revealed by areas of high concentrations shown in deep purple. Equivalent uranium concentrations exhibit similar anomalous patterns (GSC, 1996).

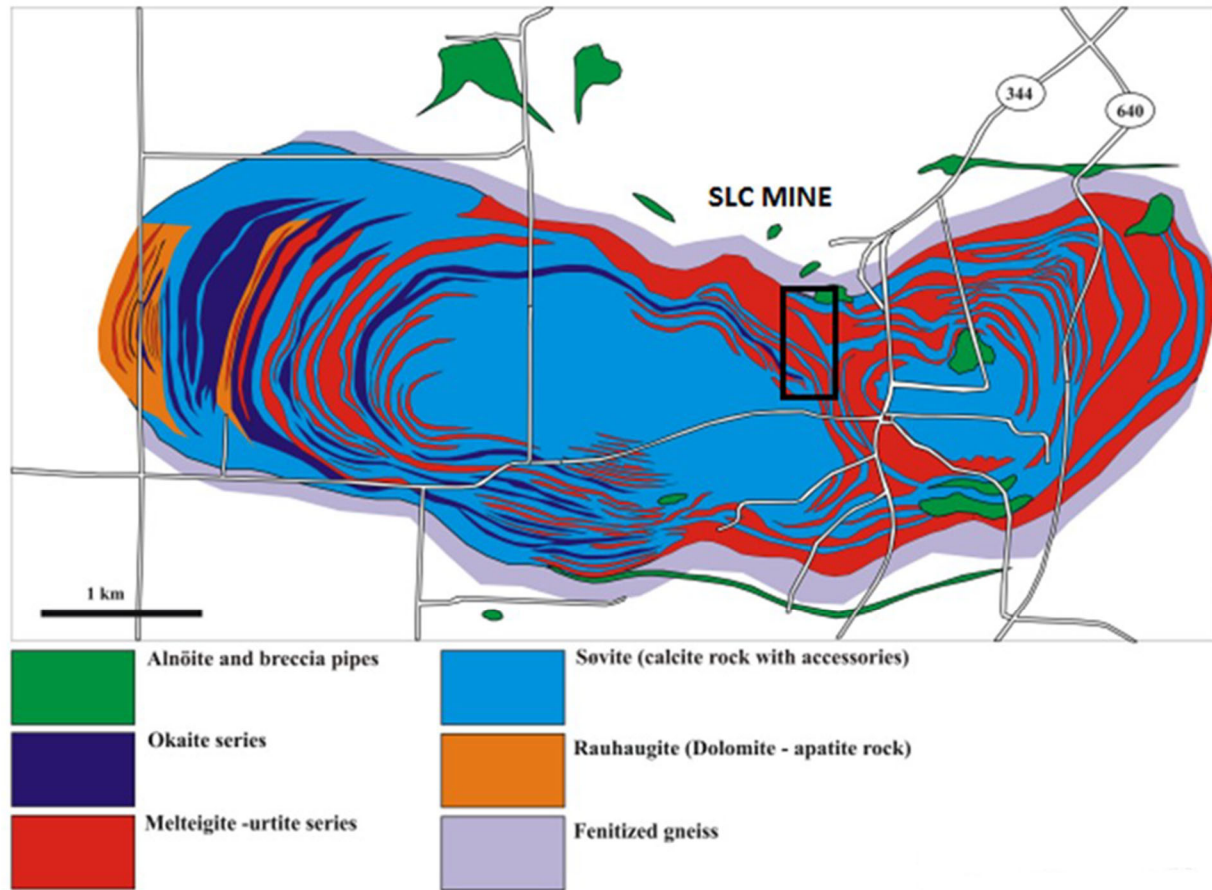


Figure 4: Simplified geology of the Oka Complex (modified from Figure 11 of Lentz et al., 2006). The black rectangle indicates the location of the SLC site.

2.3 Economic geology

All rock types within the Oka complex contain some degree of primary Nb mineralization of magmatic origin (Gold et al., 1967). However, the main economic host rock of Nb is sövite and the main Nb mineral is pyrochlore. For the purposes of grade control during mining, sövites were classified into four main types (Gold et al., 1967):

1. Early stage Nb-poor sövite containing monticellite (0.03-0.10 % Nb_2O_5)
2. Middle stage Nb-rich sövite (0.20-0.80 % Nb_2O_5)
3. Other sövite (0.10-0.50 % Nb_2O_5 and up to 1.50 % Nb_2O_5 when biotitized)
4. Late stage Nb-poor and REE-enriched sövite (0.02-0.05 % Nb_2O_5)

The silicate rocks generally contain less than 0.15 % Nb_2O_5 although in biotitized ijolite grades may reach 2 % Nb_2O_5 (Gold et al., 1967).

The SLC mine was developed on the “A” zone (Figure 2), which consists of sövites intruded by tabular bodies and irregular masses of ijolite, and some okaite (Gold et al., 1967). The main constituents of the sövite are calcite (70-85 %) and apatite (5-15 %), with variable amounts of biotite (3-15 %), pyroxene (0-10 %), monticellite (0-10 %), magnetite (0.5-5 %), sulfides (1-3 %) mainly as pyrite, pyrrhotite or sphalerite and pyrochlore (0-1 %). However, because all sövite types contain some pyrochlore, ore-waste limits were essentially economic and mining required stringent grade control measures based on sövite type (Carbonneau and Caron, 1965; Gold et al., 1967).

Pyrochlore is the main Nb-bearing mineral at the SLC mine. Other Nb-bearing minerals uniquely found within the Oka complex are perovskite (latrappite) and niocalite (Gold et al., 1967). However, neither mineral is of economic significance. Pyrochlore is a niobium oxide having the general formula (Zurevinski and Mitchell, 2004): $A_{16-x}B_{16}O_{48}(O,OH,F)_{8-y} \bullet zH_2O$ where x and y are vacant sites in the unit cell, and x, y, and z are non rational numbers. The pyrochlore structure can accommodate a wide variety of cations in the A and B sites. At the SLC mine, A site cations include Ca, Na, REE, U, Th, Sr, Mn, and Y whereas the B sites are occupied mainly by Nb, with Ti, Ta, Zr, and Fe (Gold et al., 1967). Because of these substitutions, pyrochlore compositions are extremely variable at the SLC mine and elsewhere within the Oka complex (Carbonneau and Caron, 1965; Gold et al., 1967; Petruk and Owens, 1975; Zurevinski and Mitchell, 2004). Primary pyrochlore at the SLC mine occurs mainly in disseminated form in layers or pods of specific rock types, particularly biotite-pyroxene sövite and monticellite sövite (Gold et al., 1967). Pyrochlores in these unaltered sövites contain from 47 to 66 % Nb₂O₅. Primary pyrochlore may also contain up to 0.72 % U₃O₈ and 1.51 % ThO₂ (Carbonneau and Caron, 1965). Deuteric pyrochlore is found in hydrothermally altered and biotitized zones. Although Nb grades may be higher in these zones they were avoided because the ore responded poorly to the milling process. Deuteric pyrochlore may contain up to 1.83 % U₃O₈ and 7.23 % ThO₂ (Gold et al., 1967). Uranian pyrochlore and uranpyrochlore from the SLC mine may contain up to 15.11 % and 24.79 % U₃O₈, respectively (Petruk and Owens, 1975). Mill concentrate contained on average 0.10 % U₃O₈ and 0.23 % ThO₂ (Gold et al., 1967, Table IV).

2.4 Ore Processing

Production of a marketable pyrochlore concentrate was challenging due to the variable composition of the ore and the milling process required continual improvement throughout the mine life aimed at increasing recovery efficiency (Carbonneau and Caron, 1965; Gibson et al., 2015). Carbonneau and Caron (1965) provide a relatively detailed description of the SLC milling process as it was in the early years of production. Their description is summarized here with a view toward identifying the various waste streams reporting to the tailings disposal area.

The composition of the ore feed was approximately 0.8-1.9 % pyrochlore, 50-80 % calcite, 4-8 % apatite, 0-11 % pyroxene, 3-15 % micas, 2-3 % magnetite, 1-3 % sulfides (pyrite, pyrrhotite, sphalerite), and 1-30 % other silicates including chlorite. After crushing, grinding, and desliming, the ore was fed to a sulfide flotation circuit for removal of pyrite, pyrrhotite, and sphalerite. The tailings from this circuit were sent to a magnetic separator for recovery of a magnetite by-product and additional pyrrhotite. The resulting non-magnetic concentrate was deslimed by cycloning and was then sent to a rougher bulk flotation circuit. The tailings from this circuit were sent to a cyclone with coarse particles being sent to the tailings impoundment and calcite fines being recovered for marketing as agricultural lime. The concentrate from the rougher circuit contained most of the pyrochlore as well as silicates and some apatite. After further desliming and magnetic separation steps, this concentrate was sent to a two-stage cleaning circuit in which pyrochlore was first floated

from apatite under neutral pH and then from micas and other silicates under acidic conditions promoted by the addition of hydrofluoric acid. Remaining silicates and sulfides in the concentrate were removed through two reverse flotation stages in series. The final pyrochlore concentrate contained on average 60 % Nb₂O₅, 0.10 % U₃O₈ and 0.23 % ThO₂ (Gold et al., 1967).

The types of mill waste sent to the tailings impoundment included float from the sulfide recovery circuits, coarse tailings from the rougher circuit and fines from desliming and the cleaning circuits. The sulfide float was composed mainly of pyrite, pyrrhotite, and sphalerite. The coarse tailings were composed mainly of calcite and apatite. The finer tailings and slimes contained biotite, apatite, chlorite and pyroxene. Although sometimes abundant in the ore, niocalite was not recovered by the milling process and may be found in the tailings along with unrecovered pyrochlore.

3. METHODOLOGY

3.1 Site selection

A site for detailed investigations of tailings solids and porewater chemistry was selected in the central basin of the impoundment immediately southeast of the decant pond (Figures 2, 5). This location was chosen, in part, because a large-diameter steel-cased 18 m observation well (OW-01) was already present. Unfortunately, logging information and completion details for this well were unavailable. The shallow water table near the decant pond was another reason for selecting this location. This facilitated the installation of drive-point piezometers for the sampling of tailings pore-water. Elsewhere in the impoundment, on the former tailings beaches, the water table may be over 20 m below the surface. In order to inform the interpretation of pore-water geochemical profiles from the piezometers, a cored borehole (OW-01-G) was drilled nearby. This 25 m borehole penetrated the entire thickness of accumulated tailings and was completed as a monitoring well screened in sandy material at the base of the impoundment.

3.2 Piezometer installation

A nest of 5 drive-point piezometers (Figure 5) was installed next to the pre-existing monitoring well (OW-01). The piezometers were constructed using Solinst ® Model 615 stainless steel drive points with 1 ft. screens connected to 3/4 in. commercial black steel risers. Sampling of porewater chemistry was conducted through 5/8 in. x 1/2 in. LDPE tubing running through the risers and attached to barbed fittings within the drive points. The piezometers were advanced using a Pionjar ® Model 120 pneumatic drill (Figure 6). They were completed at depths of 1.8 m (6 ft.), 2.7 m (9 ft.), 3.6 m (12 ft.), 5.4 m (18 ft.), and 7.3 m (24 ft.) below the ground surface (Figure 7). An additional piezometer (OW-01-F) was installed within the decant pond 0.9 m (3 ft.) below the bottom (Figure 5). It was constructed using a Solinst ® Model 601 Schedule 40 PVC drive point with a 1 ft. screen connected to a 3/4 in. PVC riser.



Figure 5: Layout of piezometers and observation wells in the tailings impoundment southeast of the decant pond. OW-01 coordinates: longitude -74.02625° W and latitude 45.504555° N. Google Earth image used with permission.

3.3 Borehole drilling and monitoring well installation

A stratigraphic borehole was drilled in order to obtain a detailed vertical profile of tailings textural, geochemical, and mineralogical variations across the entire thickness of the impoundment. Drilling was performed on September 20th 2018 by George Downing Estate Drilling Ltd. using a Central Mine Equipment (CME) Model 55 track-mounted rig. Hollow-stem augering (Figure 8) was used to 7.6 m (25 ft.). Because of the soft nature of the fine-grained saturated tailings, the drilling method then was switched to direct circulation with casing advancement (Figure 9) until the end of hole at 25 m (80 ft.). Core samples were obtained at regular 0.75 m (2.5 ft.) intervals by drive sampling using a 2 in. x 24 in. split-tube sampler. The borehole was completed as a 2 in. PVC monitoring well (OW-01-G) with a 1.5 m (5 ft.) screen emplaced in sandy tailings at the base of the impoundment (Figure 10). OW-01-G coordinates are: longitude -74.02628° W and latitude 45.50450° N.



Figure 6: Installation of drive-point piezometers at the study site. Photograph by J.B. Percival. NRCan Photo Database number 2020-635.



Figure 7: Piezometers OW-01-A, B, C, D, E. Observation well OW-01 is in the background. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-636.



Figure 8: Drilling of the stratigraphic borehole by hollow-stem augering. Photograph by J.B. Percival. NRCan Photo Database number 2020-637.



Figure 9: Drilling of the stratigraphic borehole by casing advancement. Photograph by J.B. Percival. NRCan Photo Database number 2020-638.



Figure 10: Completion of the stratigraphic borehole as observation well OW-01-G. Photograph by J.B. Percival. NRCan Photo Database number 2020-639.

3.4 Water level and effluent flow measurements

Spot measurements of water levels in piezometers and monitoring wells were performed manually using a Solinst 102M ® Mini Water Level meter. Automated measurements of water levels in OW-01 were recorded every 6 hours using an In-Situ TROLL 9500® data logger (24/06/2016 – 25/10/2016) and an In-Situ Aqua TROLL 200® data logger (10/06/2017 – 17/01/2018).

Measurements of effluent discharge rates in the channel at the toe of the southeastern tailings embankment were made using a 45° V-notch weir fabricated from 1/8 in. steel plate. Flow rate (Q) was calculated from the height of water above the notch (h) according to the formula: $Q = 1.38 h^{2.5}$ where h is in m and Q is in m^3/s (Streeter and Wylie, 1975).

3.5 Field sampling procedures for water chemistry

Groundwater samples from drive-point piezometers were collected using a Waterra Pegasus Athena® 12V peristaltic pump (Figure 11a). Samples from the larger diameter monitoring wells were taken using a Waterra Tornado® 5-stage 12V submersible pump (Figure 11b). Electrical power for the pumps was supplied by Power Sonic® sealed rechargeable 12 Volt (18 Amp-Hour) Absorbed Glass Mat (AGM) deep cycle battery. Groundwater samples from the MERNQ observation wells were obtained using the dedicated inertial pump in each well. The decant pond and flowing surface waters were sampled directly.

Groundwater from each piezometer or monitoring well was collected in flow cells constructed out of food-grade plastic containers fitted with a PVC inlet connector to the pump tubing, an outlet spigot, as well as ports for field parameter sensors. Groundwater pH was measured using a Denver Instruments® U-10 meter calibrated daily over the 7-10 range. Specific conductance was measured using an Oakton® CON 11 meter calibrated daily for the 0 to 1413 $\mu\text{S}/\text{cm}$ range. Temperature was also measured using the Oakton meter. Field parameters were recorded once values had stabilized in the flow cell.

Groundwater was sampled from the flow cells using all-plastic 50 mL Norm-Ject® syringes triple-rinsed in the water to be sampled. Unfiltered samples for (total) cations were collected in triple-rinsed 60 mL HDPE narrow-mouth bottles (Nalgene® 2002-0002). Filtered (Millipore® Sterivex™ 0.45 μm) samples for anions and cations were collected in 60 mL HDPE narrow-mouth bottles triple-rinsed with filtered water to be sampled. Bottles for cation samples were acidified each evening at the hotel room to pH <2 by adding 0.5 mL of 8 N ultrapure nitric acid. Duplicate water samples were collected at every tenth sample site, and sample blanks were prepared once per field campaign. Samples were stored at 4° C in a hotel room refrigerator pending return to Ottawa.

Groundwater sampling was carried out in June and October of 2016, in June and November of 2017 and in September 2018. Additional sampling of tailings effluent was conducted during site visits in July and September of 2017, and in May, June and November of 2018. Over the three field seasons, a total of 60 water samples were obtained including duplicates.

a)



b)



Figure 11: a) Groundwater sampling from drive-point piezometers using a peristaltic pump. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-640; b) Groundwater sampling from observation wells using a submersible pump. Photograph by J.B. Percival. NRCan Photo Database number 2020-641.

3.6 Laboratory Methods

Tailings granulometry and physical properties

Tailings grain-size, density, and gravimetric water content analyses were performed at the Mineralogy and Physical Properties laboratory of the Geological Survey of Canada in Ottawa. For class sizes greater than 0.063 mm, grain-size measurements were made using wet sieving followed by dynamic image analysis (Microtrac Camsizer®). For class sizes smaller than 0.063 mm, grain-sizes were determined by laser diffraction particle sizing (Beckman-Coulter LS 13-320®). Densities were measured on oven dried samples using a Micromeritics AccuPyc 1330® gas displacement pycnometer. Gravimetric water content was measured according to ASTM-D2216. Gravimetric water content is defined as the ratio of the mass of water contained in the pore spaces of the material to the solid mass of particles, expressed as a percentage.

Tailings geochemistry

Analyses of tailings solids were performed by Bureau Veritas Commodities Canada Ltd. at their Vancouver laboratory. Measurements were made using lithium borate fusion with Inductively Coupled Plasma - Emission Spectroscopy (ICP-ES) for whole rock analyses and most minor elements, and Inductively Coupled Plasma - Mass Spectrometry (ICP-MS) for trace elements. Aqua Regia digestion with ICP-MS was used for Mo, Cu, Pb, Zn, Ni, As, Cd, Sb, Bi, Ag, Au, Hg, Tl, and Se. Fluorine was determined using hydropyrolysis and ion chromatography. Certified reference materials included a silica blank, GSR-6, and Oka-1 (Steger and Bowman, 1981).

Groundwater chemistry

All groundwater samples were analyzed at the Inorganic Geochemical Research Laboratory of the Geological Survey of Canada in Ottawa. Analyses of major elements were performed by Inductively Coupled Plasma - Atomic Emission Spectroscopy (ICP-AES) using a Perkin-Elmer 3000 DV. Analyses of trace elements were performed using Inductively Coupled Plasma - Mass Spectrometry (ICP-MS) with a Thermo Corporation X-7 Series II. Determinations of anion concentrations were made with a Dionex DX-600 ion chromatograph using an AS-18 column and gradient elution. Alkalinities were determined using a PC-Titrate System. Element suites analyzed by each method and their corresponding detection limits are listed in Table 3. For each sample batch, analyses were performed on one or more certified standards of known concentrations. Analytical results for field blanks from each trip were at or below detection limits for all elements.

For most (53/59) samples, charge balance errors varied between -6.93 % and 7.24 % with a mean value of 0.04 %. Large (circa 30%) charge balance errors were obtained in two samples of very low-conductivity pond water and in a high DOC (229 mg/L) sample of green-coloured groundwater from OW-01-G. Charge balance errors in the remaining three samples were on the order of 10 %. The charge balance error was not calculated for one sample from OW-01-E for which only total cation concentrations were available.

Table 3: Analytical methods and detection limits for elements analyzed at the GSC.

ICP-MS Element Suite			
Li (0.02)*	Be (0.005)	B (0.5)	Al (2)
Ti (0.5)	V (0.1)	Cr (0.1)	Mn (0.1)
Co (0.05)	Ni (0.2)	Cu (0.1)	Zn (0.5)
Ga (0.01)	Ge (0.02)	As (0.1)	Se (0.1)
Rb (0.05)	Sr (0.5)	Y (0.01)	Zr (0.05)
Nb (0.01)	Mo (0.05)	Ag (0.005)	Cd (0.02)
In (0.01)	Sn (0.01)	Sb (0.01)	Te (0.02)
Cs (0.01)	Ba (0.2)	La (0.01)	Ce (0.01)
Pr (0.005)	Nd (0.005)	Sm (0.005)	Eu (0.005)
Tb (0.005)	Gd (0.005)	Dy (0.005)	Ho (0.005)
Er (0.005)	Tm (0.005)	Yb (0.005)	Lu (0.005)
Hf (0.01)	Ta (0.01)	W (0.02)	Re (0.005)
Tl (0.005)	Pb (0.01)	Th (0.02)	U (0.005)
* lower detection limit in ug/L shown in brackets			

ICP-AES Element Suite			
Br (0.05)*	Ca (0.02)	Cl (0.1)	Fe (0.005)
K (0.05)	Mg (0.005)	Na (0.05)	P (0.05)
S (0.05)	Sc (0.001)	Si (0.02)	
* lower detection limit in mg/L shown in brackets			

IC Element Suite			
F (0.01)*	Cl (0.01)	SO ₄ (0.02)	Br (0.02)
NO ₃ (0.02)	PO ₄ (0.02)		
* lower detection limit in mg/L shown in brackets			

4. RESULTS

4.1 Tailings physical properties

The tailings disposal area at the SLC mine (Figure 2) covers an area of approximately 19 ha (WSP, 2018) bounded by rising terrain to the northeast, and by steep berms constructed from waste rock and coarse tailings on the northwest, southeast, and southwest. In the central basin of the impoundment, the thickness of tailings reaches 29.6 m at MERN PO-13 (WSP, 2018), and 24.5 m at the GSC stratigraphic borehole (OW-01-G).

Tailings samples from OW-01-G exhibit a continuum of textures varying between clayey silt and sand (Figure 12). The vertical succession of these textures (Figure 13) was determined by tailings disposal practices over the life of the mine, in particular proximity to spigot points. A 4.4 m layer of silty sand blankets the base of the impoundment. This unit is overlain by 17.5 m of silts and clayey silts with a few interbedded sandy horizons. In turn, this sequence is capped by 2.6 m of coarse sand. Whereas fine-textured tailings are expected in the central basin of the impoundment near the decant pond, these coarse tailings are not. This material may have been bulldozed inward from tailings beaches on the periphery of the impoundment as a dust suppression measure in the early 1980s (WSP, 2018).

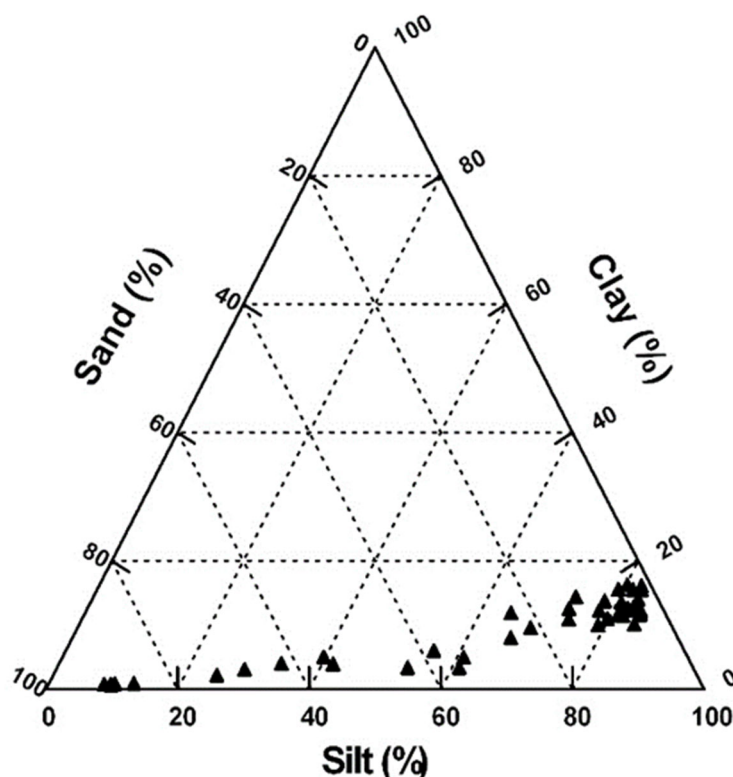


Figure 12: Tailings grain-size textures observed in split-spoon samples from the GSC stratigraphic borehole.

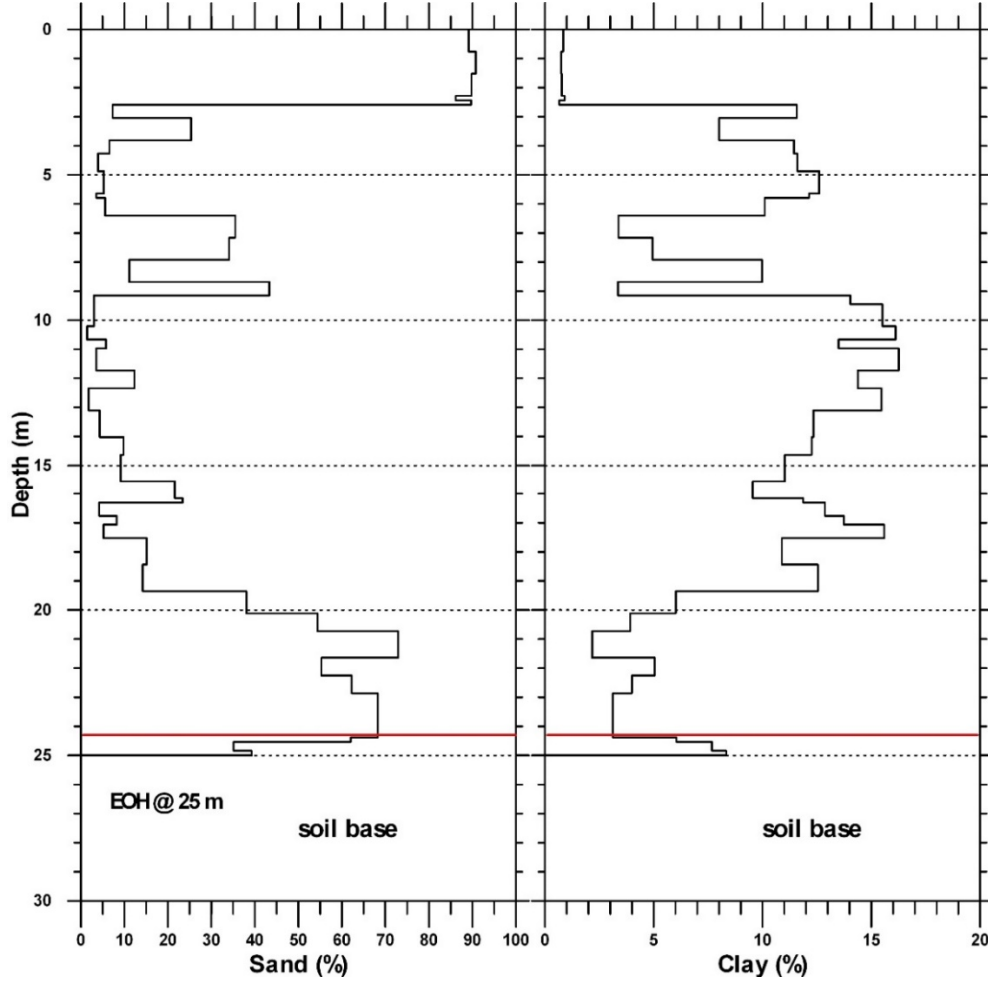


Figure 13: Downhole textural variations of tailings in the GSC stratigraphic borehole.

Across the unsaturated zone, water content increases with depth from 4 % to 20 % (Figure 14). Saturated water contents vary between 20 % in the sandy layer at the base of the tailings to over 47 % in a silty layer 4 m below ground surface. Average water content is 30 %. Tailings solids densities are generally between 2.8 and 2.9 g/cm³ (Figure 14) with an average of 2.82 g/cm³. Density spikes at depths of 9 and 20 m reflect larger amounts of apatite or magnetite in the tailings. For samples in the saturated zone, porosity (n) can be calculated from gravimetric water content (W), solid density (ρ_s), and water density (ρ_w) according to:

$$n = W \rho_s / (\rho_w + W \rho_s)$$

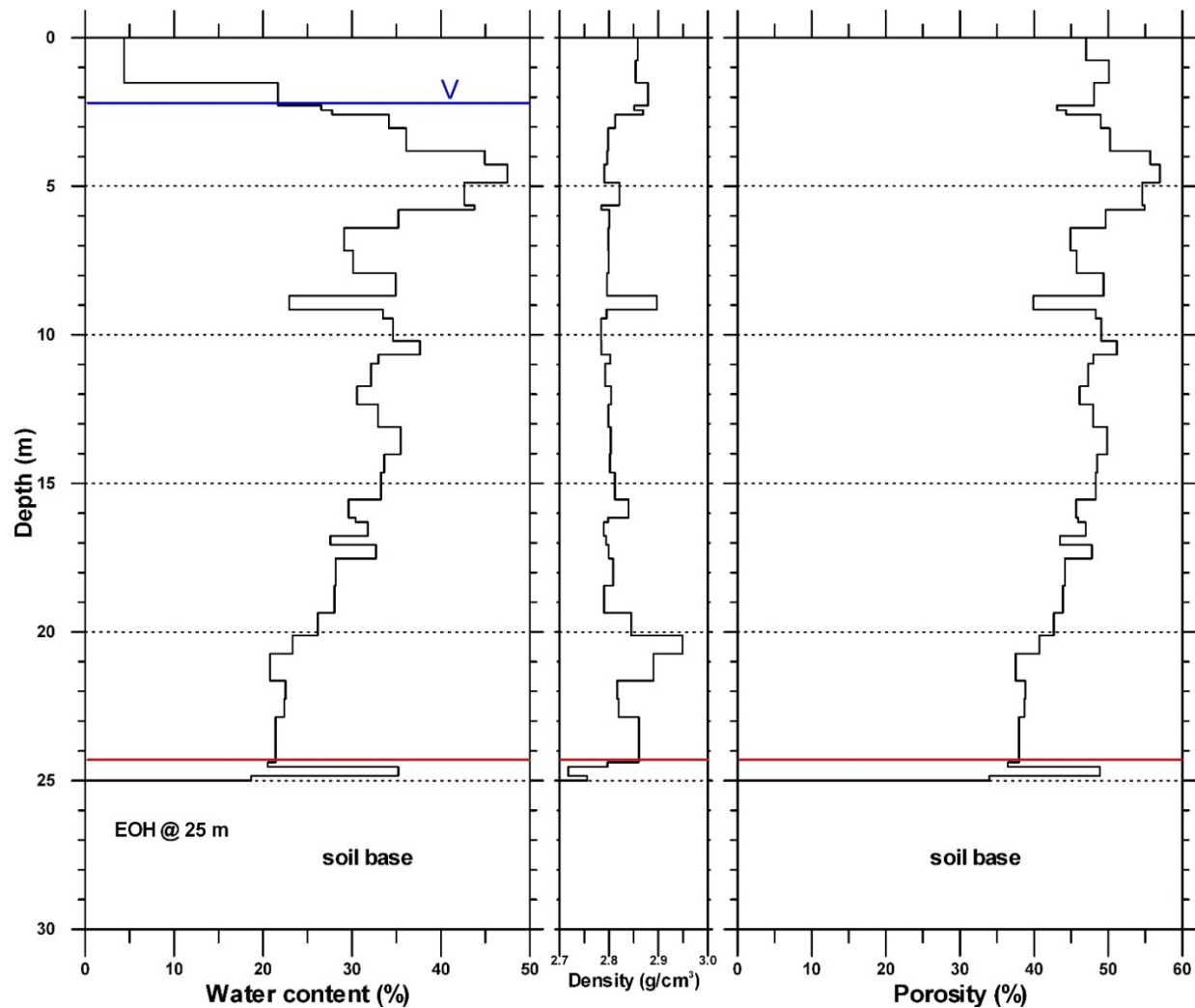


Figure 14: Downhole profiles of gravimetric water content, density, and calculated porosity in the GSC stratigraphic borehole. The approximate location of the water table is marked by a “V” on the water content profile.

For the three samples above the water table, porosity was estimated by polynomial regression on the 0.18 to 0.25 mm grain size fraction (f):

$$n = 50.801 - 2.946 f + 0.169 f^2$$

Calculated porosities in the tailings vary between 37 % in the sandy basal layer and 57 % in silty material 4 m below ground surface. The average porosity is 47 %.

The hydraulic conductivity of tailings can be estimated from grain-size distribution data using semi-empirical formulae such as those of Beyer (1964) and Aubertin et al. (1996). The Beyer equation, referred to as the “Breyer” equation in Kresic (2007), is written:

$$K = g/v C d_{10}^2$$

Where g/v is equal to 1.09×10^7 1/m.s at 25°C and C is given by:

$$C = 6 \times 10^{-4} \log (500/U)$$

The coefficient of uniformity (U) is given by d_{60}/d_{10} where the characteristic lengths, d_{10} and d_{60} , are the 10th and 60th percentiles of the grain-size distribution, respectively. The equation of Aubertin et al. (1996), developed specifically for mine tailings, is written:

$$K = g/v C e^{5.16} 1/(1+e) U^{1/3} d_{10}^2$$

Where C is equal to 0.02 for hard rock mines, and the void ration (e) is given by $n/1-n$. Downhole profiles of hydraulic conductivity estimates based on the two equations show broadly similar patterns (Figure 15) although the equation of Aubertin et al. (1996) yields values that are generally an order of magnitude higher than those obtained with the Beyer formula (Table 4).

Table 4: Summary statistics of hydraulic conductivity estimates obtained from grain-size distribution data using the equations of Beyer (1964) and Aubertin et al. (1996).

	Beyer (1964)	Aubertin et al. (1996)
Minimum (m/s)	1.62 E-08	1.31 E-07
Maximum (m/s)	6.03 E-05	6.78 E-04
Arithmetic mean (m/s)	6.29 E-06	4.33 E-05
Geometric mean (m/s)	1.45 E-07	1.52 E-06
Log standard deviation	1.16	0.97

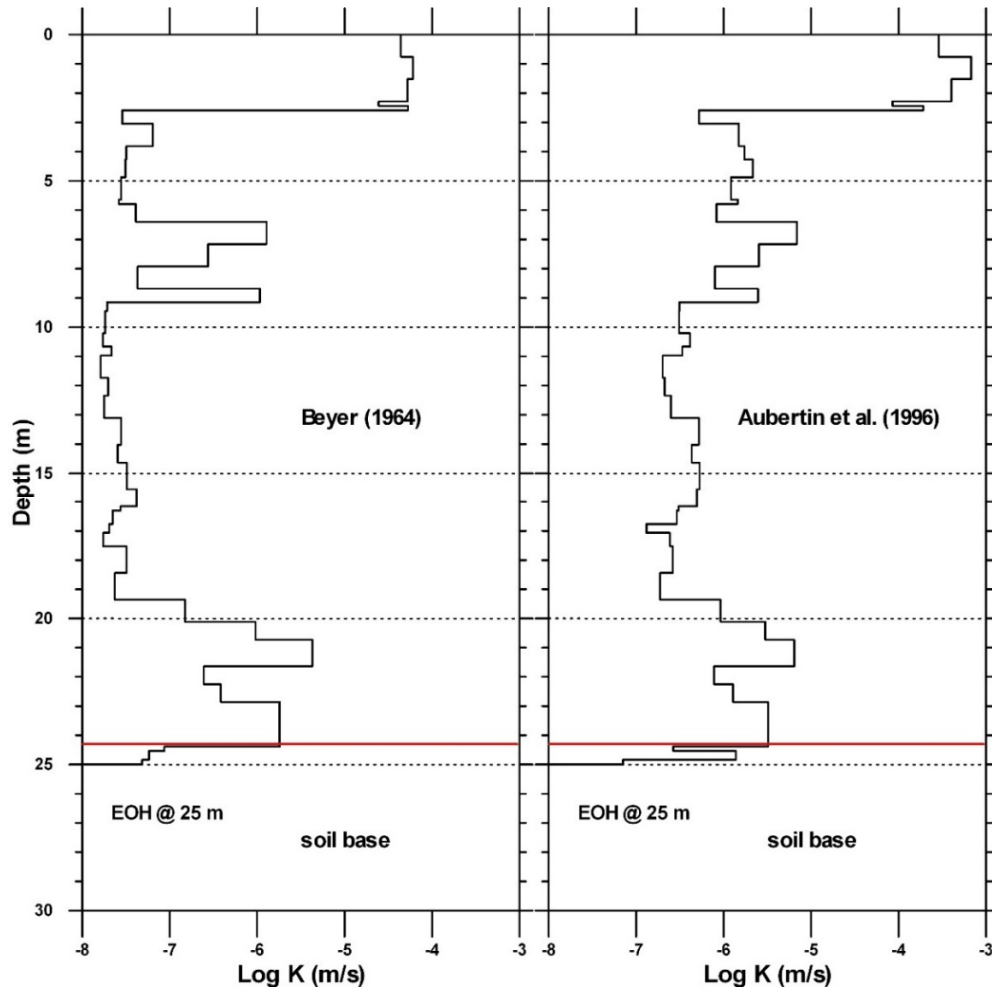


Figure 15: Downhole profiles of hydraulic conductivities estimated using the formulas of Beyer (1964) and Aubertin et al. (1996).

4.2 Tailings Mineralogy

The mineral phases forming the tailings, together with their compositions and stabilities, are key factors controlling the mobilization of trace elements during weathering processes. This section presents a summary of mineralogical observations on the tailings pending completion of a detailed report on the mineralogy of the different mine wastes at the SLC site (J. B. Percival, personal communication). Mineralogical analyses of the tailings are also reported in Des Roches (2020).

Minerals identified by bulk X-ray diffraction (XRD) on tailings samples are listed in Table 5 along with their estimated average modal abundances and ideal compositions. Depth profiles of estimated mineral abundances by XRD on bulk tailings and the clay size-fraction are presented in Tables 6 and 7, respectively. Calcite is the dominant mineral by far (64-89 %) and is distributed fairly evenly throughout the thickness of the tailings impoundment (Table 6). Dolomite is present in small amounts. The most important silicate mineral is biotite. Its abundance varies within the

tailings and in some layers it reaches 17 % (Table 6). Nepheline, in minor amounts, is distributed fairly uniformly. Chlorite, smectite, and kaolinite are detected across the tailings thickness, mainly in the clay size fraction (Table 7). Fluorapatite is the third most abundant mineral in the tailings (Table 5), reaching up to 22 % in one layer (Table 6). Other minerals detected by bulk XRD include monticellite and magnetite (Table 6). Pyrite (5 %) and clinopyroxene (3 %) were found in sample DUA-18-T28 and perovskite (1 %) was found in sample DUA-18-T29 (not shown). In the clay size-fraction, perovskite and gypsum were noted (Table 7). The last three samples in Tables 6 and 7 (shaded) are from the underlying soil and their mineralogy is dominated by quartz, plagioclase and K-feldspar (not shown).

Table 5: Summary of minerals found in SLC tailings based on bulk XRD analyses. Abundances are expressed as a weight % of minerals present above the XRD detection limit.

Mineral	Modal abundance (% weight)			Ideal Formula
	Min.	Max.	Median	
dolomite	0.0	1.0	tr	$\text{Ca(Fe,Mg,Mn)(CO}_3)_2$
biotite	6.0	17.0	12.0	$\text{K(Mg,Fe)}_3\text{AlSi}_3\text{O}_{10}(\text{F,OH})_2$
calcite	64.0	89.0	82.0	CaCO_3
chlorite	0.0	5.0	tr	$(\text{Fe,Mg,Al})_6(\text{Si,Al})_4\text{O}_{10}(\text{OH})_8$
fluorapatite	2.0	22.0	4.0	$\text{Ca(PO}_4)_3(\text{F,OH})$
gypsum	0.0	tr	tr	$\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$
kaolinite	0.0	tr	tr	$\text{Al}_2(\text{OH})_4\text{Si}_2\text{O}_5$
magnetite	0.0	1.0	tr	Fe_3O_4
monticellite	0.0	1.0	tr	CaMgSiO_4
nepheline	0.0	2.0	1.0	$(\text{Na,K})\text{AlSiO}_4$
pyrite	0.0	5.0	tr	FeS_2
perovskite	0.0	1.0	tr	CaTiO_3
smectite	0.0	4.0	tr	$(\text{Na,Ca})_{0.33}(\text{Al,Mg})_2(\text{Si}_4\text{O}_{10})(\text{OH})_2 \cdot n\text{H}_2\text{O}$

tr: trace

The semi-quantitative XRD modal abundance estimates are constrained by a detection limit of about 3 % weight and minerals present below that threshold are not reported in Table 5. Scanning Electron Microscopy (SEM) was used to identify minerals occurring below the XRD detection limit. These include barite, melilite, pyrochlore, and sphalerite. In addition to the minerals identified here, Des Roches (2020) used SEM to detect trace amounts of pyrrhotite, jarosite, niocalite, REE calcium fluoro-carbonates (parisite and synchesite), a REE-Sr carbonate (ancylite), and a REE silico-phosphate (britholite). Britholite and fluorapatite are nearly indistinguishable by XRD and some britholite may have been misidentified in this study.

Table 6: Tailings mineralogy by bulk XRD with semi-quantitative modal abundances. Abundances are expressed as a weight % of minerals present above the XRD detection limit. Shaded samples are in the soil underlying the impoundment.

Sample No.	Depth from (m)	Depth to (m)	cal	dol	bio	chl	smc	kao	mtc	nph	mag	fap
DUA-18-T1	0.00	0.61	78	tr	12	tr	4	-	-	1	-	5
DUA-18-T2	0.76	1.37	79	tr	13	-	2	-	-	1	-	5
DUA-18-T3	1.52	2.13	85	tr	6	-	2	-	-	-	-	7
DUA-18-T4A	2.29	2.44	82	1	12	tr	1	-	-	-	tr	4
DUA-18-T4B	2.44	2.59	84	1	11	tr	-	-	-	-	tr	4
DUA-18-T4C	2.59	2.90	84	tr	7	2	-	tr	-	-	-	6
DUA-18-T5	3.05	3.66	82	tr	11	2	1	-	-	1	-	3
DUA-18-T6A	3.81	4.27	79	1	12	4	1	tr	tr	1	tr	2
DUA-18-T6B	4.27	4.42	80	tr	10	2	-	-	1	1	-	4
DUA-18-T7	4.88	5.18	74	tr	16	2	-	-	tr	2	-	6
DUA-18-T8A	5.64	5.79	77	-	16	2	-	-	-	1	tr	4
DUA-18-T8B	5.79	5.94	75	1	14	4	-	-	-	1	-	5
DUA-18-T9	6.40	6.71	79	-	11	3	-	-	-	1	-	6
DUA-18-T10	7.16	7.47	73	-	16	5	-	-	-	1	tr	5
DUA-18-T11	7.92	8.23	77	tr	12	5	-	-	-	1	-	5
DUA-18-T12	8.69	8.99	77	tr	13	3	-	tr	-	1	tr	6
DUA-18-T13A	9.14	9.45	84	-	13	-	-	-	-	1	-	2
DUA-18-T13B	9.45	9.75	86	-	11	tr	-	-	-	1	-	2
DUA-18-T14	10.21	10.52	84	-	12	tr	-	-	-	1	-	3
DUA-18-T15A	10.67	10.97	82	1	12	2	-	-	-	tr	-	3
DUA-18-T15B	10.97	11.28	84	tr	12	1	-	-	-	tr	-	3
DUA-18-T16	11.73	12.04	85	tr	10	1	-	-	-	1	-	3
DUA-18-T17	12.34	12.65	83	1	11	2	-	-	-	1	-	2
DUA-18-T18	13.11	13.41	87	1	8	1	-	-	-	1	-	2
DUA-18-T19	14.02	14.33	81	1	11	3	-	-	-	1	-	3
DUA-18-T20	14.63	14.94	81	1	10	3	-	tr	-	1	-	4
DUA-18-T21	15.54	15.85	81	1	12	tr	-	-	-	1	-	5
DUA-18-T22A/B	16.15	16.61	86	tr	10	-	-	-	-	1	-	3
DUA-18-T23A/B	16.76	17.37	85	tr	8	tr	-	-	-	1	-	6
DUA-18-T24	17.53	18.14	84	1	12	tr	-	-	-	tr	-	3
DUA-18-T25	18.44	18.75	81	-	15	-	-	-	-	tr	-	4
DUA-18-T26	19.35	19.66	81	-	14	-	-	-	-	tr	-	5
DUA-18-T27	20.12	20.42	68	tr	10	-	-	-	-	tr	tr	22
DUA-18-T28	20.57	21.18	64	tr	10	-	-	-	-	tr	-	18
DUA-18-T29	21.34	21.95	89	tr	6	-	-	-	-	-	-	4
DUA-18-T30	22.25	22.56	84	-	11	-	2	tr	-	-	-	3
DUA-18-T31	22.86	23.47	76	tr	17	-	-	-	-	-	-	7
DUA-18-T32A	24.38	24.54	8	-	-	1	-	-	-	-	5	-
DUA-18-T32B	24.54	24.84	4	-	-	-	-	-	-	-	4	-
DUA-18-T32C	24.84	25.05	tr	27	-	4	-	-	-	-	-	-

cal: calcite; dol: dolomite; bio: biotite; chl: chlorite; smc: smectite; kao: kaolinite; mtc: monticellite; nph: nepheline; mag: magnetite; fap: fluorapatite; tr: trace; -: not detected.

Table 7: Tailings mineralogy by XRD on the clay (< 2 µm) size fraction with estimated modal abundances. Abundances are expressed as a weight % of minerals present above the XRD detection limit. Shaded samples are in the soil underlying the impoundment.

Sample No.	Depth from (m)	Depth to (m)	cal	dol	bio	chl	smc	kao	fap	gyp	prv
DUA-18-T1	0.00	0.61	77	1	9	tr	2	-	11	-	-
DUA-18-T2	0.76	1.37	71	1	17	-	1	-	10	-	-
DUA-18-T3	1.52	2.13	77	2	9	-	1	-	11	-	-
DUA-18-T4A	2.29	2.44									
DUA-18-T4B	2.44	2.59									
DUA-18-T4C	2.59	2.90	65	1	25	3	tr	1	5	-	-
DUA-18-T5	3.05	3.66									
DUA-18-T6A	3.81	4.27	71	2	17	5	tr	-	5	-	-
DUA-18-T6B	4.27	4.42	77	1	14	3	tr	1	4	-	-
DUA-18-T7	4.88	5.18	71	1	17	6	1	-	4	-	-
DUA-18-T8A	5.64	5.79	83	1	11	2	-	-	3	-	-
DUA-18-T8B	5.79	5.94	78	1	14	3	tr	-	4	-	-
DUA-18-T9	6.40	6.71	74	1	19	4	tr	tr	2	-	-
DUA-18-T10	7.16	7.47	78	1	17	3	tr	-	1	-	-
DUA-18-T11	7.92	8.23	76	1	18	3	-	tr	2	-	-
DUA-18-T12	8.69	8.99	75	tr	18	3	-	3	1	-	-
DUA-18-T13A	9.14	9.45	84	1	11	1	tr	tr	3	-	-
DUA-18-T13B	9.45	9.75	83	1	12	1	-	-	3	-	-
DUA-18-T14	10.21	10.52	85	1	10	2	-	tr	2	-	-
DUA-18-T15A	10.67	10.97	81	1	13	2	-	-	3	-	-
DUA-18-T15B	10.97	11.28	81	1	13	2	-	-	3	-	-
DUA-18-T16	11.73	12.04	85	1	9	2	-	-	3	-	-
DUA-18-T17	12.34	12.65	84	1	8	4	tr	tr	3	-	-
DUA-18-T18	13.11	13.41	82	1	9	4	-	-	4	-	-
DUA-18-T19	14.02	14.33	80	tr	14	3	-	-	3	-	-
DUA-18-T20	14.63	14.94	81	1	11	4	-	-	3	-	-
DUA-18-T21	15.54	15.85	78	2	13	3	-	tr	4	-	-
DUA-18-T22A/B	16.15	16.61	87	1	9	tr	-	-	3	-	-
DUA-18-T23A/B	16.76	17.37	83	1	12	1	tr	-	3	-	-
DUA-18-T24	17.53	18.14	78	1	19	-	tr	-	2	-	-
DUA-18-T25	18.44	18.75	82	1	14	tr	-	tr	3	-	-
DUA-18-T26	19.35	19.66	60	1	36	-	-	-	3	-	-
DUA-18-T27	20.12	20.42	76	1	17	-	-	-	6	-	-
DUA-18-T28	20.57	21.18	73	tr	19	1	-	tr	4	3	-
DUA-18-T29	21.34	21.95	87	tr	8	-	1	-	2	-	2
DUA-18-T30	22.25	22.56	82	1	9	1	5	-	1	tr	1
DUA-18-T31	22.86	23.47	83	tr	14	tr	-	tr	3	-	-
DUA-18-T32A	24.38	24.54	1	-	-	9	-	1	-	-	-
DUA-18-T32B	24.54	24.84	tr	-	-	7	-	-	-	-	-
DUA-18-T32C	24.84	25.05	tr	39	-	3	-	-	-	-	-

cal: calcite; dol: dolomite; bio: biotite; chl: chlorite; smc: smectite; kao: kaolinite; fap: fluorapatite; gyp: gypsum; prv: perovskite; tr: trace; -: not detected.

4.3 Tailings geochemistry

Detailed depth profiles of solids geochemistry in the GSC borehole reflect the temporal evolution of milling and tailings disposal practices over the mine life. They also provide a measure of the amounts and distribution of any potentially hazardous leachable material or unrecovered mineral resource.

Calcites from the Oka complex are known to host highly variable amounts of Sr, Mn, Mg, and Ba (Pouliot, 1970), as well as REE (Hornig-Kjarsgaard, 1998). Depth profiles for CaO and MgO show a distinct antithetical relationship indicating variable Mg substitution in calcite or greater content of ferromagnesian silicate minerals (Figure 16a). The depth profile for Ba indicates variable substitution in calcite or possible barite-rich layers (Figure 16b). The depth profile for Sr shows a spike in a calcite-rich layer (Figure 16b). Depth profiles for SiO₂, Al₂O₃, Na₂O, and K₂O clearly reveal the sharp geochemical contrast between calcitic tailings and the underlying soil layer composed mainly of quartz, plagioclase and K-feldspar possibly derived from the surrounding fenitized gneiss (Figure 17).

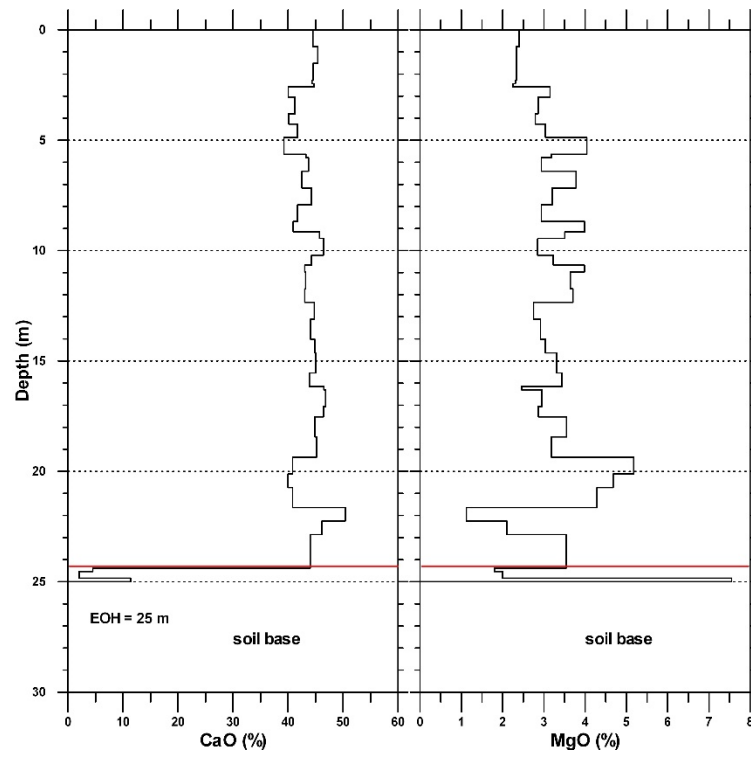
Depth profiles of Fe₂O₃ (total Fe), MnO, TiO₂, and V (Figure 18) highlight the vertical distribution of magnetite-rich horizons notably from 20 to 22 m below ground surface and at the top of the soil layer (Table 6). Titanium, Mn, and V are common substitutions in magnetite. Iron contained in biotite likely accounts for the more uniform background Fe₂O₃ concentrations in the tailings.

Fluorapatites from the Oka complex are enriched in REE (Hornig-Kjarsgaard, 1998), and depth profiles of P₂O₅, F, and total REE reveal the presence of several apatite-rich layers, at 9 m, 20-22 m, and at 23-24 m (Figure 19). While F concentrations generally follow those of P₂O₅, there are discrete F-rich horizons that do not fit this pattern. Total REE concentrations, on the other hand, follow P₂O₅ values more closely and sometimes exceed 5000 ppm.

Depth profiles of Nb, Zr, Th, and U characterize variations in pyrochlore composition and efficiency of the milling process (Figure 20). Uranium, Th, and Zr are common substitutions in the pyrochlore structure. However, the trace element composition of SLC pyrochlores varies tremendously (Gold et al., 1967; Petruk and Owens, 1975). Coincident spikes in Nb and one or more of these elements indicate layers containing significant unrecovered pyrochlore. The most important of these layers, where Nb reaches ore-grade concentrations, is between 20 and 22 m below ground surface. Uranium and Th concentrations in the layer reach 80 and 175 ppm, respectively. Higher in the tailings sequence, Nb concentrations are much lower and more uniform. This suggests a stabilization of the ore processing method, with improved pyrochlore recovery. Background Nb concentrations reflect unrecovered pyrochlore as well as Nb hosted in niocalite or perovskite (latrappite).

Total S and Zn exhibit fairly uniform depth profiles except for pronounced coincident spikes in the sandy basal layer suggesting the presence of sphalerite (Figure 21a) in addition to the pyrite noted above. Depth profiles of Pb and Mo also suggest the presence of sulfide-rich horizons (Figure 21b). From these and preceding geochemical depth profiles, it appears that coarse and/or dense tailings containing magnetite, apatite, sulphides, and unrecovered pyrochlore were deposited early in the mine life as the milling process was being adjusted.

a)



b)

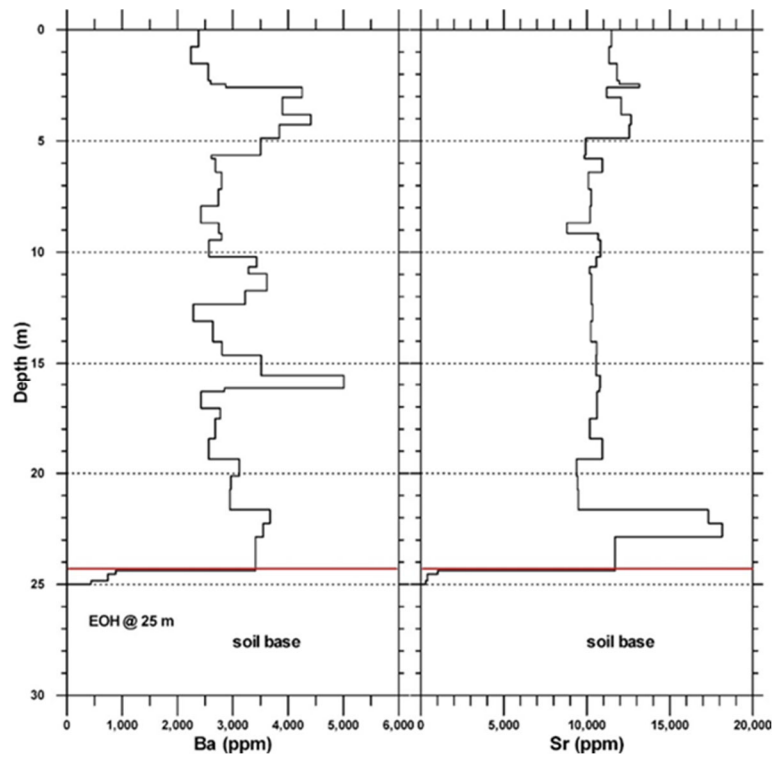
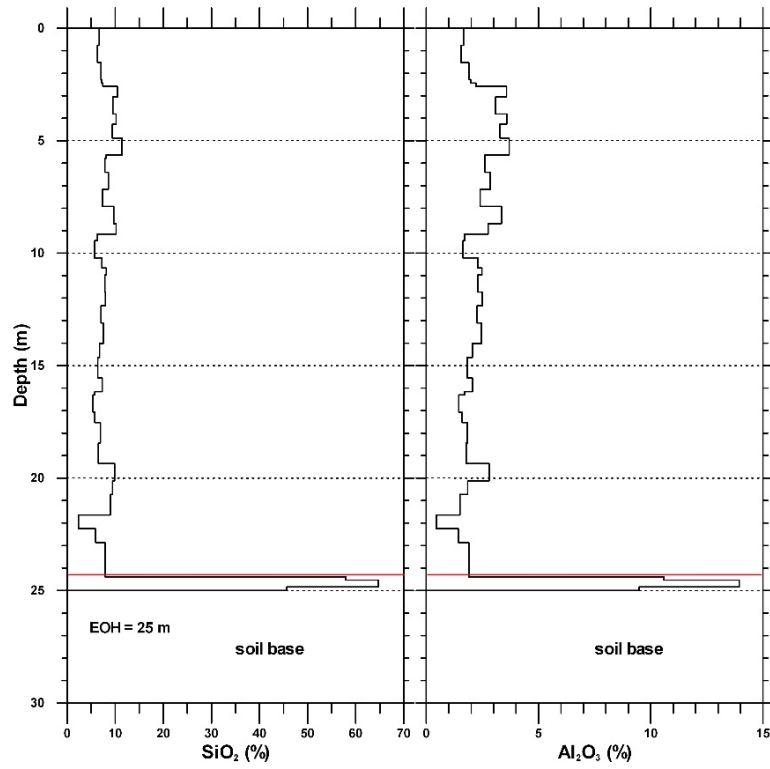


Figure 16: Downhole profiles of tailings chemistry: a) CaO and MgO; b) Ba and Sr.

a)



b)

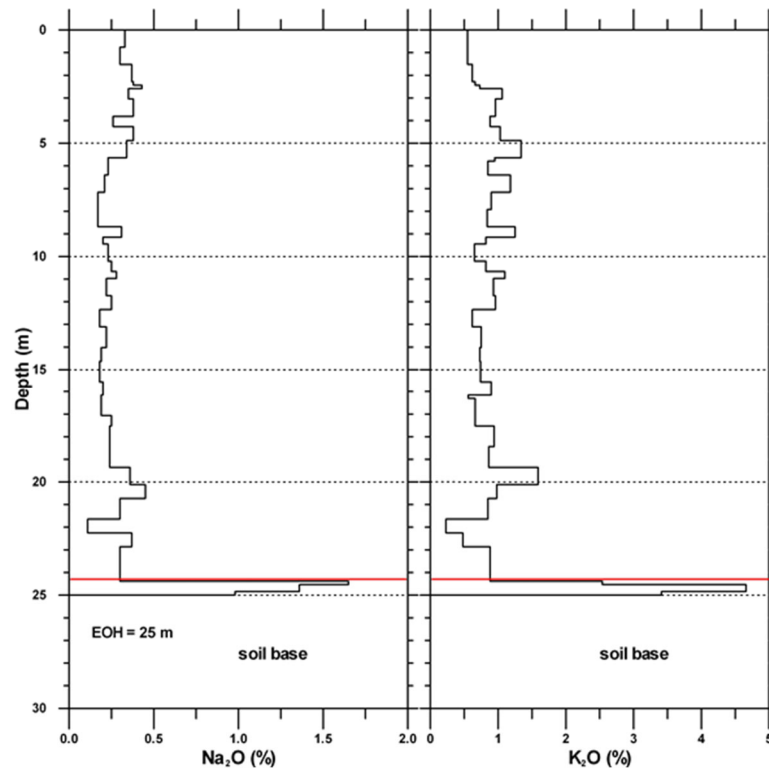
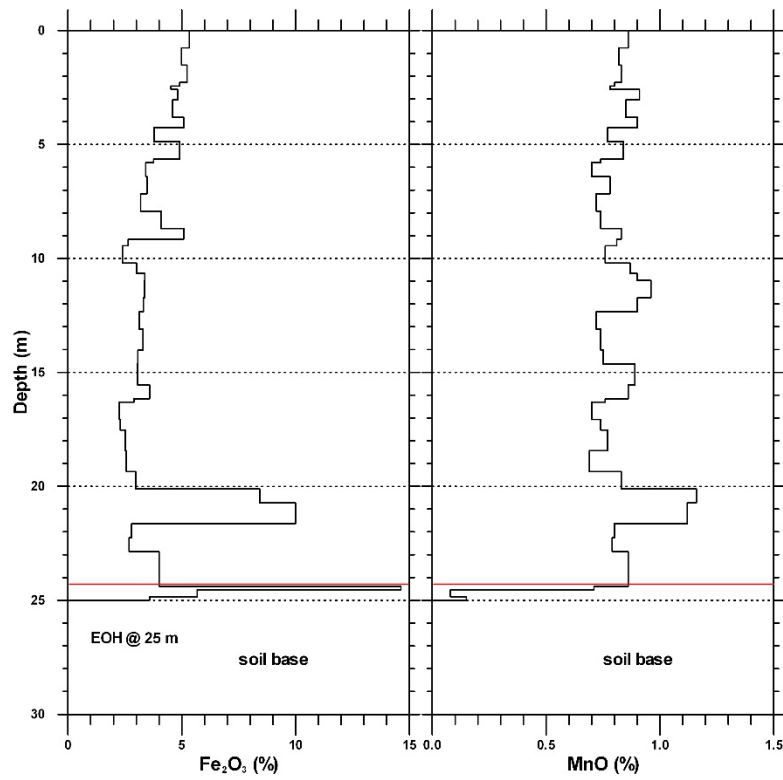


Figure 17: Downhole profiles of tailings chemistry: a) SiO_2 and Al_2O_3 ; b) Na_2O and K_2O .

a)



b)

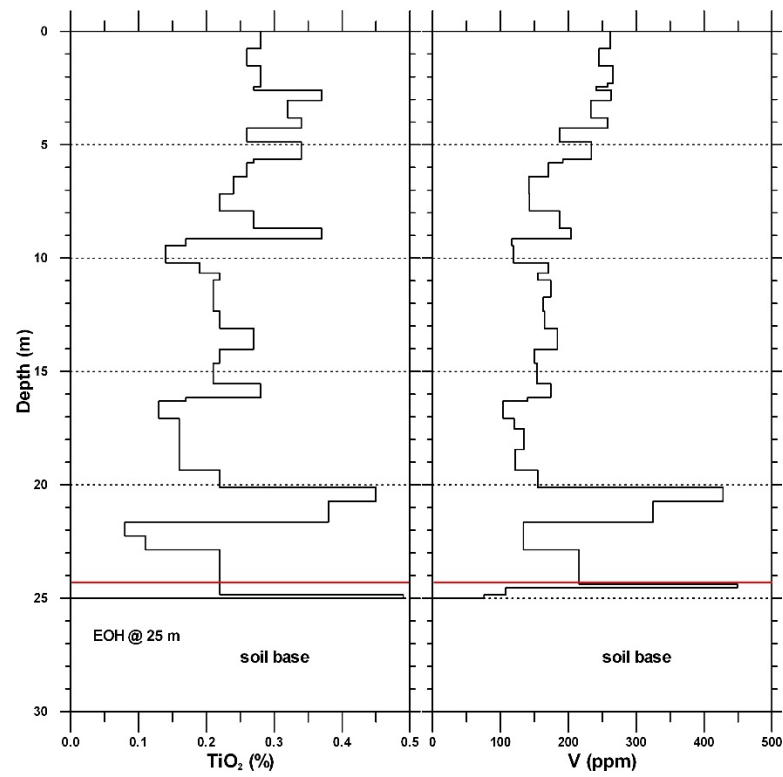
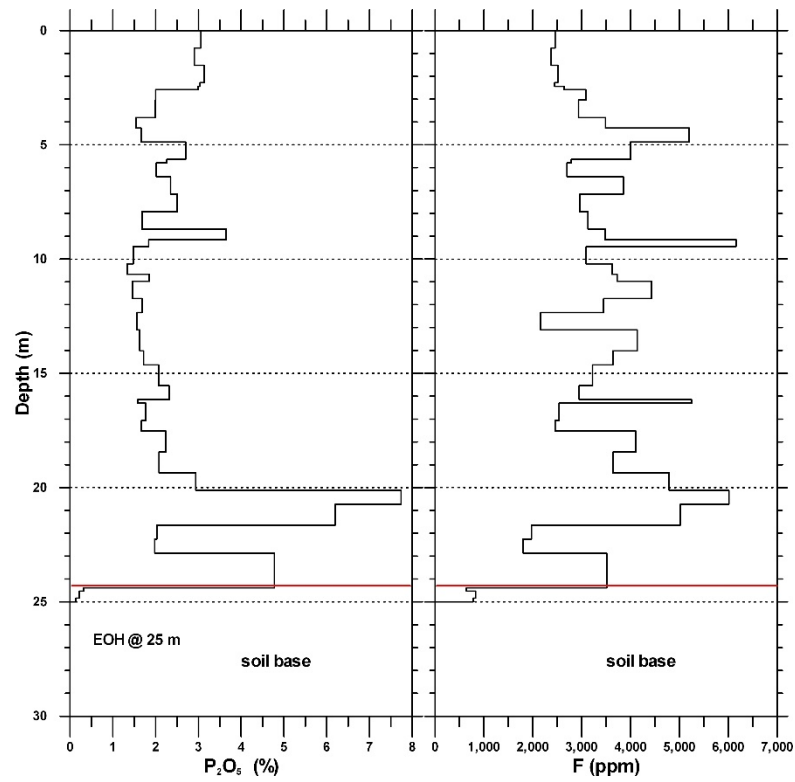


Figure 18: Downhole profiles of tailings chemistry: a) Fe_2O_3 and MnO ; b) TiO_2 and V .

a)



b)

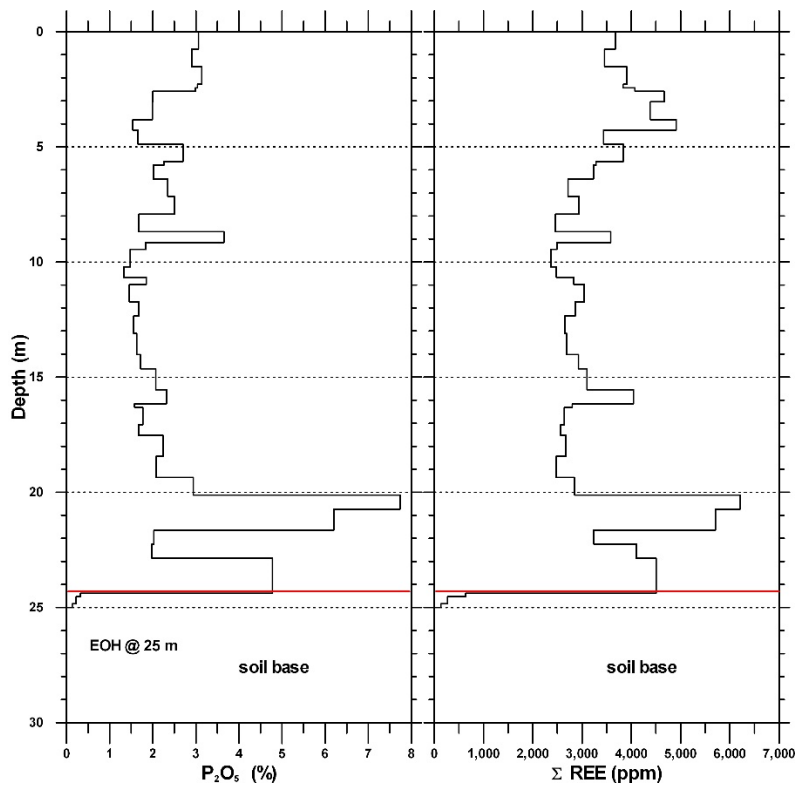
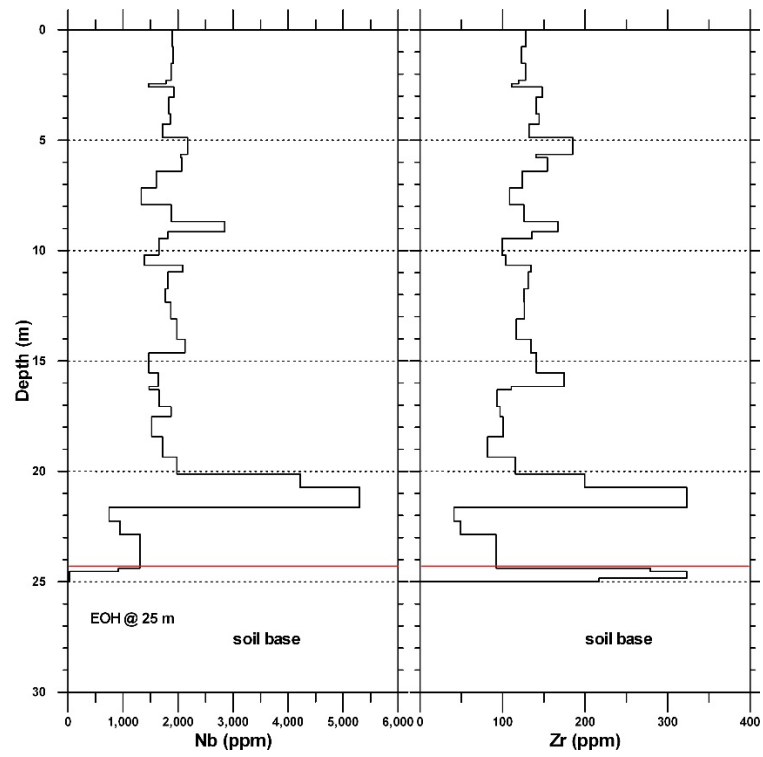


Figure 19: Downhole profiles of tailings chemistry: a) P_2O_5 and F; b) P_2O_5 and Σ REE.

a)



b)

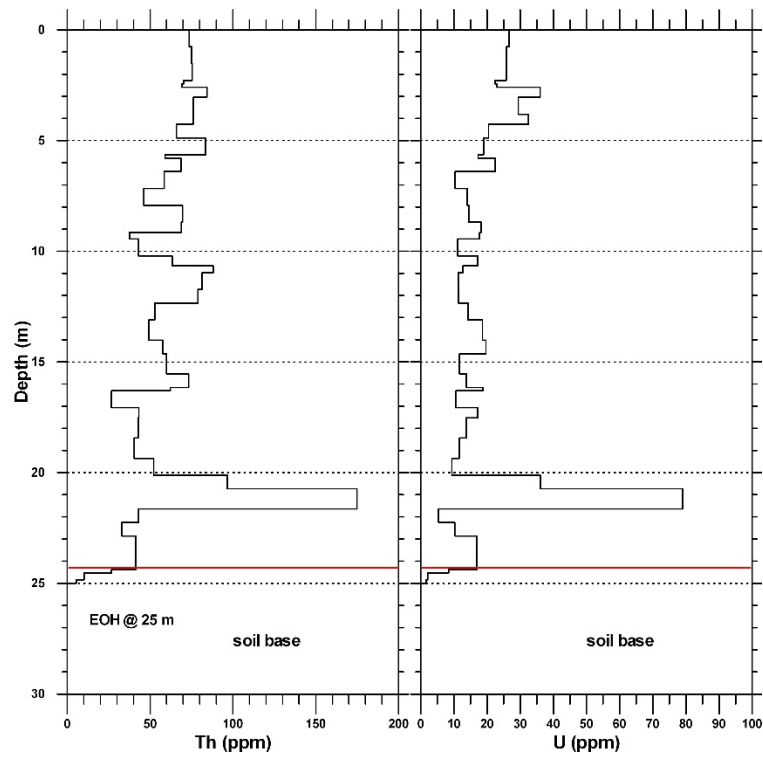
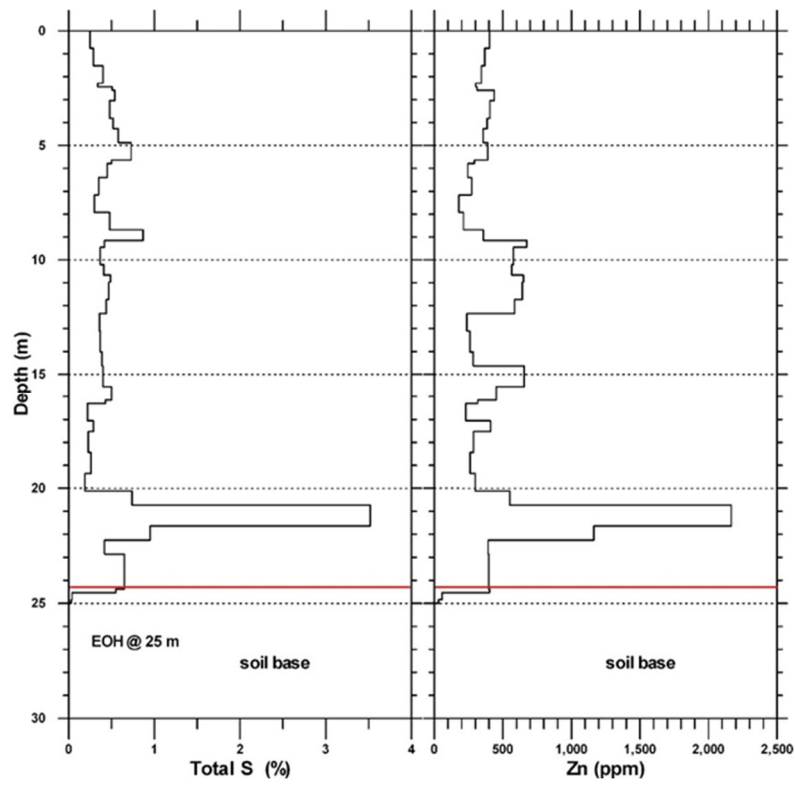


Figure 20: Downhole profiles of tailings chemistry: a) Nb and Zr; b) Th and U.

a)



b)

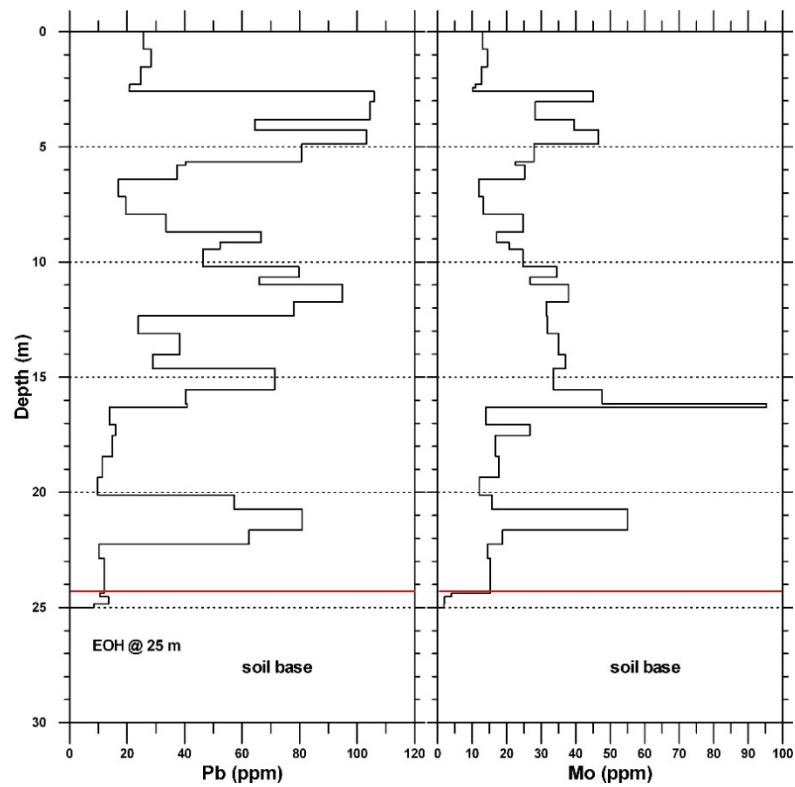


Figure 21: Downhole profiles of tailings chemistry: a) Total S and Zn; b) Pb and Mo.

4.4 Physical hydrogeology

The drive-point piezometers were emplaced at pre-determined depths as described in section 3.2. As a result, they were screened in tailings of variable texture and permeability (Figure 22): OW-01-A was screened in highly permeable sands; OW-01-B was screened in low-permeability silts; OW-01-C and OW-01-D were screened in moderately permeable sandy silts; OW-01-E and OW-01 were screened in very low-permeability clayey silts; OW-01-G was screened in highly permeable sands or silty sands; OW-01-F (not shown) was screened in low-permeability silts directly beneath the decant pond.

Head measurements made in the piezometers (Figure 23) show a very strong vertical gradient (of approximately 0.9) from the decant pond, through the fine tailings, to the top of the basal sandy layer. Within this higher permeability layer, there is little further head decline. These results indicate downward flow from the decant pond to the permeable layer whereas flow in the permeable layer is essentially horizontal.

The pluviograph and monitoring well hydrograph for the summer and fall of 2016 (Figure 24a) show that heads in OW-01 (screened in clayey silt) do not respond to discrete rainfall events. Rather, they exhibit a slow seasonal recession. Water levels during the same period in 2017 show the same general behaviour (Figure 24b). Unfortunately, the limited monitoring record is not sufficient to identify the main recharge period for the tailings impoundment although it is likely to be at the time of spring snowmelt.

Effluent from the tailings impoundment discharges at the toes of the northwest and southeast embankments. The southeast seepage flows through a weir to Saint-Pierre creek, which represents the lower reaches of a pre-existing stream that was buried beneath the tailings. Discharge rates measured at the weir are summarized in Table 8. These rates do not show much seasonal variation and average approximately 370 m³/day. The northwest seepage is captured by a ditch that eventually drains to Ruisseau Rousse. Its flow rate was not measured but is of the same magnitude as that observed on the southeast embankment.

Table 8: Flow measurements of discharge from the toe of the southeast tailings embankment.

Date	Q (L/s)	Q (m ³ /d)
23-06-2016	5.168	446.5
25-10-2016	4.813	415.8
01-11-2017	3.738	322.9
10-05-2018	4.698	405.9
11-06-2018	4.255	367.6
17-09-2018	3.639	314.4
05-11-2018	3.447	297.8
Average	4.251	367.3

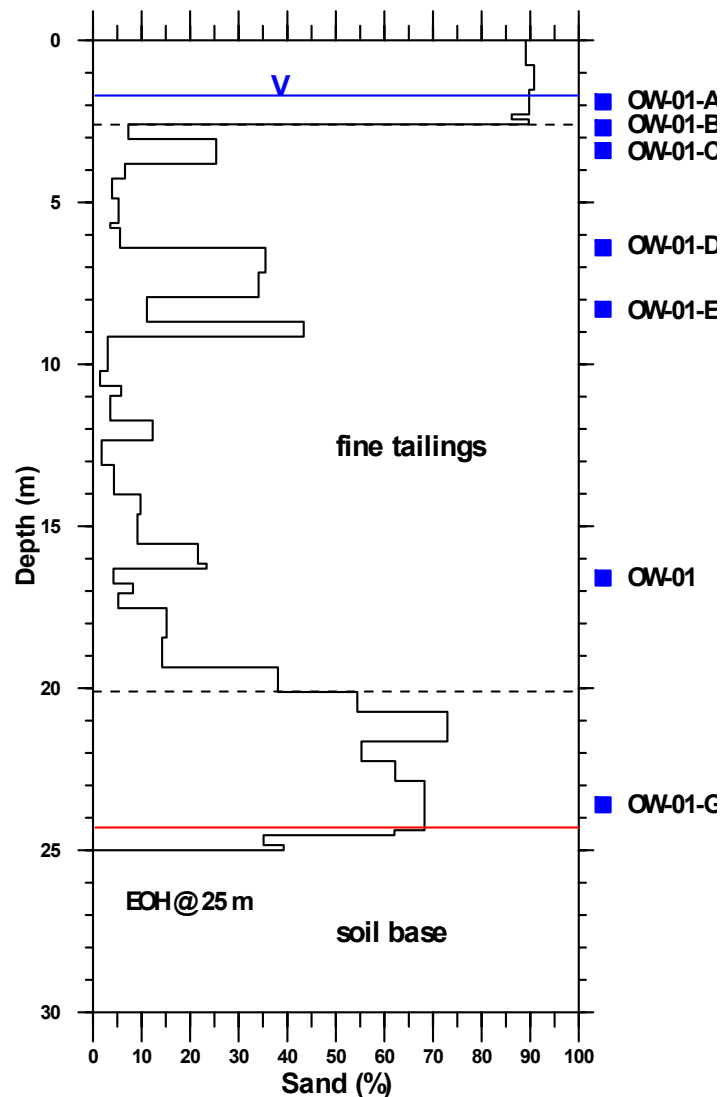


Figure 22: Vertical position of the piezometer screens with respect to sandy layers observed in the GSC stratigraphic borehole (completed as piezometer OW-01-G). The screen of piezometer OW-01-F is not shown because it is positioned in silty tailings beneath the decant pond and offset from the vertical plane containing the piezometer nest (Figure 5). The approximate position of the water table is marked by a “V”.

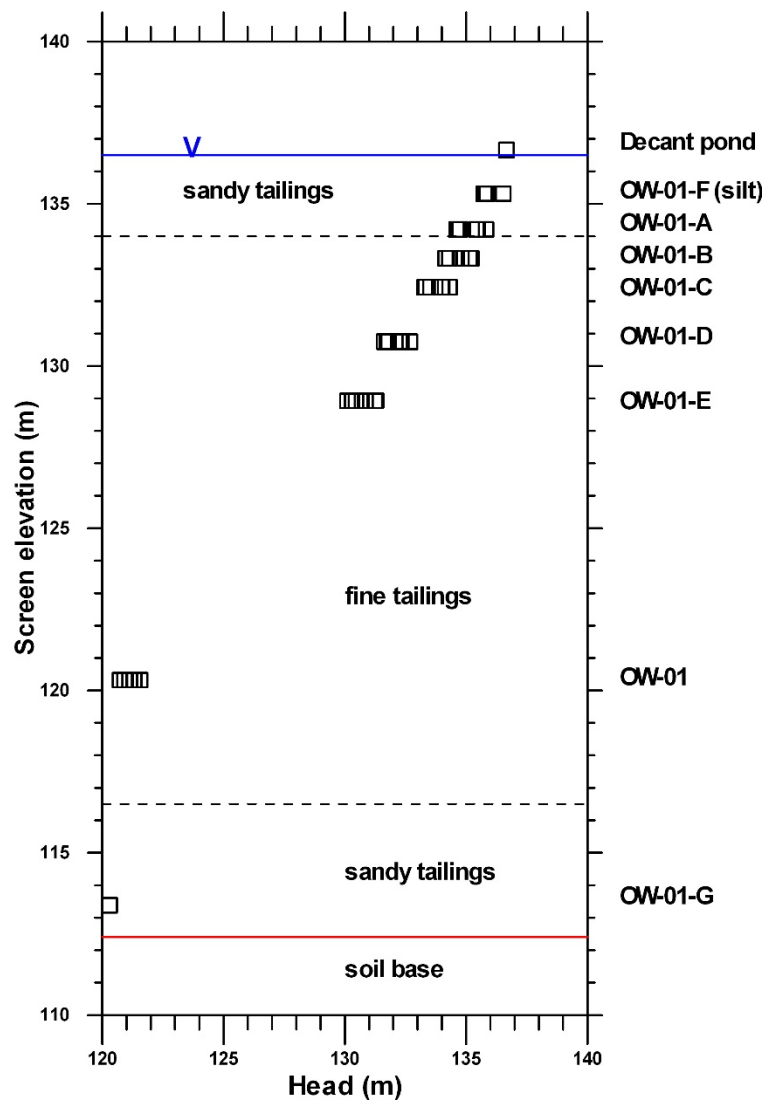
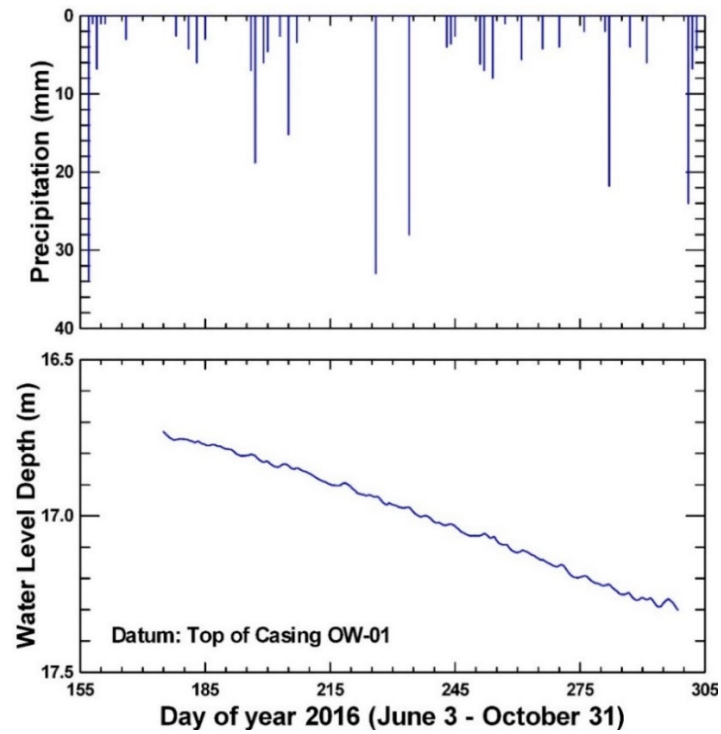


Figure 23: Head measurements from the GSC piezometer nest near the decant pond in the SLC tailings disposal area. Locations of the piezometers are shown in Figure 5. The “V” marks the approximate water level in the decant pond, which varies seasonally.

a)



b)

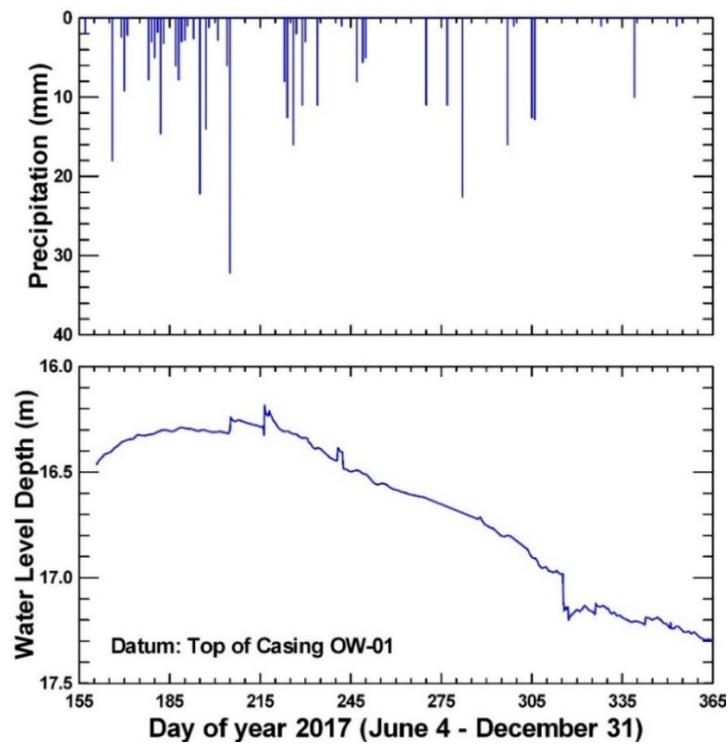


Figure 24: Pluviographs and groundwater level hydrographs measured in OW-01 for a) the summer and fall of 2016; b) the summer and fall of 2017.

4.5 Chemical hydrogeology

4.5.1 Depth profiles of tailings pore-water chemistry

The following figures show depth profiles of tailings pore-water chemistry (major dissolved ions and selected trace elements) observed at the piezometer nest in the central part of the tailings disposal area (Figure 5). Concentrations observed on different sampling dates are plotted against their corresponding piezometer screen elevation. Unless otherwise indicated, concentrations were measured on filtered ($< 0.45 \mu\text{m}$) water samples.

Specific conductance values generally increase from the decant pond down to piezometer OW-01-C (elevation 133 m). They then decrease and rise again towards the basal layer of coarse tailings (Figure 25). The pH of pore-waters decreases initially with depth to below 7.5 and then rises to near 8 (Figure 25). Values observed in piezometer OW-01 (elevation 120 m) are anomalously high. The high pH may be an artefact of groundwater reactions with the steel casing of this well (Marsh and Lloyd, 1980) or may reflect chemical weathering of Ca-Mg silicate minerals (such as diopside or monticellite) in the tailings (Roadcap et al., 2005). A high pH (>12) was observed also in a nearby PVC cased monitoring well screened in tailings (WSP, 2018). The average in-situ pore-water temperature (measured in OW-01) was 9.8°C .

Calcium concentrations increase with depth down to piezometer OW-01-B (elevation 133.4 m) and thereafter drop almost to zero before rising slightly in the layer of sandy tailings (Figure 26a). Concentrations of Mg follow a similar pattern although they peak at greater depth, at piezometer OW-01-D (elevation 131 m). Strontium, which reaches a concentration of 28.2 mg/L , has a depth profile similar to that of Ca whereas Ba generally decreases steadily throughout the fine tailings (Figure 26b). Depth profiles of Na and K (Figure 27a) are quite different, however. Concentrations of these cations increase steadily down to the base of the fine tailings before dropping in the permeable sandy layer. Dissolved SiO_2 peaks at piezometer OW-01-D (elevation 131 m), decreases through the fine tailings, before peaking again at 51.6 mg/L in the basal permeable layer (Figure 27b). Dissolved organic carbon (DOC) generally increases down the profile and exceeds 200 mg/L in a duplicate sample taken in piezometer OW-01-G (Figure 27b). Groundwater in this piezometer initially exhibited an unusual greenish colour, which went away after prolonged purging. The sample with high DOC was taken in the greenish water and the other was taken after purging.

Pore-water alkalinities within the upper portion of the tailings sequence appear to rise initially and then stabilize although there is significant scatter among measurements at different sampling dates (Figure 28a). Values in the permeable sandy layer at the base of the sequence are lower. Sulfate concentrations increase sharply with depth initially before dropping gradually towards the base of the fine tailings (Figure 28a). Except for a single outlier, Cl concentrations are low and do not show any depth trend (Figure 28b). Concentrations of F, on the other hand, exhibit a steady increase with depth and exceed 20 mg/L near the base of the fine tailings (Figure 28b). Values then drop somewhat in the sandy tailings at the base of the impoundment. Phosphate concentrations are at or below the analytical detection limit (0.02 mg/L) and are not shown.

Concentrations of redox sensitive metals Fe and Mn show significant scatter at any given piezometer depth (Figure 29a). Nonetheless, a general trend is apparent. Concentrations are highest in the shallow subsurface and decrease downward through the fine tailings sequence. Values then recover slightly in the basal sandy tailings. Concentrations of Zn rise initially in the shallow tailings before dropping sharply to very low levels (Figure 29b). Concentrations of Mo appear to rise with depth initially, then decrease before rising again near the base of the fine tailings.

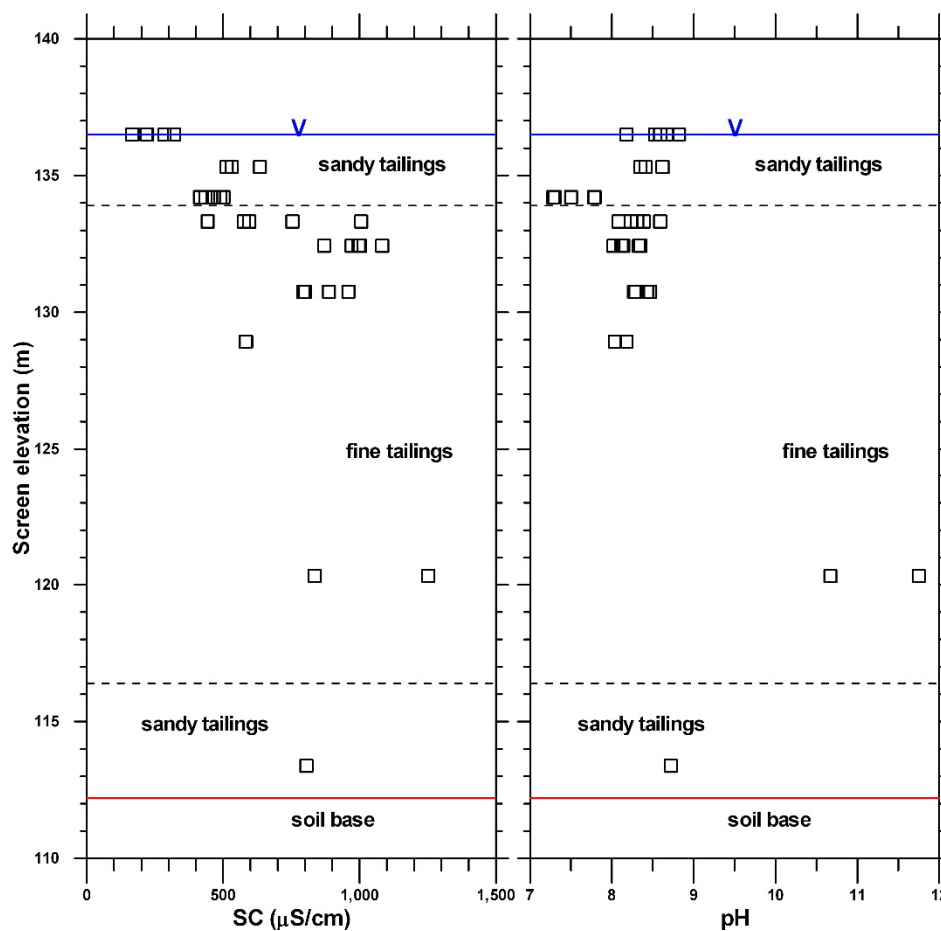
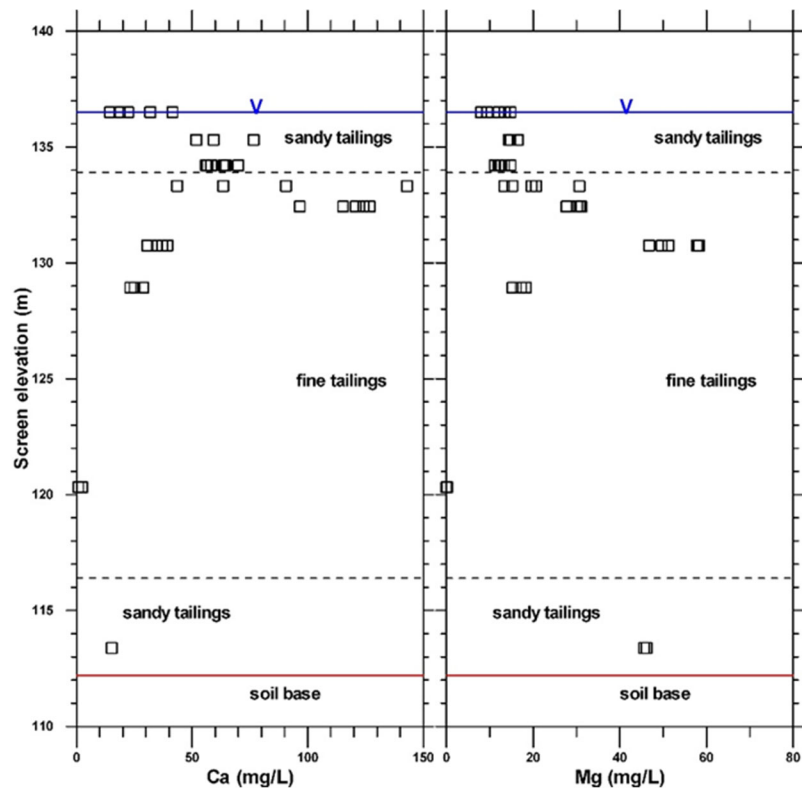


Figure 25: Depth profiles of pore-water chemistry: Specific conductance (SC) and pH

Trace element concentrations (Nb and U) show considerable scatter but are quite low and do not follow any clear depth trend (Figure 30a). Dissolved Ce concentrations do not exhibit any clear depth trend either (Figure 30b). On the other hand, total concentrations of Ce (Ce-T), measured in unfiltered samples, generally increase with depth and sometimes exceed 1000 $\mu\text{g/L}$ (Figure 30b). This suggests that there must be a significant soluble source of Ce and other REE (likely calcite) even if most of the Ce is subsequently sorbed on colloids.

a)



b)

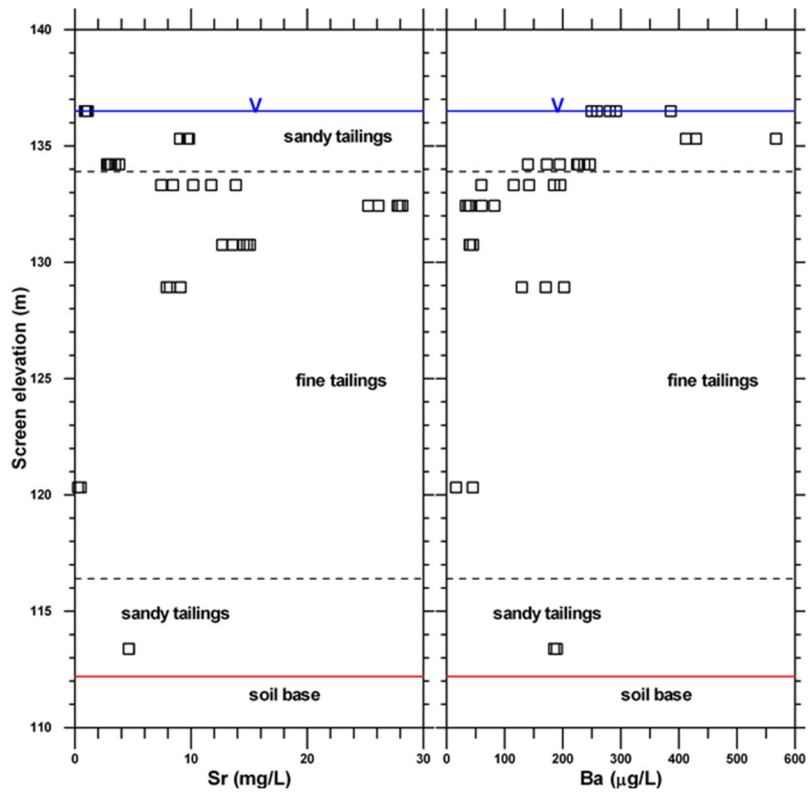
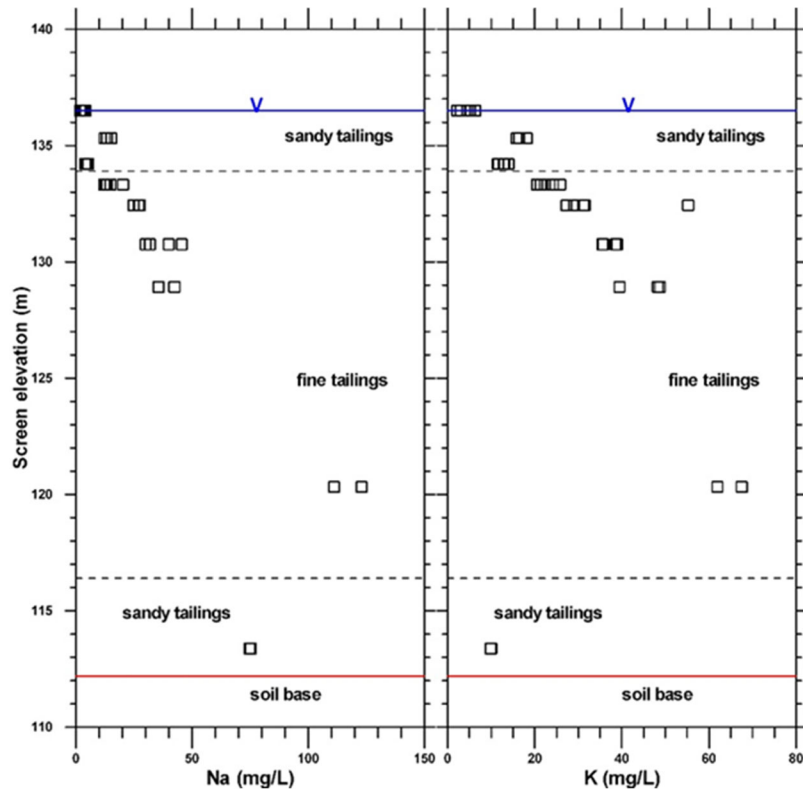


Figure 26: Depth profiles of pore-water chemistry: a) Ca and Mg; b) Sr and Ba.

a)



b)

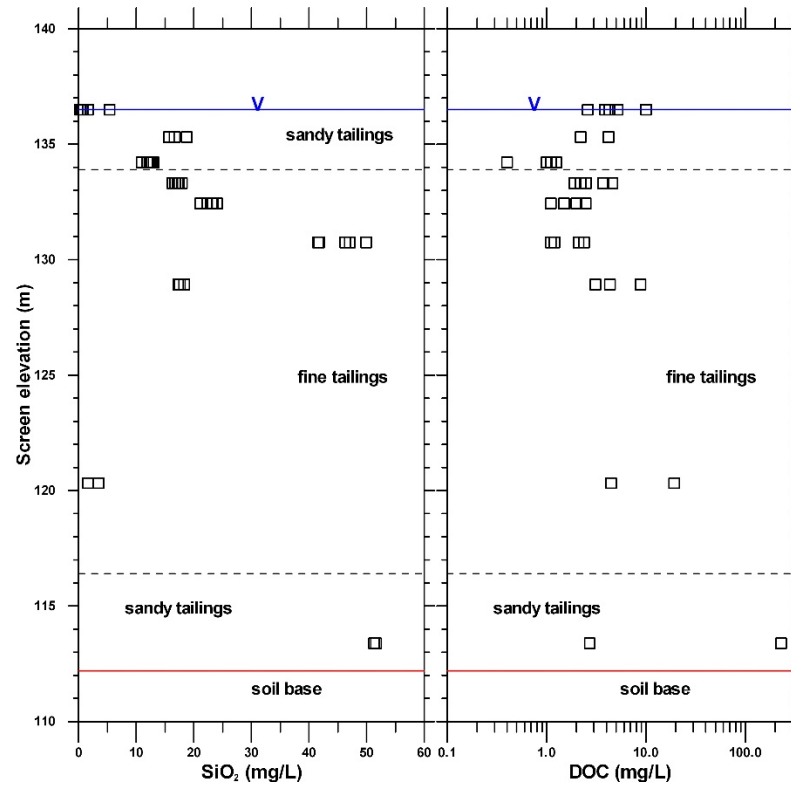
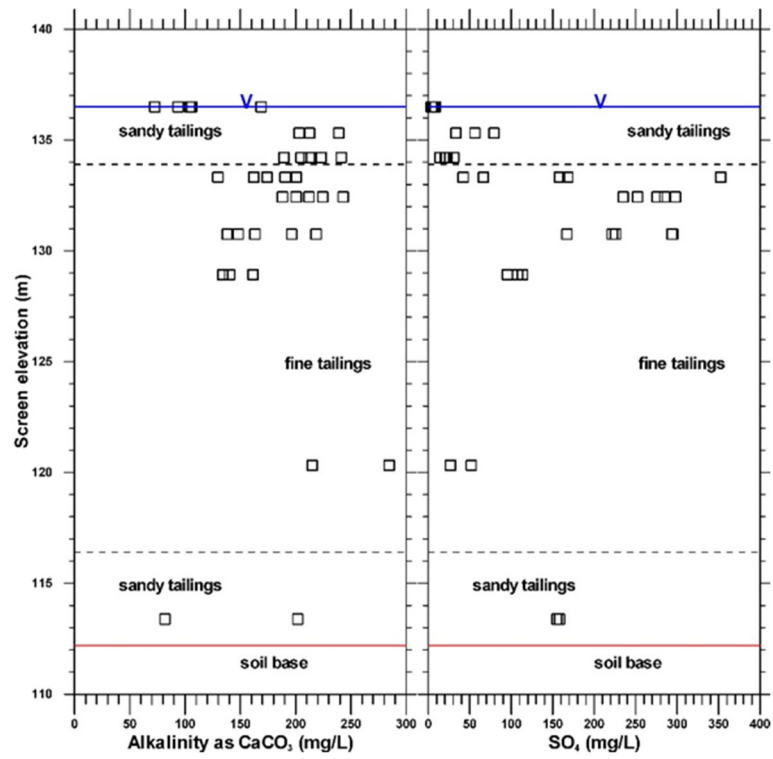


Figure 27: Depth profiles of pore-water chemistry: a) Na and K; b) SiO₂ and DOC.

a)



b)

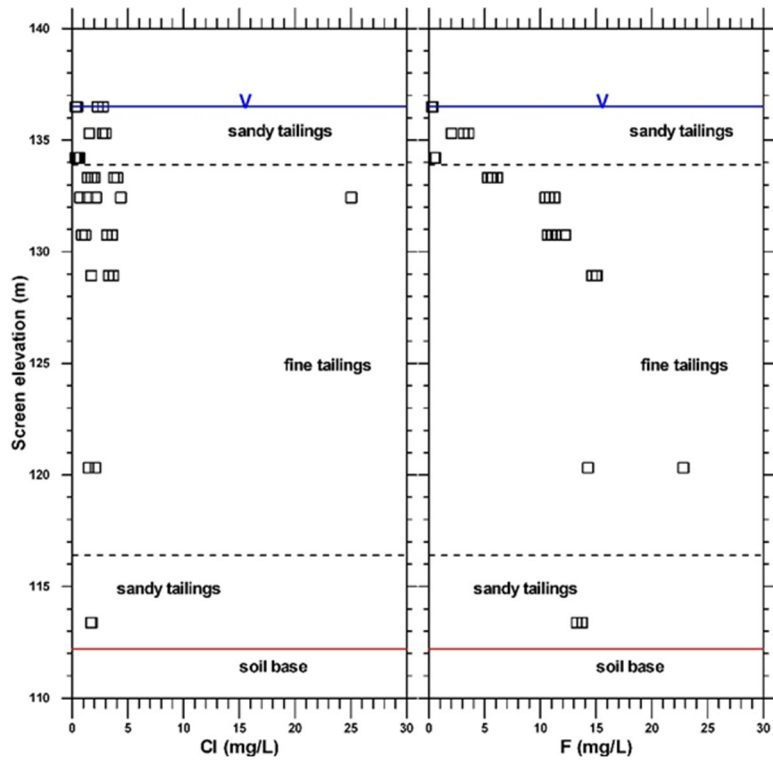
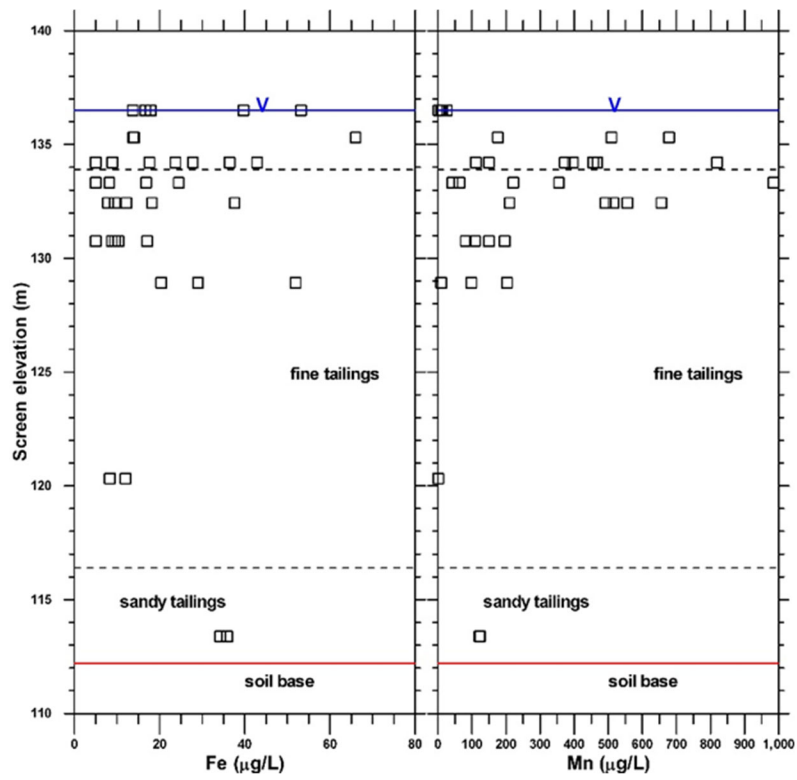


Figure 28: Depth profiles of pore-water chemistry: a) Alkalinity and SO_4 ; b) Cl and F.

a)



b)

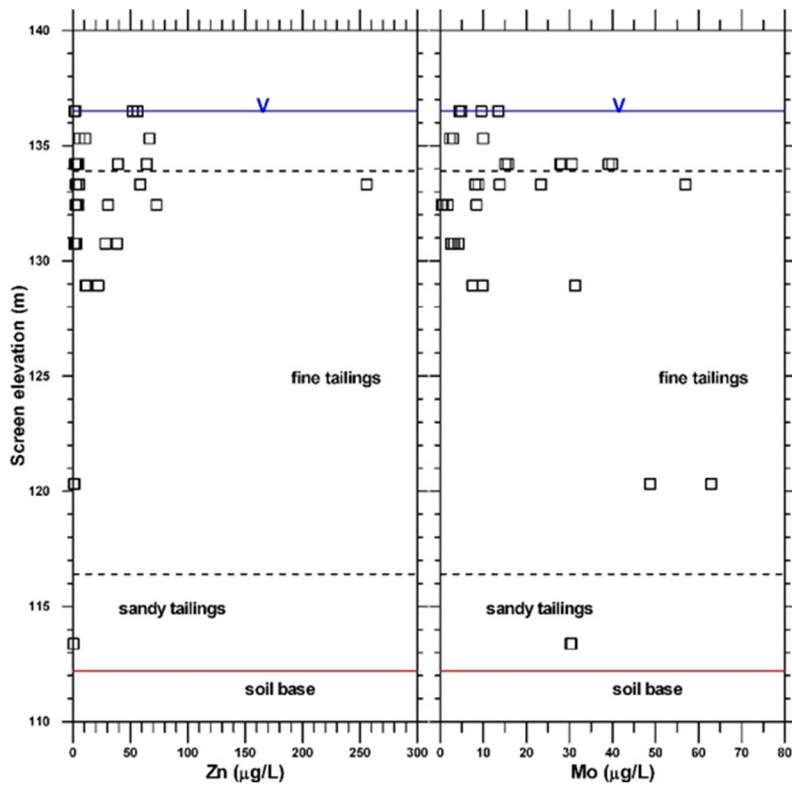
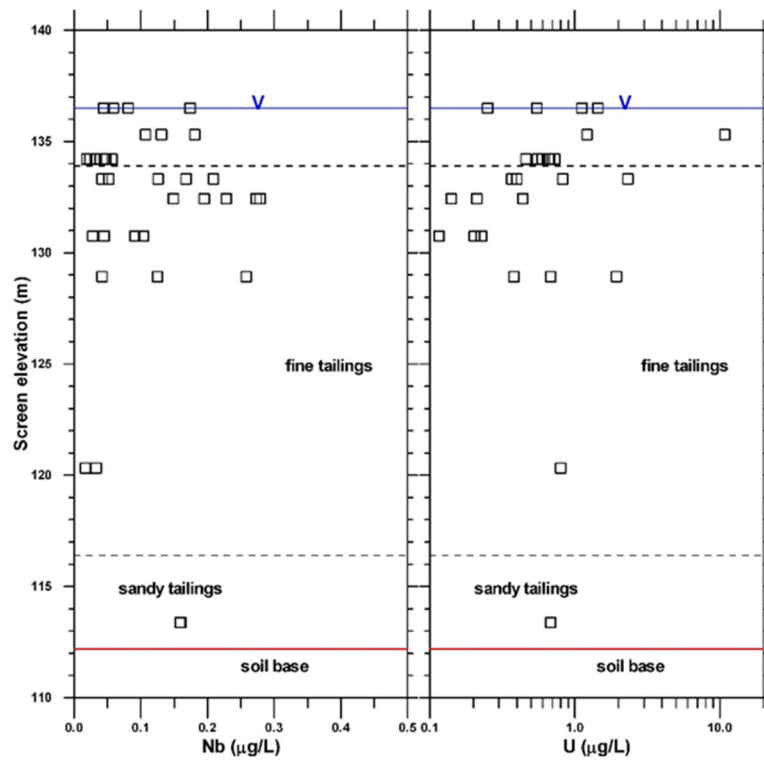


Figure 29: Depth profiles of pore-water chemistry: a) Fe and Mn; b) Zn and Mo.

a)



b)

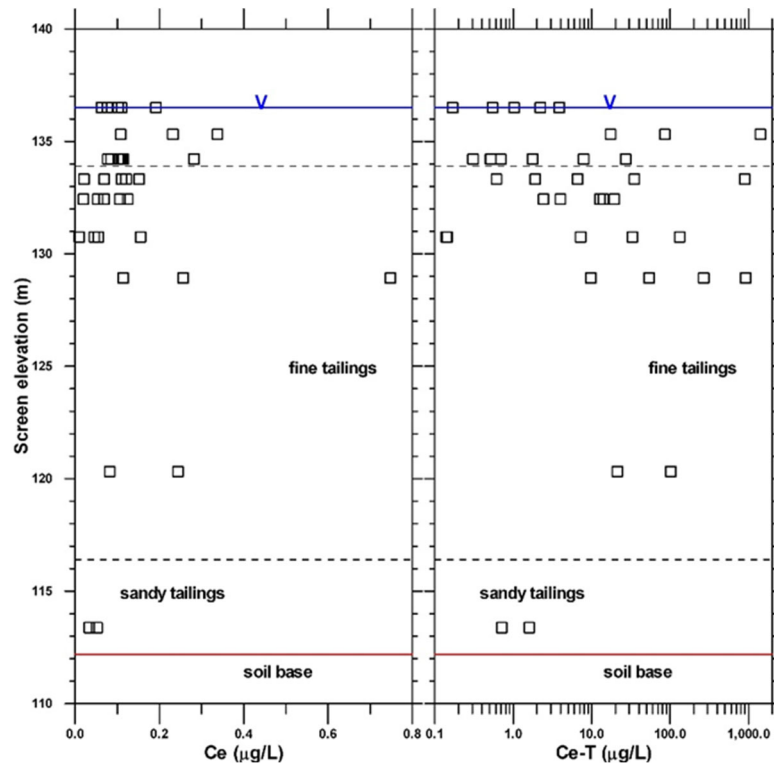


Figure 30: Depth profiles of pore-water chemistry: a) Nb and U; b) Ce and Ce-T.

4.5.2 Tailings effluent chemistry

The following figures show the temporal variations in effluent chemistry (major dissolved ions and selected trace elements) observed at the toes of the southeast (SE) and northwest (NW) embankments of the tailings disposal area. Unless otherwise indicated, concentrations were measured on filtered ($< 0.45 \mu\text{m}$) water samples.

The pH and specific conductance (SC) of seepage from the SE and NW embankments are similar, with average pH and SC values on the order of 7.8 and 1400 $\mu\text{S}/\text{cm}$, respectively (Figure 31). Effluent temperature at the point of emergence from the SE embankment varied between 9.2 and 9.8 $^{\circ}\text{C}$. At the sampling point below the NW embankment, water temperatures varied between 5.6 and 11.9 $^{\circ}\text{C}$.

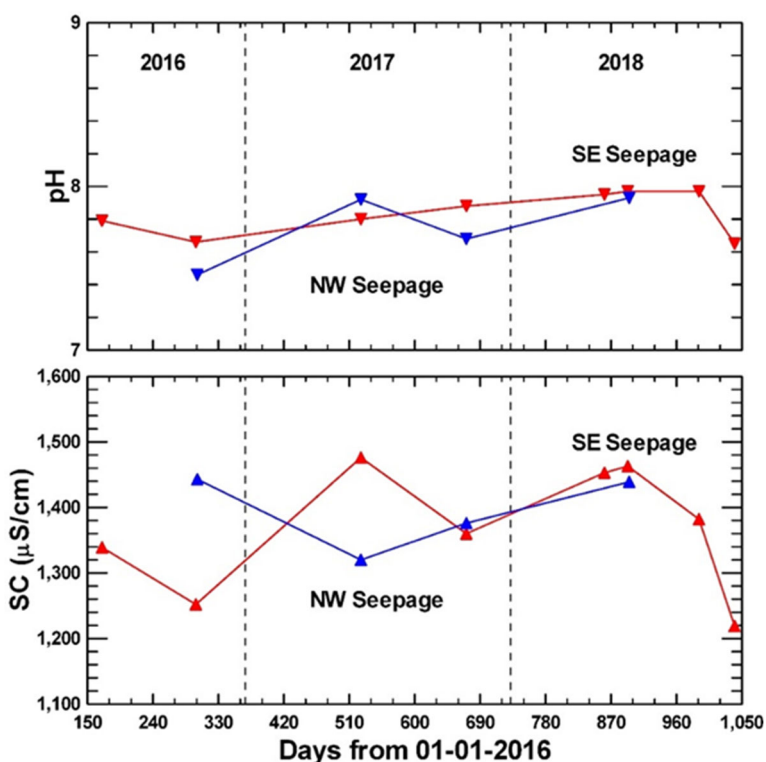
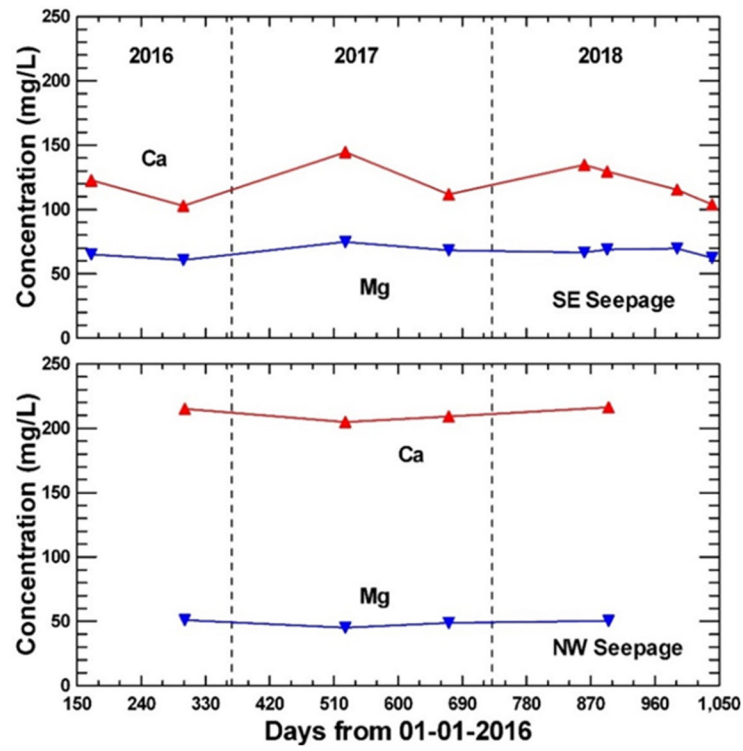


Figure 31: Temporal variations in effluent chemistry from the SE (red) and NW (blue) embankments: pH and specific conductance (SC).

Seepage from the NW embankment contains significantly higher concentrations of Ca than that from the SE embankment whereas concentrations of Mg are similar in both seepages (Figure 32a). Concentrations of Na are high in seepage from the SE embankment and much lower in the NW seepage. Concentrations of K are on the same order in both embankment seepages. (Figure 32b). Concentrations of SiO_2 exceed 30 mg/L in the SE seepage but are less than 10 mg/L in the NW seepage (Figure 33). Strontium concentrations are of the same order in both seepages (Figure 33). For all major dissolved cations and SiO_2 , concentrations show little temporal variation.

a)



b)

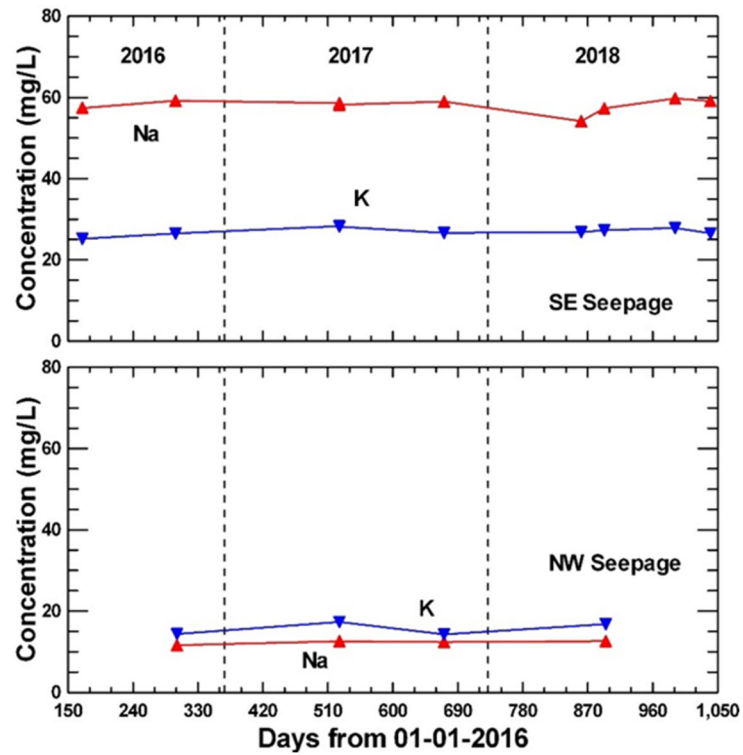


Figure 32: Temporal variations in effluent chemistry: a) Ca and Mg; b) Na and K.

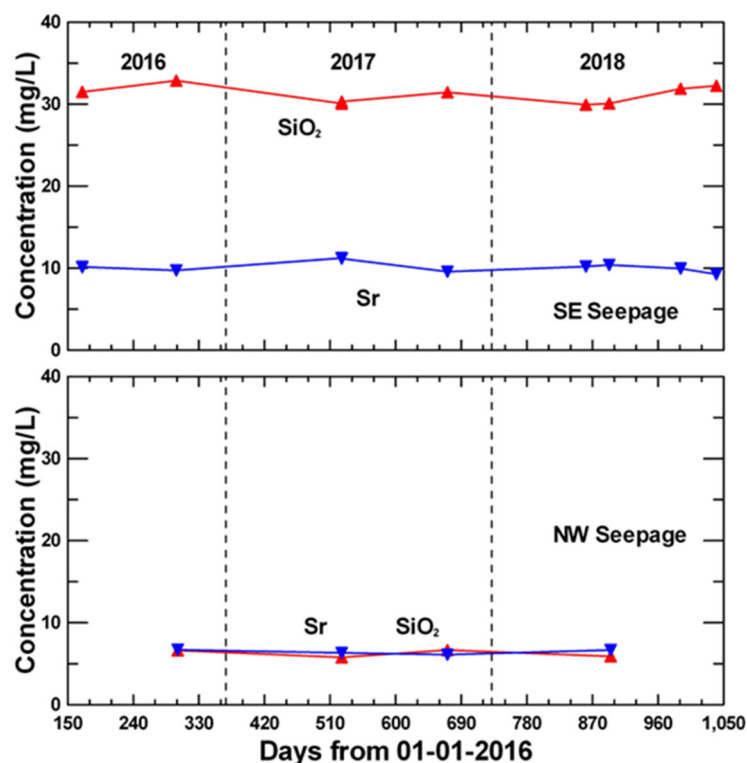
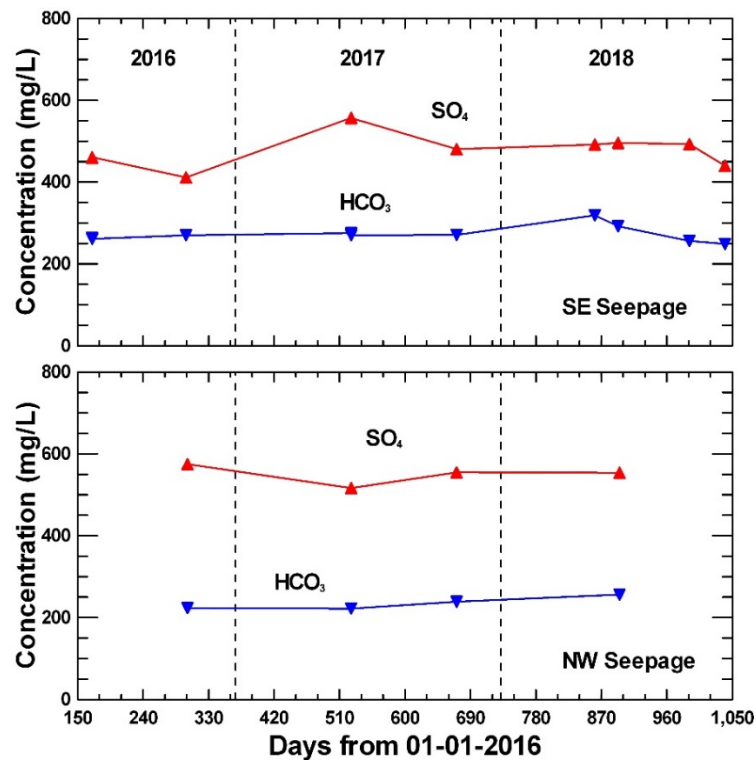


Figure 33: Temporal variations in effluent chemistry: SiO_2 and Sr (a).

Concentrations of major dissolved anions (SO_4 , HCO_3 , and Cl) are similar in both the SE and NW embankment seepages (Figure 34). Sulfate concentrations, in particular, are elevated and range between 400 and 600 mg/L. Concentrations of F in seepage from the SE embankment are very high, averaging more than 8 mg/L (Figure 34b). This value is much greater than 0.12 mg/L, the CCME (2002) inorganic fluoride guideline for the protection of freshwater aquatic life. However, in NW seepage, F is less than 1 mg/L. Phosphate concentrations were always at or below the detection limit (0.02 mg/L) and are not shown. As with the major cations, concentrations of major and minor anions show little temporal variation.

Concentrations of Mn and Fe are much higher in seepage from the SE embankment (Figure 35a). This likely reflects more reducing conditions noted at the point of emergence where samples were taken. Samples of seepage from the NW embankment were taken several metres downstream of the point of emergence. Concentrations of Ba are higher in SE seepage whereas concentrations of Mo are higher in NW seepage (Figure 35b). Concentrations of U are slightly higher in seepage from the NW embankments (Figure 36a). Concentrations of Nb are higher in the SE seepage whereas in the NW seepage values are below the detection limit (Figure 36b). Concentrations of Ce are similar in both seepages. Concentrations of U are about two order of magnitude greater than those of Nb. This suggest either a U source other than pyrochlore or much greater mobility of U than Nb. Overall, trace metal concentrations in tailings effluent are very low, reflecting the low solubility of the main mineral hosts including pyrochlore and fluorapatite.

a)



b)

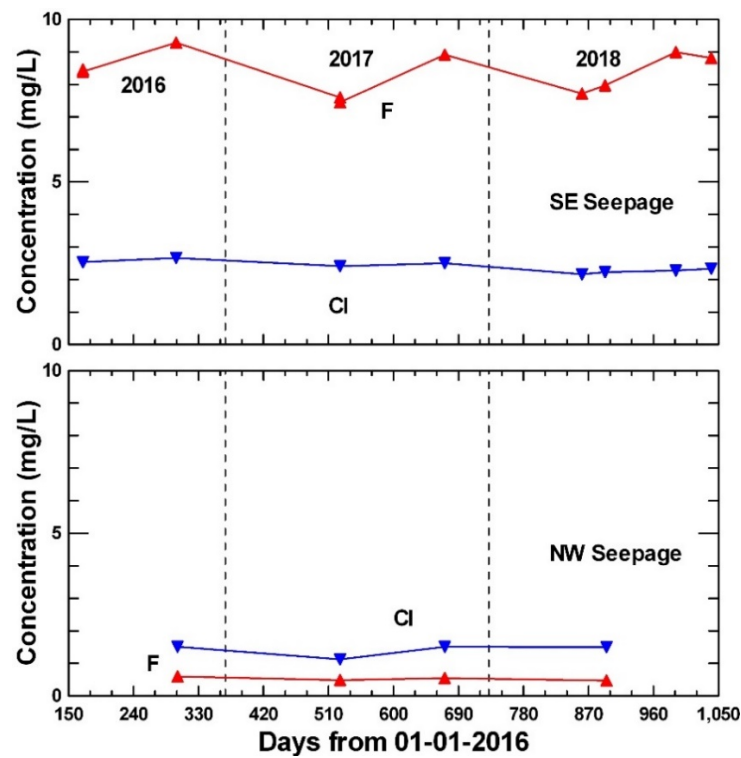
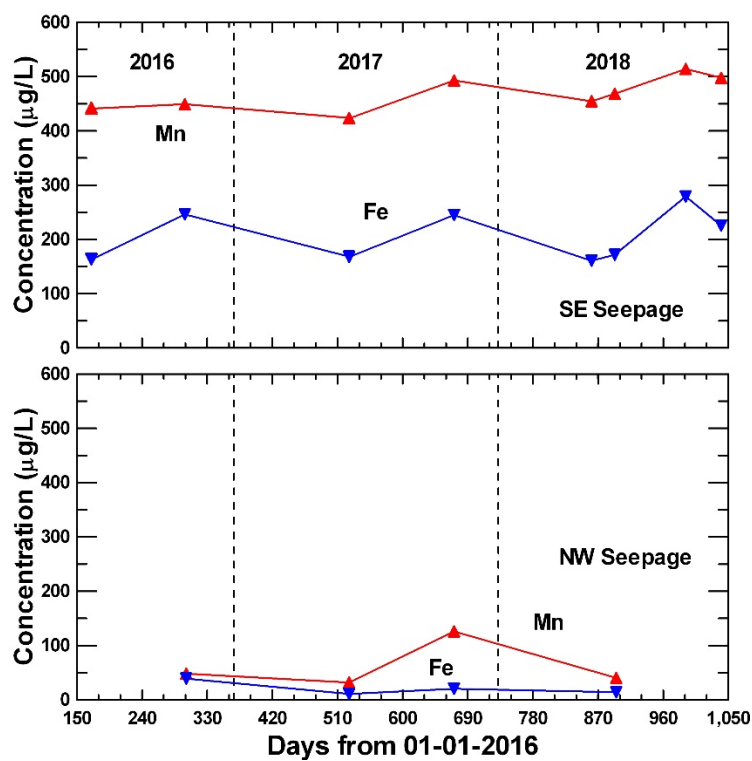


Figure 34: Temporal variations in effluent chemistry: a) SO_4 and HCO_3 ; b) Cl and F .

a)



b)

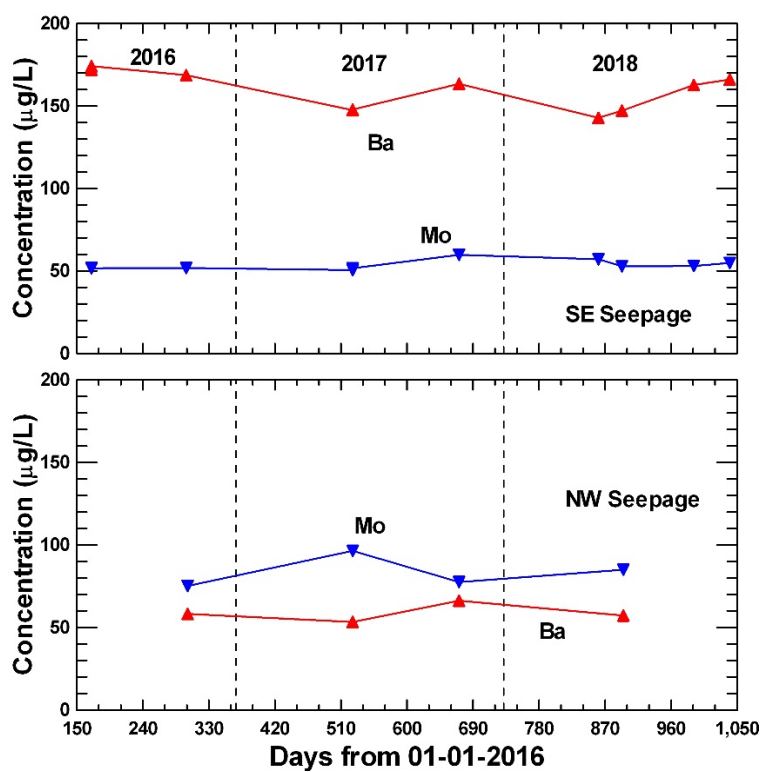
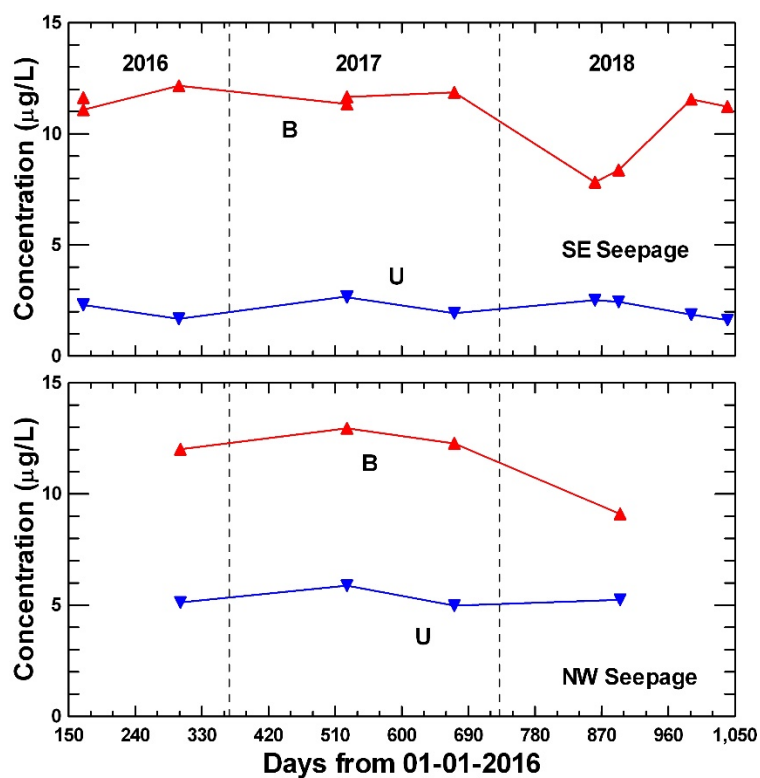


Figure 35: Temporal variations in effluent chemistry: a) Mn and Fe; b) Ba and Mo.

a)



b)

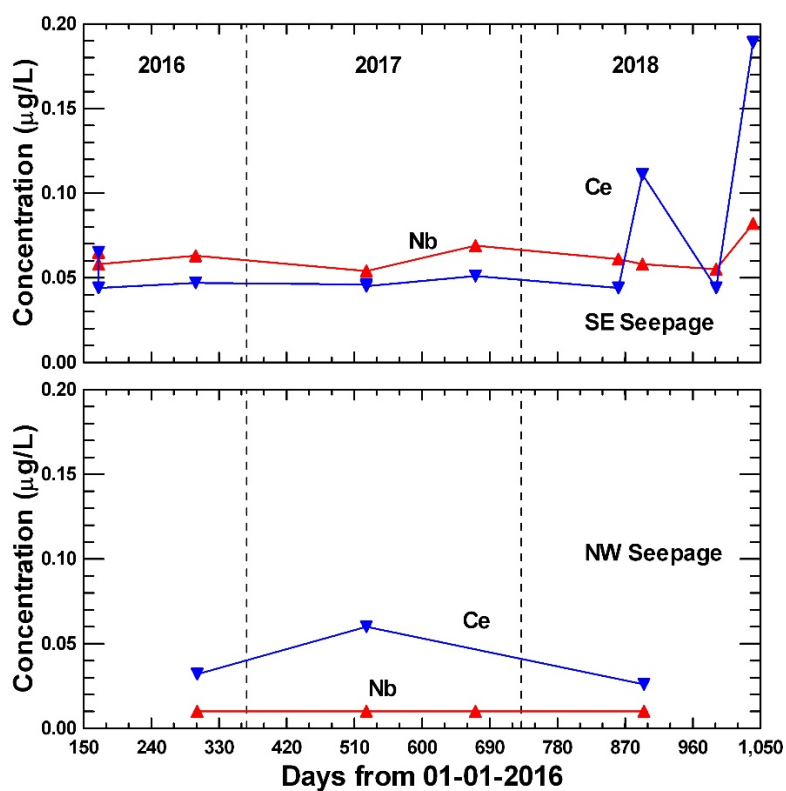


Figure 36: Temporal variations in seepage chemistry: a) B and U; b) Nb and Ce.

5. DISCUSSION

5.1 Conceptual hydrogeological model

A conceptual understanding of hydrogeological conditions within the SLC tailings impoundment provides context for interpreting groundwater chemistry and identifying pathways of solute transport to the receiving environment. Development of the conceptual model is guided by hydraulic properties of the tailings, head observations from piezometers, precipitation, and flow measurements.

Surface drainage on the impoundment is limited to a narrow channel leading from a pond on the northeastern edge of the facility to the decant pond in the central tailings basin (Figure 2). The northeast pond and channel are ephemeral whereas the decant pond was never observed to dry out completely. Based on head measurements from the MERNQ monitoring wells, WSP (2018) developed a piezometric contour map for the SLC tailings impoundment. They identified a pronounced groundwater mound, roughly aligned with the ephemeral channel and extending from the hill northeast of the impoundment to the decant pond. Beyond the decant pond, the mound drops off radially to the south, east, and west. The configuration of the water table in cross-section is illustrated in Figure 37.

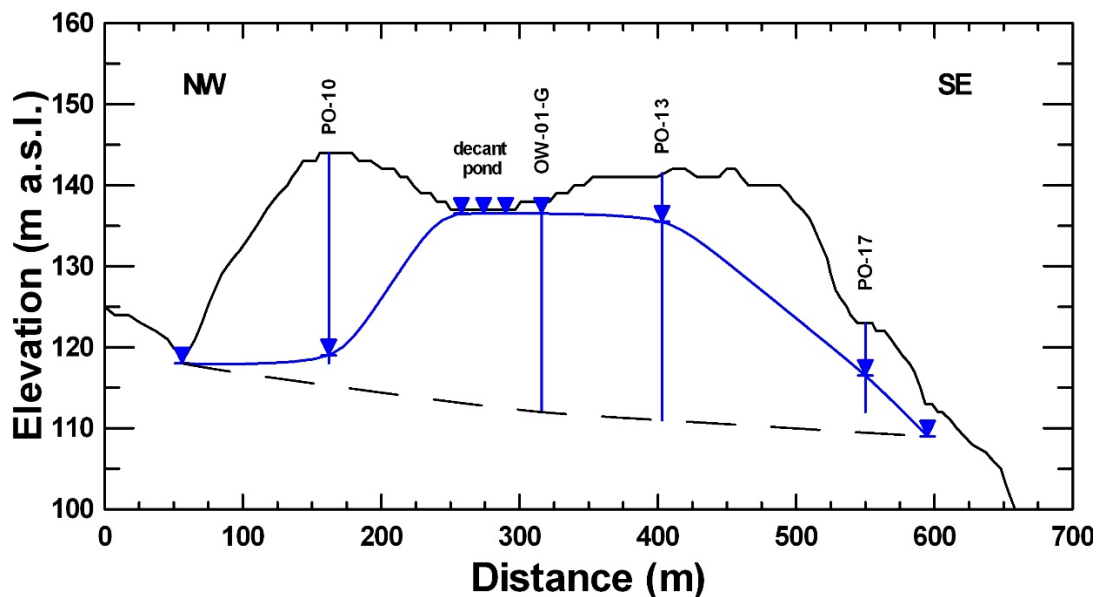


Figure 37: Schematic cross-section of the SLC tailings impoundment showing the approximate water table elevation (blue triangles). The dashed line marks the estimated base of the tailings.

The water table is deepest beneath the NW and SE embankments, which were constructed using permeable waste rock and coarse tailings. In the central basin, the water table is at surface of the decant pond, which is underlain by fine-grained tailings. In general, the position of the water table

and groundwater flow patterns within the impoundment are controlled by the tailings hydrostratigraphy. Coarser material was deposited near the embankments where tailings were discharged whereas finer low-permeability material accumulated in the central basin of the impoundment. A steep downward head gradient was observed within the fine tailings beneath the decant pond (Figure 23). By the law of refraction of flow lines (Domenico and Schwartz, 1990) groundwater flow is essentially vertical in these low-permeability tailings and horizontal in the sandy layer found at the base of the impoundment. If this basal sand is laterally extensive, it may be scavenging vertical seepage over a wide area.

Recharge to the tailings occurs as direct infiltration of rainfall and snowmelt with virtually no runoff. However, seasonal runoff from upslope of the impoundment collects in the northeast pond from where it flows in an ephemeral channel to the decant pond or infiltrates the tailings. Much of the water in the decant pond evaporates while the rest infiltrates the tailings as indicated by the downward head gradient shown in Figure 23. Visible discharge from the impoundment is limited to seepage points at the toes of the NW and SE embankments. A rough water balance calculation for the impoundment provides partial constraints on unquantified flux components.

Average annual precipitation (P) over the impoundment is 1107 mm (Table 2). The area of the impoundment is 19.2 ha (WSP, 2018). Average annual evapotranspiration (ET) is estimated at 390 mm (Statistics Canada, 2019). Assuming equal discharge rates from the NW and SE seep (Table 5), the annual surface outflow (Q_{SO}) from the impoundment is 268 130 m³, which is equivalent to a 1394 mm column of water. Assuming no change in storage, the annual water balance for the impoundment is written:

$$P - ET + Q_{SI} + Q_{GI} = Q_{SO} + Q_{GO}$$

Where Q_{SI} , Q_{GI} , and Q_{GO} represent surface water inflow, groundwater inflow, and groundwater outflow, respectively. The water balance equation can be rearranged as:

$$Q_{SI} + Q_{GI} - Q_{GO} = Q_{SO} - P + ET = 677 \text{ mm}$$

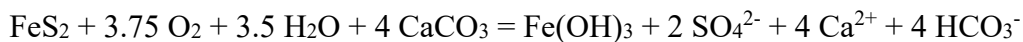
Surface runoff (Q_{SI}) from the height of land to the northeast is significant in the spring and early summer but has not been quantified. Unfortunately, the net (positive or negative) groundwater discharge to the regional flow system ($Q_{GI} - Q_{GO}$) is indeterminate based on available data.

5.2 Preliminary interpretation of hydrogeochemical data

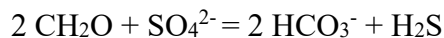
The composition of SLC tailings pore-waters shows a systematic evolution within the impoundment, with increasing sulfate and TDS concentrations between recharge and discharge points (Figure 38). However, pore-waters from OW-01 represent an exception to this trend. The waters exhibit a wide range of chemical types (Figure 39) including Ca-HCO₃ (OW-01-A), Ca-Mg-Na-HCO₃-SO₄ (OW-01-D), Na-HCO₃ (OW-01), and Ca-Mg-Na-SO₄-HCO₃ (SE embankment seep). Anion compositions are dominated by HCO₃ in the shallow piezometers of the recharge area and by SO₄ in the embankment seeps. Chloride concentrations are low although fluoride is a significant minor anion in some waters. Cation compositions of pore-waters from the shallow piezometers (OW-01-F,-A,-B,-C) show a trend of decreasing Ca with depth. The other compositions are more scattered and are not readily interpreted from the ternary plot. Greater insight into the hydrogeochemical processes operating within the tailings may be achieved by plotting depth profiles of diagnostic chemical ratios (Hounslow, 1995).

The ratio of HCO₃ (in meq) to SiO₂ (in mmol) is useful for determining the predominance of carbonate versus silicate mineral dissolution processes (Figure 40). Above a threshold ratio of 10, carbonate dissolution is indicated (Hounslow, 1995). Not surprisingly, pore-water compositions from shallow piezometers suggest carbonate dissolution. On the other hand, compositions from piezometers OW-01-D and OW-01-G indicate silicate dissolution. The high ratios observed at piezometer OW-01 are anomalous. The ratio Mg/(Ca+Mg) provides complementary information (Figure 40). If carbonate dissolution is occurring, a value less than 0.5 indicates calcite dissolution. For silicate dissolution, a value greater than 0.5 suggests weathering of ferromagnesian minerals. Thus, calcite dissolution is occurring at shallow depths within the tailings and ferromagnesian silicate dissolution is indicated in discrete permeable layers within the fine tailings (OW-01-D) and at the base of the impoundment (OW-01-G).

Since only trace amounts of gypsum have been identified in the tailings, pyrite oxidation is the most likely source of observed sulfate. Pyrite oxidation and sulfate reduction processes can be investigated by considering the ratio of HCO₃ to SO₄. Pyrite oxidation with neutralization by calcite dissolution is described by the equation:



Therefore, at equilibrium, the ratio HCO₃/SO₄ (in meq) should be approximately 1. Under anaerobic conditions, sulfate reduction by organic matter (CH₂O) is described by:



Therefore, progressive sulfate reduction is indicated by increasing HCO₃/SO₄ ratios. A depth profile of the ratio (Figure 41) suggests that pyrite oxidation starts in the shallow subsurface and reaches equilibrium with calcite dissolution at depths corresponding to piezometers OW-01-B and OW-01-C. This is consistent with the pH depth profile (Figure 25) and with decreasing availability of O₂ beneath the water table. At greater depths, the ratio increases significantly, which suggests sulfate reduction. This is consistent with a smell of H₂S noted when sampling the deeper

piezometers, and with observed elevated DOC and alkalinity values (Figures 27 and 28). Within the basal permeable layer, the ratio suggests approximate equilibrium conditions.

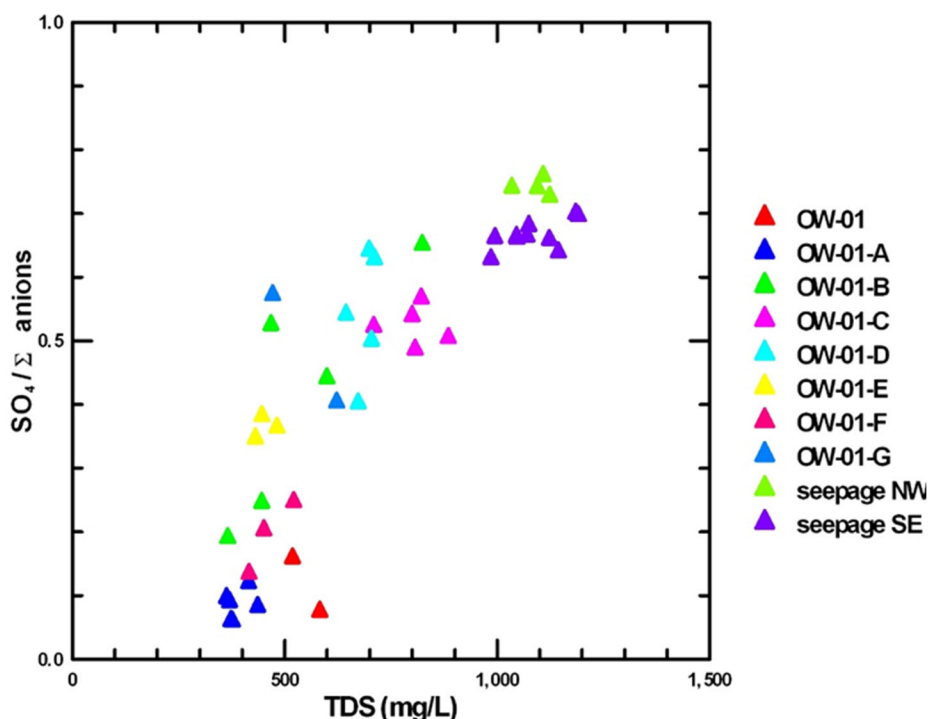
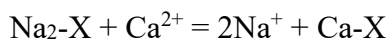


Figure 38: Cross-plot of SO₄ / Σ anions versus Total Dissolved Solids (TDS).

The ratio (Na+K-Cl)/Ca can be used to characterize the cation exchange process described by:



Where X represents a cation exchange surface such as a clay mineral. In the absence of any dissolving Na-bearing minerals, an increasing (Na+K-Cl)/Ca ratio would be indicative of Na being exchanged for Ca. Cation exchange is suggested by the monotonic increase of the ratio with depth throughout the fine tailings interval (Figure 41). Here, however, nepheline hydrolysis with formation of kaolinite may also release Na although this does not account for the removal of Ca with depth (Figure 26a).

Results presented here show that the major ion chemistry of downward flowing pore-water in the central basin of the tailings impoundment is controlled by several processes including calcite dissolution, pyrite oxidation, sulfate reduction, cation exchange and localized ferromagnesian silicate dissolution. However, pore-water compositions change abruptly in the permeable sandy layer at the base of the impoundment where groundwater flow is horizontal. This suggests that leakage from the overlying fine tailings is not the main source of water in the basal tailings or in the discharge at embankment seeps. Infiltrating waters following other, more permeable, pathways through the tailings are likely contributing more substantially to effluent from the impoundment.

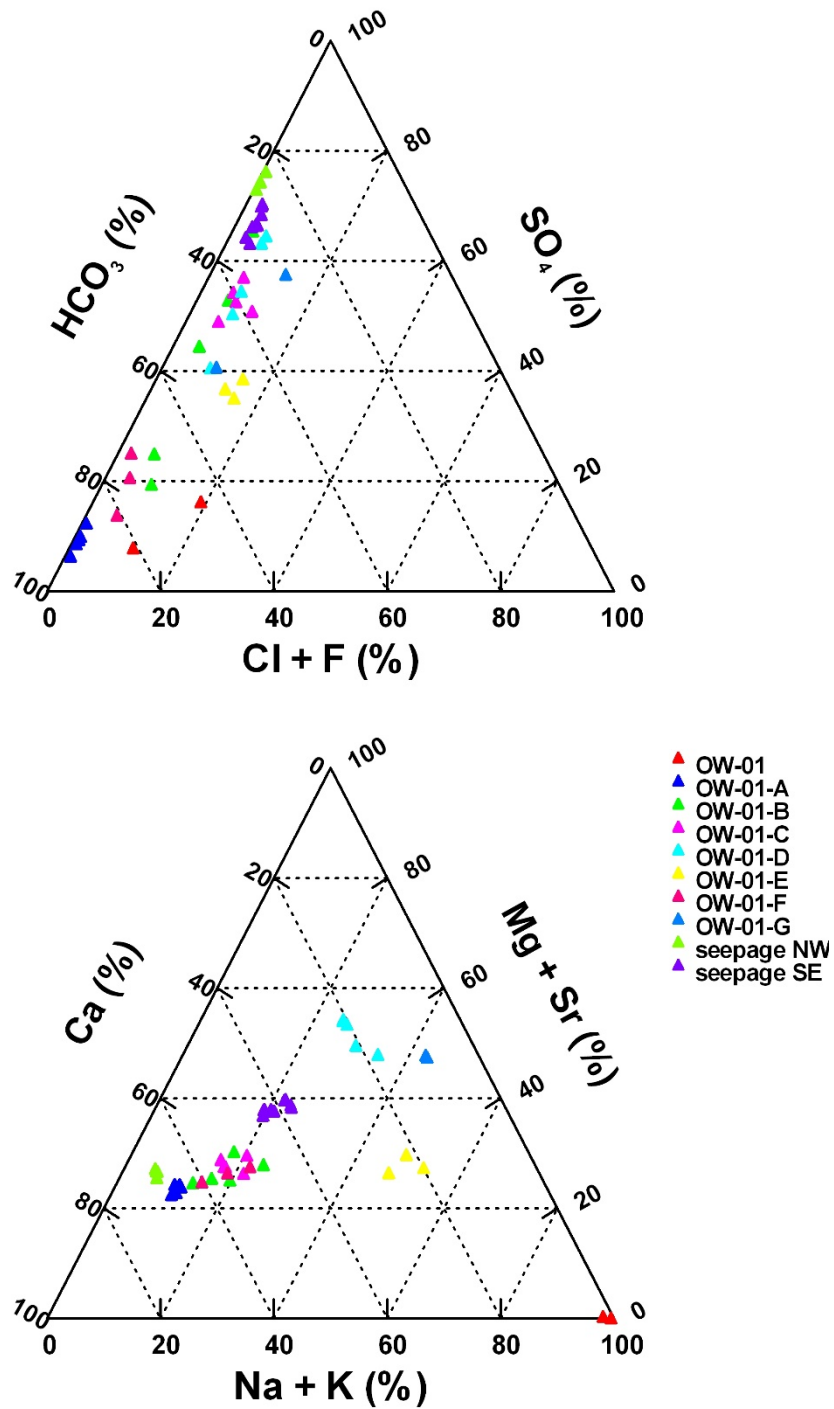


Figure 39: Ternary plots of major cations and anions from tailings piezometers and embankment seepage.

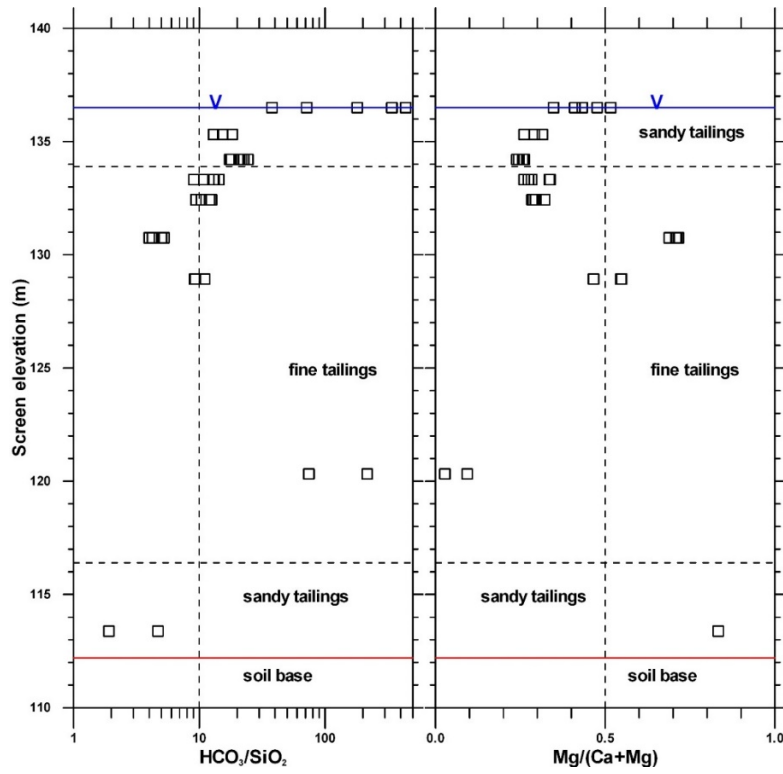


Figure 40: Depth profiles of geochemical ratios $\text{HCO}_3 / \text{SiO}_2$ and $\text{Mg} / (\text{Ca} + \text{Mg})$.

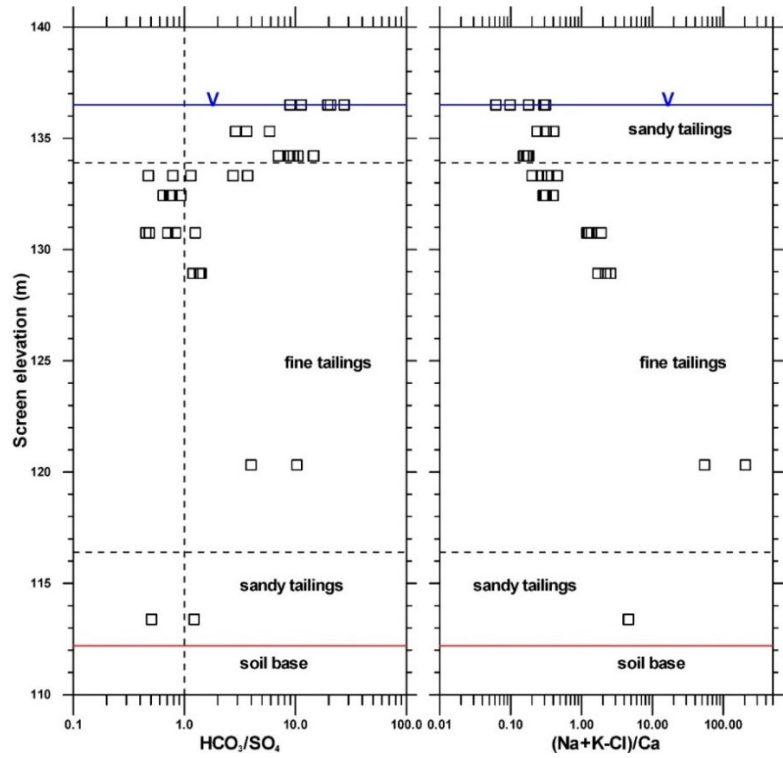


Figure 41: Depth profiles of geochemical ratios $\text{HCO}_3 / \text{SO}_4$ and $(\text{Na} + \text{K} - \text{Cl}) / \text{Ca}$

5.3 Comparisons with geochemical data from other studies

Geochemical data obtained in this study are compared with other data from the SLC site and with limited data from the active Niobec mine, which produces niobium from another carbonatite deposit in Québec. Data from the SLC site were gathered on behalf of the MERNQ as part of environmental characterization studies for eventual mine reclamation. The full report (WSP, 2018) is available online. Data from the Niobec mine were also obtained from online sources including a master's thesis on apatite recovery from mine tailings (Savard, 1981) and a report by the Ministère de l'Environnement du Québec filed as evidence during the public hearings on the proposed Niocan mine in Oka (BAPE-DB46, 2002).

Table 9 presents the major oxide and trace element compositions of tailings solids from the SLC and Niobec mines. Data from this study represent values averaged over 38 samples. Data from the WSP (2018) report are from two typical split-spoon samples obtained during the drilling of monitoring wells. Although fewer parameters are reported, the WSP results are consistent with GSC data. The Niobec ore beneficiation process produces several distinct waste streams including a carbonate concentrate and de-slimed tailings. Compared to the carbonate concentrate, the de-slimed tailings contain more SiO_2 and K_2O (likely in the form of biotite) and less P_2O_5 (in the form of apatite). Compared to SLC tailings, the Niobec wastes contain less CaO and more MgO , reflecting the dolomitic rather than calcitic composition of the ore zone. A comparison of trace element compositions between SLC and Niobec tailings suggests only a qualitative similarity in so far as Ba, Sr, and Zn appear enriched in both cases.

In many environmental assessment studies for mining projects, the geochemical composition of tailings effluent is estimated from the results of standard leach tests such as U.S. EPA 1311 TCLP (Toxic Characteristic Leaching Procedure). The TCLP procedure was developed to simulate leaching from landfills and to screen wastes for special disposal. It uses a weak acetic acid solution as an extraction fluid. The leachate is then analyzed for eight toxic metals or metalloids. Table 10 presents TCLP leach test results for two samples of SLC and Niobec tailings, respectively. These results are compared with actual seepage chemistry measured at the SE embankment of the SLC tailings disposal area. It can be seen that TCLP leach test results are very poor predictors of trace element compositions in mildly alkaline, rather than acidic, tailings effluent. Most metal concentrations are grossly overestimated whereas fluoride concentrations are underestimated.

The average composition of tailings seepage observed by the GSC at the SE embankment is compared with the compositions of two WSP (2018) samples from the same location (Table 11). In turn, these results are compared with compositions from two samples of Niobec tailings effluent. The GSC and WSP results are in good agreement, at least for the limited parameters analyzed by WSP. The Niobec seepage has a similar pH to SLC seepage but TDS and chloride concentrations are much higher. This is due to the brackish groundwater in host rocks of the Niobec deposit (Martin, 1993). Hardness and Mg concentrations are also higher, reflecting to dolomitic composition of the carbonatite core. Fluoride concentrations are very high, as they are in SLC seepage. Trace element concentrations are similar in tailings seepage from both mines.

Table 9: Comparison of tailings chemistry (solids) from the SLC and Niobec mines. Below detection (<). Not analyzed (-).

	units	SLC ¹	SLC ²	SLC ³	Niobec ⁴	Niobec ⁵	Niobec ⁶
SiO ₂	%	7.63	-	-	4.09	1.93	-
Al ₂ O ₃	%	2.23	1.71	2.02	0.83	0.23	-
Fe ₂ O ₃	%	3.92	3.89	2.96	4.82	2.71	-
MgO	%	3.15	-	-	15.80	14.56	-
CaO	%	43.76	-	-	32.14	38.19	-
Na ₂ O	%	0.27	-	-	0.08	0.09	-
K ₂ O	%	0.84	-	-	0.49	0.04	-
TiO ₂	%	0.24	-	-	-	-	-
P ₂ O ₅	%	2.46	-	-	5.16	9.45	-
MnO	%	0.82	0.61	0.76	0.99	0.85	0.84
Nb ₂ O ₅	%	0.27	-	-	0.18	0.17	-
Total S	%	0.53	-	-	-	-	0.88
Ba	ppm	3071	-	-	-	-	190
Co	ppm	5	7	3	-	-	5
Sr	ppm	11116	-	-	-	-	830
Th	ppm	65	-	-	23	30	-
U	ppm	20	-	-	5	2	< 5
Mo	ppm	27	11	11	-	-	< 2
Cu	ppm	11	8	4	14	9	4
Pb	ppm	46	18	12	58	54	10
Zn	ppm	456	211	339	420	180	160
Ni	ppm	2	7	< 2	-	-	< 2
As	ppm	4	4	4	-	-	9
Cd	ppm	4	3	2	-	-	1
Ag	ppm	< 1	< 0.5	< 0.5	-	-	< 2
Se	ppm	< 1	2	8	-	-	4
F	ppm	3466	-	-	-	-	-

¹ GSC, this study, average of 38 samples

² WSP (2018), Table 14B, sample PO11-CF7, 2017-05-15

³ WSP (2018), Table 14B, sample PO13-CF18, 2017-05-30

⁴ Savard (1981), Table 7, de-slimed tailings

⁵ Savard (1981), Table 6, carbonate concentrate

⁶ BAPE-DB46 (2002), Annex 3, Table 1, sample B (aged tailings, undifferentiated)

Table 10: Comparison of observed tailings effluent chemistry with tailings leachate (TCLP) chemistry from the SLC and Niobec mines. Below detection (<). Not analyzed (-).

	units	SLC ¹	SLC ²	SLC ³	Niobec ⁴	Niobec ⁵
pH	-	7.8	-	-	-	-
Spec. Cond.	µS/cm	1335	-	-	-	-
TDS	mg/L	1086	-	-	-	-
Alkalinity	mg/L CaCO ₃	224	-	-	-	-
Hardness	mg/L CaCO ₃	586	-	-	-	-
DOC	mg/L	1	-	-	-	-
Ca	mg/L	123	-	-	-	-
Mg	mg/L	68	-	-	-	-
Na	mg/L	58	-	-	-	-
K	mg/L	27	-	-	-	-
Sr	mg/L	10	-	-	-	-
SO ₄	mg/L	485	-	-	-	-
Cl	mg/L	2.4	-	-	-	-
F	mg/L	8.4	< 10	< 10	1.9	1.7
Fe	µg/L	199	< 10000	< 10000	-	-
Mn	µg/L	461	16400	19700	-	-
Al	µg/L	2	1200	200	-	-
Ag	µg/L	< 1	< 1	< 1	-	-
As	µg/L	< 1	< 200	< 200	< 1	< 1
B	µg/L	11	< 5000	< 5000	190	190
Ba	µg/L	159	8800	1100	5200	1900
Cd	µg/L	< 1	36	24	< 1	10
Cu	µg/L	< 1	< 100	< 100	-	-
Co	µg/L	< 1	10	20	-	-
Cr	µg/L	< 1	10	50	13	4
Mo	µg/L	53	50	< 10	-	-
Ni	µg/L	< 1	< 10	< 10	-	-
Pb	µg/L	< 1	< 50	< 50	6	1
U	µg/L	2	< 50	< 50	< 5	< 5
Zn	µg/L	21	800	900	-	-

¹ GSC, this study, SE embankment seepage, average of 10 samples

² WSP (2018), Table 12, leachate sample TR16-100-200, 2017-05-10

³ WSP (2018), Table 12, leachate sample TR19-0-100, 2017-05-10

⁴ BAPE-DB46 (2002), Annex 3, Table 2, leachate sample A (fresh tailings)

⁵ BAPE-DB46 (2002), Annex 3, Table 2, leachate sample B (aged tailings)

Table 11: Comparison of tailings effluent chemistries from the SLC and Niobec mines. Below detection (<). Not analyzed (-).

	units	SLC ¹	SLC ²	SLC ³	Niobec ⁴	Niobec ⁵
pH	-	7.8	8.0	7.9	8.0	8.1
Spec. Cond.	µS/cm	1335	1143	1316	5680	4260
TDS	mg/L	1086	-	-	3746	3406
Alkalinity	mg/L CaCO ₃	224	235	224	360	200
Hardness	mg/L CaCO ₃	586	107	576	1454	1290
DOC	mg/L	1	-	-	5	9
Ca	mg/L	123	-	-	-	-
Mg	mg/L	68	-	-	159	330
Na	mg/L	58	-	-	-	-
K	mg/L	27	-	-	51	34
Sr	mg/L	10	-	-	-	-
SO ₄	mg/L	485	616	541	432	278
Cl	mg/L	2.4	2.4	2.0	1234	1213
F	mg/L	8.4	-	-	11.5	15
Total N	mg/L	0.24	0.02	< 0.02	0.73	1.00
Total P	mg/L	< 0.05	-	-	0.02	< 0.05
Fe	µg/L	199	547	< 70	390	220
Mn	µg/L	461	512	197	550	276
Al	µg/L	2	16	< 5	< 70	21
Ag	µg/L	< 1	< 1	< 1	3	2
As	µg/L	< 1	< 1	< 1	< 10	< 1
Ba	µg/L	159	-	-	-	-
Cd	µg/L	< 1	-	< 1	< 4	13
Cu	µg/L	< 1	< 1	2	16	7
Co	µg/L	< 1	< 1	< 1	6	< 1
Mo	µg/L	53	56	57	42	36
Ni	µg/L	< 1	2	3	< 20	< 20
Pb	µg/L	< 1	< 1	< 1	40	< 30
U	µg/L	2	-	-	-	-
Zn	µg/L	21	21	9	< 10	< 10

¹ GSC, this study, SE embankment seepage, average of 10 samples

² WSP (2018), Table 18B, sample site ES-09, 2017-09-11

³ WSP (2018), Table 18C, sample site ES-09, 2017-11-10

⁴ BAPE-DB46 (2002), Annex 2, tailings effluent, 2000-07-05

⁵ BAPE-DB46 (2002), Annex 2, tailings effluent, 2001-07-04

6. SUMMARY AND CONCLUSIONS

Between 1961 and 1976, the St. Lawrence Columbium (SLC) mine produced pyrochlore concentrate from a niobium-bearing carbonatite deposit near Oka, Québec. This report is concerned with the environmental signature of tailings generated by the processing of carbonatite ore. It describes their physical properties, their mineralogy, their bulk chemistry, and the chemistry of seepage draining from the tailings impoundment. Tailings solids were sampled at closely-spaced intervals in a borehole intersecting the full thickness of the impoundment. Tailings pore-waters were sampled in a nest of eight piezometers screened at different depths beneath the decant pond. Tailings effluent was sampled at seepage points on the toes of the embankments of the facility.

Grain-size distribution data for the tailings show that they range in texture from medium sand to clayey silt. At the piezometer nest, a 2.6 m layer of coarse tailings caps 17.5 m of silty tailings with sandy beds. This sequence rests on a 4.4 m layer of sandy tailings covering the base of the impoundment. The average water content, density and porosity of the tailings are 30 %, 2.82 g/cm³, and 47 %, respectively. The geometric mean hydraulic conductivity of the tailings is estimated at 1.4 E-07 m/s using the empirical Beyer formula based on grain-size data.

The tailings are composed mainly of calcite (82 %) with biotite/phlogopite (12 %), fluorapatite (4 %). Chlorite, nepheline, smectite, kaolinite, monticellite, magnetite, pyrite, perovskite, and unrecovered pyrochlore are found in minor or trace amounts. The calcite contains significant Sr, Mn, Mg, Ba, and REE. The fluorapatite contains elevated concentrations of REE although most of the REE inventory is hosted by calcite. Fluorine is found in both the fluorapatite and biotite. In addition to Nb, the pyrochlore hosts U, Th, Ta, and Zr. A discrete layer within the tailings contains over 5000 ppm of Nb, with U and Th concentrations reaching 80 and 175 ppm, respectively. However, average U and Th concentrations in the tailings are 20 and 65 ppm, respectively. The average total S content (mainly as sulfide) of the tailings is 0.53 % although in one depth interval containing abundant pyrite it reaches 3.5 %.

Head measurements in the piezometer nest indicate a strong downward hydraulic gradient from the decant pond through the sequence of coarse and fine tailings to the top of the basal sandy layer where groundwater flow is essentially horizontal. Depth profiles of tailings pore-water chemistry observed in the piezometers show significant variations across the thickness of the impoundment. In the shallow tailings, major ion chemistry is controlled by calcite dissolution and sulfide oxidation. At greater depths, oxygen is depleted and sulfate reduction occurs. Throughout the fine tailings, Ca, Mg, and Sr concentrations decrease with depth as a result of cation exchange whereas Na and K concentrations increase steadily. Fluoride concentrations also increase with depth and approach 25 mg/L. Concentrations of Zn and Mo peak in the shallow subsurface as a result of sulfide oxidation. Dissolved concentrations of other trace elements (U, Nb, REE) are very low and do not show any clear depth trend. However, total REE concentrations in pore-water may reach 1 mg/L. The average pH and specific conductance of effluent from the tailings impoundment are approximately 8 and 1400 µs/cm, respectively. Effluent from the SE embankment is a mixed cation (Ca-Mg-Na) and mixed anion (SO₄-HCO₃) type water containing significant sulfate (500 mg/L) and an average fluoride concentration of 8 mg/L in excess of the 0.12 mg/L guideline for the protection of freshwater aquatic life. Concentrations of U and other trace elements are low.

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APPENDIX
PHOTOGRAPHS OF TAILINGS CORE SAMPLES



Figure A-1: Split-spoon core sample from the stratigraphic borehole interval 0-2.5 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-642.



Figure A-2: Split-spoon core sample from the stratigraphic borehole interval 2.5-4.5 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-643.



Figure A-3: Split-spoon core sample from the stratigraphic borehole interval 5-7 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-644.



Figure A-4: Split-spoon core sample from the stratigraphic borehole interval 7.5-9.5 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-645.

10' - 12' No Recovery

Sample DUA18-T5
from top 3" of next core

Figure A-5: Split-spoon core sample from the stratigraphic borehole interval 10-12 ft.



Figure A-6: Split-spoon core sample from the stratigraphic borehole interval 12.5-14.5 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-646.



Figure A-7: Split- spoon core sample from the stratigraphic borehole interval 15-17 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-647.



Figure A-8: Split- spoon core sample from the stratigraphic borehole interval 17.5-19.5 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-648.



Figure A-9: Split-spoon core sample from the stratigraphic borehole interval 20-22 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-649.



Figure A-10: Split-spoon core sample from the stratigraphic borehole interval 22.5-24.5 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-650.



Figure A-11: Split-spoon core sample from the stratigraphic borehole interval 25-27 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-651.



Figure A-12: Split-spoon core sample from the stratigraphic borehole interval 27.5-29.5 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-652.



Figure A-13: Split-spoon core sample from the stratigraphic borehole interval 30-32 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-653.



Figure A-14: Split-spoon core sample from the stratigraphic borehole interval 32.5-34.5 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-654.



Figure A-15: Split-spoon core sample from the stratigraphic borehole interval 35-37 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-655.



Figure A-16: Split-spoon core sample from the stratigraphic borehole interval 37.5-39.5 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-656.



Figure A-17: Split-spoon core sample from the stratigraphic borehole interval 40-42 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-657.



Figure A-18: Split-spoon core sample from the stratigraphic borehole interval 42.5-44.5 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-658.



Figure A-19: Split-spoon core sample from the stratigraphic borehole interval 45-47 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-659.



Figure A-20: Split-spoon core sample from the stratigraphic borehole interval 47.5-49.5 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-660.



Figure A-21: Split-spoon core sample from the stratigraphic borehole interval 50-52 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-661.



Figure A-22: Split-spoon core sample from the stratigraphic borehole interval 52.5-54.5 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-662.



Figure A-23: Split-spoon core sample from the stratigraphic borehole interval 55-57 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-663.



Figure A-24: Split-spoon core sample from the stratigraphic borehole interval 57.5-59.5 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-664.



Figure A-25: Split- spoon core sample from the stratigraphic borehole interval 60-62 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-665.



Figure A-26: Split- spoon core sample from the stratigraphic borehole interval 62.5-64.5 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-666.



Figure A-27: Split-spoon core sample from the stratigraphic borehole interval 65-67 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-667.



Figure A-28: Split-spoon core sample from the stratigraphic borehole interval 67.5-69.5 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-668.



Figure A-29: Split-spoon core sample from the stratigraphic borehole interval 70-72 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-669.



Figure A-30: Split-spoon core sample from the stratigraphic borehole interval 72.5-74.5 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-670.



Figure A-31: Split-spoon core sample from the stratigraphic borehole interval 75-77 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-671.



Figure A-32: Split-spoon core sample from the stratigraphic borehole interval 80-82 ft. Photograph by A.J. Desbarats. NRCan Photo Database number 2020-672.