Environmental Impacts of Historical Mine Tailings Disposal at Cochrane Hill Gold District, Nova Scotia

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Abstract

Since the first Nova Scotian gold rush in the early 1860s, gold mining and milling processes have generated tailings piles containing mercury, arsenic, cyanide, and other potentially toxic elements. Most of the gold deposits occur in the Cambro-Ordovician Meguma Group of southern Nova Scotia, and mining has been carried out at more than 60 formal gold districts for a total production of 37 t of gold. The Cochrane Hill gold deposit is located in Guysborough County, approximately 15 km north of Sherbrooke. The host rocks consist of amphibolite-facies metagreywacke and slates, and most of the gold is associated with quartz veins that intrude slates rich in arsenopyrite. Mining and milling of gold ore at Cochrane Hill took place from 1877 to 1928, and again from 1981 to 1988, resulting in two separate tailings piles. During the first period of operation, stamp milling and mercury amalgamation were used to extract gold from the ore, and the tailings were slurried into a local stream. In the 1980s, ball milling and cyanidation were used to process the ore, and the tailings were deposited into an on-site impoundment. The main objectives of this study are to: (1) characterize the mineralogy and metal concentrations in the two tailings piles; (2) assess the relative reactivity of metals and metalloids in the amalgamation versus cyanidation tailings; and (3) examine the downstream impacts of drainage from the tailings piles. Forty five samples of tailings were collected from 16 different sites at Cochrane Hill in September 2003, and water samples were collected at 12 locations within and downstream of the tailings in November 2003. Efflorescent salts were also collected from the surface of the cyanidation tailings. X-ray diffraction, scanning electron microscopy and electron microprobe analyses have been completed on select tailings samples. The primary mineralogy of the tailings includes quartz, muscovite, biotite, staurolite, arsenopyrite, anorthite, and actinolite/tremolite. Secondary mineralogy includes ferric hydroxide, basaluminite, gypsum and Turnbull's Blue $Fe_3(Fe(CN)_6)_2$. Geochemical results show that the amalgamation tailings contain significantly higher Hg concentrations (21-63,000 ppb) than the cyanidation tailings (<5-25 ppb). Both tailings piles also contain high concentrations of As (280-41,000 ppm), which occurs naturally in the ore. Concentrations of Hg, As, Al, Mn, Ni, Zn, Pb, and S in waters at the mine site are elevated relative to background levels. These concentrations drop quickly downstream from the mine. The tailings are having a direct effect on the chemistry of the waters close to the mine site but this effect is diminished by 1.5 km downstream.

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CHAPTER 1: INTRODUCTION

1.1 Opening Statement

The history of gold mining in Nova Scotia dates back to the 1850s, when gold-bearing quartz was first discovered near Mooseland by Captain L'Estrange. Since that time, there have been three main gold rushes in Nova Scotia and some periodic mining activity between the 1970s and the present. The last gold rush ended in 1942. In the early days, most gold was extracted from the ore using mercury and the wastes were dumped in nearby lakes, swamps, and rivers. Starting in the 1890s, cyanide was also used to extract the gold; this was the preferred method for the past 30 years. More recent wastes were placed in impoundments in an attempt to limit their environmental footprint (Bates, 1987).

The wastes that result from the processing of ore for gold are called tailings. Tailings material is generally sand- to silt-sized as a result of crushing. At every point during the processing of ore some cyanide or mercury is lost and accumulates in the tailings piles (Nriagu and Wong, 1997). There are also naturally-occurring toxic elements such as arsenic present in the tailings piles (Dale and Freedman, 1982). As a result, windblown dust and runoff from the tailings piles may have an adverse effect on the surrounding environment (Mudroch and Clair, 1986).

The present study focuses on the Cochrane Hill Gold District in Nova Scotia, where there are at least two distinct tailings piles. The first pile was deposited from 1877-1928 (Malcolm, 1929) when a 10-stamp mill and mercury amalgamation were used to process the ore. The second was deposited from 1981-1988 (Northumberland Mines Ltd., 1982; Novagold Resources Inc., 1988) when a ball mill and cyanidation were used to extract gold from the ore. This thesis examines the mineralogy and metal concentrations within the two piles and the relative reactivities of the old and new tailings. This information, along with water sample analyses, is used to assess the impacts of the tailings on the surrounding environment.

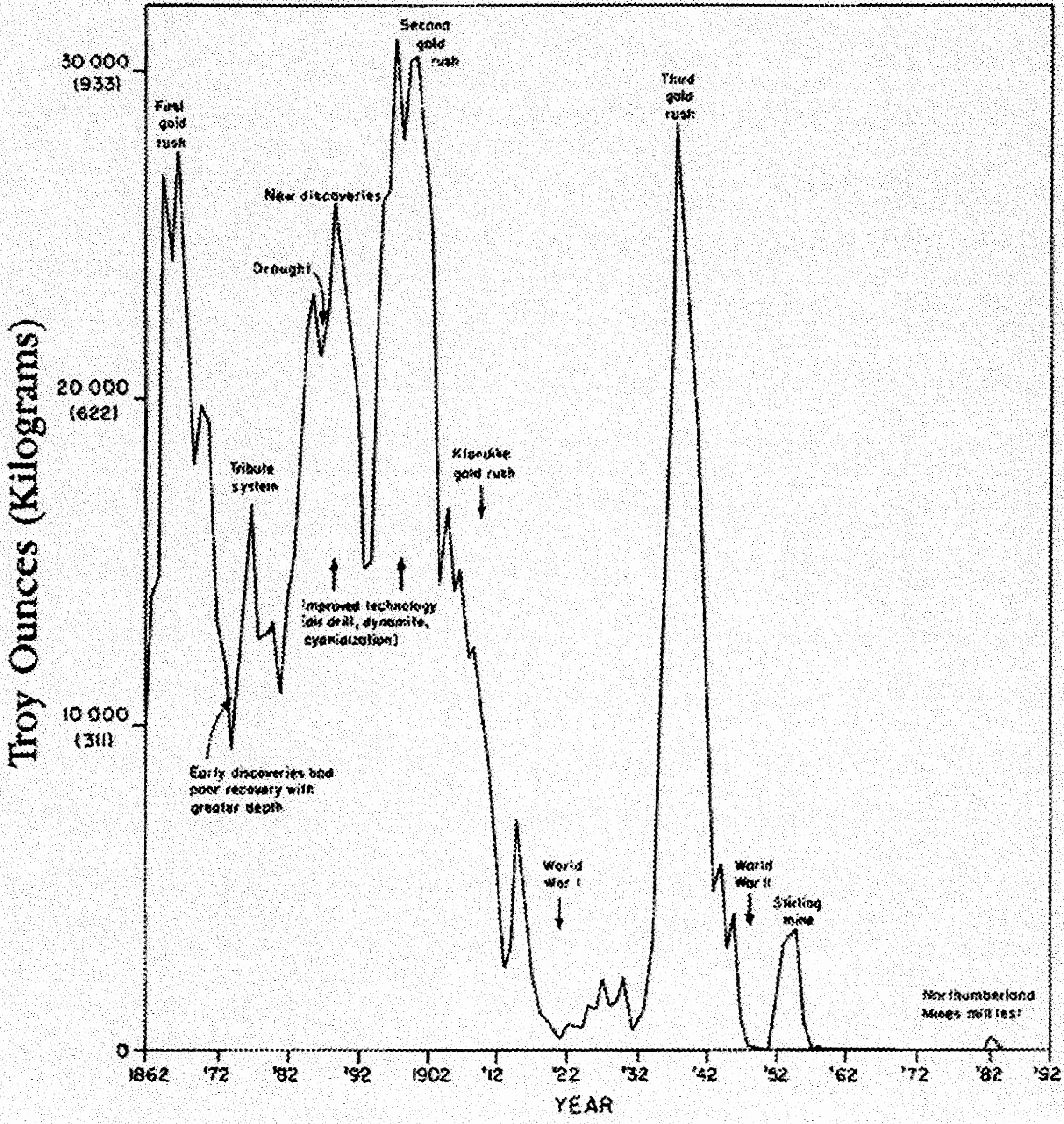
1.2 History of Gold Mining in Nova Scotia

There are three main gold rushes that have played a role in defining the history of many towns and villages in Nova Scotia. Place names like Bras d'Or and Cape d'Or indicate that French settlers may have found gold; however, there is no evidence of ancient workings from this time. During the 1830s people building roads saw gold, but its value was not appreciated at the time (Malcolm, 1929; Bates, 1987). In 1849 the California gold rush caused people throughout North America to search for gold on their properties. That same year John Campbell made the first explorations for gold after he successfully panned gold at locations along the Atlantic Coast (Archibald, 1939). In 1858 the first authentic discovery of gold in quartz (i.e. lode gold) was made by Captain L'Estrange while hunting in Mooseland (Messervey, 1937).

John Gerrish Pulsiver, a farmer from Musquodoboit, essentially began the *first gold rush* in Nova Scotia when he found gold in 1860 (Fig. 1.1). Other discoveries in Tangier, Lawrencetown, The Ovens, Wine Harbour, Isaacs Harbour, and Gold River were quick to follow. Exploration also spread southwest of the Eastern Shore. The highest total yield during the first gold rush was 27,538 ounces (780,702 g) of gold in 1867. During the 1870s poor mining methods, bad management and incompetency led to a decline in the gold industry of Nova Scotia. At the time there were no means of predicting where more gold may be and the money that had been made had been spent (Bates, 1987).

By the 1890s, technological improvements including the introduction of dynamite for blasting, the use of cyanide for extraction, and more efficient machinery and mills allowed lower grade ore to be worked and the *second gold rush* began (Fig 1.1). It lasted from 1896 to 1903, with a peak yield of 31,113 ounces (882,054 g) of gold in 1898. When the Klondike gold rush took place in the late 1890s there was a move west and a decline in gold production in Nova Scotia (Bates, 1987).

In 1921, C.S. Parsons examined the possibility that there might be a large amount of unrecovered gold still left in the tailings and waste rock at various gold mines throughout Nova Scotia. He found that a maximum of 70 percent of the gold in the rock was actually recovered. This promoted some mine owners to rework some of the tailings and the waste rock piles (Parsons, 1921)



The quantity of gold produced in Nova Scotia from 1862 to present day. The suspected causes for particular peaks and troughs are indicated on the graph.

Figure 1.1 Gold produced in Nova Scotia from 1862 to 1992 (Bates, 1987)

When the price of gold rose and there was great demand for arsenic produced from arsenopyrite there was a *third gold rush* from 1932 to 1942 (Fig 1.1). Arsenic was in great demand because it was used as a pesticide to treat wood that would be used for building.

Historical gold mining between the 1860s and mid-1940s took place in approximately 64 official gold mining districts yielding about 1.2 million ounces of gold (Blakeman, 1978). During the late 1980s the price of gold rose to US $820 per ounce, old mines were examined closer and more prospecting took place (Bates, 1987). Today the search for gold continues in Nova Scotia.

1.3 Gold Milling Technology and Methods

Most historical gold production in Nova Scotia has come from lode gold deposits, in which gold is associated with quartz-carbonate veins (Robert, 1996). In general, these veins are either well defined, or form smaller veins or stringers which are found dispersed throughout narrow slate belts (Smith, 1982; Sangster, 1990). Mining therefore must take place along very selective tracts (Messervey, 1937). In Nova Scotia, the first mining operations worked the quartz veins at the surface and involved some trenching using picks, shovels and muscle (Bates, 1987).

The first underground mine was the Tangier Mine in 1860 (Bates, 1987). At underground operations the ore would be hoisted to the surface and deposited in an elevated bin. By gravitation the ore would then pass through a feeder, across a grizzly (coarse sieve) and then onto a sorting table. The grizzly separated the small ore, less than 2 inches, from the larger ore, greater than 2 inches. The larger ore would be sorted by hand on the sorting table and the valueless rock would be thrown in a waste pile. The early gold extraction methods were often wasteful, for in many cases the quartz ore was only brought to the surface if gold was visible. This means that much of the fine gold would have been lost during early production. At the end of the sorting table there was a jaw crusher that would crush the rock into 1 and ½ inch cubes that would then be added to the less than 2 inch pieces that fell through the grizzly. All material less than 2 inches in diameter was then subjected to further crushing (Archibald, 1939).

In the very early days, an arrastra was used to crush the gold ore by the movement of a large rock attached to a wooden arm. Another method used to crush gold ore in Nova Scotia was the Chilean mill. This mill was composed of massive wheels of granite that revolved on edge in an iron pan to crush the quartz into a watery paste. Eventually, these mills were replaced by the more efficient stamp mill, which consisted of vertical iron or steel rods that were raised by a cam 10-18 inches (25-45 cm) and dropped at a speed of 50-80 drops per minute. Each battery included 5 of these stamps, and at most mines there was more than one battery to increase crushing capacity (Faribault, 1903; Bates, 1987). To keep the ore equally distributed, the stamps would fall in specific orders, for example 1,4,2,5,3 or 1,3,5,2,4. The stamp mill would be fed with water and this along with the pulverized rock would splash against screens that were set inside the battery allowing for the fine ore to be ejected (Archibald, 1939). As time went on, stamp mills were replaced by rod or ball mills, which consisted of a large cylindrical tumbler containing steel rods or balls that would grind the ore. These new mills were capable of processing much larger volumes of ore (Hudson et al., 1999).

Once the ore was crushed, there were three main methods used in Nova Scotia to extract gold from the crushed material:

1) Gravity separation: This method is a physical separation based on differences in specific gravity. Water is added to the crushed ore in a gold pan or sluice box and the resulting slurry is moved about in order to collect the denser gold on the bottom.

2) Amalgamation: This process involves dissolving gold in mercury (Bates, 1987). The crushed ore is slurried over two mercury coated copper plates. There is a 2-inch drop between the two plates in order to catch the amalgam that has mixed with the crushed ore. The correct amount of mercury must be maintained on the plates as to keep the plates in a pasty condition. The amalgam (i.e. gold dissolved in mercury) accumulates on the plates in ridges and knots before being scraped off. The ore that is not dissolved by the mercury is disposed of and called tailings (Archibald, 1939). Figure 1.2 is a picture of these plates. Mercury is also added to the battery box of the stamp mill and huge losses of mercury would occur as a result of flouring caused by the battering action of the stamps. Flouring is the break up of mercury into fine particles that will not amalgamate. The lost mercury would be deposited on the floor of the mill or with the tailings (Ashley, 2002).

Figure 1.2 Cleaning up the 20 Stamp Mill Dufferin Gold Mine Salmon River, Halifax County, Nova Scotia. 1893. Photo by E.R. Faribault, Geological Survey of Canada. Reproduced with permission from the Earth Sciences Sector Photo Library Collection, Ottawa.

The gold and mercury mixture from the plates and battery box is placed in a leather bag which is squeezed to remove the excess mercury. The amalgam is then heated in a closed system to vaporize the mercury and collect the gold, which is then melted into blocks. During the process of heating, some mercury can be lost to the atmosphere and historically there were cases of mercury poisoning among the people who worked in the mills (Bates, 1987).

3) Cyanidation: This process replaced amalgamation during the 1880s as a more efficient means of extraction. Crushed ore is added to a mixture of lime and cyanide, the gold is complexed by the cyanide and dissolves, and unwanted solids are removed by filtering. Zinc dust is then added to settle gold out of solution. This is now the most widely used technique in Canada and is the basis for the modern heap leaching process, in which cyanide solutions are poured over piles of crushed ore and waste rock to extract residual gold. The leach solutions are then collected and refined to extract the dissolved gold (Bates, 1987).

1.4 Previous Environmental Work

Mercury has been used in gold mining since Roman times. From 1550 to 1930 over 260,000 t of mercury were released to the biosphere due to large scale gold production in the Americas, Australia, Southeast Asia and England. Presently mercury amalgamation is used as the main technique for gold production in South America, China, Southeast Asia and some African countries (Lacerda, 1997)

At every stage in the amalgamation process mercury is lost to the surrounding environment, and often with use the mercury would get greasy so it would be discarded. When mercury is lost during processing it is lost to the atmosphere or is deposited with the tailings and on the floor and ground around the mill. The current loss of mercury in the Amazon of Brazil is 1.3-1.7 kilogram of mercury per kilogram of gold recovered. In the atmosphere mercury has a residence time of 6-24 months; therefore, it can be transported over great distances. Once the mercury is deposited it is either re-emitted to the atmosphere and transported again or it is sequestered in organic soil layers or the water column (Nriagu and Wong, 1997).

Studies of gold mine tailings over the past 20-30 years have documented negative environmental impacts in many parts of the world.

For example a study of the Jack of Clubs Lake in Wells, British Columbia by Azcue et al. (1994) showed elevated levels of arsenic and lead in the lake sediments as a result of nearby abandoned gold mine tailings. High concentrations of As, Cd, Cu and Pb were found in invertebrates collected from streams flowing through the mine tailings (Azcue et al., 1994).

In 1977 an investigation into the headwater lakes and bottom sediments of the Shubenacadie River in Nova Scotia revealed that they were contaminated with arsenic, mercury, lead and other metals. A study in the Waverley area concluded that 17% of the house wells contain arsenic levels above 50 µg/L, which is a limit set by the World Health Organization for drinking water (Grantham et al., 1977). The results of the Grantham et al., 1977 study indicate that arsenic, mercury, lead and zinc from past milling activities are being deposited in downstream lakes. This accumulation is gradually contaminating a larger part of the Shubenacadie River system (Mudroch and Clair, 1986).

In 1978 the Environmental Protection Service in the Atlantic Region carried out a study with two main objectives. The first was to identify the background levels of mercury in the soils and sediments in areas surrounding former gold mining sites where the amalgamation process was used extensively. Based on those results, a detailed biological and chemical sampling program was carried out in areas below selected milling operations to establish residual mercury levels in the terrestrial and aquatic environments. These detailed studies took place at the Montague, Mount Uniacke and Oldham gold districts. The results showed higher levels of mercury in soils and sediments in areas adjacent to or directly affected by gold milling activity. Mercury present in the tailings is relatively insoluble, and is being released very slowly into the surrounding environment (EPS Atlantic Region Report, 1978). In 1982, arsenic pollution was also examined at the Montague Gold District. Arsenic was found to be present mainly as arsenopyrite which is naturally occurring. Much of the arsenic input associated with the Montague Gold District will reach sinks like soil and sediment eventually. The study at Montague included plant collection and fish caging experiments and concluded that the tailings are naturally being vegetated by arsenic-tolerant plants. It also recommended that reclamation be done at the Montague site (Dale and Freedman, 1982).

More recently, Wong et al. (1999) characterized the Goldenville gold mine tailings. At the Goldenville mills 63 tons of mercury were lost and released into the environment between 1862 and 1945 (Nriagu and Wong, 1997). The Goldenville study examined the metal contents of the stream water, stream and lake sediments, tailings and vegetation. The results showed that the tailings have been distributed over time across the stream basin forming a tailings area of about 2 km^2. There is a continuous release of arsenic, mercury and lead from the tailings to the ecosystems downstream. The mines in this area closed over 50 years ago but sedimentary records of metal accumulation in Gegogan Lake, Nova Scotia show the release of metals from the tailings has not slowed down (Wong et al., 1999).

Cyanide (CN^-) is used extensively world wide to separate gold from pyrite and arsenopyrite; however, there are no published studies of cyanide impacts on the environment surrounding Nova Scotia gold mine sites. Solutions of cyanide and water are not thermodynamically stable, but at a high pH oxidation of cyanide is very slow. Lime or sodium hydroxide is normally added to cyanide to prevent the outgassing of hydrogen cyanide gas (HCN). Cyanide is not only lost through evaporation but through hydrolysis, conversion to thiocyanate, precipitation of cyanometallic compounds and by adsorption. When cyanide is released it first dilutes. Complex cyanide will likely remain mobile and not adsorb. After that point what happens is determined by the amount of light, Eh, pH, presence and amount of other compounds and elements and the presence or absence and amount of certain microorganisms. Natural UV light will destroy cyanide. The iron cyanide solid, Prussian Blue, is stable in an oxidizing environment. Another similar solid is Turnbull's Blue, which dominates a low Eh environment (Souren, 2000). Residual cyanide concentrations from past gold mining activities are likely to be low as a result of the above degradation processes but no further studies have been done on this in Nova Scotia to date.

1.5 Problem and Objectives

At the Cochrane Hill Gold District there were two predominant stages of mining and milling for gold ore. The first stage lasted from 1877 to 1928 (Malcolm, 1929) while the second was from 1981 to 1988 (Northumberland Mines Ltd., 1981; Novagold Resources Inc., 1988). During the first stage stamp milling and mercury amalgamation were used to retrieve gold from the ore. The tailings from this process were deposited in a local stream. During the second stage ball milling and cyanidation were used to retrieve gold from the ore. The tailings from this process were deposited into an on-site impoundment (Malcolm, 1929; Northumberland Mines Ltd., 1981; Novagold Resources Inc., 1988). There has been little previous environmental work done at the site but when gold is extracted using either mercury amalgamation or cyanidation, mercury and/or cyanide can be lost and deposited with the tailings (Nriagu et al., 1997). Potentially toxic elements like arsenic and thallium can also occur naturally in the ore. Previous studies have shown that windblown tailings as well as the runoff from the tailings piles can have a significant effect on the surrounding environment at other abandoned gold mine sites (e.g. Murdoch and Clair, 1986).

The main objectives of this thesis are to:

1) characterize the mineralogy and metal concentrations of the two tailings piles;
2) assess the relative reactivity of metals and metalloids in the amalgamation versus cyanidation tailings;
3) examine downstream impacts of drainage from the tailings piles.

1.6 Thesis Organization

This thesis examines and compares the geochemistry of the amalgamation and cyanidation tailings piles at Cochrane Hill. This study focuses on the geochemistry of the tailings; therefore, only a brief description of the geological setting and history of the area is given. This project augments an ongoing Geological Survey of Canada study of gold districts across Nova Scotia being conducted as part of the Metals in the Environment (MITE) program.

Chapter 2 outlines the geological setting, character of the gold deposits, and the history of the mining, milling, and mine waste disposal at Cochrane Hill. Field and lab procedures along with the sampling methods are described in Chapter 3. Chapter 4 summarizes the field observations, and results of the chemistry and mineralogy of the tailings and the water chemistry. Chapter 5 compares the "new" and "old" tailings and discusses the impact of the tailings on the surrounding water chemistry. Chapter 6 discusses the conclusions that were made as a result of this study.

CHAPTER 2: THE COCHRANE HILL GOLD DISTRICT

2.1 Introduction

The Cochrane Hill Gold District is located approximately 15 km north of Sherbrooke in Guysborough County on the eastern shore of Nova Scotia (Figure 2.1). Gold was first discovered at Cochrane Hill in 1868, and there have been two main periods of mining operations. The first stage lasted from 1877 to 1928, when mercury amalgamation and stamp milling were used to recover the gold. During this period production was episodic and the mine stood idle for years at a time (Malcolm, 1929). Throughout the second stage, from 1981 to 1988, ball milling and cyanidation were used to extract gold from the ore. During this latter period production was steadier (Nova Scotia Department of Public Works and Mines; Annual Mines Reports for Nova Scotia). Currently, Scorpio Mining Corporation is exploring Cochrane Hill for a potential high-grade underground gold mine. Drilling by SCOMINEX and Novagold resources has recently identified some high-grade veins (www.scorpiomining.com)

The gold at Cochrane Hill is associated with quartz-carbonate veins that intrude the slates and greywackes of the Cambro-Ordovician Meguma Group bedrock. Throughout most of Nova Scotia the Meguma Group has been metamorphosed to greenschist facies, but in the Cochrane Hill area the metamorphic grade was increased to amphibolite facies as a result of regional metamorphism and the Sherbrooke Pluton to the south which was emplaced 369 +/- 2 Ma. Porphyroblasts in the host rock at Cochrane Hill include muscovite, biotite, garnet, staurolite, andalusite and arsenopyrite (Smith, 1982). The quartz veins are mostly concordant with the stratigraphy and are deformed. There are six types of quartz veins and five of these have gold associated with them.

2.2 Geological Setting

The Cochrane Hill Gold District is hosted by the Meguma Terrane of Cambro-Ordovician age. The Meguma Terrane is 10-14 km thick and is a deformed set of metagreywackes and slates. The metagreywackes were deposited over 500 million years

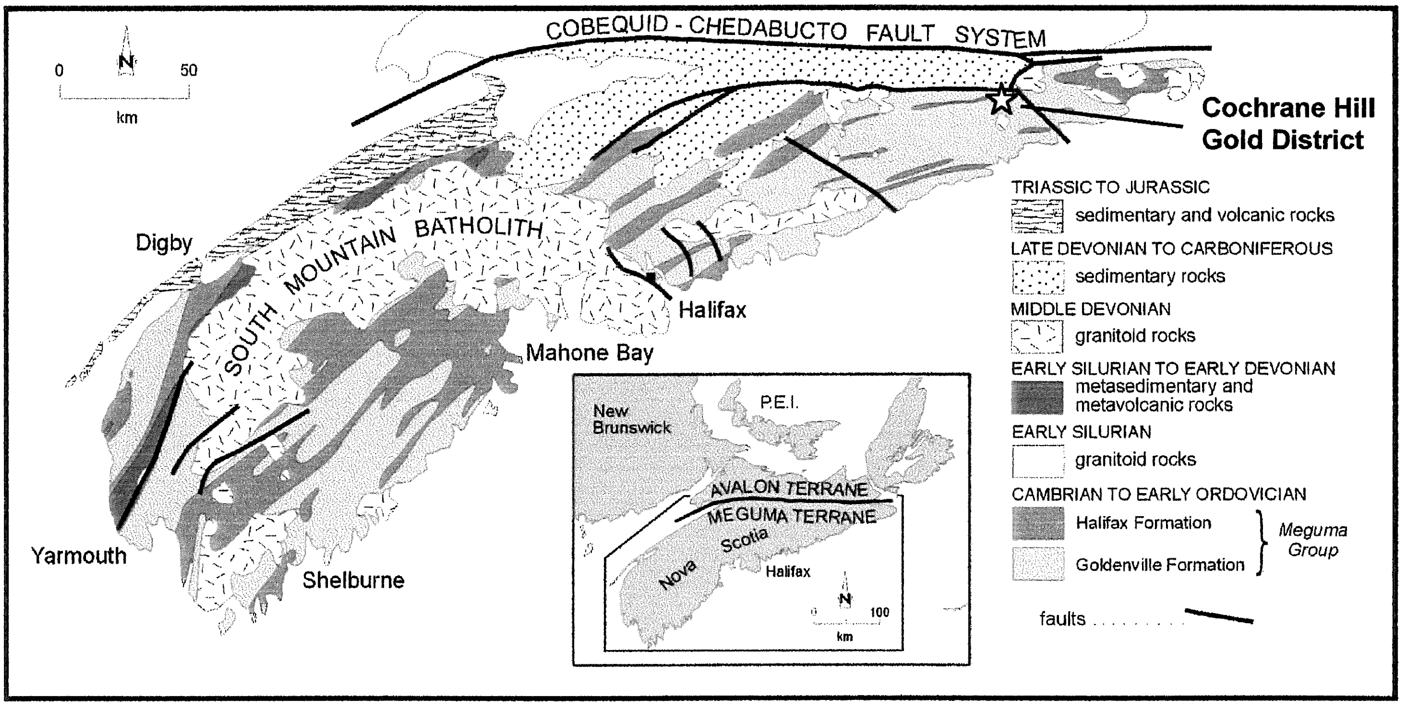


Figure 2.1 This map of Nova Scotia shows the location of the Cochrane Hill Gold District within the Meguma Group (base map courtesy of Nova Scotia Department of Natural Resources)

ago as turbidites and the slates were deposited as clays on an ancient continental rise and slope. These two units were deposited as a package together. When Pangea formed, 400 million years ago, Africa collided with North America along the Cobequid-Chedabucto Fault. This was called the Acadian orogeny and at this time the slates and metagreywackes were compressed into folds and metamorphosed to greenschist grade. At about 360-370 million years ago granite magmas intruded the metasediments in the root of the Appalachian Mountains (Schenk, 1970; Brooks et al., 1982).

The Meguma Group is divided into the lower Goldenville Formation and the upper Halifax Formation. The Goldenville Formation consists of metagreywacke and interbedded slate while the Halifax Formation consists of thinly laminated slate with small amounts of interbedded metasiltstone and metagreywacke. The boundary between these two formations is conformable and can be sharp or gradational (Schenk, 1970; Brooks et al., 1982). Gold in Nova Scotia is found associated with groups of quartz veins that are concordant to the stratigraphy in the Meguma Group. These veins are thought to result from early metamorphism and deformation (Graves and Zentilli, 1982).

Meguma Group rocks in Guysborough County are divided into northern and southern domains based on structural and metamorphic characteristics. In the northern domain where Cochrane Hill is located, the metamorphic grade has been raised to amphibolite facies as a result of nearby intruding granites. Figure 2.2 is a geological map of the area surrounding Cochrane Hill and shows the boundary between the northern and southern domains as well as the Sherbrooke pluton to the south of Cochrane Hill. In the southern domain the metamorphic facies is greenschist, which is similar to the rest of the Meguma Group in Nova Scotia. The primary mineralogy at Cochrane Hill includes quartz, muscovite, biotite, garnet, staurolite, andalusite and arsenopyrite. All of these minerals, with the exception of quartz, are also present as porphyroblasts (Smith, 1982).

Detailed mapping by Smith (1983) shows that there are two types of slates at Cochrane Hill. The more prominent slate is characterized by a high content of staurolite porphyroblasts, the presence of thin interbeds of calc-silicates, and a lack of quartz veins. Abundant sulphides, quartz veins, the general lack of calc-silicates and lack of visible gold characterize the less prominent slate. There are also two types of metagreywackes present that differ on the basis of colour and metamorphic alteration. The silicified and

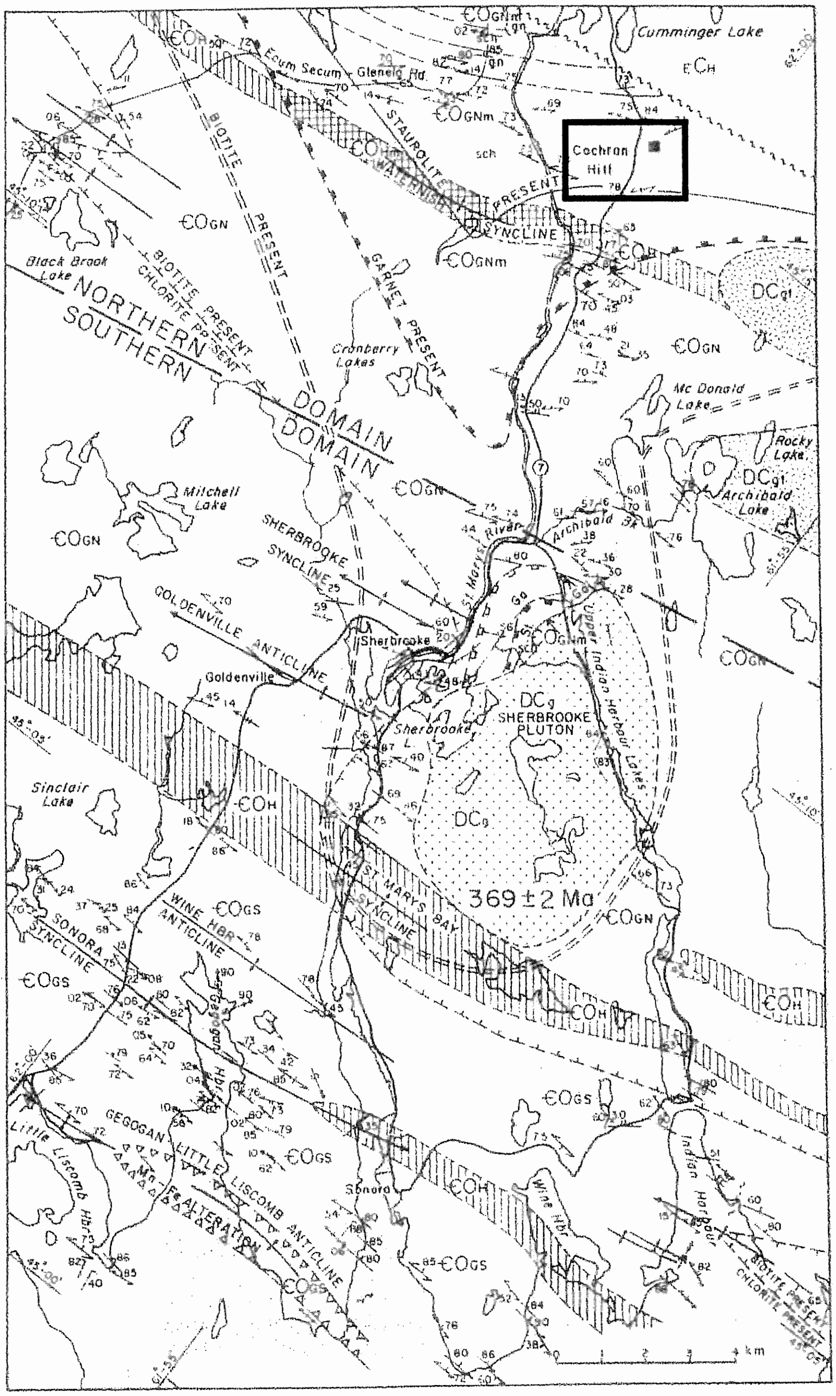


Figure 2.2 Geology map showing the location of Cochrane Hill in relation to the Sherbrooke Pluton and the boundary between the Northern and Southern domains (Smith, 1982).

altered metagreywacke is light green and contains numerous thin crosscutting stringers filled with chlorite and quartz. This grades into the unaltered metagreywacke that is medium to fine grained, blue grey, and contains quartz veins that lack the common crosscutting stringers. Arsenopyrite is the most abundant sulphide at Cochrane Hill and is found along the quartz veins in the alteration zone as well as in the host rock. Many of the arsenopyrite crystals show exsolution textures of pyrrhotite, sphalerite, galena, chalcopyrite, and loellingite; sphalerite has also been observed rimming arsenopyrite. The slates are strongly enriched in Ag, As, Au, B, Ca, Cd, Co, Mo, Pb, S, Sb, Sr, and Zn (Smith, 1983)

2.3 Character of the Gold Deposits

At Cochrane Hill, Smith (1983) distinguished six types of quartz-carbonate veins based on texture, mineralogy and relative age. The earliest veins (type A) are micro-crosslaminated and show low-angle crosscutting with the bedding. They were formed after sedimentation and before deformation. The veins also display a texture that results from later quartz emplacement. The second vein type (type B) is the most abundant at Cochrane Hill and frequently crosscuts biotite, garnet, and staurolite porphyroblasts. Other minerals commonly found along these veins are plagioclase, actinolite, apatite, sphene, rutile and other sulphide minerals. The third vein type (type C) is parallel to bedding and very similar to type B only smaller and less abundant. The fourth type (type D) is uncommon and crosscutting. Type E veins are pegmatoid veins characterized by the presence of andalusite, muscovite, biotite, plagioclase, sphene, apatite and rutile. Tourmaline is also present in alteration haloes in the wall rock around these veins. The sixth type (type F) are undeformed ac stringers that are unimportant volumetrically but do contain gold. Ac stringers are veins that are discordant and angular with respect to the bedding of the bedrock (Smith, 1983).

Visible gold occurs adjacent to type B and C veins. This gold varies from being platy and crenulated, or sandwiched between deformed biotite grains, or as irregular disseminations. Within type B veins, the gold occurs as irregular disseminations within the actinolite- and tremolite-rich section. In and adjacent to type C veins the gold can in some places protrude into the host rock. Gold also occurs as disseminations or inclusions in arsenopyrite porphyroblasts. It can be found as disseminations in the slates as well as inclusions in garnet and other silicate porphyroblasts. Gold has been found as stringers and disseminations that crosscut garnet porhyroblasts as well as in undeformed ac stringers that cut the beds at high angles (Smith, 1982).

Novagold Resources Incorporated describes the character of the gold deposit at Cochrane Hill in a report done in 1988. The gold mineralization is located within a steeply north-dipping vein zone up to 30 m wide which is parallel to the stratigraphy and occurs as a single vein or as a group of veins. The veins are often irregular and contain pieces and stringers of the host rock. The gold found within these veins is somewhat free of impurities (Novagold Resources Inc., 1988). Drilling by SCOMINEX and Novagold Resources show that there are high grade veins present at Cochrane Hill that have significant widths and are continuous along strike and down dip.

2.4 Mining and Milling History

Since gold was discovered at Cochrane Hill in 1868, mining and milling activity there has been episodic. This can be mainly attributed to fluctuations in the price of gold throughout the years. The following historical timeline of the mining and milling work at Cochrane Hill is based on information from the Nova Scotia Department of Public Works and Mines, the Annual Mines Reports for Nova Scotia, Malcolm (1929), Northumberland Mines Ltd. (1982), and Novagold Resources Inc. (1988).

1868 – Several lode gold deposits are discovered at Cochrane Hill

1869 – A shaft is sunk 100 ft deep, and a 15 stamp mill is built at Melrose 2 ½ miles northwest, and a small amount of prospecting is carried out.

1877 – 118 tons of quartz are mined and yield about 48 oz of gold when crushed at Melrose.

1879 – Cochrane Hill is formally proclaimed as an official gold district separate from Sherbrooke to the south.

1893-95 – The shafts at Cochrane Hill are deepened and extended, some trenching is completed, but otherwise the mine was idle. The California Gold Mining Company owned the mine and only worked it occasionally.

1897 – The Mitchell shaft is dewatered and 57 oz of gold are produced.

1900 – 1500 tons of ore are mined and crushed at Cochrane Hill and Crow's Nest

1901-06 – Extensive work and maintenance carried out on the shafts but the mine did not produce any gold.

1907 – 500 tons of quartz are crushed, yielding 113 oz of gold. Mining was done on the Mitchell belt by way of open cuts and underground mining. A 20 stamp mill and two Wifely tables were operated at Cochrane Hill.

1908-25 – Some buildings were built and maintenance was done on the mine as well as some prospecting but no gold was produced.

1926 – 139 oz of gold were produced at Cochrane Hill.

1927 – Novamac Mines and Power Corporation Limited erect a 10 stamp mill and deepen the Mitchell shaft by 100 ft. The mine yields 152 oz of gold from 962 tons of mined ore.

1928 – Last mention of Cochrane Hill in the Annual Mines reports for the province of Nova Scotia.

1877-1974 – 11,649 tons of ore were mined yeilding 1192 oz of gold from the Cochrane Hill Gold Mine.

1933-1980 – Episodic sampling, exploration and drilling carried out at the mine site.

1981 – An open pit is excavated and 13,500 tons of ore are mined and processed using cyanidation. About 550 oz of gold are produced by Novagold resources.

1982 – A 100 ton heap leach test is done on ore from the open pit. The ore has a head grade of 0.057 oz of gold/ton.

1985-88 – Northumberland completes work on the Mitchell shaft, and conducts mapping and exploration activities. This is the most recent time any mining was done at the site.

2.5 Distribution of Mine Wastes

During the first stage of operation (1877-1928) at Cochrane Hill stamp milling and mercury amalgamation were used to process the ore. The tailings were then deposited in a nearby stream (Fig. 2.3). This stream begins on site as drainage from the tailings area and eventually passes through a few lakes before entering the St. Mary's River. This area is now overgrown with small trees and shrubs and is waterlogged; therefore, it is difficult to determine the spatial extent of the older tailings pile and the amount of material that has been transported downstream.

During the second stage of operation (1981-1988) ball milling and cyanidation were used to process the ore for gold. The tailings were deposited in an unlined impoundment adjacent to the mill building. There is a dam of rock and gravel built on the west side of the impoundment to keep the tailings from travelling down the drainage at the site. This younger cyanidation tailings pile is located directly west of the older amalgamation tailings and is not overgrown with vegetation. The approximate size of this more recent impoundment is 10,000 m^2 (Fig. 2.3).

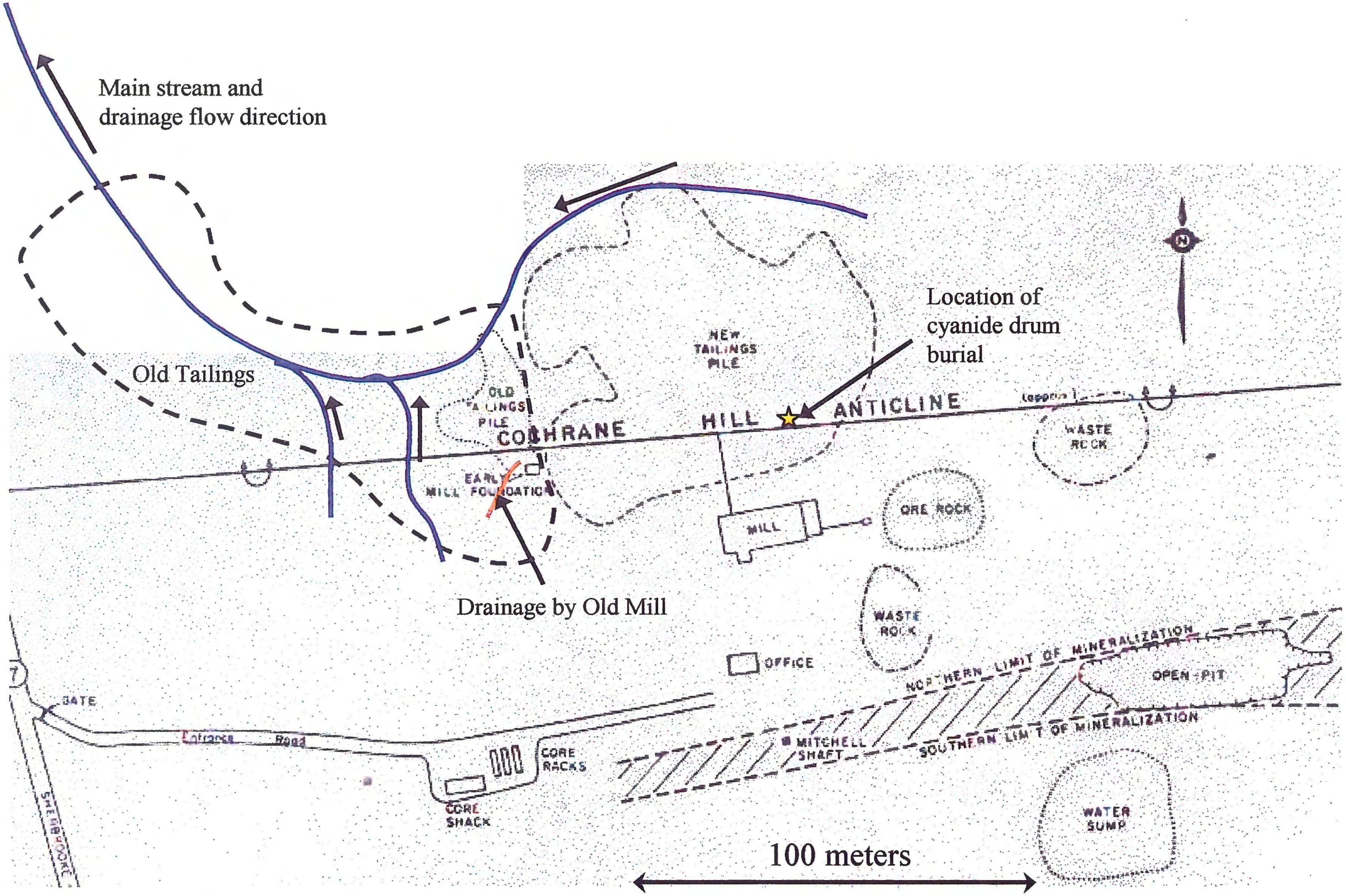


Figure 2.3 Map showing the distribution of the old and new tailings with respect to one another (Base map provided by Paul Smith, Nova Scotia Department of Natural Resources).

CHAPTER 3: METHODOLOGY

3.1 Introduction

Detailed field and laboratory studies were carried out to assess the distribution, composition, mineralogy, and downstream impacts of historical gold mine tailings at Cochrane Hill. Results from previous geological research that had been done in the area surrounding Cochrane Hill were used to help determine what areas needed to be sampled. The field and laboratory methods employed during this study were based on previous work done by Dr. Michael Parsons at other gold mine sites in Nova Scotia. Care was taken in the field and the lab to avoid potential contamination and to ensure accurate results. A detailed description of the methods used follows in this chapter.

3.2 Field Measurements and Sampling Methods

Sampling of tailings and water at Cochrane Hill took place on two different occasions. On September 24 and 25, 2003, sampling of the tailings was done at 16 sites in total, distributed evenly geographically over the old and new tailings piles. The samples were taken from holes dug in the tailings with a shovel; each hole was 30-40 cm in diameter and depth ranged from 10-85 cm. Within most holes, the tailings displayed a well-developed vertical stratigraphy, with reddish-brown oxidized tailings overlying grey unoxidized tailings. In waterlogged areas, the tailings were generally unoxidized, and dark grey throughout. In the cyanidation tailings, alternating bands of sandy tailings and clay rich tailings were evident in some holes. The number of samples taken from each site reflects the amount of variation seen within the hole.

Figure 3.1 shows a detailed map of the tailings area showing the locations where tailings samples were taken. A small stream runs from a pond near the northeast corner of the tailings impoundment, west along the north edge of the impoundment, through the old tailings and then turns northward. Water was sampled at 12 sites along this stream on November 4 and 5, 2003. The first, most distant site was 2 km downstream of the mine, approximately 200 m north of the Melrose-Country Harbour Road, and the subsequent

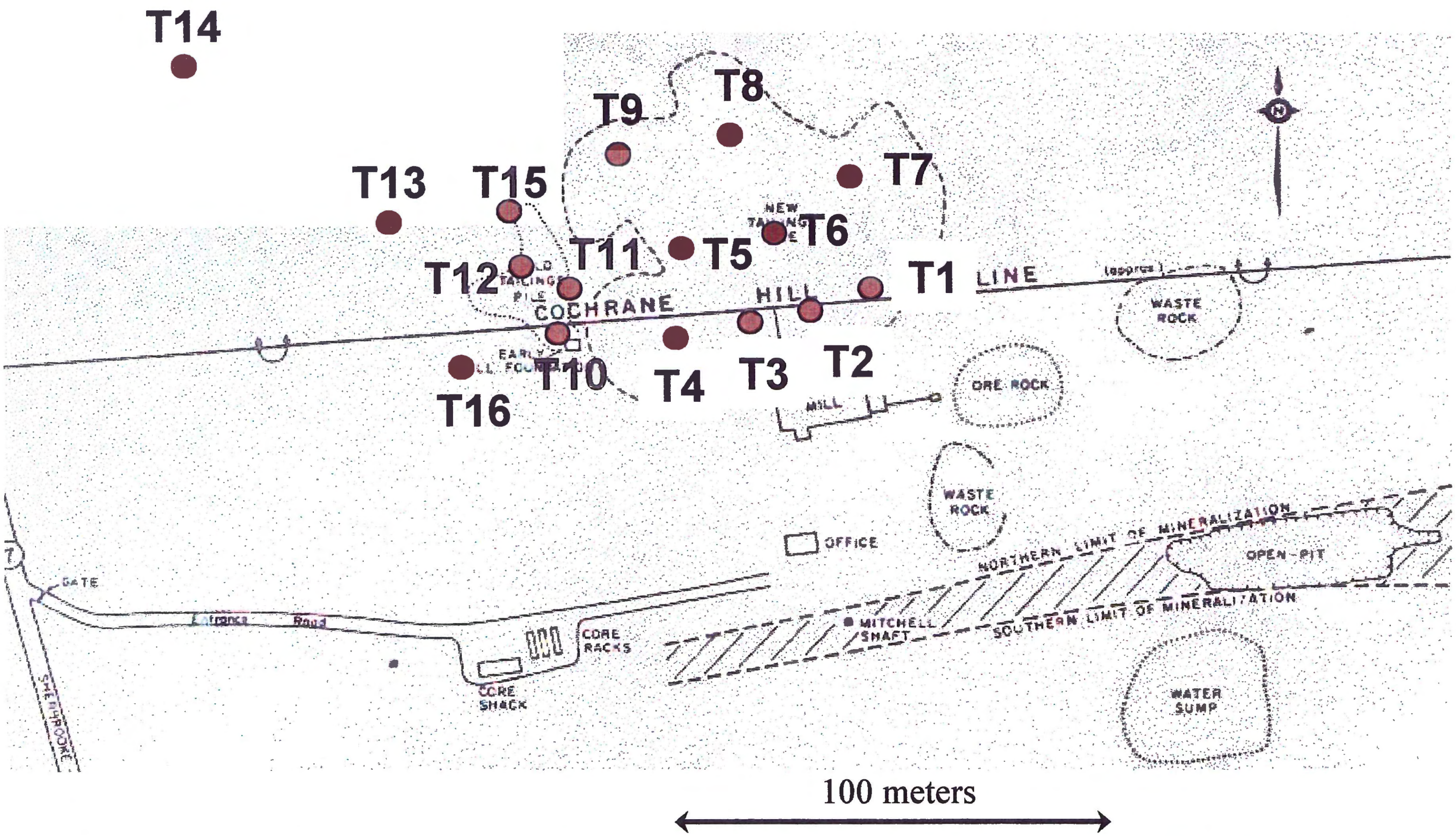


Figure 3.1 Map showing the locations where the tailings were sampled, with the appropriate sample numbers (Base map provided by Paul Smith, Nova Scotia Department of Natural Resources).

samples were taken in an upstream direction towards the tailings to minimize possible contamination of the downstream water samples. Two of these sites were holes dug in the tailings that were allowed to fill with water for approximately 24 hours. The samples taken at these two sites were taken to represent the pore waters of the tailings. At the sixth site (W6), a duplicate sample was taken to evaluate the precision of the sampling and analytical measurements. Figure 3.2 shows the water sample locations.

When sampling for water, care must be taken not to contaminate the sample with any foreign material. Nitrile gloves were used when handling the sampling equipment, and the acid-washed 60 mL containers were rinsed on-site using the water to be sampled. A new 0.45 μm filter (Sterivax capsule filter) or set of filters would also be used at each site reducing the risk of contamination. At each site the pH, temperature and conductivity of the water were measured. The pH meter was calibrated using standard pH buffer solutions at pH 4, 7, and 10. Five 60 mL samples were collected at each site: filtered (0.45 μm) and unfiltered waters for cation and mercury analyses, and one filtered sample for anion determination. At three of the sites, unfiltered samples for cyanide analysis were collected. The samples for cyanide were preserved by adding sodium hydroxide, which was done on site. Bromine monochloride (2 vol.%) was used to preserve the samples for mercury analysis, and ultrapure nitric acid (1 vol. %) was used to preserve the samples for analysis of cations. All preservation was done within 24 hours of sample collection, and samples were stored at 4°C prior to analysis. The water samples collected for cyanide analysis were submitted to Philip Analytical Services in Bedford, Nova Scotia; all other water analyses were carried out at the Analytical Method Development labs at GSC-Ottawa.

3.3 Laboratory Analyses

All tailings samples collected during this study were analyzed for their bulk chemical composition, and the mineralogy and chemistry of selected samples were characterized using X-ray diffraction (XRD), electron microprobe, and energy dispersive spectrometry (EDS) analyses. Water samples from all sites were analyzed for cations, anions, Hg, dissolved organic carbon, and alkalinity. The mineralogical analyses of the tailings and

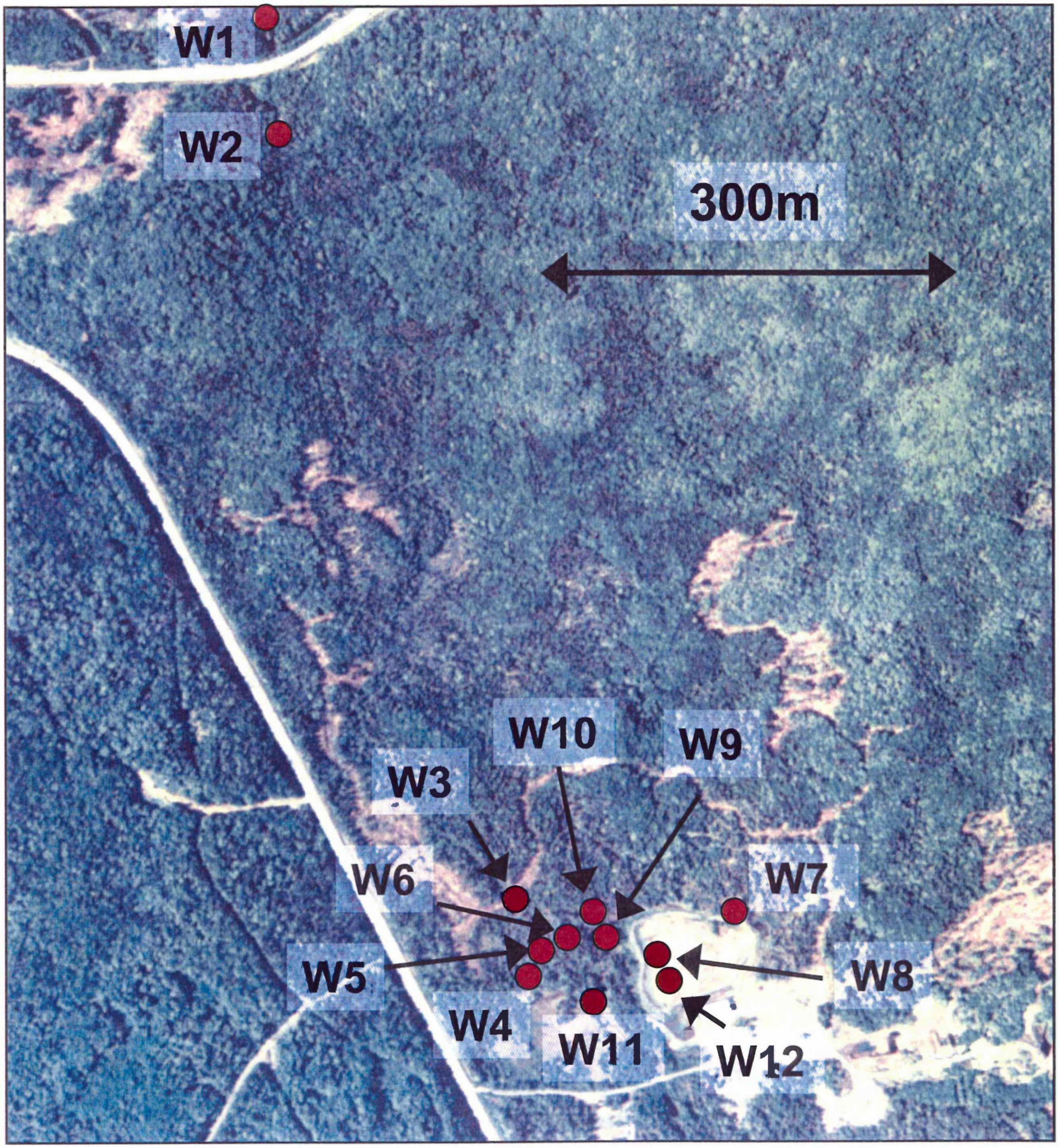


Figure 3.2 This photo shows the locations where water was sampled, with the corresponding sample numbers (Airphoto 11 E/8 © 1973 Her Majesty the Queen in Right of Canada Reproduced Courtesy of Land Information Services, 2003).

precipitates as well as the carbon analyses were completed by the author at the Geological Survey of Canada (Atlantic). The rest of the chemical analyses were done externally at ACME Analytical Laboratories in Vancouver, British Columbia and the Analytical Method Development labs at GSC-Ottawa. These results are used in this thesis to make conclusions about the tailings geochemistry and associated environmental effects.

3.3.1 Bulk tailings composition

Upon returning to GSC-Atlantic, the tailings samples were all sub-sampled and freeze dried, and 5 g of each sample was sent to ACME Analytical Laboratories in Vancouver, British Columbia for bulk chemical analysis. All samples were digested in a 2:2:2 mixture of nitric acid : hydrochloric acid : water at 95°C for 1 hour, then analyzed for 37 elements using inductively coupled plasma – mass spectrometry (ICP–MS). This strong acid digestion is capable of decomposing metal salts, carbonates, sulfides, most sulfates and some oxides and silicates (Hall, 1997). Duplicate samples and certified reference materials (CANMET STSDs 1-4) were submitted with the unknown samples to monitor analytical precision and accuracy.

3.3.2 Tailings mineralogy

X-ray diffraction (XRD) analysis was carried out on tailings samples from five sites, three of which are from the new tailings pile and the other two from the old tailings pile. These samples were chosen because they best represent the vertical stratigraphy observed in each tailings pile. This analysis was done to identify the main minerals present in the tailings. Minerals present below 1-2% are undetectable by bulk XRD; therefore, trace quantities of primary and secondary phases could not be identified using this method.

Two sample sites were selected for more detailed mineralogical analysis using the electron microprobe at Dalhousie University. One site is from the old tailings pile and the other is from the younger tailings pile. This was done to clarify the compositions of sulfide minerals and any weathering products, and also to compare the results from each tailings deposit. Grain mounts were produced for each sample within each hole. Mineral separation using sodium polytungstate with a density of about 2.84 g/cm^3 was completed on a composite sample from each site. Grain mounts were then made of the heavy and light mineral separates that were produced during separation. The grain mounts were prepared using glass slides, epoxy and a teflon surface. The mounts were polished using 600 grit silicon, carbide tin oxide and polycrystalline diamond. Polishing was done by hand and by using a Planopol-V polisher with a PdM-Force semi-automatic sample rotator. In preparation for microprobe analysis the grain mounts were examined using a reflected-light microscope to locate sulfide grains. A scanning electron microscope was also used to examine the two grain mounts of heavy separates more closely.

X-ray diffraction analysis was carried out on the heavy and light mineral separates from the composite samples, as well as a milky precipitate found on the younger tailings pile and an orange precipitate found in a stream near the old stamp mill. Scanning electron microscope (SEM) analysis was also done on these precipitates, along with the white and blue efflorescent material found on the cyanidation tailings. After initial SEM analysis of the blue material failed to clarify its identity, an effort was made to concentrate it by using a binocular microscope and tweezers.

3.3.3 Water chemistry

Five 60 mL samples were collected at each site: filtered (0.45 μm) and unfiltered waters for cation and mercury analyses, and one filtered sample for anion determination. All water analyses except for cyanide determinations were carried out by the Analytical Method Development labs at GSC-Ottawa. The concentration of major elements in the water samples was determined using inductively coupled plasma atomic emission spectrometry (ICP–AES), and all minor and trace elements were measured using ICP-ES/MS. Concentrations of Hg less than 10 ng/L (parts-per-trillion) were quantified using a Tekran Hg analyzer (detection limit, 0.5 ng/L). Anion concentrations were determined using a Dionex ion chromatograph, and dissolved organic carbon was measured using a high-temperature combustion method following removal of inorganic carbon using phosphoric acid. The alkalinity of all water samples was determined using automated

Gran titrations. Analysis of total cyanide concentrations was carried out by Philip Analytical Services (Bedford, Nova Scotia) using automated colourimetry following strong acid digestion and distillation (detection limit, 1 µg/L). Analyses of field blanks and blind duplicate samples by all methods suggest minimal levels of contamination during sampling, and indicate that the precision of most analyses is better than 5%.

CHAPTER 4: RESULTS

4.1 Introduction

In order to fully understand the geochemistry of the tailings and the water at Cochrane Hill the results from observations made in the field, mineralogical and chemical analysis of the tailings and chemical analysis of the water must be compiled. In this chapter, the field observations and analytical results are summarized with minimal interpretation; these results are discussed in detail in the Chapter 5. The field observations help give a better understanding of what interactions are taking place on a large scale. These observations can then be used to determine where it is important to examine the interactions on a smaller scale. The observations made at Cochrane Hill helped to identify the distribution of the mine wastes; this information was used to determine the most appropriate locations for detailed sampling of tailings, weathering products, and water. In the following sections, results from analysis of the tailings and water are described separately but their interaction will be discussed in Chapter 5.

4.2 Field Observations

During the first visit to the Cochrane Hill site, Paul Smith from the Nova Scotia Department of Natural Resources pointed out the old and new mill structures (Fig 4.1) the different tailings piles, and other points of interest. The main visible difference between the two tailings deposits is the lack of vegetation on the cyanidation tailings as compared to the small trees and brush growing on the older amalgamation tailings. There is a stream that runs from a pond at the northeast corner of the cyanidation tailings, westward along the north edge of the impoundment, through the amalgamation tailings, then northward as seen in Figure 2.3. On the younger tailings there are a couple of ponds located on the west side of the impoundment, and a pile of mixed tailings and waste rock in the middle. In the southeast corner of the younger tailings where some cyanide drums were apparently buried, there is moss growing on the surface which may reflect addition of lime over the buried drums. Immediately downhill of this location, blue efflorescent

Figure 4.1a This picture shows what the old stamp mill structure looks like today (Picture taken by M. Parsons, 2003).

Figure 4.1b This picture shows the new mill structure while it was in use during the 1980s. Tailings are actively being discharged near the right side of this image (photo courtesy of Nova Scotia Department of Natural Resources).

salts are present on the tailings surface. Figures 4.2a, b show an overview of the younger tailings pile as well as the location of buried cyanide drums and the associated blue efflorescent salt. Elsewhere on the younger tailings there is white efflorescent material present in patches on the surface. In November 2003, a white precipitate was visible in a small rivulet on the north side of the cyanidation tailings. On the edge of the ponds seen on the west side there are bubbles forming in the tailings; when these are cut into a distinct odour of hydrogen sulfide is released. Figures 4.3a, b show one of these bubbles as well as the small rivulet containing the white precipitate.

The full spatial extent of the amalgamation tailings is difficult to ascertain because of the abundant vegetation growing on the surface. Timbers and stone walls provide some evidence of the old stamp mill on the southeast corner of the old tailings pile (Fig. 4.1a). The drainage mentioned above runs through the northern edge of the old tailings, and tailings are visible in the stream banks for at least 500 m downstream of the mill site. In September, there was a fluffy orange material present in the water amongst the tailings; however, in early November this material was no longer present suggesting that it may have been biological in origin. While sampling water at Cochrane Hill, another ephemeral drainage was located near the old mill structure which contained a red-orange precipitate (Fig. 4.4). An additional area of old tailings was also found to the east of the mill structure which expands the known area covered by old tailings. These tailings were sampled from a hole dug at one site within that area (Site T15, Fig. 3.1).

Figures 4.5a, b compare the stratigraphy that was seen in the holes dug in the old and younger tailings. In the younger tailings the stratigraphy is more varied than in the older tailings. The stratigraphy in the younger tailings is defined by grain size as well as color differences from red-brown to grey. In both tailings it is apparent that there is a red-orange, oxidized layer above a grey, unoxidized layer. In some of the holes dug in the older tailings there was very little of the oxidized layer above the unoxidized layer.

4.3 Chemistry and Mineralogy of Tailings

Results from the chemical analysis of the tailings using ICP – ES/MS reveal concentrations of mercury ranging from <5 to 63,200 ppb and arsenic from 275 to

Figure 4.2a This overview picture of the younger tailings shows the moss growing on the surface where the cyanide drums and lime were buried with the blue efflorescent salt in the distance (Picture taken by M. Parsons, 2003).

Figure 4.2b This picture shows the blue efflorescent salt on the surface of the new tailings (Picture taken by M. Parsons, 2003).

Figure 4.3a Picture of the rivulet on the cyanidation tailings that contains the white precipitate (Picture taken by M. Parsons, 2003).

Figure 4.3b Picture of the bubbles forming in the cyanidation tailings on the edge of the ponds (Picture taken by M. Parsons, 2003).

Figure 4.4 Picture of the drainage near the old mill structure, containing the red-orange precipitate (Picture taken by M. Parsons, 2003).

Figure 4.5a Picture showing the vertical stratigraphy within the old tailings (Picture taken by M. Parsons, 2003).

Figure 4.5b Picture showing the vertical stratigraphy within the younger tailings (Picture taken by M. Parsons, 2003).

178,200 ppm. Results for some of the other elements analysed are summarized in Table 4.1. Figures 4.6 and 4.7 show the spatial distribution of depth-averaged mercury and arsenic concentrations from all sample sites.

The primary minerals within the tailings can be determined from analysis of the XRD results. The minerals that were identified in both the old and new tailings are muscovite, staurolite, biotite, quartz and anorthite. Each of these minerals was present in all samples that were analysed, and the proportions of each phase were very similar in all samples. The peak heights got higher for samples taken deeper in the holes, but that was the only significant change. For sample T3 the peak heights were higher in the middle of the hole than the top and the bottom. The X-ray diffraction results for the heavy and light mineral separates indicated there may also be other minor phases present, including tremolite / actinolite in the heavy separate. Further examination of the mineralogy using a scanning electron microscope and qualitative electron microprobe analyses revealed arsenopyrite, pyrite, pyrrhotite and small amounts of monazite. A summary of the quantitative microprobe results is provided in Table 4.2. Chemical maps of some sulphide grains are shown in Figures 4.8a, b. The maps show the distribution of antimony, mercury, arsenic, zinc and iron. The map of arsenic shows increased concentration in the center of the grains and a rind of decreased concentration around the outside. There are also small concentrations of zinc associated with the arsenic. The mercury and antimony maps show that there is little to no mercury or antimony associated with the grains.

Energy dispersive spectrometry analysis was carried out on all secondary minerals collected from the old and new tailings. The orange-red precipitate in a drainage near the old mill was determined to be arsenic-bearing ferric hydroxide (Fig. 4.9a). The white, milky precipitate collected from the rivulet on the cyanidation tailings was determined to be basaluminite [$Al_4(SO_4)(OH)_{10}\cdot 5H_2O$] with traces of calcium and iron (Fig. 4.9b). The X-ray diffraction spectra for the ferric hydroxide and basaluminite suggests that these phases are amorphous. The white salt present on the younger tailings was shown to be gypsum (Fig. 4.10a). The blue salt present near the buried cyanide drums was difficult to isolate. It is present as a thin coating on quartz and muscovite grains and is very delicate. Results from EDS analyses indicate high levels of C and Fe (Fig. 4.10b), but the high concentrations of Si, O, and Ca seem to be derived from the underlying grain. High levels

Table 4.1

Depth-averaged concentrations of selected elements analysed in the tailings using ICP-ES/MS

	Element	Ag ppb	As ppm	Au ppb	Hg ppb	Mn ppm	Ni ppm	Pb ppm	Sb ppm	Zn ppm
Younger Tailings	T1	64	8960	73	7.5	846	33	339	4.2	698
	T2	61	3630	187	5.5	315	27	75	1.9	206
	T3	43	1210	82	7.7	302	32	55	0.76	209
	T4	78	3840	191	14.7	440	37	177	2.0	327
	T5	60	4130	140	10.3	387	34	145	2.3	376
	T6	55	1580	117	13.5	452	32	87	0.85	220
	T7	85	5710	1091	6.3	293	18	94	3.3	239
	T8	34	2260	71	5.0	379	15	44	1.1	215
	T9	72	4330	173	8.5	441	20	158	2.1	284
Older Tailings	T10	3540	70330	20530	61700	308	51	243	97	111
	T11	195	8340	1130	7850	496	11	105	3.9	179
	T12	137	16500	542	9540	1150	6.8	95	2.0	75
	T13	82	4470	146	14.0	812	26	135	2.7	405
	T14	349	5620	2300	11700	398	17	184	4.7	266
	T15	122	4860	391	1690	478	20	174	2.7	281
	T16	180	2770	761	9040	847	15	57	1.2	150

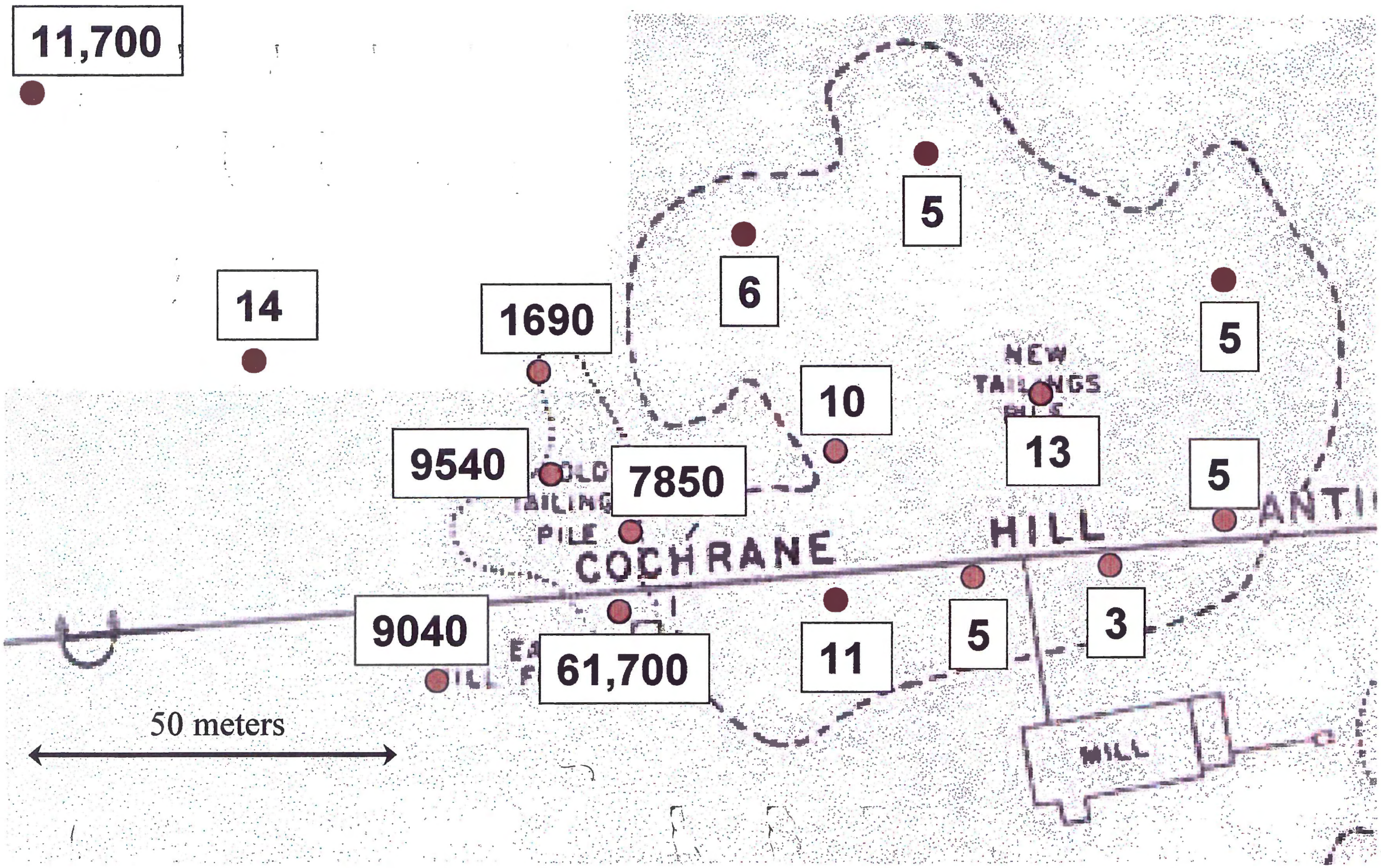


Figure 4.6 The distribution of depth-averaged mercury concentration (ppb) for all sample sites (Base map provided by Paul Smith, Nova Scotia Department of Natural Resources).

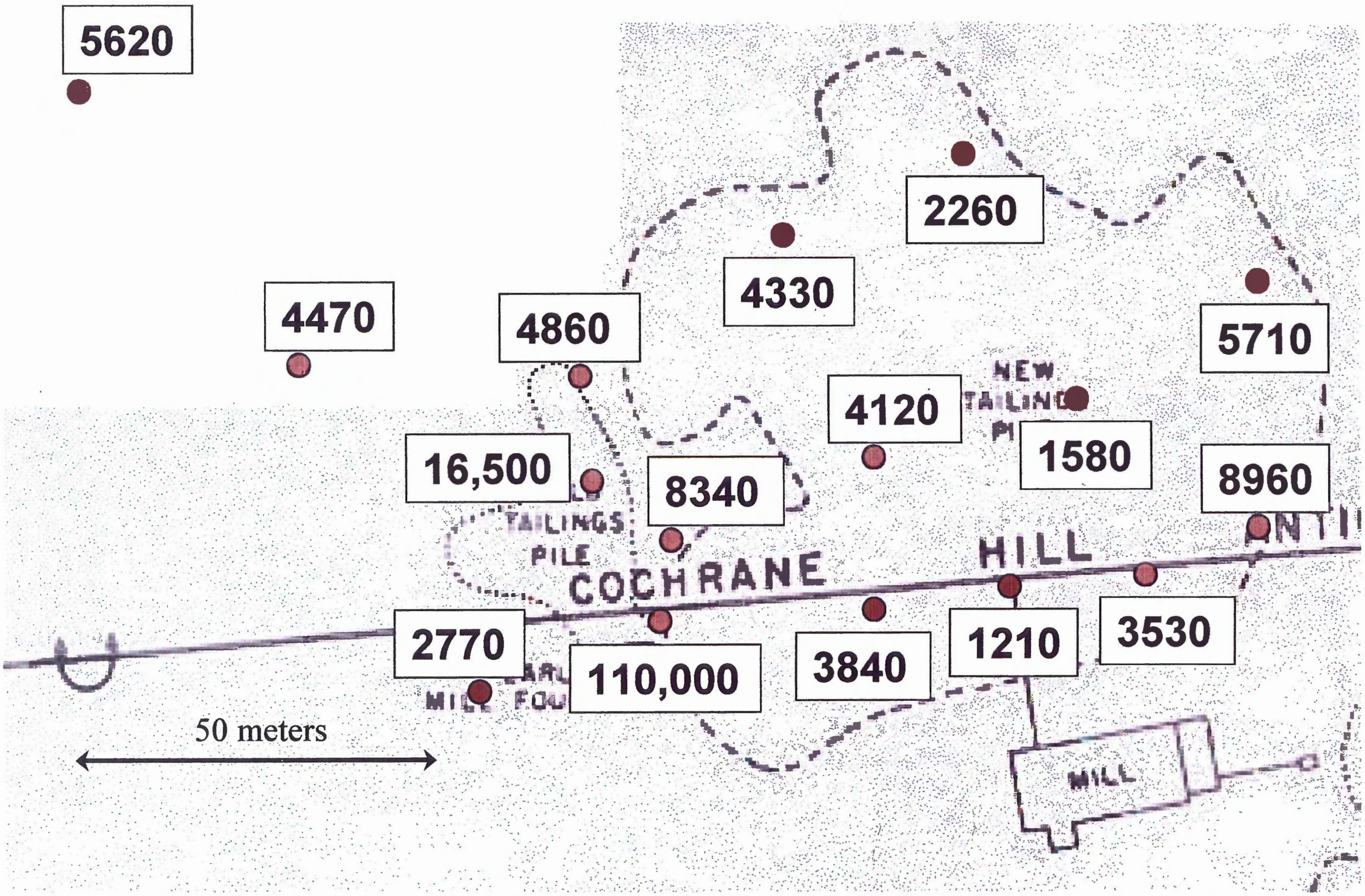


Figure 4.7 The distribution of depth-averaged arsenic concentration (ppm) for all sample sites (Base map provided by Paul Smith, Nova Scotia Department of Natural Resources).

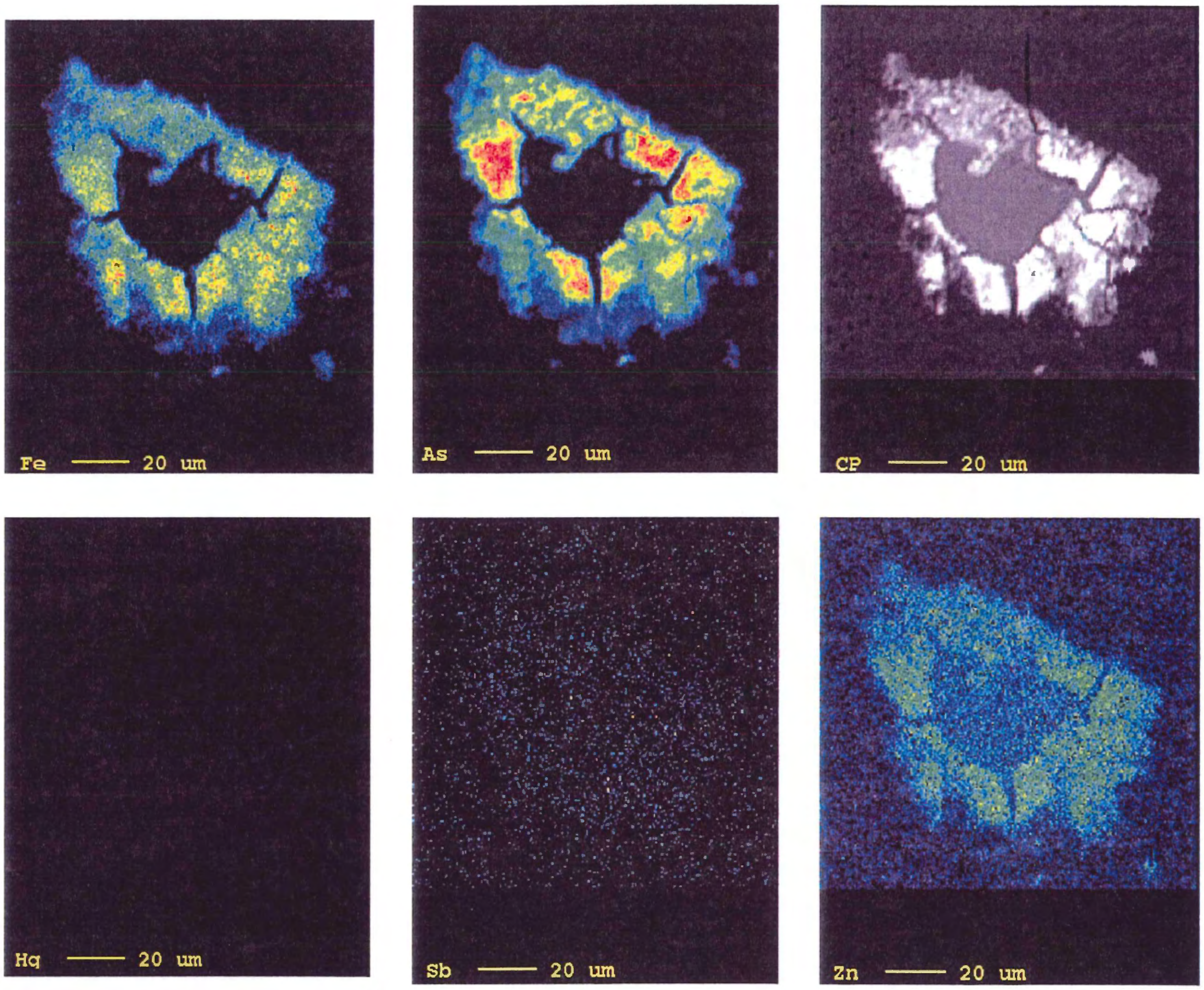
Table 4.2

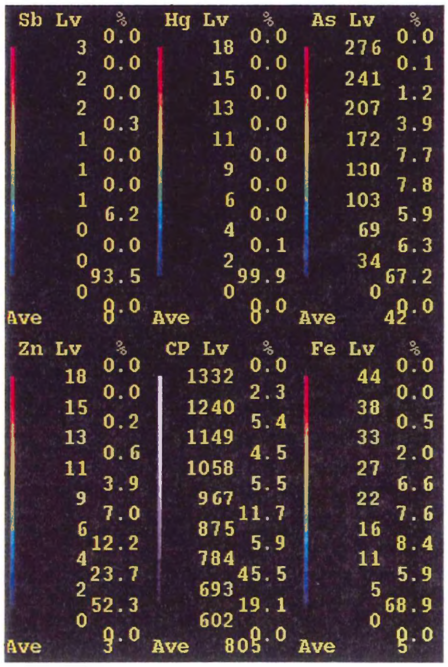
Summary of quantitative microprobe results (Mass %)

Element	Arsenopyrite (n=37)[a]			Pyrite (n=31)[a]		
	Min	Max	Mean	Min	Max	Mean
S	16.5	21.1	18.8	38.6	55.8	47.2
Fe	32.7	35.6	34.2	45.0	62.0	53.5
Zn[b]	0	2.3	1.2	0	1.8	0.90
As[b]	43.6	53.4	48.5	0	0.35	0.17
Ag[b]	0	0.15	0.076	0	0.0039	0.002
Au[b]	0	0.13	0.063	0	0.13	0.064
Cu[b]	0	6.6	3.3	0	2.1	1.0
Cd[b]	0	0.14	0.071	0	0.015	0.0074
Co[b]	0	1.4	0.70	0	0.90	0.45
Sb[b]	0	0.25	0.13	0	0.0008	0.0004
Ni[b]	0	1.3	0.63	0	0.83	0.41
Hg[b]	0	0.013	0.006	0	0	0
Pb[b]	0	0.086	0.043	0	0.081	0.041

[a] n = number of analyses

b All results of 0 are below the detection limits for that element: Zn 702, As 210, Ag 187, Au 657, Cu 569, Cd 166, Co 325, Sb 206, Ni 459, Hg 417, Pb 368





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Sb Lv     %   Hg Lv     %   As Lv     %
         0.0           0.0           0.0
     3             18           276
         0.0           0.0           0.1
     2             15           241
         0.0           0.0           1.2
     2             13           207
         0.3           0.0           3.9
     1             11           172
         0.0           0.0           7.7
     1              9           130
         0.0           0.0           7.8
     1              6           103
         6.2           0.0           5.9
     0              4            69
         0.0           0.1           6.3
     0              2            34
        93.5          99.9          67.2
     0              0             0
         0.0           0.0           0.0
Ave  0        Ave  0        Ave  42
Zn Lv     %   CP Lv     %   Fe Lv     %
         0.0           0.0           0.0
    18           1332            44
         0.0           2.3           0.0
    15           1240            38
         0.2           5.4           0.5
    13           1149            33
         0.6           4.5           2.0
    11           1058            27
         3.9           5.5           6.6
     9            967            22
         7.0          11.7           7.6
     6            875            16
        12.2           5.9           8.4
     4            784            11
        23.7          45.5           5.9
     2            693             5
        52.3          19.1          68.9
     0            602             0
         0.0           0.0           0.0
Ave  3        Ave 805       Ave  5
```

Figure 4.8a Chemical maps for an arsenopyrite grain from the amalgamation tailings. From sample T11. The different colors represent the number of counts for that particular element at that spot.

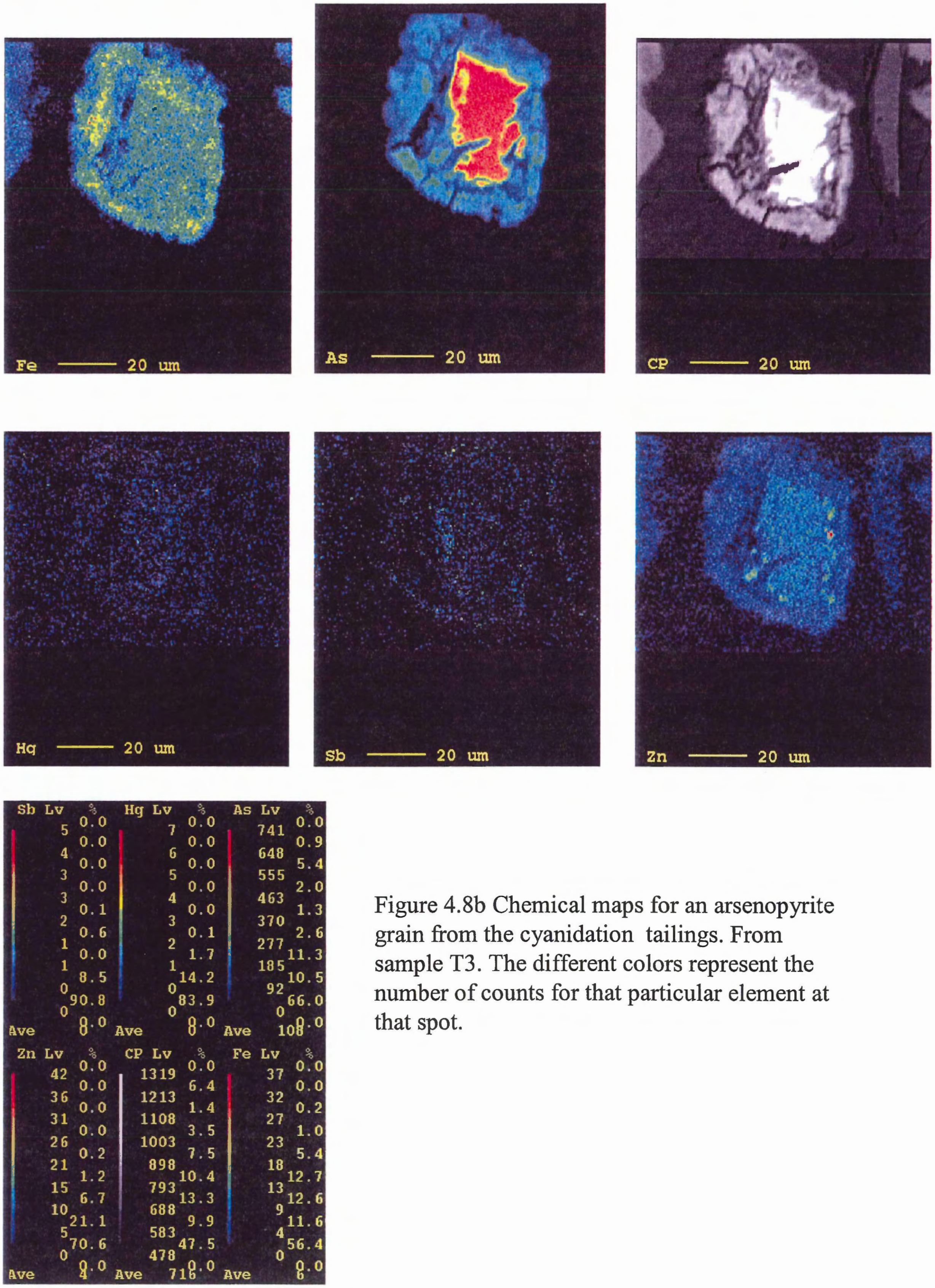


Figure 4.8b Chemical maps for an arsenopyrite grain from the cyanidation tailings. From sample T3. The different colors represent the number of counts for that particular element at that spot.

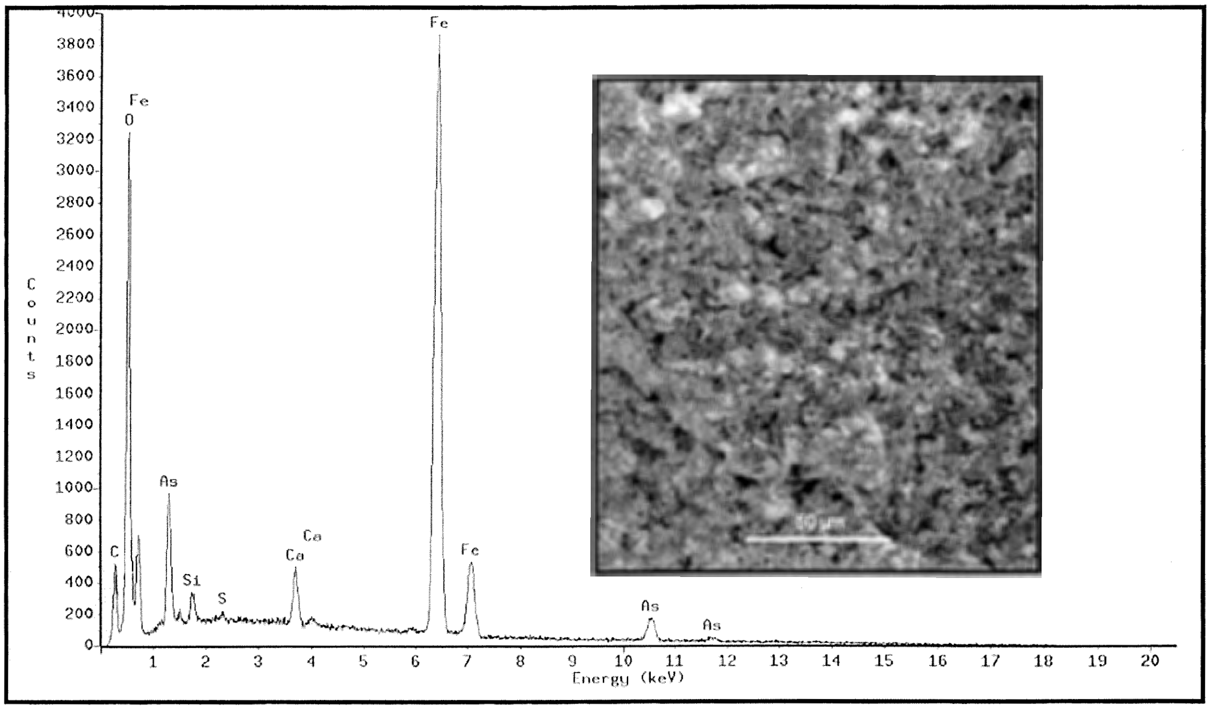


Figure 4.9a EDS spectra and SEM picture of the As-bearing ferric hydroxide $Fe(OH)_3$ in the drainage near the old mill structure.

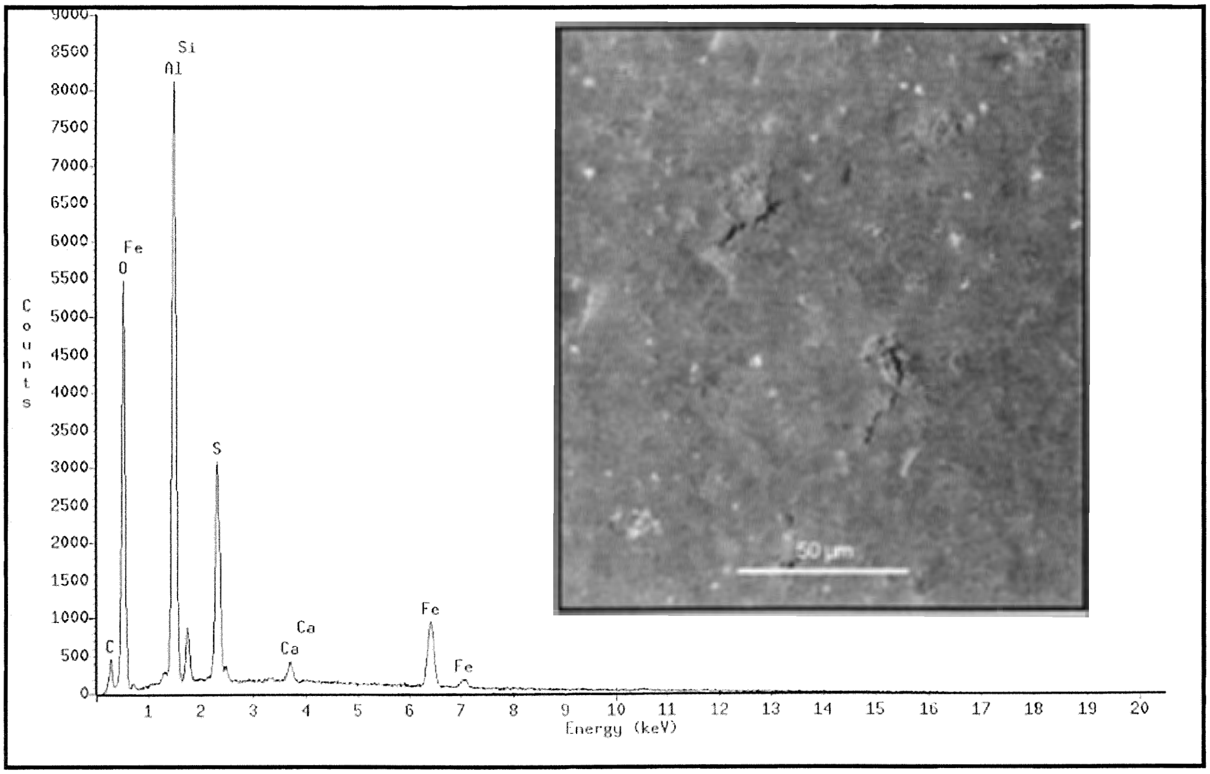


Figure 4.9b EDS spectra and SEM picture of the basaluminite ($Al_4(SO_4)(OH)_{10}{\cdot}5H_2O$) collected from the rivulet on the cyanidation tailings.

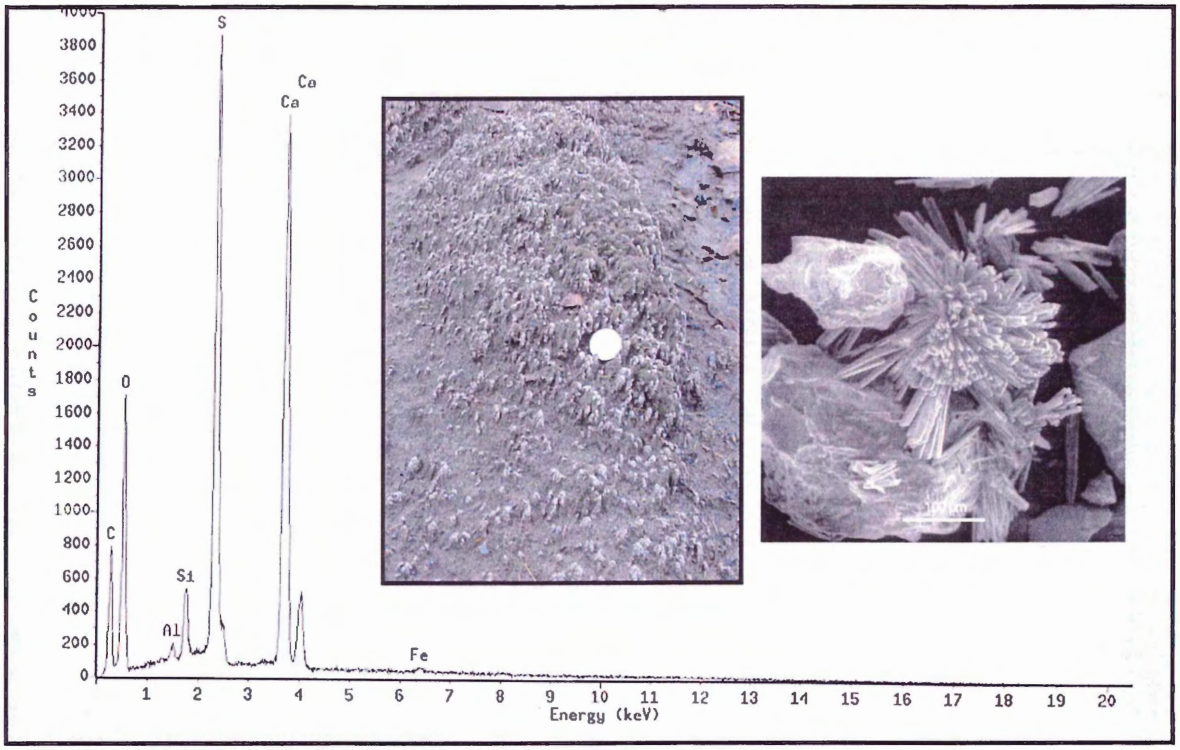


Figure 4.10a EDS and SEM spectra for the gypsum $CaSO_4 \cdot 2H_2O$ collected from the surface of the cyanidation tailings.

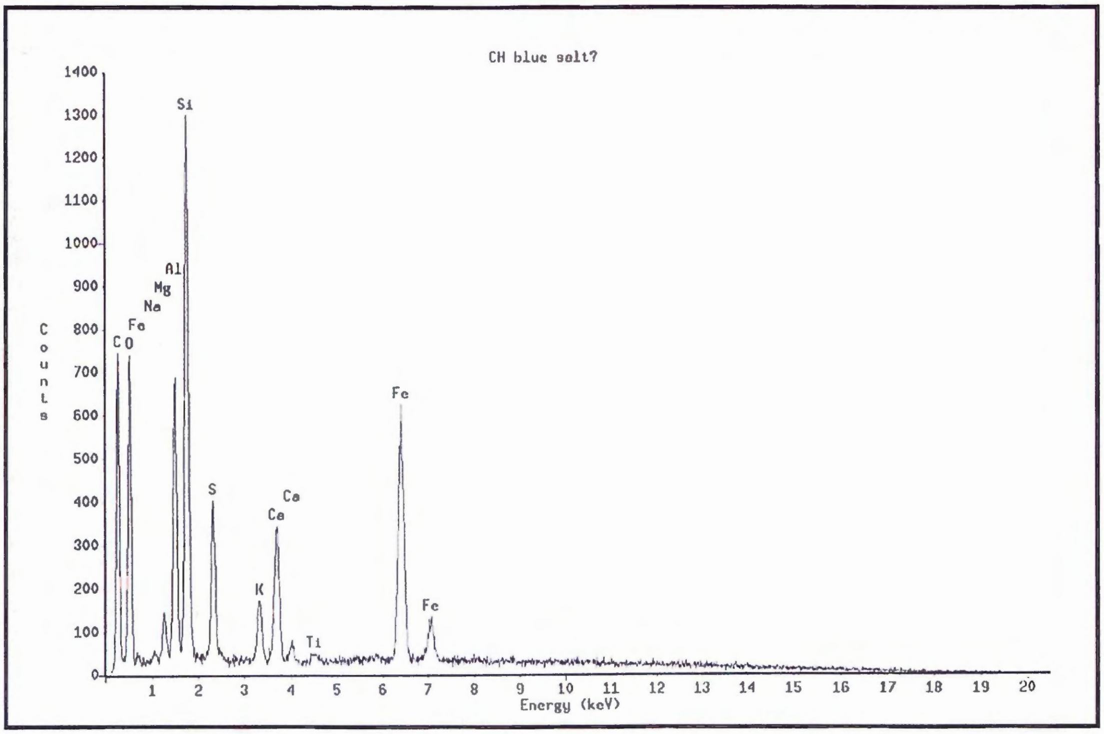


Figure 4.10b EDS spectra for the Turnbull's Blue, $Fe_3(Fe(CN)_6)_2$ collected from the surface of the cyanidation tailings. Elements present other than Fe, and C are results from the grain below the Turnbull's Blue which coats the mineral.

of C and Fe may indicate that the coating is Turnbull's Blue [$Fe_3(Fe(CN)_6)_2$], but unequivocal identification would require additional study.

4.4 Water Chemistry

Unlike the very acidic drainage associated with runoff from Meguma Group slates in other areas of Nova Scotia (e.g. King and Hart, 1990), the pH of the waters near the Cochrane Hill mine site ranged from 3.5 to 6 during the sampling in November 2003. The most acidic pH values were measured in standing waters on the surface of the cyanidation tailings. Most pH values ranged from pH 5 to 6, which suggests that any acidity generated by sulfide oxidation is effectively buffered by other minerals in the tailings. The average temperature of the water samples in November was 6.2 °C, and the conductivity of the water ranged from 43.5 – 680 μS/cm. Figure 4.11 shows the distribution of the conductivity values.

Water samples from all locations were analysed for the total and dissolved (<0.45 μm) concentrations of cations and mercury. This was done to determine if the elements are present as dissolved species or as part of suspended sediment. If the elements are dissolved, they are generally more bioavailable and thus more likely to have adverse effects on aquatic biota (Smith and Huyck, 1999). The concentration of dissolved mercury in all samples ranges from 2-15 ppt and dissolved arsenic ranges from 14-6580 ppb. The concentration range of total mercury is 2-182 ppt and total arsenic is 17-6070 ppb. The water chemistry data for all samples is summarized in Table 4.3. Figures 4.12 and 4.13 show the spatial distribution of the mercury and arsenic concentrations at and downstream of the Cochrane Hill mine site. The concentrations of cyanide in the three water samples collected near the cyanidation tailings were all very low. Cyanide concentrations in the pond above the tailings, in standing water on the tailings, and in the drainage below the tailings impoundment were <2 ppb, 2 ppb, and <2 ppb, respectively. Cyanide can be removed from the water by a number of different methods including; complexation, cyanide complex precipitation, adsorption, oxidation to cyanate, volatization, biodegradation, formation of thiocyanate and hydrolysis/saponification (Smith and Mudder, 1998)

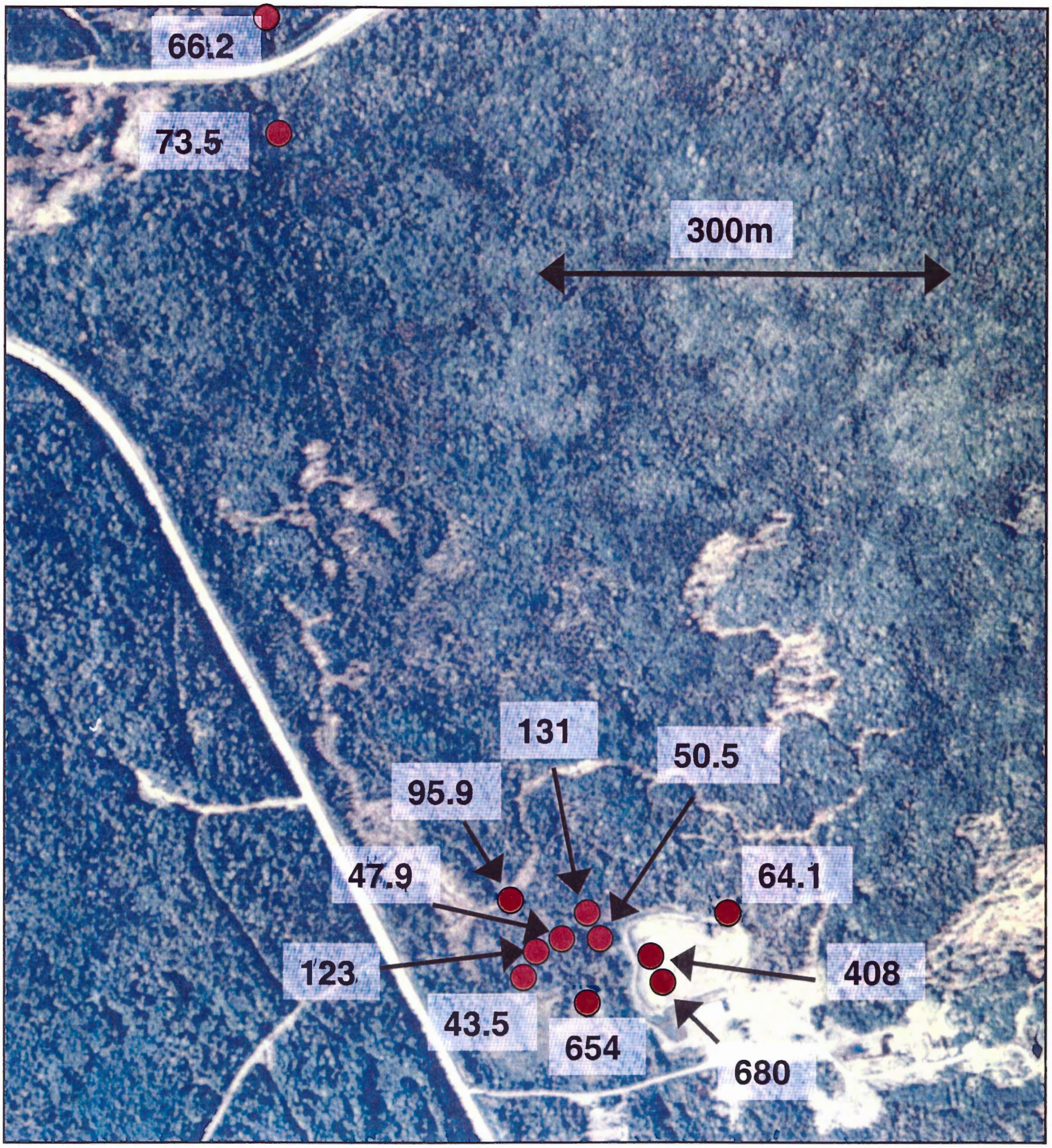


Figure 4.11 Map showing the distribution of conductivity (μS/cm) in the water sampled (Airphoto 11 E/8 © 1973 Her Majesty the Queen in Right of Canada Reproduced Courtesy of Land Information Services, 2003).

Table 4.3
Summary of selected water chemistry results.

		Alkalinity[a] PC-Titrate as $CaCO_3$ FA (ppm)	Field pH	Temp. (C°)	Conductivity (µS/cm)	As FA/UA ICP-MS (ppb)	Hg FA/UA TEKRAN* (ppt)	Mn FA/UA ICP-MS (ppb)	Ni FA/UA ICP-MS (ppb)	Zn FA/UA ICP-MS (ppb)	Pb FA/UA ICP-MS (ppb)	Al FA/UA ICP-MS (ppb)	Fe FA/UA ICP-ES (ppm)	S FA/UA ICP-ES (ppm)	SO_4 FA EG-40 (ppm)
Above Mine	W4	2	5.60	6.5	43.5	0.93/1.3	1.4/1.6	25.5/28.6	0.2/0.3	2.2/2.3	0.07/0.09	53/56	0.15/0.17	0.78/0.78	2.27
	W7	< 1	6.06	4.9	64.1	13.6/28.0	4.9/8.0	621/631	0.5/0.6	2.4/3.8	0.11/0.27	114/190	0.81/1.96	0.81/0.81	1.92
At Mine	W5	4	5.47	5.8	122.7	44.7/75.8	6.8/9.5	241/257	2.1/2.5	52.4/53.1	0.29/0.83	121/152	0.57/0.99	6.29/6.35	17.88
	W6	< 1	5.35	5.5	47.9	45.2/44.4	3.3/3.3	264/262	0.5/0.4	4.7/4.4	0.15/0.16	91/91	0.29/0.30	1.39/1.38	3.69
	W8	< 1	3.50	5.4	408	47.5/80.2	1.8/2.1	4300/4389	147/165	1624/1664	0.73/1.58	2750/2820	0.18/0.26	57.6/57.1	183.649
	W9	2	4.88	5.8	50.5	23.1/23.8	2.8/2.9	259/268	0.5/0.4	4.5/4.7	0.14/0.15	85/87	0.31/0.33	1.41/1.40	3.98
	W10	11	5.70	6.1	130.8	6580/6072	5.4/183	796/1259	70.2/ 133	37.6/375	3.75/399	66/6120	21.3/30.7	2.11/2.41	6.22
	W11	70	5.98	8.9	654	776/4290	14.6/152	5955/5716	3.8/4.3	9.2/14.2	0.40/2.70	12.0/78.0	5.21/25.9	76.6/76.9	243.263
	W12	11	5.43	6.5	680	520/1051	4.0/9.0	3650/3852	174/215	968/1250	0.28/28.35	55/4150	0.99/9.35	107/105	313.106
Below Mine	W1	< 1	6.01	6.1	66.2	15.4/16.5	4.2/4.4	23.1/31.1	0.6/0.6	9.4/9.3	0.11/0.15	72/78	0.12/0.16	1.71/1.72	4.62
	W2	2	5.33	5.6	73.5	27.3/31.0	6.1/7.1	40.4/47.0	0.9/1.0	15.6/16.0	0.21/0.28	89/97	0.23/0.30	2.47/2.43	6.59
	W3	3	5.41	6.7	95.9	41.5/55.6	5.6/8.1	164/180	1.5/1.7	32.9/34.0	0.40/0.60	105/124	0.54/0.71	4.44/4.42	12.64

* All Hg concentrations >10ppt were analysed by VG-ICP-MS

[a] Detection limit = 1

FA/UA = filtered/ unfiltered results

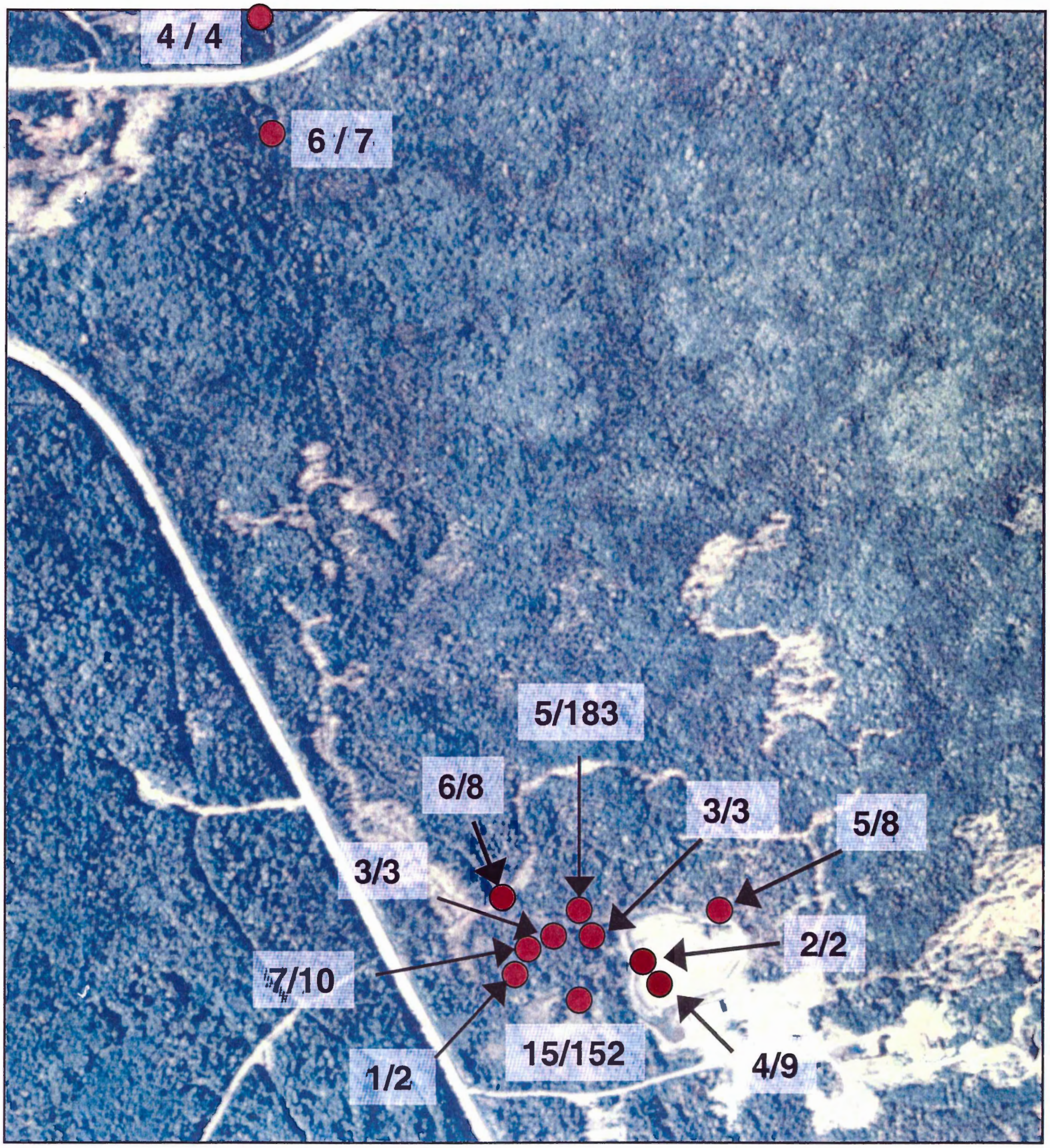


Figure 4.12 Spatial distribution of mercury in water (ppt, Diss / Tot) (Airphoto 11 E/8 © 1973 Her Majesty the Queen in Right of Canada Reproduced Courtesy of Land Information Services, 2003).

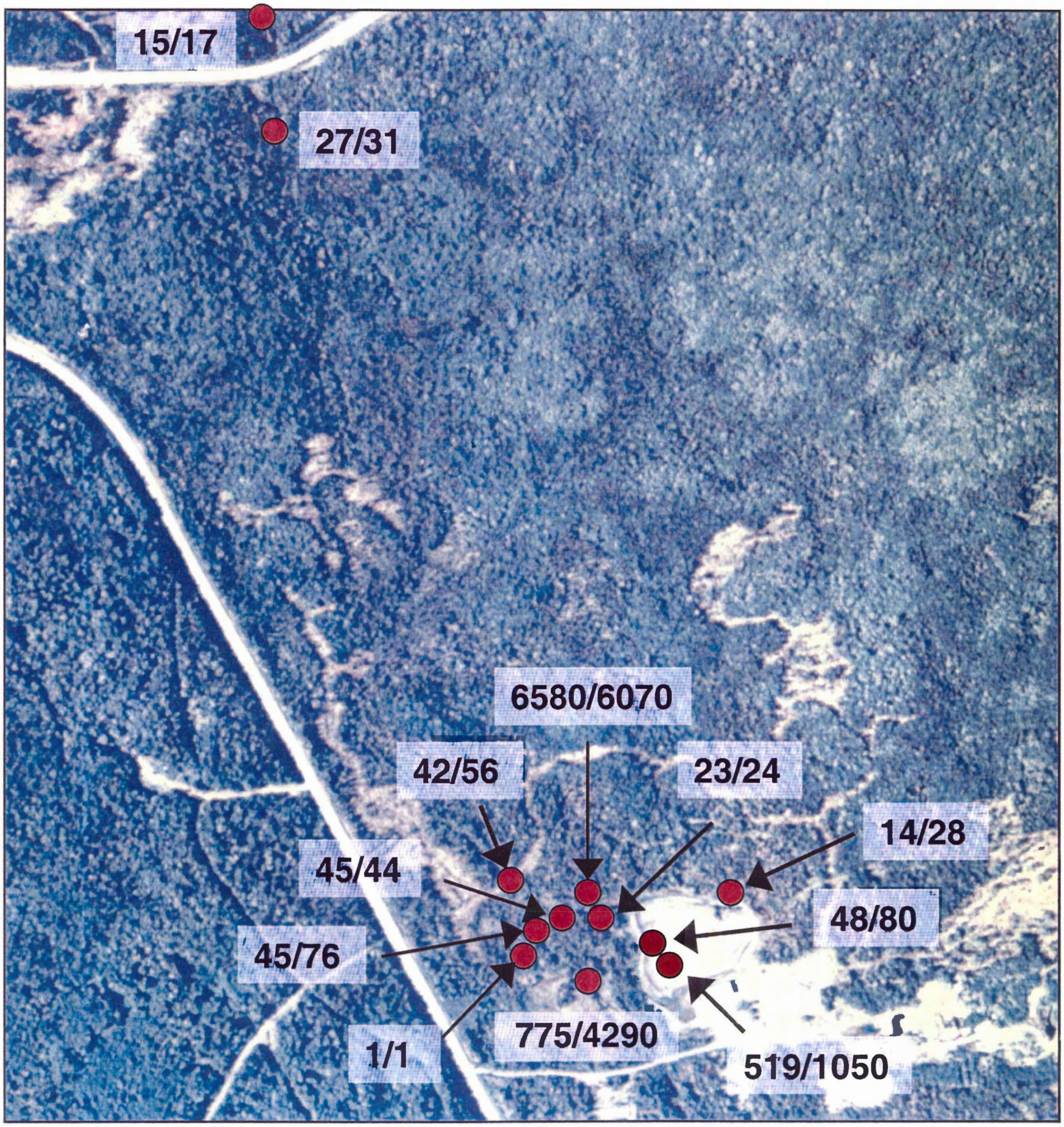


Figure 2.13 Spatial distribution of arsenic in water (ppb, Diss / Tot) (Airphoto 11 E/8 © 1973 Her Majesty the Queen in Right of Canada Reproduced Courtesy of Land Information Services, 2003).

4.5 Summary

Field observations indicate that sulphide oxidation is taking place in both the old and new tailings at Cochrane Hill, and there are a variety of reaction products (precipitates) forming on the surface of the tailings. These secondary minerals are serving as solubility controls on the dissolved concentrations of various elements, including Al, As, Ca, Fe, and S. The precipitates include ferric hydroxide, basaluminite, gypsum and possibly Turnbull's Blue. The primary mineralogy of the tailings includes arsenopyrite, pyrite, muscovite, staurolite, biotite, quartz, and anorthite with smaller amounts of monazite, tremolite/actinolite and pyrrhotite. Chemical maps of sulphides indicate traces of zinc are present. These maps also show that the arsenic is concentrated in the center of the grains and there is a rind of decreased arsenic concentration around the edge of the grains. The concentration of mercury in the tailings is <5-63,200 ppb and arsenic is 275-178,200 ppm. The pH of the water at Cochrane Hill is mildly acidic and contains very low concentrations of cyanide. Dissolved concentrations of mercury in the water are 5-15 ppt while arsenic is 14-6580 ppb. Total concentrations of mercury in the water are 2-182 ppt while arsenic is 17-6070 ppb. These results show that both the old and new tailings at Cochrane Hill are having an effect on the surrounding surface waters, which will be discussed in more detail in Chapter 5.

CHAPTER 5: DISCUSSION

5.1 Introduction

Using the results mentioned in the previous chapter some inferences can be made about the geochemistry of the tailings. There are some differences as well as similarities between the old amalgamation tailings and the younger cyanidation tailings in their chemistry and their mineralogy, and there are also some variations present within each of the tailings piles. A comparison of the secondary mineralogy present can indicate the level of reactivity that is present within each of the tailings. This chapter discusses the concentration of the elements present in the tailings as well as the correlation between some of these elements. These correlations indicate what minerals may be hosting various elements may be present in the tailings. This chapter discusses the concentration of elements present in the water samples. The spatial distribution of the concentrations can indicate how the tailings may be impacting upon the water chemistry.

5.2 Comparison of "Old" and "New" Tailings

The primary mineralogy of the tailings is similar for both piles and the sulphides from the younger tailings presently appear more altered than the sulphides from the older tailings. This is because the older tailings are now in a reducing environment because of being waterlogged and vegetated. When originally deposited these tailings would also have been in an oxidizing environment like the one the new tailings are in presently. Both piles show the same signs of oxidation within the stratigraphy of the tailings.

The amount of vegetation has influenced the reactivity within the tailings. There is evidence of more recent reactivity in the younger tailings. These tailings are not covered by vegetation and have more varied stratigraphy than the older tailings. The older tailings are overgrown with vegetation and the amount of oxidized layer is significantly less than in the younger tailings. As a result of being vegetated the tailings are not as exposed to the oxygen in the atmosphere and therefore are not presently reacting as much as the younger tailings.

More evidence to support this comes from the extent of secondary minerals associated with the younger tailings in comparison to the older tailings. Within the older tailings an arsenic-bearing ferric hydroxide is present in a drainage near the old mill structure. As much of the tailings would have been near to the mill, this evidence of reactivity is not unexpected near the old mill. Other than this there is significantly more secondary mineralization associated with the younger tailings. There is basaluminite, gypsum and Turnbull's Blue present on the younger tailings, indicating that there is some reaction taking place within the tailings.

The electron microprobe results and chemical maps of sulphide grains indicate that zinc is associated with the sulphide grains in the old and young tailings. The rinds of decreased arsenic concentration on the grains indicate that oxidation is taking place within both tailings to some degree. When arsenopyrite is oxidized As and S are released leading to acid drainage (Nesbitt et al., 1995).

Table 5.1 shows the comparison of element concentrations for the old and young tailings. Chemical analyses of the tailings show that the concentrations of arsenic for both tailings piles are similar as is the case for many of the other elements that were analysed. Mercury, silver and gold are present at higher concentrations in the older tailings than in the younger tailings. This is expected since the older tailings were produced from the mercury amalgamation process. There is a strong correlation between the concentration of silver and gold (Fig 5.1a). There is also a strong correlation between gold and mercury (Fig 5.1b). This is because the gold and silver are probably present as part of an amalgam with the mercury in the amalgamation tailings.

Arsenic and iron also show a strong correlation in the old and new tailings (Fig 5.1c), as is expected given that the arsenic is associated with the arsenopyrite in the tailings. Sulphur also has strong correlations with other elements like antimony that would be associated with sulphide minerals like the ones found in the tailings.

The strong correlations mentioned above are not present as strongly in younger tailings as they are in the older tailings. This indicates that there are more sulphides and amalgam-forming elements present in the older tailings than in the younger tailings.

Mercury is present in Nova Scotia bedrock at concentrations of <2-29 ppb and arsenic is present at concentrations generally <20 ppm in Nova Scotia soils (Smith, 2000; and

	Total Carbon (% dry wt.)	Organic Carbon (% dry wt.)	Inorganic Carbon (% dry wt.)	Fe (% dry wt.)	Al (% dry wt.)	K (% dry wt.)	Mg (% dry wt.)	Na (% dry wt.)	P (% dry wt.)	S (% dry wt.)	Ca (% dry wt.)	Ti (% dry wt.)
Min. Young	0.13	0.16	0	2.58	1.50	0.51	0.77	0.01	0.02	0.01	0.08	0.08
Max. Young	0.52	0.55	0.11	4.40	2.15	0.92	1.17	0.04	0.05	0.67	0.65	0.13
Mean Young	0.27	0.27	0	3.34	1.72	0.73	0.99	0.02	0.04	0.18	0.21	0.11
Median Young	0.25	0.25	0	3.37	1.69	0.73	0.99	0.02	0.04	0.15	0.15	0.11
Min. Old	0.34	0.25	0	1.67	0.07	0.02	0.02	0.01	0.01	0.02	0.01	0.03
Max. Old	3.62	3.31	0.37	14.8	1.91	0.82	1.13	0.02	0.05	8.05	0.33	0.13
Mean Old	1.43	1.29	0.14	4.57	1.42	0.53	0.80	0.01	0.04	0.90	0.15	0.09
Median Old	0.75	0.75	0.09	3.65	1.59	0.53	0.91	0.01	0.04	0.25	0.14	0.09

	Au ppb	Hg ppb	Ag ppb	Ga ppm	Bi ppm	Ba ppm	La ppm	B ppm	Mn ppm	Mo ppm	Cd ppm	Ni ppm	Cr ppm	Cu ppm
Min. Young	20.4	5.00	31.0	4.70	0.09	51.6	6.00	1.00	212	0.11	0.18	7.80	27.9	8.18
Max. Young	4013	25.0	185	6.90	0.60	88.3	30.6	1.00	1017	1.07	2.29	81.8	43.9	36.3
Mean Young	248	9.73	61.2	5.88	0.32	73.6	13.4	1.00	417	0.43	0.97	27.9	35.9	17.0
Median Young	124	7.50	56.0	5.90	0.26	73.5	12.4	1.00	360	0.44	0.73	22.5	35.7	16.7
Min. Old	128	14.0	67.0	0.30	0.28	11.1	0.70	1.00	257	0.19	0.19	4.40	7.40	5.62
Max. Old	25421	63162	3543	6.30	15.2	112	14.5	2.00	1723	3.44	4.50	96.7	42.2	94.5
Mean Old	3868	15320	690	4.95	2.39	66.1	9.45	1.33	635	0.67	1.16	19.9	28.6	28.2
Median Old	864	9062	176	5.40	0.60	67.4	9.70	1.00	539	0.34	0.84	9.40	28.7	23.2

	Pb ppm	Co ppm	Sb ppm	Sc ppm	Se ppm	Sr ppm	Te ppm	Th ppm	Tl ppm	U ppm	V ppm	W ppm	Zn ppm	As ppm
Min. Young	35.7	3.10	0.30	3.50	0.10	2.60	0.02	2.20	0.26	0.30	37.0	0.10	70.6	392
Max. Young	393	30.6	6.47	5.50	0.50	17.7	0.14	8.10	0.43	1.20	47.0	0.50	1006	12113
Mean Young	125	12.1	1.97	4.31	0.16	6.53	0.05	3.76	0.32	0.53	41.5	0.23	298	3766
Median Young	79.7	9.35	1.64	4.20	0.10	4.75	0.05	3.40	0.32	0.50	41.0	0.20	198	3064
Min. Old	46.9	1.50	0.69	1.10	0.10	1.10	0.02	0.50	0.20	0.30	15.0	0.10	51.3	275
Max. Old	398	101	148	4.50	8.70	14.4	2.16	3.50	0.89	0.50	45.0	3.90	463	178200
Mean Old	136	16.8	17.3	3.48	1.42	4.59	0.34	2.75	0.37	0.40	36.0	0.81	188	23252
Median Old	131	9.80	3.17	3.70	0.45	3.60	0.07	2.90	0.34	0.40	38.0	0.30	147	5618

Table 5.1 Table of elements analysed in the tailings, comparing the minimum, maximum, mean and median for the old and new tailings

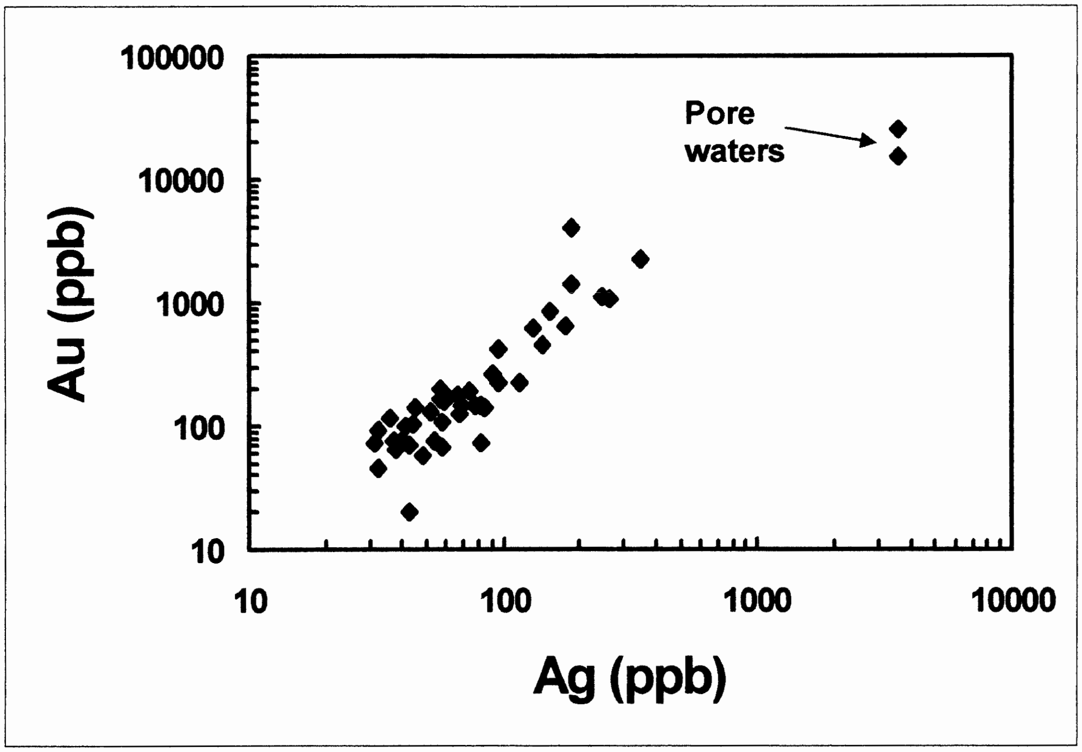


Figure 5.1a Graph showing the correlation between gold and silver for all tailings samples.

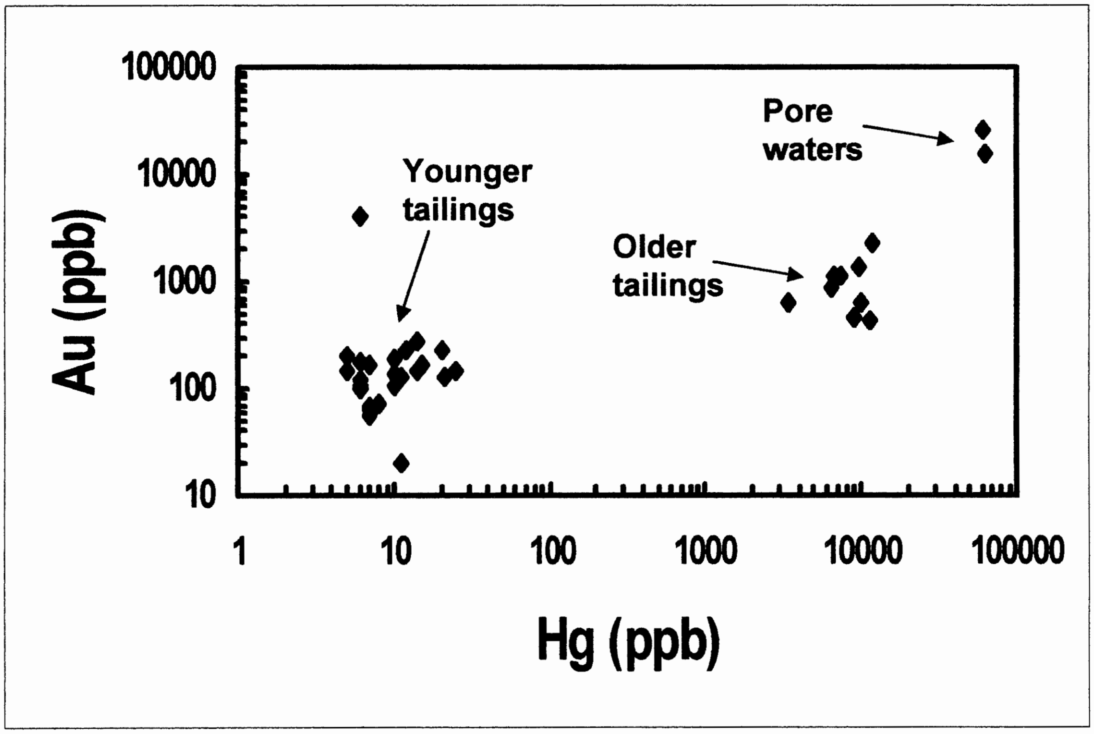


Figure 5.1b Graph showing the correlation between gold and mercury for all tailings samples.

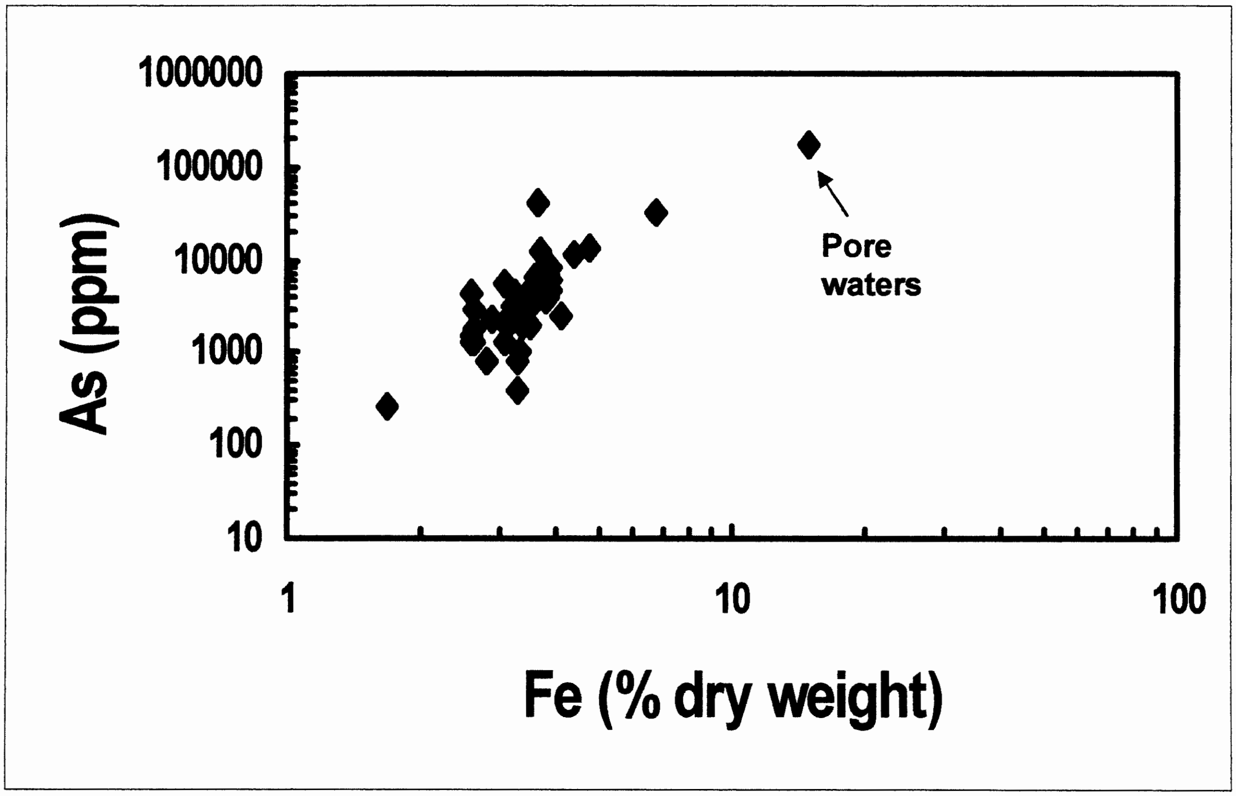


Figure 5.1c Graph showing the correlation between arsenic and iron for all tailings samples.

P.K. Smith, personal communication, 2004). The concentrations of arsenic in the tailings are well above of this and the mercury concentrations are well above the background levels in the older tailings.

The concentrations of zinc and calcium are higher in the younger tailings than the old. This is the result of the cyanidation process when lime ($Ca(OH)_2$) is added to control the pH of the cyanidation solution and zinc is added to remove the gold from the cyanide solution.

5.3 Impact of Tailings on Water Chemistry

The concentrations of elements like mercury and arsenic are at high levels in the water closer to the tailings and decrease rapidly downstream from the tailings. This indicates that the elements are not being transported over large distances but are staying localized. The highest concentrations are associated with the tailings pore waters and the water in the drainage near the old mill. For the pore waters the mercury and arsenic concentrations are both high. The unfiltered samples for mercury are much higher than for filtered, which could be the result of the tailings being disturbed when the hole was dug (Sites W10 and W12, Fig 5.2). The unfiltered and filtered samples both contain high concentrations of arsenic in the pore waters, which indicates that the concentration of arsenic is indeed higher in the pore waters of the tailings (Fig 5.3). Other elements that are present at high concentrations in the waters at the mine site are Al, Mn, Ni, Zn, Pb, and S.

The Guidelines for Canadian Drinking Water Quality have specified limits of 25 ppb for arsenic and 1 ppb for mercury. The concentrations of arsenic found in the waters at the mine site are well above these guidelines (Summary of Guidelines for Canadian Drinking Water Quality, 2003).

Figure 5.4 shows the strong positive correlation between dissolved mercury and dissolved organic carbon (DOC). It shows that there is a strong association between the concentration of dissolved mercury and DOC. This suggests that the carbon is acting as a carrier for the mercury and the two have a strong affinity for one another. There are two outliers one being the drainage near the old mill where there is a large amount of mercury

Figure 5.2 Comparison of filtered and unfiltered concentrations of Hg

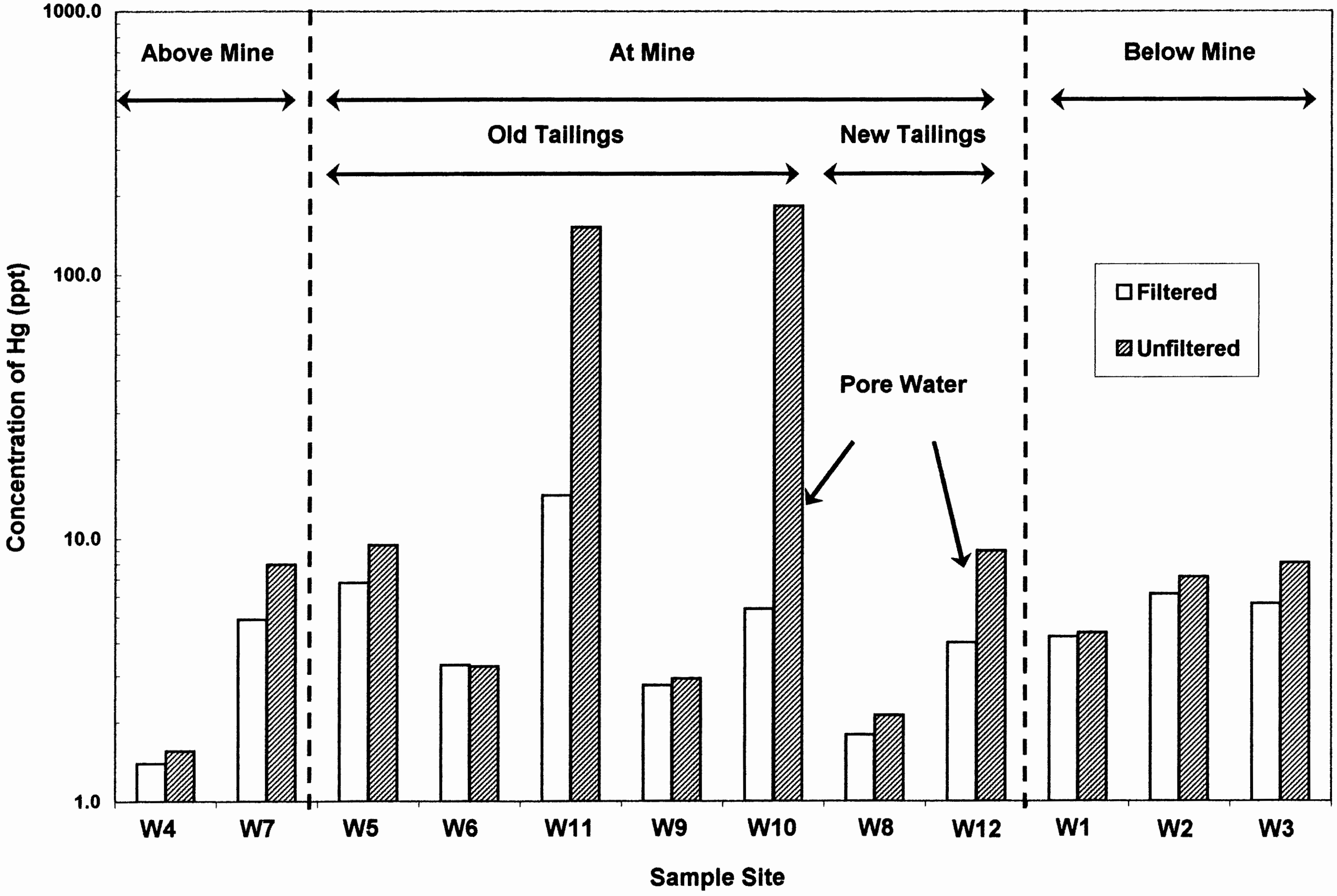
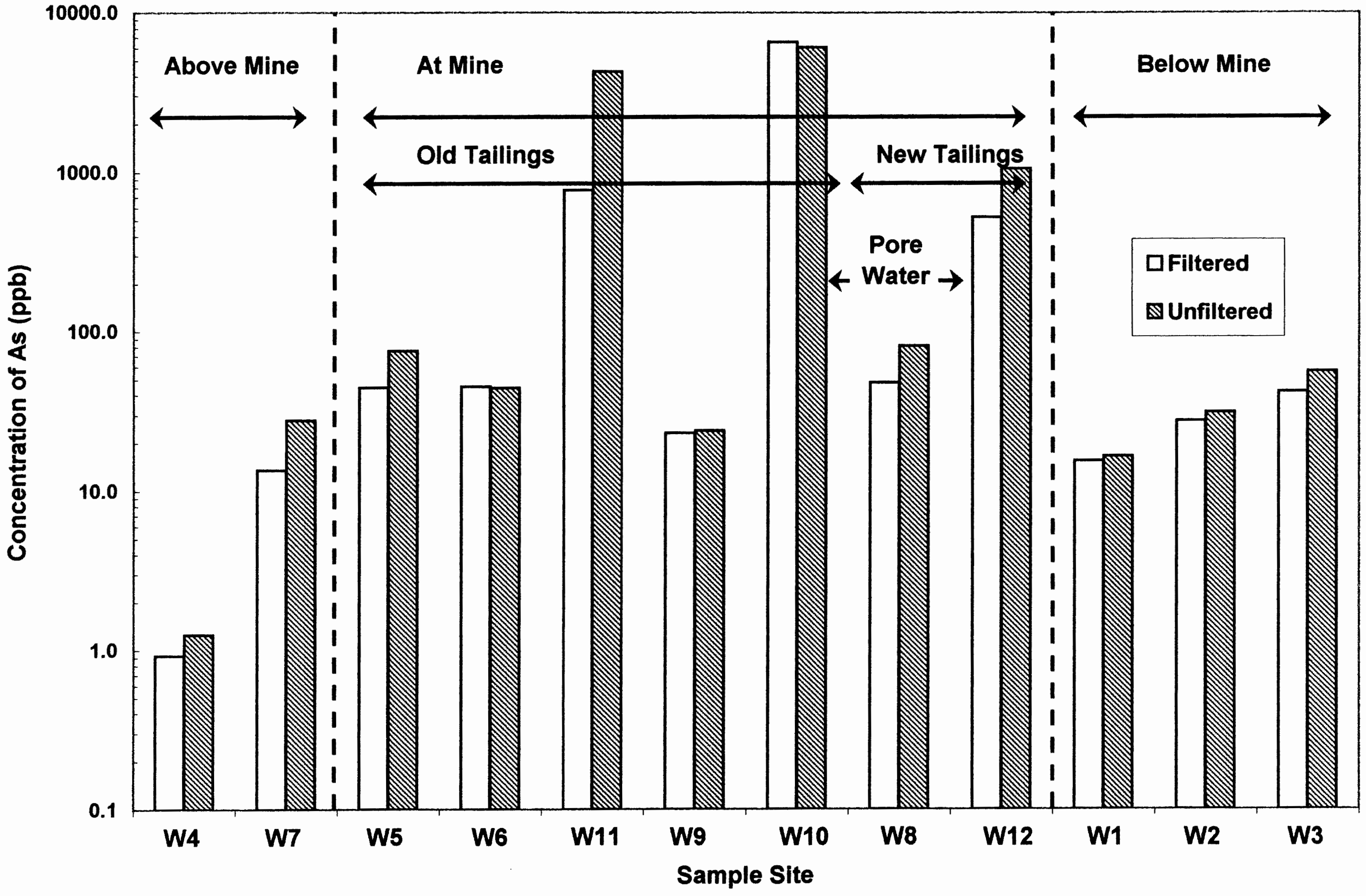


Figure 5.3 Comparison of filtered and unfiltered concentrations of As



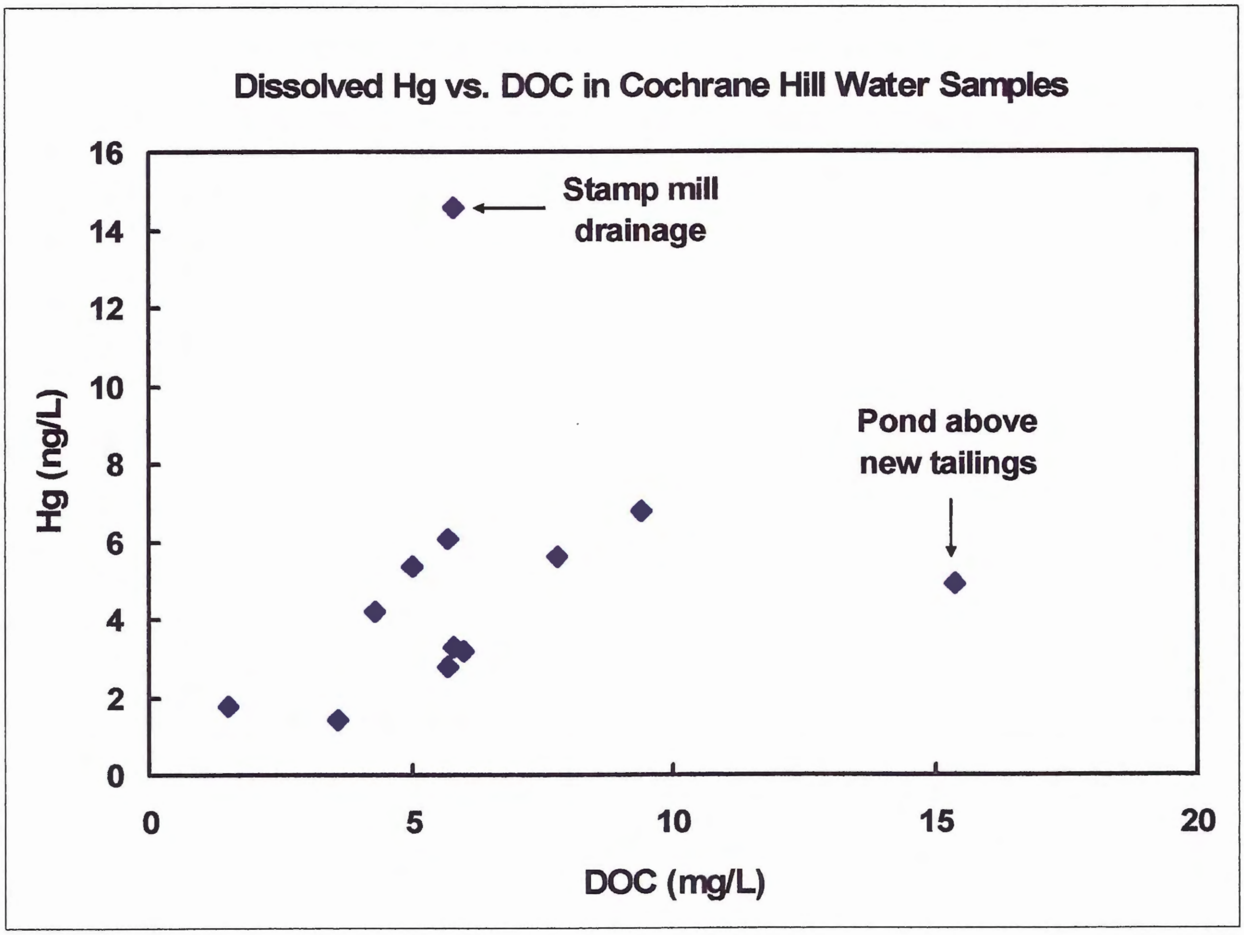


Figure 5.4 Graph of the dissolved mercury versus the dissolved organic carbon for all water samples.

present and little DOC, the other being the pond above the tailings where there is a large amount of DOC present and little mercury. The proximity of the drainage to the old mill results in higher concentrations of mercury because the mill is where most of the mercury would have been deposited with the tailings during amalgamation. The pond has high concentrations of DOC because it receives drainage directly from a forested watershed and is far enough up stream from the old tailings to not have been effected by their high Hg content.

Although there were some cyanide drums buried in the younger tailings the concentration of cyanide in the water draining out of the tailings is very low, and well below the drinking water guideline of 200 ppb (Summary of Guidelines for Canadian Drinking Water Quality, 2003). This could be the result of volatization which is a major loss mechanism for cyanide at a pH below 9. The pH of the water at Cochrane Hill is 3.5-6 which would allow for volatization. The cyanide could also be present in the blue precipitate that is possibly Turnbull's Blue (Fig 4.10b). This precipitate is present on the surface directly beside where the drums were buried and only in this location. Closer analysis of this precipitate would need to be done to confirm this, but the research done in this project suggests that the precipitate is likely Turnbull's Blue. Dissolved cyanide may still be present deeper within the tailings and could be the focus of further study.

The secondary minerals that are present as precipitates could be limiting the concentrations of metals from the water as a result of co-precipitation reactions.

CHAPTER 6: CONCLUSIONS

The primary mineralogy of the younger and older tailings is quite similar. They both show signs of reactivity; however, the younger tailings are presently more reactive. Within the younger tailings there is more variation present in the stratigraphy and there are more secondary minerals associated with them. This is partially the result of lack of vegetation on the younger tailings as compared to the overgrown older tailings.

There are elevated concentrations of Hg, Au, and Ag in the older tailings, which suggests that there is amalgam present in the older tailings. This indicates that amalgam was deposited with the tailings during the amalgamation process. It also indicates that the cyanidation process is more effective at extracting gold since there are significantly lower concentrations of gold present in the cyanidation tailings.

The levels of mercury and arsenic in the tailings are well above the background levels in Nova Scotia.

There are elevated concentrations of Zn, and Ca present in the younger tailings, indicating that some of these elements were deposited with the tailings during cyanidation process.

The water found at the mine site is not suitable for drinking. The As, and Hg concentrations are well above the Canadian Drinking Water Guidelines. There are also elevated concentrations of Al, Mn, Ni, Zn, Pb, and S found in the waters at the mine site. This indicates that the tailings are having a significant effect on the water chemistry; however, the effect does not extend far downstream. This is shown by the sharp drop in concentration of these elements about 1.5 km downstream.

The low concentrations of cyanide indicate that the cyanide has likely been lost through volatilization or is present as Turnbulls Blue. Future study would need to be done in order to make sure this is the case.

The secondary minerals could also be co-precipitating elements out of the water and therefore decreasing their concentration in the water.

The water sampling that was done is also only a snapshot in time and the concentrations of the elements analysed for could vary seasonally. In the spring for example when there is an influx of melt water and rain water the tailings could have a larger effect on the chemistry of the water. Further study should focus on how the tailings are affecting the water chemistry throughout the year and the speciation of the elements in the water.

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Summary of Guidelines for Canadian Drinking Water Quality, Federal-Provincial-Territorial Committee on Drinking Water of the Federal-Provincial-Territorial Committee on Environmental and Occupational Health, 2003

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APPENDIX A : SAMPLE SITE DESCRIPTIONS

CYANIDATION TAILINGS

T1

Tailings 5 m from buried cyanide drums, surface is covered with rocks stained with a blue precipitate, 4 samples taken T1A-D. Stratigraphy includes 30cm olive brown tailings above 5cm rusty, mottled tailings above dark grey clay rich tailings.

T2

Tailings north of the mill ~30m west of buried cyanide drums, surface is covered with white salts, 4 samples taken T2A-D. Stratigraphy includes 30cm of dark brown sandy tailings above 5 cm of rusty, mottled tailings above clay rich dark grey tailings.

T3

Tailings ~ 30m west of T2, surface is actively reworked by erosion, 5 samples taken T3A-E. Stratigraphy includes 20cm dark brown sandy tailings above 5cm dark grey tailings above 10cm dark brown sandy tailings above 3cm dark coarse tailings above 12cm dark brown sandy tailings above dark grey clay rich tailings.

T4

Tailings near the southwest corner of impoundment, 4 samples taken T4A-D. Stratigraphy includes 33cm of dark brown sandy tailings above 5cm of coarse dark tailings above 22cm of olive mottled tailings above dark grey clay rich tailings.

T5

Tailings mid-way across lower end of impoundment, 3 samples taken T5A-C. Stratigraphy includes 64cm of dark brown coarse tailings above dark grey clay rich tailings.

T6

Tailings flat in the mixed waste rock/ tailings area, 4 samples taken T6A-D. Stratigraphy includes 20cm of sandy grey brown tailings above 16cm of dark grey and rusty hard tailings above 19cm of sandy dark brown and rusty tailings above dark grey tailings.

T7

Tailings in the northeast corner of the impoundment, 4 samples taken T7A-D. Stratigraphy includes 25cm of grey brown tailings above 1cm of olive yellow tailings above 34cm of grey brown tailings above dark grey clay rich tailings.

T8

Tailings near northern rock berm, 3 samples taken T8A-C. Stratigraphy includes 55cm brown olive tailings above dark grey tailings.

T9

Tailings in the northwest corner beneath the bubbled surface area, 3 samples taken T9A-C. Stratigraphy includes 20cm of light grey and rusty tailings above 7cm coarse dark grey tailings above 23cm light grey brown tailings above grey clay rich tailings.

AMALGAMATION TAILINGS

T10

Tailings near northwest corner of old mill, 2 samples taken T10A-B. Stratigraphy includes 20cm of light grey olive coarse tailings above 2cm dark grey tailings above wood.

T11

Tailings are vegetated, 3 samples taken T11A-C. Stratigraphy includes 14cm of brown grey tailings above 5cm rusty brown tailings above 7cm dark grey tailings above rocks.

T12

Tailings in open vegetated area, 2 samples taken T12A-B. Stratigraphy includes 5cm rusty brown tailings above dark grey tailings.

T13

Tailings in marshy area, organic soils overlying saturated grey tailings, 1 sample taken T13A

T14

Tailings along the bank of stream running north from the tailings, tailings are rooty and organic rich.

T15

Tailings sampled when the water samples were taken, mossy tailings west of the northwest corner of the impoundment, 2 samples taken T15A-B. Stratigraphy includes 20cm of dark grey silty tailings above 20cm rooty brown soil and tailings above rocks.

APPENDIX B : X-RAY DIFFRACTION RESULTS

Appendix B: Key to X-ray Diffraction Results

CH03T3A-E, tailings samples taken at different depths within the hole dug at site T3.

CH03T3H, CH03T3LT, heavy (H) and light (LT) separates of tailings sampled at site T3.

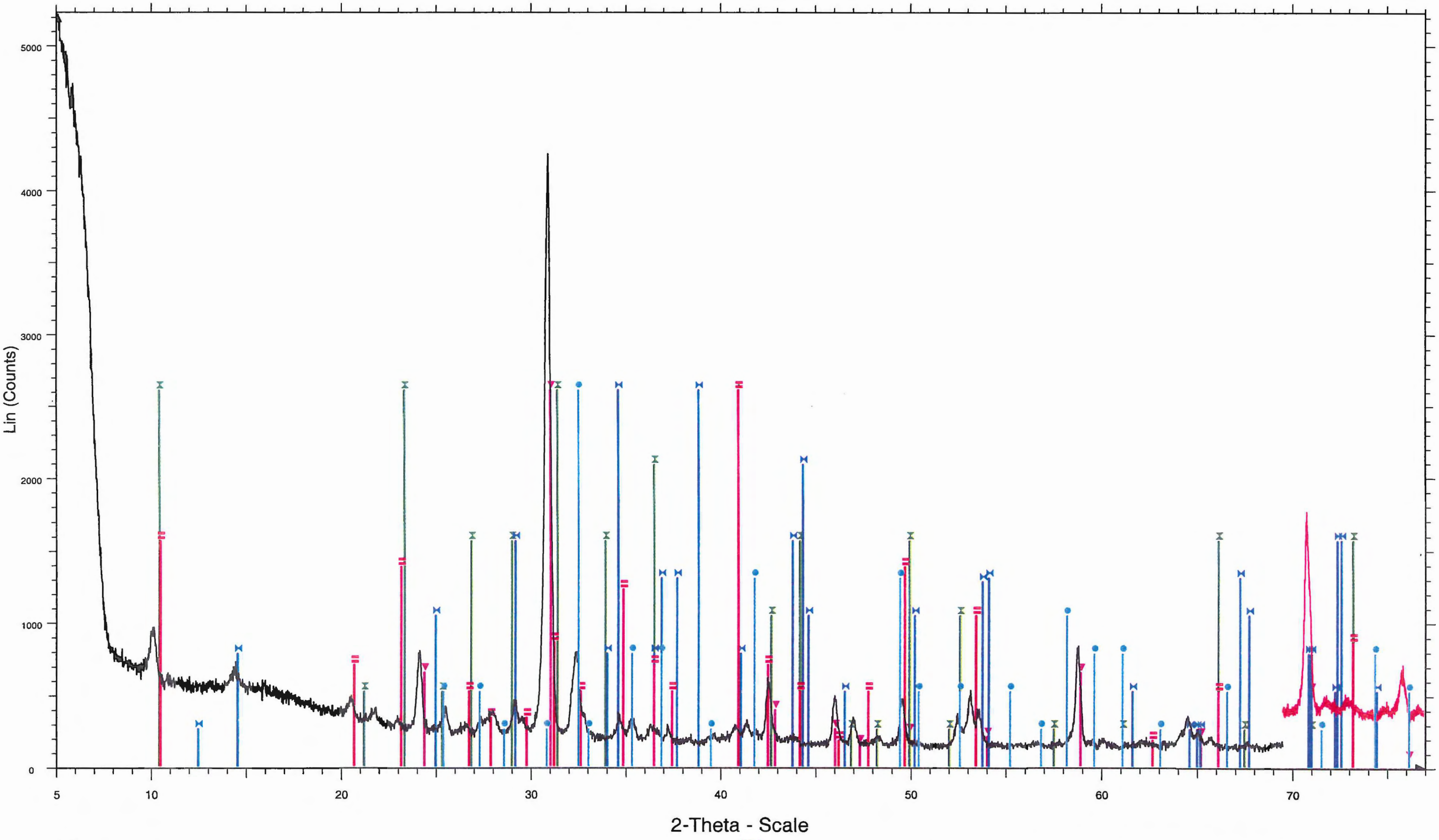
CH03T11A-C, tailings samples taken at different depths within the hole dug at site T11.

CH03T11H, CH03T11LT, heavy (H) and light (LT) separates of tailings sampled at site T11.

CH3MILKY, milky precipitate sampled from the rivulet on the younger tailings.

CH3OLDML, orange precipitate sampled from the drainage by the old mill.

CH03T3A



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15-0397 (D) - Staurolite - (Fe,Mg)4Al18H2Si8O48 - Y: 50.00 % - d x by: 1.000 - WL: 1.78897
01-1098 (D) - Muscovite - H2KAl3(SiO4)3 - Y: 50.00 % - d x by: 1.000 - WL: 1.78897

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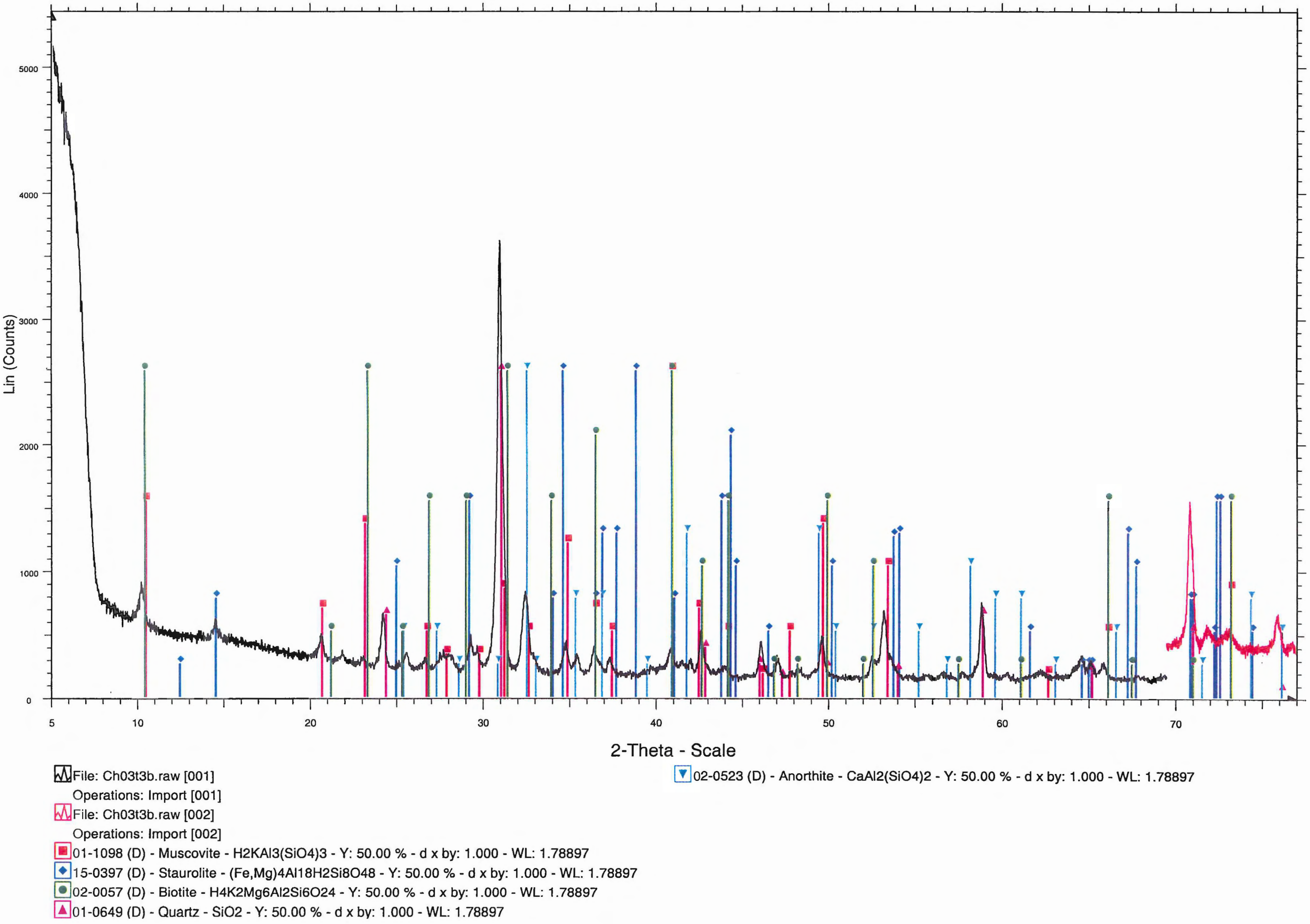
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● 02-0057 (D) - Biotite - H4K2Mg6Al2Si6O24 - Y: 50.00 % - d x by: 1.000 - WL: 1.78897
▲ 01-0649 (D) - Quartz - SiO2 - Y: 50.00 % - d x by: 1.000 - WL: 1.78897
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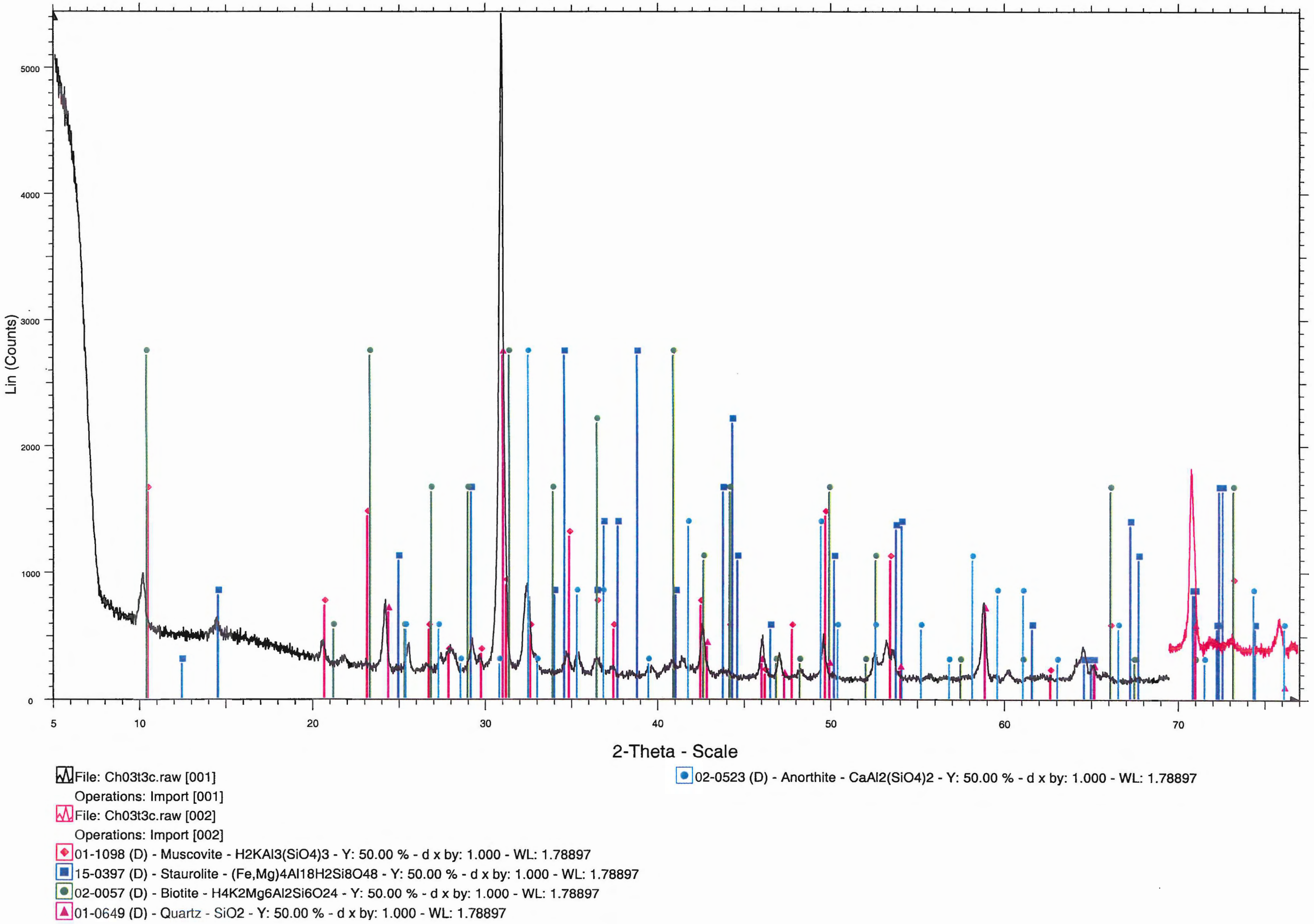
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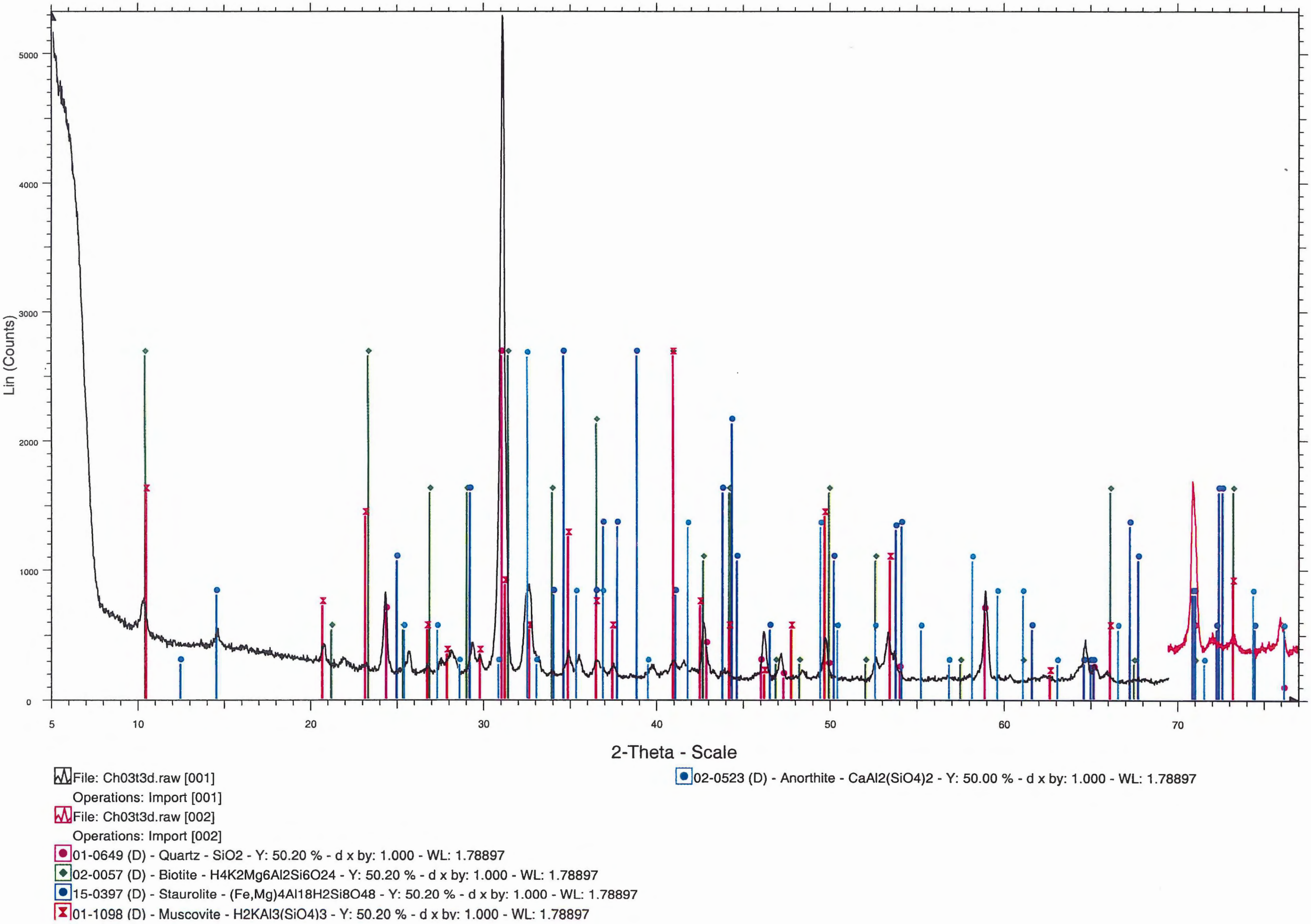
01-0649 (D) - Quartz - SiO2 - Y: 50.00 % - d x by: 1.000 - WL: 1.78897
02-0523 (D) - Anorthite - CaAl2(SiO4)2 - Y: 50.00 % - d x by: 1.000 - WL: 1.78897

CH03T3D



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15-0397 (D) - Staurolite - (Fe,Mg)4Al18H2Si8O48 - Y: 50.20 % - d x by: 1.000 - WL: 1.78897
01-1098 (D) - Muscovite - H2KAl3(SiO4)3 - Y: 50.20 % - d x by: 1.000 - WL: 1.78897
02-0523 (D) - Anorthite - CaAl2(SiO4)2 - Y: 50.00 % - d x by: 1.000 - WL: 1.78897

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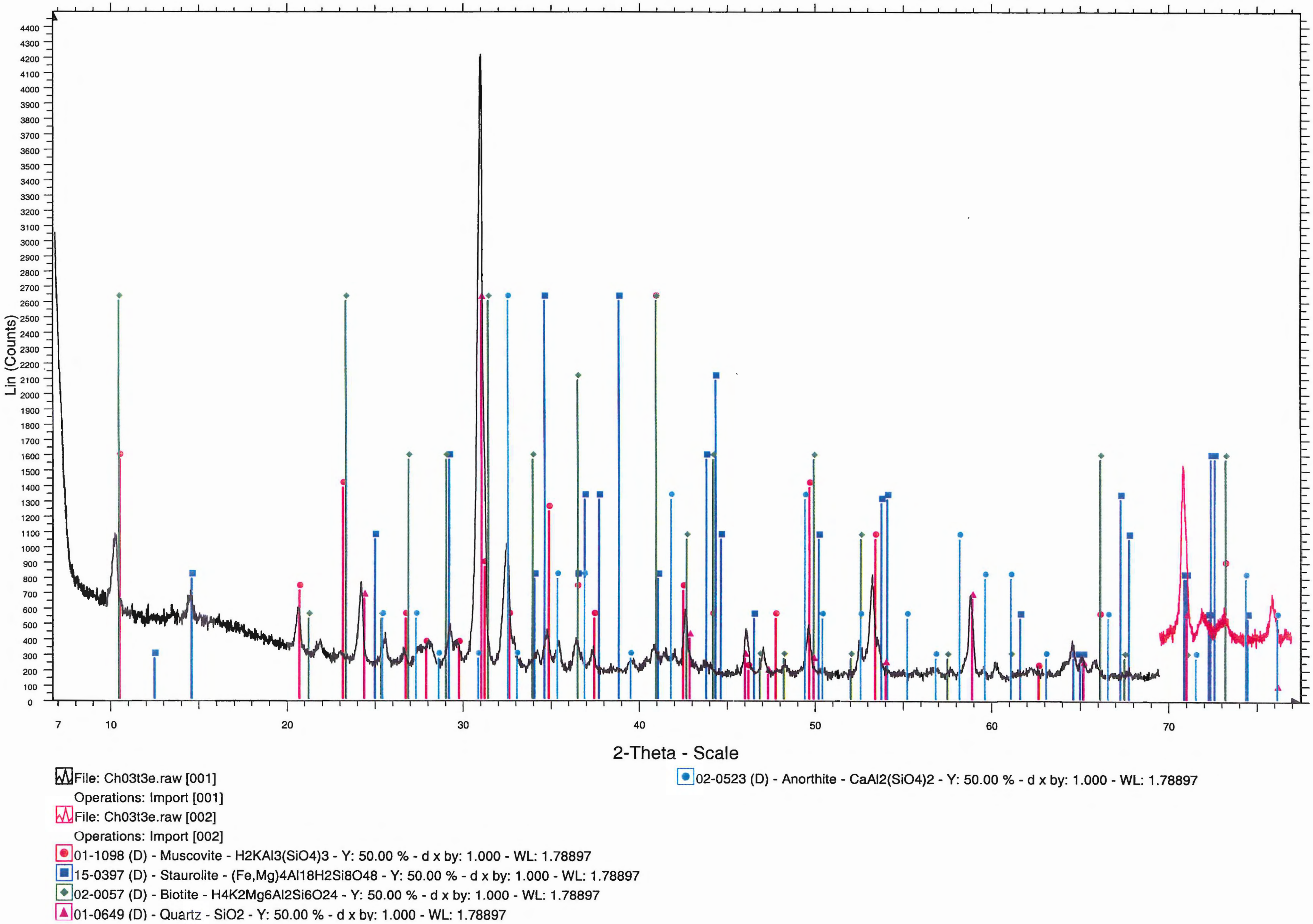
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◆ 02-0057 (D) - Biotite - H4K2Mg6Al2Si6O24 - Y: 50.00 % - d x by: 1.000 - WL: 1.78897
▲ 01-0649 (D) - Quartz - SiO2 - Y: 50.00 % - d x by: 1.000 - WL: 1.78897
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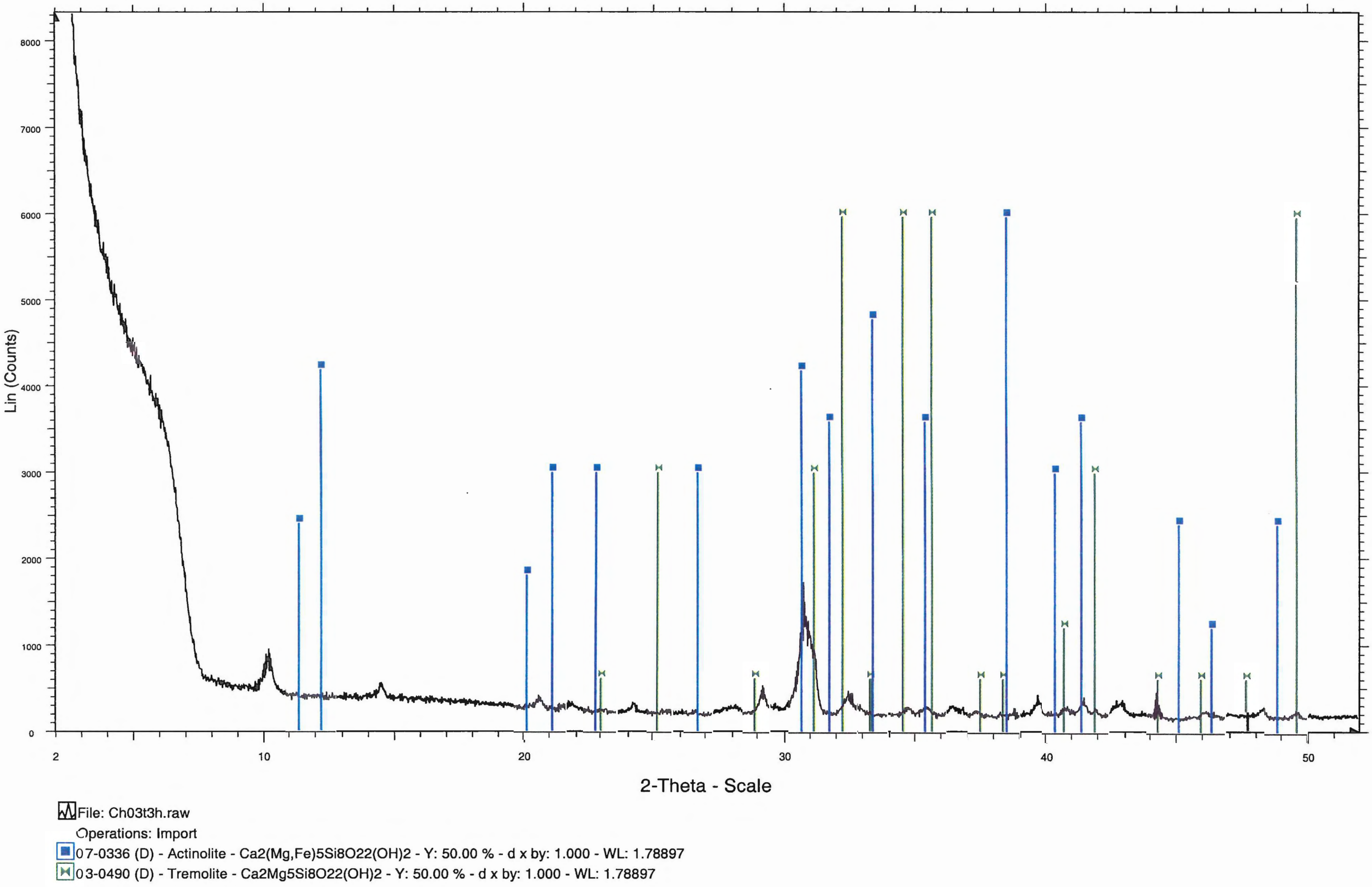
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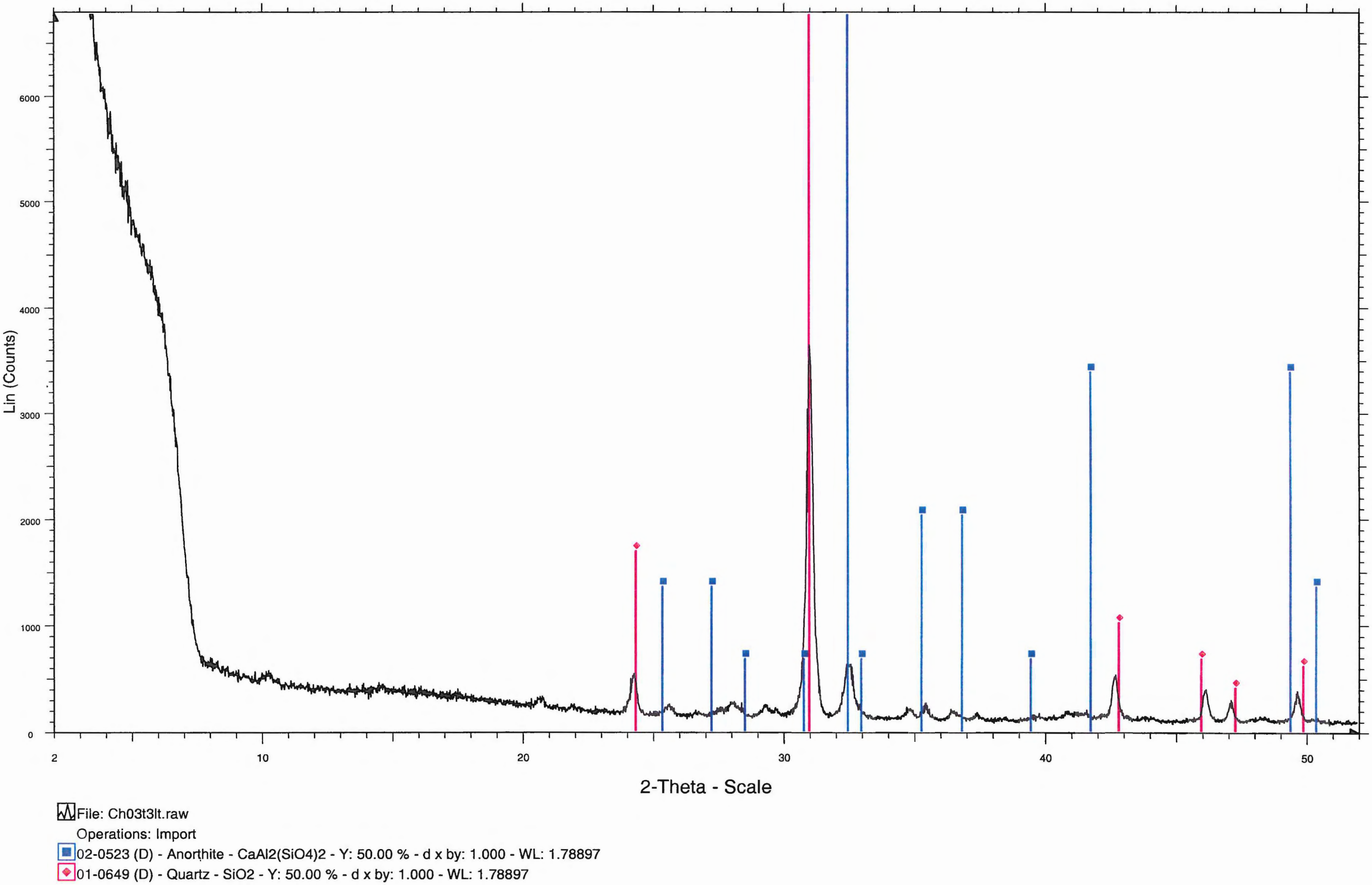
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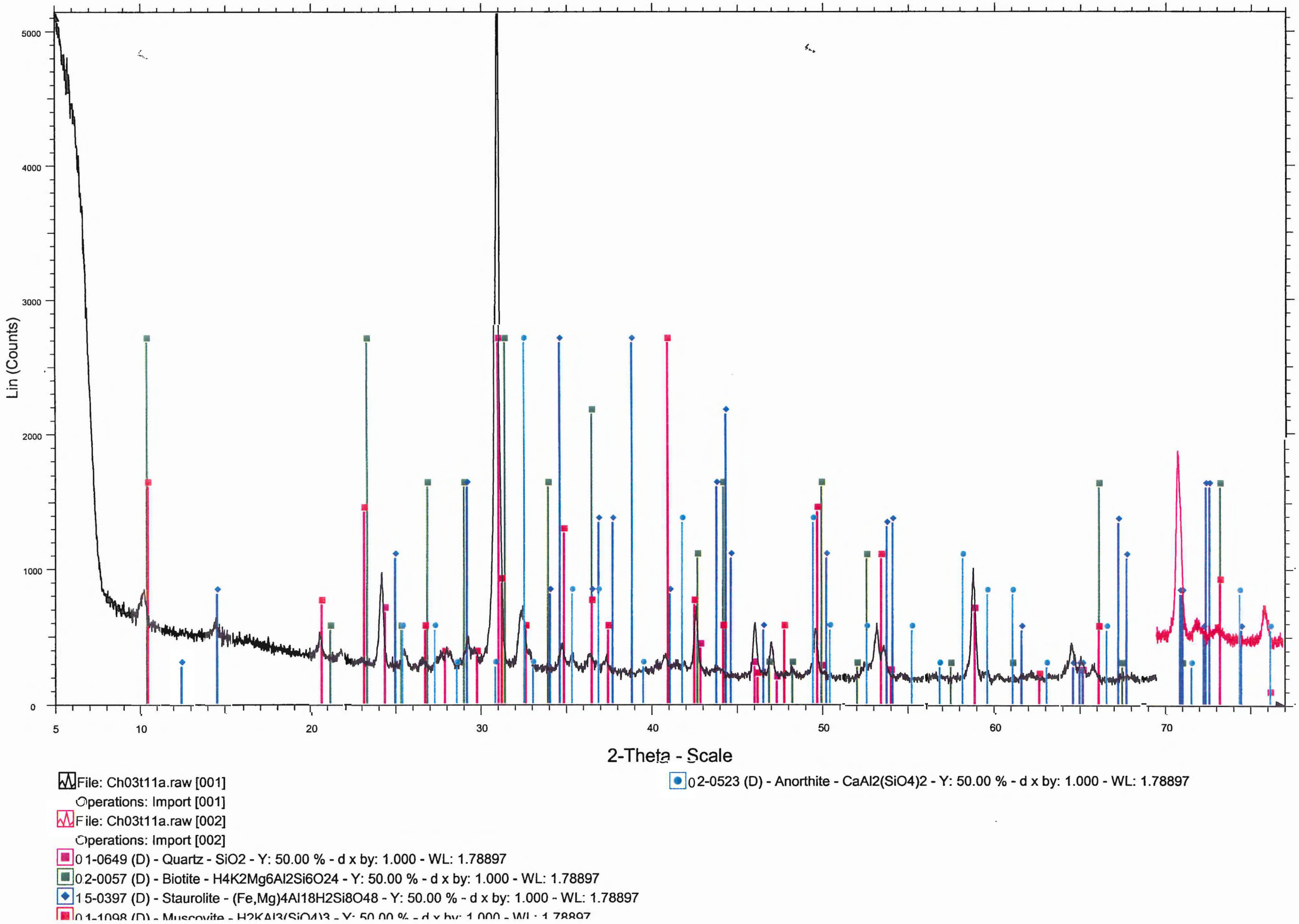
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◆ 15-0397 (D) - Staurolite - (Fe,Mg)4Al18H2Si8O48 - Y: 50.00 % - d x by: 1.000 - WL: 1.78897
🟥 01-1098 (D) - Muscovite - H2KAl3(SiO4)3 - Y: 50.00 % - d x by: 1.000 - WL: 1.78897
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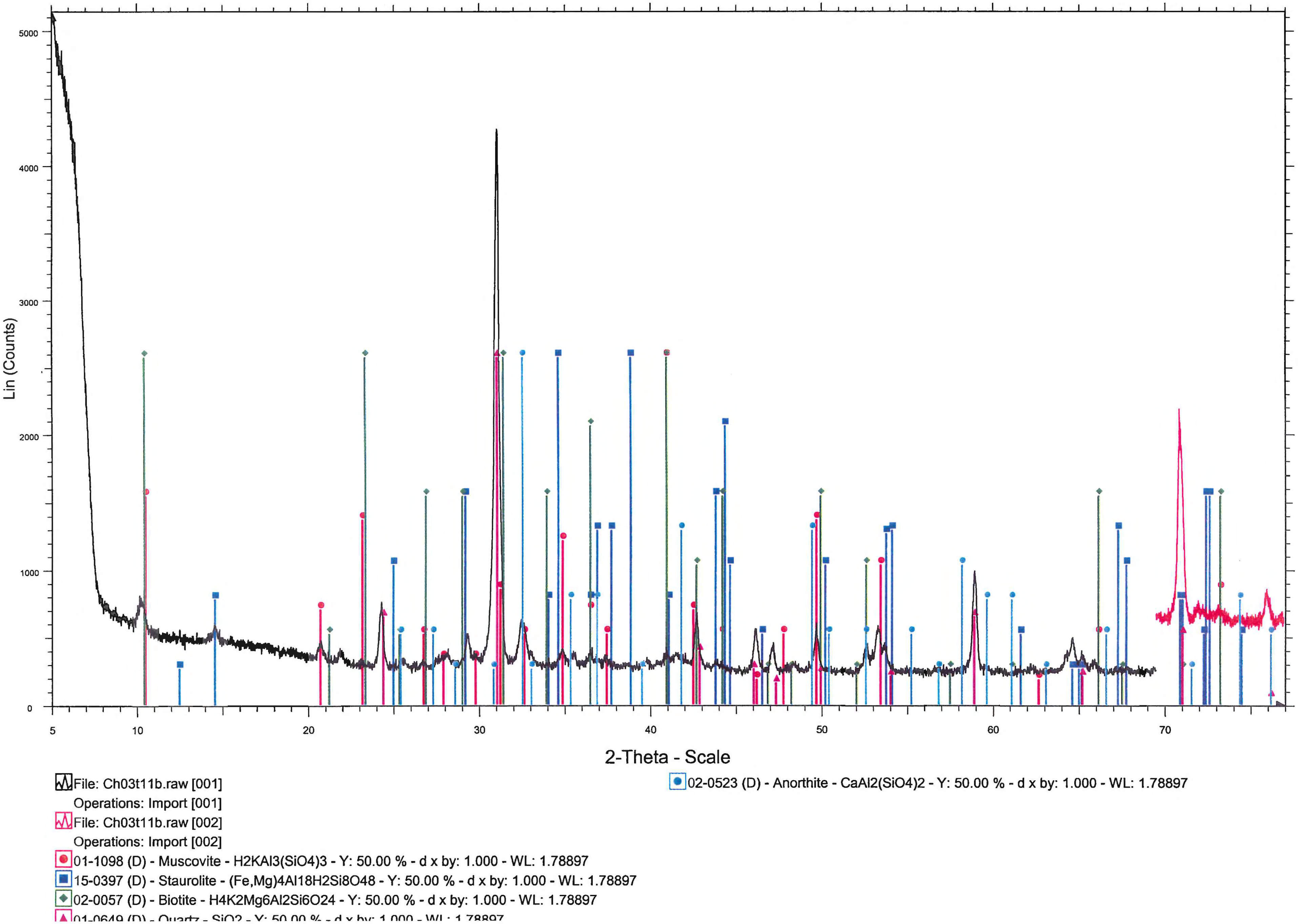

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15-0397 (D) - Staurolite - (Fe,Mg)4Al18H2Si8O48 - Y: 50.00 % - d x by: 1.000 - WL: 1.78897
02-0057 (D) - Biotite - H4K2Mg6Al2Si6O24 - Y: 50.00 % - d x by: 1.000 - WL: 1.78897
01-0649 (D) - Quartz - SiO2 - Y: 50.00 % - d x by: 1.000 - WL: 1.78897
02-0523 (D) - Anorthite - CaAl2(SiO4)2 - Y: 50.00 % - d x by: 1.000 - WL: 1.78897

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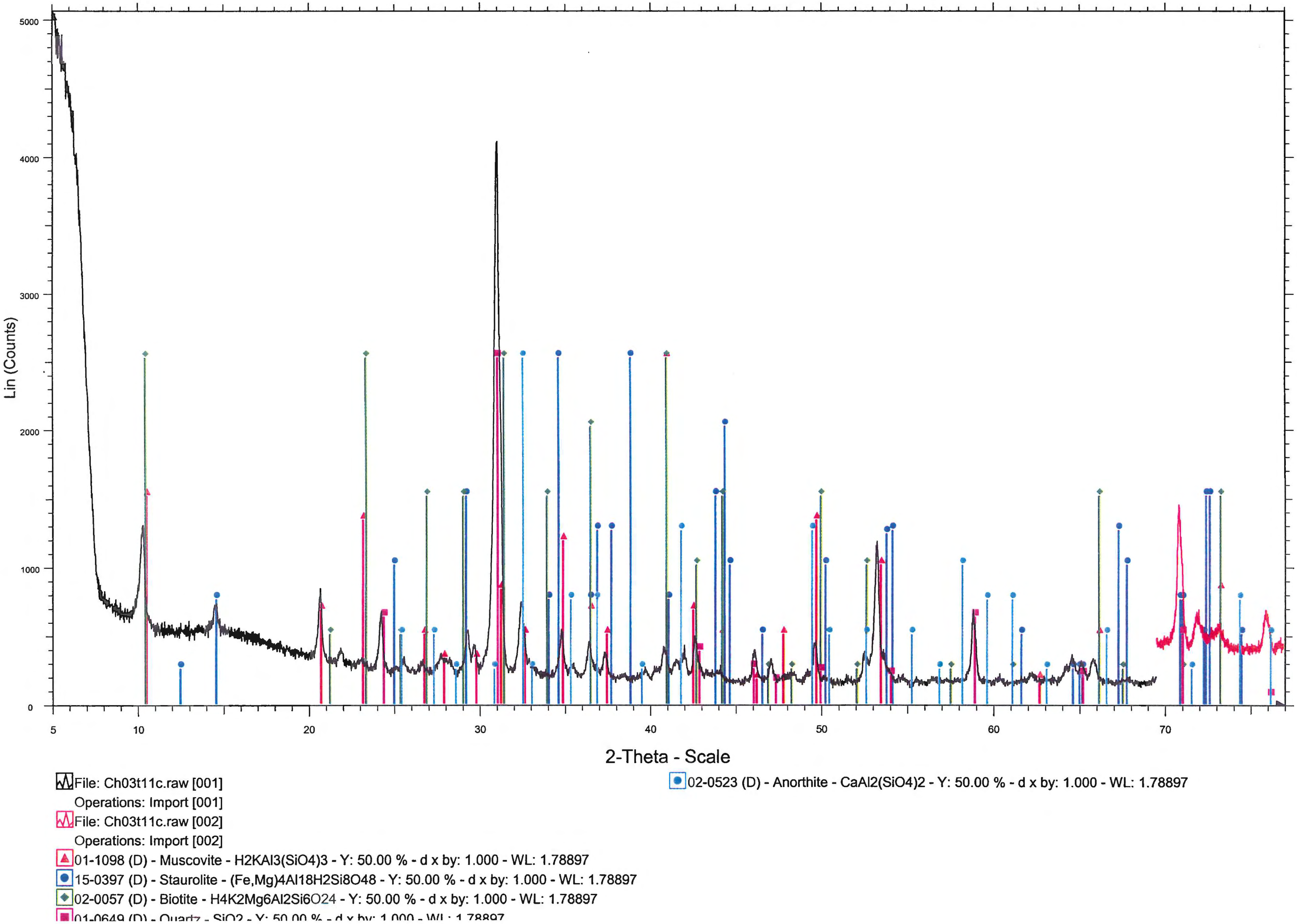
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02-0057 (D) - Biotite - H4K2Mg6Al2Si6O24 - Y: 50.00 % - d x by: 1.000 - WL: 1.78897
01-0649 (D) - Quartz - SiO2 - Y: 50.00 % - d x by: 1.000 - WL: 1.78897
02-0523 (D) - Anorthite - CaAl2(SiO4)2 - Y: 50.00 % - d x by: 1.000 - WL: 1.78897

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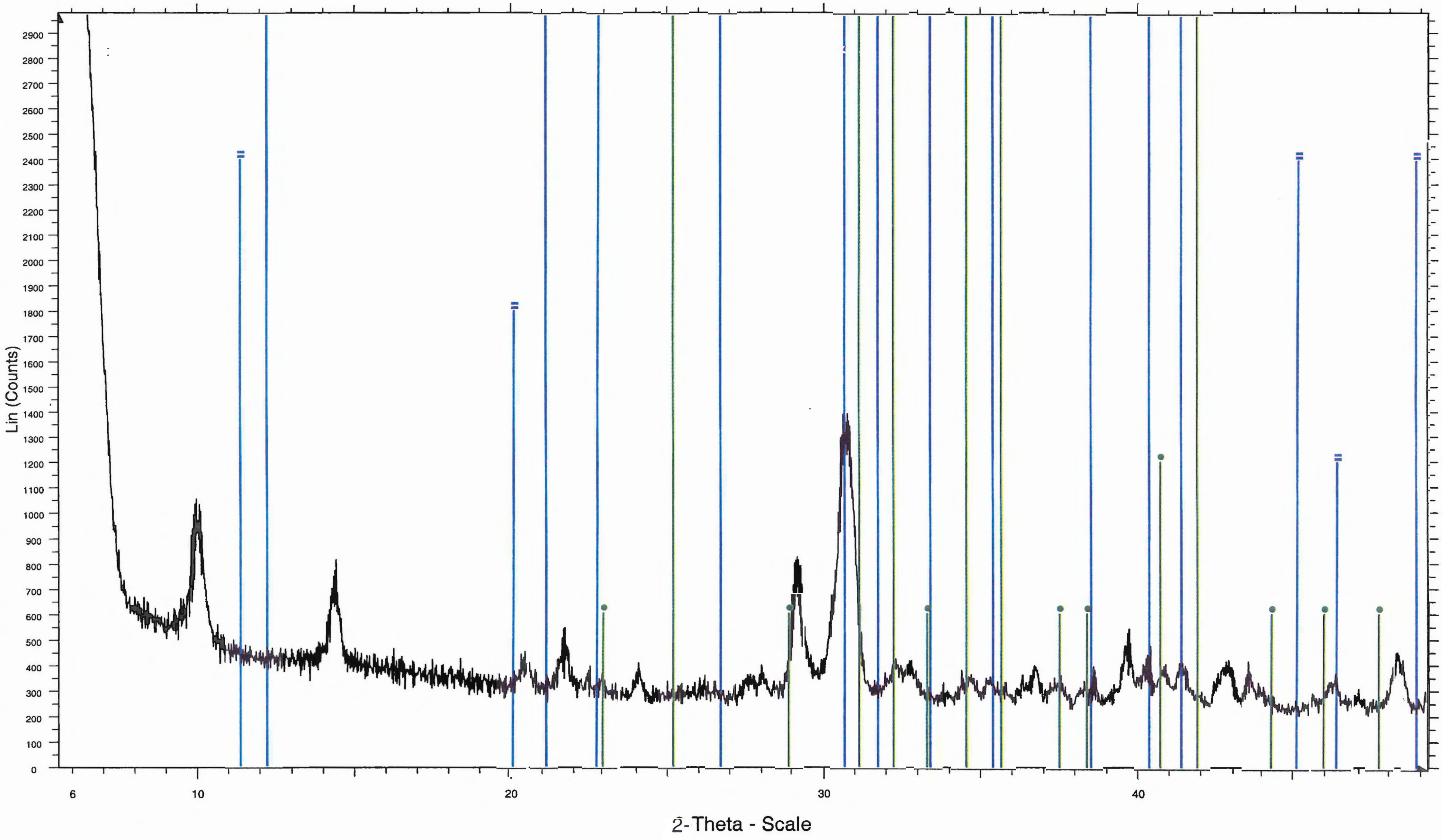

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☰ 07-0336 (D) - Actinolite - Ca2(Mg,Fe)5Si8O22(OH)2 - Y: 50.00 % - d x by: 1.000 - WL: 1.78897

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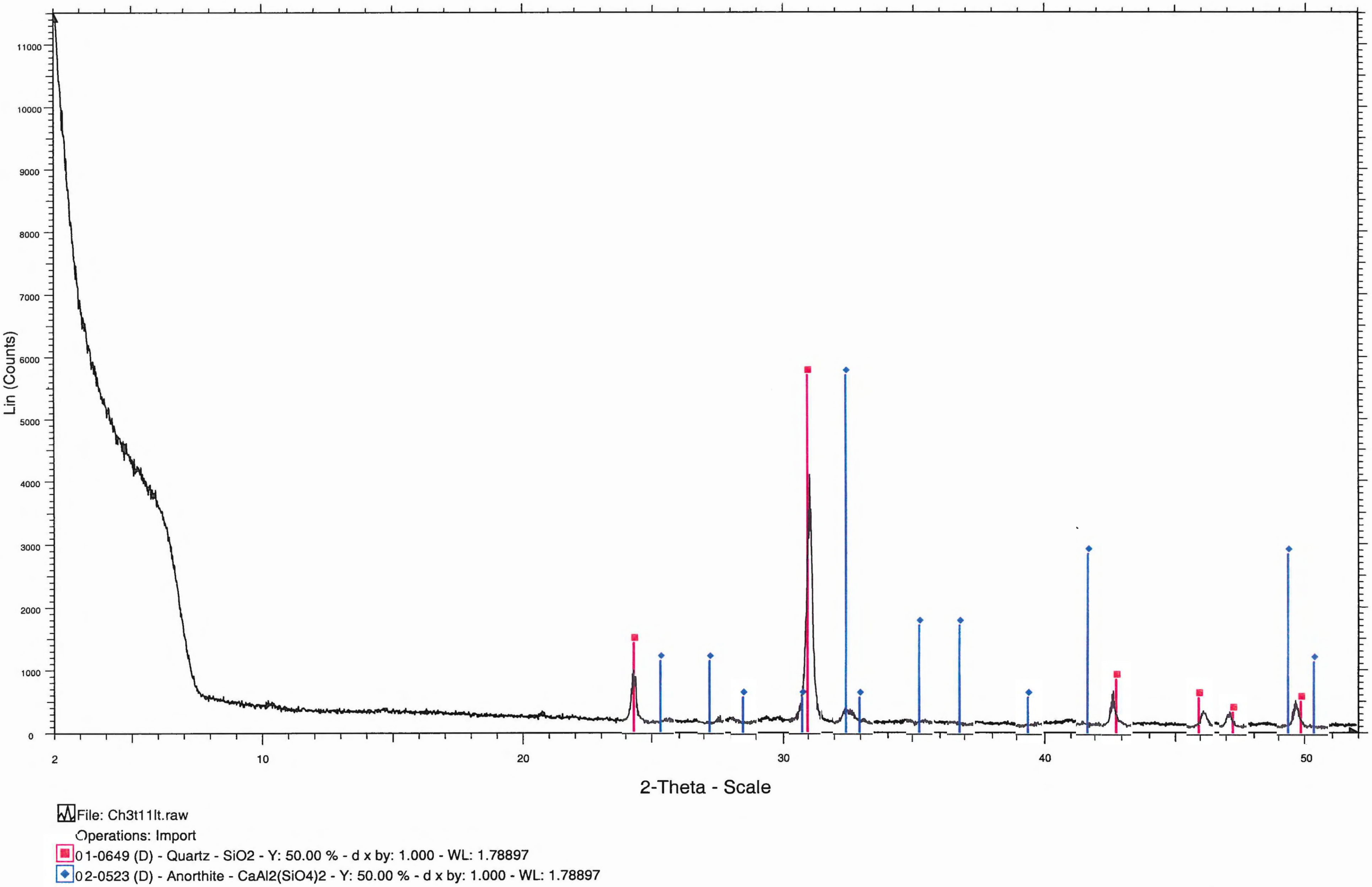
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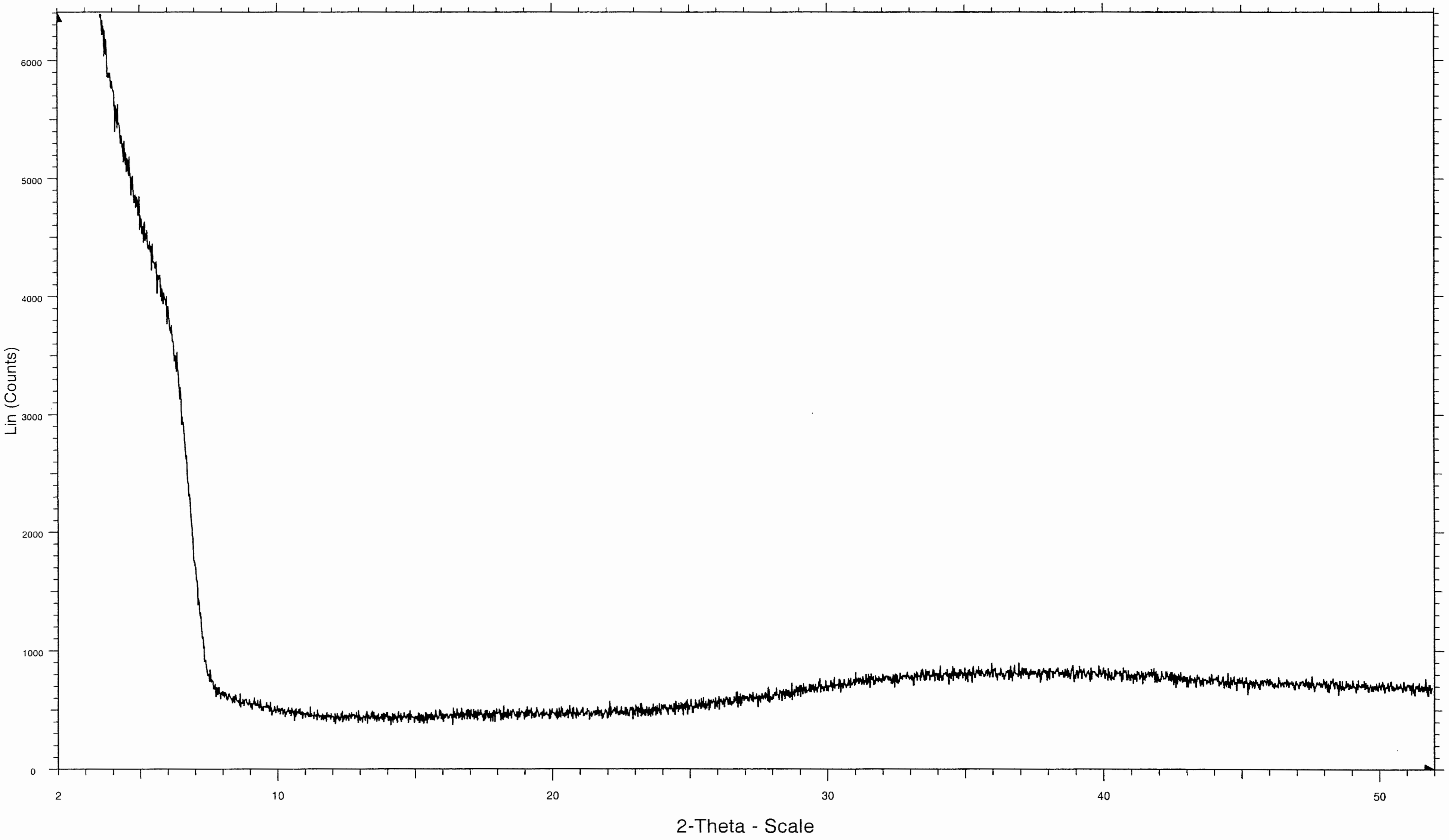

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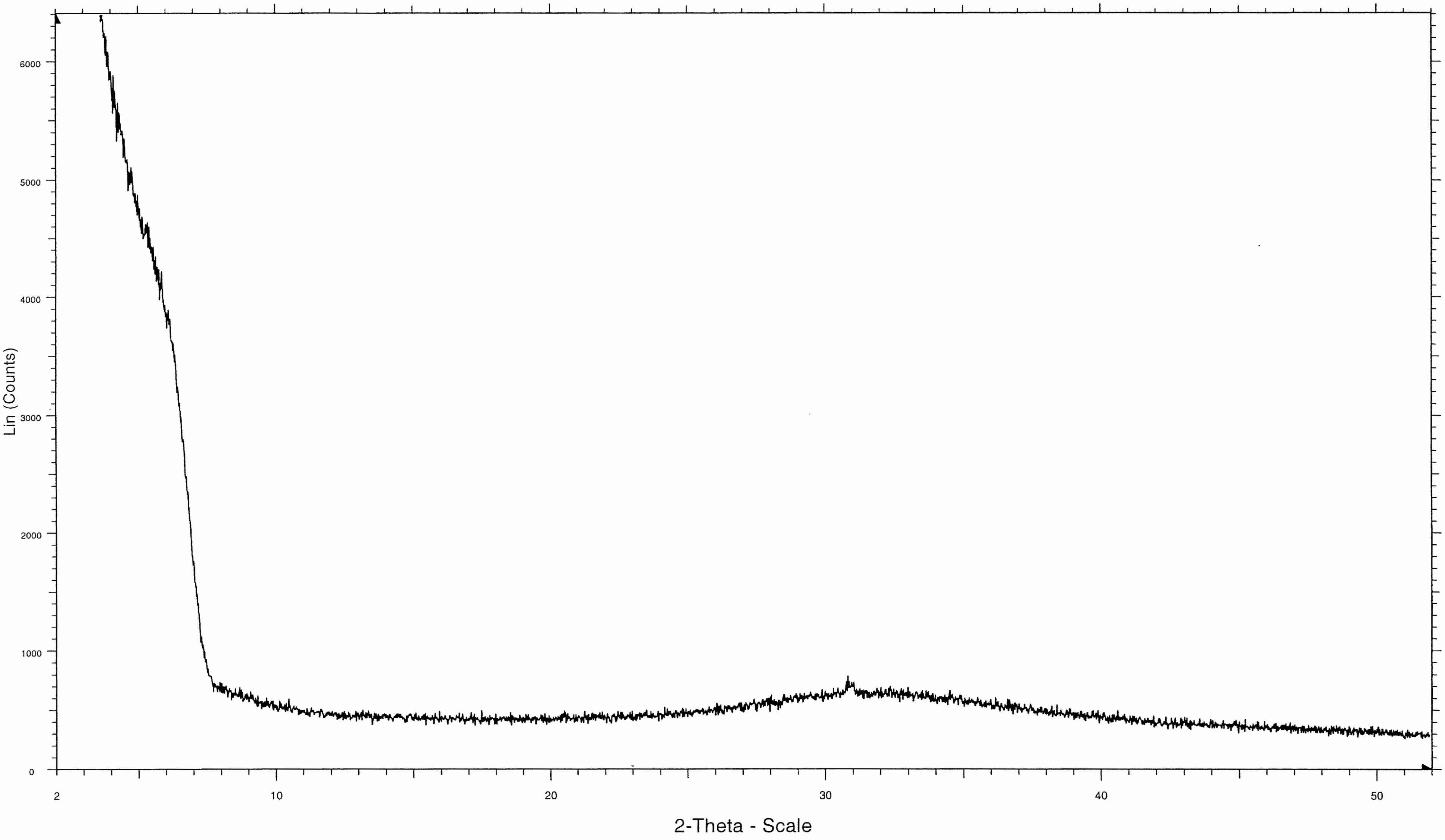

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APPENDIX C : TAILINGS CHEMISTRY

Gold District	Sample Site	Tallings Depth (cm)	Easting (20T, NAD 27)	Northing (20T, NAD 27)	Sub-sample ID #	Total Carbon (% dry wt.)	Organic Carbon (% dry wt.)	Inorganic Carbon (% dry wt.)	Ag ppb	Al (% dry wt.)	As ppm	Au ppb	B ppm	Ba ppm	Bi ppm	Ca (% dry wt.)	Cd ppm	Co ppm	Cr ppm
Cochrane Hill (CH)	T1A	0	0577211	5010992	stained pebbles														
	T1B	10	0577211	5010992	195914	0.28	0.32	-0.04	57	1.83	7964.4	68.1	1	83.9	0.6	0.21	2.29	22.3	42.2
	T1C	25-30	0577211	5010992	195915	0.33	0.35	-0.02	54	1.75	11196.7	77.4	< 1	80.7	0.54	0.21	1.85	11.2	38.7
	T1D	50	0577211	5010992	195916	0.36	0.29	0.07	81	1.8	7708.3	74.5	< 1	88.3	0.58	0.44	1.99	15.6	41.7
	T2A	0	0577183	5010955	white salts														
	T2B	15	0577183	5010955	195917	0.19	0.23	-0.04	56	1.65	4414.9	205.3	< 1	77.7	0.27	0.11	0.42	4.9	33.9
	T2C	30-35	0577183	5010955	195918	0.20	0.22	-0.02	66	1.57	2374.5	181.6	< 1	63.8	0.34	0.19	0.4	8.7	30.6
	T2D	50	0577183	5010955	195919	0.26	0.25	0.01	61	1.88	4089.9	173.5	1	87.1	0.28	0.34	1.14	22.3	35.7
	T3A	10	0577160	5010947	195920	0.16	0.20	-0.04	44	1.61	2110.1	104.7	< 1	73.9	0.16	0.11	0.35	6	33.3
	T3B	20-25	0577160	5010947	195921	0.24	0.27	-0.03	41	1.77	1940	100.7	< 1	78.6	0.21	0.25	0.72	7.2	40.1
	T3C	35-38	0577160	5010947	195922	0.15	0.18	-0.03	40	1.56	843.5	77.3	< 1	69	0.22	0.11	0.56	7.1	31.9
	T3D	50	0577160	5010947	195923	0.13	0.16	-0.03	43	1.73	392.4	71.4	< 1	81.9	0.23	0.28	1.77	30.6	35.7
	T3E	70	0577160	5010947	195924	0.30	0.22	0.08	48	1.84	780.8	56.8	1	83.8	0.22	0.54	0.47	10.6	41.3
	T4A	20	0577131	5010936	195925	0.24	0.26	-0.02	73	1.66	6604.2	195.4	< 1	75.7	0.41	0.11	0.69	3.6	38.6
	T4B	33-38	0577131	5010936	195926	0.19	0.19	0.00	31	1.5	1767.6	72.1	< 1	66	0.12	0.1	0.28	7.4	30
	T4C	60	0577131	5010936	195927	0.29	0.30	-0.01	91	1.97	4535.8	270.2	< 1	80.6	0.58	0.22	1.52	25.2	43.9
	T4D	80	0577131	5010936	195928	0.45	0.46	-0.01	116	2.15	2448.9	227.9	< 1	78.8	0.6	0.26	1.48	29	40.6
	T5A	30	0577128	5010983	195929	0.19	0.20	-0.01	36	1.56	5338.4	117.7	< 1	70.9	0.25	0.09	0.48	3.1	31.3
	T5B	64	0577128	5010983	195930	0.30	0.30	0.00	59	1.68	3832.4	162.6	< 1	66.7	0.39	0.16	0.99	16.4	36
	T5C	85	0577128	5010983	195931	0.32	0.32	0.00	85	1.91	3204.5	140.4	1	86.8	0.46	0.3	1.61	19.1	38.5
	T6A	16	0577171	5010987	195932	0.35	0.35	0.00	52	1.87	2028.8	129.5	< 1	83.6	0.24	0.08	1.05	12.9	40
	T6B	29	0577171	5010987	195933	0.37	0.40	-0.03	68	1.86	1985.6	148.5	< 1	73.1	0.38	0.19	0.7	15.4	41.4
	T6C	46	0577171	5010987	195934	0.52	0.55	-0.03	43	1.86	1031.2	20.4	< 1	65	0.18	0.11	0.63	18.2	32.5
	T6D	60	0577171	5010987	195935	0.27	0.27	0.00	56	1.7	1275	168.2	< 1	76.6	0.23	0.61	0.68	16.9	33.8
	T7A	0-25	0577203	5011011	195936	0.16	0.18	-0.02	38	1.64	3465.2	65.1	< 1	68.7	0.18	0.1	0.2	4.1	33.6
	T7B	24	0577203	5011011	195937	0.18	0.19	-0.01	57	1.59	2923.5	108.2	1	63.4	0.43	0.11	0.58	3.6	28.9
	T7C	25-26	0577203	5011011	195938	0.18	0.20	-0.02	185	1.5	4348.1	4013.1	1	51.6	0.2	0.09	1.1	4.8	27.9
	T7D	67	0577203	5011011	195939	0.22	0.24	-0.02	59	1.67	12113.3	180.8	1	61.2	0.45	0.11	1.88	19.8	39.1
	T8A	7	0577157	5011028	195940	0.20	0.19	0.01	37	1.61	3870.4	76.8	< 1	68.9	0.23	0.09	0.18	3.8	31.5
	T8B	54	0577157	5011028	195941	0.19	0.18	0.01	32	1.55	1329.1	45	< 1	63.9	0.09	0.14	1.31	8.2	31
	T8C	64	0577157	5011028	195942	0.26	0.19	0.07	32	1.56	1583.5	92.4	< 1	65.6	0.18	0.24	0.73	7.2	31.9
	T9A	13	0577114	5011019	195943	0.49	0.41	0.08	95	1.67	6800.9	230.7	1	68.5	0.54	0.14	0.81	5.1	39.6
	T9B	22	0577114	5011019	195944	0.23	0.24	-0.01	45	1.75	3821.7	142.7	< 1	71.9	0.23	0.13	0.64	5.9	34.6
	T9C	55	0577114	5011019	195945	0.37	0.26	0.11	77	1.86	2378.2	146.1	1	80.2	0.32	0.65	1.49	10	39.7
	T10A	13	0577095	5010953	195946	0.47	0.45	0.02	3543	0.25	40666.7	15646.9	< 1	23.4	9.55	0.05	0.35	1.7	7.4
	T10B	21	0577095	5010953	195947	0.75	0.75	0.00	3537	0.07	178200	25421.4	< 1	11.1	15.17	0.01	0.58	100.8	< .5
	T11A	10	0577106	5010964	195948	1.56	1.57	-0.01	244	1.59	8231.2	1127.3	< 1	111.7	0.92	0.17	0.84	2.4	28.6
	T11B	16	0577106	5010964	195949	0.48	0.39	0.09	187	1.57	13554.2	1396.7	< 1	68.2	0.7	0.14	0.42	1.5	29
	T11C	22	0577106	5010964	195950	0.39	0.38	0.01	154	1.87	3241.4	864	< 1	67.4	0.61	0.12	2.67	10.1	31.7
	T12A	3	057704	5010997	195951	2.43	2.06	0.37	131	1.55	32741.3	623.2	< 1	101.1	0.55	0.33	0.84	4.9	26.5
	T12B	10	057704	5010997	195952	3.47	3.17	0.30	142	1.33	275.2	460	< 1	46	0.36	0.15	0.19	9.8	22.3
	T13A	8	0577009	5011024	195953	0.34	0.29	0.05	82	1.73	4469.1	145.9	< 1	78.1	0.32	0.09	1.23	10	41.7
	T14A		0576950	5011065	195954	2.99	2.68	0.31	349	1.61	5618.4	2299	< 1	63	1.03	0.12	4.5	15.8	28.3
	T15A	16	0577081	5011039	195955	0.52	0.25	0.27	67	1.8	5966.6	128.4	1	77.5	0.37	0.19	1.47	10.4	42.2
	T15B	26	0577081	5011039	195956	3.62	3.31	0.31	176	1.91	3761.1	653.9	2	64.1	0.6	0.17	0.57	2.5	28.8
	T16A	7	0577065	5010930	195996	1.09	1.09	0	263	1.55	4211.7	1100.9	1	80.5	0.58	0.25	0.94	45.8	27.3
	T16B	20	0577065	5010930	195997	0.45	0.36	0.09	96	1.64	1332.7	421.7	< 1	66.6	0.28	0.14	0.46	2.5	29.7
Min.						0.13	0.16	-0.04	31	0	275	20	1	11	0	0	0	2	7
Max.						3.62	3.31	0.37	3543	2	178200	25421	2	112	15	1	5	101	44
Mean						0.60	0.56	0.04	243	2	9395	1294	1	71	1	0	1	13	34
Median						0.30	0.29	0.00	66	1.67	3821.70	148.50	1.00	71.90	0.36	0.14	0.73	9.80	33.70
n						45	45	45	45	45	45	45	12	45	45	45	45	45	44
Std Dev						0.84	0.76	0.10	714.23	0.35	26499.98	4331.12	0.28	16.36	2.55	0.14	0.79	15.94	6.65
95th						2.878	2.556	0.308	331.8	1.91	28903.88	3670.28	1.45	88.06	1.008	0.52	2.23	30.28	42.125
90th						1.372	1.378	0.206	221.2	1.876	11746.66	1288.94	1	85.64	0.664	0.336	1.868	24.04	41.61
75th						0.47	0.4	0.07	116	1.83	5966.6	421.7	1	80.5	0.58	0.24	1.47	16.4	39.625
50th						0.3	0.29	0	66	1.67	3821.7	148.5	1	71.9	0.36	0.14	0.73	9.8	33.7
25th						0.2	0.22	-0.02	45	1.57	1985.6	92.4	1	65.6	0.23	0.11	0.48	4.9	29.925

Cu ppm	Fe (% dry wt.)	Ga ppm	Hg ppb	K (% dry wt.)	La ppm	Mg (% dry wt.)	Mn ppm	Mo ppm	Na (% dry wt.)	Ni ppm	P (% dry wt.)	Pb ppm	S (% dry wt.)	Sb ppm	Sc ppm	Se ppm	Sr ppm	Te ppm	Th ppm	Ti (% dry wt.)
19.33	3.79	6.8	7	0.85	13.4	1.08	1017	0.45	0.023	41.7	0.041	393.43	0.25	3.89	4.9	0.2	16.4	0.11	3.9	0.128
16.66	4.4	6.7		0.81	13.2	1.08	754	0.45	0.019	25.1	0.04	306.37	0.37	4.48	4.2	0.2	17.7	0.1	3.7	0.119
16.77	3.78	6.9	8	0.92	12.9	1.06	766	0.44	0.024	30.6	0.041	317.55	0.53	4.36	5	0.2	11.2	0.12	3.4	0.119
13.41	3.47	6	5	0.78	8.7	1.01	287	0.42	0.014	12.2	0.035	86.5	0.08	2.25	4.3	0.1	2.8	0.05	3.4	0.117
29.41	2.87	5.4	6	0.73	8.9	0.9	268	0.33	0.015	18.3	0.033	57.87	0.2	1.4	3.7	0.1	3.7	0.03	3.3	0.103
20.92	3.87	6.6		0.9	9.9	1.12	390	0.38	0.02	50.5	0.037	80.8	0.56	1.96	4.3	0.1	5.8	0.06	2.6	0.132
10.95	3.09	5.7	10	0.84	9.8	0.96	289	0.28	0.015	12.4	0.036	61.12	0.03	1.49	4.2	< .1	3.2	0.03	3.7	0.106
18.65	3.13	6.1	6	0.7	13.8	1	326	0.36	0.017	25	0.037	71.11	0.06	1.15	4.3	0.1	4.9	0.04	3.9	0.12
17.87	2.79	5.4		0.78	6	0.91	212	0.23	0.014	15.7	0.031	50.54	0.16	0.47	4.2	< .1	2.7	< .02	2.3	0.101
15.11	3.27	6.2		0.81	30.6	0.99	274	0.34	0.019	81.8	0.039	35.69	0.33	0.3	4.5	< .1	5	< .02	3.3	0.122
13.29	3.29	6.3	7	0.91	14.7	1.05	410	0.4	0.024	27	0.041	55.58	0.25	0.39	5.1	0.1	10.1	< .02	4	0.117
16.7	3.62	5.9	10	0.77	13.9	0.99	321	1.07	0.015	12.1	0.038	175.36	0.04	3.33	4.3	0.2	3.6	0.09	4	0.12
12.75	2.62	5.2		0.71	8.3	0.87	233	0.26	0.013	10.6	0.032	49.48	0.03	1.08	3.7	0.1	2.8	0.03	3	0.092
23.34	3.92	6.4	14	0.68	22.1	1.07	575	0.54	0.02	61.2	0.045	237.41	0.15	2.21	5	0.2	12.5	0.07	5.9	0.106
36.33	4.11	6.7	20	0.61	29.4	1.17	632	0.61	0.024	63.5	0.042	244.41	0.19	1.42	4.7	0.2	10.9	0.06	8.1	0.095
10.9	3.1	5.3	6	0.75	9.5	0.93	264	0.45	0.013	7.8	0.033	78.61	0.05	2.78	3.9	0.2	3	0.06	2.8	0.107
16.9	3.38	5.5	15	0.66	14.5	0.94	397	0.45	0.016	42.3	0.035	145.79	0.19	2.32	3.9	0.2	6.9	0.06	4	0.098
21.49	3.43	6.2	10	0.74	22.9	1.04	500	0.44	0.038	51.3	0.042	210.57	0.26	1.71	5	0.1	11.3	0.03	6	0.092
18.05	3.5	6.2	11	0.78	13.8	1.07	392	0.44	0.017	20.2	0.031	88.03	0.01	1.08	4.5	0.1	2.9	0.03	3.4	0.126
20.48	3.4	6	25	0.6	19.7	0.98	519	0.51	0.021	40.9	0.036	160.24	0.09	0.99	4.3	0.1	10.7	0.04	5.2	0.086
12.41	3.35	5.8	11	0.51	15.5	0.77	553	0.48	0.01	24.8	0.031	45.48	0.09	0.59	4.2	0.5	3.9	0.02	3	0.079
13.73	3.08	5.5	7	0.7	11.3	0.93	345	0.44	0.015	42.7	0.037	53.4	0.42	0.74	4.1	0.2	7.8	0.03	3.4	0.104
12.05	3.55	5.5	7	0.79	8.9	0.98	294	0.97	0.012	9.1	0.032	50.64	0.07	1.63	3.9	0.1	2.6	0.03	3.9	0.101
13.6	2.64	5.5	6	0.67	8.2	0.9	259	0.33	0.014	9.5	0.032	92.49	0.08	2.34	4.1	0.1	3.1	0.06	3	0.095
17.53	2.59	4.7	6	0.61	8.1	0.85	269	0.27	0.012	10.7	0.028	58.31	0.16	2.71	3.5	0.1	3.5	0.05	2.4	0.088
19.35	3.71	5.3		0.65	9.1	1	349	0.44	0.014	41.1	0.033	175.87	0.67	6.47	3.9	0.3	2.9	0.14	2.6	0.11
8.31	3.45	5.5		0.68	8.3	0.97	287	0.31	0.014	8.9	0.032	52.86	0.04	1.74	4.1	0.1	2.7	0.05	3.4	0.109
10.29	2.63	5.2		0.77	7.4	0.94	367	0.11	0.016	18.7	0.027	41.07	0.12	0.72	4.5	< .1	4.6	0.02	2.2	0.11
8.18	2.58	5.4		0.72	6.9	0.95	483	0.12	0.017	16.3	0.024	38.85	0.14	0.87	4	< .1	5.8	0.02	2.5	0.118
20.55	3.86	6.1	12	0.65	16.9	0.96	368	0.71	0.017	17.8	0.041	181.97	0.04	3.52	4.2	0.2	5.6	0.09	5.2	0.09
12.63	3.31	5.9	5	0.71	11.8	1.03	352	0.36	0.016	15.6	0.029	77.03	0.03	1.64	4	0.1	4.6	0.04	3.3	0.108
24.77	3.19	6.3		0.7	20.6	1.06	602	0.46	0.031	27.2	0.039	214.06	0.19	1.03	5.5	< .1	13.7	0.03	5.5	0.103
27.37	3.65	1.9	63162	0.08	7.9	0.1	294	1.48	0.005	4.4	0.022	398.42	0.25	46.08	2.3	4.6	1.1	1.45	2.9	0.03
94.45	14.82	0.3	60166	0.02	0.7	0.02	321	3.44	0.005	96.7	0.005	87.25	8.05	148.36	1.1	8.7	2.1	2.16	0.5	< .001
23.16	3.93	5.8	7407	0.53	11.3	0.91	633	0.34	0.012	7.5	0.043	170.87	0.02	5.41	3.9	0.6	6.4	0.1	3.1	0.099
5.62	4.75	5.3	9748	0.7	9.5	1	539	0.19	0.012	4.6	0.033	63.61	< .01	2.73	3.8	0.2	6.7	0.07	3.2	0.1
31.08	3.21	6.3	6400	0.79	6.9	1.13	316	0.29	0.014	19.5	0.045	81.35	0.26	3.58	4.4	0.1	2	0.06	2.5	0.113
13.6	6.69	5.8	10024	0.53	9.6	0.89	1723	0.37	0.02	5.8	0.048	131.36	0.02	3.26	3.6	0.5	14.4	0.07	3	0.086
16.23	1.67	5.4	9062	0.34	14.2	0.62	569	0.24	0.014	7.7	0.027	57.71	0.03	0.69	2.7	0.4	3.6	0.02	2.2	0.049
15.53	3.24	5.9	14	0.81	8.2	1.07	812	0.27	0.016	25.5	0.032	135.35	0.33	2.73	4.4	< .1	2.8	0.07	2.7	0.121
32.04	3.6	5.3	11720	0.35	10	0.77	398	0.78	0.01	17.3	0.049	183.9	0.41	4.71	3.7	0.7	3.3	0.14	2.6	0.066
17.26	3.93	6.3	21	0.82	9.7	1.11	627	0.49	0.017	32.5	0.039	169.04	0.36	3.17	4.5	0.1	4.4	0.09	2.9	0.126
17.55	3.81	6	3359	0.6	14.5	1	329	0.27	0.013	7.3	0.049	178.28	0.19	2.3	3.9	0.7	3.8	0.06	3.5	0.081
41.57	3.49	5	6578	0.51	10	0.85	1436	0.38	0.012	20.9	0.035	46.92	< .01	1.42	3.3	0.3	6	0.06	3.3	0.086
30.68	2.6	5	11501	0.81	10.3	0.93	257	0.21	0.016	9.4	0.038	67.13	0.02	0.88	3.7	0.1	3.1	0.03	3.3	0.1
6	2	0	5	0	1	0	212	0	0	4	0	36	0	0	1	0	1	0	1	0
94	15	7	63162	1	31	1	1723	3	0	97	0	398	8	148	6	9	18	2	8	0
20	4	6	5696	1	12	1	480	1	0	26	0	128	0	6	4	1	6	0	3	0
17.26	3.43	5.80	12.00	0.71	10.00	0.98	368.00	0.40	0.02	18.70	0.04	86.50	0.16	1.96	4.20	0.20	4.40	0.06	3.30	0.10
45	45	45	35	45	45	45	45	45	45	45	45	45	43	45	45	38	45	42	45	44
13.46	1.83	1.11	14314.11	0.18	5.74	0.21	296.30	0.51	0.01	20.45	0.01	92.75	1.20	22.40	0.73	1.52	4.11	0.38	1.22	0.02
35.472	4.68	6.7	26253.8	0.89	22.74	1.118	976	1.05	0.024	63.04	0.0474	315.314	0.557	6.258	5	1.285	14.26	0.14	5.82	0.126
30.92	4.038	6.52	10910.2	0.832	20.24	1.08	761.2	0.752	0.0236	50.98	0.0442	241.61	0.418	4.618	4.96	0.63	12.02	0.119	5.2	0.1217
21.49	3.79	6.2	6489	0.79	14.2	1.05	569	0.46	0.019	32.5	0.041	175.87	0.26	3.26	4.4	0.275	6.9	0.085	3.9	0.11725
17.26	3.43	5.8	12	0.71	10	0.98	368	0.4	0.015	18.7	0.036	86.5	0.16	1.96	4.2	0.2	4.4	0.06	3.3	0.1035
13.41	3.1	5.4	7	0.61	8.7	0.91	294	0.29	0.013	10.6	0.032	57.71	0.045	1.08	3.9	0.1	3	0.03	2.8	0.092

Tl ppm	U ppm	V ppm	W ppm	Zn ppm
0.39	0.7	47	0.2	1005.6
0.39	0.5	45	0.2	508.2
0.43	0.5	45	0.2	579.1
0.33	0.4	43	0.1	106.5
0.29	0.5	37	0.2	134.6
0.42	0.4	47	0.2	377.6
0.34	0.4	39	0.1	104.3
0.31	0.5	44	0.3	192.9
0.31	0.3	41	< .1	202.8
0.34	0.4	43	0.2	395.1
0.32	0.6	45	0.3	148.8
0.31	0.5	42	0.2	152
0.31	0.4	38	0.1	99.6
0.34	0.9	44	0.4	579.2
0.3	1.2	45	0.5	476.9
0.32	0.4	38	0.2	99.3
0.31	0.6	38	0.2	558.7
0.32	0.9	41	0.3	471.2
0.34	0.5	47	0.2	142.5
0.27	0.7	40	0.3	328.6
0.26	0.6	43	< .1	139.3
0.3	0.6	40	0.4	270.4
0.3	0.7	41	0.2	70.6
0.29	0.4	38	0.2	169.3
0.29	0.3	37	< .1	159.4
0.35	0.4	39	0.2	558.5
0.3	0.4	39	0.1	76.5
0.32	0.3	41	0.1	289.4
0.34	0.3	39	0.3	278.1
0.31	0.6	37	0.3	170.3
0.32	0.4	43	0.2	188.9
0.31	0.8	43	0.3	492.1
0.23	< .1	23	3.9	74.4
0.89	< .1	15	2.6	146.6
0.36	0.4	39	0.3	155.8
0.34	0.4	38	0.3	60.2
0.46	0.3	43	0.2	320
0.36	0.3	37	0.4	97.7
0.2	0.5	28	0.1	51.3
0.38	0.4	44	0.2	405.4
0.28	0.4	40	0.5	265.6
0.39	0.4	45	0.2	462.5
0.34	0.5	42	< .1	99.6
0.3	0.4	36	0.2	211.3
0.33	0.4	38	< .1	88.7
0	0	15	0	51
1	1	47	4	1006
0	1	40	0	266
0.32	0.40	41.00	0.20	188.90
45	43	45	40	45
0.10	0.19	5.82	0.68	197.15
0.428	0.89	46.6	0.605	575.02
0.39	0.7	45	0.41	538.38
0.34	0.6	43	0.3	395.1
0.32	0.4	41	0.2	188.9
0.3	0.4	38	0.2	106.5

Cochrane Hill Amalgamation Tailings: Correlation Coefficients

	TC	OC	IC	Ag	Al	As	Au	B	Ba	Bi	Ca	Cd	Co	Cr	Cu	Fe	Ga	Hg	K	La	Mg	Mn	Mo	Na	Ni	P	Pb	S	Sb	Sc	Se	Sr	Te	Th	Ti	Tl	U	V	W	Zn
TC	1.00																																							
OC	1.00	1.00																																						
IC	0.75	0.69	1.00																																					
Ag	-0.27	-0.24	-0.40	1.00																																				
Al	0.21	0.18	0.36	-0.96	1.00																																			
As	-0.19	-0.17	-0.27	0.79	-0.81	1.00																																		
Au	-0.25	-0.22	-0.39	0.96	-0.95	0.92	1.00																																	
B	0.98	0.96	0.60	0.06	0.74	-0.66	0.05	1.00																																
Ba	0.09	0.08	0.18	-0.78	0.77	-0.62	-0.78	-0.99	1.00																															
Bi	-0.25	-0.23	-0.39	0.97	-0.95	0.92	1.00	0.57	-0.78	1.00																														
Ca	0.34	0.31	0.49	-0.64	0.58	-0.48	-0.65	-0.69	0.76	-0.65	1.00																													
Cd	0.13	0.11	0.19	-0.23	0.31	-0.20	-0.19	-0.81	0.13	-0.21	-0.05	1.00																												
Co	-0.15	-0.13	-0.33	0.56	-0.60	0.85	0.72	-0.64	-0.51	0.71	-0.34	-0.03	1.00																											
Cr	-0.20	-0.23	0.09	-0.78	0.85	-0.63	-0.79	-0.42	0.56	-0.77	0.18	0.26	0.11	1.00																										
Cu	-0.20	-0.16	-0.42	0.67	-0.66	0.85	0.80	-0.49	-0.58	0.79	-0.48	0.04	0.93	-0.21	1.00																									
Fe	-0.15	-0.14	-0.18	0.63	-0.64	0.96	0.79	0.25	-0.42	0.79	-0.30	-0.14	0.83	0.02	0.79	1.00																								
Ga	0.25	0.22	0.41	-0.96	0.97	-0.86	-0.97	0.30	0.80	-0.97	0.62	0.27	-0.69	0.83	-0.77	-0.71	1.00																							
Hg	-0.24	-0.21	-0.37	0.99	-0.97	0.77	0.95	0.01	-0.78	0.95	-0.61	-0.23	0.51	-0.86	0.64	0.60	-0.96	1.00																						
K	-0.25	-0.28	0.07	-0.81	0.87	-0.66	-0.80	-0.24	0.66	-0.79	0.39	0.10	-0.53	0.87	-0.59	-0.52	0.82	-0.83	1.00																					
La	0.63	0.61	0.54	-0.66	0.63	-0.80	-0.76	1.00	0.46	-0.75	0.52	-0.10	-0.72	-0.08	-0.75	-0.78	0.69	-0.63	0.34	1.00																				
Mg	0.01	-0.01	0.24	-0.93	0.97	-0.76	-0.91	0.09	0.79	-0.90	0.54	0.25	-0.58	0.90	-0.66	-0.58	0.95	-0.95	0.94	0.50	1.00																			
Mn	0.12	0.10	0.21	-0.32	0.22	-0.13	-0.31	-0.71	0.57	-0.31	0.80	-0.11	0.01	0.11	-0.20	0.01	0.28	-0.32	0.12	0.12	0.23	1.00																		
Mo	-0.19	-0.17	-0.30	0.88	-0.87	0.96	0.97	-0.87	-0.73	0.97	-0.60	-0.07	0.83	-0.65	0.88	0.87	-0.92	0.85	-0.76	-0.79	-0.84	-0.27	1.00																	
Na	0.12	0.07	0.48	-0.83	0.78	-0.58	-0.80	-0.33	0.73	-0.79	0.73	0.00	-0.51	0.68	-0.61	-0.42	0.81	-0.81	0.78	0.46	0.80	0.50	-0.73	1.00																
Ni	-0.27	-0.26	-0.29	0.54	-0.54	0.86	0.72	-0.89	-0.51	0.72	-0.48	0.03	0.92	0.73	0.88	0.85	-0.63	0.48	-0.38	-0.80	-0.48	-0.16	0.84	-0.41	1.00															
P	0.38	0.35	0.48	-0.77	0.85	-0.75	-0.82	0.96	0.76	-0.82	0.64	0.46	-0.68	0.40	-0.64	-0.59	0.85	-0.75	0.60	0.63	0.78	0.24	-0.77	0.63	-0.66	1.00														
Pb	-0.01	-0.02	0.02	0.51	-0.38	0.02	0.32	0.55	-0.22	0.33	-0.27	0.05	-0.29	-0.51	-0.13	-0.10	-0.29	0.48	-0.39	0.00	-0.39	-0.24	0.19	-0.41	-0.21	-0.02	1.00													
S	-0.21	-0.19	-0.34	0.67	-0.69	0.96	0.84	-1.00	-0.61	0.84	-0.54	-0.13	0.99	0.31	0.96	0.94	-0.78	0.64	-0.56	-0.79	-0.65	-0.21	0.93	-0.57	0.95	-0.75	-0.20	1.00												
Sb	-0.23	-0.20	-0.36	0.86	-0.86	0.98	0.96	0.00	-0.71	0.96	-0.61	-0.18	0.84	-0.74	0.88	0.90	-0.91	0.83	-0.70	-0.80	-0.81	-0.27	0.99	-0.69	0.86	-0.81	0.10	0.96	1.00											
Sc	-0.05	-0.08	0.21	-0.84	0.92	-0.81	-0.87	0.00	0.76	-0.86	0.45	0.35	-0.70	0.91	-0.72	-0.66	0.92	-0.86	0.90	0.48	0.95	0.14	-0.83	0.72	-0.55	0.81	-0.12	-0.74	-0.83	1.00										
Se	-0.23	-0.20	-0.38	0.94	-0.93	0.94	0.99	0.94	-0.76	0.99	-0.68	-0.21	0.75	-0.84	0.82	0.83	-0.96	0.92	-0.80	-0.77	-0.90	-0.28	0.98	-0.76	0.77	-0.83	0.28	0.88	0.98	-0.88	1.00									
Sr	0.26	0.23	0.42	-0.39	0.29	-0.12	-0.36	-0.71	0.65	-0.36	0.85	-0.15	-0.19	0.07	-0.35	0.07	0.35	-0.34	0.17	0.22	0.30	0.82	-0.32	0.57	-0.30	0.41	-0.22	-0.23	-0.30	0.20	-0.35	1.00								
Te	-0.27	-0.25	-0.39	0.97	-0.95	0.90	1.00	-0.50	-0.78	1.00	-0.65	-0.21	0.69	-0.74	0.78	0.77	-0.97	0.95	-0.79	-0.75	-0.91	-0.31	0.96	-0.79	0.71	-0.82	0.36	0.82	0.95	-0.85	0.99	-0.36	1.00							
Th	0.09	0.07	0.19	-0.60	0.66	-0.84	-0.75	0.76	0.64	-0.74	0.58	-0.05	-0.80	0.02	-0.78	-0.76	0.67	-0.58	0.58	0.72	0.64	0.25	-0.83	0.48	-0.86	0.72	0.15	-0.90	-0.85	0.69	-0.79	0.32	-0.73	1.00						
Ti	-0.48	-0.49	-0.18	-0.67	0.77	-0.46	-0.68	-0.59	0.64	-0.66	0.20	0.12	0.02	0.91	-0.17	0.16	0.74	-0.73	0.95	-0.29	0.91	0.12	-0.64	0.64	0.60	0.37	-0.49	0.16	-0.62	0.93	-0.73	0.14	-0.63	0.19	1.00					
Tl	-0.32	-0.31	-0.34	0.48	-0.45	0.86	0.67	-0.06	-0.36	0.67	-0.41	-0.04	0.81	0.70	0.81	0.90	-0.55	0.43	-0.25	-0.82	-0.36	-0.16	0.78	-0.32	0.92	-0.53	-0.25	0.92	0.83	-0.45	0.72	-0.14	0.65	-0.77	0.88	1.00				
U	0.51	0.53	0.25	0.06	-0.17	-0.56	-0.09	1.00	-0.50	-0.13	-0.30	-0.34	-0.03	-0.18	-0.16	-0.55	-0.23	-0.15	-0.33	0.85	-0.42	-0.38	-0.14	-0.39	-0.18	-0.36	0.07	-0.06	-0.41	-0.42	0.34	-0.41	-0.24	0.08	-0.47	-0.67	1.00			
V	0.02	-0.01	0.26	-0.86	0.95	-0.79	-0.87	0.19	0.77	-0.87	0.47	0.40	-0.63	0.91	-0.67	-0.62	0.92	-0.89	0.89	0.50	0.96	0.17	-0.81	0.71	-0.51	0.84	-0.16	-0.70	-0.82	0.99	-0.88	0.22	-0.86	0.69	0.88	-0.42	-0.33	1.00		
W	-0.29	-0.26	-0.35	0.97	-0.92	0.63	0.87	#####	-0.74	0.88	-0.59	-0.27	0.33	-0.77	0.49	0.45	-0.89	0.97	-0.77	-0.56	-0.88	-0.39	0.75	-0.78	0.33	-0.67	0.69	0.47	0.71	-0.73	0.83	-0.40	0.89	-0.40	-0.66	0.28	-0.40	-0.77	1.00	
Zn	-0.35	-0.38	-0.06	-0.26	0.37	-0.17	-0.23	-0.74	0.25	-0.22	-0.04	0.54	0.04	0.74	-0.02	-0.11	0.35	-0.37	0.46	-0.22	0.44	0.05	-0.10	0.28	0.26	0.19	-0.01	-0.07	-0.15	0.53	-0.19	-0.22	-0.20	-0.05	0.65	0.15	-0.29	0.53	-0.35	1.00

Cochrane Hill Cyanidation Tailings: Correlation Coefficients

	TC	OC	IC	Ag	Al	As	Au	B	Ba	Bi	Ca	Cd	Co	Cr	Cu	Fe	Ga	Hg	K	La	Mg	Mn	Mo	Na	Ni	P	Pb	S	Sb	Sc	Se	Sr	Te	Th	Ti	Tl	U	V	W	Zn
TC	1.00																																							
OC	0.92	1.00																																						
IC	0.38	0.01	1.00																																					
Ag	0.27	0.26	0.08	1.00																																				
Al	0.66	0.68	0.09	0.23	1.00																																			
As	0.11	0.13	-0.02	0.21	0.01	1.00																																		
Au	-0.14	-0.12	-0.09	0.78	-0.24	0.06	1.00																																	
B	#####	#####	#####	#####	#####	#####	#####	1.00																																
Ba	0.27	0.22	0.19	-0.11	0.69	0.05	-0.44	#####	1.00																															
Bi	0.47	0.48	0.09	0.43	0.53	0.63	-0.10	#####	0.37	1.00																														
Ca	0.28	0.05	0.59	0.08	0.42	-0.17	-0.14	#####	0.54	0.13	1.00																													
Cd	0.28	0.26	0.10	0.32	0.48	0.50	0.05	#####	0.42	0.63	0.30	1.00																												
Co	0.34	0.42	-0.14	0.17	0.71	0.03	-0.15	#####	0.47	0.44	0.34	0.68	1.00																											
Cr	0.55	0.50	0.28	0.15	0.77	0.33	-0.30	#####	0.70	0.65	0.40	0.57	0.52	1.00																										
Cu	0.42	0.45	0.02	0.55	0.60	0.11	0.07	#####	0.25	0.61	0.24	0.41	0.50	0.48	1.00																									
Fe	0.51	0.56	0.00	0.16	0.67	0.60	-0.26	#####	0.58	0.70	0.11	0.48	0.47	0.73	0.40	1.00																								
Ga	0.52	0.48	0.25	0.06	0.80	0.24	-0.38	#####	0.88	0.62	0.44	0.56	0.54	0.80	0.43	0.76	1.00																							
Hg	0.58	0.64	-0.05	0.20	0.58	-0.15	-0.14	#####	0.10	0.39	-0.03	0.20	0.55	0.48	0.50	0.47	0.29	1.00																						
K	-0.35	-0.45	0.19	-0.30	-0.03	0.12	-0.26	#####	0.60	-0.01	0.28	0.17	-0.01	0.19	-0.19	0.16	0.40	-0.44	1.00																					
La	0.48	0.49	0.09	0.30	0.75	-0.12	-0.13	#####	0.50	0.45	0.34	0.47	0.70	0.61	0.56	0.46	0.59	0.74	-0.16	1.00																				
Mg	0.22	0.15	0.18	0.11	0.69	0.30	-0.27	#####	0.77	0.53	0.38	0.53	0.44	0.74	0.46	0.70	0.79	0.24	0.46	0.45	1.00																			
Mn	0.58	0.55	0.25	0.17	0.61	0.39	-0.15	#####	0.50	0.65	0.35	0.72	0.49	0.64	0.32	0.57	0.72	0.30	0.13	0.37	0.51	1.00																		
Mo	0.30	0.35	-0.02	0.19	0.30	0.32	-0.12	#####	0.20	0.40	-0.03	-0.01	0.02	0.42	0.25	0.57	0.28	0.23	-0.05	0.30	0.24	0.12	1.00																	
Na	0.37	0.22	0.41	0.23	0.63	0.01	-0.17	#####	0.68	0.46	0.62	0.55	0.43	0.61	0.47	0.32	0.63	0.27	0.25	0.60	0.63	0.55	0.04	1.00																
Ni	0.26	0.31	-0.09	0.19	0.68	-0.02	-0.14	#####	0.50	0.43	0.41	0.64	0.95	0.54	0.51	0.44	0.52	0.60	0.01	0.78	0.48	0.38	0.05	0.51	1.00															
P	0.40	0.38	0.17	0.27	0.66	0.27	-0.23	#####	0.72	0.70	0.48	0.47	0.52	0.77	0.53	0.69	0.77	0.33	0.25	0.67	0.60	0.50	0.43	0.62	0.59	1.00														
Pb	0.46	0.46	0.16	0.34	0.52	0.66	-0.10	#####	0.47	0.89	0.22	0.75	0.40	0.71	0.49	0.69	0.69	0.32	0.13	0.39	0.59	0.85	0.33	0.56	0.36	0.67	1.00													
S	0.05	0.00	0.11	0.13	0.23	0.44	-0.02	#####	0.29	0.35	0.49	0.62	0.56	0.28	0.24	0.35	0.30	-0.06	0.31	0.09	0.34	0.32	-0.09	0.29	0.54	0.31	0.34	1.00												
Sb	0.07	0.10	-0.02	0.27	-0.05	0.97	0.12	#####	-0.04	0.63	-0.20	0.48	0.02	0.28	0.11	0.51	0.16	-0.17	0.08	-0.15	0.22	0.32	0.29	-0.02	-0.03	0.23	0.62	0.45	1.00											
Sc	0.41	0.25	0.48	0.07	0.69	-0.05	-0.32	#####	0.76	0.40	0.62	0.52	0.43	0.71	0.32	0.38	0.74	0.22	0.33	0.58	0.63	0.60	0.11	0.80	0.45	0.66	0.53	0.19	-0.09	1.00										
Se	0.56	0.62	-0.01	-0.04	0.21	0.26	-0.15	#####	-0.14	0.21	-0.05	0.28	0.38	0.12	-0.01	0.29	0.07	0.17	-0.39	0.20	-0.28	0.35	0.20	-0.21	0.21	0.08	0.22	0.22	0.27	0.04	1.00									
Sr	0.48	0.41	0.29	0.23	0.62	0.30	-0.12	#####	0.58	0.65	0.56	0.67	0.48	0.67	0.41	0.54	0.71	0.39	0.17	0.50	0.59	0.87	0.10	0.71	0.49	0.67	0.80	0.35	0.20	0.66	0.10	1.00								
Te	0.12	0.13	0.07	0.23	0.13	0.92	-0.01	#####	0.19	0.74	-0.03	0.61	0.26	0.49	0.20	0.61	0.38	0.00	0.24	0.06	0.37	0.42	0.31	0.07	0.24	0.46	0.71	0.55	0.95	0.14	0.28	0.33	1.00							
Th	0.56	0.54	0.15	0.39	0.74	-0.01	-0.15	#####	0.42	0.59	0.29	0.25	0.41	0.63	0.68	0.52	0.54	0.69	-0.23	0.77	0.54	0.41	0.46	0.63	0.47	0.70	0.52	-0.10	-0.04	0.54	-0.02	0.55	0.04	1.00						
Ti	-0.28	-0.33	0.08	-0.32	0.08	0.24	-0.27	#####	0.56	0.04	0.19	0.30	0.12	0.31	-0.13	0.27	0.43	-0.41	0.76	-0.10	0.57	0.19	-0.06	0.11	0.13	0.14	0.15	0.33	0.18	0.27	-0.24	0.14	0.32	-0.26	1.00					
Tl	-0.04	-0.11	0.17	-0.07	0.19	0.52	-0.17	#####	0.58	0.36	0.22	0.60	0.31	0.38	0.00	0.47	0.58	-0.24	0.74	-0.05	0.60	0.48	-0.10	0.29	0.26	0.32	0.50	0.61	0.48	0.32	-0.04	0.39	0.56	-0.15	0.74	1.00				
U	0.60	0.61	0.08	0.34	0.80	-0.01	-0.17	#####	0.43	0.58	0.36	0.31	0.53	0.62	0.68	0.53	0.54	0.62	-0.23	0.73	0.49	0.52	0.49	0.60	0.53	0.68	0.55	0.03	-0.04	0.56	0.16	0.60	0.03	0.93	-0.27	-0.13	1.00			
V	0.30	0.32	0.06	-0.08	0.71	0.07	-0.27	#####	0.79	0.25	0.34	0.48	0.49	0.63	0.23	0.57	0.82	0.04	0.46	0.36	0.71	0.58	0.15	0.41	0.41	0.44	0.41	0.27	-0.02	0.65	0.06	0.49	0.19	0.22	0.61	0.55	0.32	1.00		
W	0.63	0.59	0.21	0.64	0.69	-0.14	0.48	#####	0.29	0.44	0.51	0.24	0.48	0.53	0.60	0.28	0.37	0.49	-0.38	0.61	0.41	0.33	0.26	0.45	0.54	0.49	0.29	0.16	-0.15	0.39	0.27	0.48	0.00	0.71	-0.25	-0.19	0.75	0.20	1.00	
Zn	0.27	0.27	0.08	0.20	0.49	0.45	-0.11	#####	0.41	0.69	0.33	0.88	0.69	0.60	0.41	0.48	0.56	0.28	0.15	0.40	0.53	0.80	0.01	0.55	0.65	0.51	0.81	0.57	0.43	0.51	0.25	0.76	0.56	0.32	0.26	0.55	0.43	0.42	0.29	1.00

APPENDIX D : WATER CHEMISTRY

Sample	Li ICP-MS DIRECT ppb	Be ICP-MS DIRECT ppb	B ICP-MS DIRECT ppb	Al ICP-MS DIRECT ppb	Ti ICP-MS DIRECT ppb	V ICP-MS DIRECT ppb	Cr ICP-MS DIRECT ppb	Mn ICP-MS DIRECT ppb	Co ICP-MS DIRECT ppb	Ni ICP-MS DIRECT ppb	Cu ICP-MS DIRECT ppb	Zn ICP-MS DIRECT ppb	As ICP-MS DIRECT ppb	Se ICP-MS DIRECT ppb	Rb ICP-MS DIRECT ppb	Sr ICP-MS DIRECT ppb	Y ICP-MS DIRECT ppb	Mo ICP-MS DIRECT ppb	Ag ICP-MS DIRECT ppb	Cd ICP-MS DIRECT ppb
D.L.	0.02	0.005	0.5	2	0.5	0.1	0.1	0.1	0.05	0.2	0.1	0.5	0.1	1	0.05	0.5	0.01	0.05	0.005	0.02
CH 03 TB C	< 0.02	< 0.005	< 0.5	< 2	< 0.5	< 0.1	< 0.1	< 0.1	< 0.05	< 0.2	< 0.1	< 0.5	< 0.1	< 1	< 0.05	< 0.5	< 0.01	< 0.05	< 0.005	< 0.02
CH 03 AB C	< 0.02	< 0.005	< 0.5	< 2	< 0.5	< 0.1	0.1	0.1	< 0.05	< 0.2	< 0.1	3.6	< 0.1	< 1	< 0.05	< 0.5	< 0.01	< 0.05	< 0.005	< 0.02
CH 03 SB C	< 0.02	< 0.005	< 0.5	< 2	< 0.5	< 0.1	0.1	< 0.1	< 0.05	< 0.2	< 0.1	2.3	< 0.1	< 1	< 0.05	< 0.5	< 0.01	< 0.05	< 0.005	< 0.02
CH 03 W1 FA C	0.55	0.019	4.1	72	< 0.5	< 0.1	< 0.1	23.1	< 0.05	0.6	0.3	9.4	15.4	< 1	1.00	10.9	0.07	< 0.05	< 0.005	0.02
CH 03 W2 FA C	0.64	0.015	4.3	89	0.8	0.1	0.1	40.4	0.11	0.9	0.3	15.6	27.3	< 1	1.26	11.8	0.10	< 0.05	< 0.005	0.02
CH 03 W3 FA C	0.98	0.019	4.6	105	1.0	0.1	0.1	164.4	0.42	1.5	0.5	32.9	41.5	< 1	1.61	17.2	0.09	< 0.05	< 0.005	0.03
CH 03 W4 FA C	0.32	0.011	3.2	53	< 0.5	< 0.1	< 0.1	25.5	0.06	0.2	0.2	2.2	0.9	< 1	0.74	5.9	0.05	< 0.05	< 0.005	< 0.02
CH 03 W5 FA C	1.35	0.025	5.5	121	1.1	0.1	0.2	241.1	0.60	2.1	0.5	52.4	44.7	< 1	2.08	25.1	0.11	< 0.05	< 0.005	0.05
CH 03 W6 FA C	0.35	0.008	4.6	91	< 0.5	< 0.1	< 0.1	263.6	0.26	0.5	0.2	4.7	45.2	< 1	1.25	9.0	0.04	< 0.05	< 0.005	< 0.02
CH 03 W6D FA C	0.35	0.009	4.2	91	< 0.5	0.1	0.1	266.0	0.27	0.4	0.2	4.8	44.5	< 1	1.29	9.1	0.04	< 0.05	< 0.005	0.02
CH 03 W7 FA C	0.60	0.007	5.9	114	1.0	0.2	0.3	620.7	0.89	0.5	0.6	2.4	13.6	< 1	2.47	13.2	0.06	< 0.05	< 0.005	< 0.02
CH 03 W8 FA C	18.94	0.763	13.8	2746	2.2	< 0.1	0.2	4300.1	74.29	146.5	15.6	1623.9	47.5	< 1	5.09	87.4	3.49	< 0.05	< 0.005	4.14
CH 03 W9 FA C	0.32	0.005	4.0	85	< 0.5	< 0.1	0.1	259.1	0.22	0.5	0.2	4.5	23.1	< 1	1.23	9.2	0.04	< 0.05	< 0.005	< 0.02
CH 03 W10 FA C	2.84	0.020	4.9	66	1.0	0.4	0.6	795.7	12.20	70.2	0.3	37.6	6575.9	< 1	7.46	16.9	0.25	0.77	< 0.005	0.02
CH 03 W11 FA C	1.52	< 0.005	17.1	12	3.0	0.2	0.3	5954.6	7.20	3.8	0.3	9.2	775.6	< 1	9.29	187.1	0.11	0.41	< 0.005	0.03
CH 03 W12 FA C	26.75	0.137	15.4	55	4.6	< 0.1	< 0.1	3649.6	51.87	173.6	0.3	967.5	519.7	< 1	12.30	207.0	2.60	0.65	< 0.005	2.25
CH 03 W1 UA C	0.55	0.014	4.2	78	0.5	< 0.1	< 0.1	31.1	0.06	0.6	0.3	9.3	16.5	< 1	1.07	11.0	0.08	< 0.05	< 0.005	< 0.02
CH 03 W2 UA C	0.65	0.015	4.0	97	0.8	< 0.1	0.1	47.0	0.12	1.0	0.5	16.0	31.0	< 1	1.36	12.2	0.10	< 0.05	< 0.005	0.03
CH 03 W3 UA C	1.01	0.016	4.7	124	1.4	0.1	0.2	180.2	0.44	1.7	0.5	34.0	55.6	< 1	1.78	18.0	0.10	< 0.05	< 0.005	0.03
CH 03 W4 UA C	0.31	< 0.005	3.3	56	< 0.5	< 0.1	< 0.1	28.6	0.06	0.3	0.2	2.3	1.3	< 1	0.81	6.2	0.06	< 0.05	< 0.005	< 0.02
CH 03 W5 UA C	1.35	0.017	5.1	152	1.6	0.2	0.2	257.2	0.63	2.5	0.7	53.1	75.8	< 1	2.24	24.4	0.13	< 0.05	< 0.005	0.06
CH 03 W6 UA C	0.36	< 0.005	4.2	91	< 0.5	< 0.1	0.1	261.9	0.24	0.4	0.2	4.4	44.4	< 1	1.26	8.9	0.04	< 0.05	< 0.005	< 0.02
CH 03 W6D UA C	0.37	0.007	4.6	98	0.7	0.1	0.2	275.9	0.27	0.5	0.2	4.8	48.0	< 1	1.38	9.6	0.05	< 0.05	< 0.005	< 0.02
CH 03 W7 UA C	0.67	0.007	6.0	190	3.7	0.4	0.4	631.0	1.04	0.6	0.4	3.8	28.0	< 1	2.63	13.9	0.07	< 0.05	< 0.005	< 0.02
CH 03 W8 UA C	18.52	0.863	18.3	2822	3.8	< 0.1	0.3	4389.2	76.86	165.3	17.1	1663.8	80.2	< 1	5.37	91.3	3.63	< 0.05	< 0.005	4.14
CH 03 W9 UA C	0.34	0.007	4.3	87	< 0.5	0.1	0.1	268.3	0.21	0.4	0.3	4.7	23.8	< 1	1.29	9.7	0.05	< 0.05	< 0.005	< 0.02
CH 03 W10 UA C	10.89	0.572	5.9	6116	173.7	10.5	15.8	1258.7	22.19	133.6	14.3	375.1	6072.4	< 1	20.12	25.2	3.16	0.21	< 0.005	1.51
CH 03 W11 UA C	1.62	0.014	17.2	78	2.2	0.5	0.5	5716.5	8.48	4.3	1.0	14.2	4286.2	< 1	9.37	192.6	0.43	0.62	< 0.005	0.05
CH 03 W12 UA C	36.66	0.482	14.6	4150	152.0	6.4	5.9	3852.2	63.50	215.3	8.0	1254.8	1051.2	< 1	20.53	204.1	4.24	0.55	0.005	2.80
REPEATS																				
CH 03 AB C	< 0.02	< 0.005	< 0.5	< 2	< 0.5	< 0.1	0.1	0.1	< 0.05	< 0.2	< 0.1	3.6	< 0.1	< 1	< 0.05	< 0.5	< 0.01	< 0.05	< 0.005	< 0.02
CH 03 AB C REP	< 0.02	< 0.005	< 0.5	< 2	< 0.5	< 0.1	< 0.1	0.1	< 0.05	< 0.2	< 0.1	3.9	< 0.1	< 1	< 0.05	< 0.5	< 0.01	< 0.05	< 0.005	< 0.02
CH 03 W8 FA C	18.94	0.763	13.8	2746	2.2	< 0.1	0.2	4300.1	74.29	146.5	15.6	1623.9	47.5	< 1	5.09	87.4	3.49	< 0.05	< 0.005	4.14
CH 03 W8 FA C REP	19.82	0.811	15.0	2831	2.4	< 0.1	0.2	4468.3	77.86	155.0	16.1	1647.8	47.5	< 1	5.34	89.4	3.56	< 0.05	< 0.005	4.13
CH 03 W9 FA C	0.32	0.005	4.0	85	< 0.5	< 0.1	0.1	259.1	0.22	0.5	0.2	4.5	23.1	< 1	1.23	9.2	0.04	< 0.05	< 0.005	< 0.02
CH 03 W9 FA C REP	0.35	0.009	3.7	85	< 0.5	0.1	0.1	268.4	0.20	0.4	0.2	4.6	23.1	< 1	1.28	9.5	0.05	< 0.05	< 0.005	< 0.02
CH 03 W5 UA C	1.35	0.017	5.1	152	1.6	0.2	0.2	257.2	0.63	2.5	0.7	53.1	75.8	< 1	2.24	24.4	0.13	< 0.05	< 0.005	0.06
CH 03 W5 UA C REP	1.41	0.025	5.1	156	1.5	0.2	0.2	246.9	0.64	2.3	0.7	52.0	76.3	< 1	2.20	24.2	0.13	< 0.05	< 0.005	0.05
CONTROLS																				
BLANK	< 0.02	< 0.005	< 0.5	< 2	< 0.5	< 0.1	< 0.1	< 0.1	< 0.05	< 0.2	< 0.1	< 0.5	< 0.1	< 1	< 0.05	< 0.5	< 0.01	< 0.05	< 0.005	< 0.02
BLANK	< 0.02	< 0.005	< 0.5	< 2	< 0.5	< 0.1	< 0.1	< 0.1	< 0.05	< 0.2	< 0.1	< 0.5	< 0.1	< 1	< 0.05	< 0.5	< 0.01	< 0.05	< 0.005	< 0.02
BLANK	< 0.02	< 0.005	< 0.5	< 2	< 0.5	< 0.1	< 0.1	< 0.1	< 0.05	< 0.2	< 0.1	< 0.5	< 0.1	< 1	< 0.05	< 0.5	< 0.01	< 0.05	< 0.005	< 0.02
SLRS-4	0.51	0.006	4.6	52	1.2	0.3	0.3	3.2	< 0.05	0.6	1.6	0.9	0.7	< 1	1.46	27.5	0.13	0.20	< 0.005	< 0.02
SLRS-4	0.48	0.006	4.3	51	1.3	0.3	0.3	3.2	< 0.05	0.6	1.6	0.9	0.6	< 1	1.44	26.7	0.13	0.18	< 0.005	< 0.02
SLRS-4	0.49	< 0.005	4.5	54	1.2	0.3	0.3	3.2	< 0.05	0.7	1.7	1.0	0.6	< 1	1.46	28.3	0.14	0.19	< 0.005	< 0.02
CERTIFIED		0.007 ± 0.002		54 ± 4		0.32 ± 0.03	0.33 ± 0.02	3.37 ± 0.18	0.033 ± 0.006	0.67± 0.08	1.81 ± 0.08	0.93 ± 0.1	0.68 ± 0.06			26.3 ± 3.2		0.21 ± 0.02		0.012 ± 0.002
TMRAIN-95	0.39	0.254	23.8	2	< 0.5	0.6	0.8	5.9	0.22	0.8	5.8	12.4	0.9	< 1	< 0.05	1.6	< 0.01	0.15	0.135	0.46
CERTIFIED	0.39 ± 0.078	0.27 ± 0.061		1.7 ± 0.91	0.47	0.64 ± 0.12	0.79 ± 0.17	6.1 ±0.78	0.22 ± 0.037	0.80 ± 0.17	6.2 ± 0.93	11.1 ± 2.36	1.07± 0.25	0.74 ± 0.29		1.7 ± 0.26		0.17 ± 0.10		0.48 ± 0.12
OTT 2K	0.54	< 0.005	7.3	60	1.1	0.4	0.3	2.1	< 0.05	0.6	1.1	0.6	0.5	< 1	1.54	46.4	0.12	0.22	< 0.005	< 0.02
OTT 2K	0.53	< 0.005	6.6	60	1.2	0.3	0.3	2.1	< 0.05	0.6	1.0	0.6	0.5	< 1	1.54	45.0	0.12	0.23	< 0.005	< 0.02
OTT 2K	0.50	< 0.005	6.1	60	1.2	0.3	0.3	2.1	< 0.05	0.6	1.0	0.6	0.6	< 1	1.51	44.9	0.12	0.19	< 0.005	< 0.02
OTT 2K	0.56	0.007	8.1	65	1.2	0.4	0.3	2.2	< 0.05	0.7	1.1	0.7	0.7	< 1	1.62	46.1	0.13	0.21	< 0.005	< 0.02

In	Sb	Cs	Ba	La	Ce	Pr	Nd	Sm	Eu	Tb	Gd	Dy	Ho	Er	Tm	Yb	Lu	Re	Tl	Pb	U
ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS
DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT
ppb	ppb	ppb	ppb	ppb	ppb	ppb	ppb	ppb	ppb	ppb	ppb	ppb	ppb	ppb	ppb	ppb	ppb	ppb	ppb	ppb	ppb
0.01	0.01	0.01	0.2	0.01	0.01	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.005	0.01	0.005
< 0.01	< 0.01	< 0.01	< 0.2	< 0.01	< 0.01	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.01	< 0.005
< 0.01	< 0.01	< 0.01	0.3	< 0.01	< 0.01	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.01	< 0.005
< 0.01	0.01	< 0.01	< 0.2	< 0.01	< 0.01	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.01	< 0.005
< 0.01	0.02	0.01	6.5	0.06	0.07	0.016	0.068	0.016	< 0.005	< 0.005	0.018	0.016	< 0.005	0.006	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	0.11	< 0.005
< 0.01	0.04	0.02	5.5	0.07	0.09	0.020	0.083	0.021	< 0.005	< 0.005	0.021	0.017	< 0.005	0.012	< 0.005	0.006	< 0.005	< 0.005	0.005	0.21	< 0.005
< 0.01	0.05	0.03	5.6	0.07	0.14	0.023	0.103	0.018	0.006	< 0.005	0.024	0.016	< 0.005	0.010	< 0.005	0.006	< 0.005	< 0.005	< 0.005	0.40	0.008
< 0.01	0.01	< 0.01	4.7	0.03	0.05	0.008	0.031	0.008	< 0.005	< 0.005	0.010	0.007	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	0.07	< 0.005
< 0.01	0.07	0.04	6.2	0.09	0.17	0.023	0.111	0.031	0.006	< 0.005	0.027	0.020	< 0.005	0.009	< 0.005	0.009	< 0.005	< 0.005	0.005	0.29	0.006
< 0.01	0.03	0.01	3.7	0.03	0.06	0.009	0.039	0.012	< 0.005	< 0.005	0.013	0.009	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	0.006	0.15	< 0.005
< 0.01	0.03	0.01	3.7	0.03	0.05	0.009	0.044	0.011	0.005	< 0.005	0.008	0.007	< 0.005	0.009	< 0.005	< 0.005	< 0.005	< 0.005	0.008	0.15	< 0.005
< 0.01	0.04	0.03	3.3	0.03	0.06	0.009	0.039	0.011	< 0.005	< 0.005	0.008	0.011	< 0.005	< 0.005	< 0.005	0.007	< 0.005	< 0.005	< 0.005	0.11	< 0.005
< 0.01	0.46	0.03	7.5	7.39	13.26	1.472	5.599	0.885	0.210	0.121	0.963	0.645	0.106	0.244	0.029	0.163	0.021	< 0.005	0.014	0.73	0.079
< 0.01	0.02	0.01	3.6	0.02	0.05	0.011	0.033	0.007	< 0.005	< 0.005	0.008	0.010	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	0.005	0.14	< 0.005
< 0.01	2.01	0.04	10.7	0.22	0.58	0.074	0.343	0.065	0.017	0.008	0.075	0.047	0.010	0.021	< 0.005	0.018	< 0.005	< 0.005	< 0.005	3.75	0.021
< 0.01	0.17	0.05	28.4	0.09	0.19	0.020	0.090	0.019	0.006	< 0.005	0.021	0.017	< 0.005	0.011	< 0.005	0.008	< 0.005	< 0.005	< 0.005	0.40	0.024
< 0.01	0.51	0.11	19.6	7.54	11.94	1.215	4.448	0.633	0.127	0.081	0.669	0.408	0.073	0.151	0.017	0.074	0.012	< 0.005	0.018	0.28	0.020
< 0.01	0.02	0.01	6.6	0.06	0.08	0.019	0.075	0.019	0.005	< 0.005	0.019	0.015	< 0.005	0.007	< 0.005	0.006	< 0.005	< 0.005	< 0.005	0.15	< 0.005
< 0.01	0.04	0.02	5.6	0.08	0.11	0.022	0.100	0.025	0.008	< 0.005	0.018	0.019	< 0.005	0.009	< 0.005	0.008	< 0.005	< 0.005	< 0.005	0.28	< 0.005
< 0.01	0.05	0.03	5.7	0.08	0.17	0.024	0.097	0.026	0.007	< 0.005	0.026	0.018	< 0.005	0.012	< 0.005	0.006	< 0.005	< 0.005	< 0.005	0.60	0.010
< 0.01	< 0.01	< 0.01	4.7	0.03	0.05	0.009	0.044	0.011	< 0.005	< 0.005	0.009	0.008	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	0.09	< 0.005
< 0.01	0.07	0.04	6.0	0.11	0.22	0.033	0.132	0.035	0.008	< 0.005	0.023	0.025	< 0.005	0.014	< 0.005	0.012	< 0.005	< 0.005	0.006	0.83	0.009
< 0.01	0.03	0.01	3.6	0.03	0.05	0.010	0.038	0.014	< 0.005	< 0.005	0.008	0.008	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	0.005	0.16	< 0.005
< 0.01	0.04	0.01	3.8	0.03	0.05	0.010	0.035	0.007	< 0.005	< 0.005	0.010	0.006	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	0.18	< 0.005
< 0.01	0.04	0.04	3.8	0.05	0.21	0.019	0.061	0.016	< 0.005	< 0.005	0.015	0.015	< 0.005	0.007	< 0.005	0.006	< 0.005	< 0.005	0.006	0.27	0.007
< 0.01	0.47	0.04	7.8	7.54	13.58	1.436	5.719	0.935	0.215	0.116	0.956	0.616	0.111	0.256	0.031	0.164	0.023	< 0.005	0.010	1.58	0.079
< 0.01	0.02	0.01	3.6	0.02	0.05	0.009	0.038	0.012	< 0.005	< 0.005	0.010	0.008	< 0.005	< 0.005	< 0.005	0.006	< 0.005	< 0.005	< 0.005	0.15	< 0.005
0.02	1.25	0.98	47.6	8.23	23.51	2.152	8.552	1.687	0.414	0.178	1.348	0.860	0.131	0.285	0.038	0.212	0.030	< 0.005	0.118	398.75	0.389
< 0.01	0.32	0.05	37.4	0.77	1.42	0.145	0.579	0.097	0.026	0.012	0.120	0.069	0.012	0.028	< 0.005	0.023	< 0.005	< 0.005	< 0.005	2.70	0.059
< 0.01	0.55	0.93	36.5	10.66	23.18	1.958	7.499	1.301	0.280	0.161	1.328	0.853	0.148	0.309	0.039	0.201	0.029	< 0.005	0.065	28.35	0.190
< 0.01	< 0.01	< 0.01	0.3	< 0.01	< 0.01	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.01	< 0.005
< 0.01	0.01	< 0.01	0.3	< 0.01	< 0.01	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.01	< 0.005
< 0.01	0.46	0.03	7.5	7.39	13.26	1.472	5.599	0.885	0.210	0.121	0.963	0.645	0.106	0.244	0.029	0.163	0.021	< 0.005	0.014	0.73	0.079
< 0.01	0.46	0.03	7.4	7.68	13.53	1.493	5.636	0.917	0.219	0.117	1.003	0.646	0.109	0.254	0.027	0.153	0.022	< 0.005	0.014	0.70	0.081
< 0.01	0.02	0.01	3.6	0.02	0.05	0.011	0.033	0.007	< 0.005	< 0.005	0.008	0.010	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	0.005	0.14	< 0.005
< 0.01	0.03	0.01	3.7	0.03	0.05	0.009	0.034	0.013	< 0.005	< 0.005	0.006	0.005	< 0.005	0.006	< 0.005	< 0.005	< 0.005	< 0.005	0.007	0.14	< 0.005
< 0.01	0.07	0.04	6.0	0.11	0.22	0.033	0.132	0.035	0.008	< 0.005	0.023	0.025	< 0.005	0.014	< 0.005	0.012	< 0.005	< 0.005	0.006	0.83	0.009
< 0.01	0.06	0.05	6.4	0.12	0.22	0.033	0.146	0.037	0.009	< 0.005	0.032	0.023	< 0.005	0.016	< 0.005	0.011	< 0.005	< 0.005	< 0.005	0.83	0.009
< 0.01	< 0.01	< 0.01	< 0.2	< 0.01	< 0.01	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.01	< 0.005
< 0.01	< 0.01	< 0.01	< 0.2	< 0.01	< 0.01	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.01	< 0.005
< 0.01	< 0.01	< 0.01	< 0.2	< 0.01	< 0.01	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.01	< 0.005
< 0.01	0.25	< 0.01	12.8	0.28	0.37	0.073	0.258	0.049	0.008	< 0.005	0.033	0.029	< 0.005	0.011	< 0.005	0.009	< 0.005	0.007	0.005	0.07	0.048
< 0.01	0.24	< 0.01	12.5	0.28	0.35	0.066	0.287	0.045	0.006	< 0.005	0.032	0.025	0.006	0.012	< 0.005	0.012	< 0.005	0.008	0.010	0.08	0.048
< 0.01	0.24	< 0.01	12.8	0.27	0.35	0.067	0.262	0.052	0.007	< 0.005	0.045	0.026	0.005	0.012	< 0.005	0.012	< 0.005	0.006	0.008	0.08	0.055
	0.23 ± 0.04		12.2 ± 0.6																	0.086 ± 0.007	0.050 ± 0.003
< 0.01	0.31	< 0.01	0.8	< 0.01	< 0.01	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	< 0.005	0.330	0.26	0.253
	0.35 ± 0.1		0.73 ± 0.15																0.33 ± 0.072	0.29 ± 0.93	0.25 ± 0.60
< 0.01	0.06	< 0.01	17.6	0.21	0.24	0.053	0.192	0.032	0.006	< 0.005	0.026	0.021	< 0.005	0.011	< 0.005	0.012	< 0.005	< 0.005	0.007	0.09	0.079
< 0.01	0.06	< 0.01	17.5	0.20	0.22	0.056	0.215	0.032	0.006	< 0.005	0.026	0.023	< 0.005	0.015	< 0.005	0.012	< 0.005	0.005	0.009	0.08	0.075
< 0.01	0.05	< 0.01	17.1	0.21	0.24	0.056	0.205	0.034	0.006	< 0.005	0.030	0.026	0.005	0.013	< 0.005	0.010	< 0.005	< 0.005	0.009	0.09	0.072
< 0.01	0.05	< 0.01	17.7	0.22	0.24	0.053	0.193	0.042	0.006	< 0.005	0.037	0.022	< 0.005	0.012	< 0.005	0.013	< 0.005	0.007	0.011	0.08	0.077

Ga	Ge	Zr	Nb	Sn	Te	Hf	Ta	W
ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS	ICP-MS
DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT	DIRECT
ppb	ppb	ppb	ppb	ppb	ppb	ppb	ppb	ppb
0.01	0.02	0.05	0.01	0.01	0.02	0.01	0.01	0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
0.02	< 0.02	0.06	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
0.04	0.05	0.09	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
0.06	0.04	< 0.05	< 0.01	< 0.01	< 0.02	0.01	< 0.01	< 0.02
0.02	0.04	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
0.01	< 0.02	0.07	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
0.03	< 0.02	0.14	0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
0.03	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
1.46	0.11	2.11	0.24	0.07	< 0.02	0.11	0.02	< 0.02
0.08	0.08	0.13	< 0.01	0.01	< 0.02	< 0.01	< 0.01	0.03
0.84	0.07	0.15	0.11	0.05	< 0.02	< 0.01	< 0.01	0.05
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	0.02	< 0.01	< 0.02
0.01	< 0.02	0.07	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
0.02	< 0.02	0.07	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	0.09	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	0.10	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
0.01	< 0.02	0.09	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
< 0.01	< 0.02	< 0.05	< 0.01	0.74	< 0.02	< 0.01	< 0.01	< 0.02
				0.77				
0.01	< 0.02	0.07	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
0.01	< 0.02	0.06	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
0.01	< 0.02	0.06	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02
0.02	< 0.02	0.09	< 0.01	< 0.01	< 0.02	< 0.01	< 0.01	< 0.02

Sample	Ca ICP-ES DIRECT ppm 0.02	Fe ICP-ES DIRECT ppm 0.005	K ICP-ES DIRECT ppm 0.05	Mg ICP-ES DIRECT ppm 0.005	Na ICP-ES DIRECT ppm 0.05	P ICP-ES DIRECT ppm 0.01	S ICP-ES DIRECT ppm 0.05	Sc ICP-ES DIRECT ppm 0.001	Si ICP-ES DIRECT ppm 0.02
CH 03 TB C	< 0.02	< 0.005	< 0.05	< 0.005	< 0.05	< 0.01	< 0.05	< 0.001	< 0.02
CH 03 AB C	< 0.02	< 0.005	< 0.05	< 0.005	< 0.05	< 0.01	< 0.05	< 0.001	< 0.02
CH 03 SB C	< 0.02	< 0.005	< 0.05	< 0.005	< 0.05	< 0.01	< 0.05	< 0.001	< 0.02
CH 03 W1 FA C	2.14	0.121	0.40	0.730	8.18	< 0.01	1.71	< 0.001	2.14
CH 03 W2 FA C	3.04	0.232	0.54	0.763	8.44	< 0.01	2.47	< 0.001	2.23
CH 03 W3 FA C	5.56	0.542	0.66	1.015	9.08	< 0.01	4.44	< 0.001	2.36
CH 03 W4 FA C	0.95	0.147	0.27	0.420	5.62	< 0.01	0.78	< 0.001	2.12
CH 03 W5 FA C	7.82	0.566	0.88	1.311	10.87	< 0.01	6.29	< 0.001	2.47
CH 03 W6 FA C	2.25	0.287	0.46	0.864	3.82	< 0.01	1.39	< 0.001	1.75
CH 03 W6D FA C	2.25	0.281	0.47	0.867	3.89	< 0.01	1.39	< 0.001	1.76
CH 03 W7 FA C	4.21	0.810	1.37	1.635	4.07	< 0.01	0.81	< 0.001	1.58
CH 03 W8 FA C	46.74	0.179	5.71	4.693	5.75	< 0.01	57.61	0.001	6.94
CH 03 W9 FA C	2.38	0.313	0.45	0.905	3.85	< 0.01	1.41	< 0.001	1.75
CH 03 W10 FA C	6.10	21.275	2.02	0.793	3.79	< 0.01	2.11	< 0.001	4.07
CH 03 W11 FA C	95.32	5.214	8.52	7.365	13.12	< 0.01	76.59	0.001	6.07
CH 03 W12 FA C	114.79	0.986	9.41	8.349	5.13	< 0.01	107.25	0.001	11.31
CH 03 W1 UA C	2.14	0.156	0.38	0.734	8.15	< 0.01	1.72	< 0.001	2.75
CH 03 W2 UA C	3.04	0.295	0.53	0.768	8.37	< 0.01	2.43	< 0.001	3.10
CH 03 W3 UA C	5.56	0.705	0.72	1.010	9.14	< 0.01	4.42	< 0.001	2.75
CH 03 W4 UA C	0.95	0.169	0.33	0.426	5.64	< 0.01	0.78	< 0.001	2.78
CH 03 W5 UA C	7.94	0.987	0.91	1.332	10.79	< 0.01	6.35	< 0.001	3.30
CH 03 W6 UA C	2.29	0.301	0.44	0.865	3.88	< 0.01	1.38	< 0.001	2.14
CH 03 W6D UA C	2.27	0.314	0.44	0.860	3.84	< 0.01	1.37	< 0.001	2.23
CH 03 W7 UA C	4.29	1.963	1.34	1.646	4.07	< 0.01	0.81	< 0.001	2.35
CH 03 W8 UA C	47.08	0.263	5.76	4.664	5.81	< 0.01	57.08	0.001	2.12
CH 03 W9 UA C	2.39	0.331	0.46	0.906	3.87	< 0.01	1.40	< 0.001	2.47
CH 03 W10 UA C	7.62	30.679	3.84	3.597	4.56	0.15	2.41	0.001	1.75
CH 03 W11 UA C	95.94	25.939	8.51	7.391	13.21	< 0.01	76.92	0.001	1.75
CH 03 W12 UA C	113.18	9.354	10.92	10.381	5.28	0.09	105.42	0.002	1.65
REPEATS									
CH 03 W5 FA C	7.82	0.566	0.88	1.311	10.87	< 0.01	6.29	< 0.001	2.47
CH 03 W5 FA C REP	7.88	0.568	0.93	1.331	10.85	< 0.01	6.32	< 0.001	2.45
CH 03 W4 UA C	0.95	0.169	0.33	0.426	5.64	< 0.01	0.78	< 0.001	2.12
CH 03 W4 UA C REP	0.97	0.171	0.29	0.434	5.55	< 0.01	0.79	< 0.001	2.16
CONTROLS									
BLANK	< 0.02	< 0.005	< 0.05	< 0.005	< 0.05	< 0.01	< 0.05	< 0.001	< 0.02
BLANK	< 0.02	< 0.005	< 0.05	< 0.005	< 0.05	< 0.01	< 0.05	< 0.001	< 0.02
SLRS-4	6.03	0.104	0.68	1.657	2.30	< 0.01	2.61	< 0.001	2.05
CERTIFIED (ppm)	6.2 ± 0.2		0.68 ± 0.02	1.6 ± 0.1	2.4 ± 0.2				
CERTIFIED (ppb)		103 ± 5							
TMDA-51.2	6.35	0.118	0.23	1.490	2.63	< 0.01	2.15	< 0.001	0.67
CERTIFIED (ppm)	6.2		0.3	1.4	2.0				
CERTIFIED (ppb)		111 ± 25.8							
BATT-1	39.06	< 0.005	7.98	16.104	50.64	< 0.01	28.51	< 0.001	0.68
CERTIFIED (ppm)	40.5 ± 4.03		8.2 ± 0.85	16.5 ± 1.85	52.4 ± 5.27		27.14 ± 2.48		0.67 ± 0.071

Sample	Hg TEKRAN* Waters ppt 0.5
CH 03 TB Hg	1.0
CH 03 AB Hg	< 0.5
CH 03 SB Hg	< 0.5
CH 03 W1 FA Hg	4.2
CH 03 W2 FA Hg	6.1
CH 03 W3 FA Hg	5.6
CH 03 W4 FA Hg	1.4
CH 03 W5 FA Hg	6.8
CH 03 W6 FA Hg	3.3
CH 03 W6D FA Hg	3.2
CH 03 W7 FA Hg	4.9
CH 03 W8 FA Hg	1.8
CH 03 W9 FA Hg	2.8
CH 03 W10 FA Hg	5.4
CH 03 W11 FA Hg	14.6
CH 03 W12 FA Hg	4.0
CH 03 W1 UA Hg	4.4
CH 03 W2 UA Hg	7.1
CH 03 W3 UA Hg	8.1
CH 03 W4 UA Hg	1.6
CH 03 W5 UA Hg	9.5
CH 03 W6 UA Hg	3.3
CH 03 W6D UA Hg	3.4
CH 03 W7 UA Hg	8.0
CH 03 W8 UA Hg	2.1
CH 03 W9 UA Hg	2.9
CH 03 W10 UA Hg	182.5
CH 03 W11 UA Hg	151.5
CH 03 W12 UA Hg	9.0

REPEATS

CH 03 TB Hg	1.0
CH 03 TB Hg	0.9
CH 03 AB Hg	< 0.5
CH 03 AB Hg	< 0.5
CH 03 W5 FA Hg	6.8
CH 03 W5 FA Hg	6.8
CH 03 W3 UA Hg	8.1
CH 03 W3 UA Hg	8.0

CONTROLS

1642b	1480.0
1642b	1460.0
CERTIFIED (ppb)	1.48 ± 0.06
BLANK	< 0.5
BLANK	< 0.5

** All sample >10ppt were analysed by VG-ICP-MS*

Sample	F Dionex EG-40 ppm	Cl Dionex EG-40 ppm	SO_4 Dionex EG-40 ppm	Br Dionex EG-40 ppm	NO_3 Dionex EG-40 ppm	PO_4 Dionex EG-40 ppm
D.L.	0.010	0.01	0.02	0.02	0.02	0.02
CH 03 W1 FU A	0.020	13.47	4.62	0.03	< 0.02	< 0.02
CH 03 W2 FU A	0.019	12.95	6.59	0.05	0.03	< 0.02
CH 03 W3 FU A	0.021	14.33	12.64	0.05	0.06	< 0.02
CH 03 W4 FU A	0.018	9.28	2.27	< 0.02	< 0.02	< 0.02
CH 03 W5 FU A	0.021	16.78	17.88	0.06	0.07	< 0.02
CH 03 W6 FU A	0.011	7.67	3.69	0.06	< 0.02	< 0.02
CH 03 W6D FU A	0.012	7.79	3.69	0.03	< 0.02	< 0.02
CH 03 W7 FU A	0.018	9.03	1.92	0.05	< 0.02	0.02
CH 03 W8 FU A	0.079	5.58	183.65	0.13	0.39	< 0.02
CH 03 W9 FU A	0.012	8.33	3.98	0.04	< 0.02	< 0.02
CH 03 W10 FU A	0.026	7.87	6.22	0.06	0.02	< 0.02
CH 03 W11 FU A	0.054	3.71	243.26	0.54	0.08	< 0.02
CH 03 W12 FU A	0.069	3.08	313.11	< 0.02	0.45	< 0.02
BF 03 W1 FU A	< 0.010	3.65	1.76	0.04	0.22	< 0.02
BF 03 W2 FU A	0.025	3.49	16.27	< 0.02	0.15	< 0.02
BF 03 W3 FU A	0.028	3.61	22.61	< 0.02	0.12	< 0.02
BF 03 W4 FU A	0.025	4.55	16.75	0.59	0.17	0.04
BF 03 W5 FU A	0.046	5.25	61.01	0.45	0.17	< 0.02
REPEATS						
CH 03 W5 FU A	0.021	16.78	17.88	0.06	0.07	< 0.02
CH 03 W5 FU A REP	0.023	17.63	18.76	0.06	0.09	< 0.02
BF 03 W1 FU A	< 0.010	3.65	1.76	0.04	0.22	< 0.02
BF 03 W1 FU A REP	0.012	3.81	1.86	< 0.02	0.23	< 0.02
CONTROLS						
BLANK	< 0.010	< 0.01	< 0.02	< 0.02	< 0.02	< 0.02
BLANK	< 0.010	< 0.01	< 0.02	< 0.02	< 0.02	< 0.02
BMoos-01	0.070	0.44	5.42	0.10	1.77	< 0.02
CERTIFIED		0.44 ± 0.095	5.5 ± 0.4		1.82 ± 0.14	
Plastic-94	0.066	0.54	6.52	< 0.02	0.03	0.03
CERTIFIED		0.52 ± 0.094	6.2 ± 0.42		0.02 ± 0.044	
Hamilton-90	0.498	57.55	50.29	0.13	6.01	0.06
CERTIFIED		61.5 ± 5.17	52.0 ± 5.60		6.33 ± 0.540	
Batt-01	0.157	10.22	78.22	< 0.02	0.18	< 0.02
CERTIFIED		10.8 ± 1.70	81.3 ± 7.45		0.20 ± 0.14	

Sample	Alkalinity PC-TITRATE (as $CaCO_3$) ppm
D.L.	1
CH 03 W1 FU A	< 1
CH 03 W2 FU A	2
CH 03 W3 FU A	3
CH 03 W4 FU A	2
CH 03 W5 FU A	4
CH 03 W6 FU A	< 1
CH 03 W6D FU A	< 1
CH 03 W7 FU A	< 1
CH 03 W8 FU A	< 1
CH 03 W9 FU A	2
CH 03 W10 FU A	11
CH 03 W11 FU A	70
CH 03 W12 FU A	11
BF 03 W1 FU A	4
BF 03 W2 FU A	121
BF 03 W3 FU A	118
BF 03 W4 FU A	74
BF 03 W5 FU A	146
REPEATS	
CH 03 W6D FU A	< 1
CH 03 W6D FU A REP	< 1
CH 03 W12 FU A	11
CH 03 W12 FU A REP	10
BF 03 W2 FU A	121
BF 03 W2 FU A REP	122
CONTROLS	
HAMILTON-94	102
CERTIFIED (ppm)	100. ± 5.40
BATT-01	190
CERTIFIED (ppm)	188. ± 8.31