Distribution of platinum group elements in the New Afton
alkalic Cu-Au porphyry deposit, Kamloops, British
Columbia

by

Samuel Robb

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Abstract

New Afton is an alkalic Cu-Au porphyry deposit situated 10 km west of Kamloops, British Columbia, within the Quesnel island arc terrane, a Late Paleozoic to Mesozoic arc complex that occupies the eastern part of the Intermontane Belt of the Canadian Cordillera. Bulk rock and sulfide minerals from New Afton contain potentially economic concentrations of platinum group elements (PGE). Primary PGE ($\text{Pd} \pm \text{Pt}$) enrichment occurs within the main Cu-Au ore body and is spatially associated with the Cherry Creek monzonite. Platinum group element mineralization occurs as both discrete platinum group minerals (PGM) and as trace components in nickeliferous pyrite. The early hydrothermal fluids that caused primary Cu-Au-PGE mineralization precipitated both Pd, transported as a HS^- complex, and Ni during cooling and the formation of the main ore body. The initially deposited PGE were susceptible to subsequent hydrothermal re-mobilization and reprecipitation as platinum group minerals.

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Chapter 1: Introduction

1.1 Platinum group element potential of alkalic porphyry deposits

That's where the money is! Porphyry Cu deposits (including $\pm$ Mo $\pm$ Au $\pm$ Ag subtypes) are one of the most economically significant ore deposit types, representing between 57% and 69% of the world's current and past Cu resources (Singer 1995, 2017). The tectonic setting and character of the arc magmas that produce porphyry Cu deposits may further endow them in economically significant quantities of Au and Ag. Some Canadian examples of precious metal endowed porphyry deposits are Copper Mountain, Mount Milligan, and New Afton, all examples of alkalic porphyry deposits. Active porphyry mines in British Columbia produced approximately \$2.5 billion worth of Cu and Au in 2019.

The platinum group elements (PGE; Pt, Pd, Rh, Ru, Os, Ir) are a group of six economically important metals with similar chemical properties that are typically found together in mineral deposits (Mungall and Naldrett, 2008). The PGE may be further subdivided into two groups on the basis of magmatic and mineralogical association as the Ir-group, or IPGE, consisting of Ru, Os, Ir, and the Pd-group, or PPGE, consisting of Rh, Pt, Pd, Au (Barnes et al, 1985). The research presented in this thesis makes frequent reference to PGE but is principally concerned with the PPGE, particularly Pt and Pd.

Platinum group elements are largely used for pollution abatement in the automotive industry in the manufacture of catalytic converters, making up 60% of their global usage

(NRCAN, 2019). Platinum group element exploration expenditures in Canada have modestly increased over the past decade from 2.3% in 2008 to 2.6% in 2018 (PDAC, 2019). Over the same period, Pd prices have increased ~600%, from an average price of \$354 in 2008 to \$2151 in 2020, at the time of drafting this thesis. The PGE are exceptionally rare metals within the silicate earth with current worldwide production dominated by reef-type deposits associated with layered mafic intrusions, such as the Bushveld Complex, and intrusive Ni-Cu-PGE magmatic sulfide deposits (e.g. Norilsk), but the substantial increase in the price of Pd over the past decade combined with a global drive for environmentally friendly technology necessitates the discovery of new PGE deposits that would represent a significant contribution to the Canadian economy. Once again: that's where the money is!

Unlike Cu and Cu-Mo porphyry deposits, which are typically dominated by magmas of calc-alkaline affinity with a large degree of differentiation and crustal interaction, some Cu-Au porphyry deposits form from magmas with strongly alkaline to shoshonitic affinities, particularly in island-arc settings (Richards, 2003). Historically, although these alkalic porphyry deposits have sizeable tonnages of Au and Ag, they have not been considered as major carriers of PGE and are seldom assayed for precious metal concentrations apart from Ag and Au. Interest in the potential of precious metal-rich alkalic porphyry deposits as sources of PGE has grown over the past few decades since elevated concentrations of Pt and Pd were discovered in alkaline porphyry rocks of the North American Cordillera, the Alpine-Balkan-Carpathian Dinaride belt of eastern Europe, and the Phillipines (e.g. Werle et al, 1984; Tarkian & Koopmann, 1995; Tarkian and Stribrny, 1999), although the hallmarks of a Pd-mineralized porphyry system still remain enigmatic.

As an example, the Skouries Cu-Au $\pm$ (Pd, Pt, Te) porphyry deposit in Greece hosts the bulk of its PGE mineralization as discrete platinum group minerals and within Cu-sulfides associated with potassic alteration (Economou-Eliopolous, 2005; Economou-Eliopolous and Eliopolous, 2005; McFall et al., 2018). There, PGE enrichment is considered to have occurred contemporaneously with the main stage Cu and Au mineralization (Economou-Eliopolous and Eliopolous, 2005). In contrast, PGE mineralization at Mount Milligan in the Canadian Cordillera is found adjacent to primary Cu-Au mineralization in late-stage sub-epithermal veins marking the transition from the porphyry to epithermal environment (LeFort et al., 2011). Despite these studies, relatively little work has been completed overall on the presence of PGE mineralization porphyry systems, especially when compared with that of Au.

1.2 Objectives of thesis

This research was conducted to further study the PGE potential of alkalic Cu-Au porphyry deposits. Many of the previous studies that have investigated the elevated concentrations of PGE in alkalic porphyries based their conclusions on only a handful of data collected from hand samples, core, and flotation concentrates. This shortage of information motivated the present deposit-wide study of a porphyry deposit with known enrichments in PGE to accurately determine whether these systems may be prospective PGE exploration targets. Drill assays provided by New Gold Inc. aided in the selection of drill holes for sampling by choosing those that had relatively large enrichments in Pd over the assayed 2 m intervals. Sampled intervals were then sub-sampled at smaller spacings

and analyzed for precious metals using a Pb fire assay. The samples that returned the highest concentrations of PGE + Au were then prepared for petrographic analysis by polished thin section and microanalytical techniques to determine the mineralogical hosts of PGE. The primary objective of this thesis is to establish the controls on Pd mobility and precipitation at New Afton and see if the resulting concepts can be extended to the porphyry environment in general and under what conditions PGE mineralization is expected to occur.

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Chapter 2: Distribution and controls on platinum group element mineralization at the New Afton alkalic Cu-Au porphyry deposit, Kamloops, British Columbia

Abstract

New Afton is an alkalic Cu-Au porphyry deposit situated 10 km west of Kamloops, British Columbia, within the Quesnel island arc terrane, a Late Paleozoic to Mesozoic arc complex that comprises the eastern part of the Intermontane Belt of the Canadian Cordillera. Bulk rock and sulfide minerals from New Afton were analyzed for Pd, Pt, and Au concentrations and the presence of platinum group minerals (PGM). Fire assay data show a range of concentrations of Pd between 0.5 to 33600 ppb, Pt between 0.2 to 37800 ppb, and Au between 3 and 18600 ppb. The PGM temagamite (Pd_3HgTe_3), isomertieite ($\text{Pd}_{11}\text{Sb}_2\text{As}_2$), mertieite-II ($\text{Pd}_8[\text{Sb,As}]_3$), and cooperite ($[\text{Pt,Pd}]_2\text{S}$) were identified in most samples with elevated Pt and Pd (> 400 ppb). Chalcopyrite and pyrite were analyzed for platinum group elements (PGE) using LA-ICP-MS. Ni $\pm$ Co-rich pyrite is the main host of PGE at New Afton; concentrations of Pd and Pt in pyrite range from below detection limit to 30.8 ppm and 6.9 ppm, respectively. Chalcopyrite has modest enrichments in Pd, ranging from below detection limit to 13.4 ppm and Pt from below detection limit to 0.015 ppm. Precious metal ratios, particularly Pd/Au in the bulk rock samples, show that PGE mineralization may be grouped in three domains: (1) high Pd, low Pd/Au, (2) high Pd, high Pd/Au, and (3) low Pd, variable Pd/Au. These domains are interpreted as (1) representing primary PGE mineralization, related to primary hypogene Cu-Au mineralization, (2) secondary enrichment of PGE via hydrothermal redistribution, and (3) the depleted source of the

secondarily enriched domains. In this scenario, domains (2) and (3) represent remobilization of PGE, leading to local enrichment and depletion, with no net addition of PGE to the system. Consequently, initial PGE (Pd $\pm$ Pt) deposition at New Afton occurred during the primary hypogene mineralization event where Pd was transported from an underlying magma reservoir into the porphyry system by a magmatic-hydrothermal fluid as a HS⁻ complex and incorporated in nickeliferous pyrite precipitated with the main Cu-Au mineralization. The initially deposited PGE were susceptible to subsequent hydrothermal remobilization due to recrystallization of early hypogene sulfide minerals and were reprecipitated as platinum group minerals.

2.1 Introduction

The New Afton mine is an alkalic Cu-Au porphyry deposit located approximately 10 km west of Kamloops, British Columbia. It is one of several alkalic porphyry deposits directly associated with the Late Triassic Iron Mask batholith in the Quesnel terrane of the Intermontane Belt (Logan & Mihalynuk, 2005). The mine has been in production since 2012 and is expected to continue producing until 2030. Current proven and probable reserves at New Afton are 47.3 Mt grading 0.66 g/t Au, 1.9 g/t Ag, and 0.77% Cu with an additional measured and indicated resource of 57 Mt grading 0.61 g/t Au, 2.1 g/t Ag, and 0.74% Cu (New Gold, 2019). New Afton is currently mined as an underground block caving operation and is situated below the historic Afton open pit mine, which was owned by Teck Corporation, operated from 1978 until 1991. Prior to its acquisition by New Gold Inc the deposit was known to host potentially economic to sub-economic grades of platinum group elements (PGE), particularly Pd (Thompson et al., 2001; Nixon, 2003).

Porphyry deposits containing Cu and Au may be generated by mantle sourced arc magmas with alkaline to strongly alkaline affinities (Lang et al., 1995; Richards, 2011). The fertility of these arc magmas requires the removal of chalcophile elements from the mantle sulfide phases to the melt. Gold concentrations in the mantle are estimated at 1 ppb (McDonough & Sun, 1995; Saunders et al., 2018) and platinum group element (PGE) concentrations are estimated at 6 ppb Pt and 5 ppb Pd (Barnes et al., 2015), with sulfide phases as the main hosts of these metals. Melting the mantle at oxidizing conditions destabilizes and removes sulfide phases, and the coexisting melt may then be enriched in Pt, Pd, and Au by an order of magnitude relative to their primitive mantle values (Mungall, 2002; Mungall et al., 2006). Palladium and Pt are mobile as both bisulfide and chloride complexes in hydrothermal fluids at temperatures, oxygen fugacities (fO_2), and acidities (pH) relevant to conditions of porphyry formation (e.g., Hanley, 2005). If the PGE can be effectively transported from their mantle source to upper crustal levels, in a similar manner to Au, then these fertile, mantle-derived magmas with the potential to form Cu and Au porphyry deposits may also have the potential of forming deposits of Pt and Pd.

Previous studies have documented elevated PGE concentrations and platinum group minerals (PGM) in Cu flotation concentrates and bulk samples of alkaline and calc-alkaline porphyry deposits (Eliopoulos & Economou-Eliopoulos, 1991; Tarkian & Koopmann, 1995; Tarkian & Stribny, 1999; Thompson et al., 2001; Auge et al., 2005; Economou-Eliopoulos, 2005). Potentially economic to sub-economic grades of PGE have been observed in alkalic deposits of the Canadian Cordillera, such as Mt. Milligan (LeFort

et al., 2011), Galore Creek (Thompson et al., 2001), Whiterocks Mountain (Nixon & Carbno, 2001), and New Afton/Afton (Nixon, 2003; Garagan, 2014; this study), but their source, transportation, and depositional controls remain enigmatic. Here, we show that hydrothermal PGE mineralization at New Afton occurred during the main stage of primary Cu-Au precipitation but was susceptible to secondary hydrothermal remobilization.

2.2 Geological Setting

2.2.1 Regional geology

The Intermontane Belt is one of the five northwest-trending morphogeological belts that together form the Canadian Cordillera (Fig. 2.1). It comprises a series of amalgamated Paleozoic to Mesozoic terranes, including from east to west, the Quesnel island arc terrane (Quesnellia), the oceanic Cache Creek terrane, and the Stikine island arc terrane (Stikinia). Together, Quesnellia and Stikinia form a 2000 km-long metallogenic belt with prevalent calc-alkaline and alkaline-style porphyry mineralization. These belts contain many of the Cu occurrences in British Columbia and account for 75% of its metallic mineral production (Logan & Mihalynuk, 2014). Some of the principal porphyry deposits include Red Chris, Galore Creek, Mount Polley, Mount Milligan, Copper Mountain, and Afton/Ajax.

The Quesnel terrane, which hosts the New Afton deposit, is an ancient island arc formed off the western coast of ancestral North America in the upper Paleozoic until its eventual Mesozoic accretion on to the cratonic margin *ca.* 186 Ma (Fig. 2.1; Logan &

Mihalynuk, 2014). Its early history, prior to accretion, involved submarine to subaerial clastic and carbonate sedimentation with subsequent arc growth while combined with the Stikine arc at its northern terminus (Mihalynuk et al., 1994). The stratigraphy of southern Quesnellia in the Kamloops area is subdivided into three major sequences: the late Paleozoic Harper Ranch Group, the Late Triassic Nicola Group, and the Jurassic Rossland Group (Smith, 1979; Beatty et al., 2006). Construction of the arc complex began in the Late Devonian with the deposition of the Harper Ranch Group, a thick succession of steeply dipping, northwest-striking interbedded mudstones and siltstones, and volcanoclastic sandstone and conglomerate deposited during the opening of the marginal Slide Mountain Basin along western North America (Roback et al., 1994; Beatty et al., 2006). Late Mississippian carbonates interbedded with volcanoclastic and sedimentary layers overlie the early arc succession, marking a rapid shift from rifting and sedimentation to a period of volcanic quiescence and carbonate formation lasting until the Permian (Beatty, 2002; Beatty et al., 2006).

Pyroxene-phyric volcanic and volcanoclastic rocks of shoshonitic affinity of the Late Triassic Nicola Group rest unconformably on top of the carbonate and sedimentary rocks of the Harper Ranch Group (Mortimer, 1987). Sporadic submarine volcanism in the Nicola arc began *ca.* ~238 Ma with the eruption of pyritic felsic pyroclastics, lavas, and pillow basalts (Mihalynuk et al., 2016). Arc growth accelerated after the initial episodic magmatism, erupting shoshonitic augite-porphyritic picrite, basalt, and andesite and calc-alkaline augite $\pm$ plagioclase basalts and andesites (Mortimer, 1987; Mihalynuk et al., 2016). A 210 Ma collision between the Sitlika-Kutcho-Venables arc with the paired

Quesnel-Stikine arc caused an arc-parallel slab tear, eventually culminating in the generation and emplacement of voluminous magmas *ca.* 203 Ma and a parallel belt of alkalic and calc-alkalic porphyry deposits prior to the 201 Ma cessation of Nicola arc growth (Logan and Mihalynuk, 2014; Mihalynuk et al., 2016). Eventual counterclockwise rotation of the combined Quesnel-Stikine arc due to subduction of the Cache Creek ocean along their eastern margin resulted in the collision of Stikine and Quesnel and the enclosure of the Cache Creek terrane (Mihalynuk et al., 1994). Post-accretionary magmatism in the Kamloops area was dominated by calc-alkaline volcanic rocks of the Eocene Kamloops Group (Ewing, 1981). These high-K basalts, andesites, and rhyolites rest unconformably on top of Quesnel stratigraphic units north of New Afton but extend as far south as Idaho because of the subduction of the Kura-Farallon spreading center (Breitsprecher et al., 2003).

2.2.2 Deposit geology

New Afton is one of several Cu-Au porphyry deposits associated with the Iron Mask batholith (Fig. 2.1). The batholith is a polyphase, silica-saturated alkaline intrusive complex that intruded the sedimentary and volcanic rocks of the coeval Nicola Group during the Late Triassic (Snyder & Russell, 1993). It is considered to consist of four distinct phases, from oldest to youngest: Pothook diorite, Hybrid Unit, Cherry Creek monzonite, and the Sugarloaf diorite (Stanley et al., 1994). At the New Afton property all Iron Mask phases apart from the Hybrid Unit are present; the Cherry Creek monzonite is the main intrusive phase, emplaced in the Nicola Group volcanic rocks *ca.* 204.5 ± 0.6 Ma (Fig. 2.1;

Mortensen et al., 1995). Individual strata of the Nicola Group in the New Afton area have not been correlated due to the large amount of alteration and structural modification. The group consists of picritic and andesitic flows, monomictic and polymictic tuffs and breccias. Along the eastern margin of the deposit, the Cherry Creek monzonite is in fault contact along a massive picrite flow (Fig. 2.2). The flow is thought to have acted as a chemical buffer and aquitard during volatile exsolution by interacting with CO₂-enriched fluids leading to the precipitation of Cu-Fe sulfides and focusing mineralization within the volcanic units east of the picrite (Nixon, 2003). The emplacement of the monzonite was controlled by a set of northeast trending faults named the Hanging wall fault (HWF), Footwall fault (FWF), and J-fault. The HWF strikes northeast, dipping along the contact of the picrite with the Nicola Group volcanic rocks.

Alteration paragenesis follows a model of early calcic and potassic alteration outward to propylitic assemblages (Lipske and Wade, 2014). Calcic alteration is characterized by large magnetite + apatite + actinolite veins with local pink potassium feldspar staining along their margins. The veins are generally a few centimeters to several meters wide but may reach up to 100 m in some localities (Lipske et al., 2020). Potassic alteration is defined by the assemblage K-feldspar-biotite $\pm$ magnetite $\pm$ quartz, and further subdivided as biotite-dominant and K-feldspar-dominant, based on the altered lithology (Lipske and Wade, 2014). Potassically altered volcanic rocks of the Nicola Group are characterized by biotite > K-feldspar whereas K-feldspar-dominant potassic alteration is generally restricted to the Cherry Creek monzonite. In general, the potassic alteration halo is found adjacent to the northern margin of the Cherry Creek monzonite (Fig. 2.2). Later

phyllic alteration consists of the assemblage muscovite-carbonate-quartz-pyrite, encompassing a large portion of the deposit and is predominately controlled by high-angle structures (Fig. 2.2; Lipske et al., 2020). Late argillic alteration consists of kaolinite and white mica and is generally restricted to high angle faults crosscutting previously altered rocks.

Primary Cu-Au hypogene mineralization at New Afton is associated with biotite-dominant potassic alteration in Nicola Group volcanic rocks (Fig. 2.2). Copper mineralization is present as chalcopyrite and chalcopyrite + bornite in some areas of the deposit. Gold is present as small inclusions of native Au and electrum in chalcopyrite and pyrite. A secondary hypogene mineralization event that was accompanied by argillic alteration occurred in the Late Jurassic *ca.* 155 Ma (unpublished whole-rock K-Ar date; Lipske et al., 2018), overprinting primary Cu-Au mineralization. Secondary hypogene mineralization is defined by the presence of tennantite-tetrahedrite. A supergene enrichment zone containing native Cu occurs at the shallowest level of the deposit.

2.3 Methods

2.3.1 Sampling and fire assays

The samples used in this study are half core and quarter core from drill holes collected by New Gold Inc between 2001 and 2017. Samples from this archive were selected with guidance from the exploration department and through analysis of the New

Afton drill assay database. Core samples with Pd concentrations > 0.4 ppm over the 2-meter assay interval were selected. These samples were taken from drill holes where Pd was either correlated with $\text{Au} \pm \text{Cu}$ or showed no correlation to any precious metal. The selected 2-meter core intervals were then subsampled for separate lithologies at lengths between 5 to 30 cm. Overall, a total of 220 samples were collected from 19 drill holes that covered the vertical expanse of the deposit (Fig. 2.3).

A total of 218 samples were analyzed for their precious metal concentrations using Pb fire assay at Activation Laboratories in Ancaster, Ontario. The method allows for the accurate measurement of Au, Pt, and Pd with a detection limit of 1 ppb, 0.1 ppb, and 0.1 ppb, respectively. The fire assay results were used to select a subset of 50 samples for petrographic analysis using a conventional transmitted and reflected light microscope.

2.3.2 EPMA and LA-ICPMS microanalysis

Petrographic samples were subsequently scanned for platinum group minerals (PGM) using a JEOL 6610LV scanning electron microscope (SEM) at the University of Ottawa. Compositional analyses of PGM were acquired on a JEOL JXA-8230 SuperProbe electron probe microanalyzer (EPMA) with 5 tunable wavelength dispersive spectrometers at the University of Ottawa. The PGM were analyzed for S, Pt, Pd, Fe, Cu, Hg, Sb, Te, Ag, and As using a 20 keV beam with a 40 nA current and 20 s count time for each element.

Sulfide mineral (chalcopyrite and pyrite) trace element concentrations were acquired using a laser ablation inductively coupled plasma mass spectrometer (LA-ICP-MS) at Dalhousie University. The sulfides were ablated using an ESI NWR213 Nd:YAG laser operating at 213 nm with He used as the carrier gas and analyzed with a ThermoScientific iCAP quadrupole ICP-MS. Most chalcophile elements were quantified using the MSS5 standard (Dare et al., 2010). Lithophile elements, as well as chalcophiles not present in MSS5, were standardized using NIST610 glass. Iron was previously quantified in the selected sulfides via EPMA and used as an internal standard for both chalcopyrite and pyrite in the data reduction. Both minerals showed nearly stoichiometric Fe concentrations and so values of 46.5 wt% for pyrite and 30.4 wt% for chalcopyrite were used as an internal standard during data reduction. Elemental Pd concentrations in the sulfides are reported based on measured Pd^{106} due to the interference of Pd^{105} with Cu-Ar production in the plasma.

2.3.3 Statistical treatment of data

Statistical sorting and tests were conducted on the whole rock fire assay data and the sulfide LA-ICPMS analyses. The statistical analyses were completed in MATLAB®, using both native statistics functions included with the software and custom programs.

2.3.3.1 K-means clustering

Palladium, Pt, Au, and Pd-Au ratios determined via fire assay were sorted into statistically similar groups using k -means clustering. The method creates a group of k clusters from a set of n observations (where $k \leq n$) by minimizing the total variance within each cluster. To determine the optimum number of clusters to use for the dataset, we iteratively maximized the mean silhouette number, or, informally, the average similarity of data within a cluster compared to other clusters, from 2 to 10 clusters. The optimum number of clusters given the fire assay data was found to be 3 (Fig. 2.4), a choice which was further supported by the textural and mineralogical similarities between samples within each of the three groups.

2.3.3.2 Principal component analyses

A principal component analysis (PCA) of the LA-ICPMS data for both chalcopyrite and pyrite was performed to further elucidate the statistical relationships between trace elements within the sulfide minerals. The analysis takes n observations of m variables as input, or each trace element analyzed in this case, and transforms them into a set of linearly independent variables, or principal components. The resulting principal components are weighted linear combinations of the input variables and placed in descending order based on how much of the total variance of the dataset is contained within the component, such that principal component 1 (PC1) contains the largest proportion of the total variance, PC2 the second largest, and so on. Therefore, we can use only the first few principal components to describe most of the variance in the dataset. The loading of an input variable on a principal component is the weight of that variable within the component. The score of an

observation is the value of the principal component given that observations data as input. The trace element data in this study were all recast to elemental molar values and $\log_{10}$ -transformed prior to PCA such that the data are normally distributed.

2.3.4 Geological modelling

The spatial distribution of PGE at New Afton was investigated on a deposit scale using the 3D geological modelling software Leapfrog Geo™ and the drill database provided by New Gold, Inc. Probability grade shells were created using radial basis function (RBF) interpolation with a predefined cut-off grade to create the indicator variable. High assay values were clipped using a top-cut value and the data were log-transformed prior to interpolation. A global trend oriented with a dip of 90°, dip azimuth of 5°, and a pitch of 70° and ellipsoid axis ratios of 3:2:1 was applied to the data to match the general shape of the ore body. The data were then interpolated using a spheroidal interpolant function that applies spatial weights to the data and estimates the grade shell boundaries.

Block kriging was performed to constrain the mineralizing domain of Pd, determine its spatial distribution, and to accurately estimate the amount of metal added to the system during metasomatism. Spatial weightings used in the interpolation were estimated by computing anisotropic 3D variograms for each interpolated element. Spherical variogram models were used, and their parameters were determined by iteratively rotating the anisotropy ellipsoid to maximize the variogram sill along each axis. Variography was

performed on normal-score transformed values and the variogram models were back-transformed prior to interpolation. Pt assays are commonly at or below detection limit and higher values suffer from an extreme nugget effect. After back-transforming the Pt variogram model the nugget was only slightly lower than the total sill and so Pt tonnages could not be accurately estimated (Chiles and Delfiner, 2012).

Two separate block models were produced: one which encompassed the entire property and another that was constrained only to areas affected by hypogene (Cu-Au) mineralization as identified by core logging. The hypogene zone model was created to determine the spatial association and distribution of Pd in the deposit using blocks with a length of 5 m along each axis. Investigation of contact plots of mineralized domains, alteration domains, and lithological domains shows that Pd mineralization was primarily associated with whole rock hypogene mineralization. The property scale model was created to determine an order-of-magnitude estimation of metal and sulfur enrichment at New Afton using blocks with a length of 10 m in each direction. Both models were validated using E-W, N-S, and depth swath plots against concurrently produced nearest neighbor and inverse-distance interpolations along with comparison of grade-tonnage curves between each method of interpolation. Grades within the property scale interpolation that are in the hypogene zone are compared against kriged grade interpolations constrained by the hypogene zone and were in reasonable agreement with one another; property scale grades in general are lower due to the diluting effects of unmineralized domains outside the ore zone.

2.3.5 Thermodynamic modelling

The evolution of the early hypogene mineralizing fluid was modelled in Geochemist's Workbench (Bethke, 1996) using a modified LLNL thermodynamic database (Worley, 1992). The aqueous species, minerals, and reactions in the LLNL database were calculated in the temperature range of 250 °C to 450 °C using the SUPCRT-based program DBCreate (Kong et al., 2013). Equilibrium constants for aqueous Pd(II)-chloride complexes were taken from the experimental data of Wood et al. (1992). The equilibrium constant for the Pd(II)-bisulfide complex $\text{Pd}(\text{HS})_2$ was taken from the experimental data of Pan and Wood (1994). The data for solid Pd phases, hydroxide-, and oxide-complexes were calculated in SUPCRT92 (Johnson et al., 1992) using the slop07 database with free energy data from Sassani and Shock (1998). Data for aqueous Ni species were compiled from UV-Visible spectroscopic experiments on Ni(II)-chloride complexation (Liu et al., 2012) and the SUPCRT database. Equilibrium constants for the solid Ni-S phases pentlandite ($\text{Ni}_{4.5}\text{Fe}_{4.5}\text{S}_8$), millerite (NiS), heazlewoodite (Ni_3S_2), and vaesite (NiS_2) and the solid Co-S phases cobaltpentlandite (Co_9S_8), jaipurite (CoS), and cattierite (CoS_2) were taken from the thermodynamic extrapolations of Klein and Bach (2009). Data for Au HS-complexes and native Au were obtained from Stefansson and Seward (2004) and Cl-complexes from Stefansson and Seward (2003). Some of the experimentally determined equilibrium constants used were extrapolated to higher temperatures using a non-linear curve fit. The modified and extrapolated equilibrium constants and reactions used in the thermodynamic modelling are listed in Table 2.3.

Details of the parameters and choice of reactions used in the models are discussed in Appendix A.

2.4 Results

2.4.1 Fire assays

Fire assayed samples exhibited a wide range of Pt, Pd, and Au concentrations (Fig. 2.5). Samples range between 0.2-37800 ppb Pt, 0.5-33600 ppb Pd, and 3-18600 ppb Au. The data for Pt and Pd are both highly skewed to the left with a skewness of 14.3 and 7.1, respectively. Pd concentrations average 942.5 ppb, significantly higher than that of Pt at 227.5 ppb. Au is significantly more enriched on average than Pt and Pd, with an average of 1259.1 ppb. Calculated Pd/Au ratios of the assayed samples show three distinct populations within the data (Fig. 2.6). *K*-means clustering of Pt, Pd, Au, and Pd/Au in the assayed samples placed the samples into three groups: (1) high Pd, low Pd/Au (Au-rich samples), (2) high Pd, high Pd/Au (Au-poor samples), and (3) low Pd, variable Pd/Au (Fig. 2.7). The fire assay data are summarized in Table B.1.

2.4.2 Grouped sampled petrography

The *k*-means algorithm had no *a priori* information on the petrography of the samples prior to clustering. The sample groups (Group 1, Group 2, and Group 3)

determined through *k*-means clustering are all distinct from one another, while the samples within each individual group are petrologically similar.

Samples in the first group are characterized by the presence of weakly to strongly disseminated, pervasive, or vein-controlled chalcopyrite $\pm$ pyrite (Fig. 2.8). Most of the samples in Group 1 have primary potassic alteration that may be variably overprinted by muscovite and/or clay minerals. Potassic alteration is dominated by K-feldspar and biotite with albite, quartz, and chlorite. Chlorite occurs either as a primary alteration mineral replacing mafic phenocrysts from the host rock or as the retrograde alteration product, with Fe-Mg carbonates, of primary biotite; fine to coarse grained biotite is generally present in potassically altered volcanic rocks that host hypogene sulfide mineralization, but not consistently correlated with PGE enrichment and in many cases is altered to chlorite and carbonates. Strong potassic alteration, restricted to vein contacts where K-feldspar is the dominant alteration phase, is distinguished from potassically altered monzonite by the lack of primary igneous textures (Fig. 2.8A, B). Phyllic overprinting of early potassic alteration in Group 1 occurs as the breakdown of early feldspar to white mica and quartz (Fig. 2.8B). Pyrite precipitated along with potassic alteration is characterized by an association with chalcopyrite and the mineral assemblage k-feldspar-biotite-chlorite-quartz (Fig. 2.8A-D). The crystals are generally small ($< 75 \mu\text{m}$) with a subhedral to euhedral cubic habit. Pyrite grains deposited during the phyllic stage of alteration are large ($> 1 \text{ mm}$) and euhedral with cubic and dodecahedral habits surrounded by quartz and white mica.

Magnetite-apatite veins with potassic alteration haloes are found in some samples, with magnetite having undergone partial martitization to hematite (Fig. 2.8D). All group 1 samples were hosted by both monomictic and polymictic tuffs and andesite flows of the Nicola Group. Andesitic lavas are identified by the presence of relict amphibole, pyroxene, and feldspar phenocrysts within the altered samples. The samples within Group 1 may be found proximal to large scale brittle faults and fractures, but the cataclastite remains unmineralized with respect to PGE. Where Pd enrichment is spatially associated with later argillic alteration, the mineralization consistently occurs within potassically altered volcanic rocks in fault contact with the argillic alteration or within earlier hypogene sulfide assemblages; Group 1 PGE mineralization is hosted within sulfide-rich volcanic rocks where early potassic alteration has not been completely overprinted by later phyllic and argillic alteration (Fig. 2.8E).

Group 2 samples are characterized by the presence of partially martitized magnetite, apatite, and carbonates (Fig. 2.9A-C). Primary alteration facies are obscured by late structurally controlled carbonate alteration. Group 3 samples share similar mineralogical and textural characteristics as those in Group 2 except that magnetite has typically undergone complete martitization to hematite. The Group 2 and 3 samples are in direct contact with or within carbonate-cemented breccia, rubble, and fault gouge in the upper portions of the deposit (Fig. 2.9D). Fe-rich carbonate minerals fill grain fractures and surround partially or fully martitized magnetite crystals. Chalcopyrite is generally absent in most samples but, where it is present, occurs only as fine, sporadic disseminations. All samples in Group 2 were located in the upper levels in the deposit,

slightly below the supergene zone, and in close proximity to monzonite dykes. Although some Group 1 samples are also associated with partially oxidized magnetite-apatite veins, they are distinguished from those in Group 2 by their association with early potassic alteration and abundant primary sulfide mineralization that is notably absent in all Group 2 samples.

Samples of Cherry Creek monzonite tend to be extremely depleted in all precious metals relative to the mineralized portions of the deposit. However, in all analyzed monzonite samples, Pt, Pd, and Au were well above average continental crust concentrations. The monzonite is typically more enriched in both Pt and Pd relative to Au, unlike the primary hypogene mineralization associated with it. On average, the samples had an average of 10.19 ppb Pt, 11.23 ppb Pd, and 7.75 ppb Au. In general, the monzonite is silica-undersaturated and predominantly consists of hematite-dusted alkali feldspar and plagioclase crystals and minor biotite. Mirolitic cavities approximately 2 cm in diameter are present within the center of the monzonite at the upper levels of the deposit and decrease in size and abundance moving outwards towards the edges of the stock. The quasi-spherical cavities crystallized a mineral assemblage of actinolite, chlorite, and minor epidote consistent with an outer propylitic alteration assemblage. A series of steeply dipping parallel veins with the same alteration mineralogy also appear within the center of the intrusion. The monzonite samples are all characterized by pervasive K-feldspar-dominant alteration with minor biotite. White micas are found along the contacts of the outer propylitic veins and mirolitic cavities. In all samples primary igneous textures and fabrics

are visible and well preserved (Fig. 2.10). Distinct textural changes occur laterally across the monzonite in the upper parts of the deposit. (Fig. 2.10).

2.4.3 Platinum group minerals

Twenty-five petrographic thin sections were selected for SEM imaging and EPMA analysis based on their PGE concentrations and alteration lithofacies to identify the host minerals of PGE mineralization. Platinum group minerals (PGM) were found in 11 of the 25 analyzed thin sections and are summarized in Table 2.1. The PGM identified were temagamite (Pd_3HgTe_3), isomertieite ($\text{Pd}_{11}\text{Sb}_2\text{As}_2$), mertieite-II ($\text{Pd}_8[\text{Sb,As}]_3$), and cooperite ($[\text{Pt,Pd}]_2\text{S}$). Samples that assayed $\text{Pd} \pm \text{Pt}$ each greater than 1 ppm with abundant sulfide mineralization (Group 1 samples) typically host visible PGM. Group 2 samples, characterized by elevated Pd and low Au along with the presence of martite and apatite, rarely hosted visible PGM. The grain size of the individual PGM identified varied between less than 1 μm to 90 μm and showed variable morphologies.

Group 1 samples with abundant disseminated sulfides or sulfide flooding host their identified PGM either within chalcopyrite or along the grain boundary contacts of chalcopyrite and pyrite (Fig. 2.11A). The Group 1 samples with visible (under petrographic microscope and/or SEM) PGM are listed in Table 2.1. Individual PGM grains vary in size throughout the sample group but are on average larger than PGM found in Group 2. The largest grains occur along chalcopyrite-pyrite interfaces whereas smaller grains typically occur as stand-alone inclusions within chalcopyrite. Ag-telluride minerals (e.g. hessite) and

electrum commonly occur associated with the PGM (Fig. 2.11B-C). Pyrite is associated with samples that host PGM and is present as wispy stringers or euhedral grains within chalcopyrite (Fig. 2.11D).

Sample AF8103-052A hosts the largest grains (e.g. Fig. 2.11A) and also returned the highest assay values for Pt and Pd, at 37.8 ppm and 33.6 ppm, respectively. Individual PGM grains up to 90 μm occur along the contacts of chalcopyrite and pyrite in the presence of magnetite. Sample 052A and 052B are both polymictic volcanoclastic rocks characterized by pervasive alteration consisting of K-feldspar, biotite, and chlorite, in the clasts and chalcopyrite-pyrite overprinting of earlier magnetite-apatite veins. In both samples, PGM were restricted to the areas of sulfide flooding, with the clasts partially altered to white mica unmineralized with respect to PGE.

Platinum group minerals were identified in Group 2 samples (Table 2.1). Samples UA7606-056 and AF0135-122 are both from carbonate-healed faults (e.g. Fig. 2.11D). The margins of the faults are massive, brecciated magnetite-apatite lodes. Fe-carbonate alteration is present along magnetite grain boundaries and fractures. The PGM in sample AF0135-122 are located within pervasive magnetite clusters between large apatite crystals, as well as within the carbonate-filled margins along magnetite grains (Fig. 2.11E-G). Platinum group minerals in UA7906-056 occur both in magnetite, in a similar style to the PGE mineralization in sample AF0135-122, and as discrete grains within brecciated apatite (Fig. 2.11H). The Pd-rich PGM analyzed on the SEM (EDS) in samples AF0135-122 and UA7906-056 had no detectable concentrations of Pt. Rather, Pt mineralization was

concentrated in a few sparse grains of cooperite (PtS). Sample EA13041-111 lacks the pervasive magnetite-apatite alteration of samples UA7606-056 and AF0135-122 and was instead dominated by K-feldspar crystals surrounded by chlorite. Platinum group element mineralization in sample EA13041-111 is hosted by disseminated grains of mertieite-II and temagamite that form simple intergrowths with finely disseminated chalcopyrite.

The compositions of PGM from four samples were quantified using EPMA, with analyses (Table 2.2) in fair agreement with the semiquantitative EDS analyses of PGM grains on the SEM, so SEM data are presented where EPMA data are absent. In general, Pd-bearing PGM are deficient in Pt. Temagamite has no detectable concentrations of Pt in any analyzed grains. Isomertieite and mertieite-II has minor enrichments in Pt, up to 0.4 wt%, compared with 70 wt% Pd, on average. The main Pt-bearing phase is cooperite with Pt/Pd = 11.15 but it was only identified in one sample. All grains, except for cooperite, have detectable amounts of Ag, with the largest concentrations in isomertieite and mertieite-II. Two electrum grains measured, with ~82 wt% Au in each, have elevated Pd concentrations of 0.25 and 0.27 wt%, respectively.

2.4.4 Sulfide trace element composition

Chalcopyrite and pyrite identified in thin section were analyzed by LA-ICPMS to determine their trace element concentrations and to determine whether they are major carriers of PGE mineralization at New Afton. Twelve thin sections of Group 1 and 2 samples with the highest concentrations of PGE were selected for analyses to characterize the trace

element composition of chalcopyrite ($n = 145$) and pyrite ($n = 88$) grains. The general absence of sulfide phases in Group 2 samples skewed the choice of analyzed samples towards those in Group 1; where possible, Group 2 samples with some chalcopyrite and pyrite were analyzed. The compiled LA-ICPMS data for both chalcopyrite and pyrite are listed in Appendix C. The EPMA analyses of the corresponding grains are listed in Appendix D. The trace element data were analyzed for their statistical correlations using the Pearson correlation coefficients and the association between each element through principal component analysis. Platinum was excluded from both the correlation matrices and principal component analysis due to it being absent or below detection limit in most grains.

2.4.4.1 Chalcopyrite

Most chalcopyrite crystals analyzed are >1 mm and generally free of silicate and sulfide inclusions as indicated by homogenous time-spectra over each analyzed grain. Palladium concentrations in chalcopyrite follow a positively skewed log-normal distribution with a mean of 0.22 ppm and range between 0.004 to 13.4 ppm, although most analyses are tightly clustered between 0.01 and 0.1 ppm (Fig. 2.12). The variance between most the chalcopyrite trace element data is small (Fig. 2.13). Arsenic, Sb, and Ag form one group of correlated elements, and Pd, Te, and Mn form another (Fig. 2.13). Bismuth and Sb are also correlated with one another, but Bi is only weakly correlated with As and Ag (Fig. 2.13) Note that the correlations between elements are those with Pearson correlation coefficients >0.5 . Overall, most of the elements in chalcopyrite are present in relatively

low abundances and do not define unique populations within the sample group. The first two principal components of the chalcopyrite dataset (Fig. 2.14) describe some variance present in the data, where PC1 (predominately Mn, As, Pd, and Bi) contains 48% of the variance and PC2 (Mn, Ni, Bi, Sb, and As) contains 12% of the variance, but the scores of each of the analyses remain tightly clustered around the origin.

2.4.4.2 Pyrite

Pyrite has a much larger variation in Pt and Pd content than chalcopyrite and contains higher Pd on average (Fig. 2.12). A correlation matrix of all the analyzed elements and their Pearson correlation coefficients is shown in Figure 2.15. Out of the precious metals (Pt, Pd, Au, Ag), Pd and Ag are the most abundant and similarly distributed with concentrations between 0.01 and 30 ppm and 0.47 and 30.5 ppm, respectively (Fig. 2.15). Although the amount of the two metals are similar in the whole dataset, they are only weakly correlated within individual analyzed crystals (Fig. 2.15). Silver is most strongly correlated with enrichments in Au, Mn, Cu, Sb, Pb, and Bi, which encompass the main ore metals at New Afton. Palladium, on the other hand, is strongly correlated with Ni and Se, lacking any statistically relevant correlation with the Ag-group of metals and metalloids (Fig. 2.15). Au in pyrite is low, with a range of concentrations from below detection limit to ~2 ppm. Most of the analyzed grains are also low in As, with rare weight-percent levels (up to 2.93 wt%), but in general contain relatively low concentrations (median 58.7 ppm As) that correlate with Au.

For the LA-ICPMS analyses the beam was rastered across the crystals to limit metal fractionation, allowing the unresolved time-spectra to be used qualitatively to detect metal zonation in pyrite as the beam swept back and forth across compositional zones. Pd-rich zones in pyrite are associated with elevated $\text{Ni} \pm \text{Co} \pm \text{Se}$ (Fig. 2.16). Cobalt and Se zoning are the most prominent in the analyzed pyrite grains, and are typically, but not always, found with similarly elevated Pd concentrations (Fig. 2.16), although Co-enrichment in general is associated with elevated Pd. Pd-rich zones in pyrite are systematically tied to Ni enrichment. Figure 2.16 outlines the association between Pd-zoning with that of Ni, Co, and Se. In some samples, Pd-Ni-Co-Se zoning occurs concurrently (Fig. 2.16A) while in others Pd zones occur without regard to changes in Co concentration (Fig. 2.16B). Sample AF0183-052A, with the highest concentrations of Pd from the whole rock fire assays, contains zoned pyrite where Pd-Ni-Se zones occur adjacent to Co-rich zones (Fig. 2.16C). Cobalt zonation may also occur with no similar enrichments in Pd and Ni (Fig. 2.16D). Because the zonation was not mapped, its spatial patterns remain unknown.

The principal component analysis of the pyrite grains shows that ~80% of the variance in the data is contained in the first 3 principal components, where PC1 describes 58.54%, PC2 describes 14.83%, and PC3 describes 7.2% of the variance. The loadings of each of the trace elements on the first two principal components, listed in Table 2.3., are plotted against one another in Figure 2.14. The main ore elements (Cu, Au, Ag, and Pd) all have large loadings on PC1, which also includes the non-ore metals (Ni, Pb, and Mn) and the metalloids (As, Bi, and Sb). Se and Te both have negligible loadings on PC1. The

second component, PC2, is dominated by the effects of Ni and Pd with some contribution from Se and Pb.

2.4.5 Geological modelling

Probabilistic grade shells were created using indicator cut-off grades of 0.02, 0.07, and 0.1 ppm Pd as well as 0.01 and 0.005 ppm Pt with estimated probabilities greater than 50%. The values of 0.02 ppm Pd and 0.005 ppm Pt were selected to represent the extents of porphyry mineralization of the two metals, respectively. Grade shells below these values expand out to the limits of the spatial extents of the data set covering the entire deposit. Furthermore, the average concentrations of Pd and Pt in weakly altered Nicola Group volcanic rocks found in the assay database are 0.011 ppm and 0.003 ppm, respectively. Although it is likely that the samples used in calculating the average underwent some minor degree of porphyry enrichment, these values combined with the observation that grade shells extend out to the limits of the extent of the drilled property make it a reasonable assumption that the grades are representative of background PGE concentrations. The upper bound on the Pd grade shell (0.1 ppm) was selected to coincide with the average grade of Pd within the estimated mineral resource at New Afton (RPA, 2015). The “high-grade” Pt shell (0.01 ppm) was arbitrarily set to an order of magnitude less than Pd based on the general paucity of Pt relative to Pd throughout the deposit, where it occurs in sparse pockets where Pd grades are highest. The Cu and Au grade shells were created using cut-off grades of 4600 ppm and 0.4 ppm, respectively. These values were chosen as they are cut-off grades used in the New Afton resource model.

Palladium grades show varying degrees of control at different depths within the deposit. Palladium grades either decrease outwards, or are slightly offset, from the monzonite intrusion, or grades decrease sharply over distances less than 10 m (Fig. 2.17A-C). The latter case is generally restricted to areas where deposit-scale structures intersect one another. High Cu, Au, and Ag grades typically show the same pattern within the hypogene ore zone of the deposit. High Pd shells are offset from the highest grades of Cu and Au, with some minor overlap, such as in areas where grades sharply decrease due to fault offset. The highest-grade zones of Pd are directly adjacent to the main monzonite stock below the 4500 m level and surrounding smaller monzonite dykes above the 5150 m level of the deposit, which host extensive, concentric shells around them (Fig. 2.17A). Sinuous monzonite dykes in the southern portion of the 5150 m deposit level are surrounded by outward-decreasing Pd grades and locally bounded to the north on the hanging wall side of the J-fault (Fig. 2.17A). Some higher concentrations of Pd occur farther north, on the footwall side of the J-fault, along the southern contact of a monzonite dyke. This zone is a continuous shell with the main body of Pd mineralization in the central part of the deposit, along with two small zones of low Pd grades at the intersection of the K-fault at Footwall fault (Fig. 2.17A). Grade shells extending eastward of the main monzonite body in the central part of the deposit at the 5150 m level are generally low and extensive, while the northern edge, hosting the highest degrees of Pd mineralization, abruptly drops from 0.2 ppm to <0.02 ppm over a few meters. The J-fault, slightly further north, dips south toward the monzonite, eventually following the plunge of the intrusion at depth.

Lower in the deposit, Pd grades are highest close to the main monzonite porphyry body (Fig. 2.17B). Along the western monzonite-volcanic contact, grade shells wrap around the intrusion, but the porphyry itself is unmineralized. An elongate 0.02 ppm grade shell extends along the northern margin of the intrusion, eastward and parallel to the J-fault until its termination at the intersection of the B and Z-faults (Fig. 2.17B). Copper, Au, and Ag enrichment is absent in the western PGE mineralization but follows the eastward trending shell. Isolated porphyritic fingers of the Pothook diorite are found intermingled with the monzonite at and above the 4500 m level shown in Figure 2.17B. Moving a further 100 meters lower, the J-fault continues dipping south and is offset from Pd mineralization (Fig. 2.17C). Here, Pd grades follow the northern margin of the intrusion, with the highest values (>0.2 ppm) slightly offset from the monzonite. The eastern and western edges of the grade shell extend outwards towards background Pd levels. A high-grade zone is present ~25 m away from the main intrusion, extending south towards the intrusion but sharply terminated to the north. Both Cu and Au mineralization coincide with Pd. The J-fault splay to the north dips southerly and wraps around the monzonite and joins at both ends to the main J-fault.

2.4.6 Fluid evolution

The thermodynamic modelling considers a system dominated by either low-salinity vapor (e.g. Williams-Jones and Heinrich, 2005) or high-salinity liquid (Audétat et al., 2008; Lerchbaumer and Audétat, 2012) and shows that the co-precipitation of Pd and Ni can be explained solely through the cooling of a hydrothermal system, regardless of the fluid

conditions that may have prevailed during mineralization. Reliable experimental data on the solubility of PdS_2 unfortunately do not exist (Pan and Wood, 1994), so we tentatively allow the phase PdS to represent a Pd component in pyrite.

2.4.6.1 Low-salinity vapor

In the case of the low-salinity vapor ($\Sigma\text{Cl} = 1 \text{ m}$), Ni and Co are both transported as Cl^- complexes at all temperatures (Fig. 2.18A). The solubility of both metals depends strongly on temperature and the a_{K^+} , being diminished at lower values of both. For a cooling system containing the mineral assemblage pyrite and magnetite, a fluid containing 10 ppm NiCl_3^{2-} would begin to precipitate NiS_2 at $\sim 438^\circ\text{C}$ for an $a_{\text{K}^+} = 10^{-3}$ (Fig. 2.18A). In the same system, but instead with $a_{\text{K}^+} = 10^{-1}$, NiCl_3^{2-} would precipitate as NiS_2 at $< 300^\circ\text{C}$. Ten ppm CoCl^+ would precipitate at $\sim 495^\circ\text{C}$ for $a_{\text{K}^+} = 10^{-3}$ or $\sim 460^\circ\text{C}$ by increasing a_{K^+} to 10^{-1} . Palladium would be transported mainly as $\text{Pd}(\text{HS})_2$ at most temperatures (Fig. 2.18B). The solubility of Pd as $\text{Pd}(\text{HS})_2$ is relatively unaffected by changes in the a_{K^+} in the fluid, unless a_{K^+} rises above a value of $\sim 10^{-1}$ when the PdCl_3^- complex would begin to predominate over $\text{Pd}(\text{HS})_2$ at higher $f\text{S}_2$. A fluid carrying 1 ppm Pd as $\text{Pd}(\text{HS})_2$ would precipitate PdS at $\sim 445^\circ\text{C}$.

Copper solubility is similarly dependent on both the temperature and the relative acidity (through varying a_{K^+}) of the fluid (Fig. 2.19A). A fluid carrying 0.1 wt% CuCl_3^{2-} in solution with an $a_{\text{K}^+} = 10^{-3}$ would first precipitate chalcopyrite in the presence of pyrite and magnetite at $\sim 480^\circ\text{C}$, with the solubility decreasing to 0.01 wt% over about 15°C .

Increasing the a_{K^+} to 10^{-1} decreases the saturation temperature of 0.1 wt% CuCl_3^{2-} by ~ 30 °C, to 445 °C. The increase in a_{K^+} also increases the overall solubility of Cu in the fluid, as indicated by the larger gap between the 0.1 wt% and 0.01 wt% CuCl_3^{2-} solubility contours (Fig. 2.19A).

2.4.6.2 High-salinity brine

A high-salinity liquid would also transport both Ni and Co as Cl^- complexes regardless of temperature (Fig. 2.20A). The higher ΣCl concentration (8 m) moderately increases the solubility of the metals, decreasing the temperature at which they would begin to precipitate (in a solution carrying 10 ppm Ni and 10 ppm Co with an $a_{K^+} = 10^{-3}$) to ~ 415 °C and ~ 490 °C, respectively. The solubilities of both metals in the high-salinity liquid depend on the a_{K^+} in the fluid, in the same manner as in the low-salinity vapor. The ligand responsible for Pd transport in the high-salinity liquid depends on the a_{K^+} and, to a smaller degree, on the amount of Pd in solution (Fig. 2.20B). Below an a_{K^+} of $\sim 10^{-3}$, Pd is transported as the $\text{Pd}(\text{HS})_2$ complex in the presence of pyrite and magnetite. On the other hand, a fluid with a_{K^+} at or above 10^{-1} will transport Pd mainly as PdCl_4^{2-} , until temperatures between 420 °C and 400 °C, when $\text{Pd}(\text{HS})_2$ becomes the predominant complex for a fluid with ~ 10 ppb Pd in solution (Fig. 2.20B).

The increase in ΣCl in the fluid has a marked effect on the solubility of Cu (Fig. 2.19B). The solubility temperature of 0.1 wt% CuCl_3^{2-} is reduced from ~ 480 °C in the low-salinity vapor phase to ~ 435 °C in the high-salinity brine, for an $a_{K^+} = 10^{-3}$ or 380 °C for

$a_{K^+} = 10^{-1}$. For low K^+ activities (10^{-5}), the solubility of 0.1 wt% Cu increases to ~ 460 °C. Although the relative temperature of precipitation is affected by altering the ΣCl and/or the a_{K^+} in the fluid, chalcopyrite precipitation still occurs largely in response to decreasing temperature (Fig. 2.19).

2.5 Discussion

2.5.1 Spatial distribution of PGE and Au

Combining the results of the samples collected in this study with the drill assay database provided by New Gold allows us to analyze the distribution of PGE mineralization in the New Afton deposit below the surface. Group 2 samples, with high Pd and low concentrations of Au, are almost entirely contained within the shallow levels of the deposit and are found near or within carbonate-healed faults and structures but may also occur sparsely at depth. Block Pd-Au ratios, calculated using the kriged Pd and Au models, are generally in agreement with the Pd-Au ratios of samples collected in this study. Overall, although the manifestation of PGE mineralization as discrete, intergrown PGM in these samples might be favorable for beneficiation, their occurrence is sporadic and represents an insignificant fraction of the total PGE budget of the system. Samples in Group 1, where Au is equal to or greater than Pd, are dispersed throughout the entire deposit. The highest-grade samples, either measured through Pb fire assay in this study or routine fire assays during drilling, are always surrounded by lower, but significant (>0.1 ppm), Pd enrichments. Primary Cu-Au mineralization is present in all Group 1 samples and Pd

occurs in ppm concentrations as a trace constituent in pyrite that was precipitated either before or during the main stage ore enrichment, indicating that PGE mineralization initially occurred during the high temperature, hypogene event.

Deposit-scale elevated Pd grades (Fig. 2.17) are located around the Cherry Creek monzonite and commonly occur associated with Cu and Au enrichment. The grade shells show that Pd may either extend outwards, with little disruption by mapped structures (Fig. 2.17C) or sharply terminate over distances of a couple meters (Fig. 2.17A-C). It is unlikely that the processes controlling Pd deposition changed sharply within the main ore zone leading to two separate modes of Pd occurrence. Rather, the areas that show sharp discontinuities in Pd are spatially associated with pre-mineralization deposit-scale structures (J-fault, HWF, FWF). For example, in Figure 2.17C, in the central portion of the Pd grade shell where grades decrease by an order of magnitude over ~5 meters, the traces of the J-fault splay and the Footwall Fault are located only a few meters to the north. Both faults dip to the south, toward the Pd mineralization and monzonite intrusion. Either these structures acted as barriers to early magmatic-hydrothermal fluids, thereby restricting the domain of mineralization (evident by the zones not bounded by faults, where Pd zones are broad), or post-mineralization reactivation of the faults displaced part of the enriched zone. The latter case is unlikely because of the continuity between grade shells on either side of the J-fault in the upper levels of the deposit (Fig. 2.17A).

The monzonite, as currently mapped at depth, represents only 0.01 km³ of magma. Mass balance calculations using kriged ore tonnages for Pd (23.4 to 36.9 t) show that ~1

km³ of magma is required to produce the total amount of Pd in the deposit for a magma that contains 10 ppb Pd. Instead, it is more likely that the PGE-bearing fluids were brought up from depth, possibly sourced from underlying Iron Mask magmas, exploiting pre-mineralization deposit-scale structures and were limited to the south by the hanging wall fault and picrite. Later fluids acted to remobilize and precipitate Pd in small, high grade pockets unrelated to primary Cu and Au mineralization (Group 2 samples).

2.5.2 Fluid evolution and controls on Pd mineralization

The discovery of high Pd tenors in pyrite (Hanley et al., 2010; this study) in the presence of Cu-sulfide mineralization in Au-rich samples indicates that Pd enrichment occurred early in the magmatic-hydrothermal history at New Afton, co-precipitating with the early, high temperature Cu-Au hypogene mineralization. This is further supported by the close association between pervasive low-grade deposit-scale Pd concentration (Pd > 0.1 ppm) and the main Cu-Au orebodies, as shown by the grade shell and block model analysis. Although coarse-grained PGM are found within high Pd samples, they likely represent only a fraction of the entire Pd budget and, in some cases, are representative of localized redistribution and subsequent enrichment of Pd.

2.5.2.1 Pd incorporation in pyrite

LA-ICPMS time spectra of rastered pyrite analyses and PCA from our study show that elevated Pd systematically occurred with both Ni and Se enrichment (Fig. 2.16A-D).

The first principal component (PC1) contains significant loadings from all potentially economic metals, including Pd (Fig. 2.14). This suggests that this variable reflects the composition and abundance of pyrite grains that are associated with the hypogene ore zone; however, comparison with pyrite from outside the main ore zone would be required to further support this interpretation. Meanwhile, the large loading of both Pd and Ni on the second principal component show that it records a population of pyrite within the hypogene ore zone that is interpreted to have been affected by the main episode of PGE enrichment, directly related to the formation of Pd-Ni-rich pyrite.

In the analyzed pyrite, Co-Pd decoupling is present in only a few of the Pd-rich grains and, in some cases, Co and Pd enrichment occurred at the same time (Fig. 2.16A-B, D). A previous study by Hanley et al. (2010) suggested that Pd mineralization at New Afton was contemporaneous with early Co-Ni enrichment and the precipitation of Co-Ni-rich pyrite, where they identified pyrite with early Co-rich cores and later Ni-Pd-rich rims. They found that Pd enrichment was decoupled from the earliest Co-enriched pyrite and precipitated, along with Ni, along the rims of pyrite crystals. Our results show that although Pd and Co correlate (Co concentrations could not be quantified due to the lack of a standard reference material), Co enrichment in pyrite is not necessarily a vector towards Pd enrichment. Instead, Pd in pyrite consistently occurs with elevated Ni contents that are also detectable through routine EPMA analysis (Table D.2.).

The occurrence of Pd and Ni in pyrite at the New Afton deposit may be described with reference to the sulfidation reaction

where vaesite (NiS_2) represents the Ni component in the “bravoite” solid solution $(\text{Fe, Ni, Co})\text{S}_2$. Reaction (1) shows that under highly sulfidizing and/or oxidized conditions the formation of the vaesite component in pyrite is expected to be favored over pentlandite. We suggest that whereas pentlandite is known to host Pd as lattice-substitutions in Ni-Cu-PGE massive sulfide deposits (Barnes et al., 2006; Dare et al., 2010; Mansur et al., 2019), at higher $f\text{S}_2$ Pd instead will tend to follow Ni into the vaesite component of pyrite.

The incorporation of structurally bound Pd into pyrite can be described similarly to that of pentlandite. Pentlandite has a cubic structure and the formula M_9S_8 . It readily accommodates mixtures of Fe, Co, and Ni in both the octahedral and tetrahedral coordinated *M*-site and Pd and Ag into octahedral *M*-sites exclusively (Hall and Stewart, 1973; Burdett and Miller, 1987). Similarly, metals in the pyrite structure occur in the octahedrally coordinated *M*-site (MX_2 ; $\text{X} = \text{S, Sb, Se, As, Te, Bi}$), while the introduction of Pd^{2+} into the crystal may remove the requirement for octahedral coordination around M and instead bond as the square-planar $\text{PdS}_2/\text{PdSe}_2$ pseudotetragonal structure type with S-S and Se-Se pairs (Gronvold and Rost, 1957; Brostigen and Kjekshus, 1970). It is not necessarily the case that Pd^{2+} must bond in pyrite as a pseudotetragonal structure, as other Pd-homeotypes of pyrite exist, notably PdSb_2 and PdAs_2 (Brese and von Schnering, 1994) as well as Pd-sulfarsenides, -sulfantimonides, and -sulfobismutides (Makovicky, 2006). The effective ionic radius of the Pd^{2+} ion decreases from 0.86 Å in octahedral coordination to 0.64 Å in square-planar coordination, closer to the 0.61 Å and 0.69 Å effective ionic radius of the octahedrally coordinated low-spin Fe^{2+} ion and Ni^{2+} ion, respectively

(Shannon, 1976). This suggests that Pd^{2+} may be permitted into crystallizing pyrite in a square-planar coordination with Se-Se and/or S-S, substituting for Fe^{2+} and Ni^{2+} , or in octahedral coordination with any remaining covalent X-X pair. The above arguments do not preclude the possibility that Pd was incorporated as nanoparticles in pyrite, rather than a lattice substitution (González-Jiménez and Reich, 2017; Kravtsova et al., 2020).

In hydrothermal systems, Ni transport is dominated by the chloride complexes NiCl^+ , $\text{NiCl}_{2(\text{aq})}$, and NiCl_3^- at temperatures up to 700 °C and its solubility is markedly lower than that of Co (Liu et al., 2012). The reaction

describes the precipitation of dissolved NiCl_2 as vaesite in the presence of magnetite and pyrite. A similar reaction can be defined to describe the precipitation of the Co-component in pyrite. Its solubility is therefore controlled by decreasing temperature, increasing $f\text{S}_2$, decreasing $f\text{O}_2$, decreasing pH, and decreasing Cl content of the fluid under isobaric conditions. The limited solubility of Ni relative to Co in the fluid suggests that it would be depleted with respect to Ni shortly after reaching vaesite saturation.

Studies on the speciation and complexation of Pd in hydrothermal fluids have shown that in the 2+ valence state, Pd is transported as a chloride complex at high temperatures (>500 °C) and moderate pressures (~200 MPa) (Tagirov et al., 2013; Sullivan, 2020). At temperatures below 300 °C, the complexing behavior of Pd becomes largely dependent on the pH, $f\text{O}_2$, and $f\text{S}_2$ conditions of the fluid. In acidic ($\text{pH} < 4$) and oxidizing fluids (hematite-stable), Pd remains soluble as a chloride complex, particularly PdCl_4^{2-}

(Mountain and Wood, 1988; Wood et al., 1992; Tagirov et al., 2013). Fluids with high fS_2 , slightly acidic to near-neutral pH, and relatively reduced fO_2 ($H_2S_{(aq)}$ predominant) are capable of transporting more than 1 ppm Pd as bisulfide-complexes (Pan and Wood, 1994; Barnes and Liu, 2012).

The precipitation of Pd from a hydrothermal fluid as a structural component in pyrite is described by the reaction

or

in the presence of magnetite and pyrite. Both reactions are temperature and fS_2 dependent, but the effects of Cl^- and pH are only relevant controls on the solubility of Pd as Cl^- complexes. The precipitation of Pd as PdS_2 occurs primarily in response to decreasing temperature. This shows that in a cooling system, at certain fS_2 conditions, dissolved Pd and Ni in solution may share similar constraints on their aqueous solubility, reaching saturation and favoring their incorporation into pyrite under the same conditions.

2.5.2.2 Precipitation of platinum group minerals

The precipitation of PGM throughout the deposit was largely controlled by local processes during the evolution of the mineralizing fluids. This is supported by the presence of individual grains and clusters of PGM in both highly sulfidizing (Group 1) and/or highly

oxidized (Group 2) environments, characterized the prevalence of Cu sulfides or martite within the samples.

The nature of the PGM in Group 1 samples is enigmatic due to their multiple mineralogical associations, variable morphologies, and association with Pd-rich pyrite. Platinum group minerals associated with primary sulfide mineralization and early, high temperature potassic alteration have been identified in the alkalic Cu-Au $\pm$ (Pd, Pt, Te) Skouries porphyry deposit in Greece (McFall et al., 2018), the Elastite Cu-Au porphyry deposit in Bulgaria (Tarkian et al., 2003), and the Santo Thomas II porphyry deposit in the Philippines (Tarkian and Koopmann, 1995), with the main PGE-bearing phases being Pd-Sb-Bi-Te minerals and associated Ag-Au tellurides. It is unlikely then that a separate hydrothermally-deposited Bi-Te melt phase was the host for the PGM observed at New Afton or present in the hydrothermal fluids, as has been proposed for other Pd-rich alkalic porphyry deposits (McFall et al., 2018). The PGE-bearing phases identified at New Afton all notably lacked Bi and were not associated with other Bi-bearing minerals. Instead, they comprised some combination of metals in the system Pd-Pt-Au-Ag-Sb-As-Te-Hg. Additionally, the sulfide-associated Group 1 PGM in our samples (Fig. 2.11) do not share the globular morphology noted by McFall et al. (2018) at the Skouries deposit and were never found to be hosted within hydrothermal quartz. Instead, the PGM were intergrown with chalcopyrite, typically in contact with Pd-rich pyrite, and forming simple to complex intergrowths with electrum and hessite.

Coexisting pairs of hessite (Ag_2Te) and electrum (Au-Ag alloy) act to buffer the $f\text{Te}_2$ in hydrothermal systems, given the reaction

where Ag is a component in the electrum solid solution (Afifi et al., 1988a). Limited and transient input of magmatic H_2Te into a hydrothermal system, following the input of magmatic sulfur, may stabilize Te-bearing precious metal phases (Afifi et al., 1988b). At New Afton, there is a ubiquitous association between electrum and hessite $\pm$ PGM. In general, the electrum grains are much larger than the coexisting hessite crystals and Pd-Sb $\pm$ Hg $\pm$ Te $\pm$ As minerals along their margins (Fig. 2.11C). Two electrum crystals analyzed by EPMA in this study, due to mistaken identification as PGM during probing, were found to contain 0.02 wt% Pd.

Platinum group minerals associated with Group 2 samples are, like those in Group 1, predominately Pd-Sb-As $\pm$ Te $\pm$ Hg phases but differ in their mineralogical association and host phases (Fig. 2.11E-H). They are always associated with porous magnetite that has undergone partial martitization, and apatite, Fe-carbonate alteration, and pervasive fracturing of the host rock. The PGM are found along grain boundaries of martite, within magnetite that has not yet undergone oxidation, or, in one case, as a free grain in a cataclastite. In these samples, the total absence of sulfide minerals suggests that the PGE-bearing fluid carried the PGE as Cl complexes due to the lack of reduced S in the fluid, which requires the fluids to have been quite oxidized, where hematite is stable instead of magnetite.

2.5.2.3 Hypogene fluid evolution

Early mineralizing veins at New Afton, and those sampled in this study, defined by the assemblage chalcopyrite-pyrite-magnetite, are crosscut by later chalcopyrite-pyrite-hematite (var. specularite) veins (Fig. 2.8). In these samples of primary hypogene Cu-Au mineralization, Pd-Ni-Co-rich pyrite occurs within the earlier chalcopyrite-pyrite-magnetite assemblage. The fluid must have deposited its Pd prior to the system reaching the reaction pyrite-magnetite-hematite, while its redox and sulfidation state was still largely buffered by the host rock, perhaps due to rapid cooling as the fluid entered previously unaltered rock. Figures 2.18 and 2.20 show the T-log f_{S_2} evolution of the system prior to and during hypogene mineralization for Ni, Co, and Pd for both the low-salinity and high-salinity fluids, respectively.

In both systems, the country rocks were initially heated and subjected to the influx of magmatic buffered hydrothermal fluids until their subsequent partial equilibration with the country rocks. The early rock buffer mineral assemblage consisted of pyrite-magnetite-pyrrhotite-ilmenite-K-feldspar-muscovite-quartz. Through the subsequent addition of fluids, conditions evolved to the pyrite-magnetite-ilmenite stability field either prior to or during Co-pyrite and chalcopyrite precipitation depending on the amount of pyrrhotite initially present and the activity of ilmenite in rhombohedral oxide in the unaltered country rocks. Interaction between the fluid and country rocks caused the fluid to cool, leading to the early precipitation of Co-pyrite (Hanley et al., 2010) followed closely by the deposition of chalcopyrite (Fig. 2.19). Further cooling of the fluid led to the mutual precipitation of

both Ni and Pd as lattice substitutions in pyrite, while Co continued to precipitate as a component in pyrite, although becoming increasingly depleted in the fluid (e.g., Figs. 2.18, 2.20). Palladium, Ni, and Co were all subsequently depleted from the fluid upon its continued cooling and prior to the system reaching the pyrite-magnetite-hematite reaction. Hypogene Cu mineralization continued as the system cooled along the pyrite-magnetite-hematite reaction line (Fig. 2.19).

Whether the mineralizing fluids were low-salinity or high-salinity has no bearing on the final order of mineral precipitation. Our models predict the precipitation of the minerals (in order by decreasing temperature): Co-rich pyrite, chalcopyrite, Ni-Pd-rich pyrite, all within the pyrite-magnetite stability field (Figs. 2.18-2.20). The precipitation of the Ni-Pd±Co-rich pyrite is entirely described by a simple cooling model; a precious-metal bearing hydrothermal fluid equilibrating with surrounding country rocks will precipitate Pd (and Ni) solely in response to decreasing temperature.

The precipitation temperature for each of the previously mentioned phases that we have presented here are meant to be qualitative; the actual temperatures at which the phases would begin to precipitate are dependent on several other factors. Perhaps the most significant influence is the activity of the PdS_2 and NiS_2 components in pyrite. If those components are in solid solution within pyrite and we assume ideal mixing, then the activities of the PdS_2 and NiS_2 components are on the order of 10^{-6} and 10^{-2} , respectively. Reactions (3) and (4) show that if a_{PdS_2} or a_{NiS_2} are less than unity, the reaction will proceed to the right and favour the precipitation of PdS_2 or NiS_2 . Another factor that was

briefly mentioned in the preceding text is the effects of changing the a_{K^+} while buffered by the mineral assemblage K-feldspar-muscovite-quartz. Decreasing a_{K^+} increases the temperature that any species transported as a Cl^- complex reaches saturation in the fluid. The precipitation of the minerals from the real system would certainly have been affected by a combination of all the above factors, but we again stress that deposition of Pd and Ni, as well as Cu and Co, to form primary hypogene mineralization is most adequately explained by the simple cooling of a magmatic-hydrothermal fluid. Changing assumptions regarding the type of fluid present do not affect this first order temperature control on metal solubilities.

The association of Pd-Sb-As $\pm$ Te $\pm$ Hg phases with both primary Cu-Au mineralization and later carbonate-rich, oxidized alteration suggests that the PGM were not precipitated during the primary hypogene event, but rather at some subsequent stage of alteration. Platinum group minerals in Group 1 samples are found along the grain boundaries of chalcopyrite and pyrite, whereas compositionally similar PGM are also found within martite in Group 2 sample. One possibility is that the precipitation of PGM was caused by secondary remobilization and reprecipitation of previously deposited Pd by an oxidized, sulfur-poor fluid (Fig. 2.21) that destroyed metastable primary hypogene pyrite-vaesite solid solutions. A hematite-stable hydrothermal fluid (with a non-zero salinity) is capable of oxidizing PdS (our proxy for the Pd-rich pyrite component) to stabilize metallic Pd (our proxy for PGM) at isothermal conditions (Fig. 2.21). If the fluid is sufficiently oxidized or Cl-rich, in the absence of sulfur, it has the capacity to transport ~ 1 ppm Pd as $PdCl_3^-$ (Fig. 2.21). Localized interaction of that fluid with magnetite would

buffer to more reduced conditions, forcing the precipitation of metallic Pd or its analogs in other PGM in the presence of other bonding elements such as Sb, As, etc.

It is intriguing to consider the possibility that there exists a Pd-rich porphyry end member deposit type that has escaped recognition because it is not spatially associated with ore-grade Cu or Au mineralization. The results of the modeling performed here indicate that there is a range of conditions in hypersaline brines in which Pd is expected to precipitate at temperatures well above 400 °C whereas Cu remains in solution to temperatures closer to 300 °C. The effect is accentuated in K-rich systems. It is therefore possible that under some circumstances in Cl-rich potassic magmas hydrothermal Pd mineralization might occur in a zone inside the Cu-rich hypogene shell, perhaps not associated with obvious signs of porphyry-style mineralization.

2.6 Conclusions

Platinum group element mineralization at New Afton occurred during the deposition of the primary Cu-Au ore body. Palladium was deposited primarily as a trace component in nickeliferous pyrite during the precipitation of chalcopyrite-pyrite $\pm$ magnetite. Pd-rich pyrite that may contain >10 ppm Pd is always associated with simultaneous weight-percent enrichments in Ni $\pm$ Co that are detectable by both EPMA and SEM. The co-precipitation of both Pd and Ni as components in pyrite can be described entirely by the cooling of an orthomagmatic hydrothermal fluid, regardless of whether it is a low-salinity vapor or a high-salinity liquid phase.

Compositionally similar PGM within separate alteration facies are suggestive that their formation is not directly linked to the deposition of Pd with the main Cu-Au ore. A model where the secondary remobilization of Pd as Cl^- complexes by an oxidized fluid at a lower temperature caused local depletions and enrichments in Pd adequately describes the mineralogical occurrences of the PGM in each alteration facies. Although the PGM are intriguing, their economic viability as a source of PGE is quite low; the PGM are volumetrically insignificant compared to the main stage Pd-rich pyrite.

The porphyry mapped at New Afton formed from an influx of metals and sulfur transported by a volatile phase from an underlying, degassing magma reservoir that continuously exploited the same structures that controlled the emplacement and elongation of the dioritic and monzonitic intrusions. Although the magma was certainly enriched in precious metals, it only had the capacity to deposit one fifth of its Pd relative to Au. Recent experimental work suggests that Au is more readily partitioned than Pd from intermediate magmas to an exsolving vapor phase (Sullivan et al., 2018; Sullivan, 2020). Whether the PGE that are unaccounted for remained behind in the solidifying magma or were transported outside of the porphyry environment is beyond the scope of this study, but there is undoubtedly a major PGE anomaly surrounding New Afton, as documented by our estimate of the average background Pd concentration in weakly altered volcanic host rocks. Comparison with more deposit-scale studies is necessary to fully address the behaviour of PGE in Au-bearing porphyry systems. Pd mineralization in chloride-rich potassic

magmatic-orthomagmatic fluid systems may occur at higher temperatures than the corresponding Cu deposits, leading to failure to observe them.

2.7 References

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2.8 Figures

Figure 2.1: Regional and local geology around New Afton . *A*: Regional geology around the Iron Mask batholith compiled from data from the British Columbia Geological Survey. *B*: Deposit geology around the New Afton mine property at ~600 meters below the surface. The coordinates of the deposit map are in Mine Grid, which is a 40° rotation of the UTM grid. Letters denote the fault names used by the New Gold geologists. Reproduced from Lipske and Wade (2014). *Inset*: Map of British Columbia and the five morphogeological belts that make up the Canadian Cordillera.

Figure 2.2: Cross sections of the New Afton deposit facing east showing major lithologies, alteration, and mineralization domains. The major pre-mineralization, regional faults (HWF, FWF, and J) are shown in each cross section. Copper and Au grade shells are plotted and are closely associated with the Cherry Creek monzonite and biotite-dominant potassic alteration. Depths are present in mine reference level, with the surface located at ~5700 m.

Figure 2.3: Locations of samples (at depth) used in this study. The figure shows the location of the collected samples in green, against the Cherry Creek monzonite (orange). The historic Afton pit is above the shallowest parts of the porphyry.

Figure 2.4: Silhouette plot of k -means clustered data. The optimum number of clusters corresponds to the maximum silhouette number, such that data within each cluster is most similar to each other and least similar to data in other clusters.

Figure 2.5: Histogram of Pt, Pd, and Au fire assays from collected samples.

Figure 2.6: Cumulative probability plot of $\log_{10}$ -transformed Pd/Au ratios in the fire assayed samples. The plot shows two distinct log-normal populations, one below $\log_{10}(\text{Pd/Au}) = -1$ and the other between approximately -0.5 to 1.5. A third, albeit insufficiently sampled, population is emerging from log-ratios greater than ~ 1.5 . These three populations correspond to the statistical sample groups 1, 3, and 2, respectively.

Figure 2.7: Distribution of Pd, Pt, and Au in the grouped samples. The data were grouped using a k-means clustering algorithm. Three clusters were selected as being the optimum number of clusters for the dataset, determined by iteratively maximizing the silhouette number for a range of cluster sizes and backed by mineralogical and textural similarities between samples in each group. The central line in each box is the median of the data and the top and bottom of the boxes are representative of the 75th and 25th percentiles. The whiskers correspond to $\sim 2.7\sigma$, or the 99th percentile; outliers are all data plotted beyond the whiskers. The statistics of each group are shown on the right side of the figure. The sample groups are organized into: (1) high Pd, low Pd/Au, (2) high Pd, high Pd/Au, and (3) low Pd. The groups are considered to reflect different processes controlling Pd remobilization and deposition.

Figure 2.8: Representative Group 1 samples. The scale bars in A – D are all 1 cm. Mineral abbreviations are: Ap – apatite, Bt – biotite, Carb – carbonate, Ccp – chalcopyrite, Chl – chlorite, Ksp – k-feldspar, Mag – magnetite, Py – pyrite, Qtz – quartz, Spec – specularite. (A) Hypogene sulfide mineralization surrounded by potassium feldspar dominant potassic alteration with biotite and chlorite. (B) Hypogene sulfide mineralization associated with potassium feldspar alteration in contact with volcanic rocks selectively overprinted by white mica and quartz. (C) Biotite dominant alteration with chalcopyrite + pyrite flooding.

Early biotite is partially retrograde altered to chlorite. The sample is cut by a late quartz-carbonate vein with no visible surrounding alteration. (D) Early magnetite-apatite vein overprinted with primary hypogene Cu-Au-PGE mineralization, crosscut by a later chalcopyrite-specularite vein. (E) Potassically altered volcanic rocks (biotite-dominant) with chalcopyrite and pyrite (Cu-Au-Pd mineralization) in fault contact with highly argillic altered rocks whose primary lithology is completely overprinted.

Figure 2.9: Representative Group 2 samples. The scale bars in A – C are all 1 cm. Mineral abbreviations are: Ap – apatite, Carb – carbonate, Hem – hematite, Mag – magnetite, Qtz – quartz. The group is dominated by early magnetite-apatite lodes that have been variably faulted and fractured and modified by late CO₂-rich fluids.

Figure 2.10: Cherry Creek monzonite as seen in drill core. The top photo (A) is in the center of a monzonite dyke in the upper levels of the deposit, (B) is approximately 10 meters from the center, near the hornfelsed contact with Nicola volcanic rocks. Abundant miarolitic cavities are present in the center of the intrusion accompanied by a set of parallel propylitic alteration veins. Primary igneous textures are visible in the samples, including the preservation of trachytic feldspars in B. Post-solidification hydrofracturing with quartz-carbonate infill is present throughout the monzonite.

Figure 2.11: SEM BSE images of platinum group minerals from sample groups 1 and 2. Mineral abbreviations are: Ap – apatite, Carb – carbonate, Ccp – chalcopyrite, Mag – magnetite, Mart – martite, Py – pyrite. Platinum group minerals (PGM) in Group 1 samples are associated with sulfide minerals (A-D), often occurring along chalcopyrite-pyrite grain boundaries (A,B) and intergrown with electrum and hessite (B,C). Group 2 samples, characterized predominately by the presence of magnetite and apatite (E-H), host their PGM as inclusions in partially martitized magnetite (E-G), in association with Fe-carbonates (F) and often near large amounts of structural modification (H).

Figure 2.12: Precious metal distribution in all analyzed pyrite and chalcopyrite grains. The boxes cover the 25th to the 75th percentiles and the whiskers extend to the 99th percentile. Data outside the whiskers are considered outliers.

Figure 2.13: Correlation matrix of log-transformed molar chalcopyrite trace element data.

The data were log-transformed prior to analysis. The number in the upper left corner of each plot is the Pearson rank coefficient. Groups or pairs of elements that are correlated with one another are highlighted in the colored boxes.

Figure 2.14: Biplots of the first two principal components for chalcopyrite and pyrite. The left column shows the loadings of each variable on the chalcopyrite principal components and the right column shows them for pyrite. The blue lines (loadings) show the magnitude and direction of a variable on that component. The red dots are the scores of each analyzed mineral. Although the principal component analysis identifies some variation in chalcopyrite trace element concentrations, most of the data are clustered around the origin, the dataset records only minor changes in the composition of chalcopyrite. Conversely, the principal components of pyrite show systematic variations in the trace element composition of the crystals. PC1 in pyrite records the main stage of hypogene ore mineralization, including Cu, Au, Ag, and Pd enrichment, while PC2 exclusively records the period of Pd (+ Ni) enrichment. Although the variations among the chalcopyrite data are generally low, the first two components record variation similar to that found in the principal components of pyrite.

Figure 2.16: LA-ICPMS time spectra of select pyrite grains. (A) Pd rich zones in pyrite are associated with similar zoning in Co, Ni, and Se. (B) Pd rich zoning occurs irrespective of Co zoning. (C) Pd-Ni-Se zoning offset from highest Co-rich zones. (D) Co zoning occurs without enrichments in Ni and Pd.

Figure 2.17: Plans of Pd grade shells at (A) 5150 m level, (B) 4500 m level, and (C) 4400 m level. Deposit-scale structures are labelled with white letters. Palladium grade shells are represented by the white contours. Copper and Au grade shells, representative of hypogene mineralization, are shown as orange and yellow contours, respectively. Palladium grade

shells are in all instances associated with the Cherry Creek monzonite intrusion but are affected in their extent largely by the surrounding deposit-scale structures. Grade shells either extend outwards without disruption or abruptly decrease over a few meters. The highest Pd grades are adjacent to the monzonite, but closely associated with similar lineation.

Figure 2.18: Temperature-log/S₂ diagram of (A) Ni and (B) Pd solubility in a low-salinity vapor dominated system. Solubility contours are shown for the Cl⁻ and HS⁻ complexes stable at the modeled fluid conditions. Mineral abbreviations are Cpn – cobaltpentlandite, Hem – hematite, Hz – heazlewoodite, Mag – magnetite, Mil – millerite, Pn – pentlandite, Py – pyrite, Vs – vaesite. Contours for the activity of ilmenite are shown. Partial phase stability lines are shown for the soluble mineral phases. Log *f*O₂ contours are calculated using reaction 8 above the pyrite-magnetite-hematite reaction line and via the reaction of pyrite and hematite below. The S gas buffer was calculated between SO₂/H₂S = 1 and 10 assuming $a[\text{H}_2\text{S}_{(\text{aq})}] = 0.1$ (Inan and Einaudi, 2002; Einaudi et al., 2003). The approximate rock buffered conditions are shown by the arrow and are representative of an initial assemblage of pyrite-magnetite-pyrrhotite-ilmenite. The activity ratio $a_{\text{H}^+}/a_{\text{K}^+}$ is controlled by the K-feldspar-muscovite-quartz (KMq) buffer, and the effects of changing a_{K^+} on the solubility of the metals are shown. The diagram for Ni shows the effect on solubility of increasing the a_{K^+} to 10⁻³, which raises the temperature at which vaesite precipitates by ~40 °C (relative to an activity of K⁺ = 0.1), close the temperature where Pd precipitates as PdS. The decrease in a_{K^+} has no effect on the solubility of Pd as a HS⁻ complex at conditions relevant to our system. Also shown is the solubility of Co as CoCl⁺, which would precipitate from the fluid as CoS₂ at a temperature ~40 °C above that of Ni.

Figure 2.19: Temperature-log/S₂ diagram of Cu and Co solubility for a low-salinity vapor and high-salinity liquid. The parameters used in this diagram are the same as Figs. 2.15 and 2.16. The precipitation of chalcopyrite from CuCl₃²⁻ is largely affected by decreasing temperature. The relative temperature that chalcopyrite will precipitate is affected by the total Cl, a_{K^+} , and Cu concentration in the fluid. In all cases, CoS₂ precipitates prior to the onset of chalcopyrite formation.

Figure 2.20: Temperature- $\log/S_2$ diagram of (A) Ni and (B) Pd solubility in a high-salinity brine dominated system. Solubility contours are shown for the Cl^- and HS^- complexes stable at the modeled fluid conditions. Mineral abbreviations are Cpn – cobaltpentlandite, Hem – hematite, Hz – heazlewoodite, Mag – magnetite, Mil – millerite, Pn – pentlandite, Py – pyrite, Vs – vaesite. The parameters used in this diagram are the same as Fig. 2.15, except that the total Cl in the system is increased to 8 m.

Figure 2.21: fO_2 vs. pH in the system Fe-O-Cl-Pd at 300 °C. Previously deposited Pd (as PdS) can be remobilized as a Cl^- complex by an oxidized, S-poor fluid. Pd is completely stable over PdS at values of $\log fS_2 < -6$, or oxidized conditions (hematite stable) at higher $\log fS_2$ values. Pyrite has no stable field at values of $\log fS_2$ below approximately -3.5. Pd solubility as the $PdCl_3^-$ complex increases at higher total Cl concentrations. Palladium transported in solution by the oxidized fluid will precipitate upon interaction of the fluid with a reductant, supported by the mineralogy of Group 2 samples where PGM are hosted by martite.

2.9 Tables

Table 2.1. Summary of samples with identified platinum group minerals.

Sample ID	Group	PGM	PGM Host
AF17176-003A	1	x	ccp, py
AF17176-003B	1	x	ccp, py
EA1205-032A	1	x	ccp, py
EA1205-034	1	x	ccp, py
AF8103-052A	1	x	ccp, py
AF8103-052B	1	x	ccp, py
UA7906-057A	1	-	-
EA17196-143	1	-	-
EA17196-144	1	-	-
EA14141-166	1	x	ccp
UA12206-211	1	-	-
EA17183-225	1	x	ccp, ksp
AF3601-027B	2	-	-
AF3601-030	2	-	-
AF8103-048	2	-	-
UA7906-056	2	x	mag, ap
EA13041-107	2	-	-
EA13041-111	2	x	ccp, chl
AF0135-122	2	x	mag, ap, crb
AF0135-129	2	-	-
AF0135-133	2	-	-
PO1634-156	2	-	-
PO1634-160	2	-	-
EA14141-165	2	-	-
EA14141-168	2	-	-

* Group refers to the group number assigned by *k*-means clustering. Ccp – chalcopyrite; Py – pyrite; Mag – magnetite; Ap – apatite; Chl – chlorite; Crb – carbonate; Ksp – k-feldspar.

Table 2.2. Platinum group mineral analyses.

Sample	S (wt%)	Pt (wt%)	Pd (wt%)	Fe (wt%)	Cu (wt%)	Hg (wt%)	Sb (wt%)	Te (wt%)	Ag (wt%)	As (wt%)	Total
003A_PGM_1	0.01	0.25	71.10	0.19	1.57	0.00	15.86	0.00	2.89	8.89	100.76
003A_PGM_2	0.03	0.34	71.77	0.36	0.52	0.01	25.35	0.00	0.15	3.09	101.62
003B_PGM_1	0.05	0.17	71.57	0.83	1.31	0.00	15.68	0.00	2.90	8.92	101.43
003B_PGM_3	0.05	0.38	71.62	0.59	0.32	0.02	25.41	0.00	0.11	3.05	101.54
003B_PGM_4	0.02	0.23	71.82	0.45	1.70	0.08	15.85	0.00	2.05	8.94	101.13
003B_PGM_5	0.02	0.29	71.54	0.32	0.45	0.03	25.71	0.00	0.06	2.89	101.32
003B_PGM_6	0.03	0.31	72.51	0.56	0.93	0.05	25.38	0.00	0.07	2.96	102.79
032A_PGM_1	0.21	0.00	35.95	2.01	0.26	22.06	0.29	43.82	0.10	0.00	104.70
034_PGM_1	0.20	0.00	35.01	1.06	1.66	20.82	0.49	42.15	0.28	0.00	101.68
034_PGM_2	0.31	0.20	68.84	1.39	1.99	0.57	10.21	0.00	5.54	12.76	101.81
034_PGM_3	0.07	0.00	35.77	0.80	1.27	21.36	0.50	43.26	0.20	0.00	103.22
034_PGM_4	0.30	0.00	35.21	1.64	1.76	20.31	0.75	42.38	0.47	0.00	102.81
034_PGM_5	0.53	0.00	0.01	1.83	0.18	0.02	0.21	37.34	59.78	0.00	99.90
056_PGM_1	17.63	77.15	6.92	1.91	0.00	0.00	0.00	0.00	0.00	0.00	103.61

Table 2.3. Principal component loadings of trace elements on pyrite.

	<i>PC1</i>	<i>PC2</i>	<i>PC3</i>
Au	0.290	-0.077	0.408
As	0.196	0.081	0.678
Pd	0.266	0.584	-0.187
Mn	0.252	-0.133	0.014
Ni	0.252	0.610	0.102
Cu	0.424	-0.087	-0.471
Se	0.118	0.250	-0.091
Ag	0.343	-0.111	-0.213
Sb	0.288	-0.151	0.166
Te	0.060	0.088	0.150
Pb	0.414	-0.365	0.025
Bi	0.332	-0.103	-0.043

* the loading of each element on the principal components is its scalar coefficient A_i in the linear combination: $PC = A_1 x_1 + A_2 x_2 + \dots + A_n x_n$ where x_i is the concentration of the i -th element.

Table 2.4. Reactions and modified logK values used in thermodynamic models.

Reaction	250°C	300°C	350°C	400°C	450°C	Ref.
<i>Aqueous Species</i>						
$\text{Pd}^{2+} + \text{Cl}^- = \text{PdCl}^+$	5.89	6.50	(7.23) ¹	(8.05) ¹	(8.92) ¹	a
$\text{Pd}^{2+} + 2\text{Cl}^- = \text{PdCl}_2$	9.00	9.90	(11.10) ¹	(12.57) ¹	(14.30) ¹	a
$\text{Pd}^{2+} + 3\text{Cl}^- = \text{PdCl}_3^-$	10.69	11.50	(12.60) ¹	(13.94) ¹	(15.50) ¹	a
$\text{Pd}^{2+} + 4\text{Cl}^- = \text{PdCl}_4^{2-}$	10.61	11.30	(12.30) ¹	(13.59) ¹	(15.12) ¹	a
$\text{PdS} + \text{HS}^- + \text{H}^+ = \text{Pd}(\text{HS})_2$	2.49	2.60	2.80	(3.21) ¹	(3.95) ¹	b
$\text{Au}^+ + 2\text{Cl}^- = \text{AuCl}_2^-$	6.40	6.16	6.01	6.18	6.73	c
$\text{Au}^+ + \text{HS}^- = \text{AuHS}$	14.50	13.43	12.57	12.12	12.67	d
$\text{Au}^+ + 2\text{HS}^- = \text{Au}(\text{HS})_2^-$	19.10	17.74	16.80	16.39	17.13	d
$\text{Ni}^{2+} + 2\text{Cl}^- = \text{NiCl}_2$	1.87	2.98	4.33	6.22	9.56	f
$\text{Ni}^{2+} + 3\text{Cl}^- = \text{NiCl}_3^-$	0.54	1.96	3.70	6.07	10.15	f
<i>Minerals</i>						
$\text{Fe}_{4.5}\text{Ni}_{4.5}\text{S}_8 + 10 \text{H}^+ = 4.5 \text{Ni}^{2+} + 4.5 \text{Fe}^{2+} + 8 \text{HS}^- + \text{H}_{2(\text{aq})}$	-61.67	-65.15	-70.08	-71.77	(-75.21) ¹	e
$\text{NiS}_2 + \text{H}_{2(\text{aq})} = \text{Ni}^{2+} + 2 \text{HS}^-$	-13.90	-14.62	-15.72	-16.65	(-17.70) ¹	e
$\text{NiS} + \text{H}^+ = \text{Ni}^{2+} + \text{HS}^-$	-8.73	-9.09	-9.64	-9.84	(-10.21) ¹	e
$\text{Co}_9\text{S}_8 + 10 \text{H}^+ = 9 \text{Co}^{2+} + 8 \text{HS}^- + \text{H}_{2(\text{aq})}$	-72.65	-74.64	-78.29	-78.83	(-80.96) ¹	e
$\text{CoS}_2 + \text{H}_{2(\text{aq})} = \text{Co}^{2+} + 2 \text{HS}^-$	-14.64	-15.17	-16.11	-16.91	(-17.81) ¹	e
$\text{CoS} + \text{H}^+ = \text{Co}^{2+} + \text{HS}^-$	-7.94	-8.29	-8.83	-9.03	(-9.42) ¹	e

¹ Extrapolated value.

Data sources: (a) Pan et al. (1992); (b) Pan and Wood (1994); (c) Stefansson and Seward (2003); (d) Stefansson and Seward (2004); (e) Klein and Bach (2009); (f) Liu et al (2012)

Appendix A

A.1 Fluid evolution model parameters

The composition, temperature, and intensive parameters of the mineralizing fluids at New Afton are unknown but can be reasonably estimated using mineralogical and textural associations and resemblances to similar modern arc systems.

Early high-temperature magmatic fluids would have been oxidized and buffered by the sulfur gas buffer, or $\text{SO}_2/\text{H}_2\text{S}$ fugacity ratio (Einaudi et al., 2003). Unreacted magmatic vapors from degassing andesitic magmas have $\text{SO}_2/\text{H}_2\text{S}$ gas ratios between 1 to 10 (Burnham and Ohmoto, 1980; Einaudi et al., 2003). The two gasses, in equilibrium with one another, act to buffer the $f\text{O}_2$ via the reaction

such that $\log f\text{O}_2 = 2/3 \log(f\text{SO}_2/f\text{H}_2\text{S}) - \log K_{\text{A1}}$. Here, we assume that the magmatic system has consumed sulfide liquid such that the sulfur has been completely converted to vapor, resulting in an $\text{SO}_2/\text{H}_2\text{S}$ ratio near unity (Mungall et al., 2015). Hence, for the following discussion, the sulfur gas buffer ranges between 1 to 10. The choice of any value within the range has no bearing on the final characteristics of the mineralizing system modeled here.

In our models, the activity ratio $a_{\text{H}^+}/a_{\text{K}^+}$ is fixed by the reaction

(KMQ buffer) that describes the hydrolysis of K-feldspar by a reactive fluid to form muscovite (sericite) and quartz. Potassium feldspar and muscovite are present in all Pd-mineralized Group 1 samples, which makes reaction (A2) a reasonable estimation of the relative acidity of the system. The activity ratio $a_{\text{H}^+}/a_{\text{K}^+}$, controlled by reaction (6), remains largely constant at in cooling magmatic-hydrothermal systems as K-feldspar present in the country rocks is progressively metasomatized to muscovite (Giggenbach, 2003). A value of $10^{3.683}$ for the activity ratio was obtained using SUPCRT and used in the following solubility models while simultaneously varying the a_{K^+} in the fluid. The modelled fluids have ΣCl of 1 m (~6 wt% NaCl equiv) and 8 m (~32 wt% NaCl equiv), to represent systems dominated by either a low-salinity vapor phase or a high-salinity liquid phase, respectively.

Magnetite-apatite vein swarms resembling iron oxide-apatite (IOA) mineralization, associated with the early history of the Iron Mask batholith, predate the precipitation of sulfide mineralization at New Afton. The magnetite-apatite veins have previously been proposed as having originated from an immiscible Fe-P-O melt which exsolved from the Iron Mask magmas (Cann, 1976). This finding is supported by recent experimental studies that found intermediate, oxidized, volatile-rich magmas may readily exsolve a Si-poor, Fe-P-oxide melt (Hou et al., 2018; Mungall et al., 2018). Consequently, the formation of early magnetite-apatite swarms can be considered as separate to the fluid evolution associated with porphyry mineralization; that is, the acidity, redox, and sulfidation conditions of the porphyry fluids are not genetically linked to the early magnetite-apatite veins. Unaltered volcanic rocks of the Nicola Group found in the Quesnel terrane are also known to contain

the assemblage pyrite $\pm$ magnetite $\pm$ ilmenite $\pm$ pyrrhotite (Symons, 1985). Although pyrrhotite has not previously been identified at New Afton, ilmenite occurs with magnetite throughout the deposit and rutile/leucoxene were identified in our samples. The fS_2 of the country rocks prior to hypogene mineralization is therefore constrained by the reaction

such that $\log fS_2 = 4 \log a[\text{FeTiO}_3] - 4 \log K_{\text{A3}}$, where $a[\text{FeTiO}_3]$ is the activity of ilmenite in rhombohedral oxide and K_{A3} is the equilibrium constant for reaction A3. Volcanic rhombohedral oxides in the White River Ash of Alaska, USA, and the Yukon, Canada, have compositions with a range of X_{FeTiO_3} between 0.45 and 0.87 (Preece et al., 2014), which are consistent with the miscibility between FeTiO_3 and Fe_2O_3 in rhombohedral oxides (Ghiorso and Evans, 2008). The fO_2 of the country rocks would have been buffered at a lower limit defined by the reaction of pyrrhotite-pyrite-magnetite that coincides with an $a[\text{FeTiO}_3] \cong 0.75$ at 400 °C, in agreement with the composition of volcanic rhombohedral oxides within the White River Ash under the assumption that $a[\text{FeTiO}_3] \approx X_{\text{FeTiO}_3}$.

The presence of early pyrite constrains the fS_2 of the unreacted fluid to within the pyrite stable field at an fO_2 controlled by the S-gas buffer. During the initial stages of metasomatism, the titration of fluids into the country rocks would push the system to evolve along the rock buffered path (Fig. 2.14). As the fluid progressively cooled, the increasing fS_2 and fO_2 would destabilize the volumetrically minor pyrrhotite in the country rocks allowing the system to evolve along a new rock buffered path defined by the equilibrium between pyrite and magnetite:

These conditions would dominate the system during the early stages of hypogene mineralization due to the abundance of magnetite from the preceding magnetite-apatite mineralization while concurrently reducing the activity of ilmenite in rhombohedral oxides. The early system, just prior to Cu-sulfide deposition, would then have an $f\text{S}_2$ greater than the reaction of pyrrhotite and pyrite, an $f\text{O}_2$ greater than the reaction of pyrrhotite-pyrite-magnetite, and a fixed $f\text{S}_2/f\text{O}_2$ ratio defined by the pyrite-magnetite boundary. This stage of cooling and fluid addition would see the formation of (Cu-Au-PGE) hypogene mineralization (Figs. 2.14, 2.15). From this point on the fluid evolves within the pyrite-magnetite stability field as it cools, oxidizing and sulfidizing the earlier reduced mineral assemblage until the start of the reaction between pyrite, magnetite, and hematite. The fluid would continue to cool, following a compositional path along the pyrite-magnetite-hematite reaction line (Fig. 2.15). As the country rocks fully equilibrate with the fluids, their continued reaction would surpass the pyrite-magnetite-hematite reaction and the system would evolve unbuffered towards the S-gas buffer, assuming that the fluid is held at a constant H_2S concentration. In reality, the precipitation of hypogene sulfide minerals would reduce the S content of the fluid, thereby moving the S-gas buffer to smaller values of $\log f\text{S}_2$. The large phyllic alteration zone that encompasses the deposit suggests that the mineralizing fluids at New Afton maintained a high S content for a protracted period, and so the effects of losing S during hypogene sulfide precipitation are minimal.

A.2 References

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Appendix B Fire assay data

Table B.1. Whole rock fire assay data of samples collected.

Sample	Hole ID	Downhole Depth (m)	Au (ppb)	Pt (ppb)	Pd (ppb)
001	AF11-176	211.58	317	11.7	56.1
002A	AF11-176	211.85	833	34.4	613
002B	AF11-176	211.851	460	25.6	474
003A	AF11-176	211.96	6360	107	3670
003B	AF11-176	211.961	1860	85.2	10300
004	AF11-176	212.33	184	9.1	139
005	AF11-176	212.78	157	24.3	506
008	AF07-125B	672.04	1670	4.3	239
009	AF07-125B	672.13	149	2.6	67.7
010	AF07-125B	672.19	679	14.5	92.8
011A	AF07-125B	672.32	1320	4.1	163
011B	AF07-125B	672.321	787	5.6	141
012	AF07-125B	672.56	18	1.6	3.6
013	AF07-125B	672.7	23	3.8	3.7
014	AF07-125B	673	4	0.9	0.6
015	AF07-125B	673.2	8	0.9	1.2
016	AF07-125B	673.46	38	4.7	8.3
017	AF07-125B	673.51	646	3.4	81.8
018	AF07-125B	673.6	549	5.2	180
019A	AF07-125B	673.74	1930	18.2	355
019B	AF07-125B	673.741	1200	8.3	330
020	AF07-125B	673.89	2690	6.7	252
022	AF07-125B	674.05	1540	7.9	294
023	AF07-125B	674.12	7880	15.9	662
024	AF07-125B	674.22	892	5.5	256
025	AF01-36	343.74	174	2.9	16.1
026	AF01-36	344.12	15	71.8	238
027A	AF01-36	342.41	43	116	990
027B	AF01-36	342.411	38	141	2500
028	AF01-36	345.12	12	110	443
029	AF01-36	345.65	7	4.7	86.9
030	AF01-36	349.5	255	76.2	1810
031A	AF01-36	349.1	60	5.9	136

Appendix B. Table B.1. (Continued)

Sample	Hole ID	Downhole Depth (m)	Au (ppb)	Pt (ppb)	Pd (ppb)
031B	AF01-36	349.101	74	31.9	747
032A	EA12-005	448	1080	99.2	1570
032B	EA12-005	448.001	329	58.6	471
032C	EA12-005	448.002	104	8.9	100
033	EA12-005	448.53	18000	106	3300
034	EA12-005	448.94	5230	35.4	21000
035A	EA12-005	449.25	3460	15.9	1010
035B	EA12-005	449.251	3090	17.6	411
036A	EA12-005	447.75	141	21.4	321
036B	EA12-005	447.751	175	29.3	269
036C	EA12-005	447.752	341	30.4	1770
038	AF02-75	734.08	9	3.7	24.6
040	AF02-75	734.42	133	4.6	32
041A	AF02-75	734.9	18	7.2	89.1
041B	AF02-75	734.901	29	6.4	115
042A	AF02-75	735.85	7	2.1	10.1
042B	AF02-75	735.851	24	3.4	15.3
042A	AF01-37	426	11	2.3	8.6
042B	AF01-37	426.001	11	2.7	4.5
043A	AF01-37	427	6	15.8	26.1
043B	AF01-37	427.001	7	3.5	9.7
044	AF01-37	427.51	91	1.3	7.9
045A	AF01-37	428.45	1200	34.8	664
045B	AF01-37	428.451	28	40.1	405
046	AF01-37	429.29	20	31.7	253
047	AF03-81	574	81	3.7	38.6
048	AF03-81	574.14	307	121	1480
049	AF03-81	574.55	133	12.2	77.3
051	AF03-81	575.14	10	8.9	52.2
052A	AF03-81	575.48	9460	37800	33600
052B	AF03-81	575.481	11200	5100	10200
053	AF03-81	575.66	274	10	422
054	AF03-81	575.9	1270	24.3	1510
055	UA06-79	574.09	570	59.1	724
056	UA06-79	574.64	5380	196	21900
057A	UA06-79	575.12	6310	17.4	974
057B	UA06-79	575.121	2920	9.8	520

Appendix B. Table B.1. (Continued)

Sample	Hole ID	Downhole Depth (m)	Au (ppb)	Pt (ppb)	Pd (ppb)
058	UA06-79	575.4	1680	10.7	489
059	UA06-79	575.61	43	131	841
060	EA13-046	730.7	38	0.6	9.9
061	EA13-046	720.17	45	4.3	8.5
062	EA13-046	736	9	4.1	6.2
063A	EA13-046	738.32	180	9	32.2
063B	EA13-046	738.321	47	5.3	24.7
064	EA13-046	743.27	3360	39.9	429
065	EA13-046	759.25	1090	61.3	434
066	EA13-046	761.61	1500	36.2	126
067	EA13-046	762.17	1330	20	186
068	EA13-046	763.2	47	4.3	47.8
069	EA13-046	782.18	12	4.6	11.2
070	EA13-046	785.19	26	11.2	68.9
071	EA13-046	791.08	57	3.4	15.4
072	EA13-046	796.54	110	50.4	343
073	EA13-046	797.35	37	3.7	16
074	EA13-046	797.97	322	2	10.2
076	EA13-046	807.63	41	0.5	6.8
077	EA13-046	808.2	13	0.6	8.4
078	EA13-041	203.25	7	3.3	11.5
079	EA13-041	205.33	4	41.5	5.3
080	EA13-041	211.67	6	6.4	16.9
081	EA13-041	213.95	3	2.1	12.4
082	EA13-041	215.3	5	12.9	11.1
083	EA13-041	217.1	27	6.8	4.9
084	EA13-041	221.23	4	6.6	4
085	EA13-041	224	6	1.9	23.7
103	EA13-041	274.86	4350	10.9	191
104	EA13-041	276.49	58	2.3	18.9
105	EA13-041	277.2	9	1.6	19.1
106	EA13-041	277.6	29	65.2	444
107	EA13-041	277.82	93	185	4520
108	EA13-041	278.27	23	127	209
109	EA13-041	278.67	43	162	780
110	EA13-041	279.33	24	47.6	227
111	EA13-041	279.77	85	161	3920

Appendix B. Table B.1. (Continued)

Sample	Hole ID	Downhole Depth (m)	Au (ppb)	Pt (ppb)	Pd (ppb)
112	EA13-041	280.13	29	9.7	139
113	EA13-041	280.76	5	4.2	20.7
114	EA13-041	281.06	12	8.6	64.2
115	EA13-041	281.66	42	43.8	804
116	EA13-041	291.42	18	228	170
117A	AF01-35	319.3	443	11.1	75.3
117B	AF01-35	319.301	4650	11.5	259
118A	AF01-35	319.8	52	119	248
118B	AF01-35	319.801	38	114	464
119	AF01-35	320.22	29	190	560
120	AF01-35	321.33	31	109	372
121	AF01-35	321.86	11	96.9	254
122	AF01-35	322.5	23	120	2640
123	AF01-35	324.2	14	200	313
124	AF01-35	324.49	19	25.6	159
125	AF01-35	325.13	18	34.9	621
126	AF01-35	325.56	6	5.4	46.9
127	AF01-35	326.43	5	4.1	35.2
128	AF01-35	327.2	21	65.1	182
129	AF01-35	329	37	66.2	1150
130	AF01-35	330	11	31.7	26.8
131	AF01-35	330.25	289	6.5	63.1
132	AF01-35	333.55	1140	18.4	637
133	AF01-35	334.5	137	165	4770
134	AF01-35	334.63	85	17	26.5
135	AF01-35	336	216	2.3	8.3
136	AF01-35	336.82	100	189	851
137	AF01-35	339.5	13	4.6	4.5
138	EA17-196	482.67	1310	14.9	30.1
139	EA17-196	483.16	2270	5.3	237
140	EA17-196	482.95	1310	2.8	7.5
141	EA17-196	484	166	26	50.4
142	EA17-196	484.6	371	0.6	29.8
143	EA17-196	485.31	18300	24.4	1830
144	EA17-196	485.7	9780	24.2	1470
145	EA17-196	486.32	5070	12.1	359
146	EA17-196	486.84	202	3.4	47

Appendix B. Table B.1. (Continued)

Sample	Hole ID	Downhole Depth (m)	Au (ppb)	Pt (ppb)	Pd (ppb)
147A	EA17-196	487.03	92	3.9	23.9
147B	EA17-196	487.031	196	2.8	22.8
148A	EA17-196	487.58	666	5.6	88
148B	EA17-196	487.581	2010	5.4	103
149	EA17-196	487.71	99	6.5	28.4
150	EA17-196	487.711	168	31.8	21.7
151	EA17-196	489.94	1450	54.9	46.5
152A	EA17-196	490.58	158	22.2	36
152B	EA17-196	490.581	93	6.4	26.4
153	EA17-196	490.6	1650	26.6	25.7
154	EA17-196	492.21	116	10.1	27.5
155	EA17-196	493.5	18	1.6	8.1
156	PO16-34	266.45	242	60.9	4680
157	PO16-34	267.6	21	26.7	1340
158	PO16-34	267.93	15	7.7	234
159	PO16-34	314.2	18	1.1	12.5
160	PO16-34	314.82	819	323	5570
161A	PO16-34	315.5	94	9.3	620
161B	PO16-34	315.501	14	12.2	343
162	PO16-34	315.3	14	4.5	216
163	EA14-141	499.92	53	23.8	203
164	EA14-141	498.58	142	10.2	180
165	EA14-141	501.33	357	27.7	1630
166	EA14-141	501.71	3330	61.7	2550
167A	EA14-141	502.31	54	6.6	75.3
167B	EA14-141	502.311	94	39.7	302
167C	EA14-141	502.312	154	19.4	332
168	EA14-141	503	504	78.4	3440
169	EA14-141	503.69	1500	12.7	446
170	EA14-141	504.24	1360	11.3	364
172	EA14-141	504.94	41	1.6	11.7
173	EA14-141	506.27	479	21.2	429
178	EA14-141	510.57	129	1.8	39.7
179	EA14-141	511.29	151	19.1	248
186	EA14-149	371.67	738	2	12.5
188	EA14-149	373.63	7290	3	27.3
189	EA14-149	377.34	2400	55.7	434

Appendix B. Table B.1. (Continued)

Sample	Hole ID	Downhole Depth (m)	Au (ppb)	Pt (ppb)	Pd (ppb)
190	EA14-149	376.9	7790	2.3	346
191A	EA14-149	378.81	520	9.1	107
191B	EA14-149	378.811	15	2.6	29.5
192A	EA14-149	380.3	292	4.5	267
192B	EA14-149	380.301	152	12.5	37.6
193	EA14-149	381.63	482	2.8	14.5
194	EA14-149	383	328	1.8	30.2
195	EA14-149	398.14	16	2	4.1
196	EA14-149	401.05	4010	14.6	307
197	EA14-149	406.39	2730	41.5	816
198	EA14-149	406.84	1560	9.8	495
199	EA14-149	407.4	24	3.1	7.6
IOA-203		surface	154	44.4	309
QFP-204		surface	4	0.2	0.5
QFP-207		surface	6	0.2	1
208	UA12206	157	18600	19.2	1680
209	UA12-206	157.62	4500	10.6	450
210	UA12-206	163.4	1560	10.5	267
211	UA12-206	168.54	7310	10.6	2630
212	UA12-206	168.8	10300	14.7	620
213	UA12-206	169.33	665	6.2	228
214	UA05-44	344.4	34	7	37.8
215	UA05-44	345	15	3	9.3
216	UA05-44	346.2	15	3.8	45.9
217	UA05-44	348.6	24	0.4	7.4
218	UA05-44	349.17	68	18.6	81.2
219	EA12-010	138	89	5.8	8.8
220	EA12-010	138.25	9	5.8	7.8
221	EA12-010	139.72	5	5.4	7.1
222	EA12-010	139.88	8	6.3	11
223	EA12-183	578.21	1200	10.4	393
224	EA12-183	578.53	2940	13.7	1270
225	EA17-183	578.74	11300	30.4	4500
226A	EA17-183	578.98	287	2.1	320
226B	EA17-183	578.981	2440	10.3	681
227	EA17-183	579.12	96	6.1	271
228	EA17-183	579.121	2850	38.9	3170

Appendix C Sulfide trace element data

Table C.1. Chalcopyrite LA-ICPMS data.

Phase	Sample	Analysis No.	Mn ⁵⁵	Ni ⁶¹	Cu ⁶³	As ⁷⁵	Se ⁷⁸	Pd ¹⁰⁶	Ag ¹⁰⁷	Sb ¹²¹	Te ¹²⁶	Pt ¹⁹⁵	Au ¹⁹⁷	Pb ²⁰⁸	Bi ²⁰⁹
Ccp	AF17611-003A	176	9.00	170.00	783000	2.76	796.00	0.20	14.30	1.70	0.22	0.02	0.00	49.00	4.20
Ccp	AF17611-003A	174	0.07	0.85	742000	0.97	773.00	0.06	27.10	0.63	0.24	0.01	0.01	70.20	2.56
Ccp	AF17611-003A	173	0.33	0.60	753000	0.49	791.00	0.04	12.53	0.33	0.17	0.00	0.00	87.40	1.43
Ccp	AF17611-003A	175	0.78	1.63	767000	0.95	778.00	0.04	13.30	0.48	0.13	0.01	0.01	72.90	1.51
Ccp	AF17611-003A	171	0.39	4.95	766000	2.70	633.00	0.03	14.60	0.32	0.68	0.02	0.01	23.00	1.91
Ccp	AF17611-003A	170	0.98		781000	11.20	561.00	0.01	20.60	0.75	0.23	0.01	0.00	8.40	2.70
Ccp	AF17611-003A	155	0.13	1.06	773000	0.91	748.00	0.01	18.10	0.42	0.22	0.01	0.01	56.00	3.67
Ccp	AF17611-003A	169	1.49	1.06	756000	3.40	452.00	0.01	17.42	1.04	0.12	0.01	0.01	20.90	4.19
Ccp	AF17611-003A	172	0.12	1.56	733000	1.86	563.00	0.01	14.20	0.43	0.21	0.01	0.01	6.80	1.30
Ccp	AF17611-003A	157	0.73	0.70	742000	0.70	775.00	0.01	9.16	0.45	0.08	0.00	0.00	66.80	0.56
Ccp	AF17611-003A	165	0.10	0.78	740000	0.39	760.00	0.01	5.12	0.25	0.16	0.08	0.01	61.00	0.94
Ccp	AF17611-003A	164	0.05	0.50	726000	1.19	554.00	0.01	7.78	0.62	0.07	0.01	0.00	8.60	1.05
Ccp	AF17611-003A	163	0.09	0.45	727000	0.76	755.00	0.00	7.41	0.42	0.06	0.00	0.00	55.30	0.85
Ccp	AF17611-003A	168	0.31	0.42	735000	0.94	775.00	0.00	11.11	0.37	0.09	0.00	0.00	85.40	2.72
Ccp	AF17611-003B	185b	3.30	42.00	756000	3.46	784.00	0.34	22.80	1.34	0.09	0.03	0.01	59.00	2.08
Ccp	AF17611-003B	199b	0.73	14.80	744000	1.52	762.00	0.09	18.50	0.70	0.12	0.01	0.00	97.00	3.68
Ccp	AF3601-030	217	4.10	2.33	762000	2.50	315.00	0.78	2.45	0.96	0.23	0.01	0.12	7.20	2.38
Ccp	AF3601-030	220	2.50	1.91	778000	2.17	241.00	0.12	5.56	1.10	0.19	0.01	0.04	37.40	5.22
Ccp	AF3601-030	222	10.60	2.05	743000	6.60	726.00	0.05	4.40	1.95	0.27	0.01	0.12	26.70	5.56
Ccp	AF3601-030	227	0.17	2.47	793000	4.86	156.00	0.04	6.50	1.51	0.28	0.01	0.03	18.50	3.40
Ccp	AF3601-030	228	9.30	2.40	782000	6.40	102.00	0.03	5.75	1.78	0.28	0.01	0.10	15.00	6.55
Ccp	AF3601-030	216	7.60	0.42	744000	1.58	380.20	0.02	3.57	0.65	0.08	0.01	0.03	13.10	4.55
Ccp	EA1205-032A	277	2.62	380.00	13.1	9.50	305.60	0.36	1.50	0.02	0.76	0.01	0.01	0.88	0.50

Appendix C. Table C.1. (Continued)

Phase	Sample	Analysis No.	Mn ⁵⁵	Ni ⁶¹	Cu ⁶³	As ⁷⁵	Se ⁷⁸	Pd ¹⁰⁶	Ag ¹⁰⁷	Sb ¹²¹	Te ¹²⁶	Pt ¹⁹⁵	Au ¹⁹⁷	Pb ²⁰⁸	Bi ²⁰⁹
Ccp	EA1205-032A	289	3.00	0.99	730000	4.54	313.70	0.06	20.60	2.45	0.22	0.01	0.00	26.00	9.14
Ccp	EA1205-032A	295	39.00	1.41	738000	1.93	333.00	0.04	11.79	0.89	0.23	0.01	0.01	17.10	4.51
Ccp	EA1205-032A	284	0.55	1.06	732000	2.88	236.00	0.04	13.98	1.54	0.22	0.01	0.02	10.60	9.05
Ccp	EA1205-032A	281	3.40	1.84	731000	2.86	187.40	0.03	16.30	2.00	0.11	0.01	0.02	8.53	6.40
Ccp	EA1205-032A	292	2.30	2.47	746000	4.33	256.00	0.03	19.60	1.82	0.34	0.01	0.01	10.00	9.30
Ccp	EA1205-032A	283	0.18	0.85	732000	0.18	234.00	0.02	4.42	0.08	0.13	0.01	0.01	2.28	1.29
Ccp	EA1205-032A	290	0.27	0.71	732000	1.23	74.70	0.02	6.70	0.38	0.35	0.00	0.00	3.50	3.08
Ccp	EA1205-032A	282	0.38	1.06	736000	0.52	209.30	0.01	5.85	0.27	0.11	0.00	0.00	2.15	1.64
Ccp	EA1205-032A	291	0.35	0.40	729000	1.34	94.90	0.01	21.20	0.36	0.21	0.00	0.00	3.78	2.96
Ccp	EA1205-034	243	19.90	6.40	735000	4.44	61.10	1.98	15.44	1.25	3.95	0.00	0.00	36.50	11.63
Ccp	EA1205-034	244	2.70	3.10	747000	0.73	39.60	1.43	14.58	0.72	5.09	0.00	0.02	10.90	7.93
Ccp	EA1205-034	237b	9.40	21.00	737000	3.78	60.70	0.59	27.50	1.17	1.45	0.00	0.00	69.00	32.70
Ccp	EA1205-034	254	3.53	0.85	743000	0.87	68.50	0.12	10.43	0.66	0.14	0.00	0.00	89.00	2.87
Ccp	EA1205-034	236	4.20	0.69	758000	2.19	41.00	0.10	13.10	0.62	0.38	0.01	0.00	56.20	13.20
Ccp	EA1205-034	235b	6.80	16.20	775000	1.47	73.00	0.09	10.70	0.84	0.25	0.02	0.01	63.00	8.61
Ccp	EA1205-034	238	9.90	1.84	770000	0.94	41.00	0.04	11.64	0.60	0.30	0.01	0.00	39.40	15.20
Ccp	EA1205-034	261	1.90	1.20	742000	1.71	69.30	0.02	11.16	1.02	0.11	0.00	0.03	16.80	4.15
Ccp	EA1205-034	260	0.48	1.13	742000	3.04	58.50	0.02	14.92	1.59	0.16	0.00	0.00	31.50	5.80
Ccp	EA1205-034	250	1.80	0.92	736000	1.26	70.10	0.02	9.65	1.00	0.11	0.00	0.00	16.30	1.63
Ccp	EA1205-034	251	1.60	0.68	732000	1.35	70.60	0.02	10.20	0.93	0.21	0.00	0.00	25.60	1.74
Ccp	EA1205-034	252	1.48	0.85	738000	2.59	67.80	0.02	12.50	1.48	0.11	0.00	0.00	9.96	2.15
Ccp	EA1205-034	258	0.15	2.19	755000	0.56	122.00	0.01	6.80	0.18	0.33	0.01	0.00	6.10	0.39
Ccp	EA1205-034	256	0.06	0.62	734000	0.25	118.70	0.01	9.96	0.29	0.12	0.01	0.00	3.30	0.42
Ccp	EA1205-034	255	0.07	0.47	737000	0.62	84.20	0.01	8.16	0.71	0.09		0.00	1.91	0.22
Ccp	EA1205-034	263	0.11	1.27	743000	0.34	99.30	0.00	6.64	0.39	0.11	0.00	0.00	2.24	1.01
Ccp	AF8103-048	305	320.00	9.00	734000	1.92	142.00	0.05	9.60	0.66	0.22	0.01	0.00	39.00	1.40

Appendix C. Table C.1. (Continued)

Phase	Sample	Analysis No.	Mn ⁵⁵	Ni ⁶¹	Cu ⁶³	As ⁷⁵	Se ⁷⁸	Pd ¹⁰⁶	Ag ¹⁰⁷	Sb ¹²¹	Te ¹²⁶	Pt ¹⁹⁵	Au ¹⁹⁷	Pb ²⁰⁸	Bi ²⁰⁹
Ccp	EA1205-032A	289	3.00	0.99	730000	4.54	313.70	0.06	20.60	2.45	0.22	0.01	0.00	26.00	9.14
Ccp	EA1205-032A	295	39.00	1.41	738000	1.93	333.00	0.04	11.79	0.89	0.23	0.01	0.01	17.10	4.51
Ccp	EA1205-032A	284	0.55	1.06	732000	2.88	236.00	0.04	13.98	1.54	0.22	0.01	0.02	10.60	9.05
Ccp	EA1205-032A	281	3.40	1.84	731000	2.86	187.40	0.03	16.30	2.00	0.11	0.01	0.02	8.53	6.40
Ccp	EA1205-032A	292	2.30	2.47	746000	4.33	256.00	0.03	19.60	1.82	0.34	0.01	0.01	10.00	9.30
Ccp	EA1205-032A	283	0.18	0.85	732000	0.18	234.00	0.02	4.42	0.08	0.13	0.01	0.01	2.28	1.29
Ccp	EA1205-032A	290	0.27	0.71	732000	1.23	74.70	0.02	6.70	0.38	0.35	0.00	0.00	3.50	3.08
Ccp	EA1205-032A	282	0.38	1.06	736000	0.52	209.30	0.01	5.85	0.27	0.11	0.00	0.00	2.15	1.64
Ccp	EA1205-032A	291	0.35	0.40	729000	1.34	94.90	0.01	21.20	0.36	0.21	0.00	0.00	3.78	2.96
Ccp	EA1205-034	243	19.90	6.40	735000	4.44	61.10	1.98	15.44	1.25	3.95	0.00	0.00	36.50	11.63
Ccp	EA1205-034	244	2.70	3.10	747000	0.73	39.60	1.43	14.58	0.72	5.09	0.00	0.02	10.90	7.93
Ccp	EA1205-034	237b	9.40	21.00	737000	3.78	60.70	0.59	27.50	1.17	1.45	0.00	0.00	69.00	32.70
Ccp	EA1205-034	254	3.53	0.85	743000	0.87	68.50	0.12	10.43	0.66	0.14	0.00	0.00	89.00	2.87
Ccp	EA1205-034	236	4.20	0.69	758000	2.19	41.00	0.10	13.10	0.62	0.38	0.01	0.00	56.20	13.20
Ccp	EA1205-034	235b	6.80	16.20	775000	1.47	73.00	0.09	10.70	0.84	0.25	0.02	0.01	63.00	8.61
Ccp	AF8103-048	299	0.27	1.84	737000	0.45	575.00	0.04	6.82	0.09	0.10	0.01	0.00	491.00	0.03
Ccp	AF8103-048	302	0.09	0.92	745000	0.09	517.00	0.03	8.07	0.07	0.06	0.00		210.00	0.03
Ccp	AF8103-048	298	1.40	1.13	733000	0.73	508.00	0.03	9.08	0.36	0.13	0.01	0.01	372.00	0.28
Ccp	AF8103-048	300	0.09	1.70	746000	0.33	554.00	0.03	9.54	0.27	0.09	0.00	0.00	434.00	0.06
Ccp	AF8103-048	297	0.31	3.39	798000	1.08	262.00	0.02	12.20	0.50	0.27	0.01	0.01	17.90	1.80
Ccp	AF8103-048	296	41.00	3.89	765000	1.67	97.00	0.02	4.90	0.87	0.23	0.01	0.00	31.00	0.25
Ccp	AF8103-048	304	7.30	1.84	755000	0.57	250.00	0.01	2.82	0.22	0.12	0.01	0.00	9.80	0.07
Ccp	AF8103-048	301	0.24	1.27	738000	0.54	532.00	0.01	7.00	0.12	0.08	0.00	0.00	262.00	0.04
Ccp	AF8103-052A	306	9.90	0.71	744000	3.42	305.70	0.09	8.10	0.93	0.06	0.00	0.01	48.00	3.38
Ccp	AF8103-052A	311	11.40	0.85	747000	6.40	340.00	0.04	10.12	2.12	0.07	0.01	0.02	25.80	3.21
Ccp	AF8103-052A	310	1.40	0.85	755000	1.07	351.90	0.03	4.95	0.37	0.07	0.00	0.04	6.38	1.61

Appendix C. Table C.1. (Continued)

Phase	Sample	Analysis No.	Mn ⁵⁵	Ni ⁶¹	Cu ⁶³	As ⁷⁵	Se ⁷⁸	Pd ¹⁰⁶	Ag ¹⁰⁷	Sb ¹²¹	Te ¹²⁶	Pt ¹⁹⁵	Au ¹⁹⁷	Pb ²⁰⁸	Bi ²⁰⁹
Ccp	EA1205-032A	289	3.00	0.99	730000	4.54	313.70	0.06	20.60	2.45	0.22	0.01	0.00	26.00	9.14
Ccp	EA1205-032A	295	39.00	1.41	738000	1.93	333.00	0.04	11.79	0.89	0.23	0.01	0.01	17.10	4.51
Ccp	EA1205-032A	284	0.55	1.06	732000	2.88	236.00	0.04	13.98	1.54	0.22	0.01	0.02	10.60	9.05
Ccp	EA1205-032A	281	3.40	1.84	731000	2.86	187.40	0.03	16.30	2.00	0.11	0.01	0.02	8.53	6.40
Ccp	EA1205-032A	292	2.30	2.47	746000	4.33	256.00	0.03	19.60	1.82	0.34	0.01	0.01	10.00	9.30
Ccp	EA1205-032A	283	0.18	0.85	732000	0.18	234.00	0.02	4.42	0.08	0.13	0.01	0.01	2.28	1.29
Ccp	EA1205-032A	290	0.27	0.71	732000	1.23	74.70	0.02	6.70	0.38	0.35	0.00	0.00	3.50	3.08
Ccp	EA1205-032A	282	0.38	1.06	736000	0.52	209.30	0.01	5.85	0.27	0.11	0.00	0.00	2.15	1.64
Ccp	EA1205-032A	291	0.35	0.40	729000	1.34	94.90	0.01	21.20	0.36	0.21	0.00	0.00	3.78	2.96
Ccp	EA1205-034	243	19.90	6.40	735000	4.44	61.10	1.98	15.44	1.25	3.95	0.00	0.00	36.50	11.63
Ccp	EA1205-034	244	2.70	3.10	747000	0.73	39.60	1.43	14.58	0.72	5.09	0.00	0.02	10.90	7.93
Ccp	EA1205-034	237b	9.40	21.00	737000	3.78	60.70	0.59	27.50	1.17	1.45	0.00	0.00	69.00	32.70
Ccp	EA1205-034	254	3.53	0.85	743000	0.87	68.50	0.12	10.43	0.66	0.14	0.00	0.00	89.00	2.87
Ccp	EA1205-034	236	4.20	0.69	758000	2.19	41.00	0.10	13.10	0.62	0.38	0.01	0.00	56.20	13.20
Ccp	EA1205-034	235b	6.80	16.20	775000	1.47	73.00	0.09	10.70	0.84	0.25	0.02	0.01	63.00	8.61
Ccp	AF8103-052A	308	0.07	1.20	757000	3.20	400.00	0.03	5.07	1.22	0.07	0.00	0.03	8.49	1.71
Ccp	AF8103-052A	313	0.09	0.99	753000	1.59	529.00	0.03	8.24	0.59	0.14	0.00	0.01	42.80	2.21
Ccp	AF8103-052A	312	0.21	2.00	748000	0.85	551.00	0.02	4.94	0.25	0.17	0.00	0.01	16.37	1.36
Ccp	AF8103-052A	307	0.23	1.13	751000	0.70	298.60	0.02	3.53	0.18	0.06	0.00	0.01	10.40	1.49
Ccp	AF8103-052A	322	0.06	0.99	752000	3.91	477.00	0.02	12.30	1.23	0.16	0.00	0.01	61.00	4.25
Ccp	AF8103-052A	309	1.40	1.48	755000	0.54	160.00	0.02	1.62	0.09	0.06	0.01	0.01	3.22	0.62
Ccp	AF8103-052A	318	0.92	0.71	753000	4.11	475.00	0.02	11.01	1.87	0.14	0.00	0.01	45.90	4.40
Ccp	AF8103-052A	319	0.08	1.27	755000	2.89	505.00	0.01	8.96	0.95	0.16	0.00	0.02	28.10	3.30
Ccp	AF8103-052A	320	0.09	0.92	757000	1.86	552.00	0.01	8.12	0.78	0.13	0.00	0.01	74.00	2.41
Ccp	AF8103-052B	337	0.09	0.85	751000	0.71	507.00	0.05	6.23	0.17	0.40	0.00	0.01	28.20	1.29
Ccp	AF8103-052B	331	0.06	0.99	750000	0.44	514.70	0.05	4.34	0.17	0.25	0.00	0.00	15.70	1.12

Appendix C. Table C.1. (Continued)

Phase	Sample	Analysis No.	Mn ⁵⁵	Ni ⁶¹	Cu ⁶³	As ⁷⁵	Se ⁷⁸	Pd ¹⁰⁶	Ag ¹⁰⁷	Sb ¹²¹	Te ¹²⁶	Pt ¹⁹⁵	Au ¹⁹⁷	Pb ²⁰⁸	Bi ²⁰⁹
Ccp	EA1205-032A	289	3.00	0.99	730000	4.54	313.70	0.06	20.60	2.45	0.22	0.01	0.00	26.00	9.14
Ccp	EA1205-032A	295	39.00	1.41	738000	1.93	333.00	0.04	11.79	0.89	0.23	0.01	0.01	17.10	4.51
Ccp	EA1205-032A	284	0.55	1.06	732000	2.88	236.00	0.04	13.98	1.54	0.22	0.01	0.02	10.60	9.05
Ccp	EA1205-032A	281	3.40	1.84	731000	2.86	187.40	0.03	16.30	2.00	0.11	0.01	0.02	8.53	6.40
Ccp	EA1205-032A	292	2.30	2.47	746000	4.33	256.00	0.03	19.60	1.82	0.34	0.01	0.01	10.00	9.30
Ccp	EA1205-032A	283	0.18	0.85	732000	0.18	234.00	0.02	4.42	0.08	0.13	0.01	0.01	2.28	1.29
Ccp	EA1205-032A	290	0.27	0.71	732000	1.23	74.70	0.02	6.70	0.38	0.35	0.00	0.00	3.50	3.08
Ccp	EA1205-032A	282	0.38	1.06	736000	0.52	209.30	0.01	5.85	0.27	0.11	0.00	0.00	2.15	1.64
Ccp	EA1205-032A	291	0.35	0.40	729000	1.34	94.90	0.01	21.20	0.36	0.21	0.00	0.00	3.78	2.96
Ccp	EA1205-034	243	19.90	6.40	735000	4.44	61.10	1.98	15.44	1.25	3.95	0.00	0.00	36.50	11.63
Ccp	EA1205-034	244	2.70	3.10	747000	0.73	39.60	1.43	14.58	0.72	5.09	0.00	0.02	10.90	7.93
Ccp	EA1205-034	237b	9.40	21.00	737000	3.78	60.70	0.59	27.50	1.17	1.45	0.00	0.00	69.00	32.70
Ccp	EA1205-034	254	3.53	0.85	743000	0.87	68.50	0.12	10.43	0.66	0.14	0.00	0.00	89.00	2.87
Ccp	EA1205-034	236	4.20	0.69	758000	2.19	41.00	0.10	13.10	0.62	0.38	0.01	0.00	56.20	13.20
Ccp	EA1205-034	235b	6.80	16.20	775000	1.47	73.00	0.09	10.70	0.84	0.25	0.02	0.01	63.00	8.61
Ccp	AF8103-052B	339	0.06	0.68	757000	0.62	519.70	0.04	5.55	0.10	0.20	0.00	0.00	19.10	1.10
Ccp	AF8103-052B	340	0.06	1.34	770000	1.12	520.00	0.04	7.66	0.37	0.27	0.01	0.00	39.40	1.57
Ccp	AF8103-052B	226	0.22	1.20	752000	0.45	500.00	0.03	5.77	0.18	0.24	0.00	0.00	27.90	1.59
Ccp	AF8103-052B	338	0.12	0.71	754000	0.34	512.00	0.03	4.12	0.13	0.11	0.00	0.00	49.60	1.40
Ccp	AF8103-052B	331b	0.38	1.20	751000	0.94	514.00	0.02	7.10	0.29	0.14	0.00	0.01	160.00	1.54
Ccp	AF8103-052B	332	0.30	0.99	759000	0.77	541.00	0.02	4.45	0.14	0.13	0.01	0.00	24.50	0.70
Ccp	AF8103-052B	334	0.14	1.20	758000	1.12	509.00	0.02	8.05	0.27	0.12	0.00	0.00	30.80	0.88
Ccp	AF8103-052B	341	0.07	0.99	758000	1.29	486.80	0.01	7.26	0.39	0.12	0.00	0.01	15.50	1.02
Ccp	AF8103-052B	333	0.36	1.20	758000	10.30	487.00	0.01	13.00	3.51	0.16	0.00	0.00	21.40	1.71
Ccp	AF8103-052B	328	0.57	1.06	753000	14.40	382.60	0.01	5.45	9.60	0.15	0.01	0.01	16.90	1.15
Ccp	AF8103-052B	327	0.06	0.92	754000	7.70	115.00	0.01	4.43	5.40	0.13	0.00	0.01	12.10	1.09

Appendix C. Table C.1. (Continued)

Phase	Sample	Analysis No.	Mn ⁵⁵	Ni ⁶¹	Cu ⁶³	As ⁷⁵	Se ⁷⁸	Pd ¹⁰⁶	Ag ¹⁰⁷	Sb ¹²¹	Te ¹²⁶	Pt ¹⁹⁵	Au ¹⁹⁷	Pb ²⁰⁸	Bi ²⁰⁹
Ccp	EA1205-032A	289	3.00	0.99	730000	4.54	313.70	0.06	20.60	2.45	0.22	0.01	0.00	26.00	9.14
Ccp	EA1205-032A	295	39.00	1.41	738000	1.93	333.00	0.04	11.79	0.89	0.23	0.01	0.01	17.10	4.51
Ccp	EA1205-032A	284	0.55	1.06	732000	2.88	236.00	0.04	13.98	1.54	0.22	0.01	0.02	10.60	9.05
Ccp	EA1205-032A	281	3.40	1.84	731000	2.86	187.40	0.03	16.30	2.00	0.11	0.01	0.02	8.53	6.40
Ccp	EA1205-032A	292	2.30	2.47	746000	4.33	256.00	0.03	19.60	1.82	0.34	0.01	0.01	10.00	9.30
Ccp	EA1205-032A	283	0.18	0.85	732000	0.18	234.00	0.02	4.42	0.08	0.13	0.01	0.01	2.28	1.29
Ccp	EA1205-032A	290	0.27	0.71	732000	1.23	74.70	0.02	6.70	0.38	0.35	0.00	0.00	3.50	3.08
Ccp	EA1205-032A	282	0.38	1.06	736000	0.52	209.30	0.01	5.85	0.27	0.11	0.00	0.00	2.15	1.64
Ccp	EA1205-032A	291	0.35	0.40	729000	1.34	94.90	0.01	21.20	0.36	0.21	0.00	0.00	3.78	2.96
Ccp	EA1205-034	243	19.90	6.40	735000	4.44	61.10	1.98	15.44	1.25	3.95	0.00	0.00	36.50	11.63
Ccp	EA1205-034	244	2.70	3.10	747000	0.73	39.60	1.43	14.58	0.72	5.09	0.00	0.02	10.90	7.93
Ccp	EA1205-034	237b	9.40	21.00	737000	3.78	60.70	0.59	27.50	1.17	1.45	0.00	0.00	69.00	32.70
Ccp	EA1205-034	254	3.53	0.85	743000	0.87	68.50	0.12	10.43	0.66	0.14	0.00	0.00	89.00	2.87
Ccp	EA1205-034	236	4.20	0.69	758000	2.19	41.00	0.10	13.10	0.62	0.38	0.01	0.00	56.20	13.20
Ccp	EA1205-034	235b	6.80	16.20	775000	1.47	73.00	0.09	10.70	0.84	0.25	0.02	0.01	63.00	8.61
Ccp	AF8103-052B	342	1.16	0.57	759000	1.69	338.00	0.01	5.46	0.54	0.12	0.00	0.00	33.00	0.39
Ccp	UA7906-056	350	5.10	0.99	736000	0.37	42.70	0.09	7.70	0.41	0.11	0.00	0.02	17.90	4.46
Ccp	UA7906-056	367	4.80	3.68	842000	0.71	59.00	0.07	4.13	0.18	0.38	0.02	0.04	4.20	3.04
Ccp	UA7906-056	365	52.00	9.19	988000	40.80	59.40	0.06	15.30	9.30	0.99	0.03	0.05	31.00	7.30
Ccp	UA7906-056	347	10.90	1.13	723000	14.10	39.20	0.02	22.20	6.10	0.15	0.01	0.16	61.00	10.20
Ccp	UA7906-056	364	0.48	2.05	791000	0.80	63.00	0.02	7.96	0.51	0.21		0.03	11.40	4.94
Ccp	UA7906-056	363	0.07	1.59	757000	0.16	20.20	0.01	3.47	0.15	0.12	0.00	0.00	3.10	1.57
Ccp	UA7906-056	366	10.60	5.66	765000	0.43	111.00	0.01	7.17	0.39	0.18	0.01	0.02	6.51	2.71
Ccp	UA7906-056	349	0.47	0.92	727000	0.50	44.80	0.01	16.20	1.39	0.11	0.01	0.03	24.80	11.02
Ccp	UA7906-057A	393	0.07	0.78	791900	3.03	15.56	0.05	12.16	1.57	0.10	0.00	0.11	22.40	7.17
Ccp	UA7906-057A	388	1.90	0.69	804000	6.00	14.14	0.04	11.26	2.69	0.22		0.06	23.70	2.70

Appendix C. Table C.1. (Continued)

Phase	Sample	Analysis No.	Mn ⁵⁵	Ni ⁶¹	Cu ⁶³	As ⁷⁵	Se ⁷⁸	Pd ¹⁰⁶	Ag ¹⁰⁷	Sb ¹²¹	Te ¹²⁶	Pt ¹⁹⁵	Au ¹⁹⁷	Pb ²⁰⁸	Bi ²⁰⁹
Ccp	EA1205-032A	289	3.00	0.99	730000	4.54	313.70	0.06	20.60	2.45	0.22	0.01	0.00	26.00	9.14
Ccp	EA1205-032A	295	39.00	1.41	738000	1.93	333.00	0.04	11.79	0.89	0.23	0.01	0.01	17.10	4.51
Ccp	EA1205-032A	284	0.55	1.06	732000	2.88	236.00	0.04	13.98	1.54	0.22	0.01	0.02	10.60	9.05
Ccp	EA1205-032A	281	3.40	1.84	731000	2.86	187.40	0.03	16.30	2.00	0.11	0.01	0.02	8.53	6.40
Ccp	EA1205-032A	292	2.30	2.47	746000	4.33	256.00	0.03	19.60	1.82	0.34	0.01	0.01	10.00	9.30
Ccp	EA1205-032A	283	0.18	0.85	732000	0.18	234.00	0.02	4.42	0.08	0.13	0.01	0.01	2.28	1.29
Ccp	EA1205-032A	290	0.27	0.71	732000	1.23	74.70	0.02	6.70	0.38	0.35	0.00	0.00	3.50	3.08
Ccp	EA1205-032A	282	0.38	1.06	736000	0.52	209.30	0.01	5.85	0.27	0.11	0.00	0.00	2.15	1.64
Ccp	EA1205-032A	291	0.35	0.40	729000	1.34	94.90	0.01	21.20	0.36	0.21	0.00	0.00	3.78	2.96
Ccp	EA1205-034	243	19.90	6.40	735000	4.44	61.10	1.98	15.44	1.25	3.95	0.00	0.00	36.50	11.63
Ccp	EA1205-034	244	2.70	3.10	747000	0.73	39.60	1.43	14.58	0.72	5.09	0.00	0.02	10.90	7.93
Ccp	EA1205-034	237b	9.40	21.00	737000	3.78	60.70	0.59	27.50	1.17	1.45	0.00	0.00	69.00	32.70
Ccp	EA1205-034	254	3.53	0.85	743000	0.87	68.50	0.12	10.43	0.66	0.14	0.00	0.00	89.00	2.87
Ccp	EA1205-034	236	4.20	0.69	758000	2.19	41.00	0.10	13.10	0.62	0.38	0.01	0.00	56.20	13.20
Ccp	EA1205-034	235b	6.80	16.20	775000	1.47	73.00	0.09	10.70	0.84	0.25	0.02	0.01	63.00	8.61
Ccp	UA7906-057A	391	1.50	2.80	808000	1.16	15.56	0.02	4.38	0.71	0.13	0.00	0.02	7.01	2.50
Ccp	UA7906-057A	389	0.95	1.13	792000	16.40	12.02	0.02	13.10	5.18	0.16	0.00	0.06	31.10	2.71
Ccp	UA7906-057A	392	0.07	0.92	805100	1.17	16.97	0.01	8.10	0.89	0.13	0.00	0.02	12.57	5.31
Ccp	UA7906-057A	390	0.26	0.70	791000	1.71	191.00	0.01	1.00	0.75	0.13	0.00	0.00	0.01	0.77
Ccp	UA7906-057A	394	0.12	0.58	795000	0.75	71.00	0.01	3.56	0.47	0.11		0.01	6.25	1.97
Ccp	UA7906-057A	395	0.06	0.78	795500	0.39	109.00	0.01	2.70	0.12	0.10	0.00	0.01	2.87	2.03
Ccp	EA17196-143	422	21.21	6.36	799000	1.10	167.00	0.50	7.00	0.32	0.47	0.04	0.01	78.00	0.33
Ccp	EA17196-143	420	16.26	6.79	832000	1.56	41.00	0.15	8.80	0.23	0.22	0.01	0.01	29.00	1.01
Ccp	EA17196-143	412	4.67	2.90	851000	3.50	125.00	0.13	17.60	0.27	0.30	0.01	0.00	32.00	1.87
Ccp	EA17196-143	424	1.18	2.40	827000	12.80	110.00	0.12	15.80	0.54	0.20		0.00	40.00	0.99
Ccp	EA17196-143	409	3.90	1.84	788000	6.70	143.00	0.11	16.70	0.33	0.23	0.01	0.00	86.00	1.28

Appendix C. Table C.1. (Continued)

Phase	Sample	Analysis No.	Mn ⁵⁵	Ni ⁶¹	Cu ⁶³	As ⁷⁵	Se ⁷⁸	Pd ¹⁰⁶	Ag ¹⁰⁷	Sb ¹²¹	Te ¹²⁶	Pt ¹⁹⁵	Au ¹⁹⁷	Pb ²⁰⁸	Bi ²⁰⁹
Ccp	EA1205-032A	289	3.00	0.99	730000	4.54	313.70	0.06	20.60	2.45	0.22	0.01	0.00	26.00	9.14
Ccp	EA1205-032A	295	39.00	1.41	738000	1.93	333.00	0.04	11.79	0.89	0.23	0.01	0.01	17.10	4.51
Ccp	EA1205-032A	284	0.55	1.06	732000	2.88	236.00	0.04	13.98	1.54	0.22	0.01	0.02	10.60	9.05
Ccp	EA1205-032A	281	3.40	1.84	731000	2.86	187.40	0.03	16.30	2.00	0.11	0.01	0.02	8.53	6.40
Ccp	EA1205-032A	292	2.30	2.47	746000	4.33	256.00	0.03	19.60	1.82	0.34	0.01	0.01	10.00	9.30
Ccp	EA1205-032A	283	0.18	0.85	732000	0.18	234.00	0.02	4.42	0.08	0.13	0.01	0.01	2.28	1.29
Ccp	EA1205-032A	290	0.27	0.71	732000	1.23	74.70	0.02	6.70	0.38	0.35	0.00	0.00	3.50	3.08
Ccp	EA1205-032A	282	0.38	1.06	736000	0.52	209.30	0.01	5.85	0.27	0.11	0.00	0.00	2.15	1.64
Ccp	EA1205-032A	291	0.35	0.40	729000	1.34	94.90	0.01	21.20	0.36	0.21	0.00	0.00	3.78	2.96
Ccp	EA1205-034	243	19.90	6.40	735000	4.44	61.10	1.98	15.44	1.25	3.95	0.00	0.00	36.50	11.63
Ccp	EA1205-034	244	2.70	3.10	747000	0.73	39.60	1.43	14.58	0.72	5.09	0.00	0.02	10.90	7.93
Ccp	EA1205-034	237b	9.40	21.00	737000	3.78	60.70	0.59	27.50	1.17	1.45	0.00	0.00	69.00	32.70
Ccp	EA1205-034	254	3.53	0.85	743000	0.87	68.50	0.12	10.43	0.66	0.14	0.00	0.00	89.00	2.87
Ccp	EA1205-034	236	4.20	0.69	758000	2.19	41.00	0.10	13.10	0.62	0.38	0.01	0.00	56.20	13.20
Ccp	EA1205-034	235b	6.80	16.20	775000	1.47	73.00	0.09	10.70	0.84	0.25	0.02	0.01	63.00	8.61
Ccp	EA17196-143	408	0.84	1.27	826000	23.00	137.00	0.11	41.70	0.43	0.18	0.00	0.01	68.00	2.83
Ccp	EA17196-143	413	10.61	3.39	811000	13.70	164.00	0.06	32.00	0.43	0.22		0.01	129.00	2.50
Ccp	EA17196-143	423	16.26	6.01	807000	11.80	132.00	0.04	26.20	0.21	0.21	0.01	0.01	35.10	1.36
Ccp	EA17196-143	415	2.40	1.41	807000	1.93	83.00	0.03	9.40	0.15	0.22	0.01	0.00	56.00	0.96
Ccp	EA17196-143	410	2.17	1.56	826000	2.30	151.00	0.03	8.48	0.07	0.21	0.01	0.01	7.40	0.42
Ccp	EA17196-143	414	0.78	0.66	809000	5.50	171.00	0.02	15.70	0.24	0.13		0.01	16.10	0.97
Ccp	EA17196-143	427	0.58	1.20	854000	114.00	45.00	0.02	29.40	0.84	0.22	0.01	0.01	27.00	0.71
Ccp	EA17196-143	425	0.19	112.30	807300	0.71	99.70	0.01	5.75	0.33	0.13	0.00	0.01	5.82	0.62
Ccp	EA17196-143	431	4.60	1073.00	803600	3.14	32.40	0.01	14.00	0.49	0.13	0.00	0.01	75.00	1.64
Ccp	EA17196-144	455	1.54	1.13	823000	13.60	99.50	0.08	54.80	0.37	0.18	0.00	0.01	62.80	1.73
Ccp	EA17196-144	156	3.48	1.13	807000	16.30	107.70	0.06	42.00	0.38	0.12	0.00	0.00	112.00	3.23

Appendix C. Table C.1. (Continued)

Phase	Sample	Analysis No.	Mn ⁵⁵	Ni ⁶¹	Cu ⁶³	As ⁷⁵	Se ⁷⁸	Pd ¹⁰⁶	Ag ¹⁰⁷	Sb ¹²¹	Te ¹²⁶	Pt ¹⁹⁵	Au ¹⁹⁷	Pb ²⁰⁸	Bi ²⁰⁹
Ccp	EA1205-032A	289	3.00	0.99	730000	4.54	313.70	0.06	20.60	2.45	0.22	0.01	0.00	26.00	9.14
Ccp	EA1205-032A	295	39.00	1.41	738000	1.93	333.00	0.04	11.79	0.89	0.23	0.01	0.01	17.10	4.51
Ccp	EA1205-032A	284	0.55	1.06	732000	2.88	236.00	0.04	13.98	1.54	0.22	0.01	0.02	10.60	9.05
Ccp	EA1205-032A	281	3.40	1.84	731000	2.86	187.40	0.03	16.30	2.00	0.11	0.01	0.02	8.53	6.40
Ccp	EA1205-032A	292	2.30	2.47	746000	4.33	256.00	0.03	19.60	1.82	0.34	0.01	0.01	10.00	9.30
Ccp	EA1205-032A	283	0.18	0.85	732000	0.18	234.00	0.02	4.42	0.08	0.13	0.01	0.01	2.28	1.29
Ccp	EA1205-032A	290	0.27	0.71	732000	1.23	74.70	0.02	6.70	0.38	0.35	0.00	0.00	3.50	3.08
Ccp	EA1205-032A	282	0.38	1.06	736000	0.52	209.30	0.01	5.85	0.27	0.11	0.00	0.00	2.15	1.64
Ccp	EA1205-032A	291	0.35	0.40	729000	1.34	94.90	0.01	21.20	0.36	0.21	0.00	0.00	3.78	2.96
Ccp	EA1205-034	243	19.90	6.40	735000	4.44	61.10	1.98	15.44	1.25	3.95	0.00	0.00	36.50	11.63
Ccp	EA1205-034	244	2.70	3.10	747000	0.73	39.60	1.43	14.58	0.72	5.09	0.00	0.02	10.90	7.93
Ccp	EA1205-034	237b	9.40	21.00	737000	3.78	60.70	0.59	27.50	1.17	1.45	0.00	0.00	69.00	32.70
Ccp	EA1205-034	254	3.53	0.85	743000	0.87	68.50	0.12	10.43	0.66	0.14	0.00	0.00	89.00	2.87
Ccp	EA1205-034	236	4.20	0.69	758000	2.19	41.00	0.10	13.10	0.62	0.38	0.01	0.00	56.20	13.20
Ccp	EA1205-034	235b	6.80	16.20	775000	1.47	73.00	0.09	10.70	0.84	0.25	0.02	0.01	63.00	8.61
Ccp	EA17196-144	455	16.20	7.78	800000	22.30	140.00	0.06	210.00	0.73	0.78	0.02	0.01	30.00	3.80
Ccp	EA17196-144	468	0.23	0.65	823000	15.60	116.00	0.06	31.10	0.38	0.11	0.00	0.03	32.00	5.65
Ccp	EA17196-144	475	1.18	1.06	822000	21.00	127.00	0.06	23.60	0.43	0.17	0.01	0.02	64.00	4.08
Ccp	EA17196-144	474	1.61	0.78	815300	9.51	139.20	0.04	21.39	0.31	0.17	0.00	0.01	119.00	3.12
Ccp	EA17196-144	463	5.80	0.85	800900	6.04	96.10	0.03	14.02	0.18	0.14	0.00	0.01	110.00	3.49
Ccp	EA17196-144	453	1.27	1.06	795000	20.30	122.00	0.02	28.30	0.37	0.20	0.00	0.01	15.80	5.15
Ccp	EA17196-144	157	6.60	0.66	816000	3.98	50.80	0.02	14.44	0.32	0.11		0.00	136.00	3.89
Ccp	EA17196-144	467	1.47	1.06	818000	7.80	117.30	0.02	14.56	0.32	0.11	0.00	0.01	70.00	2.35
Ccp	EA17196-144	160	2.75	1.06	819000	8.20	108.30	0.02	17.10	0.21	0.09		0.00	133.00	3.08
Ccp	EA17196-144	465	0.67	0.64	812900	4.50	119.30	0.02	12.28	0.35	0.11	0.00	0.01	22.20	2.36
Ccp	EA17196-144	470	0.69	0.92	827000	8.10	103.50	0.02	13.18	0.28	0.10	0.00	0.00	35.00	2.33

Appendix C. Table C.1. (Continued)

Phase	Sample	Analysis No.	Mn ⁵⁵	Ni ⁶¹	Cu ⁶³	As ⁷⁵	Se ⁷⁸	Pd ¹⁰⁶	Ag ¹⁰⁷	Sb ¹²¹	Te ¹²⁶	Pt ¹⁹⁵	Au ¹⁹⁷	Pb ²⁰⁸	Bi ²⁰⁹
Ccp	EA1205-032A	289	3.00	0.99	730000	4.54	313.70	0.06	20.60	2.45	0.22	0.01	0.00	26.00	9.14
Ccp	EA1205-032A	295	39.00	1.41	738000	1.93	333.00	0.04	11.79	0.89	0.23	0.01	0.01	17.10	4.51
Ccp	EA1205-032A	284	0.55	1.06	732000	2.88	236.00	0.04	13.98	1.54	0.22	0.01	0.02	10.60	9.05
Ccp	EA1205-032A	281	3.40	1.84	731000	2.86	187.40	0.03	16.30	2.00	0.11	0.01	0.02	8.53	6.40
Ccp	EA1205-032A	292	2.30	2.47	746000	4.33	256.00	0.03	19.60	1.82	0.34	0.01	0.01	10.00	9.30
Ccp	EA1205-032A	283	0.18	0.85	732000	0.18	234.00	0.02	4.42	0.08	0.13	0.01	0.01	2.28	1.29
Ccp	EA1205-032A	290	0.27	0.71	732000	1.23	74.70	0.02	6.70	0.38	0.35	0.00	0.00	3.50	3.08
Ccp	EA1205-032A	282	0.38	1.06	736000	0.52	209.30	0.01	5.85	0.27	0.11	0.00	0.00	2.15	1.64
Ccp	EA1205-032A	291	0.35	0.40	729000	1.34	94.90	0.01	21.20	0.36	0.21	0.00	0.00	3.78	2.96
Ccp	EA1205-034	243	19.90	6.40	735000	4.44	61.10	1.98	15.44	1.25	3.95	0.00	0.00	36.50	11.63
Ccp	EA1205-034	244	2.70	3.10	747000	0.73	39.60	1.43	14.58	0.72	5.09	0.00	0.02	10.90	7.93
Ccp	EA1205-034	237b	9.40	21.00	737000	3.78	60.70	0.59	27.50	1.17	1.45	0.00	0.00	69.00	32.70
Ccp	EA1205-034	254	3.53	0.85	743000	0.87	68.50	0.12	10.43	0.66	0.14	0.00	0.00	89.00	2.87
Ccp	EA1205-034	236	4.20	0.69	758000	2.19	41.00	0.10	13.10	0.62	0.38	0.01	0.00	56.20	13.20
Ccp	EA1205-034	235b	6.80	16.20	775000	1.47	73.00	0.09	10.70	0.84	0.25	0.02	0.01	63.00	8.61
Ccp	EA17196-144	461	6.90	0.71	840000	7.13	112.90	0.02	17.70	0.42	0.12		0.01	191.00	3.77
Ccp	EA17196-144	472	1.03	0.71	827100	2.33	112.00	0.01	10.90	0.30	0.11	0.00	0.00	23.00	1.87
Ccp	EA17196-144	446	0.20	0.67	792000	2.66	7.78	0.01	5.60	0.47	0.13	0.00	0.00	8.56	0.27
Ccp	EA17196-144	449	1.08	1.13	778000	24.00	80.00	0.01	8.76	0.15	0.13	0.00	0.00	5.66	0.33
Ccp	EA17196-144	447	0.30	0.66	781700	4.36	113.30	0.01	13.15	0.10	0.15	0.00	0.00	8.42	0.15
Ccp	EA17196-144	445	0.06	0.57	776000	2.63	19.10	0.00	4.61	0.46	0.13	0.00	0.01	8.57	0.36
Ccp	PO1634-156	478	129.00	2.26	870000	1360.00	23.33	6.40	339.00	51.00	4.60	0.05	0.22	680.00	134.00
Ccp	PO1634-156	478b	3.70	2.97	853000	124.00	78.00	0.60	930.00	1.21	1.32	0.04	0.01	101.00	18.80
Ccp	EA14141-168	504a	187.00	15.20	863000	81.00	54.00	0.10	22.10	7.54	0.23	0.01	0.05	38.90	18.20
Ccp	EA14141-168	504b	26.00	4.00	868000	22.00	39.00	0.06	8.50	3.70	0.25	0.01	0.06	29.10	7.00
Ccp	EA14141-168	505	3.00	1.27	613600	0.13	46.70	0.01	1.45	0.02	0.08	0.00	0.01	2.22	1.01

Appendix C. Table C.1. (Continued)

Phase	Sample	Analysis No.	Mn ⁵⁵	Ni ⁶¹	Cu ⁶³	As ⁷⁵	Se ⁷⁸	Pd ¹⁰⁶	Ag ¹⁰⁷	Sb ¹²¹	Te ¹²⁶	Pt ¹⁹⁵	Au ¹⁹⁷	Pb ²⁰⁸	Bi ²⁰⁹
Ccp	EA1205-032A	289	3.00	0.99	730000	4.54	313.70	0.06	20.60	2.45	0.22	0.01	0.00	26.00	9.14
Ccp	EA1205-032A	295	39.00	1.41	738000	1.93	333.00	0.04	11.79	0.89	0.23	0.01	0.01	17.10	4.51
Ccp	EA1205-032A	284	0.55	1.06	732000	2.88	236.00	0.04	13.98	1.54	0.22	0.01	0.02	10.60	9.05
Ccp	EA1205-032A	281	3.40	1.84	731000	2.86	187.40	0.03	16.30	2.00	0.11	0.01	0.02	8.53	6.40
Ccp	EA1205-032A	292	2.30	2.47	746000	4.33	256.00	0.03	19.60	1.82	0.34	0.01	0.01	10.00	9.30
Ccp	EA1205-032A	283	0.18	0.85	732000	0.18	234.00	0.02	4.42	0.08	0.13	0.01	0.01	2.28	1.29
Ccp	EA1205-032A	290	0.27	0.71	732000	1.23	74.70	0.02	6.70	0.38	0.35	0.00	0.00	3.50	3.08
Ccp	EA1205-032A	282	0.38	1.06	736000	0.52	209.30	0.01	5.85	0.27	0.11	0.00	0.00	2.15	1.64
Ccp	EA1205-032A	291	0.35	0.40	729000	1.34	94.90	0.01	21.20	0.36	0.21	0.00	0.00	3.78	2.96
Ccp	EA1205-034	243	19.90	6.40	735000	4.44	61.10	1.98	15.44	1.25	3.95	0.00	0.00	36.50	11.63
Ccp	EA1205-034	244	2.70	3.10	747000	0.73	39.60	1.43	14.58	0.72	5.09	0.00	0.02	10.90	7.93
Ccp	EA1205-034	237b	9.40	21.00	737000	3.78	60.70	0.59	27.50	1.17	1.45	0.00	0.00	69.00	32.70
Ccp	EA1205-034	254	3.53	0.85	743000	0.87	68.50	0.12	10.43	0.66	0.14	0.00	0.00	89.00	2.87
Ccp	EA1205-034	236	4.20	0.69	758000	2.19	41.00	0.10	13.10	0.62	0.38	0.01	0.00	56.20	13.20
Ccp	EA1205-034	235b	6.80	16.20	775000	1.47	73.00	0.09	10.70	0.84	0.25	0.02	0.01	63.00	8.61
Ccp	EA14141-168	497	0.11	1.48	953000	0.18	14.14	0.01	1.76	0.17	0.09	0.00	0.00	1.45	0.70
Ccp	UA12206-211	516	8.80	4.90	521000	280.00	662.00	13.40	95.00	6.60	86.00	0.15	0.12	49.00	10.50
Ccp	UA12206-211	517	2.10	1.63	557000	26.00	797.00	0.18	19.60	2.00	0.60	0.00	0.02	350.00	180.00
Ccp	UA12206-211	521	16.26	26.16	528000	0.85	386.00	0.18	9.10	0.18	0.62		0.02	4.00	0.45
Ccp	UA12206-211	527	24.00	9.19	524000	0.99	280.00	0.09	120.00	0.27	1.63	0.03		8.70	0.04
Ccp	UA12206-211	541	1.90	2.40	641000	0.72	117.00	0.02	4.40	0.29	0.16	0.01	0.02	5.60	0.07
Ccp	UA12206-211	527	3.70	1.41	526000	0.32	230.00	0.01	7.00	0.04	0.32	0.00	0.00	3.85	0.01

Table C.2. Pyrite LA-ICPMS trace element data.

Phase	Sample	Analysis No.	Mn ⁵⁵	Ni ⁶¹	Cu ⁶³	As ⁷⁵	Se ⁷⁸	Pd ¹⁰⁶	Ag ¹⁰⁷	Sb ¹²¹	Te ¹²⁶	Pt ¹⁹⁵	Au ¹⁹⁷	Pb ²⁰⁸	Bi ²⁰⁹
Py	AF17611-003A	162a	0.18	2670.00	49	65.60	760.00	18.00	0.27	0.07	0.56	1.24	0.01	0.21	0.25
Py	AF17611-003A	160	0.48	5290.00	6500	51.50	848.00	15.70	2.63	0.02	0.38	0.27	0.03	1.57	0.88
Py	AF17611-003B	167	0.07	2560.00	680	67.40	764.00	15.70	0.15	0.03	0.48	2.95	0.02	0.25	0.47
Py	AF17611-003B	162b	0.46	1491.00	760	111.60	542.00	13.10	0.37	0.05	0.41	0.32	0.04	0.40	0.76
Py	AF17611-003B	190b	0.13	2480.00	48	41.10	874.00	11.80	0.03	0.02	0.35	0.07	0.00	0.10	0.05
Py	AF17611-003B	205b	0.03	3900.00	2900	71.00	726.00	7.80	5.80	0.20	0.54	0.46	0.02	7.20	3.60
Py	AF17611-003B	194b	0.02	22.50	51	54.00	751.00	0.06	0.01	0.01	1.78	0.04	0.00	0.77	0.01
Py	EA1205-032A	288	10.30	1040.00	13100	25.70	293.00	18.60	1.23	0.07	3.28	0.18	0.21	1.05	7.40
Py	EA1205-032A	286	2.26	3530.00	1140	100.20	403.00	12.23	0.44	0.05	0.34	0.01	0.07	0.36	2.44
Py	EA1205-032A	294	0.62	645.00	830	36.20	267.00	11.04	0.87	0.02	1.74	0.48	0.03	0.59	2.34
Py	EA1205-032A	279	0.59	219.00	2300	33.90	93.40	5.88	0.60	0.02	1.96	0.05	0.06	0.40	3.78
Py	EA1205-032A	285	3.00	305.00	3900	122.30	299.00	4.00	0.58	0.03	0.72	0.18	0.06	0.34	2.60
Py	EA1205-032A	287	2.55	164.00	2100	282.00	372.00	2.00	15.20	0.28	1.42	6.90	0.01	10.80	7.00
Py	EA1205-032A	295	0.10	506.00	4200	21.80	82.00	0.90	0.99	0.03	0.31	0.02	0.02	0.32	1.99
Py	EA1205-034	248	0.39	2410.00	9300	27.90	50.20	3.70	25.40	5.20	4.90	0.02	0.00	10.50	28.00
Py	EA1205-034	239	0.19	1250.00	7.7	7.70	78.40	1.36	0.12	0.02	0.22	0.01	0.00	0.05	0.64
Py	EA1205-034	245	0.04	5090.00	4.1	45.00	45.20	1.17	0.02	0.01	1.27	0.01	0.00	0.07	0.00
Py	EA1205-034	246	0.03	1580.00	5	12.90	42.20	1.08	0.03	0.01	1.98	0.00	0.01	0.23	0.12
Py	EA1205-034	242	0.14	1370.00	0.76	37.00	80.60	0.70	0.10	0.05	1.40	0.06	0.00	0.01	0.28
Py	EA1205-034	240b	0.08	2600.00	3300	17.60	93.80	0.56	0.42	0.02	0.12	0.01	0.05	0.08	1.87
Py	EA1205-034	247	0.25	1403.00	5800	41.80	76.00	0.53	11.40	0.19	0.47	0.02	0.00	8.70	11.10
Py	EA1205-034	249	0.03	360.00	0.22	2.80	52.30	0.11	0.02	0.01	0.91	0.01	0.00	0.01	0.00
Py	AF8103-052A	315	3.40	9670.00	790	163.30	672.00	30.80	1.13	0.13	0.21	0.00	0.11	25.00	4.45
Py	AF8103-052A	314	0.02	6370.00	40	108.20	723.00	12.45	0.22	0.01	0.04		0.00	1.80	0.19
Py	AF8103-052A	325	0.12	4800.00	3300	64.80	729.00	8.56	1.09	0.02	0.16	0.02	0.02	7.50	1.20

Appendix C. Table C.2. (Continued)

Phase	Sample	Analysis No.	Mn ⁵⁵	Ni ⁶¹	Cu ⁶³	As ⁷⁵	Se ⁷⁸	Pd ¹⁰⁶	Ag ¹⁰⁷	Sb ¹²¹	Te ¹²⁶	Pt ¹⁹⁵	Au ¹⁹⁷	Pb ²⁰⁸	Bi ²⁰⁹
Py	AF8103-052A	317	1.38	760.00	830	196.00	445.00	6.60	2.41	2.40	0.31	0.19	0.00	6.70	2.80
Py	AF8103-052A	324	1.97	799.00	4400	100.90	1020.00	2.27	6.60	0.17	0.13	0.14	0.01	3900.00	8.90
Py	AF8103-052A	316	0.09	817.00	2470	242.40	337.00	1.01	1.88	0.05	0.09	0.52	0.01	5.00	1.55
Py	AF8103-052B	344	0.32	16.00	80000	#####	347.00	3.21	14.70	60.00	0.56	0.03	0.28	35.00	3.90
Py	AF8103-052B	343	2.26	64.00	110000	189.00	374.00	2.23	6.80	2.10	0.18	0.01	0.25	650.00	2.94
Py	UA7906-056	345	1.32	5940.00	2660	36.80	36.40	1.42	0.88	0.12	0.26	0.00	0.10	0.00	9.80
Py	UA7906-056	345b	3.20	3750.00	710	58.00	40.60	1.03	0.98	0.09	0.29	0.00	0.03	6.20	7.18
Py	UA7906-056	358	4.80	127.00	175	147.00	21.70	0.92	0.67	0.07	0.48	0.00	0.05	0.58	9.60
Py	UA7906-056	362	2.59	219.00	65	316.00	19.60	0.62	1.33	0.16	0.65	0.00	0.13	7.20	16.90
Py	UA7906-056	351	0.61	2820.00	195	35.20	41.20	0.61	0.17	0.07	0.19	0.00	0.03	0.58	3.13
Py	UA7906-056	357	0.83	123.00	9.5	542.00	23.30	0.56	0.15	0.13	0.38	0.01	0.02	0.96	2.52
Py	UA7906-056	353	0.72	1450.00	900	181.00	22.80	0.51	1.25	0.08	0.30	0.00	0.14	4.95	6.90
Py	UA7906-056	361	0.36	210.00	14	844.00	16.90	0.50	0.13	0.03	0.69		0.05	0.37	7.30
Py	UA7906-056	360	0.07	149.00	3.5	138.00	8.70	0.15	0.02	0.01	0.06	0.00	0.00	0.00	0.17
Py	UA7906-056	359	0.04	97.00	28	4.17	3.32	0.13	0.12	0.02	0.06	0.00	0.00	0.00	0.33
Py	UA7906-056	353	0.55	57.40	46	1080.00	12.10	0.01	0.52	0.03	0.69	0.00	0.10	1.63	1.79
Py	UA7906-056	354	0.12	172.00	3.3	228.80	19.50	0.00	0.04	0.02	0.49	0.00	0.01	0.37	0.28
Py	UA7906-056	356	0.15	109.00	22	56.00	24.40	0.00	0.09	0.01	0.22	0.00	0.02	0.24	2.10
Py	UA7906-057A	387	0.08	2110.00	42	38.90	43.90	0.90	0.28	0.03	0.57	0.00	0.00	3.10	0.78
Py	UA7906-057A	382	0.58	196.00	284	357.00	40.40	0.89	1.26	0.25	0.21	0.00	0.01	13.30	4.60
Py	UA7906-057A	383	1.53	33.00	39	181.00	12.30	0.53	0.02	0.02	0.16	0.00	0.00	0.14	1.22
Py	UA7906-057A	373	0.04	144.00	10.3	825.00	34.20	0.50	0.03	0.02	0.80	0.00	0.03	0.13	0.55
Py	UA7906-057A	386	0.04	20.00	50	46.00	36.70	0.26	0.07	0.02	0.60		0.01	0.86	0.43
Py	UA7906-057A	377	0.04	567.00	3.7	129.00	48.90	0.25	0.02	0.01	0.08	0.00		0.00	0.10
Py	UA7906-057A	380	0.04	67.00	10	320.00	17.50	0.23	0.02	0.01	0.15		0.01	0.05	0.16
Py	UA7906-057A	375	0.03	67.00	13	504.00	61.50	0.21	0.14	0.02	0.73	0.00	0.01	0.60	2.02

Appendix C. Table C.2. (Continued)

Phase	Sample	Analysis No.	Mn ⁵⁵	Ni ⁶¹	Cu ⁶³	As ⁷⁵	Se ⁷⁸	Pd ¹⁰⁶	Ag ¹⁰⁷	Sb ¹²¹	Te ¹²⁶	Pt ¹⁹⁵	Au ¹⁹⁷	Pb ²⁰⁸	Bi ²⁰⁹
Py	UA7906-057A	385	0.04	207.00	31	1820.00	22.30	0.12	0.04	0.01	0.94	0.00	0.56	0.26	0.08
Py	UA7906-057A	378	0.04	89.00	2.6	7.18	9.19	0.08	0.03	0.02	0.06		0.00	0.06	0.08
Py	UA7906-057A	372	0.03	1.28	23	3.85	4.95	0.06	0.02	0.01	0.07		0.00	0.04	0.00
Py	UA7906-057A	369	0.04	47.00	19	870.00	31.00	0.03	0.02	0.01	1.69		0.01	0.12	0.20
Py	UA7906-057A	376	0.04	45.10	0.64	59.40	7.78	0.03	0.02	0.01	0.31	0.00	0.03	0.33	0.04
Py	UA7906-057A	368	0.03	4.90	42	5.36	4.31	0.01	0.04	0.02	0.07		0.00	0.13	0.04
Py	UA7906-057A	371	0.03	107.00	11.1	5.82	4.81	0.01	0.01	0.01	0.04	0.00	0.00	0.00	0.02
Py	UA7906-057A	384	0.04	38.50	29	1000.00	15.70	0.01	0.03	0.01	0.51	0.00	0.06	0.15	0.04
Py	UA7906-057A	370	0.04	0.67	94	3.80	5.59	0.00	0.23	0.02	0.08		0.01	3.10	3.09
Py	UA7906-057A	379	8.40	219.00	15	1419.00	56.50	0.00	0.02	0.02	1.61	0.00	0.06	0.13	0.77
Py	AF0135-129	396	1.47	1560.00	3200	569.00	175.00	1.97	5.98	3.98	1.17	0.00	0.61	178.00	19.60
Py	AF0135-129	398	0.51	1990.00	37100	1723.00	39.00	1.95	13.30	11.05	0.74	0.00	1.58	138.00	12.30
Py	AF0135-129	403	19.09	1120.00	14300	1069.00	205.00	1.74	35.70	17.60	1.43	0.01	1.76	7100.00	44.70
Py	AF0135-129	397	24.00	1290.00	560	730.00	236.00	1.58	1.27	1.05	0.93	0.01	0.27	240.00	8.80
Py	AF0135-129	404	9.80	1386.00	17900	507.00	134.00	0.55	12.20	10.40	1.27	0.00	0.73	352.00	22.40
Py	AF0135-129	405	4.40	1550.00	3610	535.00	47.00	0.44	13.00	9.20	0.56		1.34	42.90	17.60
Py	AF0135-129	402	66.00	3290.00	46300	2122.00	9.19	0.34	16.50	9.70	0.07	0.01	2.07	64.20	9.90
Py	AF0135-129	400	0.68	430.00	370	1010.00	72.00	0.19	0.81	1.33	0.45	0.01	0.29	90.00	4.15
Py	AF0135-129	406	2.10	1130.00	5420	549.00	47.20	0.15	8.13	14.60	0.11	0.00	1.24	50.80	12.50
Py	AF0135-129	399	18.38	128.00	562	60.50	201.00	0.06	7.57	0.38	0.11	0.00	0.14	21.20	5.46
Py	AF0135-129	407	0.05	400.00	94	45.20	125.00	0.02	0.26	0.17	0.22	0.00	0.02	0.40	1.35
Py	EA14141-168	488	2.10	750.00	3700	8.80	24.90	0.62	0.66	0.02	0.35	0.00	0.08	0.92	6.00
Py	EA14141-168	490	3.20	210.00	2400	10.27	13.60	0.26	0.57	0.01	0.25		0.05	0.92	3.80
Py	EA14141-168	508	2.70	148.00	2000	4.60	1.98	0.24	5.70	0.21	0.11		0.01	8.80	6.00
Py	EA14141-168	495	7.00	814.00	5130	17.00	51.60	0.18	1.08	0.02	0.37	0.00	0.10	1.54	10.57
Py	EA14141-168	494	1.16	430.00	1410	9.91	30.10	0.05	0.23	0.01	0.33	0.00	0.01	0.25	1.93

Appendix C. Table C.2. (Continued)

Phase	Sample	Analysis No.	Mn ⁵⁵	Ni ⁶¹	Cu ⁶³	As ⁷⁵	Se ⁷⁸	Pd ¹⁰⁶	Ag ¹⁰⁷	Sb ¹²¹	Te ¹²⁶	Pt ¹⁹⁵	Au ¹⁹⁷	Pb ²⁰⁸	Bi ²⁰⁹
Py	EA14141-168	493	1.32	2.04	88	3.58	12.70	0.04	0.12	0.01	0.09		0.01	0.80	1.27
Py	EA14141-168	491	2.04	92.00	840	15.40	19.30	0.03	3.20	0.09	0.57	0.00	0.01	6.10	4.10
Py	EA14141-168	481	0.04	12.50	9.1	2.27	3.04	0.02	0.01	0.01	0.03	0.00	0.00	0.04	0.11
Py	EA14141-168	487	0.04	0.78	143	5.80	7.78	0.01	0.93	0.03	0.18		0.00	1.12	0.75
Py	EA14141-168	480	1.26	1.78	58	3.01	2.97	0.01	0.77	0.03	0.03	0.00	0.01	1.76	1.95
Py	EA14141-168	485	0.32	0.39	69	9.50	3.18	0.00	0.48	0.04	0.16		0.00	0.94	0.52
Py	EA14141-168	482	0.02	0.40	35	1.65	3.96	0.00	0.01	0.01	0.05		0.00	0.12	0.00
Py	EA14141-168	482b	0.02	0.46	0.13	1.65	3.68	0.00	0.01	0.01	0.04	0.00	0.00		0.00
Py	EA14141-168	484	0.02	0.29	13.3	1.54	7.00	0.00	0.01	0.01	0.04		0.00	0.05	0.05
Py	EA14141-168	483	0.02	7.70	14.2	3.28	3.11	0.00	0.01	0.01	0.21	0.00	0.00	0.03	0.00
Py	EA14141-168	492	0.35	0.28	17.4	1.62	24.20	0.00	0.01	0.01	0.14		0.00	0.08	0.37

Appendix D Sulfide EPMA analyses

Table D.1. Chalcopyrite EPMA analyses

Sample	Line	S (wt%)	Sb (wt%)	Fe (wt%)	Cu (wt%)	As (wt%)	Zn (wt%)	Ag (wt%)	Hg (wt%)	Ni (wt%)	Total
003A	155	34.94	b.d.l.	30.32	34.74	b.d.l.	b.d.l.	0.01	0.01	b.d.l.	100.03
003A	156	35.03	b.d.l.	30.3	34.84	b.d.l.	b.d.l.	b.d.l.	0.01	b.d.l.	100.18
003A	157	35.06	b.d.l.	30.46	35	b.d.l.	b.d.l.	b.d.l.	0.01	b.d.l.	100.53
003A	158	34.87	b.d.l.	30.31	34.68	b.d.l.	b.d.l.	0.01	0.04	b.d.l.	99.9
003A	159	34.8	0.02	30.38	34.48	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	99.68
003A	160	54	b.d.l.	47.66	0.04	b.d.l.	b.d.l.	b.d.l.	0.02	0.01	101.74
003A	161	54.11	0.01	47.3	0.04	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.34	101.8
003A	163	35.01	b.d.l.	30.35	34.63	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100
003A	164	34.99	b.d.l.	30.6	34.56	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	100.15
003A	165	35.03	b.d.l.	30.29	34.66	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.98
003A	166	35.13	0.01	30.42	34.61	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.17
003A	168	34.9	b.d.l.	30.2	34.67	b.d.l.	b.d.l.	0.01	0.01	b.d.l.	99.8
003A	169	34.97	0.02	29.93	34.25	b.d.l.	b.d.l.	b.d.l.	0.03	b.d.l.	99.2
003A	170	34.94	b.d.l.	30.44	34.64	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.02
003A	171	34.89	b.d.l.	30.48	34.82	0	b.d.l.	0	0.05	b.d.l.	100.25
003A	172	34.9	b.d.l.	30.23	34.85	b.d.l.	b.d.l.	b.d.l.	0.04	b.d.l.	100.03
003A	176	34.96	b.d.l.	30.27	34.81	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.04
003A	177	34.93	0.01	30.16	34.71	b.d.l.	b.d.l.	b.d.l.	0.03	0.01	99.84
003A	178	29.53	0.01	26.65	28.78	0	b.d.l.	b.d.l.	0.02	b.d.l.	84.98
003A	179	34.98	b.d.l.	30.52	34.89	b.d.l.	b.d.l.	b.d.l.	0.03	b.d.l.	100.41
003A	180	35.21	b.d.l.	30.33	34.85	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.39
003B	183	34.69	0.02	30.25	34.65	b.d.l.	b.d.l.	b.d.l.	0	b.d.l.	99.6
003B	184	34.7	0	30.07	34.72	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.49

Appendix D. Table D.1. (Continued)

Sample	Line	S (wt%)	Sb (wt%)	Fe (wt%)	Cu (wt%)	As (wt%)	Zn (wt%)	Ag (wt%)	Hg (wt%)	Ni (wt%)	Total
003B	185	34.73	b.d.l.	30.16	34.8	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.69
003B	186	34.76	b.d.l.	30.25	34.71	b.d.l.	b.d.l.	b.d.l.	0	b.d.l.	99.73
003B	187	34.88	b.d.l.	30.09	34.58	b.d.l.	b.d.l.	0.01	b.d.l.	0	99.56
003B	188	53.62	b.d.l.	47.25	0.02	b.d.l.	b.d.l.	b.d.l.	0.01	b.d.l.	100.9
003B	189	53.88	0.01	46.56	0.03	b.d.l.	b.d.l.	0	b.d.l.	0.6	101.09
003B	190	53.61	0.01	46.25	0.02	b.d.l.	b.d.l.	0.01	b.d.l.	1.18	101.07
003B	199	34.8	b.d.l.	30.11	34.55	b.d.l.	b.d.l.	0	0	b.d.l.	99.47
003B	200	34.98	b.d.l.	29.54	34.35	b.d.l.	b.d.l.	b.d.l.	0.01	b.d.l.	98.88
003B	201	34.9	b.d.l.	29.68	34.54	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	99.13
003B	202	34.77	0.01	29.78	34.47	b.d.l.	b.d.l.	0.02	b.d.l.	b.d.l.	99.06
003B	203	34.8	b.d.l.	29.76	34.43	b.d.l.	b.d.l.	0.02	b.d.l.	b.d.l.	99.02
003B	204	34.87	b.d.l.	29.85	34.5	b.d.l.	b.d.l.	b.d.l.	0.04	b.d.l.	99.27
003B	208	34.93	0.01	30.14	34.62	b.d.l.	b.d.l.	b.d.l.	0.01	b.d.l.	99.71
003B	209	34.77	b.d.l.	30.1	34.8	b.d.l.	b.d.l.	0.01	0.02	0	99.69
003B	210	34.81	b.d.l.	30.11	34.69	b.d.l.	b.d.l.	0	0.01	b.d.l.	99.62
003B	211	34.78	0	30.23	34.64	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.64
003B	212	34.77	0.03	30.28	34.73	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.82
003B	213	34.8	b.d.l.	30.13	34.69	b.d.l.	b.d.l.	0.02	b.d.l.	b.d.l.	99.64
003B	214	34.79	0	29.83	34.17	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	98.79
003B	215	34.86	b.d.l.	30.08	34.66	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.59
30	216	34.84	0.02	30.35	34.55	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	99.75
30	217	34.84	b.d.l.	30.12	34.34	b.d.l.	b.d.l.	0.02	b.d.l.	b.d.l.	99.32
30	218	35.21	b.d.l.	29.87	34.07	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	99.14
30	219	34.88	0.02	30.3	34.69	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.89
30	220	34.66	b.d.l.	30.3	34.51	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.01	99.48
30	221	34.93	b.d.l.	30.3	34.62	b.d.l.	b.d.l.	0.02	0.01	b.d.l.	99.89

Appendix D. Table D.1. (Continued)

Sample	Line	S (wt%)	Sb (wt%)	Fe (wt%)	Cu (wt%)	As (wt%)	Zn (wt%)	Ag (wt%)	Hg (wt%)	Ni (wt%)	Total
30	222	35.2	b.d.l.	30.12	34.58	b.d.l.	b.d.l.	0.02	0.01	b.d.l.	99.94
30	223	34.95	b.d.l.	30.19	34.72	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	99.87
30	224	35.1	b.d.l.	30.02	34.28	b.d.l.	b.d.l.	0.01	0.01	b.d.l.	99.41
30	225	35.09	b.d.l.	30.2	34.53	b.d.l.	b.d.l.	b.d.l.	0.03	0	99.85
30	226	34.9	b.d.l.	29.85	34.29	b.d.l.	b.d.l.	0.02	b.d.l.	b.d.l.	99.06
30	227	35.11	b.d.l.	29.98	34.25	b.d.l.	b.d.l.	0	b.d.l.	0	99.34
30	228	34.78	b.d.l.	30.23	34.52	b.d.l.	b.d.l.	b.d.l.	0.03	b.d.l.	99.57
30	229	34.96	b.d.l.	30.44	34.53	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	99.94
34	230	34.9	b.d.l.	30.34	34.45	b.d.l.	b.d.l.	b.d.l.	0.02	b.d.l.	99.71
34	231	34.9	b.d.l.	30.56	34.63	b.d.l.	b.d.l.	0.01	b.d.l.	0.01	100.12
34	232	34.89	b.d.l.	30.39	34.34	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.62
34	236	34.97	b.d.l.	30.44	34.49	b.d.l.	b.d.l.	0	0.03	b.d.l.	99.93
34	237	34.92	b.d.l.	30.68	34.6	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.21
34	238	35.06	0	30.65	34.67	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0	100.38
34	243	35	b.d.l.	30.58	34.57	b.d.l.	b.d.l.	b.d.l.	0	b.d.l.	100.15
34	244	34.91	b.d.l.	30.55	34.63	b.d.l.	b.d.l.	0.01	0	b.d.l.	100.11
34	250	35.09	b.d.l.	30.59	34.6	b.d.l.	b.d.l.	b.d.l.	0.01	b.d.l.	100.29
34	251	34.71	b.d.l.	30.61	34.55	b.d.l.	b.d.l.	0	0.01	b.d.l.	99.88
34	252	34.83	b.d.l.	30.21	34.41	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.45
34	253	34.83	b.d.l.	30.34	34.31	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.48
34	254	35.09	b.d.l.	30.2	34.13	b.d.l.	b.d.l.	b.d.l.	0	b.d.l.	99.42
34	255	35.01	0.01	30.43	34.26	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.71
34	256	34.74	b.d.l.	30.48	34.68	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	99.91
34	257	34.91	0.01	30.38	34.37	b.d.l.	b.d.l.	b.d.l.	0.02	b.d.l.	99.69
34	258	34.73	0.01	30.44	34.45	b.d.l.	b.d.l.	b.d.l.	0.04	b.d.l.	99.67
34	259	34.8	0.01	30.17	34.26	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	99.23

Appendix D. Table D.1. (Continued)

Sample	Line	S (wt%)	Sb (wt%)	Fe (wt%)	Cu (wt%)	As (wt%)	Zn (wt%)	Ag (wt%)	Hg (wt%)	Ni (wt%)	Total
34	260	34.96	b.d.l.	30.37	34.37	b.d.l.	b.d.l.	0	0.01	b.d.l.	99.71
34	261	35.11	b.d.l.	30.24	34.61	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.95
34	262	34.76	0.01	30.28	34.12	b.d.l.	b.d.l.	b.d.l.	0.02	b.d.l.	99.2
34	263	34.85	0.02	30.22	34.36	b.d.l.	b.d.l.	b.d.l.	0.02	b.d.l.	99.46
34	264	34.77	b.d.l.	30.19	34.3	b.d.l.	b.d.l.	b.d.l.	0.02	b.d.l.	99.28
34	265	35.14	0	30.21	34.32	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.68
032A	281	34.62	b.d.l.	30.59	35.29	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	100.5
032A	282	34.78	0	30.66	35.21	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.66
032A	283	34.77	0.04	30.47	35.2	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.47
032A	284	32.74	b.d.l.	28.95	33.29	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	94.98
032A	289	34.88	0	30.57	35.37	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.82
032A	290	34.94	b.d.l.	30.7	35.36	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	101.01
032A	291	34.71	0.01	30.68	35.4	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.81
032A	292	34.91	0.01	30.6	35.26	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	100.78
032A	293	34.82	0.01	30.62	35.31	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.77
48	296	34.84	b.d.l.	30.65	35.38	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.87
48	297	34.67	b.d.l.	30.64	35.28	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.6
48	298	34.82	b.d.l.	30.5	35.38	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.7
48	299	34.96	0.01	30.47	35.46	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.9
48	300	34.95	0	30.59	35.34	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.87
48	301	34.83	b.d.l.	30.52	35.36	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.71
48	302	34.79	b.d.l.	30.72	35.48	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.99
48	303	34.98	b.d.l.	30.7	35.36	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0	101.05
48	304	34.75	b.d.l.	30.71	35.34	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.81
48	305	34.71	b.d.l.	30.56	35.35	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.62
052A	306	34.9	b.d.l.	30.71	35.54	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	101.16

Appendix D. Table D.1. (Continued)

Sample	Line	S (wt%)	Sb (wt%)	Fe (wt%)	Cu (wt%)	As (wt%)	Zn (wt%)	Ag (wt%)	Hg (wt%)	Ni (wt%)	Total
052A	307	34.83	b.d.l.	30.53	35.55	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.91
052A	308	34.75	b.d.l.	30.55	35.49	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.79
052A	309	34.88	b.d.l.	30.38	35.45	b.d.l.	b.d.l.	0.01	b.d.l.	0	100.73
052A	310	34.79	b.d.l.	30.34	35.58	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.71
052A	311	35.06	b.d.l.	30.55	35.4	b.d.l.	b.d.l.	0.02	b.d.l.	b.d.l.	101.04
052A	312	34.79	b.d.l.	30.88	35.36	b.d.l.	b.d.l.	0.02	b.d.l.	b.d.l.	101.06
052A	313	34.81	b.d.l.	30.8	35.34	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0	100.95
052A	318	34.86	b.d.l.	30.64	35.5	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0	101.01
052A	319	34.82	b.d.l.	30.52	35.47	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.81
052A	320	34.73	b.d.l.	30.61	35.41	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.76
052A	321	34.93	b.d.l.	30.82	35.54	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	101.3
052A	322	34.94	b.d.l.	30.77	35.64	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	101.35
052A	323	34.85	0.02	30.78	35.47	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	101.12
052B	326	34.8	b.d.l.	30.7	35.44	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.94
052B	327	33.24	b.d.l.	29.67	34.11	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	97.03
052B	328	34.89	b.d.l.	30.82	35.42	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	101.15
052B	329	34.78	b.d.l.	30.61	35.58	b.d.l.	b.d.l.	0.02	b.d.l.	b.d.l.	100.99
052B	330	34.81	b.d.l.	30.73	35.53	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	101.06
052B	331	34.97	b.d.l.	30.68	35.59	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	101.24
052B	332	34.98	b.d.l.	30.78	35.55	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	101.32
052B	333	34.96	b.d.l.	30.16	35.58	b.d.l.	b.d.l.	0.02	b.d.l.	b.d.l.	100.72
052B	334	34.82	b.d.l.	30.74	35.55	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	101.11
052B	335	34.77	b.d.l.	30.66	35.43	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	100.87
052B	336	34.94	0	30.6	35.36	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.91
052B	337	34.76	b.d.l.	30.61	35.31	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	100.69
052B	338	34.83	b.d.l.	30.73	35.43	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.99

Appendix D. Table D.1. (Continued)

Sample	Line	S (wt%)	Sb (wt%)	Fe (wt%)	Cu (wt%)	As (wt%)	Zn (wt%)	Ag (wt%)	Hg (wt%)	Ni (wt%)	Total
052B	339	34.75	0.01	30.32	35.04	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.14
052B	340	35.03	b.d.l.	30.75	35.18	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	100.96
052B	341	34.8	0.01	30.42	35.36	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	100.59
052B	342	34.83	b.d.l.	30.75	35.38	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.97
56	347	34.72	b.d.l.	30.45	35.44	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.63
56	348	34.76	b.d.l.	30.52	35.35	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.64
56	349	34.66	b.d.l.	30.67	35.14	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.47
56	350	34.45	b.d.l.	30.5	35.24	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.19
56	363	34.74	0.01	30.72	35.5	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.97
56	364	34.75	b.d.l.	30.72	35.42	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.89
56	365	34.88	b.d.l.	31.08	35.42	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	101.39
56	366	34.82	b.d.l.	30.82	35.39	b.d.l.	b.d.l.	0.02	b.d.l.	b.d.l.	101.06
56	367	34.72	0.02	30.93	35.47	0	b.d.l.	0.01	b.d.l.	b.d.l.	101.15
057A	388	34.72	0.01	30.49	35.27	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0	100.48
057A	389	34.78	b.d.l.	30.49	35.42	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.69
057A	390	34.67	b.d.l.	30.34	35.45	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0	100.46
057A	391	34.77	b.d.l.	30.47	35.26	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	100.5
057A	392	34.65	0	30.4	35.49	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.54
057A	393	34.73	0	30.43	35.42	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.59
057A	394	34.76	b.d.l.	30.37	35.31	b.d.l.	b.d.l.	0.01	b.d.l.	0	100.45
057A	395	34.9	b.d.l.	30.53	35.22	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.65
143	408	34.62	0.02	30.26	34.95	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	99.85
143	409	34.87	b.d.l.	30.37	34.85	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.09
143	410	34.65	b.d.l.	30.36	34.68	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.69
143	411	34.72	b.d.l.	29.91	34.8	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.43
143	412	34.74	b.d.l.	30.19	34.62	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	99.55

Appendix D. Table D.1. (Continued)

Sample	Line	S (wt%)	Sb (wt%)	Fe (wt%)	Cu (wt%)	As (wt%)	Zn (wt%)	Ag (wt%)	Hg (wt%)	Ni (wt%)	Total
143	413	34.59	b.d.l.	30.29	35.18	0	b.d.l.	0	b.d.l.	b.d.l.	100.06
143	414	34.76	b.d.l.	30.48	34.18	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	99.42
143	415	35.03	0	30.36	35.07	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.46
143	416	34.67	b.d.l.	30.36	35.09	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.12
143	417	34.62	b.d.l.	30.12	34.29	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.03
143	418	34.68	0	30.05	34.61	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	99.34
143	419	34.81	0.02	30.07	34.09	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	99.01
143	420	34.76	b.d.l.	30.16	34.92	b.d.l.	b.d.l.	0	b.d.l.	0	99.84
143	421	34.6	b.d.l.	29.77	34.53	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	98.9
143	422	34.82	b.d.l.	30.08	34.95	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	99.85
143	423	34.71	b.d.l.	30.43	35.21	b.d.l.	b.d.l.	0.02	b.d.l.	b.d.l.	100.37
143	424	34.82	b.d.l.	30.42	35.14	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.01	100.39
143	425	34.76	b.d.l.	30.51	34.78	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	100.04
143	426	34.88	b.d.l.	30.45	35.41	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.76
143	427	34.95	0.01	30.2	35.13	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.29
143	428	34.72	b.d.l.	30.35	35.2	0	b.d.l.	0.02	b.d.l.	b.d.l.	100.28
143	429	34.86	b.d.l.	30.7	35.09	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0	100.65
143	430	34.9	0	30.61	35.29	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.81
143	431	34.55	b.d.l.	30.42	35.15	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.01	100.12
143	432	34.73	b.d.l.	30.4	35.31	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.45
143	433	34.71	b.d.l.	30.01	34.94	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.66
143	434	34.69	0	30.31	35.01	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	100.02
144	445	35.41	b.d.l.	30.44	35.06	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0	100.91
144	446	35.34	b.d.l.	30.24	34.98	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.56
144	447	35.24	b.d.l.	30.3	34.93	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.47
144	448	35.29	b.d.l.	30.36	34.99	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	100.65

Appendix D. Table D.1. (Continued)

Sample	Line	S (wt%)	Sb (wt%)	Fe (wt%)	Cu (wt%)	As (wt%)	Zn (wt%)	Ag (wt%)	Hg (wt%)	Ni (wt%)	Total
144	449	22.73	0.35	23.34	22.12	b.d.l.	b.d.l.	0	b.d.l.	0.02	68.57
144	450	35.57	0.01	30.38	34.92	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0	100.89
144	451	35.19	b.d.l.	30.18	34.78	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.15
144	452	35.26	b.d.l.	30.57	34.9	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	100.74
144	453	35.16	b.d.l.	30.34	34.86	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.37
144	454	35.22	b.d.l.	30.23	34.76	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.21
144	455	35.32	b.d.l.	30.32	34.89	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	100.52
144	456	35.46	0	30.5	34.87	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.83
144	457	35.49	b.d.l.	30.42	34.77	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.68
144	458	35.43	b.d.l.	30.49	34.75	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.68
144	459	35.43	0.03	30.42	34.87	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.76
144	460	35.28	b.d.l.	30.54	34.8	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.63
144	461	35.38	b.d.l.	30.4	34.93	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	100.71
144	462	35.07	b.d.l.	30.49	34.87	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.43
144	463	35.33	0.02	30.59	33.83	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.77
144	464	35.36	b.d.l.	30.51	35	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.87
144	465	35.26	0.01	30.41	35	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	100.67
144	466	35.56	0.02	30.41	35.03	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	101.03
144	467	35.48	0.02	30.48	35.05	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	101.04
144	468	35.5	b.d.l.	30.53	34.94	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0	100.98
144	469	35.38	b.d.l.	30.45	34.97	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.8
144	470	35.37	b.d.l.	30.67	35.01	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	101.05
144	471	35.22	0.06	30.48	34.83	0	b.d.l.	0.01	b.d.l.	b.d.l.	100.6
144	472	35.41	0	30.62	35.12	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	101.15
144	473	35.51	b.d.l.	30.44	34.96	0	b.d.l.	0.01	b.d.l.	b.d.l.	100.93
144	474	35.41	b.d.l.	30.49	35.16	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	101.07

Appendix D. Table D.1. (Continued)

Sample	Line	S (wt%)	Sb (wt%)	Fe (wt%)	Cu (wt%)	As (wt%)	Zn (wt%)	Ag (wt%)	Hg (wt%)	Ni (wt%)	Total
144	475	35.2	b.d.l.	30.47	35.03	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.7
144	476	34.9	0.01	30.13	34.59	b.d.l.	b.d.l.	0.02	b.d.l.	b.d.l.	99.65
156	477	35.47	b.d.l.	29.67	34.44	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	99.58
156	478	35.47	b.d.l.	30.34	34.97	b.d.l.	b.d.l.	0.03	b.d.l.	b.d.l.	100.81
156	479	35.56	0.04	30.38	35.01	0.01	b.d.l.	0.02	b.d.l.	b.d.l.	101.02
168	497	35.64	b.d.l.	30.59	34.85	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	101.08
168	498	35.46	b.d.l.	30.7	34.93	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	101.09
168	503	35.43	b.d.l.	30.55	34.91	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.91
168	504	35.59	b.d.l.	30.34	35.04	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.98
168	505	35.35	0.01	30.66	34.97	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	101
211	511	35.03	b.d.l.	29.63	34.39	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.05
211	512	35.28	b.d.l.	29.64	34.57	b.d.l.	b.d.l.	0.04	b.d.l.	0	99.53
211	515	35.08	0	29.98	34.8	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0	99.86
211	516	34.86	b.d.l.	30.29	34.21	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	99.36
211	517	35.2	0	30.06	34.71	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0	99.98
211	518	35.16	b.d.l.	29.6	34.26	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.02
211	519	35.13	0.02	29.92	34.41	b.d.l.	b.d.l.	0.02	b.d.l.	b.d.l.	99.5
211	520	35.12	b.d.l.	30.09	34.79	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0	100
211	521	35.19	b.d.l.	30.11	34.61	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	99.91
211	527	35.34	b.d.l.	30.18	34.93	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.45
211	528	35.27	b.d.l.	30.13	34.93	b.d.l.	b.d.l.	0.02	b.d.l.	b.d.l.	100.35
211	529	35.25	0.01	30.43	34.85	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0	100.54
211	530	35.09	b.d.l.	30.21	34.88	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	100.18
211	533	35.02	0.01	30.35	34.92	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.31
211	534	35.48	0.02	30.39	34.86	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.75
225	541	35.35	b.d.l.	30.39	34.97	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.71

Appendix D. Table D.1. (Continued)

Sample	Line	S (wt%)	Sb (wt%)	Fe (wt%)	Cu (wt%)	As (wt%)	Zn (wt%)	Ag (wt%)	Hg (wt%)	Ni (wt%)	Total
225	542	35.43	b.d.l.	30.4	35.14	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.01	100.98
225	543	35.41	b.d.l.	30.57	35.06	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	101.06
225	544	33.6	0	28.09	33.29	b.d.l.	b.d.l.	0	b.d.l.	0.01	95
225	545	35.38	b.d.l.	29.9	34.66	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	99.94
225	546	35.35	b.d.l.	29.22	34.31	b.d.l.	b.d.l.	0.01	b.d.l.	0	98.9
225	547	34.86	0.02	29.17	34.89	b.d.l.	b.d.l.	0.04	b.d.l.	b.d.l.	98.98
225	548	34.95	0.02	29.02	34.22	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0	98.2
225	549	35.24	b.d.l.	29.83	34.59	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.66
225	550	35.21	0.02	29.85	34.62	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	99.7
225	551	35.46	0	29.53	34.46	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.44
225	552	35.35	b.d.l.	29.78	34.61	b.d.l.	b.d.l.	0	b.d.l.	b.d.l.	99.74
225	553	35.48	b.d.l.	29.66	34.42	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.56
225	554	35.39	b.d.l.	29.22	34.25	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	98.86
225	555	35.32	b.d.l.	29.65	34.38	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	99.36
225	556	34.46	0.04	29.14	34.12	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	97.75
225	557	35.32	0	30.37	35.09	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0	100.78

Table D.2. Pyrite EPMA data.

Sample	Line	S (wt%)	Sb (wt%)	Fe (wt%)	Cu (wt%)	As (wt%)	Zn (wt%)	Ag (wt%)	Hg (wt%)	Ni (wt%)	Total
003A	162	53.31	0.02	45.19	0.02	b.d.l.	0.02	b.d.l.	b.d.l.	0.27	98.82
003A	167	53.52	b.d.l.	45.40	0.08	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.10	99.09
003A	181	53.21	b.d.l.	44.03	0.05	b.d.l.	b.d.l.	0.01	b.d.l.	0.04	97.33
003A	182	53.37	b.d.l.	44.78	0.08	b.d.l.	b.d.l.	b.d.l.	0.02	0.07	98.32
003B	191	53.29	0.01	45.94	0.01	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.00	99.25
003B	192	53.43	0.00	45.90	b.d.l.	b.d.l.	b.d.l.	0.01	b.d.l.	0.00	99.36
003B	193	53.38	b.d.l.	46.24	0.02	b.d.l.	0.02	b.d.l.	0.01	0.00	99.67
003B	194	53.13	b.d.l.	46.24	0.01	b.d.l.	0.00	0.01	b.d.l.	0.01	99.40
003B	195	53.23	b.d.l.	45.61	0.03	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.06	98.94
003B	196	53.32	0.03	45.06	0.03	b.d.l.	b.d.l.	0.00	b.d.l.	0.69	99.13
003B	197	50.24	0.02	41.39	6.20	b.d.l.	b.d.l.	0.00	b.d.l.	1.27	99.13
003B	198	53.22	0.01	44.69	0.02	b.d.l.	b.d.l.	0.00	0.00	0.33	98.28
003B	205	53.08	b.d.l.	45.02	0.14	b.d.l.	b.d.l.	b.d.l.	0.01	0.84	99.09
003B	206	53.19	0.04	45.12	0.02	b.d.l.	b.d.l.	0.01	b.d.l.	1.00	99.38
003B	207	53.06	0.02	43.71	0.01	b.d.l.	b.d.l.	b.d.l.	0.03	2.01	98.85
34	233	53.32	0.00	45.75	0.05	b.d.l.	b.d.l.	b.d.l.	0.02	0.58	99.72
34	234	53.29	b.d.l.	45.20	0.03	0.00	b.d.l.	0.00	b.d.l.	1.15	99.67
34	235	53.31	0.01	45.52	0.22	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.69	99.75
34	239	53.62	b.d.l.	45.75	0.02	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.69	100.08
34	240	53.18	b.d.l.	45.86	0.30	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.18	99.52
34	241	53.05	b.d.l.	45.88	0.27	b.d.l.	b.d.l.	b.d.l.	0.01	0.11	99.32
34	242	53.24	b.d.l.	45.89	0.12	b.d.l.	b.d.l.	0.01	0.03	0.17	99.46
34	245	53.39	b.d.l.	46.17	0.01	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.23	99.79
34	246	53.45	0.01	45.77	0.02	b.d.l.	0.04	0.00	0.00	0.51	99.80
34	247	53.33	0.01	45.63	0.09	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.20	99.26

Appendix D. Table D.2. (Continued)

Sample	Line	S (wt%)	Sb (wt%)	Fe (wt%)	Cu (wt%)	As (wt%)	Zn (wt%)	Ag (wt%)	Hg (wt%)	Ni (wt%)	Total
34	248	53.36	b.d.l.	46.28	0.08	b.d.l.	b.d.l.	b.d.l.	0.02	0.10	99.83
34	249	53.50	b.d.l.	46.49	0.01	b.d.l.	0.00	b.d.l.	b.d.l.	0.00	100.01
032A	277	53.50	b.d.l.	46.54	0.02	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.05	100.11
032A	278	53.34	b.d.l.	46.44	0.01	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.05	99.85
032A	279	53.35	0.01	45.62	0.03	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.10	99.11
032A	280	53.12	b.d.l.	45.96	0.01	b.d.l.	0.01	b.d.l.	b.d.l.	0.10	99.20
032A	285	53.23	0.04	45.29	0.01	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.74	99.30
032A	286	53.14	b.d.l.	44.97	0.02	b.d.l.	b.d.l.	0.00	b.d.l.	0.91	99.03
032A	287	53.14	0.02	44.89	0.02	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.04	98.11
032A	288	53.27	0.03	44.07	0.01	b.d.l.	b.d.l.	0.00	b.d.l.	0.00	97.40
032A	294	52.76	b.d.l.	44.15	0.04	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.00	96.96
032A	295	53.35	0.00	45.93	0.04	b.d.l.	0.01	b.d.l.	b.d.l.	0.05	99.39
052A	314	53.28	b.d.l.	45.23	0.03	b.d.l.	0.00	b.d.l.	b.d.l.	1.12	99.65
052A	315	53.36	0.02	45.53	0.01	b.d.l.	0.01	0.01	b.d.l.	1.15	100.08
052A	316	53.08	b.d.l.	44.36	0.01	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.08	97.53
052A	317	53.22	b.d.l.	45.27	0.05	b.d.l.	b.d.l.	0.01	b.d.l.	0.18	98.73
052A	324	53.29	b.d.l.	45.31	0.00	b.d.l.	0.01	0.00	b.d.l.	0.06	98.67
052A	325	53.34	b.d.l.	45.61	b.d.l.	b.d.l.	b.d.l.	0.02	b.d.l.	0.07	99.04
052B	343	53.67	b.d.l.	46.62	0.28	b.d.l.	0.02	0.01	b.d.l.	0.00	100.60
052B	344	53.43	b.d.l.	46.51	0.27	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.21
56	345	53.72	b.d.l.	46.11	0.01	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.19	100.03
56	346	53.41	b.d.l.	45.51	0.02	b.d.l.	0.00	0.00	b.d.l.	0.65	99.59
56	351	53.45	b.d.l.	46.41	b.d.l.	b.d.l.	b.d.l.	0.00	b.d.l.	0.21	100.07
56	352	53.26	b.d.l.	46.67	0.01	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.94
56	353	53.46	b.d.l.	46.53	b.d.l.	b.d.l.	0.00	b.d.l.	b.d.l.	b.d.l.	99.99
56	354	53.40	0.03	46.26	0.00	b.d.l.	b.d.l.	0.01	b.d.l.	0.00	99.71

Appendix D. Table D.2. (Continued)

Sample	Line	S (wt%)	Sb (wt%)	Fe (wt%)	Cu (wt%)	As (wt%)	Zn (wt%)	Ag (wt%)	Hg (wt%)	Ni (wt%)	Total
56	355	53.22	b.d.l.	46.55	b.d.l.	b.d.l.	0.02	0.00	b.d.l.	b.d.l.	99.79
56	356	53.33	0.01	46.58	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.92
56	357	52.79	b.d.l.	45.93	0.05	b.d.l.	b.d.l.	0.03	b.d.l.	b.d.l.	98.80
56	358	53.46	b.d.l.	46.16	0.04	b.d.l.	b.d.l.	0.01	b.d.l.	0.03	99.69
56	359	53.32	b.d.l.	46.65	b.d.l.	b.d.l.	b.d.l.	0.01	b.d.l.	0.04	100.02
56	360	53.48	b.d.l.	46.75	0.00	b.d.l.	0.02	0.01	b.d.l.	b.d.l.	100.26
56	361	53.33	b.d.l.	45.94	0.01	b.d.l.	0.01	b.d.l.	b.d.l.	0.01	99.30
56	362	53.37	b.d.l.	46.04	0.01	b.d.l.	b.d.l.	0.01	b.d.l.	0.02	99.44
057A	368	53.31	0.00	46.37	0.00	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	99.69
057A	369	53.19	b.d.l.	46.35	0.00	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.54
057A	370	53.57	b.d.l.	46.75	0.00	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.01	100.34
057A	371	53.11	b.d.l.	46.48	0.01	b.d.l.	0.00	0.01	b.d.l.	0.00	99.63
057A	372	53.37	b.d.l.	46.38	b.d.l.	b.d.l.	0.00	0.00	b.d.l.	0.01	99.77
057A	373	53.55	b.d.l.	46.44	0.00	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.04	100.03
057A	374	53.13	b.d.l.	45.77	0.01	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.07	98.98
057A	375	53.15	0.02	46.63	b.d.l.	b.d.l.	b.d.l.	0.02	b.d.l.	b.d.l.	99.83
057A	376	53.36	b.d.l.	46.59	b.d.l.	b.d.l.	b.d.l.	0.00	b.d.l.	0.00	99.95
057A	377	53.21	b.d.l.	46.09	0.02	b.d.l.	0.01	0.03	b.d.l.	0.00	99.37
057A	378	53.26	0.02	46.47	b.d.l.	b.d.l.	b.d.l.	0.02	b.d.l.	b.d.l.	99.77
057A	379	53.40	0.00	46.37	0.01	b.d.l.	b.d.l.	0.02	b.d.l.	0.02	99.82
057A	380	53.13	0.03	45.90	0.00	b.d.l.	0.01	0.01	b.d.l.	0.01	99.09
057A	381	53.45	b.d.l.	46.44	0.01	b.d.l.	0.03	b.d.l.	b.d.l.	0.00	99.93
057A	382	53.43	b.d.l.	46.55	0.01	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.99
057A	383	52.46	b.d.l.	45.94	0.01	b.d.l.	b.d.l.	0.00	b.d.l.	b.d.l.	98.41
057A	384	53.43	0.05	46.56	0.01	b.d.l.	b.d.l.	0.00	b.d.l.	0.00	100.05
057A	385	53.23	0.02	46.53	b.d.l.	b.d.l.	b.d.l.	0.01	b.d.l.	0.01	99.80

Appendix D. Table D.2. (Continued)

Sample	Line	S (wt%)	Sb (wt%)	Fe (wt%)	Cu (wt%)	As (wt%)	Zn (wt%)	Ag (wt%)	Hg (wt%)	Ni (wt%)	Total
057A	386	53.22	b.d.l.	46.50	0.01	b.d.l.	0.03	0.02	b.d.l.	b.d.l.	99.78
057A	387	53.36	b.d.l.	46.47	0.02	b.d.l.	b.d.l.	0.00	b.d.l.	b.d.l.	99.85
129	396	52.75	b.d.l.	43.27	0.46	b.d.l.	b.d.l.	0.00	b.d.l.	0.11	96.60
129	397	53.28	b.d.l.	45.20	0.02	b.d.l.	b.d.l.	0.00	b.d.l.	0.28	98.79
129	398	53.10	b.d.l.	45.16	1.10	b.d.l.	b.d.l.	0.01	b.d.l.	0.06	99.43
129	399	53.21	b.d.l.	45.86	0.02	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	99.10
129	400	53.13	b.d.l.	45.03	0.01	b.d.l.	b.d.l.	0.00	b.d.l.	0.03	98.20
129	401	52.62	0.03	43.55	0.40	b.d.l.	b.d.l.	0.01	b.d.l.	0.68	97.31
129	402	52.64	b.d.l.	43.25	1.68	b.d.l.	b.d.l.	0.00	b.d.l.	0.52	98.09
129	403	53.34	b.d.l.	45.01	0.33	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.12	98.79
129	404	52.44	b.d.l.	42.80	2.78	b.d.l.	b.d.l.	0.00	b.d.l.	0.52	98.54
129	405	53.48	b.d.l.	45.42	0.28	b.d.l.	0.00	b.d.l.	b.d.l.	0.23	99.41
129	406	53.41	b.d.l.	45.99	0.09	b.d.l.	b.d.l.	0.00	b.d.l.	0.02	99.51
129	407	53.29	0.01	45.08	b.d.l.	b.d.l.	b.d.l.	0.01	b.d.l.	0.22	98.61
168	480	54.04	b.d.l.	46.47	0.00	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.51
168	481	53.97	b.d.l.	46.42	0.01	b.d.l.	b.d.l.	0.01	b.d.l.	0.00	100.41
168	482	53.80	b.d.l.	46.54	0.00	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.01	100.35
168	483	53.82	0.01	46.74	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.58
168	484	53.81	b.d.l.	46.73	0.00	b.d.l.	0.00	b.d.l.	b.d.l.	b.d.l.	100.55
168	485	54.01	b.d.l.	46.77	0.02	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	100.80
168	486	53.78	0.02	46.66	0.02	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	100.49
168	487	53.89	0.03	46.78	0.01	b.d.l.	b.d.l.	0.02	b.d.l.	b.d.l.	100.73
168	488	54.06	b.d.l.	46.42	0.02	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.01	100.50
168	489	53.97	0.01	46.63	0.00	b.d.l.	0.03	b.d.l.	b.d.l.	0.03	100.67
168	490	54.03	b.d.l.	46.60	0.02	b.d.l.	b.d.l.	0.01	b.d.l.	0.01	100.67
168	491	54.04	b.d.l.	46.64	0.01	b.d.l.	0.01	b.d.l.	b.d.l.	0.00	100.70

Appendix D. Table D.2. (Continued)

Sample	Line	S (wt%)	Sb (wt%)	Fe (wt%)	Cu (wt%)	As (wt%)	Zn (wt%)	Ag (wt%)	Hg (wt%)	Ni (wt%)	Total
168	492	53.93	b.d.l.	46.72	b.d.l.	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.01	100.66
168	493	53.88	0.00	46.75	0.03	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.01	100.66
168	494	53.69	b.d.l.	45.44	0.01	b.d.l.	0.01	0.01	b.d.l.	0.01	99.18
168	495	53.67	0.00	45.91	0.02	b.d.l.	0.03	b.d.l.	b.d.l.	0.01	99.64
168	496	53.80	b.d.l.	46.40	0.16	b.d.l.	b.d.l.	0.01	b.d.l.	0.00	100.38
168	499	53.88	0.01	46.17	0.03	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.00	100.09
168	500	53.74	b.d.l.	45.08	0.03	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.91	99.76
168	501	0.00	0.01	66.71	0.00	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.04	66.76
168	502	0.00	b.d.l.	65.23	b.d.l.	b.d.l.	0.01	0.01	b.d.l.	0.06	65.31
168	506	54.11	0.00	46.65	0.01	b.d.l.	0.02	0.00	b.d.l.	0.01	100.81
168	507	0.00	0.02	63.29	b.d.l.	b.d.l.	0.10	0.00	b.d.l.	0.03	63.44
168	508	53.88	0.01	46.64	b.d.l.	0.00	b.d.l.	b.d.l.	b.d.l.	0.01	100.54
211	509	53.26	0.00	45.86	0.13	b.d.l.	0.01	0.02	b.d.l.	0.06	99.33
211	510	53.10	b.d.l.	45.99	0.09	b.d.l.	b.d.l.	0.00	b.d.l.	b.d.l.	99.18
225	537	53.95	b.d.l.	46.19	0.15	b.d.l.	0.00	b.d.l.	b.d.l.	0.44	100.72
225	538	54.00	b.d.l.	45.56	0.29	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.01	99.87
225	539	53.97	b.d.l.	44.70	0.01	b.d.l.	b.d.l.	b.d.l.	b.d.l.	0.05	98.73
225	540	34.70	b.d.l.	29.55	34.95	b.d.l.	b.d.l.	0.01	b.d.l.	b.d.l.	99.21

b.d.l. – below detection limit.