Appendix 6-A

Metal Leaching and Acid Rock Drainage Characterization

HARPER CREEK PROJECT

Application for an Environmental Assessment Certificate /
Environmental Impact Statement

Metal Leaching and Acid Rock Drainage Characterization Harper Creek Project

Prepared for

Harper Creek Mining Corp.

Prepared by

SRK Consulting (Canada) Inc.
1CY003.001
January 2015

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Harper Creek Project

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Executive Summary

Harper Creek Mining Corporation (HCMC) proposes to construct and operate an open pit copper, gold and silver mine near Vavenby, British Columbia (BC). This report describes metal leaching and acid rock drainage (ML/ARD) potential characterization for the purposes of the Application for an Environmental Assessment (EA) Certificate under the British Columbia Environmental Assessment Act (BC EAA) and the Environmental Impact Statement (EIS) under the Canadian Environmental Assessment Act (CEAA) in accordance with the Approved Project Application Information Requirements (AIR) issued October 21, 2011.

The host rocks of the mineralization at Harper Creek is a succession consisting of orthogneiss, metasediments, metavolcanics and metavolcanic clastics. It is characterized as “polymetallic volcanogenic sulphide” deposit of which the closest analogous mineral deposit class is Kuroko-type. However, Harper Creek lacks abundant massive sulphide mineralization typical of Kurko-type deposits.

The two objectives of ML/ARD characterization are (1) to provide waste management criteria; and (2) predict contact water chemistry (referred to as “source terms”) for input into water quality assessments for the project.

ML/ARD potential of rock, tailings, overburden and quarry materials for the project was evaluated through conventional procedures including acid-base accounting, trace element analysis, mineralogy, shake flask extractions, humidity cells, subaerial and subaqueous columns and on-site kinetic tests. Some test procedures were adapted to address site-specific questions. These tests indicated that some components of all materials except the quarry have ML/ARD potential. ML potential primarily exists for copper, selenium and zinc.

For rock (waste rock and low grade ore), ARD potential is definable at the bench scale and can be segregated into non-PAG and PAG components for separate management during mining. PAG rock is not usefully correlated to rock type but segregation by ARD potential also results in segregation by sulphur content.

Rougher tailings are non-PAG whereas cleaner tailings are PAG. Overburden from outside the pit limits is non-PAG due to low sulphur content. Within the pit footprint, near surface overburden is non-PAG but weathered bedrock may be PAG.

Source terms were developed for 20 site features. The source terms group into three general types: subaerial, flushed or inundated stockpiled rock and subaqueous. The overall source term methodology considers rates of weathering under disposal conditions and solubility of weathering products. Weathering rates were derived from laboratory and field tests. Solubility limits were derived from thermodynamic calculations, project-specific tests and a database of drainage chemistry data from analogous mine sites.

Uncertainty in source terms was considered using two cases: Expected and Reasonable Upper Limit which varied the rates of weathering and solubility of weathering products. The latter was based mainly on consideration of the uncertainty in pore water pH which is expected to be controlled by CO_2 partial pressure. The Reasonable Upper Limit Case assumes that pHs as low as 7.6 could be reached based on assessment of analog data. pHs below this level are considered to have a less than 1 in 20 chance of occurring.

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List of Abbreviations

Term	Definition
ABA	Acid-Base Accounting, a type of static test.
AIR	Approved Project Application Information Requirements
ANFO	Ammonium Nitrate Fuel Oil (5.7% fuel oil by weight and 94.3% ammonium nitrate)
AP	Acid Potential, calculated from Sulphur as Sulphide x 31.25 and expressed as parts per thousand calcium carbonate (kg $CaCO_3$/t)
ARD	Acid Rock Drainage
BC EAA	British Columbia Environmental Assessment Act
CGM	Conceptual Geochemical Model
CT	Cleaner Tailings
EA	Environmental Assessment
EA	Environmental Assessment
EAO	BC Environmental Assessment Office
EIS	Environmental Impact Statement
HCMC	Harper Creek Mining Corporation
ICP-MS	Inductively coupled plasma mass spectrometry, a laboratory technique to measure metals and some non-metals at low concentrations
KP	Knight Píesold
MINFILE	On-line database of mineral occurrences in British Columbia
ML	Metal Leaching
MWMP	Meteoric Water Mobility Procedure, a test for leachable elements using water.
NNP	Net Neutralization Potential
non-PAG	Not potentially ARD generating, or acid consuming.
NP	Neutralization Potential
NP/AP	Ratio of neutralization potential to acid potential
NPR	Neutralization Potential Ratio, the same as NP/AP
ORP	Oxidation-Reduction Potential
PAG	Potentially ARD Generating
QC	Quality Control
Rotl	Rougher Tailings
SEM, SEM-EDX	Scanning electron microscope, a method used to see very small objects and estimate their elemental composition
TIC	Total inorganic carbon, a measurement of carbonate.
TMF	Tailings Management Facility
UBCEOAS	University of BC Department of Earth Ocean and Atmospheric Scienc
XRD	X-ray diffraction, a method for determining types of minerals in a sample.

1 Introduction

1.1 Project Description

Harper Creek Mining Corporation (HCMC) proposes to construct and operate an open pit copper, gold and silver mine near Vavenby, British Columbia (BC). The proposed Harper Creek Mine Project (the Project) is expected to process nominally 70,000 tonnes of copper-gold-silver ore per day (25 million tonnes per year) to produce a copper concentrate with gold and silver by-products for export over an operating life of 28 years. Ore will be processed on site, with copper concentrate to be trucked from the mine site approximately 24 km along existing access roads to a rail load-out facility at Vavenby. The concentrate will be transported via the existing Canadian National Railway network to the existing Vancouver Wharves storage, handling and loading facilities located at the Port of Vancouver for shipment to overseas smelters.

The Project consists of an open pit mine, on-site processing facility, tailings management facility (TMF) (for tailings solids, subaqueous storage of PAG waste rock, and recycling of water for processing), waste rock and low grade ore stockpiles, overburden stockpiles, a temporary construction camp, ancillary facilities, mine haul roads, sewage and waste management facilities, a 24 km access road between the Project site and rail load-out facility, a 12 km Power Line connecting the Project site to the BC Hydro transmission line corridor in Vavenby, and a rail load-out facility located on private land owned by HCMC in Vavenby.

This report describes metal leaching and acid rock drainage (ML/ARD) potential characterization for the purposes of the Application for an Environmental Assessment (EA) Certificate under the British Columbia Environmental Assessment Act (BC EAA) and the Environmental Impact Statement (EIS) under the Canadian Environmental Assessment Act (CEAA) in accordance with the Approved Project Application Information Requirements (AIR) issued October 21, 2011.

1.2 Project Location

The Project is located in the Thompson-Nicola area of BC, approximately 150 km north of Kamloops along the Yellowhead #5 Highway. The Harper Creek Property Area covers a total of 42,636.48 ha, geographically centered at 51 º30'N and 119º48'W. The closest population centre is the unincorporated municipality of Vavenby located about 10 km northeast of the proposed mine.

1.3 Project Proponent

The Proponent of the Harper Creek Project is Harper Creek Mining Corp. (HCMC). HCMC is a wholly owned subsidiary of Yellowhead Mining Inc. (YMI). YMI was formed in 2005 as a private British Columbia company specifically to acquire, explore and, if feasible, develop the Harper Creek deposit. YMI is now a publicly owned BC mineral exploration and development company trading on the Toronto Stock Exchange (TSX) in Canada. HCMC's development strategy is to engineer, permit, finance, construct, and operate the Harper Creek Project.

1.4 Geological Setting

1.4.1 Property Geology

The following summary of the property geology is as reported by Christopher Naas, PGeo and contained in HCMC's Feasibility Study (Merit Consultants 2012).

The property is located within structurally complex low-grade metamorphic rocks of the Eagle Bay Assemblage which is part of the Kootenay Terrane on the western margin of the Omineca Belt. This assemblage hosts numerous small polymetallic massive sulphide deposits found mainly within in Devonian age felsic volcanic rocks. These deposits formed in an arc volcanic environment in response to eastward subduction of a paleo‐Pacific ocean.

The host rocks of the mineralization at Harper Creek is a succession consisting of orthogneiss, metasediments, metavolcanics and metavolcanic clastics. A property scale geology map is provided in Figure 1.

Drill core logging codes are shown in Table 1. The dominant rock types at the site are 7 (phyllites), 8 (schist with minimal quartz content) and 9 (Schist with quartz content and/or quartz eyes). While these are recognizable rock types, they cannot be correlated between drill holes due to structural polyphase deformation from thrust faulting, intrusion-related folding and faulting, and strike slip and normal faulting.

For the purpose of interpretation of the property geology and resource modelling, HCMC's geologists have recognized ten distinctive large-scale lithological packages (labelled A to H) that have been correlated between drill holes. Structurally, the packages strike roughly northeast-southwest and dip to the northwest and package A is structurally lowest. The Harper Creek Fault divides the site into two domains (east and west) which show some differences in the characteristics of the packages. Table 2 shows the lithological characteristics of each package. Detailed descriptions are provided by Merit Consultants (2012).

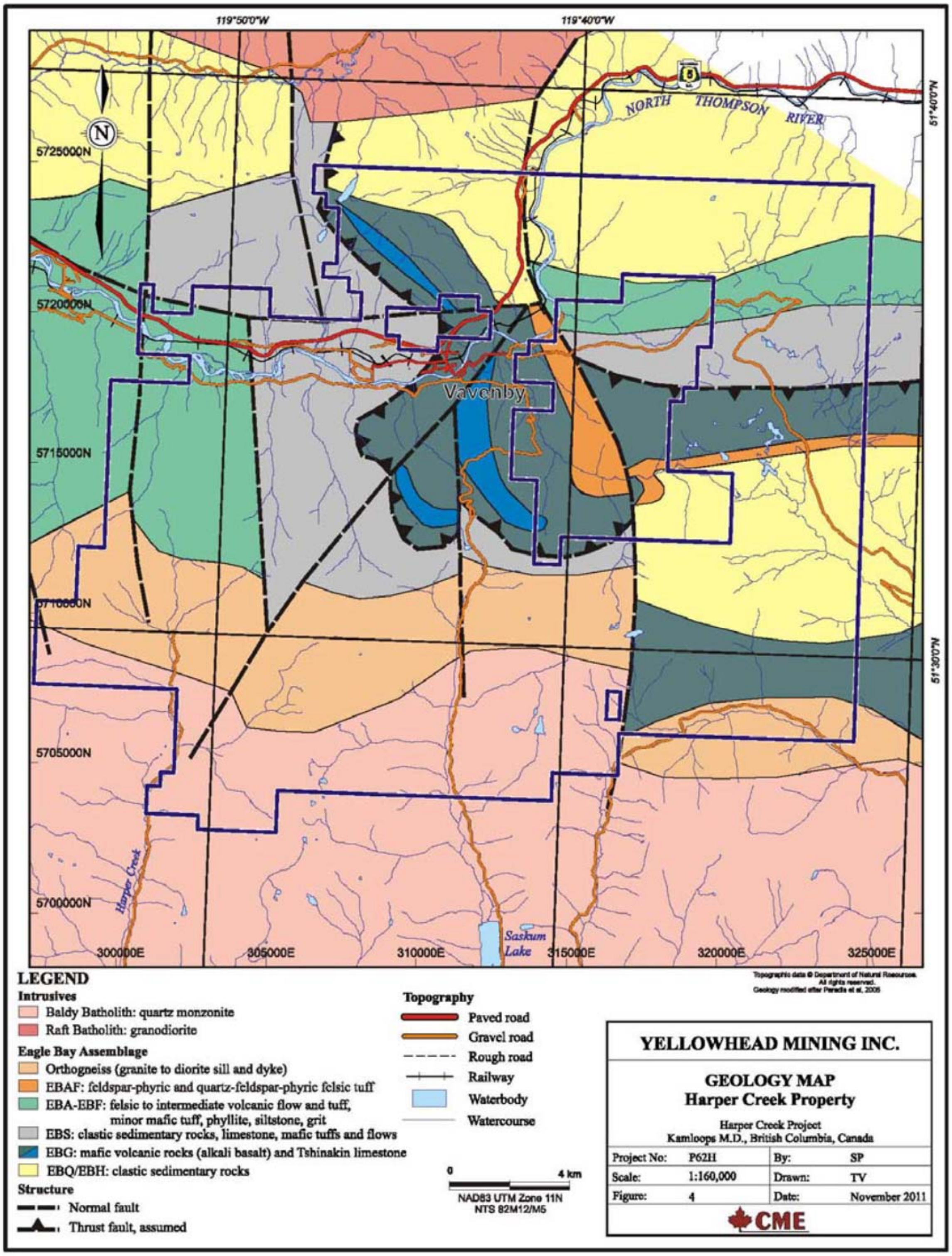

Figure 1: Property Scale Geology Map (Merit Consultants 2012).

Table 1: Lithological Logging Codes

Unit No.		Unit Name
1	1 a	Fault Zone
	1b	Fault Gouge
	1 c	Fault Breccia
	1 d	Fault Healed
	1 e	Fault Breccia & Healed
	1 f	Mylonite/Mylonite Zone
	1 g	Shear Zone
	1h	Graphitic
2	2a	Quartz dominant
	2b	Carbonate dominant
	2c	Quartz-Carbonate dominant
	2d	Sulphide Vein
	2e	Tourmaline Vein
	2f	Chlorite Vein
	2g	Biotite Vein
3	3a	Granodiorite
	3b	Hornblende-Biotite Granodiorite
	3c	Quartz-Monzonite
4	4a	Andesite
	4b	Rhyolite
	4c	Lamprophyre
5	5a	Agglomerate
	5b	Ignimbrite
	5c	Tuff
	5d	Lapilli-Tuff
6	6a	Conglomerate
	6b	Sandstone
	6c	Siltstone
7	7a	Graphitic
	7b	Sericite-Chlorite
	7c	Calcareous Chlorite-Sericite
	7d	Sericite-Chlorite-Quartz
8	8a	Sericite-Chlorite
	8b	Sericite-Chlorite-Fuchsite
	8c	Chlorite-Sericite Fragmental
9	9a	Sericite-Hornblende-Quartz-Feldspar
	9b	Sericite-Chlorite-Quartz
	9c	Sericite-Chlorite-Quartz-Feldspar
	9d	Sericite-Augen Quartz
	9e	Siliceous Chlorite-Sericite Quartz
10	10a	Orthogneiss
11	11a	Silica Alteration
	11b	Chlorite Alteration
12	12a	Sulphide-dominant, undivided
	12b	Magnetite-dominant
	12c	Pyrrhotite-dominant
	12d	Pyrite-dominant
	12e	Chalcopyrite-dominant

Source: HCMC

Table 2: Package Descriptions

West	Package ID East	Composition	Copper Mineralization
H	H	Mafic Polymictic Volcaniclastics 8c, 8a, 7c +/- 9a hornblende crystals, frequently calcareous, and deformed 7d	No
G	G	Graphitic Horizon, somewhat calcareous	No
Fb	Fb	Intermediate to Mafic Polymictic Volcaniclastics 8c, 8a, 7c +/- 9a hornblende crystals, somewhat calcareous	No
Fb	Fb	Intermediate to Mafic Polymictic Volcaniclastics 8c, 8a, 7c	Yes
	FD	Elements of D and Fa combined in the East Domain	Yes
Fa	Fa	Felsic to Intermediate Volcaniclastics: 9c, 8c & 8a	Yes
E		Graphitic Horizon: mixed 1 1a silicified+7a	Yes
D	D	Intermediate Volcaniclastics & Fragmentals, somewhat calcareous: dominated by 8c/7c	Remobilized
C		Graphitic Horizon	Remobilized
B	B	Sandy Sediment dominated: 9b+8a mafic sediments	Remobilized
A	A	Orthogneiss: 10a and associated border phases 9d and others	Remobilized

Source: HCMC

Table 2 shows that the packages have been assigned distinctive volcanic protoliths which are mainly various types of volcaniclastics. Graphitic horizons are suspected to be mudstones but also may be shear zones. Copper mineralization, which is mainly chalcopyrite and to a much lesser extent bornite occurs as wisps parallel to foliation, disseminations and fracture filling mainly in packages E and Fa with Fa carrying the highest grades. Pyrite and pyrrhotite occur throughout the sequence as fine disseminated grains, porphyroblasts (e.g. Package C and G) and lenses (e.g. up to 5 m thick in Package D).

Molybdenite is particularly noted in Package E. Other sulphides noted by Merit Consultants (2012) and the MINFILE description (082M 009) indicated the presence of trace levels of sphalerite, galena, arsenopyrite, tetrahedrite-tennanite and cubanite.

As shown, some packages are noted to be calcareous. Carbonate minerals include calcite, dolomite and iron-carbonates. Dolomite is specifically noted in Package Fb. The presence of iron carbonates is apparent by the abundant characteristic staining of old core by weathering.

1.4.2 Mineral Deposit Classification

The classification of mineral deposits is important because it allows a subject property to be compared with other sites and possible provides a source of analogous geochemical data which may be used to predict the performance of the site under study.

Merit Consultants (2012) indicated the deposit is a “polymetallic volcanogenic sulphide” deposit because the mineralization consists of iron and copper sulphides with accessory magnetite and the mineralization is generally conformable with the host rock stratigraphy. The MINFILE description (082M 009) (Ministry of Energy and Mines 2011) classifies the deposit as “Noranda/Kuroko massive sulphide Cu-Pb-Zn” though Harper Creek has very little massive sulphide mineralization with the exception of 5 m thick lenses in Package D. Harper Creek lacks the distinctive different types of sulphide mineralization often observed in Kuroko-type deposits including the presence of barite, sphalerite and galena. Important Kuroko-type deposits in B.C. include the nearby Samatosum Deposit, Britannia, Myra Falls and Eskay Creek. These deposits are all characterized by more complex massive sulphide mineralization than observed at Harper Creek.

It is concluded that there are no direct analogs for the Harper Creek deposit; however, based on the MINFILE classification of the Harper Creek deposit, Kuroko-type volcanogenic massive sulphide deposits are considered the nearest similar deposit type.

1.5 Harper Creek Mine Plan

The mine plan was developed using initial observations of the geological and geochemical setting and refined as geochemical data became available. The proposed final plan contains the following elements in the context of ML/ARD management (Figure 2):

- An open pit which will flood at closure addressing ML/ARD of pit walls at closure.
- A segregation plan for waste rock and low grade ore to separate potentially ARD generating (PAG) and non-PAG waste rock.
- Upland disposal of non-PAG waste rock in waste rock dumps designed to contain drainage.
- Use of non-PAG waste rock for construction purposes within the mine site.
- Disposal of PAG waste rock in submerged conditions in the tailings impoundment.
- Construction of the tailings dam using locally quarried non-PAG rock and glacial till.
- Conventional beached disposal of non-PAG rougher flotation tailings.
- Subaqueous disposal of sulphidic PAG cleaner tailings in the tailings impoundment.
- Contingency for water treatment.

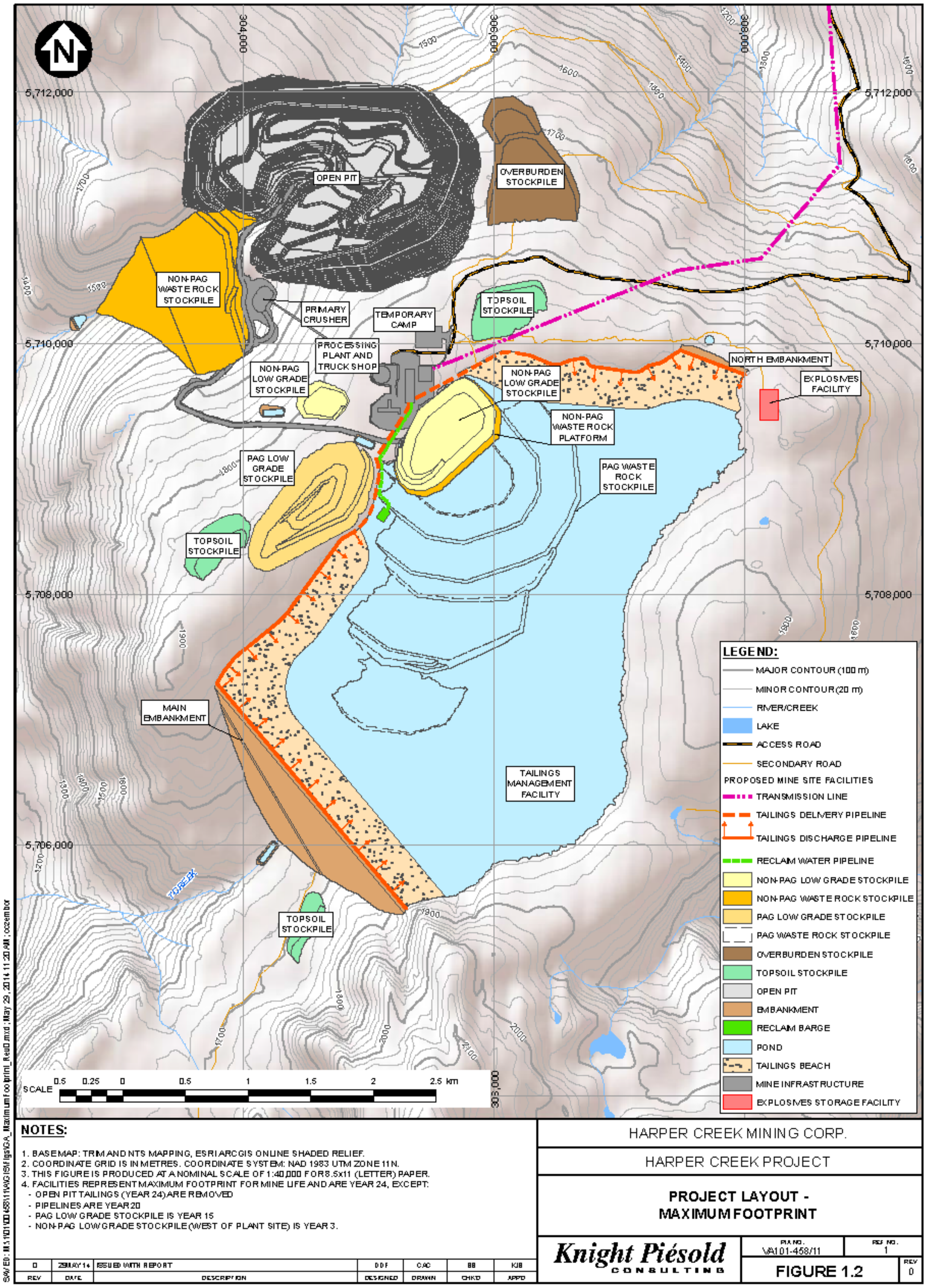

Figure 2: Final Configuration of Mine Site at Completion. Source: KP.

1.6 Report Structure

This report contains the following main headings:

- Chapter 2 (Characterization Design) explains the design of the geochemical testing program in the context of project data requirements.
- Chapter 3 (Characterization Design) summarizes the geochemical characterization methods.
- Chapter 4 (Results) describes the results of the geochemical characterization program.
- Chapter 5 (Development of Source Terms) provides water chemistry predictions for each facility at the site.

The separate ML/ARD Management Plan provides interpretation of the data to yield geochemical design criteria for the management of wastes and construction of the disposal facilities.

1.7 Acknowledgements

This report was prepared by SRK Consulting (Canada) Inc. (SRK) with input from the following organizations:

- Harper Creek Mining Corp. (HCMC) – Collection of rock samples, set up of field tests, waste rock block model and scheduling.
- Inspectorate – Trace element analysis.
- Maxxam Analytics – Geochemical testing of rock, tailings, and borrow source materials.
- Knight Píesold (KP) – Operation of field tests, collection of borrow and quarry samples, waste facility configuration, water balance data.

2 Background Review

2.1 Legislation, Regulations and Guidelines

The Project is subject to both provincial and federal EAs under the BC Environmental Assessment Act (2002) and Canadian Environmental Assessment Act 1992 (CEAA; 1992). The EA will undergo a coordinated review in accordance with the 2004 Canada-BC Agreement on Environmental Assessment Cooperation. The requirements for the EA are defined in the AIR for the Project, approved by the BC Environmental Assessment Office (EAO) on October 21, 2011. This baseline report has been prepared to support the submission of the Application/EIS.

The following regulatory guidance documents were referenced to develop the ML/ARD characterization plan.

- BC MEM. 1998a. Policy for Metal Leaching and Acid Rock Drainage at Minesites in British Columbia. July 1998.

- BC MEM. 1998b. Guidelines for Metal Leaching and Acid Rock Drainage at Minesites in British Columbia. August 1998.
- BC MOE. 2012a. Water and Air Baseline Monitoring Guidance Document for Mine Proponents and Operators. October 2012.
- Price, W.A. 2009. Prediction Manual for Drainage Chemistry from Sulphidic Materials. Report prepared for MEND Program. MEND Report 1.20.1.

2.2 Previous Environmental Studies

No previous work on ML/ARD Characterization has been published.

3 Methodology

3.1 Characterization Design

3.1.1 Introduction

The two objectives of ML/RD characterization are to:

- Provide design criteria for the operation and management of the various facilities containing geological materials at the site to the project engineers. These criteria can include segregation criteria to address ARD potential, criteria to define exposure times for reaction materials, and recommendations for construction of facilities such as placement methods; and
- Predict the chemistry of water coming into contact with geological materials as “source terms” for inputs into water quality models for the site.

The underlying basis for the design of the program is the development of Conceptual Geochemical Models (CGMs) which capture the expected geochemical performance of each Project component for which a source term is required. CGMs frame the geochemical questions that need to be answered for each component and therefore focus the characterization program by selection of appropriate methodologies for sample collection, testing and data interpretation. The following sections describe the CGMs.

3.1.2 Conceptual Geochemical Models

Overall

Review of the geological setting indicates the following general observations on geochemical performance of wastes and facilities at the site:

- Both pyrite and pyrrhotite occur throughout the host rocks implying that at least potential acid generation with associated metal leaching is a consideration for the project.
- Other than the gangue sulphide minerals, the dominant base metal sulphide is chalcopyrite indicating that copper leaching is a potential concern. The absence of abundant sulphides of

lead, zinc, arsenic and antimony, which are often associated with volcanogenic sulphide deposits, indicates that neutral pH leaching of these elements may not be important.

- Locally massive sulphide horizons are present indicating potential for acid rock drainage.
- The deposit does not have a gossan and no naturally acidic seeps have been encountered in the area.
- Some of the host rocks are described as calcareous which can be expected to delay or prevent acid rock drainage. Significant (decades) delay in onset of ARD may be anticipated.
- Carbonate mineralogy includes calcite, dolomite and iron carbonates, which influence the application and interpretation of standard acid-base accounting procedures.
- The mineral deposit model indicates that sulphide mineralization is stratiform and therefore that sulphide content may occur in predictable correlated zones.

These features indicate that ML/ARD potential needs to be considered for the Project. ARD is a function of the coupled processes of iron sulphide oxidation to generate acid, and neutralization of the resulting acidity by carbonate minerals. The tendency for ARD to actually form depends on the balance between the amounts of sulphide minerals and carbonates, and the disposal conditions. If amounts of sulphide minerals exceed the carbonate minerals, ARD may develop at some future time if the disposal conditions allow the sulphide minerals to oxidize readily.

ML refers to general processes of leaching of rock components (which include metals and other regulated elements and ions) into contact waters at levels that could result in a concern for receiving waters if not managed appropriately. All rocks and the related mines wastes contain regulated substances but the abundance and mineralogical occurrence of them ultimately determines whether leaching processes are significant.

ML potential at the Project has a strong linkage to the processes that result in ARD. Sulphide minerals are often the principle sources for heavy trace elements (for example, copper) because oxidation of sulphide minerals converts these elements to water soluble forms and the acidic conditions can increase solubility in some cases. Neutralization of the acidity by reaction with carbonates lowers the concentrations of some of these elements (for example, copper) in contact waters by forming precipitates while others can persist (for example, cadmium and zinc). Dissolution of carbonates can add to leaching of metals depending on the types of carbonate minerals.

ML potential can also result from simple dissolution of soluble rock components without acid generation but these processes are not expected to be significant for the Project. Examples would include dissolution of calcium sulphate minerals to leach sulphate, and dissolution of fluoride minerals to leach fluoride.

The following sections indicate the CGMs for specific site facilities and material types.

Non-PAG Waste Rock

Non-PAG waste rock will be placed in dedicated waste rock storage areas and used for fill inside the site. It will originate by mining in the open pit and quarrying. By definition, this waste rock will be segregated from PAG waste rock to result in rock that will not weather to result in acidic drainage; however, some PAG rock can be expected to be incorporated in non-PAG waste rock due to inefficiencies in segregation and operational upsets. The following components of the CGM have been identified:

- Weathering will occur under well-oxygenated conditions with movement of oxygen into the facilities driven by diffusive, convective and advective processes.
- Sulphide minerals will weather to leach acidity, sulphate and elements contained in the sulphides which will include iron, heavy metals (including iron and copper) and other heavy elements. The latter will include elements contained within the structure of pyrite.
- Dissolved bicarbonate and carbonate minerals will neutralize acidity adding Ca, Mg, Fe and Mn to solution.
- Non-acidic conditions are expected to be dominant throughout the facilities except in the immediate micro-environments adjacent to oxidizing sulphide grains and in larger scale environments where unsegregated PAG materials are present. Excess of neutralization potential results in non-acidic drainage but variations in pH can be expected to occur. The expected range of pH is 7 to 8.5 representing the effect of variable pore gas carbon dioxide partial pressures.
- Solubility of leached components is constrained by the formation of specific secondary minerals (e.g. gypsum for sulphate, iron oxyhydroxides for iron and copper oxides for copper) and sorptive processes (eg. adsorption of manganese to iron oxyhydroxides). Under neutral to basic conditions, oxyanions are expected to be more mobile than cations. Variations in pH will affect solubility of most metals and heavy elements.

Tailings Facility

The tailings facility will contain three main waste types: PAG waste rock, PAG cleaner tailings and non-PAG rougher tailings which will be disposed permanently or temporarily in subaqueous conditions and subaerial conditions. These tailings will be carried to the pond by process water.

PAG waste rock will initially be placed sub-aerially and will weather under dominantly non-acidic conditions. As the process pond rises, PAG waste rock will be inundated resulting in dissolution of accumulated secondary minerals formed under exposed conditions. Dissolution will occur initially under oxidation conditions until concentrations in the rock pore spaces reach levels constrained by mineral solubility. Displacement of pore water will result in a loading source to the pond. In the long term, inundated waste rock will be exposed to low oxygen concentrations controlled by advective flux of water through the rock. This could result in dissolution of secondary iron oxides formed under oxidizing conditions and resultant release of sorbed trace elements though in general the limited exposure to oxidizing conditions is not expected to result in significant accumulation of oxidation products. Reductive dissolution is also a possibility but is expected to be limited due to the short period of oxidation before flooding.

Cleaner tailings deposited into the process pond result in immediate and permanent saturation. Under these conditions, minimal weathering of the tailings is expected. Reductive dissolution effects are expected to be minimal when fresh ore is being processed due to the limited formation of secondary minerals in ore prior to processing. Processing of oxidized low grade ore at the end of mine life could result in secondary minerals (primarily hydrous ferric oxides) in the tailings. Residual organic process reagents could facilitate reductive dissolution of these secondary minerals.

Rougher tailings will be disposed of on the tailings beaches, some of which will remain partially as an unsaturated wedge against the dam. A large part of the rougher tailings will become saturated as the phreatic surface in the facility rises. Oxidation rates in well and fully saturated tailings near and below the phreatic surface will be very low. If oxidation takes place, it will occur as oxygen penetrates the tailings mass due to diffusion. The rate of diffusion of oxygen will be controlled by the physical characteristics of the tailings, the degree of saturation and the rate of oxidation. Oxidation can be expected to be most intense at the surface and processes will be broadly similar to weathering of waste rock. The depth of the weathering front in the tailings will depend on the sulphide content of the tailings. Lower sulphide content is expected to result in greater oxygen penetration assuming that oxidation rates correlate with sulphur content. As with waste rock, reductive dissolution of oxidation products is also expected to be limited.

The tailings dams will be constructed from non-PAG pit rock, overburden and non-PAG quarry rock. These components will be partly unsaturated. CGM's are described in other sections.

Chemical loadings in process water will be derived from process chemicals, leaching of secondary minerals formed in ore prior to processing and oxidation of sulphides occurring during processing. Oxidation of pyrrhotite during processing may result in formation of thiosalts. Processing of stockpiled low grade ore at the conclusion of mining can be expected to result in greater chemical loads in the process than when processing freshly mined ore.

Overburden and Soil

Overburden will consist of surficial material which is predominantly glacial till stripped from the pit area and local borrow pits throughout the site. These materials are expected to consist of transported and locally derived weathered rock. Where locally derived rock is present, weathering and leaching will occur under the same processes as described for non-PAG rock.

Contact water characteristics are expected to be controlled mainly by secondary minerals formed by weathering over geological time.

Open Pit

Operational

Water chemistry of the operational pit sump will be a combination of inflows from groundwater, direct precipitation and contact water flow over broken rock on benches and the pit walls. Non-PAG and PAG walls are expected to be non-acidic during operations with greatest loadings coming from shattered bedrock on benches and less load from walls. Broken rock will weather and leach with the same processes as indicated for waste rock.

Closure

During closure, flooding of the pit will occur resulting in submergence of walls. As the water level rises, oxidation of flooding walls will be reduced but any residual oxidation products will be flushed from the walls and contribute to total load in the pit lake. High walls remaining unsubmerged after final flooding will continue to contribute loadings to the discharging pit lake.

Nitrogen Model

Conventional ammonium nitrate fuel oil (ANFO) based explosives will be used for blasting in the pit. Incomplete combustion will result in explosives residuals which contribute to nitrogen forms (nitrate, nitrite and ammonia) in waters contacting blasted rock including waste rock, construction fill, ore and pit walls.

3.1.3 Characterization Program Design

The characterization program was designed using the mine facilities as a check-list and incorporating data needs indicated by the CGM's. Twenty-five different mine component/facilities that could be sources of chemical loads and other parameters to surface and groundwater were identified during development of the mine plan for the site. Variants within the sources (for example, sub-aerial, flushing and submerged for PAG rock) resulted in 34 individual source terms for input into the overall site water quality model. Three sources were eliminated during the planning process.

For each source term, design criteria requirements and water chemistry inputs were identified and used to determine testwork design components.

3.2 Characterization Methods

3.2.1 Sample Acquisition Methods

Waste Rock

Waste rock sampling specifically for geochemical purposes occurred in two phases.

Phase 1 was designed to provide an initial assessment of the performance of analytical methods, correlation of geochemical characteristics to rock types and downhole variation in geochemical characterization. Sample selection occurred in February 2011 from two diamond drill holes (HC10-76 and BH10-77) determined based on completion of lithological re-coding by HCMC and availability of multi-element ICP data that included sulphur analyses. The latter was sought to allow evaluation of analytical surrogates for acid-base accounting parameters in the exploration database.

SRK selected 40 samples from lithologies 5, 7, 8, 9 and 11 to characterize waste materials occurring over intervals of about 6 m and representing a range of sulphur concentrations. Each sample was a contiguous composite of shorter (typically 2 m) intervals sampled for exploration purposes. The samples were recovered from the pulp archive of previously tested samples.

Interpretation of the Phase 1 data set indicated that ARD potential was variable and that a block model would be needed to quantify amounts of potentially ARD generating (PAG) and non-PAG wastes. Phase 1 results indicated the block model could be built using analytical surrogates for ABA parameters in the exploration database. These results are described in subsequent sections. Phase 2 was therefore designed to expand coverage throughout the deposit area and also to provide data for several minor rock types to evaluate the robustness of the proposed surrogates. Phase 1 did not include testing of ore so Phase 2 sampling was expanded to include ore and low-grade ore rock.

Samples for static testing were selected as follows:

- Using the coverage provided by the exploration database, Ron Simpson of GeoSim Services Inc. identified 647 intervals for sampling to provide in-fill data for ICP analysis to support block modelling as not all exploration samples were tested for sulphur content.
- SRK randomly selected 88 samples for ABA testing.
- In addition, 12 samples of minor rock types were selected by SRK.

The samples were recovered from the pulp archive of previously tested samples.

Following interpretation of Phase 2 static data, samples were selected for laboratory kinetic testing (humidity cells). Two samples were selected for each major numerical lithological unit (1, 4, 5, 6, 7, 8, 9, 11) to evaluate reactivity of median and elevated (95th percentile) sulphide concentrations. The samples selected also targeted lower NPs to evaluate onset to ARD, and a range in concentrations of enriched elements. For unit 8, three samples were selected because the approach of characterizing the range in sulphide content did not result in samples representing the range of lead and zinc concentrations.

Figure 3 shows the locations of all drill holes from which samples were obtained for geochemical testing including those tested for ABA, and those used to calculate surrogate ABAs from sulphur and calcium determined by ICP (Figure 6 and Figure 9, respectively)..

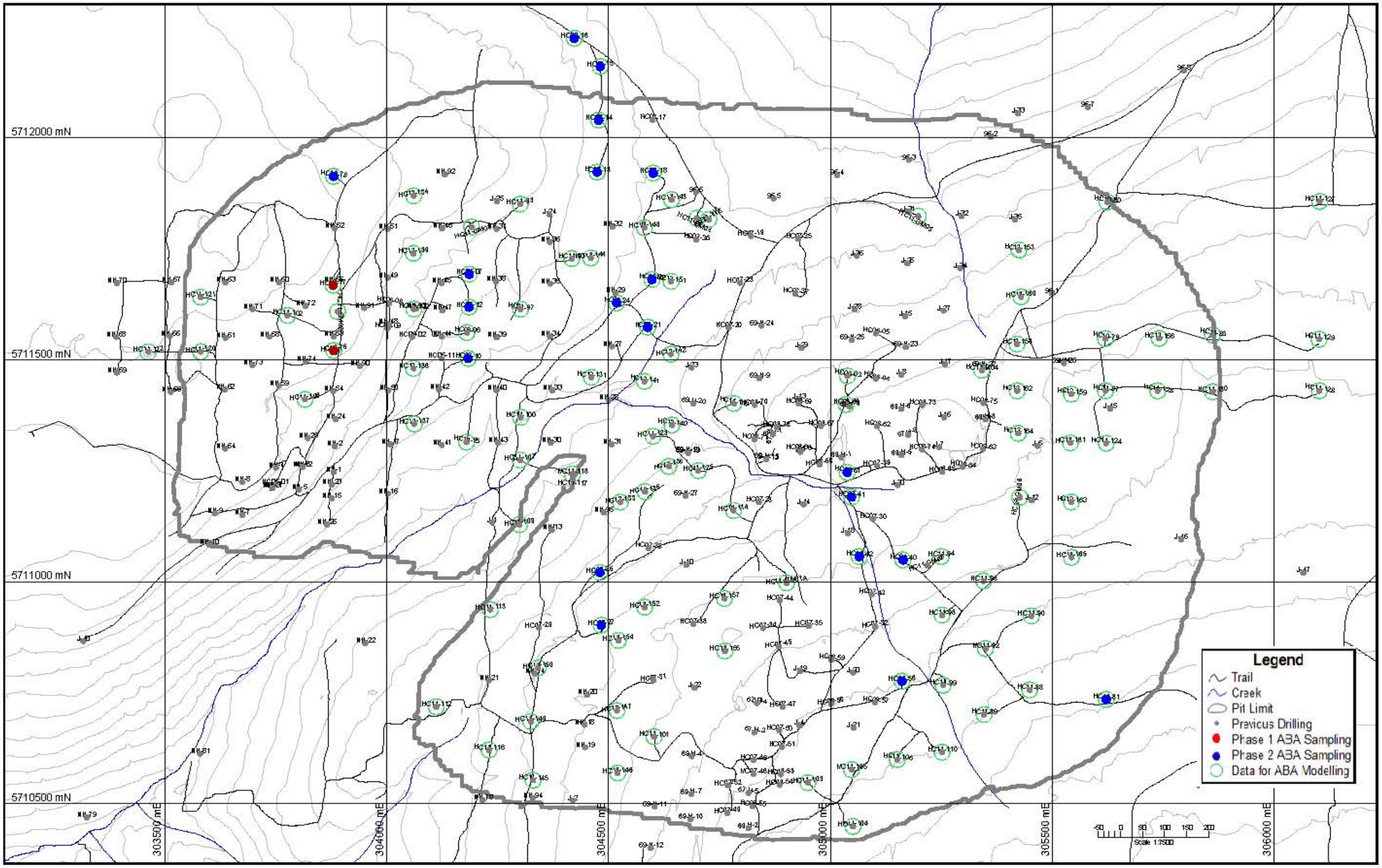

Figure 3: Proposed Mining Area Showing Collars of Drillholes for Core Samples (Phase 1, Phase 2 and All Drillholes Used in the Block Model). Source: HCMC

Tailings

Metallurgical testing was performed by G & T Metallurgical under the direction of HCMC from a master composite (2916-12/13) created from a range of lithologies to represent the first 10 years of mining. Testwork was completed in three phases and provided tailings which were samples for geochemical testing:

- 2916-12: Locked Cycle Test, representative of grind sizes 189 µm and 27µm).
- 2916-13: Locked Cycle Test, representative of grind sizes 189µm and 21µm).
- 3221: Pilot Plant Test representing final process to generate concentrate.

Overburden

Three phases of overburden sampling were completed:

- 16 samples from road cut faces in the open pit area;
- 13 samples from test pits in the footprint of the tailings management facility; and
- 23 samples from drill holes in the open pit area. Ten samples were described as overburden while the balance were weathered bedrock collected below overburden.

All samples were collected by Knight Piésold (KP) and provided to SRK for analysis. Sampling locations are provided in Figure 4.

Quarry Borrow Samples

Six samples of granodiorite were collected from diamond drill core obtained by Knight Piésold from two drill holes in the vicinity of the proposed rock borrow source located near the TMF dam. Locations are shown in Figure 4.

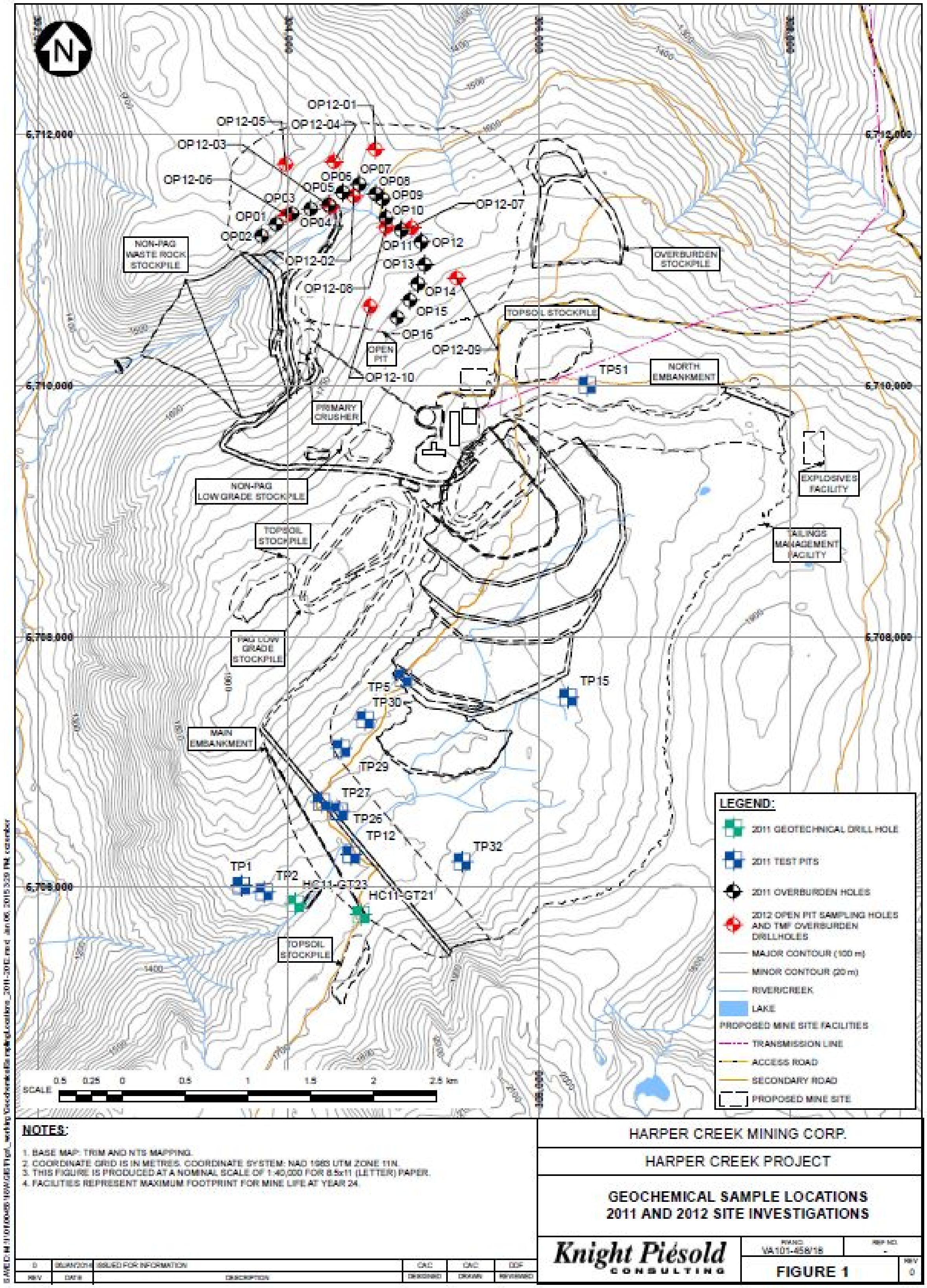

Figure 4: Overburden and Quarry Sampling Locations. Source: KP

3.2.2 Analytical Methods

Sample Preparation

Rock samples obtained from core were prepared in several ways to obtain samples for the various analytical methods:

- Static analyses were performed on a pulp prepared to pass a 200 mesh sieve.
- All laboratory kinetic testing was performed on samples jaw crushed to pass a ¼" sieve.
- Field kinetic testing was performed on samples jaw crushed to pass a 1" sieve.

Samples for Shake Flask Extraction (SFE, Price 2009) were performed on unconsolidated materials screened to pass a 2 mm screen. Samples for Meteoric Water Mobility Procedure (MWMP, NDEP 1997) were performed on unconsolidated materials screened to pass 2 inch screen.

Tailing samples were tested without particle size screening. Four rougher tailings samples were also screened to three size fractions using sieves at 100 and 200 mesh.

Physical Analyses

All samples submitted for kinetic testing were characterized for particle size distribution using particle sieves at 1/4" and meshes of 5, 9, 20, 35, 60, 115 and 250.

Mineralogical Analyses

Mineralogical analyses included:

- Optical mineralogy performed by Kathryn Dunne, PGeo.
- Rietveld x-ray diffraction (XRD) performed by University of BC Department of Earth Ocean and Atmospheric Sciences (UBCEOAS).
- Electron microprobe on mineral grains performed by UBCEOAS.
- QEMSCAN performed by ALS Metallurgy Kamloops.

Twenty-two samples were selected from all rock samples tested under Phase 1 and 2 primarily to support evaluation of the performance of static results. The target for selection was three samples per rock type and spanning ore and low-grade ore materials. Within the rock type groupings, the range of the ratio of carbonate content to neutralization potential was also considered as an indicator of iron carbonate content.

QEMSCAN work was performed to evaluate elevated lead concentrations in overburden samples

Static Geochemical Tests

Standard static geochemical tests performed on all types of samples included:

- Total sulphur by Leco furnace;
- Sulphate determined using hydrochloric acid;
- Neutralization potential by Modified Acid Base Account (Coastech 1991) method;
- Total inorganic carbon determined by coulometric methods;
- Paste pH determined by the Sobek et al (1978);
- Paste conductivity using the same procedure as the paste pH; and
- Element scan (including sulphur) using ICP following a 4-acid digestion, including low level Hg.

Leach Tests

Two types of leach tests were performed on overburden samples to evaluate load flushed by contact with meteoric water following excavation:

- For samples with limited volume for testing, the SFE (Price 2009) method was used. Leachates were analysed using the methods and detection limits indicated in Section 3.2.10.
- For larger samples, the MWMP (NDEP 1997) method was used. Leachates were analysed using the methods and detection limits indicated in Section 3.2.10.

Leach tests used the SFE method were also performed on three stored humidity cell head samples to evaluate accumulation of weathering products in storage.

Humidity Cells

Rock Samples

Seventeen rock core samples were selected for humidity cell samples testing. Seven samples were classified as ore or low-grade ore based on copper grade. The samples were selected on a rock type basis using sulphur content as the primary selection criterion. For each rock type, at least two samples were selected to represent near median sulphur content and sulphur content exceeding the 90th percentile of the rock type. Samples with lower NP were chosen to evaluate if acidic conditions could develop and the characteristics of acidic leachate. In practice, NPs were too high to observe onset of acidic conditions under the timeframe of the environmental assessment.

The inclusion of ore and low grade ore materials ensured that the range of copper content in the samples was considered. Concentrations of arsenic, lead, selenium and zinc were also included as secondary selection criteria so that leaching characteristics could be evaluated with respect to these elements.

In addition, four composite samples tested in field barrels were tested in four humidity cells.

The humidity cells were operated using the procedure described by Price (2009). Leachates were collected and analysed as follows.

Weekly analyses performed were:

- Volume recovered
- pH
- Conductivity
- ORP

The following parameters were measured weekly for the first two weeks and then every two weeks thereafter:

- Acidity
- Alkalinity
- pH
- Sulphate
- Chloride
- Fluoride
- Metal scan (ICP package to include selenium)
- Low level mercury

The 17 rock core humidity cell tests were initiated on December 20, 2011. In mid-July 2013, the tests had yielded 83 weeks of data and were determined, in most cases, to be showing stable leaching conditions. Four tests were selected for continuation based on the following criteria (Table 3):

- Coverage of major rock types and distinctive higher oxidation rates of sedimentary rock types.
- Ore vs waste.
- Lower NPs to evaluation to onset to acidification.
- Range of parameters of interest (sulphide, copper, selenium).

Table 3. Continuing Humidity Cells

Test	Rock Type	Reason
HC-7	Sedimentary	This sample is yielding higher oxidation rates than schist/phylite. It also has lower NP so acid onset may be observed.
HC-13	Schist with quartz eyes	This sample has the 2nd highest sulphide and Se content of schists and phyllites being tested.
HC-15	Schist with no quartz eyes	This sample has median sulphide and Se content for schists and phyllites
HC-16	Alteration	This sample is ore grade rock with lower NP.

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The five humidity cell samples equivalent to the barrel tests were initiated in April 2012 and terminated in July 2013 following acquisition of 68 weeks of leachate data.

Tailings Sample

Sample KM2916-14 was tested in a humidity cell using the configuration for tailings humidity cell specified by Price (2009). This sample had been dried prior to being tested and therefore had likely oxidized. As these storage conditions were not representative of how tailings will be deposited in the TMF, prior to testing in the humidity cell, the sample was re-slurried at a ratio of one part water to one part tailings (by volume) and allowed to sit for 24 hours. Following this procedure, the sample was drained and the leachate analysed for same parameters as humidity cell leachates. The humidity cell test was then initiated.

Leachates were analysed using the same protocol as the waste rock humidity cell.

The test was initiated on date June 5, 2012 and had yielded 59 weeks of data when the test was terminated in July 2013.

Unsaturated Columns

Rock

Testing of rock in four unsaturated columns was designed to occur in parallel with the on-site kinetic tests to provide data on leaching of rock under low water contact ratios. The tests were largely established as a contingency for drought conditions at the site that would result in lack of leachate from the site tests. However, eight rounds of water samples were obtained from the site tests in 2012 and 2013 so the column tests have been discontinued.

The tests were performed by placing 5 kg of -1/4” sample in a column approximately 10 cm in diameter and 60 cm in height. The columns were irrigated with 0.5 L of deionized water every four weeks. This application rate corresponded to 64 mm every four weeks or 830 mm/year compared to average site precipitation of 1050 mm/year. The test therefore involved a higher application rate than site conditions but the flow path length was comparable to the on-site barrel (68 cm).

All leachates were analyzed for the same parameters indicated for humidity cell.

The tests were initiated July 5, 2012 and terminated in July 2013 following acquisition of 14 leachate samples.

Tailings

An unsaturated tailings column was initiated on sample KM 2916-14 Cu Rotl (rougher tail) in July 2012 following pre-treatment as described for the humidity cell. The column configuration was the same as the unsaturated rock column. The water application rate for the sample was 500 mL per week. All leachates were analyzed for the same parameters indicated for humidity cells.

The test was initiated July 12, 2012 and terminated in July 2013. Fifty-five leachate samples were obtained.

Saturated Columns

Rock

Saturated column tests were performed on three major rock type composites prepared from samples used in humidity cells (Table 4).

Table 4: Saturated Column Rock Type Composites

Hole	Sample ID	Humidity Cell	From	To	Lith Code	Rock Type	Subaqueous Column Composite
HC07-27	448	HC-9	143.79	155.475	7	Phyllites	Column 1
HC07-22	359	HC-10	253.55	263.75	7	Phyllites	
HC06-07	61	HC-11	185.19	196.227	8	Schists (Q eyes)	Column 2
HC07-16	256	HC-12	210.72	220.807	8	Schists (Q eyes)	
HC10-77	110250-24	HC-13	172.72	176	8	Schists (Q eyes)	
HC07-14	180	HC-14	264.2	277.41	9	Schists (No Q eyes)	
HC07-21	339	HC-15	207.9	217.9	9	Schists (No Q eyes)	
HC07-14	187	HC-16	373.8	380.09	11	Alteration	Column 3
HC10-77	110250-39	HC-17	293	296.5	11	Alteration	

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The tests were performed on 5 kg of each sample in a 14 cm diameter column. The rock was covered by 60 cm of deionized water which was maintained to make-up for removal by sampling and evaporation. Each week, 300 mL of water was recovered from the base of the column for analysis for the same parameters as humidity cells.

The tests were initiated in July 2012 and terminated in July 23 following acquisition of 55 weeks of data.

Tailings

Samples 6.2.2 KM 2916-14 Cu Rotl (rougher) and KM 2916-14 Cu 1CT (cleaner) were tested in saturated columns following the same pre-treatment as described for the humidity cell.

Sample 6.2.2 KM 2916-14 Cu Rotl was tested in a column with the same configuration as waste rock. Less than 0.5 kg of sample was available for KM 2916-14 Cu 1CT. As a result, the column dimension was reduced to an inside diameter of 5 cm.

Leachates were recovered and analysed using the same protocol as the waste rock saturated column.

The tests were initiated in July 2012 and terminated in July 23 following acquisition of 55 weeks of data.

On-Site Kinetic Tests

On-site kinetic or barrel tests were constructed in 2012 and were sampled using protocols provided by SRK.

Sample feed was prepared using the following sequence:

- Lithological coding in the core database was used to sort the dataset into lithology sub-sets.
- The total available mass of core in each subset was estimated.
- Rock types expected to yield less than 300 kg of sample were not considered further as they would not yield sufficient sample for testing.
- All intervals in the major rock types (codes 7, 8, 9 and 11) were assigned a random decimal number from 0 to 1.
- All intervals with a random number less than the ratio of 300/total mass available were selected for inclusion in the composite thereby yielding a composite of about 300 kg.
- Using the resulting list of samples, core was recovered and crushed to pass a 1” screen.
- All crushed intervals were composited at Inspectorate. A split of the composite was retained for static and mineralogical testing in humidity cells as indicated in Section 3.2.6.
- The crushed material was returned to the site and placed in the barrels by HCMC.

Ongoing monitoring of the barrels was performed as part of baseline water quality sampling. The collection pails were inspected roughly once each month. The volume of leachate was recorded along with pH, conductivity and temperature and sampled if sufficient leachate was available for testing. The remaining leachate was discarded following sampling.

Four barrels containing each rock type composite were constructed in May 2012. Data available at the time of interpretation included leachate samples obtained from each barrel on eight occasions through the summers of 2012 and 2013.

Solution Analyses

Solutions produced by various test procedures were analysed for the parameters indicated in Table 5.

Table 5: List of Analytes and Detection Limits Used for Testing Solutions Produced by Extraction, Laboratory Kinetic Tests and Site Kinetic Tests

Parameter	Unit	Barrel	Columns	HCT	MWMP	SFE	Parameter	Unit	Barrel	Columns	HCT	MWMP	SFE
Conductivity	uS/cm	2	0.5	0.5	0.5	0.5	Cadmium (Cd)-Dissolved	mg/L	0.00001	0.00003	0.000005	0.000005	0.000005
--	mV	--	0	5	0	0	Calcium (Ca)-Dissolved	mg/L	0.05	0.3	0.05	0.05	0.05
Hardness (as $CaCO_3$)	mg/L	0.5	0.5	0.5	0.5	0.5	Cesium (Cs) - Dissolved	mg/L	--	0.0003	0.00005	0.00005	0.00005
pH	pH	0.1	N/A	0.5	N/A	N/A	Chromium (Cr)-Dissolved	mg/L	0.0001	0.0005	0.0001	0.0001	0.0001
Total Suspended Solids	mg/L	1	--	--	--	--	Cobalt (Co)-Dissolved	mg/L	0.0001	0.00003	0.000005	0.000005	0.000005
Total Dissolved Solids	mg/L	10	--	--	--	--	Copper (Cu)-Dissolved	mg/L	0.0002	0.0003	0.00005	0.00005	0.00005
Turbidity	NTU	0.1	--	--	--	--	Iron (Fe)-Dissolved	mg/L	0.01	0.005	0.001	0.001	0.001
Acidity (mg $CaCO_3$/L) 4.5	mg/L	--	0.5	0.5	--	0.5	Lanthanum (La) - Dissolved	mg/L	--	0.0003	0.00005	0.00005	0.00005
Acidity (mg $CaCO_3$/L) 8.3	mg/L	--	0.5	0.5	--	0.5	Lead (Pb)-Dissolved	mg/L	0.00005	0.00003	0.000005	0.000005	0.000005
Alkalinity, Bicarbonate (as $CaCO_3$)	mg/L	2	--	--	--	0.5	Lithium (Li)-Dissolved	mg/L	0.0005	0.003	0.0005	0.0005	0.0005
Total Alkalinity	mg/L	2	0.5	0.5	0.5	0.5	Magnesium (Mg)-Dissolved	mg/L	0.1	0.3	0.05	0.05	0.05
Alkalinity, Hydroxide (as $CaCO_3$)	mg/L	2	--	--	--	0.5	Manganese (Mn)-Dissolved	mg/L	0.00005	0.0003	0.00005	0.00005	0.00005
Alkalinity, Total (as $CaCO_3$)	mg/L	2	--	--	--	--	Mercury (Hg)-Dissolved	mg/L	0.00001	0.00005	--	--	--
Ammonia, Total (as N)	mg/L	0.005	--	--	--	--	Molybdenum (Mo)-Dissolved	mg/L	0.00005	0.0003	0.00005	0.00005	0.00005
Bromide (Br)	mg/L	0.5	--	--	--	--	Nickel (Ni)-Dissolved	mg/L	0.0005	0.0001	0.00002	0.00002	0.00002
Chloride (Cl)	mg/L	5	0.5	0.5	0.5	0.5	Phosphorus (P)-Dissolved	mg/L	0.3	0.01	0.002	0.002	0.002
Fluoride (F)	mg/L	0.2	0.05	0.01	0.01	0.01	Potassium (K)-Dissolved	mg/L	0.05	0.3	0.05	0.05	0.05
Nitrate and Nitrite (as N)	mg/L	0.051	--	--	--	--	Rubidium (Rb) -Dissolved	mg/L	--	0.0003	0.00005	0.00005	0.00005
Nitrate (as N)	mg/L	0.05	--	--	0.02	--	Selenium (Se)-Dissolved	mg/L	0.0001	0.0002	0.00004	0.00004	0.00004
Nitrite (as N)	mg/L	0.01	--	--	0.005	--	Silicon (Si)-Dissolved	mg/L	0.05	0.5	0.1	0.1	0.1
Orthophosphate-Dissolved (as P)	mg/L	0.001	--	--	--	--	Silver (Ag)-Dissolved	mg/L	0.00001	0.00003	0.000005	0.000005	0.000005
Phosphorus (P)-Total	mg/L	0.002	--	--	0.005	--	Sodium (Na)-Dissolved	mg/L	0.05	0.3	0.05	0.05	0.05
Sulfate (SO4)	mg/L	5	5	0.5	0.5	0.5	Strontium (Sr)-Dissolved	mg/L	0.0002	0.0003	0.00005	0.00005	0.00005
Cyanides	mg/L	0	--	--	--	--	Sulphur (S) -Dissolved	mg/L	--	10	10	10	10
Cyanide, Weak Acid Diss	mg/L	0.005	--	--	--	--	Tellurium (Te) -Dissolved	mg/L	--	0.0001	0.00002	0.00002	0.00002
Cyanide, Total	mg/L	0.005	--	--	--	--	Thallium (Tl)-Dissolved	mg/L	0.00001	0.00001	0.000002	0.000002	0.000002
Thiocyanate (SCN)	mg/L	0.5	--	--	--	--	Thorium (Th) -Dissolved	mg/L	--	0.00003	0.000005	0.000005	0.000005
Cyanide, Free	mg/L	0.005	--	--	--	--	Tin (Sn)-Dissolved	mg/L	0.0001	0.00005	0.0002	0.0002	0.0002
Organic / Inorganic Carbon	mg/L	0	--	--	--	--	Titanium (Ti)-Dissolved	mg/L	0.01	0.003	0.0005	0.0005	0.0005
Total Organic Carbon	mg/L	0.5	--	--	--	--	Tungsten (W) - Dissolved	mg/L	--	0.00005	0.00001	0.00001	0.00001
Aluminum (Al)-Dissolved	mg/L	0.001	0.001	0.0002	0.0002	0.0002	Uranium (U)-Dissolved	mg/L	0.00001	0.00001	0.000002	0.000002	0.000002
Antimony (Sb)-Dissolved	mg/L	0.0001	0.0001	0.00002	0.00002	0.00002	Vanadium (V)-Dissolved	mg/L	0.001	0.001	0.0002	0.0002	0.0002
Arsenic (As)-Dissolved	mg/L	0.0001	0.0001	0.00002	0.00002	0.00002	Zinc (Zn)-Dissolved	mg/L	0.001	0.0005	0.0001	0.0001	0.0001
Barium (Ba)-Dissolved	mg/L	0.00005	0.0001	0.00002	0.00002	0.00002	Zirconium (Zr) -Dissolved	mg/L	--	0.0005	0.0001	0.0001	0.0001
Beryllium (Be)-Dissolved	mg/L	0.0001	0.00005	0.00001	0.00001	0.00001	Mercury (Hg) - Dissolved	ug/L	--	0.002	0.002	--	--
Bismuth (Bi)-Dissolved	mg/L	0.0005	0.00003	0.000005	0.000005	0.000005	Mercury (Hg) - Dissolved	mg/L	--	--	--	0.00005	0.00005
Boron (B)-Dissolved	mg/L	0.01	0.3	0.05	0.05	0.05							

Source: P:\01_SITES\Harper_Creek\1CY003.001_MLARD_Feasability&EA\1000_Reporting\1050_Draft_EA\Tables\[DetectionLimitTable_1CY003001_Rev00_AML.xlsx]

Quality Control

Quality control (QC) measures for the analytical procedures are specified by agreement between SRK and Maxxam Analytics the analytical laboratory (SRK 2011). For the Harper Creek Project, the specific QC measures are indicated in Table 6.

Table 6: Quality Control Measure by Program

Procedure	Number of Tests	Blank Tests	Replicates
Static tests	191	8	15
Humidity cells	21	1	2
SFE	8	2	2
MWMP	5	1	2
Saturated columns	5	1	1
Unsaturated columns	5	1	
Field barrels	4		

Source: \\VAN-SVR0\Projects\01_SITES\Harper_Creek\1CY003.001_MLARD_Feasability&EA\1000_EA_Reporting\1060_Final_EA\Tables\Table_5_QAQC_20130308

4 Results

4.1 Waste Rock

4.1.1 Sulphur Occurrence

Mineralogy

Petrographic descriptions, presented in Appendix A1, indicated for the samples tested in humidity cells, sulphur occurs mainly as pyrite with significant chalcopyrite. Pyrrhotite tended to be much less abundant than either pyrite or chalcopyrite. Sphalerite was observed in five samples and traces of galena were observed in one sample. All minerals occurred mainly in disseminated form as free rather than occluded grains. Sphalerite where present had very small occluded chalcopyrite but the latter mainly occurred as unoccluded grains. Descriptions of the barrel test composites noted traces of marcasite and molybdenite and possibly also arsenopyrite along with pyrite, chalcopyrite, sphalerite and pyrrhotite.

Sulphur Speciation

Complete sulphur speciation results are presented in Appendix B1. Analyses showed low levels of sulphate sulphur with median concentrations around 0.01% though sulphate ranged up to 0.43% sulphur.

Paste pHs were negatively correlated with sulphate concentrations and paste conductivity (Figure 5) implying that sulphate in the samples is not a primary mineral (such as gypsum) but more likely secondary minerals formed while the core was in storage. The correlation reflects dissolution of acid sulphates and partial neutralization during the paste pH procedure. Higher levels of sulphate release proportionately greater acidity which may not be completely neutralized by reaction with carbonates in the procedure. The paste pHs obtained therefore may reflect minimum actual pHs. Paste pHs must also be interpreted with respect to the pH of the deionized water used in the test (pH typically near 5.5). Paste pHs above this level can be taken as an indication that the sample has at least consumed the acidity of the deionized water. One sample of ore (HC07-14 373.8-380.09) had a truly acidic paste pH of 4.0 but it also contained 21 kg$CaCO_3$/t carbonate and was non-acidic in a humidity cell. This indicates that the depressed paste pH reflected slow neutralization of acidity during pH procedure.

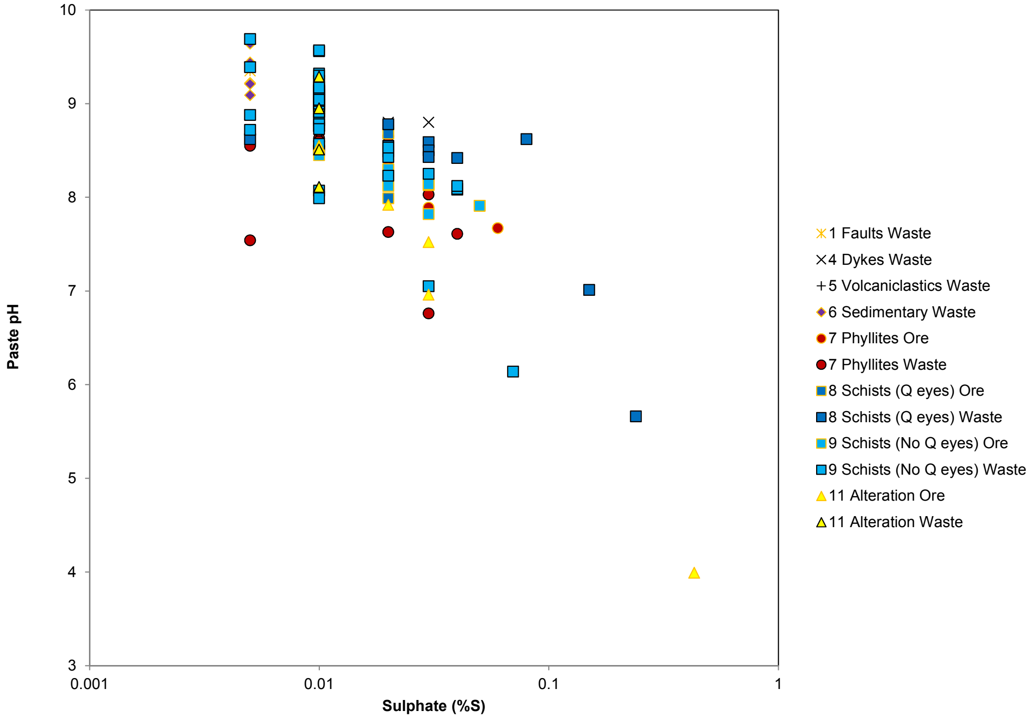
The age of the core varied from fresh to five years old at the time of testing. The highest sulphate concentrations were in the oldest core but these samples also contained the highest sulphide concentrations.

ICP Sulphur

Full results are presented in Appendix B2. Exploration analyses for the project have recently included an ICP scan which included sulphur concentrations. Strong correlation of total sulphur (by Leco) with ICP sulphur (as determined following a 4-acid digestion) is apparent (Figure 6) and indicates that ICP sulphur concentrations can be used to estimate total sulphur concentrations. The resulting regression equation:

$$\text{Log Total S(S,\%)} = 1.1 \cdot \log(\text{S(4-acid),\%}) - 0.04$$

indicated that sulphur by ICP slightly under-estimates total S by Leco at low sulphur concentrations and over-estimates total S at higher concentrations but in the mid-range the two measures are nearly equivalent.



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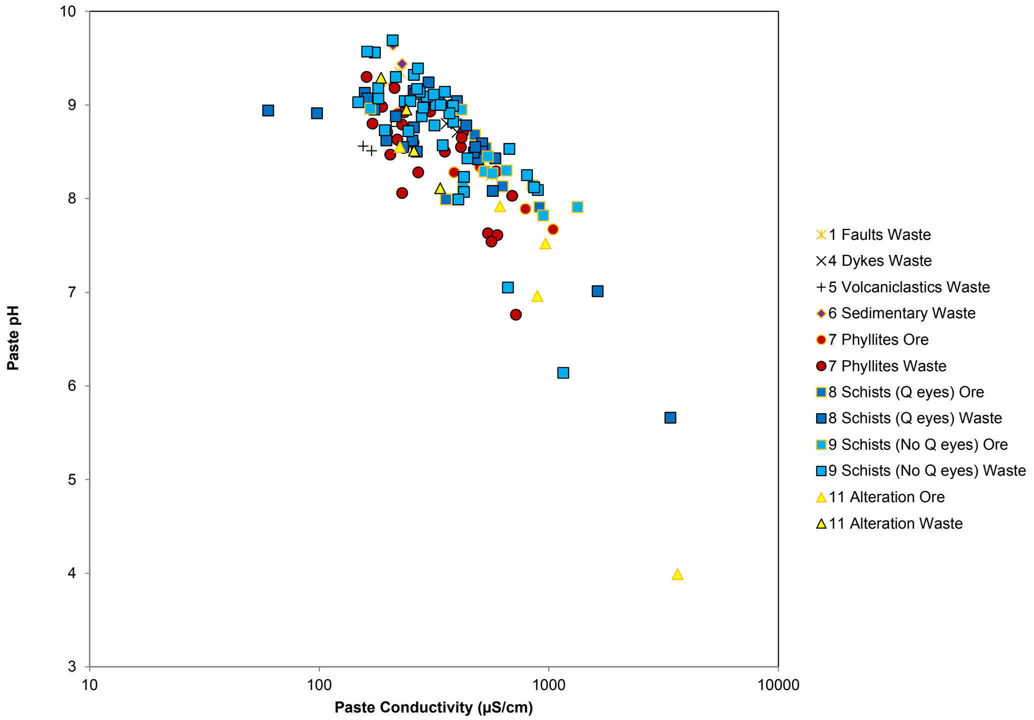

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Figure 5: Paste pH vs Sulphate Sulphur Concentration and Paste Conductivity in Rock Samples.

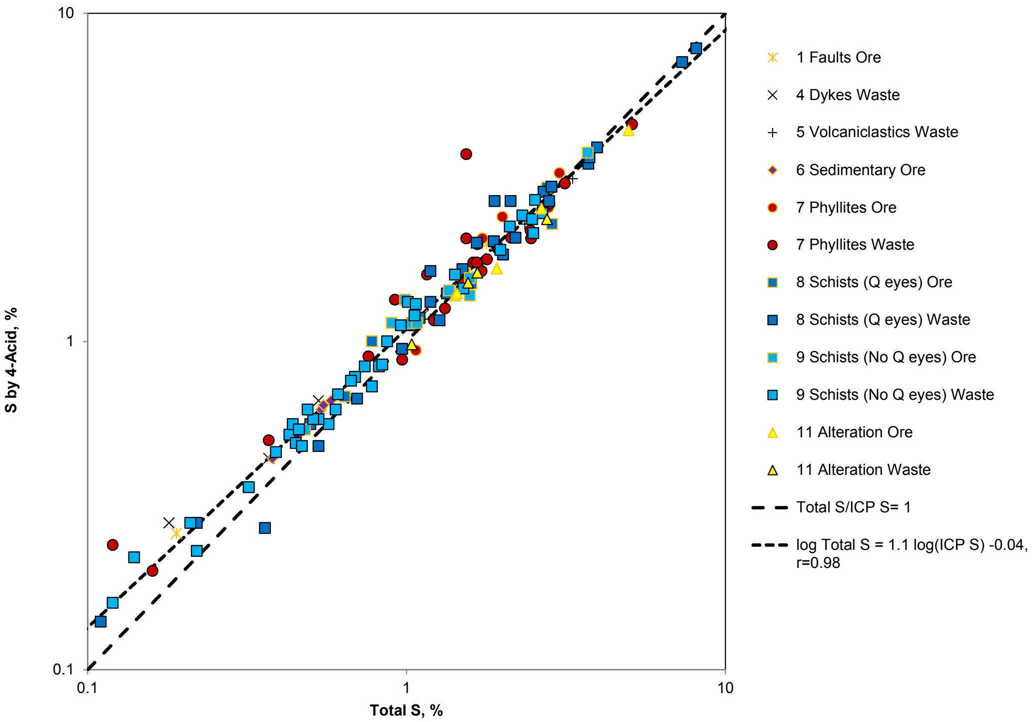

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Figure 6: Correlation of Total Sulphur (by Leco) with ICP Sulphur

For the purpose of subsequent ARD potential block modelling, it was concluded that sulphur determined by ICP could be used to calculate total sulphur using the above equation, which in turn is an appropriate method to calculate acid potential AP.

4.1.2 Neutralization Potential Occurrence

Carbonate Mineralogy

Table 7 provides results of XRD analyses on 22 samples for the major rock types. Full results are presented in Appendix A2. All four major carbonate minerals were determined to be widespread. The presence of iron carbonates (ankerite and siderite) confirms the observation of distinctive weathering of core. The occurrence of siderite appears to be partially rock type controlled as shown by its dominance in dyke samples. Ankerite appears to occur mainly in dykes and volcaniclastics and unit 6 lacks ankerite and calcite.

Table 7: X-Ray Diffraction Results for Rock Samples

ID	Rock Type Code	Rock Type	Ankerite	Calcite	Dolomite	Siderite
			%	%	%	%
648	1	Faults	0	0	3.3	2.7
650	1	Faults	0	0	2.5	0
654	1	Faults	3.7	3.3	12	0
330	4	Dykes	2.1	1.2	6.2	12
336	4	Dykes	3.4	0.54	6.8	13
337	4	Dykes	4.4	0.97	6.3	15
110250-4	5	Volcaniclastics	14	9.7	0	0.34
110250-1	5	Volcaniclastics	3.9	20	0	0
659	6	Sedimentary	0	0	9.1	0
660	6	Sedimentary	0	0	0.22	5.2
663	6	Sedimentary	0	0	1.4	1.2
64	7	Phyllites	0	0	2.8	1.7
76	7	Phyllites	0	0.13	0.35	4.5
194	7	Phyllites	0	25	0.37	0
39	8	Schists (Q eyes)	0	0.17	2.9	3.4
169	8	Schists (Q eyes)	16	2.6	0	0.74
286	8	Schists (Q eyes)	0	9.6	0	0
202	9	Schists (No Q eyes)	0	8.2	5.6	0
339	9	Schists (No Q eyes)	0	0.37	0.22	8.9
489	9	Schists (No Q eyes)	0	0	2	0.96
187	11	Alteration	0	0	0	1.4
331	11	Alteration	0	0	2.1	0

Source: P:\01_SITES\Harper_Creek\1CY003.000_MLARD_Characterization\2011-07_Static_Testing\3.Interpretation\[Carbonate_NP_Calibration_1CY003000_SJD_20111124_VER00.xlsx]

Neutralization Potential

Analytical carbonate content is correlated with modified neutralization potential (NP) but carbonate nearly always exceeds NP when expressed as kg $CaCO_3$/t as expected based on the presence of iron carbonate minerals (Table 7). Full results are presented in Appendix B1.

Site Specific Neutralization Potential

An equation for estimating site specific neutralization potential (NP*) was developed using the method of Day (2009). The inputs to the method were the x-ray diffraction (XRD) data in Table 7 and the chemical formulae of the carbonate minerals. For the latter, it was assumed initially that carbonate minerals had their pure end-member compositions pending determination of whether this allowed for a reasonable fit to the carbonate analytical data. The first step in the method was to calculate mineralogical carbonate content using the mineralogical data. Comparison of “mineralogical” carbonate content with analytical carbonate content showed an excellent correlation with values near parity (Figure 7). Most rock types showed an equal scatter around the equivalence line though the samples coded as sedimentary (Unit 6) had analytical TIC exceeding mineralogical. TIC for all three sample.

This comparison showed that mineralogical data and the underlying assumption of carbonate mineral composition was reasonable.

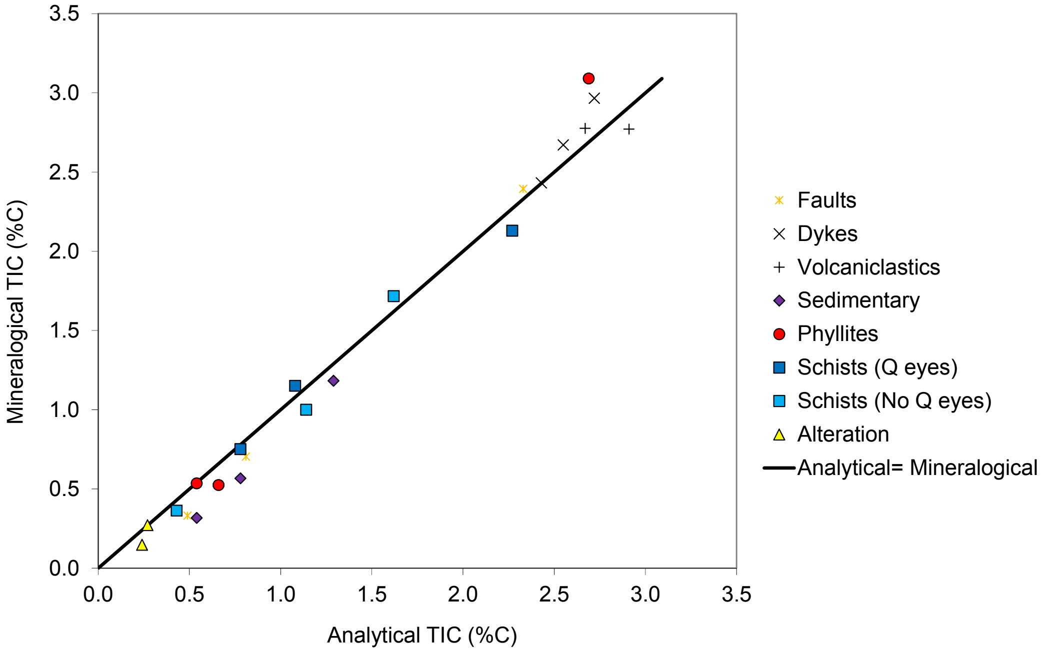

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Figure 7: Comparison of Analytical and Mineralogical TIC

The next step in the calculation was to compare the estimated calcium and magnesium carbonate content with modified neutralization potential (Figure 8).

At lower NP, there is some indication that the results of the Modified NP method used in this evaluation is under-estimating Ca and Mg carbonate contents. A similar effect has been observed in other datasets (Red Chris 2004) and reflects the low acid strength used in the NP method which may not fully quantify the Ca and Mg content of slower reacting carbonates like ankerite.

At higher NP, the reverse appears to be true in that NP exceeds Ca and Mg carbonate for most samples. This probably reflects the digestion of silicate minerals by the higher acid addition used in the NP determination at higher NP values.

This comparison allows for calculation of a site specific NP measure (NP*) as follows:

- For NP<100 kg $CaCO_3$/t: NP* (kg $CaCO_3$/t) = NP

- For NP≥100 kg $CaCO_3$/t: NP* (kg $CaCO_3$/t) = NP-18

The value of 18 kg $CaCO_3$/t used to correct NP is estimated from one sample that yielded an acidic paste pH but reported an NP of 18 kg $CaCO_3$/t. The average shift in Figure 8 is -13 kg $CaCO_3$/t.

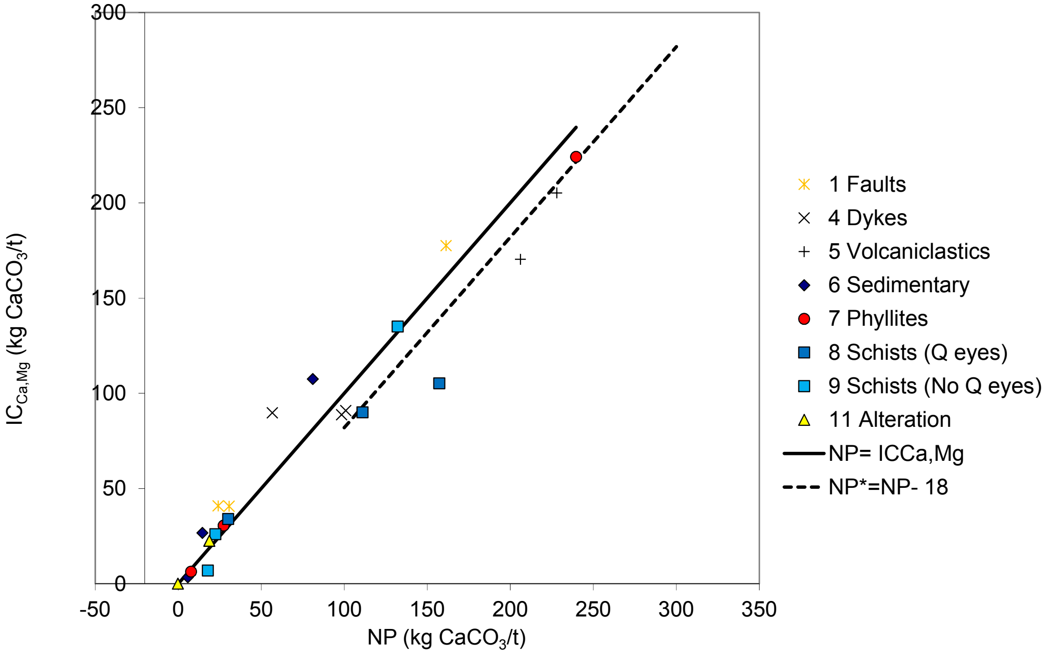

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Figure 8: Comparison of Modified NP with Calcium and Magnesium Carbonate Content

NP Surrogate

The exploration ICP database contains calcium and magnesium concentrations determined following a 4-acid digestion. As a result of the large number of analyses available and the potential opportunity to use the data for block modelling of ARD potential, correlations of Ca and Mg with NP were investigated to determine if these results could be a reliable surrogate for NP.

Figure 9 shows that calcium concentrations are correlated with NP with a correlation coefficient (r) of 0.95. Throughout most of the concentration range, NP exceeds calcium content when expressed in the same units but the difference decreases as NP increases. The resulting regression equation for the whole dataset is:

$$\text{Log NP (kg } CaCO_3\text{/t)} = 0.8 \cdot \log(\text{Ca, kg } CaCO_3\text{/t}) + 0.49$$

In general the equation holds for all rock types though is weakest for Unit 8 (schists with quartz eyes) at highest NP concentrations. Magnesium was evaluated for its effect either on its own or in combination with Ca; however, correlations were much weaker and it was not considered further.

The finding that Ca determined following a 4-acid digestion correlates well with NP was unexpected because the Ca concentrations determined by this method would be expected to report silicates in addition to carbonates. The explanation appears to lie in the dominance of non-calcic silicates in the samples. Quartz is a major component whereas the only calcium-bearing mineral other than carbonates is plagioclase which occurs at concentrations generally less than 10% and is sometimes absent. Furthermore, plagioclase in these settings may be more albitic than anorthitic which both limits calcium concentrations and limits attack by acid digestions. The weak correlation of magnesium with NP may reflect the presence of chlorite which contains Mg and is susceptible to acid attack.

The potential error in ARD potential classification resulting from the use of the surrogate is evaluated in Section 4.1.4.

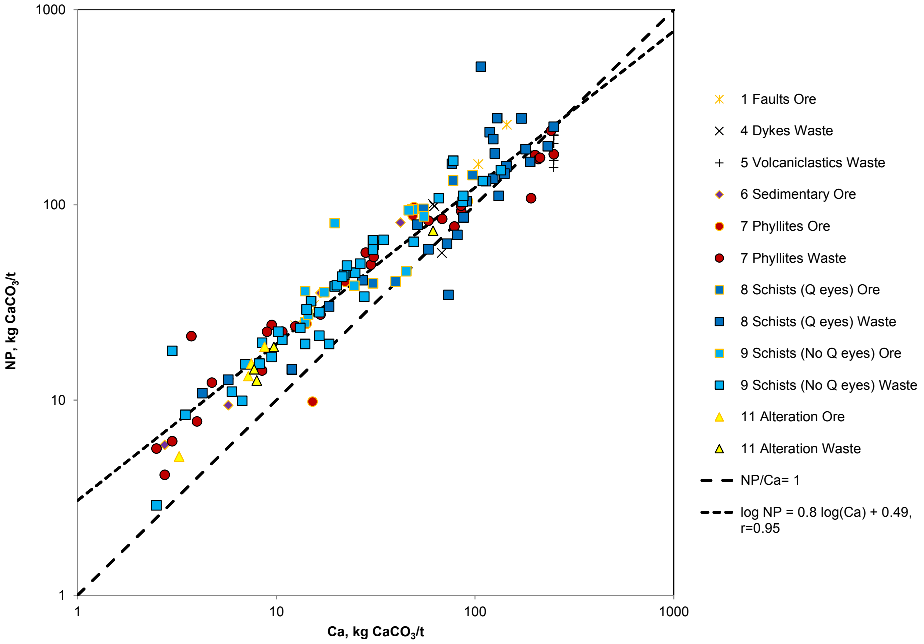

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Figure 9: Comparison of NP and with Ca

4.1.3 Trace Element Occurrence

Complete trace element results are presented in Appendix B2. A correlation matrix for trace elements was calculated to evaluate associations between elements and possible linkages to mineralogical hosts. Significant relationships were assessed from a correlation coefficient of 0.21 (1% significance level) but strong correlations were determined from much higher correlation coefficients. The matrix indicated that:

- Sulphur concentrations were correlated with many elements expected to occur in sulphide minerals including Ag, As, Cd, Co, Cu, Fe, Pb, Sb, Se and Zn. The strongest correlation was for selenium (r=0.7).

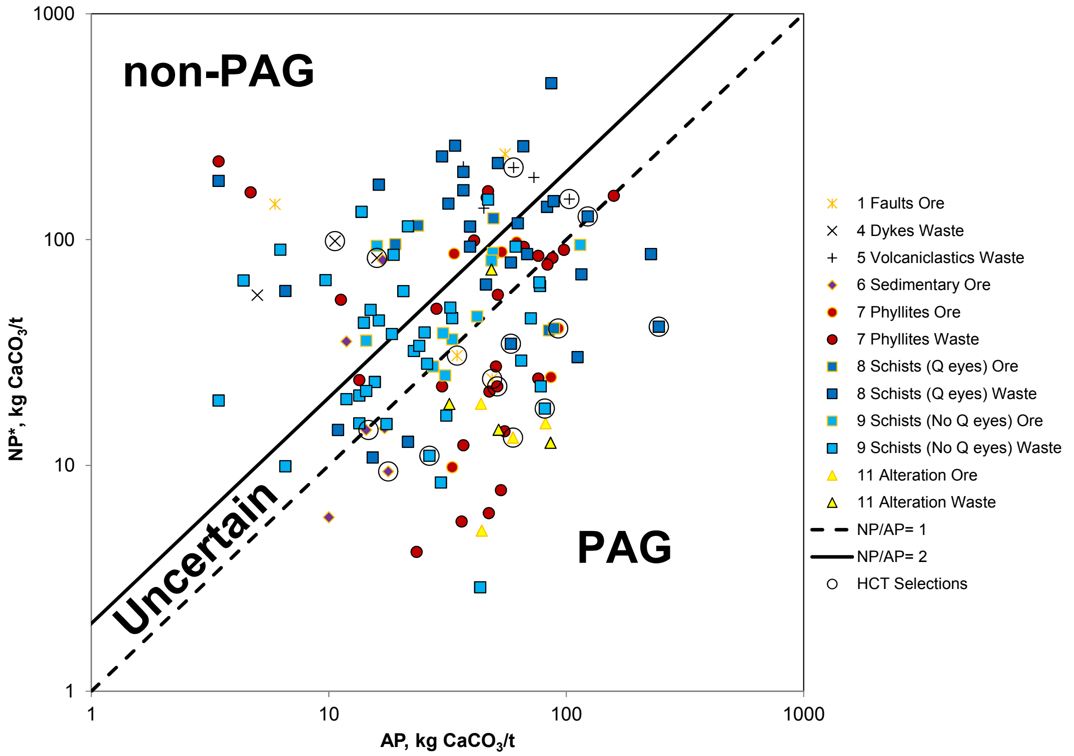
- Copper is mainly present as chalcopyrite and was correlated with Ag and Se suggesting these elements mainly associate with chalcopyrite.
- Zinc is mainly present as sphalerite. As is commonly observed, cadmium was strongly correlated with zinc (r=0.8).
- Galena was observed at trace levels. Lead is only weakly correlated with sulphur content which implies that it is mainly associated with galena.
- Arsenopyrite was observed at trace levels. Arsenic was only correlated with sulphur which suggests it is mainly hosted by arsenopyrite.
- Molybdenite is present at trace levels. Molybdenum is weakly correlated with Ag, Cu and Se suggesting an association with chalcopyrite.

4.1.4 ARD Potential

By Rock Type

Figure 10 shows NP* vs AP with samples coded according to rock type. Results are shown with respect to NP*/AP classification lines of 1 and 2. Since NP* is refined site specific correction of NP based on carbonate mineralogy, NP/AP>2 is an appropriate definition for non-PAG rock. As shown by Figure 10, ARD potential varies widely from strongly PAG to strongly non-PAG. Rock type may be a control for some aspects:

- Schist with quartz eyes (Unit 8) has highest NP of all rock types. AP is also higher than Unit 9 but Unit 8 tends to be non-PAG.
- Unit 9 varies from PAG to non-PAG.
- Phyllites (Unit 7) have widely varying NP and AP but generally classified as PAG or uncertain.
- Sampling was limited in the faults (Unit 1), dykes (Unit 4) and volcaniclastics (Unit 5) but samples from these units were mainly non-PAG.



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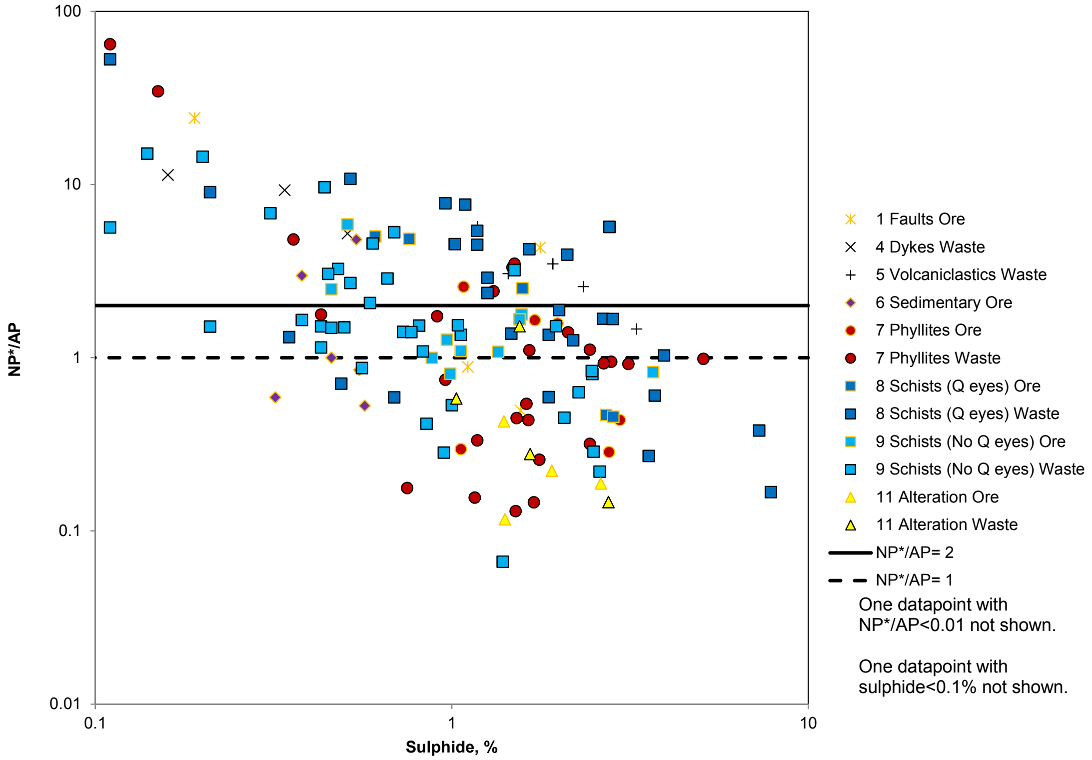

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Figure 10: NP vs AP (top) and NP*/AP vs Sulphide Content (bottom) by Rock Type By Package

Figure 11 shows the same data as Figure 10 but with data coded by rock package rather than rock type. Coding by data package provides some additional resolution of ARD potential:

- Waste samples from Packages D (Intermediate Calcareous Volcaniclastics) and Fb (Intermediate to Mafic Volcaniclastics) have NP*/AP typically near or above 1 indicating lower ARD potential.
- Package Fa (Felsic and Intermediate Volcaniclastics) have NP*/AP less than 2 extending to lower NP*/AP than the above packages indicating higher ARD potential.
- Package E (Graphitic Horizon) is mainly PAG based on NP*/AP less than 1.
- Package B shows greatest variability of NP*/AP including PAG and non-PAG classification.

Spatial

Using the site-specific surrogate method, downhole plots of total sulphur and NP/AP were prepared for six holes throughout the proposed area of the open pit to evaluate the scale of occurrence of PAG (defined as NP*/AP*<2) and non-PAG intervals (Appendix B3). As sampling intervals were less than 2 m, this method provides a high level of resolution of ARD potential. This assessment provides an indication of the level of effort which will be needed to separate PAG and non-PAG waste rock and low grade ore during mining.

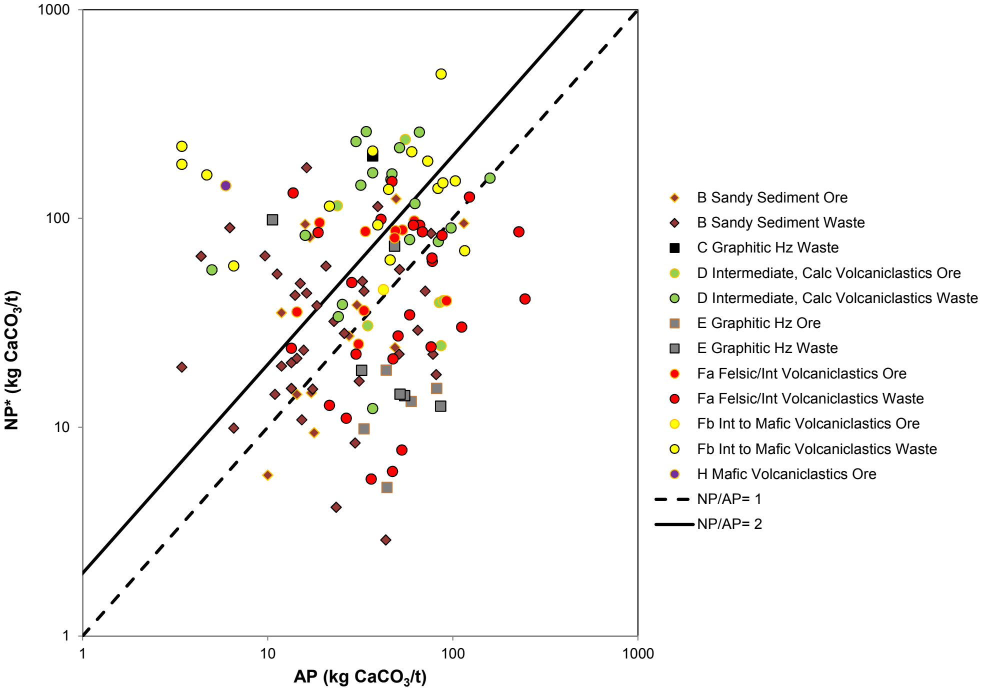
The plots were manually annotated to classify intervals of dominantly PAG (i.e. PAG with isolated single intervals of non-PAG), non-PAG (i.e. non-PAG with isolated non-contiguous intervals of PAG) and mixed (alternating PAG and non-PAG). The latter intervals would be classified as PAG or non-PAG during mining because the PAG and non-PAG intervals would become mixed during blasting, loading into haul trucks and dumping.

The assessment indicated that consistent PAG layers commonly occur over intervals exceeding 15 m and therefore that PAG rock will rarely contain significant entrained non-PAG rock (for example, see 240 to 370 m in HC11-93, 150 to 300 m in HC10-76, 180 to 235 m in HC11-115, 70 to 260 m in HC11-119. 70 to 170 in HC11-101) .

Non-PAG intervals tend to be narrower (less than 20 m) though locally are tens of metres thick (for example, 30 to 80 m HC10-76 and 110 m to 170 m in HC11-94). The generally smaller scale of non-PAG occurrence indicates that management at the bench scale will require blast hole sampling and modeling to define dig limits. Where non-PAG rock occurs in very narrow intervals with narrow intervals of PAG rock, non-PAG rock is probably not physically separable and the overall mixture may be PAG.

Use of Surrogate Parameters for ARD Potential Classification

As discussed in Section 4.1.2, calcium has been identified as a potential surrogate for neutralization potential. The use of a surrogate potentially results in classification errors due to the imperfect correlation between measured values and those based on the surrogate. Figure 12 compares NP*/AP with the ratio calculated using sulphur and calcium concentrations determined by ICP to estimate NP and AP. NP_{Ca}^{*} was calculated using the equation that links Ca and NP to calculate NP_{Ca} then the correction to NP based on the mineralogical correction to obtain NP_{Ca}^{*}. $AP_{ICP\text{-}S}$ was calculated from the equation linking total sulphur determined by Leco and ICP sulphur.



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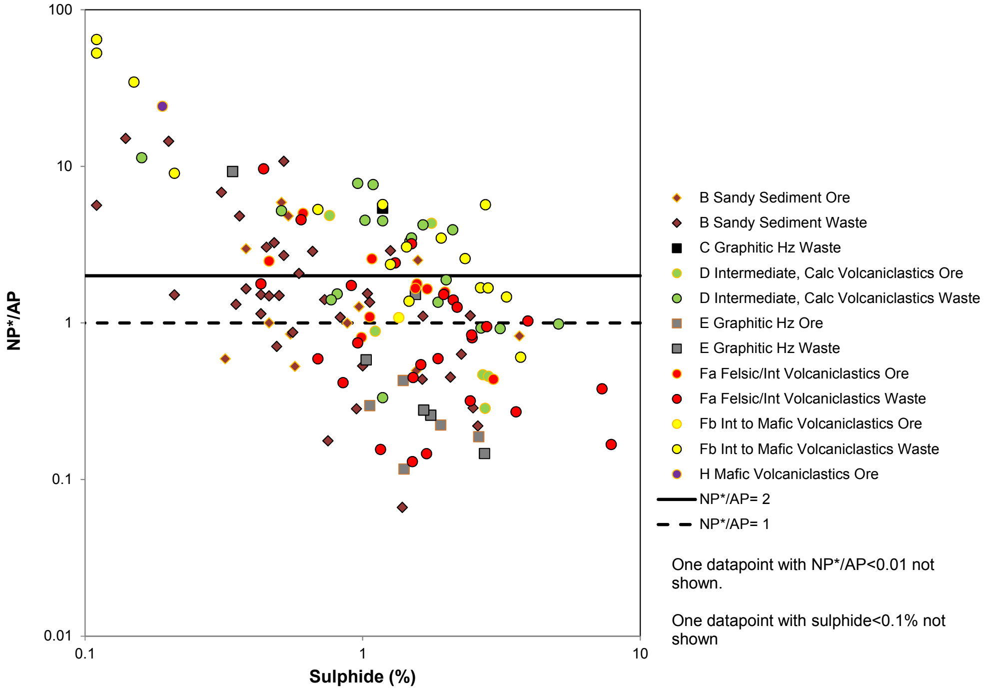

One datapoint with NP*/AP<0.01 not shown.

One datapoint with sulphide<0.1% not shown

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Figure 11: NP* vs AP (top) and NP*/AP vs Sulphide Content (bottom) By Stratigraphic Package

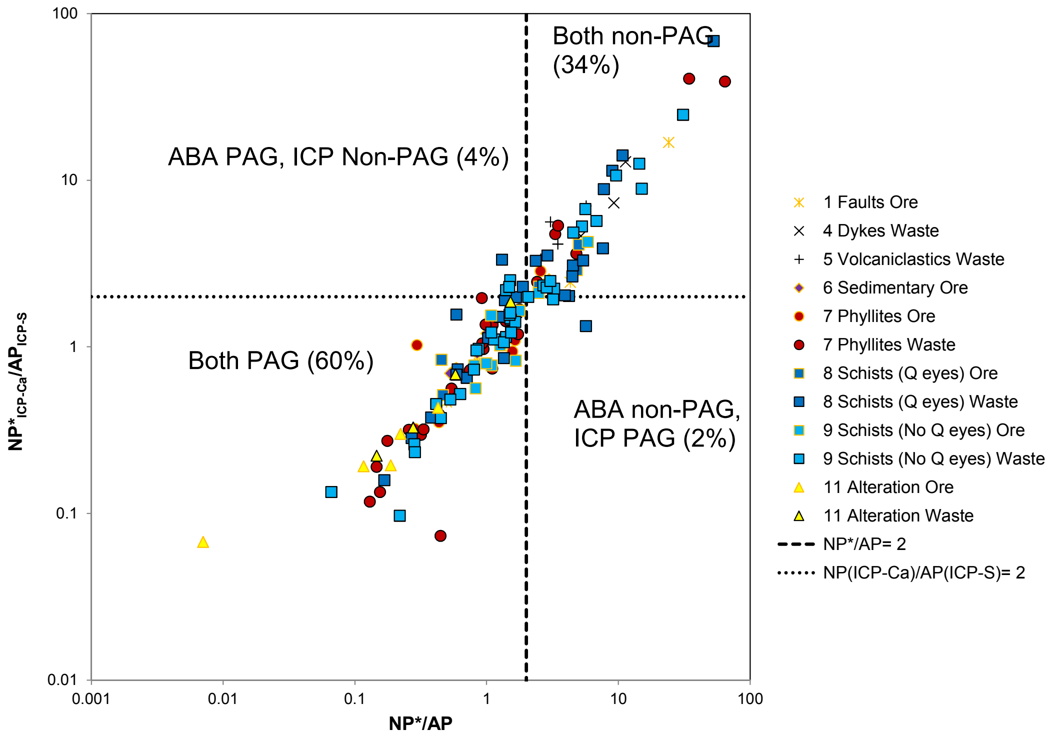

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Figure 12: Evaluation of ARD Potential Miss-Classification as a Result of Using the Surrogate Method

The graph in Figure 12 is divided into four fields. Two fields indicate that both methods produce the same result (PAG or non-PAG). Two fields recognize errors in classification. The field in which ABA classifies a result as PAG whereas the surrogate method indicates non-PAG represents the more significant error for determining quantities of PAG waste in the block model because they potentially result in under-estimation of the requirements for PAG rock storage.

Due to the range of NP/AP values, 94% of results were classified as PAG or non-PAG by both methods. Of the remaining 6%, 4% were miss-classifications of the type indicated above. These percentages might be regarded as the uncertainty in volume requirements but is a relatively low proportion in the context of numerous other factors and is not considered significant. The surrogate method appears to provide a robust method for expanding the data coverage using ICP data.

Effect of Accidental PAG Rock on Segregated Non-PAG Rock

The mine plan involves segregation of PAG and non-PAG rock so that PAG rock can be submerged in the TMF. Due to inefficiencies of segregation and operational upsets, some PAG rock will inevitably be incorporated into the non-PAG dump resulting in lowering of the NP*/AP of the mixture. The effect of PAG rock on NP*/AP was evaluated from the characteristics of PAG and non-PAG rock and the calculated NP*/AP for various proportions of PAG rock. The calculation was performed for three cases (Table 8):

- Average NP* and AP for PAG and non-PAG rock.
- Best case represented by mixing non-PAG and PAG each containing upper confidence limit on mean NP and lower confidence limit on AP.
- Worst case represented by mixing non-PAG and PAG each containing lower confidence limit on mean NP and upper confidence limit on AP.

Table 8. Characteristics of non-PAG and PAG Rock Used to Evaluate Accidental PAG Rock in Segregated Non-PAG Rock

			Case		
Category	**Parameter**	**Unit**	**Best**	**Average**	**Worst**
Non-PAG	NP*	kg $CaCO_3$/t	165	134	103
	AP	kg $CaCO_3$/t	19	27	34
	NP*/AP	Ratio	8.5	5.0	3.0
PAG	NP*	kg $CaCO_3$/t	54	43	33
	AP	kg $CaCO_3$/t	43	55	68
	NP*/AP	Ratio	1.3	0.8	0.5

Source: P:\01_SITES\Harper_Creek\1CY003.000_MLARD_Characterization\2011-07_Static_Testing\3.Interpretation\[MixingNPR_1CY003001_SJD_REV00.xlsx]

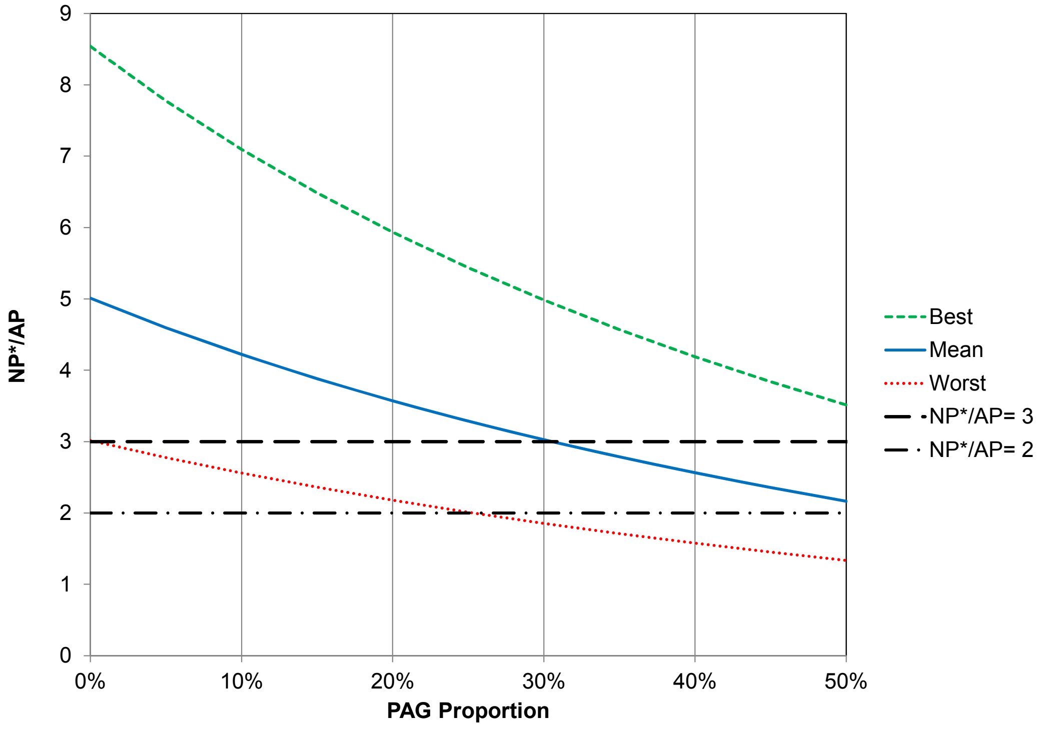

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Figure 13. Effect on NP*/AP PAG Rock Mixed with Non-PAG Rock

The resulting NP*/AP of the mixtures is shown in Figure 13 with reference to NP*/AP of 2 and 3. NP*/AP is considered a reasonable criterion for evaluating the effect of segregation errors because it is above the range of the theoretical balance of NP*/AP (1 to 2). For mean mixtures, PAG proportions less than 30% will have NP*/AP exceeding 3. For the worst case, non-PAG rock has NP*/AP of 3 but 25% PAG rock would be required to reduce mixed NP*/AP to 2. For the best case, the mixed NP*/AP is above 3 for mixtures where PAG dominates non-PAG rock.

The calculation illustrates that for typical classification errors (which can be expected to be less than 10% PAG), typical mixtures can be expected to have NP*/AP well above 3 with upset conditions between 2 and 3. The presence of excess NP in typical non-PAG rock will offset the upset conditions but careful monitoring of segregation success in the ML/ARD Management Plan will be needed.

ARD Potential by Mine Unit

Rock at the Project will be classified during mining according to five categories:

- Ore
- PAG Low Grade Ore.
- Non-PAG Low Grade Ore
- PAG waste rock.
- Non-PAG waste rock

ARD potential characteristics of these categories are shown in Table 9, Table 10 and Table 11 calculated from the exploration database, NP* was obtained from calcium concentrations as described in Section 4.1.2.

Ore has variable ARD potential but is dominantly classified as PAG as indicated by median NP*/AP of 0.98.

Table 9. ARD Potential of Ore

Statistic	NP* kg $CaCO_3$/t	AP kg $CaCO_3$/t	NP*/AP	Cu %
N	3117	3117	3117	3117
Min	1	1.9	0.0044	0.22
P5	6.4	11	0.071	0.23
P50	44	42	0.98	0.36
Mean	53	61	0.86	0.47
P95	130	190	6.7	1.1
Max	260	310	32	8

Source: P:\01_SITES\Harper_Creek\1CY003.001_MLARD_Feasability&EA\120_Additional_ABA_Analysis\2012-10_Exploration_database\[Exploration_Database_Ore-Waste_Stats_1CY003001_SJD_REV00.xlsx]

Low grade ore is defined based on a lower copper concentration of 0.14% and a variable upper concentration that is higher during the earlier stages of mining. The range of copper concentrations that could report to the low grade ore stockpiles is 0.14% to 0.24%. Low grade ore will be segregated based on ARD potential using the same methods as waste rock (see Section 5.1.1). Geochemical characteristics of low grade ore according to ARD potential are shown in Table 10. Sulphide content of PAG low grade ore is greater than non-PAG low grade resulting in higher APs for the former. PAG low grade ore tends to be more sulphidic than PAG waste rock but there is little difference between non-PAG low grade ore and non-PAG waste rock (Table 11).

Table 10. ARD Potential of Low Grade Ore According to PAG and non-PAG Categories

	PAG Low Grade Ore				non-PAG Low Grade Ore			
Statistic	NP* kg $CaCO_3$/t	AP kg $CaCO_3$/t	NP*/AP	Cu %	NP* kg $CaCO_3$/t	AP kg $CaCO_3$/t	NP*/AP	Cu %
Number of Samples	1148	1148	1148	1148	623	623	623	623
Minimum	1	7.2	0.0099	0.14	18	1.3	2	0.14
5th Percentile	5.6	17	0.076	0.14	31	5.3	2.1	0.14
Median	24	48	0.58	0.18	62	15	4.1	0.18
Mean	35	64	0.54	0.18	71	18	4.1	0.18
95th Percentile	99	160	1.8	0.23	140	42	12	0.23
Maximium	230	310	2	0.24	230	79	50	0.24

Source: P:\01_SITES\Harper_Creek\1CY003.001_MLARD_Feasability&EA\120_Additional_ABA_Analysis\2012-10_Exploration_database\[Exploration_Database_Ore-Waste_Stats_1CY003001_SJD_REV00.xlsx]

Waste rock is defined as rock with copper concentrations less than 0.14% and will be segregated into PAG and non-PAG components (see Section 5.1.1). As for Low Grade Ore, PAG rock has distinctively higher sulphur concentrations resulting in higher AP (average 53 kg $CaCO_3$/t) compared to non-PAG rock (average 17 kg $CaCO_3$/t). Copper concentrations are also higher for PAG rock than non-PAG rock.

Table 11. ARD Potential of Waste Rock According to PAG and non-PAG Categories

	PAG Waste Rock				non-PAG Waste Rock			
Statistic	NP* kg $CaCO_3$/t	AP kg $CaCO_3$/t	NP*/AP	Cu %	NP* kg $CaCO_3$/t	AP kg $CaCO_3$/t	NP*/AP	Cu %
Number of Samples	6571	6571	6571	6571	5240	5240	5240	5240
Minimum	1	3.1	0.012	0.001	5.9	0.31	2	0.001
5th Percentile	5.9	11	0.12	0.004	20	1.9	2.2	0.002
Median	23	38	0.74	0.03	64	12	4.6	0.01
Mean	35	53	0.66	0.042	81	17	4.9	0.026
95th Percentile	110	150	1.8	0.12	190	50	37	0.11
Maximum	260	310	2	0.14	360	130	780	0.14

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4.1.5 Metal Leaching Potential

ML potential was evaluated at a screening level by comparison of trace element concentrations with ten times global averages values tabulated by Price (1997). The geological setting indicates the presence of a wide range of rock types varying from mudstone to sandstone protoliths, intermediate volcanic rocks and felsic intrusives. Metamorphism of the geological sequence has occurred obliterating some of the original protoliths. For this reason, “Low Calcium Granite” was selected for comparison because it represents a rock type without extreme average characteristics.

Concentrations of silver, arsenic, cadmium, cobalt, chromium, copper, mercury, molybdenum, nickel, lead, antimony, selenium and zinc were enriched in at least one sample. With the exception of chromium, all these elements are associated with sulphide mineralization and are correlated with sulphur concentrations. Several elements on this list appear enriched but the enrichment factor is relatively low and probably is indicative of rock type variations (for example, Cr, Ni, Co). Enrichment of Ag, As, Cd, Hg, Mo and Sb is also weak. Elements most enriched are copper, selenium and zinc.

4.1.6 Humidity Cells

Characteristics

Characteristics of samples of rock core and barrel test composites in humidity cells are provided in Table 12 and 9, respectively. Complete acid base accounting results for the humidity cell test material are provided in Appendix B4, trace element analysis results are provided in Appendix B5.

Table 12: Characteristics of Rock Core Samples Tested in Humidity Cells

Hole	Sample ID	Humidity Cell	From	To	Lith Code	Rock Type	Ore/Waste	Paste pH	Total S	S-SO$_4$ HCl	AP	NP (Modified)	NP/AP	Cu 4-Acid	Cu AR
			m	m				pH Units	%, S	%, S	kg CaCO$_3$/t	kg CaCO$_3$/t		mg/kg	mg/kg
HC07-21	648	HC-1	148.35	152.67	1	Faults	Waste	8.8	1.12	0.01	34.375	31	0.9	231.90	-
HC07-42	650	HC-2	319.28	322.17	1	Faults	Waste	8.3	1.58	0.02	48.4375	24	0.5	230.30	-
HC07-21	330	HC-3	69.46	83.47	4	Dykes	Waste	8.8	0.37	0.03	11.25	98	8.7	173.10	-
HC07-21	336	HC-4	157.46	172.29	4	Dykes	Waste	8.7	0.53	0.02	16.25	101	6.2	193.70	-
HC10-77	110250-3	HC-5	40	45	5	Volcaniclastics	Waste	8.9	1.93	0.01	59.375	226	3.8	232.00	216
HC10-77	110250-5	HC-6	53	58	5	Volcaniclastics	Waste	8.5	3.3	0.02	100.0	170.0	1.7	350	350
HC10-81	661	HC-7	140.00	145.65	6	Sedimentary	Waste	9.0	0.6	0.01	18.0	9.4	0.53	93	-
HC10-81	662	HC-8	150.50	154.75	6	Sedimentary	Waste	9.2	0.5	-0.01	14.0	14.0	1	66	-
HC07-27	448	HC-9	143.79	155.475	7	Phyllites	Ore	7.7	3.0	0.06	93.0	40.0	0.44	2400	-
HC07-22	359	HC-10	253.55	263.75	7	Phyllites	Waste	8.5	1.7	0.02	51.0	22.0	0.44	180	-
HC06-07	61	HC-11	185.19	196.227	8	Schists (Q eyes)	Waste	5.7	8.1	0.24	240.0	41.0	0.17	710	-
HC07-16	256	HC-12	210.72	220.807	8	Schists (Q eyes)	Waste	8.8	1.9	0.01	58.0	35.0	0.59	260	-
HC10-77	110250-24	HC-13	173	176	8	Schists (Q eyes)	Waste	8.8	4.0	0.02	120.0	140.0	1.2	760	730
HC07-14	180	HC-14	264.2	277.41	9	Schists (No Q eyes)	Waste	8.2	0.9	0.02	27.0	11.0	0.42	680	-
HC07-21	339	HC-15	207.9	217.9	9	Schists (No Q eyes)	Waste	6.1	2.7	0.07	81.0	18.0	0.22	590	-
HC07-14	187	HC-16	373.8	380.09	11	Alteration	Ore	4.0	5.0	0.43	150.0	-0.1	-0.0009	5800	-
HC10-77	110250-39	HC-17	293	297	11	Alteration	Ore	8.6	1.9	0.01	59.0	13.0	0.22	1100	1100
-	Comp #7	HC-19	-	-	7	Phyllites	Waste	9.2	1.0	0.01	32.0	52.0	1.6	-	410
-	Comp #8	HC-20	-	-	8	Schists (Q eyes)	Waste	9.8	3.2	0.011	100.0	120.0	1.2	-	620
-	Comp #8 Duplicate	HC-20D	-	-	8	Schists (Q eyes)	Waste	10.0	1.6	0.012	49.0	91.0	1.9	-	400
-	Comp #9	HC-21	-	-	9	Schists (No Q eyes)	Waste	8.2	1.2	0.0093	37.0	24.0	0.64	-	710
-	Comp #11	HC-22	-	-	11	Alteration	Waste	9.4	2.2	0.011	68.0	21.0	0.31	-	800

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Table 13. Characteristics of Field Barrel Samples.

Barrel ID	Column Number	Rock Type Code	Rock Type	Classification
Comp 7	6	7	Phyllites	Composite
Comp 8	7	8	Schists (no Quartz eyes)	Composite
Comp 9	8	9	Schists (quartz eyes)	Composite
Comp 11	9	11	Alteration	Composite

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Leachate Chemistry and Trends

Charts illustrating results are provided in Appendix C1.

Leachates from rock type humidity cells have shown pHs above 7, ranging up to near 8.5 including HC-16 which had a paste pH of 4 before tested. HC-16 has showed the lowest pH (7 to 7.5) and lowest alkalinity though has not yielded acidic pH after 120 weeks of testing.

Leachates from the tests were dominated by sulphate, bicarbonate, calcium and magnesium. Release rates generally stabilized and were not trending significantly up or down. Initially elevated sulphate release was observed for most tests in the first few weeks probably reflecting flushing of oxidation products accumulated prior to testing. Trace element leaching also stabilized for most tests. Gentle downward trends were apparent for some parameters including antimony, arsenic, lead, molybdenum, selenium, and uranium. Highest leaching rates were shown by HC-16 particularly for cadmium, cobalt, copper, manganese, nickel and zinc which in part may be due to lower pHs.

Charts illustrating results of lab HCT containing the same material as on-site barrel tests are provided in Appendix C4. Leachate trends were similar for all five tests. After 10 weeks, leachate pH varied in the range 7.6 to 8.2 and sulphate release was stable with no clear increasing or decreasing trends. Release of cadmium was distinctive. Greatest release was shown by Comp #9 followed by Comp #11.

4.1.7 Unsaturated Columns

Characteristics of waste rock samples in unsaturated columns are provided in Table 12. Charts illustrating results are provided in Appendix C2. The columns were operated for 1 year and yielded 14 leachates samples on the four-week sampling cycle.

Leachate pHs were highly uniform and varied between 7.8 and 8.4 with no apparent trend. Leachate chemistry was dominated by sulphate, calcium and magnesium. Trace element concentrations tended to be in the low parts per billion range. These results were interpreted in the context of understanding trace element solubility controls (section 5.3.3).

4.1.8 Saturated Columns

Characteristics of rock in saturated columns are provided in Table 15. Charts illustrating results are provided in Appendix C3. These tests were operated for 54 weeks.

Leachate pHs varied from 7.5 to 8.3 initially then converged to 7.8 to 8.2 as the test proceeded. An initial sulphate flush (concentrations up to 530 mg/L) was observed followed by stable sulphate concentrations in the range 1 to 15 mg/L. Stable leachate chemistry was dominated by alkalinity, calcium and magnesium. Trace element concentrations were typically in the sub-parts per billion range. Manganese concentrations ranged up to 0.1 mg/L.

4.1.9 On-Site Kinetic Tests

Characteristics of waste rock samples in barrel tests are provided in Table 12 (HC19 to HC22). Charts illustrating results are shown in Appendix C5. Eight leachate samples were obtained from most tests during the summers of 2012 and 2013. In 2012, samples were collected in June, July and August, and 2013 were collected in July, August, September and October. The results therefore spanned spring and summer conditions, followed by summer and fall. Lowest volumes of water were during the summer months, with higher volumes in the spring and fall. 2013 yielded lower total water volumes than 2012.

Leachate pHs varied from 7.9 to 8.4. Lower pHs were apparent during spring and fall. Concentrations of sulphate, calcium and magnesium were highly correlated. Highest total ions concentrations were apparent in the first samples collected in June 2012, followed by near lowest concentrations in July 2012, and higher concentrations in fall 2012 in parallel with decreasing leachate volumes. Concentrations again decreased in July 2013 then increased and peaked in September 2013 before decreasing again in October 2013. Alkalinity showed mostly decreasing trends through 2012 and erratic to decreasing trends in 2013.

An initial flush was observed for several trace elements. Arsenic, selenium and molybdenum leaching paralleled the major ion trend whereas copper concentrations decreased following the initial flush. Cadmium and zinc showed no indication of an initial flush and increased between 2012 and 2013.

4.1.10 Stored Sample Shake Flask Tests

Three archived samples of rock tested in humidity cells (HC-9, HC-16, HC-17) were leached using shake flask extractions (SFEs) to evaluate dependence of frequency of flushing on oxidation rate. Previous studies (Day et al. 1997) showed that the rate of accumulation on a per unit time basis decreases as the time between flushes increases.

The samples were selected to evaluate the effect of re-processing stockpiled low grade ore on tailings supernatant water chemistry. The three archived samples had been stored for about 2 years and were leached using the SFE method of Price (2009). The resulting “SFE Rate” shown in Table 14 was obtained from the amount of sulphate leached in the SFE divided by the number of weeks. The SFE HC-9 was performed in duplicate and yielded a relative percent difference of 14%.

The calculation indicates that the average rate of sulphate accumulation in storage was over an order of magnitude less than indicated by a conventional humidity cell flushed weekly. Since the experiment consisted of two datapoints, it is not possible to determine how the rate ratio varies with time but since stockpiling times exceed 2 years, the test provides a useful indication of the decrease in accumulation rates. A separate more rigorous experiment has been initiated to evaluate this effect for a number of storage periods. This experiment is ongoing but results were not available to develop the source terms.

Table 14. Oxidation Rate Effects for Stored Samples

HCT	HCT Rate $mgSO_4$/kg/week	SFE Rate $mgSO_4$/kg/week	SFE/HCT
HC-9	13	0.94	0.07
HC-16	16	0.59	0.04
HC-17	7.3	0.50	0.07

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4.2 Tailings

4.2.1 Sulphur Occurrence

The petrographic report, presented in Appendix D1, indicated that the dominant form of sulphur in both rougher and cleaner tailings samples KM 2916-14 was pyrite as liberated grains. Traces of chalcopyrite were also observed along with rare pyrrhotite, galena, sphalerite and molybdenite.

Chemical analysis of tailings, presented in Appendix D2, indicated negligible sulphate in rougher tailings (Table 16). Sulphur concentrations ranging from 0.7% to 1.0% in four samples is dominantly present as pyrite. Cleaner tailings contained detectable levels of sulphate (0.04 to 0.4% as sulphur), which based on depressed paste pHs and elevated paste conductivity likely reflects oxidation of the sample prior to testing. Sulphide sulphur concentrations in the two cleaner tailings samples were 5.8% and 8.9%.

4.2.2 Carbonate Occurrence

Rietveld X-ray diffraction analyses on ore feed and rougher tailings samples indicated the presence of ankerite-dolomite, magnesite and calcite (full results are presented in Appendix D3). Electron probe determinations on individual grains returned compositions consistent with dolomite, magnesite and dolomite (Figure 14). The composition of dolomite approached that of ankerite. Siderite grains were Mg-rich and magnesite grains were Fe-rich resulting in a continuum between siderite and magnesite. The composition of siderite is therefore not iron-rich. Carbonate content exceeds neutralization potential (Figure 15) which is consistent with the presence of iron carbonate minerals.

Using the interpretation of method of Day (2009), it was determined that modified neutralization potential is equivalent to calcium and magnesium carbonate content. The finding was the same for waste rock containing NP less than 100 kg $CaCO_3$/t. Rougher tailing NP is about 75 kg $CaCO_3$/t.

Table 15: Characteristics of Rock Samples in Saturated Columns

Test	Code	Rock Type	Paste pH	CO_2	Total S	S-SO_4 HCl	S-SO4 Na_2CO_3	AP	NP (Modified)	NP/AP
			pH Units	kg $CaCO_3$/t	%, S	%, S	%, S	kg $CaCO_3$/t	kg $CaCO_3$/t	
Column 1	7	Phyllites	8	73	2.4	0.041	0.041	73	32	0.44
Column 2	8/9	Schists	7.3	77	3.4	0.079	0.099	100	34	0.33
Column 3	11	Alteration	5.6	23	3.9	0.28	0.22	110	4.7	0.041

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Table 16: Static Geochemical Characteristics of Tailings Samples

Sample ID	Type	Paste pH	CO_2	Total S	Sulphate	AP	NP	NP/AP	Cu
			kg $CaCO_3$/t	%	%	kg $CaCO_3$/t	kg $CaCO_3$/t		mg/kg
3321-P2 WSB CL TAILS	Cleaner	8	100	5.9	0.04	180	76	0.41	530
KM 2916-14 Cu 1CT	Cleaner	6.8	130	9.3	0.37	290	80	0.28	2900
KM 2916-14 Cu Con	Concentrate	3.5	5.7	34	0.71	1100	-8	-0.008	10000
KM 2916 MCI	Feed	8.9	110	1.8	<0.01	55	72	1.3	3200
2916-12	Rougher	9.4	110	0.79	<0.01	25	76	3.1	250
2916-13	Rougher	9.5	100	0.72	<0.01	23	78	3.5	250
3321-P2 WSB RO TAILS	Rougher	9.3	96	0.96	0.01	30	76	2.5	220
KM 2916-14 Cu Rotl	Rougher	8.7	110	0.87	<0.01	27	73	2.7	190
2916-12 DUP	Rougher	9.4	96	0.77	<0.01	24	74	3.1	240
2916-13 DUP	Rougher	9.4	110	0.76	<0.01	24	77	3.2	250

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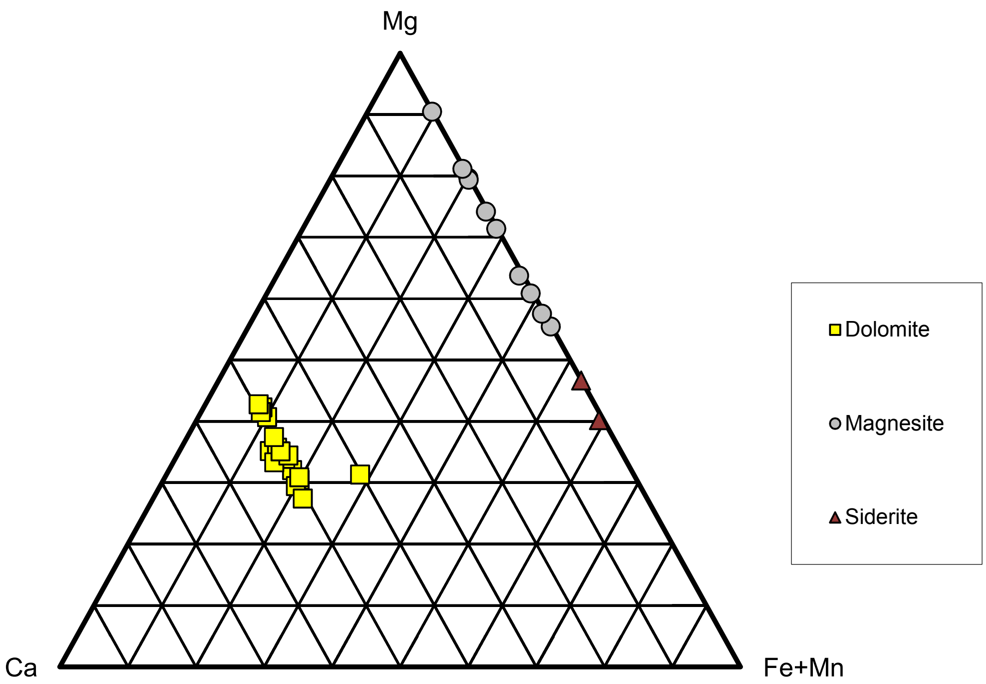

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Figure 14: Composition of Carbonate Grains in Tailings Samples

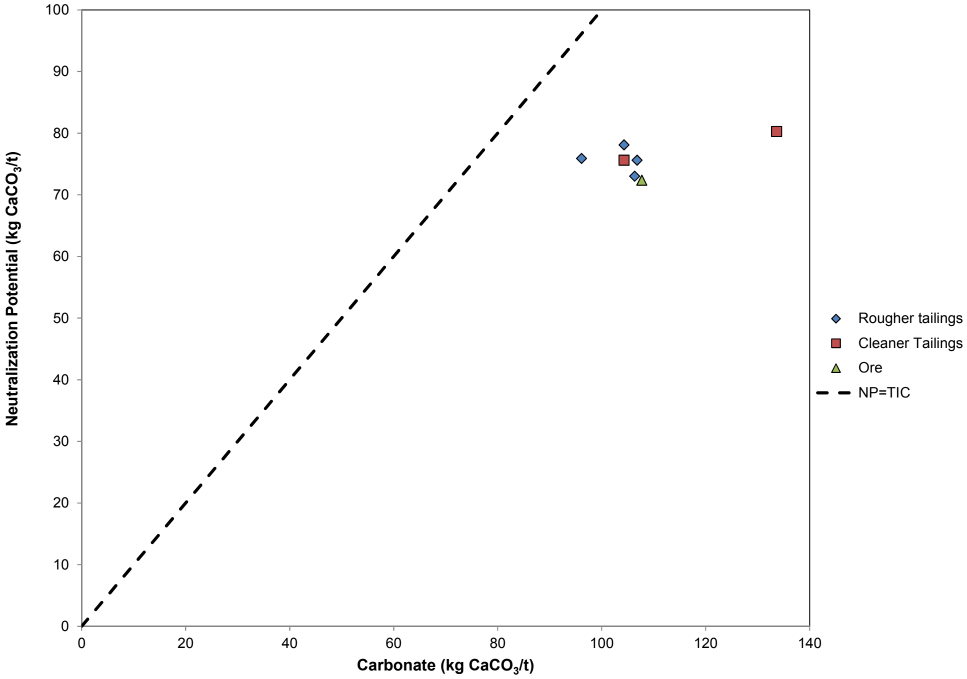

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Figure 15: Carbonate vs NP for Tailings Samples

4.2.3 Trace Element Occurrence

Complete trace element results are presented in Appendix D2. Trace element occurrence is expected to be the same as waste rock based on the similar trace element composition. Comparison of the trace element content of cleaner and rougher tailings shows enriched concentrations of Mo, Co, Cu, Pb, Ni, As and Se in cleaner tailings indicating these elements are associated with pyrite in the cleaner tailings. Ag, Cd, Sb, Tl, Hg and Zn were also enriched in the cleaner tailings but to a lesser degree. The concentrate sample shows enrichment of Cu, Pb, Zn, Ag, Cd, Sb and Se relative to the cleaner tailings indicating that these elements are associated with chalcopyrite, galena and sphalerite in the concentrate. This provides some indication that Mo, Co, Ni and As are associated with pyrite. The relative enrichment of Se in the concentrate implies that it may be associated with chalcopyrite, as well as pyrite, and therefore that copper and selenium leaching may be expected to be linked.

4.2.4 ARD Potential

Figure 16 shows the ARD potential of tailings. The concentration of pyrite by flotation into the cleaner tailings results in these samples being classified as PAG because NP*/AP is well below one. Rougher tailings have NP*/AP between 2.5 and 3.5 and are classified as non-PAG. Combined tailings would have characteristics similar to the ore feed (after removal of sulphur associated with chalcopyrite). The feed in this case had NP*/AP of 1.3 indicating uncertain potential for ARD. These results indicate the benefit of separate disposal of cleaner and rougher tailings.

Figure 17 shows characteristics of tailings size fractions. For three of four samples, higher sulphide concentrations occurred in the two coarser fractions. As a result, NP/AP was near 2 in the coarse fractions. This implies that some degree of sulphide enrichment might be expected on the tailings beaches due to gravitational separation as might be anticipated.

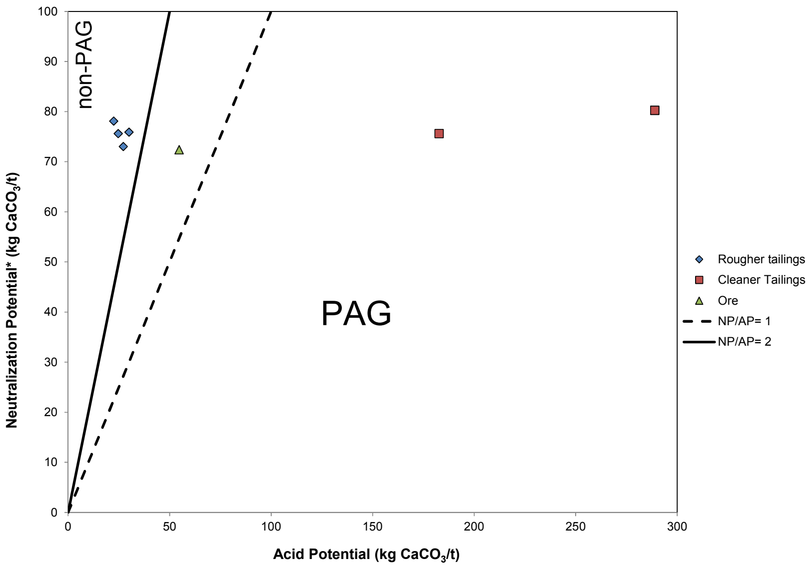

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Figure 16: NP* vs AP for Tailings Samples

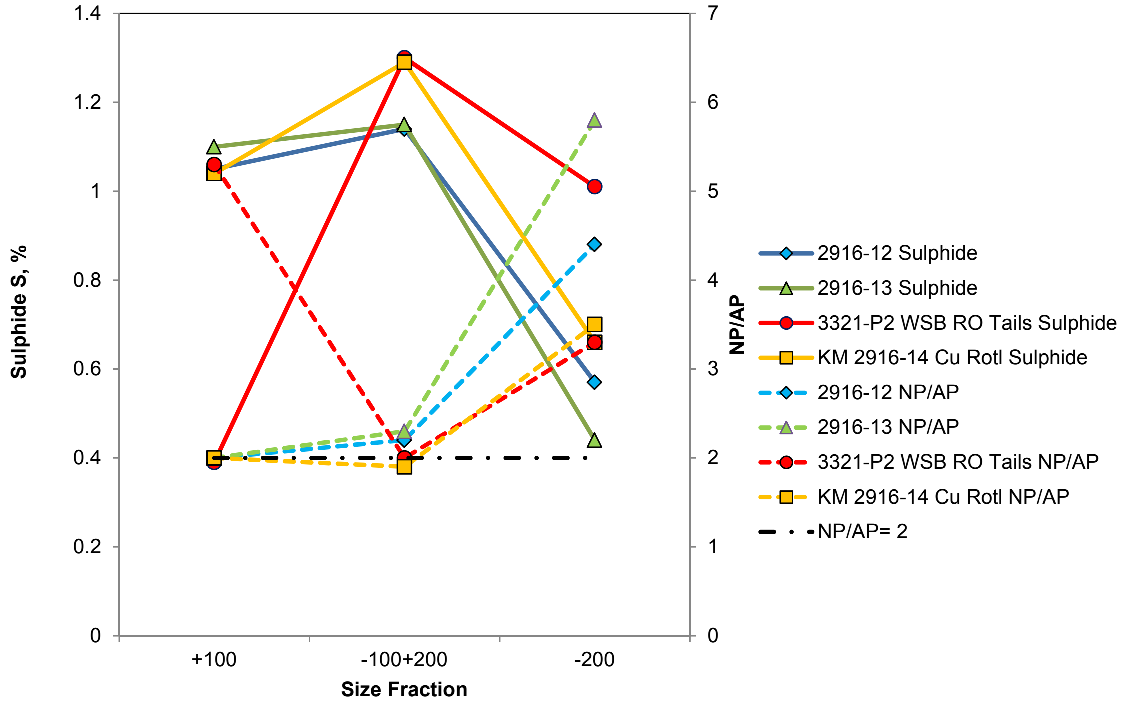

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Figure 17: Characteristics of Tailings Size Fractions

4.2.5 Metal Leaching Potential

Metal leaching potential can be evaluated by comparing concentrations of trace elements in the tailings to global crustal values as described in Section 4.1.5. Flotation of chalcopyrite and pyrite from the rougher tailings reduces the rougher tailings to low concentrations of trace elements indicating relatively low leaching potential. Re-slurring of KM2916-14 CU ROTL with water prior to kinetic testing to address accumulated oxidation products indicating parts per billion and low leachable concentrations of trace elements including arsenic (2 µg/L), copper (1 µg/L), selenium (7 µg/L) and zinc (1 µg/L). Sulphate concentrations were 95 mg/L implying that the sample had oxidized. These concentrations compare to the oxidized cleaner tailings sample with 4000 mg/L sulphate, and relatively elevated copper (23 µg/L), selenium (400 µg/L) and zinc (20 µg/L).

4.2.6 Humidity Cells

Fifty-eight weeks of humidity cell data for KM2916-14 CU ROTL were obtained for sample KM2916-14 CU ROTL. Charts showing results are provided in Appendix D4.

Except for an initial flush pH of 7.7, leachate pHs were between 8.0 and 8.4 with no apparent trend. As a result of the re-slurrying procedure to remove accumulated oxidation products, the initial period of sharp reduction in leaching lasted 2 weeks after which release of sulphate and most other parameters stabilized with some fluctuations. The initial flushing effect was more prolonged for some parameters (antimony, arsenic, fluoride and selenium); however, once the test was completed leachate chemistry had stabilized.

4.2.7 Unsaturated Column

Fifty-two weeks of data were obtained for one saturated column test on KM2916-14 CU ROTL. Charts showing results are provided in Appendix D5.

Overall trends and concentrations were similar to the humidity cell.

4.2.8 Saturated Columns

Fifty-two weeks of data were obtained for saturated column tests on KM2916-14 CU ROTL and KM2916-14 CU 1CT at the time of reporting. Charts showing results are provided in Appendix D5.

Leachate pHs were above 7.8 except for 7.7 in initial flush. The cleaner tailings sample yielded lower pH (no higher than 8.1) compared to pHs up to 8.5 as the test progressed. The cleaner tailings sample also showed fluctuations in sulphate concentrations until week 14 but then declined to converge with the rougher tailings at about 11 to 13 mg/L. Calcium showed similar concentration trends whereas magnesium did not. As the test progressed, higher trace element leaching concentrations were shown by the cleaner tailings samples for cadmium, cobalt, copper, molybdenum, and nickel. In most cases, the differences between rougher and cleaner tailings diminished as the test progressed though for copper the difference stabilized.

4.3 Overburden

Complete static test results are presented in Appendix E. Results are summarized below.

4.3.1 Sulphur Occurrence

Chemical analysis of overburden samples indicated total sulphur levels ranging from 0.01% to 4.5%. The maximum concentration reflects the local presence of mineralized rock in the overburden. Sulphate concentrations were less than detection limits (0.01%) up to 0.03%. Low sulphate concentrations indicated that sulphide is the dominant form of sulphur in these samples. There was a strong correlation between ICP sulphur after 4 acid digestion, and total sulphur (by Leco).

Total sulphur concentrations in the TMF area samples were between 0.01% and 0.21%.

4.3.2 Carbonate Occurrence

Neutralization potential in the TMF area samples was generally low, with the majority (70%) of samples having NP less than 10 kg $CaCO_3$/t. NP ranged from -8 kg $CaCO_3$/t to 120 kg $CaCO_3$/t in the TMF footprint. Neutralization potential and carbonate content were strongly correlated.

4.3.3 Trace Element Occurrence

A correlation matrix for trace elements was calculated to evaluate associations between elements and possible linkages to mineralogical hosts. Significant relationships were assessed from a correlation coefficient of 0.21 (1% significance level) but strong correlations were determined from much higher correlation coefficients. The matrix indicated that:

- Sulphur concentrations were correlated with Zn, Au and Cd (r>0.7).
- Cd was strongly correlated with zinc (r=0.9) and arsenic (0.8).

4.3.4 ARD Potential

Acid-base accounting of overburden showed that the majority of samples had very low AP (less than 10 kg $CaCO_3$/t) and sufficient NP to be classified as non-PAG (Figure 18). The exceptions were five samples from drilling in the pit area which had more than 1% total sulphur and were classified as barely non-PAG to PAG. These samples were obtained in weathered bedrock but showed little indication of weathering (for example, depressed pH and sulphate) despite presumably being weathered for millennia followed the last glaciation. Depressed paste pHs (including three results between pH of 4.4 and 5) were obtained from five samples; however, these samples contained negligible sulphur and acid potential was very low. These acidic pHs had low associated conductivity, and negligible sulphate and carbonate indicating that the pHs reflect weak buffering of the deionized water in the procedure by natural alumino-silicate weathering products. All acidic samples were obtained from near surface and therefore probably reflect weathering of soils rather than sulphide oxidation.

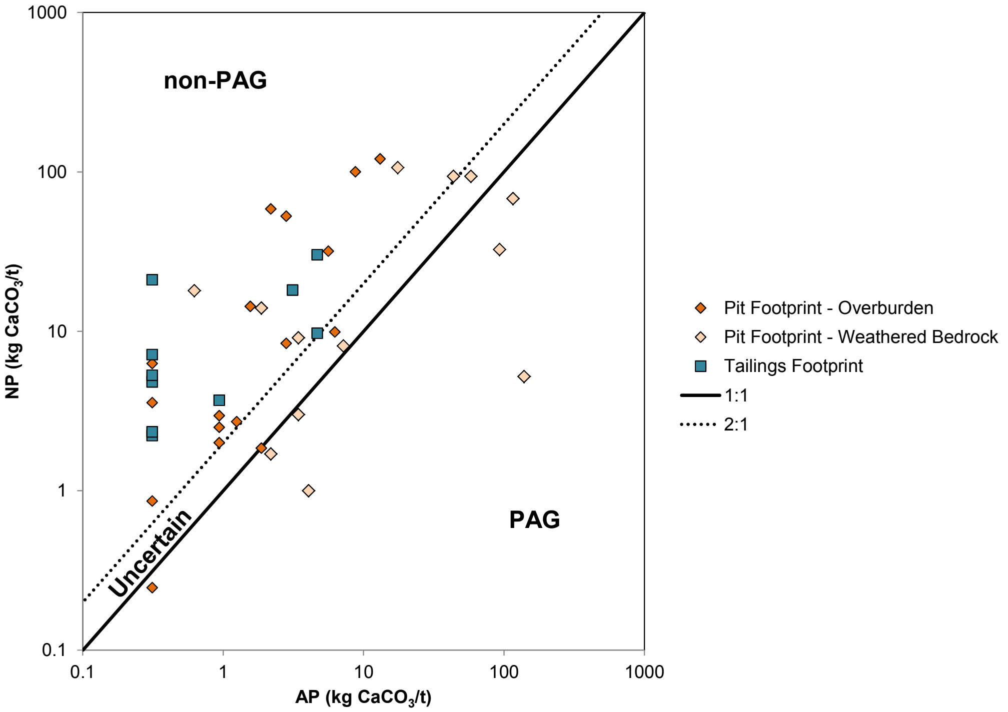

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Figure 18: NP vs AP for Overburden Samples from the Pit area and Tailings footprint area

4.3.5 ML Potential

Trace element analyses showed that overburden in the pit area was enriched with similar elements to the deposit host rocks (arsenic, copper, molybdenum, selenium and zinc). Correlations of bulk characteristics to leachable concentrations were only apparent for copper, selenium and zinc.

Overburden in the TMF area showed widespread lead enrichment (average of 710 mg/kg compared to assumed global average of 7 mg/kg). Enrichment of arsenic was also apparent (average 80 mg/kg compared to global average of 1 to 3 mg/kg). However, there is no indication that these elevated concentrations correlate with elevated leachability based on shake flask extraction, nor is there a geological explanation such as the documented presence of lead mineralization.

Mineralogical characterization for TMF area overburden showed the presence of metallic lead particles (Appendix E3). Since native lead is very rare as a natural mineral and not expected to occur in overburden, the presence of lead is believed to indicate the samples were contaminated, and therefore that no additional testing beyond leach tests was warranted.

4.4 Quarry

Complete static test results are presented in Appendix F. Results are summarized below.

4.4.1 Sulphur Occurrence

Six quarry samples were selected across various depths in two drill holes to characterize quarry material. Chemical analysis indicates samples have low sulphur levels, ranging between 0.03 and 0.04%. Sulphate levels were below the detection limit of 0.01%, indicating sulphide sulphur is the dominant form in these samples.

4.4.2 Carbonate Occurrence

Comparison of neutralization potential and TIC (both in units of kg $CaCO_3$/t) indicated that the NP in the quarry samples exceeds TIC by a factor of about 1.5 (Figure 19).

4.4.3 ARD Potential

These samples both have low levels of both neutralization potential and acid generation potential. All six samples were classified as non-PAG (Figure 20).

4.4.4 ML Potential

Metal leaching potential can be evaluated by comparing metal analysis results to global crustal averages. When compared to low Ca granite, chromium was the only element elevated in the quarry samples.

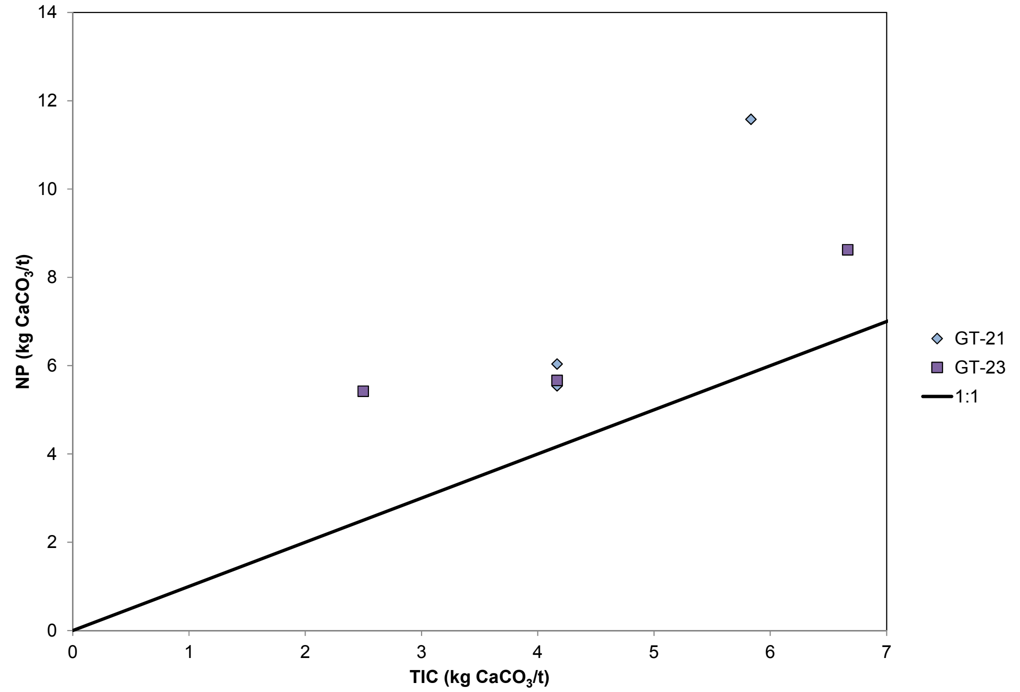

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Figure 19: Comparison of NP and TIC in quarry samples

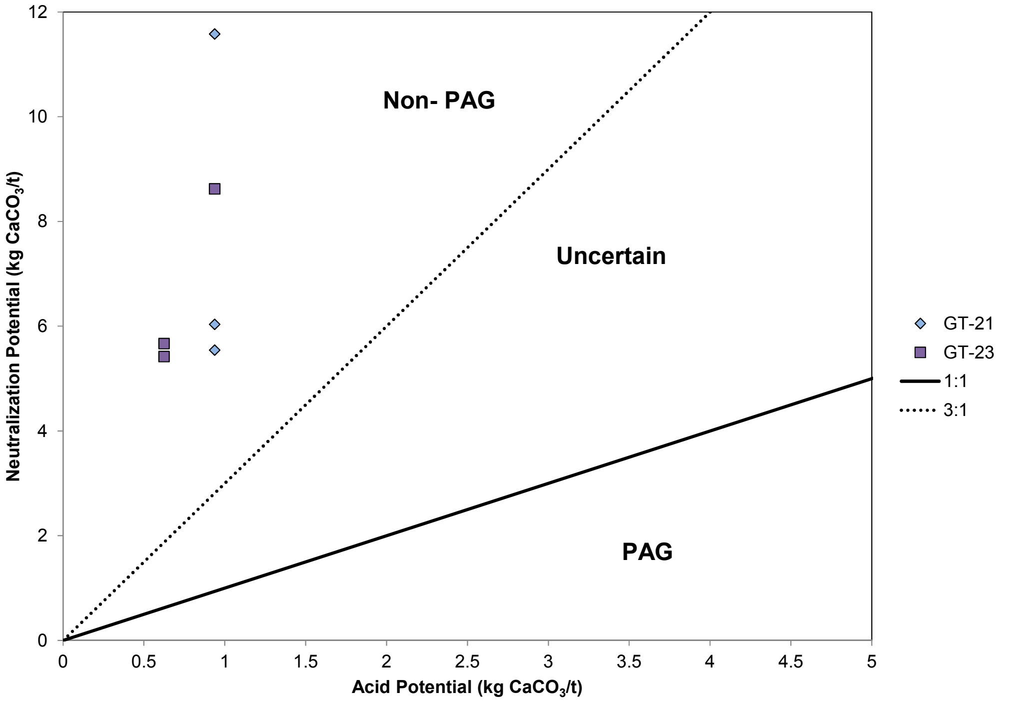

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Figure 20: NP vs AP for Quarry Samples

5 Source Term Development

5.1 Introduction

5.1.1 Definition of “Source Term”

“Source terms” refers to predicted concentrations for waters in contact with various types of geologically-sourced wastes and surfaces under the expected disposal conditions at the site. For example, the source term for a waste rock storage area is the concentration found in pore waters within the facilities. These predictions become inputs (or terms) to the overall site wide water and load balance model used to assess potential effects of the project on the receiving environment. This model has been developed by KP and is not presented here.

5.1.2 Evaluation of Uncertainty

Source terms are calculated based on information acquired from various sources including results of laboratory testing and data from other sites. As such, the calculations contain uncertainty which needs to be reflected in the subsequent affects assessment for the project. The approach used for the Harper Creek Project reflects typical practice in B.C. by modelling to reflect two conditions or cases. The overall intent of source term predictions is to err on the side of conservatism. The first case (“Expected Case”) uses typical inputs derived from what is considered to be the most likely chemistry of contact water. A second case (“Reasonable Upper Limit Case”) considers upper limit uncertainty in the inputs that are intended to bound calculated concentrations in the source terms.

Two inputs to the source terms were considered to be primary sources of uncertainty: the rates at which materials weather converting relatively insoluble minerals to more soluble weathering products, and the uncertainty in the solubility of the resulting weathering products. Variability of rates has been determined based on laboratory measurements (Section 5.3.2). For the majority of heavy metal parameters, the latter is strongly controlled by pH because the solubility reactions are of the form (for example, iron at lower pHs):

$$Fe^{3+} + 3H_2O \rightarrow Fe(OH)_3 + 3H^+$$

The mathematical equation that describes this reaction indicates that iron concentrations are a function of pH and that one unit decrease in pH can result in a three orders-of-magnitude increase in iron concentrations (depending on pH). The pH of contact waters therefore has a strong influence on metal concentrations and the source of uncertainty is the pH of the pore waters. Consideration of pore water pH is discussed in Section 5.2.6.

5.1.3 Use of Source Terms

Outputs from source term calculations should be used in a relative rather than absolute sense when drawing conclusions about potential effects on the receiving environment, and should be used primarily to inform Project design and the need to add mitigation measures. In general:

- If the Expected Case source terms result in predicted concentrations above guideline levels in the receiving environment, the result is considered likely in the project configuration and additional mitigation, management action or contingencies are probably warranted.
- In contrast, if predicted concentrations are slightly above guidelines for the Reasonable Upper Limit Case but well below guidelines for the Expected Case, it is considered low probability that actual concentrations will exceed water guidelines and therefore significant design modifications are probably not warranted. Instead, the project should include the ability to monitor contact water and compare predicted versus observed concentrations early in the mine operation.

Regardless of case, the terms were calculated as annual average dissolved concentrations for each year of the mine and do not consider seasonal transient effects such as dilution in contact waters that commonly occurs during snow melt events. Predicted concentrations were calculated on a constant annual basis which is considered conservative. When these predictions are coupled with the seasonal hydrograph, elevated loads occur during spring runoff and other high flow events which are commonly observed at mine sites.

The resulting source term concentrations are largely developed for input into the water quality model to assess potential for impacts on the receiving environment in respect of meeting relevant water quality guidelines and therefore the resulting solution chemistries do not have closed ion balances.

5.1.4 Conservatism in Source Terms

The main sources of conservatism in the current source term predictions are as follows:

- Seasonal variations were not considered. The calculations assume constant concentrations through the year but concentrations are expected to decrease during flow events. Loadings are expected to increase during the freshet but the model method probably over-estimates the effect.
- Solubility limits are assumed to be reached. In reality, lower concentrations may be reached due to other processes.
- Wastes are assumed to have unlimited oxygen availability for oxidation. Oxygen limitations may occur due to consumption of oxygen and slow re-supply due to the physical properties of the wastes. This is particularly the case for tailings but also could be a factor in limiting the weathering of waste rock.
- Understanding of the effect on age of stockpiling on accumulation of soluble load is limited to a test that evaluated the mechanism for rock that was 2 years old. Older rock can be expected to accumulate load more slowly but this could not be quantified in the source terms.

The Reasonable Upper Limit Case incorporates a greater degree of conservatism because it assumes that weathering rates are near maximum observed and solubility limits are controlled by lower pHs.

5.2 Source Term Methods

The following sections describe calculation methods for each term. Inputs are described in Section 6.3.

5.2.1 Unsaturated Sources

Unsaturated terms apply to the non-PAG waste rock piles, non-PAG low grade ore stockpiles, PAG low grade ore stockpile, the PAG waste rock stockpile before submergence, soil and overburden stockpiles, unsaturated tailings dams constructed from rock and overburden, pit walls and quarry walls.

Source terms for unsaturated waste rock sources were calculated using scaling methods to convert laboratory measured weathering rates to field scale using scaling factors followed by calculation of load released and concentrations from waste scheduling and climatic information supplied by HCMC and KP.

The steps in the calculation are:

- Obtain input weathering rates (R_P) for each parameter (P) from humidity cells.
- Estimate field scale weathering rates ($R_P^{'}$) using scaling factors to account for lower temperatures at site (k_t), particle size differences between laboratory rock and run of mine waste (k_D) and different oxygen availability (k_{O2}):

$$R_p^{'} = R_p.k_t.k_D.k_{O2}$$

- Calculate weathering rate ($R^{''}_p(t)$) for mass of the facility at time t (M(t)).

$$R_p^{''}(t) = R^{'}_P.M(t)$$

- Calculate potential load available from flushing ($L_p(t)$) based on the assumed fraction of rock (k_c) flushed by infiltration:

$$L_p(t) = R^{''}_P(t).k_c$$

- Dissolve the load flushed into the infiltration volume (I(t)) to obtain potential concentration in contact water ($C_p(t)$):

$$C_p(t) = L_p(t)/I(t)$$

- Compare $C_p(t)$ to expected solubility limits for parameter p (C_p’) and use the following rules to calculate the source term at time t for parameter p (C_p”(t):

$$\text{If } C_p(t)>C_p' \text{ then } C_p''(t) = C_p'$$

$$\text{If } C_p(t)<C_p' \text{ then } C_p''(t) = C_p(t)$$

Uncertainty in the calculation is considered for R_p and C_p’. The Expected Case was based on median weathering rates and assumption that pore water pH exceeds 8 based on results of the field barrels. The potential for depressed pH (7 to 8) was used for the Reasonable Upper Limit Case. This results in order of magnitude variation in $C_p^{'}$.

The scaling factors were not varied because they are well-bounded by experience and were found to have limited control on overall predictions when compared to the assigned variation in $C_p^{'}$.

For the purpose of source term development the following assumptions regarding onset of leaching effects were made:

- All unsaturated non-PAG sources begin leaching at non-acidic pH (as defined by the case being tested) and contributing to downgradient water quality as soon as they are placed in their respective stockpiles. This includes non-PAG waste rock, non-PAG overburden, non-PAG low grade ore and non-PAG pit walls.
- All unsaturated PAG sources also begin leaching at non-acidic pH (as defined by the case being tested) and contributing to downgradient water quality as soon as they are placed in their respective stockpiles. This includes PAG waste rock and PAG low grade ore. These two sources are also assumed to be managed to prevent the initiation of acidic conditions and therefore they are not assumed to become acidic. PAG waste rock will be submerged and encapsulated in tailings in the TMF and PAG low grade ore will be processed resulting in pyrite being extracted and disposed sub-aqueously as cleaner tailings.

5.2.2 PAG Pit Walls

The mine has been designed to minimize the onset of acidic conditions by submerging reactive wastes. However, based on block modelling, 69% of the permanently exposed pit highwall can be expected to acidify at some time in the distant future due to depletion of neutralization potential. Onset of acidification will occur at different times depending on the NP/AP of each component of the wall, and intensity of acid generation will depend on the amount of sulphide remaining when NP is depleted and acid generation begins.

The timing to onset of acid generation was calculated from NP/AP:

$$t_{onset} = \frac{\left({NP^*}/{AP^*} \right)_0}{k.\left({NP^*}/{AP^*} \right)_{crit}}$$

Where k is the rate constant for the oxidation of pyrite (assumed to be zero order), $(NP^*/AP^*)_{crit}$ is the relative rate at which NP is depleted relative to AP, and $(NP^*/AP^*)_0$ is the initial NP*/AP* of the rock. Further detail on the calculation of ARD onset is provided in the ML/ARD Management Plan.

The rate of acid generation as acidification occurs was calculated from:

$$R_p^{'} \cdot \left(\frac{AP^* - {NP^*}\big/{\left(NP^*/AP^*\right)_{crit}}}{AP^*} \right) .k_a$$

Assuming that depletion of sulphur in parallel with NP will occur, k_a represents acceleration of oxidation rates due to acidification. As AP depletes, oxidation rates will decrease.

The total acid load released from the highwall was calculated as the sum of the acid load from each component of the wall represented by the distribution of NP/AP indicated by the ABA database.

5.2.3 Flushing Waste Rock Sources

Flushing terms were needed for PAG waste rock as it is submerged by the rising water level in the TMF, ore and low grade ore as it is processed in the mill and pit walls as they are inundated by water following completion of mining. Weathering minerals accumulated in these materials but not flushed by infiltrating waters will be dissolved in the water contacting the waste. Flushed load originates from two sources:

- Load generated outside zones unflushed by infiltrating water ($\sum L_u$). This is calculated from the field weathering rate ($R_p^{'}$) the decrease in rate expected due to lack of flushing (k_u), the quantity of material not flushed by infiltration ($M(t).(1-k_c)$), and total time ($\sum t$) before flushing by inundating water occurs:

$$\sum L_{p,u} = R_p^{'}.k_u.M(t).(1-k_c).\sum t$$

- Load generated inside zones flushed by infiltrating water ($\sum L_f$) but is not flushed due to a solubility limit being reached. This load is the difference between total potential load ($\sum L_p(t)$) and actual load removed by infiltration ($\sum C_p(t).\mathbf{I}(t)$):

$$L_{,p,f} = (\sum L_p(t)) - \sum C_p(t).\mathbf{I}(t)$$

The resulting total load is ($L_{p,u}+L_{p,f}$) is then dissolved in the volume of contact water (V) and compared to solubility limits to obtain the concentration in the flushing water.

$$C_p(t) = (L_{p,u}+L_{p,f})/V$$

$$\text{If } C_p(t)>C_p' \text{ then } C_p''(t) = C_p'$$

$$\text{If } C_p(t)<C_p' \text{ then } C_p''(t) = C_p(t)$$

A factor to account for decreasing rates of oxidation (L_p) due to reduced flushing was applied based on the findings in Section 4.1.10.

5.2.4 Saturated (Subaqueous) Sources

Subaqueous terms were needed primarily for PAG rock, cleaner tailings and rougher tailings in the TMF. Contact water chemistry in this disposal condition is expected to represent equilibration of somewhat soluble minerals with pore waters. The main minerals expected to dissolve are carbonates because sulphates are assumed to be removed for the flushing term. Oxygen concentrations are not expected to be significant and therefore oxidation of sulphide minerals is not expected to occur at significant rates.

Subaqueous terms were derived directly from column tests performed on each type of waste material.

The conceptual geochemical model recognizes the potential for reductive dissolution of oxidation products in tailings resulting from processing of low grade ore. The potential reductant in this case would be dissolved organic carbon originating from process reagents. This process is considered unlikely to be significant for water quality management, however, due to the low levels of process reagents used, their tendency to rapidly bio-degrade and the fact that the primary reagents to be used (a collector and a frother), in keeping with their intended purpose, selectively report with the concentrate. As a result of the limited information available in the literature, the potential for such an effect is not easily quantifiable for the purpose of the EA with current knowledge.

5.2.5 Explosive Residue Terms

Explosive residue terms refer to loadings derived from explosives use. Leaching of explosive residues to yield soluble nitrate, nitrite and ammonia was calculated using the Ferguson and Leask (1988) method.

5.2.6 Consideration of Uncertainty

Sources of Uncertainty

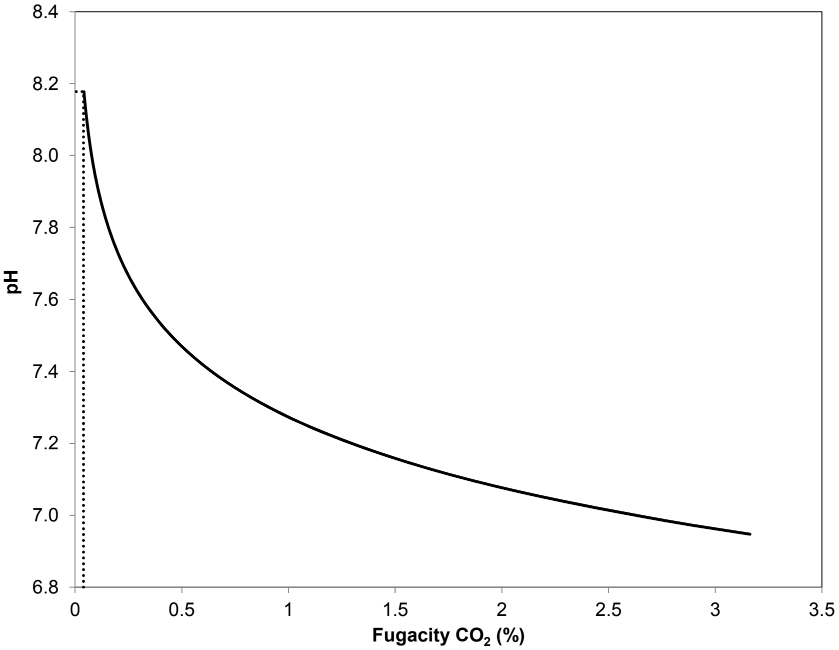
As described in Section 5.1.2, uncertainty in the calculations was considered for R_p and C_p’. The Expected Case was based on median weathering rates whereas the Reasonable Upper Limit Case was based on 95th percentile rates. The scaling factors were not varied because they are well-bounded by experience and were found to have limited control on overall predictions when compared to the assigned variation in C_p’ as described below.

Variation in $C_p^{'}$

Variation in solubility of most trace elements (C_p’) is driven by pH; hence the uncertainty in pH needs to be considered. For systems in which carbonate minerals are present, pH is controlled by the partial pressure of carbon dioxide because it dissolves in water to form carbonic acid which in dissociates to release protons and bicarbonate. pH is buffered by the reaction with carbonate minerals. Carbon dioxide is produced when acid generated by sulphide oxidation is neutralized by carbonate minerals. It is also introduced from the atmosphere.

Figure 21 shows pH as a function of CO_2 with calcite present to buffer acidity. The graph shows that pHs near atmospheric CO_2 partial pressures (about 0.04% CO_2) pH is about 8.2. As CO_2 increases, pHs decrease most rapidly at first but calcite buffering causes the curve to flatten. A nearly two orders of magnitude increase in CO_2 pressure is needed to produce a one unit drop in pH.

For the purpose of defining pH for the Expected and Reasonable Upper Limit Cases, the Expected case was set as pH 8 and above which indicates exchange of CO_2 generated by carbonate neutralization of acidity with the atmosphere. The Reasonable Upper Limit Case was set based on a pH range of 7 to 8 which indicates slower exchange of CO_2 with the atmosphere.



C:\Users\sday\Documents\!Geochemistry_Toolbox\2007_Mind_Maps\5.Tools\Mineral Solubilities\GWB\[CO2.xlsx]

Figure 21. Modelled pH as a Function of Carbon Dioxide Fugacity (~Partial Pressure) Buffered by Calcite Dissolution at 5°C

Refinement of Reasonable Upper Limit Case

Initially, a lower limit pH of 7.0 was used to define an upper limit case but this was considered to have an unrealistically low probability of occurring. It requires that PAG rock is widely acidifying and causing high demand for neutralization by carbonate minerals thereby generating high carbon dioxide partial pressures, or that the waste rock is very fine grained and pore gases were not exchanging with the atmosphere.

At the Project, waste rock will be managed to limit inclusion of PAG rock in the non-PAG rock, and PAG rock will be submerged before it acidifies. Grain scale acidification will occur where sulphide minerals are oxidizing but propagation of acidic zones beyond this scale is not expected.

To provide a refined estimate of the range of pHs that can be expected for waste rock weathering under dominantly non-acidic conditions (i.e. lacking widespread acidifying PAG rock), drainage data for near source seeps at three interior copper mines in B.C were compiled and interpreted with the permission of their operators. These sites were selected because SRK is directly familiar with the sampling locations and reliable data exist on the characteristics of the source materials including the proportion of PAG and non-PAG components. Characteristics of the three sites are provided in Table 17.

Table 17. Characteristics of Sites Used to Define pH Distribution in Seeps

Site	Geological Setting	Dominant Sulphide Minerals	Approx Average Sulphur Content (%)	Carbonate Minerals	Number of Seeps	Number of Measurements
Copper Mountain	Intermediate to mafic volcanics, alkalic intrusives	Pyrite, chalcopyrite, bornite	0.5	Calcite, dolomite	7	75
Highland Valley Copper	Intermediate intrusives	Pyrite, chalcopyrite, bornite, molybdenite	0.3	Calcite	3	68
Mount Polley	Alkalic intrusives	Pyrite, chalcopyrite	0.5	Calcite	3	54
Harper Creek	Felsic to intermediate meta-volcanics	Pyrite, pyrrhotite, bornite	1	Calcite, ankerite, dolomite, siderite	-	-

Source: P:\01_SITES\Harper_Creek\1CY003.001_MLARD_Feasability&EA\900_Water_Quality_Predictions\2014-07_Revized_Terms\pCO2_Assessment\[AnalogData_1CY003001_SJD_REV01.xlsx]

At Copper Mountain Mine, three seeps drain to Wolfe Creek and originate from contact between recent segregated non-PAG waste rock and legacy mixed waste rock. The latter is dominantly non-PAG but will contain some PAG components because waste rock was not segregated based on ARD potential prior to current operations. Supporting data on this site can be found in SRK (2009) prepared in support of the Mines Act Permit Amendment Application submitted to re-start operations.

Most seepage data for Highland Valley Copper Mine are from the Bethlehem area. Waste rock was not segregated based on ARD potential; however, the vast majority of waste rock is non-PAG with only minor PAG components in the Bethlehem area (SRK 2014).

Mount Polley Mine waste rock seeps reflect dominantly non-PAG rock because the mine segregates waste rock based on ARD potential.

General chemistry for the waters was evaluated to investigate the degree to which the waters represent contact with waste rock. Sulphate concentrations ranged up to 1400 mg/L. For all sites (for number of measurements indicated in Table 17, maximum gypsum saturation indices (calculated using PHREEQC (version 2.18.3, with MINTEQ version 4 database) were -0.2 and maximum partial carbon dioxide pressures (pCO_2) were $10^{-2.3}$ atm (compared to $10^{-3.4}$ atm for atmospheric conditions). These modelled results implied that the waters have a strong influence from contact with waste rock containing oxidizing pyrite and neutralization of acid by carbonate minerals. The elevated pCO_2 levels also imply that equilibration with the atmosphere has not fully occurred. This remains a possible concern for the use of these pHs values to represent contact water because carbon dioxide pore gas concentrations within the rock dump have not been measured. However, off-gassing in actual drainage from waste rock at the Project would occur and cause pHs to increase.

Figure 22 shows cumulative histograms of pH in seep waters. For Copper Mountain Mine, the pH distribution is from field readings whereas for Highland Valley Copper and Mount Polley the data are from laboratory analyses. pH distributions are very similar. Medians varied from 8.0 to 8.2. Copper Mountain showed the narrowest variation. The reasons for subtle variability are not known. Seasonal variation was eliminated as a potential cause but other factors such as the effect of dilution by non-contact waters, pH changes in transportation, and differences in analytical protocols may all be involved. The dataset as a whole indicates a 5th percentile, median and 95th percentile pHs 7.6, 8.1 and 8.4.

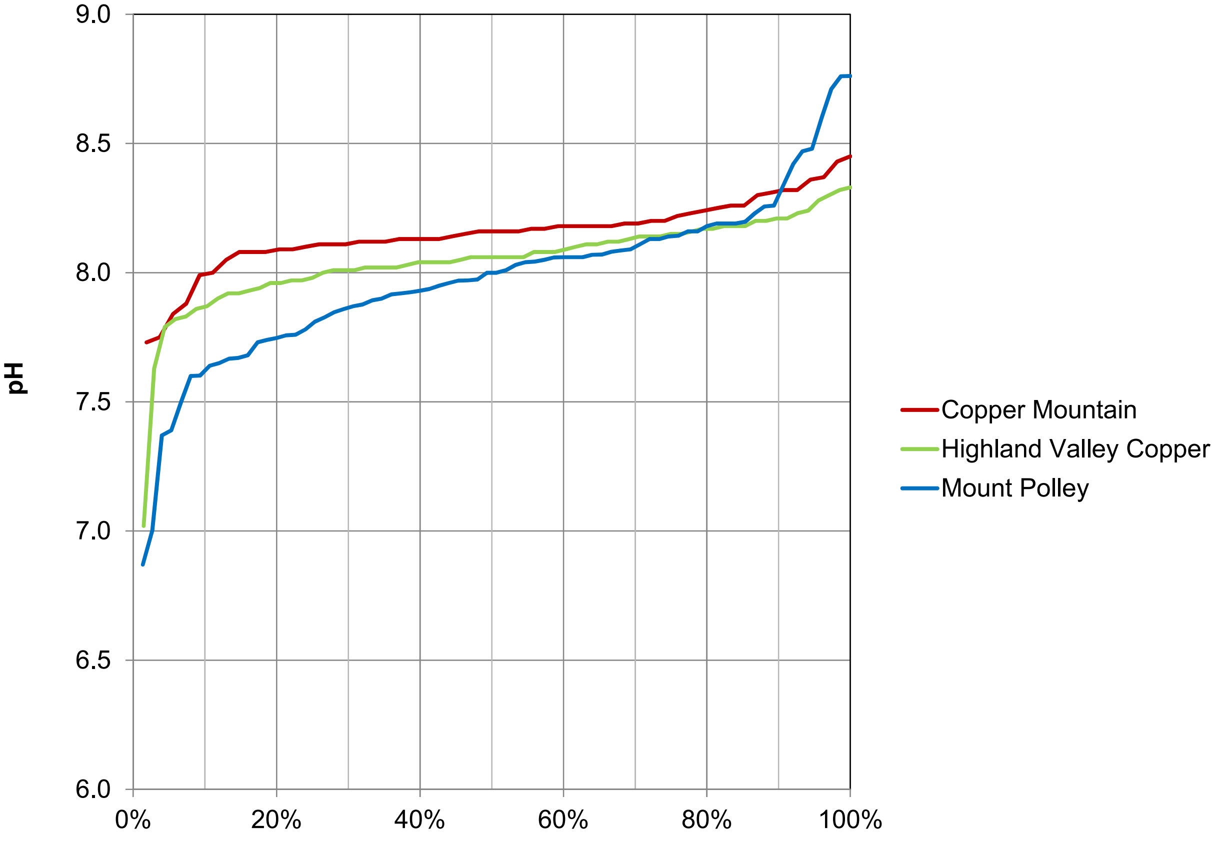


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Figure 22. Distribution of pH in Waste Rock Seep Waters from Copper Mountain, Highland Valley Copper and Mount Polley.

For subsequent source term derivation, the Expected Case was defined using a pH range of 7 to 8, and the Reasonable Upper Limit Case pH was set at 7.6. For definition of C_p’, this case can be considered to represent a 1 in 20 probability of exceeding due to lower pHs (Figure 22).

5.3 Inputs

5.3.1 Summary

Table 18 and Table 19 summarize input terms for the Expected Case and Reasonable Upper Limit Cases. The inputs for the initial upper limit based on a pH range of 7 to 8 is provided in Appendix G. The following sections describe the source of each input.

5.3.2 Rates

Inputs were calculated as averages from humidity cell rates following the initial flush of soluble weathering products accumulated prior to testing of the sample. Figure 23 shows that between the start of the tests and about 10 weeks, sulphate release rates decreased rapidly but after 10 weeks most of the tests were showing stable release rates, For each test, the beginning of stable sulphate release rates was estimated based on the end of the rapidly decreasing trend. Average release rates were calculated for all parameters beginning at that time.

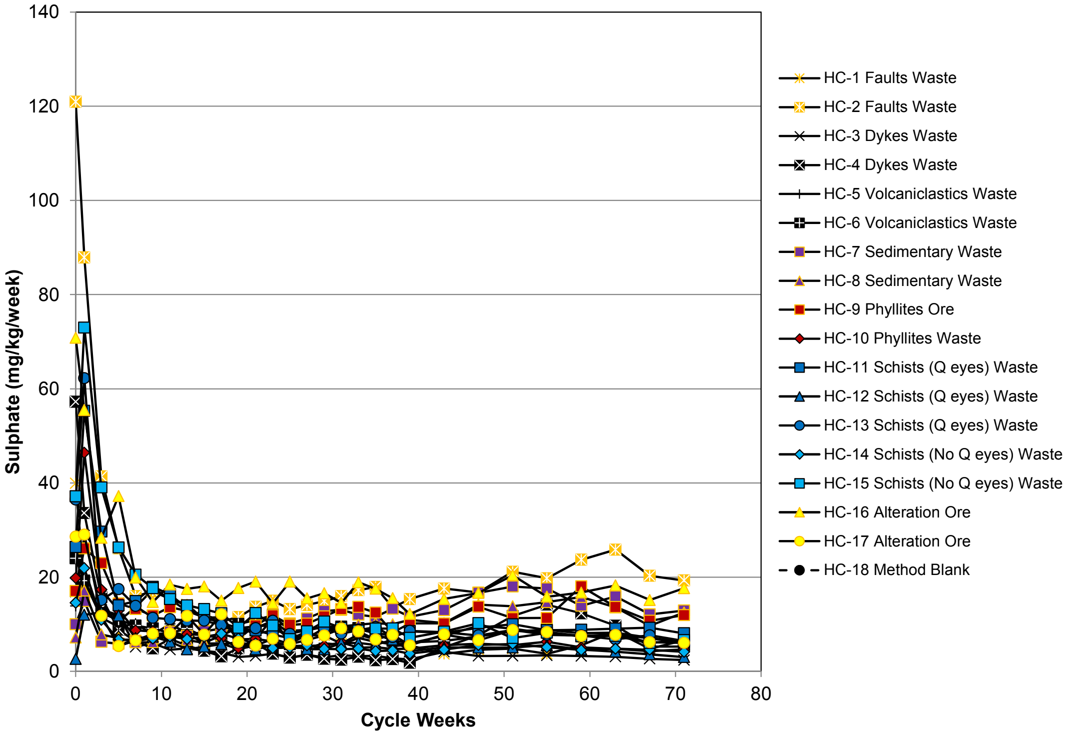

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Figure 23. Sulphate Release Rates Shown by Humidity Cells

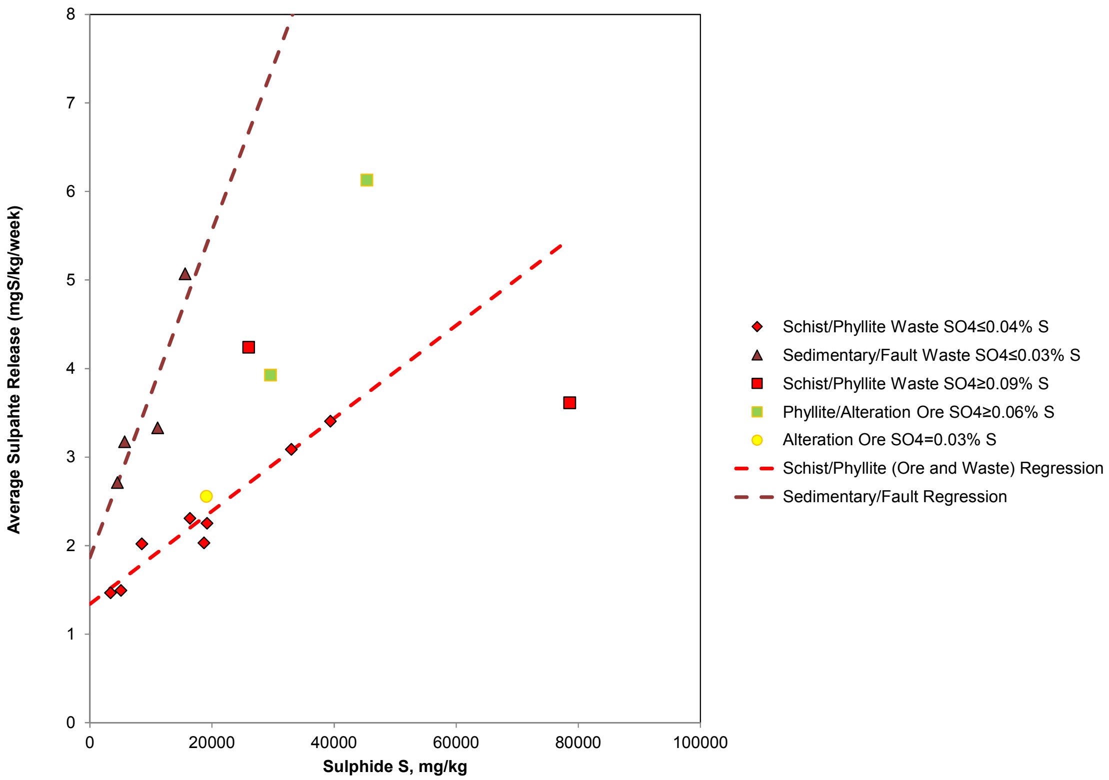
Waste rock and ore rates were calculated based on samples classified as each type of material. The 50th and 95th percentile of all rates obtained for humidity cells of each material type were calculated without considering the relative proportions of different material types. This approach was used because the distribution and proportion of rock types cannot practically be modelled for the Project, and subsequent source terms were primarily determined to be based on solubility controls (Section 5.3.3) rather than these rates.

For waste rock and low grade ore, separate source terms were not calculated for the non-PAG and PAG categories. While it is apparent that higher sulphur content (i.e. AP) is correlated with higher ARD potential (Table 10 and Table 11), oxidations rates indicated by humidity cells were not correlated with sulphide content unless rock type was considered a variable (Figure 24), but rock type is not modelled separately and therefore cannot be correlated to material type. As a result, oxidation rates are not correlated to metal release. Copper release is strongly correlated with copper content of the rock which supports use of separate source terms for low grade ore and waste rock (Figure 24).

Quarry samples were not tested in humidity cells due to the very low sulphur content and trace element content near global average values. The input rate for quarry type materials was calculated as 1/100th the rate for humidity cells based on the sulphur content of quarry rock being about two orders of magnitude lower than the mineralized rock. The same rate was used as the basis for overburden samples.

For tailings, the average rate for the humidity cell sample was used in the calculation. As one sample was tested, no basis for evaluating uncertainty in rate was available; however, the solubility controls were found to determine release so that uncertainty in the rate was unimportant.

The resulting rates are shown in Table 20.



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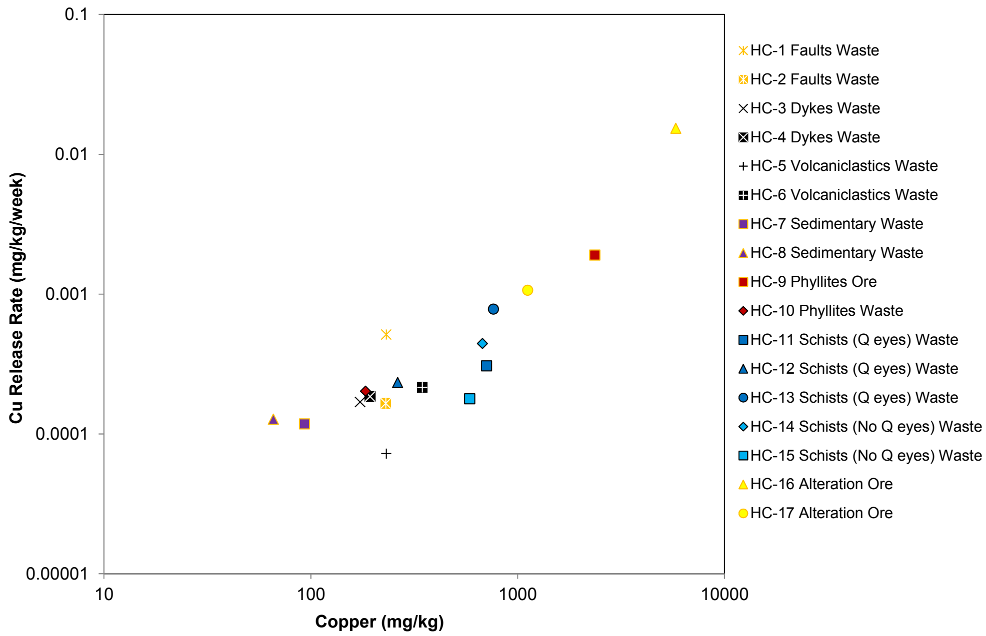

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Figure 24. Comparison of Sulphide Content with Sulphate Release (by rock type) and Copper Content with Copper Release

Table 18: Summary of Expected Case Inputs by Term

Term	Term Location	Material Type	Term Type	Rate Type		Solubility Control		Infiltration	Rate Scaling			Reporting Units
				Type	Statistic	Type	P95 Statistic	mm/year	Temperature	Particle Size	Contact	
2	North Non-PAG Stockpile	Pit Rock	Subaerial	Waste Rock	P50	Analog	pH≥8	564.5	0.5	0.2	0.5	mg/L
3	Non-PAG Low Grade Ore Stockpile Outside TMF	Pit Rock	Subaerial	Ore	P50	Analog	pH≥8	564.5	0.5	0.2	0.5	mg/L
25	Non-PAG Low Grade Ore Stockpile Inside TMF	Pit Rock	Subaerial	Ore	P50	Analog	pH≥8	564.5	0.5	0.2	0.5	mg/L
4	PAG Low Grade Ore Stockpile	Pit Rock	Subaerial	Ore	P50	Analog	pH≥8	564.5	0.5	0.2	0.5	mg/L
7.1	PAG Stockpile	Pit Rock	Subaerial	Waste Rock	P50	Analog	pH≥8	564.5	0.5	0.2	0.5	mg/L
7.2	PAG Stockpile	Pit Rock	Flushing	Waste Rock	P50	Analog	pH≥8	-	0.5	0.2	0.5	mg/year
7.3	PAG Stockpile	Pit Rock	Subaqueous	Not Applicable	Not Applicable	Testwork	P50	Not Applicable	Not Applicable	Not Applicable	Not Applicable	mg/L
9	East Overburden Stockpile	Overburden	Subaerial	Quarry Rock	P50	OP Overburden	P50	564.5	0.3	0.2	0.5	mg/L
10.1	Pit walls	Wall	Subaerial	Waste Rock	P50	Analog	pH≥8	101.61	0.3	0.1	0.5	mg/L
10.2	Acidified Highwall	Wall	Subaerial	Waste Rock	Accelerated	Analog	4<pH<5	101.61	0.3	0.1	0.5	mg/L
10.3	Flushed Pit Walls	Wall	Flushing	Waste Rock	P50	Analog	pH≥8	-	0.3	0.1	0.5	mg/m2
11.1	Tailings Dam	Pit Rock	Subaerial	Waste Rock	P50	Analog	pH≥8	564.5	0.5	0.2	0.5	mg/L
11.2	Tailings Dam	Quarry Rock	Subaerial	Quarry Rock	P50	Analog	pH≥8	564.5	0.3	0.2	0.5	mg/L
11.3	Tailings Dam	TMF Overburden	Subaerial	Quarry Rock	P50	TMF Overburden	P50	790.3	0.3	0.2	0.5	mg/L
11.4	Tailings Dam	Pit Overburden	Subaerial	Quarry Rock	P50	OP Overburden	P50	790.3	0.3	0.2	0.5	mg/L
11.5	Tailings Dam	Tailings	Subaeral	Tailings	Initial	Analog	pH≥8	790.3	0.3	1	1	mg/L
12.1	Seepage dams	TMF Overburden	Subaerial	Quarry Rock	P50	TMF Overburden	P50	790.3	0.3	0.2	0.5	mg/L
12.2	Seepage dams	Pit Overburden	Subaerial	Quarry Rock	P50	OP Overburden	P50	790.3	0.3	0.2	0.5	mg/L
12.3	Seepage dams	Local Overburden	Subaerial	Quarry Rock	P50	TMF Overburden	P50	790.3	0.3	0.2	0.5	mg/L
14	Haul Road and Crusher Pad	Pit Rock	Subaerial	Waste Rock	P50	Analog	pH≥8	564.5	0.5	0.2	0.5	mg/L
15	Live ore pad and Live ore	Pit Rock	Subaerial	Ore	P50	Analog	pH≥8	564.5	0.5	0.2	0.5	mg/L
16	Cleaner tailings	Tailings	Subaqueous	Not Applicable	Not Applicable	Testwork	P50	Not Applicable	Not Applicable	Not Applicable	Not Applicable	mg/L
17.1	Rougher tailings	Tailings	Subaerial	Tailings	Initial	None	-	-	0.3	1	1	mg/m2/year
17.2	Rougher tailings	Tailings	Subaerial	Tailings	Initial	Analog	pH≥8	790.3	0.3	1	1	mg/L
18	Rougher tailings	Tailings	Subaqueous	Not Applicable	Not Applicable	Testwork	P50	Not Applicable	Not Applicable	Not Applicable	Not Applicable	mg/L
19	Topsoil Stockpiles	Overburden	Subaerial	Quarry Rock	P50	TMF Overburden	P50	790.3	0.3	0.2	0.5	mg/L
20	Plant site	Pit Rock	Subaerial	Waste Rock	P50	Analog	pH≥8	564.5	0.5	0.2	0.5	mg/L
21	Quarry	Wall	Subaerial	Quarry Rock	P50	Analog	pH≥8	101.61	0.3	0.1	0.5	mg/L
22	Ore to Process	Pit Rock	Flushing	Ore	P50	Analog	pH≥8	-	0.5	0.1	0.5	mg/tonne
23.1	Low grade ore to process	Pit Rock	Flushing	Ore	P50	Analog	pH≥8	-	0.5	0.2	0.5	mg/tonne
23.2	Process and Pond Constraints	-	-	-	-	Calculated	Process Pond pH	-	-	-	-	mg/L
24	Reagents	Reagents	Flushing	-	-	-	-	-	-	-	-	-

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Note:

“ – “ – Not applicable

Table 19: Summary of Reasonable Upper Limit Case Inputs by Term

Term	Term Location	Material Type	Term Type	Rate Type		Solubility Control		Infiltration	Rate Scaling			Reporting Units
				Type	Statistic	Type	P95 Statistic	mm/year	Temperature	Particle Size	Contact	
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16	Cleaner tailings	Tailings	Subaqueous	Not Applicable	Not Applicable	Testwork	P95	Not Applicable	Not Applicable	Not Applicable	Not Applicable	mg/L
17.1	Rougher tailings	Tailings	Subaerial	Tailings	Initial	None	-	-	0.3	1	1	mg/m2/year
17.2	Rougher tailings	Tailings	Subaerial	Tailings	Initial	Analog	7.6≤pH<8	790.3	0.3	1	1	mg/L
18	Rougher tailings	Tailings	Subaqueous	Not Applicable	Not Applicable	Testwork	P95	Not Applicable	Not Applicable	Not Applicable	Not Applicable	mg/L
19	Topsoil Stockpiles	Overburden	Subaerial	Quarry Rock	P95	TMF Overburden	P95	790.3	0.3	0.2	0.5	mg/L
20	Plant site	Pit Rock	Subaerial	Waste Rock	P95	Analog	7.6≤pH<8	564.5	0.5	0.2	0.5	mg/L
21	Quarry	Wall	Subaerial	Quarry Rock	P95	Analog	7.6≤pH<8	101.61	0.3	0.1	0.5	mg/L
22	Ore to Process	Pit Rock	Flushing	Ore	P95	Analog	7.6≤pH<8	-	0.5	0.1	0.5	mg/tonne
23.1	Low grade ore to process	Pit Rock	Flushing	Ore	P95	Analog	7.6≤pH<8	-	0.5	0.2	0.5	mg/tonne
23.2	Process and Pond Constraints	-	-	-	-	Calculated	Process Pond pH	-	-	-	-	mg/L
24	Reagents	Reagents	Flushing	-	-	-	-	-	-	-	-	-

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Table 20: Humidity Cell Rates (mg/kg/week)

Type	Statistic	SO_4	Alkalinity	Cl	F	Al	Sb	As	Ba	Be	B	Cd	Cr	Co	Cu	Fe	Pb	Mn	Mo	Ni	Se	Ag	Ti	U	Zn	Hg
Waste Rock	P50	7.4	16	0.27	0.0059	0.009	0.000081	0.00013	0.00091	0.0000048	0.024	2.7E-06	0.000059	0.00002	0.0002	0.0016	0.000014	0.00089	0.00048	0.00013	0.00024	0.0000024	0.00026	0.000042	0.00042	0.00000096
	P95	17	28	0.33	0.016	0.05	0.00022	0.0022	0.025	0.0000049	0.024	0.000069	0.000082	0.00018	0.00087	0.0045	0.0001	0.018	0.0052	0.00046	0.0017	0.0000031	0.00036	0.00018	0.0025	0.00000098
Ore, Low Grade Ore	P50	15	11	0.26	0.0073	0.0045	0.000085	0.000059	0.0012	0.000006	0.024	0.0001	0.000057	0.00089	0.0086	0.0015	0.00001	0.027	0.00036	0.00049	0.00048	0.0000025	0.00025	0.000043	0.014	0.00000096
	P95	16	13	0.26	0.0087	0.0069	0.00012	0.00006	0.0013	0.0000071	0.024	0.00017	0.000062	0.0016	0.015	0.0017	0.000012	0.035	0.00063	0.00077	0.00062	0.0000025	0.00025	0.00008	0.027	0.00000096
Tailings		4.6	25	0.25	0.0081	0.0081	0.000094	0.00076	0.0016	0.0000046	0.023	5.2E-06	0.000054	7.5E-06	0.00015	0.0011	0.00015	0.0022	0.0002	0.000099	0.00055	0.0000023	0.00023	0.00006	0.00094	0.00000093

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5.3.3 Solubility Controls

Basis

Solubility control input is based on the assumption that each parameter is contained in one or more mineral phases and that mineral phases have finite solubility under the prevailing disposal conditions defined by pH, Eh, temperature and bulk ion chemistry. Understanding of solubility controls can be derived from first principles (i.e, mineral-water equilibria and thermodynamics), experimental data and monitoring data from analogous sites. These approaches and application to the Harper Creek Project are described below.

Estimation of Solubility Controls from First Principals

For Harper Creek, only sulphate concentrations were determined using thermodynamic data. Solubility of sulphate is expected to be controlled by the mineral gypsum. While gypsum contains calcium and sulphate, the concentration of sulphate in equilibrium with gypsum is a function of all major cations in solution (calcium and magnesium in this case) due to the common ion effect for sulphate minerals. Essentially, as the proportion of ions forming sulphates more soluble than gypsum increases, so does the sulphate concentration. The range in molar ratio of magnesium to calcium concentrations (1.7 to 2.3) indicated by the barrel test data was used to calculate a range in constraining sulphate concentrations of 3,000 mg/L to 3,600 mg/L.

Using the expectation that selenium can be expected to co-precipitate with gypsum (Fernández-González et al. 2006), selenium concentrations were constrained using the ratio of selenium to sulphate observed in the barrel tests ($9.2\text{x}10^{-5}$ to $1.4\text{x}10^{-4}$ mgSe/mgSO_4) coupled with the constraining sulphate concentration. This resulting range in constraining selenium concentrations was 0.3 to 0.5 mg/L when gypsum was saturated.

Estimation of Solubility Controls from Experimental Data

In general, leaching testwork performed at small scale and under laboratory conditions provides limited data on solubility of most trace elements because the liquid-to-solid ratio is too high to allow solubility constraints to be reached unless adaptations are made to the procedures to understand the effect of leaching ratios.

For Harper Creek Project, testwork was performed using three procedures (humidity cells, leach columns and field barrel tests) on the same four rock type composites and provided an opportunity to evaluate differences in contact ratios (0.5, 0.03, and 0.01 (average) L/kg/week, respectively). For comparison, contact ratios for run-of-mine waste rock after accounting for particle size differences and reduced flow contact are typically about 0.001 L/kg/week, which is also the low end of contact ratios for the barrel tests.

Distributions of results for the three scales of testing were compared using box-and-whisker plots. Figure 25 illustrates typical results for selected parameters. Leachate pH was lowest for the HCTs compared to the columns and barrel tests. This is consistent with the use of deionized water with a pH between 5 and 6 for leaching and the highest leachate contact ratio.

Sulphate concentrations were much higher in both columns and barrels. Sulphate generation rates are discussed in Section 5.3.4. Columns and HCTs yielded similar generation rates but sulphate concentrations were higher in the columns due to the lower contact ratio. Barrels yielded lower sulphate generate rates compared to columns due to slower oxidation rates under site conditions. Concentrations were lower despite the lower contact ratio.

Copper showed a tendency toward increasing concentrations through the different scales though higher end concentrations appeared to be converging toward concentrations in the 0.05 to 0.1 mg/L range. The higher copper concentrations were consistent with copper oxide (tenorite, CuO) as a solubility control (Figure 26). The selenium distribution resembles that of sulphate in that there was a tendency toward higher selenium concentrations in column leachates but the difference is only reflected in the highest concentrations.

For most other parameters, the trend between tests is for concentrations to be similar in the barrel and column tests, or higher in the column tests. Concentrations were lowest for HCTs with the exception of aluminum which was probably influenced by the lower pH of the humidity cell leachates. The only parameter to show a consistent increase from HCTs to barrels was antimony.

Due to the common similarity of concentration ranges in the columns and barrels and the apparent attainment of equilibrium with copper secondary minerals, the on-site barrel tests are considered to indicate solubility controls for pHs above 8 under site conditions.

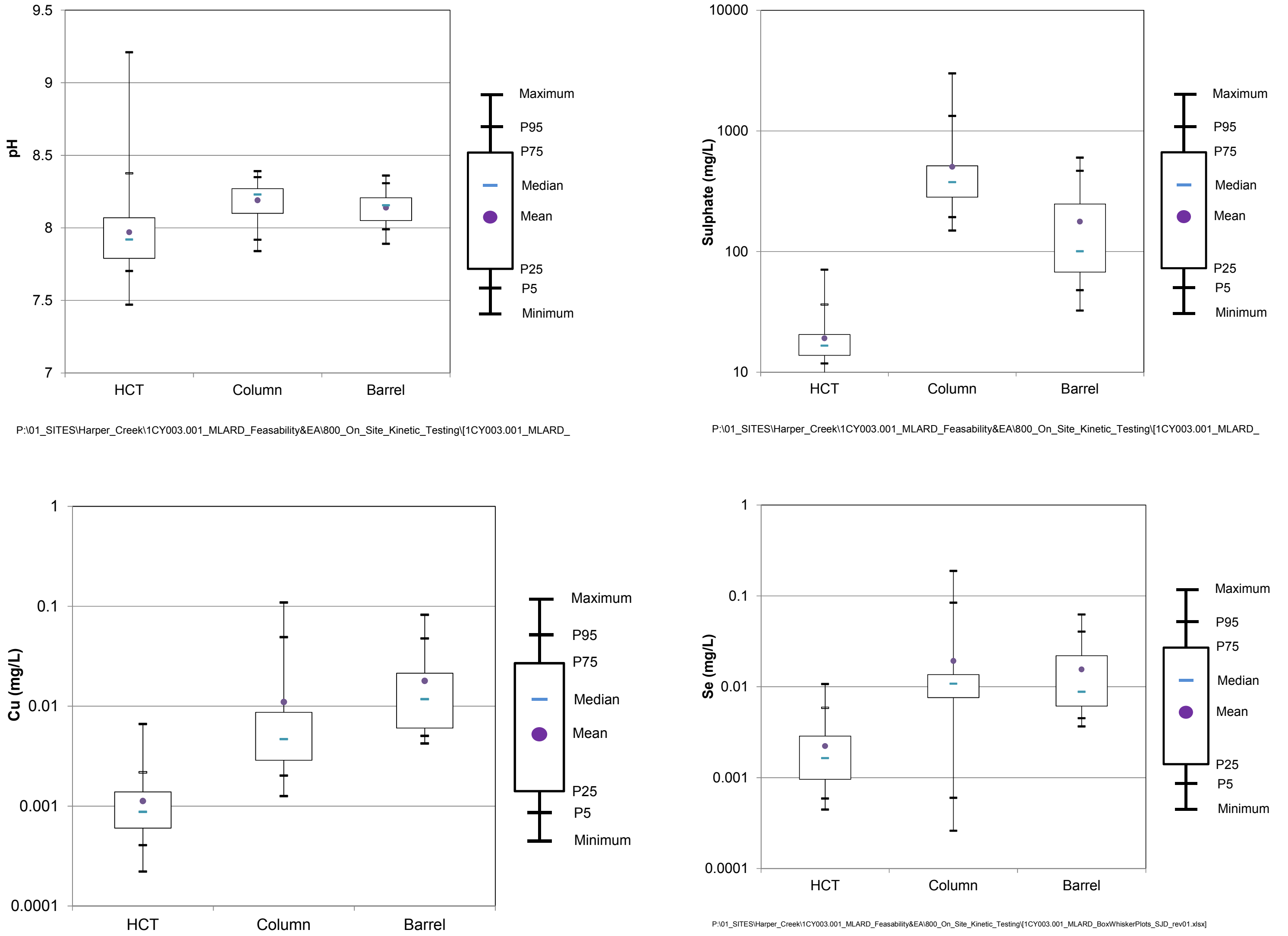

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Figure 25. Examples of Box-and-Whisker Plots to Compare HCT. Column and Barrel Test Data

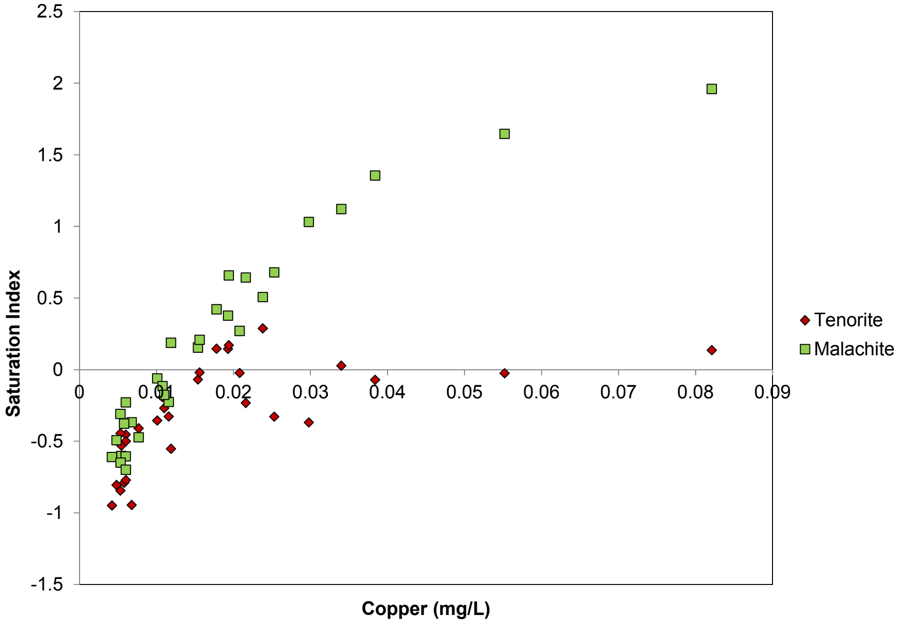

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Figure 26. Saturation Indices for Barrel Leachates for Copper Minerals Tenorite and Malachite

Mine Drainage Chemistry Analog Database

Mine drainage chemistry for geologically similar (analogous) mineral deposits can be an invaluable source of supplementary data for both ground-truthing of site data and providing understanding of water chemistry outside the conditions represented by the site data. Analog data should be used with caution because it is derived from sites that may not have implemented the same management strategies proposed for the Project. Kuroko-type deposits commonly yield ARD and have significant ML issues but the Project has been designed to address ML potential.

For comparison with solubility indicated by site data and to determine solubility under lower pH conditions, analog drainage data for Kuroko-type volcanogenic massive sulphide deposits were assembled. As discussed in Section 1.4.2, Kuroko-type is an imperfect analog for the Project because the Harper Creek deposit lacks abundant massive sulphide mineralization including lead and zinc sulphides and barite. However, this is considered to be the nearest similar deposit and is consistent with BC MINFILE classification (BC Ministry of Energy and Mines 2011).

The relevant data for the analog database are drainage collected as near as possible to the source material so that datapoints may come close to representing contact water. It is recognized that the majority of mine site drainages can be influenced by non-contact waters to some degree due to local runoff, source runoff, and groundwater inflows to monitoring points, though by compiling data from many sources, direct contact water has a high likelihood of being represented.

A global search for mines exploiting Kuroko-type mineral deposits using United States Geological Survey and BC MINFILE websites resulted in identification of 25 candidate sites. Field-scale water chemistry data were found for 12 mines (Britannia (B.C.), Eskay Creek (B.C.), Flin Flon

(MB), Heath Steele (NB), Kutcho Creek (B.C.), Parys Mountain (Wales, UK), Premier Gold (B.C.), Tulsequah (B.C.), Myra Falls (B.C.), and Greens Creek (Alaska)).

Review of the datasets obtained indicated that Eskay Creek, Harper Creek, Heath Steele, Kutcho Creek, Premier Gold, Tulsequah, and Myra Falls had useable near contact water chemistry for waste rock. Drainage data for Britannia Mine and Flin Flon Area mines is for underground mines and therefore likely reflects limited contact and/or dilution. Parys Mountain data was of uncertain value. The monitoring points were apparently distant from the site and the source of the drainage is unknown. As this site is highly acidic, it has limited value in understanding mobility under non-acidic conditions. Greens Creek has relevant waste rock dump drainage but only limited summary data were available.

Table 21 summarizes information on the scale of sites and the typical sulphur content of wastes. Since most Kuroko-type deposits yield massive sulphide ores, the quantity of wastes tends to be low relative to the Harper Creek Project. Early higher concentration data from barrel tests at Harper Creek and similar small scale-on site tests at Kutcho Creek were included in the analog database because interpretation of the Harper Creek barrel tests indicated that small scale tests can provide useful solubility data for non-acidic conditions.

Table 21: Sites in the Kuroko-Type Analog Database

Mine or Site	Type of Data	Facility Scale tonnes	Typical S %
Eskay	Waste rock dump drainage	10^4	1
Harper Creek	On site barrel tests	10^{-1}	1
Heath Steele	Waste rock dump drainage	10^4	5
Kutcho	Field test pads	10^1	Several percent
Premier Gold	Waste rock dump drainage	10^6	1
Tulsequah	Portal waste rock dump seepage	10^4	Unknown
Myra Falls	Waste rock dump drainage	10^5 to 10^6	4

Source: P:\01_SITES\Harper_Creek\1CY003.001_MLARD_Feasability&EA\900_Water_Quality_Predictions\2012-07_Initial_Terms\Analog_Database\[AnalogSites_Locked_for_Harper_1CY003001_SJD_REV08.1.xlsx]

To develop the solubility constraints for use in the source term predictions, the following procedure was used:

- Scatter plots of concentrations vs pH were evaluated for each parameter to determine if expected solubility relationships existed.
- The 50th and 95th percentiles for each parameter were determined for pH categories at each site. The pH categories were defined for pHs of 4<pH<5, 7.6≤pH<8 and pH≥8. .
- The solubility constraint was selected as the highest 95th percentile value in a given category for the majority of parameters.

- For Cd, Fe, Pb, Ag and Zn, the highest median value was selected because the dataset includes several sites which are massive lead and zinc sulphide deposits. Harper Creek does not contain massive sulphide mineralization or significant galena or sphalerite, and therefore the elevated 95th percentile values for these sites were not considered appropriate for application to Harper Creek.

The 95th percentile was selected to provide a systematic basis for exclusion of outliers which commonly exist in water chemistry data sets. This is equivalent to concluding that 1 in 20 datapoints may not have acceptable quality. The 95th percentiles were calculated on a site basis rather than using the whole dataset in an effort to avoid bias resulting from mixing datasets of variable size. Results of the assessment of the analog database are shown in Table 22. The table also provides similar numbers from a porphyry analog database (Day and Rees 2006) because the mineralization at Harper Creek is disseminated chalcopyrite and the scale of porphyry mines is comparable to Harper Creek. The porphyry database was not re-evaluated for the purpose of this project hence the statistics compared are not exactly the same but provide a general indication of concentration ranges at similar pHs.

Comparison of concentrations for the pH ranges indicates the expected relationships for most parameters. Elements occurring as cations show higher concentrations at lower pHs (e.g. Al, Cd, Co, Cu, Mn, Fe, Ni, Zn). Sulphate concentrations increase as pH decreases due to the shift in chemistry from Ca dominated to products of silicate weathering (Ca, Mg, K and Na). Several elements showed amphoteric behaviour (i.e., elevated concentrations at both acidic and basic pHs). Statistically, this translated to little difference across the pH range (Mo, Se and U). Relationships between pH and ions are provided in Appendix H with examples in Figure 27.

The Kuroko and Porphyry analog datasets show similar concentrations for many parameters likely reflecting similar underlying mineralogical controls regardless of the overall geological setting. Order of magnitude greater concentrations for the porphyry analog dataset at near neutral pH are apparent for Be, Cd, Pb, Mo, Ni and Zn. For Be and Hg, the differences reflect the effect of variable detection limits. The difference for Cd, Pb, and Zn is likely due to the common occurrence of sphalerite and galena in porphyries and the use of pH 6 to bound the range of pH to calculate statistics. The pH difference may explain the difference for Ni. Molybdenite in porphyries results in elevated Mo concentrations in drainage.

Table 22 also compares concentrations in the analog database for $pH \geq 8$ and the 95th percentile concentrations indicated by the on-site barrel tests. Concentrations were very similar and always within the same order of magnitude providing confidence that the site data appropriately represent full scale.

Based on these findings, the Kuroko database was considered appropriate to constrain source term predictions for the Harper Creek Project under the Reasonable Upper Limit cases when pHs could be lower than indicated by the barrels.

Table 22: Assessment of Controls on Element Mobility

Parameter	Correlation with pH	Mineral Source	Mineralogical Control	Waste Rock						Overburden			
				Kuroko			Barrels	Porphyry		TMF		Open Pit	
				4<pH<5	7.6≤pH<8	pH≥8	pH>8	pH<4	pH>6	Median	UBC	EC	UBC
Alkalinity	Positive	Carbonates	Calcite	57	250	190	190	2	7.5	190	220	190	220
Cl	None	Silicates	None	4.4	39	70	42	3.8	6.5	13	20	25	53
F	Variable by site	Apatite	Fluorite	0.079	0.77	0.87	0.86	1	0.9	0.87	0.77	0.68	0.77
SO4	Weak negative	Sulphides	Gypsum	11000	1400	1400	1300	2400	1500	26	690	100	4900
Al	Negative	Silicates	Basic Al sulphates	75	0.2	0.12	0.034	40	0.17	0.12	0.46	0.12	0.46
Sb	None	Pyrite, tetrahedrite	Sorbed with ferric hyroxides	0.05	NA	0.038	0.037	NA	0.075	0.011	0.047	0.0023	0.023
As	Weak Negative	Pyrite, arsenopyrite	Sorbed with ferric hyroxides	5.1	0.028	0.027	0.025	0.0006	0.03	0.027	0.028	0.02	0.028
Ba	Weak positive	Silicates	Barite	0.054	0.2	0.077	0.071	0.043	0.076	0.077	0.21	0.077	0.21
Be	None	Silicates	Unknown	0.0064	0.005	0.0001	0.00005	0.028	0.003	0.0001	0.00077	0.0001	0.002
B	None	Silicates	Unknown	0.084	0.3	0.3	0.3	0.5	0.41	0.3	0.3	0.3	0.3
Cd	Negative	Sphalerite	Sorbed with ferric hyroxides, carbonate co-precipitate	0.79	0.00047	0.00043	0.00047	0.04	0.0065	0.00043	0.00085	0.00043	0.00085
Cr	Weak negative	Oxides	Sorbed with ferric hyroxides	0.015	0.015	0.0011	0.0005	NA	0.0023	0.0011	0.01	0.0011	0.015
Co	Negative	Pyrite, pyrrhotite	Sorbed with ferric hyroxides	1.5	0.17	0.12	0.11	1.6	0.33	0.00083	0.05	0.044	0.76
Cu	Negative	Chalcopyrite	Malachite, tenorite	45	0.056	0.1	0.049	340	0.14	0.048	0.13	0.1	4.5
Fe	Negative	Pyrite	Ferric hydroxides	5.7	0.079	0.01	0.005	3.4	0.2	0.01	0.28	0.01	0.28
Pb	Weak negative	Galena	Sorbed with ferric hyroxides	0.41	0.001	0.00058	0.00057	0.018	0.0036	0.00058	0.003	0.00058	0.003
Mn	Weak negative	Pyrite, carbonates	Sorbed with ferric hyroxides, carbonate co-precipitate	150	2	0.39	0.42	26	4.5	0.39	3.5	0.39	86
Hg	None	Pyrite	Sorbed with ferric hyroxides	0.00032	0.0012	0.00001	NA	0.0006	0.00043	0.00001	0.00071	0.00001	0.0012
Mo	Weak Amphoteric	Pyrite, molybdenite	Sorbed with ferric hyroxides	0.03	0.07	0.054	0.12	0.026	0.33	0.0071	0.07	0.0075	0.07
Ni	Negative	Pyrite, pyrrhotite	Sorbed with ferric hyroxides	1.2	0.039	0.033	0.026	1.4	0.21	0.0057	0.02	0.033	0.16
Se	Weak Amphoteric	Pyrite, chalcopyrite	Gypsum, sorbed with ferric hydroxides	0.5	0.2	0.11	0.084	0.085	0.27	0.0033	0.082	0.027	0.5
Ag	Negative	Pyrite, tetrahedrite, chalcopyrite	Sorbed with ferric hyroxides	0.0005	0.00015	0.00011	0.00019	0.01	0.00001	0.000093	0.00011	0.00011	0.00049
Tl	None	Sulphides	Unknown	0.00014	0.00028	0.00029	0.00026	NA	0.00005	0.00011	0.00028	0.00029	0.00028
U	Weak Amphoteric	Oxides	Sorbed with ferric hyroxides	0.036	0.045	0.038	0.037	0.0005	0.0014	0.0033	0.0065	0.0017	0.043
Zn	Negative	Sphalerite	Sorbed with ferric hyroxides, carbonate co-precipitate	160	0.025	0.037	0.039	4	0.81	0.037	0.054	0.037	0.054

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Note:

All concentrations are in mg/L.

NA – Insufficient data to define

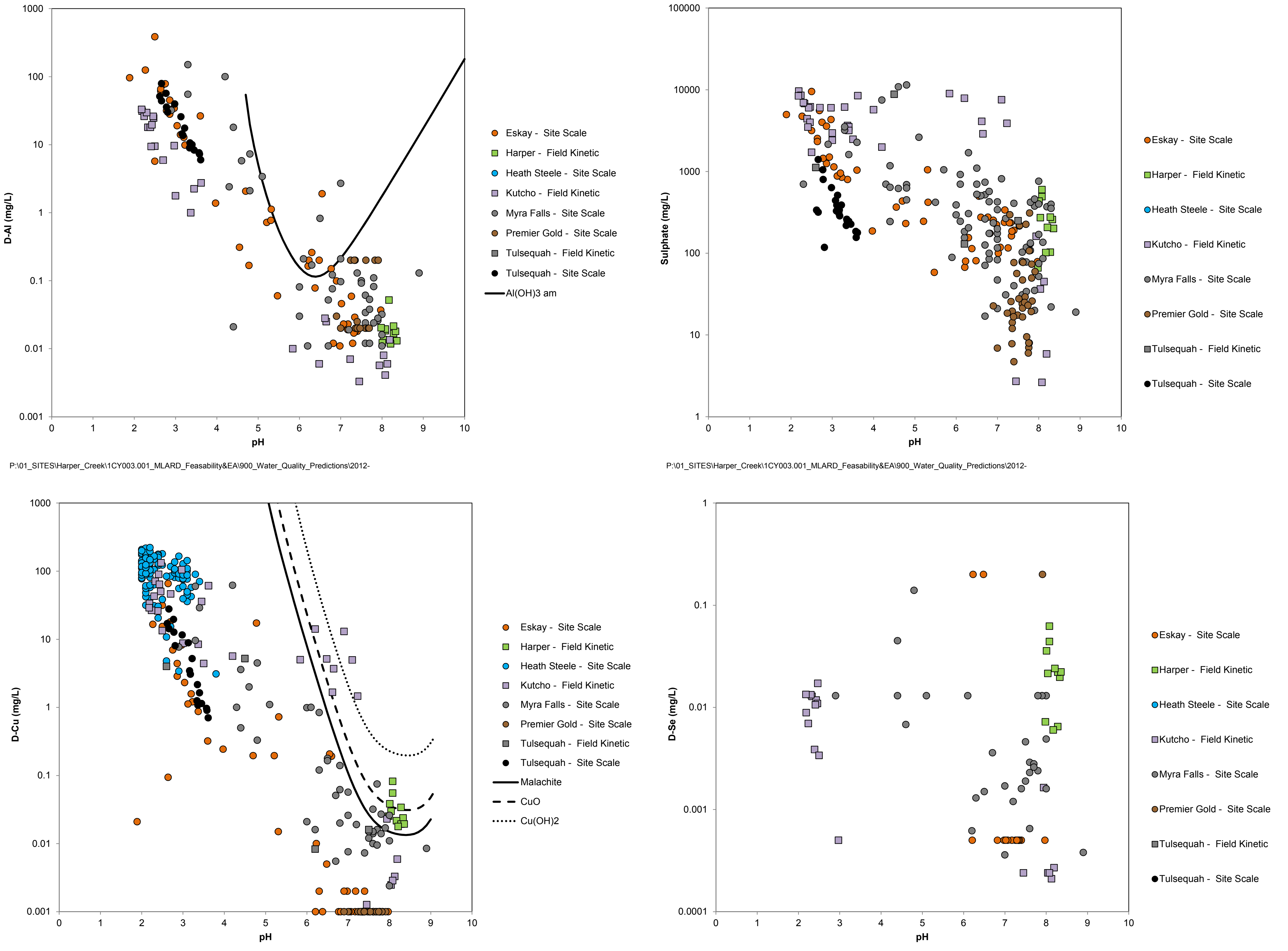

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Figure 27: Examples of Relationships between pH and Concentrations in the Kuroko-Type Analog Database for Selected Elements

Overburden

For overburden, solubility constraints were based on data obtained from leaching tests. For TMF area overburden, the 50th and 95th percentile of concentrations were used for the Expected and Reasonable Upper Limit Cases, respectively.

For the open pit area, the concentrations were adjusted using the sample moisture content to account for the high liquid to solid ratio in the test and to account for the presence of mineralized material. The adjusted concentration was compared to the analog database concentration so that concentrations did not exceed those for waste rock. Concentrations are shown in Table 22.

Saturated Wastes

For saturated wastes, in situ pore water concentrations were calculated as the early data 50th and 95th percentile of concentrations indicated by column testwork for the Expected and Reasonable Upper Limit Cases, respectively. Concentrations used are shown in Table 23.

Process Plant and Tailings Pond

For Term 23.2, solubility constraints were developed for water reclaimed for use in the process plant. The effect of increasing pH to 11 by the use of lime was modelled using Geochemist's Workbench (Bethke 2009). Parameters affected by the pH adjustment are shown in Table 24. The majority metal cations precipitated as a result of reactions of the type:

$$M^{n+} + (n/2)Ca(OH)_2 \rightarrow M(OH)_n + (n/2)Ca^{2+}$$

Sulphate solubility is much lower than in waste rock seepage due to removal of magnesium by precipitation to form magnesium hydroxide:

$$Mg^{2+} + Ca(OH)_2 \rightarrow Mg(OH)_2 + Ca^{2+}$$

This in turn results in solution chemistry dominated by Ca^{2+} which allows gypsum to precipitate.

A solubility control was applied to the tailings pond which was set at the same concentrations as the waste rock concentrations shown in Table 22.

Table 23: Saturated Source Terms

Parameter	Waste Rock		Tailings	
	P_{50}	P_{95}	P_{50}	P_{95}
pH	8.1	8.3	8.1	8.4
SO_4	40	140	180	1300
Alkalinity	74	110	84	120
Cl	1.6	4.1	2	8.4
F	0.31	0.75	0.25	0.78
Hardness	85	170	270	1400
Al	0.036	0.079	0.016	0.031
Sb	0.00097	0.005	0.0015	0.0022
As	0.0055	0.063	0.0015	0.0023
Ba	0.012	0.05	0.0099	0.062
Be	<0.00001	<0.00001	<0.00001	<0.00001
B	<0.05	<0.05	<0.05	0.01
Cd	0.000019	0.000042	0.000018	0.00011
Cr	<0.0001	0.0001	<0.0001	0
Co	0.0004	0.0044	0.00035	0.0055
Cu	0.0014	0.0056	0.0026	0.018
Fe	0.035	0.77	0.011	0.046
Pb	0.00016	0.00039	0.000098	0.00018
Mn	0.033	0.092	0.15	0.89
Mo	0.0037	0.0063	0.029	0.049
Ni	0.001	0.0051	0.0023	0.042
Se	0.0017	0.0022	0.0025	0.025
Ag	$<5\text{x}10^{-6}$	$<5\text{x}10^{-6}$	0.00001	0.00044
Tl	0.000019	0.000032	0.000005	0.000045
U	0.0008	0.003	0.00049	0.019
Zn	0.0018	0.004	0.0024	0.0055
Hg	<0.00001	0.000029	<0.00001	0.000015

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Note: All concentrations are in mg/L.

Table 24: Process Solubility Constraints

Parameter	Process Constraint mg/L
SO_4	1,500
Al	0.65
Cd	0.0093
Co	0.12
Cu	0.036
Fe	0.000044
Pb	0.0028
Ni	1.5
Zn	0.8
Hg	0.000002

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5.3.4 Scaling Factors

Scaling factors are described below and summarized in Table 18 and Table 19.

Temperature Factor - k_T

Two temperature factors were selected. For facilities expected to be heated by exothermic reactions, a scaling factor of 0.5 was used. Using the Arrhenius Equation and assuming pyrite is the dominant sulphide mineral, this corresponds to an average internal temperature of 15^{o}C compared to average ambient temperature at the site of 0^{o}C. For facilities with low oxidation rates, or expected to be cooled atmospheric conditions a factor of 0.3 was used. This corresponds to an average temperature of 11^{o}C.

Particle Size Factor - k_p

For waste rock, the particle size factor was set at 0.2 based on experience elsewhere (e.g. SRK 2011). For tailings, k_p = 1, because laboratory testing was performed on samples of tailings generated from metallurgical testing with a particle size distribution that is very close to eventual production values.

Contact Factor - k_c

The contact factor was set at 0.5 for rock sources and 1.0 for tailings. The factor for waste rock reflects that preferential flowpaths can be expected to develop whereas in tailings, an even wetting front can be expected to develop.

Oxygen Factor – k_{O2}

The oxygen factor was set at 1 which assumes that oxygen availability is the same as in humidity cells (i.e. expected atmospheric). This is considered to be conservative because it is expected that oxygen depletion inside waste facilities will decrease due to consumption by oxidation reactions.

Acidification Factor - k_a

The acidification factor was applied for Term 10.2 (long term pit highwall) and was set at five (i.e. the primary rate of weathering). The factor cannot be determined for Harper Creek directly because samples have not generated acid. For Huckleberry Mine, SRK (2002) found that the factor was about five for pyritic rock. SRK (2007, NorthMet) reported a factor of ten for pyrrhotite-rich rock.

Unflushed Factor – k_u

This factor was set at 0.1 for the Expected Case based on testwork performed at the Kudz Ze Kayah Project (Day et al. 1997) and supported by scoping level tests for this project (Section 4.1.10). The time between flushing was varied up to four weeks. For the Reasonable Upper Limit Case, the factor was set at one.

Evaluation of Scaling Factors for Site Conditions

Barrel test and humidity cell data can be compared to provide an indication of scaling factors for the difference between site and laboratory conditions. Due to the lack of large run-of-mine particles in the barrel tests, short flow paths and expected lack of oxygen depletion at this scale, the difference in rates is mainly due to temperature differences.

Sulphate release rates for barrel tests were calculated using chemistry data and records of water volumes in the collection pails. Leached loads were calculated for winter assuming the first water collected represents leaching of oxidation products accumulated through the winter. The summer rate was calculated based on cumulative loads leached through the summer months.

Table 25 shows average release rates for HCTs and columns on the same materials. The similarity of rates obtained by the two laboratory tests shows that the differences in test protocols had a limited effect on oxidation rates and the laboratory procedures acted as near duplicates. The summer barrel test results showed narrow variability in sulphate release rates and ratios to humidity rates of 0.1 to 0.3. As indicated above, these differences are mainly expected to reflect differences in temperatures though the ratios are lower than the annual value for K_T indicated above. The difference could reflect particle size and contact effects in addition to temperature. It is also likely that the barrels freeze completely in winter leading to very low weathering rates.

The findings of this interpretation were not used directly in the source term calculations because the results provided confirmation that lower rates can be expected under site conditions. Since the ratio obtained can be explained by a reasonable combination of factors, the overall approach was not changed.

Table 25. Calculation of Scaling Factors for Sulphate

			Barrel		
Composite	**HCT mg/kg/week**	**Column mg/kg/week**	**Winter mg/kg/week**	**Summer mg/kg/week**	**Summer/HCT Ratio**
COMP 7	7.6	6.7	0.038	1.4	0.2
COMP 8	7.1	10	0.021	2.3	0.3
COMP 9	10	10	0.081	1.3	0.1
COMP 11	6.0	6.4	0.032	1.0	0.2
Average	7.7	8.2	0.043	1.5	0.2

Source:P:\01_SITES\Harper_Creek\1CY003.001_MLARD_Feasability&EA\800_On_Site_Kinetic_Testing\Barrels\Interpretations\[BarrelInterp.1CY003001_SJD_REV00.xlsx]

5.3.5 Waste Quantities and Geometry

Waste quantities were provided by Knight Piésold (KP) on an annual basis. For the purpose of source terms, rates indicated in Table 20 were applied to all wastes above the water table.

The two low grade ore piles will be actively managed by addition and removal of material as mining progresses. For the purpose of determining the time factor for input into the calculation of stored soluble load (Section 5.2.3), the average age of a low grade pile was assumed to be 50% of the Project Year. For example, in Project Year 10, the average age is assumed to be 5 years. This approach was selected because the mining schedule allows a determination of when rock is placed on a particular pile but does not provide a linkage to when rock placed in a particular year is recovered for processing.

5.3.6 Infiltration

Average annual infiltration values were provided by KP and are tabulated in Table 18 and Table 19. Infiltration was not changed for the cases.

5.3.7 Nitrogen Leaching

Inputs to the nitrogen leaching calculation were provided by HCMC:

- Powder factors of 0.6 kg/m^3.
- Blasting Products.
- Dry Conditions: Fortan Advantage 35 - 35% emulsion and 65% ANFO.
- Wet Conditions: Fortan Advantage 70 – 70% emulsion and 30% ANFO.

ANFO is 5.7% fuel oil by weight and 94.3% ammonium nitrate. Emulsion is 3.5% fuel oil, 3.5% mineral oils and the balance as ANFO.

5.4 Results

Outputs from the two cases were provided to KP.

6 Conclusions

The ML/ARD baseline characterization and prediction program indicates:

- Waste rock and low grade ore has mixed ARD potential ranging from PAG to non-PAG. For both classes of waste, carbonate content is elevated and ARD potential of PAG materials is not expected to be realized for decades based on kinetic test data, observations of historic core and non-acidic conditions observed for in situ weathered PAG rock.
- The mine plan includes segregation of waste rock by ARD potential so that PAG rock is disposed underwater in the TMF.
- Regardless of pH, waste rock ML potential is greatest for sulphate, copper, selenium and zinc. ML potential is correlated with economic categories and waste categories and as a result wastes with higher ML potential will be submerged.
- Overburden in the pit footprint has variable ARD potential and should managed using the same programs as waste rock.
- Overburden in the TMF area has low ML/ARD potential. Elevated lead and arsenic concentrations were found in this overburden but it does not appear to be leachable.
- Bulk tailings have uncertain ARD potential but rougher tailings are non-PAG and cleaner tailings are PAG. The two tailing streams will be separated and the latter will be disposed in permanently saturated conditions in the TMF.
- Geochemical characterization data were used to develop contact water chemistry predictions for 32 individual sources in the water quality model for the project.

This report, “Metal Leaching and Acid Rock Drainage Characterization - Harper Creek Project”, has been prepared by SRK Consulting (Canada) Inc.

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The opinions expressed in this report have been based on the information available to SRK at the time of preparation. SRK has exercised all due care in reviewing information supplied by others for use on this project. Whilst SRK has compared key supplied data with expected values, the accuracy of the results and conclusions from the review are entirely reliant on the accuracy and completeness of the supplied data. SRK does not accept responsibility for any errors or omissions in the supplied information, except to the extent that SRK was hired to verify the data.

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Appendices

Appendix A: Rock Mineralogy

A1: Petrographic Descriptions

PETROGRAPHIC REPORT ON 18 WASTE ROCK SAMPLES FROM HARPER CREEK (FOR SRK YELLOWHEAD MINING INC.)

Report for: Tim O'Hearn
Maxxam Analytical
4606 Canada Way
Burnaby, B.C. V5G 1K5

Invoice 120072
ARD 2-21-900

Feb. 10, 2012.

SUMMARY:

The 18 samples all consist mainly of somewhat variable, mixed chips of fine-grained, variably foliated metamorphic rock (schist or sub-schist/phyllite) and lesser coarser-grained, vein material that contains the bulk of the sulfide minerals (which are locally or in some cases commonly closely associated or coated by the carbonate). Minerals identified, in roughly decreasing order of abundance, are quartz, sericite (or muscovite, depending on grain size), carbonate, chlorite, local K-feldspar (2 samples only), minor plagioclase, variable sulfides (mainly pyrite; but minor chalcopyrite occurs in 12 of the samples, local pyrrhotite, sphalerite in 5 samples, trace galena in one sample) and accessory rutile, rare tourmaline. One sample (12) contains significant amphibole, epidote and sphene. Carbonate appears to be mostly dolomite or ferroan dolomite (ankerite); there may be rare siderite (?). Chlorite tends to be slightly to locally somewhat magnesian based on optical properties (F:M ratio mostly 0.4, locally 0.4- 0.5?). Plagioclase appears to be mainly albite. The amphibole appears to include both relict hornblende or actinolitic hornblende as well as finer-grained, fibrous, pale green tremolite-actinolite (?). Epidote appears to be Fe-poor (clinozoisite?). Tourmaline appears to be intermediate dravite-schorl with F:M in the 0.3-0.5 range (?). Sphalerite appears to be Fe-poor.

The samples appear to be mostly derived by metamorphism of volcanic rocks, based on the presence of relict quartz and possible mafic and rare feldspar phenocrysts. On the basis of relict phenocrysts and essential mineralogy, they may be roughly divided into three categories: possible felsic (mainly quartz-eye bearing), intermediate/mafic, and mixtures of these two, as follows.

Relict quartz-eye bearing, felsic meta-volcanic (8 samples, 1, 2, 10, 10D, 14, 15, 16, 17) composed of chips that vary from weakly foliated, quartz-minor sericite-carbonate to strongly foliated, almost 100% sericite (accessory rutile, pyrite in both).

Relict mafic phenocryst (?) bearing, intermediate-mafic meta-volcanic (7 samples, 5, 6, 7, 8, 11, 12, possibly 13?) composed of carbonate, quartz, chlorite, sericite in variable proportions plus accessory rutile, pyrite and pyrrhotite (with the addition of amphibole, epidote and sphene in 12).

Mixtures of the above (3 samples, 3, 4, 9) in which the felsic meta-volcanic chips contain significant K-feldspar.

Capsule descriptions are as follows:

1: mainly quartz-sericite-minor carbonate-pyrite-rutile schist (possibly in part developed from felsic, quartz phyric volcanic/hypabyssal rocks) veined by quartz-carbonate-pyrite ±chalcopyrite, rutile. Carbonate appears likely to be variably ferroan dolomite or ankerite (?), and is commonly in contact with, or locally surrounds/coats, the sulfides.

2: mainly quartz-sericite-minor chlorite-carbonate-pyrite-rutile schist (in part after felsic, quartz phyric volcanic/hypabyssal rocks?) locally veined by quartz ± carbonate-pyrite-pyrrhotite-chalcopyrite-rutile. Carbonate appears likely to be variably ferroan ankerite (?), and is commonly in contact with, or locally surrounds/coats, sulfides.

3: chips of carbonate-quartz and/or carbonate-Kspar-minor pyrite-rutile-trace pyrrhotite-sphalerite-chalcopyrite altered rock of uncertain origin (possibly intermediate amygdular volcanic?), with minor chips of quartz ±sericite-carbonate-sulfide. Sulfides are relatively minor and only locally in contact with carbonate.

4: most chips appear to represent strongly quartz-carbonate-Kspar?-minor pyrite-trace pyrrhotite-chalcopyrite-accessory rutile altered, possibly originally carbonate-quartz amygdular intermediate/ mafic volcanic rock; some are quartz-sericite ±rutile schists derived from quartz phyric felsic volcanics. Minor sulfides are mostly in veins associated with or touching carbonate and quartz.

5: this sample contains quartz-carbonate-chlorite-rutile ±pyrite and quartz-sericite ±rutile, rare tourmaline semi-schists, both with coarse blastic carbonate or sericite aggregates (after mafic minerals?), suggestive of derivation from intermediate/mafic volcanic rocks partly mineralized with pyrite-trace chalcopyrite associated with veins of carbonate-quartz-plagioclase-chlorite.

6: mainly quartz-carbonate-chlorite-epidote-rutile ±sphene, pyrite schist, with lensy or blastic carbonate aggregates (possibly after mafic minerals, now deformed?), suggestive of derivation from intermediate/mafic volcanic rocks partly mineralized with pyrite-trace chalcopyrite associated with irregular, poorly defined veins of carbonate-quartz-chlorite.

7: mainly quartz-sericite-chlorite-minor carbonate-accessory sulfide (pyrrhotite ±pyrite)-rutile schist (after intermediate volcanic, locally quartz phyric?), with sulfides, locally in contact with carbonate, partly controlled by vein-like aggregates or segregations of carbonate-quartz along the foliation.

8: mainly quartz-sericite-chlorite-minor carbonate-accessory sulfide (pyrrhotite ±pyrite, trace chalcopyrite, rare pentlandite?)-rutile schist (after intermediate volcanic, locally quartz phyric?), in which sulfides are partly controlled by vein-like aggregates or segregations of carbonate-quartz along the foliation, and are locally in contact with carbonate.

9: quartz-chlorite-carbonate-rutile ±pyrite schist with local lensy or blastic carbonate ±sericite aggregates (after mafic phenocrysts?), suggestive of derivation from intermediate/mafic volcanics, and quartz-sericite ±chlorite-rutile schists with local relict phyric quartz, both partly mineralized with pyrite-minor chalcopyrite, pyrrhotite partly associated with veins of carbonate-quartz-chlorite.

10: chips of strongly foliated, sericite-rich (minor quartz-carbonate-local pyrite-accessory rutile) or less foliated, quartz-minor carbonate-sericite-accessory pyrite-chalcopyrite-sphalerite-pyrrhotite-rutile schist, plus coarse-grained, vein quartz with minor carbonate, sericite, and sulfides (the latter generally not in contact with carbonate).

10D: chips of strongly foliated, sericite-rich (minor quartz-carbonate-local pyrite-accessory rutile) or less foliated, quartz-minor carbonate-sericite-pyrite-chalcopyrite-rare pyrrhotite-accessory rutile schist, plus coarse-grained, vein quartz with minor carbonate, sericite, chlorite and sulfides. Sulfides are only locally in contact with carbonate.

11: chlorite-, sericite-, or quartz-rich schist or semi-schist, the latter typically with variable carbonate, sericite/muscovite and minor chlorite; all contain very fine-grained accessory rutile and local pyrite. Most sulfides (pyrite, trace chalcopyrite) occur in vein chips of carbonate-quartz, where they are in contact with or enclosed in the carbonate. Chlorite may be slightly magnesian (F:M ~0.4?).

12: mainly fine-grained amphibole-chlorite-epidote-sphene-minor quartz-carbonate schist (local blastic amphibole, carbonate, albite possibly representing former mafic and rare feldspar phenocryst sites?), cut by coarser-grained veins and veinlets of carbonate-quartz-chlorite-amphibole-pyrite ±chalcopyrite, local epidote.

13: mainly fine-grained, weakly/moderately foliated, quartz-carbonate-sericite-chlorite-accessory pyrite-rutile schist, cut by veins of coarser-grained secondary quartz-carbonate-pyrite-sphalerite-chalcopyrite-trace galena, local chlorite-muscovite.

14: mostly fine-grained, variably textured, variably foliated quartz-sericite-minor carbonate-accessory pyrite-rutile schist or phyllite, locally with quartz "eyes" or relict phenocrysts and rare blastic carbonate possibly after mafic phenocrysts, cut by irregular, poorly defined veins of coarser-grained quartz-carbonate-pyrite-chalcopyrite-local minor chlorite.

15: mostly fine-grained, variably textured, variably foliated quartz-carbonate-minor sericite-accessory pyrite-rutile schist or phyllite, locally with quartz "eyes" or relict phenocrysts and rare blastic carbonate possibly after mafic phenocrysts suggestive of derivation from felsic volcanic rock, cut by irregular, poorly defined veins of coarser-grained quartz-carbonate-pyrite-chalcopyrite locally partly oxidized to minor limonite and in contact with Fe-carbonate.

16: medium/fine-grained, variably textured, variably foliated quartz-minor sericite-carbonate-accessory pyrite-rutile phyllite or sericite schist, locally with quartz "eyes" or relict phenocrysts and rare blastic carbonate possibly after mafic phenocrysts suggestive of derivation from felsic volcanic rock, cut by irregular, poorly defined veins of coarser-grained quartz-carbonate-pyrite-chalcopyrite-trace sphalerite, locally in contact with Fe-carbonate.

17: medium/fine-grained, variably textured, variably foliated quartz-lesser sericite-minor carbonate-accessory pyrite-rutile phyllite, or sericite schist, locally with quartz "eyes" or relict phenocrysts and rare blastic carbonate possibly after mafic phenocrysts, suggestive of derivation from felsic volcanic rock, cut by irregular, poorly defined veins of coarser-grained quartz-carbonate-pyrite-chalcopyrite-sphalerite, all locally in contact with Fe-carbonate.

Detailed petrographic description and photomicrographs are appended (on CD). If you have any questions regarding the petrography, please do not hesitate to contact me.

1 (HC07-21): QUARTZ-SERICITE-MINOR CARBONATE-PYRITE-RUTILE SCHIST WITH QUARTZ-CARBONATE-PYRITE-TRACE CHALCOPYRITE-RUTILE VEINS

Sample consists of flat, slabby chips of pale beige-white coloured schistose rock mostly <1.5 cm long (cone crushed -1/4 fraction). Most chips are highly siliceous and contain finely disseminated sulfides (mainly pyrite?). The rock is not magnetic, shows no reaction to cold dilute HCl (even where powdered), and no stain for K-feldspar in the etched offcut prepared from a grain mount in epoxy. Modal mineralogy in polished thin section prepared from the grain mount is approximately:

Quartz (partly secondary)	65%
Sericite	25%
Pyrite	~5%
Carbonate (ankerite?)	3-5%
Rutile	<1%
Chalcopyrite	<<1%

This sample consists of chips of quartz-rich or sericite-rich, or variably quartz-sericite-accessory pyrite-carbonate-trace rutile schist (rarely with small quartz "eyes" or relict phenocrysts, suggestive of derivation from a felsic volcanic or hypabyssal rock), plus lesser chips of quartz-minor pyrite-carbonate veins (?).

In the quartz-sericite schist chips, proportions of quartz and sericite vary from ~90:10 to 10:90, with variable carbonate, accessory pyrite and rutile together generally making up <5%. The quartz ranges from very fine-grained (ragged tightly interlocking sub/anhedra mostly <50 um) to fine-grained (up to 0.25 mm) but grades to secondary quartz (see below). Relict quartz phenocrysts have rounded subhedral outlines mainly <0.6 mm in diameter. Sericite forms mainly euhedral flakes <50 um in size, but they range up to 0.15 mm, and due to the strong alignment, they may be semi-continuous optically for up to 1 mm (especially in sericite-rich chips). Carbonate occurs as sub/euhedral crystals up to 0.1 mm, locally arranged in vein-like aggregates. The lack of reaction to HCl in hand specimen suggests it may be ankerite (ferroan dolomite). Accessory pyrite forms sub- to euhedral crystals mostly <0.2 mm, locally associated with rutile as slender stubby golden brown euhedra <25 um long, locally aggregating to 0.1 mm. Rutile also occurs within the carbonate aggregates as minute (10 um) dark brown subhedra.

In the possible vein chips, quartz forms interlocking sub/anhedra up to about 1.25 mm, with moderate strain indicated by undulose extinction, sub-grain development, and local suturing of grain boundaries. Carbonate forms interlocking sub/euhedra up to about 1 mm in size that vary from relatively clear (dolomitic?) rims to pale brownish (ankeritic?) cores, possibly depending on Fe content. The reverse (brownish ankeritic rims on clearer dolomitic cores?) is also locally seen. Pyrite forms subhedra up to about 1 mm in size (in aggregates up to ~3 mm across) and is commonly although not everywhere in contact with, or at least associated with, the carbonate. Locally pyrite is fractured, or in places veined by dark brown (Fe-rich) carbonate and is associated with minor rutile as aggregates to ~0.1 mm of golden brown euhedra <35 um long. Rare chalcopyrite forms rounded subhedral blebs <50 um across, interstitial to or rarely included within, pyrite crystals, or within quartz (but surrounded by carbonate). Minor chalcopyrite also locally occurs as ragged subhedra to 0.1 mm partly encased in carbonate (subhedra <50 um) in siliceous or partly silicified chips.

In summary, this sample appears to represent mainly quartz-sericite-minor carbonate-pyrite-rutile schist (possibly in part developed from felsic, quartz phyric volcanic/hypabyssal rocks) veined by quartz-carbonate-pyrite ±chalcopyrite, rutile. Carbonate appears likely to be variably ferroan dolomite or ankerite (?), and is commonly in contact with, or locally surrounds/coats, the sulfides.

2: QUARTZ-SERICITE-MINOR CHLORITE-CARBONATE-PYRITE-RUTILE SCHIST WITH QUARTZ-CARBONATE-PYRITE-PYRRHOTITE-TRACE CHALCOPYRITE-RUTILE VEINS

Sample (from HC07-42) consists of flat, slabby chips of pale beige-white coloured schistose rock mostly <1.5 cm long (cone crushed -1/4 fraction). Some chips are highly siliceous and contain finely disseminated sulfides (mainly pyrite?). The chips are locally weakly magnetic, but show no reaction to cold dilute HCl (even where powdered), and no stain for K-feldspar in the etched offcut prepared from a grain mount in epoxy. Modal mineralogy in polished thin section prepared from the grain mount is approximately:

Quartz (partly secondary)	55%
Sericite	35%
Chlorite	3-5%
Fe-carbonate (ankerite?)	2-3%
Pyrite	1-2%
Pyrrhotite	~1%
Rutile	~1%
Chalcopyrite	<<1%

This sample consists of chips of quartz-rich or sericite-rich, or variably quartz-sericite-accessory chlorite-sulfide-carbonate-rutile schist (locally with small quartz “eyes” or relict phenocrysts, suggestive of derivation from a felsic volcanic or hypabyssal rock), plus lesser chips of quartz-minor sulfide-carbonate veins (?).

In the quartz-sericite schist chips, proportions of quartz and sericite vary from ~85:15 to 10:90, with variable chlorite, carbonate, rare sulfide and rutile together generally making up <10%. The quartz ranges from very fine-grained (ragged tightly interlocking sub/anhedra mostly <50 um) to fine-grained (up to 0.25 mm) but grades to secondary quartz (see below). Relict quartz phenocrysts have rounded subhedral outlines up to almost 1mm in diameter. Sericite forms mainly euhedral flakes < 0.15 mm, and due to the strong alignment, they may be semi-continuous optically for up to 1.5 mm (especially in sericite-rich chips). Chlorite occurs as subhedral flakes mostly <0.1 mm that are intimately mixed with/aligned with sericite, difficult to distinguish from sericite due to very pale green colour, except by length-fast character that is suggestive of somewhat magnesian composition (Fe:Fe+Mg, or F:M, ratio ~0.4?) Carbonate occurs as brownish sub/euhedral crystals rarely over 0.1 mm, locally arranged in vein-like or irregular aggregates <0.5 mm. The lack of reaction to HCl in hand specimen and the brown colour in thin section suggests it is likely ankerite (ferroan dolomite). Accessory pyrite forms sub- to euhedral crystals mostly <0.2 mm, locally associated with rutile as slender acicular golden brown euhedra up to 55 um long (locally aggregating to 0.1 mm).

In the possible vein chips, quartz forms interlocking sub/anhedra mostly <0.5 mm, with moderate strain indicated by undulose extinction, sub-grain development, and local suturing of grain boundaries. Carbonate forms interlocking sub/euhedra up to about 0.5 mm mostly stained brownish (ankeritic?), possibly with substantial Fe content. Pyrite forms ragged subhedra up to about 1 mm (locally needle-like, suggestive of having formed after marcasite?), in aggregates up to 2.5 mm across locally in contact with, or at least associated with, the carbonate. Pyrrhotite occurs as aggregates to 2 mm of rounded subhedra mainly <0.2 mm, also locally partly rimmed by the Fe-carbonate. Locally pyrite is fractured and is associated with minor rutile as aggregates to ~0.1 mm of golden brown euhedra <45 um long. Rare chalcopyrite forms irregular subhedra <50 um across, associated with pyrite or pyrrhotite, or within quartz (also commonly partly surrounded by carbonate).

In summary, this sample appears to represent mainly quartz-sericite-minor chlorite-carbonate-pyrite-rutile schist (possibly in part developed from felsic, quartz phyric volcanic/hypabyssal rocks) veined by quartz ± carbonate-pyrite-pyrrhotite-chalcopyrite, rutile. Carbonate appears likely to be variably ferroan ankerite (?), and is commonly in contact with, or locally surrounds/coats, sulfides.

3: CHIPS OF CARBONATE±QUARTZ AMYGDULAR, CARBONATE-QUARTZ±PYRITE-RUTILE, OR KSPAR-CARBONATE-QUARTZ ROCK, MINOR QUARTZ-SERICITE, VEIN QUARTZ-CARBONATE-SULFIDES (PYRITE, RARE SPHALERITE, CHALCOPYRITE)

Sample (from HC07-21) consists of angular chips of grey or buff-beige coloured, partly schistose rock mostly <1 cm across (cone crushed -1/4 fraction). Some chips are very soft (highly sericitic?); others are feldspathic (strong stain for K-feldspar in the etched offcut). The chips are locally weakly magnetic, and grey ones show local minor reaction to cold dilute HCl. Modal mineralogy in polished thin section prepared from the grain mount in epoxy is approximately:

Carbonate (dolomite, ankerite?)	30%
Quartz (partly secondary, vein?)	30%
K-feldspar (partly secondary?)	20%
Sericite	17%
Pyrite	1-2%
Rutile	<1%
Pyrrhotite	<1%
Sphalerite	<1%
Chalcopyrite	<1%

Most chips in this sample consist of 1) carbonate-minor quartz amygdules (?) in fine-grained carbonate-quartz-accessory sulfide-rutile, or 2) mostly K-feldspar, lesser carbonate and quartz (also with blastic or amygdular carbonate?). Chips composed of sericite or quartz-sericite and vein fragments composed mostly of coarser-grained quartz and minor sericite, local accessory sulfide and carbonate, are uncommon to relatively rare.

In the first type, rounded to ovoid or lensoid aggregates up to 1.2 mm long consist of interlocking sub/anhedral crystals of carbonate (mainly <0.25 mm, with undulose extinction, mainly relatively clear, possibly dolomite, but with local partial cores of brownish, finer-grained possible ankerite?) and quartz (subhedra <0.2 mm). The groundmass consists of tightly interlocking, randomly oriented, sub/anhedral crystals of brownish carbonate (ankerite?) mostly <50 um in size, intergrown with quartz as similar sized ragged sub/anhedra and possibly local K-feldspar as ragged anhedra <35 um, plus accessory sulfide (mainly pyrite as cubic subhedra <0.1 mm, trace chalcopyrite <15 um) and rutile (minute subhedra <10 um). Rarely, carbonate-rich chips up to 1 mm across contain sphalerite (colourless, low Fe) as rounded subhedra <0.3 mm, associated with minor chalcopyrite as subhedra <0.1 mm).

In the second type, sub/euhedral K-feldspar forming lath-like crystals to 0.35 mm long with random orientations are set in a groundmass of finer-grained (<45 um) Kspar and dark brownish (ankeritic?) carbonate, with accessory pyrite (cubic subhedra <0.1 mm) and rutile (aggregates to 0.1 mm of minute dark brown euhedra <25 um), containing local rounded or blastic, euhedral crystals of carbonate up to 0.6 mm in diameter that are mostly relatively clear (possibly dolomite?). The blastic carbonate crystals are particularly well-developed along zones (veins?) in which the rock appears to be replaced by fine-grained secondary quartz as ragged subhedra mostly <50 um long, with random orientations, associated with rare pyrite as rounded aggregates to 0.2 mm.

In the quartz-sericite chips, quartz forms interlocking sub/anhedra mostly <0.7 mm with moderate strain indicated by undulose extinction, sub-grain development, and rare suturing of grain boundaries; sericite is mainly interstitial, forming deformed/bent subhedra mostly <0.1 mm. These chips grade to what appear to be vein chips, with similar sized or coarser quartz and carbonate (aggregates to 0.6 mm with brown cores, likely ankerite, surrounded by relatively clear rims, likely dolomite, locally in contact with sulfides (aggregates to 0.6 mm of sub-cubic pyrite <0.3 mm).

In summary, this sample appears to be composed mainly of chips of carbonate-quartz and/or carbonate-Kspar-minor pyrite-rutile-trace pyrrhotite-sphalerite-chalcopyrite altered rock of uncertain origin (possibly intermediate amygdular volcanic?), with minor chips of quartz ±sericite-carbonate-sulfide. Sulfides are relatively minor and only locally in contact with carbonate.

4: STRONGLY QUARTZ-CARBONATE-KSPAR?-MINOR PYRITE-RUTILE ALTERED MAFIC VOLCANIC, QUARTZ-SERICITE-RUTILE SCHIST AFTER FELSIC VOLCANIC, AND CARBONATE-QUARTZ VEINS WITH MINOR PYRITE-PYRRHOTITE-CHALCOPYRITE

Sample (from HC07-21) consists of angular chips of grey or buff-beige coloured, partly schistose rock mostly <1 cm across (cone crushed -1/4 fraction). Some chips are white and soft (partly sericitic?); others are feldspathic (moderate stain for K-feldspar in the etched offcut). The chips are essentially non-magnetic, and show no reaction to cold dilute HCl. Modal mineralogy in polished thin section prepared from the grain mount in epoxy is approximately:

Quartz (largely secondary, minor vein?)	35%
Carbonate (mainly ankerite?)	30%
K-feldspar (partly secondary?)	20%
Sericite	13%
Pyrite	1%
Rutile	<1%
Pyrrhotite, chalcopyrite	<1% each

The chips in this sample are mostly divisible into end-member types: fine-grained carbonate, quartz and interstitial very fine-grained K-feldspar, or variable quartz-sericite (from quartz-rich to sericite-rich). Both contain accessory rutile, but neither contains significant sulfide, which tends to occur as separate grains or with thin, poorly developed veins of carbonate and/or quartz. In many of the first type, relict feldspar laths appear to have been extensively replaced by secondary quartz, and possible relict mafic sites by carbonate and accessory rutile, suggestive of an intermediate volcanic precursor.

In the former, rounded to ovoid or lensoid aggregates up to 1 mm long consist of relatively coarse carbonate (interlocking subhedra <0.5 mm, likely dolomite and ankerite since they are relatively clear and brownish respectively) and local minor quartz (euhedral prisms up to 0.1 mm long with radiating habit). These strongly resemble amygdules in a volcanic rock, and this impression is reinforced by the finely trachytic texture in the groundmass, created by partly aligned euhedral lath shapes of former feldspar (likely mainly plagioclase) up to about 0.2 mm long but now mostly replaced by secondary quartz as minute, randomly oriented crystals <10 um in size, separated by small aggregates with subhedral outlines mostly <0.1 mm composed of distinctly brown carbonate (interlocking subhedra mainly <40 um, probably ankerite) that likely represent the sites of former mafic minerals such as pyroxene, since they contain the bulk of the accessory rutile as minute sub/-euhedra <20 um, and are associated with pyrite, rare pyrrhotite, chalcopyrite as subhedra <0.1 mm. The groundmass consists of feathery, interlocking crystals of quartz, carbonate and lesser interstitial K-feldspar (negative relief against quartz, stained yellow in offcut) all mostly <25 um in size.

In the latter, local relict quartz “eyes” have rounded/irregular outlines <0.35 mm composed of recrystallized, interlocking quartz <0.1 mm, set in a matrix of quartz (rounded subhedra mainly <50 um) and sericite (aligned subhedral flakes mostly <35 um). The proportions of quartz to sericite are widely variable from about 80% quartz/10% sericite (10% carbonate) to 90% sericite/10% quartz (minor rutile as golden brown acicular euhedra mainly <45 um long). Carbonate aggregates are similar to those described above, also likely ankerite. The presence of carbonate in the former, and the presence of thin carbonate veins both in the chips and separate, suggests some of these chips may also be derived from volcanic rocks by more intense alteration.

In possible veins and vein chips, carbonate forms interlocking subhedra to 0.2 mm that are clear or brown (dolomite or ankerite?); quartz forms rounded sub/anhedra to 0.3 mm with moderate undulose extinction and sub-grain development, but only rare suturing of grain boundaries. Sulfides including pyrite as fractured euhedra to 0.6 mm, and rare chalcopyrite as aggregates to 0.3 mm, are commonly in contact with or associated with carbonate.

In summary, most chips appear to represent strongly quartz-carbonate-Kspar?-minor pyrite-trace pyrrhotite-chalcopyrite-accessory rutile altered, possibly originally carbonate-quartz amygdular intermediate/mafic volcanic rock; some are quartz-sericite ±rutile schists derived from quartz phyric felsic volcanics. Minor sulfides are mostly in veins associated with or touching carbonate and quartz.

5: CARBONATE±SERICITE BLASTIC, QUARTZ-CHLORITE/SERICITE-RUTILE SCHIST WITH VEINS OF CARBONATE-QUARTZ-PYRITE±CHALCOPYRITE-PLAGIOCLASE

Sample (from HC10-77) consists of angular chips of pale greenish- or buff-beige coloured, locally schistose rock mostly <1.5 cm across (cone crushed -1/4 fraction). Some chips are white and soft (partly sericitic?); others are grey and partly soft (carbonate-rich?). The chips are locally slightly magnetic, and show strong reaction to cold dilute HCl where powdered, but there is no stain for K-feldspar in the etched offcut. Modal mineralogy in polished thin section prepared from the grain mount in epoxy is approximately:

Carbonate (mainly dolomite, minor ankerite?)	35%
Quartz (partly secondary)	30%
Chlorite (magnesian?)	15%
Sericite	10%
Pyrite (partly after pyrrhotite?), trace chalcopyrite	5%
Plagioclase (albite, secondary)	3%
Rutile	1-2%
Tourmaline (intermediate dravite-schorl?)	<1%

Most of the chips in this sample may be roughly divided into 1) fine-grained quartz-chlorite-rutile with coarse blastic carbonate-local sericite (muscovite)-minor sulfide, or more strongly foliated quartz-sericite-minor rutile, also with coarse blastic carbonate-sulfide, grading to 3) vein-like aggregates of carbonate-quartz-sulfide-minor chlorite-local plagioclase.

In the first type, the blastic aggregates of carbonate (mostly irregular, <1 mm, but locally up to almost 2 mm long where gradational to vein-like aggregates, all mostly composed of interlocking sub/euhedra <0.5 mm, likely mainly dolomite to judge by the reaction in hand specimen, although there could be lesser ankerite?) and sericite or muscovite (euhedral flakes to 0.25 mm) may represent the sites of former mafic phenocrysts in mafic/intermediate rock. This conclusion is supported by the presence in the matrix of significant amounts of chlorite (somewhat aligned, subhedral flakes mostly <50 um, almost colourless, with length-fast, weakly anomalous first-order greenish-grey birefringence suggestive of F:M around 0.4?) and rutile (golden brown stubby euhedral prisms to 50 um), mixed with quartz as subhedra mostly <0.1 mm long (somewhat flattened in the plane of foliation, with length:width up to ~2:1). Sulfide is only locally present (aggregates to 0.25 mm of cubic pyrite mainly <0.1 mm), associated and locally intergrown with carbonate and quartz in foliation-parallel stringers or bands up to 0.5 mm thick that may grade to the veins (see below).

In the second type, quartz and sericite are present in varying proportions, either segregated into discontinuous foliae of quartz (interlocking subhedra <0.2 mm but commonly in aggregates with strongly flattened outlines up to 1.5 mm long and weak/moderate strain indicated by undulose extinction, minor sub-grain development, rare suturing of grain boundaries) or sericite (aligned sub/euhedral flakes <65 um), the latter typically with minute hair-like (sagenitic) rutile mainly <20 um long by <3 um thick. Rare tourmaline forms stubby euhedral prisms <0.1 mm long, oriented across the foliation, with brownish pleochroism suggestive of intermediate dravite-schorl (?).

In the vein (?) chips, carbonate forming interlocking subhedra to 1.5 mm, quartz forming subhedra to 1 mm, and local sulfide (mainly pyrite as cubic euhedra up to ~2 mm, rare chalcopyrite <0.2 mm, both commonly in contact with carbonate) are intergrown. Rarely, minor chlorite forming subhedral flakes to 0.1 mm (optical properties as described above) and local plagioclase (subhedra to 1.2 mm with vague twinning and negative relief compared to quartz indicative of albite, likely secondary) are intergrown with quartz. Some of the pyrite aggregates have a porous texture composed of minute crystallites <15 um suggestive of pyrite/marcasite after former pyrrhotite, and contain inclusions of rutile (subhedra to 25 um) or chalcopyrite (subhedra to 70 um).

In summary, this sample contains quartz-carbonate-chlorite-rutile ±pyrite and quartz-sericite ±rutile, rare tourmaline semi-schists, both with coarse blastic carbonate or sericite aggregates (after mafic minerals?), suggestive of derivation from intermediate/mafic volcanic rocks partly mineralized with pyrite-trace chalcopyrite associated with veins of carbonate-quartz-plagioclase-chlorite.

6: CARBONATE±PYRITE-QUARTZ-CHLORITE BLASTIC, QUARTZ-CHLORITE-EPIDOTE-RUTILE-SPHENE SCHIST, VEINS OF CARBONATE-QUARTZ-PYRITE±CHALCOPYRITE

Sample (from HC10-77) consists of angular chips of medium greenish- grey coloured, partly schistose rock mostly <1 cm across (cone crushed -1/4 fraction). Some chips are white and harder than steel (mainly quartz veins?). The chips are not magnetic, show strong reaction to cold dilute HCl where powdered, and no stain for K-feldspar in the etched offcut. Modal mineralogy in polished thin section prepared from the grain mount in epoxy is approximately:

Carbonate (mainly dolomite, minor ankerite?)	30%
Quartz (partly secondary)	30%
Chlorite (slightly magnesian?)	30%
Pyrite (partly after pyrrhotite?)	5%
Epidote, sphene (difficult to separate)	2%
Rutile	2%
Sphene	<1%
Chalcopyrite	<1%

Most of the chips in this sample consist of fine-grained quartz-chlorite-minor epidote-rutile-sphene, with lens-like or locally blastic aggregates of carbonate-minor sulfide-quartz-chlorite, grading to vein-like aggregates of carbonate-quartz-sulfide-minor chlorite-rutile.

In the first type, the lensy aggregates of carbonate (mostly irregular/flattened, <1 mm thick by up to 3 mm long, aligned with the foliation, possibly gradational to vein-like aggregates, all mostly composed of interlocking sub/euhedra <0.35 mm long (length:width ratios mostly <2:1 in the plane of foliation), are likely mainly dolomite to judge by the reaction in hand specimen, although there could be lesser ankerite around the margins where carbonate is more brownish. These could represent the sites of former mafic phenocrysts in mafic/intermediate rock although given the level of metamorphism/deformation this is speculative. However, it is supported by the presence in the matrix of abundant chlorite (strongly aligned, mainly euhedral flakes up to 0.1 mm, with weak but distinct green pleochroism but length-fast, weakly anomalous green birefringence suggestive of F:M around 0.4-0.5?) and common epidote-rutile ±sphene (aggregates to 0.2 mm of almost colourless to pale yellow, i.e. low Fe, epidote, golden brown acicular prisms to 0.1 mm of rutile, and pale brownish rounded subhedra <50 um of sphene), typically mixed with quartz as subhedra mostly <0.15 mm long (somewhat flattened in the plane of foliation, with length:width up to 1.5:1). Sulfide is only locally present (aggregates to 0.5 mm of cubic pyrite mainly <0.2 mm), associated and locally intergrown with carbonate and quartz in foliation-parallel stringers or bands up to 1.5 mm thick that may grade to the veins (see below).

In the veins (?) or vein-like segregations, carbonate forming interlocking ragged subhedra <1 mm, quartz forming subhedra <0.5 mm, and local sulfide (mainly pyrite as ragged euhedra up to ~2 mm, rare chalcopyrite <0.4 mm, both commonly in contact with carbonate) are intergrown with local chlorite forming subhedral flakes to 0.35 mm (optical properties as described above). Some of the pyrite aggregates have a porous texture composed of minute crystallites <20 um possibly suggestive of pyrite/marcasite after former pyrrhotite (?), and contain inclusions of or are intergrown with rutile (subhedra to 75 um) or chalcopyrite (subhedra to 0.15 mm).

In summary, this sample consists of quartz-carbonate-chlorite-epidote-rutile ±sphene, pyrite schist, with lensy or blastic carbonate aggregates (possibly after mafic minerals, now deformed?), suggestive of derivation from intermediate/mafic volcanic rocks partly mineralized with pyrite-trace chalcopyrite associated with irregular, poorly defined veins of carbonate-quartz-chlorite.

7: QUARTZ-SERICITE-CHLORITE-MINOR CARBONATE-PYRRHOTITE±PYRITE-RUTILE SCHIST (SULFIDES MAINLY ASSOCIATED WITH VEINS OF CARBONATE-QUARTZ)

Sample (from HC10-81) consists of angular chips of medium/dark greenish-grey coloured, mainly schistose rock mostly <1 cm across (cone crushed -1/4 fraction). Some chips are white and harder than steel (mainly quartz veins?). The chips are locally weakly magnetic, but show no reaction to cold dilute HCl (even where powdered), and no stain for K-feldspar in the etched offcut. Modal mineralogy in polished thin section prepared from the grain mount in epoxy is approximately:

Quartz (partly secondary)	55%
Sericite	20%
Chlorite	20%
Carbonate (ankerite?), mainly veins	3%
Pyrrhotite, trace included chalcopyrite	1%
Pyrite	<1%
Rutile	<1%

Most of the chips in this sample consist of foliated, fine-grained quartz and sericite/chlorite (so intimately mixed that proportions are difficult to estimate) with minor carbonate (mostly associated with vein-like segregations of quartz and rare sulfide, accessory rutile). Alternating bands or layers up to about 2 mm thick are enriched in quartz (minor carbonate, lesser sulfides) or sericite-chlorite. A few chips show remnant quartz-phyric texture

In the quartz-rich bands, quartz forms interlocking ragged sub/anhedra mostly in the 50 um to 0.15 mm size range, locally displaying minor flattening/elongation with length:width ratios up to 1.5:1 in the plane of foliation (defined mostly by aligned sericite flakes), except for the relict quartz eyes which have rounded/ellipsoid outlines up to almost 1.5 mm long (locally recrystallized, with moderate to strong undulose extinction, sub-grain and planar feature development, local suturing of grain boundaries). However, in vein-like aggregates or segregations (possibly metamorphic “sweats”?) quartz may form interlocking subhedra up to almost 1 mm in diameter, typically strained as described above.

In the mica-rich bands, sericite forms mainly euhedral, strongly aligned/locally crenulated flakes <0.2 mm (but commonly semi-continuous optically for up to 1 mm). The sericite is commonly mixed with sub-parallel oriented, lesser or sub-equal amounts of chlorite of similar or smaller size that are difficult to detect due to 1) almost complete lack of colour (very faint greenish cast) and 2) near-zero birefringence, suggestive of F:M possibly in the 0.4-0.5 range (?) although this is speculative and would require SEM analysis to determine.

Carbonate is typically confined to quartz-rich bands or possibly vein-like segregations, where it forms sub/euhedral crystals mainly <0.5 mm in diameter (aggregates locally to ~1 mm long, or along thin veins up to 0.35 mm thick). The carbonate is possibly mainly ankerite since no reaction to HCl is detected even in powdered material (although the small quantities could render this conclusion suspect), and colour is brownish in thin section.

Sulfide is mainly pyrrhotite, forming aggregates to 1 mm long of interlocking ragged sub- to anhedra mainly <0.2 mm in diameter (rarely with included chalcopyrite <35 um), or lesser pyrite as cubic euhedra to ~1 mm, rarely with inclusions of rutile <20 um. Rutile more commonly occurs as euhedral acicular golden brown crystals to 75 um long, locally aggregating to ~0.15 mm. Sulfides, especially pyrrhotite, are locally in contact with carbonate.

In summary, this sample consists mainly of quartz-sericite-chlorite-minor carbonate-accessory sulfide (pyrrhotite ±pyrite)-rutile schist (after intermediate volcanic, locally quartz phyric?), in which sulfides are partly controlled by vein-like aggregates or segregations of carbonate-quartz along the foliation, and are locally in contact with carbonate.

8: QUARTZ-SERICITE-CHLORITE-MINOR CARBONATE-PYRRHOTITE±PYRITE-RUTILE SCHIST (SULFIDES MAINLY ASSOCIATED WITH VEINS OF CARBONATE-QUARTZ)

Sample (from HC10-81) consists of angular chips of medium/dark greenish-grey coloured, mainly schistose rock mostly <1 cm across (cone crushed -1/4 fraction). Some chips are white and harder than steel (mainly quartz veins?). The chips are locally weakly magnetic, but show only trace reaction to cold dilute HCl (only where powdered), and no stain for K-feldspar in the etched offcut. Modal mineralogy in polished thin section prepared from the grain mount in epoxy is approximately:

Quartz (partly secondary)	55%
Sericite	20%
Chlorite	20%
Carbonate (dolomite/ankerite?), mainly veins	3%
Pyrrhotite (trace inclusions of pentlandite?)	1%
Pyrite	<1%
Rutile	<1%
Chalcopyrite	<<1%

Most of the chips in this sample consist of fine-grained, foliated quartz and sericite/chlorite (so intimately mixed that proportions are difficult to estimate) with minor carbonate (mostly in coarse-grained quartz veins or vein-like segregations of quartz and rare sulfide, accessory rutile). Alternating bands or layers up to about 3 mm thick are enriched in quartz (minor carbonate, lesser sulfides) or sericite-chlorite. A few chips show remnant quartz-phyric texture

In the quartz-rich bands, quartz forms interlocking ragged sub/anhedra mostly in the <50 um to 0.15 mm size range, locally displaying minor flattening/elongation with length:width ratios up to 3.5:1 in the plane of foliation (defined mostly by aligned sericite flakes), except for the relict quartz eyes which have rounded/ellipsoid outlines up to almost 1.2 mm long (locally recrystallized, with moderate to strong undulose extinction, sub-grain and planar feature development, local suturing of grain boundaries). However, in veins and vein-like aggregates/segregations (possibly metamorphic “sweats”), quartz forms highly interlocked, ragged subhedra up to almost 4 mm in diameter, either strained as described above and containing minor carbonate and sulfides, or almost unstrained.

In the mica-rich bands, sericite forms mainly euhedral, strongly aligned/locally crenulated flakes <0.2 mm (but commonly semi-continuous optically for up to 1 mm). The sericite is commonly mixed with sub-parallel oriented, lesser or sub-equal amounts of chlorite of similar or smaller size that are difficult to detect due to 1) almost complete lack of colour (very faint greenish cast) and 2) near-zero to locally slightly length-fast, green anomalous birefringence, suggestive of F:M possibly in the 0.4-0.5 range (?) although this is speculative and would require SEM analysis to determine.

Carbonate is typically confined to quartz-rich bands, veins or vein-like segregations, where it forms sub/euhedral crystals mainly <0.5 mm in diameter (locally to ~1 mm long, or along thin veins up to 2 mm thick that are partly ribbon-banded)). The carbonate is possibly mainly ankerite and minor dolomite since reaction to HCl is muted even in powdered material (although the small quantities could render this conclusion suspect), and colour is partly brownish in thin section.

Sulfide is mainly pyrrhotite, forming aggregates to 1.5 mm long of interlocking ragged sub- to anhedra mainly <0.8 mm in diameter (rarely associated with chalcopyrite as subhedra to 0.1 mm, or containing flame-like <35 um long inclusions of pentlandite?), or lesser pyrite as cubic euhedra to ~1 mm, rarely with inclusions of euhedral rutile <35 um and amoeboid-shaped pyrrhotite <45 um long. Rutile more commonly occurs as euhedral acicular golden brown crystals up to 60 um long, locally aggregating to ~0.3 mm long. Sulfides, especially pyrrhotite, are locally in contact with carbonate.

In summary, this sample consists mainly of quartz-sericite-chlorite-minor carbonate-accessory sulfide (pyrrhotite ±pyrite, trace chalcopyrite, rare pentlandite?)-rutile schist (after intermediate volcanic, locally quartz phyric?), in which sulfides are partly controlled by vein-like aggregates or segregations of carbonate-quartz along the foliation, and are locally in contact with carbonate.

9: QUARTZ-CHLORITE-CARBONATE-RUTILE (INTERMEDIATE, MAFIC PHYRIC?) OR QUARTZ-SERICITE-RUTILE (FELSIC, QUARTZ PHYRIC) SCHIST, VEINS OF QUARTZ-CARBONATE-CHLORITE-PYRITE±CHALCOPYRITE, PYRRHOTITE

Sample (from HC07-27) consists of angular chips of pale greenish-grey coloured, mainly schistose rock mostly <1 cm across (cone crushed -1/4 fraction). Some chips are white and harder than steel (mainly quartz veins?), some are white and softer than steel (sericitic?). The chips are locally weakly magnetic, but show no visible reaction to cold dilute HCl (even where powdered), and no stain for K-feldspar in the etched offcut. Modal mineralogy in polished thin section prepared from the grain mount in epoxy is approximately:

Quartz (partly secondary)	40%
Sericite	25%
Chlorite	15%
Carbonate (mainly ankerite?)	15%
Pyrite (partly after pyrrhotite?), trace chalcopyrite, pyrrhotite	4-5%
Rutile	<1%

Most of the chips in this sample consist of fine-grained quartz-sericite-chlorite-rutile with local lensy or blastic carbonate ±sericite (muscovite)-minor sulfide, or more strongly foliated quartz-sericite-minor rutile with local relict phyric quartz, locally with lensy carbonate-sulfide, grading to vein-like aggregates of carbonate-quartz-local chlorite-sulfide.

In the first type, the aggregates of carbonate (mostly irregular, <1 mm, but locally up to almost 2 mm long where gradational to vein-like aggregates, all mostly composed of interlocking sub/euhedra <0.35 mm, likely mainly ankerite to judge by lack of reaction in hand specimen?) and local sericite or muscovite (euhedral flakes to 0.15 mm) may represent the sites of former mafic phenocrysts in mafic/intermediate rock. This conclusion is supported by the presence in the matrix of locally significant amounts of chlorite (somewhat aligned, subhedral flakes mostly <0.1 mm, almost colourless, with length-fast, weakly anomalous first-order greenish-grey birefringence suggestive of F:M around 0.4?) and rutile (golden brown stubby euhedral prisms mostly <30 um), mixed with abundant quartz as subhedra mostly <0.15 mm in size (only rarely flattened in the plane of foliation, with length:width up to ~1.5:1). Sulfide is only locally present (aggregates to 0.235 mm of cubic pyrite mainly <0.2 mm), associated/commonly intergrown with carbonate and quartz in foliation-parallel stringers or bands up to 1.5 mm thick that may grade to the veins (see below). These chips may represent intermediate, originally mafic phyric volcanics.

In the second type, quartz and sericite are present in varying proportions, generally segregated into discontinuous foliae of quartz (interlocking subhedra <0.2 mm but locally in relict phenocrysts with rounded/ellipsoid outlines up to 1.5 mm long and moderate/strong strain indicated by undulose extinction, sub-grain development, suturing of grain boundaries) or of sericite (aligned sub/euhedral flakes <0.5 mm but commonly semi-continuous optically for up to ~1 mm), the latter typically with scattered aggregates of minute golden brown rutile mainly <20 um long. In some chips, there may be lesser chlorite intimately mixed with the sericite, and minor carbonate mixed with the quartz. These chips may represent more felsic/intermediate, quartz-phyric volcanics (?).

In the vein (?) chips, carbonate forming interlocking subhedra <0.5 mm, quartz forming subhedra to 0.7 mm, and local sulfide (mainly pyrite as ragged subhedra up to ~2 mm, locally with inclusions or veined by chalcopyrite <0.3 mm, both locally in contact with carbonate) are intergrown. Locally, significant chlorite forming subhedral flakes to 0.7 mm (optical properties as described above) is intergrown with quartz and sulfide. Relatively rare pyrrhotite forms aggregates to 0.2 mm, locally with inclusions of rutile (subhedra to 15 um) or chalcopyrite (subhedra to 40 um).

In summary, this sample contains quartz-chlorite-carbonate-rutile ±pyrite schist with local lensy or blastic carbonate ±sericite aggregates (after mafic phenocrysts?), suggestive of derivation from intermediate/mafic volcanics, and quartz-sericite ±chlorite-rutile schists with local relict phyric quartz, both partly mineralized with pyrite-minor chalcopyrite, pyrrhotite partly associated with veins of carbonate-quartz-chlorite.

10: FOLIATED SERICITE-MINOR QUARTZ-CARBONATE-PYRITE-RUTILE, AND COARSER QUARTZ-SERICITE-CARBONATE-PYRITE±CHALCOPYRITE-PYRRHOTITE-SPHALERITE-RUTILE SCHIST; COARSE QUARTZ-MINOR CARBONATE-SULFIDE VEINS

Sample (from HC07-22) consists of angular chips of pale grey/white coloured, mainly schistose rock mostly <1 cm across (cone crushed -1/4 fraction). Some chips are white and harder than steel (mainly quartz veins?), some are buff and softer than steel (sericitic?). The chips are locally weakly magnetic, but show no visible reaction to cold dilute HCl (even where powdered), and no stain for K-feldspar in the etched offcut. Modal mineralogy in polished thin section prepared from the grain mount in epoxy is approximately:

Quartz (partly secondary)	55%
Sericite	25%
Carbonate (mainly ankerite?)	15%
Pyrite	3%
Rutile	<1%
Chalcopyrite	<1%
Pyrrhotite	<1%
Sphalerite	<1%
Tourmaline (intermediate dravite-schorl?)	trace

Most of the chips in this sample consist of either strongly foliated, fine-grained sericite or quartz-sericite-carbonate-sulfide-rutile schist, or are chips of coarser, quartz-carbonate vein material.

Quartz-sericite schist consists of variable proportions of sericite as strongly aligned euhedral flakes mostly <0.15 mm (but commonly optically continuous for up to 2 mm along the foliation) and lesser quartz as mostly flattened/elongated subhedra <0.15 mm long (length:width up to 3:1) plus local minor carbonate (subhedra <50 um) or pyrite (see below) and accessory rutile (euhedra <30 um long).

Somewhat coarser-grained, more quartz-rich, carbonate-bearing schist chips consist of interlocking sub/anhedral quartz commonly <0.15 mm, but up to 0.3 mm in size, with either interstitial carbonate (aggregates to 0.5 mm of sub/anhedra <0.15 mm, likely mainly ankerite since there is no reaction in hand specimen) or sericite (wispy foliae of sub/euhedral flakes as described above), accessory pyrite and rutile (as described below). Sulfides tend to be less common in this rock type than in the sericite-rich schist, with the result that much of the pyrite is not in contact with carbonate in this sample. Very rare tourmaline forming slender acicular needles to 75 um long oblique to foliation shows pale greenish/brownish pleochroism suggestive of F:M around 0.4-0.5 (?).

Vein fragments consist mainly of very coarse-grained quartz (subhedra up to 2.5 mm, or aggregates of <1 mm ragged sub/anhedra, both displaying strong undulose extinction, sub-grain development, and suturing of grain boundaries), local carbonate (ragged aggregates to 1 mm long of brownish subhedra <0.1 mm, likely ankeritic) and pyrite (elongated aggregates to 1.5 mm that are typically in contact with either quartz or sericite).

Pyrite forms fractured cubic sub/euhedra up to 1.5 mm in coarse carbonate-quartz veins, or aggregates to 2 mm of subhedra mainly <0.3 mm, aligned along foliation in schists and intergrown with sericite and accessory rutile forming aggregates to 0.4 mm of acicular euhedra to 0.3 mm long. Chalcopyrite forms ragged subhedra <0.15 mm, typically intergrown with the carbonate in quartz-carbonate veins, rarely with sphalerite (subhedra <0.1 mm, virtually opaque due to micron-sized inclusions of chalcopyrite). Pyrrhotite is relatively rare, forming ragged aggregates to 0.2 mm of subhedra <50 um, typically enclosed in carbonate.

In summary, this sample contains chips of strongly foliated, sericite-rich (minor quartz-carbonate-local pyrite-accessory rutile) or less foliated, quartz-minor carbonate-sericite-accessory pyrite-chalcopyrite-sphalerite-pyrrhotite-rutile schist, plus coarse-grained, vein quartz with minor carbonate, sericite, and sulfides (the latter generally not in contact with carbonate).

10D: FOLIATED SERICITE-MINOR QUARTZ-CARBONATE-PYRITE-RUTILE, COARSER QUARTZ-SERICITE-CARBONATE-PYRITE-RUTILE ±CHALCOPYRITE-PYRRHOTITE SCHIST; AND COARSE QUARTZ-MINOR CARBONATE-SULFIDE-CHLORITE VEINS

Sample (from HC07-22D) consists of angular chips of pale grey/white coloured, mainly schistose rock mostly <1 cm across (cone crushed -1/4 fraction). Some chips are white and harder than steel (mainly quartz veins?), some are buff and softer than steel (sericitic?). The chips are locally weakly magnetic, but show no visible reaction to cold dilute HCl (even where powdered), and no stain for K-feldspar in the etched offcut. Modal mineralogy in polished thin section prepared from the grain mount in epoxy is approximately:

Quartz (partly secondary)	55%
Sericite	30%
Carbonate (mainly ankerite?)	10%
Chlorite	2-3%
Pyrite	2-3%
Rutile	<1%
Chalcopyrite	<1%
Pyrrhotite	<1%
Tourmaline (intermediate dravite-schorl?)	trace

Most of the chips in this sample consist of either strongly foliated, fine-grained sericite or quartz-sericite-carbonate-sulfide-rutile schist, or of coarser, quartz-carbonate ±sulfide-chlorite veins (?).

Quartz-sericite schist consists of variable proportions of sericite as strongly aligned euhedral flakes mostly <0.25 mm (but locally semi-continuous optically for up to 1.5 mm along the foliation) and lesser quartz as subhedra <0.1 mm (rarely flattened/elongated with length:width up to 2:1) plus local minor carbonate (subhedra <50 um, typically distributed along the foliation) or pyrite (see below) and accessory rutile (euhedra <30 um long).

Somewhat coarser-grained, more quartz-rich, carbonate-bearing schist chips consist of interlocking sub/anhedral quartz commonly <0.15 mm, but locally up to 0.3 mm in size, with either interstitial or blastic carbonate (aggregates to 0.75 mm of sub/anhedra mostly <0.25 mm or rarely euhedra to 0.75 mm, likely mainly ankerite since it is commonly brown, and no reaction noted in hand specimen) or sericite (wispy foliae of sub/euhedral flakes as described above), local chlorite (subhedral flakes <0.1 mm), accessory pyrite and rutile (see below). Sulfides tend to be more common in this rock type than in sericite-rich schist, with the result that some pyrite is in contact with carbonate in this sample. Very rare tourmaline forms slender acicular needles to 0.15 mm long oblique to foliation, with very pale greenish-brown pleochroism suggestive of F:M around 0.3-0.4?.

Vein fragments consist mainly of very coarse-grained quartz (subhedra up to 4.5 mm, or aggregates of <2 mm ragged sub/anhedra, both displaying strong undulose extinction, sub-grain development, and suturing of grain boundaries indicative of strain), local carbonate (ragged aggregates to 1 mm long of brownish subhedra <0.1 mm or euhedra to 0.5 mm, likely ankerite?) and pyrite (elongated aggregates to 1.5 mm that are typically in contact with either quartz or sericite) or chlorite (subhedral flakes to 0.3 mm with pale green pleochroism/near-zero birefringence, F:M 0.5?).

Pyrite forms fractured cubic sub/euhedra up to 1 mm in coarse carbonate-quartz veins, or aggregates to 2.5 mm of subhedra <0.2 up to 1 mm, aligned along foliation in schists and intergrown with sericite, chlorite and locally carbonate. Accessory rutile forms aggregates to 0.3 mm of acicular euhedra to 0.15 mm long. Pyrite rarely contains chalcopyrite<0.1 mm; chalcopyrite also forms ragged subhedra <0.2 mm, locally intergrown with the carbonate in quartz-carbonate veins. Pyrrhotite is rare, forming aggregates or ragged subhedra <0.1 mm, typically enclosed in carbonate.

In summary, this sample contains chips of strongly foliated, sericite-rich (minor quartz-carbonate-local pyrite-accessory rutile) or less foliated, quartz-minor carbonate-sericite-pyrite-chalcopyrite-rare pyrrhotite-accessory rutile schist, plus coarse-grained, vein quartz with minor carbonate, sericite, chlorite and sulfides. Sulfides are mostly only locally in contact with carbonate.

11: CHLORITE-, SERICITE-, OR QUARTZ-CARBONATE-MUSCOVITE±CHLORITE-PYRITE-RUTILE SCHIST/PHYLLITE; VEINS OF CARBONATE-QUARTZ-PYRITE±CHALCOPYRITE

Sample (from HC06-07) consists of angular chips of pale greenish buff coloured, mainly schistose rock mostly <1 cm across (cone crushed -1/4 fraction). Some chips are white and harder than steel (mainly quartz veins?), some are buff and softer than steel (sericitic?). The chips are locally slightly magnetic, but show no visible reaction to cold dilute HCl (even where powdered), and no stain for K-feldspar in the etched offcut. Modal mineralogy in polished thin section prepared from the grain mount in epoxy is approximately:

Quartz (partly secondary)	45%
Chlorite	20%
Sericite	20%
Carbonate (mainly ankerite?)	10%
Pyrite	3-5%
Plagioclase (albite, secondary/vein)	<1%
Rutile	<1%
Chalcopyrite	<1%

Most of the chips in this sample may be classified as fine-grained, either chlorite- or sericite-rich schist, or quartz-carbonate-sericite/muscovite-minor chlorite- accessory sulfide (mainly pyrite)-rutile semi-schist, plus chips of vein material (coarse-grained carbonate-quartz-sulfides)

The former are dominated by either chlorite (closely packed, strongly aligned, subhedral flakes mostly <0.1 mm with almost no colour/pleochroism and length-fast, weakly anomalous, first-order greenish-grey birefringence suggestive of slightly magnesian composition with F:M around 0.4?) or sericite (typically also closely packed, generally aligned, sub/euhedral flakes mostly <0.2 mm but locally optically continuous for at least 1 mm). Both contain minor quartz as subhedral, locally somewhat elongated, crystals mostly <65 um long (length:width ratios up to 1.5:1, partly aligned with the foliation), particularly concentrated in thin discontinuous bands or laminae up to ~0.5 mm thick. In places the chlorite- or sericite-rich rock can be seen adjacent or attached to the coarser-grained, quartz-rich rock, suggesting they occur as alternating bands in the undisturbed rock from which the chips are derived. Accessory rutile (see next paragraph) is common in chlorite-rich and to a lesser extent in sericite-rich rock.

The quartz-rich rock consists mainly of interlocking, more or less randomly oriented, ragged subhedra of quartz mostly <0.1 but up to 0.25 mm, intergrown with variable carbonate (ragged subhedra mostly <0.1 but up to 0.7 mm, locally aggregating to 1.7 mm, possibly ankerite since they are commonly brownish in thin section and show no reaction to HCL in hand specimen), sericite or muscovite as mainly euhedral, randomly oriented flakes up to 0.5 mm, local chlorite as sub/euhedral flakes to 0.5 mm (as described above), and rutile as aggregates mostly <0.1 mm of pale golden yellow sub/euhedra mostly <50 um long, locally intergrown with euhedral pyrite of similar size.

Vein chips are composed of coarse carbonate (interlocking subhedra to almost 2.5 mm, pale brownish, likely ferroan dolomite, i.e. ankerite) and lesser quartz (interlocking ragged sub/anhedra <0.6 mm) with local variable sulfides (mainly pyrite) and trace rutile. Both quartz and carbonate show evidence of strain (undulose extinction, sub-grain development, suturing of grain boundaries). Pyrite forms cubic sub/euhedra mostly <1 mm but locally in loose aggregates up to 3 mm across in veins, where the crystals are commonly in contact with, or locally partly to completely encased in, carbonate, which also tends to contain the very minor chalcopyrite as ragged subhedra <0.1 mm. Traces of rutile occur as inclusions (sub/euhedra <50 um) both in pyrite and in the carbonate. Rare thin (<0.5 mm) veinlets of albitic plagioclase (subhedra <1mm with negative relief compared to quartz and extinction on 010 to 16 degrees), carbonate and pyrite cut the quartz-rich rock chips.

In summary, this sample consists of chlorite-, sericite-, or quartz-rich schist or semi-schist, the latter typically with variable carbonate, sericite/muscovite and minor chlorite; all contain accessory rutile and local pyrite. Most sulfides (pyrite, trace chalcopyrite) occur in vein chips of carbonate-quartz where they are in contact with or enclosed in the carbonate.

12: FINE AMPHIBOLE-CHLORITE-QUARTZ-CARBONATE-EPIDOTE-SPHENE SCHIST (BLASTIC ALBITE), VEINS OF CARBONATE-QUARTZ-CHLORITE-PYRITE-AMPHIBOLE

Sample (from HC07-16) consists of angular chips of medium green coloured (chloritic?), mainly schistose rock mostly <1 cm across (cone crushed -1/4 fraction). Some chips are entirely pyrite (aggregates to 4 mm). The chips are locally slightly magnetic, and show minor reaction to cold dilute HCl (only where powdered), but no stain for K-feldspar in the etched offcut. Modal mineralogy in polished thin section prepared from the grain mount in epoxy is approximately:

Amphibole (tremolite-actinolite, hornblende?)	35%
Chlorite	20%
Quartz (largely secondary)	20%
Carbonate (mainly ankerite?)	10%
Epidote (clinozoisite?)	5%
Pyrite	5%
Sphene (trace rutile inclusions <20 um)	3%
Plagioclase (albite, blastic)	1-2%
Chalcopyrite	<1%

Most chips in this sample consist of fine-grained, amphibole-chlorite-epidote-sphene-minor quartz-carbonate schist. There are also common coarse-grained, mainly vein chips composed of carbonate, quartz, chlorite (partly after amphibole), sulfides (mainly pyrite), and minor epidote.

In the schist chips, amphibole and chlorite are typically intimately intermixed or interleaved. Amphibole forms mainly euhedral fibrous crystals <0.15 mm long, with interstitial chlorite as mainly euhedral flakes <0.1 mm in diameter, both generally strongly aligned in and defining the foliation. Most chlorite has distinct green pleochroism and length-fast, weakly anomalous green birefringence indicative of F:M around 0.5 (?). Proportions vary between mainly amphibole-minor chlorite and mainly chlorite-minor amphibole, typically with minor to significant epidote and sphene as small clots along the foliation, locally mixed with carbonate. Epidote (mainly colourless, low Fe, possibly clinozoisite?) and sphene occur as small (<0.1 mm) abundant irregular aggregates of euhedra <25 um or brownish subhedra <20 um, respectively, typically intergrown with the chlorite and sericite. Rarely, the aggregates contain minor red-brown pleochroic allanite (?) as subhedra <50 um, and are surrounded by fibrous colourless/pale green amphibole euhedra to 0.3 mm long with small extinction angle (tremolite-actinolite?), partly altered at margins to chlorite. Some amphibole is concentrated in separate bands up to 1.5 mm thick (that appear to grade to veins), or in blastic crystals up to 1.5 mm of darker green amphibole. Quartz is present as flattened/elongated subhedra mainly <0.1 mm long (length:width up to 2:1) and is locally concentrated in bands up to 1.5 mm thick that appear to grade to the veins. Carbonate locally occurs as blastic sub/euhedra mainly <0.25 mm, or as thin veinlets <0.2 mm thick, mainly dolomite to judge by the reaction in hand specimen (± dark brown ankerite?). Rare albite (vaguely twinned, negative relief against quartz) forms blastic sub/euhedra <0.5 mm.

In the veins, which are up to at least 3 mm thick, carbonate forms interlocking sub/euhedra up to ~2 mm (mostly dolomite), with lesser quartz as rounded subhedra to 1.2 mm. Quartz in particular displays moderate/strong undulose extinction, sub-grain development, and local suturing of grain boundaries. Chlorite as subhedral flakes to 0.5 mm locally surrounds/replaces amphibole as dark green ragged subhedra to 2 mm (hornblende?), associated with epidote to 0.5 mm. Sulfides are mainly intergrown with chlorite and quartz, rarely in contact with carbonate. Pyrite occurs mostly in vein chips where it forms sub/euhedral cubic crystals <1.5 mm in size (but in aggregates up to 3.5 mm across), locally with chalcopyrite as rounded subhedral inclusions to 0.2 mm or as adjoining aggregates to 1 mm long. Separate cubic crystals to 4 mm, with porous cores caused by inclusions of gangues (quartz and sericite?) <0.2 mm are probably also from veins.

In summary, this sample is mainly fine-grained amphibole-chlorite-epidote-sphene-minor quartz-carbonate schist (local blastic amphibole, carbonate, albite possibly representing former mafic and rare feldspar phenocryst sites?), cut by coarser-grained veins and veinlets of carbonate-quartz-chlorite-amphibole-pyrite ±chalcopyrite, local epidote.

13: FINE-GRAINED VARIABLY FOLIATED QUARTZ-CARBONATE-SERICITE-CHLORITE± PYRITE-RUTILE SCHIST VEINED BY COARSER-GRAINED QUARTZ-CARBONATE-PYRITE-SPHALERITE-CHALCOPYRITE±GALENA, LOCAL CHLORITE, MUSCOVITE

Sample (from HC10-77) consists of flat, angular chips of pale greenish coloured (chloritic?), mainly schistose rock up to 2.5 cm across (cone crushed -1/4 fraction). Some chips are mainly veins of quartz-pyrite (aggregates to 4 mm). The chips are not magnetic, show only trace reaction to cold dilute HCl (only where powdered), and no stain for K-feldspar in the etched offcut. Modal mineralogy in polished thin section prepared from the grain mount in epoxy is approximately:

Quartz (partly secondary)	35%
Carbonate (mainly dolomite/ankerite?)	25%
Sericite	15%
Chlorite (slightly magnesian?)	10%
Pyrite (trace inclusions of chalcopyrite, sphalerite, galena)	10%
Rutile	~2%
Sphalerite (trace inclusions of chalcopyrite)	1-2%
Chalcopyrite	~1%
Galena (inclusions in pyrite)	<<1%

Most of the chips in this sample consist of variable proportions of fine-grained quartz, carbonate, sericite, chlorite, accessory pyrite and rutile semi-schist or phyllite with weak to moderate foliation, or coarser-grained veins of quartz-carbonate-sulfides-chlorite-muscovite.

Wallrock typically consists mostly of quartz (interlocking ragged sub/anhedra mainly <0.2 mm, with moderate/strong strain indicated by undulose extinction, sub-grain development, and suturing of grain boundaries, local planar features), carbonate as interlocking subhedra mainly <0.15 mm (dolomite and/or ankerite to judge by the lack of reaction in hand specimen for such an abundance), and lesser sericite (mainly euhedral flakes <0.2 mm but locally optically continuous for up to 2-3 mm where strongly aligned in and defining the foliation) and chlorite (sub/euhedral flakes mostly <50 um, locally but not everywhere aligned with the foliation, virtually colourless and with length-fast, first-order grey to weakly anomalous greenish birefringence indicating F:M around 0.4?). Locally, minor pyrite occurs as cubic sub/euhedra mainly <0.2 mm in diameter, in part associated with or separate from locally significant concentrations of rutile (aggregates to 0.4 mm of minute, randomly oriented dark brown/opaque acicular euhedra mostly <10 um long, or in places pale golden brown euhedra to 45 um).

In the veins, which are up to at least 0.5 cm thick and commonly sub-parallel to foliation, quartz forms interlocking, commonly fractured, strained subhedra up to at least 2.5 mm long displaying strong undulose extinction, sub-grain/planar feature development, and local suturing of grain boundaries. Carbonate forms mostly interlocking subhedra to about 1 mm diameter, also with indications of strain (undulose extinction, sub-grain development, suturing of grain boundaries), and possibly mainly dolomite (where clear) or ankerite (where brownish). Sulfides are mainly pyrite (subhedra to 2 mm, aggregates to 0.5 cm, local inclusions of chalcopyrite <0.1 mm, galena <45 um) with lesser sphalerite (orange-brown, i.e. low/moderate Fe, subhedra to 1.5 mm with trace inclusions of chalcopyrite <50 um) and minor chalcopyrite (ragged aggregates to 0.7 mm of subhedra <0.3 mm). Chlorite locally occurs as euhedral flakes to 0.6 mm with optical properties as described above, and sericite (muscovite) forms euhedral flakes to 0.35 mm. Only part of the pyrite is in contact with the carbonate (except where associated with sphalerite), but most of the sphalerite and chalcopyrite are enclosed in or partly surrounded by carbonate.

In summary, this sample consists mainly of fine-grained, weakly/moderately foliated, quartz-carbonate-sericite-chlorite-accessory pyrite-rutile schist cut by veins of coarser-grained secondary quartz-carbonate-pyrite-sphalerite-chalcopyrite-trace galena, local chlorite-muscovite.

14: FINE-GRAINED VARIABLY FOLIATED QUARTZ-SERICITE-CARBONATE± PYRITE-RUTILE SCHIST VEINED BY COARSER-GRAINED QUARTZ-CARBONATE-PYRITE ± CHALCOPYRITE, LOCAL CHLORITE

Sample (from HC07-14) consists of flat, angular chips of pale buff coloured, soft (sericitic?), mainly schistose rock mostly <1.5 cm across (cone crushed -1/4 fraction). Some chips are mainly veins of quartz-minor pyrite. The chips are not magnetic, show no reaction to cold dilute HCl (even where powdered), and no stain for K-feldspar in the etched offcut. Modal mineralogy in polished thin section prepared from the grain mount in epoxy is approximately:

Quartz (partly secondary)	60%
Sericite	25%
Carbonate (mainly ankerite?)	10%
Pyrite (minor inclusions of chalcopyrite)	2-3%
Chlorite (veins only, slightly magnesian?)	~1%
Rutile	~1%
Chalcopyrite	~1%

Most of the chips in this sample consist of mostly fine-grained, variably textured, variably foliated quartz-sericite-minor carbonate-accessory pyrite-rutile schist or phyllite, locally with quartz “eyes” or relict phenocrysts or cut by irregular, poorly defined veins of coarser-grained quartz-carbonate-sulfides-local minor chlorite.

In the wallrock, proportions vary from mostly (>75%) quartz, minor sericite and carbonate, accessory pyrite and rutile, to mainly (>90%) sericite (minor carbonate, pyrite, rutile, mostly concentrated in foliation-parallel, poorly defined zones that are gradational to the veins). In the former, quartz occurs as tightly interlocking, sub/anhedral crystals mostly <0.15 mm in diameter, typically without noticeable flattening/elongation in the plane of foliation defined by alignment of sericite flakes (mainly sub/euhedral, <0.1 mm) and local concentrations of carbonate forming blastic aggregates to ~1 mm in size (relatively clear, possibly dolomite, after former mafic crystals?) or elongated aggregates generally closely associated/intergrown with the accessory pyrite forming subhedral cubic crystals mostly <0.2 mm and rutile (aggregates to 0.2 mm of stubby euhedra mostly <30 um long). In the latter, sericite forms strongly aligned sub/euhedral flakes mainly <0.1 mm in size (but commonly optically continuous for up to several mm), carbonate occurs in irregular elongated aggregates <0.3 mm thick of ragged subhedra mostly <0.2 mm (distinctly brown, likely mainly ankerite), and quartz occurs as scattered subhedra mainly <0.1 mm. Rutile forms pale golden brown euhedral prisms <40 um long. Relict quartz phenocrysts in both types have rounded subhedral outlines up to about 1 mm and are recrystallized; with moderate strain in both phenocrysts and matrix indicated by undulose extinction, sub-grain development, and rare suturing of grain boundaries.

In the veins, which are poorly defined, irregular, up to about 3 mm thick, quartz forms interlocking ragged sub/anhedra mainly <0.5 mm with moderate undulose extinction, sub-grain development, and rare suturing of grain boundaries. Carbonate forms ragged subhedra to 0.6 mm that are mostly brownish (ankerite?) and rarely surround or vein pyrite crystals, where they are associated with chlorite as subhedral flakes to 0.3 mm (almost colourless, length-fast first-order grey birefringence indicative of slightly magnesian F:M around 0.4?). Pyrite forms sub- to euhedral cubic crystals up to ~1 mm diameter (locally fractured and with chalcopyrite either as inclusions <0.1 mm or along the fractures mostly <50 um thick), mostly enclosed in quartz and not in contact with carbonate. Chalcopyrite also occurs as ragged subhedra to 0.5 mm, mostly in contact with or partly surrounded by/associated with carbonate.

In summary, this sample consists of mostly fine-grained, variably textured, variably foliated quartz-sericite-minor carbonate-accessory pyrite-rutile schist or phyllite, locally with quartz “eyes” or relict phenocrysts and rare blastic carbonate possibly after mafic phenocrysts, cut by irregular, poorly defined veins of coarser-grained quartz-carbonate-pyrite-chalcopyrite-local minor chlorite.

15: FINE-GRAINED VARIABLY FOLIATED QUARTZ-CARBONATE-SERICITE± PYRITE-RUTILE SCHIST VEINED BY COARSER-GRAINED QUARTZ-CARBONATE-PYRITE ± CHALCOPYRITE, LOCALLY OXIDIZED TO LIMONITE

Sample (from HC07-21) consists of angular chips of pale grey-buff coloured (sericitic?), mainly schistose rock mostly <1.5 cm across (cone crushed -1/4 fraction). Some chips are mainly veins of quartz-minor pyrite; some are stained red by limonite. The chips are not magnetic, show no reaction to cold dilute HCl (even where powdered), and no stain for K-feldspar in the etched offcut. Modal mineralogy in polished thin section prepared from the grain mount in epoxy is approximately:

Quartz (partly secondary)	65%
Carbonate (mainly ankerite, minor siderite?)	15%
Sericite	15%
Pyrite (trace inclusions of chalcopyrite)	3-5%
Chalcopyrite	<1%
Rutile	<1%
Limonite (hematitic, after pyrite)	<1%

Most chips in this sample consist of mainly fine-grained, variably textured, variably foliated quartz-sericite-local carbonate-accessory pyrite-rutile schist/phyllite, locally with prominent quartz “eyes” or relict phenocrysts, or cut by irregular, poorly defined veins of coarser-grained quartz-carbonate-sulfides (mainly pyrite, partly oxidized locally to limonite).

In the wallrock, proportions vary from mostly (>75%) quartz, minor sericite and carbonate, accessory pyrite and rutile, to mainly (>75%) carbonate (minor sericite, pyrite, rutile, mostly concentrated in foliation-parallel, poorly defined zones that are gradational to the veins, or mainly (>90%) sericite. In the first, quartz occurs as tightly interlocking, sub/anhedral crystals mostly <0.2 mm in diameter, typically without noticeable flattening/elongation in the plane of foliation that is more or less defined by alignment of sericite flakes (mainly sub/euhedral, <0.2 mm) and local concentrations of carbonate forming either blastic aggregates <0.5 mm (brownish, possibly ankerite, after former mafic crystals?) or elongated aggregates along what may be fractures/microveinlets, closely associated/intergrown with the accessory pyrite forming subhedral cubic crystals mostly <0.2 mm or the rutile, which occurs in aggregates <0.1 mm of stubby euhedra mostly <30 um long. In the second (carbonate-rich), carbonate occurs as weakly foliated aggregates to ~1 mm thick of ragged interlocking dark brown (ankeritic?) sub/anhedra mainly <0.1 mm associated/intergrown with pyrite as described above, minor sericite forms euhedra <0.1 mm. In the third, sericite forms strongly aligned sub/euhedral flakes mainly <0.1 mm but locally optically continuous for up to several mm, and quartz occurs as scattered subhedra mainly <0.1 mm; rutile forms pale golden brown euhedral prisms <40 um long. Relict quartz phenocrysts mostly in the first have rounded subhedral outlines up to ~1.5 mm long or are recrystallized; with moderate strain in both phenocrysts and matrix indicated by undulose extinction, sub-grain development, and rare suturing of grain boundaries.

In the veins, which are poorly defined, irregular, mostly <1 mm thick, quartz forms ragged interlocking sub/anhedra mainly <0.75 mm with moderate undulose extinction, sub-grain development, and rare suturing of grain boundaries. Carbonate forms ragged subhedra to 0.6 mm that are dark brownish or limonite-stained (ankerite or siderite?) and rarely surround or vein pyrite crystals, where they are associated with limonite as microcrystalline (dark red-brown, hematitic?) likely after pyrite. Pyrite forms sub- to euhedral cubic crystals up to ~1 mm diameter (locally fractured and with chalcopyrite either as inclusions <0.1 mm or along the fractures mostly <50 um thick), mostly enclosed in quartz and not in contact with carbonate. Chalcopyrite also occurs as ragged subhedra to ~1 mm, mostly in contact with or partly surrounded by/associated with carbonate.

In summary, this sample consists of mostly fine-grained, variably textured, variably foliated quartz-carbonate-minor sericite-accessory pyrite-rutile schist or phyllite, locally with quartz “eyes” or relict phenocrysts and rare blastic carbonate possibly after mafic phenocrysts suggestive of derivation from felsic volcanic rock, cut by irregular, poorly defined veins of coarser-grained quartz-carbonate-pyrite-chalcopyrite locally partly oxidized to minor limonite and in contact with Fe-carbonate.

16: MEDIUM/FINE-GRAINED VARIABLY FOLIATED QUARTZ-SERICITE±CARBONATE-PYRITE-RUTILE OR SERICITE-RICH SCHIST VEINED BY COARSER-GRAINED QUARTZ-CARBONATE-PYRITE-CHALCOPYRITE ± SPHALERITE

Sample (from HC07-14) consists of angular chips of pale grey-greenish/buff coloured (chloritic/sericitic?), partly schistose rock mostly <1 cm across (cone crushed -1/4 fraction). Many chips are mainly veins of quartz-pyrite. The chips are not magnetic, show no reaction to cold dilute HCl (even where powdered), and no stain for K-feldspar in the etched offcut. Modal mineralogy in polished thin section prepared from the grain mount in epoxy is approximately:

Quartz (partly/largely secondary)	65%
Sericite	15%
Carbonate (mainly ankerite?)	10%
Pyrite (trace inclusions of chalcopyrite)	8-10%
Chalcopyrite	~1%
Rutile	<1%
Sphalerite	trace

Most of the chips in this sample consist of medium-grained, variably textured, mainly weakly foliated quartz-sericite-local carbonate-pyrite-accessory rutile metamorphic rock with local quartz “eyes” or relict phenocrysts, or rarely well-foliated sericite schist, strongly affected by irregular, poorly defined veins of coarser-grained quartz-carbonate-sulfides (mainly pyrite, minor chalcopyrite).

In the wallrock chips, proportions vary from mostly (>75%) quartz, lesser sericite and minor carbonate, accessory pyrite and rutile, to mainly (>90%) sericite (minor quartz, rutile, rare pyrite). Irregular foliation-parallel/oblique, poorly defined zones or concentrations of coarser quartz, local carbonate, and sulfides appear to be gradational to the veins. In the first, quartz-rich type, quartz occurs as tightly interlocking, sub/anhedral crystals mostly <0.35 mm in diameter, only locally with noticeable flattening/elongation in the plane of foliation that is more or less defined by alignment of sericite flakes (mainly sub/euhedral, <0.25 mm) and local concentrations of carbonate forming either blastic aggregates <0.5 mm (brownish, possibly ankerite, after former mafic crystals?) or elongated aggregates along what are likely fractures/veinlets up to 0.25 mm thick. Minor accessory pyrite as subhedral cubic crystals mostly <0.2 mm (irregular aggregates to 1 mm) is locally associated with the carbonate, as is rutile, which occurs in aggregates <0.1 mm of stubby euhedra mostly <30 um long. In the second, sericite forms strongly aligned sub/euhedral flakes mainly <0.1 mm but generally optically continuous for up to 1 mm, and quartz occurs as scattered subhedra mainly <0.1 mm. Rutile forms pale golden brown euhedral prisms <40 um long. Relict quartz phenocrysts mostly in the first have rounded subhedral outlines <1 mm long or are partly recrystallized; with moderate to locally strong strain in phenocrysts (less in matrix) indicated by undulose extinction, sub-grain development, and local suturing of grain boundaries.

In the veins, which are poorly defined, irregular, up to 3 mm thick, quartz forms ragged interlocking sub/anhedra mainly <1 mm with moderate to strong undulose extinction, sub-grain development, and rare suturing of grain boundaries. Carbonate forms ragged subhedra to 0.6 mm that are mainly brownish (ankerite?) and only rarely surround pyrite and lesser chalcopyrite. Pyrite forms irregular sub- to locally cubic euhedral crystals up to 2.5 mm diameter (locally fractured and with chalcopyrite either as inclusions <0.1 mm or along grain boundaries/fractures mostly <0.2 mm thick). Pyrite is mostly enclosed in quartz and not in contact with carbonate, but chalcopyrite, which also occurs as ragged subhedra to ~1 mm (with rare pale yellow-brown, i.e. low Fe, sphalerite as sub/euhedra to 0.1 mm), is commonly in contact with or partly surrounded by/associated with carbonate.

In summary, this sample consists of medium/fine-grained, variably textured, variably foliated quartz-minor sericite-carbonate-accessory pyrite-rutile phyllite or sericite schist, locally with quartz “eyes” or relict phenocrysts and rare blastic carbonate possibly after mafic phenocrysts suggestive of derivation from felsic volcanic rock, cut by irregular, poorly defined veins of coarser-grained quartz-carbonate-pyrite-chalcopyrite-trace sphalerite, locally in contact with Fe-carbonate.

17: MEDIUM/FINE-GRAINED VARIABLY FOLIATED QUARTZ-SERICITE±CARBONATE-PYRITE-RUTILE OR SERICITE-RICH SCHIST VEINED BY COARSER-GRAINED QUARTZ-CARBONATE-PYRITE-CHALCOPYRITE-SPHALERITE

Sample (from HC10-77) consists of angular chips of pale grey-white coloured (sericitic?), partly schistose rock mostly <1 cm across (cone crushed -1/4 fraction). Many chips are mainly veins of quartz-sulfides. The chips are not magnetic, show no reaction to cold dilute HCl (even where powdered), and no stain for K-feldspar in the etched offcut. Modal mineralogy in polished thin section prepared from the grain mount in epoxy is approximately:

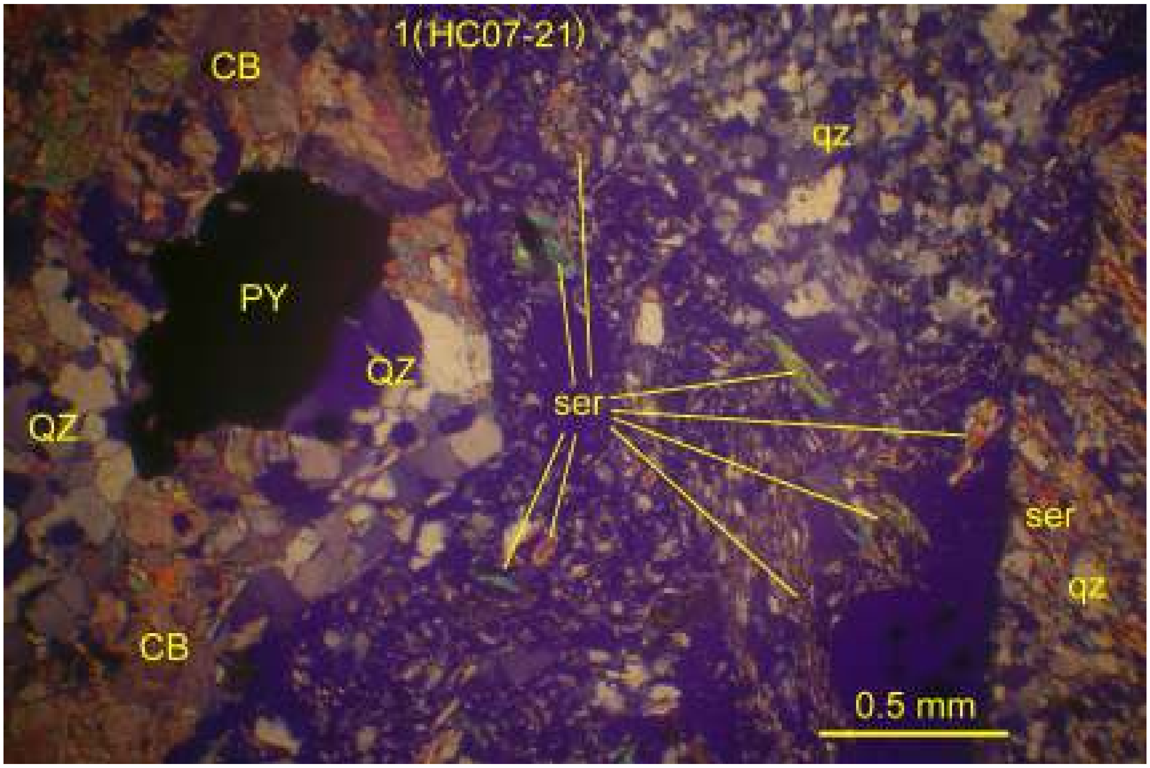
Quartz (partly/largely secondary)	70%
Sericite	20%
Pyrite (trace inclusions of chalcopyrite)	5%
Carbonate (dolomite, ankerite?)	3%
Chalcopyrite	~1%
Sphalerite	<1%
Rutile	<1%

Most chips consist of medium-grained, variably textured, weakly to moderately foliated quartz-sericite-local carbonate-pyrite-accessory rutile metamorphic rock with local quartz “eyes” or relict phenocrysts, or rarely well-foliated sericite schist, both strongly affected by irregular, poorly defined veins of coarser-grained quartz-carbonate-sulfides (mainly pyrite, minor chalcopyrite, sphalerite).

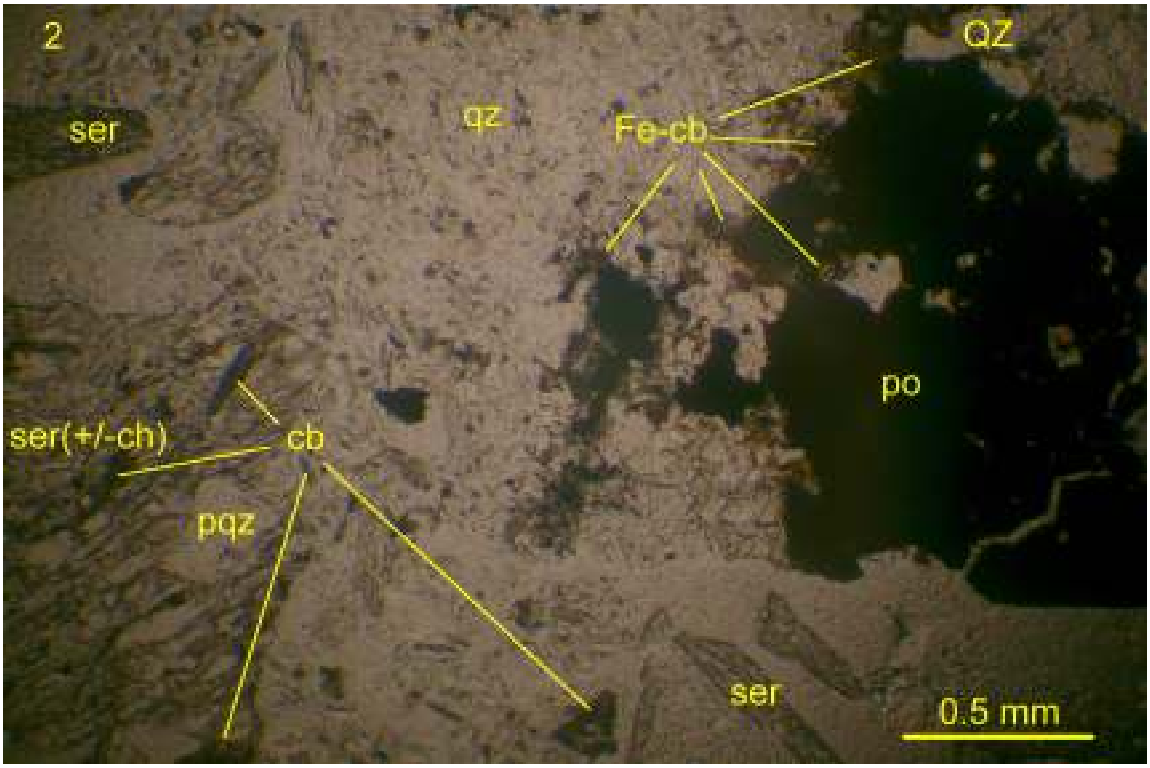
In the wallrock chips, proportions vary from mostly (>75%) quartz, lesser sericite and minor carbonate, accessory pyrite and rutile, to mainly (>90%) sericite (minor quartz, rutile, rare pyrite), through intermediate ratios around 50/50 quartz/sericite. Irregular foliation-parallel/oblique, poorly defined zones or concentrations of coarser quartz, local carbonate/sulfides appear to be gradational to the veins. In the first, quartz-rich type, quartz occurs as tightly interlocking, sub/anhedral crystals mostly <0.25 mm in size, only rarely with noticeable flattening/elongation in the plane of foliation that is more or less defined by alignment of sericite flakes (mainly euhedral, up to 0.3 mm) and local concentrations of carbonate forming either blastic aggregates <0.5 mm (brownish, possibly ankerite, after former mafic crystals?) or wispy aggregates along the foliation or veinlets, <0.1 mm thick. Minor accessory pyrite as subhedral cubic crystals mostly <0.2 mm (loose aggregates to 1 mm) is locally associated with the carbonate along margins of veins. Rutile occurs in aggregates <0.2 mm of stubby euhedra mostly <30 um long. In the second, sericite forms strongly aligned sub/euhedral flakes mainly <0.3 mm but commonly optically continuous for up to 2 mm, and quartz occurs as scattered subhedra mainly <0.15 mm. Rutile forms pale golden brown euhedral prisms <30 um long. Relict quartz phenocrysts mostly in the first have rounded subhedral outlines up to 1.5 mm long, mostly recrystallized; with moderate to locally strong strain in phenocrysts and in matrix indicated by undulose extinction, sub-grain development, and local suturing of grain boundaries.

In the veins, which are poorly defined, irregular, and up to at least 3 mm thick, quartz forms ragged interlocking sub/anhedra mainly <0.75 mm with moderate to strong undulose extinction, sub-grain development, and common suturing of grain boundaries. Carbonate forms ragged subhedra to ~1 mm that are pale brownish (dolomite/ankerite?) that only partly contact pyrite, lesser chalcopyrite and local sphalerite. Pyrite forms irregular sub- to locally cubic euhedral crystals mostly <1 mm in diameter (locally fractured and with chalcopyrite either as inclusions <0.1 mm or along grain boundaries/fractures mostly <0.2 mm thick). Pyrite is mostly enclosed in quartz and not in contact with carbonate, but chalcopyrite, which also occurs as ragged subhedra to ~1 mm (with local pale yellow-brown, i.e. low Fe, sphalerite as sub/ euhedra to 0.5 mm), is more commonly in contact with or partly surrounded by/associated with carbonate.

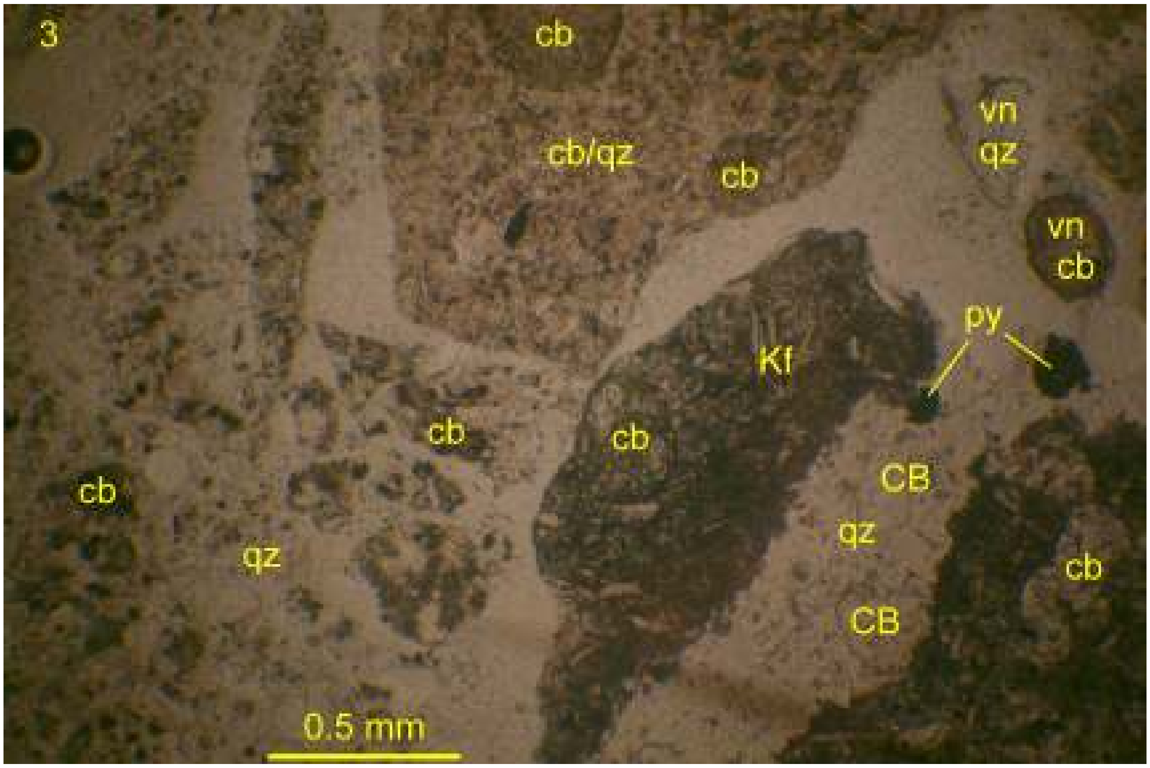
In summary, this sample consists of medium/fine-grained, variably textured, variably foliated quartz-lesser sericite-minor carbonate-accessory pyrite-rutile phyllite, or sericite schist, locally with quartz “eyes” or relict phenocrysts and rare blastic carbonate possibly after mafic phenocrysts, suggestive of derivation from felsic volcanic rock, cut by irregular, poorly defined veins of coarser-grained quartz-carbonate-pyrite-chalcopyrite-sphalerite, all locally in contact with Fe-carbonate.



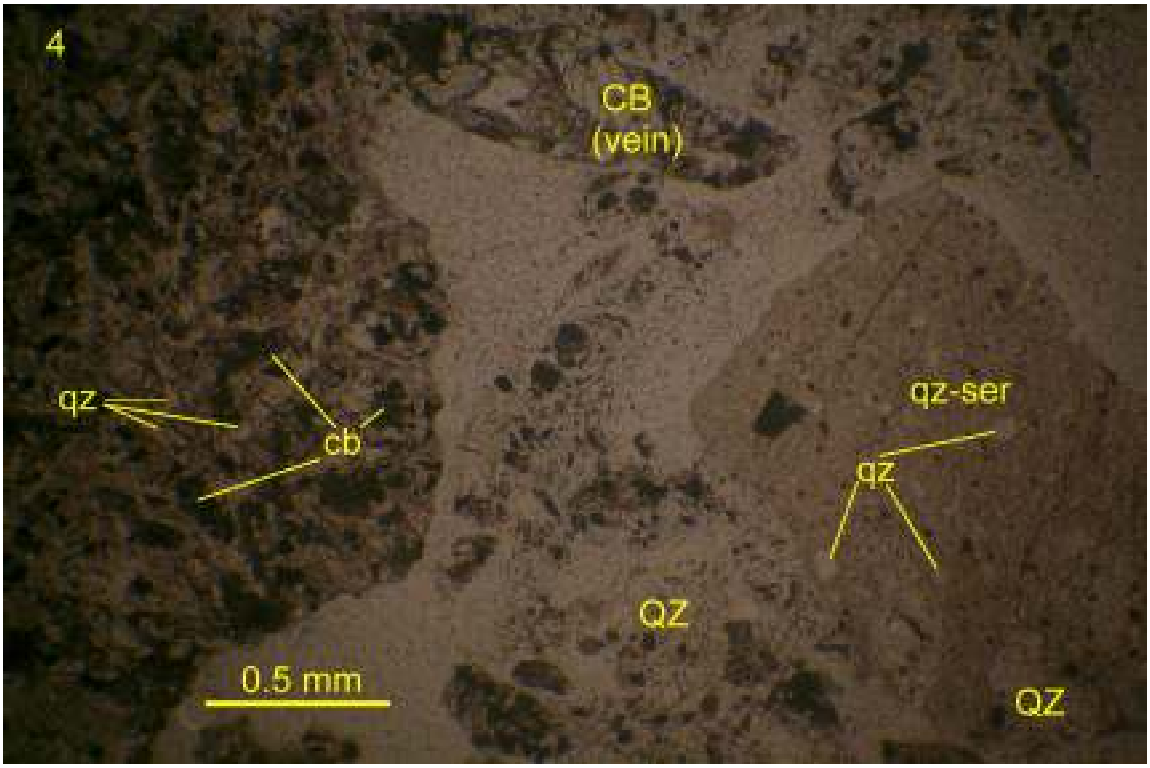
1(HC07-21): Chips of variably quartz (qz) or sericite (ser) rich schist, and coarser-grained, carbonate (CB)-quartz (QZ)-pyrite (PY) vein chips, with the sulfide in contact with or partly surrounded by the carbonate. Carbonate may be ferroan dolomite (ankerite). Transmitted light, crossed polars, field of view 3.0 mm wide.



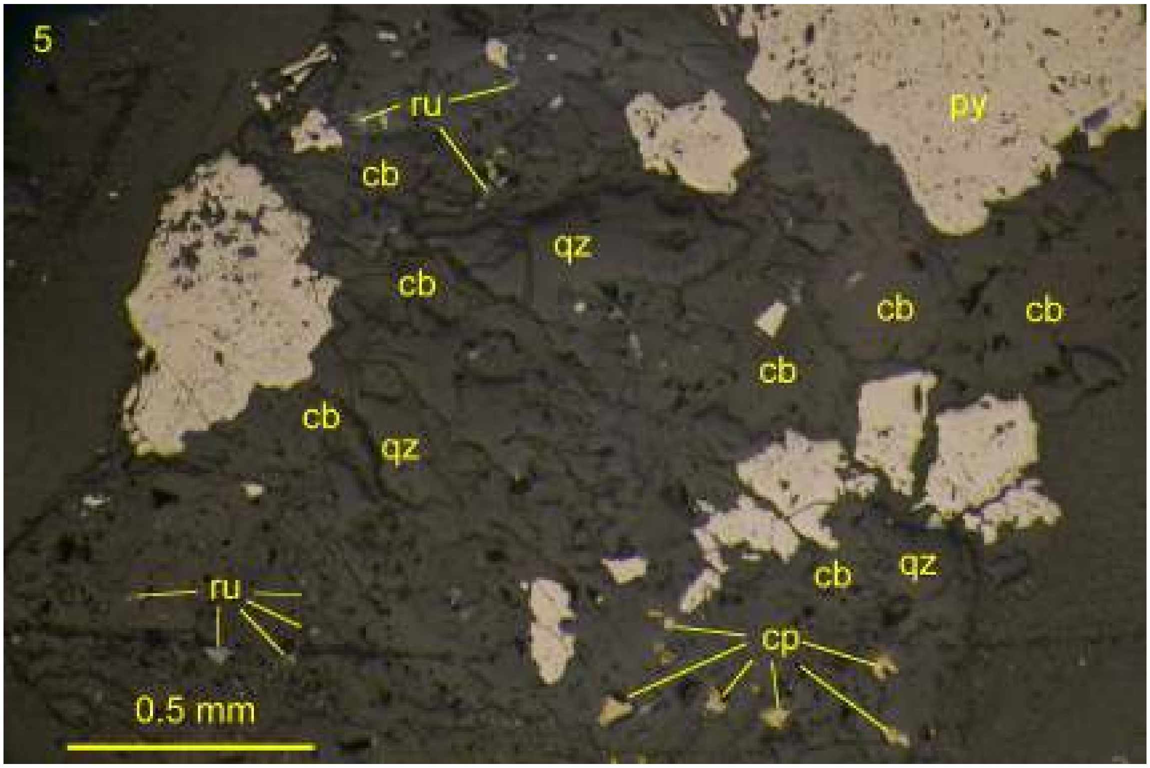
2: Variably quartz (qz)-rich or sericite (ser)-rich schist, the latter with minor chlorite (indistinguishable from sericite) and carbonate (cb), local quartz “eyes”), local veins of pyrrhotite (po) partly surrounded by brown-stained ferroan dolomite or ankerite (Fe-cb), intergrown with quartz (QZ). Transmitted plane light, field of view 3.0 mm wide.



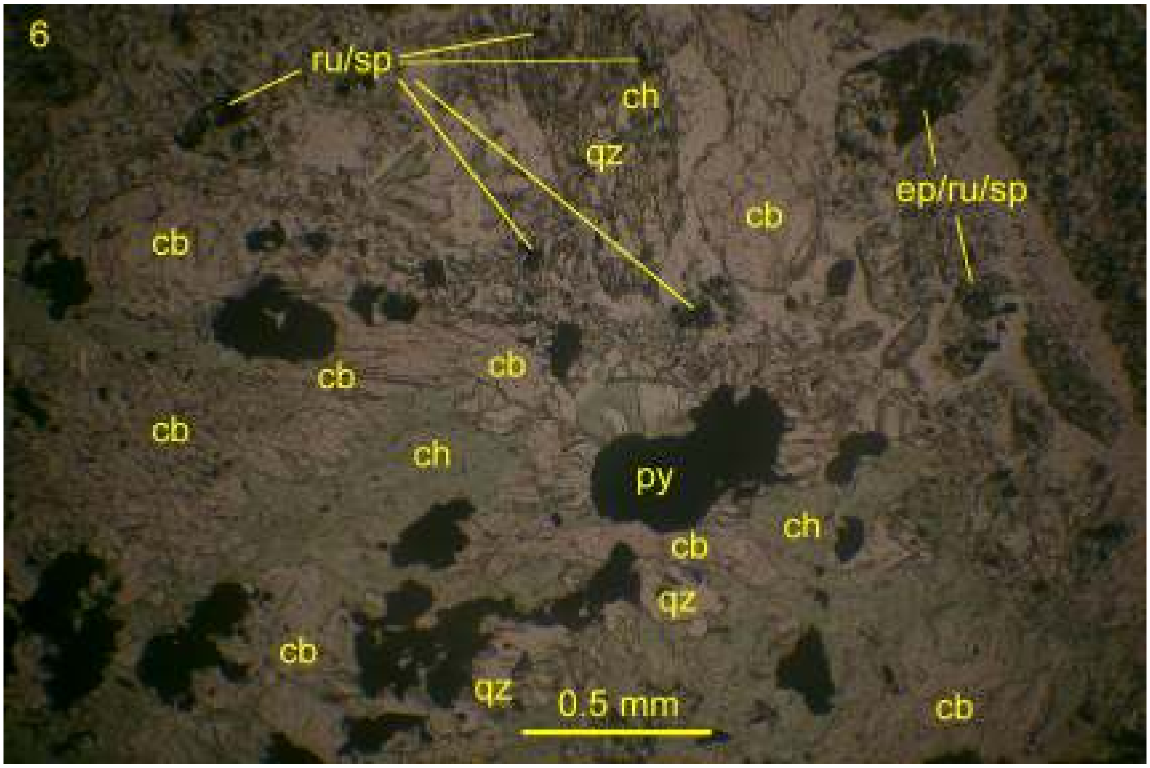
3: Chips of K-feldspar (lath-like, Kf) cut by variable zones of coarse blastic carbonate (CB) in fine-grained secondary quartz (qz) with minor pyrite (py), or chips with amygdular carbonate (cb) in fine-grained carbonate-quartz, or siliceous chips (qz) with minor carbonate, or vein quartz (QZ) and carbonate. Transmitted plane light, field of view 3.0 mm wide.



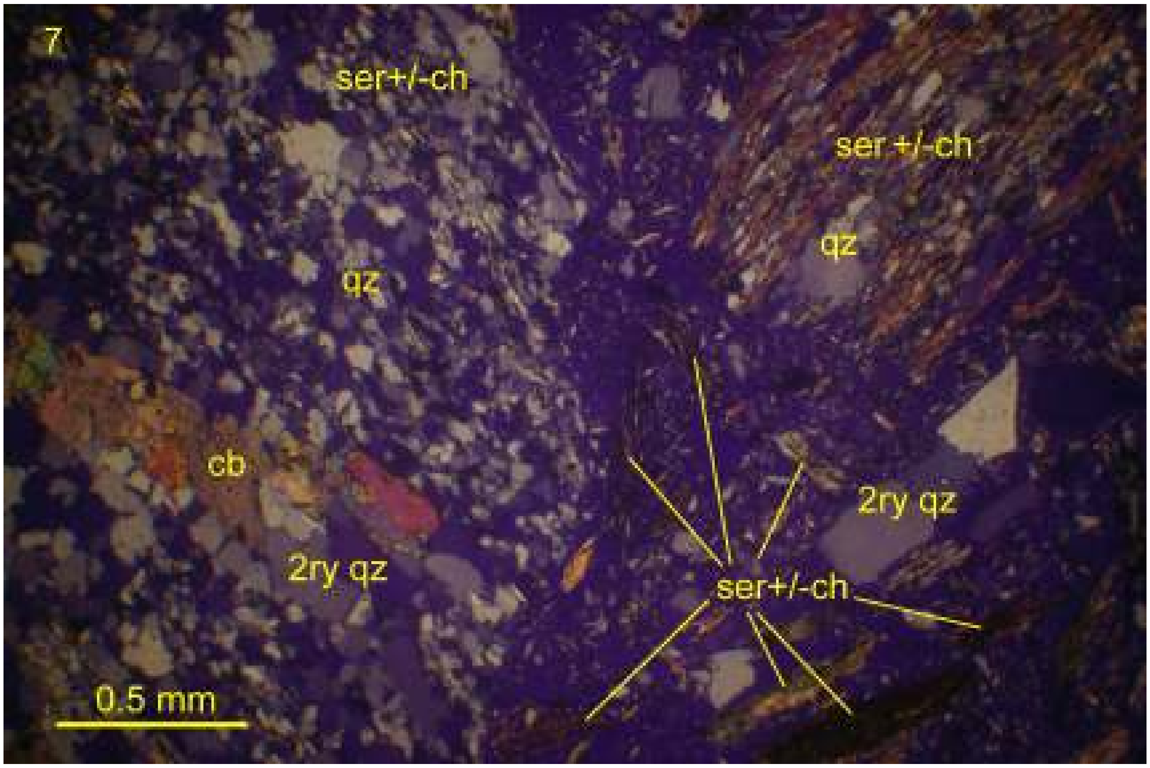
4: Strongly altered mafic/intermediate volcanic composed of quartz (clear) and carbonate (brownish, ankerite?) in matrix of Kspar and quartz, and felsic volcanic composed of rare relict quartz eyes (QZ) in weakly foliated quartz-sericite (qz-ser), both locally with accessory sulfides and rutile (opaques). Transmitted plane light, field of view 3.0 mm wide.



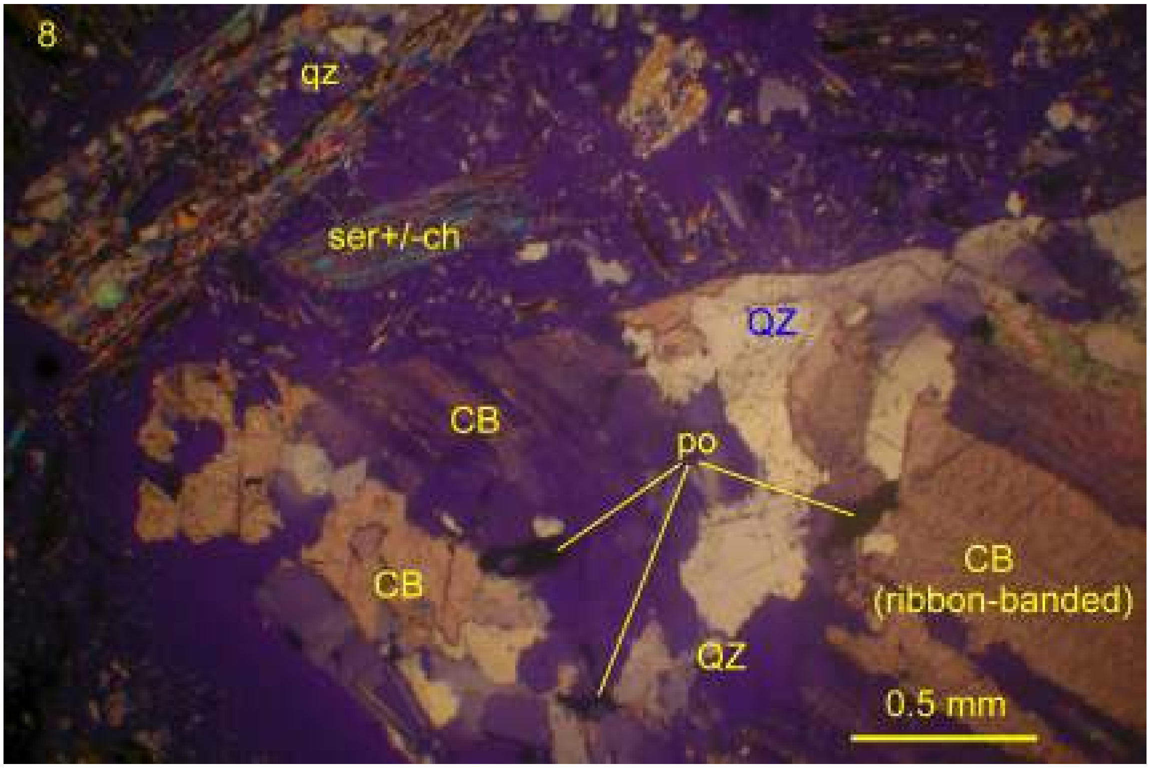
5: Vein-like chip composed of aggregates of pyrite (py, some with porous texture suggestive of being after former pyrrhotite?), minor chalcopyrite (cp), both partly in contact with carbonate (cb) in matrix of secondary quartz (qz) and minor rutile (ru). Reflected light, uncrossed polars, field of view 2.25 mm wide.



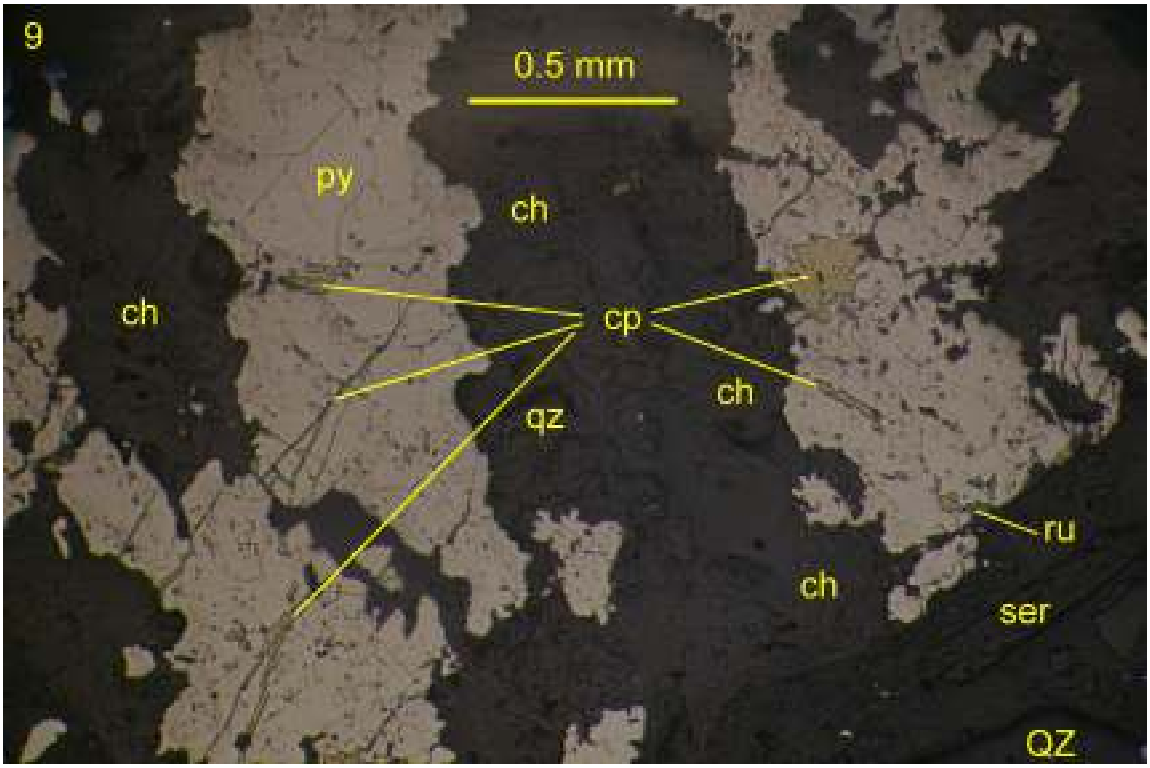
6: Relatively coarse-grained (gradational to vein?) chip of quartz (qz), chlorite (ch), carbonate (cb) and pyrite (py, opaque, commonly in contact with carbonate), or fine-grained quartz-chlorite-local epidote/rutile/sphene (ep/ru/sp) chips, likely derived by alteration/metamorphism of intermediate/mafic volcanic rock. Transmitted plane light, field of view 3.0 mm.



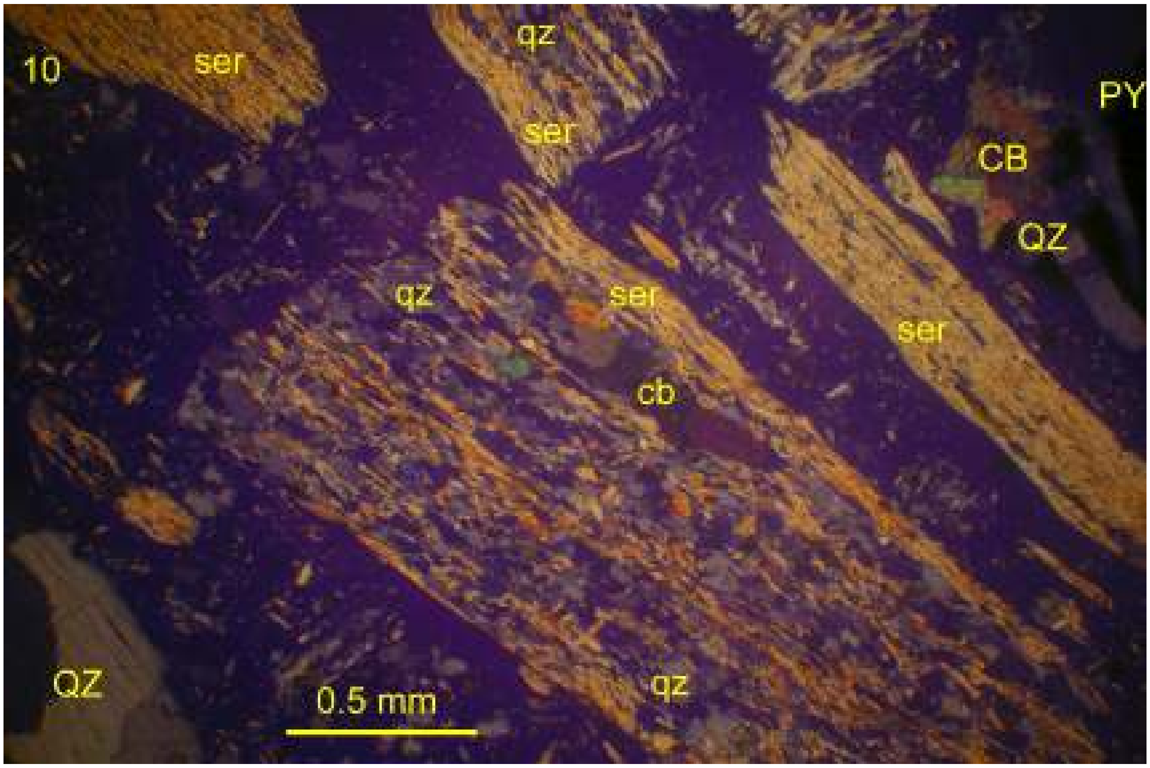
7: Variation from quartz (qz)-rich to sericite ± chlorite (ser±ch)-rich schist, the former with local minor carbonate (cb) possibly partly distributed along thin veins with secondary quartz (2ry qz). Transmitted light, crossed polars, field of view 3.0 mm wide.



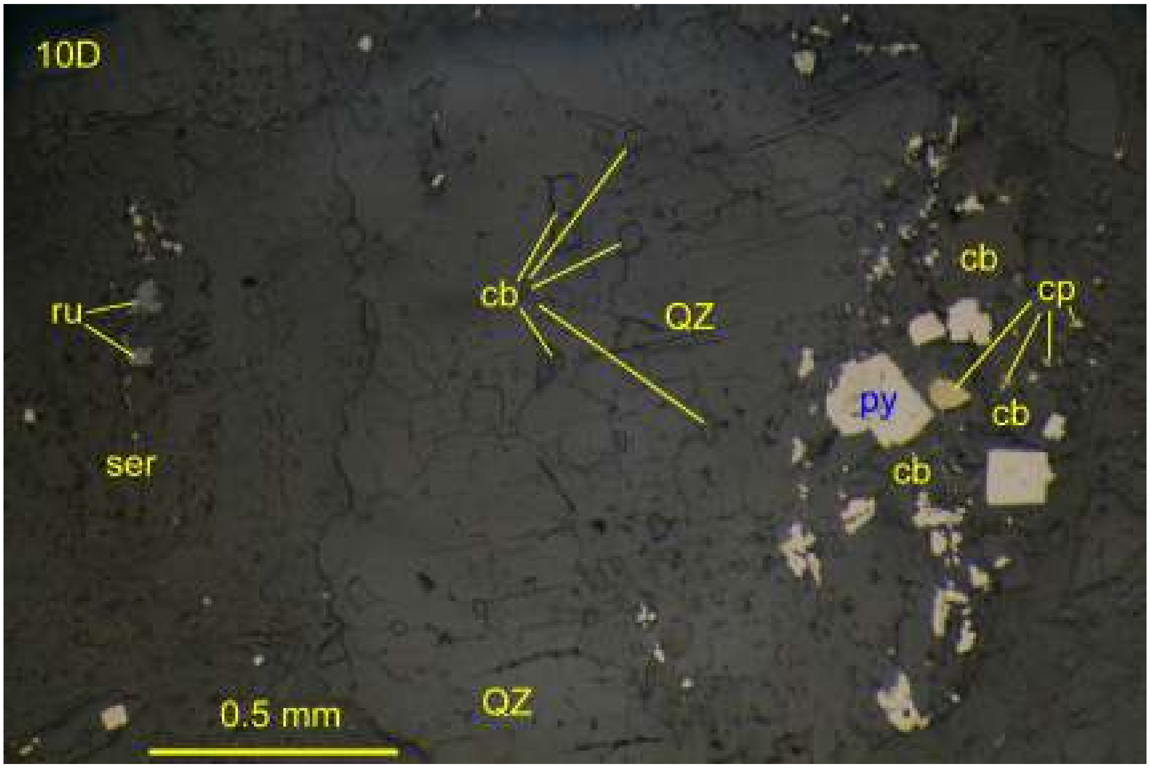
8: Chips of fine-grained, laminated, foliated quartz-sericite ±chlorite schist as in the previous sample, and coarse-grained, partly ribbon-banded, carbonate (CB) and quartz (QZ) veins (note minor pyrrhotite, po, largely in contact with carbonate). Transmitted light, crossed polars, field of view 3.0 mm wide.



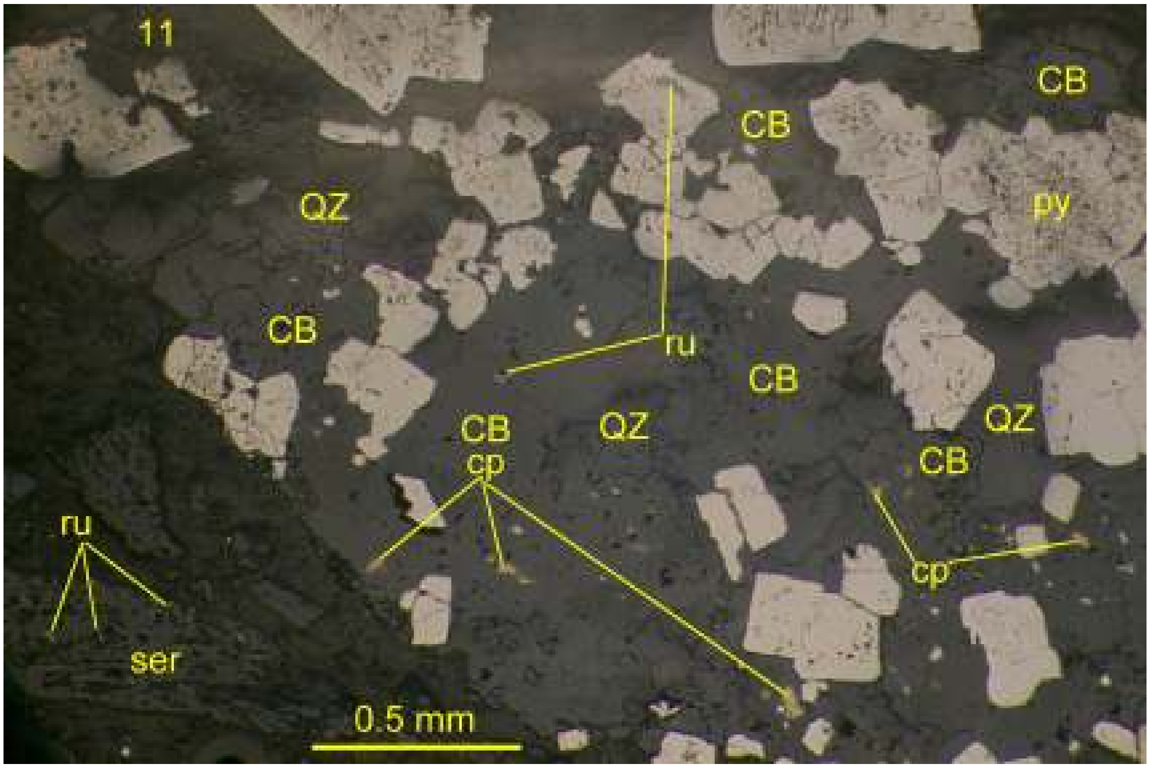
9: Aggregates of pyrite (py) locally with inclusions of, or veined by, chalcopyrite (cp), and rare inclusions of rutile (ru), in vein chip composed mainly of chlorite (ch) and minor quartz (qz); note also coarse vein quartz (QZ) and small chip of sericite-rich schist (ser). Reflected light, uncrossed polars, field of view 2.75 mm wide.



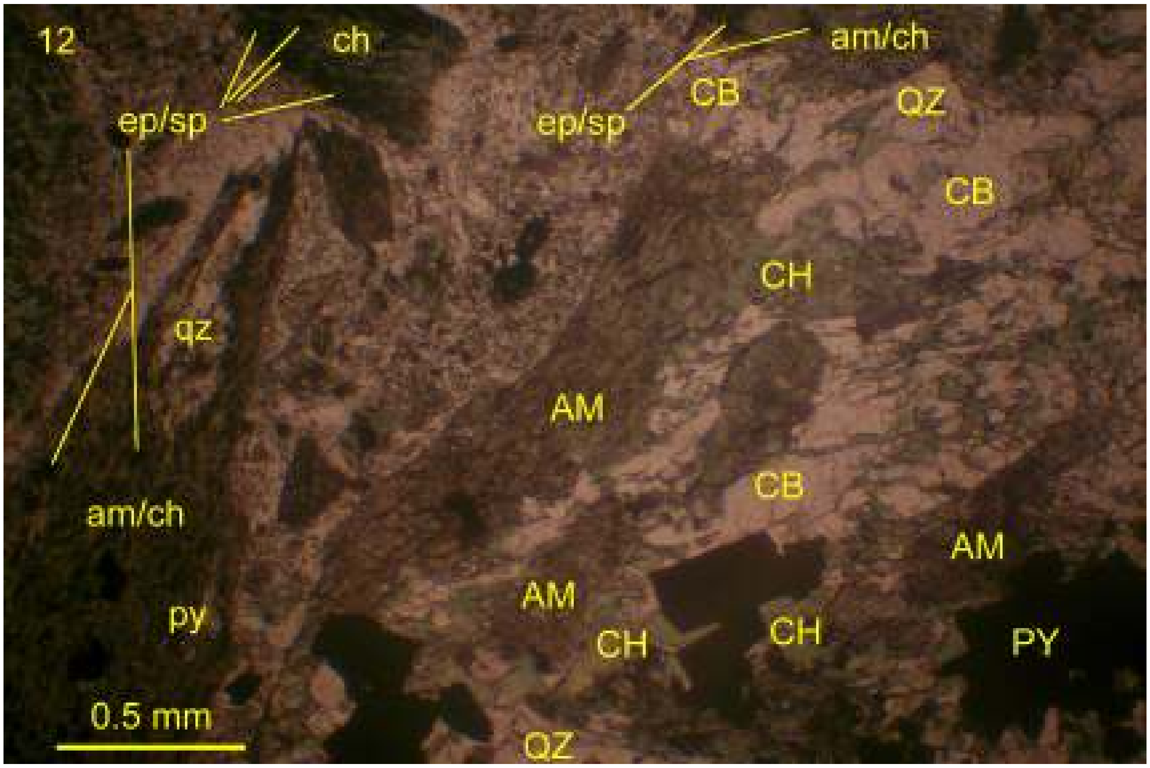
10: Variably sericite (ser) or quartz (qz) rich schist chips, the latter with local carbonate (cb), plus chips of coarse vein quartz (QZ) or quartz with pyrite (PY) and minor carbonate (CB) not touching most of the pyrite. Transmitted light, crossed polars, field of view 3.0 mm wide.



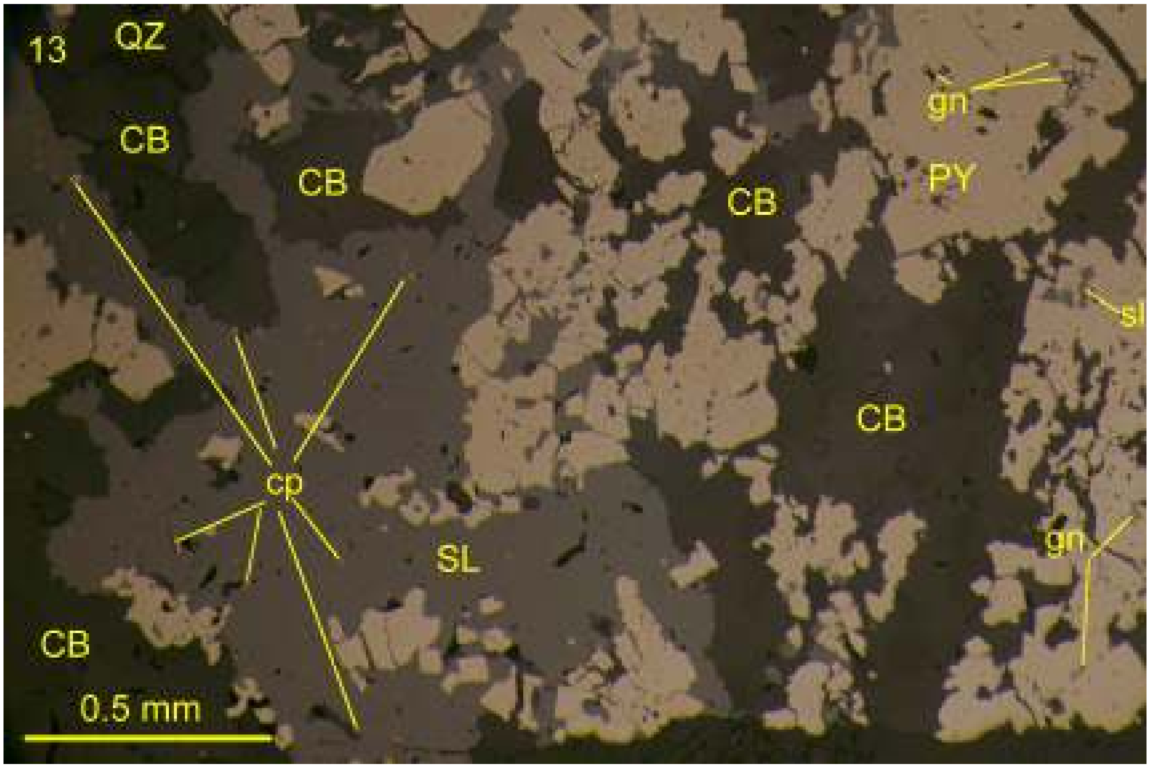
10D: Pyrite (py) and minor chalcopyrite (cp) associated with and in contact with carbonate (cb) along margin of coarse quartz (QZ)-minor carbonate vein. Note rutile (ru) in adjacent chip of sericite (ser) schist. Reflected light, uncrossed polars, field of view 2.25 mm wide.



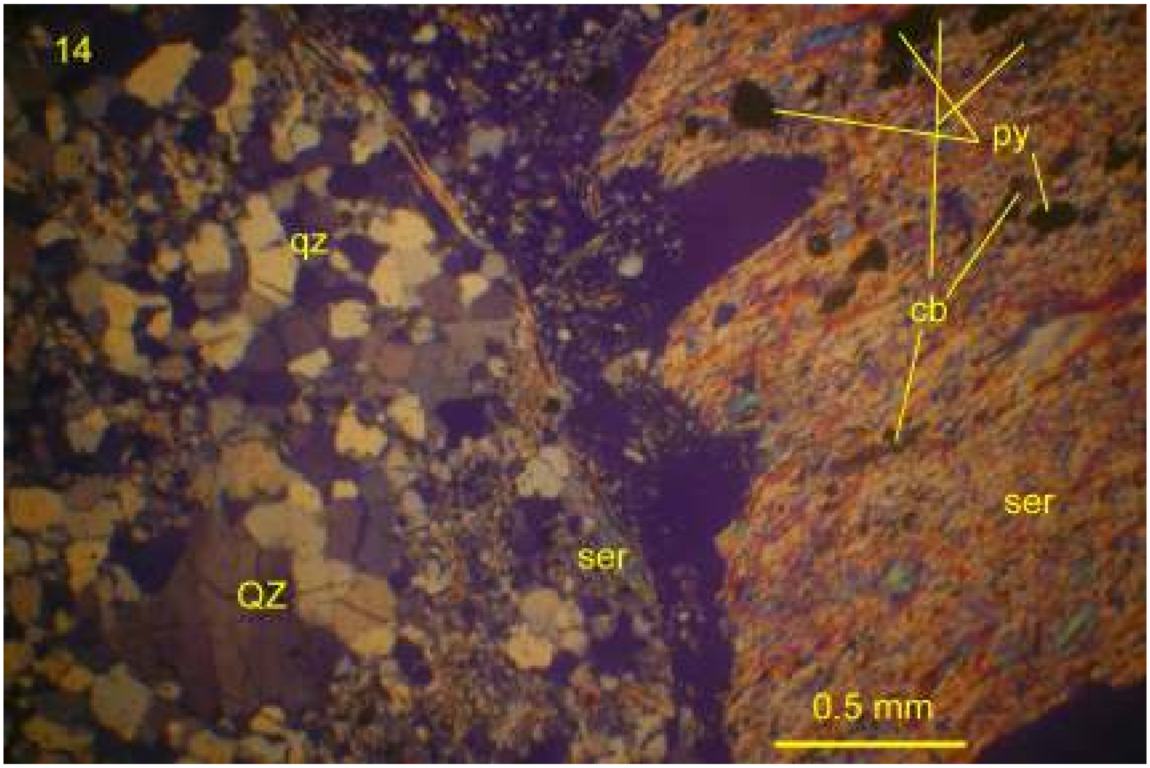
11: Pyrite (py) and minor chalcopyrite (cp) generally enclosed in carbonate (CB) and minor quartz (QZ) vein. Note rutile (ru) included in both pyrite and carbonate, as well as in adjacent chip of sericite (ser) schist. Reflected light, uncrossed polars, field of view 2.75 mm wide.



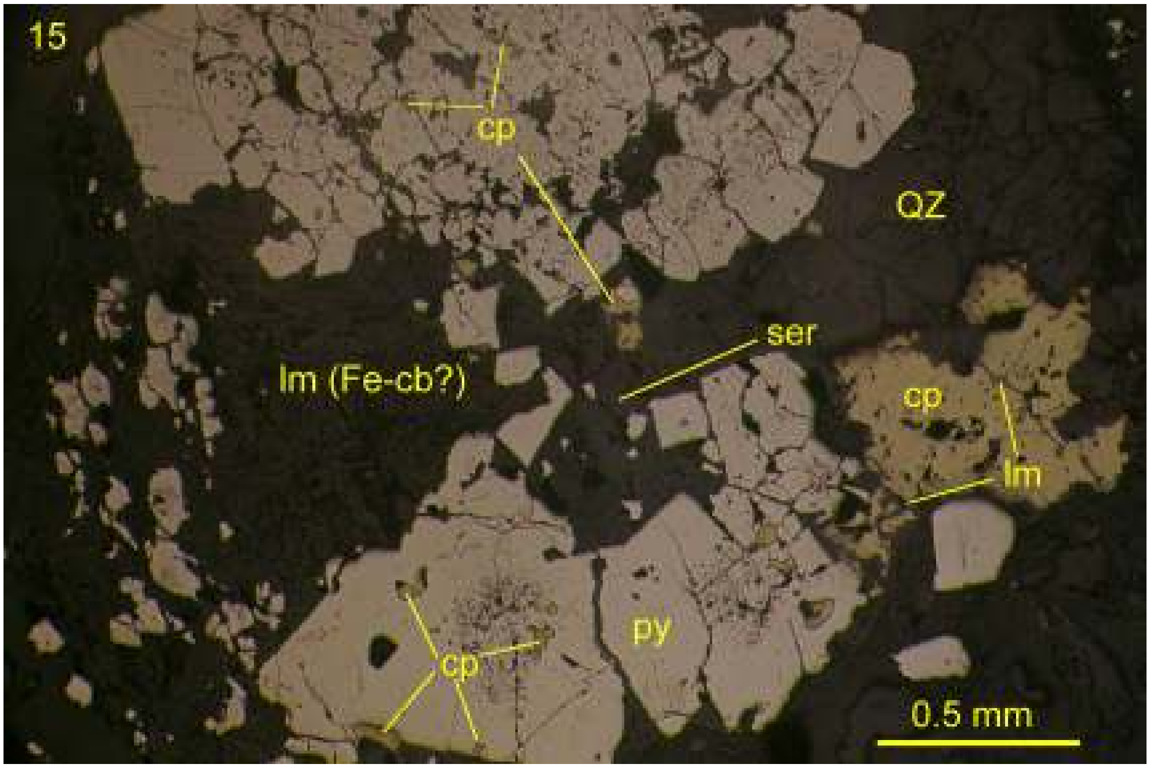
12: Chips of chlorite (ch) or amphibole (am) rich schist containing minor lenses of quartz (qz) and accessory epidote-sphene (dark clots), with veins of coarse quartz (QZ)-carbonate (CB)-amphibole (AM, partly altered to chlorite (CH) associated with pyrite, PY). Transmitted plane light, field of view 3.0 mm wide.



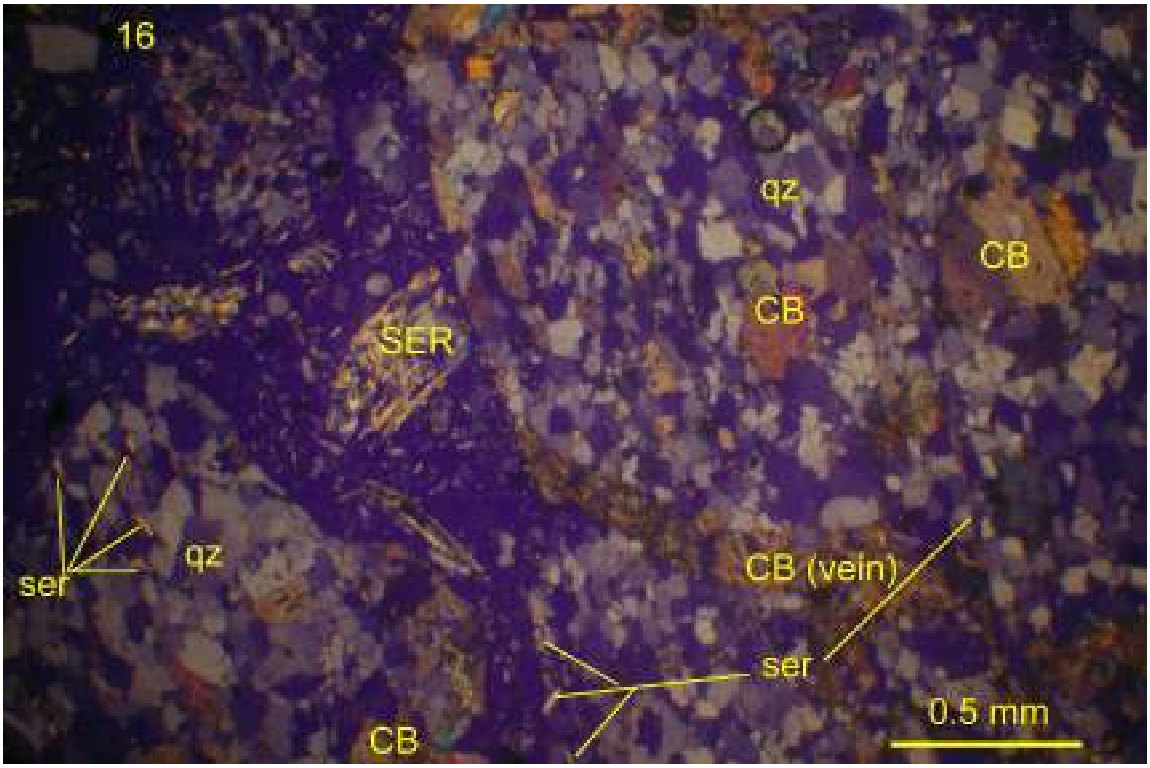
13: Vein with abundant pyrite (PY, containing minor inclusions of galena, gn, or sphalerite, sl) associated with sphalerite (SL, containing minor inclusions of chalcopyrite, cp), mostly surrounded/enclosed in carbonate (CB), local quartz (QZ). Reflected light, uncrossed polars, field of view 2.25 mm wide.



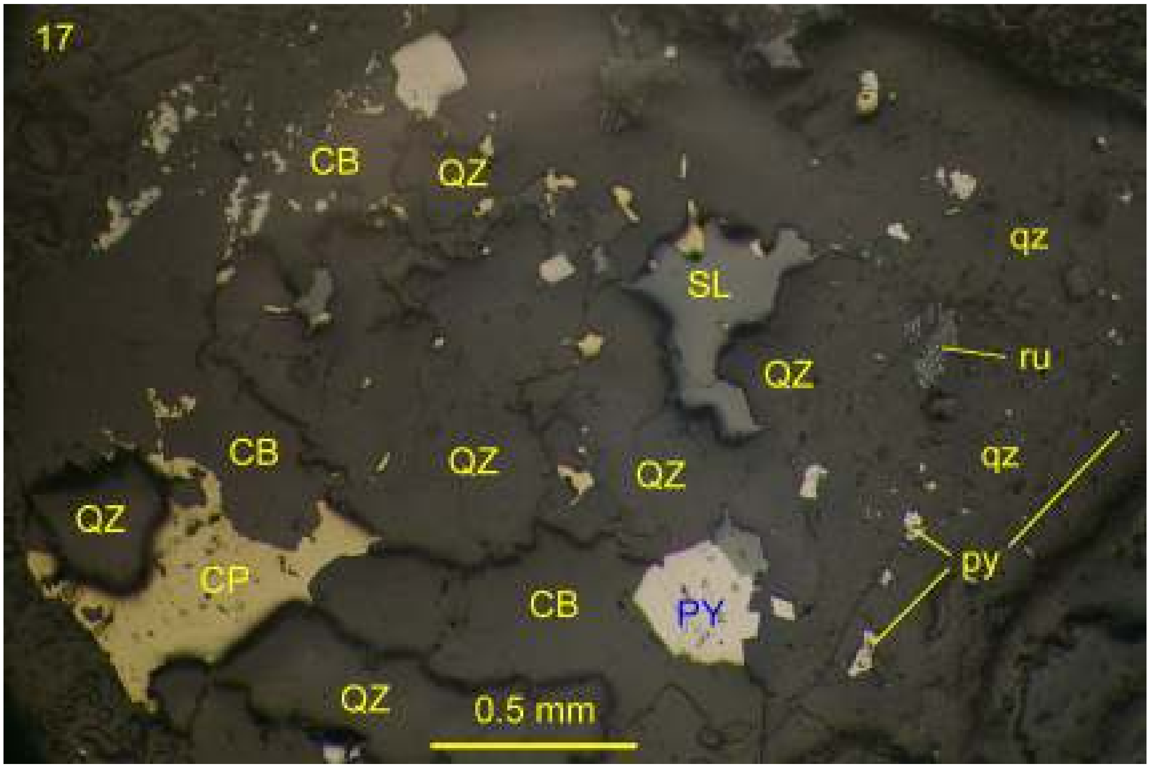
14: Typical end-members of sericite-rich (ser), minor carbonate (cb)-pyrite (py) schist and quartz-rich (qz), minor sericite semi-schist with relict quartz eyes or phenocrysts (QZ)., Transmitted light, crossed polars, field of view 3.0 mm wide.



15: Fractured pyrite (py) and lesser chalcopyrite (cp) both partly oxidized at margins to hematitic limonite (lm) that is in part mixed with or after Fe-carbonate (cb), in irregular vein composed mainly of relatively coarse-grained, secondary quartz (QZ). Reflected light, uncrossed polars, field of view 2.75 mm wide.



16: Weakly foliated quartz-rich chips with somewhat elongated/flattened quartz (qz), minor sericite (ser) and blastic or vein-like carbonate (CB), plus rare or lesser, more strongly foliated, sericite-rich chips (SER). Transmitted light, crossed polars, field of view 3.0 mm wide.



17: Vein chip composed of relatively coarse-grained quartz (QZ), carbonate (CB), the latter locally in contact with sulfides including pyrite (PY), chalcopyrite (CP) and sphalerite (SL); note minor pyrite (py) and rutile (ru) in finer-grained adjacent wallrock. Reflected light, uncrossed polars, field of view 2.75 mm wide.

A2: XRD Test Results

#1		#2		#3		#4		#5		#6		#7		#8		#9		#10		#11	
Comp # 648		**Comp # 650**		**Comp # 654**		**Comp # 330**		**Comp # 336**		**Comp # 337**		**110250-4**		**110250-1**		**Comp # 659**		**Comp # 660**		**Comp # 663**	
Quartz	63.5	Quartz	46.9	Quartz	35.9	Quartz	38.8	Quartz	34.6	Quartz	26.5	Quartz	21.2	Quartz	26.7	Quartz	68.0	Quartz	34.3	Quartz	46.3
Muscovite	25.4	Muscovite	36.0	Clinochlore	3.7	Calcite	1.2	Calcite	0.5	Kaolinite	9.3	Pyrite	5.3	Pyrite	2.6	Muscovite	16.6	Muscovite	48.4	Muscovite	39.1
Kaolinite	1.1	Pyrite	1.9	Kaolinite	8.1	Muscovite-Illite	4.7	Muscovite-Illite	7.3	Calcite	1.0	Calcite	9.7	Calcite	19.5	Pyrite	0.4	Clinochlore	8.6	Clinochlore	10.0
Pyrite	2.1	Dolomite	2.5	Plagioclase	24.9	Ankerite	2.1	Ankerite	3.4	Muscovite-Illite	9.4	Plagioclase	16.3	Plagioclase	6.3	Dolomite	9.1	Rutile	0.8	Rutile	0.9
Dolomite	3.3	Clinochlore	8.9	Calcite	3.3	Dolomite	6.2	Dolomite	6.8	Ankerite	4.4	Rutile	3.9	Rutile	3.3	Clinochlore	3.8	Siderite	5.2	Siderite	1.2
Siderite	2.7	Rutile	0.5	Muscovite-Illite	4.9	Pyrite	1.1	Pyrite	1.6	Dolomite	6.3	Ankerite	14.2	Ankerite	3.9	Rutile	0.2	Plagioclase	2.5	Dolomite	1.4
K-feldspar	1.9	Pyrrhotite	0.9	Paragonite	3.1	Siderite	8.8	Siderite	10.6	Hematite	1.3	Muscovite	1.9	Clinochlore	37.6	Pyrrhotite	0.6	Dolomite	0.2	Pyrrhotite	1.0
		K-feldspar	1.5	Ankerite	3.7	Anatase	1.0	Anatase	1.0	Siderite	10.1	Paragonite	2.1			K-feldspar	1.4				
		Hydroxylapatite	0.8	Dolomite	12.2	K-feldspar	12.8	K-feldspar	13.5	Anatase	1.3	Siderite	0.3								
				Hematite	0.2	Siderite	3.1	Siderite	2.2	Siderite	4.7	Clinochlore	25.2								
						Plagioclase	3.7	Plagioclase	3.2	Plagioclase	9.7										
						Kaolinite	16.4	Kaolinite	15.2	Clinochlore	2.3										
										K-feldspar	12.2										
										Titanite	1.8										
Total	**100.0**		**100.0**		**100.0**		**100.0**		**100.0**		**100.0**		**100.0**		**100.0**		**100.0**		**100.0**		**100.0**

#12		#13		#14		#15		#16		#17		#18		#19		#20		#21		#22	
Comp # 64		**Comp # 76**		**Comp # 194**		**Comp # 39**		**Comp # 169**		**Comp # 286**		**Comp # 202**		**Comp # 339**		**Comp # 489**		**Comp # 187**		**Comp # 331**	
Quartz	39.5	Quartz	47.3	Quartz	16.3	Quartz	42.0	Quartz	17.7	Quartz	11.7	Quartz	42.5	Quartz	62.7	Quartz	61.2	Quartz	74.2	Quartz	79.6
Muscovite	45.1	Muscovite	39.3	Clinochlore	25.7	Muscovite	29.4	Clinochlore	18.5	Calcite	9.6	Calcite	8.2	Muscovite	18.4	Pyrite	4.0	Muscovite	12.3	Muscovite	12.7
Clinochlore	6.2	Clinochlore	4.1	Calcite	25.4	Clinochlore	11.9	Calcite	2.6	Plagioclase	29.0	Plagioclase	14.6	Rutile	0.5	Clinochlore	4.7	Siderite	1.4	Pyrite	2.7
Rutile	0.8	Siderite	4.5	Plagioclase	16.9	Siderite	3.4	Plagioclase	17.6	Hydroxylapatite	3.3	Pyrite	1.2	Siderite	8.9	Muscovite	23.5	Pyrite	8.0	K-feldspar	1.9
Siderite	1.7	Pyrite	3.5	Biotite	10.4	Pyrite	7.2	Rutile	3.4	Pyrite	3.5	Clinochlore	22.6	Pyrite	5.1	Dolomite	2.0	Chalcopyrite	1.2	Dolomite	2.1
Dolomite	2.8	Rutile	0.9	Rutile	2.5	Rutile	1.2	Hydroxylapatite	3.2	Clinochlore	30.5	Rutile	1.5	Kaolinite	2.4	Siderite	1.0	Jarosite	1.2	Chalcopyrite	0.4
Pyrite	3.7	Calcite	0.1	Hydroxylapatite	0.8	Calcite	0.2	Ankerite	15.6	Titanite	10.5	Muscovite	3.7	Clinochlore	1.4	Talc	1.0	K-feldspar	1.6	Kaolinite	0.7
Grossular	0.3	Dolomite	0.3	Ilmenite	0.9	Dolomite	2.9	Pyrite	6.4	Biotite	1.1	Dolomite	5.6	Calcite	0.4	Pyrrhotite	1.0				
				Grossular	0.8	Talc	1.5	Muscovite	12.4	Ilmenite	0.7			Dolomite	0.2	Magnetite	0.5				
				Dolomite	0.4	Magnetite	0.4	Kaolinite	1.8							K-feldspar	1.2				
								Siderite	0.7												
Total	**100.0**		**100.0**		**100.0**		**100.0**		**100.0**		**100.0**		**100.0**		**100.0**		**100.0**		**100.0**		**100.0**

A3: XRD Test Results for Humidity Cell Material

QUANTITATIVE PHASE ANALYSIS OF 18 POWDER SAMPLES USING THE RIETVELD METHOD AND X-RAY POWDER DIFFRACTION DATA.

Project: SRK-Yellowhead Mining Inc - Harper Creek (Waste Rock Humidity Cell Program)

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March 1, 2012

EXPERIMENTAL METHOD

The eighteen samples of **Project SRK Yellowhead – Harper Creek (Waste Rock Humidity Cell Program)** were reduced to the optimum grain-size range for quantitative X-ray analysis (<10 μm) by grinding under ethanol in a vibratory McCrone Micronising Mill for 7 minutes. Step-scan X-ray powder-diffraction data were collected over a range 3-80°2θ with CoKa radiation on a Bruker D8 Focus Bragg-Brentano diffractometer equipped with an Fe monochromator foil, 0.6 mm (0.3°) divergence slit, incident- and diffracted-beam Soller slits and a LynxEye detector. The long fine-focus Co X-ray tube was operated at 35 kV and 40 mA, using a take-off angle of 6°.

RESULTS

The X-ray diffractograms were analyzed using the International Centre for Diffraction Database PDF-4 and Search-Match software by Siemens (Bruker). X-ray powder-diffraction data of the samples were refined with Rietveld program Topas 4.2 (Bruker AXS). The results of quantitative phase analysis by Rietveld refinements are given in Table 1 (separate file, ***Maxxam Analytics Results Mar 1 2012 - Proj SRK Yellowhead - Harper Creek.xls***). These amounts represent the relative amounts of crystalline phases normalized to 100%. The Rietveld refinement plots are shown in Figures 1 – 18. Ideal formulae of the phases present are given in Table 2.

Table 2.

Mineral	**Ideal Formula**
Actinolite	$Ca_2(Mg,Fe)_5Si_8O_{22}(OH)_2$
Anatase	TiO_2
Ankerite – Dolomite	$Ca(Fe^{2+},Mg,Mn)(CO_3)_2$ – $CaMg(CO_3)_2$
Biotite	$K(Mg,Fe)_3(AlSi_3O_{10})(OH)_2$
Birnessite ?	$Na_4Mn_{14}O_{27}$ $9H_2O$
Calcite	$CaCO_3$
Chalcopyrite	$CuFeS_2$
Clinochlore	$(Mg,Fe^{2+})_5Al(Si_3Al)O_{10}(OH)_8$
Clinozoisite	$Ca_2Al_3(SiO_4)_3(OH)$
Galena	PbS
Hydroxylapatite	$Ca_5(PO_4)_3(OH)$
Kaolinite	$Al_2Si_2O_5(OH)_4$
K-Feldspar	$KAlSi_3O_8$
Magnetite	$Fe^{2+}Fe^{3+}{}_2O_4$
Muscovite	$KAl_2(AlSi_3O_{10})(OH)_2$
Paragonite	$NaAl_2AlSi_3O_{10}(OH)_2$
Plagioclase	$NaAlSi_3O_8$ – $CaAl_2Si_2O_8$
Powellite?	$CaMoO_4$
Pyrite	FeS_2
Pyrrhotite	$Fe_{1-x}S$
Quartz	SiO_2
Rutile	TiO_2
Siderite	$Fe^{2+}CO_3$
Talc	$Mg_3Si_4O_{10}(OH)_2$
Titanite	$CaTiSiO_5$

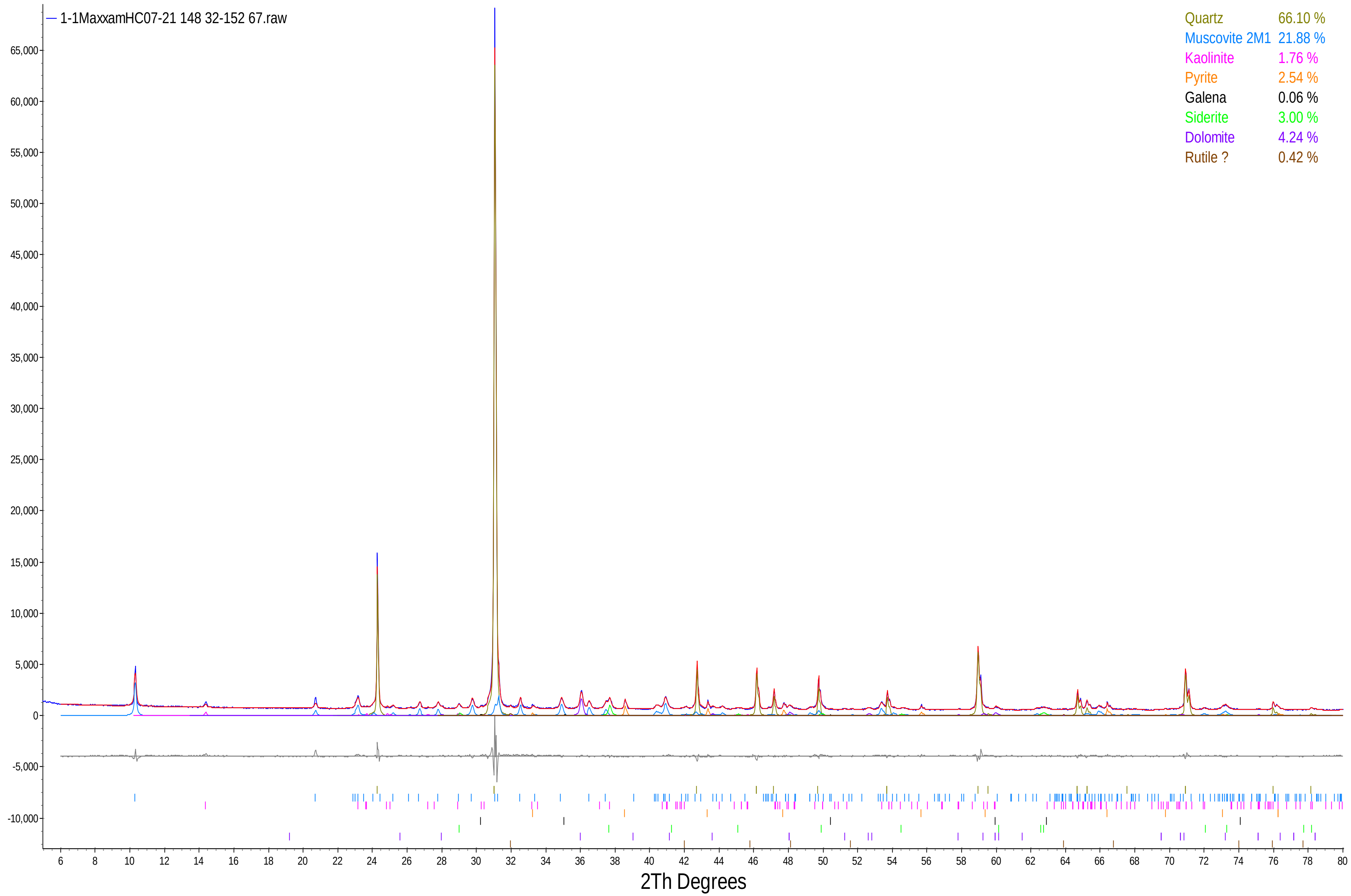


Figure 1. Rietveld refinement plot of sample **“HC07-21 From 148.32 to 152.67”** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

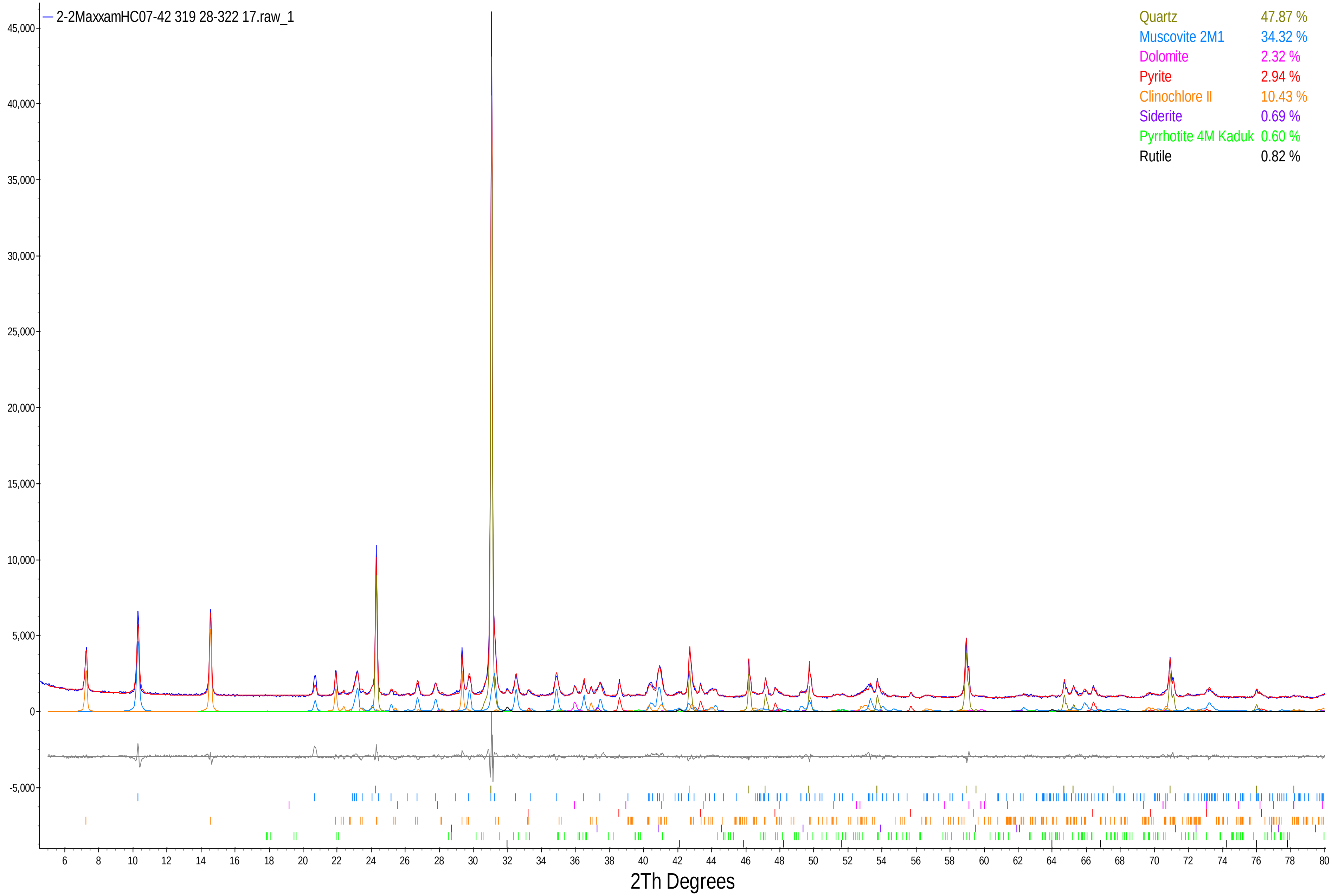


Figure 2. Rietveld refinement plot of sample **"HC07-42 From 319.28 to 322.17"** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

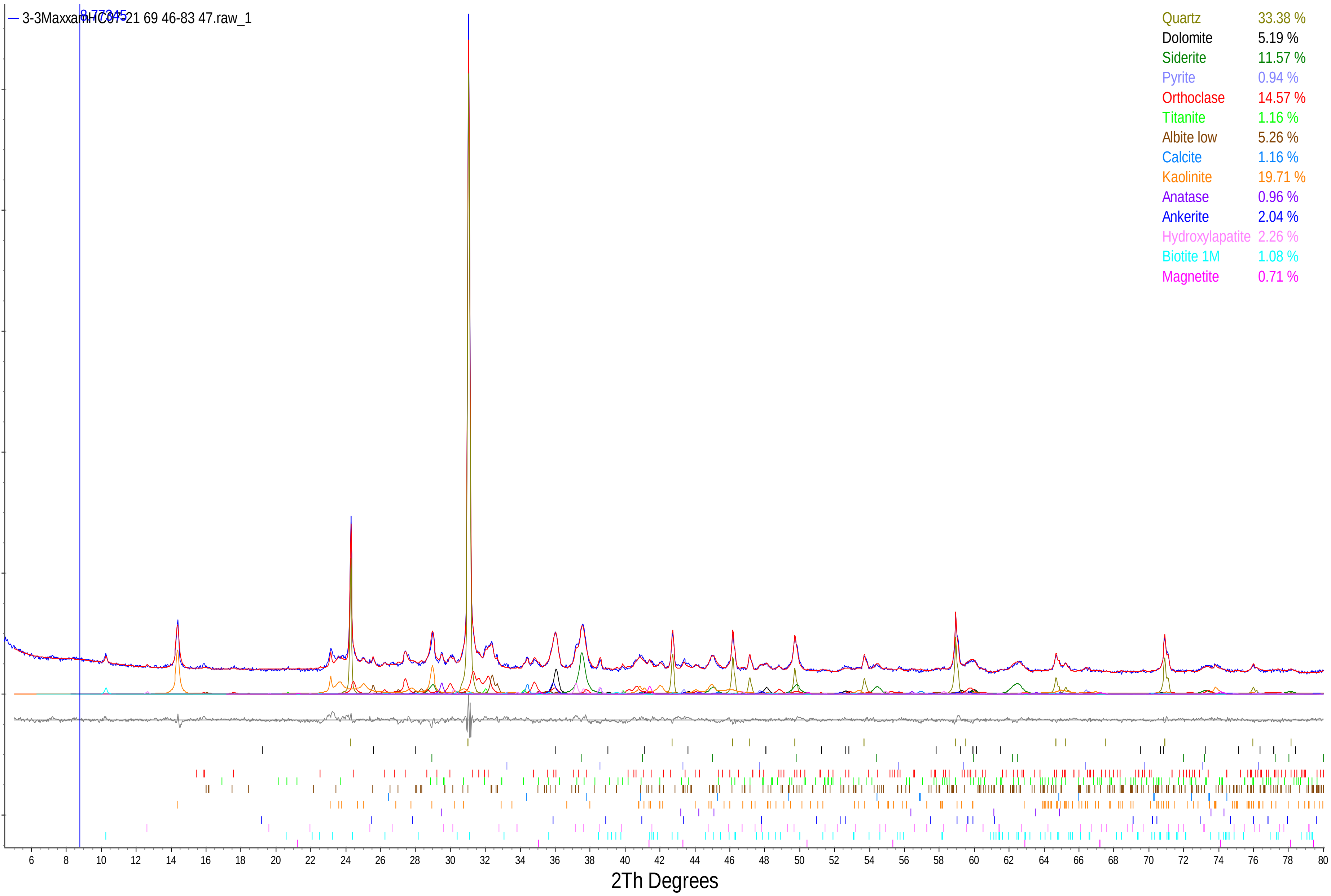


Figure 3. Rietveld refinement plot of sample **"HC07-21 From 69.46 to 83.47"** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

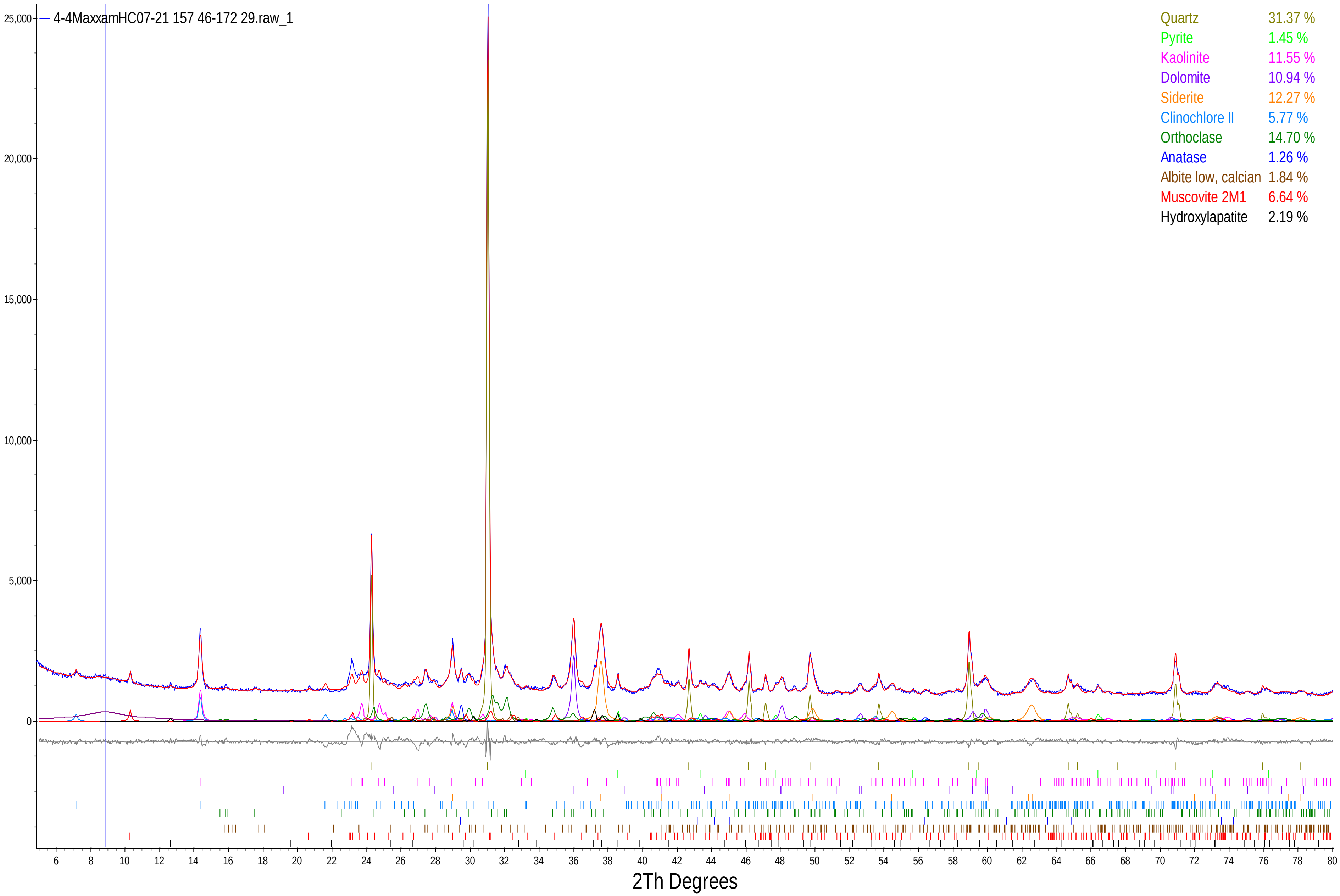


Figure 4. Rietveld refinement plot of sample **“HC07-21 From 157.46 to 172.29”** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

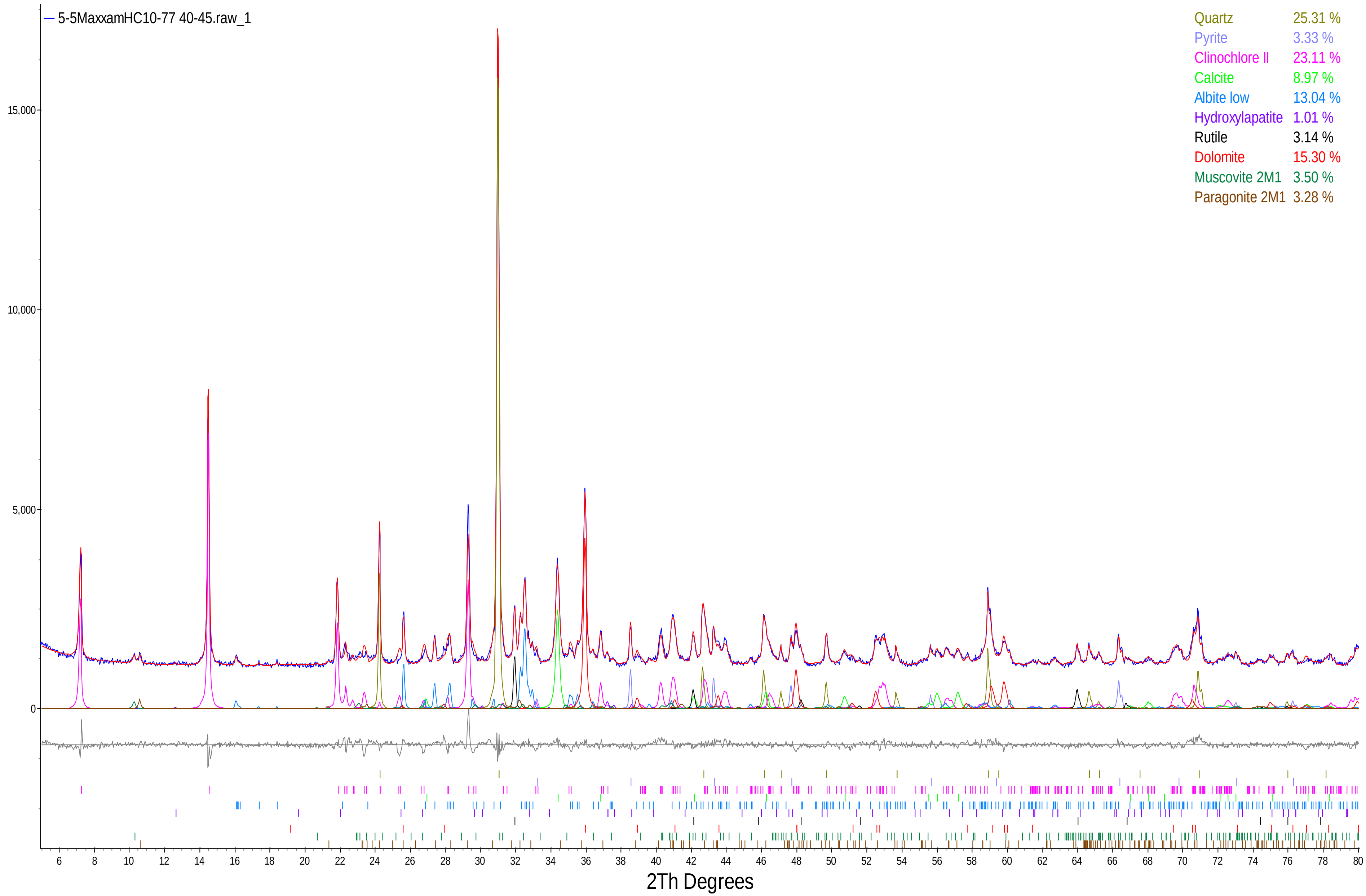


Figure 5. Rietveld refinement plot of sample **"HC10-77 From 40 to 45"** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

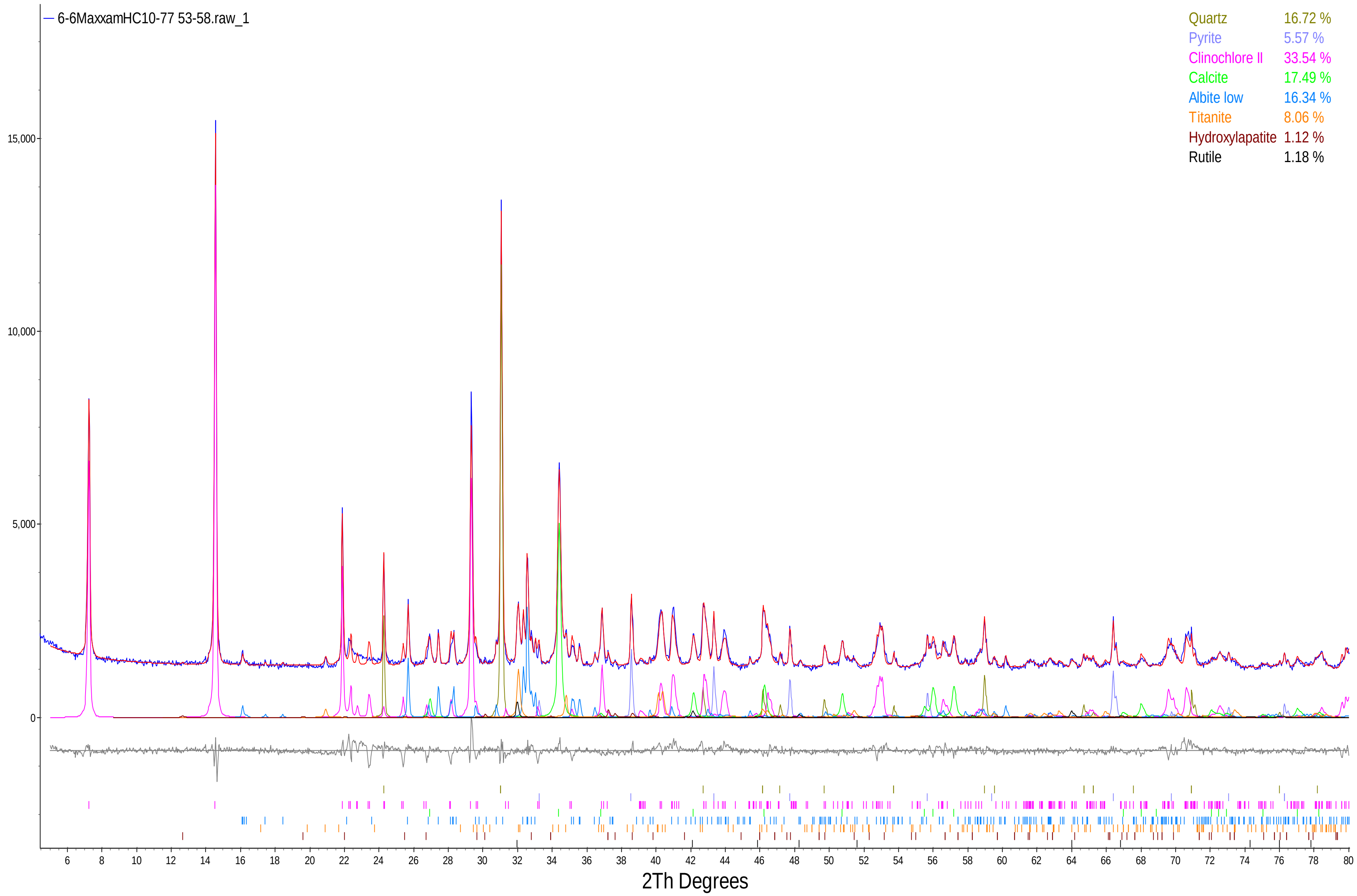


Figure 6. Rietveld refinement plot of sample **"HC10-77 From 53 to 58"** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

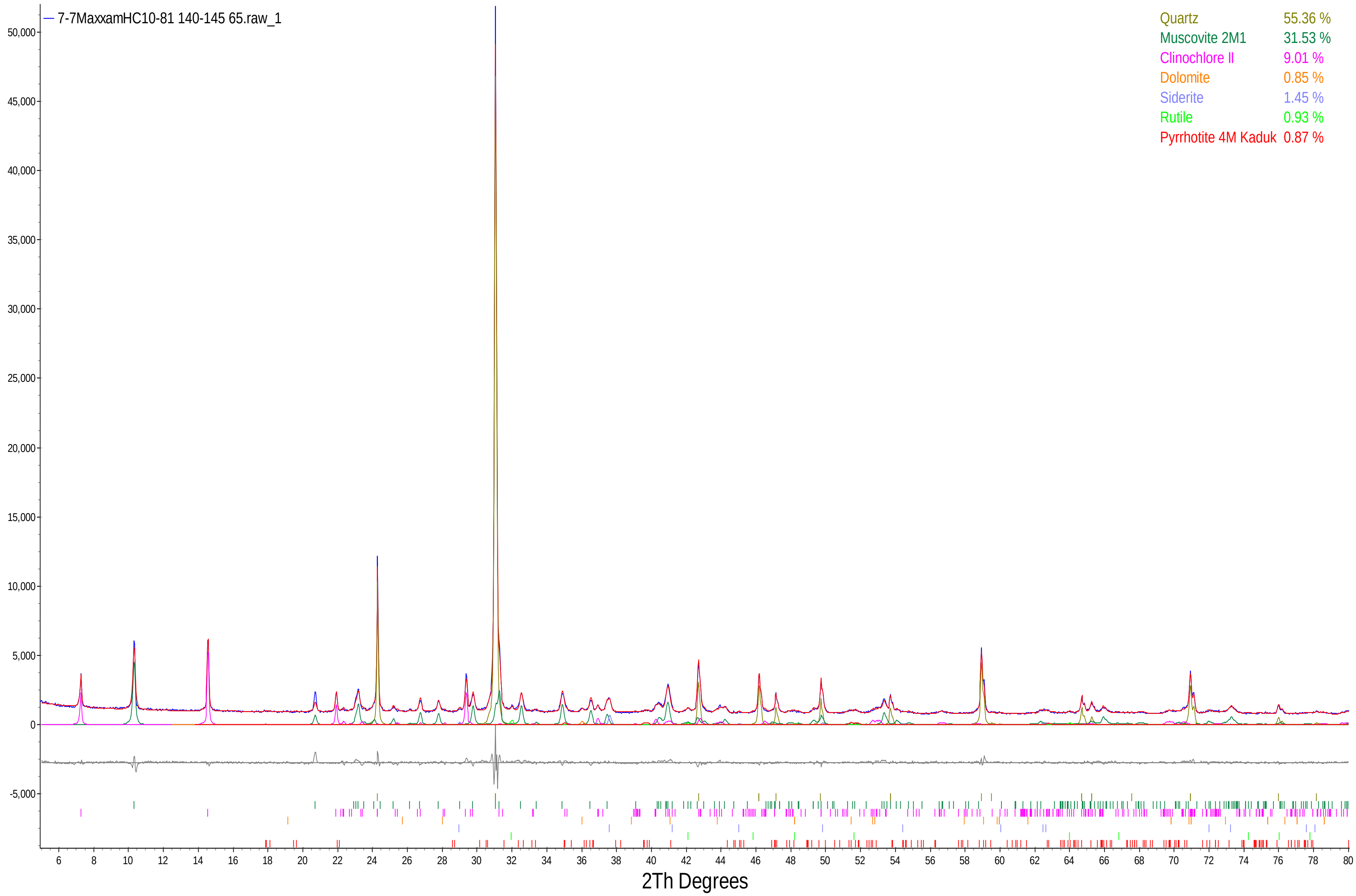


Figure 7. Rietveld refinement plot of sample **"HC10-81 From 140.00 to 145.65"** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

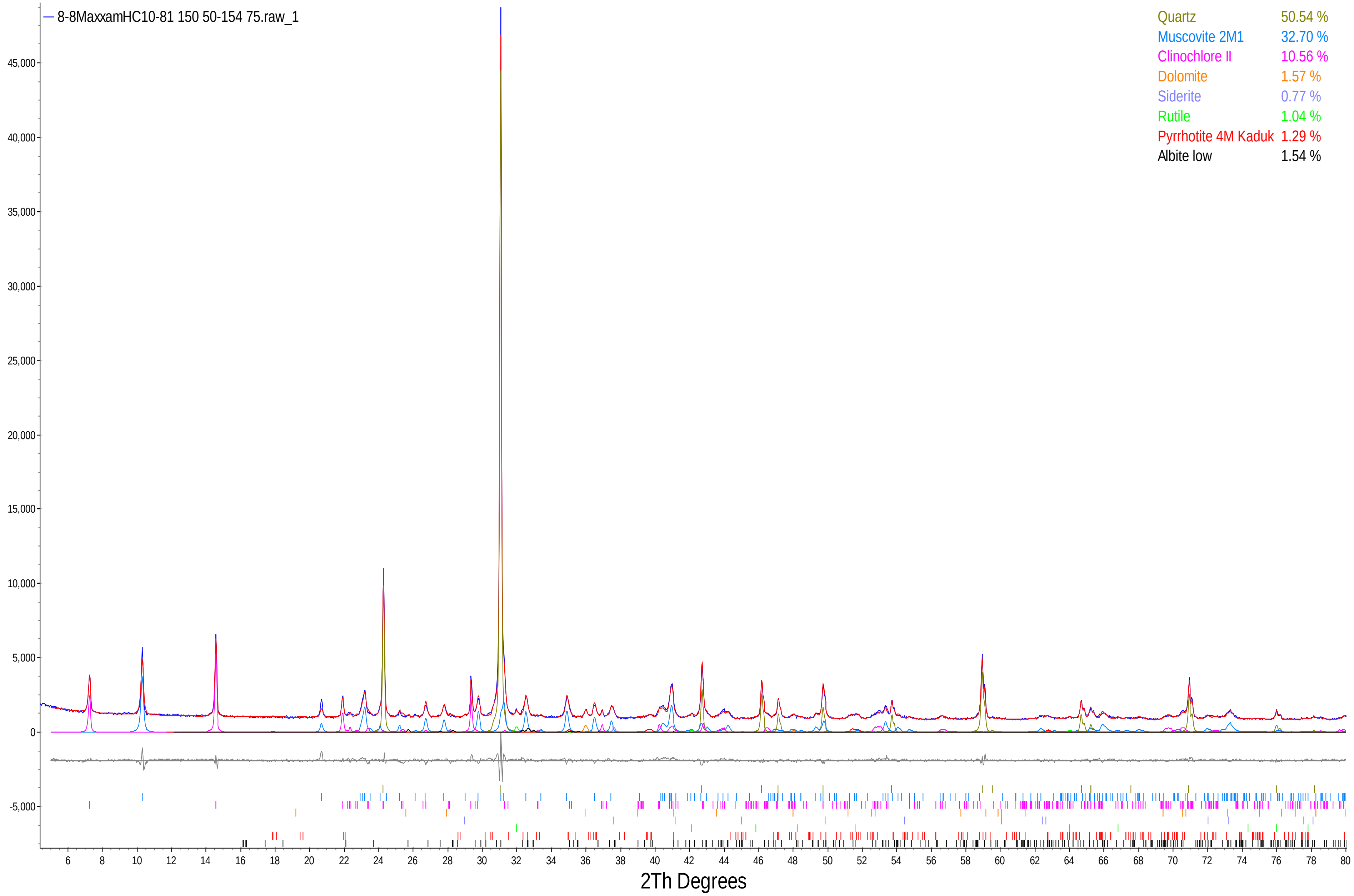


Figure 8. Rietveld refinement plot of sample **"HC10-81 From 150.50 to 154.75"** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

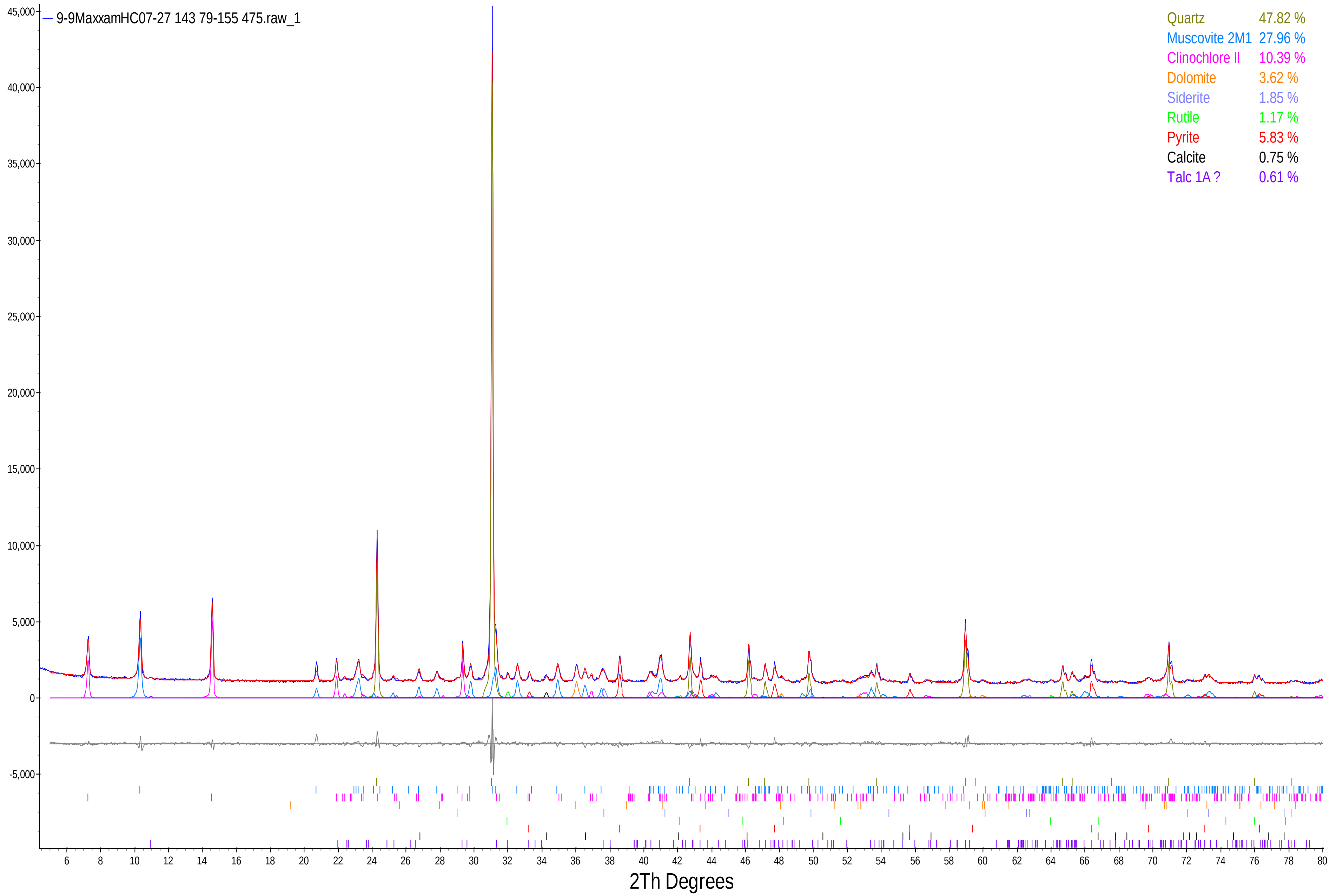


Figure 9. Rietveld refinement plot of sample **"HC07-27 From 143.79 to 155.475"** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

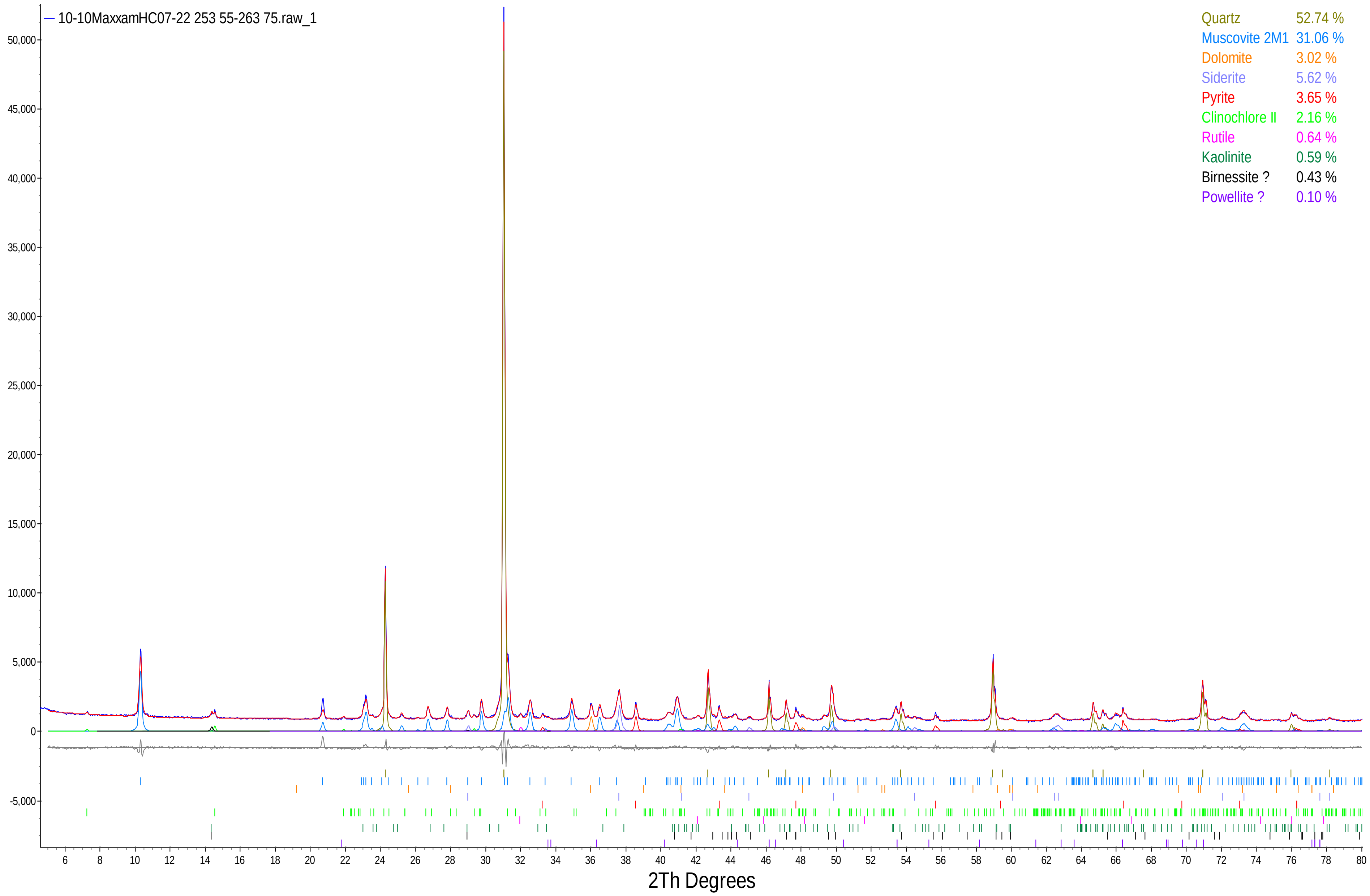


Figure 10. Rietveld refinement plot of sample **“HC07-22 From 253.55 to 263.75”** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

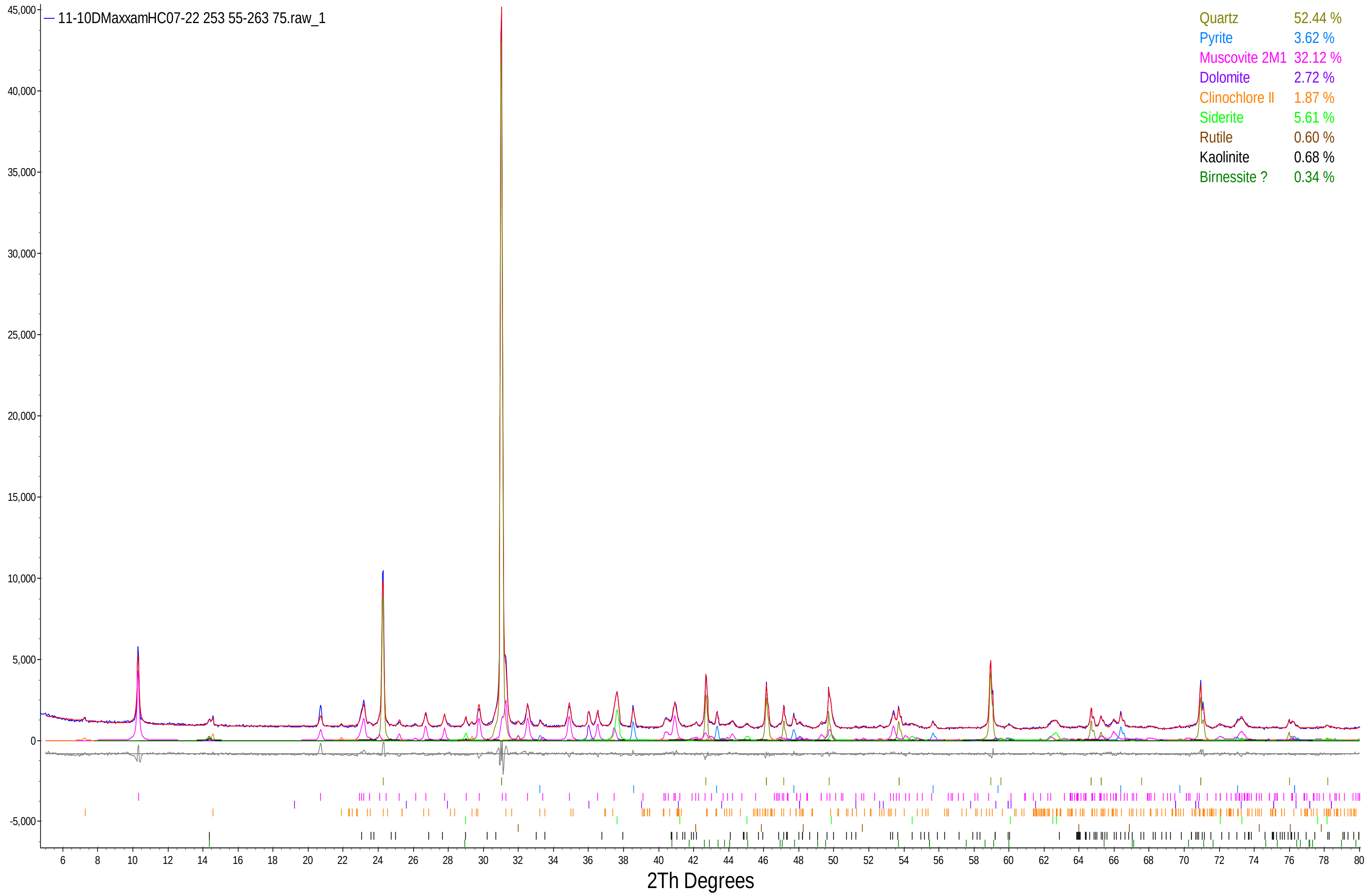


Figure 11. Rietveld refinement plot of sample **"HC07-22 From 253.55 to 263.75"** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

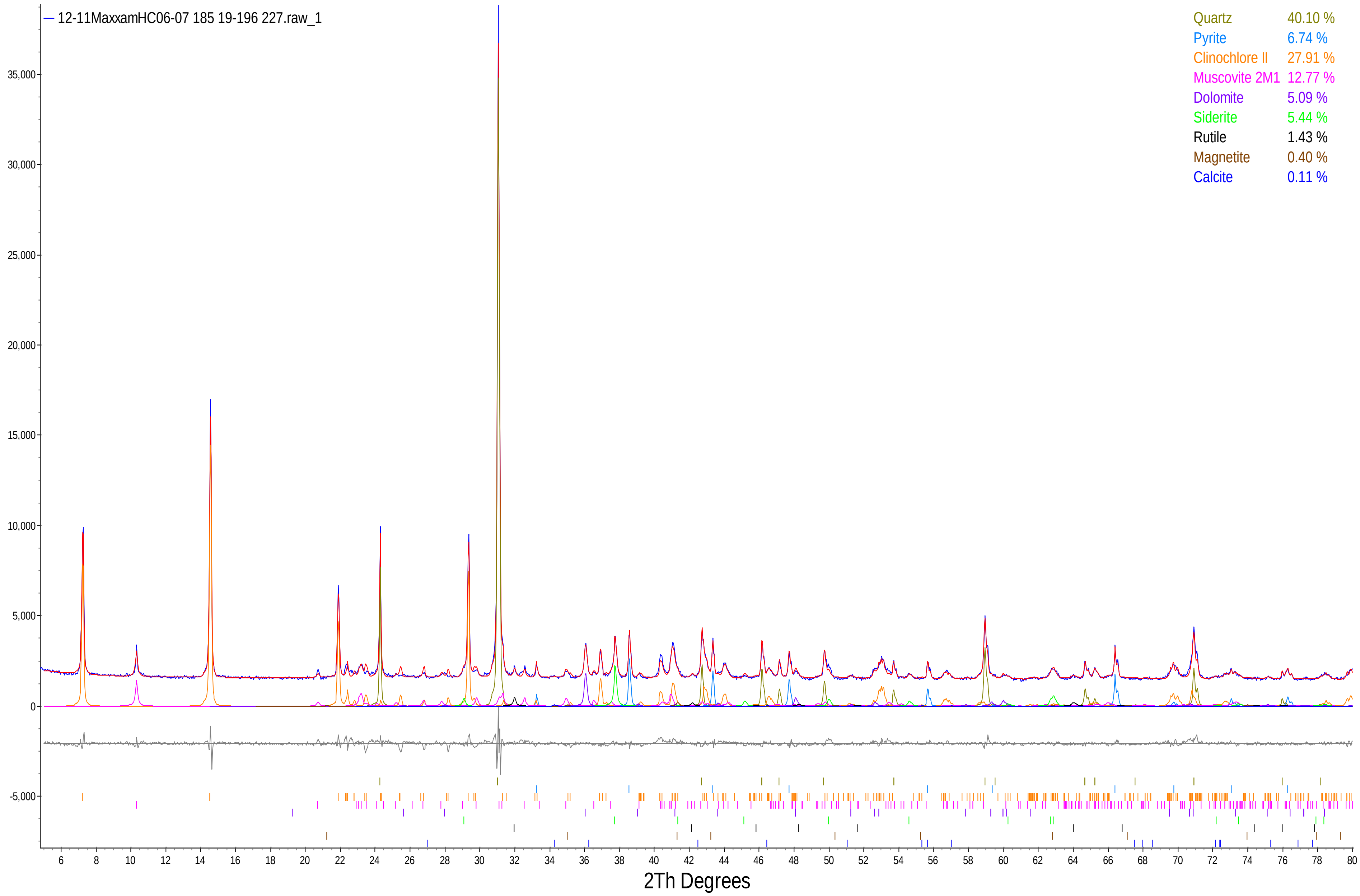


Figure 12. Rietveld refinement plot of sample **“HC06-07 From 185.19 to 196.227”** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

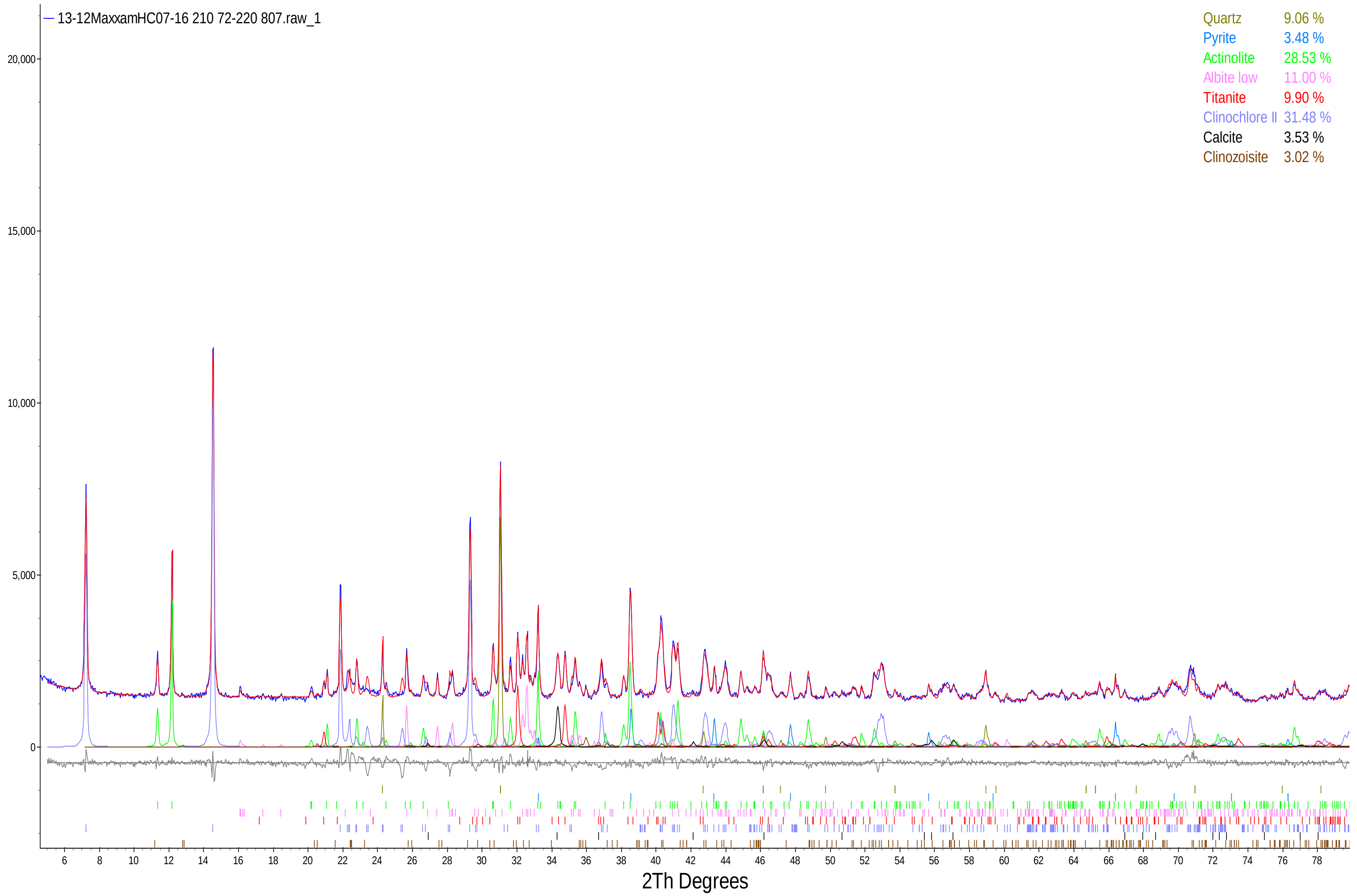


Figure 13. Rietveld refinement plot of sample **“HC07-16 From 210.72 to 220.807”** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

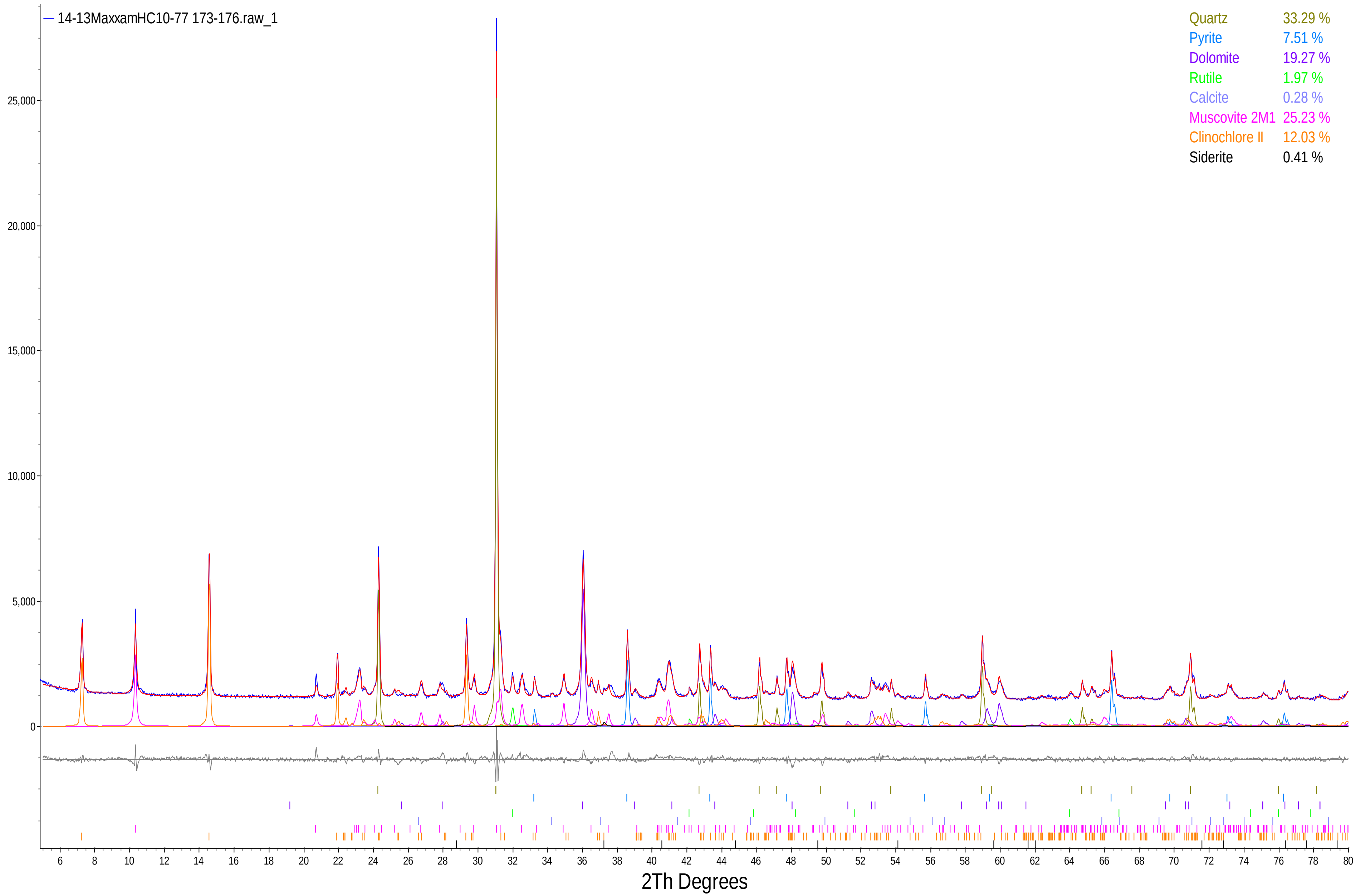


Figure 14. Rietveld refinement plot of sample **"HC10-77 From 173 to 176"** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

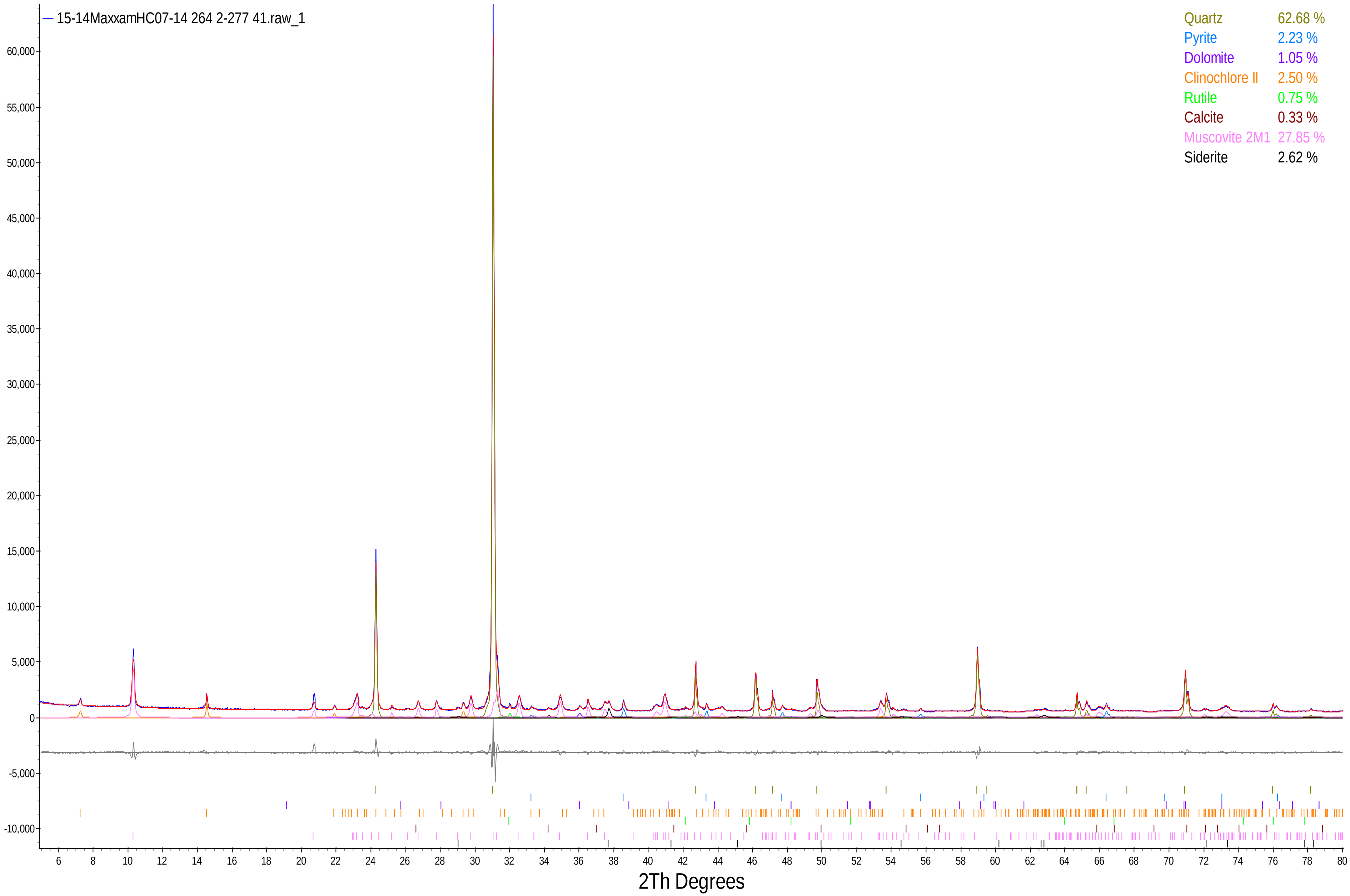


Figure 15. Rietveld refinement plot of sample **"HC07-14 From 264.2 to 277.41"** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

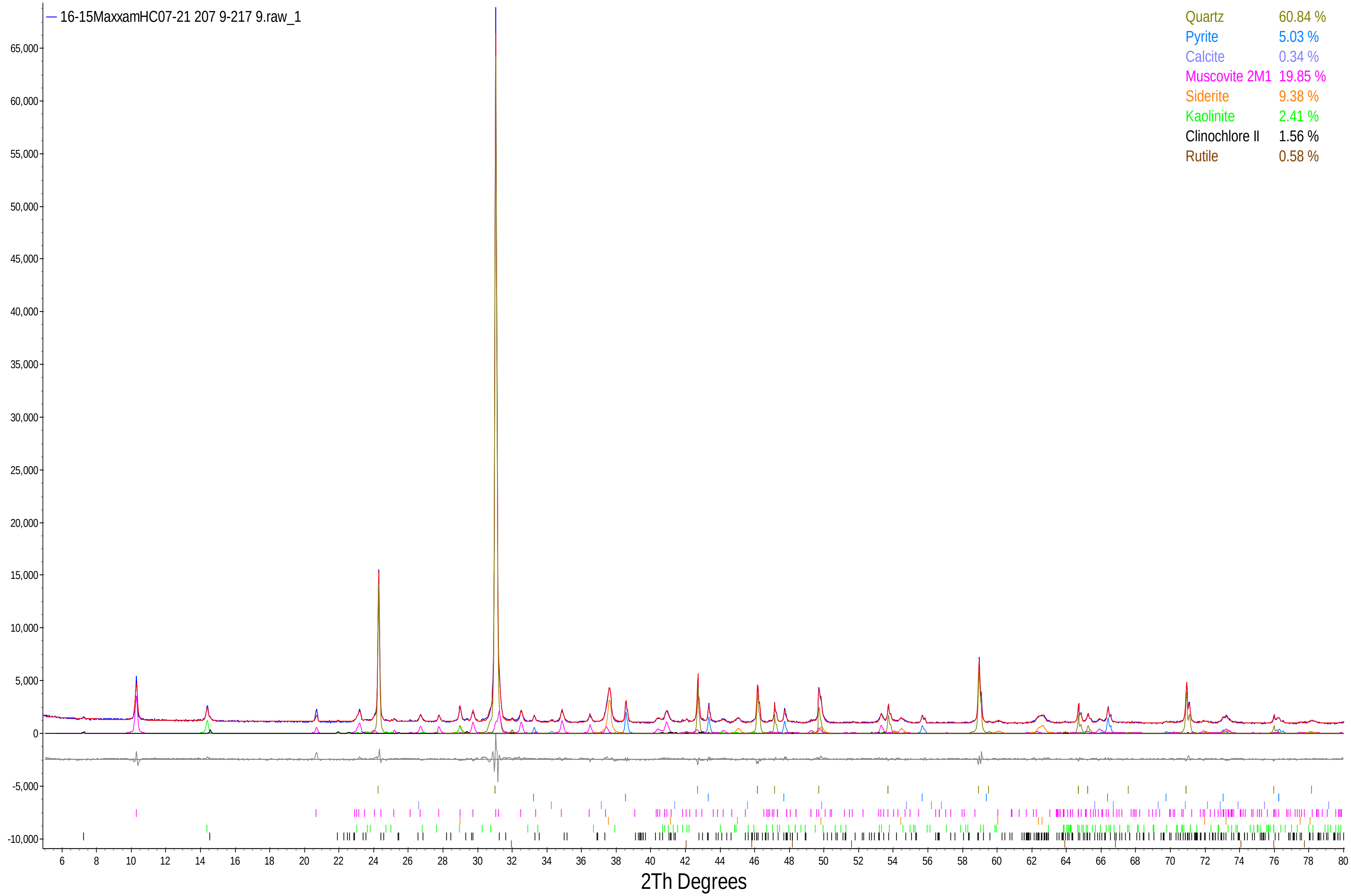


Figure 16. Rietveld refinement plot of sample **"HC07-21 From 207.9 to 217.9"** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

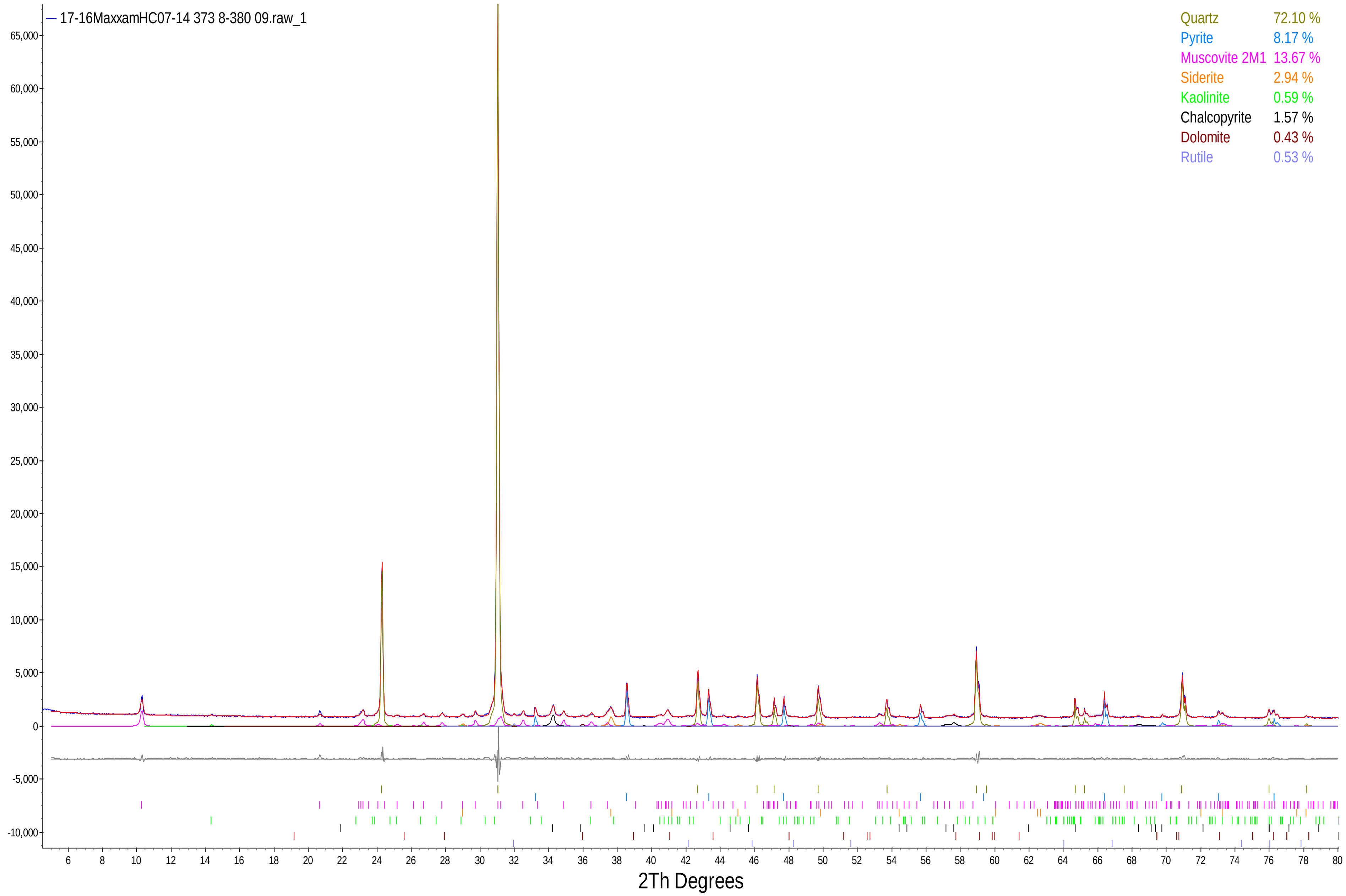


Figure 17. Rietveld refinement plot of sample **"HC07-14 From 373.8 to 380.09"** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

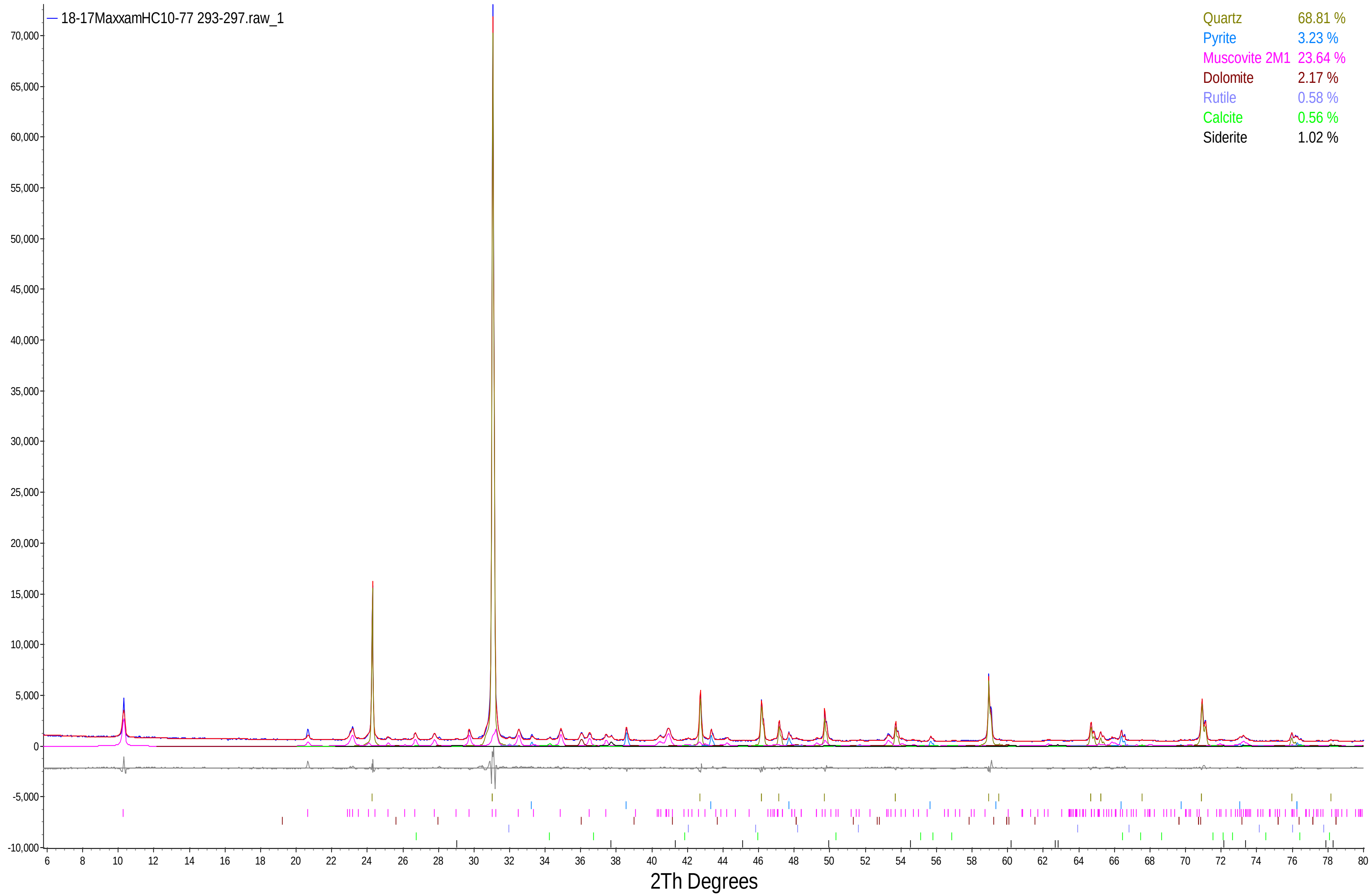


Figure 18. Rietveld refinement plot of sample **"HC10-77 From 293 to 297"** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

Appendix A3: XRD Test Results for Humidity Cell Material

	1		2		3		4		5		6
	HC07-21		HC07-42		HC07-21		HC07-21		HC10-77		HC10-77
	From 148.32 to 152.67		From 319.28 to 322.17		From 69.46 to 83.47		From 157.46 to 172.29		From 40 to 45		From 53 to 58
Quartz	66.1	Quartz	47.9	Quartz	33.4	Quartz	31.4	Quartz	25.3	Quartz	16.7
Muscovite	21.9	Muscovite	34.3	Dolomite	5.2	Pyrite	1.5	Pyrite	3.3	Pyrite	5.6
Kaolinite	1.8	Dolomite	2.3	Siderite	11.6	Kaolinite	11.6	Clinochlore	23.1	Clinochlore	33.5
Pyrite	2.5	Pyrite	2.9	Pyrite	0.9	Dolomite	10.9	Calcite	9.0	Calcite	17.5
Galena	0.1	Clinochlore	10.4	K-feldspar	14.6	Siderite	12.3	Plagioclase	13.0	Plagioclase	16.3
Siderite	3.0	Siderite	0.7	Titanite	1.2	Clinochlore	5.8	Hydroxylapati	1.0	Titanite	8.1
Dolomite	4.2	Pyrrhotite	0.6	Plagioclase	5.3	K-feldspar	14.7	Rutile	3.1	Hydroxylapati	1.1
Rutile ?	0.4	Rutile	0.8	Calcite	1.2	Anatase	1.3	Dolomite	15.3	Rutile	1.2
				Kaolinite	19.7	Plagioclase	1.8	Muscovite	3.5		
				Anatase	1.0	Muscovite	6.6	Paragonite	3.3		
				Ankerite	2.0	Hydroxylapatite	2.2				
				Hydroxylapat	2.3						
				Biotite	1.1						
				Magnetite	0.7						
Total	100.0		100.0		100.0		100.0		100.0		100.0

	10		10D		11		12		13		14
	HC07-22		HC07-22		HC06-07		HC07-16		HC10-77		HC07-14
	From 253.55 to 263.75		From 253.55 to 263.75		From 185.19 to 196.227		From 210.72 to 220.807		From 173 to 176		From 264.2 to 277.41
Quartz	52.7	Quartz	52.4	Quartz	40.1	Quartz	9.1	Quartz	33.3	Quartz	62.7
Muscovite	31.1	Pyrite	3.6	Pyrite	6.7	Pyrite	3.5	Pyrite	7.5	Pyrite	2.2
Dolomite	3.0	Muscovite	32.1	Clinochlore	27.9	Actinolite	28.5	Dolomite	19.3	Dolomite	1.1
Siderite	5.6	Dolomite	2.7	Muscovite	12.8	Plagioclase	11.0	Rutile	2.0	Clinochlore	2.5
Pyrite	3.6	Clinochlore	1.9	Dolomite	5.1	Titanite	9.9	Calcite	0.3	Rutile	0.8
Clinochlore	2.2	Siderite	5.6	Siderite	5.4	Clinochlore	31.5	Muscovite	25.2	Calcite	0.3
Rutile	0.6	Rutile	0.6	Rutile	1.4	Calcite	3.5	Clinochlore	12.0	Muscovite	27.9
Kaolinite	0.6	Kaolinite	0.7	Magnetite	0.4	Clinozoisite	3.0	Siderite	0.4	Siderite	2.6
Birnessite ?	0.4	Birnessite ?	0.3	Calcite	0.1						
Powellite ?	0.1										
Total	100.0		100.0		100.0		100.0		100.0		100.0

	7		8		9
	HC10-81		HC10-81		HC07-27
	From 140.00 to 145.65		From 150.50 to 154.75		From 143.79 to 155.475
Quartz	55.4	Quartz	50.5	Quartz	47.8
Muscovite	31.5	Muscovite	32.7	Muscovite	28.0
Clinochlore	9.0	Clinochlore	10.6	Clinochlore	10.4
Dolomite	0.9	Dolomite	1.6	Dolomite	3.6
Siderite	1.5	Siderite	0.8	Siderite	1.9
Rutile	0.9	Rutile	1.0	Rutile	1.2
Pyrrhotite	0.9	Pyrrhotite	1.3	Pyrite	5.8
		Plagioclase	1.5	Calcite	0.8
				Talc ?	0.6
	100.0		100.0		100.0

	15		16		17
	HC07-21		HC07-14		HC10-77
	From 207.9 to 217.9		From 373.8 to 380.09		From 293 to 297
Quartz	60.8	Quartz	72.1	Quartz	68.8
Pyrite	5.0	Pyrite	8.2	Pyrite	3.2
Calcite	0.3	Muscovite	13.7	Muscovite	23.6
Muscovite	19.8	Siderite	2.9	Dolomite	2.2
Siderite	9.4	Kaolinite	0.6	Rutile	0.6
Kaolinite	2.4	Chalcopyrite	1.6	Calcite	0.6
Clinochlore	1.6	Dolomite	0.4	Siderite	1.0
Rutile	0.6	Rutile	0.5		
	100.0		100.0		100.0

Appendix B: Waste Rock Static Test Results

B1: Acid Base Accounting Results

SRK Project No: 1CY003.000

ABA Test Results for 100 Yellowhead-Harper Creek Pulp Samples - October 2011

Report to Chris Kennedy, Stephen Day & Greg Smyth: 17-Oct-2011

SRK Consulting (Canada) Inc., Yellowhead Mining Inc.-Harper Creek Project, Rec'd 31-Aug-11

S. No.	Sample ID (Composite No.)	Hole	From m	To m	Lith	Paste pH (pH Units)	Rinse EC (µS/cm)	Total Carbon (Wt.%)	CO2 (Wt.%)	CaCO3 Equiv.* (Kg CaCO3/Tonne)	Total Sulphur (Wt.%)	Sulphate Sulphur (Wt.%)	Sulphate Sulphur (Wt.%)
22 Pulps (Ore ABA)													
1	3	HC06-03	37.09	48.71	9	8.5	540	1.3	4.8	110	1.6	0.01	0.02
2	12	HC06-03	174.06	182.30	7	7.9	790	0.54	2.2	49	2.8	0.03	0.03
3	19	HC06-03	290.63	298.36	8	8.7	480	3	10	240	1.6	0.02	0.02
4	29	HC06-06	13.45	25.07	9	7.8	950	0.49	2	44	1.6	0.03	0.04
5	175	HC07-14	213.48	226.09	9	7.9	1300	0.5	2	44	1	0.05	0.05
6	187	HC07-14	373.80	380.09	11	4	3600	0.24	0.9	20	5	0.43	0.33
7	191	HC07-14	427.52	437.18	8	7.9	910	0.84	3.1	70	2.8	0.05	0.03
8	215	HC07-15	274.32	281.35	9	8.3	530	0.77	2.9	67	0.48	0.02	0.02
9	262	HC07-16	274.17	285.76	7	8.3	500	1.2	4.6	100	1.1	0.02	0.02
10	295	HC07-18	123.00	129.75	8	8.1	630	1.3	4.6	110	0.64	0.03	0.02
11	303	HC07-18	212.17	224.89	11	7.5	970	0.21	0.87	20	2.7	0.03	0.04
12	316	HC07-18	384.34	392.28	9	8.1	850	1.7	6.9	160	3.7	0.03	0.03
13	331	HC07-21	83.47	96.91	11	7.9	610	0.27	1.1	24	1.4	0.02	0.03
14	341	HC07-22	23.33	34.46	9	8.3	570	1.1	4.2	95	1.1	0.02	0.02
15	350	HC07-22	131.89	143.82	11	7	890	0.08	0.29	6.6	1.4	0.03	0.04
16	400	HC07-24	427.29	436.15	9	8.1	420	0.95	3.5	78	0.99	0.02	0.02
17	444	HC07-27	81.09	92.83	7	8.3	520	1.2	4.8	110	1.7	0.02	0.03
18	448	HC07-27	143.79	155.48	7	7.7	1000	0.74	2.9	65	3	0.06	0.06
19	449	HC07-27	155.48	167.16	7	8.3	390	1.3	5.1	120	2	0.02	0.02
20	511	HC07-41	116.54	123.77	8	8.5	530	2.1	8.2	190	0.78	0.02	0.02
21	582	HC08-56	106.72	118.53	9	8.3	660	0.55	2.2	50	0.9	0.02	0.02
22	604	HC08-61	124.99	136.31	9	9	420	1.4	5.5	120	0.52	0.01	0.01
68 Pulps (Waste ABA)													
23	39	HC06-06	96.84	109.50	8	7	1600	0.78	2.9	66	3.7	0.15	0.14
24	38	HC06-06	119.93	130.37	7	8.5	350	0.49	1.8	40	1.5	0.02	0.02
25	61	HC06-07	185.19	196.23	8	5.7	3400	1.5	5	110	8.1	0.24	0.29
26	64	HC06-07	218.30	228.65	7	8.6	420	0.54	2	46	1.6	<0.01	0.02
27	76	HC06-10	78.71	91.21	7	7.6	540	0.66	2.3	53	1.7	0.02	0.02
28	78	HC06-10	105.54	114.55	7	8.3	590	0.86	3.2	72	2.5	0.02	0.02
29	82	HC06-10	159.50	173.31	7	7.6	600	0.77	2.8	63	1.2	0.04	0.04
30	93	HC06-12	103.76	116.00	9	8.4	440	0.93	3.5	79	2.5	0.02	0.03
31	116	HC06-12	413.61	423.48	9	8.8	380	1.2	4.4	100	0.6	0.01	0.02
32	117	HC06-12	423.48	433.35	9	8.6	350	0.85	3.2	73	0.57	0.01	0.01
33	135	HC07-13	218.91	227.81	7	8.7	420	0.61	2.3	53	0.97	0.01	0.02
34	148	HC07-13	391.25	404.75	9	9	380	0.84	3.1	71	0.82	0.01	0.01
35	159	HC07-14	30.30	36.30	8	9.2	260	0.63	2.3	53	0.22	0.01	0.01
36	169	HC07-14	134.70	144.72	8	8.6	510	2.3	8.4	190	2.7	0.03	0.03
37	180	HC07-14	264.20	277.41	9	8.2	430	0.39	1.5	33	0.87	0.02	0.02
38	194	HC07-15	16.05	26.31	7	9.2	210	2.7	9.9	230	0.12	0.01	0.01
39	196	HC07-15	43.42	56.88	8	8.5	510	0.98	3.7	83	2.8	0.03	0.03
40	200	HC07-15	92.47	101.52	9	9.7	210	0.16	0.54	12	<0.02	<0.01	0.01
41	202	HC07-15	114.48	123.97	9	8.7	250	1.6	6	140	0.69	<0.01	0.02
42	207	HC07-15	176.16	187.55	8	8.4	490	0.58	2.2	50	3.8	0.04	0.03
43	220	HC07-15	364.30	373.99	7	8.8	230	1	3.9	88	0.44	0.01	0.01
44	242	HC07-16	41.26	54.00	7	8.6	220	2.1	7.8	180	0.16	0.01	0.01
45	243	HC07-16	54.00	64.50	8	8.6	200	2.4	8.8	200	0.11	<0.01	<0.01
46	249	HC07-16	131.55	143.20	8	8.4	590	0.68	2.5	56	1.5	0.03	0.03
47	256	HC07-16	210.72	220.81	8	8.8	260	0.57	2.1	48	1.9	0.01	0.02
48	281	HC07-16	516.80	527.32	7	8	690	1.5	2.8	64	1.8	0.03	0.03
49	286	HC07-18	17.61	29.13	8	8.6	260	1.1	4.1	93	1.3	0.08	0.02
50	287	HC07-18	29.13	40.64	8	8.5	470	2	6.9	160	2.9	0.02	0.03
51	319	HC07-18	423.59	432.37	9	9	320	0.72	2.7	62	0.74	0.01	0.01
52	321	HC07-18	445.55	458.54	9	8.1	900	1	3.8	86	2.1	0.04	0.04
53	330	HC07-21	69.46	83.47	4	8.8	360	2.4	9.2	210	0.37	0.03	0.01
54	336	HC07-21	157.46	172.29	4	8.7	400	2.6	9	200	0.53	0.02	0.01
55	337	HC07-21	172.29	187.12	4	8.8	400	2.7	10	230	0.18	0.02	0.01
56	339	HC07-21	207.90	217.90	9	6.1	1200	1.1	4	91	2.7	0.07	0.09
57	359	HC07-22	253.55	263.75	7	8.5	490	0.97	3.6	82	1.7	0.02	0.02
58	366	HC07-22	350.71	363.65	9	8.9	280	0.68	2.4	54	0.43	<0.01	0.01
59	371	HC07-24	32.33	48.67	7	9	380	0.89	3	68	0.92	0.01	0.02

SRK Project No: 1CY003.000

ABA Test Results for 100 Yellowhead-Harper Creek Pulp Samples - October 2011

Report to Chris Kennedy, Stephen Day & Greg Smyth: 17-Oct-2011
SRK Consulting (Canada) Inc., Yellowhead Mining Inc.-Harper Creek Project, Rec'd 31-Aug-11

S. No.	Sample ID	Hole	From	To	Sulphide Sulphur**	Maximum Potential Acidity***	Neutralization Potential	Net Neutralization Potential****	Neutralization Potential Ratio (NPR)*****	Fizz Rating
22 Pulps (Ore ABA)	**(Composite No.)**		**m**	**m**	**(Wt.%)**	**(Kg CaCO3/Tonne)**	**(Kg CaCO3/Tonne)**	**(Kg CaCO3/Tonne)**	**(dimensionless; no unit)**	
1	3	HC06-03	37.09	48.71	1.6	49	87	38	1.8	Slight
2	12	HC06-03	174.06	182.30	2.8	86	25	-62	0.28	None
3	19	HC06-03	290.63	298.36	1.6	49	140	93	2.9	Slight
4	29	HC06-06	13.45	25.07	1.6	48	81	32	1.7	Slight
5	175	HC07-14	213.48	226.09	0.99	31	25	-5.9	0.81	Slight
6	187	HC07-14	373.80	380.09	4.5	140	-0.13	-142	0.00	None
7	191	HC07-14	427.52	437.18	2.7	85	40	-45	0.47	Slight
8	215	HC07-15	274.32	281.35	0.46	14	36	21	2.5	Slight
9	262	HC07-16	274.17	285.76	1.1	34	100	71	3.1	Moderate
10	295	HC07-18	123.00	129.75	0.61	19	95	76	5	Slight
11	303	HC07-18	212.17	224.89	2.6	82	15	-67	0.19	Slight
12	316	HC07-18	384.34	392.28	3.7	110	95	-20	0.83	Slight
13	331	HC07-21	83.47	96.91	1.4	44	19	-25	0.43	Slight
14	341	HC07-22	23.33	34.46	1.1	33	36	3	1.1	Slight
15	350	HC07-22	131.89	143.82	1.4	44	5.1	-39	0.12	None
16	400	HC07-24	427.29	436.15	0.97	30	38	8.2	1.3	Slight
17	444	HC07-27	81.09	92.83	1.7	53	88	35	1.6	Slight
18	448	HC07-27	143.79	155.48	3	93	40	-52	0.44	Slight
19	449	HC07-27	155.48	167.16	2	62	97	35	1.6	Slight
20	511	HC07-41	116.54	123.77	0.76	24	130	110	5.6	Slight
21	582	HC08-56	106.72	118.53	0.88	28	27	-0.12	1	Slight
22	604	HC08-61	124.99	136.31	0.51	16	94	78	5.9	Slight
68 Pulps (Waste ABA)										
23	39	HC06-06	96.84	109.50	3.6	110	30	-81	0.27	Slight
24	38	HC06-06	119.93	130.37	1.5	48	21	-26	0.45	Slight
25	61	HC06-07	185.19	196.23	7.9	250	41	-205	0.17	Slight
26	64	HC06-07	218.30	228.65	1.6	51	27	-23	0.54	Slight
27	76	HC06-10	78.71	91.21	1.7	53	7.8	-45	0.15	None
28	78	HC06-10	105.54	114.55	2.4	76	24	-52	0.32	Slight
29	82	HC06-10	159.50	173.31	1.2	37	12	-25	0.33	None
30	93	HC06-12	103.76	116.00	2.5	78	62	-15	0.8	Slight
31	116	HC06-12	413.61	423.48	0.59	18	38	20	2.1	Slight
32	117	HC06-12	423.48	433.35	0.56	18	15	-2.27	0.87	Slight
33	135	HC07-13	218.91	227.81	0.96	30	22	-7.63	0.75	Slight
34	148	HC07-13	391.25	404.75	0.81	25	39	13	1.5	Slight
35	159	HC07-14	30.30	36.30	0.21	6.6	59	53	9	Strong
36	169	HC07-14	134.70	144.72	2.7	83	160	74	1.9	Slight
37	180	HC07-14	264.20	277.41	0.85	27	11	-16	0.42	None
38	194	HC07-15	16.05	26.31	0.11	3.4	240	240	70	Strong
39	196	HC07-15	43.42	56.88	2.8	87	510	420	5.9	Strong
40	200	HC07-15	92.47	101.52	<0.02	<0.6	19	19	#N/A	Slight
41	202	HC07-15	114.48	123.97	0.69	22	130	110	6.1	Strong
42	207	HC07-15	176.16	187.55	3.7	120	70	-46	0.6	Strong
43	220	HC07-15	364.30	373.99	0.43	13	24	10	1.8	Slight
44	242	HC07-16	41.26	54.00	0.15	4.7	180	170	38	Strong
45	243	HC07-16	54.00	64.50	0.11	3.4	200	200	58	Strong
46	249	HC07-16	131.55	143.20	1.5	46	63	17	1.4	Strong
47	256	HC07-16	210.72	220.81	1.9	58	35	-24	0.59	Strong
48	281	HC07-16	516.80	527.32	1.8	55	14	-41	0.26	None
49	286	HC07-18	17.61	29.13	1.3	39	110	72	2.8	Strong
50	287	HC07-18	29.13	40.64	2.8	88	170	77	1.9	Strong
51	319	HC07-18	423.59	432.37	0.73	23	32	9.3	1.4	Slight
52	321	HC07-18	445.55	458.54	2.1	65	29	-36	0.45	Slight
53	330	HC07-21	69.46	83.47	0.34	11	98	88	9.3	Slight
54	336	HC07-21	157.46	172.29	0.51	16	100	85	6.3	Slight
55	337	HC07-21	172.29	187.12	0.16	5	57	52	11	None
56	339	HC07-21	207.90	217.90	2.6	81	18	-63	0.22	Slight
57	359	HC07-22	253.55	263.75	1.6	51	22	-29	0.44	Slight
58	366	HC07-22	350.71	363.65	0.43	13	20	6.9	1.5	Slight
59	371	HC07-24	32.33	48.67	0.91	28	49	21	1.7	Slight

SRK Project No: 1CY003.000

ABA Test Results for 100 Yellowhead-Harper Creek Pulp Samples - October 2011

Report to Chris Kennedy, Stephen Day & Greg Smyth: 17-Oct-2011
SRK Consulting (Canada) Inc., Yellowhead Mining Inc.-Harper Creek Project, Rec'd 31-Aug-11

S. No.	Sample ID (Composite No.)	Hole	From m	To m	Lith	Paste pH (pH Units)	Rinse EC (µS/cm)	Total Carbon (Wt.%)	CO2 (Wt.%)	CaCO3 Equiv.* (Kg CaCO3/Tonne)	Total Sulphur (Wt.%)	Sulphate Sulphur (Wt.%)	Sulphate Sulphur (Wt.%)
22 Pulps (Ore ABA)													
60	382	HC07-24	178.82	189.23	8	8.6	480	1.5	5.8	130	1.9	0.02	0.02
61	387	HC07-24	253.39	265.75	8	9	300	2.7	9.6	220	1.2	0.01	0.02
62	394	HC07-24	362.34	371.53	9	9.1	180	0.53	1.9	43	0.12	0.01	0.01
63	414	HC07-26	112.71	123.23	7	7.5	560	0.41	1.5	34	1.2	<0.01	0.02
64	416	HC07-26	138.00	152.90	7	6.8	720	0.68	2.4	55	1.5	0.03	0.03
65	433	HC07-26	374.50	391.05	7	8.9	310	1.5	5.3	120	0.37	0.01	0.01
66	453	HC07-27	207.80	219.07	9	8.1	430	0.34	1.4	31	0.96	0.01	0.02
67	456	HC07-27	247.10	261.88	9	7.1	660	0.37	1.4	33	1.4	0.03	0.04
68	464	HC07-27	355.81	366.51	9	9.1	300	0.62	2.2	51	0.39	0.01	0.01
69	472	HC07-40	30.59	42.70	8	9	290	4	14	330	1.1	0.01	0.01
70	474	HC07-40	56.32	68.31	8	8.9	220	2.3	8.3	190	1	0.01	0.02
71	487	HC07-40	281.17	292.14	9	8.1	870	0.68	2.5	56	2.3	0.04	0.04
72	489	HC07-40	303.89	316.55	9	8.3	810	0.43	1.4	33	2.5	0.03	0.03
73	493	HC07-40	372.26	379.55	7	8.1	230	0.14	0.45	10	0.76	0.01	0.02
74	496	HC07-40	401.42	411.24	9	8	400	0.51	1.8	40	1	0.01	0.02
75	497	HC07-40	411.24	417.95	7	8.7	420	0.95	3.5	79	1.7	0.02	0.02
76	506	HC07-41	53.85	61.15	8	9.1	330	3.7	12	280	1.7	0.01	0.02
77	534	HC07-41	421.90	432.93	9	9.3	260	1.9	6.3	140	0.21	0.01	0.01
78	540	HC07-42	29.57	40.17	8	9	400	4.1	14	310	2.1	0.01	0.02
79	552	HC07-42	175.86	189.15	9	9.1	350	0.96	3.5	79	0.32	0.01	0.01
80	553	HC07-42	189.15	202.43	9	9.1	260	0.66	2.3	53	0.53	0.01	0.02
81	566	HC07-42	366.60	375.82	8	8.5	270	0.28	0.93	21	0.5	0.01	0.01
82	578	HC08-56	59.43	68.89	9	9	280	0.49	1.8	40	0.51	0.01	0.01
83	585	HC08-56	142.15	153.95	9	8.8	320	0.31	1.1	25	0.44	0.01	0.01
84	589	HC08-56	198.61	215.99	9	8.9	370	0.88	3	68	1.1	0.01	0.02
85	599	HC08-61	42.38	56.69	8	9.2	300	3.6	12	270	1.2	0.01	0.01
86	615	HC08-61	244.94	255.27	9	9.4	270	1	3.4	77	0.14	<0.01	0.01
87	618	HC08-61	279.00	289.55	9	9.1	320	0.99	3.4	77	0.49	0.01	0.01
88	627	HC08-61	378.63	391.19	9	8.5	670	0.81	2.8	63	1.1	0.02	0.03
89	628	HC08-61	391.19	403.74	9	9	340	0.93	3.3	75	0.67	0.01	0.02
90	629	HC08-61	403.74	416.30	9	9.1	270	0.79	2.6	58	0.46	0.01	0.01
10 Pulps (Additional ABA)													
91	648	HC07-21	148.35	152.67	1	8.8	380	0.81	2.9	65	1.1	0.01	0.02
92	649	HC07-42	61.11	65.26	1	9	370	3.9	13	290	1.8	0.01	0.02
93	650	HC07-42	319.28	322.17	1	8.3	570	0.49	1.7	39	1.6	0.02	0.03
94	654	HC10-78	30.50	34.60	1	9.4	230	2.3	7.4	170	0.19	<0.01	0.01
95	656	HC10-81	41.70	46.30	6	9.4	230	1.1	3.2	73	0.38	<0.01	0.01
96	659	HC10-81	132.55	136.30	6	9.6	210	1.3	4.2	95	0.54	<0.01	0.01
97	660	HC10-81	138.20	140.00	6	9.2	210	0.78	2.6	60	0.32	<0.01	0.01
98	661	HC10-81	140.00	145.65	6	9	230	0.5	1.1	25	0.58	0.01	0.01
99	662	HC10-81	150.50	154.75	6	9.2	190	0.45	1.1	25	0.46	<0.01	0.01
100	663	HC10-81	154.75	157.50	6	9.1	170	0.54	1.2	26	0.55	<0.01	0.01
Detection Limits						*0.5*	*0.5*	*0.02*	*0.02*	*0.5*	*0.02*	*0.01*	*0.01*
Maxxam SOP No:						*7160*	*7190*	*LECO*	*LECO*	*Calculation*	*LECO*	*HCl Leach (SOP: 7410)*	*Na2CO3 Leach*

SRK Project No: 1CY003.000

ABA Test Results for 100 Yellowhead-Harper Creek Pulp
Samples - October 2011

Report to Chris Kennedy, Stephen Day & Greg Smyth: 17-Oct-2011
SRK Consulting (Canada) Inc., Yellowhead Mining Inc.-Harper Creek Project, Rec'd 31-Aug-11

S. No.	Sample ID (Composite No.)	Hole	From m	To m	Sulphide Sulphur** (Wt.%)	Maximum Potential Acidity*** (Kg CaCO3/Tonne)	Neutralization Potential (Kg CaCO3/Tonne)	Net Neutralization Potential**** (Kg CaCO3/Tonne)	Neutralization Potential Ratio (NPR)***** (dimensionless; no unit)	Fizz Rating
22 Pulps (Ore ABA)										
60	382	HC07-24	178.82	189.23	1.9	58	79	21	1.4	Slight
61	387	HC07-24	253.39	265.75	1.2	37	180	150	5	Slight
62	394	HC07-24	362.34	371.53	0.11	3.4	19	16	5.6	Slight
63	414	HC07-26	112.71	123.23	1.2	36	5.6	-31	0.16	None
64	416	HC07-26	138.00	152.90	1.5	47	6.1	-41	0.13	None
65	433	HC07-26	374.50	391.05	0.36	11	54	43	4.8	Slight
66	453	HC07-27	207.80	219.07	0.95	30	8.4	-21	0.28	None
67	456	HC07-27	247.10	261.88	1.4	43	2.9	-41	0.066	None
68	464	HC07-27	355.81	366.51	0.38	12	20	7.7	1.7	Slight
69	472	HC07-40	30.59	42.70	1.1	34	280	240	8.2	Slight
70	474	HC07-40	56.32	68.31	1	32	160	130	5.1	Slight
71	487	HC07-40	281.17	292.14	2.3	71	45	-26	0.63	Slight
72	489	HC07-40	303.89	316.55	2.5	78	22	-56	0.29	Slight
73	493	HC07-40	372.26	379.55	0.75	23	4.1	-19	0.18	None
74	496	HC07-40	401.42	411.24	1	31	17	-15	0.53	Slight
75	497	HC07-40	411.24	417.95	1.7	52	57	5.3	1.1	Slight
76	506	HC07-41	53.85	61.15	1.7	52	240	180	4.6	Slight
77	534	HC07-41	421.90	432.93	0.2	6.3	110	100	17	Slight
78	540	HC07-42	29.57	40.17	2.1	66	280	210	4.2	Moderate
79	552	HC07-42	175.86	189.15	0.31	9.7	66	56	6.8	Slight
80	553	HC07-42	189.15	202.43	0.52	16	44	28	2.7	Slight
81	566	HC07-42	366.60	375.82	0.49	15	11	-4.5	0.71	Slight
82	578	HC08-56	59.43	68.89	0.5	16	23	7.7	1.5	Slight
83	585	HC08-56	142.15	153.95	0.43	13	15	1.9	1.1	Slight
84	589	HC08-56	198.61	215.99	1.1	33	45	12	1.4	Slight
85	599	HC08-61	42.38	56.69	1.2	37	220	180	5.9	Slight
86	615	HC08-61	244.94	255.27	0.14	4.4	66	61	15	Slight
87	618	HC08-61	279.00	289.55	0.48	15	49	34	3.3	Slight
88	627	HC08-61	378.63	391.19	1	33	50	17	1.5	Slight
89	628	HC08-61	391.19	403.74	0.66	21	59	38	2.9	Slight
90	629	HC08-61	403.74	416.30	0.45	14	43	29	3	Slight
10 Pulps (Additional ABA)										
91	648	HC07-21	148.35	152.67	1.1	35	31	-4.0	0.88	Slight
92	649	HC07-42	61.11	65.26	1.8	55	260	200	4.6	Slight
93	650	HC07-42	319.28	322.17	1.6	49	24	-25	0.49	Slight
94	654	HC10-78	30.50	34.60	0.19	5.9	160	160	27	Strong
95	656	HC10-81	41.70	46.30	0.38	12	35	23	3	Slight
96	659	HC10-81	132.55	136.30	0.54	17	81	64	4.8	Slight
97	660	HC10-81	138.20	140.00	0.32	10	5.9	-4.1	0.59	None
98	661	HC10-81	140.00	145.65	0.57	18	9.4	-8.4	0.53	None
99	662	HC10-81	150.50	154.75	0.46	14	14	-0.03	1	Slight
100	663	HC10-81	154.75	157.50	0.55	17	15	-2.6	0.85	Slight
Detection Limits					*0.02*	*0.6*				
Maxxam SOP No:					*Calculated from HCL Leach*	*Calculation*	*7150*	*Calculation*	*Calculation*	*7150*

Notes:
100% rush charges for 3 week TAT on ABA, sulphate S by Na2CO3 leach & CO2.
Rinse EC & Paste pH was done on as-rec'd pulp.
Total sulphur, total carbon and carbonate carbon (CO2; HCl direct method) done by Leco at Acme Labs.
Calculations:
*CaCO3 equivalents is based on carbonate carbon.
**Sulphide sulphur is based on difference between total sulphur and sulphate sulphur (by HCl leach).
***MPA (Maximum Potential Acidity) is based on sulphide sulphur .
****NNP (Net Neutralization Potential) is based on difference between neutralization potential (NP) and MPA.
*****NPR (Neutralization Potential Ratio) is NP divided by MPA.
References:
Reference for Mod ABA NP method (SOP No. 7150): MEND Acid Rock Drainage Prediction Manual, MEND Project 1.16.1b (pages 6.2-11 to 17), March 1991.

B2: Trace Element Analysis Results

SRK Project No: 1CY003.000

Trace Element Test Results for 100 Yellowhead-Harper Creek Pulp Samples - October 2011

S. No. 22 Pulps (Ore ABA)	Sample ID (Composite No.)	Hole	From m	To m	Lith	Ag ppm	Al %	As ppm	Ba ppm	Be ppm	Bi ppm	Ca %	Cd ppm	Ce ppm	Co ppm	Cr ppm	Cu ppm	Fe %	Ga ppm	Ge ppm	Hg ppb	K %	La ppm	Li ppm	Mg %	Mn ppm	Mo ppm
1	3	HC06-03	37.09	48.71	9	0.8	0.87	2.6	22	<0.1	2.7	2.2	0.42	5.2	12	150	2600	3.5	2.3	1.2	5	0.07	2.5	9.6	1.7	530	18
2	12	HC06-03	174.06	182.30	7	1.3	0.44	35	58	0.3	1	0.57	0.33	20	20	89	2300	3.6	1.6	1.3	<5	0.15	10	2	0.68	150	3.4
3	19	HC06-03	290.63	298.36	8	0.8	1.6	120	26	0.1	0.74	3.9	0.46	4.9	35	120	1200	7.7	5.4	2.8	5	0.1	2	13	3.4	750	3.5
4	29	HC06-06	13.45	25.07	9	2.9	1.8	18	24	0.5	1.5	0.79	1.5	4.6	16	98	6400	3.9	4.4	1.7	20	0.1	2.5	19	2.1	350	16
5	175	HC07-14	213.48	226.09	9	1.6	1.4	4.1	47	<0.1	2.1	0.56	0.62	13	15	210	3900	3.3	3.1	2.7	5	0.15	6.5	11	1.5	260	21
6	187	HC07-14	373.80	380.09	11	2.8	0.22	72	34	0.1	3.5	0.17	1.5	8.6	47	210	5800	5.2	1	3.6	10	0.09	4	<0.1	0.23	170	5.7
7	191	HC07-14	427.52	437.18	8	0.6	3.5	25	37	0.4	3.7	1.2	0.38	12	41	180	1100	>10	12	8	<5	0.18	6	24	3.3	1000	3.5
8	215	HC07-15	274.32	281.35	9	1.2	0.9	4.4	41	<0.1	1.1	0.7	1.9	13	7.6	180	1300	2.1	2.1	1.6	5	0.1	6.5	9.8	1.4	300	8.7
9	262	HC07-16	274.17	285.76	7	0.9	4	21	30	0.7	0.38	3.7	0.43	50	32	290	3200	7.3	15	22	<5	0.09	24	21	4.1	1000	20
10	295	HC07-18	123.00	129.75	8	2.1	3.4	7.3	19	0.2	0.76	2.2	0.61	16	34	280	4500	6.3	13	5.2	10	0.03	7	34	4	650	13
11	303	HC07-18	212.17	224.89	11	1.9	0.22	27	39	<0.1	4.6	0.3	2.3	11	15	170	1500	2.5	0.7	2	25	0.1	5.5	<0.1	0.19	66	7.8
12	316	HC07-18	384.34	392.28	9	0.9	0.72	34	34	0.4	2.7	1.9	2	11	16	130	1700	6.4	2.6	4.8	10	0.12	5	6.1	1.5	640	2.7
13	331	HC07-21	83.47	96.91	11	1.3	0.2	88	60	<0.1	2	0.35	0.83	12	7	200	1500	1.5	0.6	1.3	15	0.08	6	0.3	0.22	91	7.9
14	341	HC07-22	23.33	34.46	9	0.8	0.75	4.8	38	0.2	0.74	0.56	0.29	8.6	10	130	1700	2.6	1.9	2.2	<5	0.12	4.5	6.6	1.6	380	4.6
15	350	HC07-22	131.89	143.82	11	0.9	0.16	120	32	<0.1	2.3	0.13	1.6	11	6.1	260	1200	1.5	0.5	1.3	20	0.06	5.5	0.3	0.07	39	8.6
16	400	HC07-24	427.29	436.15	9	0.7	1.5	7.1	87	0.2	2.5	0.99	0.66	21	19	150	1300	5.9	5	18	<5	0.16	12	9.3	1.3	1000	5
17	444	HC07-27	81.09	92.83	7	1.3	1.5	17	33	0.4	1	1.9	1.8	23	28	170	1600	5	5.8	3.2	15	0.14	12	17	2	570	30
18	448	HC07-27	143.79	155.48	7	2.1	1.7	46	48	0.3	4.7	0.89	0.93	17	23	130	2400	6	5.4	4.3	10	0.18	9.5	14	2	560	13
19	449	HC07-27	155.48	167.16	7	0.8	3.5	68	23	0.2	2.2	2	0.63	12	32	220	1000	7.7	12	5.9	<5	0.08	6.5	45	4.8	810	6.4
20	511	HC07-41	116.54	123.77	8	1.6	1.9	21	32	0.2	0.44	3.1	0.72	15	24	180	3700	5.3	6.4	16	10	0.18	7.5	18	3.2	1200	8.5
21	582	HC08-56	106.72	118.53	9	1.7	1	23	53	0.2	0.66	0.58	0.46	24	15	170	1700	3.5	3.2	2.9	10	0.19	14	7.9	1.2	440	3.2
22	604	HC08-61	124.99	136.31	9	1.3	0.92	9.4	26	0.3	0.28	1.9	0.53	20	12	210	2800	2.9	3.1	1.9	10	0.11	11	8.7	1.8	660	8.8
68 Pulps (Waste ABA)																											
23	39	HC06-06	96.84	109.50	8	0.4	2	58	48	0.4	3.1	0.74	1.7	10	39	180	560	8.4	6.4	3.5	<5	0.23	5	14	2.3	690	3.3
24	38	HC06-06	119.93	130.37	7	0.9	1.1	47	40	<0.1	2.6	0.15	0.51	15	24	120	1300	6.7	3.2	2.8	<5	0.14	7	4.3	1.1	360	2
25	61	HC06-07	185.19	196.23	8	1	3.3	370	18	0.3	4.4	1.1	2.2	4.2	46	230	710	>10	11	7.4	15	0.06	2	19	4.7	1300	3.9
26	64	HC06-07	218.30	228.65	7	<0.1	1.4	23	76	0.3	0.64	0.67	0.31	26	21	110	63	4.9	4.2	2.3	5	0.29	13	7.3	1.4	680	2
27	76	HC06-10	78.71	91.21	7	0.3	0.78	22	43	0.3	1.6	0.16	0.17	24	19	120	320	4.5	2.7	2.3	10	0.19	12	5.5	1	400	2.3
28	78	HC06-10	105.54	114.55	7	0.1	0.37	15	54	<0.1	1.1	0.38	0.17	20	14	170	140	3.7	1.5	1.7	5	0.22	9.5	3.2	0.85	390	2.1
29	82	HC06-10	159.50	173.31	7	0.4	0.6	170	48	0.3	1.1	0.19	0.21	24	18	110	340	4	2.3	1.8	10	0.23	12	3.9	0.78	460	1.1
30	93	HC06-12	103.76	116.00	9	0.6	3.4	48	14	0.4	1.6	1.2	0.67	5.3	34	200	600	6.9	9.8	3.2	10	0.07	3	13	3.6	720	2.6
31	116	HC06-12	413.61	423.48	9	<0.1	0.45	3.9	48	<0.1	0.9	0.79	0.07	25	12	150	100	3.3	1.7	1.6	5	0.16	12	4.4	0.88	500	1.4
32	117	HC06-12	423.48	433.35	9	0.1	0.31	4.4	52	0.1	0.94	0.28	0.29	29	12	130	100	2.6	1.3	1.3	5	0.19	14	4	0.55	340	1.5
33	135	HC07-13	218.91	227.81	7	0.5	0.75	18	57	0.1	0.48	0.36	0.16	22	8.1	130	640	2.2	2.4	1.1	5	0.19	11	5.7	1.1	250	4.4
34	148	HC07-13	391.25	404.75	9	0.3	0.96	8.1	35	0.1	1.1	0.81	0.21	22	12	160	390	3.3	3.9	1.7	5	0.16	11	6.2	1.3	480	2.1
35	159	HC07-14	30.30	36.30	8	<0.1	2	5.6	45	0.4	0.16	2.3	0.16	26	17	220	160	3.2	7.1	4.2	<5	0.11	12	12	2	370	6.6
36	169	HC07-14	134.70	144.72	8	0.5	3.2	30	30	0.5	0.9	5.8	1.1	74	41	59	720	>10	14	9.6	10	0.08	35	19	3.2	1000	3.5
37	180	HC07-14	264.20	277.41	9	0.5	0.62	20	47	<0.1	0.52	0.24	0.27	24	7.9	150	680	2.2	1.7	1.9	5	0.14	12	3.2	0.7	140	10
38	194	HC07-15	16.05	26.31	7	<0.1	3.7	3.1	330	1.6	0.02	9.7	0.17	32	46	480	68	7.3	15	4.9	<5	0.98	15	33	4.3	1200	1.4
39	196	HC07-15	43.42	56.88	8	0.2	3.1	14	90	1.2	0.92	4.3	0.39	68	39	93	530	8.5	14	6	5	0.43	30	20	2.8	850	3.1
40	200	HC07-15	92.47	101.52	9	<0.1	1.6	1.7	22	<0.1	0.24	0.74	0.02	9.3	4.8	99	55	1.3	5	1.1	<5	0.05	3.5	10	2.1	190	4.9
41	202	HC07-15	114.48	123.97	9	0.3	2.8	27	6	0.3	2.2	4.4	0.62	40	17	230	510	4.5	10	3.2	<5	0.02	19	23	3.6	710	3.6
42	207	HC07-15	176.16	187.55	8	0.6	4	90	22	0.8	0.62	3.3	0.46	37	53	84	510	>10	17	8.1	<5	0.06	17	17	3.7	810	1.5
43	220	HC07-15	364.30	373.99	7	<0.1	0.58	5.6	44	0.2	0.42	0.5	0.37	34	11	130	59	3	2.1	2.4	10	0.13	16	3.8	0.94	390	1.5
44	242	HC07-16	41.26	54.00	7	<0.1	4.3	8.4	46	0.9	0.02	8.1	0.15	43	43	470	62	7.6	17	33	<5	0.18	20	21	4.9	1100	2.2
45	243	HC07-16	54.00	64.50	8	<0.1	4.8	10	22	0.9	0.1	9.3	0.21	26	49	630	79	7.9	17	33	<5	0.11	12	18	5.4	1400	1.3
46	249	HC07-16	131.55	143.20	8	0.5	2	7.9	11	0.2	1.6	2.9	0.5	33	29	110	1200	4.8	5.4	18	20	0.04	19	5.8	1.9	570	3.3
47	256	HC07-16	210.72	220.81	8	<0.1	3.9	37	2	0.5	0.36	3	0.11	14	60	560	260	8.4	14	27	<5	<0.01	6.5	11	3.9	870	1.3
48	281	HC07-16	516.80	527.32	7	0.3	0.96	20	40	0.5	0.92	0.34	0.2	21	20	140	590	4.6	3	11	<5	0.2	12	5	1.1	740	4
49	286	HC07-18	17.61	29.13	8	0.2	4	20	16	1	0.82	5.3	0.23	88	34	56	190	9.1	21	21	<5	0.02	40	15	3.4	1100	1.7
50	287	HC07-18	29.13	40.64	8	0.6	3.5	48	47	0.9	3	7.6	1	110	40	95	640	8.5	13	17	<5	0.12	56	15	3	1200	3.2
51	319	HC07-18	423.59	432.37	9	0.1	0.67	6.1	46	0.1	0.56	0.6	0.13	23	8.9	140	190	2.7	2.1	2.5	<5	0.16	12	2.9	0.88	420	3.4
52	321	HC07-18	445.55	458.54	9	0.3	0.57	8.8	48	0.4	1.9	0.57	0.53	19	11	140	300	5.1	2.1	4	<5	0.14	9.5	2.6	0.85	470	2.6
53	330	HC07-21	69.46	83.47	4	0.2	0.87	12	230	1.1	0.52	2.5	0.45	58	15	74	170	4.3	3.5	3.6	25	0.15	25	13	1.6	750	4.7
54	336	HC07-21	157.46	172.29	4	0.2	0.81	4.9	110	1.4	0.18	2.5	0.17	63	16	64	190	4.7	3.5	4	30	0.14	28	29	1.9	860	3.4
55	337	HC07-21	172.29	187.12	4	0.1	1.1	2.3	330	0.6	0.08	2.7	0.15	70	17	54	110	4.8	4.4	4.1	25	0.18	31	16	2.1	910	3.3
56	339	HC07-21	207.90	217.90	9	0.6	0.39	100	35	0.3	4.9	0.12	0.21	18	18	170	590	5.5	1.9	4.1	10	0.11	8.5	3.6	0.77	350	3.1
57	359	HC07-22	253.55	263.75	7	0.3	0.51	12	60	0.2	1.7	0.43	0.21	24	16	170	180	3.9	1.9	3.2	<5	0.16	12	1.4	0.8	480	3.7
58	366	HC07-22	350.71	363.65	9	0.2	1.1	3.1	48	0.1	0.56	0.43	0.15	32	13	140	330	3.7	3.4	3.3	<5	0.13	16	6	0.91	480	2.8
59	371	HC07-24	32.33	48.67	7	0.7	0.84	11	60	0.1	0.72	1.2	0.62	24	7.9	170	1100	2.1	2.3	9.1	15	0.2	13	4.9	1.6	310	14
60	382	HC07-24	178.82	189.23	8	0.5	2.8	20	44	0.4	1.7	2.1	0.22	16	39	200	550	9.4	12	32	<5	0.21	8	21	3.2	1400	2.4
61	387	HC07-24	253.39	265.75	8	0.3	2.7	24	31	0.2	1.3	5	0.33	13	38	200	440	8.6	10	27	<5	0.11	6.5	25	3.6	1900	2.8
62	394	HC07-24	362.34	371.53	9	<0.1	1.5	3.6	51	0.2	0.36	0.56	0.48	39	10	170	60	3.9	5.2	12	<5	0.17	21	11	1.2	630	4.1
63	414	HC07-26	112.71	123.23	7	0.8	0.96	25	65	0.1	1.2	0.1	0.23	33	14	160	1000	3.6	3.4	11	<5	0.21	18	6.4	0.94	180	2.9
64	416	HC07-26	138.00	152.90	7	0.8	0.65	57	64	0.3	5.7	0.12	0.31	37	17	170	910	4.3	2.6	13	<5	0.24	20	3.1	0.78	300	4.2
65	433	HC07-26	374.50	391.05	7	0.1	0.77	17	62	0.5	0.88	1.2	0.27	31	13	150	140	4.5	3.2	14	<5	0.19	17	5.3	1.4	910	2.9

SRK Project No: 1CY003.000

Trace Element Test Results for 100 Yellowhead-Harper Creek Pulp Samples - October 2011

S. No. **22 Pulps (Ore ABA)**	**Sample ID (Composite No.)**	**Hole**	**From m**	**To m**	**Lith**	**Ag ppm**	**Al %**	**As ppm**	**Ba ppm**	**Be ppm**	**Bi ppm**	**Ca %**	**Cd ppm**	**Ce ppm**	**Co ppm**	**Cr ppm**	**Cu ppm**	**Fe %**	**Ga ppm**	**Ge ppm**	**Hg ppb**	**K %**	**La ppm**	**Li ppm**	**Mg %**	**Mn ppm**	**Mo ppm**
66	453	HC07-27	207.80	219.07	9	0.5	0.71	17	65	0.3	1.2	0.14	0.14	30	12	180	760	2.7	2.3	2.2	<5	0.2	17	4.6	0.69	160	4.3
67	456	HC07-27	247.10	261.88	9	1	0.69	90	63	0.1	1.5	0.1	0.95	24	15	170	760	3.5	2.4	2.8	10	0.18	14	3.5	0.65	240	3.2
68	464	HC07-27	355.81	366.51	9	<0.1	1.1	5.8	93	0.6	0.66	0.34	0.06	43	12	140	120	3.4	3.5	3	<5	0.26	24	6.5	0.9	480	3
69	472	HC07-40	30.59	42.70	8	0.6	1.8	22	17	<0.1	1.9	5.2	0.16	4.7	41	220	66	6.3	6.9	3.9	<5	0.05	2.5	15	4	1400	4.4
70	474	HC07-40	56.32	68.31	8	0.2	3.6	48	9	0.3	0.7	3.1	0.12	5.3	44	380	82	6.8	12	4.7	<5	0.04	3	38	5.7	840	3.2
71	487	HC07-40	281.17	292.14	9	0.8	2.1	21	38	0.2	1.8	0.96	0.63	12	18	120	580	6.3	5.9	4.4	5	0.15	6.5	17	1.9	760	1.1
72	489	HC07-40	303.89	316.55	9	0.9	1	21	57	0.3	3.4	0.41	0.85	21	14	180	410	5.6	3.5	3.9	<5	0.21	12	6	0.9	510	1.4
73	493	HC07-40	372.26	379.55	7	<0.1	2.2	1.6	53	0.2	0.6	0.11	0.03	38	17	93	93	5.6	6.6	4.5	<5	0.16	23	19	1.3	340	0.54
74	496	HC07-40	401.42	411.24	9	0.1	0.6	67	63	0.2	0.68	0.38	0.22	24	13	170	150	3.5	2	2.9	<5	0.18	14	3	0.65	450	1.8
75	497	HC07-40	411.24	417.95	7	0.2	1.3	10	60	0.4	4.2	1.1	3.8	21	20	120	210	5.6	4.1	4.3	20	0.22	13	11	1.7	990	0.98
76	506	HC07-41	53.85	61.15	8	0.1	1.6	29	28	<0.1	1.1	4.8	0.18	2.6	43	180	170	7.3	5.9	4.6	<5	0.1	1	18	3.6	950	8.7
77	534	HC07-41	421.90	432.93	9	<0.1	1.2	10	61	0.3	0.76	2.6	0.15	40	19	140	160	4.2	4.4	13	<5	0.2	23	8.1	1.9	1200	2.1
78	540	HC07-42	29.57	40.17	8	0.3	1.2	74	47	0.2	0.06	6.9	0.56	12	45	180	75	7.7	4.6	23	5	0.19	6	7.3	3.3	2700	2
79	552	HC07-42	175.86	189.15	9	0.5	0.65	9.7	29	0.1	0.34	1.4	0.2	36	5.8	130	950	1.4	2.4	4.6	5	0.16	20	4.4	1.1	370	19
80	553	HC07-42	189.15	202.43	9	0.8	1.1	18	34	0.2	0.52	0.88	3.2	30	8	130	820	1.9	3.8	6	15	0.17	16	10	1.4	350	21
81	566	HC07-42	366.60	375.82	8	0.2	1.5	16	65	0.2	0.38	0.17	0.44	34	13	180	76	3.7	4.9	11	10	0.3	18	6.7	1.2	520	2.9
82	578	HC08-56	59.43	68.89	9	0.3	1.5	10	71	0.2	0.36	0.53	0.2	38	14	140	380	3.8	4.9	11	<5	0.39	21	6.7	1.2	470	1.6
83	585	HC08-56	142.15	153.95	9	0.3	1.6	12	66	0.1	0.24	0.33	0.23	35	15	230	410	3.7	5	3.1	<5	0.23	20	11	1.3	350	2.8
84	589	HC08-56	198.61	215.99	9	0.4	1.3	8	64	0.2	0.98	1	1.8	22	13	130	290	4.4	3.9	3.5	10	0.26	13	6.3	1.6	750	1.3
85	599	HC08-61	42.38	56.69	8	0.3	1.4	350	25	0.3	0.46	4.9	6	7.3	38	250	140	5.9	5.2	3.9	10	0.07	4	16	3.7	1300	1.8
86	615	HC08-61	244.94	255.27	9	0.5	0.51	3.8	41	0.2	0.42	1.2	0.2	37	3.1	190	710	1.1	1.2	0.9	<5	0.17	21	2.5	1.1	280	19
87	618	HC08-61	279.00	289.55	9	0.2	0.82	15	52	0.4	0.7	0.91	1.4	31	11	210	280	2.7	2.7	2.1	10	0.19	17	4.7	1.4	550	5.4
88	627	HC08-61	378.63	391.19	9	1.3	0.62	4.9	31	0.3	2.3	1.1	1.4	18	15	210	1600	3.1	2	2.3	<5	0.09	10	3.9	0.81	650	1.7
89	628	HC08-61	391.19	403.74	9	0.3	0.94	6.9	47	0.2	0.76	1.2	0.31	24	12	170	380	3.2	3.2	2.5	<5	0.14	14	5.2	1.1	700	1.1
90	629	HC08-61	403.74	416.30	9	0.2	1	5.6	48	<0.1	0.56	0.86	0.15	30	13	150	210	3.1	3.5	2.6	<5	0.14	17	6.2	1.1	540	1.8
10 Pulps (Additional ABA)																											
91	648	HC07-21	148.35	152.67	1	0.2	0.33	20	71	<0.1	0.42	0.62	0.11	21	9.3	200	230	2.3	1.4	1.8	<5	0.16	12	1.2	0.69	420	3.3
92	649	HC07-42	61.11	65.26	1	0.1	1.3	27	39	0.5	2.9	5.8	0.13	12	42	190	130	7.3	4.7	5	<5	0.11	6.5	10	3.2	1900	1.8
93	650	HC07-42	319.28	322.17	1	0.2	1.5	1.8	50	0.5	1.1	0.5	0.04	36	13	120	230	4.9	4.9	4	<5	0.17	20	14	1.4	420	0.9
94	654	HC10-78	30.50	34.60	1	<0.1	1	9.4	14	<0.1	0.32	4.2	0.07	18	7.8	77	93	2.1	3.7	1.7	20	0.05	10	14	1.9	400	1.8
95	656	HC10-81	41.70	46.30	6	0.1	0.86	9.8	53	0.2	0.38	0.67	0.14	38	13	200	84	3.3	2.9	2.4	<5	0.17	22	5.2	1.2	540	0.89
96	659	HC10-81	132.55	136.30	6	0.3	0.77	10	42	0.2	0.98	1.7	0.21	23	7.1	120	96	2.4	2.6	1.8	<5	0.13	13	6	1.5	560	1.5
97	660	HC10-81	138.20	140.00	6	<0.1	1.5	25	65	0.4	0.24	0.11	0.06	48	23	66	91	4.9	5	3.7	<5	0.19	28	8.8	1.4	1600	0.87
98	661	HC10-81	140.00	145.65	6	<0.1	1.6	11	59	0.4	0.5	0.23	0.04	40	15	110	93	4.4	5	3.6	<5	0.17	22	10	1.2	590	1.2
99	662	HC10-81	150.50	154.75	6	0.1	1.6	6	48	0.4	0.58	0.33	0.04	36	16	120	66	4.1	5.2	3.6	<5	0.15	21	12	1.2	650	0.56
100	663	HC10-81	154.75	157.50	6	0.2	1.7	4.6	57	0.4	1.3	0.31	0.06	42	19	170	82	4.3	5.2	3.8	<5	0.17	25	11	1.3	530	1.9

SRK Project No: 1CY003.000

Trace Element Test Results for 100 Yellowhead-Harper Creek Pulp Samples - October 2011

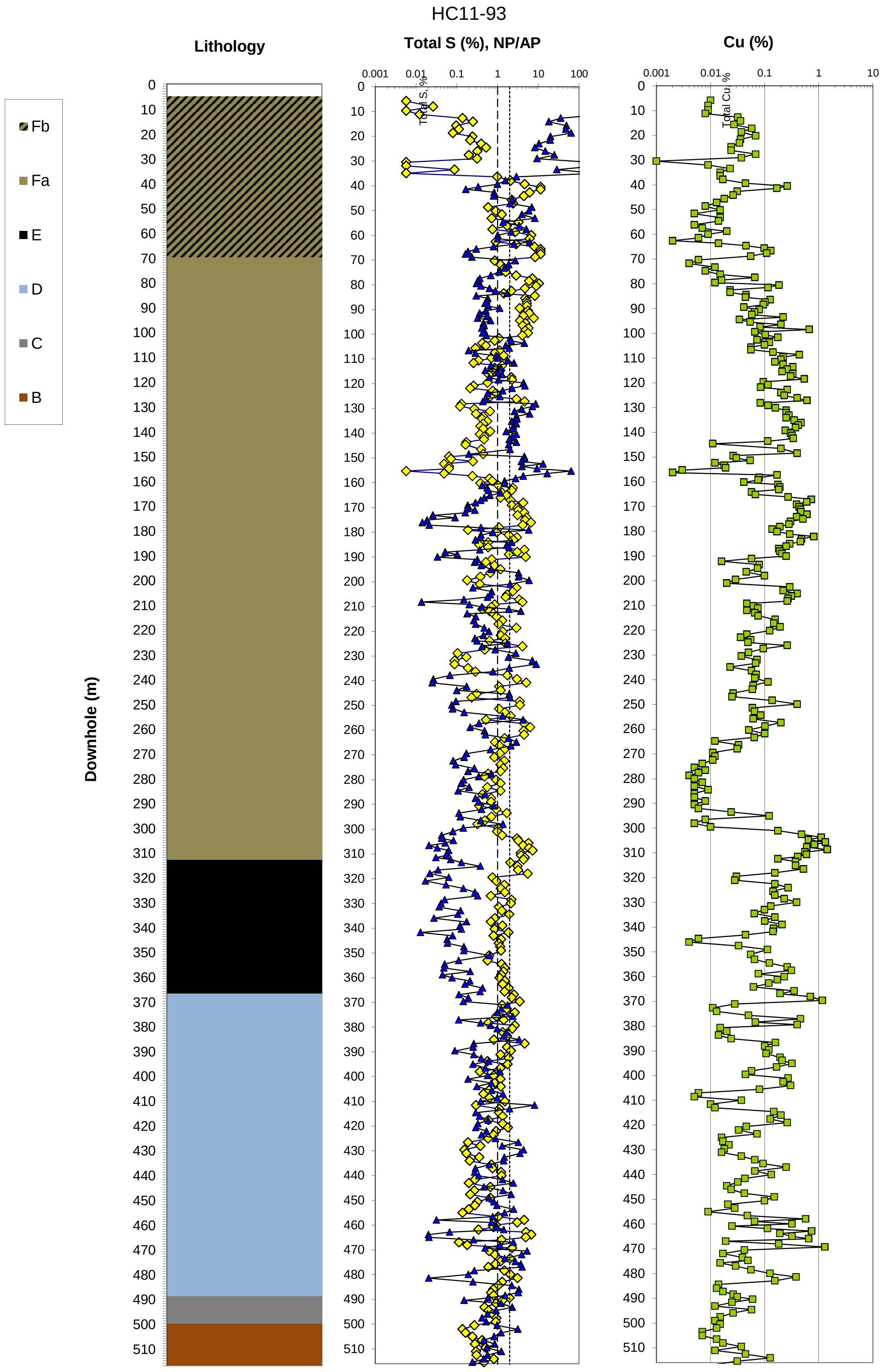
S. No.	Sample ID	Na	Nb	Ni	P	Pb	Rb	S	Sb	Sc	Se	Sn	Sr	Ta	Te	Th	Ti	Tl	U	V	W	Y	Zn	Zr	Ba	F (fusion method)
22 Pulps (Ore ABA)	**(Composite No.)**	**%**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**%**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**%**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**PPM**	**%**
1	3	0.092	0.04	9.1	270	4.7	2.9	1.6	0.16	2.8	7	0.5	38	<0.05	2	2.1	0.001	0.06	0.4	8	0.5	2.3	97	1.9	330	0.07
2	12	0.078	0.02	30	390	13	6.6	2.6	0.6	1	5	0.5	21	<0.05	0.64	7	0.001	0.08	1.2	<2	0.3	2	79	2.2	1100	0.11
3	19	0.11	<0.02	81	1200	13	4.6	1.5	1.3	5	4.6	0.2	55	<0.05	0.38	0.5	0.001	0.06	<0.1	32	0.1	3.6	180	0.94	260	0.11
4	29	0.079	<0.02	11	170	120	4.5	1.4	0.46	2.3	12	1.9	20	<0.05	1.5	2.7	0.001	0.04	0.3	16	<0.1	1.4	410	1.9	300	0.12
5	175	0.1	0.04	13	200	5.6	5.9	1.1	0.2	1.8	6.3	0.8	21	<0.05	1.7	4.7	0.001	0.06	0.4	12	0.5	1.4	140	2.4	500	0.09
6	187	0.075	0.06	31	170	12	4	4.4	2	0.6	11	3	32	<0.05	0.62	2.3	0.001	0.08	0.4	<2	0.3	1	260	4.1	280	0.03
7	191	0.085	0.1	52	1900	28	9.9	2.9	0.4	6.7	5.2	1.6	62	<0.05	0.2	1	0.027	0.12	0.3	66	0.4	4.8	230	1.8	320	0.08
8	215	0.084	<0.02	14	140	210	4.4	0.54	0.14	1.4	3.5	0.6	32	<0.05	0.52	5.6	0.001	0.04	0.5	8	0.2	1.5	340	1.4	470	0.07
9	262	0.059	0.02	130	3500	32	3.5	1.2	0.36	8.8	4.4	0.3	130	<0.05	0.38	4	0.007	0.04	0.4	88	<0.1	13	110	2.1	270	0.16
10	295	0.074	0.04	140	1200	27	1.5	0.68	0.18	11	6.1	0.8	39	0.1	0.54	1	0.005	0.06	0.1	110	0.4	6.4	140	1.3	190	0.16
11	303	0.068	0.02	22	130	360	3.6	2.5	0.5	0.6	5	1.1	17	<0.05	1.4	3.1	0.001	0.06	0.4	6	0.2	0.8	530	1.9	460	0.04
12	316	0.072	0.02	17	270	14	6.1	3.8	0.72	1.7	3.9	1.6	36	<0.05	0.16	3.4	0.001	0.1	0.5	12	<0.1	2.2	310	3.3	390	0.05
13	331	0.054	0.02	11	89	41	3.1	1.4	2.3	0.5	3.2	0.8	24	<0.05	0.78	3.2	0.001	0.02	0.3	4	<0.1	0.9	140	2.1	400	0.04
14	341	0.074	<0.02	6.9	200	15	5.4	1.1	0.24	2	3.4	0.8	30	<0.05	0.64	2.9	0.001	0.04	0.3	12	<0.1	1.3	100	3	430	0.06
15	350	0.057	0.06	9.7	44	54	2.3	1.4	6.3	0.3	2.7	0.6	13	<0.05	0.92	3	0.001	<0.02	0.2	4	<0.1	0.6	360	2.3	250	0.02
16	400	0.074	0.02	22	360	3.9	7.9	1.3	0.3	2.6	2.1	1.1	35	<0.05	0.16	6.2	0.002	0.08	1.3	20	0.2	2.5	140	2.8	960	0.05
17	444	0.085	0.04	55	790	19	7.3	2.1	0.44	2.6	5.5	1.5	42	0.05	0.2	5.1	0.002	0.14	0.6	18	0.5	4.1	350	3.1	310	0.11
18	448	0.08	0.06	51	780	25	8.9	3.3	0.58	3.2	6.6	1.3	18	<0.05	0.88	2.8	0.004	0.12	1.3	24	0.4	2.9	170	7.5	500	0.10
19	449	0.069	0.04	110	1200	23	3.4	2.4	0.48	7.8	4.9	1.2	33	<0.05	0.54	2.5	0.005	0.06	1.9	90	0.3	3.5	290	6.1	280	0.11
20	511	0.095	0.02	55	1200	27	7.9	1	0.38	5	4.9	0.6	72	0.05	0.52	1.7	0.001	0.1	0.4	34	0.6	4.5	160	3.4	260	0.11
21	582	0.075	0.06	22	240	20	9.9	1.1	0.54	1.9	2.8	0.7	24	0.05	0.22	8.4	0.005	0.18	0.9	14	0.7	2.4	93	2.8	520	0.05
22	604	0.12	0.02	35	290	24	5	0.58	0.2	2.3	3.7	0.6	37	<0.05	0.36	5.9	0.001	0.06	0.6	12	0.2	2.5	100	1.2	280	0.08
68 Pulps (Waste ABA)																										
23	39	0.082	0.06	92	490	34	14	3.5	0.44	4	6.4	0.9	20	<0.05	0.44	3.3	0.011	0.12	1	34	0.2	1.6	330	5.1	510	0.09
24	38	0.076	0.02	34	340	9.1	6.9	3.7	0.5	1.7	4.8	0.7	8.5	<0.05	0.44	4	0.002	0.06	0.6	6	<0.1	1.3	83	3.1	760	0.12
25	61	0.073	<0.02	130	490	170	2.8	7.8	1	7	12	1.3	21	<0.05	0.54	1.2	0.003	0.06	1.8	64	<0.1	1.8	630	4.7	140	0.07
26	64	0.084	0.08	41	380	16	17	1.7	0.42	2.6	2	0.6	27	<0.05	0.18	6.9	0.009	0.14	0.7	12	<0.1	1.9	110	3.2	810	0.08
27	76	0.066	0.04	32	380	2.9	6.9	1.6	0.26	1.4	3.4	0.6	10	<0.05	0.34	5.3	0.001	0.08	0.7	8	0.3	1.4	52	3.8	720	0.06
28	78	0.061	0.04	24	190	4.8	8.8	2.1	0.42	1.4	3.2	0.7	27	<0.05	0.16	5.3	0.001	0.1	0.5	4	0.2	1.2	50	3.7	740	0.04
29	82	0.055	0.06	23	250	9.9	9.9	1.2	1.3	1.5	2.6	0.8	13	<0.05	0.22	5.7	0.003	0.12	0.5	6	0.2	1.3	69	3.8	690	0.07
30	93	0.064	<0.02	95	570	19	2.3	2.1	0.48	5.7	6	0.7	24	<0.05	0.92	1.2	0.002	0.04	0.7	64	<0.1	1.7	200	3.4	200	0.07
31	116	0.058	0.02	25	200	1.9	6.3	0.62	0.22	1.2	1.4	0.5	31	<0.05	0.06	6.6	0.001	0.04	0.5	6	<0.1	1.7	42	2.4	620	0.04
32	117	0.059	0.02	17	260	12	7.4	0.56	0.56	1	1.2	0.5	18	<0.05	0.04	8.1	0.001	0.06	0.5	2	<0.1	1.6	60	2	620	0.04
33	135	0.058	0.02	21	260	13	6.3	0.88	0.2	1.3	1.9	0.3	22	<0.05	0.26	5.3	0.001	0.04	0.6	8	<0.1	1.6	68	3.3	820	0.05
34	148	0.077	0.02	22	250	7.7	5.7	0.84	0.18	1.8	1.9	0.5	37	<0.05	0.16	5	0.001	0.04	0.4	14	<0.1	1.7	77	2.3	340	0.04
35	159	0.1	0.72	65	930	3.6	4.8	0.28	0.18	4.2	0.7	1.2	75	0.05	0.1	4.3	0.24	0.06	0.5	50	0.8	7.5	39	2.6	330	0.04
36	169	0.085	0.1	24	6500	33	3.7	2.9	0.74	4.9	6.1	0.6	200	0.05	0.56	3	0.008	0.06	0.2	62	1.6	20	260	3.1	260	0.17
37	180	0.08	0.04	19	270	45	5.7	1	0.32	0.8	1.8	0.3	10	<0.05	0.3	7.1	0.001	0.06	0.8	4	0.3	1.3	86	3.2	680	0.05
38	194	0.083	0.24	170	2000	10	65	0.24	0.12	20	0.4	0.8	410	<0.05	0.06	1.7	0.43	0.44	<0.1	220	0.1	10	100	3	370	0.07
39	196	0.071	1.2	47	3800	21	25	2.7	0.34	8.3	6.3	1.6	140	0.05	0.58	3.2	0.42	0.28	0.4	120	0.5	13	95	6.2	540	0.13
40	200	0.094	0.28	17	410	6	2.3	0.06	0.08	4.8	0.2	1.1	19	<0.05	0.2	3.2	0.13	<0.02	0.5	50	0.3	5.1	33	3.5	230	0.06
41	202	0.073	0.08	56	990	21	1.1	0.78	0.26	9.3	2	1.4	85	<0.05	0.88	3.6	0.011	<0.02	1.1	84	0.1	6	120	2.8	40	0.07
42	207	0.081	1.4	38	5000	130	2.5	3.6	1.1	5.1	8.8	1.9	66	0.05	0.52	2.4	0.35	0.04	0.2	120	0.9	9.2	150	3.4	170	0.09
43	220	0.08	0.02	24	290	12	6.9	0.52	0.14	1.3	0.5	0.4	25	<0.05	0.08	8.6	0.001	0.06	0.8	4	<0.1	1.9	150	3	640	0.04
44	242	0.06	0.7	130	3100	11	10	0.2	0.16	22	0.5	0.5	410	0.05	0.22	2.6	0.45	0.1	0.2	230	0.2	11	110	5.6	58	0.07
45	243	0.054	0.48	190	2200	18	7.6	0.14	0.2	24	0.5	0.4	350	<0.05	0.2	1.1	0.49	0.08	0.1	230	0.2	7.7	120	4	23	0.07
46	249	0.069	0.24	38	760	19	2	1.7	0.28	3.2	10	1	57	<0.05	0.82	3.6	0.12	0.02	0.7	40	0.2	8	99	4.6	73	0.08
47	256	0.069	0.58	170	2000	11	0.3	2	0.56	6.1	2.9	1.5	71	<0.05	0.22	0.8	0.47	<0.02	0.2	150	0.8	5.6	58	2.3	<5	0.06
48	281	0.066	<0.02	50	350	12	7.2	1.8	0.22	1.8	2.8	0.3	19	<0.05	0.12	4	0.001	0.06	1.2	6	<0.1	1.5	70	3.6	660	0.06
49	286	0.063	2	25	6200	38	0.8	1.4	0.52	7.8	2.6	2	220	0.05	0.3	3	0.36	<0.02	0.3	140	0.4	13	130	4.7	82	0.12
50	287	0.061	0.8	40	3700	83	3.8	3	0.94	5.4	5.5	1.2	250	<0.05	1	3.5	0.15	0.04	0.2	84	0.1	16	240	4.2	440	0.10
51	319	0.062	0.02	20	290	16	8.3	0.84	0.18	1.2	0.9	0.7	24	<0.05	0.08	5	0.002	0.1	0.4	10	<0.1	1.7	62	3.4	580	0.04
52	321	0.065	0.04	19	410	22	7.6	2.2	0.66	1.8	2.3	0.8	25	<0.05	0.1	5.4	0.002	0.1	0.7	10	0.1	1.9	150	3.4	520	0.03
53	330	0.064	0.08	22	2900	15	11	0.44	0.76	9	0.9	0.6	160	<0.05	0.42	2.9	0.014	0.1	0.5	76	0.1	16	140	4.4	680	0.07
54	336	0.063	0.04	22	3000	8.3	12	0.66	0.58	9.2	1	0.7	170	<0.05	0.06	3	0.009	0.16	0.4	72	0.3	17	89	3.8	510	0.06
55	337	0.074	0.08	23	3300	6.3	15	0.28	0.12	9.6	0.7	0.6	220	<0.05	0.06	2.5	0.037	0.1	0.4	86	0.1	20	89	4.1	910	0.10
56	339	0.057	0.02	14	130	8.7	5.3	2.5	0.16	1	3.4	1	8.5	<0.05	0.38	4.9	0.001	0.06	0.5	8	<0.1	0.9	110	2.5	370	0.04
57	359	0.07	0.02	31	260	9.2	9.2	1.7	0.18	1.6	2.7	0.5	22	<0.05	0.16	5.8	0.001	0.06	0.5	10	0.1	1.5	86	3.5	690	0.04
58	366	0.07	<0.02	27	260	3.5	7	0.52	0.1	1.3	0.9	0.5	18	<0.05	0.06	7.6	0.001	0.04	0.7	12	<0.1	1.5	78	2.6	690	0.04
59	371	0.091	0.02	38	440	18	8.2	1.3	2	1.9	2.6	0.4	57	<0.05	0.58	6.7	0.001	0.06	0.7	8	0.6	2.3	180	3.9	680	0.06
60	382	0.082	0.06	53	1400	12	12	2.7	0.66	10	2.1	1	92	<0.05	0.28	2.4	0.013	0.14	0.3	96	0.2	4.9	120	2.5	320	0.05
61	387	0.11	0.04	48	1900	10	5.1	1.6	0.36	6.6	1.9	0.4	100	<0.05	0.12	0.5	0.003	0.04	<0.1	50	0.2	5.9	170	1.9	300	0.05
62	394	0.078	<0.02	24	560	2.1	8.4	0.16	0.08	1.7	0.4	0.5	21	<0.05	0.06	8.8	0.002	0.06	0.8	12	0.1	2.4	160	2.5	530	0.03
63	414	0.069	<0.02	27	390	5.9	9.1	1.6	0.56	1.4	3.6	0.9	9.5	<0.05	0.28	7.9	0.001	0.08	0.7	6	<0.1	1.7	64	4	760	0.05
64	416	0.08	<0.02	29	470	7.4	11	2.1	0.62	1.7	4	0.8	16	<0.05	2.1	9.2	0.001	0.1	1.1	4	<0.1	2	72	5	780	0.06
65	433	0.08	<0.02	26	320	5.1	8.8	0.5	0.7	3.1	0.7	0.6	39	<0.05	0.26	8.1	0.001	0.12	0.8	16	0.2	3.2	120	4.5	600	0.03

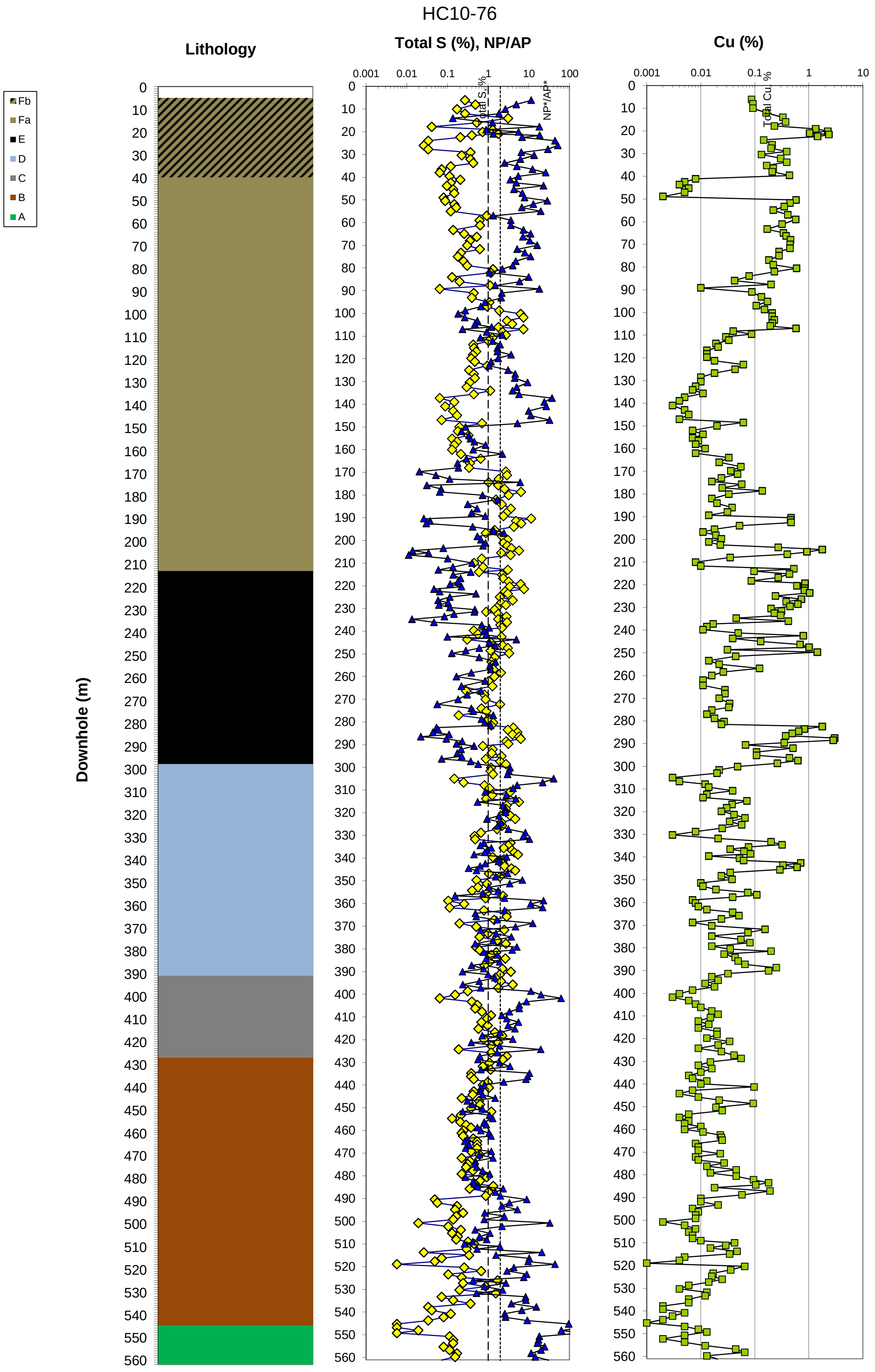
SRK Project No: 1CY003.000

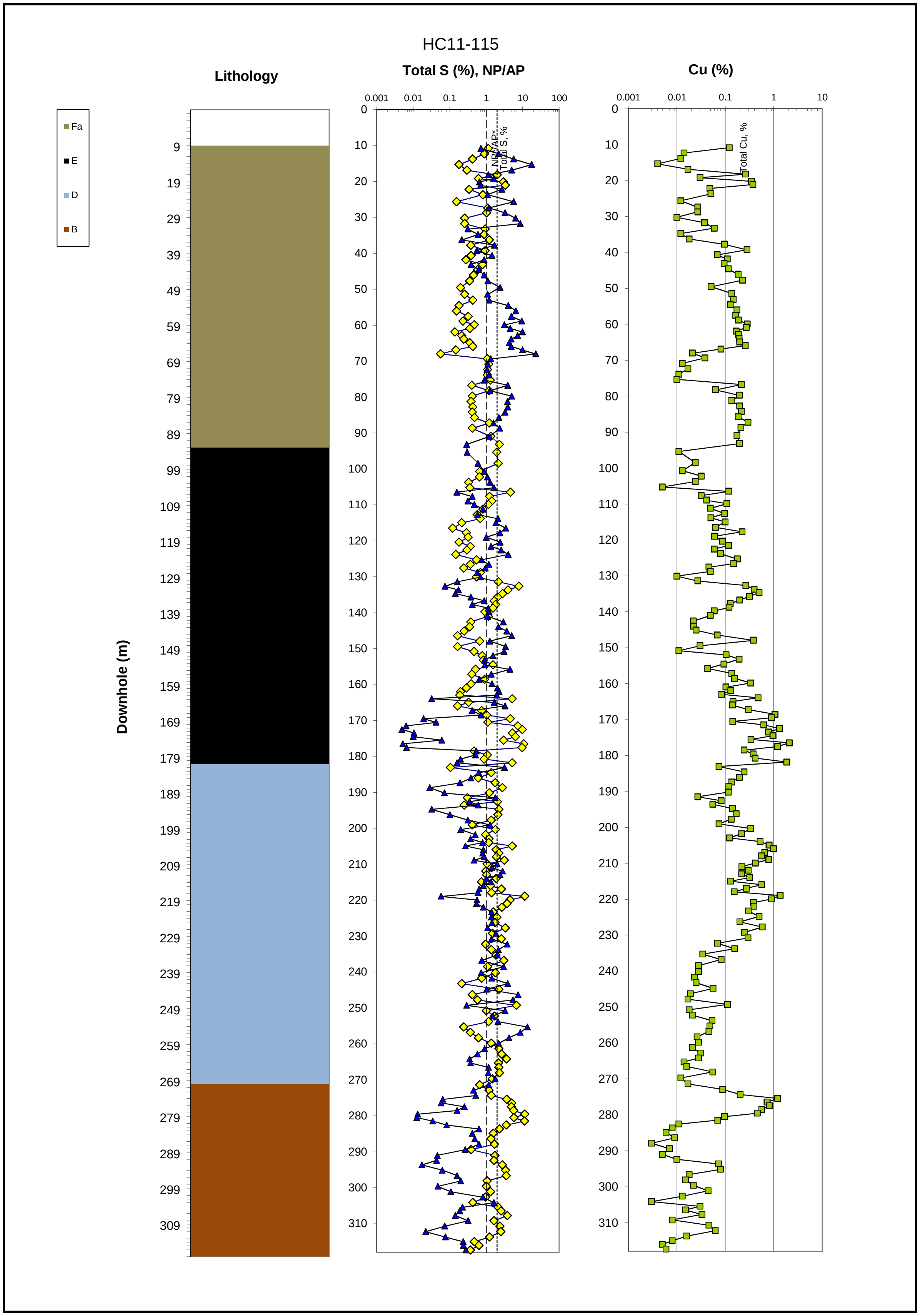
Trace Element Test Results for 100 Yellowhead-Harper Creek Pulp Samples - October 2011

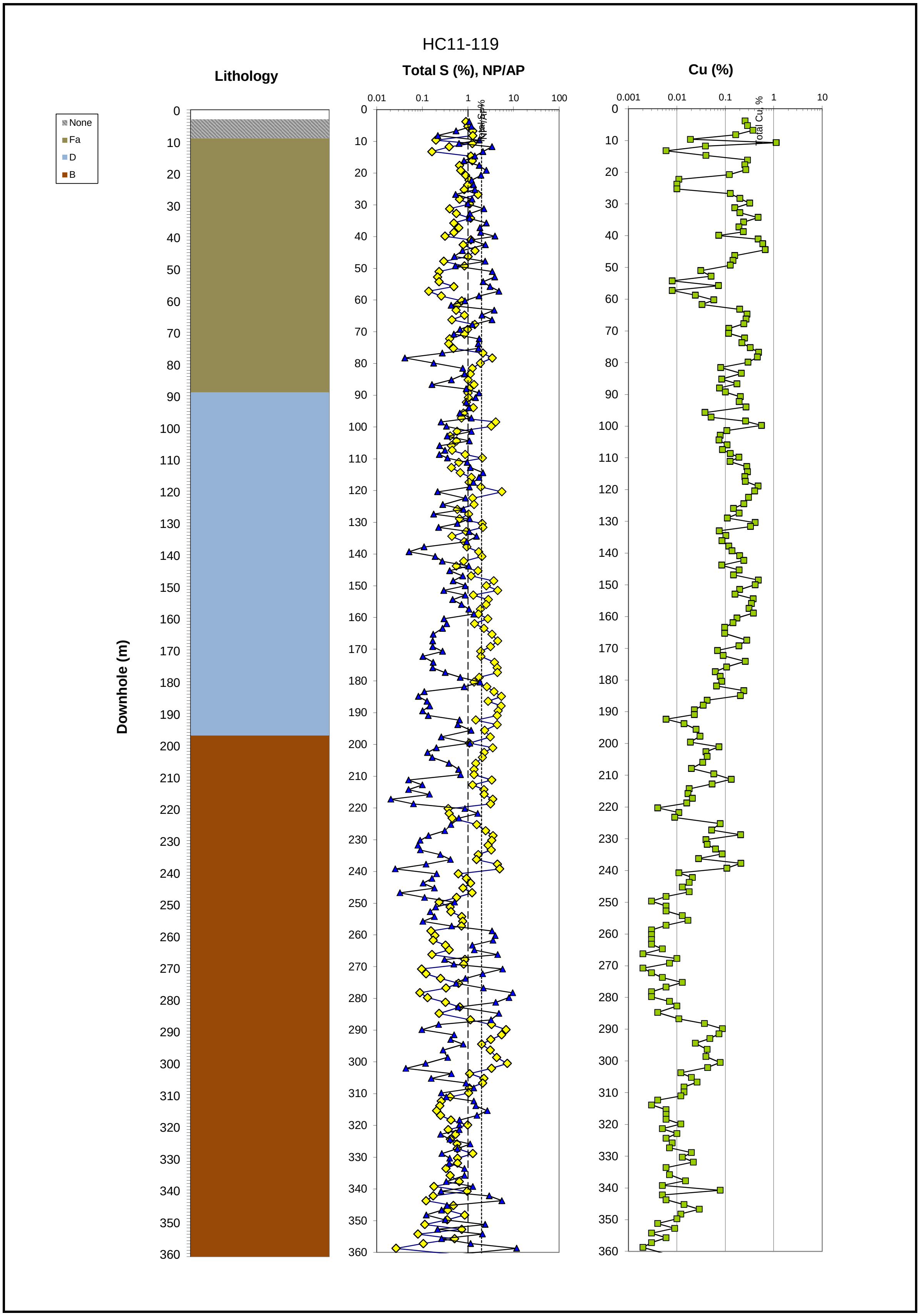
S. No.	Sample ID	Na	Nb	Ni	P	Pb	Rb	S	Sb	Sc	Se	Sn	Sr	Ta	Te	Th	Ti	Tl	U	V	W	Y	Zn	Zr	Ba	F (fusion method)
22 Pulps (Ore ABA)	**(Composite No.)**	**%**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**%**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**%**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**PPM**	**%**
66	453	0.079	0.04	26	300	3.6	9.7	1.1	0.4	1.3	2.6	0.5	11	<0.05	0.22	6.3	0.002	0.1	0.9	8	0.3	1.8	36	4.2	770	0.05
67	456	0.086	0.08	25	220	25	9.9	1.6	0.72	1.2	3.6	0.6	10	<0.05	0.38	5.6	0.004	0.12	0.8	8	0.3	1.4	210	4.5	640	0.03
68	464	0.11	0.04	29	270	1.9	14	0.46	1	1.7	0.7	0.5	20	<0.05	0.06	7.4	0.006	0.14	0.7	12	0.3	2.8	51	3.5	690	0.04
69	472	0.093	0.06	190	640	47	2.2	1.2	0.16	6.9	0.7	0.2	120	<0.05	0.48	0.5	0.002	0.02	0.2	56	<0.1	3	130	2.8	160	0.03
70	474	0.084	<0.02	260	570	19	1.6	1.1	0.4	11	1	0.2	58	<0.05	0.14	1	0.005	0.02	0.8	96	<0.1	2.3	110	3.1	74	0.03
71	487	0.086	0.02	16	510	49	6.6	2.4	0.34	3	3	0.6	20	<0.05	0.5	3	0.004	0.06	0.6	26	0.1	2	210	3.1	440	0.04
72	489	0.072	0.12	19	250	110	13	2.7	0.34	1.2	3.5	1.2	8.5	<0.05	0.42	5.4	0.012	0.14	0.9	8	0.4	1.5	200	2.5	460	0.03
73	493	0.091	0.04	47	590	3.1	7.9	0.9	0.08	1.8	1	0.3	9.5	<0.05	0.1	7.9	0.003	0.06	1.7	18	<0.1	2.1	140	3.4	1000	0.08
74	496	0.074	0.04	21	740	8.1	10	1.3	0.32	1.2	1.5	0.5	16	<0.05	0.04	6	0.003	0.1	0.9	8	0.2	2.3	70	1.9	540	0.04
75	497	0.089	0.06	43	450	11	13	2	0.2	2.1	1.7	0.9	23	<0.05	0.14	4.3	0.006	0.12	0.9	24	0.1	2.4	760	2.7	600	0.06
76	506	0.11	<0.02	110	470	6.2	4.7	2	0.44	5.9	1.3	0.3	110	<0.05	0.12	0.3	0.002	0.04	0.2	50	<0.1	3.3	87	2.1	250	0.05
77	534	0.098	<0.02	44	490	5.1	8.1	0.28	0.48	2.5	0.6	0.4	51	<0.05	0.1	11	0.001	0.08	1.6	18	0.3	3.3	130	2.7	640	0.07
78	540	0.076	0.02	150	2100	140	6.9	2.7	0.7	4.3	0.5	0.2	120	<0.05	0.1	1.1	0.002	0.06	<0.1	26	0.2	6.5	220	1.5	410	0.07
79	552	0.11	<0.02	20	270	34	5.7	0.36	0.28	1.4	2	0.3	40	<0.05	0.24	8	<0.001	0.04	0.7	6	0.1	2.5	55	2.1	300	0.05
80	553	0.092	<0.02	24	380	320	6.5	0.58	0.36	1.6	2.4	0.5	29	<0.05	0.24	9.6	0.001	0.06	0.9	10	0.1	2.6	420	2.4	310	0.07
81	566	0.074	0.06	24	160	58	15	0.56	0.24	1.6	0.9	0.5	9	<0.05	0.04	8	0.009	0.1	0.6	10	0.2	1.4	160	2.8	590	0.06
82	578	0.068	0.06	31	330	15	18	0.58	0.18	2.5	0.8	0.5	17	<0.05	0.08	8.6	0.011	0.14	0.9	14	0.2	2.5	82	3.8	680	0.07
83	585	0.073	0.08	36	340	9.1	12	0.56	0.16	1.9	1.2	0.4	10	0.1	0.14	8.3	0.006	0.14	0.9	16	0.6	2.4	100	3.9	710	0.06
84	589	0.088	0.08	25	290	38	15	1.3	0.26	2.5	1.4	0.6	21	0.05	0.14	5.8	0.01	0.18	0.7	16	0.5	2.5	400	4.3	630	0.05
85	599	0.11	0.02	140	790	120	3.1	1.3	0.32	6	1.6	0.3	110	<0.05	0.1	1.5	0.001	0.04	0.5	38	0.2	3.4	670	3.6	190	0.04
86	615	0.078	0.04	13	190	12	8	0.22	0.1	1.3	1.3	0.3	45	<0.05	0.16	8.9	0.001	0.08	0.5	6	0.2	2.7	51	1.6	410	0.04
87	618	0.085	0.04	25	460	15	9.5	0.62	0.14	2.6	1.1	0.8	35	<0.05	0.18	6.5	0.005	0.08	0.6	16	0.2	2.4	280	3.1	450	0.05
88	627	0.078	0.04	20	110	26	4.5	1.2	0.1	1.1	2.3	1.4	27	<0.05	0.2	5.4	0.001	0.04	0.4	6	0.1	1.9	130	1.4	200	0.03
89	628	0.072	0.02	22	260	12	6.8	0.76	0.1	1.4	1.1	0.6	31	<0.05	0.18	7.9	0.002	0.06	1	12	0.2	2.4	87	1.7	390	0.04
90	629	0.083	0.02	30	330	3.9	7.2	0.54	0.06	1.6	0.9	0.4	21	<0.05	0.06	9.1	0.002	0.06	1.2	16	0.1	2	85	1.8	490	0.05
10 Pulps (Additional ABA)																										
91	648	0.066	0.02	18	160	7	8.9	1.2	0.56	1.2	1.2	0.9	39	<0.05	0.12	6.2	0.001	0.08	0.4	6	0.2	1.6	30	3	560	0.07
92	649	0.06	0.02	160	940	8.6	5.6	2	0.42	4.2	0.9	0.3	77	<0.05	0.24	2.4	0.002	0.08	0.3	26	0.1	5.1	64	1.6	220	0.05
93	650	0.059	<0.02	27	220	3.5	8.5	1.7	0.16	1.6	2.4	0.5	37	<0.05	0.16	11	0.002	0.1	1.2	12	0.2	2.8	70	5.4	610	0.06
94	654	0.097	<0.02	9.7	270	7	4.5	0.26	0.16	12	0.8	0.6	240	<0.05	0.28	5	0.001	0.04	0.7	52	0.1	10	24	2.8	82	0.07
95	656	0.084	<0.02	33	260	39	9	0.44	0.1	1.5	0.5	0.3	18	<0.05	0.1	9.5	0.001	0.06	0.9	8	0.1	2.3	80	3.8	540	0.06
96	659	0.067	<0.02	16	160	45	6.7	0.62	0.06	0.9	0.6	0.2	30	<0.05	0.16	5.8	0.001	0.04	0.6	8	0.1	1.9	69	2	330	0.04
97	660	0.086	0.08	50	270	3.1	11	0.36	0.1	2	0.6	0.2	8.5	<0.05	0.1	11	0.013	0.06	0.6	14	0.2	1.8	95	1.8	930	0.06
98	661	0.07	0.08	37	470	10	8.7	0.66	0.1	1.4	0.7	0.3	13	<0.05	0.1	10	0.011	0.06	1.3	12	0.2	2.1	99	2.6	700	0.06
99	662	0.077	0.06	36	420	21	7.7	0.52	0.16	1.7	0.6	0.3	13	<0.05	0.06	9	0.003	0.06	1.1	16	<0.1	2	110	2.3	540	0.03
100	663	0.076	0.04	45	410	35	8.7	0.64	0.08	1.5	0.8	0.3	14	<0.05	0.4	9.9	0.002	0.06	1.2	14	0.1	2.1	120	2.7	720	0.04

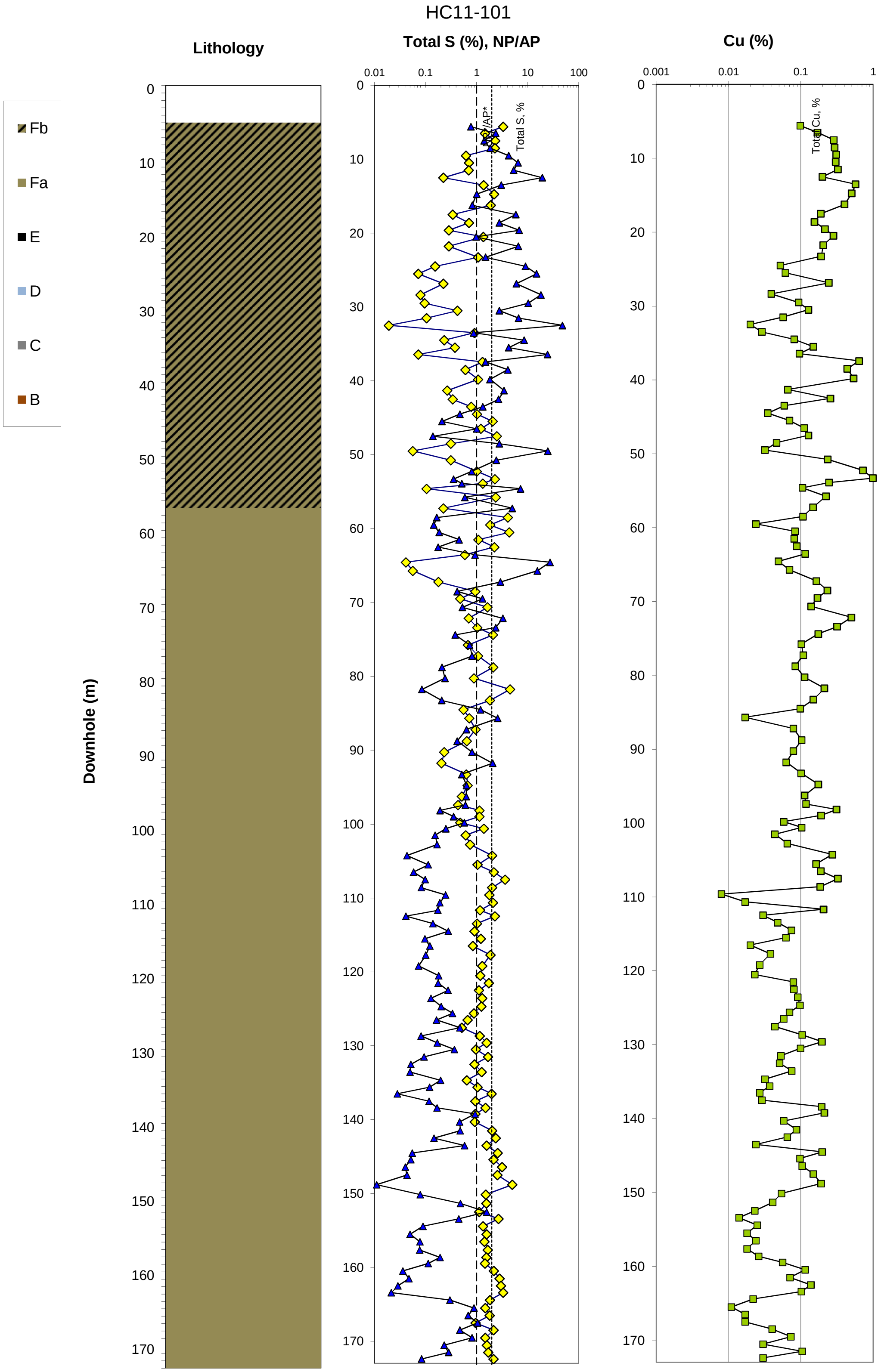
B3: Downhole Static Test Graphics

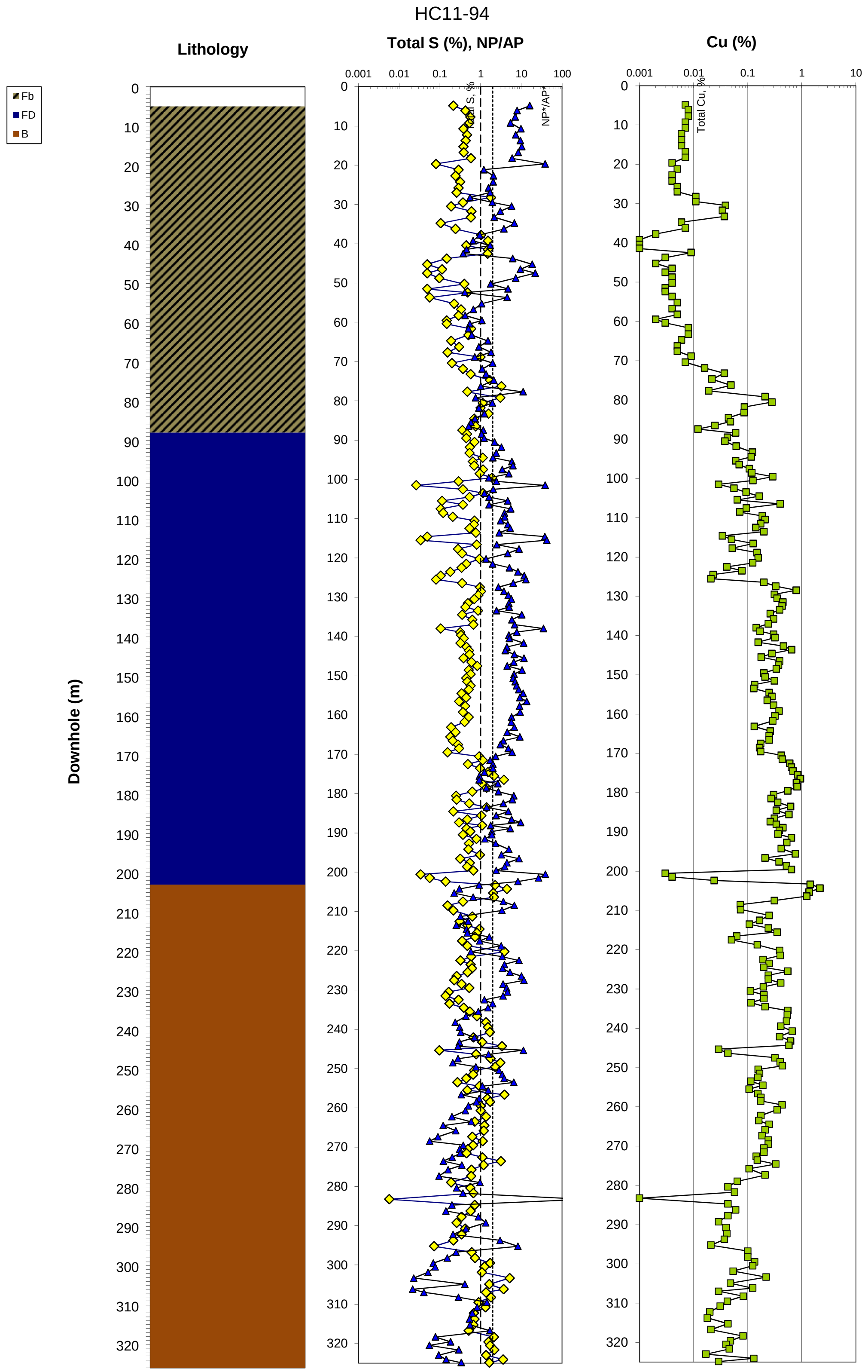

HC11-93
Lithology
Total S (%), NP/AP
Cu (%)
Fb
Fa
E
D
C
B
Downhole (m)
0
10
20
30
40
50
60
70
80
90
100
110
120
130
140
150
160
170
180
190
200
210
220
230
240
250
260
270
280
290
300
310
320
330
340
350
360
370
380
390
400
410
420
430
440
450
460
470
480
490
500
510
0.001
0.01
0.1
1
10
100
Total S, %
0.001
0.01
0.1
1
10
Total Cu, %


HC10-76
Lithology
Total S (%), NP/AP
Cu (%)
Fb
Fa
E
D
C
B
A
Downhole (m)
0
10
20
30
40
50
60
70
80
90
100
110
120
130
140
150
160
170
180
190
200
210
220
230
240
250
260
270
280
290
300
310
320
330
340
350
360
370
380
390
400
410
420
430
440
450
460
470
480
490
500
510
520
530
540
550
560
0.001
0.01
0.1
1
10
100
Total S, %
NP*/AP*
Total Cu, %


HC11-115
Lithology
Total S (%), NP/AP
Cu (%)
Fa
E
D
B
Downhole (m)
9
19
29
39
49
59
69
79
89
99
109
119
129
139
149
159
169
179
189
199
209
219
229
239
249
259
269
279
289
299
309
0.001
0.01
0.1
1
10
100
0
10
20
30
40
50
60
70
80
90
100
110
120
130
140
150
160
170
180
190
200
210
220
230
240
250
260
270
280
290
300
310
NP/AP*
Total S, %
0.001
0.01
0.1
1
10
Total Cu, %


HC11-119
Lithology
Total S (%), NP/AP
Cu (%)
None
Fa
D
B
Downhole (m)
0.01
0.1
1
10
100
0.001
10
Total S%
NP/AP
Total Cu, %
0
10
20
30
40
50
60
70
80
90
100
110
120
130
140
150
160
170
180
190
200
210
220
230
240
250
260
270
280
290
300
310
320
330
340
350
360


HC11-101
Lithology
Total S (%), NP/AP
Cu (%)
Fb
Fa
E
D
C
B
Downhole (m)
0
10
20
30
40
50
60
70
80
90
100
110
120
130
140
150
160
170
0.01
0.1
1
10
100
NP/AP*
Total S, %
0.001
0.01
0.1
1
Total Cu, %


HC11-94
Lithology
Total S (%), NP/AP
Cu (%)
Fb
FD
B
Downhole (m)
0
10
20
30
40
50
60
70
80
90
100
110
120
130
140
150
160
170
180
190
200
210
220
230
240
250
260
270
280
290
300
310
320
0.001
0.01
0.1
1
10
100
Total S, %
NP*/AP*
Total Cu, %

B4: Acid Base Accounting Results for Humidity Cells

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Table 1: ABA Test Results for project Harper Creek Waste Rock Humidity Cell Program 1CY003.001

Maxxam Sample No	HC ID	Sample ID	Paste pH	CO2	CaCO3 Equiv.	Total S	HCl Extractable Sulphur	Sulphide Sulphur (by diff.)	Acid Generation Potential	Mod. ABA Neutralization Potential	Fizz Rating	Net Neutralization Potential (Mod. ABA)	Neutralization Potential Ratio
		Units	pH Units	wt%	Kg CaCO3/T	wt%	wt%	wt%	Kg CaCO3/T	Kg CaCO3/T	N/A	Kg CaCO3/T	N/A
CL5451	HC-1	HC07-21 (148.35-152.67)	8.9	3.85	87.50	1.16	<0.01	1.16	36.3	46.1	SLIGHT	9.9	1.3
CL5452	HC-2	HC07-42 (319.28-322.17)	8.1	1.83	41.59	1.54	0.02	1.52	47.5	29.8	SLIGHT	-17.7	0.6
CL5453	HC-3	HC07-21 (69-46-83.47)	8.6	9.35	212.5	0.37	0.03	0.34	10.6	134.8	SLIGHT	124.2	12.7
CL5454	HC-4	HC07-21 (157.46-172.29)	8.6	10.49	238.6	0.46	0.02	0.44	13.8	130.3	SLIGHT	116.6	9.5
CL5455	HC-5	HC07-77 (40-45.00)	8.8	9.39	213.4	1.61	<0.01	1.61	50.3	209.0	STRONG	158.7	4.2
CL5456	HC-6	HC07-77 (53-58.00)	8.8	6.07	137.9	2.39	0.02	2.37	74.1	161.7	STRONG	87.6	2.2
CL5457	HC-7	HC07-81 (140-145.75)	9.2	0.81	18.41	0.44	<0.01	0.44	13.8	14.6	NONE	0.9	1.1
CL5458	HC-8	HC07-81 (150.5-154.75)	9.2	0.85	19.32	0.59	<0.01	0.59	18.4	19.0	NONE	0.6	1.0
CL5459	HC-9	HC07-27 (143.79-155.48)	8.7	1.96	44.55	3.03	0.01	3.02	94.4	44.0	SLIGHT	-50.4	0.5
CL5460	HC-10	HC07-22 (253.55-263.75)	8.7	3.18	72.27	1.71	0.01	1.70	53.1	34.5	SLIGHT	-18.6	0.6
CL5461	HC-10D	HC07-22D (253.55-263.75)	8.7	3.16	71.82	1.66	0.01	1.65	51.6	31.5	SLIGHT	-20.1	0.6
CL5462	HC-11	HC06-07 (185.19-196.23)	9.3	3.83	87.05	3.11	0.01	3.10	96.9	46.9	SLIGHT	-50.0	0.5
CL5463	HC-12	HC07-16 (210.72-220.81)	9.0	1.27	28.86	1.49	0.01	1.48	46.3	36.1	STRONG	-10.2	0.8
CL5464	HC-13	HC10-77 (173.00-176.00)	9.1	7.16	162.7	3.49	0.01	3.48	108.8	158.4	SLIGHT	49.7	1.5
CL5465	HC-14	HC07-14 (264.20-277.41)	7.6	1.18	26.82	1.05	<0.01	1.05	32.8	11.6	NONE	-21.2	0.4
CL5466	HC-15	HC07-21 (207.90-217.90)	7.1	3.88	88.18	2.58	0.01	2.57	80.3	20.4	NONE	-59.9	0.3
CL5467	HC-16	HC07-14 (373.80-380.09)	8.8	1.69	38.41	4.74	0.02	4.72	147.5	12.1	NONE	-135.4	0.1
CL5468	HC-17	HC10-77 (293.00-297.00)	8.6	1.47	33.41	1.96	<0.01	1.96	61.3	18.8	NONE	-42.5	0.3
Detection Limits			*0.5*	*0.02*		*0.02*	*0.01*	*0.02*	*0.3*			*0.1*	*0.1*
Maxxam SOP #			*7160*	*LECO*	*Calculation*	*LECO*	*7410*	*Calculation*	*Calculation*	*7150*	*7150*	*Calculation*	*Calculation*

Notes:

Total sulphur, total carbon and carbonate carbon (CO2; direct HCl method) by Leco furnace done at Acme Labs.

CO2 Analysis: A 0.2g pf pulp sample is digested with 6 ml of 1.8N HCl in a hot water bath of 70 ˚C for 30 minutes. The CO2 that evolves is trapped in a gas chamber that is controled with a stopcock, once the stopcock is is opened the CO2 gas is swept into the Leco analyser with a oxygen carrier gas. Leco then determines the CO2 as total-carbon which is calculated to total CO2.

References:

Reference for Mod ABA NP method (Maxxam SOP No. 7150): MEND Acid Rock Drainage Prediction Manual, MEND Project 1.16.1b (pages 6.2-11 to 17), March 1991.

Reference for HCl extractable Sulphate Sulphur (Maxxam SOP No. 7410): Modified ASTM D2492-02 Method (The S extracted is determined by analysing the extract for sulphate).

Sulphide Sulphur (by.diff.) = Total S - HCl Extractable Sulphur

Acid Generation Potential = Sulphide Sulphur (by diff.)*31.25

Net Neutralization Potential = (Modified ABA Neutralization Potential)-(Acid Generation Potential)

Neutralization Potential Ratio = (Neutralization Potential)/(Acid Generation Potential)

CaCO3 Equivalency = Carbonate Carbon (CO2)*(100/44)*10

B5: Trace Element Analysis Results for Humidity Cells

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Table 3: Ultratrace Metals Test Results for project Harper Creek Waste Rock Humidity Cell Program 1CY003.001

Maxxam Analytics 4606 Canada Way, Burnaby,

Table 3: Ultratrace Metals Test Result

Maxxam Sample No	HC ID	Sample ID	Mo	Cu	Pb	Zn	Ag	Ni	Co	Mn	Fe	As	U	Au	Th	Sr	Cd	Sb	Bi	V	Ca	P	La	Cr
		Units	ppm	ppm	ppm	ppm	ppb	ppm	ppm	ppm	%	ppm	ppm	ppb	ppm	ppm	ppm	ppm	ppm	ppm	%	%	ppm	ppm
CL5451	HC-1	HC07-21 (148.35-152.67)	0.32	307	9.21	35.5	382	16.9	10.0	517	2.61	21.2	0.3	7.7	2.7	43.5	0.14	0.35	0.56	3	1.00	0.019	3.6	68.3
CL5452	HC-2	HC07-42 (319.28-322.17)	0.36	235	3.53	71.8	344	24.5	14.2	457	4.94	1.8	1.0	1.7	8.1	35.2	0.06	0.11	0.88	11	0.54	0.023	8.3	64.7
CL5453	HC-3	HC07-21 (69-46-83.47)	5.21	319	16.5	180	343	25.4	19.2	898	4.68	13.7	0.5	4.4	2.6	204	0.59	0.31	0.60	77	2.85	0.328	29.1	60.1
CL5454	HC-4	HC07-21 (157.46-172.29)	1.59	219	8.78	95.7	237	25.5	18.7	999	4.84	4.0	0.4	2.0	3.1	204	0.18	0.18	0.15	69	2.75	0.331	29.7	41.0
CL5455	HC-5	HC07-77 (40-45.00)	3.26	233	23.9	82.0	348	84.4	40.0	1010	7.09	58.9	0.1	9.2	1.6	163	0.22	0.17	5.11	107	5.54	0.228	9.7	204
CL5456	HC-6	HC07-77 (53-58.00)	2.72	339	12.4	69.0	188	80.6	53.5	941	8.99	17.8	<0.1	8.9	1.5	219	0.13	0.22	0.70	165	5.48	0.251	14.9	216
CL5457	HC-7	HC07-81 (140-145.75)	0.48	86.9	8.09	87.7	114	36.4	15.3	637	4.06	17.0	1.2	<0.2	5.1	12.5	0.05	0.03	1.04	10	0.29	0.043	8.4	84.4
CL5458	HC-8	HC07-81 (150.5-154.75)	0.50	71.9	20.4	121	273	39.9	23.0	605	4.59	14.2	1.1	<0.2	5.1	14.3	0.05	0.03	0.73	13	0.39	0.039	9.1	81.9
CL5459	HC-9	HC07-27 (143.79-155.48)	7.05	2250	21.7	153	2070	51.8	23.6	541	6.14	63.5	1.5	12.4	2.1	17.5	0.78	0.29	3.99	21	0.80	0.067	3.3	69.2
CL5460	HC-10	HC07-22 (253.55-263.75)	1.52	273	9.85	102	404	32.7	19.2	612	4.15	15.8	0.5	4.7	2.8	26.7	0.28	0.08	1.99	5	0.65	0.025	3.9	61.7
CL5461	HC-10D	HC07-22D (253.55-263.75)	1.16	290	8.66	88.4	402	32.1	17.9	558	4.00	16.1	0.5	3.5	2.7	24.2	0.22	0.09	1.86	4	0.61	0.028	4.0	58.7
CL5462	HC-11	HC06-07 (185.19-196.23)	2.66	441	59.1	558	1450	131	37.1	1040	9.60	77.4	1.0	10.4	0.6	24.0	2.07	0.30	1.82	66	0.97	0.056	0.9	196
CL5463	HC-12	HC07-16 (210.72-220.81)	0.68	268	4.03	54.4	135	159	50.2	779	6.92	29.5	0.1	1.3	0.9	56.9	0.13	0.29	0.38	106	1.94	0.182	5.2	442
CL5464	HC-13	HC10-77 (173.00-176.00)	3.30	684	380	1020	1410	56.2	34.0	863	6.93	42.4	<0.1	11.0	0.7	102	5.53	0.59	1.71	22	3.52	0.244	4.5	85.0
CL5465	HC-14	HC07-14 (264.20-277.41)	4.67	637	30.1	71.3	680	17.2	10.2	148	2.21	29.8	0.6	11.5	4.5	9.6	0.19	0.22	0.62	4	0.23	0.024	6.4	58.0
CL5466	HC-15	HC07-21 (207.90-217.90)	0.37	488	8.43	110	1090	15.9	19.7	379	5.82	50.6	0.5	4.1	3.8	7.1	0.20	0.12	4.89	6	0.10	0.015	4.6	62.2
CL5467	HC-16	HC07-14 (373.80-380.09)	0.46	5550	14.2	244	3590	32.2	50.6	233	5.37	112	0.4	34.8	1.8	37.5	1.41	1.28	3.72	4	0.16	0.019	3.1	105
CL5468	HC-17	HC10-77 (293.00-297.00)	0.32	768	112	285	968	15.0	11.4	135	2.21	338	0.5	63.2	4.3	13.6	1.10	0.59	1.58	2	0.38	0.011	5.5	75.3
QAQC																								
Duplicates																								
CL5454 Dup		HC07-21 (157.46-172.29)	1.52	213	8.17	89.3	232	24.7	17.9	973	4.67	3.8	0.4	1.5	2.9	194	0.17	0.23	0.14	68	2.66	0.326	27.4	37.7
References																								
STD DS8			14.36	114.07	137.75	333	1819	39.6	8	636	2.62	25.7	3	107.7	7.5	66.7	2.49	4.86	7.21	43	0.74	0.089	15.7	125.5
STD DS8 True Values			13.44	110.00	123.00	312.0	1690	38.1	7.5	615	2.46	26	2.8	107.0	6.89	67.7	2.38	4.80	6.67	41.1	0.70	0.080	14.6	115.0
STD OREAS45CA			0.9	521.63	21.69	63.1	282	258	90.8	962	16.52	3.4	1.2	38.2	7.3	16.3	0.11	0.11	0.2	215	0.43	0.041	17	782.8
STD OREAS45CA True Values			1.00	494.00	20.00	60	275	240.0	92.0	943	15.69	3.8	1.2	43	7.0	15.1	0.10	0.13	0.19	215	0.43	0.039	15.9	709.0
Detection Limits			0.01	0.01	0.01	0.1	2	0.1	0.1	1	0.01	0.1	0.05	0.2	0.1	0.5	0.01	0.02	0.002	2	0.01	0.001	0.5	0.5
Method Blank							<2							<0.2										
Method Blank			<0.01	<0.01	<0.01	<0.1		<0.1	<0.1	<1	<0.01	<0.1	<0.1		<0.1	<0.5	<0.01	<0.02	<0.02	<2	<0.01	<0.001	<0.5	<0.5
Acme Method			*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*

Maxxam

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Table 3: Ultratrace Metals Test Res for project Harper Creek Waste Rock Humidity Cell Program 1CY003.001

Maxxam Sample No	HC ID	Mg %	Ba ppm	Ti %	B ppm	Al %	Na %	K %	W ppm	Sc ppm	Tl ppm	S %	Hg ppb	Se ppm	Te ppm	Ga ppm
CL5451	HC-1	0.86	35.0	<0.001	<20	0.19	0.012	0.11	0.2	1.4	0.05	1.32	<5	1.1	0.21	0.6
CL5452	HC-2	1.45	30.0	0.001	<20	1.73	0.010	0.16	0.1	1.8	0.10	1.63	<5	2.3	0.06	4.7
CL5453	HC-3	1.82	306	0.010	<20	0.92	0.025	0.21	<0.1	10.4	0.14	0.43	27	0.5	0.48	2.9
CL5454	HC-4	2.06	135	0.003	<20	0.78	0.020	0.17	<0.1	10.3	0.19	0.54	35	0.4	<0.02	2.5
CL5455	HC-5	3.31	6.5	0.005	<20	3.17	0.014	0.02	<0.1	13.2	<0.02	1.61	<5	4.1	0.74	13.5
CL5456	HC-6	3.51	2.0	0.125	<20	4.28	0.007	<0.01	<0.1	14.5	<0.02	2.29	6	3.7	0.12	16.3
CL5457	HC-7	1.13	38.7	0.003	<20	1.59	0.014	0.13	<0.1	1.6	0.03	0.51	<5	0.5	0.35	4.3
CL5458	HC-8	1.33	38.3	0.001	<20	1.97	0.019	0.16	<0.1	1.8	0.03	0.68	<5	0.5	0.05	5.2
CL5459	HC-9	1.91	38.6	0.003	<20	1.78	0.016	0.19	<0.1	3.3	0.07	3.01	7	5.8	0.80	5.5
CL5460	HC-10	0.94	64.2	<0.001	<20	0.31	0.016	0.13	0.2	1.6	0.05	1.91	<5	3.0	0.11	1.3
CL5461	HC-10D	0.89	66.7	<0.001	<20	0.29	0.015	0.11	0.2	1.5	0.04	1.82	<5	2.9	0.12	1.2
CL5462	HC-11	4.44	9.4	0.003	<20	3.69	0.011	0.04	<0.1	7.9	<0.02	3.17	<5	6.1	0.28	11.7
CL5463	HC-12	3.51	2.7	0.259	<20	3.56	0.017	<0.01	1.4	4.1	<0.02	1.52	<5	2.3	0.19	13.2
CL5464	HC-13	2.88	23.0	0.005	<20	1.96	0.017	0.08	<0.1	3.0	0.04	3.51	44	10.8	0.64	6.9
CL5465	HC-14	0.69	32.7	0.001	<20	0.46	0.014	0.12	0.2	0.9	0.04	1.14	<5	1.9	0.26	1.4
CL5466	HC-15	0.84	24.8	<0.001	<20	0.23	0.008	0.09	<0.1	1.2	0.06	2.59	9	3.3	0.36	1.5
CL5467	HC-16	0.30	37.5	<0.001	<20	0.20	0.012	0.11	<0.1	0.8	0.06	4.83	11	11.2	0.60	1.1
CL5468	HC-17	0.30	42.4	<0.001	<20	0.22	0.010	0.15	<0.1	0.7	0.05	1.93	8	3.1	0.18	0.7
QAQC																
Duplicates																
CL5454 Dup		1.99	128	0.003	<20	0.70	0.018	0.16	<0.1	9.9	0.18	0.53	27	0.5	<0.02	2.2
References																
STD DS8		0.64	316.4	0.111	<20	0.96	0.09	0.43	2.4	2.6	6	0.18	202	5.4	5.44	5
STD DS8 True Values		0.645	279.0	0.113	2.6	0.93	0.0883	0.41	3.0	2.3	5.40	0.1679	192	5.23	5.00	4.7
STD OREAS45CA		0.15	177.7	0.11	<20	3.57	0.014	0.07	<0.1	45.9	0.1	0.03	26	0.3	0.05	20.1
STD OREAS45CA True Values		0.14	164.0	0.128		3.59	0.0075	0.0717		39.7	0.07	0.021	30	0.5	0.06	18.4
Detection Limits		0.01	0.5	0.001	20	0.01	0.001	0.01	0.05	0.1	0.02	0.02	5	0.1	0.02	0.1
Method Blank													<5			
Method Blank		<0.01	<0.5	<0.001	<20	<0.01	<0.001	<0.01	<0.1	<0.1	<0.02	<0.02		<0.1	<0.02	<0.1
Acme Method		*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*	*1F*

Table 4: Mercury on solids CVAF Method Project Harper Creek Waste Rock Humidity Cell Program 1CY003.001

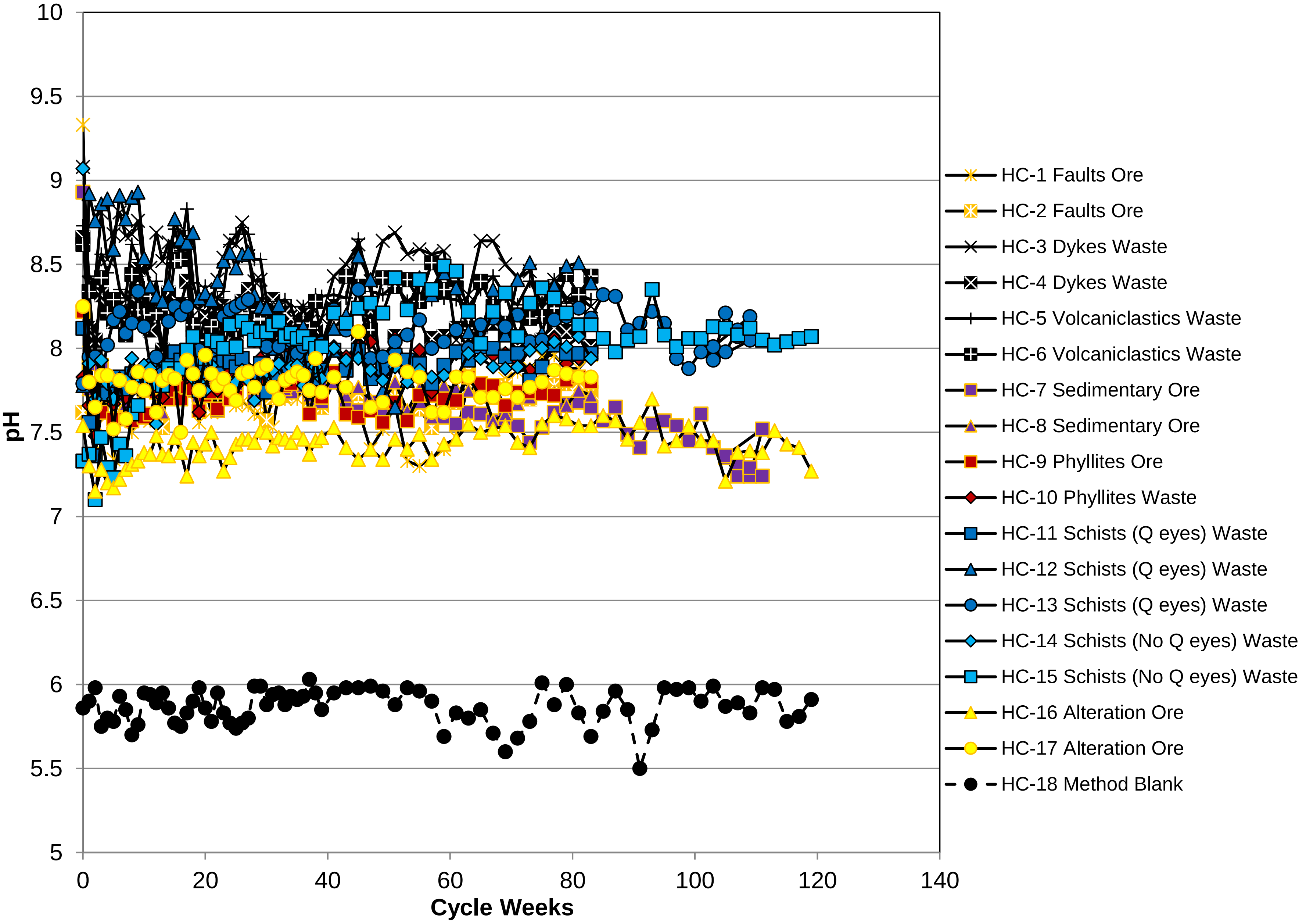
Maxxam Sample No	Sample ID	Hg
	Units	mg/kg
CL5451	HC07-21 (148.35-152.67)	<0.01
CL5452	HC07-42 (319.28-322.17)	<0.01
CL5453	HC07-21 (69-46-83.47)	0.03
CL5454	HC07-21 (157.46-172.29)	0.03
CL5455	HC07-77 (40-45.00)	<0.01
CL5456	HC07-77 (53-58.00)	<0.01
CL5457	HC07-81 (140-145.75)	<0.01
CL5458	HC07-81 (150.5-154.75)	<0.01
CL5459	HC07-27 (143.79-155.48)	<0.01
CL5460	HC07-22 (253.55-263.75)	<0.01
CL5461	HC07-22D (253.55-263.75)	<0.01
CL5462	HC06-07 (185.19-196.23)	<0.01
CL5463	HC07-16 (210.72-220.81)	<0.01
CL5464	HC10-77 (173.00-176.00)	0.03
CL5465	HC07-14 (264.20-277.41)	<0.01
CL5466	HC07-21 (207.90-217.90)	<0.01
CL5467	HC07-14 (373.80-380.09)	<0.01
CL5468	HC10-77 (293.00-297.00)	<0.01
QAQC		
Duplicate		
CL5451	HC07-21 (148.35-152.67)	<0.01
Detection Limits		0.01
Maxxam SOP #		CVAF
Method Blank		<0.01

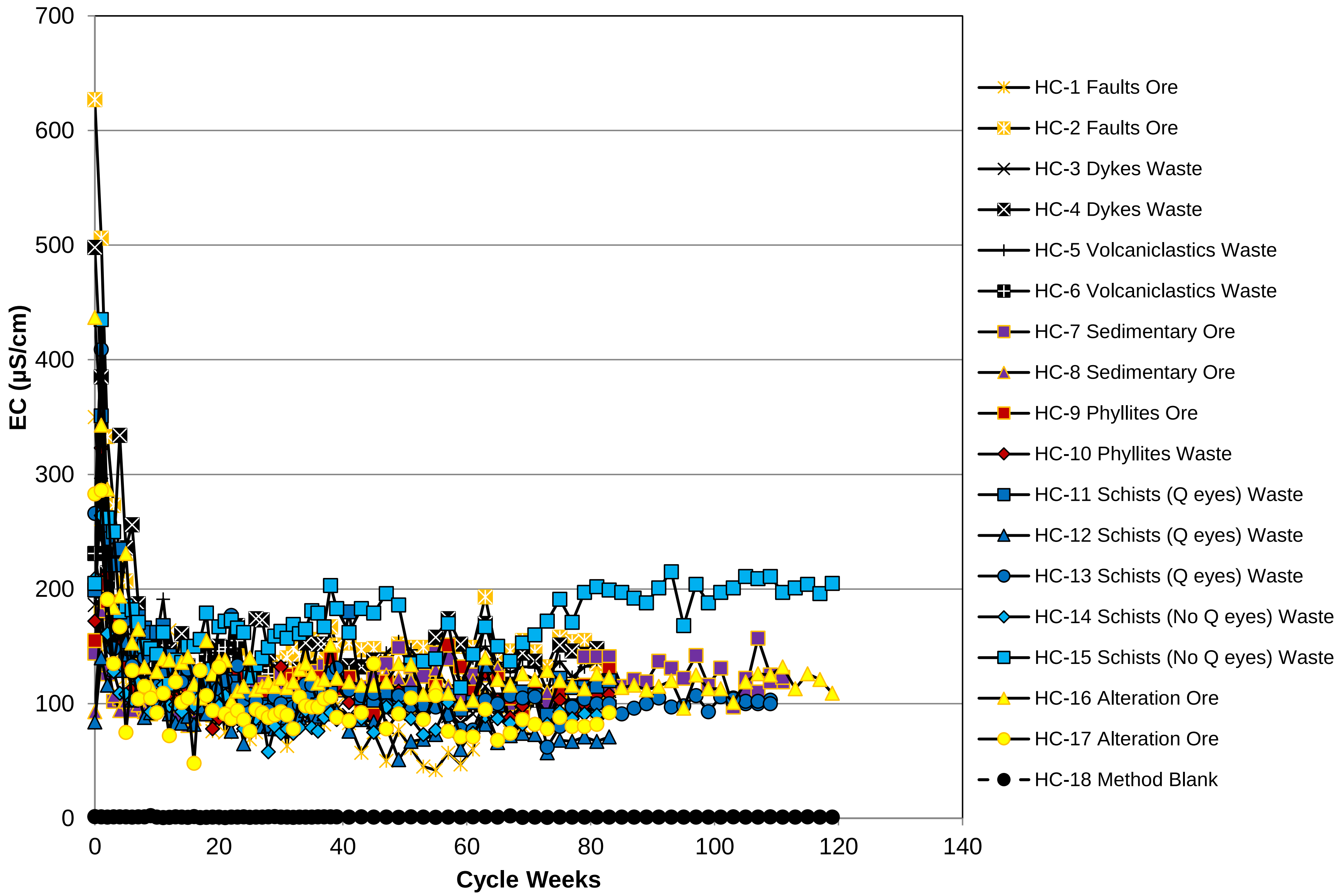
Table 5: Fluoride on solids for Project Harper Creek Waste Rock Humidity Cell Program 1CY003.001

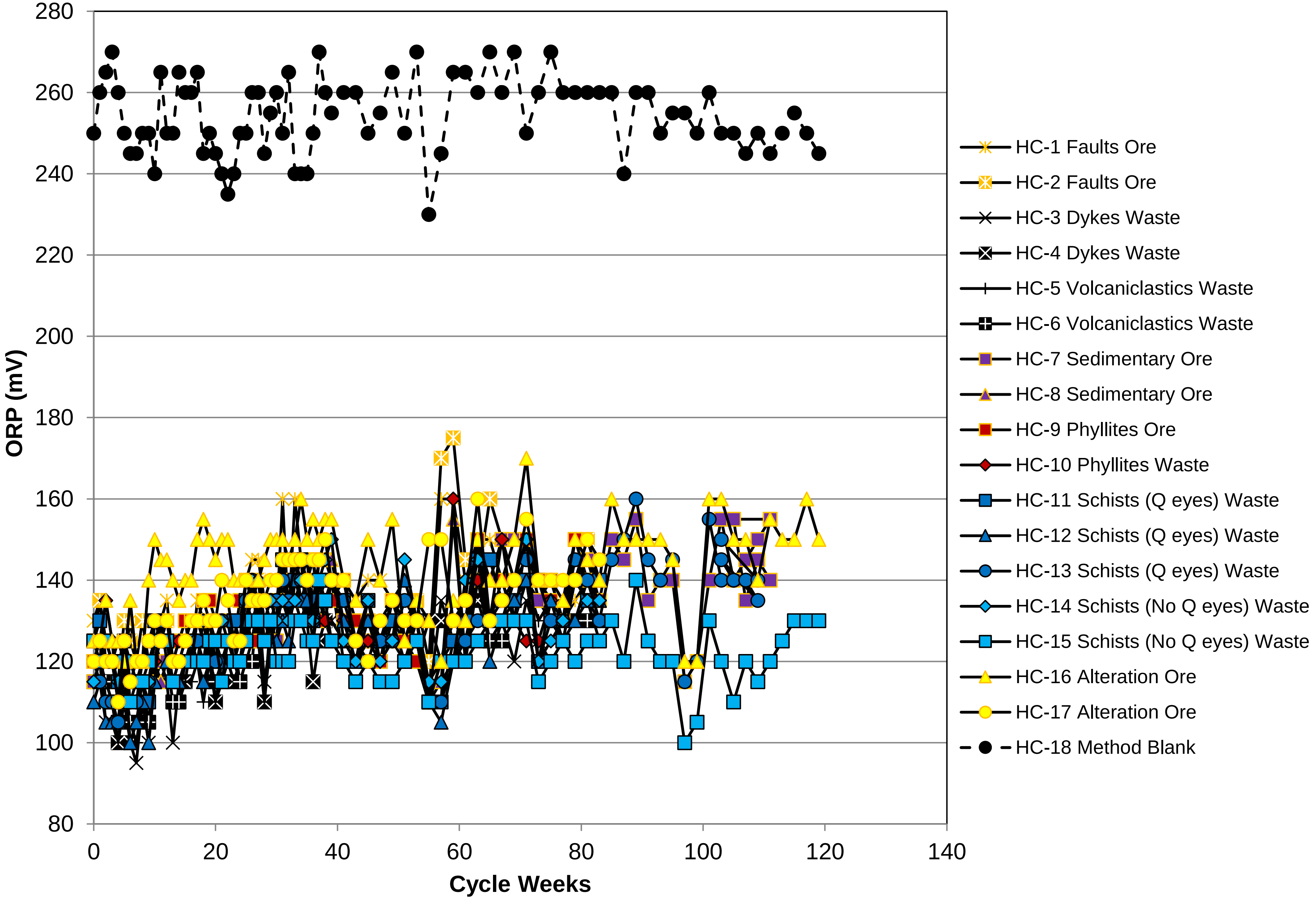
Maxxam Sample No	Sample ID	F
	Units	%
CL5451	HC07-21 (148.35-152.67)	0.02
CL5452	HC07-42 (319.28-322.17)	0.02
CL5453	HC07-21 (69-46-83.47)	0.05
CL5454	HC07-21 (157.46-172.29)	0.05
CL5455	HC07-77 (40-45.00)	0.04
CL5456	HC07-77 (53-58.00)	0.05
CL5457	HC07-81 (140-145.75)	0.01
CL5458	HC07-81 (150.5-154.75)	0.02
CL5459	HC07-27 (143.79-155.48)	0.05
CL5460	HC07-22 (253.55-263.75)	0.01
CL5461	HC07-22D (253.55-263.75)	<0.01
CL5462	HC06-07 (185.19-196.23)	0.04
CL5463	HC07-16 (210.72-220.81)	0.04
CL5464	HC10-77 (173.00-176.00)	0.08
CL5465	HC07-14 (264.20-277.41)	0.02
CL5466	HC07-21 (207.90-217.90)	0.02
CL5467	HC07-14 (373.80-380.09)	0.01
CL5468	HC10-77 (293.00-297.00)	0.02
QAQC		
Reference		
STD STSD-1		0.08
STD STSD-1 True Value		0.095
Detection Limits		0.01
Acme Method		*G803*
Method Blank		<0.01

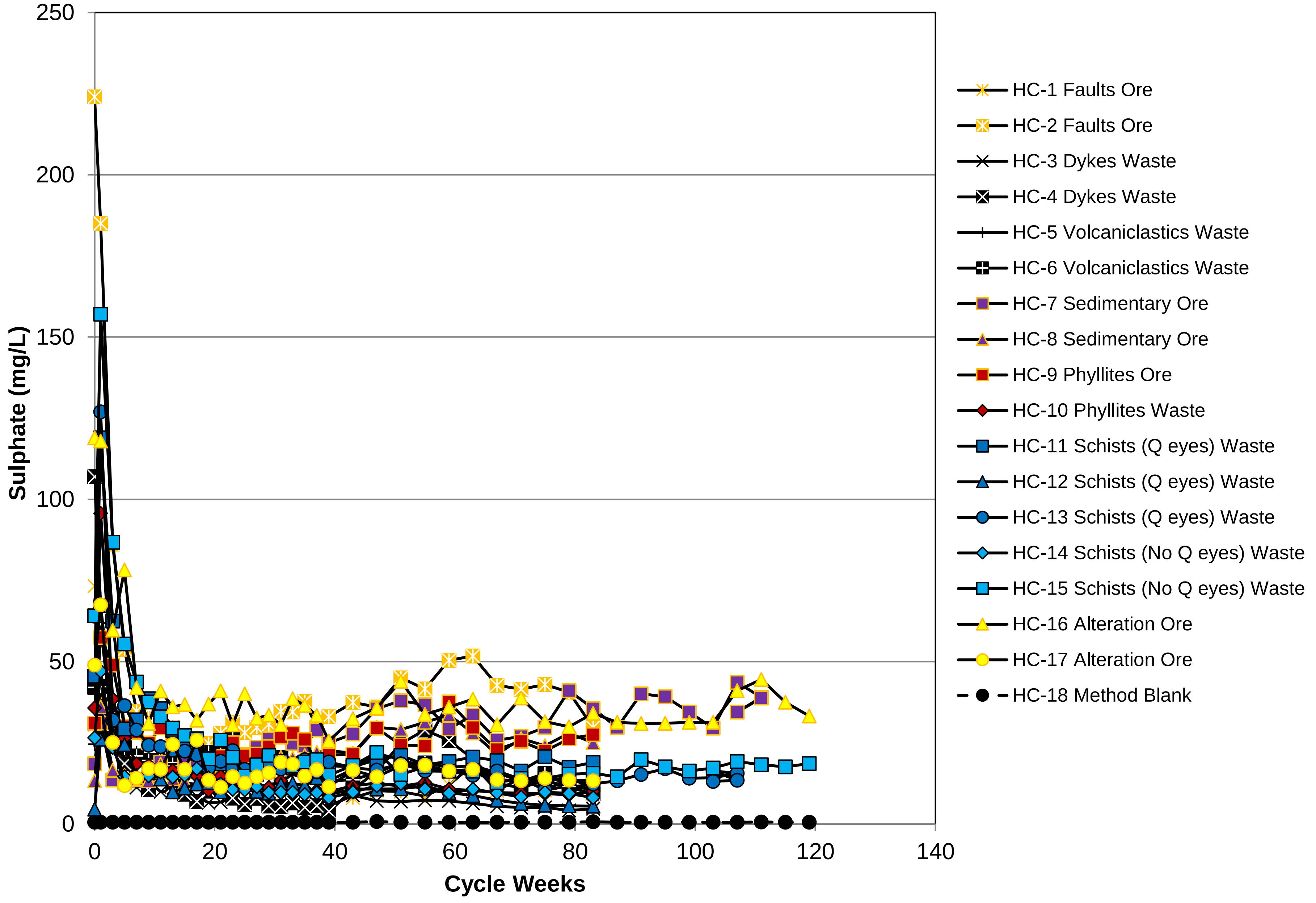
Appendix C: Kinetic Test Results

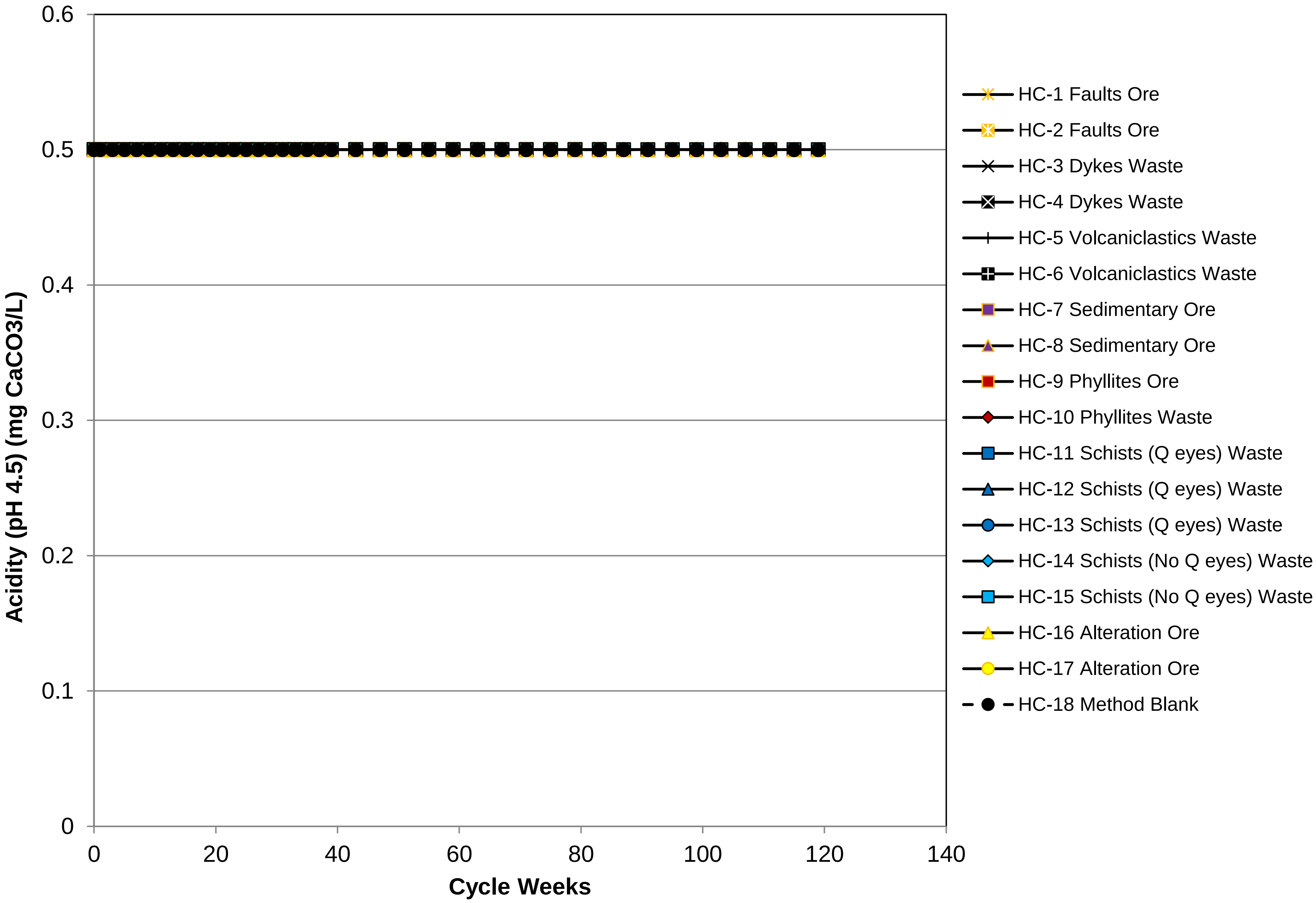
C1: Waste Rock Humidity Cells


10
9.5
9
8.5
8
7.5
7
6.5
6
5.5
5
pH
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

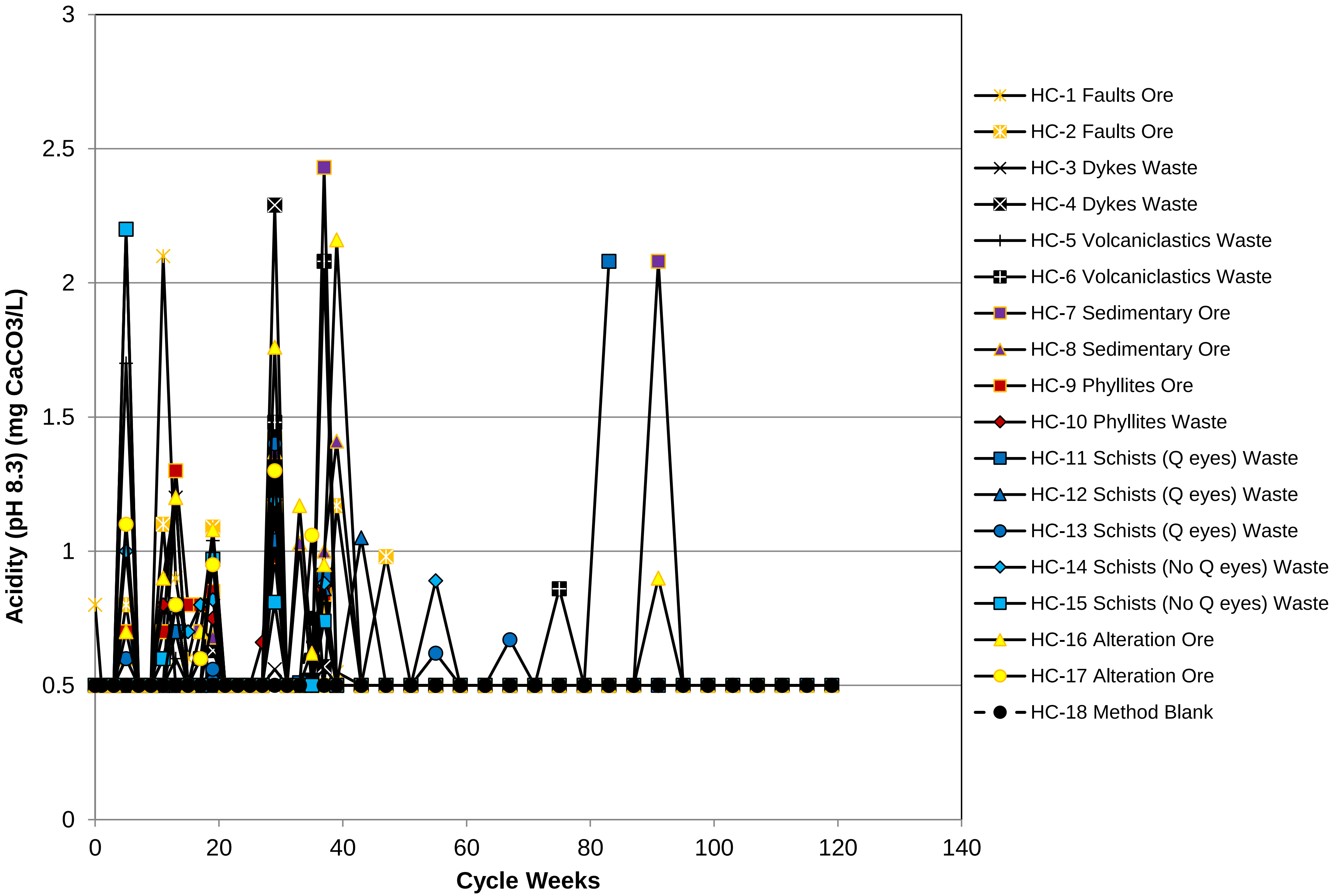

700
600
500
400
300
200
100
0
EC (µS/cm)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

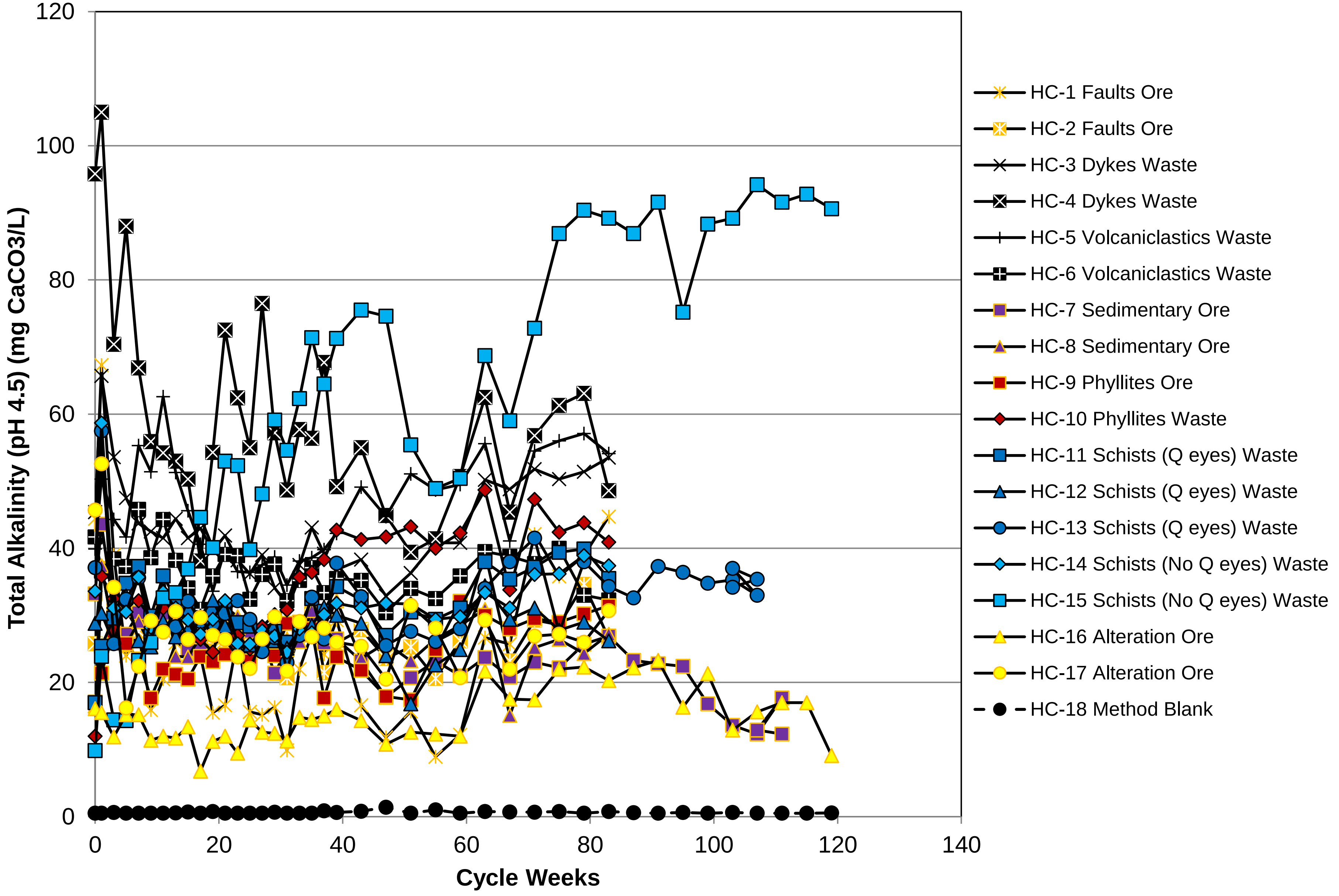

ORP (mV)
280
260
240
220
200
180
160
140
120
100
80
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

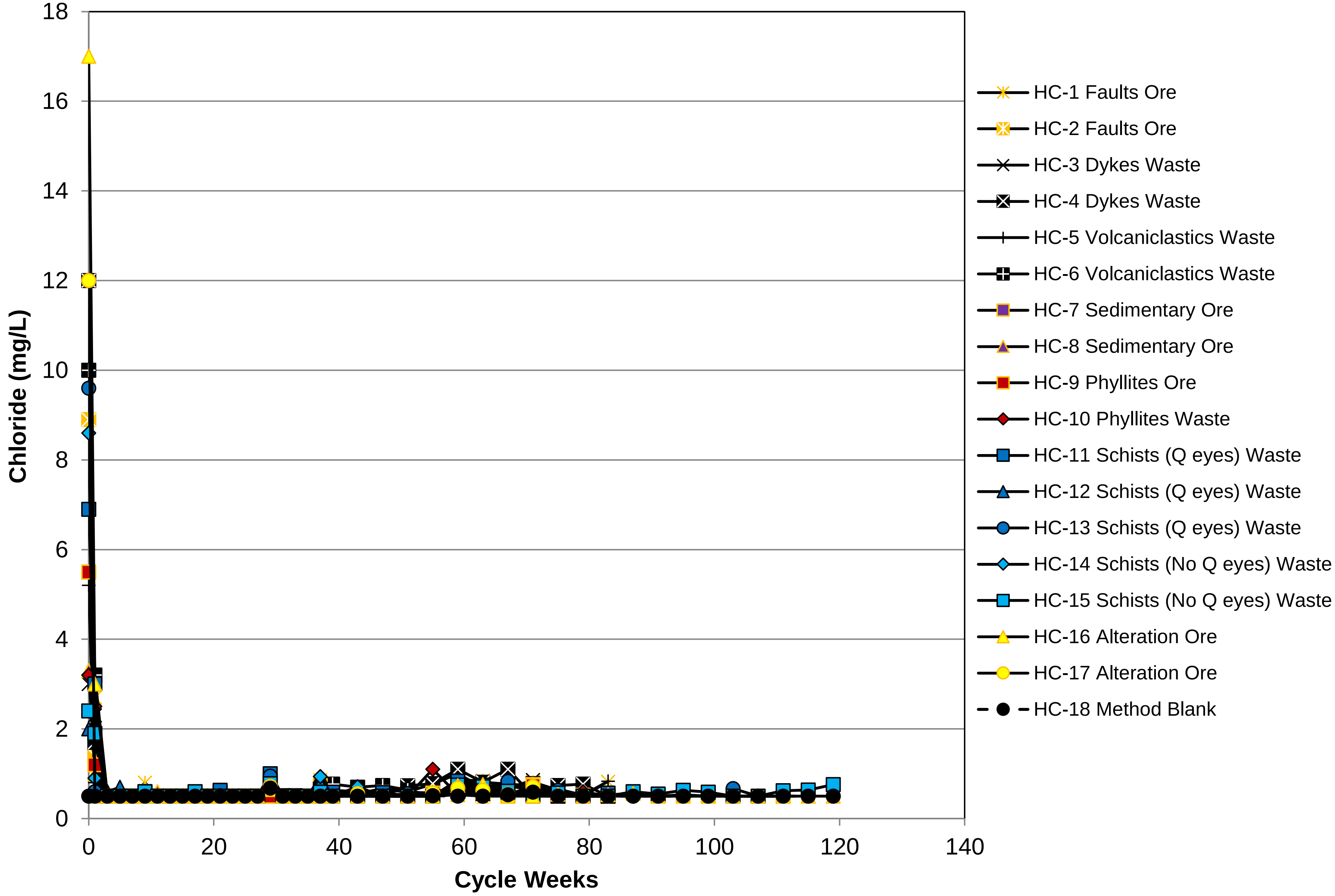

250
200
150
100
50
0
Sulphate (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

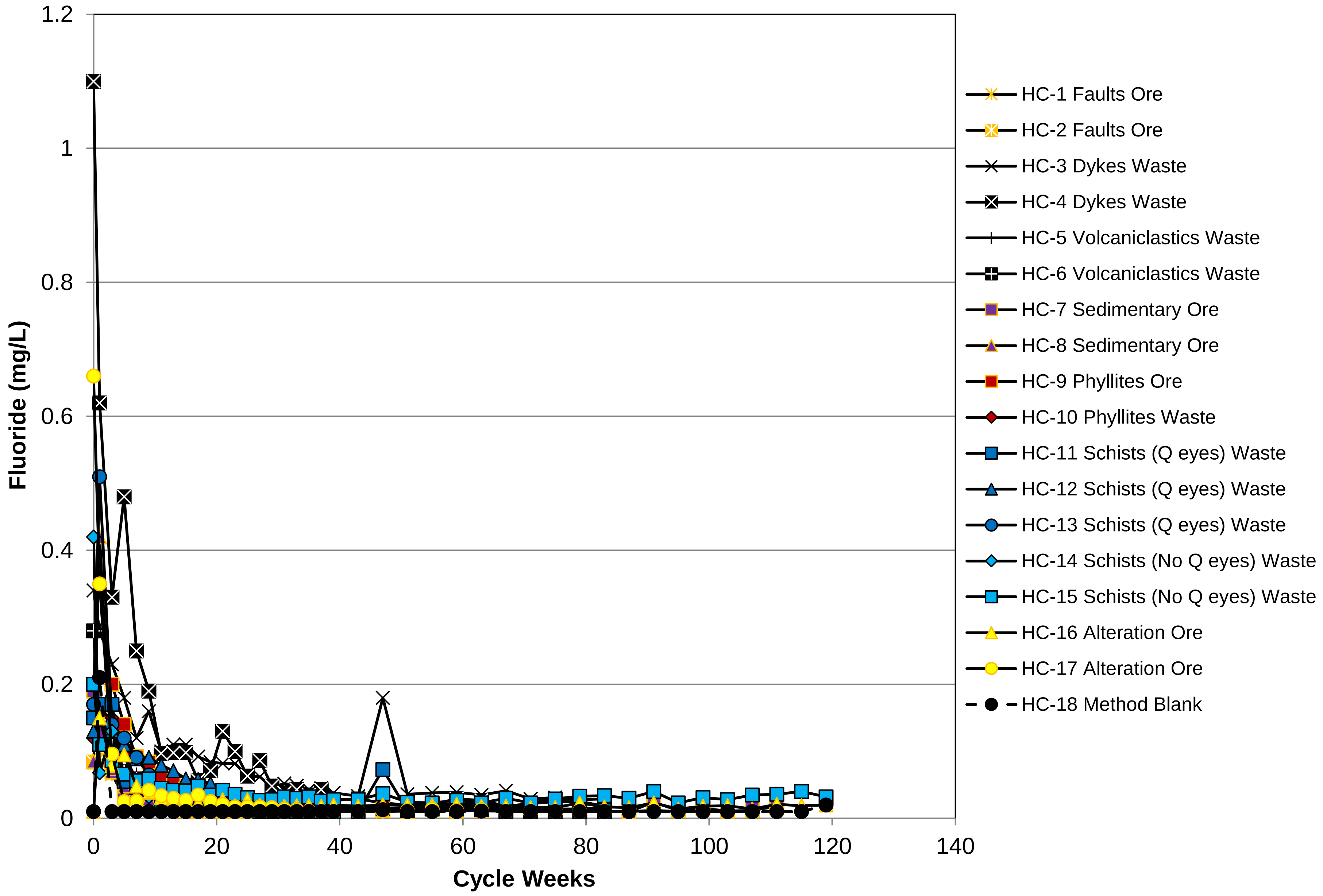


0.6
0.5
0.4
0.3
0.2
0.1
0
Acidity (pH 4.5) (mg CaCO3/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

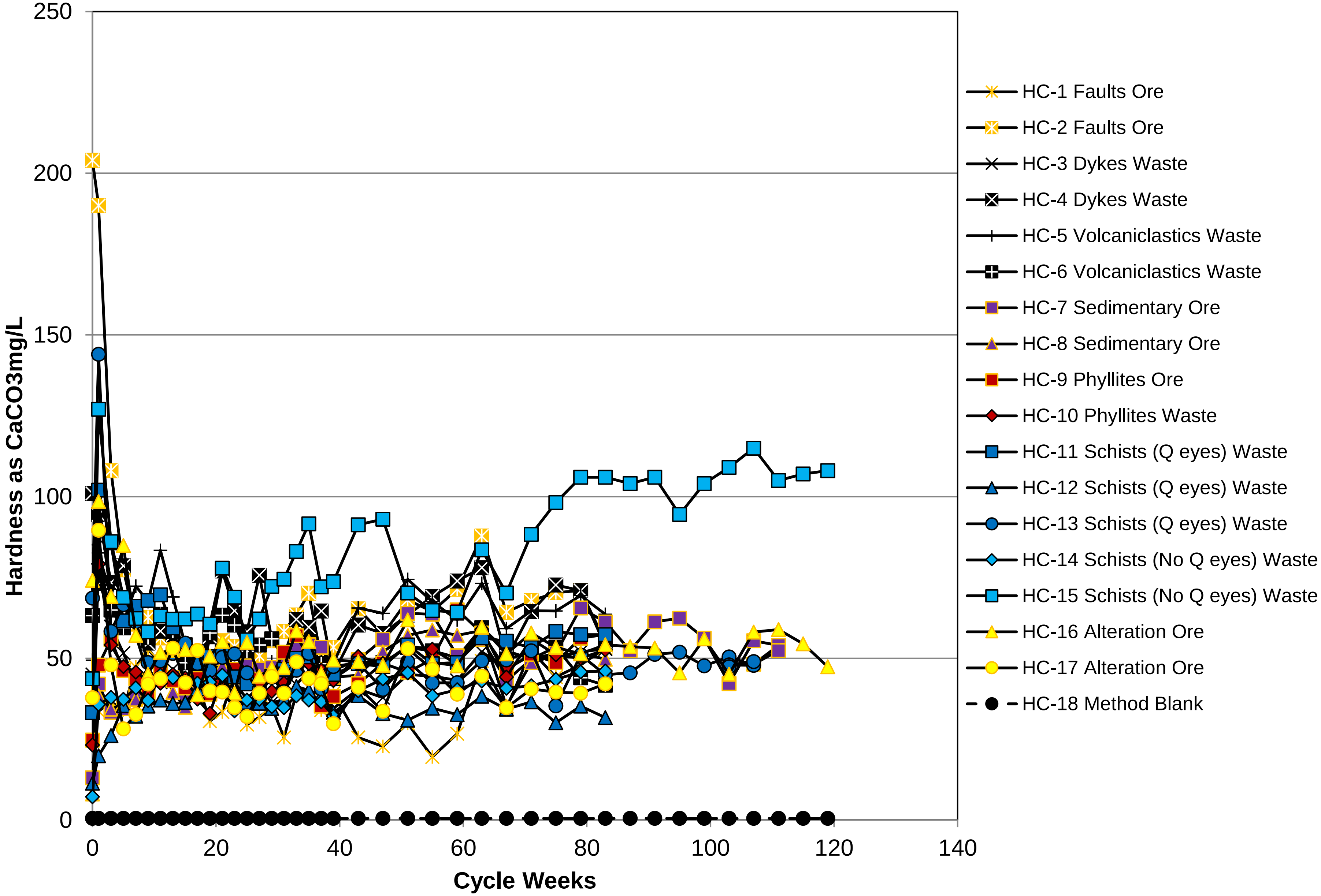

Acidity (pH 8.3) (mg CaCO3/L)
0
0.5
1
1.5
2
2.5
3
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

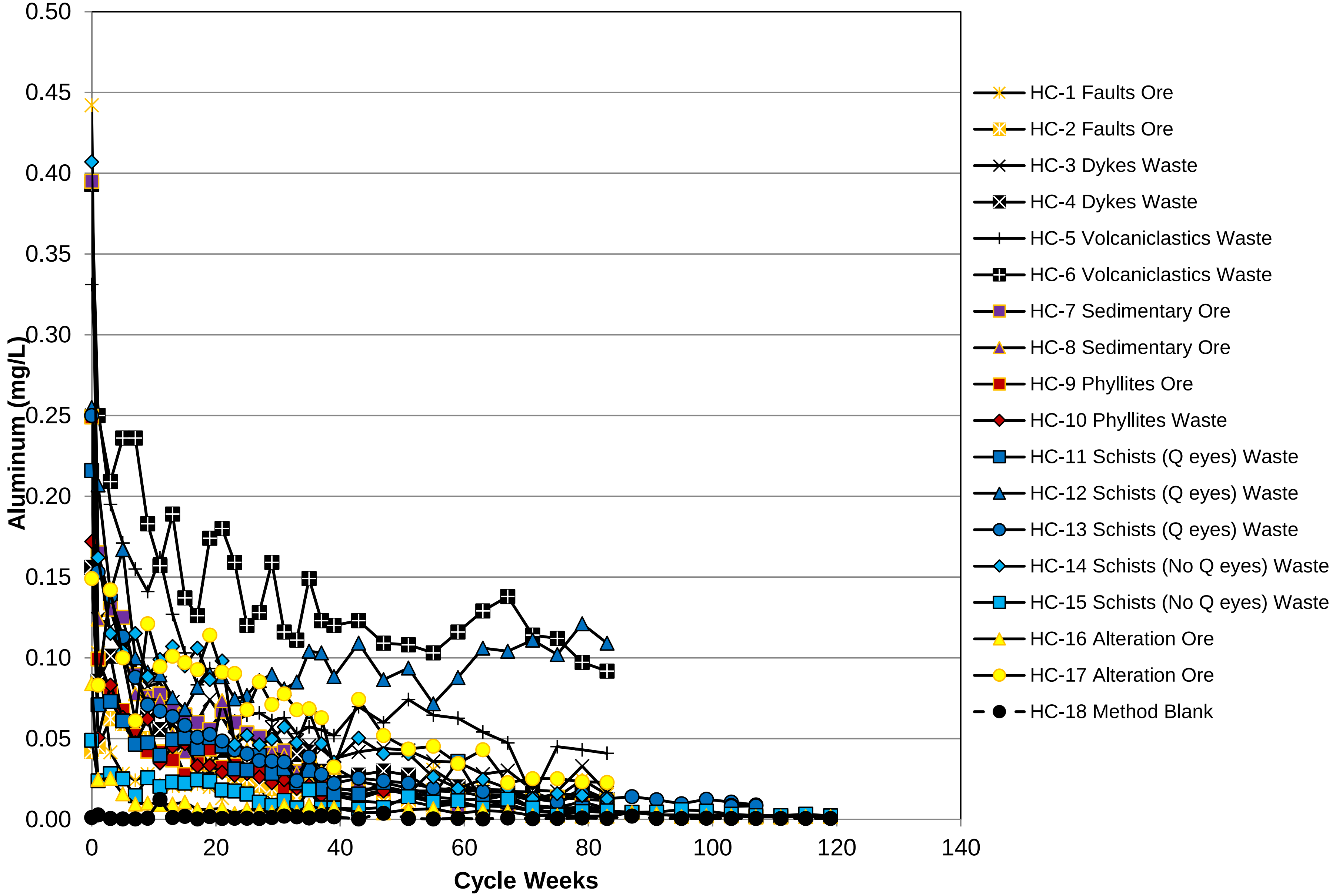

Total Alkalinity (pH 4.5) (mg CaCO3/L)
0
20
40
60
80
100
120
Cycle Weeks
0
20
40
60
80
100
120
140
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

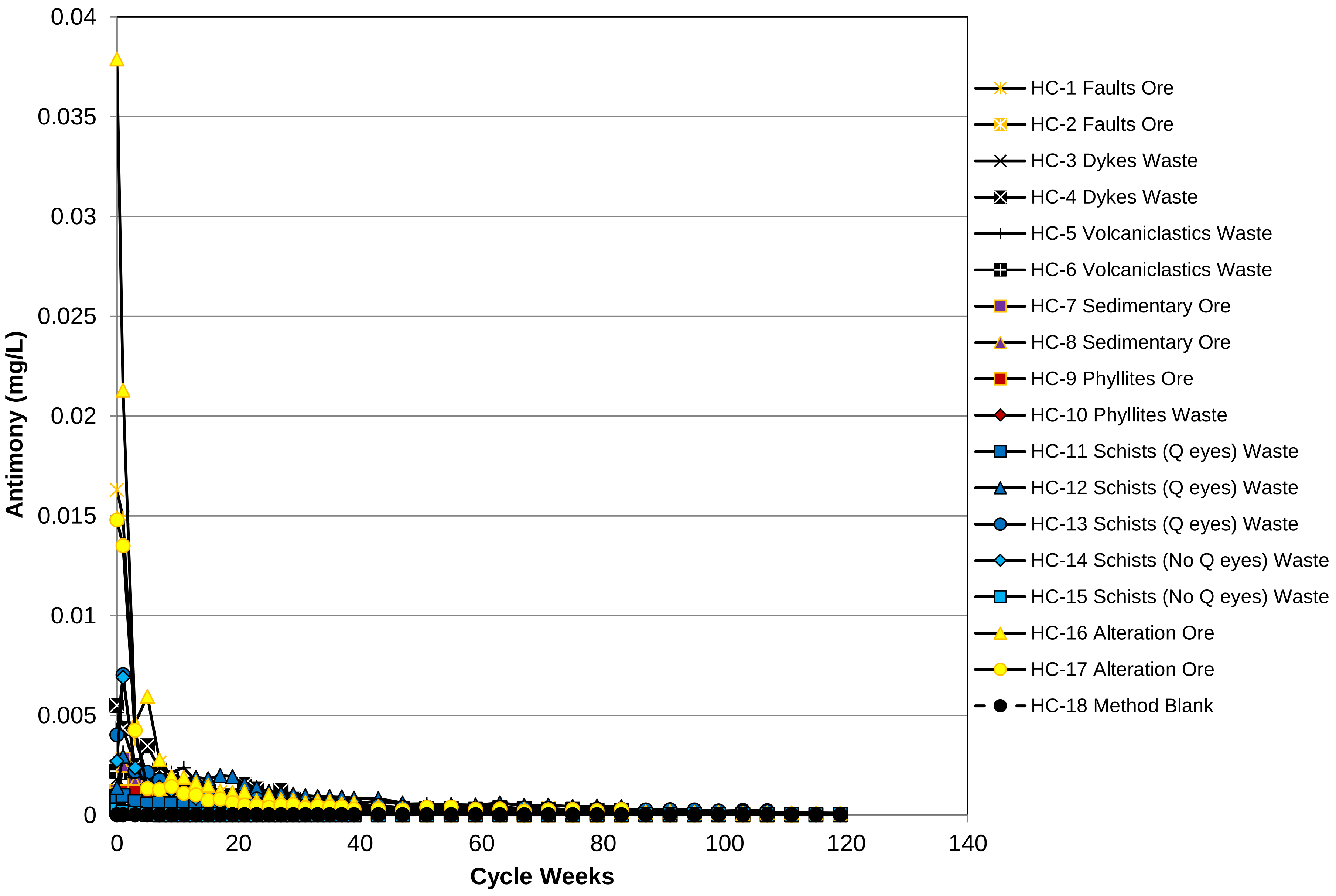

Chloride (mg/L)
0
2
4
6
8
10
12
14
16
18
Cycle Weeks
0
20
40
60
80
100
120
140
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

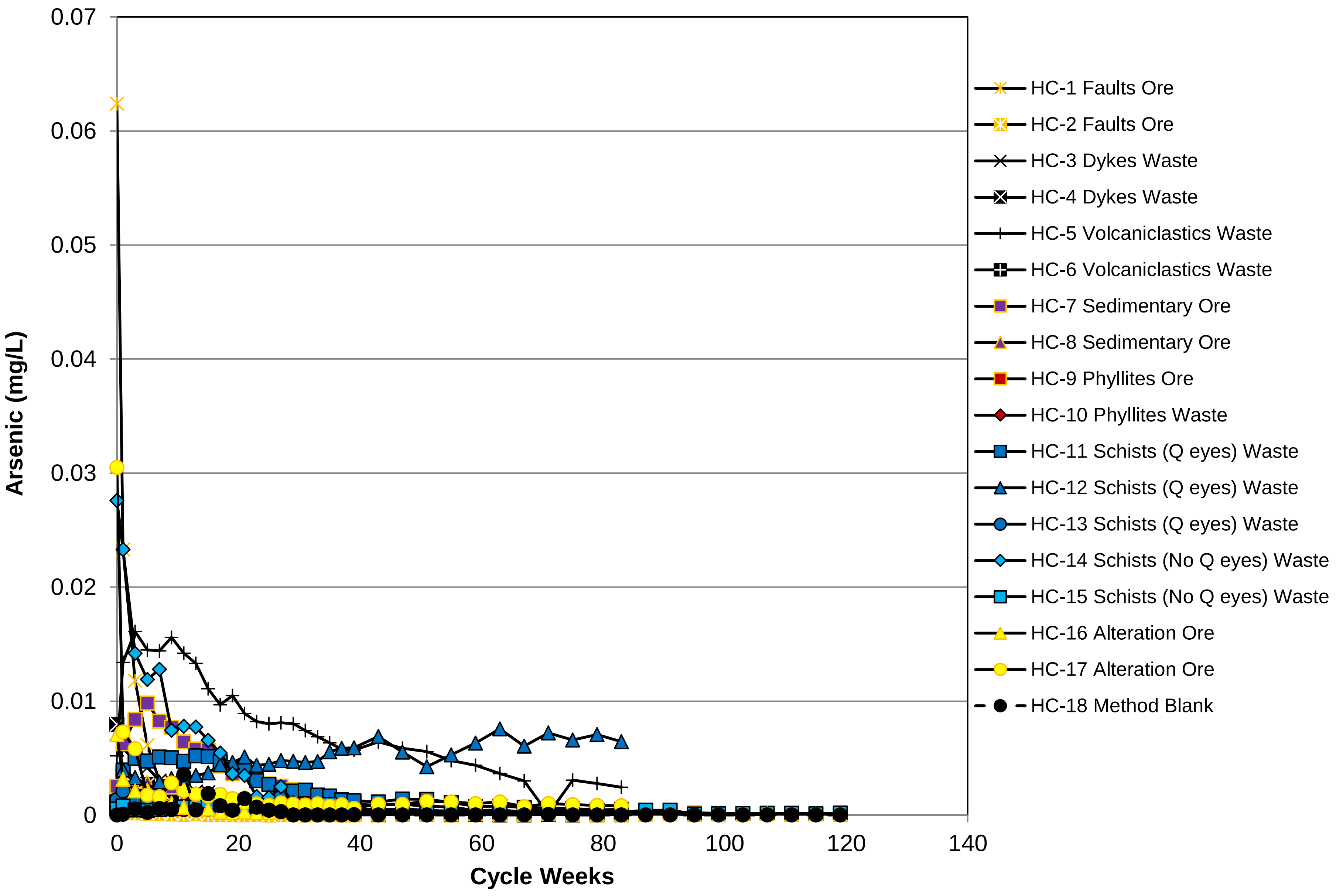


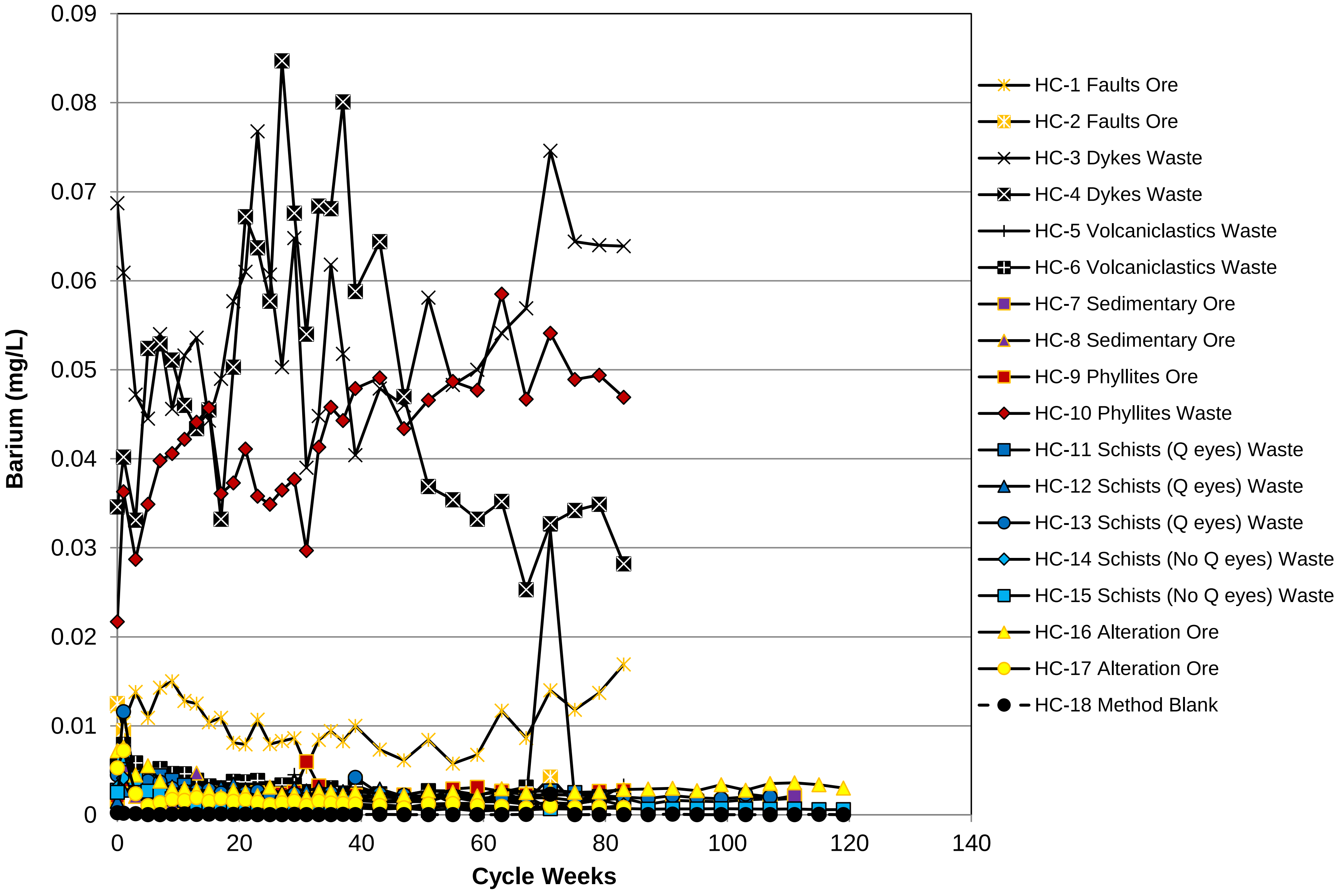
1.2
1
0.8
0.6
0.4
0.2
0
Fluoride (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

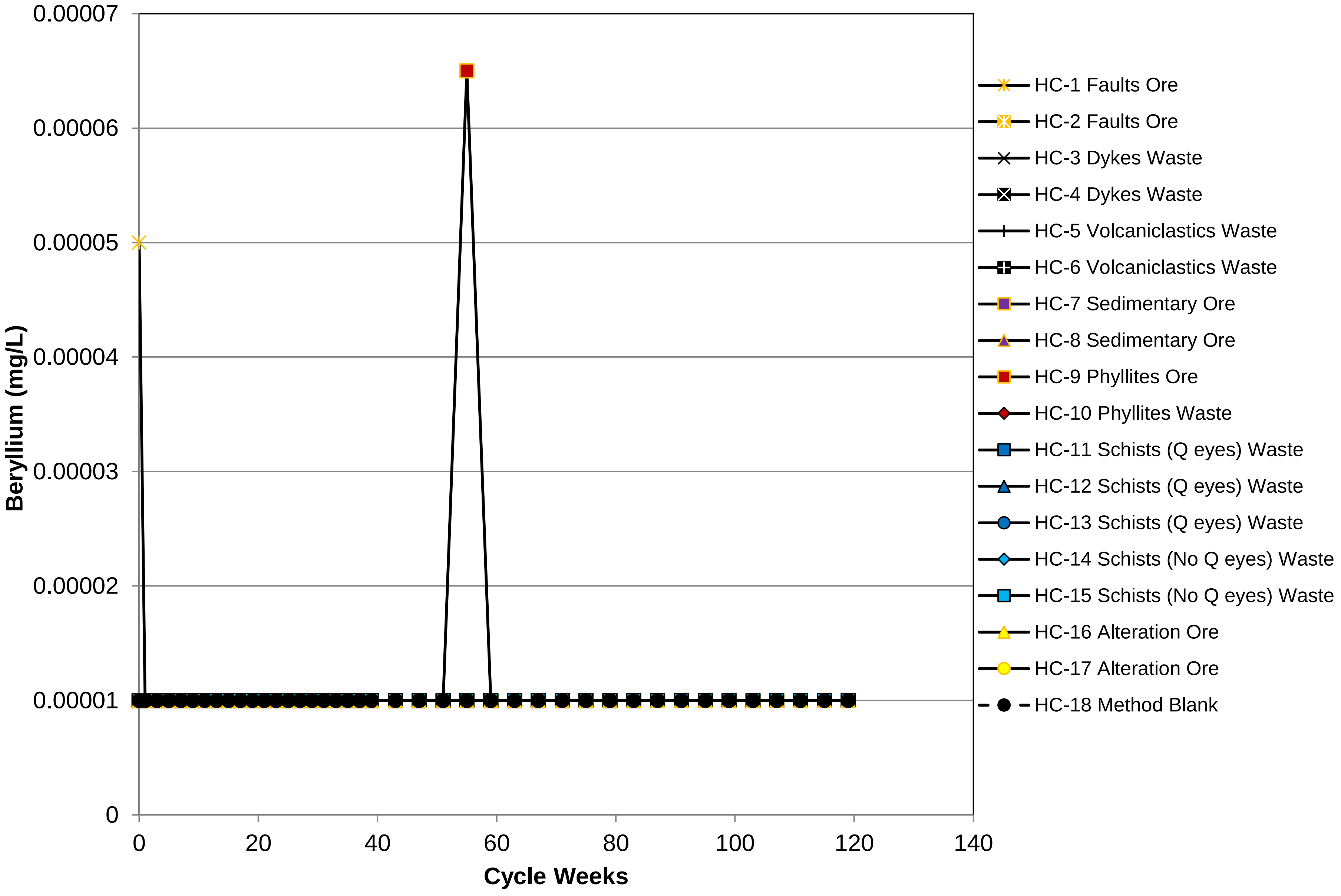

250
200
150
100
50
0
Hardness as CaCO3mg/L
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

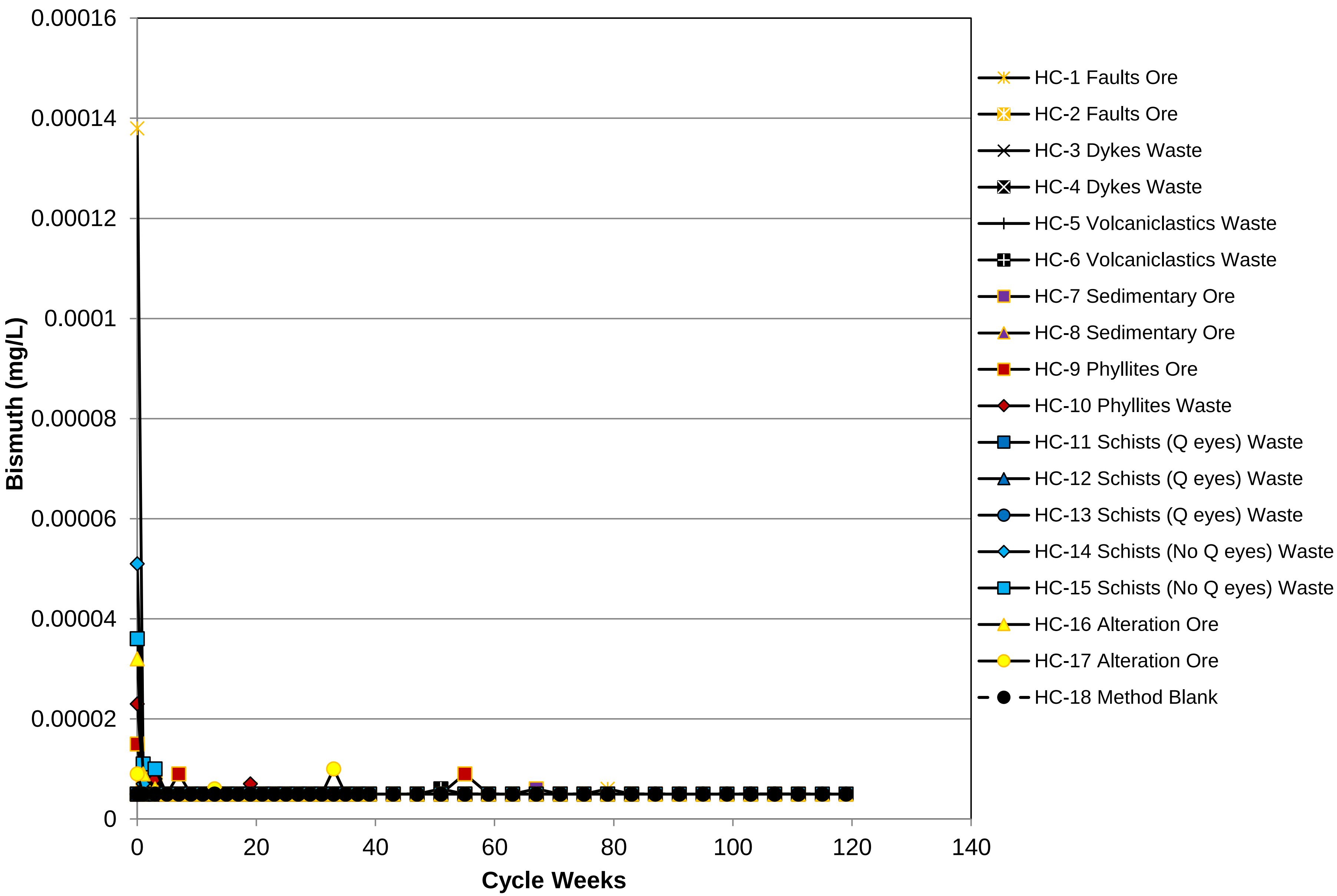

0.50
0.45
0.40
0.35
0.30
0.25
0.20
0.15
0.10
0.05
0.00
Aluminum (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

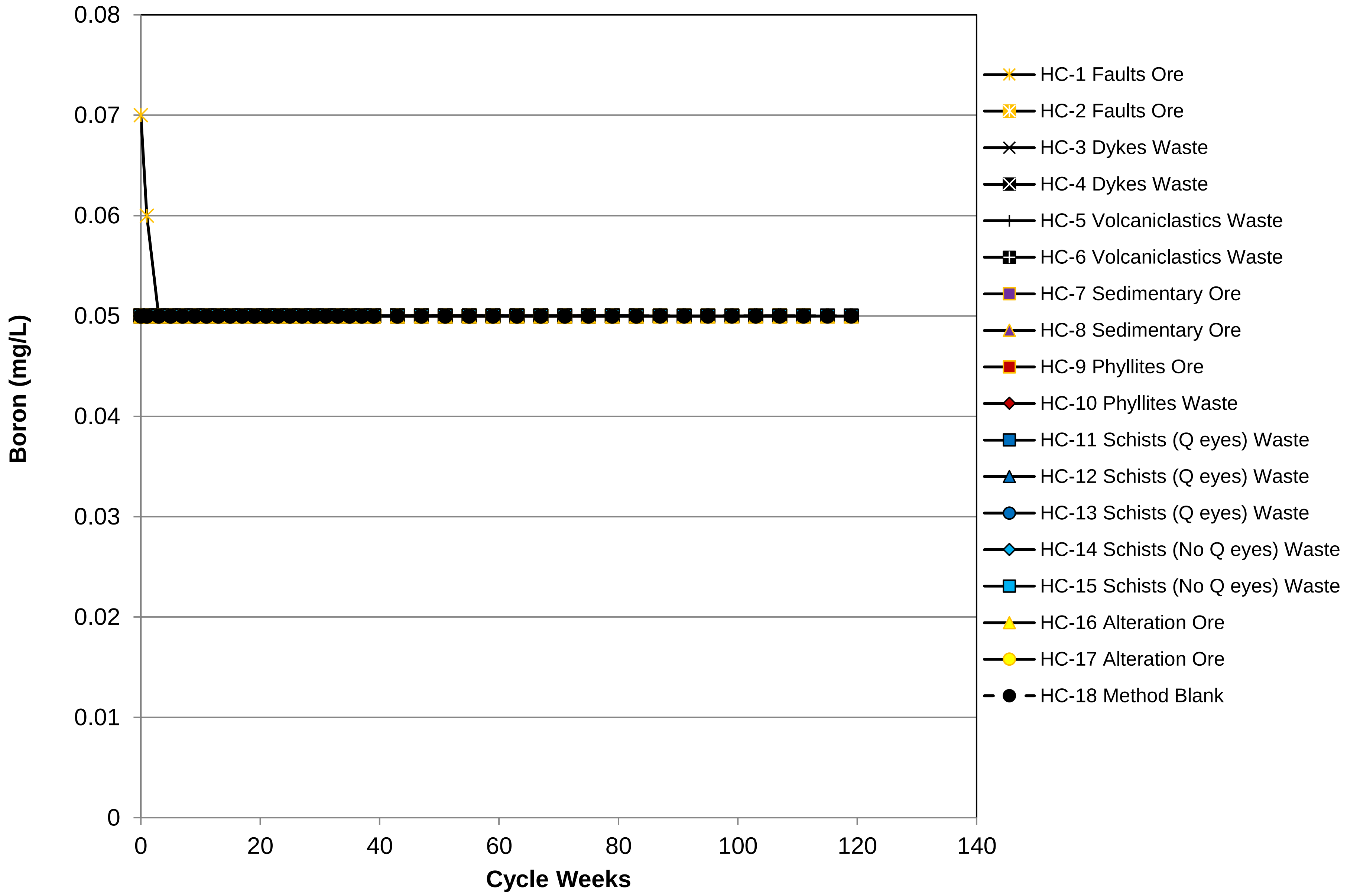

0.04
0.035
0.03
0.025
0.02
0.015
0.01
0.005
0
Antimony (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

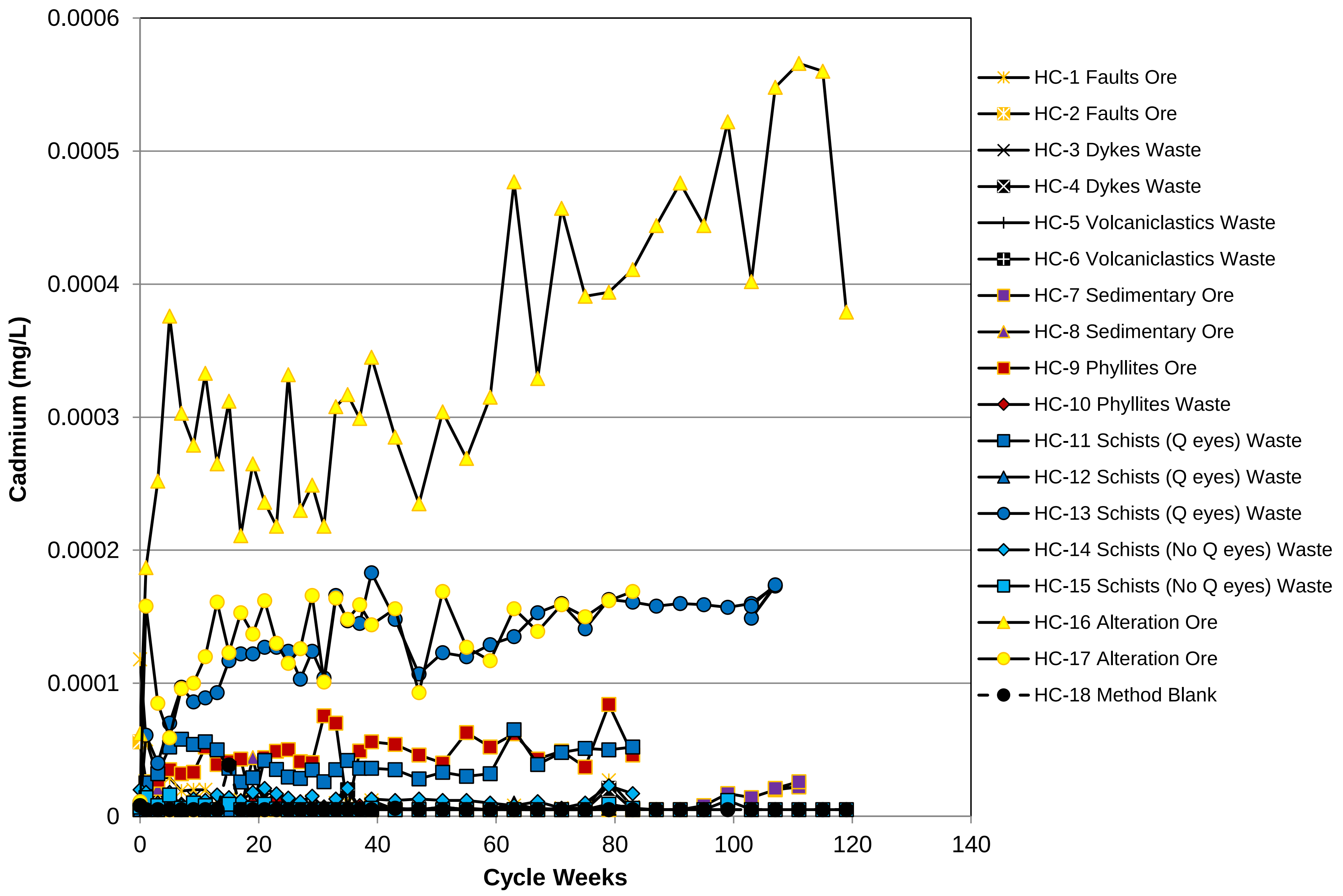

0.07
0.06
0.05
0.04
0.03
0.02
0.01
0
Arsenic (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

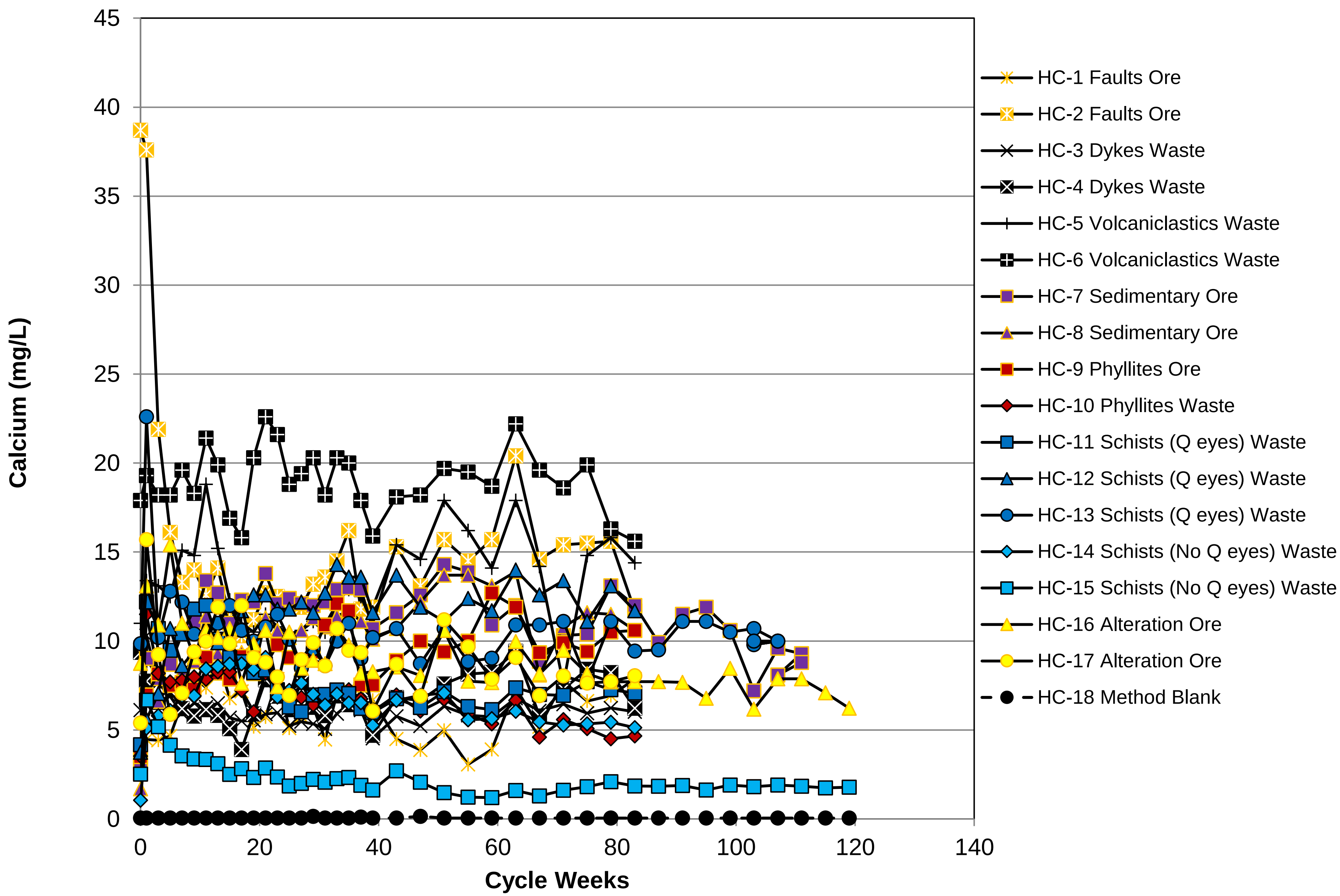

Barium (mg/L)
0.09
0.08
0.07
0.06
0.05
0.04
0.03
0.02
0.01
0
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

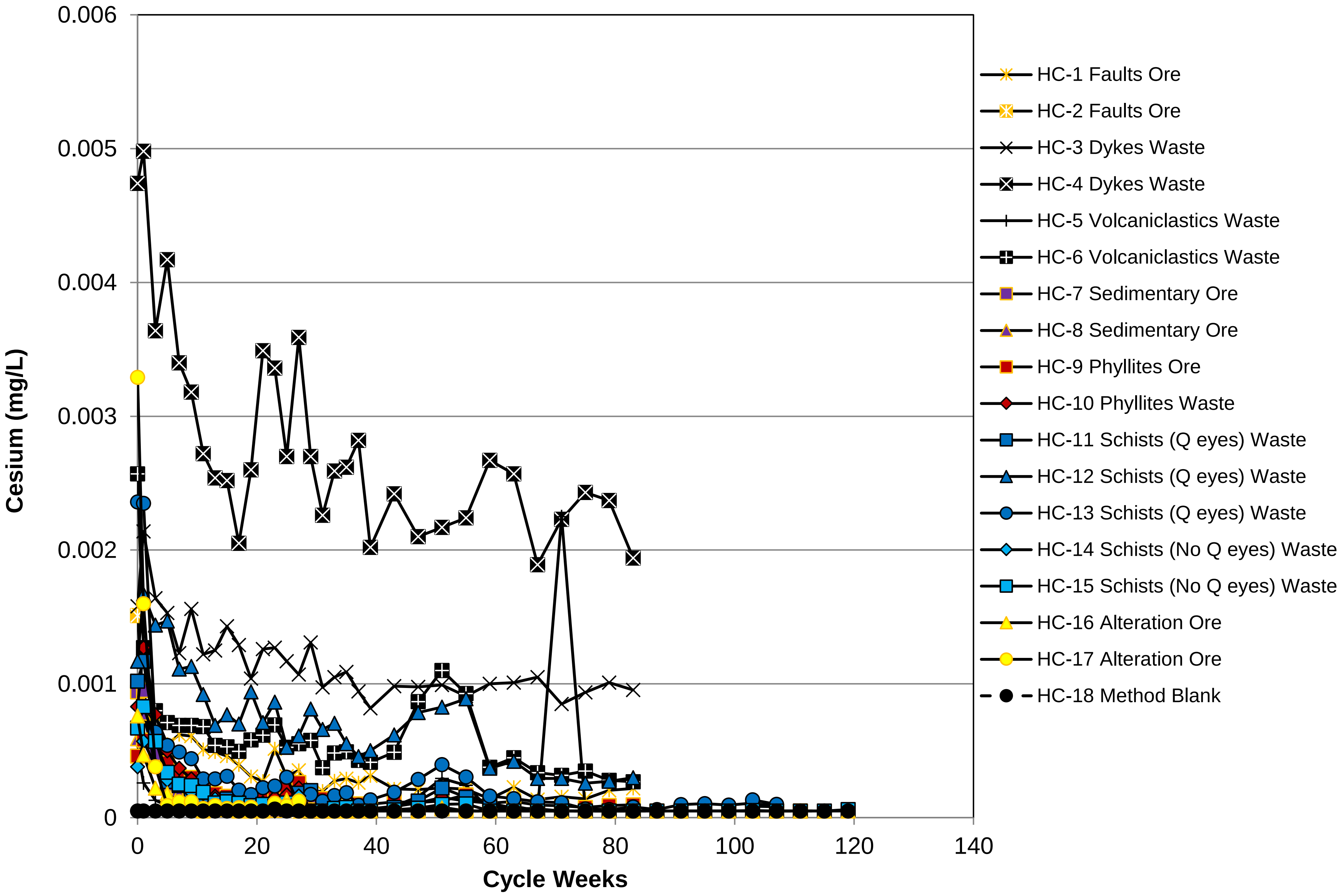

0.00007
0.00006
0.00005
0.00004
0.00003
0.00002
0.00001
0
Beryllium (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

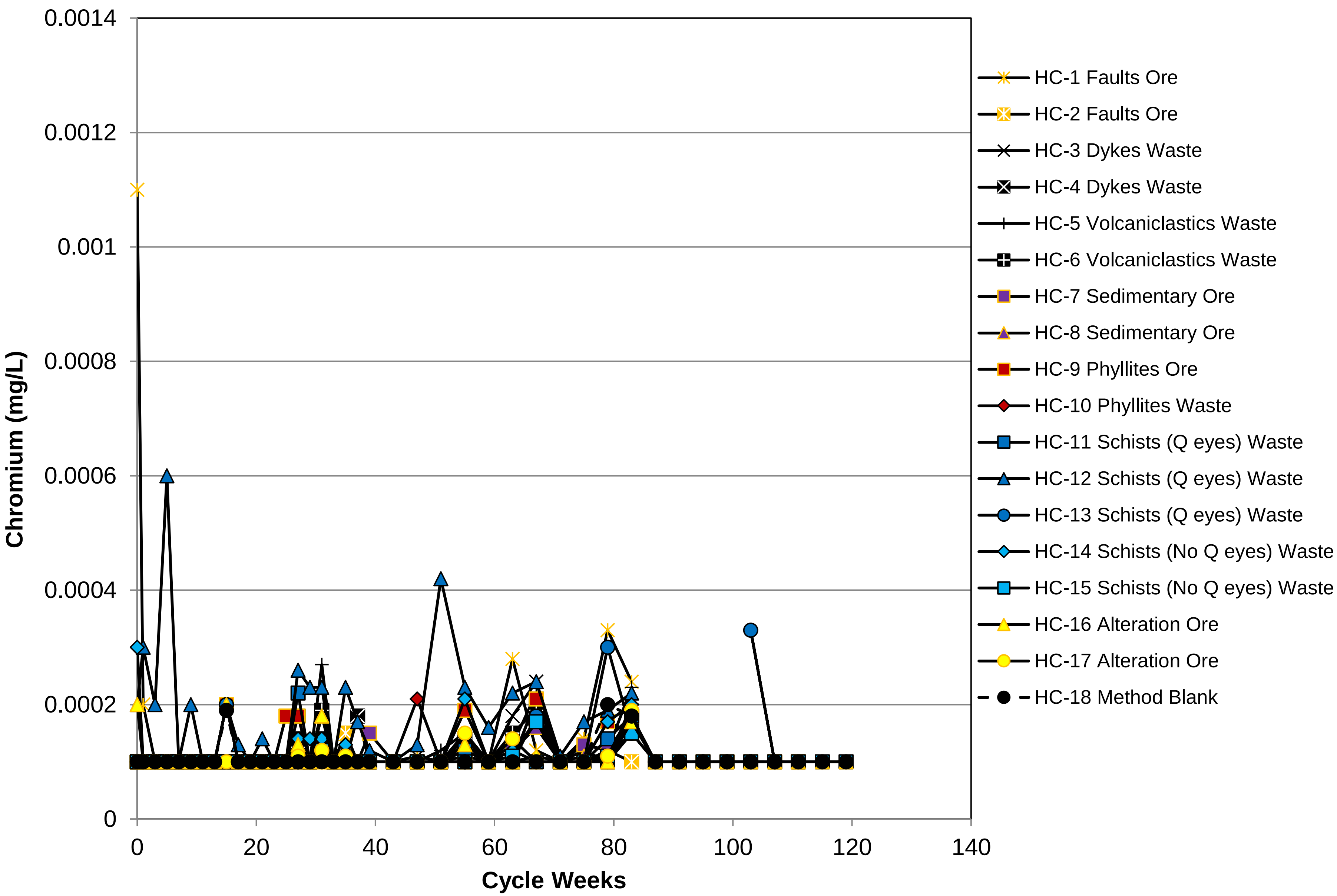

Bismuth (mg/L)
0.00016
0.00014
0.00012
0.0001
0.00008
0.00006
0.00004
0.00002
0
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

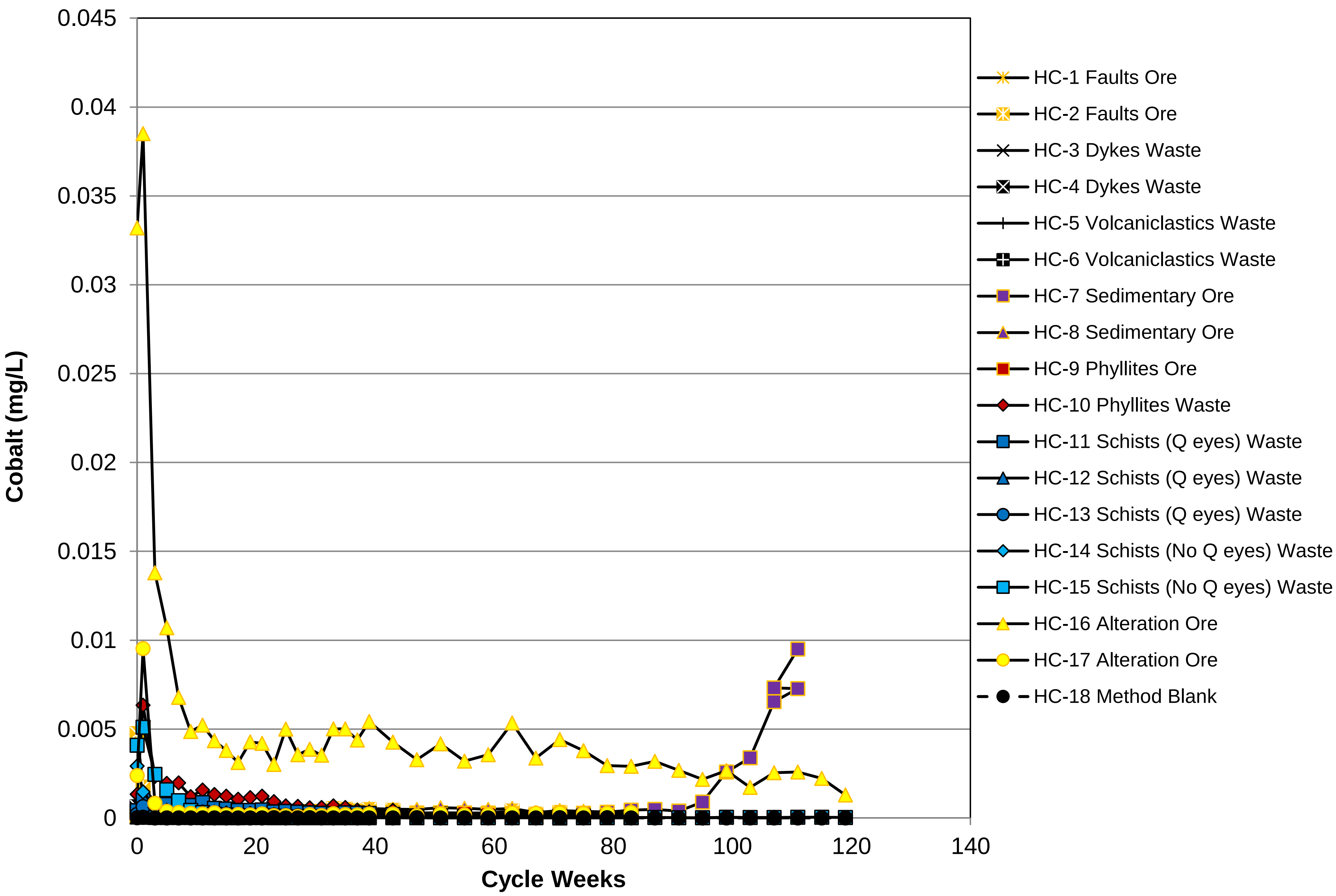

0.08
0.07
0.06
0.05
0.04
0.03
0.02
0.01
0
Boron (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

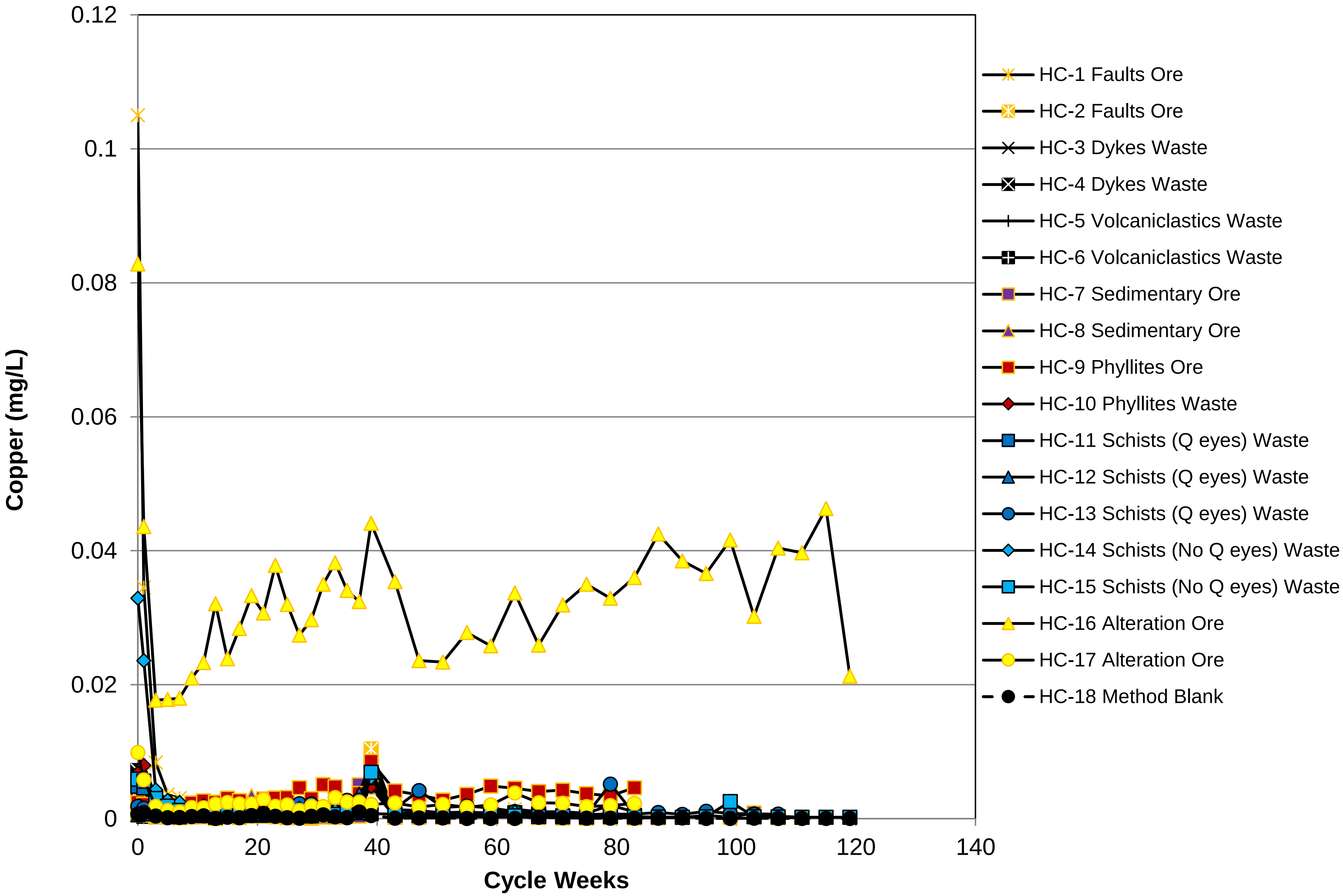

Cadmium (mg/L)
0.0006
0.0005
0.0004
0.0003
0.0002
0.0001
0
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

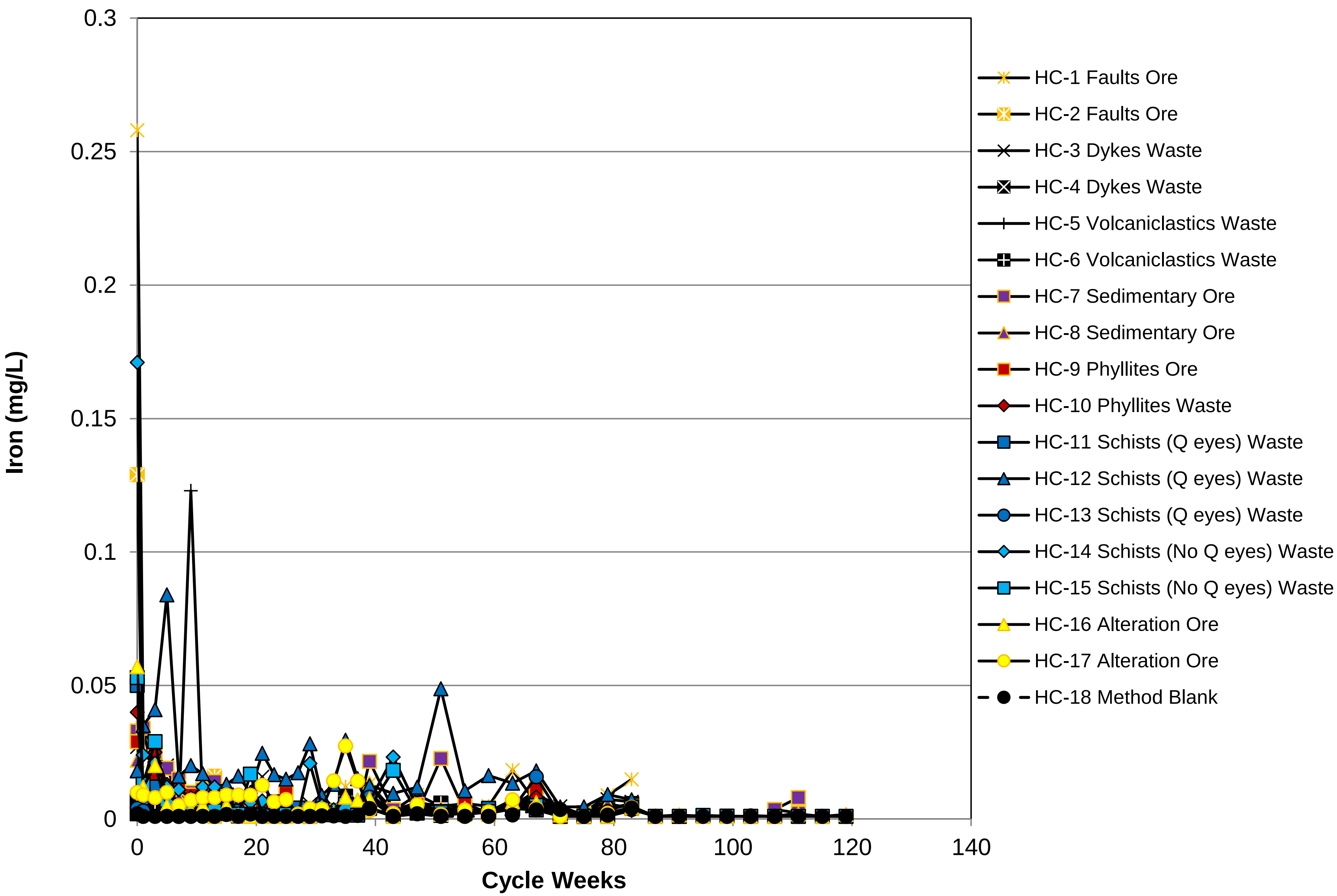

Calcium (mg/L)
0
5
10
15
20
25
30
35
40
45
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

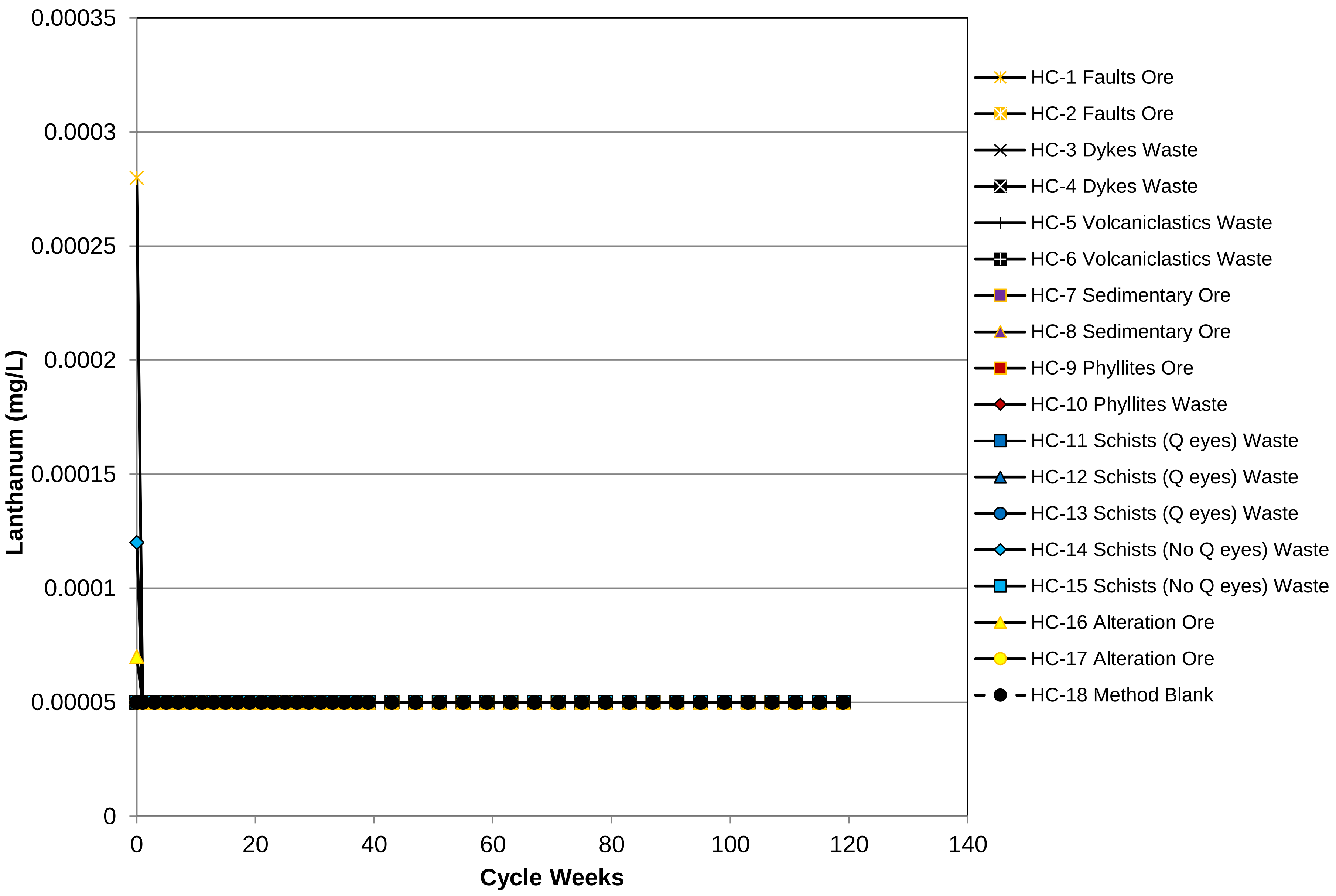

0.006
0.005
0.004
0.003
0.002
0.001
0
Cesium (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

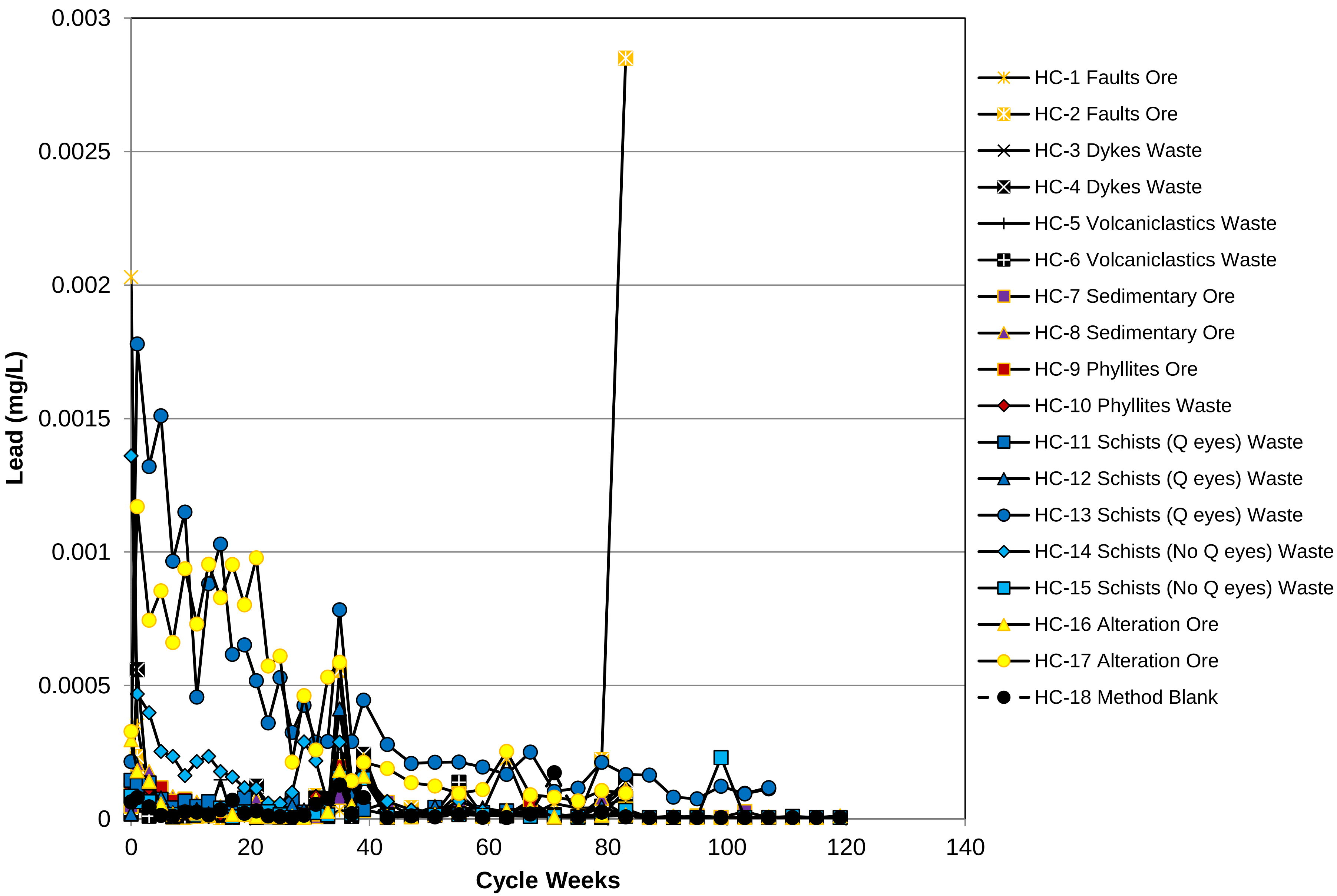

Chromium (mg/L)
0.0014
0.0012
0.001
0.0008
0.0006
0.0004
0.0002
0
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

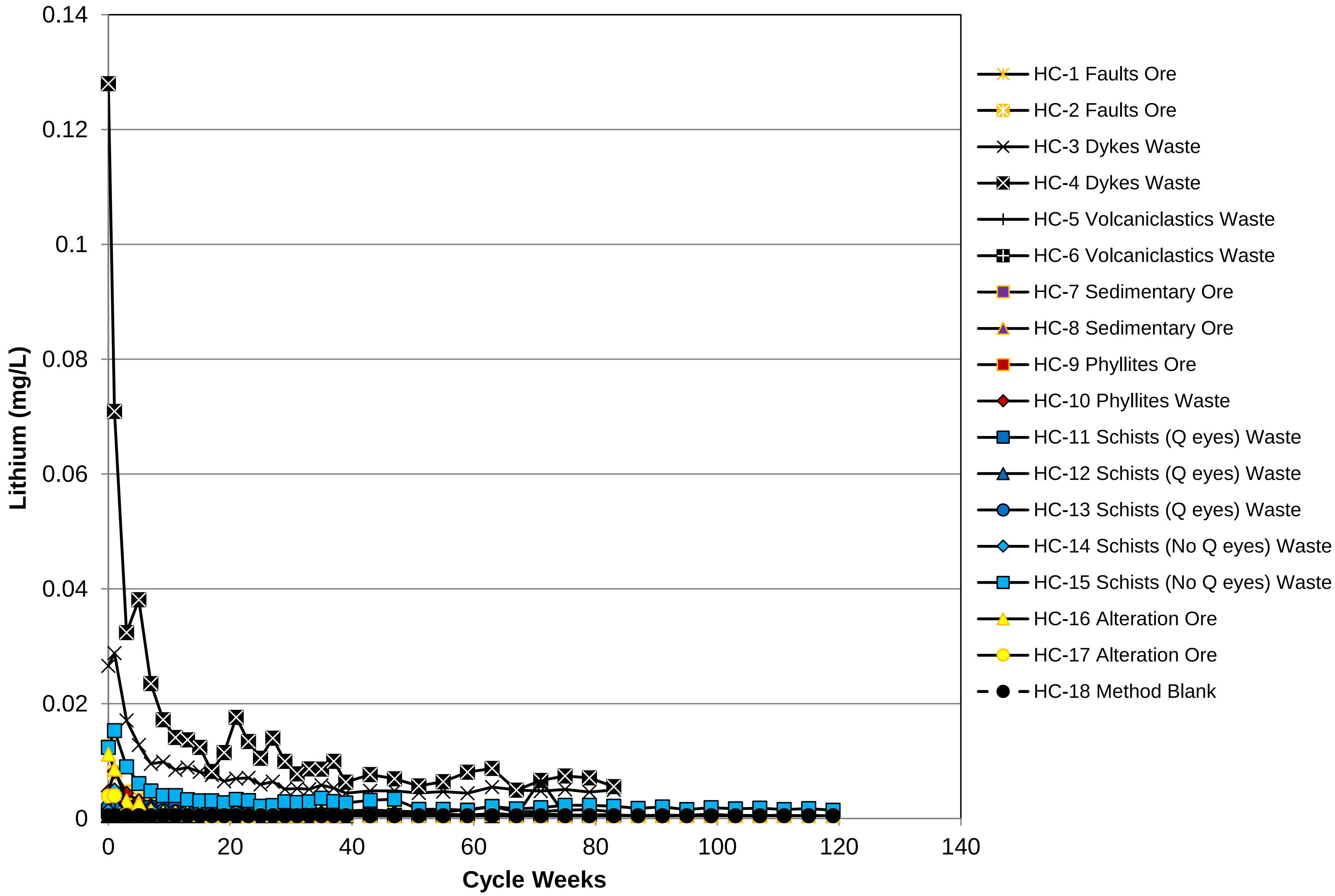

Cobalt (mg/L)
0.045
0.04
0.035
0.03
0.025
0.02
0.015
0.01
0.005
0
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

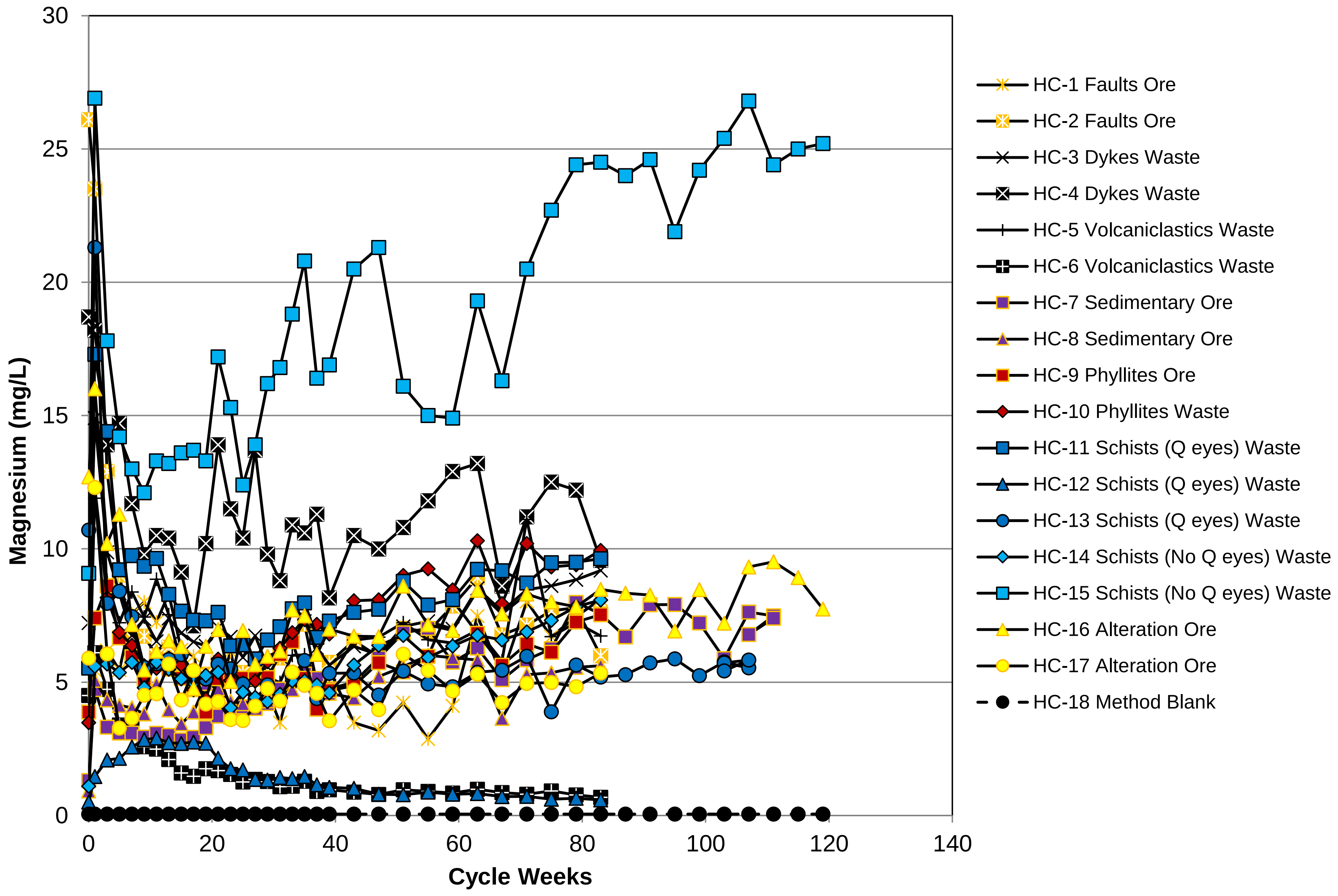

0.12
0.1
0.08
0.06
0.04
0.02
0
Copper (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

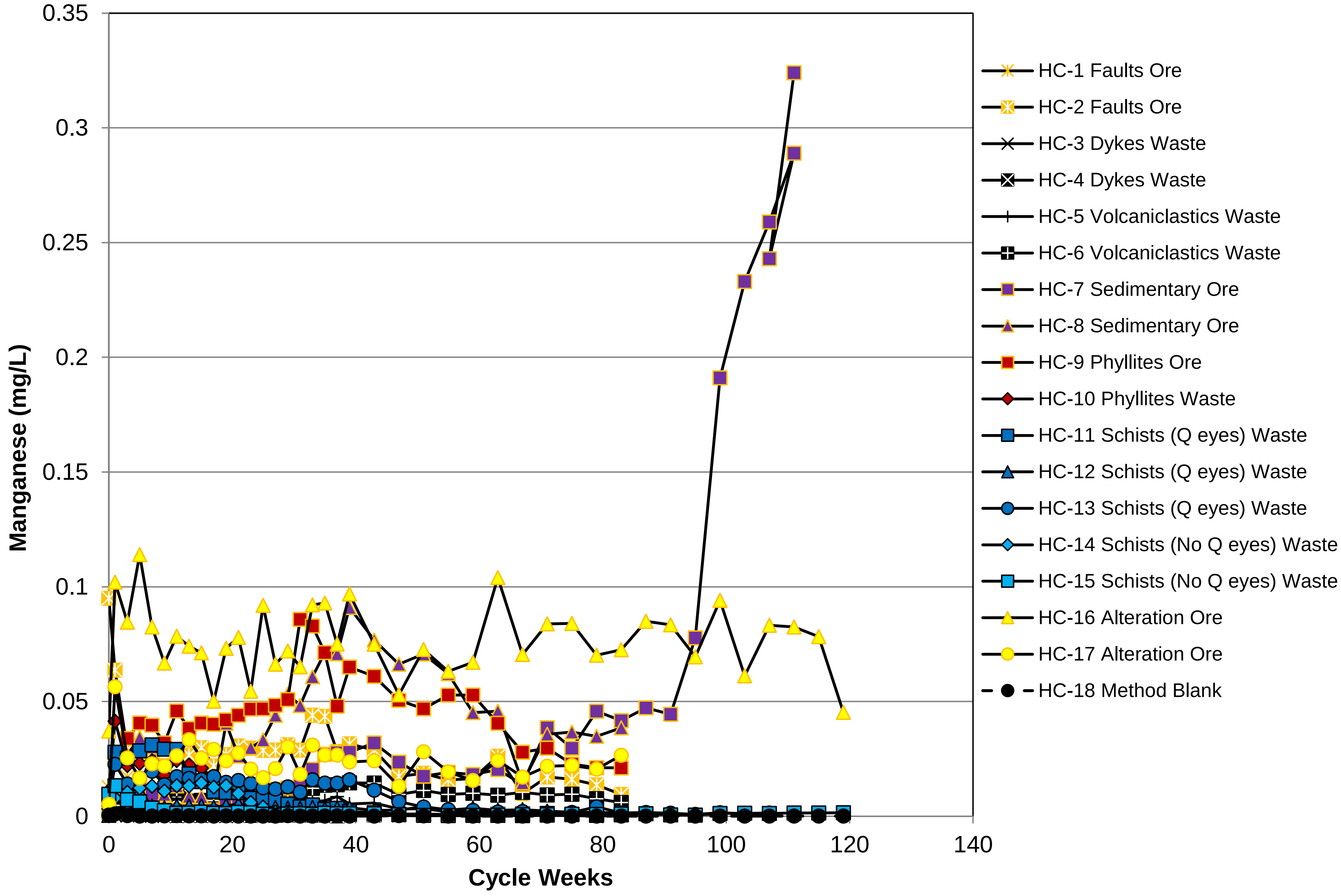

0.3
0.25
0.2
0.15
0.1
0.05
0
Iron (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

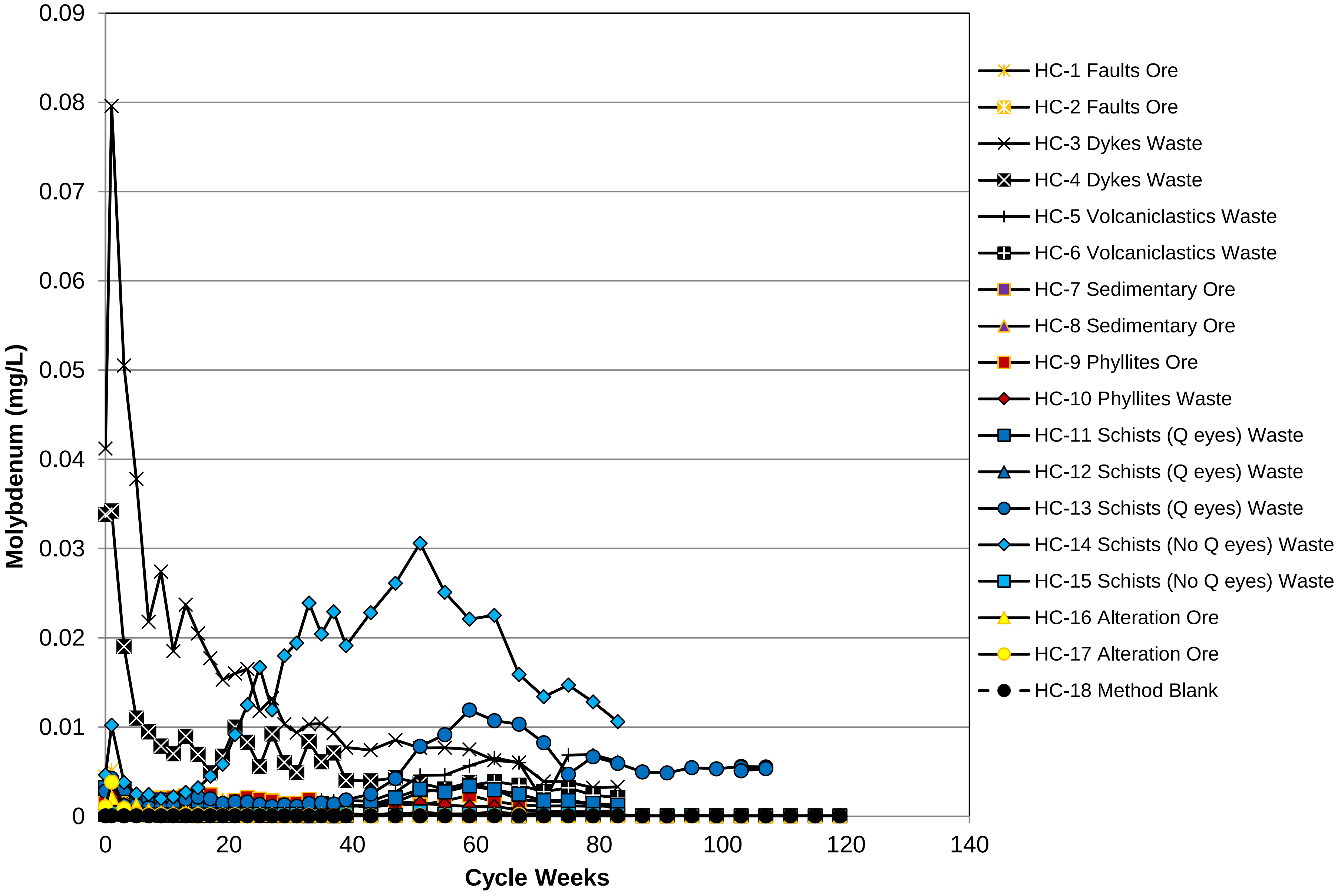

0.00035
0.0003
0.00025
0.0002
0.00015
0.0001
0.00005
0
Lanthanum (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

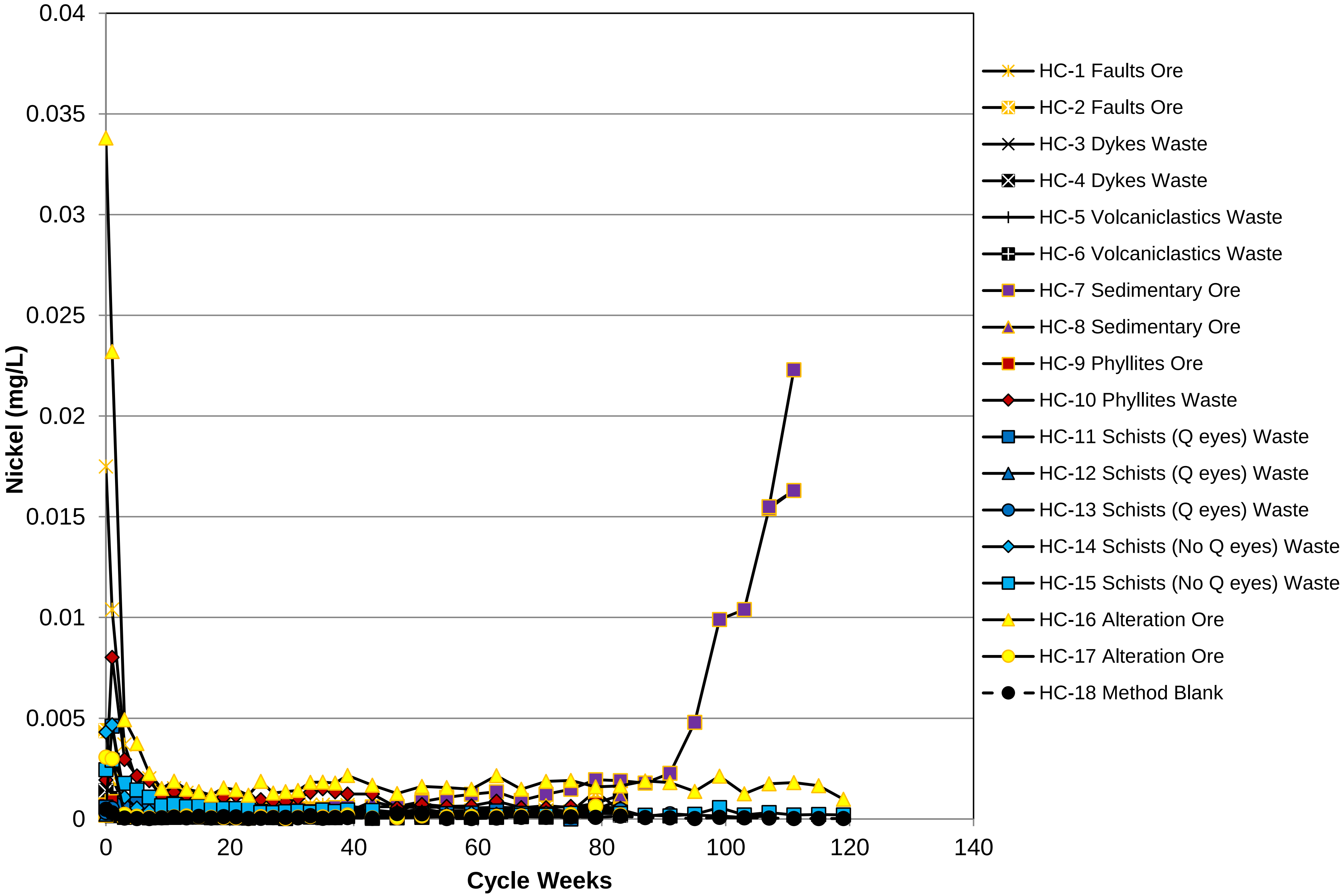

0.003
0.0025
0.002
0.0015
0.001
0.0005
0
Lead (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

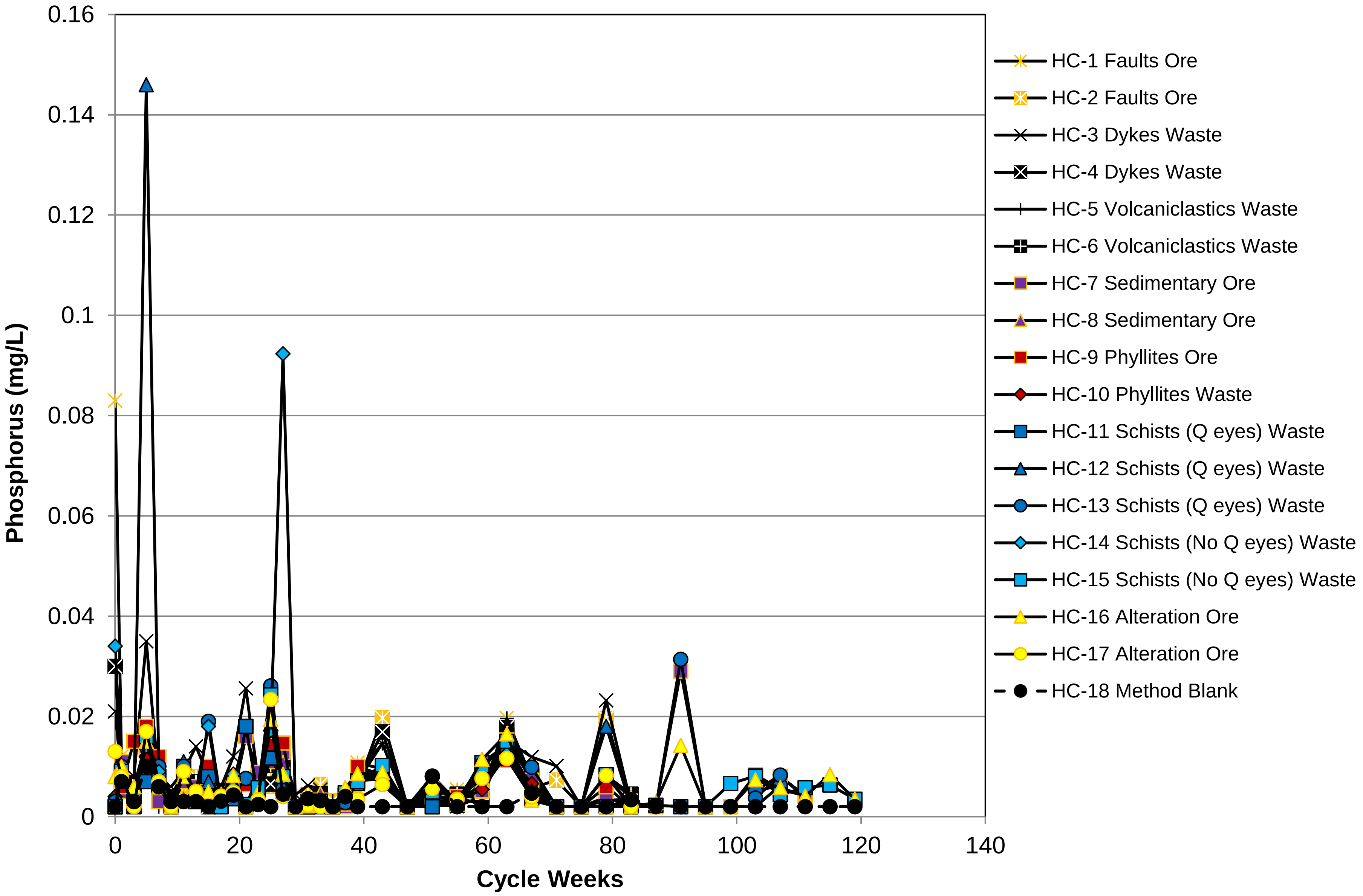

0.14
0.12
0.1
0.08
0.06
0.04
0.02
0
Lithium (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

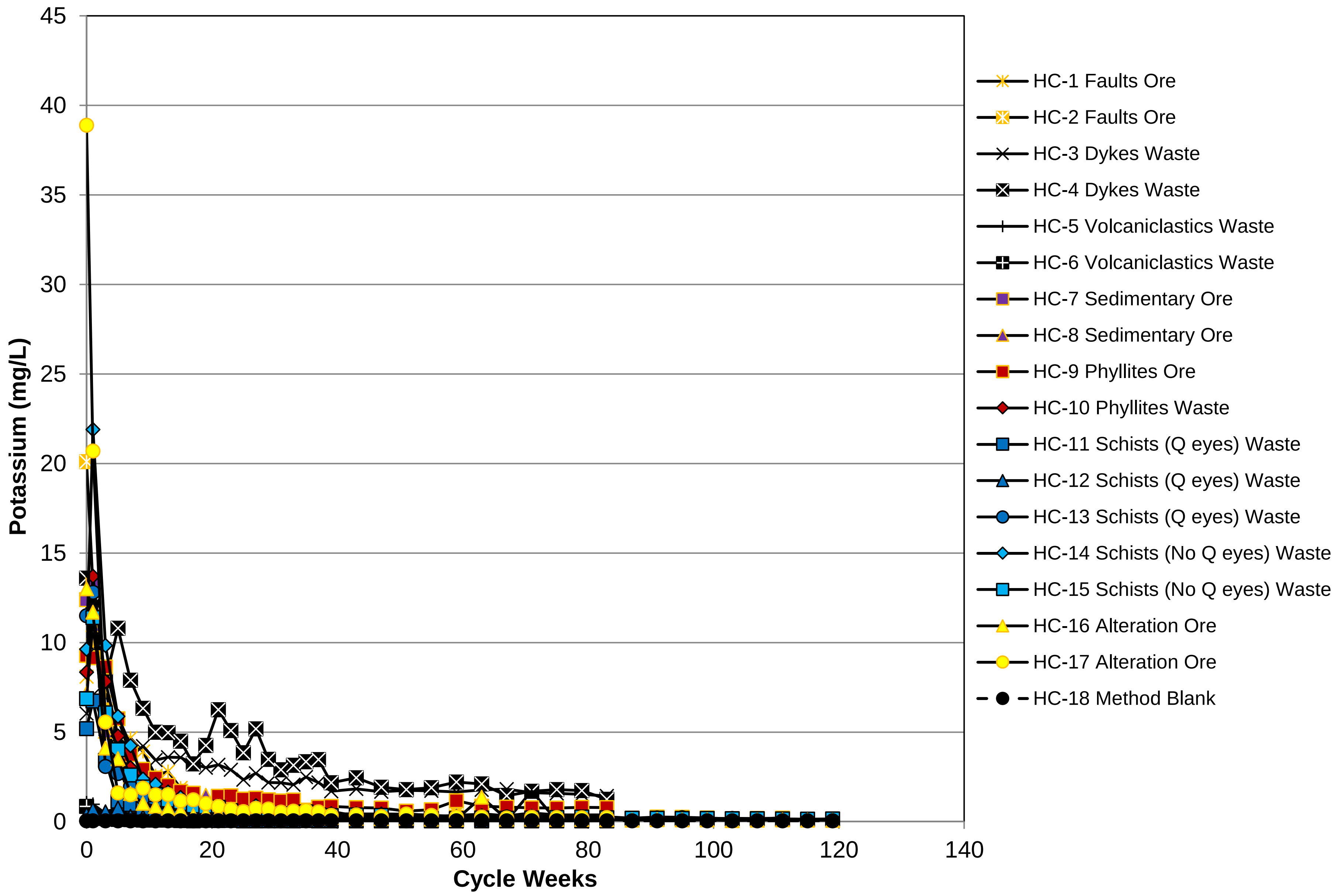

Magnesium (mg/L)
30
25
20
15
10
5
0
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

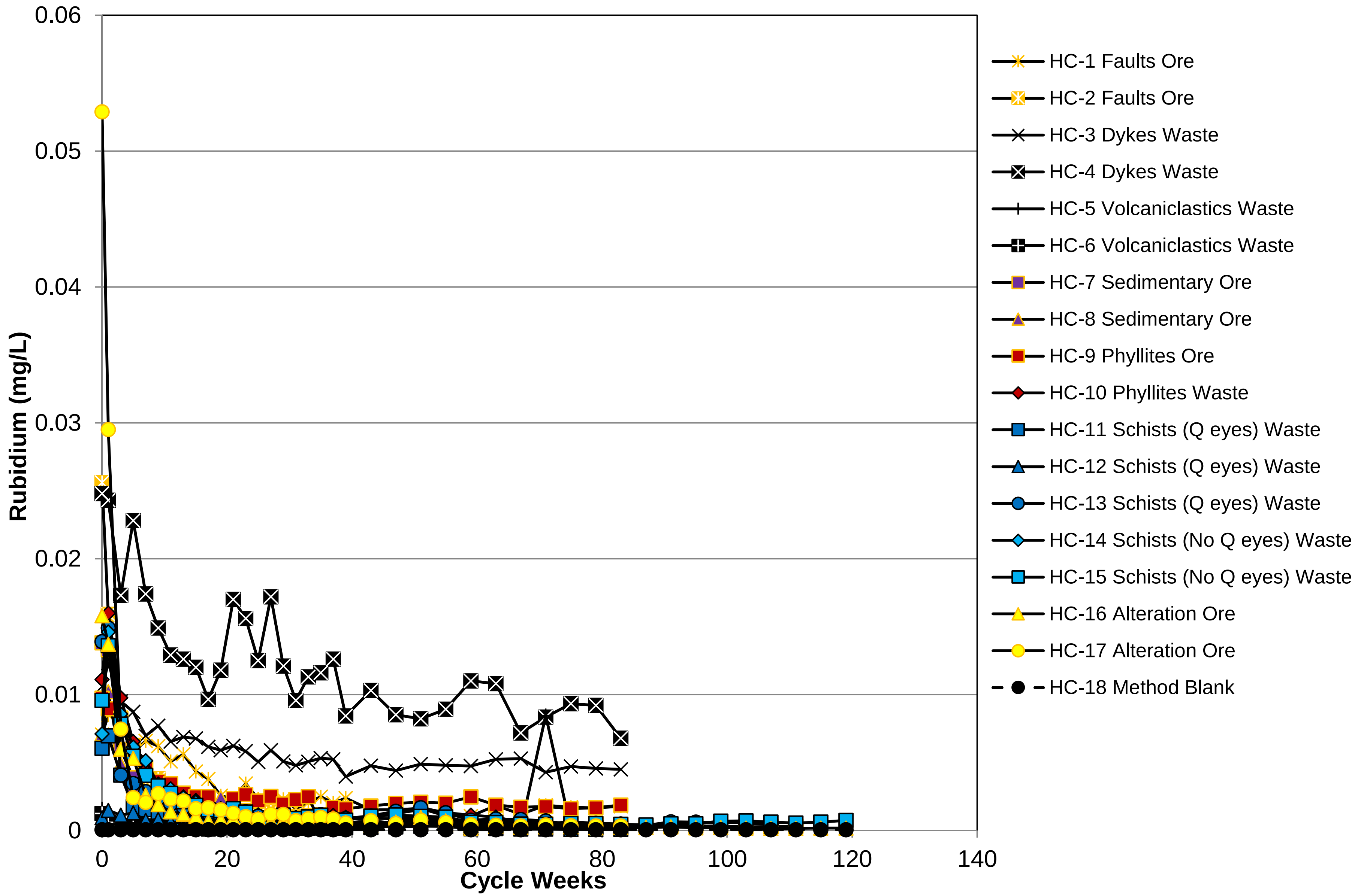

Manganese (mg/L)
0.35
0.3
0.25
0.2
0.15
0.1
0.05
0
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank


0.09
0.08
0.07
0.06
0.05
0.04
0.03
0.02
0.01
0
Molybdenum (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

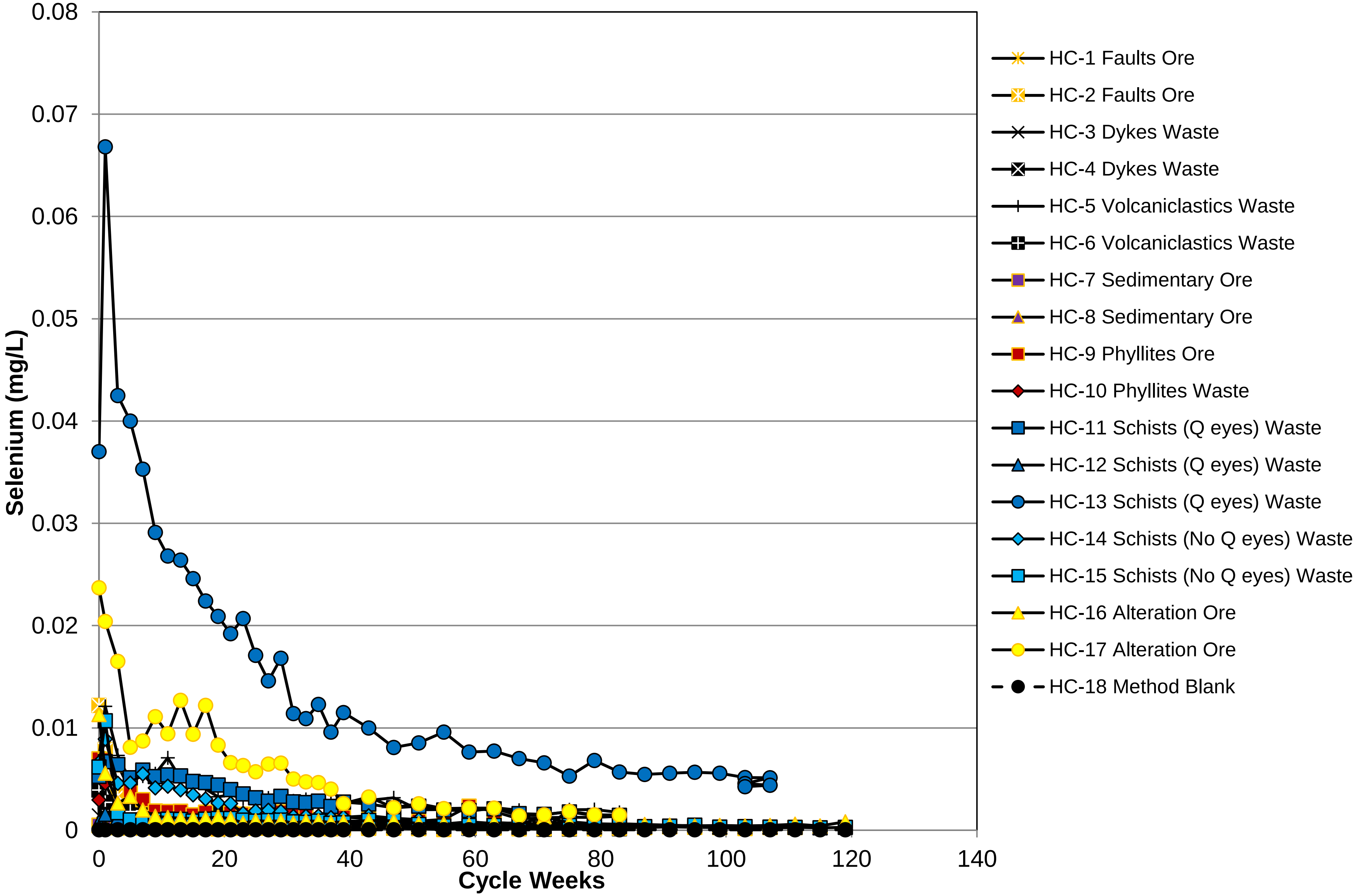

0.04
0.035
0.03
0.025
0.02
0.015
0.01
0.005
0
Nickel (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

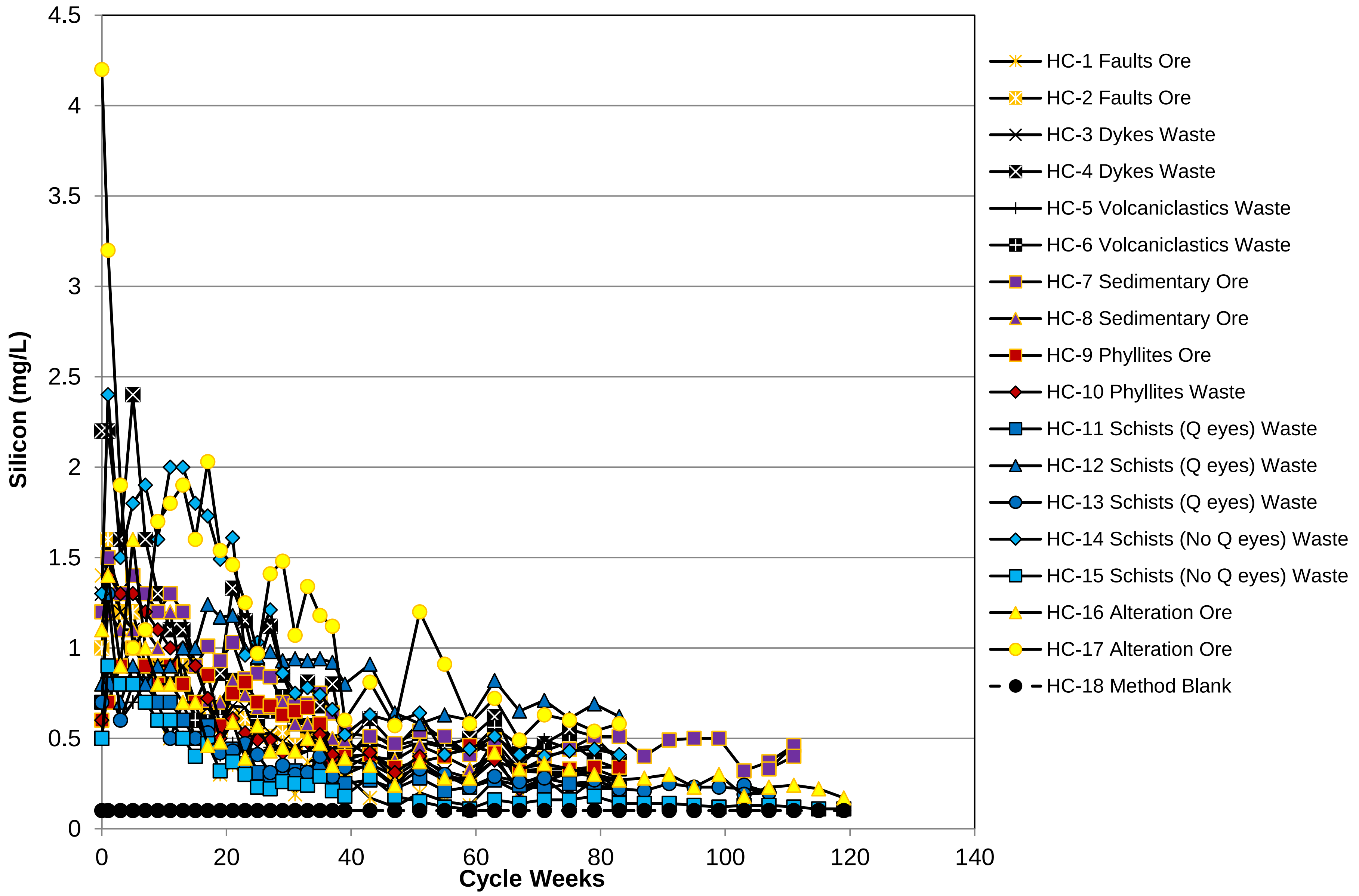

0.16
0.14
0.12
0.1
0.08
0.06
0.04
0.02
0
Phosphorus (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

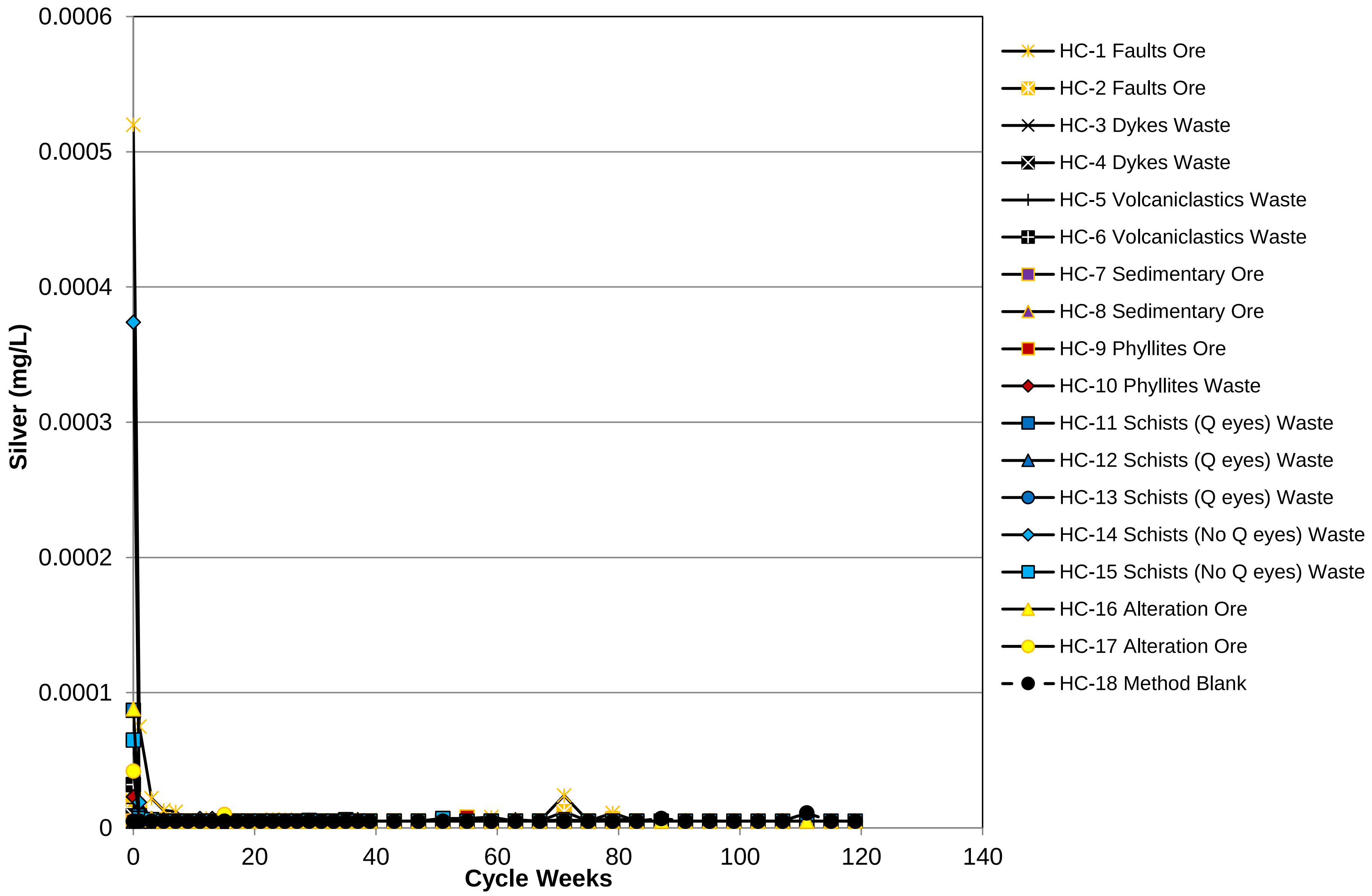

45
40
35
30
25
20
15
10
5
0
Potassium (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

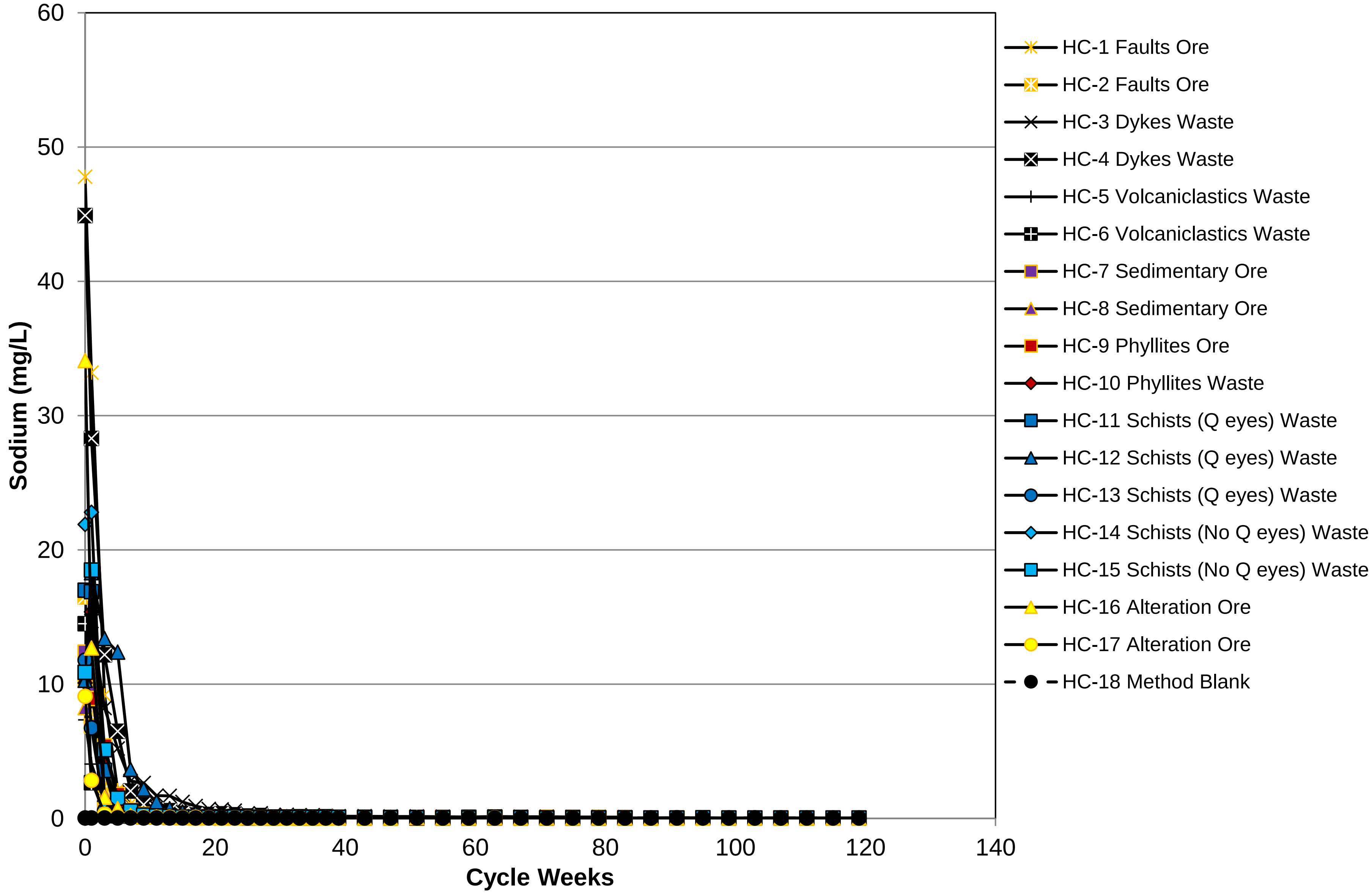


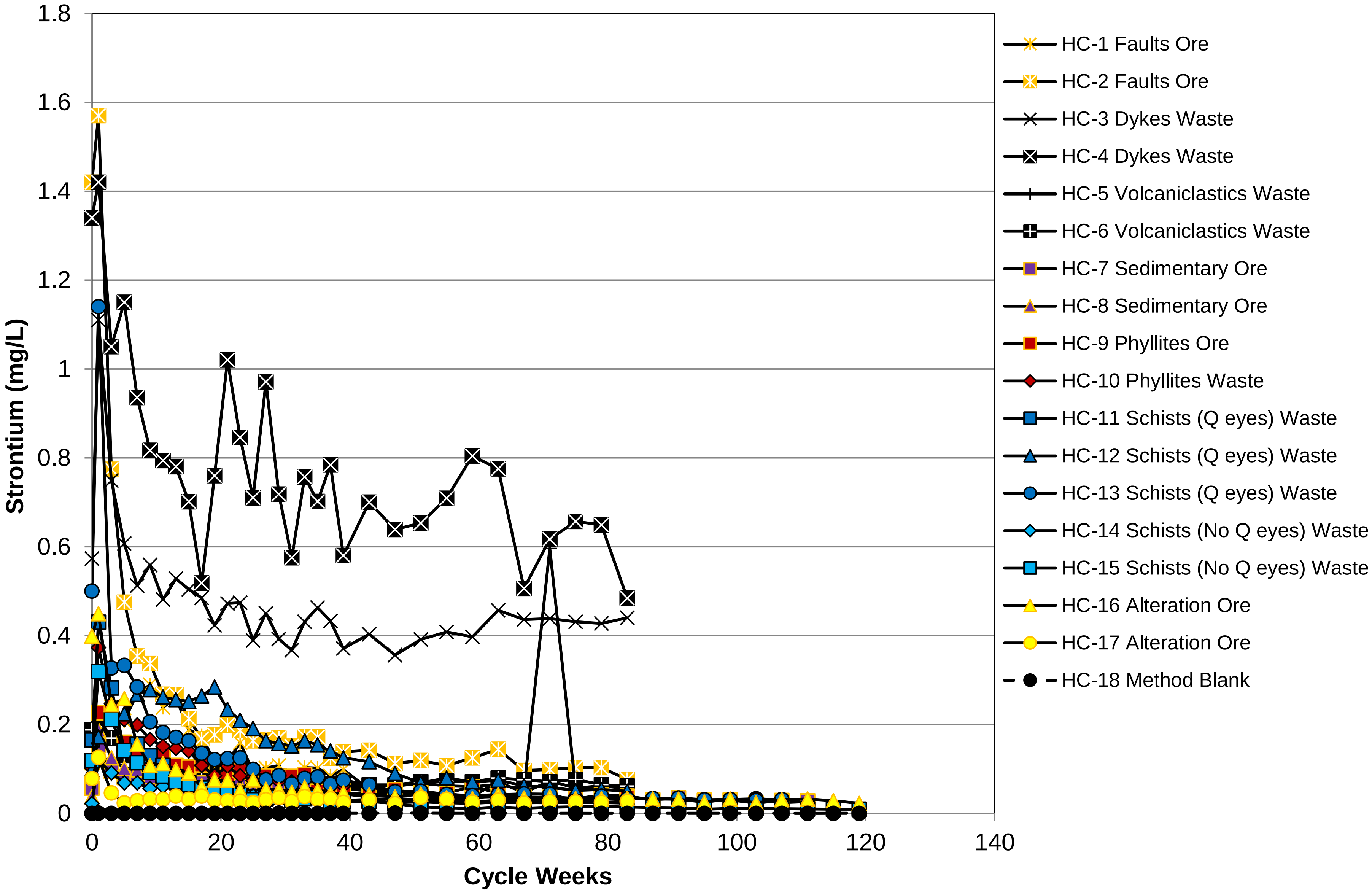
0.06
0.05
0.04
0.03
0.02
0.01
0
Rubidium (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

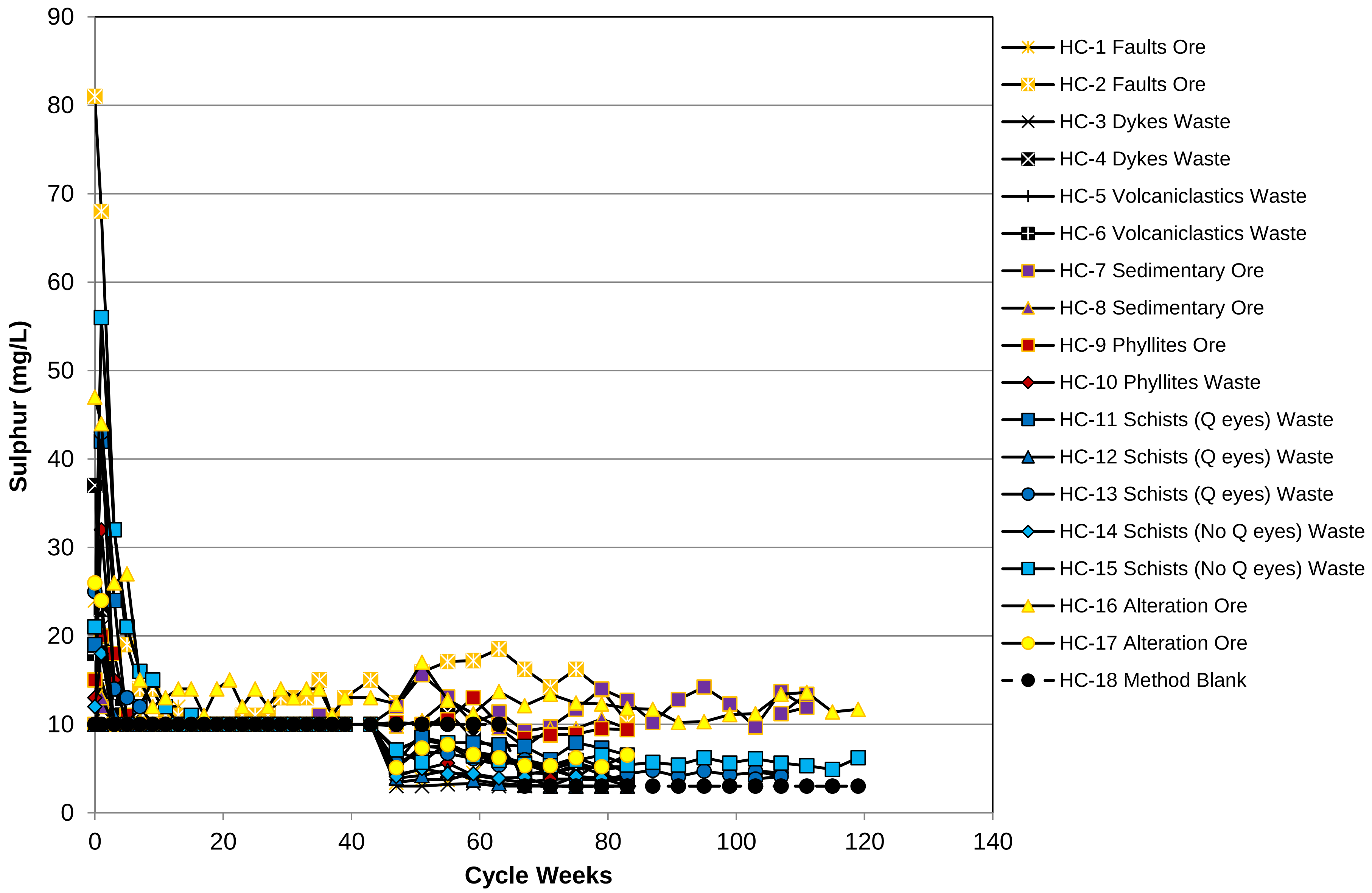

Selenium (mg/L)
0.08
0.07
0.06
0.05
0.04
0.03
0.02
0.01
0
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

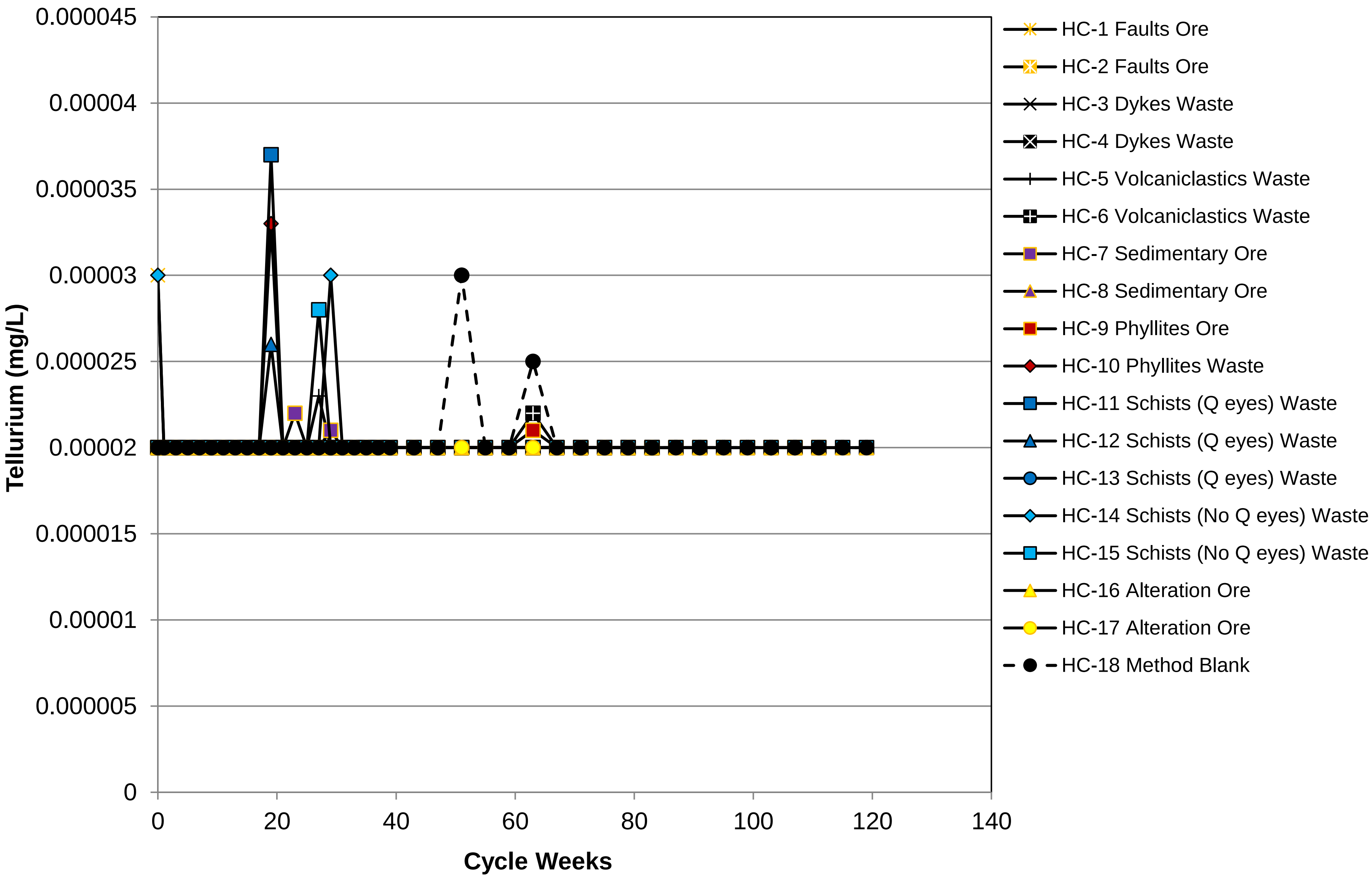

Silicon (mg/L)
4.5
4
3.5
3
2.5
2
1.5
1
0.5
0
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

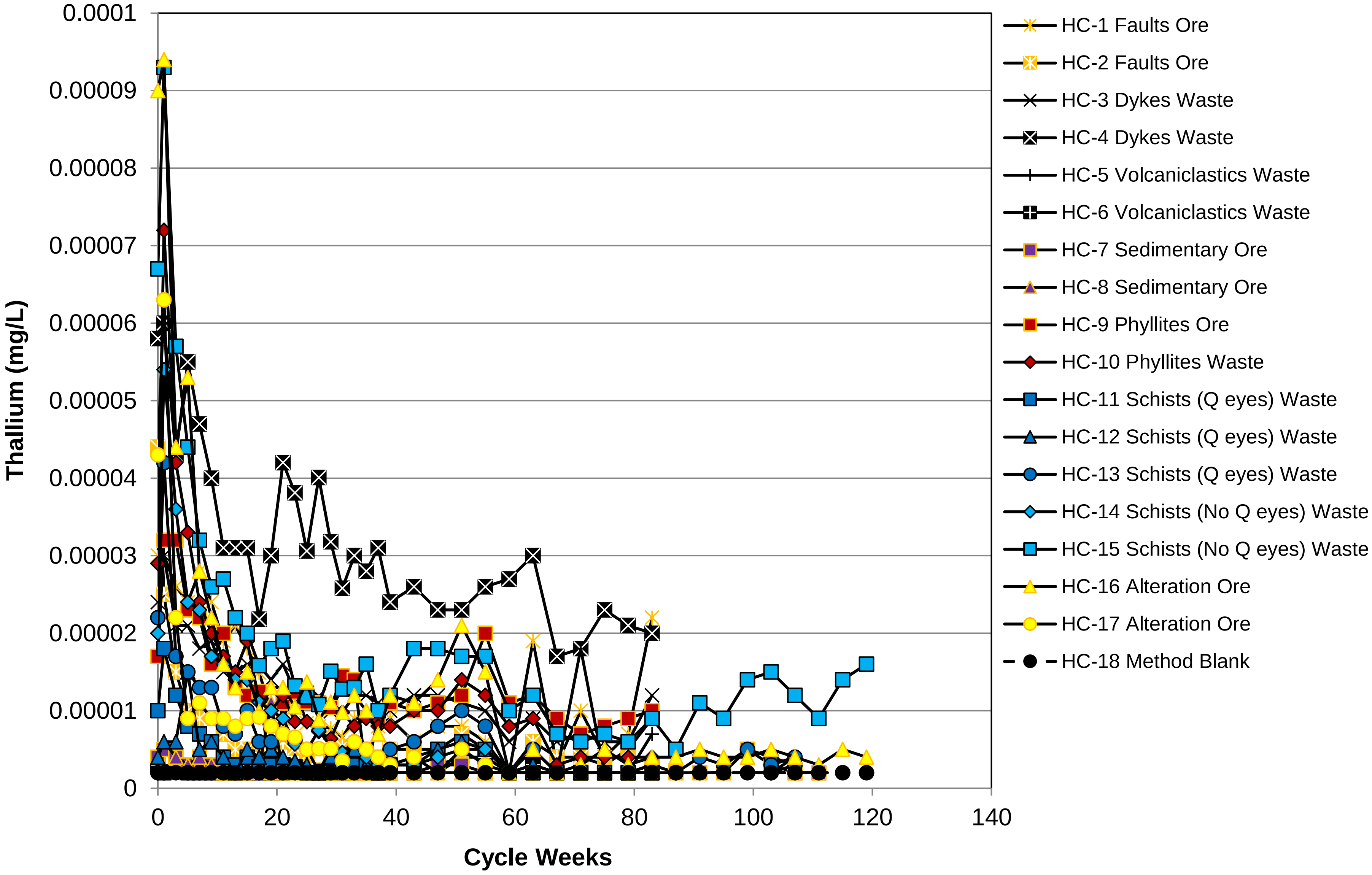

0.0006
0.0005
0.0004
0.0003
0.0002
0.0001
0
Silver (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

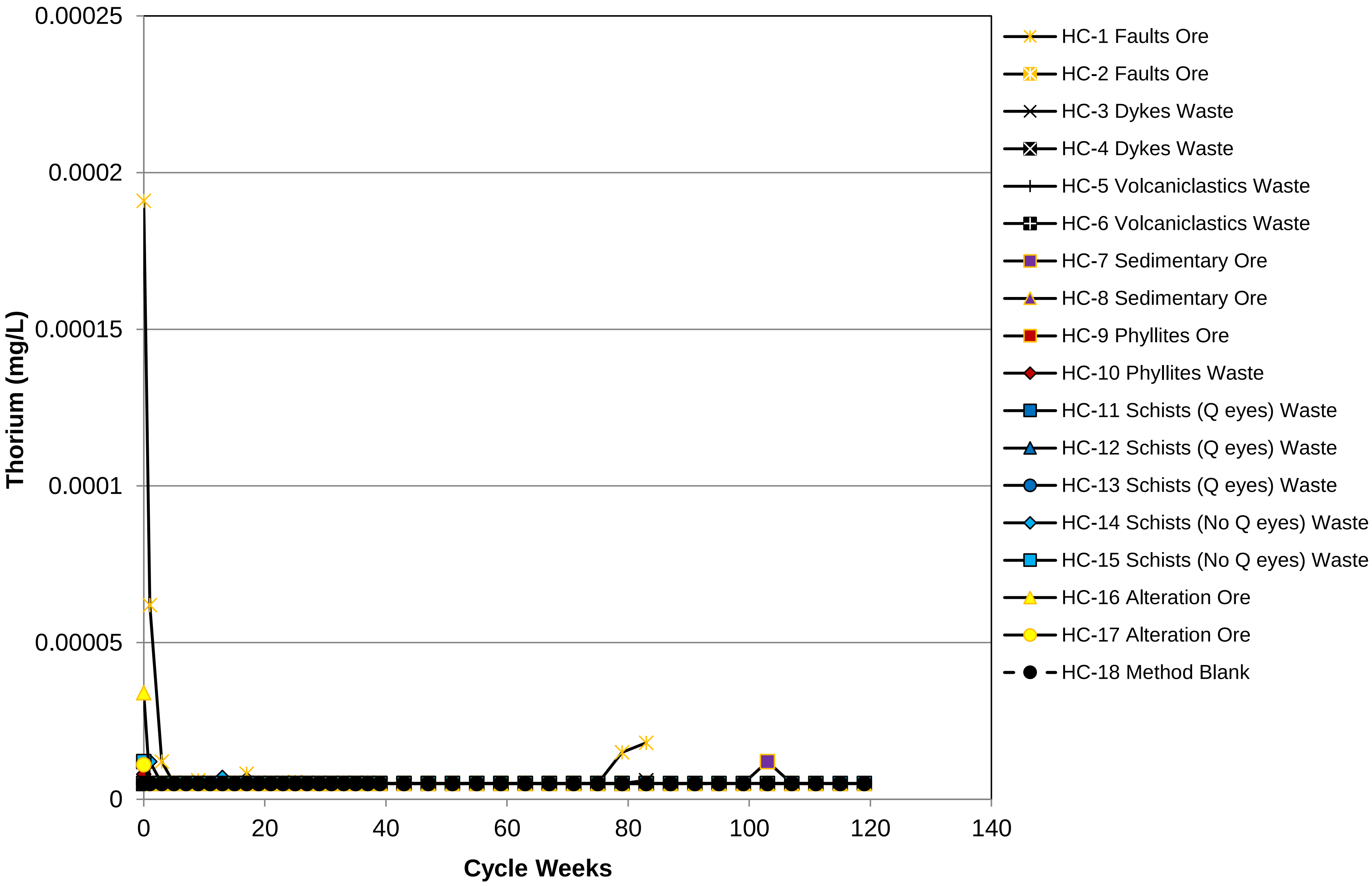

60
50
40
30
20
10
0
Sodium (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

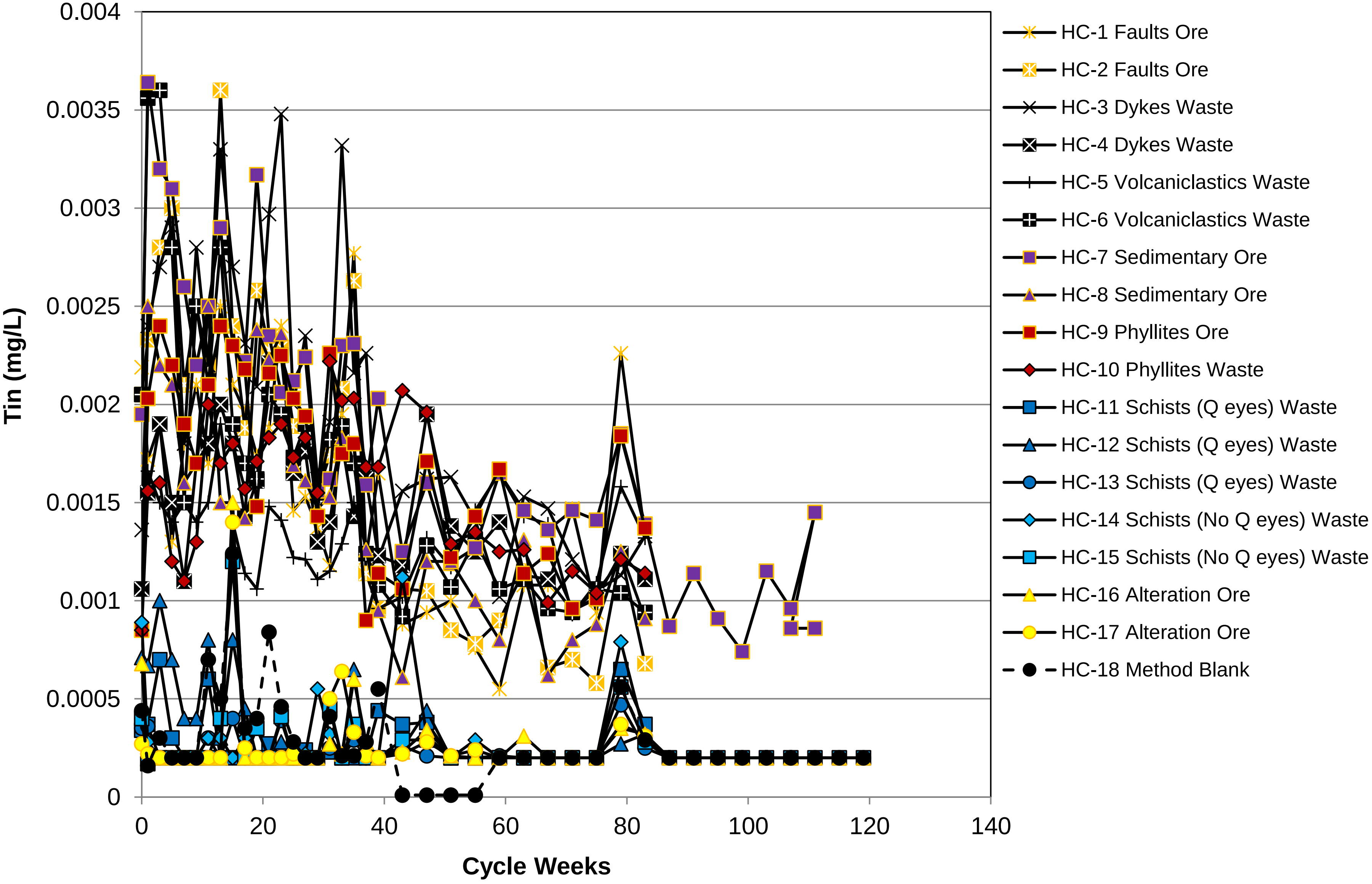

1.8
1.6
1.4
1.2
1
0.8
0.6
0.4
0.2
0
Strontium (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank


Sulphur (mg/L)
Cycle Weeks
0
10
20
30
40
50
60
70
80
90
0
20
40
60
80
100
120
140
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

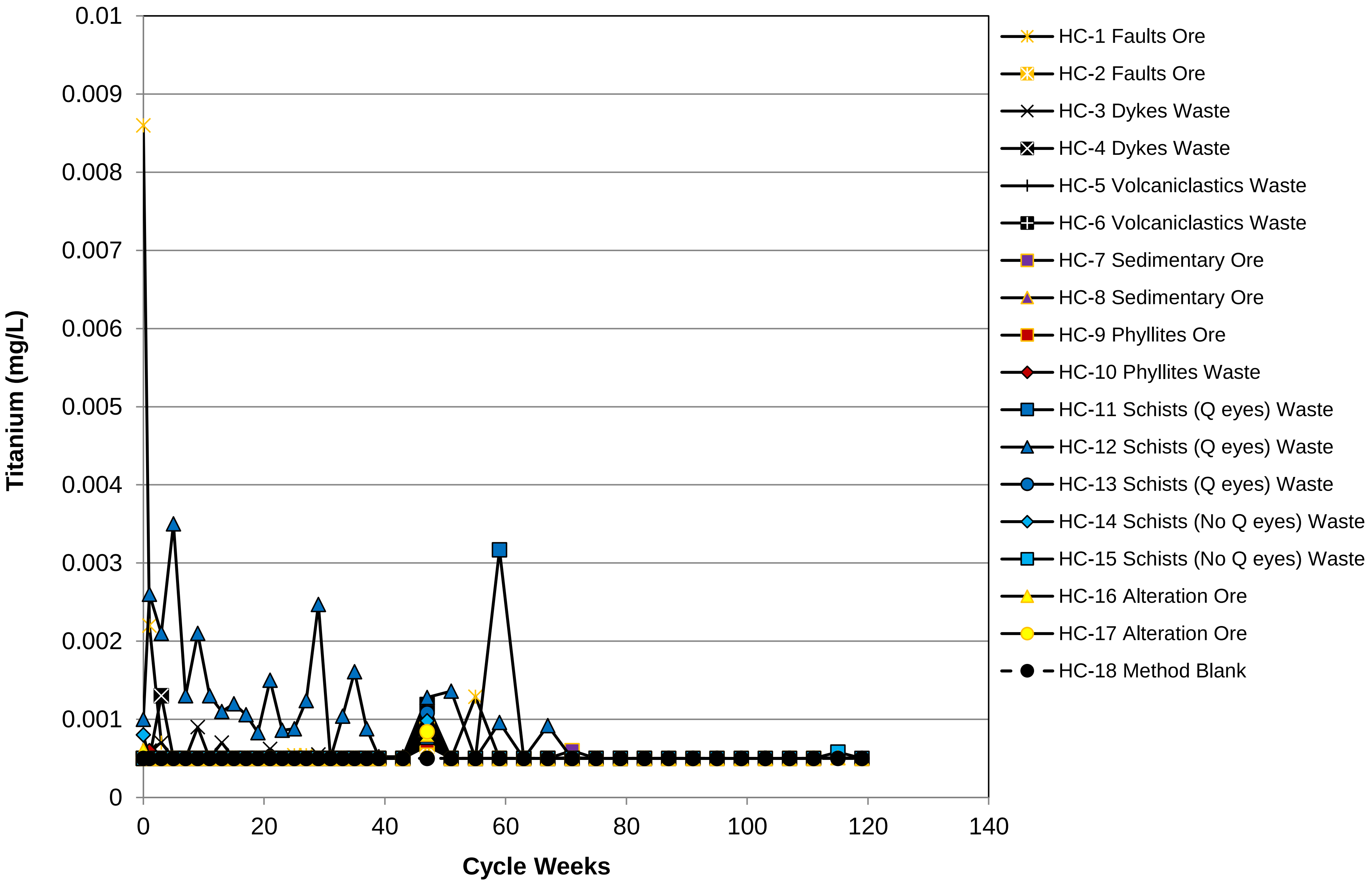

0.000045
0.00004
0.000035
0.00003
0.000025
0.00002
0.000015
0.00001
0.000005
0
Tellurium (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

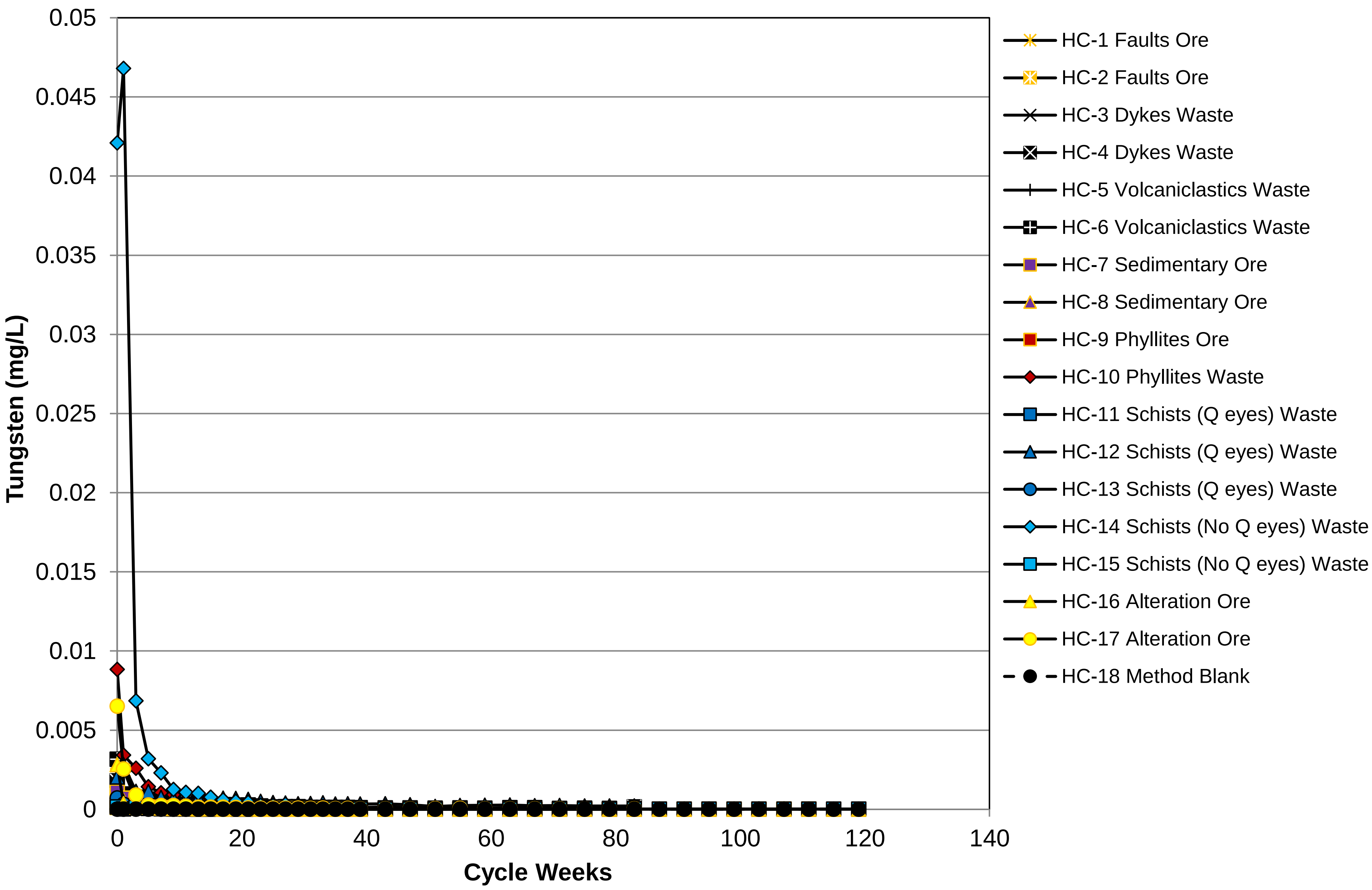

Thallium (mg/L)
0.0001
0.00009
0.00008
0.00007
0.00006
0.00005
0.00004
0.00003
0.00002
0.00001
0
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

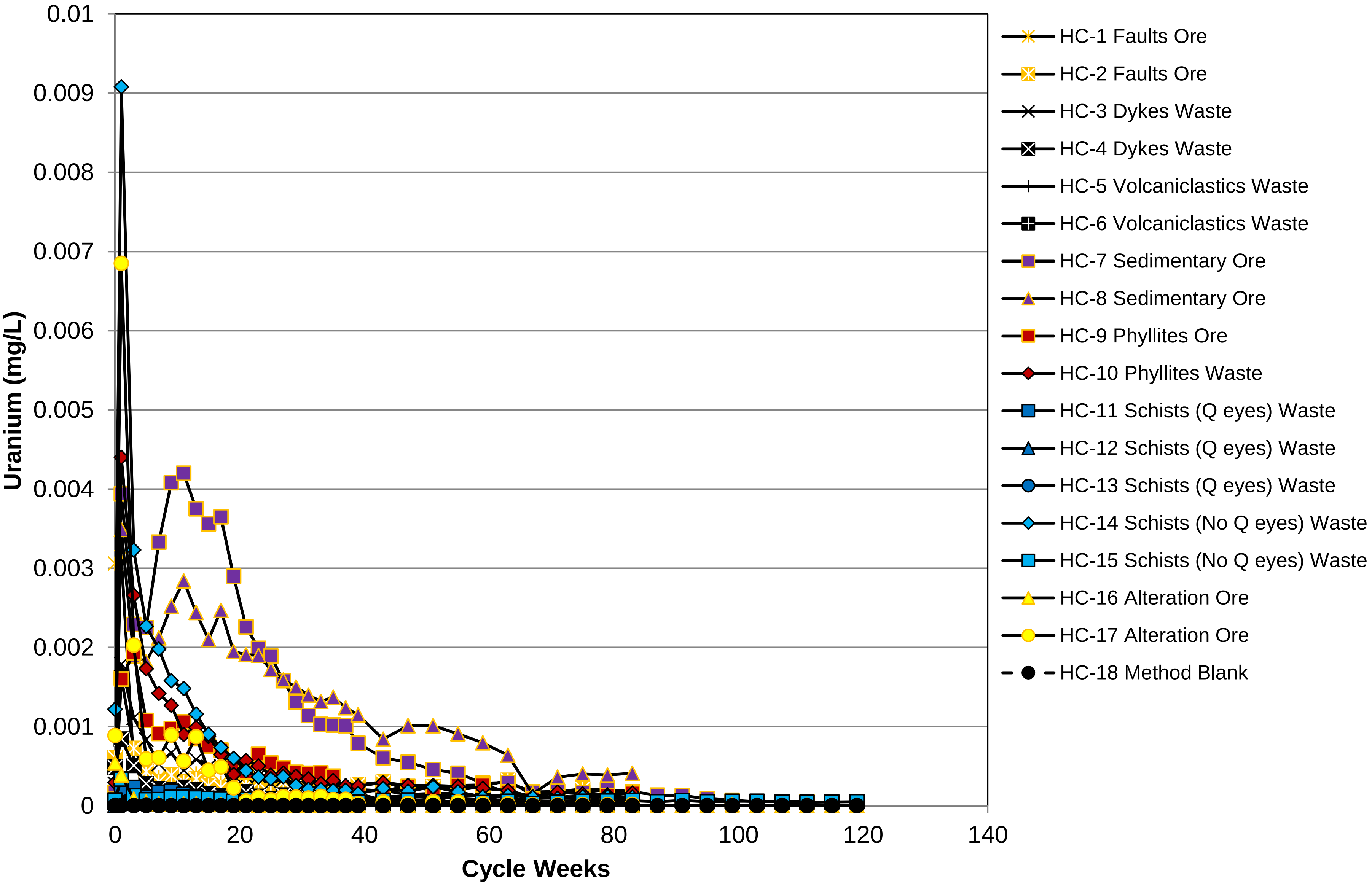

0.00025
0.0002
0.00015
0.0001
0.00005
0
Thorium (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

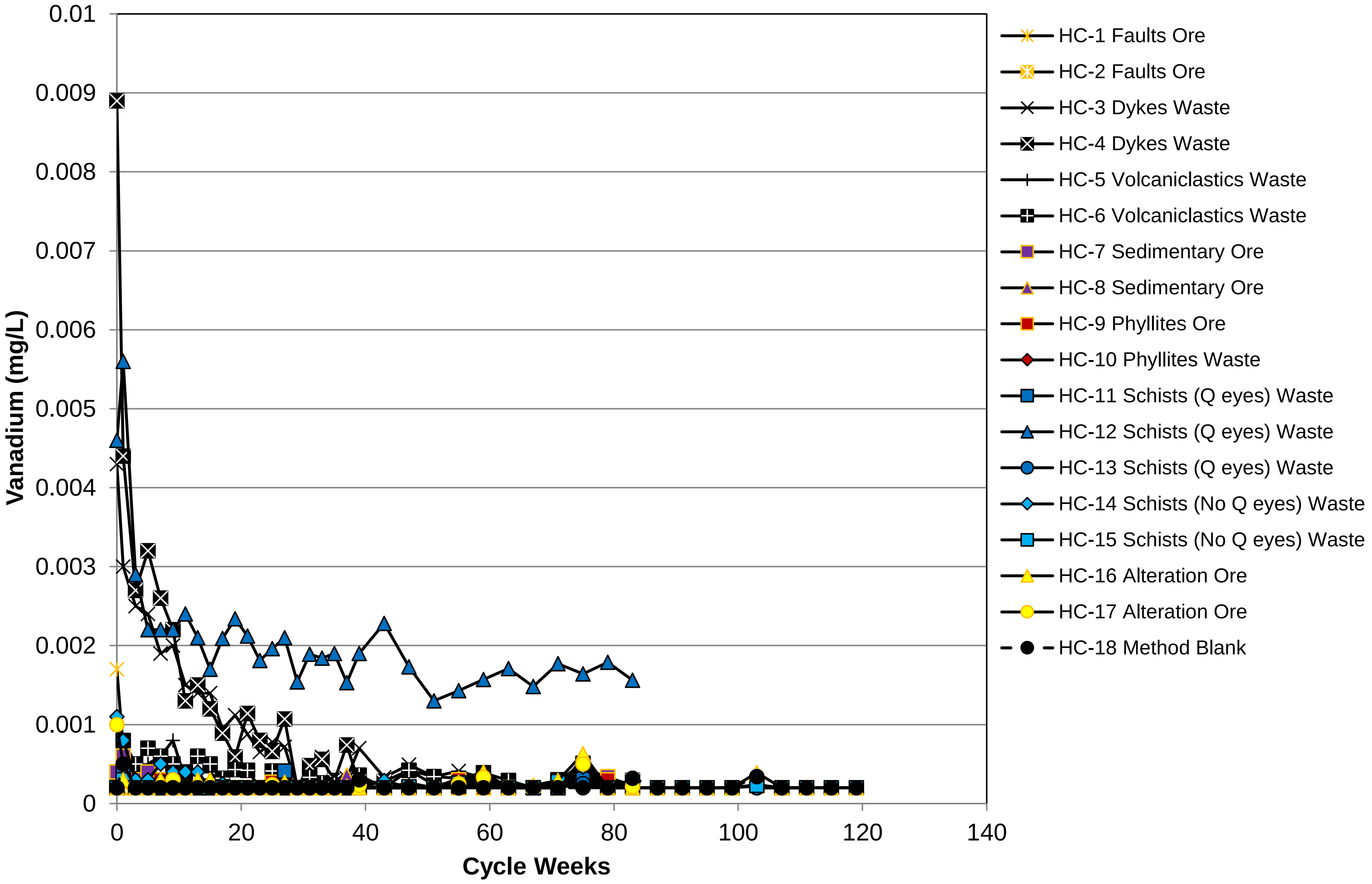


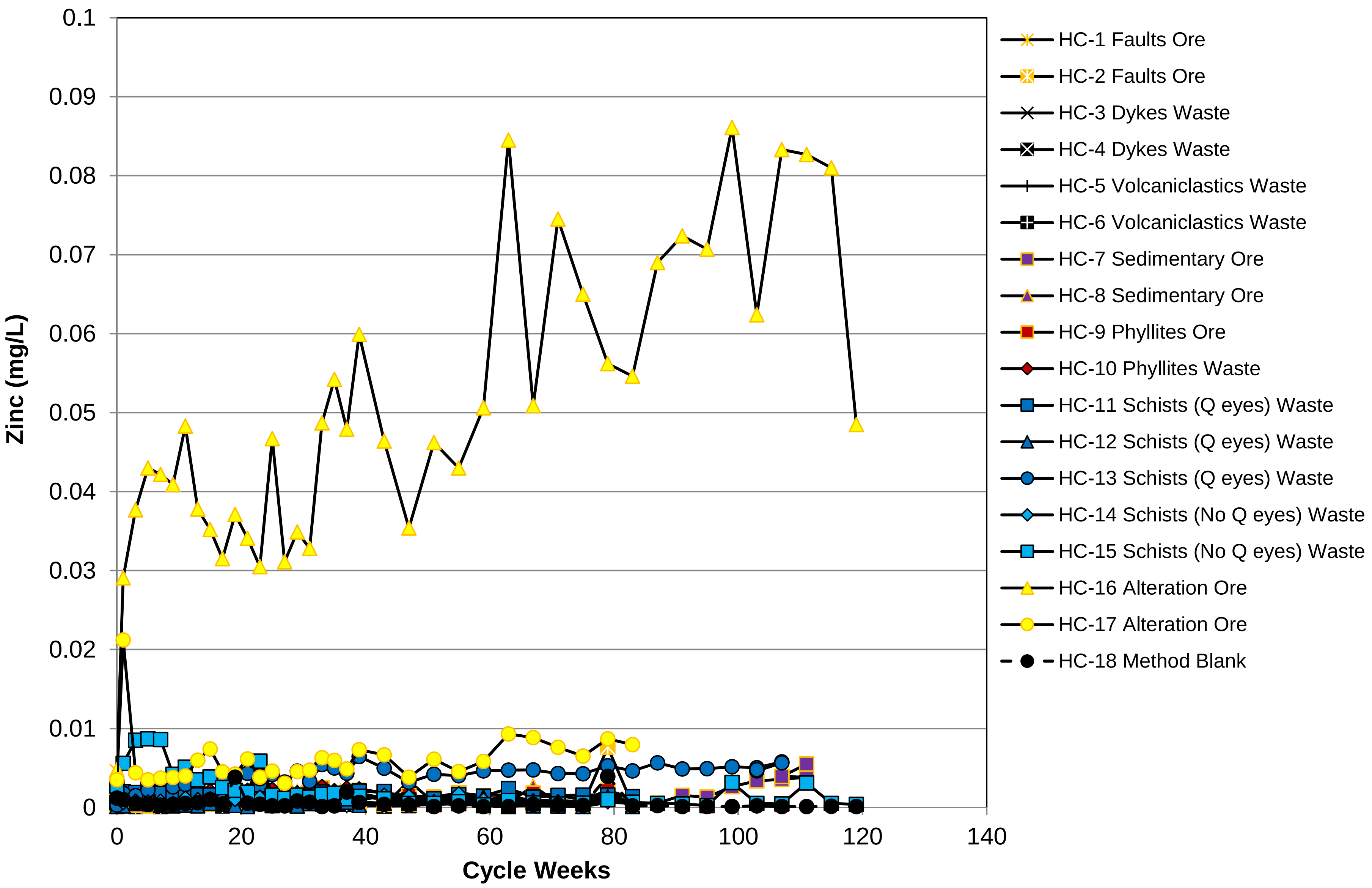
Tin (mg/L)
0.004
0.0035
0.003
0.0025
0.002
0.0015
0.001
0.0005
0
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

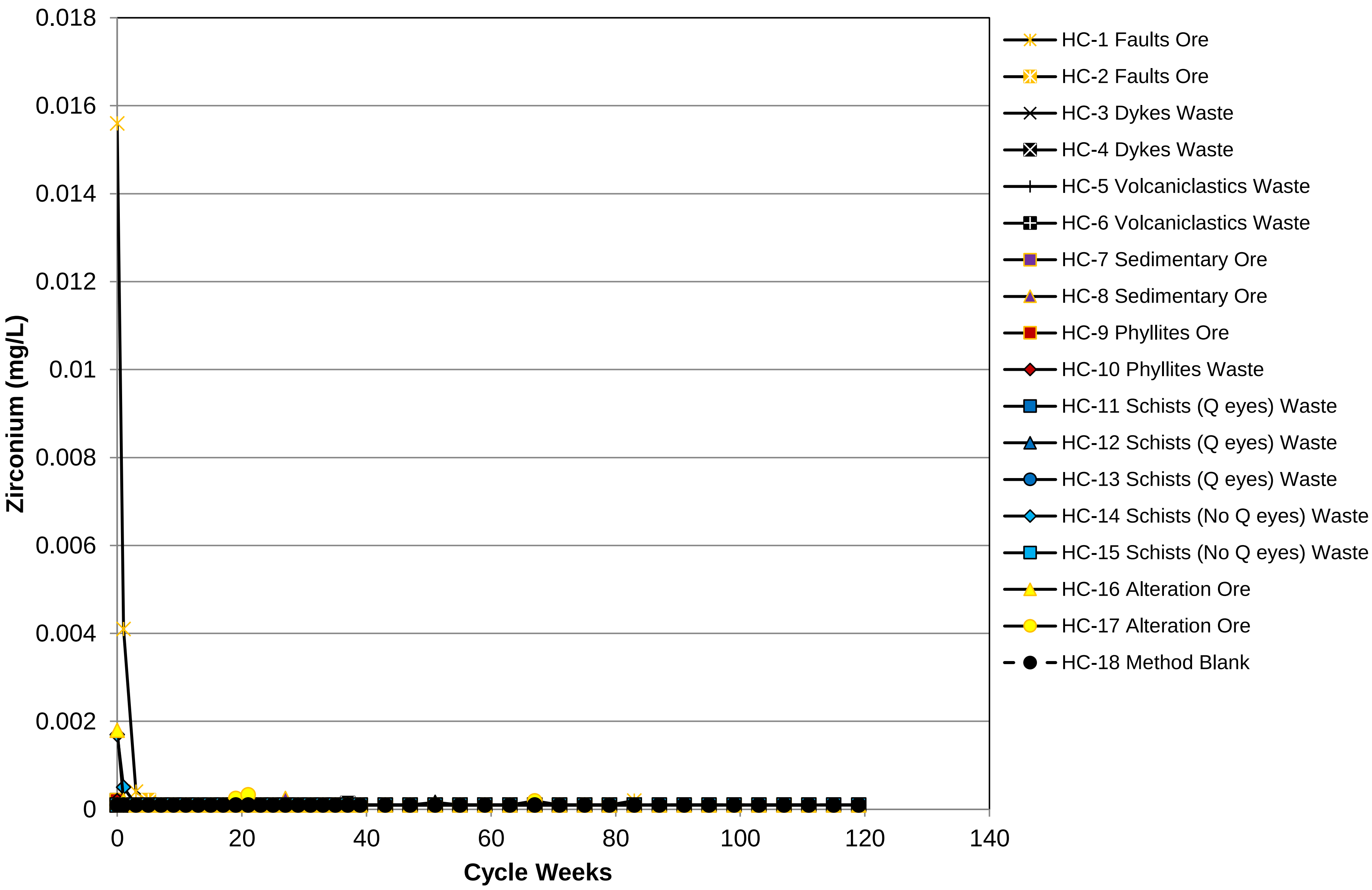

Titanium (mg/L)
0.01
0.009
0.008
0.007
0.006
0.005
0.004
0.003
0.002
0.001
0
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

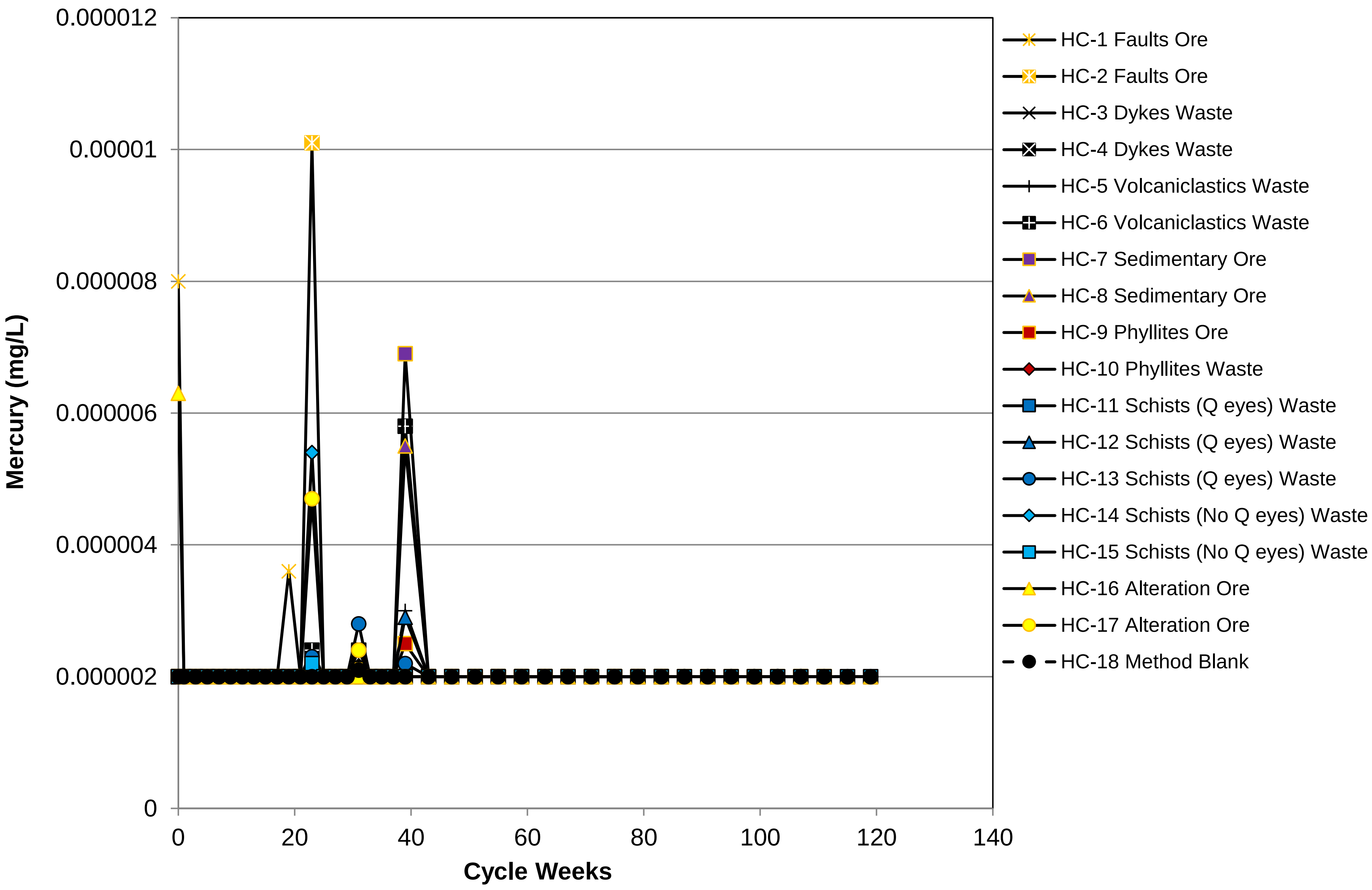

0.05
0.045
0.04
0.035
0.03
0.025
0.02
0.015
0.01
0.005
0
Tungsten (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank


0.01
0.009
0.008
0.007
0.006
0.005
0.004
0.003
0.002
0.001
0
Uranium (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

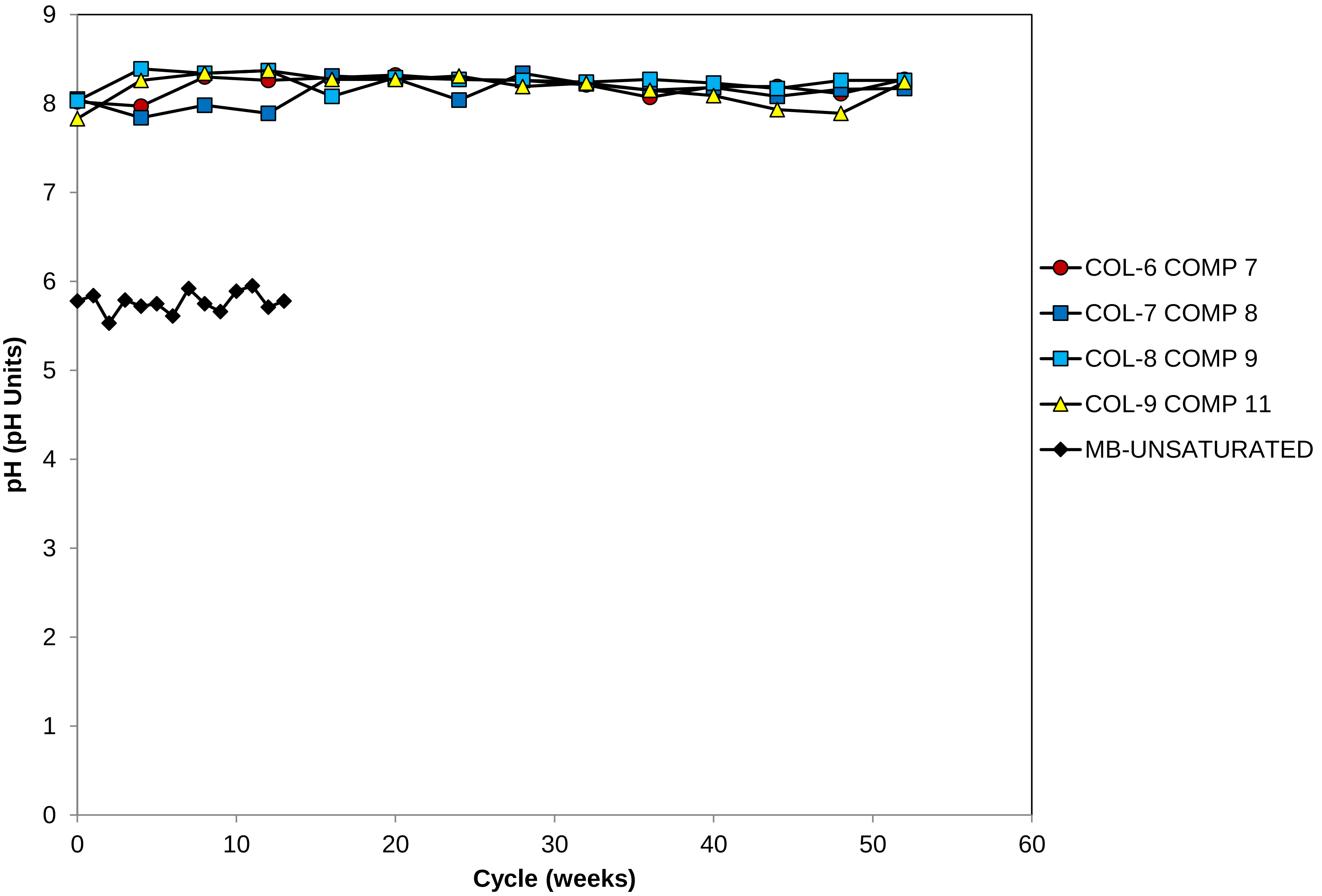

Vanadium (mg/L)
0.01
0.009
0.008
0.007
0.006
0.005
0.004
0.003
0.002
0.001
0
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

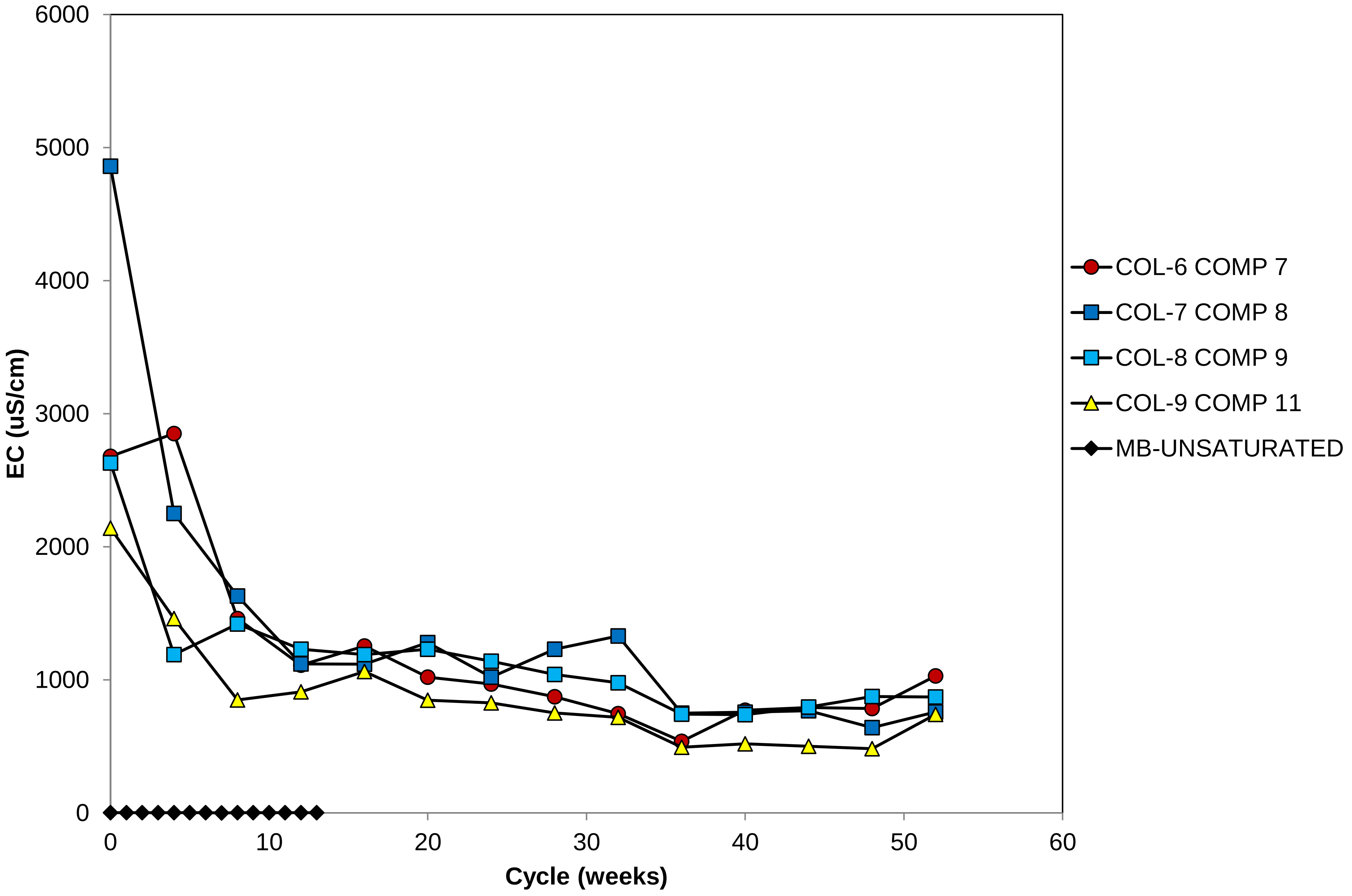

0.1
0.09
0.08
0.07
0.06
0.05
0.04
0.03
0.02
0.01
0
Zinc (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

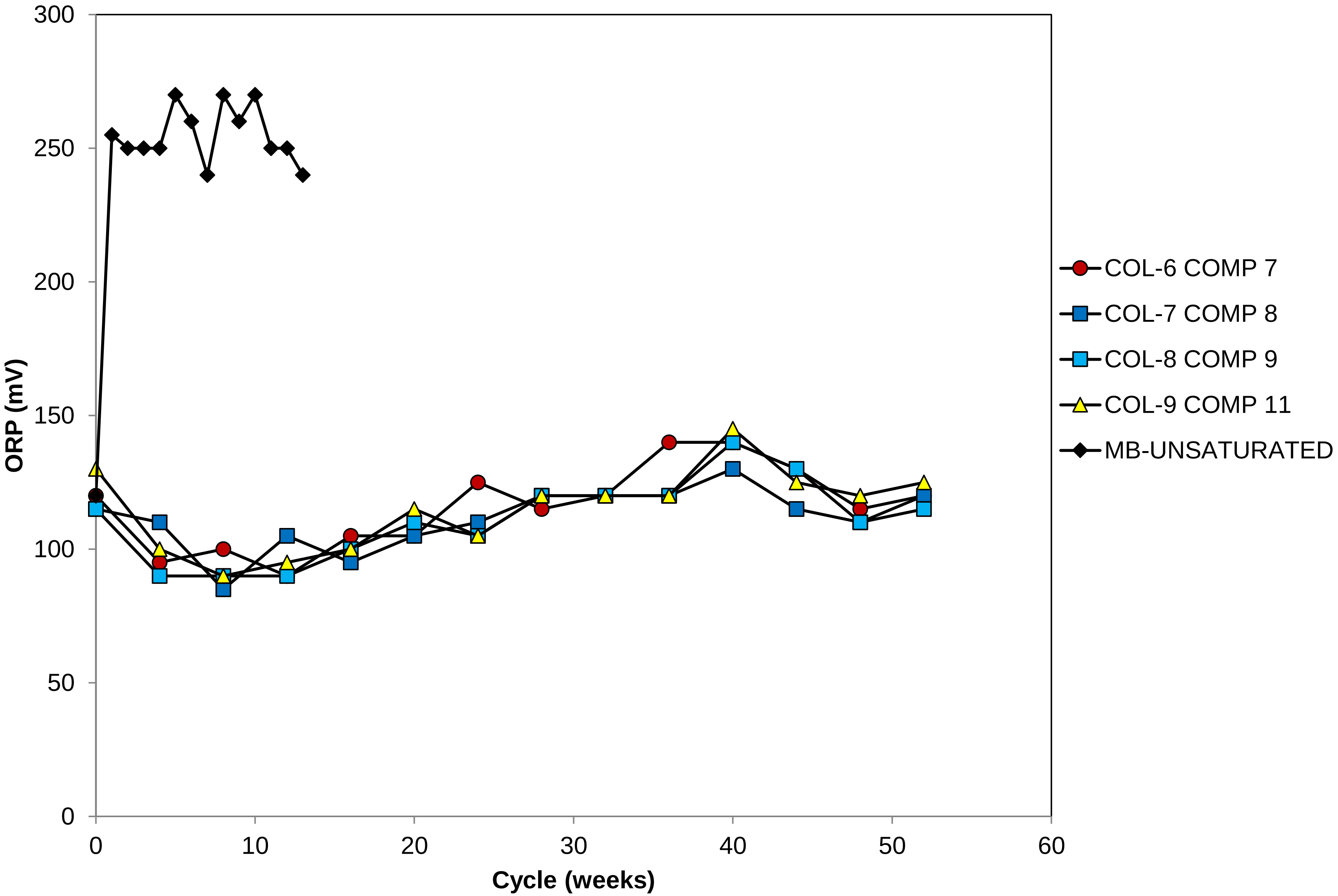

0.018
0.016
0.014
0.012
0.01
0.008
0.006
0.004
0.002
0
Zirconium (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

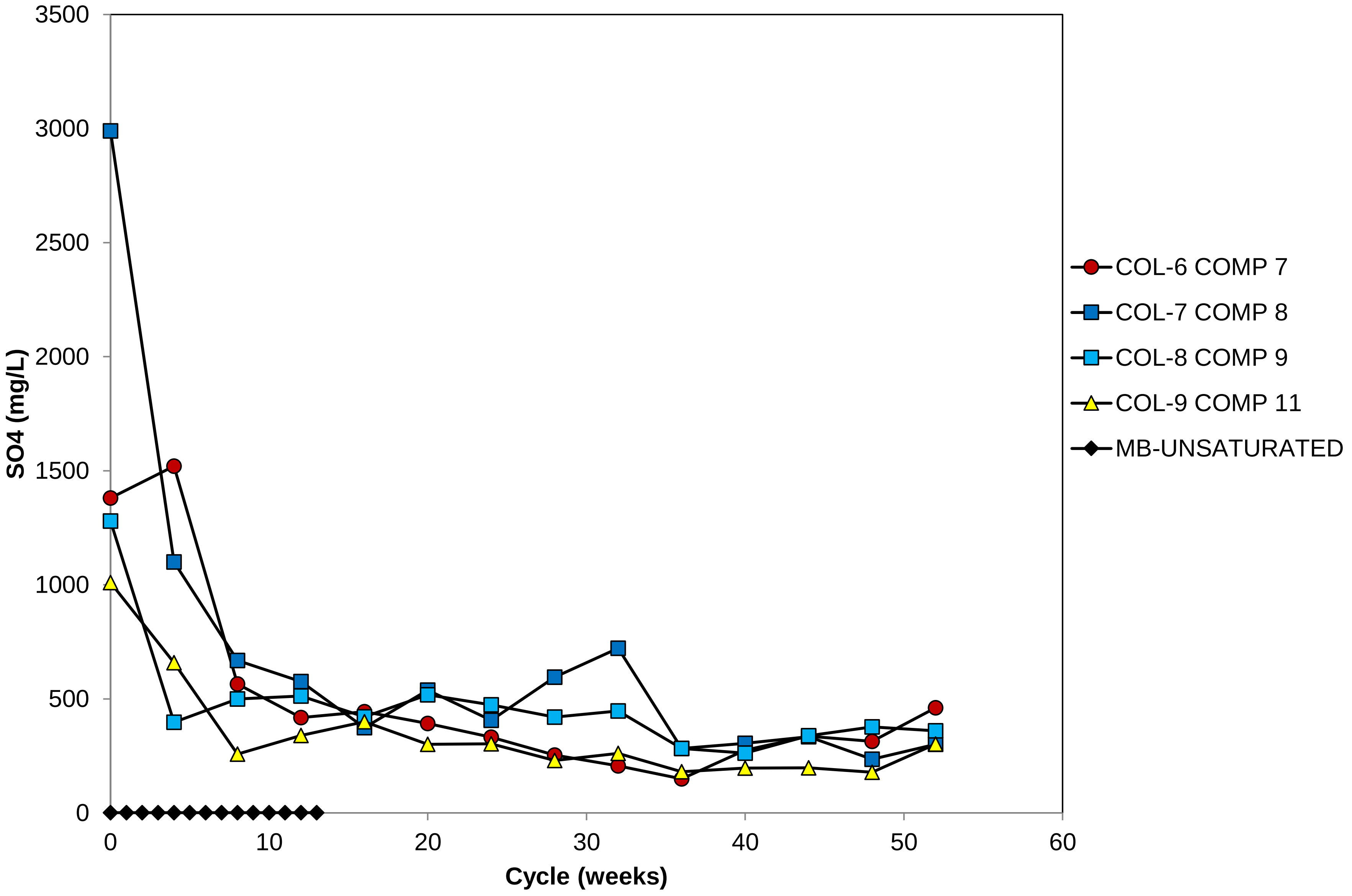

0.000012
0.00001
0.000008
0.000006
0.000004
0.000002
0
Mercury (mg/L)
0
20
40
60
80
100
120
140
Cycle Weeks
HC-1 Faults Ore
HC-2 Faults Ore
HC-3 Dykes Waste
HC-4 Dykes Waste
HC-5 Volcaniclastics Waste
HC-6 Volcaniclastics Waste
HC-7 Sedimentary Ore
HC-8 Sedimentary Ore
HC-9 Phyllites Ore
HC-10 Phyllites Waste
HC-11 Schists (Q eyes) Waste
HC-12 Schists (Q eyes) Waste
HC-13 Schists (Q eyes) Waste
HC-14 Schists (No Q eyes) Waste
HC-15 Schists (No Q eyes) Waste
HC-16 Alteration Ore
HC-17 Alteration Ore
HC-18 Method Blank

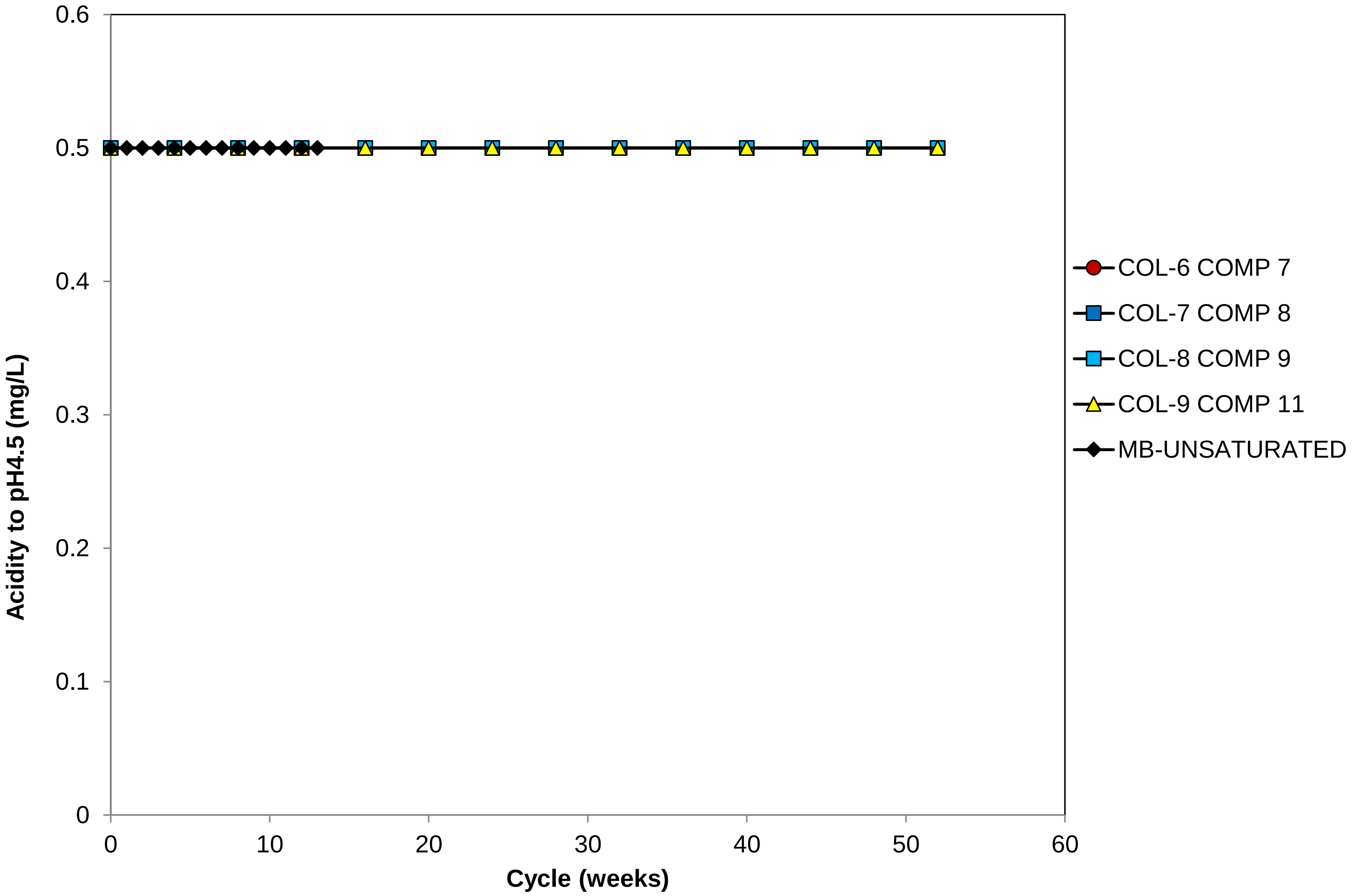
C2: Unsaturated Columns

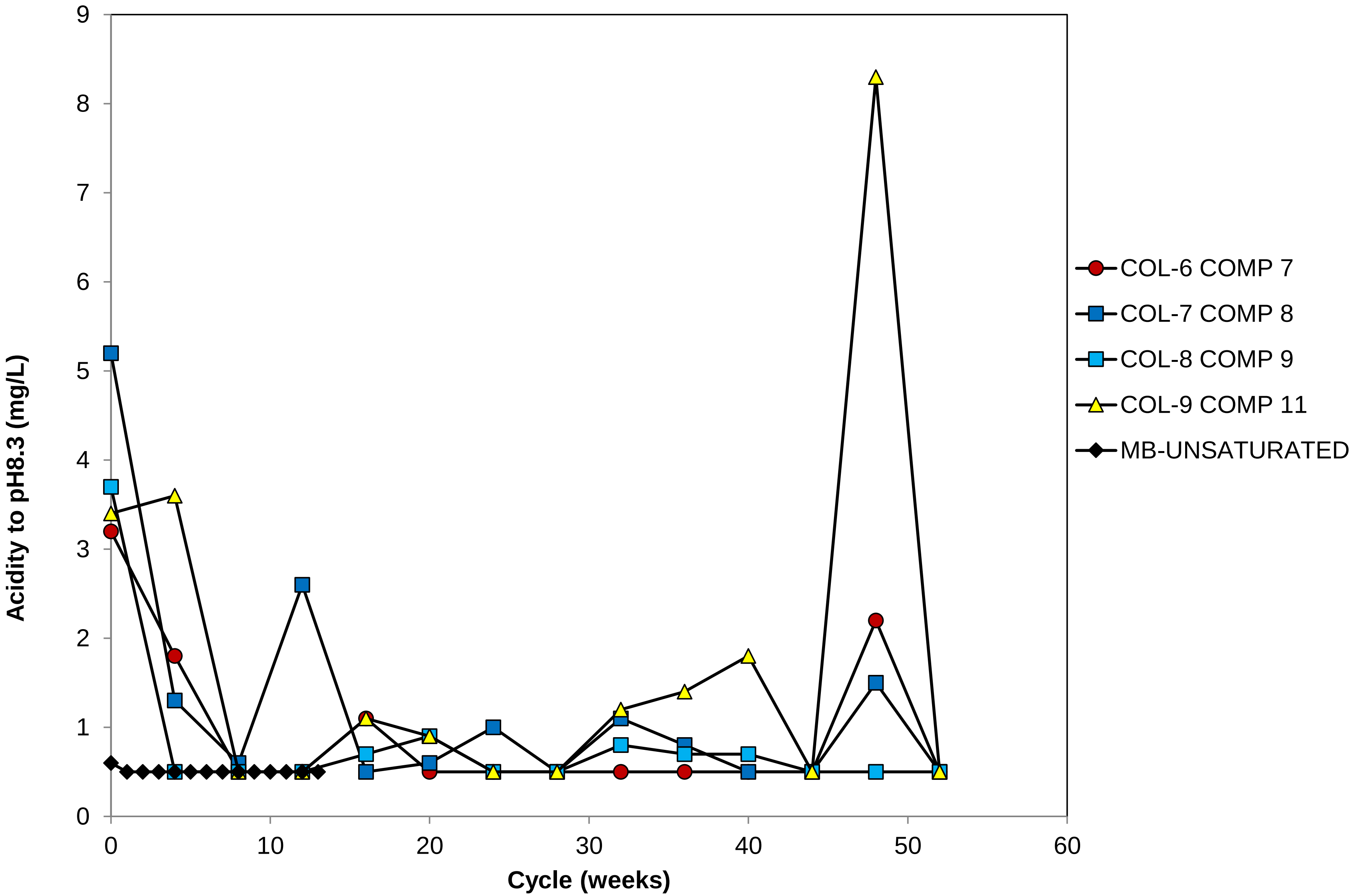

9
8
7
6
5
4
3
2
1
0
pH (pH Units)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED


6000
5000
4000
3000
2000
1000
0
EC (uS/cm)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

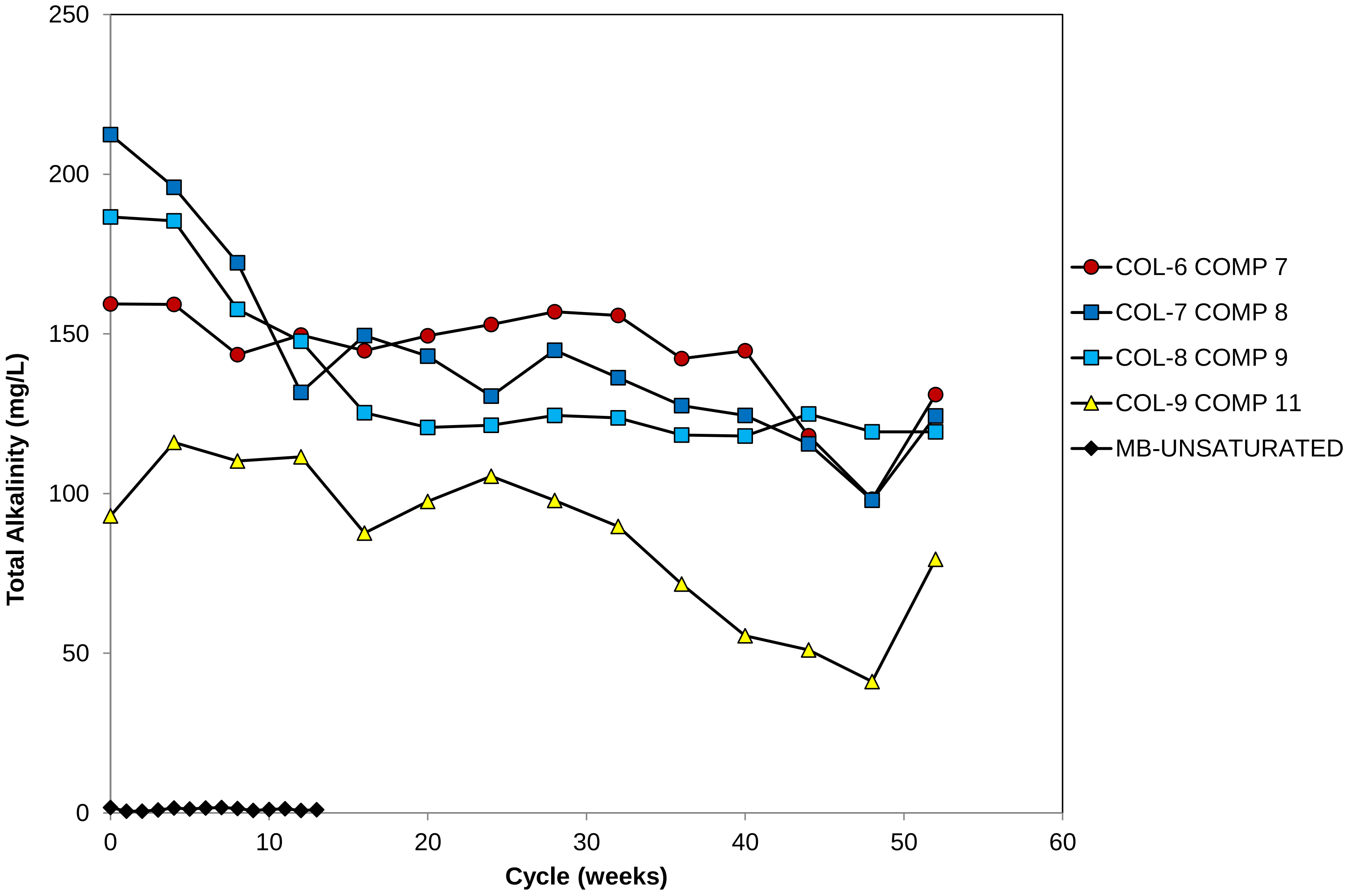

300
250
200
150
100
50
0
ORP (mV)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

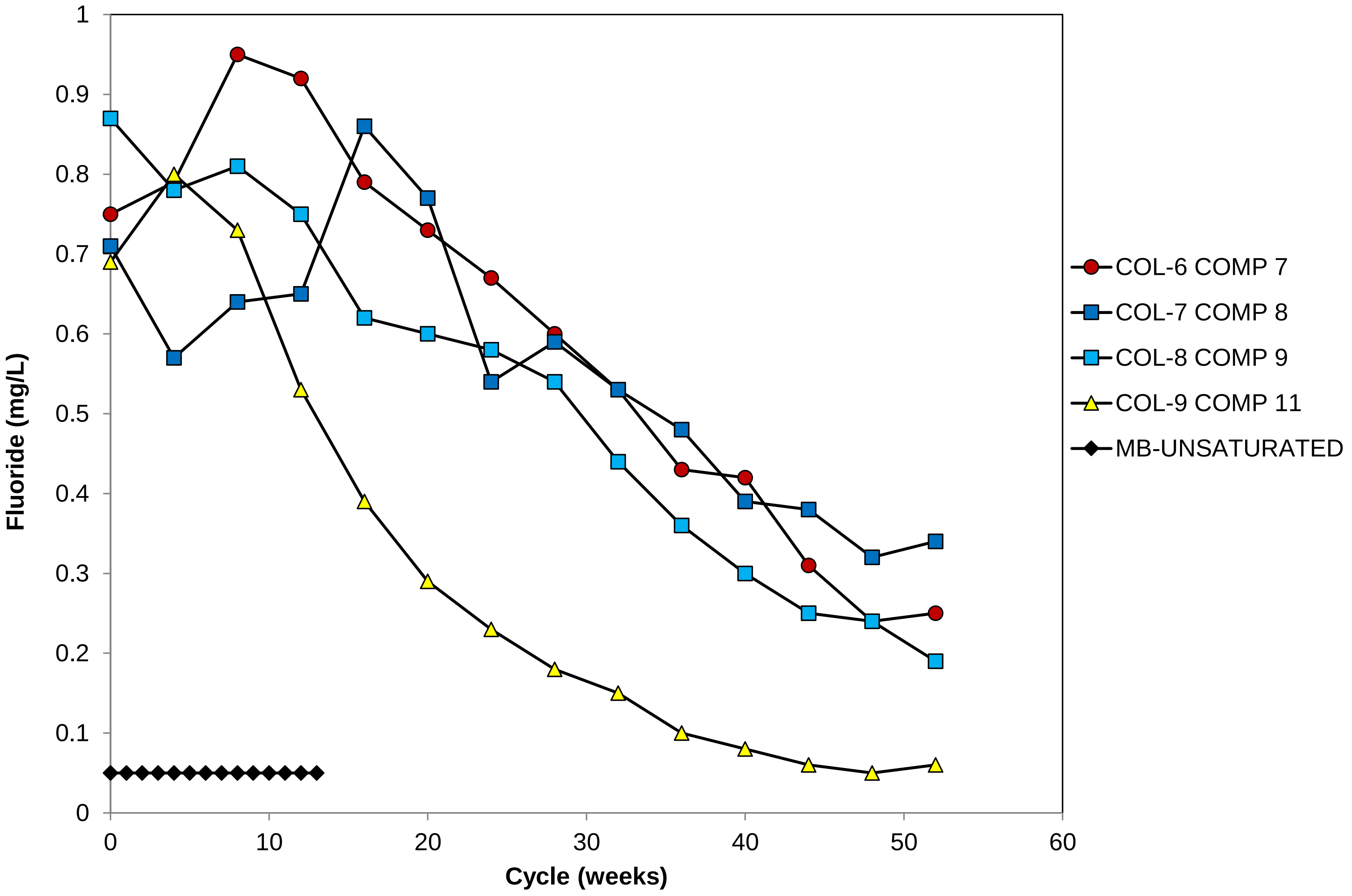

SO4 (mg/L)
3500
3000
2500
2000
1500
1000
500
0
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

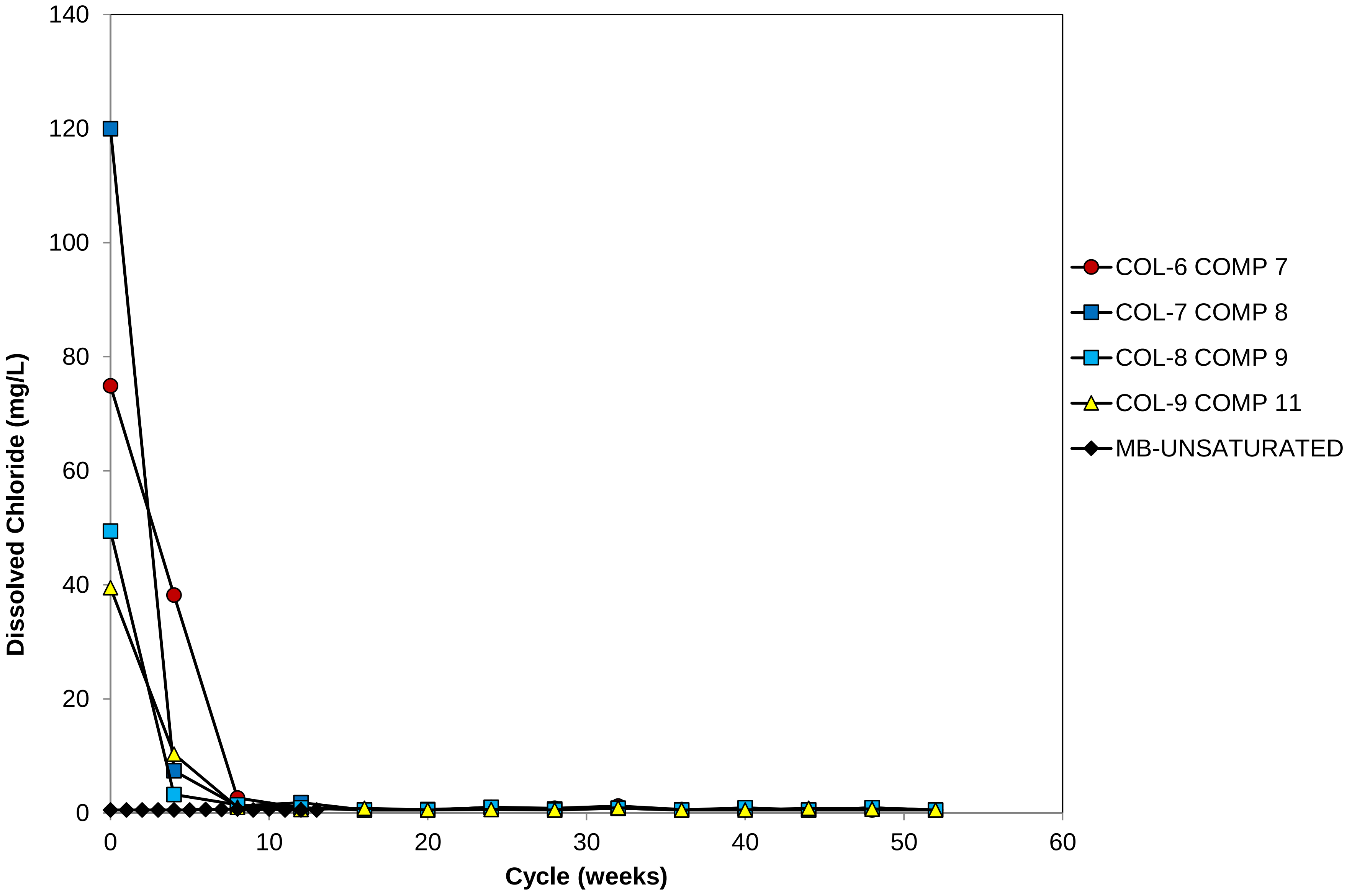

0.6
0.5
0.4
0.3
0.2
0.1
0
Acidity to pH4.5 (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

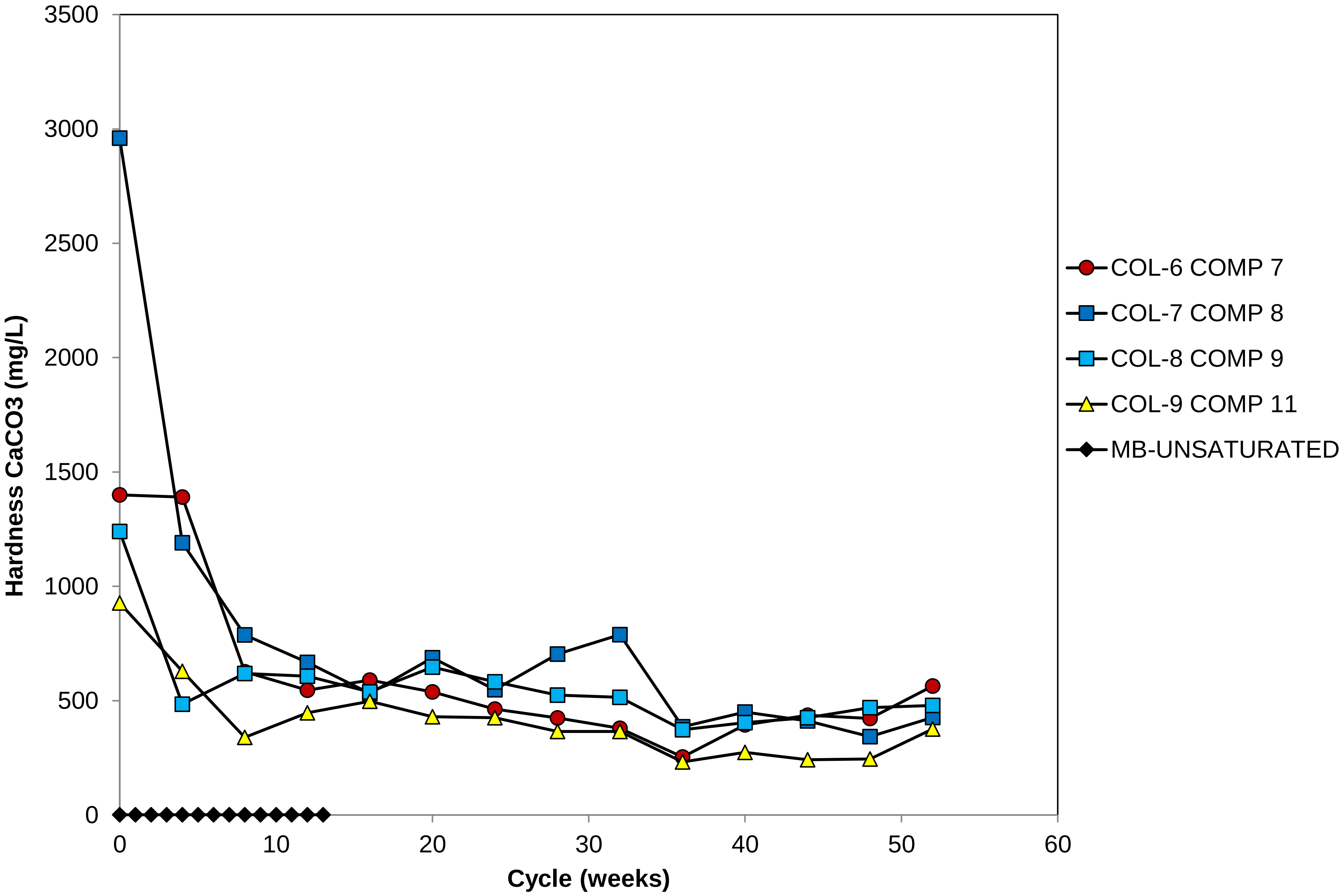


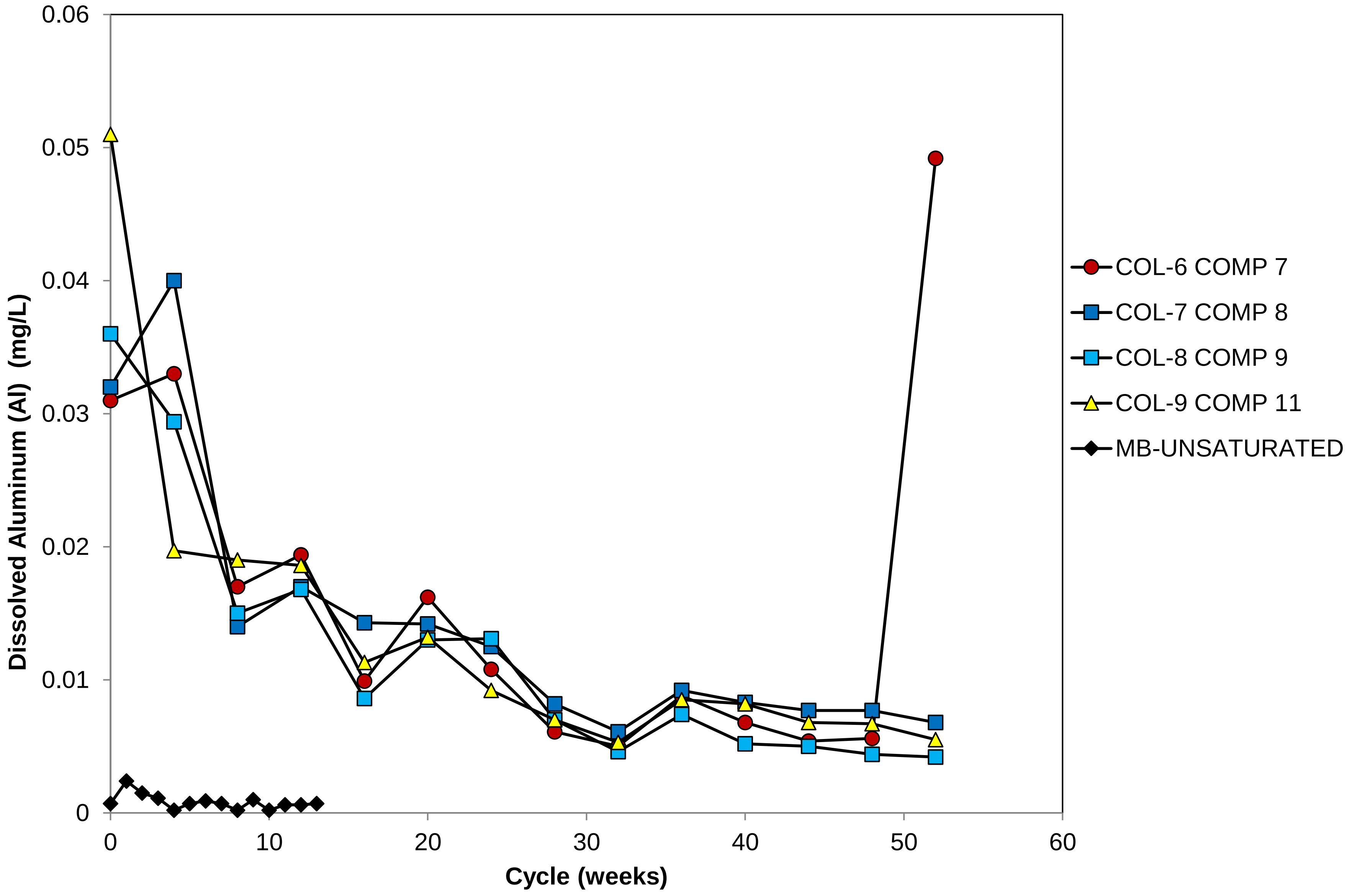
9
8
7
6
5
4
3
2
1
0
Acidity to pH8.3 (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

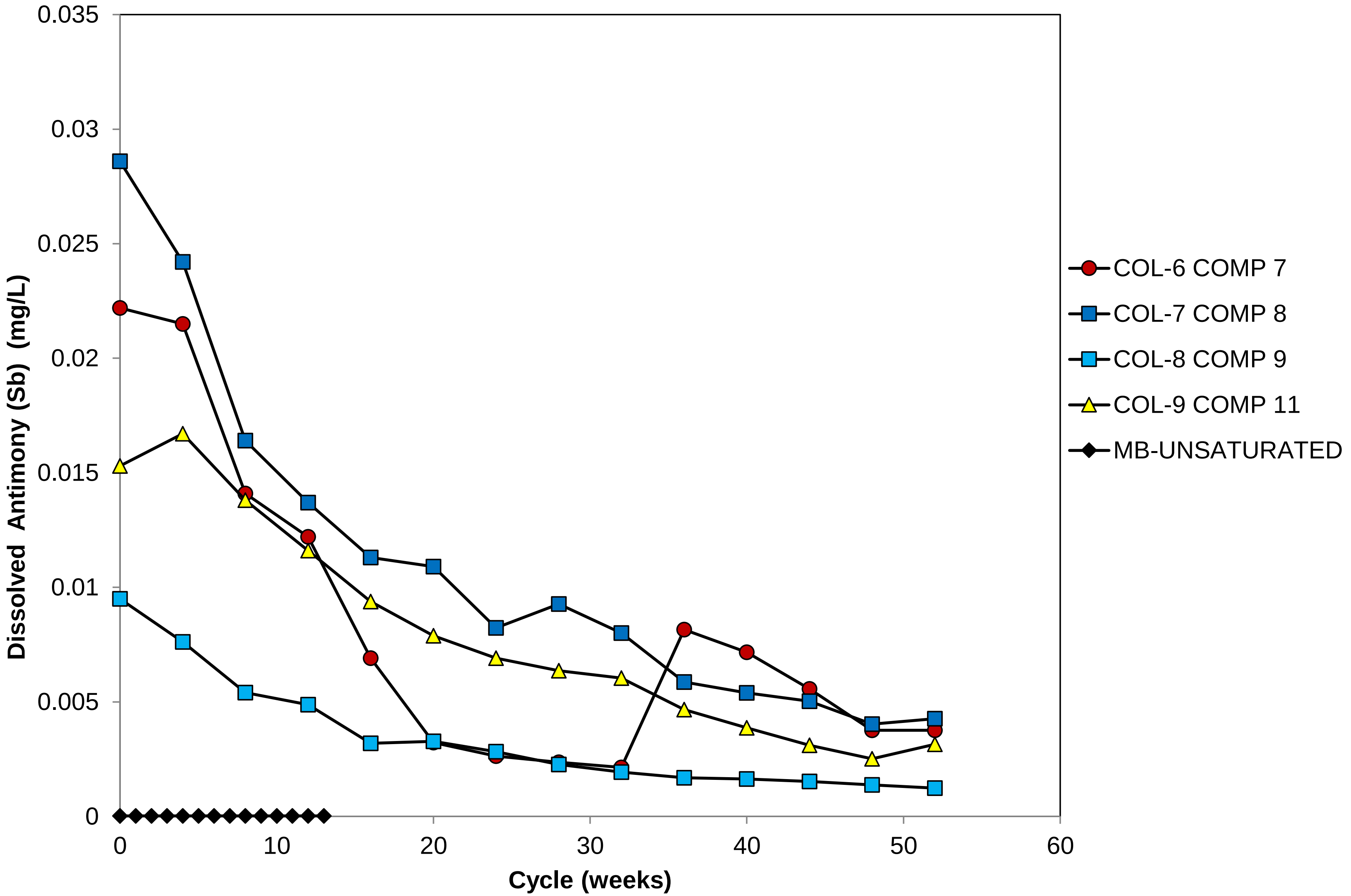

250
200
150
100
50
0
0
10
20
30
40
50
60
Total Alkalinity (mg/L)
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

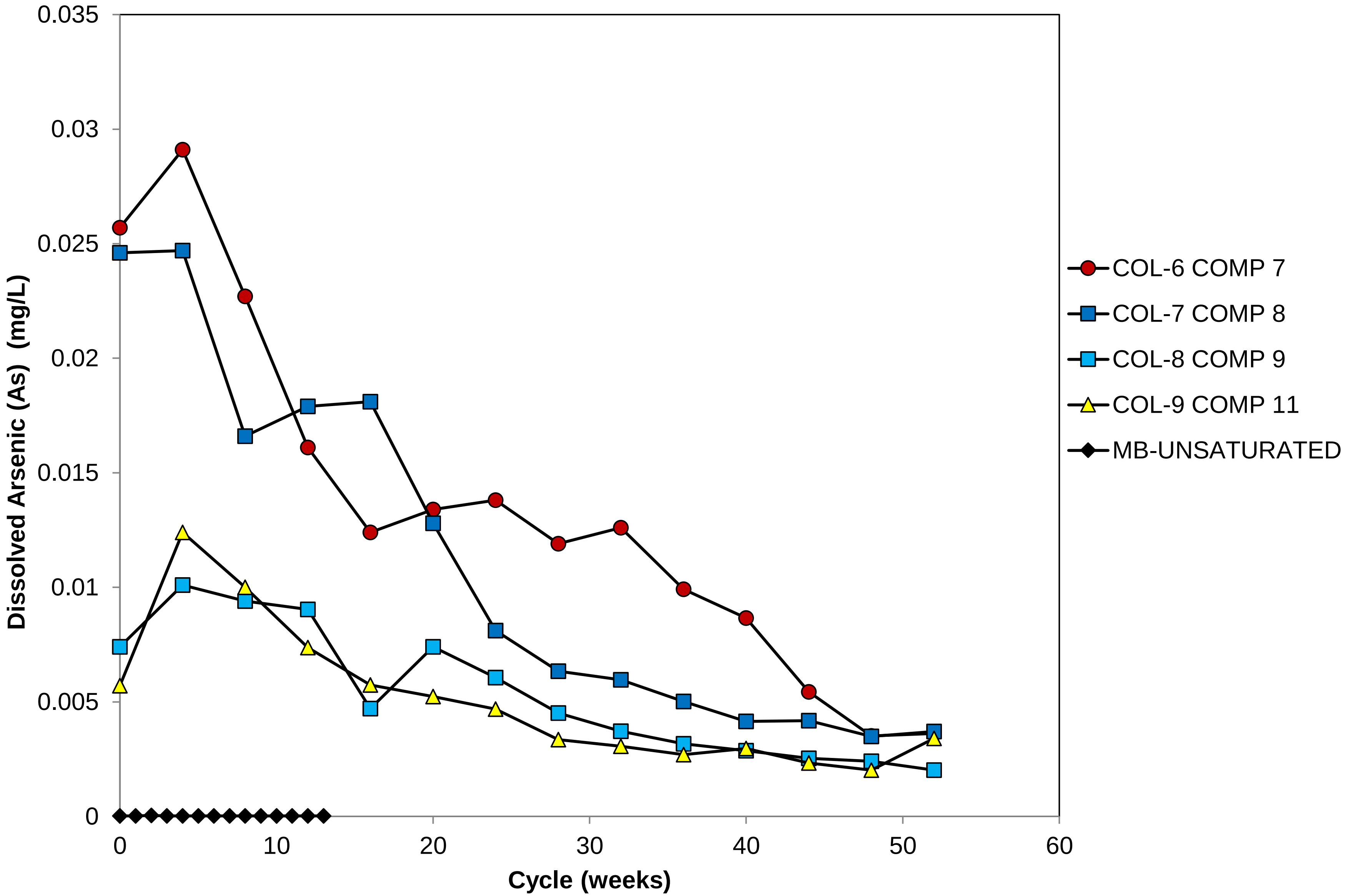

1
0.9
0.8
0.7
0.6
0.5
0.4
0.3
0.2
0.1
0
Fluoride (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

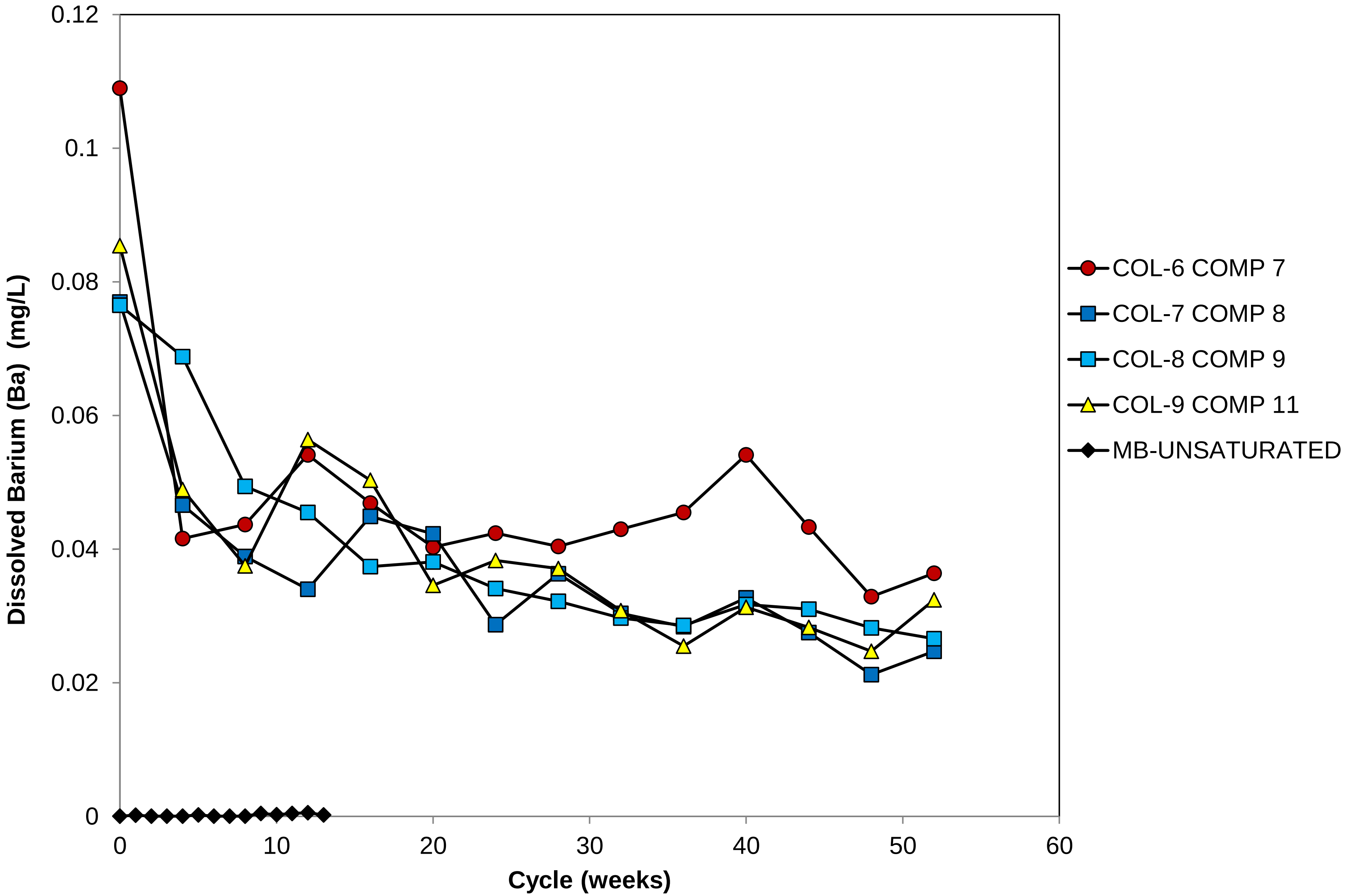

140
120
100
80
60
40
20
0
Dissolved Chloride (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED


3500
3000
2500
2000
1500
1000
500
0
Hardness CaCO3 (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

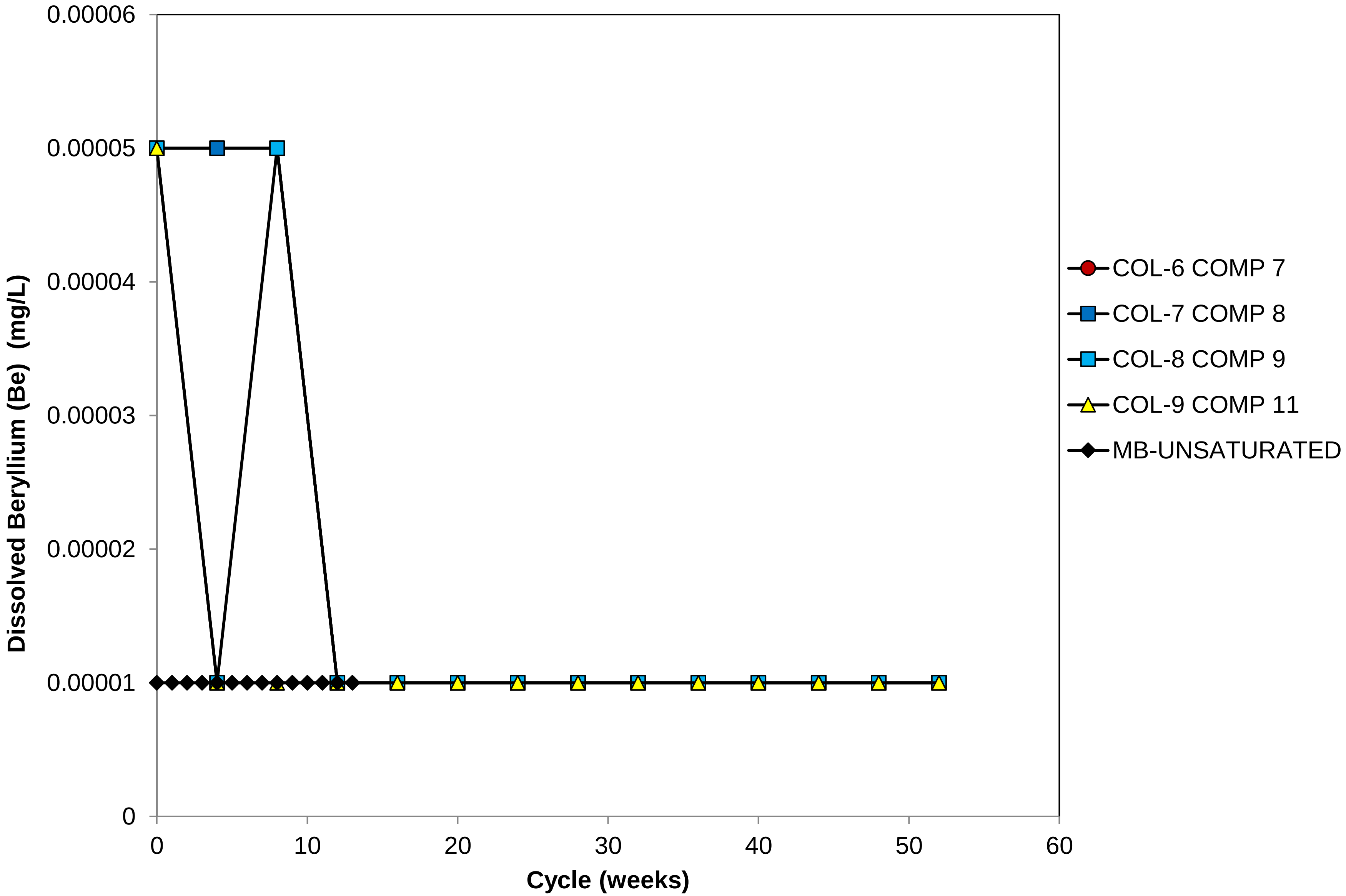

0.06
0.05
0.04
0.03
0.02
0.01
0
Dissolved Aluminum (Al) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

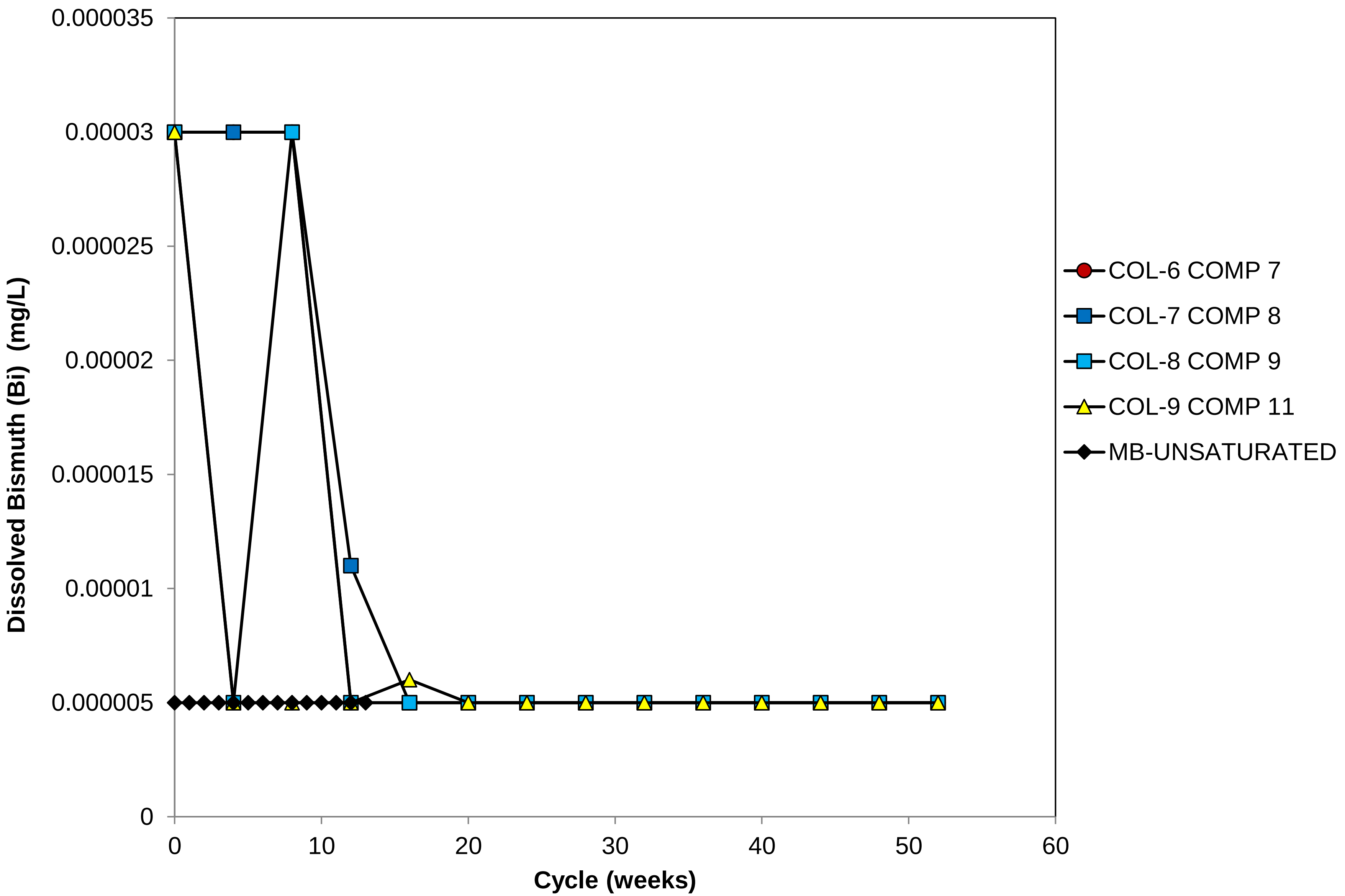

0.035
0.03
0.025
0.02
0.015
0.01
0.005
0
Dissolved Antimony (Sb) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

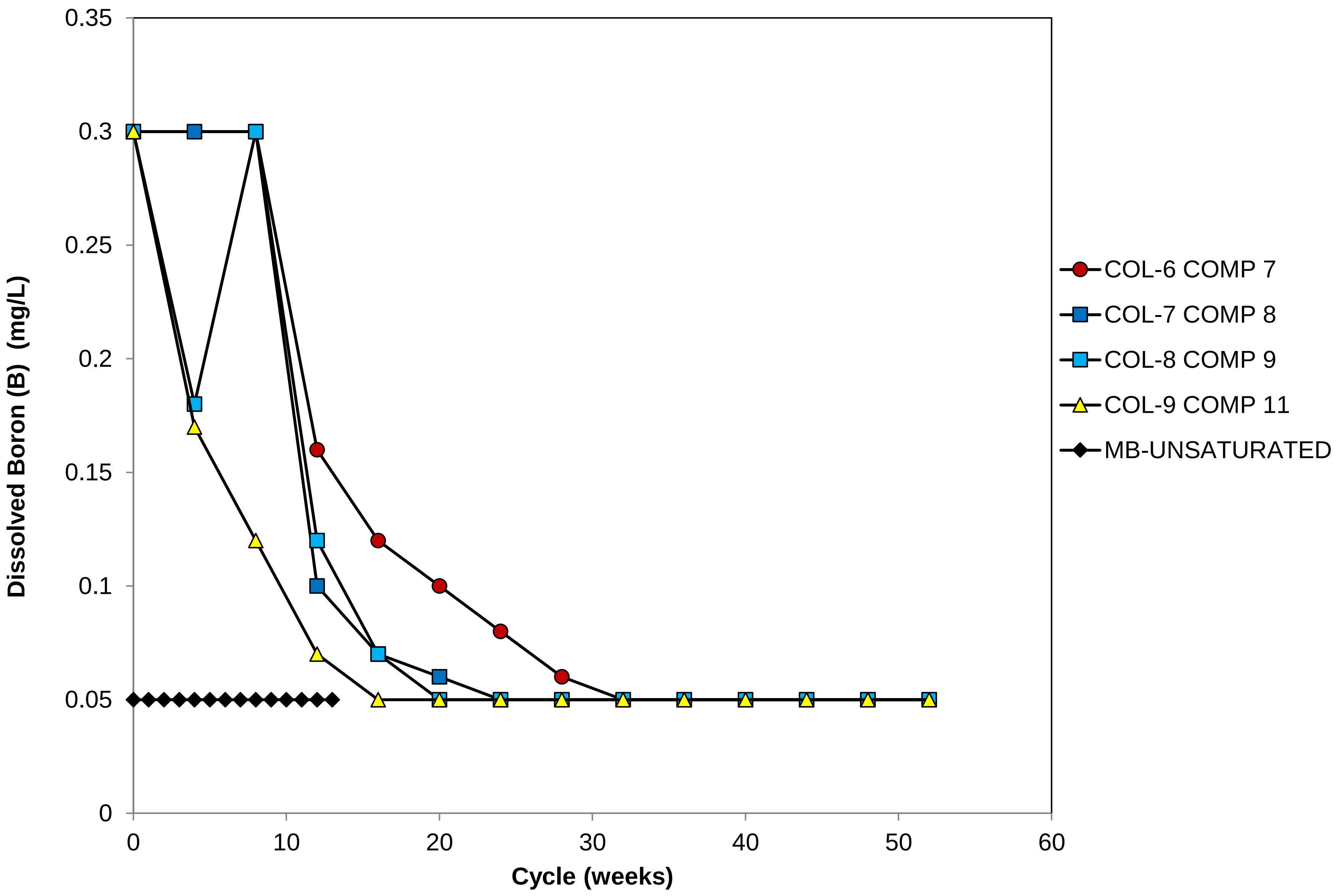

0.035
0.03
0.025
0.02
0.015
0.01
0.005
0
Dissolved Arsenic (As) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

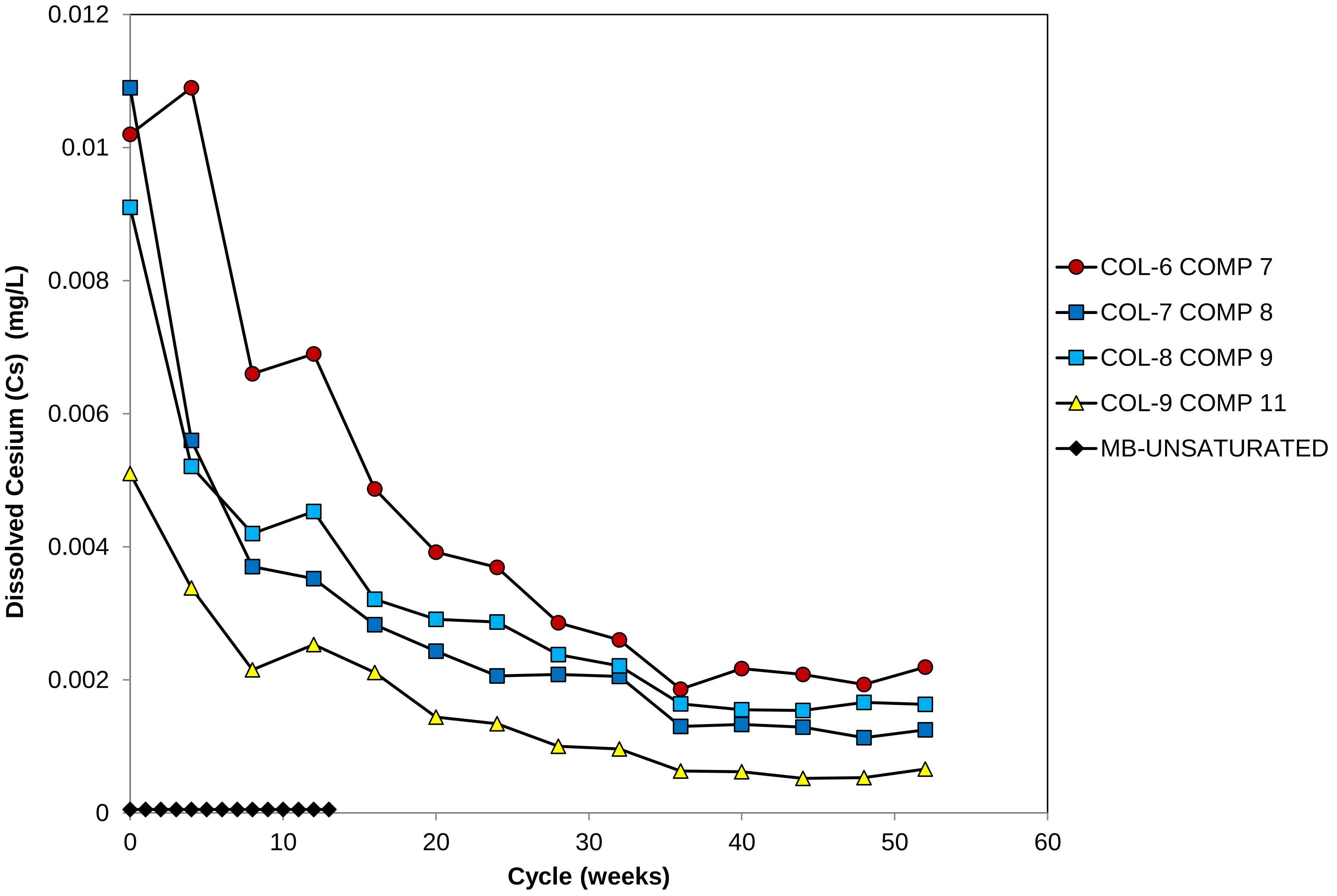


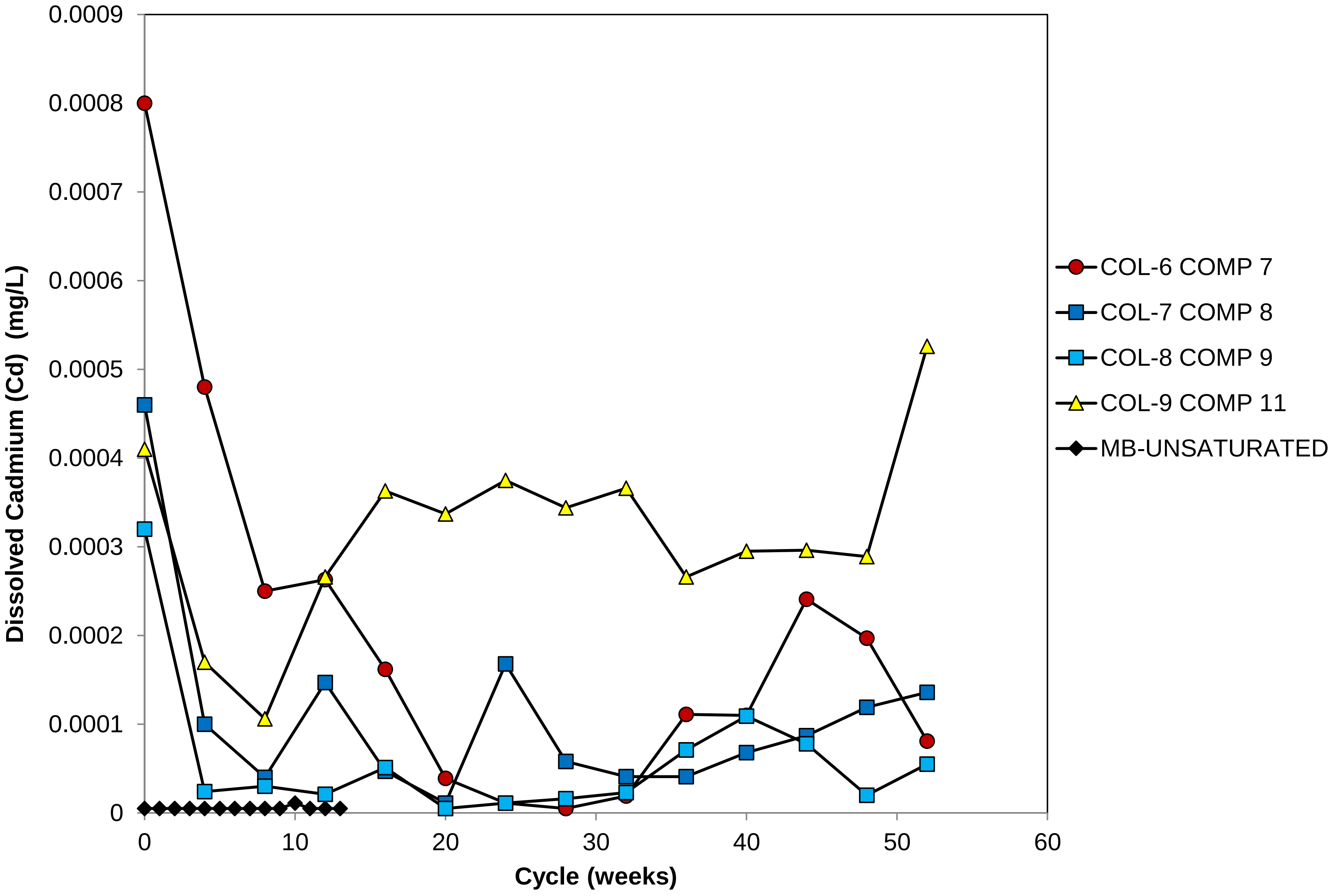
\\VAN-SVR0\Projects\01_SITES\Harper_Creek\1CH014.001_EA_Re-Submission\300_Reporting\310_Update SRK Report\030_Appendices\Appendix C\Working Files\Appendix C2_Harper_Columns_CONC charts_BarrelMaterial

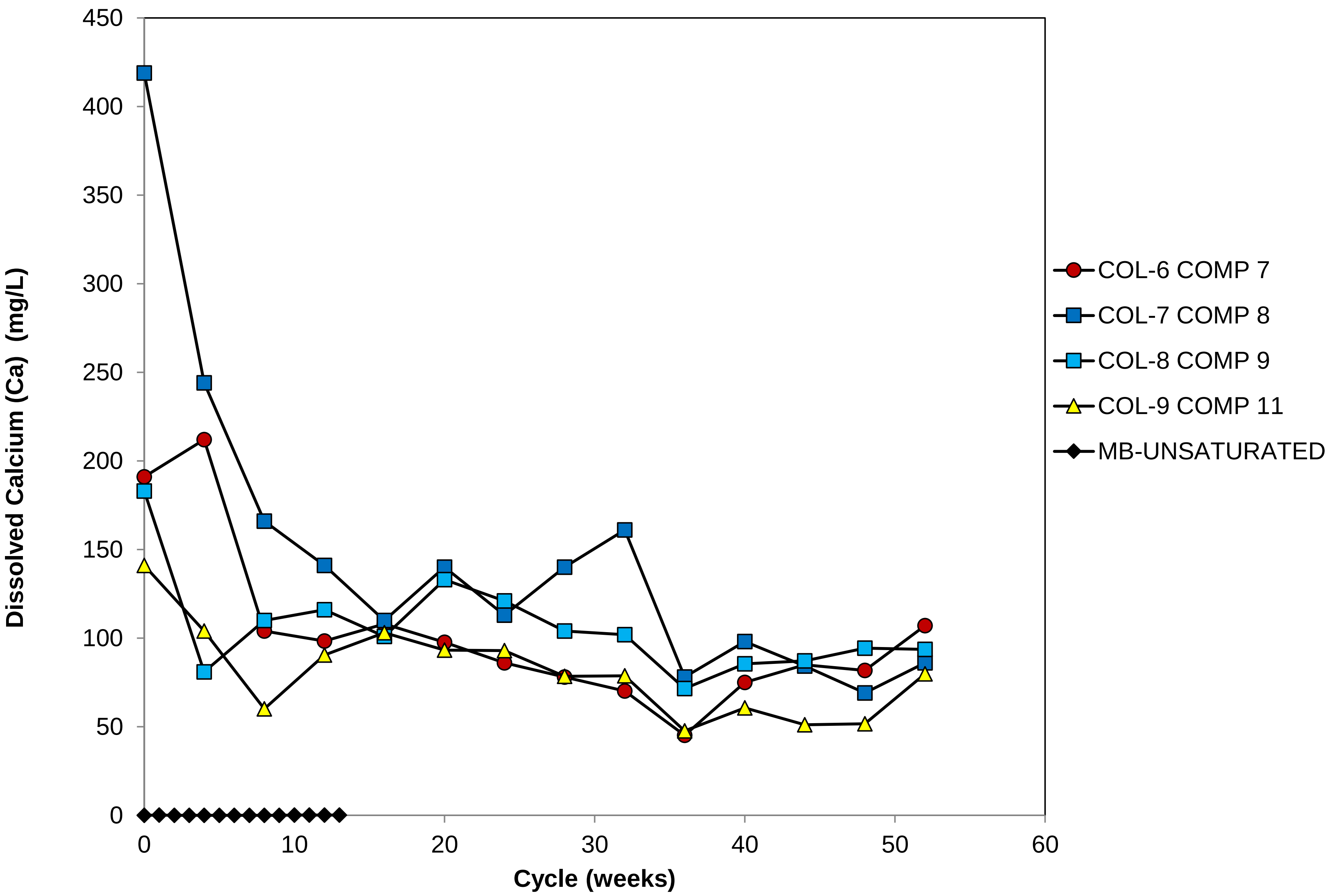

0.00006
0.00005
0.00004
0.00003
0.00002
0.00001
0
Dissolved Beryllium (Be) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

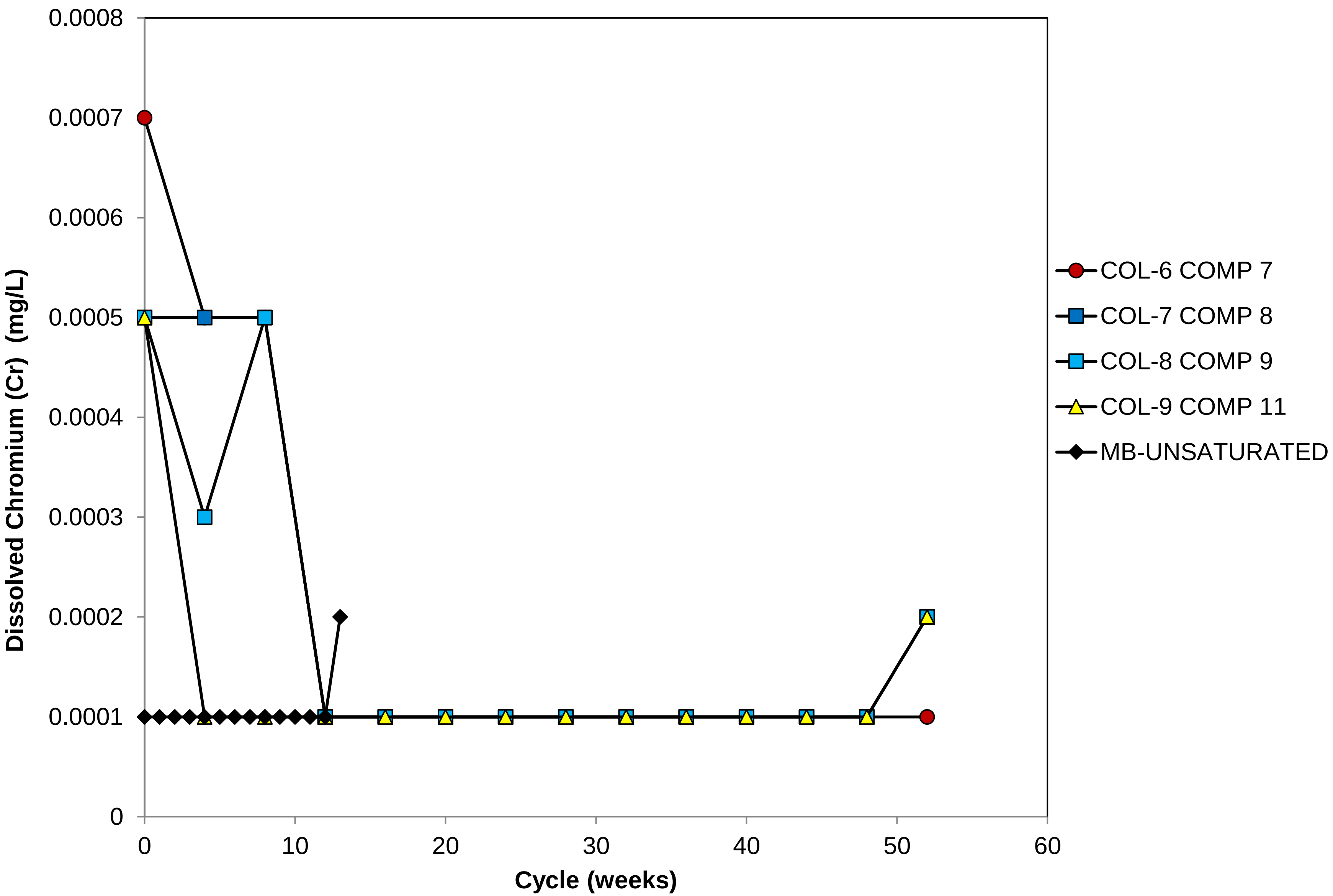

0.000035
0.00003
0.000025
0.00002
0.000015
0.00001
0.000005
0
Dissolved Bismuth (Bi) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

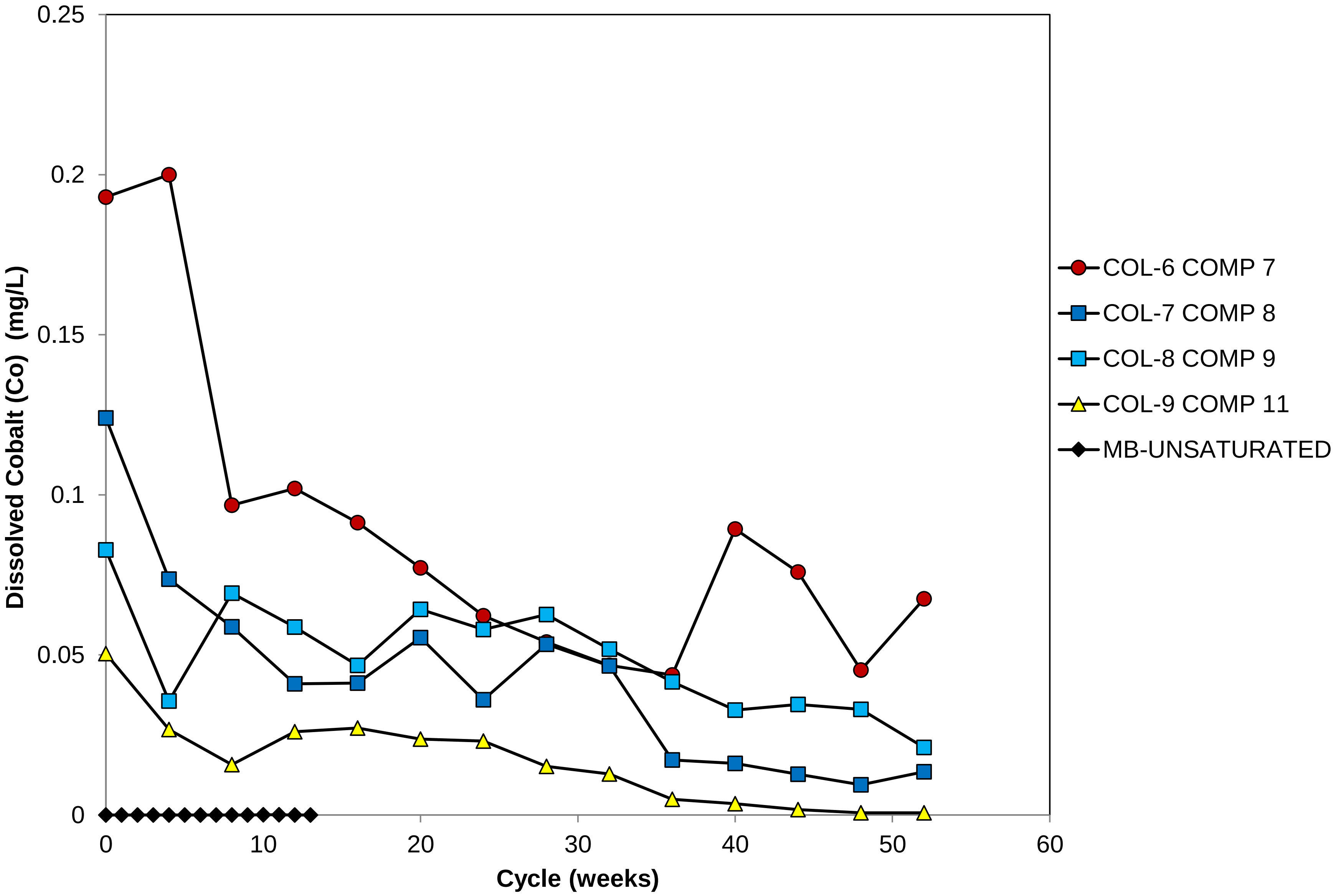

0.35
0.3
0.25
0.2
0.15
0.1
0.05
0
Dissolved Boron (B) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

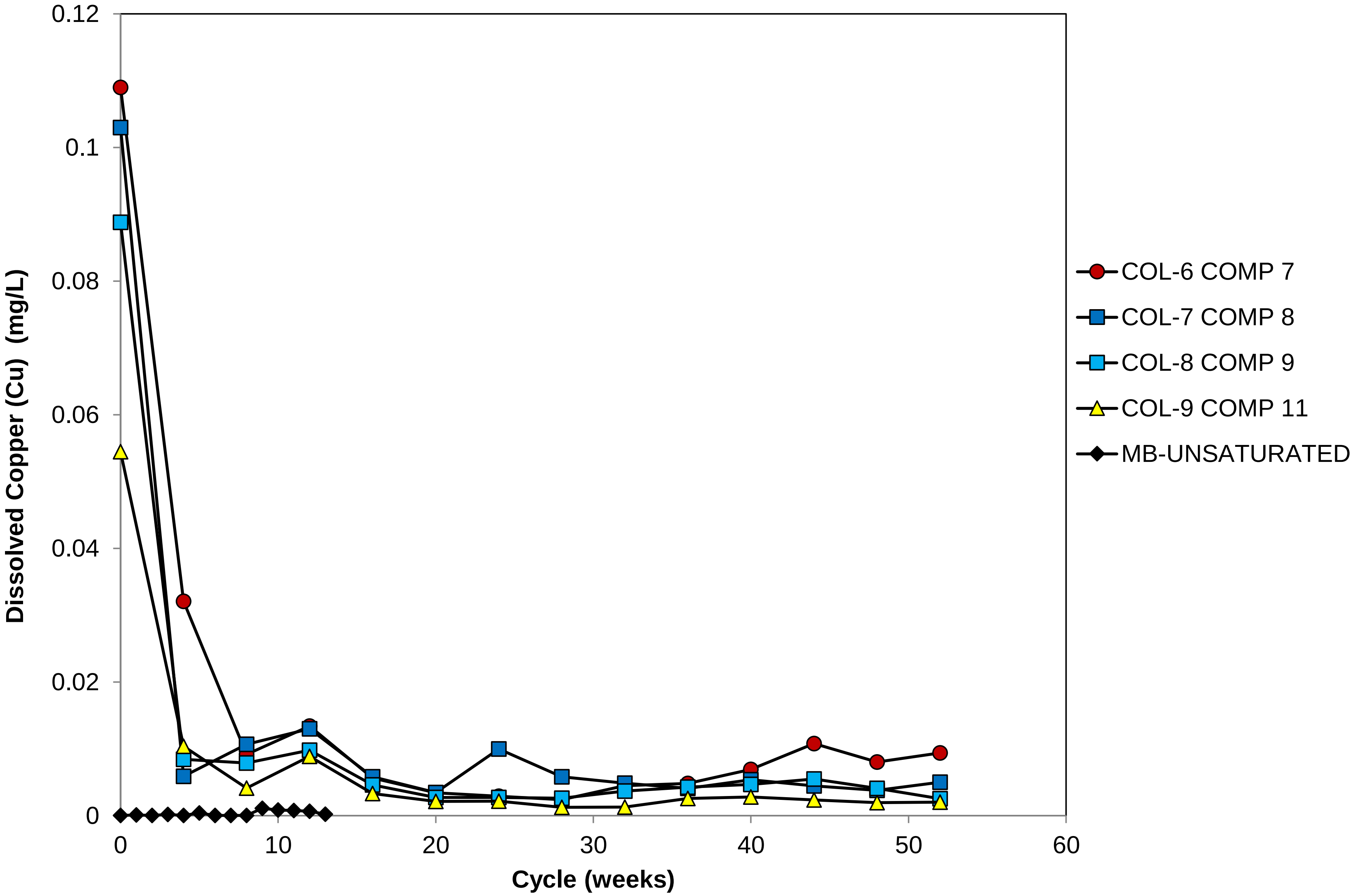

Dissolved Cesium (Cs) (mg/L)
0.012
0.01
0.008
0.006
0.004
0.002
0
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

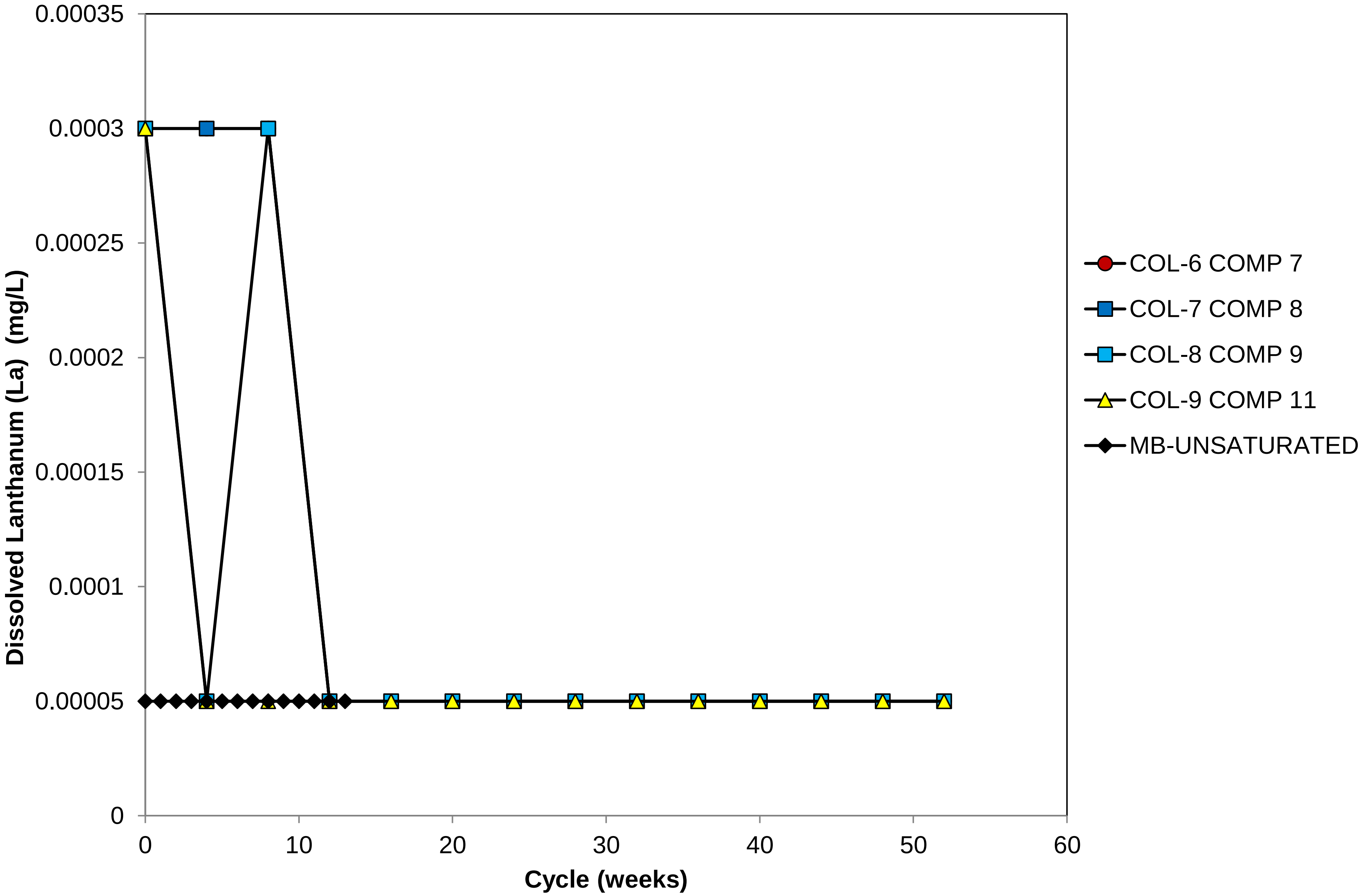

0.0009
0.0008
0.0007
0.0006
0.0005
0.0004
0.0003
0.0002
0.0001
0
Dissolved Cadmium (Cd) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

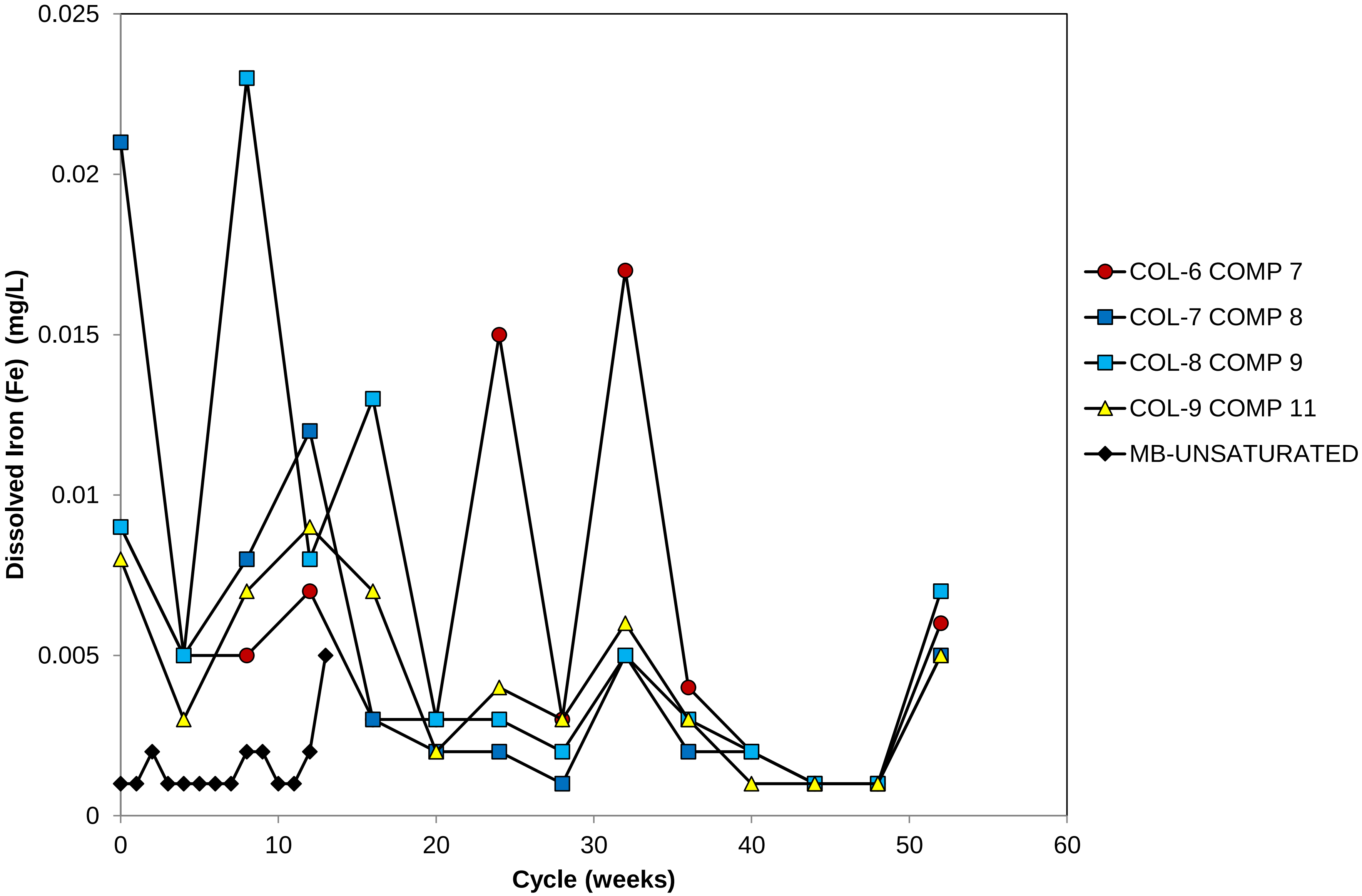

450
400
350
300
250
200
150
100
50
0
Dissolved Calcium (Ca) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

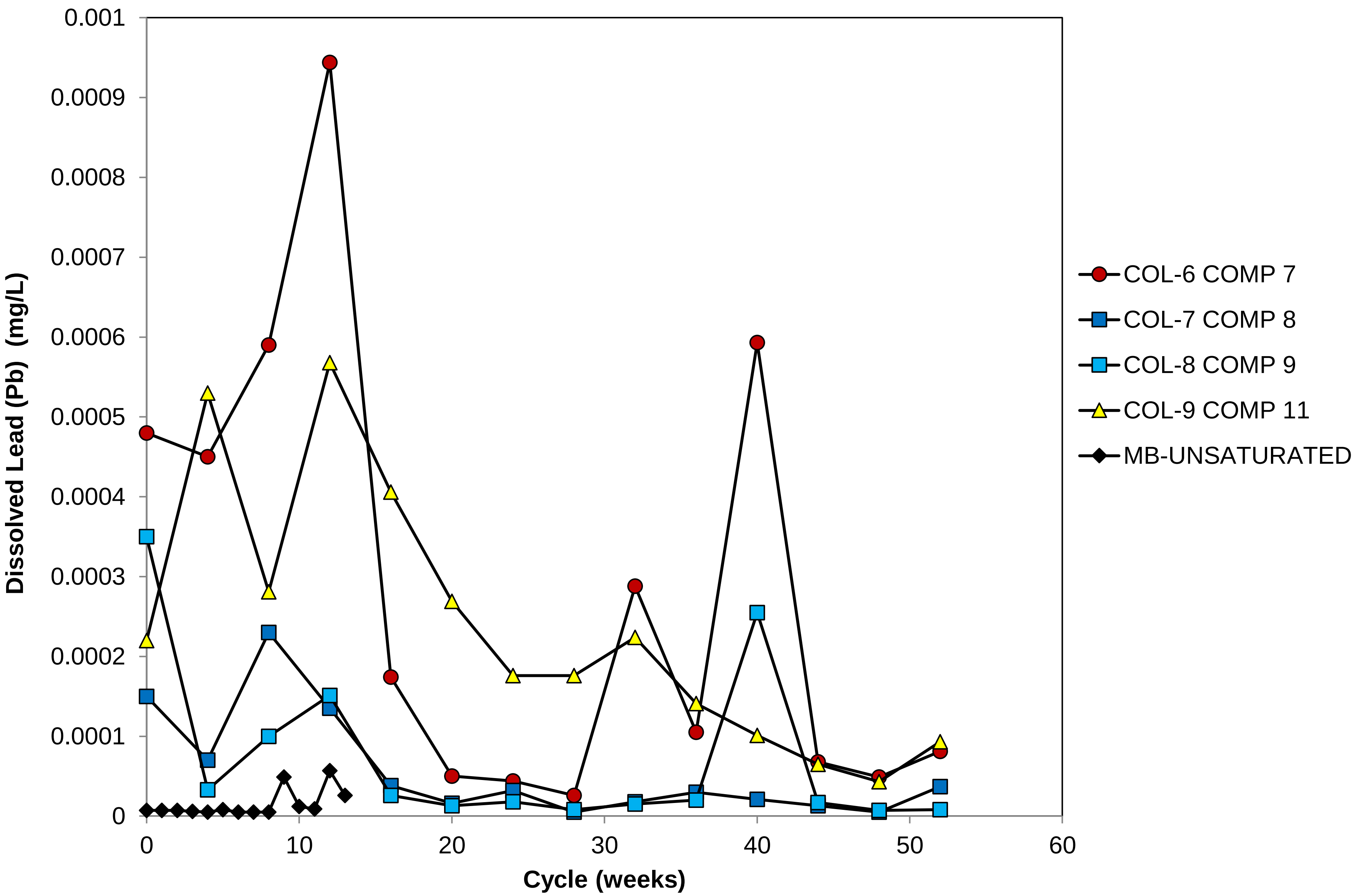

0.0008
0.0007
0.0006
0.0005
0.0004
0.0003
0.0002
0.0001
0
Dissolved Chromium (Cr) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

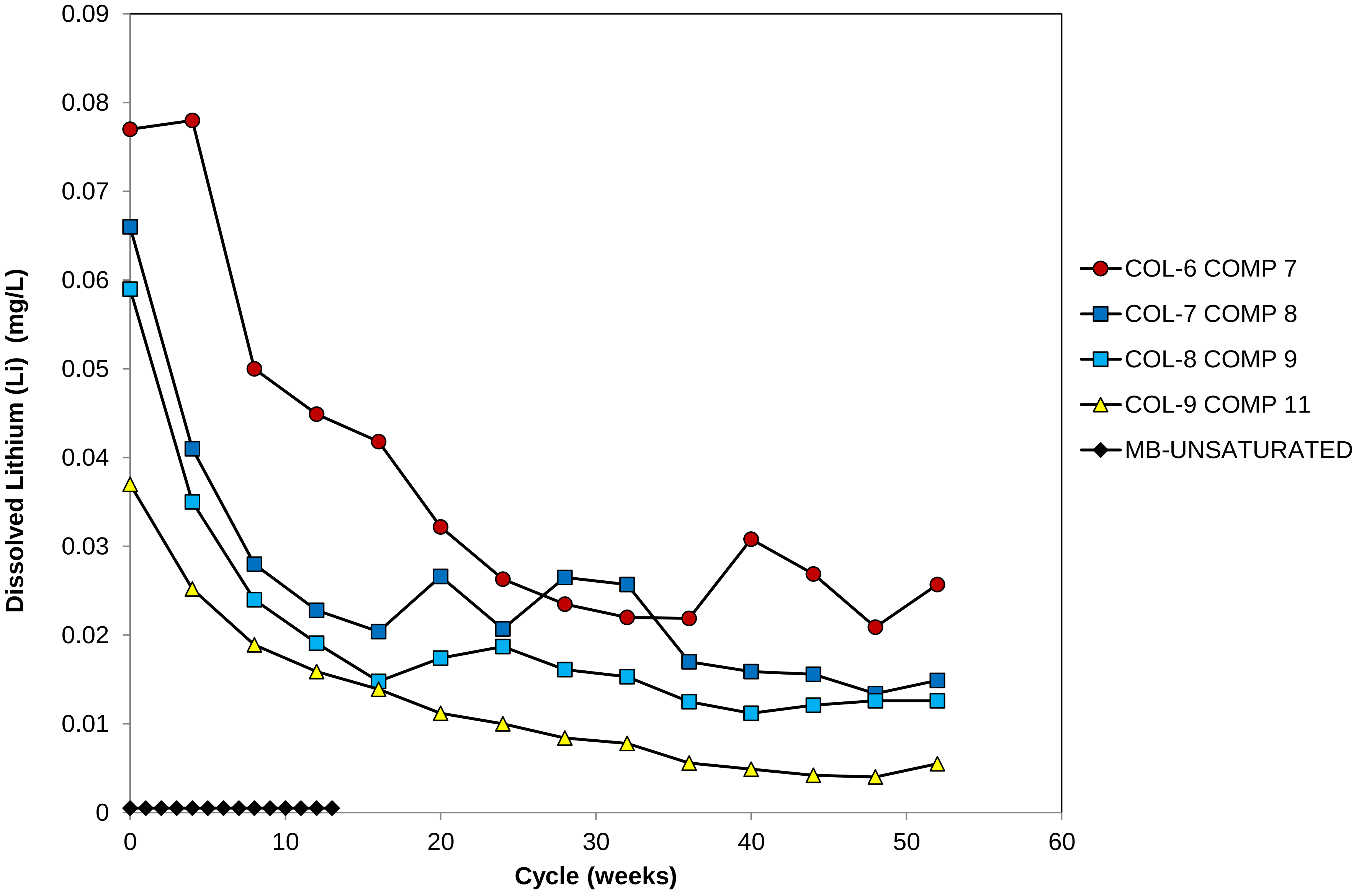

0.25
0.2
0.15
0.1
0.05
0
Dissolved Cobalt (Co) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

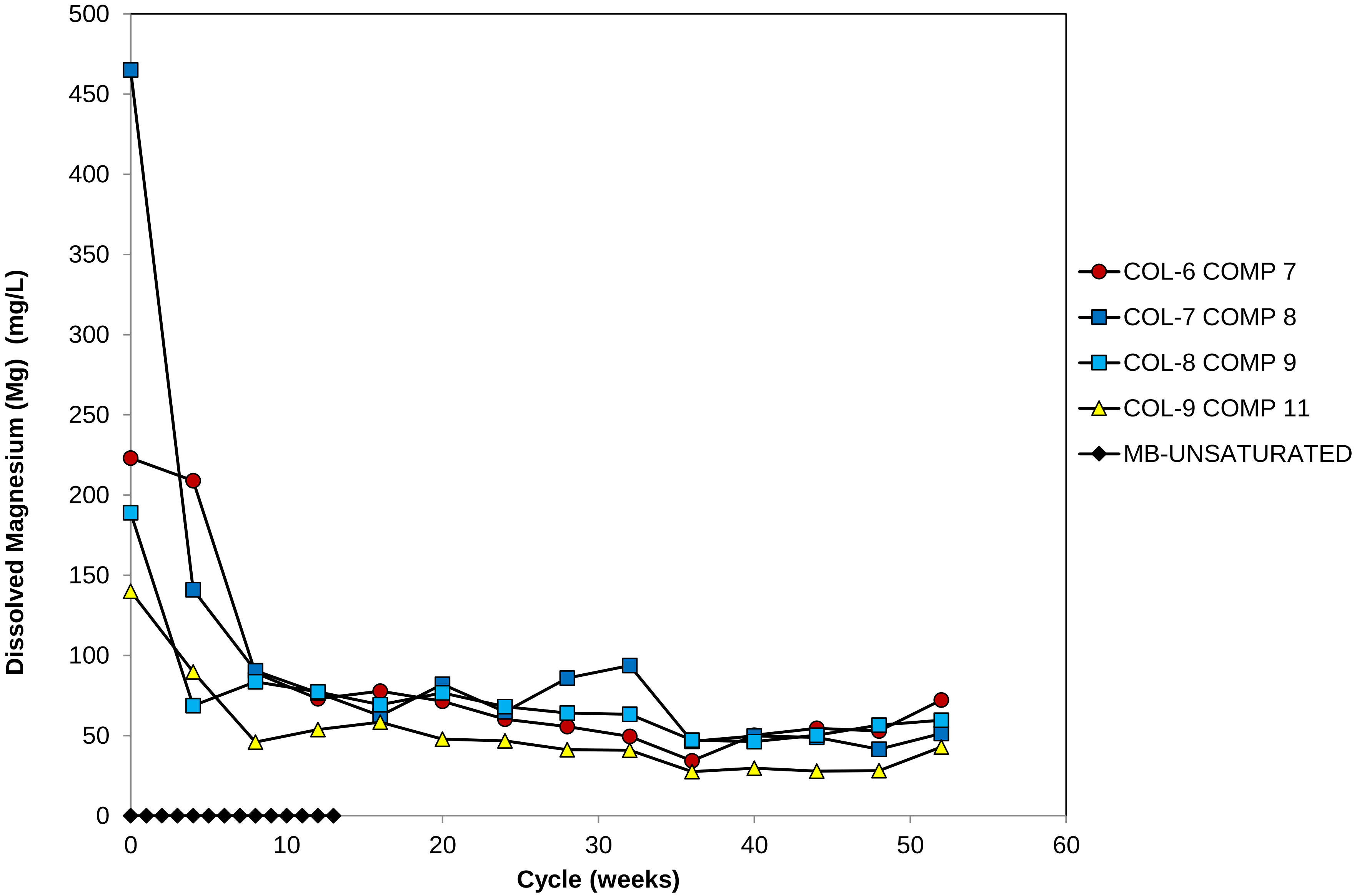

0.12
0.1
0.08
0.06
0.04
0.02
0
Dissolved Copper (Cu) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

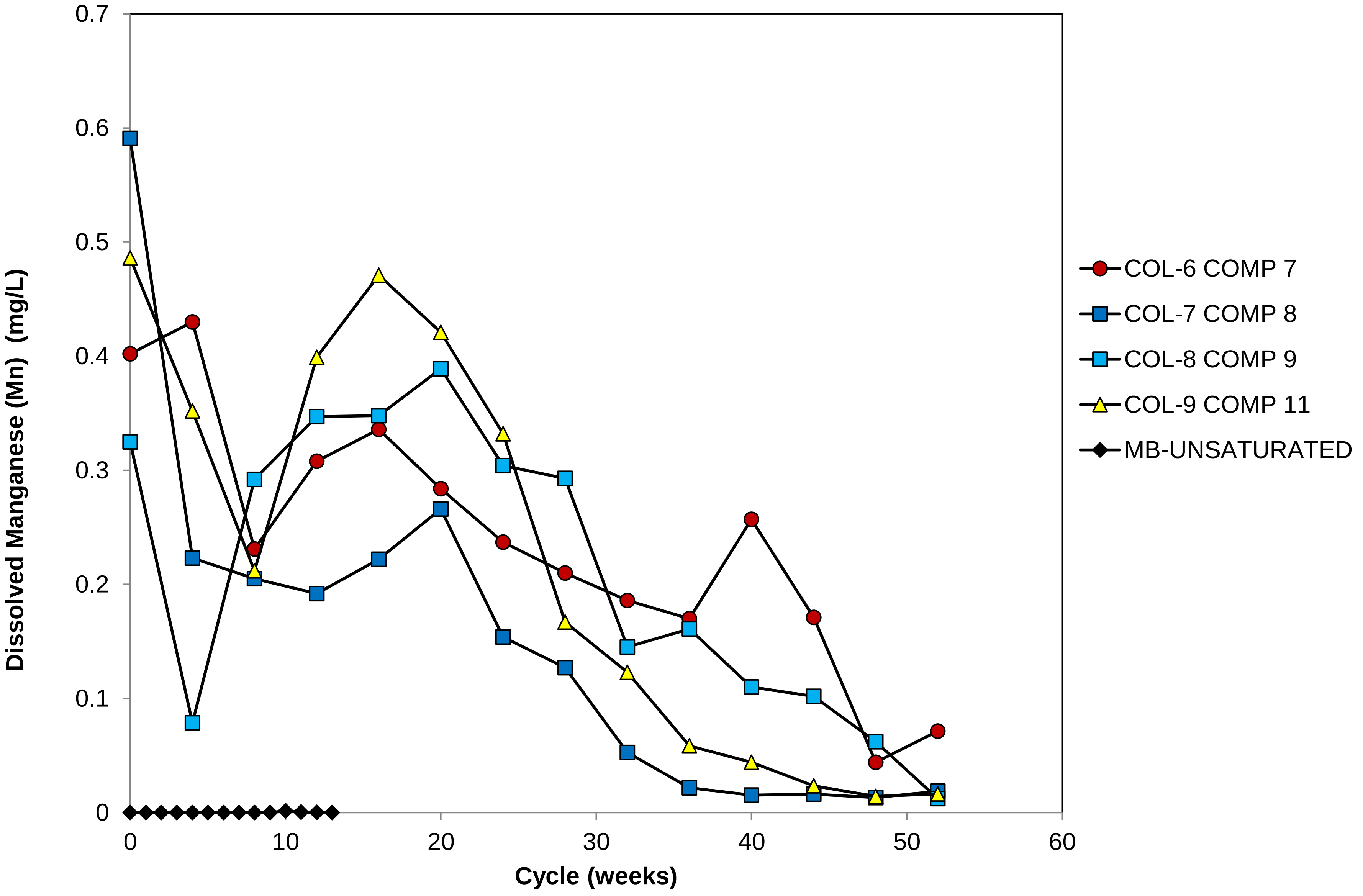

0.00035
0.0003
0.00025
0.0002
0.00015
0.0001
0.00005
0
Dissolved Lanthanum (La) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

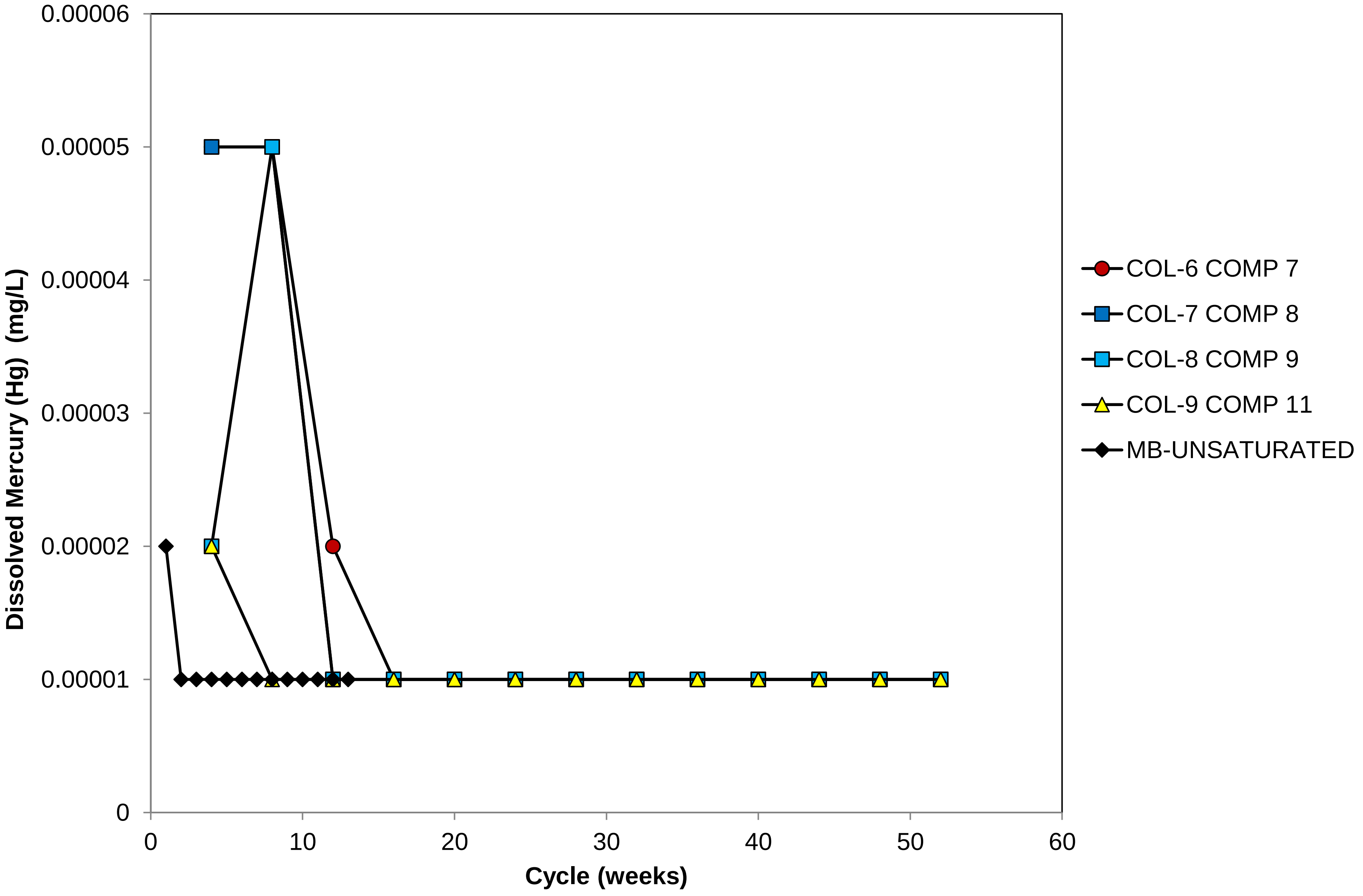

Dissolved Iron (Fe) (mg/L)
0.025
0.02
0.015
0.01
0.005
0
Cycle (weeks)
0
10
20
30
40
50
60
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

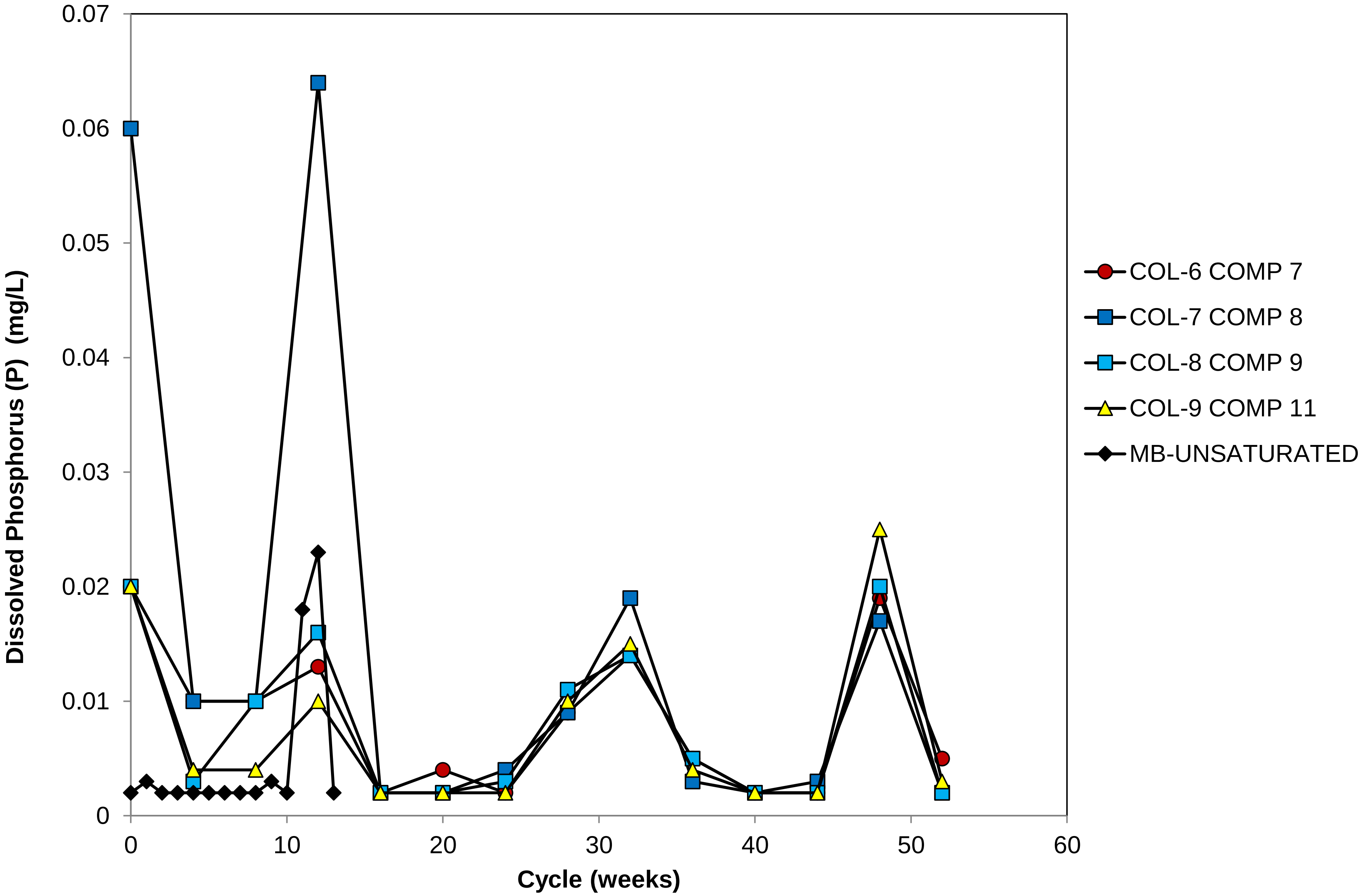

0.001
0.0009
0.0008
0.0007
0.0006
0.0005
0.0004
0.0003
0.0002
0.0001
0
Dissolved Lead (Pb) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

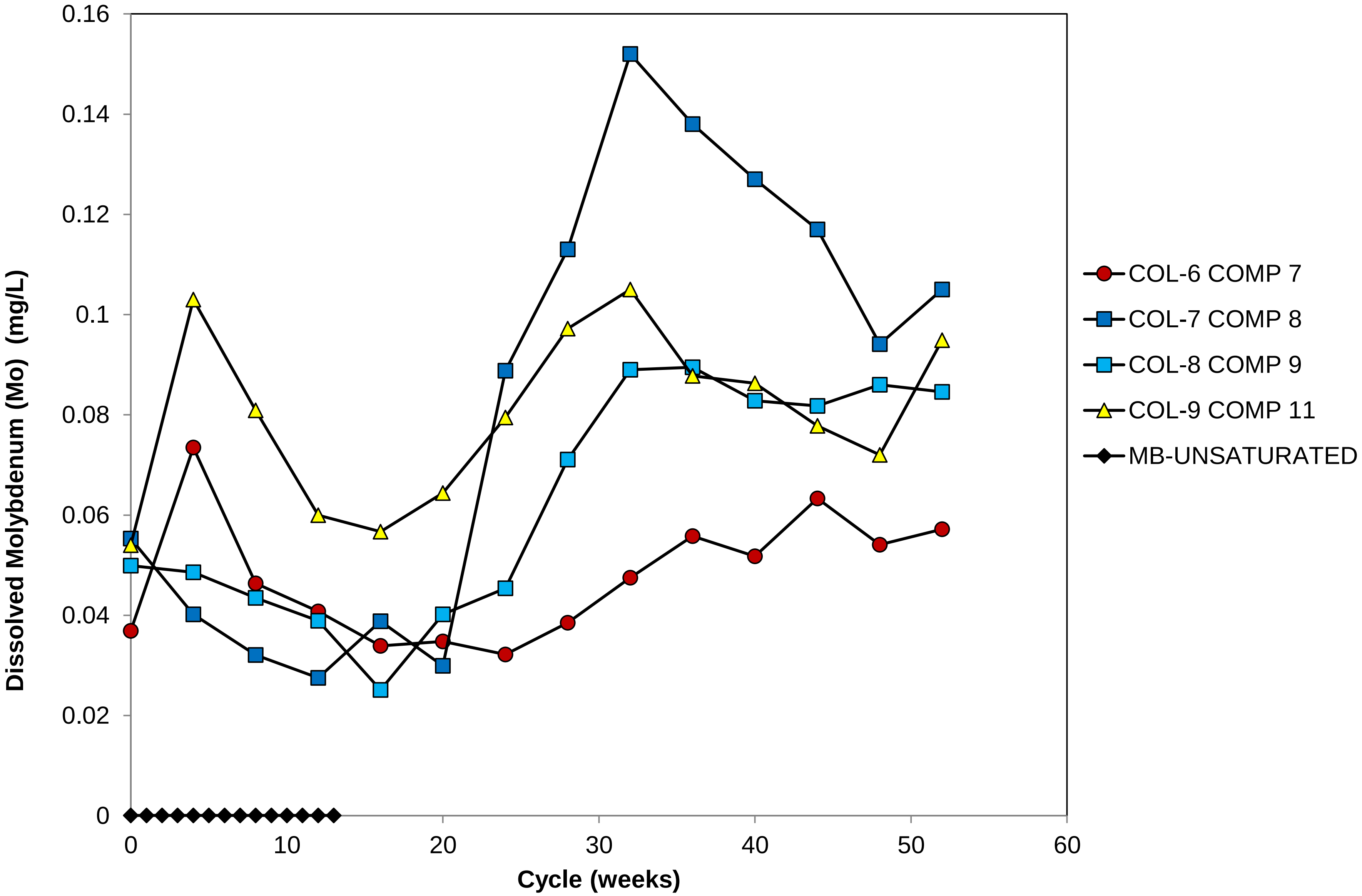

0.09
0.08
0.07
0.06
0.05
0.04
0.03
0.02
0.01
0
Dissolved Lithium (Li) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

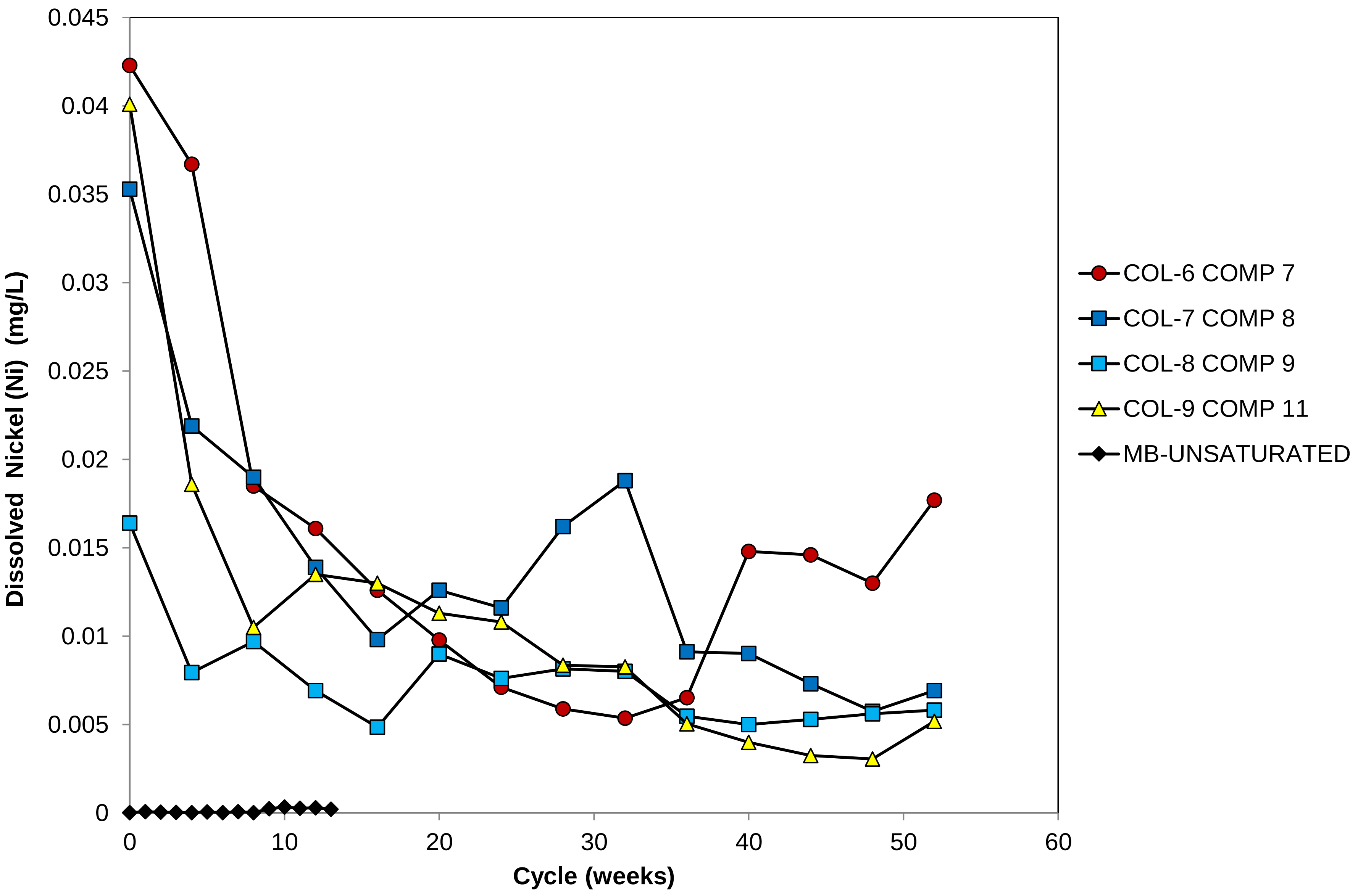

Dissolved Magnesium (Mg) (mg/L)
500
450
400
350
300
250
200
150
100
50
0
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

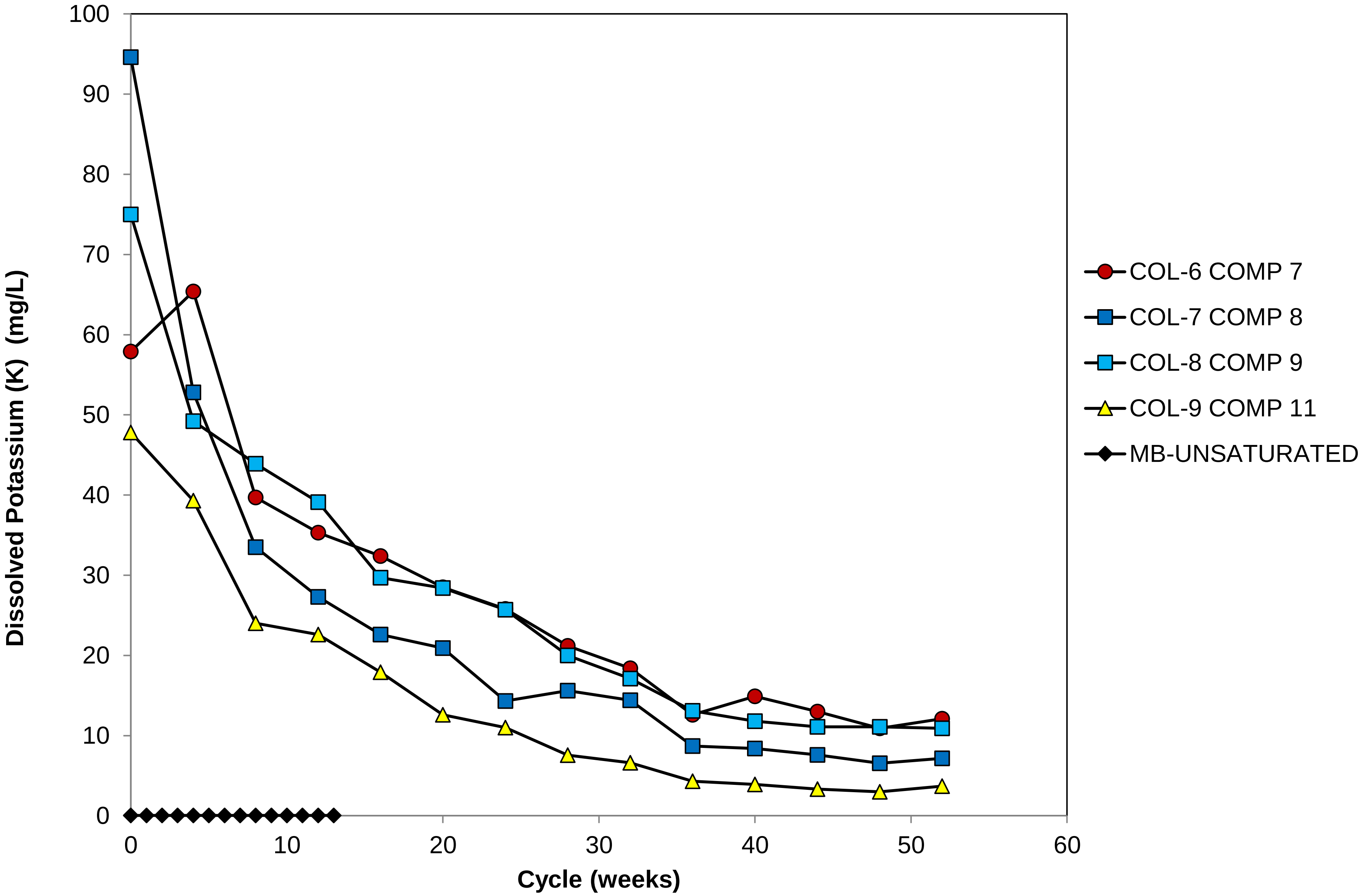

0.7
0.6
0.5
0.4
0.3
0.2
0.1
0
Dissolved Manganese (Mn) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

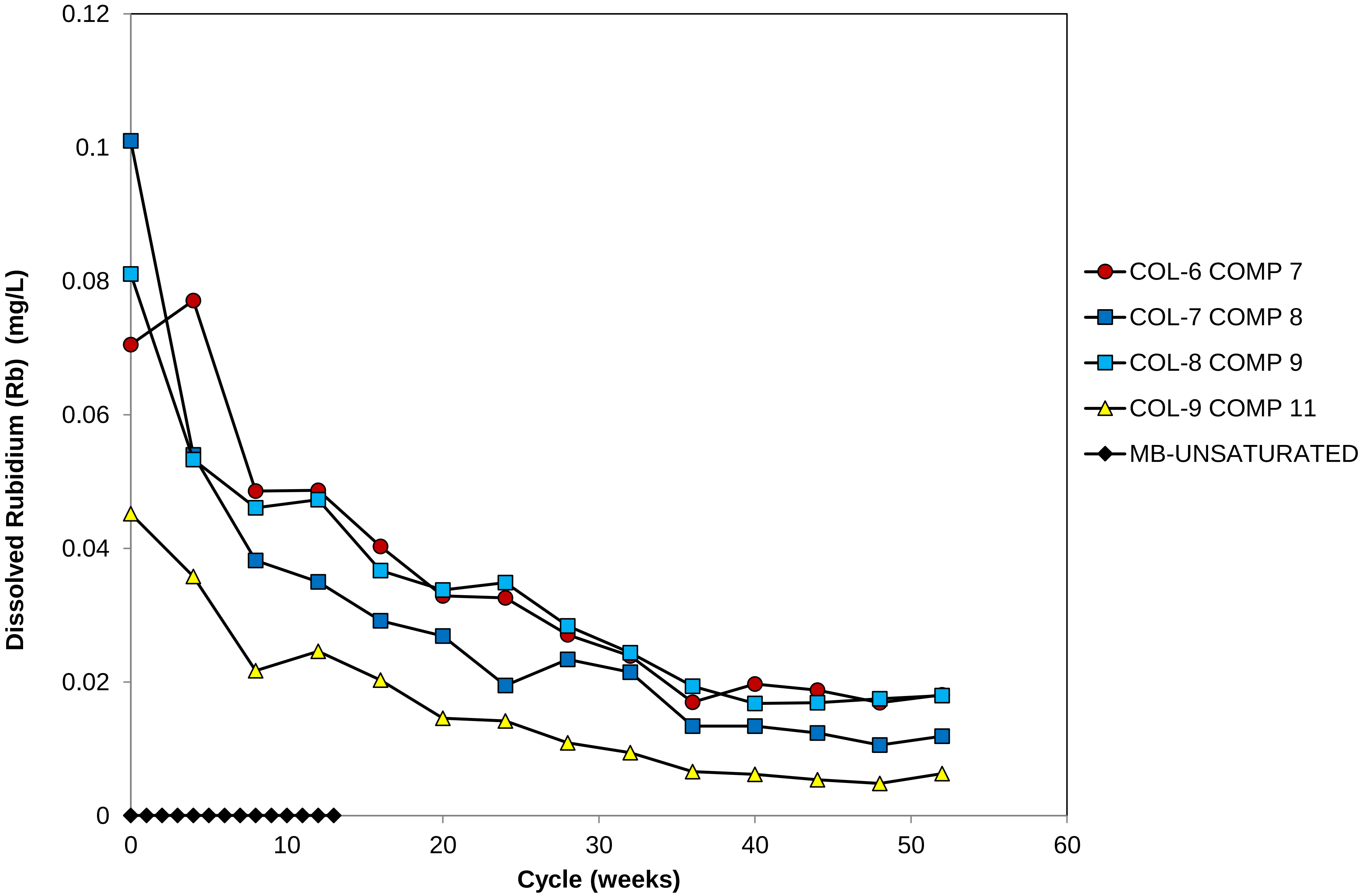

0.00006
0.00005
0.00004
0.00003
0.00002
0.00001
0
Dissolved Mercury (Hg) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

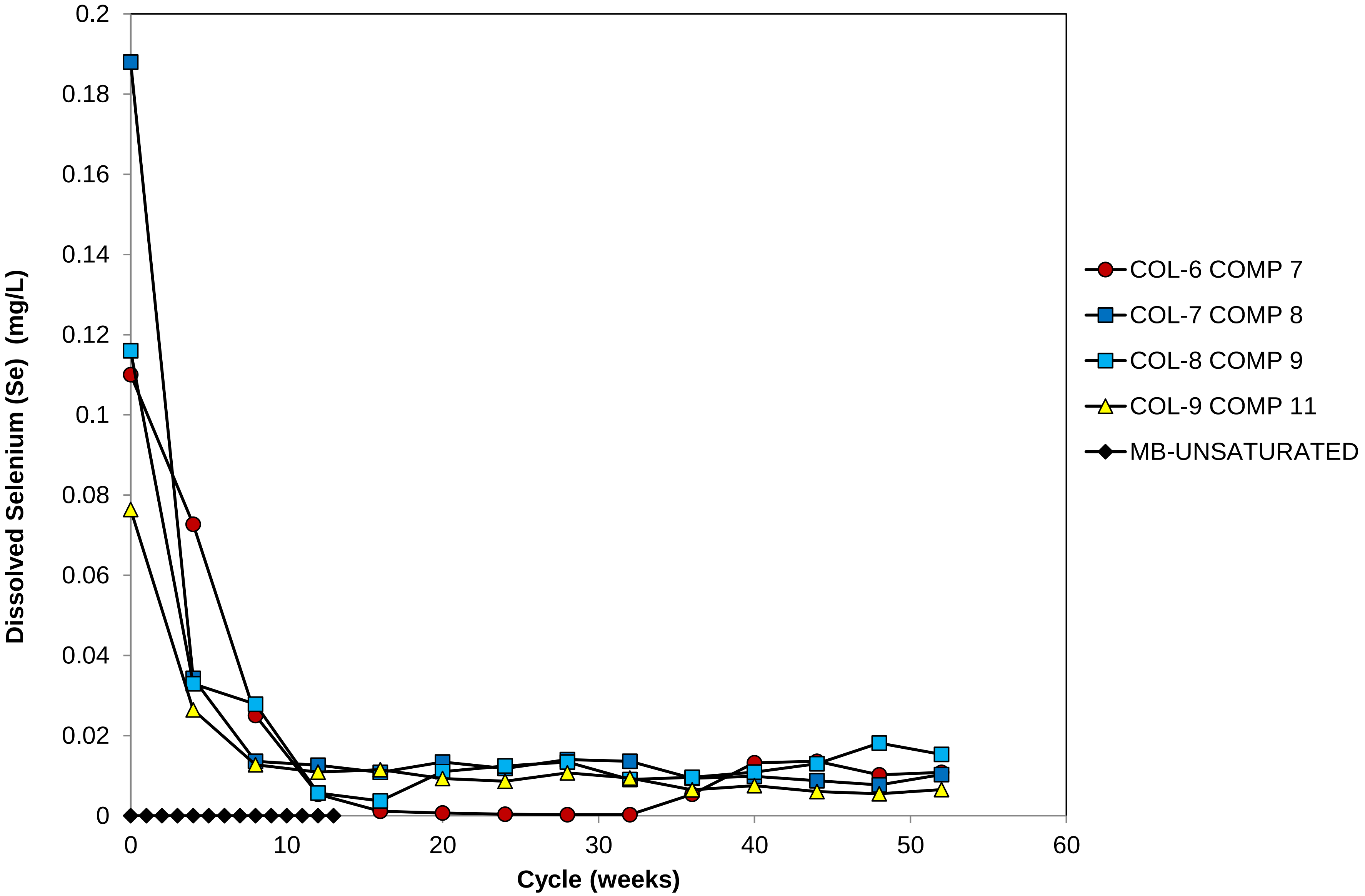

0.07
0.06
0.05
0.04
0.03
0.02
0.01
0
Dissolved Phosphorus (P) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

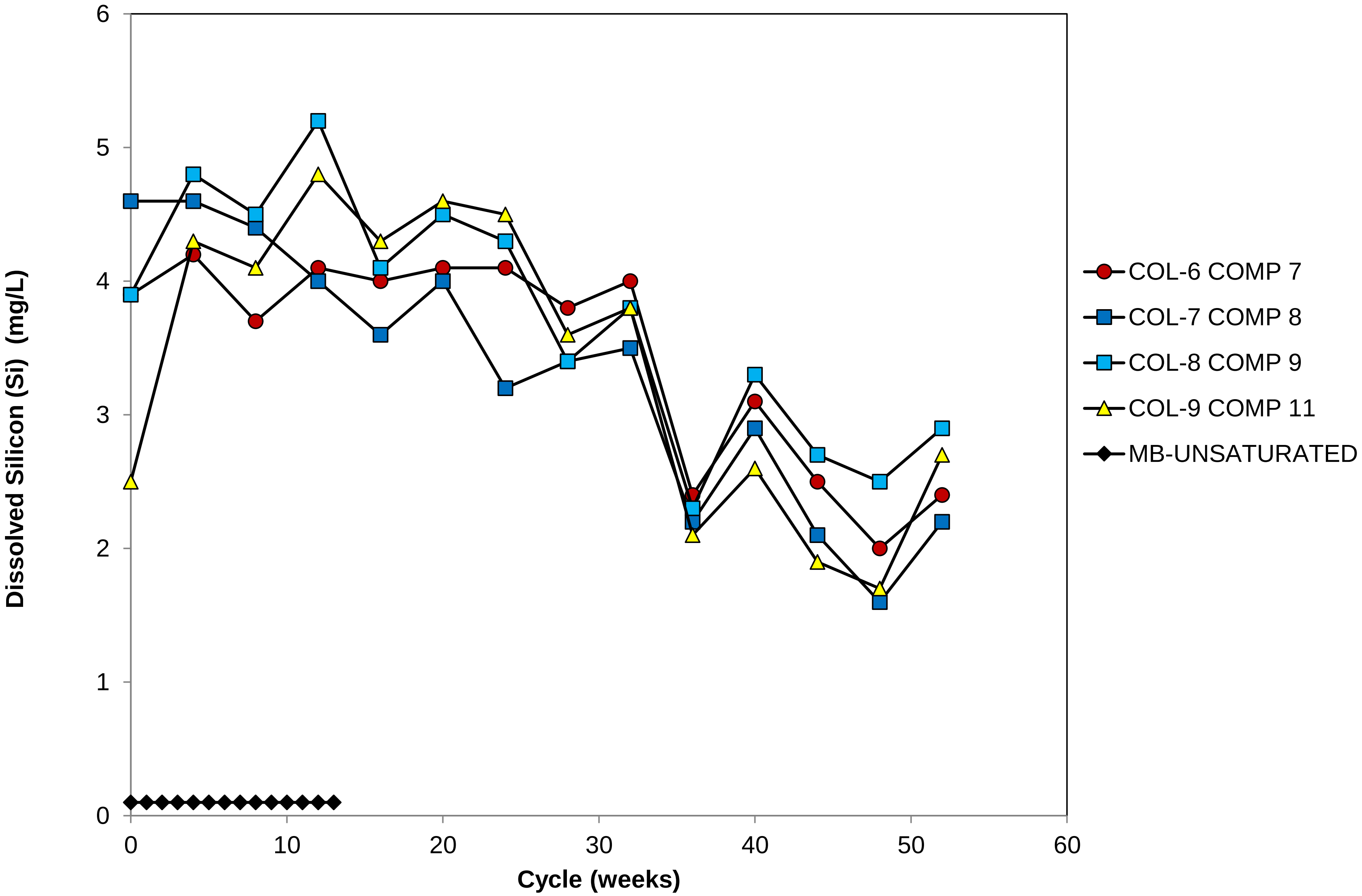

0.16
0.14
0.12
0.1
0.08
0.06
0.04
0.02
0
Dissolved Molybdenum (Mo) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

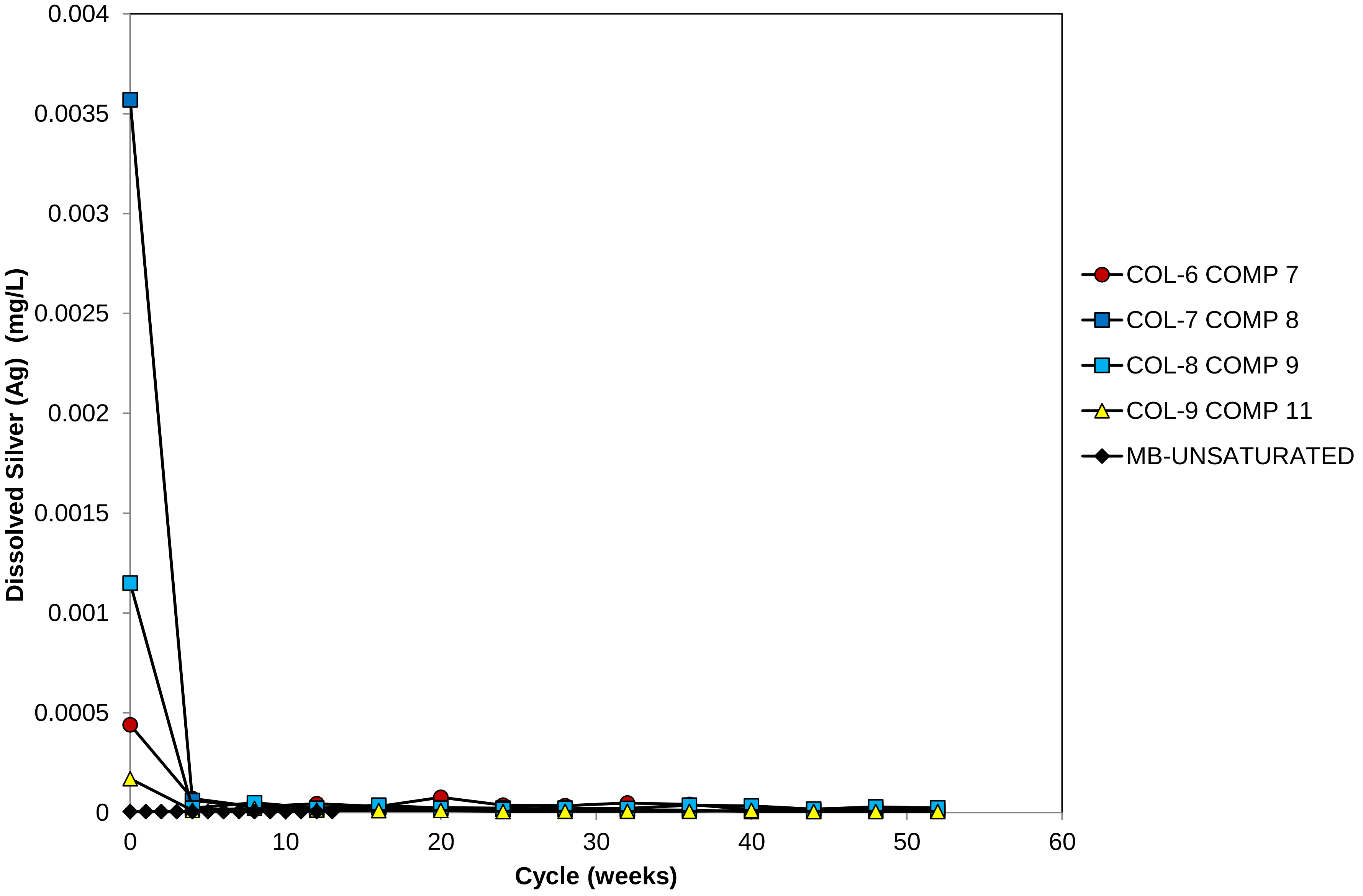

Dissolved Nickel (Ni) (mg/L)
0.045
0.04
0.035
0.03
0.025
0.02
0.015
0.01
0.005
0
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

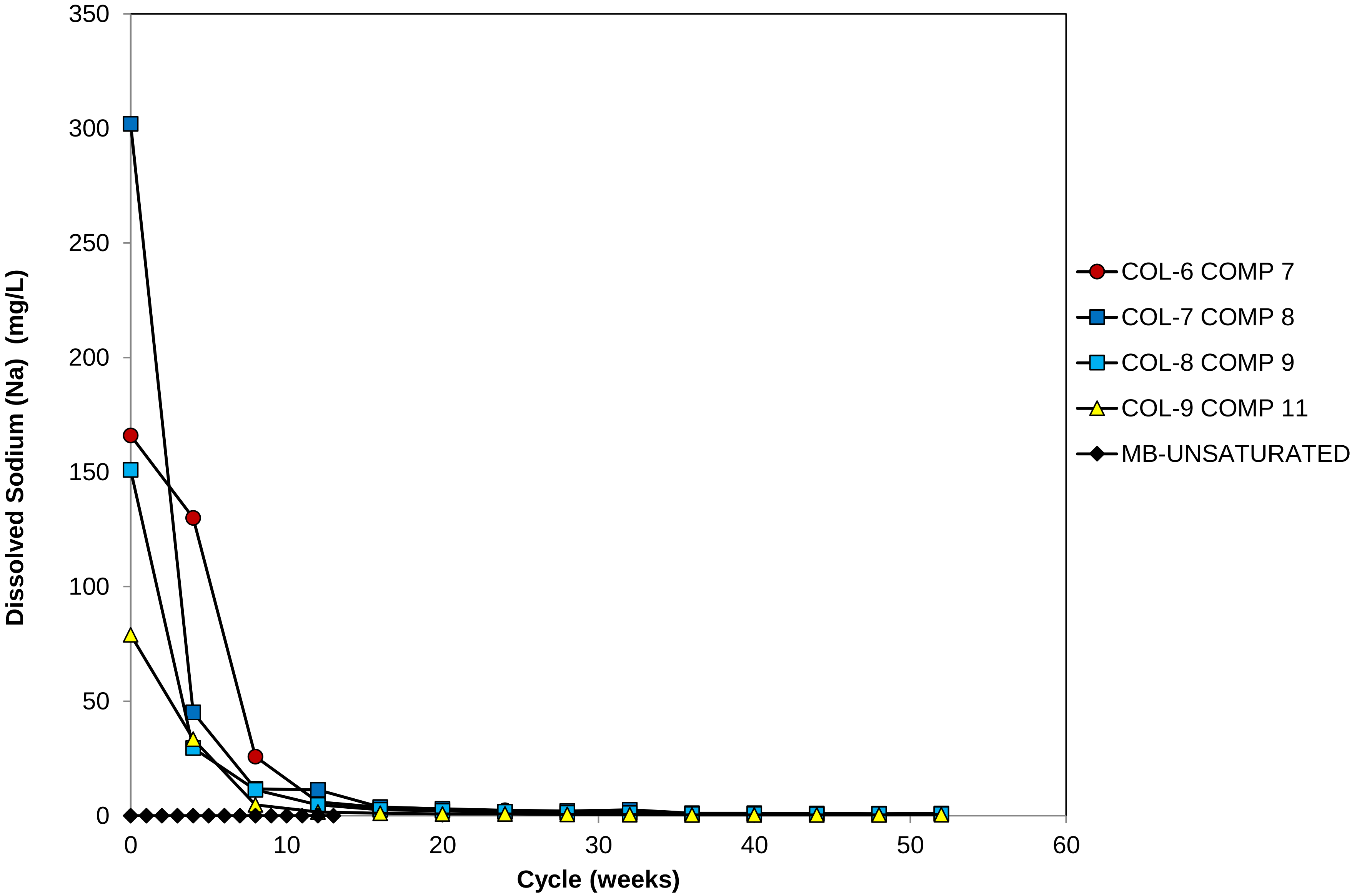

100
90
80
70
60
50
40
30
20
10
0
Dissolved Potassium (K) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

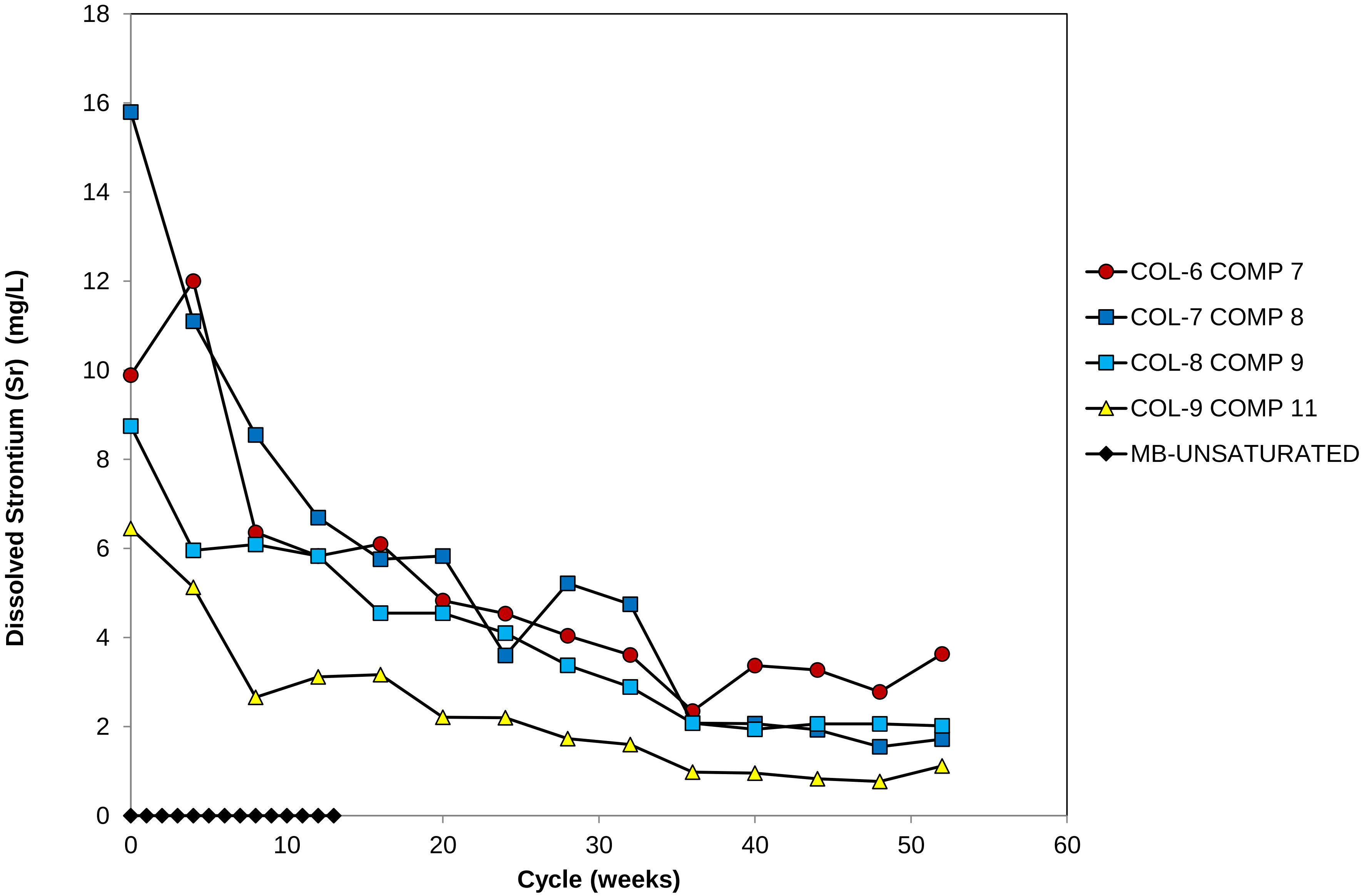

0.12
0.1
0.08
0.06
0.04
0.02
0
Dissolved Rubidium (Rb) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

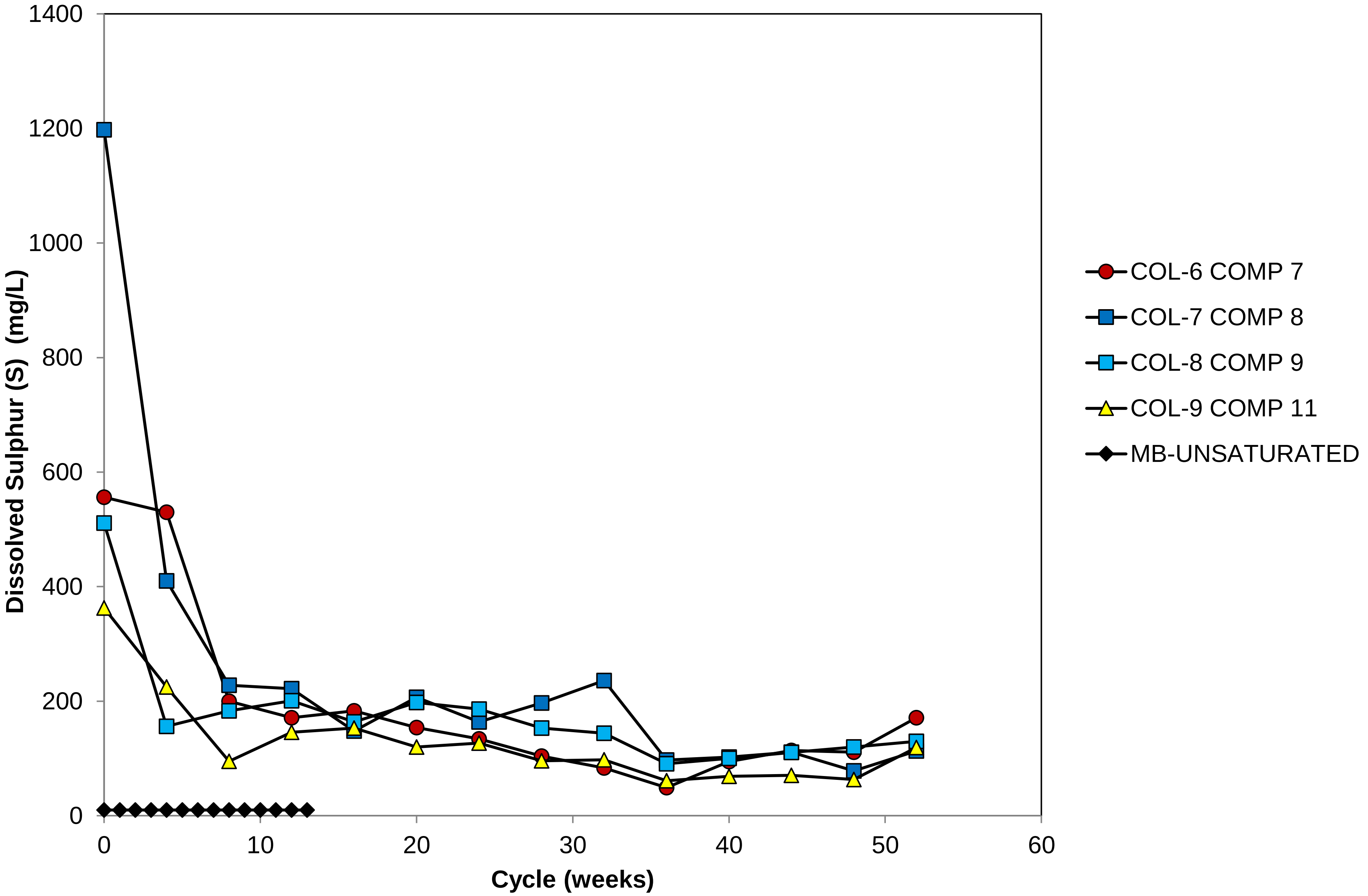

0.2
0.18
0.16
0.14
0.12
0.1
0.08
0.06
0.04
0.02
0
Dissolved Selenium (Se) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

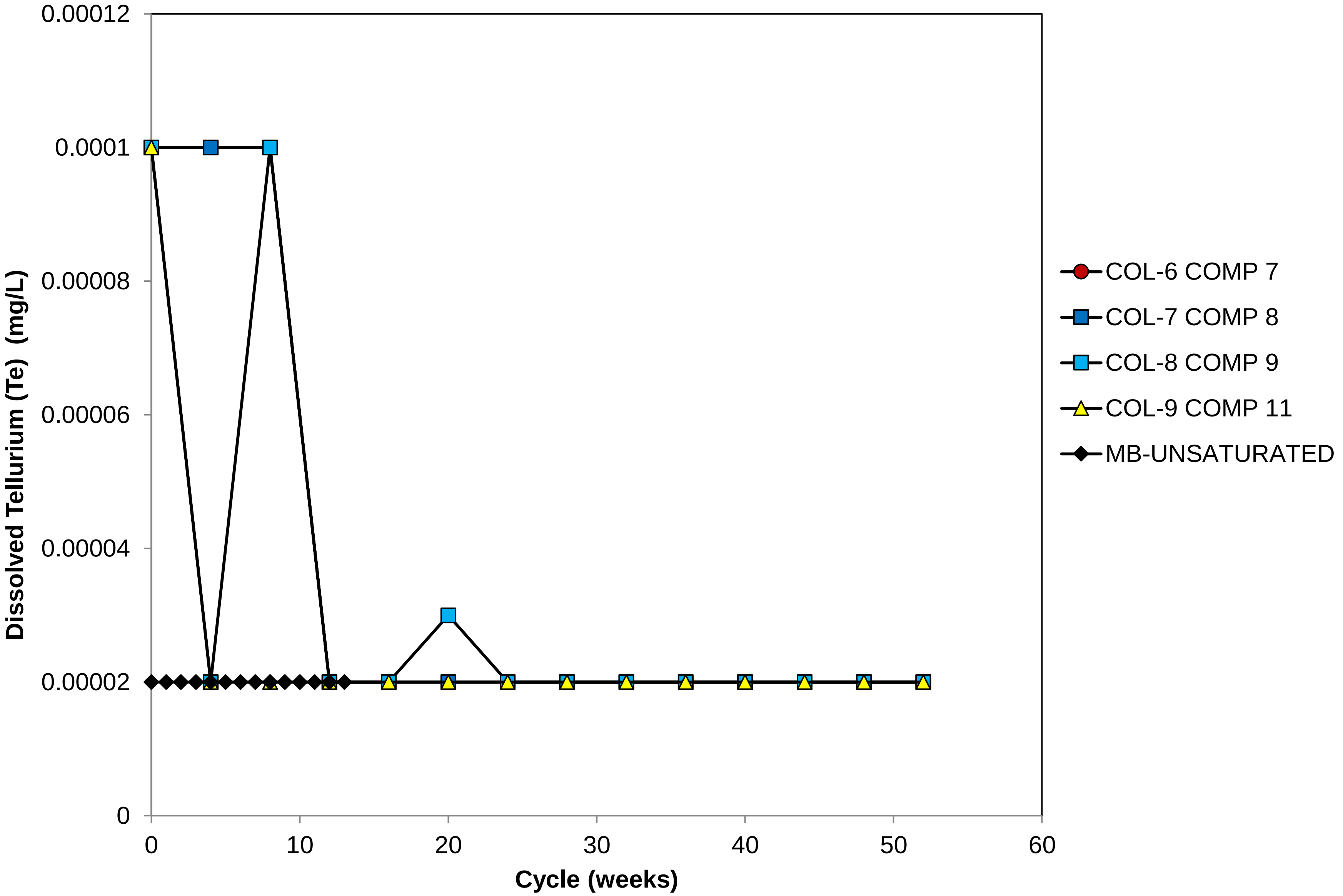

Dissolved Silicon (Si) (mg/L)
6
5
4
3
2
1
0
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

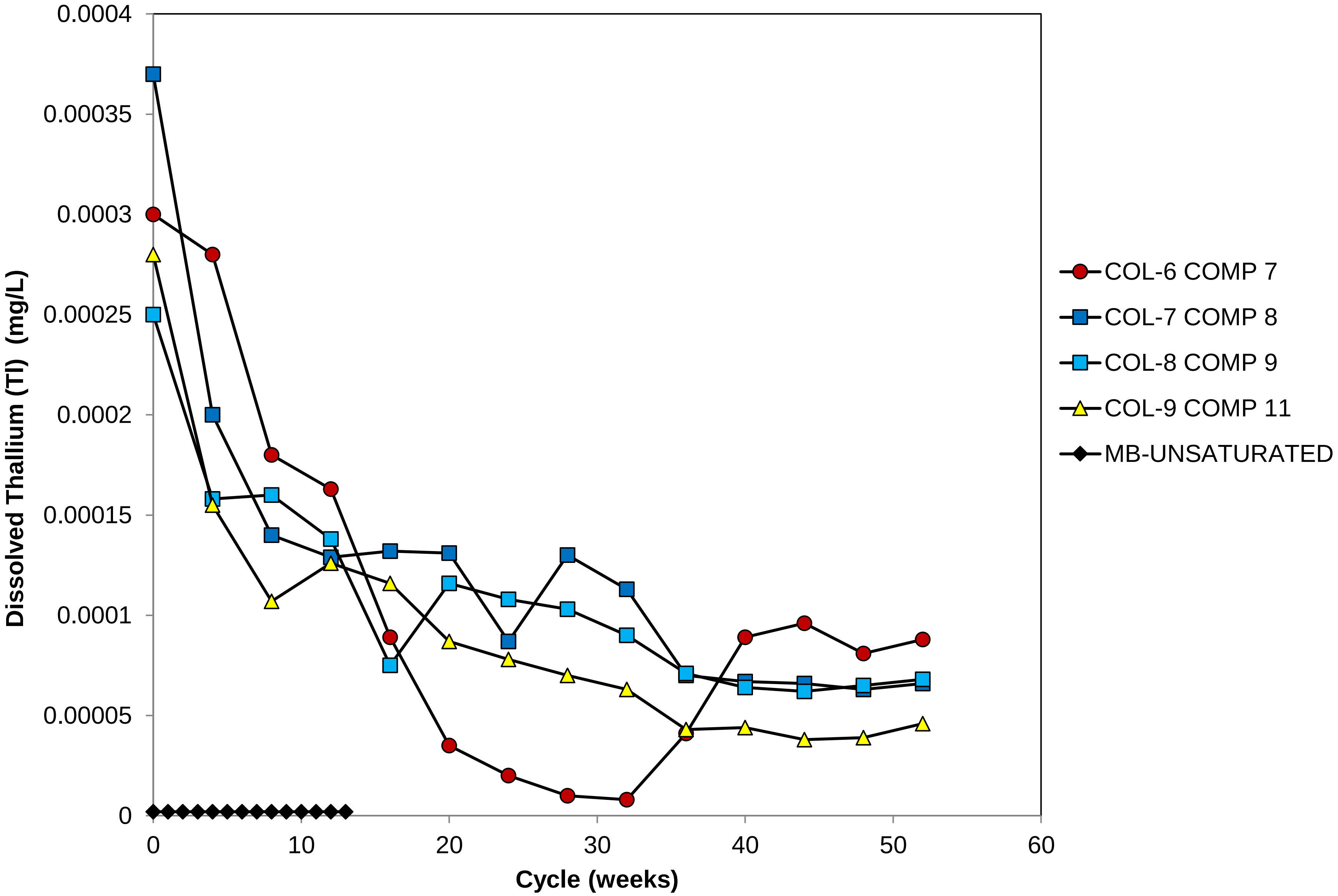

0.004
0.0035
0.003
0.0025
0.002
0.0015
0.001
0.0005
0
Dissolved Silver (Ag) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

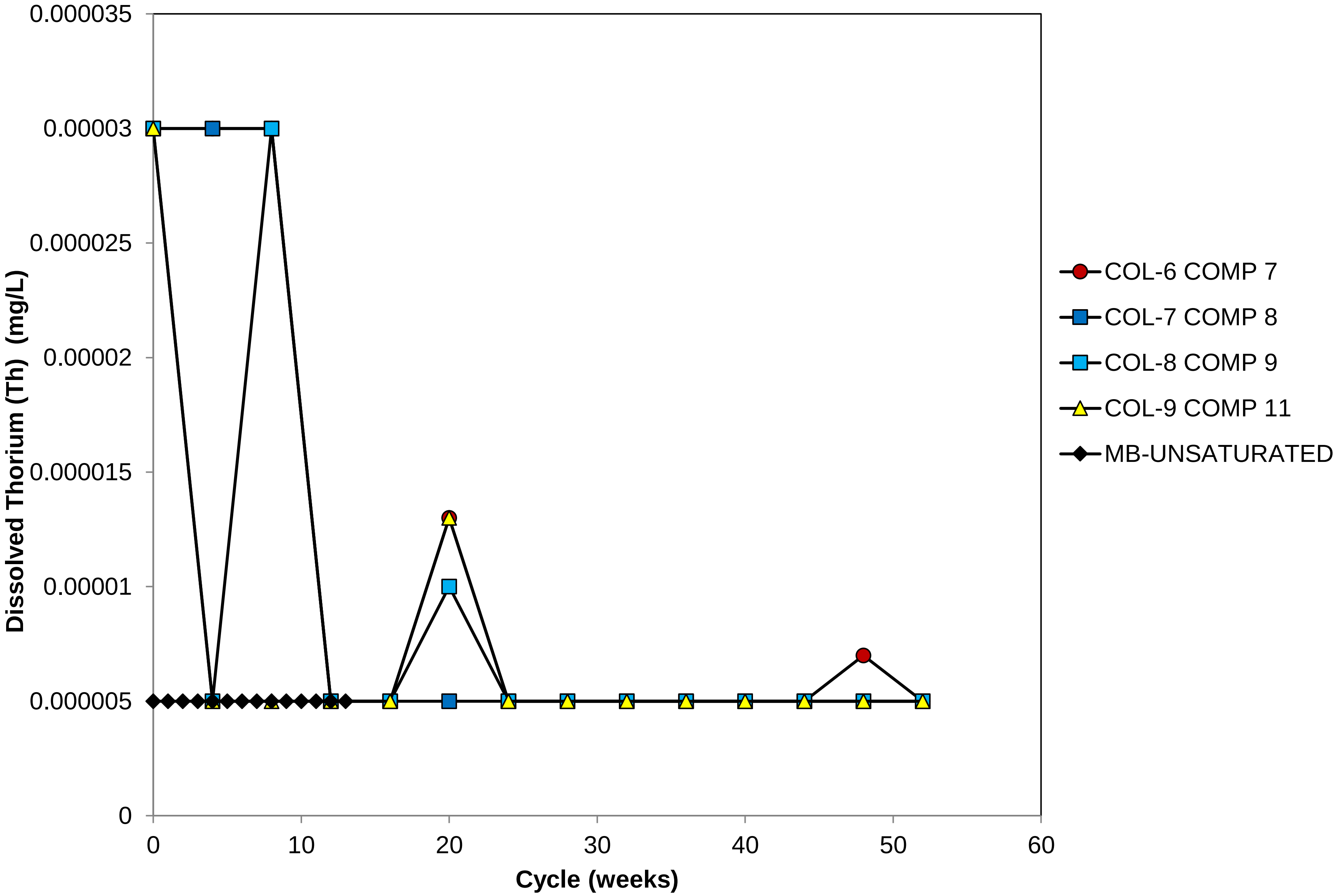

Dissolved Sodium (Na) (mg/L)
350
300
250
200
150
100
50
0
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

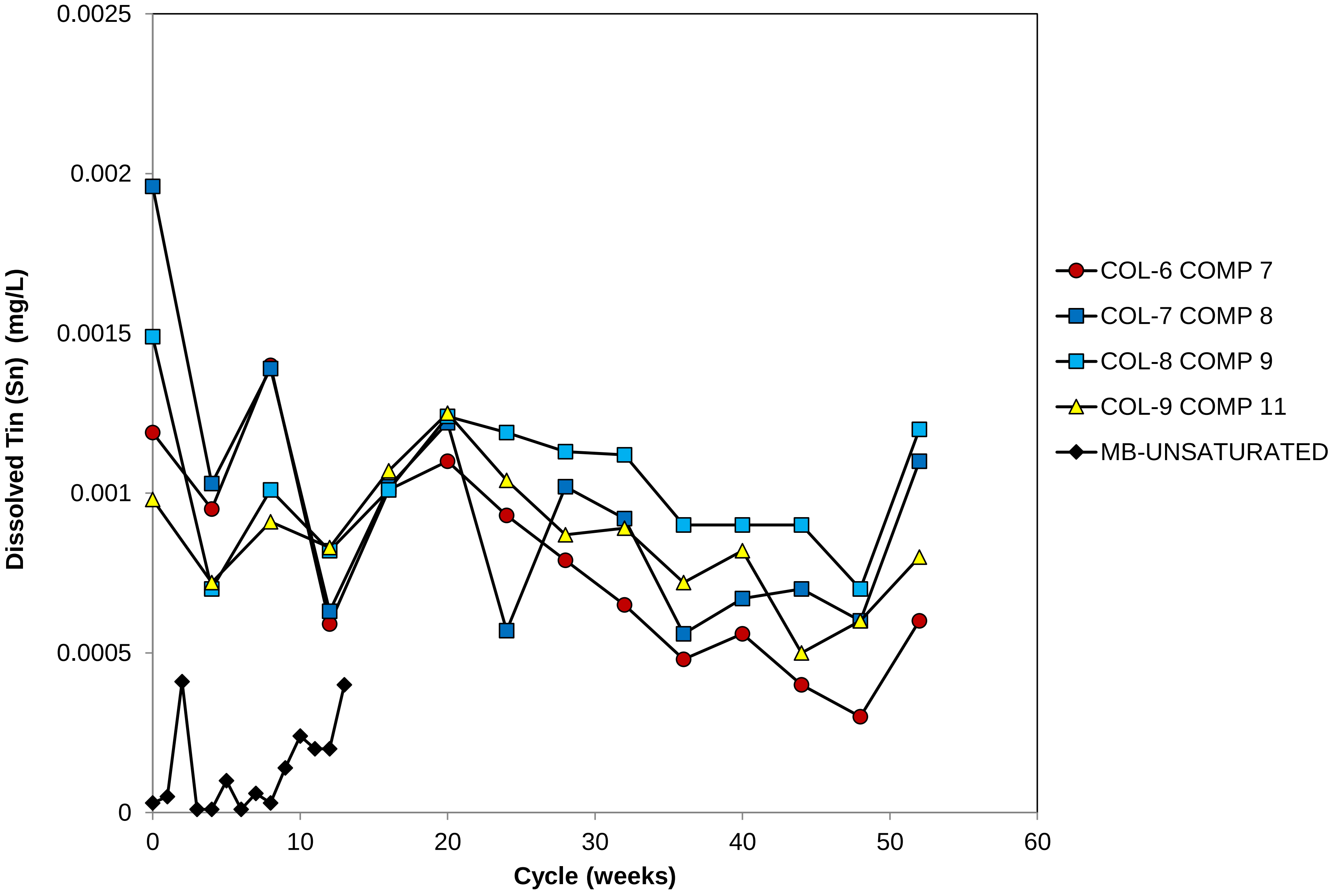

Dissolved Strontium (Sr) (mg/L)
0
2
4
6
8
10
12
14
16
18
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

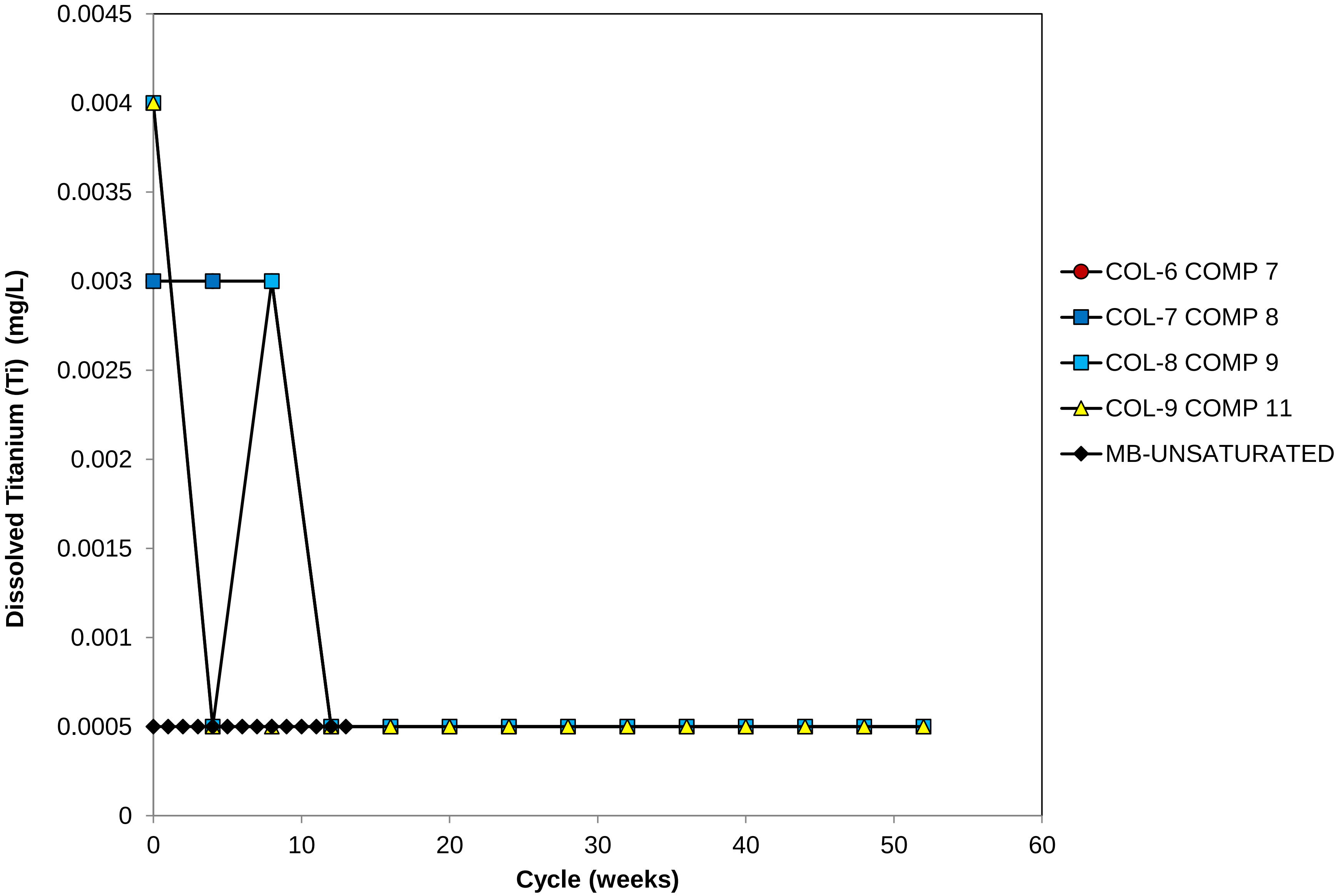

1400
1200
1000
800
600
400
200
0
Dissolved Sulphur (S) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

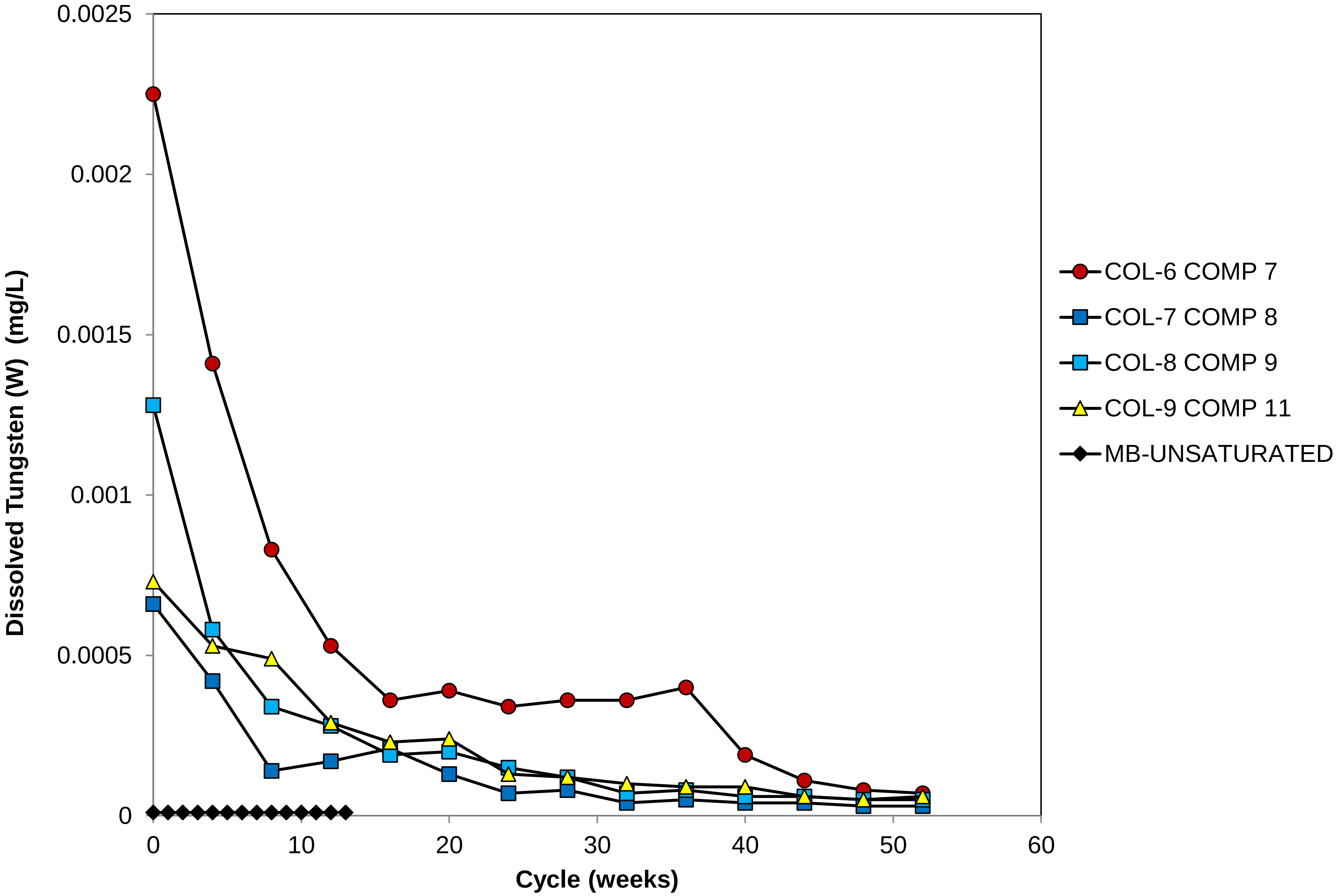

0.00012
0.0001
0.00008
0.00006
0.00004
0.00002
0
Dissolved Tellurium (Te) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

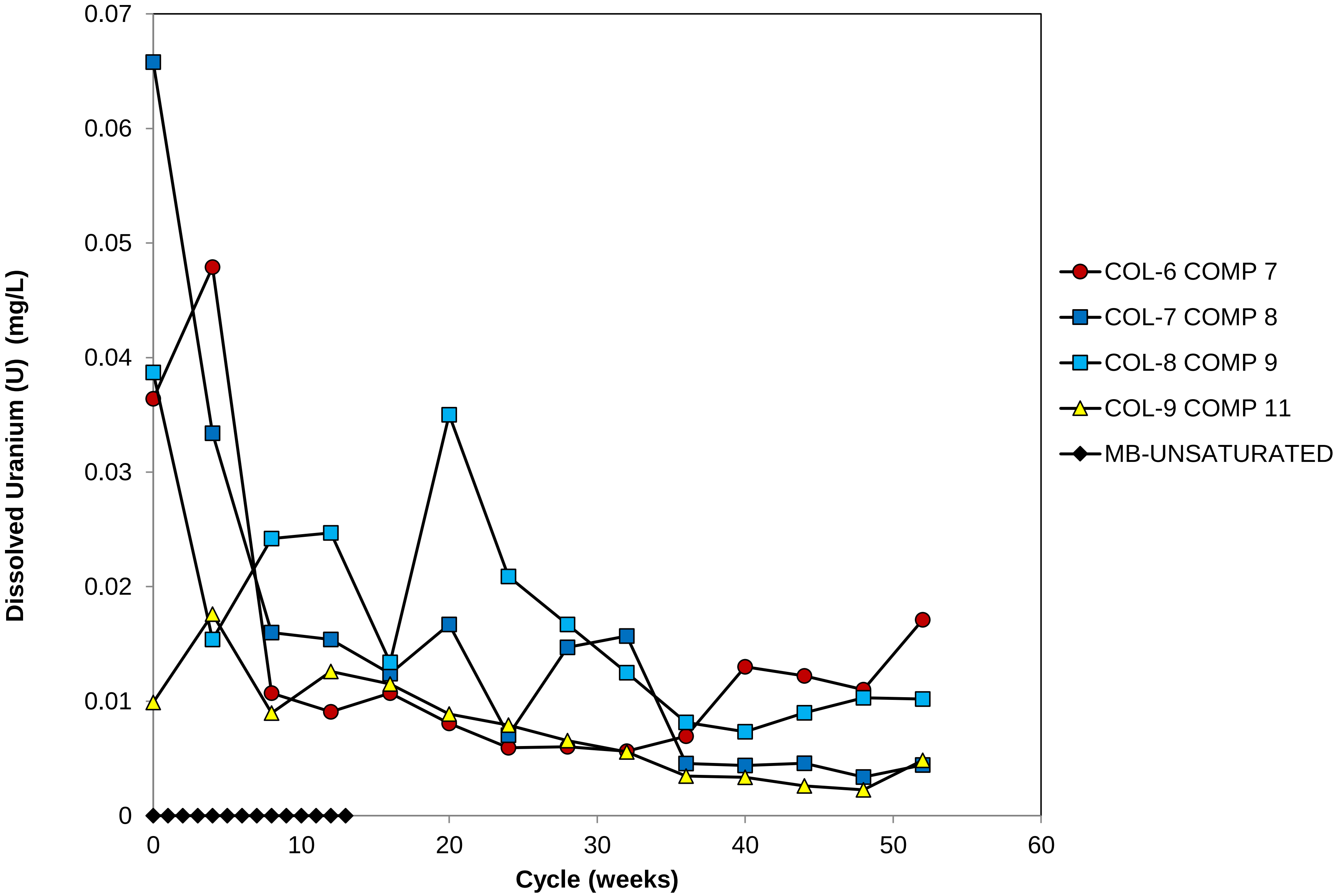

0.0004
0.00035
0.0003
0.00025
0.0002
0.00015
0.0001
0.00005
0
Dissolved Thallium (Tl) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

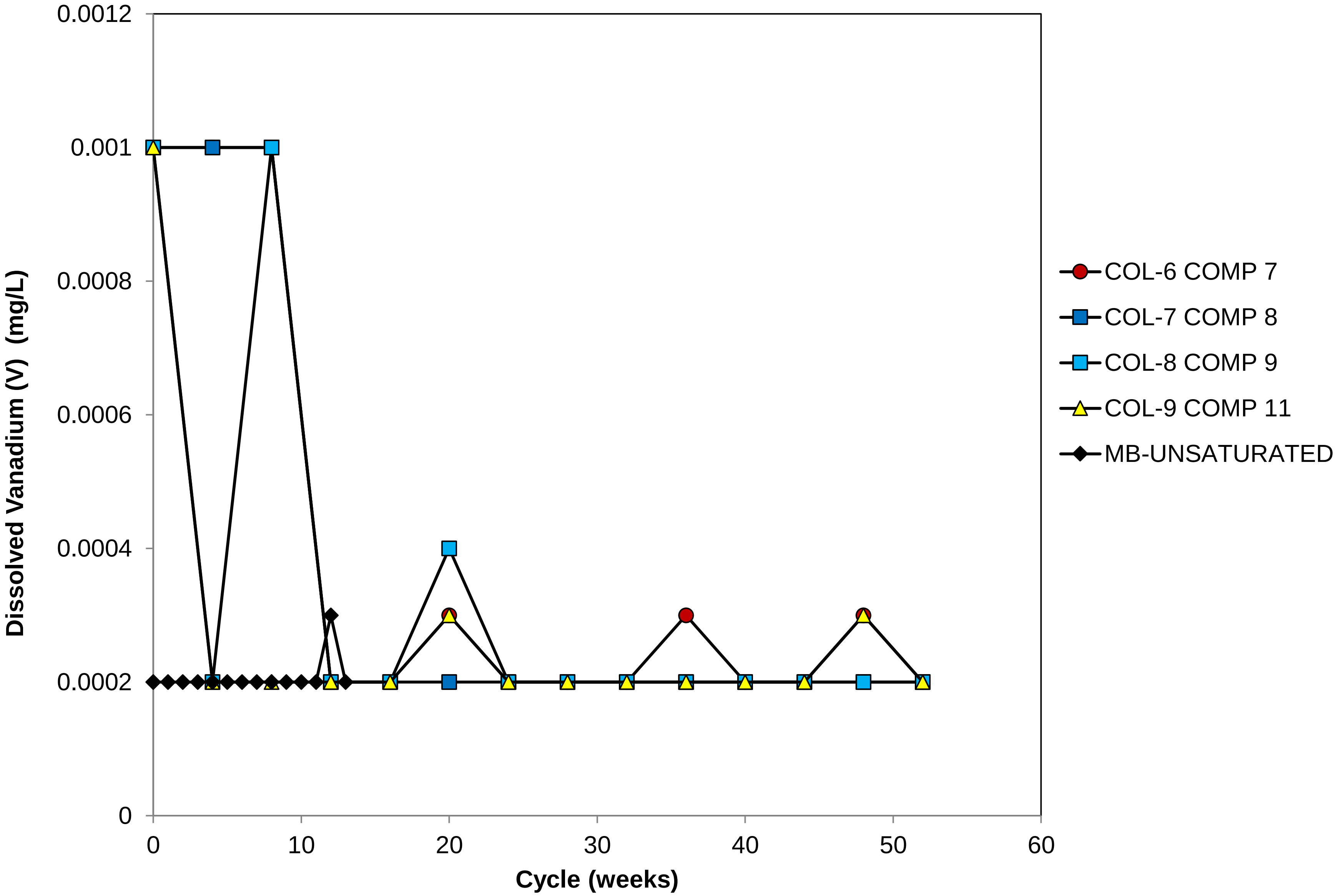

0.000035
0.00003
0.000025
0.00002
0.000015
0.00001
0.000005
0
Dissolved Thorium (Th) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

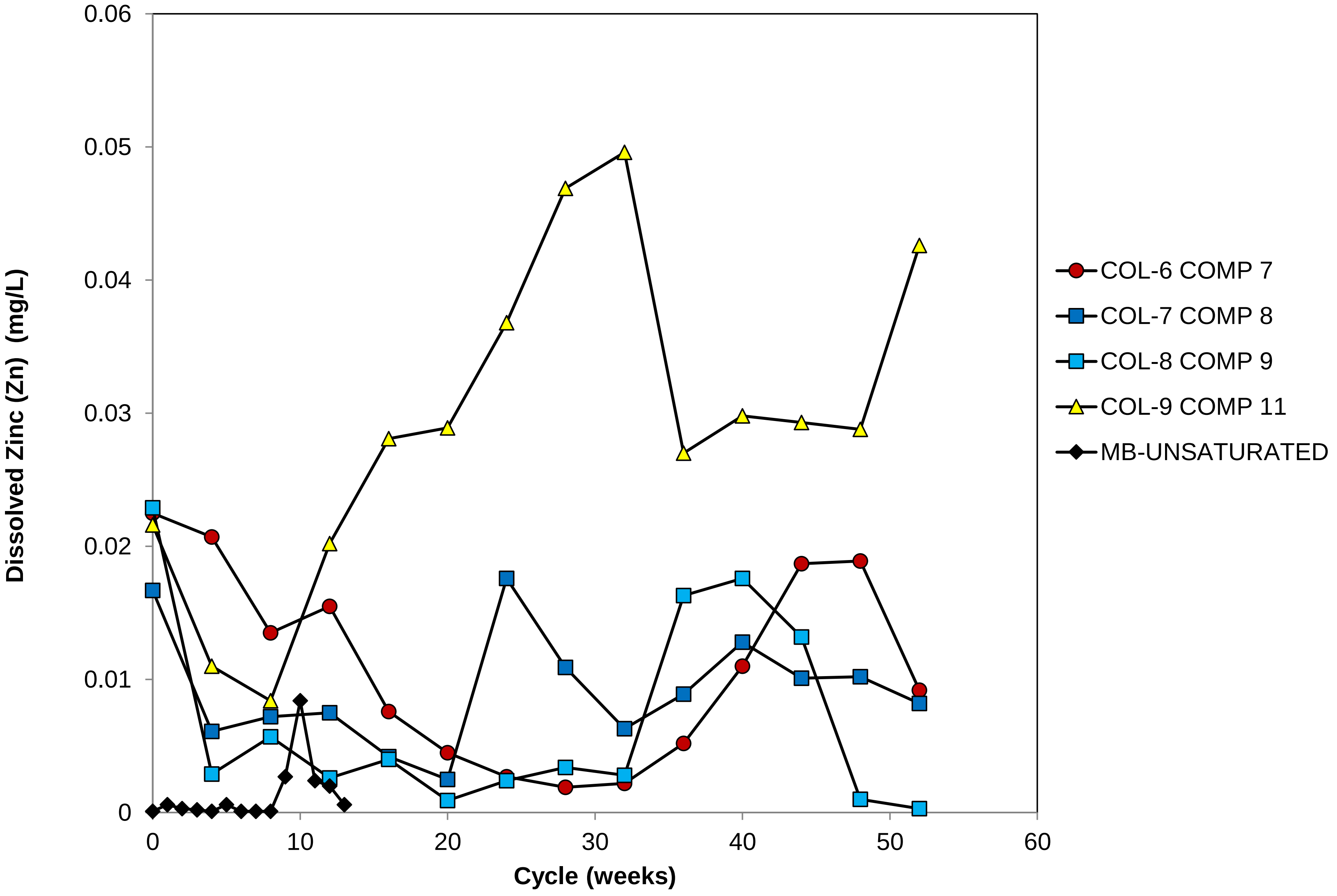

0.0025
0.002
0.0015
0.001
0.0005
0
Dissolved Tin (Sn) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

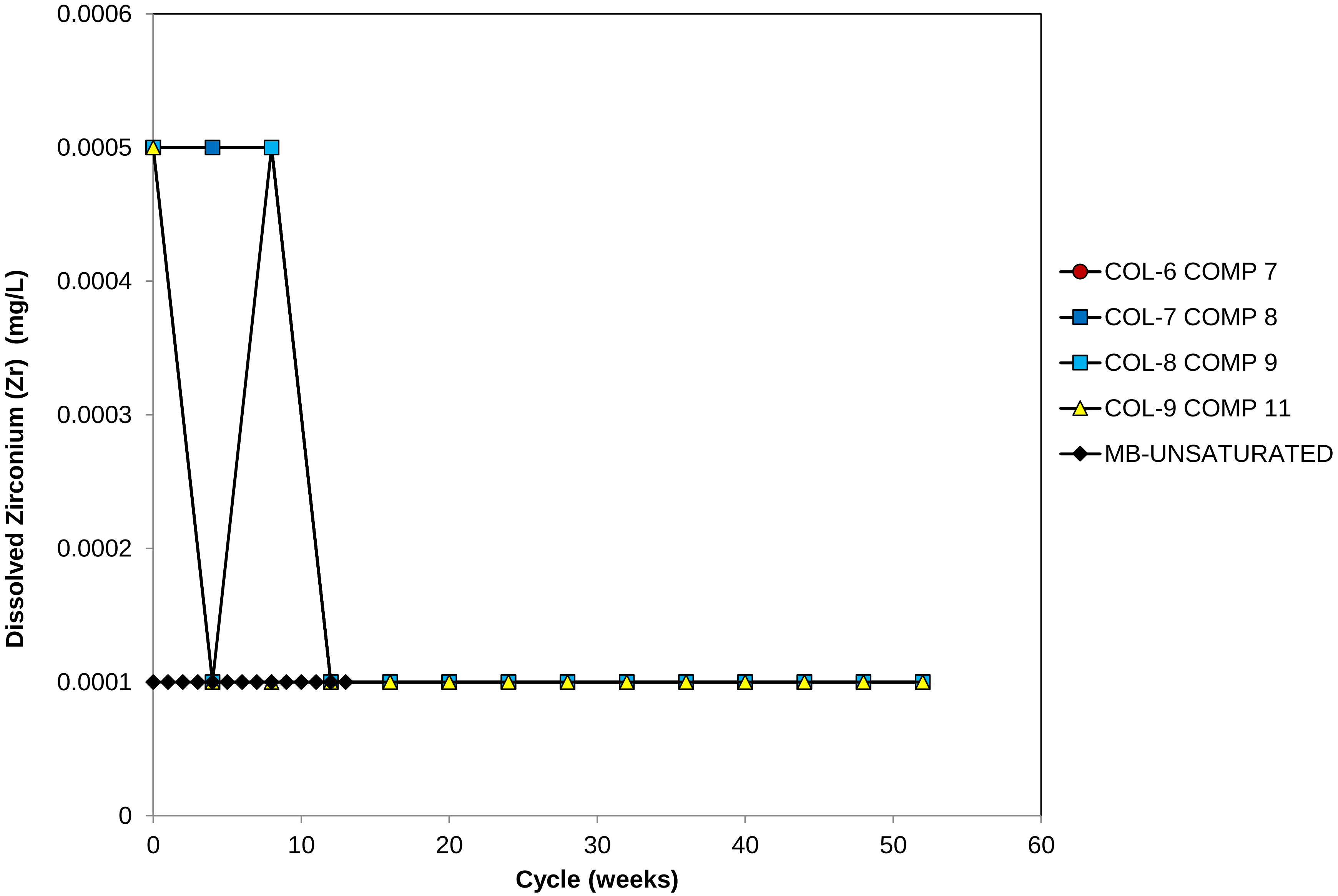

0.0045
0.004
0.0035
0.003
0.0025
0.002
0.0015
0.001
0.0005
0
Dissolved Titanium (Ti) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

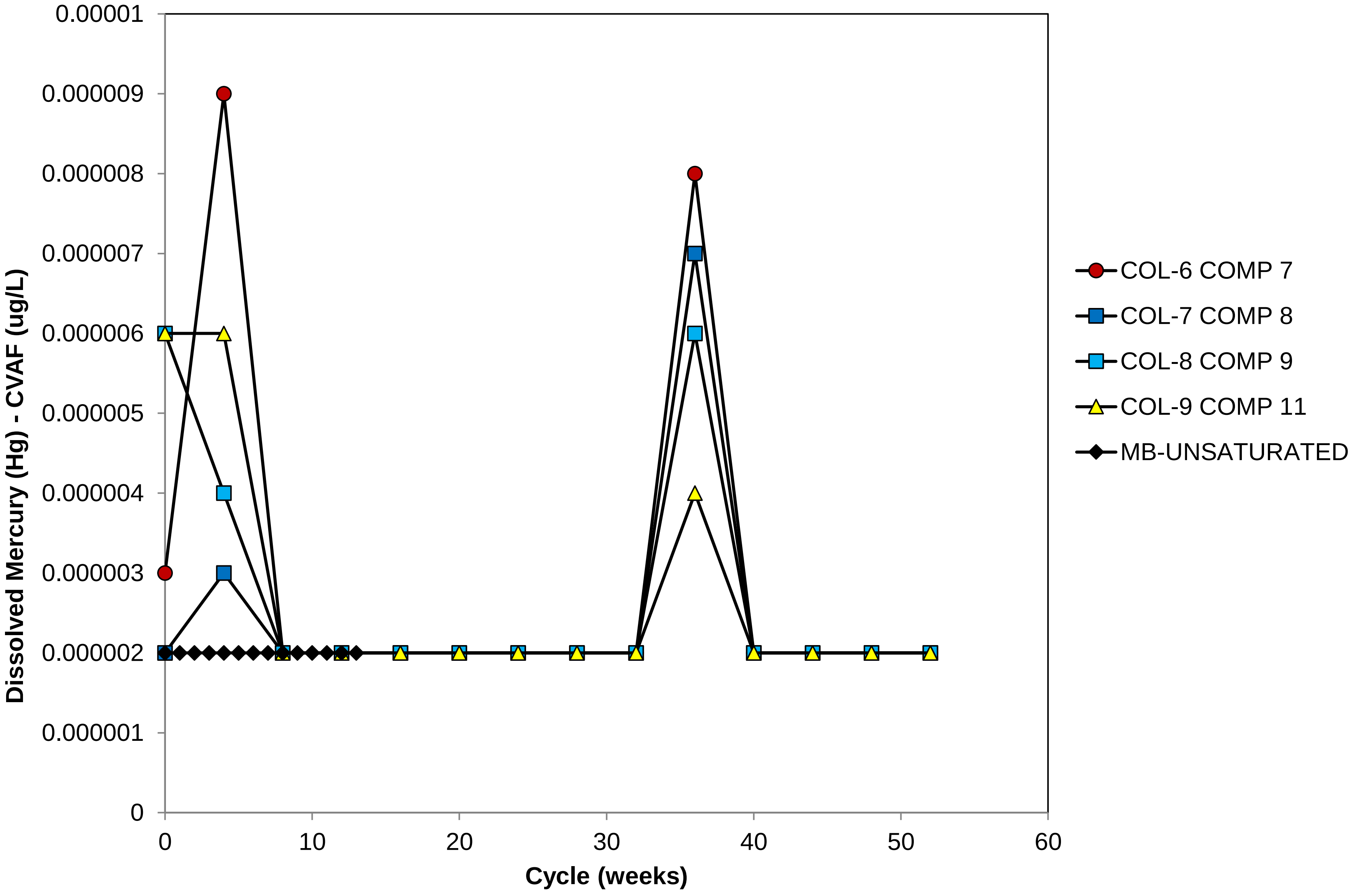

0.0025
0.002
0.0015
0.001
0.0005
0
Dissolved Tungsten (W) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED


0.07
0.06
0.05
0.04
0.03
0.02
0.01
0
Dissolved Uranium (U) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

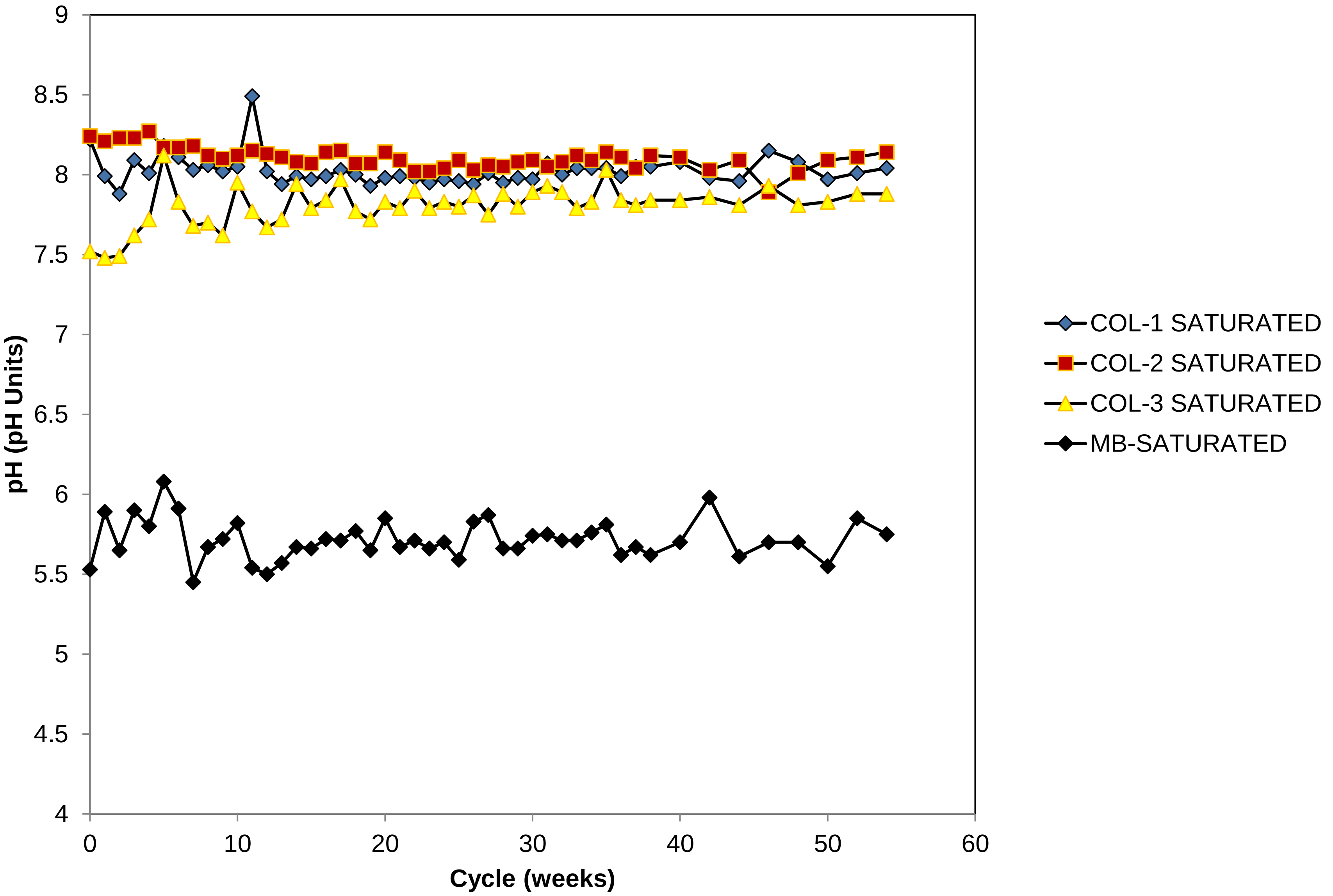

0.0012
0.001
0.0008
0.0006
0.0004
0.0002
0
Dissolved Vanadium (V) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED


0.06
0.05
0.04
0.03
0.02
0.01
0
Dissolved Zinc (Zn) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED


0.0006
0.0005
0.0004
0.0003
0.0002
0.0001
0
Dissolved Zirconium (Zr) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED


0.00001
0.000009
0.000008
0.000007
0.000006
0.000005
0.000004
0.000003
0.000002
0.000001
0
Dissolved Mercury (Hg) - CVAF (ug/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-6 COMP 7
COL-7 COMP 8
COL-8 COMP 9
COL-9 COMP 11
MB-UNSATURATED

C3: Saturated Columns



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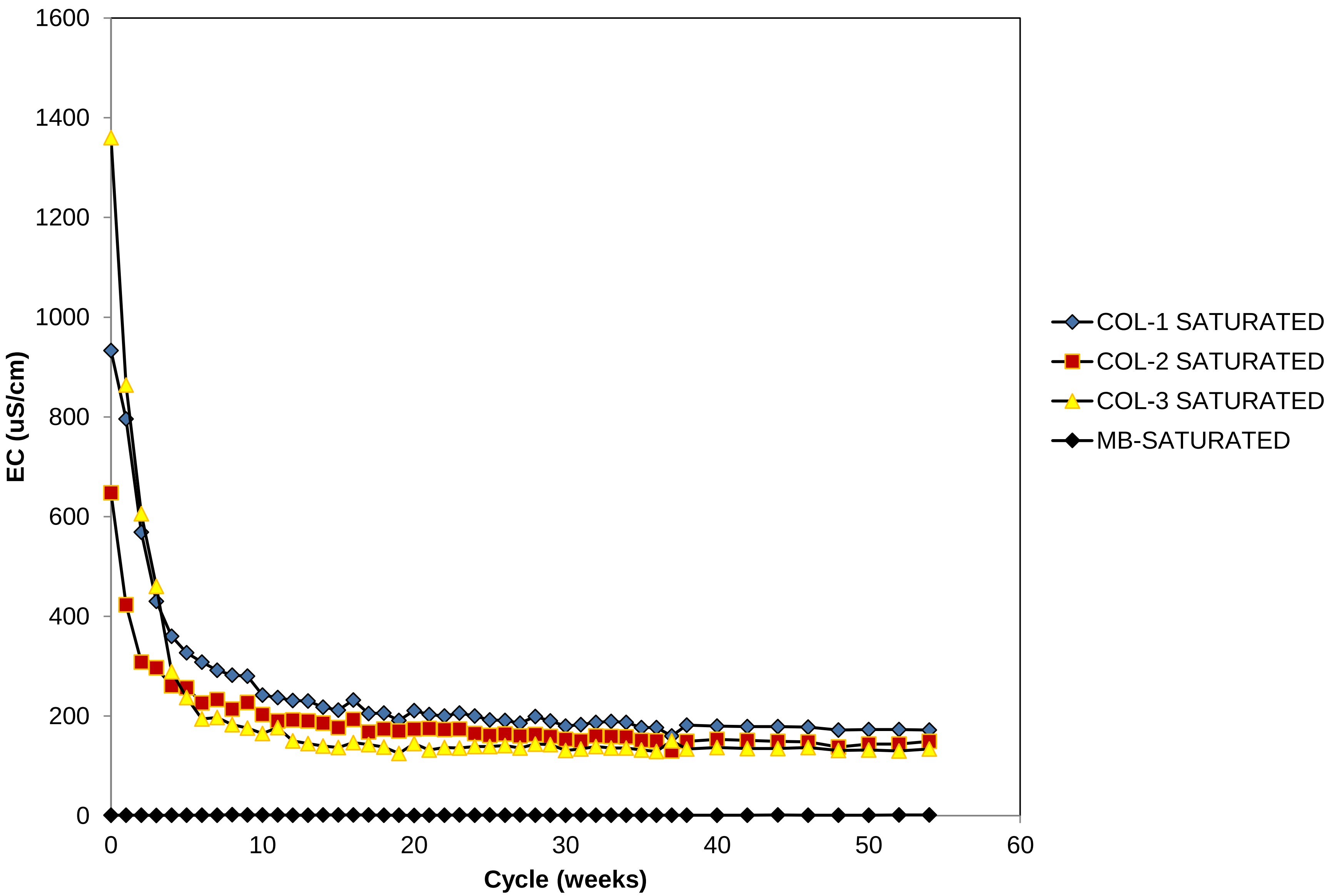

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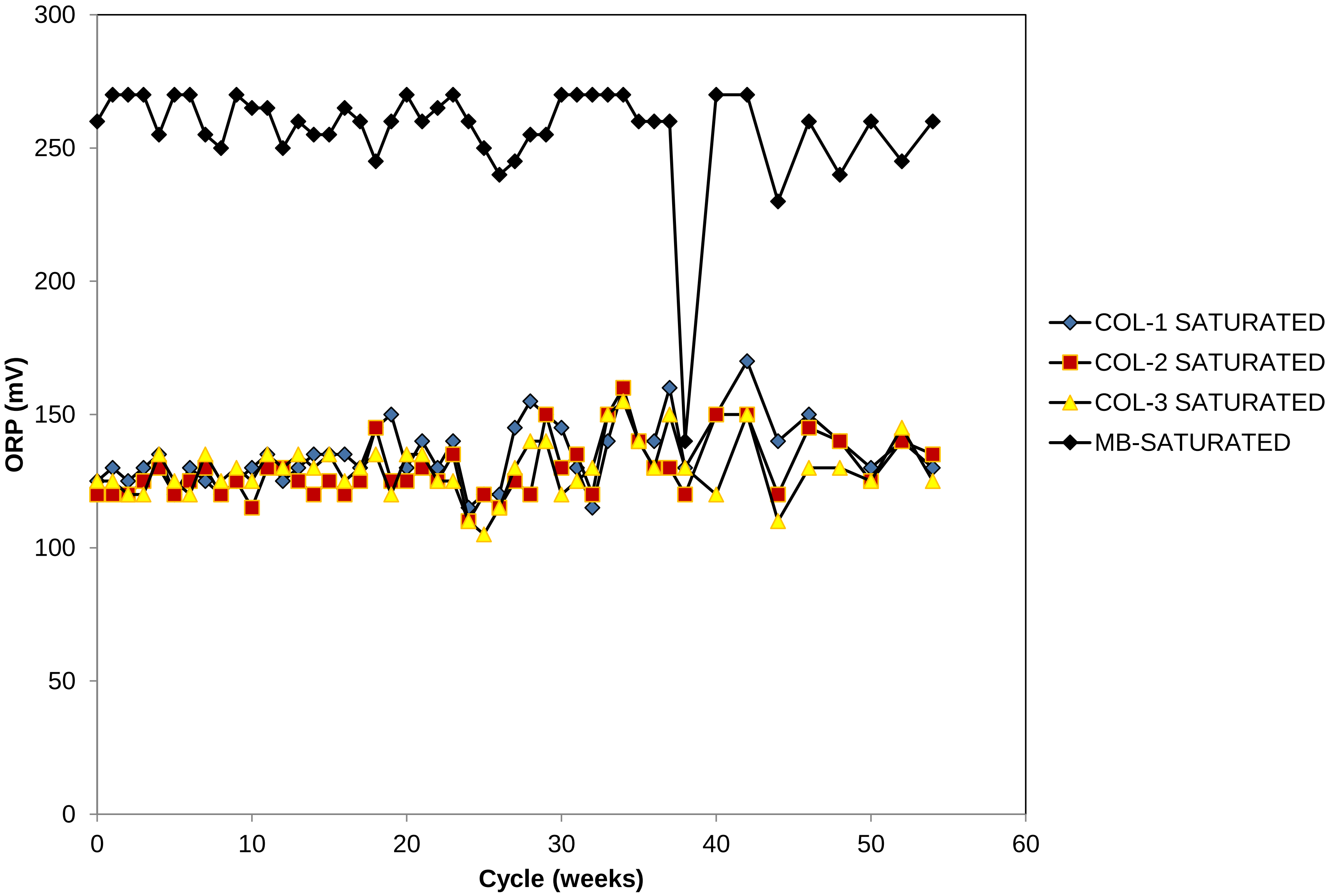

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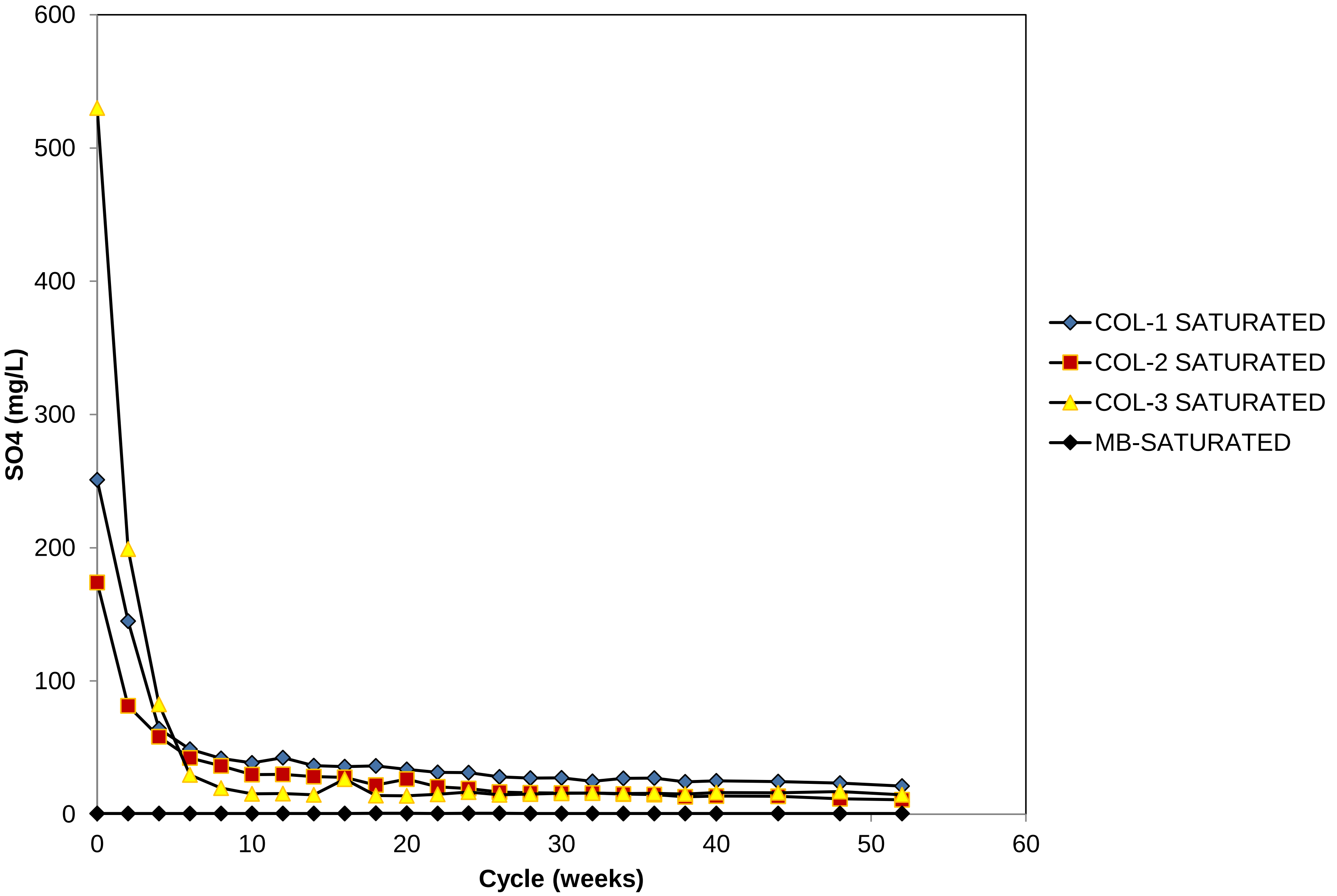

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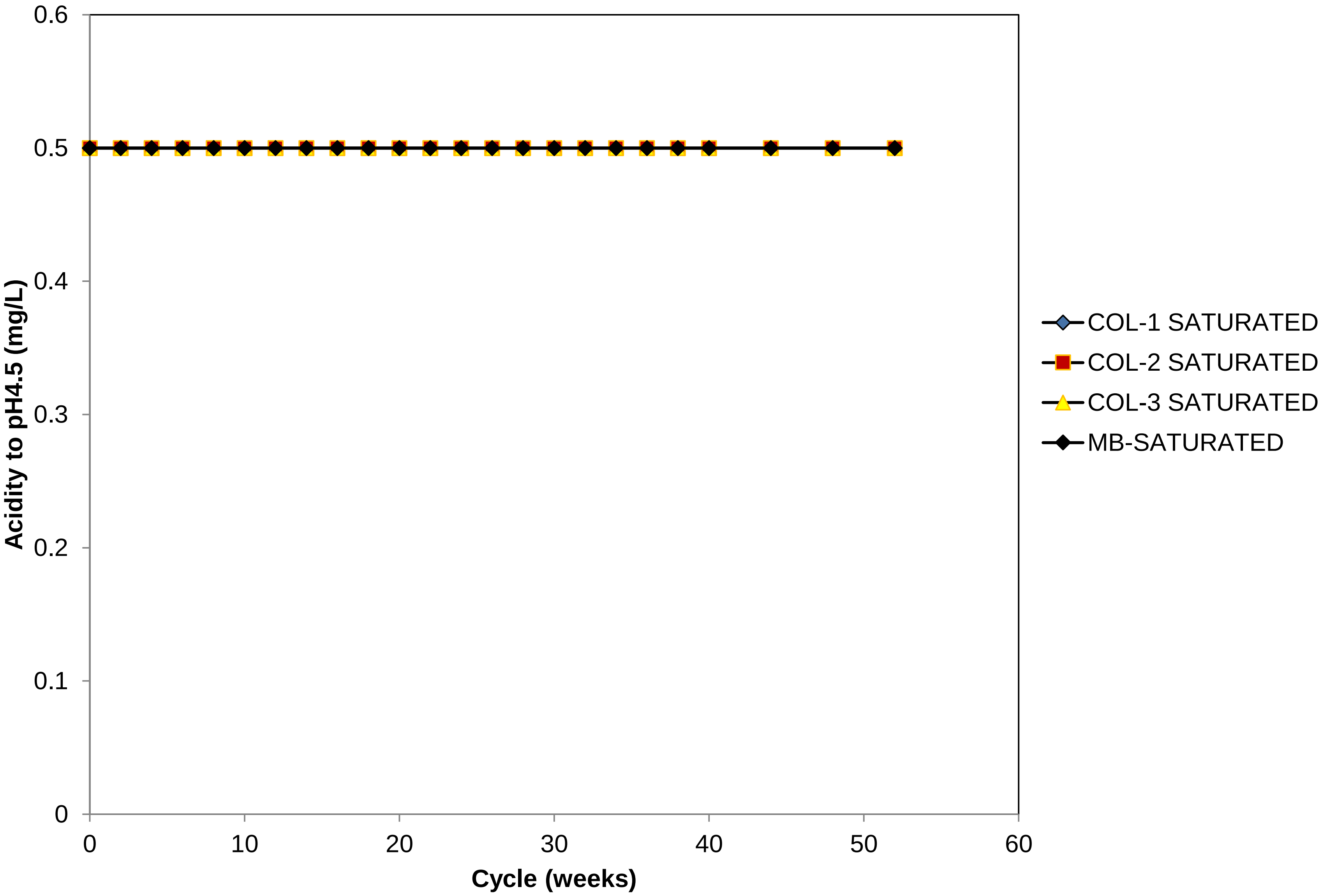

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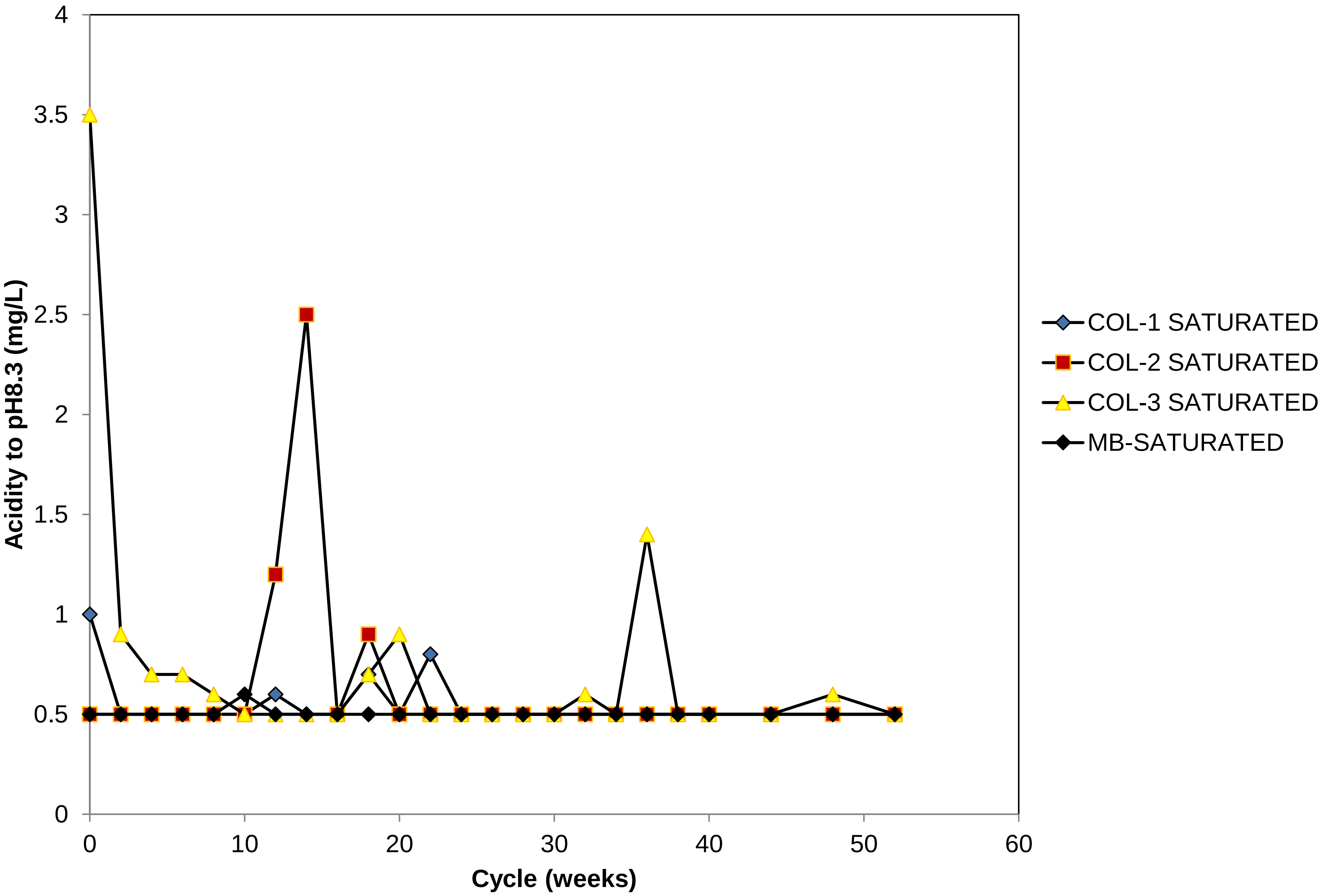

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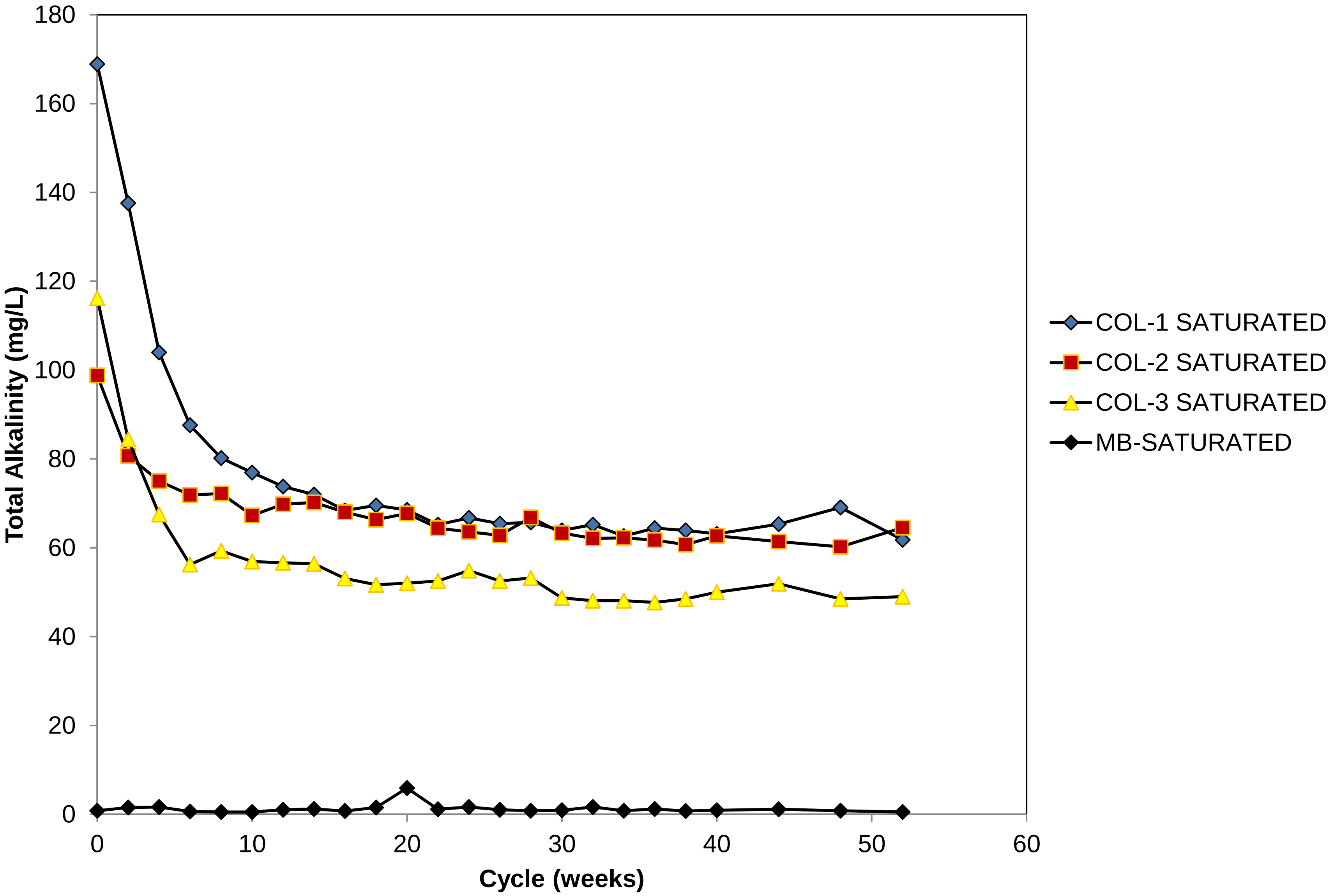

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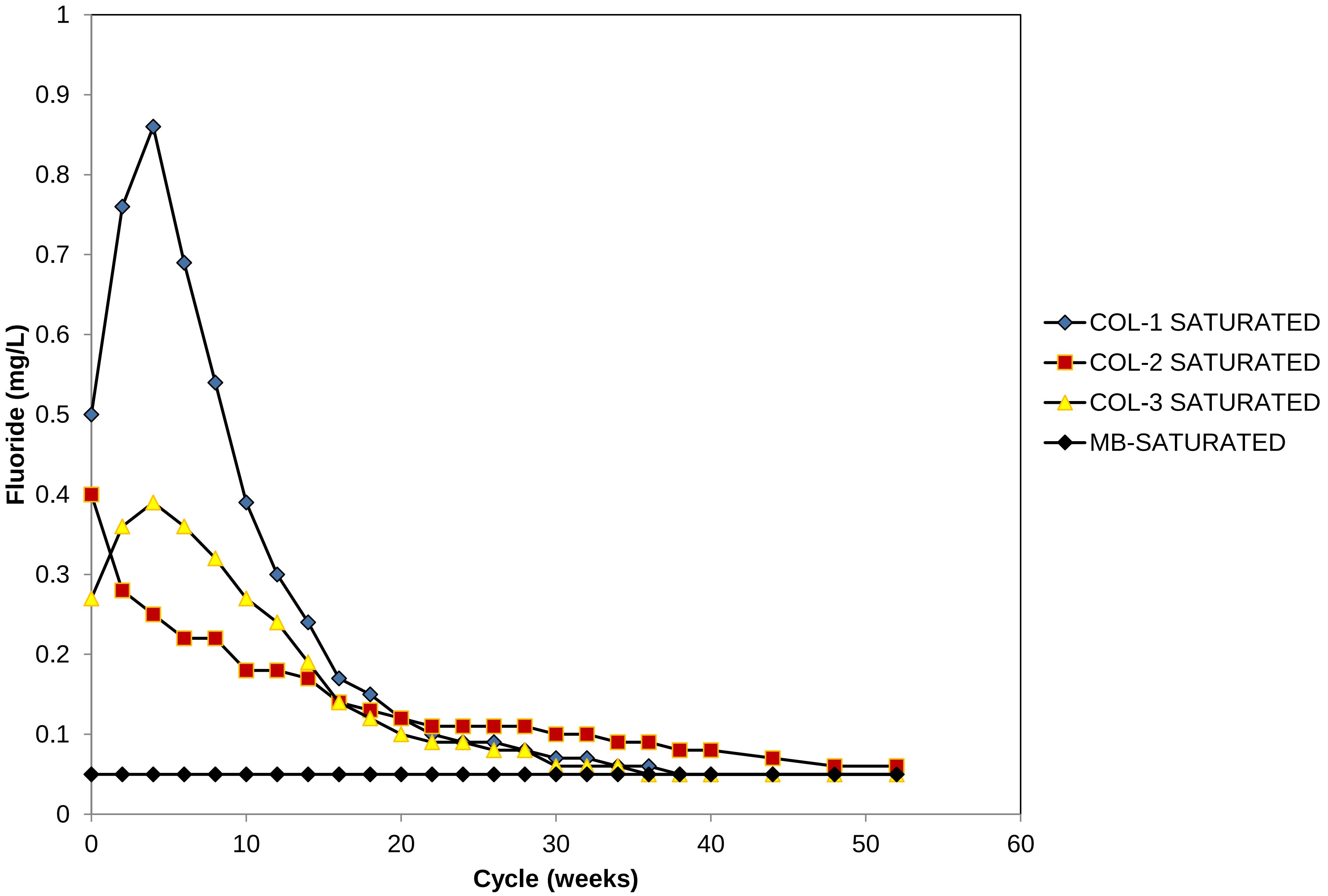

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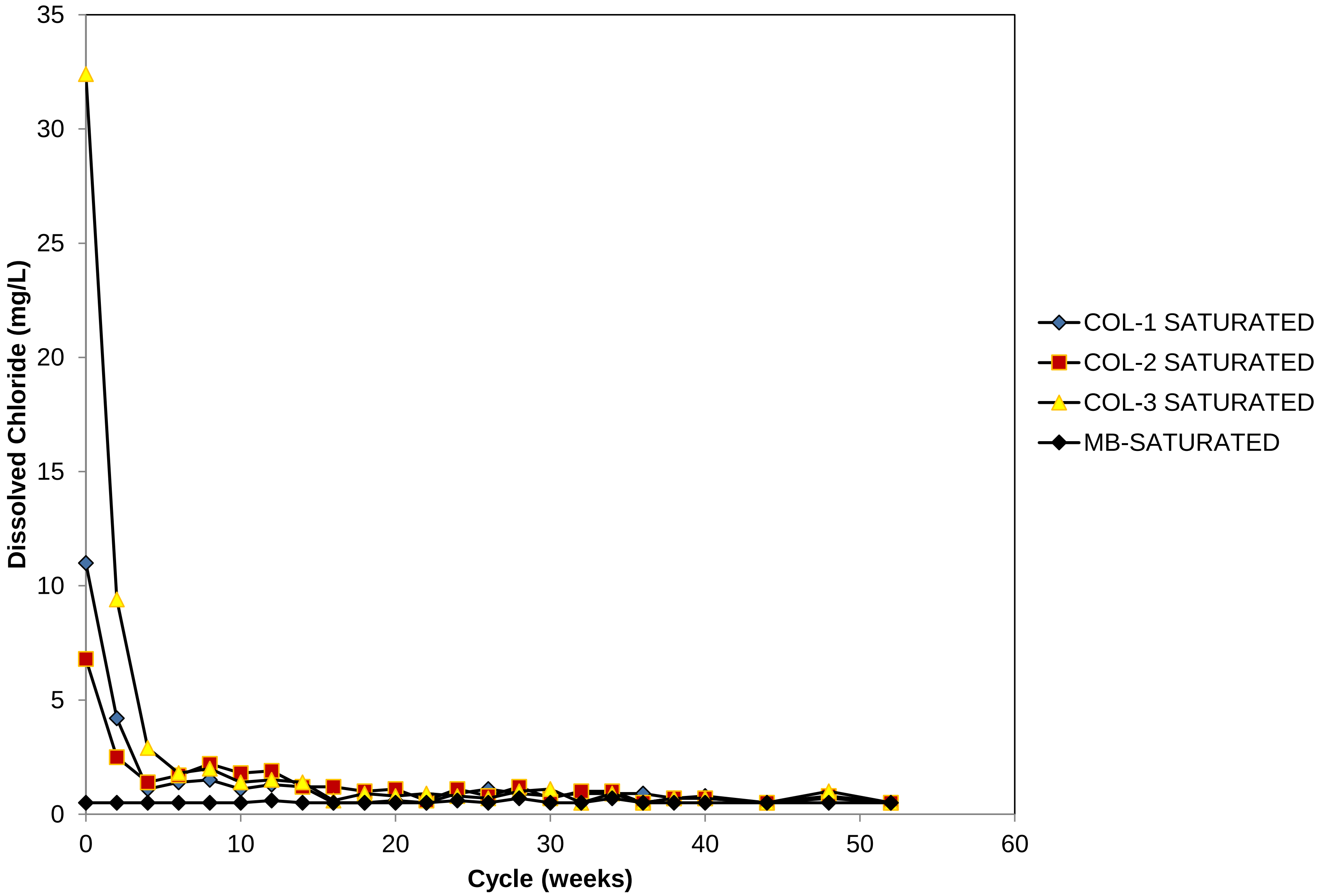

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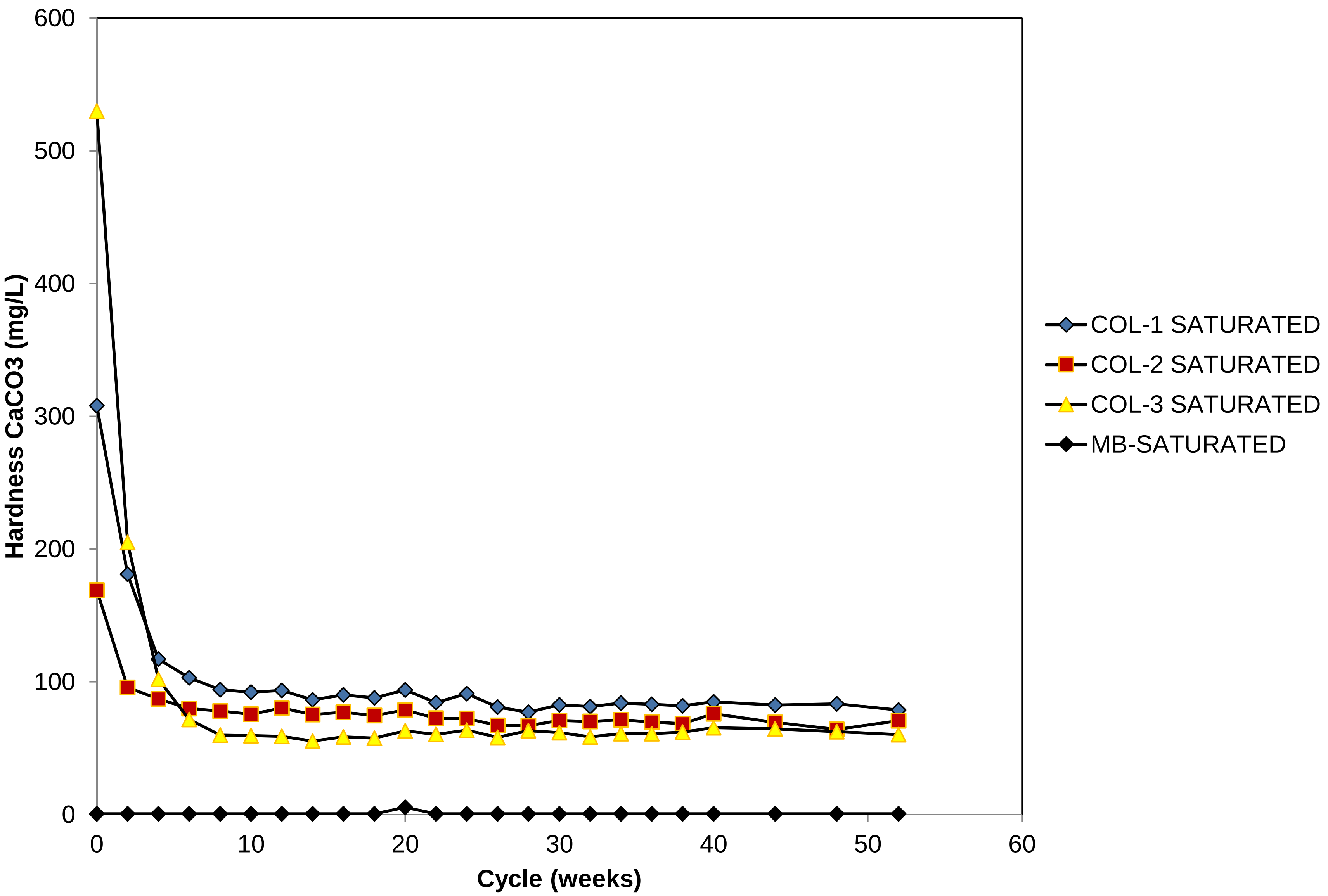

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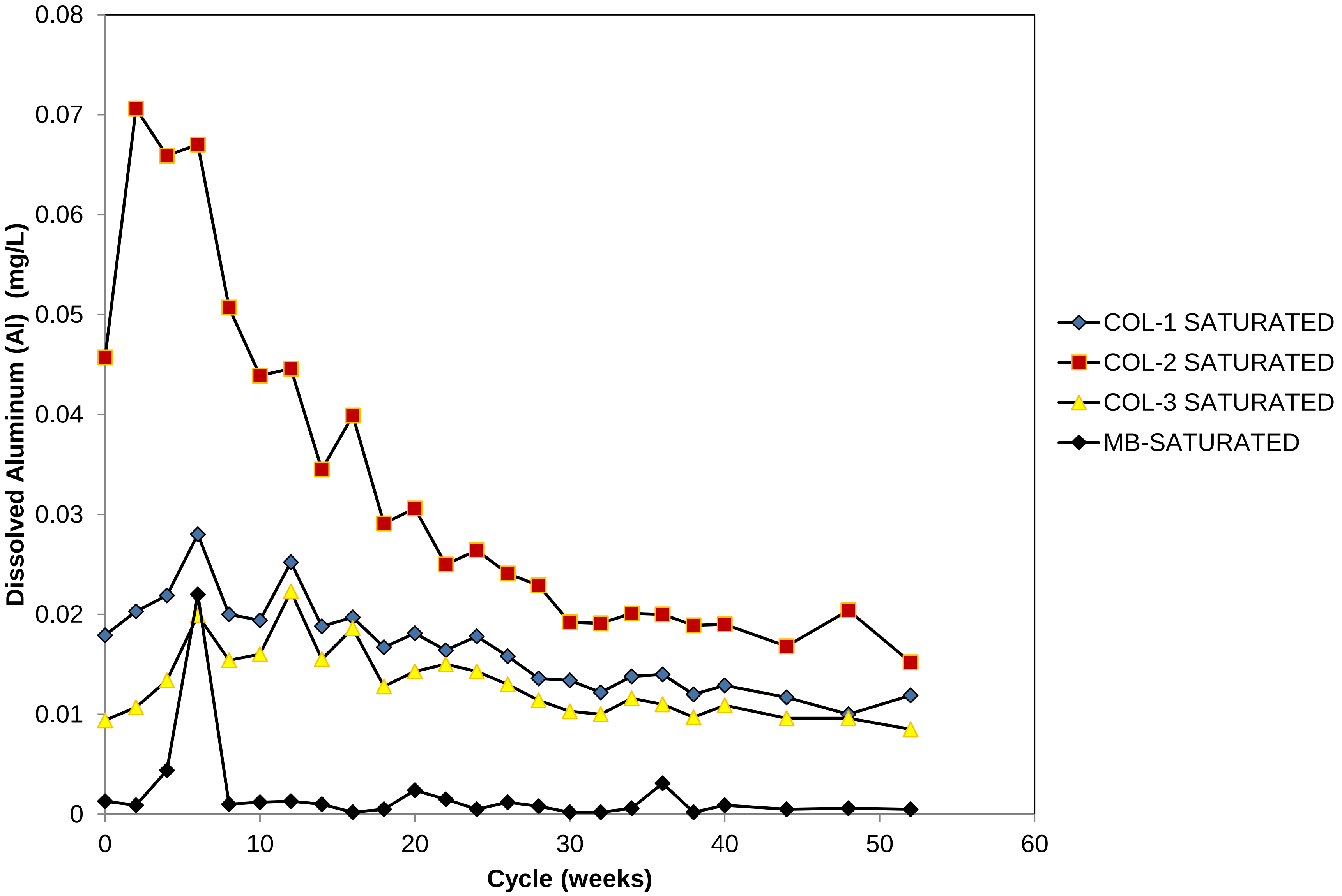

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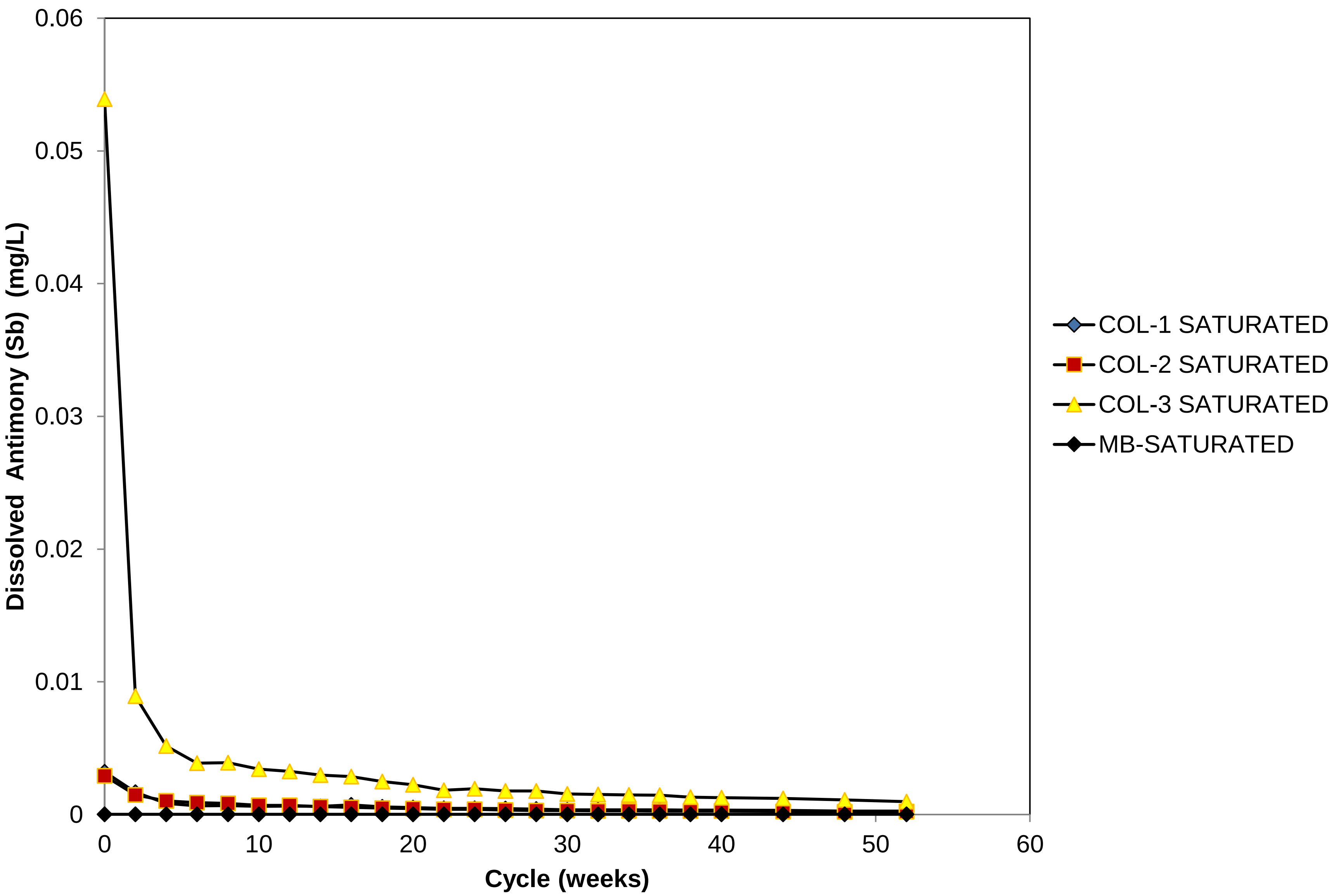

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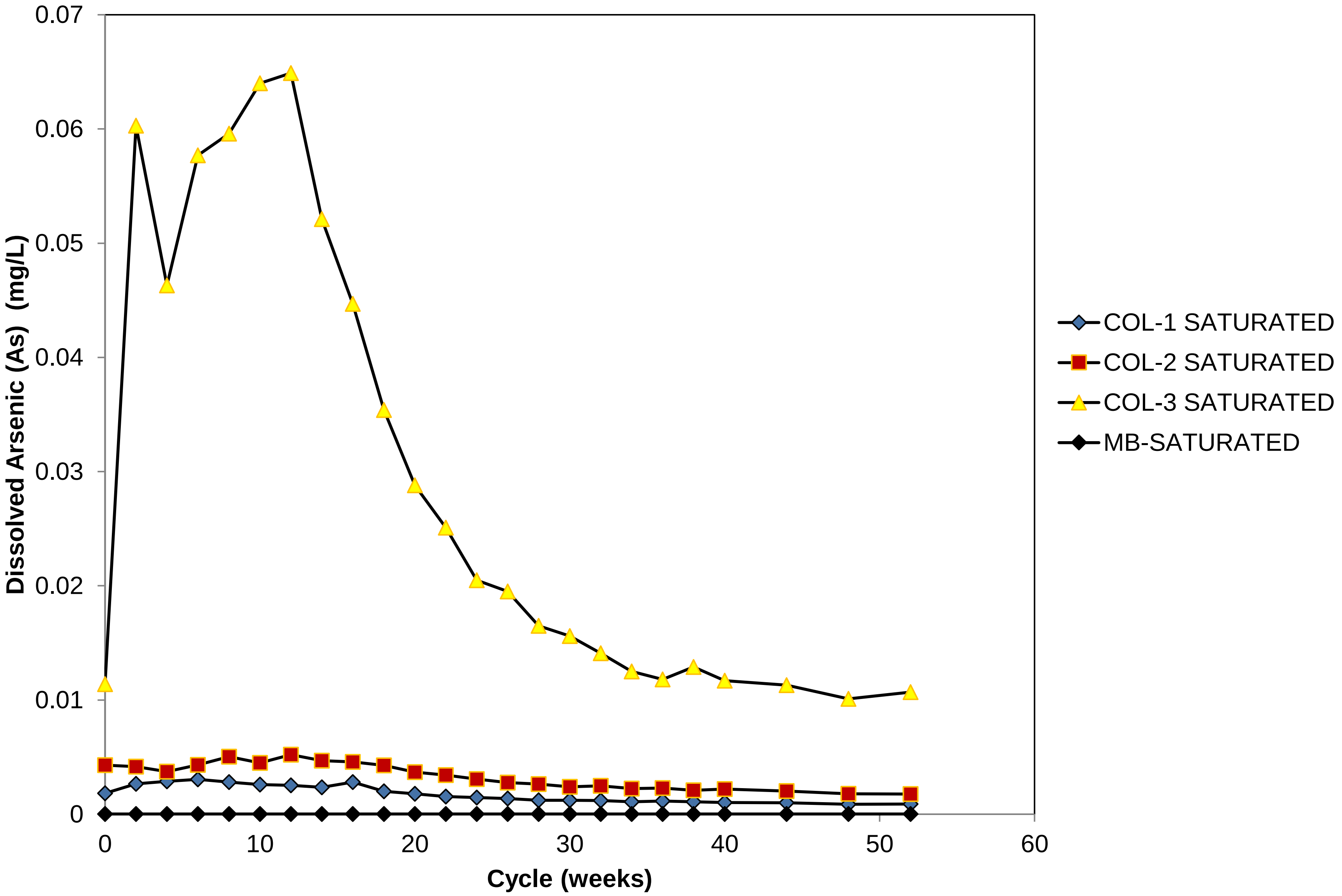

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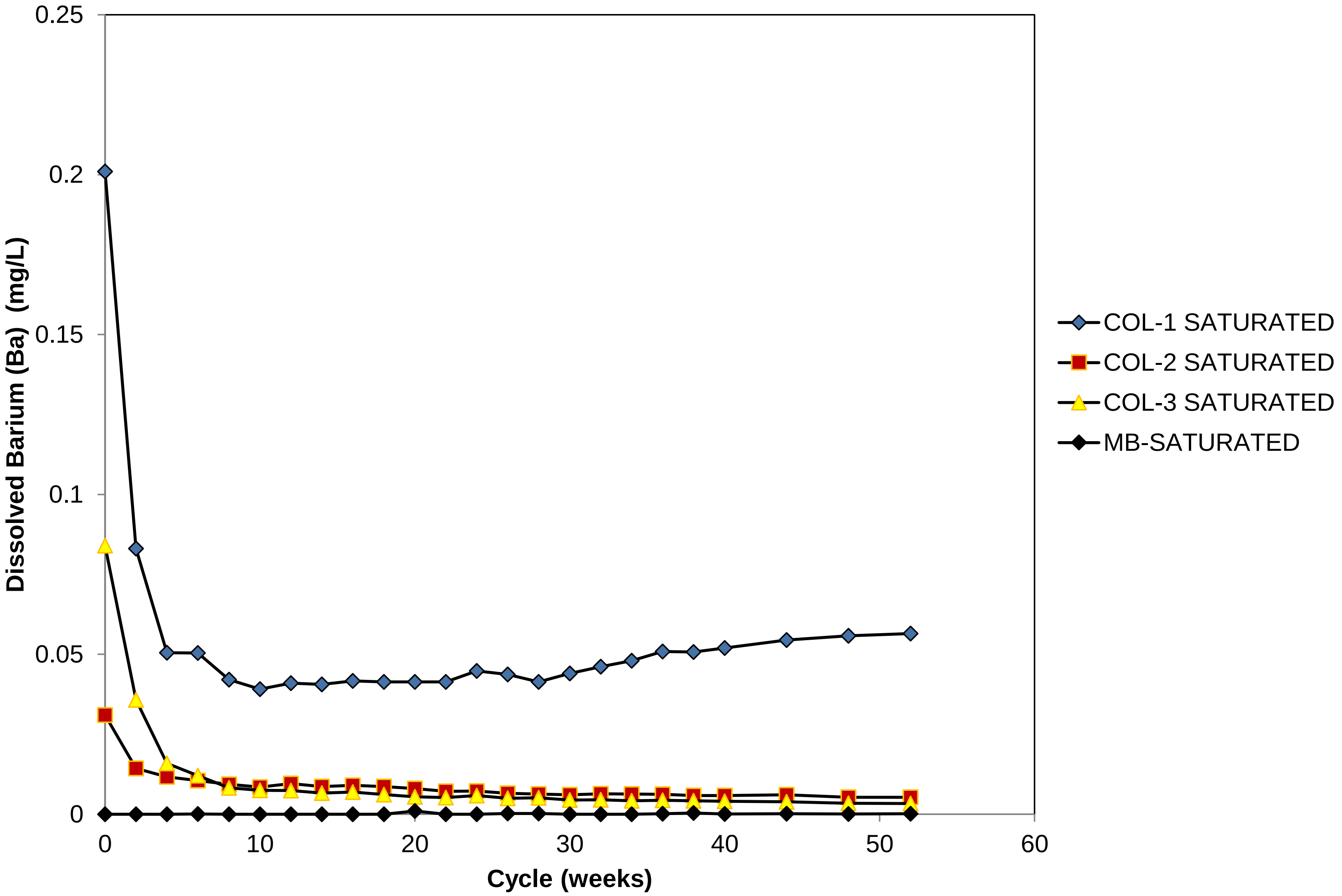

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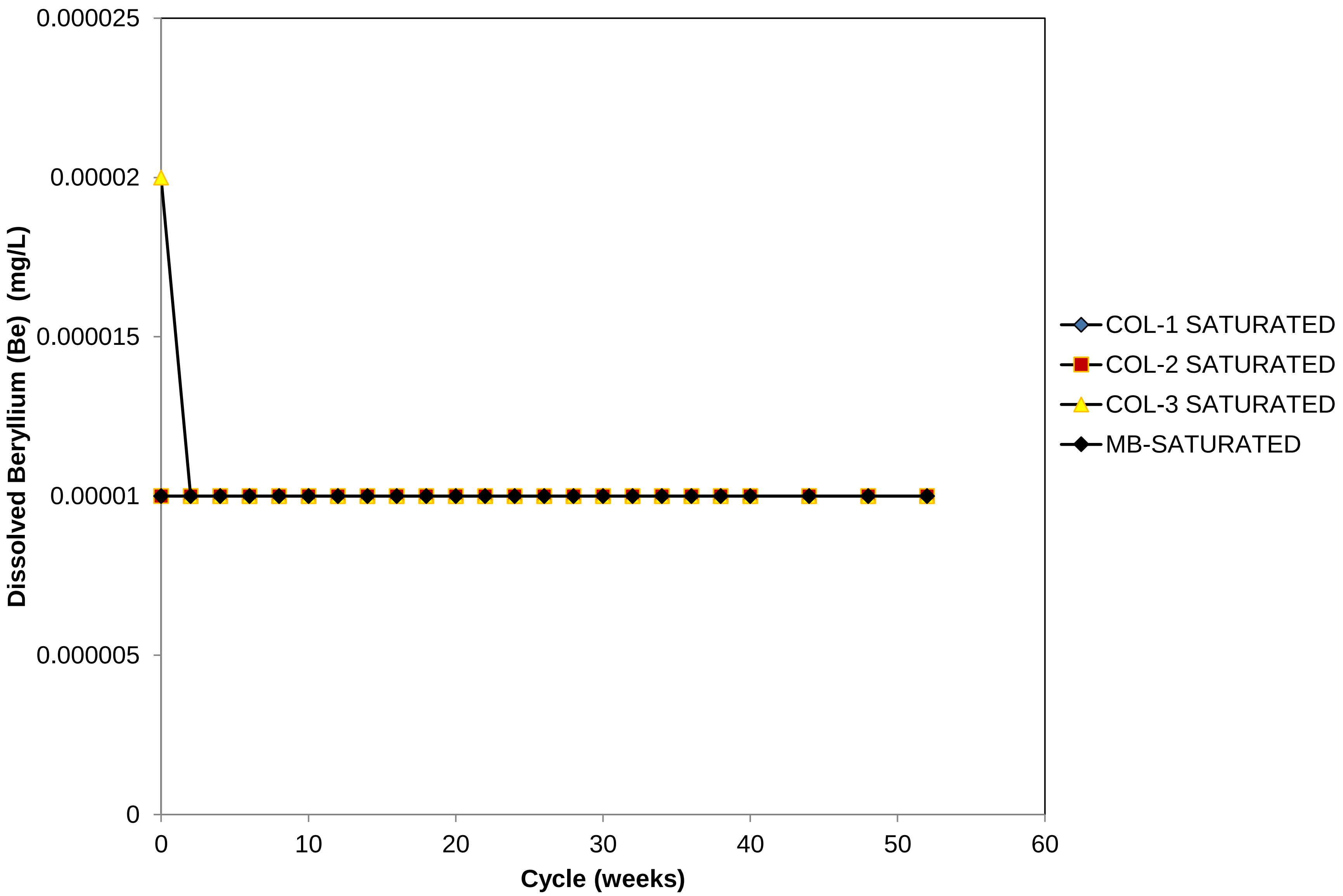

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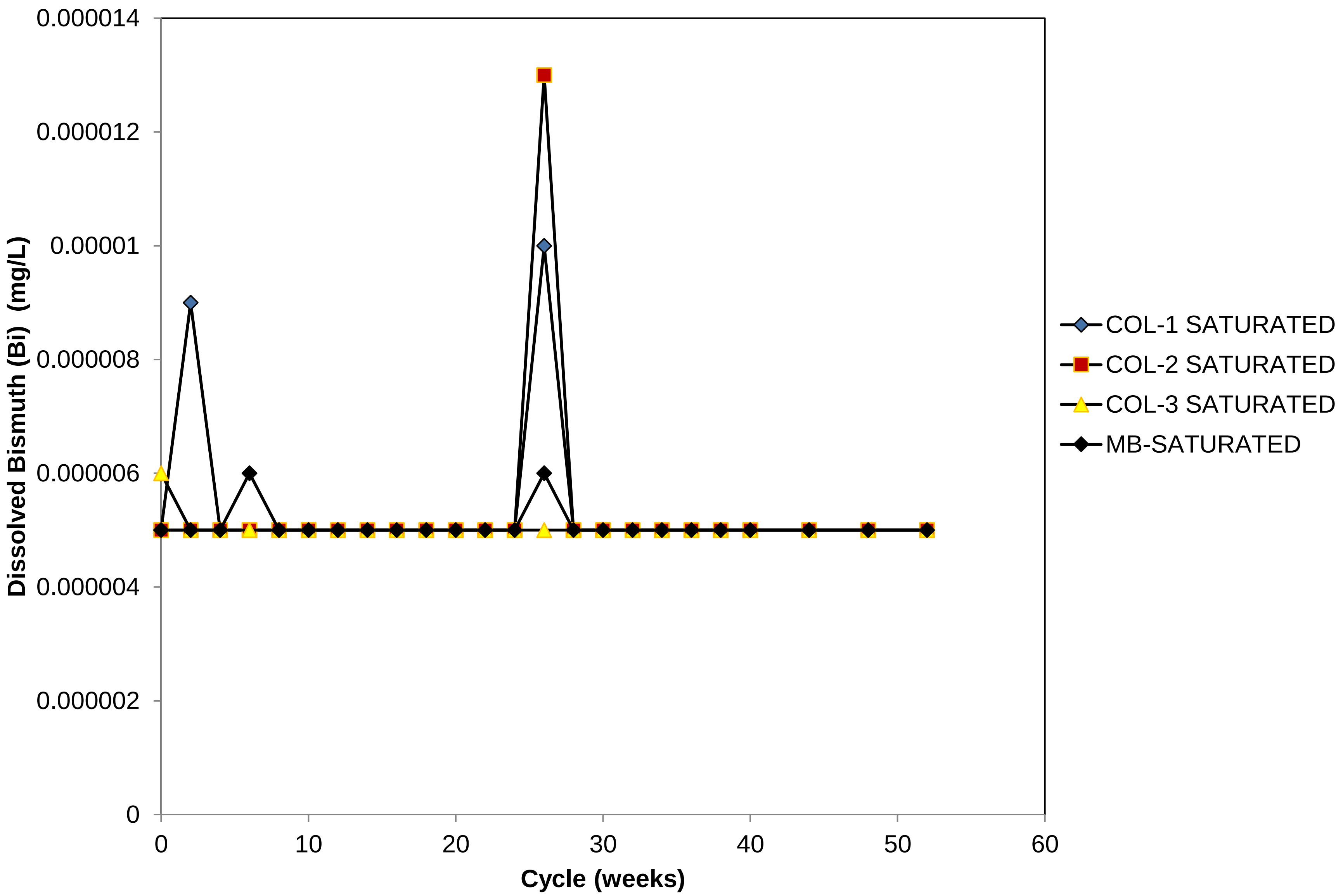

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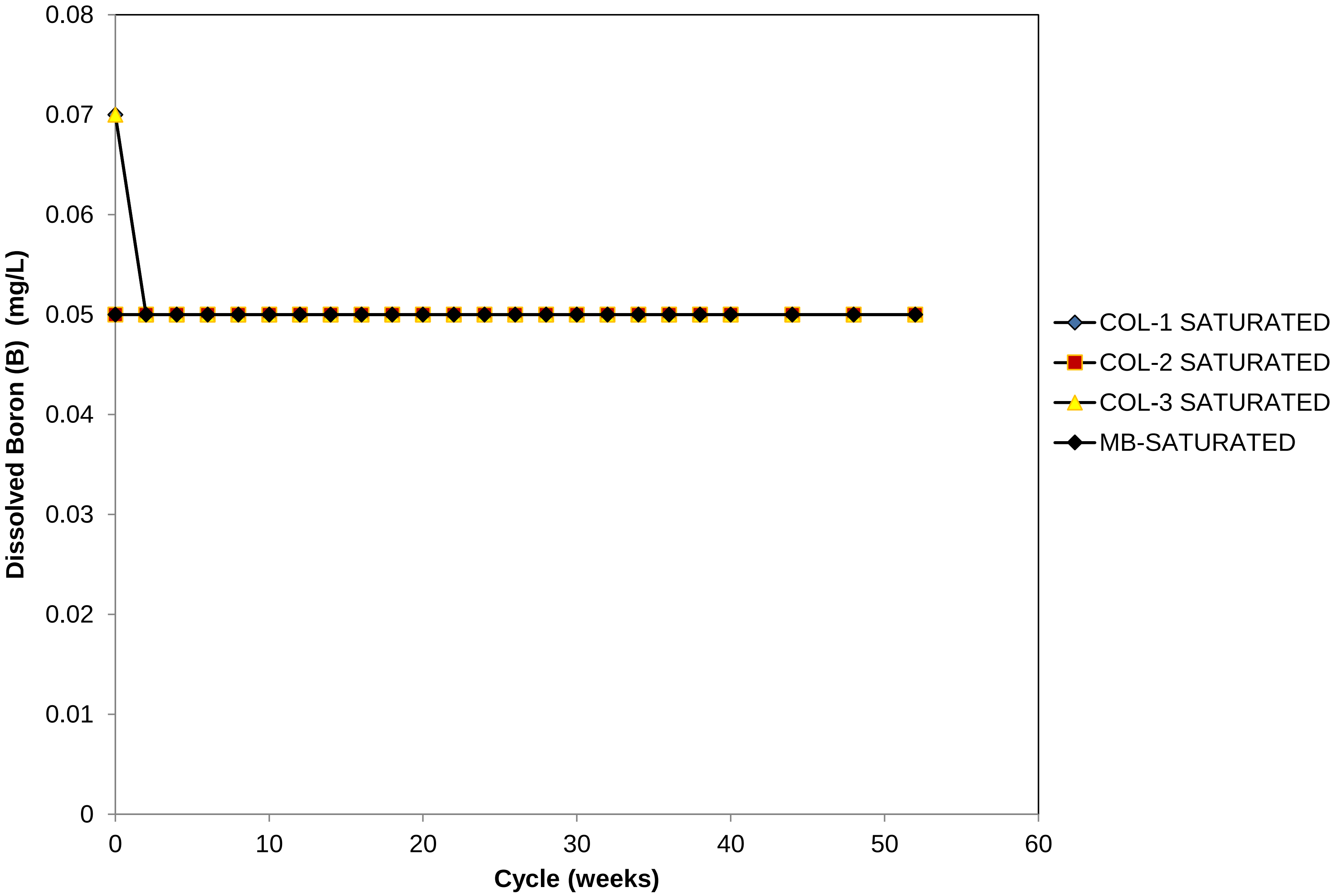

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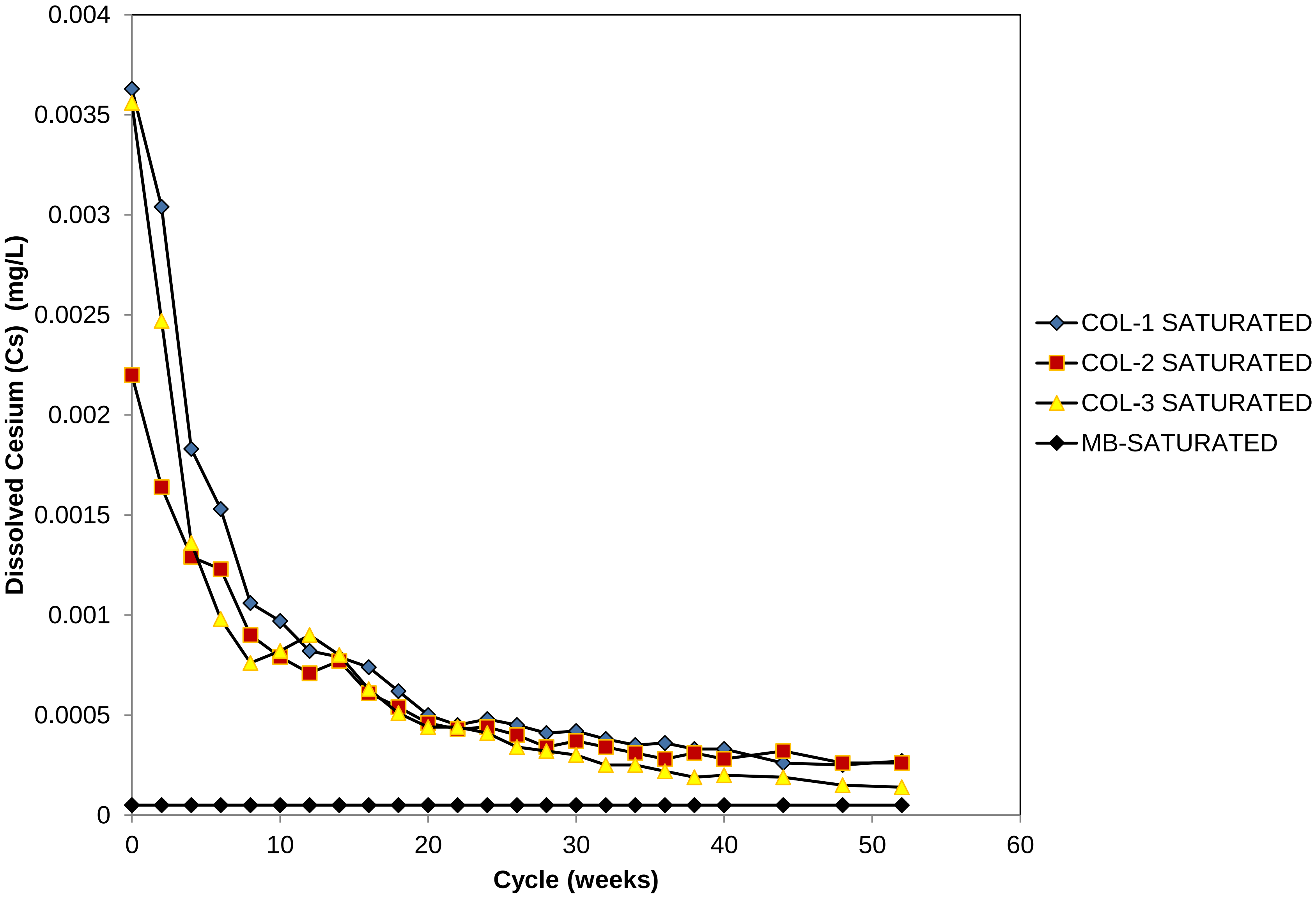

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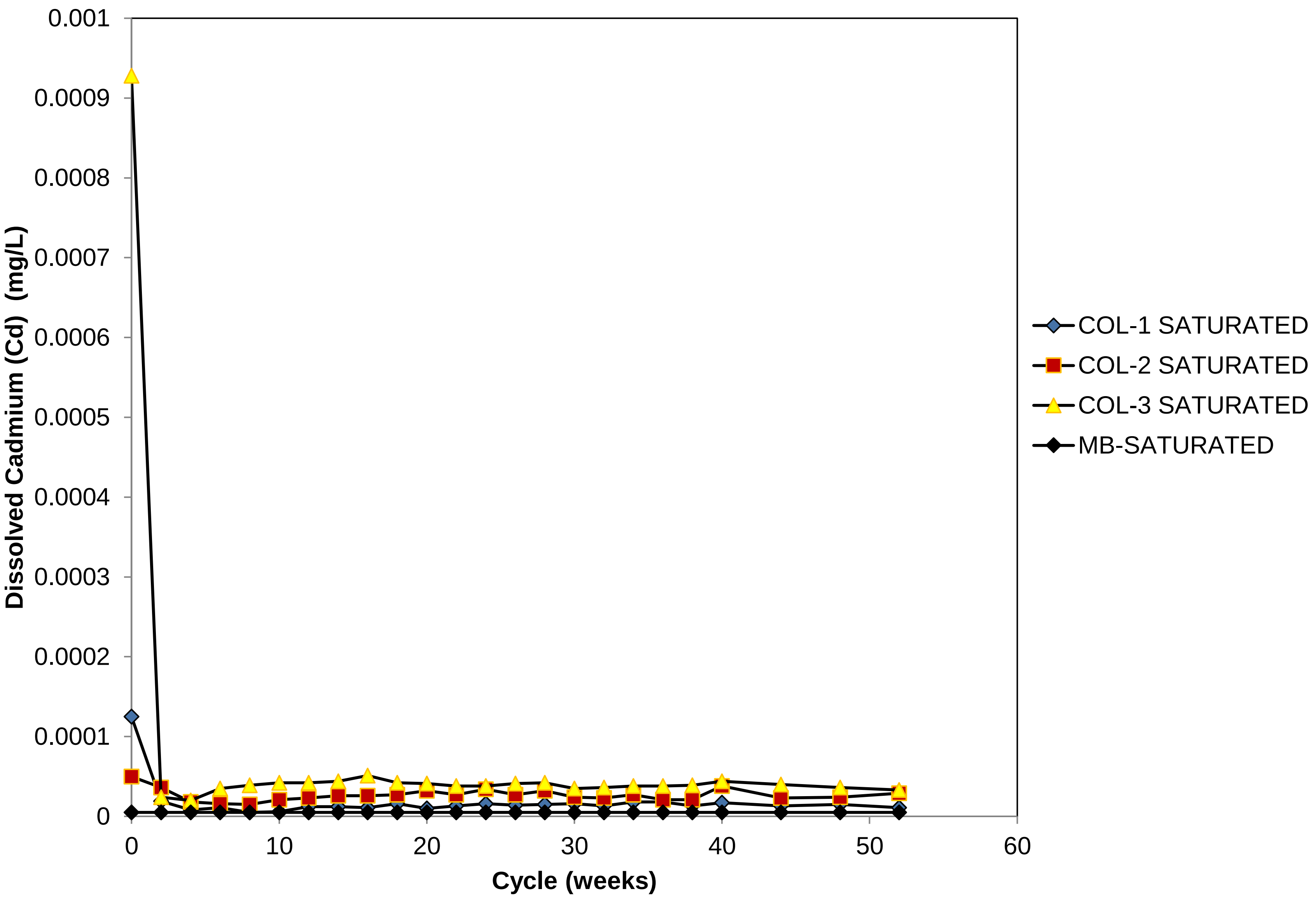

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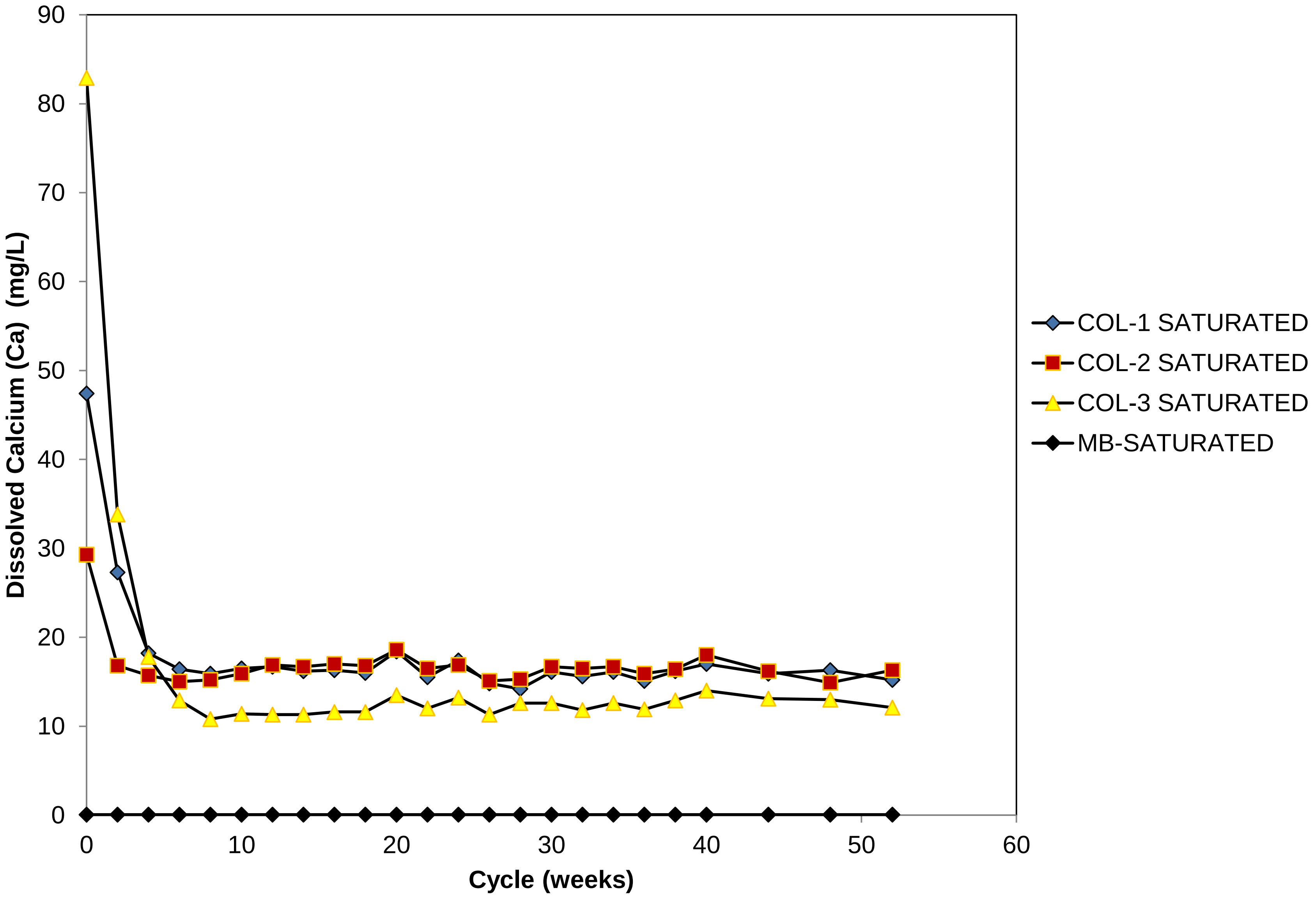

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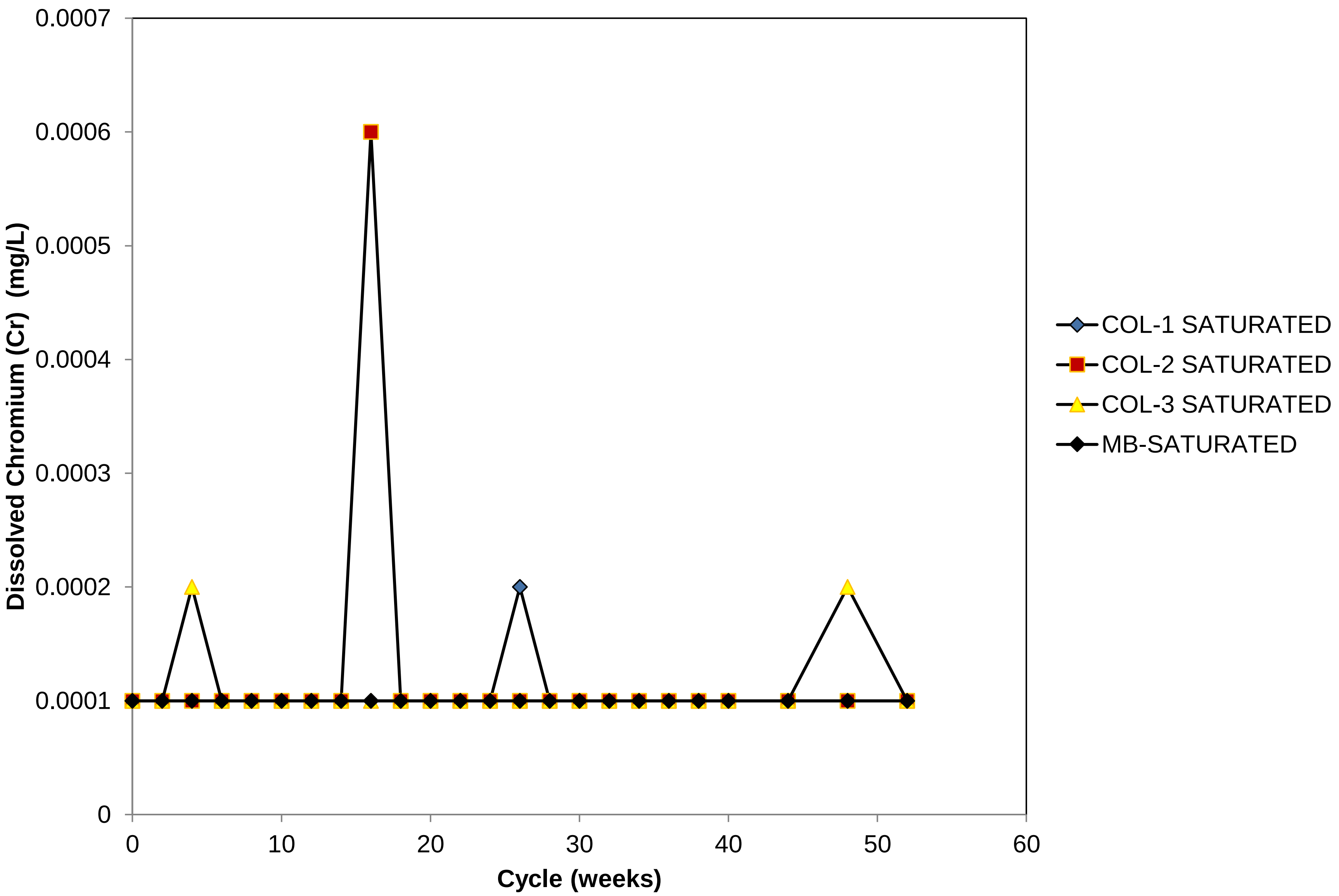

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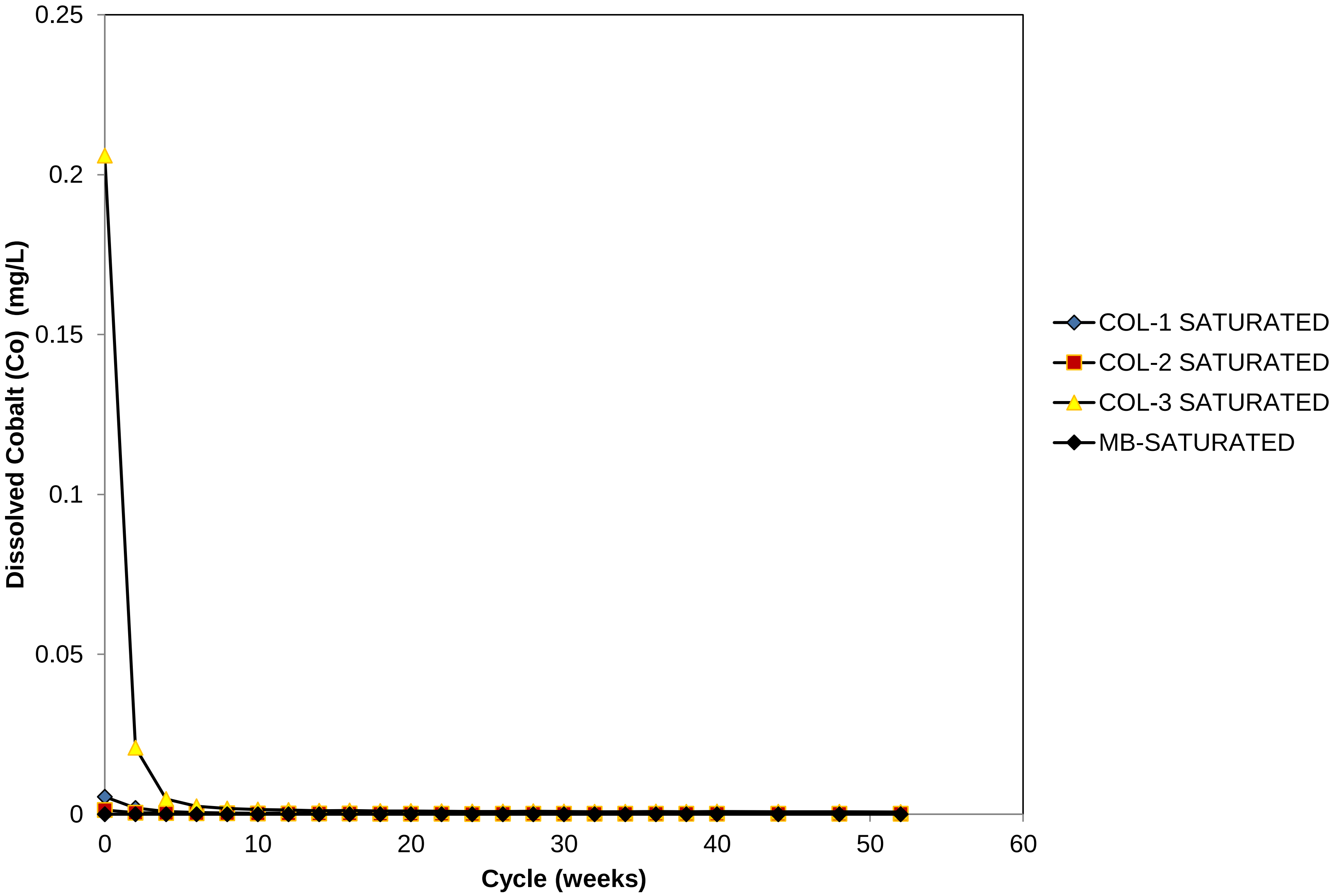

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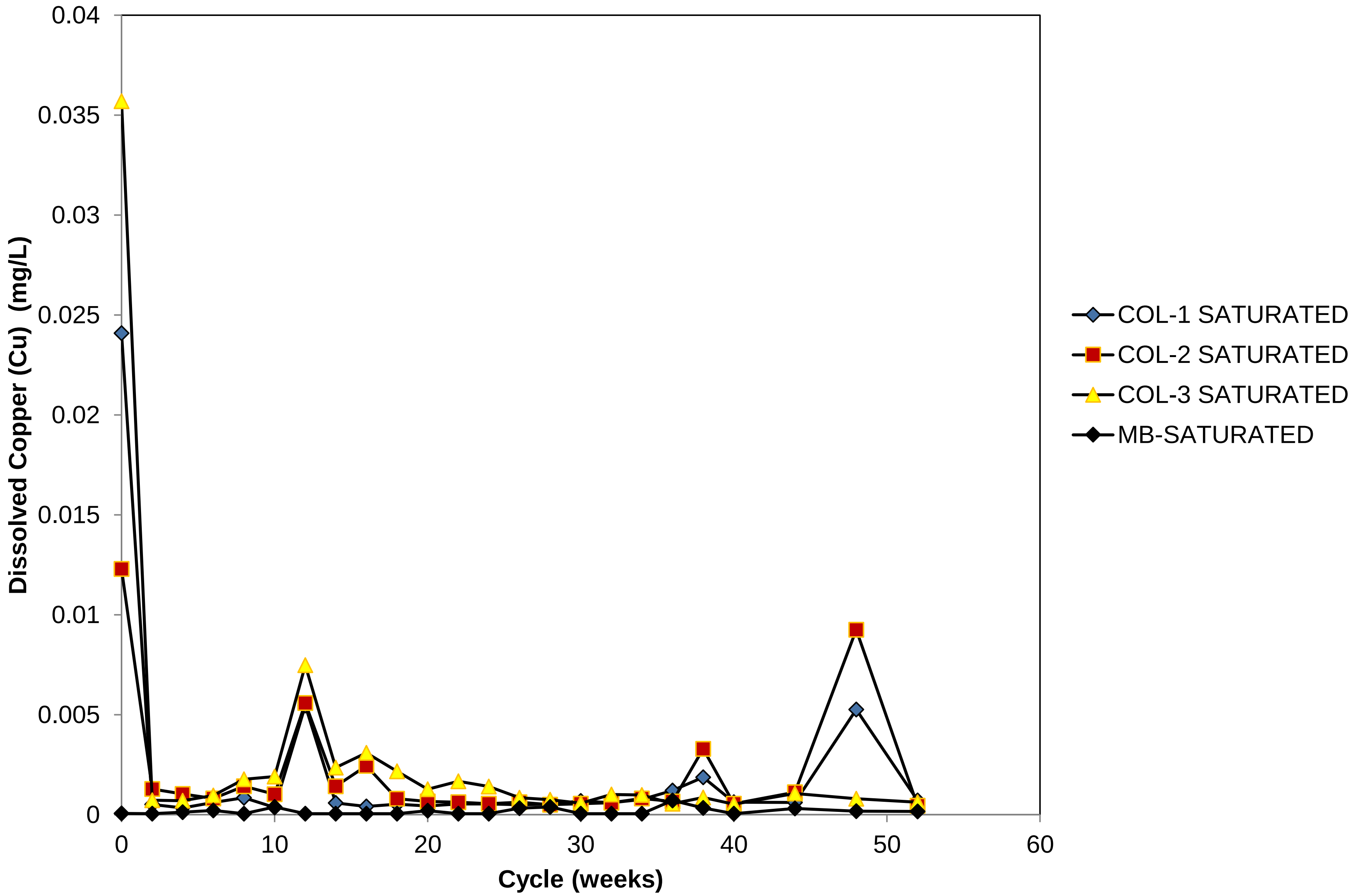

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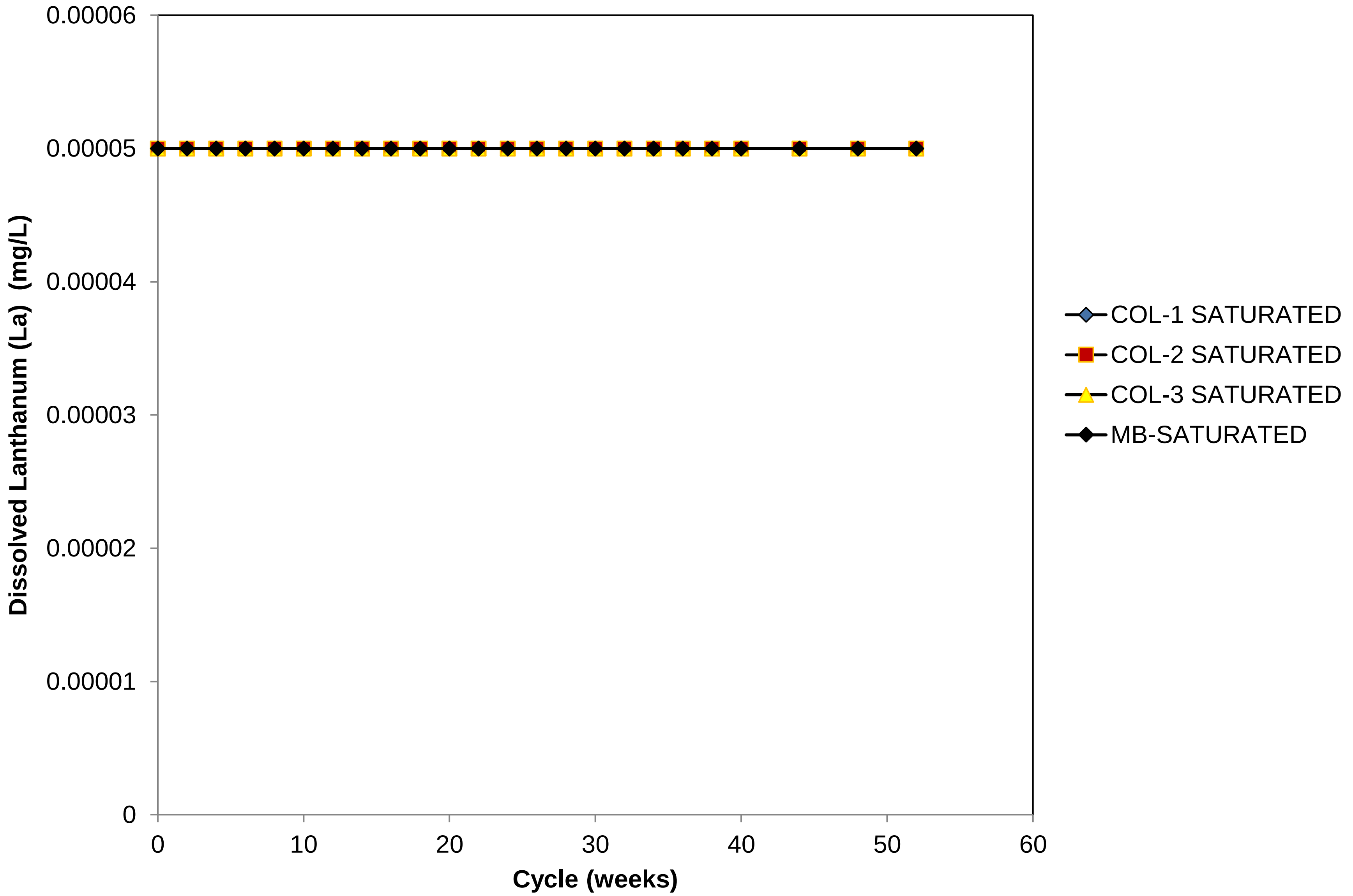

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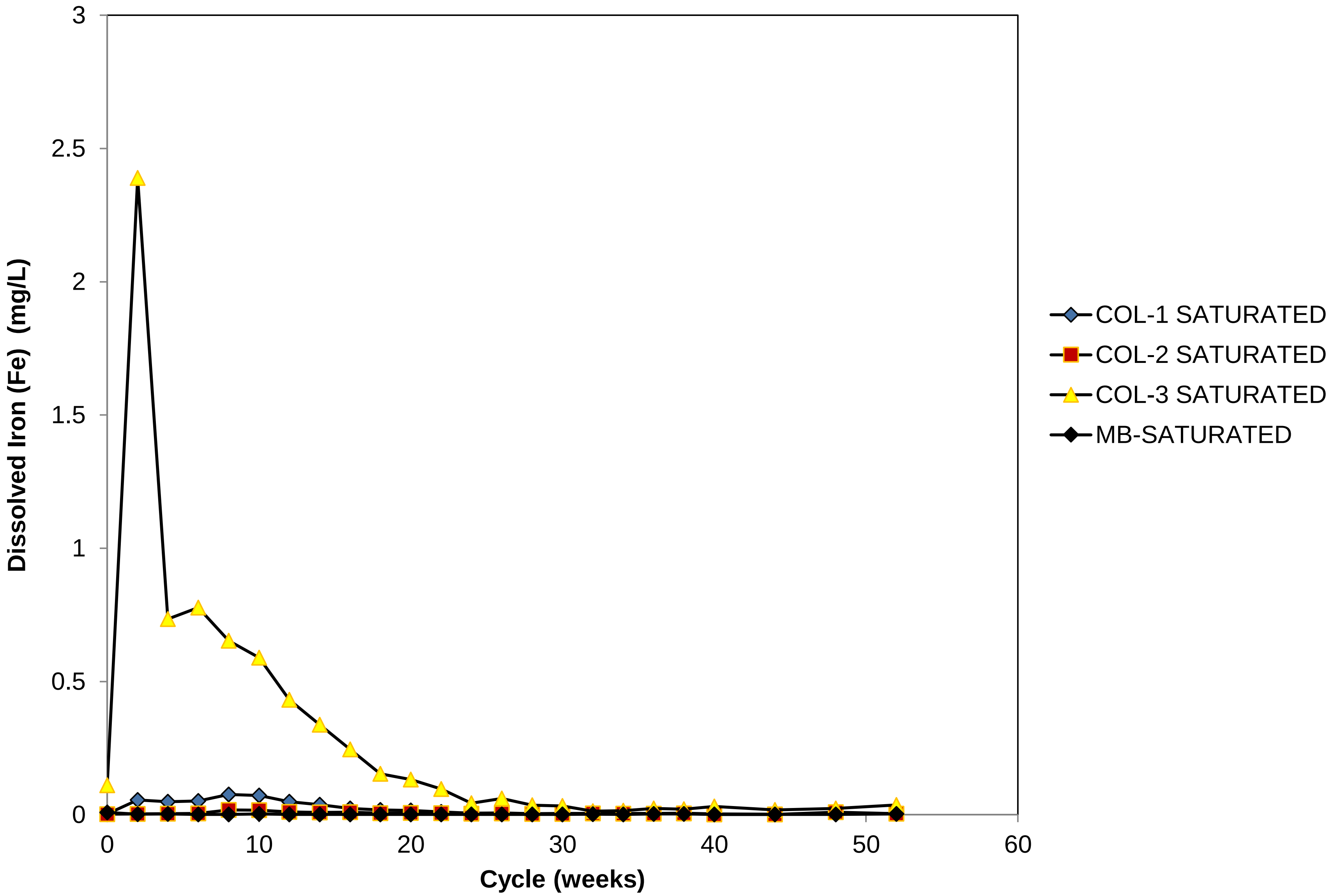

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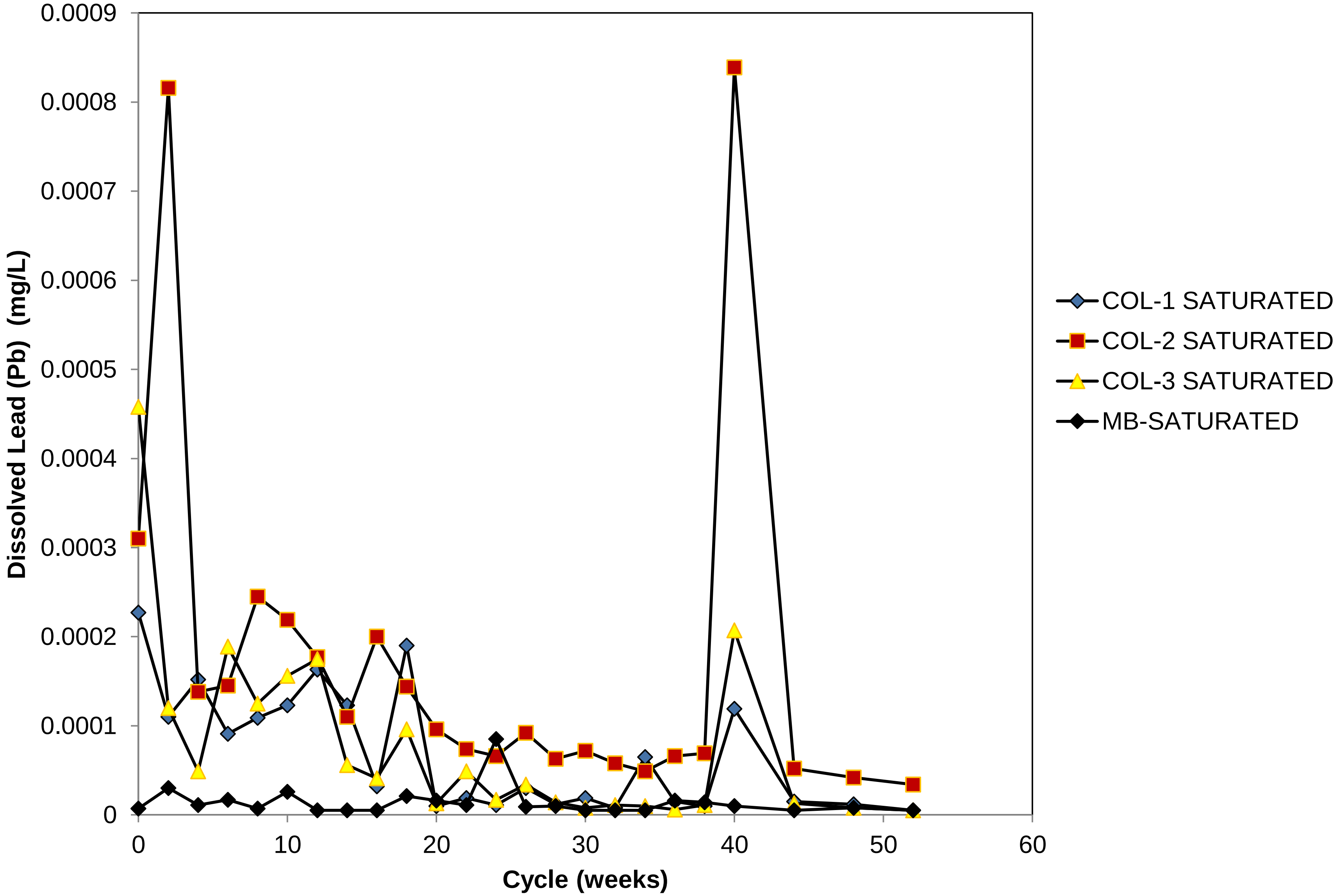

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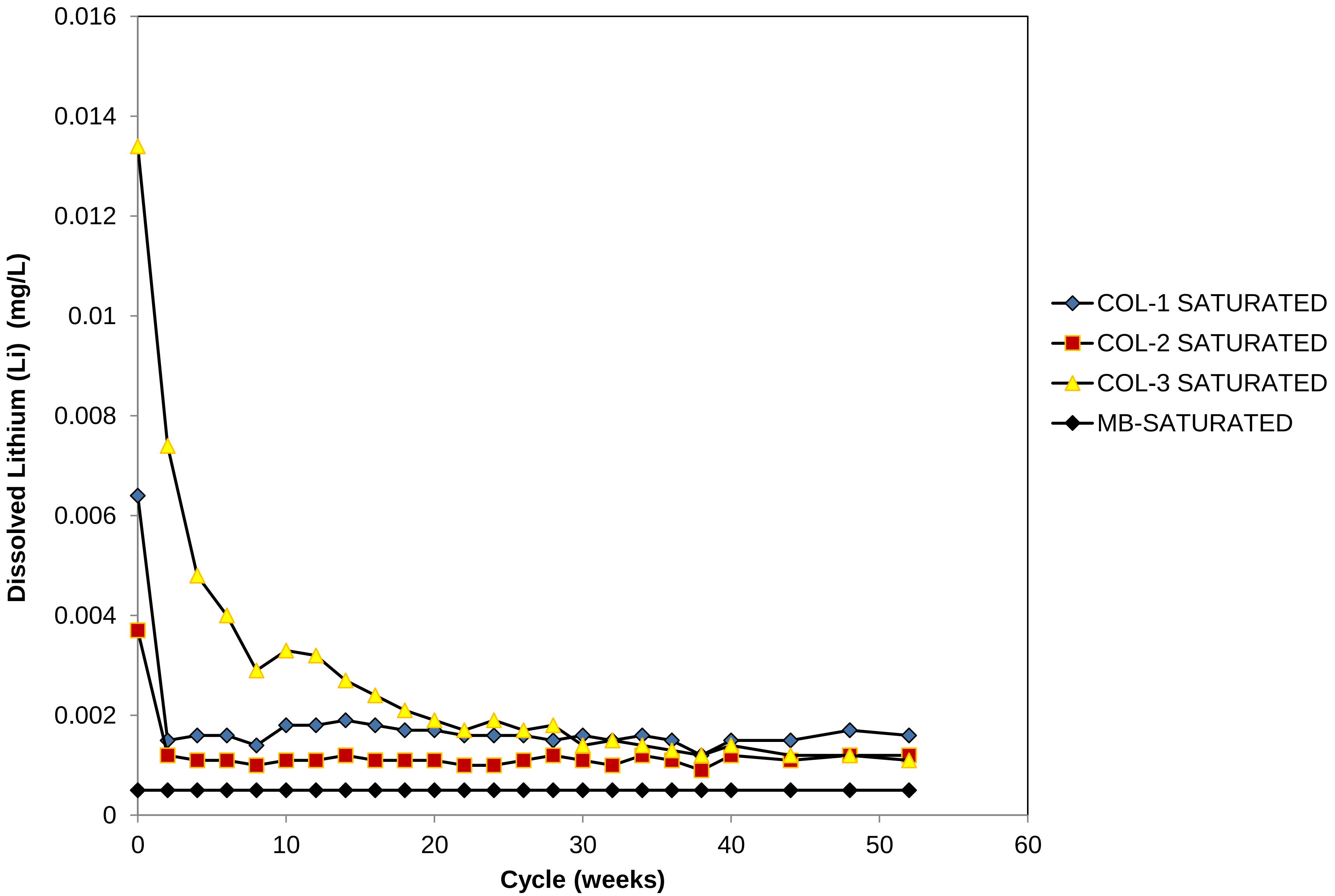

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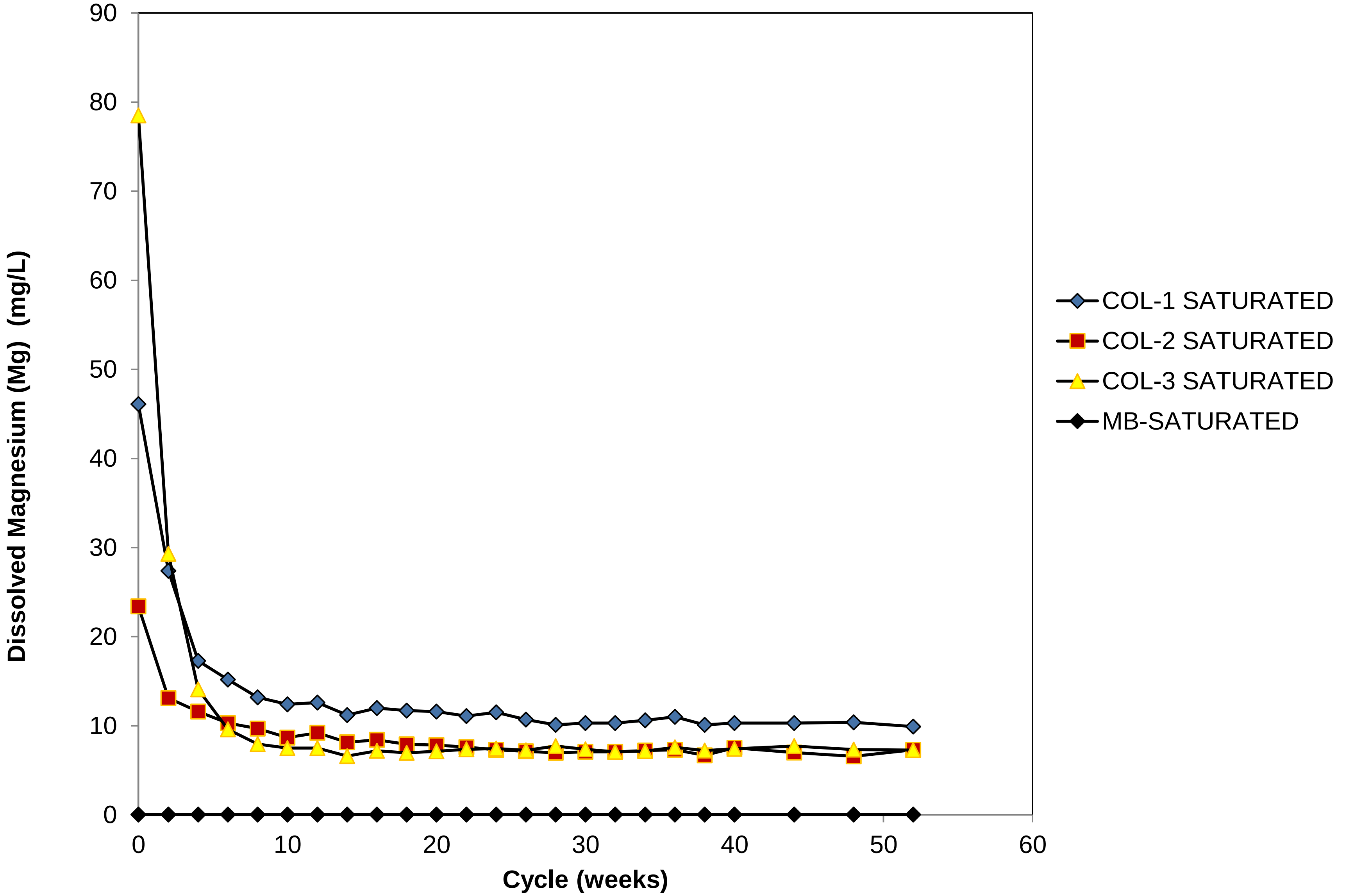

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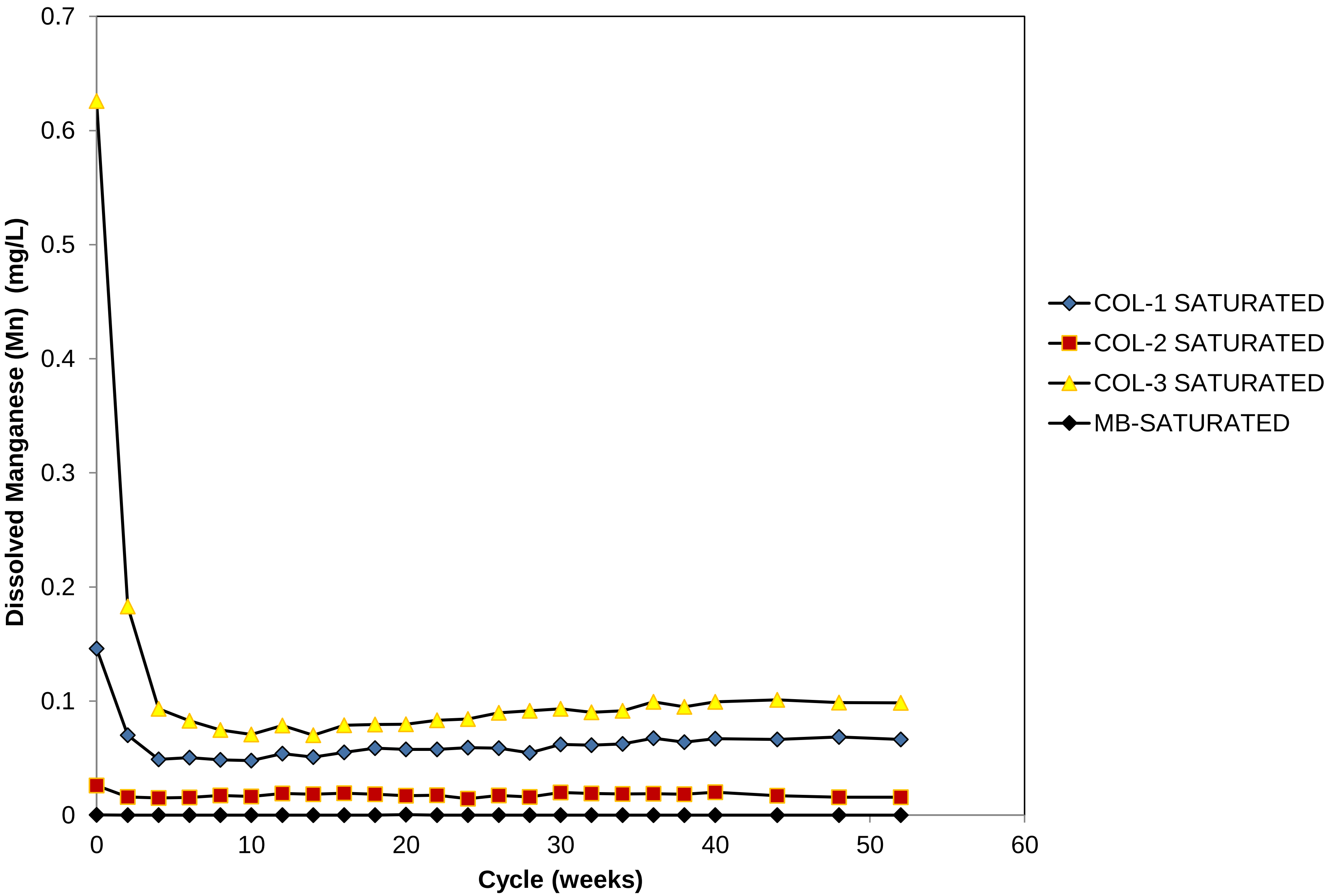

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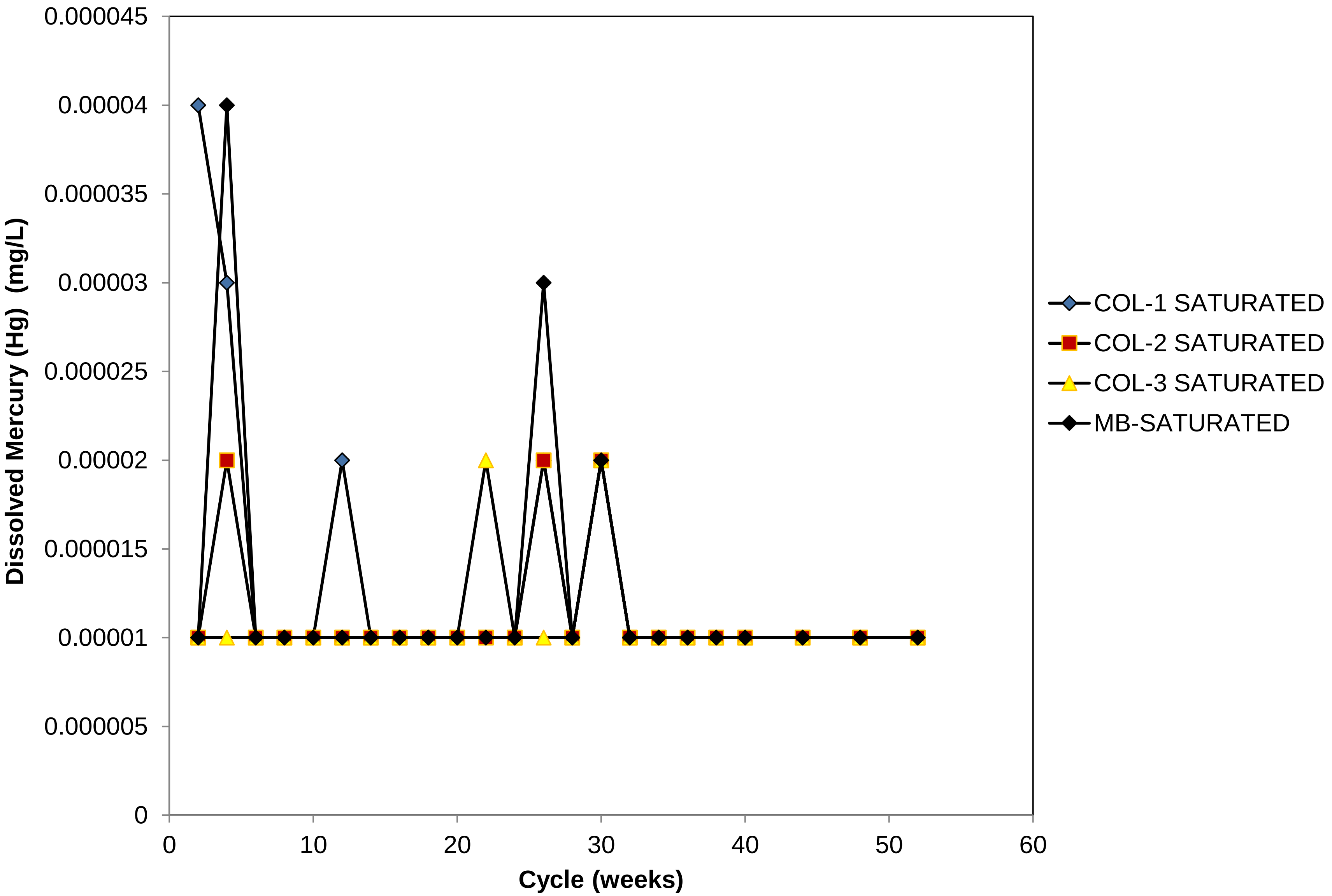

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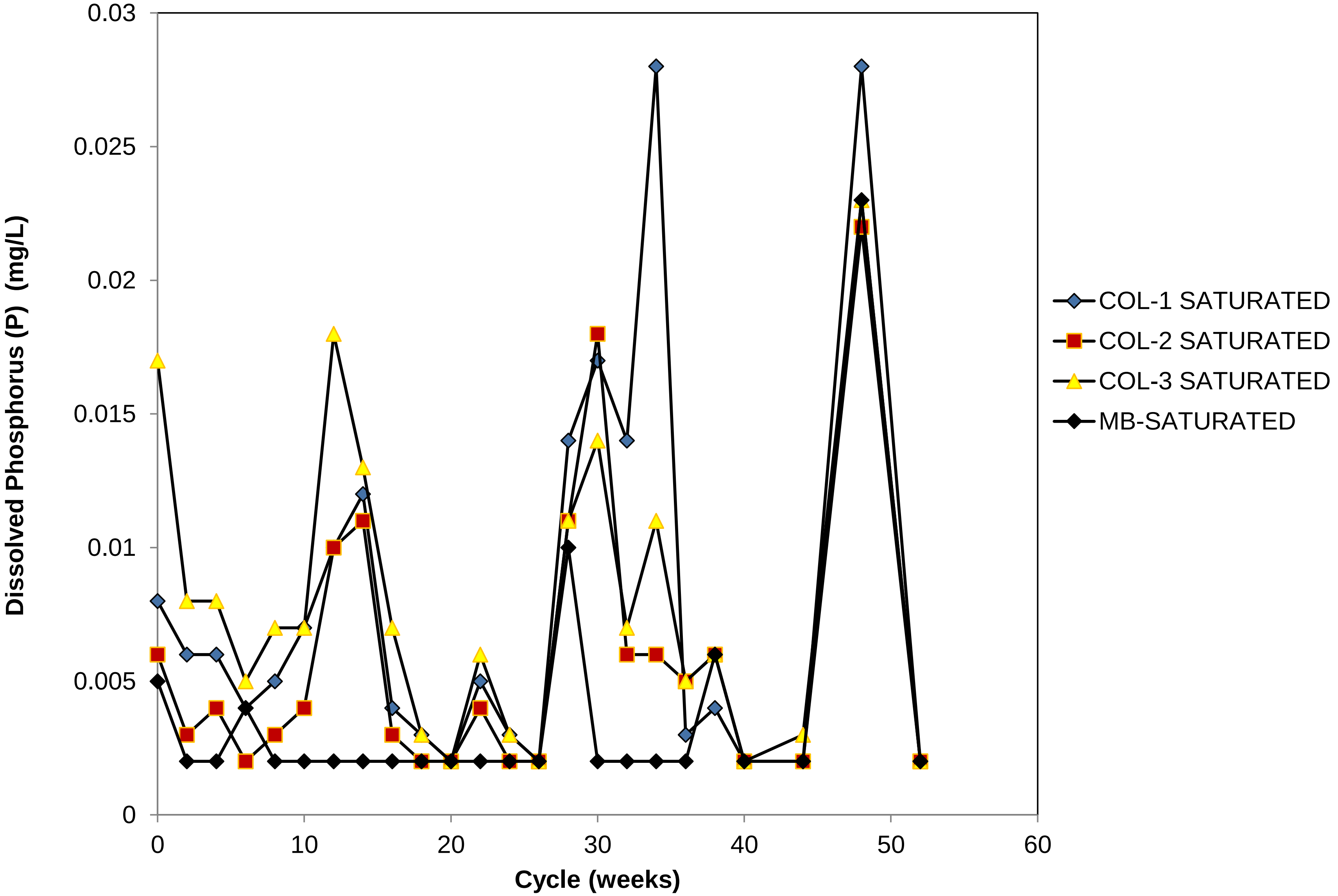

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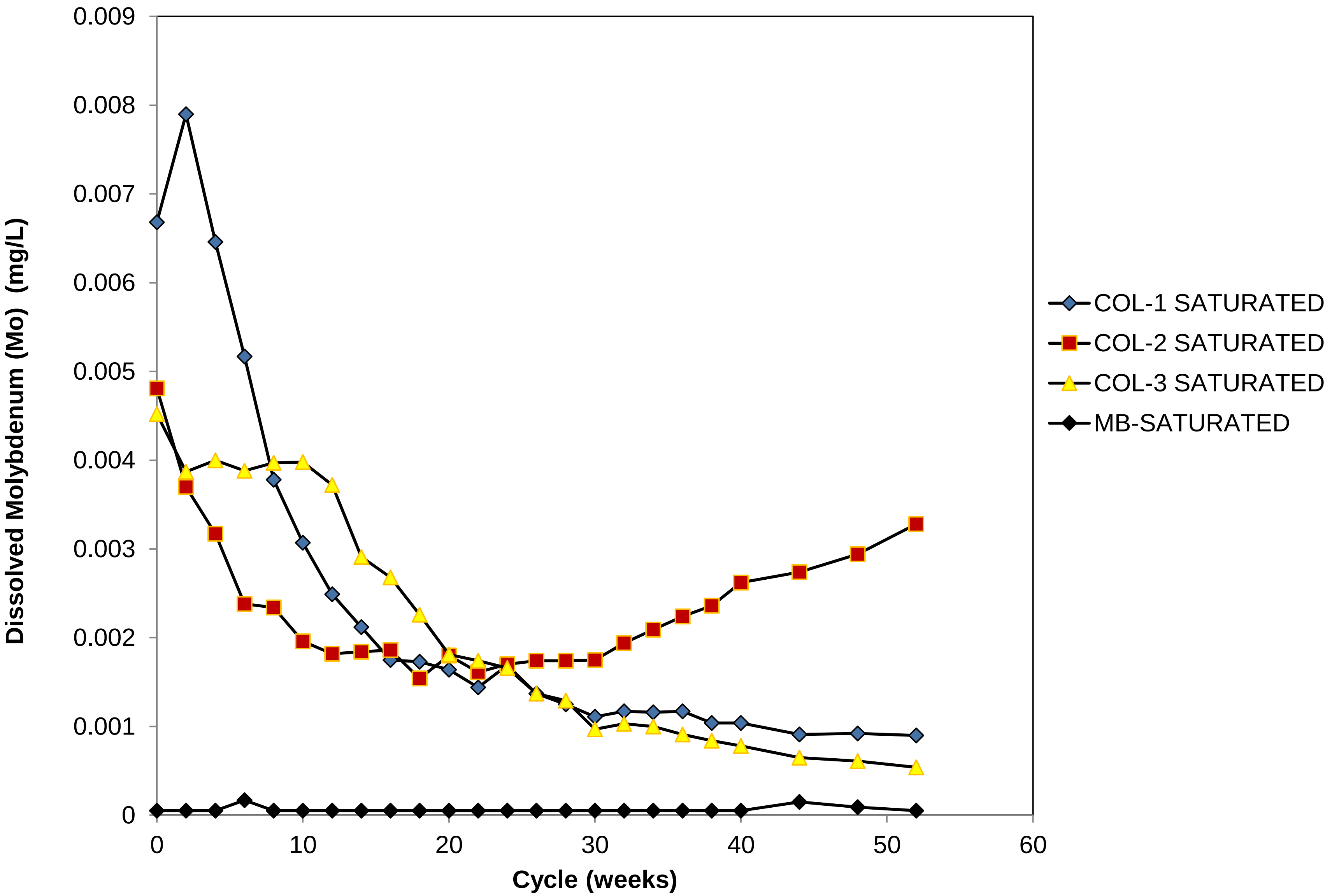

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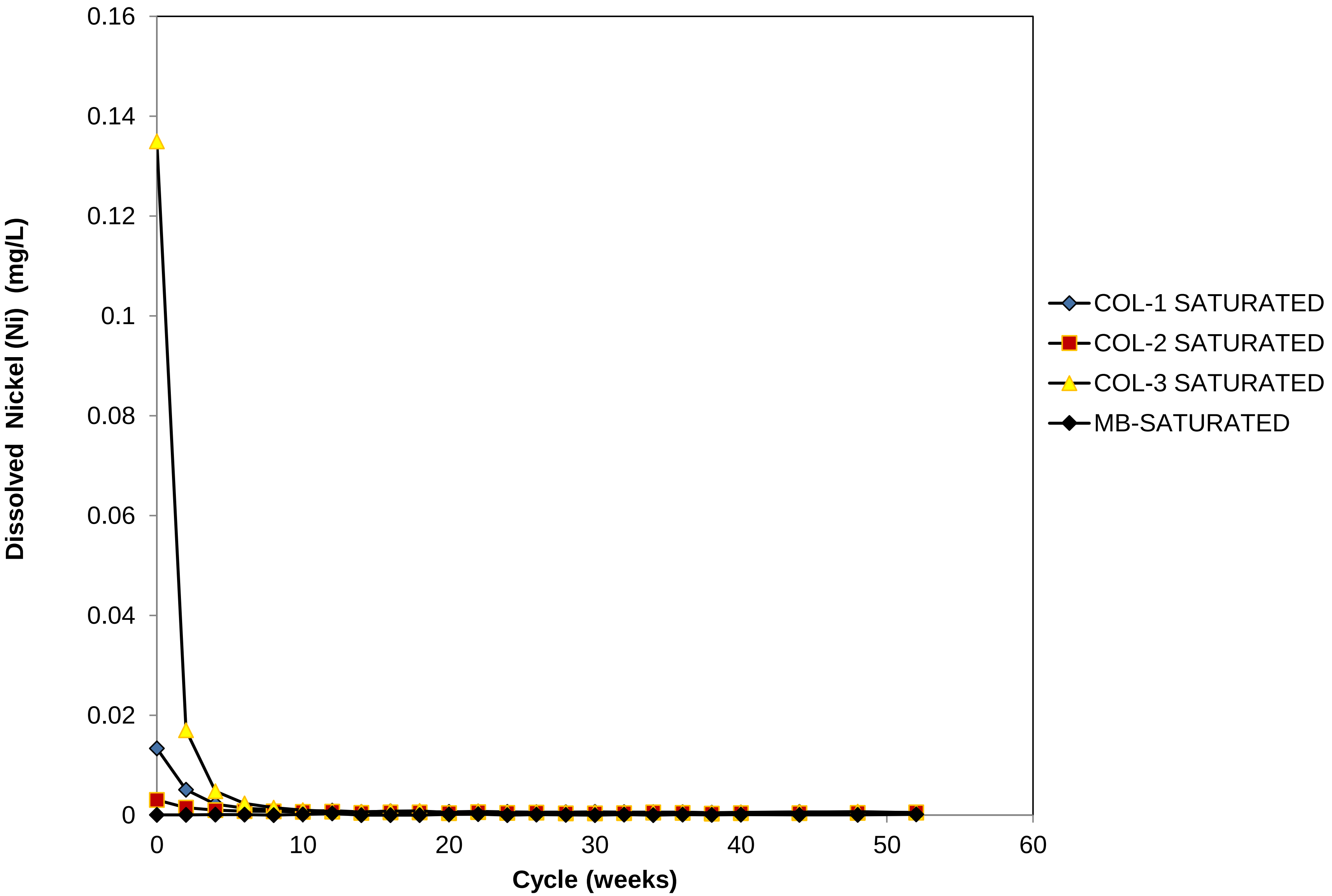

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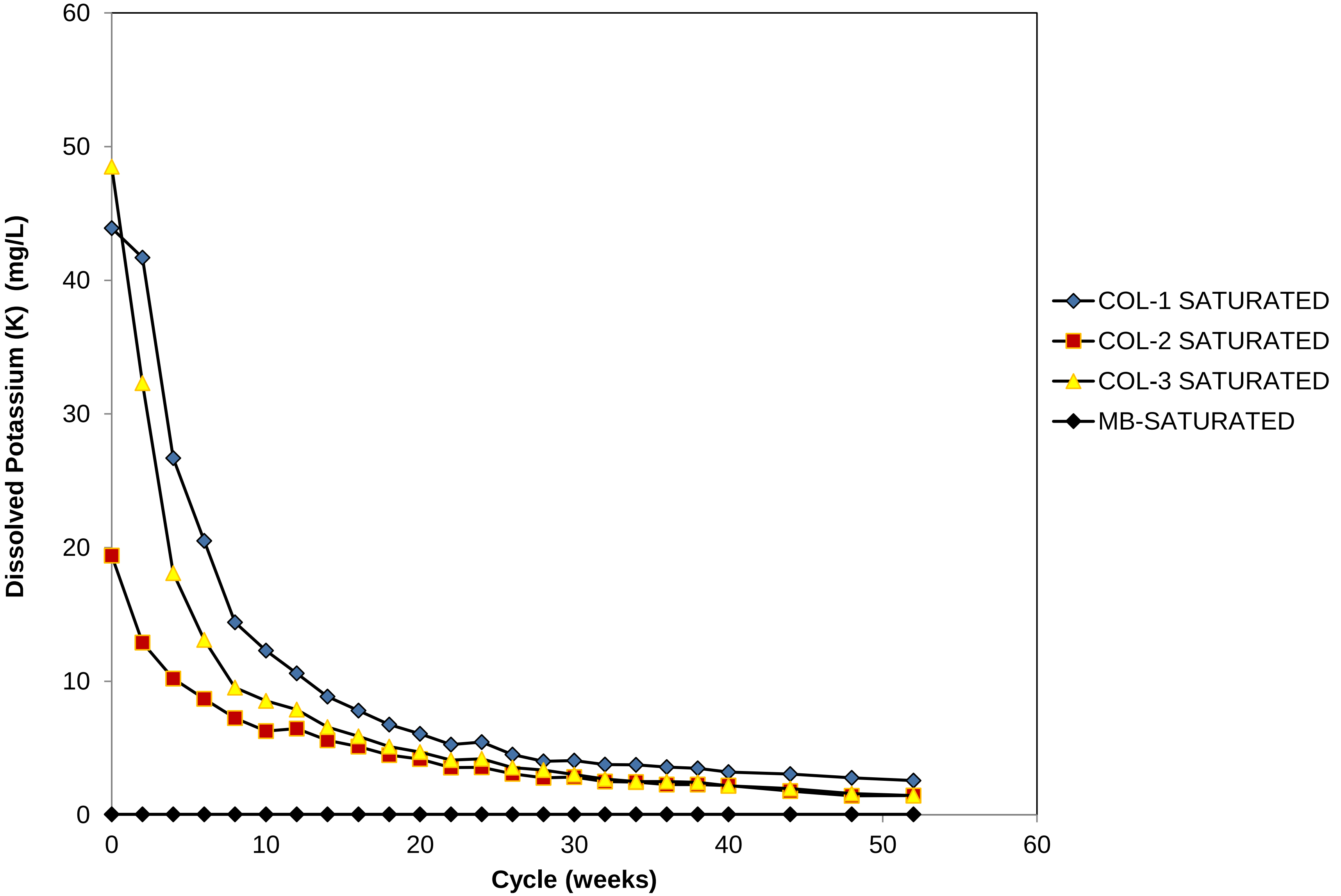

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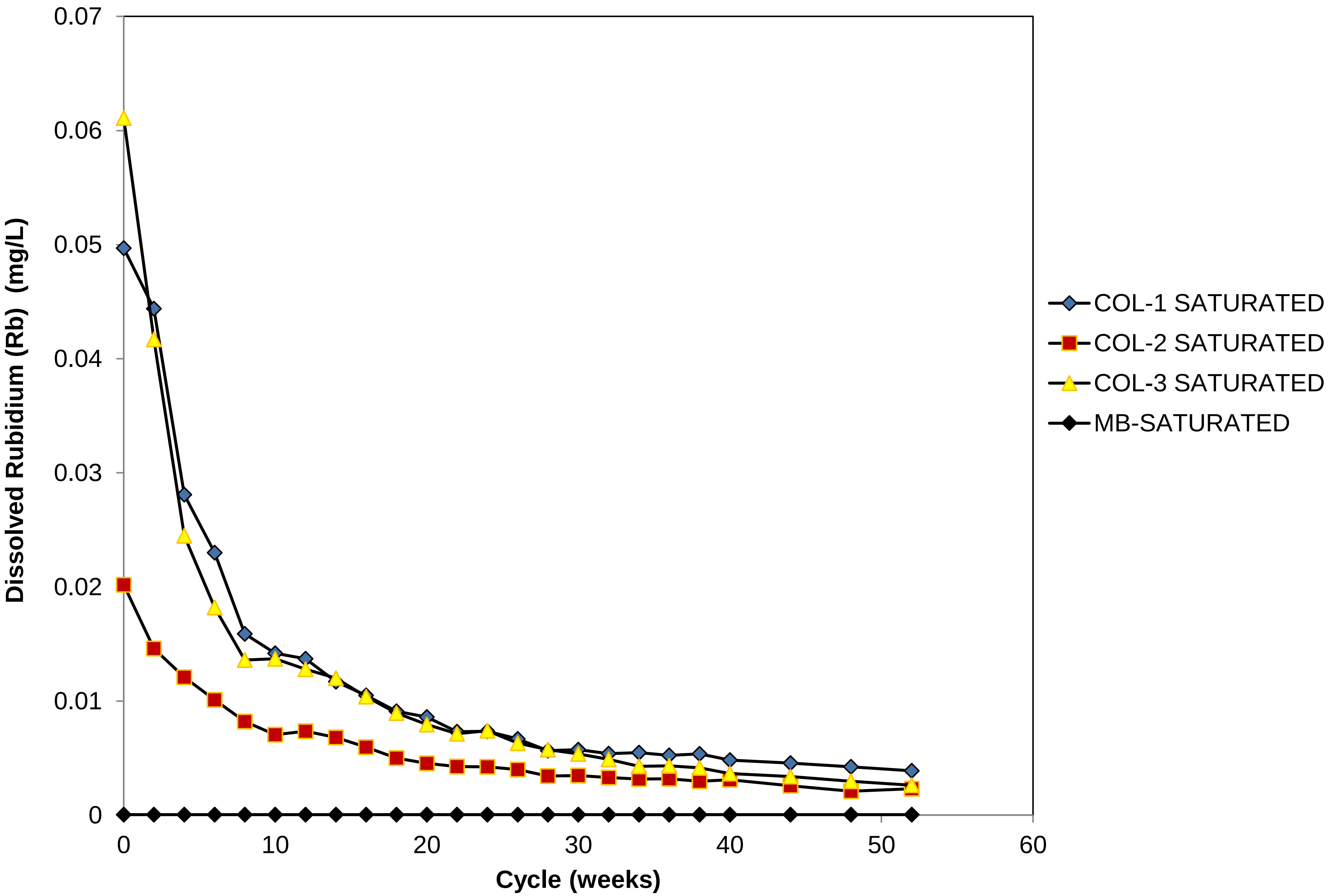

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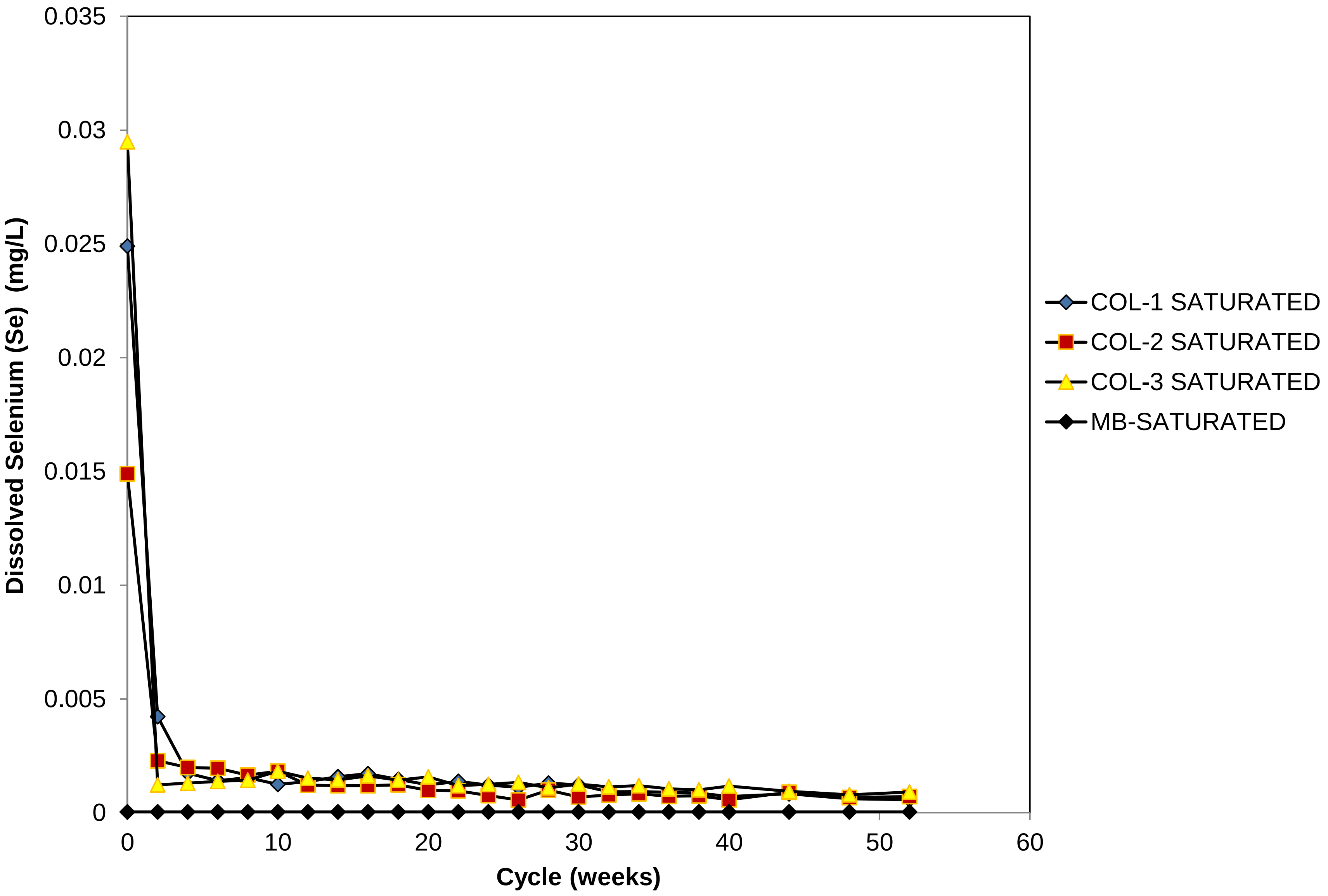

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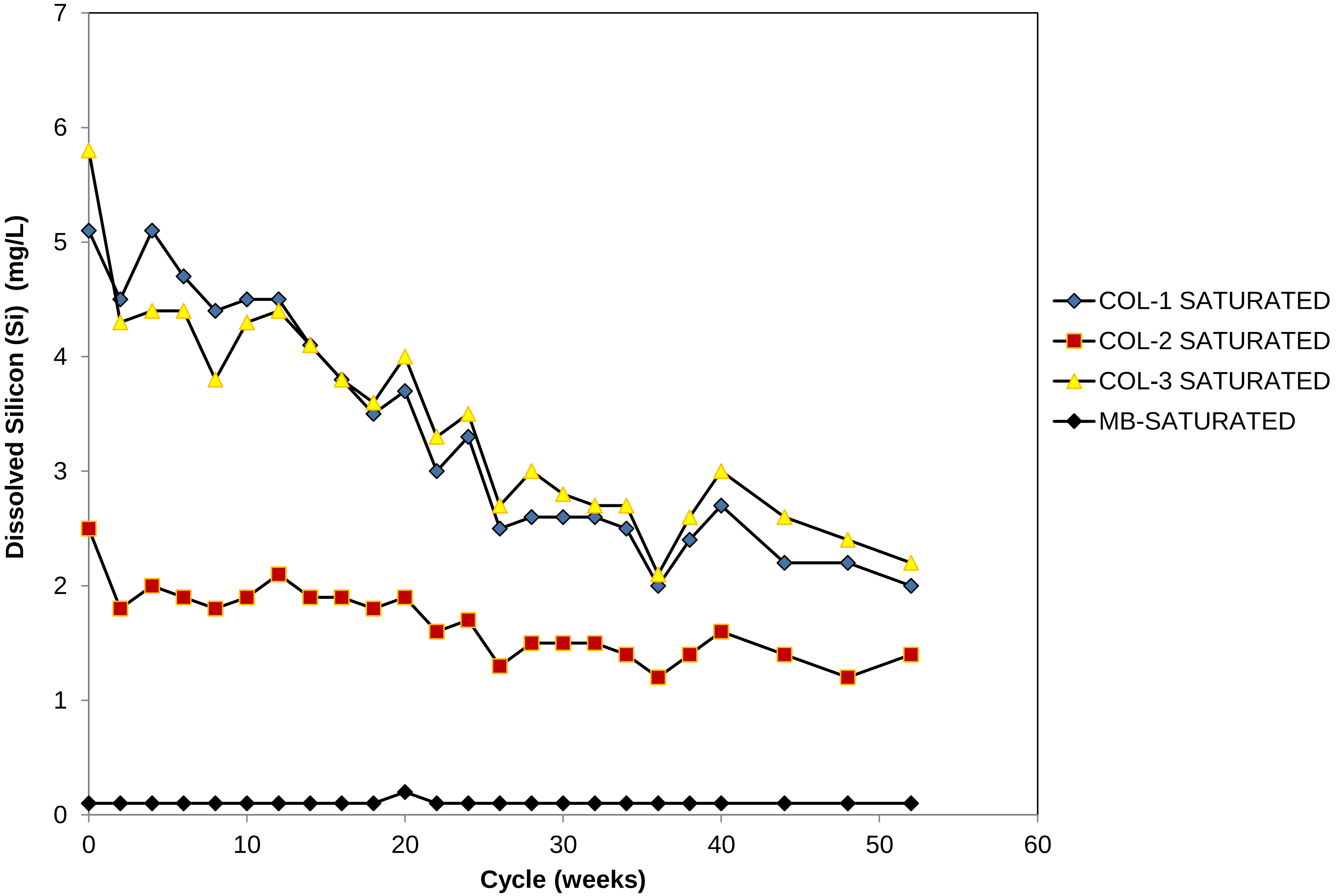
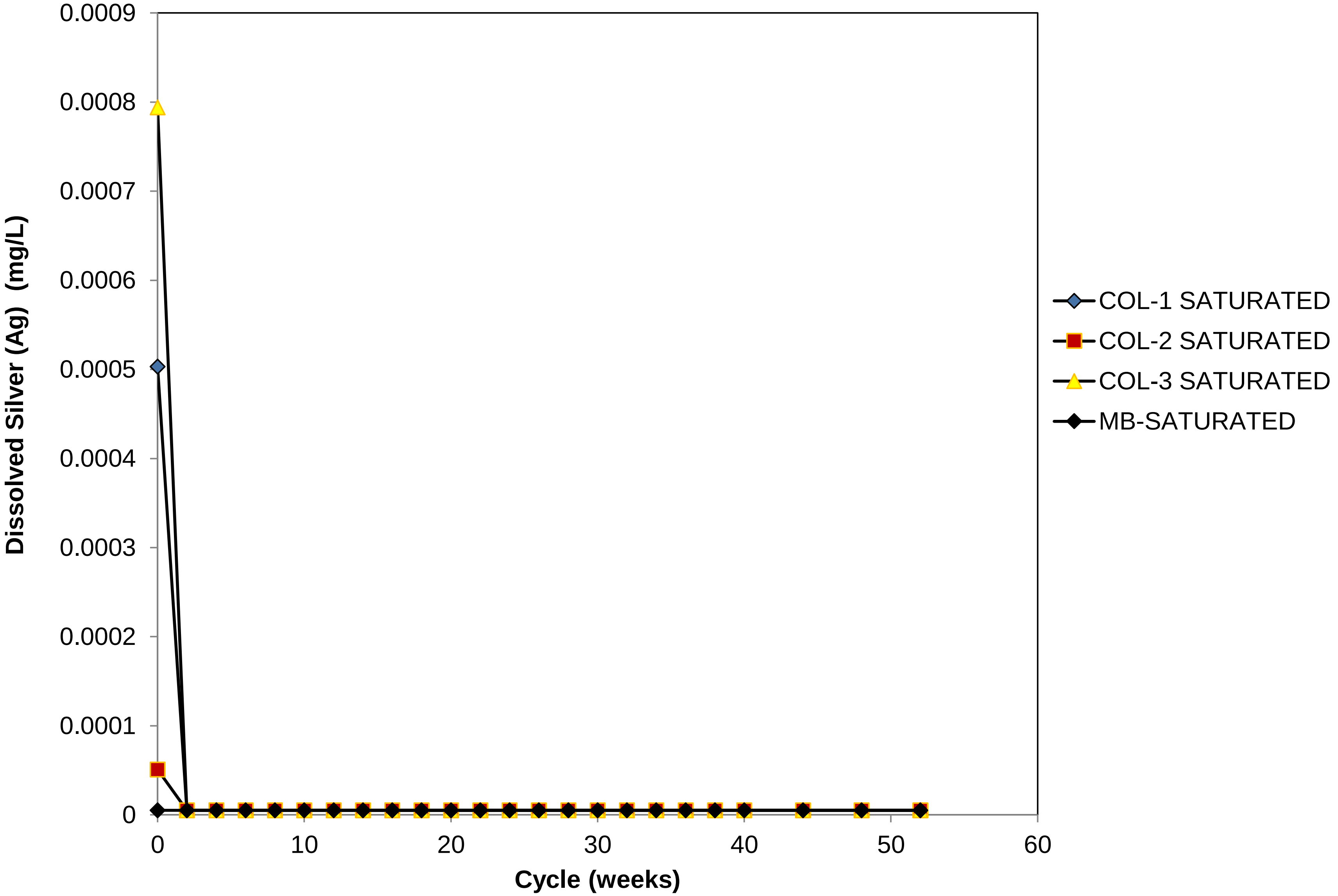


\\VAN-SVR0\Projects\01_SITES\Harper_Creek\1CH014.001_EA_Re-Submission\300_Reporting\310_Update SRK Report\030_Appendices\Appendix C\Working Files\[Appendix C3_Harper_Columns_CONC charts_1CY003.001.xlsx]



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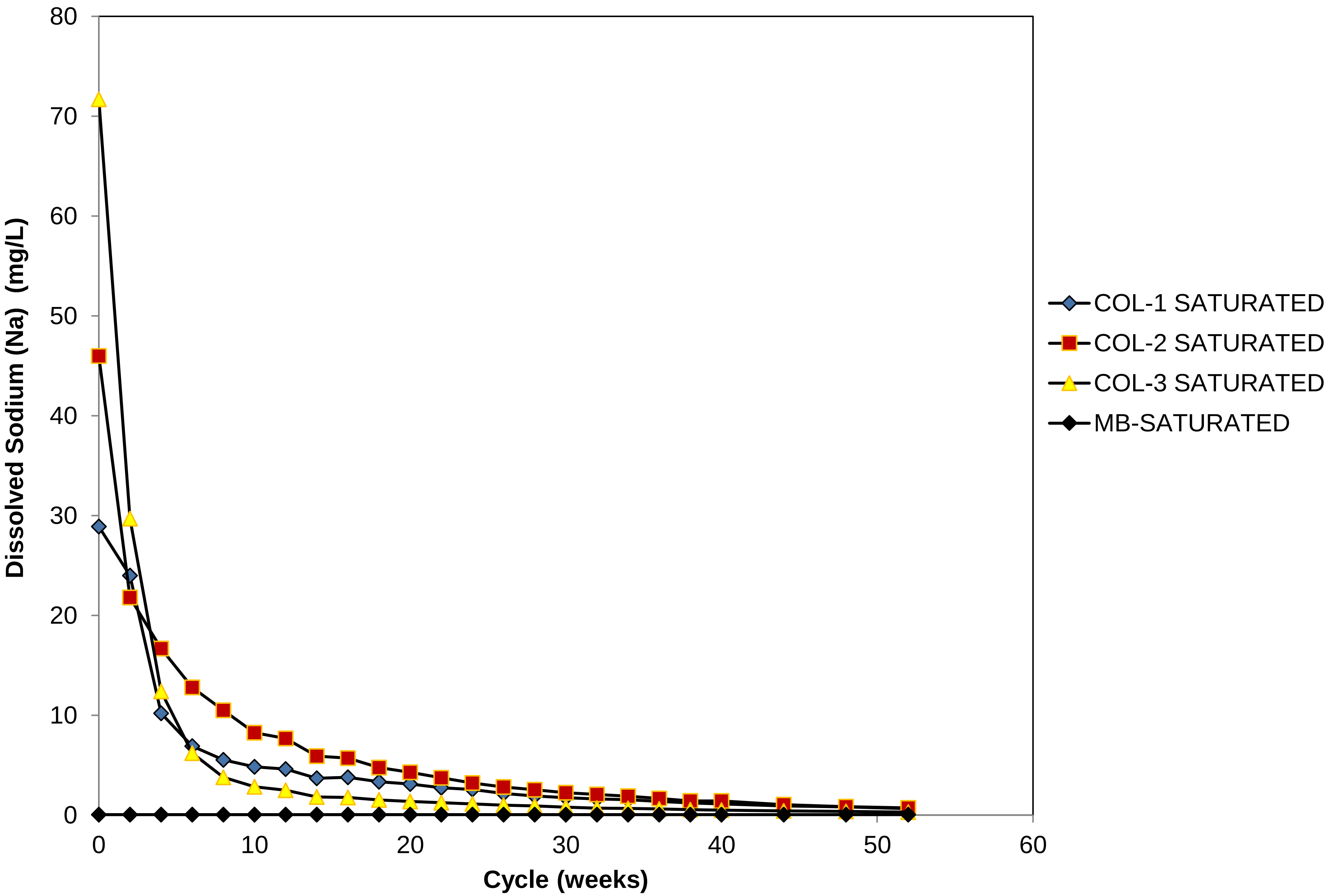

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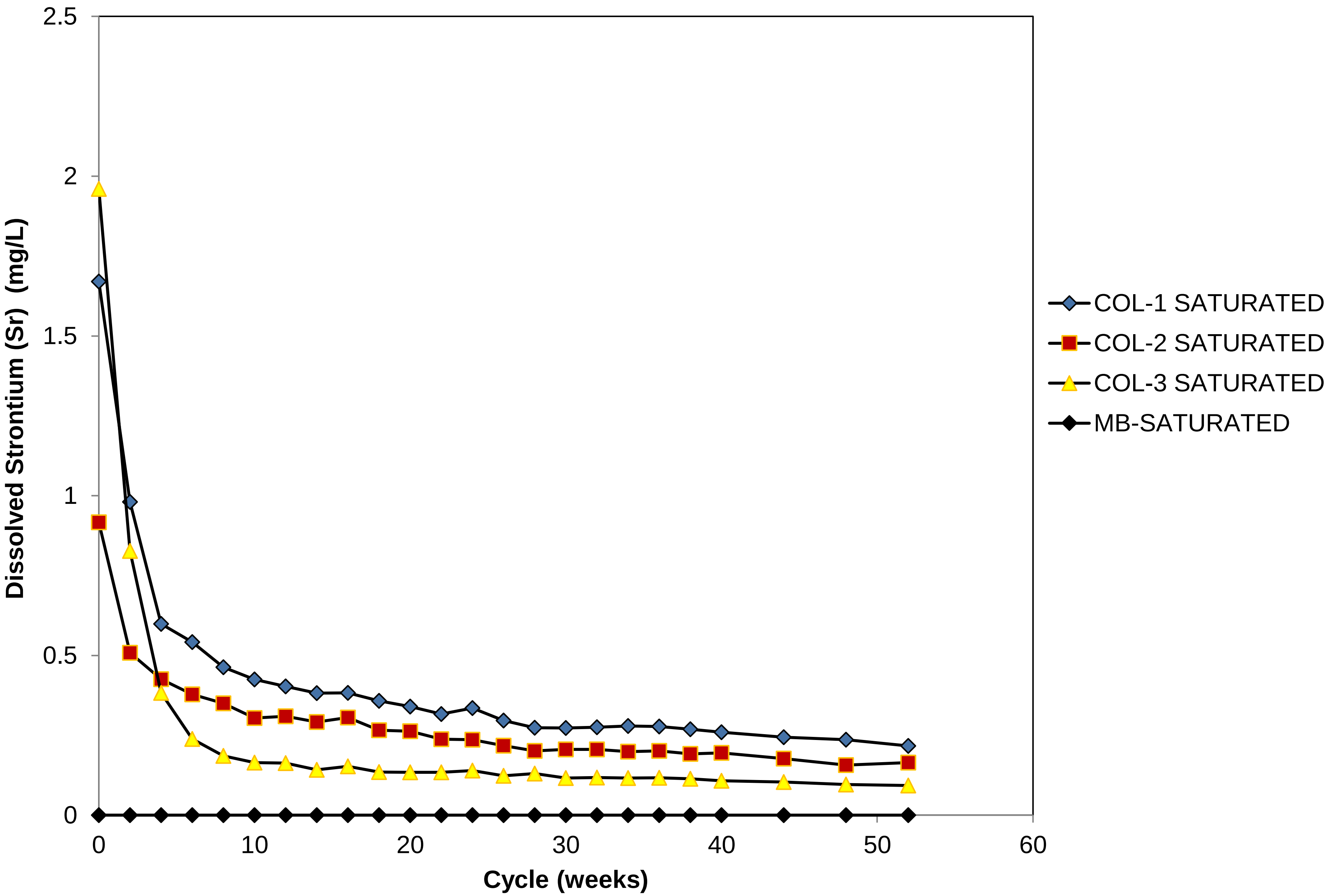
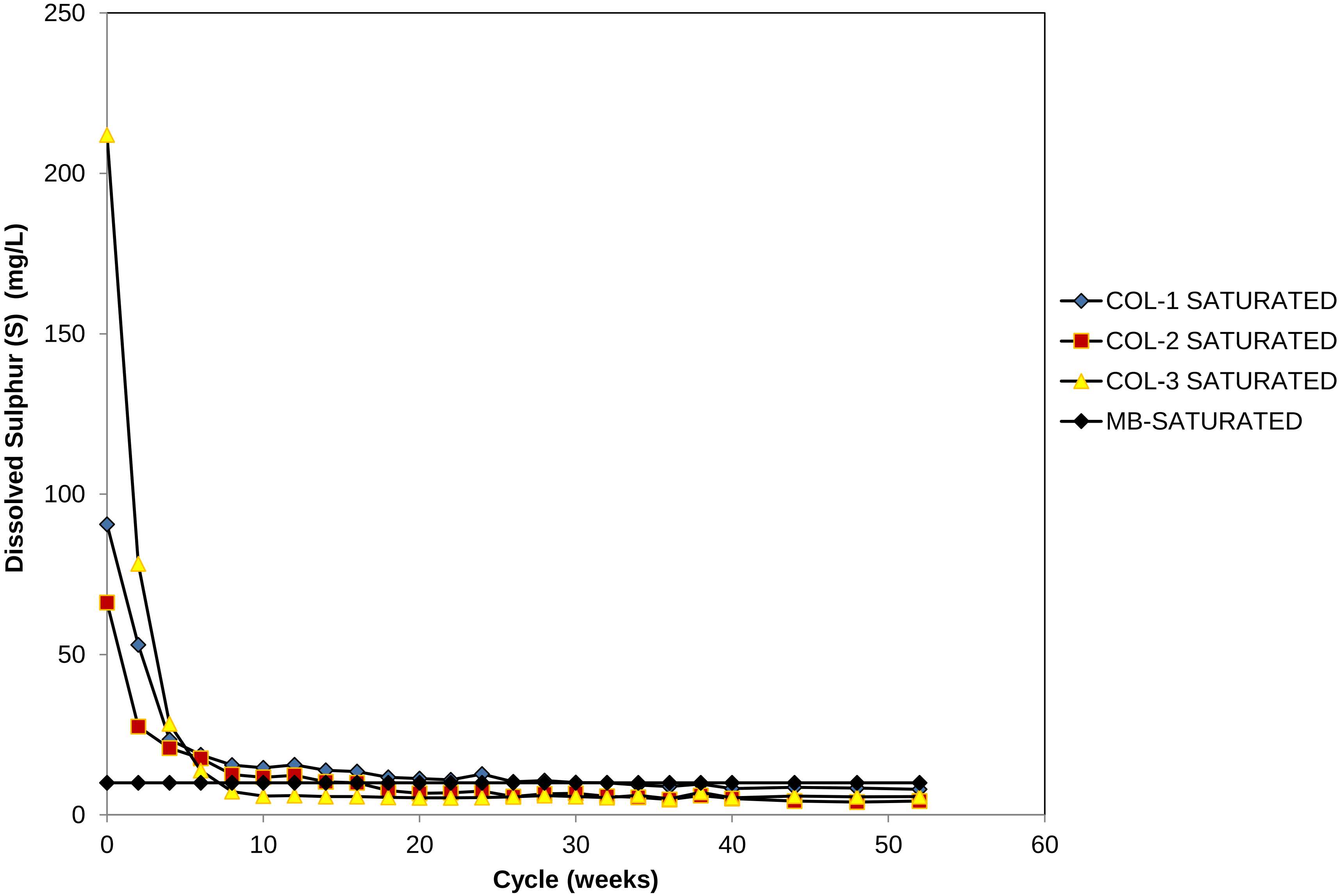


\\VAN-SVR0\Projects\01_SITES\Harper_Creek\1CH014.001_EA_Re-Submission\300_Reporting\310_Update SRK Report\030_Appendices\Appendix C\Working Files\[Appendix C3_Harper_Columns_CONC charts_1CY003.001.xlsx]



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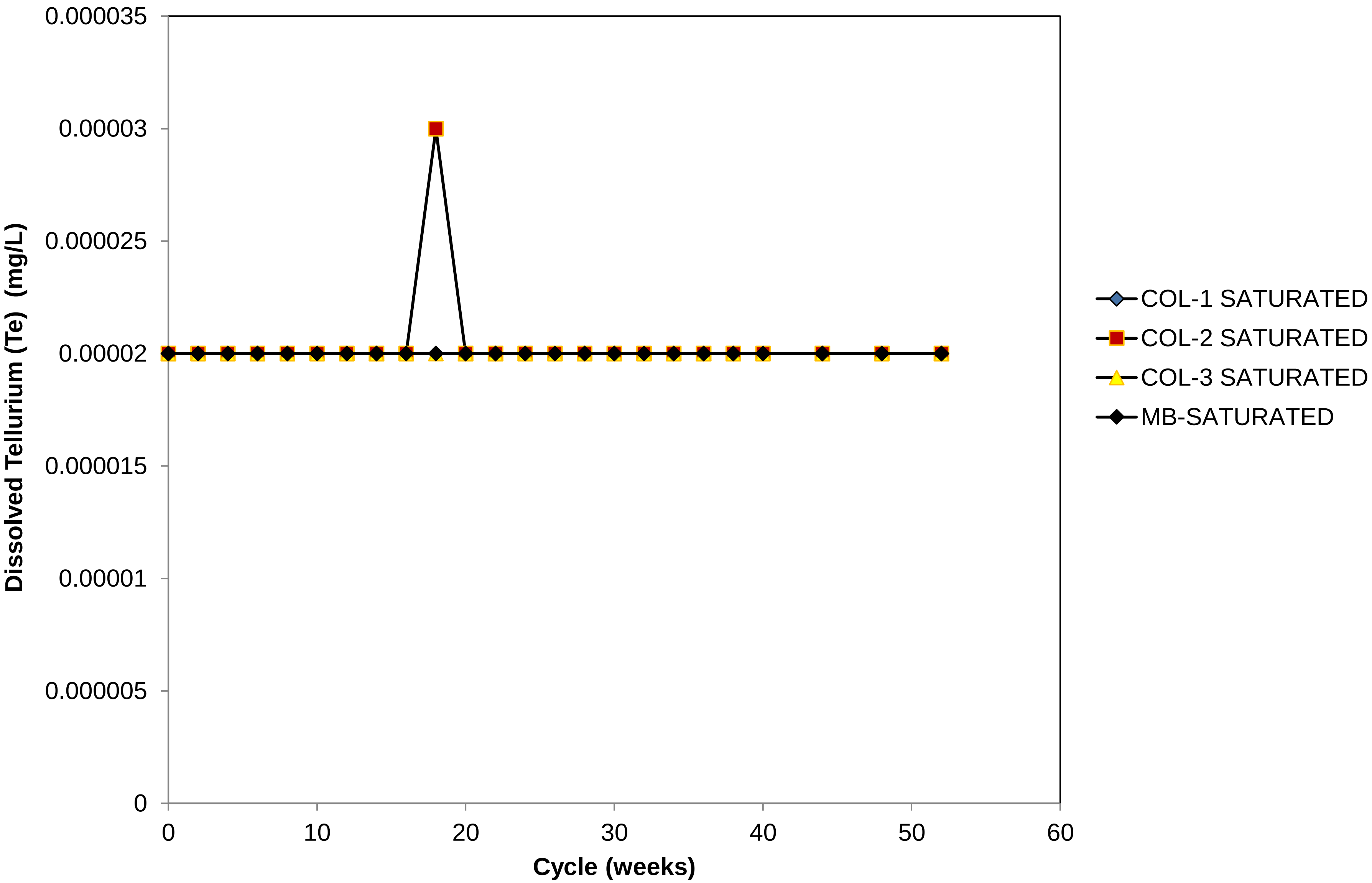

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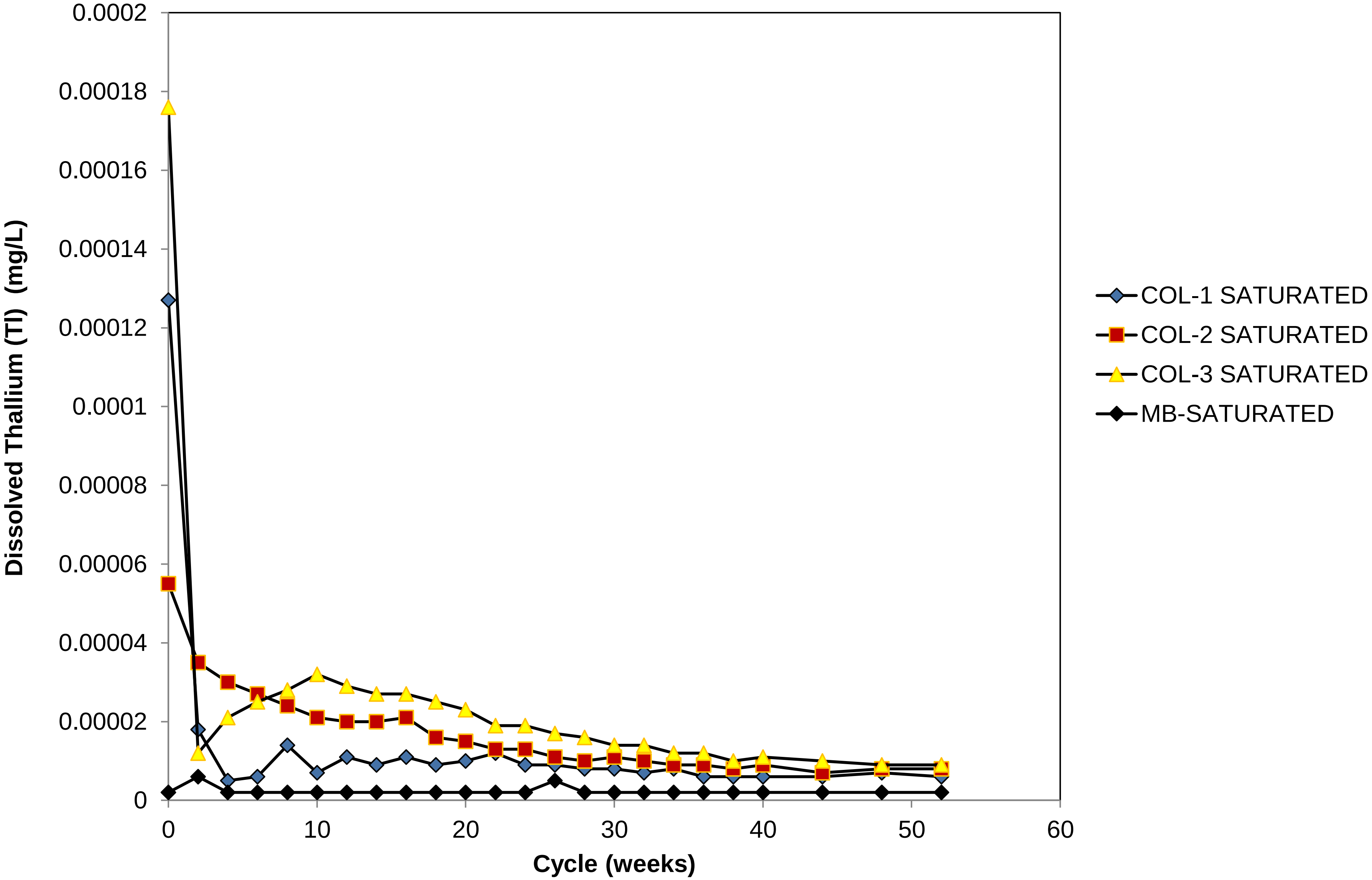

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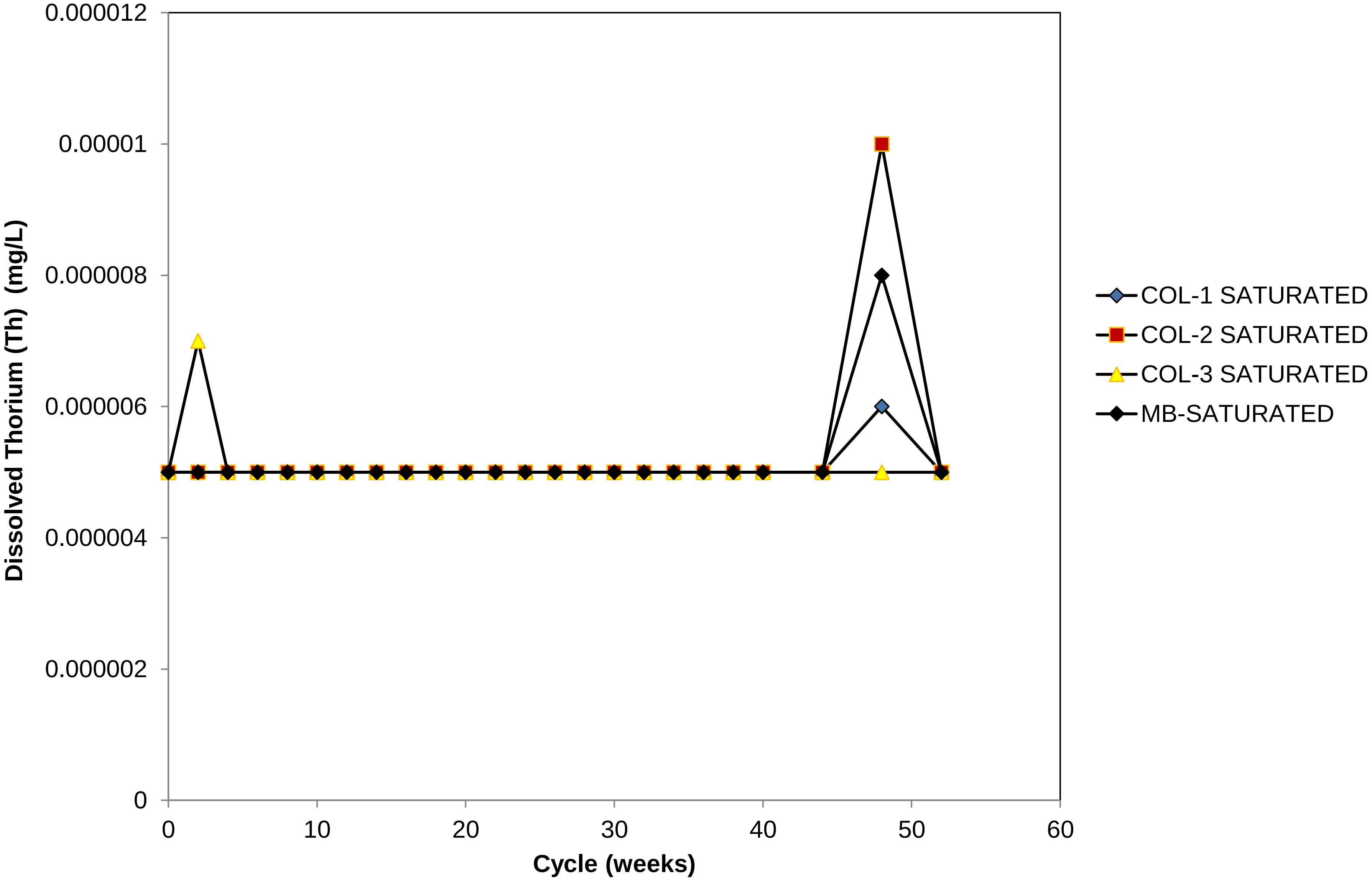
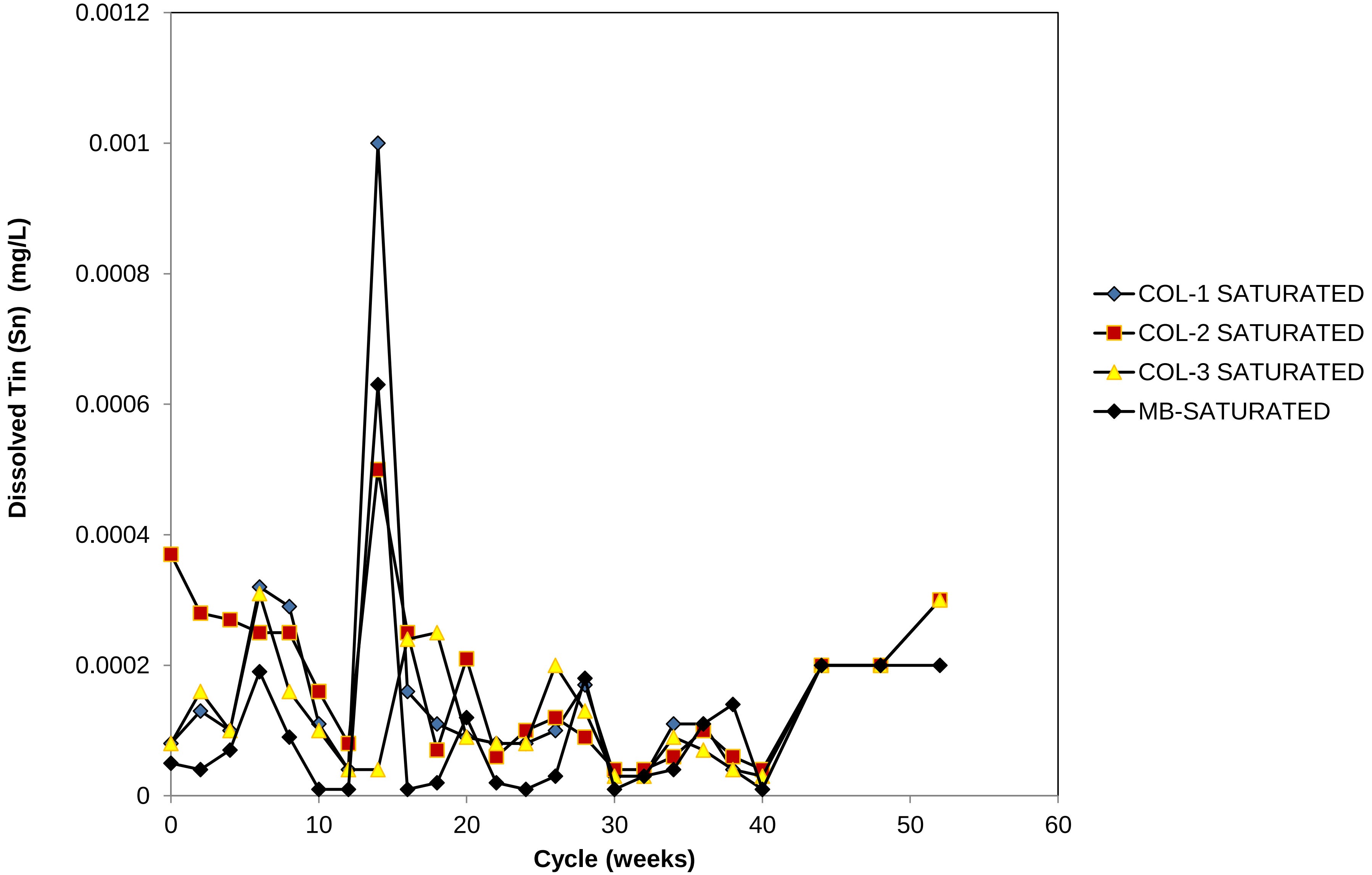


\\VAN-SVR0\Projects\01_SITES\Harper_Creek\1CH014.001_EA_Re-Submission\300_Reporting\310_Update SRK Report\030_Appendices\Appendix C\Working Files\[Appendix C3_Harper_Columns_CONC charts_1CY003.001.xlsx]



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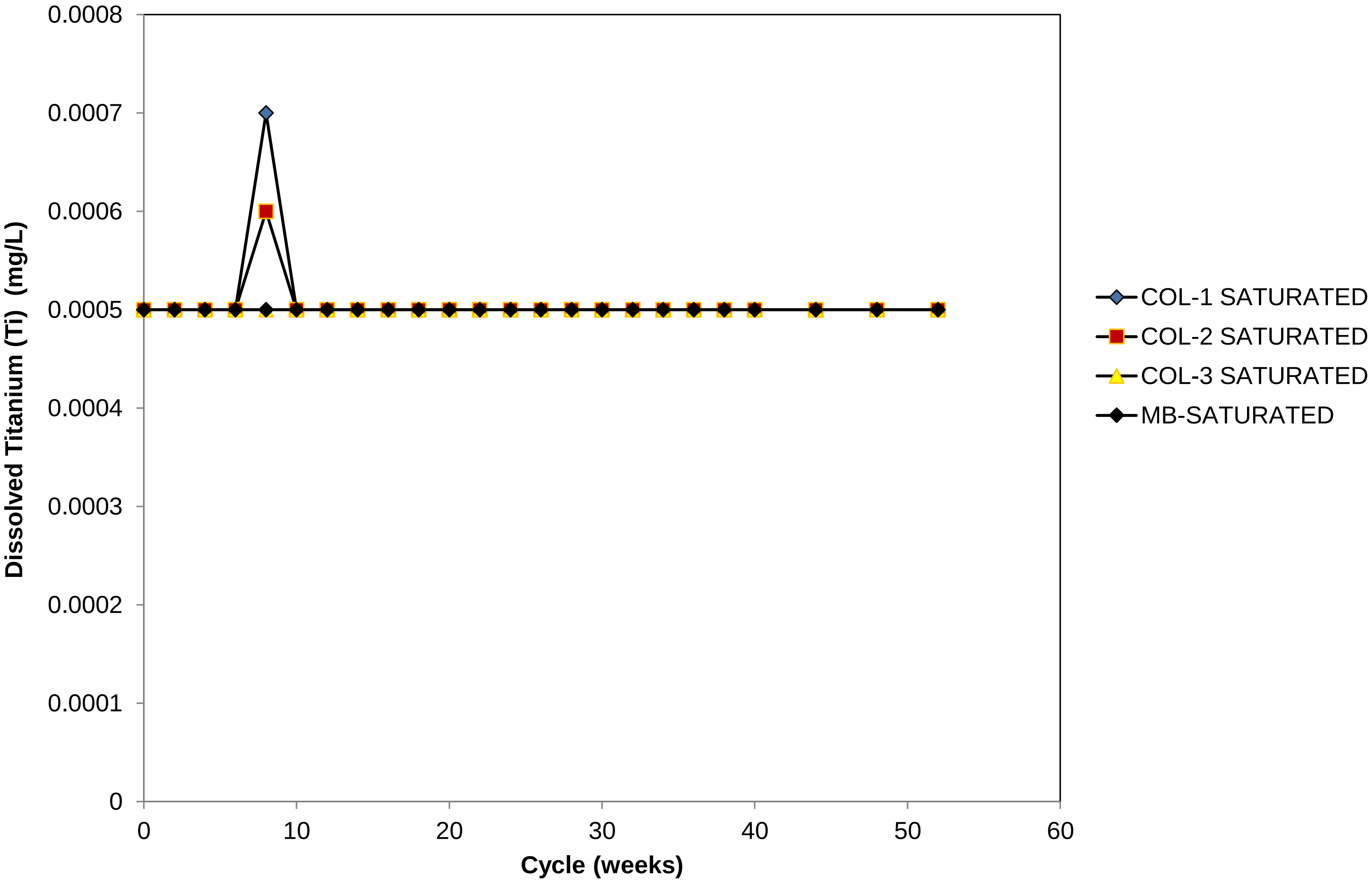

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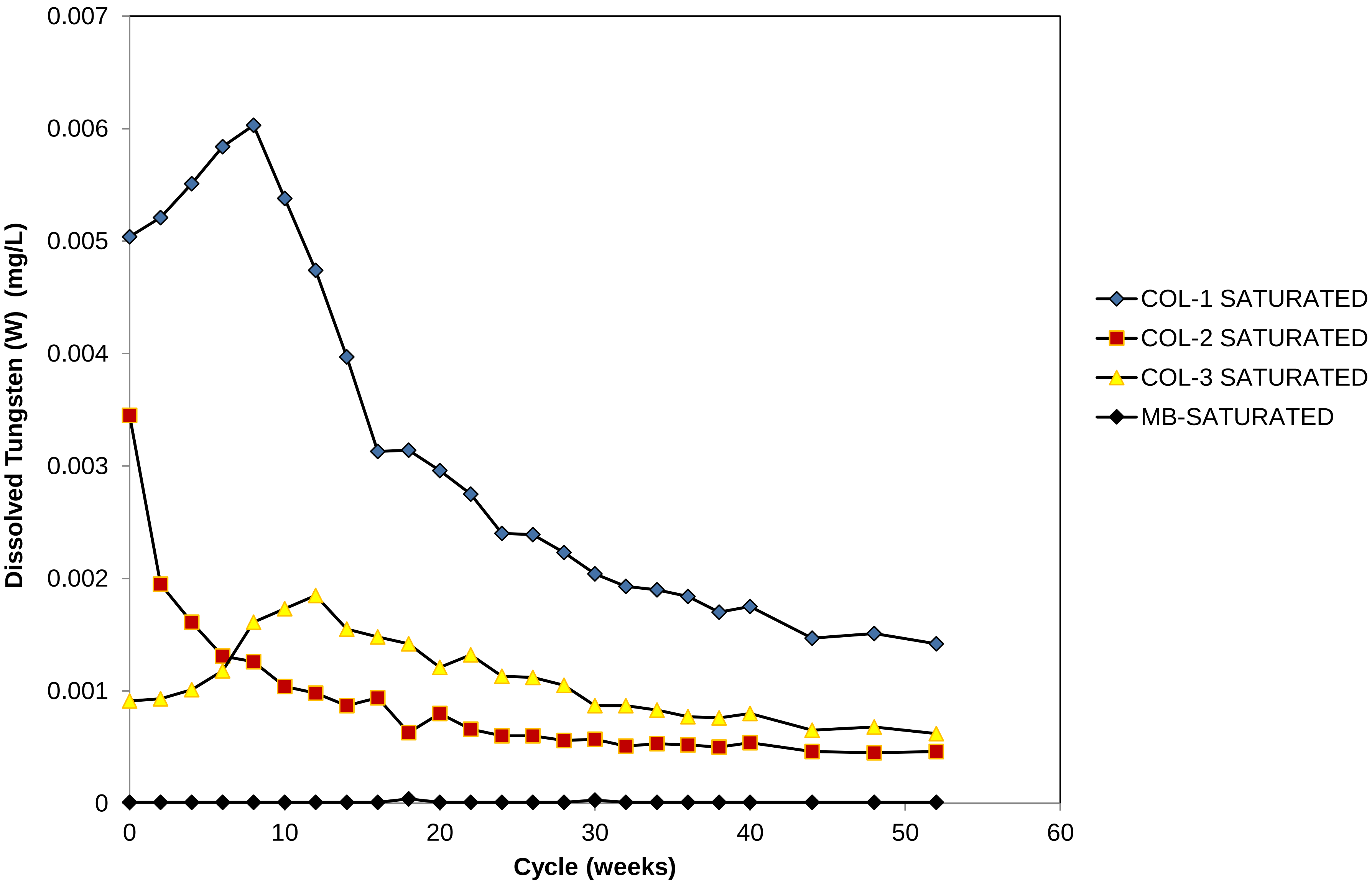

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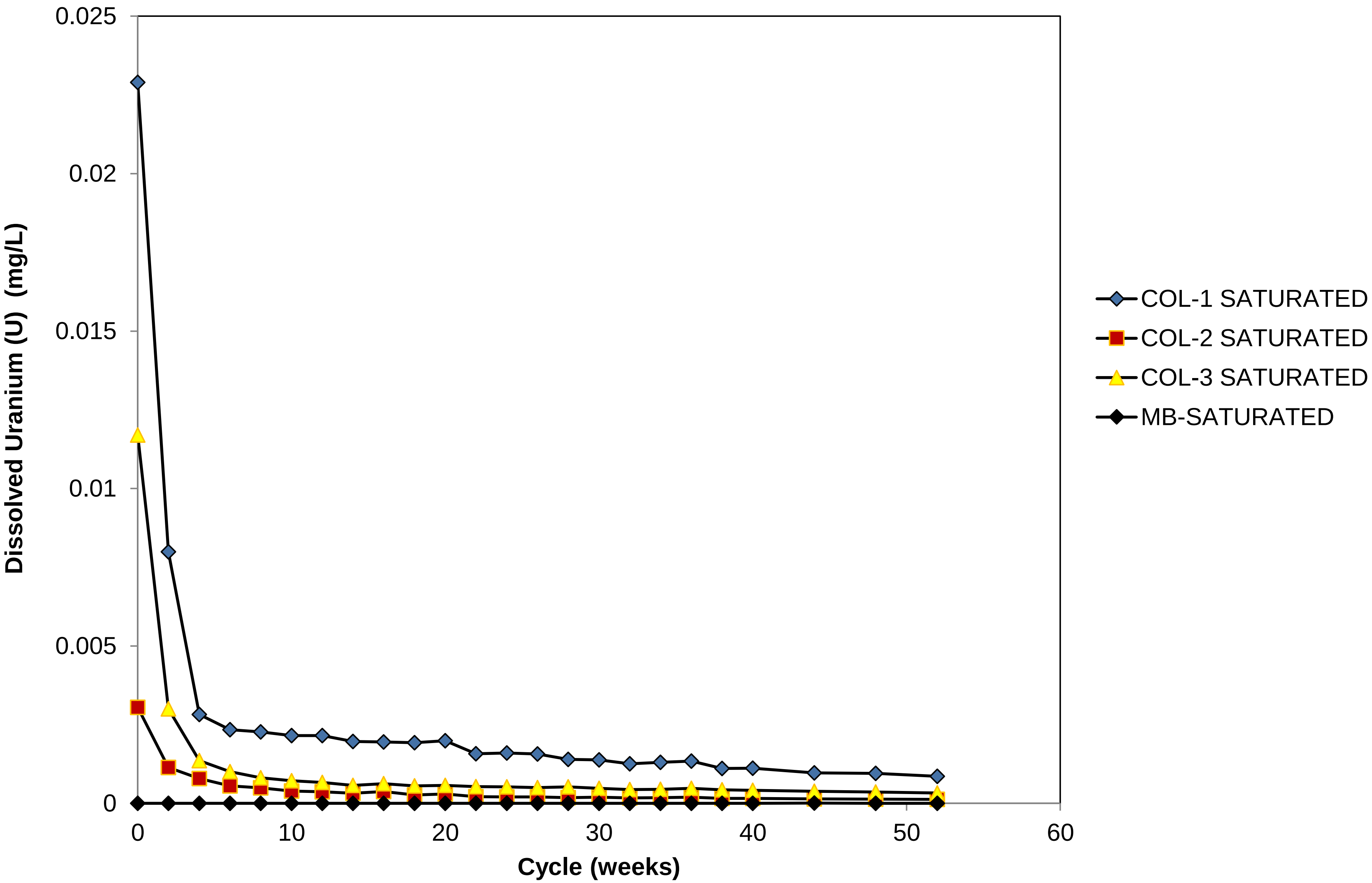

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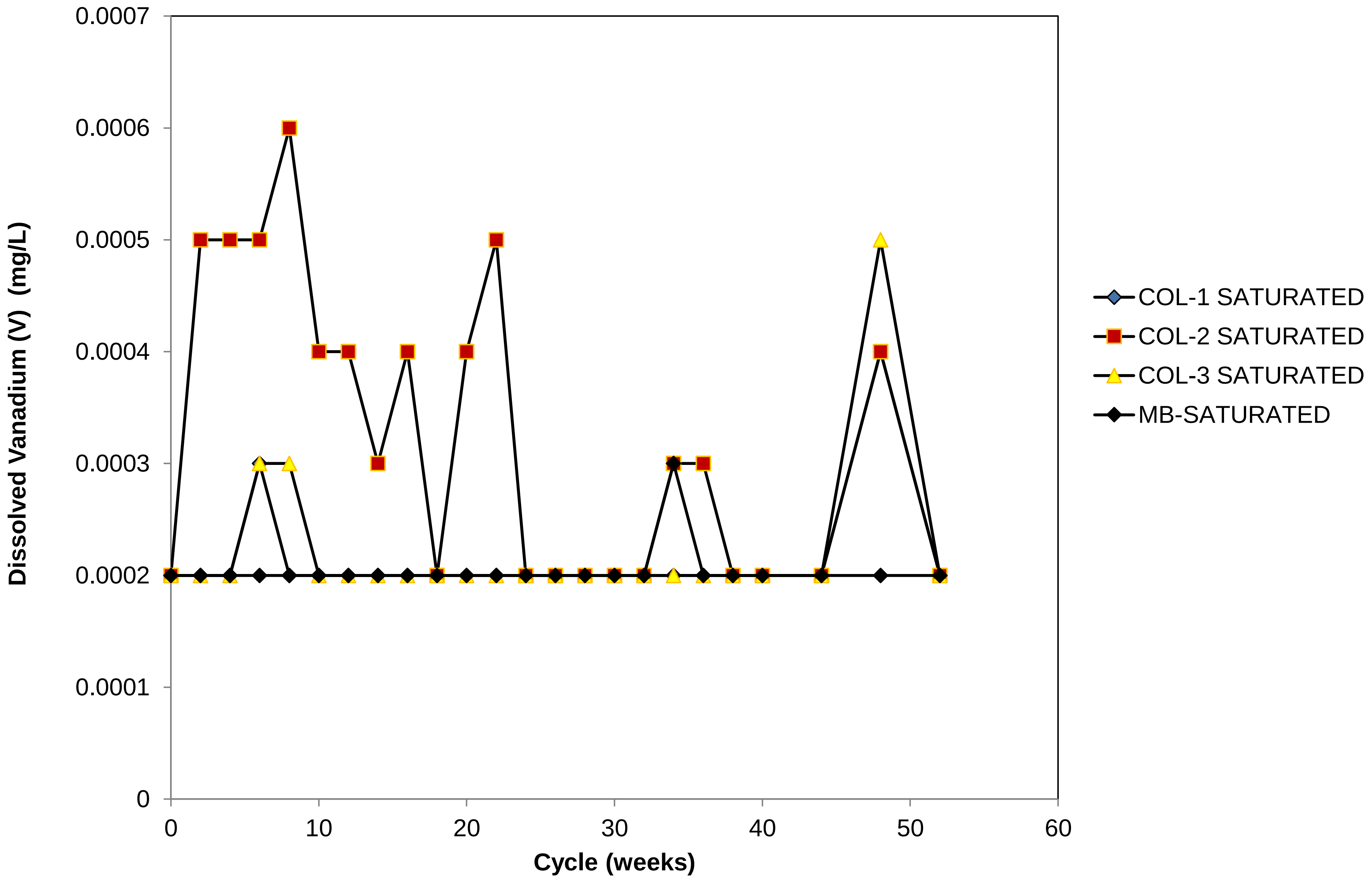

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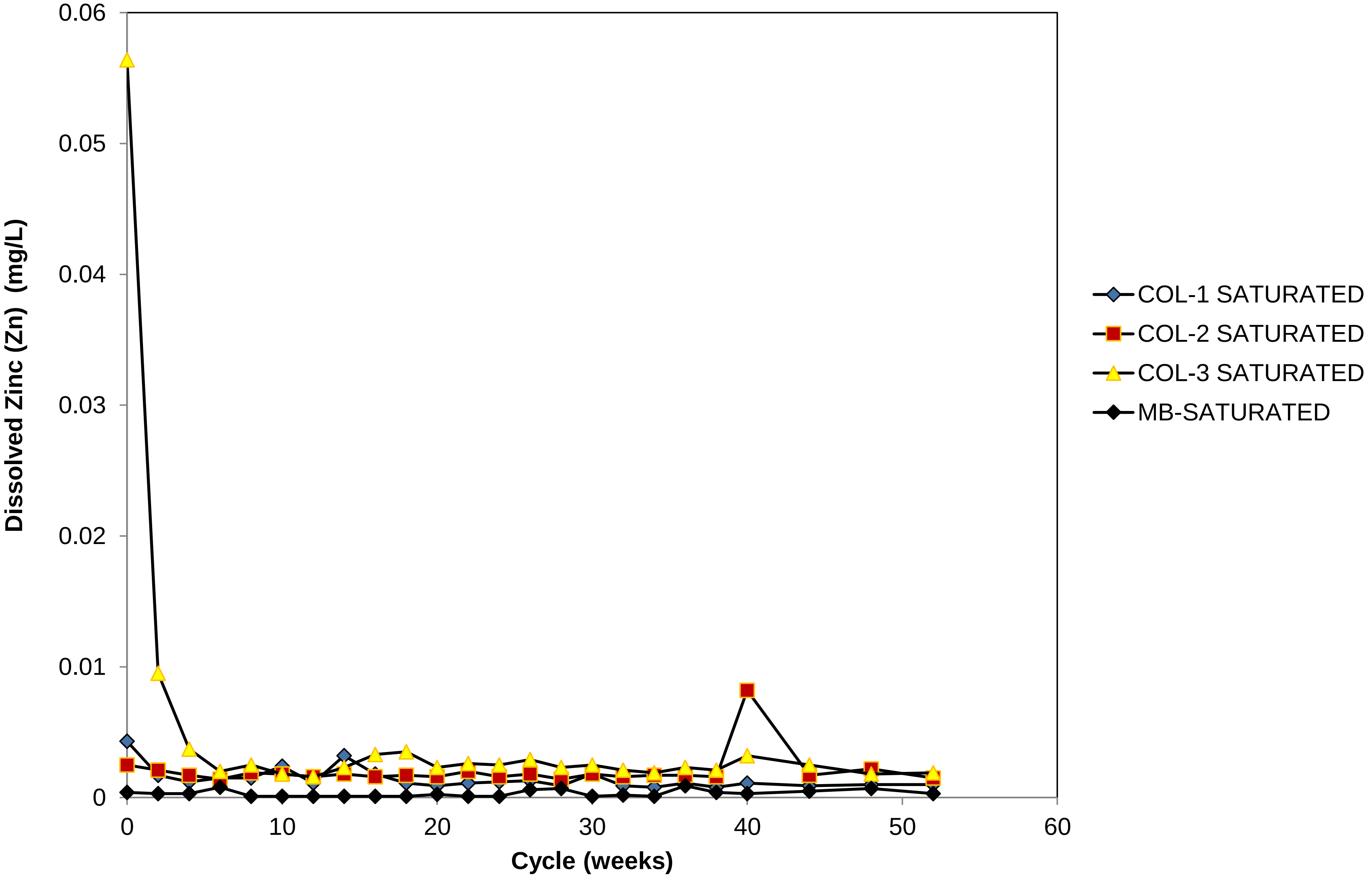

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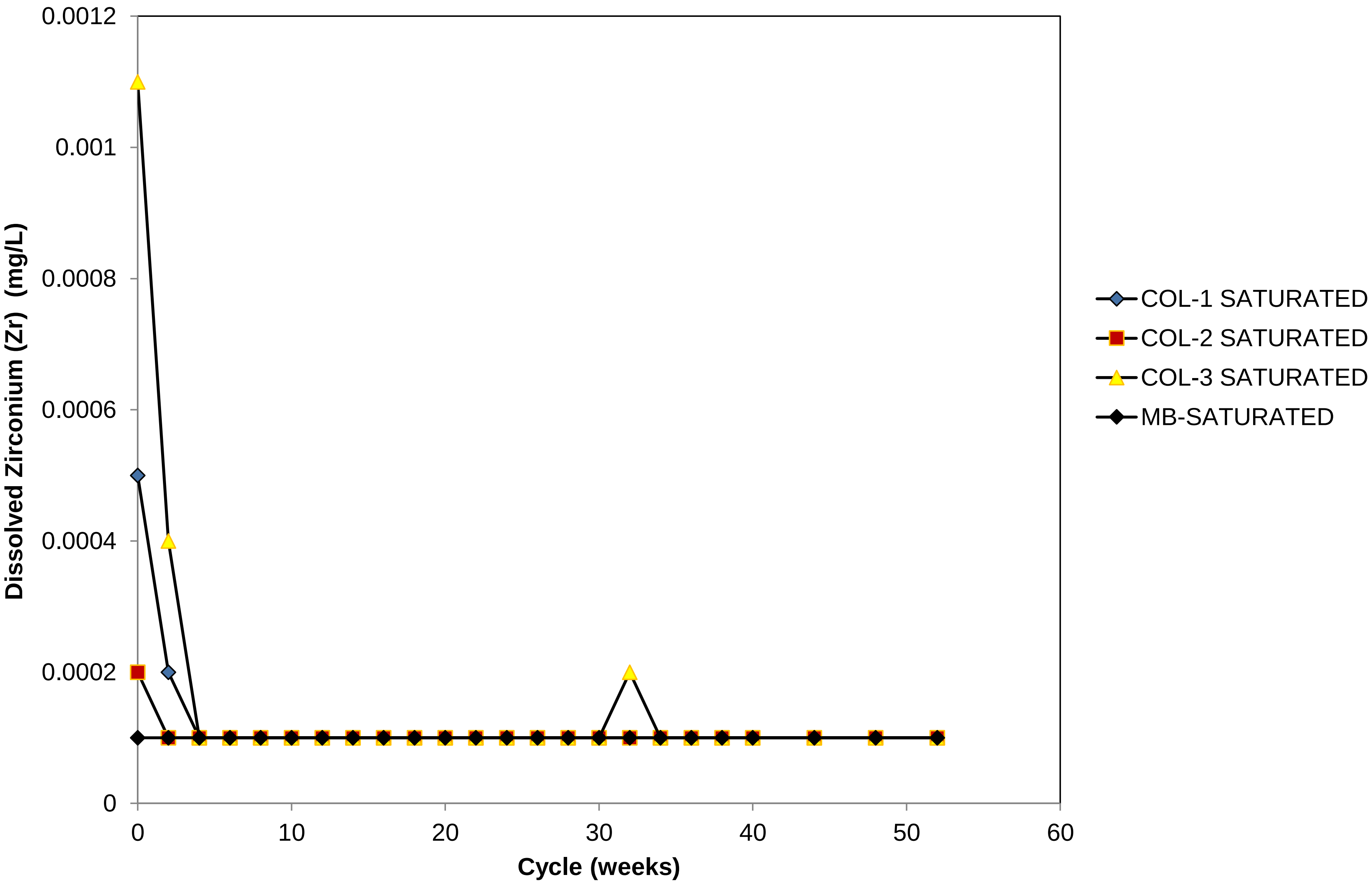

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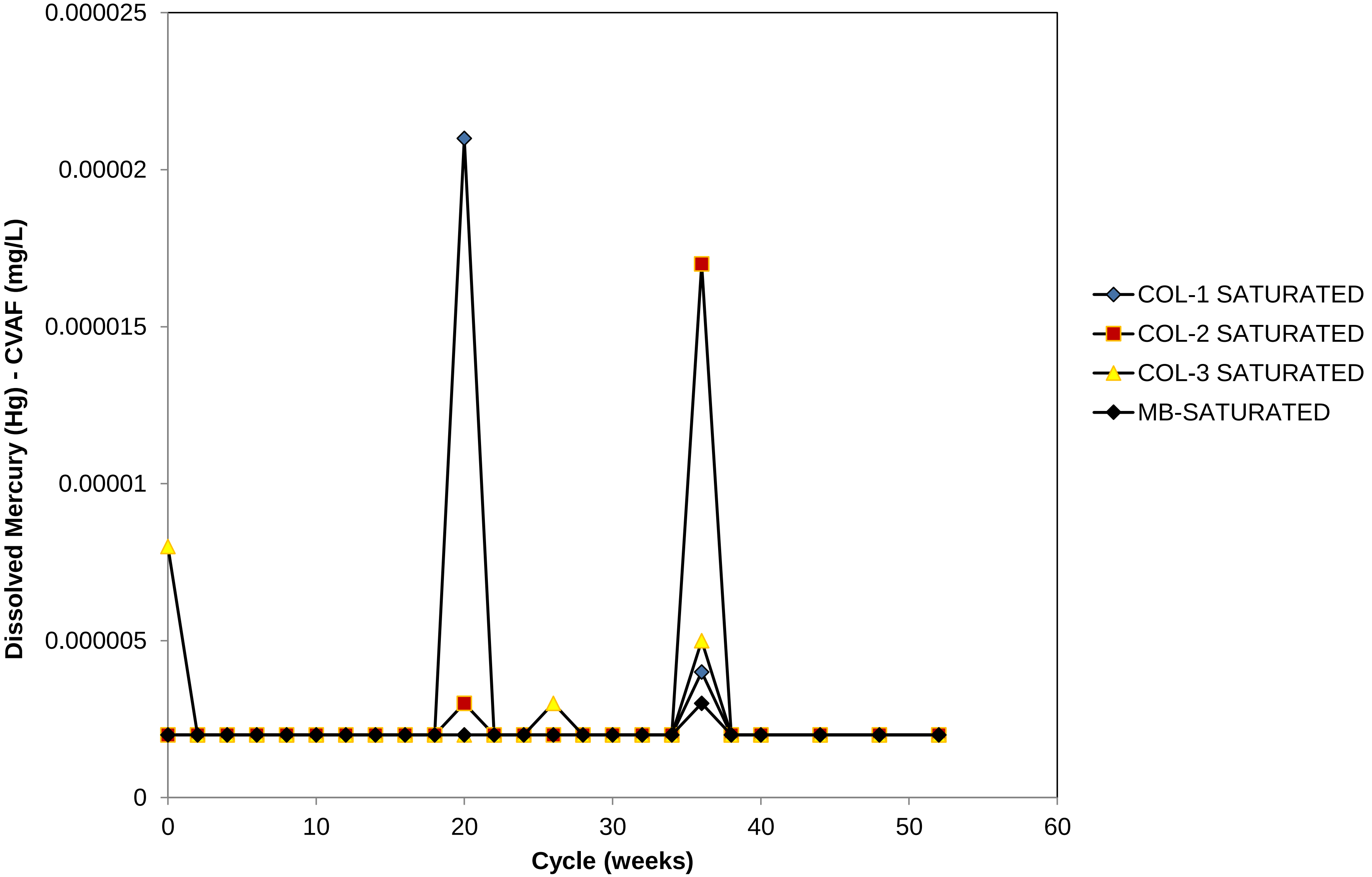
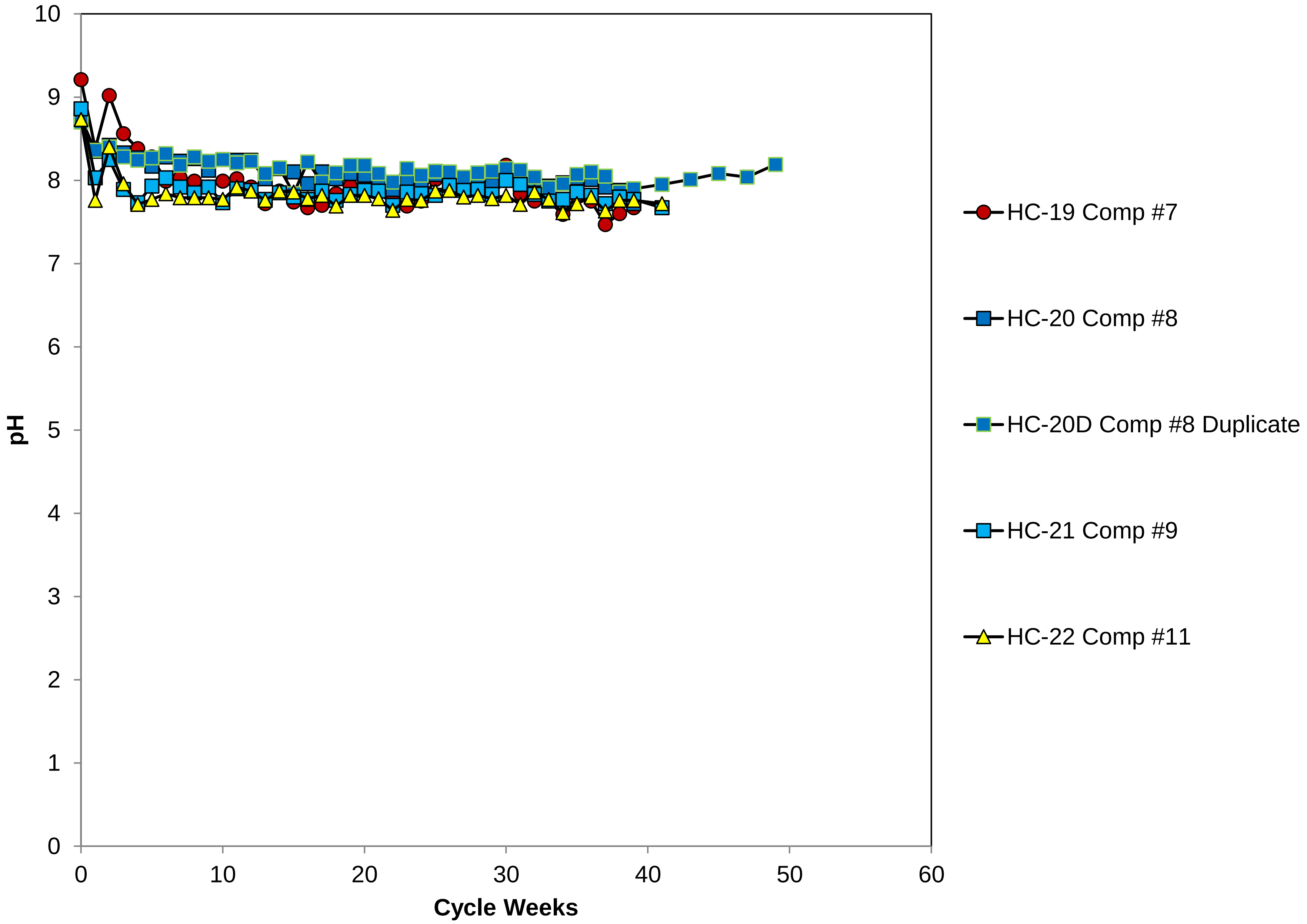
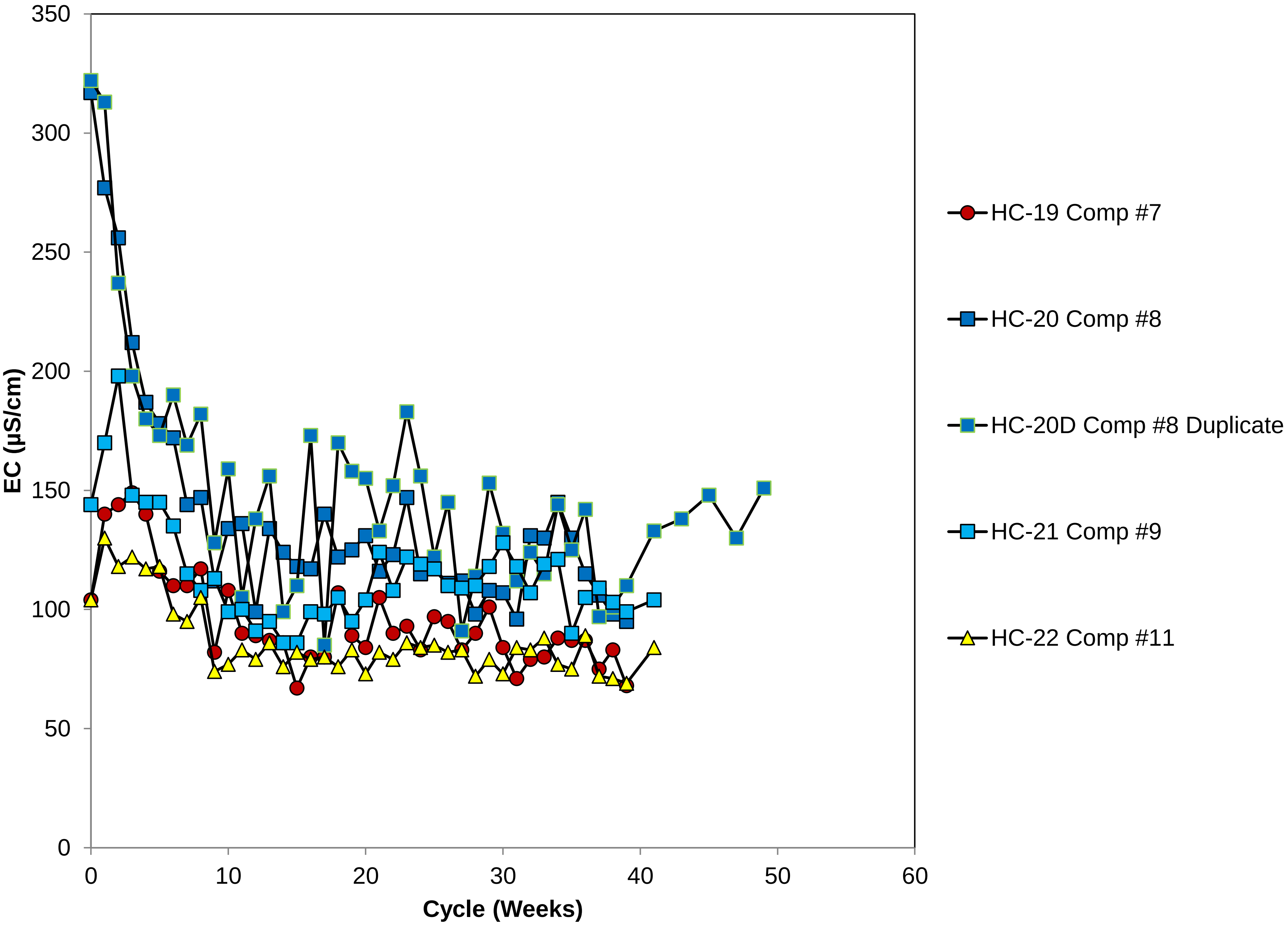
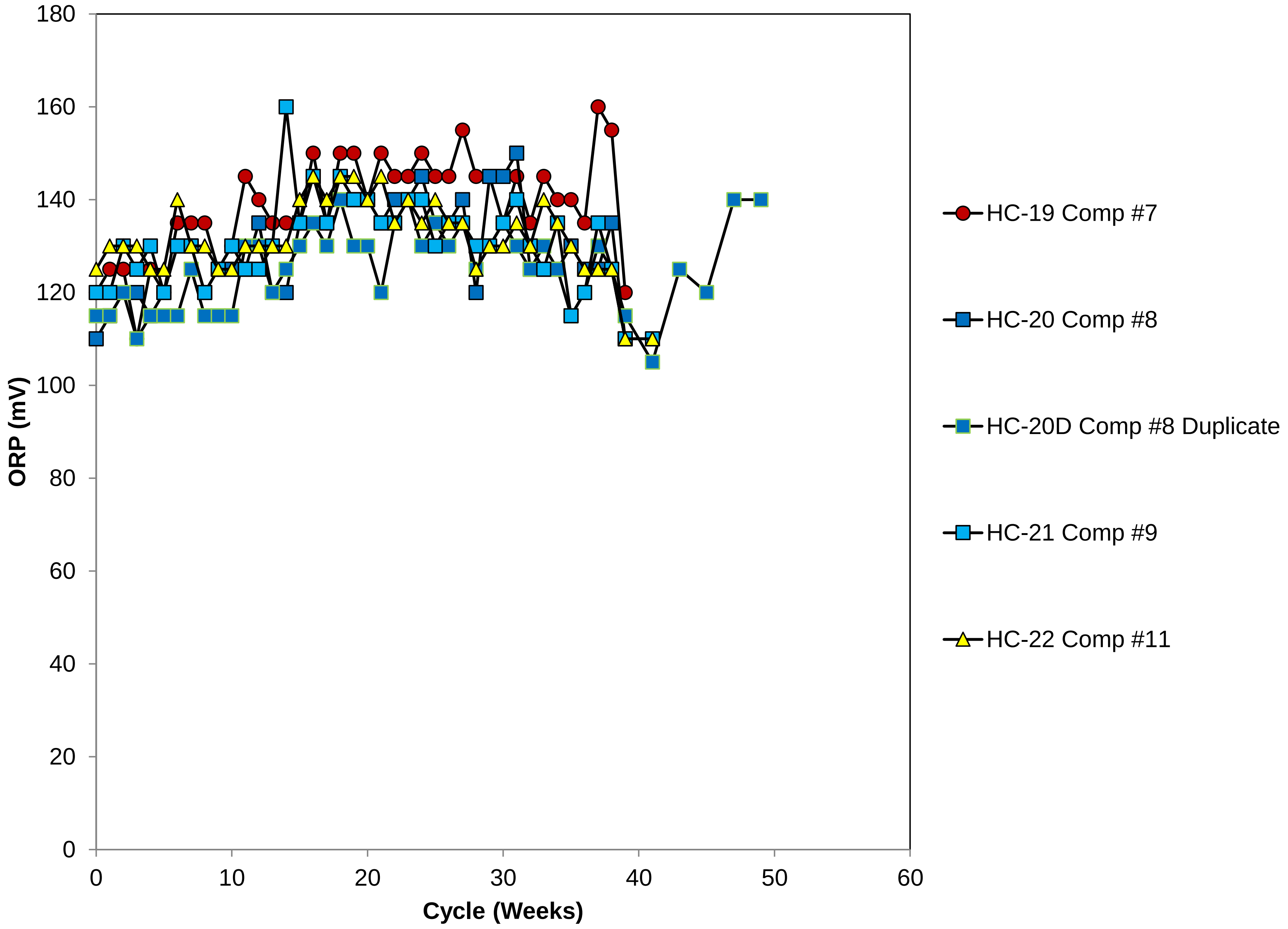


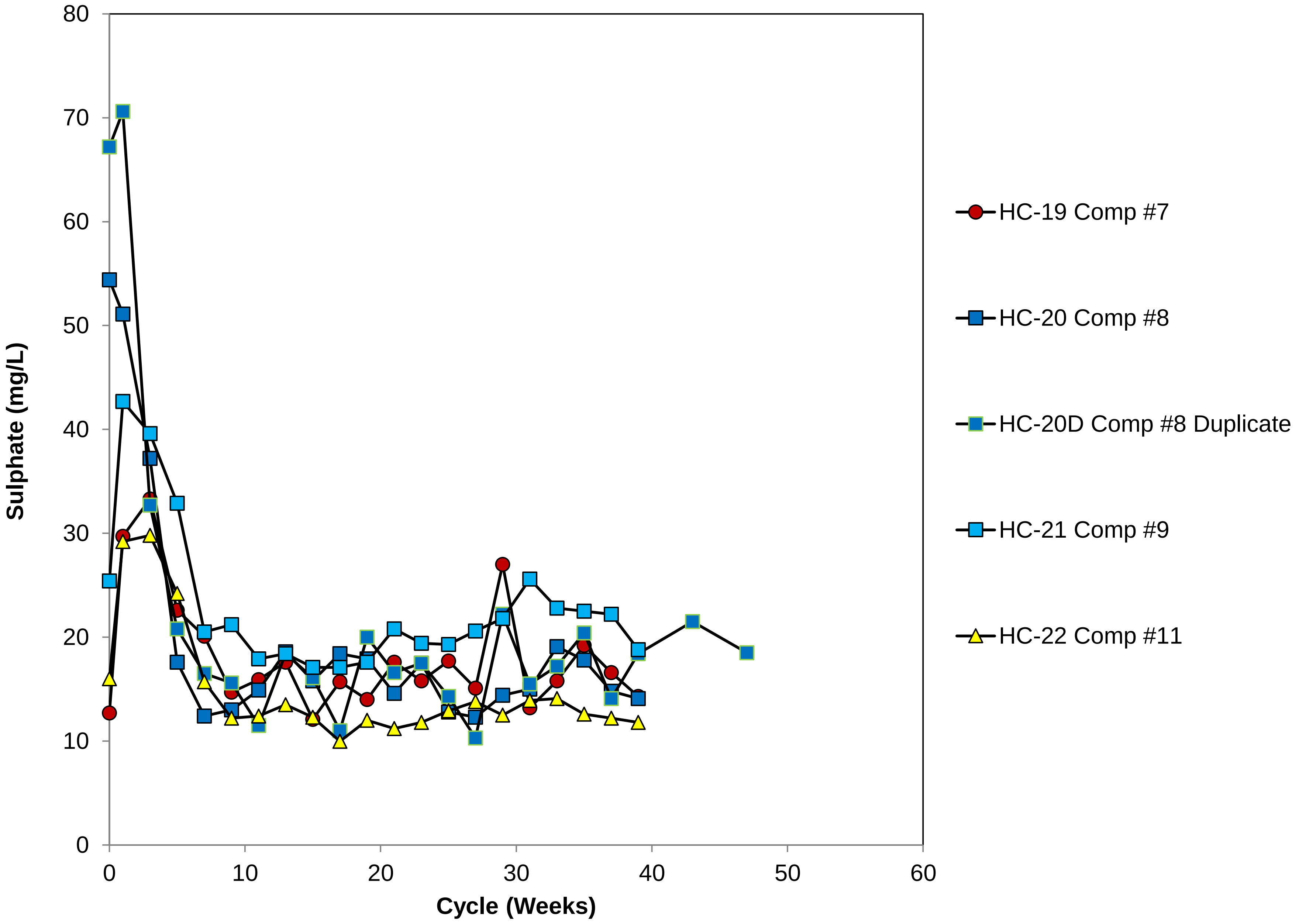
\\VAN-SVR0\Projects\01_SITES\Harper_Creek\1CH014.001_EA_Re-Submission\300_Reporting\310_Update SRK Report\030_Appendices\Appendix C\Working Files\[Appendix C3_Harper_Columns_CONC charts_1CY003.001.xlsx]

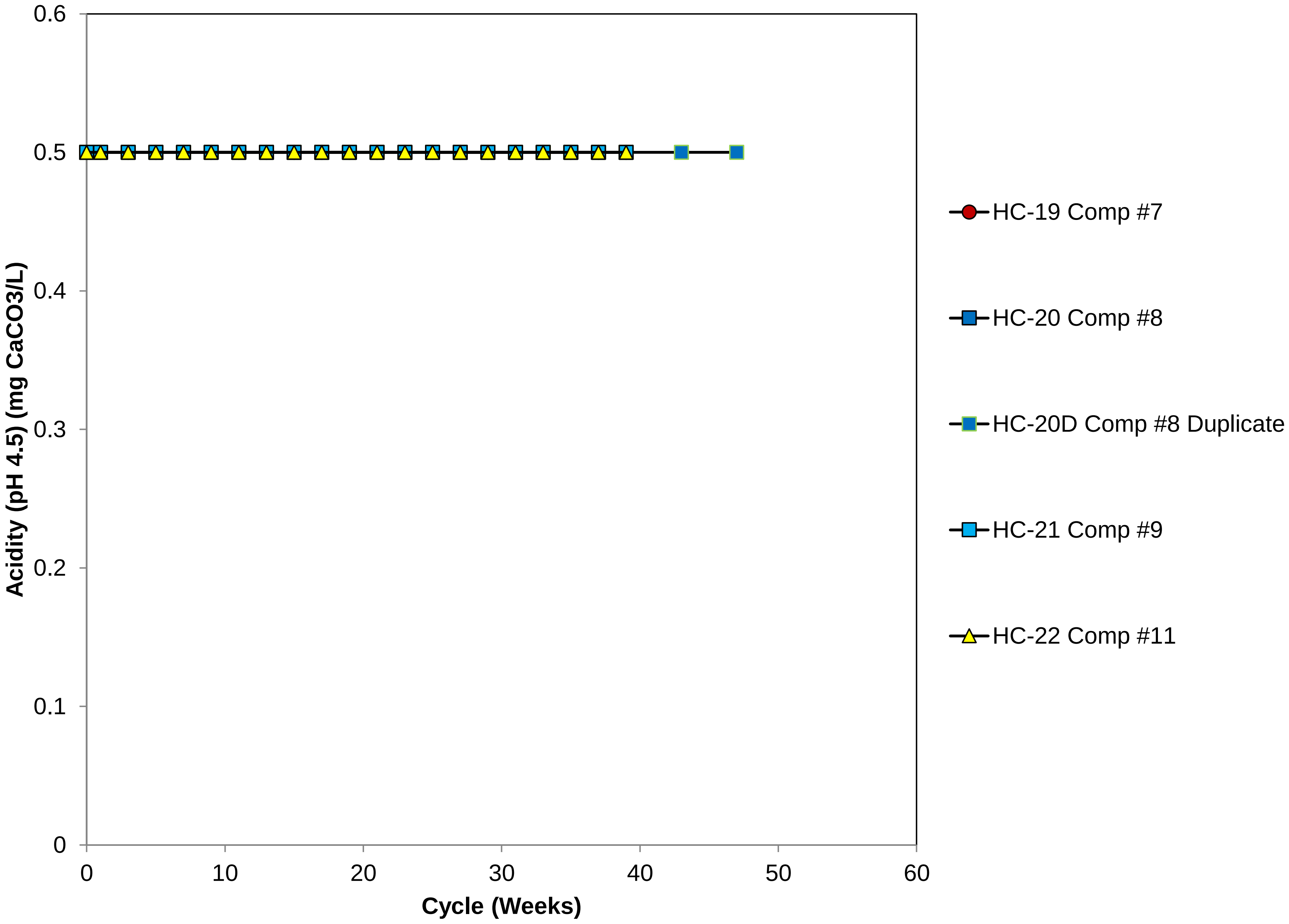
C4: Lab Humidity Cell Tests of Barrel Material

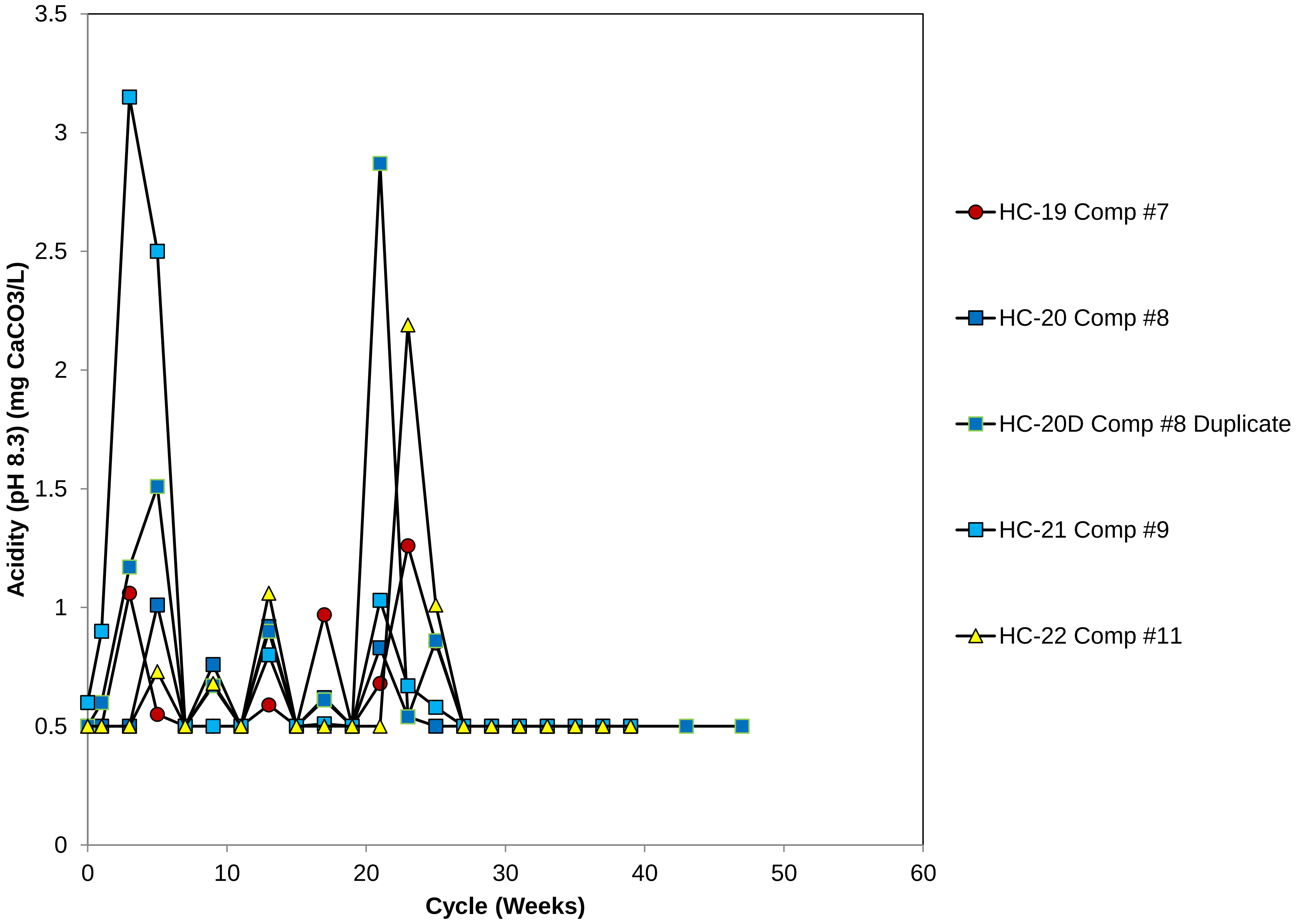

10
9
8
7
6
5
4
3
2
1
0
pH
0
10
20
30
40
50
60
Cycle Weeks
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

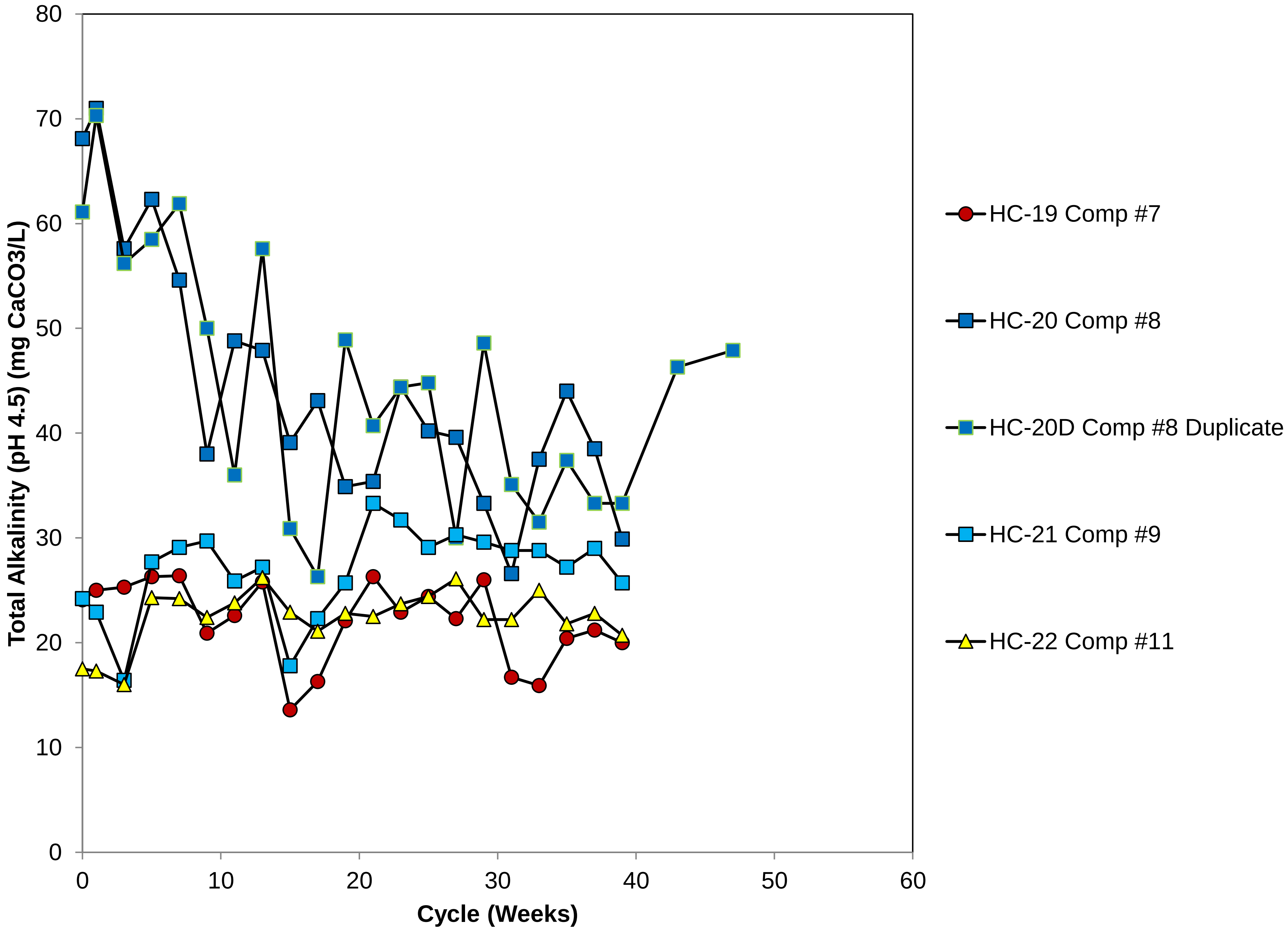

350
300
250
200
150
100
50
0
EC (µS/cm)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

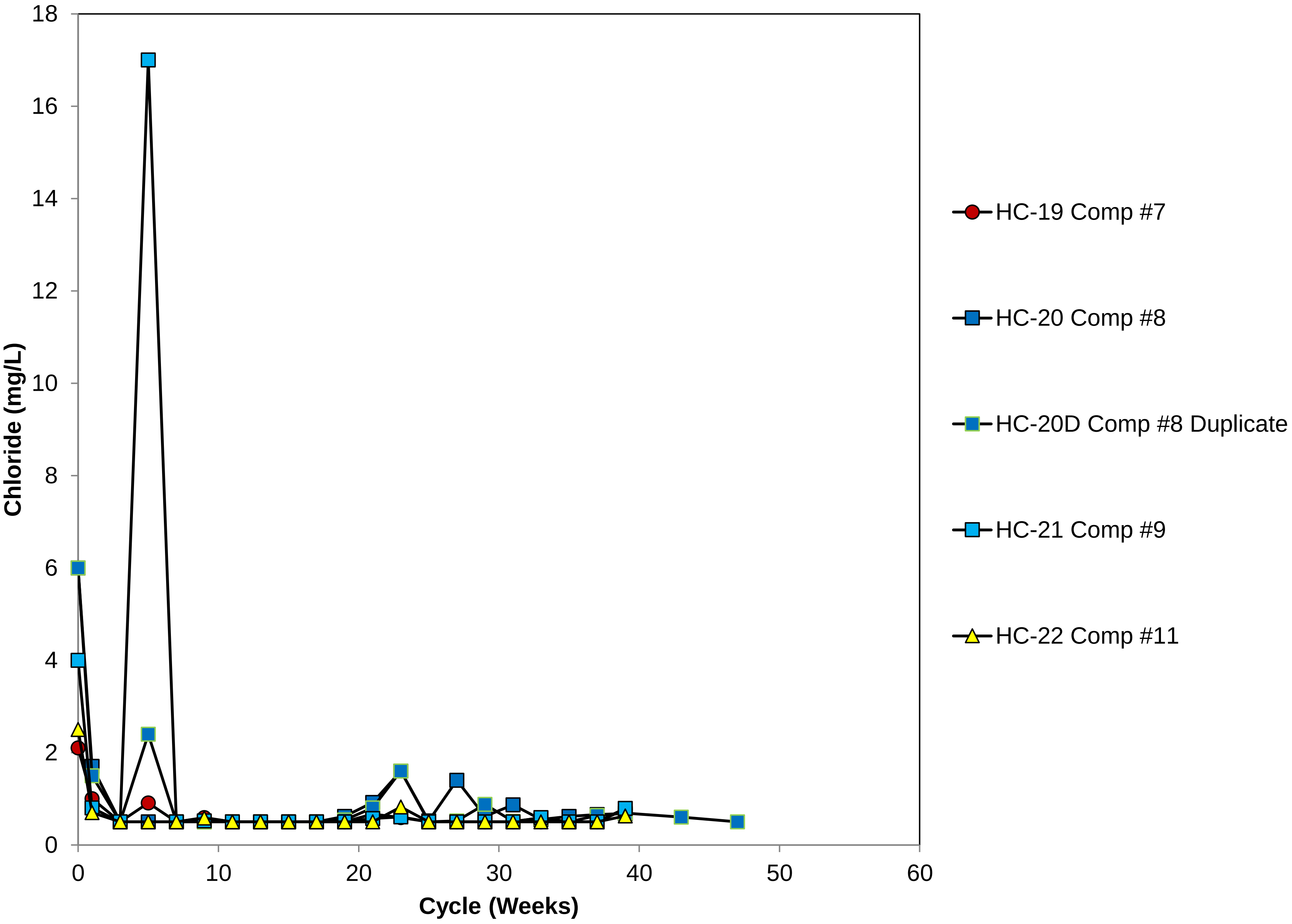

180
160
140
120
100
80
60
40
20
0
ORP (mV)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

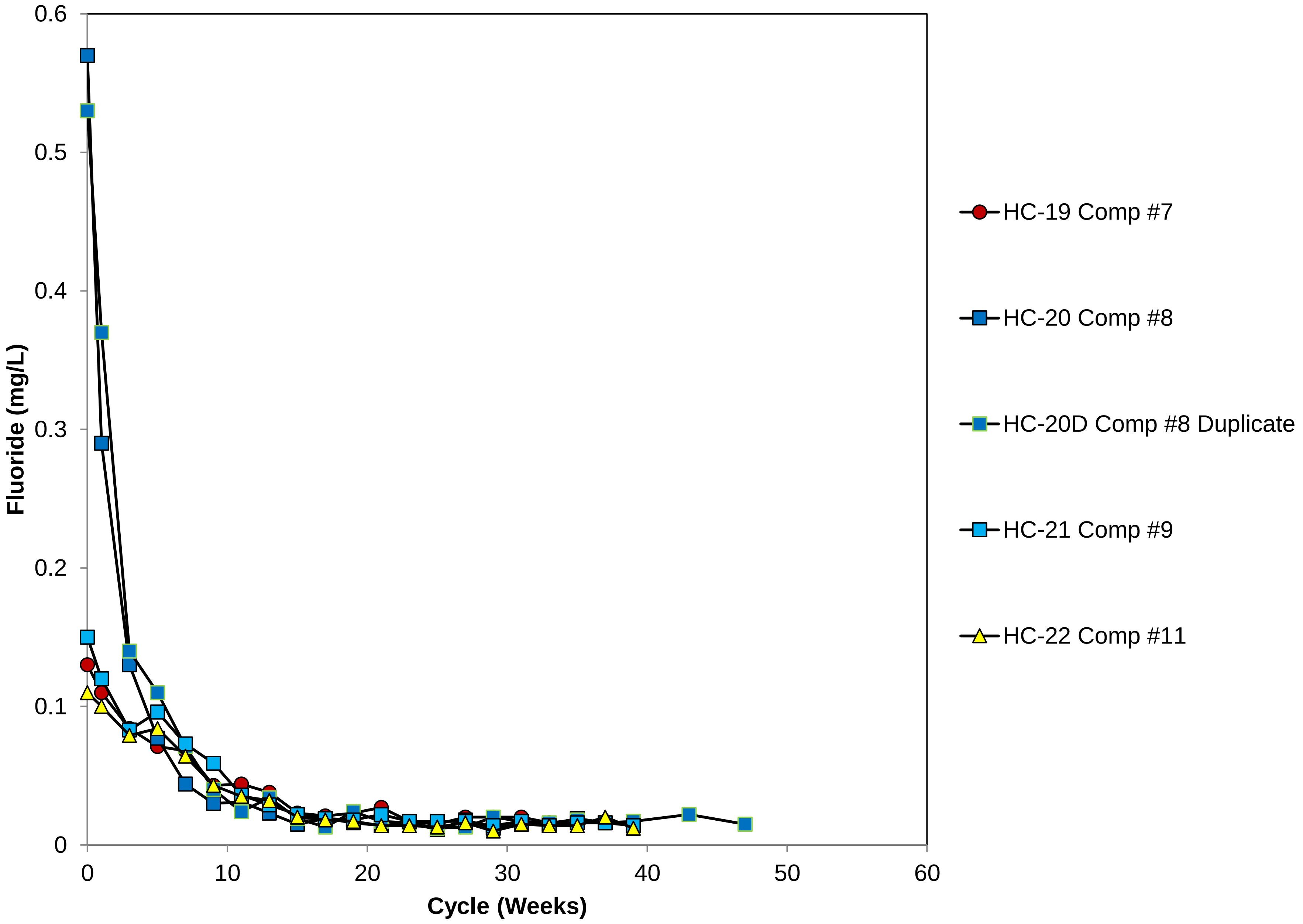

Sulphate (mg/L)
80
70
60
50
40
30
20
10
0
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

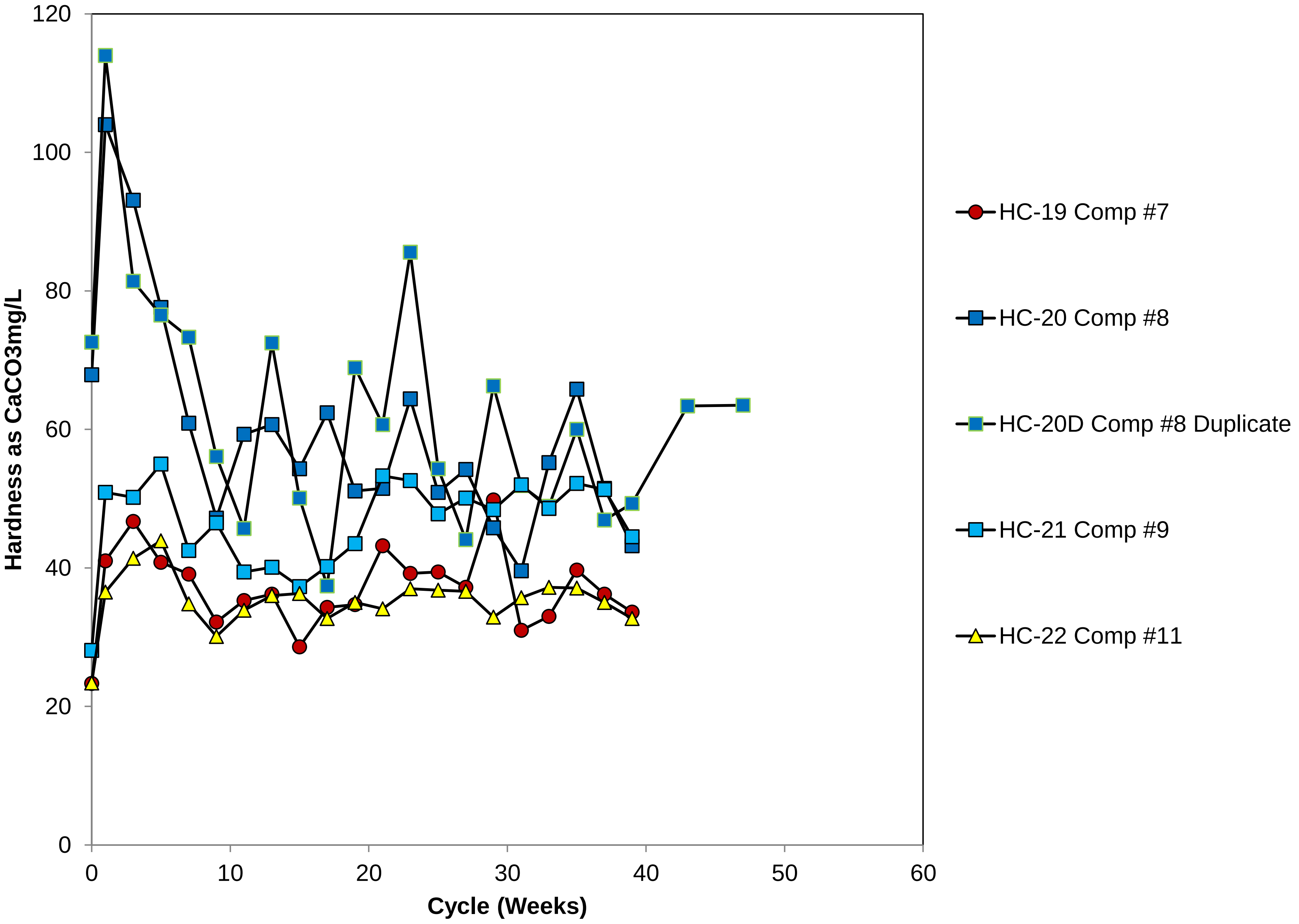

0.6
0.5
0.4
0.3
0.2
0.1
0
Acidity (pH 4.5) (mg CaCO3/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

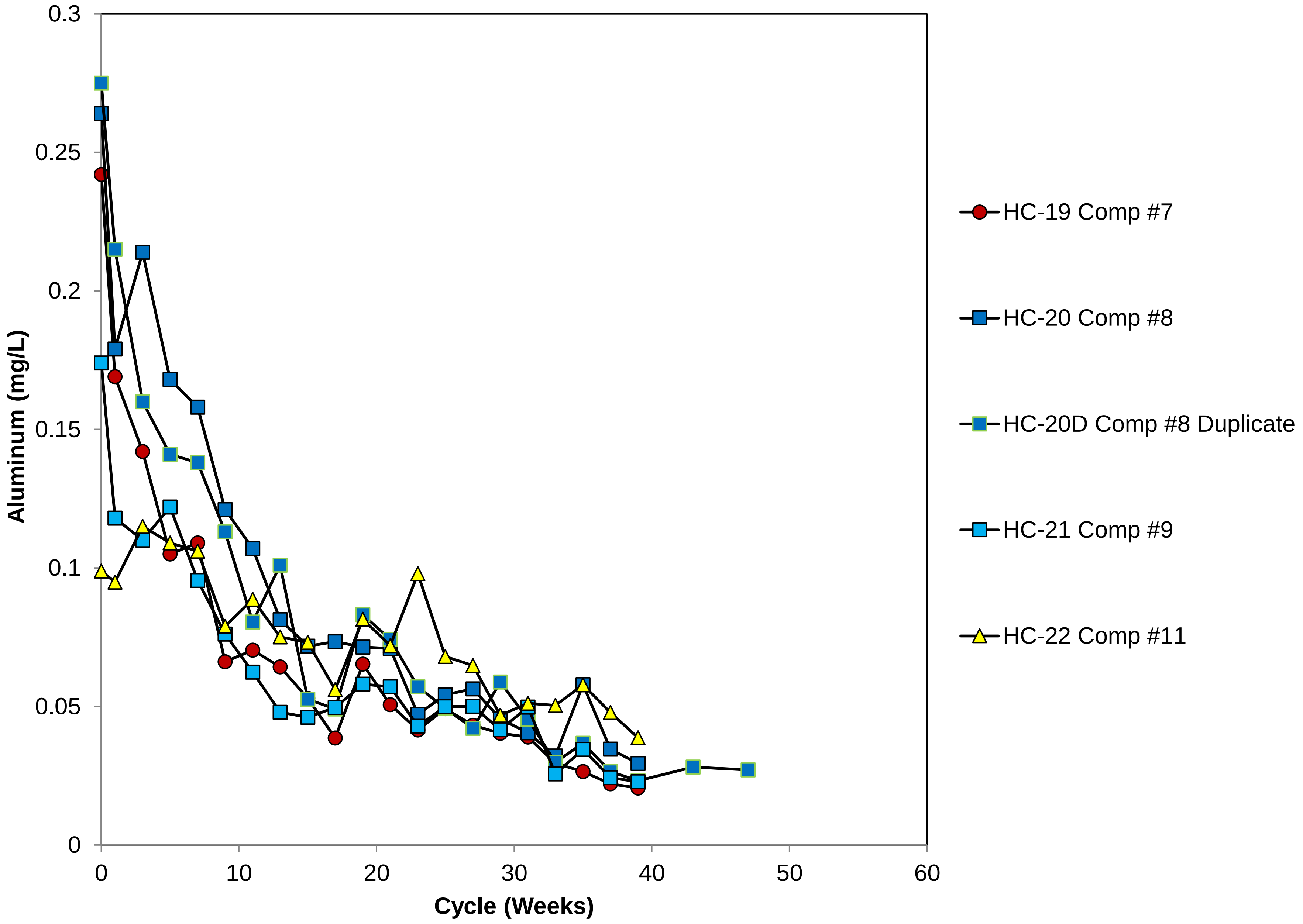

3.5
3
2.5
2
1.5
1
0.5
0
Acidity (pH 8.3) (mg CaCO3/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

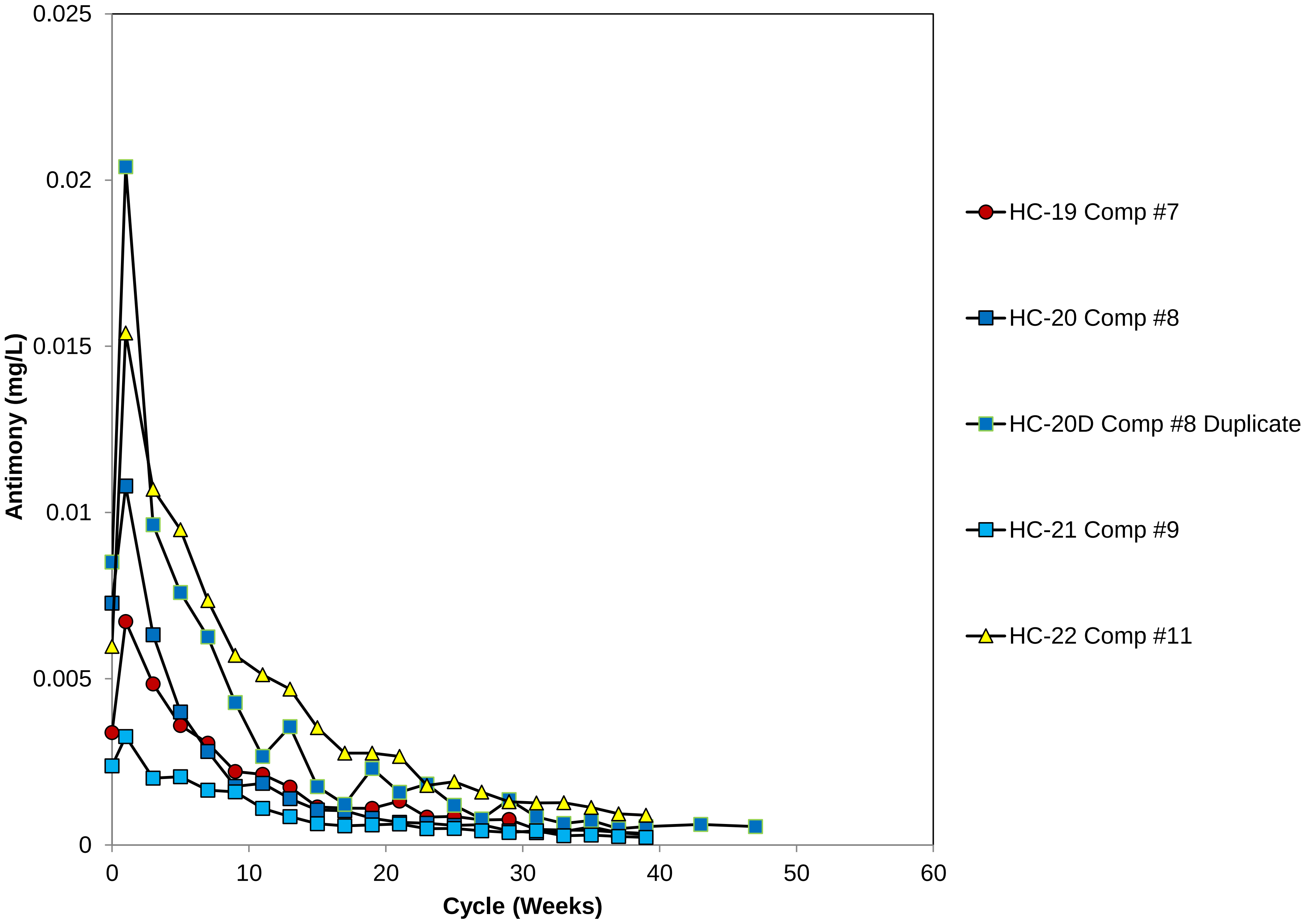

80
70
60
50
40
30
20
10
0
Total Alkalinity (pH 4.5) (mg CaCO3/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

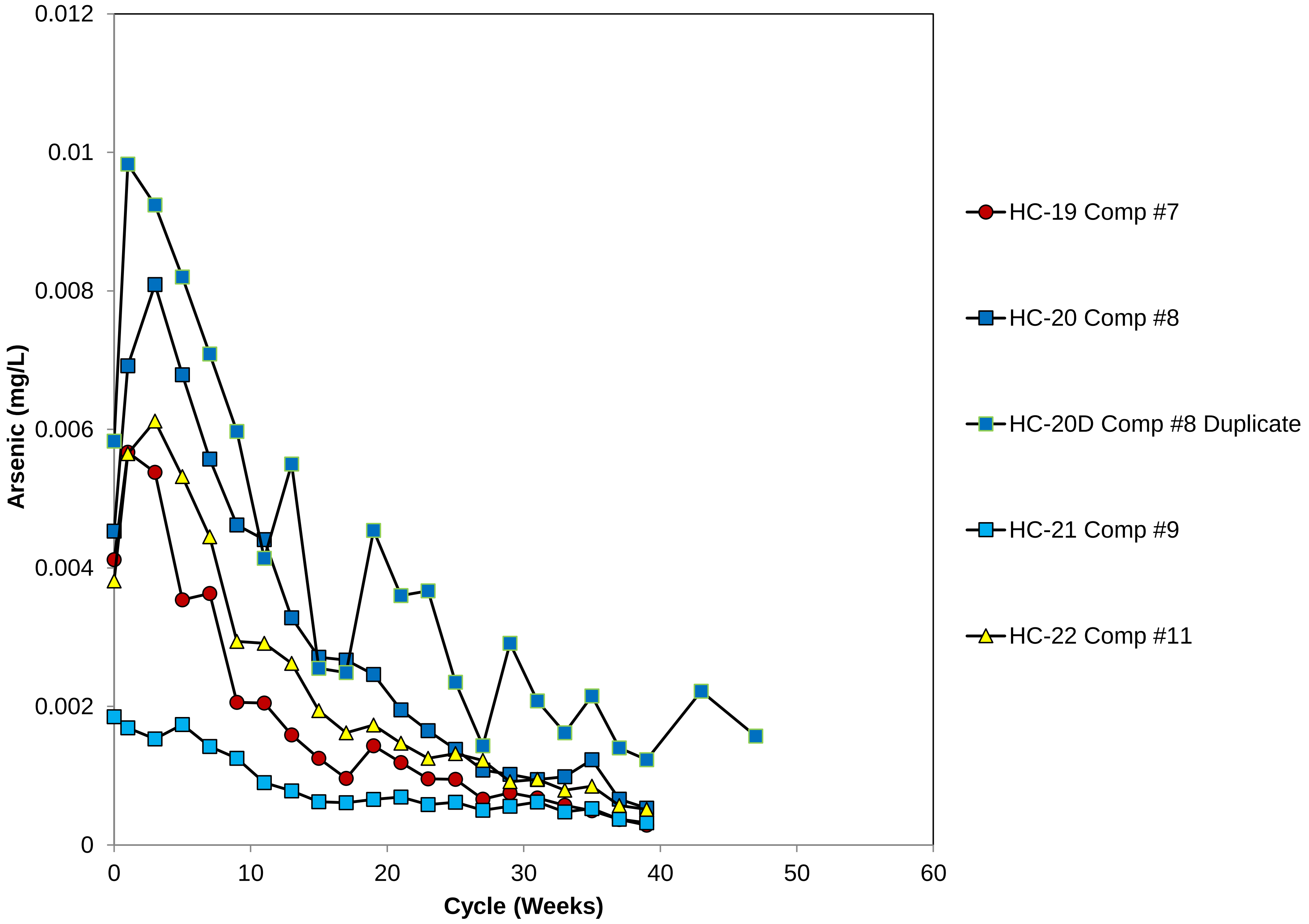

18
16
14
12
10
8
6
4
2
0
Chloride (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

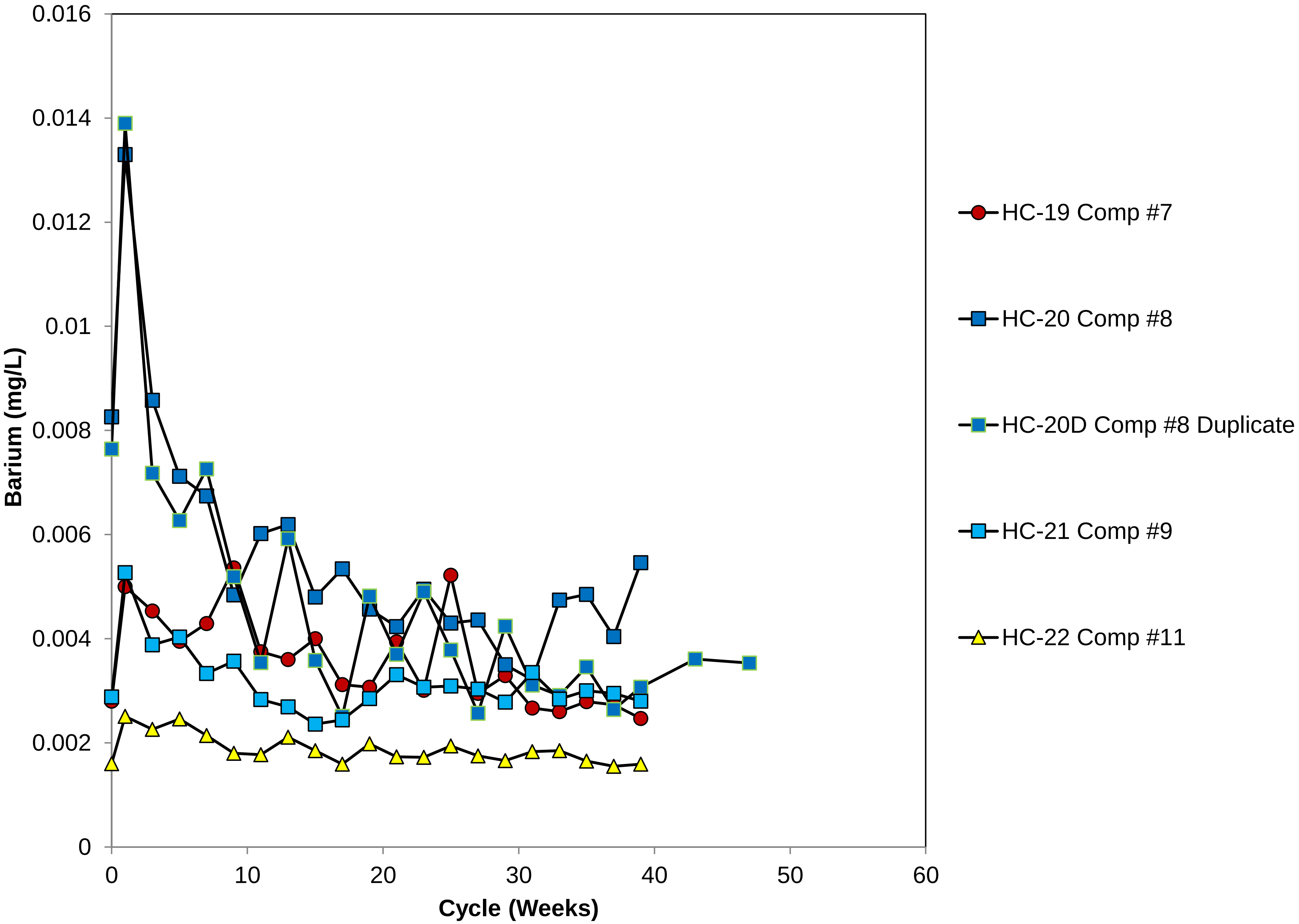

0.6
0.5
0.4
0.3
0.2
0.1
0
Fluoride (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

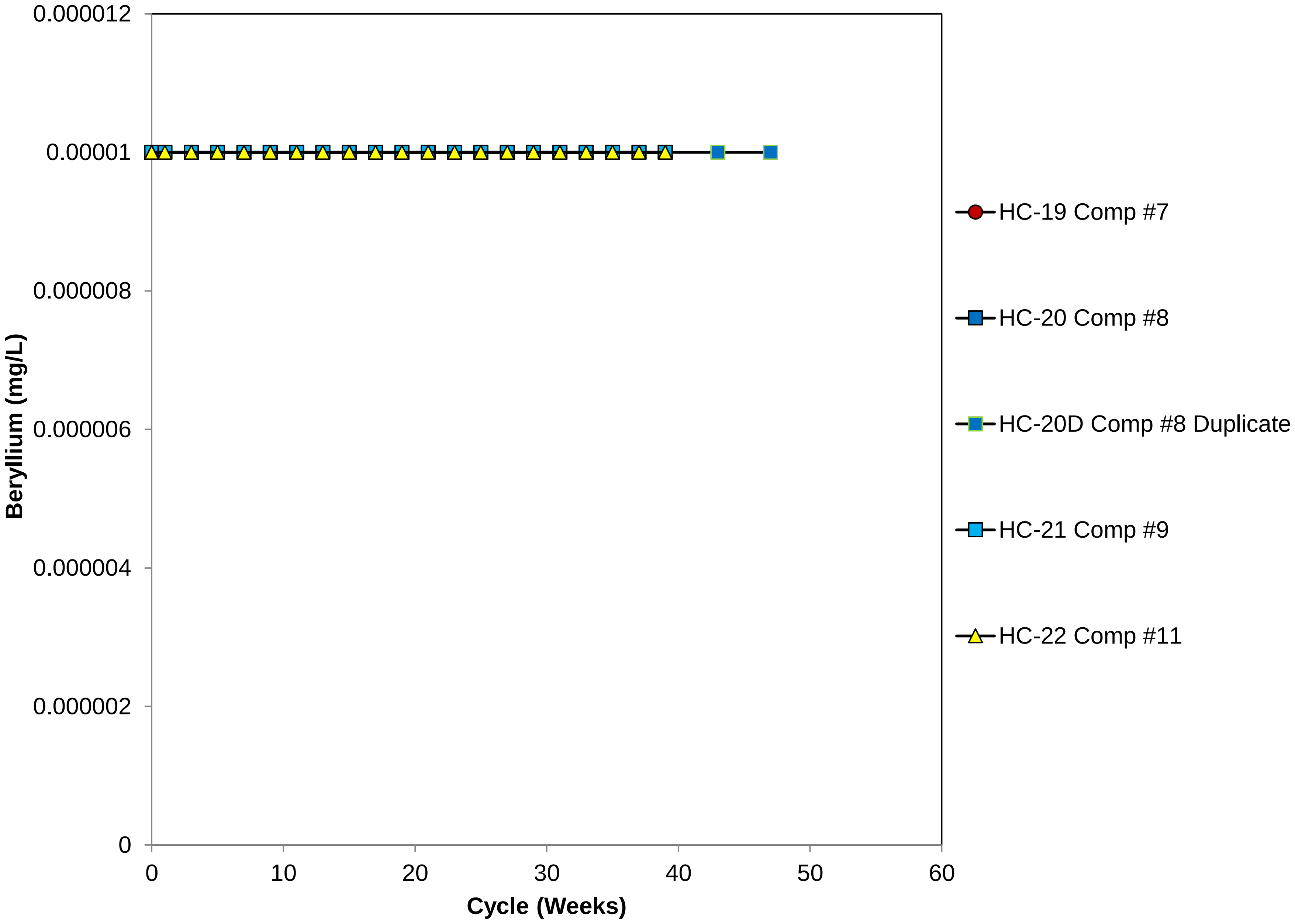

120
100
80
60
40
20
0
Hardness as CaCO3mg/L
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

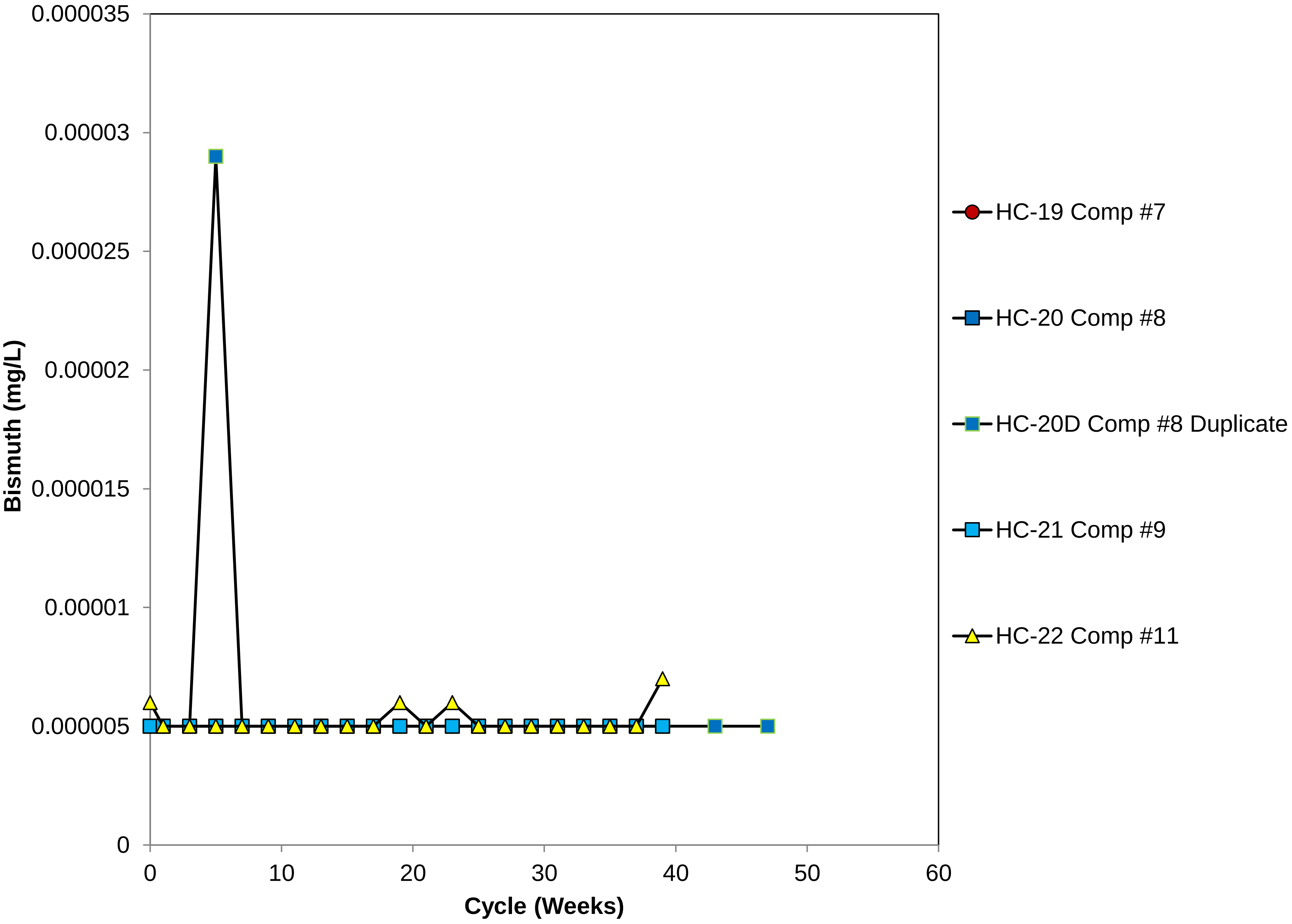

0.3
0.25
0.2
0.15
0.1
0.05
0
Aluminum (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

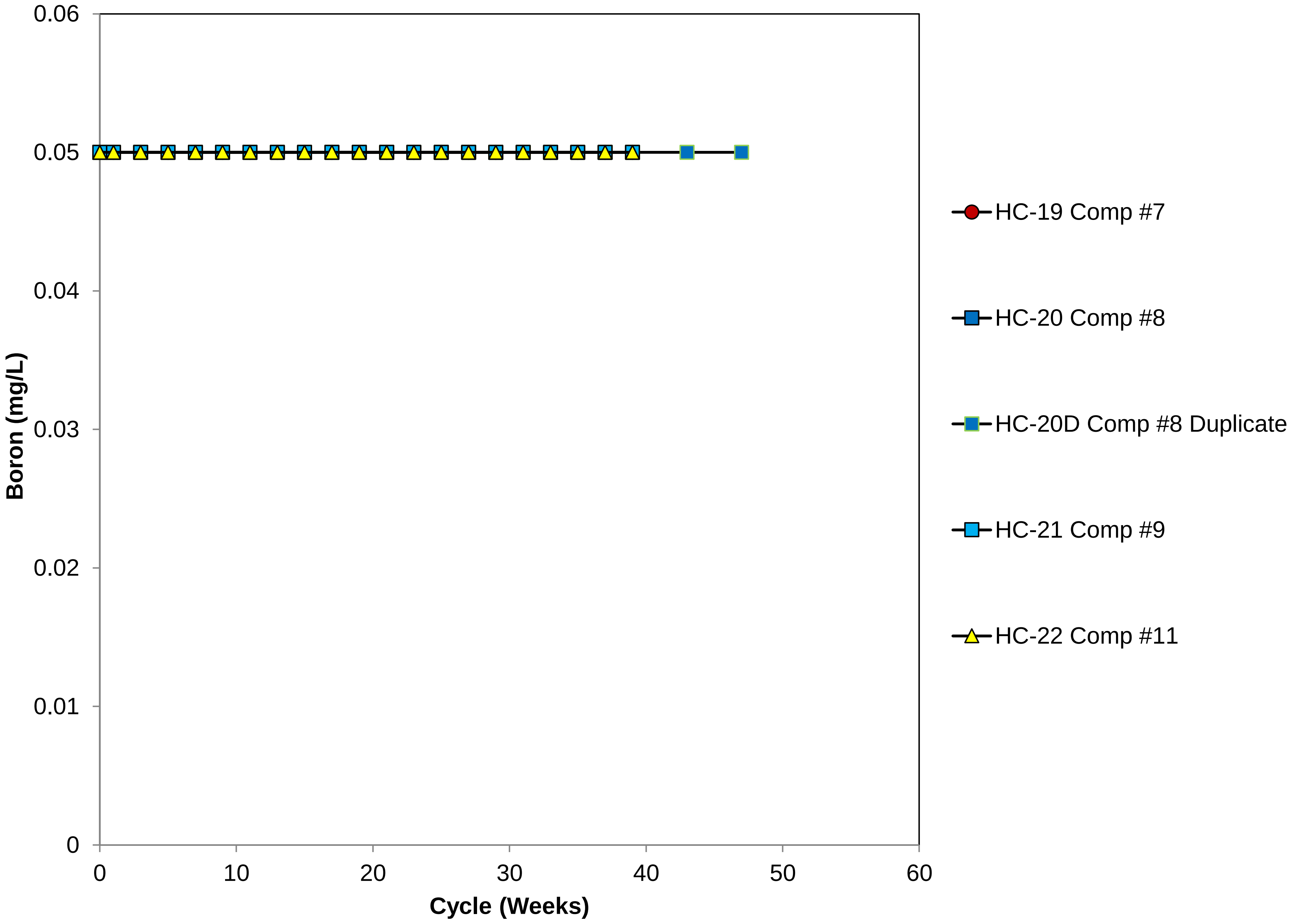

0.025
0.02
0.015
0.01
0.005
0
Antimony (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

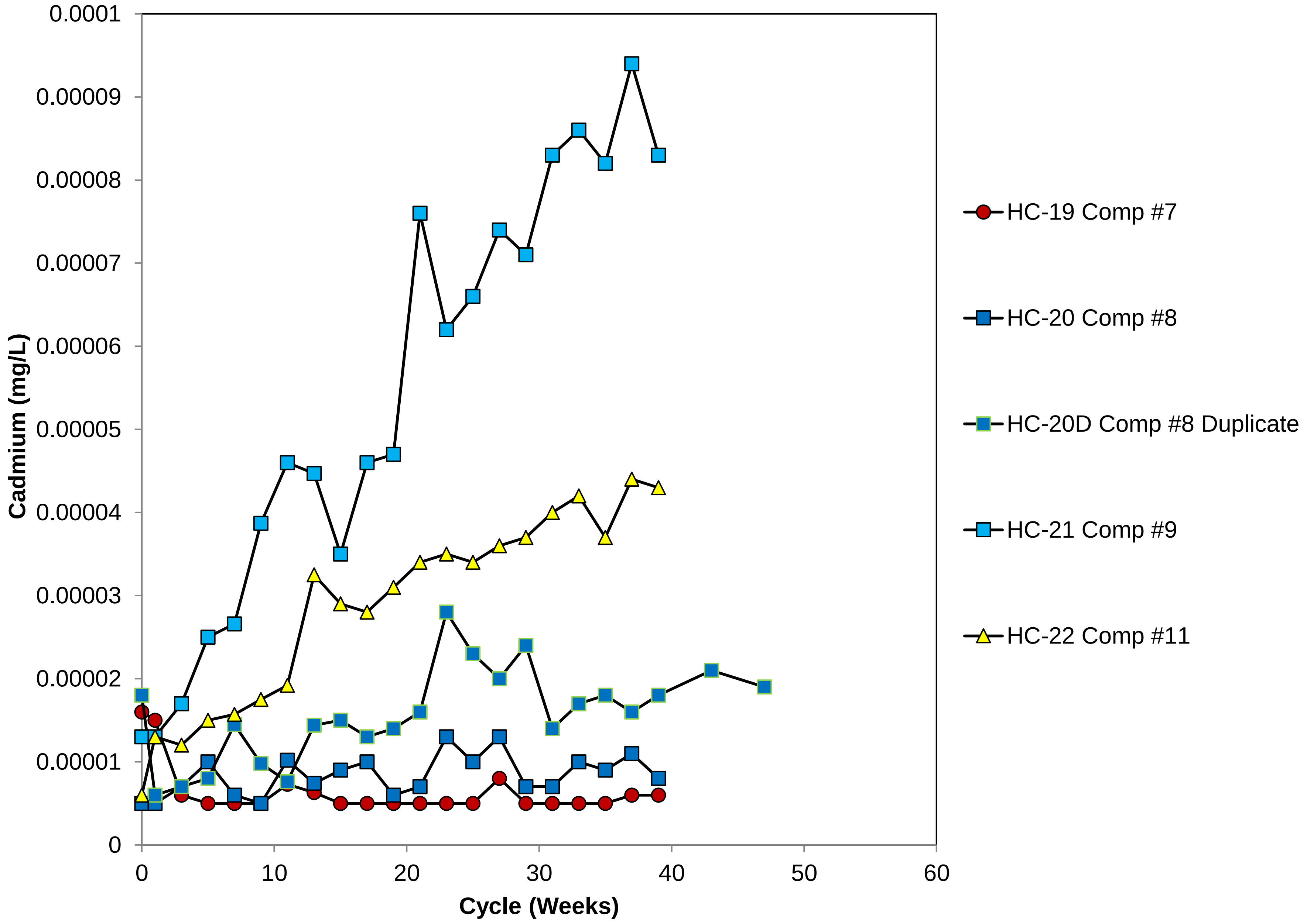

0.012
0.01
0.008
0.006
0.004
0.002
0
Arsenic (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

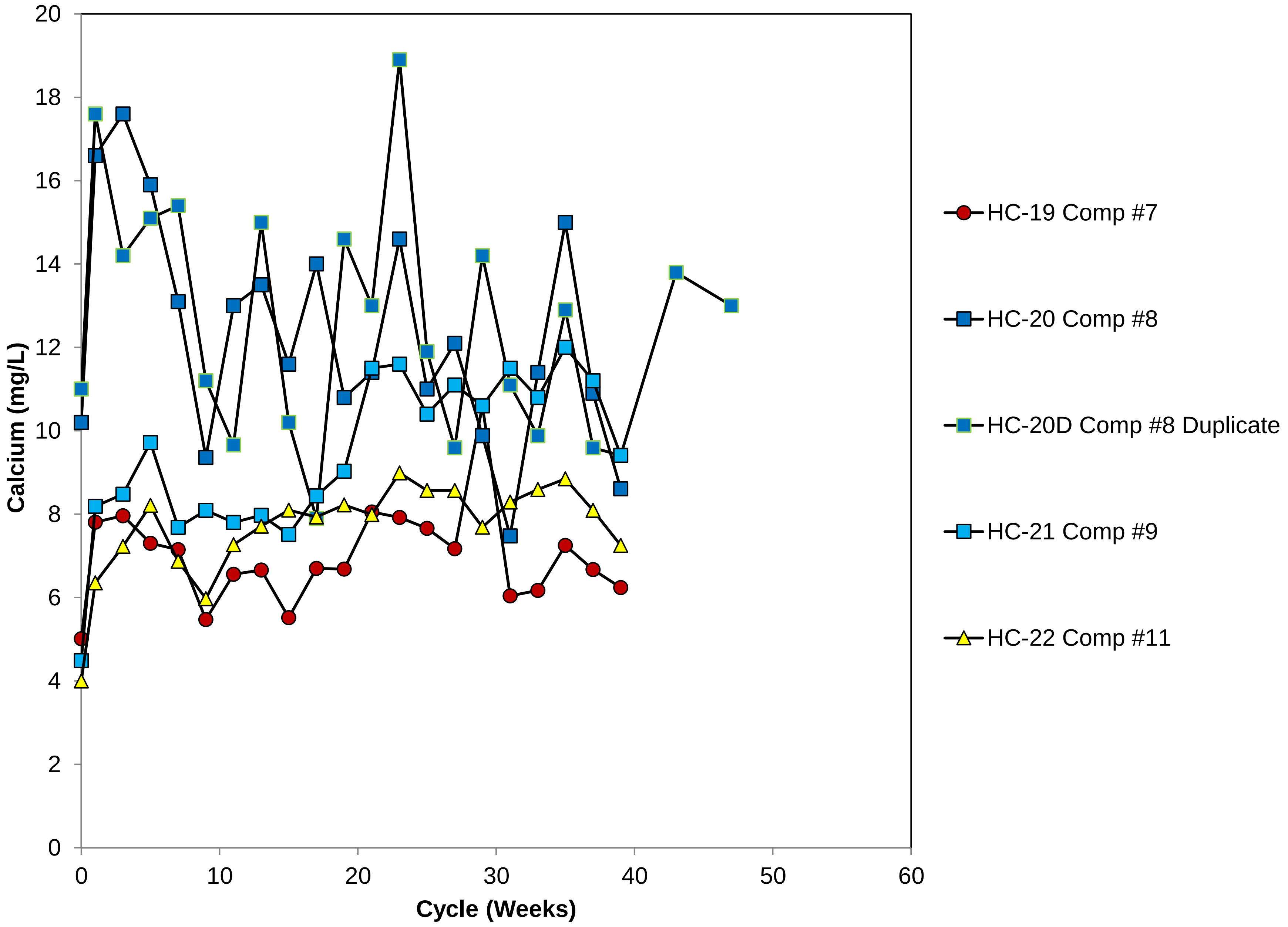

0.016
0.014
0.012
0.01
0.008
0.006
0.004
0.002
0
Barium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

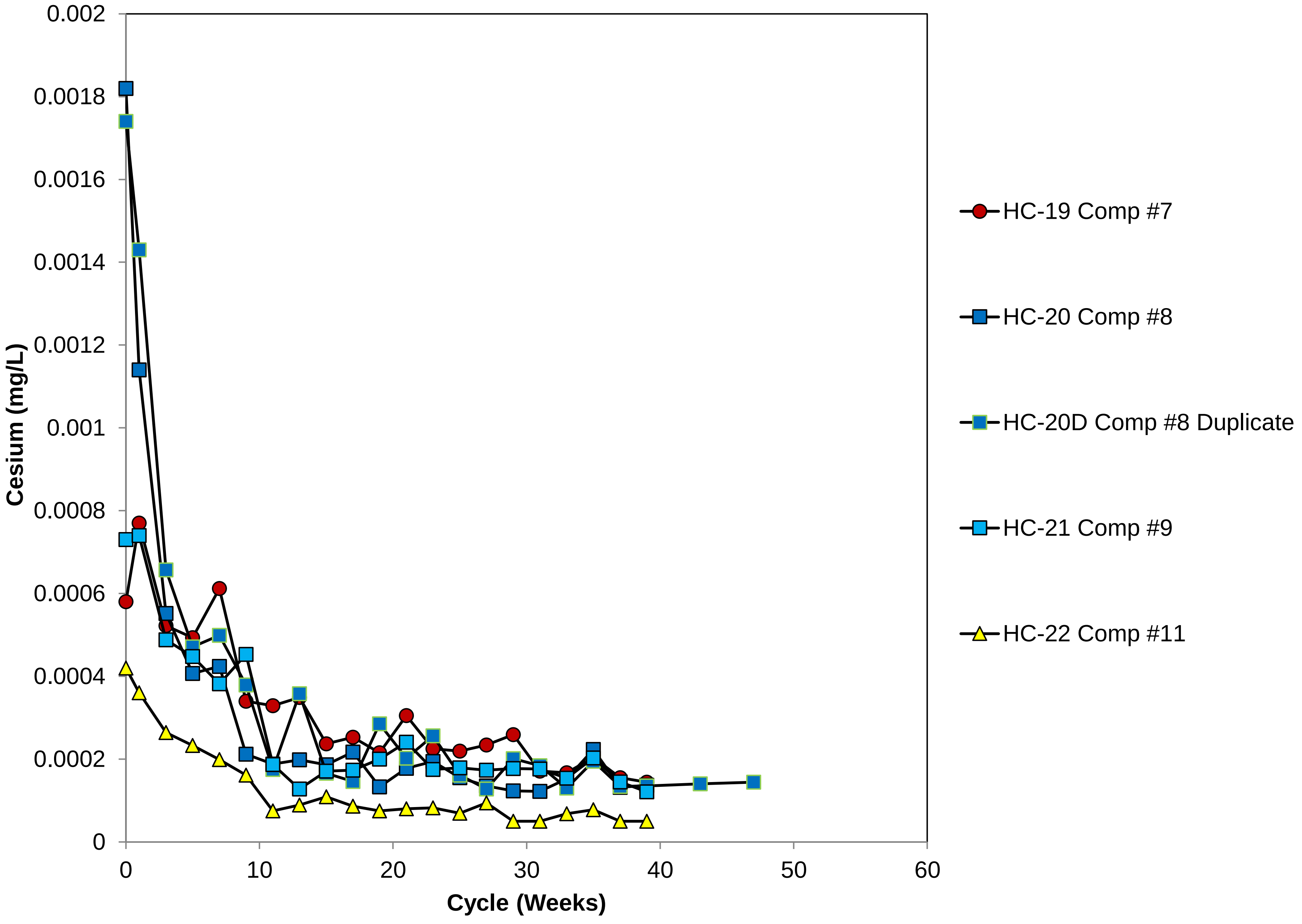

0.000012
0.00001
0.000008
0.000006
0.000004
0.000002
0
Beryllium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

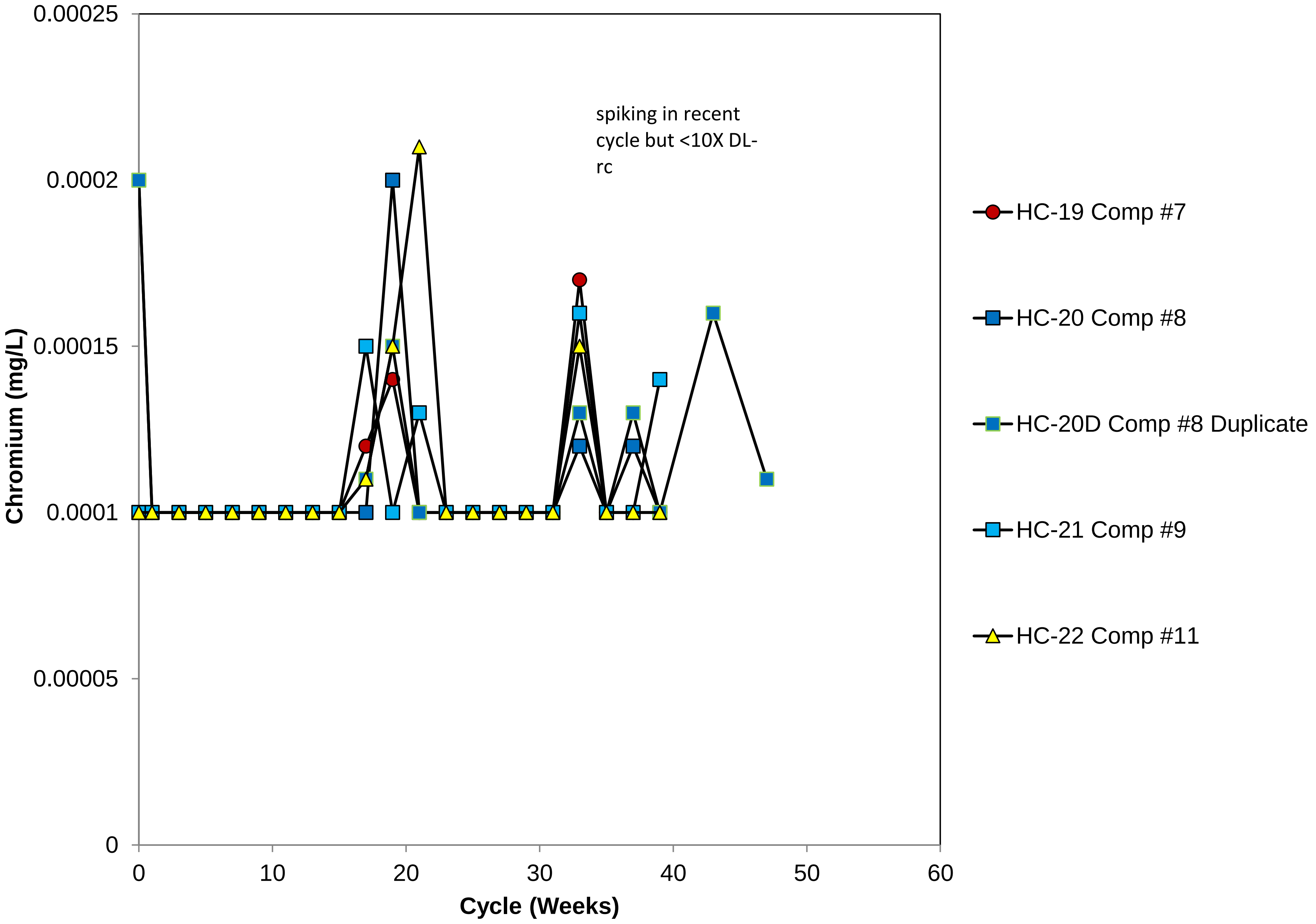

0.000035
0.00003
0.000025
0.00002
0.000015
0.00001
0.000005
0
Bismuth (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

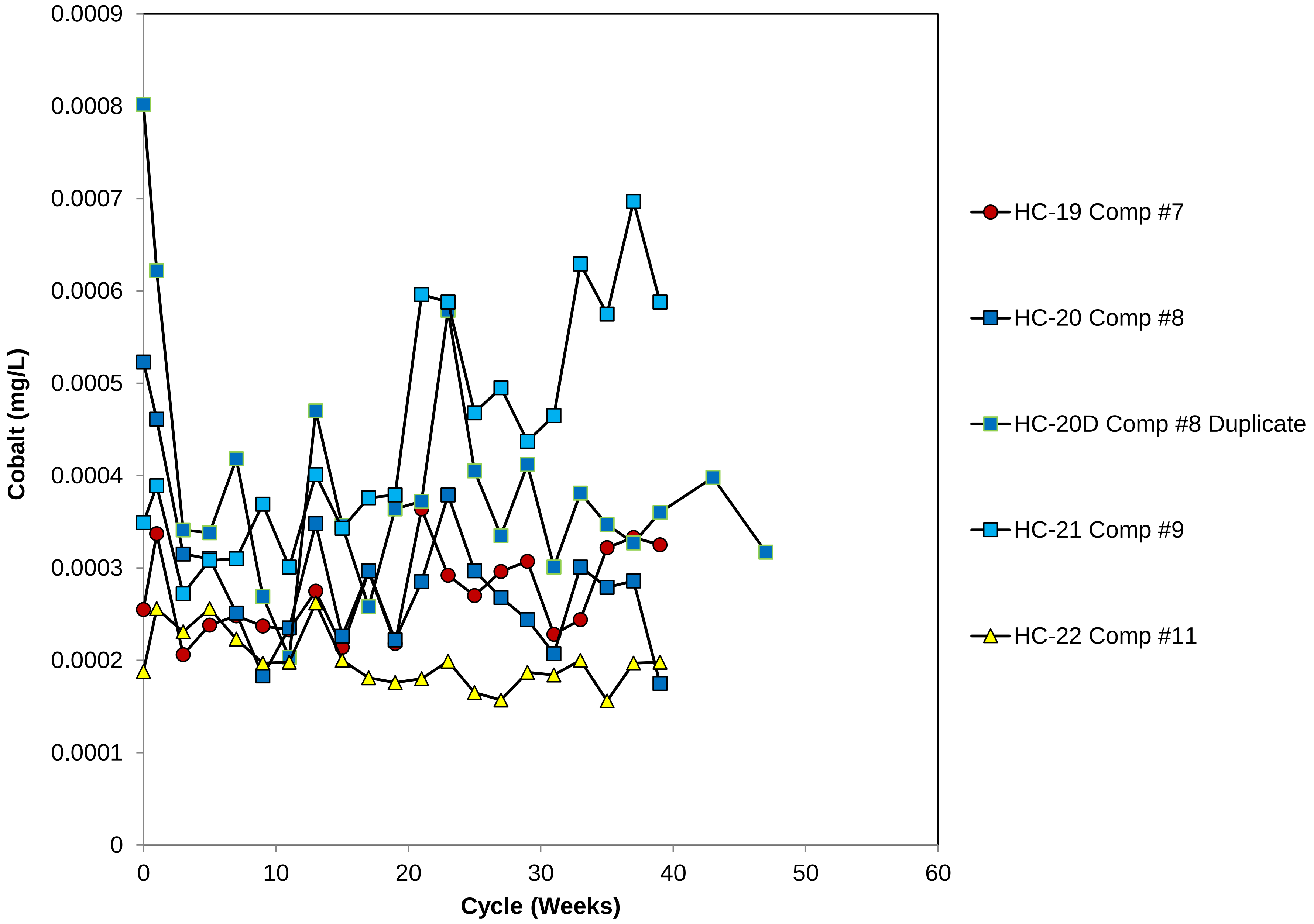

0.06
0.05
0.04
0.03
0.02
0.01
0
Boron (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

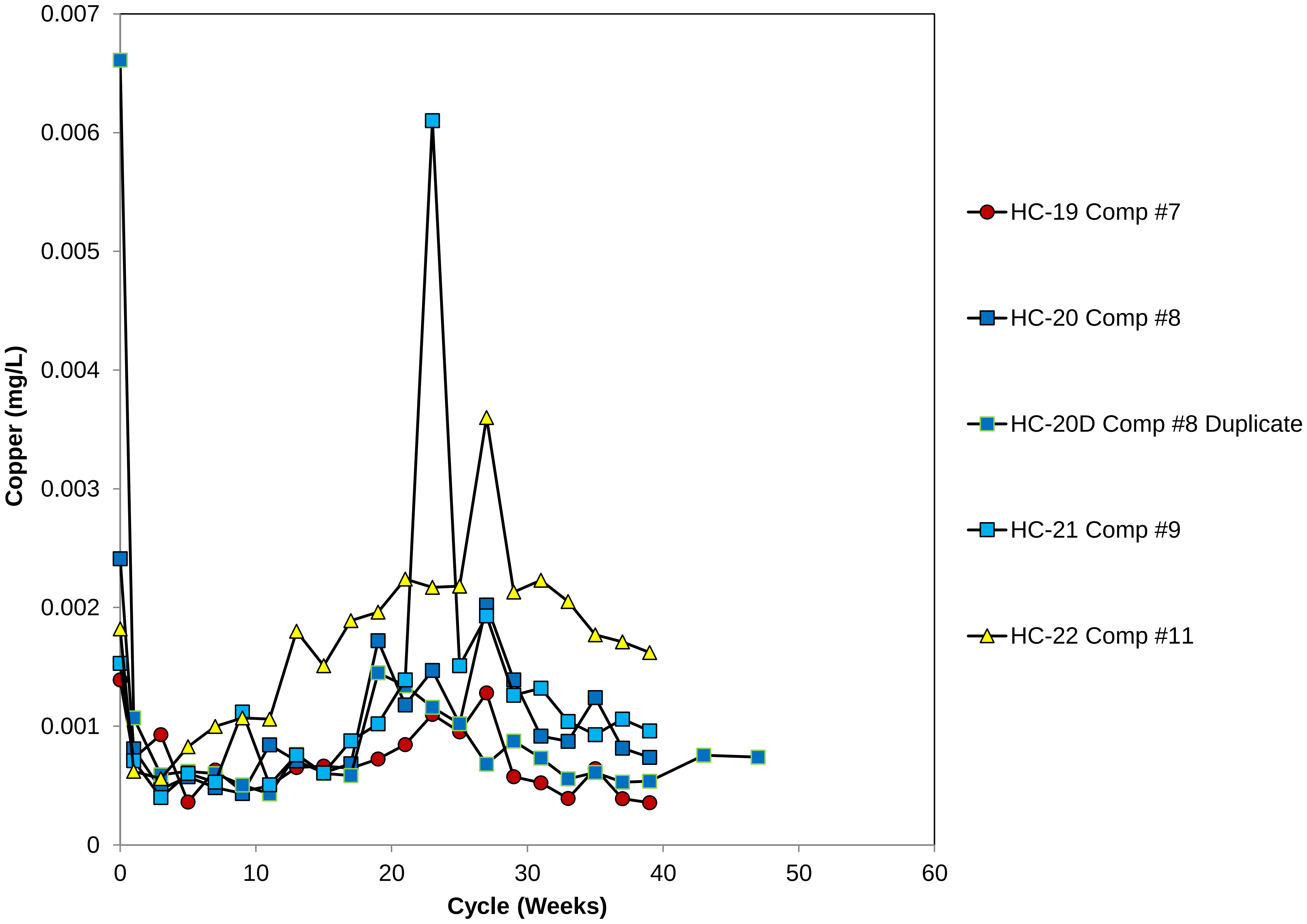

0.0001
0.00009
0.00008
0.00007
0.00006
0.00005
0.00004
0.00003
0.00002
0.00001
0
Cadmium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

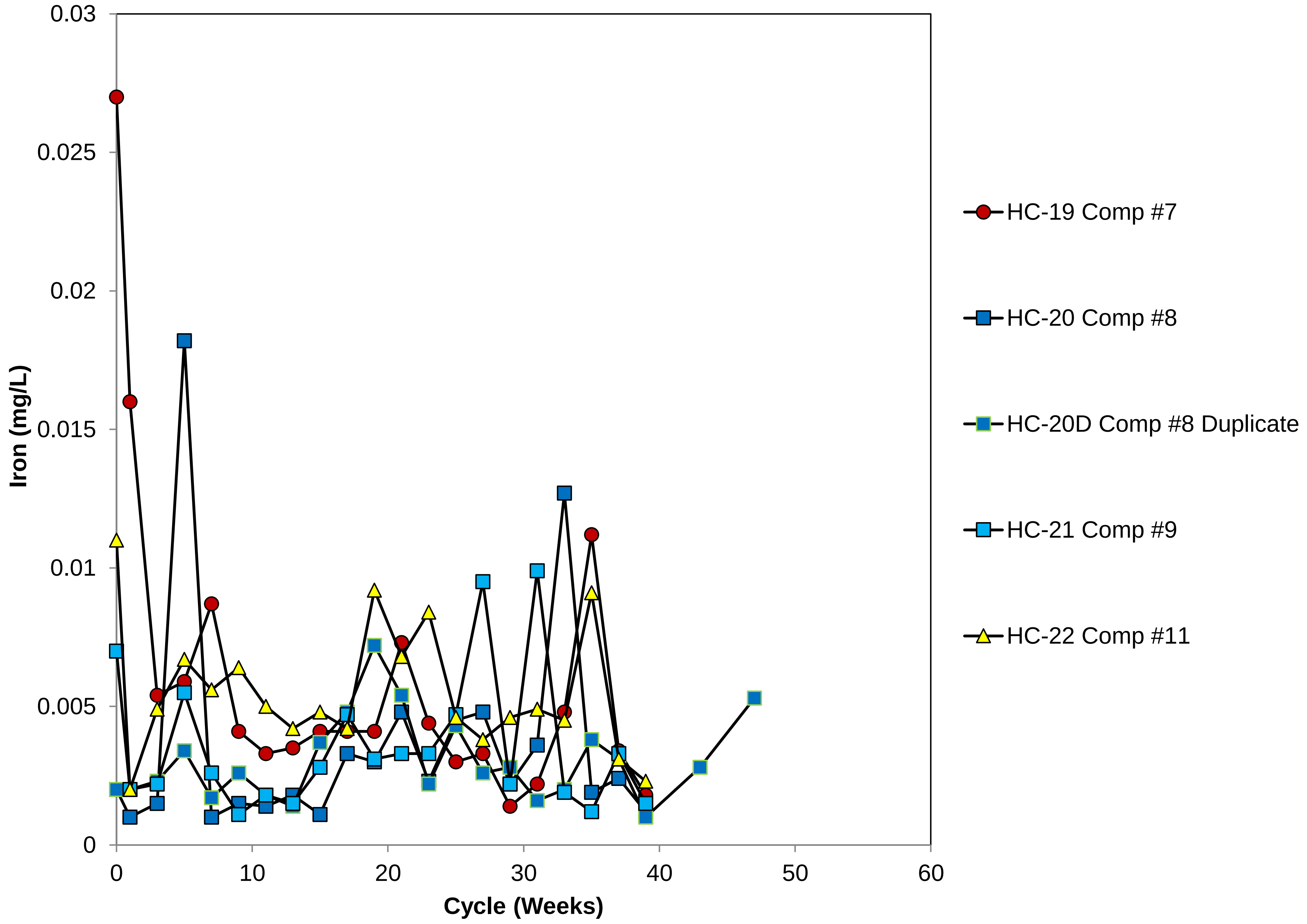

20
18
16
14
12
10
8
6
4
2
0
Calcium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11


Cesium (mg/L)
0.002
0.0018
0.0016
0.0014
0.0012
0.001
0.0008
0.0006
0.0004
0.0002
0
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

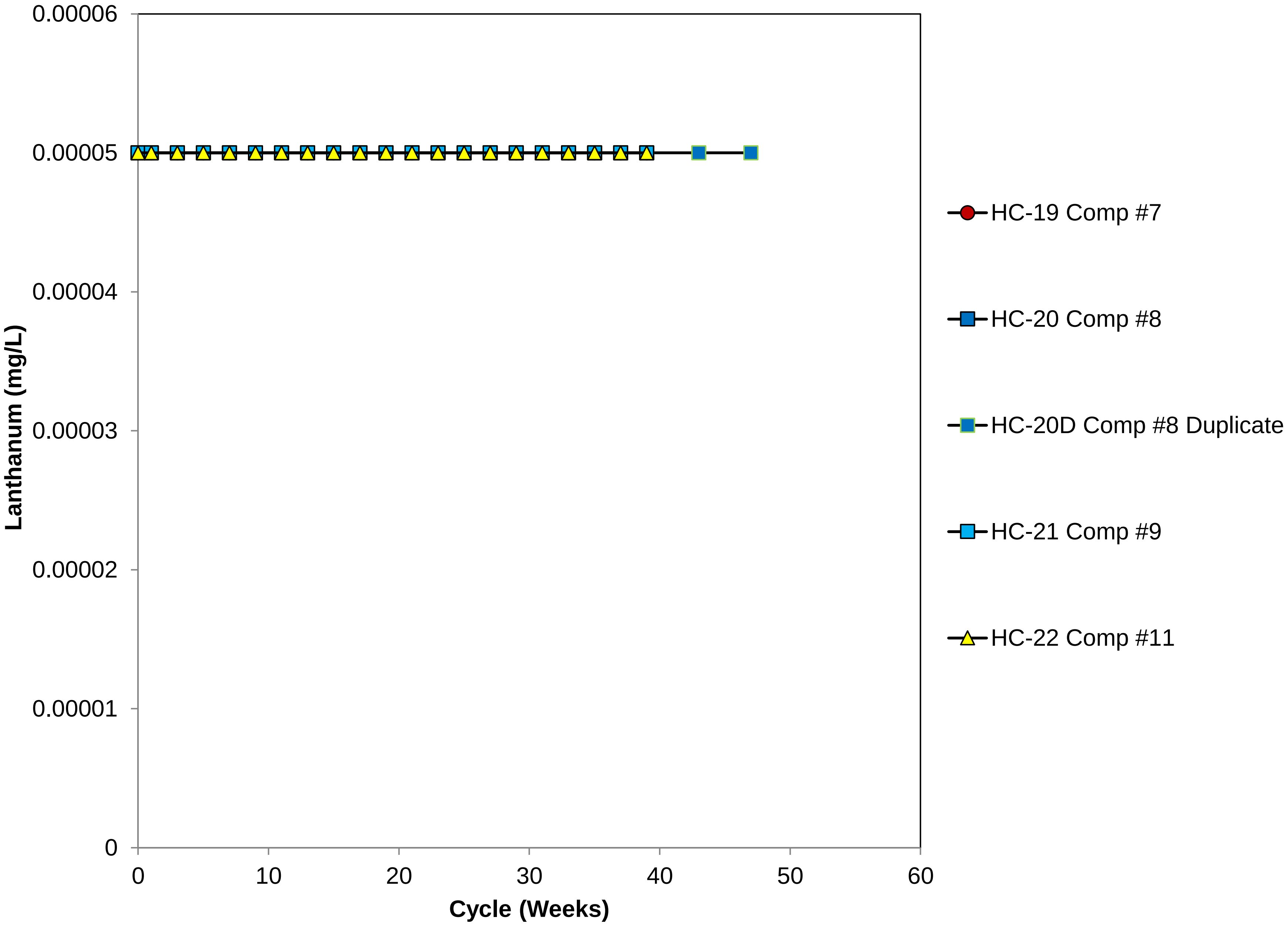

0.00025
0.0002
0.00015
0.0001
0.00005
0
Chromium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
spiking in recent cycle but <10X DL-rc
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

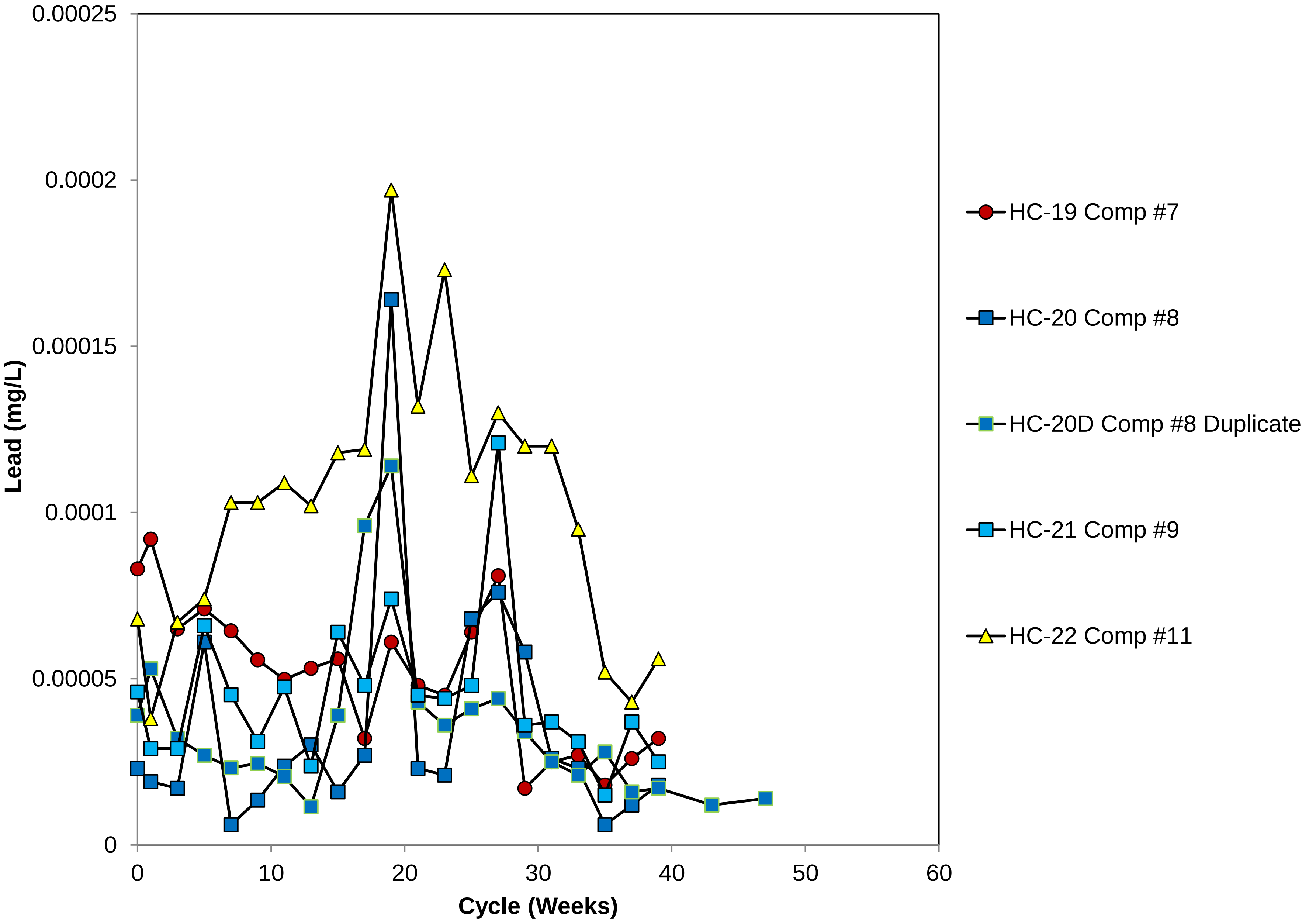

0.0009
0.0008
0.0007
0.0006
0.0005
0.0004
0.0003
0.0002
0.0001
0
Cobalt (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

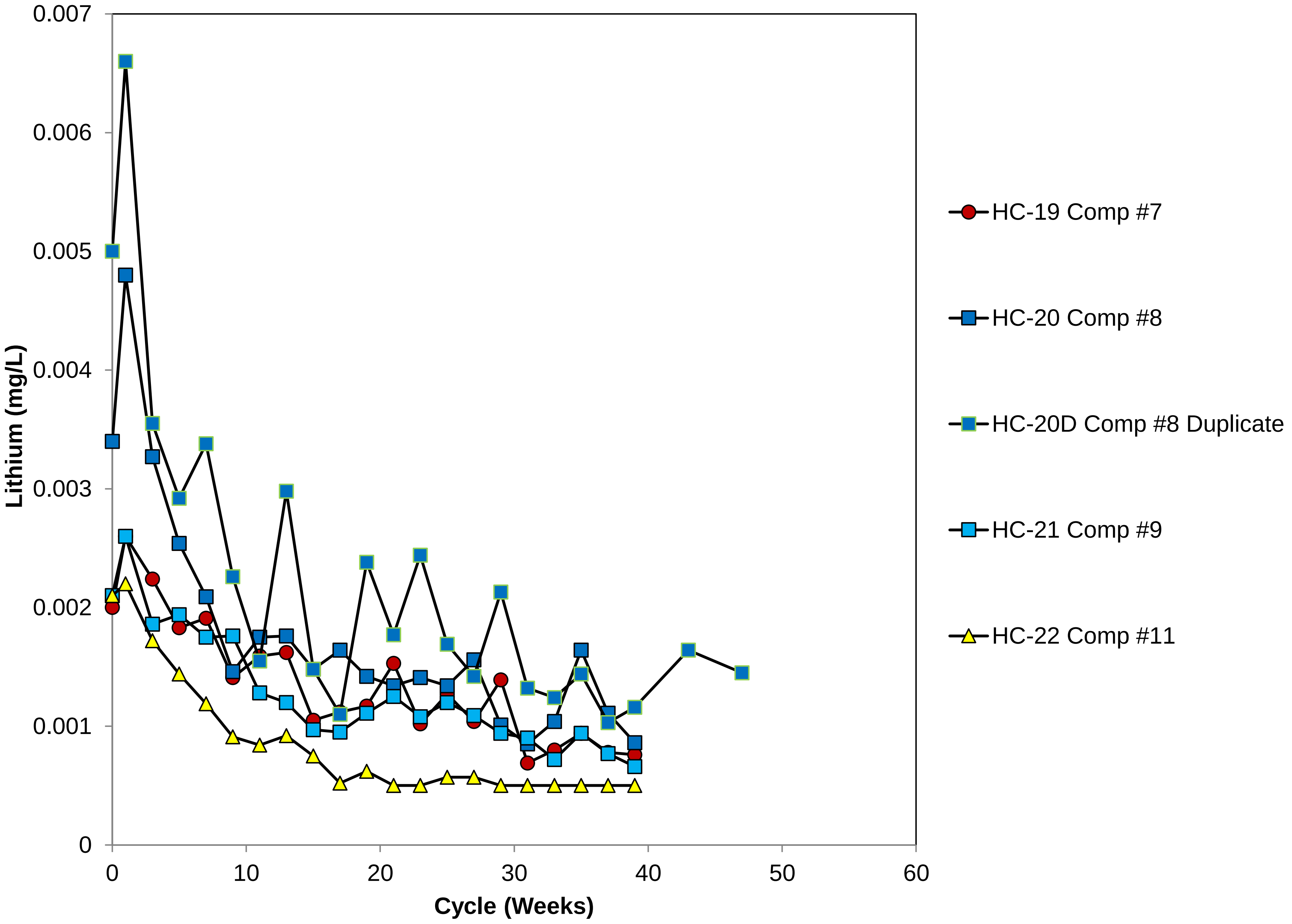

0.007
0.006
0.005
0.004
0.003
0.002
0.001
0
Copper (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

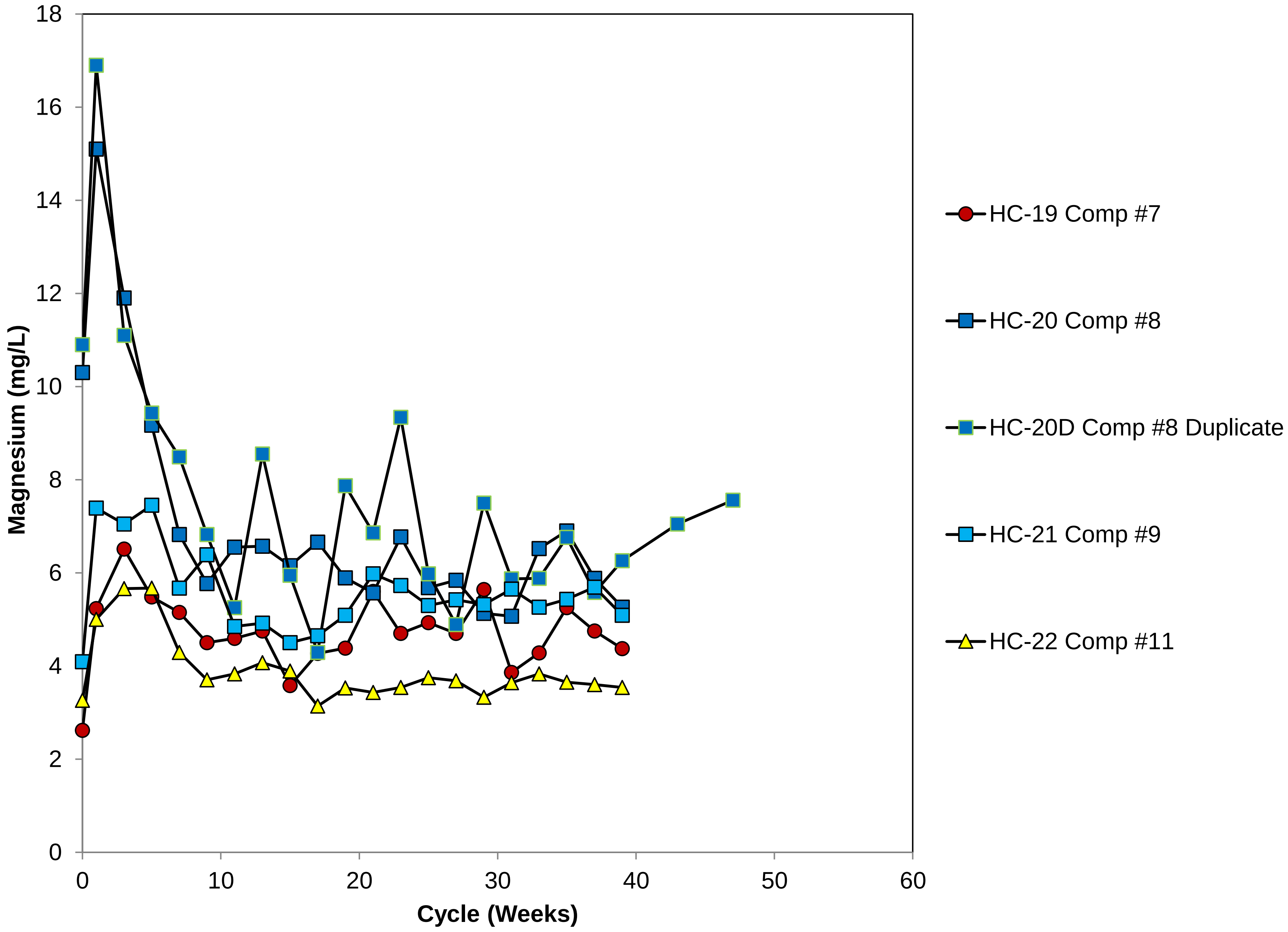


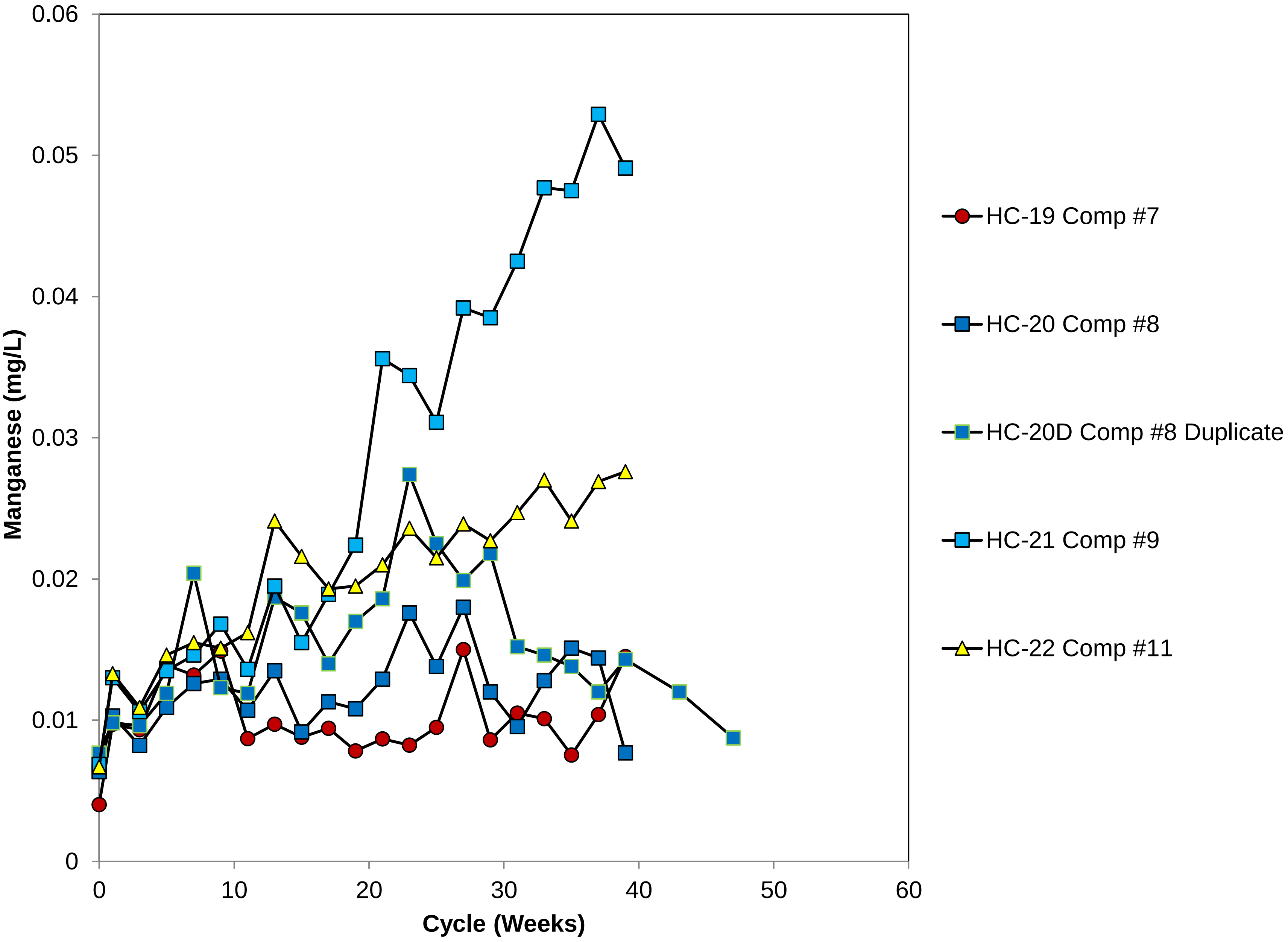
Iron (mg/L)
0.03
0.025
0.02
0.015
0.01
0.005
0
Cycle (Weeks)
0
10
20
30
40
50
60
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

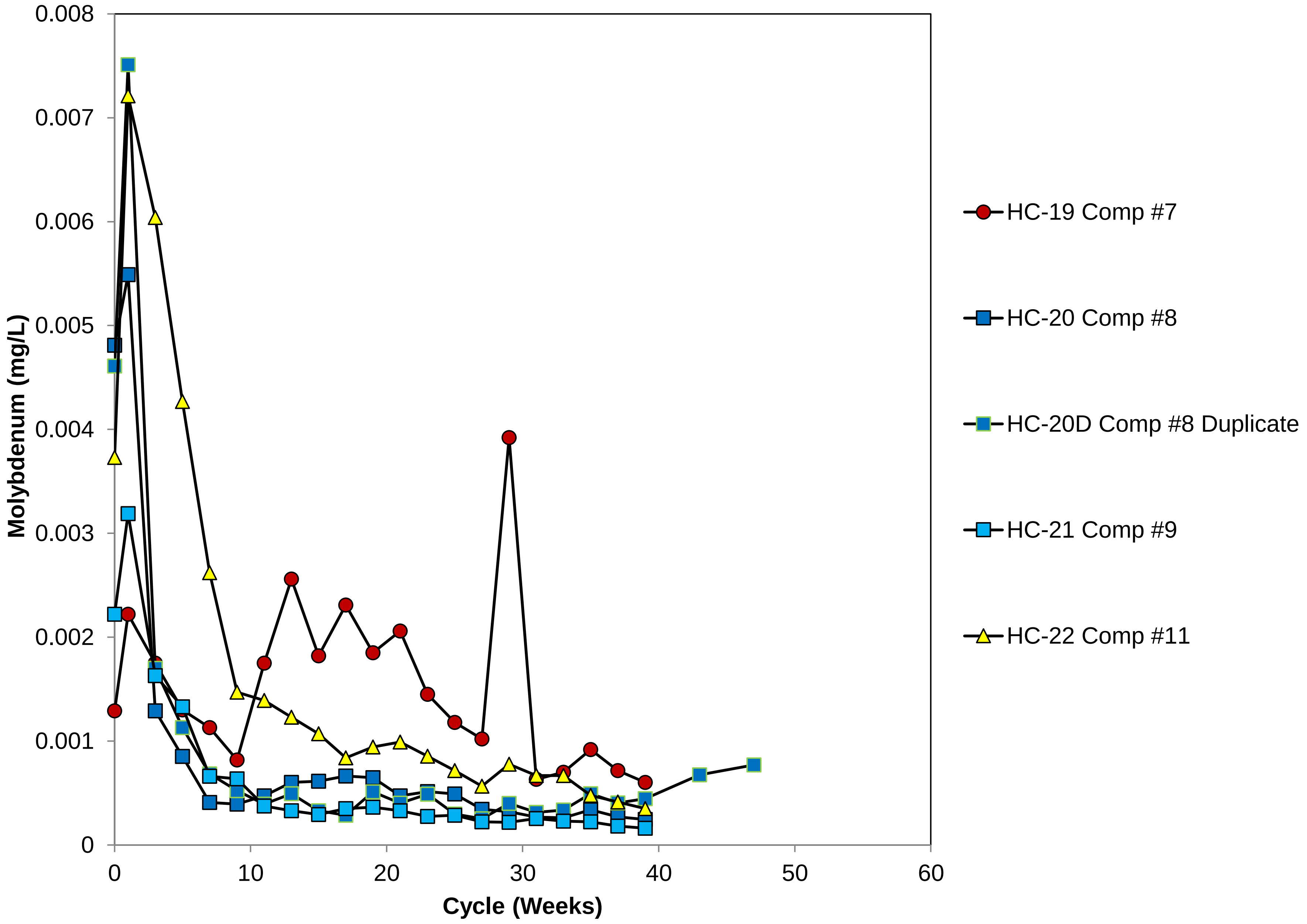

0.00006
0.00005
0.00004
0.00003
0.00002
0.00001
0
Lanthanum (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

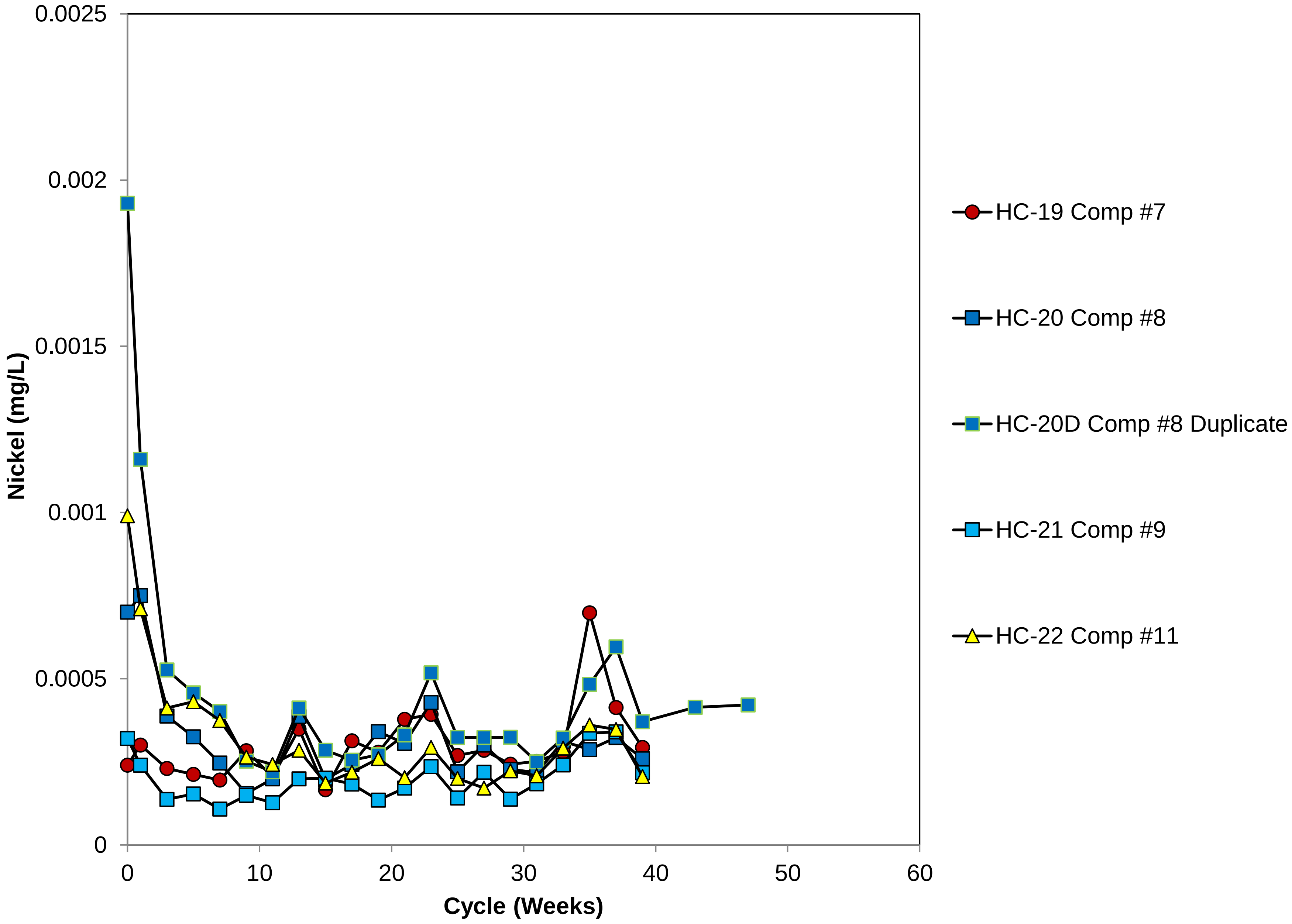

0.00025
0.0002
0.00015
0.0001
0.00005
0
Lead (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

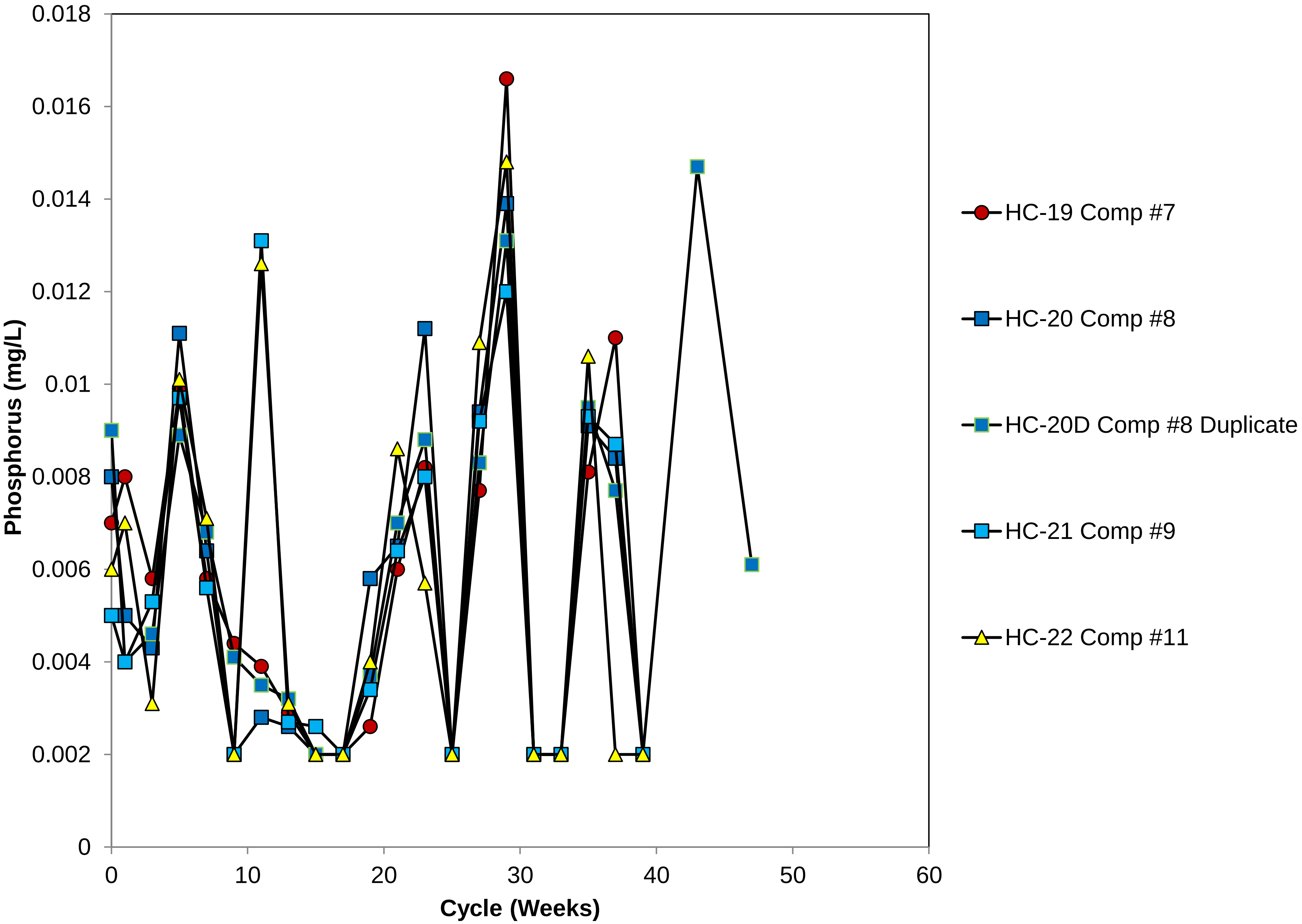

0.007
0.006
0.005
0.004
0.003
0.002
0.001
0
Lithium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

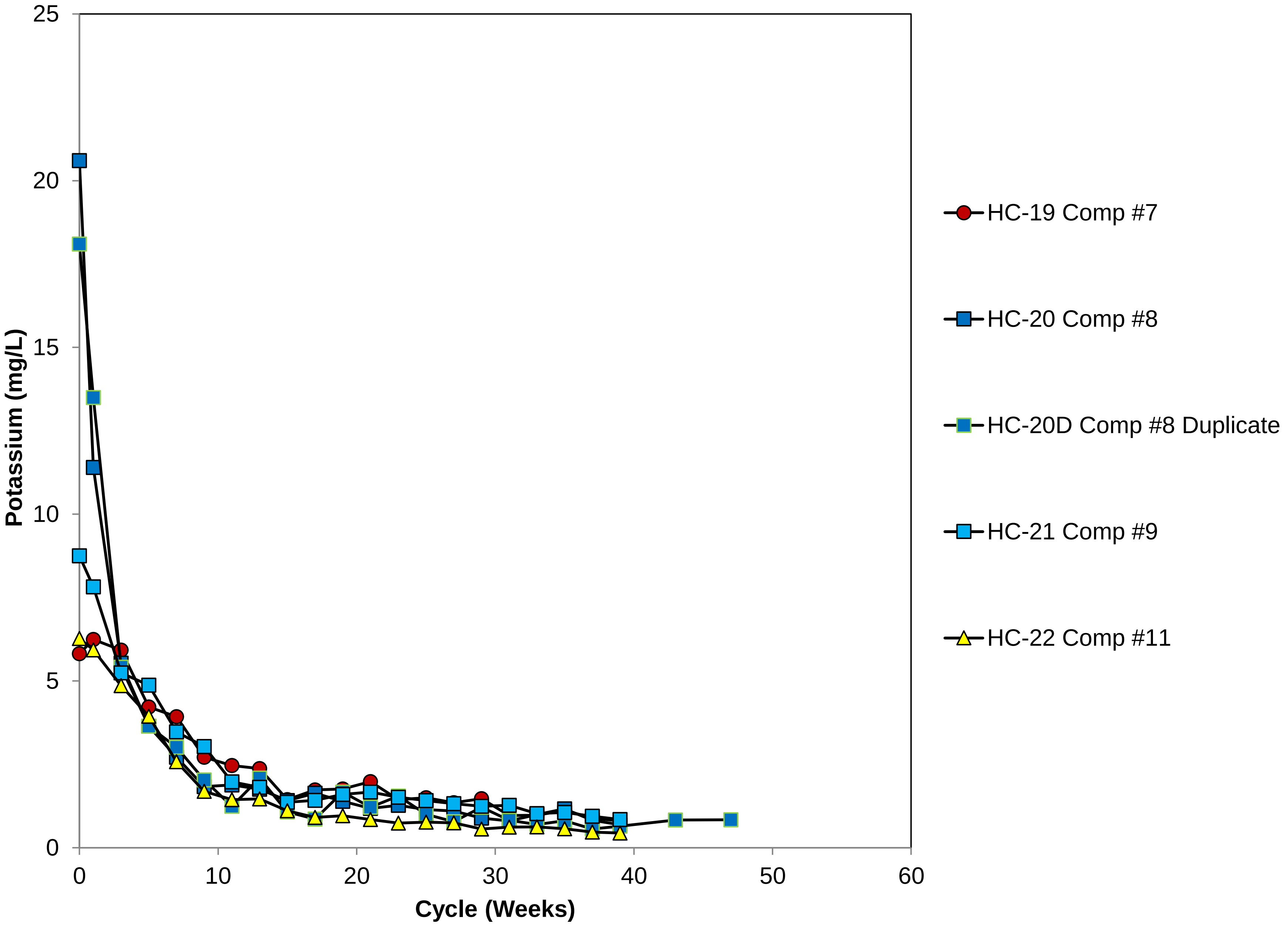

Magnesium (mg/L)
0
2
4
6
8
10
12
14
16
18
Cycle (Weeks)
0
10
20
30
40
50
60
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

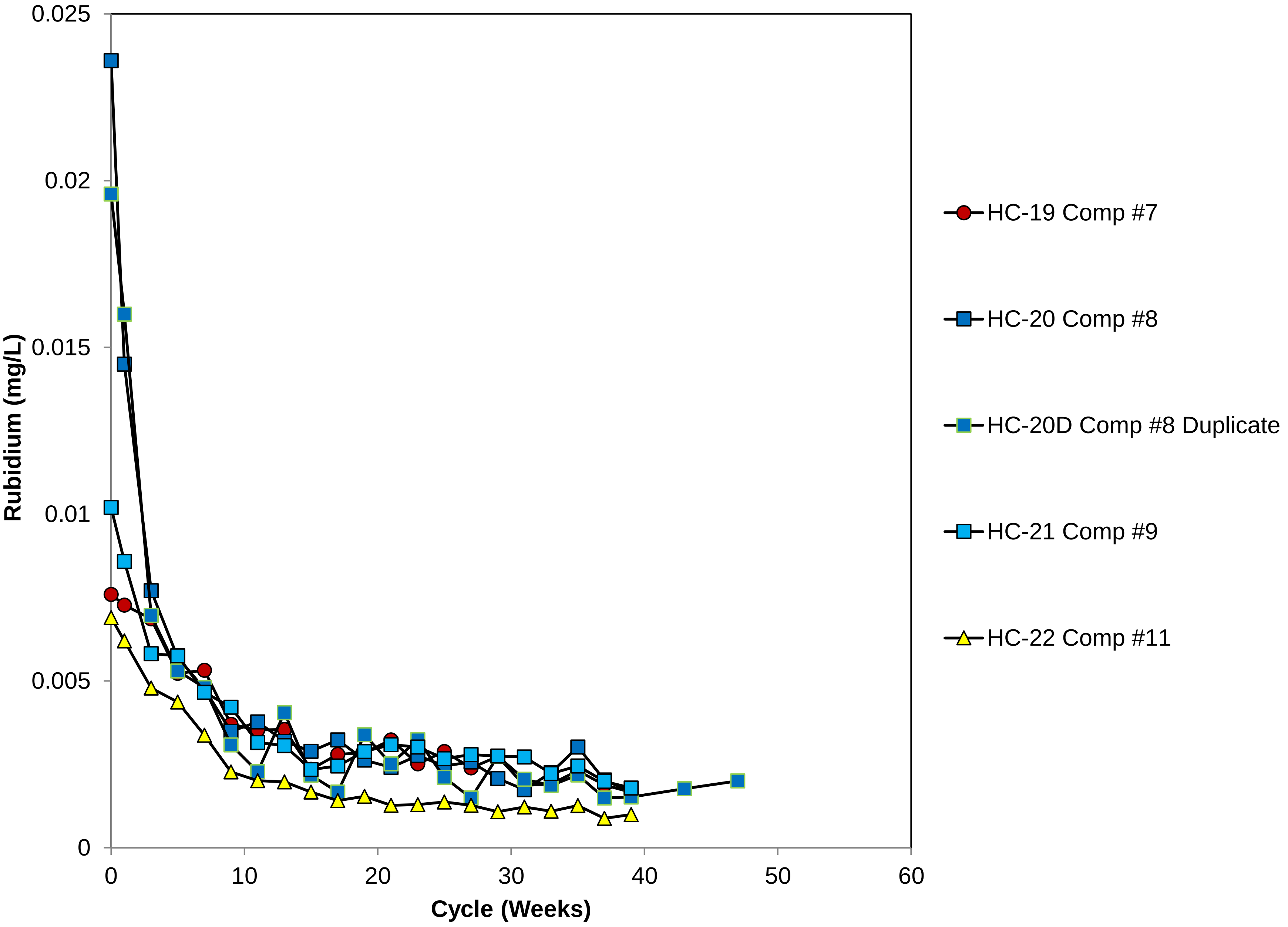

0.06
0.05
0.04
0.03
0.02
0.01
0
Manganese (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

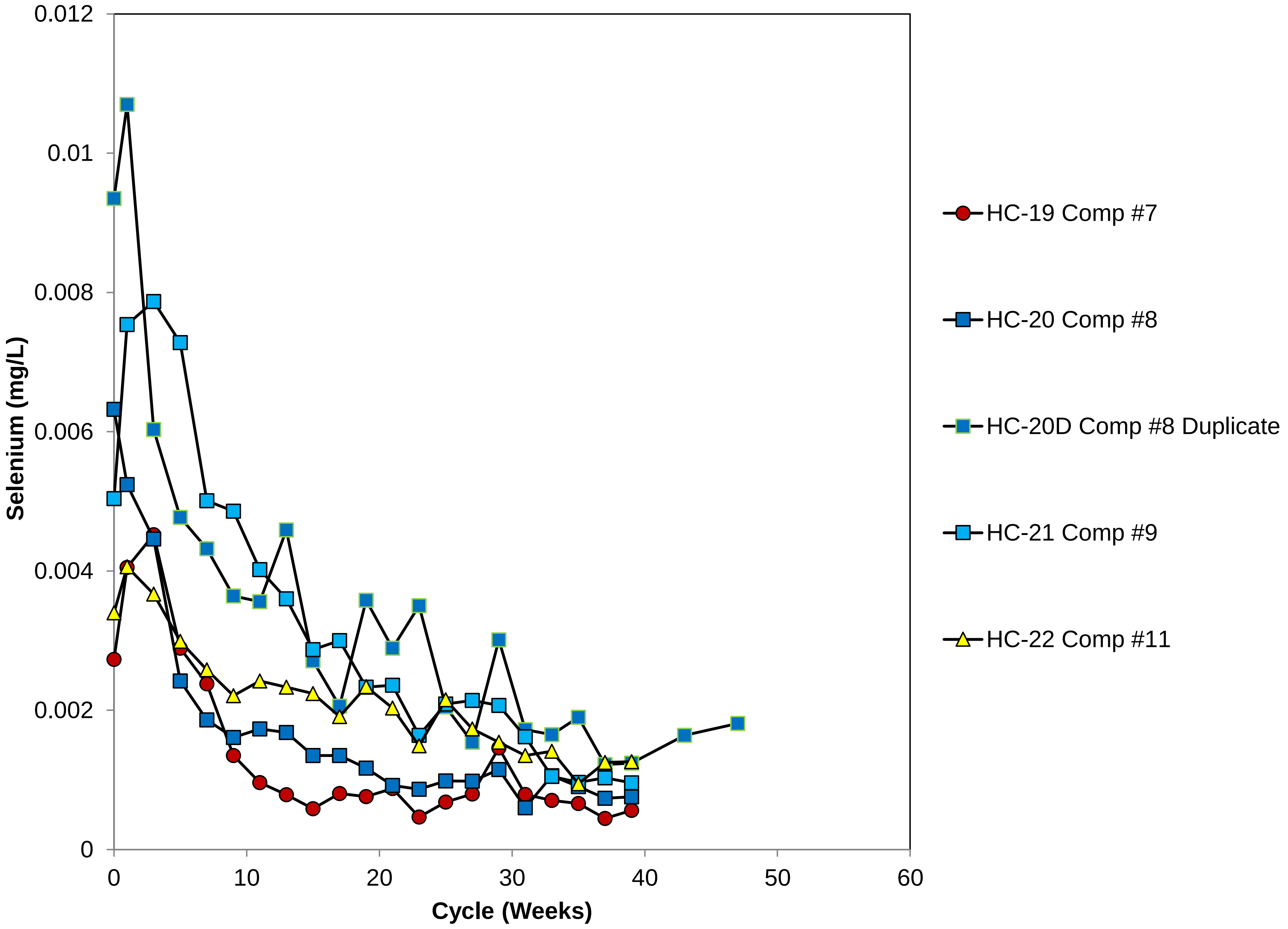

0.008
0.007
0.006
0.005
0.004
0.003
0.002
0.001
0
Molybdenum (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

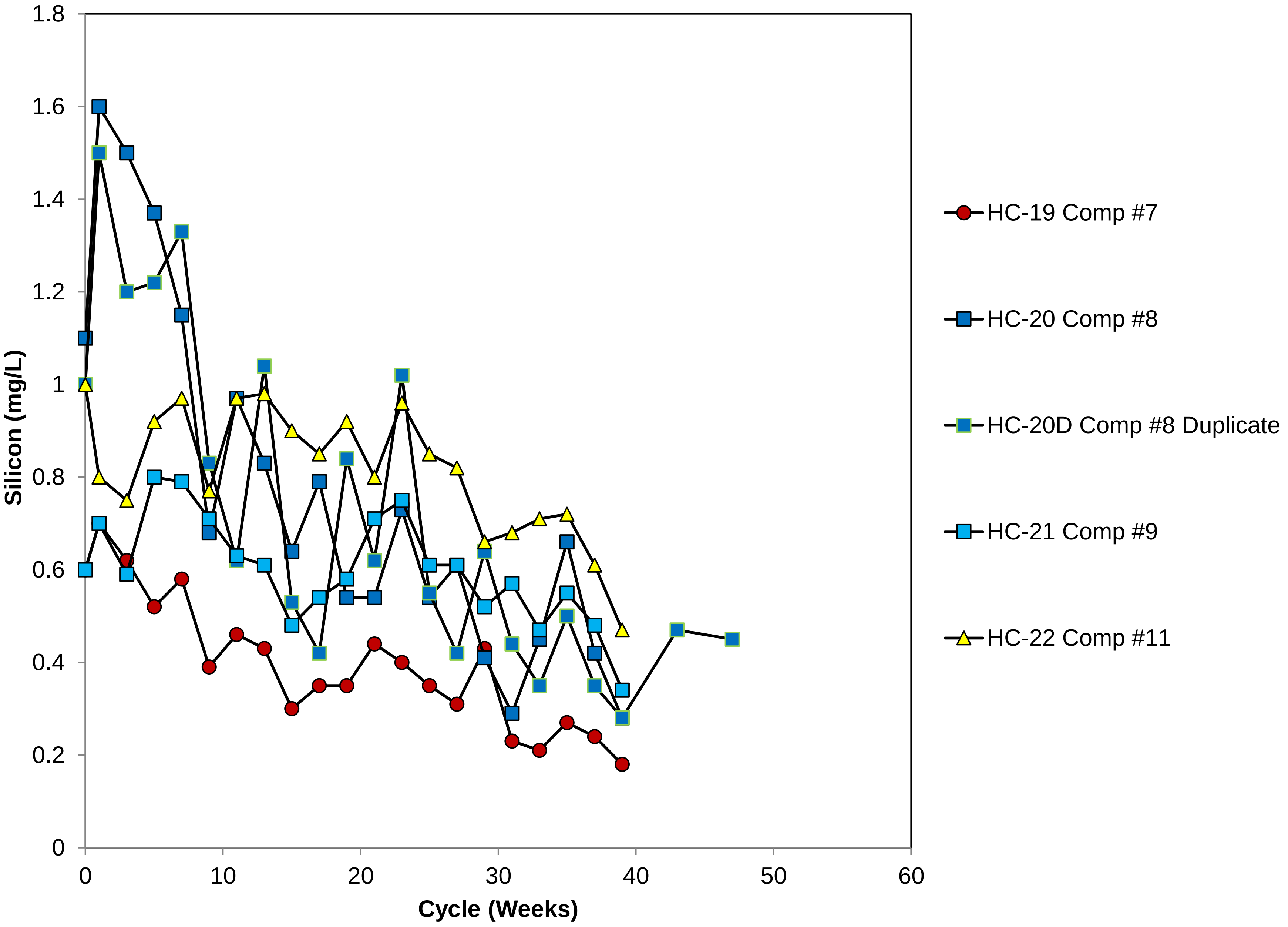

0.0025
0.002
0.0015
0.001
0.0005
0
Nickel (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

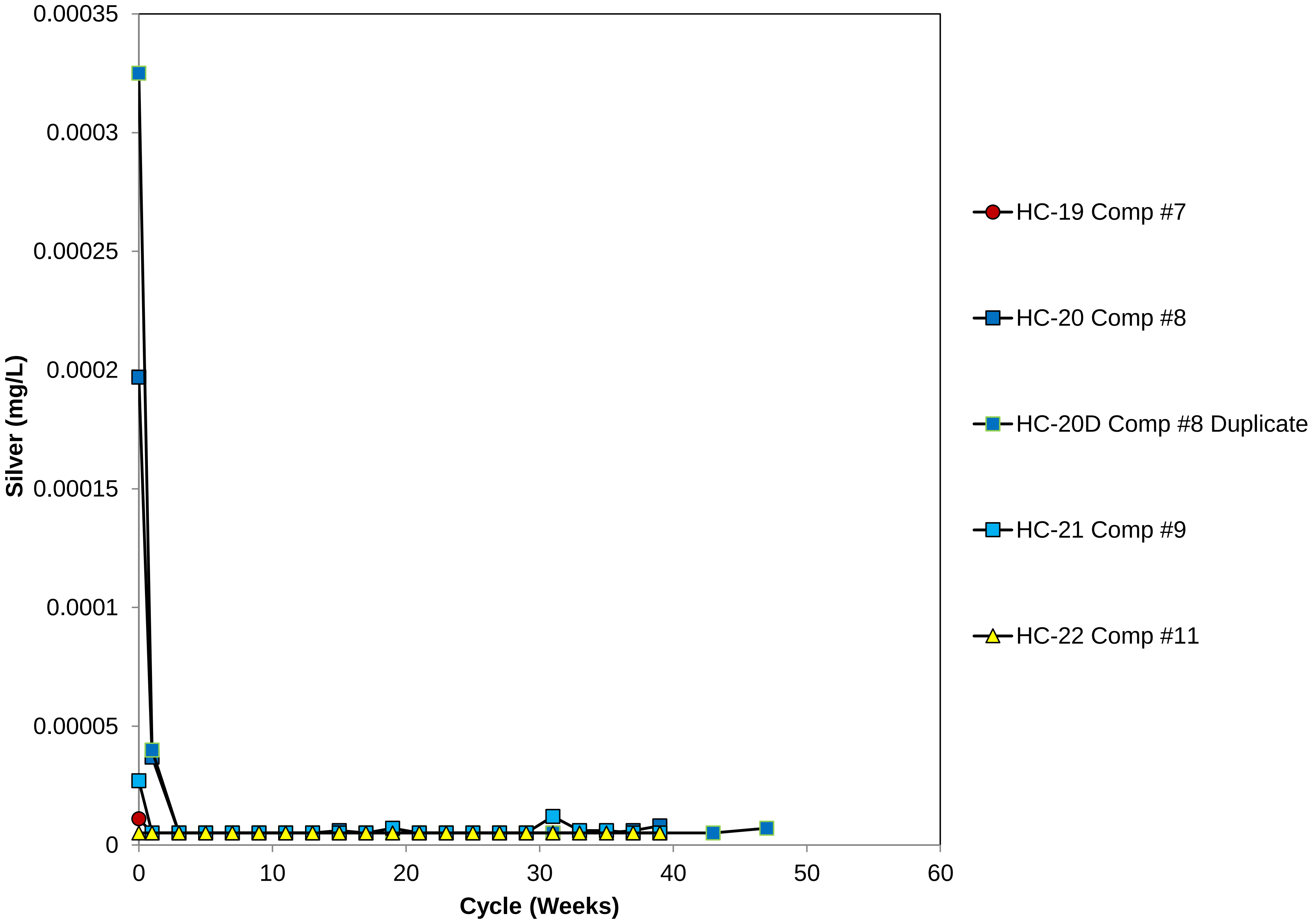

0.018
0.016
0.014
0.012
0.01
0.008
0.006
0.004
0.002
0
Phosphorus (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

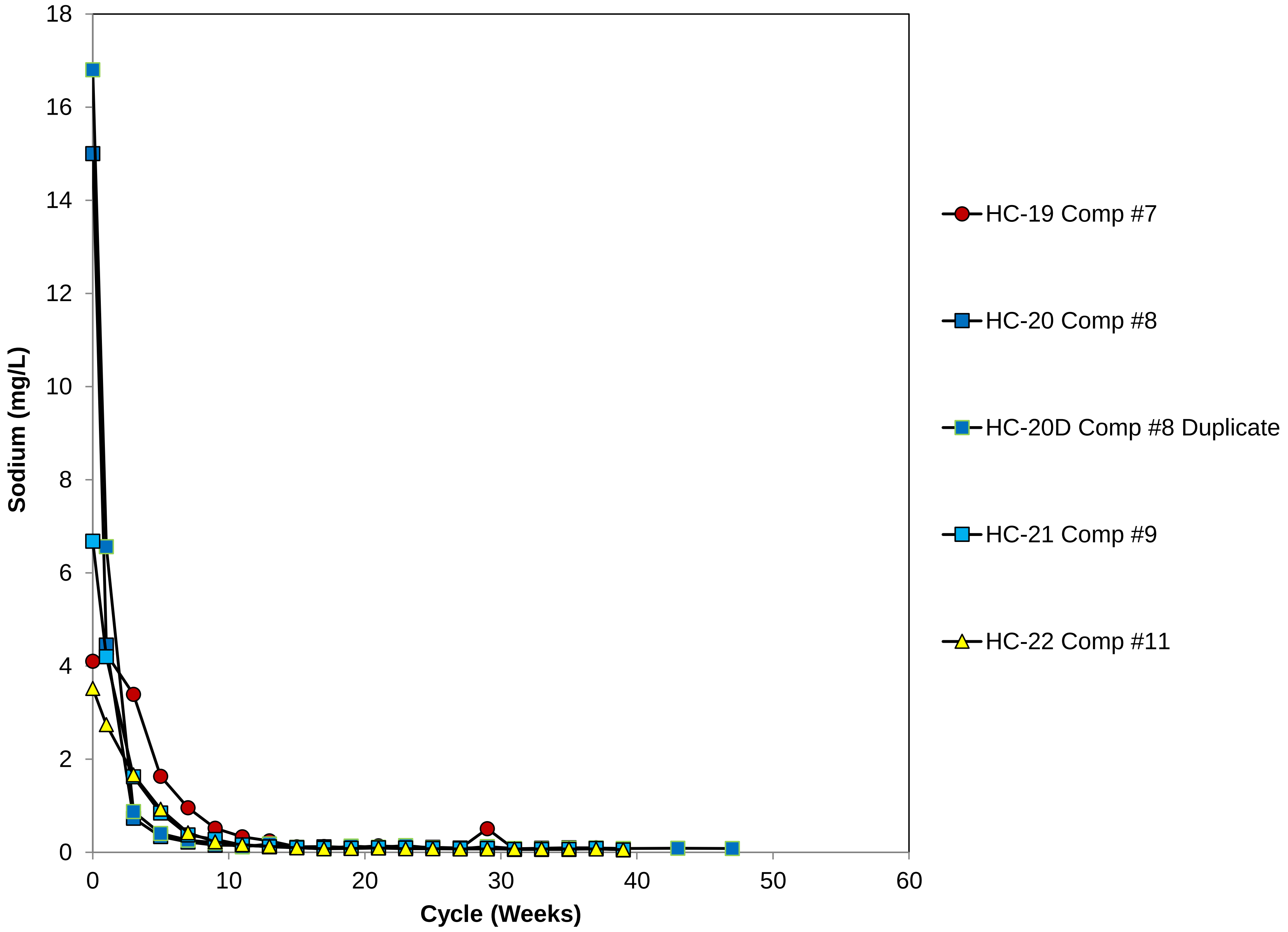

25
20
15
10
5
0
Potassium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

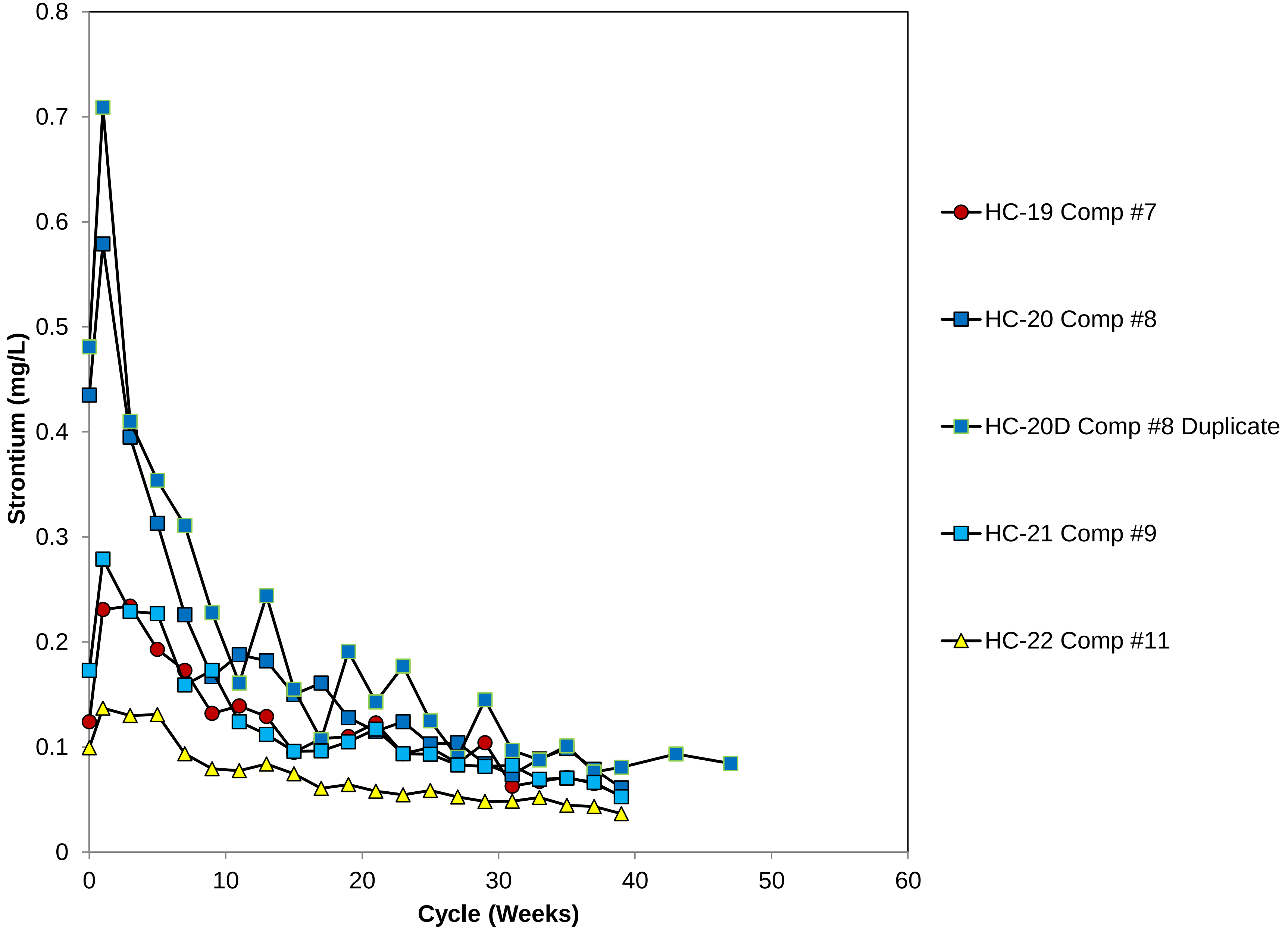

0.025
0.02
0.015
0.01
0.005
0
Rubidium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

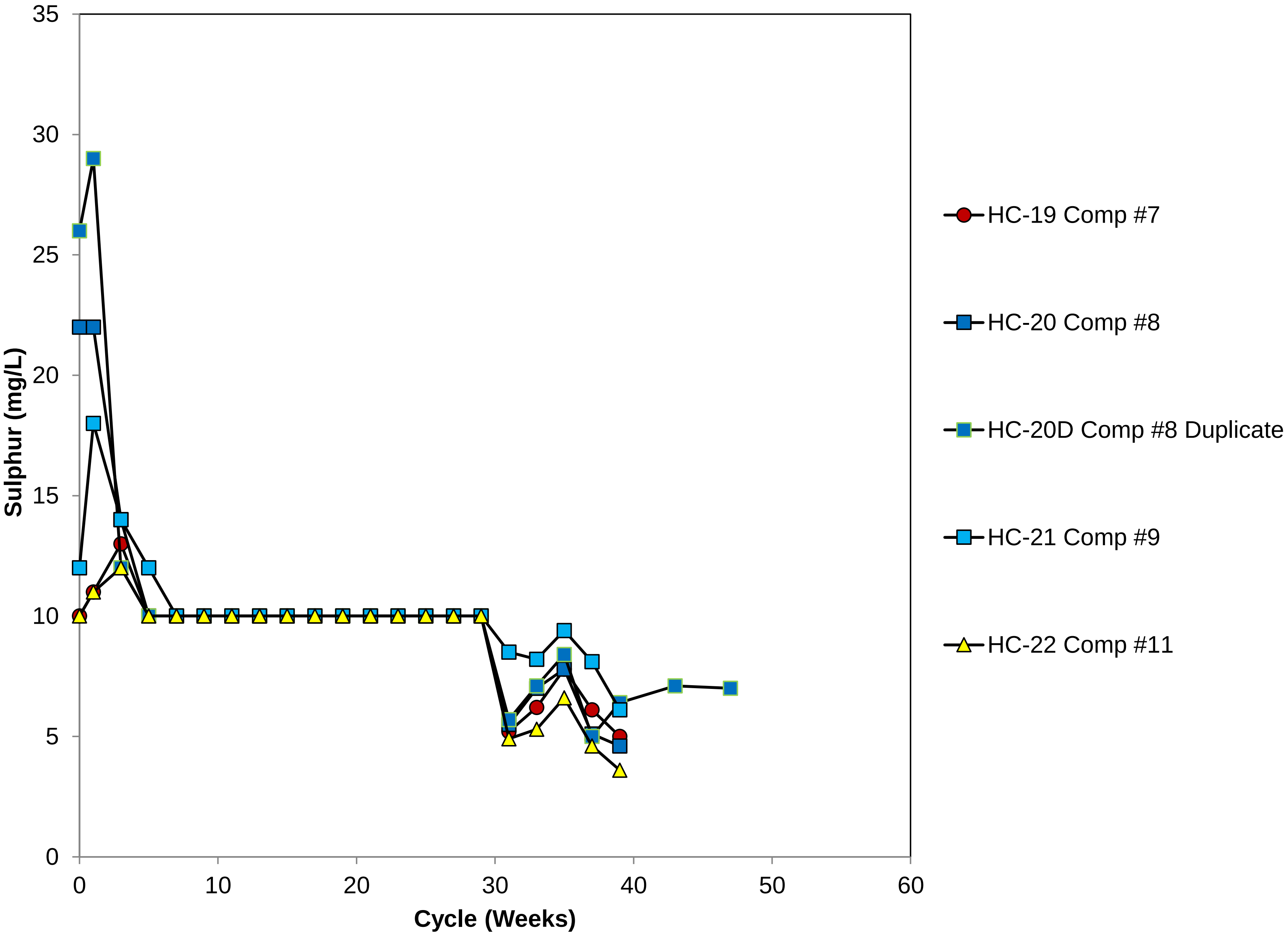

0.012
0.01
0.008
0.006
0.004
0.002
0
Selenium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

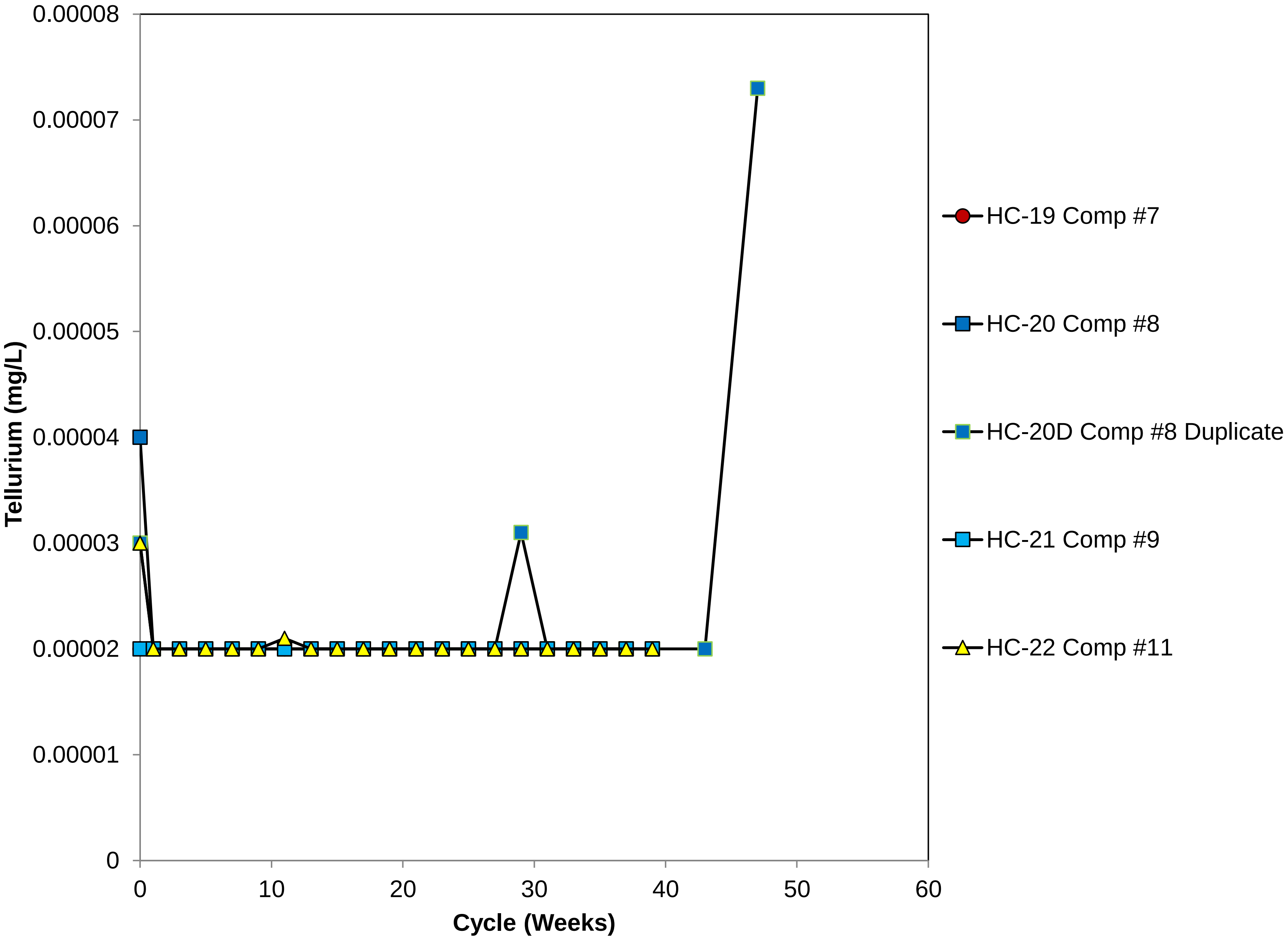

1.8
1.6
1.4
1.2
1
0.8
0.6
0.4
0.2
0
Silicon (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

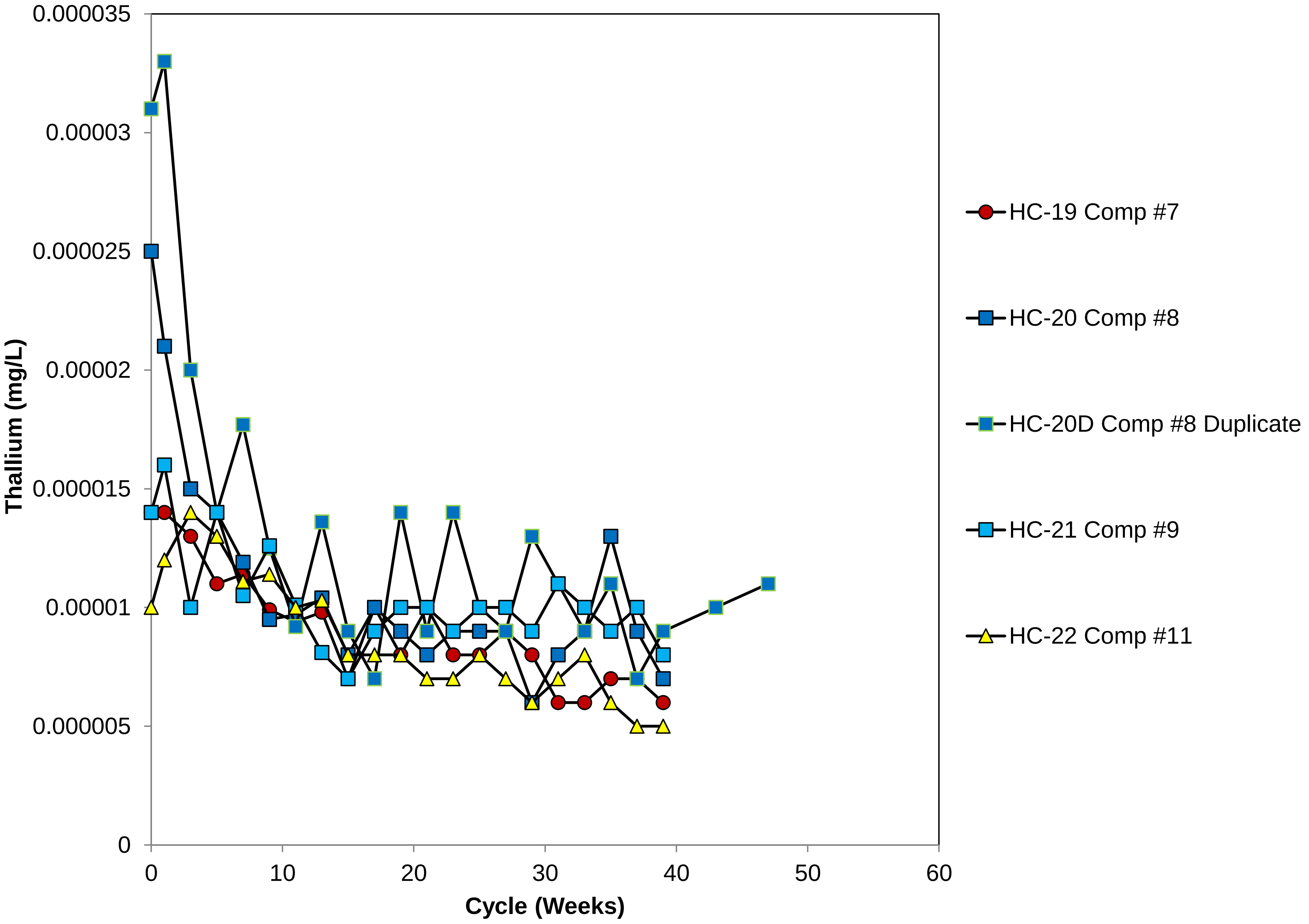

0.00035
0.0003
0.00025
0.0002
0.00015
0.0001
0.00005
0
Silver (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

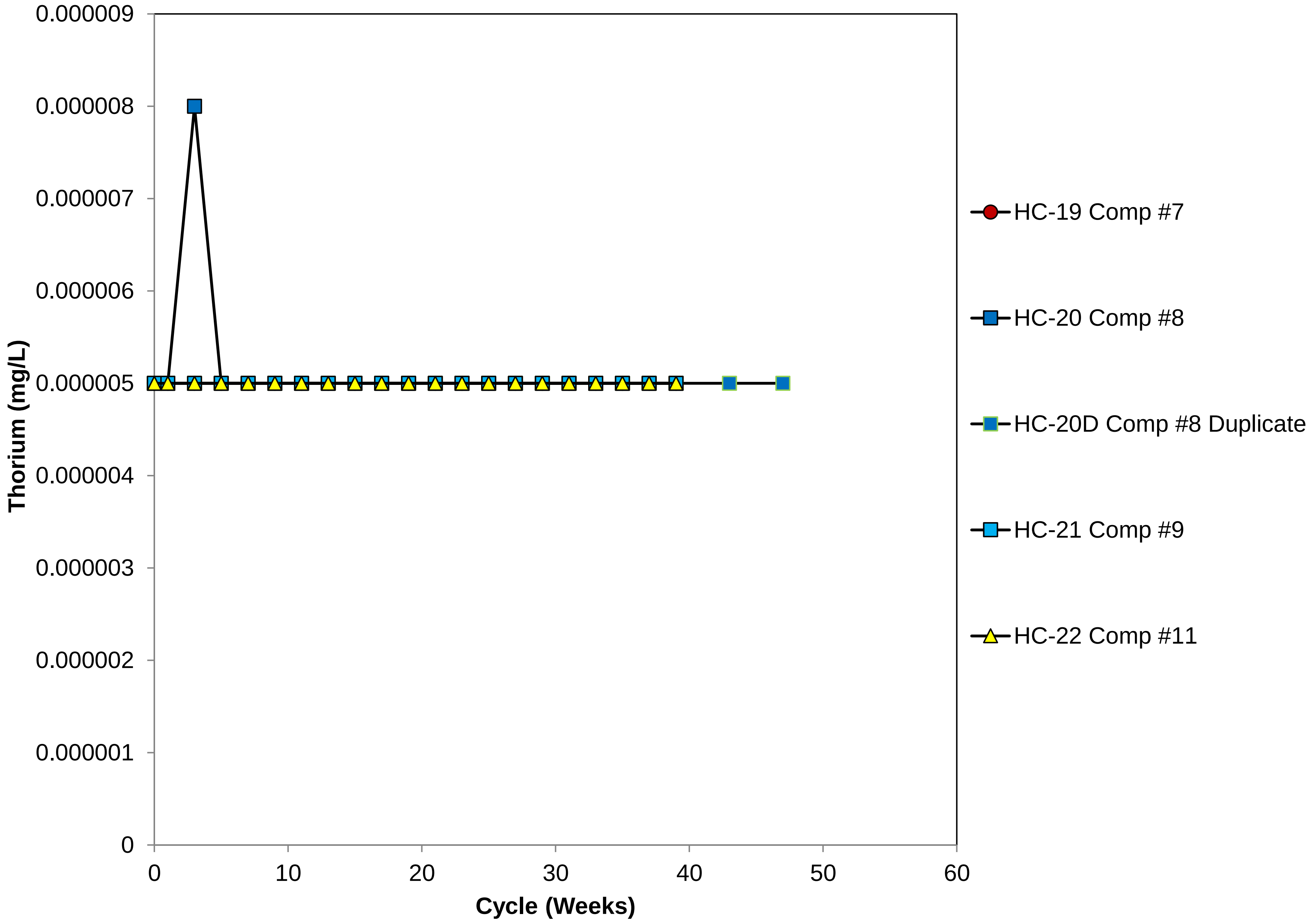

Sodium (mg/L)
0
2
4
6
8
10
12
14
16
18
Cycle (Weeks)
0
10
20
30
40
50
60
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

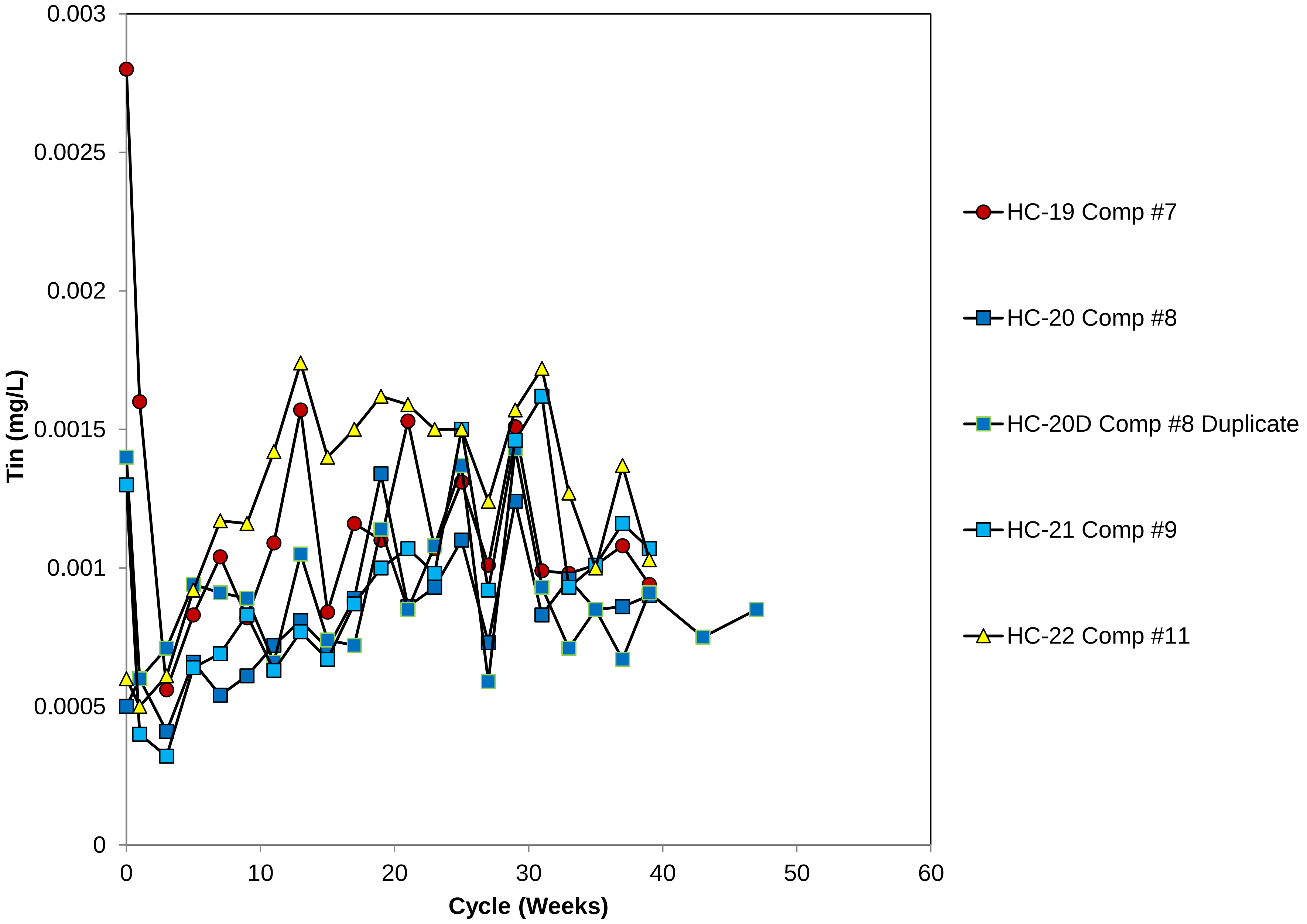

0.8
0.7
0.6
0.5
0.4
0.3
0.2
0.1
0
Strontium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

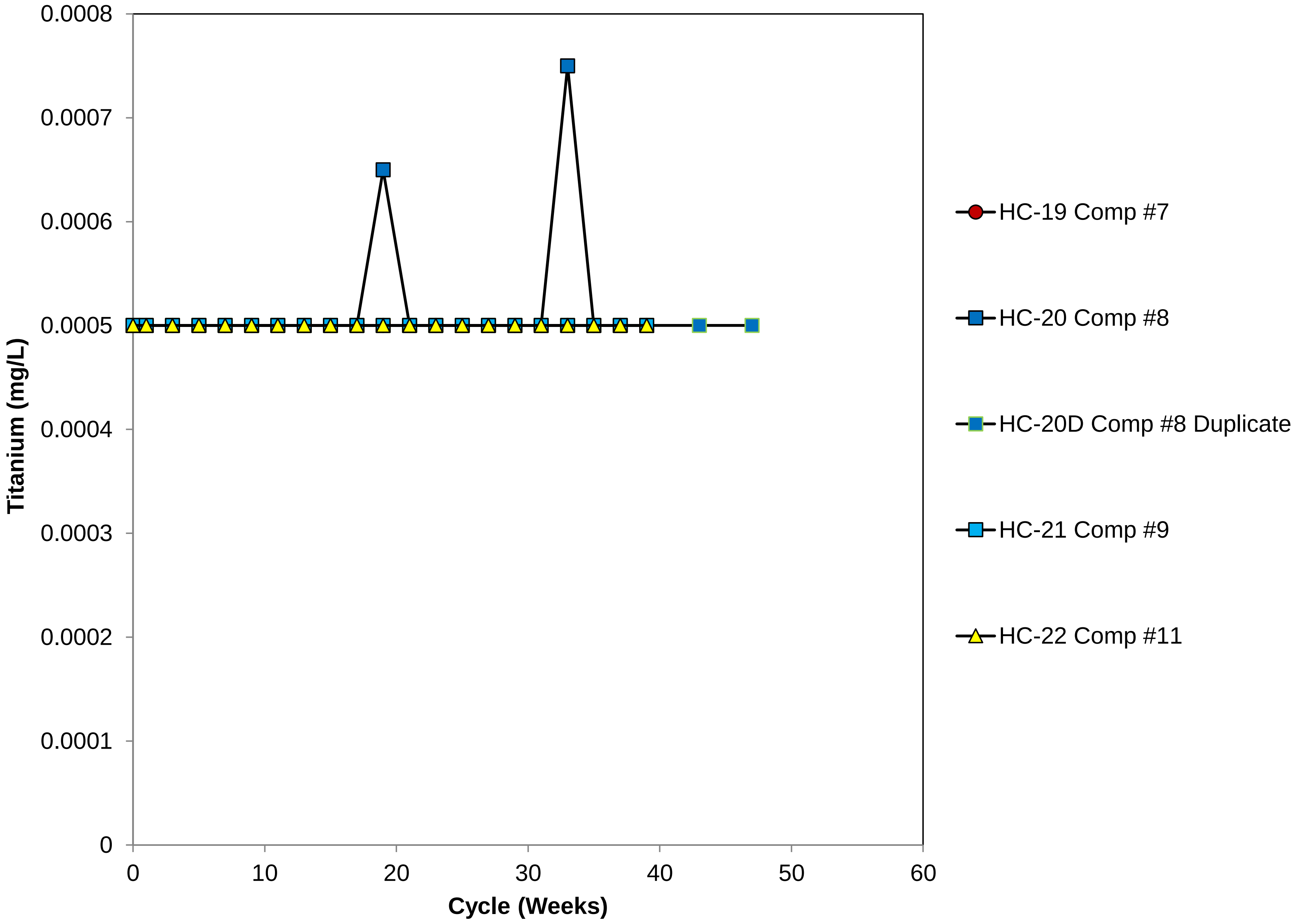

35
30
25
20
15
10
5
0
Sulphur (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

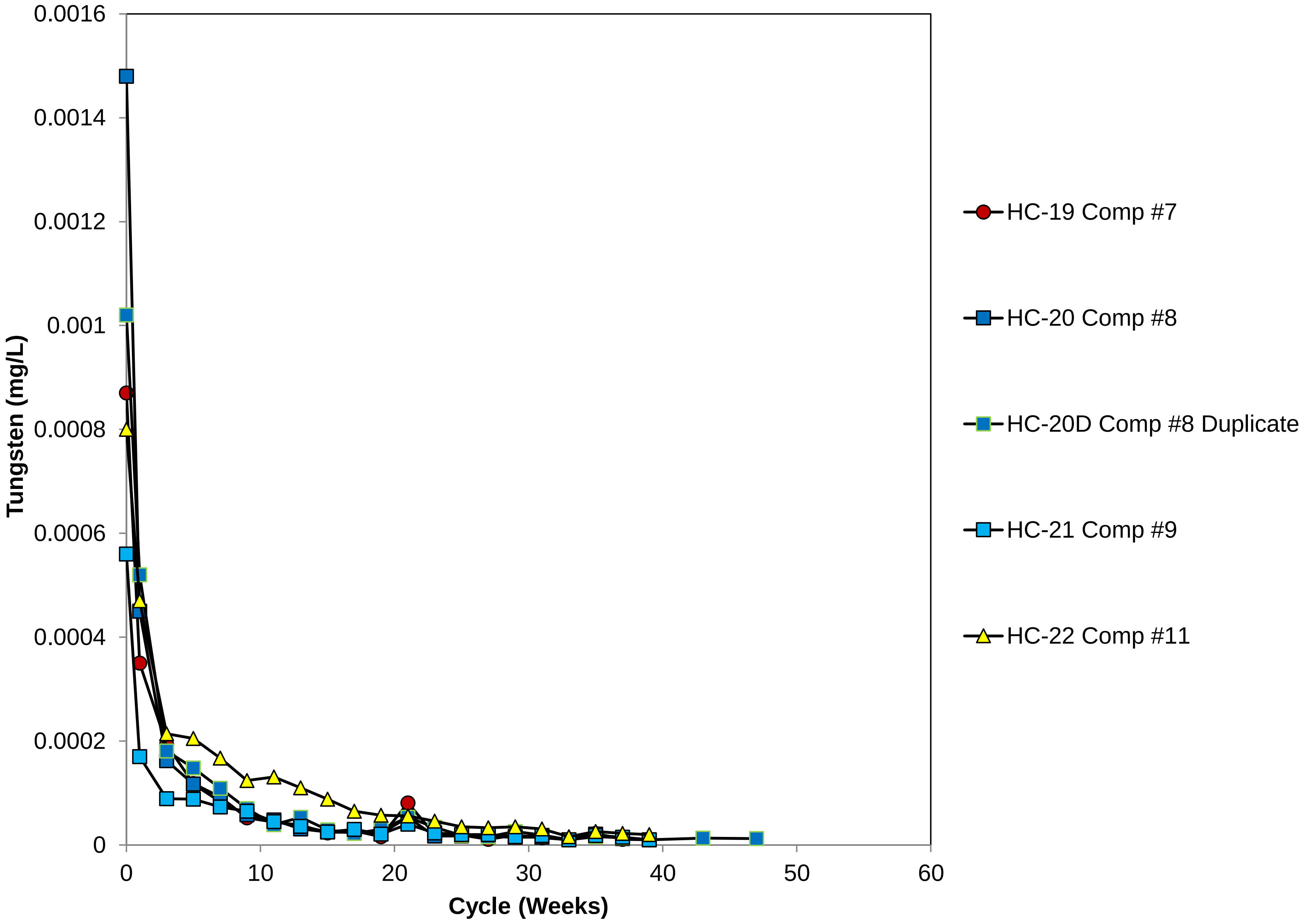

0.00008
0.00007
0.00006
0.00005
0.00004
0.00003
0.00002
0.00001
0
Tellurium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

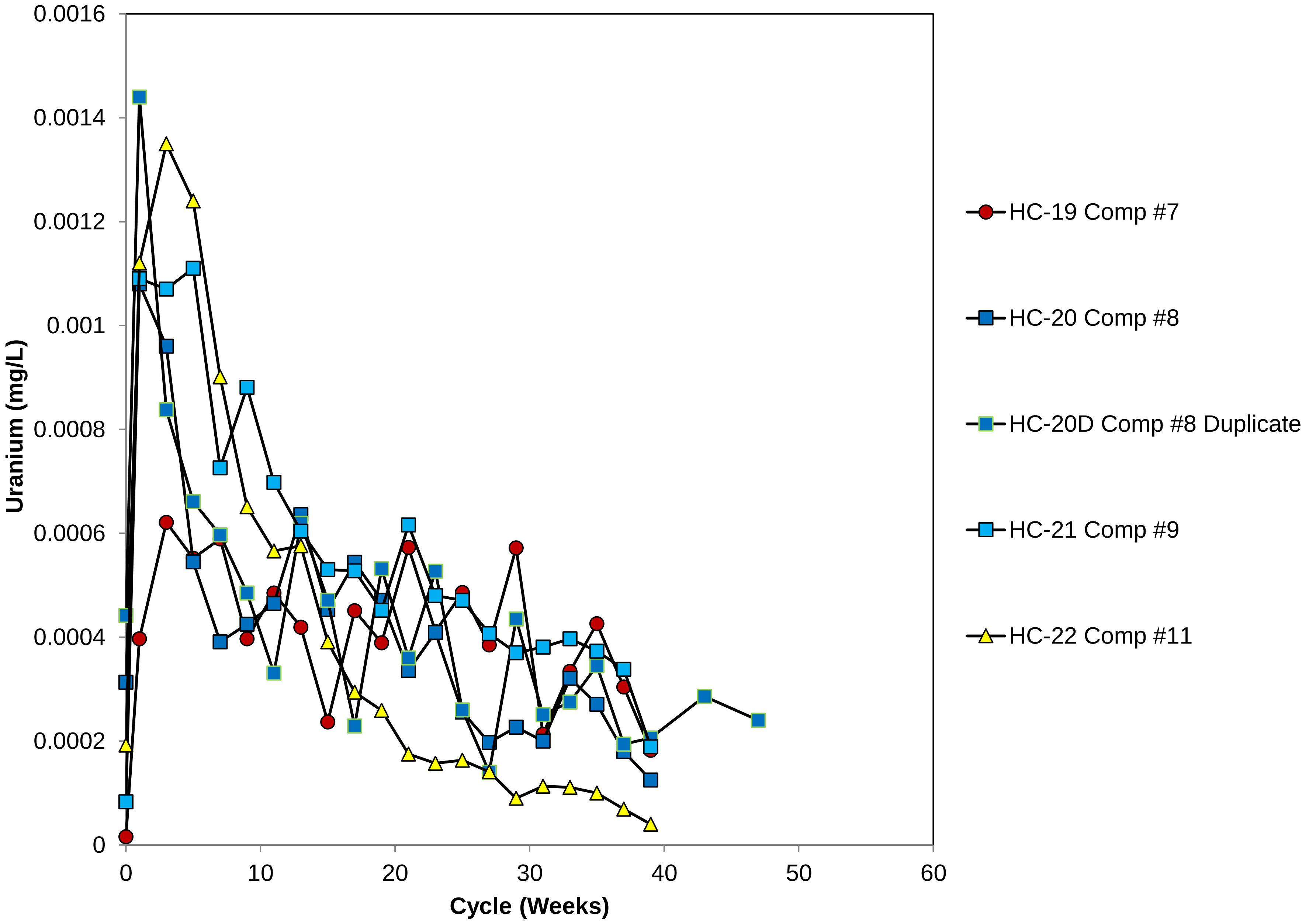

0.000035
0.00003
0.000025
0.00002
0.000015
0.00001
0.000005
0
Thallium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

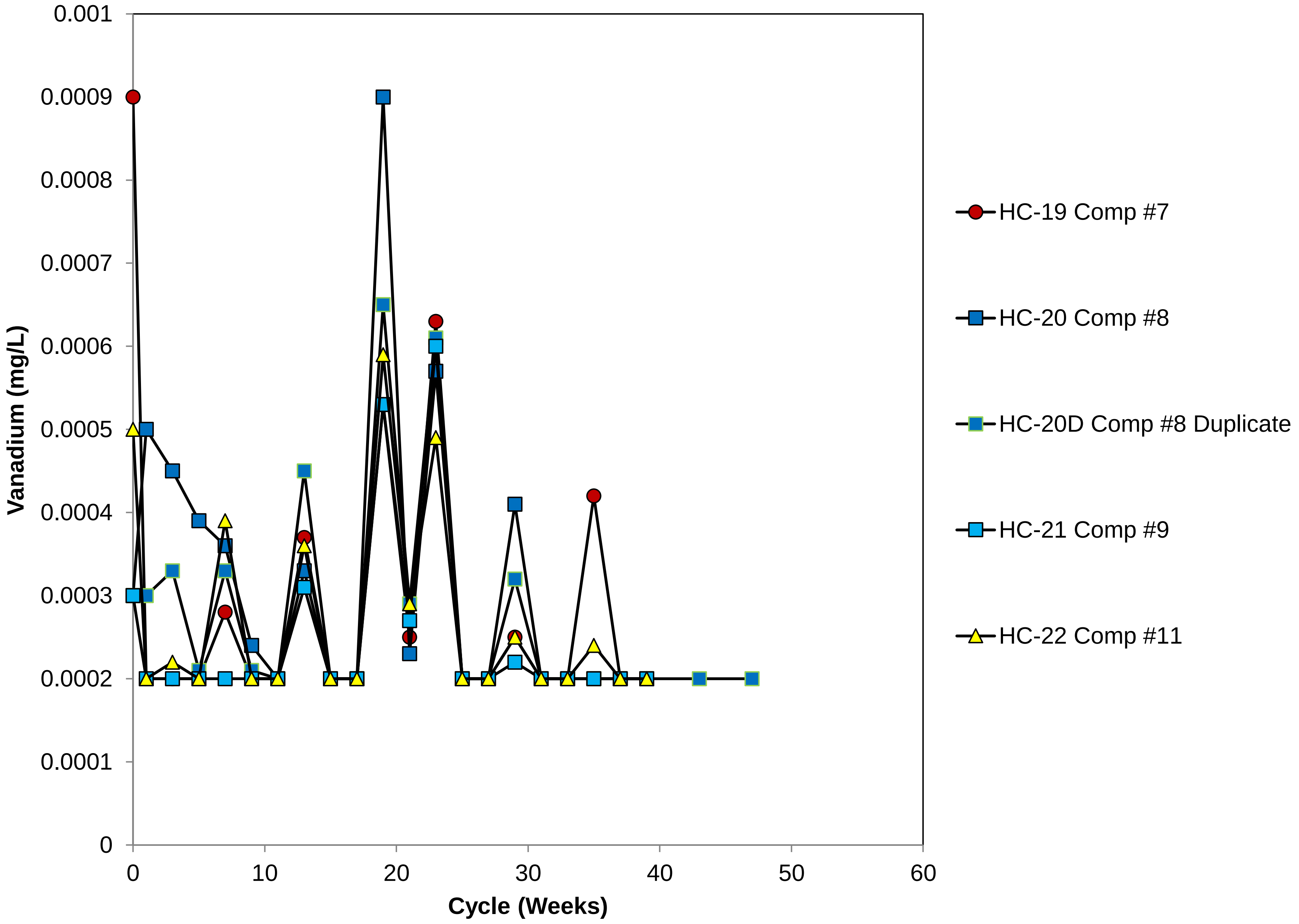

0.000009
0.000008
0.000007
0.000006
0.000005
0.000004
0.000003
0.000002
0.000001
0
Thorium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

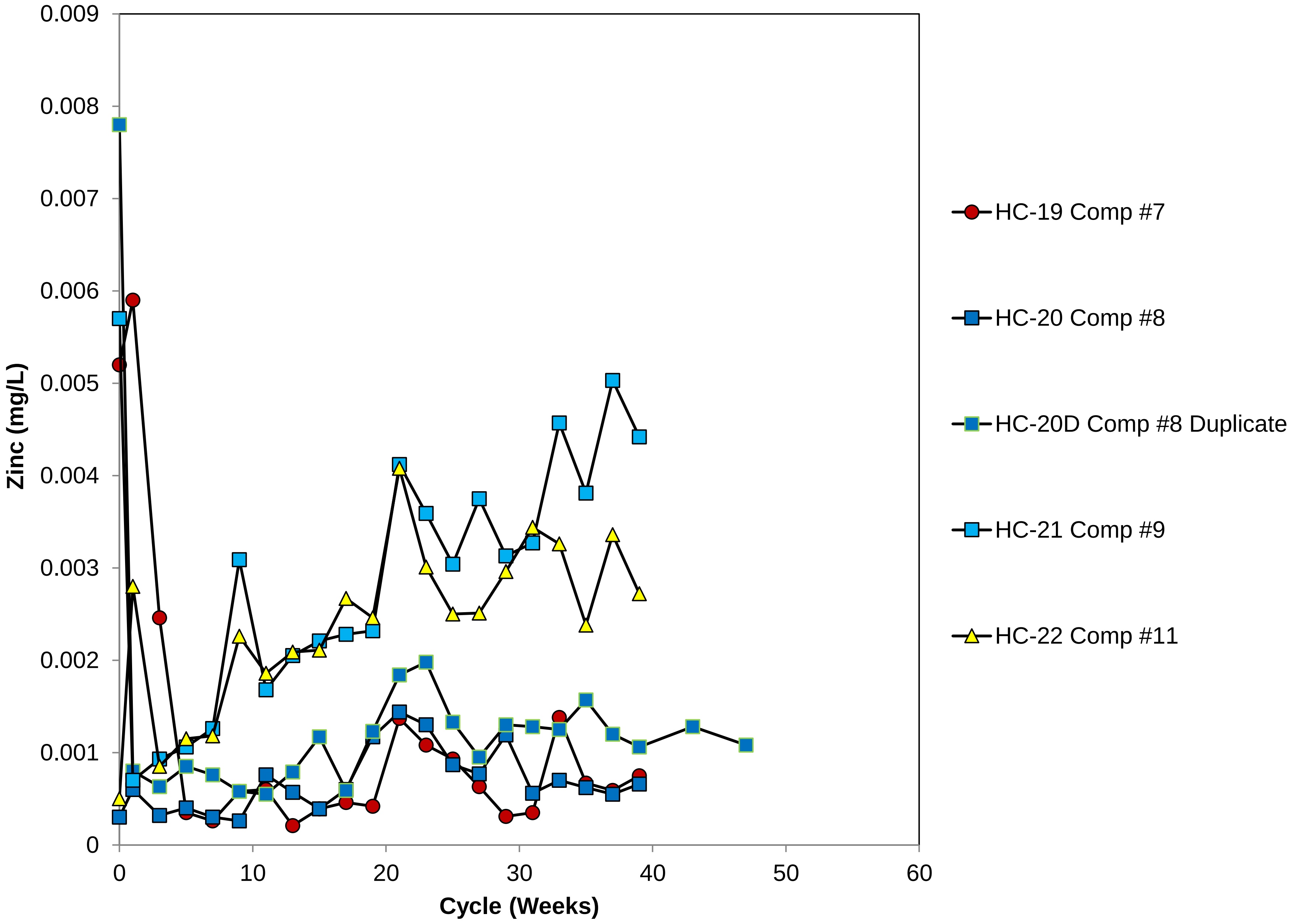

0.003
0.0025
0.002
0.0015
0.001
0.0005
0
Tin (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

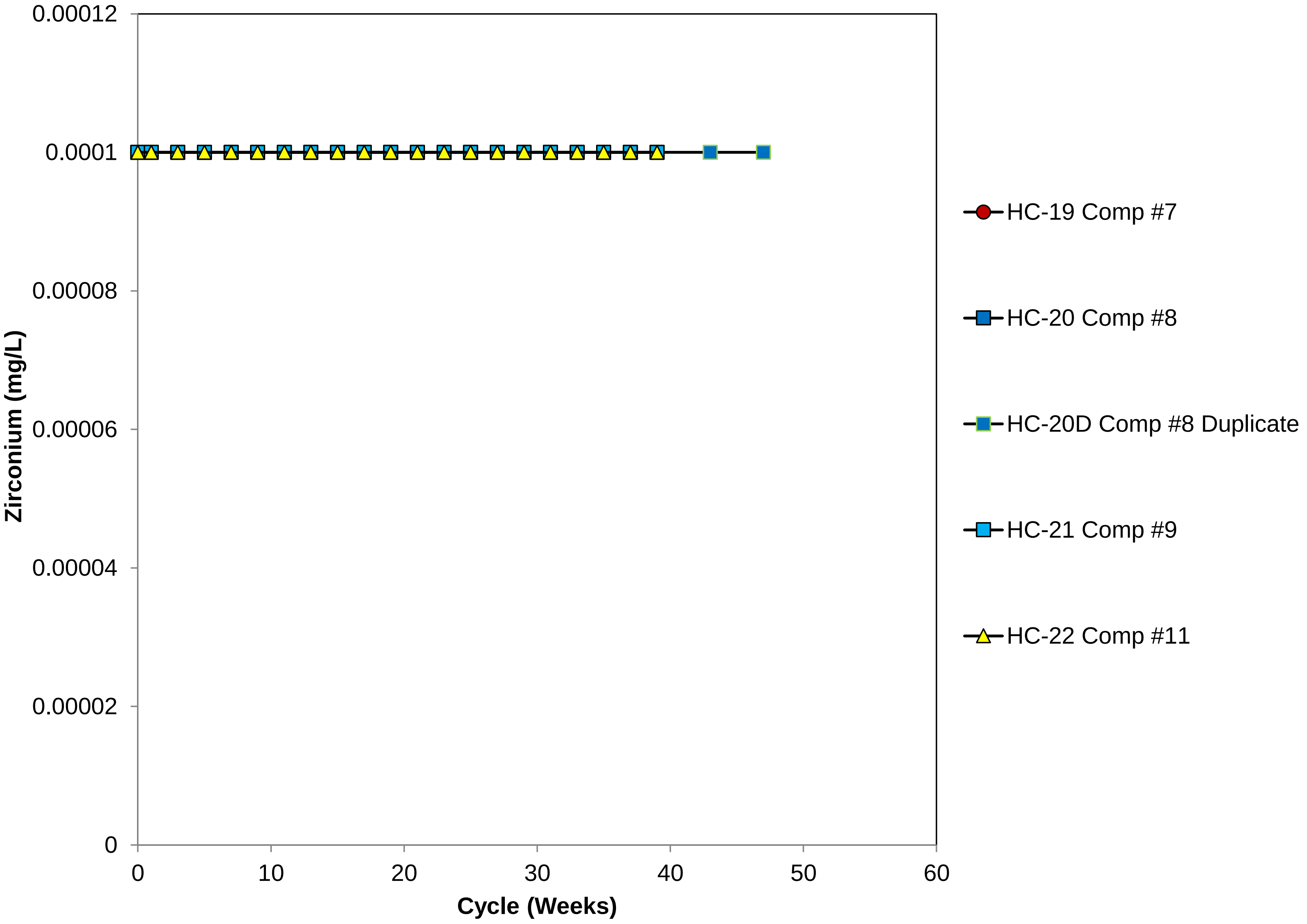

0.0008
0.0007
0.0006
0.0005
0.0004
0.0003
0.0002
0.0001
0
Titanium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

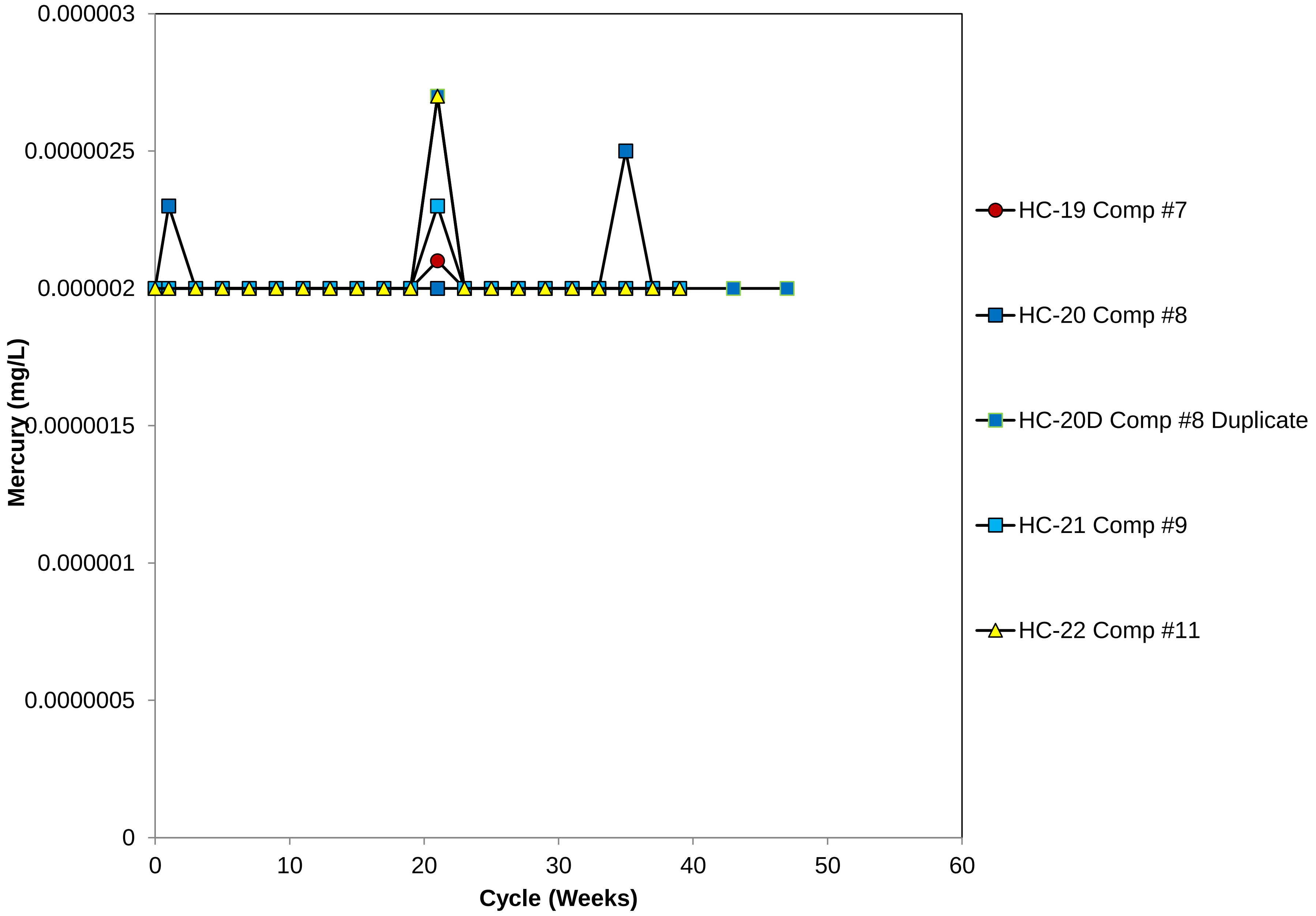

0.0016
0.0014
0.0012
0.001
0.0008
0.0006
0.0004
0.0002
0
Tungsten (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11


0.0016
0.0014
0.0012
0.001
0.0008
0.0006
0.0004
0.0002
0
Uranium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

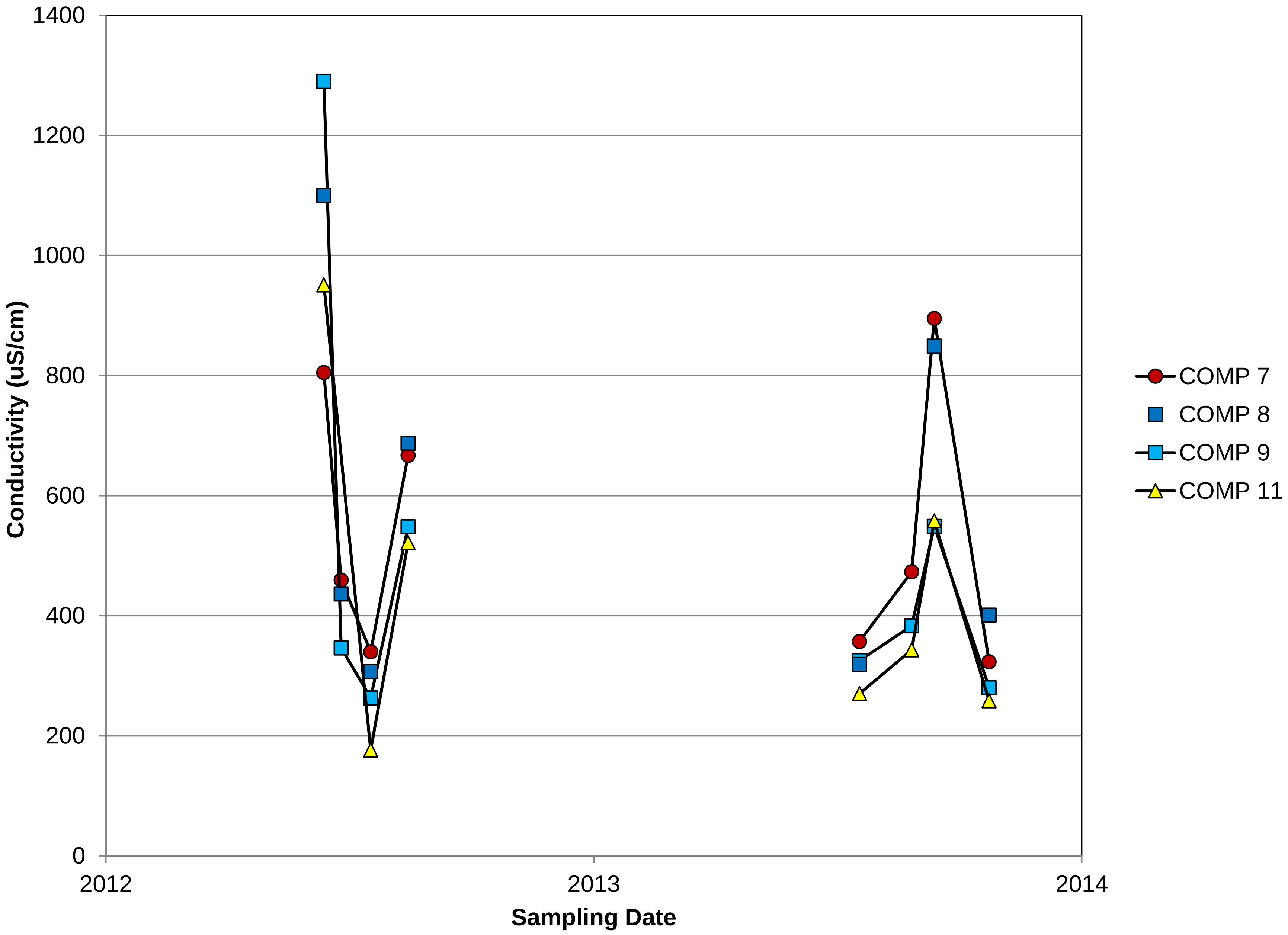

Vanadium (mg/L)
0.001
0.0009
0.0008
0.0007
0.0006
0.0005
0.0004
0.0003
0.0002
0.0001
0
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11


0.009
0.008
0.007
0.006
0.005
0.004
0.003
0.002
0.001
0
Zinc (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

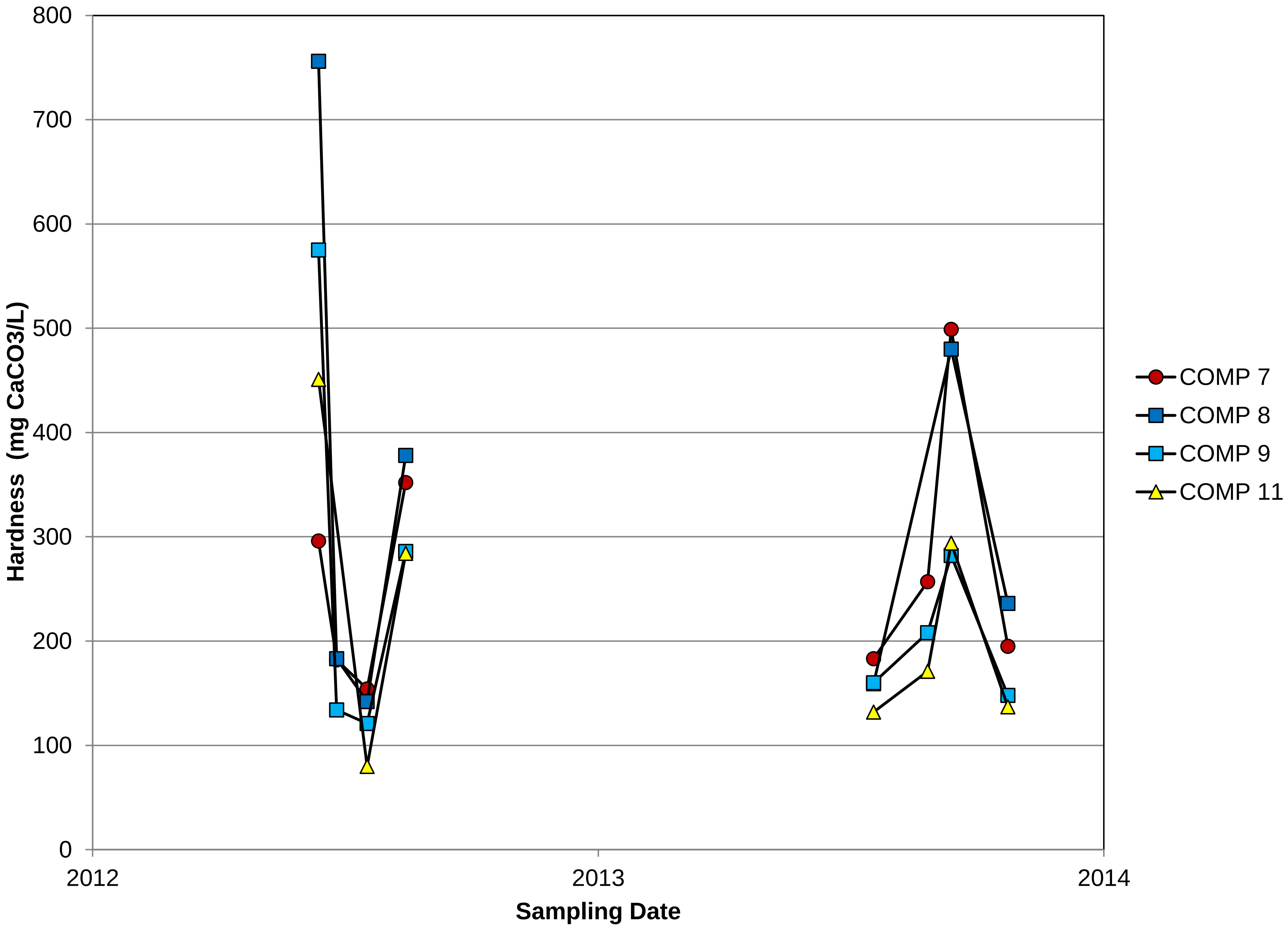

0.00012
0.0001
0.00008
0.00006
0.00004
0.00002
0
Zirconium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11


0.000003
0.0000025
0.000002
0.0000015
0.000001
0.0000005
0
Mercury (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-19 Comp #7
HC-20 Comp #8
HC-20D Comp #8 Duplicate
HC-21 Comp #9
HC-22 Comp #11

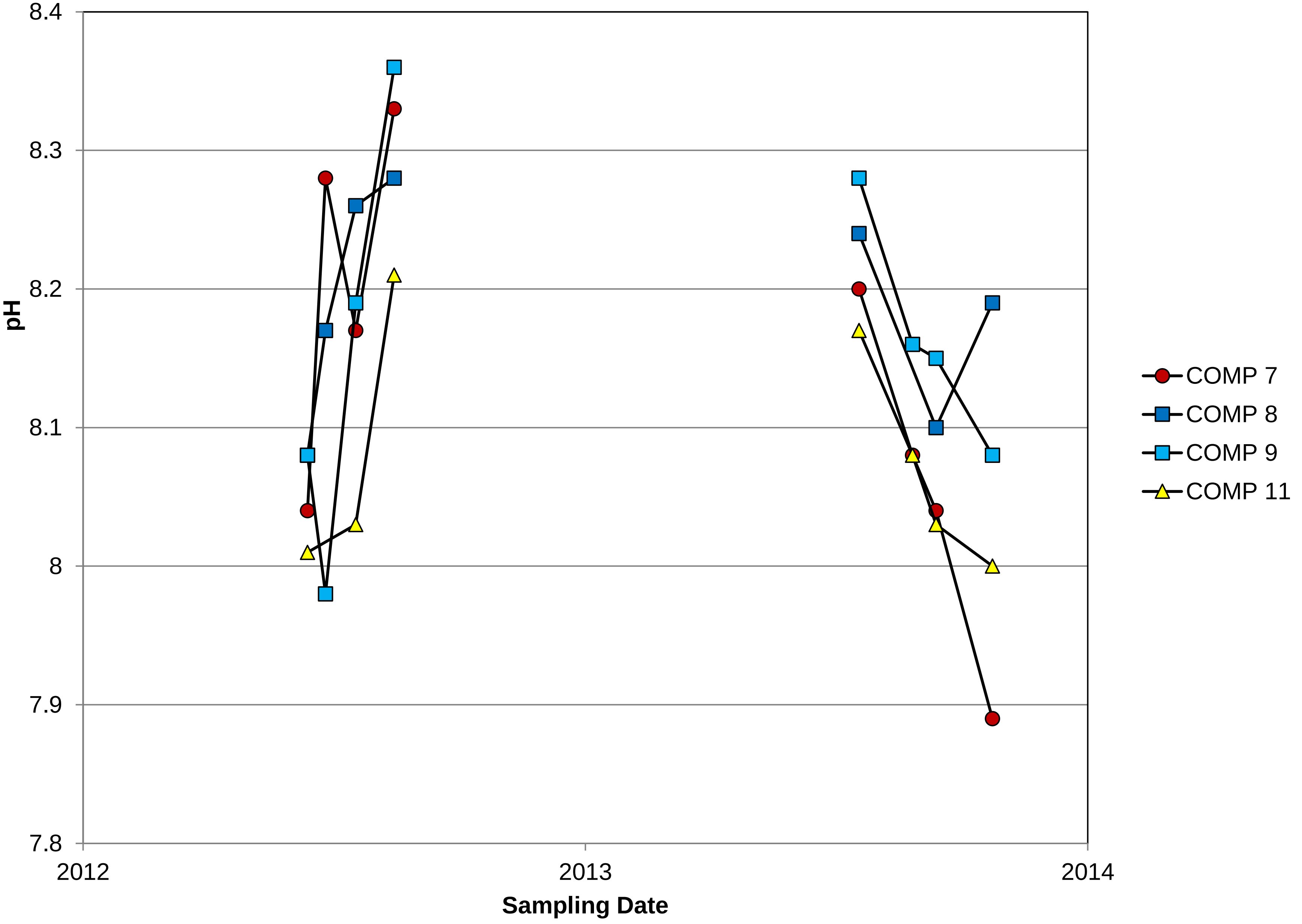
C5: Barrel Tests



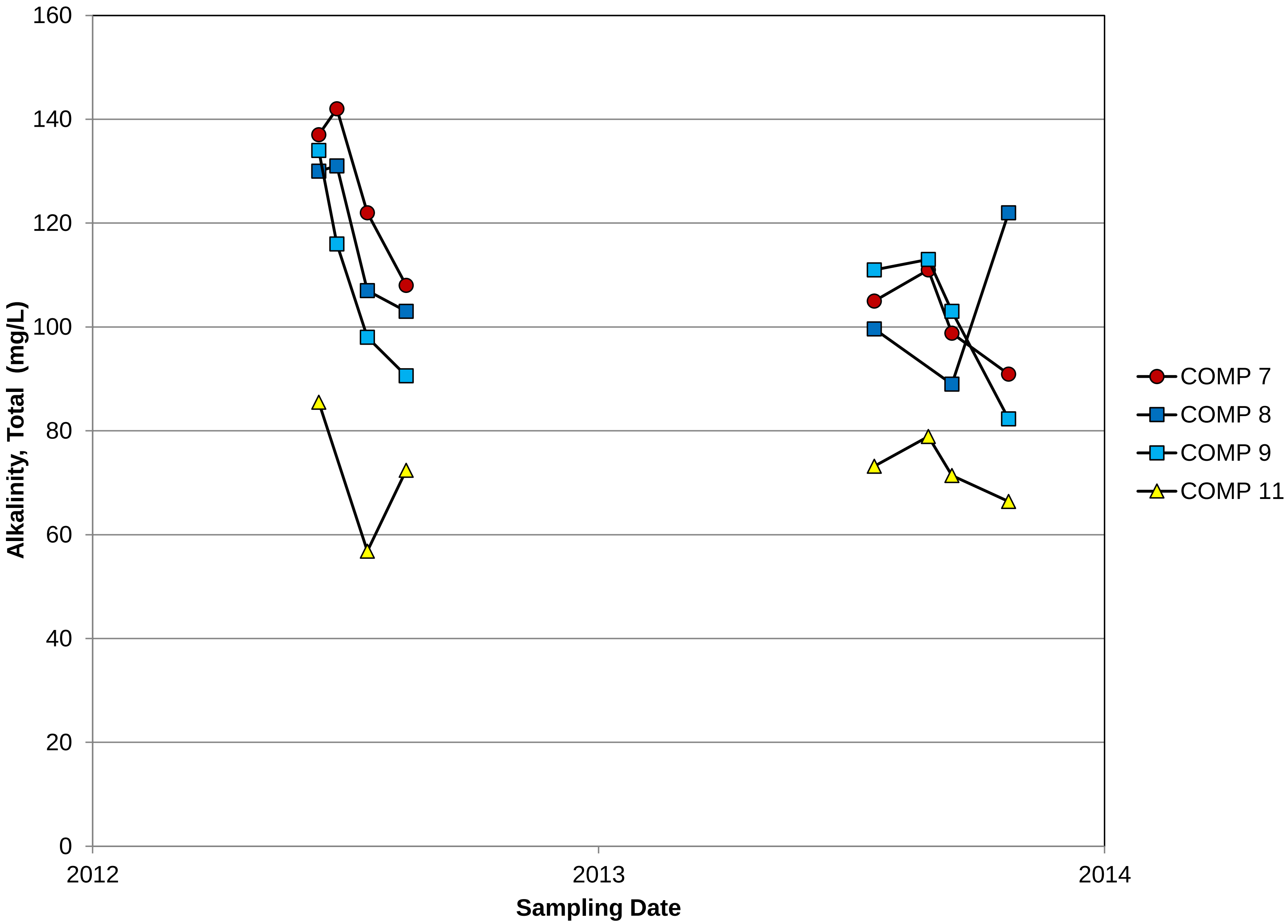
P:\01_SITES\Harper_Creek\1CY003.001_MLARD_Feasability&EA\800_On_Site_Kinetic_Testing\Barrels\Calcs and Charts\[Conc_Charts_1CY003.001_AML_rev02.xlsx]



P:\01_SITES\Harper_Creek\1CY003.001_MLARD_Feasability&EA\800_On_Site_Kinetic_Testing\Barrels\Calcs and Charts\[Conc_Charts_1CY003.001_AML_rev02.xlsx]



P:\01_SITES\Harper_Creek\1CY003.001_MLARD_Feasability&EA\800_On_Site_Kinetic_Testing\Barrels\Calcs and Charts\[Conc_Charts_1CY003.001_AML_rev02.xlsx]



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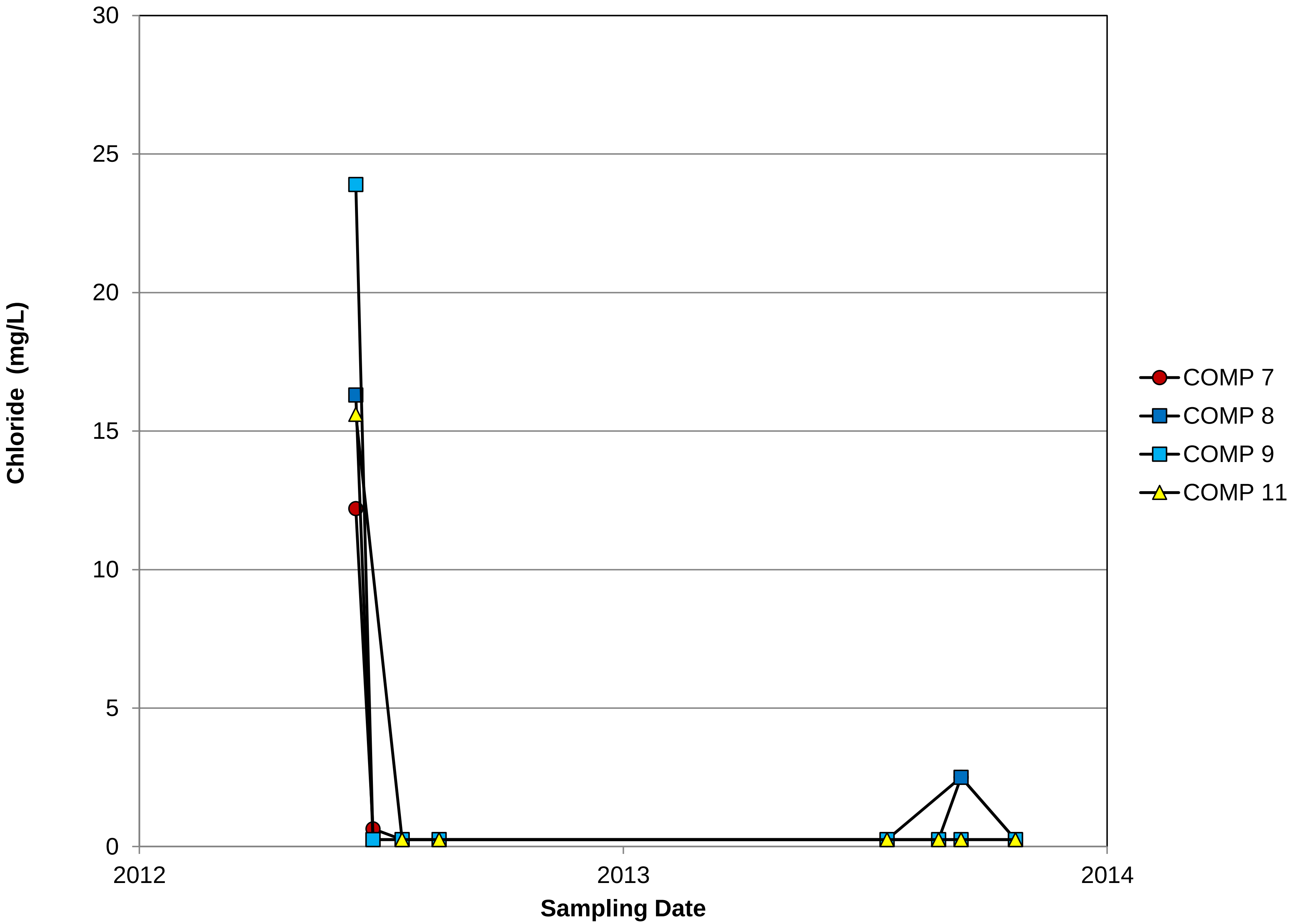

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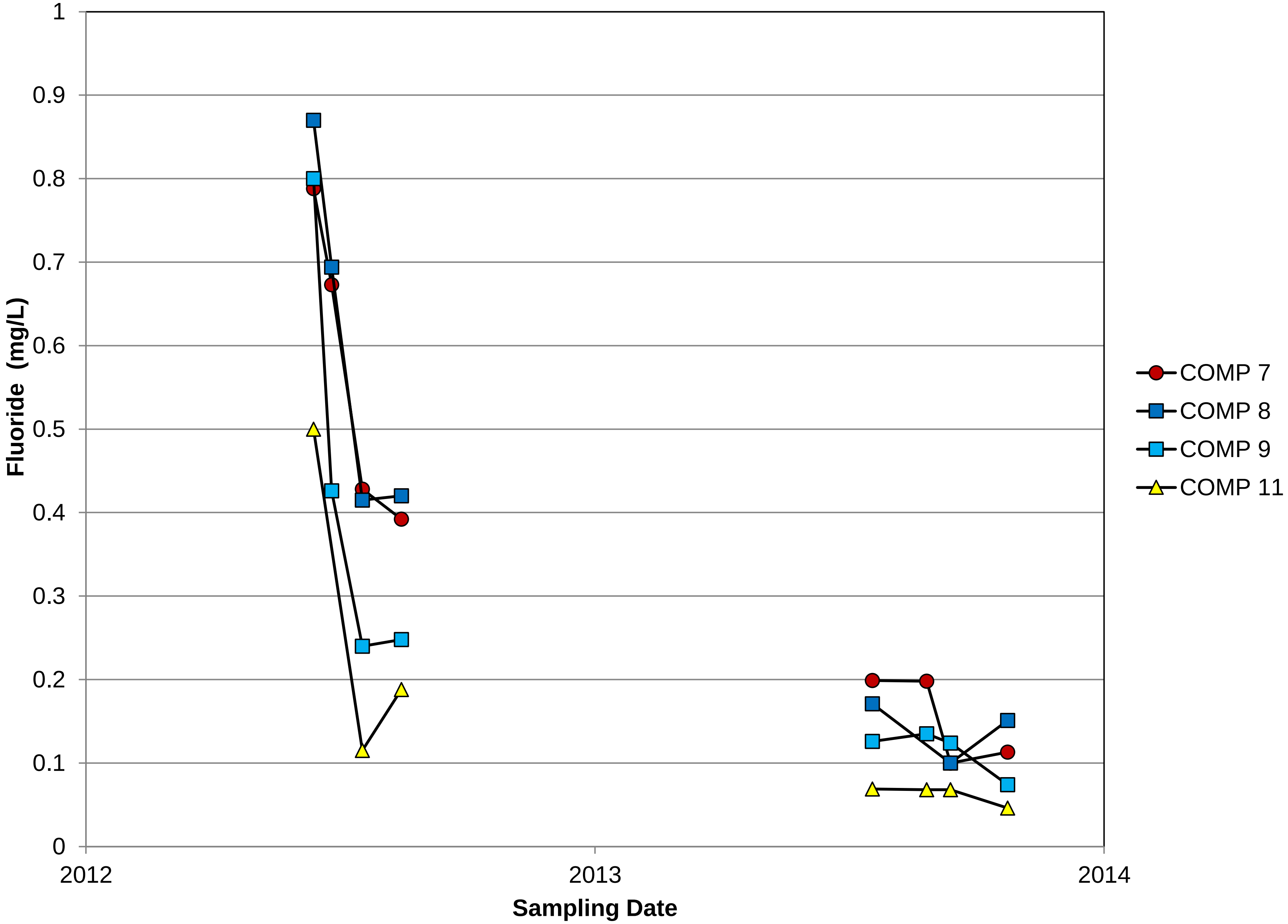

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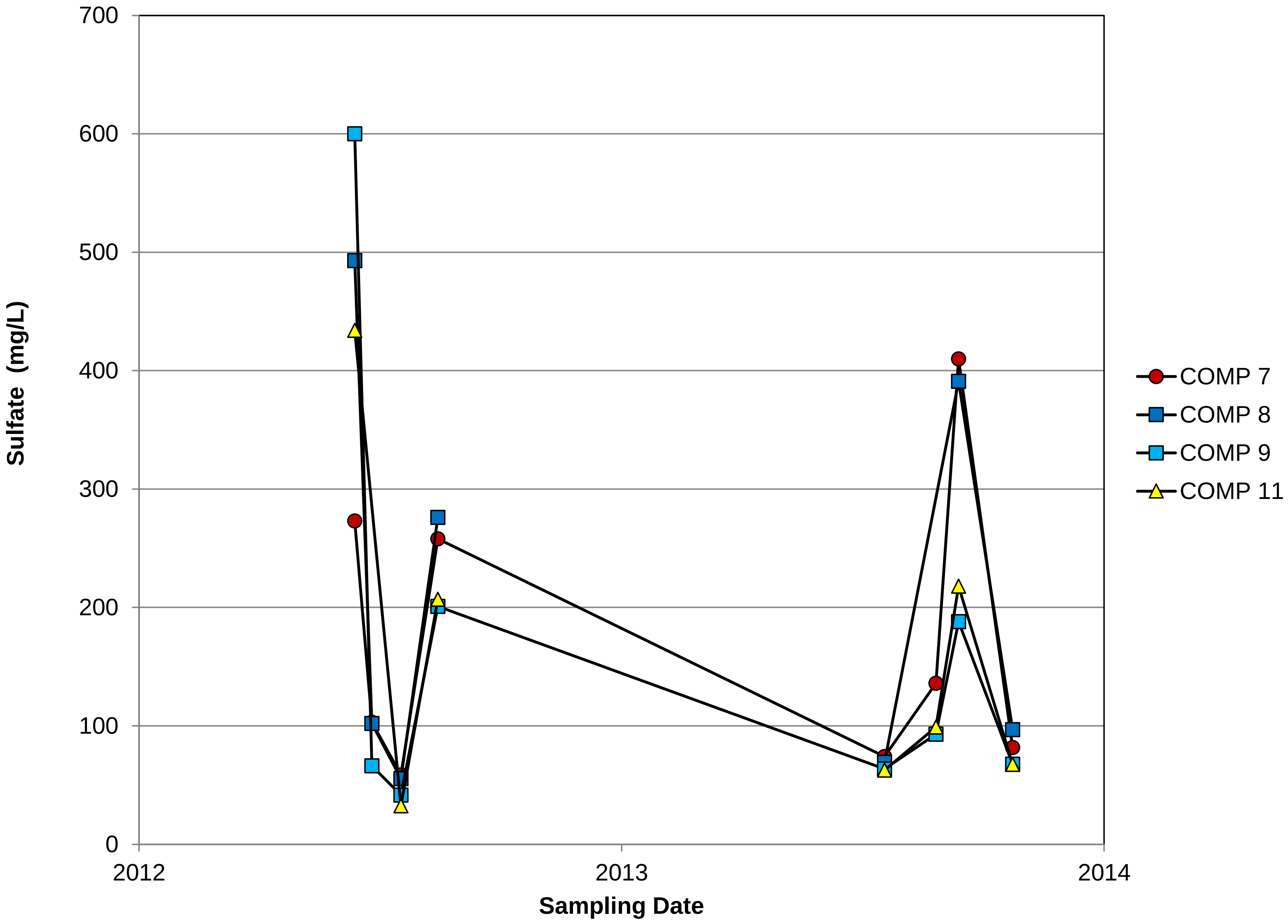

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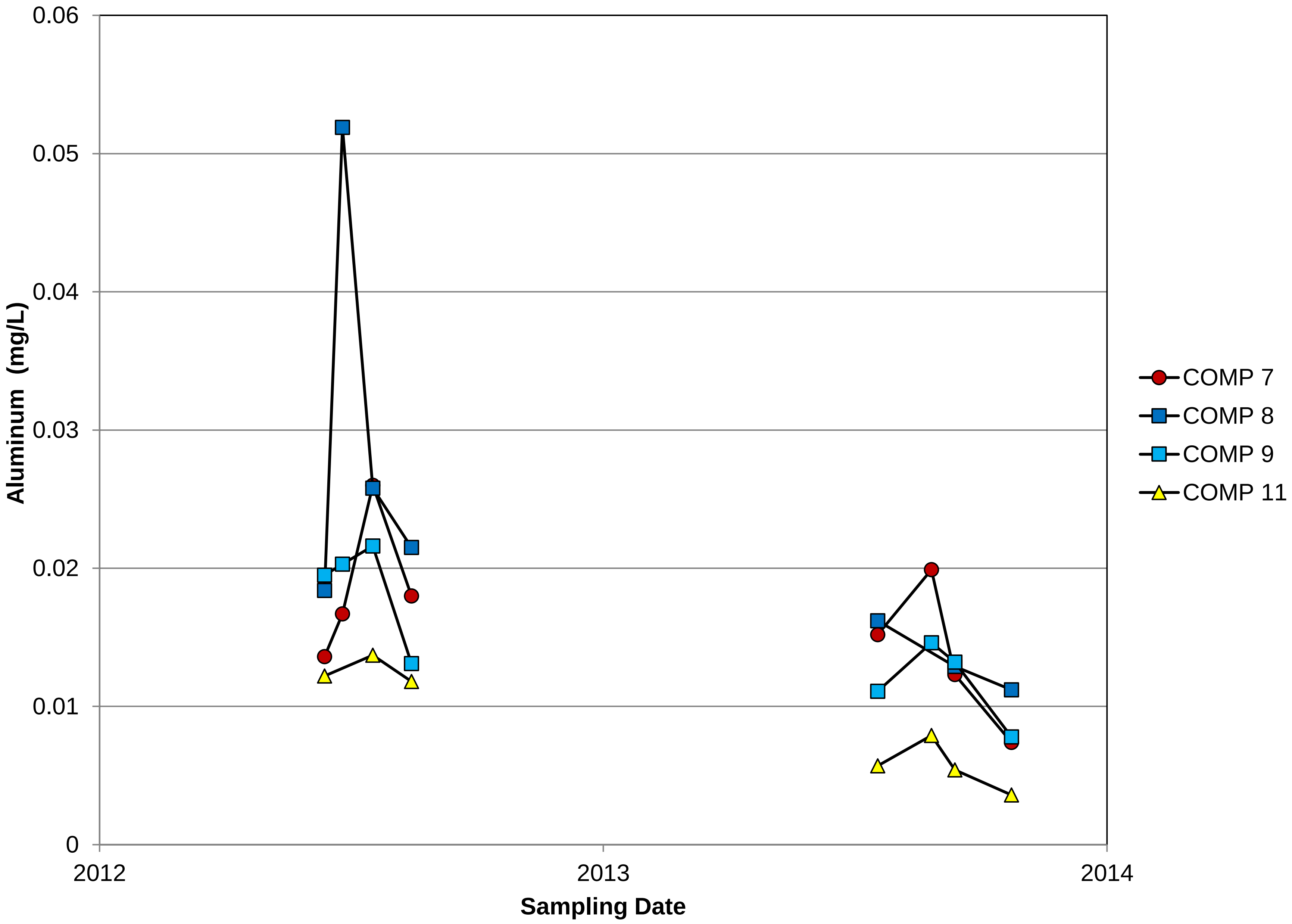

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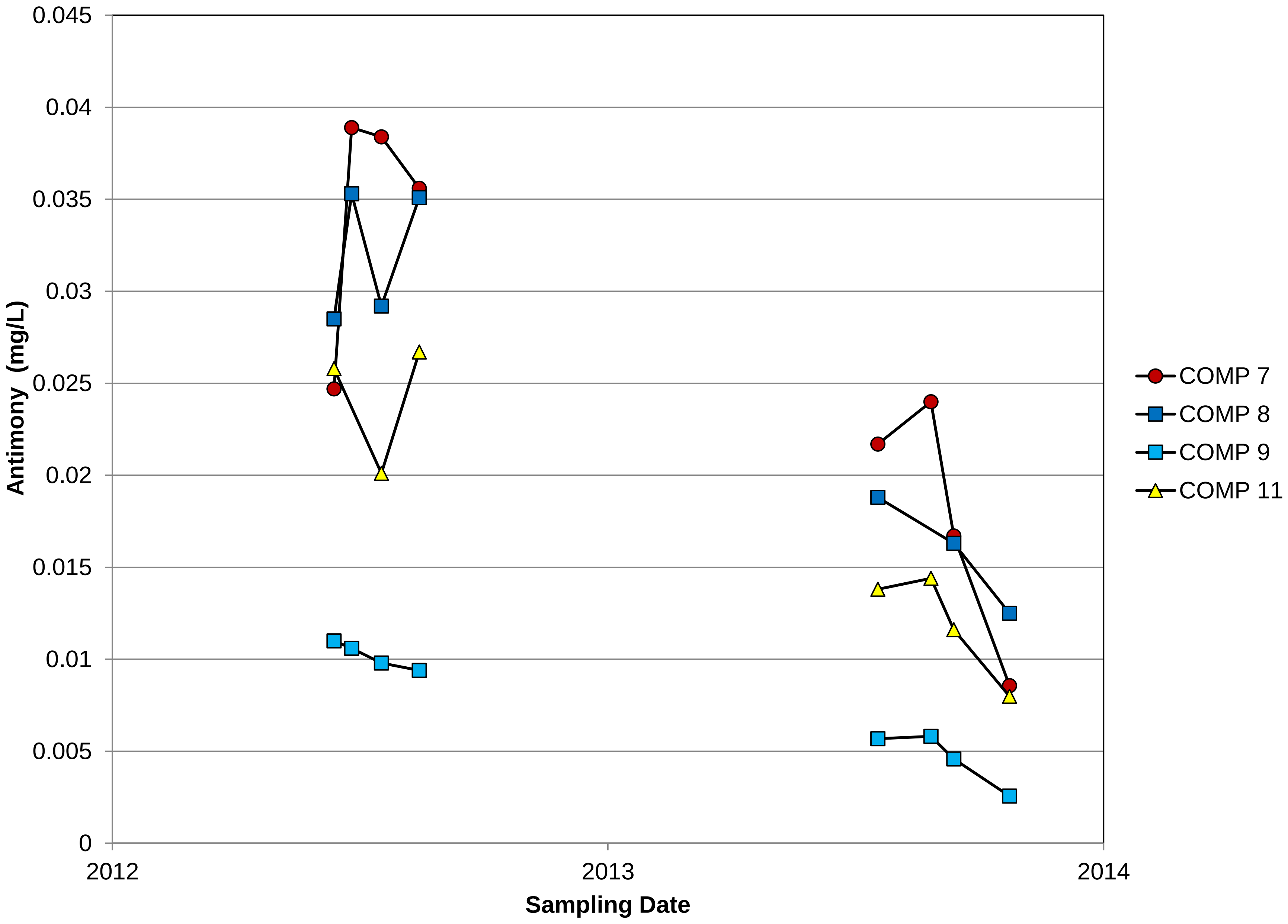

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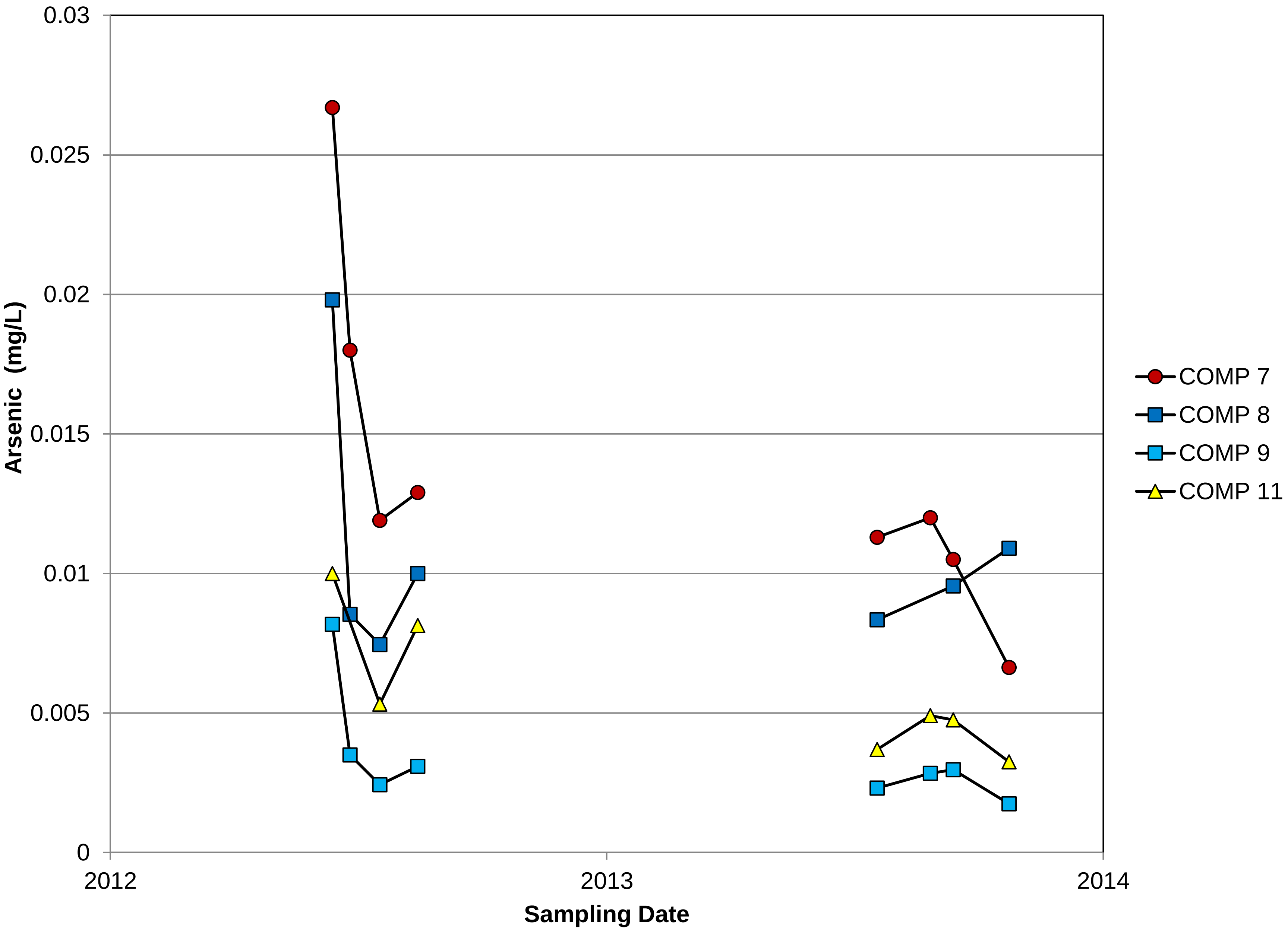

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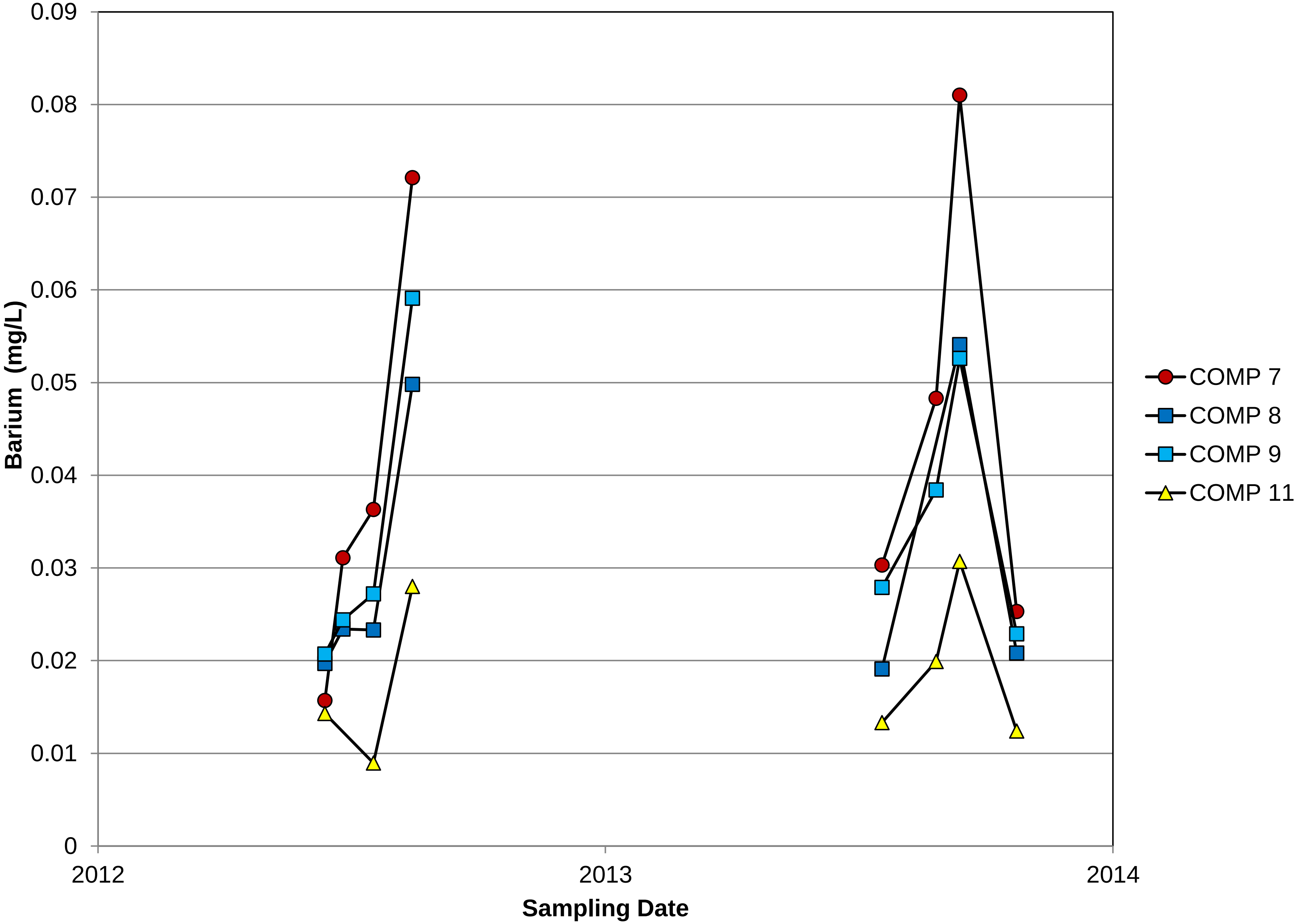

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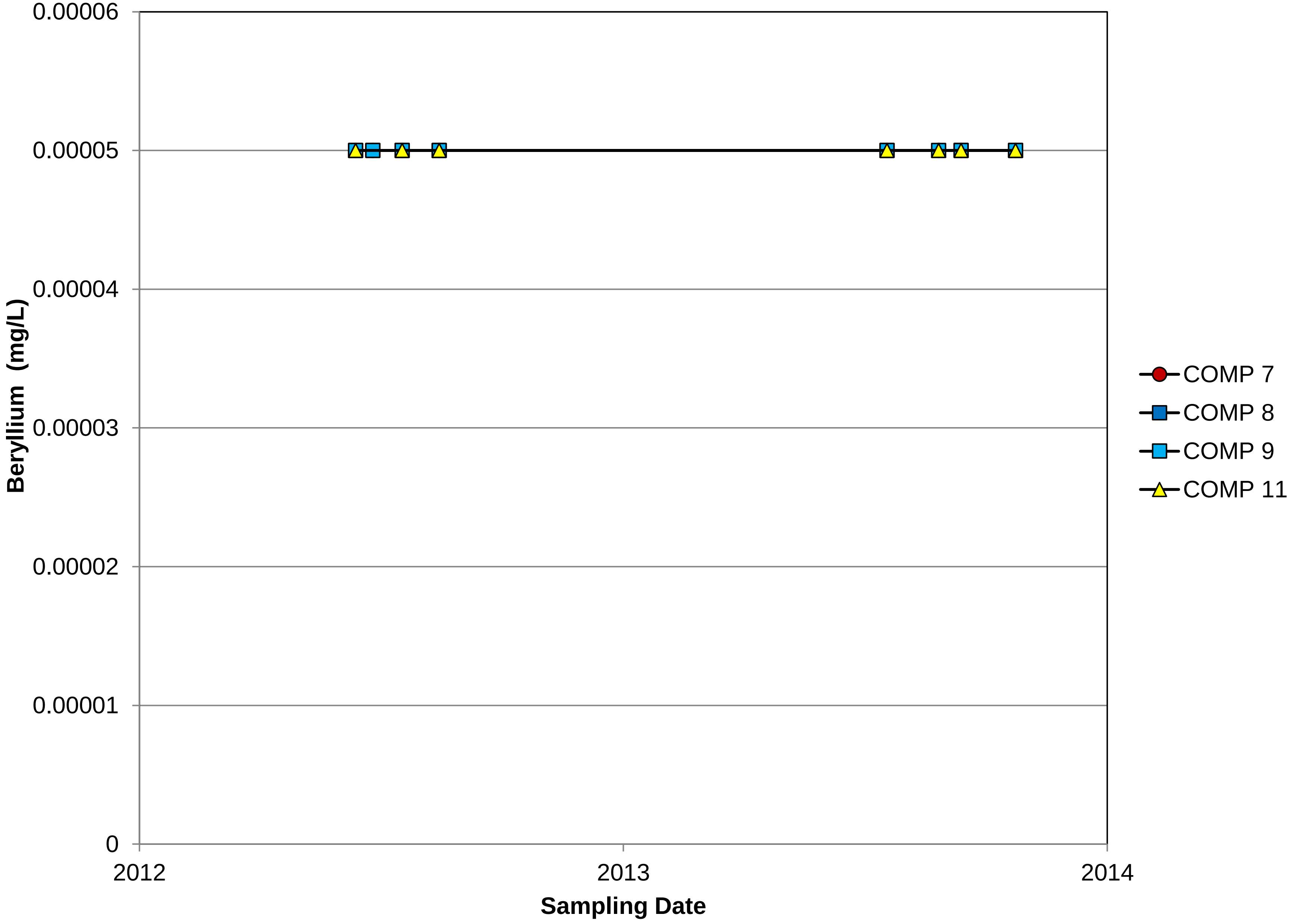

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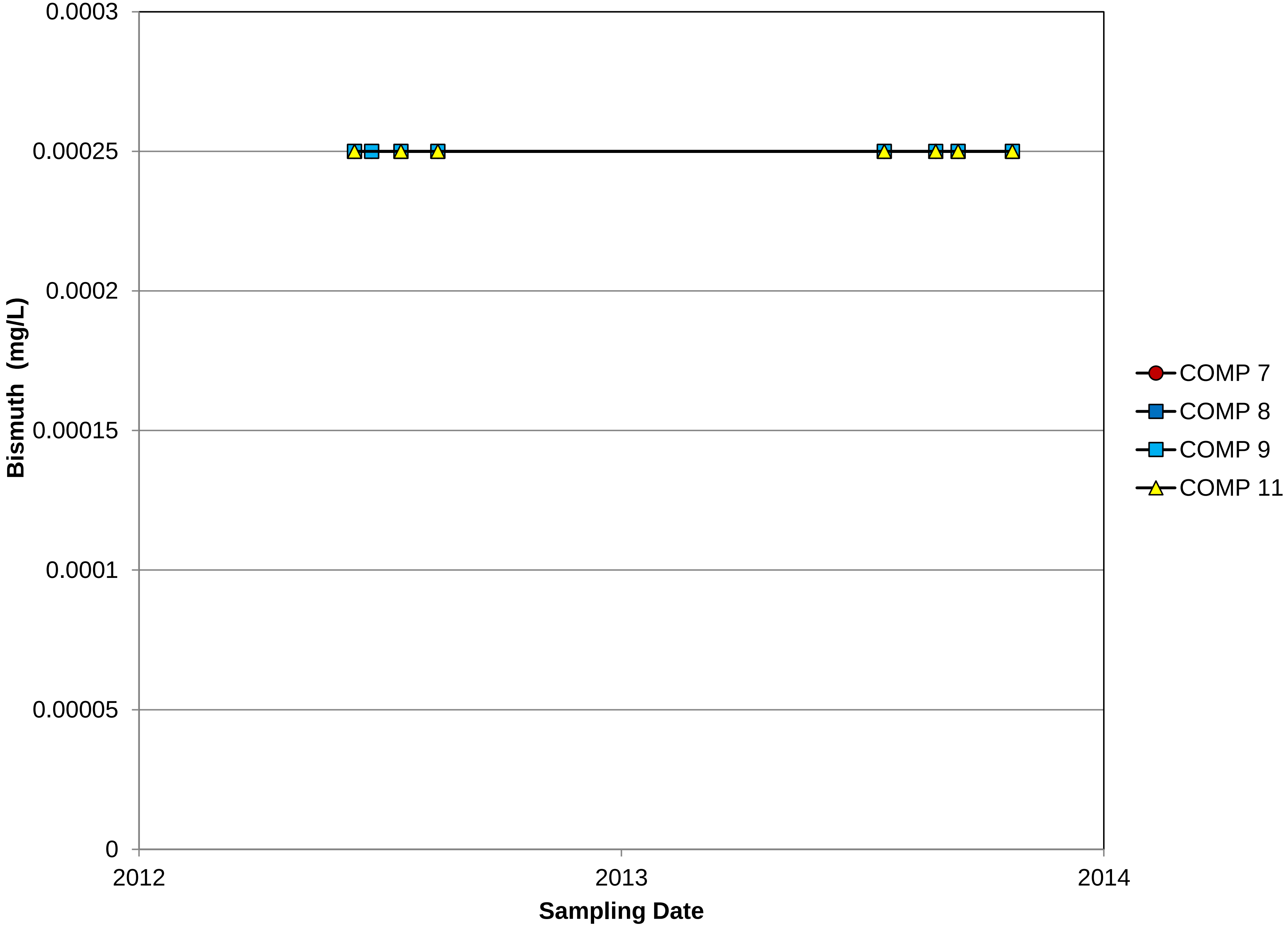

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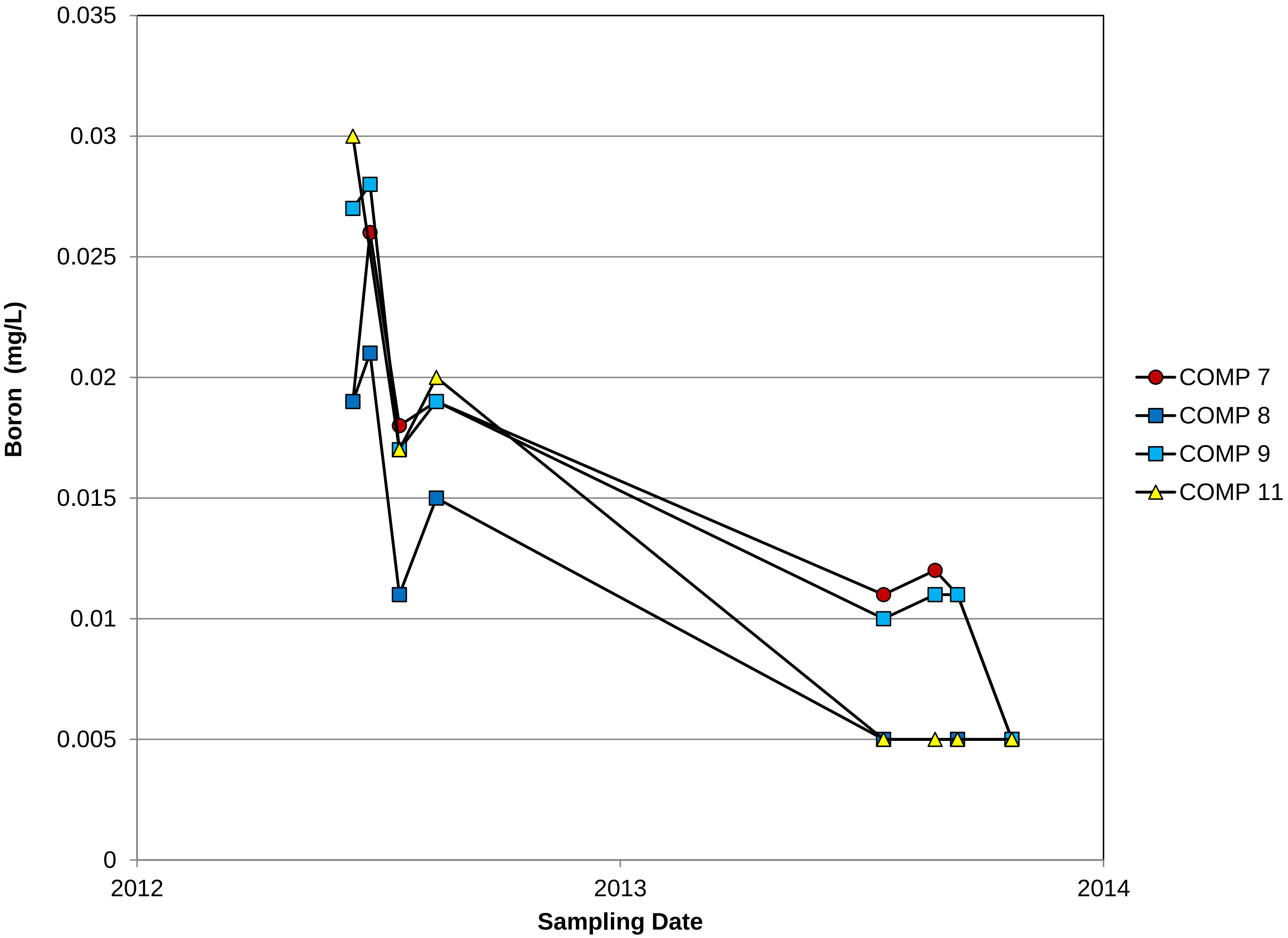

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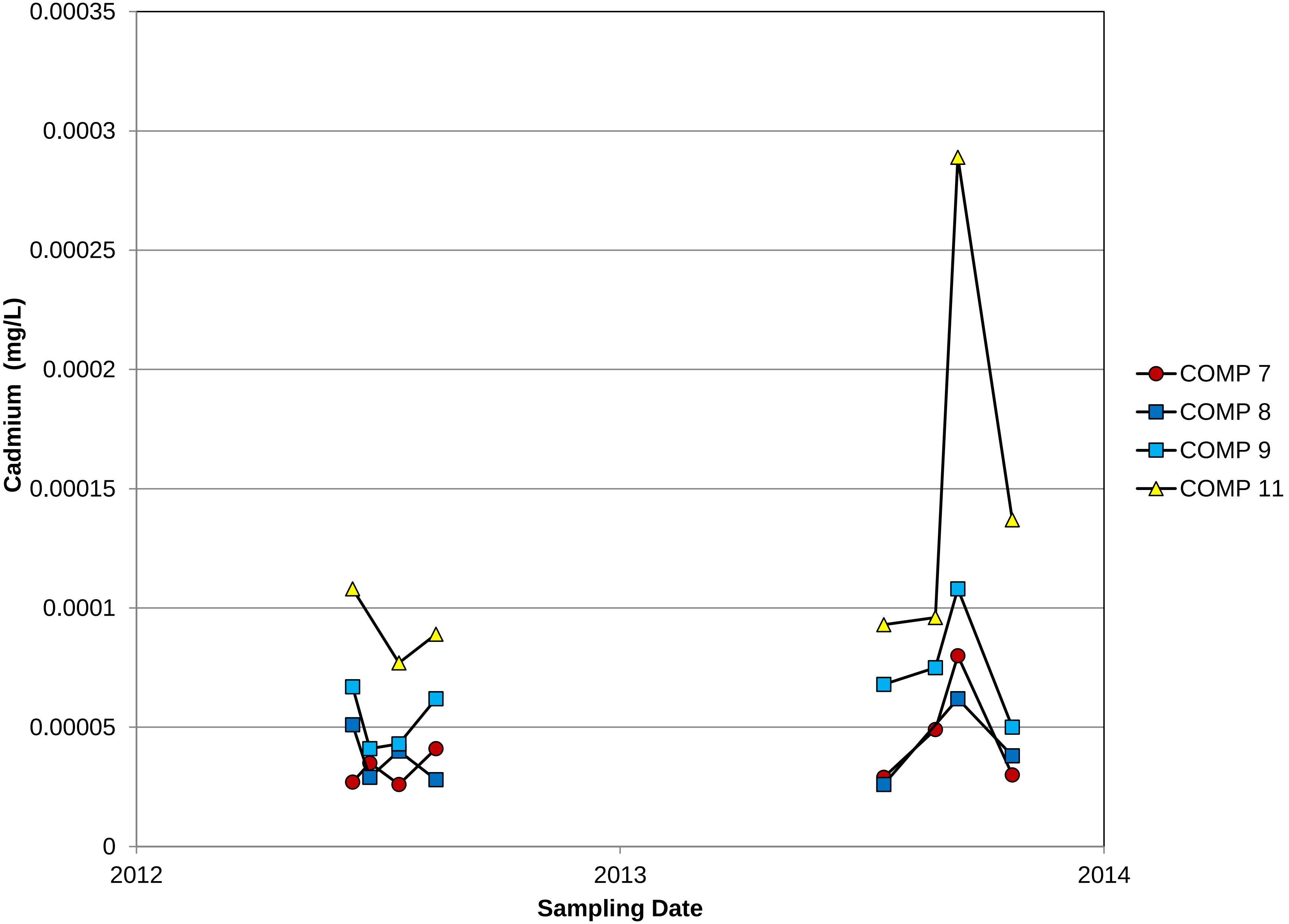

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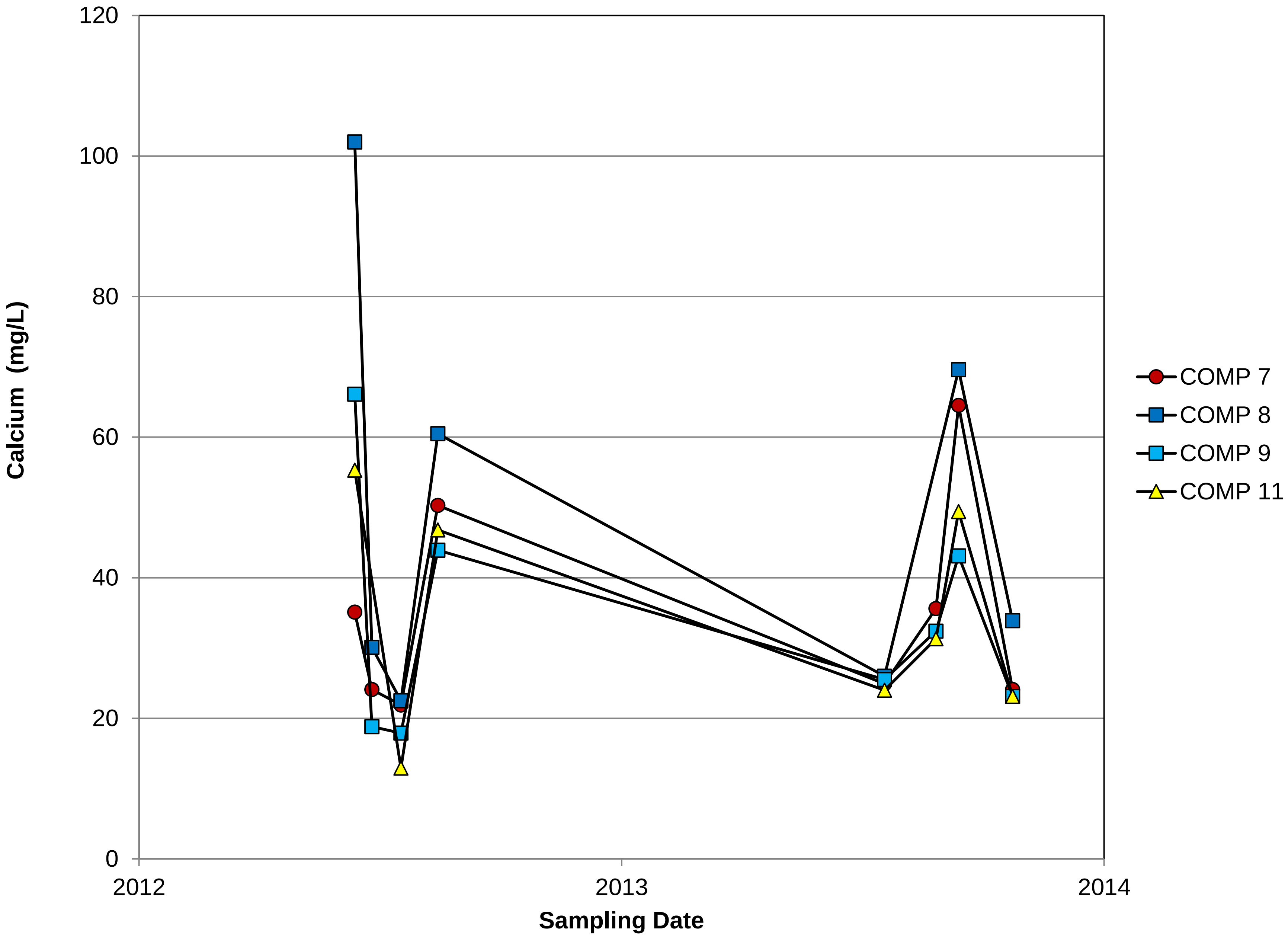

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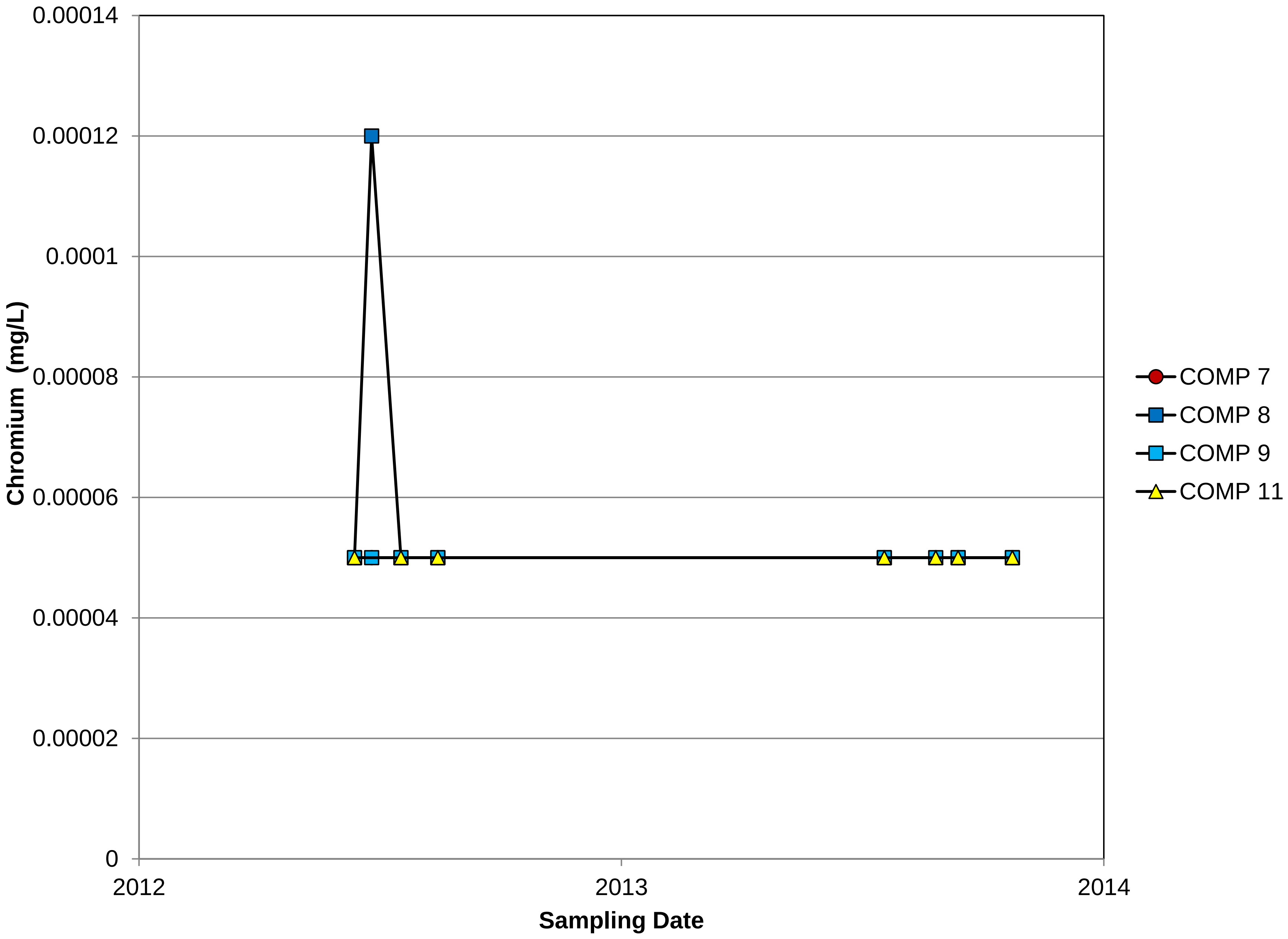

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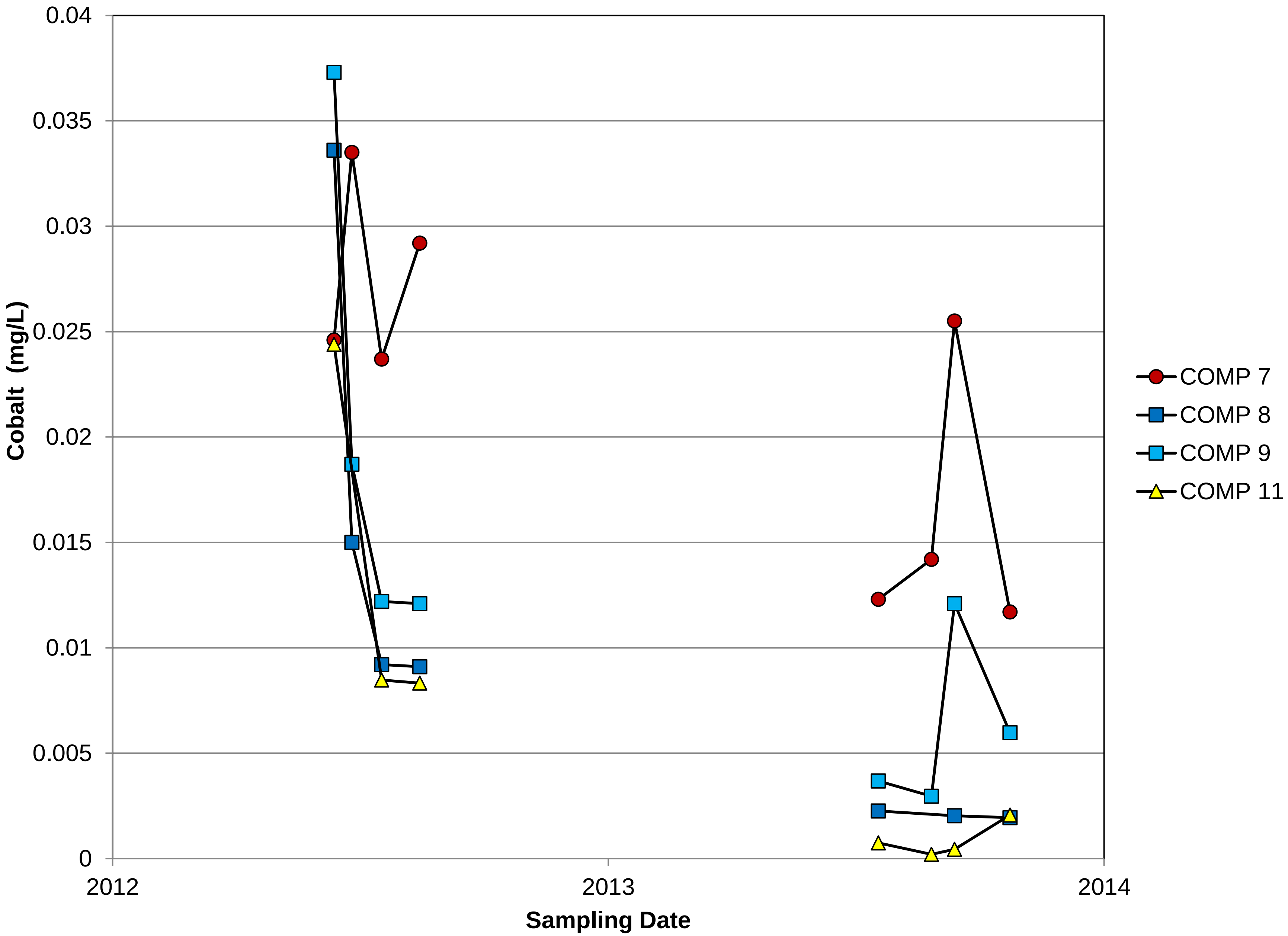

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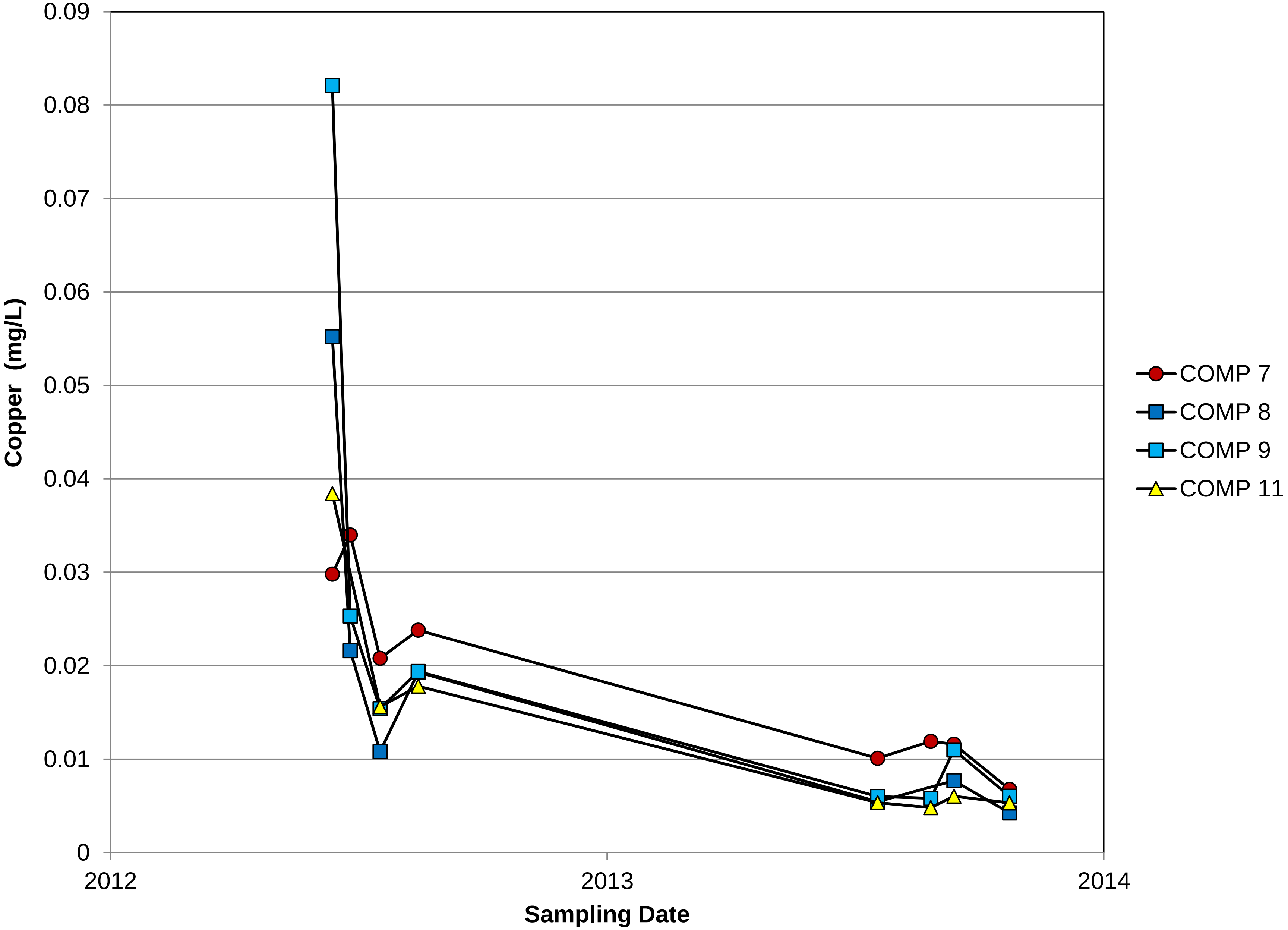

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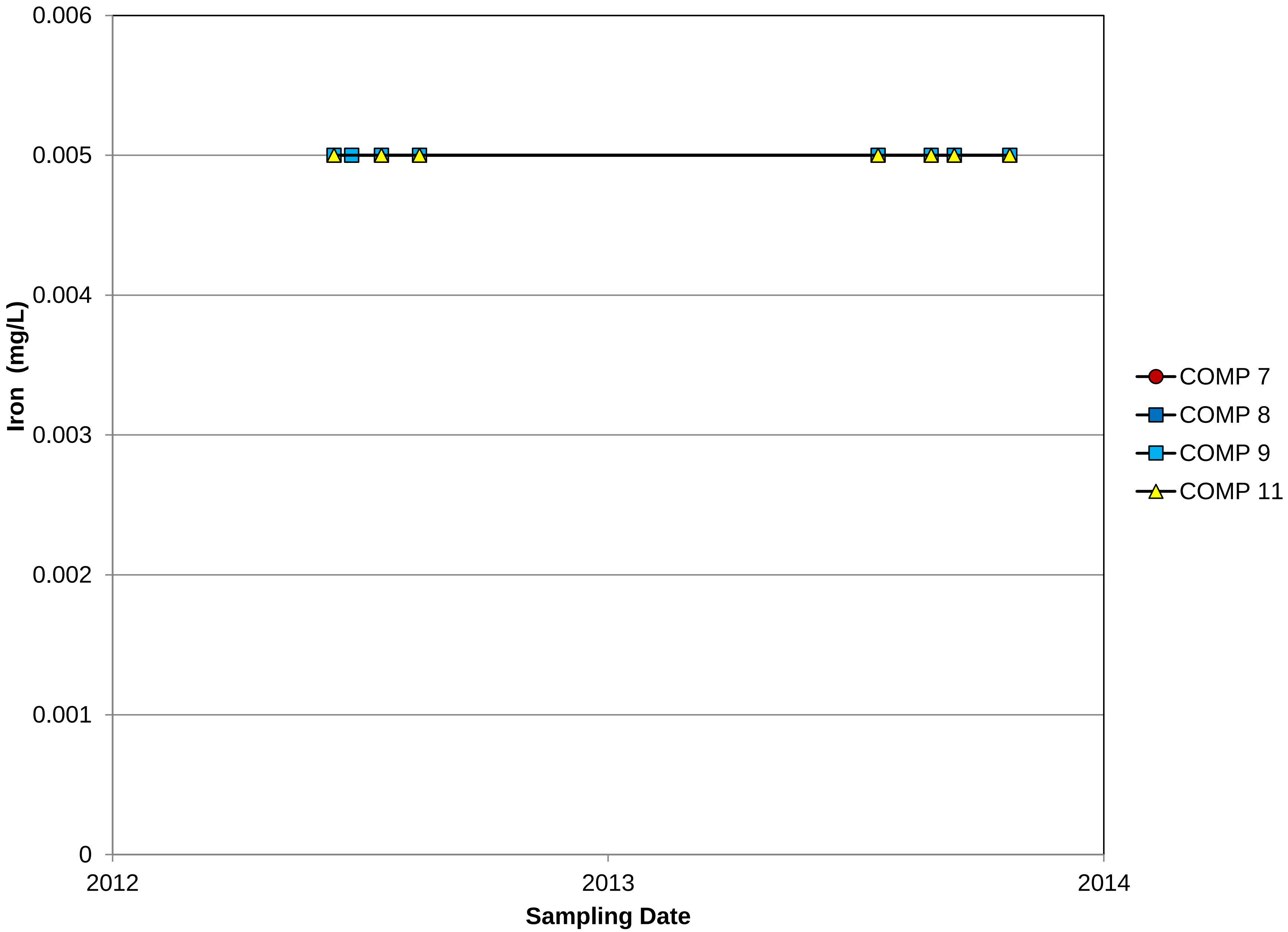

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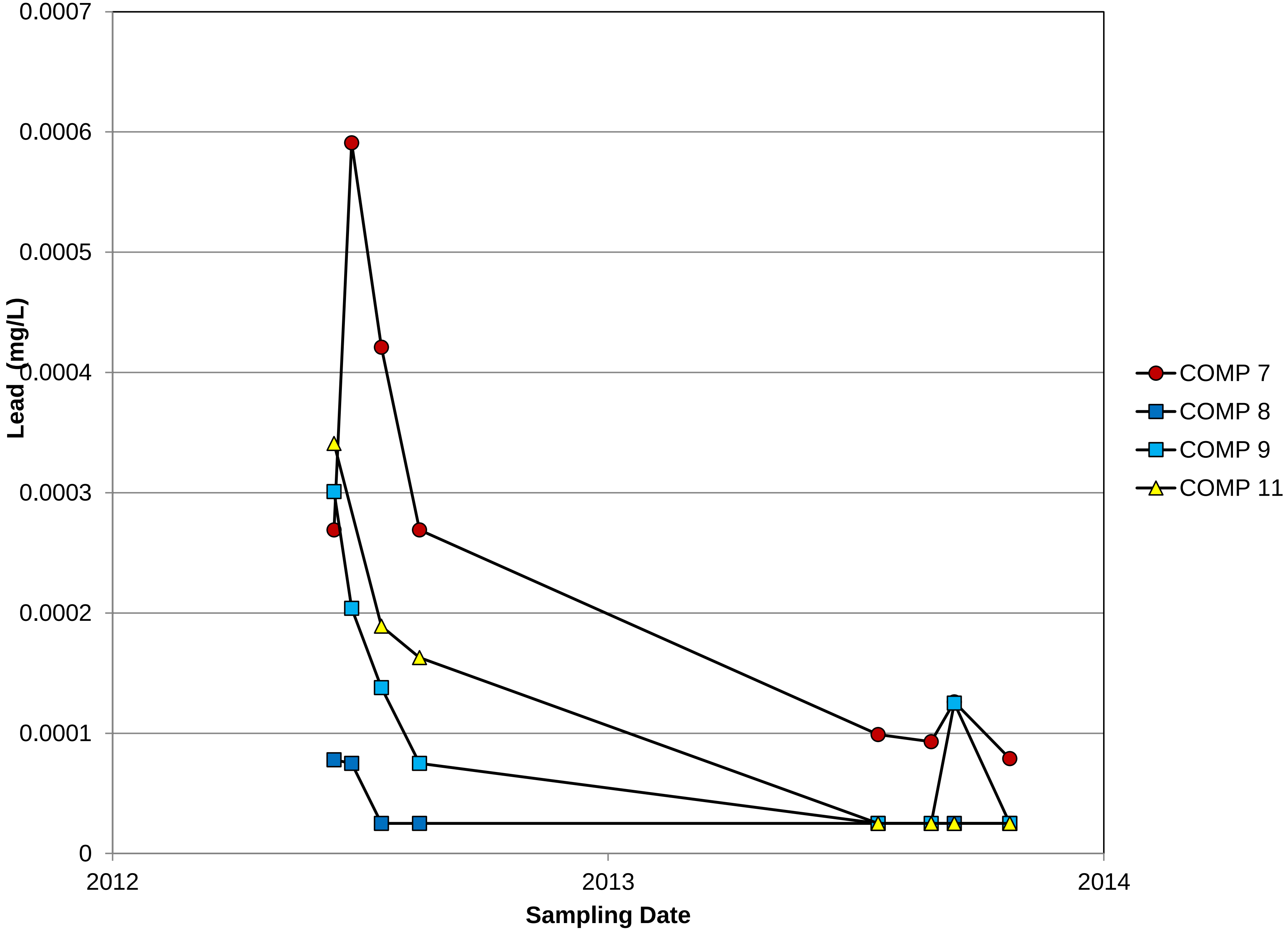

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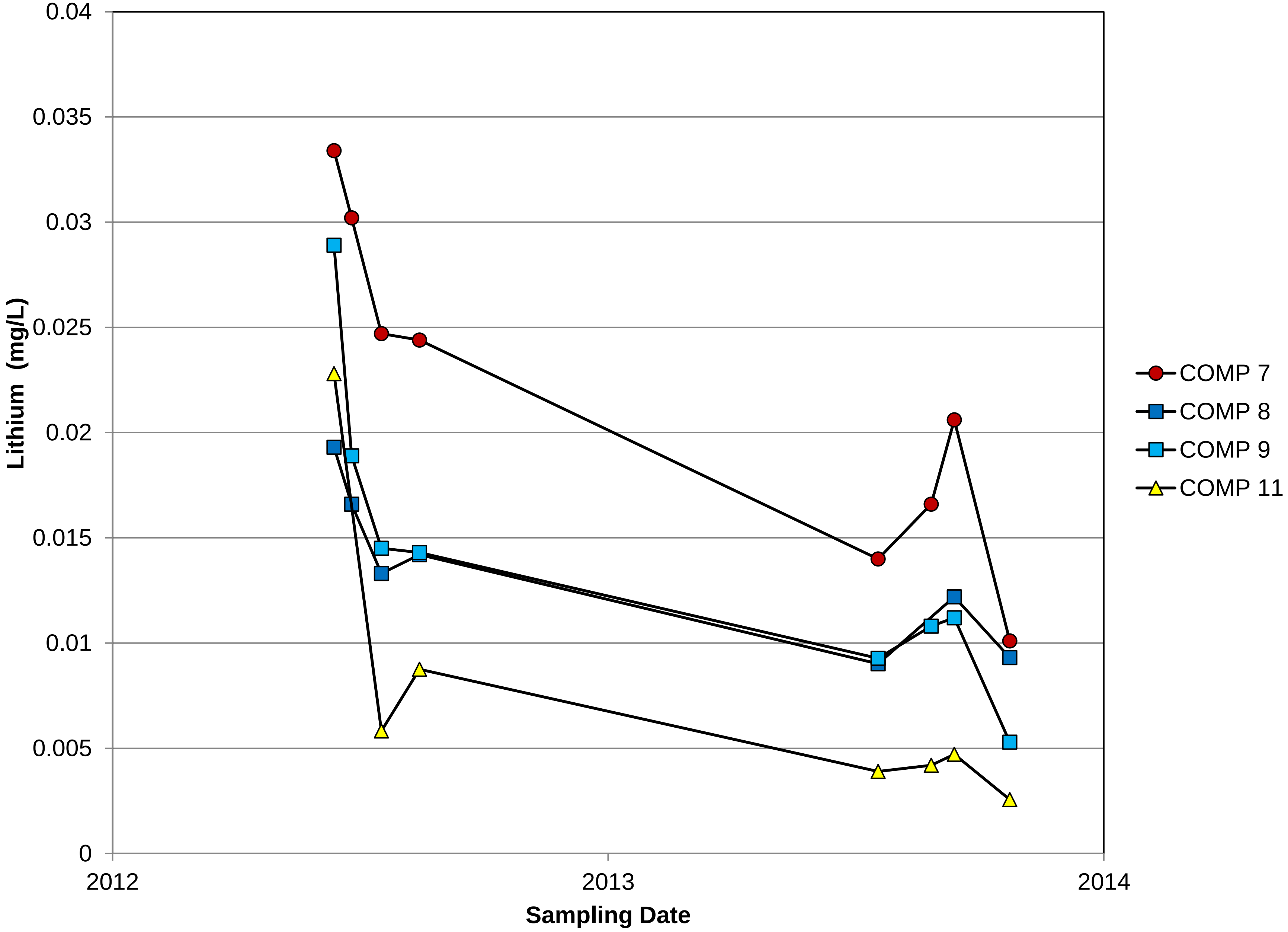

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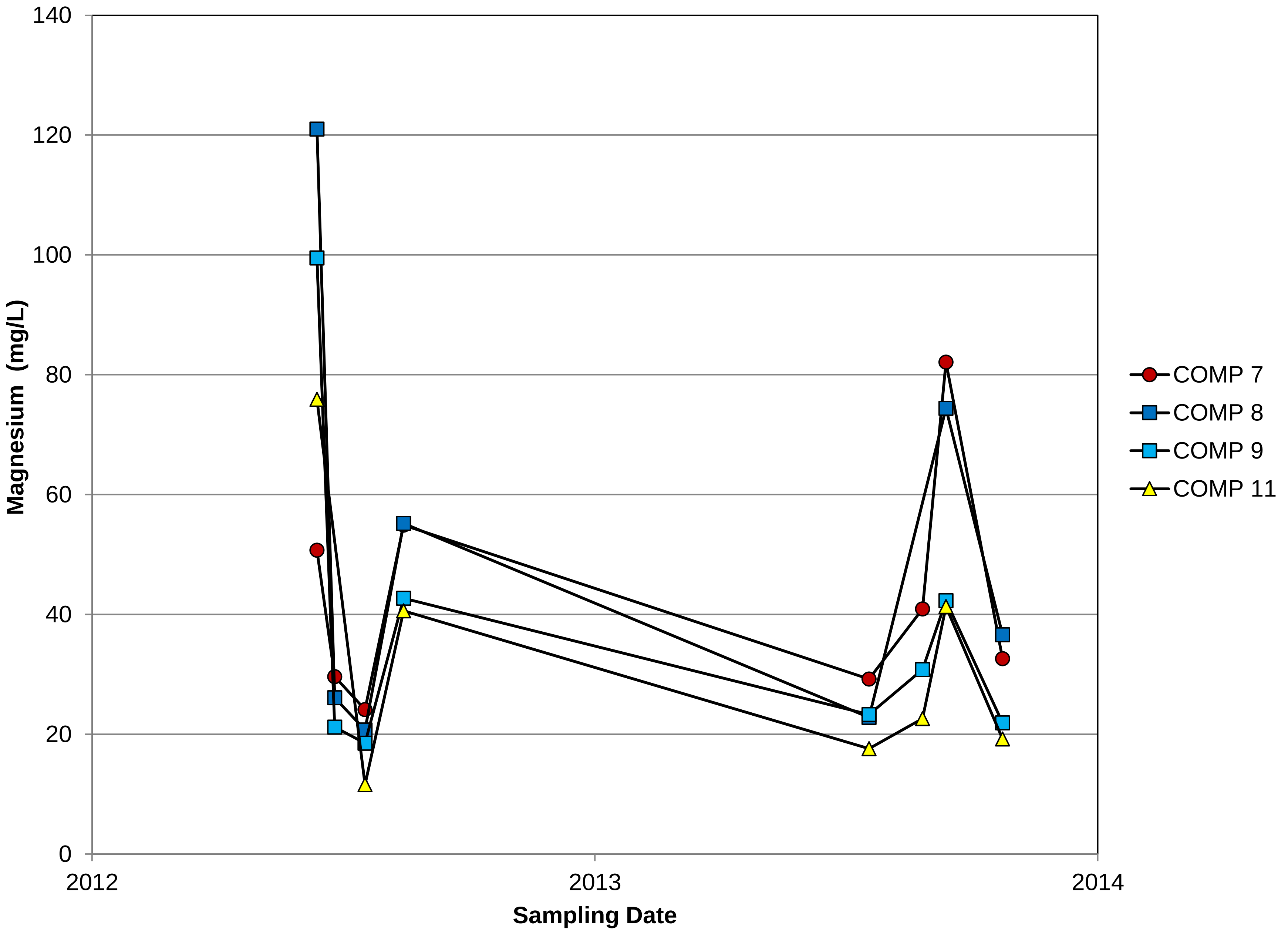

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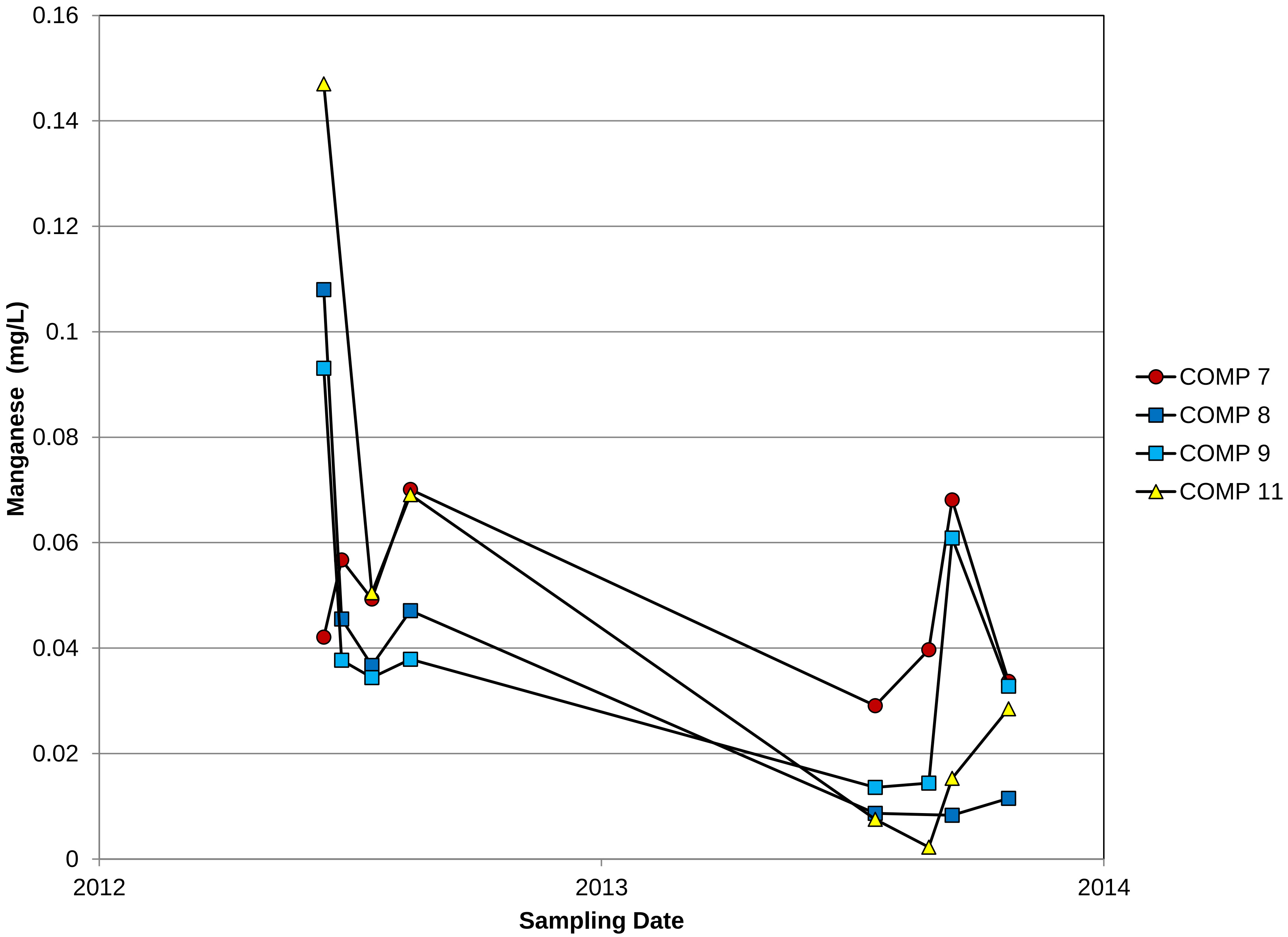

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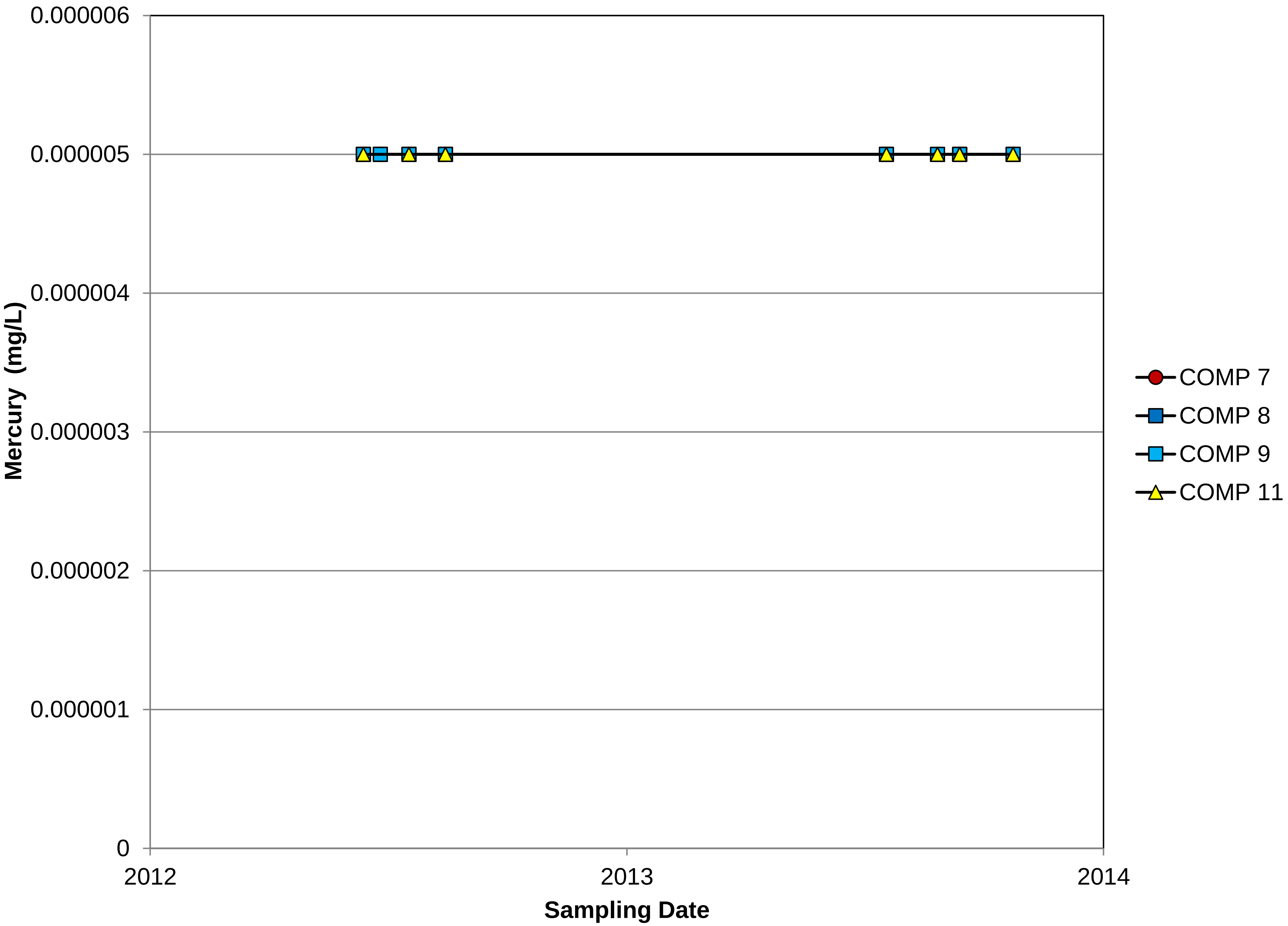

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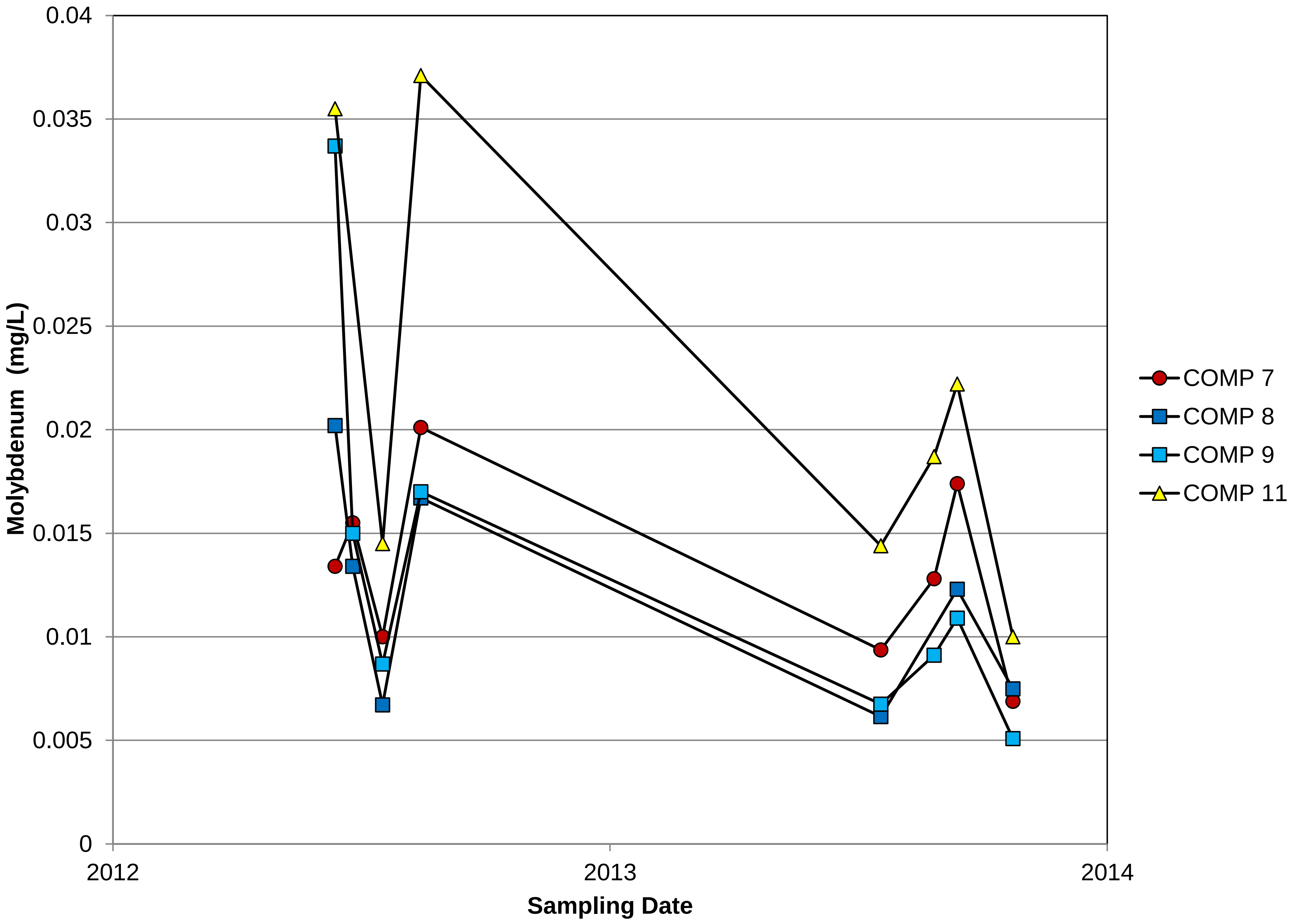

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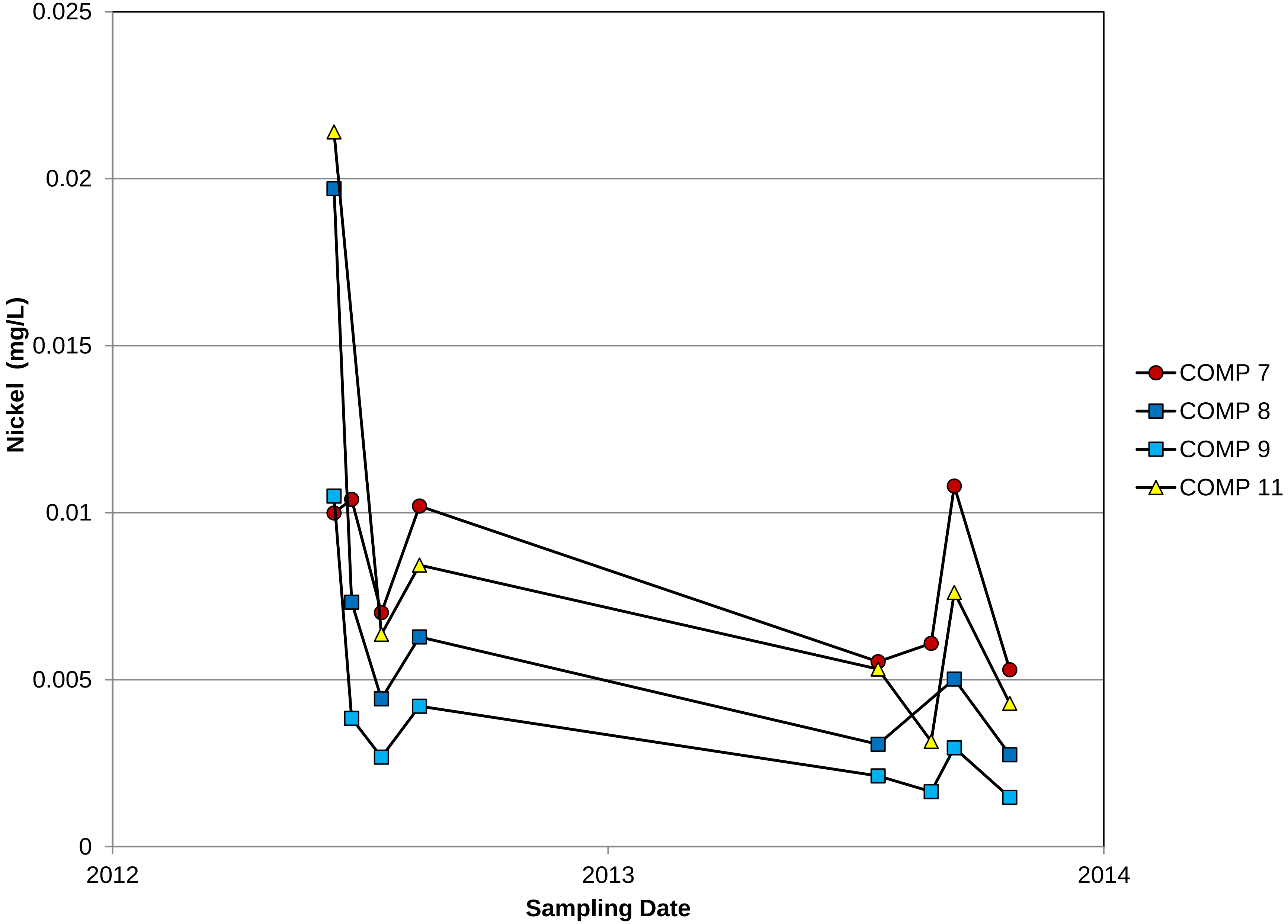

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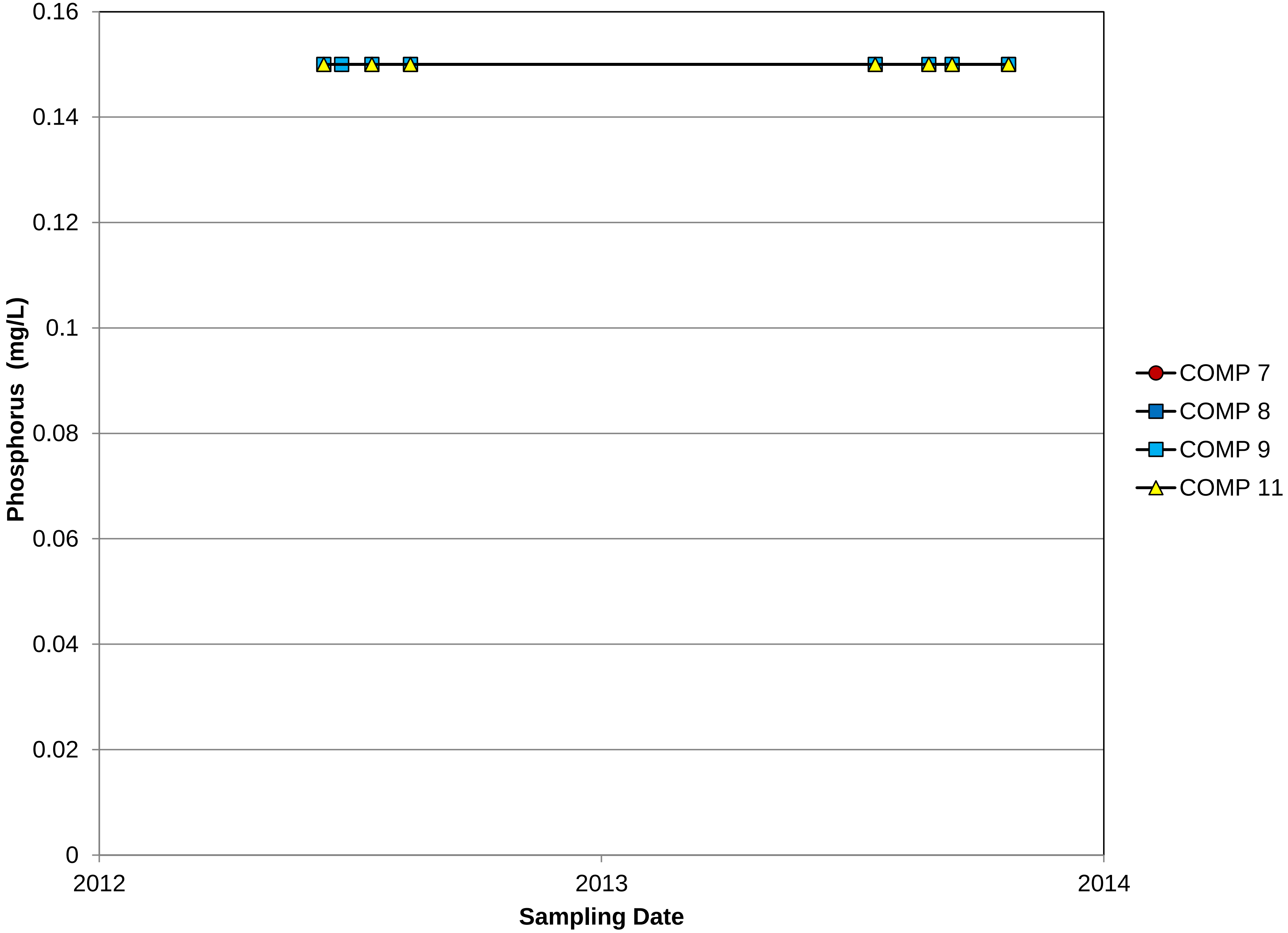

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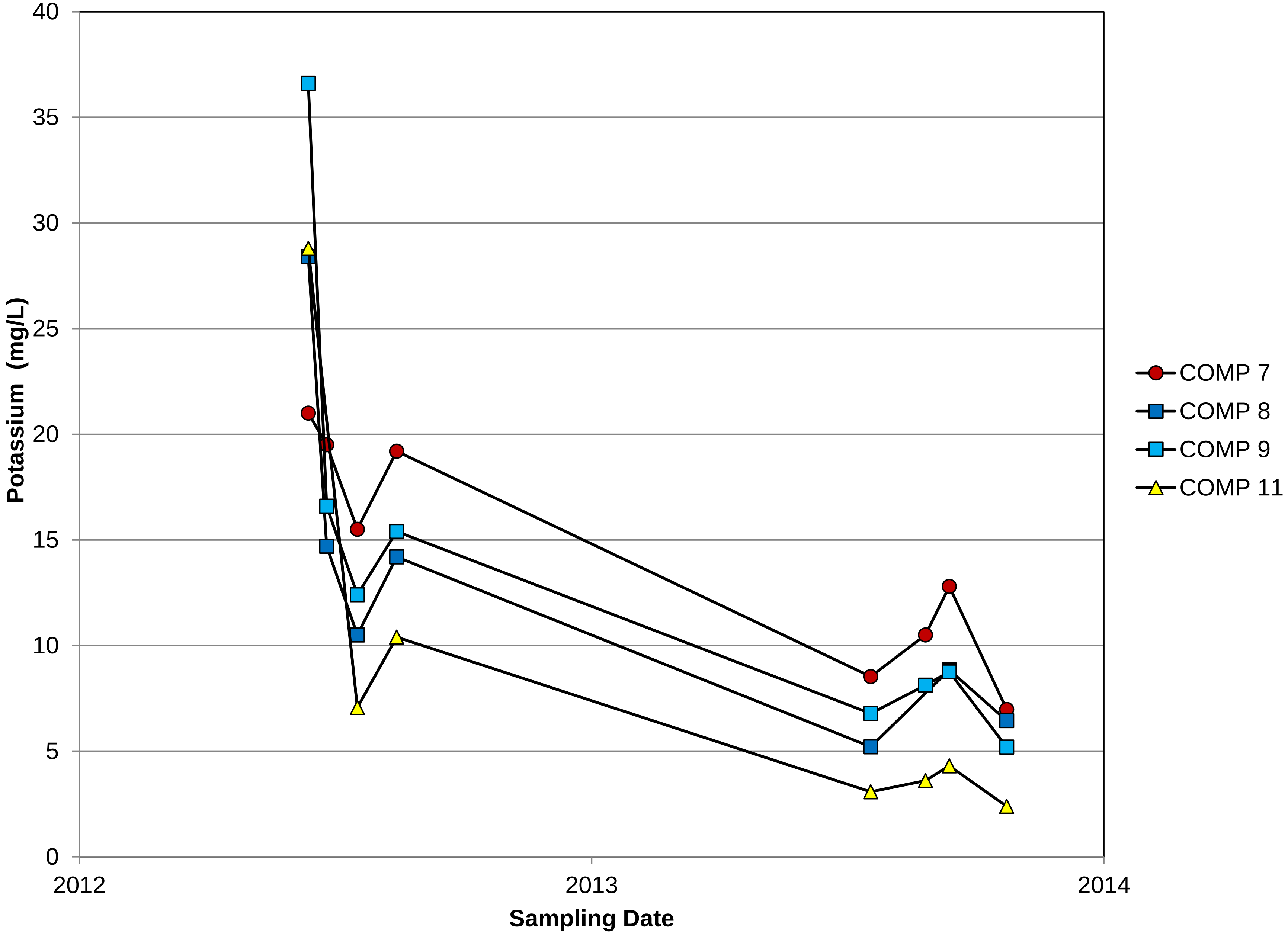

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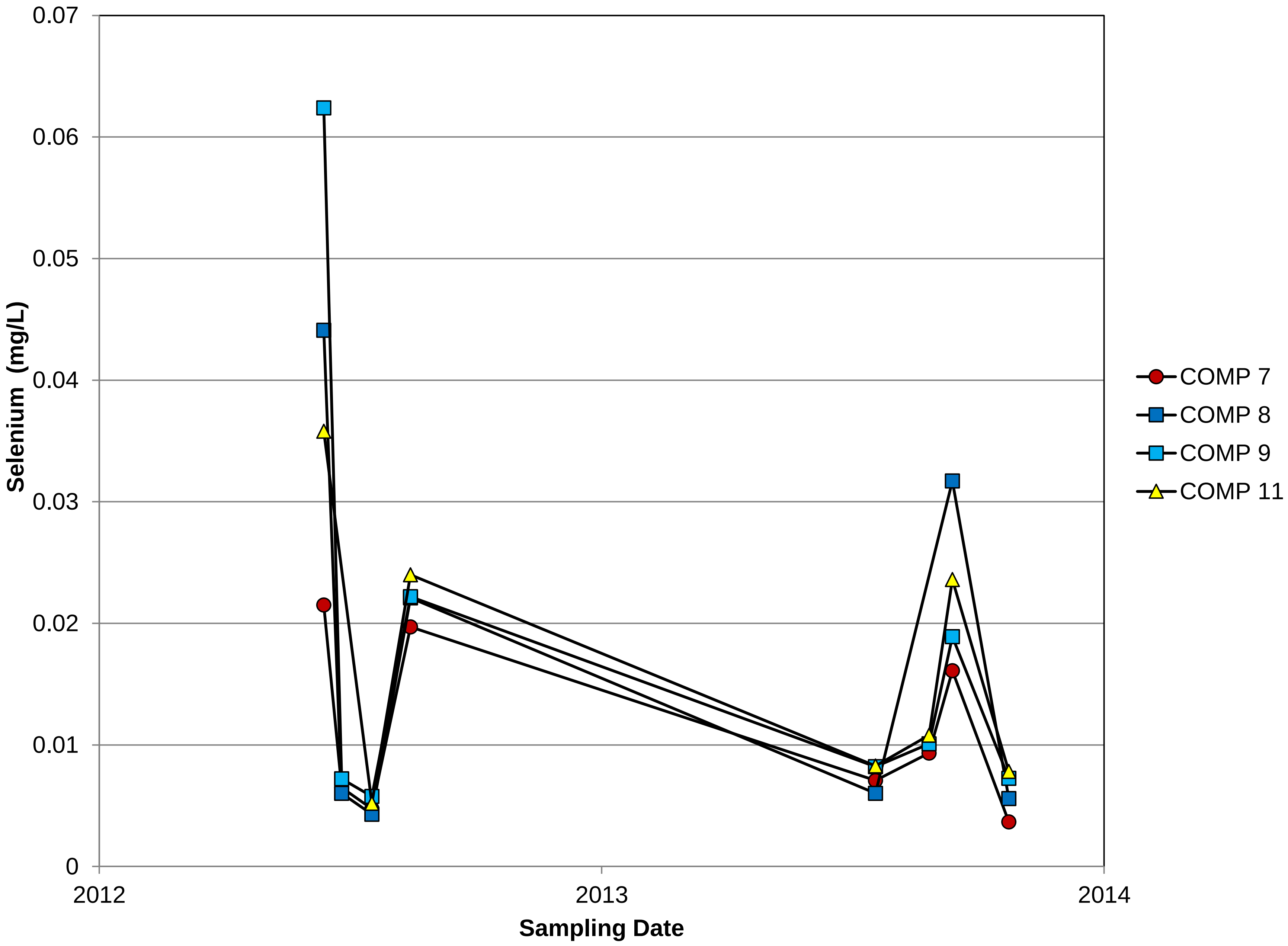

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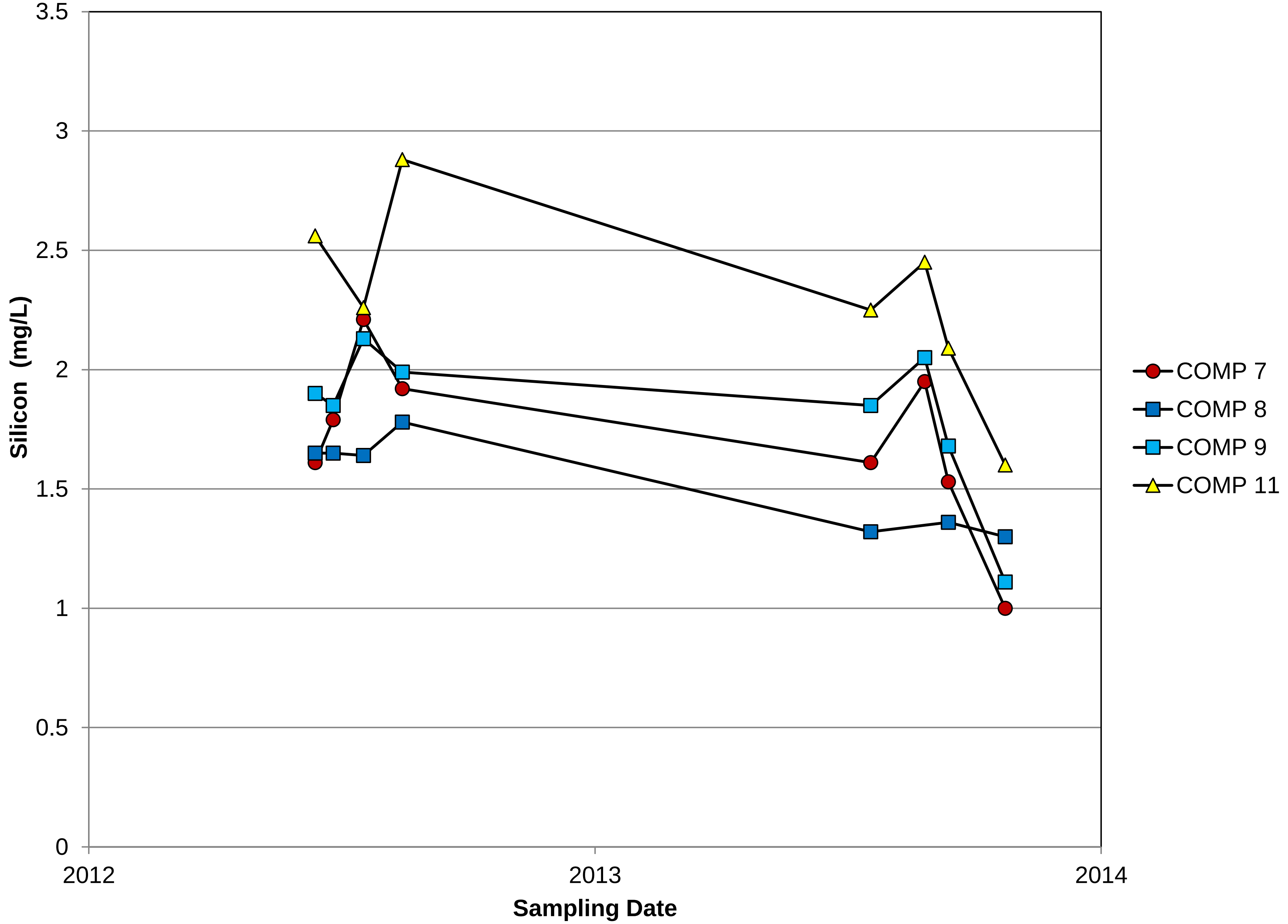

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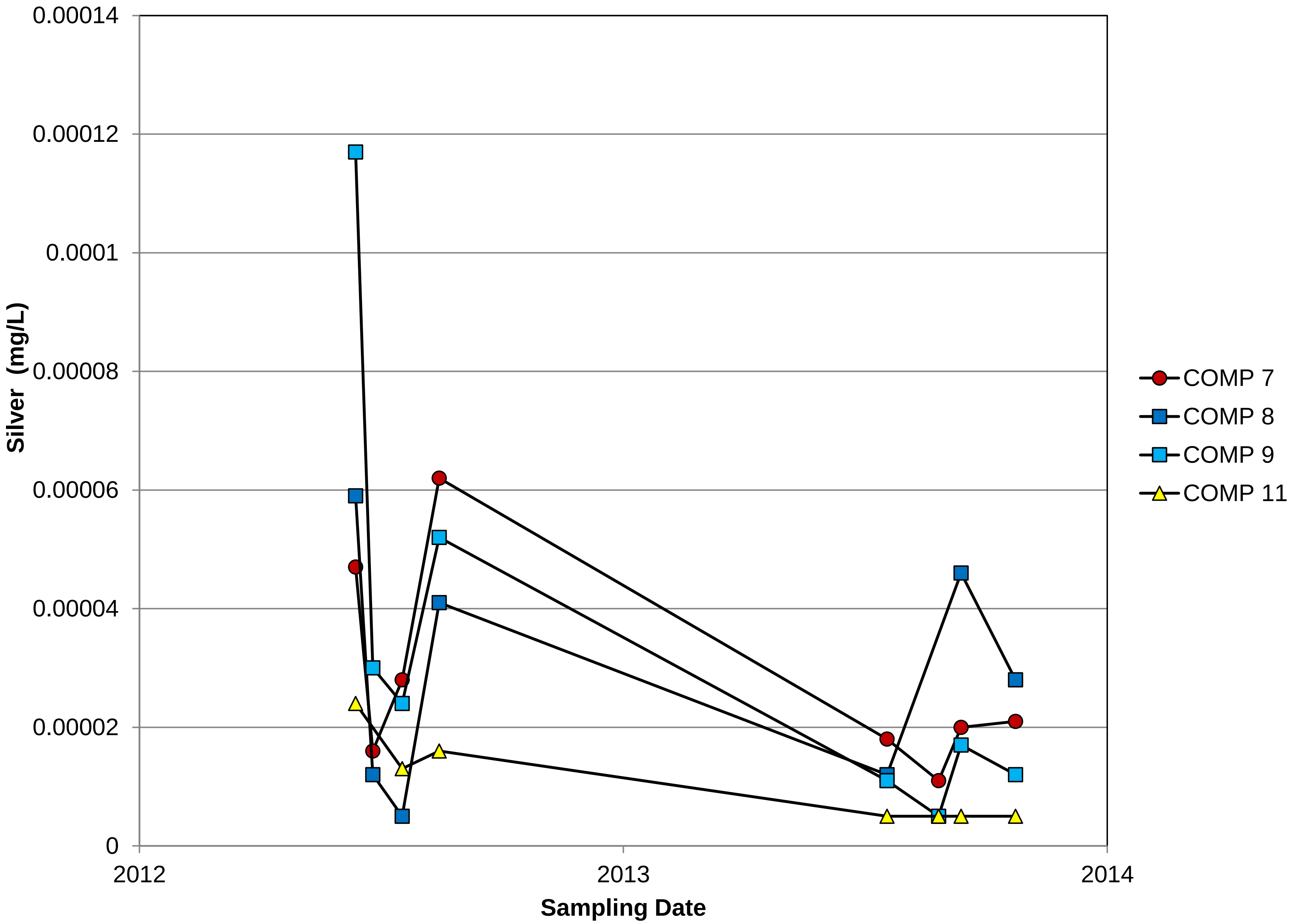

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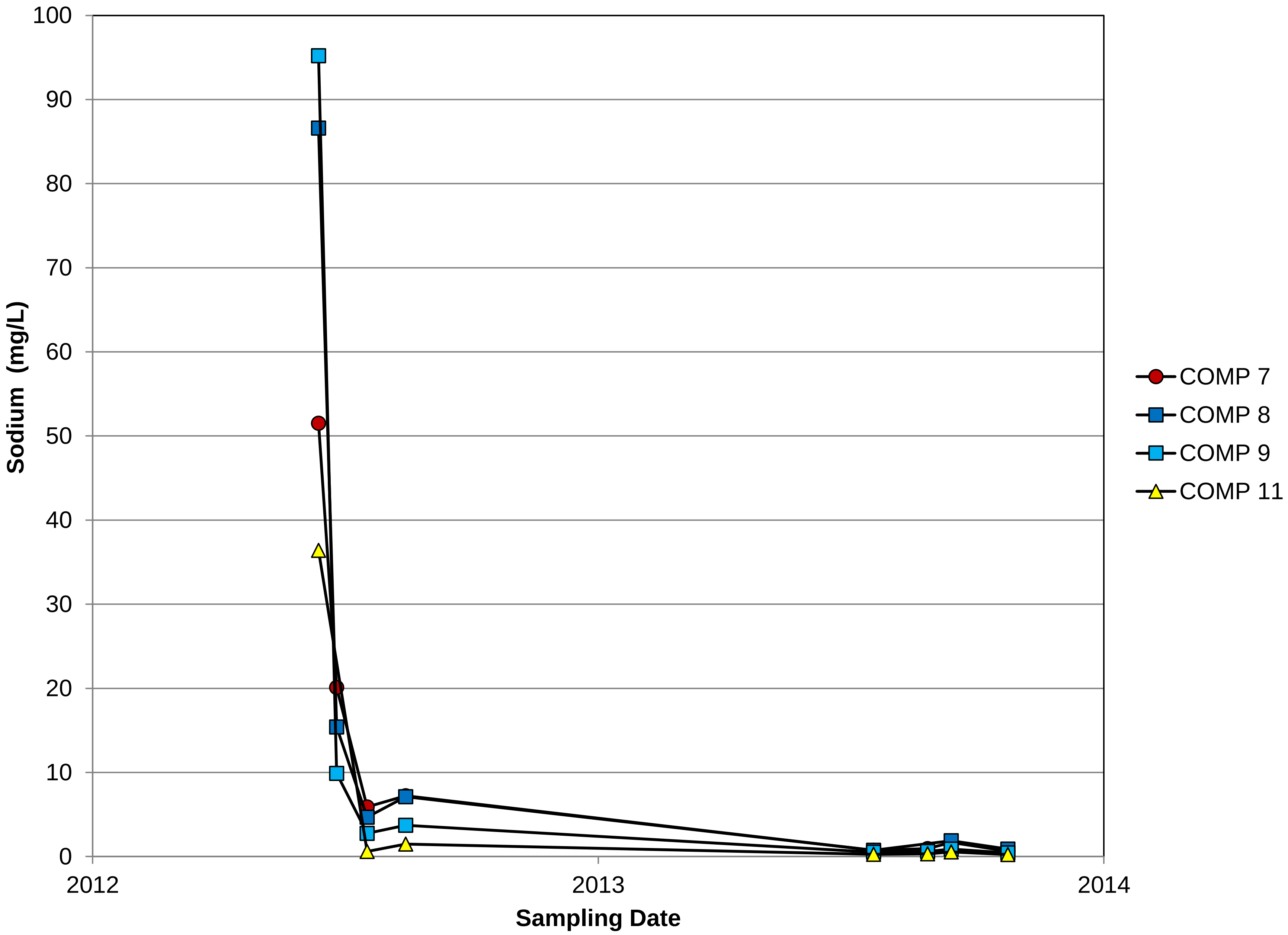

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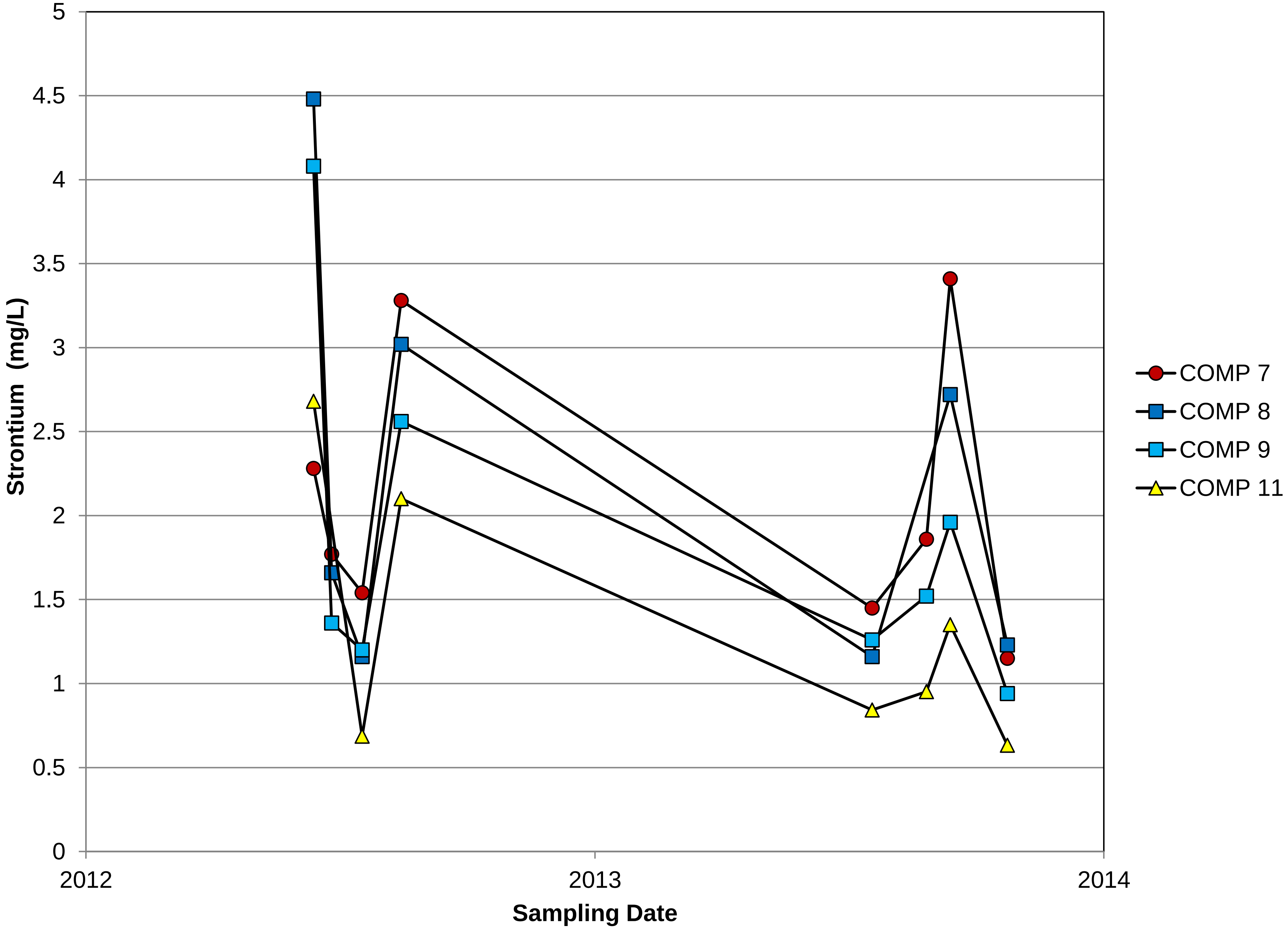

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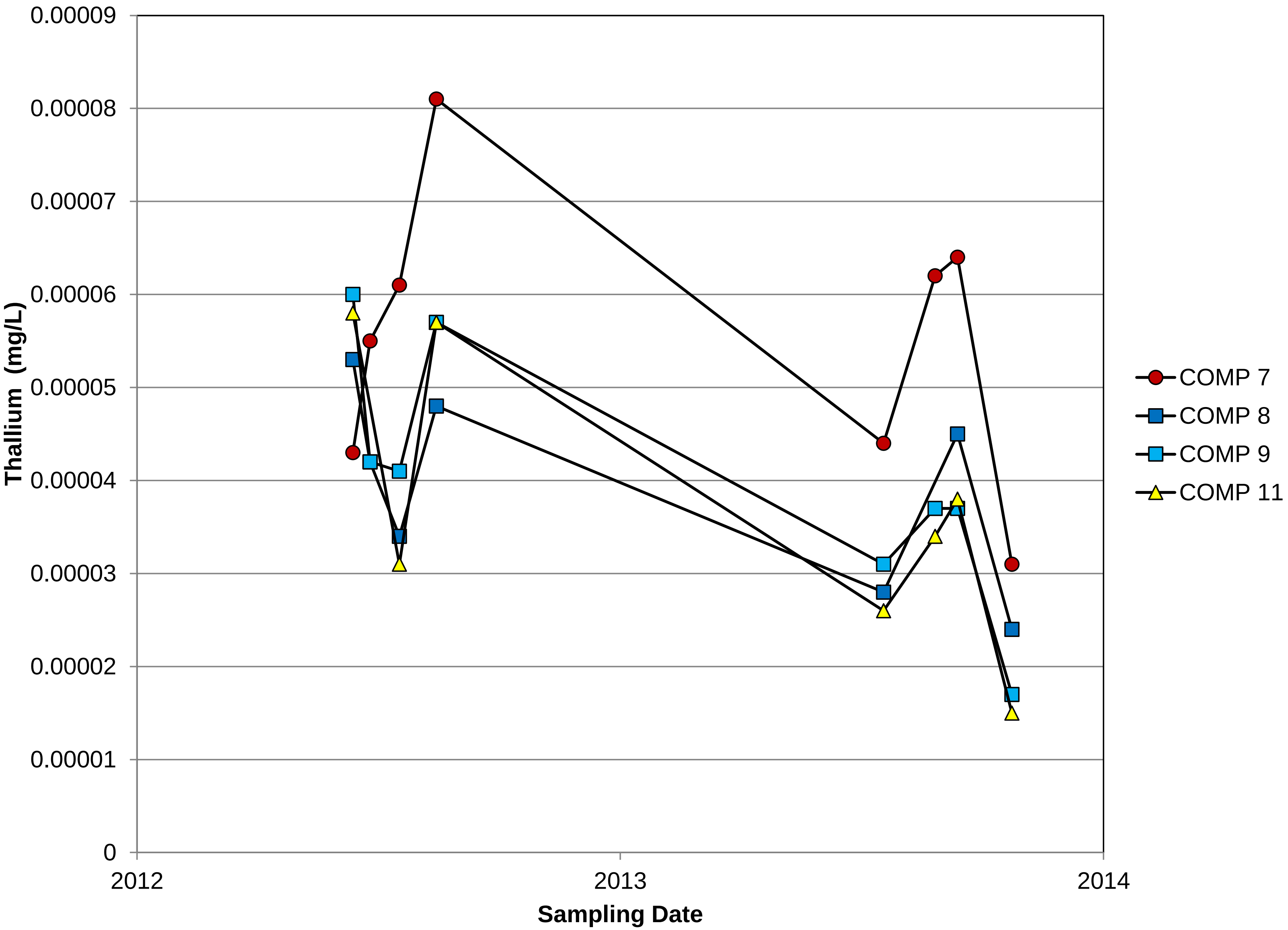

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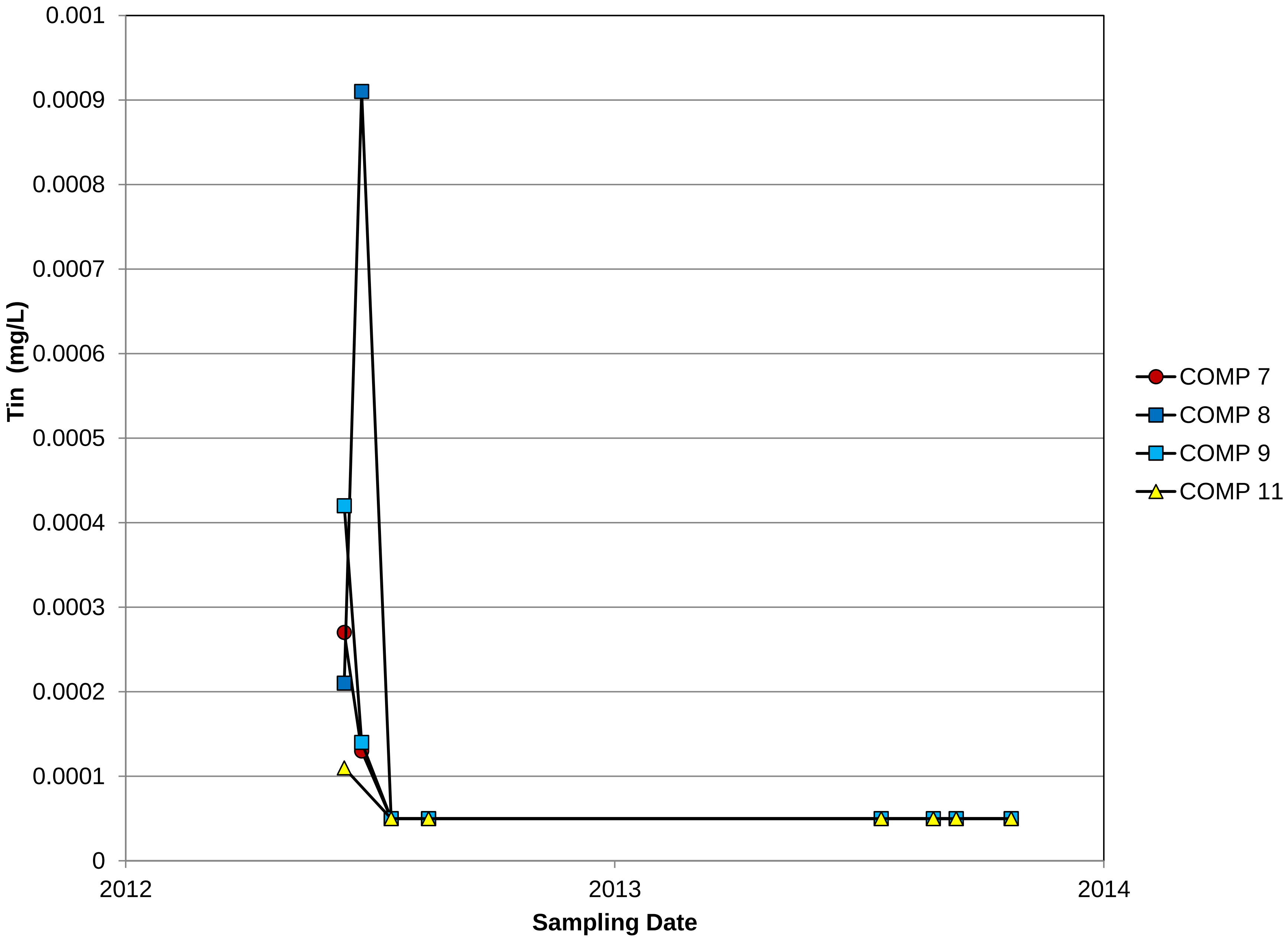

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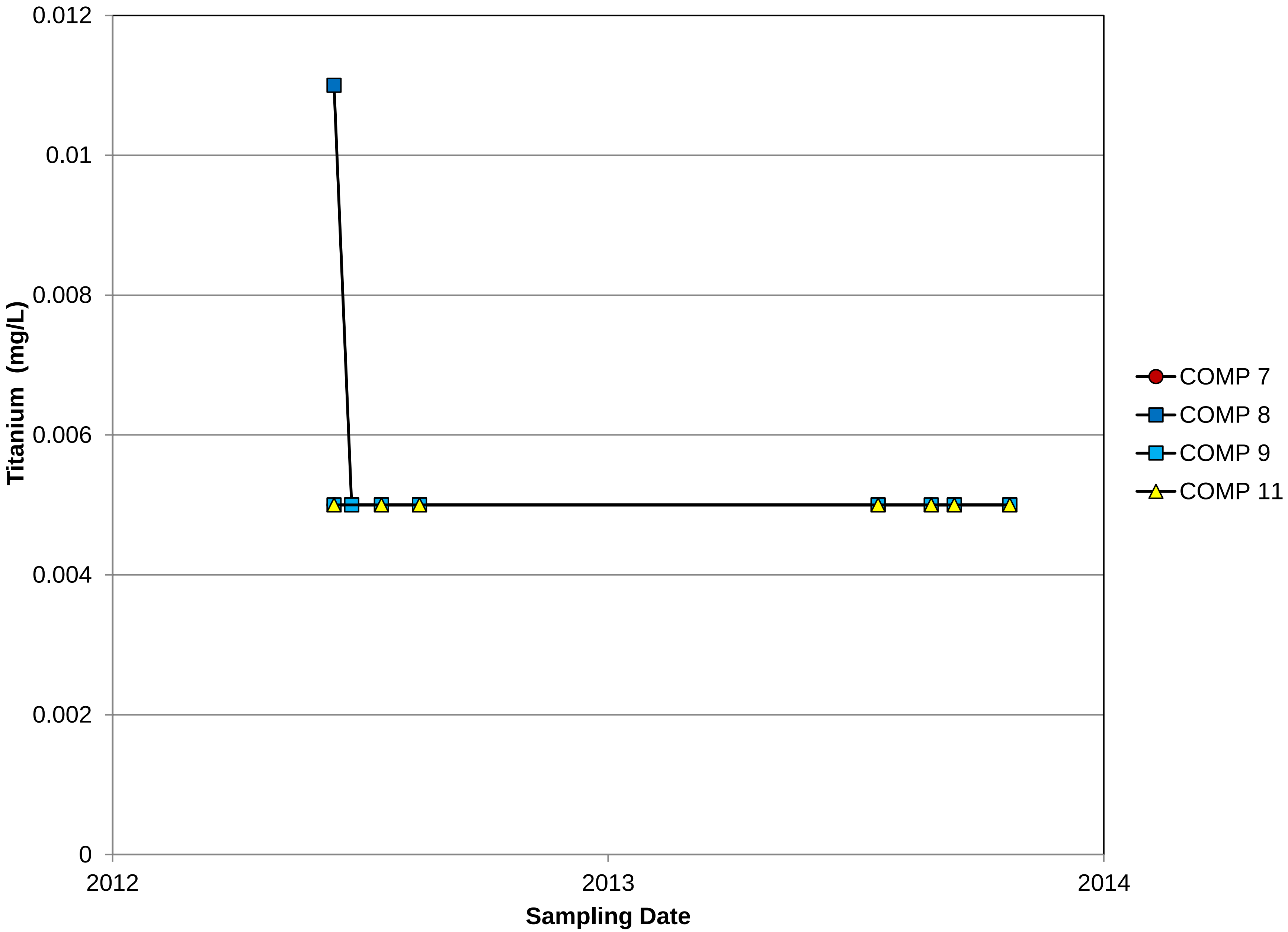

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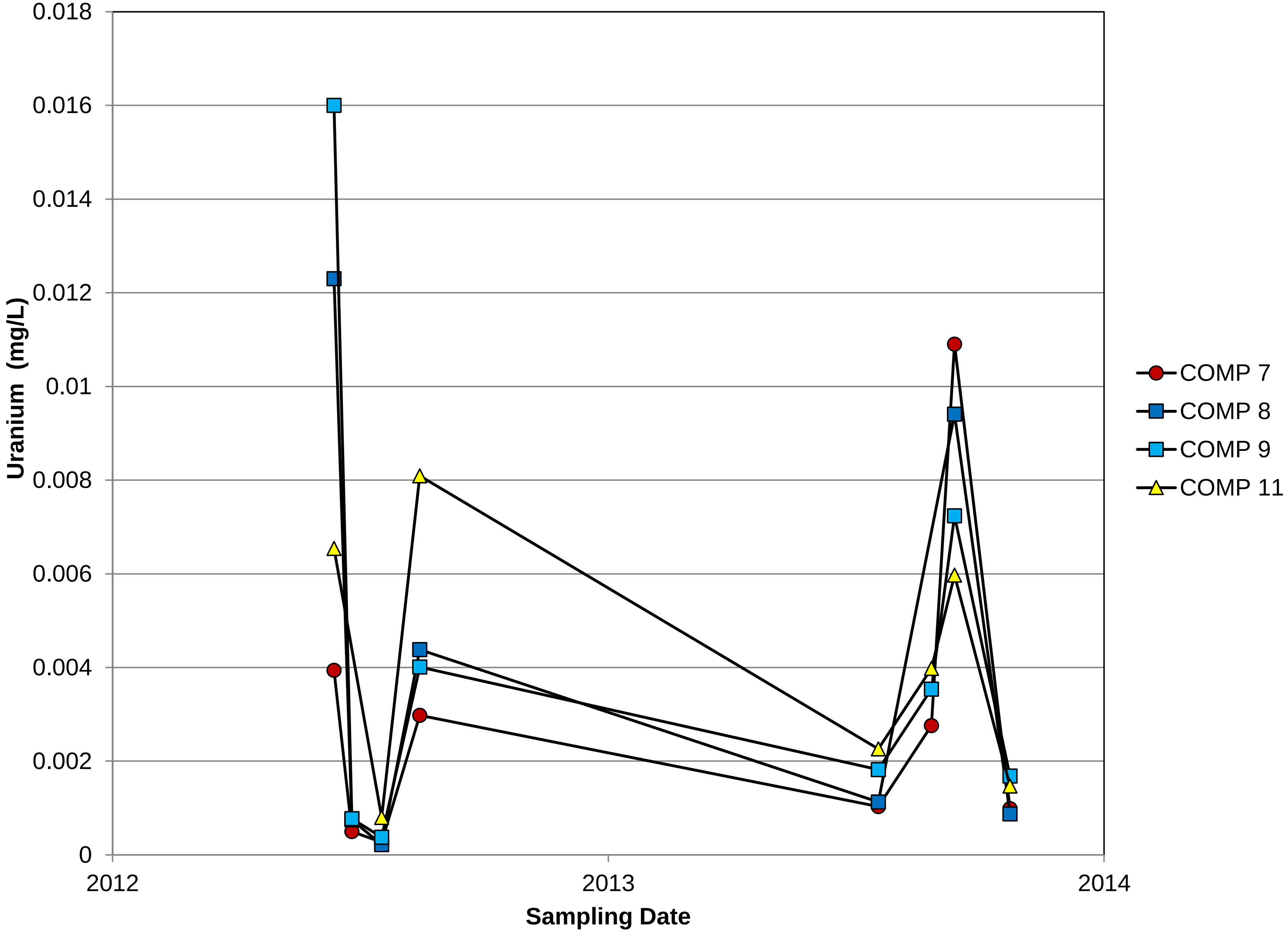

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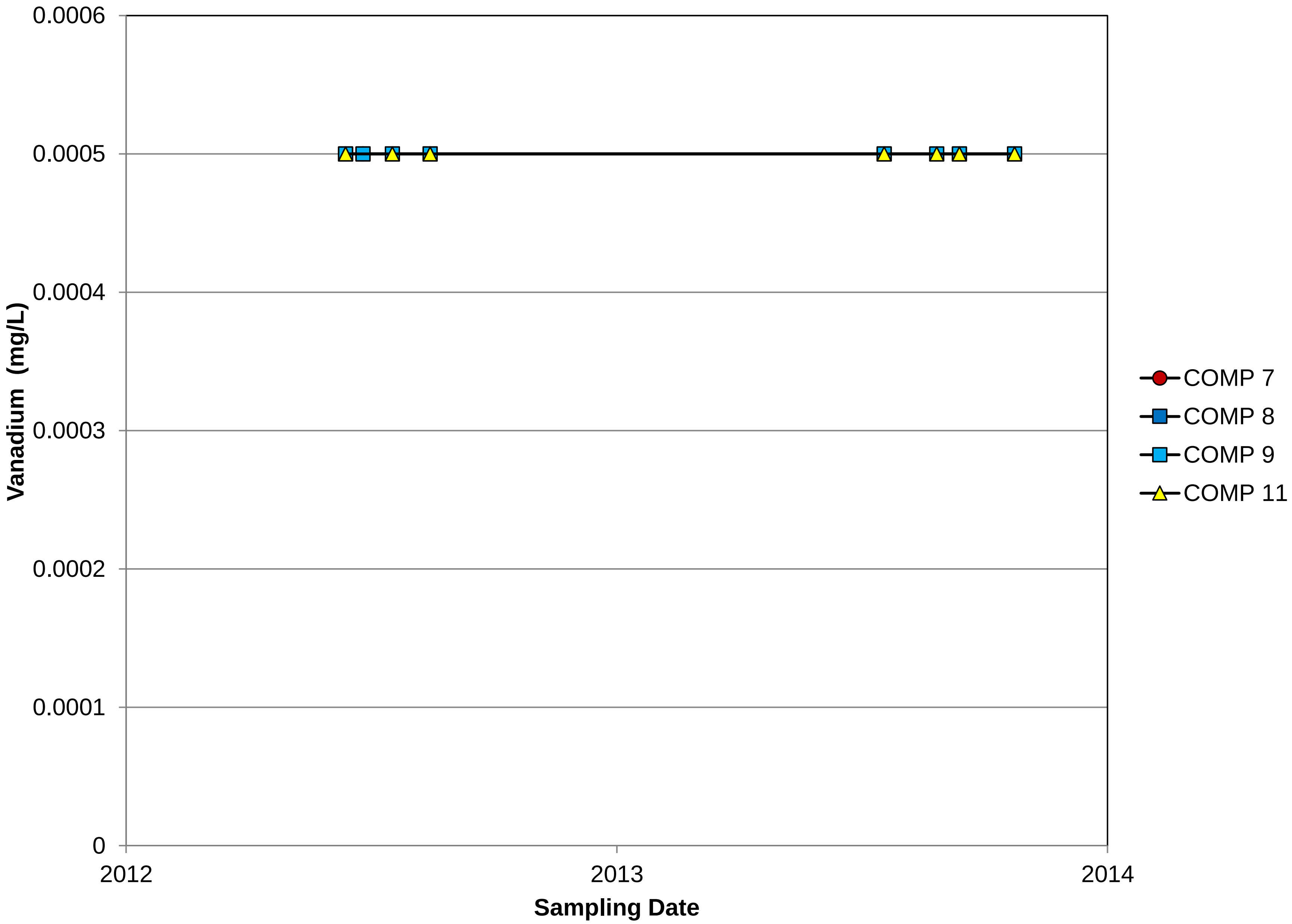

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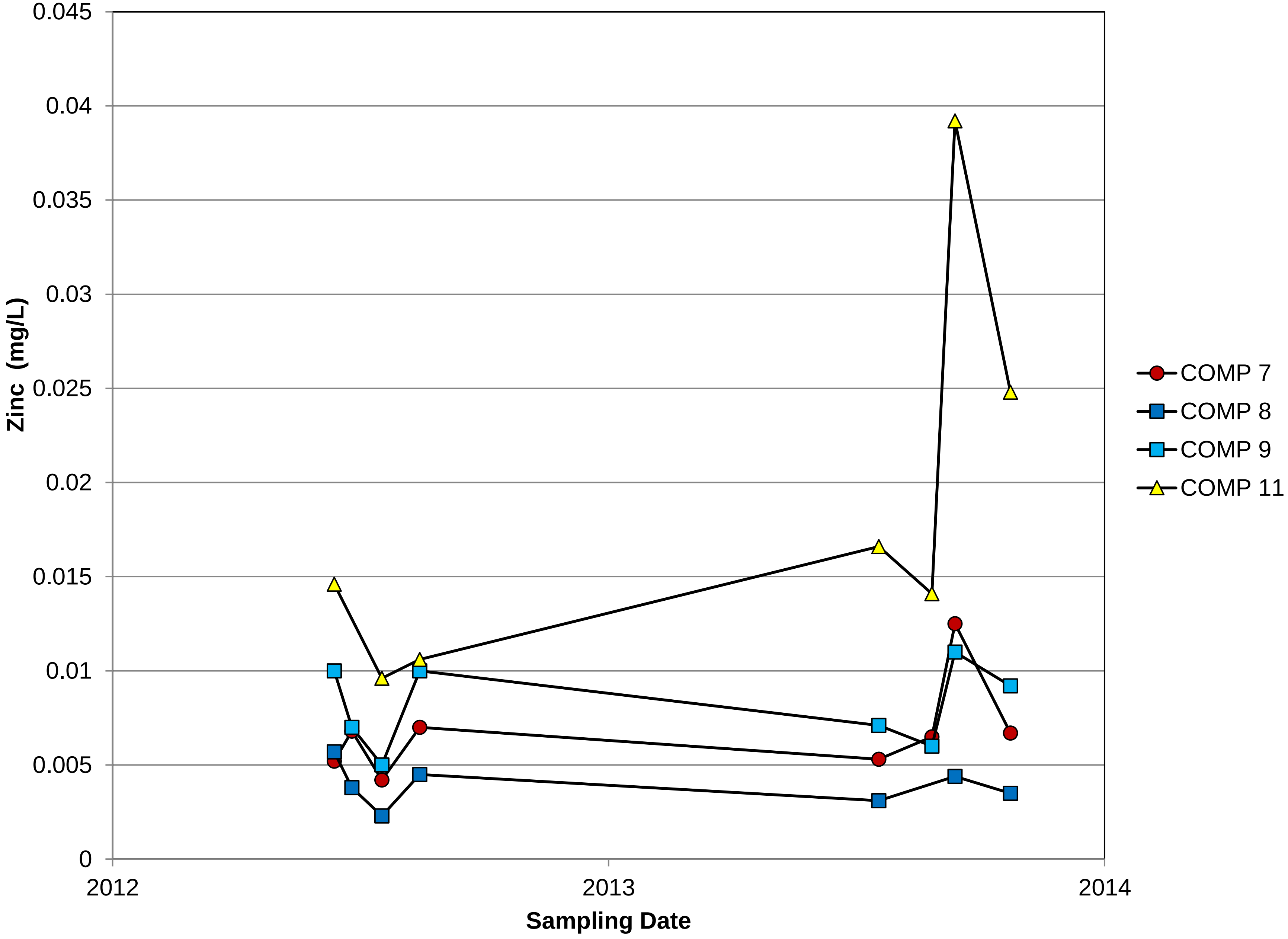

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Appendix D: Tailings

D1: Petrographic Report

Mineralogy Report

OPTICAL MINERALOGY &
RESULTS OF MICROPROBE TEST WORK (EPMA):

FOR SAMPLES:
KM 2916-14 CU 1CT
KM2916-14 CU ROTL

HARPER CREEK PROJECT, BC
CANADA

July 13, 2012

Maxxam Project #:
2-21-900

Prepared for:
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Background

Two tailings samples from the Harper Creek Project are characterized in this report (Maxxam Project No. 2-21-900). Optical reporting, carbonate and sulphide grain selection and microprobe test work (EPMA) were requested by Tim O'Hearn of Maxxam Analytics.

The samples were submitted for polished thin section production at Vancouver Petrographics Ltd. by Francis Chiu of Maxxam Analytics on May 25, 2012. The samples, polished thin sections and offcut mounts were then sent to Kathryn Dunne, P.Geo. for optical analysis. Petrography was undertaken on the samples, polished thin sections and offcut mounts by Kathryn Dunne, M.Sc., P.Geo (Petrography Report, June 29, 2012, revised). The purpose of the optical study was to characterize the mineralogy with particular emphasis on sulphide minerals and any carbonate minerals present.

This mineralogy report includes part of the petrography report of June 29, 2012 which has been revised to incorporate results of carbonate and sulphide microprobe test work (EPMA) at UBC. Photographs of the carbonate and sulphide grains selected for the EPMA are in Appendix 1 (this mineralogy report). Carbonate and sulphide determinations by Electron Probe Microanalysis (EPMA) were undertaken by Edith Czech at UBC in July. The EPMA data are listed in Appendix 2A & 2B (this mineralogy report). Microsoft Excel files with the same data will be sent directly via e-mail to Tim O'Hearn of Maxxam Analytics.

The optical observations are summarized below and petrographic descriptions of polished thin sections with representative photomicrographs follow the summary. All percentages in the descriptions are approximate based on visual estimation.

Conventions

Please note the following grain size conventions are used in the report:

very fine-grained (< 50 µm), fine-grained (> 50 µm and less than 1 mm),
medium-grained (> 1mm and < 5 mm), coarse-grained (> 5mm).

Definition of terms

Please note that the term “Fe-ox” is used in this report for unknown iron-oxides, -hydroxides, -oxyhydroxides, hydroxy-sulphate and/or oxyhydroxy-sulphate minerals. The term “rutile” is used in this report for very fine-grained Ti-oxides which may also include leucoxene and anatase. The term “sericite” is used in this report as a fine to very fine-grained colourless mica, either muscovite or illite.

Summary

The samples comprise mixed 1) fine to very fine liberated mineral grains, 2) fine rock fragments, 3) minor clumps of grains, rock fragments and powder and 4) rare tramp particles. Grains distinguished optically include major quartz, sericite and carbonate, minor to major pyrite, minor chlorite in one sample, trace to minor rutile and trace amounts of a number of other minerals. Rock fragments include variably sulphide-bearing, fine to very fine-grained quartz ±carbonate aggregate, sericite ±rutile aggregate, quartz- sericite ±plagioclase ±carbonate ±rutile aggregate. Some of the material in the sections is unidentified due to very fine-grain size of particles.

Carbonate occurs in major amounts ~ 15% as fine to very fine patchy aggregates in rock fragments and as liberated grains. Carbonate grains were tested by EPMA (Appendix 2A). Rarely, in sample KM 2916-14, Fe-ox occurs staining liberated carbonate fragments and carbonate adjacent to sulphide grains.

Sulphide occurs in minor to major amounts, (~3% and ~15%, respectively), as pyrite, traces of chalcopyrite and rarely as sphalerite, galena, and in some samples pyrrhotite or molybdenite. Pyrite occurs as anhedral liberated grains, as sub-anhedral grains in rock fragments and in powder clumps. Pyrite grain boundaries are typically irregular but unaltered. Pyrite grains were tested by EPMA (Appendix 2B). Chalcopyrite occurs as anhedral grains disseminated in rock fragments and as inclusions in pyrite. Chalcopyrite grains are unaltered.

Summary cont.
Rare particles of tramp Fe ±Cr ±Ni, (±Si ±S) occur as scattered liberated particles and within clumps of mineral grains in the samples. These tramp particles were described as unknown particles in the petrographic report of June 29, 2012 and tested by EPMA (Appendix 2B). Some of the tramp particles are partly replaced by red-brown Fe-ox material.

Traces of red-brown Fe-ox occurs as liberated grains and aggregates. In sample KM 2916-14 Cu ROTL, traces of red-brown Fe-ox occurs as anhedral aggregates within matrix of some powder clumps and as liberated aggregates. Rarely in sample KM 2916-14 Cu ROTL, Fe-ox occurs partly replacing pyrrhotite and tramp material.

Tabular summary (Maxxam Project No. 2-21-900):

Sample #	**Sulphide**	**% ~**	**Carbonate occurrence**	**% ~**	**Fe-ox* occurrence**	**% ~**
KM 2916 -14 Cu ROTL	pyrite chalcopyrite sphalerite pyrrhotite galena	3 tr r. r. r.	liberated grains, within rock fragments, in powder clumps	15	red-brown, unknown	tr
KM 2916 -14 Cu 1CT	pyrite chalcopyrite sphalerite galena molybdenite	15 tr r. r. r.	liberated grains, within rock fragments, in powder clumps	15	red-brown, unknown	r.

tr = trace (< 1%); r. = rare; x = none observed

Maxxam Project #: 2-21-900
Sample ID: KM 2916 -14 Cu ROTL

Description of powder:

Yellowish-grey powder. No reaction to magnet. Very weak reaction to cold, dilute HCl. No reaction of offcut mount to etching with HF and staining with sodium cobaltinitrite solution (no yellow stain).

Polished Thin Section Description:

Mineral	%	Grain Size (min) mm	Grain Size (max) mm	Description
Quartz	50	< 0.01	0.5	liberated grains, within rock fragments, in powder clumps
Sericite	20	< 0.01	0.025	liberated flakes, within rock fragments and in powder clumps
Carbonate	15	<0.01	0.55	liberated grains, within rock fragments, in powder clumps
Chlorite	3	< 0.01	0.2	liberated plates, within rock fragments and in powder clumps
Rutile	1	<0.005	0.010	aggregate within rock fragments
Albite	tr	0.05	0.075	aggregate within rock fragments
K-feldspar	tr	0.05	0.4	cloudy aggregate within rock fragments
Fe-ox	tr	<0.005	<0.005	red-brown aggregate, patchy cement in powder clumps
Clinopyroxene	rare		0.6	liberated grain
Tramp material	rare		0.05	liberated particles, within powder clump
Unidentified	~7	<0.01	<0.01	

Sulphide Mineral	%	Grain Size (min) mm	Grain Size (max) mm	Description
Pyrite	3	0.007	0.7	liberated grains, within rock fragments, in powder clumps
Chalcopyrite	trace	0.02	0.2	subhedral grains in rock fragments, inclusions in pyrite, within powder clump
Sphalerite	rare		0.025	within carbonate aggregate
Pyrrhotite	rare		0.2	anhedral aggregate with carbonate
Galena	rare		0.035	liberated grain

Mixed fine to very fine liberated grains (typically < 0.3 mm, maximum 0.7 mm), rock fragments (up to 0.7 mm) and minor clumps of grains, rock fragments and powder (up to 2 mm clumps. Grains that can be distinguished optically include major quartz, sericite and carbonate, minor chlorite, pyrite and rutile and traces of feldspar, Fe-ox aggregate and chalcopyrite. Sphalerite, pyrrhotite, galena, clinopyroxene and tramp material occur rarely. Approximately 7% of the section is unidentified due to very fine-grain size of particles. Rock fragments include fine-grained quartz aggregate, sericite±rutile aggregate and quartz-sericite±plagioclase±carbonate aggregate.

Carbonate occurs in major amounts ~ 15% as fine to very fine (< 0.55 mm) patchy aggregates in rock fragments and as liberated grains. Rarely, carbonate grains are stained red-brown by Fe-ox aggregate.

Sulphide occurs in minor amounts, ~3%, as pyrite, traces of chalcopyrite and rarely as sphalerite, pyrrhotite and galena. Pyrite occurs as anhedral liberated grains, as sub-anhedral grains in rock fragments and in powder clumps. Pyrite grain boundaries are typically irregular but unaltered. Chalcopyrite occurs in trace amounts as anhedral grains disseminated in rock fragments and as inclusions in pyrite. Chalcopyrite grains are unaltered.

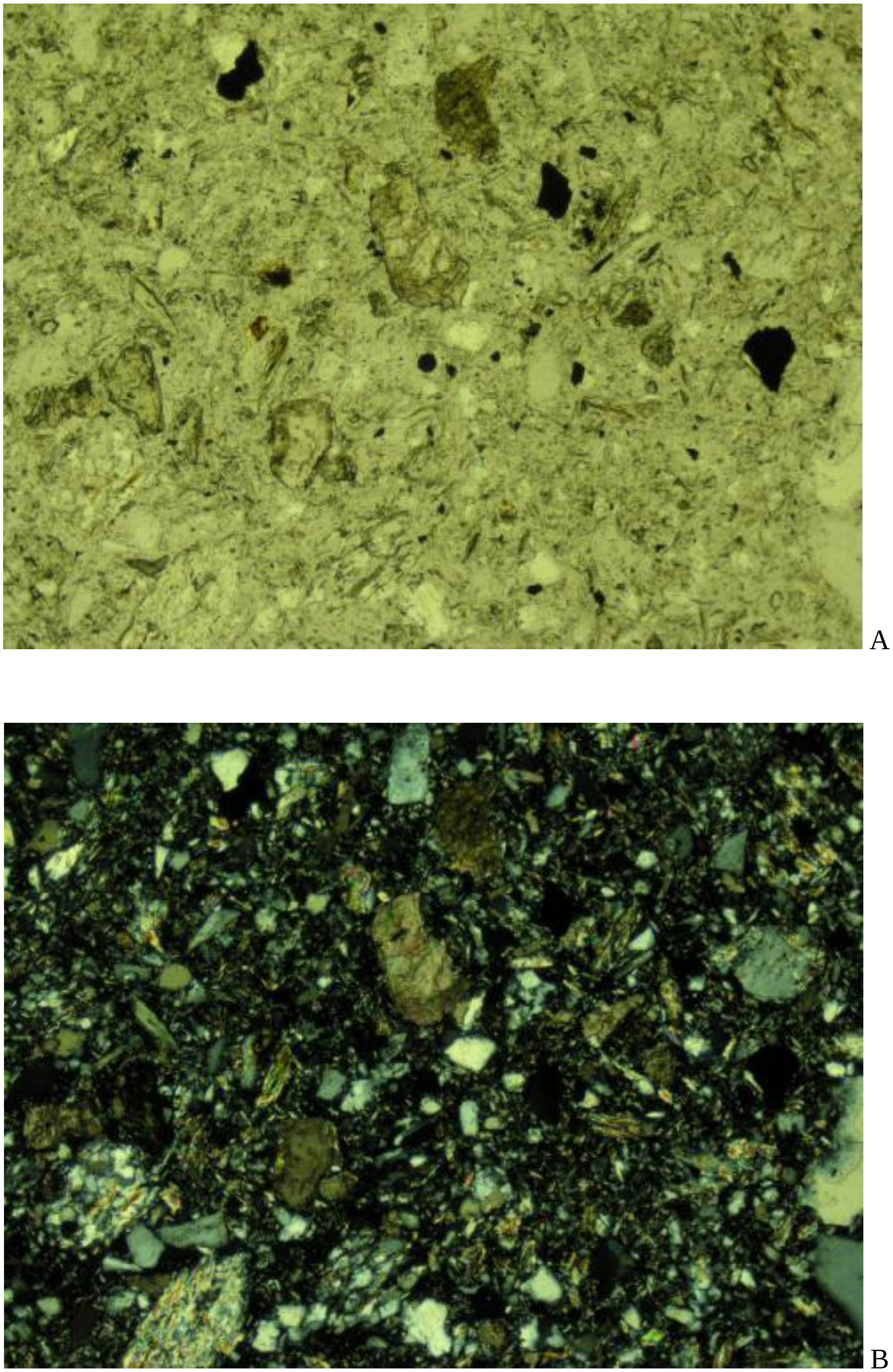
Maxxam Project #: 2-21-900
Sample ID: KM 2916 -14 Cu ROTL

Polished Thin Section Description: (cont.)

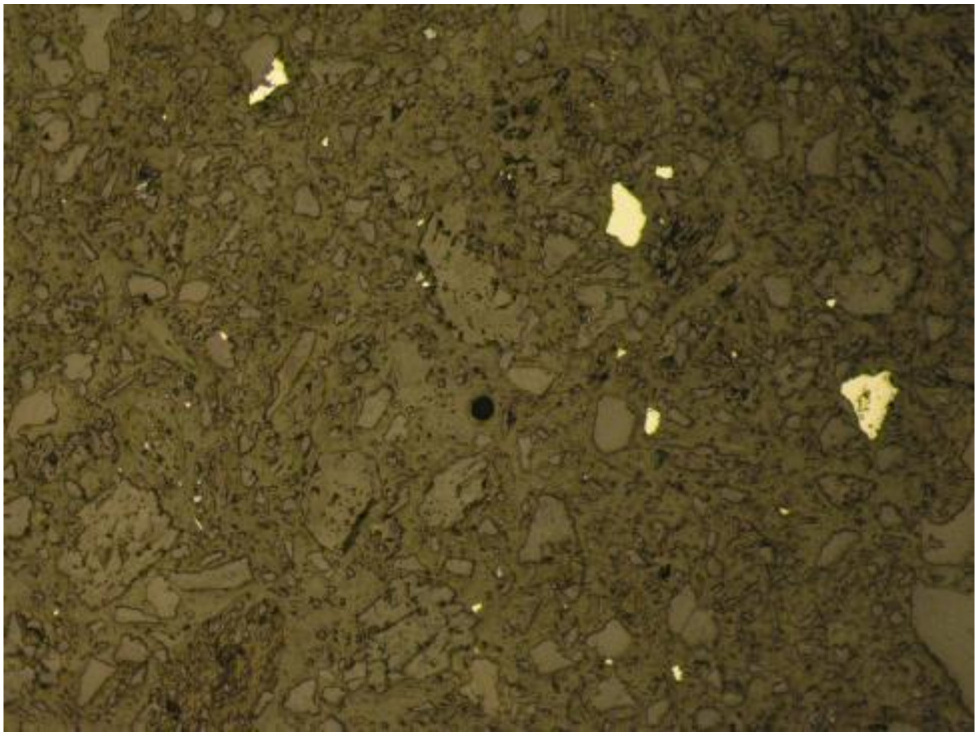
Pyrrhotite occurs rarely associated with carbonate in one fragment. Pyrrhotite grains are weakly altered along rims and fractures. Sphalerite occurs rarely associated with chalcopyrite and carbonate. Galena occurs rarely as one liberated very fine grain.

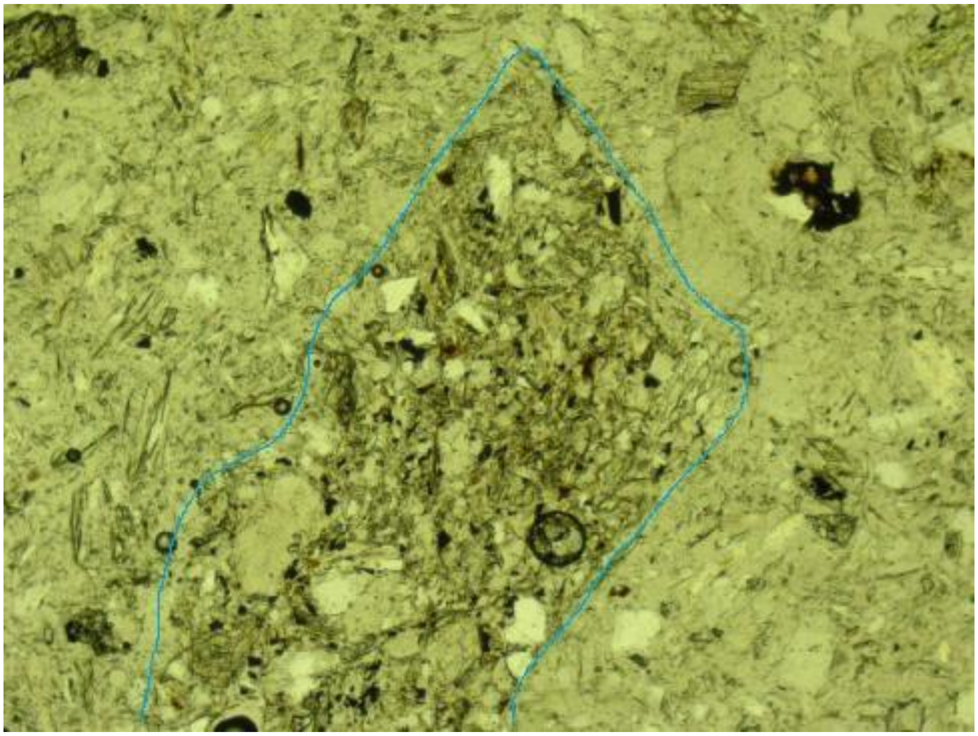
A particle of tramp Fe occurs within a clump of mineral grains in the sample (Photo H). Other tramp Fe ±Cr (±Si) particles occur as liberated grains. The tramp particles were described as unknown particles in the petrographic report of June 29, 2012 and tested by EPMA (Appendix 2B). The tramp Fe particle is partly replaced by red-brown Fe-ox material.

Traces of red-brown Fe-ox occurs as anhedral aggregates within matrix of powder clumps and as liberated aggregates. Rarely Fe-ox occurs partly replacing pyrrhotite and tramp material. Rarely, Fe-ox occurs staining liberated carbonate fragments and carbonate adjacent to sulphide grains.

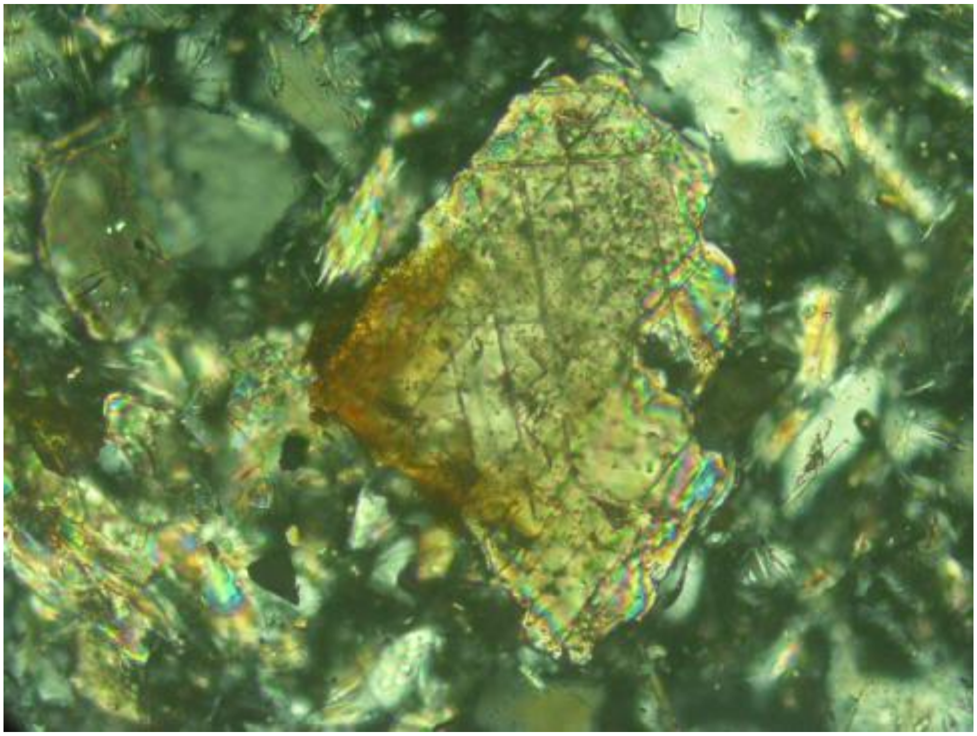


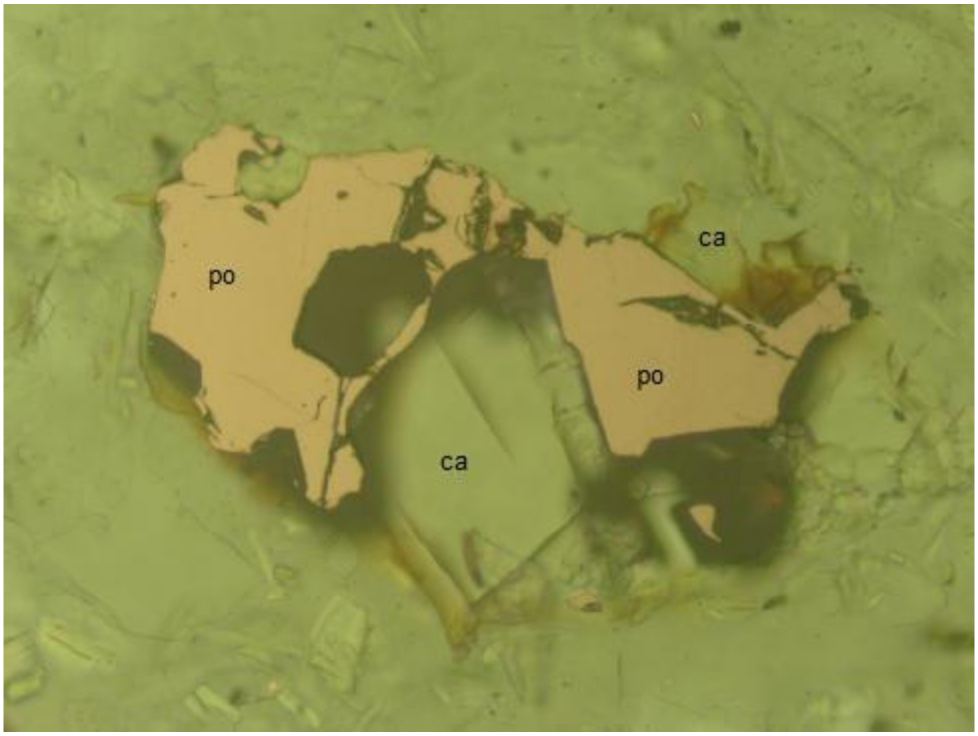
KM 2916 -14 Cu ROTL: A&B) Representative fine to very fine grains, rock fragments and clumps of powder and grains. A) PPL, B) XPL, FOV = ~ 2.6 mm.



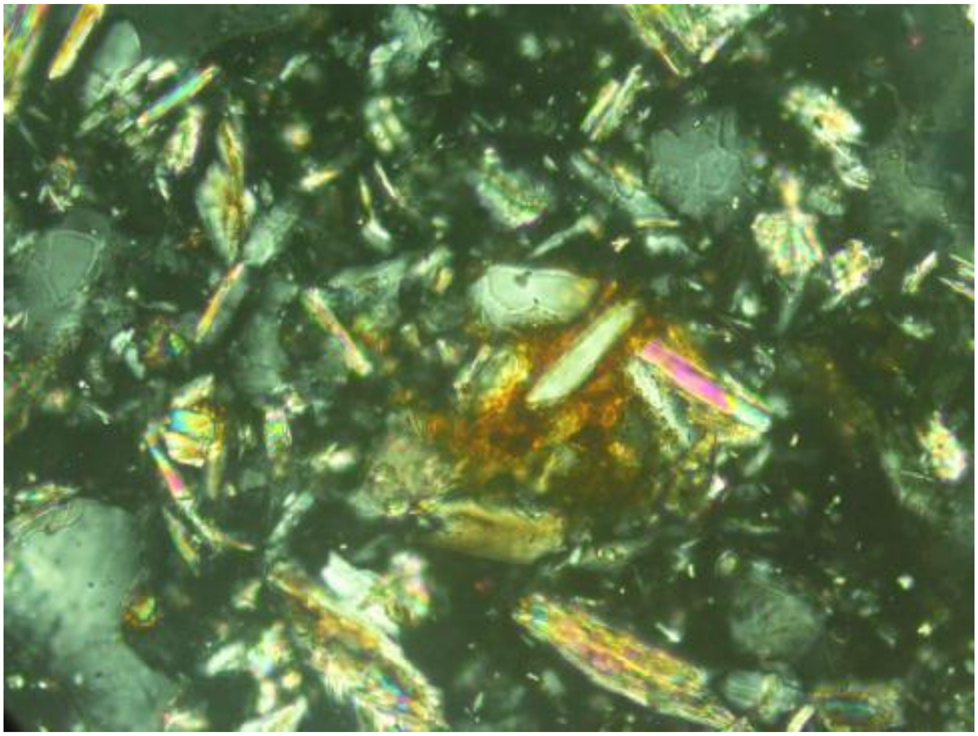


KM 2916 -14 Cu ROTL: C) Representative fine to very fine grains and clumps of powder and grains. (same view as photos A & B). RL, FOV =~ 2.6 mm, D) Clump of mineral grains and powder (blue outline). PPL, FOV =~ 2.6 mm.

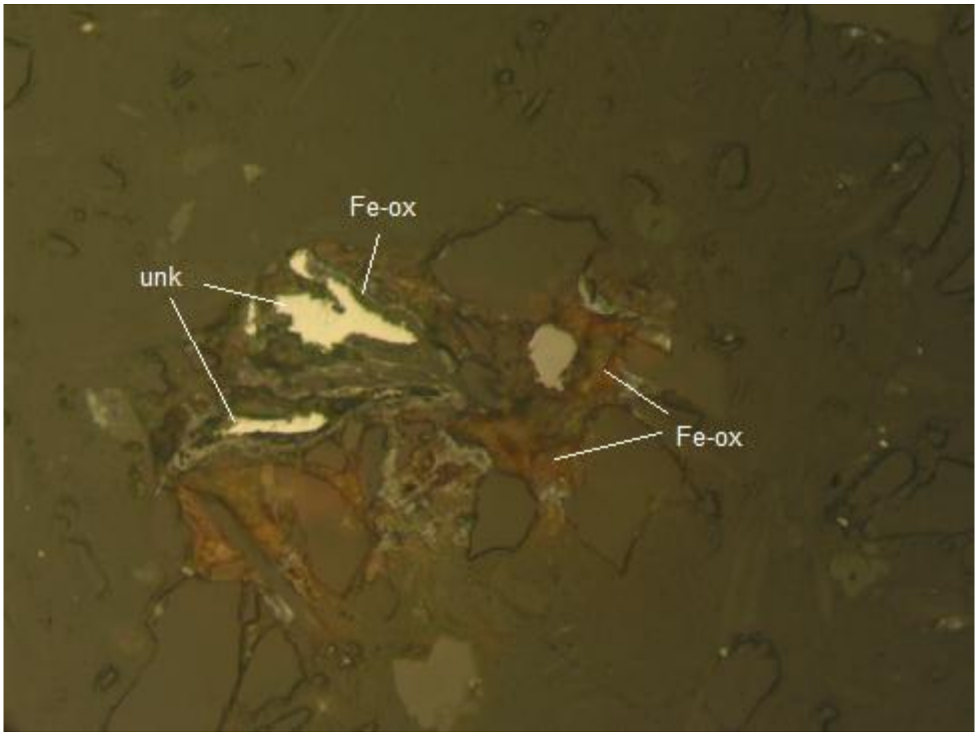




KM 2916 -14 Cu ROTL: E) Carbonate grain (centre) partly replaced by red-brown Fe-ox aggregate. XPL. FOV =~ 0.3 mm. F) Weakly altered pyrrhotite (po) aggregate associated with carbonate (ca). Note carbonate adjacent to pyrrhotite is stained red-brown by Fe-ox aggregate. PPL+RL. FOV =~ 0.3 mm.

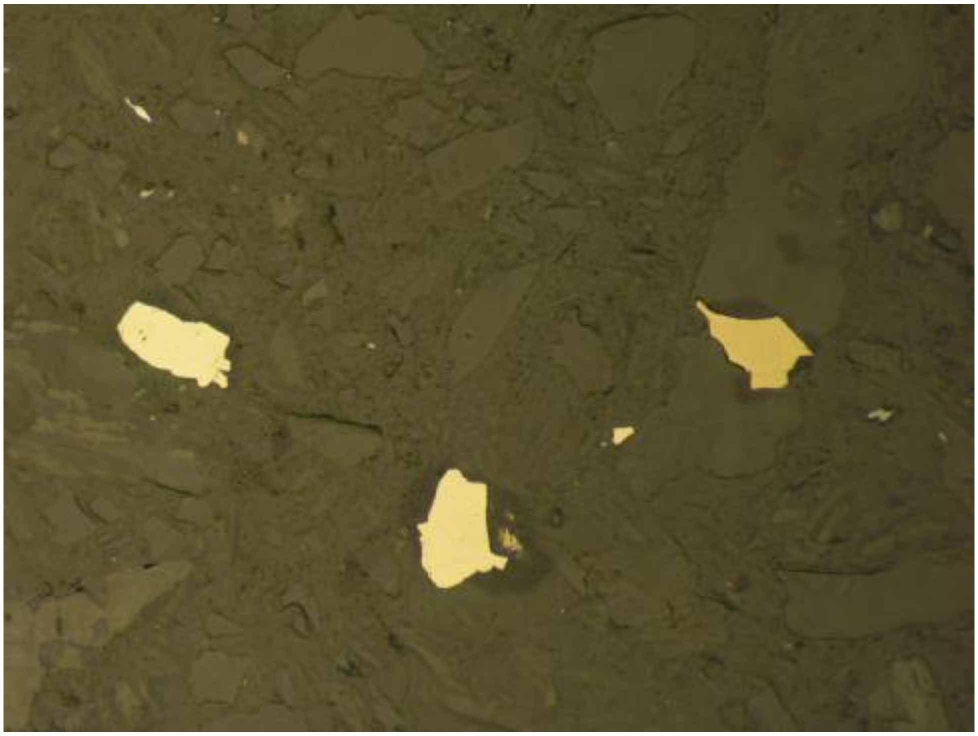


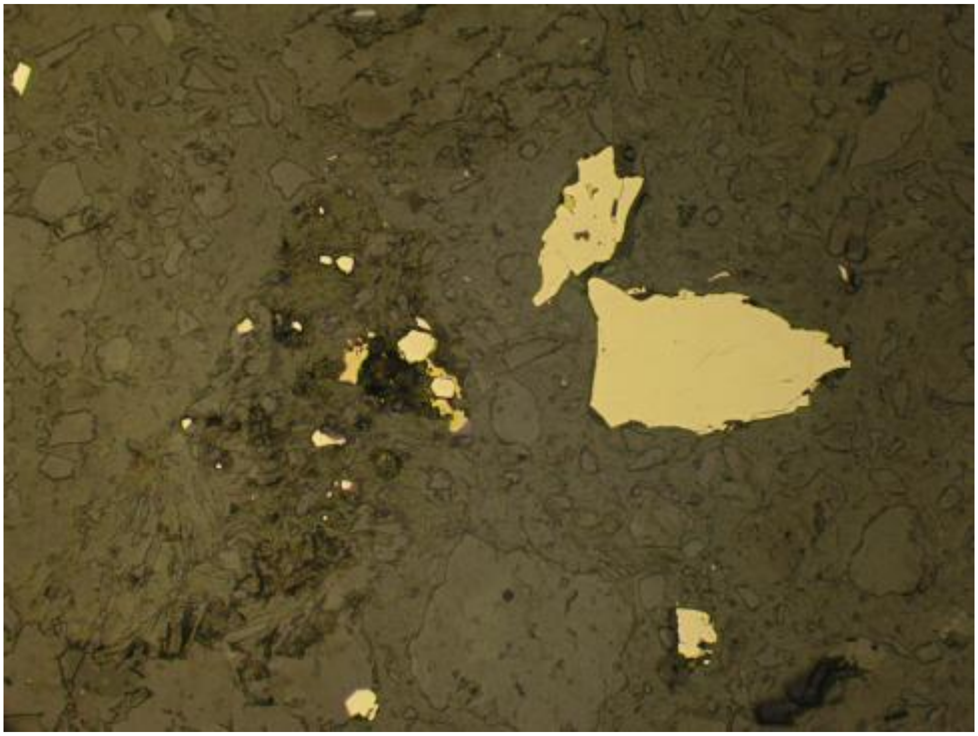
G



H

KM 2916 -14 Cu ROTL: G) Red-brown Fe-ox aggregate as cement to quartz and sericite grains. XPL. FOV =~ 0.3 mm. H) Tramp particle (labeled unk) is partly replaced by Fe-ox aggregate within clump of mineral grains cemented by Fe-ox aggregate. RL. FOV = ~0.3 mm.





KM 2916 -14 Cu ROTL: I) Liberated pyrite grains (centre and left) and chalcopyrite grain within rock fragment. RL. FOV =~ 0.7 mm. J) Liberated pyrite grains (right) and scattered pyrite and chalcopyrite particles within powder clump (left). RL. FOV = ~1.3 mm.

Maxxam Project #: 2-21-900
Sample ID: KM 2916 -14 Cu 1CT

Description of powder:

Medium-grey powder and clumps of powder (up to 4 mm size). No reaction to magnet. Moderate reaction to cold, dilute HCl. No reaction of offcut mount to etching with HF and staining with sodium cobaltinitrite solution (no yellow stain).

Polished Thin Section Description:

Mineral	%	Grain Size (min) mm	Grain Size (max) mm	Description
Quartz & plagioclase (undifferentiated)	20	< 0.01	0.05	liberated grains, within rock fragments, in powder clumps
Sericite	20	< 0.01	0.07	liberated flakes, within rock fragments and in powder clumps
Carbonate	15	< 0.01	0.3	liberated grains, within rock fragments, in powder clumps
Quartz	2	< 0.05	0.3	liberated grains, within rock fragments and in powder clumps
Unknown	2	<0.001	< 0.001	cryptocrystalline material forms opaque powder clumps
Rutile	trace	<0.005	0.010	aggregate within rock fragments and in powder clumps
Biotite	rare	0.02	0.035	within rock fragments
Fe-ox	rare	<0.005	0.1	liberated red-brown grains, within powder clumps
Tramp material	rare	0.01	0.035	liberated particles
Unidentified	~25	<0.01	<0.01	

Sulphide Mineral	%	Grain Size (min) mm	Grain Size (max) mm	Description
Pyrite	15	0.007	0.35	liberated grains, within rock fragments, in powder clumps
Chalcopyrite	trace	0.01	0.075	grains in rock fragments, inclusions in pyrite
Sphalerite	rare		0.1	inclusions in pyrite
Galena	rare		0.075	liberated grain, within powder clump
Molybdenite	rare		0.05	liberated grain

Mixed fine to very fine liberated grains (typically < 0.05 mm, maximum 0.35 mm), rock fragments (up to 0.35 mm) and minor clumps of grains, rock fragments and powder (up to 3 mm clumps. The very fine-grain size of most constituents in the section precludes accurate visual identification and percentage estimates of anisotropic minerals with low birefringence. Some grains that can be distinguished optically include quartz, sericite, carbonate, pyrite and traces of rutile and chalcopyrite. Biotite, Fe-ox, sphalerite, galena, molybdenite and tramp particles occur rarely. Unknown cryptocrystalline material occurs as fragments of powder clumps (up to 0.35 mm diameter). Approximately 25% of the section is unidentified due to very fine-grain size of particles. Rock fragments include variably pyrite or chalcopyrite bearing, fine-grained quartz±carbonate aggregate, quartz-sericite ±plagioclase ±carbonate aggregate and quartz-biotite aggregate.

Carbonate occurs in major amounts ~ 15% as fine to very fine (< 0.3 mm) patchy aggregates in rock fragments, as liberated grains and within clumps of powder.

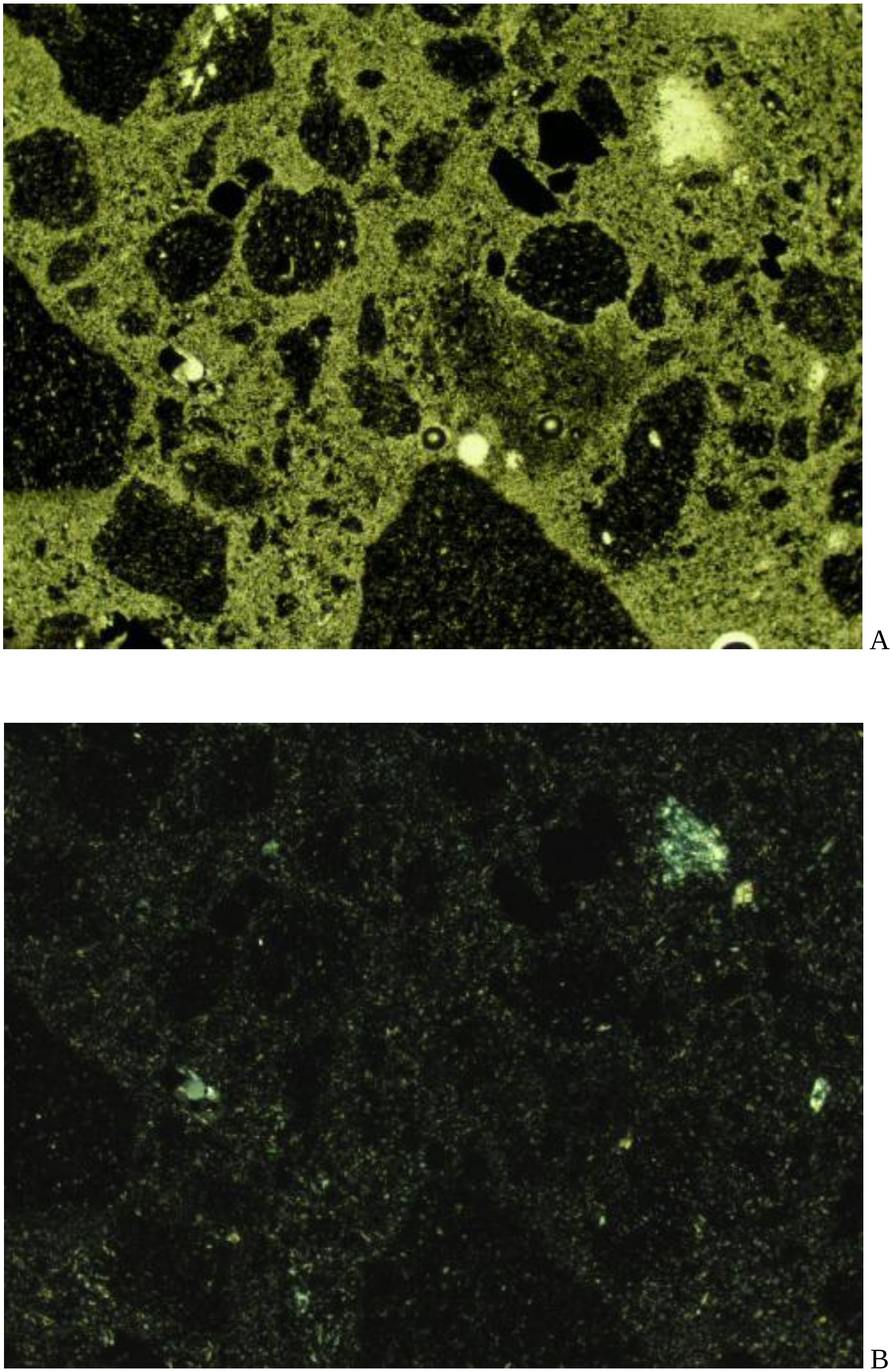
Sulphide occurs in major amounts, ~15%, as pyrite, traces of chalcopyrite and rarely as sphalerite, galena and molybdenite. Pyrite occurs as anhedral liberated grains, as sub-anhedral grains in rock fragments and in powder

Maxxam Project #: 2-21-900
Sample ID: KM 2916 -14 Cu 1CT

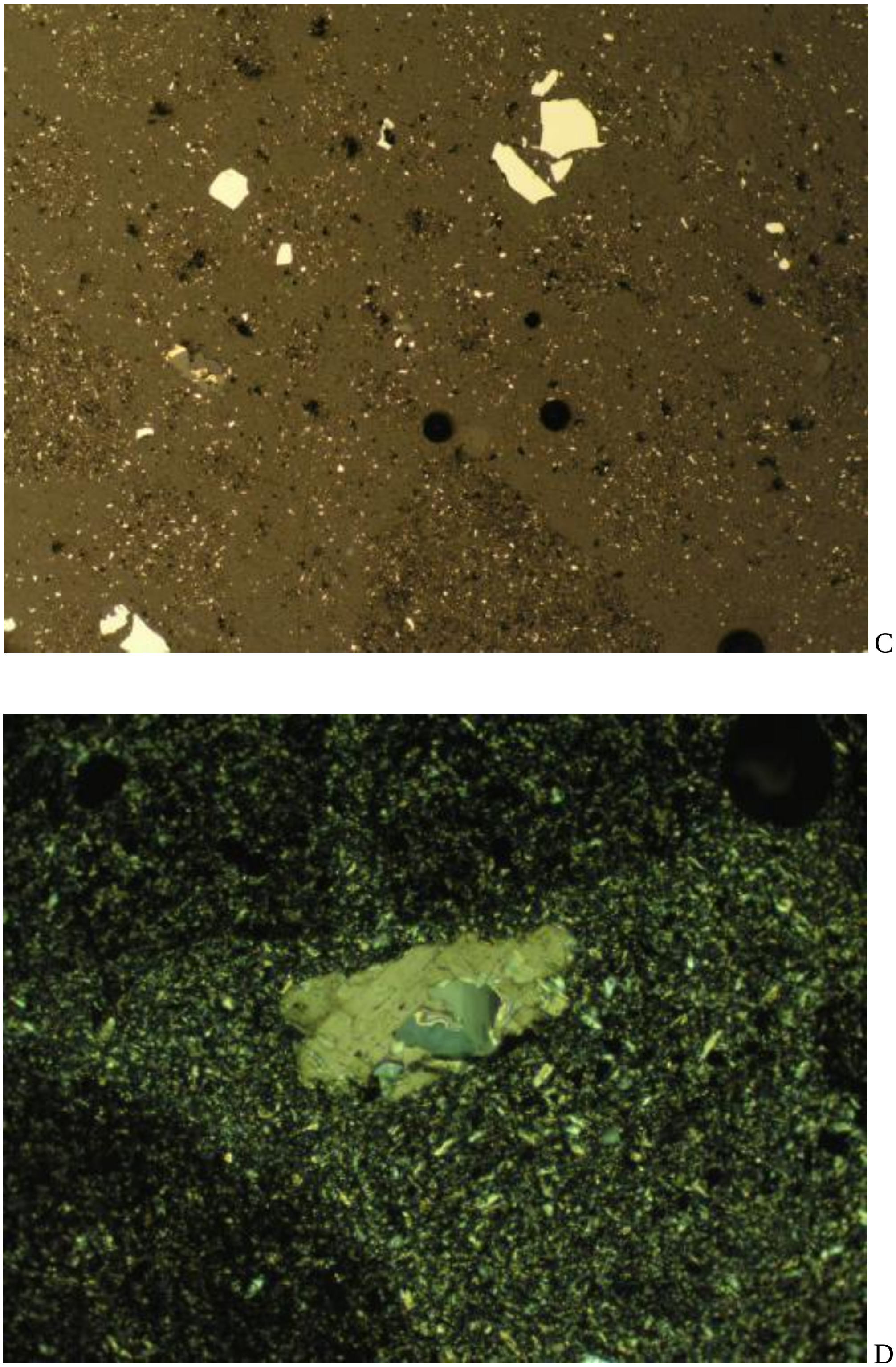
Polished Thin Section Description: (cont.)
clumps. Pyrite grain boundaries are typically irregular but unaltered. Chalcopyrite occurs in trace amounts as anhedral grains disseminated in rock fragments and as inclusions in pyrite. Chalcopyrite grains are unaltered. Sphalerite occurs rarely with chalcopyrite as inclusions in pyrite. Galena occurs rarely as a liberated very fine grain. Molybdenite is observed as a liberated very fine grains and particles within powder clump.

Particles of tramp Fe ±Cr-Ni (±Si) occur scattered as liberated particles in the sample. The tramp particles were described as unknown particles in the petrographic report of June 29, 2012 and tested by EPMA (Appendix 2B).

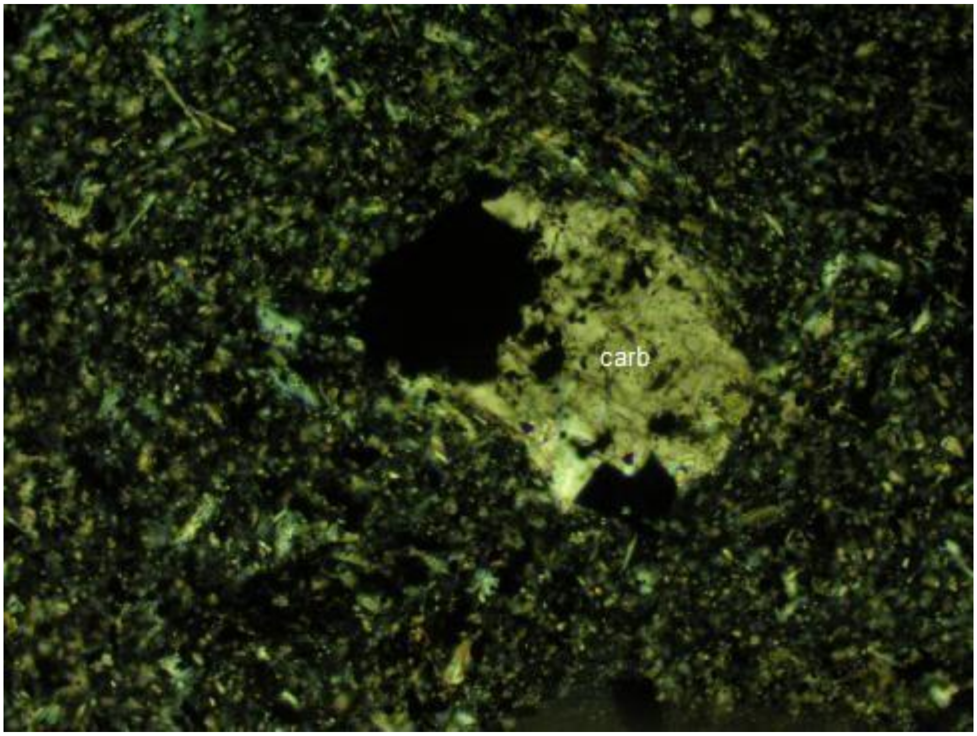
Traces of red-brown Fe-ox occurs as liberated grains and aggregates.



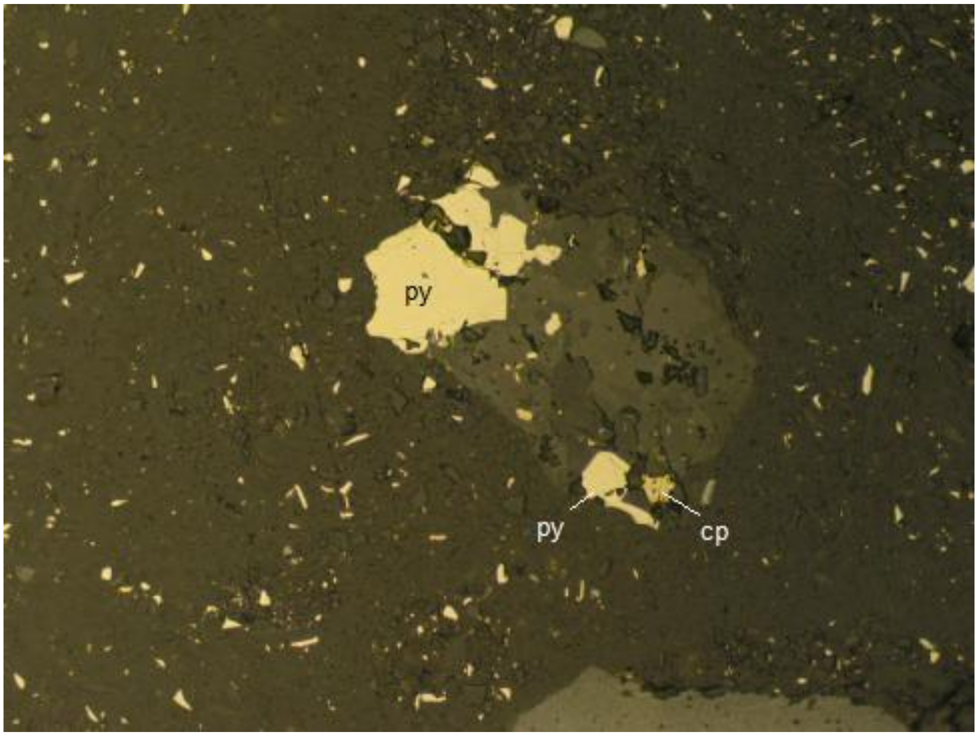
KM 2916 -14 Cu 1CT: A&B) Representative fine to very fine grains, rock fragments and clumps of powder and grains. A) PPL, B) XPL, FOV = ~ 2.6 mm.



KM 2916 -14 Cu 1CT: C) Representative fine to very fine grains and clumps of powder and grains. (same view as photos A & B). RL, FOV =~ 2.6 mm, D) Carbonate-quartz rock fragment (centre). XPL, FOV =~ 1.3 mm.

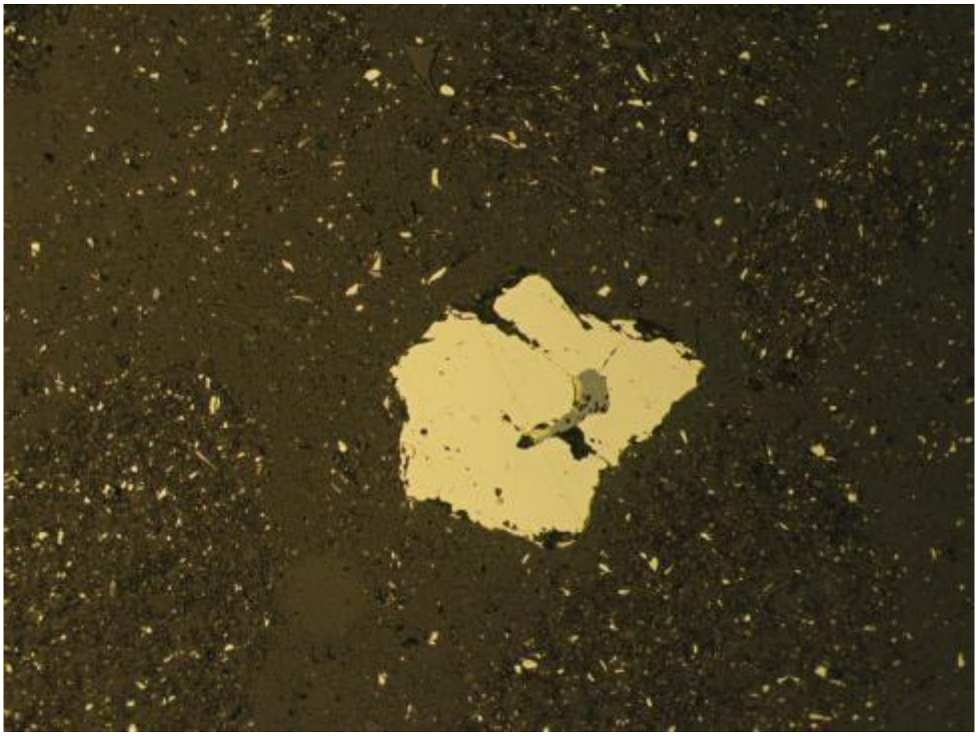


E



F

KM 2916 -14 Cu 1CT: E&F) Carbonate-pyrite-chalcopyrite aggregate (centre). E) XPL, F) RL. FOV =~ 0.7 mm.



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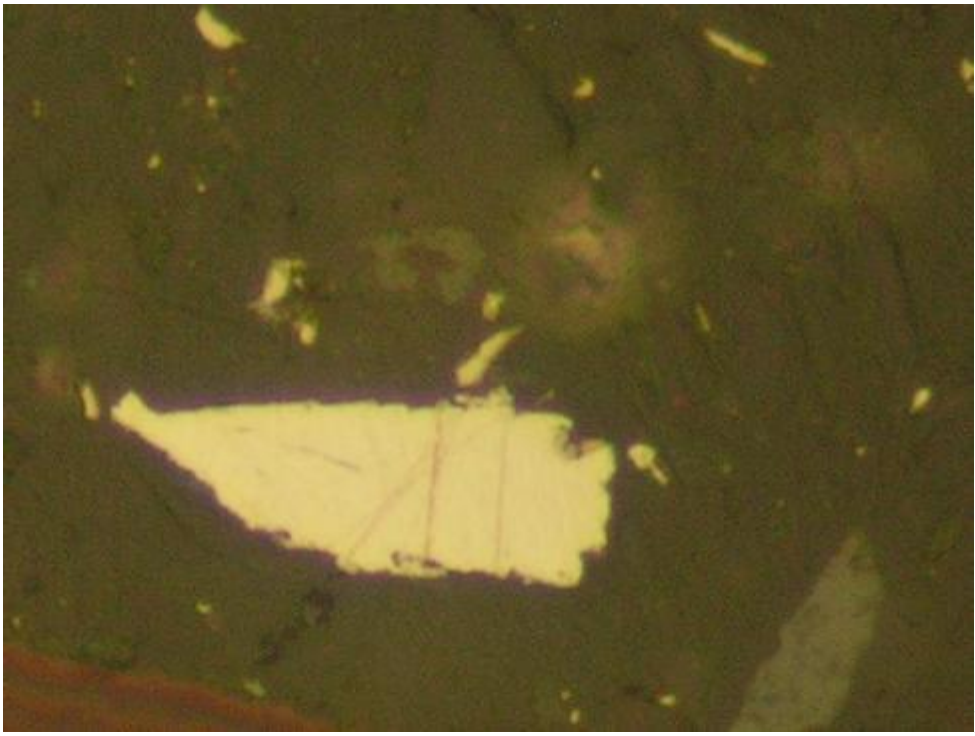
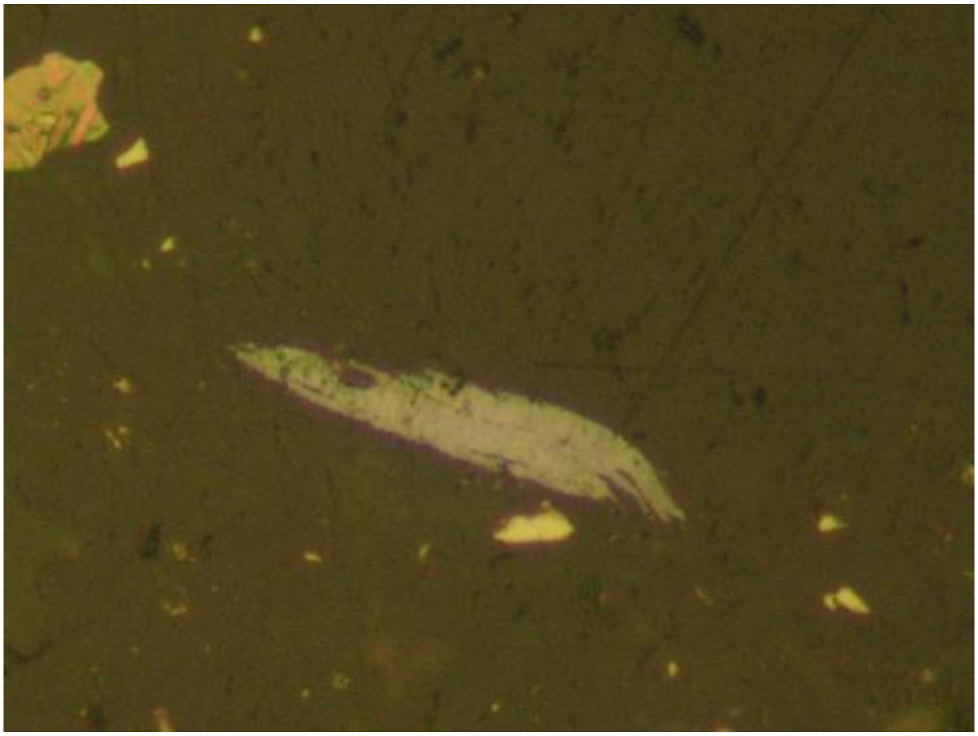
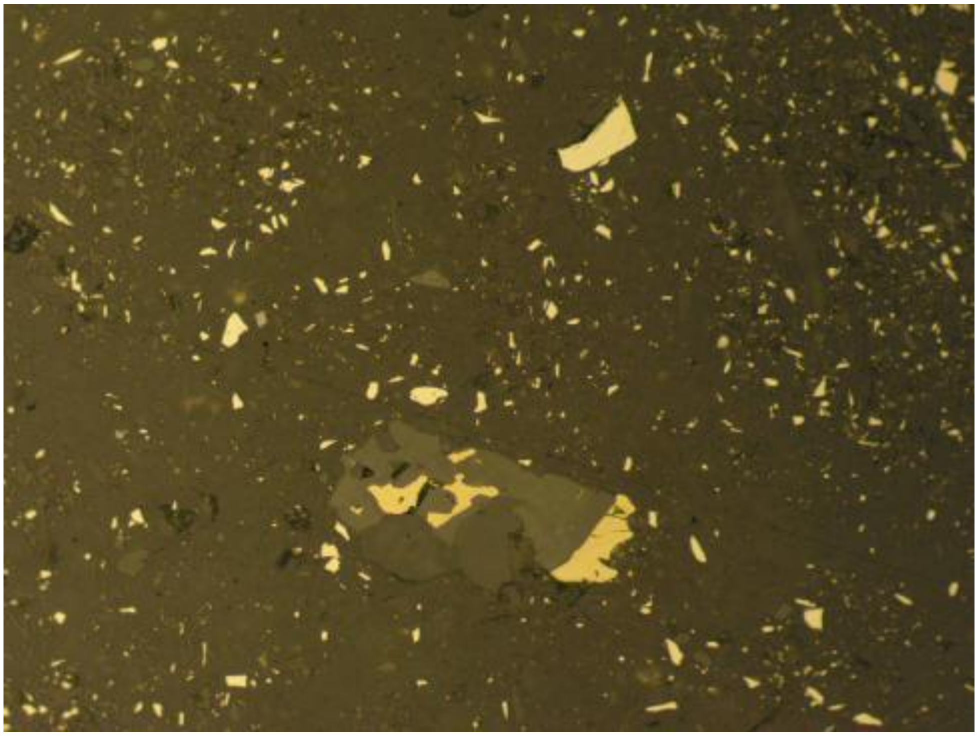


H

KM 2916 -14 Cu 1CT: G) Very fine pyrite fragments scattered as liberated particles and within powder clumps. Fine-grained, liberated anhedral pyrite grain (centre) with sphalerite and chalcopyrite inclusions. RL. FOV =~ 1.3 mm. H) Liberated tramp particle. RL. FOV = ~0.07 mm.





KM 2916 -14 Cu 1CT: I) Liberated molybdenite particle. RL. FOV =~ 0.1 mm. J) Liberated pyrite grains and disseminated chalcopyrite within rock fragment (below centre). RL. FOV = ~0.7 mm.

Statement of qualifications: Kathryn P.E. Dunne

I, Kathryn P.E. Dunne, of the City of Salmon Arm, province of British Columbia, do hereby certify that:

1. I am an independent consulting geologist, with a business office at 4610 Lakeshore Road NE, Salmon Arm, B.C., Canada. My business mailing address is: Bag 9000, # 207, 190B Trans Can Hwy NE, Salmon Arm, BC, V1E 1S3.

2. I am a graduate in geology, with a BSc in geology from The University of British Columbia (1985).

3. I received my Masters degree in geology from The University of British Columbia, Vancouver, B.C. in 1988.

4. I am a registered member of the Association of Professional Engineers and Geoscientists of the Province of British Columbia (No. 18674).

5. I am a fellow of the Geological Association of Canada and a member of the Society of Economic Geologists and Mineralogical Association of Canada.

6. I have practiced my profession as a geologist for approximately 23 years: 4 years as geologist with the British Columbia Geological Survey Branch, 3 years as research coordinator at the Mineral Deposit Research Unit housed within the Department of Earth and Ocean Sciences at the University of British Columbia, and 16 years as an independent consultant.

7. The petrographic data of this report was collected by me in June 2012 and revised based on EPMA results in July 2012.

……………………..…………………
Kathryn P.E. Dunne, M.Sc., P.Geo.
Consulting Geologist
July 13, 2012

APPENDIX 1

Summary of Carbonate & Sulphide Grain Selection for Electron Probe Microanalysis

2 TAILINGS SAMPLES,

HARPER CREEK PROJECT, BC

July 2, 2012

Prepared for:
Mati Raudsepp, Professor (Hon.)
Director: Electron-microbeam/X-ray Diffraction Facility
Department of Earth & Ocean Sciences
6339 Stores Road, The University of British Columbia
Vancouver, BC V6T 1Z4,

AND
Tim O'Hearn, M.Eng., P.Eng.
Director, ARD
Maxxam Analytics
4606 Canada Way
Burnaby, BC V5G 1K5

Prepared by:
Kathryn Dunne, M.Sc. P.Geo.
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Salmon Arm, BC
Canada V1E 1S3

phone: 250-804-0729
kgeo@telus.net

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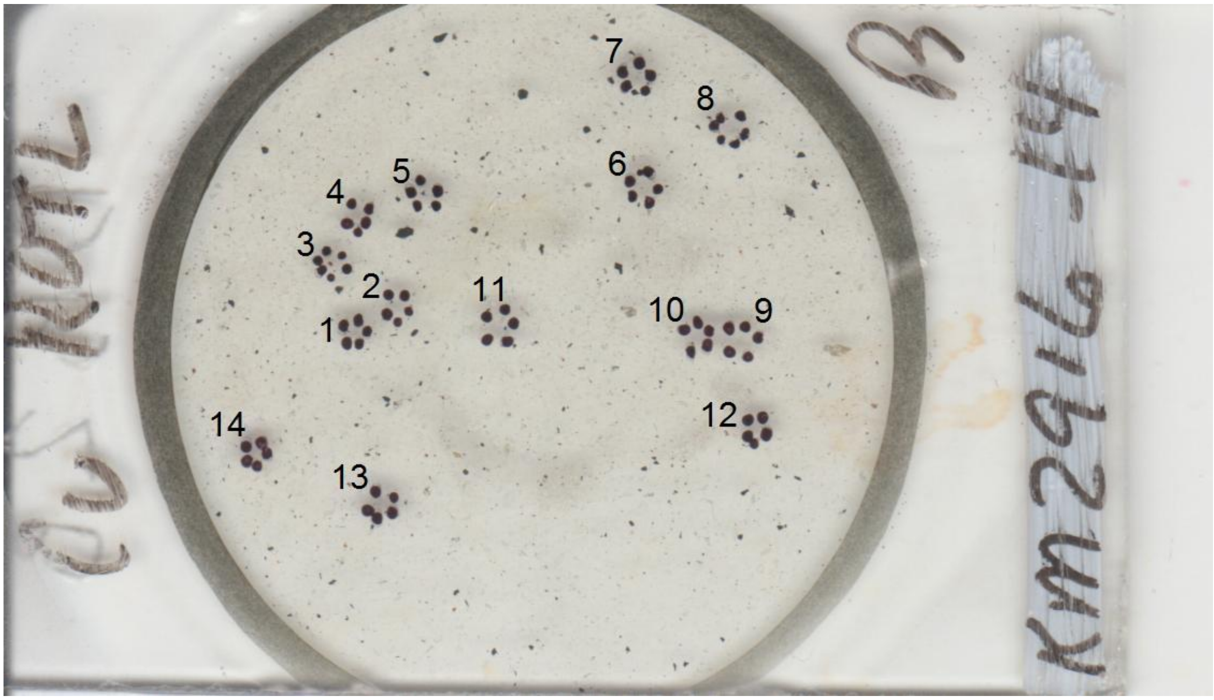
SAMPLE #: KM 2916-14 Cu ROTL #1

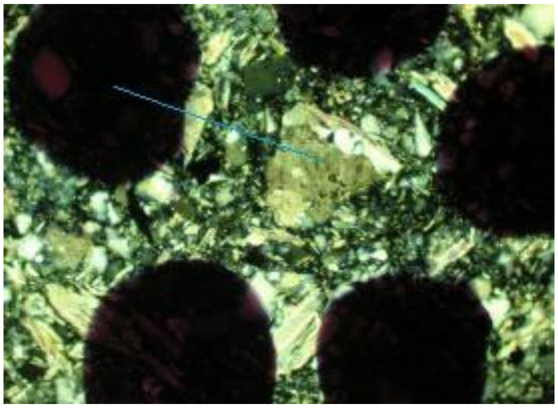


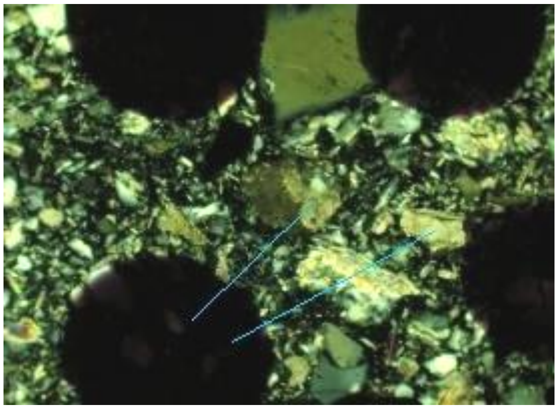
→ (right side of section)

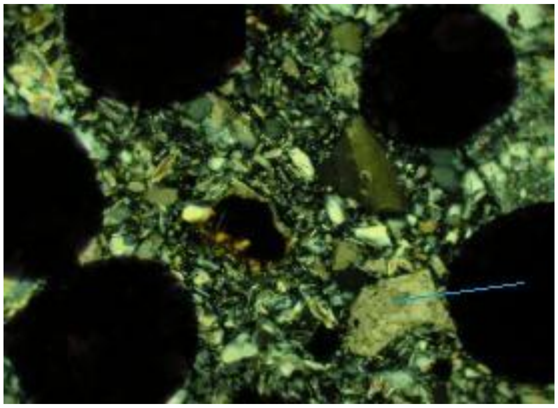
Note: The orientation of detailed circle photos in the following pages is same as image above. The polished thin section identifying number and right edge of above slide is always parallel to the right side of the circle images.

SAMPLE #: KM 2916-14 Cu ROTL #1

15 Grains total for Carbonate Identification: → (right side of section)


Circle #1 (XPL) FOV = 1.3 mm
Blue line from an ink dot to carbonate grain


Circle #2-Detailed view (XPL) FOV = 1.3 mm
Blue line from an ink dot to carbonate grains (2)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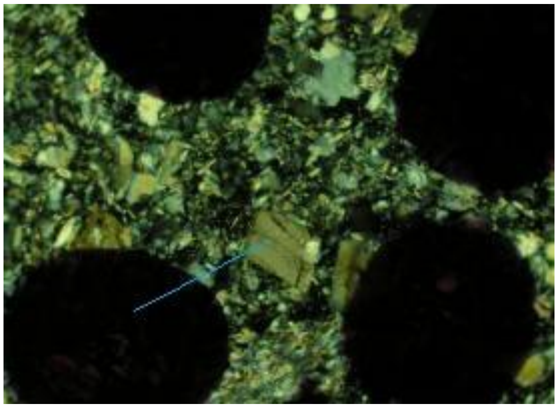
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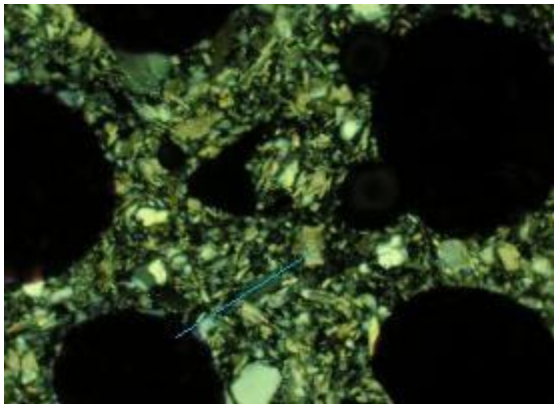
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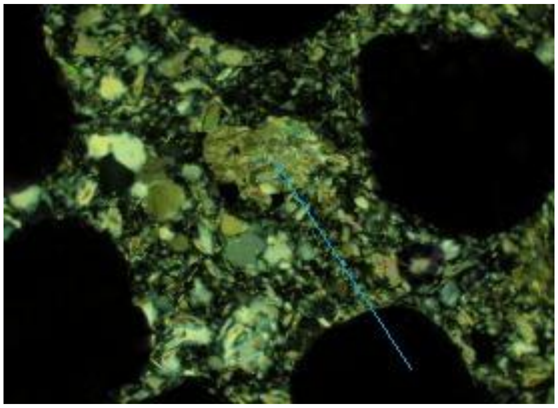
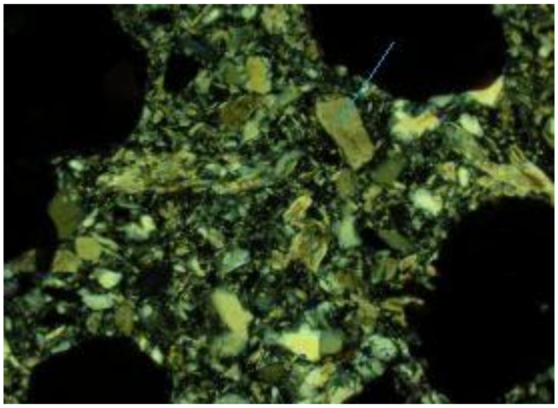
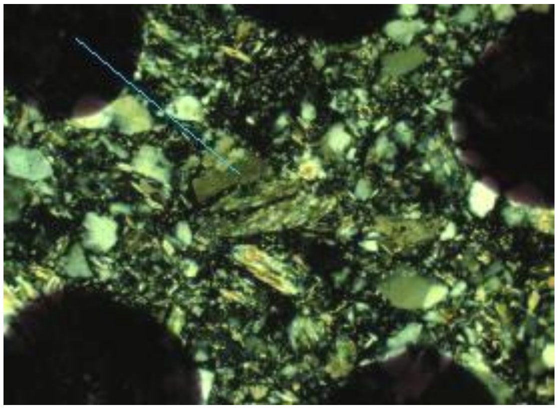

Circle #7-Detailed view (XPL) FOV = 1.3 mm
Blue line from an ink dot to carbonate grain

SAMPLE #: KM 2916-14 Cu ROTL #1

15 Grains total for Carbonate Identification: → (right side of section)



Circle #8 (XPL) FOV = 1.3 mm
Blue line from an ink dot to carbonate grain



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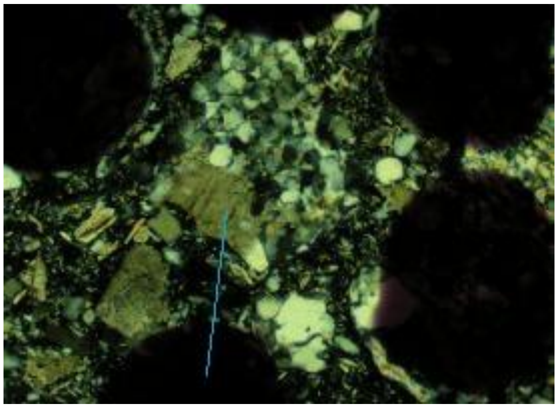

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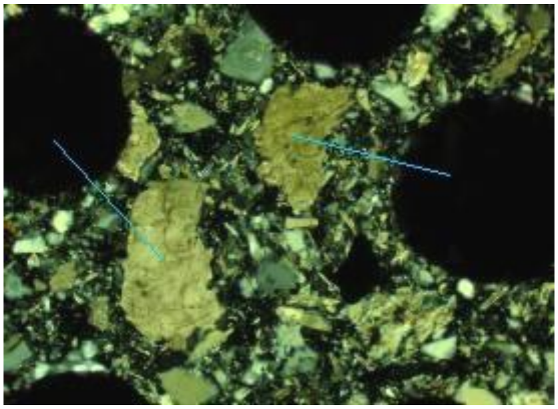

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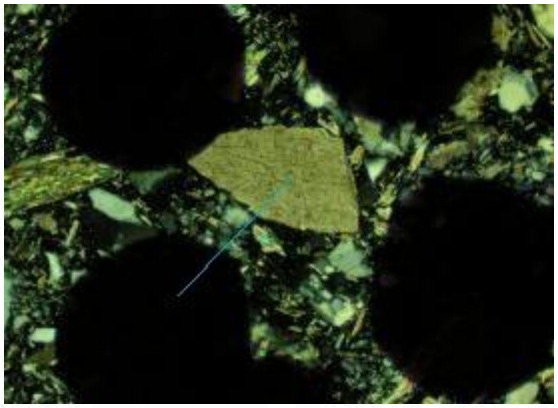

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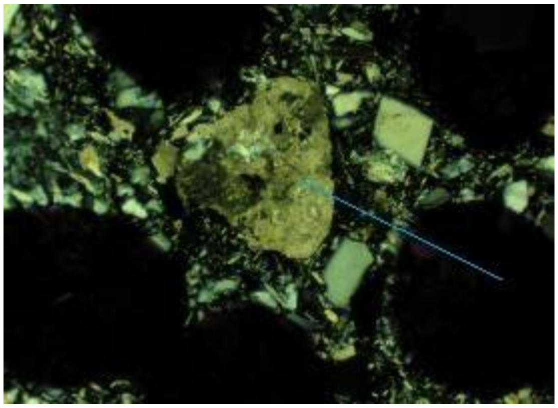
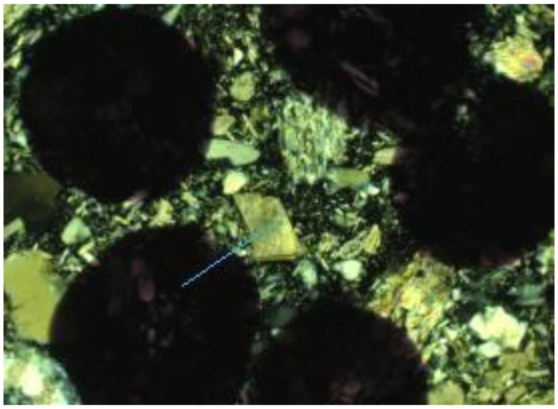


Circle #13-Detailed view (XPL) FOV = 1.3 mm
Blue line from an ink dot to carbonate grain

SAMPLE #: KM 2916-14 Cu ROTL #1

<u>15 Grains total for Carbonate Identification:</u> → (right side of section)

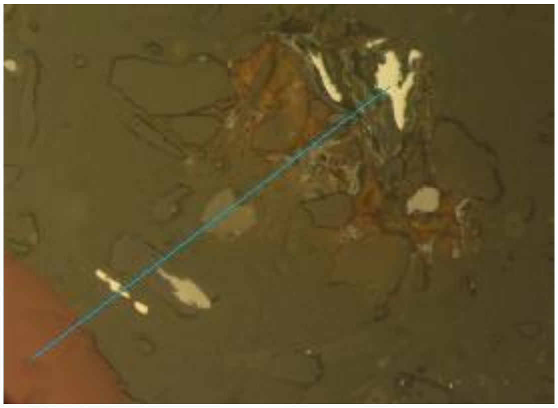


<u>Circle #14</u> (XPL) FOV = 1.3 mm
Blue line from an ink dot to carbonate grain

SAMPLE #: KM 2916-14 Cu ROTL #1

<u>5 Grains total for Sulphide Identification:</u> → (right side of section)

<u>Circle #3</u> (RL) FOV = 1.3 mm
Blue line from an ink dot to unknown grain



<u>Circle #3-Detailed view</u> (RL) FOV = 0.3 mm
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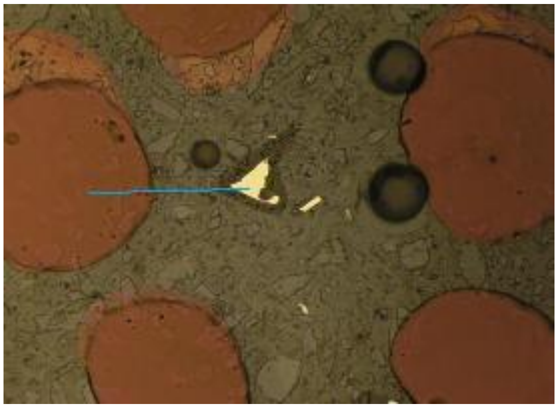

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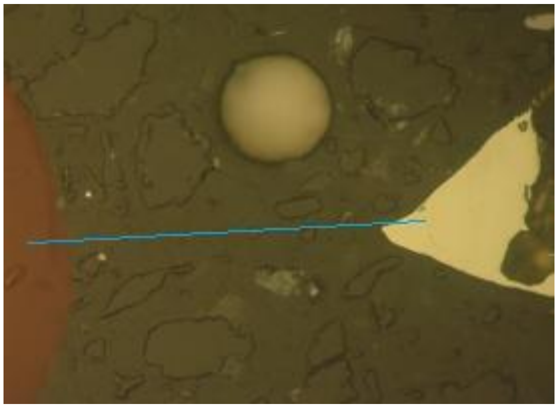

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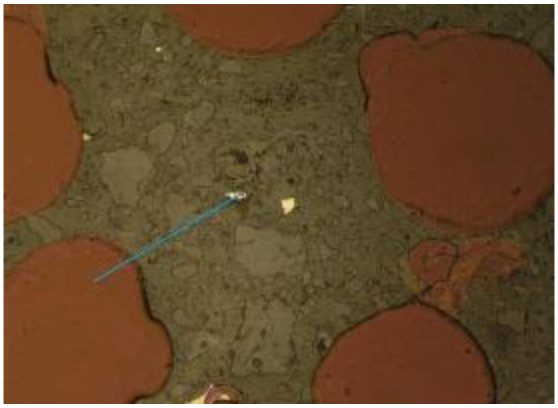

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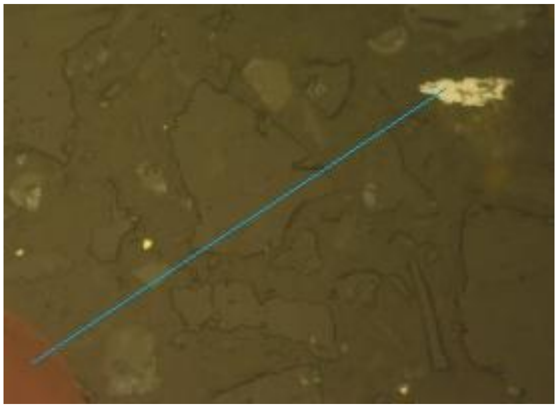
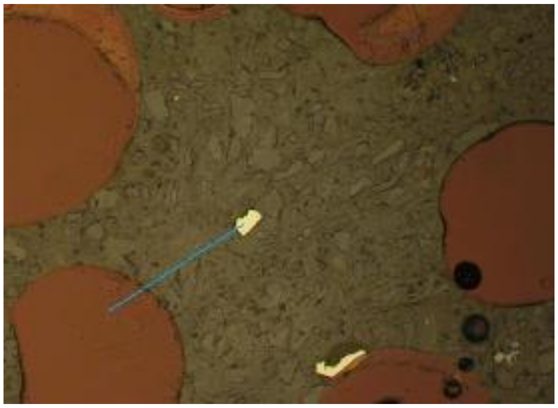


<u>Circle #6-Detailed view</u> (RL) FOV = 0.3 mm
Blue line from an ink dot to unknown grain

SAMPLE #: KM 2916-14 Cu ROTL #1

<u>5 Grains total for Sulphide Identification:</u> → (right side of section)



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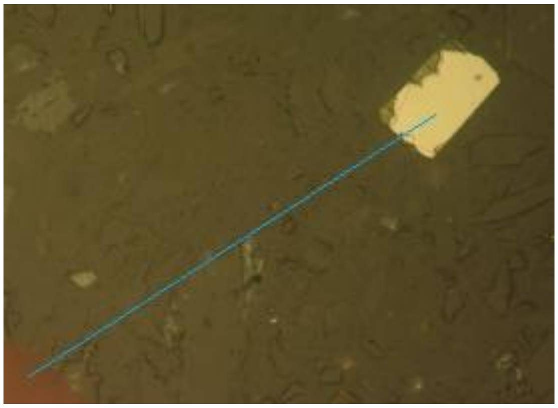
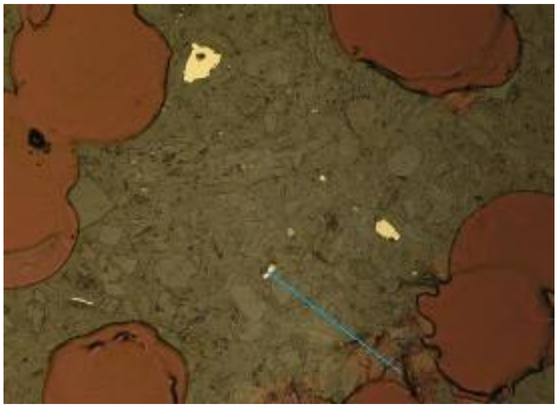


<u>Circle #7-Detailed view</u> (RL) FOV = 0.3 mm
Blue line from an ink dot to pyrite grain



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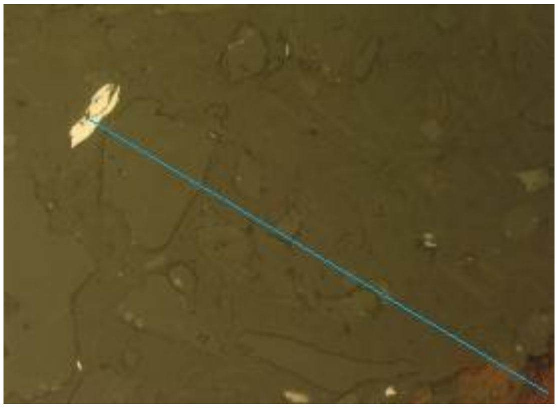

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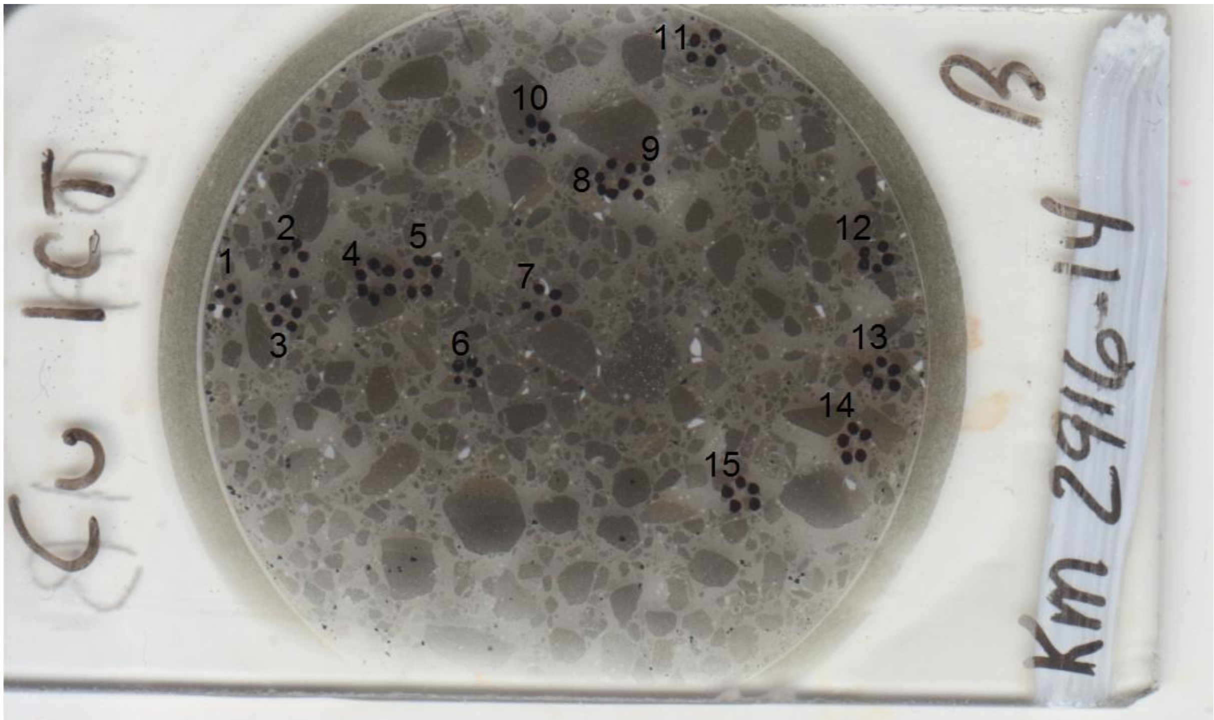
Blue line from an ink dot to unknown grain

SAMPLE #: KM 2916-14 Cu 1CT #2



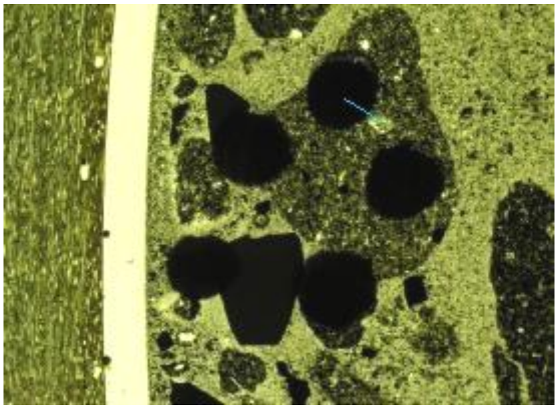
→ (right side of section)

Note: The orientation of detailed circle photos in the following pages is same as image above. The polished thin section identifying number and right edge of above slide is always parallel to the right side of the circle images.

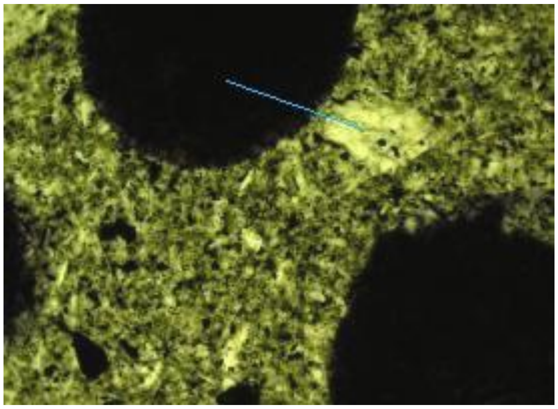
*Note: The sulphide grains selected in this sample are very tiny and may be difficult to locate due to the particle size of sample.

SAMPLE #: KM 2916-14 Cu 1CT #2

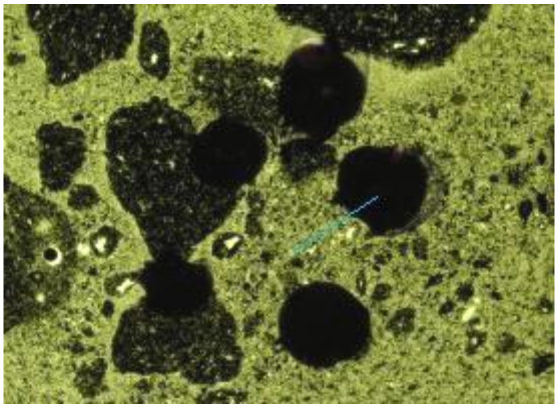
15 Grains total for Carbonate Identification: → (right side of section)



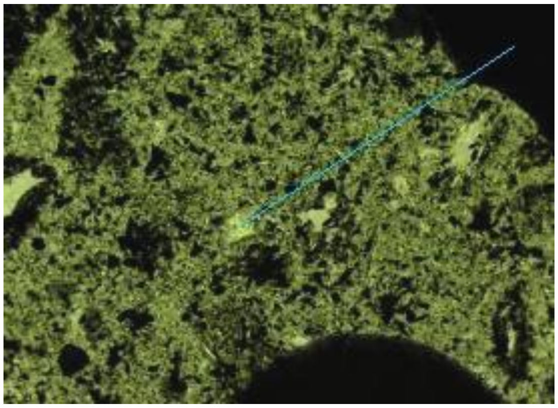
Circle #1 (PPL) FOV = 2.6 mm
Blue line from an ink dot to carbonate grain



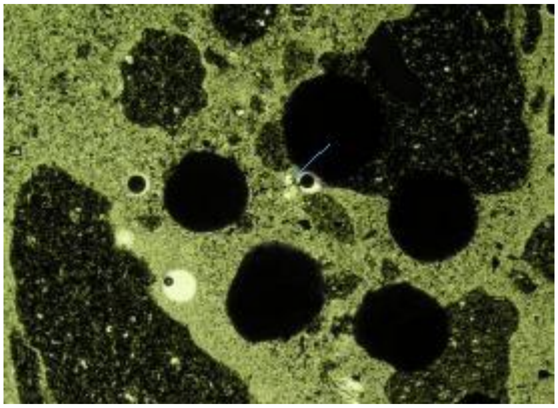
Circle #1-Detailed view (PPL) FOV = 0.7 mm
Blue line from an ink dot to carbonate



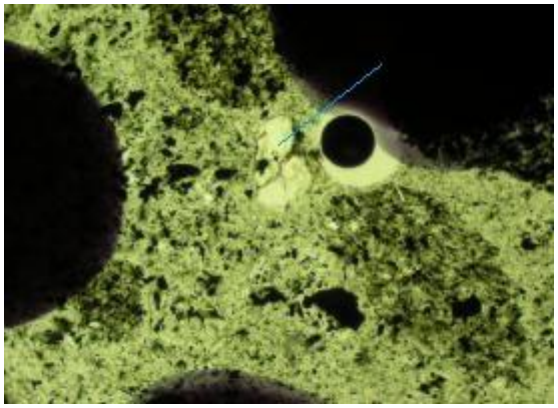
Circle #2 (PPL) FOV = 2.6 mm
Blue line from an ink dot to carbonate grain



Circle #2-Detailed view (PPL) FOV = 0.7 mm
Blue line from an ink dot to carbonate grain



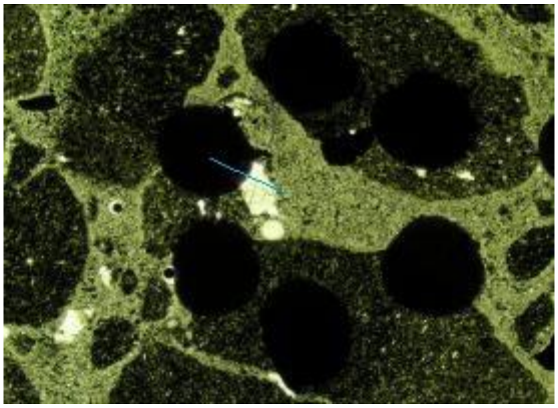
Circle #3 (PPL) FOV = 2.6 mm
Blue line from an ink dot to carbonate grain



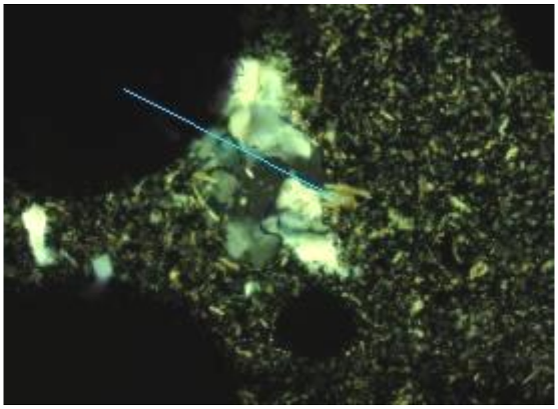
Circle #3-Detailed view (PPL) FOV = 0.7 mm
Blue line from an ink dot to carbonate grain

SAMPLE #: KM 2916-14 Cu 1CT #2

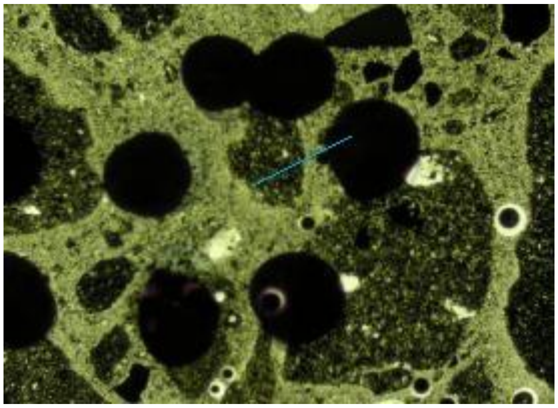
15 Grains total for Carbonate Identification: → (right side of section)



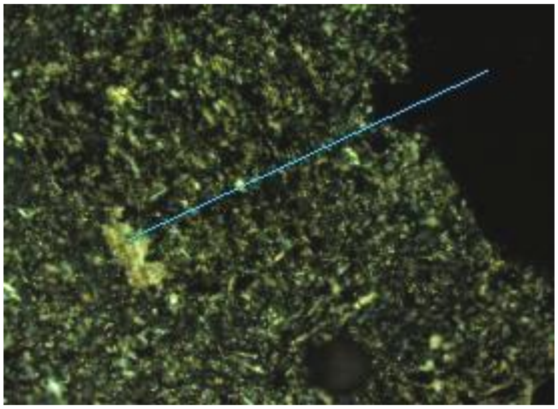
Circle #4 (PPL) FOV = 2.6 mm
Blue line from an ink dot to carbonate grain



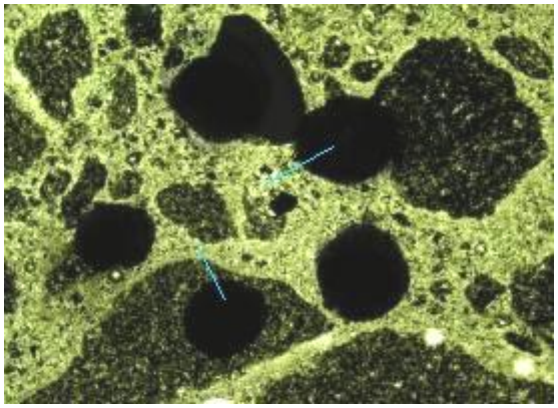
Circle #4-Detailed view (XPL) FOV = 0.7 mm
Blue line from an ink dot to carbonate grain



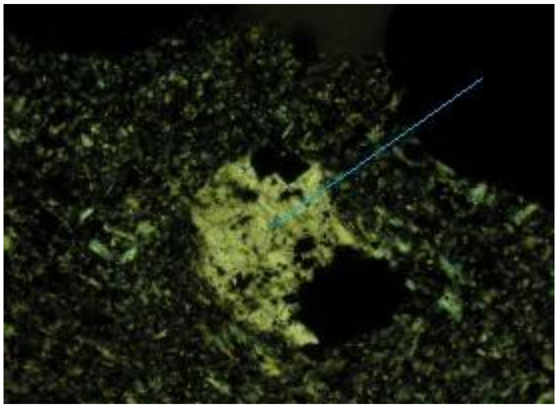
Circle #5 (PPL) FOV = 2.6 mm
Blue line from an ink dot to carbonate grain



Circle #5-Detailed view (XPL) FOV = 0.7 mm
Blue line from an ink dot to carbonate



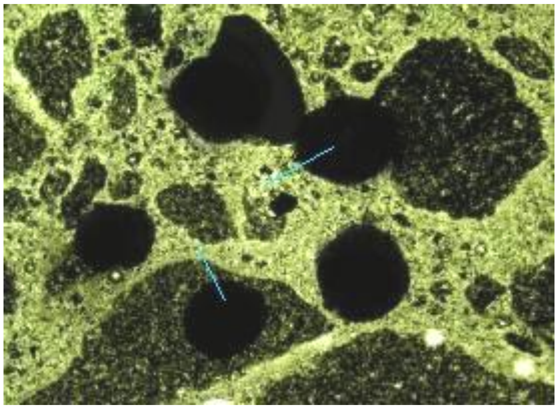
Circle #7 (PPL) FOV = 2.6 mm
Blue line from an ink dot to carbonate grains (2)



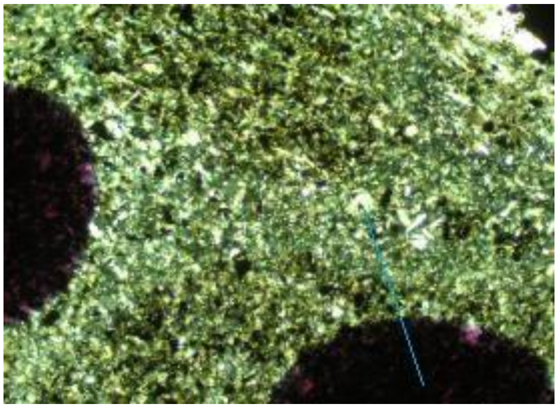
Circle #7-Detailed view (XPL) FOV = 0.7 mm
Blue line from an ink dot to carbonate

SAMPLE #: KM 2916-14 Cu 1CT #2

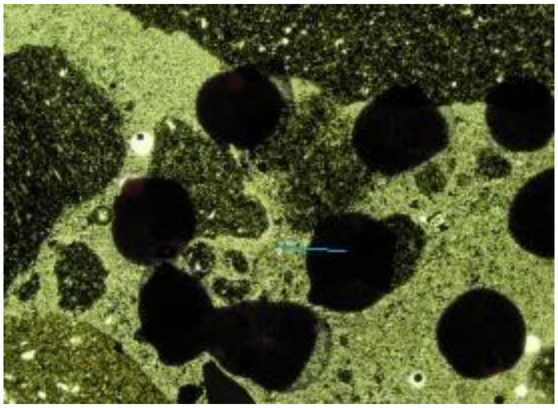
15 Grains total for Carbonate Identification: → (right side of section)



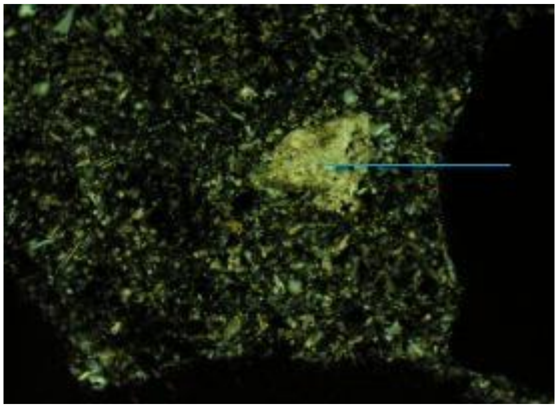
Circle #7 (PPL) FOV = 2.6 mm
Blue line from an ink dot to carbonate grains (2)



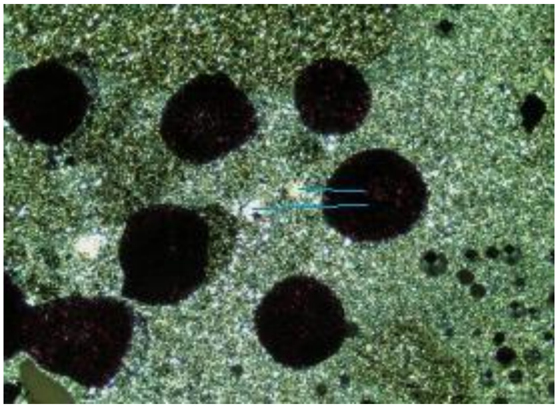
Circle #7-Detailed view (XPL) FOV = 0.7 mm
Blue line from an ink dot to carbonate grain



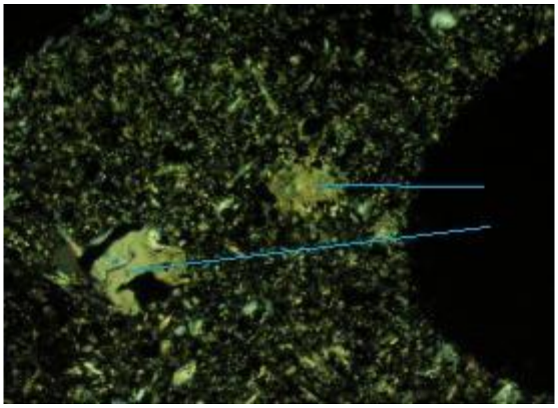
Circle #8 (PPL) FOV = 2.6 mm
Blue line from an ink dot to carbonate grain



Circle #8-Detailed view (XPL) FOV = 0.7 mm
Blue line from an ink dot to carbonate



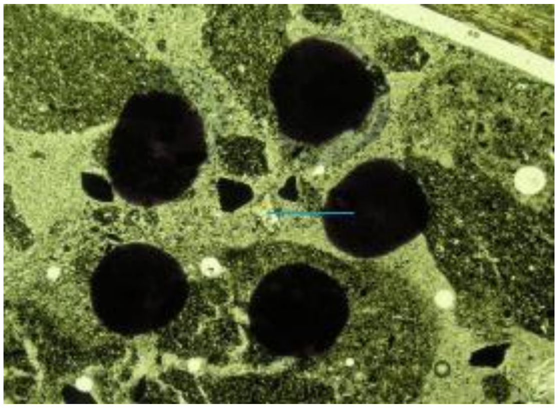
Circle #9 (PPL) FOV = 2.6 mm
Blue line from an ink dot to carbonate grains (2)



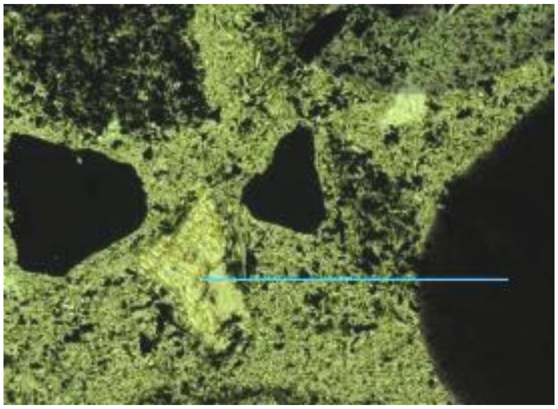
Circle #9-Detailed view (XPL) FOV = 0.7 mm
Blue line from an ink dot to carbonate grains (2)

SAMPLE #: KM 2916-14 Cu 1CT #2

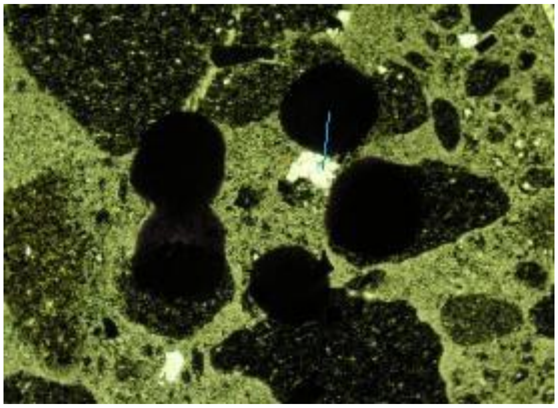
15 Grains total for Carbonate Identification: → (right side of section)



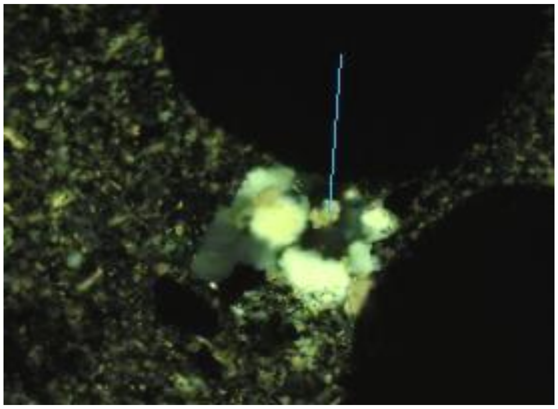
Circle #11 (PPL) FOV = 2.6 mm
Blue line from an ink dot to carbonate grain



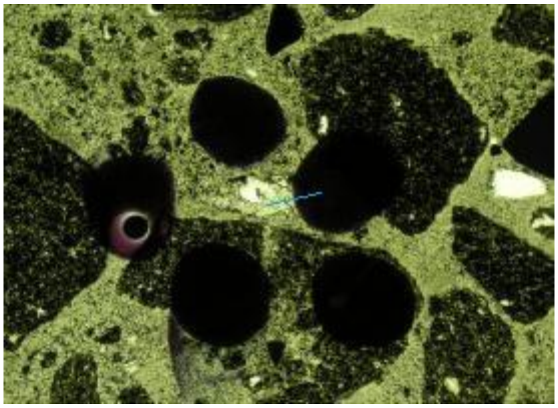
Circle #11-Detailed view (PPL) FOV = 0.7 mm
Blue line from an ink dot to carbonate grain



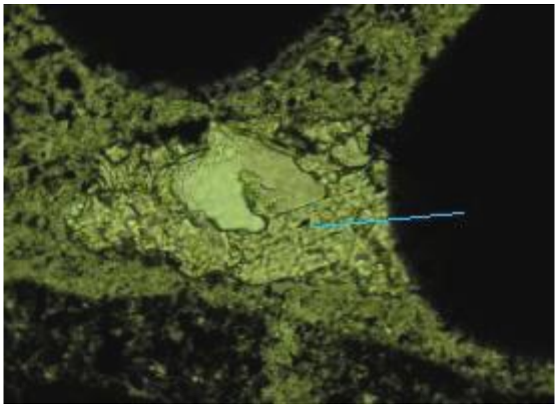
Circle #12 (PPL) FOV = 2.6 mm
Blue line from an ink dot to carbonate grain



Circle #12-Detailed view (XPL) FOV = 0.7 mm
Blue line from an ink dot to carbonate



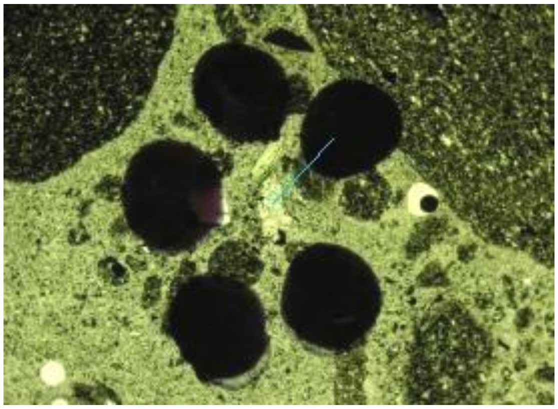
Circle #13 (PPL) FOV = 2.6 mm
Blue line from an ink dot to carbonate grain



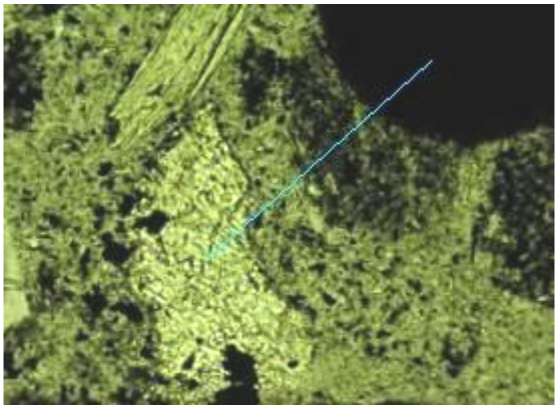
Circle #13-Detailed view (PPL) FOV = 0.7 mm
Blue line from an ink dot to carbonate

SAMPLE #: KM 2916-14 Cu 1CT #2

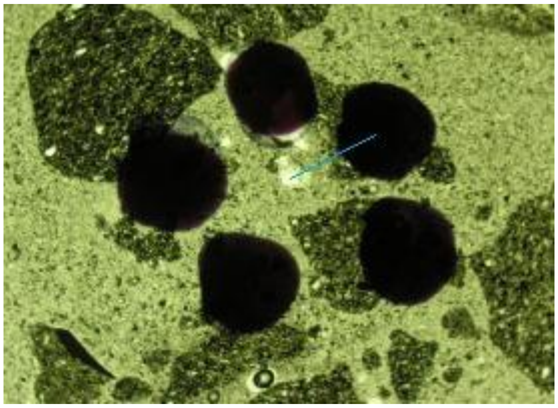
15 Grains total for Carbonate Identification: → (right side of section)



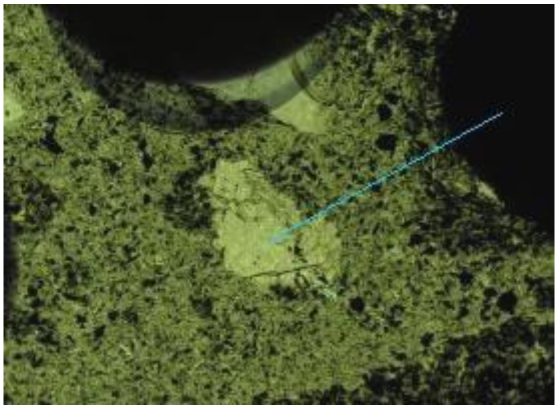
Circle #14 (PPL) FOV = 2.6 mm
Blue line from an ink dot to carbonate grain



Circle #14-Detailed view (PPL) FOV = 0.7 mm
Blue line from an ink dot to carbonate grain



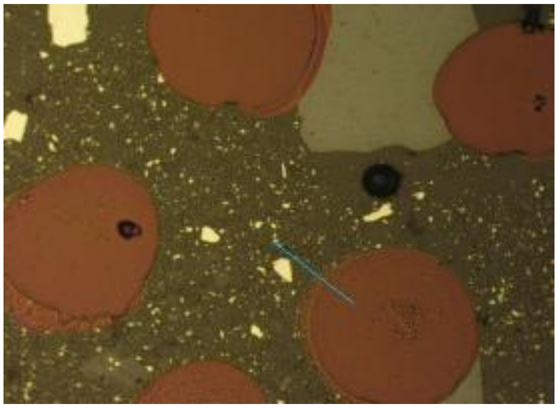
Circle #15 (PPL) FOV = 2.6 mm
Blue line from an ink dot to carbonate grain



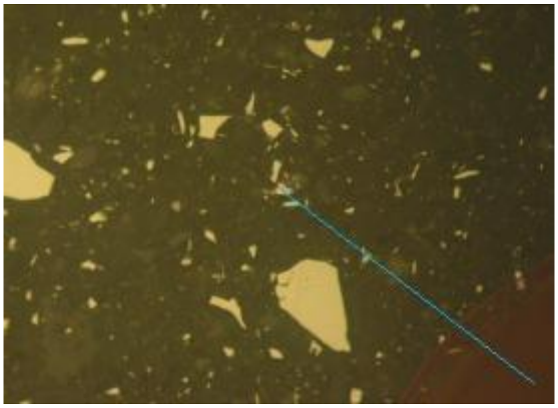
Circle #15-Detailed view (PPL) FOV = 0.7 mm
Blue line from an ink dot to carbonate

SAMPLE #: KM 2916-14 Cu 1CT #2

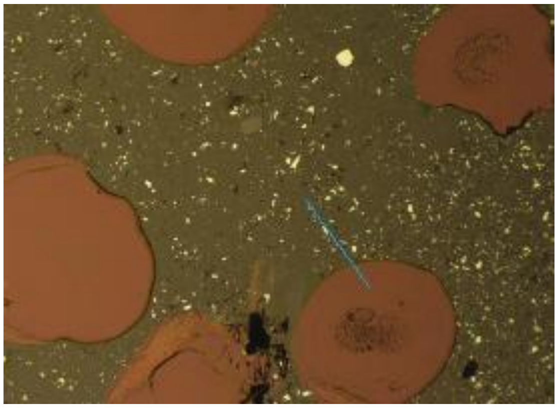
5 Grains total for Sulphide Identification: → (right side of section)



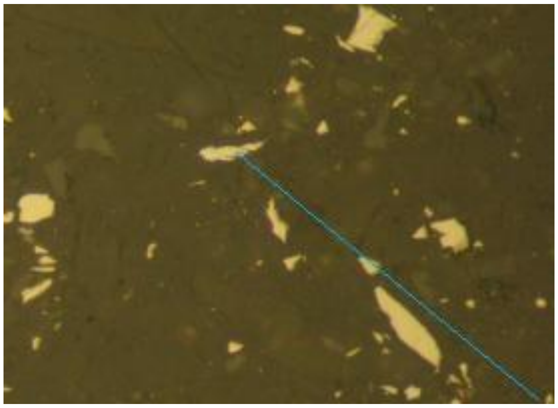
Circle #1 (RL) FOV = 1.3 mm
Blue line from an ink dot to unknown grain



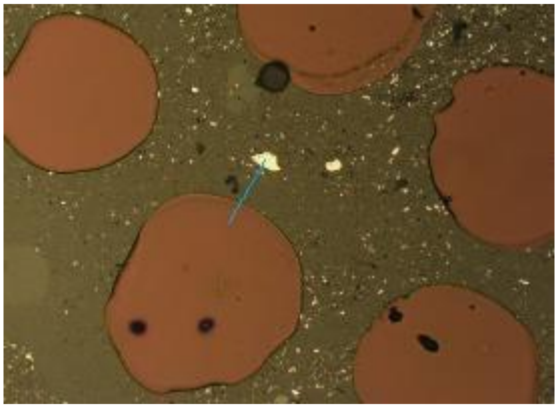
Circle #1-Detailed view (RL) FOV = 0.3 mm
Blue line from an ink dot to unknown grain



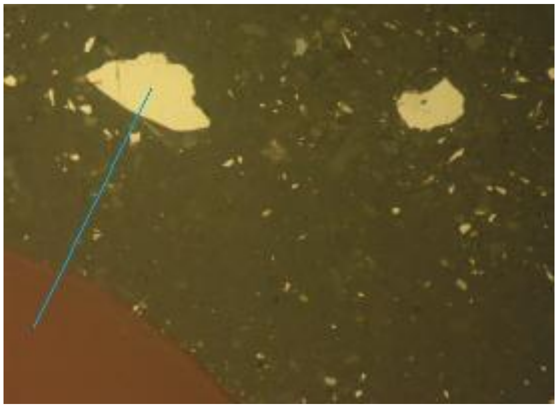
Circle #2 (RL) FOV = 1.3 mm
Blue line from an ink dot to unknown grain



Circle #2-Detailed view (RL) FOV = 0.3 mm
Blue line from an ink dot to unknown grain



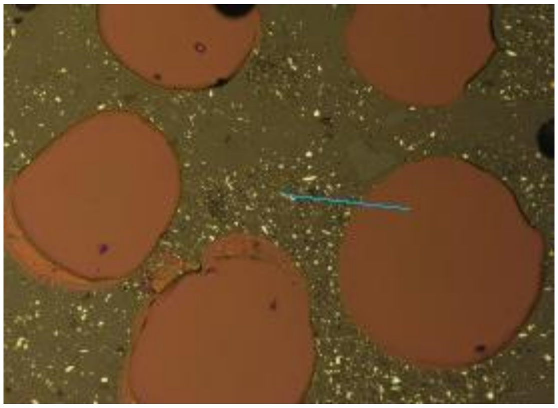
Circle #3 (RL) FOV = 1.3 mm
Blue line from an ink dot to pyrite grain



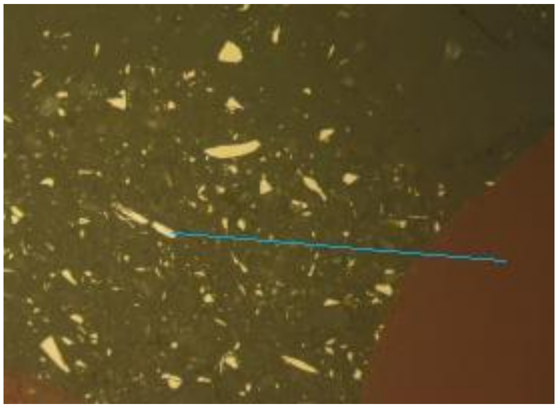
Circle #3-Detailed view (RL) FOV = 0.3 mm
Blue line from an ink dot to pyrite grain

SAMPLE #: KM 2916-14 Cu 1CT #2

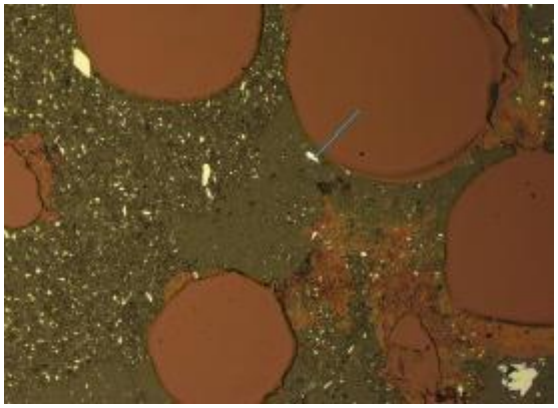
5 Grains total for Sulphide Identification: → (right side of section)



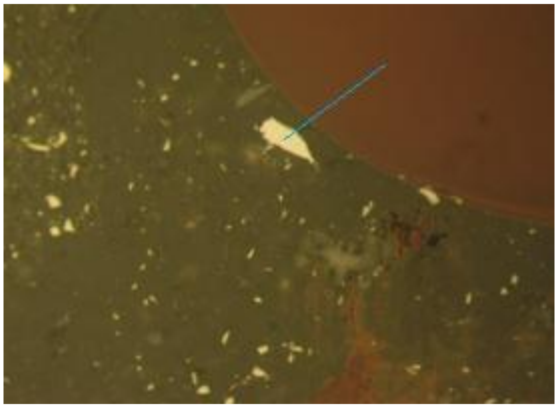
Circle #6 (RL) FOV = 1.3 mm
Blue line from an ink dot to unknown grain



Circle #6-Detailed view (RL) FOV = 0.3 mm
Blue line from an ink dot to unknown grain



Circle #10 (RL) FOV = 1.3 mm
Blue line from an ink dot to unknown grain



Circle #10-Detailed view (RL) FOV = 0.3 mm
Blue line from an ink dot to unknown grain

APPENDIX 2A

Carbonate determinations by Electron Probe Microanalysis (EPMA) undertaken by Edith Czech at UBC, Vancouver

July 6, 2012

Carbonate Grains for Electron Probe Microanalysis

	MGO	CAO	MNO	FEO	CO2 *	TOTAL	MG2+	CA2+	MN2+	FE2+	C4+
KM2916-14CuROTL-C1-1	12.44	28.18	1.67	12.88	44.63	99.80	0.61	0.99	0.05	0.35	2.00
KM2916-14CuROTL-C2-2	17.07	28.75	0.12	7.46	45.85	99.25	0.81	0.98	0.00	0.20	2.00
KM2916-14CuROTL-C3-1	13.18	28.51	0.67	12.56	44.88	99.80	0.64	1.00	0.02	0.34	2.00
KM2916-14CuROTL-C4-2	12.13	28.99	0.77	13.87	44.97	100.73	0.59	1.01	0.02	0.38	2.00
KM2916-14CuROTL-C5-1	11.16	28.65	0.92	15.01	44.44	100.18	0.55	1.01	0.03	0.41	2.00
KM2916-14CuROTL-C6-1	17.92	29.00	0.41	6.13	46.34	99.80	0.85	0.98	0.01	0.16	2.00
KM2916-14CuROTL-C7-1	14.01	28.73	0.88	11.04	45.15	99.81	0.68	1.00	0.02	0.30	2.00
KM2916-14CuROTL-C8-1	14.63	29.83	0.65	9.19	45.42	99.72	0.70	1.03	0.02	0.25	2.00
KM2916-14CuROTL-C9-2	25.43	0.24	0.58	28.26	45.63	100.14	1.22	0.01	0.02	0.76	2.00
KM2916-14CuROTL-C10-2	14.49	28.83	0.61	10.22	45.09	99.24	0.70	1.00	0.02	0.28	2.00
KM2916-14CuROTL-C11-1	26.71	0.39	0.28	26.32	45.77	99.47	1.28	0.01	0.01	0.71	2.00
KM2916-14CuROTL-C12-2	14.75	28.85	0.88	9.50	45.11	99.09	0.71	1.00	0.02	0.26	2.00
KM2916-14CuROTL-C13-1	13.78	29.84	0.77	10.16	45.17	99.72	0.67	1.04	0.02	0.28	2.00
KM2916-14CuROTL-C14-2	17.42	29.05	0.92	5.65	45.85	98.89	0.83	0.99	0.03	0.15	2.00
KM2916-14CuROTL-C15-2	18.18	29.25	0.40	5.54	46.45	99.82	0.86	0.99	0.01	0.15	2.00

Carbonate Grains for Electron Probe Microanalysis

	MGO	CAO	MNO	FEO	CO2 *	TOTAL	MG2+	CA2+	MN2+	FE2+	C4+
KM2916-14Cu1CT-C1-1	30.79	0.13	0.10	21.78	47.13	99.93	1.43	0.00	0.00	0.57	2.00
KM2916-14Cu1CT-C2-1	35.44	0.08	1.04	14.95	48.56	100.07	1.59	0.00	0.03	0.38	2.00
KM2916-14Cu1CT-C3-2	32.35	0.18	0.10	19.73	47.61	99.97	1.48	0.01	0.00	0.51	2.00
KM2916-14Cu1CT-C4-2	12.55	27.98	0.44	13.91	44.46	99.34	0.62	0.99	0.01	0.38	2.00
KM2916-14Cu1CT-C5-2	14.17	28.18	1.04	10.99	44.97	99.35	0.69	0.98	0.03	0.30	2.00
KM2916-14Cu1CT-C6-1	15.94	0.40	1.22	40.60	43.35	101.51	0.80	0.01	0.04	1.15	2.00
KM2916-14Cu1CT-C7-1	12.67	22.63	0.88	19.60	44.15	99.93	0.63	0.81	0.03	0.54	2.00
KM2916-14Cu1CT-C8-2	14.60	28.99	0.85	10.22	45.48	100.14	0.70	1.00	0.02	0.28	2.00
KM2916-14Cu1CT-C9-2	19.01	0.07	0.13	38.52	44.49	102.22	0.93	0.00	0.00	1.06	2.00
KM2916-14Cu1CT-C10-1	35.21	0.16	0.20	15.95	48.47	99.99	1.59	0.01	0.01	0.40	2.00
KM2916-14Cu1CT-C11-2	23.24	0.10	0.08	33.11	45.79	102.32	1.11	0.00	0.00	0.89	2.00
KM2916-14Cu1CT-C12-2	15.67	29.04	0.66	8.85	45.73	99.95	0.75	1.00	0.02	0.24	2.00
KM2916-14Cu1CT-C13-2	23.92	0.24	2.10	29.08	45.42	100.76	1.15	0.01	0.06	0.78	2.00
KM2916-14Cu1CT-C14-2	36.75	0.15	0.19	14.82	49.44	101.35	1.62	0.01	0.01	0.37	2.00
KM2916-14Cu1CT-C15-1	41.90	0.06	0.12	7.68	50.58	100.34	1.81	0.00	0.00	0.19	2.00

APPENDIX 2B

Sulphide and Unknown particle determinations by Electron Probe Microanalysis (EPMA)
undertaken by Edith Czech at UBC, Vancouver
July 6, 2012

Sulphide and Unknown Particles for Electron Probe Microanalysis

	S	Mn	Fe	Cu	Zn	As	Cd	Sb	Pb	Total	Comments
KM2916-14CuROTL-S1	0.02	0.79	98.13	0.09	0.05	0.26	0.00	0.01	0.01	99.36	**EDS**: Fe
KM2916-14CuROTL-S2	54.37	0.02	47.44	0.06	0.03	0.07	0.00	0.00	0.00	101.99	
KM2916-14CuROTL-S3	0.02	0.70	97.15	0.41	0.00	0.19	0.00	0.00	0.06	98.52	**EDS**: Fe (traces of Si)
KM2916-14CuROTL-S4	53.58	0.01	47.42	0.00	0.00	0.12	0.00	0.03	0.00	101.16	
KM2916-14CuROTL-S5	0.00	0.69	83.03	0.06	0.02	0.33	0.00	0.00	0.01	84.14	**EDS**: Fe, Cr (traces of Si)
KM2916-14Cu1CT-S1	0.46	1.31	58.32	0.64	0.00	0.05	0.02	0.00	0.00	60.80	**EDS**: Fe, Cr, Ni (traces of Si and S)
KM2916-14Cu1CT-S2	0.21	0.71	91.87	0.18	0.03	0.10	0.00	0.04	0.00	93.13	**EDS**: Fe (traces of Si and S)
KM2916-14Cu1CT-S3	53.44	0.00	47.25	0.04	0.01	0.00	0.00	0.04	0.00	100.78	
KM2916-14Cu1CT-S4	8.01	1.05	58.49	0.62	0.03	0.13	0.00	0.00	0.00	68.32	**EDS**: Fe, Cr , Ni (S and Si minor elements)
KM2916-14Cu1CT-S5	0.03	1.52	65.99	0.50	0.03	0.17	0.04	0.00	0.00	68.28	**EDS**: Fe, Cr, Ni (traces of Si and S)

D2: Static Test Results- ABA and Trace Element

Maxxam Analytics 4606 Canada Way, Burnaby, BC Canada V5G 1K5 Tel: 604 734 7276 Fax: 604 731 2386 www.maxxam.ca

Final Update: Includes Hg on Solids Reported to Stephen Day, Alastair Tiver: 13-Dec-2011

Update: Includes 1DX Reported to Stephen Day, Alastair Tiver: 1-Dec-2011

Reported to Stephen Day, Alastair Tiver; Marcus Hanemaayer, John Andrew & Gregory Smyth: 22-Nov-2011

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Table 1: ABA Test Results for 4 Yellowhead-Harper Creek Pulp Samples - November 2011

				Acme			Acme	HCl Leach (SOP: 7410)	Na2CO3 Leach			Mod. ABA NP			
S. No.	**Sample ID**	**Paste pH (pH Units)**	**Rinse EC (µS/cm)**	**Total Carbon (Wt.%)**	**CO2 (Wt.%)**	**CaCO3 Equiv.* (kg CaCO3/Tonne)**	**Total Sulphur (Wt.%)**	**Sulphate Sulphur (Wt.%)**	**Sulphate Sulphur (Wt.%)**	**Sulphide Sulphur** (Wt.%)**	**Maximum Potential Acidity*** (kg CaCO3/Tonne)**	**Neutralization Potential (kg CaCO3/Tonne)**	**Net Neutralization Potential**** (kg CaCO3/Tonne)**	**Neutralization Potential Ratio (NPR)***** (dimensionless)**	**Fizz Rating**
1	KM 2916-14 Cu Rotl	8.7	588	1.23	4.7	106.4	0.87	<0.01	0.01	0.87	27.2	73.0	45.8	2.7	Slight
2	KM 2916-14 Cu Con	3.5	5920	0.12	0.25	5.7	34.07	0.71	0.72	33.36	1042.5	-8.1	-1050.6	0.0	Slight
3	KM 2916-14 Cu 1CT	6.8	2830	1.66	5.88	133.6	9.26	0.37	0.40	8.89	277.8	80.3	-197.6	0.3	Slight
4	KM 2916 MCI	8.9	488	1.26	4.74	107.7	1.75	<0.01	0.01	1.75	54.7	72.4	17.7	1.3	Slight
Detection Limits		*0.5*	*0.5*	*0.02*	*0.02*	*0.5*	*0.02*	*0.01*	*0.01*	*0.02*	*0.6*				
Maxxam SOP No:		*7160*	*7190*	*LECO*	*LECO*	*Calculation*	*LECO*	*HCl Leach (SOP: 7410)*	*Na2CO3 Leach*	*from HCL Leach*	*Calculation*	*7150*	*Calculation*	*Calculation*	*7150*

Notes:

100% rush charges for 3 week TAT on ABA, sulphate S by Na2CO3 leach & CO2.

Rinse EC & Paste pH was done on as-rec'd pulp.

Total sulphur, total carbon and carbonate carbon (CO2; HCl direct method) done by Leco at Acme Labs.

CO2 Analysis: A 0.2g of pulp sample is digested with 6 ml of 1.8N HCl in a hot water bath of 70 ˚C for 30 minutes. The CO2 that evolves is trapped in a gas chamber that is controlled with a stopcock, once the stopcock is opened the CO2 gas is swept into the Leco analyser with an oxygen carrier gas. Leco then determines the CO2 as total-carbon which is calculated to total CO2.

Calculations:

*CaCO3 equivalents is based on carbonate carbon.

**Sulphide sulphur is based on difference between total sulphur and sulphate sulphur (by HCl leach).

***MPA (Maximum Potential Acidity) is based on sulphide sulphur .

****NNP (Net Neutralization Potential) is based on difference between neutralization potential (NP) and MPA.

*****NPR (Neutralization Potential Ratio) is NP divided by MPA.

References:

Reference for Mod ABA NP method (SOP No. 7150): MEND Acid Rock Drainage Prediction Manual, MEND Project 1.16.1b (pages 6.2-11 to 17), March 1991.

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Table 2: QAQC for 4 Yellowhead-Harper Creek Pulp Samples - November 2011

QA/QC for Paste pH & NP Determination

S. No.	Sample ID	Paste pH (pH Units)	
QAQC		***Reported***	***Duplicate***
Duplicates			
1	KM 2916-14 Cu Rotl	8.7	8.8
S. No.	**Sample ID**	**Rinse EC (µS/cm)**	
QAQC		***Reported***	***Duplicate***
Duplicates			
1	KM 2916-14 Cu Rotl	588	600
S. No.	**Sample ID**	**Modified ABA NP (kgCaCO3/Tonne)**	
QAQC		***Reported***	***Duplicate***
Duplicates			
1	KM 2916-14 Cu Rotl	73	72
Reference Material			
KZK-1 Reference (NP = 58.9) for slight fizz rating		57.1	

QA/QC for Sulphur Speciation

S. No.	Sample ID	Total Sulphur (Wt.%) (Acme)	
QAQC (ACME Labs)		***Reported***	***Duplicate***
Method Blank			
Method Blank		<0.02	
Reference Material			
Maxxam Ref. (0.11% S)		0.11	
STD CSC (4.25% S)		4.23	
STD OREAS76A (18.00% S)		17.06	
S. No.	**Sample ID**	**Sulphate Sulphur (Wt.%)**	
QAQC (Sulphate S (HCl Leach))		***Reported***	***Duplicate***
Duplicates			
1	KM 2916-14 Cu Rotl	<0.01	<0.01
Method Blank			
Method Blank		<0.01	
Reference Material			
Maxxam Ref. (0.06% SO4-S)		0.04	
S. No.	**Sample ID**	**Sulphate Sulphur (Wt.%)**	
QAQC (Sulphate S (Na_2CO_3 Leach))		***Reported***	***Duplicate***
Duplicates			
1	KM 2916-14 Cu Rotl	0.01	0.01
Method Blank			
Method Blank		<0.01	
Reference Material			
Maxxam Ref. (1.2% SO4-S)		1.34	

QA/QC for Carbon Speciation

S. No.	Sample ID	Total Carbon (Wt.%)	
QAQC (ACME Labs)		***Reported***	***Duplicate***
Method Blank			
Method Blank		<0.02	
Reference Material			
STD CSC (2.94% C)		3.00	
STD OREAS76A (0.16% C)		0.15	
S. No.	**Sample ID**	**Carbonate Carbon (Wt.%)**	
QAQC (ACME Labs)		***Reported***	***Duplicate***
Method Blank			
Method Blank		0.02	
Reference Material			
STD CSC (1.55% CO2)		1.69	
STD FER4 (4.86% CO2)		5.28	

Note:

For Maxaam Data (NP, paste pH and sulphate sulphur): The acce

For Acme's Data (total sulphur and carbonate carbon (CO2): The

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Table 3: Aqua Regia digest with ICPMS finsih on 4 Yellowhead-Harper Creek Pulp Samples - November 2011

S. No:	Sample ID	Mo	Cu	Pb	Zn	Ag	Ni	Co	Mn	Fe	As	U	Au	Th	Sr	Cd	Sb	Bi	V	Ca	P
		PPM	PPM	PPM	PPM	PPM	PPM	PPM	PPM	%	PPM	PPM	PPB	PPM	PPM	PPM	PPM	PPM	PPM	%	%
1	KM 2916-14 Cu Rotl	2.6	186.7	10.6	81.00	0	22.5	11.8	508	2.86	13.5	0.4	5.7	2.5	48	0.2	0.1	0.5	15	1.66	0.051
2	KM 2916-14 Cu Con	47.2	>10000.0	1598.3	3271.00	98	135.5	87.9	66	27.04	67.3	0.3	1878.4	0.5	8	21.9	6.6	19.9	<2	0.32	0.002
3	KM 2916-14 Cu 1CT	131.6	2930.4	138.8	160.00	6	332.3	106.5	736	10.82	121.2	1.3	6452.5	3.2	59	1.5	0.8	7.4	17	1.89	0.026
4	KM 2916 MCI	8.5	3157.2	38.0	135.00	2	28.1	17.4	519	3.49	20.1	0.5	28.4	2.4	51	0.6	0.1	1.1	15	1.67	0.050
QAQC																					
Duplicates																					
4	KM 2916 MCI	7.2	3181.4	37.0	138.00	2.4	29.8	18.0	518.00	3.55	21.5	0.5	29.6	2.4	49.00	0.7	0.2	1.1	15.00	1.66	0.05
Method Blank																					
Method Blank		<0.1	<0.1	<0.1	<1	<0.1	<0.1	<0.1	<1	<0.01	<0.5	<0.1	<0.5	<0.1	<1	<0.1	<0.1	<0.1	<2	<0.01	<0.001
Reference Material (1)																					
STD DS8		13.0	109.9	120.8	302.00	2	36.7	7.6	580	2.45	26.5	2.7	104.9	6.8	65	2.4	4.3	6.1	41	0.73	0.090
True Values STD DS8		**13.4**	**110.0**	**123.0**	**312.00**	**2**	**38.1**	**7.5**	**615**	**2.46**	**26.0**	**2.8**	**107.0**	**6.9**	**68**	**2.4**	**4.8**	**6.7**	**41**	**0.70**	**0.080**
Percent Difference		-3.3	-0.1	-1.8	-3.2	0.6	-3.7	1.3	-5.7	-0.4	1.9	-3.6	-2.0	-1.3	-4.0	0.8	-10.4	-8.5	-0.2	4.3	12.5
Reference Material (2)																					
STD OREAS45CA		0.8	532.2	20.3	64.00	0	264.4	93.3	972	16.30	4.0	1.2	44.4	6.9	19	0.1	<0.1	0.2	203	0.45	0.043
True Values STD OREAS45CA		**1.0**	**494.0**	**20.0**	**60.00**	**0**	**240.0**	**92.0**	**943**	**15.69**	**3.8**	**1.2**	**43.0**	**7.0**	**15**	**0.1**	**0.1**	**0.2**	**215**	**0.43**	**0.039**
Percent Difference		-20.0	7.7	1.5	6.7	9.1	10.2	1.4	3.1	3.9	5.3	0.0	3.3	-1.4	26.7	0.0		5.3	-5.6	5.5	11.7
Detection Limits		*0.1*	*0.1*	*0.1*	*1*	*0.1*	*0.1*	*0.1*	*1*	*0.01*	*0.5*	*0.1*	*0.5*	*0.1*	*1*	*0.1*	*0.1*	*0.1*	*2*	*0.01*	*0.001*
Maxxam Method / Acme Group No.		*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*

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S. No:	**Sample ID**	**La**	**Cr**	**Mg**	**Ba**	**Ti**	**B**	**Al**	**Na**	**K**	**W**	**Hg**	**Sc**	**Tl**	**S**	**Ga**	**Se**	**Te**
		PPM	**PPM**	**%**	**PPM**	**%**	**PPM**	**%**	**%**	**%**	**PPM**	**PPM**	**PPM**	**PPM**	**%**	**PPM**	**PPM**	**PPM**
1	KM 2916-14 Cu Rotl	4	41	1.71	25	0.001	<20	1.08	0.015	0.10	<0.1	<0.01	2.3	<0.1	0.82	3	1.7	0.2
2	KM 2916-14 Cu Con	<1	58	0.15	1	<0.001	<20	0.05	<0.001	<0.01	0.5	0.24	0.2	0.30	>10.00	4	>100.0	11.5
3	KM 2916-14 Cu 1CT	4	312	2.25	27	0.003	<20	1.14	0.019	0.09	0.4	0.03	3.2	<0.1	7.94	3	18.9	4.8
4	KM 2916 MCI	3	76	1.78	29	0.001	<20	1.13	0.021	0.12	<0.1	<0.01	2.4	<0.1	1.57	3	5.2	0.8
QAQC																		
Duplicates																		
4	KM 2916 MCI	3.00	76.00	1.78	28.00	0.00	<20	1.14	0.02	0.12	<0.1	<0.01	2.50	<0.1	1.60	3.00	4.9	0.8
Method Blank																		
Method Blank		<1	<1	<0.01	<1	<0.001	<20	<0.01	<0.001	<0.01	<0.1	<0.01	<0.1	<0.1	<0.05	<1	<0.5	<0.2
Reference Material (1)																		
STD DS8		16	112	0.61	278	0.118	<20	0.96	0.096	0.42	2.1	0.20	2.4	5.20	0.16	5	4.7	4.9
True Values STD DS8		**15**	**115**	**0.60**	**279**	**0.113**	**3**	**0.93**	**0.088**	**0.41**	**3.0**	**0.19**	**2.3**	**5.40**	**0.17**	**5**	**5.2**	**5.0**
Percent Difference		9.6	-2.6	0.9	-0.4	4.4		3.2	8.7	2.4	-30.0	4.2	4.3	-3.7	-4.7	6.4	-10.1	-2.0
Reference Material (2)																		
STD OREAS45CA		18	720	0.18	181	0.137	<20	4.14	0.010	0.08	<0.1	0.03	46.1	0.10	<0.05	21	0.6	<0.2
True Values STD OREAS45CA		**16**	**709**	**0.14**	**164**	**0.128**		**3.59**	**0.008**	**0.07**		**0.03**	**39.7**	**0.07**	**0.02**	**18**	**0.5**	
Percent Difference		13.2	1.6	29.9	10.4	7.0		15.3	33.3	11.6		0.0	16.1	42.9		14.1	20.0	
Detection Limits		*1*	*1*	*0.01*	*1*	*0.001*	*20*	*0.01*	*0.001*	*0.01*	*0.1*	*0.01*	*0.1*	*0.1*	*0.05*	*1*	*0.5*	*0.2*
Maxxam Method / Acme Group No.		*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*	*1DX*

Notes:

1DX: Aqua regia digestion followed by ICPMS finish at ACME labs

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Table 4: Total Barium & Fluoride Results on 4 Yellowhead-Harper Creek Pulp Samples - November 2011

S. No:	**Sample ID**	**Ba**	**F (fusion method)**
		PPM	**%**
1	KM 2916-14 Cu Rotl	370	0.08
2	KM 2916-14 Cu Con	11	<0.01
3	KM 2916-14 Cu 1CT	293	0.06
4	KM 2916 MCI	344	0.06
QAQC			
Ba: Reference Material			
STD SO-18 (1)		473	
STD SO-18 (2)		489	
True Values STD SO-18		**515**	
Percent Difference (1)		-8.2	
Percent Difference (2)		-5.0	
F: Reference Material (1)			
STD STSD-1 (True Value: 0.095%)			0.08
True Values STD STSD-1			**0.095**
Percent Difference (1)			-15.8
F: Reference Material (2)			
STD LIBF (True Value: 13.4%)			13.41
True Values STD LIBF			**13.40**
Percent Difference (2)			0.07
Method Blank			
Method Blank		<5	<0.01
Detection Limits		*5*	*0.01%*
Maxxam Method / Acme Group No.		*4A*	*G803*

Notes:

Total Ba on solids:

Fusion ICP assay Method Group 4A

Fluoride on solids:

Fusion ICP assay method (Group: G803) done at Acme Labs.

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Table 4: Total Mercury on 4 Yellowhead-Harper Creek Pulp Samples - November 2011

S. No:	**Sample ID**	**Hg**
		mg/kg
1	KM 2916-14 Cu Rotl	<0.01
2	KM 2916-14 Cu Con	0.12
3	KM 2916-14 Cu 1CT	<0.01
4	KM 2916 MCI	<0.01
QAQC		
Lab Duplicate		
1	KM 2916-14 Cu Rotl	<0.01
Method Blank		
Method Blank		<0.01
Detection Limits		*0.01*
Maxxam Job #		*B1B8270*

Notes:

Total Hg on Solids

Hg by aqua regia digestion and CVAF finsih

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Table 5: ABA Test Results for 2 Yellowhead-Harper Creek Supernatant Samples - November 2011

Parameters	Units	Instrument	RDL	Sample ID KM 2916-14 Unfiltered Cu Ict Supernatant	KM 2916-14 Unfiltered Cu Ict Supernatant (LAB DUP)	KM 2916-14 Cu Rotl Supernatant	QC Batch
pH	pH Units	pH Meter	0.01	9.0		10.3	5291716
Electric Conductivity	µS/cm	Conductivity Meter	0.01	207		206	5291715
Inorganics							
Acidity (pH 4.5)	mg/L	<0.5		<0.5		<0.5	5293789
Acidity (pH 8.3)	mg/L	<0.5		<0.5		<0.5	5293789
Alkalinity (Total as CaCO3)	mg/L	Auto-titrator	0.5	30		68	5219331
Bicarbonate (HCO3)	mg/L	Auto-titrator	0.5	28		6.5	5219331
Carbonate (CO3)	mg/L	Auto-titrator	0.5	4.4		38	5219331
Hydroxide (OH)	mg/L	Auto-titrator	0.5	<0.5		<0.5	5219331
Dissolved Sulphate (SO4)	mg/L	Colourimetic	0.5	26.2		18.6	5223402
Fluoride (F)	mg/L	ISE	0.01	0.14		0.31	5293359
Dissolved Chloride (Cl)	mg/L	Colourimetic	0.5	34		9.9	5308697
Nitrite (N)	mg/L	Colourimetic	0.002	<0.005		<0.005	5300390
Nitrate plus Nitrite (N)	mg/L	Colourimetic	0.002	<0.02		0.03	5300388
Nitrate (N)	mg/L	Calculated	0.002	<0.02		0.03	
Ammonia (N)	mg/L	Colourimetic	0.005	0.07		0.25	5303564
Orthophosphate (P)	mg/L	Colourimetic	0.001	<0.005	<0.005	0.024	5293957
Dissolved Metals by ICP-MS							
Dissolved Hardness (CaCO3)	mg/L	Calc. from Ca & Mg	0.5	84.6		143	5290411
Dissolved Aluminum (Al)	mg/L	ICP-MS	0.0002	2		3.33	5297567
Dissolved Antimony (Sb)	mg/L	ICP-MS	0.00002	0.00359		0.00642	5297567
Dissolved Arsenic (As)	mg/L	ICP-MS	0.00002	0.0017		0.00108	5297567
Dissolved Barium (Ba)	mg/L	ICP-MS	0.00002	0.0172		0.0225	5297567
Dissolved Beryllium (Be)	mg/L	ICP-MS	0.00001	<0.00001		<0.00001	5297567
Dissolved Bismuth (Bi)	mg/L	ICP-MS	0.000005	0.000045		<0.000005	5297567
Dissolved Boron (B)	mg/L	ICP-MS	0.05	<0.05		0.07	5297567
Dissolved Cadmium (Cd)	mg/L	ICP-MS	0.000005	0.000046		0.000008	5297567
Dissolved Calcium (Ca)	mg/L	ICP-ES	0.05	33.3		57.4	5290412
Dissolved Cesium (Cs)	mg/L	ICP-MS	0.00005	0.00013		0.00062	5297567
Dissolved Chromium (Cr)	mg/L	ICP-MS	0.0001	0.0017		0.0002	5297567
Dissolved Cobalt (Co)	mg/L	ICP-MS	0.000005	0.000237		0.000009	5297567
Dissolved Copper (Cu)	mg/L	ICP-MS	0.00005	0.0184		0.00014	5297567
Dissolved Iron (Fe)	mg/L	ICP-MS	0.001	0.353		0.025	5297567
Dissolved Lanthanum (La)	mg/L	ICP-MS	0.00005	<0.00005		<0.00005	5297567
Dissolved Lead (Pb)	mg/L	ICP-MS	0.000005	0.00161		0.000281	5297567
Dissolved Lithium (Li)	mg/L	ICP-MS	0.0005	<0.0005		<0.0005	5297567
Dissolved Magnesium (Mg)	mg/L	ICP-ES	0.05	0.38		<0.05	5290412
Dissolved Manganese (Mn)	mg/L	ICP-MS	0.00005	0.00496		0.00011	5297567
Dissolved Molybdenum (Mo)	mg/L	ICP-MS	0.00005	0.0328		0.0173	5297567

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Table 5: ABA Test Results for 2 Yellowhead-Harper Creek Supernatant Samples - November 2011

Parameters	Units	Instrument	RDL	Sample ID			QC Batch
				KM 2916-14 Unfiltered Cu Ict Supernatant	KM 2916-14 Unfiltered Cu Ict Supernatant (LAB DUP)	KM 2916-14 Cu Rotl Supernatant	
Dissolved Nickel (Ni)	mg/L	ICP-MS	0.00002	0.00099		0.0001	5297567
Dissolved Phosphorus (P)	mg/L	ICP-MS	0.002	0.006		0.005	5297567
Dissolved Potassium (K)	mg/L	ICP-ES	0.05	6.2		16.6	5290412
Dissolved Rubidium (Rb)	mg/L	ICP-MS	0.00005	0.00507		0.00753	5297567
Dissolved Selenium (Se)	mg/L	ICP-MS	0.00004	0.0284		0.0167	5297567
Dissolved Silicon (Si)	mg/L	ICP-MS	0.1	1.2		6.3	5297567
Dissolved Silver (Ag)	mg/L	ICP-MS	0.000005	0.000075		<0.000005	5297567
Dissolved Sodium (Na)	mg/L	ICP-ES	0.05	4.58		9.08	5290412
Dissolved Strontium (Sr)	mg/L	ICP-MS	0.00005	0.175		0.404	5297567
Dissolved Sulphur (S)	mg/L	ICP-ES	10	32		<10	5290412
Dissolved Tellurium (Te)	mg/L	ICP-MS	0.00002	0.00016		0.00104	5297567
Dissolved Thallium (Tl)	mg/L	ICP-MS	0.000002	0.000007		0.000007	5297567
Dissolved Thorium (Th)	mg/L	ICP-MS	0.000005	<0.000005		<0.000005	5297567
Dissolved Tin (Sn)	mg/L	ICP-MS	0.00001	0.0001		0.00007	5297567
Dissolved Titanium (Ti)	mg/L	ICP-MS	0.0005	0.0006		<0.0005	5297567
Dissolved Tungsten (W)	mg/L	ICP-ES	0.00001	0.00185		0.00421	5297567
Dissolved Uranium (U)	mg/L	ICP-ES	0.000002	0.000014		<0.000002	5297567
Dissolved Vanadium (V)	mg/L	ICP-ES	0.0002	0.0006		0.0051	5297567
Dissolved Zinc (Zn)	mg/L	ICP-ES	0.0001	0.0022		0.0007	5297567
Dissolved Zirconium (Zr)	mg/L	ICP-ES	0.0001	0.0018		<0.0001	5297567
Dissolved Mercury (Hg)	ug/L	CVAF	<0.002	<0.002		<0.002	5233328

Ion Balance:							
Anians				2.11		2.04	Maxxam Job #: B192079
Cations				2.29		4.06	
Balance %				-4.09		-32.97	

Rep: Replicates tests performed on the same extract as the original
(1) DL raised do to sample dilution

Table 6: ABA Test Results for 2 Yellowhead-Harper Creek Supernatant Samples - November 2011
Maxxam Job Number: VB1A1734

QA/QC Batch Num	QC Type	Parameter	Date Analyzed yyyy/mm/dd	Value	Recovery	Units	QC Limits
5291711	Matrix Spike	Alkalinity (Total as CaCO3)	10/23/2011		95	%	80 - 120
	Spiked Blank	Alkalinity (Total as CaCO3)	10/23/2011		94	%	80 - 120
	Method Blank	Alkalinity (Total as CaCO3)	10/23/2011	<0.5		mg/L	
		Alkalinity (PP as CaCO3)	10/23/2011	<0.5		mg/L	
		Bicarbonate (HCO3)	10/23/2011	<0.5		mg/L	
		Carbonate (CO3)	10/23/2011	<0.5		mg/L	
		Hydroxide (OH)	10/23/2011	<0.5		mg/L	
	RPD	Alkalinity (Total as CaCO3)	10/23/2011	NC		%	20
		Alkalinity (PP as CaCO3)	10/23/2011	NC		%	20
		Bicarbonate (HCO3)	10/23/2011	NC		%	20
		Carbonate (CO3)	10/23/2011	NC		%	20
		Hydroxide (OH)	10/23/2011	NC		%	20
5291715	Spiked Blank	Conductivity	10/23/2011		96	%	80 - 120
	Method Blank	Conductivity	10/23/2011	<1		uS/cm	
	RPD	Conductivity	10/23/2011	0.9		%	20
5293359	Matrix Spike	Fluoride (F)	10/24/2011		101	%	80 - 120
	Spiked Blank	Fluoride (F)	10/24/2011		102	%	80 - 120
	Method Blank	Fluoride (F)	10/24/2011	<0.01		mg/L	
	RPD	Fluoride (F)	10/24/2011	3.8		%	20
5293789	Spiked Blank	Acidity (pH 8.3)	10/24/2011		102	%	80 - 120
	Method Blank	Acidity (pH 4.5)	10/24/2011	<0.5		mg/L	
		Acidity (pH 8.3)	10/24/2011	<0.5		mg/L	
	RPD	Acidity (pH 4.5)	10/24/2011	NC		%	20
		Acidity (pH 8.3)	10/24/2011	NC		%	20
5293957	Matrix Spike	Orthophosphate (P)	10/24/2011		95	%	80 - 120
	Spiked Blank	Orthophosphate (P)	10/24/2011		97	%	80 - 120
	Method Blank	Orthophosphate (P)	10/24/2011	<0.005		mg/L	
	RPD	Orthophosphate (P)	10/24/2011	NC		%	20
5297567	Matrix Spike	Dissolved Antimony (Sb)	10/26/2011		105	%	80 - 120
		Dissolved Arsenic (As)	10/26/2011		98	%	80 - 120
		Dissolved Barium (Ba)	10/26/2011		NC	%	80 - 120
		Dissolved Beryllium (Be)	10/26/2011		91	%	80 - 120
		Dissolved Bismuth (Bi)	10/26/2011		99	%	80 - 120
		Dissolved Cadmium (Cd)	10/26/2011		103	%	80 - 120
		Dissolved Cesium (Cs)	10/26/2011		106	%	80 - 120
		Dissolved Chromium (Cr)	10/26/2011		95	%	80 - 120
		Dissolved Cobalt (Co)	10/26/2011		95	%	80 - 120
		Dissolved Copper (Cu)	10/26/2011		93	%	80 - 120
		Dissolved Iron (Fe)	10/26/2011		99	%	80 - 120
		Dissolved Lanthanum (La)	10/26/2011		101	%	80 - 120
		Dissolved Lead (Pb)	10/26/2011		101	%	80 - 120
		Dissolved Lithium (Li)	10/26/2011		NC	%	80 - 120
		Dissolved Manganese (Mn)	10/26/2011		98	%	80 - 120
		Dissolved Molybdenum (Mo)	10/26/2011		NC	%	80 - 120

SRK Consulting (Canada) Inc., Yellowhead Mining Inc.-Harper Creek Project, Rec'd 31-Aug-11
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Table 6: ABA Test Results for 2 Yellowhead-Harper Creek Supernatant Samples - November 2011
Maxxam Job Number: VB1A1734

QA/QC Batch Num	QC Type	Parameter	Date Analyzed yyyy/mm/dd	Value	Recovery	Units	QC Limits
		Dissolved Nickel (Ni)	10/26/2011		94	%	80 - 120
		Dissolved Selenium (Se)	10/26/2011		104	%	80 - 120
		Dissolved Silver (Ag)	10/26/2011		109	%	80 - 120
		Dissolved Strontium (Sr)	10/26/2011		NC	%	80 - 120
		Dissolved Tellurium (Te)	10/26/2011		99	%	80 - 120
		Dissolved Thallium (Tl)	10/26/2011		95	%	80 - 120
		Dissolved Tin (Sn)	10/26/2011		102	%	80 - 120
		Dissolved Titanium (Ti)	10/26/2011		109	%	80 - 120
		Dissolved Uranium (U)	10/26/2011		103	%	80 - 120
		Dissolved Vanadium (V)	10/26/2011		96	%	80 - 120
		Dissolved Zinc (Zn)	10/26/2011		105	%	80 - 120
	Spiked Blank	Dissolved Antimony (Sb)	10/26/2011		103	%	80 - 120
		Dissolved Arsenic (As)	10/26/2011		98	%	80 - 120
		Dissolved Barium (Ba)	10/26/2011		106	%	80 - 120
		Dissolved Beryllium (Be)	10/26/2011		88	%	80 - 120
		Dissolved Bismuth (Bi)	10/26/2011		92	%	80 - 120
		Dissolved Cadmium (Cd)	10/26/2011		99	%	80 - 120
		Dissolved Cesium (Cs)	10/26/2011		113	%	80 - 120
		Dissolved Chromium (Cr)	10/26/2011		96	%	80 - 120
		Dissolved Cobalt (Co)	10/26/2011		97	%	80 - 120
		Dissolved Copper (Cu)	10/26/2011		96	%	80 - 120
		Dissolved Iron (Fe)	10/26/2011		102	%	80 - 120
		Dissolved Lanthanum (La)	10/26/2011		106	%	80 - 120
		Dissolved Lead (Pb)	10/26/2011		103	%	80 - 120
		Dissolved Lithium (Li)	10/26/2011		97	%	80 - 120
		Dissolved Manganese (Mn)	10/26/2011		101	%	80 - 120
		Dissolved Molybdenum (Mo)	10/26/2011		104	%	80 - 120
		Dissolved Nickel (Ni)	10/26/2011		97	%	80 - 120
		Dissolved Selenium (Se)	10/26/2011		104	%	80 - 120
		Dissolved Silver (Ag)	10/26/2011		106	%	80 - 120
		Dissolved Strontium (Sr)	10/26/2011		103	%	80 - 120
		Dissolved Tellurium (Te)	10/26/2011		97	%	80 - 120
		Dissolved Thallium (Tl)	10/26/2011		96	%	80 - 120
		Dissolved Tin (Sn)	10/26/2011		99	%	80 - 120
		Dissolved Titanium (Ti)	10/26/2011		103	%	80 - 120
		Dissolved Uranium (U)	10/26/2011		105	%	80 - 120
		Dissolved Vanadium (V)	10/26/2011		100	%	80 - 120
		Dissolved Zinc (Zn)	10/26/2011		97	%	80 - 120
	Method Blank	Dissolved Aluminum (Al)	10/26/2011	0.0003	RDL=0.0002	mg/L	
		Dissolved Antimony (Sb)	10/26/2011	<0.00002		mg/L	
		Dissolved Arsenic (As)	10/26/2011	<0.00002		mg/L	
		Dissolved Barium (Ba)	10/26/2011	<0.00002		mg/L	
		Dissolved Beryllium (Be)	10/26/2011	<0.00001		mg/L	
		Dissolved Bismuth (Bi)	10/26/2011	<0.000005		mg/L	

Table 6: ABA Test Results for 2 Yellowhead-Harper Creek Supernatant Samples - November 2011
Maxxam Job Number: VB1A1734

QA/QC Batch Num	QC Type	Date Analyzed Parameter	yyyy/mm/dd	Value	Recovery	Units	QC Limits
		Dissolved Boron (B)	10/26/2011	<0.05		mg/L	
		Dissolved Cadmium (Cd)	10/26/2011	<0.000005		mg/L	
		Dissolved Cesium (Cs)	10/26/2011	<0.00005		mg/L	
		Dissolved Chromium (Cr)	10/26/2011	<0.0001		mg/L	
		Dissolved Cobalt (Co)	10/26/2011	<0.000005		mg/L	
		Dissolved Copper (Cu)	10/26/2011	<0.00005		mg/L	
		Dissolved Iron (Fe)	10/26/2011	<0.001		mg/L	
		Dissolved Lanthanum (La)	10/26/2011	<0.00005		mg/L	
		Dissolved Lead (Pb)	10/26/2011	<0.000005		mg/L	
		Dissolved Lithium (Li)	10/26/2011	<0.0005		mg/L	
		Dissolved Manganese (Mn)	10/26/2011	<0.00005		mg/L	
		Dissolved Molybdenum (Mo)	10/26/2011	<0.00005		mg/L	
		Dissolved Nickel (Ni)	10/26/2011	<0.00002		mg/L	
		Dissolved Phosphorus (P)	10/26/2011	<0.002		mg/L	
		Dissolved Rubidium (Rb)	10/26/2011	<0.00005		mg/L	
		Dissolved Selenium (Se)	10/26/2011	<0.00004		mg/L	
		Dissolved Silicon (Si)	10/26/2011	<0.1		mg/L	
		Dissolved Silver (Ag)	10/26/2011	<0.000005		mg/L	
		Dissolved Strontium (Sr)	10/26/2011	<0.00005		mg/L	
		Dissolved Tellurium (Te)	10/26/2011	<0.00002		mg/L	
		Dissolved Thallium (Tl)	10/26/2011	<0.000002		mg/L	
		Dissolved Thorium (Th)	10/26/2011	<0.000005		mg/L	
		Dissolved Tin (Sn)	10/26/2011	<0.00001		mg/L	
		Dissolved Titanium (Ti)	10/26/2011	<0.0005		mg/L	
		Dissolved Tungsten (W)	10/26/2011	<0.00001		mg/L	
		Dissolved Uranium (U)	10/26/2011	<0.000002		mg/L	
		Dissolved Vanadium (V)	10/26/2011	<0.0002		mg/L	
		Dissolved Zinc (Zn)	10/26/2011	<0.0001		mg/L	
		Dissolved Zirconium (Zr)	10/26/2011	<0.0001		mg/L	
	RPD	Dissolved Aluminum (Al)	10/26/2011	5.3		%	20
		Dissolved Antimony (Sb)	10/26/2011	3.8		%	20
		Dissolved Arsenic (As)	10/26/2011	1.6		%	20
		Dissolved Barium (Ba)	10/26/2011	2.9		%	20
		Dissolved Beryllium (Be)	10/26/2011	NC		%	20
		Dissolved Bismuth (Bi)	10/26/2011	NC		%	20
		Dissolved Boron (B)	10/26/2011	NC		%	20
		Dissolved Cadmium (Cd)	10/26/2011	NC		%	20
		Dissolved Cesium (Cs)	10/26/2011	NC		%	20
		Dissolved Chromium (Cr)	10/26/2011	NC		%	20
		Dissolved Cobalt (Co)	10/26/2011	3		%	20
		Dissolved Copper (Cu)	10/26/2011	8.3		%	20
		Dissolved Iron (Fe)	10/26/2011	2.3		%	20
		Dissolved Lanthanum (La)	10/26/2011	NC		%	20
		Dissolved Lead (Pb)	10/26/2011	1.4		%	20

SRK Consulting (Canada) Inc., Yellowhead Mining Inc.-Harper Creek Project, Rec'd 31-Aug-11
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Table 6: ABA Test Results for 2 Yellowhead-Harper Creek Supernatant Samples - November 2011
Maxxam Job Number: VB1A1734

QA/QC Batch Num	QC Type	Date Analyzed Parameter	yyyy/mm/dd	Value	Recovery	Units	QC Limits
		Dissolved Lithium (Li)	10/26/2011	0.8		%	20
		Dissolved Manganese (Mn)	10/26/2011	3.5		%	20
		Dissolved Molybdenum (Mo)	10/26/2011	7		%	20
		Dissolved Nickel (Ni)	10/26/2011	12		%	20
		Dissolved Phosphorus (P)	10/26/2011	NC		%	20
		Dissolved Rubidium (Rb)	10/26/2011	4.6		%	20
		Dissolved Selenium (Se)	10/26/2011	2.4		%	20
		Dissolved Silicon (Si)	10/26/2011	4.3		%	20
		Dissolved Silver (Ag)	10/26/2011	NC		%	20
		Dissolved Strontium (Sr)	10/26/2011	1.1		%	20
		Dissolved Tellurium (Te)	10/26/2011	NC		%	20
		Dissolved Thallium (Tl)	10/26/2011	NC		%	20
		Dissolved Thorium (Th)	10/26/2011	NC		%	20
		Dissolved Tin (Sn)	10/26/2011	1.1		%	20
		Dissolved Titanium (Ti)	10/26/2011	NC		%	20
		Dissolved Tungsten (W)	10/26/2011	NC		%	20
		Dissolved Uranium (U)	10/26/2011	NC		%	20
		Dissolved Vanadium (V)	10/26/2011	NC		%	20
		Dissolved Zinc (Zn)	10/26/2011	2		%	20
		Dissolved Zirconium (Zr)	10/26/2011	NC		%	20
5299152	Matrix Spike	Dissolved Chloride (Cl)	10/24/2011		NC	%	80 - 120
	Spiked Blank	Dissolved Chloride (Cl)	10/24/2011		106	%	80 - 120
	Method Blank	Dissolved Chloride (Cl)	10/24/2011	<0.5		mg/L	
	RPD	Dissolved Chloride (Cl)	10/24/2011	0.5		%	20
5300238	Matrix Spike	Dissolved Sulphate (SO4)	10/24/2011		NC	%	80 - 120
	Spiked Blank	Dissolved Sulphate (SO4)	10/24/2011		99	%	80 - 120
	Method Blank	Dissolved Sulphate (SO4)	10/24/2011	<0.5		mg/L	
	RPD	Dissolved Sulphate (SO4)	10/24/2011	NC		%	20
5300388	Matrix Spike	Nitrate plus Nitrite (N)	10/25/2011		NC	%	80 - 120
	Spiked Blank	Nitrate plus Nitrite (N)	10/25/2011		109	%	80 - 120
	Method Blank	Nitrate plus Nitrite (N)	10/25/2011	<0.02		mg/L	
	RPD	Nitrate plus Nitrite (N)	10/25/2011	NC		%	25
5300390	Matrix Spike	Nitrite (N)	10/25/2011		105	%	80 - 120
	Spiked Blank	Nitrite (N)	10/25/2011		103	%	80 - 120
	Method Blank	Nitrite (N)	10/25/2011	<0.005		mg/L	
	RPD	Nitrite (N)	10/25/2011	NC		%	20
5301898	Matrix Spike	Dissolved Mercury (Hg)	10/26/2011		95	%	80 - 120
	Spiked Blank	Dissolved Mercury (Hg)	10/26/2011		101	%	80 - 120
	Method Blank	Dissolved Mercury (Hg)	10/26/2011	<0.002		ug/L	
	RPD	Dissolved Mercury (Hg)	10/26/2011	NC		%	20
5303564	Matrix Spike	Ammonia (N)	10/26/2011		99	%	80 - 120
	Spiked Blank	Ammonia (N)	10/26/2011		100	%	80 - 120
	Method Blank	Ammonia (N)	10/26/2011	0.007	RDL=0.005	mg/L	
	RPD	Ammonia (N)	10/26/2011	3.1		%	20

Table 6: ABA Test Results for 2 Yellowhead-Harper Creek Supernatant Samples - November 2011
Maxxam Job Number: VB1A1734

QA/QC Batch Num	QC Type	Parameter	Date Analyzed yyyy/mm/dd	Value	Recovery	Units	QC Limits
5303864	Matrix Spike	Alkalinity (Total as CaCO3)	10/26/2011		NC	%	80 - 120
	Spiked Blank	Alkalinity (Total as CaCO3)	10/26/2011		96	%	80 - 120
	Method Blank	Alkalinity (Total as CaCO3)	10/26/2011	0.6	RDL=0.5	mg/L	
		Alkalinity (PP as CaCO3)	10/26/2011	<0.5		mg/L	
		Bicarbonate (HCO3)	10/26/2011	0.7	RDL=0.5	mg/L	
		Carbonate (CO3)	10/26/2011	<0.5		mg/L	
		Hydroxide (OH)	10/26/2011	<0.5		mg/L	
	RPD	Alkalinity (Total as CaCO3)	10/27/2011	0.6		%	20
5308697	Spiked Blank	Dissolved Chloride (Cl)	10/27/2011		111	%	80 - 120
	Method Blank	Dissolved Chloride (Cl)	10/27/2011	<0.5		mg/L	

Duplicate: Paired analysis of a separate portion of the same sample. Used to evaluate the variance in the measurement.
Matrix Spike: A sample to which a known amount of the analyte of interest has been added. Used to evaluate sample matrix interference.
Spiked Blank: A blank matrix to which a known amount of the analyte has been added. Used to evaluate analyte recovery.
Method Blank: A blank matrix containing all reagents used in the analytical procedure. Used to identify laboratory contamination.
NC (Matrix Spike): The recovery in the matrix spike was not calculated. The relative difference between the concentration in the parent sample and the spiked amount was not sufficiently significant to permit a reliable recovery calculation.
NC (RPD): The RPD was not calculated. The level of analyte detected in the parent sample and its duplicate was not sufficiently significant to permit a reliable calculation.

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Sample Summary: SRK Consulting (Canada) Inc., Yellowhead Mining Inc.-Harper Creek Project, Rec'd 6-Sep-11
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Date Samples Received: 6-Sep-11

Date Instructions Received: As per email instructions from Stephen Day finalized 25-Oct-11

Sample Prep: None.

Date of Analysis: Mod. ABA NP: 28 to 29-Oct-11;
SO4-S (HCl leach): 25/27-Oct-11 (Maxxam QC Batch: B1A2898);
SO4-S (Na2CO3 leach): 26/28-Oct-11 (Maxxam QC Batch: B1A3325).
Effluent Analysis: 28-Oct-11 (Maxxam Batch B1A1734)

Other Analysis Requested: None.

Client: SRK Consulting (Canada) Inc.
Name of Project: Yellowhead-Harper Creek
Client Project No: 1CY003.000

Contact Person (s): 1) Stephen Day (SRK Consulting (Canada) Inc.
2) Alastair Tiver (Yellowhead Mining)
3) Marcus Hanemaayer (Stewart Group Global)
4) John Andrew (Stewart Group Global)

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Contact No: **SRK (General):** 604-681-4196; Direct: 604-601-8421; Cell: 604-862-4097.
Greg Smyth: 604-685-0543

Fax No: **SRK:** 604-687-5532
Greg Smyth: 604-685-0147

S. No.	Sample ID	Sample Wt. (kg)	Sample Type & Condition
1	KM 2916-14 Cu Rotl 1 of 7	2.3	Tailing
2	KM 2916-14 Cu Rotl 2 of 7	2.28	Tailing
3	KM 2916-14 Cu Rotl 3 of 7	2.3	Tailing
4	KM 2916-14 Cu Rotl 4 of 7	2.26	Tailing
5	KM 2916-14 Cu Rotl 5 of 7	2.33	Tailing
6	KM 2916-14 Cu Rotl 6 of 7	2.24	Tailing
7	KM 2916-14 Cu Rotl 7 of 7	2.24	Tailing
8	KM 2916-14 Cu Con	0.14	Tailing
9	KM 2916-14 Cu 1CT	0.94	Tailing
10	KM 2916 MCI	2.01	Tailing
11	KM 2916-14 Unfiltered Cu Ict Supernatant	12.32	Water
12	KM 2916-14 Cu Rotl Supernatant	12.74	Water

44.1
97.02

Sign:

Report Released by: Kevin Bunten
Position: Senior Scientific Specialist, ARD Division, Maxxam Analytics Inc.
Report Verified by:
Position:
Report Validated by: Tim O'Hearn
Position: Director, ARD Division, Maxxam Analytics Inc.

ARD Project No: 2-21-0900
Acme Group No: VAN11005752

Contact No: 604-734-7276 x2601; Direct: 604-639-2601 (Ivy Rajan)
Contact No:
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Maxxam

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Client: Yellowhead Mining Inc.

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Table 1: ABA Test Results for project HARPER CREEK

Maxxam Sample No	Sample ID	Paste pH	Paste EC	CO2	CaCO3 Equiv.	Total S	HCl Extractable Sulphur	Sulphide Sulphur (by diff.)	Acid Generation Potential	Mod. ABA Neutralization Potential	Fizz Rating	Net Neutralization Potential	Neutralization Potential Ratio
	Units	pH Units	uS/cm	wt%	Kg CaCO3/T	wt%	wt%	wt%	Kg CaCO3/T	Kg CaCO3/T	N/A	Kg CaCO3/T	N/A
CM2696	2916-12	9.4	173	4.70	106.8	0.79	<0.01	0.79	24.7	75.6	SLIGHT	50.9	3.1
CM2697	2916-12 DUP	9.4	180	4.22	95.91	0.77	<0.01	0.77	24.1	74.1	SLIGHT	50.0	3.1
CM2698	2916-13	9.5	184	4.59	104.3	0.72	<0.01	0.72	22.5	78.1	SLIGHT	55.6	3.5
CM2699	2916-13 DUP	9.4	176	4.70	106.8	0.76	<0.01	0.76	23.8	76.6	SLIGHT	52.9	3.2
Detection Limits		*0.5*	*1*	*0.02*		*0.02*	*0.01*	*0.02*	*0.3*				*0.1*
Maxxam SOP #		*7160*	*7160*	*LECO*	*Calculation*	*LECO*	*7410*	*Calculation*	*Calculation*	*7150*	*7150*	*Calculation*	*Calculation*

Notes:

Total sulphur, total carbon and carbonate carbon (CO2; direct HCl method) by Leco furnace done at Acme Labs.

CO2 Analysis: A 0.2g pf pulp sample is digested with 6 ml of 1.8N HCl in a hot water bath of 70 ˚C for 30 minutes. The CO2 that evolves is trapped in a gas chamber that is controled with a stopcock, once the stopcock is is opened the CO2 gas is swept into the Leco analyser with a oxygen carrier gas. Leco then determines the CO2 as total-carbon which is calculated to total CO2.

References:

Reference for Mod ABA NP method (Maxxam SOP No. 7150): MEND Acid Rock Drainage Prediction Manual, MEND Project 1.16.1b (pages 6.2-11 to 17), March 1991.

Reference for HCl extractable Sulphate Sulphur (Maxxam SOP No. 7410): Modified ASTM D2492-02 Method (The S extracted is determined by analysing the extract for sulphate).

Sulphide Sulphur (by.diff.) = Total S - HCl Extractable Sulphur

Acid Generation Potential = Sulphide Sulphur (by diff.)*31.25

Net Neutralization Potential = (Modified ABA Neutralization Potential)-(Acid Generation Potential)

Neutralization Potential Ratio = (Neutralization Potential)/(Acid Generation Potential)

CaCO3 Equivalency = Carbonate Carbon (CO2)*(100/44)*10

Table 2: ABA QAQC Test Results for project HARPER CREEK

	Duplicate QC							
Maxxam Sample No	**Sample ID**	Paste pH Reported	**Paste pH Dup**	CO2 Reported	**CO2 Dup**	**Total S Reported**	Mod. ABA Neutralization Potential Reported	**Mod. ABA Neutralization Potential Reported Dup**
	Units	pH Units	pH Units	wt%	wt%	wt%	Kg CaCO3/T	Kg CaCO3/T
CM2696 Dup	2916-12	9.4	9.4				75.60	73.90
CM2698 Dup	2916-13			4.59	4.60			

Reference Material QC	Total S	Mod. ABA Neutralization Potential Reported
Reference Material (58.9 Kg CaCO3/T)		57.5
STD CSC (2.94% C, 4.25% S)	4.47	
STD GS910-4 (2.65% C, 8.27% S)	8.43	
Units	wt%	Kg CaCO3/T
Blank QC		
Method Blank		<0.1

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Client: Yellowhead Mining Inc.

Table 3: Ultratrace Metals Test Results for project HARPER CREEK

Maxxam Sample No	**Sample ID**	**Mo**	**Cu**	**Pb**	**Zn**	**Ag**	**Ni**	**Co**	**Mn**	**Fe**	**As**	**U**	**Au**	**Th**
	Units	**ppm**	**ppm**	**ppm**	**ppm**	**ppb**	**ppm**	**ppm**	**ppm**	**%**	**ppm**	**ppm**	**ppb**	**ppm**
CM2696	2916-12	2.79	246	14.9	83.7	304	23.9	10.4	523	2.87	12.5	0.5	7.8	2.8
CM2697	2916-12 DUP	3.12	244	15.0	81.1	289	23.0	10.8	521	2.91	11.6	0.5	10.2	3.0
CM2698	2916-13	3.38	249	14.7	84.9	294	23.4	11.3	576	2.96	11.1	0.5	6.7	3.0
CM2699	2916-13 DUP	2.63	248	13.7	78.8	284	24.3	10.7	533	2.94	12.6	0.5	7.2	3.0
Reference Materials														
STD DS8		13.4	107.93	125.02	312.1	1923	36.4	7.4	611	2.47	25.7	2.8	114	7.3
STD DS8 True Values		13.44	110.00	123.00	312.0	1690	38.1	7.5	615	2.46	26	2.8	107.0	6.89
STD OREAS45CA		0.48	524.19	22.18	69.5	274	255.9	89.2	955	15.7	3.5	1.4	38.1	7.9
STD OREAS45CA True Values		1.00	494.00	20.00	60	275	240.0	92.0	943	15.69	3.8	1.2	43	7.0
Detection Limits		0.01	0.01	0.01	0.1	2	0.1	0.1	1	0.01	0.1	0.05	0.2	0.1
Maxxam SOP #														
Method Blank										<0.01				
Method Blank						<2							<0.2	
Method Blank		<0.01	<0.01	<0.01	<0.1		<0.1	<0.1	<1		<0.1	<0.05		<0.1

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Client: Yellowhead Mining Inc.

Table 3: Ultratrace Metals Test Results for project HARPER CREEK

Maxxam Sample No	Sample ID	Sr	Cd	Sb	Bi	V	Ca	P	La	Cr	Mg	Ba	Ti	B
	Units	ppm	ppm	ppm	ppm	ppm	%	%	ppm	ppm	%	ppm	%	ppm
CM2696	2916-12	52.8	0.21	0.12	0.77	15	1.74	0.042	4.3	71.0	1.70	26.9	0.001	<20
CM2697	2916-12 DUP	55.3	0.23	0.13	0.85	15	1.75	0.045	4.0	70.4	1.68	27.6	0.001	<20
CM2698	2916-13	55.0	0.18	0.10	0.80	16	1.82	0.047	4.6	103	1.71	32.8	0.001	<20
CM2699	2916-13 DUP	57.5	0.22	0.09	0.75	15	1.81	0.044	4.1	71.9	1.68	27.3	0.001	<20
Reference Materials														
STD DS8		73.1	2.25	4.49	6.93	42	0.71	0.081	16	122.1	0.6	297.9	0.111	<20
STD DS8 True Values		67.7	2.38	4.80	6.67	41.1	0.70	0.080	14.6	115.0	0.645	279.0	0.113	2.6
STD OREAS45CA		17	0.11	0.09	0.21	212	0.43	0.038	17.6	667.5	0.15	170	0.13	<20
STD OREAS45CA True Values		15.1	0.10	0.13	0.19	215	0.43	0.039	15.9	709.0	0.14	164.0	0.128	
Detection Limits		0.5	0.01	0.02	0.002	2	0.01	0.001	0.5	0.5	0.01	0.5	0.001	20
Maxxam SOP #														
Method Blank							<0.01	<0.001			<0.01		<0.001	
Method Blank														
Method Blank		<0.5	<0.01	<0.02	<0.002	<2			<0.5	<0.5		<0.5		<20

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Client: Yellowhead Mining Inc.

Table 3: Ultratrace Metals Test Results for project HARPER CREEK

Maxxam Sample No	Sample ID Units	Al %	Na %	K %	W ppm	Sc ppm	Tl ppm	S %	Hg ppb	Se ppm	Te ppm	Ga ppm
CM2696	2916-12	1.07	0.019	0.10	<0.05	2.4	0.04	0.73	8	1.5	0.40	2.9
CM2697	2916-12 DUP	1.05	0.018	0.10	<0.05	2.0	0.04	0.79	<5	1.6	0.44	3.0
CM2698	2916-13	1.12	0.022	0.11	<0.05	2.3	0.04	0.74	6	1.5	0.36	3.2
CM2699	2916-13 DUP	1.05	0.018	0.10	<0.05	2.3	0.03	0.78	<5	1.7	0.42	3.0
Reference Materials												
STD DS8		0.91	0.088	0.41	2.6	2.3	5.37	0.15	179	5.1	4.9	4.7
STD DS8 True Values		0.93	0.0883	0.41	3.0	2.3	5.40	0.1679	192	5.23	5.00	4.7
STD OREAS45CA		3.79	0.01	0.07	<0.1	46	0.08	<0.02	28	0.6	0.03	18.6
STD OREAS45CA True Values		3.59	0.0075	0.0717		39.7	0.07	0.021	30	0.5	0.06	18.4
Detection Limits		0.01	0.001	0.01	0.05	0.1	0.02	0.02	5	0.1	0.02	0.1
Maxxam SOP #												
Method Blank		<0.01	<0.001	<0.01				<0.02				
Method Blank									<5			
Method Blank					<0.05	<0.1	<0.02			<0.1	<0.02	<0.1

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Table 5: Sample Summary for project HARPER CREEK

Yellowhead Mining Inc., HARPER CREEK
Page 4 of 4

Date Samples Rec'd by Maxxam: 4 sample were rec'd on 9-Jan-2012.

Sample Prep Conducted by Maxxam: YES

Date of Analysis: February 2012

Client:	Yellowhead Mining Inc.
Client Project Name:	HARPER CREEK
Client Project No:	
ARD Project #:	
Maxxam Job No:	B201776
Contact Person:	Alastair Tiver: ativer@yellowheadmining.com
E-mail Address:	**Ashley Landriault: alandriault@srk.com** **Rosemarie Cocuaco: rcocuaco@srk.com** **Steve Day: sday@srk.com**

Data Validated by:	Ashley Leow
Position:	Burnaby ARD Supervisor

Sample Storage

Sample rejects (and selected test residues where applicable) have been archived
Standard archive protocol is archiving for samples for 3 months after testing is complete.
If archiving is required past 3 months a fee will be required.

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Client: Yellowhead Mining Inc.

Table 1: ABA Test Results for project YHM Harper Creek 1CY003.001

Maxxam Sample No	Sample ID	Paste pH	Paste EC	CO2	CaCO3 Equiv.	Total S	HCl Extractable Sulphur	Sulphide Sulphur (by diff.)	Acid Generation Potential	Mod. ABA Neutralization Potential	Fizz Rating	Net Neutralization Potential	Neutralization Potential Ratio
	Units	**pH Units**	**uS/cm**	**wt%**	**Kg CaCO3/T**	**wt%**	**wt%**	**wt%**	**Kg CaCO3/T**	**Kg CaCO3/T**	**N/A**	**Kg CaCO3/T**	**N/A**
DW8018	3321-P2 WSB CL TAILS	8.01	1050	4.59	104.3	5.85	0.04	5.81	181.6	75.6	SLIGHT	-106.0	0.4
DW8019	3321-P2 WSB RO TAILS	9.26	310	4.23	96.1	0.96	0.01	0.95	29.7	75.9	SLIGHT	46.2	2.6
Detection Limit		*N/A*	*1*	*0.02*	*0.5*	*0.02*	*0.01*	*0.02*	*0.6*	*0.1*	*N/A*	*0.1*	*N/A*
Maxxam SOP #		*7160*		*LECO*	*Calculation*	*LECO*	*BBY0SOP-00009*	*Calculation*	*Calculation*	*BBY0SOP-00010*	*BBY0SOP-00*	*Calculation*	*Calculation*

References:

Acid Generation Potential = Sulphide Sulphur (by diff.)*31.25

CaCO3 Equivalency = Carbonate Carbon (CO2)*(100/44)*10

Fizz Rating - Reference method used is based on NP method.

Net Neutralization Potential = (Modified ABA Neutralization Potential)-(Acid Generation Potential (S-S by diff))

Mod. ABA Neutralization Potential - MEND Acid Rock Drainage Prediction Manual, MEND Project 1.16.1b (pages 6.2-11 to 17), March 1991.

Neutralization Potential Ratio = (Neutralization Potential)/(Acid Generation Potential)

Paste EC - based on Field and Laboratory Methods Applicable to Overburdens and Minesoils, (EPA 600 / 2-78-054, March 1978).

Paste pH - Field and Laboratory Methods Applicable to Overburdens and Minesoils, (EPA 600 / 2-78-054, March 1978).

Sulphide Sulphur = (Total Sulphur)-(Sulphate Sulphur)

Client: Yellowhead Mining Inc.

Table 2: ABA QAQC Test Results for project YHM Harper Creek 1CY003.001

	Duplicate QC										
Maxxam Sample No	**Sample ID**	Paste pH Reported	**Paste pH Dup**			HCl Extractable Sulphur Reported	**HCl Extractable Sulphur**	Mod. ABA Neutralization Potential Reported	**Mod. ABA Neutralization Potential Reported Dup**	Fizz Rating Reported	**Fizz Rating Dup**
	Units	pH Units	pH Units			wt%	wt%	Kg CaCO3/T	Kg CaCO3/T	N/A	N/A
DW8018 Dup	3321-P2 WSB CL TAILS	8.01	7.97			0.04	0.04	75.6	74.7	SLIGHT	SLIGHT

Reference Material QC	CO2	Total S	HCl Extractable Sulphur	Mod. ABA Neutralization Potential Reported
Units	wt%	wt%	wt%	Kg CaCO3/T
ARD Ref Mat C&S (6037874) (8.27 wt%)		8.3		
ARD Spike C02 (6037882) (4.86 wt%)	4.6			
KZK-1ModS Slight (6042235) (58.9 Kg CaCO3/T)				60.1
ARD Ref Mat C&S (6037874) (2.35 wt%)		2.4		
ARD Spike C02 (6037882) (1.55 wt%)	1.3			
ARD Ref Mat C&S (6037874) (0.16 wt%)		0.17		
RS10 STD (0.06 % S)			0.07	
Blank QC				
Method Blank			<0.01	
Method Blank		<0.02		
Method Blank	<0.02			

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Client: Yellowhead Mining Inc.

Table 3: Ultratrace Metals Test Results for project YHM Harper Creek 1CY003.001

Maxxam Sample No	**Sample ID**	**Mo**	**Cu**	**Pb**	**Zn**	**Ag**	**Ni**	**Co**	**Mn**	**Fe**	**As**	**U**	**Au**	**Th**	**Sr**	**Cd**	**Sb**	**Bi**	**V**	**Ca**	**P**
	Units	**ppm**	**ppm**	**ppm**	**ppm**	**ppb**	**ppm**	**ppm**	**ppm**	**%**	**ppm**	**ppm**	**ppb**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**%**	**%**
DW8018	3321-P2 WSB CL TAILS	15.1	532	80.7	132	1860	109	71.1	673	7.56	71.1	0.9	62.3	3.5	58.6	0.57	0.50	4.24	16	1.92	0.064
DW8019	3321-P2 WSB RO TAILS	5.62	215	23.4	82.8	347	34.4	13.5	585	3.31	14.3	0.5	14.9	3.3	55.5	0.24	0.11	1.14	15	1.91	0.056
		0.01	0.01	0.01	0.1	2	0.1	0.1	1	0.01	0.1	0.05	0.2	0.1	0.5	0.01	0.02	0.002	2	0.01	0.001
QA/QC																					
Blanks																					
Method Blank										<0.01										<0.01	<0.001
Method Blank						6							<0.2								
Method Blank		<0.01	0.12	0.02	<0.1		<0.1	<0.1	<1		<0.1	<0.05		<0.1	<0.5	<0.01	<0.02	<0.002	<2		
Reference Material																					
REF DS9% (6037999)										2.36										0.740	0.0800
True Values REF DS9%										**2.37**										**0.776**	**0.0844**
Percent Difference (6037999)										-0.42										-4.64	-5.21
Reference Material																					
REF DS9 PPB (6038000)													106								
True Values REF DS9 PPB													**102**								
Percent Difference (6038000)													3.92								
Reference Material																					
REF DS9 PPM (6038001)		13.1	111	125	308		40.8	7.3	569		25.0	2.90		7.90	78.2	2.3	4.52	6.69	40		
True Values REF DS9 PPM		**12.74**	**104**	**126**	**322**		**39.5**	**7.6**	**586**		**27**	**2.9**		**7.15**	**76.1**	**2.3**	**4.84**	**6.78**	**40**		
Percent Difference (6038001)		2.82	6.73	-0.79	-4.35		3.29	-3.95	-2.90		-7.41	0.00		10.49	2.76	0.00	-6.61	-1.33	0.00		
Reference Material																					
REFMAT OREAS45CA (%) (6037999)										15.9										0.460	0.0400
True Values REFMAT OREAS45CA										**15.69**										**0.4265**	**0.0385**
Percent Difference (6037999)										-5.04										7.85	3.89
Reference Material																					
REFMAT OREAS45CA PPB (6038000)						291							42.1								
True Values REFMAT OREAS45CA PPB						**275**							**43**								
Percent Difference (6038000)						5.82							-2.09								
Reference Material																					
REFMAT OREAS45CA PPM (6038001)		0.98	531	23.2	61.1		261	100			3.10	1.30		6.90	17.4	0.110	0.10	0.18	218		
True Values REFMAT OREAS45CA PPM		**1**	**494**	**20**	**60**		**240**	**92**			**3.8**	**1.2**		**7**	**15**	**0.1**	**0.13**	**0.19**	**215**		
Percent Difference (6038001)		-2.00	7.49	16.0	1.83		8.75	8.00			-18.42	8.33		-1.43	16.0	10.0	-23.08	-5.26	1.40		
Detection Limits		0.01	0.01	0.01	0.1	2	0.1	0.1	1	0.01	0.1	0.05	0.2	0.1	0.5	0.01	0.02	0.002	2	0.01	0.001
Acme Method		*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*

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Client: Yellowhead Mining Inc.

Table 3: Ultratrace Metals Test Results for project YHM Harper Creek 1CY003.001

Maxxam Sample No	Sample ID Units	La ppm	Cr ppm	Mg %	Ba ppm	Ti %	B ppm	Al %	Na %	K %	W ppm	Sc ppm	Tl ppm	Hg ppb	Se ppm	Te ppm	Ga ppm	S %
DW8018	3321-P2 WSB CL TAILS	3.5	151	1.88	24.4	0.001	<20	1.03	0.010	0.06	0.4	3.0	0.08	49	11.4	2.40	3.3	5.42
DW8019	3321-P2 WSB RO TAILS	5.4	75.5	1.70	43.3	0.001	<20	1.10	0.023	0.12	<0.1	2.7	0.05	6	1.9	0.56	3.4	0.88
		0.5	0.5	0.01	0.5	0.001	20	0.01	0.001	0.01	0.05	0.1	0.02	5	0.1	0.02	0.1	0.02
QA/QC																		
Blanks																		
Method Blank				<0.01		<0.001		<0.01	<0.001	<0.01								<0.02
Method Blank														<5				
Method Blank		<0.5	<0.5		<0.5		<20				<0.05	<0.1	<0.02		<0.1	<0.02	<0.1	
Reference Material																		
REF DS9% (6037999)				0.640		0.154		0.970	0.0850	0.410								0.170
True Values REF DS9%				**0.6437**		**0.1239**		**0.9915**	**0.0905**	**0.3874**								**0.1737**
Percent Difference (6037999)				-0.57		24.29		-2.17	-6.08	5.83								-2.13
Reference Material																		
REF DS9 PPB (6038000)																		
True Values REF DS9 PPB																		
Percent Difference (6038000)																		
Reference Material																		
REF DS9 PPM (6038001)		18.3	112		324		<20				<0.05	2.6	5.20		5.4	5.07	4.60	
True Values REF DS9 PPM		**15.7**	**119**		**308**						**3**	**2.8**	**5.48**		**5.4**	**5**	**4.84**	
Percent Difference (6038001)		16.56	-5.88		5.19							-7.14	-5.38		0.00	1.40	-4.96	
Reference Material																		
REFMAT OREAS45CA (%) (6037999)				0.140		0.115		3.68	0.0080	0.0710								<0.02
True Values REFMAT OREAS45CA				**0.1358**		**0.128**		**3.592**	**0.0075**	**0.0717**								**0.021**
Percent Difference (6037999)				3.09		-10.16		2.45	6.67	-0.98								
Reference Material																		
REFMAT OREAS45CA PPB (6038000)														36.0				
True Values REFMAT OREAS45CA PPB														**30**				
Percent Difference (6038000)														20.0				
Reference Material																		
REFMAT OREAS45CA PPM (6038001)		14.7	717		172		<20				3.00	46.20	0.07		0.3	0.0600	19.80	
True Values REFMAT OREAS45CA PPM		**15.9**	**709**		**164**							**39.7**	**0.07**		**0.5**	**0.06**	**18.4**	
Percent Difference (6038001)		-7.55	1.13		97.50							16.37	0.00		-40.0	0.00	7.61	
Detection Limits		0.5	0.5	0.01	0.5	0.001	20	0.01	0.001	0.01	0.05	0.1	0.02	5	0.1	0.02	0.1	0.02
Acme Method		*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*	*1F-MS*

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Table 4: Hg-CVAF Test Results for project YHM Harper Creek 1CY003.001

Maxxam Sample No	Sample ID	Hg on Solids
	Units	**mg/kg**
DW8018	3321-P2 WSB CL TAILS	0.043
DW8019	3321-P2 WSB RO TAILS	<0.010
Reportable Detection Limits		*0.010*
QAQC		
Duplicates		
DW8018 Dup	3321-P2 WSB CL TAILS	0.042
Blanks		
Method Blank		<0.010
Reference Material		
Hg Soil Spike 1 ppm (6039017)		1.01
True Values Hg Soil Spike 1 ppm		**1**
Percent Difference (6039017)		1.00
Reference Material		
Hg Soil CRM SS-2 (6039017)		0.32
True Values Hg Soil CRM SS-2		**0.33**
Percent Difference (6039017)		-3.03
Detection Limits		0.010
Maxxam SOP #		65-C-015-03

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Table 5: Fluorine Test Results for project YHM Harper Creek 1CY003.001

Maxxam Sample No	Sample ID	Fluorine
	Units	%
DW8018	3321-P2 WSB CL TAILS	0.08
DW8019	3321-P2 WSB RO TAILS	0.05
Reportable Detection Limits		*0.01*
QAQC		
Duplicates		
DW8019 Dup	3321-P2 WSB RO TAILS	0.06
Blanks		
Method Blank		<0.01
Reference Material		
STD STSD-1 (6037891)		0.09
True Values STD STSD-1		**0.095**
Percent Difference (6037891)		1.00
Reference Material		
STD LIBF (6037891)		13.77
True Values STD LIBF		**13.4**
Percent Difference (6037891)		-3.03
Detection Limits		0.010
Acme Method		*G803*

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Table 6: Sample List for project YHM Harper Creek 1CY003.001

Maxxam Sample ID	Client Sample ID	Sample Form	Dry Weight Received (kg)
DW8018	3321-P2 WSB CL TAILS	Dry Pulp	0.2260
DW8019	3321-P2 WSB RO TAILS	Dry Tail	0.5000

Total Weight	0.73
Total Samples Received	2

Maxxam Analytics 4606 Canada Way, Burnaby, BC Canada V5G 1K5 Tel: 604 734 7276 Fax: 604 731 2386 www.maxxam.ca

Table 7: Sample Summary for project YHM Harper Creek 1CY003.001

Yellowhead Mining Inc.,

Date Samples Rec'd by Maxxam: 2 sample were rec'd on 19-June-2012.

Sample Prep Conducted by Maxxam: YES

Date of Analysis: July 2012

Client:	Yellowhead Mining Inc.
Client Project Name:	
Client Project No:	HARPER CREEK
ARD Project #:	
Maxxam Job No:	B259070
Contact Person:	Ashley Landriault: alandriault@srk.com
E-mail Address:	Steve Day: sday@srk.com

Data Validated by:	Ashley Leow
Position:	Burnaby ARD Supervisor

Sample Storage

Sample rejects (and selected test residues where applicable) have been archived
Standard archive protocol is archiving for samples for 3 months after testing is complete.
If archiving is required past 3 months a fee will be required.

D3: XRD Test Results

QUANTITATIVE PHASE ANALYSIS OF TWO POWDER SAMPLES USING THE RIETVELD METHOD AND X-RAY POWDER DIFFRACTION DATA.

Project: Yellowhead Mining – Harper Creek Tailings

Kevin Bunten, Ph.D.
Maxxam Analytics Inc.
4606 Canada Way
Burnaby, BC V5G 1K5

Mati Raudsepp, Ph.D.
Elisabetta Pani, Ph.D.
Edith Czech, M.Sc.
Jenny Lai, B.Sc.

Dept. of Earth & Ocean Sciences
The University of British Columbia
6339 Stores Road
Vancouver, BC V6T 1Z4

June 11, 2012

EXPERIMENTAL METHOD

The two samples of **Project Yellowhead Mining – Harper Creek Tailings** were reduced to the optimum grain-size range for quantitative X-ray analysis (<10 μm) by grinding under ethanol in a vibratory McCrone Micronising Mill for 7 minutes. Step-scan X-ray powder-diffraction data were collected over a range 3-80°2θ with CoKa radiation on a Bruker D8 Focus Bragg-Brentano diffractometer equipped with an Fe monochromator foil, 0.6 mm (0.3°) divergence slit, incident- and diffracted-beam Soller slits and a LynxEye detector. The long fine-focus Co X-ray tube was operated at 35 kV and 40 mA, using a take-off angle of 6°.

RESULTS

The X-ray diffractograms were analyzed using the International Centre for Diffraction Database PDF-4 and Search-Match software by Siemens (Bruker). X-ray powder-diffraction data of the samples were refined with Rietveld program Topas 4.2 (Bruker AXS). The results of quantitative phase analysis by Rietveld refinements are given in Table 1. These amounts represent the relative amounts of crystalline phases normalized to 100%. The Rietveld refinement plots are shown in Figures 1 – 2.

Table 1. Results of quantitative phase analysis (wt.%)

Mineral	**Ideal Formula**	**1** **KM2916 MCI**	**2** **KM2916-14 Cu ROTL**
Quartz	SiO_2	51.2	52.2
Clinochlore	$(Mg,Fe^{2+})_5Al(Si_3Al)O_{10}(OH)_8$	5.7	5.9
Kaolinite	$Al_2Si_2O_5(OH)_4$	1.1	1.0
Muscovite	$KAl_2(AlSi_3O_{10})(OH)_2$	18.8	20.2
Paragonite	$NaAl_2AlSi_3O_{10}(OH)_2$	1.4	1.5
Plagioclase	NaAlSi3O8 – CaAl2Si2O8	4.4	4.5
K-feldspar	KAlSi3O8	1.5	1.3
Ankerite/Dolomite	$Ca(Fe^{2+},Mg,Mn)(CO_3)_2/CaMg(CO_3)_2$	8.1	8.0
Magnesite	$MgCO_3$	2.5	2.6
Calcite	$CaCO_3$	2.0	0.5
Pyrite	FeS_2	2.8	1.7
Rutile	TiO_2	0.5	0.7
Total		100.0	100.0

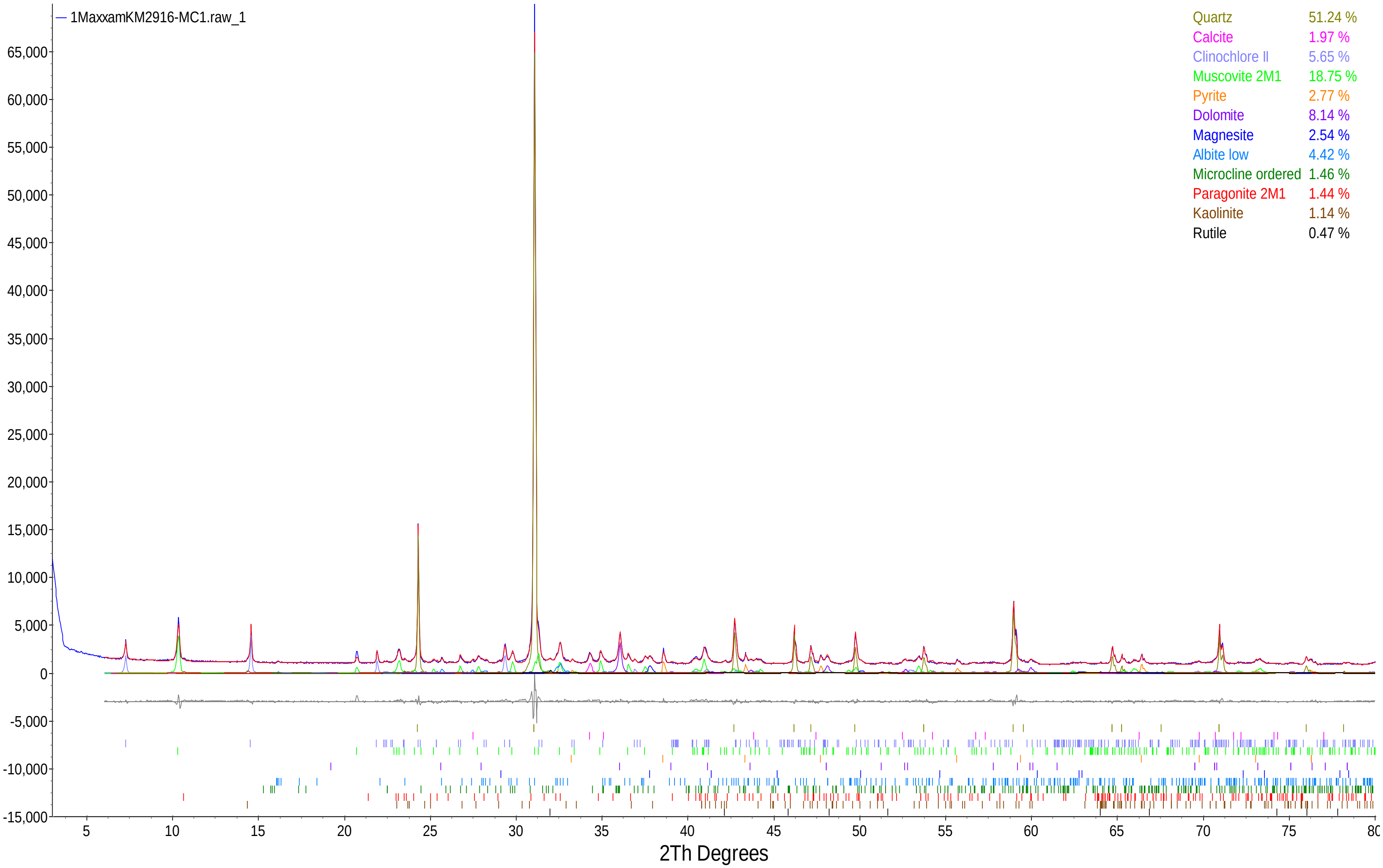


Figure 1. Rietveld refinement plot of sample **"1: KM2916 MCI"** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

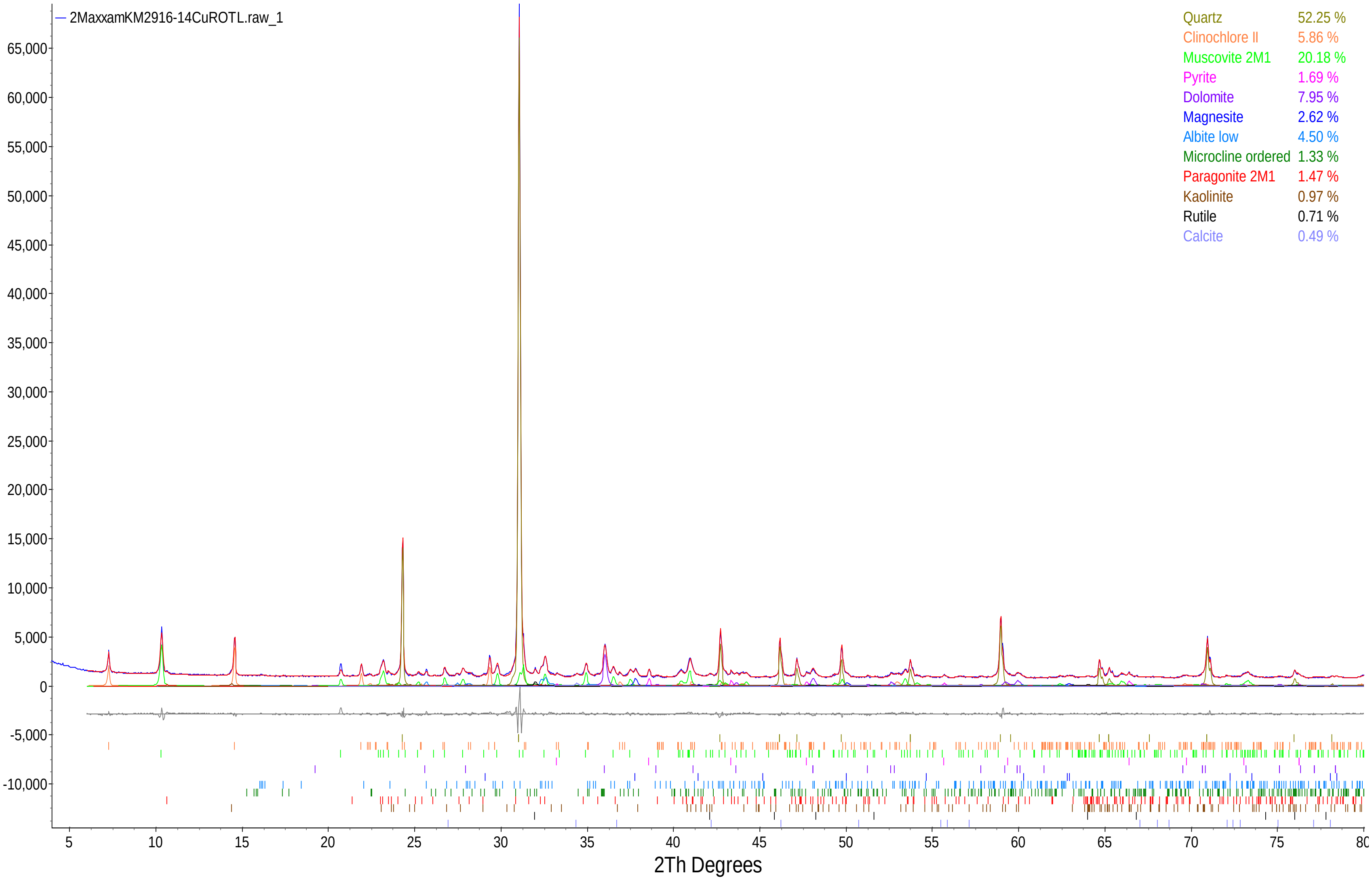


Figure 2. Rietveld refinement plot of sample **“2: KM2916-14 Cu ROTL”** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

QUANTITATIVE PHASE ANALYSIS OF ONE POWDER SAMPLE USING THE RIETVELD METHOD AND X-RAY POWDER DIFFRACTION DATA.

Project: Yellowhead Mining – Harper Creek Ore Comp.

Kevin Bunten, Ph.D.
Maxxam Analytics Inc.
4606 Canada Way
Burnaby, BC V5G 1K5

Mati Raudsepp, Ph.D.
Elisabetta Pani, Ph.D.
Jenny Lai, B.Sc.
Edith Czech, M.Sc.

Dept. of Earth & Ocean Sciences
The University of British Columbia
6339 Stores Road
Vancouver, BC V6T 1Z4

June 5, 2012

EXPERIMENTAL METHOD

The sample of **Project Yellowhead Mining – Harper Creek Ore Comp.** was reduced to the optimum grain-size range for quantitative X-ray analysis (<10 μm) by grinding under ethanol in a vibratory McCrone Micronising Mill for 7 minutes. Step-scan X-ray powder-diffraction data were collected over a range 3-80°2θ with CoKa radiation on a Bruker D8 Focus Bragg-Brentano diffractometer equipped with an Fe monochromator foil, 0.6 mm (0.3°) divergence slit, incident- and diffracted-beam Soller slits and a LynxEye detector. The long fine-focus Co X-ray tube was operated at 35 kV and 40 mA, using a take-off angle of 6°.

RESULTS

The X-ray diffractogram was analyzed using the International Centre for Diffraction Database PDF-4 and Search-Match software by Siemens (Bruker). X-ray powder-diffraction data of the sample was refined with Rietveld program Topas 4.2 (Bruker AXS). The results of quantitative phase analysis by Rietveld refinements are given in Table 1. These amounts represent the relative amounts of crystalline phases normalized to 100%. The Rietveld refinement plot is shown in Figure 1.

Table 1. Results of quantitative phase analysis (wt.%)

Mineral	**Ideal Formula**	**KM 2916-14 Cu 1CT**
Quartz	SiO_2	34.2
Clinochlore	$(Mg,Fe^{2+})_5Al(Si_3Al)O_{10}(OH)_8$	6.1
Muscovite	$KAl_2(AlSi_3O_{10})(OH)_2$	16.3
Plagioclase	NaAlSi3O8 – CaAl2Si2O8	2.6
K-feldspar	KAlSi3O8	1.8
Calcite	$CaCO_3$	1.4
Ankerite/Dolomite	$Ca(Fe^{2+},Mg,Mn)(CO_3)_2/CaMg(CO_3)_2$	8.0
Magnesite	$MgCO_3$	6.8
Cerussite ?	$PbCO_3$	0.1
Gypsum	$CaSO_4 \cdot 2H_2O$	1.8
Pyrite	FeS_2	18.2
Zircon	$ZrSiO_4$	2.7
Total		100.0

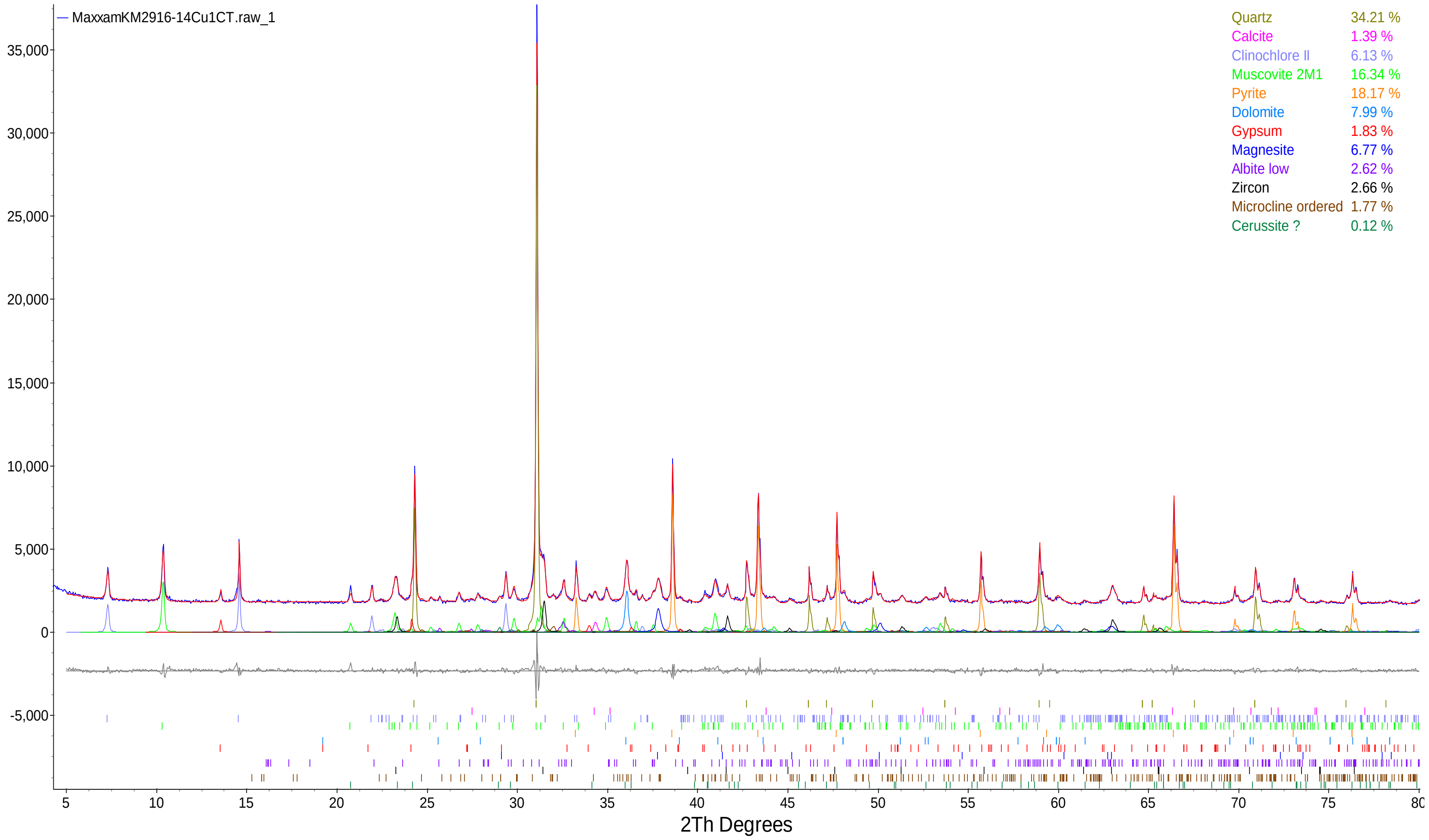
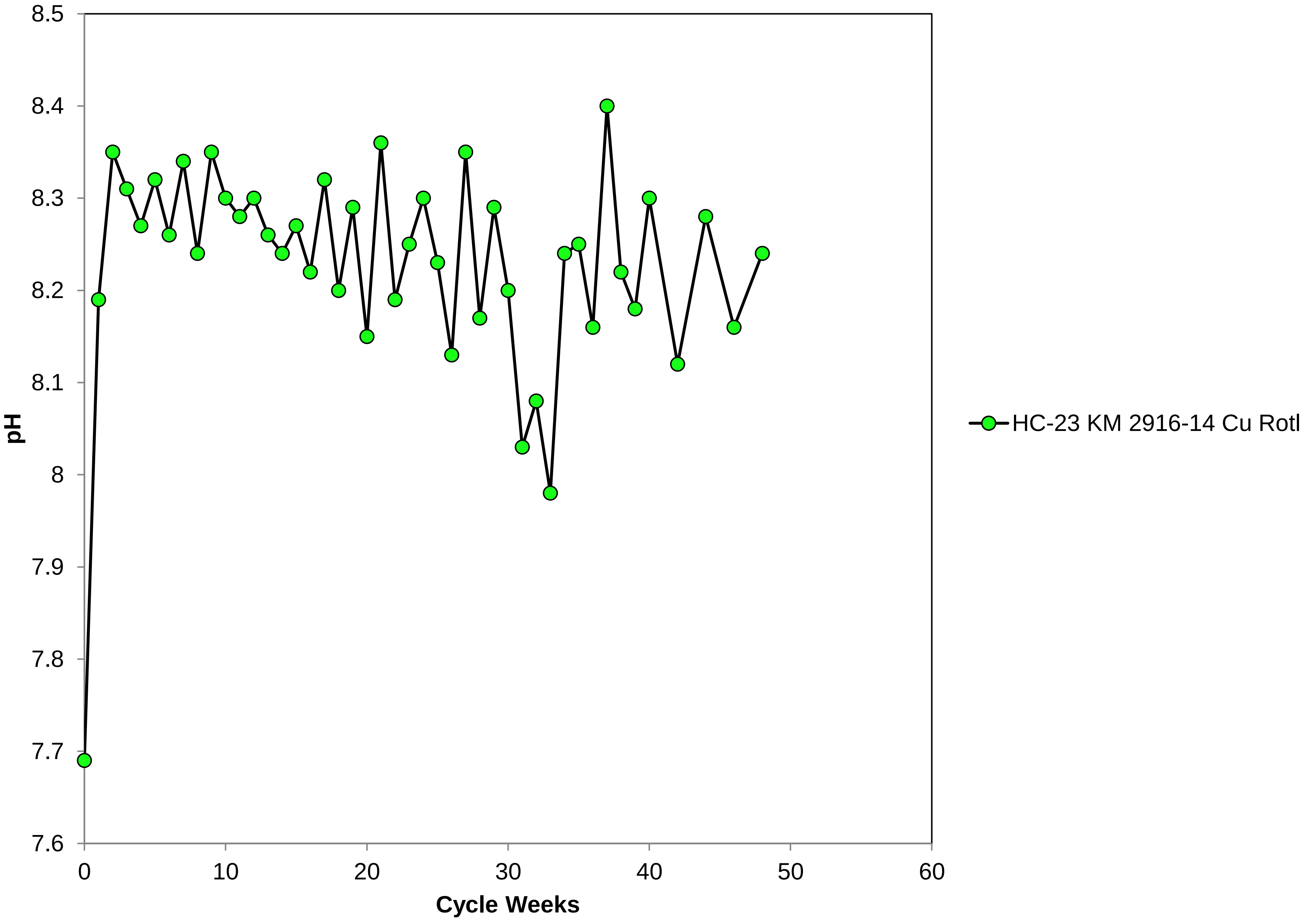
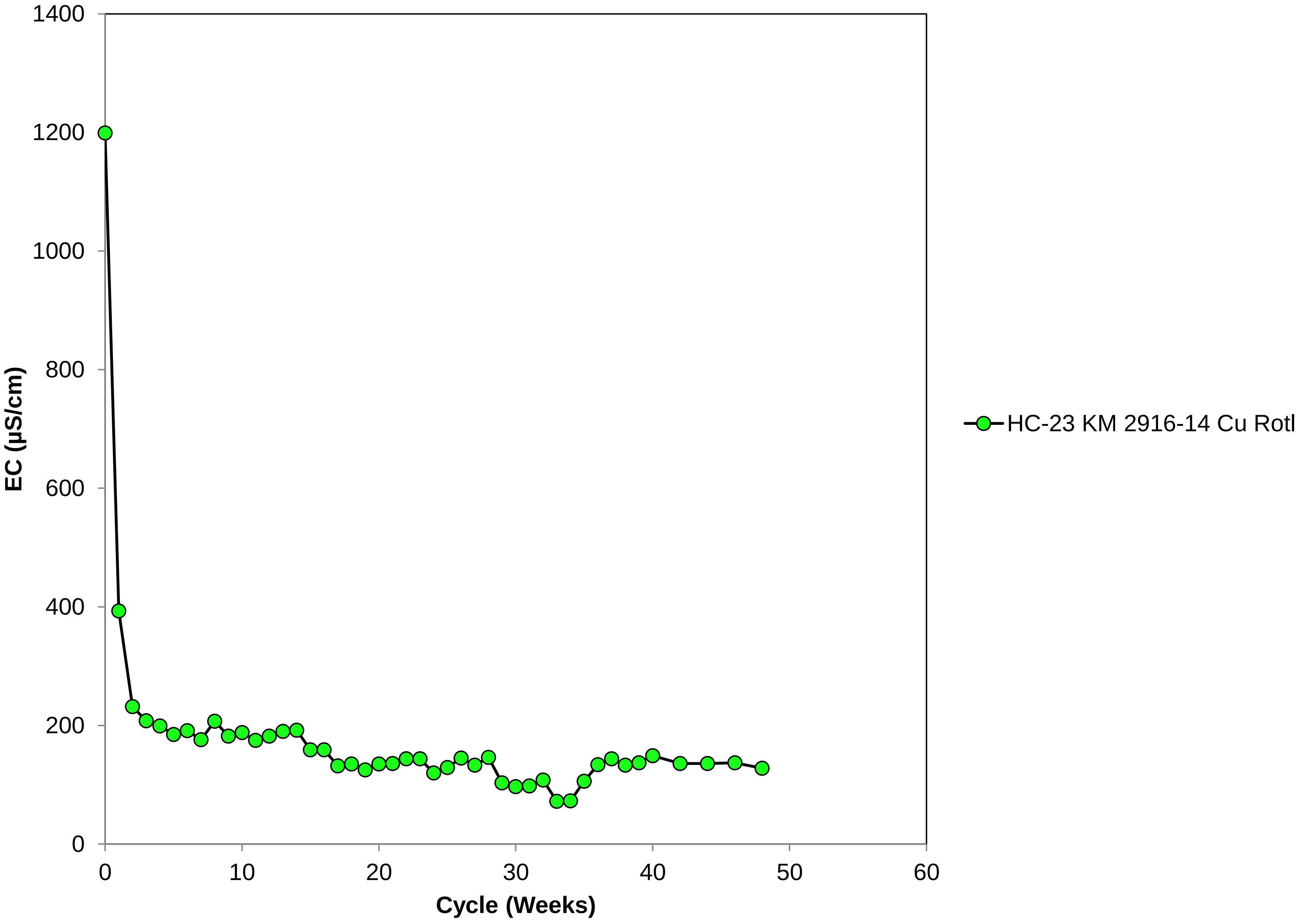
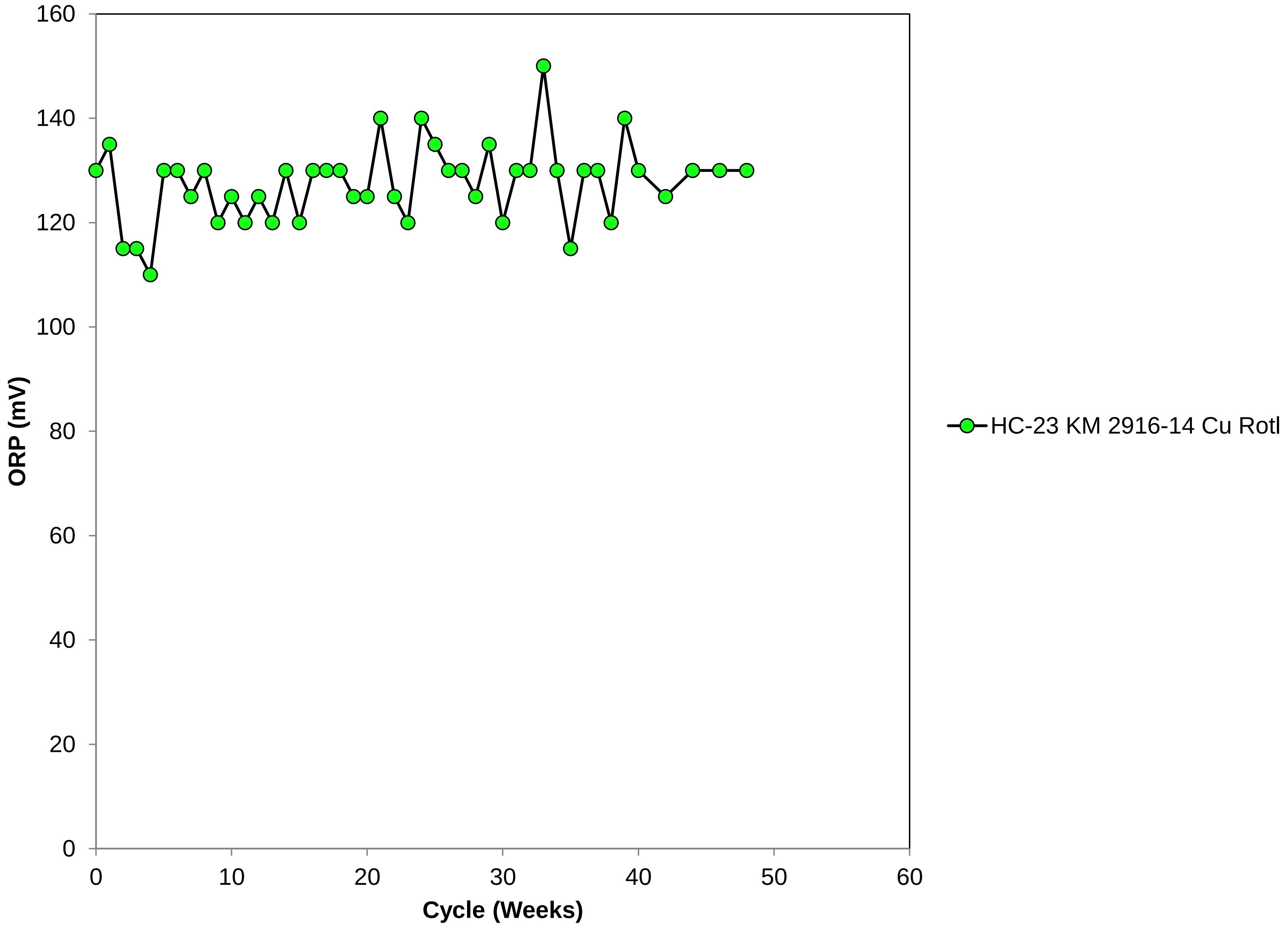


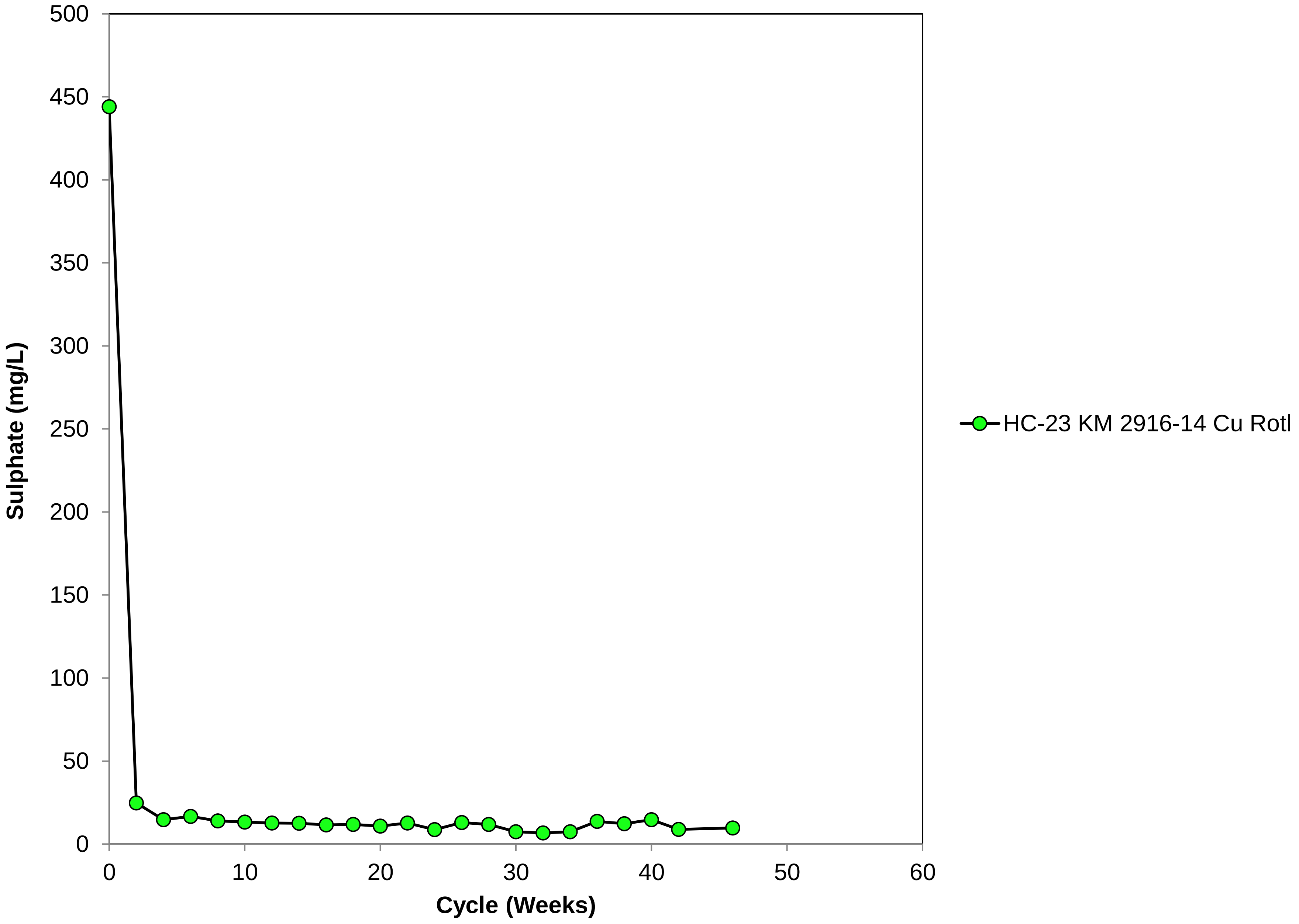
Figure 1. Rietveld refinement plot of sample **“KM 2916-14 Cu 1CT”** (blue line - observed intensity at each step; red line - calculated pattern; solid grey line below – difference between observed and calculated intensities; vertical bars, positions of all Bragg reflections). Coloured lines are individual diffraction patterns of all phases.

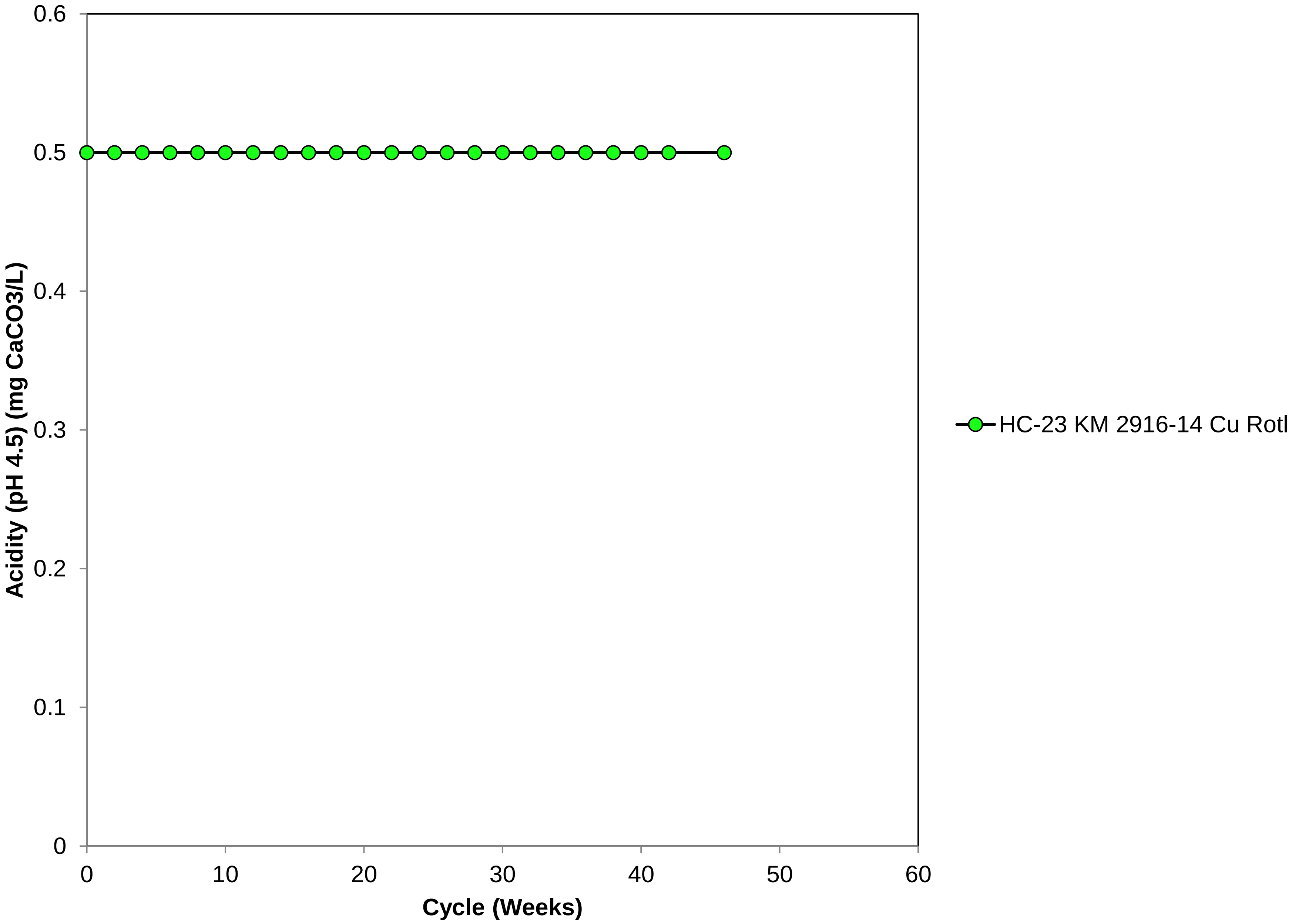
D4: Humidity Cells

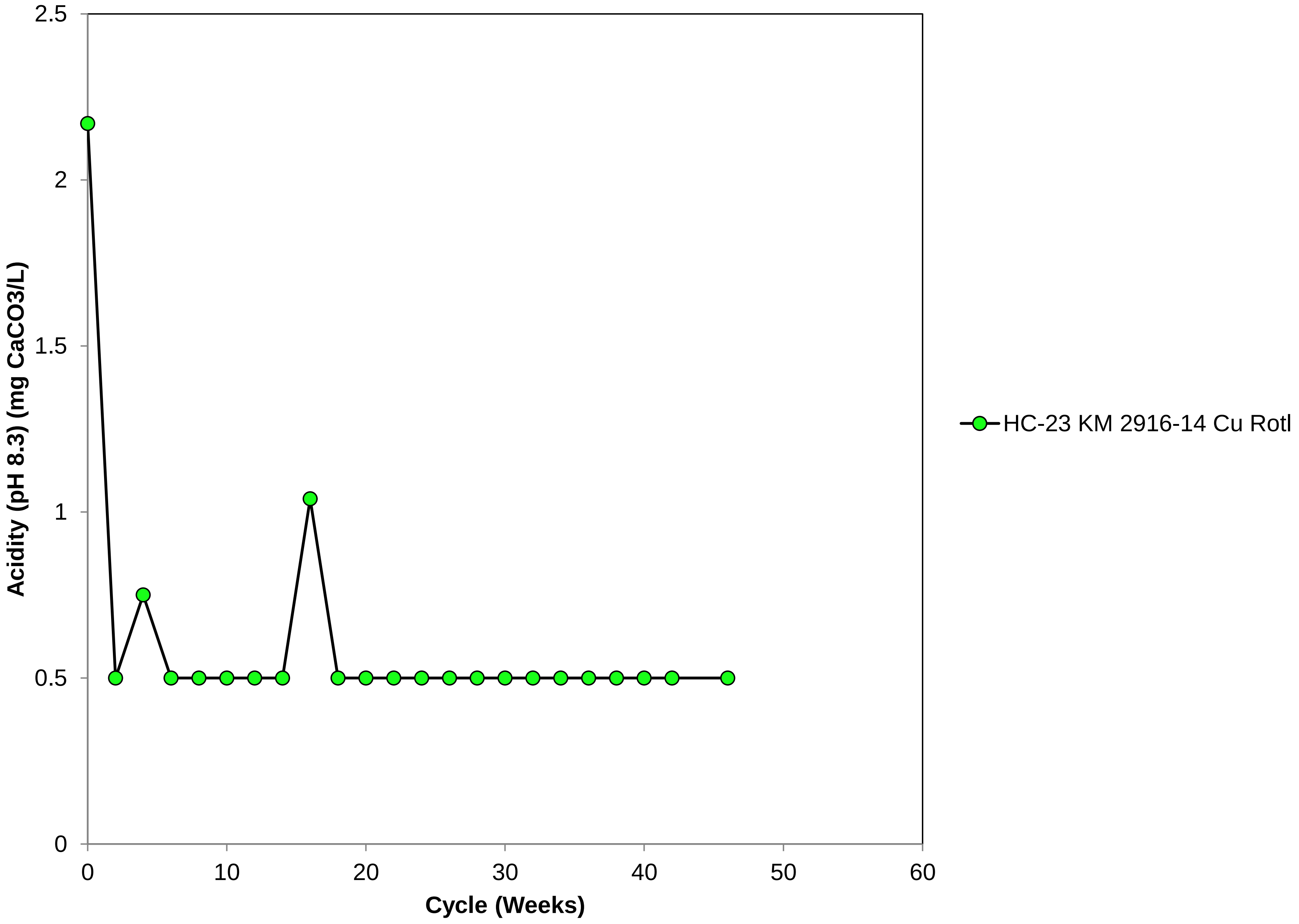

8.5
8.4
8.3
8.2
8.1
8
7.9
7.8
7.7
7.6
pH
0
10
20
30
40
50
60
Cycle Weeks
HC-23 KM 2916-14 Cu Rotl

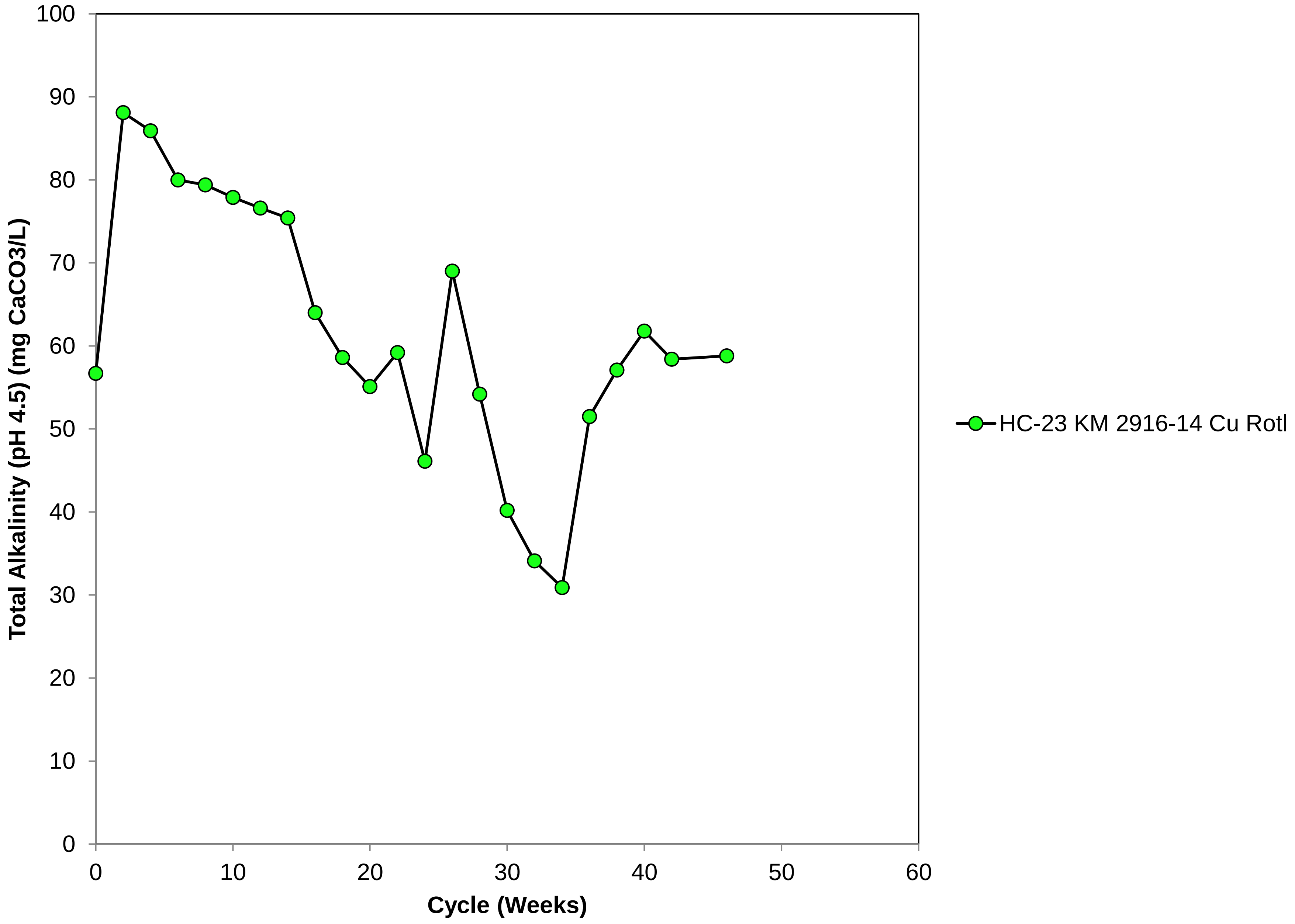

EC (µS/cm)
1400
1200
1000
800
600
400
200
0
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

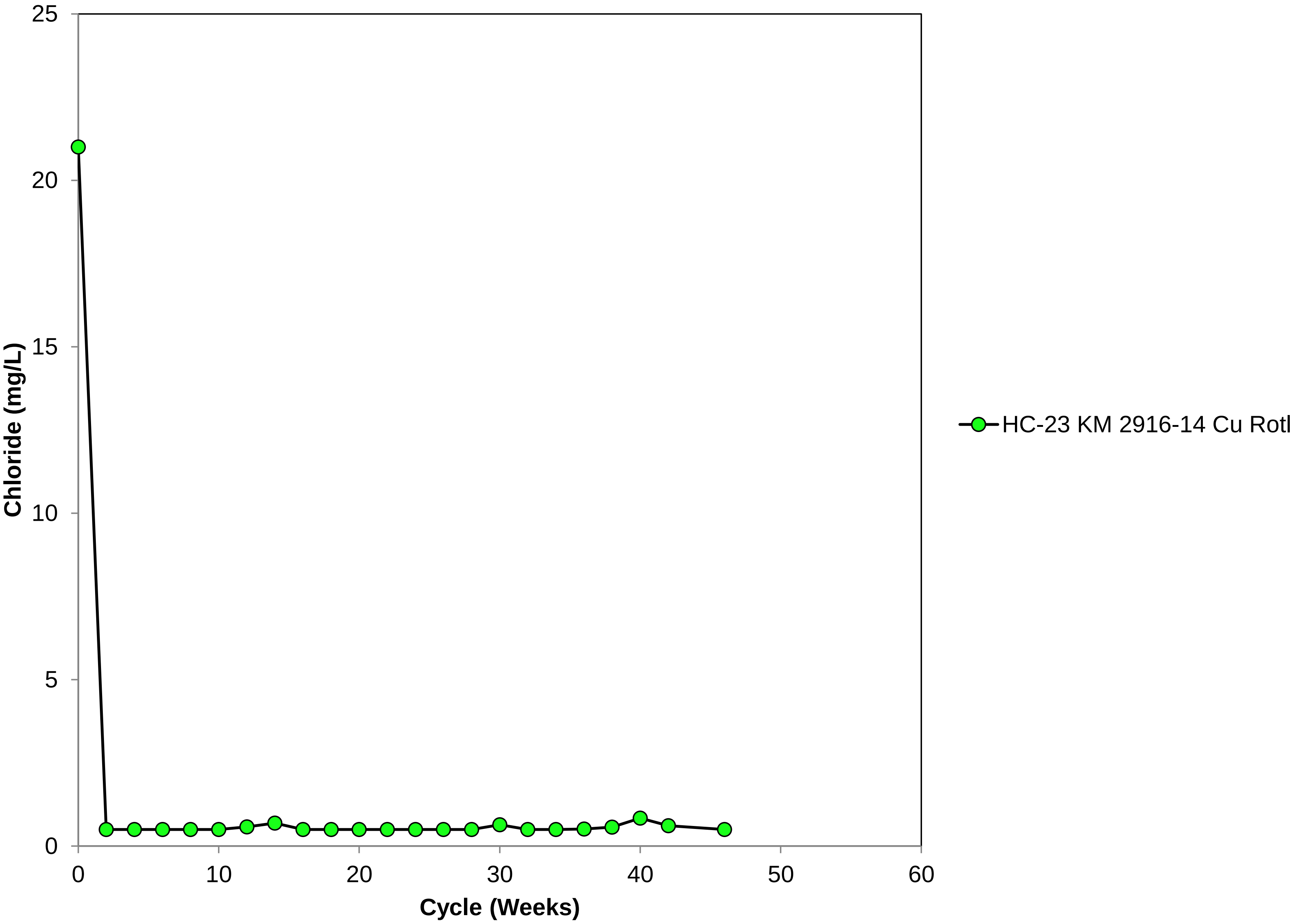

160
140
120
100
80
60
40
20
0
ORP (mV)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

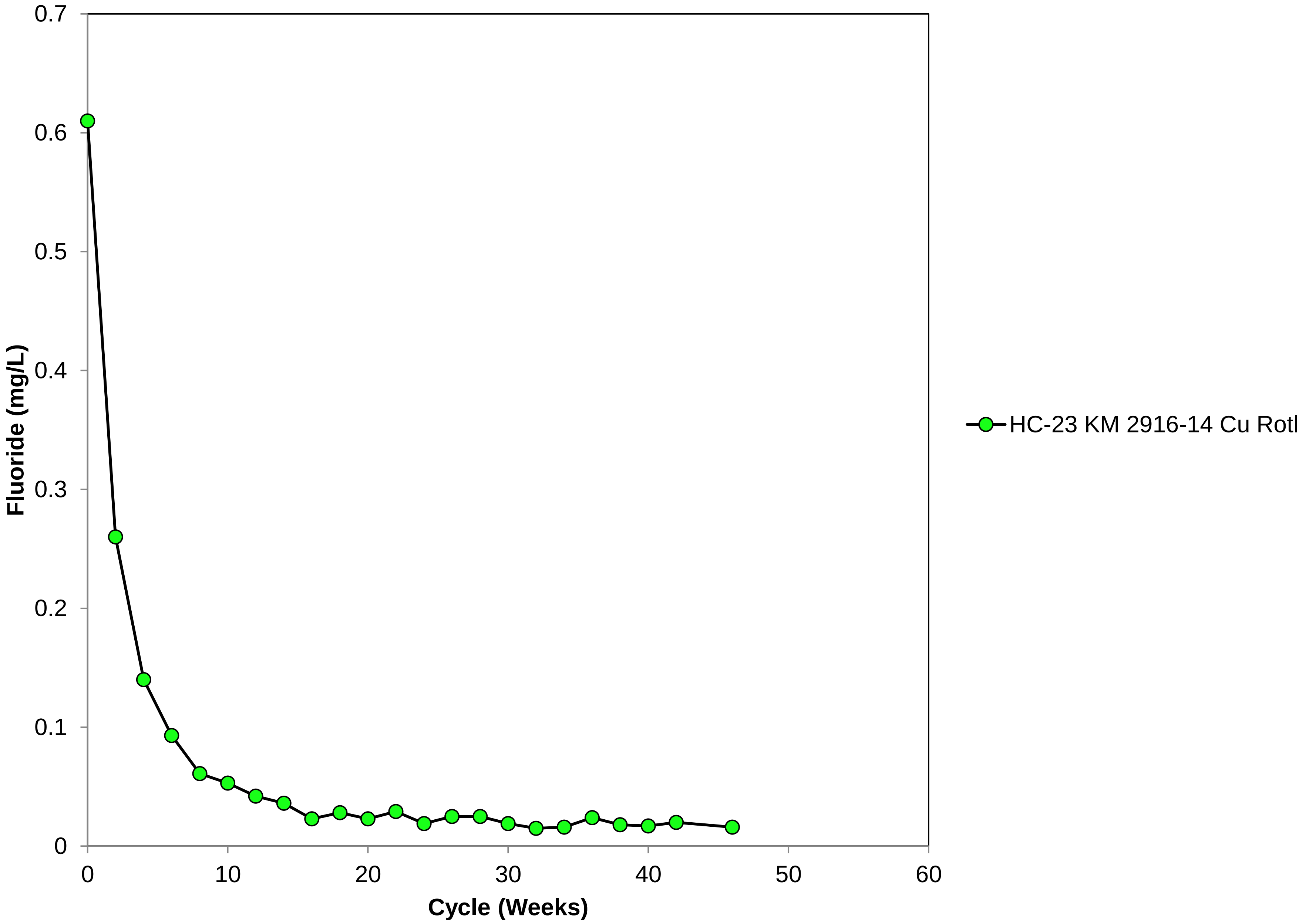

500
450
400
350
300
250
200
150
100
50
0
Sulphate (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

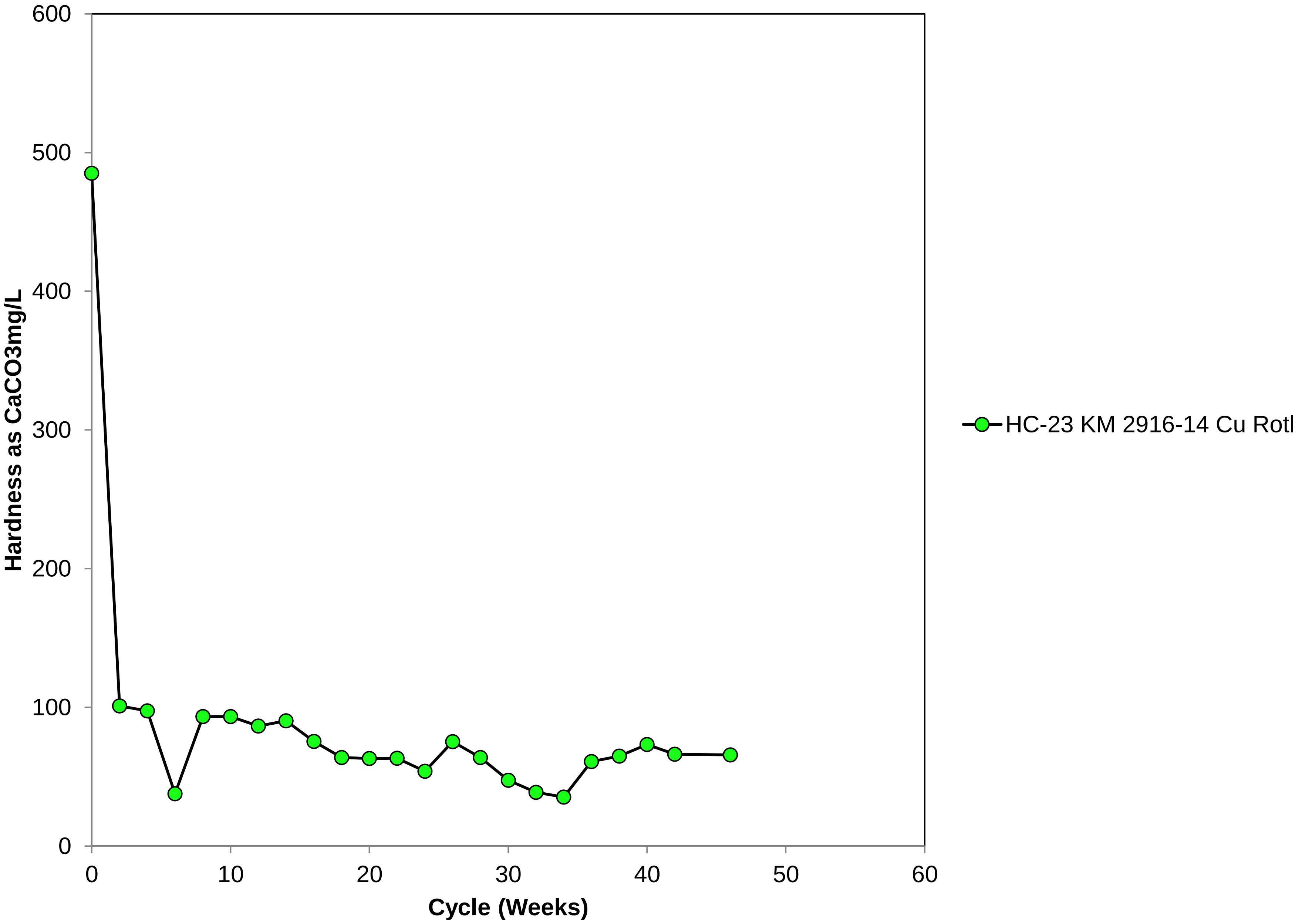

0.6
0.5
0.4
0.3
0.2
0.1
0
Acidity (pH 4.5) (mg CaCO3/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

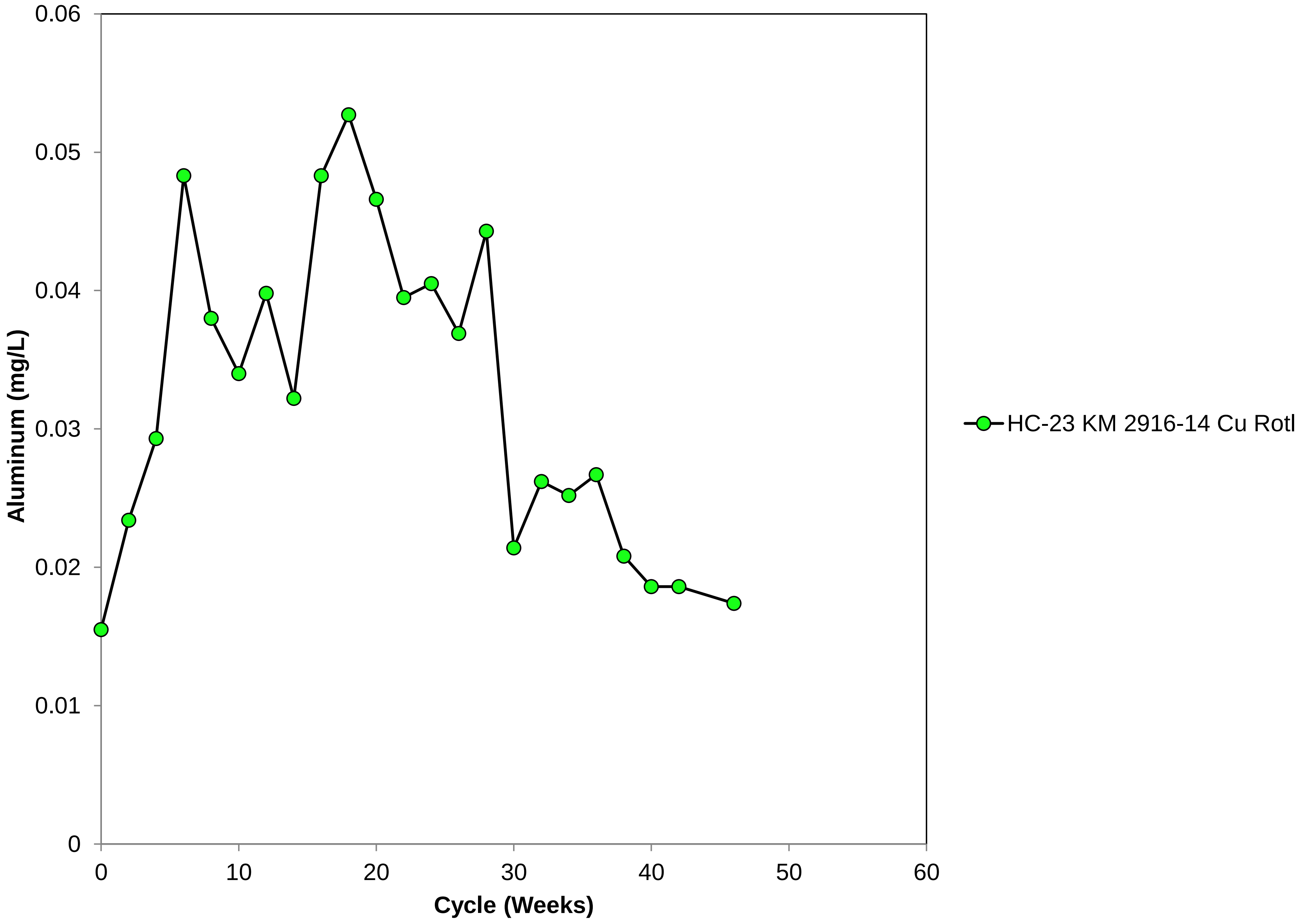

2.5
2
1.5
1
0.5
0
0
10
20
30
40
50
60
Acidity (pH 8.3) (mg CaCO3/L)
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

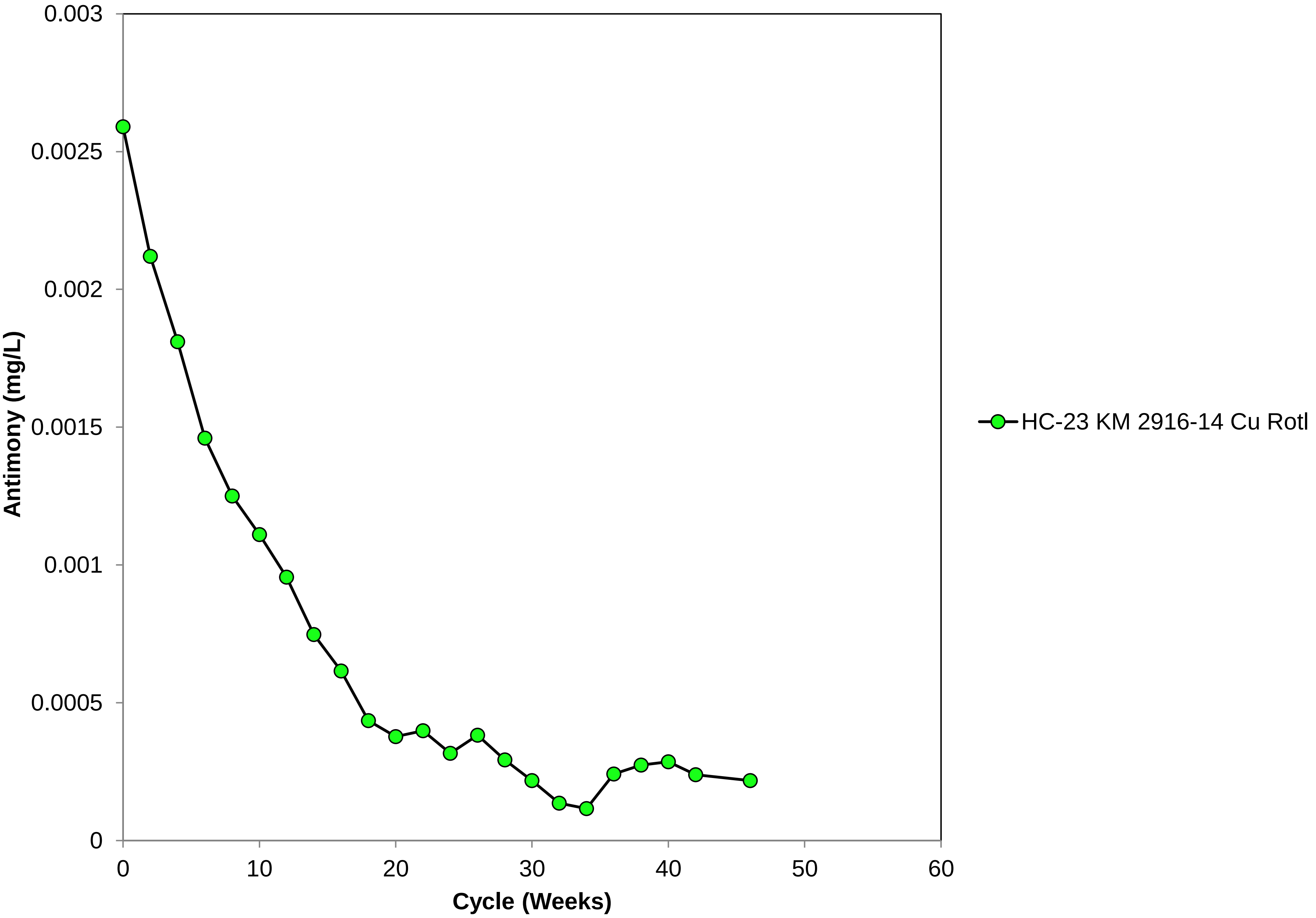

100
90
80
70
60
50
40
30
20
10
0
Total Alkalinity (pH 4.5) (mg CaCO3/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

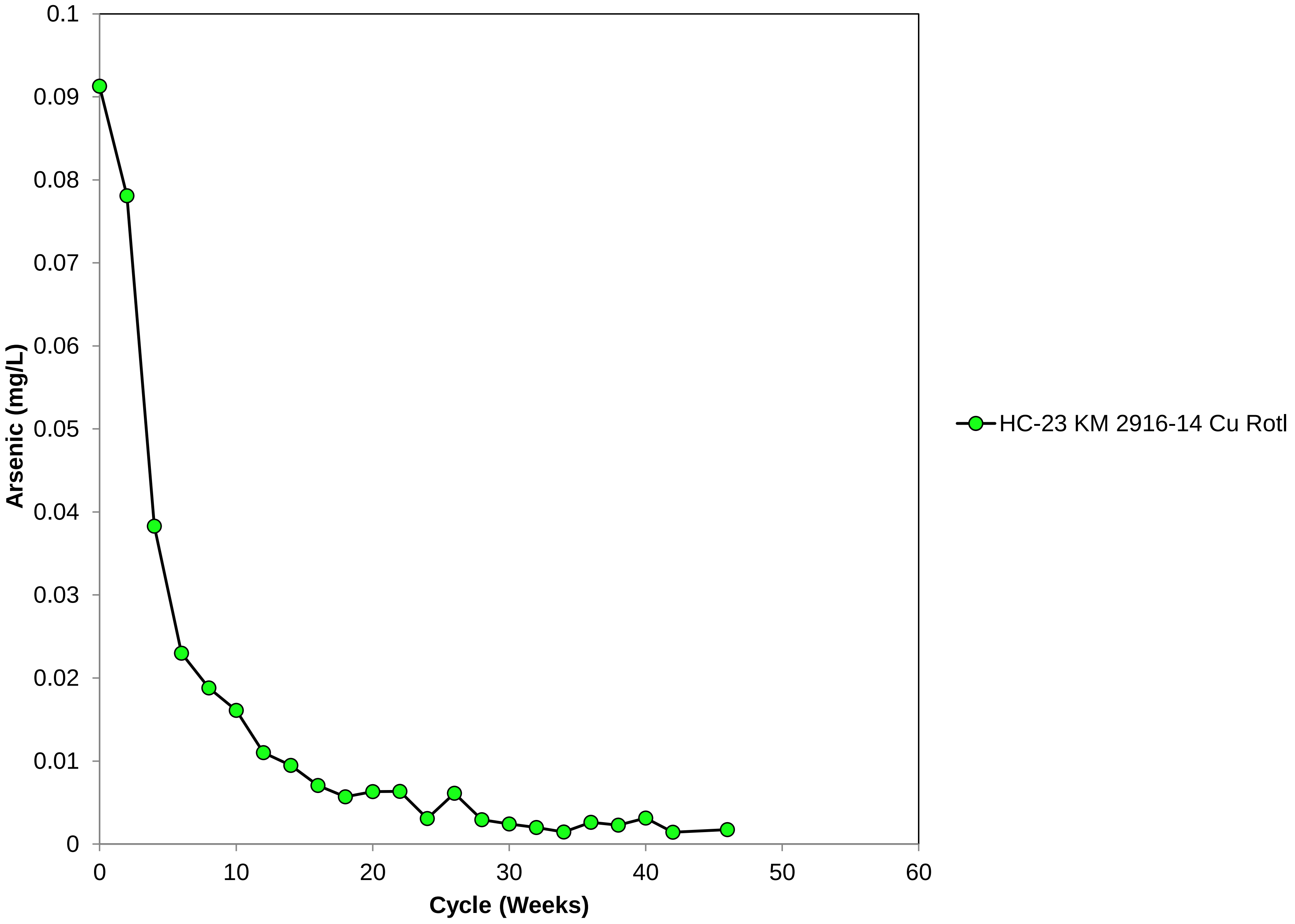

25
20
15
10
5
0
Chloride (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

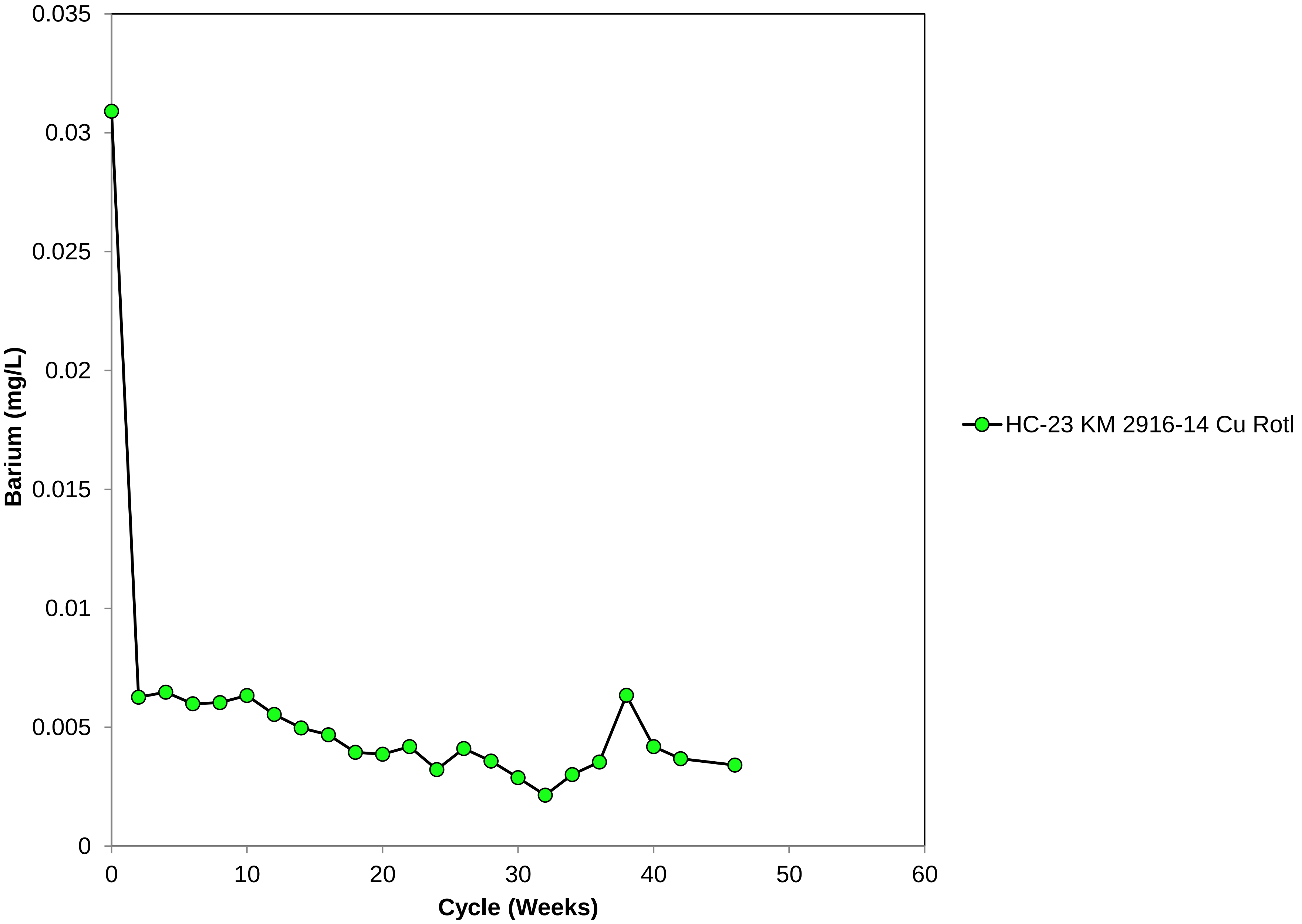

0.7
0.6
0.5
0.4
0.3
0.2
0.1
0
Fluoride (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

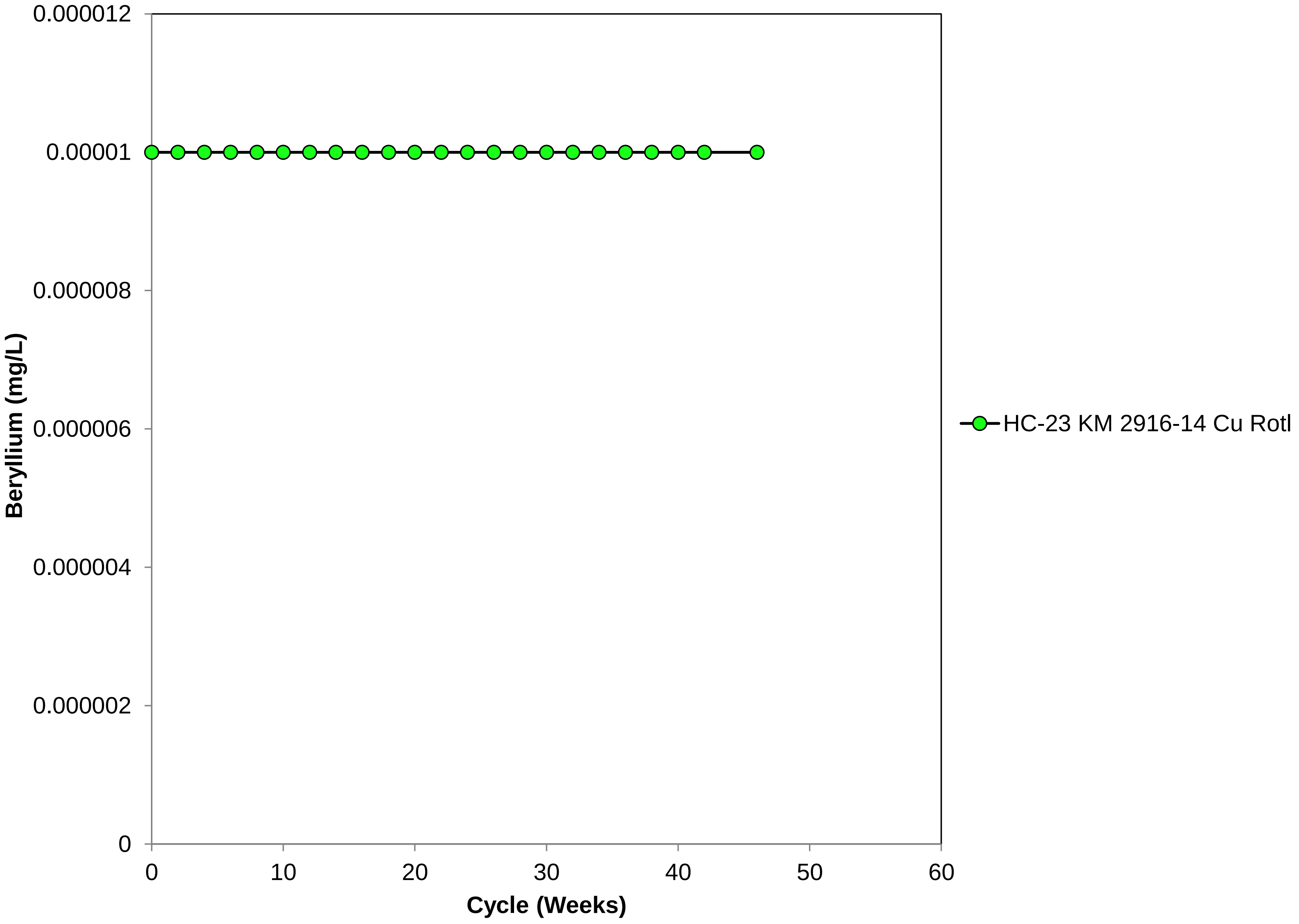

600
500
400
300
200
100
0
Hardness as CaCO3mg/L
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

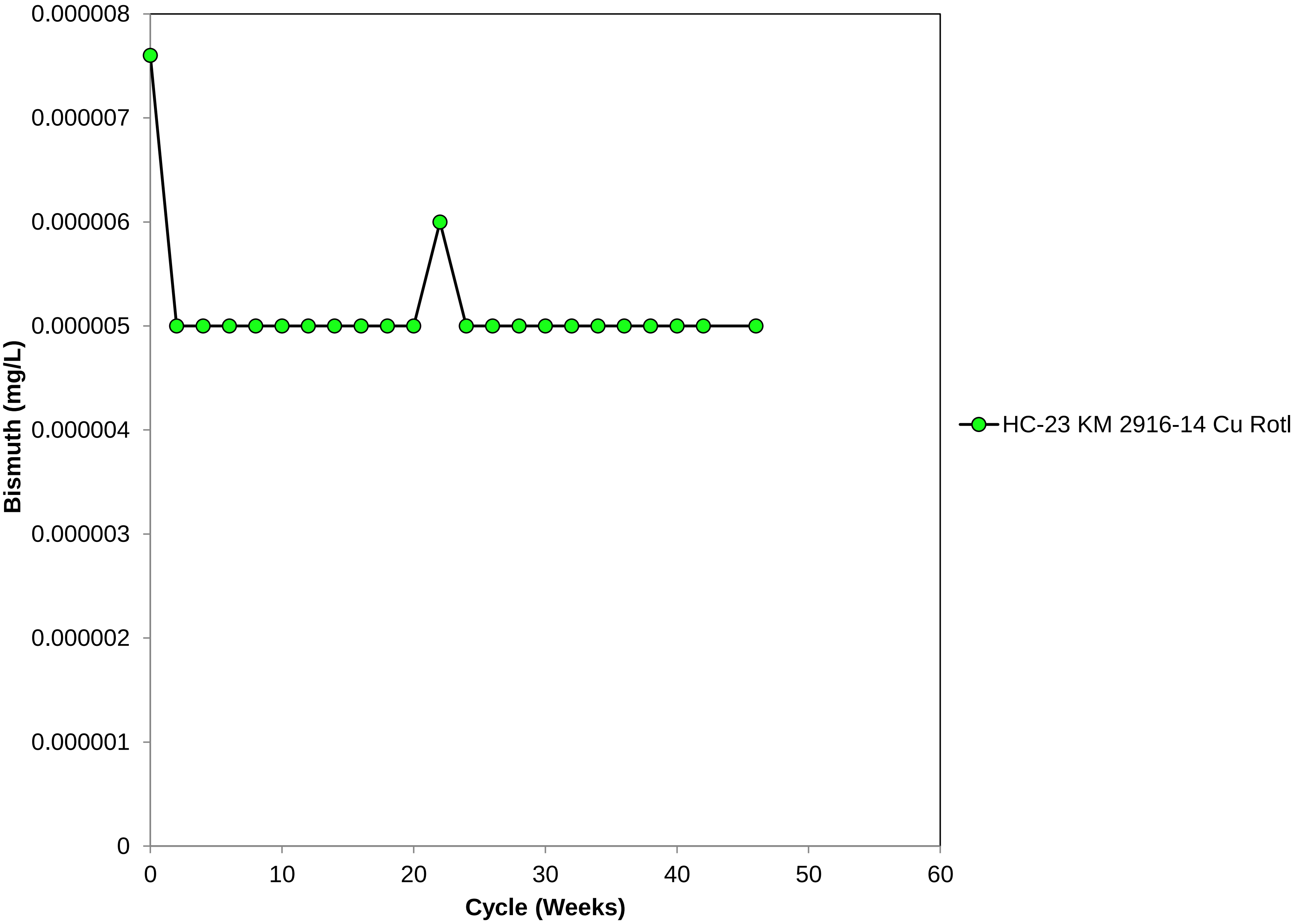

0.06
0.05
0.04
0.03
0.02
0.01
0
Aluminum (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

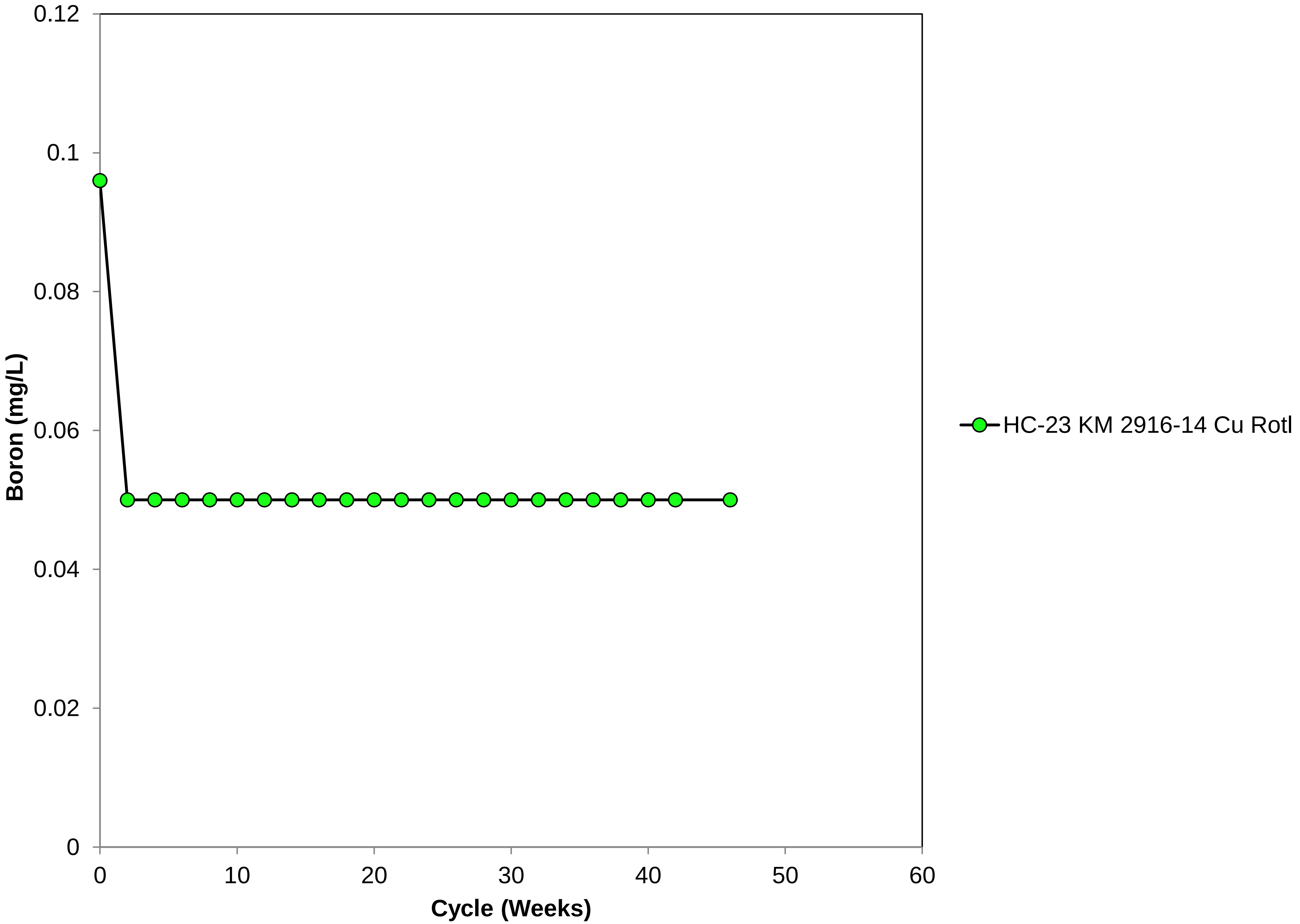

0.003
0.0025
0.002
0.0015
0.001
0.0005
0
Antimony (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

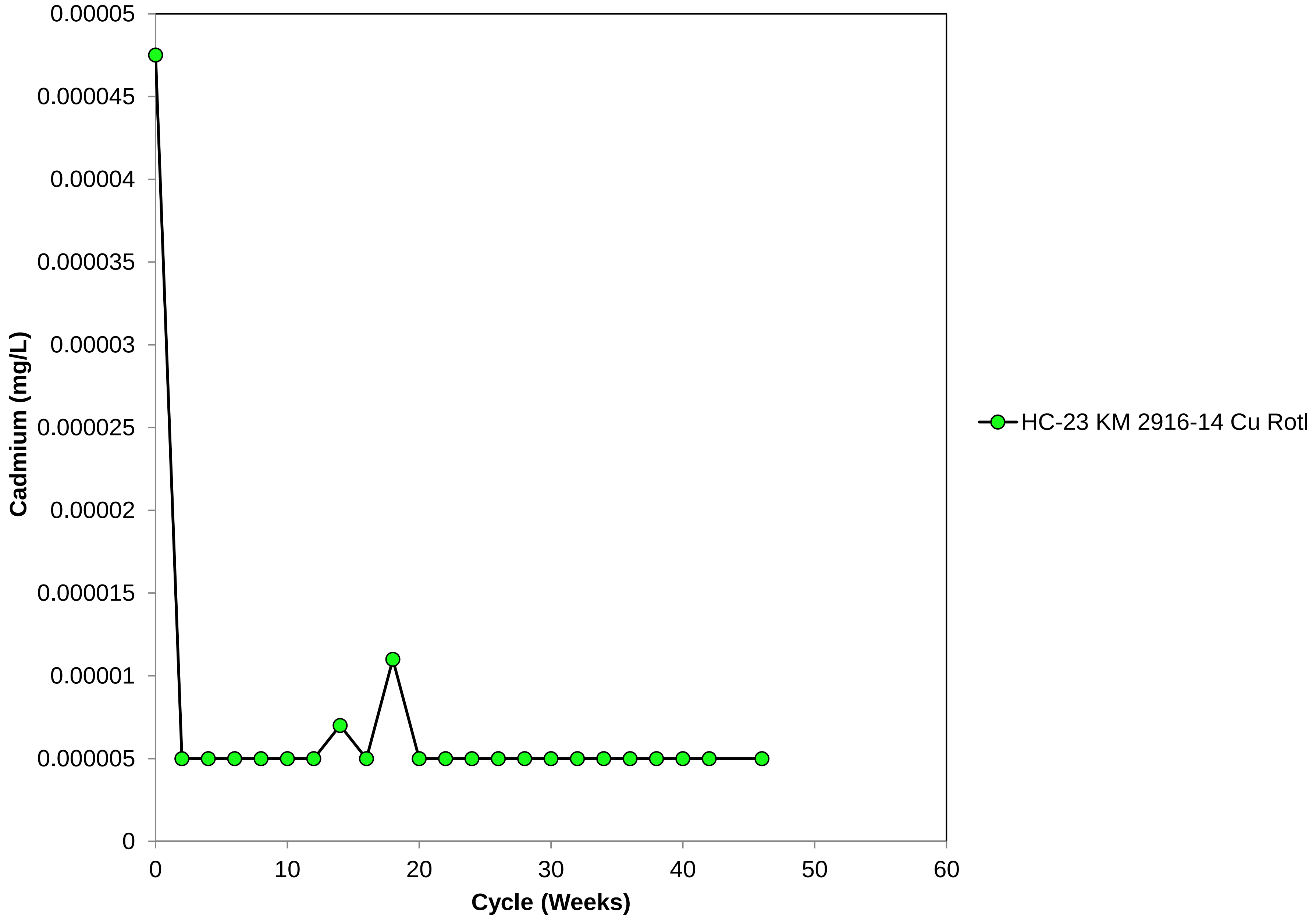

0.1
0.09
0.08
0.07
0.06
0.05
0.04
0.03
0.02
0.01
0
Arsenic (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl


0.035
0.03
0.025
0.02
0.015
0.01
0.005
0
Barium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl


0.000012
0.00001
0.000008
0.000006
0.000004
0.000002
0
Beryllium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl


0.000008
0.000007
0.000006
0.000005
0.000004
0.000003
0.000002
0.000001
0
Bismuth (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl


0.12
0.1
0.08
0.06
0.04
0.02
0
Boron (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl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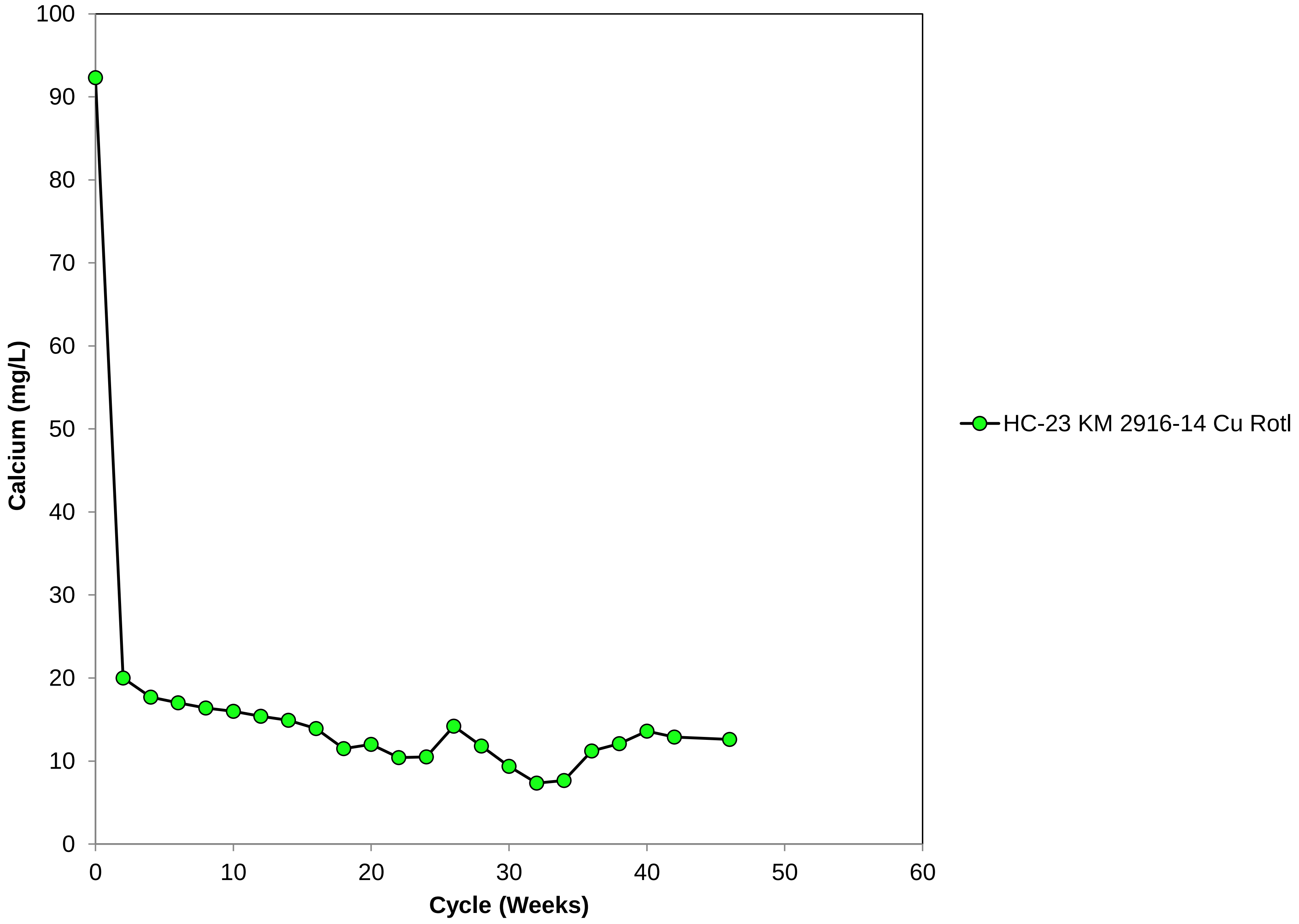
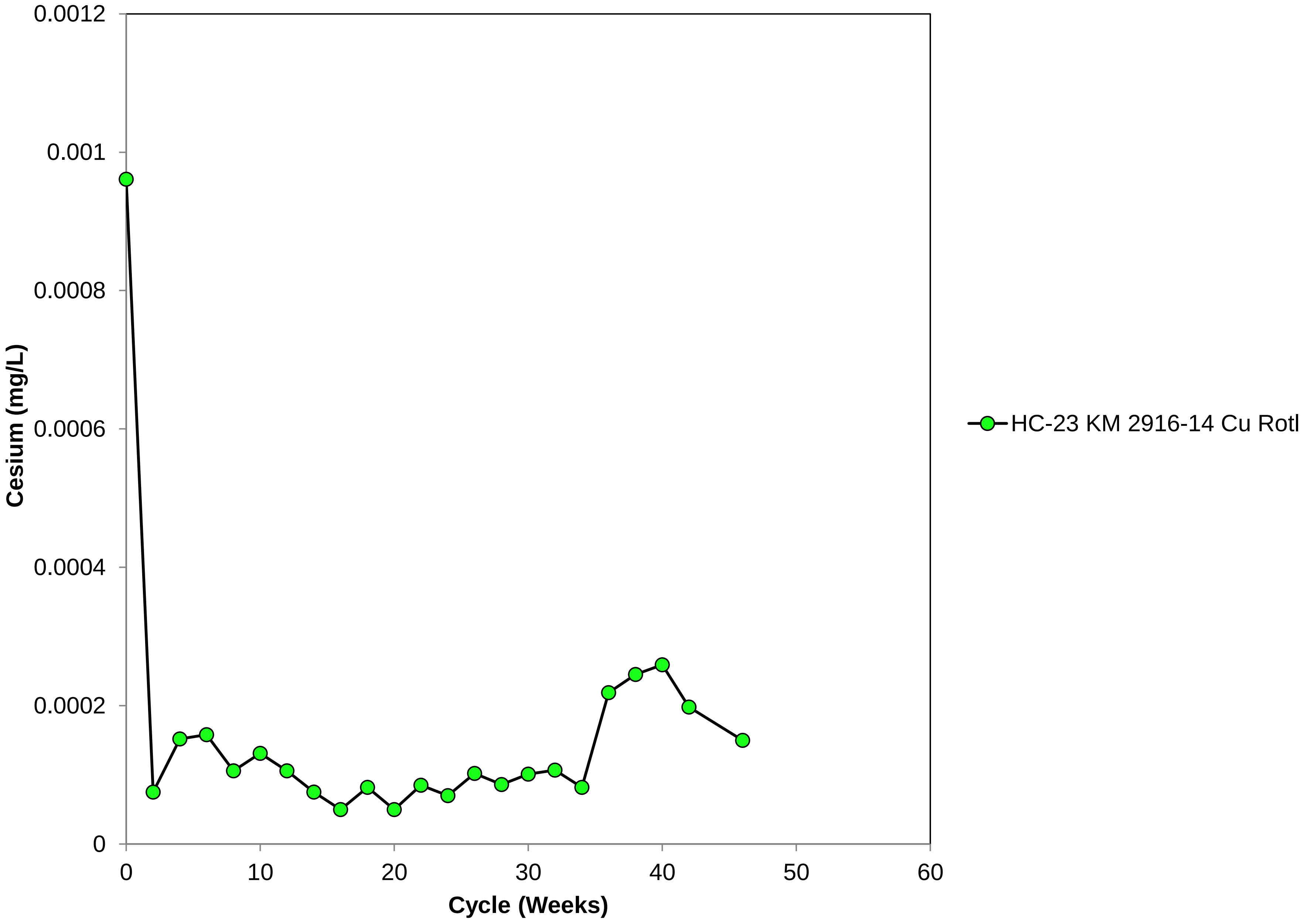
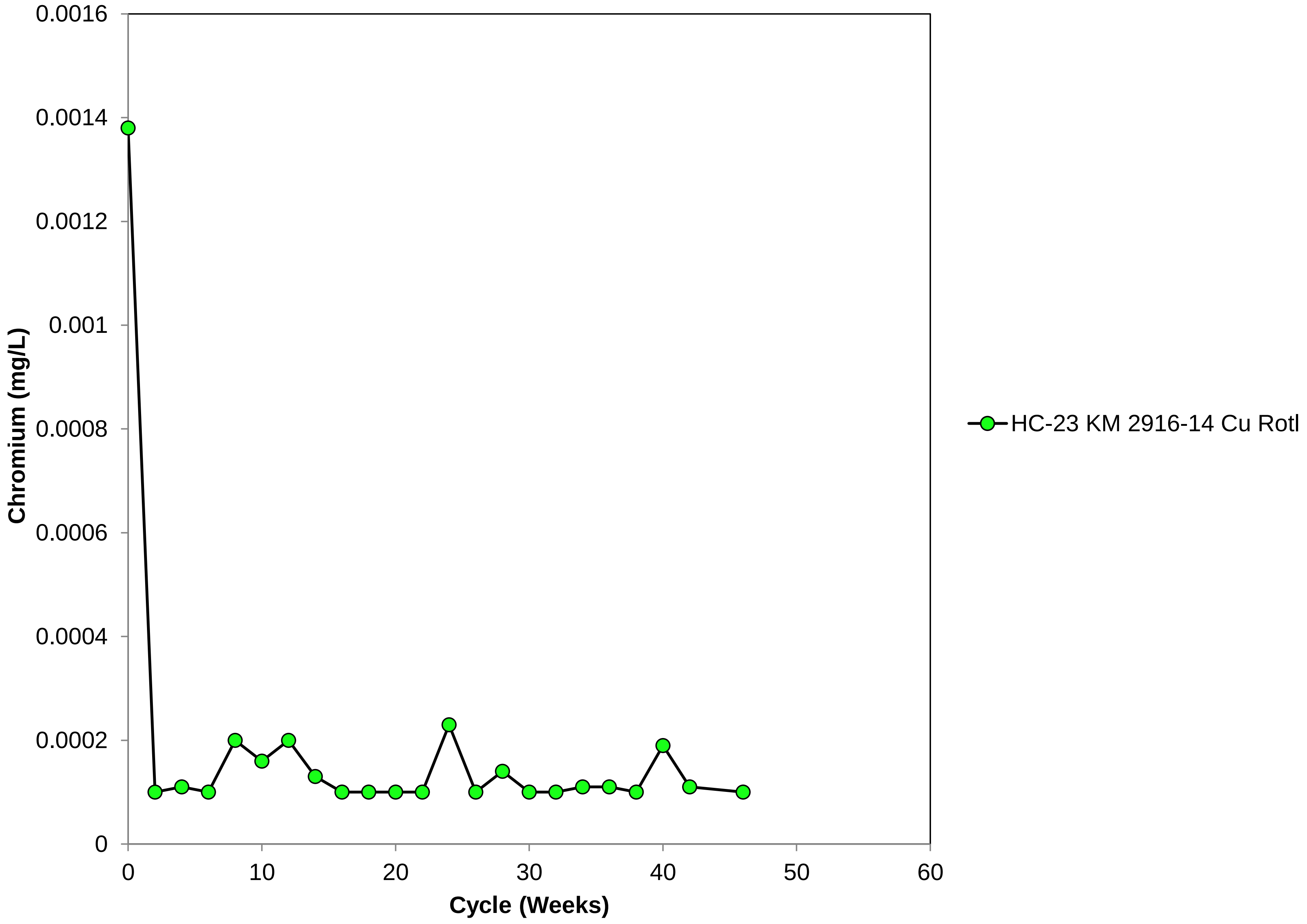
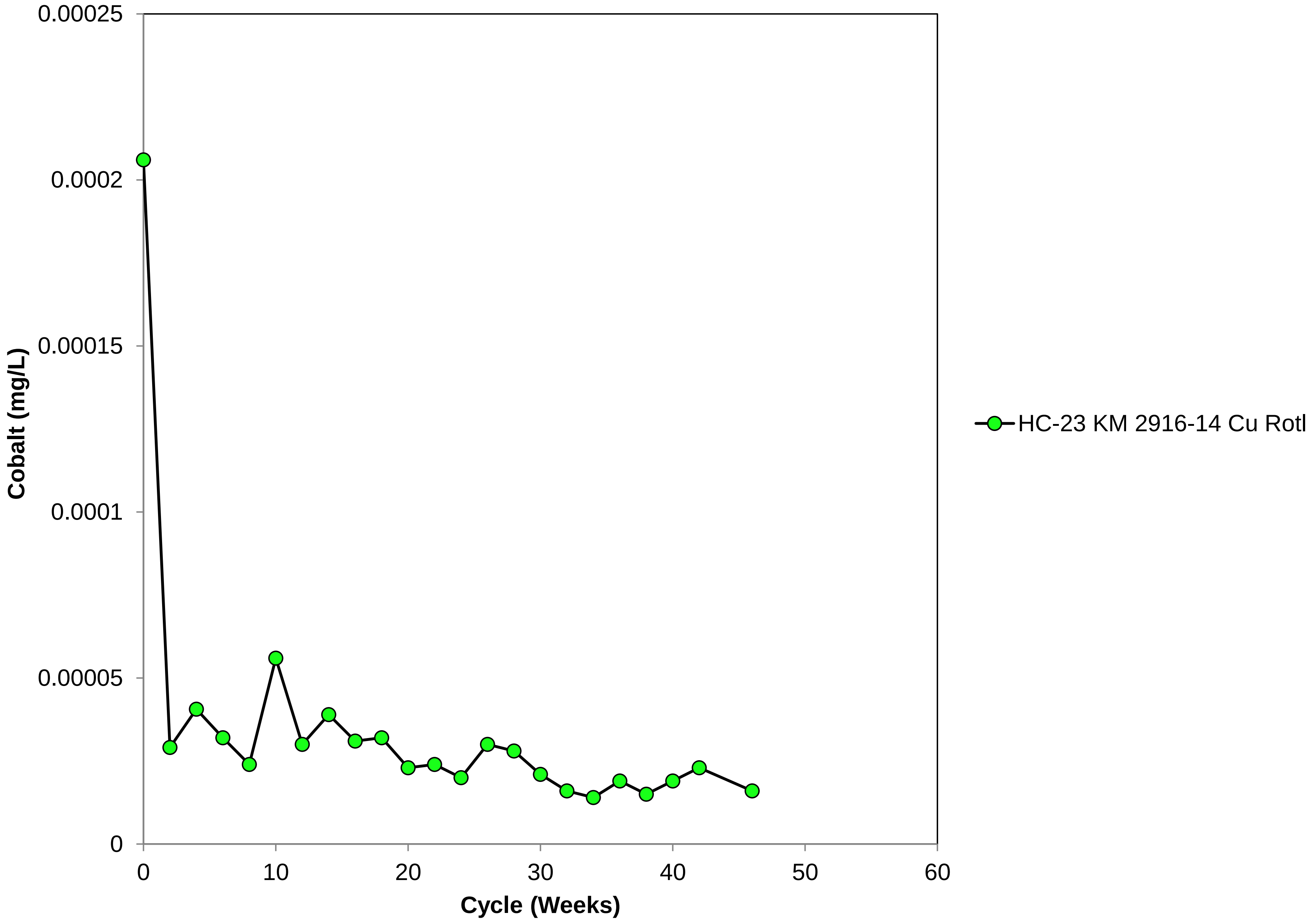
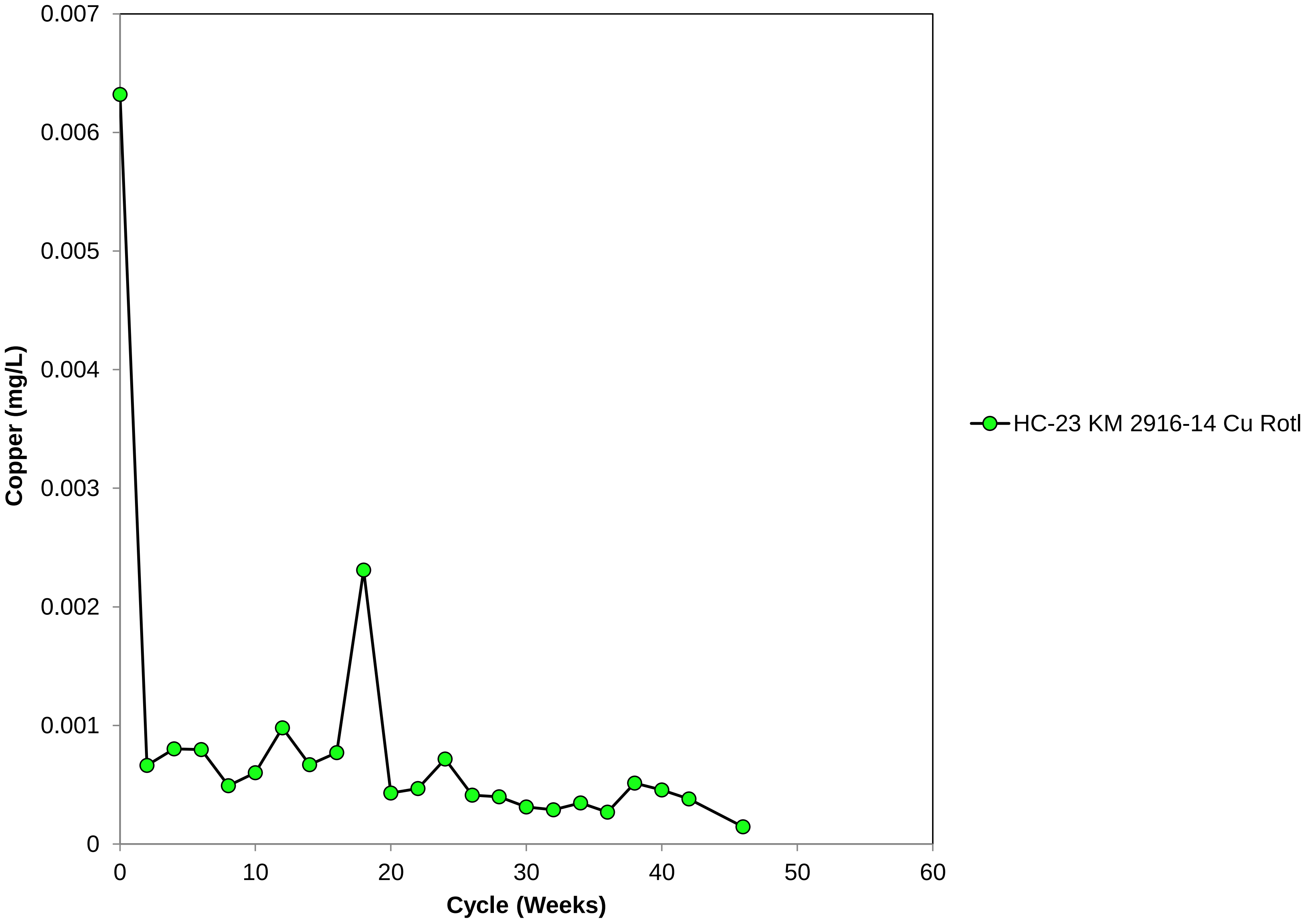
0.000045
0.00004
0.000035
0.00003
0.000025
0.00002
0.000015
0.00001
0.000005
0
Cadmium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

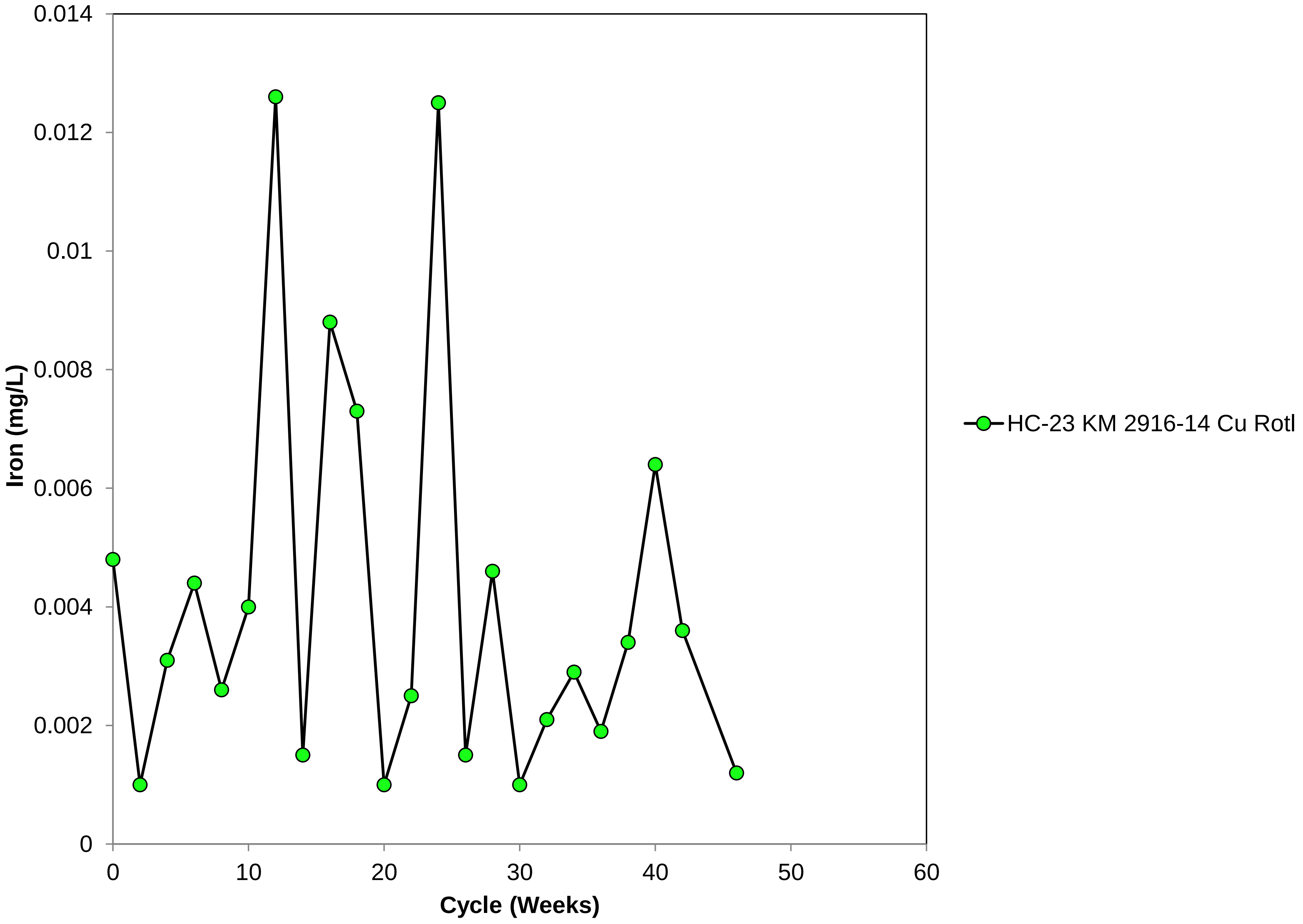

100
90
80
70
60
50
40
30
20
10
0
Calcium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

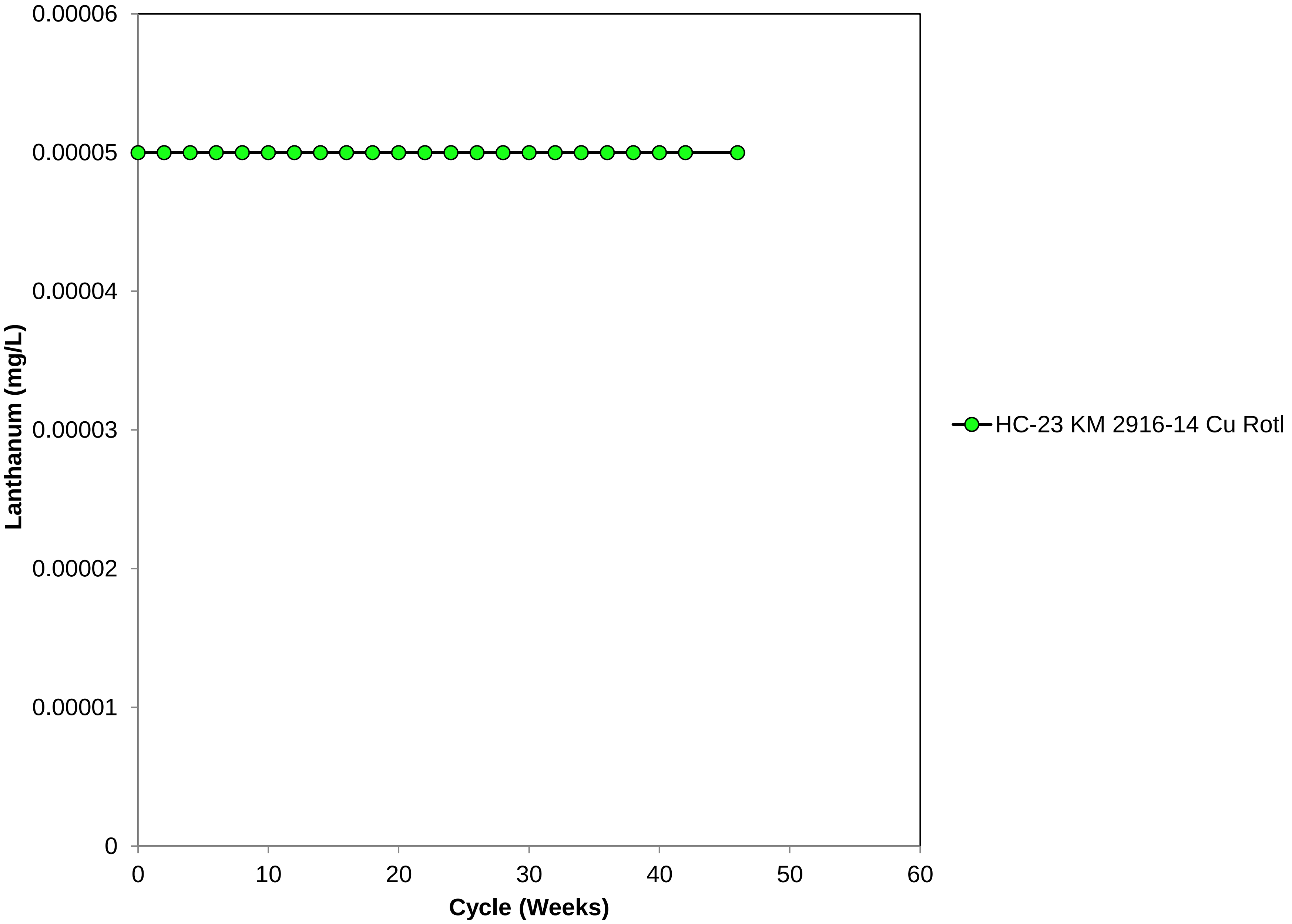

0.0012
0.001
0.0008
0.0006
0.0004
0.0002
0
Cesium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

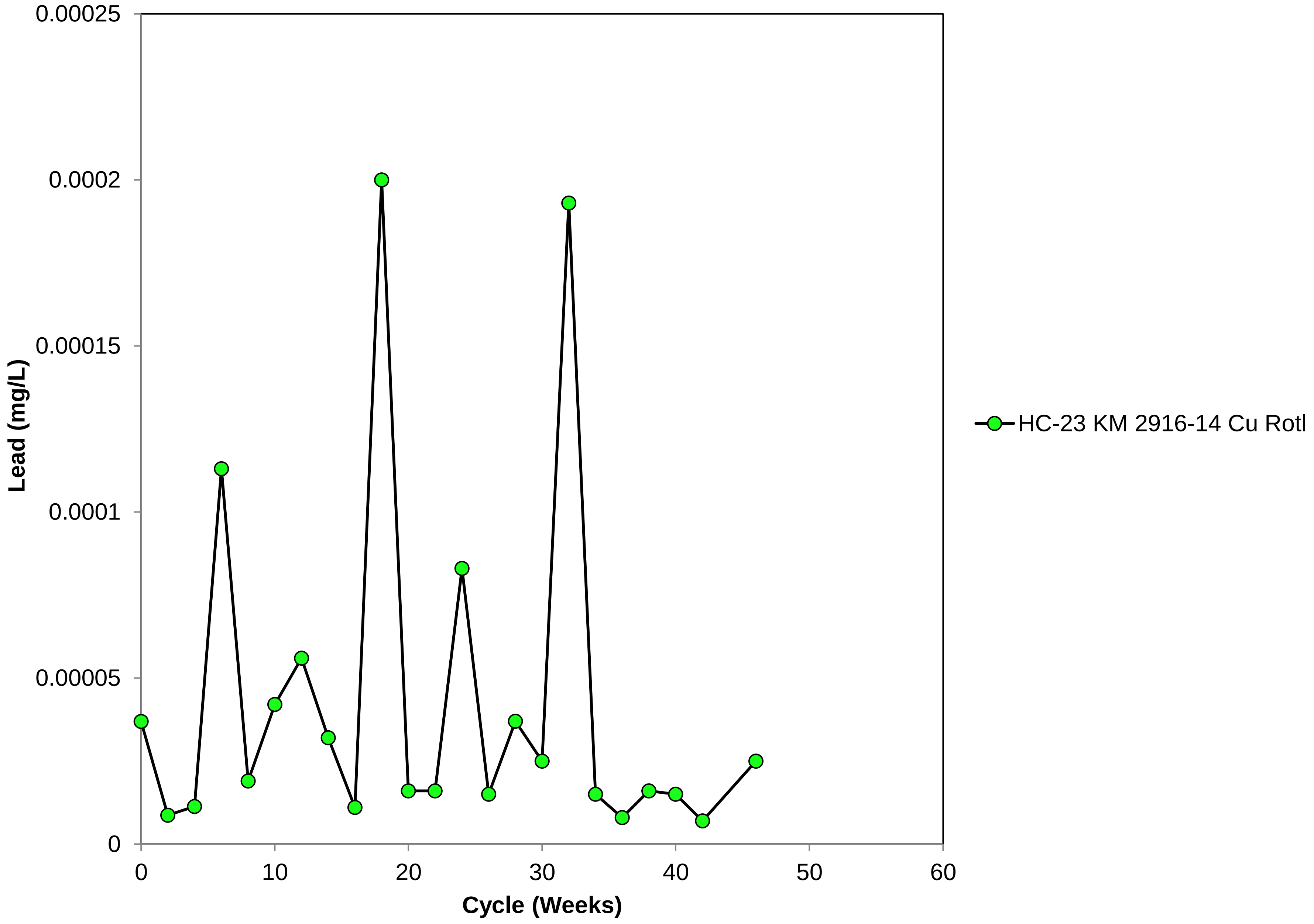

0.0016
0.0014
0.0012
0.001
0.0008
0.0006
0.0004
0.0002
0
Chromium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl


0.00025
0.0002
0.00015
0.0001
0.00005
0
Cobalt (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl


0.007
0.006
0.005
0.004
0.003
0.002
0.001
0
Copper (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl


0.014
0.012
0.01
0.008
0.006
0.004
0.002
0
Iron (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl


0.00006
0.00005
0.00004
0.00003
0.00002
0.00001
0
Lanthanum (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl


0.00025
0.0002
0.00015
0.0001
0.00005
0
Lead (mg/L)
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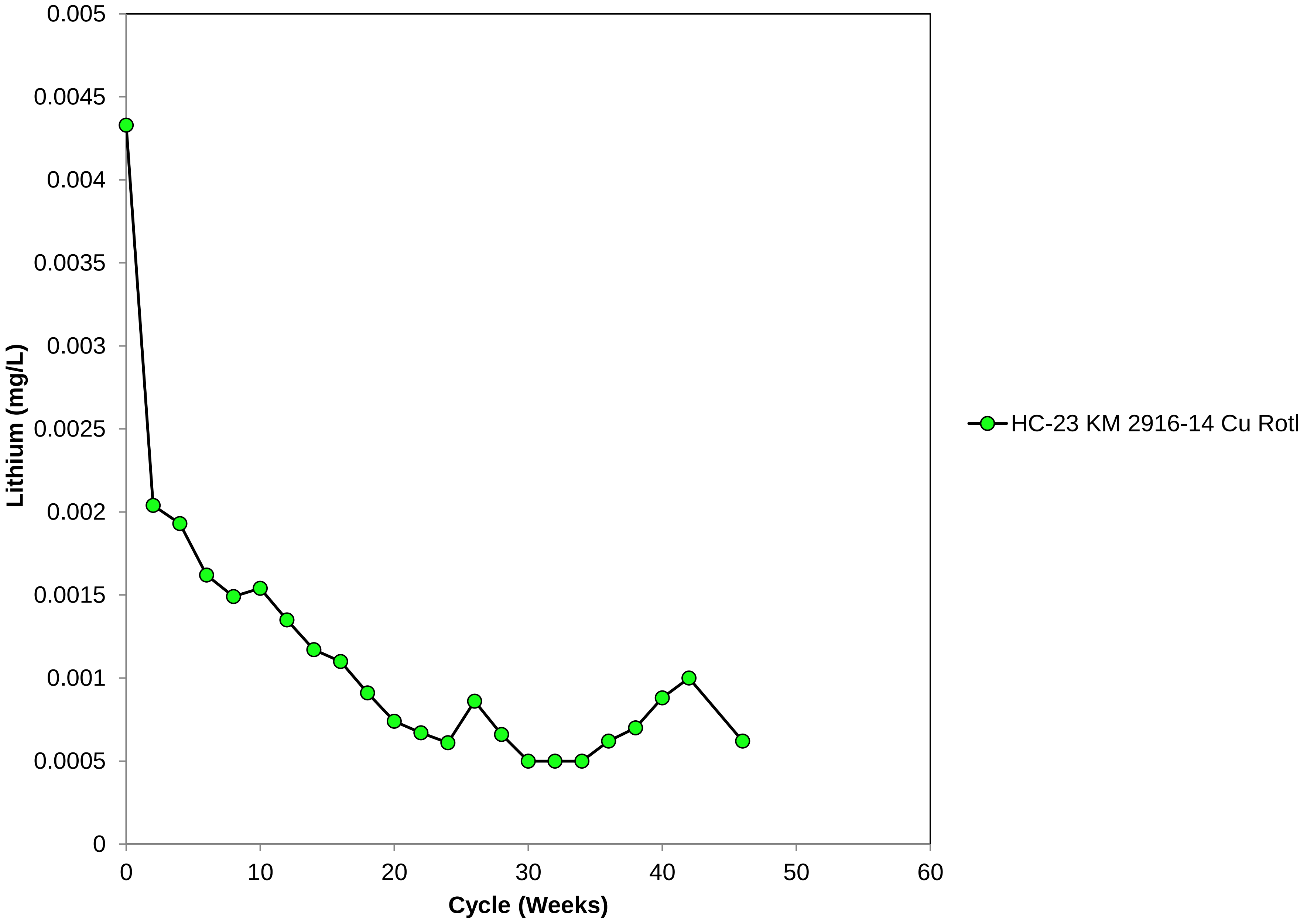
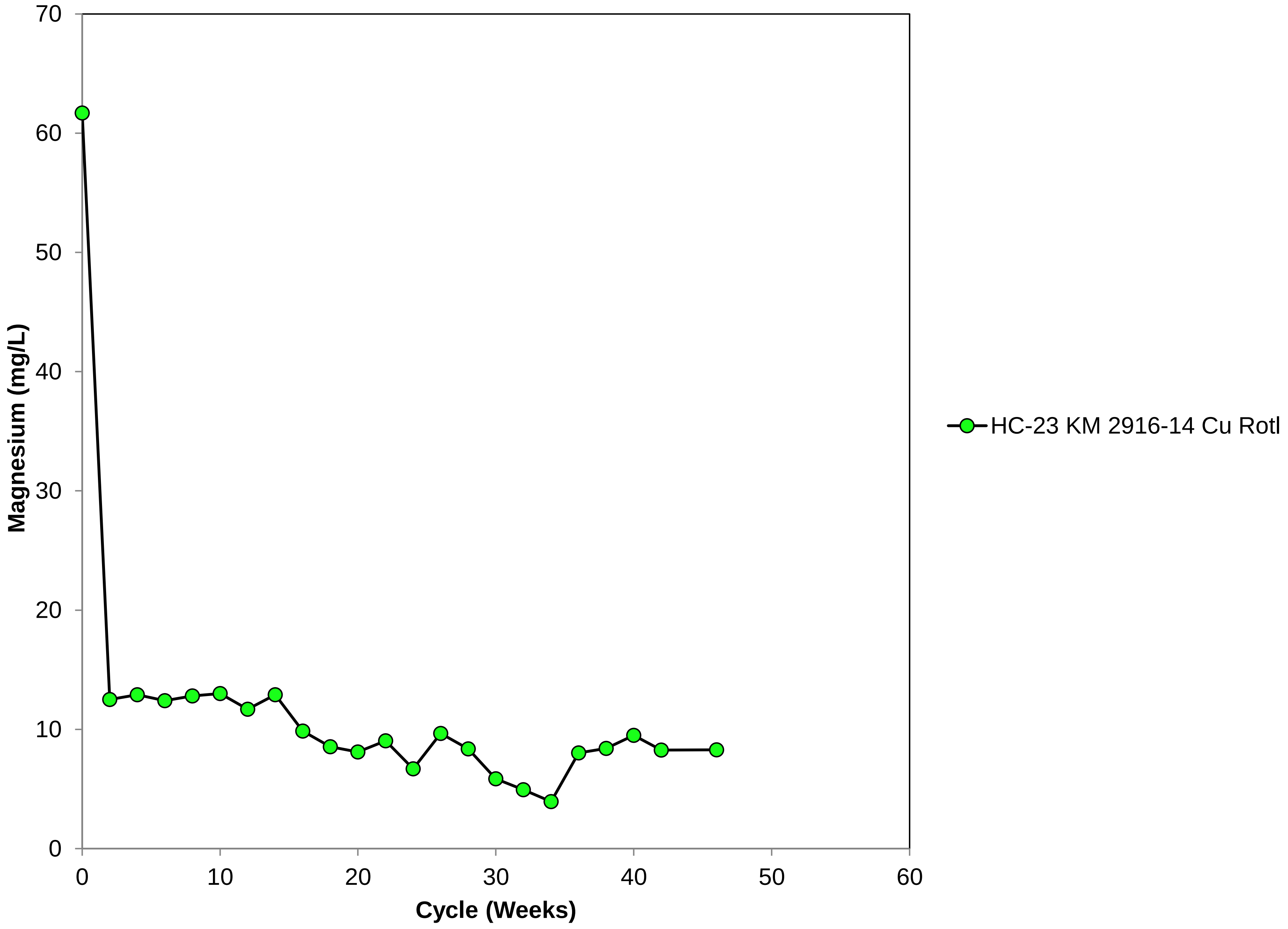
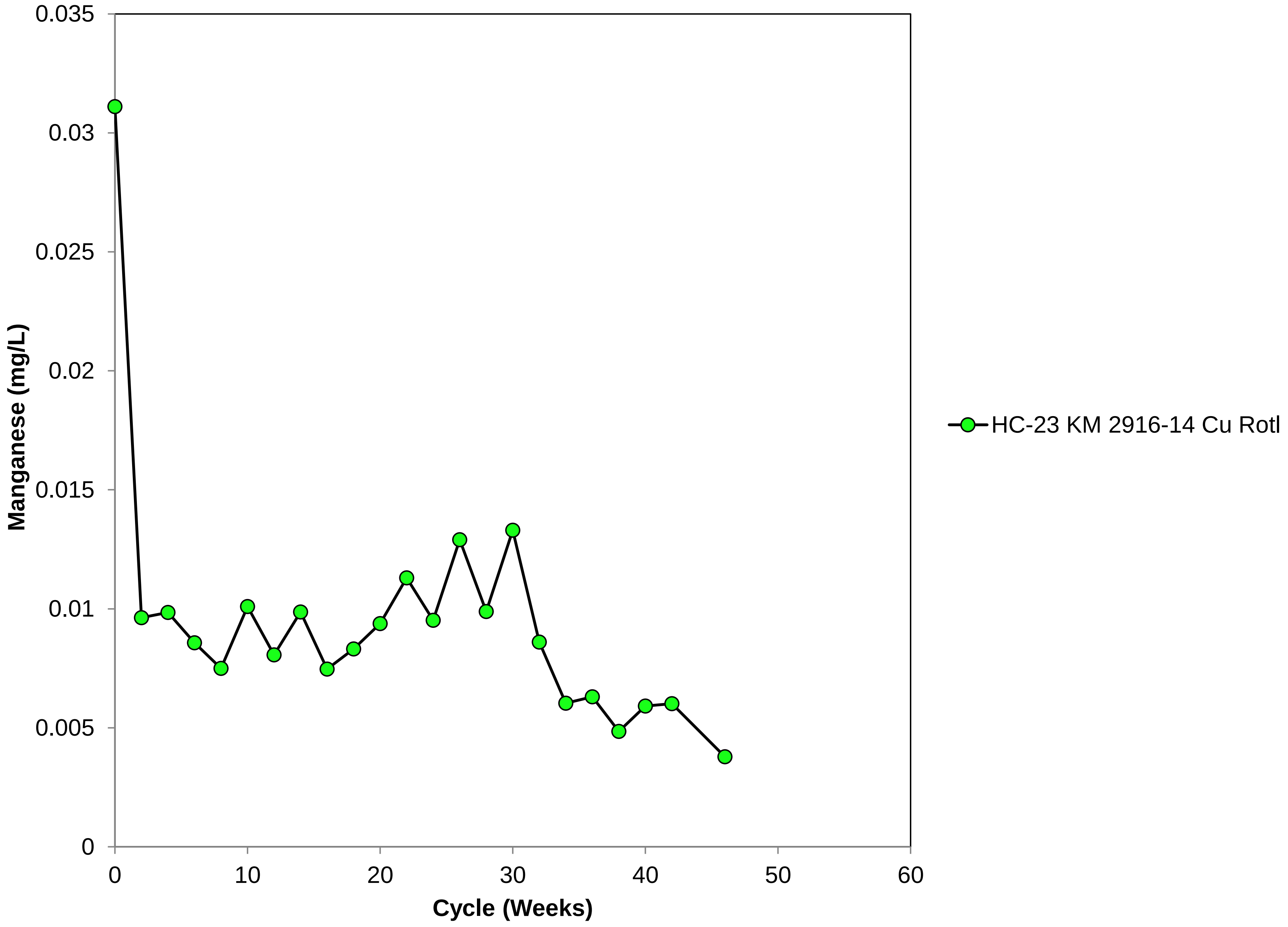
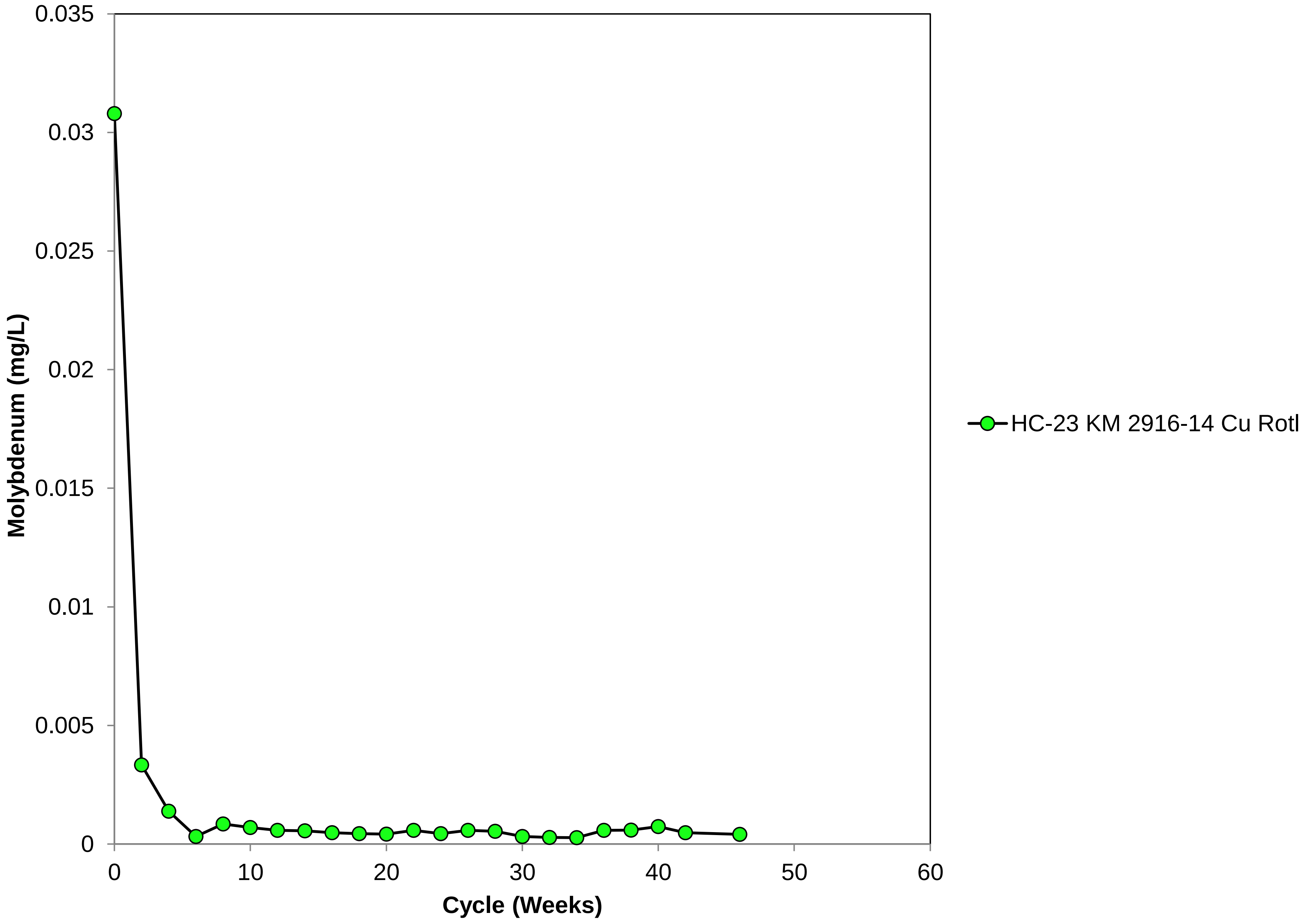
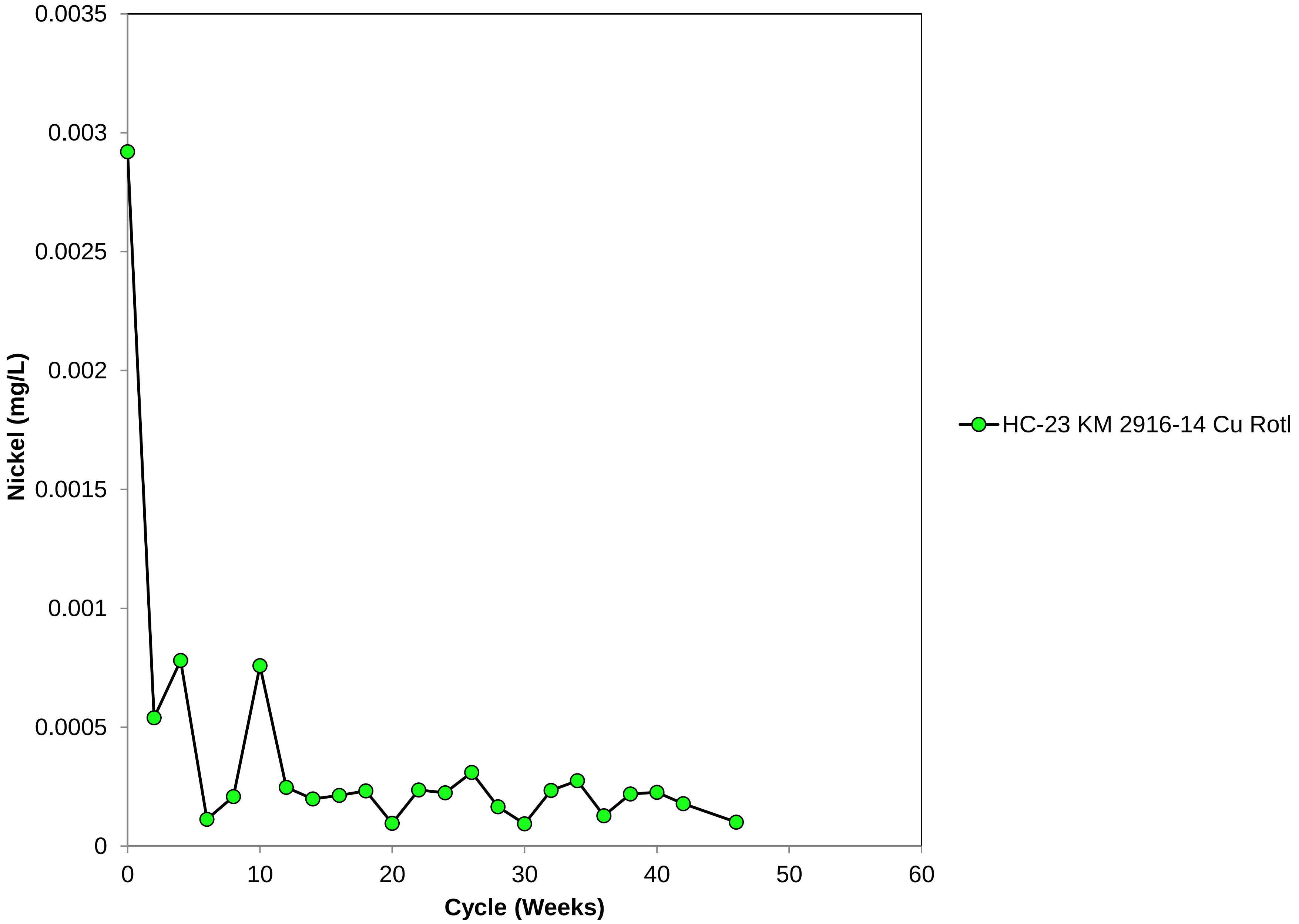
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

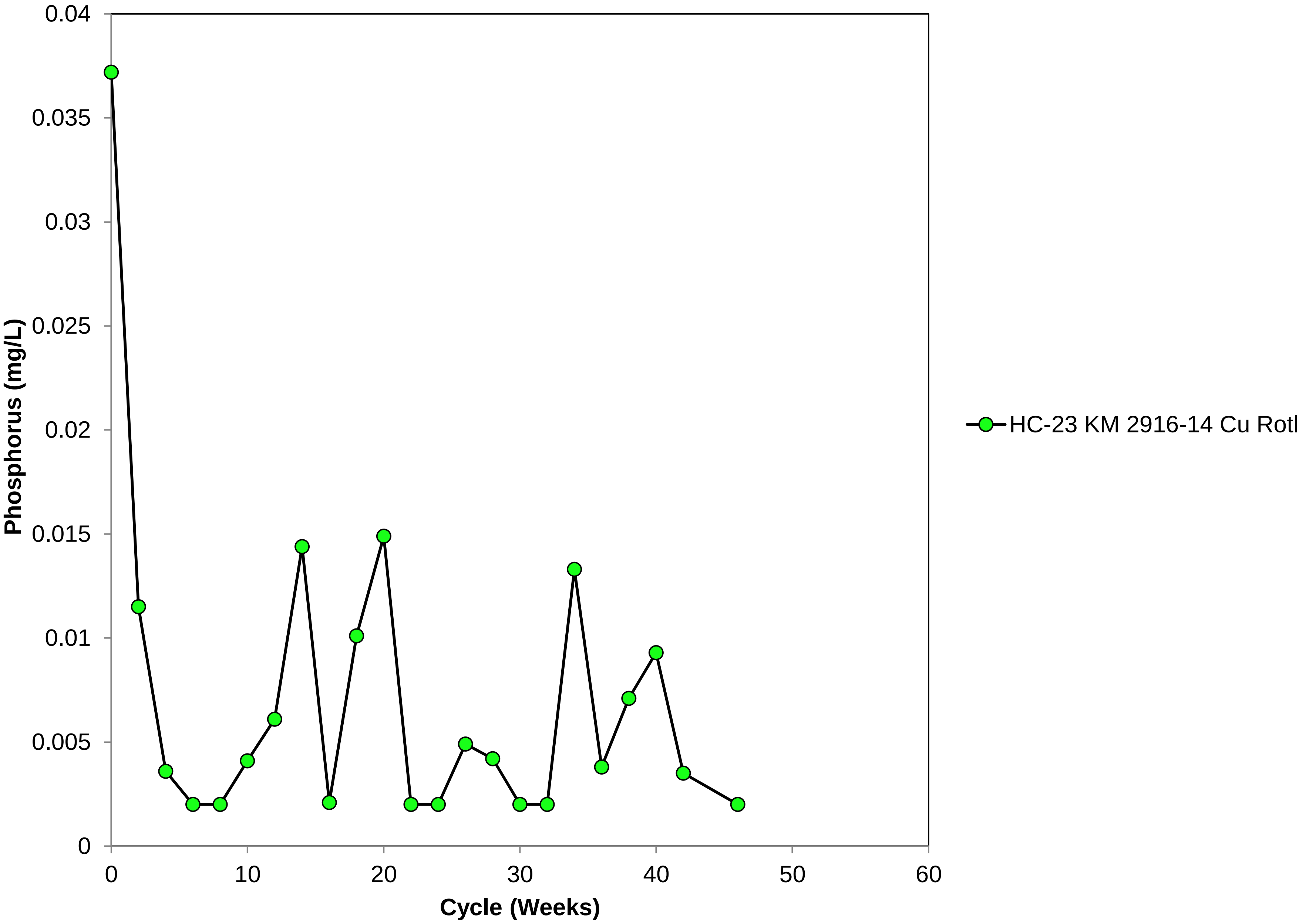

0.005
0.0045
0.004
0.0035
0.003
0.0025
0.002
0.0015
0.001
0.0005
0
Lithium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

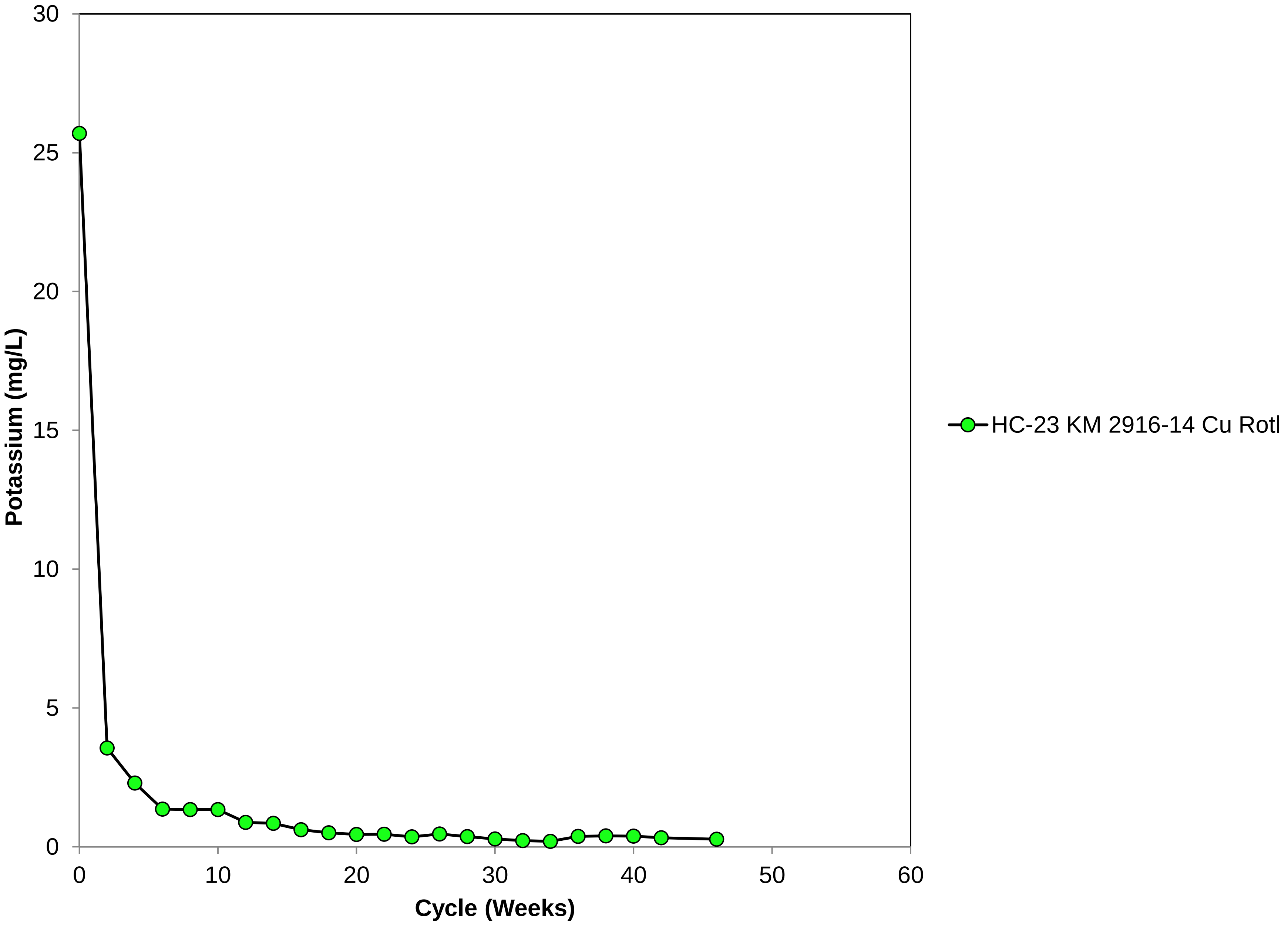

70
60
50
40
30
20
10
0
Magnesium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

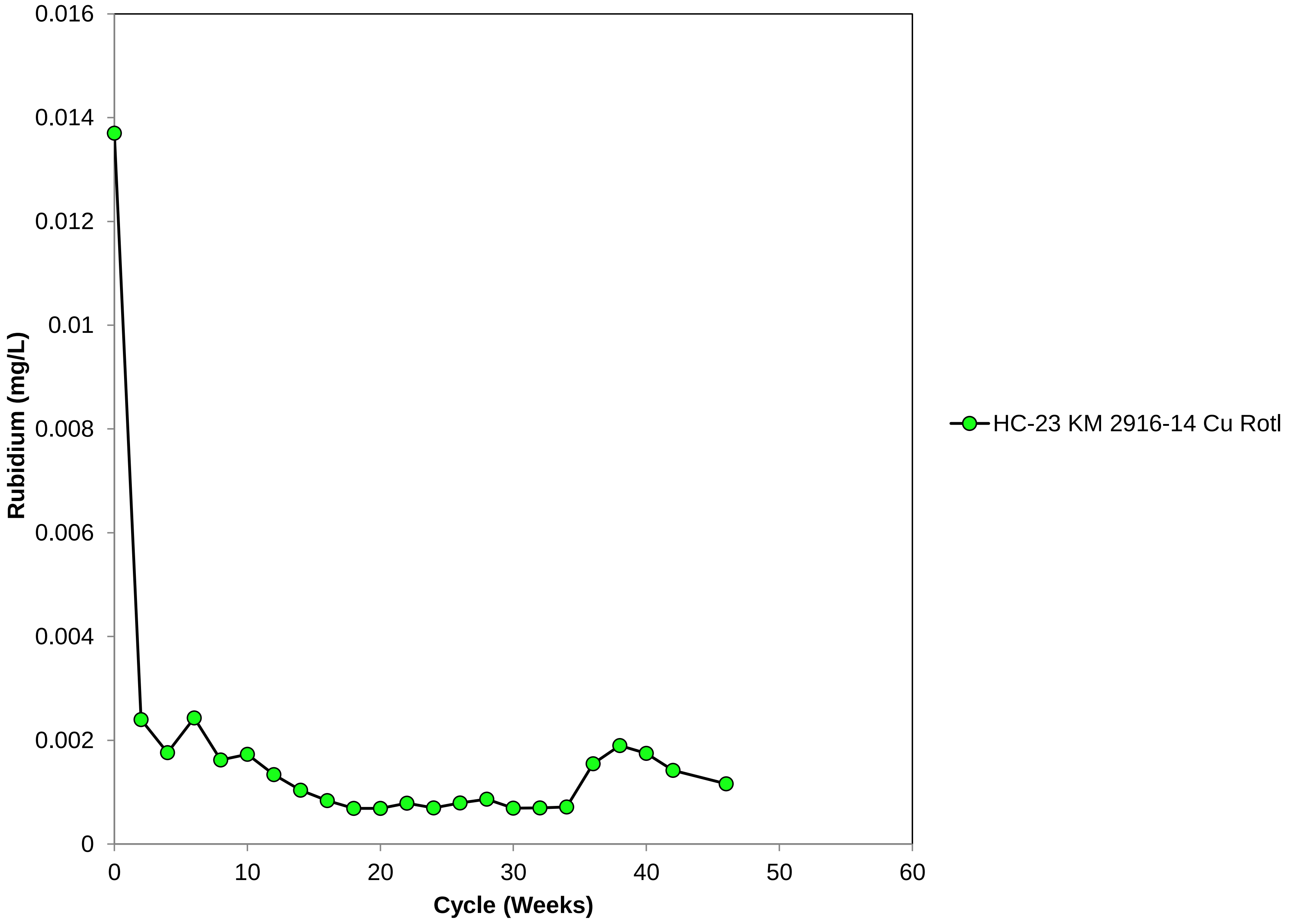

0.035
0.03
0.025
0.02
0.015
0.01
0.005
0
Manganese (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

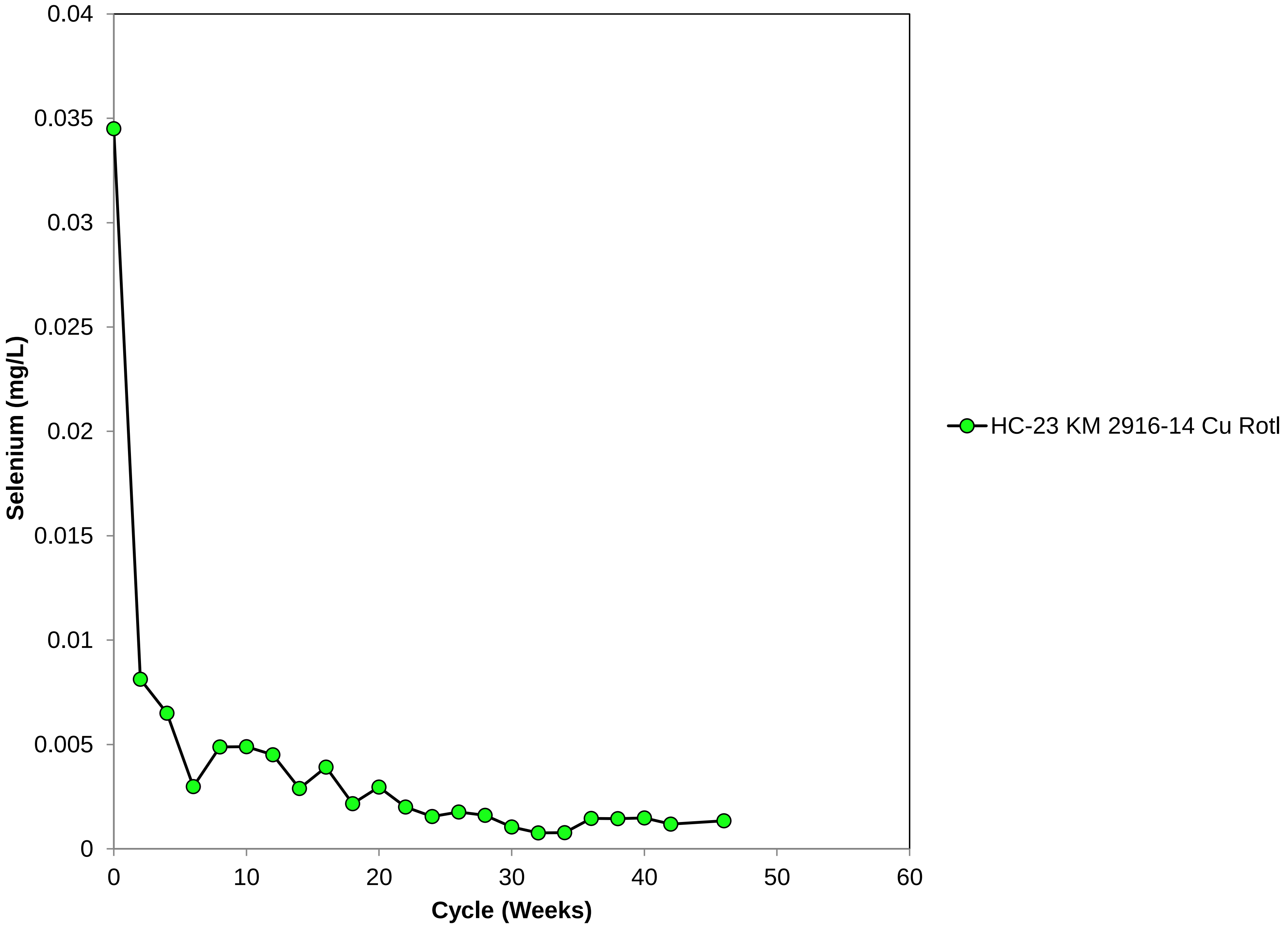

0.035
0.03
0.025
0.02
0.015
0.01
0.005
0
Molybdenum (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

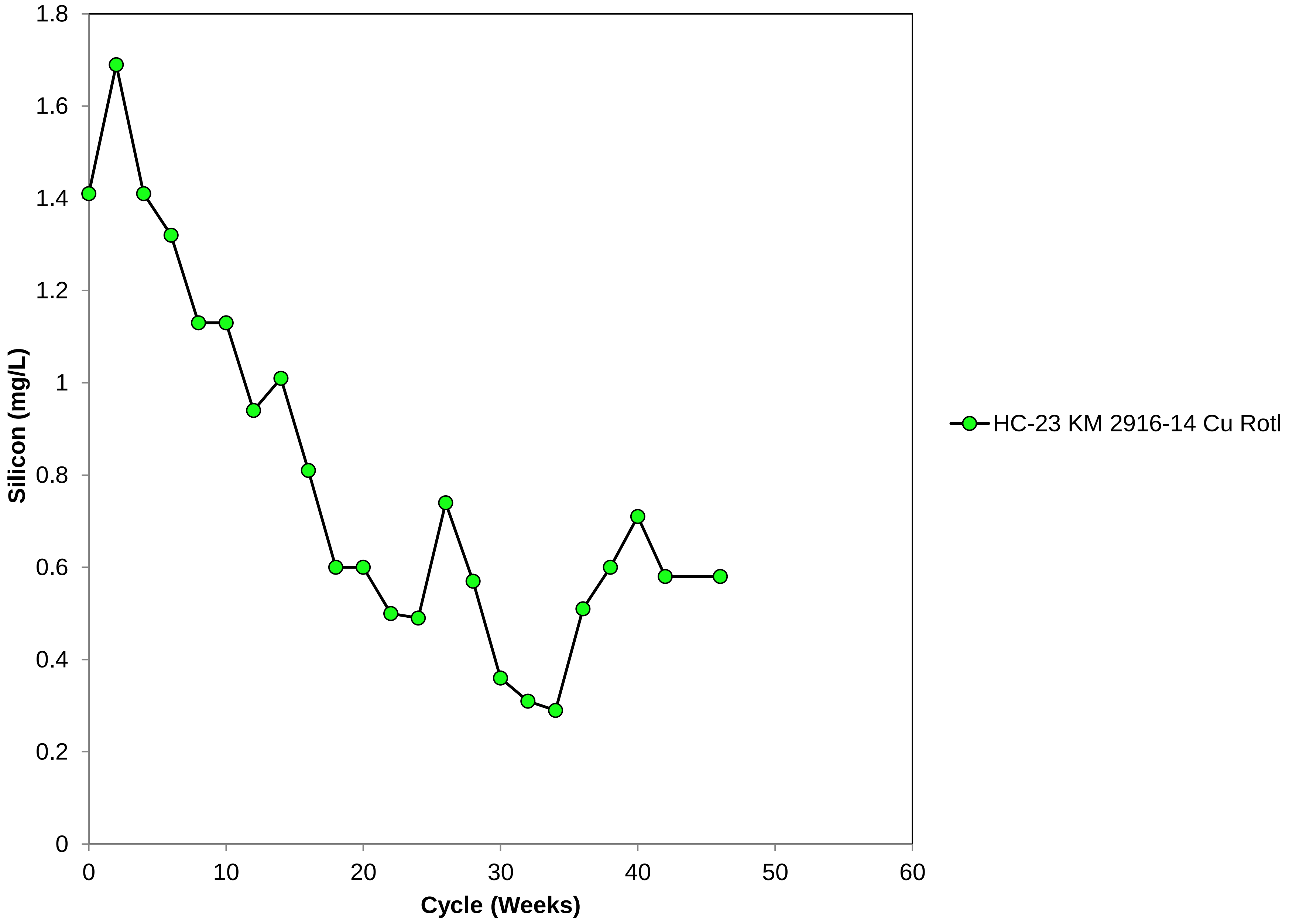

0.0035
0.003
0.0025
0.002
0.0015
0.001
0.0005
0
Nickel (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

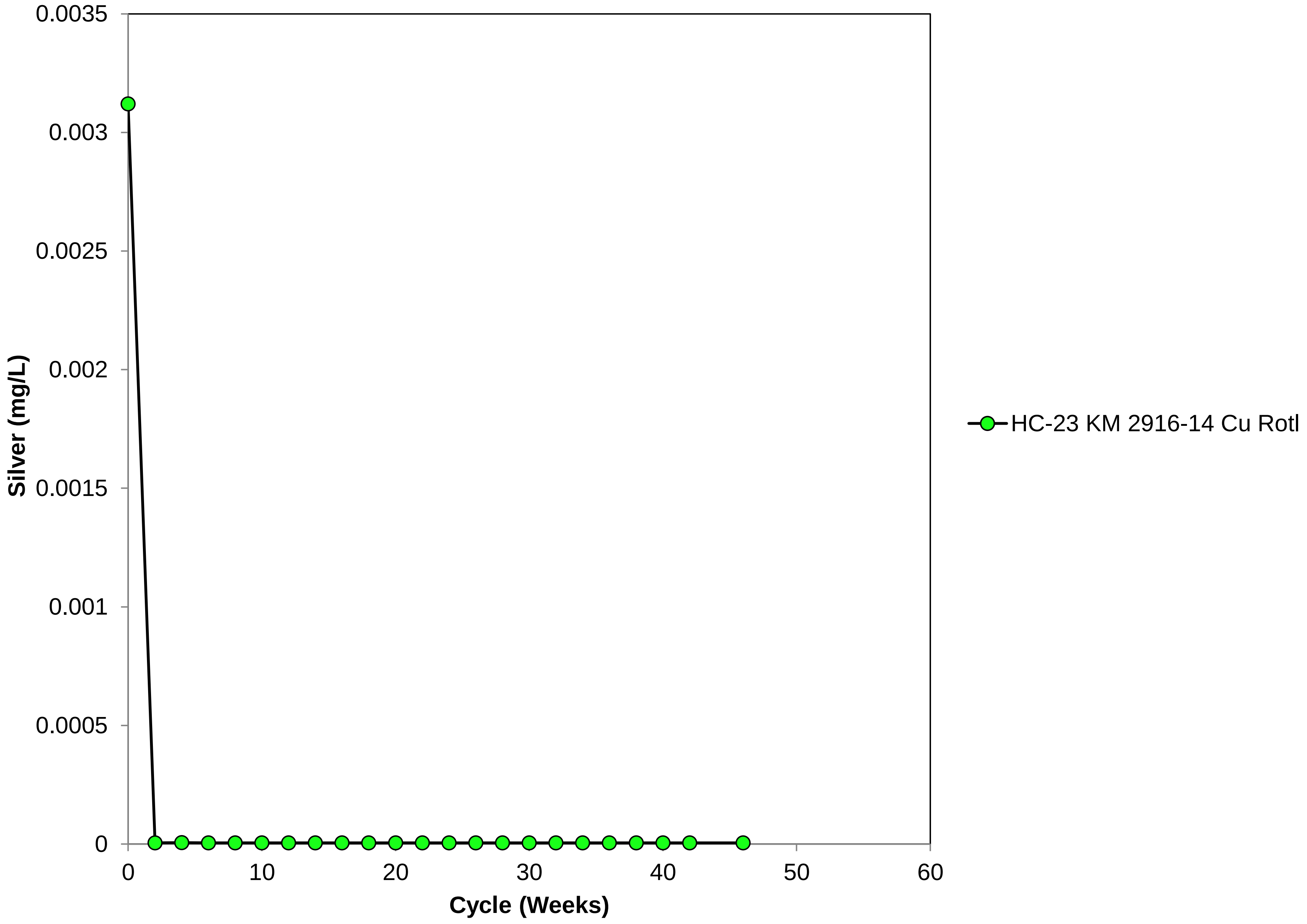

0.04
0.035
0.03
0.025
0.02
0.015
0.01
0.005
0
Phosphorus (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

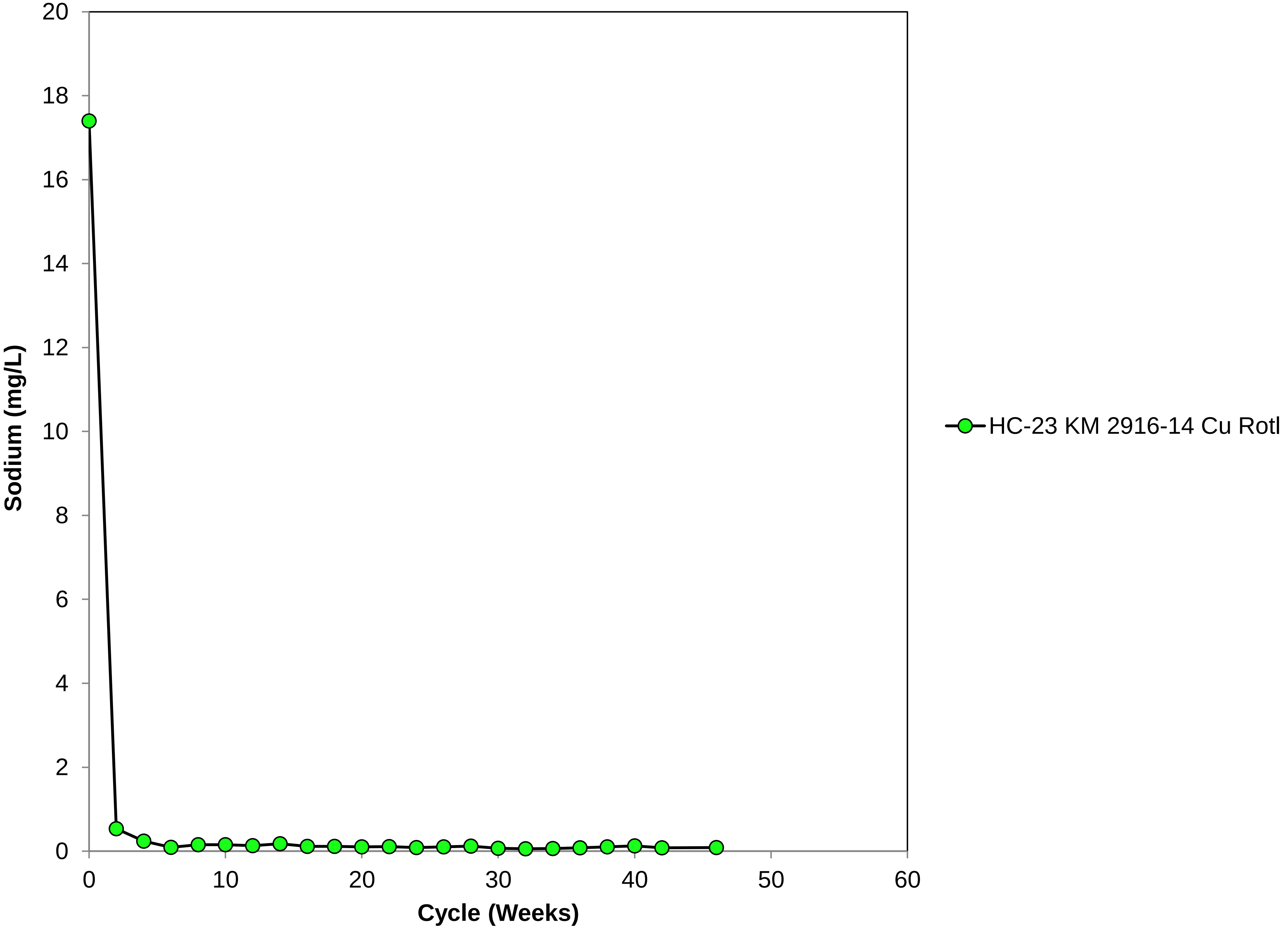

30
25
20
15
10
5
0
Potassium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl


0.016
0.014
0.012
0.01
0.008
0.006
0.004
0.002
0
Rubidium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

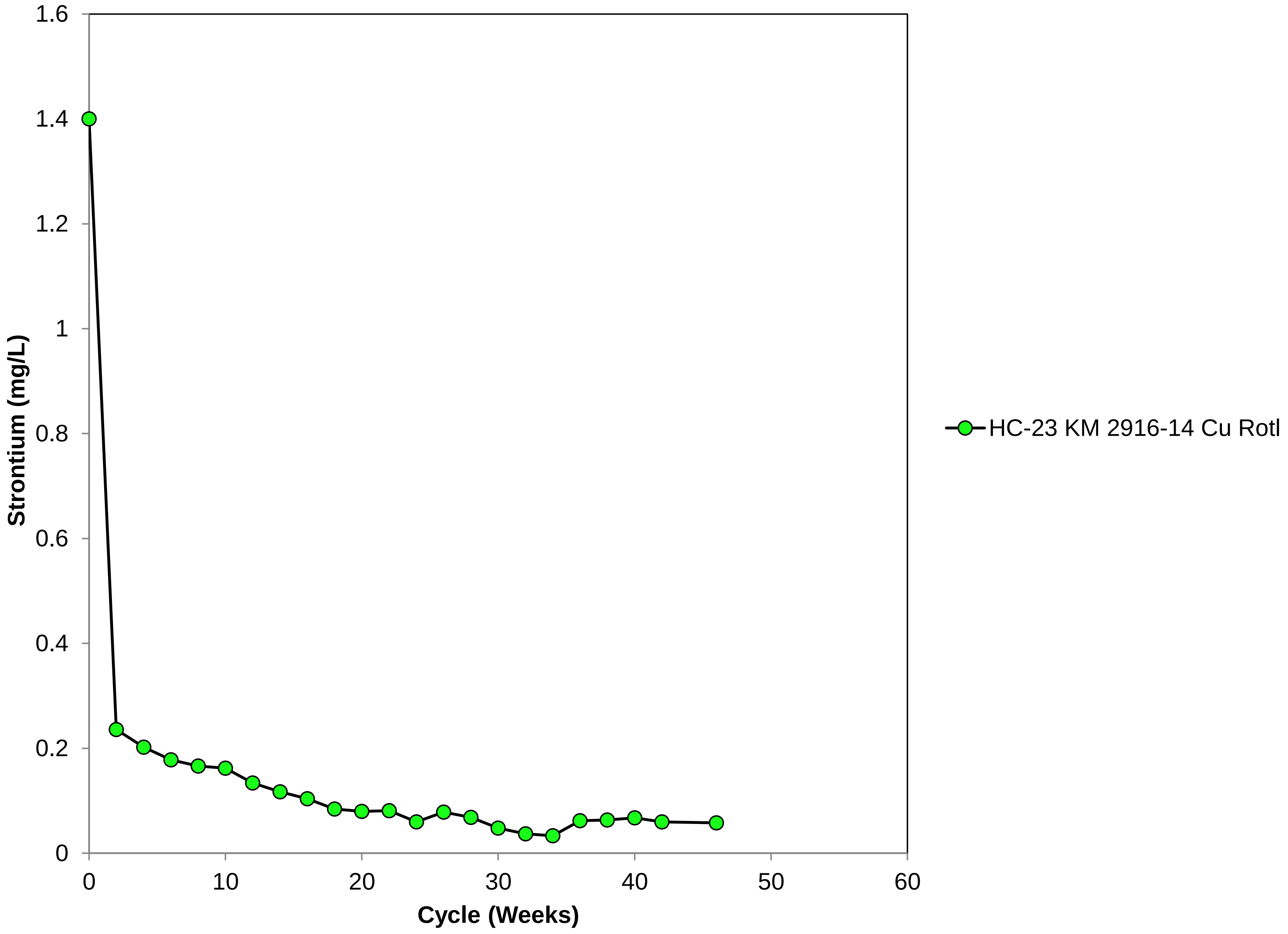

0.04
0.035
0.03
0.025
0.02
0.015
0.01
0.005
0
Selenium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

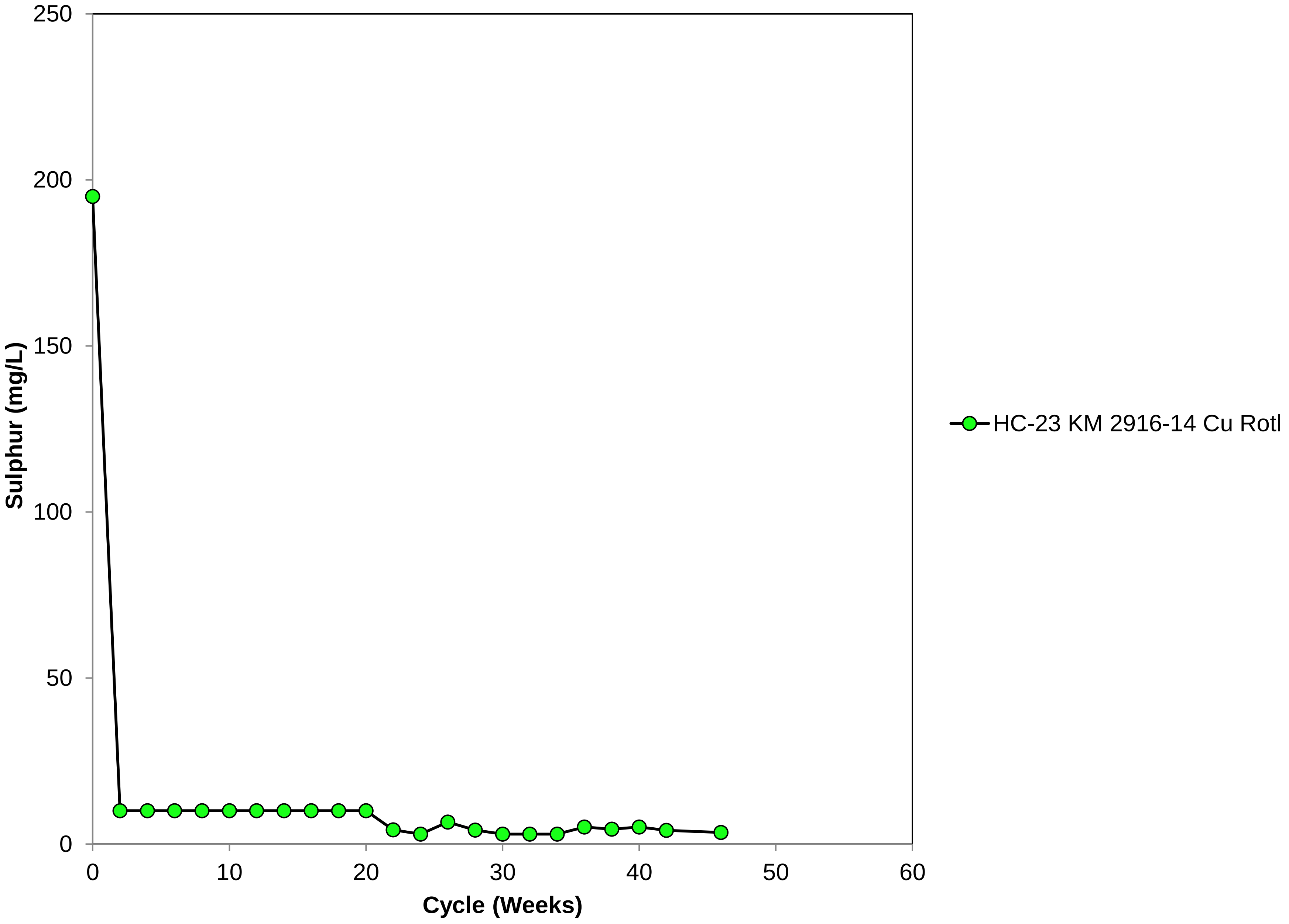

1.8
1.6
1.4
1.2
1
0.8
0.6
0.4
0.2
0
Silicon (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

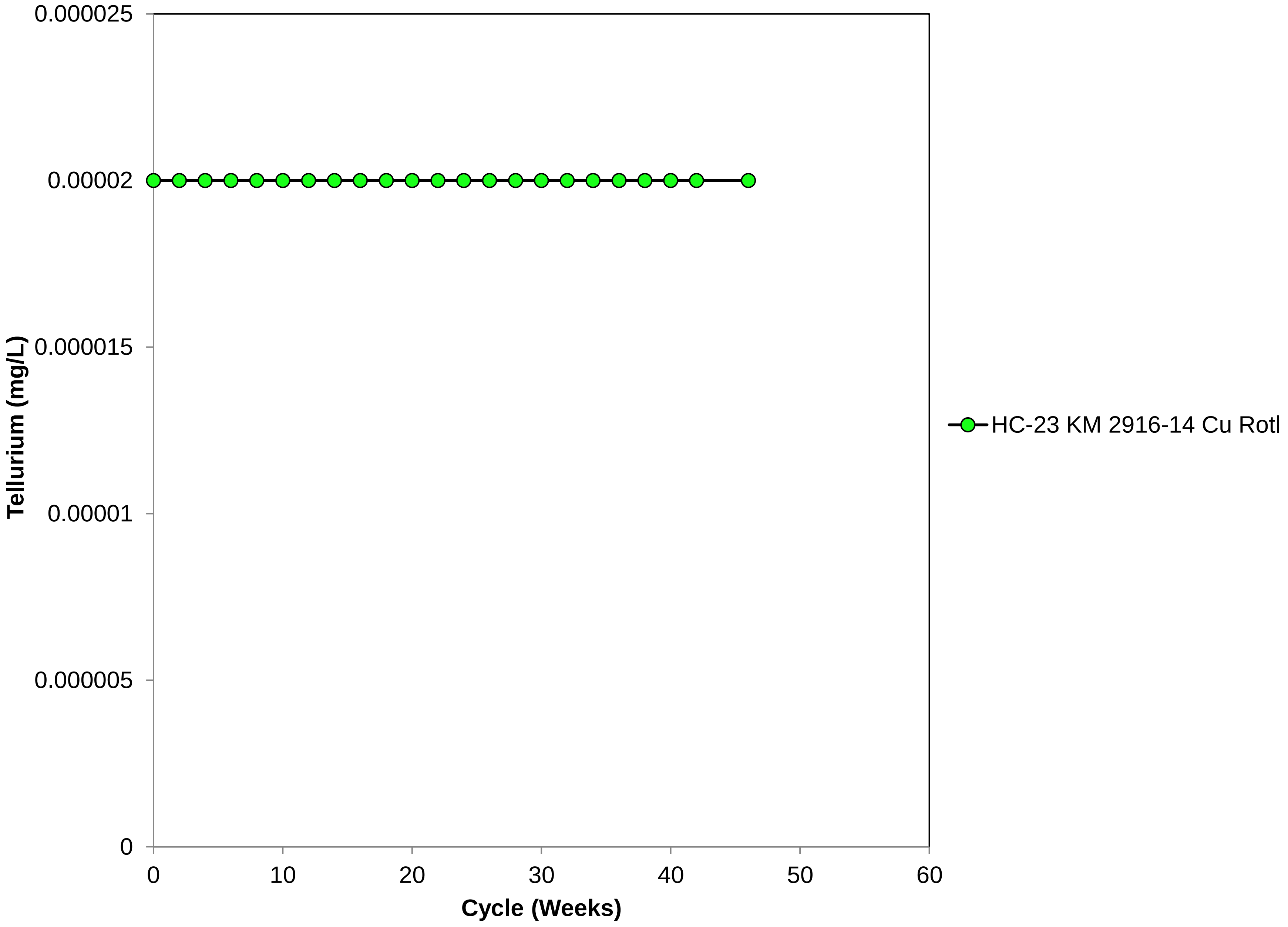

0.0035
0.003
0.0025
0.002
0.0015
0.001
0.0005
0
Silver (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

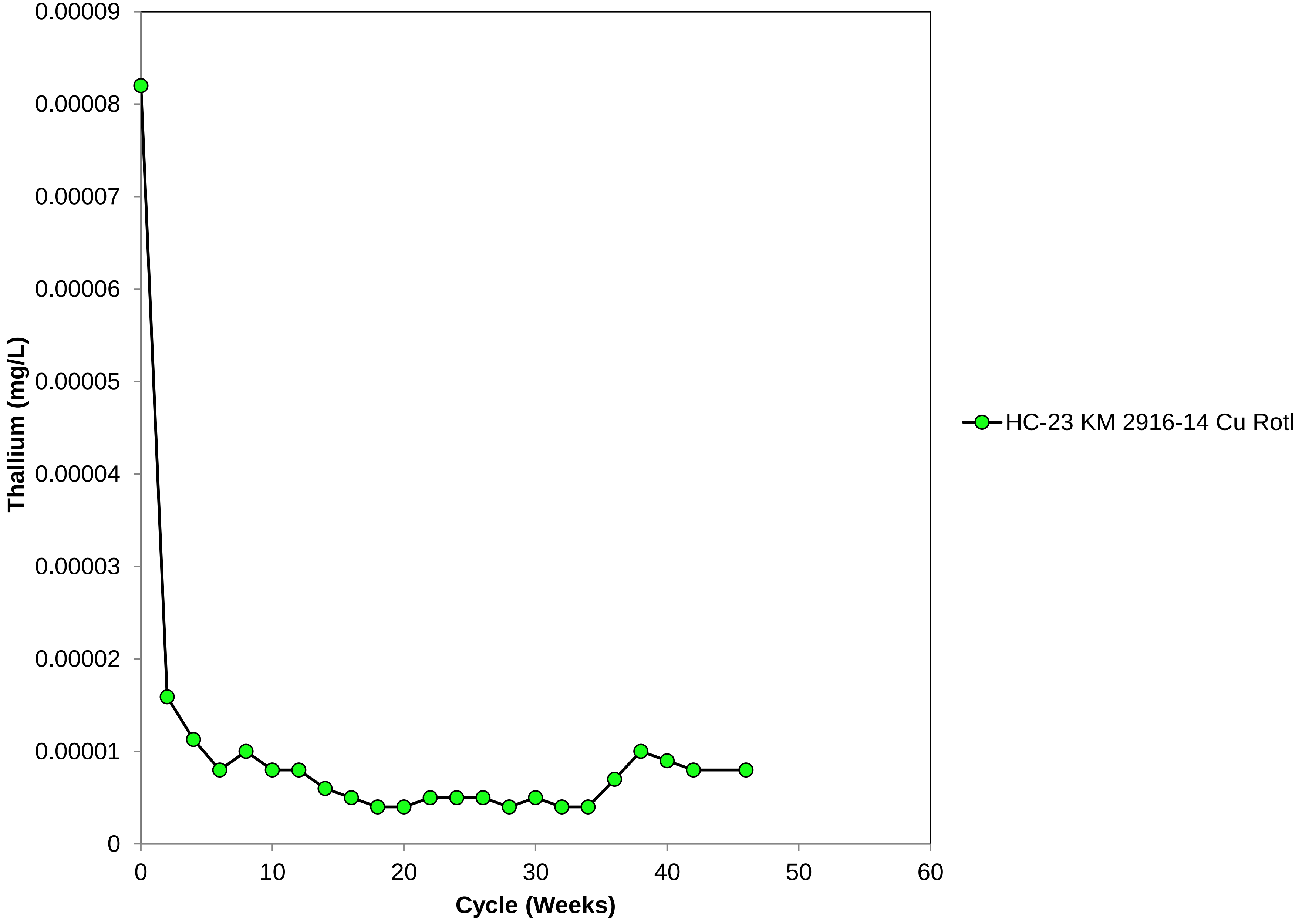


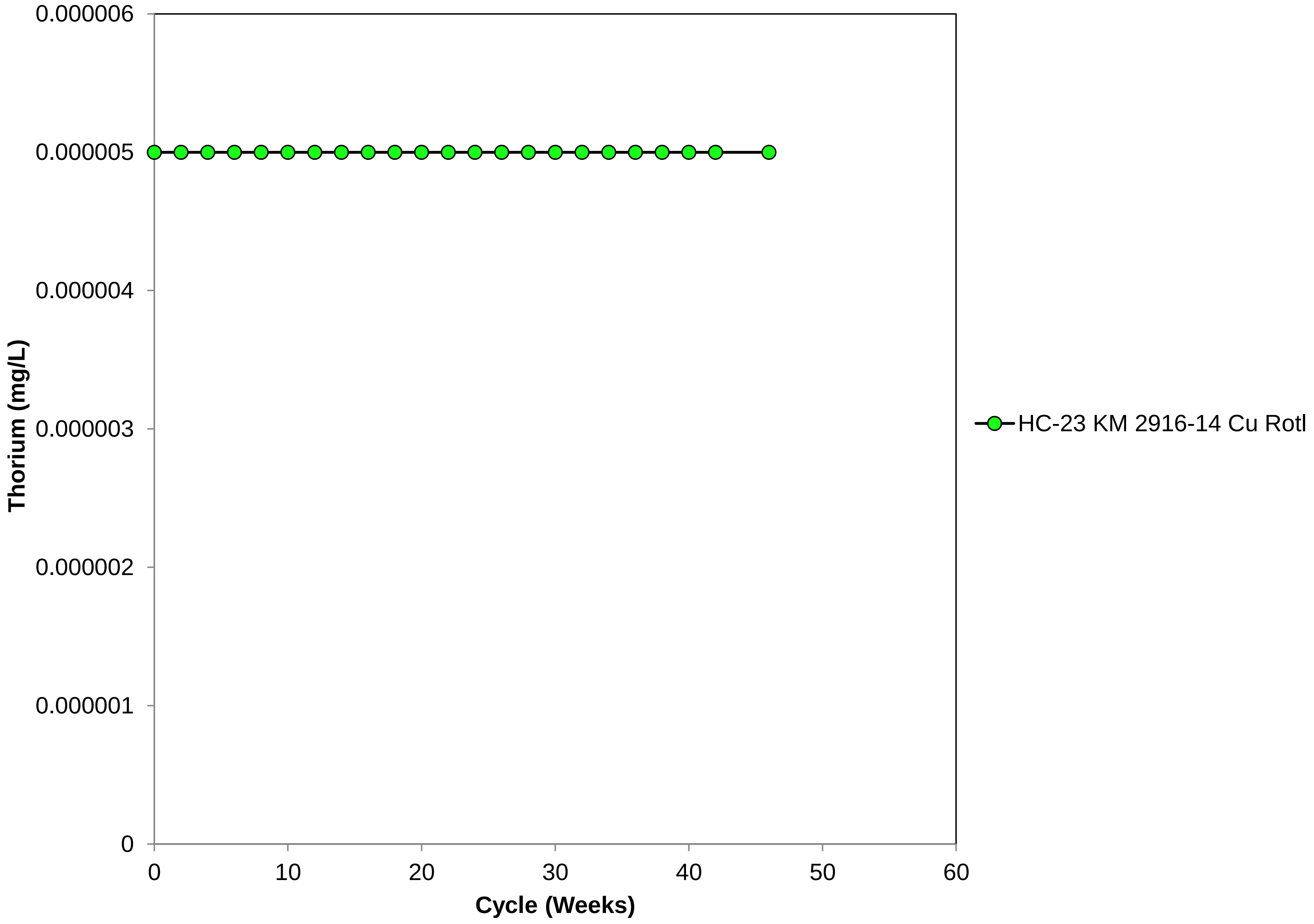
20
18
16
14
12
10
8
6
4
2
0
Sodium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

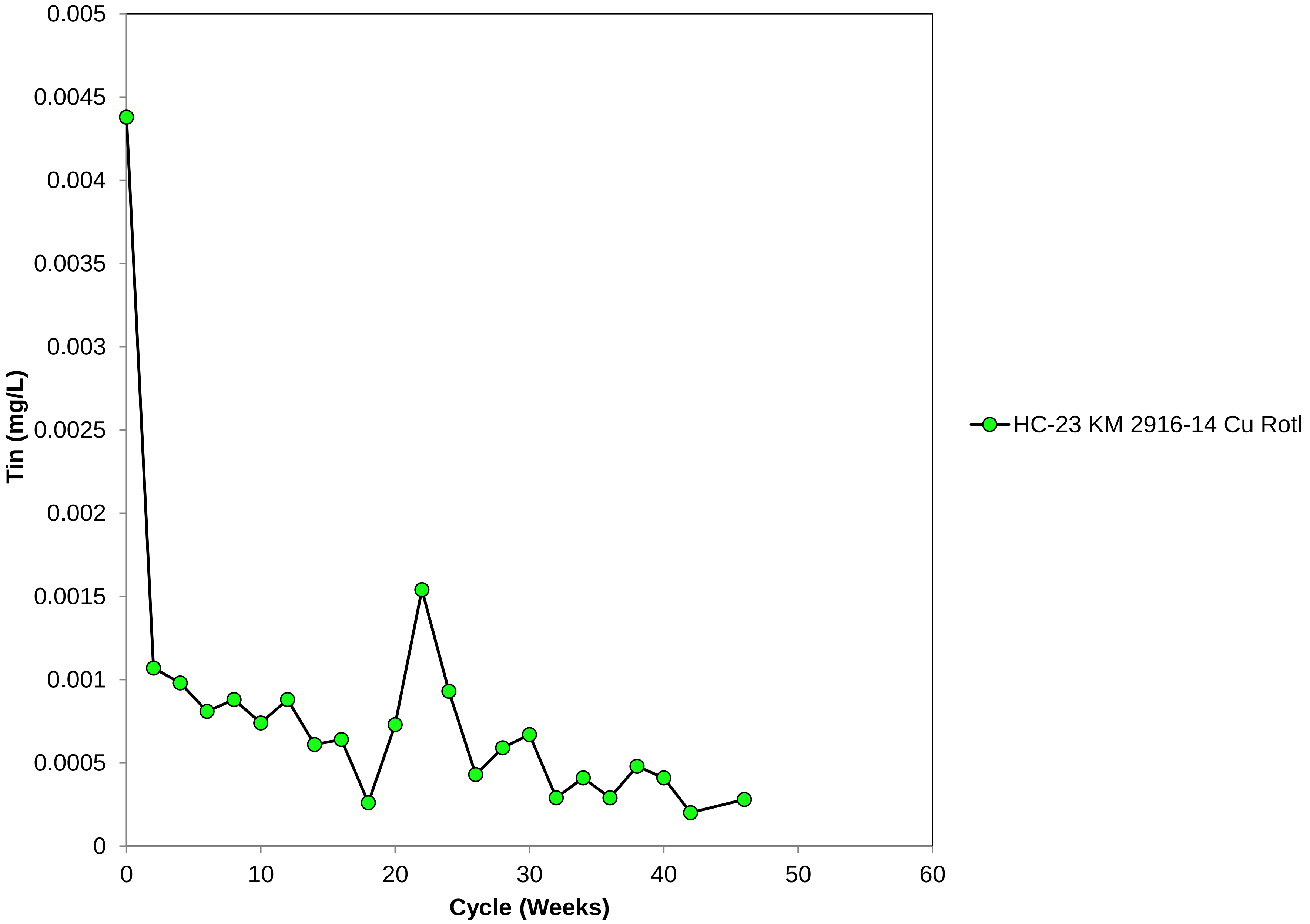

1.6
1.4
1.2
1
0.8
0.6
0.4
0.2
0
Strontium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

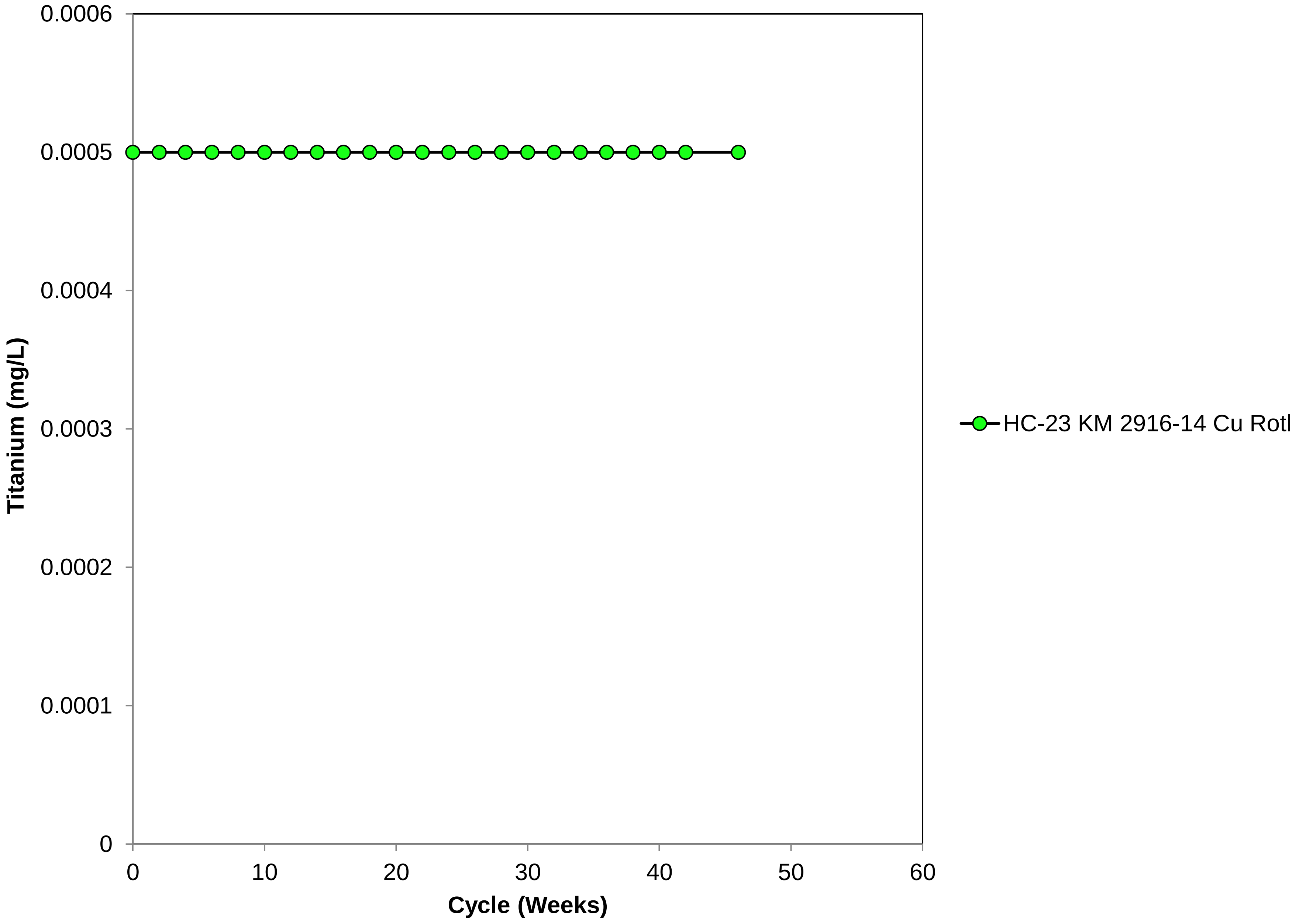

250
200
150
100
50
0
Sulphur (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

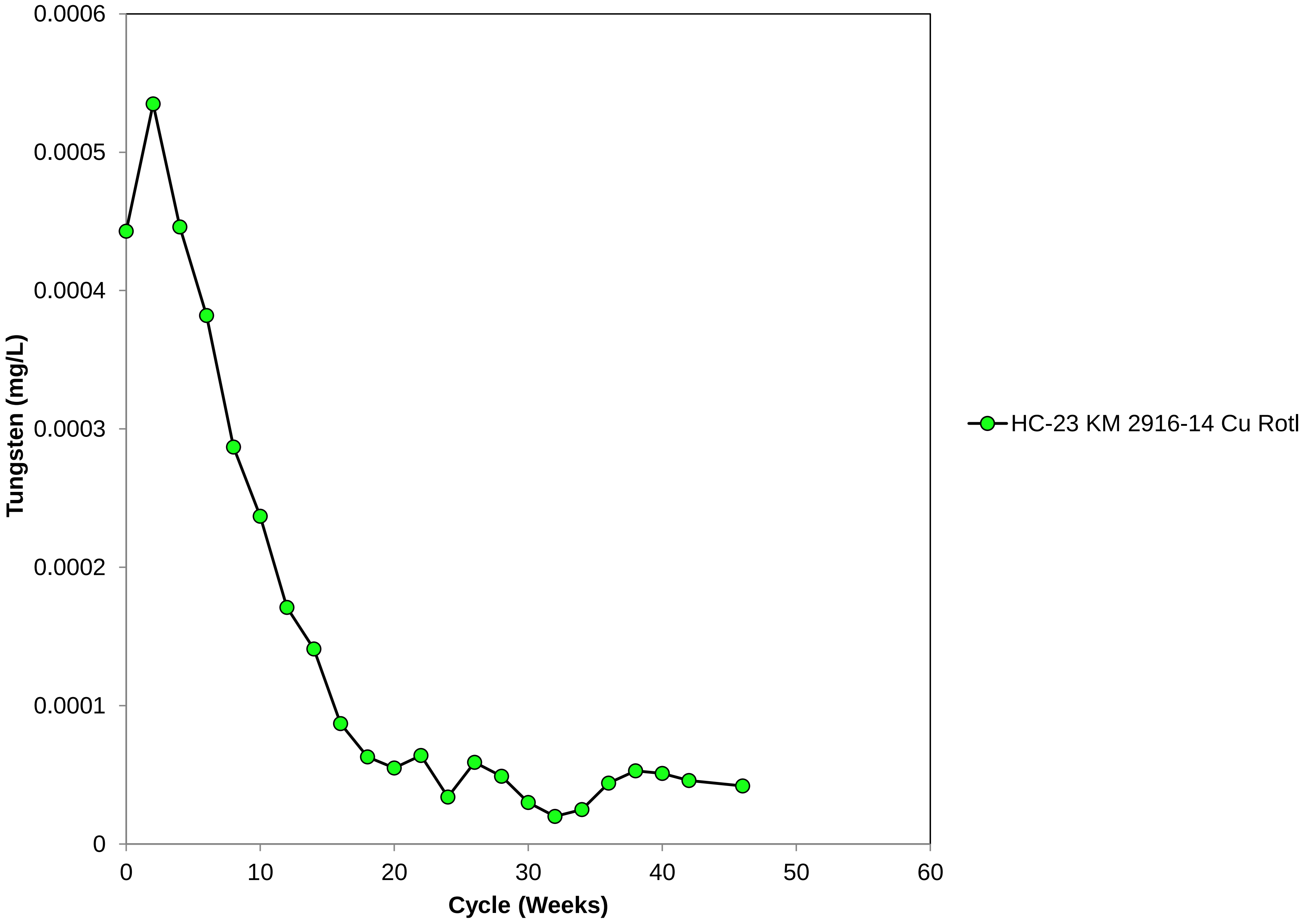

0.000025
0.00002
0.000015
0.00001
0.000005
0
Tellurium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl


0.00009
0.00008
0.00007
0.00006
0.00005
0.00004
0.00003
0.00002
0.00001
0
Thallium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

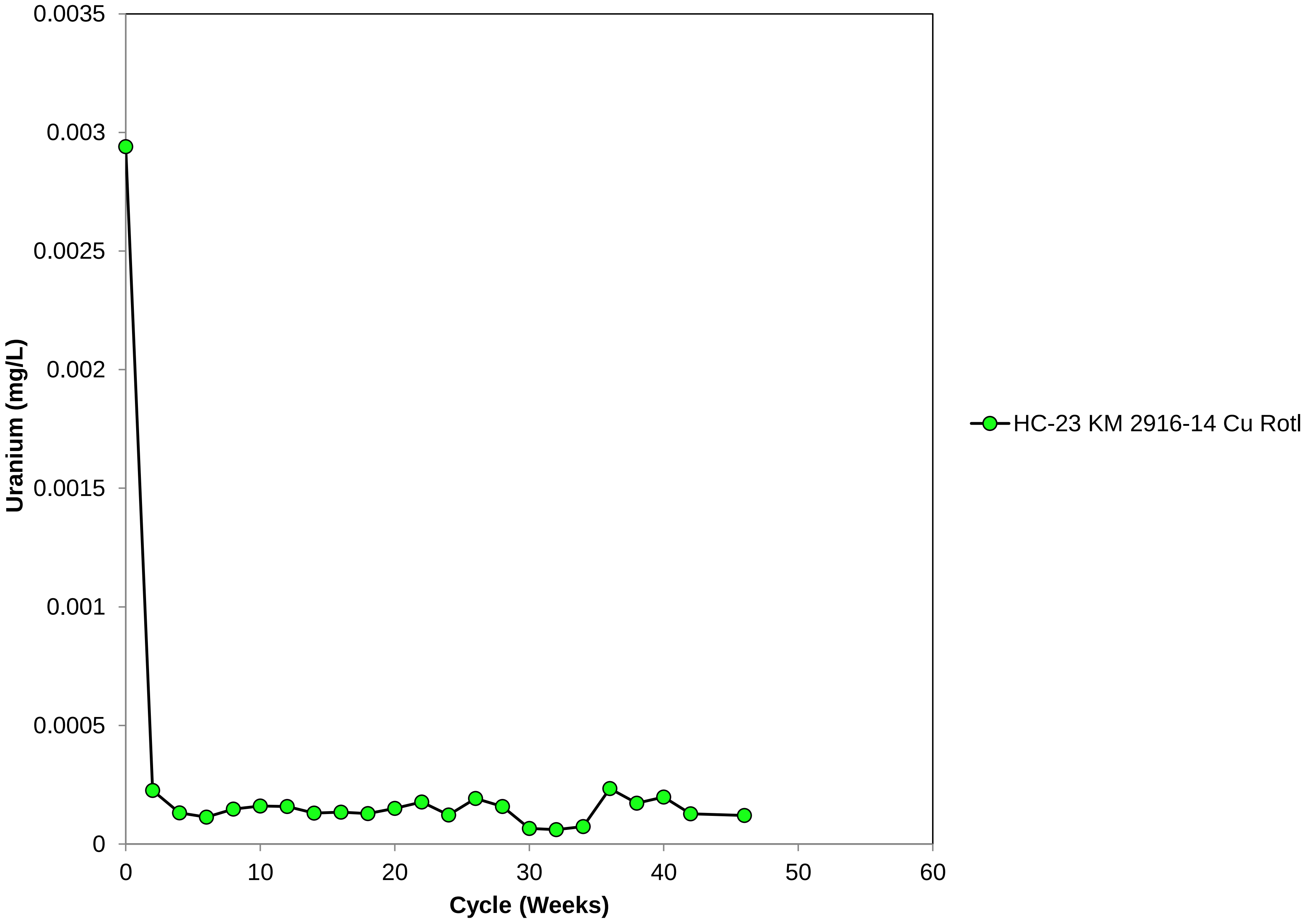

0.000006
0.000005
0.000004
0.000003
0.000002
0.000001
0
Thorium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

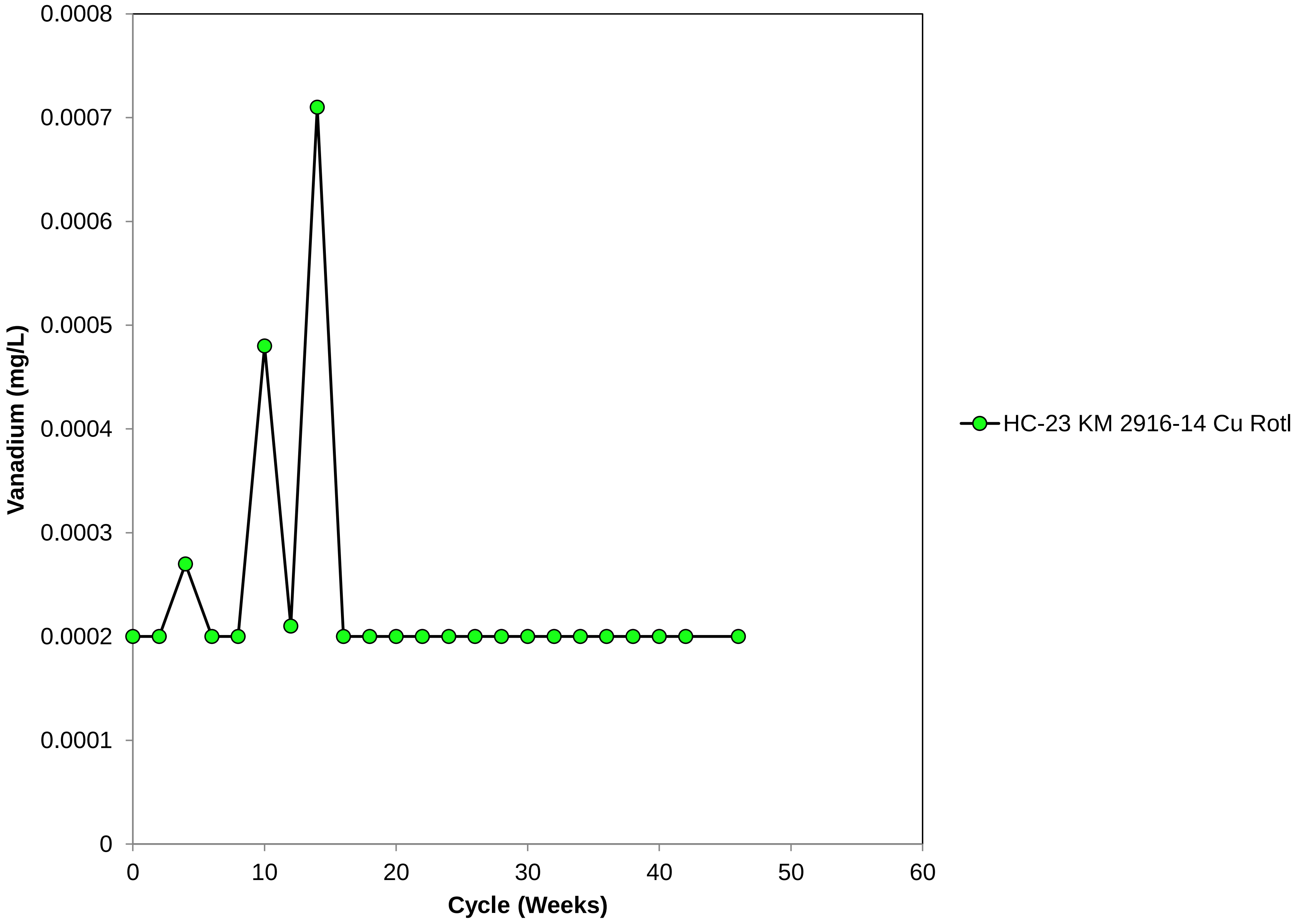

0.005
0.0045
0.004
0.0035
0.003
0.0025
0.002
0.0015
0.001
0.0005
0
Tin (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

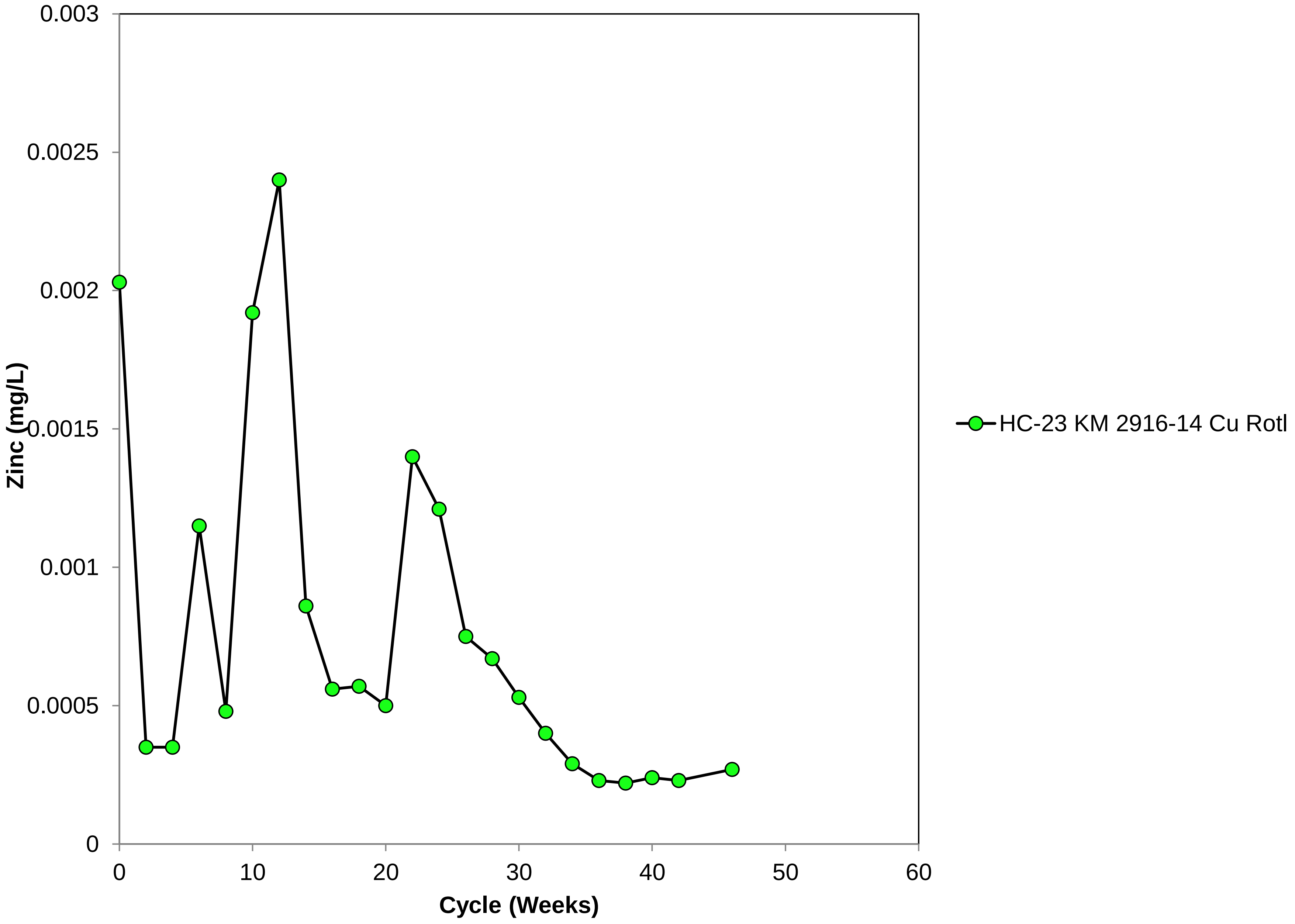

0.0006
0.0005
0.0004
0.0003
0.0002
0.0001
0
Titanium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

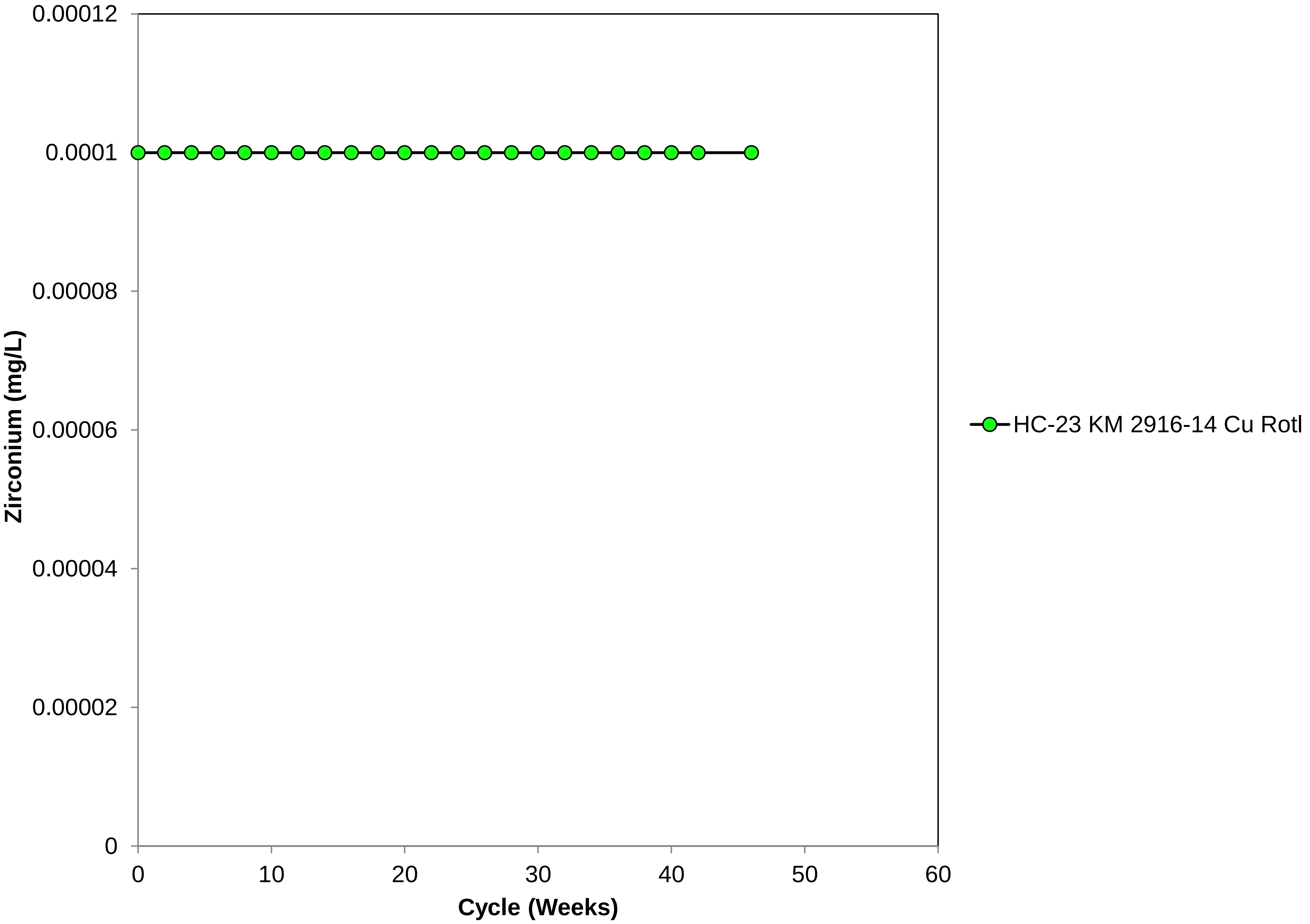


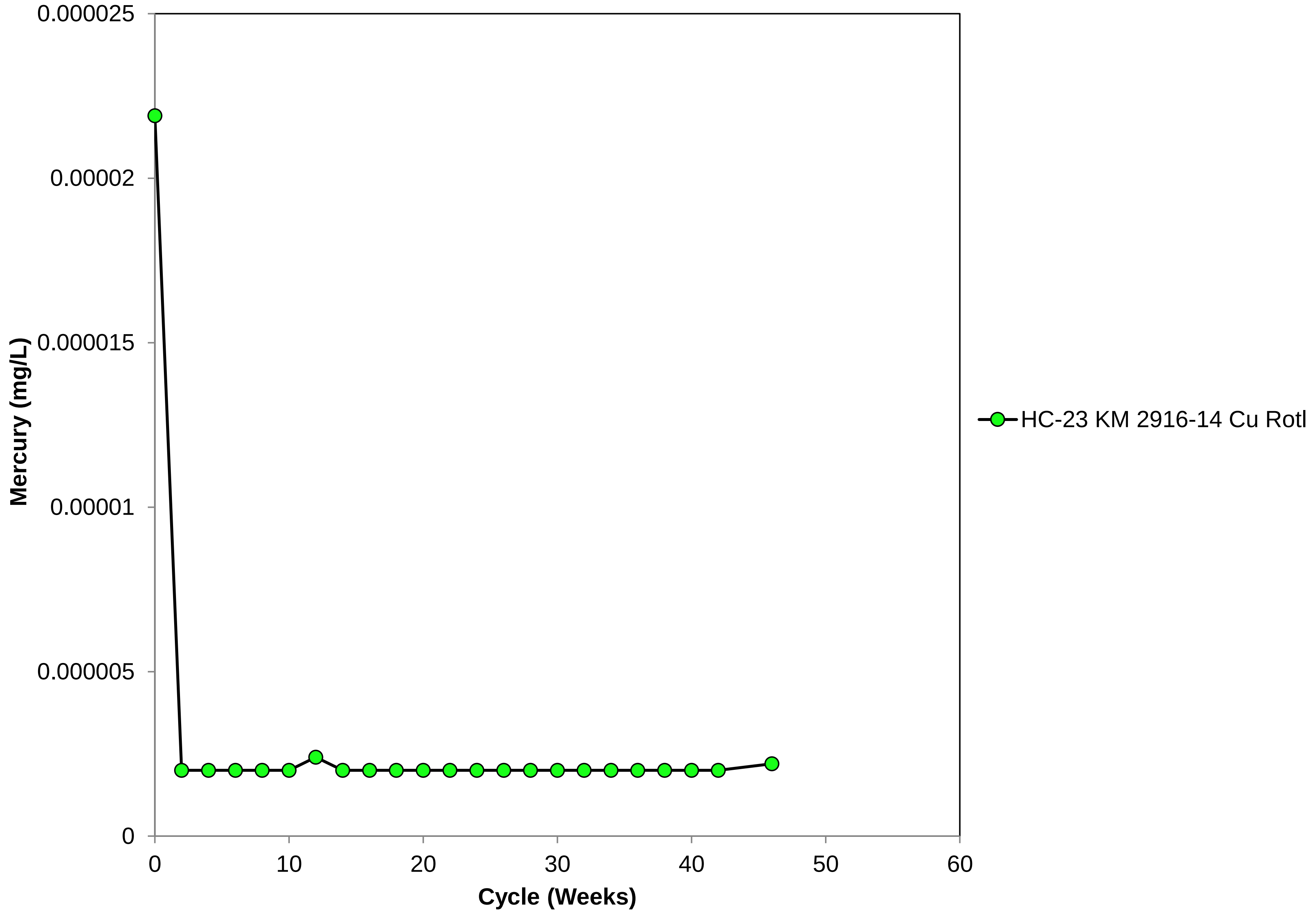
\\VAN-SVR0\Projects\01_SITES\Harper_Creek\1CH014.001_EA_Re-Submission\300_Reporting\310_Update SRK Report\030_Appendices\Appendix D\Working Files\Appendix D4_Harper_TailingsHCTConc_Charts


0.0035
0.003
0.0025
0.002
0.0015
0.001
0.0005
0
0
10
20
30
40
50
60
Uranium (mg/L)
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

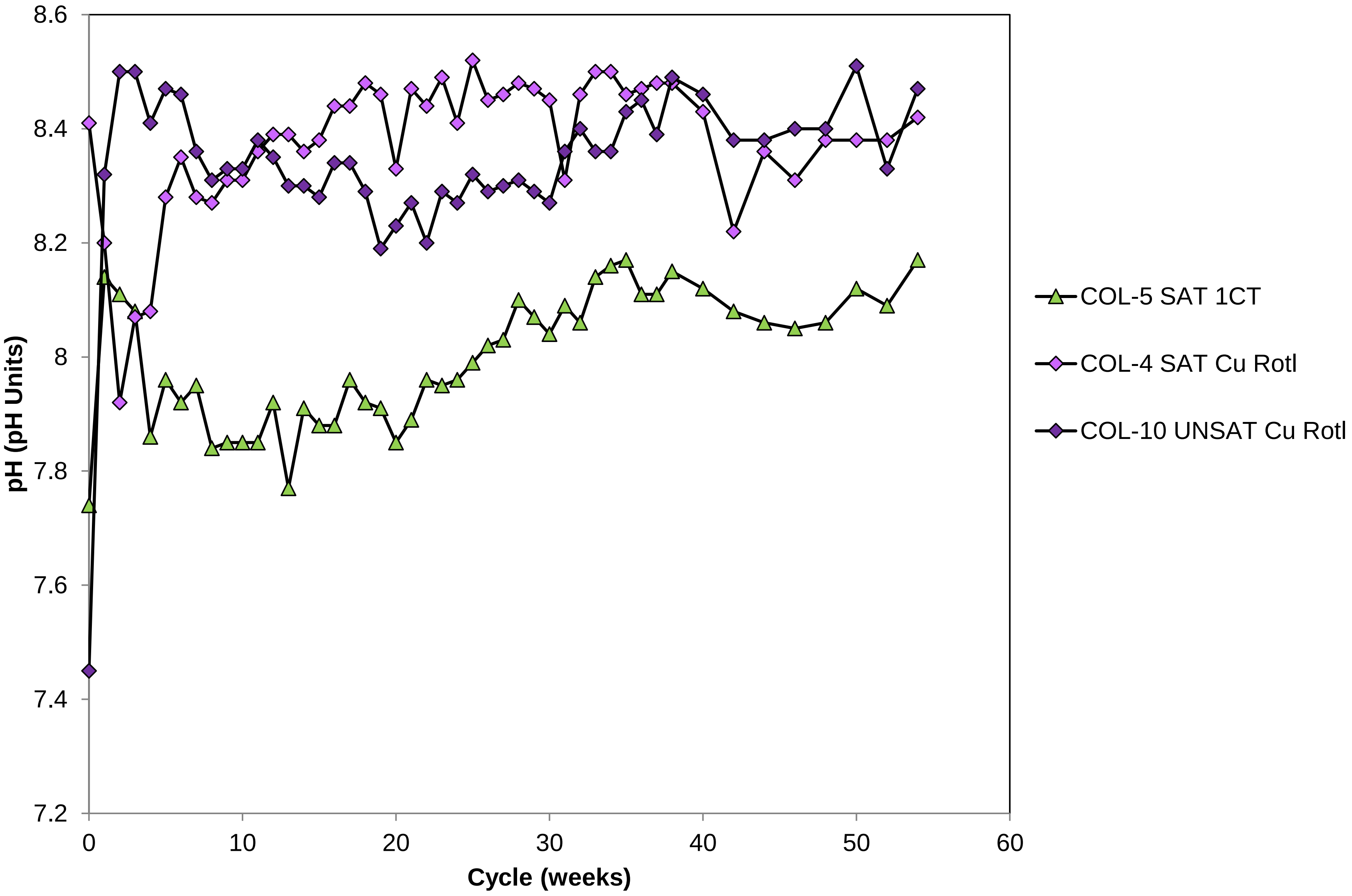

0.0008
0.0007
0.0006
0.0005
0.0004
0.0003
0.0002
0.0001
0
Vanadium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

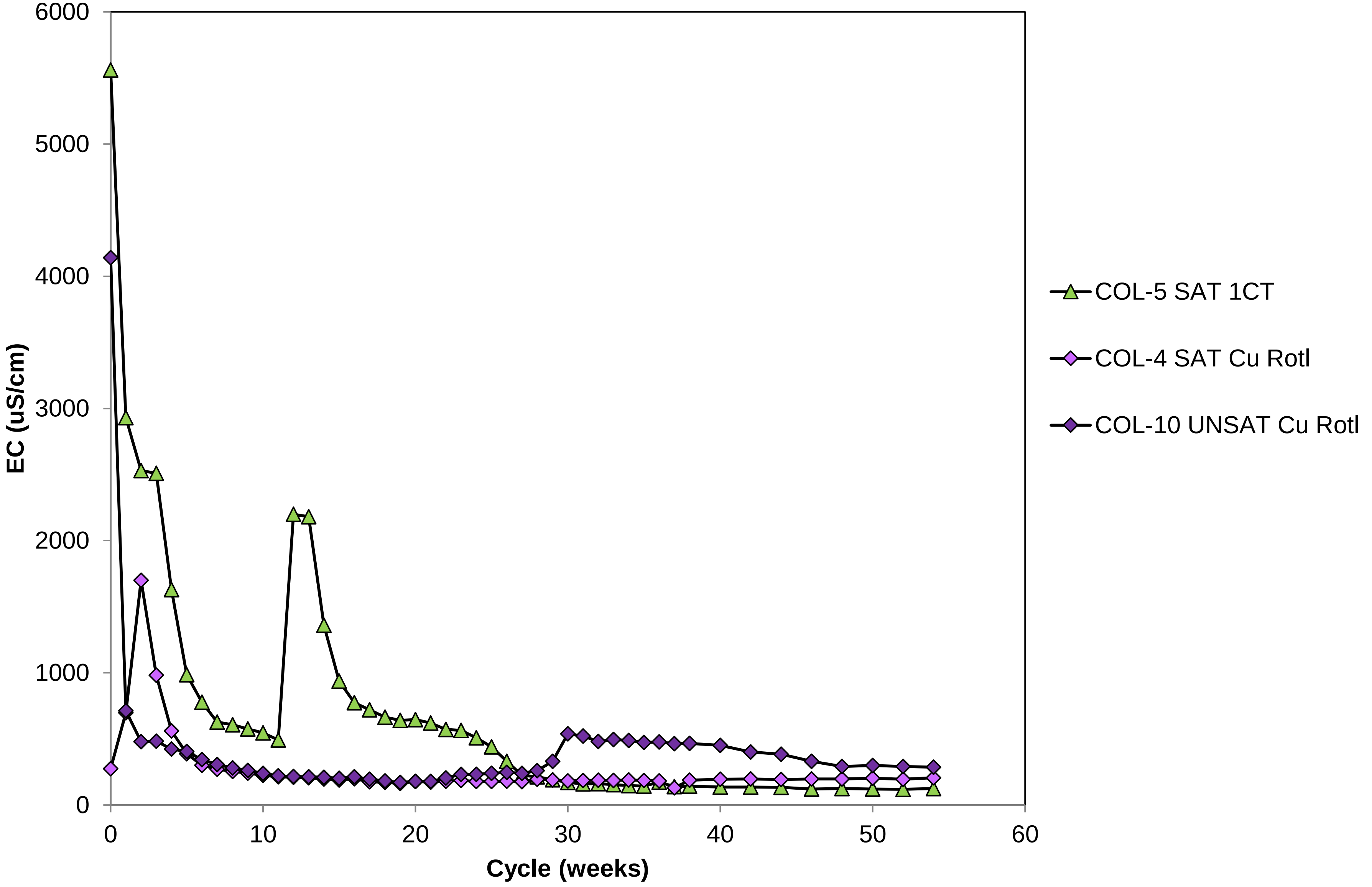

0.003
0.0025
0.002
0.0015
0.001
0.0005
0
Zinc (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

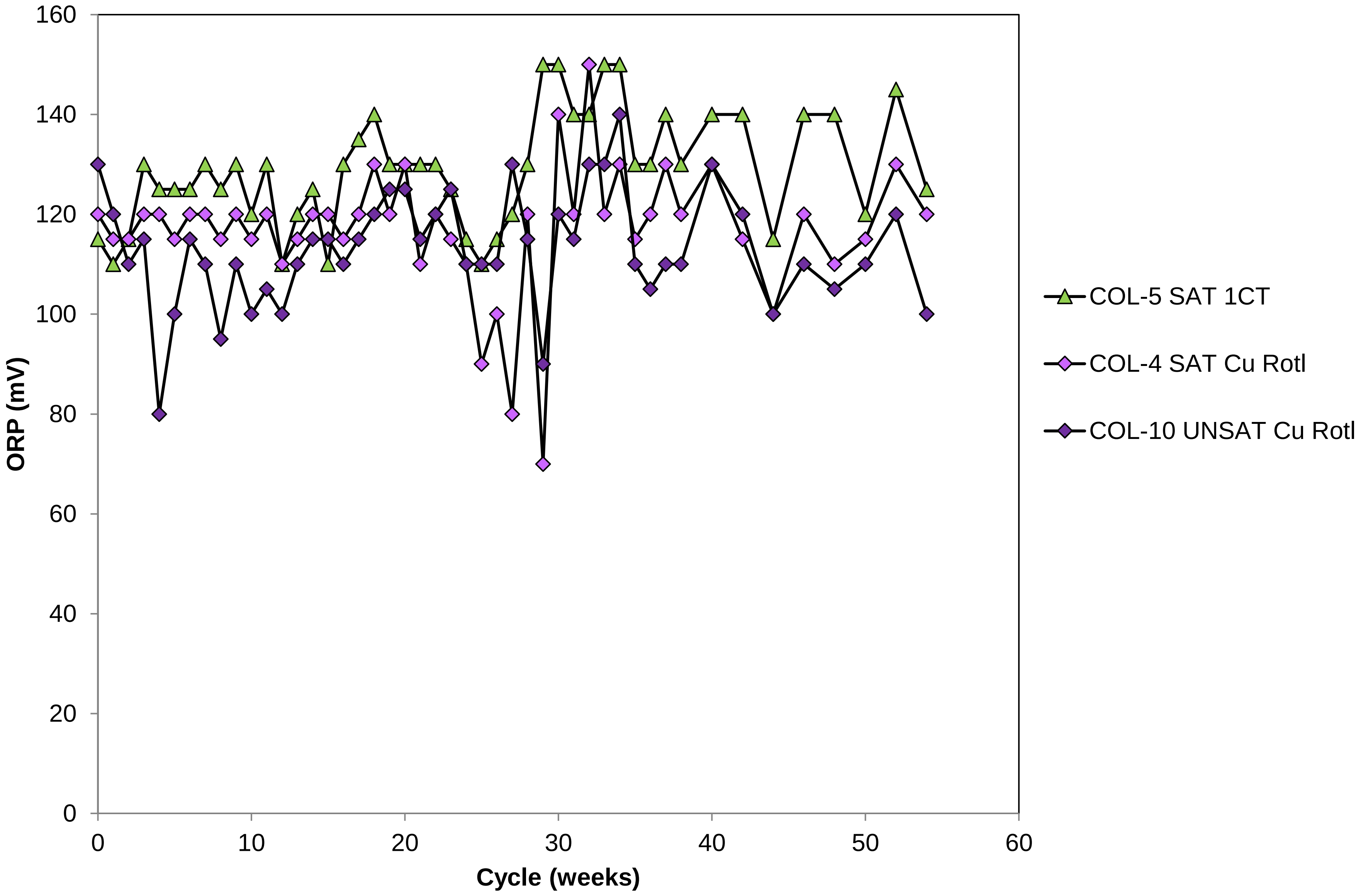

0.00012
0.0001
0.00008
0.00006
0.00004
0.00002
0
Zirconium (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

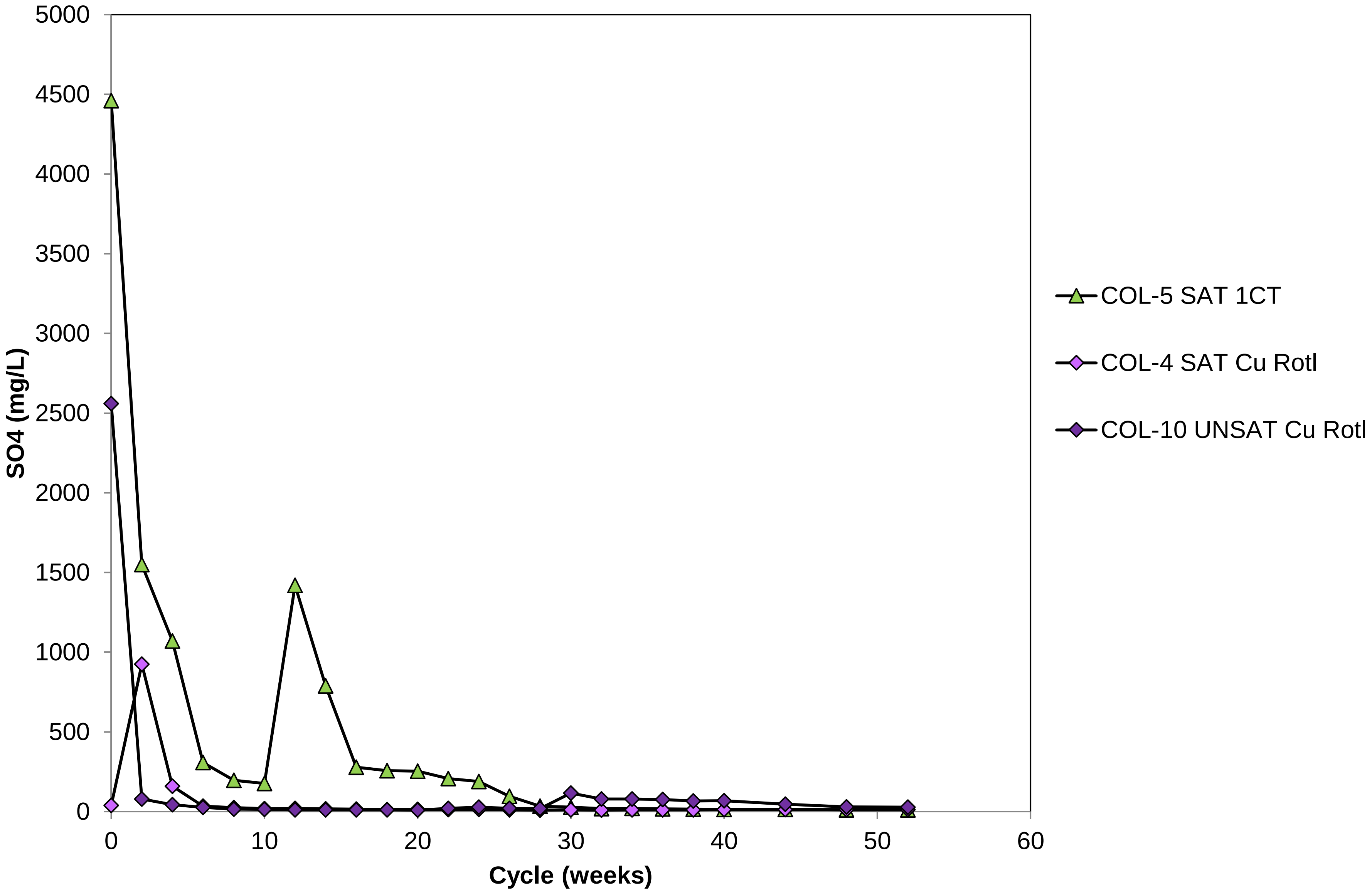

0.000025
0.00002
0.000015
0.00001
0.000005
0
Mercury (mg/L)
0
10
20
30
40
50
60
Cycle (Weeks)
HC-23 KM 2916-14 Cu Rotl

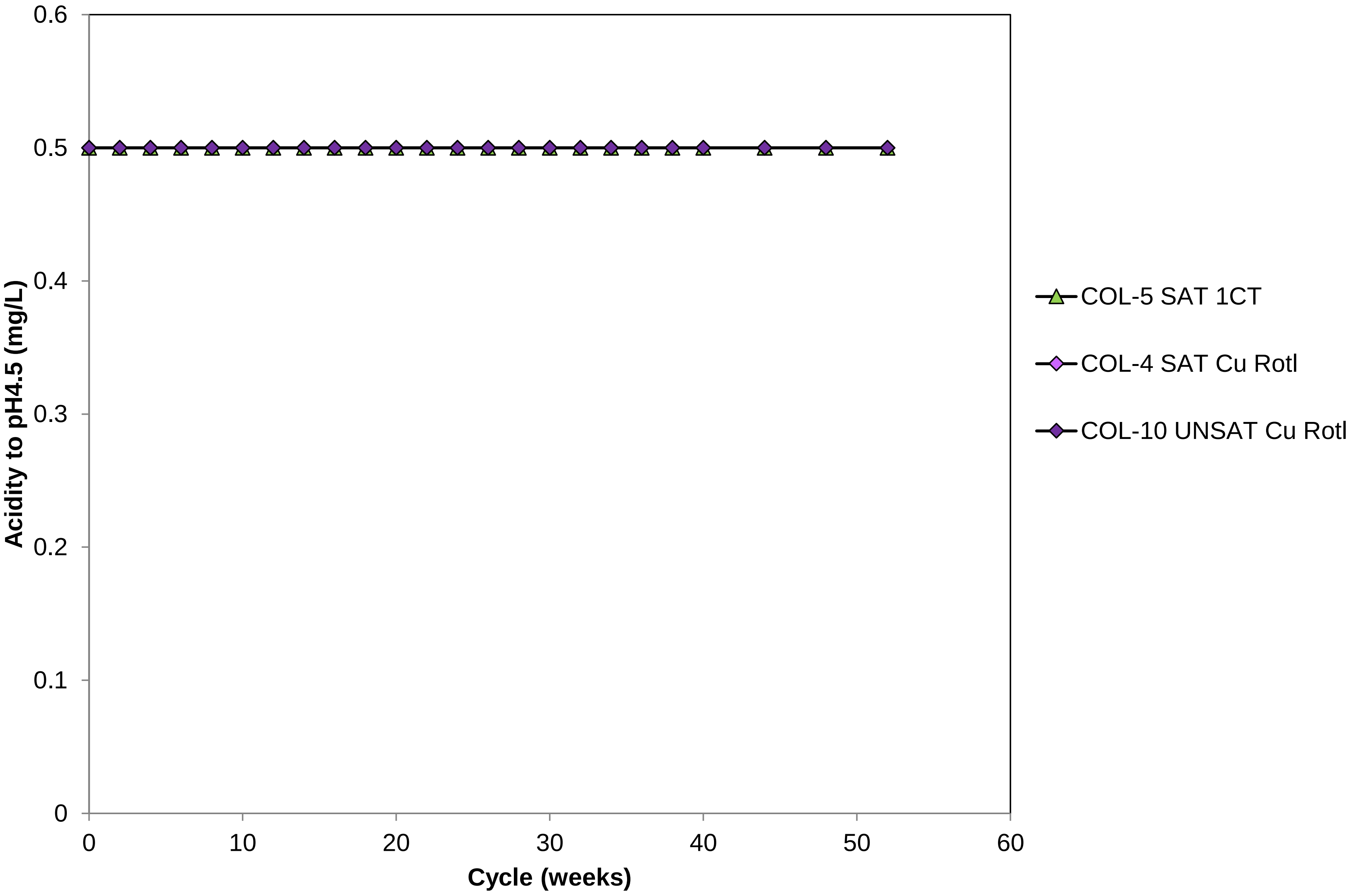
D5: Saturated and Unsaturated Columns

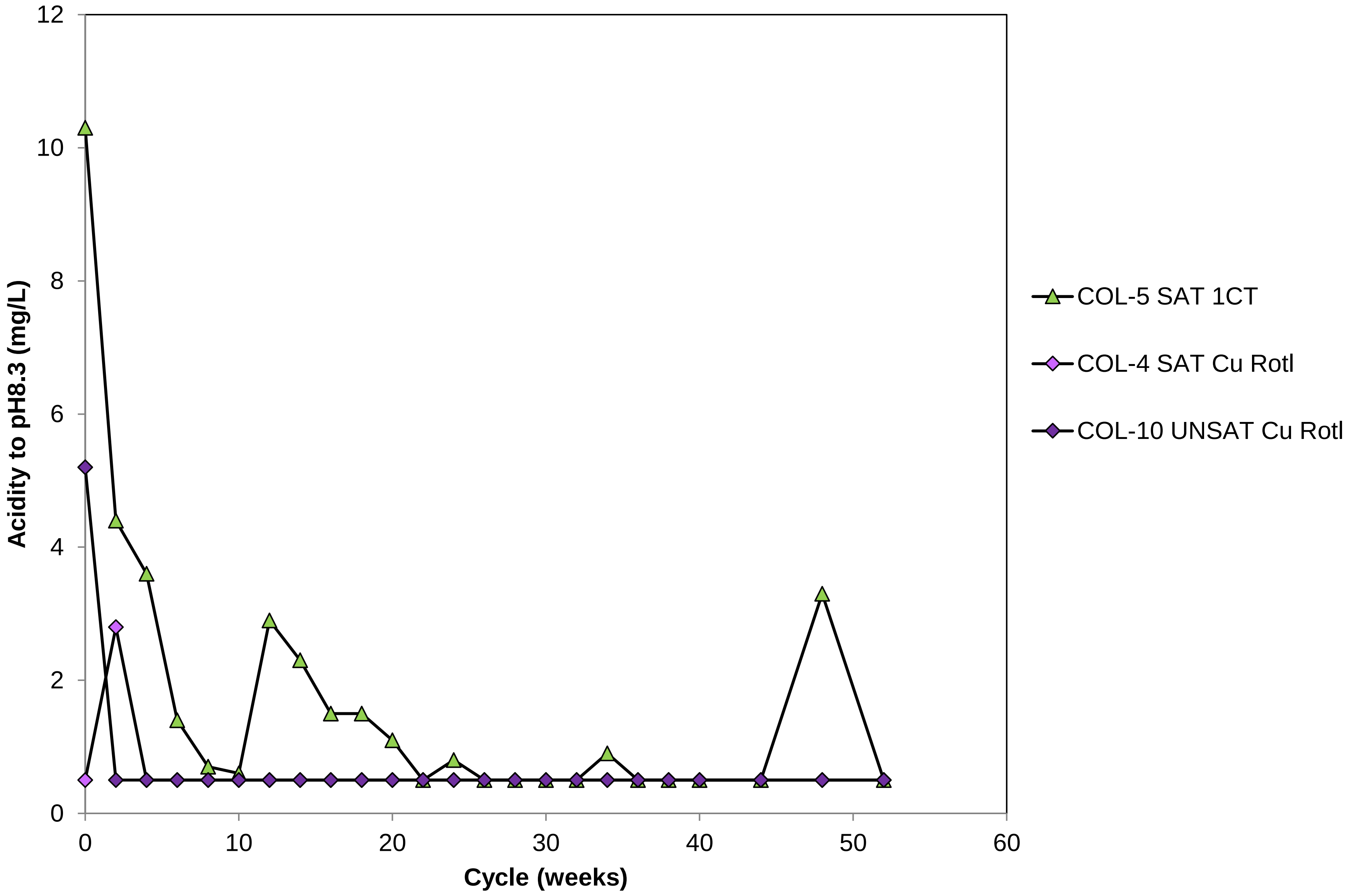

8.6
8.4
8.2
8
7.8
7.6
7.4
7.2
pH (pH Units)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

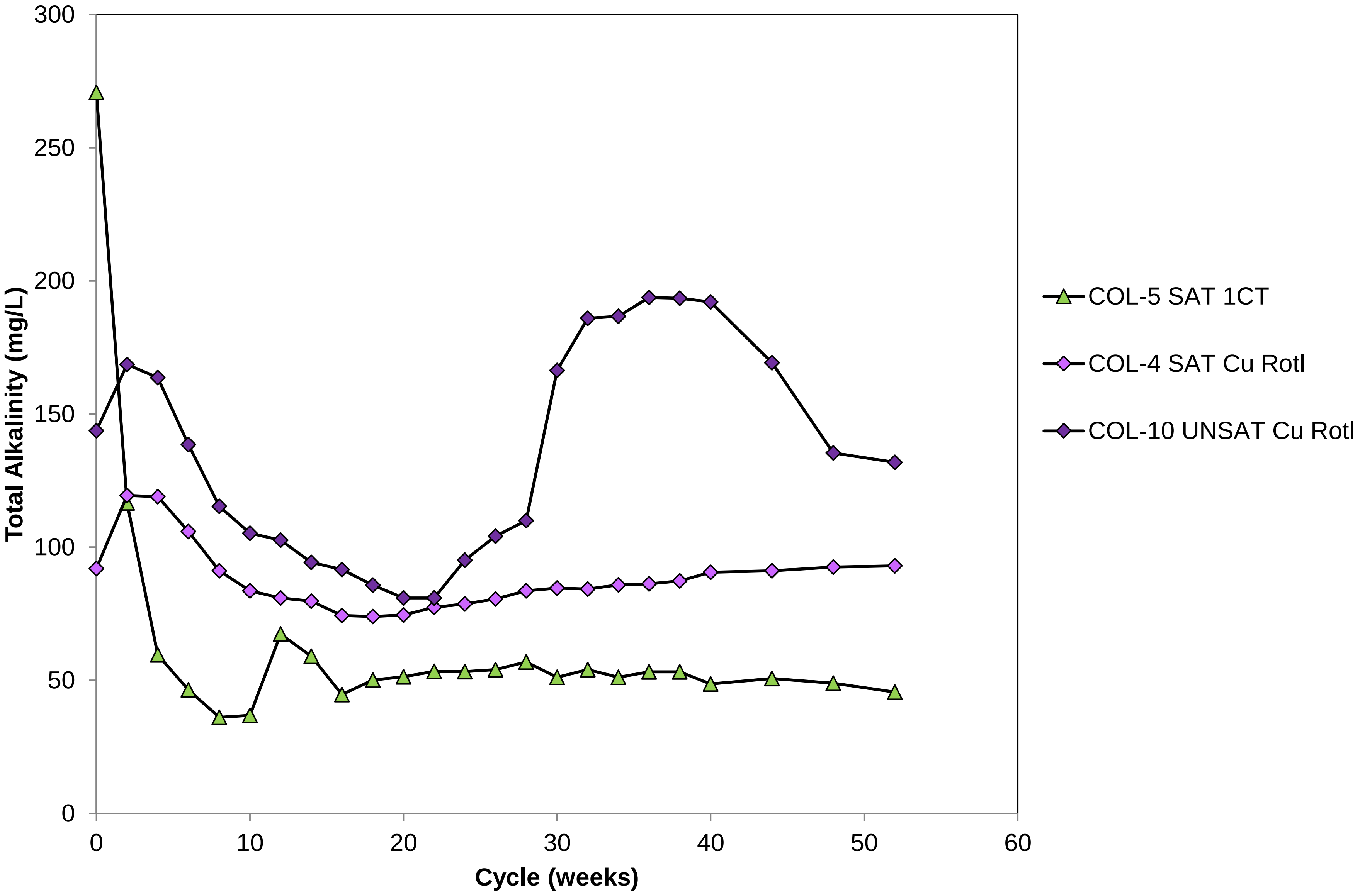

6000
5000
4000
3000
2000
1000
0
EC (uS/cm)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

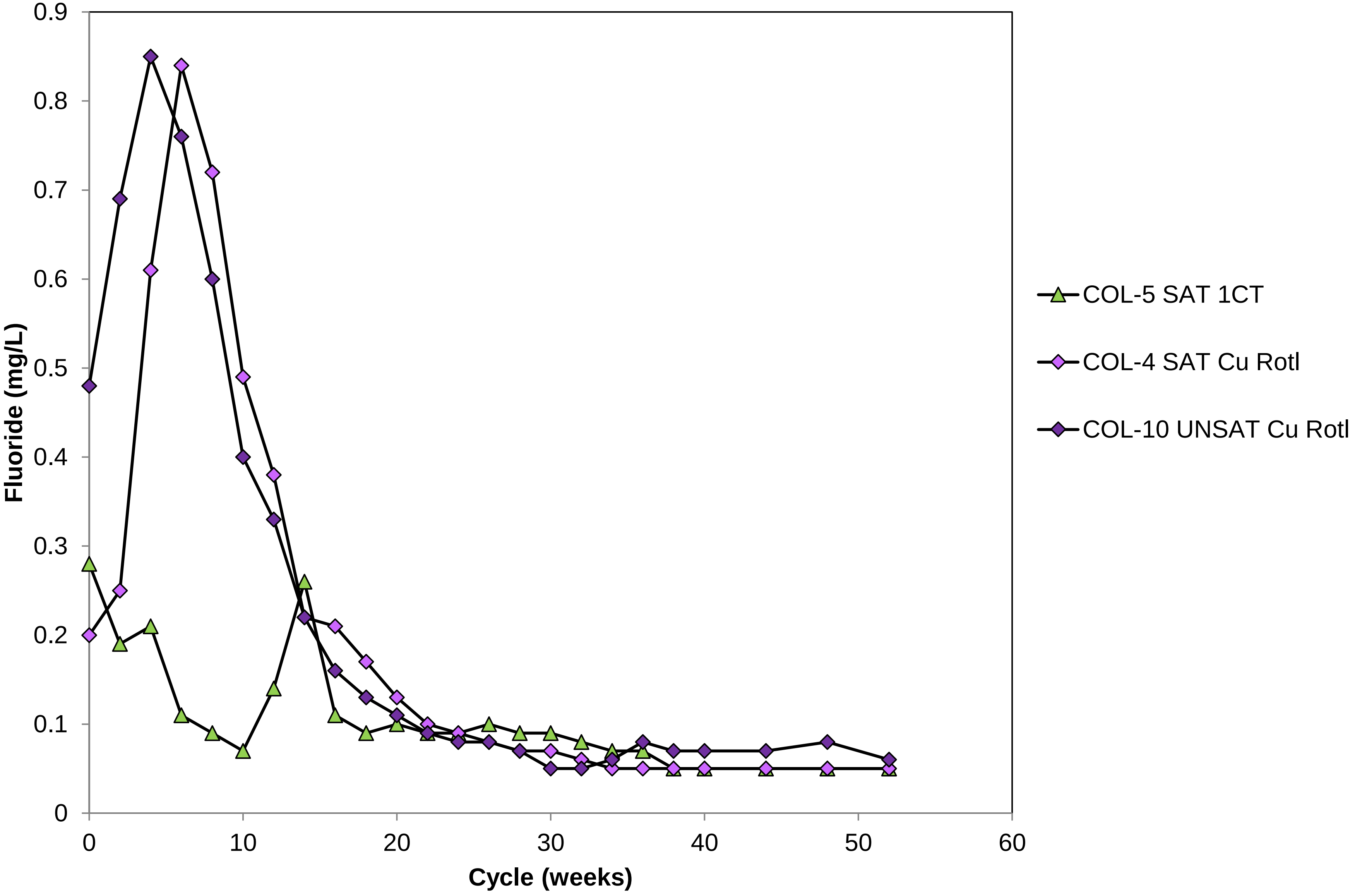

ORP (mV)
0
20
40
60
80
100
120
140
160
Cycle (weeks)
0
10
20
30
40
50
60
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

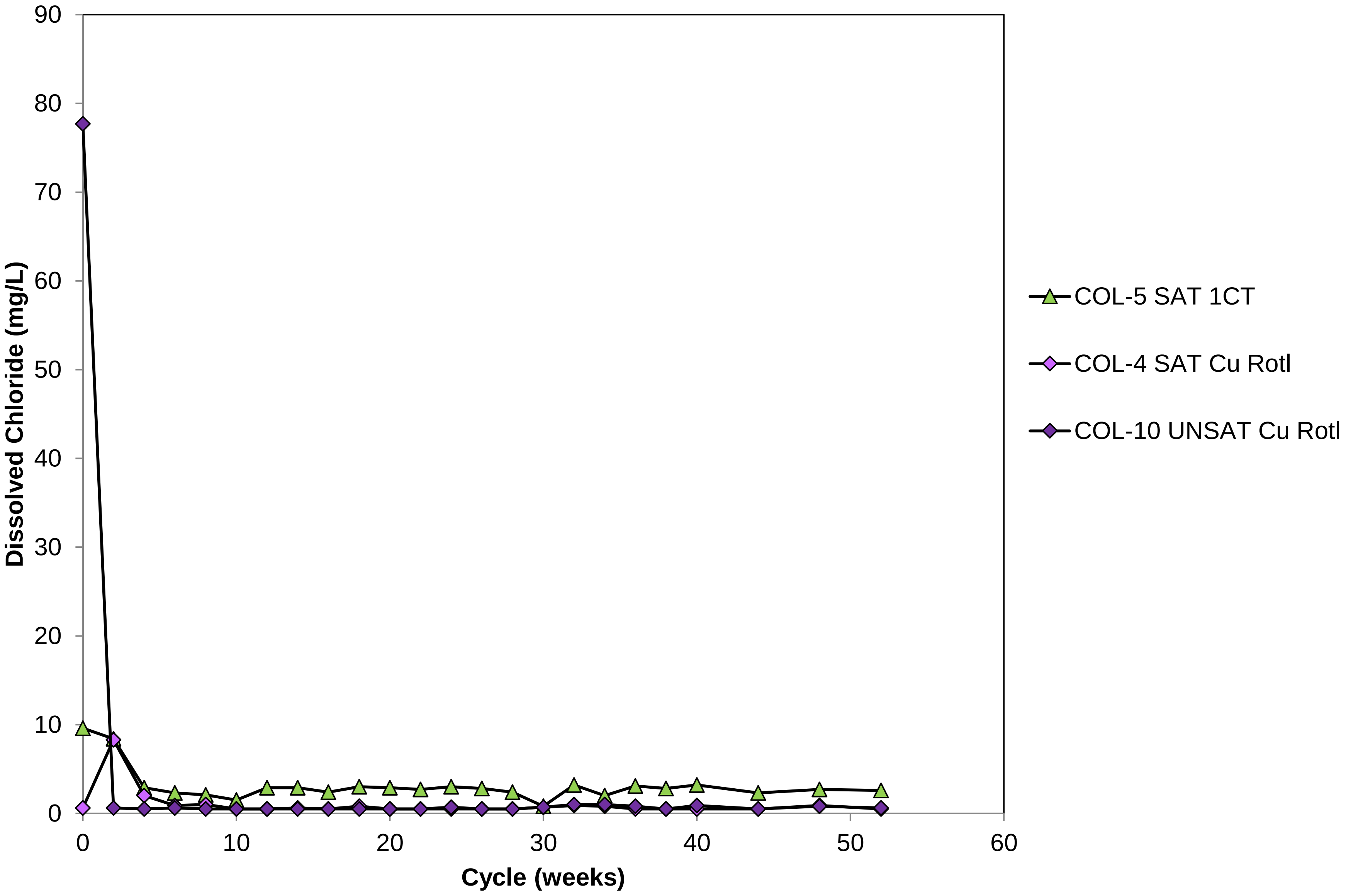

5000
4500
4000
3500
3000
2500
2000
1500
1000
500
0
SO4 (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

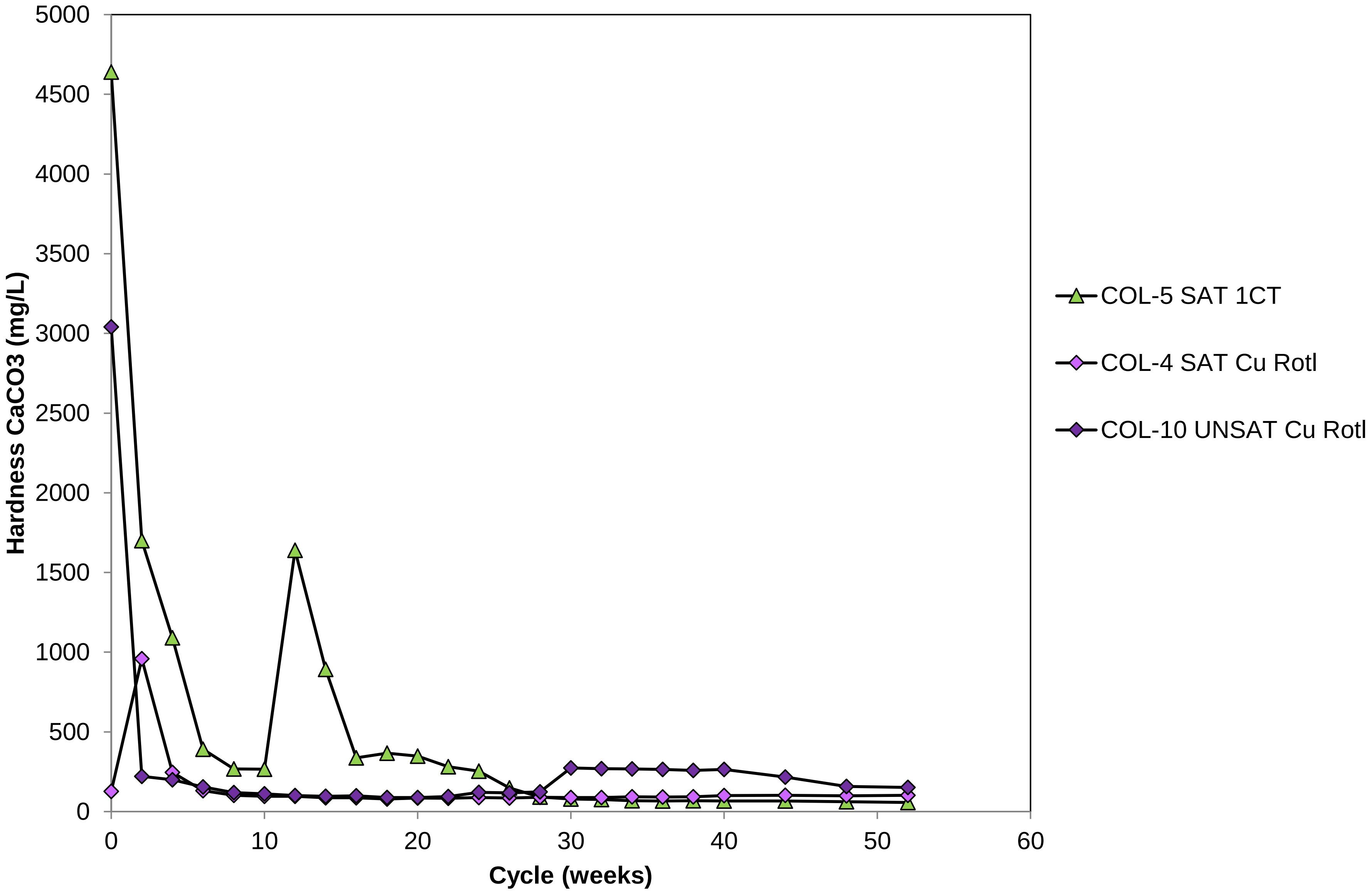

0.6
0.5
0.4
0.3
0.2
0.1
0
Acidity to pH4.5 (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

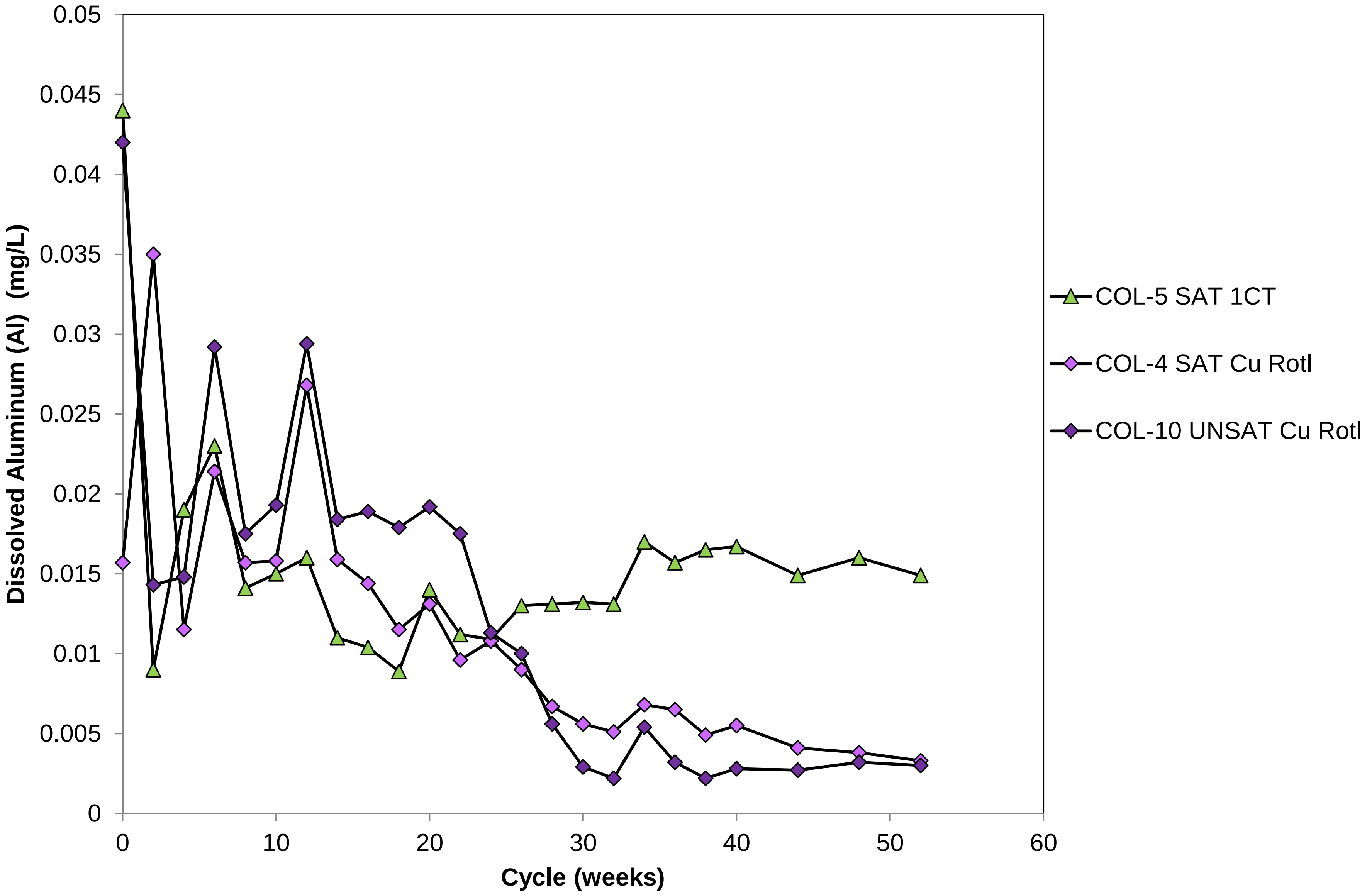

12
10
8
6
4
2
0
Acidity to pH8.3 (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

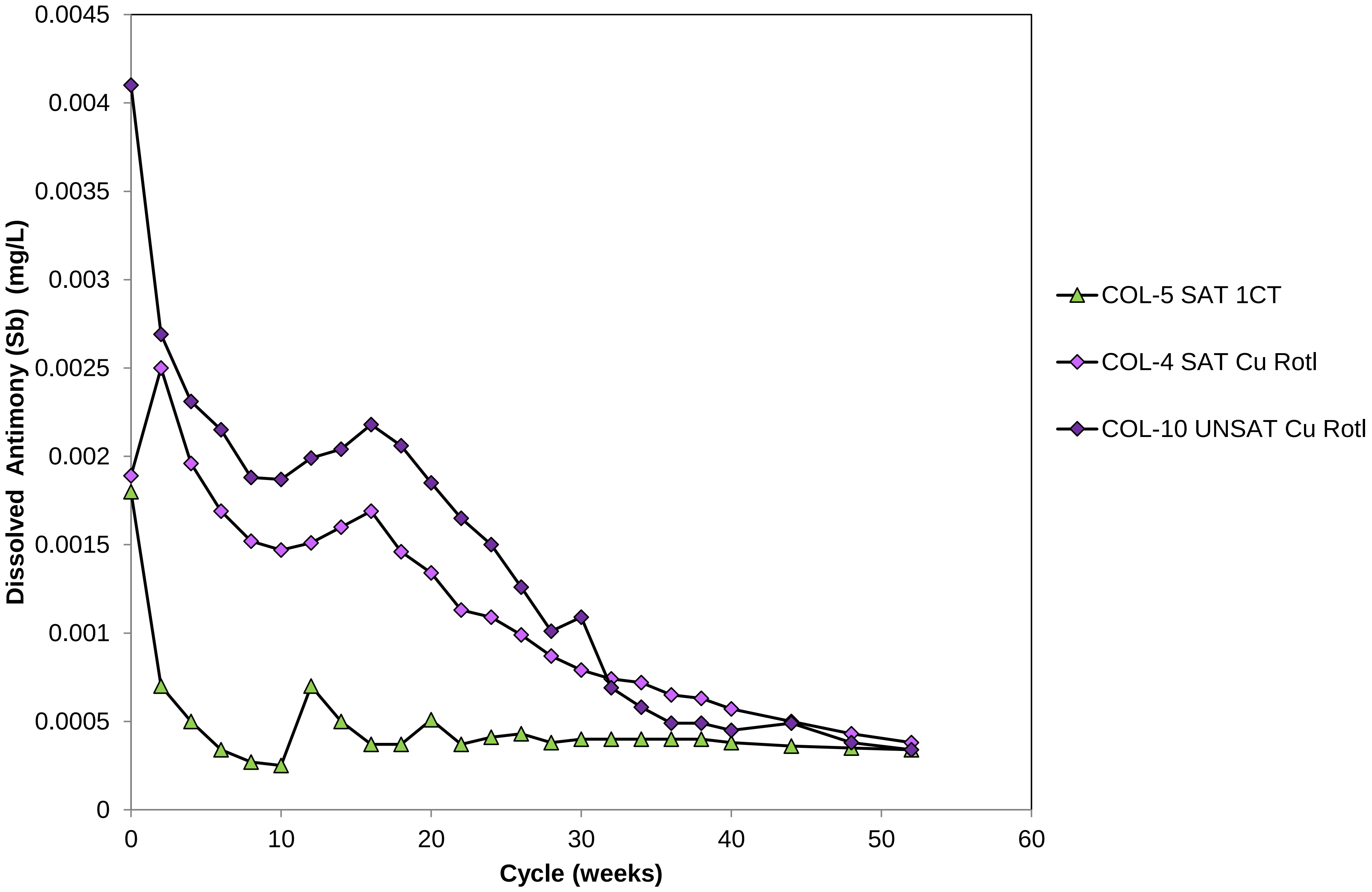

300
250
200
150
100
50
0
0
10
20
30
40
50
60
Total Alkalinity (mg/L)
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

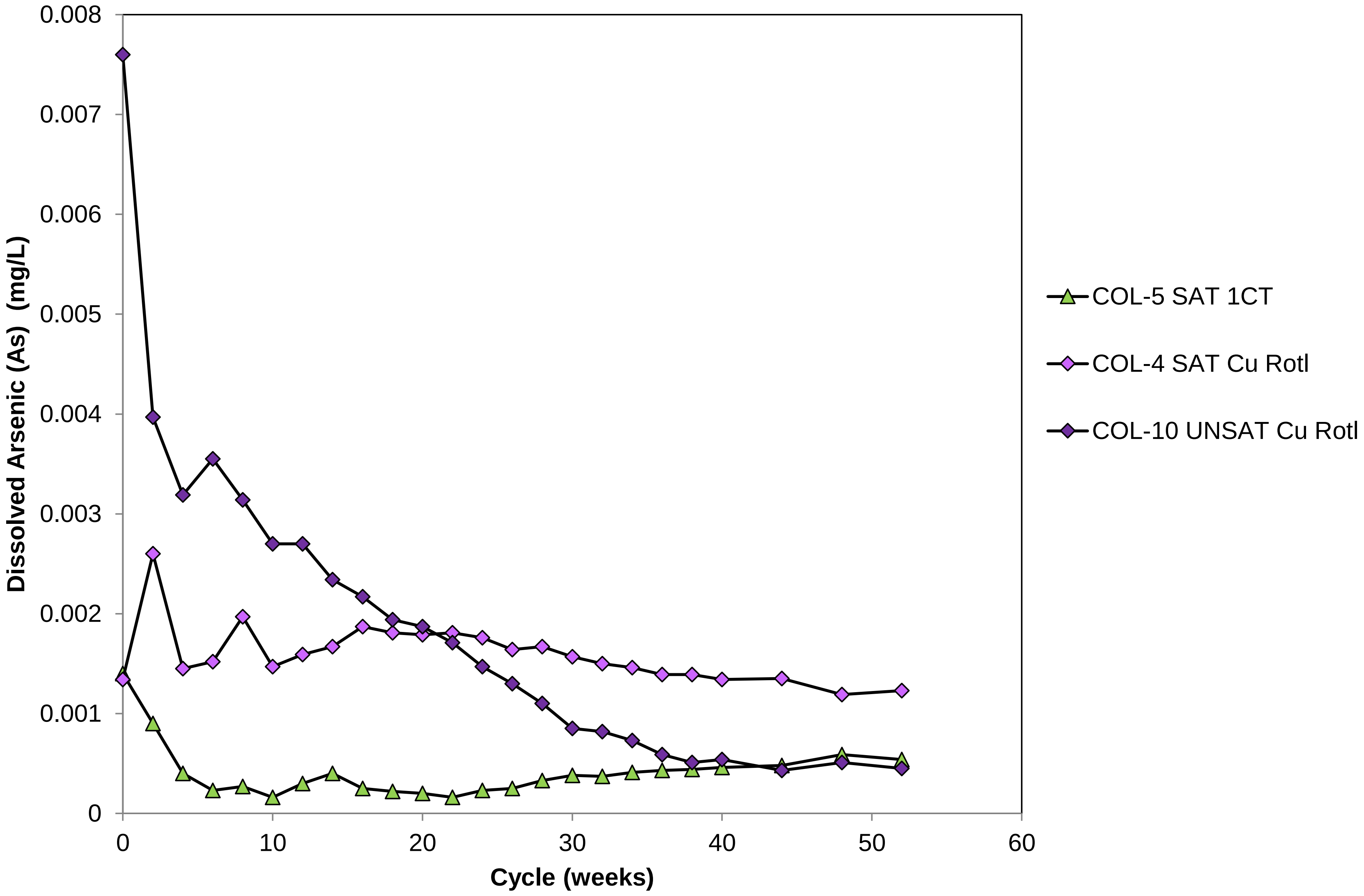

0.9
0.8
0.7
0.6
0.5
0.4
0.3
0.2
0.1
0
Fluoride (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

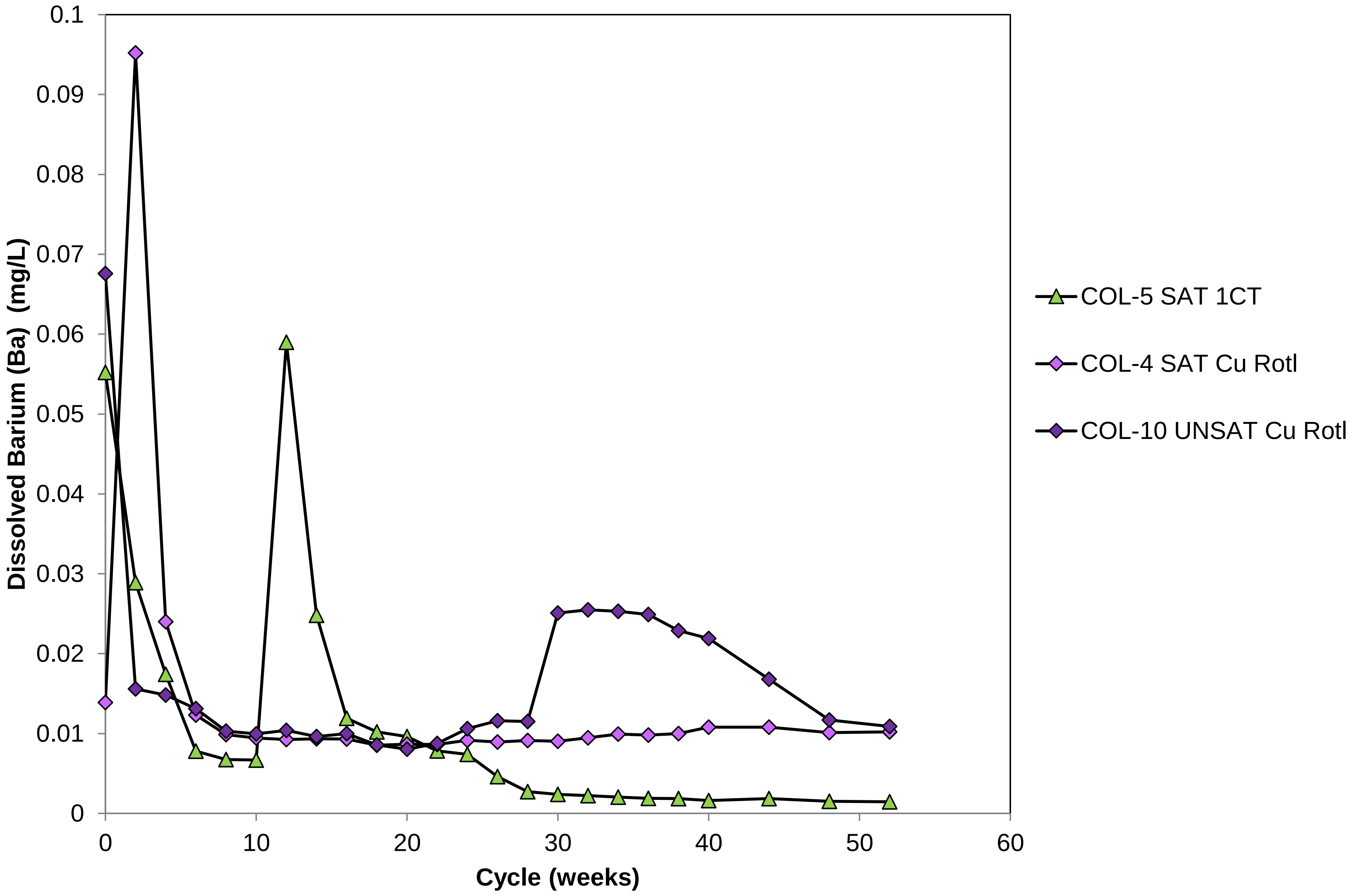

90
80
70
60
50
40
30
20
10
0
Dissolved Chloride (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

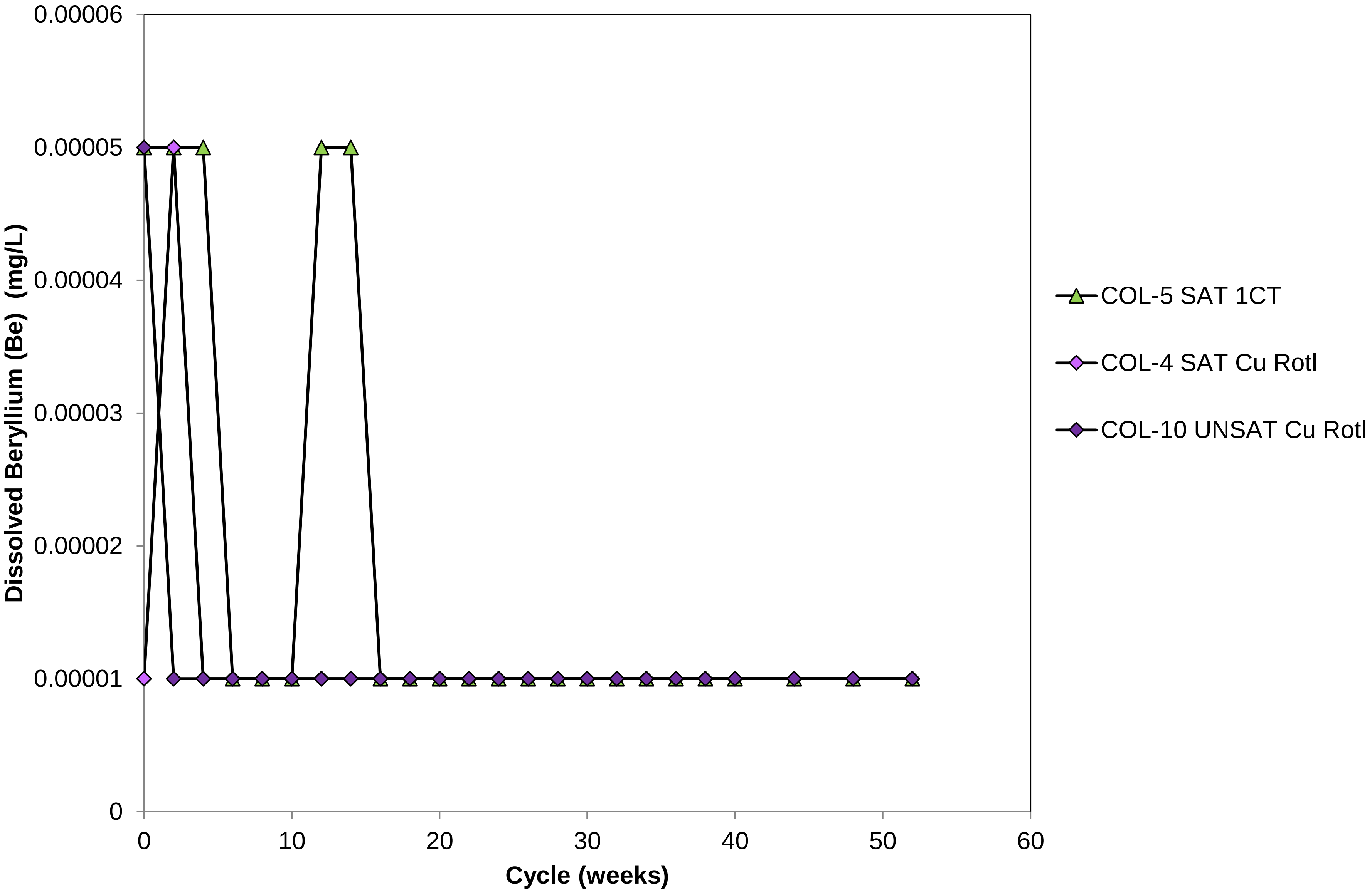

5000
4500
4000
3500
3000
2500
2000
1500
1000
500
0
Hardness CaCO3 (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

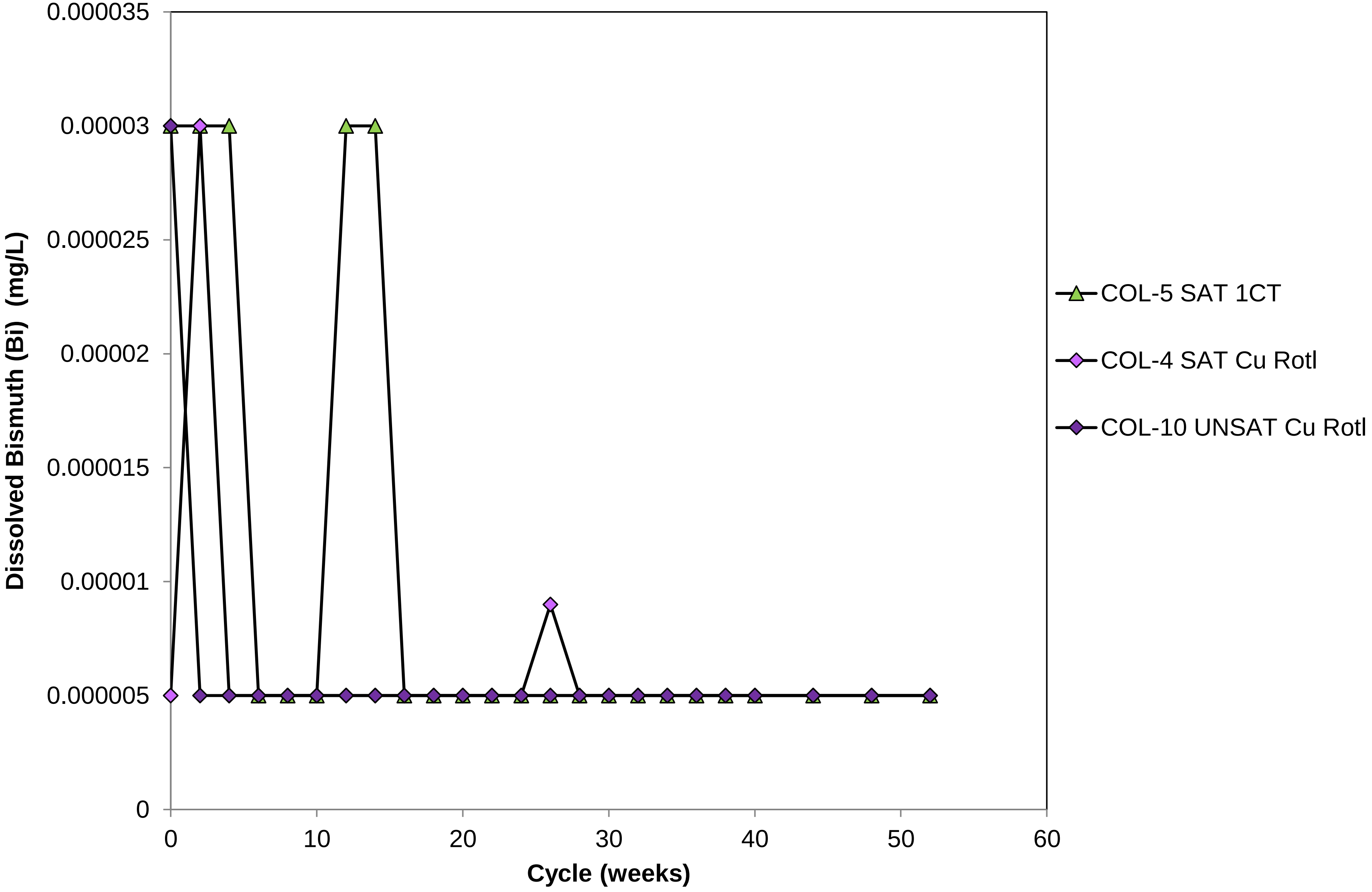

0.05
0.045
0.04
0.035
0.03
0.025
0.02
0.015
0.01
0.005
0
Dissolved Aluminum (Al) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

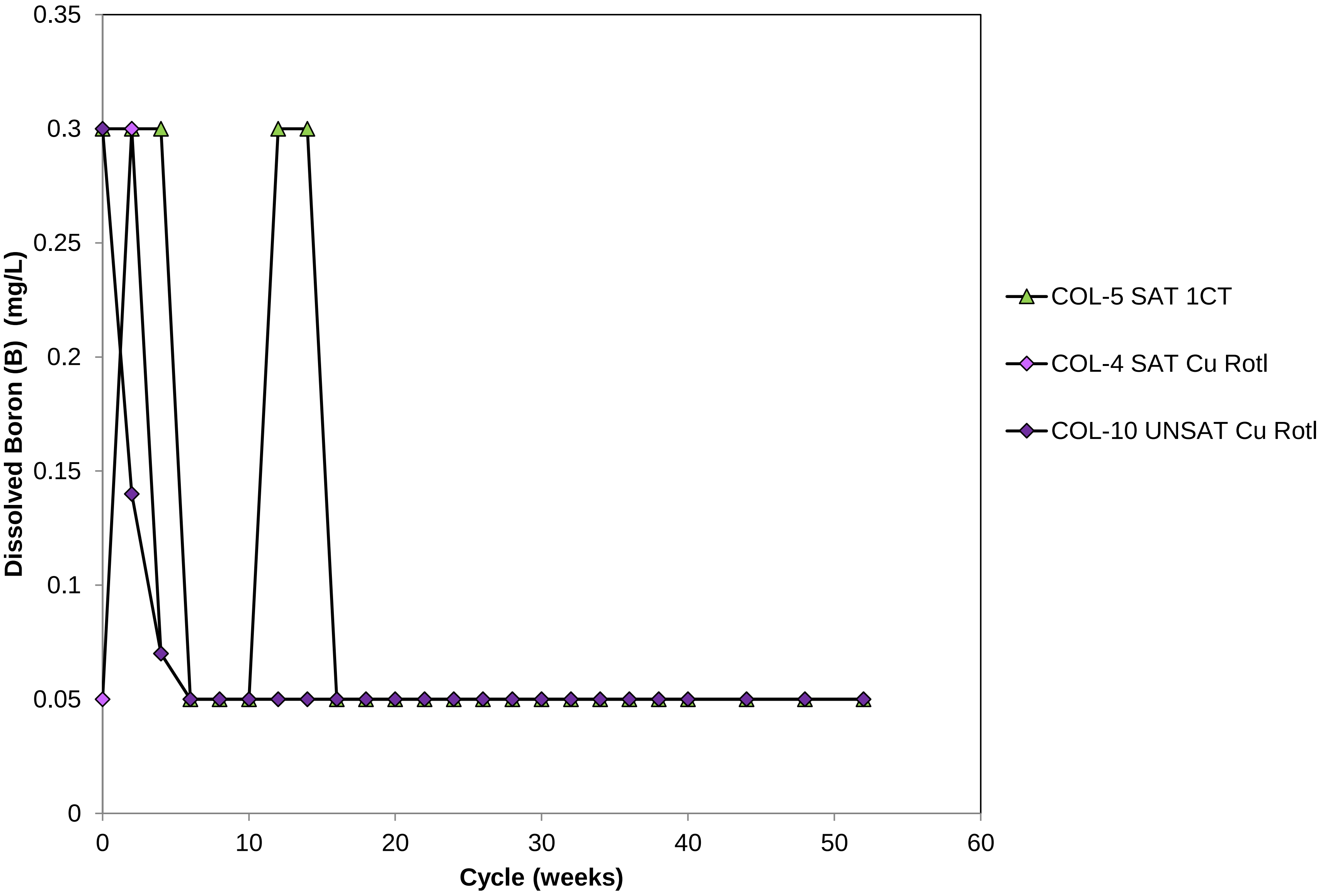

0.0045
0.004
0.0035
0.003
0.0025
0.002
0.0015
0.001
0.0005
0
Dissolved Antimony (Sb) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

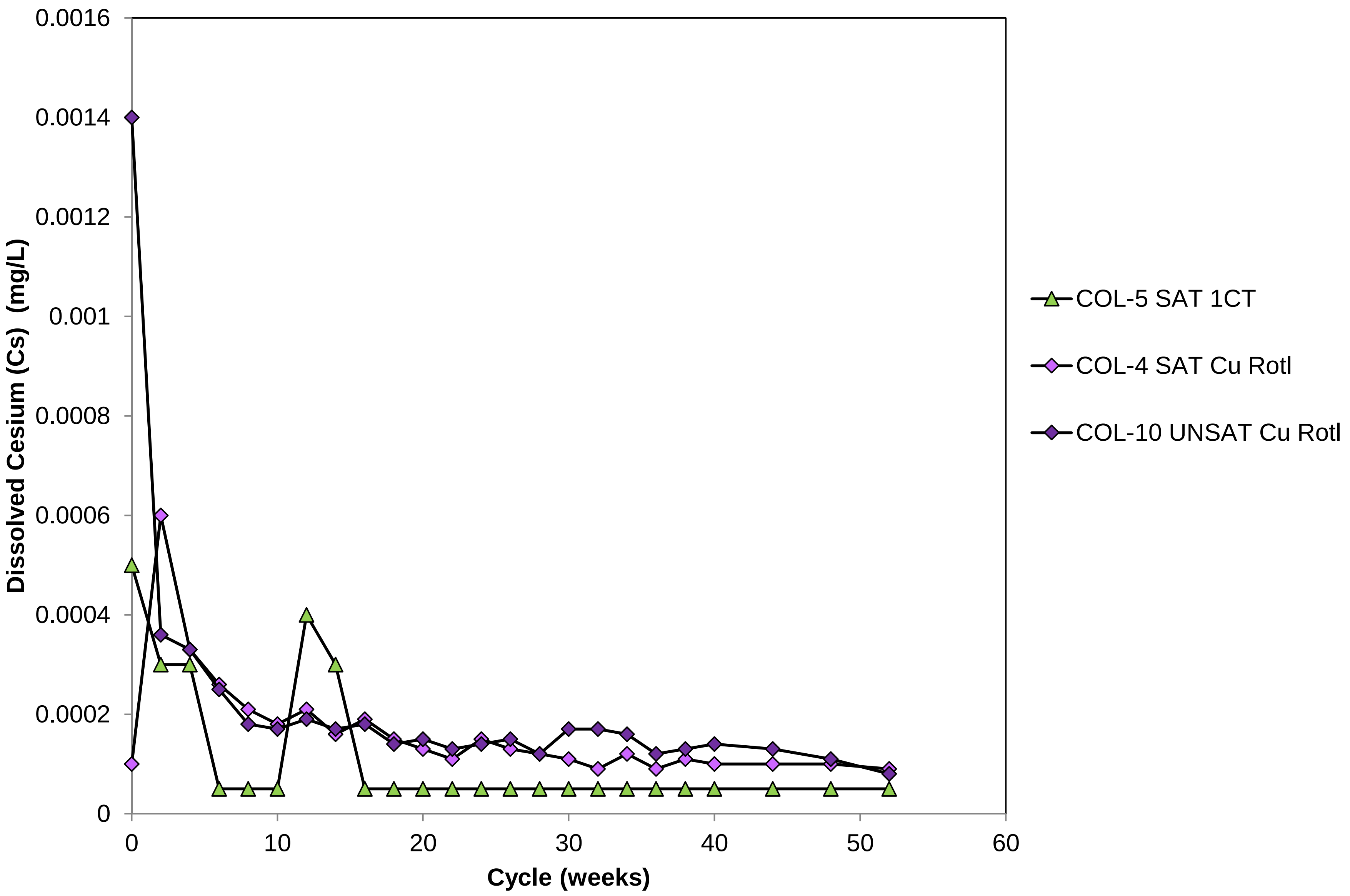

0.008
0.007
0.006
0.005
0.004
0.003
0.002
0.001
0
Dissolved Arsenic (As) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

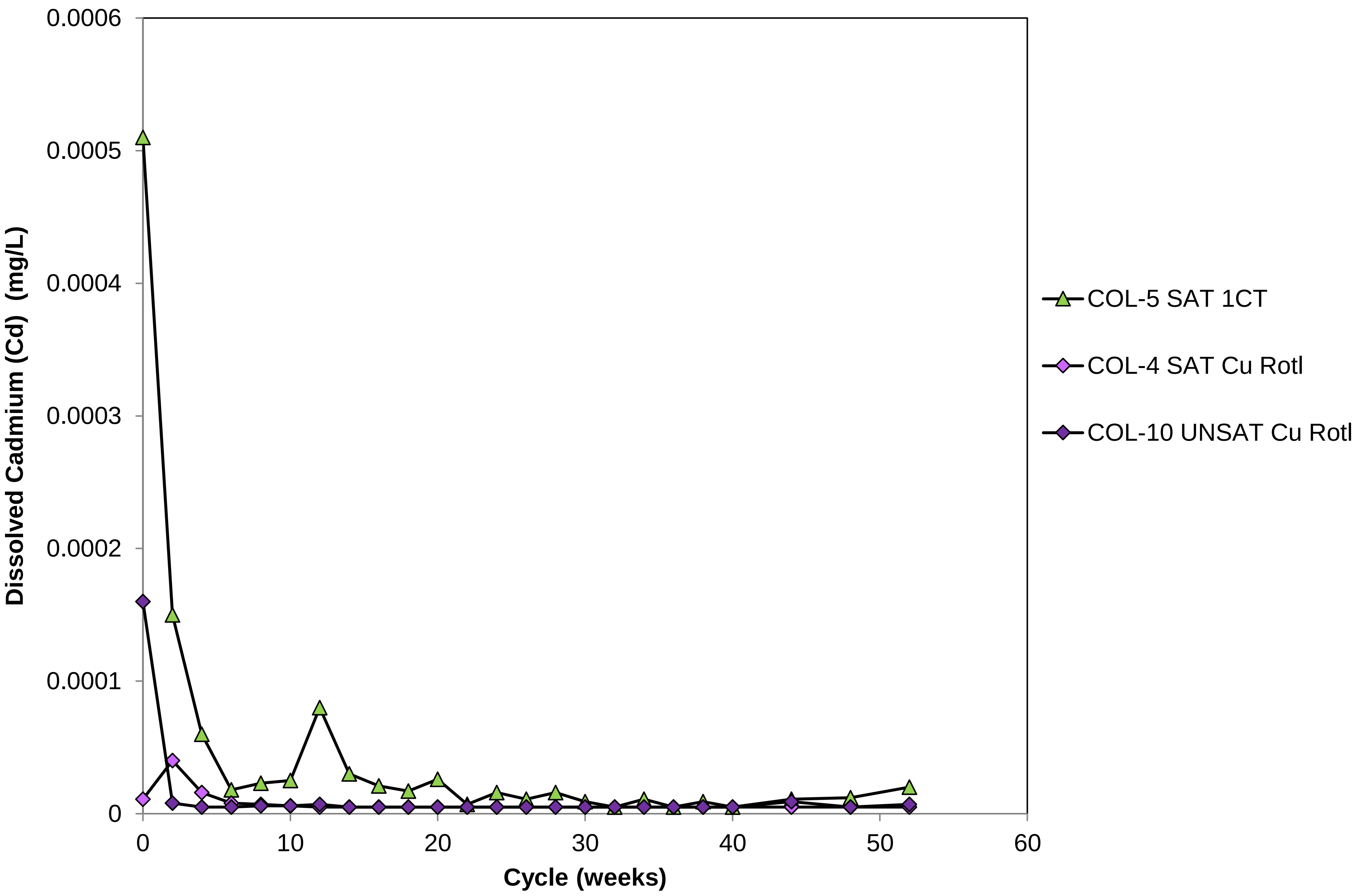

Dissolved Barium (Ba) (mg/L)
0.1
0.09
0.08
0.07
0.06
0.05
0.04
0.03
0.02
0.01
0
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

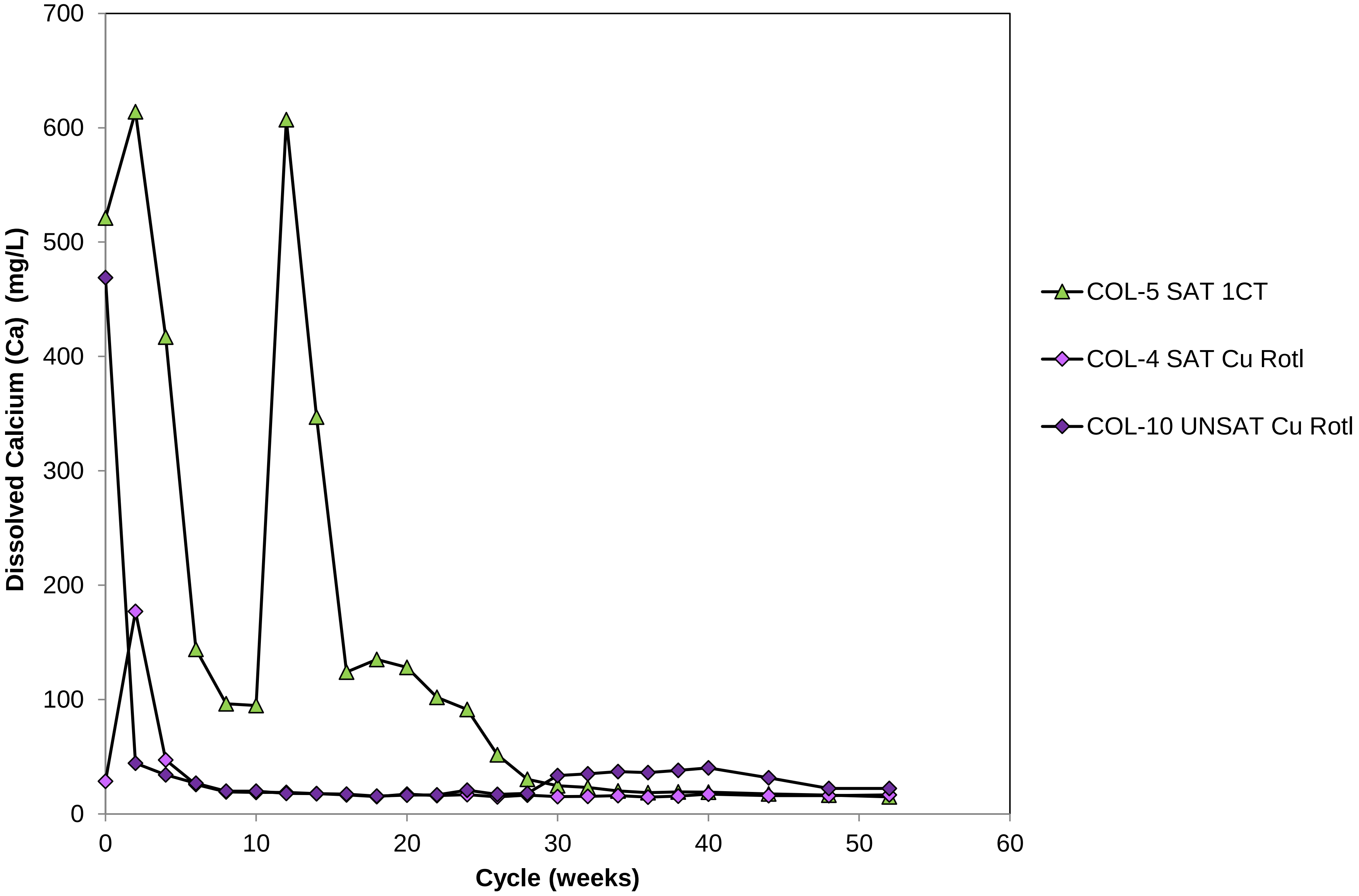

0.00006
0.00005
0.00004
0.00003
0.00002
0.00001
0
Dissolved Beryllium (Be) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

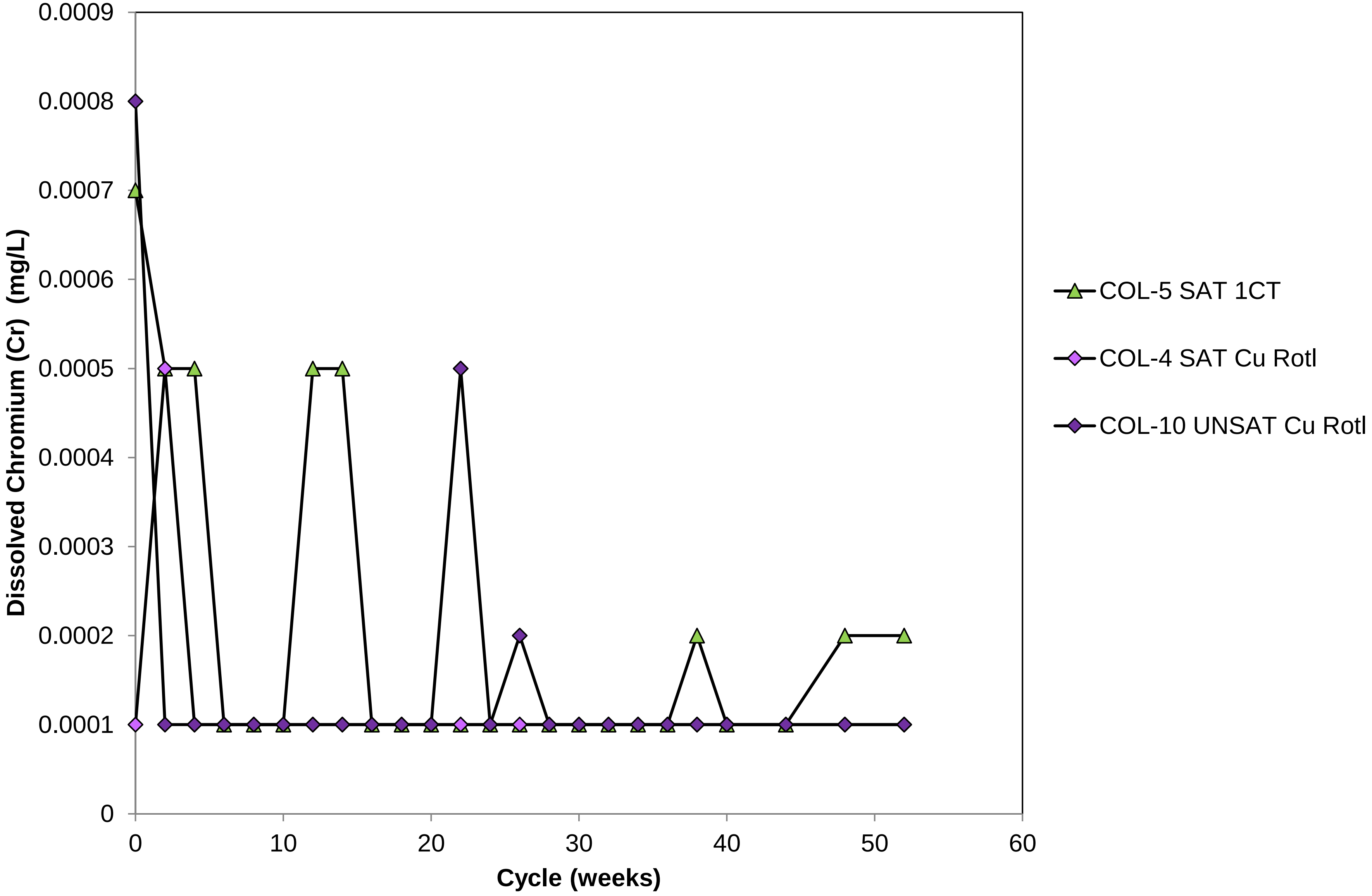

Dissolved Bismuth (Bi) (mg/L)
0.000035
0.00003
0.000025
0.00002
0.000015
0.00001
0.000005
0
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

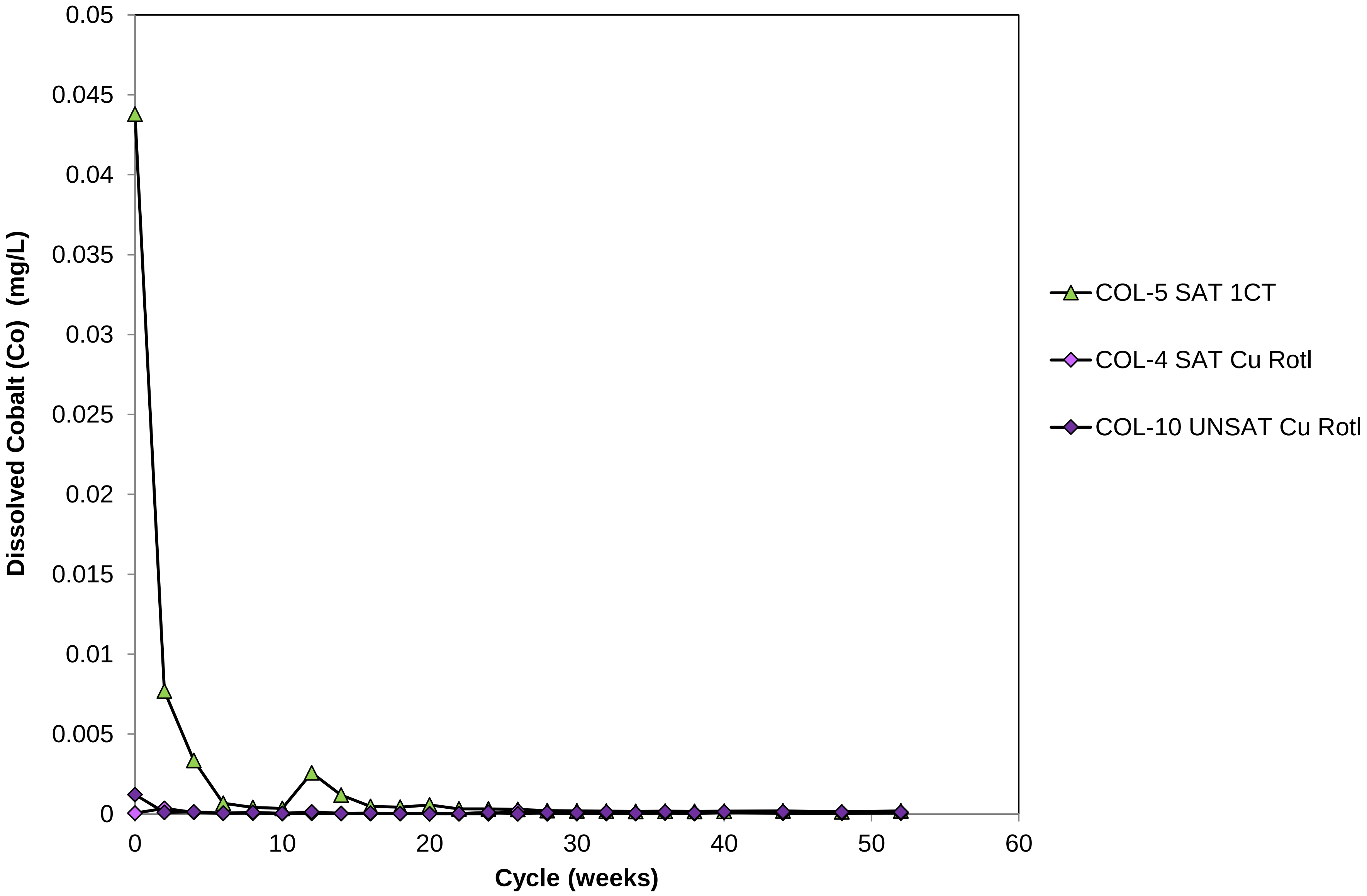

0.35
0.3
0.25
0.2
0.15
0.1
0.05
0
Dissolved Boron (B) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

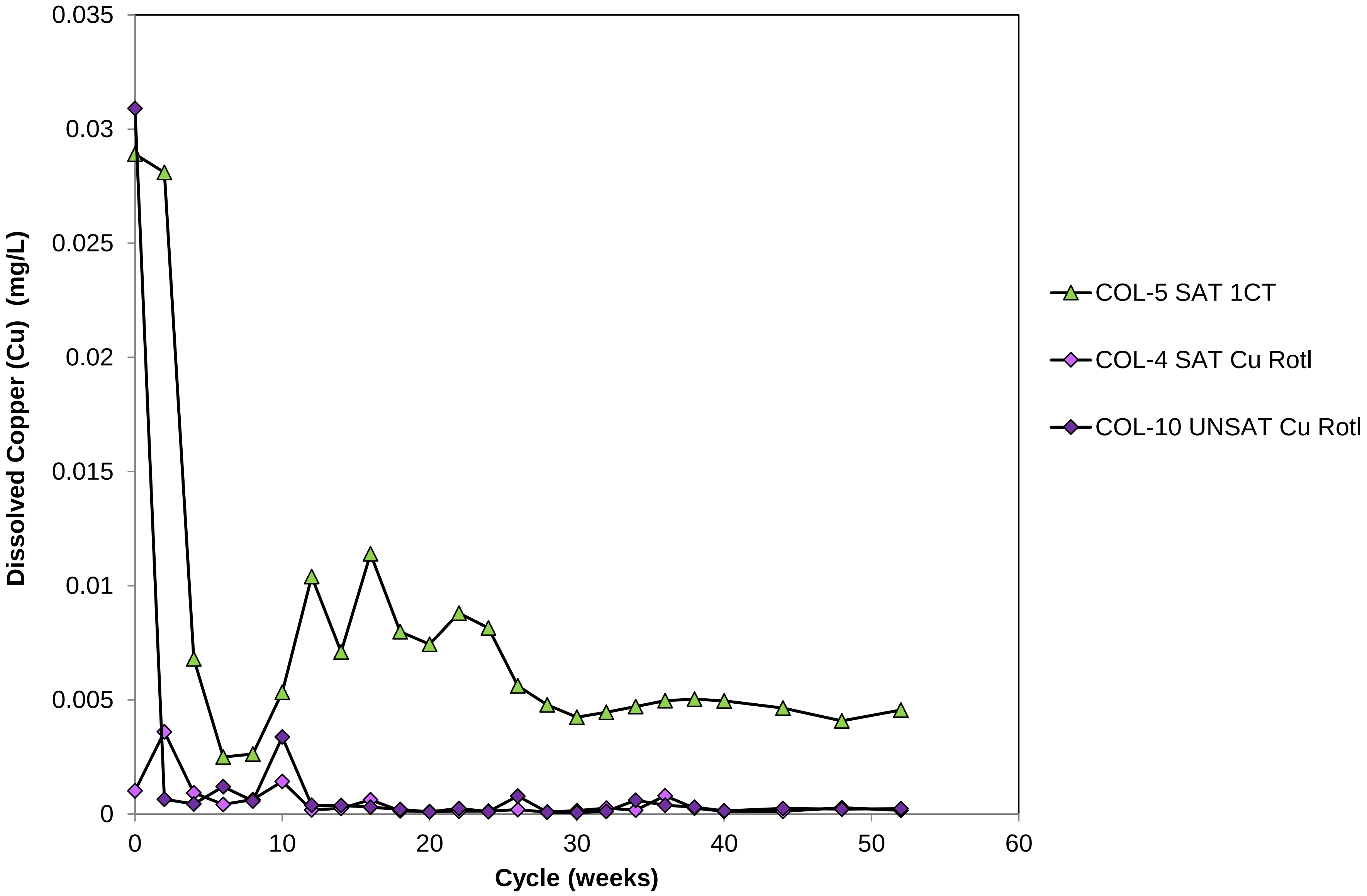

0.0016
0.0014
0.0012
0.001
0.0008
0.0006
0.0004
0.0002
0
Dissolved Cesium (Cs) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

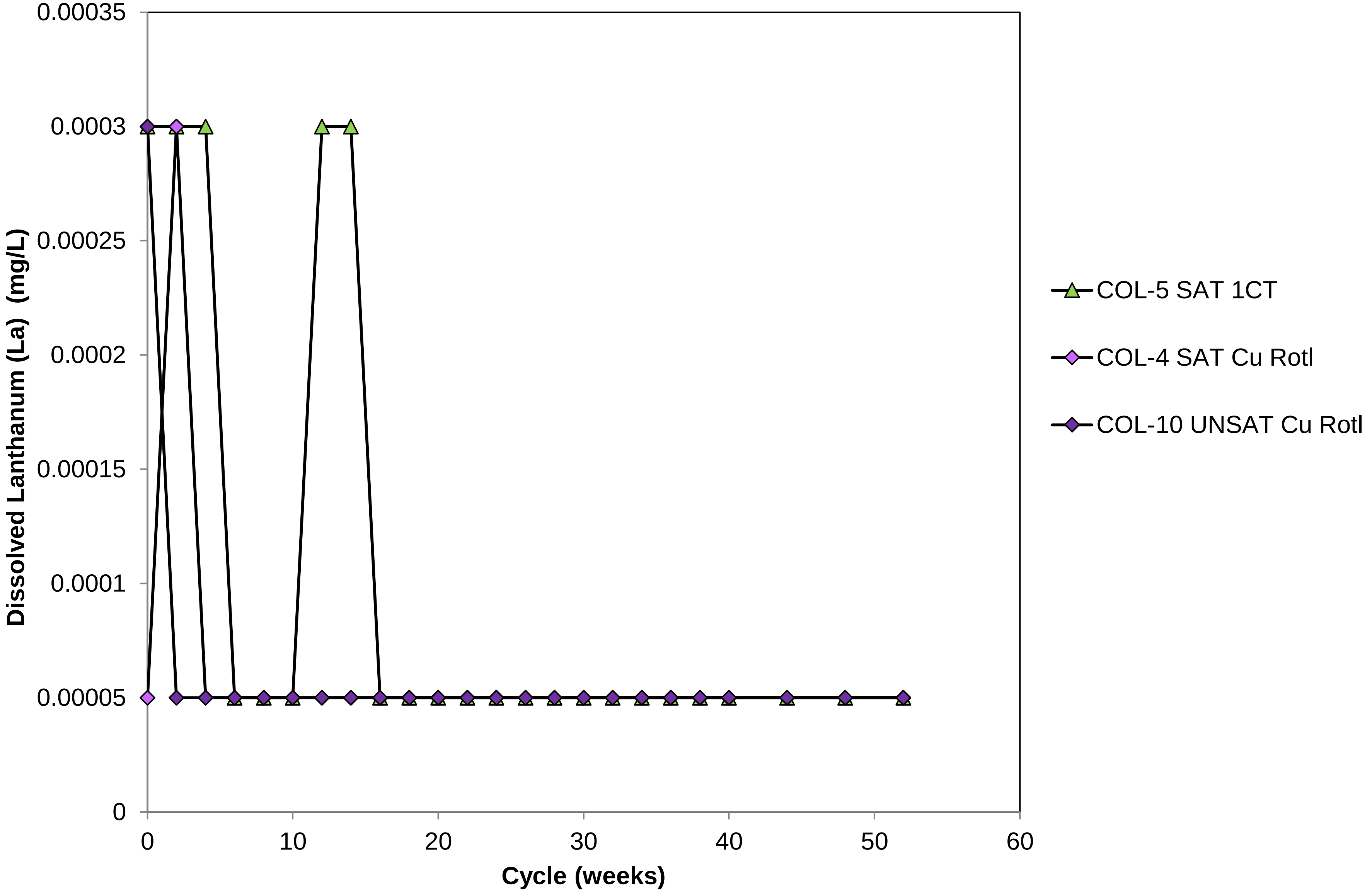

0.0006
0.0005
0.0004
0.0003
0.0002
0.0001
0
Dissolved Cadmium (Cd) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

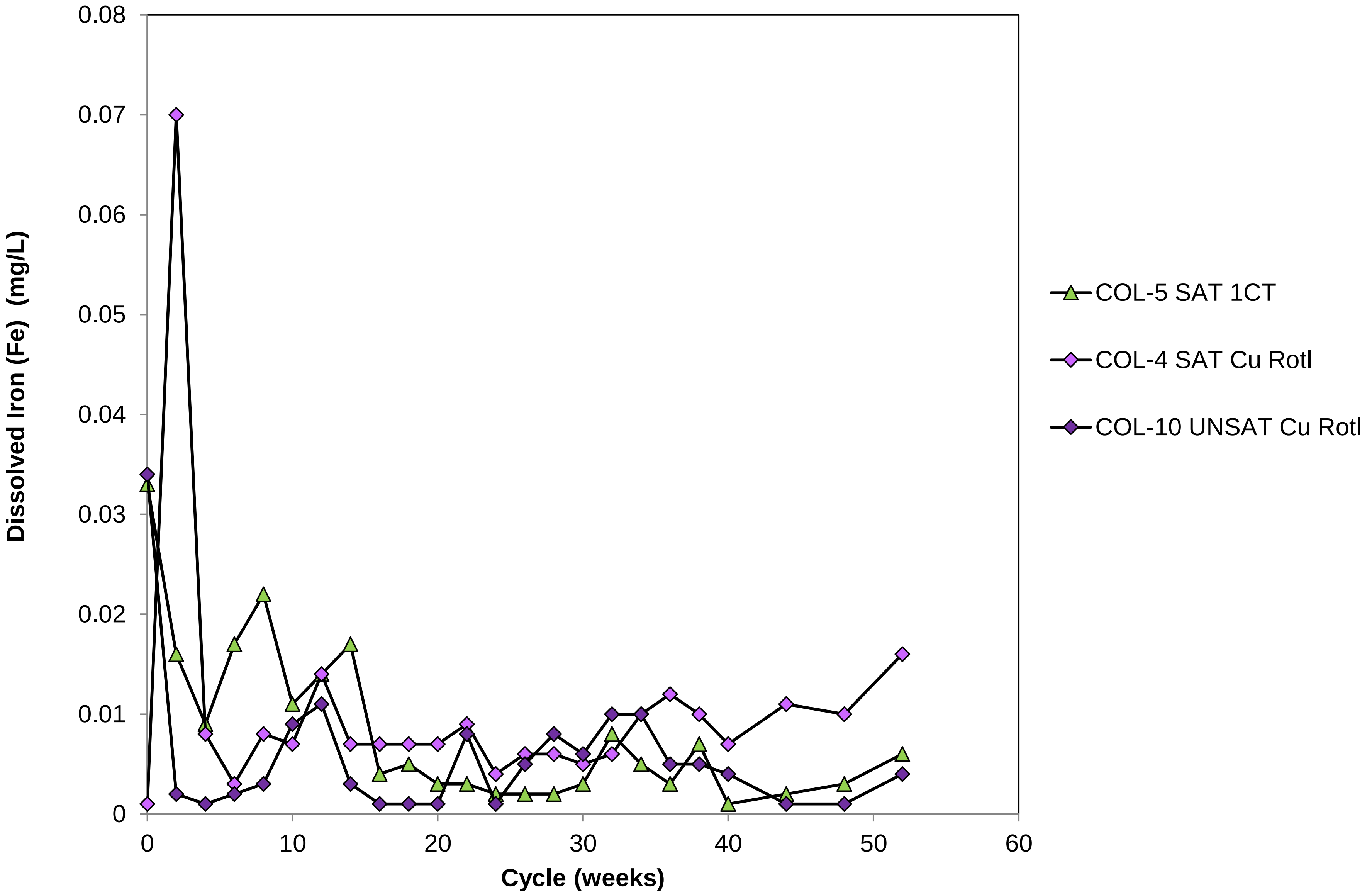

Dissolved Calcium (Ca) (mg/L)
700
600
500
400
300
200
100
0
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

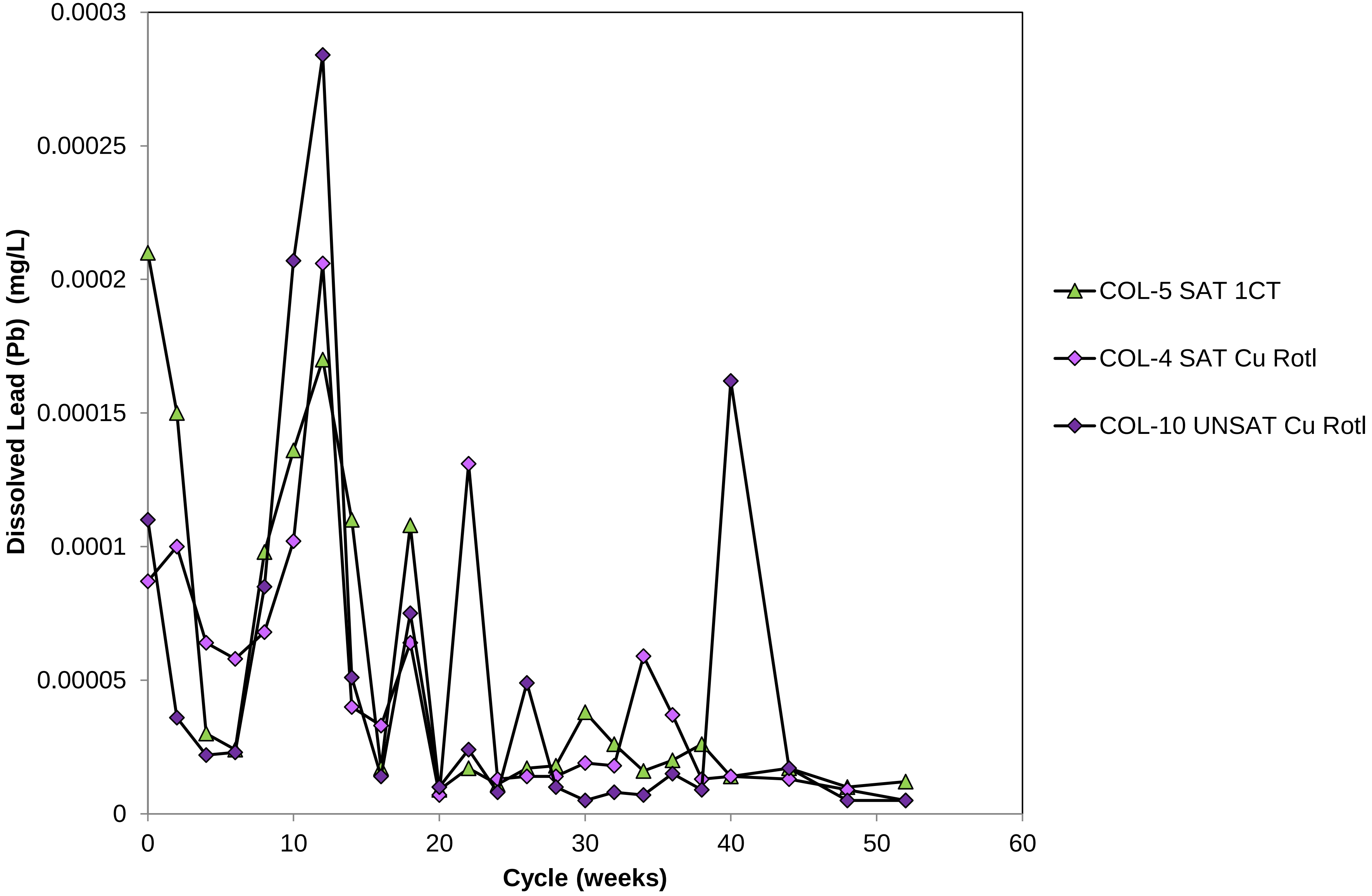

0.0009
0.0008
0.0007
0.0006
0.0005
0.0004
0.0003
0.0002
0.0001
0
Dissolved Chromium (Cr) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

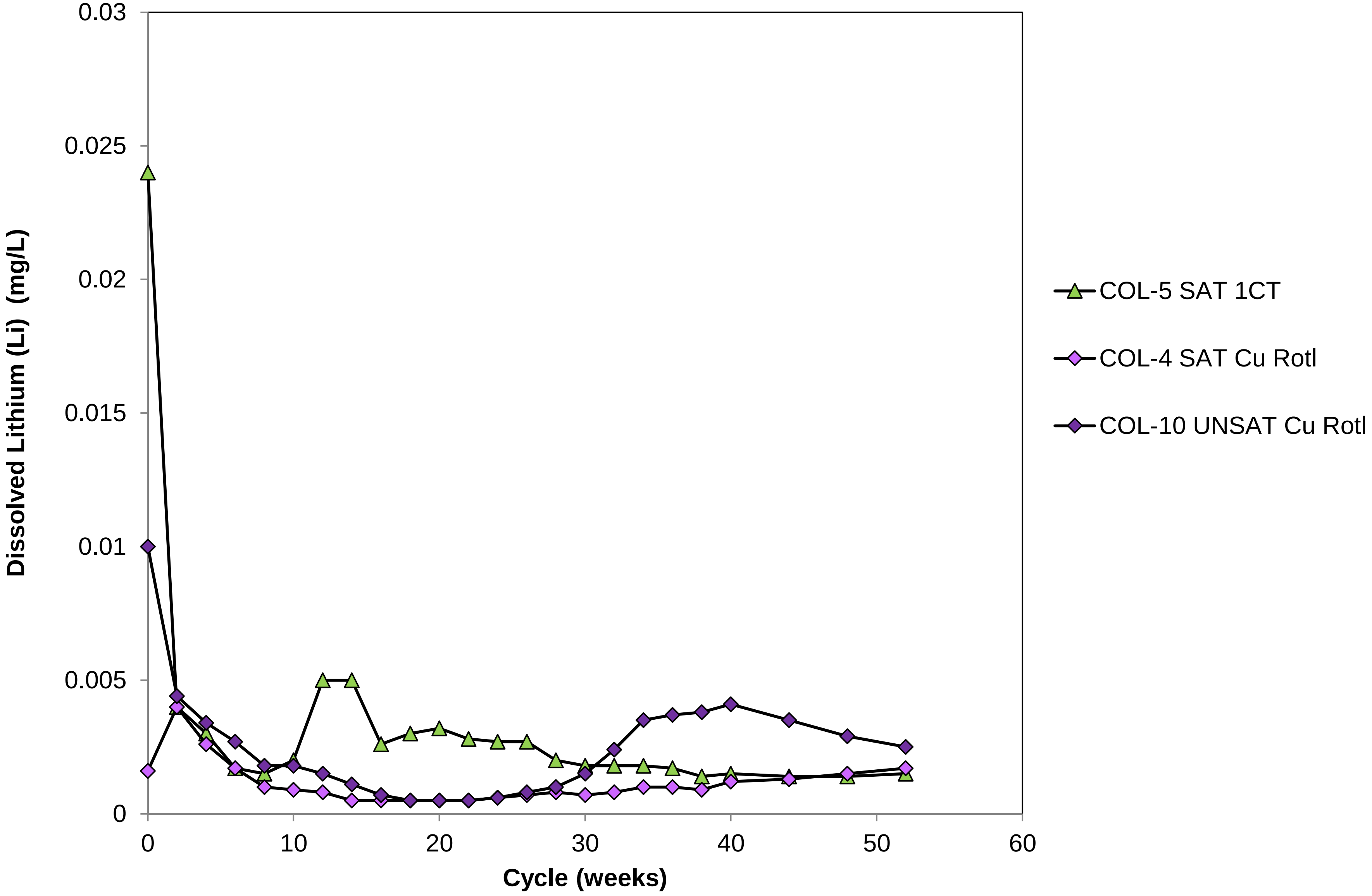

0.05
0.045
0.04
0.035
0.03
0.025
0.02
0.015
0.01
0.005
0
Dissolved Cobalt (Co) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

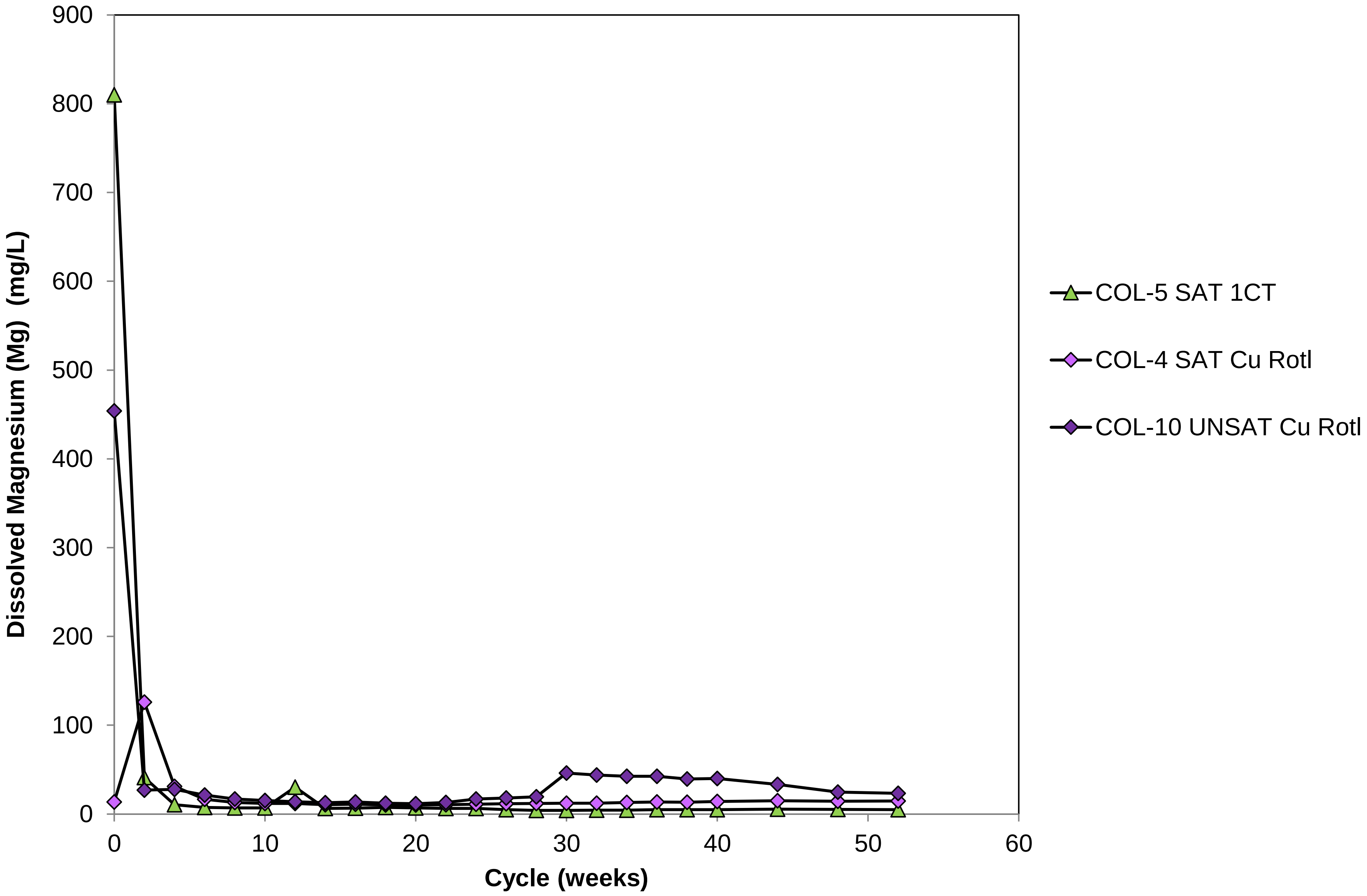

0.035
0.03
0.025
0.02
0.015
0.01
0.005
0
Dissolved Copper (Cu) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

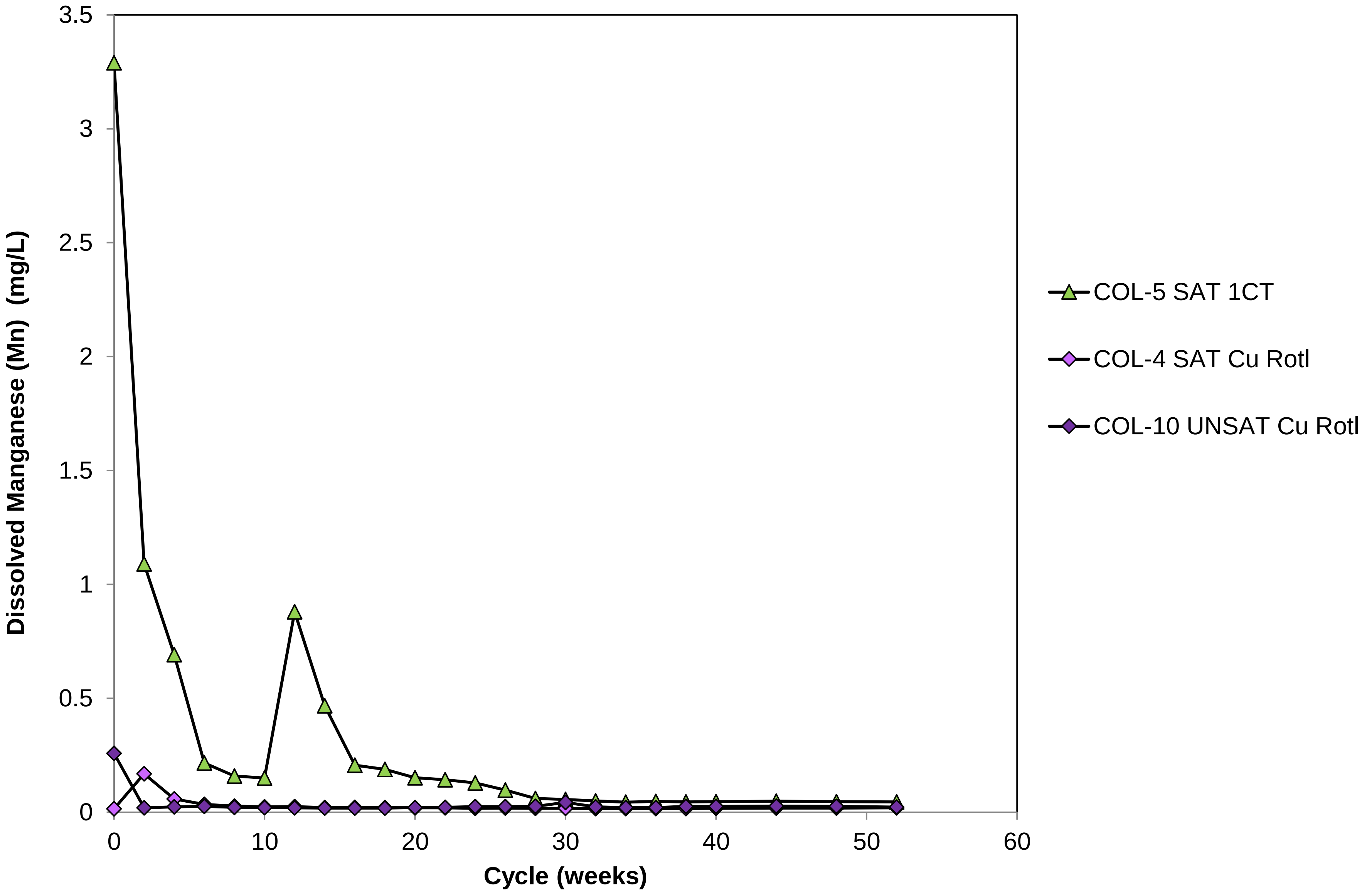

0.00035
0.0003
0.00025
0.0002
0.00015
0.0001
0.00005
0
Dissolved Lanthanum (La) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

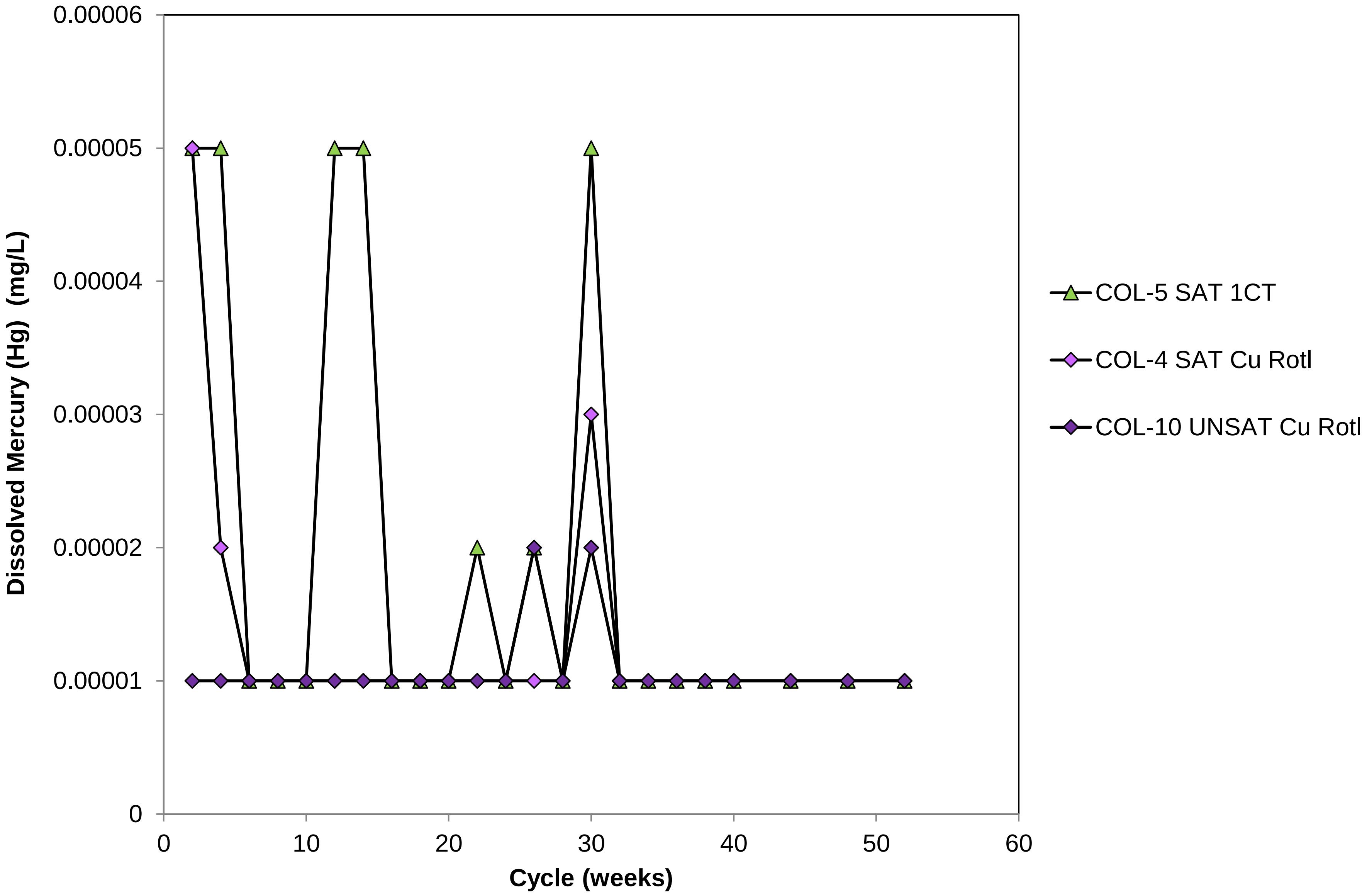

0.08
0.07
0.06
0.05
0.04
0.03
0.02
0.01
0
Dissolved Iron (Fe) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

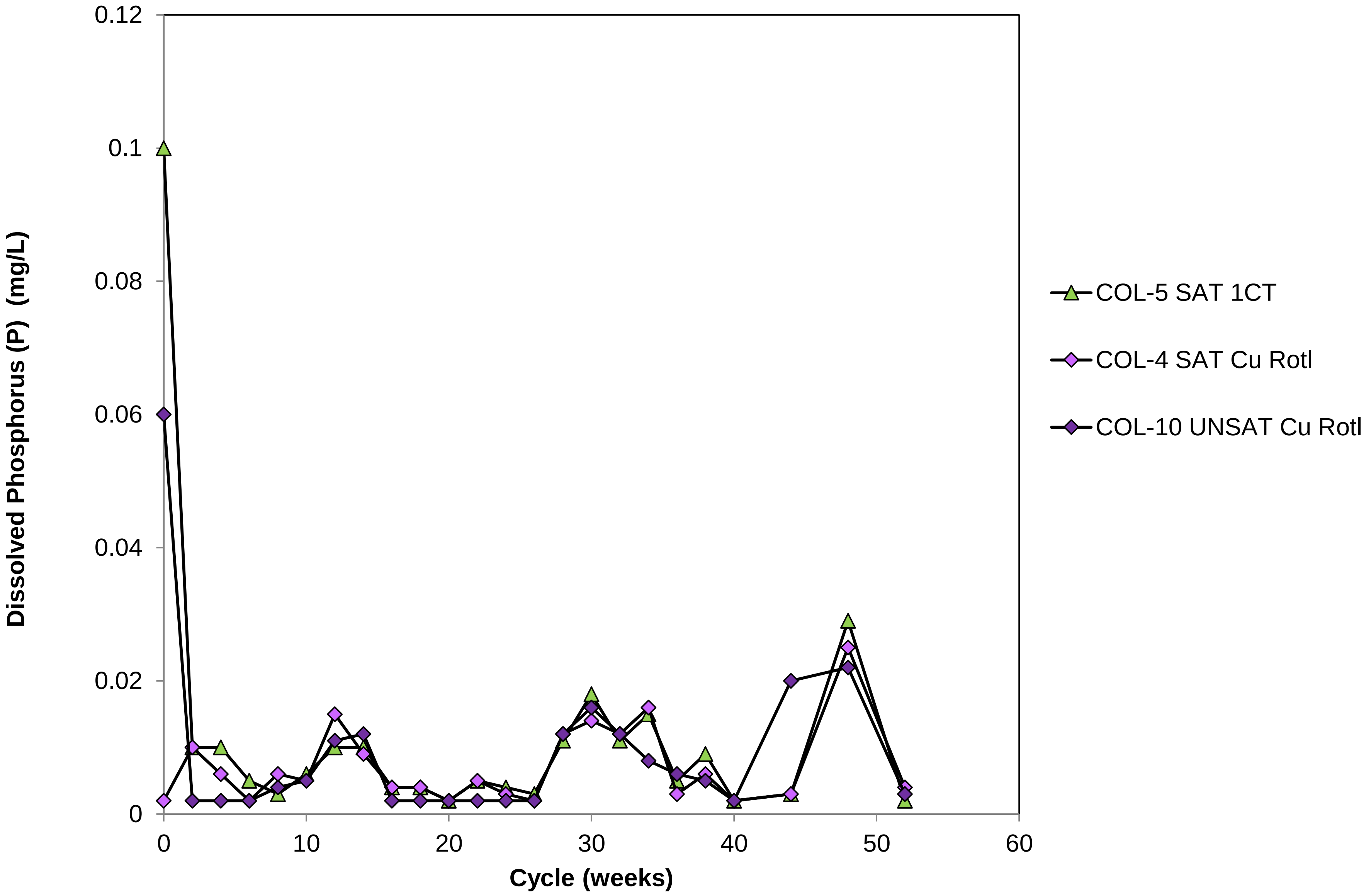

0.0003
0.00025
0.0002
0.00015
0.0001
0.00005
0
Dissolved Lead (Pb) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

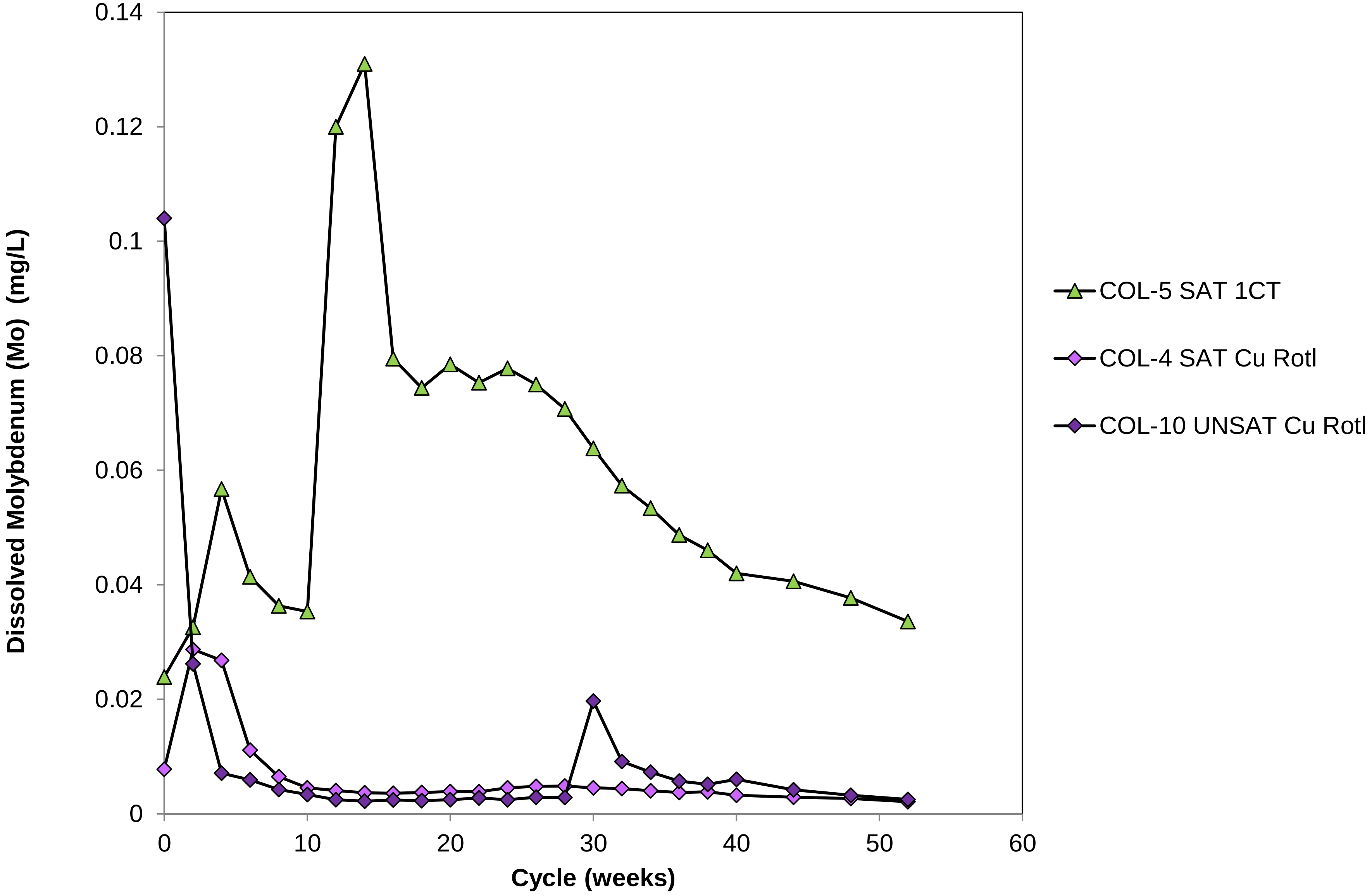

0.03
0.025
0.02
0.015
0.01
0.005
0
0
10
20
30
40
50
60
Dissolved Lithium (Li) (mg/L)
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

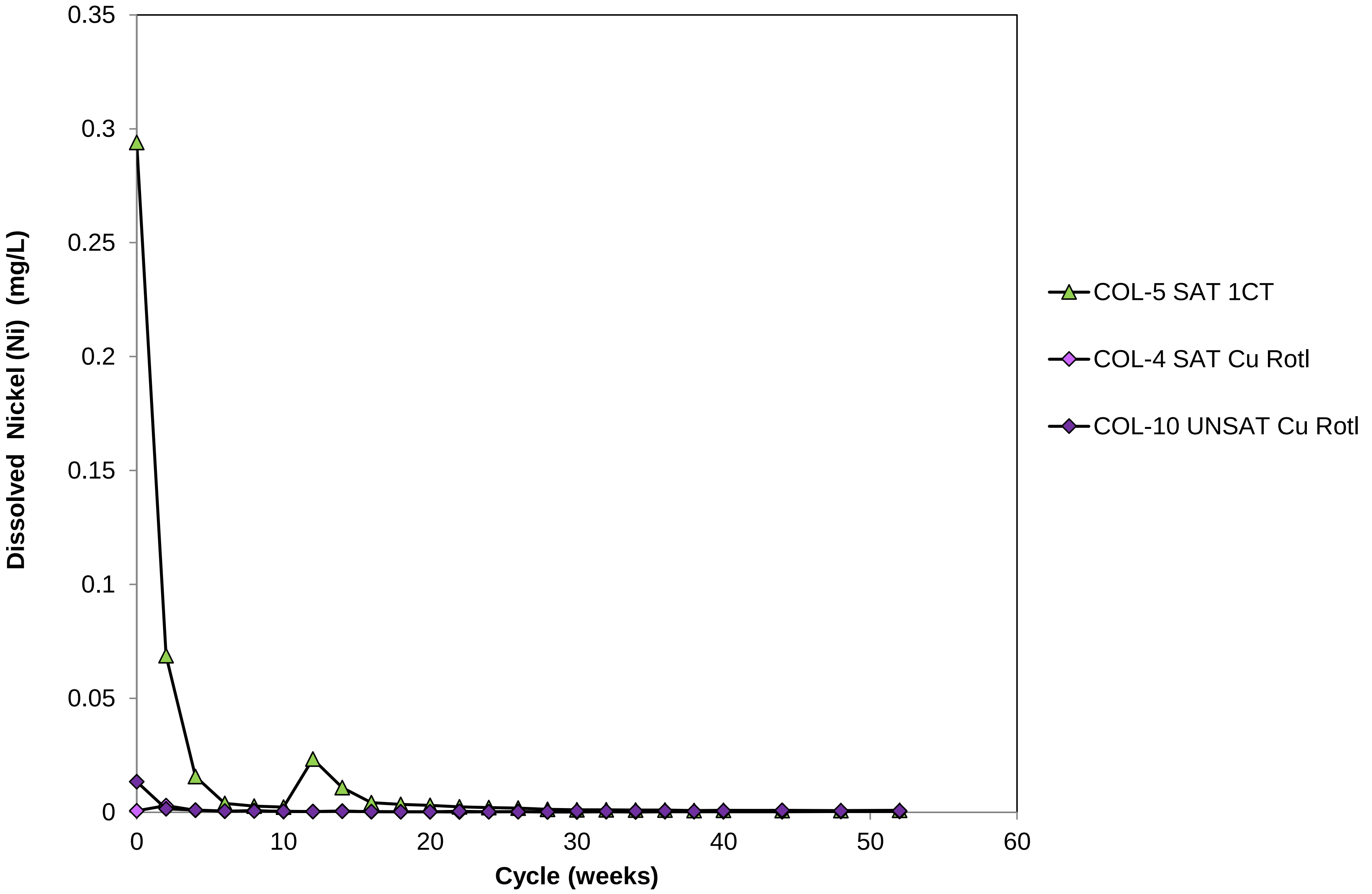

Dissolved Magnesium (Mg) (mg/L)
900
800
700
600
500
400
300
200
100
0
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

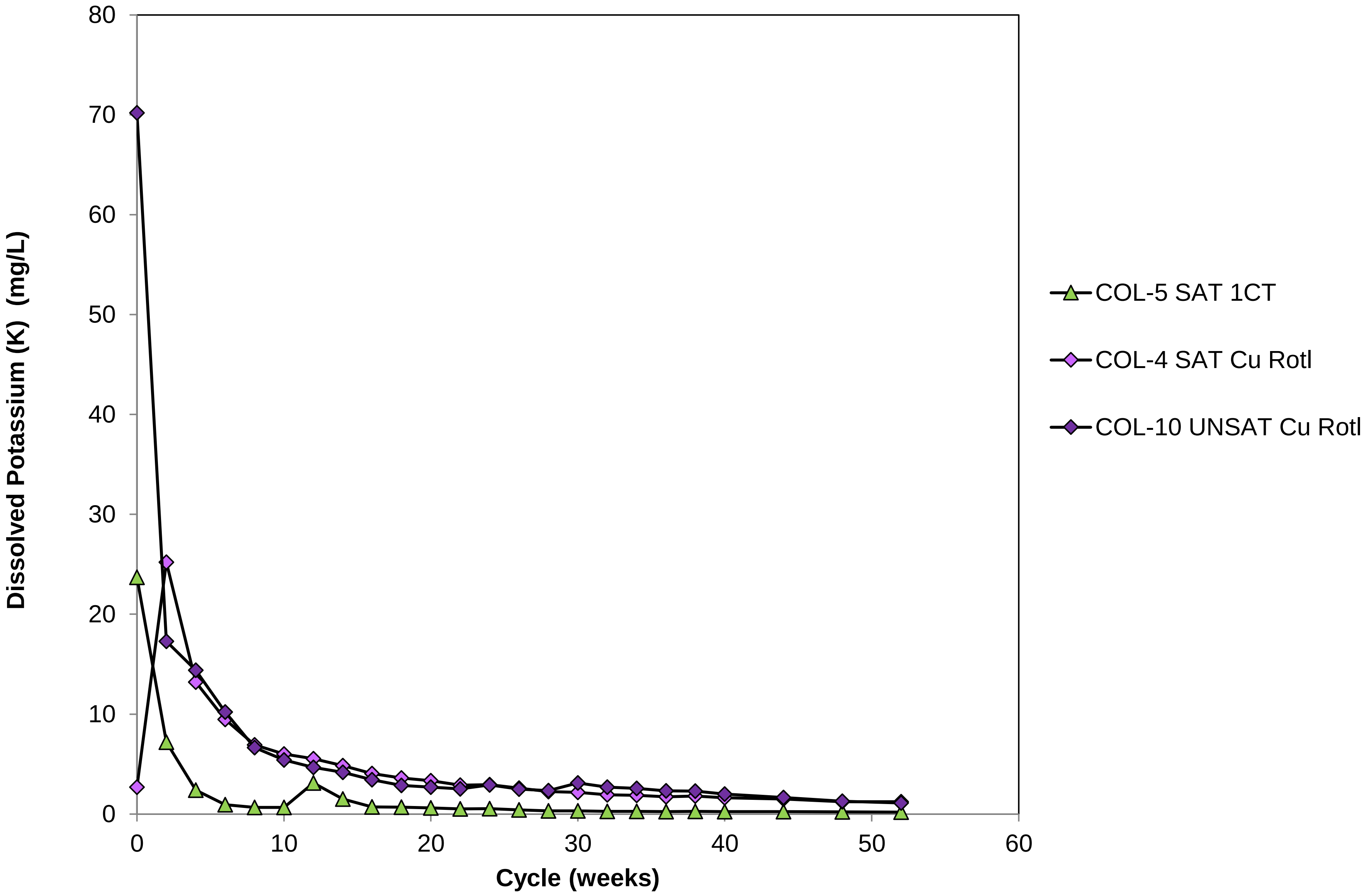

3.5
3
2.5
2
1.5
1
0.5
0
Dissolved Manganese (Mn) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

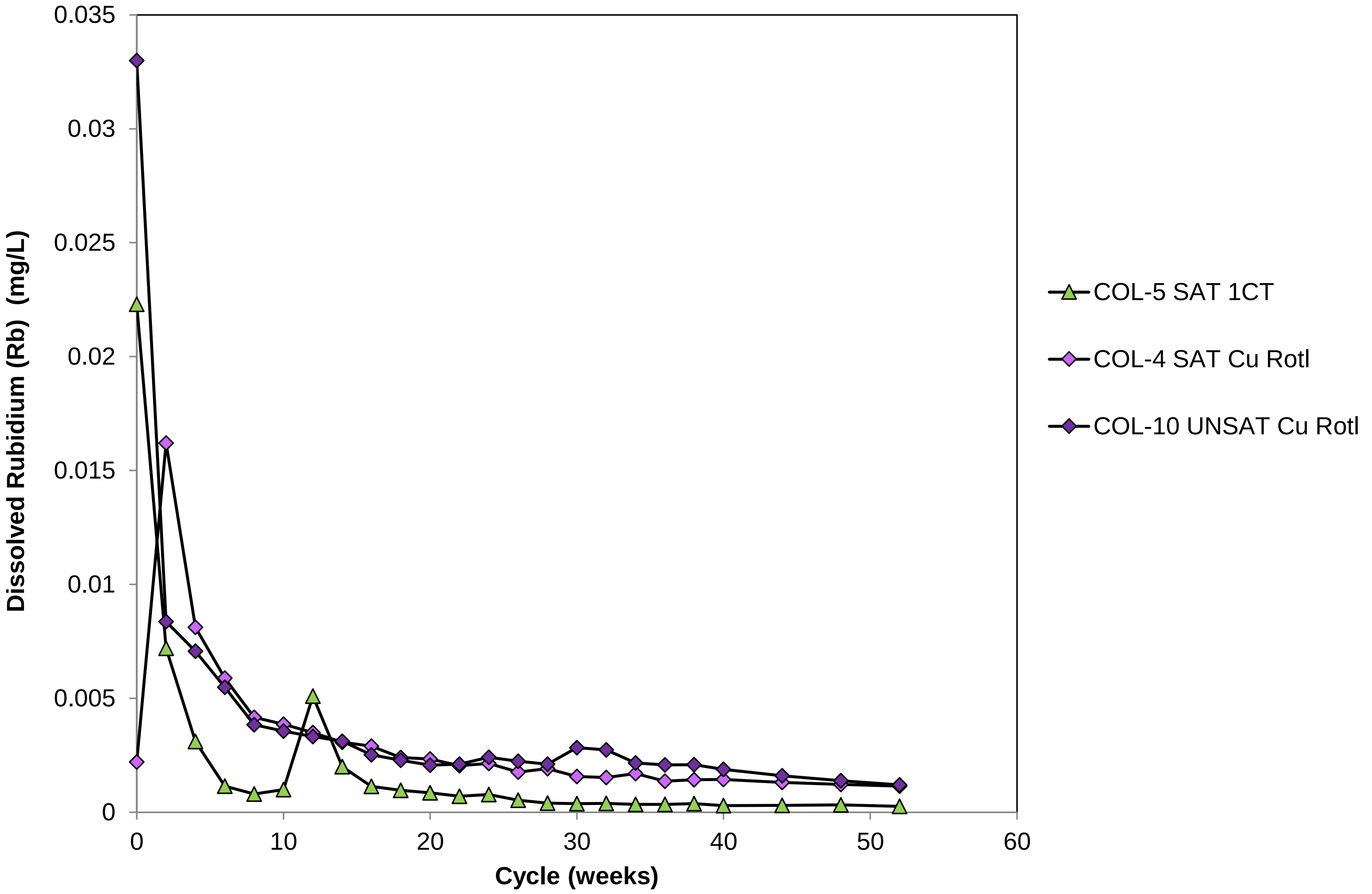

0.00006
0.00005
0.00004
0.00003
0.00002
0.00001
0
0
10
20
30
40
50
60
Dissolved Mercury (Hg) (mg/L)
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

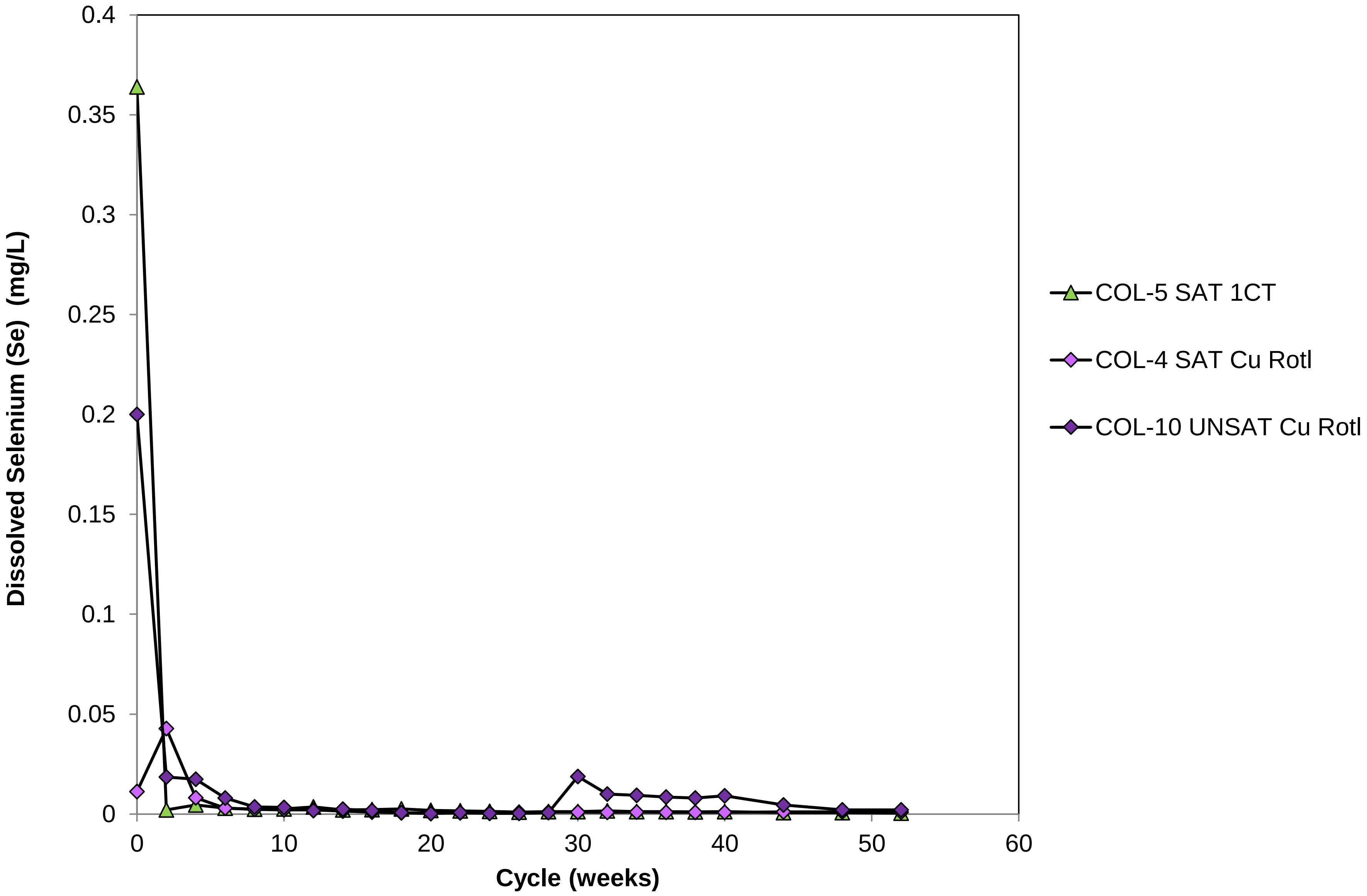

0.12
0.1
0.08
0.06
0.04
0.02
0
Dissolved Phosphorus (P) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

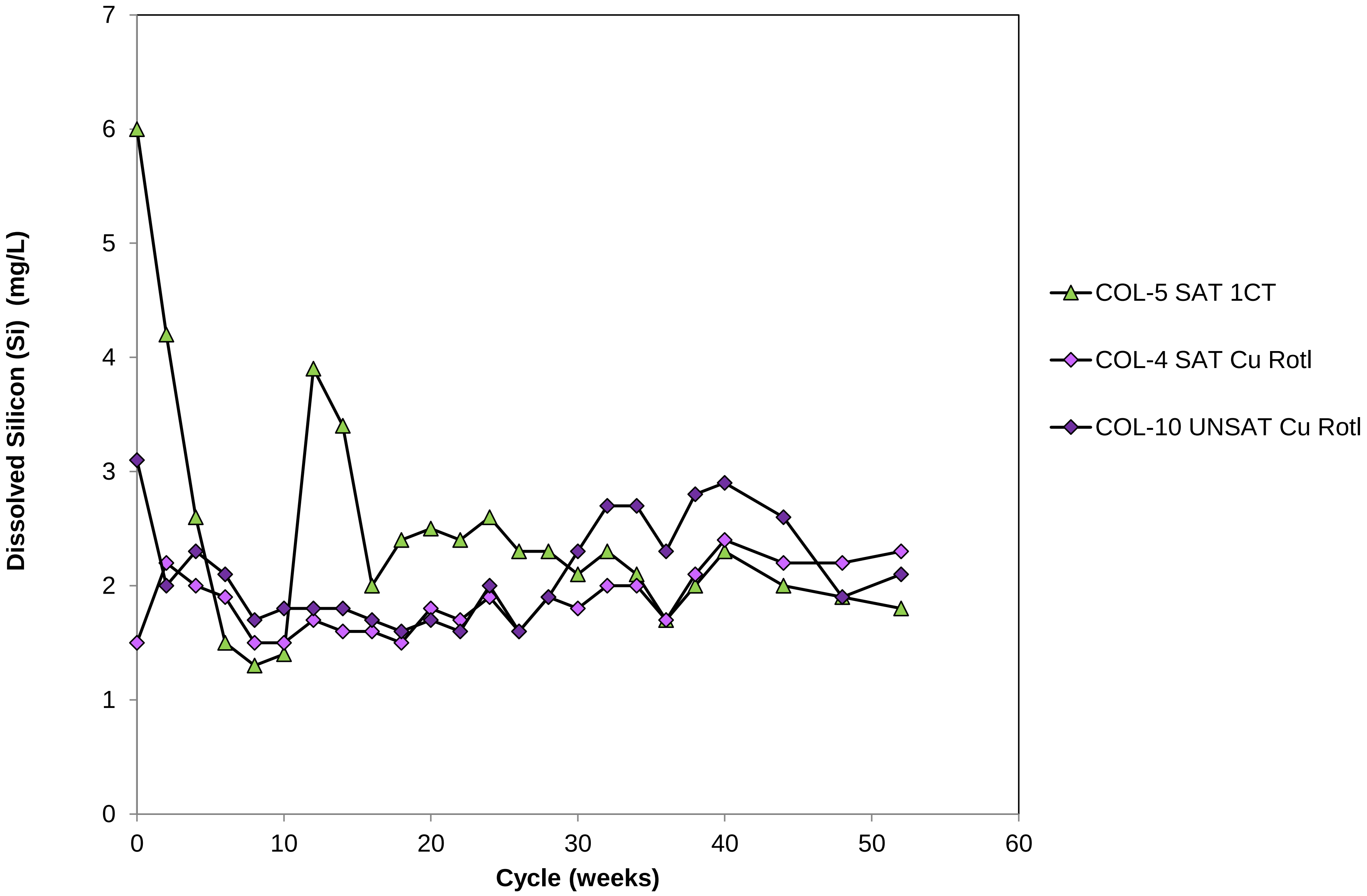

0.14
0.12
0.1
0.08
0.06
0.04
0.02
0
Dissolved Molybdenum (Mo) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

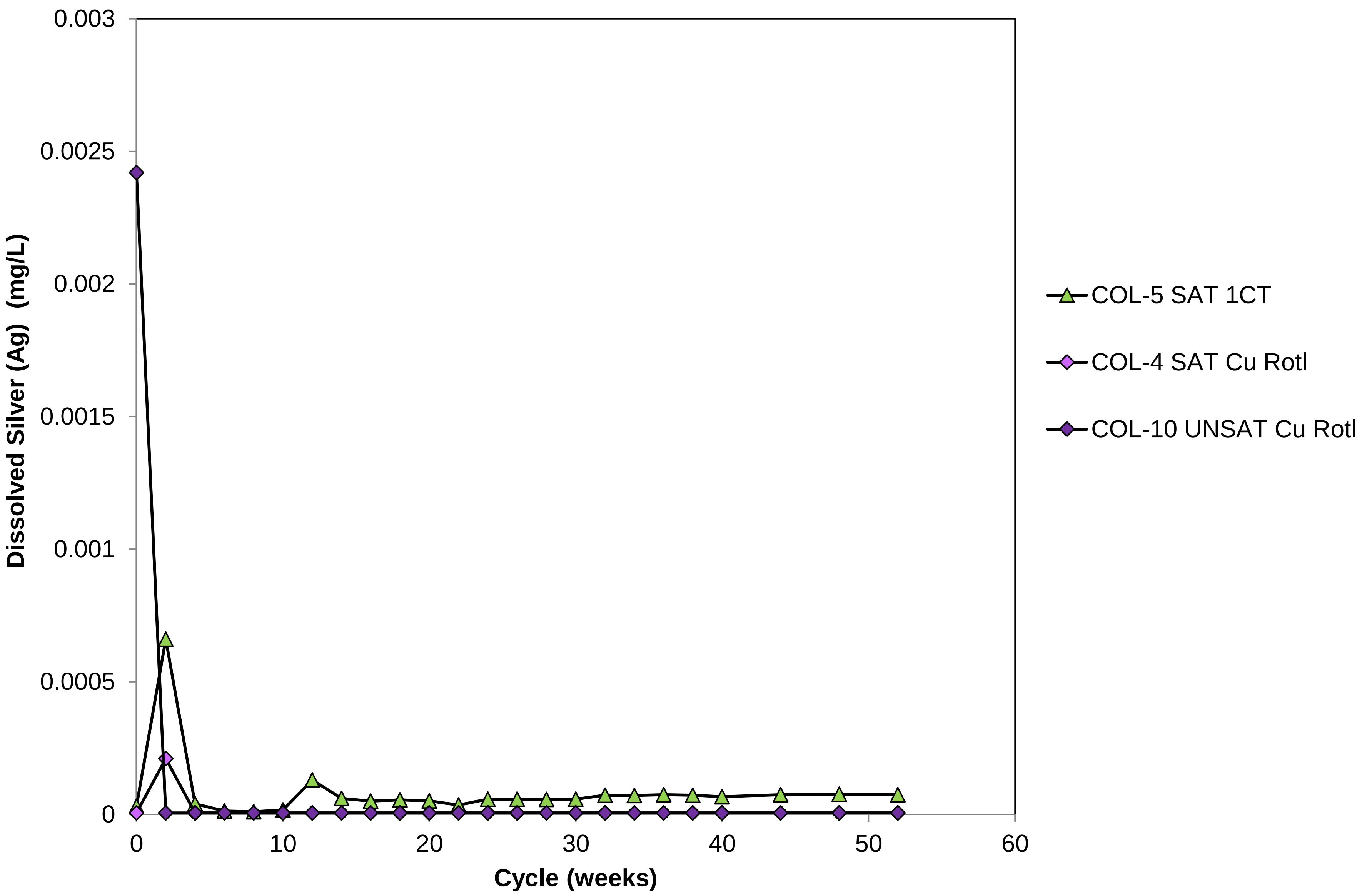

Dissolved Nickel (Ni) (mg/L)
0.35
0.3
0.25
0.2
0.15
0.1
0.05
0
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

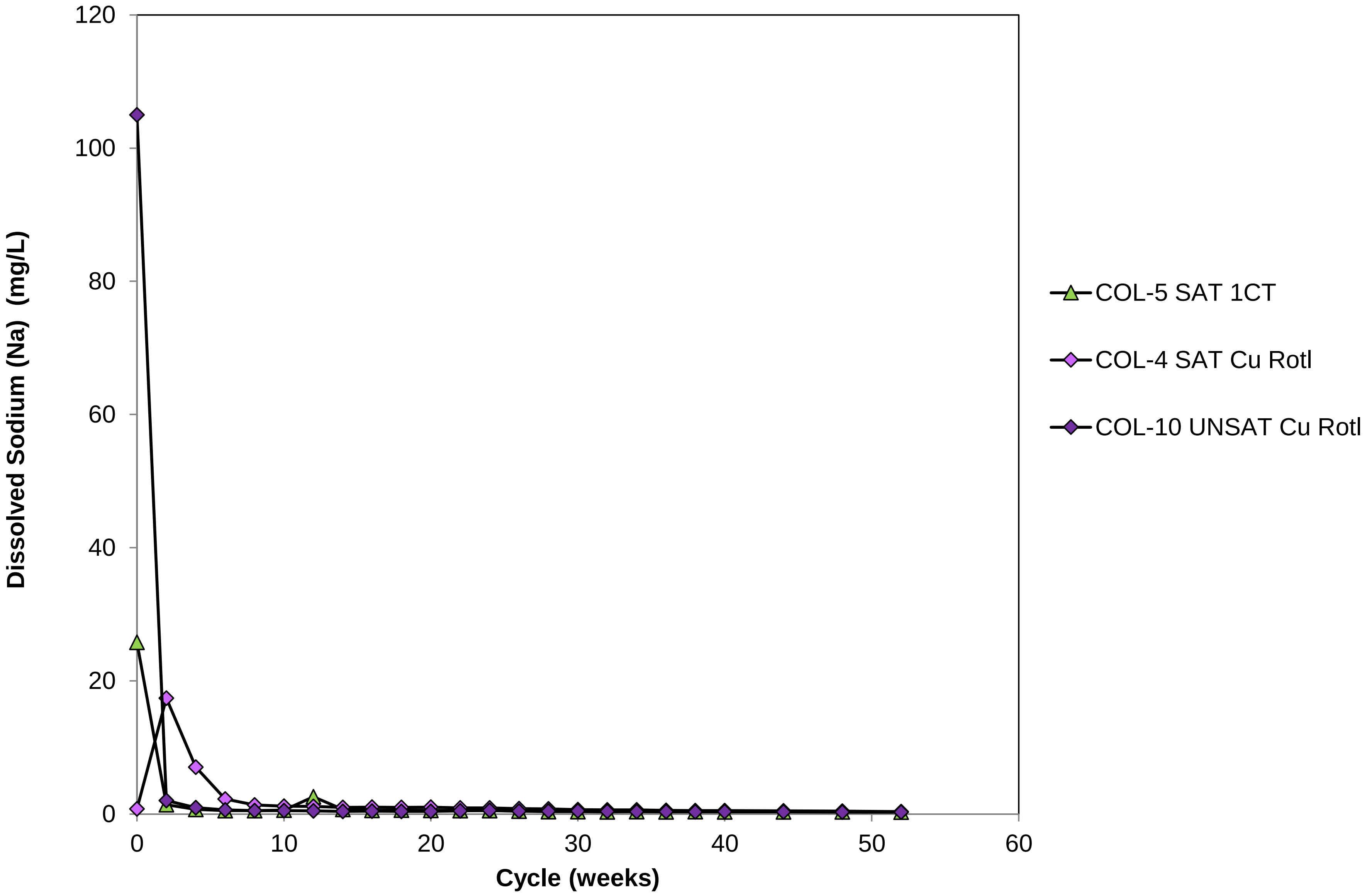

Dissolved Potassium (K) (mg/L)
80
70
60
50
40
30
20
10
0
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

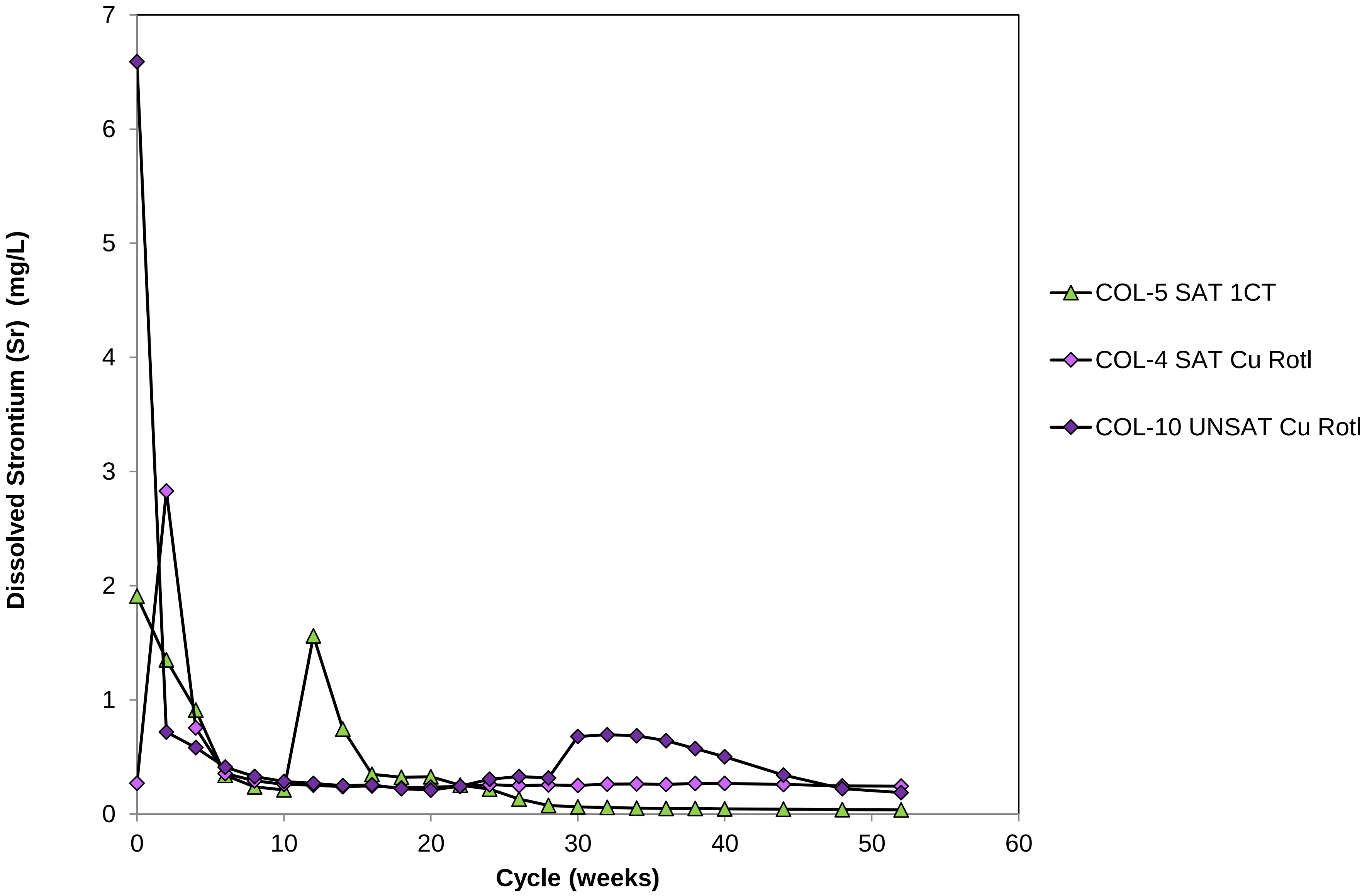

0.035
0.03
0.025
0.02
0.015
0.01
0.005
0
Dissolved Rubidium (Rb) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

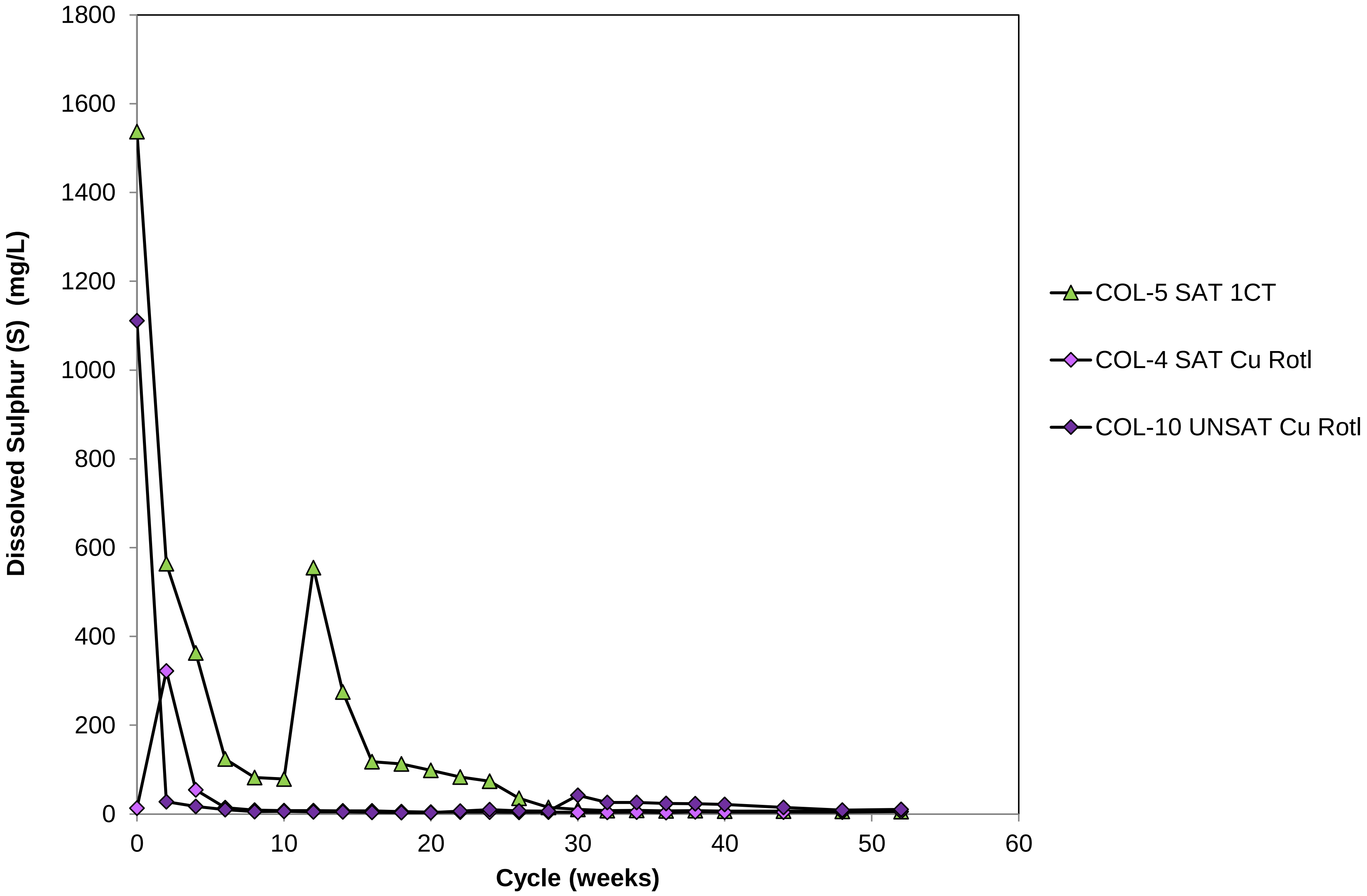

0.4
0.35
0.3
0.25
0.2
0.15
0.1
0.05
0
Dissolved Selenium (Se) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

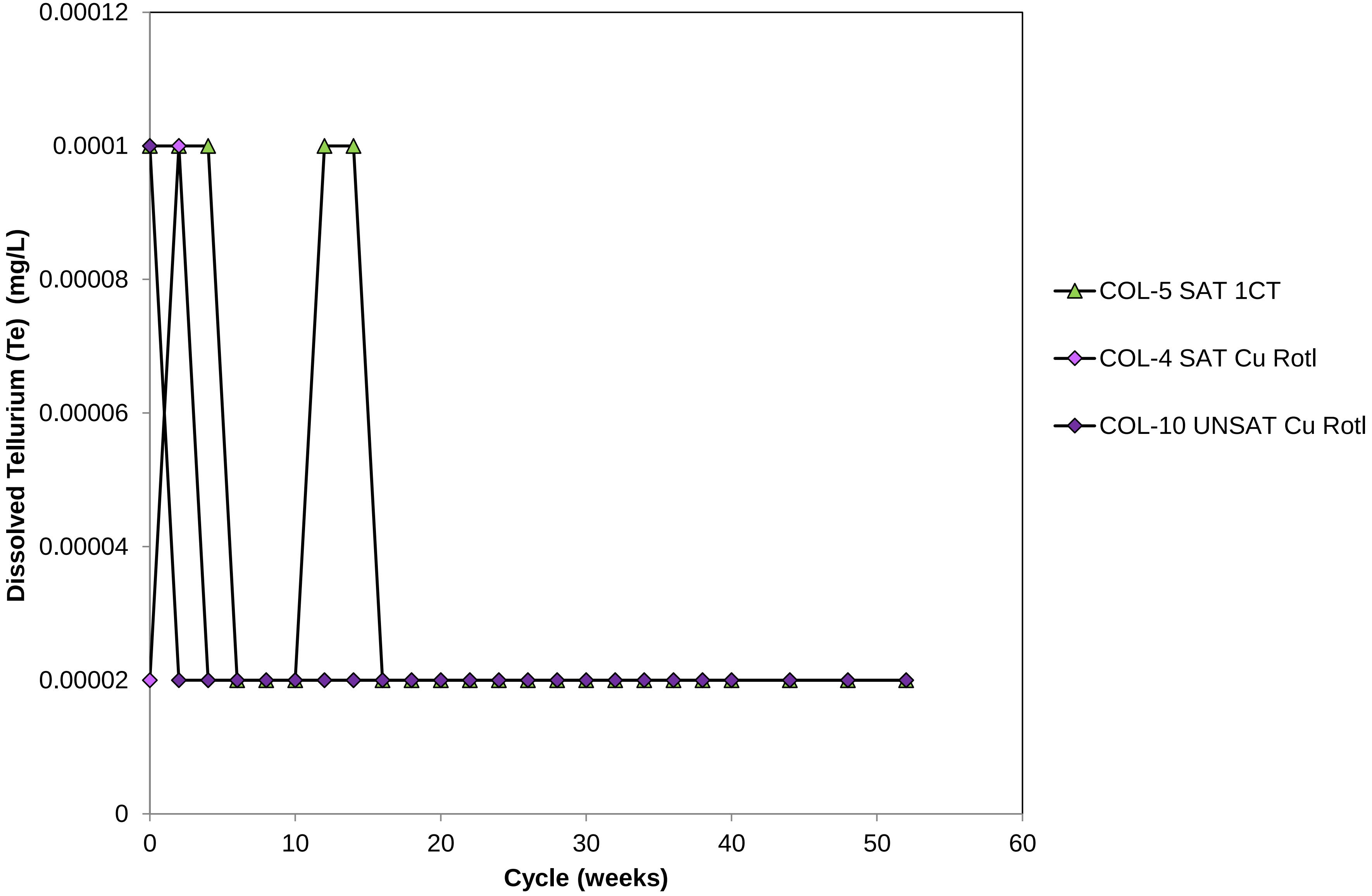

Dissolved Silicon (Si) (mg/L)
0
1
2
3
4
5
6
7
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

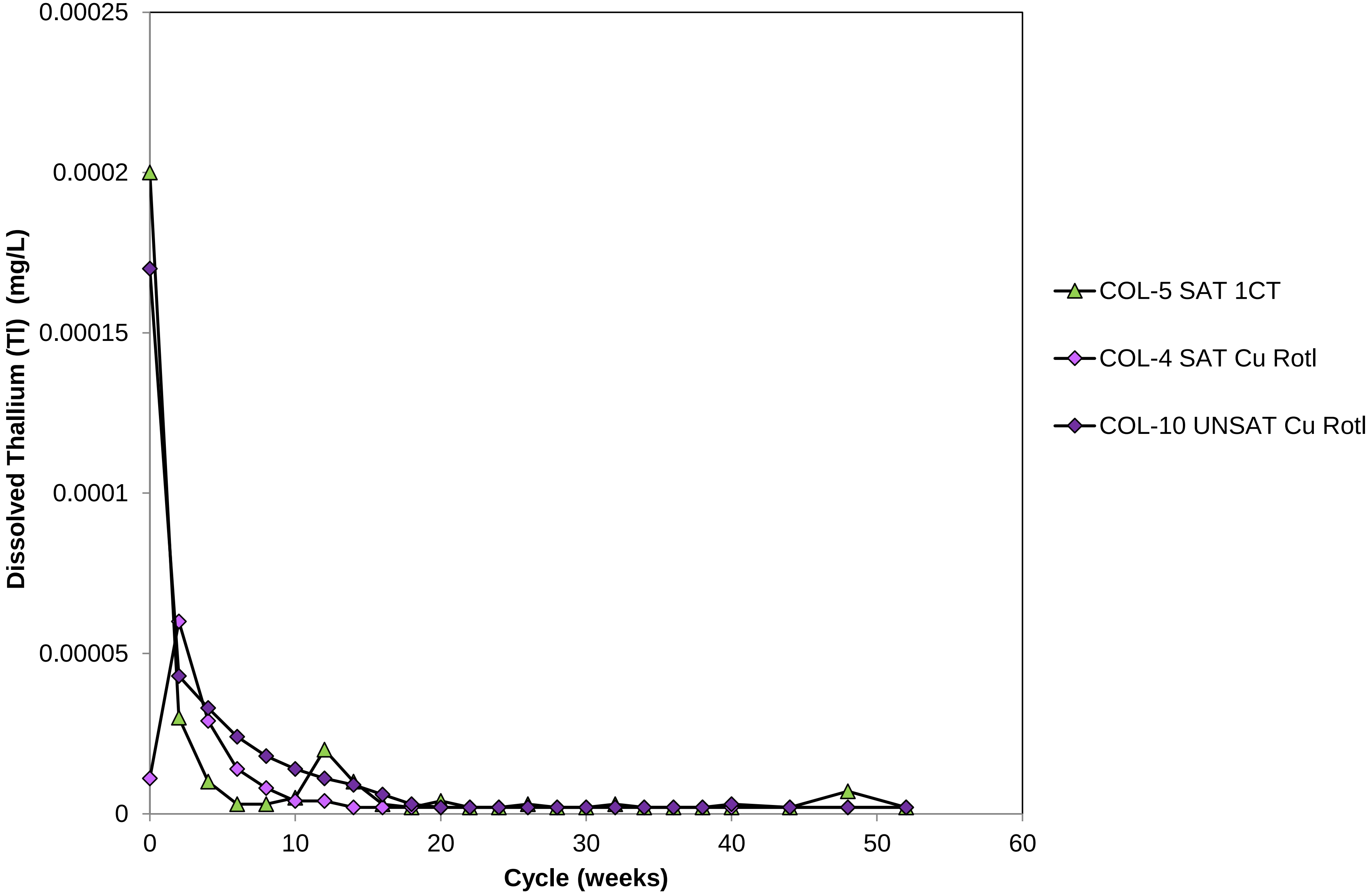

0.003
0.0025
0.002
0.0015
0.001
0.0005
0
Dissolved Silver (Ag) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

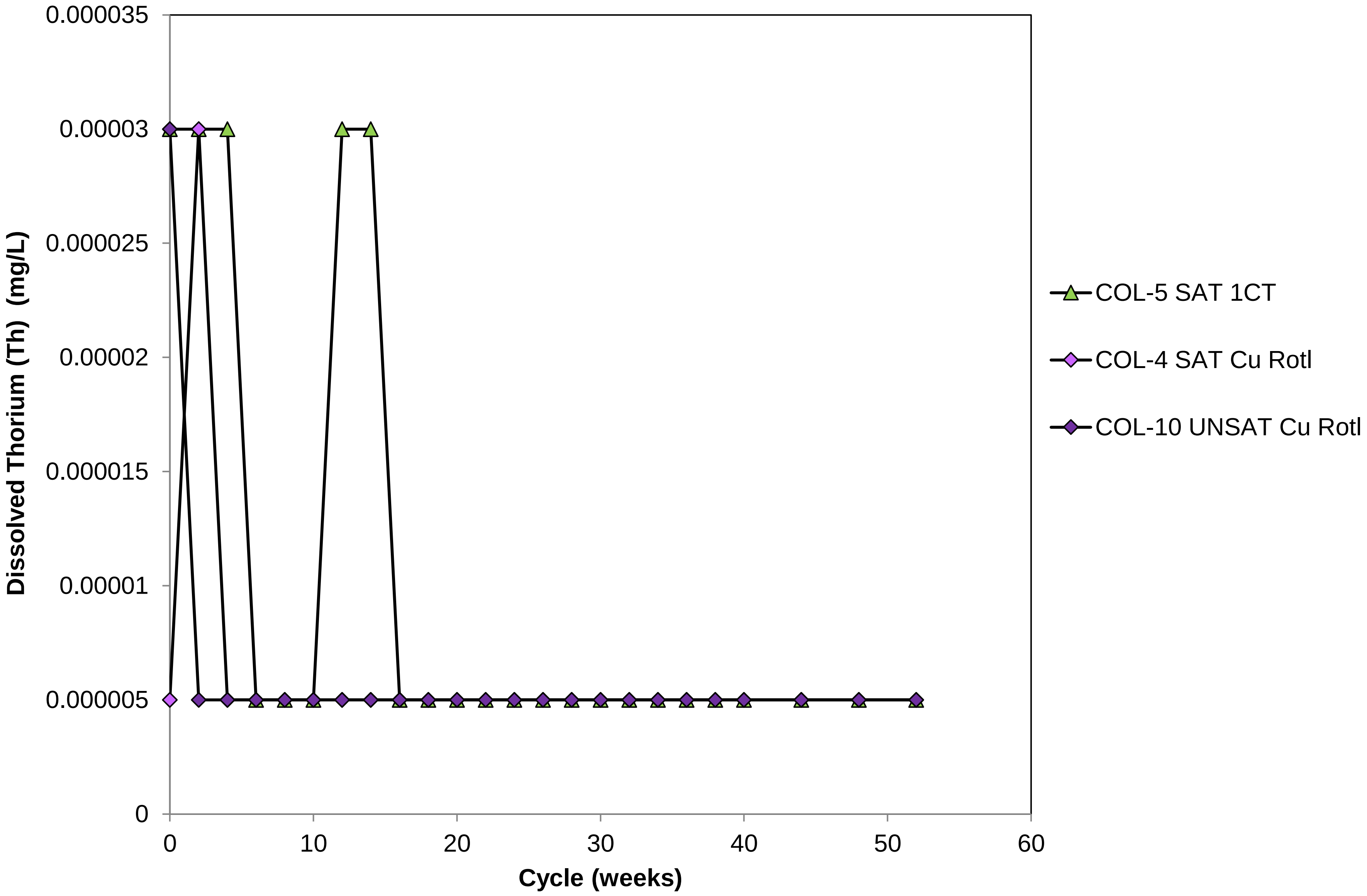

Dissolved Sodium (Na) (mg/L)
120
100
80
60
40
20
0
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

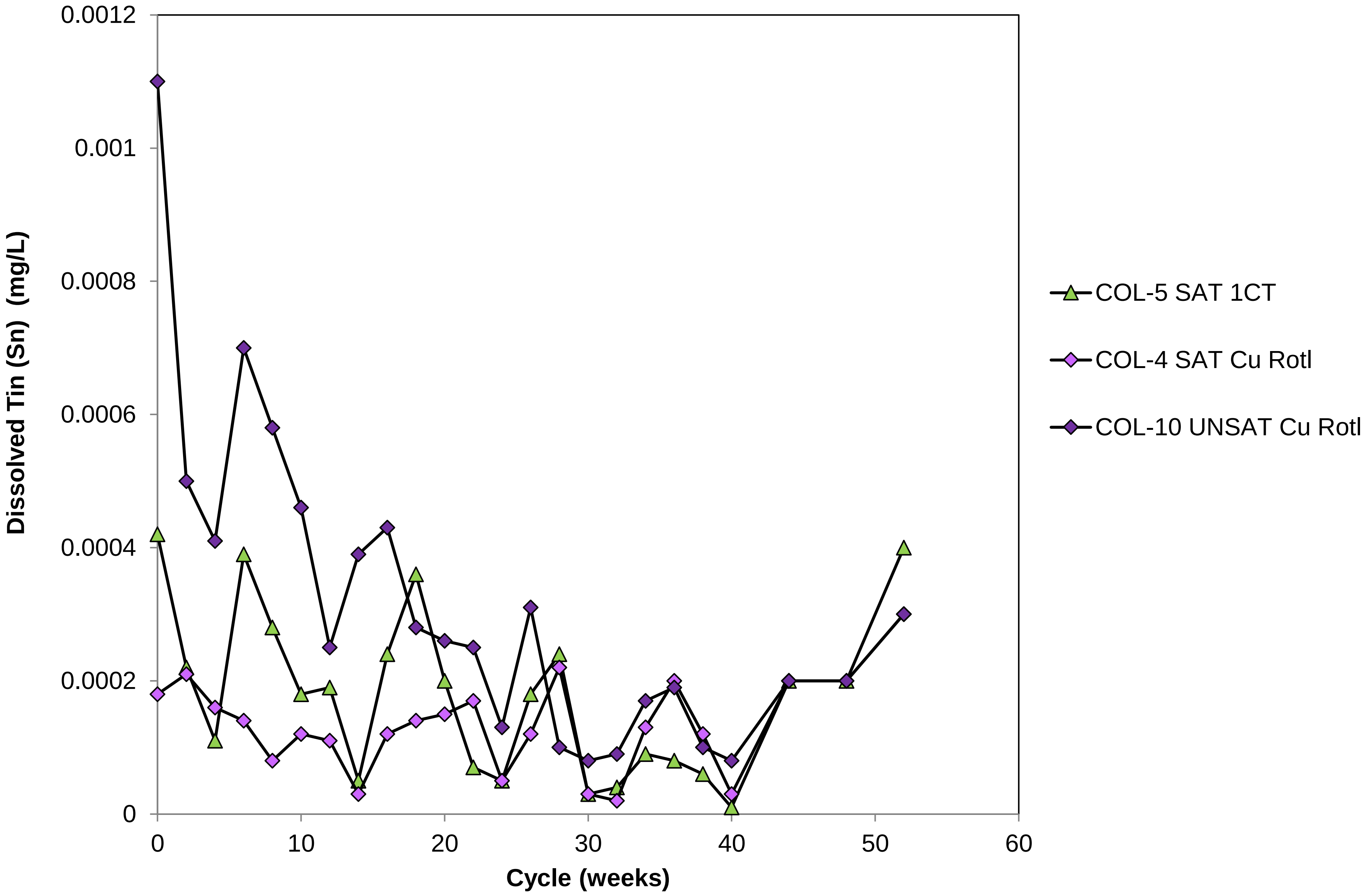

Dissolved Strontium (Sr) (mg/L)
0
1
2
3
4
5
6
7
Cycle (weeks)
0
10
20
30
40
50
60
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

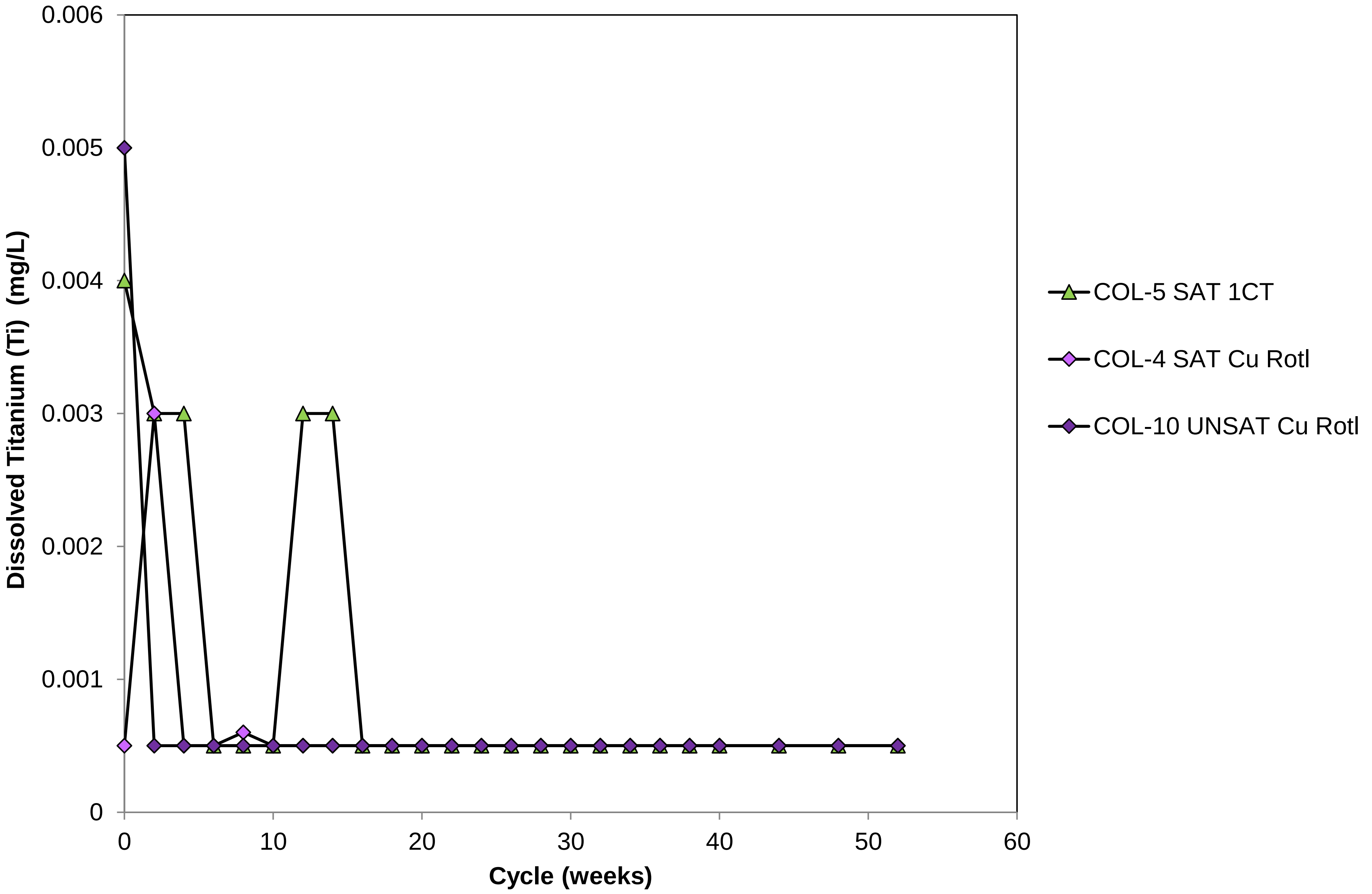

1800
1600
1400
1200
1000
800
600
400
200
0
Dissolved Sulphur (S) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

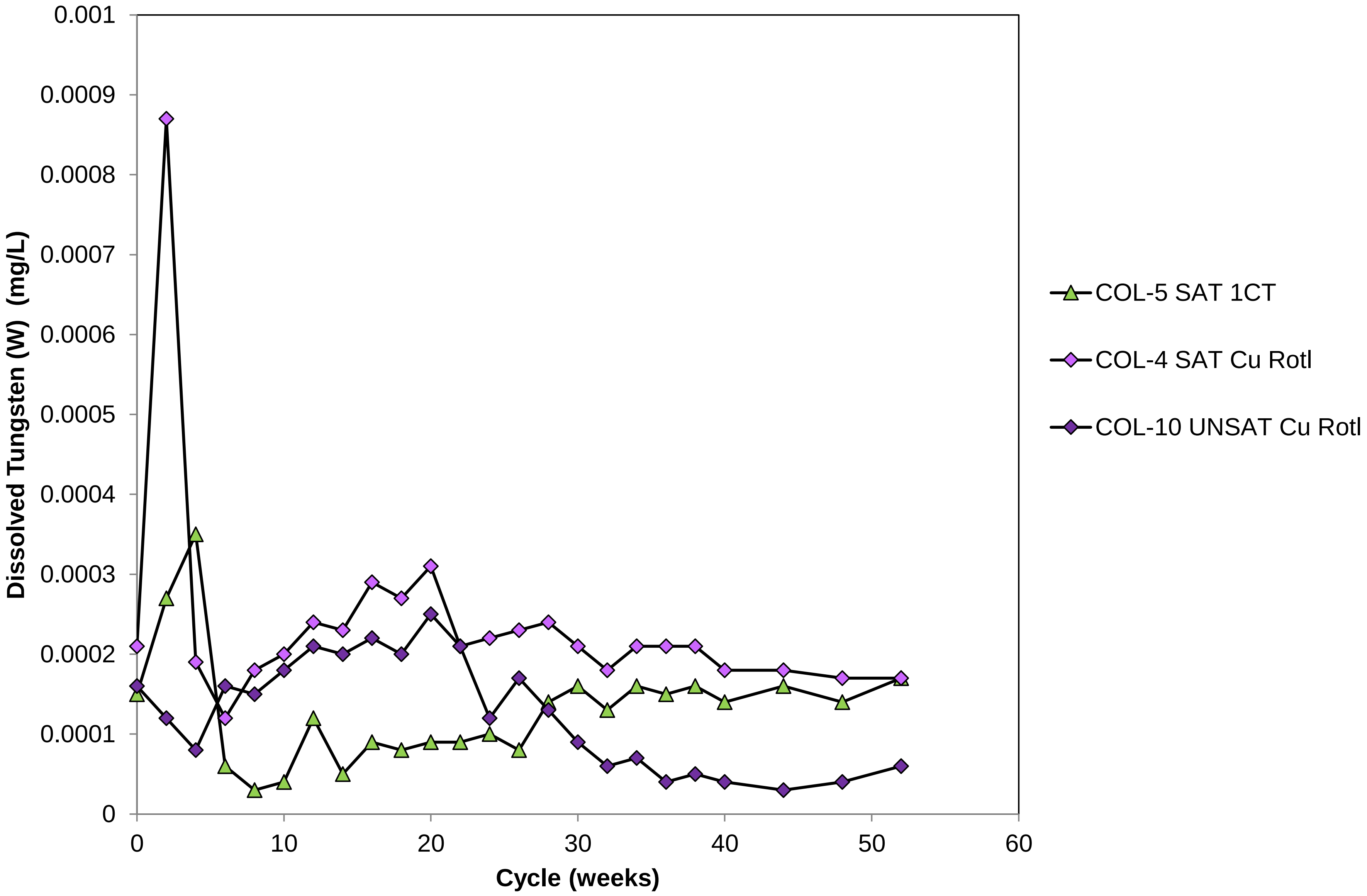

0.00012
0.0001
0.00008
0.00006
0.00004
0.00002
0
Dissolved Tellurium (Te) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

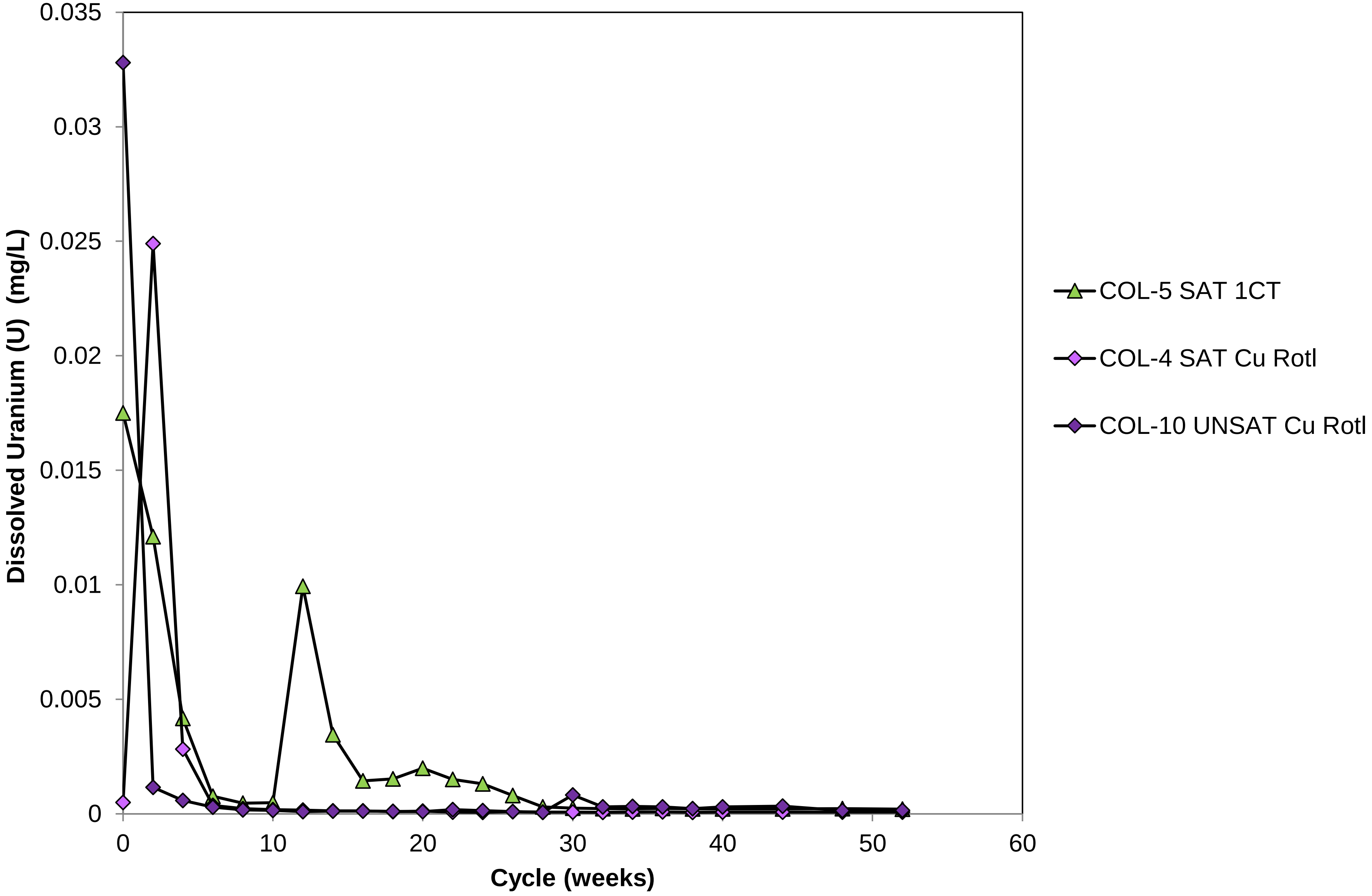

0.00025
0.0002
0.00015
0.0001
0.00005
0
Dissolved Thallium (Tl) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

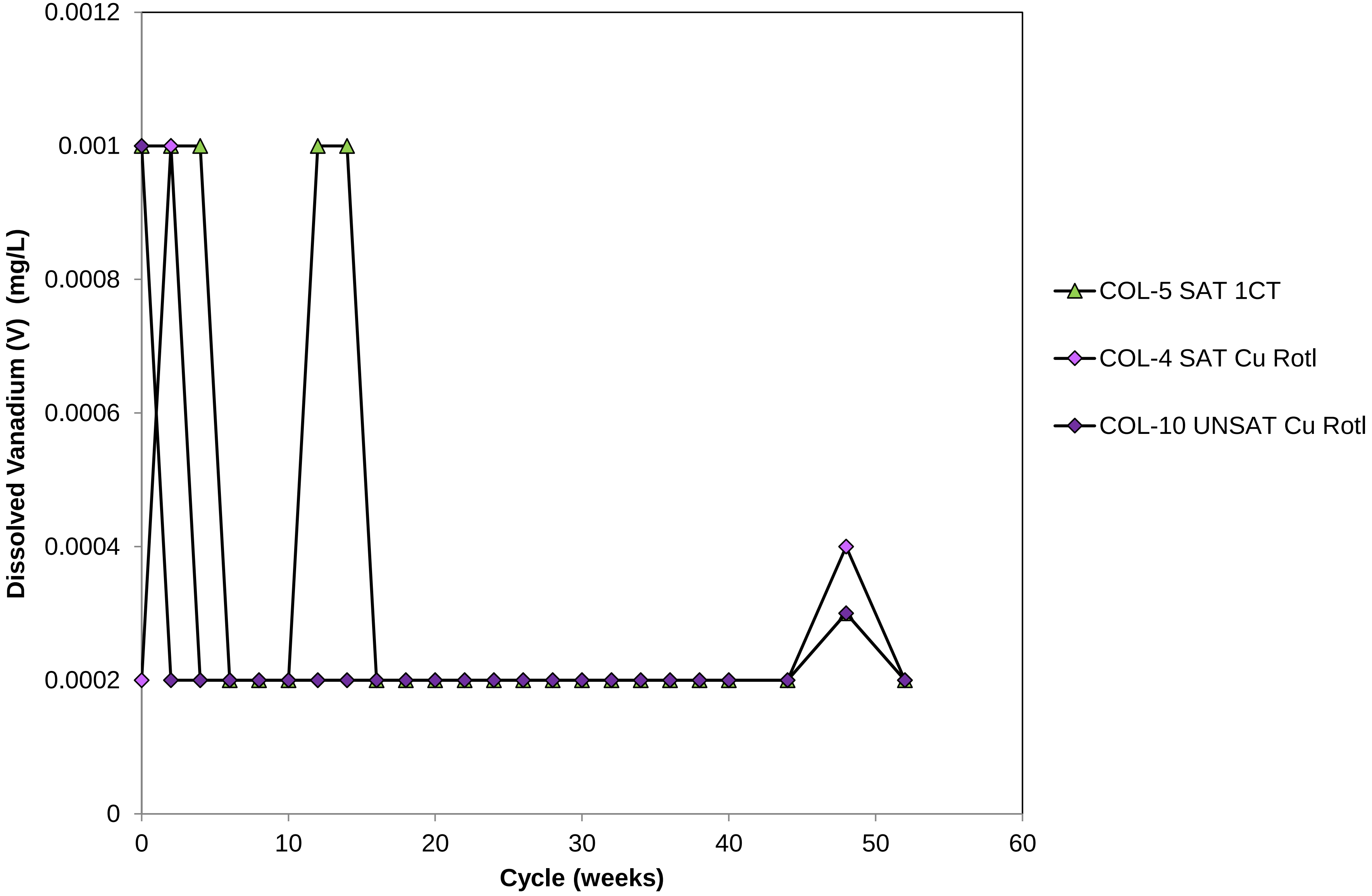

0.000035
0.00003
0.000025
0.00002
0.000015
0.00001
0.000005
0
Dissolved Thorium (Th) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

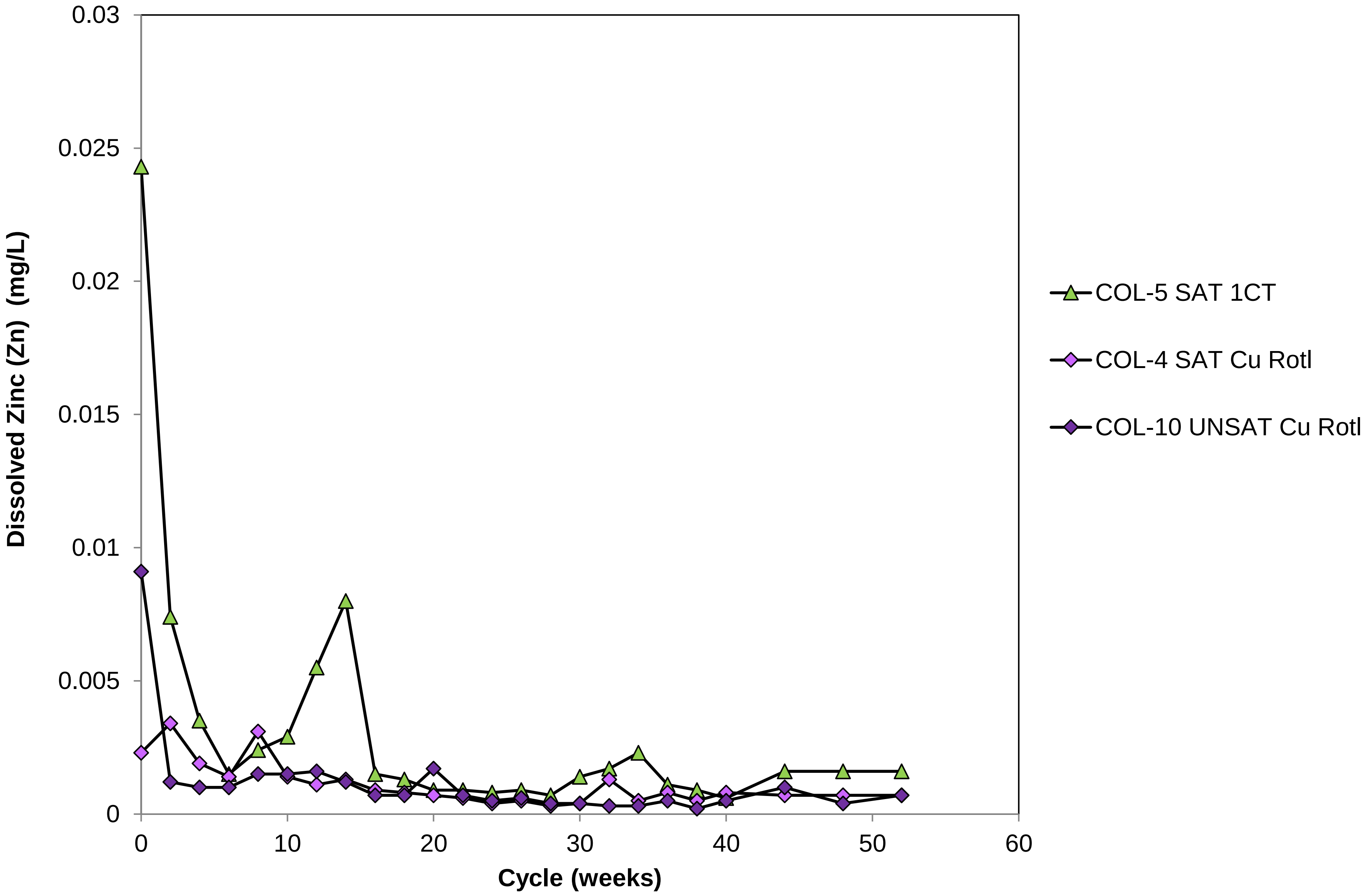

0.0012
0.001
0.0008
0.0006
0.0004
0.0002
0
Dissolved Tin (Sn) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

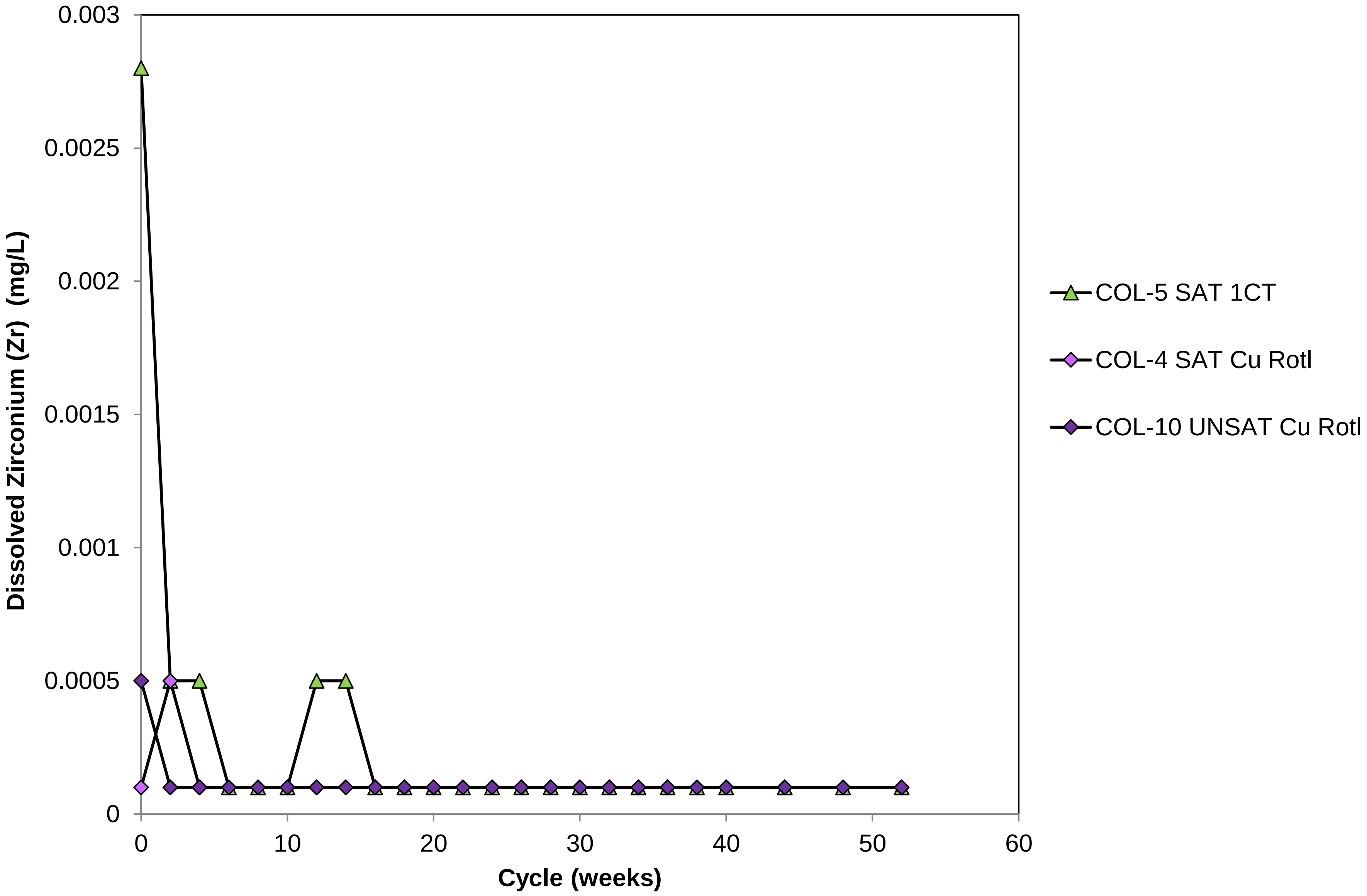

0.006
0.005
0.004
0.003
0.002
0.001
0
Dissolved Titanium (Ti) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

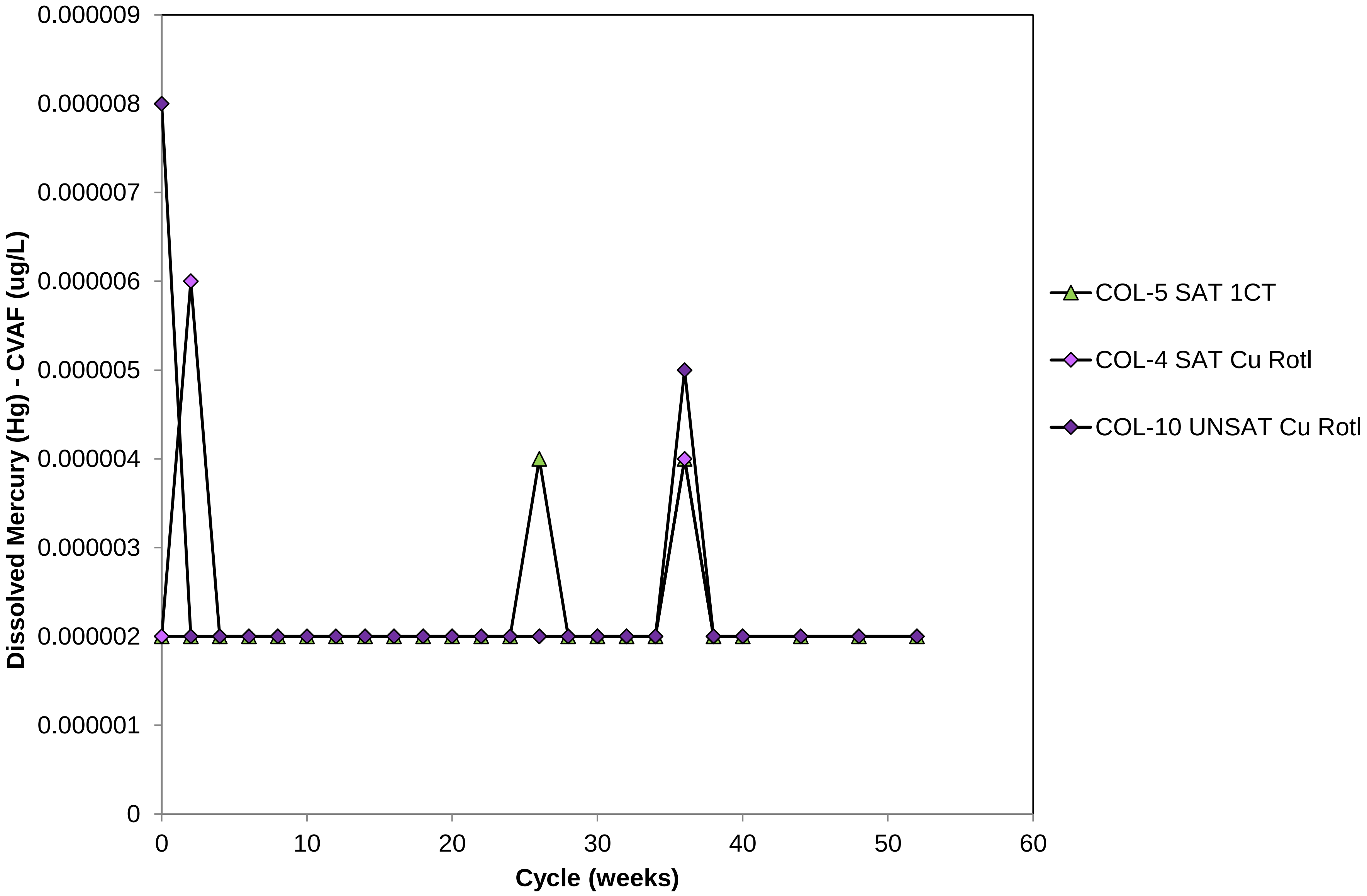

0.001
0.0009
0.0008
0.0007
0.0006
0.0005
0.0004
0.0003
0.0002
0.0001
0
Dissolved Tungsten (W) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl


Dissolved Uranium (U) (mg/L)
0.035
0.03
0.025
0.02
0.015
0.01
0.005
0
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl


0.0012
0.001
0.0008
0.0006
0.0004
0.0002
0
Dissolved Vanadium (V) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl


0.03
0.025
0.02
0.015
0.01
0.005
0
Dissolved Zinc (Zn) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl


0.003
0.0025
0.002
0.0015
0.001
0.0005
0
Dissolved Zirconium (Zr) (mg/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl


0.000009
0.000008
0.000007
0.000006
0.000005
0.000004
0.000003
0.000002
0.000001
0
Dissolved Mercury (Hg) - CVAF (ug/L)
0
10
20
30
40
50
60
Cycle (weeks)
COL-5 SAT 1CT
COL-4 SAT Cu Rotl
COL-10 UNSAT Cu Rotl

Appendix E: Overburden

E1: Static Test Results ABA and Trace Element

Acid Base Accounting

Sample ID	Area	Type	Rinse pH	Rinse EC	Paste EC	Rinse ORP	Paste	TIC	CO2	CaCO3	S(T)	S(SO4)	Sulfide	AP	Modified NP	NP/AP	TIC/AP	Fizz Test
Units				uS/cm	uS/cm		pH	%	wt%	kg CaCO3/t	%	%	%	kg CaCO3/t	t CaCO3/1000t			
Method Code			MEND	MEND		MEND	Sobek	CSB02V		Calc.	CSA06V	CSA07V	Calculated	Calculated	Modified NP	Calculated	Calculated	Sobek
LOD			0.01	1		1	0.2	0.01		#N/A	0.01	0.01			0.5			#N/A
OP01 <2mm	Pit	Road Cut	5.1	33	#N/A	663	4.73	0.01	#N/A	0.8	0.04	0.02	0.02	0.625	-3.2	-5.1	1.3	none
OP02 <2mm	Pit	Road Cut	5.45	70	#N/A	669	5.47	0.02	#N/A	1.7	0.05	0.01	0.04	1.25	2.7	2.2	1.3	none
OP03 <2mm	Pit	Road Cut	5.21	59	#N/A	626	4.98	0.01	#N/A	0.8	0.04	0.01	0.03	0.9375	3.0	3.2	0.9	none
OP04 <2mm	Pit	Road Cut	5.01	37	#N/A	645	5.15	0.01	#N/A	0.8	0.07	0.01	0.06	1.875	1.8	1.0	0.4	none
OP05 <2mm	Pit	Road Cut	5.37	20	#N/A	662	5.08	0.01	#N/A	0.8	0.1	0.02	0.08	2.5	-0.1	0.0	0.3	none
OP06 <2mm	Pit	Road Cut	5.56	33	#N/A	658	5.39	0.01	#N/A	0.8	0.03	0.02	0.01	0.3125	5.3	16.9	2.7	none
OP07 <2mm	Pit	Road Cut	5.49	236	#N/A	606	5.14	0.02	#N/A	1.7	0.03	0.02	0.01	0.3125	0.9	2.8	5.3	none
OP08 <2mm	Pit	Road Cut	5.63	88	#N/A	622	5.19	0.01	#N/A	0.8	0.03	0.01	0.02	0.625	-1.5	-2.4	1.3	none
OP09 <2mm	Pit	Road Cut	5.68	43	#N/A	632	5.62	<0.01	#N/A	0.8	0.01	0.01	0.01	0.3125	0.2	0.8	2.6	none
OP10 <2mm	Pit	Road Cut	5.16	26	#N/A	636	4.67	0.01	#N/A	0.8	0.03	0.01	0.02	0.625	-3.4	-5.5	1.3	none
OP11 <2mm	Pit	Road Cut	5.76	49	#N/A	658	6.54	<0.01	#N/A	0.8	0.02	0.01	0.01	0.3125	6.3	20.1	2.6	none
OP12 <2mm	Pit	Road Cut	5.53	77	#N/A	612	5.05	0.02	#N/A	1.7	0.02	0.01	0.01	0.3125	-3.7	-11.8	5.3	none
OP13 <2mm	Pit	Road Cut	6.34	96	#N/A	617	6.55	0.02	#N/A	1.7	0.01	0.01	0.01	0.3125	3.6	11.4	5.3	none
OP14 <2mm	Pit	Road Cut	4.74	120	#N/A	599	4.36	0.02	#N/A	1.7	0.02	0.01	0.01	0.3125	-7.6	-24.4	5.3	none
OP15 <2mm	Pit	Road Cut	5.26	48	#N/A	674	5.18	<0.01	#N/A	0.8	0.01	0.01	0.01	0.3125	-0.9	-2.8	2.6	none
OP16 <2mm	Pit	Road Cut	5.71	40	#N/A	669	5.94	0.01	#N/A	0.8	0.02	0.01	0.01	0.3125	5.2	16.6	2.7	none
OP12-01 Overburden Sample #1	Pit	Deep Overburden	5.73	616	291	#N/A	7.76	#N/A	4.24	96.4	0.28	0.01	0.28	8.75	100.2	11.5	#N/A	MODERATE
OP12-01 Overburden Sample #2	Pit	Deep Overburden	6.94	432	175	#N/A	8.57	#N/A	4.94	112.3	0.42	0.01	0.42	13.125	120.6	9.2	#N/A	MODERATE
OP12-02 Overburden Sample #1	Pit	Deep Overburden	7.67	289	268	#N/A	7.93	#N/A	1.23	28	0.18	0.01	0.18	5.625	31.8	5.7	#N/A	SLIGHT
OP12-02Overburden Sample #2	Pit	Deep Overburden	7.32	230	269	#N/A	7.78	#N/A	0.24	5.5	0.09	0.01	0.09	2.8125	8.4	3.0	#N/A	NONE
OP12-03 Overburden Sample #1	Pit	Deep Overburden	7.45	382	335	#N/A	7.81	#N/A	2.2	50	0.07	0.01	0.07	2.1875	58.6	26.8	#N/A	MODERATE
OP12-05 Overburden Sample #1	Pit	Deep Overburden	5.62	169	314	#N/A	6.84	#N/A	0.02	0.5	0.03	0.01	0.03	0.9375	2.5	2.7	#N/A	NONE
OP12-06Overburden Sample #1	Pit	Deep Overburden	7.38	190	250	#N/A	8	#N/A	0.18	4.1	0.2	0.01	0.2	6.25	9.9	1.6	#N/A	SLIGHT
OP12-07 Overburden Sample #1	Pit	Deep Overburden	7.6	411	357	#N/A	7.96	#N/A	2.05	46.6	0.09	0.01	0.09	2.8125	52.7	18.7	#N/A	MODERATE
OP12-09 Overburden Sample #1	Pit	Deep Overburden	6.14	44	204	#N/A	7.35	#N/A	0.02	0.5	0.03	0.01	0.03	0.9375	2	2.1	#N/A	NONE
OP12-10 Overburden Sample #1	Pit	Deep Overburden	5.28	48	92	#N/A	7.11	#N/A	0.05	1.1	0.05	0.01	0.05	1.5625	14.3	9.2	#N/A	NONE
OP12-01 Weathered Rock Sample #1	Pit	Weathered Bedrock	7.31	490	475	#N/A	7.97	#N/A	1.78	40.5	2.98	0.01	2.98	93.125	32.6	0.4	#N/A	SLIGHT
OP12-01 Weathered Rock Sample #2	Pit	Weathered Bedrock	7.69	596	628	#N/A	8.29	#N/A	4.22	95.9	3.71	0.01	3.71	115.9375	68	0.6	#N/A	SLIGHT
OP12-02 Weathered Rock Sample #1	Pit	Weathered Bedrock	7.09	114	217	#N/A	7.68	#N/A	0.03	0.7	0.07	0.01	0.07	2.1875	1.7	0.8	#N/A	NONE
OP12-03 Weathered Rock Sample #1	Pit	Weathered Bedrock	6.55	69	87	#N/A	7.57	#N/A	0.04	0.9	0.13	0.01	0.13	4.0625	1	0.2	#N/A	NONE
OP12-04 Weathered Rock Sample #1	Pit	Weathered Bedrock	6.4	335	266	#N/A	8.28	#N/A	3.26	74.1	1.87	0.01	1.86	58.125	93.8	1.6	#N/A	MODERATE
OP12-05 Weathered Rock Sample #1	Pit	Weathered Bedrock	7.35	112	202	#N/A	8.81	#N/A	0.19	4.3	0.02	0.01	0.02	0.625	18	28.8	#N/A	SLIGHT
OP12-05 Weathered Rock Sample #2	Pit	Weathered Bedrock	7.42	174	159	#N/A	8.55	#N/A	0.25	5.7	0.06	0.01	0.06	1.875	14	7.5	#N/A	SLIGHT
OP12-06 Weathered Rock Sample #1	Pit	Weathered Bedrock	6.42	314	409	#N/A	7.31	#N/A	0.63	14.3	4.45	0.01	4.45	139.0625	5.2	0.04	#N/A	NONE
OP12-07 Weathered Rock Sample #1	Pit	Weathered Bedrock	8.17	159	273	#N/A	9.05	#N/A	5.96	135.5	0.56	0.01	0.56	17.5	106.3	6.1	#N/A	SLIGHT
OP12-08 Weathered Rock Sample #1	Pit	Weathered Bedrock	7.95	52	212	#N/A	9.09	#N/A	0.3	6.8	0.11	0.01	0.11	3.4375	9.1	2.6	#N/A	NONE
OP12-09 Weathered Rock Sample #1	Pit	Weathered Bedrock	5.94	60	269	#N/A	7.64	#N/A	0.13	3	0.11	0.01	0.11	3.4375	3	0.9	#N/A	NONE
OP12-09 Weathered Rock Sample #2	Pit	Weathered Bedrock	5.92	180	310	#N/A	7.6	#N/A	2.15	48.9	0.23	0.01	0.23	7.1875	8.1	1.1	#N/A	NONE
OP12-10 Weathered Rock Sample #1	Pit	Weathered Bedrock	6.98	164	316	#N/A	8.78	#N/A	3.38	76.8	1.4	0.01	1.4	43.75	93.7	2.1	#N/A	MODERATE
TP01-1 <2mm	TMF	Overburden	5.53	56	#N/A	639	5.2	0.01	#N/A	0.8	0.02	0.01	0.01	0.3125	-3.9	-12.6	2.7	none
TP5-1 <2mm	TMF	Overburden	6.84	60	#N/A	614	6.7	0.01	#N/A	0.8	0.02	0.01	0.01	0.3125	4.8	15.4	2.7	none
TP12-1 <2mm	TMF	Overburden	6.15	66	#N/A	637	6.21	0.02	#N/A	1.7	0.04	0.01	0.03	0.9375	3.7	3.9	1.8	none
TP12-2 <2mm	TMF	Overburden	7.58	208	#N/A	590	7.57	0.15	#N/A	12.5	0.16	0.01	0.15	4.6875	9.7	2.1	2.7	none
TP15-4 <2mm	TMF	Overburden	7.78	359	#N/A	594	8	0.4	#N/A	33.3	0.16	0.01	0.15	4.6875	30.2	6.4	7.1	slight
TP2 <2mm	TMF	Overburden	8.18	128	#N/A	581	7.95	0.03	#N/A	2.5	0.02	0.01	0.01	0.3125	7.1	22.9	8.0	none
TP26-1 <2mm	TMF	Overburden	7.91	347	#N/A	585	7.88	0.24	#N/A	20.0	0.11	0.01	0.1	3.125	18.1	5.8	6.4	slight
TP27-1 <2mm	TMF	Overburden	6.48	36	#N/A	611	6.61	<0.01	#N/A	0.8	0.01	0.01	0.01	0.3125	2.2	7.1	2.6	none
TP29-1 <2mm	TMF	Overburden	6.7	21	#N/A	623	6.8	<0.01	#N/A	0.8	0.01	0.01	0.01	0.3125	2.3	7.5	2.6	none
TP30-1 <2mm	TMF	Overburden	5.14	83	#N/A	590	4.96	0.01	#N/A	0.8	0.03	0.01	0.02	0.625	-7.1	-11.4	1.3	none
TP30-2 <2mm	TMF	Overburden	6.68	42	#N/A	595	7.06	<0.01	#N/A	0.8	0.01	0.01	0.01	0.3125	5.3	16.9	2.6	none
TP32-1 <2mm	TMF	Overburden	8.09	187	#N/A	566	7.91	0.23	#N/A	19.2	0.01	0.01	0.01	0.3125	21.1	67.4	61.3	slight
TP51-3 <2mm	TMF	Overburden	7.24	140	#N/A	568	7.13	0.01	#N/A	0.8	0.03	0.03	0.01	0.3125	-0.2	-0.8	2.7	none

				Metals																		
Sample ID	**Area**	**Type**	**Ca**	**Al**	**B**	**Ba**	**Ca**	**Cr**	**Cu**	**Fe**	**K**	**Li**	**Mg**	**Mn**	**Na**	**Ni**	**P**	**S**	**Sr**	**Ti**	**V**	**Zn**	**Zr**	**Ag**
Units			**kg CaCO3/t**	**%**	**ppm**	**ppm**	**%**	**ppm**	**ppm**	**%**	**%**	**ppm**	**%**	**ppm**	**%**	**ppm**	**ppm**	**%**	**ppm**	**%**	**ppm**	**ppm**	**ppm**	**ppm**
Method Code				ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B
LOD				0.01	10	5	0.01	1	0.5	0.01	0.01	1	0.01	2	0.01	0.5	50	0.01	0.5	0.01	1	1	0.5	0.01
OP01 <2mm	Pit	Road Cut	2.5	2.69	50	67	0.1	125	500	5.63	0.1	22	1.61	378	0.03	41.9	960	0.04	9.6	0.03	46	212	2	0.39
OP02 <2mm	Pit	Road Cut	7.7	3.34	60	39	0.31	208	976	8.66	0.08	23	2.61	769	0.03	82.6	1440	0.05	13.5	0.06	80	288	4	0.54
OP03 <2mm	Pit	Road Cut	10.5	2.18	50	34	0.42	75	1850	6.08	0.06	15	1.46	675	0.02	28.8	2180	0.04	15.1	0.04	44	165	2.6	1.4
OP04 <2mm	Pit	Road Cut	12.7	2.62	60	52	0.51	163	803	7.28	0.09	16	1.81	1050	0.02	69.3	2410	0.08	22.1	0.13	63	110	3	0.26
OP05 <2mm	Pit	Road Cut	10.5	2.48	60	49	0.42	95	1400	6.46	0.08	14	1.45	776	0.02	42.8	2000	0.13	17.9	0.13	50	86	3.1	0.39
OP06 <2mm	Pit	Road Cut	15.5	2.64	50	67	0.62	96	423	6.67	0.12	18	1.91	774	0.02	55.3	2810	0.03	28.3	0.13	62	93	2.6	0.16
OP07 <2mm	Pit	Road Cut	17.0	2.73	60	96	0.68	105	125	5.67	0.09	22	1.69	1300	0.02	51.9	2170	0.04	33.2	0.14	61	119	3.6	0.4
OP08 <2mm	Pit	Road Cut	7.0	1.77	50	48	0.28	74	309	5.06	0.06	15	1.14	754	0.01	46.5	1080	0.03	14.5	0.04	39	126	1	0.23
OP09 <2mm	Pit	Road Cut	5.0	2.69	50	94	0.2	119	125	5.69	0.1	20	1.95	976	0.01	71.7	870	0.01	11.8	0.05	63	115	2.3	0.07
OP10 <2mm	Pit	Road Cut	1.5	2.35	50	72	0.06	88	333	4.81	0.09	17	1.03	323	0.02	40.6	680	0.03	5.5	0.03	38	98	3.2	0.29
OP11 <2mm	Pit	Road Cut	8.7	2.31	60	72	0.35	138	4300	5.94	0.18	18	1.71	1030	0.02	69.5	1790	0.02	19.4	0.05	52	188	6.1	0.39
OP12 <2mm	Pit	Road Cut	4.5	1.74	50	56	0.18	175	684	4.32	0.11	14	0.68	490	0.05	33.6	670	0.03	13.2	0.05	36	98	1.7	0.87
OP13 <2mm	Pit	Road Cut	7.5	1.93	50	63	0.3	106	1580	5.58	0.15	17	1.32	909	0.02	81.3	1360	0.01	18.2	0.01	33	172	5.8	0.64
OP14 <2mm	Pit	Road Cut	2.0	1.66	50	57	0.08	108	187	5.34	0.06	13	0.68	458	0.03	43.4	560	0.02	7	0.05	50	82	1.3	0.25
OP15 <2mm	Pit	Road Cut	2.7	1.34	50	56	0.11	59	520	4.02	0.09	11	0.76	663	0.01	44.6	660	0.01	6.8	0.01	23	157	4.2	0.09
OP16 <2mm	Pit	Road Cut	11.0	2.59	60	57	0.44	146	961	6.83	0.12	20	1.83	1010	0.02	106	2450	0.01	23.9	0.03	55	168	4.8	0.25
OP12-01 Overburden Sample #1	Pit	Deep Overburden	97.8	2.69	20	60	3.92	186	156	4.8	0.14	#N/A	2.27	858	0.008	74.4	1710	0.19	137	0.054	61	92.4	#N/A	0.117
OP12-01 Overburden Sample #2	Pit	Deep Overburden	116.0	2.5	20	49.7	4.65	168	153	4.3	0.14	#N/A	2.23	810	0.009	71.5	1730	0.4	142	0.064	56	76.3	#N/A	0.14
OP12-02 Overburden Sample #1	Pit	Deep Overburden	31.4	1.84	20	53.3	1.26	124	292	4.22	0.14	#N/A	1.35	731	0.012	56.3	850	0.18	38.1	0.024	35	178	#N/A	0.374
OP12-02Overburden Sample #2	Pit	Deep Overburden	9.7	1.44	20	53.6	0.39	136	519	3.78	0.13	#N/A	0.86	470	0.015	44.1	560	0.08	16.4	0.015	24	102	#N/A	0.512
OP12-03 Overburden Sample #1	Pit	Deep Overburden	62.6	2.35	20	55.5	2.51	126	202	4.58	0.16	#N/A	1.77	802	0.007	72.7	1670	0.06	85	0.042	52	101	#N/A	0.15
OP12-05 Overburden Sample #1	Pit	Deep Overburden	5.5	2.38	20	39.8	0.22	102	117	3.07	0.07	#N/A	2.37	454	0.017	22.7	500	0.02	9.6	0.021	50	48	#N/A	0.034
OP12-06Overburden Sample #1	Pit	Deep Overburden	14.0	1.88	20	40.9	0.56	95.1	530	4.54	0.12	#N/A	1.39	932	0.021	41.3	910	0.18	17.8	0.045	38	129	#N/A	0.272
OP12-07 Overburden Sample #1	Pit	Deep Overburden	57.9	1.96	20	53.6	2.32	156	1180	4.41	0.15	#N/A	1.44	808	0.013	55.4	1250	0.09	68.6	0.036	42	109	#N/A	0.621
OP12-09 Overburden Sample #1	Pit	Deep Overburden	1.5	0.46	20	45.8	0.06	64.1	56.3	3.14	0.18	#N/A	0.12	425	0.012	32.3	340	0.03	4.3	0.001	4	144	#N/A	0.089
OP12-10 Overburden Sample #1	Pit	Deep Overburden	34.7	4.06	20	30	1.39	57.1	952	8.15	0.06	#N/A	3.18	994	0.003	39.6	6400	0.04	33.6	0.089	67	107	#N/A	0.654
OP12-01 Weathered Rock Sample #1	Pit	Weathered Bedrock	20.9	1.23	20	26.7	0.84	98	179	4.47	0.14	#N/A	1.26	404	0.019	10.3	230	2.91	31.1	0.001	15	770	#N/A	0.992
OP12-01 Weathered Rock Sample #2	Pit	Weathered Bedrock	44.1	1	20	56.9	1.77	88.5	250	5.38	0.13	#N/A	1.46	842	0.013	9.2	220	3.7	55.7	0.001	11	618	#N/A	1.51
OP12-02 Weathered Rock Sample #1	Pit	Weathered Bedrock	2.5	0.43	20	39.2	0.1	78.7	762	2.52	0.12	#N/A	0.16	147	0.014	17.9	410	0.08	7.5	0.005	8	46.5	#N/A	0.544
OP12-03 Weathered Rock Sample #1	Pit	Weathered Bedrock	0.7	0.49	20	35.9	0.03	114	282	2.59	0.13	#N/A	0.23	80	0.01	14.7	150	0.14	2.1	0.001	4	31.5	#N/A	0.147
OP12-04 Weathered Rock Sample #1	Pit	Weathered Bedrock	116.7	3.93	20	33.6	4.68	75	157	8.44	0.05	#N/A	2.9	1300	0.003	36.3	5240	1.77	136	0.162	112	111	#N/A	0.159
OP12-05 Weathered Rock Sample #1	Pit	Weathered Bedrock	19.5	1.67	20	24.6	0.78	120	26	1.73	0.09	#N/A	1.7	228	0.025	10.3	320	0.02	27.2	0.02	30	51.4	#N/A	0.019
OP12-05 Weathered Rock Sample #2	Pit	Weathered Bedrock	14.7	2.37	20	32.1	0.59	121	152	2.96	0.09	#N/A	2.29	369	0.021	15	290	0.06	20.4	0.011	39	221	#N/A	0.208
OP12-06 Weathered Rock Sample #1	Pit	Weathered Bedrock	3.0	2.25	20	32	0.12	80.1	451	9.31	0.18	#N/A	1.81	458	0.012	15	260	4.49	4.3	0.006	29	408	#N/A	0.695
OP12-07 Weathered Rock Sample #1	Pit	Weathered Bedrock	59.6	0.28	20	28.2	2.39	82.2	1710	1.51	0.11	#N/A	1.66	429	0.032	15.1	200	0.54	53.8	0.001	8	43.2	#N/A	0.545
OP12-08 Weathered Rock Sample #1	Pit	Weathered Bedrock	5.0	0.24	20	17.7	0.2	98.3	240	0.63	0.07	#N/A	0.16	170	0.047	10.7	90	0.11	12.6	0.001	2	15.1	#N/A	0.075
OP12-09 Weathered Rock Sample #1	Pit	Weathered Bedrock	2.2	0.31	20	41.2	0.09	107	72.4	2.02	0.16	#N/A	0.09	249	0.012	23.8	240	0.12	5	0.002	3	79.3	#N/A	0.115
OP12-09 Weathered Rock Sample #2	Pit	Weathered Bedrock	4.0	0.26	20	31.2	0.16	80.4	49.4	2.49	0.14	#N/A	0.6	336	0.009	25.4	270	0.23	6.4	0.002	5	89.1	#N/A	0.148
OP12-10 Weathered Rock Sample #1	Pit	Weathered Bedrock	105.5	3.29	20	7.8	4.23	140	6210	6.97	0.01	#N/A	2.56	849	0.013	87.1	3850	1.32	126	0.144	85	73.2	#N/A	1.71
TP01-1 <2mm	TMF	Overburden	4.7	2.4	50	85	0.19	70	87.7	3.02	0.21	27	0.81	535	0.03	12.1	460	0.02	12.2	0.09	46	118	1.9	0.15
TP5-1 <2mm	TMF	Overburden	11.0	2.49	60	213	0.44	65	149	4.24	0.65	17	1.35	995	0.06	14.2	580	0.02	19.7	0.1	66	137	2.7	0.12
TP12-1 <2mm	TMF	Overburden	9.7	1.96	50	140	0.39	73	91.7	3.68	0.34	15	0.89	556	0.06	11	510	0.03	17.2	0.07	51	78	2.6	0.19
TP12-2 <2mm	TMF	Overburden	17.7	1.82	50	141	0.71	70	88.5	3.35	0.41	15	0.95	625	0.06	14	490	0.19	22.4	0.05	44	85	3.6	0.2
TP15-4 <2mm	TMF	Overburden	34.2	1.88	50	152	1.37	57	123	3.82	0.36	15	1.09	789	0.04	18.7	450	0.21	28.2	0.02	35	81	4.3	0.2
TP2 <2mm	TMF	Overburden	12.7	2.16	50	172	0.51	64	83.1	3.33	0.53	17	1.03	652	0.07	10.6	450	0.03	24.9	0.07	51	90	3.4	0.16
TP26-1 <2mm	TMF	Overburden	23.4	1.89	50	147	0.94	69	90.6	3.49	0.4	17	1.03	708	0.05	17.7	470	0.14	28.3	0.04	42	70	4	0.19
TP27-1 <2mm	TMF	Overburden	4.5	1.17	40	81	0.18	53	94.4	2.05	0.29	13	0.59	440	0.03	5.2	340	0.01	9.3	0.05	26	45	1.5	0.23
TP29-1 <2mm	TMF	Overburden	6.0	0.77	30	74	0.24	66	533	4.61	0.28	27	0.3	1160	0.01	7.7	390	0.01	9	0.01	36	80	2.2	0.49
TP30-1 <2mm	TMF	Overburden	3.7	2.51	40	116	0.15	49	139	3.65	0.16	15	0.81	420	0.02	15.1	370	0.03	8.5	0.03	47	112	2.4	0.18
TP30-2 <2mm	TMF	Overburden	15.0	2.05	50	188	0.6	75	160	3.98	0.38	15	0.94	823	0.07	15.5	900	0.01	25.2	0.05	52	128	2.9	0.26
TP32-1 <2mm	TMF	Overburden	29.2	0.89	40	176	1.17	49	296	4.17	0.28	9	0.34	1200	0.02	7.2	290	0.01	18.7	0.01	30	106	2.2	5.05
TP51-3 <2mm	TMF	Overburden	2.2	0.45	50	42	0.09	94	151	9.03	0.15	2	0.13	638	0.02	83.9	880	0.03	9.6	0.01	1	108	23.3	0.35

Sample ID	Area	Type	As	Be	Bi	Cd	Ce	Co	Cs	Ga	Ge	Hf	Hg	In	La	Lu	Mo	Nb	Pb	Rb	Sb	Sc	Se
Units			ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Method Code			ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B
LOD			1	0.1	0.02	0.01	0.05	0.1	0.05	0.1	0.1	0.05	0.01	0.02	0.1	0.01	0.05	0.05	0.2	0.2	0.05	0.1	1
OP01 <2mm	Pit	Road Cut	34	0.3	1.06	0.32	25.3	15.2	0.78	6.6	0.1	0.05	0.02	0.2	12.1	0.04	2.7	1.06	91.1	7.2	0.3	3.7	2
OP02 <2mm	Pit	Road Cut	176	0.4	1.39	1.67	48.8	44.5	0.72	9.9	0.1	0.09	0.03	0.57	19.2	0.08	3.38	1.88	264	4.2	0.5	7.5	2
OP03 <2mm	Pit	Road Cut	66	0.3	0.97	0.42	47	27.4	0.35	6.9	0.1	0.06	0.02	0.26	28.2	0.11	2.75	1.34	88.7	3.6	0.81	4.1	2
OP04 <2mm	Pit	Road Cut	53	0.5	1.03	0.32	56.2	34.9	0.79	8.1	0.1	0.06	0.02	0.18	23.9	0.07	3.03	3.72	46.8	6	0.52	4.7	1
OP05 <2mm	Pit	Road Cut	60	0.5	1.08	0.28	48.7	30.3	0.85	7.3	0.1	0.07	0.02	0.14	22.5	0.07	2.84	4.15	34.4	5.6	0.45	3.6	2
OP06 <2mm	Pit	Road Cut	94	0.6	0.64	0.27	63.8	33.7	1.61	8.5	0.1	0.05	0.03	0.08	27.4	0.12	1.99	3.61	26.1	11.2	0.44	4.3	1
OP07 <2mm	Pit	Road Cut	49	0.5	0.65	0.46	48.4	24.9	2.63	9.2	0.1	0.07	0.06	0.07	20.7	0.09	2.25	4.33	28	12	0.33	3.6	1
OP08 <2mm	Pit	Road Cut	53	0.3	0.65	0.31	49.1	27.1	1	6.2	0.1	0.05	0.02	0.06	20	0.05	4.78	1.37	39	5.9	0.26	2.3	1
OP09 <2mm	Pit	Road Cut	43	0.4	0.37	0.28	58.3	28.3	0.64	8.7	0.1	0.05	0.03	0.05	26.8	0.08	2.24	1.1	25.1	6.1	0.18	5.7	1
OP10 <2mm	Pit	Road Cut	26	0.4	0.62	0.14	33.1	16.4	0.76	5.7	0.1	0.05	0.04	0.08	16.9	0.03	1.93	1.59	24.2	8.3	0.22	3	1
OP11 <2mm	Pit	Road Cut	55	0.4	0.6	0.56	58.1	28.5	0.96	7.4	0.1	0.1	0.02	0.08	30.6	0.12	4.05	0.79	34.1	7.8	0.19	5.7	1
OP12 <2mm	Pit	Road Cut	23	0.3	0.91	0.27	33.8	16.5	0.72	6.1	0.1	0.05	0.03	0.12	16.7	0.06	3.87	1.46	27.5	6	0.15	1.8	1
OP13 <2mm	Pit	Road Cut	38	0.2	0.85	0.54	41.5	27.9	0.79	5.8	0.1	0.1	0.03	0.13	22.6	0.08	5.76	0.34	29.8	6.3	0.19	4.3	1
OP14 <2mm	Pit	Road Cut	22	0.2	0.74	0.19	27.1	14.3	0.81	8	0.1	0.05	0.03	0.07	13.3	0.03	2.17	1.21	18.9	6	0.17	1.7	1
OP15 <2mm	Pit	Road Cut	24	0.3	0.48	0.38	51.7	16	0.63	4.3	0.1	0.08	0.01	0.04	26.2	0.06	3.07	0.45	41.7	5.8	0.13	2.8	1
OP16 <2mm	Pit	Road Cut	35	0.3	0.5	0.4	59.7	31.3	1.26	8.5	0.1	0.1	0.02	0.13	28.8	0.1	3.88	0.66	35	6.9	0.23	5.2	1
OP12-01 Overburden Sample #1	Pit	Deep Overburden	25.4	#N/A	0.29	0.26	#N/A	29.9	#N/A	8.5	#N/A	#N/A	0.005	#N/A	#N/A	20.1	4.13	#N/A	22.9	#N/A	0.16	6	0.4
OP12-01 Overburden Sample #2	Pit	Deep Overburden	19.5	#N/A	0.34	0.22	#N/A	27	#N/A	8.2	#N/A	#N/A	0.005	#N/A	#N/A	18.7	2.56	#N/A	35.9	#N/A	0.17	5.7	0.7
OP12-02 Overburden Sample #1	Pit	Deep Overburden	92	#N/A	0.59	0.93	#N/A	22.4	#N/A	5.7	#N/A	#N/A	0.005	#N/A	#N/A	14.9	3.11	#N/A	91.8	#N/A	0.26	4.2	0.8
OP12-02Overburden Sample #2	Pit	Deep Overburden	30.9	#N/A	0.66	0.38	#N/A	18.1	#N/A	4.4	#N/A	#N/A	0.005	#N/A	#N/A	17.2	5.79	#N/A	41.2	#N/A	0.22	3.2	0.7
OP12-03 Overburden Sample #1	Pit	Deep Overburden	45.2	#N/A	0.25	0.28	#N/A	28.4	#N/A	7.9	#N/A	#N/A	0.005	#N/A	#N/A	20.6	2.34	#N/A	21.9	#N/A	0.16	5.5	0.3
OP12-05 Overburden Sample #1	Pit	Deep Overburden	22.6	#N/A	0.09	0.09	#N/A	13.4	#N/A	6.6	#N/A	#N/A	0.005	#N/A	#N/A	20.2	1.24	#N/A	10.5	#N/A	0.14	6.8	0.2
OP12-06Overburden Sample #1	Pit	Deep Overburden	24	#N/A	0.32	0.43	#N/A	22	#N/A	5.1	#N/A	#N/A	0.012	#N/A	#N/A	12.6	3.7	#N/A	41.3	#N/A	0.22	5.1	1
OP12-07 Overburden Sample #1	Pit	Deep Overburden	26.9	#N/A	0.31	0.37	#N/A	24.2	#N/A	6.2	#N/A	#N/A	0.005	#N/A	#N/A	18.5	4.93	#N/A	15.3	#N/A	0.12	4.5	0.9
OP12-09 Overburden Sample #1	Pit	Deep Overburden	10.5	#N/A	0.31	0.36	#N/A	14.4	#N/A	1.3	#N/A	#N/A	0.005	#N/A	#N/A	22.6	0.79	#N/A	34.1	#N/A	0.03	1.4	0.1
OP12-10 Overburden Sample #1	Pit	Deep Overburden	9.2	#N/A	0.08	0.17	#N/A	39.9	#N/A	17.3	#N/A	#N/A	0.005	#N/A	#N/A	17	5.61	#N/A	35.8	#N/A	0.18	3.3	0.7
OP12-01 Weathered Rock Sample #1	Pit	Weathered Bedrock	1010	#N/A	2.44	4.7	#N/A	12.8	#N/A	3.1	#N/A	#N/A	0.005	#N/A	#N/A	3.7	1.51	#N/A	191	#N/A	1.84	2.7	1.4
OP12-01 Weathered Rock Sample #2	Pit	Weathered Bedrock	1360	#N/A	3.55	3.85	#N/A	13.8	#N/A	2.3	#N/A	#N/A	0.005	#N/A	#N/A	2.4	1.27	#N/A	244	#N/A	1.88	2.5	2.4
OP12-02 Weathered Rock Sample #1	Pit	Weathered Bedrock	16.2	#N/A	0.63	0.31	#N/A	8.1	#N/A	1.3	#N/A	#N/A	0.005	#N/A	#N/A	11.8	8.83	#N/A	10.6	#N/A	0.13	1.2	0.8
OP12-03 Weathered Rock Sample #1	Pit	Weathered Bedrock	40.9	#N/A	0.74	0.1	#N/A	4.2	#N/A	1.5	#N/A	#N/A	0.005	#N/A	#N/A	12.7	1.48	#N/A	4.52	#N/A	0.1	0.9	1.5
OP12-04 Weathered Rock Sample #1	Pit	Weathered Bedrock	36.3	#N/A	0.4	0.24	#N/A	38.3	#N/A	16.7	#N/A	#N/A	0.005	#N/A	#N/A	19.3	2.01	#N/A	40.1	#N/A	0.57	6.2	0.9
OP12-05 Weathered Rock Sample #1	Pit	Weathered Bedrock	26.8	#N/A	0.03	0.08	#N/A	5.9	#N/A	4.1	#N/A	#N/A	0.005	#N/A	#N/A	9.2	0.81	#N/A	5.76	#N/A	0.06	6.2	0.1
OP12-05 Weathered Rock Sample #2	Pit	Weathered Bedrock	164	#N/A	0.26	0.81	#N/A	11.9	#N/A	5.5	#N/A	#N/A	0.006	#N/A	#N/A	10.8	2.69	#N/A	113	#N/A	0.19	7.7	0.1
OP12-06 Weathered Rock Sample #1	Pit	Weathered Bedrock	99.7	#N/A	2.2	2.19	#N/A	25.8	#N/A	4.9	#N/A	#N/A	0.005	#N/A	#N/A	2	2.4	#N/A	93	#N/A	0.43	4.9	5.6
OP12-07 Weathered Rock Sample #1	Pit	Weathered Bedrock	0.4	#N/A	0.52	0.26	#N/A	7	#N/A	0.9	#N/A	#N/A	0.005	#N/A	#N/A	1.5	12.5	#N/A	6.67	#N/A	0.04	2.8	3.9
OP12-08 Weathered Rock Sample #1	Pit	Weathered Bedrock	1	#N/A	0.16	0.13	#N/A	2.1	#N/A	0.4	#N/A	#N/A	0.005	#N/A	#N/A	2.6	9.31	#N/A	3.31	#N/A	0.02	0.5	0.5
OP12-09 Weathered Rock Sample #1	Pit	Weathered Bedrock	19.2	#N/A	0.21	0.26	#N/A	11.6	#N/A	0.8	#N/A	#N/A	0.005	#N/A	#N/A	16.8	0.89	#N/A	25.4	#N/A	0.04	0.9	0.1
OP12-09 Weathered Rock Sample #2	Pit	Weathered Bedrock	14.2	#N/A	0.36	0.24	#N/A	11.7	#N/A	0.7	#N/A	#N/A	0.01	#N/A	#N/A	9.8	1.03	#N/A	21.6	#N/A	0.04	0.7	0.1
OP12-10 Weathered Rock Sample #1	Pit	Weathered Bedrock	8.7	#N/A	0.04	0.69	#N/A	34.6	#N/A	16.2	#N/A	#N/A	0.043	#N/A	#N/A	12.7	2.96	#N/A	9.54	#N/A	0.24	5.1	8.1
TP01-1 <2mm	TMF	Overburden	16	0.4	0.32	0.27	58	10.3	2.59	5.8	0.1	0.1	0.03	0.04	31.9	0.21	0.79	2.64	177	23.6	0.24	5	1
TP5-1 <2mm	TMF	Overburden	9	0.3	0.45	0.29	26.9	14	2.93	6.9	0.1	0.08	0.02	0.06	14.6	0.11	0.97	0.48	450	41.5	0.19	8.4	1
TP12-1 <2mm	TMF	Overburden	17	0.3	0.5	0.12	25.2	10.8	2.4	5.5	0.1	0.05	0.01	0.03	13.4	0.11	0.79	1.15	790	24.4	0.24	6.4	1
TP12-2 <2mm	TMF	Overburden	18	0.3	0.5	0.27	24.5	11.9	2.47	5.1	0.1	0.09	0.01	0.03	13.2	0.09	1.2	0.34	1480	22.6	0.29	5.8	1
TP15-4 <2mm	TMF	Overburden	11	0.4	0.72	0.23	27.3	14.3	1.69	4.8	0.1	0.11	0.01	0.03	14.6	0.11	0.97	0.14	1090	17	0.12	4.7	1
TP2 <2mm	TMF	Overburden	16	0.3	0.47	0.24	27	11.2	2.78	6	0.1	0.09	0.01	0.03	14.7	0.1	1.03	0.22	1720	29.6	0.21	6.8	1
TP26-1 <2mm	TMF	Overburden	9	0.3	0.51	0.17	28.3	12.1	1.85	5.1	0.1	0.09	0.01	0.02	15.2	0.09	1.06	0.23	1310	21.6	0.12	5	1
TP27-1 <2mm	TMF	Overburden	34	0.3	0.32	0.15	23.9	6.8	1.93	3.3	0.1	0.05	0.01	0.04	13.4	0.05	0.52	0.54	193	20.9	1.59	3.5	1
TP29-1 <2mm	TMF	Overburden	325	1.1	1.07	0.28	37.5	28.1	8.41	2.2	0.1	0.05	0.01	0.07	23.2	0.15	1.68	0.11	1880	20.1	16.3	12.1	1
TP30-1 <2mm	TMF	Overburden	27	0.5	0.58	0.23	26.3	12.5	3.55	5.3	0.1	0.05	0.04	0.06	11.9	0.06	0.83	1.3	22	16	0.6	5.7	1
TP30-2 <2mm	TMF	Overburden	30	0.5	0.61	0.41	32.9	15.4	4.03	5.4	0.1	0.08	0.01	0.06	18.2	0.13	0.87	0.33	39.1	23.9	0.61	6.9	1
TP32-1 <2mm	TMF	Overburden	503	1.6	0.61	0.45	43.2	12.9	8.01	2.9	0.1	0.05	0.03	0.05	25.6	0.17	2.59	0.22	98.2	17.6	15.1	8.1	1
TP51-3 <2mm	TMF	Overburden	23	0.4	3.86	0.29	65.2	25.4	0.9	1.9	0.1	0.45	0.02	0.02	35.8	0.09	2.58	0.07	20.3	6.3	0.41	1.5	1

Sample ID	Area	Type	Sn	Ta	Tb	Te	Th	Tl	U	W	Y	Yb	Au	Hg on Solids
Units			ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppb	mg/kg
Method Code			ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B		
LOD			0.3	0.05	0.02	0.05	0.1	0.02	0.05	0.1	0.05	0.1		
OP01 <2mm	Pit	Road Cut	0.8	0.05	0.23	0.54	3.2	0.05	0.52	0.1	3.08	0.3	#N/A	#N/A
OP02 <2mm	Pit	Road Cut	1.5	0.05	0.44	0.71	3.3	0.05	0.58	0.1	7	0.6	#N/A	#N/A
OP03 <2mm	Pit	Road Cut	0.9	0.05	0.63	0.57	3.6	0.04	0.69	0.1	11.4	0.8	#N/A	#N/A
OP04 <2mm	Pit	Road Cut	0.9	0.05	0.53	0.32	3.3	0.07	0.57	0.2	7.76	0.6	#N/A	#N/A
OP05 <2mm	Pit	Road Cut	0.7	0.05	0.47	0.43	3	0.07	0.57	0.2	7.74	0.6	#N/A	#N/A
OP06 <2mm	Pit	Road Cut	0.6	0.05	0.61	0.29	3.3	0.13	0.6	0.2	10.9	0.9	#N/A	#N/A
OP07 <2mm	Pit	Road Cut	0.7	0.05	0.44	0.2	1.5	0.12	0.77	0.2	7.81	0.7	#N/A	#N/A
OP08 <2mm	Pit	Road Cut	0.3	0.05	0.34	0.28	2	0.05	0.75	0.1	5.01	0.4	#N/A	#N/A
OP09 <2mm	Pit	Road Cut	0.4	0.05	0.51	0.11	5.5	0.08	1.32	0.1	7.55	0.6	#N/A	#N/A
OP10 <2mm	Pit	Road Cut	0.6	0.05	0.26	0.24	4.3	0.06	0.63	0.1	3.23	0.3	#N/A	#N/A
OP11 <2mm	Pit	Road Cut	0.5	0.05	0.63	0.27	7.5	0.08	0.91	0.1	10	0.8	#N/A	#N/A
OP12 <2mm	Pit	Road Cut	0.8	0.05	0.31	0.43	1.7	0.05	1.01	0.2	5.12	0.4	#N/A	#N/A
OP13 <2mm	Pit	Road Cut	0.5	0.05	0.48	0.43	7.2	0.06	1.33	0.1	7.89	0.6	#N/A	#N/A
OP14 <2mm	Pit	Road Cut	0.5	0.05	0.2	0.11	0.8	0.06	0.62	0.2	2.55	0.2	#N/A	#N/A
OP15 <2mm	Pit	Road Cut	0.3	0.05	0.4	0.09	8.8	0.08	1.74	0.1	5.25	0.4	#N/A	#N/A
OP16 <2mm	Pit	Road Cut	0.4	0.05	0.68	0.11	5.6	0.1	1.24	0.1	10.2	0.7	#N/A	#N/A
OP12-01 Overburden Sample #1	Pit	Deep Overburden	#N/A	#N/A	#N/A	0.2	4.1	0.05	0.3	0.1	#N/A	#N/A	6.8	0.01
OP12-01 Overburden Sample #2	Pit	Deep Overburden	#N/A	#N/A	#N/A	0.23	4.2	0.05	0.4	0.2	#N/A	#N/A	5.3	0.01
OP12-02 Overburden Sample #1	Pit	Deep Overburden	#N/A	#N/A	#N/A	0.23	5.5	0.07	0.9	0.3	#N/A	#N/A	11.8	0.01
OP12-02Overburden Sample #2	Pit	Deep Overburden	#N/A	#N/A	#N/A	0.6	7.1	0.06	0.9	0.2	#N/A	#N/A	19.7	0.01
OP12-03 Overburden Sample #1	Pit	Deep Overburden	#N/A	#N/A	#N/A	0.1	5.2	0.07	0.5	0.2	#N/A	#N/A	7.1	0.01
OP12-05 Overburden Sample #1	Pit	Deep Overburden	#N/A	#N/A	#N/A	0.06	5.3	0.04	0.5	0.1	#N/A	#N/A	3	0.01
OP12-06Overburden Sample #1	Pit	Deep Overburden	#N/A	#N/A	#N/A	0.12	4.2	0.04	0.6	0.1	#N/A	#N/A	6.6	0.01
OP12-07 Overburden Sample #1	Pit	Deep Overburden	#N/A	#N/A	#N/A	0.27	5.2	0.07	0.5	0.1	#N/A	#N/A	7	0.01
OP12-09 Overburden Sample #1	Pit	Deep Overburden	#N/A	#N/A	#N/A	0.02	10.6	0.07	1.2	0.1	#N/A	#N/A	2.5	0.01
OP12-10 Overburden Sample #1	Pit	Deep Overburden	#N/A	#N/A	#N/A	0.19	2.5	0.02	0.2	0.4	#N/A	#N/A	15.6	0.01
OP12-01 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	0.7	2	0.06	0.4	0.5	#N/A	#N/A	114	0.01
OP12-01 Weathered Rock Sample #2	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	0.35	1.5	0.09	0.4	0.8	#N/A	#N/A	109	0.014
OP12-02 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	0.55	6	0.04	0.9	1	#N/A	#N/A	6.2	0.01
OP12-03 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	0.45	5.8	0.04	0.4	0.6	#N/A	#N/A	3.7	0.01
OP12-04 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	0.11	1.5	0.04	0.1	0.3	#N/A	#N/A	9.4	0.01
OP12-05 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	0.02	4.1	0.02	0.4	0.1	#N/A	#N/A	1	0.01
OP12-05 Weathered Rock Sample #2	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	0.17	3.8	0.03	0.6	0.1	#N/A	#N/A	9.2	0.014
OP12-06 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	0.5	1.1	0.09	0.4	0.2	#N/A	#N/A	34.2	0.01
OP12-07 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	0.86	1.4	0.03	0.3	0.5	#N/A	#N/A	20.5	0.01
OP12-08 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	0.35	5	0.02	0.1	0.4	#N/A	#N/A	5.5	0.01
OP12-09 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	0.04	8.5	0.05	0.9	0.5	#N/A	#N/A	6.1	0.01
OP12-09 Weathered Rock Sample #2	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	0.04	4.9	0.04	1.2	0.7	#N/A	#N/A	0.4	0.01
OP12-10 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	0.41	2.8	0.02	0.2	6.7	#N/A	#N/A	58.9	0.03
TP01-1 <2mm	TMF	Overburden	0.7	0.05	0.56	0.05	3.6	0.2	4.76	0.5	13.9	1.4	#N/A	#N/A
TP5-1 <2mm	TMF	Overburden	0.9	0.05	0.29	0.08	6.5	0.29	0.98	0.5	7.01	0.7	#N/A	#N/A
TP12-1 <2mm	TMF	Overburden	0.9	0.05	0.27	0.06	6.4	0.18	0.96	0.5	6.64	0.7	#N/A	#N/A
TP12-2 <2mm	TMF	Overburden	0.7	0.05	0.26	0.06	6.4	0.17	1	0.4	5.93	0.6	#N/A	#N/A
TP15-4 <2mm	TMF	Overburden	0.5	0.05	0.28	0.15	6.7	0.14	0.96	0.3	6.6	0.7	#N/A	#N/A
TP2 <2mm	TMF	Overburden	0.9	0.05	0.27	0.06	6.7	0.21	1.19	0.5	6.24	0.6	#N/A	#N/A
TP26-1 <2mm	TMF	Overburden	0.6	0.05	0.26	0.05	6.9	0.16	1.09	0.5	5.68	0.6	#N/A	#N/A
TP27-1 <2mm	TMF	Overburden	0.7	0.05	0.18	0.05	7	0.15	1.04	0.5	4	0.4	#N/A	#N/A
TP29-1 <2mm	TMF	Overburden	1.6	0.05	0.4	0.1	7.2	0.71	2.75	1.1	9.78	1	#N/A	#N/A
TP30-1 <2mm	TMF	Overburden	0.9	0.05	0.21	0.08	4.5	0.14	0.65	0.5	4.35	0.4	#N/A	#N/A
TP30-2 <2mm	TMF	Overburden	1	0.05	0.37	0.06	6	0.21	0.8	0.4	8.63	0.9	#N/A	#N/A
TP32-1 <2mm	TMF	Overburden	0.7	0.05	0.41	0.05	8.9	0.18	1.5	1.3	11.4	1.2	#N/A	#N/A
TP51-3 <2mm	TMF	Overburden	0.5	0.05	0.47	0.29	12.1	0.24	3.87	0.1	4.87	0.6	#N/A	#N/A

E2: Shake Flask Extraction Test Results

Shake Flask Extraction

Sample ID	**Area**	**Type**	**Sample Weight**	**Volume Used**	**pH**	**EC**	**ORP**	**SO4**	**Acidity to pH4.5**	**Acidity to pH8.3**	**Total Alkalinity**	**Bicarbonate**	**Carbonate**	**Hydroxide**	**Fluoride**	**Dissolved Chloride**	**Hardness CaCO3**	**Dissolved Aluminum (Al)**	**Dissolved Antimony (Sb)**
Units			**g**	**mL**	**pH Units**	**uS/cm**	**mV**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**
Method Code								BBY6SOP-00017	BBY6SOP-00037		BBY6SOP-00026, BBY0SOP-00002				BBY6SOP-00038	BBY6SOP-00011	Calculation		
LOD					N/A	0.5		0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.01	0.5	0.5	0.0002	0.00002
OP02 <2mm	Pit	Road Cut	250	750	5.92	36.2	170	3	0.5	2.6	7.0	8.6	0.50	0.50	0.0	1.3	12.1	0.352	0.000058
OP07 <2mm	Pit	Road Cut	250	750	5.82	90.6	150	0.5	0.5	9.1	28	34	0.50	0.50	0.1	2.1	41.7	0.934	0.000162
OP11 <2mm	Pit	Road Cut	250	750	6.09	24.1	210	1.1	0.5	1	8	9.8	0.5	0.5	0.03	0.6	8.6	0.337	0.000037
OP14 <2mm	Pit	Road Cut	250	750	4.91	63.8	160	0.5	0.5	17.5	11	14	0.5	0.5	0.04	1.4	22	1.68	0.000096
OP12-01 Overburden Sample #1	Pit	Deep Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP12-10 Overburden Sample #1	Pit	Deep Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP12-02 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP12-06 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP12-07 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
TP15-4 <2mm	TMF	Overburden	250	750	8.49	234	150	56.7	0.5	0.9	26	31	0.5	0.5	0.1	1	88.7	0.0688	0.000233
TP29-1 <2mm	TMF	Overburden	250	750	7.58	11.2	155	0.9	0.5	0.5	3	3.7	0.5	0.5	0.09	0.5	2.09	0.0717	0.00127
TP30-1 <2mm	TMF	Overburden	250	750	5.19	52.1	160	0.5	0.5	8.9	12	15	0.5	0.5	0.03	1.5	21.1	1.1	0.000115
TP30-2 <2mm	TMF	Overburden	250	750	8.59	95.8	120	2.7	0.5	0.5	42	51	0.5	0.5	0.19	0.8	38.6	0.0993	0.00369

Sample ID	Area	Type	Dissolved Arsenic (As)	Dissolved Barium (Ba)	Dissolved Beryllium (Be)	Dissolved Bismuth (Bi)	Dissolved Boron (B)	Dissolved Cesium (Cs)	Dissolved Cadmium (Cd)	Dissolved Calcium (Ca)	Dissolved Chromium (Cr)	Dissolved Cobalt (Co)	Dissolved Copper (Cu)	Dissolved Lanthanum (La)	Dissolved Iron (Fe)	Dissolved Lead (Pb)
Units			mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
Method Code																
LOD			0.00002	0.00002	0.00001	0.000005	0.05	0.00005	0.000005	0.05	0.0001	0.000005	0.00005	0.00005	0.001	0.000005
OP02 <2mm	Pit	Road Cut	0.00173	0.0135	0.000015	0.000005	0.05	0.00005	0.000339	2.9	0.00042	0.00311	0.171	0.000432	0.203	0.00176
OP07 <2mm	Pit	Road Cut	0.00267	0.0429	0.00005	0.000005	0.05	0.00005	0.000271	15	0.00052	0.00302	0.0264	0.000609	0.17	0.000245
OP11 <2mm	Pit	Road Cut	0.00102	0.0113	0.000016	0.000005	0.05	0.000072	0.000229	2.2	0.00084	0.000967	0.674	0.000379	0.407	0.000658
OP14 <2mm	Pit	Road Cut	0.00141	0.0422	0.000052	0.000005	0.05	0.00005	0.000435	7.06	0.00154	0.00645	0.0468	0.000468	0.949	0.000341
OP12-01 Overburden Sample #1	Pit	Deep Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP12-10 Overburden Sample #1	Pit	Deep Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP12-02 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP12-06 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP12-07 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
TP15-4 <2mm	TMF	Overburden	0.000809	0.0474	0.00001	0.000005	0.05	0.00005	0.0000649	32.7	0.0001	0.0000906	0.0043	0.000068	0.0477	0.000461
TP29-1 <2mm	TMF	Overburden	0.00335	0.00155	0.00001	0.000005	0.05	0.00008	0.0000055	0.706	0.00077	0.0000195	0.00111	0.00005	0.0354	0.000253
TP30-1 <2mm	TMF	Overburden	0.0014	0.0798	0.000065	0.000005	0.05	0.000066	0.000294	6.9	0.00042	0.00408	0.0101	0.000367	0.229	0.000457
TP30-2 <2mm	TMF	Overburden	0.0351	0.0102	0.00001	0.000005	0.05	0.000248	0.0000466	13.2	0.00054	0.0000256	0.00237	0.00005	0.0341	0.000479

Sample ID	**Area**	**Type**	**Dissolved Lithium (Li)**	**Dissolved Magnesium (Mg)**	**Dissolved Manganese (Mn)**	**Dissolved Phosphorus (P)**	**Dissolved Molybdenum (Mo)**	**Dissolved Nickel (Ni)**	**Dissolved Potassium (K)**	**Dissolved Rubidium (Rb)**	**Dissolved Selenium (Se)**	**Dissolved Silicon (Si)**	**Dissolved Silver (Ag)**	**Dissolved Sodium (Na)**	**Dissolved Strontium (Sr)**	**Dissolved Sulphur (S)**
Units			**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**
Method Code			BBY7SOP-00002													
LOD			0.0005	0.05	0.00005	0.002	0.00005	0.00002	0.05	0.00005	0.00004	0.1	0.000005	0.05	0.00005	10
OP02 <2mm	Pit	Road Cut	0.0005	1.17	0.0955	0.0534	0.000077	0.00389	0.992	0.00334	0.000892	1.69	0.0000141	0.812	0.0117	10
OP07 <2mm	Pit	Road Cut	0.0005	0.999	0.449	0.183	0.000279	0.004	0.989	0.00452	0.00199	3.2	0.0000346	1.16	0.0533	10
OP11 <2mm	Pit	Road Cut	0.00112	0.756	0.0901	0.0246	0.00007	0.00274	0.827	0.00219	0.000272	2.04	0.0000093	0.227	0.0105	10
OP14 <2mm	Pit	Road Cut	0.00167	1.05	0.373	0.127	0.00005	0.00667	0.498	0.00148	0.000534	3.65	0.0000218	2.32	0.0267	10
OP12-01 Overburden Sample #1	Pit	Deep Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP12-10 Overburden Sample #1	Pit	Deep Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP12-02 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP12-06 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP12-07 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
TP15-4 <2mm	TMF	Overburden	0.00101	1.69	0.0514	0.0321	0.000693	0.000485	2.83	0.000651	0.00675	1.49	0.0000076	1.2	0.0886	23
TP29-1 <2mm	TMF	Overburden	0.00305	0.079	0.00315	0.0387	0.000306	0.000314	0.963	0.000312	0.00005	1.78	0.000005	0.576	0.00191	10
TP30-1 <2mm	TMF	Overburden	0.0005	0.935	0.28	0.0673	0.000053	0.00154	0.508	0.00398	0.000305	4.25	0.0000073	0.67	0.0337	10
TP30-2 <2mm	TMF	Overburden	0.00171	1.36	0.00162	0.0259	0.00876	0.000279	4.12	0.00313	0.000155	2.17	0.0000057	1.67	0.0342	10

														MWMP		
Sample ID	**Area**	**Type**	**Dissolved Tellurium (Te)**	**Dissolved Thallium (Tl)**	**Dissolved Thorium (Th)**	**Dissolved Tin (Sn)**	**Dissolved Titanium (Ti)**	**Dissolved Tungsten (W)**	**Dissolved Uranium (U)**	**Dissolved Vanadium (V)**	**Dissolved Zinc (Zn)**	**Dissolved Zirconium (Zr)**	**Dissolved Mercury (Hg)**	**Sample Weight**	**Volume In (DI Water)**	**Volume Out (extract)**
Units			**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**kg**	**mL**	**mL**
Method Code																
LOD			0.00002	0.000002	0.000005	0.0002	0.0005	0.00001	0.000002	0.0002	0.0001	0.0001	0.00005			
OP02 <2mm	Pit	Road Cut	0.00002	0.0000208	0.000143	0.00026	0.00131	0.000012	0.0000829	0.00039	0.0301	0.00157	0.00005	#N/A	#N/A	#N/A
OP07 <2mm	Pit	Road Cut	0.000037	0.0000556	0.000195	0.0002	0.00397	0.000024	0.000145	0.00052	0.00687	0.00453	0.00005	#N/A	#N/A	#N/A
OP11 <2mm	Pit	Road Cut	0.00005	0.0000101	0.0000316	0.00025	0.00998	0.00001	0.0000214	0.00068	0.014	0.00033	0.00005	#N/A	#N/A	#N/A
OP14 <2mm	Pit	Road Cut	0.000033	0.0000099	0.000253	0.0002	0.00662	0.00001	0.000171	0.0004	0.0234	0.00533	0.00005	#N/A	#N/A	#N/A
OP12-01 Overburden Sample #1	Pit	Deep Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	5.4	5300	5015
OP12-10 Overburden Sample #1	Pit	Deep Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	5.4	5300	5115
OP12-02 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	5.7	5300	5008
OP12-06 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	5.1	5050	5000
OP12-07 Weathered Rock Sample #1	Pit	Weathered Bedrock	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	5.4	5300	5180
TP15-4 <2mm	TMF	Overburden	0.00002	0.0000049	0.000005	0.0002	0.00077	0.00023	0.000476	0.00034	0.00441	0.0001	0.00005	#N/A	#N/A	#N/A
TP29-1 <2mm	TMF	Overburden	0.00002	0.0000088	0.000005	0.0002	0.0005	0.000021	0.0000072	0.0002	0.00491	0.0001	0.00005	#N/A	#N/A	#N/A
TP30-1 <2mm	TMF	Overburden	0.00002	0.0000289	0.000254	0.0002	0.00223	0.000012	0.000145	0.00065	0.0178	0.00135	0.00005	#N/A	#N/A	#N/A
TP30-2 <2mm	TMF	Overburden	0.00002	0.0000066	0.0000059	0.0002	0.00063	0.00084	0.000322	0.0002	0.00543	0.0001	0.00005	#N/A	#N/A	#N/A

Sample ID	Area	Type	Moisture	pH	EC	ORP	SO4	Total Alkalinity	Fluoride	Dissolved Chloride	Nitrate-N	Nitrite-N	Dissolved Phosphorus (P)	Hardness CaCO3	Dissolved Aluminum (Al)	Dissolved Antimony (Sb)
Units			%	pH Units	uS/cm	mV	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
Method Code							BBY6SOP-00017	BBY6SOP-00026, BBY0SOP-00002	BBY6SOP-00038	BBY6SOP-00011	BBY6SOP-00010		BBY6SOP-00013			
LOD				N/A	0.5		0.5	0.5	0.01	0.5	0.02	0.005	0.005	0.5	0.0002	0.00002
OP02 <2mm	Pit	Road Cut	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP07 <2mm	Pit	Road Cut	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP11 <2mm	Pit	Road Cut	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP14 <2mm	Pit	Road Cut	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP12-01 Overburden Sample #1	Pit	Deep Overburden	6.6	7.17	978	240	99.2	193.6	0.11	11.4	0.2	0.05	0.128	529	0.006	0.0017
OP12-10 Overburden Sample #1	Pit	Deep Overburden	6.7	5.46	479	270	6.9	81	0.03	1.7	0.02	0.009	0.082	208	0.065	0.0002
OP12-02 Weathered Rock Sample #1	Pit	Weathered Bedrock	11.8	6.9	217	190	53.6	39	0.08	0.7	0.02	0.005	0.169	82.8	0.0459	0.000133
OP12-06 Weathered Rock Sample #1	Pit	Weathered Bedrock	2.3	6.77	695	250	221	43.5	0.14	2.2	0.02	0.008	0.008	314	0.0022	0.00027
OP12-07 Weathered Rock Sample #1	Pit	Weathered Bedrock	7.7	7.68	159.3	225	14.1	48.2	0.48	2.4	0.02	0.005	0.186	46.3	0.18	0.00138
TP15-4 <2mm	TMF	Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
TP29-1 <2mm	TMF	Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
TP30-1 <2mm	TMF	Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
TP30-2 <2mm	TMF	Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A

Sample ID	**Area**	**Type**	**Dissolved Arsenic (As)**	**Dissolved Barium (Ba)**	**Dissolved Beryllium (Be)**	**Dissolved Bismuth (Bi)**	**Dissolved Boron (B)**	**Dissolved Cesium (Cs)**	**Dissolved Cadmium (Cd)**	**Dissolved Calcium (Ca)**	**Dissolved Chromium (Cr)**	**Dissolved Cobalt (Co)**	**Dissolved Copper (Cu)**	**Dissolved Lanthanum (La)**	**Dissolved Iron (Fe)**	**Dissolved Lead (Pb)**
Units			**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**
Method Code																
LOD			0.00002	0.00002	0.00001	0.000005	0.05	0.00005	0.000005	0.05	0.0001	0.000005	0.00005	0.00005	0.001	0.000005
OP02 <2mm	Pit	Road Cut	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP07 <2mm	Pit	Road Cut	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP11 <2mm	Pit	Road Cut	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP14 <2mm	Pit	Road Cut	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP12-01 Overburden Sample #1	Pit	Deep Overburden	0.00189	0.038	0.000116	0.000025	0.25	0.00025	0.000295	179	0.0005	0.0173	0.076	0.00025	0.0137	0.000165
OP12-10 Overburden Sample #1	Pit	Deep Overburden	0.00054	0.0679	0.0001	0.00005	0.5	0.0005	0.0013	47.5	0.001	0.463	1.06	0.00128	0.037	0.000364
OP12-02 Weathered Rock Sample #1	Pit	Weathered Bedrock	0.00135	0.0169	0.00001	0.000005	0.05	0.00005	0.000062	30	0.0001	0.000932	0.0068	0.00005	0.0755	0.000129
OP12-06 Weathered Rock Sample #1	Pit	Weathered Bedrock	0.00063	0.0132	0.00005	0.000025	0.25	0.00025	0.00749	77.9	0.0005	0.0208	0.0172	0.00025	0.005	0.000086
OP12-07 Weathered Rock Sample #1	Pit	Weathered Bedrock	0.00146	0.00311	0.00001	0.000013	0.05	0.000373	0.000007	7.9	0.00025	0.00024	0.00665	0.00005	0.138	0.000234
TP15-4 <2mm	TMF	Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
TP29-1 <2mm	TMF	Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
TP30-1 <2mm	TMF	Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
TP30-2 <2mm	TMF	Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A

Sample ID	**Area**	**Type**	**Dissolved Lithium (Li)**	**Dissolved Magnesium (Mg)**	**Dissolved Manganese (Mn)**	**Dissolved Phosphorus (P)**	**Dissolved Molybdenum (Mo)**	**Dissolved Nickel (Ni)**	**Dissolved Potassium (K)**	**Dissolved Rubidium (Rb)**	**Dissolved Selenium (Se)**	**Dissolved Silicon (Si)**	**Dissolved Silver (Ag)**	**Dissolved Sodium (Na)**	**Dissolved Strontium (Sr)**	**Dissolved Sulphur (S)**
Units			**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**
Method Code			BBY7SOP-00002													
LOD			0.0005	0.05	0.00005	0.002	0.00005	0.00002	0.05	0.00005	0.00004	0.1	0.000005	0.05	0.00005	10
OP02 <2mm	Pit	Road Cut	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP07 <2mm	Pit	Road Cut	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP11 <2mm	Pit	Road Cut	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP14 <2mm	Pit	Road Cut	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP12-01 Overburden Sample #1	Pit	Deep Overburden	0.0213	20.1	8.68	0.128	0.00846	0.0652	8.19	0.00094	0.00486	5.6	0.000025	4.66	0.536	50
OP12-10 Overburden Sample #1	Pit	Deep Overburden	0.005	21.8	19.6	0.082	0.0005	0.0234	5.71	0.00648	0.00182	4.9	0.00005	3.12	0.273	100
OP12-02 Weathered Rock Sample #1	Pit	Weathered Bedrock	0.00056	1.89	0.293	0.169	0.00804	0.00322	3.49	0.00117	0.00114	1.78	0.000005	3.11	0.104	21
OP12-06 Weathered Rock Sample #1	Pit	Weathered Bedrock	0.0052	29	7.06	0.008	0.00146	0.00827	14.1	0.00347	0.0149	4.57	0.000025	5.29	0.176	101
OP12-07 Weathered Rock Sample #1	Pit	Weathered Bedrock	0.00281	6.45	0.0442	0.186	0.0218	0.00162	7.78	0.00357	0.0219	1.93	0.000005	9.72	0.059	10
TP15-4 <2mm	TMF	Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
TP29-1 <2mm	TMF	Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
TP30-1 <2mm	TMF	Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
TP30-2 <2mm	TMF	Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A

Sample ID	**Area**	**Type**	**Dissolved Tellurium (Te)**	**Dissolved Thallium (Tl)**	**Dissolved Thorium (Th)**	**Dissolved Tin (Sn)**	**Dissolved Titanium (Ti)**	**Dissolved Tungsten (W)**	**Dissolved Uranium (U)**	**Dissolved Vanadium (V)**	**Dissolved Zinc (Zn)**	**Dissolved Zirconium (Zr)**	**Dissolved Mercury (Hg)**
Units			**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**	**mg/L**
Method Code													
LOD			0.00002	0.000002	0.000005	0.0002	0.0005	0.00001	0.000002	0.0002	0.0001	0.0001	0.00005
OP02 <2mm	Pit	Road Cut	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP07 <2mm	Pit	Road Cut	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP11 <2mm	Pit	Road Cut	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP14 <2mm	Pit	Road Cut	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
OP12-01 Overburden Sample #1	Pit	Deep Overburden	0.0001	0.00001	0.000025	0.001	0.0025	0.000842	0.000162	0.001	0.00383	0.0005	0.00025
OP12-10 Overburden Sample #1	Pit	Deep Overburden	0.0002	0.000037	0.00005	0.002	0.005	0.0001	0.00002	0.002	0.0513	0.001	0.0005
OP12-02 Weathered Rock Sample #1	Pit	Weathered Bedrock	0.00002	0.000011	0.000008	0.00047	0.00113	0.000026	0.000198	0.0002	0.00505	0.0001	0.00005
OP12-06 Weathered Rock Sample #1	Pit	Weathered Bedrock	0.0001	0.00004	0.000025	0.001	0.0025	0.000064	0.00001	0.001	0.06	0.0005	0.00025
OP12-07 Weathered Rock Sample #1	Pit	Weathered Bedrock	0.000043	0.000028	0.000008	0.00032	0.00069	0.00752	0.00538	0.00169	0.00188	0.0001	0.00005
TP15-4 <2mm	TMF	Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
TP29-1 <2mm	TMF	Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
TP30-1 <2mm	TMF	Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A
TP30-2 <2mm	TMF	Overburden	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A

E3: Mineralogy Report for TMF Area Samples

QEMSCAN TRACE MINERAL SEARCH ON SAMPLES FROM THE HARPER CREEK PROJECT

BRITISH COLUMBIA, CANADA

KM3767

March 15, 2013

ISO 9001:2008
Certificate No. FS 63170

Work performed on behalf of SRK Consulting (Canada) Inc.

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ALS Metallurgy

March 15, 2013

Mr. Steven Day
Corporate Consultant (Geochemistry)
SRK Consulting (Canada) Inc.
22nd Floor, 1066 West Hastings Street
Vancouver, BC V6E 3X2

Dear Mr. Day;

Re: QEMSCAN Trace Mineral Search on Samples from the Harper Creek Project
KM3767

We are pleased to report that we have now completed the mineralogical study on three pulp samples identified as TP2, TP29-1, and TP51-3, which were reportedly from the Harper Creek Project. The principal objective of this study was a search for lead bearing minerals or contaminants present in the samples. To achieve the objective, QEMSCAN Trace Mineral Search (TMS) was performed on each of the three un-sized samples.

Three samples were received on February 19, 2013, and the analysis was completed on March 7, 2013. All information produced in this program is presented in detail in three appendices of data:

Appendix I – Sample Origin
Appendix II – Spectra Data
Appendix III – Mineralogical Data

A summary of the QEMSCAN Trace Mineral Search is presented in Figure 1. The following comments summarize our findings based on the available data:

- Geochemical analyses data[*] of the metal contents in the samples indicates elevated concentrations of lead in samples TP2 and TP29-1, assaying 1,720 ppm and 1,880 ppm, respectively. The measured concentration of lead in sample TP51-3 was about 20 ppm.

- After searching over one million particles, several hundred lead bearing particles were detected in samples TP2 and TP29-1 (See Figure 1). The majority of the lead bearing particles presented as metallic lead. The mean projected diameter of lead bearing particles in samples TP2 and TP29-1 was about 28µm and 21µm, respectively, and varied from less than 10µm up to a maximum of 390µm. The QEMSCAN Backscatter Image 1 shows the typical structure of a large metallic lead particle. More backscatter images, as well as the spectra data for lead bearing particles, can be located in Appendix II.

- Only four lead bearing particles were located in sample TP51-3. Mean projected diameter of lead bearing particles in the sample varied from less than 10µm up to a maximum of 32µm. The average diameter of lead bearing particles in this sample is 13µm.

In summary, the elevated concentrations of lead in samples TP2 and TP29-1 was dominantly due to the presence of metallic lead grains. The sources of these metallic lead particles should be invested.

---

[*] The geochemical analysis data was provided by SRK.

Thank you for the opportunity to participate in your mineralogical studies. If you have any questions regarding this report, or the results generated by this program, please contact us directly.

Regards,

Kathryn Dunne, M.Sc., P. Geo.
Mineralogist

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March 15, 2013
KM3767

Report Distribution:
Steven Day, SRK Consulting (Canada) Inc., Vancouver, BC – Electronic Copy

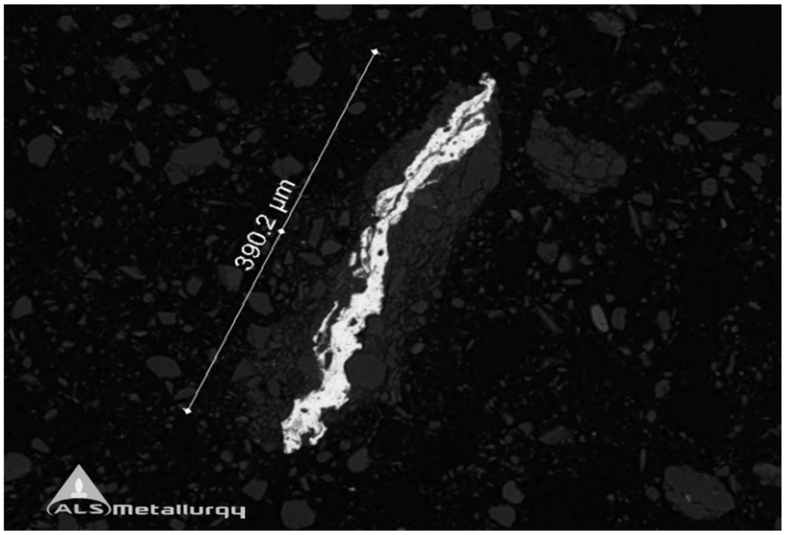
FIGURE 1
SUMMARY OF QEMSCAN TRACE MINERAL SEARCH
KM3767 Harper Creek - Pulp Samples

Parameter	Samples		
	TP2	TP29-1	TP51-3
Size Fraction	Unsized	Unsized	Unsized
Number of Blocks Scanned	1	1	1
Backscattered Electrons (BSE) Limit	60	60	60
Total Found Particles	1.22 $\times 10^6$	1.40 $\times 10^6$	2.15 $\times 10^6$
Number of Lead-Bearing Particles	300	187	4
Mean Projected Diameter of Lead-Bearing Particles	28.4 µm	20.5 µm	12.9 µm
Measured Lead Content*	1720 ppm	1880 ppm	20.3 ppm

* Aqua Regia Digestion with ICP-MS Finish (SRK Consulting data)

QEMSCAN BACKSCATTER IMAGE 1
HARPER CREEK – TP2

Example of Lead Particle



APPENDIX I – KM3767

SAMPLE ORIGIN

1.0 Sample Origin

On February 19, 2013, three pulp samples, reportedly from the Harper Creek deposit, were received at ALS Metallurgy Kamloops. The weight of the samples ranged from about 270 to 530 grams each. The weights of each sample are listed in Table I-1.

Geochemical analysis data of the metals in the samples by aqua regia digestion with ICP-MS finish were provided by SRK Consulting (Canada) Inc. These geochemical results are shown in Table I-2.

The samples were then mounted and polished in preparation for QEMSCAN Trace Mineral Analysis.

TABLE I-1
MASS AND IDENTIFICATION OF SAMPLES RECEIVED

Sample ID*	Sample Form	Weight – kg
TP2 <2mm	pulp	0.0533
TP29-1 <2mm	pulp	0.0273
TP51-3 <2mm	pulp	0.0404
Total		0.121

*Samples are identified as TP2, TP29-1 and TP51-3.

TABLE I-2
GEOCHEMICAL ANALYSIS OF THE METALS IN THE PULP SAMPLES

Element	Symbol	Unit	Tolerance +/-	TP2 <2mm	TP29-1 <2mm	TP51-3 <2mm
Aluminum	Al	%	0.41	2.16	0.77	0.45
Boron	B	ppm	N/A	50	30	50
Barium	Ba	ppm	70	172	74	42
Calcium	Ca	%	0.15	0.51	0.24	0.09
Chromium	Cr	ppm	17	64	66	94
Copper	Cu	ppm	202	83.1	533	151
Iron	Fe	%	0.54	3.33	4.61	9.03
Potassium	K	%	0.27	0.53	0.28	0.15
Lithium	Li	ppm	5.5	17	27	2
Magnesium	Mg	%	0.2	1.03	0.3	0.13
Manganese	Mn	ppm	38	652	1160	638
Sodium	Na	%	0.04	0.07	0.01	0.02
Nickel	Ni	ppm	5.4	10.6	7.7	83.9
Phosphorus	P	ppm	120	450	390	880
Sulphur	S	%	0.26	0.03	<0.01	0.03
Strontium	Sr	ppm	5	24.9	9	9.6
Titanium	Ti	%	0.036	0.07	<0.01	<0.01
Vanadium	V	ppm	12	51	36	<1
Zinc	Zn	ppm	26	90	80	108
Zirconium	Zr	ppm	5	3.4	2.2	23.3
Silver	Ag	ppm	0.8	0.16	0.49	0.35
Arsenic	As	ppm	2.3	16	325	23
Beryllium	Be	ppm	1	0.3	1.1	0.4
Bismuth	Bi	ppm	0.2	0.47	1.07	3.86
Cadmium	Cd	ppm	0.2	0.24	0.28	0.29
Cerium	Ce	ppm	7	27	37.5	65.2
Cobalt	Co	ppm	4.7	11.2	28.1	25.4
Cesium	Cs	ppm	0.6	2.78	8.41	0.9
Gallium	Ga	ppm	2	6	2.2	1.9
Germanium	Ge	ppm	0.2	<0.1	<0.1	<0.1
Hafnium	Hf	ppm	0.1	0.09	0.05	0.45
Mercury	Hg	ppm	0.023	<0.01	0.01	0.02
Indium	In	ppm	0.01	0.03	0.07	0.02
Lanthanum	La	ppm	4	14.7	23.2	35.8
Lutetium	Lu	ppm	N/A	0.1	0.15	0.09
Molybdenum	Mo	ppm	0.6	1.03	1.68	2.58
Niobium	Nb	ppm	N/A	0.22	0.11	0.07
Lead	Pb	ppm	4.5	1720	1880	20.3
Rubidium	Rb	ppm	12	29.6	20.1	6.3
Antimony	Sb	ppm	0.3	0.21	16.3	0.41
Scandium	Sc	ppm	1.4	6.8	12.1	1.5
Selenium	Se	ppm	2.3	<1	<1	1
Tin	Sn	ppm	0.1	0.9	1.6	0.5
Tantalum	Ta	ppm	0.2	<0.05	<0.05	<0.05
Terbium	Tb	ppm	N/A	0.27	0.4	0.47
Tellurium	Te	ppm	0.1	0.06	0.1	0.29
Thorium	Th	ppm	0.5	6.7	7.2	12.1
Thallium	Tl	ppm	0.1	0.21	0.71	0.24
Uranium	U	ppm	0.1	1.19	2.75	3.87
Tungsten	W	ppm	0.7	0.5	1.1	0.1
Yttrium	Y	ppm	1.1	6.24	9.78	4.87
Ytterbium	Yb	ppm	N/A	0.6	1	0.6

APPENDIX II – KM3767

SPECTRA DATA

INDEX

TABLE 1
QEMSCAN X-RAY ANALYSIS ON LEAD PARTICLES
KM3767 TP2

Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
1	Oxygen	K-series	3.1	3.8	33.3	0.4
	Aluminium	K-series	0.4	0.5	2.4	0.0
	Lead	L-series	76.2	95.7	64.2	2.0
		Sum:	79.6	100	100	
Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
2	Oxygen	K-series	3.0	3.5	31.8	0.4
	Aluminium	K-series	0.0	0.0	0.0	0.0
	Lead	L-series	82.3	96.5	68.2	2.2
	Sulfur	K-series	0.0	0.0	0.0	0.0
		Sum:	85.2	100	100	
Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
3	Oxygen	K-series	3.6	4.3	37.0	0.5
	Lead	L-series	80.3	95.7	63.0	2.1
		Sum:	84.0	100	100	
Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
4	Oxygen	K-series	5.6	5.7	43.9	0.7
	Lead	L-series	92.4	94.3	56.1	2.4
		Sum:	98.0	100	100	
Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
5	Oxygen	K-series	5.4	6.1	45.8	0.7
	Lead	L-series	82.4	93.9	54.2	2.2
		Sum:	87.7	100	100	
Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
6	Oxygen	K-series	4.0	4.9	40.0	0.5
	Lead	L-series	78.3	95.1	60.0	2.1
		Sum:	82.3	100	100	
Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
7	Oxygen	K-series	3.9	4.5	37.8	0.5
	Lead	L-series	83.8	95.5	62.2	2.2
		Sum:	87.7	100	100	

TABLE 1 CONTINUED
QEMSCAN X-RAY ANALYSIS ON LEAD PARTICLES
KM3767 TP2

Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
8	Oxygen	K-series	4.6	5.6	43.5	0.6
	Lead	L-series	76.6	94.4	56.5	2.0
		Sum:	81.2	100	100	
Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
9	Oxygen	K-series	5.1	5.8	44.5	0.6
	Lead	L-series	82.0	94.2	55.5	2.1
		Sum:	87.1	100	100	
Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
10	Oxygen	K-series	3.6	4.1	35.9	0.5
	Lead	L-series	83.0	95.9	64.1	2.2
		Sum:	86.6	100	100	

FIGURE 1
QEMSCAN X-RAY ANALYSIS ON LEAD PARTICLES
KM3767 TP2

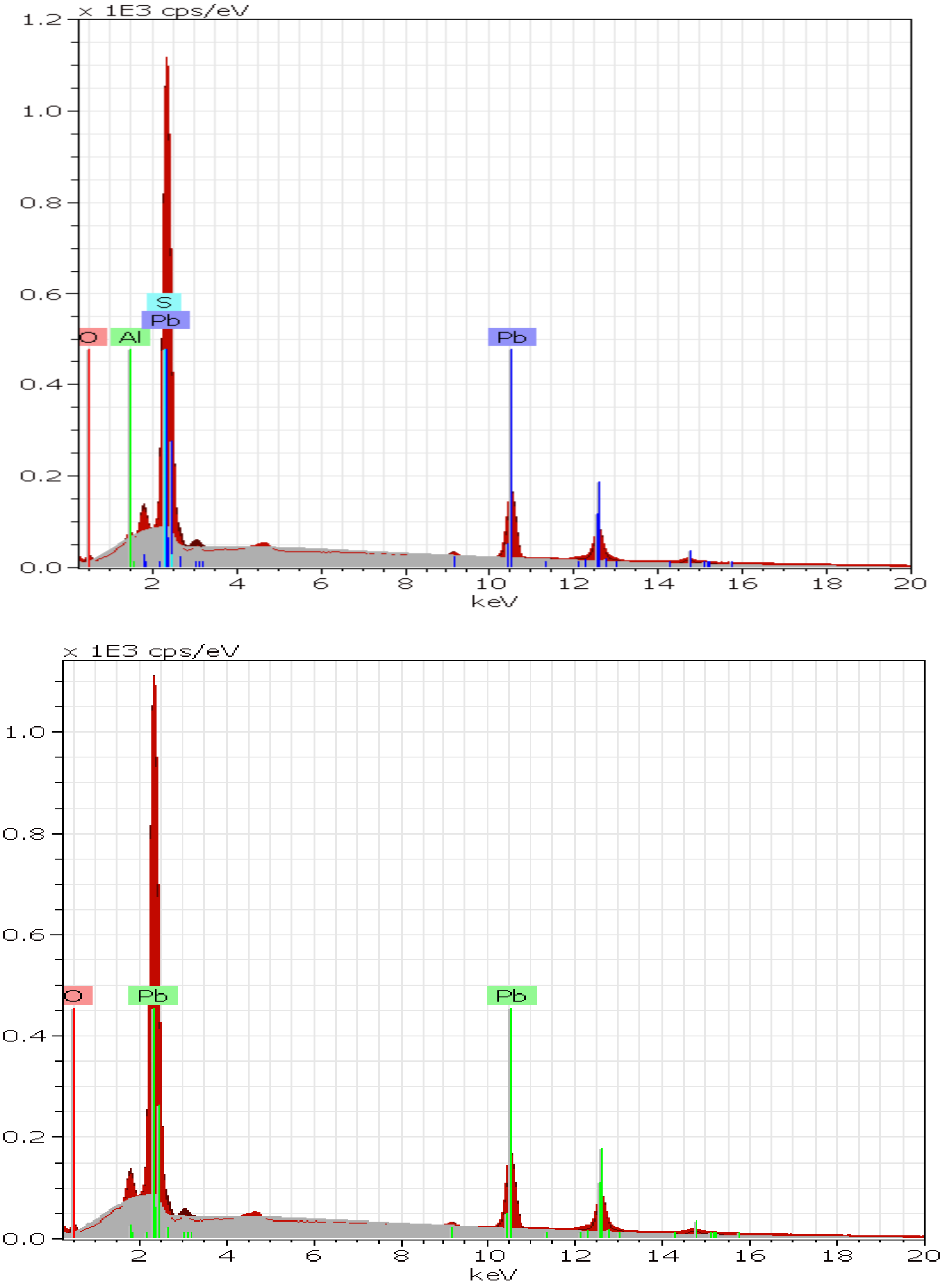


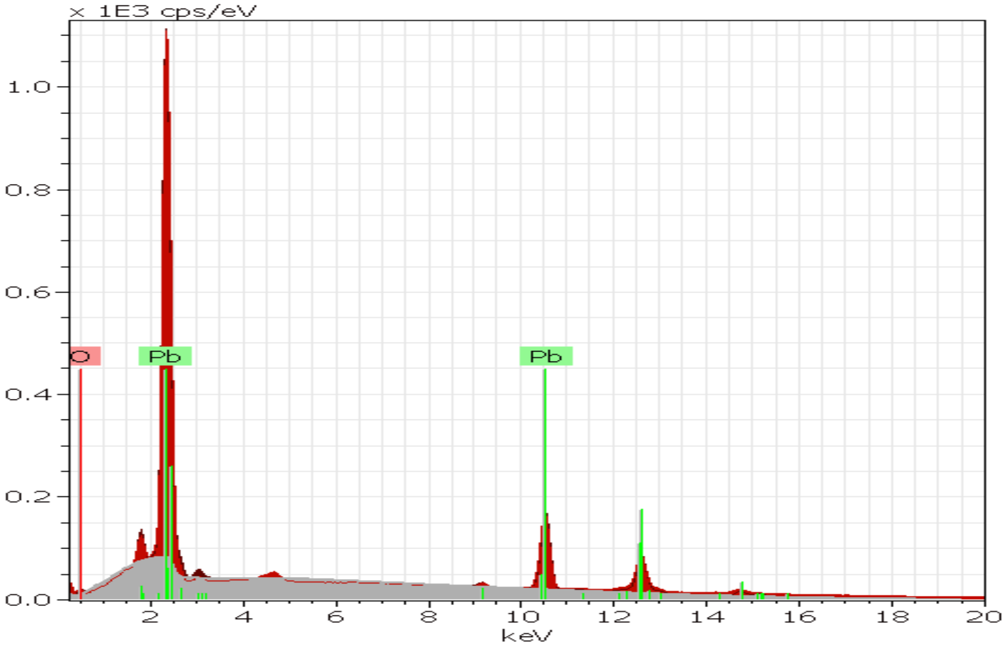
TABLE 2
QEMSCAN X-RAY ANALYSIS ON LEAD PARTICLES
KM3767 TP29-1

Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
1	Oxygen	K-series	2.1	2.7	26.1	0.3
	Lead	L-series	78.6	97.3	73.9	2.1
		Sum:	80.8	100	100	
Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
2	Oxygen	K-series	3.1	3.9	34.7	0.4
	Lead	L-series	76.4	96.1	65.3	2.0
		Sum:	79.5	100	100	
Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
3	Oxygen	K-series	3.0	3.6	32.8	0.4
	Lead	L-series	80.1	96.4	67.2	2.1
		Sum:	83.1	100	100	
Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
4	Oxygen	K-series	2.8	3.2	29.7	0.4
	Lead	L-series	85.2	96.8	70.3	2.2
		Sum:	88.0	100	100	
Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
5	Oxygen	K-series	2.9	3.5	32.2	0.4
	Lead	L-series	79.9	96.5	67.8	2.1
		Sum:	82.8	100	100	
Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
6	Oxygen	K-series	2.4	2.9	28.0	0.3
	Lead	L-series	80.9	97.1	72.0	2.1
		Sum:	83.4	100	100	
Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
7	Oxygen	K-series	2.2	2.7	26.8	0.3
	Lead	L-series	79.6	97.3	73.2	2.1
		Sum:	81.9	100	100	
Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
8	Oxygen	K-series	3.9	4.7	39.2	0.5
	Lead	L-series	77.4	95.3	60.8	2.0
		Sum:	81.3	100	100	

TABLE 2 CONTINUED
QEMSCAN X-RAY ANALYSIS ON LEAD PARTICLES

Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
9	Oxygen	K-series	4.2	5.7	44.0	0.5
	Lead	L-series	69.4	94.3	56.0	1.8
		Sum:	73.6	100	100	
Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
10	Oxygen	K-series	6.1	8.1	53.2	0.8
	Lead	L-series	69.1	91.9	46.8	1.8
		Sum:	75.1	100	100	

FIGURE 2
QEMSCAN X-RAY ANALYSIS ON LEAD PARTICLES
KM3767 TP29-1



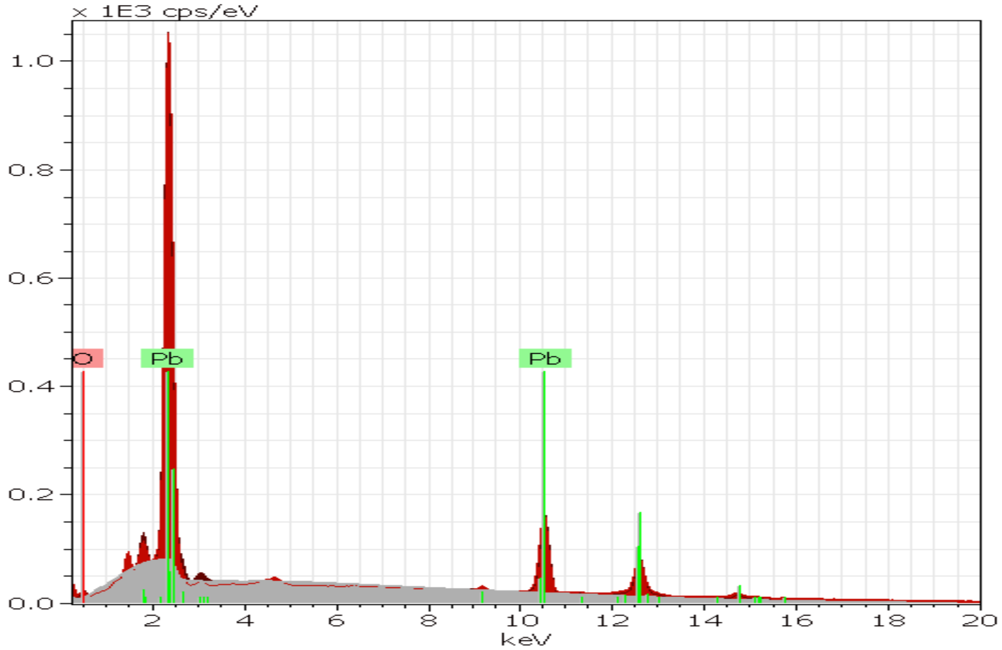
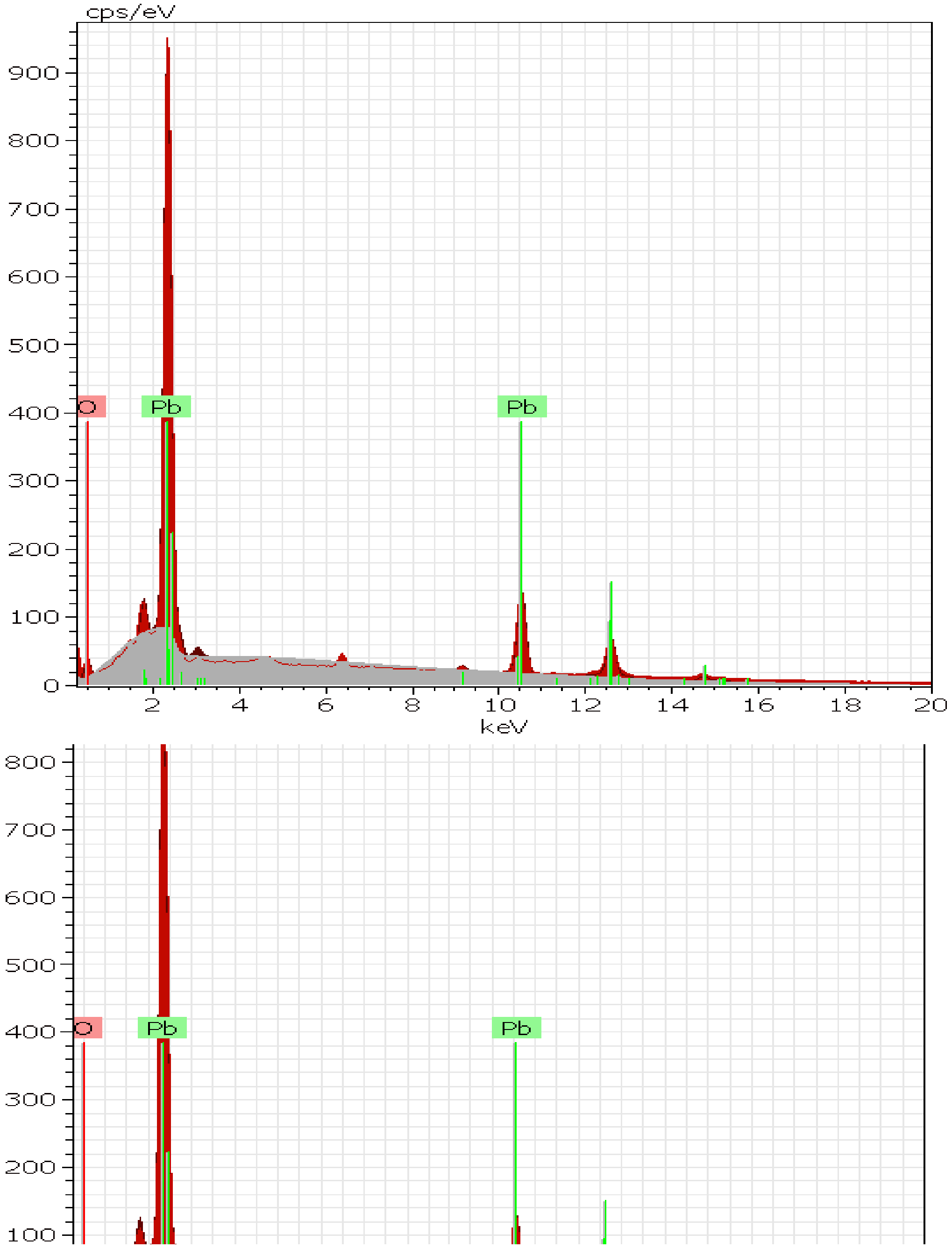


TABLE 3
QEMSCAN X-RAY ANALYSIS ON LEAD PARTICLES
KM3767 TP51-3

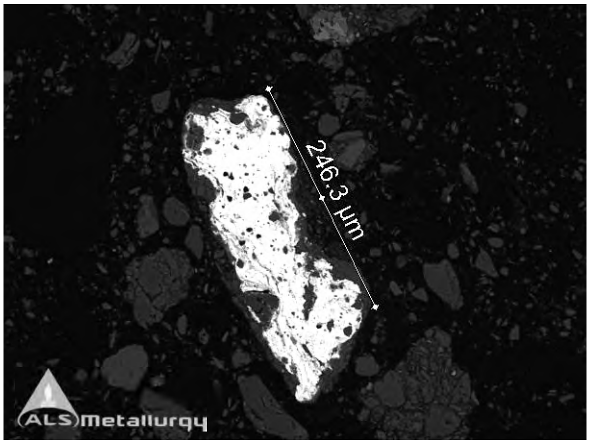
Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
1	Oxygen	K-series	9.8	12.4	64.7	1.2
	Lead	L-series	69.2	87.6	35.3	1.8
		Sum:	79.0	100	100	
Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
2	Oxygen	K-series	5.7	8.8	55.5	0.7
	Lead	L-series	58.8	91.2	44.5	1.5
		Sum:	64.4	100	100	
Particle	Element	Series	Weight Percent	Normal Weight Percent	Normal Atomic Percent	Error in Percent
3	Oxygen	K-series	7.1	9.2	56.8	0.9
	Lead	L-series	69.8	90.8	43.2	1.8
		Sum:	76.8	100	100	

FIGURE 3
QEMSCAN X-RAY ANALYSIS ON LEAD PARTICLES
KM3767 TP51-3

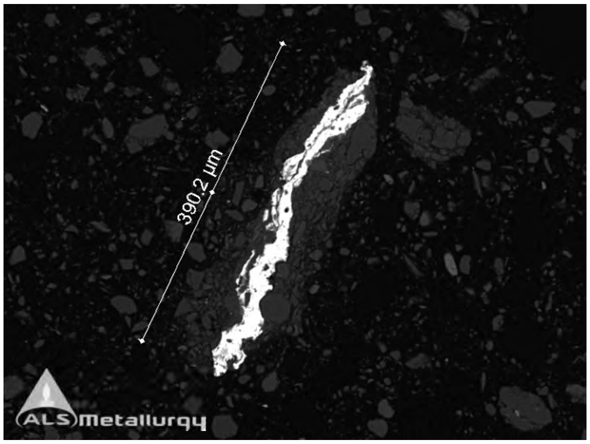


PHOTOMICROGRAPH 1A
HARPER CREEK – TP2
KM3767

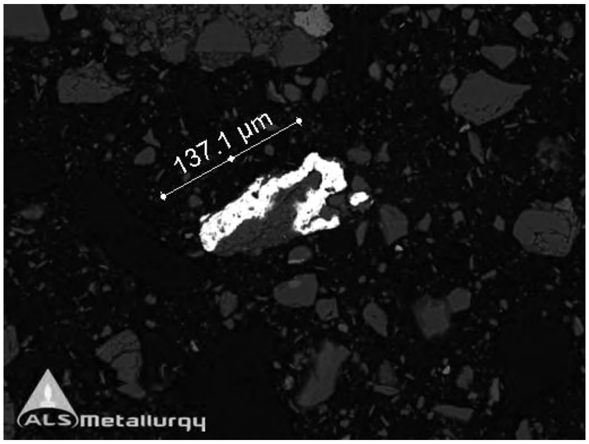
Particle 1



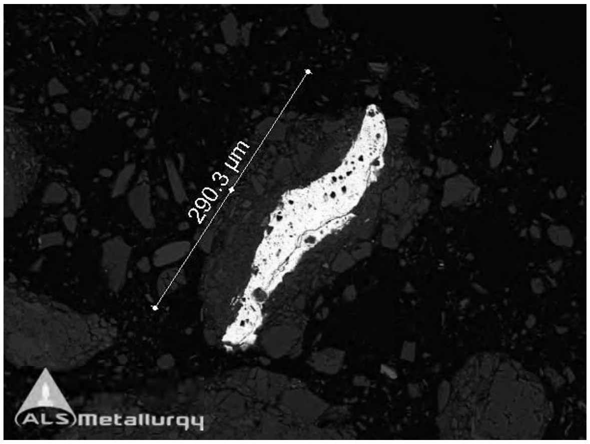
Particle 2



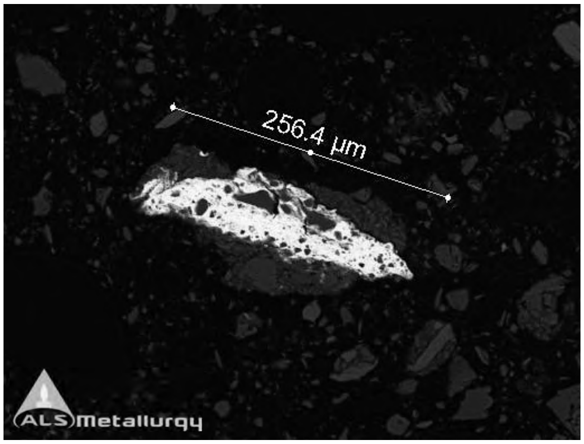
Particle 3



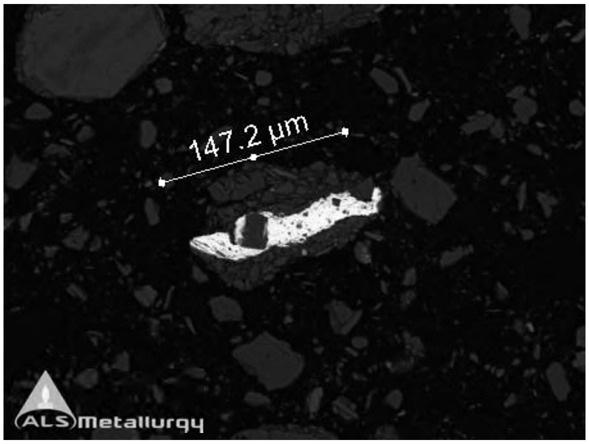
Particle 4



Particle 5



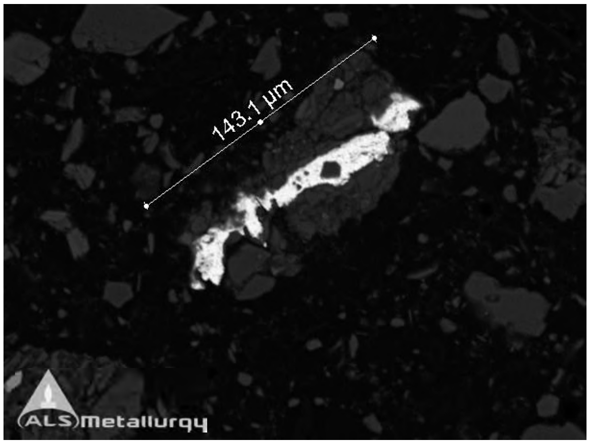
Particle 6



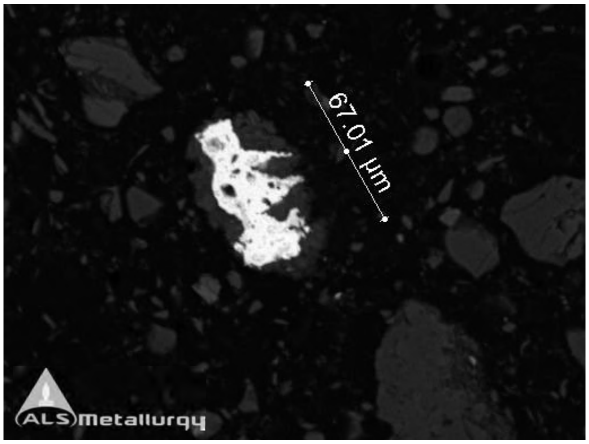
*Photos taken using QEMSCAN.

PHOTOMICROGRAPH 1B
HARPER CREEK – TP2
KM3767

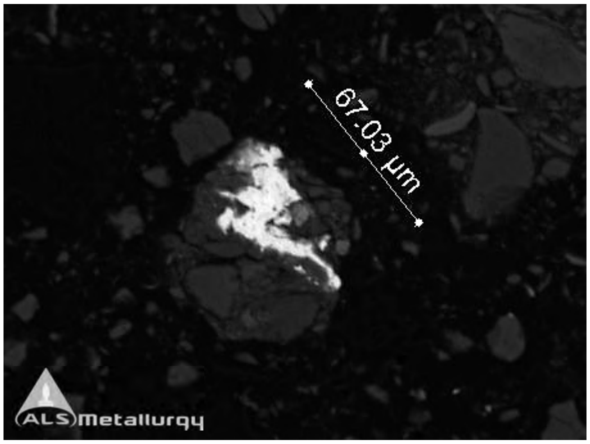
Particle 7



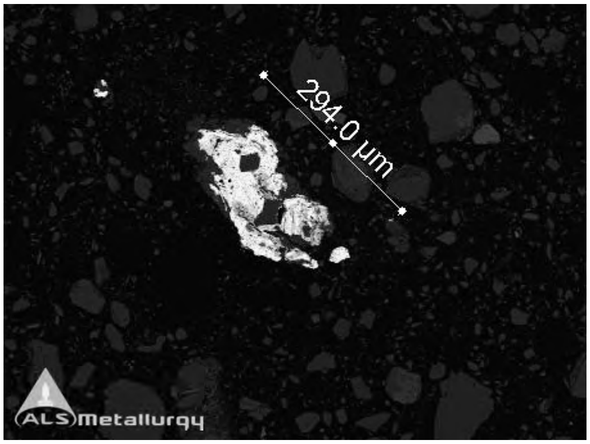
Particle 8



Particle 9



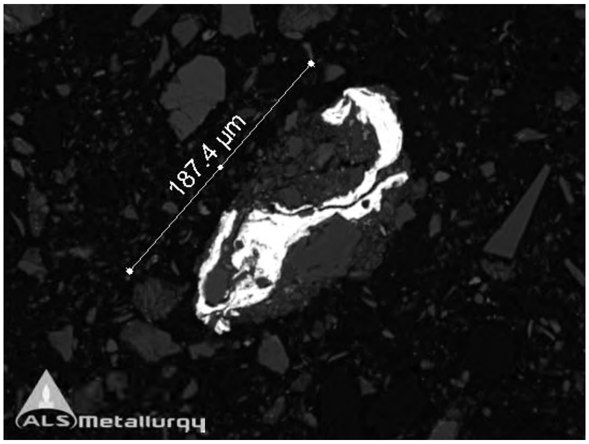
Particle 10



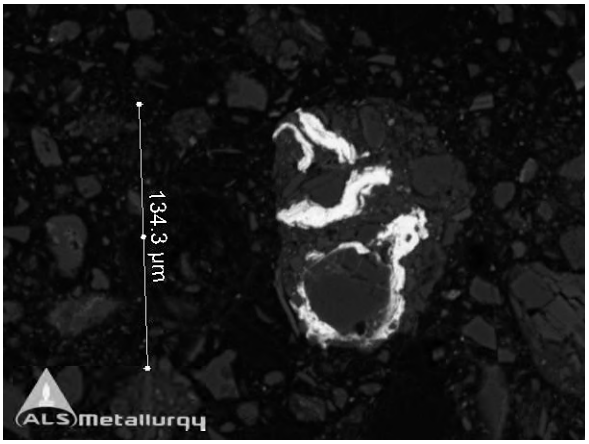
*Photos taken using QEMSCAN.

PHOTOMICROGRAPH 2A
HARPER CREEK – TP29-1
KM3767

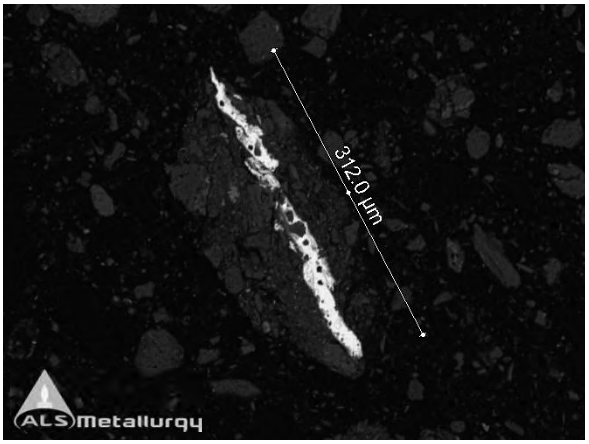
Particle 1



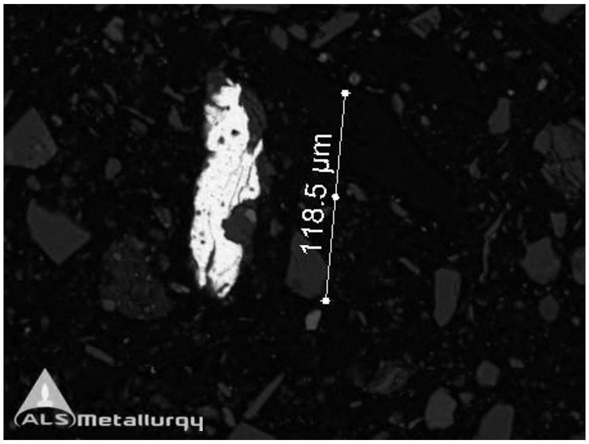
Particle 2



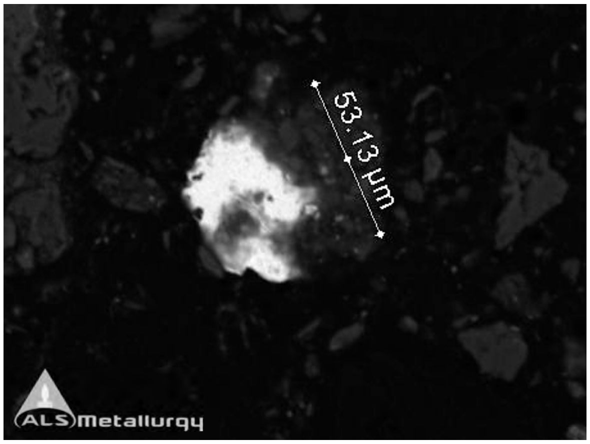
Particle 3



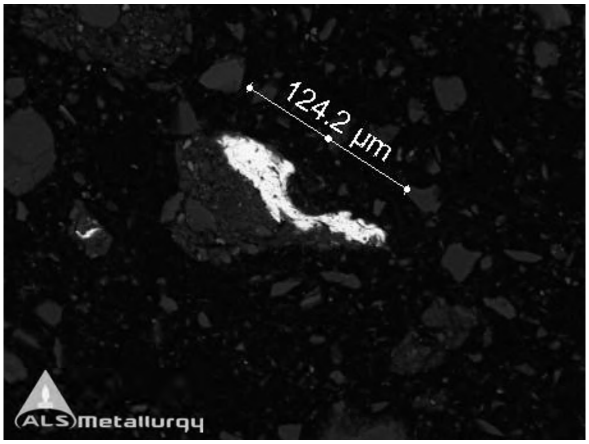
Particle 4



Particle 5



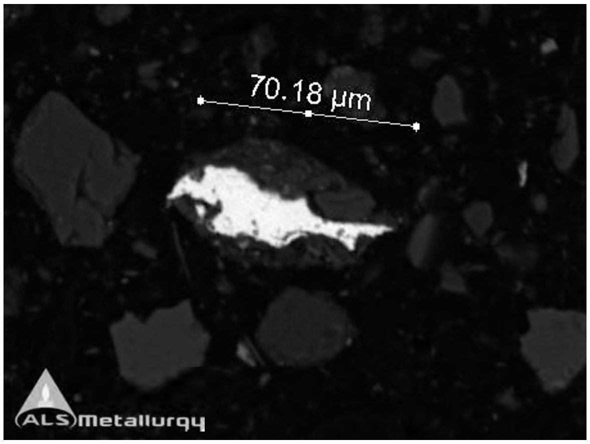
Particle 6



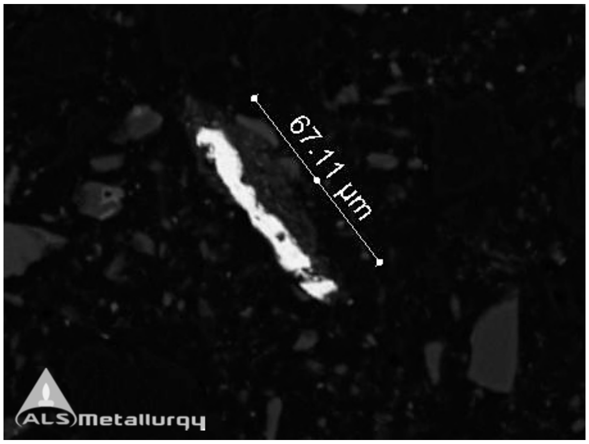
*Photos taken using QEMSCAN.

PHOTOMICROGRAPH 2B
HARPER CREEK – TP29-1
KM3767

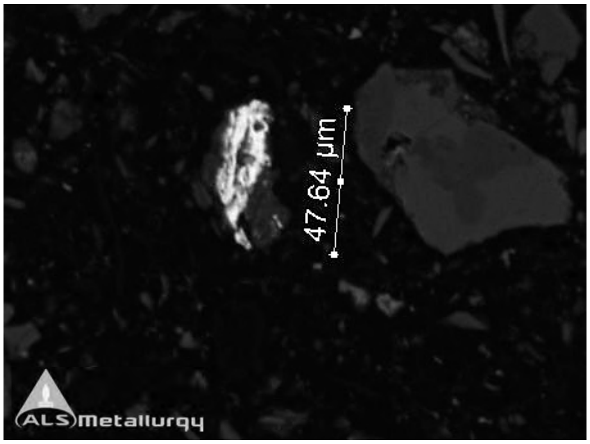
Particle 7



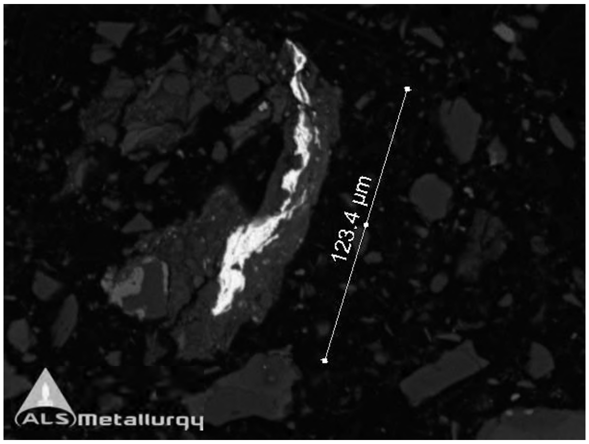
Particle 8



Particle 9



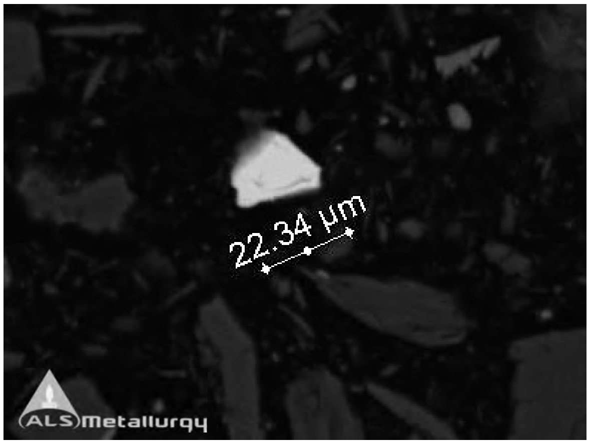
Particle 10



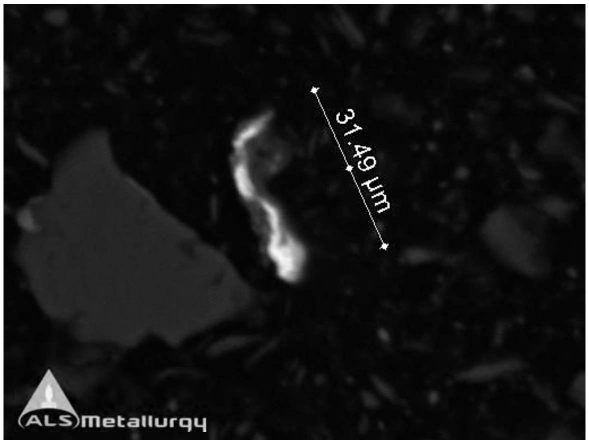
*Photos taken using QEMSCAN.

PHOTOMICROGRAPH 3
HARPER CREEK – TP51-3
KM3767

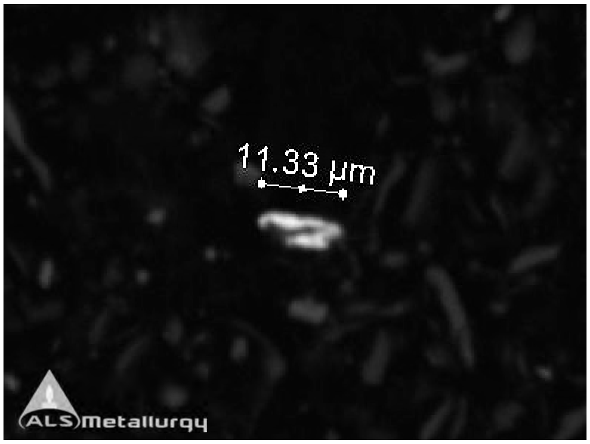
Particle 1



Particle 2



Particle 3



*Photos taken using QEMSCAN.

APPENDIX III – KM3767

MINERALOGICAL DATA

INDEX

TABLE 1
MINERAL COMPOSITION OF THE TRACE MINERAL SEARCH PARTICLE SET - UNSIZED
KM3767 Harper Creek

Minerals	TP2 <2mm	TP29-1 <2mm	TP51-3 <2mm
Lead-bearing particles	8.94	13.7	0.13
Pyrite	0.59	0.21	1.29
Chalcopyrite	0.00	0.00	0.00
Iron Oxides	0.11	1.2	4.94
Quartz	33.7	28.4	17.3
Micas	21.6	34.5	51.9
Feldspars	25.8	15.0	6.05
Ti-Minerals	3.37	4.31	13.7
Kaolinite	0.76	1.61	1.13
Chlorite	1.53	0.14	0.13
Epidote	1.52	0.01	0.00
Amphibole (Hornblende)	1.00	0.04	0.02
Ce-phosphate (Monazite)	0.05	0.08	2.25
Others	1.1	0.88	1.28
Total	100	100	100

Notes: 1) Iron Oxides include Hematite, Goethite and Limonite
2) Micas include Muscovite and Biotite/Phlogopite.
3) Feldspars includes Plagioclase Feldspar, K-Feldspar, Feldspar-Albite, and Alkali-Feldspar.
4) Ti-Minerals includes Rutile/Anatase, Ilmentite and Sphene.
5) Others includes trace amounts of Sphalerite, Garnet, Pyroxene and unresolved mineral species.

TABLE 2
SUMMARY OF QEMSCAN TRACE MINERAL SEARCH
KM3767 Harper Creek - Pulp Samples

Parameter	Samples		
	TP2 < 2mm	TP29-1 <2mm	TP51-3 <2mm
Size Fraction	Unsized	Unsized	Unsized
Number of Blocks Scanned	1	1	1
Backscattered Electrons (BSE) Limit	60	60	60
Total Found Particles	1.22×10^6	1.40×10^6	2.15×10^6
Rejected - Non Target:	1.22×10^6	1.40×10^6	2.15×10^6
Remaining Candidates:	1.18×10^3	1.19×10^3	1.62×10^3
Total Finally Accepted	1.18×10^3	1.01×10^3	1.30×10^3
Total Finally Accepted (%)	0.1	0.07	0.06
Number of Lead-Bearing Particles	300	187	4
Mean Projected Diameter of Lead-Bearing Particles	28.39 µm	20.51 µm	12.89 µm
Measured Lead Content*	1720 ppm	1880 ppm	20.3 ppm

* Aqua Regia Digestion with ICP-MS Finish (SRK Consulting data)

Appendix F: Quarry Static Test Results ABA and Trace Element

	Acid Base Accounting											Trace Elements								
Sample ID	**Paste**	**TIC**	**CaCO3**	**S(T)**	**S(SO4)**	**Sulfide**	**AP**	**Modified NP**	**NP/AP**	**TIC/AP**	**Fizz Test**	**Al**	**B**	**Ba**	**Ca**	**Cr**	**Cu**	**Fe**	**K**	**Li**
Units	**pH**	**%**	**kg CaCO3/t**	**%**	**%**	**%**	**kg CaCO3/t**	**t CaCO3/1000t**				**%**	**ppm**	**ppm**	**%**	**ppm**	**ppm**	**%**	**%**	**ppm**
Method Code	Sobek	CSB02V	Calc.	CSA06V	CSA07V	Calculated	Calculated	Modified NP	Calculated	Calculated	Sobek	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B
LOD	0.20	0.01	#N/A	0.01	0.01			0.5			#N/A	0.01	10	5	0.01	1	0.5	0.01	0.01	1
GT-21-01(6.98-7.18)	9.36	0.07	5.8	0.04	*<0.1*	0.03	0.94	11.6	12.3	6.2	slight	0.74	50	233	0.44	88	8.9	2.42	0.65	36
GT-21-02(33.48-33.66)	9.59	0.05	4.2	0.04	*<0.1*	0.03	0.94	5.5	5.9	4.4	none	0.88	60	294	0.44	96	8.8	2.65	0.78	44
GT-21-03(63.09-63.31)	9.83	0.05	4.2	0.04	*<0.1*	0.03	0.94	6.0	6.4	4.4	none	0.88	60	290	0.45	101	5.5	2.63	0.77	33
GT-23-01(6.67-6.88)	9.66	0.05	4.2	0.03	*<0.1*	0.02	0.63	5.7	9.1	6.7	none	0.69	60	189	0.4	102	6.1	2.24	0.56	19
GT-23-02(24.81-24.99)	9.78	0.03	2.5	0.03	*<0.1*	0.02	0.63	5.4	8.7	4.0	none	0.73	60	213	0.39	107	5.7	2.35	0.6	34
GT-23-03(47.23-47.71)	9.62	0.08	6.7	0.04	*<0.1*	0.03	0.94	8.6	9.2	7.1	none	0.72	50	190	0.48	89	5.2	2.36	0.56	20

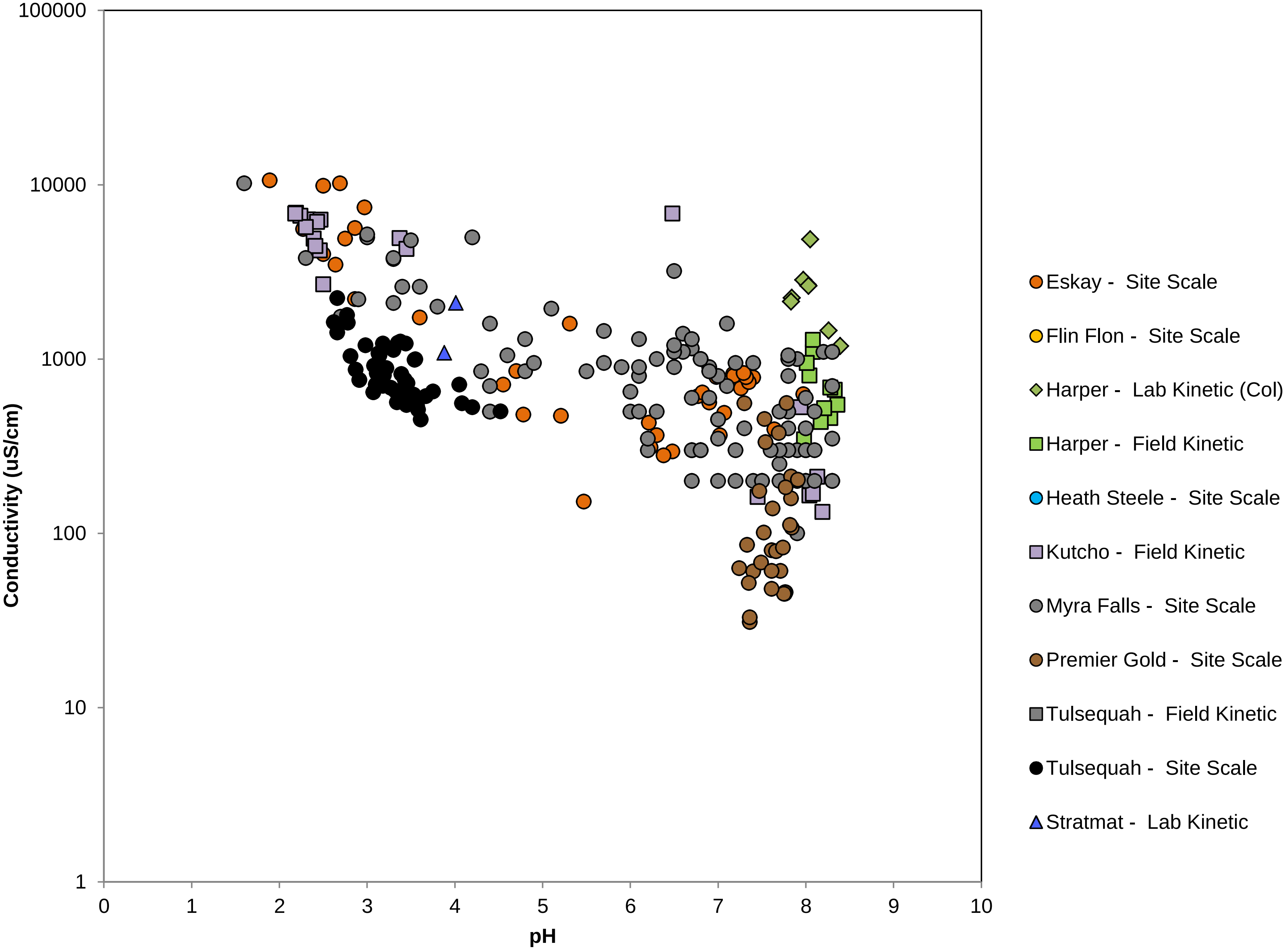
Sample ID	**Mg**	**Mn**	**Na**	**Ni**	**P**	**S**	**Sr**	**Ti**	**V**	**Zn**	**Zr**	**Ag**	**As**	**Be**	**Bi**	**Cd**	**Ce**	**Co**	**Cs**	**Ga**	**Ge**	**Hf**	**Hg**
Units	**%**	**ppm**	**%**	**ppm**	**ppm**	**%**	**ppm**	**%**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**
Method Code	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B
LOD	0.01	2	0.01	0.5	50	0.01	0.5	0.01	1	1	0.5	0.01	1	0.1	0.02	0.01	0.05	0.1	0.05	0.1	0.1	0.05	0.01
GT-21-01(6.98-7.18)	0.97	401	0.04	36.8	1090	0.05	16.3	0.18	50	48	1.6	0.04	<1	*<0.1*	0.06	0.03	54.9	6.2	1.71	4.1	*<0.1*	0.07	*<0.01*
GT-21-02(33.48-33.66)	0.83	482	0.07	6.3	1210	0.06	20.1	0.22	58	55	2.7	0.03	<1	0.1	0.1	0.04	76.5	5.7	3.63	5	*<0.1*	0.11	*<0.01*
GT-21-03(63.09-63.31)	0.8	460	0.07	6.5	1260	0.04	19.2	0.24	61	55	2.5	0.03	<1	0.1	0.09	0.01	51.9	5.7	2.48	4.6	*<0.1*	0.11	*<0.01*
GT-23-01(6.67-6.88)	0.66	364	0.06	6	1000	0.05	18.6	0.18	46	43	1.9	0.04	<1	*<0.1*	0.05	0.02	61.3	4.9	1.29	4	*<0.1*	0.09	*<0.01*
GT-23-02(24.81-24.99)	0.7	387	0.06	5.8	1010	0.04	15.8	0.2	49	48	1.9	0.03	<1	0.1	0.04	0.02	65.8	5.3	1.36	4.4	*<0.1*	0.09	*<0.01*
GT-23-03(47.23-47.71)	0.66	399	0.05	5.6	1030	0.04	33.5	0.15	47	46	1.2	0.03	<1	0.2	0.06	0.02	83.7	5.3	1.52	4.5	*<0.1*	0.06	*<0.01*

Sample ID	**In**	**La**	**Lu**	**Mo**	**Nb**	**Pb**	**Rb**	**Sb**	**Sc**	**Se**	**Sn**	**Ta**	**Tb**	**Te**	**Th**	**Tl**	**U**	**W**	**Y**	**Yb**
Units	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**	**ppm**
Method Code	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B	ICM14B
LOD	0.02	0.1	0.01	0.05	0.05	0.2	0.2	0.05	0.1	1	0.3	0.05	0.02	0.05	0.1	0.02	0.05	0.1	0.05	0.1
GT-21-01(6.98-7.18)	*<0.02*	39.3	0.05	0.4	0.85	1.4	38.4	*<0.05*	2.8	*<1*	0.5	*<0.05*	0.17	*<0.05*	7.9	0.28	1.51	0.1	3.67	0.3
GT-21-02(33.48-33.66)	0.03	53.1	0.07	0.69	1.58	2	58.1	*<0.05*	4.1	*<1*	1.5	*<0.05*	0.23	*<0.05*	8.9	0.41	1.54	0.9	4.79	0.5
GT-21-03(63.09-63.31)	0.02	33	0.09	0.53	1.37	1.4	46.8	*<0.05*	2.8	*<1*	0.8	*<0.05*	0.21	*<0.05*	6.3	0.35	1.88	0.2	4.52	0.5
GT-23-01(6.67-6.88)	*<0.02*	44.6	0.07	0.51	1.29	1.9	34.6	*<0.05*	2.9	*<1*	0.6	*<0.05*	0.2	*<0.05*	11.3	0.25	1.82	0.2	4.24	0.4
GT-23-02(24.81-24.99)	*<0.02*	47	0.06	0.56	1.32	2.1	37.8	*<0.05*	2.6	*<1*	0.6	*<0.05*	0.19	*<0.05*	12.1	0.27	2.52	0.1	4.1	0.4
GT-23-03(47.23-47.71)	0.02	53.6	0.1	0.39	0.92	1.5	34.9	*<0.05*	3.3	*<1*	0.7	*<0.05*	0.34	*<0.05*	13.4	0.24	2.26	*<0.1*	7.26	0.6

Appendix G: Initial Upper Limit Case

Term	Term Location	Material Type	Term Type	Rate Type		Solubility Control		Infiltration	Rate Scaling			Reporting Units
				Type	Statistic	Type	P95 Statistic	mm/year	Temperature	Particle Size	Contact	
2	North Non-PAG Stockpile	Pit Rock	Subaerial	Waste Rock	P95	Analog	7<pH<8	564.5	0.5	0.2	0.5	mg/L
3	Non-PAG Low Grade Ore Stockpile	Pit Rock	Subaerial	Ore	P95	Analog	7<pH<8	564.5	0.5	0.2	0.5	mg/L
4	PAG Low Grade Ore Stockpile	Pit Rock	Subaerial	Ore	P95	Analog	7<pH<8	564.5	0.5	0.2	0.5	mg/L
7.1	PAG Stockpile	Pit Rock	Subaerial	Waste Rock	P95	Analog	7<pH<8	564.5	0.5	0.2	0.5	mg/L
7.2	PAG Stockpile	Pit Rock	Flushing	Waste Rock	P95	Analog	7<pH<8	-	0.5	0.2	0.5	mg/year
7.3	PAG Stockpile	Pit Rock	Subaqueous	Not Applicable	Not Applicable	Testwork	P95	Not Applicable	Not Applicable	Not Applicable	Not Applicable	mg/L
9	East Overburden Stockpile	Overburden	Subaerial	Quarry Rock	P95	OP Overburden	P95	564.5	0.3	0.2	0.5	mg/L
10.1	Pit walls	Wall	Subaerial	Waste Rock	P95	Analog	7<pH<8	101.61	0.3	0.1	0.5	mg/L
10.2	Acidified Highwall	Wall	Subaerial	Waste Rock	Accelerated	Analog	4<pH<5	101.61	0.3	0.1	0.5	mg/L
10.3	Flushed Pit Walls	Wall	Flushing	Waste Rock	P95	Analog	7<pH<8	-	0.3	0.1	0.5	mg/m^2
11.1	Tailings Dam	Pit Rock	Subaerial	Waste Rock	P95	Analog	7<pH<8	564.5	0.5	0.2	0.5	mg/L
11.2	Tailings Dam	Quarry Rock	Subaerial	Quarry Rock	P95	Analog	7<pH<8	564.5	0.3	0.2	0.5	mg/L
11.3	Tailings Dam	TMF Overburden	Subaerial	Quarry Rock	P95	TMF Overburden	P95	790.3	0.3	0.2	0.5	mg/L
11.4	Tailings Dam	Pit Overburden	Subaerial	Quarry Rock	P95	OP Overburden	P95	790.3	0.3	0.2	0.5	mg/L
11.5	Tailings Dam	Tailings	Subaeral	Tailings	Initial	Analog	7<pH<8	790.3	0.3	1	1	mg/L
12.1	Seepage dams	TMF Overburden	Subaerial	Quarry Rock	P95	TMF Overburden	P95	790.3	0.3	0.2	0.5	mg/L
12.2	Seepage dams	Pit Overburden	Subaerial	Quarry Rock	P95	OP Overburden	P95	790.3	0.3	0.2	0.5	mg/L
12.3	Seepage dams	Local Overburden	Subaerial	Quarry Rock	P95	TMF Overburden	P95	790.3	0.3	0.2	0.5	mg/L
14	Haul Road and Crusher Pad	Pit Rock	Subaerial	Waste Rock	P95	Analog	7<pH<8	564.5	0.5	0.2	0.5	mg/L
15	Live ore pad and Live ore	Pit Rock	Subaerial	Ore	P95	Analog	7<pH<8	564.5	0.5	0.2	0.5	mg/L
16	Cleaner tailings	Tailings	Subaqueous	Not Applicable	Not Applicable	Testwork	P95	Not Applicable	Not Applicable	Not Applicable	Not Applicable	mg/L
17.1	Rougher tailings	Tailings	Subaerial	Tailings	Initial	None	-	-	0.3	1	1	mg/m2/year
17.2	Rougher tailings	Tailings	Subaerial	Tailings	Initial	Analog	7<pH<8	790.3	0.3	1	1	mg/L
18	Rougher tailings	Tailings	Subaqueous	Not Applicable	Not Applicable	Testwork	P95	Not Applicable	Not Applicable	Not Applicable	Not Applicable	mg/L
19	Topsoil Stockpiles	Overburden	Subaerial	Quarry Rock	P95	TMF Overburden	P95	790.3	0.3	0.2	0.5	mg/L
20	Plant site	Pit Rock	Subaerial	Waste Rock	P95	Analog	7<pH<8	564.5	0.5	0.2	0.5	mg/L
21	Quarry	Wall	Subaerial	Quarry Rock	P95	Analog	7<pH<8	101.61	0.3	0.1	0.5	mg/L
22	Ore to Process	Pit Rock	Flushing	Ore	P95	Analog	7<pH<8	-	0.5	0.1	0.5	mg/tonne
23.1	Low grade ore to process	Pit Rock	Flushing	Ore	P95	Analog	7<pH<8	-	0.5	0.2	0.5	mg/tonne
23.2	Process and Pond Constraints	-	-	-	-	Calculated	Process Pond pH	-	-	-	-	mg/L
24	Reagents	Reagents	Flushing	-	-	-	-	-	-	-	-	-

Appendix H: Analog Database



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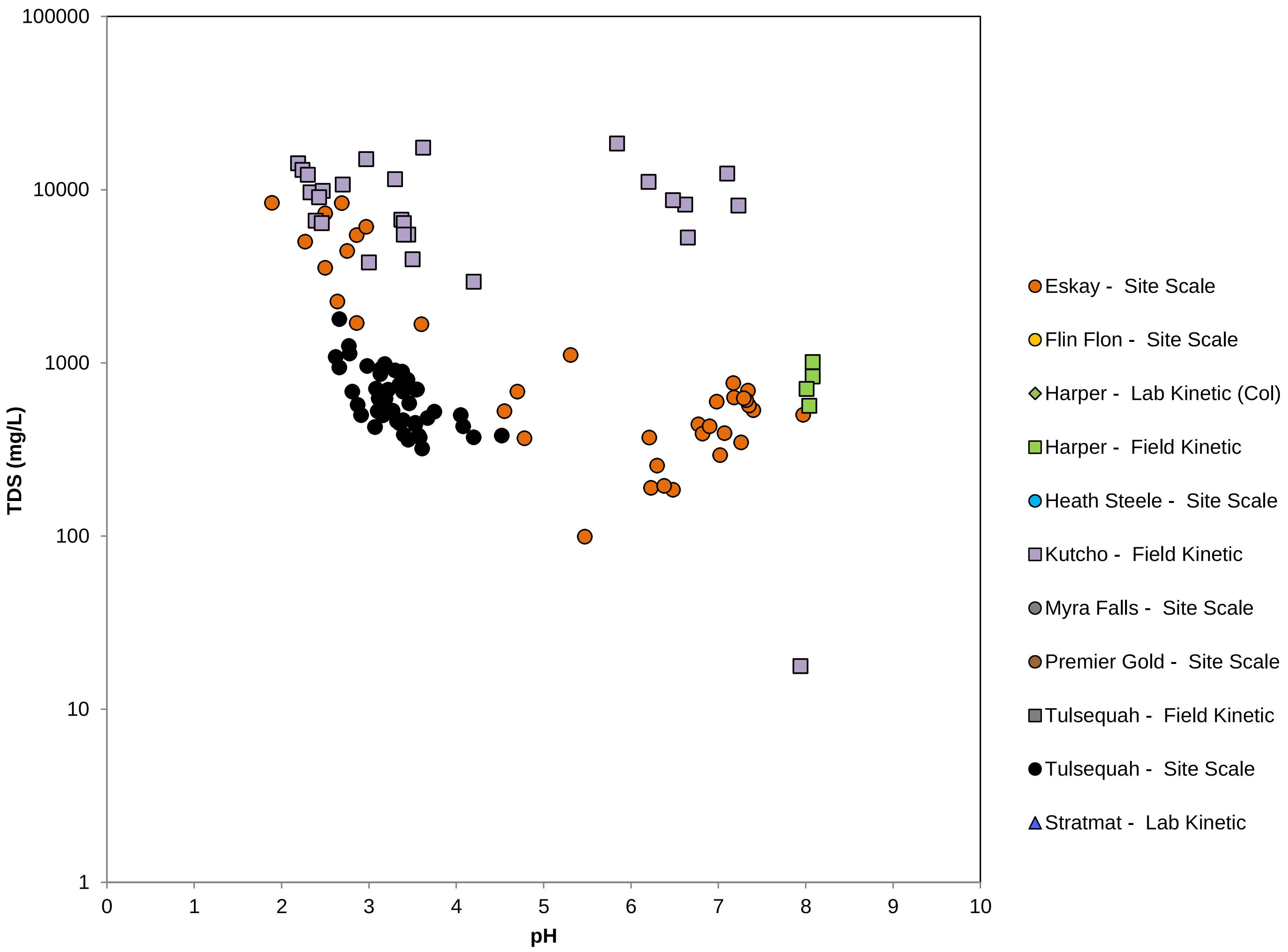

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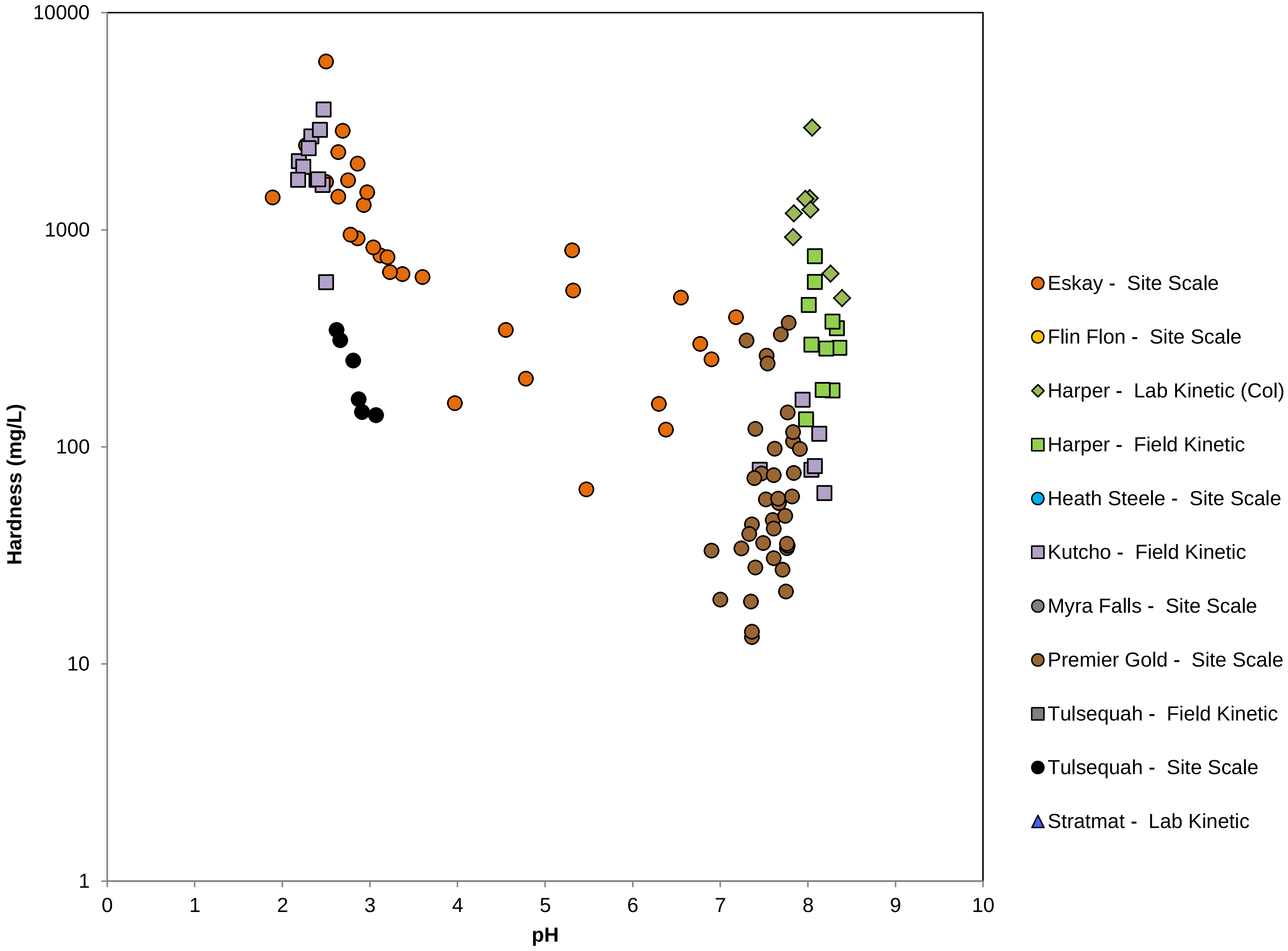

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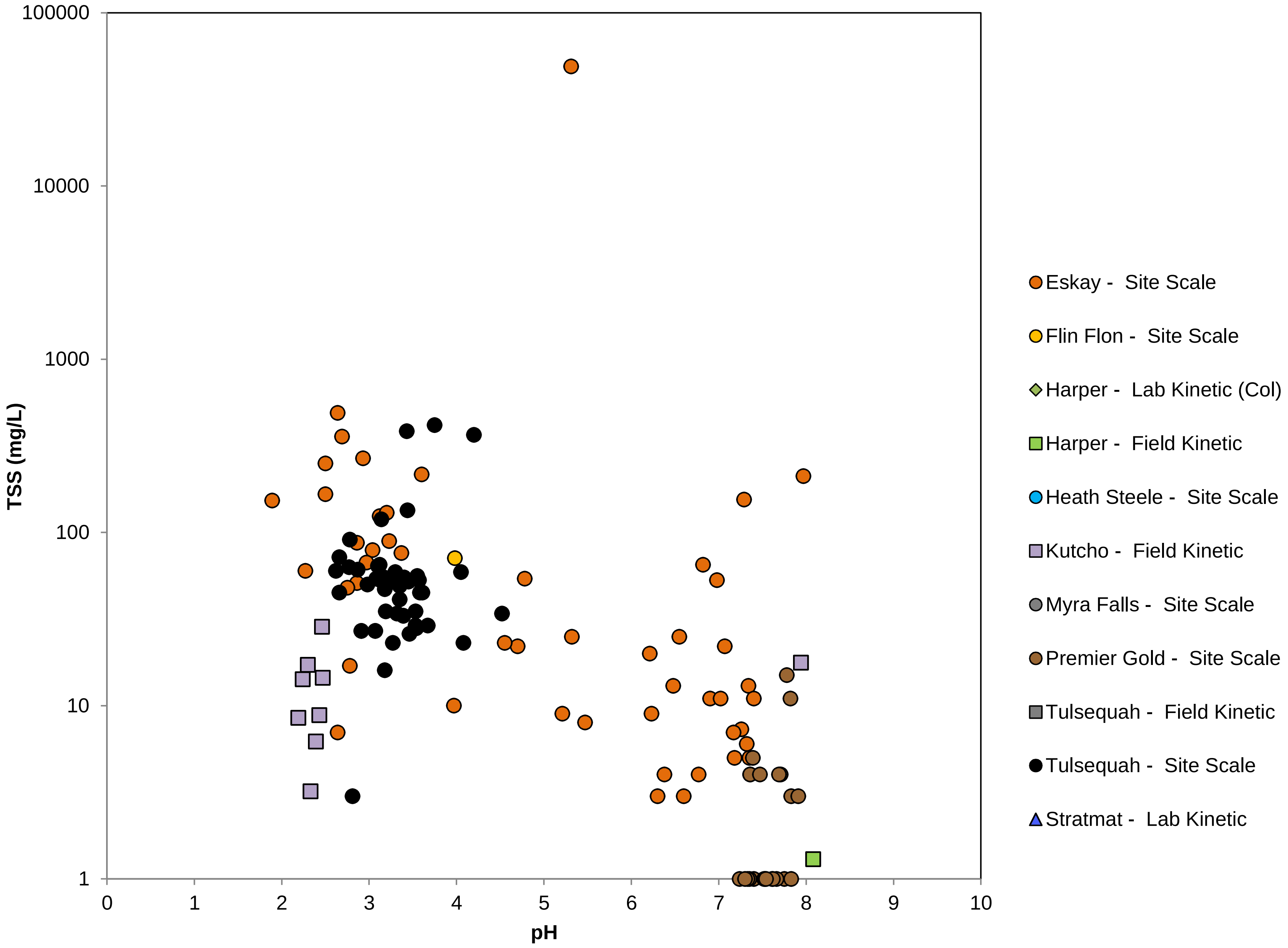

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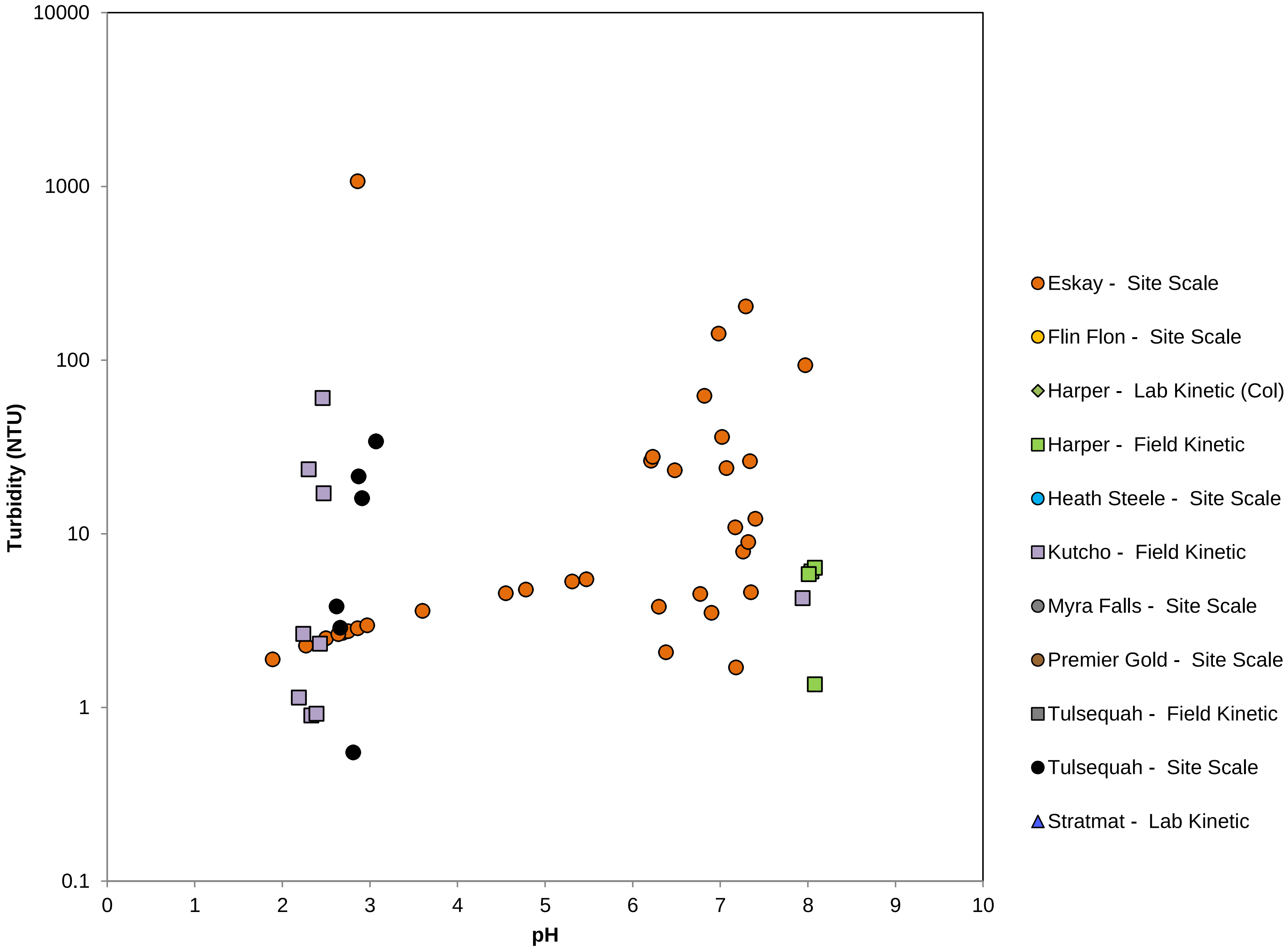

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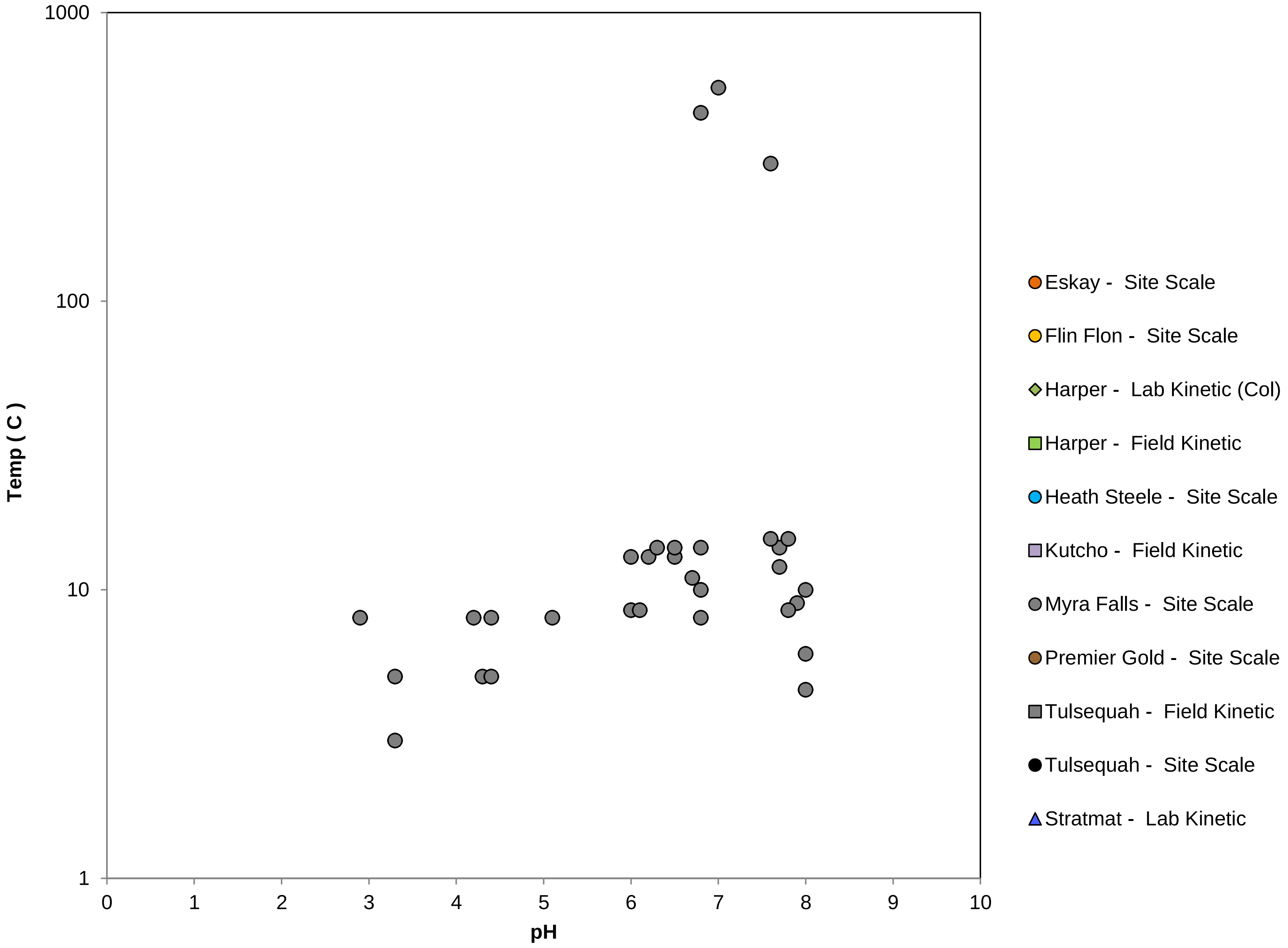

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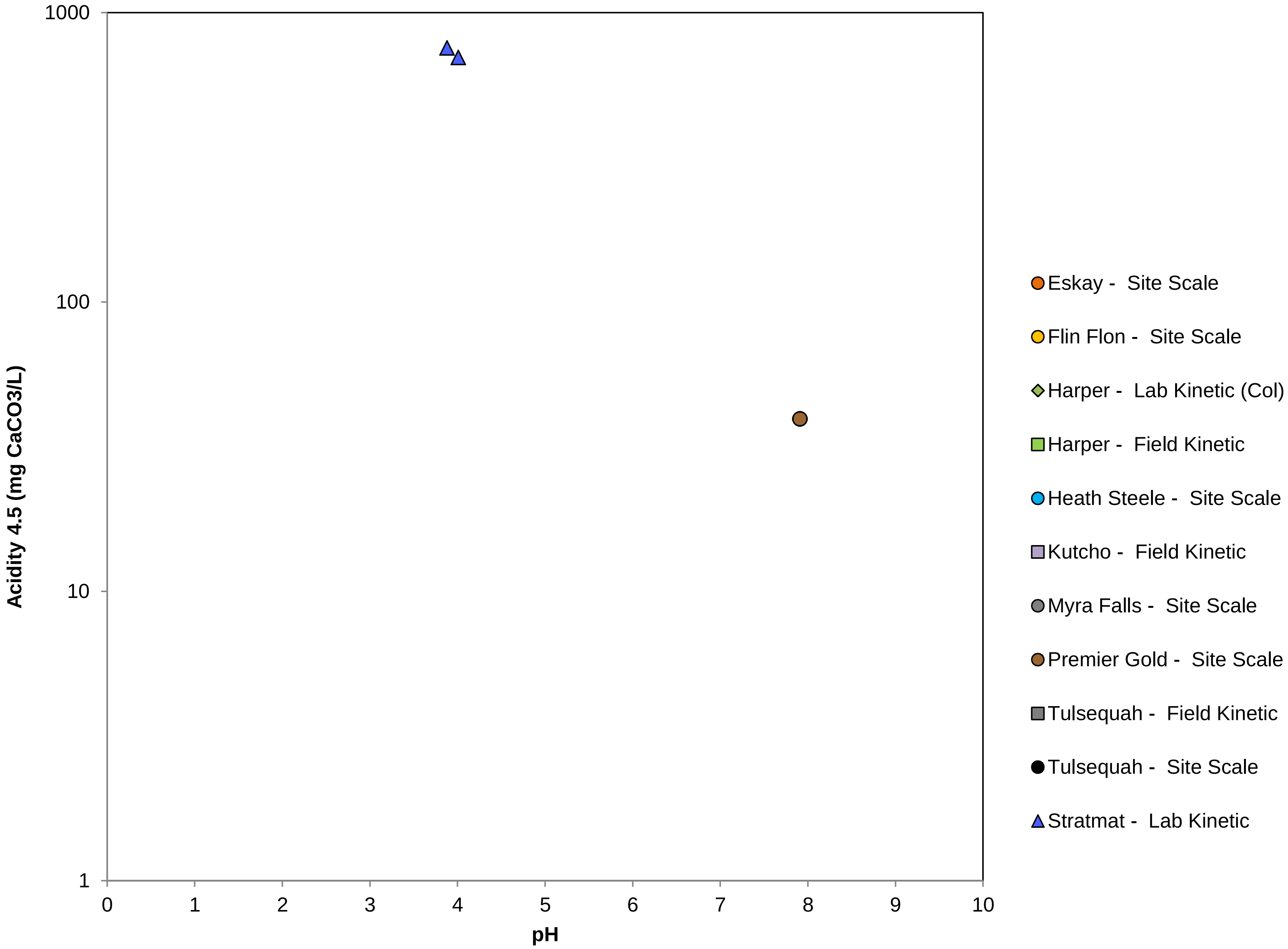

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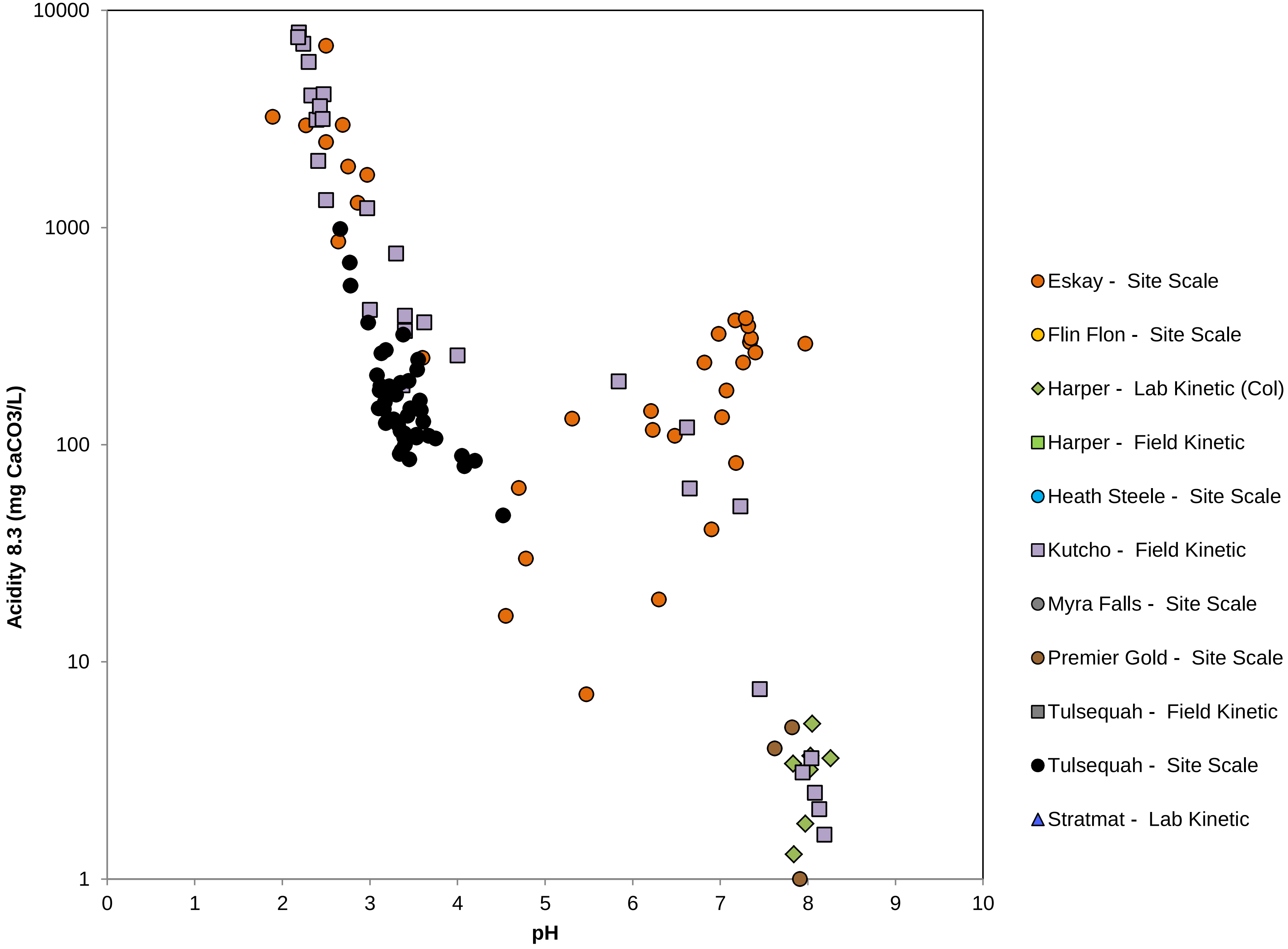

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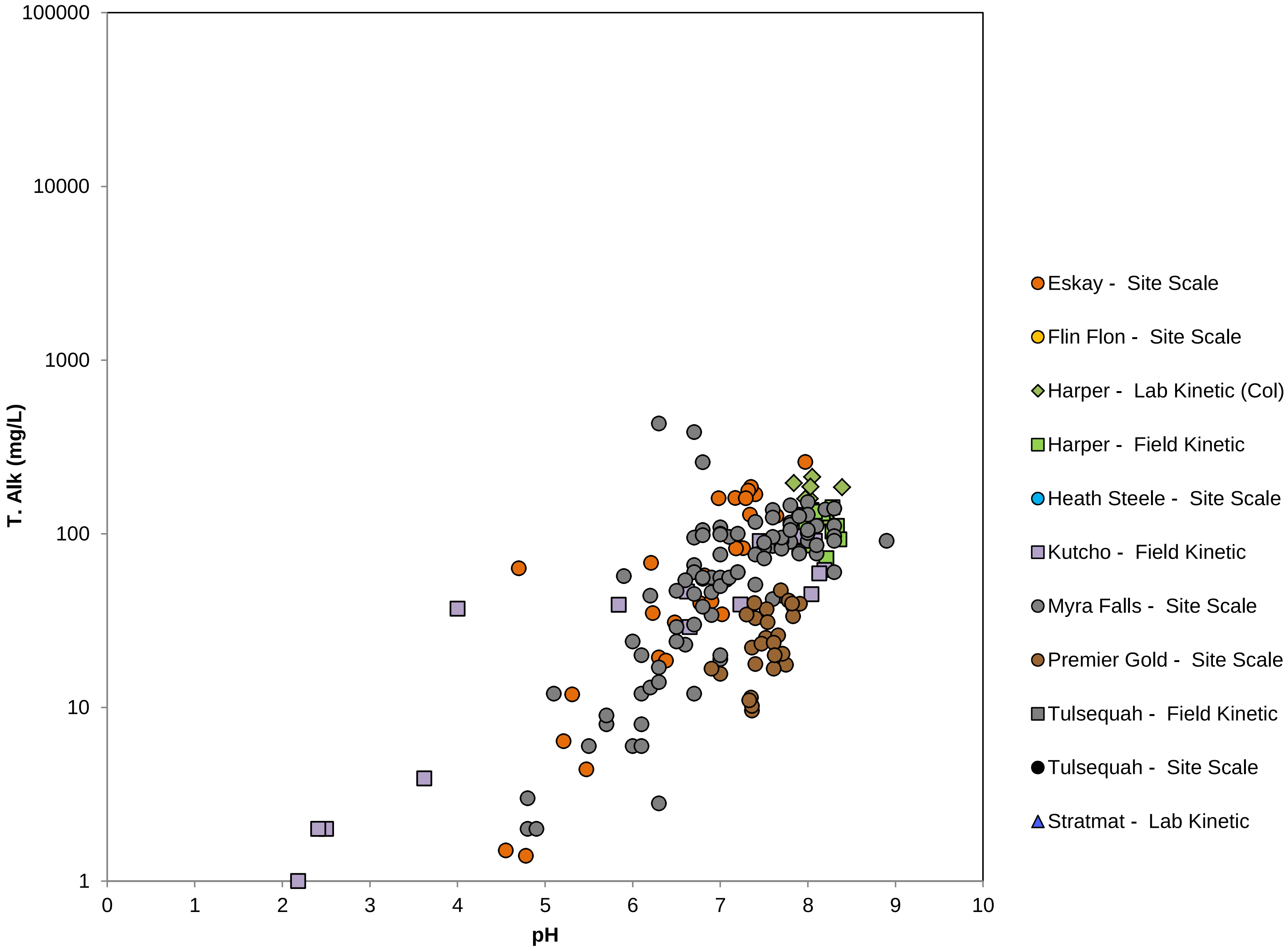

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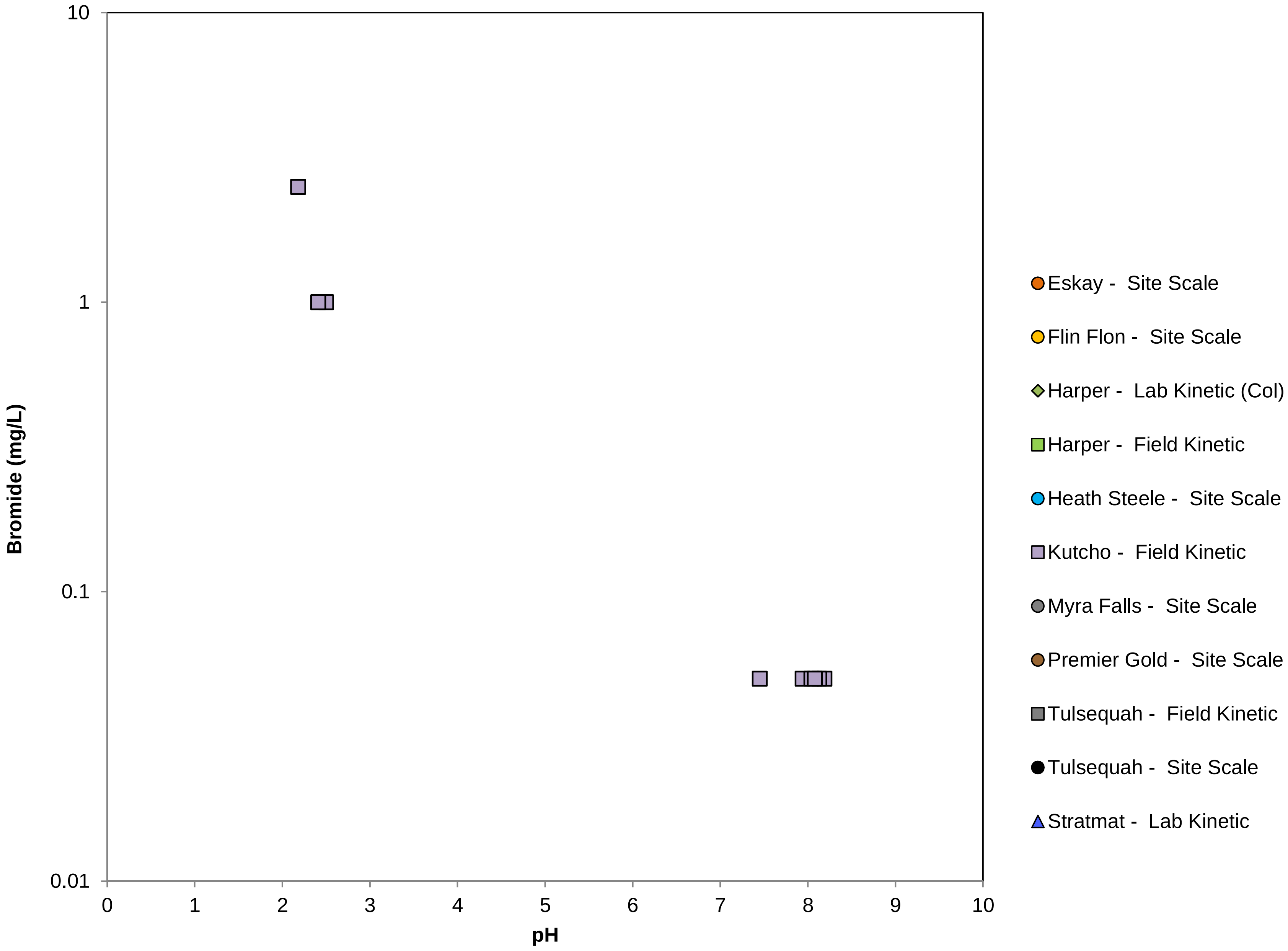

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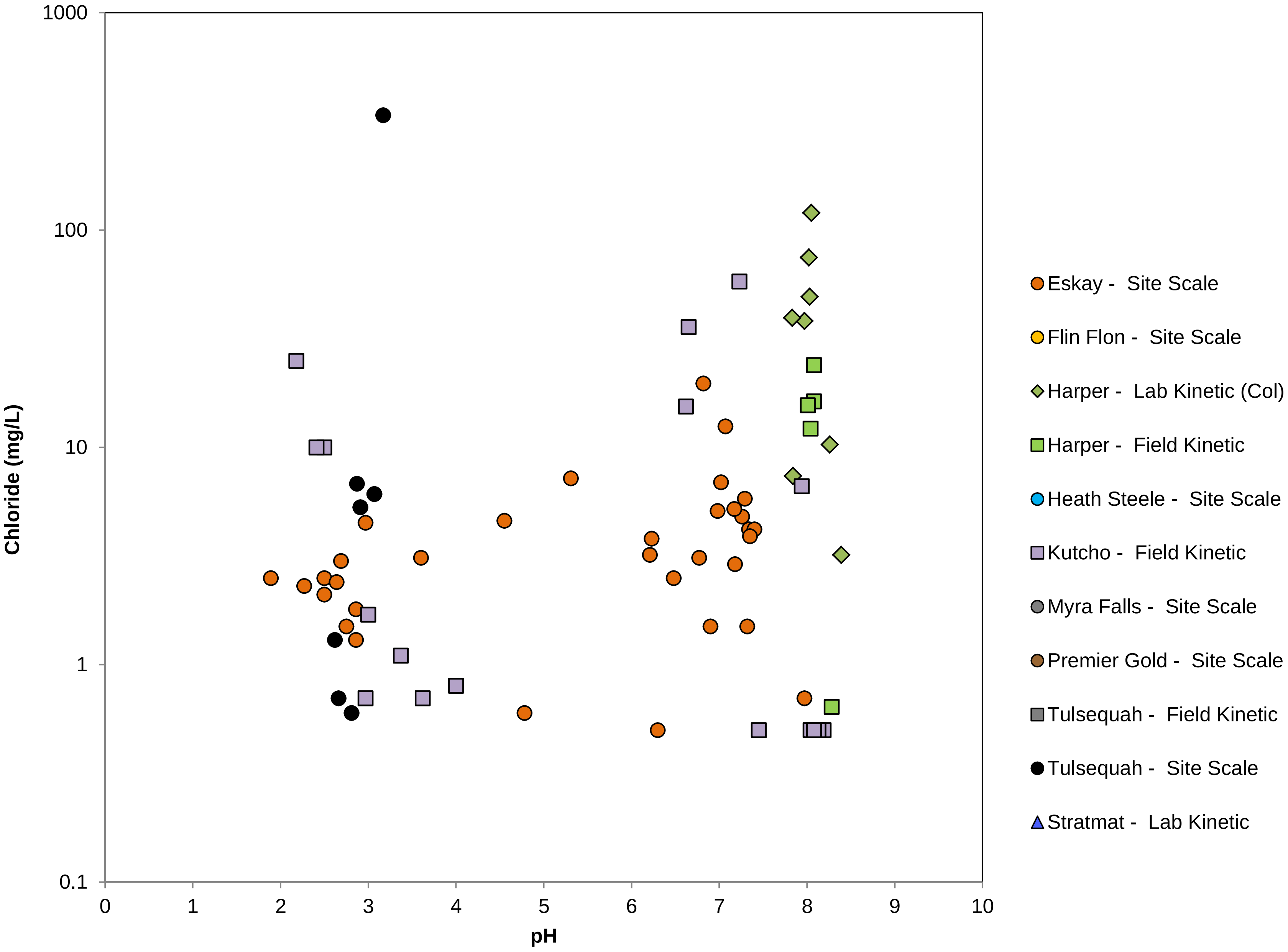

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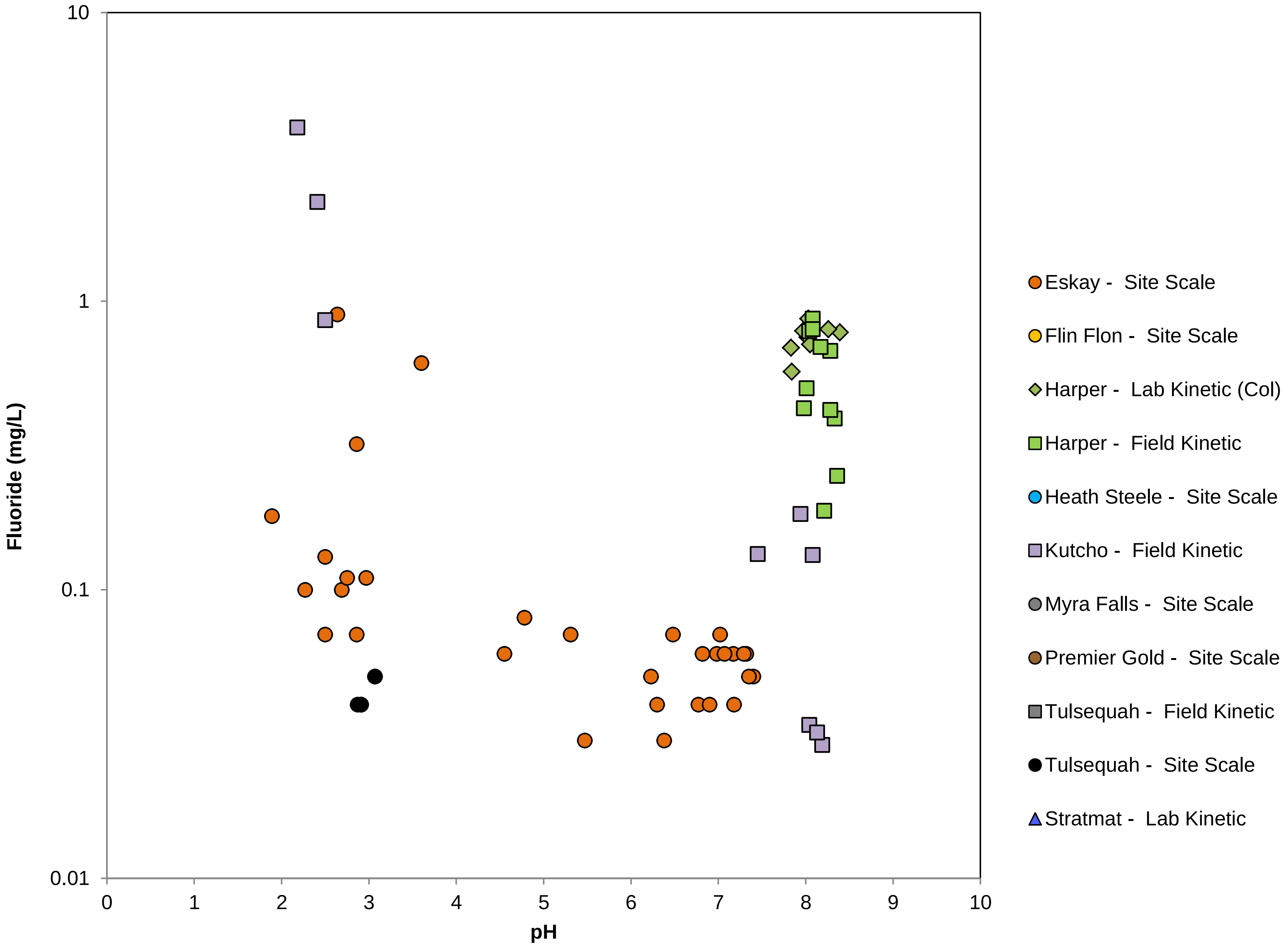

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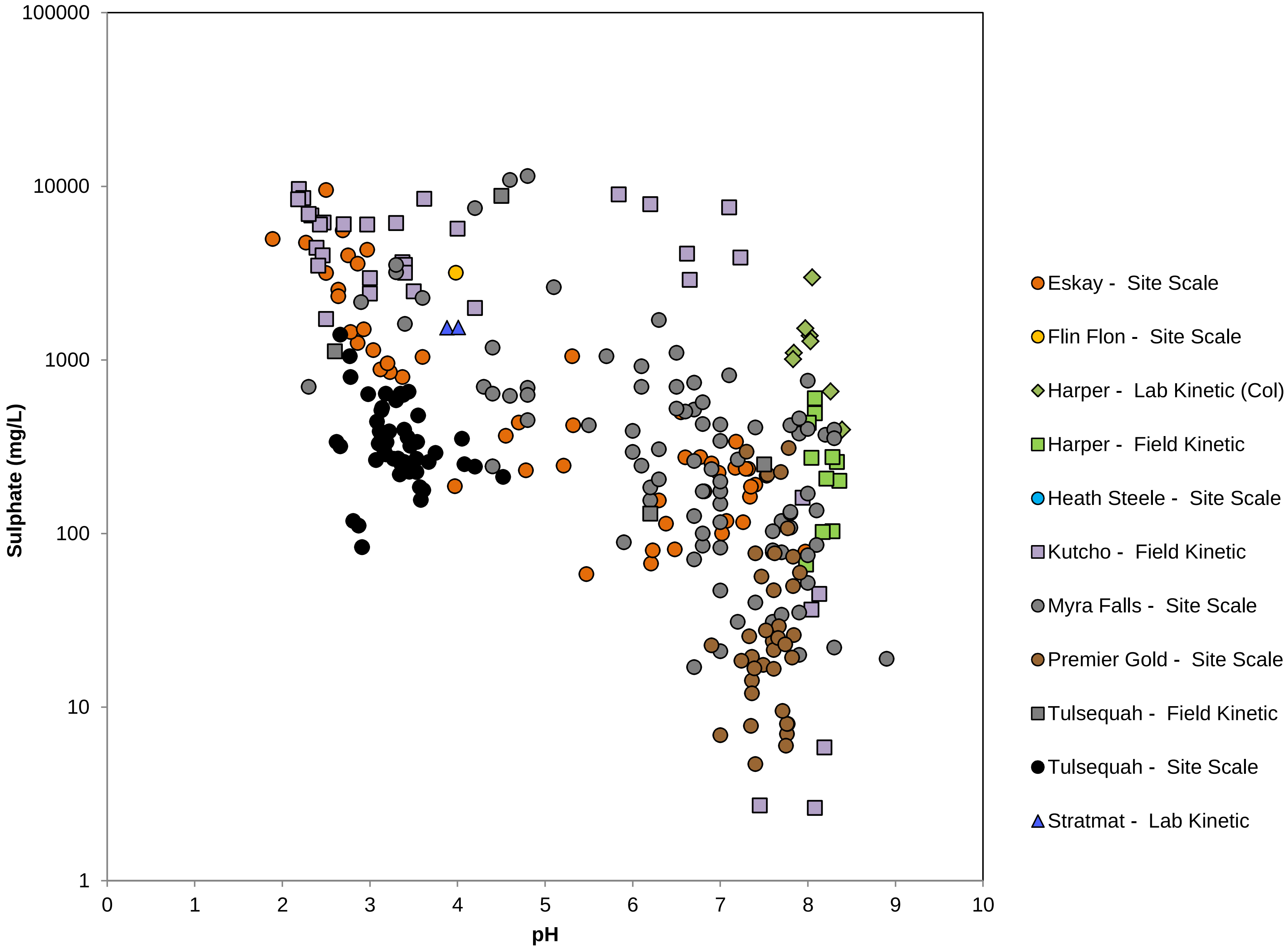

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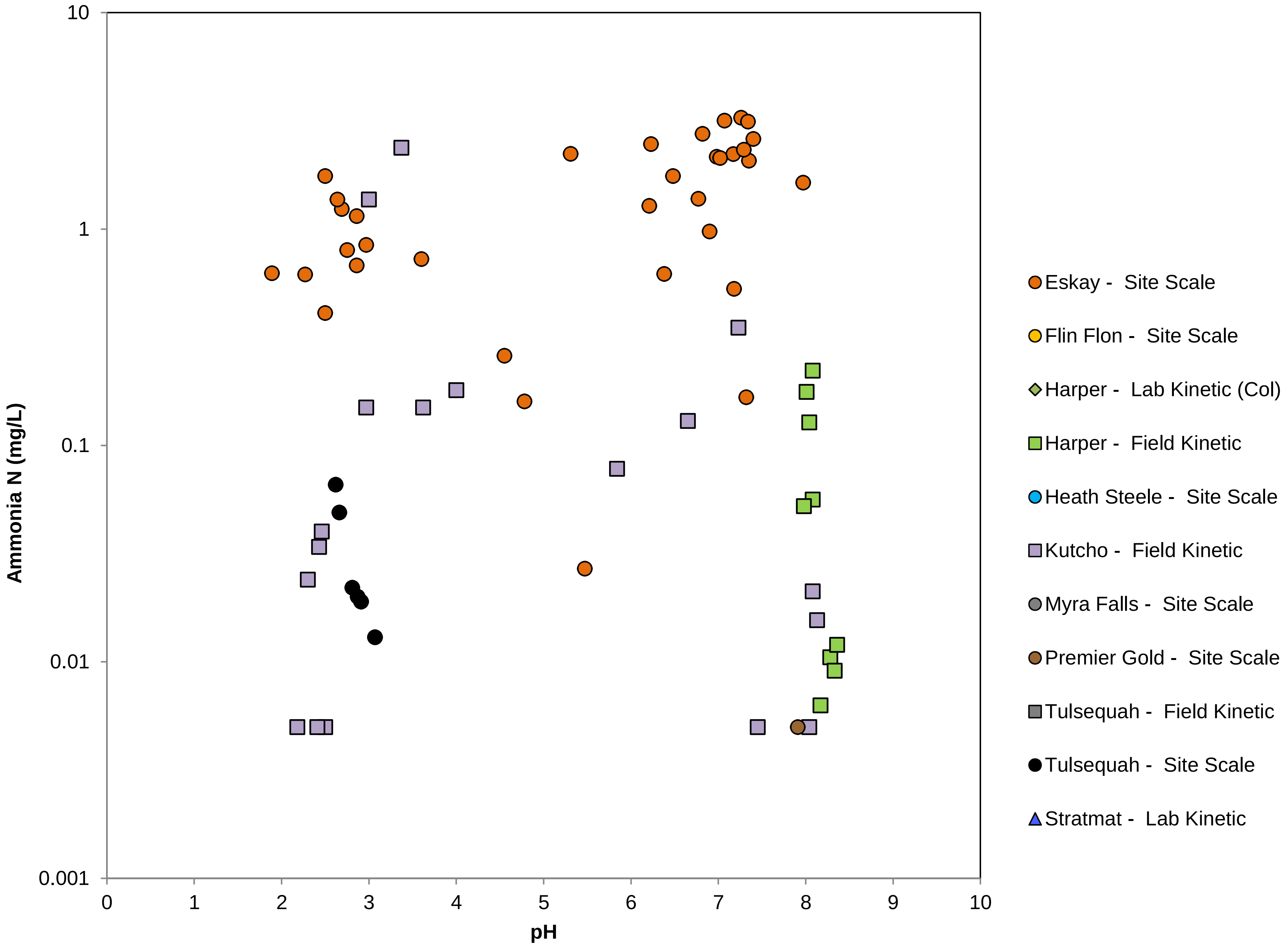

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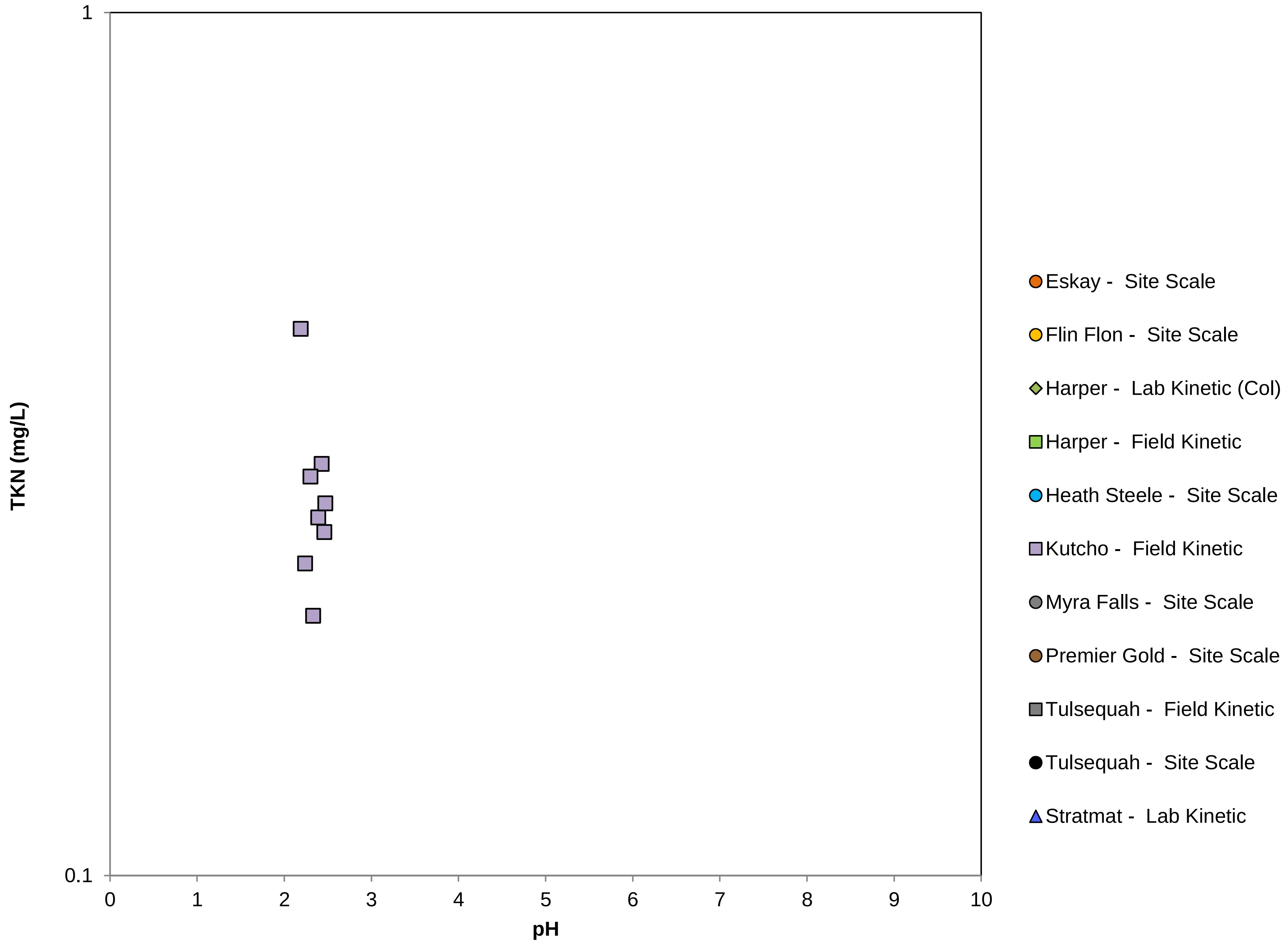

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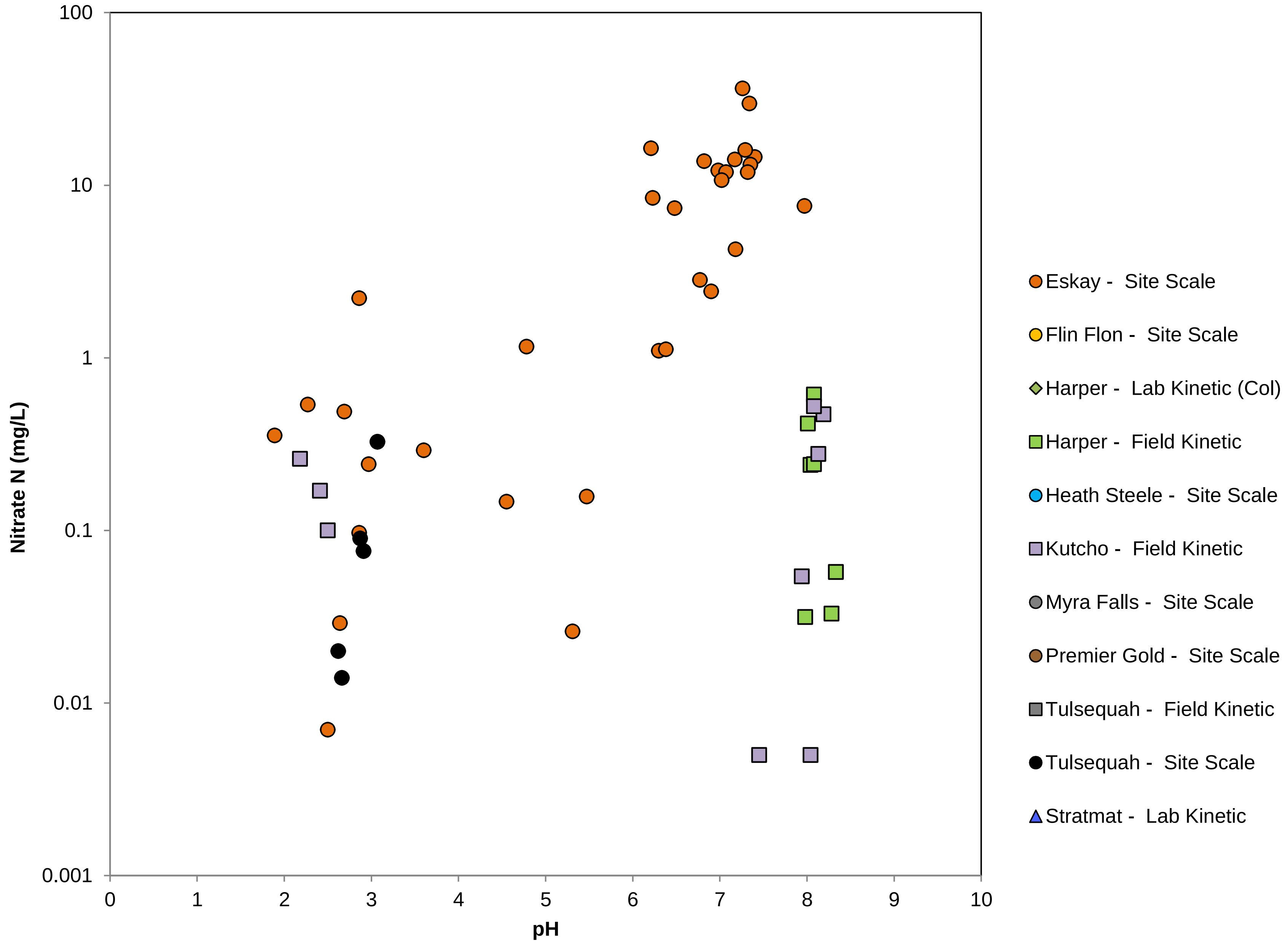

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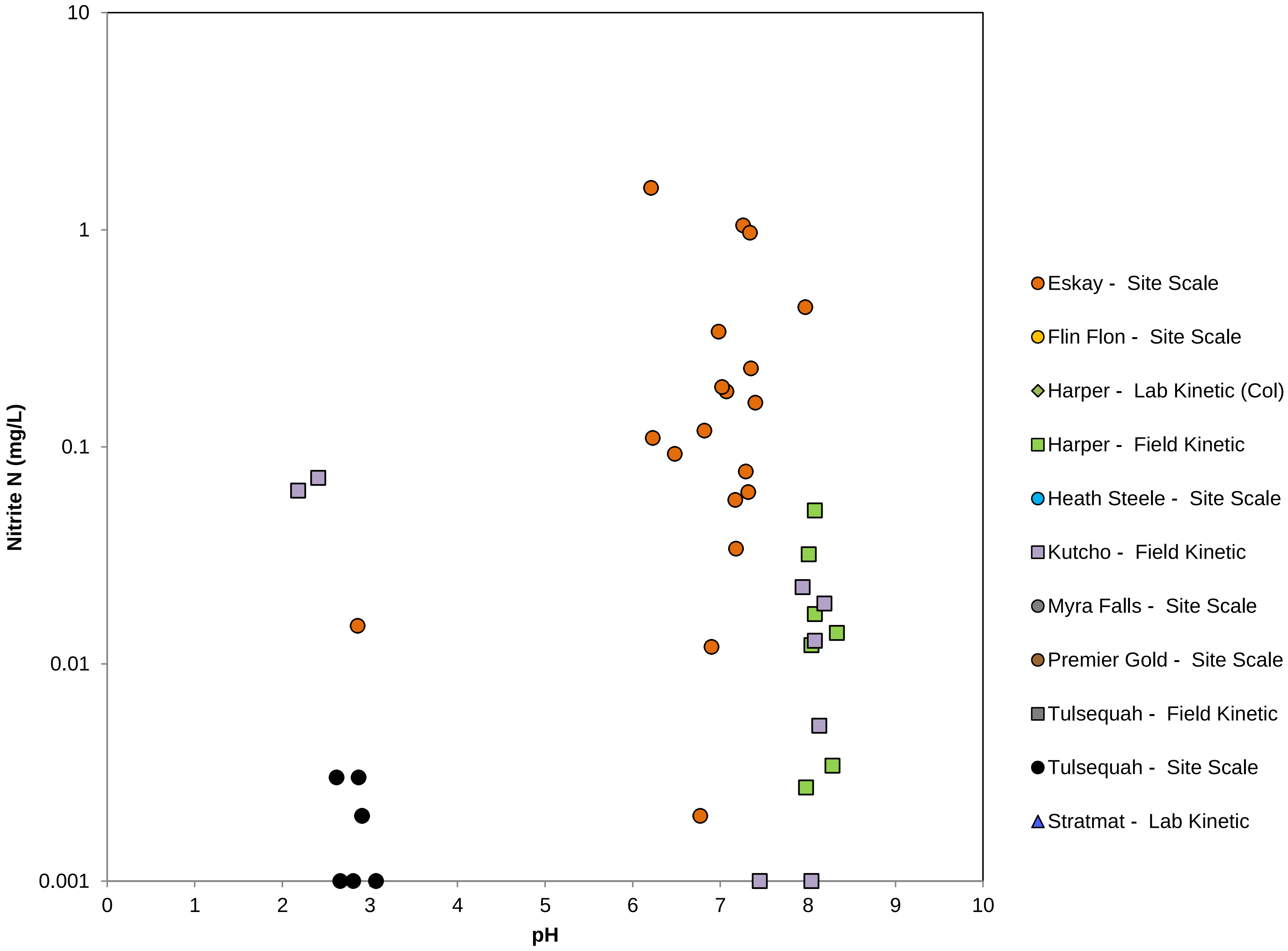

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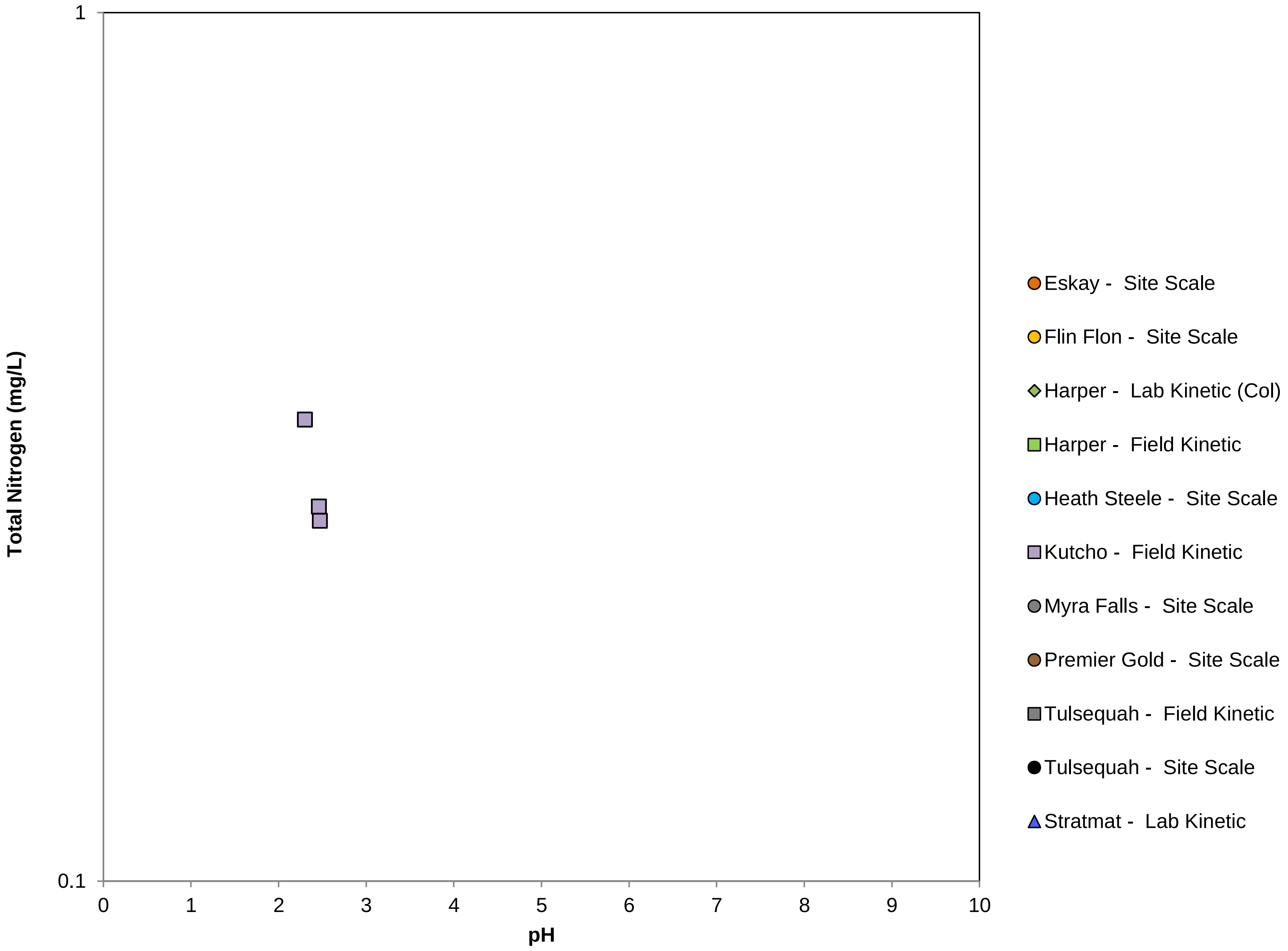

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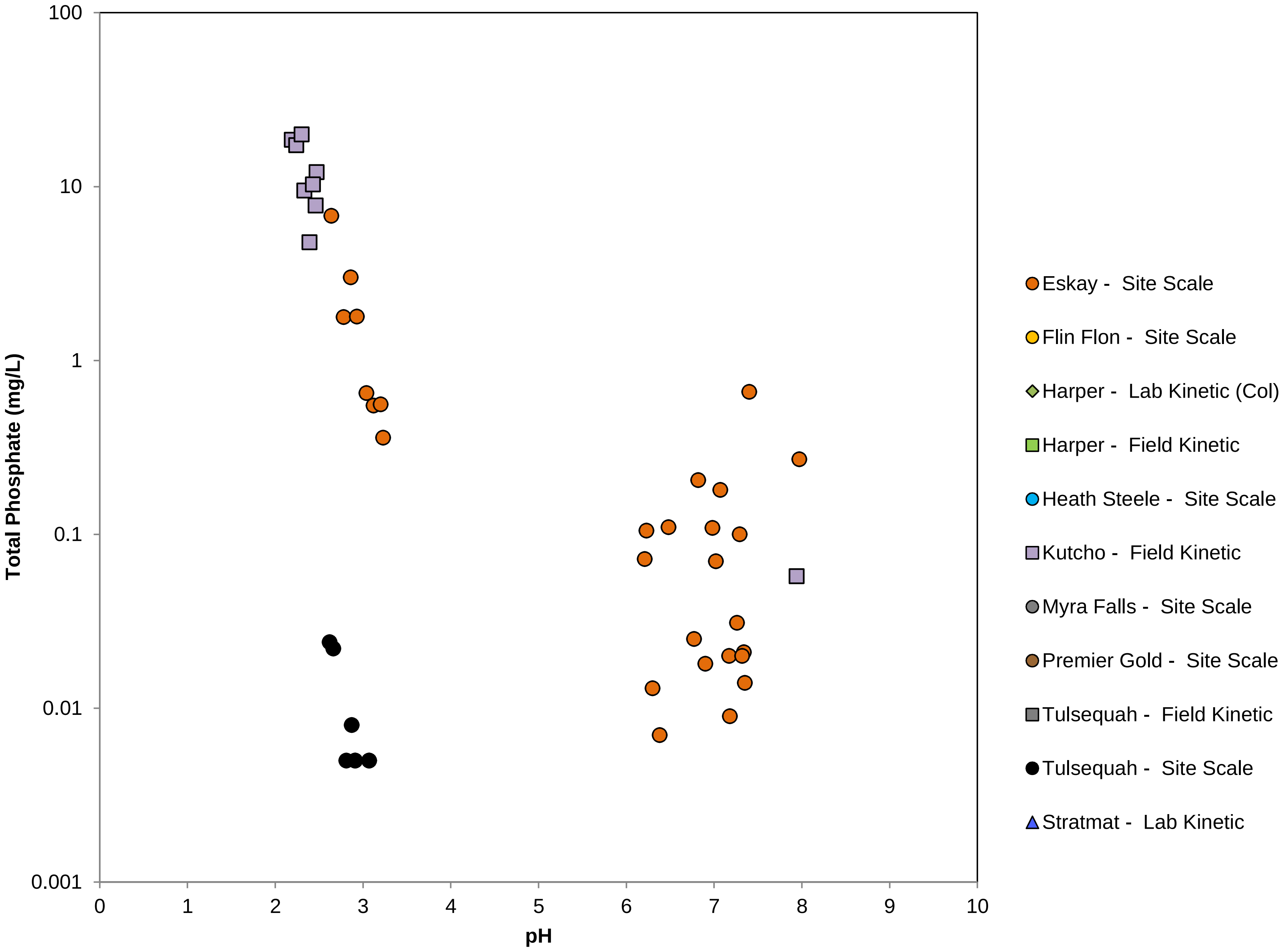

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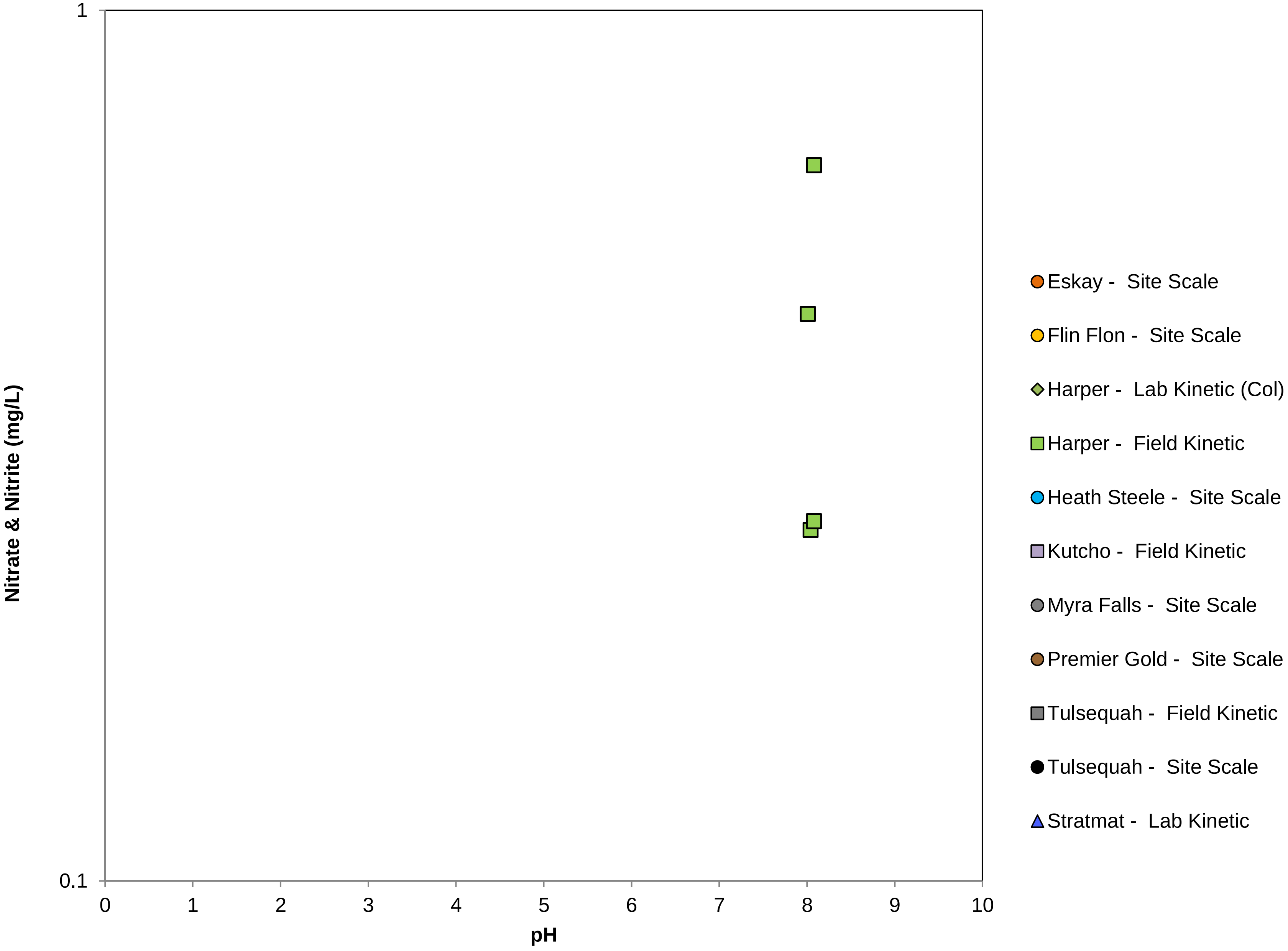

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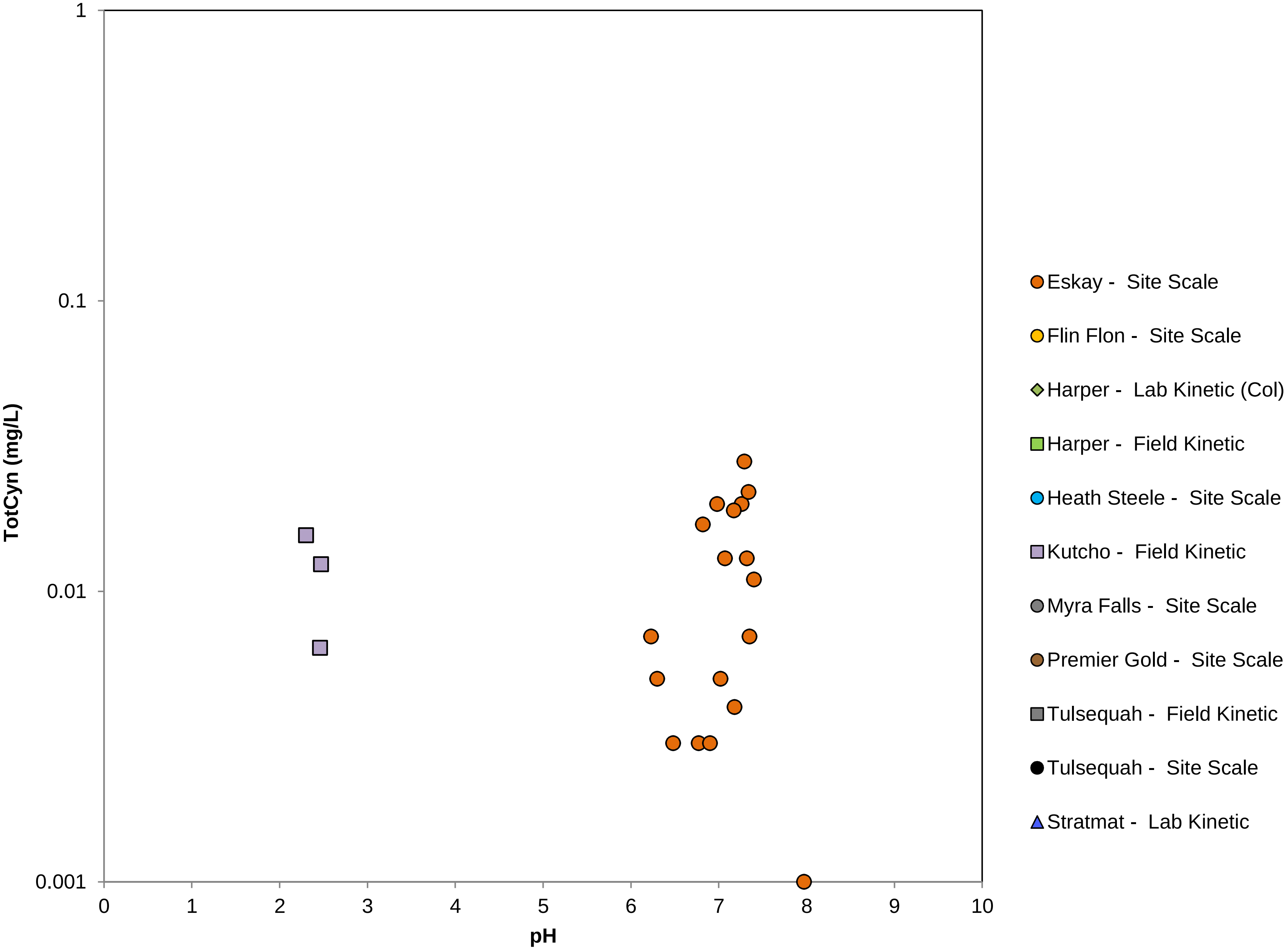

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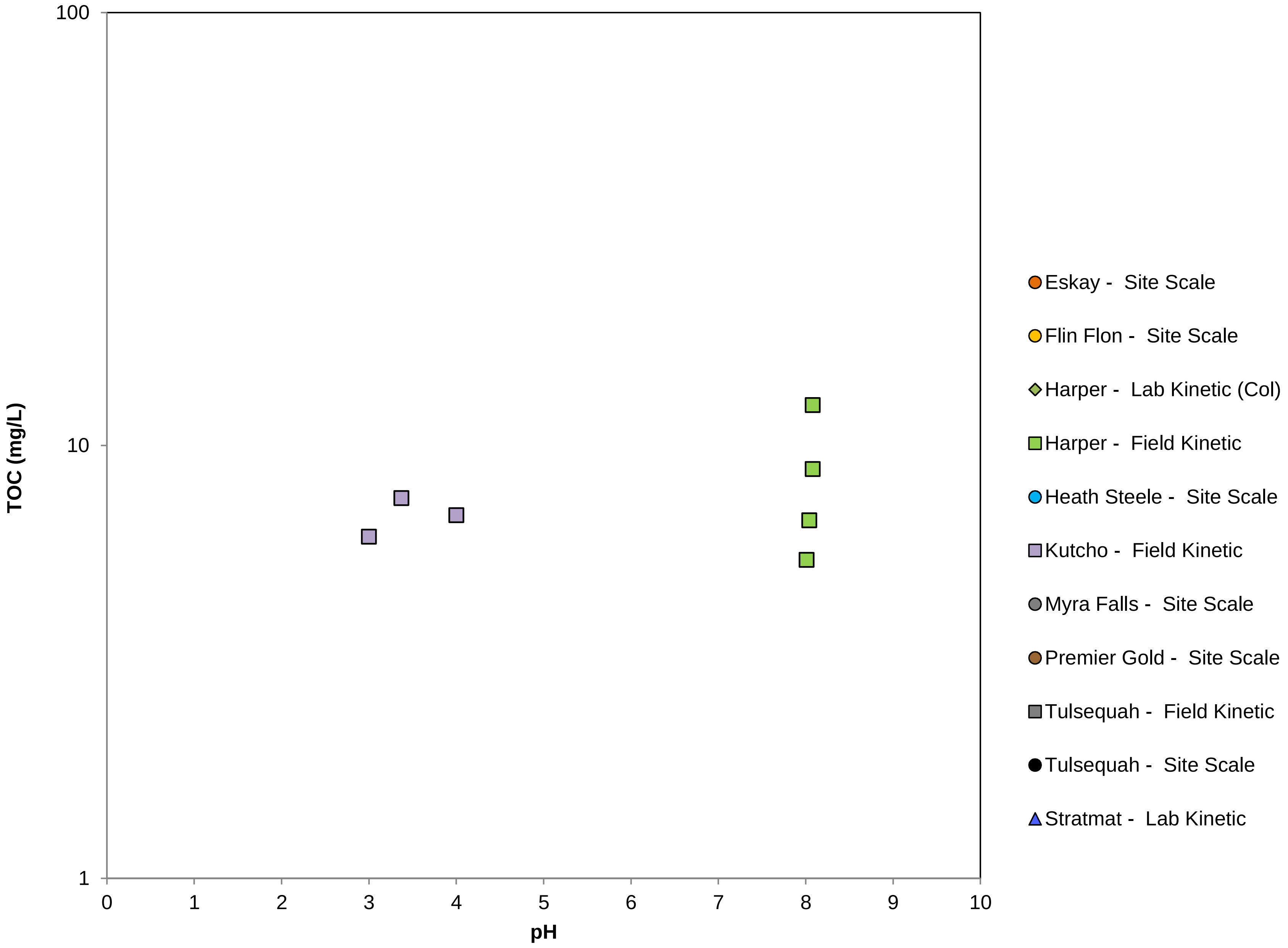

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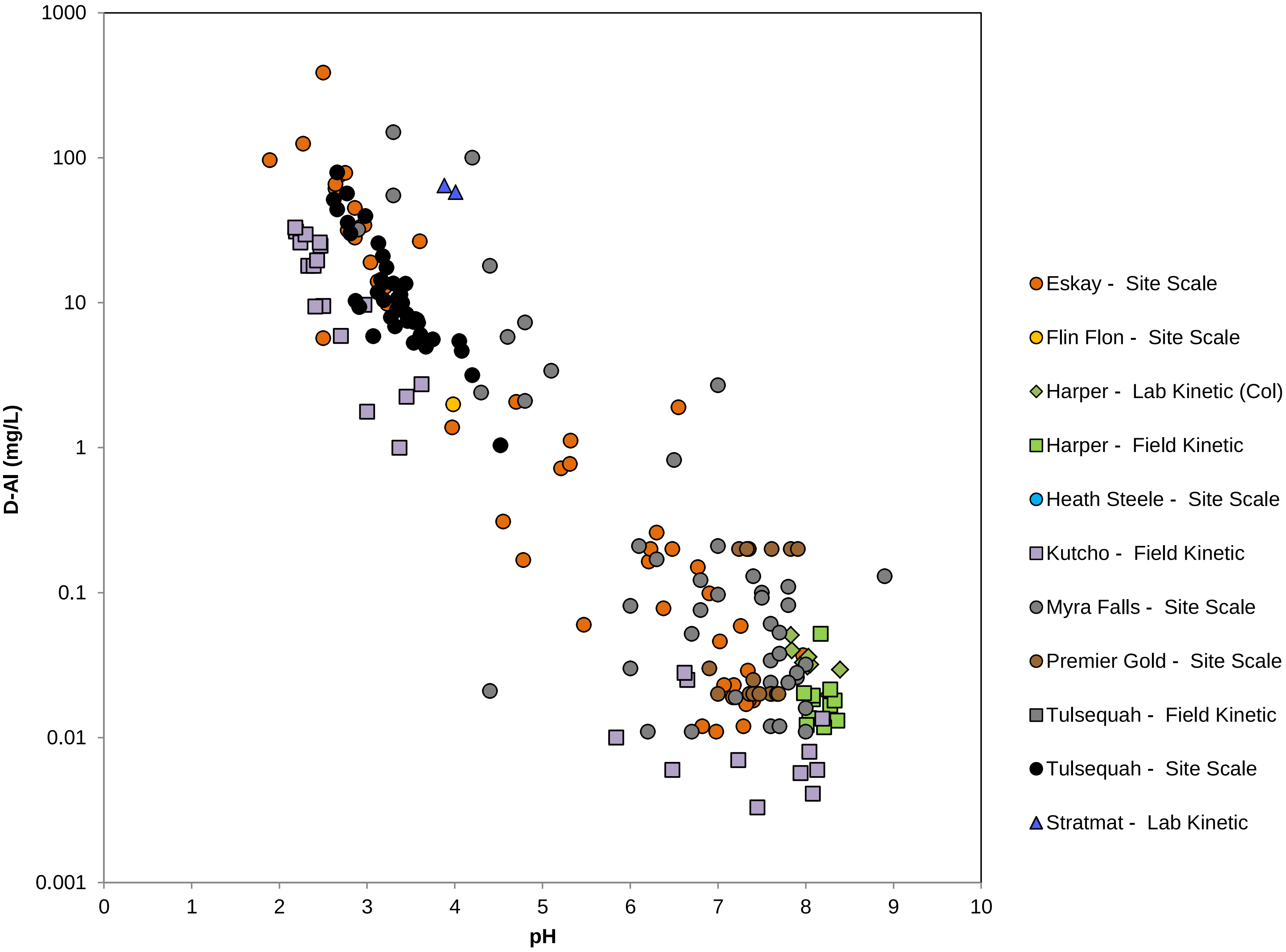

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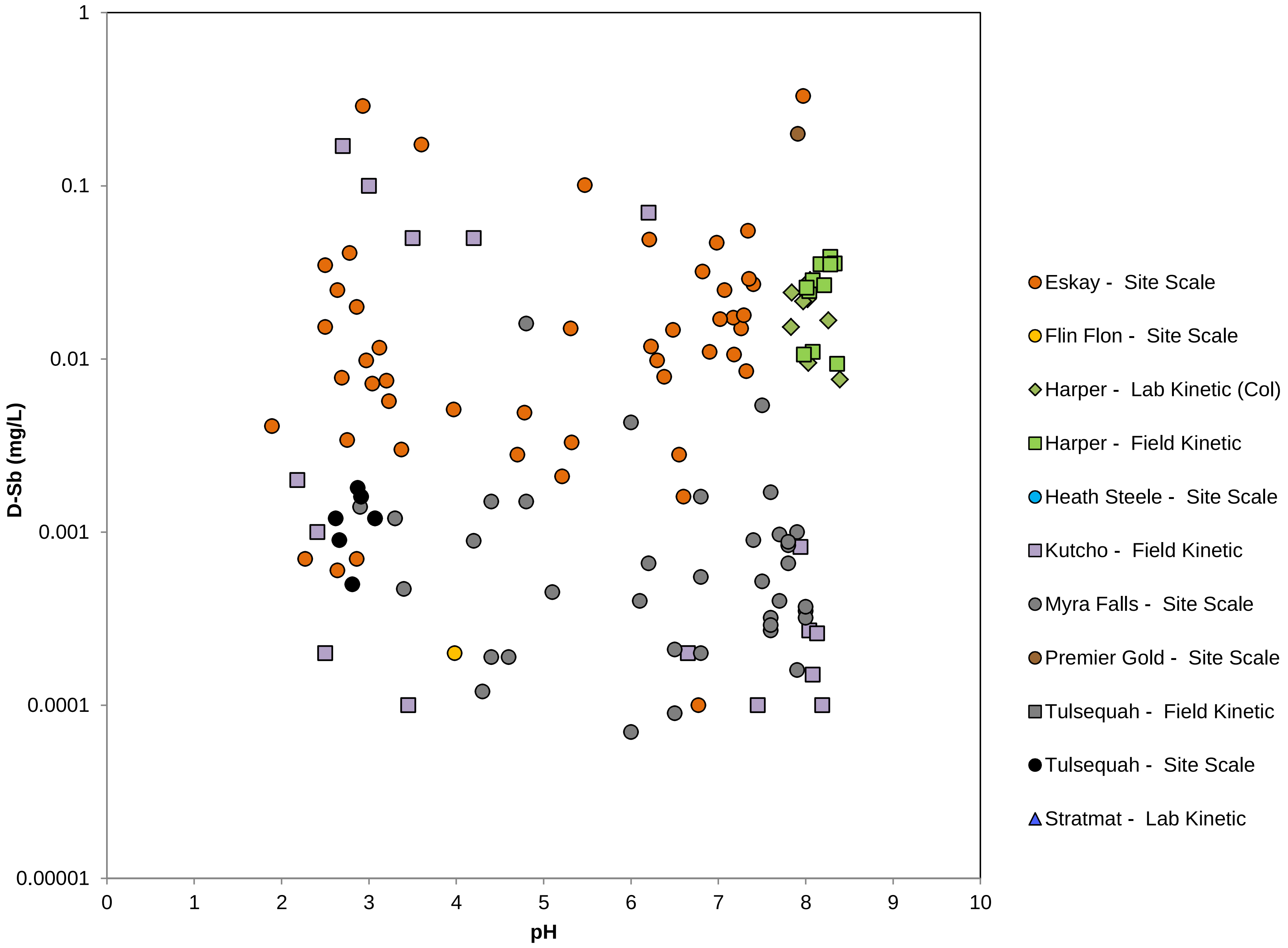

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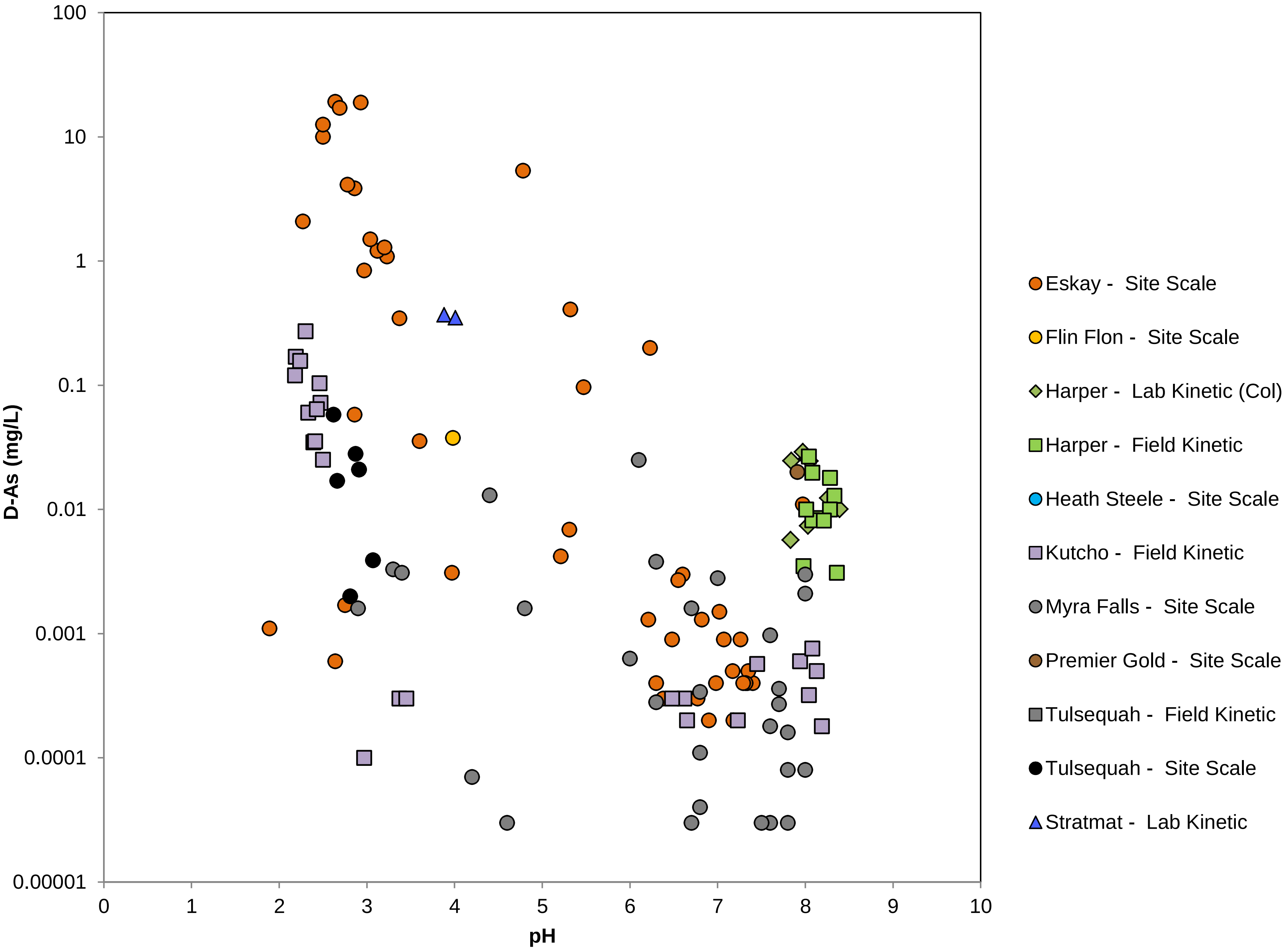

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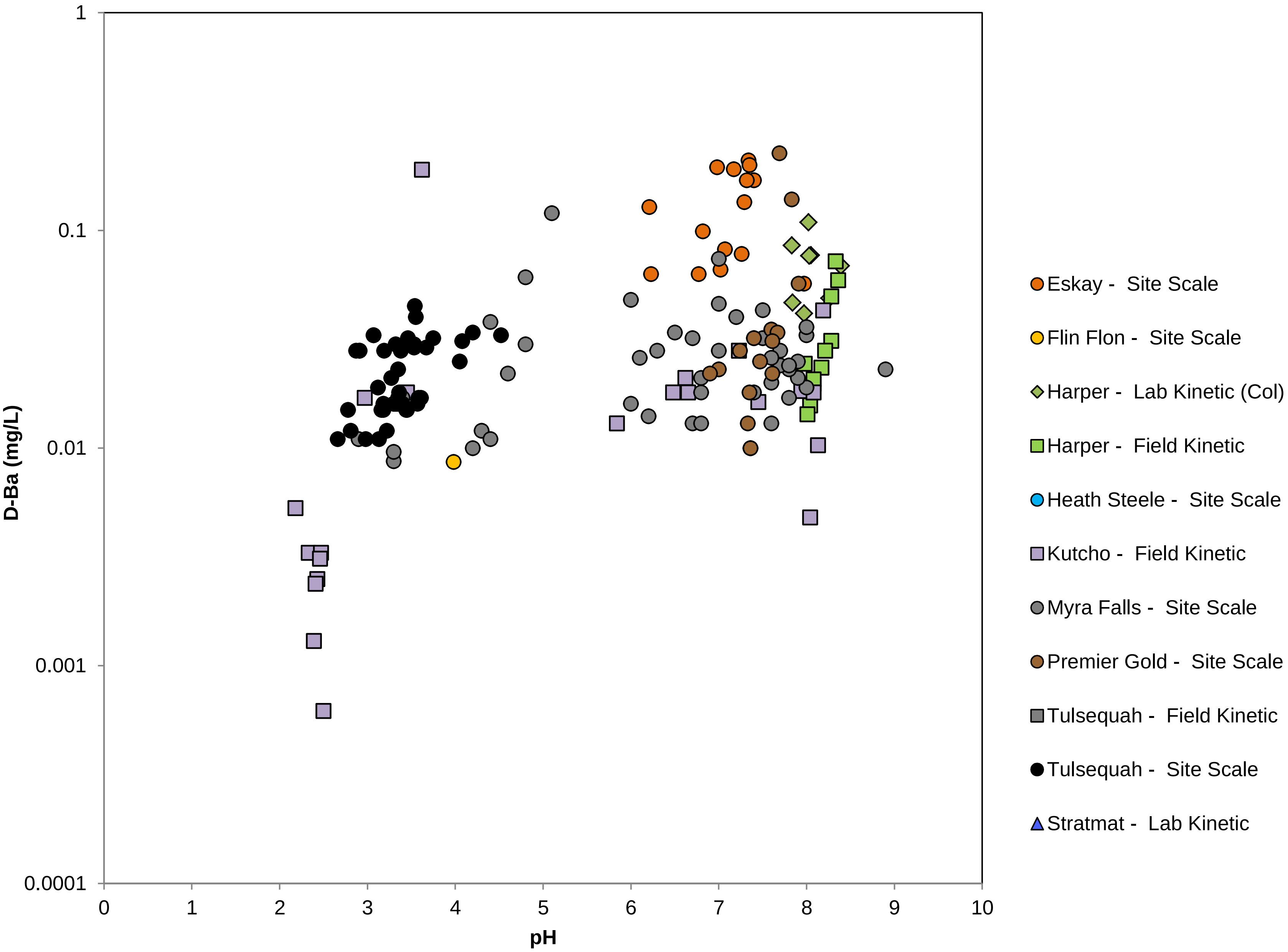
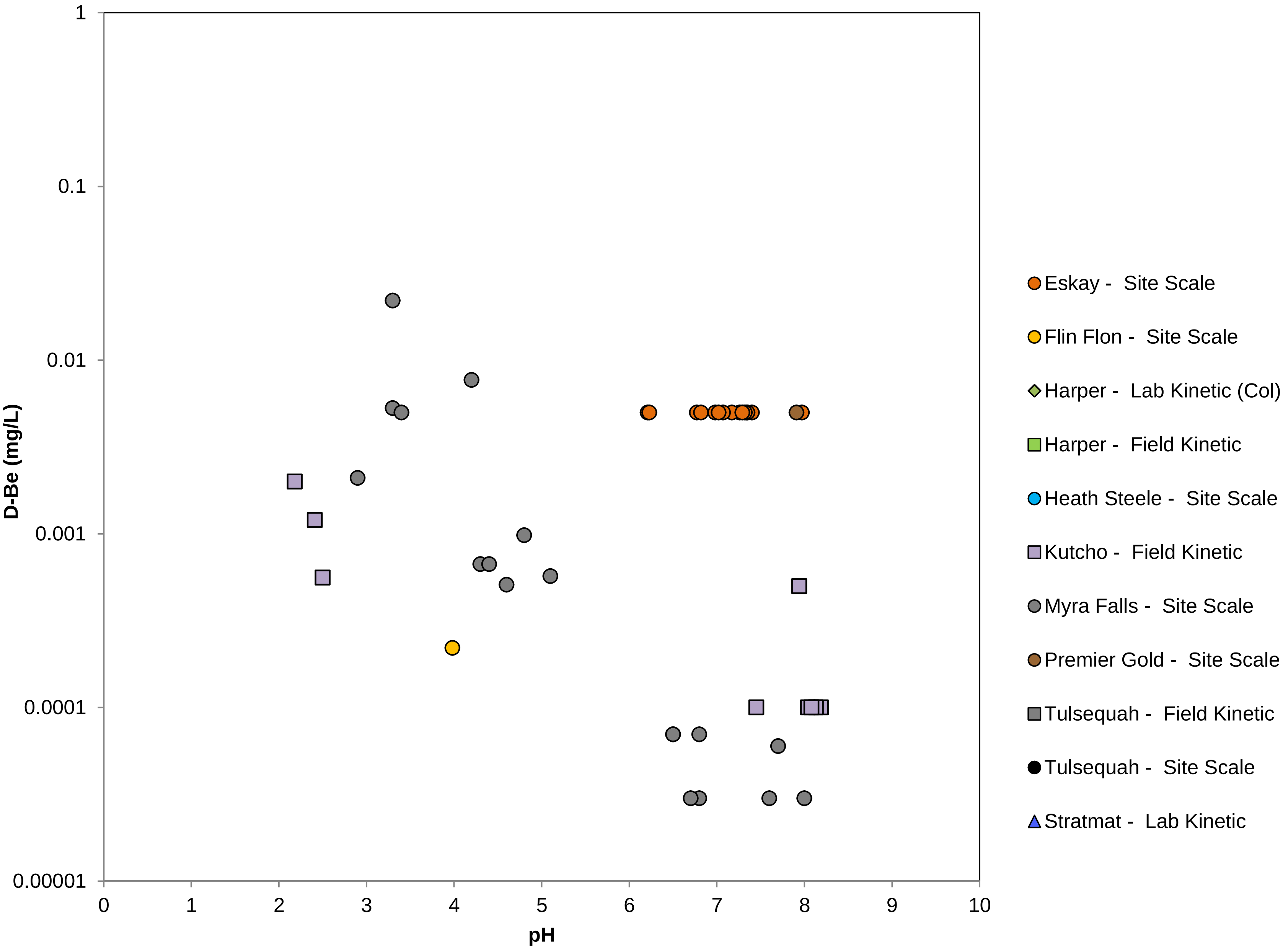

1
0.1
0.01
0.001
0.0001
D-Ba (mg/L)
0
1
2
3
4
5
6
7
8
9
10
pH
● Eskay - Site Scale
● Flin Flon - Site Scale
◆ Harper - Lab Kinetic (Col)
■ Harper - Field Kinetic
● Heath Steele - Site Scale
■ Kutcho - Field Kinetic
● Myra Falls - Site Scale
● Premier Gold - Site Scale
■ Tulsequah - Field Kinetic
● Tulsequah - Site Scale
▲ Stratmat - Lab Kinetic



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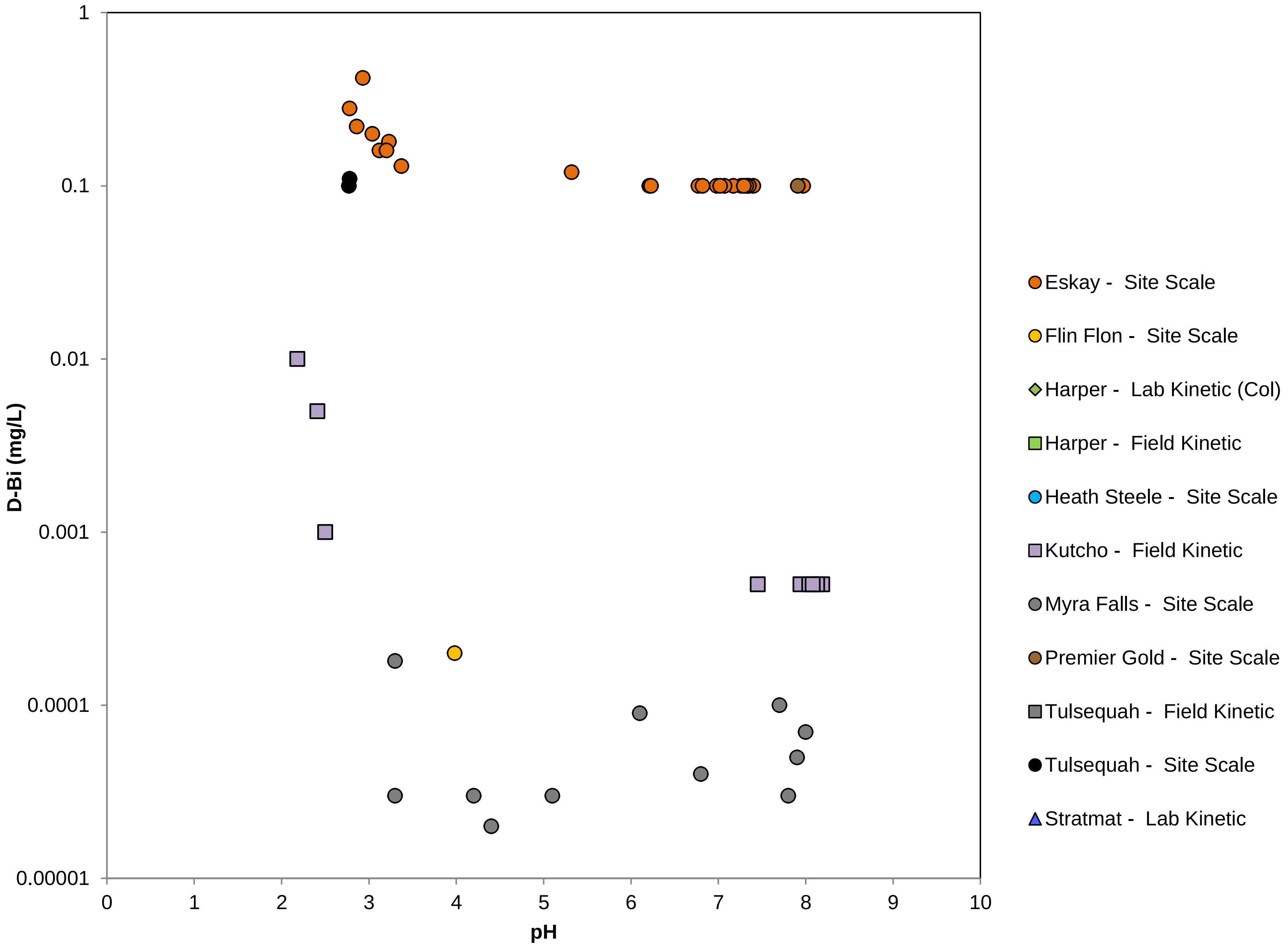

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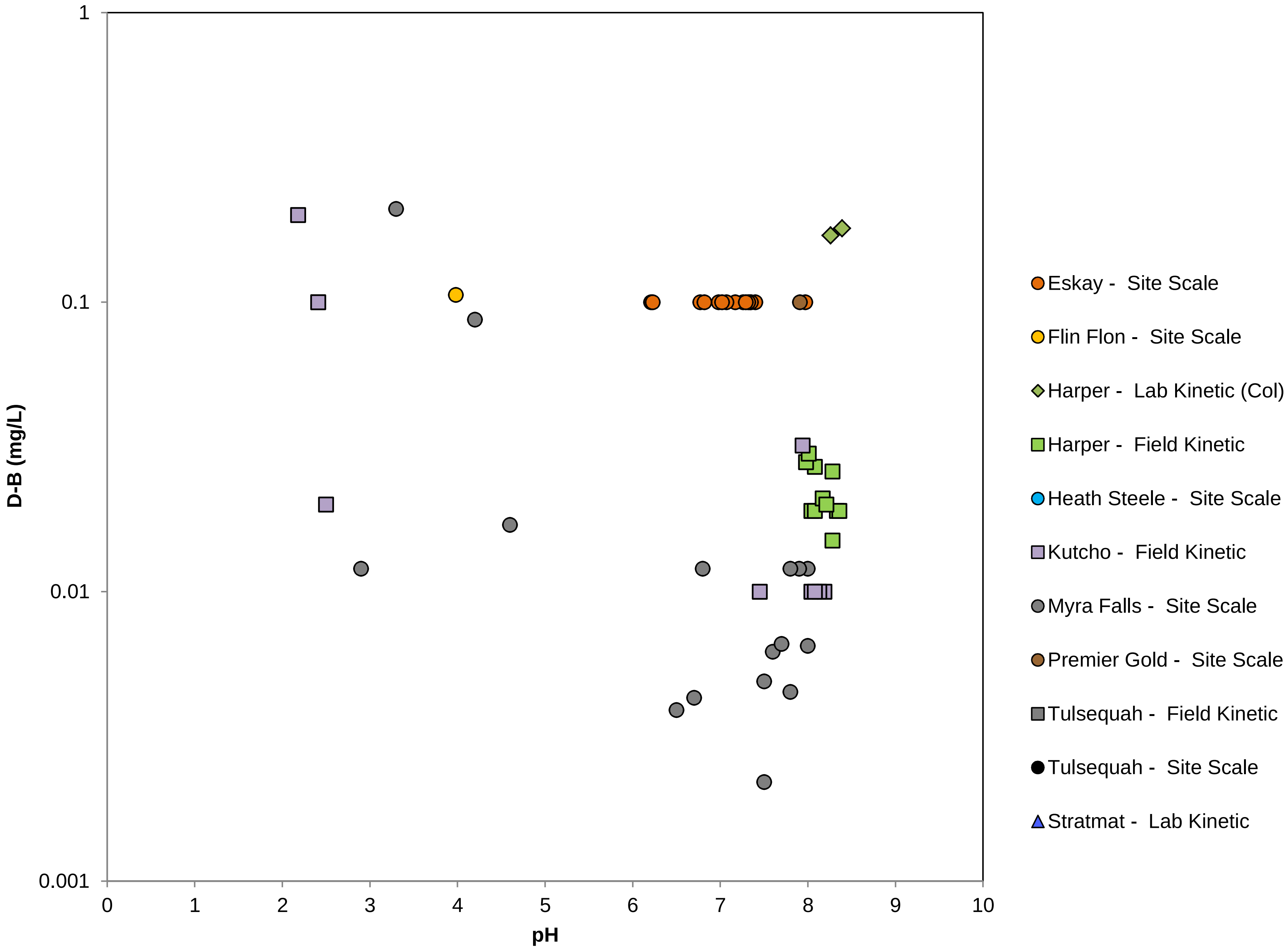

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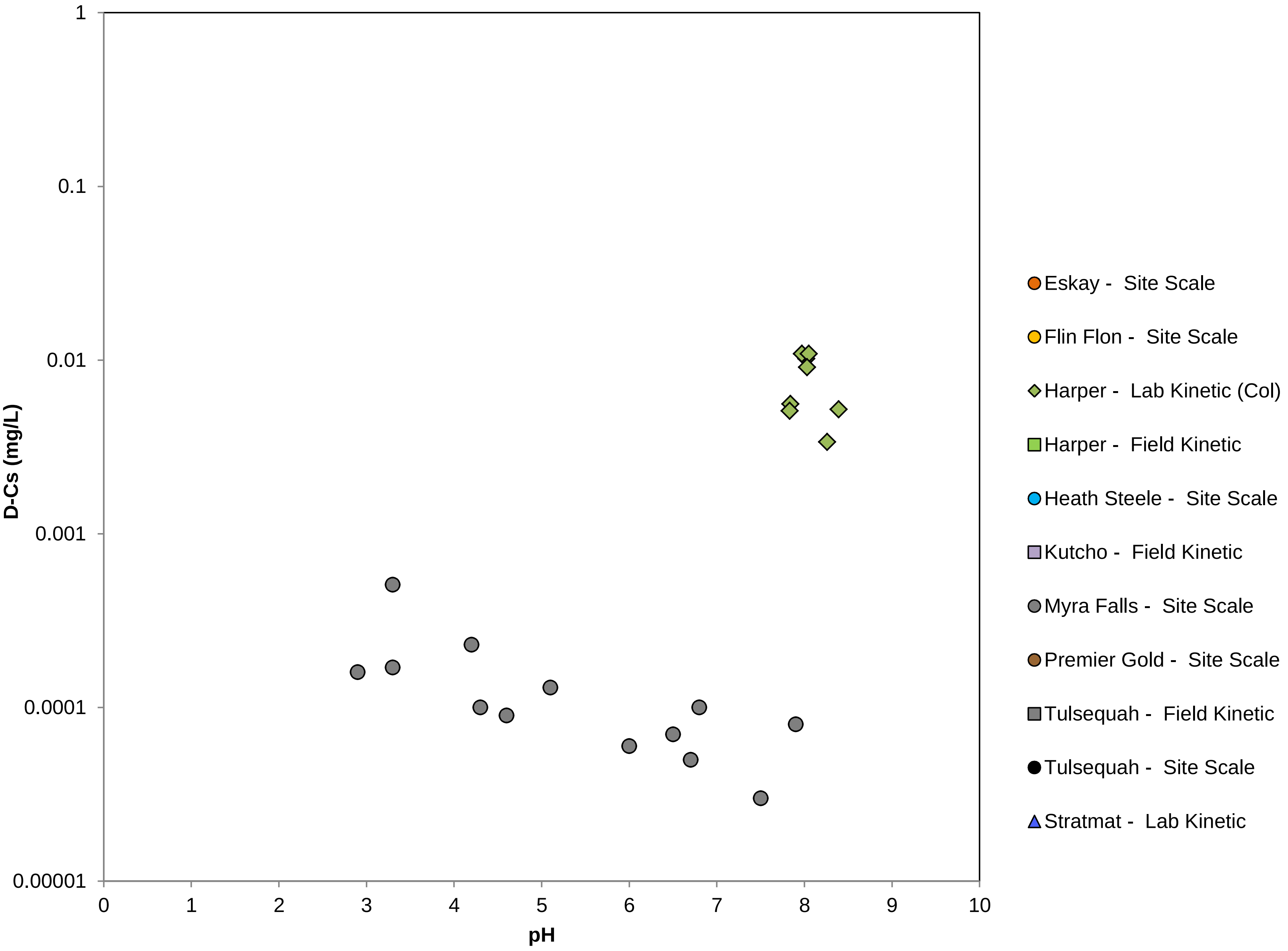

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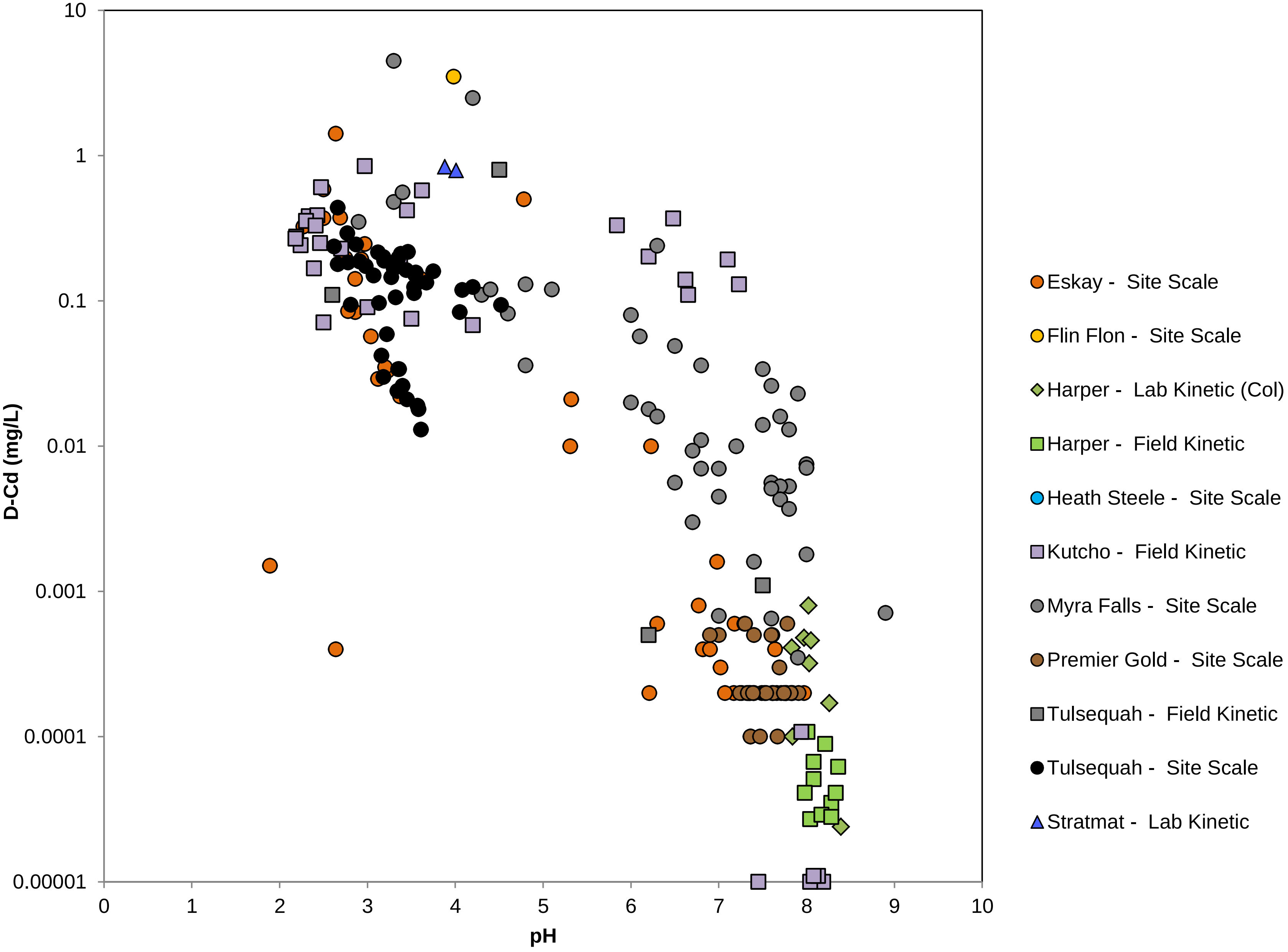

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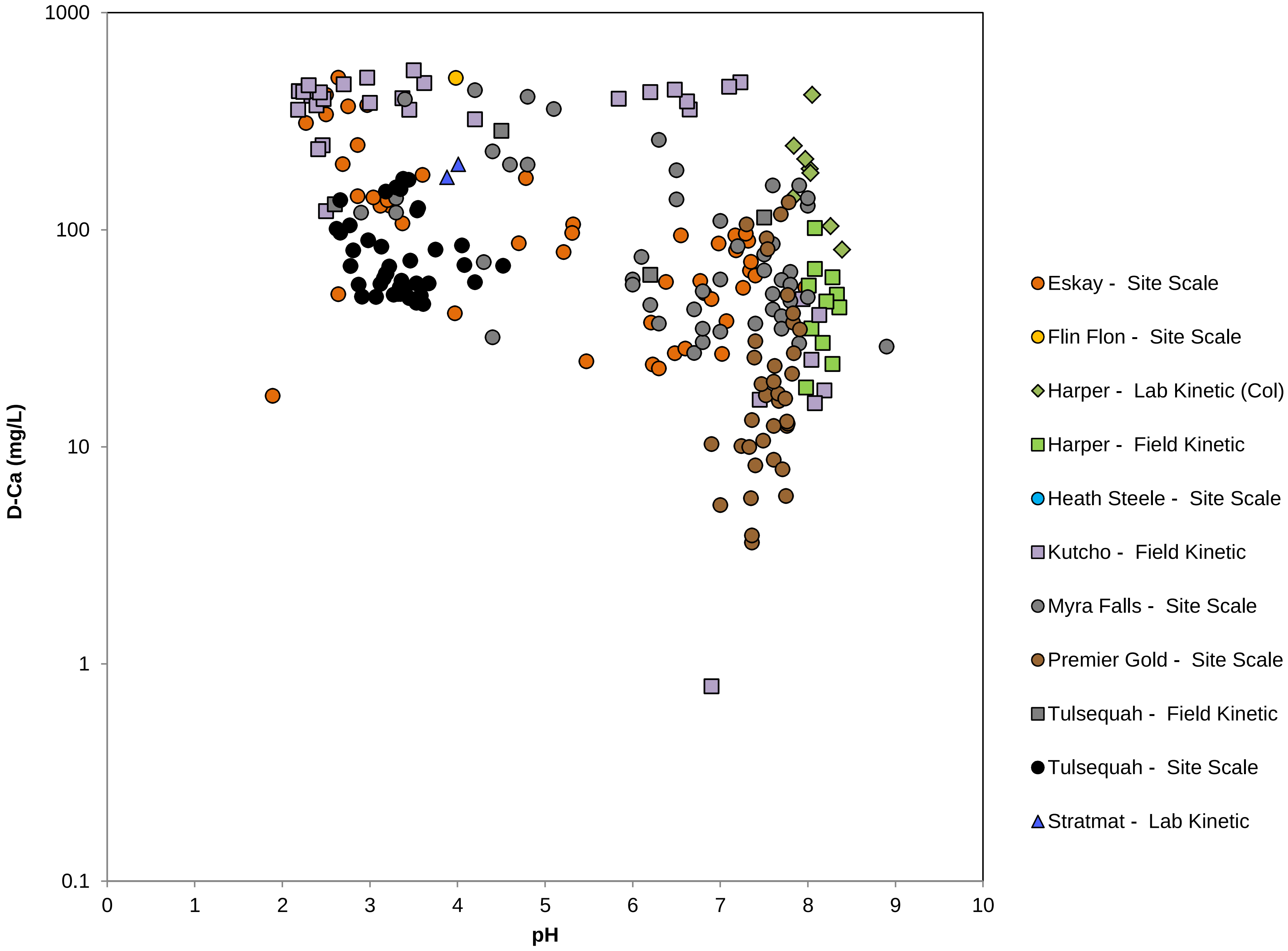

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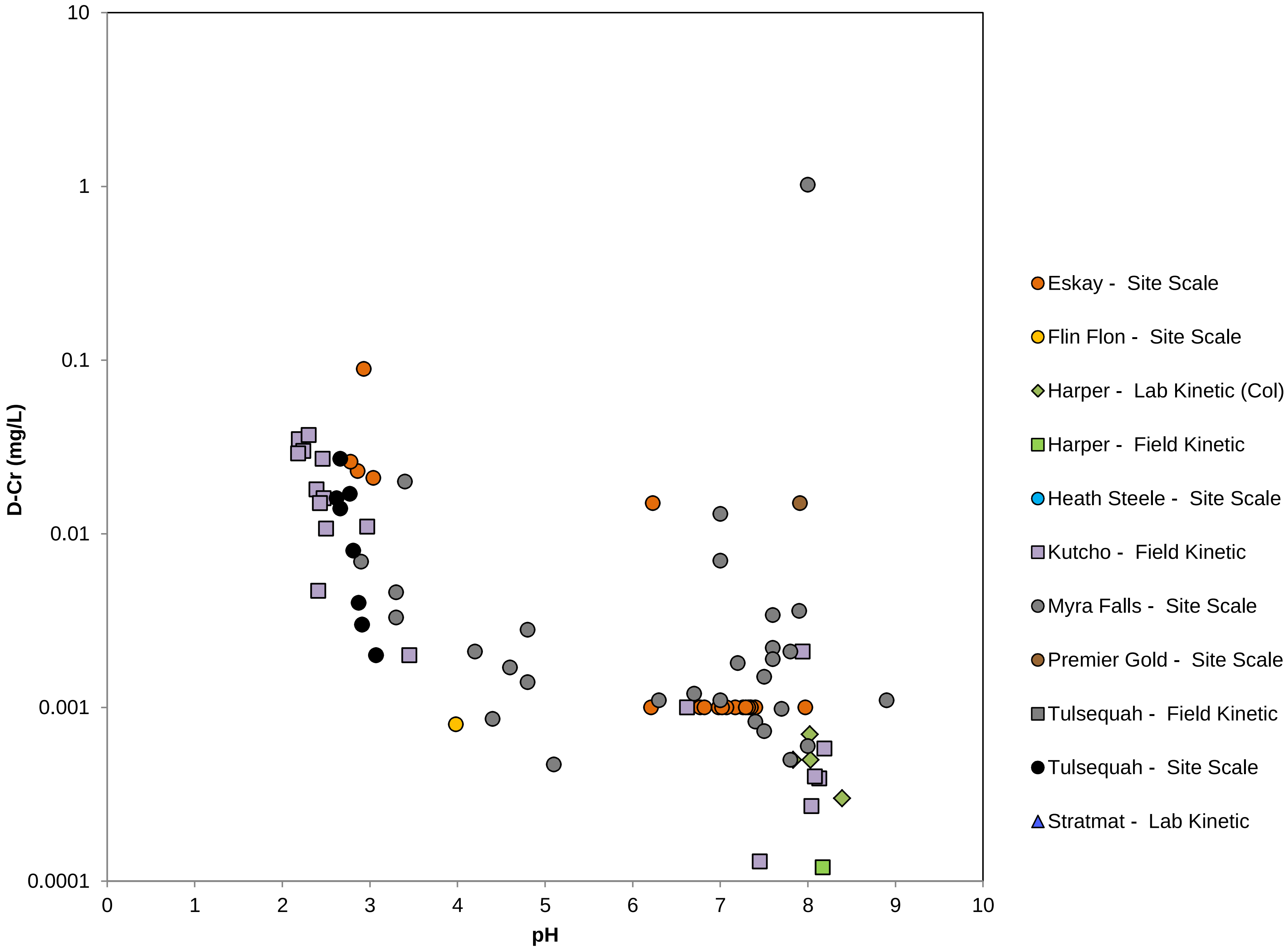

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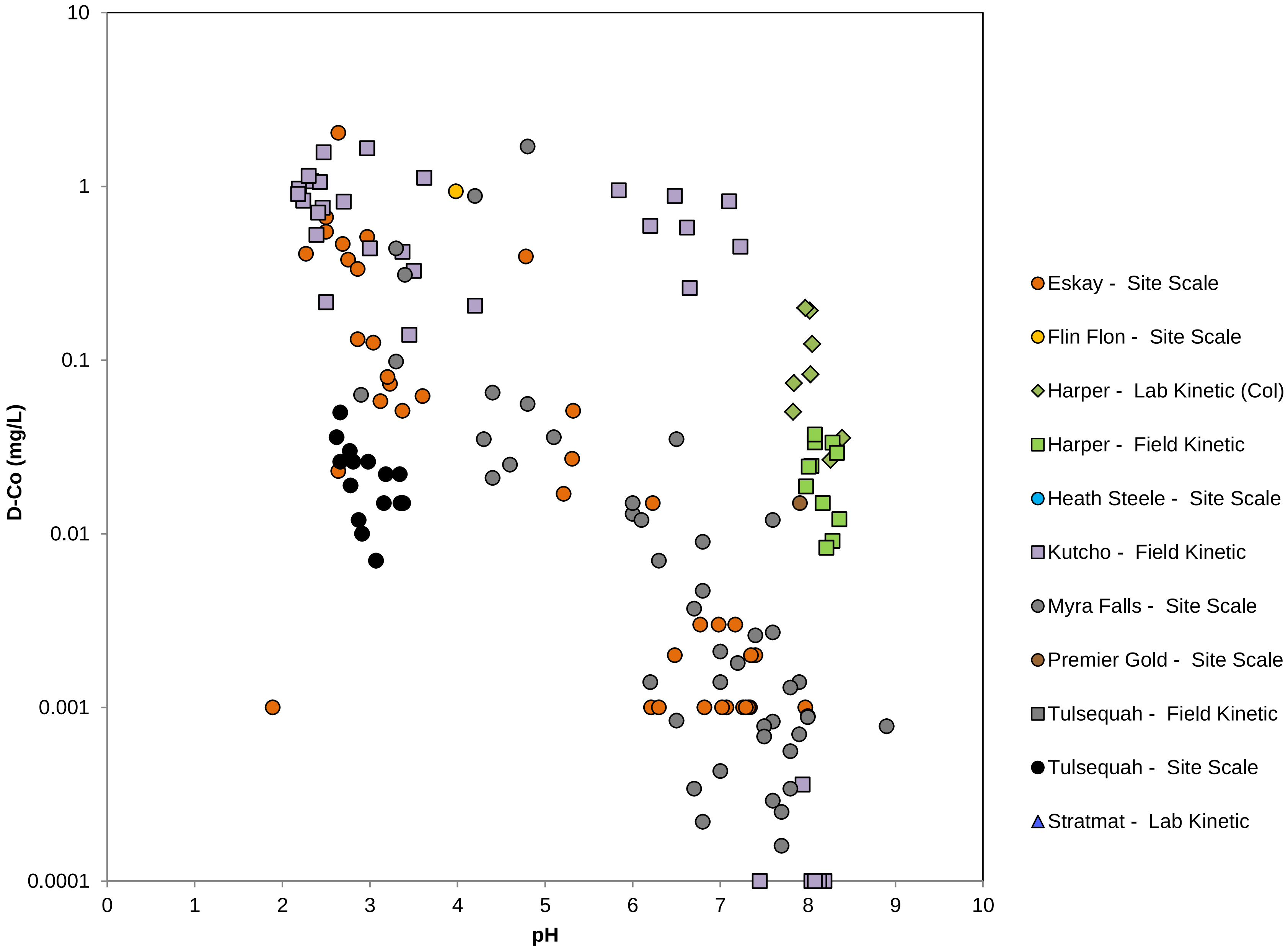

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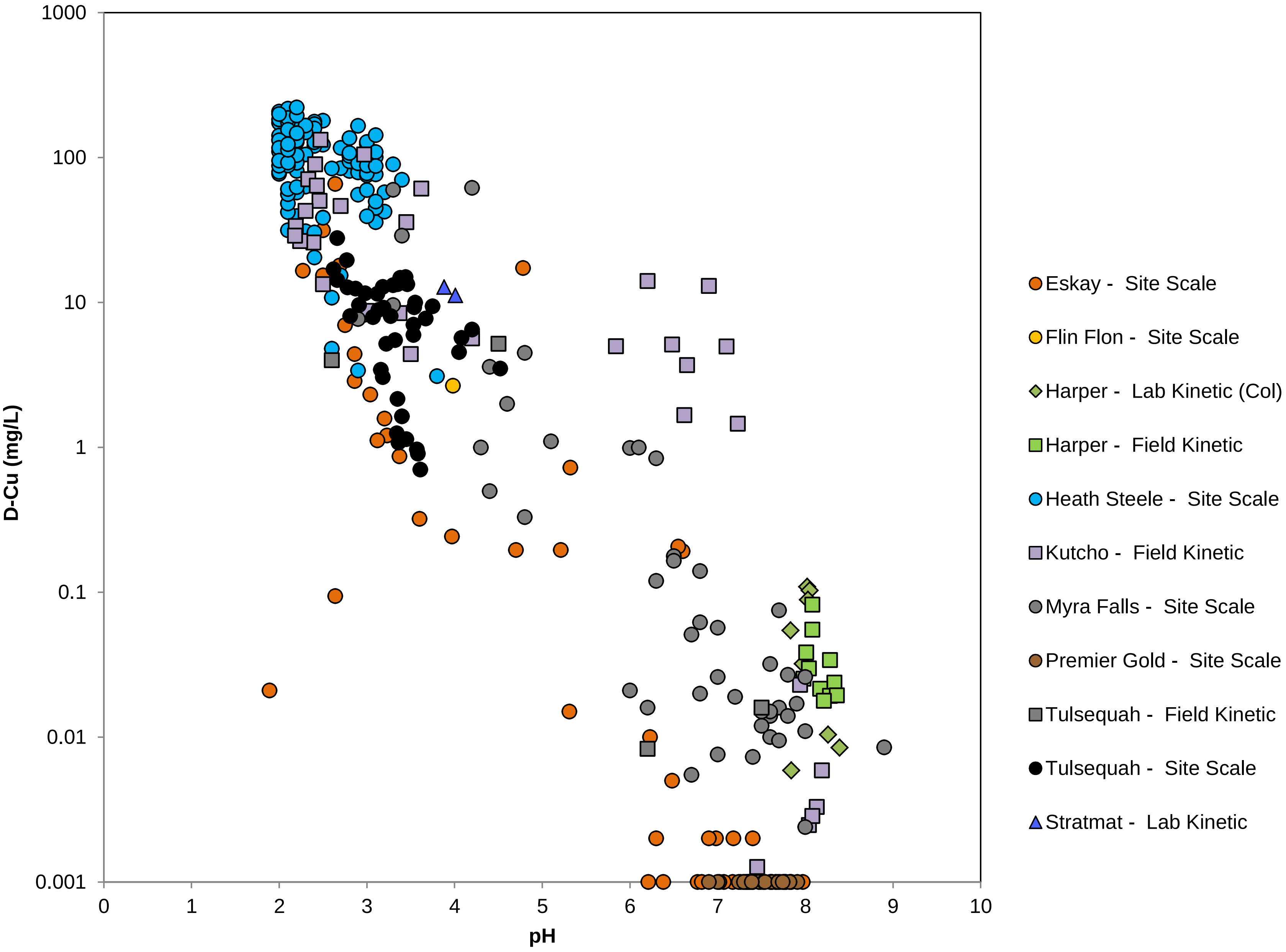

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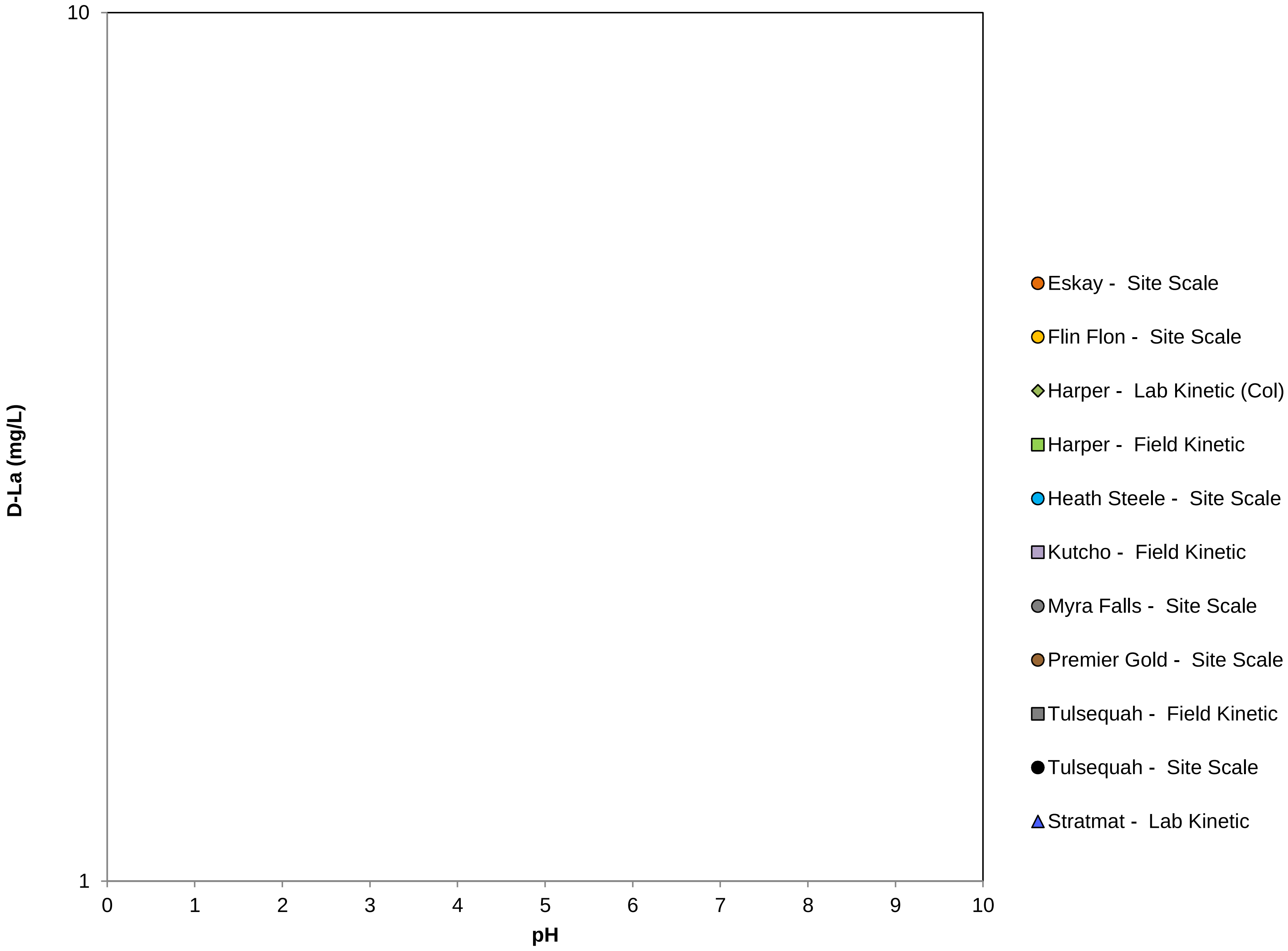

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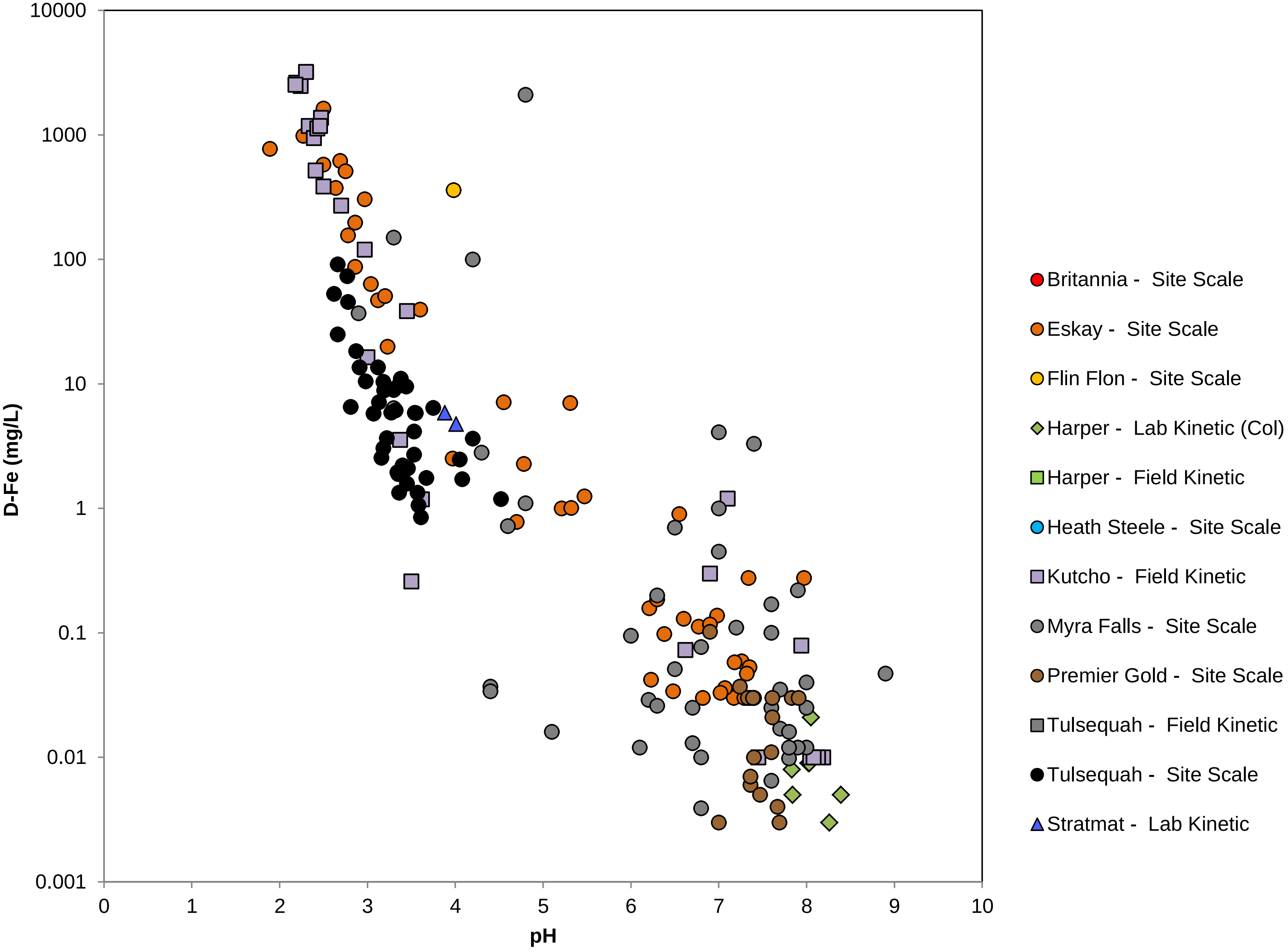

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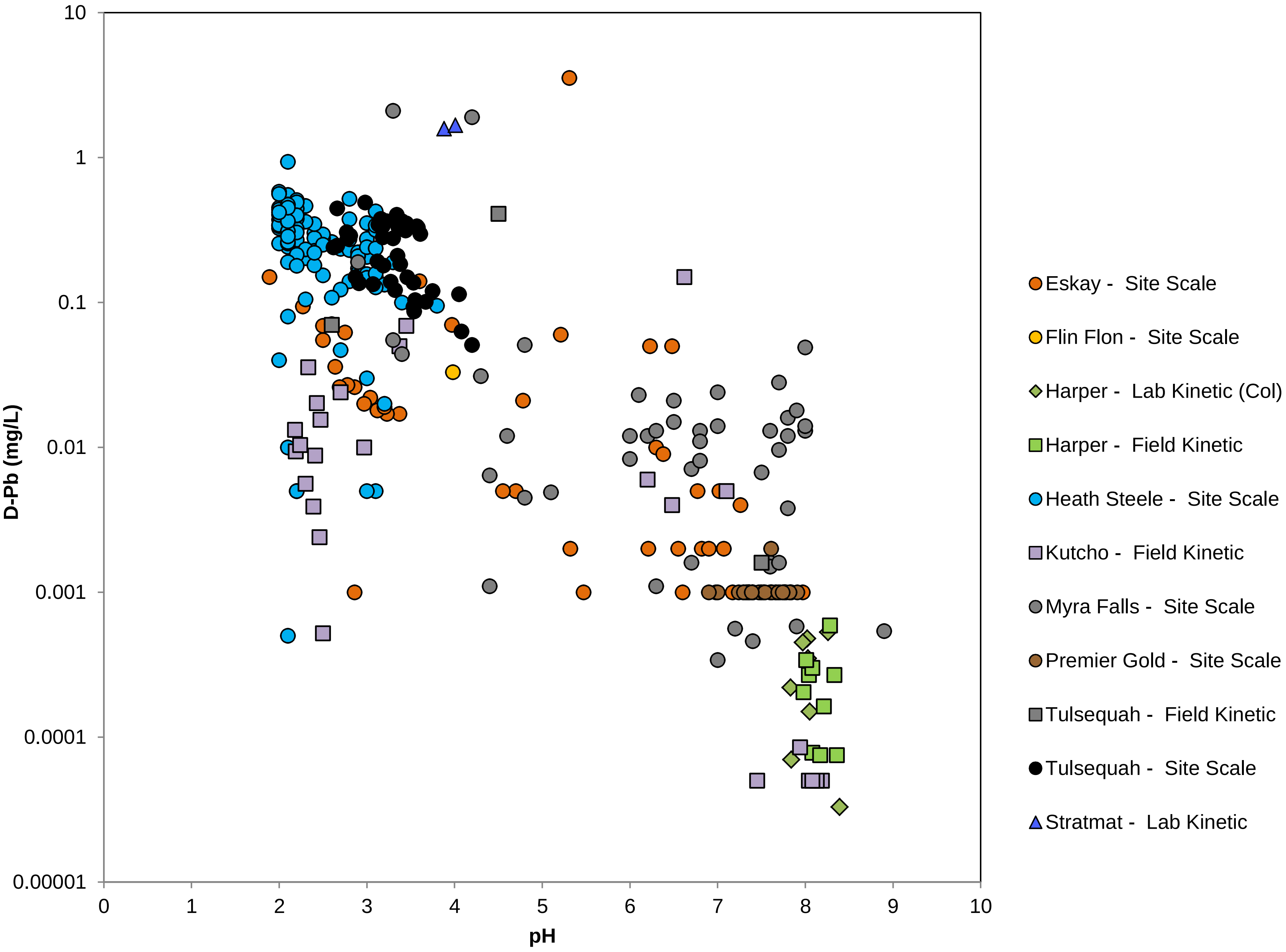

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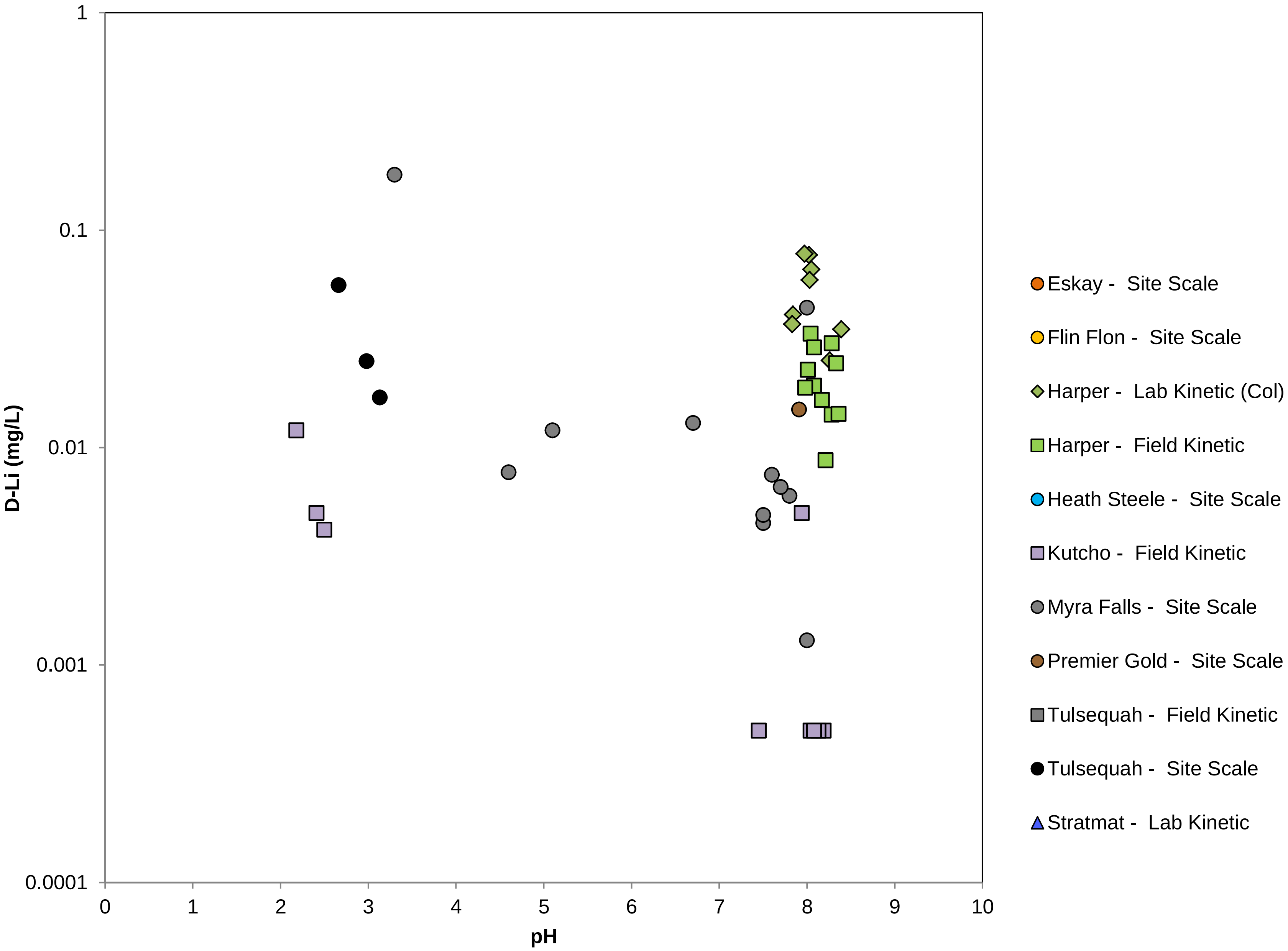

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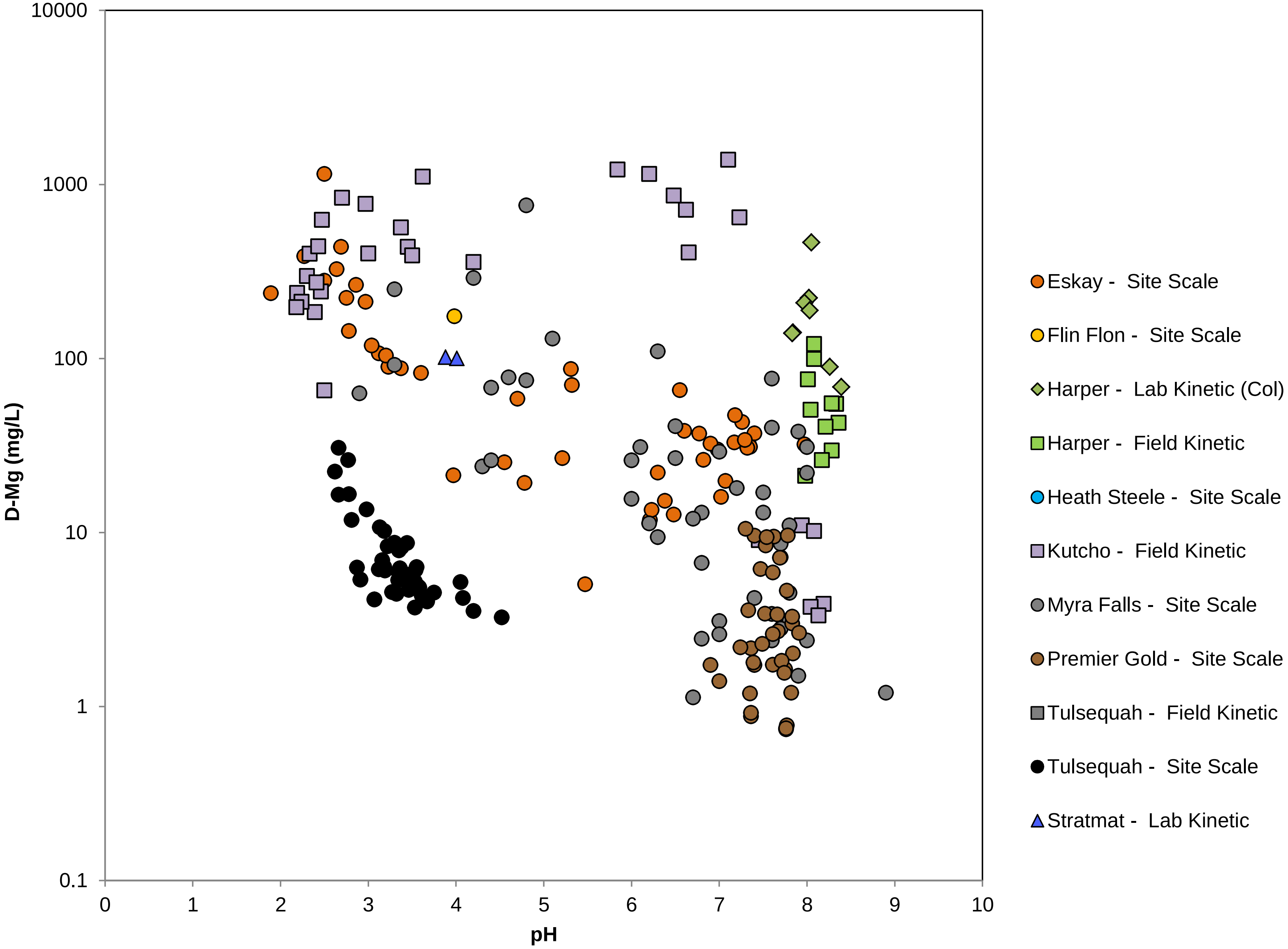

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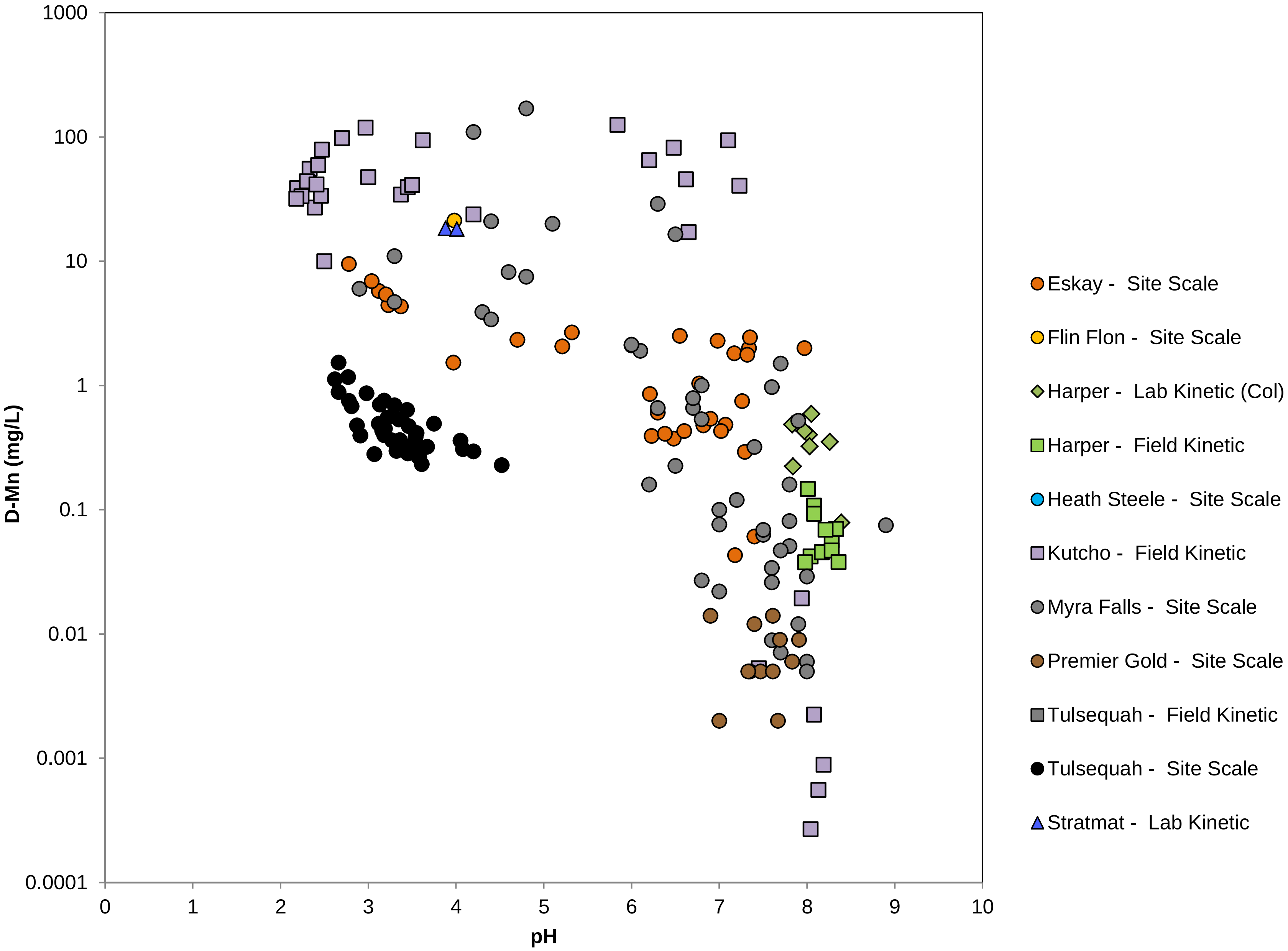

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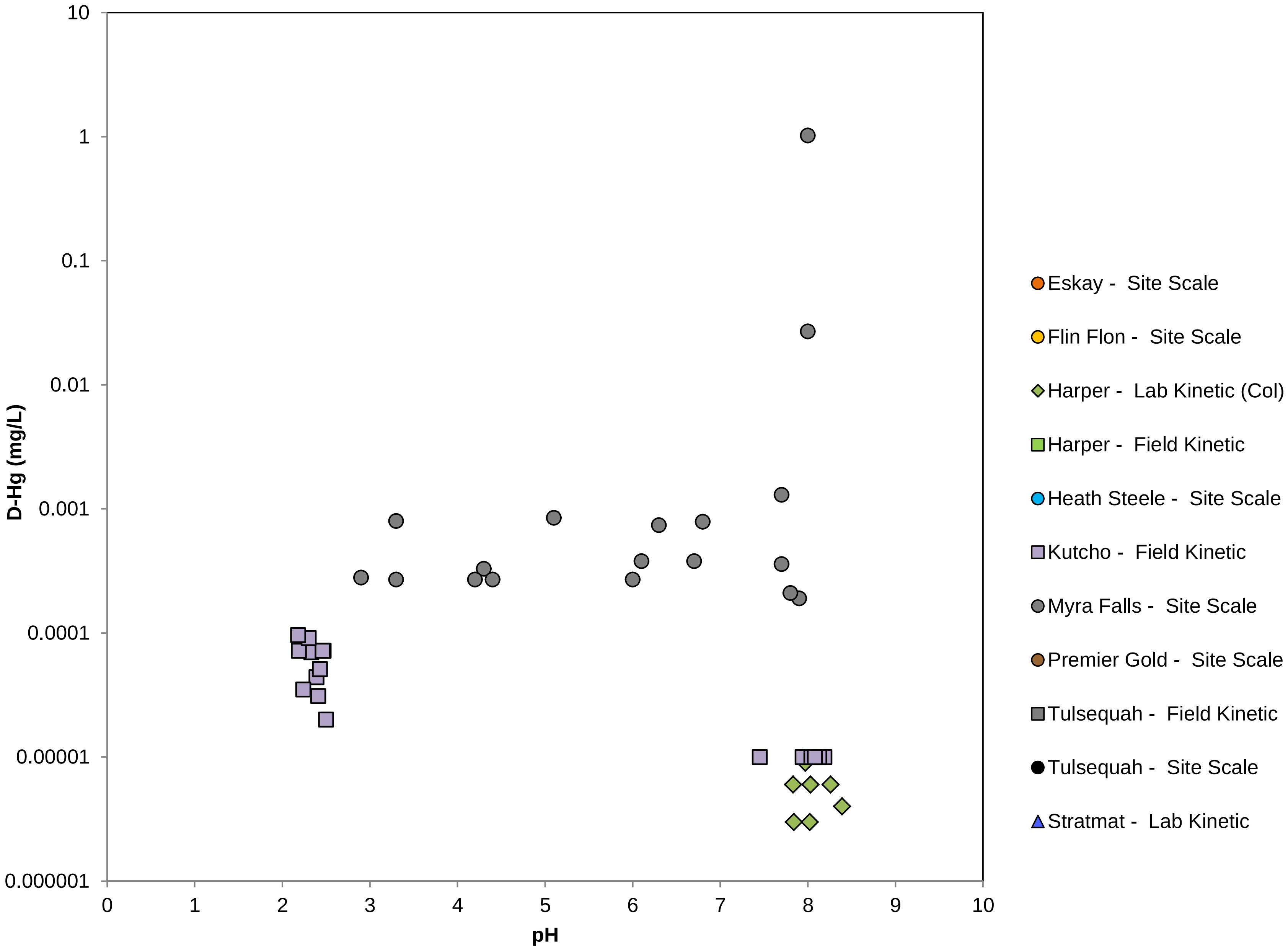

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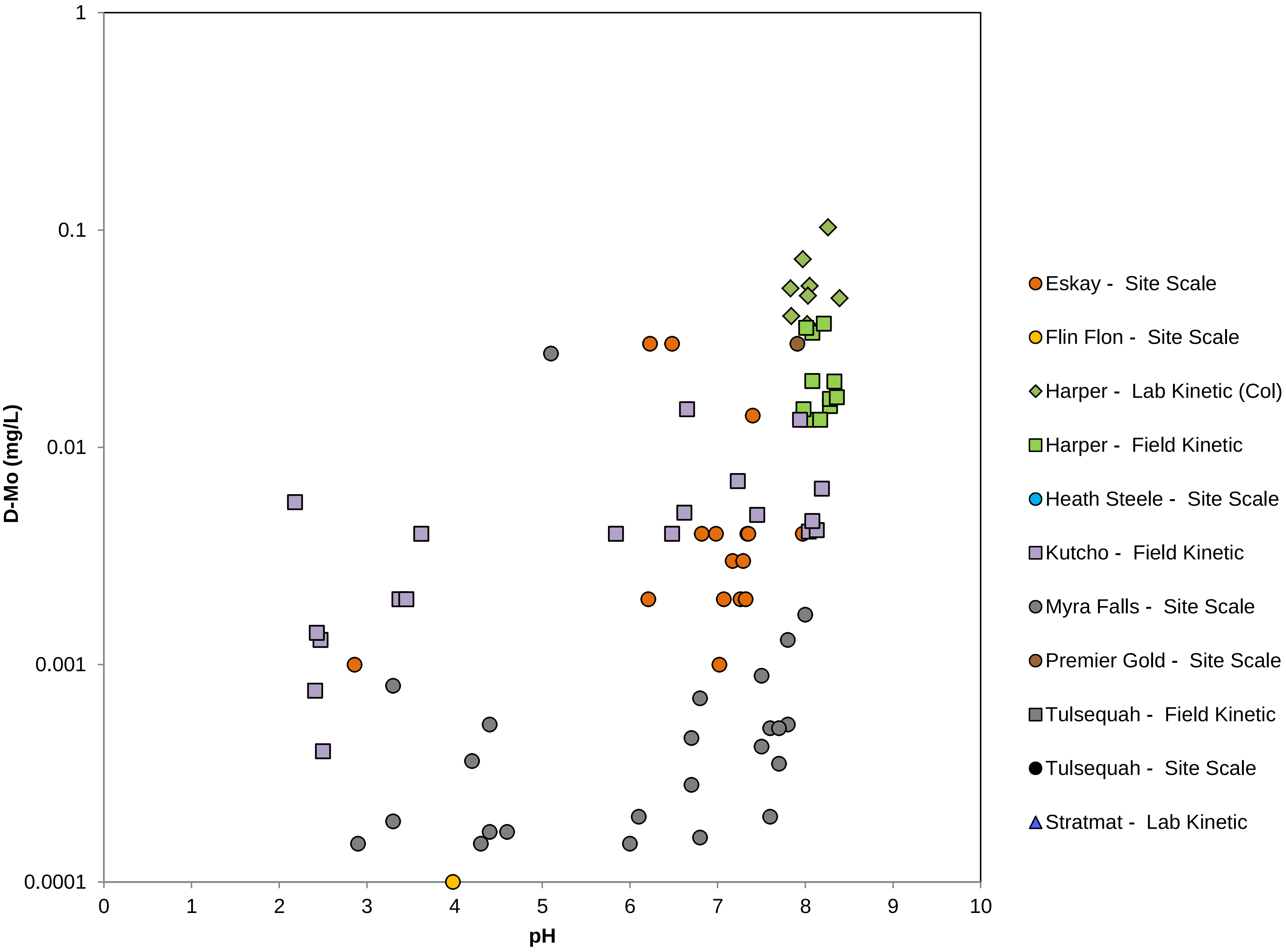

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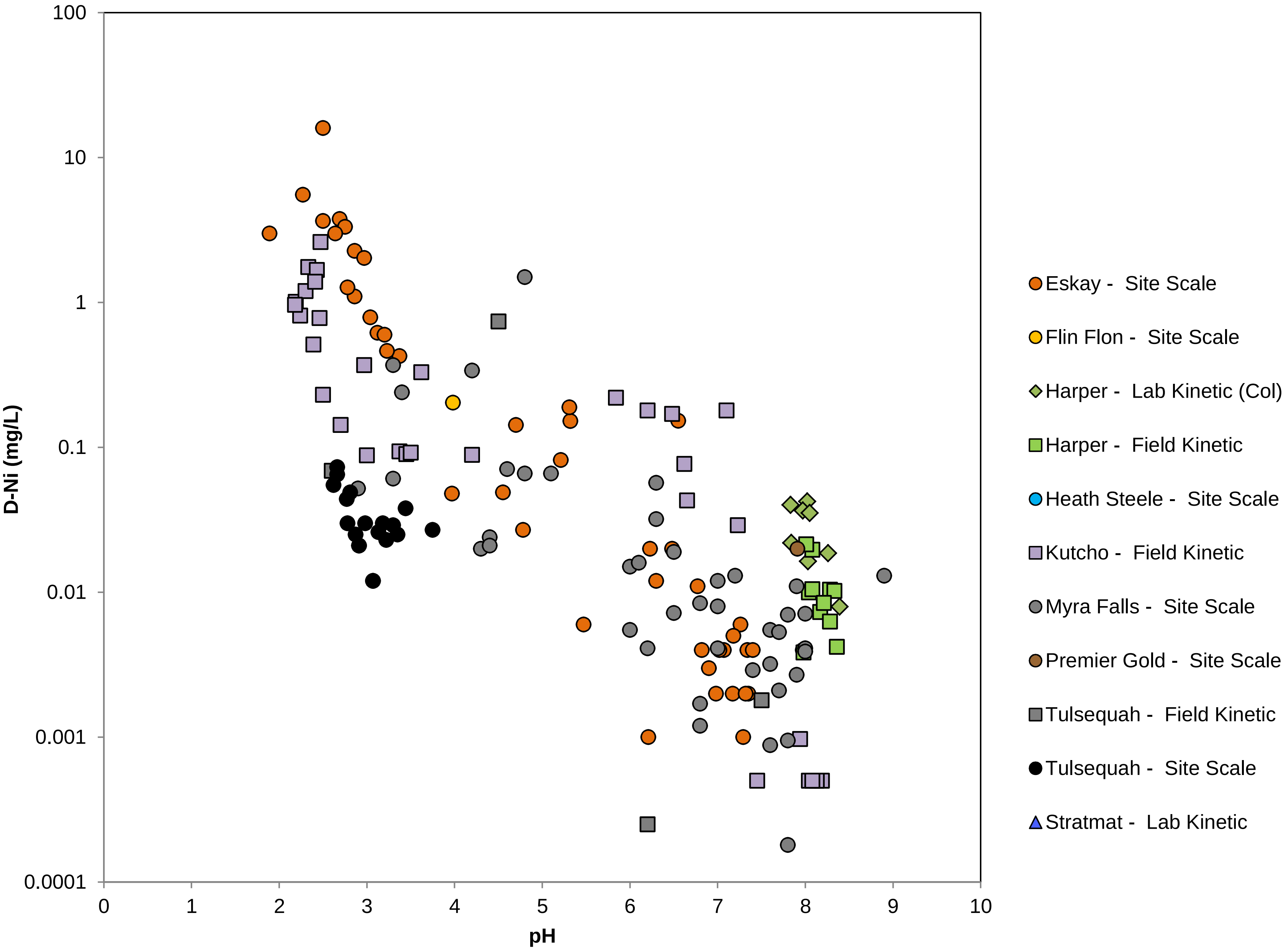

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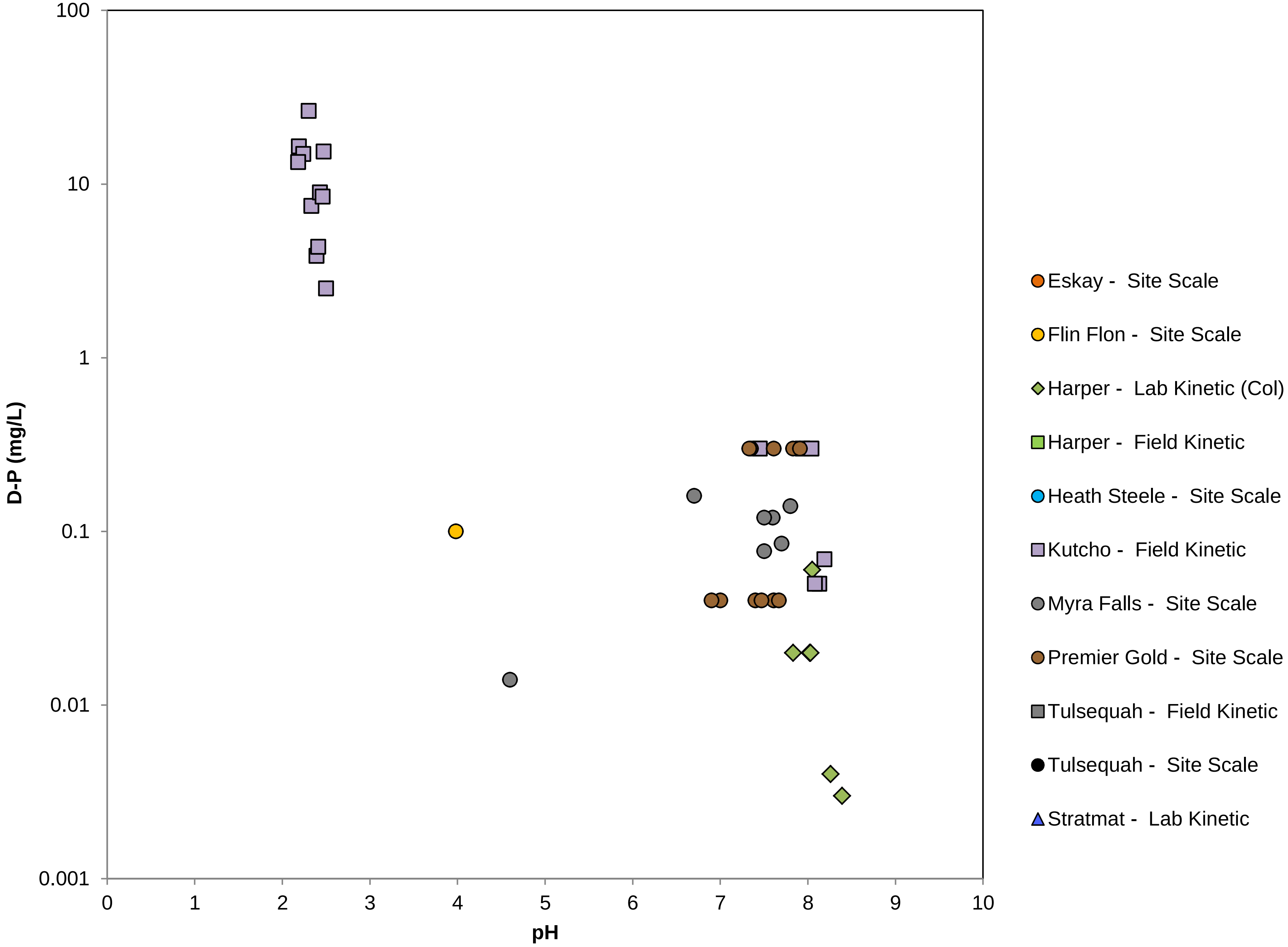
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Flin Flon - Site Scale
Harper - Lab Kinetic (Col)
Harper - Field Kinetic
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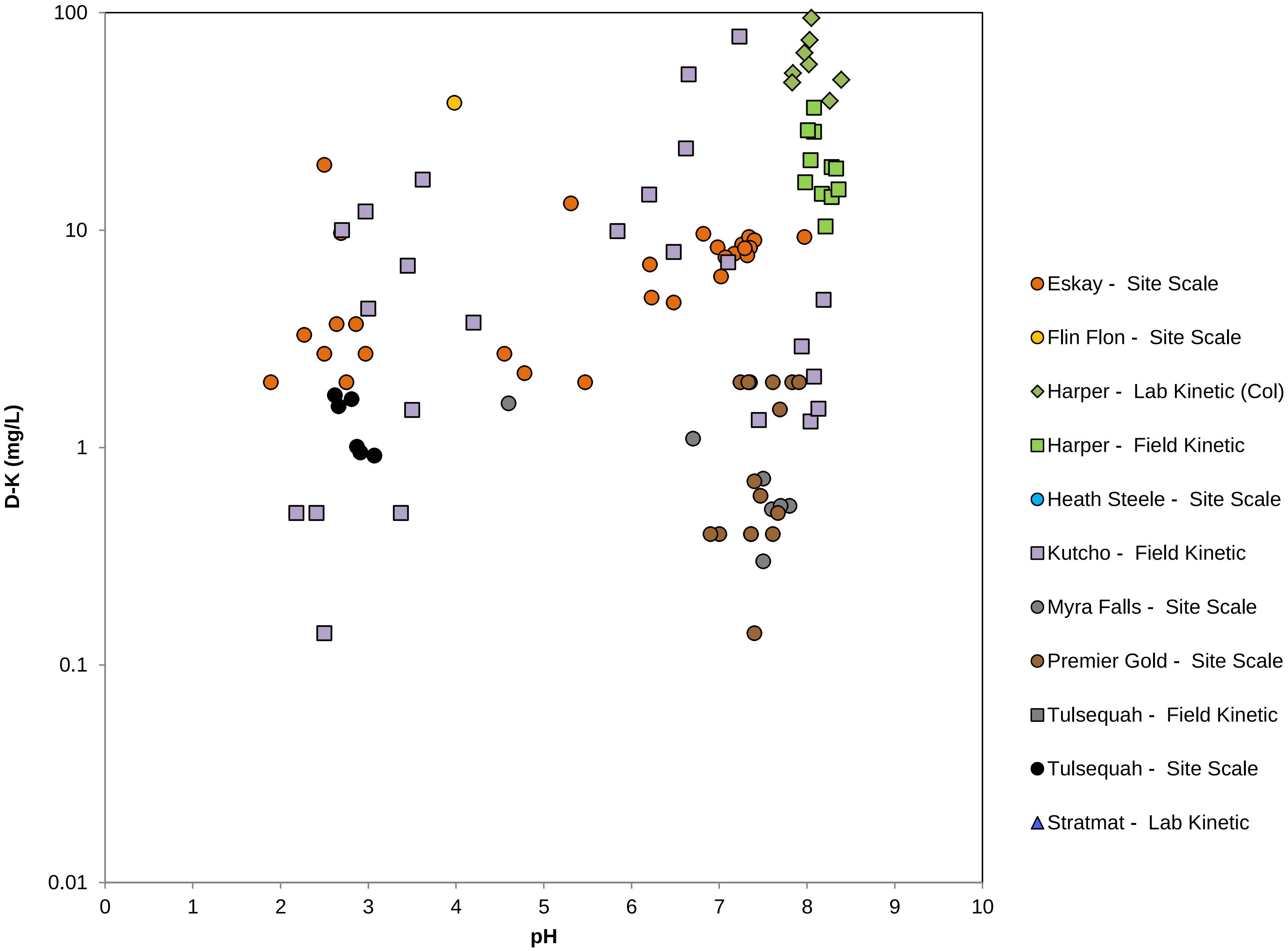

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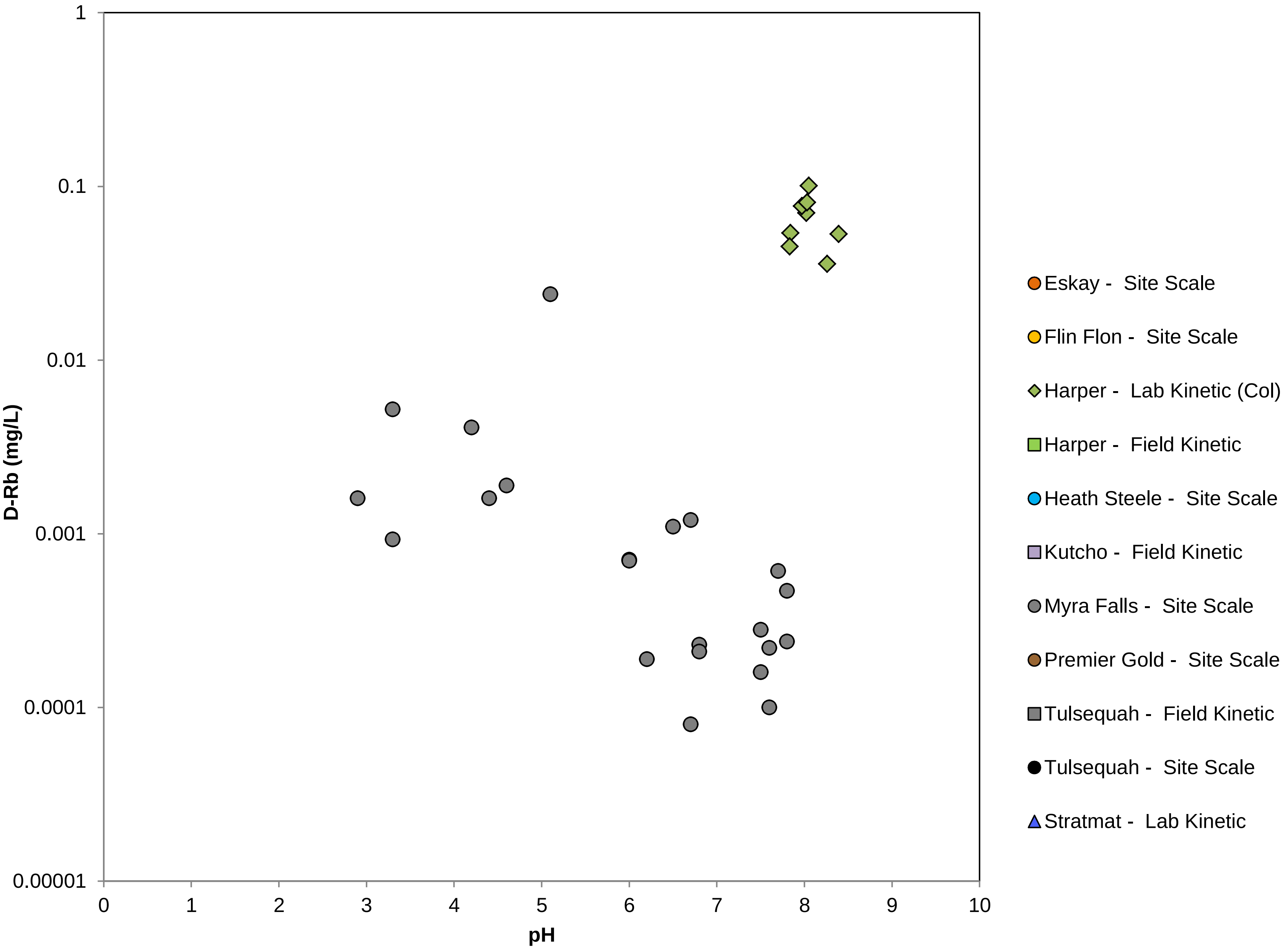

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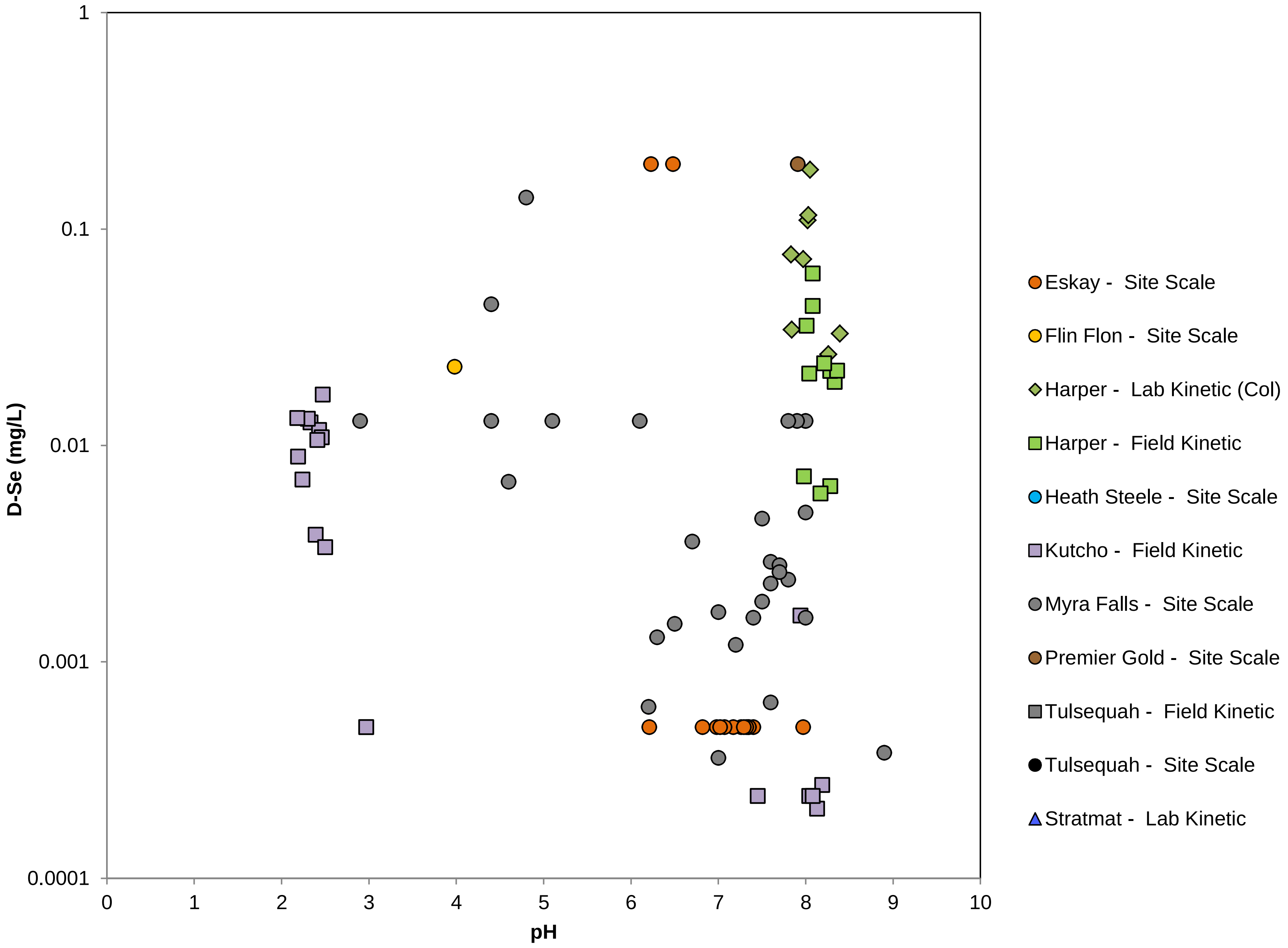

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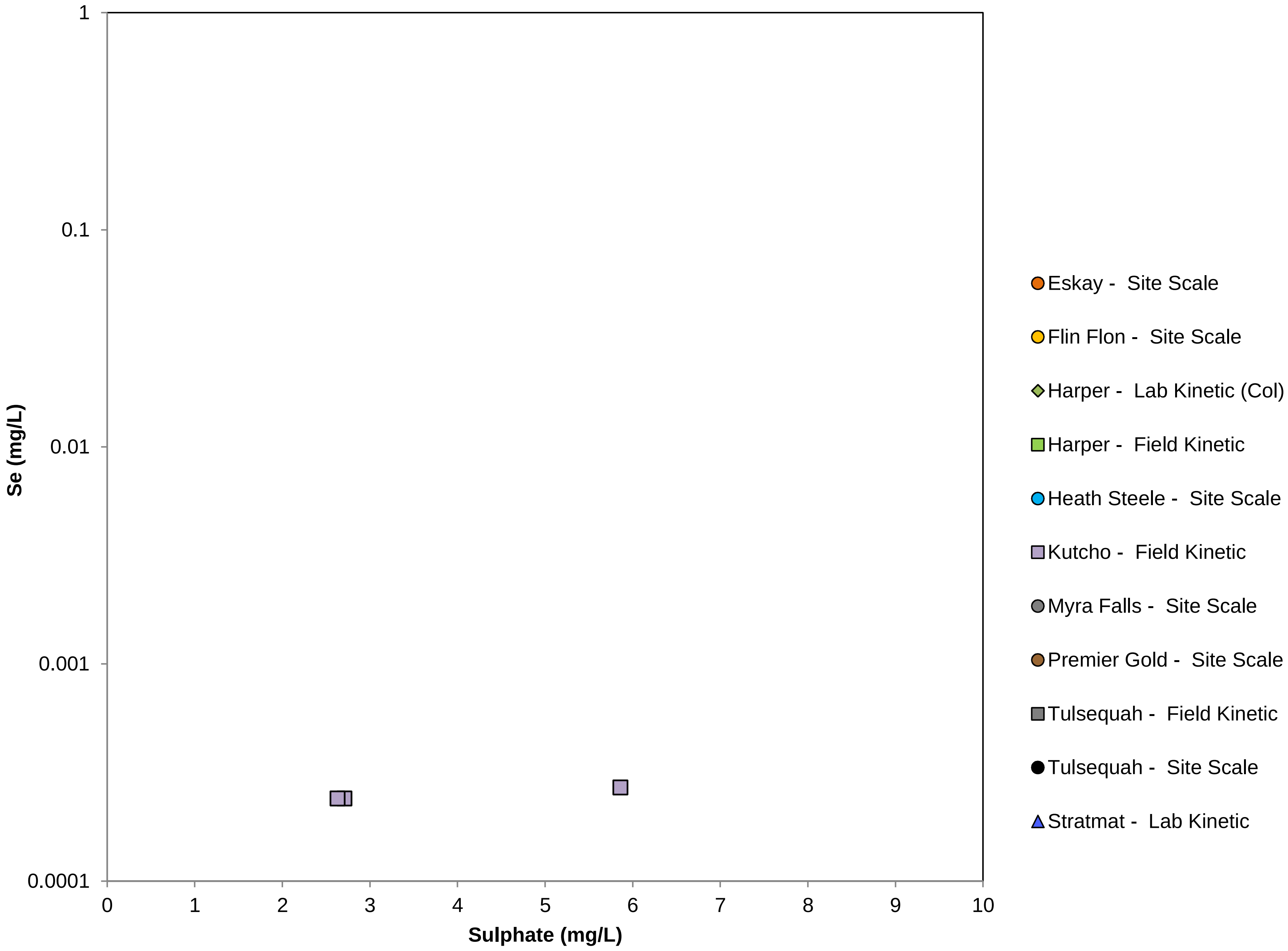

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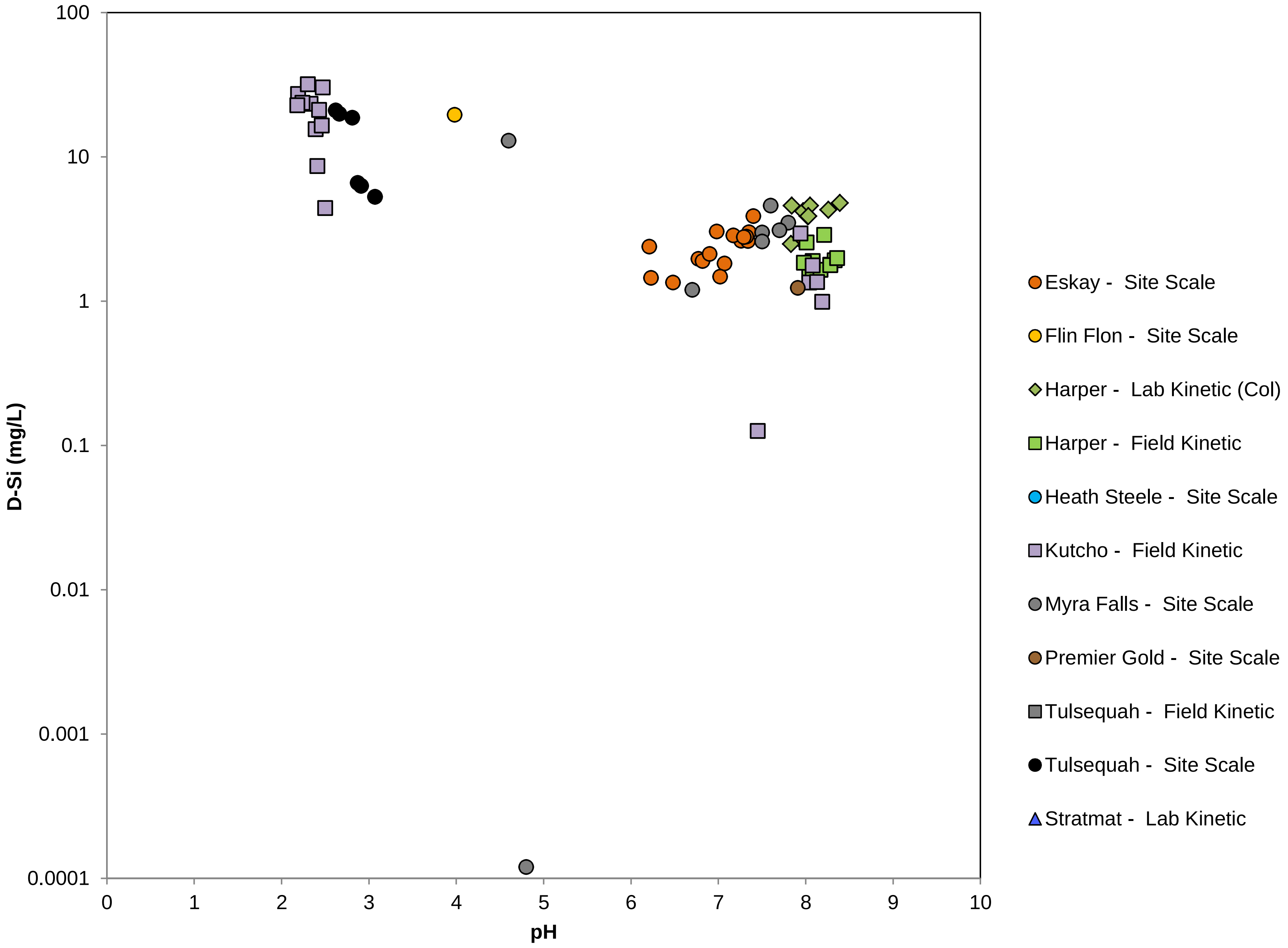

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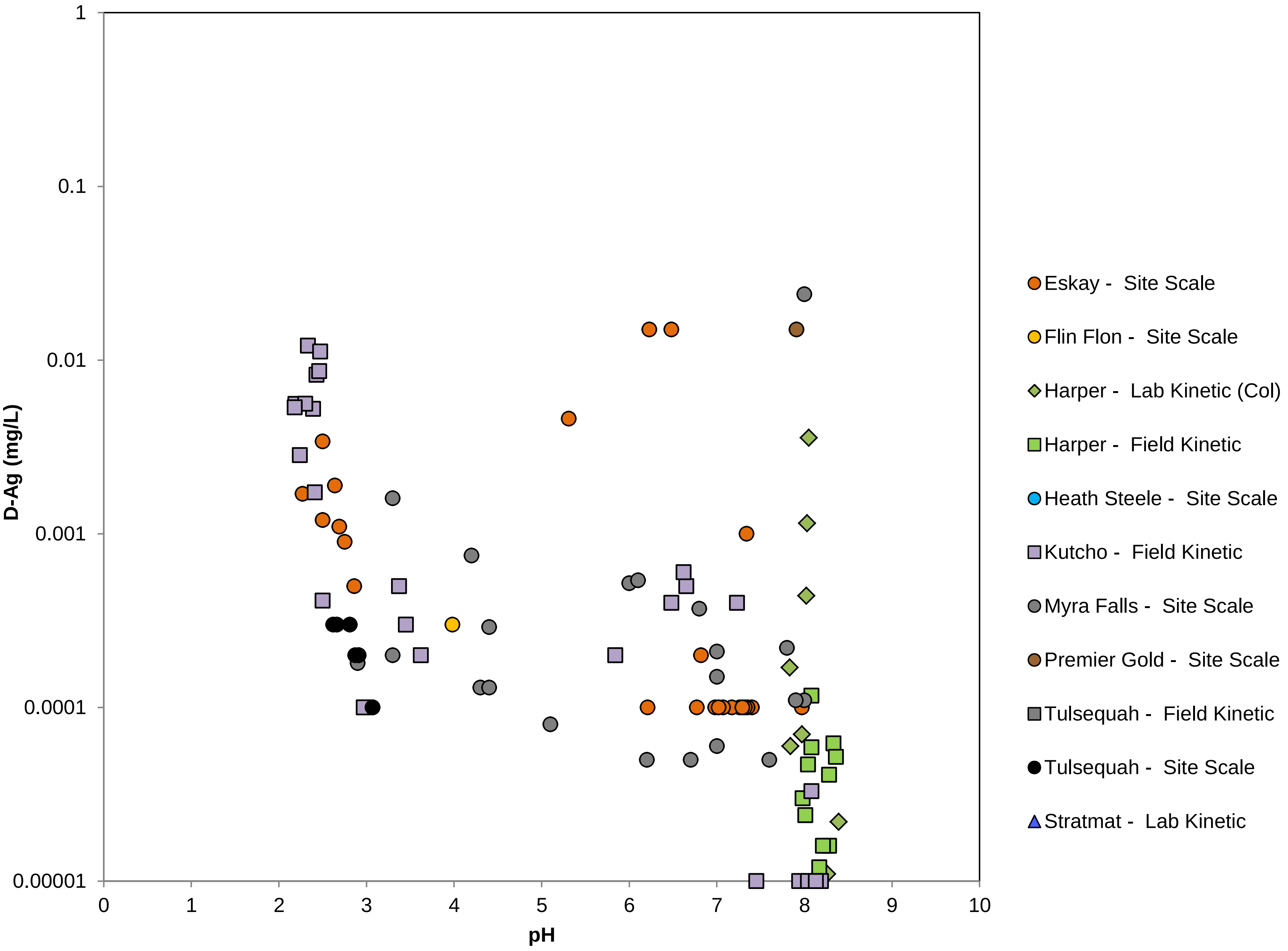

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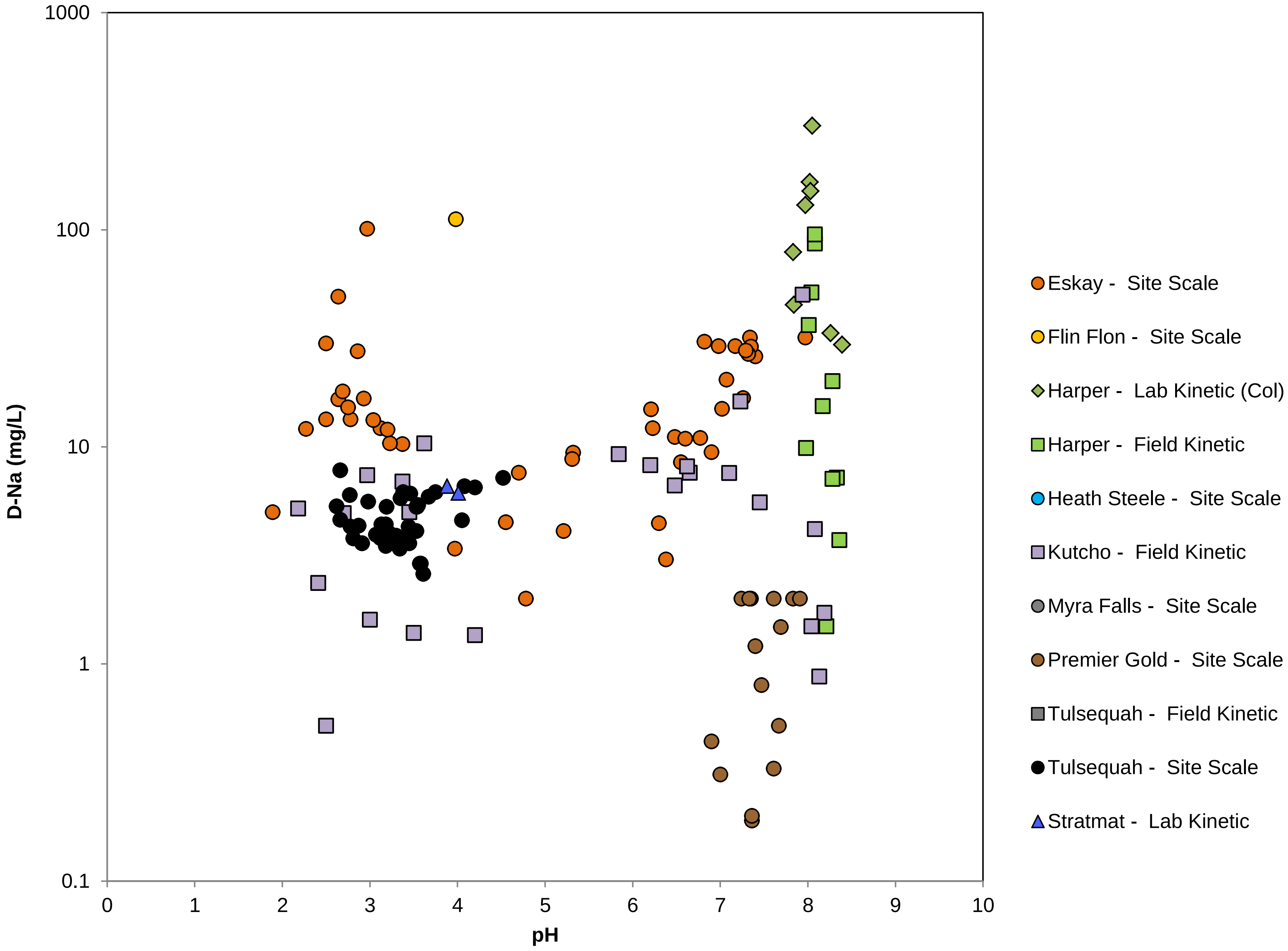

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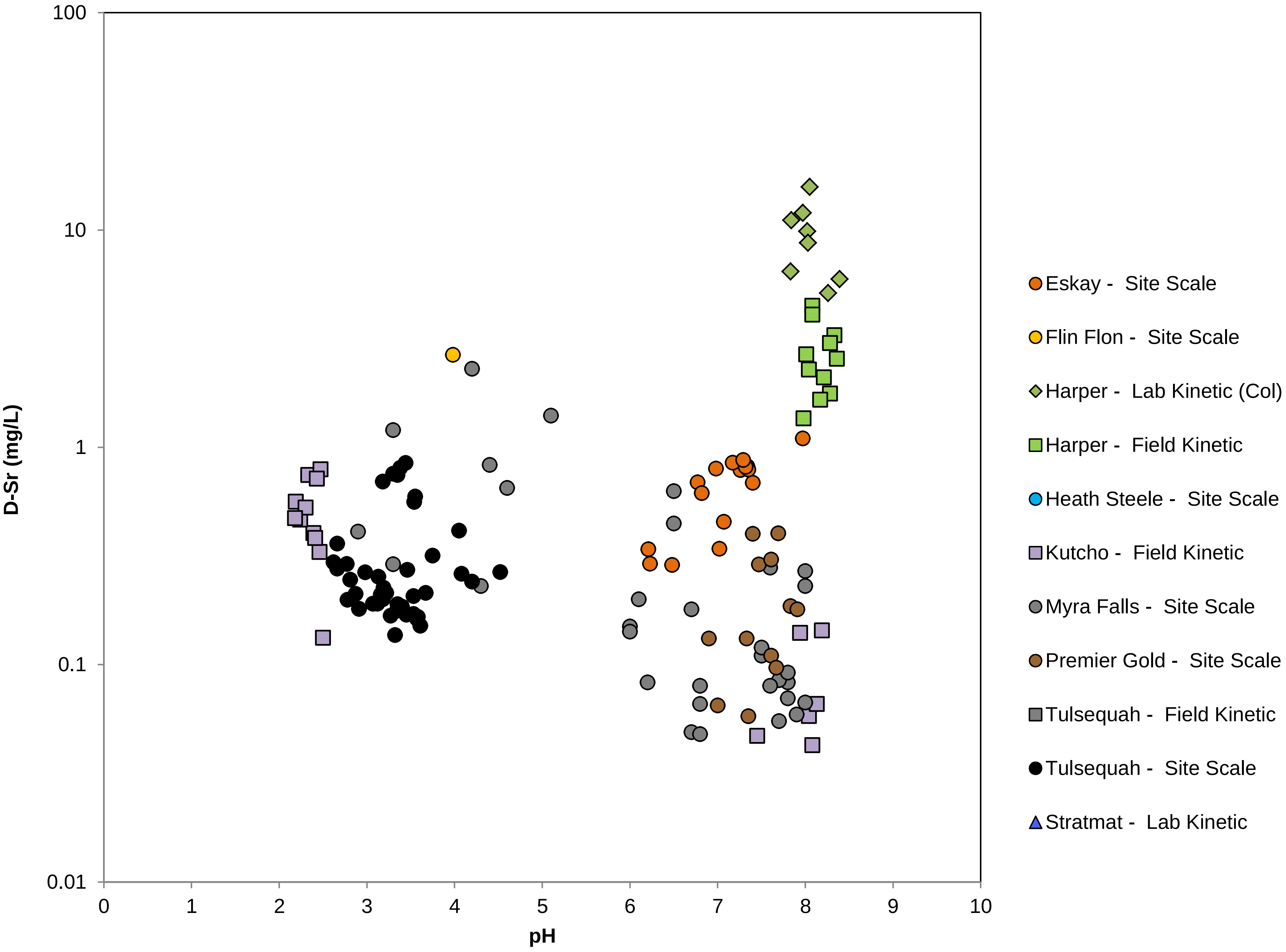

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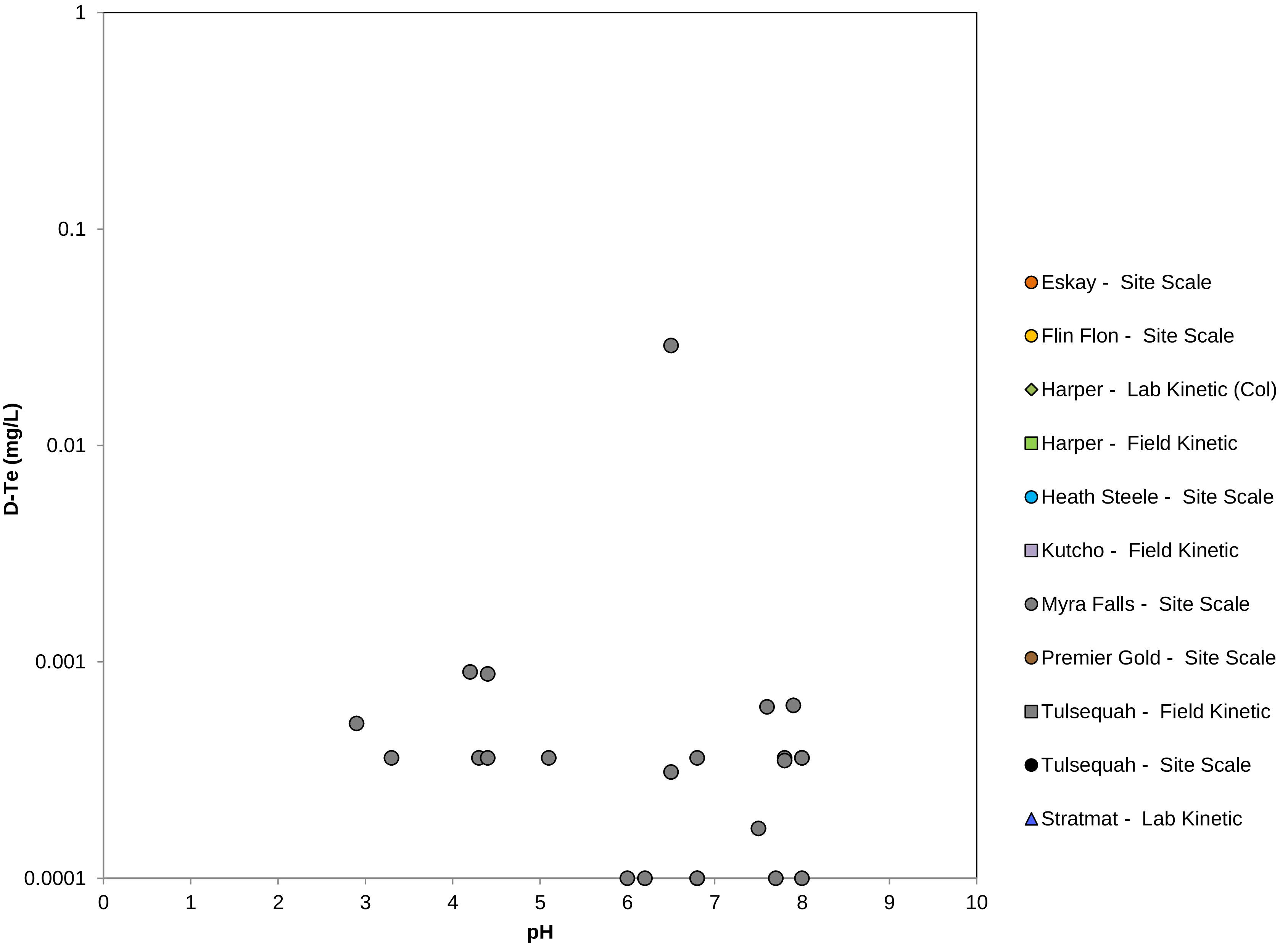

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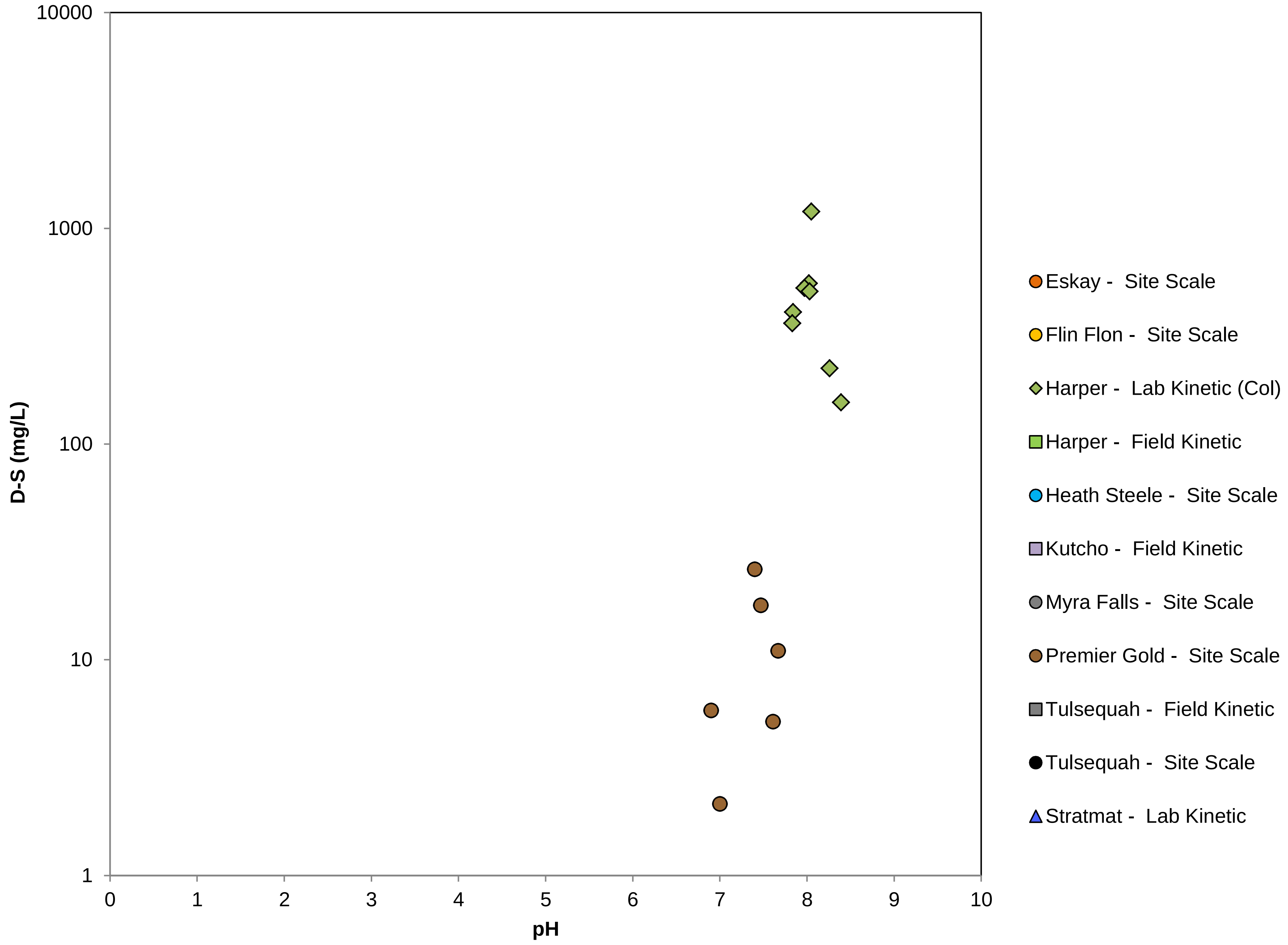

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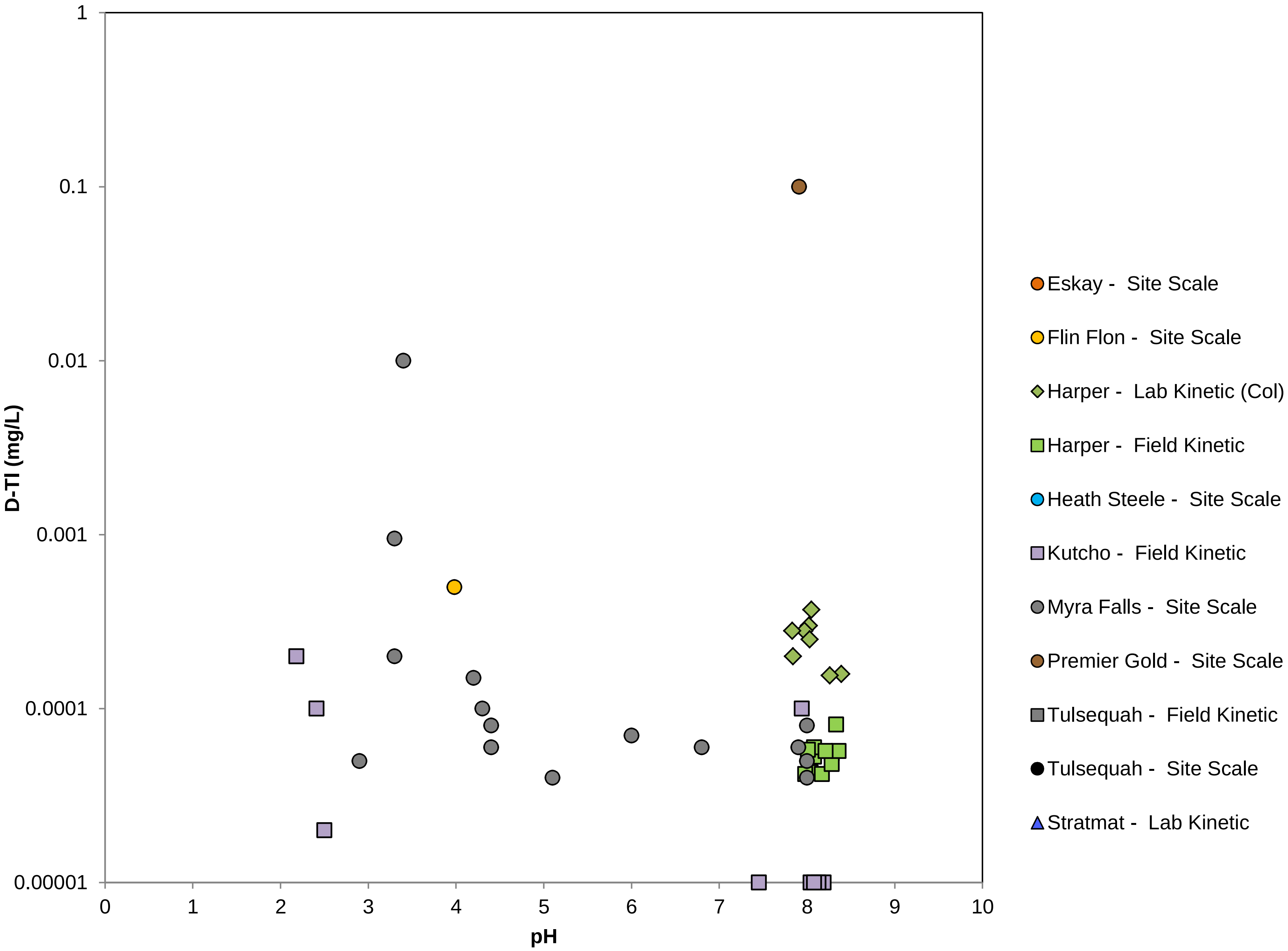

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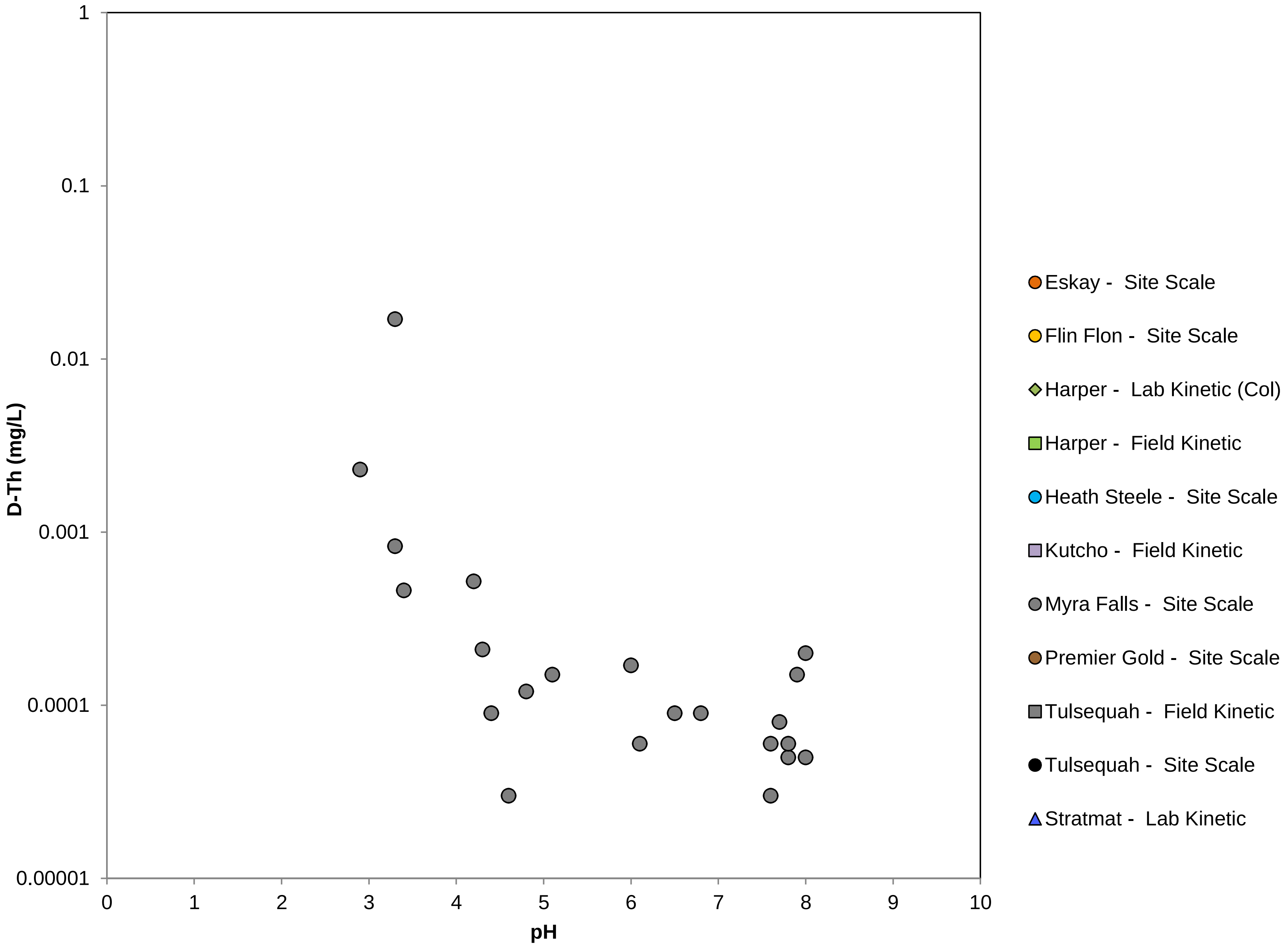

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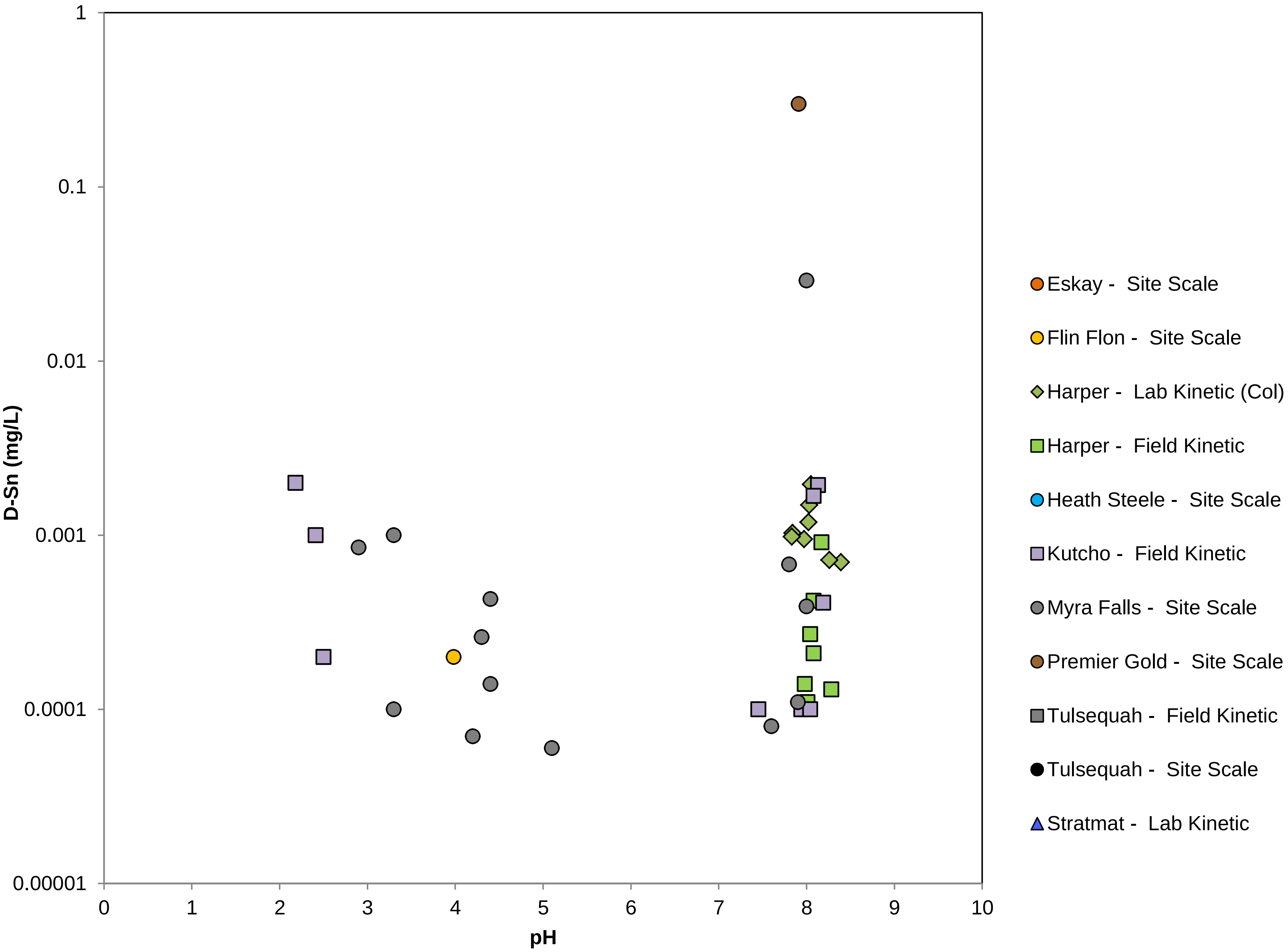

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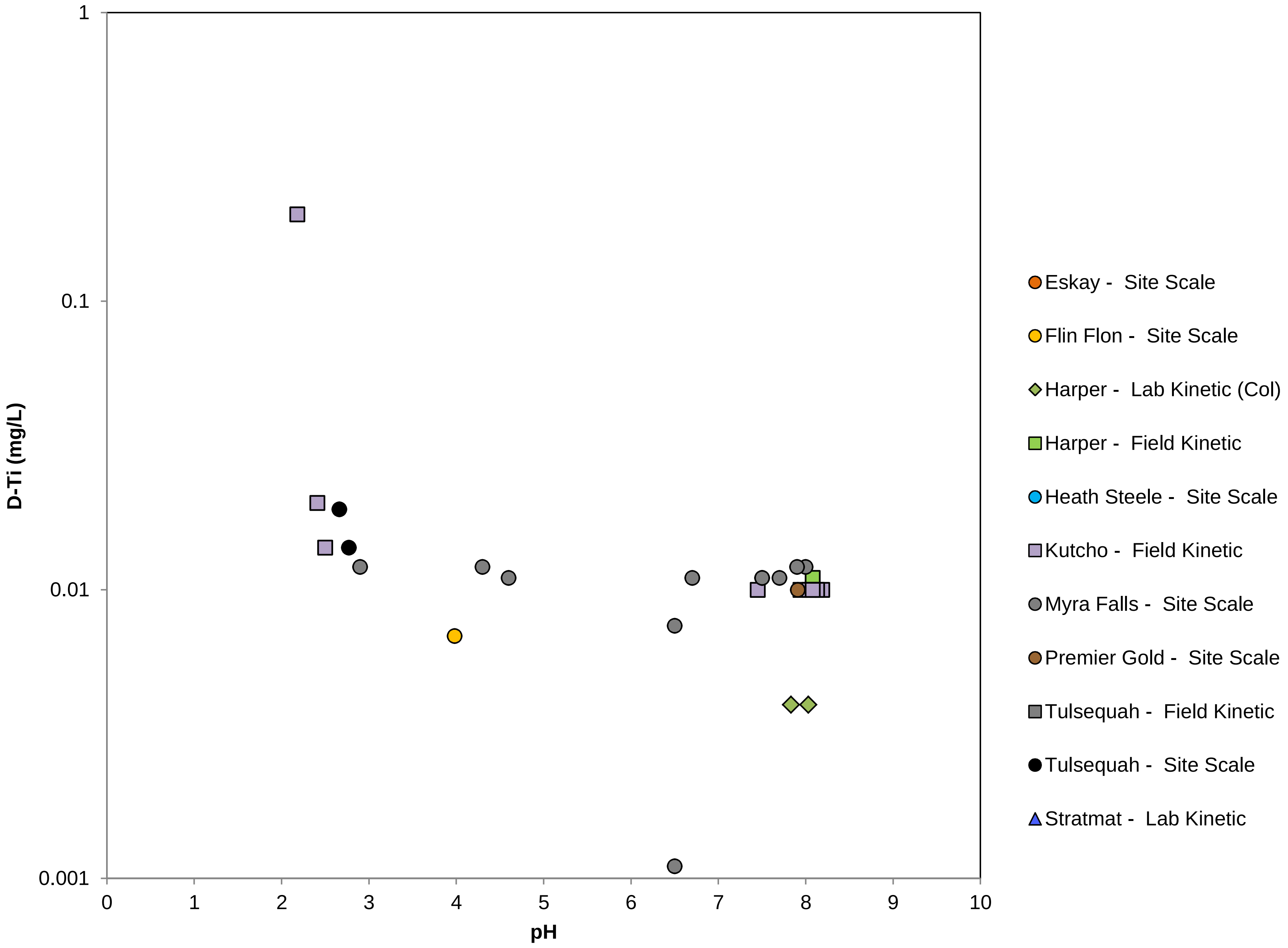

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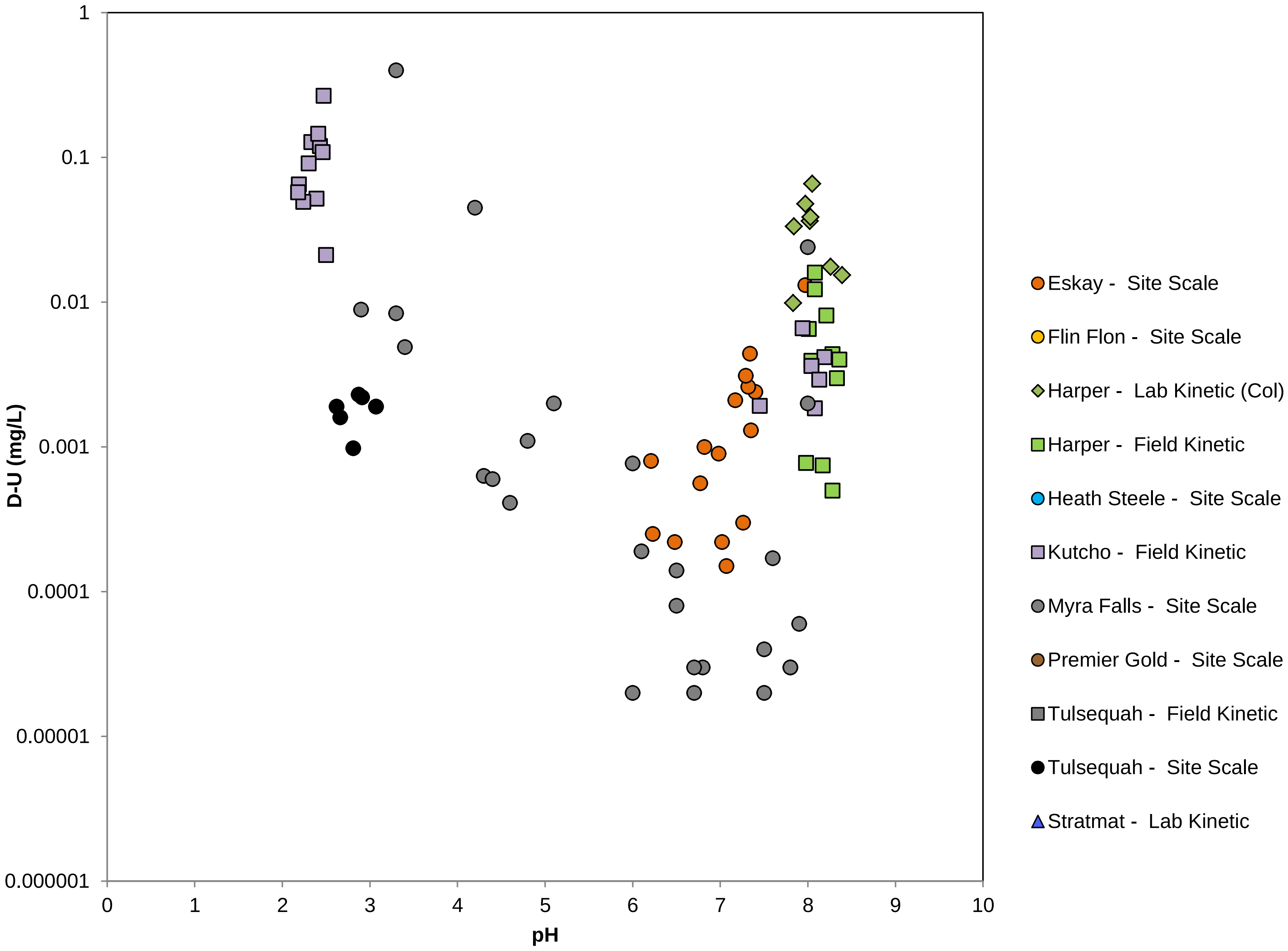

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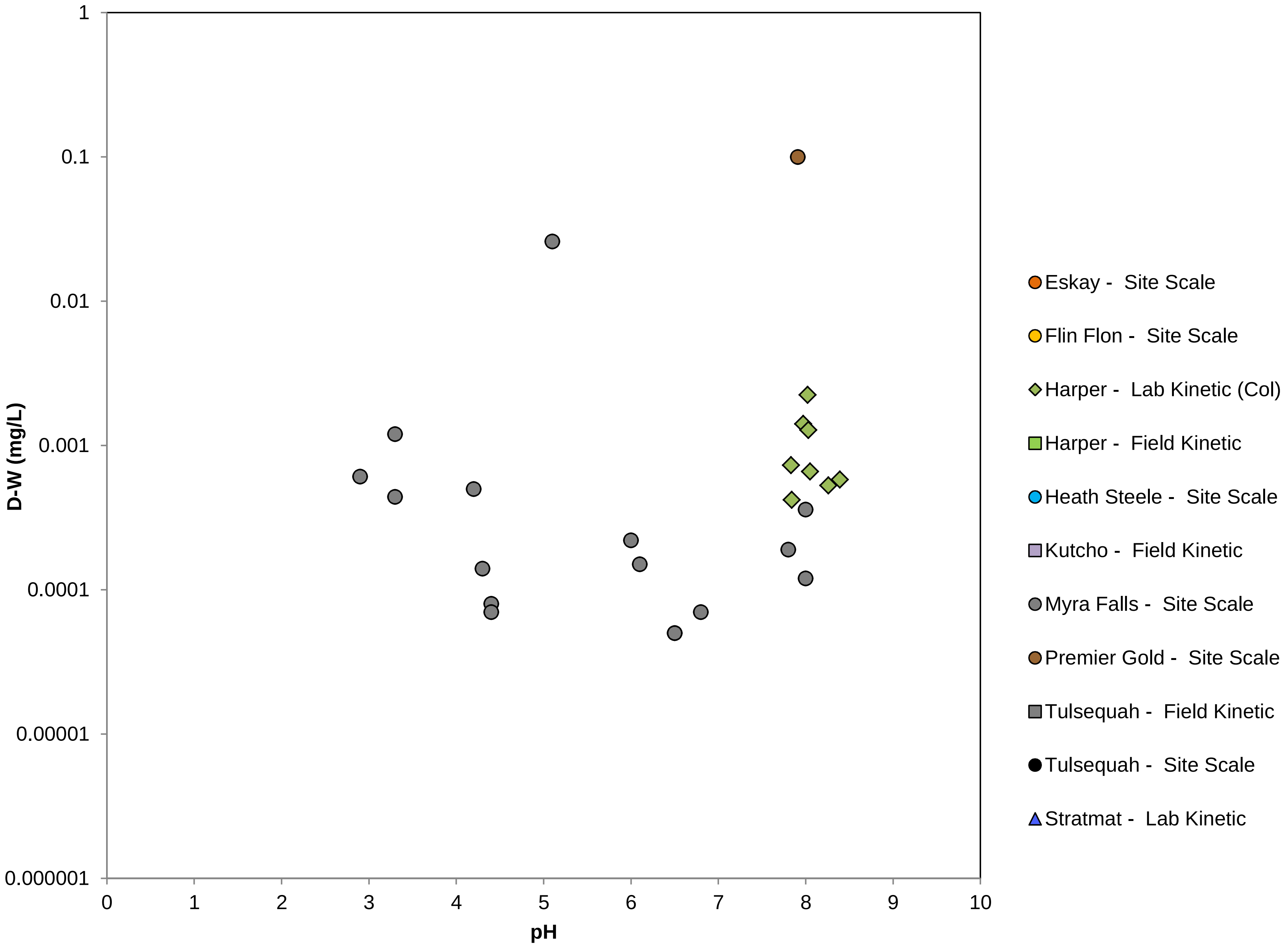

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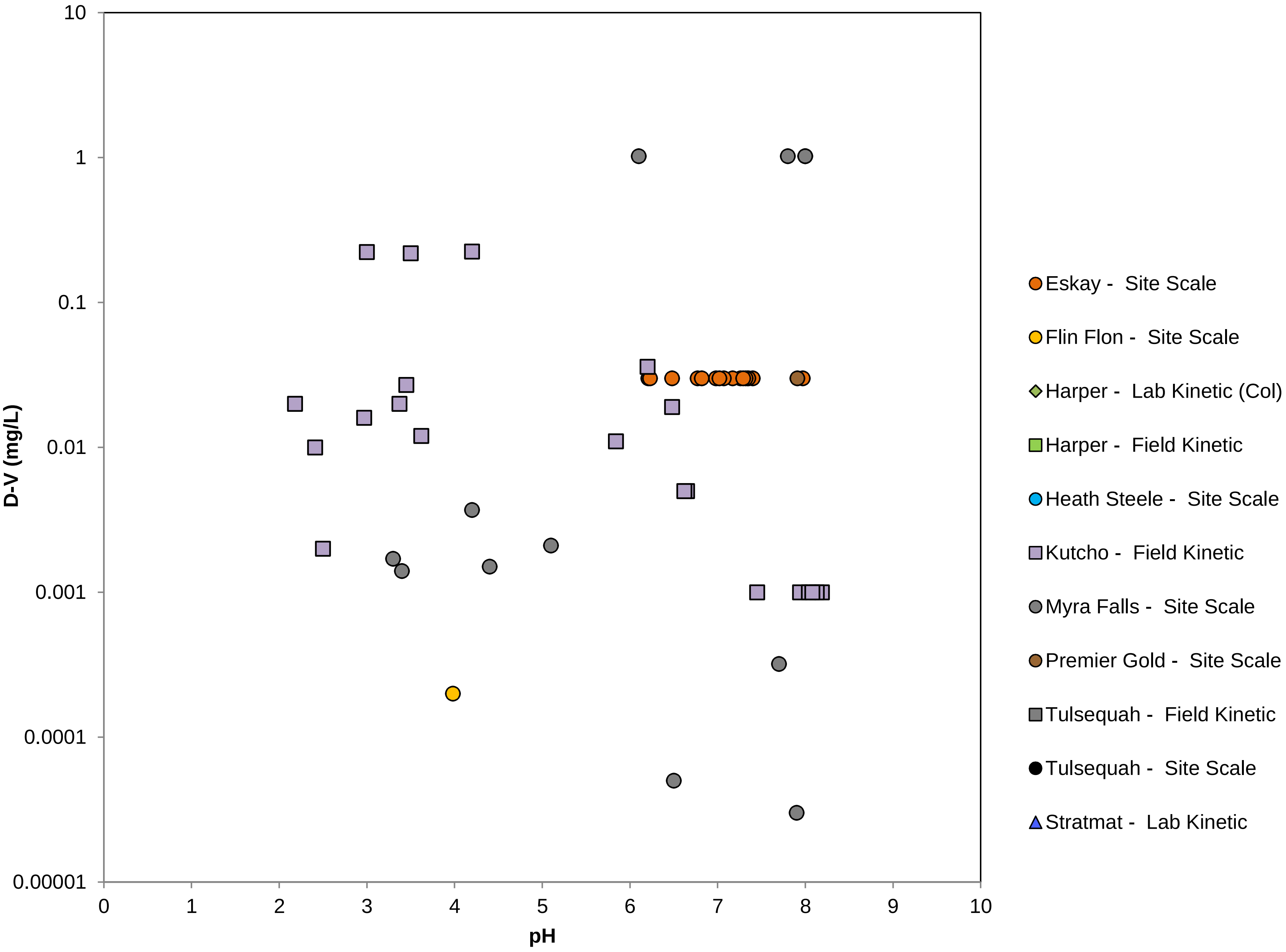

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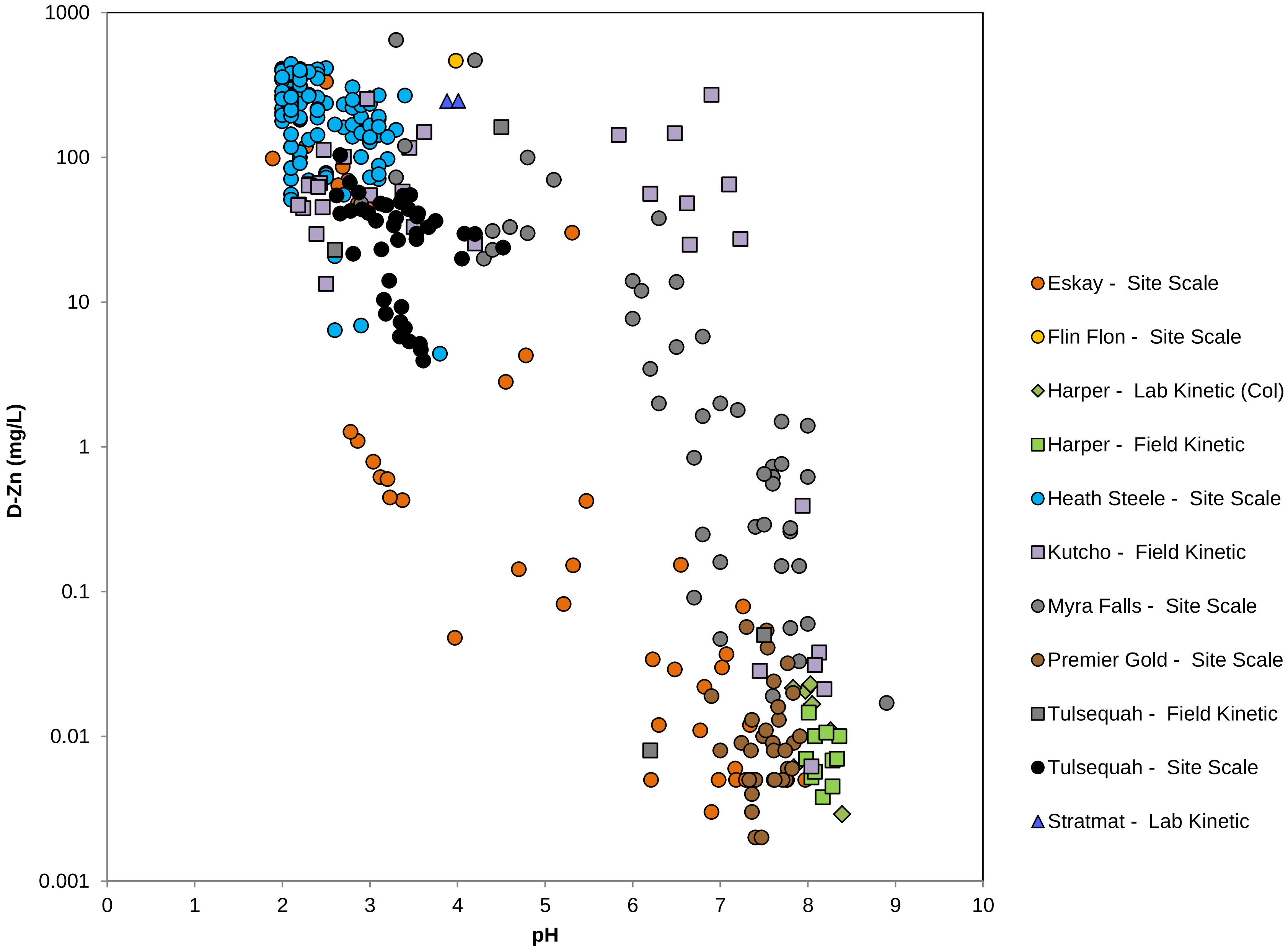

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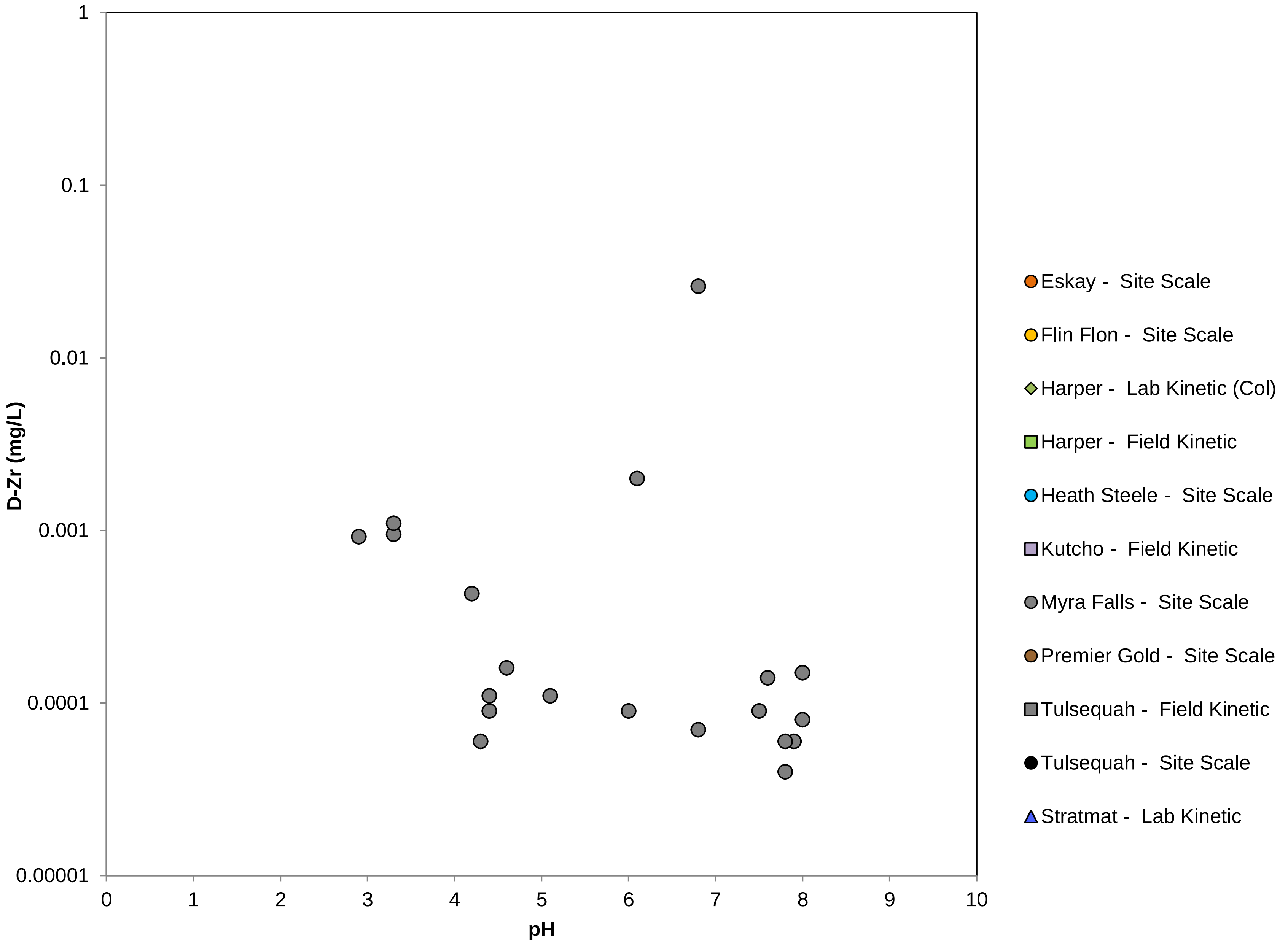

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