Attachment 2

Water Chemistry of Mine Drainage Water and TDRSA Water

TECHNICAL MEMORANDUM

DATE: 7 November 2005

TO: **Kennecott Eagle Mining Company**

FROM: Mark J. Logsdon (Geochimica)

SUBJECT: **Water Quality in Underground Mine During Operational Conditions**

BACKGROUND

The water-quality model for the underground mine in the Phase II Geochemistry Report (Geochimica, 2005) is for an underground mine at full life-of-mine, *without* backfill. This is the hydraulic basis of the Golder Associates flow values (180 Usgpm total inflow) used in the model. At the end of mining almost all ore-grade rock will have been removed, so the water quality impacts are controlled significantly (although not exclusively) by the surface-area-weighted geochemical reactivity of unmineralized intrusives and the country rock that are exposed in the walls of the underground workings.

During the operational period, once the development workings have advanced to the ore zones, there will be ore-grade rock exposed in the rib and back, and because of the relatively high reactivity of the ore-grade rocks compared to the unmineralized zones, this material is expected to exert a greater impact on water quality during the operational period than the small volumes left in the workings at the end of mining. This memorandum describes and produces the results for a water-quality model of the operational period that assumes, very conservatively, that 50% of all the rock exposed in the underground workings is Massive Sulfide Unit ore.

why is this conservative?

TECHNICAL APPROACH

The initial model for water quality in the underground mine during operations has been coordinated with the current hydrogeological investigations in order to develop a common basis for wall-rock surface area and total mine inflows. The crucial concern is what will be the impacts of mine walls on water quality *during* mining. This is the problem addressed by this evaluation, using data updated through the most recent laboratory reports in August 2005.

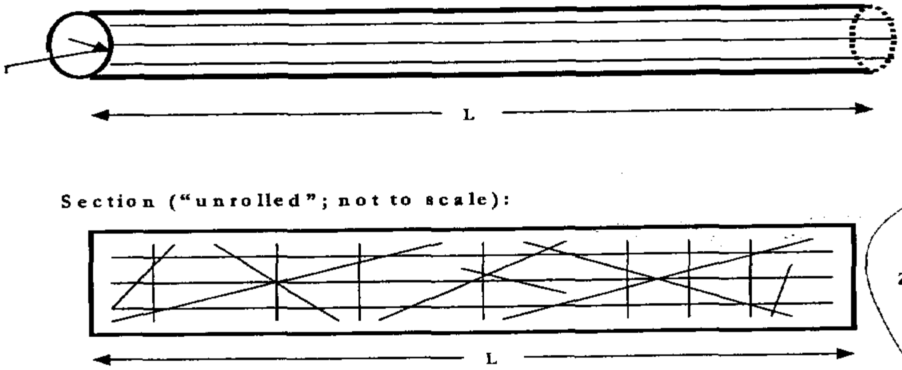
The technical approach to the problem is identical to that used for the mine at end of operations, as documented in the Phase II Report(Geochimica, 2005). The conceptual model for the physical system of the underground workings can be considered in terms of a unit portion of tunnel-like openings. The open workings have some geometric surface area that can be calculated simply from the dimensions of the workings. For the purposes of illustration, the tunnel is shown as circular in cross section, but this is not essential, and the geometric surface areas used in the model are derived from computations provided by the mining-engineering team that is planning the underground system. Because of natural fracturing and faulting and the effects of blasting, the total available surface area of the mine walls will be much greater than the apparent geometric surface area. Morin and Hutt (1997) have compiled data showing that available surface area on mine walls may vary from about 20 to about 125 times the apparent geometric surface area. At this time, there are no data on blasting effects, so we have made a reasonably conservative assumption that the actual surface area will be 100 times the apparent geometric surface area. The basic conceptual model is illustrated in Figure 1 (identical to Figure 4-6 in the Phase II Report, Geochimica, (2005).

The approach to developing a geochemical estimate is the same asthat used in the Phase II Report for both the surface stockpile and the underground workings at the end of mining. The total surface area is calculated as described in the paragraph above (i.e., adjusted to account for surface roughness and fracturing), and the inflow water volume is taken from the hydrogeologic model. Based on the 15-Feb 05 mine plan, Golder Associates Ltd. developed surface areas for the exposed walls and calculated mine-water inflow. Based on their results, the total geometric[1] surface area exposed in the underground will be 1.29 E+05 m^2. For the analysis of the operational condition, we have assumed that the system is to be evaluated when the total surface area is 50% of the final surface area, representing a period in which the maximum amount of ore-grade rock would be exposed. The long-term inflow for the entire open workings, based on the Golder Associates hydrologic modeling, is assumed to be 981 m^3/d (180 USgpm).

Figure 1 Conceptual Model for Controls on Leaching in Underground Workings

Schematic Cylindrical Tunnel (not to scale):

Section ("unrolled"; not to scale):



$2\pi r$

Surface Area Model:

Geometric Surface Area: (2*r) * L

Roughness and Fracture Adjustment: 20 – 120 * Geometric (Morin & Hutt, 1997) – Choose Factor of 100

LOM Geometric Surface Area: 1.29 E+05 m^2 (GAI, 2005)

Model Underground Surface Area: 1.29 E+07 m^2

What is LOM?

The computational model begins with a mass-balance calculation that converts the observed leachate chemistry of test charges of representative rock samples (see below) to leach rates per unit surface area per unit time. The model then takes the total surface area of exposed rock (of each type) and computes an apparent mass leached. This mass is divided by the total inflow volume, producing an average concentration in the aqueous phase. Finally, this total mass-balance approach is provided as input to a numerical geochemical model [in this case REACT (Bethke, 1996; 2005)] to compute possible solubility or other geochemical controls. For example, if one assumes that there will be breathable air in the mine workings during operations (which of course must be the case for worker safety), then it is reasonable to assume that the solutions will equilibrate with atmospheric oxygen. This modeling assumption implies that dissolved iron would be predominantly in the ferric state, allowing, for a high SO_4 solution to precipitate a ferric hydroxysulfate like jarosite [$KFe(SO_4)_2(OH)_6$]. This, in turn limits the total dissolved concentration of Fe and SO_4 (also K).

LEACHATE CHEMISTRY IN MODEL INPUT

In the conceptual model for the operational mine, one can usefullydistinguish two parts of the system:

- The access developments (modeled as 50% unmineralized intrusive (itself 50% low-S and 50% high-S) and 50% country rock)
- The ore faces (modeled as Massive Sulfide Unit (MSU), the maximum-reactivity ore-grade rock).

The access development will have been exposed to the underground air for a considerable period of time (several years), whereas the ore faces will be mined expeditiously and therefore represent short-term leaching. To represent this conceptualization, we take long-term (40 – 70 week data) from the column-leach tests for the unmineralized intrusives and country rock. We take short-term (i.e., maximum concentration) column-test data for the MSU, the arithmetic average of the first fiveweeks of leaching. Input data, updated since Geochimica (2005) to include the most recent results are summarized in Table 1. When input data are less than the limit of detection, they are entered in the model as ½ the limit of detection.

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[1] As explained in Figure 1, the total surface area exposed to weathering is assumed at this stage to be 100 times the geometric area to account for roughness and fracturing (Morin and Hutt, 1994). Therefore, the surface area used for calculations of the Life-of-Mine model of water-quality impacts is 1.29 E+07 m^2.

Table 1 Input Leachate Chemistry for UG Mine during Operations[2] (mg/L except pH in su and Alkalinity in mg $CaCO_3$/L)

PARAMETER	**VALUE**		
	Country Rock	**Intrusive**	**MSU**
pH	6.1	7.1	3.8
Alkalinity	6.8	8.4	0
Total Dissolved Solids	145	164	1893
Calcium	12	9	7
Magnesium	8	16	11
Sodium	2	3	7
Potassium	2	6	4
Bicarbonate	5.5	6.9	nil
Sulfate	72	82	1054
Cl	0.5	0.6	15.2
F	0.07	0.09	0.88
Aluminum	0.32	0.62	0.27
Antimony	0.0004	0.0002	0.165
Arsenic	0.01	0.001	0.16
Barium	0.007	0.015	0.026
Boron	0.03	0.13	0.58
Cadmium	0.0020	0.0021	0.093
Chromium, Total*	<0.0005	.0009	<0.065
Cobalt	0.111	.053	7.15
Copper	0.40	1.13	0.49
Lithium	0.007	.009	0.26
Lead	0.007	.019	0.077
Iron	2.51	4.77	233
Manganese	0.48	0.17	9.10
Mercury *	<0.00005	<0.00008	<0.00005
Molybdenum*	<0.0004	<0.0001	<0.027
Nickel	0.63	1.65	365
Selenium	0.0005	0.0032	0.205
Silver*	<0.00002	<0.00001	<0.009
Strontium	0.104	.09	0.084
Thallium*	<0.0001	<0.0001	<0.165
Vanadium	<0.001	.0015	0.074
Zinc	0.32	0.18	1.16

*: Values less than detection entered in model as ½ detection limit for calculational purposes

OPERATIONAL MODEL RESULTS

The results of the equilibrated model are presented in Table 2 for the same parameters that are shown for input waters in Table 1. In cases where input data were less than detection, output data are modeled using a value of one half the largest detection limit in the set.

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[2] The reported values are observed concentrations form the Phase I and Phase II column tests. Background concentrations of groundwater infiltrating the workings are addressed by others.

Readers are directed to the Phase II Report for further discussion about the modeling approach and assumptions.

Table 2 Incremental Water Quality for UG Mine during Operations[3] (mg/L except pH in su and Alkalinity in mg $CaCO_3$/L)

Parameter	**Value**
pH	3.6
Alkalinity	0
Total Dissolved Solids	184
Calcium	4
Magnesium	5
Sodium	1
Potassium	1
Bicarbonate*	nil
Sulfate	110
Cl	1.58
F	0.098
Aluminum	0.088
Antimony	0.016
Arsenic	0.017
Barium	0.004
Boron	0.07

Parameter	**Value**
Cadmium	0.01
Chromium, Total**	0.0045
Cobalt	0.72
Copper	0.15
Lithium	0.026
Lead	0.009
Iron	6.4
Manganese	0.95
Mercury **	0.00004
Molybdenum**	0.013
Nickel	36.4
Selenium	0.02
Silver**	0.0045
Strontium	0.02
Thallium**	0.008
Vanadium	0.007
Zinc	0.15

* Although there is no titratable alkalinity, the model assumes equilibrium with atmosphere CO_2, so the nominal bicarbonate concentration is 0.75 mg/L, which is below analytical detection.

** Input values for source rocks are less than detection, assumed to be ½ detection limit.

DISCUSSION

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[3] The reported values are incremental concentrations due to wall-rock leaching. Background concentrations of groundwater infiltrating the workings (evaluated by others) would be additive to these.

Table 2 shows that important aspects of the water chemistry in the underground mine during operations will be dominated by the reactivity of the ore that is exposed during operations. However, the operational concentrations do not approach (in fact are about one-tenth) the maximum concentrations seen in the column tests themselves. This is primarily because the effective water:rock ratio in the mine will be very much greater than in the column tests. Secondarily, the ore provides only ½ the reactive mass in the mine model, and there are reductions in some important water-quality parameters due to common solubility controls.

The values shown in Table 2 derive from the specific assumptions (e.g., August 2005 data; 50% MSU; mine volumes; groundwater inflow rate) used in the model. If other modeling assumptions had been used, the concentrations would have differed. However, provided there is a significant fraction of exposed surface that is ore-grade mineralization, the general outcome of the modeling (low pH, elevated concentrations of some metals and sulfate) would be repeated. Thus, we consider that the model successfully represents the major chemical conditions expected in the underground mine during operations so that water-management and treatment options can be evaluated.

The two parameters that are noticeably different from common water-quality criteria are pH and nickel. The low pH derives from the acidity released by the oxidation of sulfide minerals. This is high-grade nickel ore, and nickel will be released during the incipient oxidation of pentlandite and pyrrhotite in the ore-grade rock exposed temporarily during mining. Unlike some other transition metals (e.g Fe), there is no plausible solubility control for Ni once it has been released, so its elevated concentration during operational conditions is entirely expected.

Finally, groundwater flowing into the workings will have reacted with surrounding bedrock over many of years and will already have a solute load that makes it different chemically from de-ionized water used in the column tests. Therefore, the outcomes from this modeling exercise need to be understood as incremental concentrations, the values of which need to be added to the likely groundwater chemistry in order to estimate the water that actually would report to sumps in the underground at the end of mining.

REFERENCES

Bethke, C.M. 1996. *Geochemical Reaction Modeling.* New York: Oxford University Press.

Bethke, C.M., 2005. *The Geochemist's Workbench, Release 6.0, Users' Guide.* Urbana-Champagne, IL: University of Illinois.

Geochimica, Inc., 2005. Geochemistry Phase II. Report of Investigations for Eagle Project, Marquette County, Michigan. Contractor Report to Kennecott Eagle Mining Company.

Morin, K.A. and N.M. Hutt, 1997. *Environmental Geochemistry of Minesite Drainage: Practical Theory and Case Studies.* Vancouver, BC: MDAG Publishing.

TECHNICAL MEMORANDUM

DATE: 7 November 2005

TO: **Kennecott Eagle Mining Company**

FROM: Mark J. Logsdon (Geochimica)

SUBJECT: **Water Quality from the Development Rock Storage Pad During Operations**

BACKGROUND

The water-quality model for the Development Rock Stockpile in the Phase II Geochemistry Report (Geochimica, 2005) is for a full-scale pad that contains the entire volume of development rock that will ultimately be placed there before backfilling begins. The model results in the report assume no amendment of the stockpile with limestone. The hydraulic portion of the analysis is based on effective infiltration of 20% of the average annual precipitation, estimated from long-term data at Marquette. This memorandum describes and produces the results for a water-quality model for the Development Rock Stockpile, including the expected geochemical effects of amending the development rock with limestone to extend the lag period for the onset of possible acidification. This analysis uses the most recent and complete data for simulated weathering in column tests, received in August 2005.

TECHNICAL APPROACH

Based on current mine planning, KEMC will produce a total of approximately 379,000 tonnes[1] of development rock that will be stored on a rectangular pad with plan dimensions of approximately 200 m by 100 m for periods of up to seven years at the surface. Mid-way through the mine life (beginning about Mine Year 4), development rock will be returned to the underground workings as cemented backfill, and by Mine Year 7 all development will be backfilled into the deep underground.

At this stage of the geochemical investigations, one can develop a simple model of a development rock stockpile undergoing surficial weathering in the vadose zones with rinsing by infiltrating meteoric waters (rain and snowmelt) typical of average annual precipitation in the Upper Peninsula. One uses the results of the ongoing kinetic tests and available data on site hydrology to characterize the system behavior. The basic conceptual model for a development-rock stockpile undergoing surficial weathering is illustrated in Figure 1.

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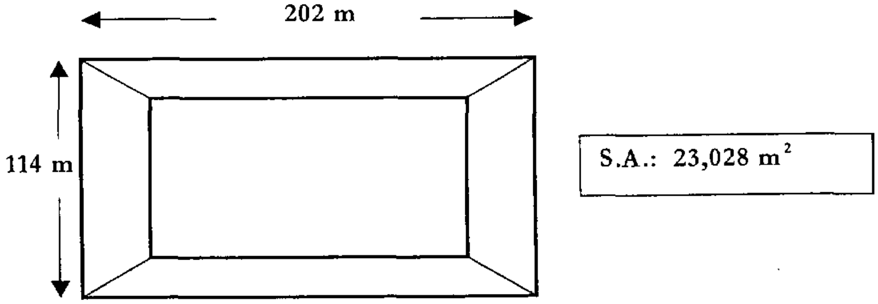
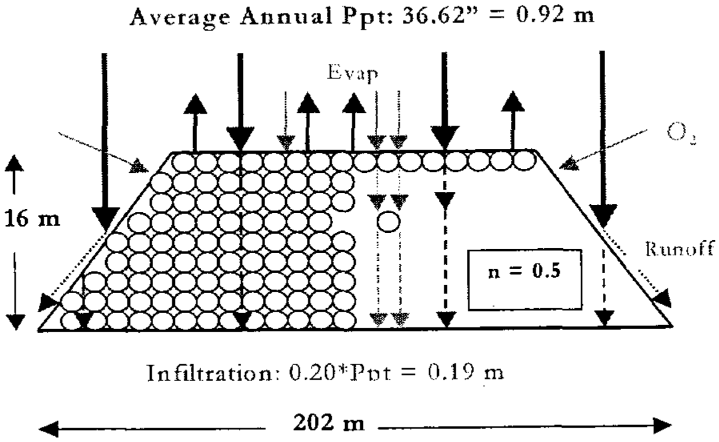
[1] Tonne is a metric ton, 1000 kg, or 2200 pounds (1.1 US short ton).

Figure 1 Conceptual Model for a 379,000-tonne Development-Rock Stockpile

Plan (not to scale):



Section (not to scale):



The model assumes that the development rock stockpile would be an equal mixture of Intrusive and Country Rock that has been well represented by the Phase I and Phase II kinetic tests. For each rock group, we have considered high-sulfur and low-sulfur samples from the kinetic tests. In this most recent version, we have used data from both the Phase I and II tests because Phase II samples represent a higher-sulfur range and show more sign of reactivity. Sulfide oxidation in the stockpile occurs because O_2 migrates into the pile; this model does not depend on the specific mechanism of gas transport, but does assume that oxygen is not limiting. Water needed for the reactions and also for transport through the pile comes from annual precipitation that is intercepted on the plan surface area of the pile. The average-annual precipitation value, 36.6 inches (0.92 m) per year is the long-term average from Weather Service records at Marquette. For the calculation, we assume that 20% of average annual precipitation infiltrates through the pile on a steady-state basis. The bulk porosity of the pile is assumed to be 50%, and the mean diameter of rock particles in the pile is assumed to be 10 cm (0.1 m). Leach rates for weighted average country-rock and intrusives are calculated in terms of mass per unit surface area per unit time from the kinetic column tests, with the surface areas calculated from the sieve analyses of sample splits measured before the leach testing begins. Because the rock on the full-scale pad will include much more coarse rock than can be tested in the columns, the column leach rates are then scaled to field conditions for the surface area of the total pad. The spreadsheet model used to calculate expected water quality is provided in Appendix F to Geochimica (2005), and also includes a thermodynamically-based geochemical model of the estimated solution to account for precipitation and sorption reactions that could be expected within the pile prior to discharge.

The basis for the mass-balance calculation is as follows:

- Compile basic data on the column tests (mass of charge; particle-size distribution of the fragments; weekly effluent volumes)
- Compile column test effluent chemistries for a representative period over which chemistries are essentially constant.. Because natural rain water is compositionally close to de-ionized water, the conceptual model of the column tests is apt, and the effluent chemistries can be taken as representative, for the experimental conditions of the columns.
- Convert the observed concentrations to release rates in units of mass/unit surface area/unit time.
- Calculate the total surface area for rock in the stockpile based on a model of particle size distribution for the rock.
- Calculate the equivalent annual mass released from weathering of the full stockpile, assuming that release rates per unit surface area in the columns is equivalent to release rates per unit surface area of the full-scale pile.
- Calculate the volume of infiltration expected to report through the pile on an annual basis.
- Divide the mass released by the amount of water available to carry the mass. This has units of concentration (Mass solute/Volume solvent).
- Finally, evaluate the mass-balanced chemistry in terms of equilibrium with geochemically credible phases (e.g., gypsum; ferrihydrite), and the pH is calculated for the resultant solution as controlled with atmospheric CO_2.

LEACHATE CHEMISTRY IN MODEL INPUT

In the conceptual model for the Development-Rock Stockpile, there is no ore-grade rock (Massive Sulfide Unit or Semi-Massive Sulfide Unit). Based on advice form mine planners, the total access development is modeled as 50% unmineralized intrusive and 50% country rock. Both types of development rock are assumed to include both high-sulfur and low-sulfur rocks, represented by the range of samples that have been weathered in KEMC's ongoing column-test program. To represent this conceptualization, we take long-term (40 –

70 week) data from the most recent column-leach tests for the unmineralized intrusives and country rock. When input data are less than the limit of detection, they are entered in the model as ½ the limit of detection.

Table 1 Input Leachate Chemistry for Development-Rock Stockpile during Operations (mg/L except pH in su and Alkalinity in mg $CaCO_3$/L)

Parameter	**Values**	
	Country Rock	**Intrusive**
pH	6.1	7.1
Alkalinity	6.8	8.4
Total Dissolved Solids	145	164
Calcium	12	9
Magnesium	8	16
Sodium	2	3
Potassium	2	6
Bicarbonate	5.5	6.9
Sulfate	72	82
Cl	0.5	0.6
F	0.07	0.09
Aluminum	0.32	0.62
Antimony	0.0004	0.0002
Arsenic	0.01	0.001
Barium	0.007	0.015
Boron	0.03	0.13
Cadmium	0.0020	0.0021
Chromium, Total*	<0.0005	.0009
Cobalt	0.111	.053
Copper	0.40	1.13
Lithium	0.007	.009
Lead	0.007	.019
Iron	2.51	4.77
Manganese	0.48	0.17
Mercury *	<0.00005	<0.00008
Molybdenum	<0.0004	<0.0001
Nickel	0.63	1.65
Selenium	0.0005	0.0032
Silver*	<0.00002	<0.00001
Strontium	0.104	.09
Thallium*	<0.0001	<0.0001
Vanadium	<0.001	.0015
Zinc	0.32	0.18

*: Values less than the detection limit are entered in model as ½ detection limit for calculational purposes

DEVELOPMENT-ROCK STOCKPILE MODEL RESULTS

The results of the equilibrated model are presented in Table 2 for the same parameters that are shown for input waters in Table 1. In cases where input data were less than detection, output data are modeled using a value of one half the largest detection limit in the set. For additional detail and discussion of specific geochemical reactions, readers are directed to the Phase II Geochemistry Report (Geochimica, 2005).

Table 2 Modeled Water-Quality for Development-Rock Stockpile – No Limestone Addition (mg/L except pH in su and Alkalinity in mg $CaCO_3/L$)

Parameter	**Value**	**Parameter**	**Value**
pH	7.1	Cadmium	0.0002
Alkalinity	66	Chromium, Total**	0.00045
Total Dissolved Solids	865	Cobalt	0.0008
Calcium	79	Copper	0.29
Magnesium	88	Lithium	0.019
Sodium	19	Lead	0.0004
Potassium	29	Iron	0.005
Bicarbonate	54	Manganese	0.005
Sulfate	575	Mercury **	0.00004
Cl	11	Molybdenum	0.0015
F	2	Nickel	8.33
Aluminum	0.001	Selenium	0.004
Antimony	0.0004	Silver**	0.00005
Arsenic	0.083	Strontium	0.50
Barium	0.03	Thallium**	0.00005
Boron	0.58	Vanadium**	0.00075
		Zinc	1.90

**: Input values less than the detection limit were modeled as ½ detection limit for calculational purposes

The mass-balanced values were adjusted in the geochemical model REACT (Bethke, 2005) for solubility controls. Geochemically credible phases that affect the solution chemistry included barite [Ba], ferrihydrite [Fe], gibbsite [Al], manganite [Mn], malachite [Cu], and amorphous silica [Si]. The model considered the solution for solubility with gypsum [$CaSO_4.2H_2O$], but found the waters somewhat undersaturated. The geochemical model predicts extremely low values (<0.001 mg/L for Al, Fe and Mn). For this evaluation, we have adjusted them slightly upward to account for sub-part per million levels of these metals observed in almost all natural groundwaters that have TDS values greater than a few hundred mg/L.

No pure-metal solubility controls in low-temperature waters were found for trace metals and metalloids such as As, Sb, B, Cd, Cr, Pb, Ni, Se, or Zn. Because the input values for Hg in both rock types are less than detection, the model returns less-than-detection values for Hg; concentrations of other trace metals (Cr, Ag, Tl and V) also are limited by the non-detect values in one or both of the input parameters. It is expected that there could be sorption controls on some of these metals and metalloids, especially As and Hg. Near pH 7, sorption of B, Cd, Ni, Se and Zn would be slight; relatively little is known about the sorption behavior of Sb, but it probably behaves somewhat like As. Although the model solution is saturated in ferrihydrite, the amount of ferric hydroxide precipitated is so small that sorption onto the ferric hydroxide precipitated would be minimally effective at pH near 7.

DISCUSSION

General Results

Table 2 shows that important aspects of the effluent chemistry in the stockpiled development rock, especially the concentrations of sulfate and trace metals, will be dominated by the reactivity of the minor to trace levels of sulfides that are exposed during operations. The mineralogy of these rocks is described in detail in the Phase II Geochemistry Report (Geochimica, 2005).

The specific values shown in Table 2 derive from the specific assumptions (e.g., 50% proportions of both country rock and intrusives; volume of development rock on the stockpile; infiltration rate) used in the model. If other modeling assumptions had been used, the concentrations would have differed. However, the general outcome of the modeling (neutral pH; slightly elevated sulfate; part-per million concentration for Ni) would be repeated for any plausible combination of assumptions that are based on the existing test data. Thus, we consider that the model successfully represents the major chemical conditions expected in the development rock storage pad so that water-management and treatment options can be evaluated.

Column-test data indicate that the neutralization potential measured in the static tests (Phase II report, Geochimica [2005]) are sufficiently available to overcome the acidity, averaged over the tested samples, released from oxidation of the minor to trace amounts of sulfide minerals for generally extensive periods. The parameters that are noticeably different from common water-quality criteria are boron, sulfate and nickel. Boron derives from leaching of the intrusive rocks, all of which have elevated concentrations of B. The slightly elevated sulfate is due to oxidation of pyrite and pyrrhotite in both rock types, and nickel will be released during the incipient oxidation of sulfide minerals (probably largely pyrrhotite) in the development rock. Unlike some other transition metals (e.g. Fe), there is no plausible solubility control for Ni once it has been released, so its elevated concentration during operational conditions is entirely expected.

Unlike groundwater flowing into the workings that will have reacted with bedrock over thousands of years and will already have a solute load, the water infiltrating the development rock stockpile will be a very low and poorly buffered rain water or snow melt. Therefore, the outcomes from this modeling exercise do not need to be understood as incremental concentrations, but rather can be taken as estimates of effluent from the development-rock stockpile.

Effects of Adding Limestone to Extend Lag Period for Onset of Acidification

The initial modeling of the Development-Rock Stockpile, although producing a pH of 7.1 (Table 2), is slightly undersaturated in calcite ($CaCO_3$). This suggests a possible strategy for simply amending the system with readily-available High-Calcium limestone. This would have two, closely related functions:

- Add additional acid-neutralization capacity, in a readily available form, to the pad, thereby extending the lag period for potential onset of acidification.
- Raise the pH of effluent water, further reducing concentrations of pH-sensitive metals such as Cu and stabilizing the ferric hydroxide (ferrihydrite) that would precipitate and sorb other trace metals.

The effect of adding limestone was evaluated by simulating the reaction of the modeled effluent with calcite (CaCO3, the principal component of limestone). A source of limestone being considered by KEMC has an analyzed content of 98% CaCO3 (personal communication, KEMC, September, 2005), so this is a fair assumption regarding the mineralogy. The simulation is conducted using the numerical model REACT (Bethke 1996; 2005). In the simulation,only a small amount of limestone (600 mg CaCO3; 0.22 cm^3 of calcite) is required to bring each kg of modeled effluent into equilibrium with calcite. The results are provided in Table 3.

Table 3 Modeled Water-Quality for Development-Rock Stockpile with Limestone Addition (mg/L except pH in su and Alkalinity in mg $CaCO_3$/L)

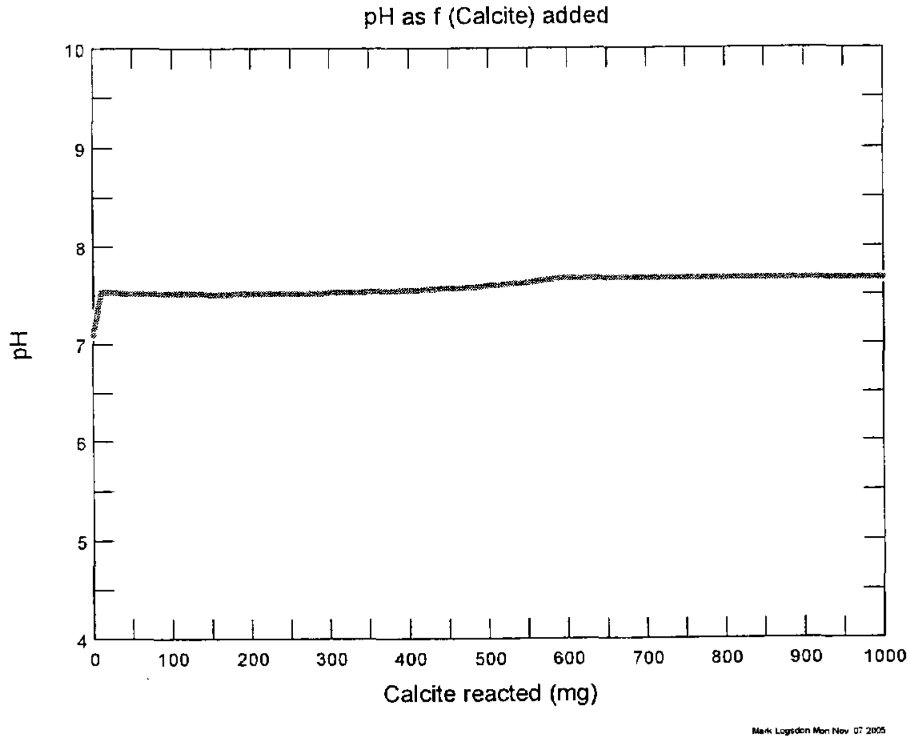
Parameter	Value
pH	7.6
Alkalinity	72
Total Dissolved Solids	932
Calcium	199
Magnesium	18
Sodium	19
Potassium	29
Bicarbonate	59
Sulfate	575
Cl	10
F	0.5
Aluminum	0.001
Antimony	0.0004
Arsenic	0.083
Barium	0.03
Boron	0.58

Parameter	Value
Cadmium	0.0002
Chromium, Total**	0.00045
Cobalt	0.0008
Copper	0.06
Lithium	0.019
Lead	0.0004
Iron	0.002
Manganese	0.0005
Mercury **	0.00004
Molybdenum	0.0015
Nickel	8.33
Selenium	0.004
Silver**	0.00005
Strontium	0.20
Thallium**	0.00005
Vanadium**	0.00075
Zinc	1.90

**: Input values less than the detection limit were modeled as ½ detection limit for calculational purposes

The principal change is an increase in pH and Ca, and a reduction in Cu. The solution remains undersaturated in gypsum. The pH increase, modeled to be to 7.6, is not sufficiently high to precipitate Ni as a hydroxide, so the Ni concentration remains unchanged. The solution does reach equilibrium with fluorite, so there is a small decrease in the concentration of F. At the point in the simulation where calcite reaches saturation, the mineralogical control on Ba changes from barite ($BaSO_4$) to witherite ($BaCO_3$), according to the thermodynamic factors, however, there is no significant change in Ba concentration expected. Major trends in the geochemistry if limestone were added to the pile are illustrated in Figures 2 and 3.

Figure 2 pH Change as a Function of Calcite added.



The main points of Figure 2 are that the pH (a) rises slightly and (b) stays nearly constant, implying the increase in buffer capacity from the additional calcite in the system.

Figure 3 Mineral Precipitation During Calcite Addition

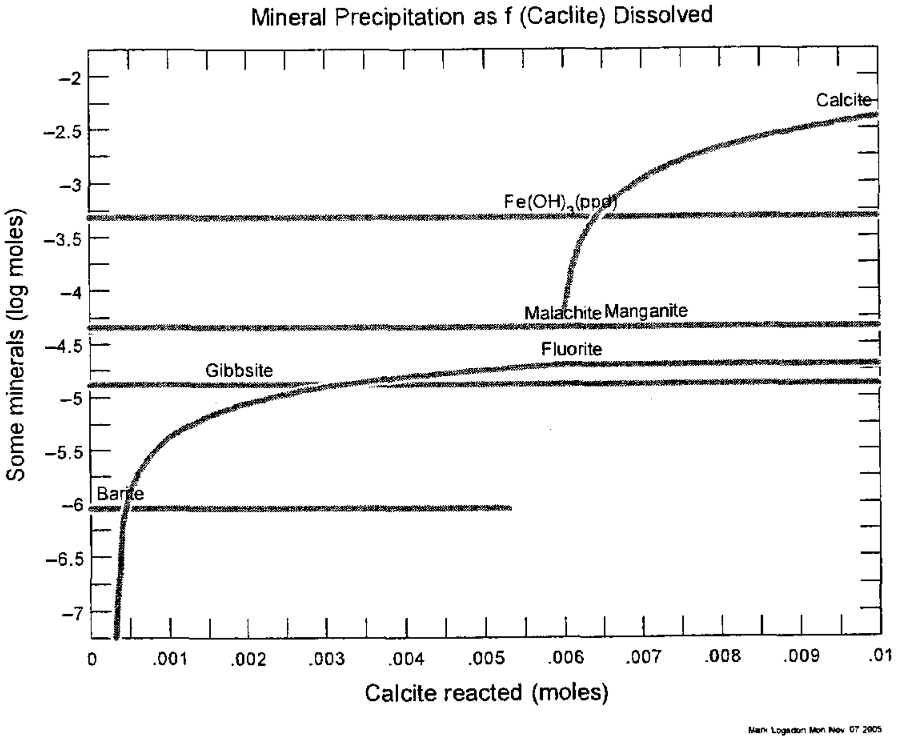


Figure 3 shows the succession of mineral precipitation reactions expected as calcite is added to the solution. Barite (controlling Ba and SO4 very slightly), Gibbsite (Al), Malachite (Cu), Manganite (Mn), and Ferrihydrite (Fe) all were saturated when the simulation began. As explained above, the termination of barite merely means that witherite ($BaCO_3$) is stable once sufficient carbonate alkalinity of available. The rise of fluorite (CaF_2) is notable, adding a control for F. The appearance of calcite after dissolution of 0.006 moles (600 mg) of calcite indicates that calcite has reached equilibrium and additional calcite would then be available as additional Acid Neutralization Capacity.

ESTIMATING LIMESTONE ADDITION

The best available point estimate of ARD risk is the Net Neutralization Potential. Using either medians or means, the best estimate of NNP for the whole of the Development Rock stockpile is approximately (–10) tonnes $CaCO_3$ eq/kt (Phase II Report, Geochimica (2005)). If one assumes that the underground workings would re-flood quickly (either under natural gradient or by intentional flooding), then the long-term acidification potential would be controlled by the inundated condition. Therefore, KEMC could consider an alkaline addition to either the stockpile or the backfill sufficient to meet the average Maximum Potential Acidity demand, assuming that if that amendment were readily available for dissolution, it would be adequate to offset the reaction rate of the backfill and wall rock until the flooded conditions can predominate. Note that this estimate assumes that it would be sufficient to bring the stoichiometric NNP value to precisely 0.

For 379,000 tonnes of Development Rock, an incremental NP of 10 t $CaCO_3$eq/kt would require about 3800 tonne $CaCO_3$ eq. The amount of limestone that would be needed is greater than this because of several sources of inefficiency.

- Firstly, natural limestones are rarely pure $CaCO_3$. Based on assays of available metallurgical-grade limestone, one may assume a 98% $CaCO_3$ eq content.
- Secondly, a great deal of field experience shows that only a portion of the actual potentially-effective CaCO3 will react, either because of inefficiencies in mixing or because some of the limestone is blinded by secondary precipitates like gypsum or iron oxides. For current planning, we recommend that KEMC assume that the average availability would be 70%.
- Therefore, the total availability of neutralization potential from the limestone would estimated at 68.6% (70% of 98%).

Therefore, a reasonable planning basis if KEMC were to strive for an even acid-base balance across the whole of the Development Rock Stockpile would be approximately 5540 tonne of limestone. If KEMC wished a factor of safety of 1.2, the value would be 6650 tonne, and for a FOS of 1.4 (the Nevada DEP factor), 7760 tonne of limestone would be needed.

If the minimum recommended value of 5540 tonne of limestone were added, this would comprise 2.31E+09 cm^3 of calcite ($\varrho = 2.4\ g.cm^{-3}$). If, as indicated by the numerical simulations described above, the calcite equilibrium requires 0.22 cm^3 of calcite per kg of effluent, then the design-basis limestone would be capable of neutralizing 1 E+10 kg of effluent. Based on the conceptual model illustrated in Figure 1, total effluent production would be approximately 4.38E+3 $m^3.a^{-1}$. For a fluid density of approximately 1E+3 $kg.m^{-3}$, the total annual mass of effluent would be 4.38 E+6 $kg.a^{-}$. Thus, there would be a very large excess of limestone in the pile, compared to that required to buffer pH to levels suggested by Table 3. Therefore, there should be sufficient calcite present to both ensure that the solution is not net acid-generating and also to buffer the pH and control metals concentrations to levels such as those estimated in this model (Table 3).

REFERENCES

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