

Appendix C

Memo (24 July 2000) – Water Chemistry Predictions

3rd Kinetic Report

Memo (5 June 2001) – Compilation of Water Chemistry Predictions

Memo (12 February 2001) – Average Case Predictions

Memo (12 February 2001) – Explanation of Differences in Predictions of Drystack Runoff Chemistry

Memo (16 January 2002) – Pogo Total & Dissolved Metals Conversion Factors

Memo (19 October 2001) – Manganese Concentrations in Development Rock Pile Drainage

MEMORANDUM

DATE: July 24, 2000

TO: Rick Zimmer, Teck

CC: Eric Konigsmann, Teck
Karl Hanneman, Teck
Bryan Nethery, Agra Simons

FROM: Stephen Day, SRK

PROJECT: 1CT002.00

RE: **POGO PROJECT WATER CHEMISTRY PREDICTIONS**

BACKGROUND

This memorandum describes water chemistry estimates for the various facilities at the Pogo Project. The estimates are updated from previous memoranda dated March 24, 2000, and April 5, 2000.

These estimates are for the following streams:

- Seepage through the co-mingled dry stacked tailings and mineralized rock.
- Runoff from the co-mingled dry stack.
- Mineralized rock stockpile.
- Underground Mine.

The information available to develop the predictions is:

- Humidity cell and column test work for rock and tailings.
- Water quality monitoring data for adit development rock piles and site wide groundwater.
- Flow data from site water balance developed by Agra Earth and Environmental (March 16, 2000 revision) with clarification provided by Mike Davies and Gary Beckstead on March 23, 2000 (Agra 2000).
- Volumes of waste rock and tailings provided by Mike Davies (Agra) on March 7, 2000.



The overall approach has been to develop initial conservative estimates for the parameters requested. More refined modeling may improve these estimates.

The chemistry estimates are provided for the majority of components initially requested by Gene Andrews (February 25, 2000):

- As, Ba, B, Cd, Cu, Fe, Pb, Hg, Mn, Ni, Se, Ag, Zn, sulfate, chloride, nitrate, ammonia, and cyanide species, TDS and hardness.

Aluminum and chromium were added. Estimates for the three nitrogen-containing species are not provided. These are being calculated separately by Tom Higgs (Agra Simons).

The general methods used for each parameter are described in the following sections. This is followed by the method used for each source/facility.

All predictions are for dissolved concentrations. Total concentrations above dissolved concentrations can be estimated by assuming a concentration of total suspended solids and applying typical background solids concentrations to the suspended component. This memorandum does not provide total concentrations.

METHODS BY PARAMETER

Ag, Al, As, B, Ba, Ca, Cd, Cu, Cr, Fe, Hg, K, Mg, Mn, Na, Ni, Pb, Se, sulfate, Zn

The general method is as follows:

1. Using humidity cell data, the release trend of each parameter from each medium (tailings, waste rock, backfill) was determined and used to characterize how the release rate would be calculated. For most elements, the trend was flat, or very slightly decreasing following rapid initial decrease. The average release rates were therefore calculated as a load weighted average on a unit mass per week basis (mg/kg/week).
2. Average release rates were compared to sulfate release and whole rock/tailings composition to detect any relationships with other parameters.
3. Based on the active rock/tailings mass, the release rate was used to calculate the total release on an annual basis. The estimated flow through the source was used to calculate the concentration ("Mass Weighted Concentration").



4. The concentrations of all parameters were then entered into MINTEQA2. The MINTEQA2 output was examined to evaluate possibly mineralogical saturation controls. Any minerals reasonably expected to form, or be present were then assumed to control the leachate chemistry and the equilibrated concentrations re-calculated ("MINTEQ Concentration").
5. Equilibrated concentrations were compared to site observed maxima (all tests and monitoring data) to include the effect of site-specific secondary mineralogical controls.
6. Reasonable Worst Case (RWC) concentration estimates were developed using the following rationale:
 - The MINTEQ Concentration (MC) was used if it was comparable to the Mass Weighted Concentration (MWC), and less than the Testwork and Site Maximum Concentrations (TSMC), that is indicating that the mass leached was not sufficient to reach the TSMC (examples, B, Se). The TSMC is assumed to be limited by processes such as formation of mineral precipitates and adsorption.
 - The MWC was used if it was less than the TSMC, and the TSMC was greater than the MC (example, Mn, Al). If the TSMC exceeds MC, it is likely that the modeling approach in MINTEQ does not represent the minerals controlling the concentrations observed in site specific testwork, and the MC should not be used.
 - The MC was used if it was constrained by well-known abundant minerals (example, alkalinity released from calcite). Certain minerals have well-defined solubility and are known to be present.
 - The TSMC was used if the concentrations were less than the MINTEQ concentration and the Mass Weighted Concentration (examples, As, Ba, Cr, Cu, g, K, Na, Ni, Pb). This allows for site specific control on element concentrations not modeled by MINTEQ. An example is formation of ferric arsenates with high Fe/As ratios which are not modeled by MINTEQ but have a significant effect on the solubility of arsenic.
7. The major element (Ca, Mg, K, Na, sulfate, alkalinity) concentrations were re-entered into MINTEQA2 and assumed to be constrained by calcite, dolomite and gypsum. The ion balance was checked and the final predicted concentrations were included in the Reasonable Worst Case.

Chloride

Chloride concentrations have been determined as part of the kinetic test study, however, the concentrations do not appear to be reliable based on comparison of directly measured chloride and chloride by ICP-MS. The mineralogical source of Cl in the material is not known but it is probably chlorite. All tests have tended to yield similar maximum concentrations of chloride (36 to 53 mg/L) it is therefore recommended that a concentration of 53 mg/L be used.

TDS and Hardness

These parameters cannot be modelled on a mass balance basis because they reflect the concentration of several parameters. Concentrations were estimated by predicting the concentrations of major ions (Ca, Mg, K, Na, SO₄, alkalinity), and summing the appropriate elements.

Comments

The majority of parameters have uniform low level release that is unrelated to sulfate or the whole rock geochemistry. Exceptions are:

- Sulfate release is correlated with total sulfur concentration.
- Arsenic release from waste rock is correlated with total arsenic concentration (with some exceptions).
- Selenium release from waste rock is strongly correlated with sulfate release. Selenium release was calculated based on sulfate release.

Minerals assumed to constrain concentrations under non-acidic conditions were calcite, dolomite, gypsum, alunite, barite, siderite, rhodochrosite, otavite, cerrusite, malachite, mercury carbonate and lead chromate. Due to the basic conditions, carbonate minerals are expected to limit concentrations of many parameters.

No mineralogical constraints on arsenic, boron, nickel, selenium or silver were identified.

In the case of arsenic it is very likely that arsenic concentrations are controlled by ferric arsenates. The solubility of ferric arsenates is strongly dependent on the Fe/As ratio and pH. Minimum solubility typically occurs in the pH 4 to 6 range. Solubility increases as Fe/As decreases, being greatest for FeAsO₄. Comparison of arsenic concentrations with pH from various laboratory tests and field monitoring indicates that upper limits on arsenic concentrations are constrained by pH, and that the constraints vary according to the degree of arsenic



mineralization (Figure 1). The rock kinetic tests indicate constraints at higher arsenic concentrations associated with mineralized rock containing arsenopyrite. Unmineralized rock shows a lower constraint. Tailings kinetic tests are generally consistent with the more mineralized rock. The groundwater data lies between the unmineralized and mineralized rock. The highest arsenic concentrations in groundwater occur in non-acidic reduced groundwater containing relatively low concentrations of iron, that is, representing high solubility conditions for arsenic. Based on these observations, arsenic solubility is controlled by the presence of iron and the pH conditions. The groundwater indicates a maximum arsenic concentration of about 5 mg/L. This was selected as the Reasonable Worst Case limit for arsenic predictions because it represents low iron and reducing conditions that will not exist in the waste disposal facilities.

METHODS BY FACILITY

Co-mingled Tailings and Waste Rock

The following specific assumptions were used:

- The configuration at Year 6 was used (ie when the bulk of mineralized rock has been placed).
- The waste rock chemical release rates were based on humidity cells. No correction was made for lower oxygen concentrations and lower temperatures expected in the mixture.
- The tailings chemical release rates for infiltrating water were based on the leach columns.
- The tailings chemical release rates for runoff water were based on the humidity cells.
- It was assumed that 5% of the rock is "fine" and 20% of the fines are contacted by water.
- It was assumed that 20% of the tailings mass is contacted by infiltrating water.
- The seepage load was calculated based on the sum of loads from the co-mingled mineralized rock, co-mingled tailings and non-mineralized toe berm.
- The load released by tailings in contact with runoff was estimated by assuming that annual release from a reactive skin (set at 1 cm) is contained in the total runoff in positive net precipitation months (snowmelt in spring, and fall rainfall).
- The waste rock is non-acid generating.
- The flow through the co-mingled mass was assumed to be 10 USgpm (Agra Reference).
- The effect of mixing with process water was not considered.

Water chemistry was predicted for

1. Seepage from the facility
2. Surface runoff



Mineralized Rock Stockpile

These predictions assume:

- Similar characteristics to the mineralized rock removed during adit development.
- Maximum size (1.2 million tons)
- Runoff/seepage at 5 USgpm.

The load released from a smaller pile could be calculated in direct proportion to the mass of the pile.

Underground Mine

During operation, water chemistry is expected to be dominated by mixing of groundwater and drainage from backfill. The chemistry of backfill drainage should be assumed to be the same as the process water. Kinetic testing indicates that the backfill is potentially acid generating, but that it will not generate acid during operation.

At closure, the effect of acidification of the backfill and leaching of oxidized backfill and any broken rock will be control water chemistry. The closure conditions have not been determined and have therefore not been modelled.

PREDICTIONS

Dry Stack Seepage

Predictions for seepage from the dry stack are shown in Table 1.

The initial mass weighted concentrations are high due to the relatively low volume of seepage compared to the mass of tailings and rock.

MINTEQ predicts that these concentrations would be limited by the formation of several secondary minerals. This causes concentrations of several major parameters to decrease substantially (Al, Ca, Mg, alkalinity), leaving high concentrations of sulfate, potassium and sodium. It is probably not valid to assume that potassium and sodium will be released to the levels indicated because they originate from silicates. The predictions of potassium and sodium were therefore reduced based on observed concentrations in tests and site waters.

Since the predictions for most parameters were adjusted to reflect solubility limitations apparent in testwork and field monitoring, changes in the physical parameters used to make the initial



mass-weighted predictions will have negligible effects on the final predictions. However, concentrations are not expected to be any greater than the estimates. The reasonable worst case shown in Table 1 could therefore also be treated as the expected case.

**TABLE 1
Drystack Seepage**

Parameter	Mass Weighted Conc. mg/L	MINTEQ Conc mg/L	Testwork Max Conc. mg/L	Site Data Max Conc. mg/L	Reasonable Worst Case Conc. mg/L	Geochemical Control
Ag	0.012	0.056	0.0018	0.00011	0.0018	Unknown
Al	1.59	0.005	0.868	1.84	1.59	Hydroxides
As	56.8	59.2	1.8	5.1	5.1	Ferric arsenates
B	3	3	0.3	0.116	3	Unknown
Ba	1.66	0.001	0.055	0.491	0.49	Sulfate
Alkalinity	21643	174	190	530	174	Carbonate
Ca	10662	195	290	936	291	Carbonates, gypsum
Cd	0.01	0.005	0.0006	0.0046	0.005	Carbonate
Cr	0.49	0.14	0.014	0.00184	0.014	Lead chromate
Cu	0.30	0.63	0.026	0.0336	0.034	Carbonate
Fe	17	3	1.3	29.6	3	Ferric hydroxide
Hg	0.16	0.001	0.002	0.0001	0.002	Carbonate
K	2738	2871	59	19	59	Silicate dissolution
Mg	2802	199	83	294	300	Carbonate
Mn	19	3	1.9	29.7	19	Carbonate
Na	578	606	130	77.9	130	Silicate dissolution
Ni	0.26	0.28	0.057	0.236	0.24	Unknown
Pb	0.45	0.18	0.005	0.00535	0.005	Carbonate
SO ₄	6897	6692	800	745	2002	Gypsum
Se	0.13	0.13	0.049	0.019	0.13	Unknown
Zn	0.37	0.387	0.11	0.699	0.37	Carbonate
Hardness				1180	1980	Carbonate
TDS	45000	10000	Not analyzed	7160	3000	Not applicable

Based on flow rate of 10 USgpm (5.3 million USg per year)

Dry Stack Runoff

Runoff concentrations are expected to be relatively low (Table 2) primarily due to the limited contact with the tailings. MINTEQ indicates minor adjustments to alkalinity and calcium. MINTEQ adjusted concentrations are similar to the mass-weighted concentrations and are mostly lower than the site or testwork maxima. Therefore, the Reasonable Worst Case concentrations are similar to the mass-weighted concentrations. The concentrations are “Reasonable Worst Case” because the leaching rates are based on an active leaching depth of 1



cm, and the rates are room temperature. A decrease in the active depth to 0.5 cm would result in 50% lower concentrations than those currently predicted.

**TABLE 2
Dry Stack Runoff**

Parameter	Mass Weighted Conc. mg/L	MINTEQ Conc mg/L	Testwork Max Conc. mg/L	Site Data Max Conc. mg/L	Reasonable Worst Case Conc. mg/L	Geochemical Control
Ag	0.0002	0.0002	0.0018	0.00011	0.0002	Unknown
Al	0.095	0.095	0.868	1.84	0.095	Hydroxides
As	0.38	0.38	1.8	5.1	0.4	Arsenates
B	0.033	0.033	0.3	0.116	0.033	Unknown
Ba	0.007	0.007	0.055	0.491	0.007	Sulfate
Alkalinity	98	158	190	530	158	Carbonate
Ca	86	42	290	936	42	Carbonates, gypsum
Cd	0.0001	0.0001	0.0006	0.0046	0.0001	Carbonate
Cr	0.003	0.003	0.014	0.00184	0.003	Lead chromate
Cu	0.005	0.01	0.026	0.0336	0.005	Carbonate
Fe	0.329	0.0004	1.3	29.6	0.0004	Ferric hydroxide
Hg	0.003	0.0012	0.002	0.0001	0.001	Carbonate
K	25	25	59	19	25	Silicate dissolution
Mg	37	37	83	294	37	Carbonate
Mn	0.105	0.10	1.9	29.7	0.10	Carbonate
Na	11	11	130	77.9	11	Silicate dissolution
Ni	0.006	0.01	0.057	0.236	0.01	Unknown
Pb	0.006	0.01	0.005	0.00535	0.005	Carbonate
SO ₄	255	255	800	745	255	Gypsum
Se	0.010	0.01	0.049	0.019	0.01	Unknown
Zn	0.007	0.007	0.11	0.699	0.007	Carbonate
Hardness	370	260		1180	260	Carbonate
TDS	500	517		7160	529	Not applicable

Based on annual flow rate of 2.4 million USg in Year 6.

TEMPORARY MINERALIZED ROCK STORAGE STOCKPILE

Table 3 shows predicted reasonable worst case concentrations in the temporary mineralized rock pile seepage and runoff. Changes in several physical and chemical variables could result in lower concentrations in drainage. Possible significant causes are the decrease in reactive mass due to internal freezing and slower reaction kinetics at the lower temperatures. Most of the trace elements could be affected by this process and the resulting concentrations would be lower than predicted for the Reasonable Worst Case.



The second last column of Table 3 compares the predictions to maximum observed concentrations from the development rock currently monitored at site locations SW25B (non-mineralized rock) and SW26 (mineralized rock) during 2000. The metals concentrations shown are for "dissolved" because the mass-weighted predictions are also based on dissolved concentrations measured in testwork. Due to the method used to estimate the reasonable worst case, the reasonable worst case concentrations are greater than the observed maximum concentrations with the exception of Ca, Mg, Mn, Na, SO₄ and TDS. The difference for TDS is substantial (772 mg/L predicted compared to 7160 mg/L observed in the non-mineralized pile drainage). A significant component of the observed TDS in the non-mineralized pile drainage is nitrate, which is present at maximum concentrations of 950 N-NO₃ mg/L, or 3500 mg NO₃/L. The presence of nitrate in turn raises the concentration of other metals (Ca, Mg, Mn, Na) due to the high solubility of their nitrates. The non-mineralized pile drainage contains higher dissolved concentrations of many elements (Ba, Cd, Co, Cu, Hg, Ni, Pb, Zn) for the same reason.

Teck has indicated that the emulsion explosives hauled to the site over the winter road deteriorated after sitting a full year before use, resulting in incomplete detonation. Although use of these explosives was discontinued on date?xxx, a larger than normal amount of undetonated explosives ended up in the non-mineralized pile. Calculations of explosive losses by others (T. Higgs, personal communication) indicate that much lower nitrate concentrations can be expected for the temporary mineralized rock pile. These concentrations will be a small component of the TDS. The predicted TDS concentrations are more realistic (and are comparable to the current mineralized pile), but should be increased slightly to account for nitrate, nitrite and ammonia.



**TABLE 3
Temporary Mineralized Rock Pile Seepage and Runoff**

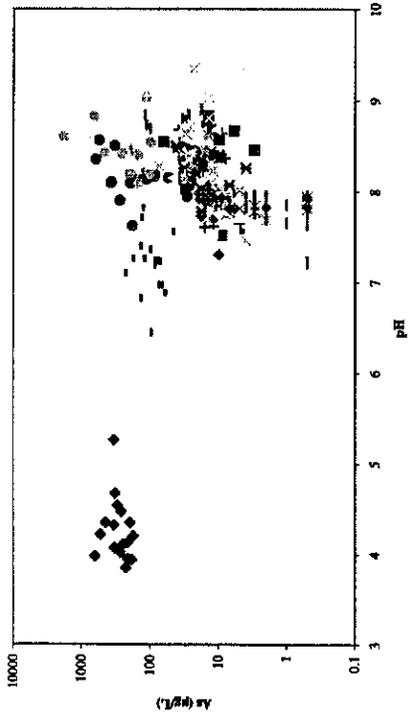
Parameter	Mass Weighted Conc. mg/L	MINTEQ Conc mg/L	Testwork Max Conc. mg/L	Site Data Max Conc. mg/L	Reasonable Worst Case Conc. mg/L	Observed Mineralized Rock mg/L	Geochemical Control
Ag	0.0026	0.0030	0.0018	0.00011	0.0018	<0.00002	Unknown
Al	2.385	0.102	0.868	1.84	1.84	0.079	Hydroxides
As	0.50	0.50	1.8	5.1	0.5	0.011	Arsenates
B	0.256	0.255	0.3	0.116	0.3	0.0295	Unknown
Ba	0.163	0.006	0.055	0.491	0.163	0.114	Sulfate
Alkalinity	840	187	190	530	187	56	Carbonate
Ca	359	31	290	936	31	936	Carbonates, gypsum
Cd	0.004	0.004	0.0006	0.0048	0.0048	0.0048	Carbonate
Cr	0.03	0.03	0.014	0.002	0.014	0.0009	Lead chromate
Cu	0.05	0.05	0.026	0.0336	0.03	0.011	Carbonate
Fe	1.45	1.45	1.3	29.6	1.45	0.084	Ferric hydroxide
Hg	0.024	0.024	0.002	0.0001	0.002	0.000005	Carbonate
K	110	109	59	19	59	19	Silicate dissolution
Mg	100	33	83	294	100	294	Carbonate
Mn	0.98	0.98	1.9	29.7	0.98	29.7	Carbonate
Na	9	9	130	77.9	9	441	Silicate dissolution
Ni	0.05	0.05	0.057	0.236	0.05	0.05	Unknown
Pb	0.05	0.05	0.005	0.00535	0.005	0.00012	Carbonate
SO ₄	274	268	800	745	386	644	Gypsum
Se	0.03	0.03	0.049	0.019	0.03	0.008	Unknown
Zn	0.056	0.056	0.11	0.699	0.06	0.029	Carbonate
Hardness	NC	NC	NC	1180	494	3410	Carbonate
TDS	NC	NC	NC	7160	772	7160	Not applicable
N-NO ₃	NC	NC	NC	NC	NC	950	Explosives

Notes:

NC – Not calculated.

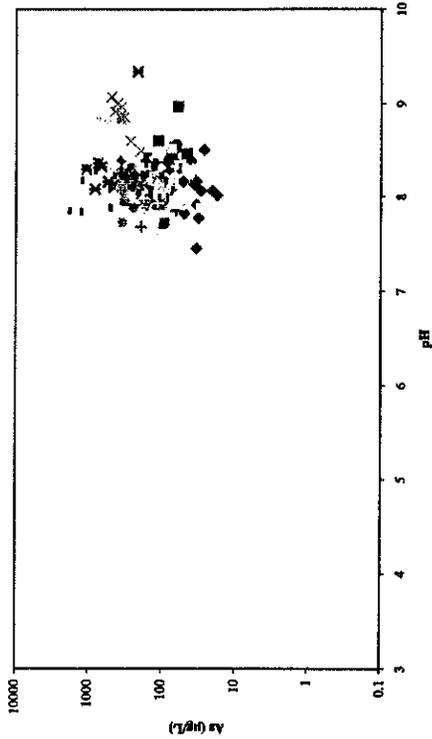
N-NO₃ – Calculated by Agra-Simons (T. Higgs)

Rock Kinetic Tests



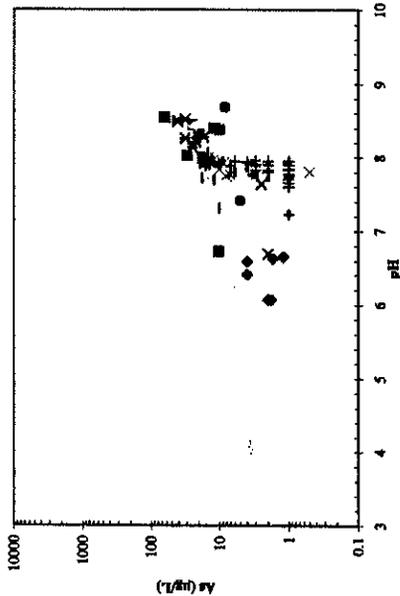
- ◆ Cell 1
- Cell 2
- Cell 3
- × Cell 4
- × Cell 5
- Cell 6
- + Cell 7
- Cell 8
- Cell 9
- Blank
- Cell 10
- Cell 11
- × Cell 22
- × Cell 23
- Cell 24
- Cell 25
- Column 1
- Column 2

Tailings Kinetic Tests



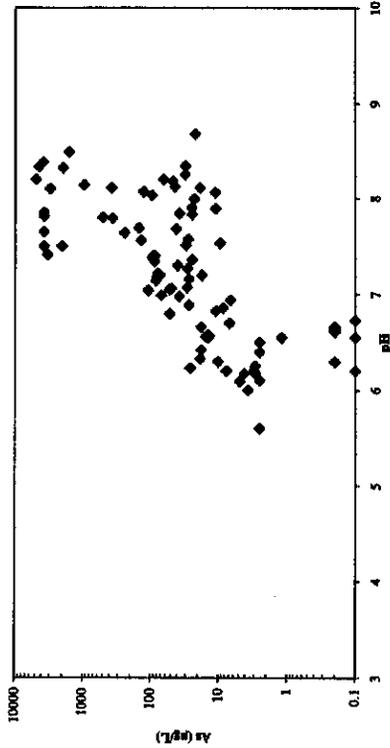
- ◆ Cell 12
- Cell 13
- Cell 14
- × Cell 15
- × Cell 18
- + Cell 20
- Cell 21
- Cell 26
- Cell 27
- Cell 28
- Cell 29
- Cell 30
- × Cell 31
- × Cell 32
- Column 4
- Column 5
- Column 6
- Column 7

Unmineralized Rock



- Cell 2
- Cell 3
- × Cell 4
- × Cell 5
- + Column 1
- Column 2
- Column 3
- ◆ Non-Mineralized File
- Mineralized File
- ▲ Adit rock MWMP
- ▲ Mineralized File - 2000
- × Non-Mineralized File - 2000
- SW31

Groundwater



TECK CORPORATION

WATER CHEMISTRY PREDICTIONS

Pogo Project

As Concentration Compared to pH

Project	Date	Approved	Figure
1CT002.00	June 2009		1

TECK-POGO INC.



1CT002.00

**THIRD PROGRESS REPORT ON
KINETIC GEOCHEMICAL TESTS
POGO PROJECT**

1CT002.00

**THIRD PROGRESS REPORT ON
KINETIC GEOCHEMICAL TESTS
POGO PROJECT**

Prepared for:

TECK-POGO INC.
600-200 Burrard Street
Vancouver, British Columbia
V6C 3L9

Prepared by:

STEFFEN ROBERTSON AND KIRSTEN (CANADA) INC.
Suite 800, 580 Hornby Street
Vancouver, B.C. V6C 3B6
Tel: (604) 681-4196 • Fax: (604) 687-5532
E-mail: vancouver@srk.com Web site: www.srk.com

FEBRUARY 2001

1CT002.00

**THIRD PROGRESS REPORT ON
KINETIC GEOCHEMICAL TESTS
POGO PROJECT**

TABLE OF CONTENTS

1.0 INTRODUCTION	1
1.1 Background	1
1.2 Study Participants	2
2.0 TESTING DESIGN	2
2.1 Testing Objectives	2
2.2 Sample Selection.....	3
2.2.1 Waste Rock.....	4
2.2.2 Ore	5
2.2.3 Borrow Source Rock	6
2.2.4 Flotation Tailings	6
2.2.5 Cemented Flotation Tailings	6
2.2.6 Backfill	7
2.3 Test Method Selection	7
2.3.1 Pre-Test Characterization	7
2.3.2 Kinetic Test Methods	8
2.3.3 Quality Control.....	10
3.0 PROGRESS RESULTS	11
3.1 Pre-Test Characterization.....	11
3.1.1 Waste Rock and Ore	11
3.1.2 Borrow Source Rock	12
3.1.3 Flotation Tailings	12
3.1.4 Cemented Flotation Tailings	12
3.1.5 Simulated Backfill	13
3.2 Kinetic Tests	13
3.2.1 Introduction	13
3.2.2 Waste Rock.....	14
3.2.3 Ore	17
3.2.4 Borrow Source.....	18

3.2.5 Flotation Tailings	19
3.2.6 Cemented Flotation Tailings	21
3.2.7 Simulated Backfill	22
4.0 CONCLUSIONS.....	24
5.0 RECOMMENDATIONS.....	25
6.0 REFERENCES AND SUPPORTING DOCUMENTS	27

LIST OF TABLES

Table 1	Test Program Samples and Test Types
Table 2	Selection of Rock Samples for Kinetic Testwork
Table 3	Kinetic Test Leachate Analysis Detection Limits
Table 4	Summary of Tests – Construction and Operation Details
Table 5	Waste Rock Test Material Characteristics and Selected Outcomes
Table 6	Ore Test Material Characteristics and Selected Outcomes
Table 7	Borrow Rock Test Material Characteristics and Selected Outcomes
Table 8	Flotation Tailings Test Material Characteristics and Selected Outcomes
Table 9	Cemented Flotation Tailings Test Material Characteristics and Selected Outcomes
Table 10	Simulated Backfill Test Material Characteristics and Selected Outcomes
Table 11	Recommendations for Each Test

LIST OF FIGURES

Figure 1	Overall Structure of the Kinetic Test Program
Figure 2	Waste Rock Humidity Cells - pH(t), SO ₄ (t), As(t)
Figure 3	Rock - Rate and Bulk Chemistry Relationships
Figure 4	Ore Humidity Cells- pH(t), SO ₄ (t), As(t)
Figure 5	Borrow Humidity Cells - pH(t), SO ₄ (t), As(t)
Figure 6	Flotation Tailings Columns - pH(t), SO ₄ (t), As(t)
Figure 7	Flotation Tailings Humidity Cells - pH(t), SO ₄ (t), As(t)
Figure 8	Process Materials - Rate and Bulk Chemistry Relationships
Figure 9	Cemented Flotation Tailings Humidity Cells - pH(t), SO ₄ (t), As(t)
Figure 10	Simulated Backfill Humidity Cells - pH(t), SO ₄ (t), As(t)

**THIRD PROGRESS REPORT ON
KINETIC GEOCHEMICAL TESTS
POGO PROJECT**

1.0 INTRODUCTION

1.1 Background

Teck-Pogo Inc.'s Pogo Project, located near Delta Junction, Alaska has progressed to the stage of advanced exploration, including development of an exploration adit, and pre-feasibility engineering design. The project will include underground mining of ore and on-site processing by gravity concentration, flotation and/or cyanide leaching to recovery gold. To the extent possible, process plant tailings will be disposed of in the underground workings as cemented backfill; those tailings not backfilled will require disposal in a dry-stacked tailings storage facility.

Based on this conceptual plan, potential environmental issues related to mineral weathering have been identified on an ongoing basis to define objectives for kinetic weathering studies, including, but not limited to:

- Leaching of contaminants (primarily arsenic) from non-acid generating development rock and mine walls exposed during development of access tunnels to the vein.
- Acid generation rates and leaching of contaminants (primarily arsenic) from acid generating and mineralized vein wall rocks.
- Leaching of contaminants from sulfide concentrate.
- Leaching of contaminants from dry-stacked flotation tailings.
- Leaching of contaminants from cemented underground backfill during operation.
- Solute release from mine walls and backfill during mine decommissioning

This report provides an update to two previous reports prepared by SRK Consulting in July 1999 (SRK Consulting 1999) and July 2000 (SRK Consulting 2000b). The second report was appended to the Water Management Plan for the project (Teck-Pogo Inc. 2000).

1.2 Study Participants

The study involves the participation of the following organizations and individuals:

- Teck Corporation (Project Manager – Rick Zimmer; Others – Karl Hanneman, Kim Bittman, Eric Konigsman, Janet Freeth) – Coordination of sample collection, Teck Corporation point-of-contact.
- SRK Consulting (Project Manager: Stephen Day) – Study design, laboratory monitoring and data interpretation.
- BC Research Inc (Project Manager: Rik Vos; Technician: Terri-Lynn Delaney) – Operation of tests, analysis of anions, supervision of analytical laboratory and analytical QA/QC.
- Acme Analytical Laboratories (Project Manager – Clarence Leong) – Trace element scans on solids and leachates.
- ALS-Chemex (Project Manager - Carolyn Low) – Acid-base accounting and trace element analysis.

2.0 TESTING DESIGN

2.1 Testing Objectives

To develop the test plan, a series of objectives was defined to guide sample and method selection:

- Define site-specific classification criteria for acid generating rock for use during mining.
- Evaluate the effect of total contaminant (particularly arsenic) concentration and mineralogy in rock and tailings on the release rate of contaminants under non-

acidic conditions, and develop predictive relationships between whole rock characteristics and leaching rates.

- Evaluate the time to onset of acid generation for potentially acid-generating materials primarily to determine materials which may become acid generating prior to decommissioning
- Evaluate release of contaminants under acidic conditions to predict the behavior of acid-generating waste rock in development rock dumps and mine walls.
- Understand the formation of secondary minerals particularly to determine the storage and release of contaminants as weathering proceeds and as the mine workings are flooded during decommissioning.
- Evaluate the effect of low temperatures on weathering rates.

2.2 Sample Selection

Six types of wastes and materials are being tested:

1. **Waste Rock** – This material type includes rock along the exploration adit alignment
2. **Ore.**
3. **Borrow Source Rock** – This is rock that will be extracted from a quarry for construction of the Recycled Tailing Pond (RTP) Dam located downstream of the Dry Stack Tailings
4. **Flotation Tailings** – These tailings will be placed in the Dry Stack.
5. **Cemented Flotation Tailings** – These materials were tested as part of an original proposal to cement flotation tailings.
6. **Cemented Backfill** – These materials will consist of blended leached and flotation tailings with cement.

The overall structure of the test program is illustrated in Figure 1. Table 1 provides more detail on the various material types and the tests conducted.

The rationale for selection of samples of each type is described in the following sections.

2.2.1 Waste Rock

The criteria for selection of rock samples were initially based a three-way matrix classifying major rock type, broad test work objective and arsenic concentration (Table 2). Gneiss and granodiorite are the main host rocks with overall geochemical characteristics described by Norecol Dames & Moore (1998).

TABLE 2
Selection of Rock Samples for Kinetic Testwork

Objective	ROCK TYPE/SAMPLES			
	A. Gneiss	B. Granodiorite	C. Altered rock near vein.	D. Ore
1. Lag time to As release, onset of acid generation and rate at pH.	None	None	One of samples below to include sample classified as potentially acid generating.	One sample
2. As concentrations under non-acidic conditions	Three samples (mean and high As concentrations)	Three samples (mean and high As concentrations).	3 samples (mean and high As concentrations)	Three samples (including the above sample) containing a range of As concentrations.
3. Effect of low temperatures	Duplicate of one of the above.		Duplicate of one of the above	

A short list of suitable samples meeting the above criteria was developed and individual samples selected based on:

- greatest likelihood of being mined or exposed as part of mining;
- even spatial coverage of samples;
- availability of sufficient sample for testing (defined as at least 1.5 kg).

The samples were selected from rock diamond drill hole (DDH) core stored inside enclosed core racks at the project site. The core was split using a diamond saw to leave sufficient material for future analyses should these be required. The core would be described a “fresh” because it showed no evidence of oxidation in the core boxes

Three samples of gneiss distant from the orebody were selected to characterize rock that would be intersected during development of the adit. Results of the testing were described by SRK (1999). Four samples of the rock excavated from the exploration adit were also collected in the winter of 2000 to correlate laboratory leaching rates with runoff from adit development rock piles (SRK 2000a). These samples were selected to test various combinations of high and low sulfur, and arsenic concentrations with reference to the non-mineralized/mineralized waste rock criteria of 0.5% and 200 mg/kg, respectively.

All rock samples were collected by Teck Corporation personnel, shipped to Chemex Laboratories in Fairbanks for initial processing, stored in plastic bags then shipped to BC Research for kinetic testing..

Geochemical characteristics of all waste rock materials tested in humidity cells are shown in Table 5. The majority of samples are not expected to generate acid based on the acid-base account. A few samples containing higher sulfur concentrations were purposely selected to evaluate acid generation (Cells 1, 4 and 8).

2.2.2 Ore

Three ore samples were selected by Teck to evaluate weathering effects in the walls of the underground mine. Three samples were selected based primarily on the sulfide mineralogy of the samples reported by CANMET (1998):

- Sample 97-43 – Dominantly pyrrhotite
- Sample 97-50 – Dominantly arsenopyrite and löllingite.
- Sample 97-41A – Dominantly arsenopyrite and pyrite.

Four samples of crushed ore used in the Phase 1 Pilot Processing Plant are being tested in leach columns.

Geochemical characteristics of the test materials are shown in Table 6.

2.2.3 Borrow Source Rock

Three composite samples of diamond drill core from the vicinity of the diorite quarry are being tested in humidity cells. The three drill holes selected (LD-1, LD-3, LD-4) were near the proposed quarry, on the same side of the valley as the quarry, and in the same rock type (diorite). The composites did not include weathered material that would be unsuitable for construction.

Geochemical characteristics of the test materials are shown in Table 7.

2.2.4 Flotation Tailings

Flotation tailings sample selection has followed process testing at the bench- and pilot plant-scales. Samples are being tested as follows:

- **F48 at Bench Scale** – Four humidity cells and four columns started in January, 1999 and March 1999.
- **F96, F97, F97 and F98 at Bench Scale** – Six humidity cells and one column started in June 1999.
- **Phase 1 Pilot Plant** – Four humidity cells and four columns started in November 2000.

For the Pilot Plant tests, two types of flotation tailings are being tested:

1. Flotation tailings produced by a gravity circuit in the grinding circuit.
2. Flotation tailings produced by a flash flotation cell in the gravity circuit.

Two samples of each of these types are being tested in both humidity cells and columns.

2.2.5 Cemented Flotation Tailings

Cemented flotation tailings tests were started in March 1999 when disposal of cemented tailings rather than dry-stacked tailings was considered.

Several types of cemented flotation tailings samples were produced for testing:

- Addition of 1%, 2% and 3% cement to flotation tailings from F48 processing.
- Addition of 5% cement to flotation tailings from FXX processing.

Lakefield Research prepared the tailings samples. The tailings were cemented into cylinders by Golder Associates and shipped packed in sand to BC Research.

2.2.6 Backfill

Cemented backfill in the underground mine would consist of various combinations of flotation tailings and leached tailings with a few percent cement. The test materials were produced from bench scale processing products. The following materials are being tested in humidity cells:

- 100% leach tailings with 10% cement (March 2000).
- Control - Uncemented 100% leach tailings (April 1999).
- 50% leach and 50% flotation tailings with 5% cement (March 2000).
- 20% leach and 80% flotation tailings with 5% cement (March 2000).
- 20% leach and 80% flotation tailings with 2% cement (March 2000).

2.3 Test Method Selection

2.3.1 Pre-Test Characterization

Prior to testing, samples are tested using the following methods:

- Acid-base accounting (ABA) using the EPA method (Sobek *et al.* 1978).
- Hydrochloric acid soluble sulfur (as a measure of soluble sulfate)
- Total inorganic carbon (TIC) (as a measure of carbonate content).
- Elemental analysis using ICP-ES following aqua regia digestion.
- Soluble element analysis using EPA method 1312 (Simulated Precipitation Leach Procedure, west of Mississippi River leach, US EPA 1996) (initial rock samples only).
- Meteoric Water Mobility Procedure (MWMP) (Exploration adit alignment samples only) (Nevada Mining Association, 1996).
- Optical mineralogy from polished thin sections.
- Mineralogy by scanning electron microscope (selected samples).

Two fractions of most rock samples were analysed following production of >2 mm and <2 mm size fractions. Each fraction was analysed for ABA, soluble sulfate, TIC and aqua regia-digestible elements. The fine fraction was analyzed using EPA method 1312.

2.3.2 Kinetic Test Methods

2.3.2.1 Rock

The base method for kinetic testing of rock samples is the humidity cell procedure recommended by the Canadian Mine Environment Neutral Drainage (MEND) Program (Coastech Research 1991). Humidity cells are used to estimate primary weathering rates. During testing, the sample is fully oxygenated and then inundated with water to dissolve weathering products. The dissolved weathering products interact with the sample matrix. Generally, the leachate to rock ratio is high ensuring that solubility of most sulfate weathering products is not limited. Solubility limitations are usually apparent for heavy elements and carbonates. The leachate collected from the cell therefore contains the soluble products of oxidation, dissolution of buffering minerals by the leaching solution and solubility-limited concentrations of heavy elements.

Each humidity cell is constructed from 11-cm diameter plexiglass pipe. The cell is 17 cm high and contains 1 kg of rock sample (crushed to $\frac{1}{4}$ " to a depth of about 17 cm. The base plate of the column is perforated and the test material rests on Nytex mesh to limit loss of fines during leaching. The cell is initially leached with 750 mL of deionized water.

The cell is aerated with dry air for three days, then with humid air for three days. On the seventh day, 500 mL of water is added to the cell as a single batch. The cell is stirred to ensure thorough contact with the leachate and leached for 1 hour. The leachate is drained from the cell. The volume of leachate extracted is recorded. About 250 mL of leachate is filtered (0.45 μ m), acidified (nitric acid) and analyzed for elements every other week. Two water analysis methods are used alternately: ICP-ES and ICP-MS. Detection limits for both methods are indicated in Table 3. The latter is used to obtain very low detection limits (particularly for As). The balance of the leachate is analyzed for pH, electrical conductivity (EC), alkalinity, acidity and sulfate.

Low temperature tests use the same procedure except that the cell is operated in a cold room at 5°C. Room temperature tests operate in a typical temperature range of 18 to 22°C.

A second type of test was operated for three samples of rock representing the alignment of the permitted 1999 exploration adit. This rock is generally very weakly mineralized and contains low concentrations of sulfur and arsenic. For this reason, it was expected that very dilute solutions would be obtained from humidity cells, which would reflect the rapid flushing environment in the cells. The test used for these samples is a column design in which the leachate is continually re-circulated (with small weekly removals) to allow contaminants to accumulate in the leachate. Each column is constructed from 10 cm diameter PVC. The length is 76.3 cm filled to a depth of about 65 cm with 8 kg of sample. As the rock was very finely crushed it was already partially weathered prior to testing. The rock was rinsed repeatedly to recover readily soluble weathering products accumulated in storage. Four to five pore volumes were used. The rinse water was collected and monitored during flushing until electrical conductivity had stabilized. Two rinse batches were analyzed for metal concentrations.

The weekly leaching cycle consists of addition of 1 pore volume of water drawn from the leachate reservoir. Each week 125 mL of leachate was withdrawn and analyzed using the same procedures as the humidity cells. The tests were terminated after 20 weeks of operation.

Detailed operating procedures for each test are shown in Table 4

2.3.2.2 Tailings

Tailings are being tested using both humidity cells and columns. Humidity cells are being used to evaluate the primary weathering conditions occurring in tailings exposed directly to the atmosphere. The column tests evaluate the interaction of leachate produced by oxidation under atmospheric conditions with deeper less oxygenated tailings.

The humidity cell design follows that recommended by the MEND Program for tailings. Each cell is constructed from 20.25-cm diameter PVC pipe. The cell is 9.5 cm high and filled to a depth of 3.5 cm with 1 kg of sample. The operating conditions

are similar to the rock humidity cells. The main difference is that air is passed over the surface of the tailings rather than upward through the test material.

Initially, slicing of the cemented tailings samples was attempted to produce cubes of tailings with defined surface areas. However, the cubes fell apart shortly after testing was started in humidity cells. Subsequent samples of cemented materials (including simulated backfill) were gently broken up prior to being placed in the humidity cell.

Each tailings column was constructed from 6.25 cm diameter PVC and contains 3 kg of tailings to a depth of 74 cm. The samples were initially flushed with 2 to 3 pore volumes of water to remove readily soluble process additives and oxidation products. Each week, the samples are flushed with one pore volume of leachate. The leachate is analyzed using the same procedure as for rock humidity cells. A similar procedure was followed for the recently started crushed ore columns (Columns 9 to 12)

Detailed operating procedures for each test are shown in Table 4

2.3.3 Quality Control

Several levels of quality control have been initiated for the test program. These include:

- A blank humidity cell is being operated to evaluate the effect of the construction materials on the leach solutions.
- Two triplicates of humidity cells were operated on rock samples. The samples were selected based on the availability of sufficient test materials. One cell in each triplicate was terminated in March 2000 to evaluate the progress of weathering in samples (SRK 2000c). The remaining duplicates are continuing.
- Two duplicates of column tests on ore type materials are being operated. These samples are taken from ore batches used for Pilot Plant testing. They are not splits of a single sample.
- Two sets of triplicate room temperature flotation tailings humidity cells are being operated (F48 Sample).

- Triplicate room temperature flotation tailings columns are being operated (F48 Sample).
- Two sets of duplicate room temperature flotation tailings humidity cells are being operated (Phase 1 Pilot Plant)
- One triplicate of a cemented flotation and leach tailings backfill sample is being operated.

Replicates are shown in Table 1.

Leachate chemistry is evaluated by re-analysis of one sample in every batch of leachates submitted (about 1 in every 20 samples) and analysis of standard reference waste water samples. Trend analysis for the tests is also used to identify results to be re-analyzed.

3.0 PROGRESS RESULTS

3.1 Pre-Test Characterization

3.1.1 Waste Rock and Ore

Pre-test characterization results for waste rock samples in humidity cells are shown in Table 5.

The four samples of rock (Cells 43 to 46) extracted from the exploration adit were selected to provide a range of sulfur and arsenic concentrations. The range of sulfur concentrations is 0.08% to 0.89%, and the range of arsenic <2 to 360 µg/g (Table 5).

The three ore samples contained a relatively narrow range of sulfur concentrations (0.5 to 1%), but a wide range of arsenic concentrations (94 to 2910 µg/g) (Table 6). One sample (Cell 42) was potentially acid generating (NP/AP 0.3) and carbonate was undetectable.

The four columns containing crushed ore used for Pilot Plant testing had a wider range of sulfur concentrations (0.6 to 1.8%) and higher arsenic concentrations (1050 to 6790 µg/g) (Table 6). Three samples had NP/AP less than 1 and are therefore predicted to

be potentially acid generating primarily due to the higher sulfur concentrations compared to the three humidity cells.

3.1.2 Borrow Source Rock

Table 7 summarizes characteristics of the three borrow source samples. In the context of other rock from the project area, these samples contained low sulfur concentrations (0.04% to 0.14%) and moderate neutralization potential. Arsenic concentrations were also low (8 to 29 µg/g).

3.1.3 Flotation Tailings

Table 8 summarizes characteristics of flotation tailings samples.

The acid-base accounts for the F48 flotation tailings sample indicated very low total sulfur concentrations (<0.1%) (Cells 12, 13, 20, 21).

The composition of the four subsequent tailings samples (F95 to F98, cells 27 to 32)) was controlled to obtain a range of sulfur (from 0.06 to 0.36%) and arsenic (0.05 to 0.36%) concentrations. Neutralization potentials for all three samples were comparable (24 to 25 kg CaCO₃/t).

The Pilot Plant flotation tailings samples (Cells 50 to 53) had comparable sulfur concentrations to the bench scale sulfur concentrations. Neutralization potentials for all flotation tailings samples were very similar (22 to 30 kg CaCO₃/t).

The samples are non-potentially acid generating due to the extraction of sulfide minerals during flotation

3.1.4 Cemented Flotation Tailings

The four cemented flotation tailings samples all contained low sulfur concentrations with higher neutralization potentials than the tailings (Table 9). The neutralization potentials reflected the addition of cement, which contributes neutralization potential. The difference between NP and TIC increases approximately in proportion to the percentage of cement.

The samples were all non-potentially acid generating.

3.1.5 Simulated Backfill

The composition of these samples reflects the varying proportions of the different tailings and the different amounts of cement (Table 10). The higher proportion of leached tailings resulted in higher total sulfur concentrations but the increase in cement resulted in higher neutralization potential. The samples containing the higher proportion of leach tailings (>50%, Cells 33 and 35) were potentially acid generating ($1 \leq \text{NP/AP}$). The samples containing the lowest proportion of leach tailings and 5% cement (Cells 37, 38 and 39) had an NP/AP of 2.4 indicating low potential for acid generation

The uncemented and cemented leach tailings (Cells 26 and 33) contained 4.9% total sulfur and comparable neutralization potential to the tailings though part of this was due to lime addition. This material was potentially acid generating. Arsenic concentrations in the leach tailings exceeded 1%. Many elements are concentrated in the leach tailings, compared to the flotation tailings, including Ag, Bi, Co, Cr, Cu, Fe, Mo, Ni, Pb, Sb and Zn.

3.2 Kinetic Tests

3.2.1 Introduction

The ongoing kinetic results were presented by SRK (1999) and updated by SRK (2000b). This report focuses on the following specific aspects of the data:

- Reproducibility of results from triplicates and duplicates.
- The stability of long term trends to determine if reduction of monitoring frequency or termination of tests is appropriate.
- The relationships between bulk test material characteristics and leachate chemistry.
- Potential for acid generation (criteria for acid generating materials and delay to onset of acid generation).

For the purpose of data review, the leachate data received from the laboratory was reduced as follows:

- Less than detection limit values were converted to half the detection limit.
- Release rates were calculated for each parameter from the concentration and volume of leachate recovered. For non-detectable concentrations, release rates are only calculated for the lower detection limit (usually ICP-MS) to avoid biasing subsequent calculations of averages.
- Average release rates were calculated by summing total release over the period of the test and dividing by the duration of the test.
- The proportion of sulfur and carbonate leached was calculated from the initial bulk concentration and the total release of sulfur, calcium and magnesium.

The average release rates do not imply that these rates are observed throughout the tests. In many cases, release rates are greatest at the start of the test, then decrease. However, the average rates are a useful basis for comparison of different tests because relative rate differences tend to be maintained for tests of similar duration. For example, Tests A and B may show a range of sulfate release rates as the tests proceed, but Test A may always have a higher release rate than Test B.

Each aspect of the data is discussed below for each type of test material.

3.2.2 Waste Rock

The following discussion applies to the waste rock humidity cells. Results of the waste rock leach columns (1, 2 and 3) were presented previously by SRK (1999).

The majority of rock sample humidity cells have been running for more than 2½ years. The recent adit development rock cells have been running for a less than a year.

3.2.2.1 Reproducibility

The triplicates for sample 90400 (2, 22 and 23) showed excellent reproducibility. Average release rates for sulfate varied from 0.9 to 1.4 mg/kg/week and from 0.005 to 0.006 mg/kg/week for arsenic (Table 5). The average sulfate release rate decreases as the length of the test increases generally because release rates tend to be higher earlier in the test. The triplicates for sample 95832 (6, 24 and 25) showed similar average

release rates for sulfate (1.6 to 4.5 mg/kg/week), but again the higher rate was associated with the shortest duration test. A similar effect was apparent for arsenic (0.07 to 0.1 mg/kg/wk).

3.2.2.2 Stability of Trends

Except for Cell 1, leachate pHs have remained in the circum-neutral to slightly alkaline range (pH to 7 to 9) with most leachate pHs between 8 and 9 (Figure 2). This is consistent with leaching of carbonates in the absence of rapid sulfide mineral oxidation.

Cell 1 pH decreased to near 4 early in the test and appears to be stable in the pH range 3.7 to 4.6. The refrigerated equivalent of this Cell (10) has shown an erratic decline in leachate pH. The lowest pH reached has been 5.4, while sulfate release has remained steady and arsenic release has increased from typical low values of 0.004 mg/kg/week to 0.02 mg/kg/week as pH decreased.

Release trends for sulfate and arsenic (Figure 2) show that release rates are variable, but within a narrow range. The fundamental difference between the responses of different test materials is apparent despite this variability.

3.2.2.3 Bulk Chemistry and Release Rate Relationships

SRK (1999) showed a positive correlation between total sulfur concentration in the test material and sulfate release. A similar relationship is apparent in the longer term data (Figure 3). SRK (1999) explained some the variability by differences in the mineralogy of sulfur (pyrrhotite and pyrite). The four adit development rock humidity cells show a strong positive correlation but at a higher level than the longer term humidity cells. The difference may be due to comparison of different time frames (as shown by the triplicate tests, see Section 3.2.2.1).

A relationship between bulk arsenic concentration and arsenic release is also apparent (Figure 3), however, the relationship is not clearly positive. Arsenic release appears to increase as arsenic concentration increases, but the reverse appears to be true at lower arsenic concentrations. A similar relationship was apparent about a year ago and has not shifted significantly. The four adit development rock cells are part of the same trend which supports the predictions for the adit rock pile based on the ore zone waste rock humidity cells (SRK 2000a).

The triplicate cells (6, 24, 25) for 95823 lie well outside the general arsenic trend (Figure 3), showing the highest arsenic release under non-acidic conditions at a relatively low bulk arsenic concentration (142 µg/g). The arsenic in this sample occurred dominantly as löllingite rather than arsenopyrite or pyrite (SRK 2000c). The residue from Cell 24 lacked löllingite, suggesting that it oxidized rapidly early in the test.

The acidic leachate from Cell 1 showed the highest arsenic release of any of the tests. The refrigerated equivalent (Cell 10) showed order of magnitude lower arsenic release, though the pH is also not as low as the room temperature cell. The refrigerated pair Cells 9 and 11 showed a typical reduction in rates of both sulfate and arsenic release (about one third).

3.2.2.4 Potential for Acid Generation

To date, Cell 1 has produced acidic leachate (Figure 2). This cell contained low NP, primarily as carbonate, and 54% of the carbonate content has been leached (based on release of calcium and magnesium) (Table 5). The cell produced acidic leachate almost immediately, implying that the small amount of NP is actually not readily available to buffer acid. Based on this finding, other potentially acid generating cells (3, 4, 8) would not be expected to generate acid for many years because they contain much higher NP.

The relative rates of release of sulfate and acid neutralization indicators (calcium and magnesium) show conservatively whether sulfides will outlast neutralization potential. In kinetic tests on weakly reactive materials, carbonates in particular are leached far more rapidly than under field conditions due to the high water to solid ratio used in the tests. Under field conditions, carbonate solubility limits depletion of these minerals. Comparison of total sulfur concentration with the molar calcium and magnesium to sulfate rate ratio (CMSR, $(Ca+Mg)/SO_4$) shows that the ratio decreases as sulfur concentration increases (Figure 3). A series of samples with higher sulfur concentrations have a ratio of 1.3 to 1.4 at neutral to basic pH. Cell 3 has a ratio of 3, though the sulfate release rate is much lower than the other cells in the same group (4, 8, 43, 46). SRK (1999) estimated that the ratio was between 1.1 and 2 based on earlier data. The more recent data has therefore refined the conclusion that the theoretical criterion for acid generating rock for the project is probably NP/AP less than 1.4.

3.2.2.5 Waste Rock - Conclusions

The waste rock humidity cells now represent more than two years of testing. The current test results show that:

- reproducibility of the test work is good and trends are stable;
- conclusions drawn by SRK (1999) based on approximately 40 weeks of data have been confirmed and refined by the long term data; and
- trends observed currently will continue at least for several years.

The exception to the third conclusion is Cell 1, which may show a further pH change as additional carbonate is removed. The time frame for this transition is possibly one or two years.

3.2.3 Ore

3.2.3.1 Stability of Trends

Three ore humidity cells (40, 41 and 42) have been operating for about 42 weeks. The four columns (9, 10, 11, 12) have only been leaching for seven weeks and will not be discussed further in this report. The following comments apply to the humidity cells.

The pH of leachates and sulfate release rates are stable (Figure 4). Sulfate release stabilized after about 4 weeks. Arsenic release has taken longer to stabilize. Cells 40 and 42 appear to have stabilized. Arsenic release from Cell 41 was initially greater than the other two but decreased rapidly as the test proceeded. The arsenic release rate for this cell is currently comparable to Cell 42.

3.2.3.2 Bulk Chemistry and Release Rate Relationships

Bulk sulfur and arsenic concentrations are compared to sulfate and arsenic release in Figure 3. The sulfur concentrations are midway between the lower and higher sulfur concentrations of the waste rock humidity cells, and the release of sulfate also lies on the same trend, confirming the relationship observed for waste rock.

Two of the tests (Cells 40 and 42) showed average arsenic release on the same trend as the waste rock samples (Figure 3). The Cell 41 average is much higher than expected based on its bulk arsenic content. The test material was selected specifically because it

contains löllingite. The results are therefore consistent with the Sample 95832 triplicate (Cells 6, 24 and 25), which implied that löllingite appears to explain the greater arsenic release rates for these samples (SRK 2000).

3.2.3.3 Potential for Acid Generation

The sample in Cell 42 contained a sulfur concentration of 1%, however it is oxidizing relatively slowly (9 mgSO₄/kg/week) (Table 6). The sample contained no detectable carbonate (<0.2% CO₂) and NP/AP of less than 0.3. It would be expected to generate acidic leachate. The observed average rate of leaching of Ca and Mg indicates that at least 13% of carbonate neutralization potential has been leached (Table 6). Based on the performance of Cell 1, Cell 42 would have been expected to generate acid already. The lower oxidation rate is probably allowing the available carbonate neutralization potential to be effective.

The other two samples (Cell 40 and 41) had higher NP/AP and higher NP. They are not expected to generate acid. The CMSRs of 0.8 to 2.6 are consistent with the correlation observed for waste rock humidity cells, and the site specific NP/AP criterion for acid generation of less than 1.4. The low-end value of 0.8 is anomalous but implies an NP/AP criterion of less than 1.4.

3.2.3.4 Ore - Conclusions

The results of the three humidity cells are consistent with the waste rock tests. Based on the stability of results observed in the waste rock tests after about 1 year, it is expected that Cells 40 and 41 will not vary markedly. Cell 42 is expected to generate acidic leachate within less than a year of additional leaching.

3.2.4 Borrow Source

3.2.4.1 Stability of Trends

The three borrow source humidity cells (47, 48 and 49) have been operating for 32 weeks. Leachate pH stabilized almost immediately and sulfate stabilized from initially elevated release after nine weeks (Figure 5). Arsenic release increased for Cells 48 and 49 then decreased. After 17 weeks, these cells had similar arsenic release rates (0.02 to 0.03 mg/kg/week). Cell 47 has shown stable arsenic release near 0.01 mg/kg/week since the test started (Table 7).

Release rates for all other parameters have been low and stable.

3.2.4.2 Bulk Chemistry and Release Rate Relationships

Relationships between bulk total sulfur and arsenic, and release rates for sulfate and arsenic are similar to this relationships seen for the waste rock and ore cells (Figure 3). The low sulfate release rates are consistent with the low sulfur concentrations. Arsenic release rates are higher than expected based on other humidity cells containing similar low arsenic concentrations, though the longer term averages may be lower than currently observed. The general trend is however part of the negative correlation between bulk arsenic concentration and arsenic release at low bulk arsenic concentrations.

This negative correlation trend may be due to the lack of carbonate buffer in the less mineralized samples. This results in higher leachate pH due to hydrolysis of silicates, and greater mobility of arsenic. This may be an artificial effect produced by the greater flushing volume and stirring of the sample during each flush. The latter causes fresh silicate surfaces to be exposed, an effect that would not occur in a waste rock dump.

3.2.4.3 Potential for Acid Generation

The three samples have NP/AP greater than 1 and low sulfur concentrations indicating low potential for generation of acidic leachate. The CMSRs are between 2.5 and 10 (Figure 3, Table 7). These are consistent with the low sulfur concentrations and are indicative of leaching of carbonate during the leaching cycle, rather than as a result of acid generation.

3.2.4.4 Borrow - Conclusions

The short-term results of the tests are consistent with the other tests on waste rock, ore and adit development rock samples. Short-term arsenic release was more rapid than expected, but appears to be stabilizing at lower levels.

3.2.5 Flotation Tailings

3.2.5.1 Reproducibility

Based on 1 to 2 years of testing, two humidity cell triplicates indicate good reproducibility for F48 and F98 tailings samples. Sulfate release rates for the F48

triplicate were 5.3 to 7.7 mg/kg/week and for the F98 triplicate the range was 17 to 20 mg/kg/week (Table 8).

Similarly, the triplicate column test on F48 tailings (2 years duration), showed good reproducibility of pH, sulfate and arsenic concentrations (Figure 6).

3.2.5.2 Stability of Trends

All ten longer term humidity cells have consistently leachate with pH between 7.8 and 8.8 (Figure 7).

Sulfate typically showed initially rapid release, followed by a long term trend of stable release at low levels (less than 5 mg/kg/wk) (Figure 7). The exception is the F98 triplicate which showed an increase in sulfate release after about one year for all three cells. Peak release rate was 48 mg/kg/week. More recent rates are nearer 15 mg/kg/week.

Arsenic followed similar trends (Figure 7). The low sulfate release cells have showed stable arsenic release at about 0.02 mg/kg/week. The F98 samples showed an increase in arsenic release after about year to about 0.08 mg/kg/week. Unlike sulfate, these higher levels have continued. No other parameters show evidence of similar increases.

The column tests on F48 and F95 samples (Columns 5 to 8) showed stable trends in pH and sulfate concentrations established after about 32 weeks (Figure 7). Sulfate concentrations decreased initially and then remained at uniform levels. Arsenic concentrations in F48 leachate increased initially and then reached stable levels. The F95 column initially increased for a year to 0.6 mg/L, but concentrations are currently decreasing rapidly with current concentrations at 0.2 mg/L.

3.2.5.3 Bulk Chemistry and Release Rate Relationships

The ten bench scale tests showed a very strong positive relationship between total sulfur and sulfate release rate (Figure 8). The four recent pilot plant tailings sampled showed high average sulfate release rates, but these tests have been operating for only 15 weeks and the average sulfate release rates are biased by the initially high sulfate release occurring at the beginning of the test. The bench scale and pilot plant test results are therefore not directly comparable.

The relationship between bulk arsenic concentration and bulk arsenic release is not as well defined, partly because bulk arsenic concentrations cluster at two extremes (Figure 8). Nonetheless, the higher arsenic release for the F98 samples was correlated with the elevated bulk arsenic concentration. The difference in release rates was about 2.5:1 for 7-fold increase in bulk arsenic concentration.

3.2.5.4 Potential for Acid Generation

The flotation tailings contain low sulfur concentrations (0.06 to 0.36%) resulting in NP/AP > 2. The relationship between total sulfur and CMSR (Figure 8) showed the same general trends as the waste rock samples. As sulfur concentration increases, the release rate ratio decreases to 1.6. This indicates a theoretical NP/AP criterion for potentially acid generating tailings of less than 1.6, which would be equivalent to a total sulfur concentration of 0.5% assuming NP of 25 mg CaCO₃/t.

None of the flotation tailings samples have potential to generate low pH leachate.

3.2.5.5 Flotation Tailings - Conclusions

Testing of flotation tailings samples with less than 0.4% sulfur has established that:

- they will generate non-acidic leachate indefinitely;
- oxidation rates are correlated with sulfur concentrations.
- arsenic release is correlated with bulk arsenic concentration.'

Arsenic release was variable for the F98 tailings sample. The reason for the variability is uncertain.

3.2.6 Cemented Flotation Tailings

3.2.6.1 Stability of Trends

Leachate from the cemented flotation tailings cells is characterized by pH between 8 and 9 over the approximately 2 years of testing (Figure 9). Sulfate release was rapid initially for the three F48 samples, then decreased to low levels with several erratic spikes. The spikes may be due to breakdown of the cemented material, which has occurred continually as the tests have progressed. Typical sulfate release rates are less than 4 mg/kg/week. The 3% cemented sample had the lowest sulfate release rate. The

FXX sample with 5% cement has shown relatively stable sulfate release. Arsenic release initially increased erratically for all cells but then decreased.

3.2.6.2 Bulk Chemistry and Release Rate Relationships

Sulfate release for the samples containing 1% and 2% sulfur was comparable to uncemented tailings samples with similar sulfur concentrations, but lower than the uncemented tailings for the higher cement amounts of 3% and 5% (Figure 8), implying that the cement reduce the rate of oxidation.

The cemented samples released arsenic much more rapidly than the uncemented tailings. Typical average arsenic release rates for cemented tailings were between 0.1 and 0.3 mg/kg/week, compared to 0.02 mg/kg/week for the same uncemented tailings (Figure 8).

3.2.6.3 Potential for Acid Generation

The cemented flotation tailings have low sulfur concentrations combined with carbonate NP and NP provided by the cement. The NPs exceeded the APs by a wide margin ($NP/AP > 20$), therefore cemented flotation tailings have no potential for acid generation. Calculated depletion of neutralization potential as carbonate and depletion of sulfur indicates that sulfur is depleting more rapidly than neutralization potential, confirming that the cemented flotation tailings will not generate acid.

3.2.6.4 Cemented Flotation Tailings - Conclusions

The principal conclusion from these tests is that arsenic is released more rapidly from cemented tailings than from uncemented equivalent samples. The difference in rates is about an order of magnitude.

3.2.7 Simulated Backfill

3.2.7.1 Reproducibility

Reproducibility of these tests has been investigated with a triplicate sample containing 80% flotation tailings, 20% leach tailings and 5% cement. The tests have been running for 11 months. Reproducibility of trends has been excellent. Leachate pH has increased for all three cells from 7.6 to 8.8 (Figure 10). Sulfate release initially peaked at more than 400 mg/kg/week, then decreased to current levels of less than 100

mg/kg/week. Arsenic release is continuing to increase. Current release rates are greater than 1 mg/kg/week.

3.2.7.2 Stability of Trends

The tests containing dominantly flotation tailings (Cells 36, 37, 38 and 39) initially showed rapid sulfate release that can probably be attributed to flushing of sulfate from the sample (Figure 10). Calculation of sulfur depletion indicates that most soluble sulfate has probably been flushed and that sulfate release is now controlled by sulfide oxidation.

The cemented cells containing more than 20% leached tailings (Cell 33 and 35) showed higher initial sulfate release peaks (up to 1500 mg/kg/week) followed by decreasing sulfate release (Figure 10). The test containing equal amounts of each type of tailings (Cell 35) appears to have stabilized, though at a higher sulfate release rate (between 200 to 300 mg/kg./week. Soluble sulfate has also been flushed from these tests. The arsenic release rates for both Cell 33 and 35 are increasing.

The uncemented leach tailings sample (Cell 26) has stable pH, and sulfate and arsenic release.

3.2.7.3 Bulk Chemistry and Release Rate Relationships

The cemented tailings showed positive correlations between sulfur content and sulfate release, and arsenic content and arsenic release (Figure 8). Both relationships appear to indicate that cemented samples are more reactive than their uncemented equivalents. This is particularly apparent for arsenic. The higher release rates found for the cemented flotation tailings are on the same trend as the cemented mixtures, and pure cemented leach tailings. A similar general trend is apparent at a lower level for the uncemented tailings.

3.2.7.4 Potential for Acid Generation

The presence of significant amounts of sulfur in the form of sulfate (probably calcium sulfate) makes interpretation of depletion of neutralization potential more complex than for the other tests. Cell 26 contained the lowest sulfur as sulfate concentration (0.04%) and this is probably entirely depleted, as shown by depletion of 0.3% sulfur (Table 10). Leaching of calcium and magnesium in this case probably represents removal of carbonate. The depletion calculation indicates that all carbonate has been

removed. This test is expected to start generating acidic leachate. The calcium and magnesium to molar sulfate release rate ratio was 1.4 on average. Based on this criterion (which is similar to all other tests), Cells 33 and 34 would be expected to generate acidic leachate (NP/AP of 0.7 and 1.0, respectively). Cell 33 is oxidizing rapidly, and 76% of carbonate has been depleted (Table 10). This suggests that this material may generate acidic leachate in the next year. Cell 34 is oxidizing less rapidly. Depletion of carbonate is not expected to occur for several years, unless the sample begins to oxidize more rapidly.

The remaining cells are not expected to generate acidic leachate

3.2.7.5 Simulated Backfill - Conclusions

The simulated backfill samples have shown that:

- After one year of testing, the sulfate release rate is stable but the arsenic release rate is increasing.
- Cement appears to increase the rate of arsenic release when compared to similar uncemented samples.
- Cement may also increase iron sulfide oxidation.
- Based on the tailings mixtures tested, mixtures containing greater than 20% leached tailings are potentially acid generating.
- Potentially acid-generating mixtures may be expected to generate acidic leachate after a few years of weathering.

4.0 CONCLUSIONS

Testing on a variety of materials representing the types of waste materials likely to be generated by the Pogo Project show a number of consistent results:

- Replicates of tests show that trends and average release rates are comparable.
- Correlation between bulk chemical characteristics and release rates for sulfate and arsenic are strong and allow release rates to be predicted from bulk chemistry.
- Rock or tailings with NP/AP less than 1.4 can theoretically be expected to generate acid. To date, this conclusion has been confirmed by one test.

- Release rates at lower (need freezing) temperatures are lower than, or comparable to release rates at room temperature.
- Addition of cement increases sulfate and arsenic release.
- Tests greater than one year in duration have generally confirmed findings after one year of testing (with some exceptions) indicating that many of the longer term tests do not need to be continued.

5.0 RECOMMENDATIONS

Based on the last general conclusion indicated above, the test program has been reviewed to determine if any of the tests can be terminated. The criteria used to select tests for continuation are as follows:

- The test was started recently (ie much less than one year).
- Leaching of significant contaminants (primarily arsenic) has not stabilized.
- A short-term (less than a year) transition (ie to lower pH) can be expected that would have a significant effect on leachate chemistry.
- Long term monitoring of selected typical samples is beneficial as mining starts and ongoing closure planning occurs.

Candidates for termination can include:

- Stable release rates and insignificant depletion of significant mineralogical components (sulfides, carbonates).
- Replicate tests for which reproducibility has been demonstrated.
- Tests on materials that are no longer relevant to the mine plan.

Table 11 provides recommendations and reasoning for each test. No recommendations have been made to terminate tests based on relevance to the mine plan.

Test residues should be analyzed using same procedures used to characterize the starting material. Mineralogical examination using SEM is also recommended to identify weathering features and secondary minerals.

This report, 1CT002.00, **Third Progress Report on Kinetic Geochemical Tests** has been prepared by:

STEFFEN, ROBERTSON AND KIRSTEN (CANADA) INC.



Feb 20/01

Stephen Day, M.Sc., P.Geo.

Principal Geochemist

6.0 REFERENCES AND SUPPORTING DOCUMENTS

CANMET, 1998, Image Analysis and Mineralogical Distribution of Au in Twelve Samples from the Pogo Deposit, Prepared by Mining and Mineral Science Laboratories. June 1998.

Coastech Research, 1991, Acid Rock Drainage Prediction Manual, MEND Project Report 1.16.1b, MEND, Ottawa, Ontario.

Nevada Mining Association, 1996, Meteoric Water Mobility Procedure (MWMP), Standardized Column Percolation Test Procedure, Nevada Mining Association, Reno, NV, 5p.

Norecol, Dames & Moore, 1998, Prediction of Acid Rock Drainage and Metal Leaching Potential Exploration Adit, Pogo Project, Alaska. Report 28497-004.

Sobek A A, Schuller W A, Freeman J R, and Smith R M., 1978, Field and laboratory methods applicable to overburden and minesoils. USEPA Report No. 600/2-78-054, 203 pp.

SRK Consulting, 1999, Progress Report on Kinetic Geochemical Tests – Pogo Project. Report 1CT002.00. July 1999.

SRK Consulting 2000a, Prediction of Arsenic Concentrations in Drainage from Development Adit Rock Piles, Pogo Project, Alaska. Letter prepared for Teck Corporation, February 7, 2000.

SRK Consulting, 2000b, Second Progress Report on Kinetic Geochemical Tests – Pogo Project. Report 1CT002.00. May 2000.

SRK Consulting, 2000b, Mineralogy of Column Leach Residues, Tailings and Waste Rock, Pogo Project, Alaska. Report U15555. July 2000.

Teck- Pogo Inc. 2000. Pogo Project Water Management Plan. Volume of Pogo Project Project Documentation.

U.S. Environmental Protection Agency, 1996, Test Methods for Evaluating Solid Waste - Physical/Chemical Methods (SW-846), USEPA, Washington, DC.

TABLES

TABLE 1
Test Program Samples and Test Types

Source	Product	Variants	Sub-variant	Tests									
				Room				Fridge					
				HC	Test Nos	Col (Aerial)	Test Nos	HC	Test Nos	Col	Test Nos		
Mine	Waste Rock	Gneiss		1	<u>1</u>				1	<u>10</u>			
		Gneiss		3	<u>2, 22, 23</u>								
		Granodiorite		1	3								
		Gneiss		1	4								
		Altered Rock		1	5								
		Granodiorite		3	<u>6, 24, 25</u>								
		Gneiss		1	7								
		Altered Rock		1	8								
		Altered Rock		1	9					1	<u>11</u>		
		Adit Pre-development						3	<u>1, 2, 3</u>				
		Adit Development Rock	Mineralized		2	<u>43, 46</u>							
			Unmineralized		2	<u>44, 45</u>							
		Ore	Core		1	<u>40, 41, 42</u>							
		Phase 1 Crushed Ore				4	<u>9, 10, 11, 12</u>						
Borrow Quarry	Borrow Rock	Diorite		3	<u>47, 48, 49</u>								
Processing	Flotation tailings	F48		3	<u>13, 20, 21</u>	3	<u>4, 5, 6, 7</u>	1	<u>12</u>	1	<u>5</u>		
		F95		1	27	1	<u>8</u>						
		F96		1	28								
		F97		1	29								
		F98		3	<u>30, 31, 32</u>	1							
		Pilot Plant		4	<u>50, 51, 52, 53</u>	4	<u>13, 14, 15, 16</u>						
		Cemented flotation tailings	F48	1% Cement	1	<u>14</u>							
			2% Cement	1	<u>15</u>								
			3% Cement	1	<u>18</u>								
	Backfill	100% Leached	0% Cement	1	<u>26</u>								
		100% Leached	10% Cement	1	<u>33</u>								
		50% Leached	5% Cement	1	<u>35</u>								
		20% Leached	2% Cement	1	<u>36</u>								
			5% Cement	3	<u>37, 38, 39</u>								
		100% Float	5% Cement	1	<u>34</u>								

Notes

HC - Number of Humidity Cells

Col - Number of Columns

Italics indicate replicate tests

Test Nos - Test designations, see Table 3 for test construction details

Underlined and bold indicates tests ongoing. Other tests stopped

TABLE 3
Kinetic Test Leachate Analysis Detection Limits

Parameter	Unit	ICP-MS	ICP-ES	Parameter	Unit	ICP-MS	ICP-ES
pH		N/A	N/A	Lu	µg/L	0.01	-
Cond.	µmhos/cm	1	1	Mg	µg/L	50	100
Sulphate	mg/L	1	1	Mn	µg/L	0.1	10
Acidity	mgCaCO ₃ /L	1	1	Mo	µg/L	0.1	5
Alkalinity	mgCaCO ₃ /L	1	1	Na	µg/L	50	100
Temp	C	N/A	N/A	Nb	µg/L	0.01	-
Ag	µg/L	0.05	5	Nd	µg/L	0.01	-
Al	µg/L	1	100	Ni	µg/L	0.2	20
As	µg/L	1	30	Os	µg/L	0.05	-
Au	µg/L	0.05	-	P	µg/L	50	20
B	µg/L	10	20	Pb	µg/L	2	10
Ba	µg/L	0.05	20	Pd	µg/L	0.05	-
Be	µg/L	0.05	2	Pr	µg/L	0.01	-
Bi	µg/L	0.05	20	Pt	µg/L	0.01	-
Br	µg/L	5	-	Rb	µg/L	0.01	-
Ca	µg/L	50	100	Re	µg/L	0.01	-
Cd	µg/L	0.05	2	Rh	µg/L	0.01	-
Ce	µg/L	0.01	30	Ru	µg/L	0.05	-
Cl	mg/L	0.1	-	Sb	µg/L	0.05	10
Co	µg/L	0.02	5	Sc	µg/L	0.05	20
Cr	µg/L	0.5	20	Se	µg/L	0.05	20
Cs	µg/L	0.01	-	Si	µg/L	#N/A	20
Cu	µg/L	1	2	Sm	µg/L	0.05	-
Dy	µg/L	0.01	-	Sn	µg/L	0.05	-
Er	µg/L	0.01	-	Sr	µg/L	0.01	-
Eu	µg/L	0.01	-	Ta	µg/L	0.05	-
Fe	µg/L	50	10	Tb	µg/L	0.01	-
Ga	µg/L	0.05	-	Te	µg/L	0.05	10
Gd	µg/L	0.01	-	Th	µg/L	0.05	-
Ge	µg/L	0.05	-	Ti	µg/L	10	10
Hf	µg/L	0.02	-	Tl	µg/L	0.01	10
Hg	µg/L	1	-	Tm	µg/L	0.01	-
Ho	µg/L	0.01	-	U	µg/L	0.05	-
I	µg/L	1	-	V	µg/L	1	10
In	µg/L	0.01	-	W	µg/L	0.1	10
Ir	µg/L	0.05	-	Y	µg/L	0.01	-
K	µg/L	50	100	Yb	µg/L	0.01	-
La	µg/L	0.01	-	Zn	µg/L	0.5	5
Li	µg/L	50	50	Zr	µg/L	0.5	-

Notes

"-" - Not analyzed

"N/A" - Detection limit not applicable

TABLE 4
Summary of Tests - Construction and Operation Details

LAB ID	Sample ID (ODH and Interval for Rock Samples)	Sample Type	Column Dimensions		Space Above Sample	Sample Wt. kg	Column Material	Column Packing	Initial Flushing ml	Flushing Rate/ Wash-Inst ^a ml	Temperature °C	Sampling Frequency	Start Date m/d/yyyy	Duration weeks	Sampling Day	Procedure	Sample prep for flushings
			Diaper	Length													
HUMIDITY CELLS																	
1	88015 (95-08 185-190)	Waste Rock	11.0	17.0	11.0	1.0	1.0 Nivex Mesh	Plexx	750	500	18-22	weekly	7/30/1998	12-28-2000	Thursday	MEND	stirred after water addition
2	90430 (96-37 112-721)	Waste Rock	11.0	17.0	11.0	1.0	1.0 Nivex Mesh	Plexx	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
3	90430 (96-37 112-721)	Waste Rock	11.0	17.0	11.0	1.0	1.0 Nivex Mesh	Plexx	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
4	95784 (96-36 627-681)	Waste Rock	11.0	17.0	11.0	1.0	1.0 Nivex Mesh	Plexx	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
5	95784 (96-36 627-681)	Waste Rock	11.0	17.0	11.0	1.0	1.0 Nivex Mesh	Plexx	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
6	95821 (96-37 1012-1022)	Waste Rock	11.0	17.0	11.0	1.0	1.0 Nivex Mesh	Plexx	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
7	95821 (96-37 1012-1022)	Waste Rock	11.0	17.0	11.0	1.0	1.0 Nivex Mesh	Plexx	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
8	95849 (96-32 212-5215.5)	Waste Rock	11.0	17.0	11.0	1.0	1.0 Nivex Mesh	Plexx	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
9	95849 (96-32 212-5215.5)	Waste Rock	11.0	17.0	11.0	1.0	1.0 Nivex Mesh	Plexx	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
10	88016 (95-08 185-190)	Waste Rock	11.0	17.0	11.0	1.0	1.0 Nivex Mesh	Plexx	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
11	88016 (95-08 185-190)	Waste Rock	11.0	17.0	11.0	1.0	1.0 Nivex Mesh	Plexx	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
12	88016 (95-08 185-190)	Waste Rock	11.0	17.0	11.0	1.0	1.0 Nivex Mesh	Plexx	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
13	F48 Comb. Tail.	Flotation Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	dry cake broken before water addition
14	F48 Comb. Tail.	Flotation Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	dry cake broken before water addition
15	F48 Comb. Tail.	Flotation Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	dry cake broken before water addition
16	F48 Comb. Tail.	Flotation Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	dry cake broken before water addition
17	F48 Comb. Tail.	Flotation Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	dry cake broken before water addition
18	F48 Comb. Tail.	Flotation Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	dry cake broken before water addition
19	F48 Comb. Tail.	Flotation Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	dry cake broken before water addition
20	F48 Comb. Tail.	Flotation Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	dry cake broken before water addition
21	F48 Comb. Tail.	Flotation Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	dry cake broken before water addition
22	90400 (96-37 112-721)	Waste Rock	11.0	17.0	11.0	1.0	1.0 Nivex Mesh	Plexx	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
23	90400 (96-37 112-721)	Waste Rock	11.0	17.0	11.0	1.0	1.0 Nivex Mesh	Plexx	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
24	95821 (96-37 1012-1022)	Waste Rock	11.0	17.0	11.0	1.0	1.0 Nivex Mesh	Plexx	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
25	95821 (96-37 1012-1022)	Waste Rock	11.0	17.0	11.0	1.0	1.0 Nivex Mesh	Plexx	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
26	CR Composite	Flotation Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
27	CR Composite	Flotation Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
28	CR Composite	Flotation Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
29	F48 Comb. Tail.	Flotation Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
30	F48 Comb. Tail.	Flotation Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
31	F48 Comb. Tail.	Flotation Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
32	F48 Comb. Tail.	Flotation Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
33	100% Wash, 10% wt. -7	Comminuted Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
34	100% Wash, 10% wt. -7	Comminuted Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
35	50% F120% Wash, 5% wt. -7	Comminuted Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
36	50% F120% Wash, 5% wt. -7	Comminuted Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
37	50% F120% Wash, 5% wt. -7	Comminuted Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
38	50% F120% Wash, 5% wt. -7	Comminuted Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
39	50% F120% Wash, 5% wt. -7	Comminuted Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
40	97-43 (leached-BCR1)	Ore	11	17	11	1.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
41	97-43 (leached-BCR1)	Ore	11	17	11	1.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
42	97-41A (leached-BCR1)	Ore	11	17	11	1.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
43	N332131 (Underground Ramp B03)	Waste Rock	11	17	11	1.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
44	N332131 (Underground Ramp B03)	Waste Rock	11	17	11	1.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
45	N332131 (Underground Ramp B03)	Waste Rock	11	17	11	1.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
46	N332131 (Underground Ramp B03)	Waste Rock	11	17	11	1.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
47	N332131 (Underground Ramp B03)	Waste Rock	11	17	11	1.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
48	LD.3 Composite	Waste Rock	11	17	11	1.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
49	LD.4 Composite	Waste Rock	11	17	11	1.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	stirred after water addition
50	102 A	Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Thursday	MEND	dry cake broken before water addition
51	102 B	Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Monday	MEND	dry cake broken before water addition
52	102 A	Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Monday	MEND	dry cake broken before water addition
53	102 B	Tailings	20.25	9.5	6.0	6.0	1.0 Nivex Mesh	PVC	750	500	18-22	weekly	7/30/1998		Monday	MEND	dry cake broken before water addition

LAB ID	Sample ID	Sample Type	Diaper	Length	Space Above Sample	Sample Wt. kg	Column Material	Column Packing	Initial Flushing ml	Flushing Rate/ Wash-Inst ^a ml	Temperature °C	Sampling Frequency	Start Date m/d/yyyy	Duration weeks	Sampling Day	Procedure	Sample prep for flushings
COLUMNS																	
1	98-80 (19-119)	Waste Rock	10.0	26.3	16.0	1.0	8 Nivex Mesh	PVC	9000	1700	16-23	weekly	11/24/1998	20	Monday	Reconditioned Water	
2	98-81 (403-503)	Waste Rock	10.0	26.3	16.0	1.0	8 Nivex Mesh	PVC	9000	1700	16-23	weekly	11/24/1998	20	Monday	Reconditioned Water	
3	98-82 (198-651)	Flotation tailings	6.3	91.5	17.5	1.0	3 Nivex Mesh	PVC	2000	2000	16-23	weekly	11/24/1998	60	Thursday	Weekly Input	
4	F48	Flotation tailings	6.3	91.5	17.5	1.0	3 Nivex Mesh	PVC	2000	2000	16-23	weekly	11/24/1998	60	Thursday	Weekly Input	
5	F48	Flotation tailings	6.3	91.5	17.5	1.0	3 Nivex Mesh	PVC	2000	2000	16-23	weekly	11/24/1998	60	Thursday	Weekly Input	
6	F48	Flotation tailings	6.3	91.5	17.5	1.0	3 Nivex Mesh	PVC	2000	2000	16-23	weekly	11/24/1998	60	Thursday	Weekly Input	
7	F48	Flotation tailings	6.3	91.5	17.5	1.0	3 Nivex Mesh	PVC	2000	2000	16-23	weekly	11/24/1998	60	Thursday	Weekly Input	
8	PV3	Ore	6.25	91.5	17.5	1.0	3 Nivex Mesh	PVC	1460	400	16-23	weekly	12/5/2000	3	Thursday	Weekly Input	
9	100A	Ore	6.25	91.5	17.5	1.0	3 Nivex Mesh	PVC	1460	400	16-23	weekly	12/5/2000	3	Thursday	Weekly Input	
10	100B	Ore	6.25	91.5	17.5	1.0	3 Nivex Mesh	PVC	1460	400	16-23	weekly	12/5/2000	3	Thursday	Weekly Input	
11	103A	Ore	6.25	91.5	17.5	1.0	3 Nivex Mesh	PVC	1460	400	16-23	weekly	12/5/2000	3	Thursday	Weekly Input	
12	103B	Ore	6.25	91.5	17.5	1.0	3 Nivex Mesh	PVC	1460								

TABLE 5
Waste Rock Test Material Characteristics and Selected Outcomes

Cell / Column	Sample ID	Start Date	Date Stopped	Rock Type	Mineralogy			TIC	AP	NP	NNP	NF/AP	AQUA REGIA DIGESTIBLE ELEMENTS			RATES (mg/kg/yr)			DEPLETION SUMMARY				
					Po	Py	As/Py						Carb	As	ppm	SO ₄	As	As	Duration Days	TIC Depleted kg/t	%	Car/Mg/SO ₄	%
1	88016	30-Jul-98	#N/A	Gneiss	0.7	0	Tr	0	4.3	25.3	5.7	-19.6	0.2	1477	22.4	0.135	1065	3	62	0.8	0.11	13	
10	88016	30-Jul-98	#N/A	Gneiss	0.7	0	Tr	0	4.3	25.3	5.7	-19.6	0.2	1477	10.9	0.009	1065	2	40	1.0	0.06	6	
2	90400	30-Jul-98	#N/A	Gneiss	0	0.5	Tr	0	6.1	6.3	7.4	1.2	36.3	0.9	0.005	1065	1	21	8.7	0.00	2		
22	90400	25-Mar-99	9-Mar-00	Gneiss	0	0.5	Tr	0	6.1	6.3	7.4	1.2	36.3	1.4	0.005	350	0	6	5.5	0.00	1		
23	90400	25-Mar-99	#N/A	Gneiss	0	0.5	Tr	0	6.1	6.3	7.4	1.2	36.3	1.2	0.006	827	1	14	5.8	0.00	2		
3	91737	30-Jul-98	30-Sep-99	Granodiorite	0.5	Tr	0	Tr	8.6	21.6	21.8	0.2	32.8	6.9	0.004	427	1	15	3.0	0.01	2		
4	95761	30-Jul-98	30-Sep-99	Gneiss	0.7	0.5	0	1	7.8	1.15	26.2	-9.1	0.7	54.7	40.7	0.003	427	4	13	1.4	0.08	7	
5	95786	30-Jul-98	30-Sep-99	Altered Rock	Tr	0.3	Tr	2	8.8	0.66	1.9	41.3	39.4	29.9	3.3	0.008	427	1	4	6.9	0.01	11	
6	95832	30-Jul-98	#N/A	Granodiorite	0	0	0	0	18.8	3.8	37.1	33.4	9.9	142	3.8	0.066	1065	3	17	5.3	0.02	16	
24	95832	25-Mar-99	9-Mar-00	Granodiorite	0	0	0	0	18.8	3.8	37.1	33.4	9.9	142	4.5	0.116	350	1	5	4.2	0.01	6	
25	95832	25-Mar-99	#N/A	Granodiorite	0	0	0	0	18.8	3.8	37.1	33.4	9.9	142	1.6	0.060	827	3	14	12.6	0.01	5	
7	8844	13-Aug-98	30-Sep-99	Gneiss	Tr	Tr	Tr	Tr	7.7	3.0	15.3	10.3	3.1	648.2	1.5	0.006	413	0	5	4.4	0.00	2	
8	92570	13-Aug-98	#N/A	Altered Rock	0	5.5	0.7	0.5	17.3	91.9	10.4	-81.3	0.1	7432.9	15.6	0.094	1051	3	19	1.3	0.08	3	
9	89549	17-Sep-98	30-Sep-99	Altered Rock	0	0.2	0.4	6.5	82.4	10.3	51.8	41.5	5.0	4311	1.4	0.022	378	1	1	8.7	0.00	1	
11	89549	17-Sep-98	#N/A	Altered Rock	0	0.2	0.4	6.5	82.4	10.3	51.8	41.5	5.0	4311	0.5	0.007	1016	1	2	19.3	0.00	1	
43	N332131	1-May-00	#N/A	Gneiss	-	-	-	-	31.8	27.8	39	11.2	1.4	360	39.6	0.002	424	4	11	1.4	0.08	9	
44	N332215	1-May-00	#N/A	Gneiss	-	-	-	-	18.2	2.5	24	21.5	9.6	6	6.0	0.008	424	1	7	3.3	0.01	15	
45	N332229	1-May-00	#N/A	Gneiss	-	-	-	-	158.9	5.0	153	148.0	30.6	28	7.5	0.009	424	1	1	2.8	0.02	9	
46	N332508	1-May-00	#N/A	Gneiss	-	-	-	-	13.6	15.6	20	4.4	1.3	360	23.6	0.002	424	2	15	1.3	0.05	10	

Notes
Cells 10 and 11 are refrigerated equivalents of Cells 1 and 9, respectively.
Cells 2, 22 and 23 are triplicates
Cells 6, 24 and 25 are triplicates
Cells 3, 4, 5, 7 and 9 were stopped on September 30, 1999
Cells 22 and 24 were shutdown on March 9, 2000

TABLE 6
Ore Test Material Characteristics and Selected Outcomes

Cell /Column	Sample ID	Start Date	ACID-BASE ACCOUNTING										DEPLETION SUMMARY											
			Mineralogy			Faste pH	S %	S-SO ₂ %	S-S ²⁻ %	TIC CO ₂ %	kg CaCO ₃ /t			NP/AP	AQUA REGIA DIGESTIBLE		RATES (mg/kg/wk)		Duration Days	TIC Depleted kg/t	% S Depleted	% of Total		
			Po	Py	Aspy						Lo	Cp	As		ppm	SO ₂	As	Ca+Mg/SO ₂						
40	97-43	4/3/2000	0.84	0.07	0.07	0.01	8.3	0.49	<0.01	#N/A	1.0	22.7	15.3	26	10.7	1.7	94	10.0	0.002	452	2	8	0.02	4
41	97-50	4/3/2000	0.26	0.01	3.04	1.61	9.0	0.30	<0.01	#N/A	1.4	31.8	9.4	44	34.6	4.7	486	9.1	0.031	452	0	1	0.8	7
42	97-41A	4/17/2000	0.23	1.96	1.41	0.07	8.5	0.97	<0.01	#N/A	<0.2	-4.5	30.3	10	-20.3	0.3	2910	8.8	0.023	438	1	>18	1.4	2
/9	100A	12/5/2000	#N/A	###	#N/A	#N/A	8.3	1.15	<0.01	#N/A	1.5	36.3	35.9	24	-11.9	0.7	1050							
/10	100B	12/5/2000	#N/A	###	#N/A	#N/A	8.5	0.59	0.01	#N/A	1.4	31.8	18.4	24	5.6	1.3	1980							
/11	103A	12/5/2000	#N/A	###	#N/A	#N/A	8.1	1.52	<0.01	#N/A	1.2	27.2	47.5	20	-27.5	0.4	6790							
/12	103B	12/5/2000	#N/A	###	#N/A	#N/A	8.1	1.79	<0.01	#N/A	1.4	31.8	55.9	17	-38.9	0.3	1570							

Notes:
 Mineralogy: Po - pyrrhotite, py - pyrite, Aspy - arsenopyrite, Lo - lollingite, cp - chalcopyrite, carb - carbonates
 ABA: S - total sulfur, S-SO₂ - Sulfur as sulfate, S-S²⁻ - Sulfur as sulfide (direct measurement), TIC - Total Inorganic Carbon
 AP - Acid Potential, NP - Neutralization Potential, NNP - Net Neutralization Potential = NP - AP
 Mineralogy reported by CANMET (1998)

TABLE 7
Borrow Rock Test Material Characteristics and Selected Outcomes

Cell	Sample ID	Start Date	Paste pH	ACID-BASE ACCOUNTING											AQUA REGIA DIGESTIBLE					RATES (mg/kg/wk)					DEPLETION SUMMARY				
				S %	S-SO ₄ %	S-S ²⁻ %	TIC CO ₂ %	TIC AP kg CaCO ₃ /t	NP	NNP	TIC/AP	As ppm	SO ₄	As	Duration Days	TIC Depleted kg/t	% of Total	Carb/Mg/SO ₄	%	S Depleted %									
POGO 47	LD1 Comp	7/6/2000	#N/A	0.07	#N/A	#N/A	0.8	62.5	2.0	IP	#N/A	#N/A	30.6	19.8	1.7	0.01	358	1	10.0	0.00	4								
POGO 48	LD2 Comp	7/6/2000	#N/A	0.14	#N/A	#N/A	0.1	8.0	4.5	IP	#N/A	#N/A	1.8	28.6	4.7	0.04	358	1	2.5	0.01	6								
POGO 49	LD3 Comp	7/6/2000	#N/A	0.04	#N/A	#N/A	0.1	11.2	1.2	IP	#N/A	#N/A	9.1	7.5	3.0	0.03	358	1	4.2	0.01	13								

Mineralogy: Po - pyrrhotite, py - pyrite, Aspy - arsenopyrite, Lo - lollingite, cp - chalcopyrite, carb - carbonates
 ABA: S - total sulfur, S-SO₄ - Sulfur as sulfate, S-S²⁻ - Sulfur as sulfide (direct measurement), TIC - Total Inorganic Carbon
 AP - Acid Potential, NP - Neutralization Potential, NNP - Net Neutralization Potential = TIC - AP
 IP - Analysis in progress

TABLE 8
Flotation Tailings Test Material Characteristics and Selected Outcomes

Cell / Column	Sample ID	Start Date	Stopped Date	ACID-BASE ACCOUNTING										AQUA REGIA DIGESTIBLE					RATES (mg/kg/wk)					DEPLETION SUMMARY				
				Mineralogy			Paste pH	S %	TIC CO ₂ %	TIC	AP	NP	NNP	NP/AP	As ppm	SO ₄	As	Duration Days	kg/t	TIC Depleted % of Total	Ca+Mg/SO ₄	S Depleted %	% of Total					
				Po	Py	Aspy																		Carb	SO ₄	As	Ca+Mg/SO ₄	S Depleted %
12/5	F43	21-Jan-99	#N/A	Tr	Tr	Tr	1.2	8.3	0.08	0.3	25	2.5	30	28	12.0	628	4.1	0.01	890	2	8	3.9	0.02	22				
13/4	F48	21-Jan-99	#N/A	Tr	Tr	Tr	1.2	8.3	0.08	0.3	25	2.5	30	28	12.0	628	5.6	0.02	890	2	8	2.7	0.02	30				
20/6	F48	18-Mar-99	2-Mar-00	Tr	Tr	Tr	1.2	8.3	0.08	0.3	25	2.5	30	28	12.0	628	7.7	0.02	330	1	3	1.8	0.01	16				
21/7	F48	18-Mar-99	#N/A	Tr	Tr	Tr	1.2	8.3	0.08	0.3	25	2.5	30	28	12.0	628	5.3	0.02	834	2	8	2.9	0.02	27				
27/8	F95	14-Jun-99	#N/A	#N/A	#N/A	#N/A	#N/A	7.9	0.06	0.4	29	1.9	24	22	12.8	540	5.8	0.02	746	2	5	2.5	0.02	34				
28	F96	14-Jun-99	#N/A	<0.1	<0.1	2.5		7.9	0.12	0.3	25	3.8	25	21	6.7	770	10.1	0.02	746	2	9	1.9	0.04	30				
29	F97	14-Jun-99	#N/A	#N/A	#N/A	#N/A	#N/A	8.0	0.18	0.3	25	5.6	25	19	4.4	1200	13.1	0.02	746	2	10	1.7	0.05	26				
30	F98	14-Jun-99	#N/A	#N/A	#N/A	#N/A	#N/A	8.0	0.36	0.3	25	11.3	25	14	2.2	3600	17.3	0.04	746	3	12	1.5	0.06	17				
31	F98	14-Jun-99	#N/A	#N/A	#N/A	#N/A	#N/A	8.0	0.36	0.3	25	11.3	25	14	2.2	3600	17.1	0.05	746	3	12	1.6	0.06	17				
32	F98	14-Jun-99	#N/A	#N/A	#N/A	#N/A	#N/A	8.0	0.36	0.3	25	11.3	25	14	2.2	3600	20.1	0.04	746	3	13	1.5	0.07	20				
50/13	102A	2-Nov-00	#N/A	#N/A	#N/A	#N/A	#N/A	8.1	0.09	0.4	36	3	24	21	8.5	342	27	0.02	239	2	5	1.9	0.03	34				
51/14	102B	2-Nov-00	#N/A	#N/A	#N/A	#N/A	#N/A	8.1	0.13	0.4	36	4	23	19	5.7	348	36	0.02	239	2	6	1.7	0.04	32				
52/15	105A	2-Nov-00	#N/A	#N/A	#N/A	#N/A	#N/A	8.3	0.06	0.3	27	2	23	21	12.3	332	35	0.03	239	2	9	1.9	0.04	67				
53/16	105B	2-Nov-00	#N/A	#N/A	#N/A	#N/A	#N/A	8.2	0.07	0.4	32	3	22	20	10.1	336	23	0.03	239	1	4	1.7	0.03	37				

Cells 13, 20 and 21 are replicates

Cells 30, 31, and 32 are replicates

Mineralogy: Po - pyrrhotite, py - pyrite, Aspy - arsenopyrite, Lo - lollingite, cp - chalcopyrite, carb - carbonates

ABA: S - total sulfur, S-SO₄ - Sulfur as sulfate, S-S²⁻ - Sulfur as sulfide (direct measurement), TIC - Total Inorganic Carbon

AP - Acid Potential, NP - Neutralization Potential, NNP - Net Neutralization Potential = NP - AP

TABLE 9
Cemented Flotation Tailings Test Material Characteristics and Selected Outcomes

Cell	Sample ID	Start Date	Paste pH	S %	S-SO ₄ %	TIC CO ₂ %	ACID-BASE ACCOUNTING				AQUA REGIA DIGESTIBLE				RATES (mg/kg/wk)				DEPLETION SUMMARY			
							TIC	AP	NP	NNP	NP/AP	As ppm	SO ₄	As	Duration Days	TIC Depleted kg/t	% of Total	Ca+Mg/SO ₄	%	S Depleted % of Total		
14	F48 - 1% Cement	3/22/1999	9.6	0.09	0.03	0.40	33	1.9	38	35.2	20	584	7.3	0.3	830	3	9	3.2	0.03	32		
15	F48 - 2% Cement	3/22/1999	9.9	0.08	0.04	0.50	42	1.3	49	46.5	39	616	7.7	0.3	830	4	10	4.4	0.03	38		
18	F48 - 3% Cement	3/15/1999	11.1	0.10	0.06	0.45	38	1.3	52	48.9	42	590	2.2	0.1	837	2	7	9.1	0.01	9		
34	FXX - 5% Cement	3/23/2000	9.8	0.13	0.12	2.00	167	0.3	68	63.9	218	586	2.0	0.2	463	3	2	23.3	0.00	3		

ABA: S - total sulfur, S-SO₄ - Sulfur as sulfate, S-S²⁻ - Sulfur as sulfide (direct measurement), TIC - Total Inorganic Carbon
 AP - Acid Potential, NP - Neutralization Potential, NNP - Net Neutralization Potential = NP - AP

TABLE 10
 Simulated Backfill Test Material Characteristics and Selected Outcomes

Cell	Start Date	Float %	Leach %	Cement %	Paste pH	S %	S-SO ₄ %	TIC CO ₂ %	AP kg CaCO ₃ /t	NP	NNP	NP/AP	As ppm	Release Rates (mg/kg/wk)			DEPLETION SUMMARY					
														SO ₄	As	As	Duration Days	TIC Depleted kg/t	TIC Depleted % of Total	Ca+Mg/SO ₄	S Depleted %	S Depleted % of Total
26	19-Apr-99	0	100	0	9.1	4.85	0.04	0.55	13	150	40	-111.6	0.3	30000	93	0.39	802	15	122	1.4	0.35	7
33	23-Mar-00	0	100	10	9.5	4.41	0.33	2.6	59	128	83	-54.8	0.7	29300	727	1.05	463	63	106	1.3	1.61	36
35	23-Mar-00	50	50	5	9.0	2.43	0.30	2.40	55	67	67	-8.9	1.0	16400	273	0.50	463	26	47	1.4	0.60	25
36	23-Mar-00	80	20	2	8.6	1.09	0.11	1.60	36	31	47	12.9	1.5	7740	137	0.49	463	13	36	1.4	0.30	28
37	23-Mar-00	80	20	5	9.3	1.07	0.19	2.40	55	28	67	33.6	2.4	7080	190	0.67	463	17	31	1.3	0.42	39
38	23-Mar-00	80	20	5	9.3	1.07	0.19	2.40	55	28	67	33.6	2.4	7080	182	0.60	463	14	26	1.1	0.40	38
39	23-Mar-00	80	20	5	9.3	1.07	0.19	1.60	36	28	67	33.6	2.4	7080	187	0.69	463	17	46	1.3	0.41	39

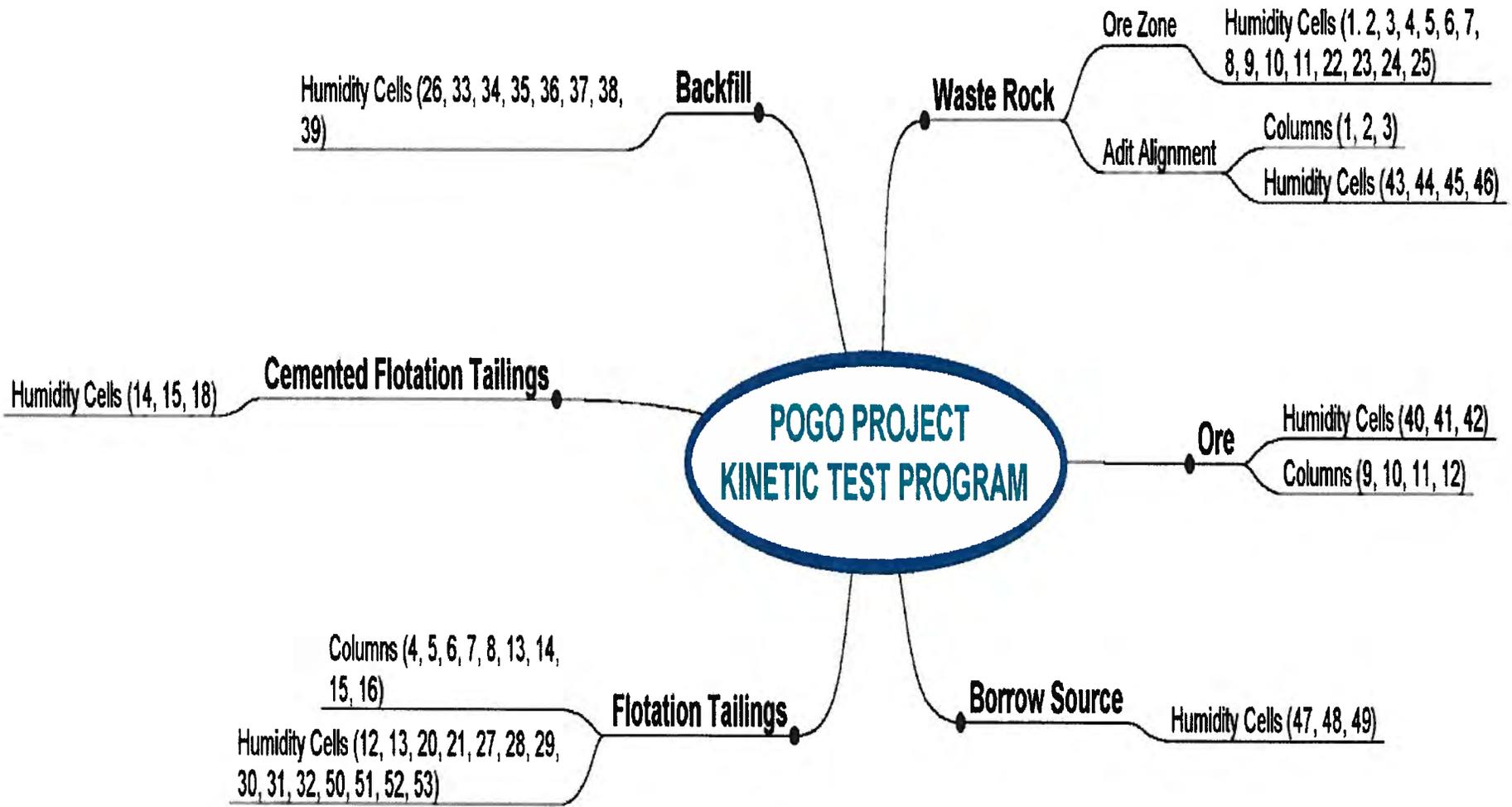
Cells 37, 38 and 39 are triplicates

ABA: S - total sulfur, S-SO₄ - Sulfur as sulfate, S-S²⁻ - Sulfur as sulfide (direct measurement), TIC - Total Inorganic Carbon
 AP - Acid Potential, NP - Neutralization Potential, NNP - Net Neutralization Potential = NP - AP

TABLE 11
Recommendations for Each Test

Test	Sample ID	Duration	Sample Type	Recommended Action	Rationale
Humidity Cell					
1	88016	126	Waste Rock	Continue	Monitor pH drop as carbonate completely depleted
2	90400	126	Waste Rock	Continue	Long term low S
3	91737	61	Waste Rock	Already stopped	Results were stable
4	95761	61	Waste Rock	Already stopped	Results were stable
5	95786	61	Waste Rock	Already stopped	Results were stable
6	95823	126	Waste Rock	Stop	Stable results
7	8544	59	Waste Rock	Already stopped	Results were stable
8	92570	124	Waste Rock	Stop	Stable results
9	89549	119	Waste Rock	Already stopped	Results were stable
10	88016	123	Waste Rock	Continue	Monitor difference between room temperature and refrigerated test
11	89549	119	Waste Rock	Stop	Stable results
12	F48	101	Flotation Tailings	Stop	Stable results
13	F48	101	Flotation Tailings	Continue	Monitor long term
14	F48	92	Cemented Flotation Tailings	Continue	Monitor Decreasing arsenic trend
15	F48	92	Cemented Flotation Tailings	Stop	Stable results
18	F48	93	Cemented Flotation Tailings	Continue	Monitor Decreasing arsenic trend
20	F48	93	Flotation Tailings	Stop	Reproducibility with Cell 13 demonstrated
21	F48	93	Flotation Tailings	Stop	Reproducibility with Cell 13 demonstrated
22	90400	50	Waste Rock	Already stopped	Mineralogical examination of residue
23	90400	92	Waste Rock	Stop	Stable reproducible results shown
24	95823	50	Waste Rock	Already stopped	Mineralogical examination of residue
25	98523	92	Waste Rock	Stop	Stable reproducible results shown
26		88	Si mulated Backfill	Continue	Monitor delay to onset of acid generation
27	F 95 Combined Tailings	80	Flotation Tailings	Stop	Stable results
28	F 96 Combined Tailings	80	Flotation Tailings	Stop	Stable results
29	F97 Combined Tailings	80	Flotation Tailings	Stop	Stable results
30	F98 Combined Tailings	80	Flotation Tailings	Continue	Monitor elevated arsenic
31	F98 Combined Tailings	80	Flotation Tailings	Stop	Reproducibility with Cell 30 demonstrated
32	F98 Combined Tailings	80	Flotation Tailings	Stop	Reproducibility with Cell 30 demonstrated
33	100% Leach, 10% wt. -7"	40	Si mulated Backfill	Continue	Results not stable
34	100% F1, 5% wt. -7"	40	Cemented Flotation Tailings	Continue	Monitor decreasing arsenic trend
35	50% F/50% leach, 5% wt. -7"	40	Si mulated Backfill	Continue	Monitor increasing arsenic trend
36	80% F1/20% leach, 2% wt. -7"	40	Si mulated Backfill	Continue	Monitor increasing arsenic trend
37	80% F/20% leach, 5% wt. -7"	40	Si mulated Backfill	Continue	Monitor increasing arsenic trend
38	80% F/20% leach, 5% wt. -7"	40	Si mulated Backfill	Stop	Reproducibility with Cell 37 demonstrated
39	80% F/20% leach, 5% wt. -7"	40	Si mulated Backfill	Stop	Reproducibility with Cell 37 demonstrated
40	97-43 (crushed-BCRI)	38	Ore	Stop	Stable results
41	97-50 (crushed-BCRI)	38	Ore	Continue	Arsenic not stable
42	97-41A (crushed-BCRI)	36	Ore	Continue	Arsenic not stable, acid generation expected.
43	N332131 (fine crushed-Chemex)	34	Waste Rock	Continue	Monitor Decreasing arsenic trend
44	N332215 (crushed-Chemex)	34	Waste Rock	Continue	Monitor Decreasing arsenic trend
45	N332229 (crushed-Chemex)	34	Waste Rock	Continue	Monitor Decreasing arsenic trend
46	N332508 (crushed-Chemex)	34	Waste Rock	Continue	Monitor Decreasing arsenic trend
47	LD 1 Composite	25	Borrow Rock	Continue	Results not stable
48	LD 3 Composite	25	Borrow Rock	Continue	Results not stable
49	LD 4 Composite	25	Borrow Rock	Continue	Results not stable
50	102 A	8	Flotation Tailings	Continue	Recently started test
51	102 B	8	Flotation Tailings	Continue	Recently started test
52	105 A	8	Flotation Tailings	Continue	Recently started test
53	105 B	8	Flotation Tailings	Continue	Recently started test
54	250A	4	Si mulated Backfill	Continue	Recently started test
55	250B	4	Si mulated Backfill	Continue	Recently started test
56	253A	4	Si mulated Backfill	Continue	Recently started test
57	253B	4	Si mulated Backfill	Continue	Recently started test
58	256A	4	Si mulated Backfill	Continue	Recently started test
59	256B	4	Si mulated Backfill	Continue	Recently started test
60	259A	4	Si mulated Backfill	Continue	Recently started test
61	259B	4	Si mulated Backfill	Continue	Recently started test
Columns					
1	Composite	20	Waste Rock	Already stopped	Results were stable
2	Composite	20	Waste Rock	Already stopped	Results were stable
3	Composite	20	Waste Rock	Already stopped	Results were stable
4	F48	101	Flotation Tailings	Continue	Long term effects
5	F48	101	Flotation Tailings	Continue	Long term effects - low temperature
6	F48	93	Flotation Tailings	Stop	Reproducibility demonstrated
7	F48	93	Flotation Tailings	Stop	Reproducibility demonstrated
8	F95	80	Flotation Tailings	Stop	Results stable
9	100A	3	Ore	Continue	Recently started test
10	100B	3	Ore	Continue	Recently started test
11	103A	3	Ore	Continue	Recently started test
12	103B	3	Ore	Continue	Recently started test
13	102A	3	Flotation Tailings	Continue	Recently started test
14	102B	3	Flotation Tailings	Continue	Recently started test
15	105A	3	Flotation Tailings	Continue	Recently started test
16	105B	3	Flotation Tailings	Continue	Recently started test
17	250B	2	Si mulated Backfill	Continue	Recently started test
18	250A	2	Si mulated Backfill	Continue	Recently started test
19	253A	2	Si mulated Backfill	Continue	Recently started test
20	253B	2	Si mulated Backfill	Continue	Recently started test
21	256A	2	Si mulated Backfill	Continue	Recently started test
22	256B	2	Si mulated Backfill	Continue	Recently started test
23	259A	2	Si mulated Backfill	Continue	Recently started test
24	259B	2	Si mulated Backfill	Continue	Recently started test
Summary					
Status		Number			
Continue		52			
Stop		20			
Already Stopped		10			

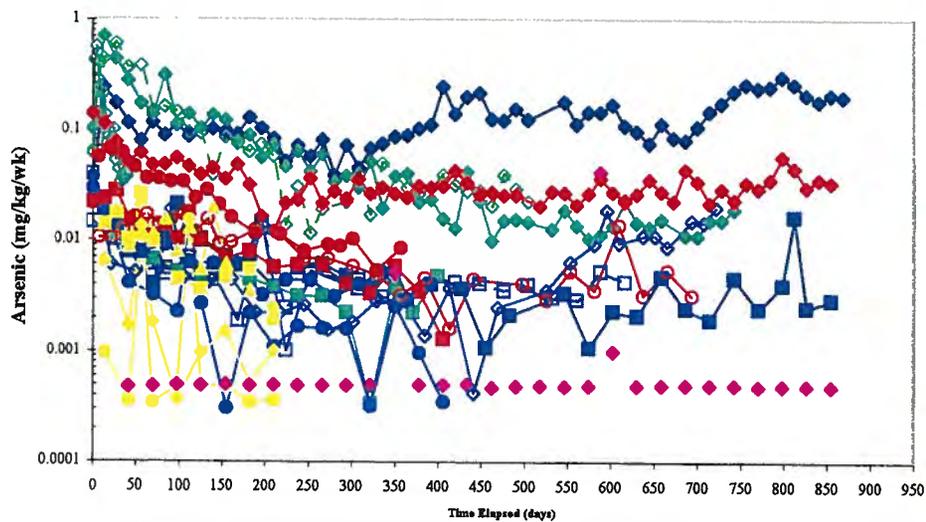
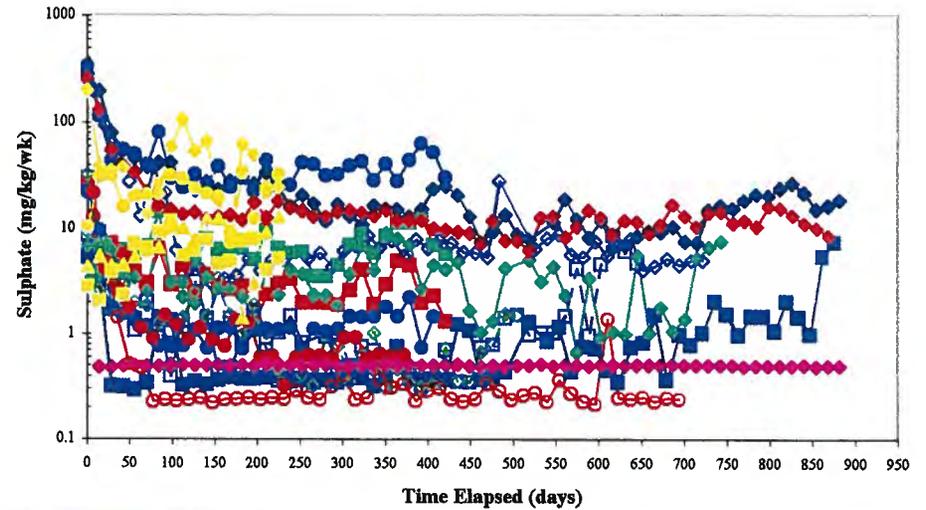
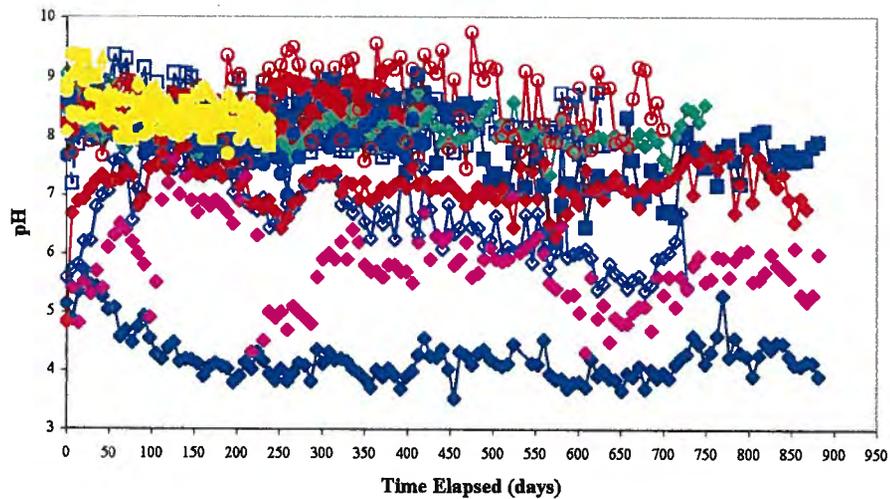
FIGURES



TECK-POGO INC.

**POGO PROJECT
FEB 2001 KINETIC TEST REPORT
Overall Structure of the Kinetic Test Program**

Project	Date	Approved	Figure
1CT002.00	FEB 2001		1



- ◆ 1 - Gneiss
- ◇ 10 - 1 Refrig
- 2 - Gneiss
- 23 - 2 Trip
- ▣ 22 - 2 Trip
- 3 - gd
- 4 - Gneiss
- 5 - Altered
- ◆ 6 - gd
- ◇ 24 - 6 Trip
- ◇ 25 - 6 Trip
- 7 - Gneiss
- ◆ 8 - Altered
- 9 - Altered
- 9 - 11 Refrig
- ◆ 43 - Adit
- 44 - Adit
- ▲ 45 - Adit
- 46 - Adit
- ◆ Blank

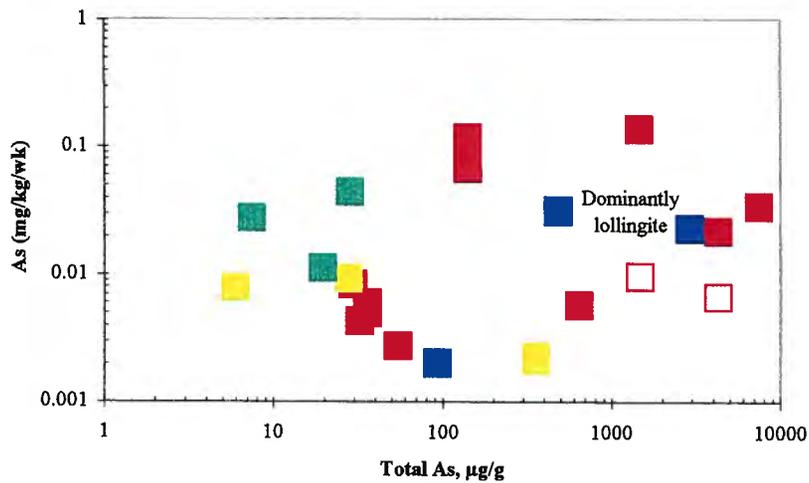
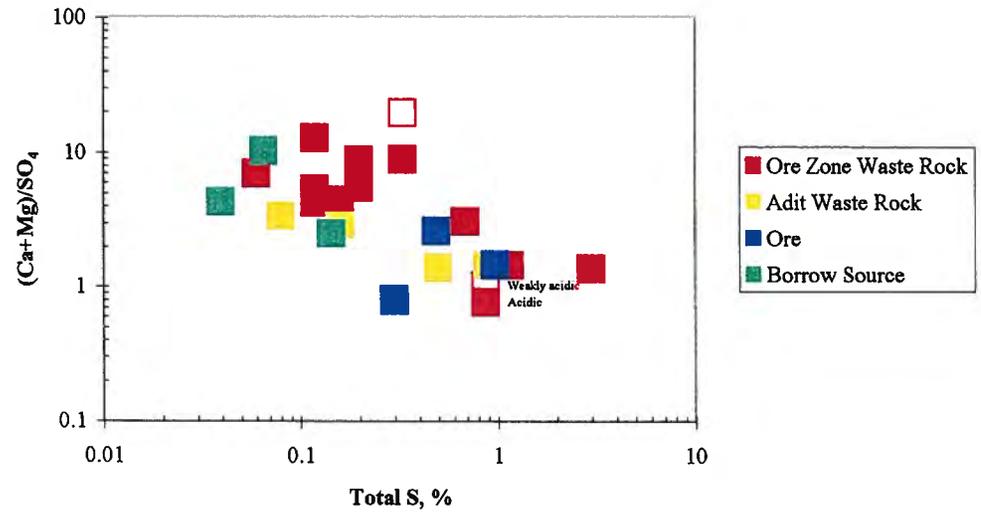
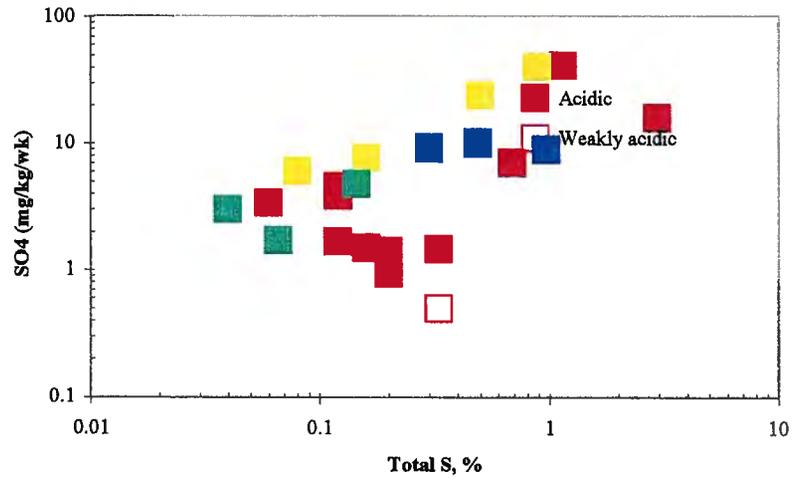


TECK-POGO INC.

**POGO PROJECT
FEB 2001 KINETIC TEST REPORT**

Waste Rock - pH(t), SO₄(t) and As(t)

Project	Date	Approved	Figure
1CT002.00	FEB 2001		2



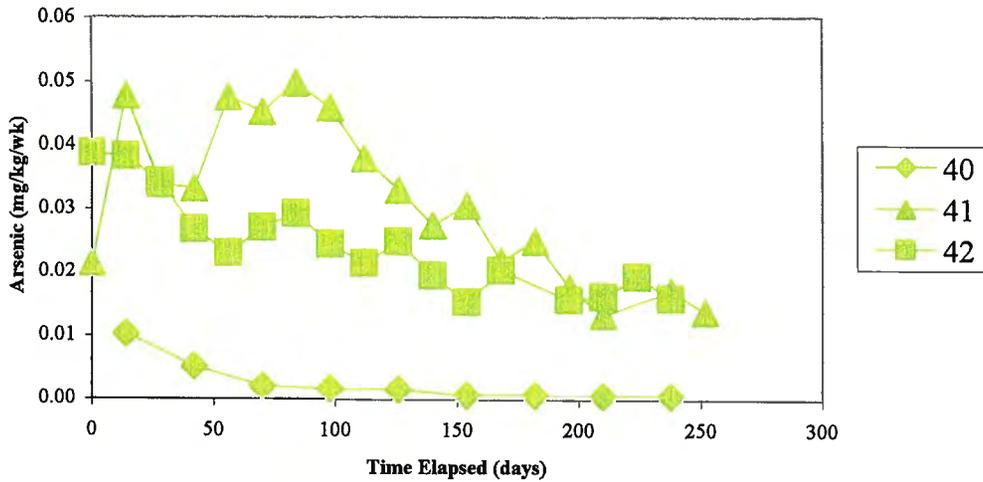
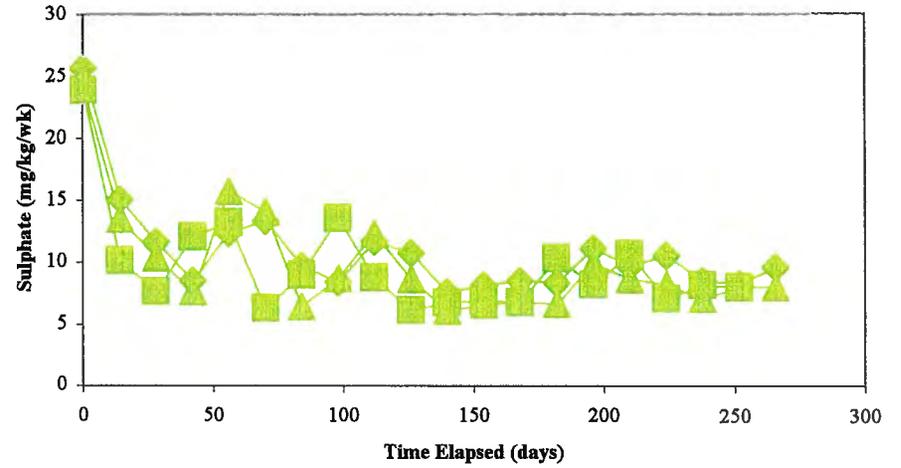
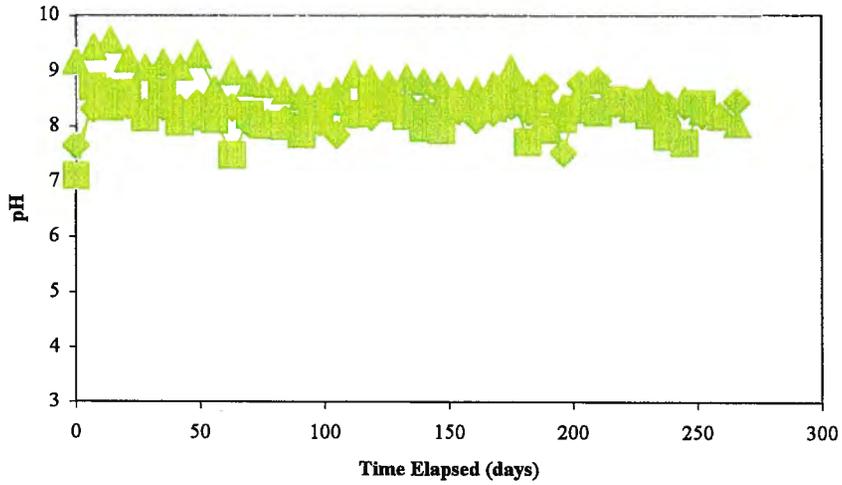
Note: Open symbols indicate refrigerated cells.



TECK-POGO INC.

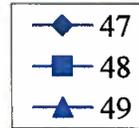
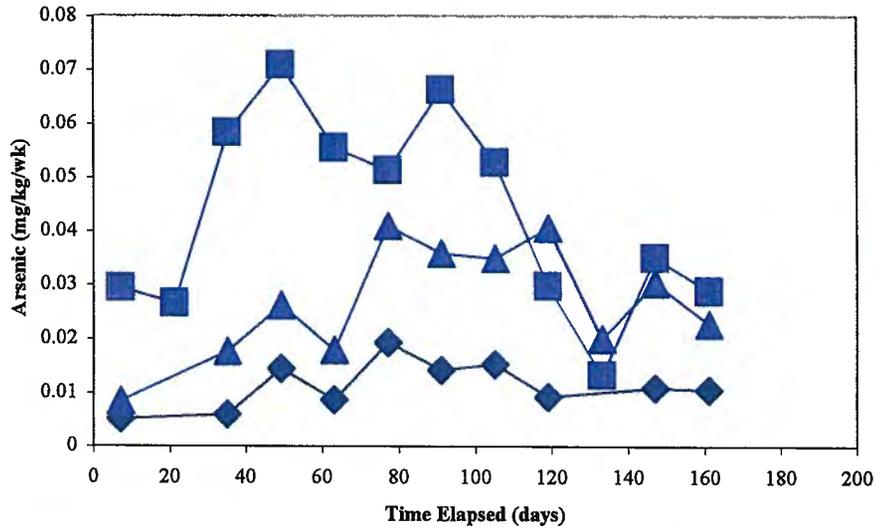
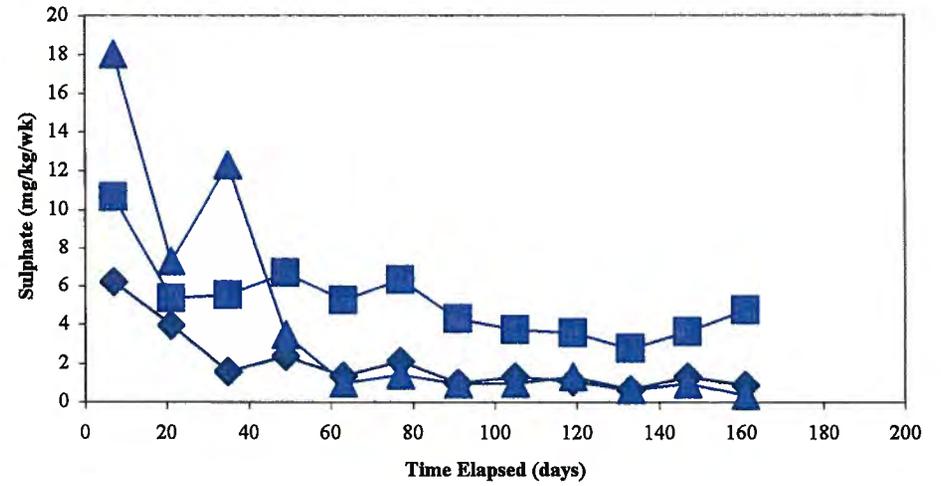
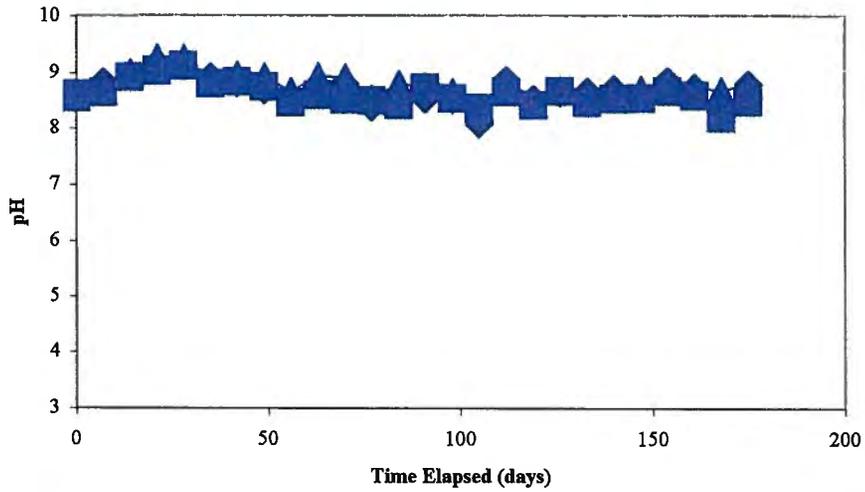
POGO PROJECT
FEB 2001 KINETIC TEST REPORT
Rock Humidity Cells - Rate and Bulk
Chemistry Relationships

Project	Date	Approved	Figure
1CT002.00	FEB 2001		3



TECK-POGO INC.

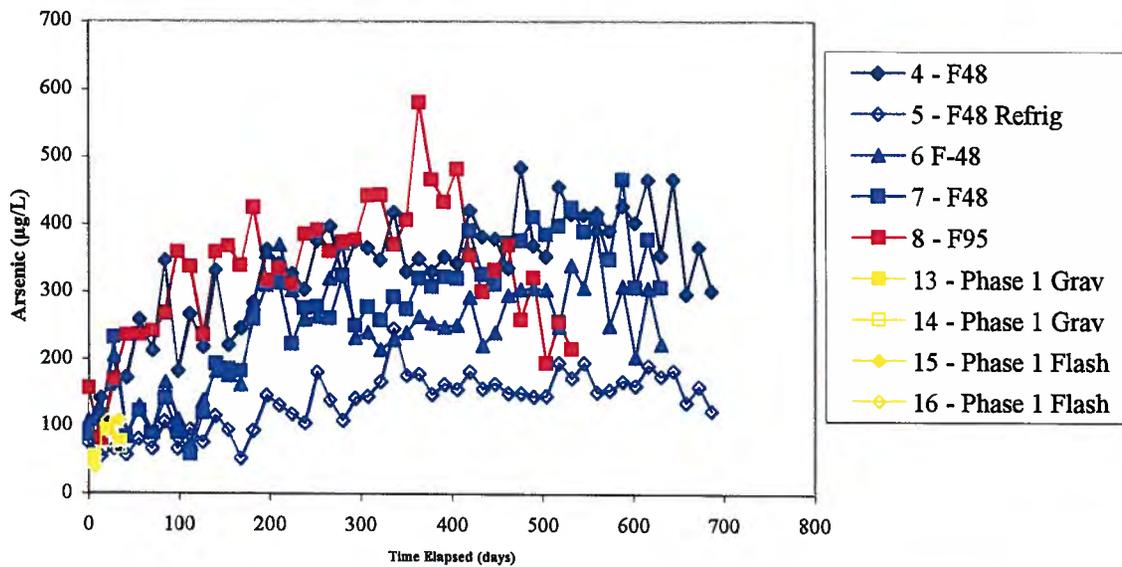
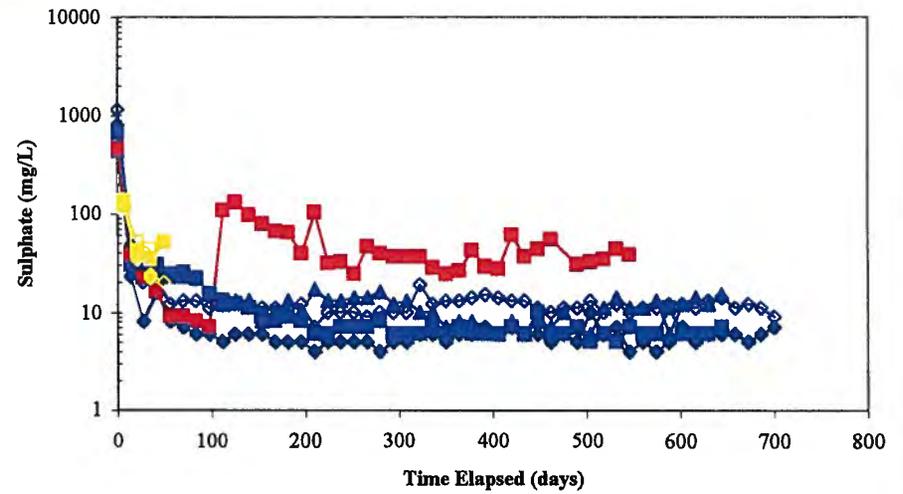
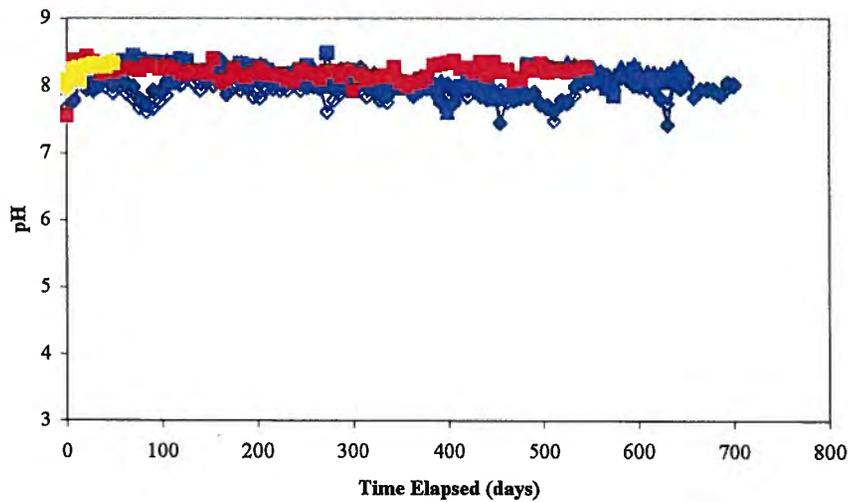
POGO PROJECT			
FEB 2001 KINETIC TEST REPORT			
Ore Humidity Cells - pH(t), SO₄(t) and As(t)			
Project	Date	Approved	Figure
1CT002.00	FEB 2001		4



TECK-POGO INC.

**POGO PROJECT
FEB 2001 KINETIC TEST REPORT
Borrow Rock Humidity Cells - pH(t),
SO₄(t) and As(t)**

Project	Date	Approved	Figure
1CT002.00	FEB 2001		5



- ◆ 4 - F48
- ◇ 5 - F48 Refrig
- ▲ 6 F-48
- 7 - F48
- 8 - F95
- 13 - Phase 1 Grav
- 14 - Phase 1 Grav
- ◆ 15 - Phase 1 Flash
- ◇ 16 - Phase 1 Flash

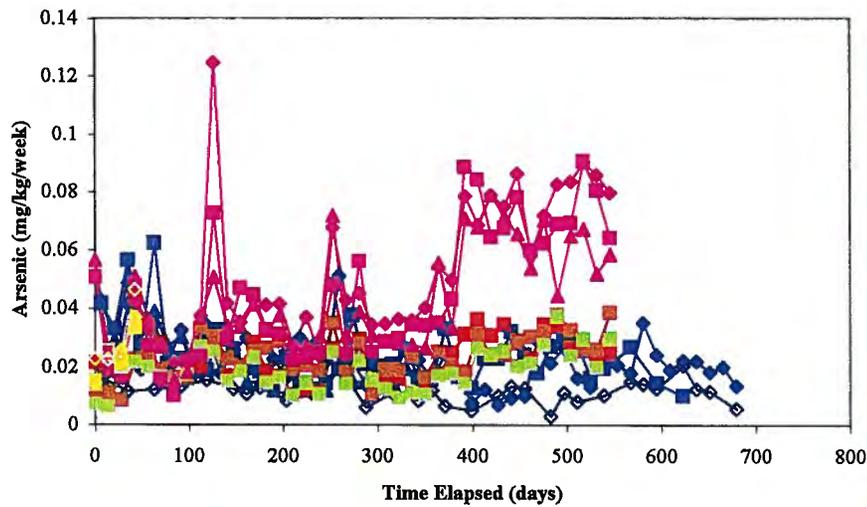
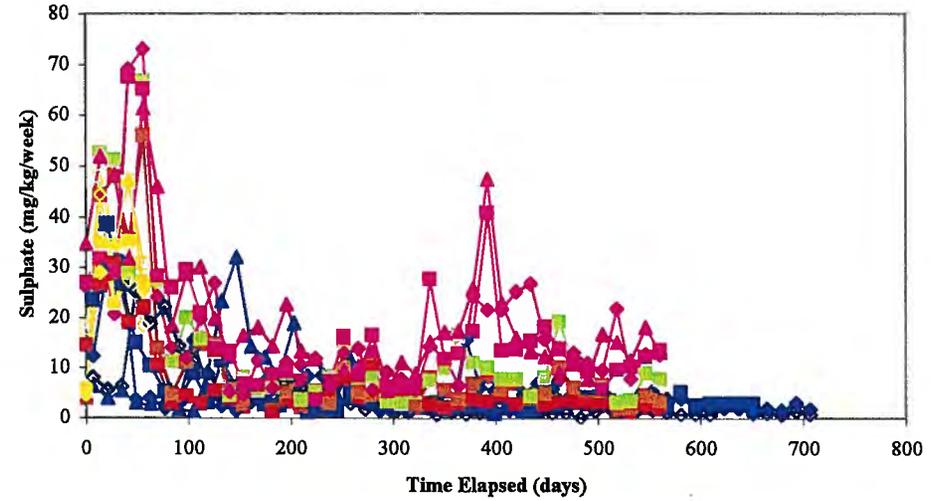
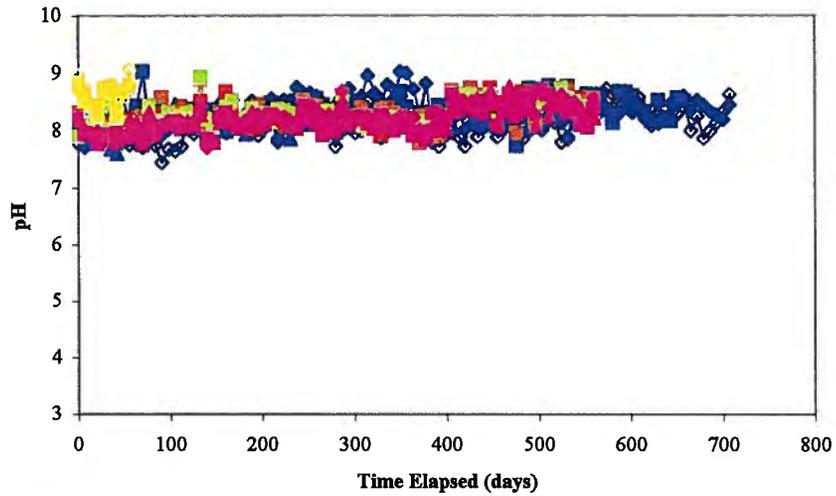
Open symbols indicate refrigerated cells.



TECK-POGO INC.

**POGO PROJECT
FEB 2001 KINETIC TEST REPORT
Flotation Tailings Columns - pH(t),
SO₄(t) and As(t)**

Project	Date	Approved	Figure
1CT002.00	FEB 2001		6



- 12 - F48 - Refrig
- 13 - F48
- ▲ 20 - F48
- 21 - F48
- 27 - F95
- 28 - F96
- 29 - F97
- 30 - F98
- ◆ 31 - F98
- ▲ 32 - F98
- 50 - Phase 1 Grav
- 51 - Phase 1 Grav
- ◆ 52 - Phase 1 Flash
- ◇ 53 - Phase 1 Flash

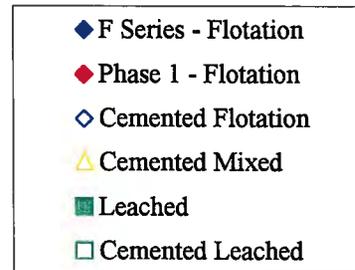
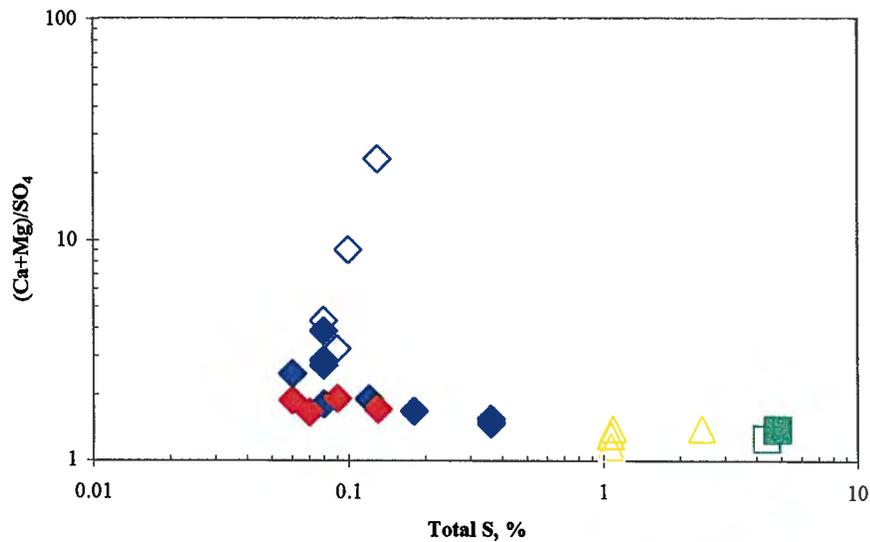
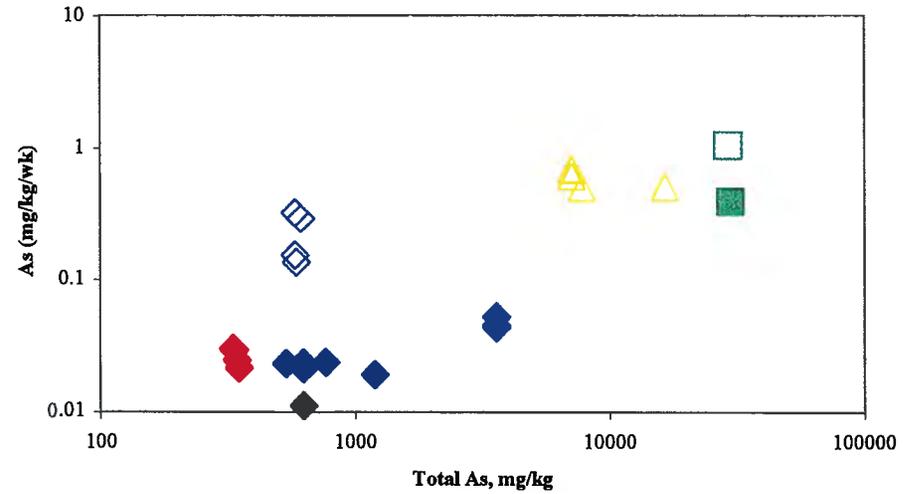
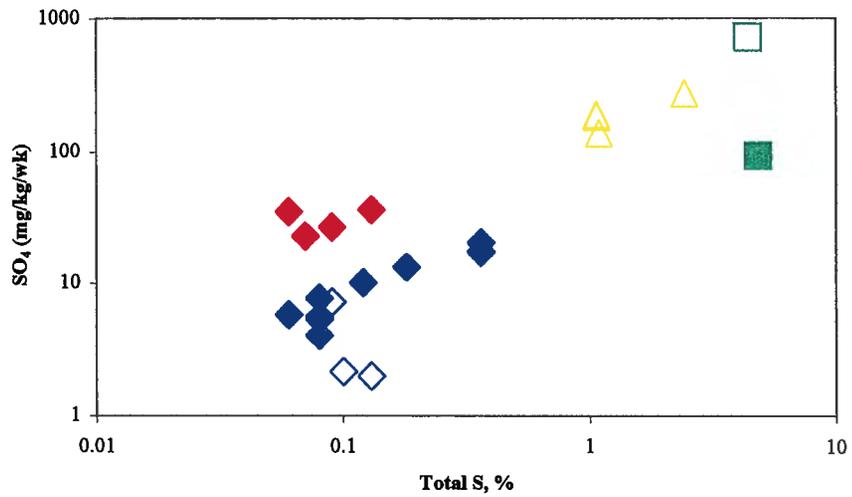
Open symbols indicate refrigerated cells.



TECK-POGO INC.

**POGO PROJECT
FEB 2001 KINETIC TEST REPORT
Flotation Tailings Humidity Cells -
pH(t), SO₄(t) and As(t)**

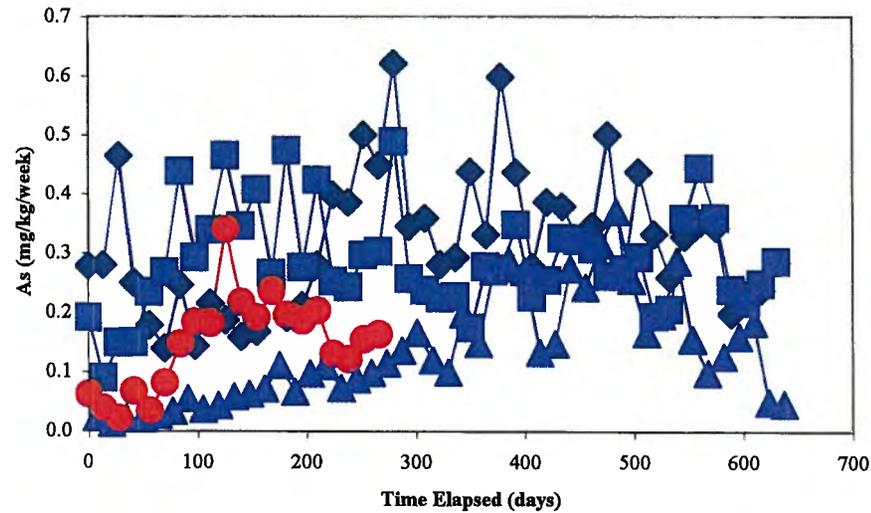
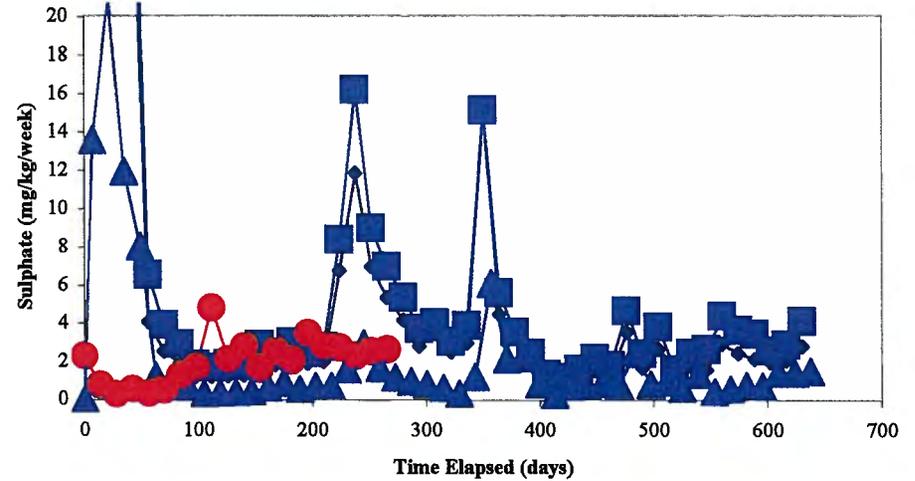
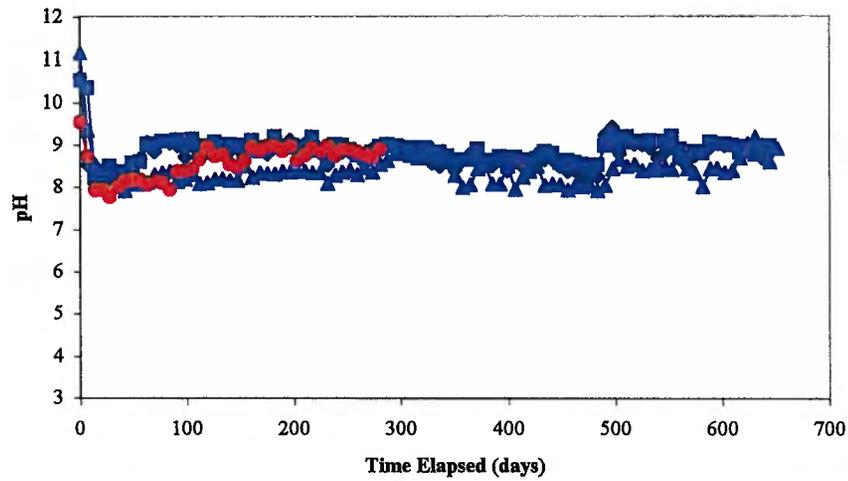
Project	Date	Approved	Figure
1CT002.00	FEB 2001		7



TECK-POGO INC.

POGO PROJECT
FEB 2001 KINETIC TEST REPORT
Process Samples - Rate and Bulk
Chemistry Relationships

Project	Date	Approved	Figure
1CT002.00	FEB 2001		8

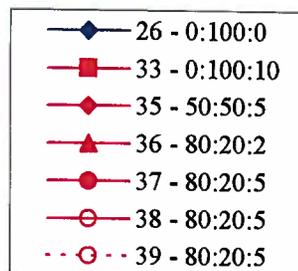
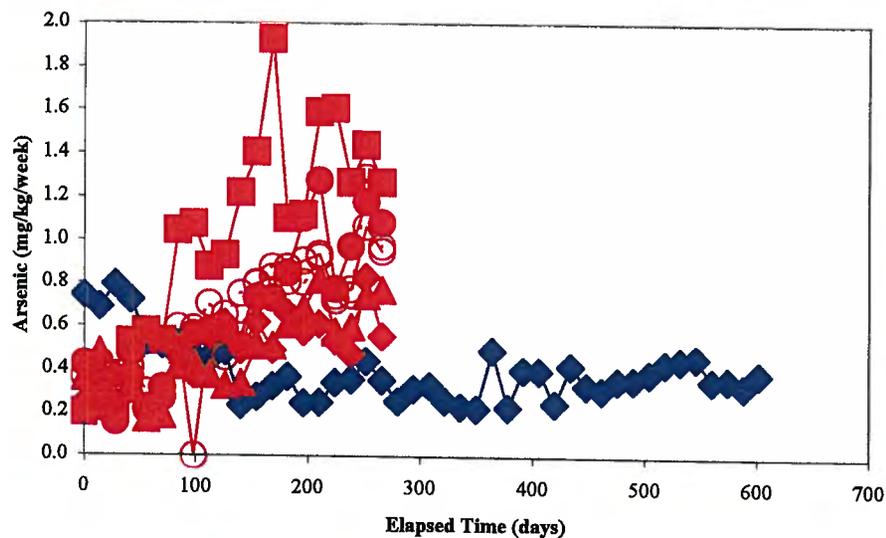
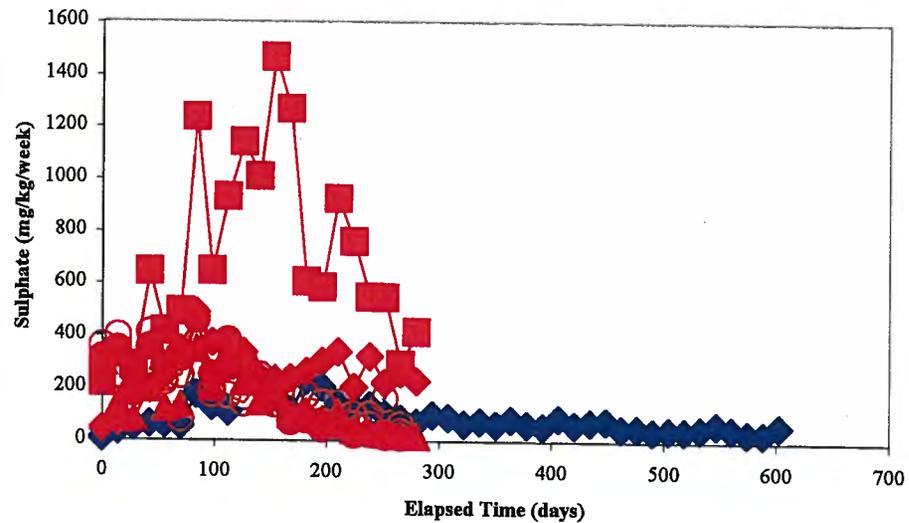
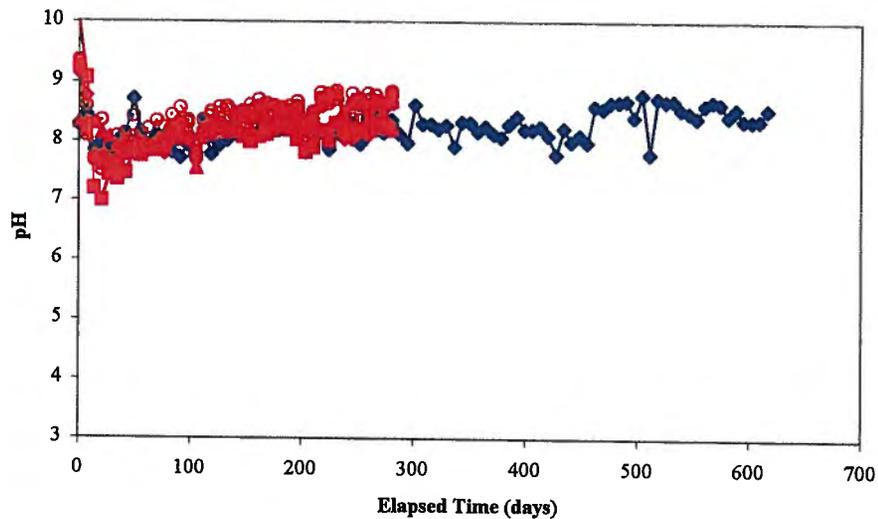


- ◆ 14 - 1% Cement
- 15 - 2% Cement
- ▲ 15 - 3% Cement
- 34 - 5% Cement



TECK-POGO INC.

POGO PROJECT			
FEB 2001 KINETIC TEST REPORT			
Cemented Flotation Tailings Humidity Cells			
- pH(t), SO₄(t) and As(t)			
Project	Date	Approved	Figure
1CT002.00	FEB 2001		9



Legend:
 Float Tailings : Leach Tailings : Cement



TECK-POGO INC.

POGO PROJECT
FEB 2001 KINETIC TEST REPORT
Simulated Backfill Humidity Cells -
pH(t), SO₄(t) and As(t)

Project	Date	Approved	Figure
1CT002.00	FEB 2001		10

MEMORANDUM

DATE: June 5, 2001

TO: Rick Zimmer, Teck

FROM: Stephen Day, SRK

CC: Karl Hanneman, Teck
Bryan Nethery, AMEC

PROJECT: 1CT002.00

RE: **COMPILATION OF WATER CHEMISTRY PREDICTIONS
POGO PROJECT**

The purpose of the memorandum is to present a compilation of the water quality predictions for dry stack seepage, dry stack runoff, and temporary mineralized stockpile runoff for the Pogo project. These values are based on the memos described below, with modifications as recommended by the EIS team.

- **July 24, 2000. "Pogo Project Water Chemistry Predictions" (included in Appendix C of the Water Management Plan document dated August 2000).** This memo set forth the original predictions for water quality coming from the dry stack seepage, drystack runoff, and temporary ore stockpile runoff.
- **February 12, 2001. "Explanation of Differences Between Predictions of Dry Stack Runoff Chemistry".** This memo set forth revised predictions for drystack runoff.
- **February 12, 2001. "Average Case Predictions".** This memo set forth the arithmetic average for all reported dissolved analyses to date from development rock piles, site runoff, and groundwater.

During a March 2, 2001 telephone conference call, the EIS Team members reviewed the above memos and indicated their preferred values for selected predictions, as described below.

- **Dry Stack Seepage**

The EIS team would use the Site Data Maximum Concentrations for Fe, Mn and Zn rather than the calculated reasonable worst case (original data in Table 1, SRK Memo July 24, 2000, Appendix C, Water Management Plan). The remaining parameters were considered acceptable.

Parameter	Testwork	Site Data	SRK	EIS Team
	Max Conc.	Max Conc.	Reasonable Worst Case Conc.	Reasonable Worst Case Conc.
	mg/L	mg/L	mg/L	mg/L
Fe	1.3	29.6	3	29.6
Mn	1.9	29.7	19	29.7
Zn	0.11	0.699	0.37	0.699

- **Drystack Runoff**

The EIS team agreed with the estimates as presented in the memo dated February 12, 2001, which were primarily based on site maximum concentrations.

- **Temporary Mineralized Rock Pile**

The EIS team indicated that they would use the Site Data or Testwork Maximum Concentrations for Mn, Ni and Zn rather than the calculated reasonable worst case (original data in Table 3, SRK Memo July 24, 2000, Appendix C, Water Management Plan). The remaining parameters were considered acceptable.

Parameter	Testwork	Site Data	SRK	EIS Team
	Max Conc.	Max Conc.	Reasonable Worst Case Conc.	Reasonable Worst Case Conc.
	mg/L	mg/L	mg/L	mg/L
Mn	1.9	29.7	0.98	29.7
Ni	0.057	0.236	0.05	0.236
Zn	0.11	0.699	0.06	0.699

Table 1 provides a summary of the reasonable worst case predictions for the Pogo project. In order to develop probability distributions for these parameters for use in project modeling, it would be appropriate to use the averages as medians, then calibrate the distribution to the reasonable worst case at the 99% confidence limit.



Please call me if you have any questions.

TABLE 1: Compilation of Recommended Concentrations

Source Parameter	Drystack Runoff (mg/L)		Drystack Seepage (mg/L)		Temporary Mineralized Rock Seepage (mg/L)	
	SRK Recommended Value	EIS Team Recommended Value	SRK Recommended Value	EIS Team Recommended Value	SRK Recommended Value	EIS Team Recommended Value
Ag	0.0002	same	0.002	same	0.002	same
Al	0.087	same	1.59	same	1.84	same
As	0.4	same	5.1	same	0.5	same
B	0.02	same	3	same	0.3	same
Ba	0.008	same	0.49	same	0.163	same
Alkalinity	90	same	174	same	187	same
Ca	97	same	291	same	31	same
Cd	0.0004	same	0.005	same	0.005	same
Cr	0.0011	same	0.014	same	0.014	same
Cu	0.003	same	0.034	same	0.03	same
Fe	0.0003	same	3	29.6	1.45	same
Hg	0.0002	same	0.002	same	0.002	same
K	1.5	same	59	same	59	same
Mg	24	same	300	same	100	same
Mn	2.4	same	19	29.7	0.98	29.7
Na	6.3	same	130	same	9	same
Ni	0.02	same	0.24	same	0.05	0.236
Pb	0.0004	same	0.005	same	0.005	same
SO ₄	302	same	2002	same	386	same
Se	0.006	same	0.13	same	0.03	same
Zn	0.06	same	0.37	0.699	0.06	0.699
Hardness	343	same	1980	same	494	same
TDS	523	same	3000	same	772	same

MWC - Mass-weighted concentrations

Bold – Different Recommendations for SRK and EIS Team

MEMORANDUM

DATE: February 12, 2001
TO: Bryan Nethery, AMEC
FROM: Stephen Day, SRK
PROJECT: 1CT002.00
RE: **“AVERAGE CASE” PREDICTIONS**

Bryan

Table 1 summarizes arithmetic average dissolved concentrations for site waters in intimate contact with possibly mineralized rock. This includes monitoring of the development rock piles, site runoff from the portal area and groundwater. The concentrations are simple arithmetic averages of all reported dissolved analyses; no attempt has been made to create weighted average concentrations. Detection limit values were used without adjustment. Different detection limits have been used for different monitoring programs.

When these numbers are used for the water quality predictions, the rules to be applied are:

- If the Average Dissolved Concentration (ADC) in Table 1 is less than the Reasonable Worst Case (RWC) concentration (developed described in the SRK July 24, 2000 memorandum to R. Zimmer, Appendix C, Water Management Plan), then use the ADC.
- If the RWC concentration is less than the ADC, use the Reasonable Worst Case (RWC) as the RWC was obtained based on adjustments of the overall chemistry using MINTEQ and mass balance considerations.

Tables 2 to 4 summarize the “Reasonable Worst Case” predictions and the average case predictions made using the above rules. Predictions shown in Table 2 and 4 for the RWC were previously presented in the July 24, 2000 memorandum. Table 3 RWC concentrations are based on the AMEC December 22, 2000 memorandum and subsequent revision as described in SRK February 12, 2001 memorandum.



For the most part the average-based predictions generated for the Dry Stack Seepage (Table 2) and Temporary Mineralized Rock Stockpile (Table 4) are significantly lower than the worst cases. In Table 4, arsenic is only slightly lower in the average case. In the case of the Dry Stack runoff (Table 3), the average concentrations are similar to the worst case for Ag, Al, As, Ba, Cr, Fe, Hg and Se.

The similarity of some of the predictions reflects:

- Both predictions are strongly influenced by close proximity to the detection limits (eg Ag, Al, Cr, Fe, Hg, Pb, Se), therefore there is not much difference between average and maximum concentrations.

- Site average concentrations are influenced by large numbers of elevated values, which occur in the database. Arsenic is the main example. Groundwater arsenic concentrations are elevated in the vicinity of the ore body. This could be addressed by calculating the average case from geometric rather than arithmetic means, although I have not done so.

Give me a call when you have looked at this.



TABLE 1
Summary of Average Concentrations

Parameter	Number of Analyses	Average Dissolved Conc (mg/L)
Ag	138	0.000022
Al	141	0.10
As	141	0.43
Ba	141	0.059
B	141	0.0065
Cd	141	0.00014
Co	141	0.0041
Cr	141	0.0028
Cu	141	0.0016
Fe	141	0.018
Hg	135	0.000040
Mn	141	0.87
Mo	141	0.0058
Ni	139	0.0064
Pb	141	0.00028
Sb	141	0.0016
Se	141	0.0017
Zn	141	0.027
TDS	141	420

TABLE 2
Drystack Seepage

Parameter	SRK July 24, 2000 Reasonable Worst Case Conc. mg/L	Site Average Case mg/L
Ag	0.0018	0.00002
Al	1.59	0.1
As	5.1	0.4
Ba	0.49	0.06
Cd	0.005	0.0001
Cr	0.014	0.003
Cu	0.034	0.002
Fe	3	0.02
Hg	0.002	0.00004
Mn	19	0.9
Ni	0.24	0.006
Pb	0.005	0.0003
Se	0.13	0.002
Zn	0.37	0.03
TDS	3000	420



**TABLE 3
Drystack Runoff**

Parameter	SRK Feb 12, 2001 Reasonable Worst Case Conc. mg/L	Site Average Case mg/L
Ag	0.0002	0.00001
Al	0.09	0.09
As	0.4	0.4
Ba	0.008	0.008
Cd	0.0004	0.0001
Cr	0.0011	0.0011
Cu	0.003	0.002
Fe	0.0003	0.0003
Hg	0.00016	0.00004
Mn	2.4	0.9
Ni	0.02	0.006
Pb	0.0004	0.0003
Se	0.006	0.002
Zn	0.06	0.03
TDS	523	420

**TABLE 4
Temporary Mineralized Rock Pile Seepage and Runoff**

Parameter	SRK July 24, 2000 Reasonable Worst Case Conc. mg/L	Average Case mg/L
Ag	0.0018	0.00002
Al	1.84	0.1
As	0.5	0.4
Ba	0.163	0.06
Cd	0.0048	0.0001
Cr	0.014	0.003
Cu	0.03	0.002
Fe	1.45	0.02
Hg	0.002	0.00004
Mn	1.0	0.9
Ni	0.05	0.006
Pb	0.005	0.0003
Se	0.03	0.002
Zn	0.06	0.03
TDS	772	420

MEMORANDUM

DATE: February 12, 2001

TO: Karl Hanneman, Teck

FROM: Stephen Day, SRK

PROJECT: 1CT002.00

RE: **EXPLANATION OF DIFFERENCES BETWEEN PREDICTIONS OF DRY
STACK RUNOFF CHEMISTRY**

RESPONSE TO BRUCE NELSON COMMENTS

The following is provided in response to Bruce Nelson's comments to Dave Bunte (Memo of January 15, 2001 – Response Review: Tailings Water Chemistry/Tailings Seepage). He requested an explanation for the decrease in predicted concentrations, despite the substantial increase in the volume of tailings potentially contributing to runoff chemistry.

The reason for the difference is in the different methods used to calculate how the oxidized load is transported to the surface.

In the original estimate, it was assumed that 1 cm of tailings contributed to runoff. It was assumed conservatively that the entire soluble load in the 1 cm of oxidized tailings would be dissolved in the runoff volume. The runoff volume was 4.6 gpm (see footnote in Table 2 of the July 24, 2000 memo). The majority of concentrations were well below the site maximum concentrations and any constraint indicated by MINTEQA2, therefore, predicted concentrations were comparable to the concentrations using the mass-weighting method described in the July 24, 2000 memorandum.

For the revised estimate, it was necessary to consider that the upward movement of water limits the load being transported to the surface. It would not be logical to assume that the load produced by oxidation at a depth of 1 foot could actually be available for dissolution by runoff. Therefore, the load transported to surface would be limited by the solubility of the oxidation products in the upward moving water. Because the depth of tailings is significant, and the volume of leaching water is small, the mass-weighted concentrations exceeded the site maximum



concentrations by significant margins. The concentrations were adjusted downward to the maximum concentrations. Therefore, the load transported is:

$$C_{\max} \cdot Q_{\text{evap}}$$

where C_{\max} is the maximum concentration and Q_{evap} is the rate of evaporation.

This load transported to surface is then diluted by the much larger volume of runoff, and the resulting concentration is:

$$C_{\max} \cdot (Q_{\text{evap}} / Q_{\text{Runoff}})$$

The runoff volume used in this calculation is 26 gpm (compared to 4.6 gpm in the July 26, 2000 memo). This reflects refinements in AMEC's water balance for the drystack.

In summary, the greater depth of oxidation theoretically increases the maximum load available, but also isolates it from surface, limiting the availability for leaching.

RESPONSE TO DAVE BUNTE COMMENTS

The following is provided in response to Dave Bunte's comments to Karl Hanneman (Memo of January 17, 2001 – Review of AMEC Memos on Pogo Tailings Drystack Runoff and Seepage (December 22, 2000)). He requested clarification of how the revised runoff chemistry predictions were developed for each element.

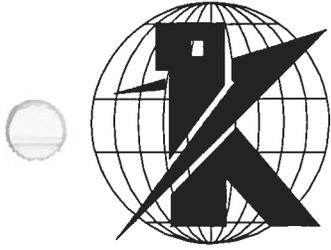
The logic used is described in the previous section. All the predicted concentrations exceeded maximum concentrations and are therefore calculated based on the maximum concentrations.

Dave Bunte correctly noted a mistake in the predictions presented in the December 22, 2000 memorandum. The concentrations were based on the site maximum concentrations rather than both laboratory and site maximum concentrations. This affected the predictions for Ag, B, Cr, Hg, and Se. The correct predictions using this approach are shown in Table 1. However, as shown in Table 1, the concentrations for these parameters are less than or equal to the original estimates (Column 3).



**TABLE 1
Dry Stack Runoff Chemistry**

Parameter	Site Data Max Conc. mg/L	SRK July 24, 2000 Reasonable Worst Case Conc. mg/L	AMEC Dec. 22, 2000 Reasonable Worst Case Conc (Based on site max only) mg/L	SRK Feb. 12, 2001 Worst Case Conc (Based on Maximum Concentrations from Testwork and Site Monitoring) mg/L
Ag	0.0018	0.0002	0.00001	0.0002
Al	1.84	0.095	0.087	0.087
As	5.1	0.4	0.4	0.4
B	0.116	0.033	0.01	0.02
Ba	0.491	0.007	0.008	0.008
Alkalinity	530	158	90	90
Ca	936	42	97	97
Cd	0.0046	0.0001	0.0004	0.0004
Cr	0.014	0.003	0.0001	0.0011
Cu	0.0336	0.005	0.003	0.003
Fe	29.6	0.0004	0.0003	0.0003
Hg	0.002	0.001	0.00001	0.00016
K	19	25	1.5	1.5
Mg	294	37	24	24
Mn	29.7	0.10	2.4	2.4
Na	77.9	11	6.3	6.3
Ni	0.236	0.01	0.02	0.02
Pb	0.0054	0.005	0.0004	0.0004
SO ₄	745	255	302	302
Se	0.049	0.01	0.0015	0.006
Zn	0.699	0.007	0.06	0.06
Hardness	1180	260	343	343
TDS	7160	529	523	523



Teck-Pogo Inc.

Memo

To: Karl Hanneman
From: Shelly Basketfield
CC: JF, DAH, BAL
File: 856F.101.03(03)
Date: 16-January-2002
Re: Pogo Total and Dissolved Metals Conversion Factors

Summary

This memo presents recommendations for site-specific conversion factors for the Pogo project for comparing modeling results completed with dissolved values to water quality criteria or effluent limitations that are based on total recoverable values.

Discussion

Several approaches may be taken to develop a conversion factor between total and dissolved metals. Although it is known that metals associate with particulate matter, the nature of this association is complex and there are several mechanisms involved: (1) physical adsorption to solid surfaces, (2) chemical sorption or binding by ligands, and (3) ion exchange (Chapra, 1997).

The Environmental Protection Agency (EPA) provides a formula for determining a partition coefficient, K_d , based on Total Suspended Solids (EPA, 1995).

$$C = \frac{C_{if}}{1 + K_d TSS \times 10^{-6}}$$

where,

C=dissolved phase metal concentration,

C_{if} =Total metal concentration,

TSS-total suspended solids concentration, and

K_d =partition coefficient.

If we define a conversion factor, CF, as

$$CF = \frac{C}{C_{df}}$$

and assume that K_d is constant, then the conversion factor, CF, decreases with increasing TSS concentration.

The EPA also provides conversion factors (CF's) for several metals to convert total to dissolved and vice versa in the water quality standards guidance (EPA, 1999). There are different conversion factors for fresh water and salt water, different factors for criterion maximum concentration (CMC) and criterion continuous concentration (CCC) and some of CF's are based on hardness.

Further complicating the definition of a CF, in anoxic sediments the metals are subject to precipitation with hydrogen sulfide. Metals precipitate in sequence depending on their solubility products, with the metals having the lowest solubility products precipitating first. The sequence is; HgS, CuS, PbS, CdS, ZnS, and NiS (Chapra, 1997).

Therefore several simplifying assumptions were made in the approach to defining a CF for this site:

1. The TSS concentration will be less than 20 mg/L. This is based on the Effluent limitation Guidelines (ELG's) for gold mining; a maximum daily of 30 mg/L TSS and a monthly average of 20 mg/L.
2. At these low TSS concentrations the CF will be constant, that is the CF will not vary with changing TSS concentrations.
3. The relationship between the total and dissolved metal fractions at these low TSS concentrations can be represented by the straight-line $y=mx+b$, where y is the dissolved fraction, x is the total fraction, m is the slope of the line and the y-intercept, b, is equal to 0.

Method

Surface water, groundwater, and water treatment plant data were extracted from the EQWin™ database for total metals, dissolved metals, and TSS. The data sets were filtered to include only those where the total metals concentration was greater than the dissolved.

The next step was to look at the filtered data to determine;

- Is a relationship present between total and dissolved, and
- Is a relationship present between total and TSS concentrations.

After a review of all the groundwater and surface water data, the data was grouped on four locations at TSS concentration less than 20 mg/L to better understand differences in correlations between total and dissolved fractions and the effect of TSS. The water quality data location groups included;

1. All surface water stations
2. All groundwater stations except those in Liese Creek or those in the Goodpaster Valley above the confluence with Liese Creek (the LL series).
3. Groundwater stations in upper Liese Creek (LB-001, LB-003A, LD-005, LL-012B, LT-003, LT-007A, LT-007B, LT-009, LW-003)

4. Groundwater Station in the Goodpaster River Valley above the confluence with Liese Creek (LL-001, LL-002, LL-003, LL-004, LL-005, LL-006A, LL-006B, LL-007, LL-008B, LL-010A, LL-010B, LL-012A, LL-012B, LL-014, LL-023, LL-024, LL-025, LL-026, LL-027, LL-028, LL-029, and LL-030).
5. Water treatment plant.

A straight-line fit was used to establish a relationship between the total and dissolved metals fractions for each parameter. A visual review was used to determine that the relationship between to each set of data paired total and dissolved metals. If the R-squared value was greater than 0.9, then the best fit was considered acceptable and the slope of the line was assumed to be a correlation between the total and dissolved fraction, that is $y = mx + b$.

A summary of the results is presented below in Table 1. Based on the amount of data in the data set, the quality of the correlation, and other factors, the recommended value is noted in bold.

Table 1. Summary of Conversion Factors (CFs) for Metals at TSS <20 mg/L

Metal	Comments	R-squared Correlation (TSS < 20mg/L)				CF (slope of line) (TSS < 20mg/L)					EPA (1999) Fresh Water CCC ²
		Surface Water	Lower Liese Series	Liese Creek Series	All Other Wells	Surface Water	Lower Liese Series	Liese Creek Series	All Other Wells	WTP ¹	
Ag	Detection limits are too high to determine a correlation.	0.69	NA	NA	NA	0.39	NA	NA	NA	DL	0.85
As	Two populations in the LL series; one at less than 2 µg/L and one between 7 and 12 µg/L.	0.98	0.85	0.86	0.98	0.90	0.82	0.80	0.87	0.84	1.0
Cd*	Scatter on LL series	0.97	NA	NA	0.49	0.92	NA	NA	0.48	DL	0.87
Cr		0.85	0.93	0.90	0.98	0.84	0.68	0.83	0.91	DL	0.86
Cu*	Scatter	0.97	0.61	0.86	0.63	0.92	0.41	0.75	0.56	0.92	0.96
Fe*	Scatter on groundwater	0.53	NA	NA	0.99	0.38	NA	NA	0.94	0.32	
Hg*	Scatter, Groundwater data near the detection limit	0.40	0.90	1	0.90	0.79	0.94	1	0.87	DL	0.85
Mn	Scatter of data for Liese Creek Series	0.99	0.99	NA	0.98	0.90	0.98	NA	0.89	1.0	
Pb*	Scatter, some data near detection limits	0.79	NA	0.33	0.73	0.52	NA	0.18	0.49	0.75	0.643
Se	One data points for Upper Liese Series, two for LL series	0.99	NA	NA	0.98	0.97	NA	NA	0.76	DL	
Zn	Scatter on LL series and other wells, Liese Creek at detection limits.	0.99	NA	NA	NA	0.95	NA	NA	NA	DL	0.986

¹ the WTP data includes two data sets, collected on 06/27/2001 and 12/05/01. In both samples, the dissolved manganese concentration is greater than the total manganese concentration. Data pairs reported as less than the method reporting limit (MRL) are noted as DL (detection limit). A ratio of dissolved to total could not be calculated for these pairs of data.

² from Appendix A - Conversion Factors for Dissolved Metals (EPA,1999). The hardness-based conversion factors (cadmium and lead) were calculated based on a hardness of 276 mg/L (the 5th percentile hardness of the existing WTP data). Measured chromium is total chromium, the EPA conversion factor is for chromium III.

Observations

General observations noted during the review of the data include;

1. The correlation between total and dissolved metal fractions is not consistent above a TSS concentration of 20 mg/L for the groundwater.
2. In the Liese Creek wells, a portion of the high TSS may be calcium.
3. There is high scatter in the Liese Creek data for manganese. The remaining groundwater data looks reasonably well correlated.
4. This analysis is based on review of available data. Metals that either had varying correlations or other scatter include As, Cd, Cu, Fe, Pb, and Zn.

References

Chapra, Steven C. (1997). Surface Water-Quality Modeling. The McGraw-Hill Companies, Inc. 1997

Environmental Protection Agency. (1999). National Recommended Water Quality Criteria--Correction. Office of Water. EPA 822-Z-99-001. April 1999

Environmental Protection Agency. (1995). Water Quality Standards Handbook, Second Edition. Office of Water . Government Institutes, Inc. Rockville, Maryland. 1995.

MEMORANDUM

DATE: October 19, 2001

TO: Karl Hanneman, Teck Pogo

FROM: Stephen Day, SRK

PROJECT: 1CT002.00

RE: **ELEVATED MANGANESE CONCENTRATIONS IN DEVELOPMENT
ROCK PILE DRAINAGE**

As you requested, this memorandum provides a discussion of the elevated manganese concentrations observed in the drainage from the non-mineralized development rock pile. These comments are based on the water quality monitoring dataset collected between August 1999 and July 2001.

BACKGROUND

Drainage samples are currently collected from four locations, three of which (SW25B to D) collect water from the non-mineralized rock pile at points along the drainage collection channel. Location SW25B is just upstream of the intake of the drainage collection culvert. Between May and August 1999, samples were also collected downstream of the final galvanized culvert receiving the drainage. These data are not considered further in this discussion because elevated zinc concentrations indicated that the culvert was affecting the water.

Location SW26 collects water from the mineralized rock pile.

GENERAL DESCRIPTION OF WATERS

Non-Mineralized Pile

The three SW25 locations show similar trends in water chemistry, and concentrations increase along the flow path from SW25D to SW25B. SW25B (upstream of culvert intake) was therefore considered representative of the cumulative flow from the non-mineralized rock pile.



The pH of the drainage was typically between 6 and 7 (with sporadic peaks above 7) in 1999 and 2000, but then increased to greater than 7 in 2001 (Figure 1). The dominant anion in the water from the non-mineralized pile has been nitrate throughout the three years of monitoring (Figure 2). Nitrogen as nitrate concentrations have slowly decreased from peaks near 1000 mg N/L (4428 mg NO₃/L) to peaks near 600 mg N/L (2657 mg NO₃/L), whereas sulfate peaks have increased from 1000 mg/L to 1500 mg/L. Alkalinity concentrations were low compared to nitrate and sulfate (near 60 mgCaCO₃/L in 1999 and 2000) but increased in 2001 to 135 mgCaCO₃/L). The dominant cations were calcium (200 to 1000 mg/L), magnesium (70 to 370 mg/L) and sodium (100 to 600 mg/L). Ion balances were typically better than ±10%.

Nitrogen as ammonia concentrations varied from less than 1 mg/L to peak concentrations near 30 mg/L.

Of the heavy elements, manganese had the highest concentrations (typical concentrations less than 30 mg/L, with a single peak at 46 mg/L). Concentrations have steadily decreased to current peaks of less than 9 mg/L (Figure 3). Concentrations of other heavy elements (including iron and arsenic) were less than 0.1 mg/L.

Mineralized Rock Pile

Drainage from the mineralized rock pile has had pHs between 7 and 8 throughout the monitoring period (Figure 1). Nitrate was again a significant anion (20 to 360 mg/L) (Figure 2), but at comparable concentrations to sulfate (10 to 870 mg/L). Neither nitrate nor sulfate concentrations have increased or decreased significantly. Calcium (11 to 440 mg/L), magnesium (3 to 120 mg/L) and sodium (2 to 280 mg/L) were again the dominant cations.

Through 1999 and 2000, manganese concentrations did not exceed 0.4 mg/L (Figure 3). In 2001, concentrations up to 3.9 mg/L were detected. Concentrations of other trace elements were of the same order-of-magnitude as the non-mineralized pile, but the mineralized pile had lower iron and copper concentrations, and higher arsenic concentrations.

DISCUSSION

Overall Processes

The presence of elevated nitrate concentrations in the drainage from both piles is believed to be a result of dissolution of ammonium nitrate-based explosives residues in the rock. Teck Pogo



indicated that relatively large amounts of explosives were not completely detonated during a period of adit development because the explosives had deteriorated in storage.

The explosives used during this period contained 75% ammonium nitrate, 4% sodium nitrate, 15% water, 4.5% mineral oil and 1.5% emulsifier.

The dissolution of ammonium nitrate releases ammonium ions which oxidize:



The bacterially-mediated oxidation reaction releases two protons (acidity) for each mole of nitrogen oxidized. The reaction requires a carbon source which could be from mineral oil in the explosives, or more likely bicarbonate (Grady and Lim 1980)¹ from carbonate minerals in the rock

Evidence that oxidation is occurring is provided by the shift in the ratio of nitrogen as ammonia to nitrogen as nitrate. In the explosives, the ratio is about 0.95 whereas in the drainage the average ratio is 0.016 (based on average N-NH₃ and N-NO₃ concentrations of 8.5 mg/L and 520 mg/L, respectively) indicating conversion of nitrogen in the form of ammonia to nitrate. Some ammonia may also be lost to volatilization but this reaction also releases a proton.

The acid released would be buffered by reaction with carbonate minerals in the rock. Evidence that this process is occurring is shown by the correlation of higher nitrate concentrations, slightly lower pH and higher calcium and magnesium concentrations in the non-mineralized pile rock drainage compared to the mineralized rock drainage. The similarity of sulfate concentrations for drainage from both piles indicates that sulfide oxidation is probably occurring at comparable rates in both piles. In this process, manganese probably originates from carbonates such as ankerite and siderite, which have been identified in these rocks (SRK 2000)².

Reaction of locally acidic pore waters (from ammonia oxidation) with calcium and magnesium carbonate releases bicarbonate, calcium and magnesium. Further acidification of bicarbonate forms carbonic acid, and carbon dioxide may be evolved. The resulting calcium, magnesium and nitrate leachate will not readily dissolve additional carbonate due to the high calcium concentrations. The alkalinity of the solution therefore remains low. As highly soluble explosives

¹ Grady, C.P.L and Lim, H.C. 1980. Biological Wastewater Treatment Theory and Applications. Marcel Dekker Inc. 963p.

² SRK(2000). Mineralogy of column leach residues, tailings and waste rock, Pogo Project Alaska. Prepared for Teck Corp. July 2000.



residues are leached from the rock, pH is expected to recover and alkalinity increases. These processes are important for understanding the behavior of manganese, as discussed below.

Manganese Geochemistry

The Eh-pH diagram for manganese indicates that the stable oxidation state of manganese is +2 for typical natural conditions (Figure 4). Under alkaline strongly oxidizing conditions, the +3 and +4 oxidation states are dominant as the common manganese oxides, but not as aqueous forms. In the absence of carbonate, Mn^{2+} is stable in solution for weakly alkaline solutions. If carbonate is present, manganese carbonate (rhodochrosite) is expected to be a controlling phase for manganese concentrations under alkaline conditions.

MINTEQA2 showed that drainage waters from the non-mineralized pile containing the highest manganese concentrations are close to saturation with respect to the manganese carbonate phase "rhodochrosite (c)" (SI = -0.22). Mineralized pile waters with manganese concentrations of near 4 mg/L, higher pH, slightly higher alkalinity and lower nitrate were calculated as being slightly over-saturated with respect to rhodochrosite (c) (SI = 0.22). Manganese carbonate therefore appears to be a potential control on manganese concentrations. The non-mineralized pile waters were well undersaturated with respect to calcite and dolomite. The mineralized pile drainage was close to equilibrium with calcite and dolomite.

MINTEQA2 was used to generate curves for manganese solubility as a function of pH using the chemistry represented by the above examples of the drainage. Curves were also generated for rhodochrosite (c) and rhodochrosite (c) in the presence of calcite in contact with pure water. These curves were superimposed on the monitoring data (Figure 5). These curves show that theoretically rhodochrosite is more soluble in the example of the non-mineralized pile drainage than in the mineralized pile drainage. This was due to the higher alkalinity and pH for the waters modeled. The solubility of rhodochrosite in pure water is lower than the drainage curves. The presence of calcite lowers the solubility of rhodochrosite further due to the resulting high alkalinity in the water.

If Pogo project area groundwater data are superimposed on the same graph, maximum manganese concentrations follow a curve of the same form as the calculated curves but between the ideal rhodochrosite and rhodochrosite with calcite curves. This is expected because groundwater is in contact with rocks containing carbonate which would be expected to control water chemistry.



These findings indicate that the drainage from the non-mineralized pile is atypical because it is affected by dissolution of explosives residues which oxidize and probably create locally acidic conditions. This in turn leads to lower alkalinity of water in the pile and higher solubility of manganese carbonate. The observed decrease in manganese concentrations and the similarity of the chemistry of drainage from the mineralized and non-mineralized piles in 2001 indicates that explosives residues have been significantly leached and that ammonia oxidation is no longer causing extensive acidification within the pile. Drainage from the mineralized rock pile probably provides a better indication of typical manganese concentrations in waste rock leachate though nitrate concentrations were also elevated in the drainage. Maximum concentrations to date (about 4 mg/L) are consistent with maximum manganese concentrations observed in groundwater (about 5 mg/L).

IMPLICATIONS TO WATER CHEMISTRY PREDICTIONS

Previous predicted manganese concentrations for components of the Pogo project were:

- Dry Stack seepage – 19 mg/L
- Dry Stack runoff – 2.4 mg/L
- Mineralized Rock Stockpile seepage – 0.98 mg/L.

The drystack seepage value was based on a mass-weighted concentration. MINTEQ indicated that a concentration of 3 mg/L would be reasonable because manganese carbonate would be expected to control concentrations. However, the mass weighted concentration was used because manganese concentrations higher than 19 mg/L had been observed in the non-mineralized stockpile drainage. Based on the above findings on the effects of explosives residues, 3 mg/L is probably a more reasonable concentration for the dry stack seepage.

The dry stack runoff value was based on a typical maximum value of 29.7 mg/L diluted using the method presented previously. If a groundwater maximum value of 4.75 mg/L is used in the calculation, the dry stack runoff concentration becomes 0.38 mg/L.

The mineralized rock stockpile seepage concentration was obtained from the mass weighted concentration and is not affected by these findings. The concentration is lower than observed in the mineralized waste rock development pile drainage but is of the same order-of-magnitude indicating that the predictions are reasonably close to observed conditions.



CONCLUSION

The presence of explosives residues believed to have been caused by deterioration of explosives in storage resulted in a dominance of nitrate in the non-mineralized pile drainage. The resulting effects appear to be:

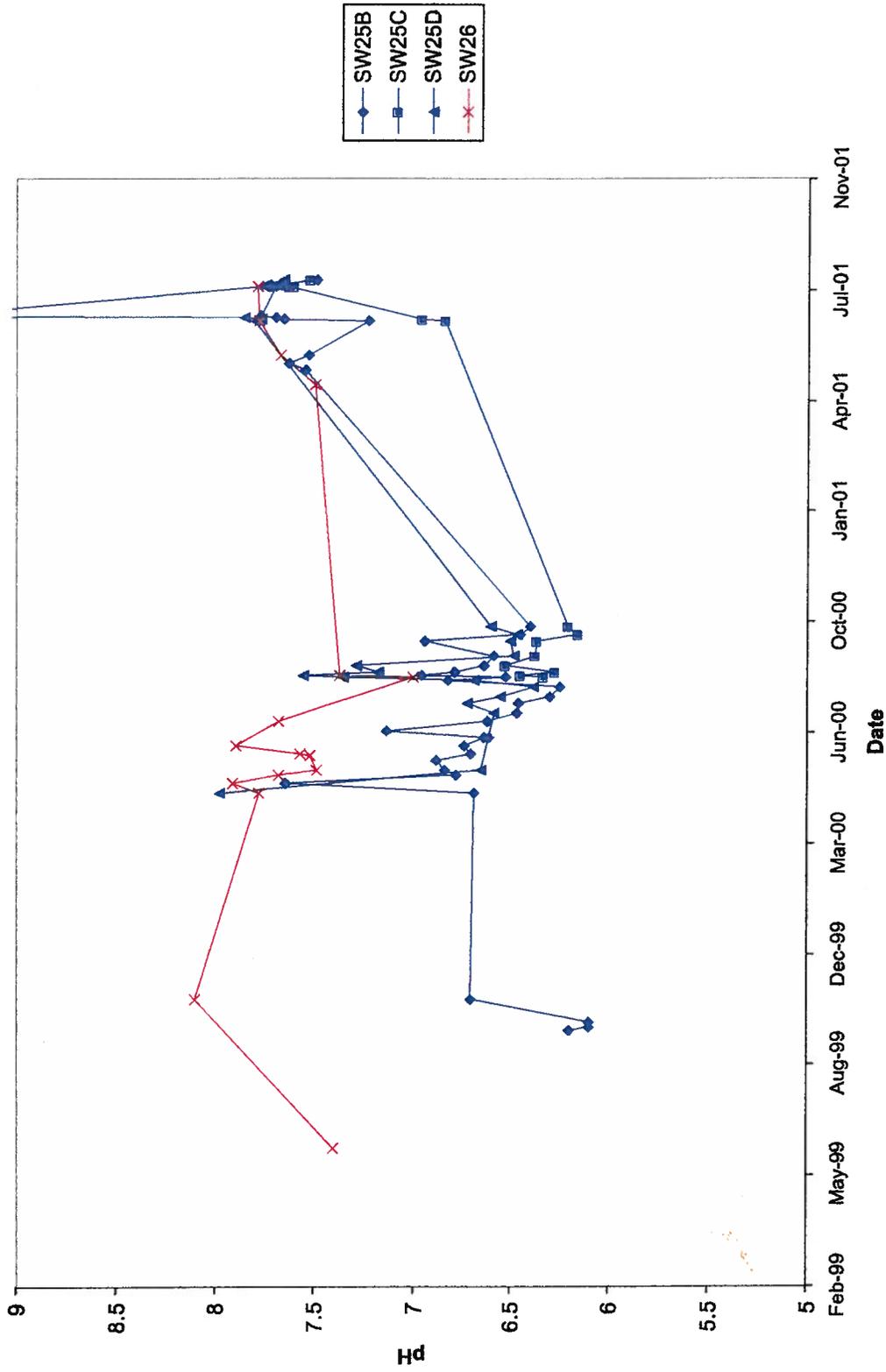
- Locally acidic leaching due to release of acidity by oxidation of ammonia.
- Drainage chemistry for the non-mineralized pile that raises the solubility of manganese carbonate.

The drainage chemistry observed from the non-mineralized pile should not be used as an indicator of drainage chemistry under full scale conditions. The mineralized pile has also been influenced by explosives residues but to a lesser degree.

The observations imply that drainage chemistry predictions for the dry stack seepage and runoff based on the non-mineralized pile drainage (as presented in the Water Management Plan) are probably over-estimates. Suggested revised concentrations (compared to original concentrations) are:

- Dry Stack seepage – 3 mg/L (19 mg/L)
- Dry Stack runoff – 0.38 mg/L (2.4 mg/L).

Stephen Day, M.Sc.
Principal Geochemist



ELEVATED Mn IN DEVELOPMENT ROCK PILE DRAINAGE

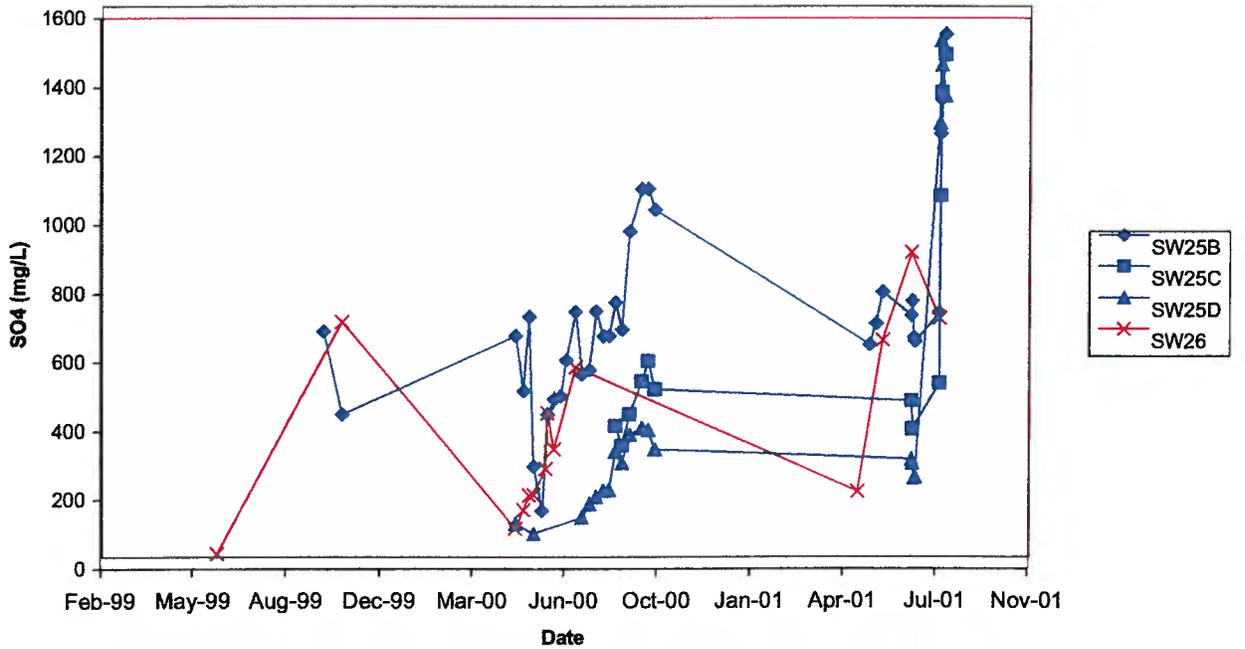
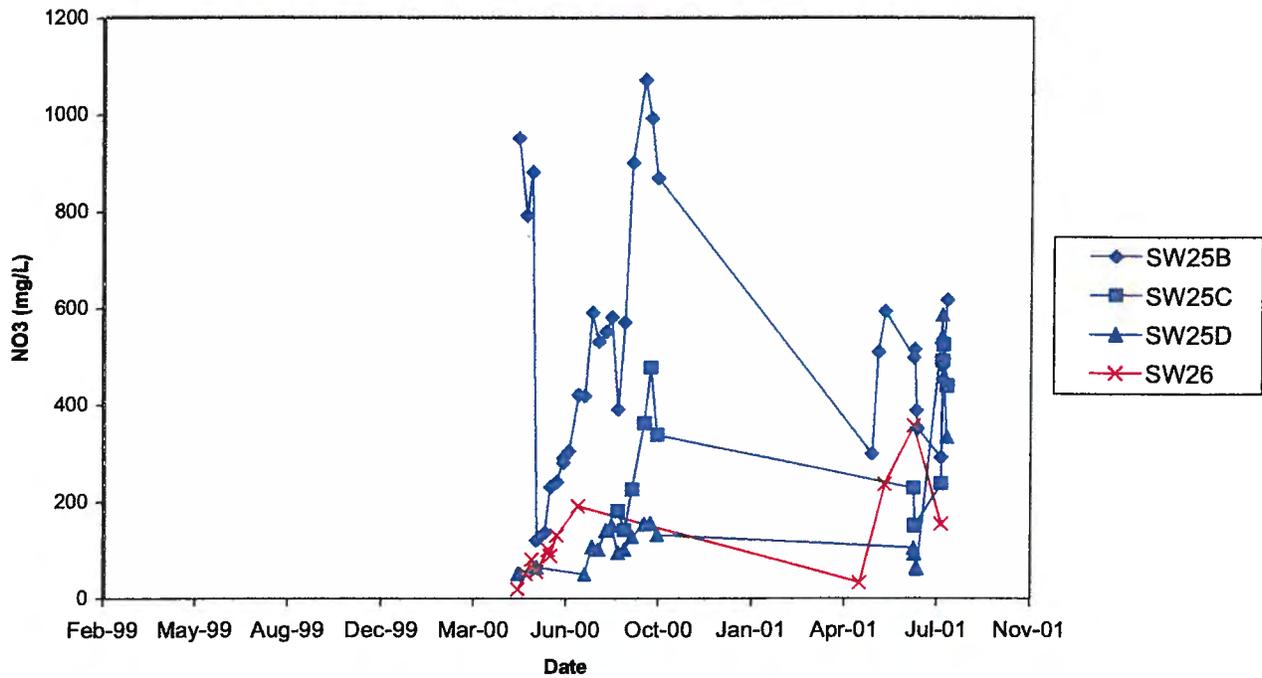
pH(t)

Project: 1CT002.00 Date: Oct, 2001 Approved: Figure: 1

SRK Consulting
Engineers and Scientists

TECK-POGO INC.



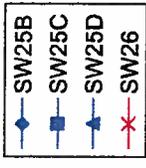
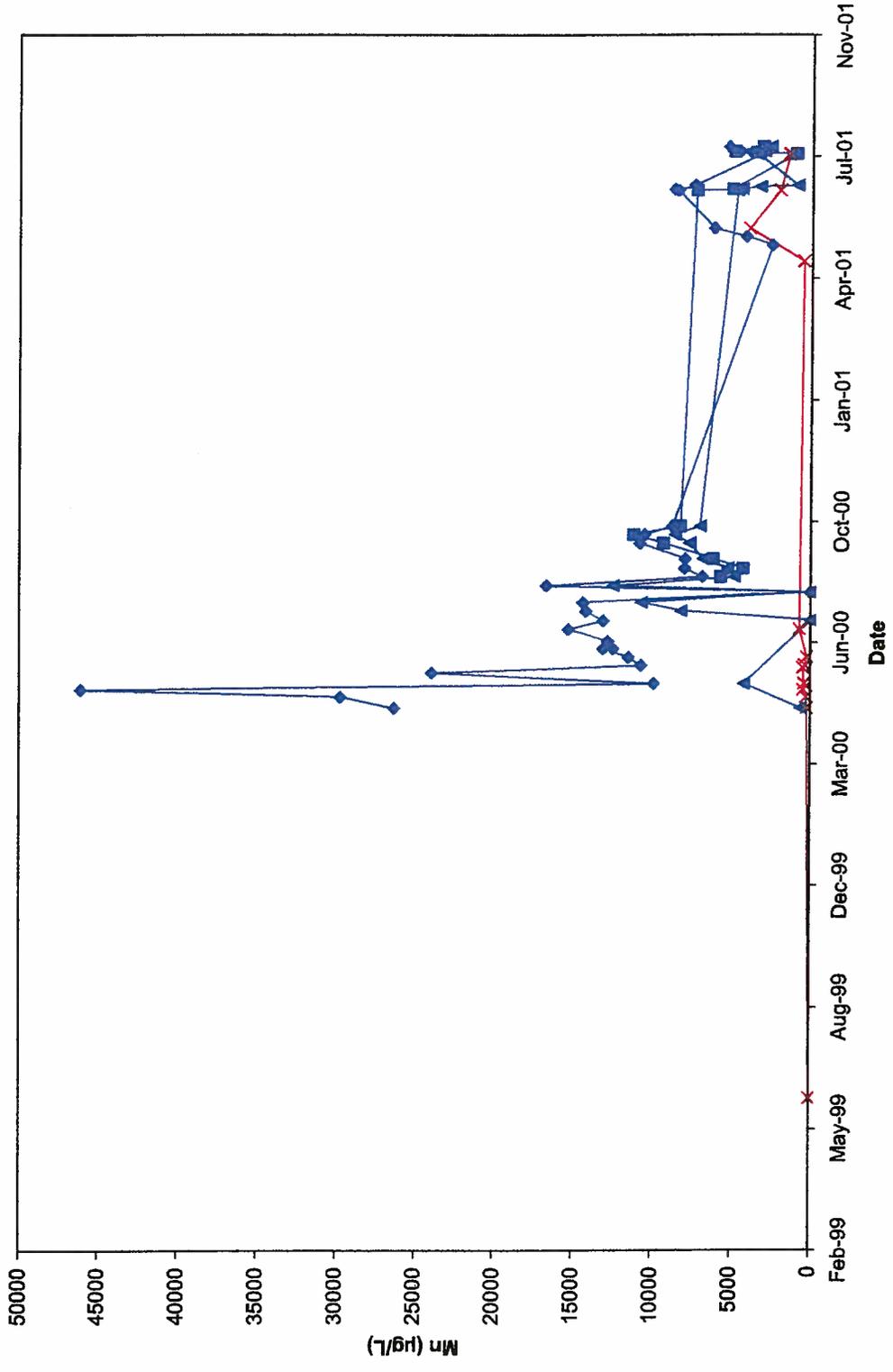
ELEVATED Mn IN DEVELOPMENT ROCK PILE DRAINAGE

NO3(t) and SO4(t)



TECK-POGO INC

Project 1CT002.00	Date Oct, 2001	Approved	Figure 2
----------------------	-------------------	----------	-------------

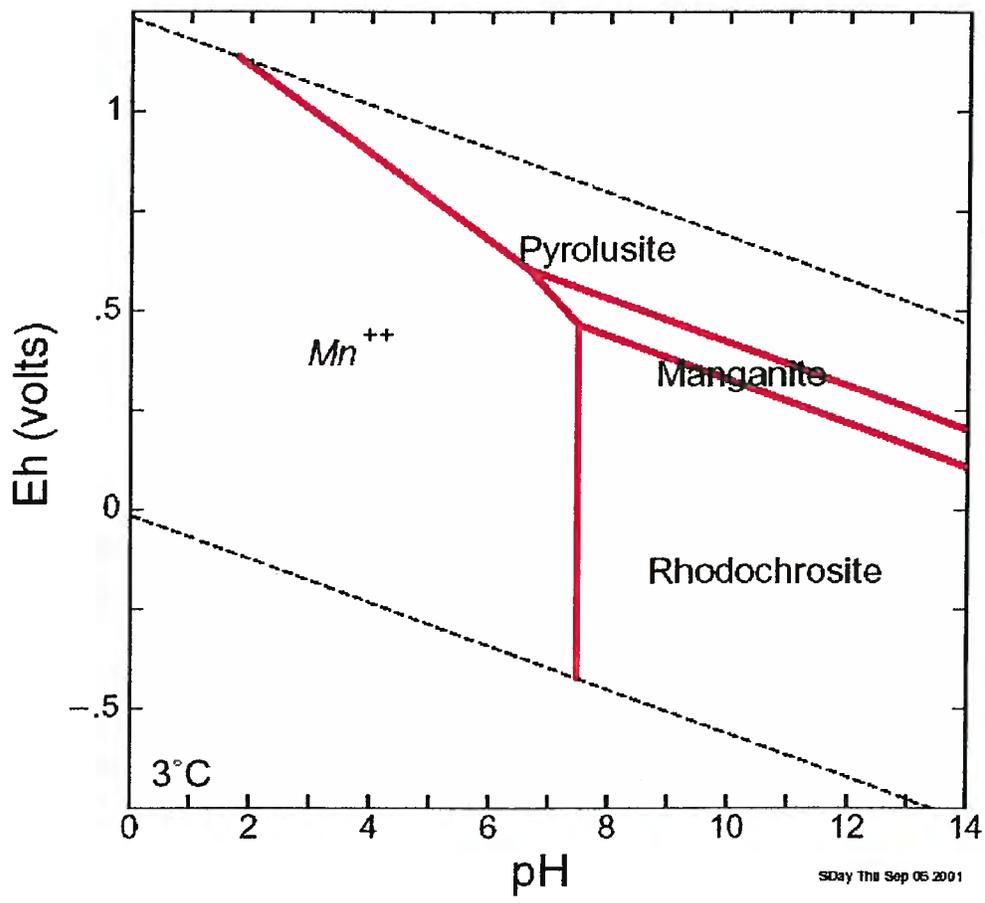


**ELEVATED Mn IN DEVELOPMENT ROCK
PILE DRAINAGE**

SRK Consulting
Engineers and Scientists

TECK-POGO INC.

Mn(t)	
Project	Date
1CT002.00	Oct, 2001
Approved	Figure
	3



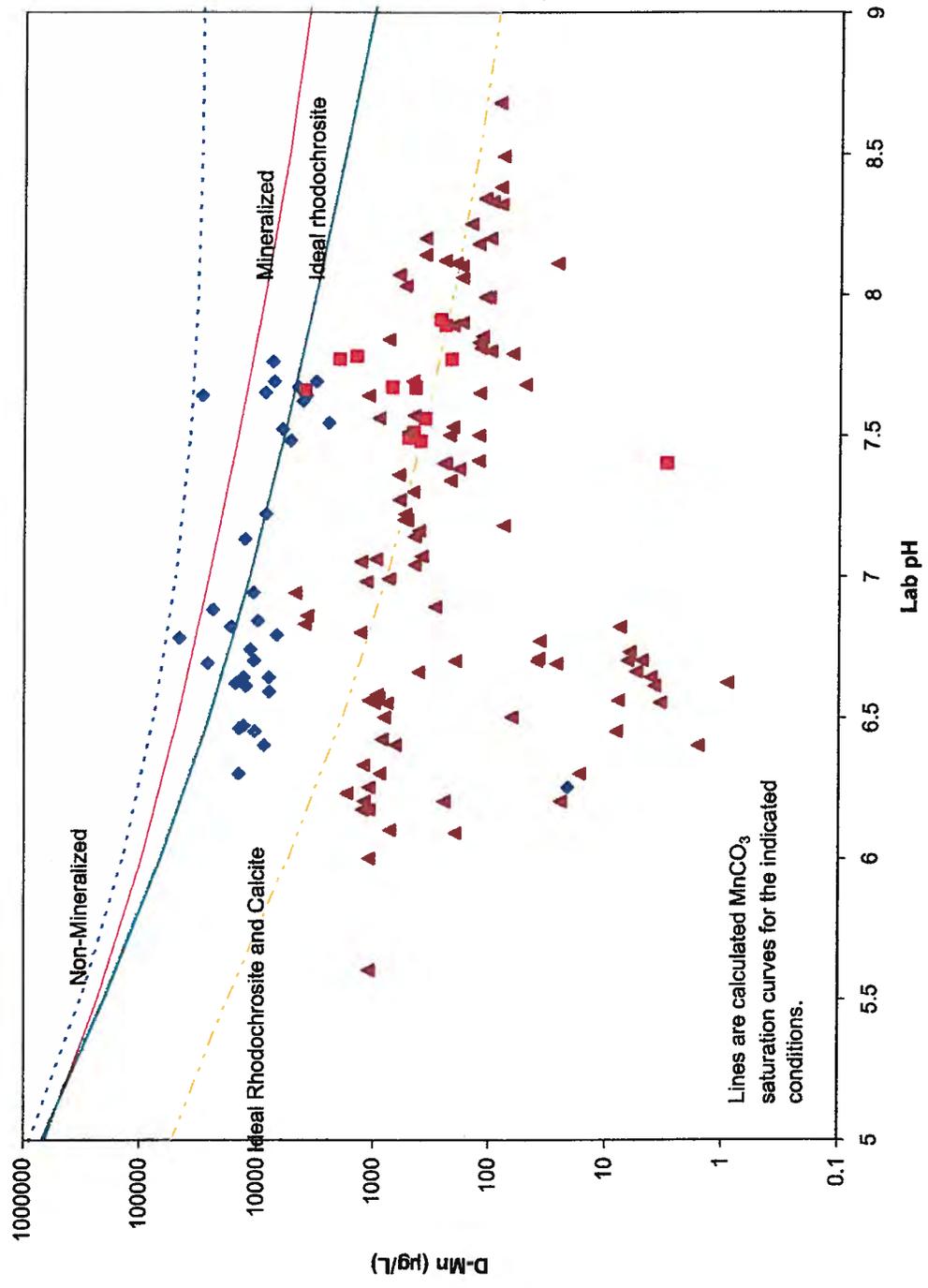
ELEVATED Mn IN DEVELOPMENT ROCK PILE DRAINAGE

Eh vs pH Diagram for Manganese



TECK-POGO INC

Project 1CT002.00	Date Oct, 2001	Approved	Figure 4
----------------------	-------------------	----------	-------------



**ELEVATED Mn IN DEVELOPMENT ROCK
PILE DRAINAGE**

Mn(t)	
Project	Date
1CT002.00	Oct, 2001
Approved	Figure
	5

