

**CASE STUDIES OF ML/ARD
ASSESSMENT AND
MITIGATION: SNIP GOLD MINE**

MEND Report 9.1b

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CASE STUDIES OF ML/ARD ASSESSMENT AND MITIGATION: SNIP GOLD MINE

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EXECUTIVE SUMMARY

The Snip Mine was an underground gold and silver mine that operated from 1991 to 1999, producing approximately 1.3 million tonnes of ore. The mine is located in the rugged coastal mountains of northern British Columbia. There is no road connection to the outside world and access is only possible by air or boat. On average, the site receives 1318 mm of annual precipitation; however, unlike other mine sites in the area that are at higher elevation (e.g., Johnny Mountain and Eskay Creek), the Snip site is close to sea level so the snow equivalent is only 619 mm. From the perspective of metal leaching and ARD, the main site components are the two waste storage facilities, namely the tailings impoundment and the underground workings. Waste rock was used as: underground backfill, either directly or crushed and cycloned to produce backfill sand; placed in dumps at the mouths of various portals; and used to construct the tailings impoundment. The majority of the tailings was cycloned to produce backfill sand for use underground. The slimes, the fine-grained reject of the cycloned tailings, and whole tailings when cycloning did not occur were placed in the impoundment on site.

The ML/ARD Assessment

The pre-mine assessment of the ARD potential was based on the assumption that a positive NNP (NP – AP) indicated materials did not have the potential to generate ARD. Using this assumption, the mine predicted that the waste rock would be non-PAG, the ore would be PAG, tailings slimes and whole tailings produced from sulphide ore were PAG, and tailings sand would be non-PAG. Flooding in the impoundment was proposed as the means of preventing PAG tailings from generating ARD.

Throughout mining, regular, usually monthly, sampling and ABA analysis were conducted on the waste rock, whole tailings and backfilled tailings sand. At closure, the 5th and 95th percentile ABA results for the waste rock were a %S of 0.31 and 5.61, a Sobek-NP of 68 and 215 kg/t, and a Sobek-NPR of 1.2 and 15.0. The 5th and 95th percentile ABA results for the whole tailings were a %S of 2.8 and 5.8, a Sobek-NP of 157 and 231 kg/t and a Sobek-NPR of 1.0 and 2.3. The 5th and 95th percentile ABA results for the tailings sand were a %S of 2.8 and 7.1, a Sobek-NP of 148 and 230 kg/t and a Sobek-NPR of 0.8 and 2.3. The assessment of the ARD potential at closure was based on the assumption that ARD was likely if the NPR (NP/AP) was < 1, unlikely with an NPR > 2, and uncertain if the NPR was between 1 and 2. In the operational characterization:

- approximately 20% of the waste rock samples had Sobek-NPR values less than 2;
- most whole tailings samples had Sobek-NPR values between 1 and 2 and approximately 5% had Sobek-NPR values slightly less than 1; and
- most of the tailings sand samples had Sobek-NPR values between 1 and 2, and approximately 20% had NPR values slightly less than 1.

Like many mines, Snip dovetailed its operational ARD sampling with sampling for ore grade or mill process control. Advantages include integration of sampling with the mining operation, and the labour savings in sampling, sample preparation and storage. However, there were also a number of disadvantages. One was that the tailings sample was taken prior to the addition of waste rock and the removal of cycloned sand, and was therefore potentially very different from the tailings material

placed in the impoundment. Sampling tailings prior to deposition may also have missed changes in composition that occurred due to particle size and mineral density segregation within the impoundment after the tailings were spiggotted.

A potential concern with the operational characterization of the waste rock was whether compositing sub-samples from several muck piles masked important variability. Sampling individual muck piles in the last few years indicated that compositing did not mask variability. Whether to composite samples can be an important issue because, depending on factors such as deposition and hydrology, the composition of a small portion of waste may be the primary determinant of drainage chemistry and source of contaminants. Another concern in sampling waste rock was whether the composition of the fines differed from that of the coarser fragments whose mass typically dominated 'whole waste rock' samples. The composition of the fines was important because they constituted the majority of the surface area and thus controlled weathering and drainage chemistry. Analysis of different particle sizes of samples from the surface dumps indicated that the NP/AP ratio in the waste rock at Snip was lower and thus the ARD potential was higher in the < 2 mm compared to the coarser fragments.

Additional sampling and analysis was conducted prior to mine closure to fill gaps in material characterization. This included sampling to check the composition of the: mine walls underground; tailings within the impoundment; waste rock in individual muck piles; and waste rock placed in dumps outside a number of portals. ABA results for tailings in the impoundment were similar to those for whole tailings. ABA analysis results for the waste rock in individual muck piles and the mainly development waste rock in surface dumps were similar to those for the operational waste rock characterization. During mining, it was assumed that all the carbonate was calcite. However, Rietveld XRD work done in support of the closure plan indicated that while calcite was the most common carbonate mineral, ankerite and siderite were also present and thus a small but potentially significant portion of the carbonate-C and the Sobek-NP may have been non-neutralizing. Based on the significant portion of each material with an NPR lower than 2, in the closure planning it was recognized that the waste rock and tailings were potentially ARD generating (PAG).

In addition to the ABA analysis, the mine also operationally measured the elemental composition of the waste rock, whole tailings and backfilled tailings sand, with additional sampling and analysis at closure to check the composition of the mine walls underground, tailings within the impoundment, and waste rock placed in dumps. The results indicated that median concentrations of total Cd, Mo, Pb and Zn in the tailings sand and the whole tailings were five to ten times higher than typical upper crustal background, while As was two hundred and fifty times higher. Waste rock exceeded typical upper crustal background in its median As and the 95th percentile concentrations of total Ag, Cd, Cu, Mo, Pb and Zn. Although the trace metal concentrations in the Snip wastes were relatively low by mine standards, the presence of elevated concentrations of metals with a relatively high solubility over a wide range in pH, such as As and Zn, indicated that there was a potential for significant metal leaching in neutral pH, as well as in acidic discharge. Notably, Se and Sb, which were not measured in the elemental assays, were observed in elevated concentrations in the humidity cell drainage.

The Tailings Impoundment

The tailings impoundment contains approximately 850,000 t of tailings and 100,000 t of waste rock. The closure objective was to maintain the water table high enough to saturate the PAG wastes and prevent significant metal leaching. Additional work immediately after the mine closed included: improvements to the two dams to ensure they could withstand a 1-in-a-1000 year flood and extreme seismic events; construction of a spillway; placing waste rock from surface dumps in the impoundment; and the addition of a soil cover. The 1-in-a-1000 year flood and additional seismic protection were achieved by adding toe berms to the dams, creating an effective slope of 6:1 at a relatively low cost. The lower NPR waste rock was placed in a depression. The higher NPR waste rock was spread over the tailings. Placement of the weathered waste rock within the impoundment has had no significant impact on water quality. After waste rock placement was complete, natural surficial materials stripped from the area of dam construction and a borrow area were placed on the surface. Geotextile fabric was used to place the soil-like material on the recently deposited tailings at the lower, wetter end of the impoundment within months of the mine closing.

Additional ongoing requirements for the impoundment include: geotechnical monitoring of the dams and spillway and any necessary maintenance, monitoring the quality and quantity of drainage discharge to the environment, monitoring the height of the water table at the high-end of the impoundment to ensure potentially ARD generating (PAG) wastes are flooded, and determining whether the small area of tailings without a soil cover at the lower-end of the impoundment should be covered. An important consideration with any monitoring and maintenance was that the mine was only accessible by helicopter or boat. Angular armour rock (boulders > 12 inch) was placed in the spillway to prevent beavers from damming the flow to reduce the required frequency of spillway inspection and maintenance. Assuming the dams remain stable and the PAG wastes at the high-end of the impoundment are saturated, the tailings impoundment was not considered a ML/ARD concern.

The Main Underground Workings

The underground workings consist of the Twin West and the Main or Twin Zone, located on the hill slopes on opposite sides of the tailings impoundment. Only the Main Underground Workings were considered a ML/ARD concern. The primary ML/ARD concern in the underground workings was the backfill, which consisted of 344,648 t of waste rock, 466,959 t of tailings sand and 1,306 t of cement. Mine walls were not considered a major metal leaching concern because of their lower surface area and because most of the workings were backfilled, which would limit wall collapse. Based on the operational ABA monitoring results and the supposition that all the tailings sand but only 25% of the waste rock occurred as reactive fines (75% of the waste rock was assumed to occur as coarse fragments that were relatively inert), the assessment at closure was:

- ARD was considered likely (NPR <1) for 93,000 t of tailings sand;
- ARD production was considered uncertain (NPR 1-2) for 326,000 t of tailings sand and 17,000 t of waste rock fines; and
- ARD was considered not likely (NPR >2) for 47,000 t of tailings sand and 69,000 t of waste rock fines.

The objectives for the mine workings at closure were to reduce the ARD potential, prevent access and the associated health and safety concerns, and ensure that the majority of drainage from the internal workings reported to the tailings impoundment. Measures planned at closure to achieve these objectives included:

- constructing concrete bulkheads in the two lowest adits to flood workings below the 300 Level;
- filling the upper portals with cemented rock fill (waste rock) to block access and reduce air entry into the upper, unflooded workings; and
- constructing a P-drain in the 300 Portal that would block air entry without blocking drainage discharge.

Removal of the ore rich crown pillar and subsequent subsidence made it impossible to flood above the 300 Level. The lower workings, below the 300 Level, were estimated to contain approximately 30% of the backfill. Observations of the mine layout and operational drainage patterns indicated that drainage from the upper unflooded workings would report to the lower flooded workings. Drainage in excess of that required to flood the lower workings was predicted to report to the 300 Portal. Potential concerns with the plan to flood the lower workings were that:

- there might be significant leakage causing the height of water table to fluctuate, causing significant weathering, metal leaching and carbonate dissolution; and
- leaching of weathered backfill in the lower workings as they flood may result in unacceptably high trace metal loadings to the tailings impoundment.

To address these concerns, as part of the closure planning, Snip attempted to predict the post-mining water balance, the rate of flooding and the resulting chemistry of discharge from the flooded workings, and proposed contingency plans where there were thought to be significant risks.

To reduce drainage losses, the mine conducted an extensive program of mapping and then capping exploration drill holes in the lower workings. The estimate of drainage losses through drill holes and fractures of 11,977 m³/month from flooded lower workings was an educated guess, because it would have been impossible to measure these values without flooding active workings. Post-mining drainage input into the lower, hopefully flooded workings, were predicted by subtracting estimated drainage additions in the backfill from measurements of the drainage collected at the bottom and pumped out of the mine in 1997 and 1998. The calculated input ranged from 11,299 to 53,553 m³/month and averaged 31,000 m³/month. In 1997 and 1998, there was only one month in which predicted drainage losses exceeded calculated drainage inputs. Consequently, the water balance was predicted to be overwhelmingly positive.

Monitoring and a contingency plan were proposed to address concerns about the uncertainty regarding potential leakage through fracture and drill holes, and the accuracy of the data used to predict the water balance. A pressure gauge was installed in the 130 Bulkhead to monitor the height of flooding. The contingency plan proposed, if significant water table fluctuations accelerated carbonate dissolution and metal leaching, was that the mine would drill large holes through the bulkheads, allowing the lower workings to be freely drained. The procedure of drilling holes through a bulkhead to prevent unacceptable flooding had been previously used at the Baker Mine in

northeast British Columbia. Surface inspection was proposed as means of spotting significant unplanned discharge through fracture and drill holes. The location of the workings on the hill above the impoundment meant there was little concern that leaks would result in drainage discharge directly to the environment.

Based on their void volume, the average groundwater inflow (31,000 m³) and the predicted leakage rate as the water table rose, the estimated time to flood the lower workings was 6.45 months. The lower bulkhead was constructed and flooding started on August 20th, 1999. The water table had already reached the 300 Portal when monitoring re-started on the 19th of January 2000, five months later. Subsequent monitoring of the pressure gauge in the 130 Bulkhead and discharge from the 300 Portal, indicated that the height of water table was fairly static. Despite the concerns that flooding would cause significant leakage through fractures, the only significant discharge observed from the partially flooded workings was drainage from the 130, 180 and 300 Portals. No new seeps were observed on the slope adjacent to the underground workings.

Since monitoring started, monthly discharge from the 300 Portal varied between 0 to 27.4 L/s, and averaged 8.4 L/s (21,334 m³/month). Monthly flow from the 180 Portal has been < 2.3 L/s, with an average of 1.1 L/s (2,904 m³/month), and monthly flow from the 130 Portal has been 2 to 6.4 L/s, and averaged 3.5 L/s (8,975 m³/month). The combined average discharge from the mine workings (flow from the 130, 180 and 300 Portals) was 33,213 m³/month. Assuming 50% of the 130 Portal and the 180 Portal discharges comes from near-surface seepage, the combined flow from the internal mine workings (flow from the unflooded upper workings, overflow from the lower workings and seepage around the bulkheads) was 27,274 m³/month. This is quite close to 31,000 m³, the pre-flooding prediction of average monthly groundwater inputs into the mine, suggesting that leakage from fractures and drill holes in the flooded workings, other than in the immediate vicinity of the bulkheads, was limited.

The pre-closure prediction of drainage chemistry from the partially flooded workings was based on weak acid soluble metal concentrations (EPA 1312), metal release during the first year of humidity cell testing, and a number of assumptions regarding the weathering and leaching of backfill and wall rock in the upper unflooded and lower flooded workings. The predicted post-flooding “Final” drainage in the flooded workings was neutral pH, with As 77 µg/L, Cd 4.4 µg/L, Cr and Pb 28 µg/L, Cu 25 µg/L, Mo 29 µg/L, Ni 56 µg/L and Zn 21 µg/L. The predicted post-flooding discharge from the 300 Portal was neutral pH drainage, with As 29 µg/L, Cd 1.8 µg/L, Cr 17 µg/L, Cu 4 µg/L, Mo 9 µg/L, Ni 21 µg/L, Pb 9 µg/L and Zn 8 µg/L. The predicted water quality was similar to the drainage chemistry underground prior to closure. These values were typically two to three, and in the case of Cu up to eight times, lower than the predicted concentrations from the unflooded workings. This was to be expected as the pre-closure underground drainage chemistry was used to calibrate the prediction model and the model assumed that 95% of the eventual discharge from the 300 Level would come from the upper unflooded workings. Based on the similarity between predicted post-flooding and the existing (pre-flooding) metal concentrations, Snip assumed that flooding of weathered backfill in the lower workings would not significantly impact discharge water quality¹.

¹ Although the predicted Cd of 1.8 µg/L exceeded the discharge limit of 0.8 µg/L, it was assumed that there would be sufficient dilution and attenuation in the impoundment prior to discharge.

The actual post-flooding trace metal concentrations in the discharge from the 300 Portal and drainage behind the 130 Bulkhead were almost all much lower than predicted values: median values for As of 0.7 and 0.45 µg/L, Cd of 0.4 and < 0.2 µg/L, Cu of < 1 and < 5 µg/L, Ni of 3 and 8 µg/L, and Pb both < 1 µg/L, respectively. Concentrations were also low for Sb (0.7 and 1.5 µg/L) and Se (1.1 and <0.5 µg/L), trace metals released in relatively high amounts in the long-term humidity cell tests (e.g., maximum values of 67 and 120 µg/L, respectively). A notable exception was Zn, whose concentration was greatly underestimated in the pre-closure water quality prediction. Maximum and median concentrations of Zn in the discharge from the 300 Portal were 388 and 80 µg/L versus predicted values of 8 µg/L. Maximum and median concentrations of Zn in the drainage behind the 130 Bulkhead were 583 and 57 µg/L versus a predicted value of 21 µg/L. The maximum Zn values occurred immediately after the lower workings were flooded. However even in 2003, four years later, the range in D-Zn of 46 to 66 µg/L in 2003 for the discharge from the 300 Portal was well above the prediction of 8 µg/L. To date, dilution and attenuation in the impoundment have reduced Zn concentrations by at least an order of magnitude prior to discharge from the impoundment and off-site.

Factors that likely contributed to the underestimation of Zn were that the tailings sand data used in the model came from test work done on fresh rather than weathered samples and that at least initially, a high proportion of the discharge may have come from the water flooding the lower workings. Unlike the waste rock samples, which came from dumps, the tailings sand used in the EPA 1312 and humidity cell tests were fresh unweathered samples, whose soluble Zn content was potentially not representative of older backfill, which had been exposed to up to eight years weathering and were relatively unleached. Because of the difficulty in accessing older stopes, older backfilled tailings sand were not analyzed prior to flooding. Access and a lack of seepage prevented the monitoring of drainage from older stopes and as a result, drainage chemistry data collected prior to flooding came primarily from areas where backfilled materials were relatively new and weathering effects were masked by the drainage additions and chemistry of active mining. The relatively high solubility of Zn at neutral pH's and the fact that galvanic effects likely accelerated the oxidation of Zn sulphide, while suppressing the oxidation of other sulphide minerals, are possible explanations for why other trace metals were not similarly underestimated. Notably, Zn and Mg were unique among the parameters monitored in the tailings sand humidity cells in that the production rate increased during the test.

Factors that may have caused a higher than predicted proportion of the 300 Portal discharge to initially come from the flooded portion of the workings included: discharge first started in mid-winter when drainage inputs to the upper workings would be relatively low and the depth of mixing would be highest when flood water first reached the discharge elevation.

Despite relatively high discharges of Mn and Zn in the initial 300 drainage, metal concentrations in the impoundment discharge have remained well below the discharge limit. The impoundment was especially effective in lowering Zn, which was 388 µg/L in the initial 300 discharge, but only 27 µg/L in the drainage discharged from the impoundment spillway.

Future Water Quality

The high NP of the backfill should ensure that relatively high drainage pH and alkalinity will continue for a number of years. There is uncertainty about what will happen in the short-term to the concentrations of trace metals. Depletion of more readily weathered minerals and finer particles and the development of protective surface coatings may reduce trace metal concentrations. Conversely, there may be increases potentially exceeding concentrations observed in the humidity cells and the initial drainage. Factors potentially causing increased trace metal loadings in neutral pH drainage from the upper workings included:

- increased drainage inputs and leaching in the upper unflooded portion of the mine as a result of surface subsidence;
- increased trace metal solubility, due to depletion of major cations and alkalinity added in process water during operation;
- increased leaching and weathering as a result of the collapse of backfill and the workings themselves; and
- the cumulative build-up of weathering products resulting in larger flushing events.

Assuming the lower workings remain flooded, the highest trace metal loadings from the underground workings are predicted when Ca and Mg CO₃ are exhausted in the PAG materials in the upper unflooded portion of the workings. The ABA results indicated that acidic weathering and acidic drainage are likely from some portion of the unflooded backfilled tailings sand and were possible from a portion of the waste rock and collapsed workings. Discussion of the factors influencing the amount and strength of ARD and the pH of the eventual discharge are discussed in Section 7.2.5. The humidity cell tests indicate that acidic weathering will not occur for at least 20 to 30 years. According to Morin and Hutt (1997), it took 15 years for the Island Copper Mine waste rock dumps, with 0 to 4% S and NP values of less than 80 kg/t, to produce measurable ARD.

Flooding is expected to minimize sulphide oxidation in the lower workings and the resulting low metal levels and alkalinity may provide some dilution or neutralization of the drainage from the upper workings. However, unless there is some sustainable mechanism of metal removal, continuous addition of drainage from the upper workings will soon result in similar water quality in the mixing zone. It is also important to note that mixing and therefore dilution may be minimal at certain times of the year or during major runoff events, and that even if drainage from PAG wastes is neutralized underground, Cd and Zn and other metals released by acidic weathering will remain entrained in the drainage causing increased trace metal loadings to the environment.

The complexity and heterogeneity of the wastes and workings, the limited extent of our generic understanding of the processes controlling ML/ARD and the lack of long-term data from other, similar mine workings make it very difficult to predict drainage chemistry and metal loading in the immediate future, let alone several decades into the future. Some of the important questions yet to be answered include the following.

- What portion of the tailings sand, waste rock and mine walls will eventually be ARD generating, how long will it take and what will be the resulting acidity and metal loadings (see Sections 3.2 and 7.2.5)?

- What is the ability of NP sources in the underground to both neutralize acidic drainage and reduce associated elevated metal concentrations prior to discharge from the workings and the tailings impoundment? NP sources include non-ARD generating wastes and alkalinity in groundwater inputs to the upper workings, drainage within the flooded workings and drainage within the tailings impoundment.
- What is the rate of air entry within different areas of the upper mine (e.g., crown pillar versus remote stopes) and within the backfilled tailings sand versus voids or the waste rock?
- How low does the rate of oxygen supply need to be to reduce the rate of sulphide oxidation to a level where ARD no longer occurs in PAG materials?
- Will future subsidence in the area of the crown pillar significantly increase oxygen and drainage entry into the upper mine workings, with a subsequent increase in sulphide oxidation and leaching?
- Will dissolution and physical pressure decrease the strength and increase the porosity of the rock around the bulkheads, and will this create significant fluctuations in the height of the floodwater? Is periodic grouting required to prevent this?

At mine closure, the proposed contingency plan, if discharge from the underground workings were to cause drainage in the tailings impoundment to exceed the discharge limits, was drainage collection and treatment within the tailings impoundment. Drainage could be briefly impounded in the tailings impoundment using stop-logs in the spillway. Because of the uncertainty regarding future drainage chemistry, no details were provided regarding treatment processes and the resources required, and there are no triggers set to indicate when additional actions will be taken. Presently the mine relies on the monitoring of the portal drainages to detect problems, assuming that changes in water quality, such as increases in sulphate and pH fluctuations, will allow increasing trace metal concentrations to be identified in time for actions to be taken to prevent significant impacts to the environment.

Discussion and Conclusion

The ML/ARD work conducted at Snip illustrates many of the challenges associated with ML/ARD. These include technical challenges:

- the large number of properties and processes in flux (e.g., air entry, backfill weathering and collapse of mine walls);
- the difficulty in measuring many key properties (e.g., mineralogy); and
- the lack of long-term data from other similar sites (e.g., underground mines).

There are also logistical challenges:

- the large costs;
- problems in accessing remote mine sites and closed underground mines;
- the large amount of information required;
- the problems of data management; and
- the departure of key staff, camp closure and the reduction in the number of personnel.

One mitigation strategy at Snip was to, where possible, flood materials with a potential to produce ARD using dams and bulkheads. This essentially converted a geochemical risk into a geotechnical risk. The success of this strategy will depend on the integrity of bulkheads and dams. Maintenance of the long-term geotechnical integrity requires regular inspection and, when required, repair. Another objective at closure was to create a single main discharge location, with discharge from the underground reporting to the tailings impoundment and discharge from the tailings impoundment reporting via the spillway to Sky Creek. The exceptions were the discharge from the 130 Portal and the Dam #3 seepage, both of which report to the Monsoon Creek system.

There are typically a number of unresolved issues with any ML/ARD mitigation measure. Unresolved geochemical questions regarding the tailings impoundment included the height of the water table and the extent of flooding at the upper end of the impoundment and the impact of discharge from the underground mine. The main outstanding geochemical issue at Snip is future drainage chemistry and metal loading from the upper, unflooded backfilled underground workings. Uncertainty about future metal leaching or acidity is an issue at many mine sites and creates considerable uncertainty about future information and resource needs. The long list of questions associated with future water quality, highlights the uncertainty that exists regarding this issue.

Presently, the mine relies on monitoring portal drainages to detect problems, assuming that changes in water quality will allow increasing trace metal concentrations to be identified in time for actions to be taken to prevent significant impacts to the environment. Use of monitoring instruments and data loggers to monitor air and water quality in the upper underground workings were suggested in the review of the Snip closure plan, but were rejected because of the difficulty in maintaining relatively inaccessible monitoring equipment, and concerns that a limited underground monitoring program would not address regulatory concerns regarding temporal and spatial heterogeneity.

The questions of temporal and spatial heterogeneity, regulatory acceptance and whether test conditions matched the site-specific conditions were also raised in the discussion of whether additional prediction studies could be used to show what portion of the tailings sand, waste rock and mine walls will eventually be ARD generating, how long will it take before the NP is depleted and what will be the resulting acidity and metal loadings. The NP depletion procedure used in several humidity cells was an example of where the specific conditions at Snip were not considered when adopting test procedures from another site. Before attempting to increase understanding, one should have a clear idea of the purpose, procedural limitations, data requirements, and the manner in which results will be interpreted.

Previously, water quality monitoring was primarily used to detect environmental impact and determine permit compliance. In an effective ML/ARD program, monitoring is used to test predictions, provide a warning of potential problems and inform corrective actions. Providing sufficient resources to maintain, regularly update and review monitoring records is an important part of successful long-term management at many mine sites. Regular review of the data is required to ensure adequate data quality, not just for the evaluation of the results.

Where long-term monitoring and maintenance are required, sites need regularly updated operating manuals for site management, maintenance, mitigation and monitoring. Another key component is maintenance of a database showing the composition, location and mass of different mine

components, and changes in flow, drainage chemistry, loadings and other properties that impact or reflect changes in weathering. Databases should be organized so that new information can be easily added. Well-maintained records are also very important when, as is more and more common, there are changes in ownership, personnel or reporting structures. Unfortunately, where resources are short, data management and review are often the first things to be dropped, especially for issues such as ARD where problems are unlikely to occur for a number of years.

ML/ARD closure plans require information from a wide range of technical disciplines (e.g., hydrology and geotechnical) and a thorough understanding of the site and its history. Periodic regulatory review is important in ensuring companies have sufficient understanding and resources to handle the liability and risk associated with the mine closure plan, and adapt to changing conditions. It is also important that government informs mines and communities of the regulatory implications of proposed actions. It takes considerable time and energy before a regulator can understand the complexity of the site, the mine history, and trends and limitations of the data. Short-time lines and the frequent modifications that typically occur during mine closure frequently challenge regulatory organizations reviewing closure plans.

Where there remains considerable uncertainty about future drainage chemistry, as is the case at Snip, the challenges may not diminish when the mine closes. While mine closure planning should be part of the initial mine plan and re-visited as more information becomes available throughout the mine life, from an environmental and financial perspective, operating mines are often only the tip of the iceberg. The vast majority of the mine sites with significant ML/ARD issues are closed mines.

Mine closure can be a difficult time to conduct work and collect data as budgets are cut and most of the people and equipment leave. In the decommissioning of the Snip mine, a major consideration in staff reductions and the removal of equipment was the mine's desire to dismantle housing facilities and reduce costs. The imminent departure of key staff and equipment create tight timelines for the proposed work and its review. Corporate memory loss regarding important aspects of the site may occur long before closure as long-term staff leaves for more secure positions. Without adequate data base management, staff departure for more permanent jobs elsewhere may result in the loss of knowledge of previous activities, what information exists, how it was collected and where it is stored. While closure planning requires hard work and may include significant financial pressure, closure can also be demoralizing for staff. Staff departure was less pronounced at Snip compared to other closures because some personnel also had or took jobs at the neighbouring Eskay Creek mine and still retained responsibility for work at Snip.

One of the lessons learned at Snip was the importance of early recognition of ML/ARD risks and information deficiencies. Recognition of the geochemical risks associated with the underground workings prior to mining would likely have resulted in modifications to the mine plan, with the mine either reducing the sulphide content of the backfill (e.g., desulphurization prior to cycloning) or not using tailings sand as backfill. Recognition of ML/ARD during exploration would also have permitted more complete grouting of exploration drill holes. Earlier recognition of the uncertainty regarding future water quality would have enabled the mine to use its operating facilities and site personnel to run long-term studies underground. Recognition of ML/ARD information deficiencies earlier in the mine life is especially important for underground mines where access is limited once mining ceases.

In closure plans the 'devil is typically in the details'. The foundation of every closure plan is the raw data. This was illustrated in the discussion of operational monitoring and the water quality model. While it is easy to get lost in the details and not see the 'forest for the trees', it is equally important not to overlook the limitations of sampling and analysis. In conducting ML/ARD work it is important to integrate the micro and macro scales, and to check the accuracy of underlying data no matter how complex or detailed the analysis. In these busy times, many will only read the executive summary of this report. However, it is important to note that the executive summary only provides an outline of the ML/ARD assessment and mitigation at Snip. Proper understanding of the work and the resources required to conduct that work can only be achieved by reading the attached report.

SOMMAIRE

La mine Snip a été exploitée comme mine souterraine d'or et d'argent de 1991 à 1999. Elle a alors produit environ 1,3 million de tonnes de minerai. Elle est située dans les montagnes côtières très accidentées du Nord de la Colombie-Britannique. Faute de route, on peut s'y rendre uniquement par bateau ou par avion. En moyenne, le site reçoit 1 318 mm de précipitations par année. Toutefois, contrairement aux autres sites miniers de la région qui se trouvent à une altitude plus élevée (p. ex., Johnny Mountain et Eskay Creek), Snip est presque au niveau de la mer; alors, l'équivalent en neige n'y est que de 619 mm. Du point de vue de la lixiviation des métaux (LM) et du drainage rocheux acide (DRA), le site renferme principalement deux emplacements de rejets, soit le parc à résidus et les ouvertures souterraines. Les stériles ont été utilisés pour remblayer le souterrain, soit qu'ils y ont été déversés directement, soit qu'ils ont été concassés et cyclonés pour être transformés en du sable de remblayage. Les stériles ont aussi été placés dans des haldes situées à divers portails et utilisés dans la construction du parc à résidus. La majorité des résidus a été cyclonée en vue de la production de sable de remblayage qui a été utilisé sous terre. Les schlamms provenant des résidus cyclonés et les résidus non-cyclonés, donc entiers, ont été déversés dans le parc à résidus.

L'évaluation de la LM/du DRA

L'évaluation du potentiel du DRA faite avant l'exploitation de la mine était basée sur l'hypothèse que les matériaux ayant un potentiel de neutralisation net (PNN, qui est égal au potentiel de neutralisation [PN] moins le potentiel de génération d'acide [PA]) positif ne pouvaient générer du DRA. En se fondant sur cette hypothèse, les responsables de la mine ont prévu que les stériles n'auront pas de PA, que le minerai aura un PA, que les schlamms et les résidus entiers issus du minerai sulfuré auront un PA et que la fraction sablonneuse n'aura pas de PA. On a proposé d'inonder le parc pour empêcher les résidus à PA de produire du DRA.

Durant toute la période d'exploitation de la mine, les stériles, les résidus entiers et les remblais en sables ont fait l'objet d'un échantillonnage et d'une analyse du bilan acide-base (ABA) régulièrement, soit généralement chaque mois. À la fermeture de la mine, les résultats ABA du 5^e et du 95^e centiles étaient pour les stériles : 0,31 % et 5,61 % S, PN-Sobek de 68 et 215 kg/t et rapport PN/PA-Sobek de 1,2 et 15,0; pour les résidus entiers : 2,8 % et 5,8 % S, PN-Sobek de 157 et 231 kg/t et rapport PN/PA-Sobek de 1,0 et 2,3; pour la fraction sablonneuse des résidus : 2,8 % et 7,1 % S, PN-Sobek de 148 et 230 kg/t et rapport PN/PA-Sobek de 0,8 et 2,3. L'évaluation du potentiel du

DRA à la fermeture était basée sur l'hypothèse que le DRA était probable avec un rapport PN/PA < 1, peu probable avec un rapport PN/PA > 2 et incertain avec un rapport PN/PA entre 1 et 2. Dans la caractérisation opérationnelle :

- environ 20 % des échantillons de stériles présentaient un rapport PN/PA-Sobek inférieur à 2;
- pour la majorité des échantillons de résidus entiers, le rapport PN/PA-Sobek se situait entre 1 et 2, et pour environ 5 % des échantillons, le rapport PN/PA-Sobek était légèrement inférieur à 1;
- pour la plupart des échantillons de fraction sablonneuse des résidus, le rapport PN/PA-Sobek se situait entre 1 et 2, et pour à peu près 20 % des échantillons, le rapport PN/PA était légèrement inférieur à 1.

Conformément à la pratique générale, les responsables de Snip ont fait concorder l'échantillonnage régulier lié au DRA ou avec l'échantillonnage relié au contrôle de procéder visant à déterminer la teneur du minerai. Les avantages étaient nombreux : intégration de l'échantillonnage avec l'extraction, réduction du coût de la main-d'œuvre pour l'échantillonnage et la préparation et l'entreposage des échantillons, etc. Plusieurs désavantages se sont aussi manifestés. Par exemple, l'échantillon de résidus a été prélevé avant l'ajout de stériles et l'enlèvement du sable cycloné. Par conséquent, cet échantillon pouvait être très différent des résidus déversés dans le parc à résidus. L'échantillonnage des résidus avant leur dépôt peut aussi avoir empêché les analystes de déceler les changements dans la composition qui se sont produits en raison de la ségrégation en fonction de la granulométrie et de la densité des minéraux qui a eu cours dans le parc après que les résidus y ont été déversés.

L'on peut se demander, au sujet de la caractérisation opérationnelle des stériles, si le mélange des sous-échantillons prélevés dans plusieurs amas de déblais a masqué une variabilité importante. L'échantillonnage des divers amas de déblais au cours des dernières années a permis de conclure que le mélange des échantillons n'a pas masqué la variabilité. Produire ou non des échantillons composites peut s'avérer une question importante parce que, selon des facteurs comme le dépôt et l'hydrologie, la composition d'une petite portion des rejets peut être le facteur déterminant de la chimie des eaux de drainage et de la source des contaminants. L'on s'est aussi demandé, quant à l'échantillonnage des stériles, si la composition des fractions fines différait de celle des fractions plus grossières, dont la masse dominait en général les échantillons de « stériles entiers ». La composition des fractions fines était importante parce que ces fractions constituaient la majorité de la superficie et, de ce fait, contrôlaient l'altération et la chimie des eaux de drainage. L'analyse des différentes granulométries constituant les échantillons prélevés dans les haldes de surface a montré que le rapport PN/PA pour les stériles à Snip était moins élevé et qu'ainsi, le potentiel de DRA était plus élevé pour les fractions < 2 mm que pour les fractions plus grossières.

Une campagne d'échantillonnage et d'analyse supplémentaire a été menée avant la fermeture de la mine afin de combler les lacunes au plan de la caractérisation des matériaux. Cette campagne a notamment compris un échantillonnage visant à vérifier la composition des parois de la mine sous terre; des résidus présents dans le parc; des stériles placés dans les piles de déblais; et des haldes de stériles situées à l'extérieur de plusieurs portails. L'ABA des résidus du parc était similaire à celui des résidus entiers. Les résultats de l'analyse du ABA des stériles présents dans les amas de déblais et des stériles accumulés en haldes de surface, ces derniers provenant principalement des travaux de développement, étaient similaires aux résultats de la caractérisation des stériles durant

l'exploitation. Durant l'exploitation de la mine, il a été assumé que tout le carbonate était de la calcite. Cependant, les travaux d'analyse effectués au moyen de la diffraction des rayons X par la méthode de Rietveld, à l'appui du plan de fermeture, ont indiqué que, même si la calcite était le minéral carbonaté le plus fréquent, l'ankérite et la sidérite étaient elles aussi présentes et bien qu'en faible portion, le C sous forme de carbonate pourrait être potentiellement important et le PN-Sobek ont pu être non neutralisants. En se fondant sur la portion importante de chacun des matériaux ayant un rapport PN/PA inférieur à 2, les auteurs du plan de fermeture de la mine ont reconnu que les stériles et les résidus pouvaient éventuellement produire du DRA.

En plus de l'analyse du ABA, les responsables de la mine ont mesuré de façon standard la composition élémentaire des stériles, des résidus entiers et des remblais et ont procédé à un échantillonnage et à une analyse supplémentaires à la fermeture de la mine, afin de vérifier la composition des parois de la mine sous terre, des résidus déversés dans le parc et des haldes de stériles. Les résultats ont montré que les concentrations médianes de Cd total, de Mo total, de Pb total et de Zn total dans la fraction sablonneuse des résidus et les résidus entiers étaient de cinq à dix fois plus élevées que les concentrations de fond généralement enregistrées dans la couche supérieure de l'écorce terrestre et que les concentrations étaient deux cent cinquante fois plus élevées pour l'As. Les concentrations dans les stériles ont surpassé les concentrations de fond généralement enregistrées dans la couche supérieure de l'écorce terrestre quant à la concentration médiane d'As et aux concentrations du 95^e centile d'Ag total, de Cd total, de Cu total, de Mo total, de Pb total et de Zn total. Les concentrations de métaux-traces dans les rejets miniers à la mine Snip étaient relativement peu élevées selon les normes, mais la présence de concentrations élevées de métaux à solubilité relativement élevée dans une large gamme de pH, notamment d'As et de Zn, a révélé un potentiel de LM important à pH neutre et dans la décharge acide. Il convient de noter tout particulièrement que le Se et le Sb, qui n'ont pas été mesurés dans les dosages élémentaires, étaient présents sous forme de concentrations élevées dans les eaux de drainage provenant des essais en cellule d'humidité.

Le parc à résidus

Le parc à résidus renferme près de 850 000 t de résidus et 100 000 t de stériles. L'objectif du plan de fermeture était de maintenir le niveau phréatique assez élevé pour saturer les rejets acidifiants et prévenir la LM. Des travaux supplémentaires ont été effectués immédiatement après la fermeture de la mine, entre autres, des améliorations ont été apportées aux deux digues afin qu'ils puissent résister à une inondation qui se produit une fois dans mille ans et à des phénomènes sismiques qui présentent une ampleur extrême; un déversoir a été construit; les stériles accumulés en haldes de surface ont été transférés dans le parc; et une couverture géologique a été aménagée. L'on a protégé les digues contre l'inondation susmentionnée et l'on a accru leur capacité de résister aux phénomènes sismiques en leur ajoutant des bermes. Un talus de 6:1 a ainsi été créé à un coût relativement peu élevé. Les stériles à rapport PN/PA peu élevés ont été placés dans une dépression. Les stériles à rapport PN/PA élevé ont été répandus sur les résidus. Le transfert des stériles altérés dans le parc à résidus n'a eu aucun effet important sur la qualité de l'eau. Après que le transfert a été terminé, on les a recouverts de matériaux naturels provenant de la zone de construction des digues et d'un banc d'emprunt. Un géotextile a été utilisé pour placer ces matériaux sur les résidus récemment déposés à l'extrémité la plus basse et la plus humide du parc, au cours des mois qui ont suivi la fermeture de la mine.

D'autres besoins doivent être comblés dans le cas du parc à résidus. Par exemple, il faut assurer un suivi géotechnique des digues et du déversoir et d'entretenir ces ouvrages; surveiller la quantité et la qualité des eaux de drainage déchargées dans l'environnement; surveiller la hauteur de la nappe phréatique à l'extrémité élevée du parc afin de veiller à ce que les rejets susceptibles d'entraîner du DRA soient inondés; et déterminer si la petite zone de résidus sans recouvrement qui se trouve à l'extrémité la moins élevée du parc devrait être recouverte. L'accès à la mine uniquement par hélicoptère ou par bateau a constitué un facteur important du point de vue du suivi et de l'entretien. Un perré angulaire (blocs > 12 po) a été aménagé dans le déversoir pour empêcher les castors d'y construire des barrages et ainsi réduire la fréquence des inspections et des travaux d'entretien à cet ouvrage. L'on a jugé que le parc à résidus ne sera pas une source de LM/DRA si les digues demeurent stables et si les rejets PA situés à l'extrémité élevée du parc sont toujours saturés.

Le chantier souterrain Main

Les ouvertures souterraines se composent du chantier Twin West et du chantier Main ou Twin. Il est situé sur les pentes de la colline qui fait face au parc à résidus. Le chantier Main est le seul qui a été jugé susceptible de présenter le problème de LM/du DRA, son remblai ayant été la principale source de préoccupation. Ce remblai était composé de 344 648 t de stériles, de 466 959 t de la fraction sablonneuse des résidus et de 1 306 t de ciment. Les parois de la mine n'étaient pas trop inquiétantes du point de vue de la LM parce que leur superficie était peu élevée et parce que le chantier était presque entièrement remblayé, ce qui réduit les effondrements des parois. D'après les résultats du suivi opérationnel du ABA et la supposition que tous les sables sauf 25 % des stériles sont des fractions fines réactives (il a été présumé que 75 % des stériles étaient des fractions plus grossières relativement inertes), l'évaluation à la fermeture était la suivante :

- la production d'un DRA est probable (rapport PN/PA < 1) pour 93 000 t de la fraction sablonneuse des résidus;
- la production d'un DRA est incertaine (rapport PN/PA entre 1 et 2) pour 326 000 t de la fraction sablonneuse des résidus et 17 000 t de stériles à grain fin;
- la production de DRA est peu probable (rapport PN/PA > 2) pour 47 000 t de la fraction sablonneuse des résidus et 69 000 t de stériles à grain fin.

Les objectifs pour les ouvertures souterraines à la fermeture de la mine étaient de réduire le potentiel de DRA, d'empêcher l'accès au site, de prévenir les problèmes de santé et de sécurité liés au site et de faire en sorte que la majorité des eaux de drainage provenant des ouvertures s'écoule dans le parc à résidus. Plusieurs mesures ont été prévues à la fermeture pour que ces objectifs soient atteints. Il a notamment été prévu de :

- construire des cloisons en béton dans les deux ouvertures les plus basses pour inonder les galeries sous le niveau 300;
- remplir les ouvertures supérieures de matériaux de remblayage (stériles) cimentés afin d'en bloquer l'accès et de réduire l'entrée d'air dans le chantier supérieur, non-inondé;
- construire un drain spécial (P-drain) dans le portail au niveau 300 qui va empêcher l'entrée d'air mais pas la décharge des eaux de drainage.

L'extraction du pilier de surface, riche en minerai, et l'affaissement de surface qui en a résulté ont rendu impossible l'inondation au-dessus du niveau 300. Les chantiers sous le niveau 300 contenaient, selon les estimations, près de 30 % des matériaux de remblayage. L'examen du plan de la mine et du tracé du réseau hydrographique a révélé que les eaux de drainage provenant du chantier supérieur non-inondé vont s'écouler dans le chantier inférieur inondé. Il a été prévu que les eaux de drainage en surplus après l'inondation du chantier inférieur seront évacuées par l'ouverture au niveau 300. L'éventuelle inondation du chantier inférieur pourrait causer ces problèmes :

- une importante fuite pourrait entraîner une fluctuation de la hauteur de la nappe phréatique, et ainsi, une altération importante, une LM et une dissolution des carbonates;
- la lixiviation des matériaux altérés du chantier inférieur suite à l'inondation de ce dernier pourrait entraîner un niveau inacceptable de métaux-traces pouvant atteindre le parc à résidus.

Pour prévenir ces problèmes, les responsables de Snip ont tenté, dans le plan de fermeture, de prévoir le bilan hydrique postérieur à l'exploitation de la mine ainsi que le degré d'inondation et son effet sur la chimie de l'eau provenant du chantier inondé, et ils ont proposé des plans d'urgence à l'égard des risques importants.

Pour réduire les pertes par ruissellement, les responsables de la mine ont exécuté un vaste programme de cartographie et ont ensuite bouché les trous des forages d'exploration dans le chantier inférieur. L'estimé de 11 977 m³/mois de la baisse du débit en raison des trous de forage et des fractures dans le chantier inférieur inondé est le résultat d'un processus raisonné parce qu'il aurait été impossible de mesurer ces valeurs sans inonder le chantier en production. L'on a prévu quel sera le débit des eaux de drainage dans le chantier inférieur, idéalement inondé, une fois que l'exploitation de la mine sera terminée, en soustrayant les ajouts estimatifs apportés aux matériaux de remblayage des mesures faites, en 1997 et en 1998, sur les eaux de drainage du fond de la mine avant leur pompage à l'extérieur de celle-ci. Le débit calculé variait de 11 299 à 53 553 m³/mois et était en moyenne de 31 000 m³/mois. En 1997 et en 1998, les baisses de débit prévues ont surpassé les gains de débit calculés pour seulement un mois. Par conséquent, l'on a prévu que le bilan hydrique sera essentiellement positif.

Un suivi et un plan d'urgence ont été proposés pour faire face à toute fuite par les fractures et les trous de forage et assurer l'exactitude des données utilisées pour prévoir le bilan hydrique. Un manomètre a été installé dans la cloison 130 pour surveiller le niveau de l'inondation. Selon le plan d'urgence proposé, si le jeu de la nappe phréatique est assez important pour accélérer la dissolution des carbonates et la LM, de gros trous seront percés dans les cloisons afin que l'eau accumulée dans le chantier inférieur soit facilement évacuée. Percer des trous dans une cloison pour empêcher une inondation inacceptable est un moyen qui avait été utilisé auparavant à la mine Baker, dans le Nord-Est de la Colombie-Britannique. Une inspection en surface a été proposée comme moyen de déceler les décharges d'eau importantes et imprévues qui s'effectuent par les fractures et les trous de forage. Comme le chantier est situé sur la colline surplombant le parc à résidus, il a été jugé peu probable que les fuites éventuelles causent une décharge des eaux de drainage directement dans l'environnement.

D'après l'espace libre dans le chantier, l'apport moyen en eau souterraine (31 000 m³) et la prévision du taux de ruissellement accompagnant la remontée du niveau de la nappe phréatique, l'on a estimé qu'il faudra 6,45 mois pour inonder le chantier inférieur. La cloison inférieure a été construite et l'inondation a commencé le 20 août 1999. La nappe phréatique avait déjà atteint l'ouverture au niveau 300 lorsque le suivi a repris le 19 janvier 2000, soit cinq mois plus tard. Le suivi ultérieur du manomètre dans la cloison 130 et de la décharge au niveau 300 a révélé que la hauteur de la nappe phréatique était plutôt stable. L'on craignait que l'inondation cause une fuite d'eau importante par les fractures, mais la seule décharge digne de mention qui a été observée à partir du chantier partiellement inondé a été celle des eaux de drainage s'écoulant des ouvertures 130, 180 et 300. Aucun suintement nouveau n'a été observé sur la pente adjacente au chantier souterrain.

Depuis le début du suivi, le débit mensuel de la décharge au niveau 300 a varié de 0 à 27,4 l/S et a été en moyenne de 8,4 l/S (21 334 m³/mois). Le débit mensuel au niveau 180 a été inférieur à 2,3 l/S et en moyenne de 1,1 l/S (2 904 m³/mois) et le débit mensuel au niveau 130 a été de 2 à 6,4 l/S et en moyenne de 3,5 l/S (8 975 m³/mois). Le débit moyen pour toute la mine (débit aux niveaux 130, 180 et 300) a été de 33 213 m³/mois. Si l'on suppose que 50 % de la décharge au niveau 130 et au niveau 180 proviennent du suintement au voisinage de la surface, le débit total pour les chantiers internes de la mine (débit issu du chantier supérieur non-inondé, trop-plein provenant du chantier inférieur et suintement autour des cloisons) s'établit à 27 274 m³/mois. Comme ce chiffre se situe assez près des 31 000 m³ représentant la prévision avant l'inondation de l'apport mensuel moyen en eau souterraine dans la mine, il est permis de croire que les fuites par les fractures et les trous de forage jusque dans le chantier inondé, exception faite du voisinage immédiat des cloisons, étaient peu importantes.

La prévision antérieure à la fermeture de la mine de la chimie des eaux de drainage provenant du chantier partiellement inondé était basée sur des concentrations de métaux solubles faiblement acides (EPA 1312), la libération de métaux durant la première année des essais en cellule d'humidité et diverses hypothèses concernant l'altération et la lixiviation du remblayage et de la roche encaissante dans le chantier supérieur non-inondé et dans le chantier inférieur inondé. L'on a prévu que les caractéristiques « définitives » des eaux de drainage après l'inondation du chantier seront les suivantes : pH neutre, As : 77 µg/l, Cd : 4,4 µg/l, Cr et Pb : 28 µg/l, Cu : 25 µg/l, Mo : 29 µg/l, Ni : 56 µg/l et Zn : 21 µg/l. Selon les prévisions, la décharge à l'ouverture du niveau 300 aura ces caractéristiques après l'inondation : drainage à pH neutre, As : 29 µg/l, Cd : 1,8 µg/l, Cr : 17 µg/l, Cu : 4 µg/l, Mo : 9 µg/l, Ni : 21 µg/l, Pb : 9 µg/l et Zn : 8 µg/l. La qualité de l'eau prévue était similaire à la chimie du drainage souterrain avant la fermeture de la mine. Ces valeurs étaient en général de deux à trois fois — et dans le cas du Cu, de huit fois — inférieures aux concentrations prévues pour l'eau provenant du chantier non-inondé. Il fallait s'y attendre parce que la chimie du drainage souterrain avant la fermeture de la mine a été utilisée pour étalonner le modèle prédictif et que le modèle supposait que 95 % de l'éventuelle décharge au niveau 300 proviendrait du chantier supérieur non-inondé. En se fondant sur la similarité entre les concentrations de métaux prévues pour la période postérieure à l'inondation et les concentrations de métaux existantes, donc les concentrations précédant l'inondation, les responsables de Snip ont assumé que l'inondation du

remblayage altérés en place dans le chantier inférieur n'aurait aucun effet important sur la qualité de l'eau rejetée².

Après l'inondation, les concentrations de métaux-traces dans les eaux rejetées au niveau 300 et dans les eaux de drainage derrière la cloison 130 étaient en fait presque toutes beaucoup moins élevées que les prévisions. En effet, les valeurs médianes étaient respectivement les suivantes : As : 0,7 et 0,45 µg/l, Cd : 0,4 et < 0,2 µg/l, Cu : < 1 et < 5 µg/l, Ni : 3 et 8 µg/l et Pb : < 1 et < 1 µg/l. Les concentrations de Sb (0,7 et 1,5 µg/l) et de Se (1,1 et < 0,5 µg/l) étaient elles aussi peu élevées. Le Sb et le Se sont deux métaux-traces qui ont été libérés en des quantités relativement élevées lors des essais à long terme en cellule d'humidité (p. ex., valeurs maximales de 67 et 120 µg/l, respectivement). Le Zn faisait manifestement exception. Sa concentration a été grandement sous-estimée lors de la prévision de la qualité de l'eau avant la fermeture de la mine. La concentration maximale et la concentration médiane de Zn dans l'eau rejetée au niveau 300 étaient de 388 et 80 µg/l, alors que les prévisions les avaient situées à 8 µg/l. La concentration maximale et la concentration médiane de Zn dans les eaux de drainage derrière la cloison 130 étaient de 583 et 57 µg/l, alors que les prévisions les avaient situées à 21 µg/l. Les valeurs maximales pour le Zn ont été obtenues immédiatement après l'inondation du chantier inférieur. Cependant, même en 2003, soit quatre ans plus tard, la gamme de concentrations de Zn dissous de 46 à 66 µg/l pour l'eau rejetée au niveau 300 était très supérieure à la prévision de 8 µg/l. À ce jour, la dilution et l'atténuation dans le parc à résidus ont réduit les concentrations de Zn d'au moins un ordre de grandeur avant le point de rejet du parc et à l'effluent final.

Ces facteurs ont vraisemblablement contribué à la sous-estimation de la concentration de Zn : les données sur les fractions sablonneuses des résidus utilisées dans le modèle provenaient d'essais effectués sur des échantillons frais plutôt qu'altérés; et une proportion élevée de la décharge a pu provenir de l'eau inondant le chantier inférieur, du moins au départ. Contrairement aux échantillons de stériles, qui provenaient des haldes, les sables utilisés dans l'EPA 1312 et les essais en cellule d'humidité étaient des échantillons frais, non altérés, dont le contenu en Zn soluble pouvait ne pas être représentatif du remblayage qui avait été altéré durant huit ans et qui était relativement peu lixivié. En raison de la difficulté d'accéder aux vieux chantiers, les sables constituant les vieux remblais n'ont pas été analysés avant l'inondation. Le drainage issu des vieux chantiers n'a fait l'objet d'aucun suivi en raison de la difficulté d'accès à ces chantiers et de l'absence de suintement. Par conséquent, les données sur la chimie du drainage recueillies avant l'inondation provenaient principalement de zones où les matériaux de remblayage étaient relativement frais et où les effets de l'altération étaient masqués par les ajouts de drainage et la chimie attribuable à l'exploitation de la mine. La solubilité relativement élevée du Zn lorsque le pH est neutre et le fait que des effets galvaniques ont vraisemblablement accéléré l'oxydation du sulfure de zinc tout en supprimant l'oxydation des autres minéraux sulfurés pourraient expliquer pourquoi les concentrations d'autres métaux-traces n'ont pas été sous-estimées. Fait à noter, le Zn et le Mg se sont distingués des autres paramètres observés lors de l'essai en cellule d'humidité sur les fractions sablonneuses des résidus, en ce que leur taux de production s'est accru durant l'essai.

L'eau rejetée au niveau 300 est venue au départ de la zone inondée du chantier dans une proportion plus élevée que celle qui avait été prévue, notamment parce que la décharge d'eau a

² La concentration de Cd prévue de 1,8 µg/l dépassait la limite de rejet de 0,8 µg/l, mais il a été supposé qu'il y aurait une dilution et une atténuation suffisante dans le parc à résidus avant la décharge.

d'abord commencé au milieu de l'hiver, lorsque les apports d'eaux de drainage dans le chantier supérieur étaient relativement peu importants et que la profondeur de mélange des eaux était à son maximum quand les eaux d'inondation ont d'abord atteint le niveau de la décharge.

En dépit de rejets de Mn et de Zn relativement importants dans le drainage initial du niveau 300, les concentrations de métaux dans l'eau rejetée du parc à résidus sont demeurées bien au-dessous de la limite de rejet. Le parc à résidus a été tout particulièrement efficace pour réduire la concentration de Zn, car celle-ci était de 388 µg/l dans l'eau rejetée initialement au niveau 300, mais de seulement 27 µg/l dans les eaux de drainage provenant du déversoir du parc.

La qualité de l'eau dans l'avenir

Le PN élevé du remblayage permet de croire que le pH du drainage et l'alcalinité relativement élevés se maintiendront durant plusieurs années. L'on ne sait trop toutefois ce qu'il adviendra à court terme des concentrations des métaux-traces. La diminution des minéraux facilement altérés et des particules fines et le développement de couches superficielles protectrices pourraient abaisser les concentrations de métaux-traces. Par contre, il pourrait y avoir des augmentations, qui pourraient éventuellement dépasser les concentrations observées dans les cellules d'humidité et le drainage initial. Parmi les facteurs qui pourraient éventuellement causer un accroissement des charges en métaux-traces dans le drainage à pH neutre issu du chantier supérieur, notons :

- une augmentation des apports de drainage et de la lixiviation dans la partie supérieure non-inondée de la mine, par suite de l'affaissement de surface;
- une hausse de la solubilité des métaux-traces due à la diminution des principaux cations et à l'alcalinité ajoutée à l'eau de procédé durant l'opération;
- un accroissement de la lixiviation et de l'altération par suite de l'effondrement des remblais et de tout le reste du chantier;
- une formation cumulative de produits d'altération qui accroît l'importance des phénomènes de lessivage.

L'on prévoit que, si le chantier inférieur demeure inondé, les charges en métaux-traces issues du chantier souterrain seront les plus élevées une fois que le Ca et le Mg CO₃ seront épuisés dans les matériaux à PA se trouvant dans la partie supérieure non-inondée du chantier. Les résultats ABA ont montré qu'une altération acide et un drainage acide se produiront vraisemblablement à partir d'une certaine zone des remblais sablonneux non-inondés et qu'ils pourraient se produire à partir d'une zone des stériles et des chantiers effondrés. Les facteurs de la quantité et de l'importance du DRA et du pH de l'eau éventuellement rejetée sont examinés à la section 7.2.5. Selon les essais en cellule d'humidité, l'altération acide ne se produira pas avant 20 à 30 ans, au moins. D'après Morin et Hutt (1997), il a fallu 15 ans aux haldes de stériles de la mine Island Copper (de 0 à 4 % S et des valeurs inférieures à 80 kg/t pour le PN) pour produire un DRA mesurable.

L'on prévoit que l'inondation réduira l'oxydation des sulfures dans le chantier inférieur. Les concentrations de métaux et l'alcalinité peu élevées qui en résulteront pourraient entraîner une certaine dilution ou neutralisation du drainage issu du chantier supérieur. Cependant, à moins qu'il y ait un mécanisme durable d'enlèvement des métaux, l'ajout continu d'un drainage provenant du chantier supérieur ne tardera pas à se traduire par une qualité d'eau similaire dans la zone de

mélange. Il est important de noter que le mélange, donc la dilution, peut être minimal à certaines périodes de l'année ou durant les écoulements de surface importants et que, même si le drainage à partir des rejets à PA est neutralisé sous terre, le Cd, le Zn et les autres métaux libérés par l'altération acide continueront d'être entraînés dans le drainage, ce qui augmentera les charges en métaux-traces dans l'environnement.

La complexité et l'hétérogénéité des rejets et des ouvertures souterraines, les limites de notre connaissance générale des processus contrôlant la LM/le DRA et le manque de données recueillies à long terme à d'autres sites miniers similaires expliquent la difficulté de prévoir quelles seront la chimie du drainage et la charge en métaux dans un avenir immédiat et, pire encore, dans plusieurs décennies. Plusieurs questions importantes demeurent encore sans réponse, par exemple :

- Quelle portion de la fraction sablonneuse des résidus, des stériles et des parois de la mine produira éventuellement du DRA? Dans combien de temps ce DRA sera-t-il produit? Quelle sera l'importance de l'acidité et des charges en métaux qui découleront de ce DRA (voir les sections 3.2 et 7.2.5)?
- Quelle est la capacité des sources souterraines de PN de neutraliser le drainage acide et d'abaisser les concentrations de métaux élevées qui l'accompagnent, avant que l'eau soit rejetée du chantier et du parc à résidus? Les sources de PN incluent les rejets non-producteurs de DRA et l'alcalinité des apports d'eau souterraine dans le chantier supérieur, le drainage dans le chantier inondé et le drainage dans le parc à résidus.
- Quelle est l'importance de l'entrée d'air dans les diverses zones de la mine supérieure (p. ex., dans le pilier de surface par rapport aux chambres éloignées) et dans les remblais par rapport aux espaces vides ou aux stériles?
- Jusqu'à quel point le taux d'oxygène doit-il être faible pour que le taux d'oxydation des sulfures soit si réduit que le DRA ne se produit plus dans les matériaux à PA?
- Est-ce que l'affaissement de surface dans la zone du pilier de surface accroîtra de beaucoup la venue d'oxygène et d'eau de drainage dans le chantier supérieur, ce qui aurait pour effet d'augmenter l'oxydation des sulfures et la lixiviation?
- Est-ce que la dissolution et la pression physique réduiront la résistance et accroîtront la porosité de la roche autour des cloisons et est-ce que cela créera d'importantes fluctuations du niveau des eaux d'inondation? Est-ce qu'il faudrait faire périodiquement des injections de coulis pour prévenir cela?

À la fermeture de la mine, la collecte des eaux de drainage et leur traitement dans le parc à résidus faisaient partie du plan d'urgence proposé au cas où la décharge du chantier souterrain soit telle que les eaux de drainage dans le parc dépassent les limites des effluents. Les eaux de drainage pourraient être brièvement endiguées dans le parc au moyen de batardeaux installés dans le déversoir. Comme l'on ne sait trop quelle sera la chimie du drainage dans l'avenir, aucun renseignement n'a été fourni au sujet des procédés de traitement et des ressources nécessaires, et aucun élément déclencheur n'a été établi pour indiquer quand des mesures supplémentaires seront prises. Présentement, les responsables de la mine s'en remettent au suivi des eaux de drainage aux portails pour déceler les problèmes, en assumant que les changements survenant dans la qualité de l'eau, par exemple les hausses de sulfate et les fluctuations de pH, permettront d'identifier l'augmentation des concentrations de métaux-traces assez rapidement pour qu'il soit possible de prévenir toute répercussion importante sur l'environnement.

Discussion et conclusion

Les travaux effectués à Snip à l'égard de la LM/du DRA illustrent de nombreux défis liés à la LM/au DRA. Ces défis sont d'ordre technique :

- le grand nombre de propriétés et de processus qui entrent en jeu (p. ex., l'entrée d'air, l'altération des matériaux de remblayage et l'effondrement des parois de la mine);
- la difficulté de mesurer de nombreuses propriétés cruciales (p. ex., la minéralogie);
- le manque de données recueillies à long terme à d'autres sites similaires (p. ex., les mines souterraines);

et relèvent aussi de la logistique :

- les coûts élevés;
- les problèmes d'accès aux sites miniers éloignés et aux mines souterraines fermées;
- la grande quantité d'information nécessaire;
- les problèmes en matière de gestion des données;
- le départ du personnel clé, la fermeture du quartier d'habitation et la réduction du nombre d'employés.

L'une des stratégies d'atténuation utilisée à Snip a consisté à inonder, autant que possible, les matériaux susceptibles de produire du DRA, en construisant des digues et des cloisons. Cette stratégie a essentiellement converti un risque géochimique en un risque géotechnique. Le succès de cette stratégie repose sur l'intégrité des cloisons et des digues. Le maintien de l'intégrité géotechnique à long terme nécessite des inspections régulières et la réalisation de toute réparation nécessaire. Un autre objectif visé à la fermeture était de créer un seul lieu de décharge principal et de faire en sorte que l'eau provenant des chantiers souterrains s'écoule dans le parc à résidus et que l'eau issue de ce parc soit acheminée au déversoir, puis dans le ruisseau Sky. L'eau rejetée au niveau 130 et le suintement de la digue n° 3 faisaient exception en étant évacués dans le réseau hydrographique du ruisseau Monsoon.

En général, toute mesure d'atténuation de la LM/du DRA s'accompagne de questions non résolues. Les questions géochimiques non résolues relativement au parc à résidus sont la hauteur de la nappe phréatique et l'importance de l'inondation à l'extrémité supérieure du parc, et l'impact de la décharge provenant de la mine souterraine. La principale question géochimique qui demeure sans réponse à Snip est la chimie du drainage dans l'avenir et la charge en métaux issue du chantier souterrain supérieur remblayé et non-inondé. L'imprévisibilité de la lixiviation des métaux ou de l'acidité dans l'avenir existe pour de nombreux sites miniers et rend très difficile l'établissement des besoins futurs en informations et en ressources. La longue liste de questions liées à la qualité future de l'eau témoigne de l'imprévisibilité qui existe dans ce dossier.

Présentement, les responsables de la mine s'en remettent au suivi des eaux de drainage aux portails pour déceler les problèmes, en assumant que les changements survenant dans la qualité de l'eau permettront d'identifier l'augmentation des concentrations de métaux-traces assez rapidement pour qu'il soit possible de prévenir toute répercussion importante sur l'environnement. L'utilisation d'instruments de suivi et d'enregistreurs de données pour surveiller la qualité de l'eau et de l'air

dans le chantier souterrain supérieur a été suggérée dans l'examen du plan de fermeture de Snip, mais cette suggestion a été rejetée parce qu'il aurait été difficile d'entretenir de l'équipement relativement inaccessible et parce qu'on craignait qu'un programme de surveillance souterraine limitée ne réponde pas aux préoccupations réglementaires au sujet de l'hétérogénéité spatiale et temporelle.

Les questions de l'hétérogénéité spatiale et temporelle, de l'acceptation réglementaire et de l'opportunité des conditions des essais étant donné les conditions particulières au site ont elles aussi été soulevées dans l'examen qui visait à déterminer si des études prévisionnelles supplémentaires permettraient de déterminer quelle portion de la fraction sablonneuse des résidus, des stériles et des parois de la mine va éventuellement produire du DRA, combien de temps cela prendra pour que le PN soit épuisé et quelles seront l'acidité et les charges en métaux subséquentes. La procédure d'épuisement du PN utilisée dans plusieurs cellules d'humidité a constitué un exemple de cas où les conditions particulières à Snip n'ont pas été prises en compte dans l'adoption de procédures d'essai mises en œuvre à un autre site. Avant d'essayer d'approfondir les connaissances, il faut avoir une idée claire de l'objectif, des limites des procédures, des besoins en données et de la méthode qui sera utilisée pour interpréter les résultats.

Auparavant, le suivi de la qualité de l'eau servait principalement à déceler tout impact sur l'environnement et à déterminer la conformité au permis. Dans les programmes d'atténuation de la LM/du DRA efficaces, le suivi est utilisé pour confirmer les prévisions, avertir les responsables des problèmes potentiels et déterminer les mesures correctrices. Fournir des ressources suffisantes pour tenir, mettre à jour régulièrement et examiner les registres du suivi constitue un volet important d'une gestion à long terme efficace, à de nombreux sites miniers. Il faut examiner régulièrement les données pour s'assurer que leur qualité est adéquate et non pas seulement pour évaluer les résultats.

Lorsque de l'entretien et le suivi à long terme sont nécessaires, les responsables des sites ont besoin de mettre à jour régulièrement, des manuels opérationnels pour la gestion et l'entretien du site, l'atténuation des répercussions et pour le suivi. Un autre facteur clé est le maintien d'une base de données montrant la composition, l'emplacement et la masse des diverses composantes de la mine ainsi que les changements de débits, la chimie du drainage, les charges et les autres paramètres qui ont des répercussions ou qui révèlent les changements du à l'altération. Les bases de données doivent être organisées de manière à ce qu'il soit facile d'y faire des ajouts. Il est aussi très important que les registres soient bien tenus, car les changements de propriétaire, de personnel ou de structure des rapports hiérarchiques sont de plus en plus fréquents. Malheureusement, l'examen et la gestion des données sont souvent les premières activités qui sont abandonnées si les ressources viennent à manquer, particulièrement lorsqu'il s'agit de DRA et des autres problèmes qui prennent généralement plusieurs années à se manifester.

Du point de vue de l'atténuation de la LM/du DRA, les plans de fermeture des mines nécessitent de l'information issue de toute une gamme de disciplines techniques (p. ex., hydrologie et géotechnique) et une connaissance approfondie du site et de son histoire. Il est important de procéder périodiquement à un examen réglementaire afin de s'assurer que les compagnies ont suffisamment de connaissances et de ressources pour faire face à la responsabilité financière et au risque liés au plan de fermeture de la mine et s'adapter aux conditions changeantes. Il est également important que le gouvernement informe les responsables des mines et les collectivités des

répercussions réglementaires des mesures proposées. Les autorités de réglementation doivent en venir à connaître le site dans toute sa complexité ainsi que l'histoire de la mine et les tendances et les limites des données, et cela leur demande beaucoup de temps et d'énergie. Les délais courts et les modifications fréquentes qui accompagnent en général les fermetures de mine constituent souvent des défis pour les autorités de réglementation chargées d'examiner les plans de fermeture.

Dans le cas de Snip et des autres mines où il est quasi impossible de prévoir la chimie du drainage dans l'avenir, les défis ne diminuent pas nécessairement avec sa fermeture. La planification de la fermeture d'une mine doit faire partie du plan initial et être rajustée à mesure que de l'information est recueillie aux diverses étapes de la vie de la mine, tant du point de vue des finances que de l'environnement, mais, souvent, les mines en production ne sont que la partie émergée de l'iceberg. La vaste majorité des sites miniers qui présentent un important problème de LM/DRA sont des mines fermées.

Il peut s'avérer difficile d'exécuter des travaux et de recueillir des données durant la fermeture d'une mine, parce que des coupures sont effectuées dans les budgets, que la majorité du personnel part et que la plupart des équipements sont déménagés. Lors du déclassement de la mine Snip, les responsables de la mine ont décidé d'y réduire le personnel et de déménager ses équipements pour diverses raisons majeures, notamment parce qu'ils désiraient démanteler les quartiers d'habitation et réduire les coûts. Le déménagement imminent d'employés et d'équipements essentiels laisse peu de temps pour les travaux proposés et leur examen. Une perte de mémoire institutionnelle quant aux aspects importants du site peut survenir longtemps avant la fermeture de la mine, soit au moment où des employés de longue date quittent la mine pour aller travailler là où la sécurité d'emploi est plus élevée. Sans une gestion adéquate des bases de données, les employés qui partent pour aller occuper un emploi permanent ailleurs peuvent être les seuls à connaître les activités exécutées précédemment et à savoir quelle information existe, comment elle a été recueillie et où elle est entreposée. La planification de la fermeture d'une mine est un dur labeur qui peut comporter une lourde pression financière, mais il ne faut pas oublier que la fermeture peut aussi être démoralisante pour le personnel. Le nombre de départs a été moins élevé à Snip qu'à d'autres mines qui ont fermé, parce que des employés avaient aussi, ou ont accepté, un emploi à la mine Eskay Creek, située non loin de Snip, et ont conservé la responsabilité des travaux à Snip.

Snip nous a livré plusieurs leçons. Elle nous a notamment appris qu'il est important d'identifier le plus tôt possible les risques de LM/DRA et l'information nécessaire. Si les risques géochimiques liés au chantier souterrain avaient été identifiés avant la mise en production de la mine, des modifications auraient probablement été apportées au plan de celle-ci et le contenu en sulfures des matériaux de remblayage aurait été réduit (p. ex., par une désulfuration avant le cyclonage) ou la fraction sablonneuse des résidus n'aurait pas été utilisés comme matériau de remblayage. Si la LM/le DRA avaient été identifiés à l'étape de l'exploration, un plus grand nombre de trous de forage d'exploration aurait été bouché par injection de coulis. Si les propriétaires de la mine avaient réalisé plus tôt l'imprévisibilité de la qualité de l'eau dans l'avenir, ils auraient pu mettre à profit les installations et le personnel de la mine pour effectuer des études à long terme sous terre. Il est particulièrement important d'identifier les besoins en informations portant sur la LM/le DRA dès le début de la vie de la mine lorsqu'il s'agit d'une mine souterraine, parce que l'accès à celle-ci est limité une fois que l'extraction de son minerai prend fin.

Dans les plans de fermeture, « ce sont généralement les détails qui font problème ». Les données brutes constituent les fondements des plans de fermeture, comme l'ont montré l'examen du suivi régulier et le modèle de la qualité de l'eau. Autant il est facile de se perdre dans les détails et de « laisser les arbres nous cacher la forêt », autant il est important de ne pas faire abstraction des limites de l'échantillonnage et de l'analyse. Lorsqu'on effectue des travaux liés à la LM/au DRA, il est important d'intégrer la micro-échelle et la macro-échelle et de vérifier l'exactitude des données sous-jacentes, peu importe le degré de complexité ou de détail de l'analyse. Le rythme de vie effréné des temps modernes en amènera plus d'un à ne lire que le sommaire de ce rapport. Mais le sommaire ne donne qu'un aperçu de l'évaluation et de l'atténuation de la LM/du DRA à Snip. Pour avoir une idée juste des travaux et des ressources nécessaires à leur exécution, il faut lire le rapport.

Acronyms

ABA:	Acid-Base Accounting
AP:	Acid Potential
ARD:	Acid Rock Drainage
Asl:	Above sea level
D-Zn	Dissolved zinc concentration
Hcell:	Humidity Cell
ML:	Metal Leaching
NNP:	Net Neutralization Potential
NP:	Neutralization Potential
NPR:	Neutralization Potential Ratio (NP/AP)
PAG:	Potentially ARD Generating
QA/QC:	Quality Assurance/Quality Control
TSS:	Total Suspended Solids

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1. General Site Conditions

1.1 Natural Setting

The Snip gold mine is located at 56° 40 08 N and 131° 06 25 W, in the rugged coast mountains of northern British Columbia, 100 km northwest of Stewart (Figure 1). The site is in the Iskut River basin, between Bronson Creek and Monsoon Creek, which drain directly into the Iskut River, and Sky Creek, which in turn drains into the Craig River (Figure 2). The airstrip and former sites of the camp and plant are located on the Bronson Creek/Iskut River flood plain (Figure 3). The underground workings and tailings impoundment are located in a steeply incised valley between Sky Creek and Monsoon Lake (Figure 4). The plant site/air strip is 75 to 125 asl, while the tailings impoundment is approximately 150 m asl and the upper underground portal is 520 m asl.

The climate is cool and wet, with an average annual precipitation of 1358 mm. Unlike other mines in the area that are at higher elevation and with the majority of the precipitation occurring as snow (e.g., Johnny Mountain and Eskay Creek), at Snip the snow equivalent is only 619 mm. Snowfall typically occurs from October to April. Maximum runoff occurs during snow melt in the spring and high rain and rain on snow events in the fall. From 1989 to 1999, the top five daily precipitation events were 145, 82, 70, 66 and 58 mm (Klohn Crippen, 1999b). The maximum monthly precipitation in this period of record was 424 mm in September 1994.

The mine has no road access to the outside world. When the mine was operating, personnel arrived by air from Terrace and Smithers. Supplies were also brought up river by boat from Wrangell, Alaska at the coast. The Iskut Canyon is a barrier to boat passage further inland. Since the mine closed, no staff are presently stationed at the site. Inspections are typically conducted when a helicopter from Smithers visits the nearby operating Eskay Creek mine to pick up the gold bullion. Fish guides, guide outfitters and members of the Tahltan First Nation reach the area by boat and are often in the area during certain times of the year.

There are a number of closed mine sites in the area. The closest is the Johnny Mountain mine, a relatively short-lived underground Au mine that operated in the late 1980s, approximately 5 km south, up on Johnny Plateau and accessible from Snip Mine by road. In the past, Johnny Mountain used the Iskut riverfront facilities.

1.2 Mining

The Snip gold showing was discovered in 1965 by geologists employed by Cominco Ltd. working on the western slope of Johnny Mountain. Little work was done until 1986 when Delaware Resources Corporation optioned the property and carried out surface drilling and sampling. Detailed underground drilling and bulk sampling in 1989 confirmed the presence of a mineable gold deposit. Subsequently, Prime Resources bought Delaware's 40% interest. Cominco, who retained its 60% interest, carried out mine development and mill construction.

Production began on January 25, 1991. Based on pre-production reserves of 940,000 t with 28.5 g/t Au and a production rate of 300 t per day, the mine life was anticipated to be 10 years. Once mining started, the production rate was soon raised to 450-460 t per day. With the exhaustion of the main ore zone in 1996, ore production decreased from 460 to 350 t per day. According to records

of the British Columbia Ministry of Energy and Mines, the mine milled 1,308,340 t of ore, producing 12,183 kg of Ag, 32,093 kg of Au and 249 t of Cu (Table 1). Mining occurred primarily in the Twin Zone underground workings, with small underground workings constructed in the Twin West Zone on the other (west) side of the tailings impoundment (Figure 3). The mine closed June 1999, with the majority of the reclamation to date conducted from June to October 1999.

Cominco operated the mine until April 30, 1996, when the Prime Resources Group Inc. purchased Cominco's interest for \$55 million and Homestake Canada Inc. took over its operation. Operation of the site was taken over by Barrick Gold Corporation when Homestake merged with Barrick in December 2001.

1.3 Metallurgy and Milling

The ore was broken using a jaw crusher and a ball mill. Approximately 30-35% of the recovered gold was obtained by a gravity circuit and converted to dore gold bars on site. The remaining 65-70% was recovered by conventional flotation methods that produced a bulk rougher concentrate together with a reground cleaner float concentrate. The combined gold-bearing sulphide concentrates were dewatered before shipment off-site. Gold recovery was approximately 92%.

For much of the mine life, a hovercraft traveling on the Iskut and Stikine rivers removed recovered gold. Towards the end of the mine, concentrate was loaded into 1.4 tonne bags and transported to Wrangell, Alaska by aircraft. For a limited time the concentrate was barged from Wrangell to Stewart and transported by truck to the Premier Gold CIL mill for gold recovery. Towards the end of mining, the ore was shipped to the DOWA smelter in Japan.

The mine decided to use conventional gravity flotation to produce a gold concentrate after concerns were raised about solution containment in the initial proposal to use cyanidation (personal communication from Doug Flynn, Mines Inspector).

1.4 Waste Materials

The mine produced two mineralized waste materials.

Waste Rock: 869,388 t

- 344,648 t directly used as backfill in the underground workings
- 281,740 t crushed and cycloned for use along with mill tailings as sand backfill, with approximately 70% (197,218 t) reporting to the sand backfill and 30% (84,522 t) reporting as slimes to the tailings impoundment
- 229,000 t placed in dumps at the mouths of various portals; at closure, waste rock from four of the five sites (130,000 t) was removed to the tailings impoundment (see waste rock dumps in outline of mine components for amount in each dump and eventual fate)
- approximately 14,000 t used to construct dykes for tailings impoundment

Tailings from Processed Ore: 1,202,949 t

- 773,408 t fine rejects (slimes) from cyclone sand production placed in impoundment
- approximately 150,000 t whole tailings placed in impoundment
- 269,741 t cycloned sand used as underground backfill
- 9,800 t backfilled cycloned tailings sand combined with 1,306 t Portland cement

During operation of the mine, the term tailings was used both for whole tailings sampled at the end of flotation circuit and the mix of whole tailings and cyclone slimes deposited in the impoundment. The term backfill was used when referring to cycloned tailings sand, although waste rock was also used as backfill.

The mine components with a potential for significant metal leaching are as follows:

- Backfilled main underground workings
- Twin West underground workings
- Waste rock placed outside the portals
- Tailings/waste rock impoundment

1.5 Reclamation Requirements

The primary environmental and reclamation objective at Snip Mine was to prevent ML/ARD impacts to downstream aquatic resources. This included ensuring that contaminant discharge was below the levels required to meet receiving environment water quality objectives (e.g., BC water quality criteria and guidelines), and there were no significant future unfunded liability or environmental risks to the Province. The two permitting agencies were the BC Provincial Ministries of Water, Land and Air Protection and Energy and Mines. For the BC Ministry of Energy and Mines, an adverse impact on the environment is defined as either precluding the reclamation objectives or exceedance of discharge limits established by Environmental Protection Program of the BC Ministry of Water, Land and Air Protection (BC MWLAP). An explanation of the BC Ministry of Energy and Mines generic expectations, information requirements and constraints for different aspects of ML/ARD prediction and prevention/mitigation are outlined in Price and Errington (1998).

The post-mining land use on portions of the sites not used for ML/ARD mitigation was the re-establishment of forest ecosystems and wildlife habitat. Due to fears that this would attract species vulnerable to hunting, the grass and legume use in reclamation was restricted to areas with immediate erosion potential. Revegetation has primarily consisted of introduced or naturally invading early successional woody species, such as alder, willow and cottonwood, that thrive in the wet conditions present at the site.

1.6 Mine Hydrology and Aquatic Receiving Environment

Drainage from the 130 portal, plant site and camp area report to Monsoon Creek, Monsoon Lake or Bronson Creek, Bronson Creek eventually reports to the Iskut River (Figures 2 and 3). Drainage from the 130 portal first reports to a small settling pond and then drains through the forest to Monsoon Lake. During mine construction, lower Monsoon Creek was diverted into an old stream channel that drained into the Iskut River rather than Bronson Creek. The approximate basin area of Monsoon Lake is 2.1 km². According to Beatty Spence (1991) and Sharpe (1995), Monsoon Lake and Upper and Lower Monsoon Creek, which are low gradient and drain relatively low lying terrain, support populations of cutthroat trout and Dolly Varden char. Bronson Creek has a catchment area of 55 km² of steep rugged terrain, much of which is glaciated, resulting in high silt loads. Salmon spawning has been observed in Bronson and Lower Monsoon Creek. Salmon use of Bronson Creek is limited to the lower-most section due to the poor habitat and high silt load.

Drainage from the Twin West zone and all portals of the main underground, except the 130, report to the tailings impoundment, which was constructed on a saddle wetland between Sky Creek and Upper Monsoon Creek (Figures 2 and 4). Prior to mining, drainage in the north eastern portion of the area now covered by the impoundment, reported to Upper Monsoon Creek and drainage in the south western portion of the area now covered by the impoundment reported to Sky Creek, which reports to the Craig River, approximately 10 km from the Craig River's confluence with Iskut River. As part of impoundment construction, Sky Creek was diverted west around the tailings impoundment. The upper reaches of Sky Creek are on Johnny Mountain and are too steep for fish passage. Below the mine, Sky Creek is low gradient, often swampy, and a good fisheries resource. There are resident Dolly Varden char and Cutthroat Trout for 2 km below the tailings impoundment. Subsequently, there is a 400 m canyon that restricts access to salmon and other fish. The lower reaches of Sky Creek support all five salmon species and resident bull trout. The Province considers the Craig and Iskut Rivers as Class 1 fisheries resources.

Construction and operation of the underground and the tailings impoundment resulted in several changes to the natural hydrology. When the mine was operating, drainage from the underground workings was primarily discharged from the 180 Portal. This included groundwater, surface runoff entering through the area of the crown pillar and water added with the hydraulic backfill. Discharge from the 180 Portal reported to the tailings impoundment, which was the source of process water. From 1991 to mid-1995, excess surface ponding in the impoundment was discharged through a decant pipe, east into Upper Monsoon Creek. From 1995 until the spillway was constructed at closure, excess surface ponding in the impoundment was decanted west into Sky Creek. At the time the decant location was changed from the Monsoon Lake to the Sky Creek side of the dam, there was a concern that the reduction in flow would significantly impact the Monsoon watershed. This did not prove to be the case presumably because the impoundment was only a small part of the contributing watershed and because much of the impoundment discharge occurs as groundwater seepage through the dams. A lack of flow data for dam seepage hampered efforts to estimate the available dilution for discharged tailings supernatant. According to Sharpe (1995), dilution in Upper Monsoon Creek was estimated to be < 1:1. While there is little or no dilution under extreme low flow conditions, on average Sky Creek provides at least six times more dilution than that available on the Monsoon Creek side.

The main changes in site hydrology at mine closure were:

- the cessation of process water inputs to the impoundment and the underground workings;
- construction of bulkheads, which flooded the lower mine workings and directed drainage out the 300 portal; and
- construction of a final rock spillway on the Sky Creek side of the impoundment.

Drainage from the 300 portal reports to the tailings impoundment.

1.7 Regulation of Drainage Discharge

Typical of mines in this region, assessment of aquatic resources and flows, along with an assessment of the operation, by the proponent (Norecol Environmental Consultants Ltd., 1990) and government (Beatty Spence, 1991, Roberts, 1991, Love, 1995 and Sharpe, 1995) were used to set

discharge limits and drainage monitoring requirements, with environmental effects monitoring required at regular intervals to verify that permit limits were protecting the receiving environment. During mining, the BC Ministry of the Environment *Waste Management Permit* (PE-9079 1996) required twice monthly monitoring of commonly measured water quality parameters in the discharge from the tailings pond. Upstream and downstream water quality in Sky Creek and various locations in the Monsoon Lake/Creek system were monitored at a similar frequency as part of an approved Receiving Environment Monitoring Program. Discharge limits for surface discharge from the tailings impoundment were 14,300 m³/day, 50 mg/L TSS, 0.3 mg/L D-Al, 1.0 mg/L total-As, 0.8 µg/L total-Cd, 0.02 mg/L total-Cr, 0.014 mg/L D-Cu, 0.6 mg/L total-Cu, 1.0 mg/L D-Fe, 0.05 mg/L D-Pb, 0.4 mg/L total-Pb, 1.0 mg/L total-Ni, 0.05 mg/L D-Zn and 1.0 mg/L total-Zn. In addition, the *Waste Management Permit* required the tailings supernatant to annually pass a 96 hr LC50 rainbow trout acute toxicity test. The mine typically did the test during spring freshet.

The initial operating *Waste Management Permit* was issued in 1991 and then amended in 1995 to authorize discharge of tailings supernatant to Sky Creek as opposed to Monsoon Lake (Love, 1995). The *Waste Management Permit* was revised in 2000 to reflect changes that occurred when the mine closed. Changes to the *Waste Management Permit* at closure included reducing monitoring of the tailings pond discharge and Sky Creek from biweekly to monthly, and to 5 times per year for the Monsoon sites. The post-closure permit also required monthly monitoring of the drainage from the 300, 180 and 130 portals. Based on a review of the geochemical characterization and ABA assessments of the tailings and waste rock, and because past concentrations were consistently below the detection limit and were not predicted to reach concentrations of concern for many years post closure, BC Ministry of the Environment concluded that various discharge limits present in the operational permit were not necessary in the post closure permit (Love, 2000). Parameters such as nitrite and ammonia were removed from the post-closure permit based on discontinuation of blasting and low levels observed in post closure monitoring of 300 level mine water discharges. Other parameters removed were: aluminium, arsenic, chromium, copper and nickel. Removal of these parameters resolved the Permittee's concern that they were paying for contaminants not present or expected to be in the discharge. However, it is important to note that although there were no longer discharge limits for these parameters, monitoring for all the dissolved metals included in a standard ICP scan is required in the post-closure monitoring program. Limits remaining in the permit for discharge from the spillway of the tailings impoundment were a pH of 6.5 to 8.5, 50 mg/L TSS, 0.8 µg/L total-Cd, 1.0 mg/L D-Fe and 0.05 mg/L D-Pb, and to annually pass a 96 hr LC50 rainbow trout acute toxicity test. In 2000 The *Waste Management Permit* discharge limit for D-Zn was increased from 0.05 mg/L to 0.2 mg/L.

Based on the low and perceived downward trend in metal concentrations in the site drainage and the downstream environment, the lack of significant variability, and high monitoring costs (Lough, 2003), further revisions in the *Waste Management Permit* in 2003 were:

- reduction in the drainage monitoring to 5 times per year, once each in late April/early May, June, late July/early August, late September/early October and late December/early January;
- removal of alkalinity from the list of measured parameters; and
- removal of the requirement for downstream water quality monitoring in either the Sky Creek or Monsoon Lake watersheds.

The 2003 permit kept the same quantitative discharge limits for the tailings impoundment. According to the 2003 permit, the characteristics of discharge from the 130 and 180 Level Portals “shall be typical of groundwater collected between the (130 or 180) portal plug and the portal entrance, along with minor seepage from behind the portal plug.” The characteristics of discharge from the 300 Level Portal “shall be typical of groundwater collected in the flooded underground workings and drainage from the workings above the 300 Level.” The intent of these two sentences was to prohibit the discharge of drainage whose water quality had deteriorated sufficiently to have a significant impact on the receiving environment (Mark Love pers. comm.).

Monitoring of environmental effects on different components of the receiving environment was conducted at regular intervals throughout the life of the mine. Conclusions of a study conducted when the mine closed in 1999 (EVS, 1999) were that there was little difference in upstream and downstream metal concentrations and no evidence of a reduction in Sky Creek water quality as a result of tailings pond discharge. In the 2000 to 2001 data presented in the 2003 review, the maximum Zn values were 0.01 mg/L in Sky Creek and 0.025 mg/L in Upper Monsoon Lake. Based on the low and perceived downward trends in metal concentrations in site drainage and the downstream environment, Lough (2003) suggested that regular water quality or environmental effects monitoring in Sky Creek or Monsoon Lake watersheds should only be re-instated if there was a deterioration in site discharge quality.

2. Geology and Mineralogy

The Snip property is underlain by upper Triassic feldspathic greywacke, siltstone and mafic tuff belonging to the Stuhini Group. Greywacke is the predominant lithology in the mine workings, with minor amounts of siltstone/mudstone and sandstone. The sedimentary rocks strike generally east-west and dip variably north.

Several feldspar porphyries intrude the sedimentary sequence. The most notable is the dike-like Red Bluff orthoclase megacrystic diorite to monzodiorite dated at 195 +/-1 Ma (Rhys, 1993). Characterization of the Red Bluff as part of the exploration work conducted by the nearby Bronson Slope Project, indicates an ARD potential. At Snip, the only excavation and/or exposure of Red Bluff rock occurred in the first 154 m of the 130 adit (L. Smithies, June 19, 1999). The waste rock resulting from 130 adit construction was presumably placed in the dump outside the 130 portal. The large natural surface exposure of Red Bluff porphyry on a rocky cliff immediately upstream on Bronson Creek is a potential source of natural ARD.

The ore at Snip occurred in mineralized veins emplaced progressively in a dynamic tectonic environment, characterized by semi-brittle deformation (Rhys, 1993). Four discrete yet interrelated ore zones were mined; the Twin Zone, the 150 vein and the 130 vein, which are physically connected, and the Twin West Zone, which occurs on the other side of the tailings impoundment. The main zone of gold mineralization, the Twin Zone lies within the greywackes one kilometer to the south and parallel to the Red Bluff porphyry. The Twin Zone was a 0.5 to 15 m wide, sheared quartz-carbonate-sulphide vein that strikes at 300° and dips southwest at 40° to 65°. It was traced by drilling over 1,000 m (ore strike length is 600 m) and had a vertical range of 500 m. The ore zone plunged 50° northwest and bottomed out at 180 m elevation. The top of the vein was truncated by

surface topography. The Twin Zone was a normally directed shear with approximately 28° oblique westerly slip. The name of the Twin Zone was derived from a post-ore biotite lamprophyre dike (known as BSU or biotite spotted unit), which lay in the plane of the ore vein and commonly divided it in two segments.

Compositional differences in waste rock primarily occurred as a result of differences in the extent of quartz veining and the degree of mineralization. Physical weakness was primarily associated with structure, primarily the occurrence of faults. The main exception was the gouge in the BSU. Within the Twin Zone, the ore was laterally zoned and the following two end members and one intermediate mineralization type were identified.

- Pink and green ore (P&G) occurred on the western end of the deposit against and west of the West-bounding fault. This ore was low in sulphide, typically less than 5% pyrite, contained no chalcopyrite but significant molybdenite. The pink colour was due to pervasive hydrothermal red-brown biotite, with subsequent calcite alteration. The green colour was attributed to chlorite. Textures varied from laminated to brecciated.
- Massive sulphide comprised mainly of pyrite and pyrrhotite, this was the predominant ore type found on the eastern end of the Twin zone. Chalcopyrite was abundant and sphalerite was present, but molybdenite was virtually absent. To the east, the Twin vein thins and/or broke down into numerous pyrite/pyrrhotite stringers and extension veins, striking about 075°, with biotite alteration haloes up to 30 m wide referred to as the py-po-bio zone. This alteration was especially prominent where the Twin vein intersects the 150 vein.
- Streaky ore (Stky) was the most common ore type in the Twin Zone shear vein, and was spatially and compositionally intermediate between pink and green and massive sulphide ore. It consisted of laminations or 'streaks' of biotite, quartz-carbonate, chlorite, sulphide and potassic-altered greywacke in varying proportion. Sheath veining occurred within streaky ore, in 5 - 25 m west plunging lenses, where there was a change in dip. These were filled with massive pyrite or "crackled quartz" (CRQ), with 5 - 15% chlorite in discontinuous micro fractures that was sometimes subdivided as a fourth ore type.

Vertical zonation was less evident, being limited primarily to the predominance of pyrrhotite at depth and pyrite at higher elevation.

The 150, 150 Footwall and 130 veins splayed off the footwall of the Twin vein at angles of 30° and 10° respectively (150° and 130° true azimuth). They were narrow but high-grade veins. The 130 vein was 10 cm wide, but was 200-300 g/t Au and constituted about 20% (200,000 t) of the Snip resource prior to mining. The northwest and southeast ends of the Twin zone were marked by cross faults, the West Bounding fault and the Red fault. The Twin vein (with Py-Po-Bio ore) was offset right laterally by the Red fault, but the BSU dike was unaffected.

Biotite was the principal alteration mineral in the wall rock. It was a widespread metamorphic mineral, but felted, hydrothermal biotite resulted in a dark coloured envelope around ore veins. Pale calcite, K-feldspar and quartz occurred as an outer envelope to veins.

Geological mapping of the Snip Mine was done on both plan and on section as mining and diamond drilling progressed. These maps show the major lithologies and structures. The drawings were done at a 1:500 scale, the sections being 12.5 m apart and the plans at elevations corresponding to the excavations. As with any mine, the level of completion of this work varied from level to level and section to section. Upon mine closure, the mine workings and all openings were mapped. Mapping primarily focused on structural features, such as faults, joints, stringer zones and fractures. Lithologies were mapped (for the most part) as ore, greywacke, biotite spotted dyke and feldspar porphyry (both as dykes and stock). Sub-units of the feldspathic wacke unit (siltstone, greywacke and fragmental) were not mapped because they did not appear to have any correlation with the ore grade or distribution and the sub-units were often intercalated with gradational contacts making the distinction between them very subjective.

During mining, the mine conducted petrographic analysis on samples of the ore. Although twenty-six mineral species and three native metal species were identified in the ore samples, more than 90% of the ore was thought to be composed of only 5 minerals: calcite, quartz, pyrite, chlorite and biotite (Table 2). Based on differences in mineralogy, the ore was separated into four different types (carbonate, sulphide, quartz and biotite/chlorite), although at any given vein location, there may be more than one ore type present. According to Table 2, the relative abundance of different sulphide minerals in the ore was pyrite (4-58%) >> pyrrhotite (0.3-9.2%) >> arsenopyrite (0.002-1.4%) = sphalerite (0.07-1%) > chalcopyrite (0.07-0.3%) > galena (0.003-0.02%) > molybdenite (0.001-0.012%). Massive pyrrhotite is noted to occur locally (NDM, 1998).

As part of the work at mine closure to fill gaps in material characterization, the mine conducted petrographic analysis on whole tailings and tailings sand samples (Tables 3). Both materials contained high concentrations of quartz, albite, biotite, carbonate and pyrite, with lesser concentrations of sericite and microcline, and 1 to 2% pyrrhotite. The backfill tailings sand contained higher concentrations of quartz and albite, while the whole tailings had higher concentrations of biotite. During mining, it was assumed that all the carbonate was calcite, a conclusion that appeared to be based on the lack of tarnishing and strong fizz observed when tested with hydrochloric acid. The notes accompanying the petrographic results shown in Table 3 stated, "the carbonate species is dominantly - if not entirely - calcite." The manner in which this conclusion was reached was not provided and there was no mention of the use of the staining required to microscopically separate different carbonate minerals.

In 1999, Snip conducted Rietveld XRD and scanning electron microscopy plus energy dispersive spectra (SEM-EDS) analysis to check blind spots in petrographic analysis, the mineralogy of grains < 100 µm and the carbonate mineralogy, and to provide a more detailed characterization of potentially neutralizing minerals. The main differences in XRD data compared to the earlier petrographic results were the reduced biotite and pyrite, increased sericite and chlorite, and identification of the carbonate minerals ankerite and siderite in addition to calcite (Table 4). The reduced biotite and pyrite and increased sericite and chlorite are attributable to the ability of XRD to analyze all particle sizes, whereas petrographic mineral identification of silicate minerals is typically limited to particles > 50 µm. The Rietveld XRD analysis indicated that although calcite was the most common carbonate species in the analyzed samples, ankerite and siderite were approximately 5% of total mineralogy and 25% of the total carbonate content. The most common potential silicate sources of NP identified by XRD were biotite (9-16%), chlorite (3-7%), sericite (7-

11%) and albite (15-26%). The implications of the XRD results in the interpretation of the NP results are discussed in Section 3.2.

3. Overview of the Metal Leaching and ARD Test Work

3.1 ABA Analysis

ABA analysis was conducted prior to, during and at the end of mining. The pre-mining analysis included (Tables 5 and 6):

- initial analysis of 12 waste rock, 8 ore and 5 tailings samples; and
- follow up analysis of 9 ore and 15 waste rock samples from the sulphidic zone.

Samples were pulverized and screened to –140 mesh. Analyses included total-S determined by the Leco procedure, NP by the Sobek analysis and paste pH. Acid soluble sulphate and a nitric acid analysis of sulphide-S conducted on the tailings and sulphidic zone samples, permitted sulphide-S rather than total-S use in calculation of the AP. Because of the higher pyrite content, special attention was paid to the ore/tailings and waste rock in the hanging wall and footwall of the sulphidic ore zone.

ABA analysis was conducted during mining in (Tables 7, 8, 9 and 10; Figures 5 to 16; Snip, 1998 and Norecol, Dames and Moore, 1998):

- composite samples of ore tailings, waste rock and tailings sand collected throughout the life of the mine;
- samples taken from the tailings beach in 1993, 1994, 1995 1997, 1998 and 1999;
- samples taken from the waste rock dumps in 1997 and 1998;
- samples taken from the borrow pits, plant site and other disturbed areas or materials around the site in 1997; and
- samples taken from mine walls in the Main and Twin West underground in 1997 and 1998.

The operational sampling and analysis of ore tailings, waste rock and tailings sand was a Mine's Act permit requirement. The waste rock samples were taken from muck piles after it was blasted. The ore tailings samples (whole) were taken from the flotation circuit in the mill. The cycloned tailings sand was sampled before it was pumped underground for use as hydraulic backfill. Like many mines, Snip dovetailed its operational ARD sampling with sampling for grade or process control. Advantages included the labour savings in sampling and sample preparation and storage, and the use of standard protocol. However, there were also a number of disadvantages. One important one was that the tailings samples were taken prior to the addition of waste rock and the removal of cycloned sand, and were therefore potentially very different from the tailings placed in the impoundment. Sampling tailings prior to deposition may also have missed changes in composition that may have occurred due to particle size and mineral density segregation within the impoundment after the tailings are spiggotted. For these reasons, additional sampling and analysis of the tailings in the impoundment were performed.

A potential concern with the operational characterization of the waste rock was whether compositing sub-samples from several muck piles masked important variability. This was checked

by sampling individual muck piles in the last few years. The results indicated that compositing did not mask variability. Variability can be an important issue because, depending on factors such as deposition and hydrology, the composition of a small portion of waste may have been the primary determinant of drainage chemistry and source of contaminants. Another concern in sampling waste rock was whether the composition of the fines differed from that of the coarser fragments who's mass typically dominates 'whole waste rock' samples. The composition of waste rock fines are important because they constitute the majority of the surface area and thus control weathering and drainage chemistry. Analysis of different particle sizes of samples from the surface dumps indicated that the NP/AP ratio in the waste rock at Snip was lower and thus the ARD potential was higher in the < 2 mm than the coarser fragments (Section 4.1).

The ABA analyses conducted on the operational samples were paste pH, total-S and NP. The NP procedure used in the weekly characterization was the Sobek procedure (McLeod & Higgs, 1995). In the closure planning, bulk NP was determined using the Modified method (Coastech, 1991). Sulphate-S was measured in some of the closure samples, with sulphide-S, calculated from total-S minus sulphate-S, used to calculate AP.

Different rock types were sampled separately in the sampling of rock chips and sections of drill core used to characterize mine walls. The narrow, banded nature of the mineralization made it impractical to separate different rock types when sampling the mine wastes.

3.2 NP and AP Results

Notable features of the wastes at Snip included the wide range of NP and AP concentrations, the relatively high NP and the high concentrations of AP in some materials (Table 7). Of the different waste materials, the tailings contained the highest NP and AP concentrations. Median values in the various tailings materials were 99 to 208 kg CaCO₃/t for AP and 180 to 353 kg CaCO₃/t for NP. The waste rock had median AP values of 16 to 104 kg CaCO₃/t and NP values of 65 to 179 kg CaCO₃/t. The median values for the mine walls were 10 to 48 kg CaCO₃/t for AP and 91 to 103 kg CaCO₃/t for NP.

As discussed in Appendix A, a key part of an ARD assessment is determining the accuracy of the laboratory AP and NP data, and, if required, making corrections. Reasons for discrepancies between measured/calculated AP and the acid generation in the actual materials under site conditions are as follows:

- use of total-S as a surrogate for sulphide-S when there is significant acid soluble or insoluble sulphate;
- incorrectly assuming all the sulphide-S produces the same acidity per mole of S as pyrite (FeS₂) or pyrrhotite (Fe_{1-x}S); and
- greater concentration of the sulphide/AP minerals relative to NP minerals in a particular fraction of the wastes, making the material more or less likely to be ARD generating than the material sampled.

The available information on these factors indicated that the use of total-S would provide a fairly accurate measure of the field AP. Comparisons of sulphate-S with total-S and the NPR values calculated using total-S versus sulphide-S indicated that sulphate-S was typically less than 0.2% and

there was little impact from using total-S to calculate the acid potential (NDM, 1998). The low concentrations of Ba and Pb indicated that the concentrations of the main non-acid soluble sulphate species (Ba-barite and Pb-anglesite) were also negligible. Both the limited mineralogical (e.g., petrographic and XRD - Tables 2, 3 and 4) and more extensive elemental (Table 11) data indicated that pyrite and pyrrhotite, which produce a similar amount of acidity per mole of S, account for almost all the mass of sulphide minerals, so a major overestimation of the AP due to a significant presence of other sulphide minerals was unlikely.

The lack of difference between the ABA data for the whole tailings and the cycloned sand suggested that there was no significant concentration of sulphide-S in either the tailings sand or slimes. Sulphide minerals disproportionately reporting to the fine fraction of the waste rock was not unexpected for a vein deposit and was shown to be the case in the waste rock dump analysis (Table 9). However, carbonate minerals also preferentially occur in veins and the NP was also much higher in the < 2 mm compared to coarser size fractions of the waste rock. Average NPR values were lower in the < 2 mm size fraction of one of the two dumps, but in both cases the NPR was greater than 2.

The most common reasons for discrepancies between laboratory measurements of NP and neutralization in the actual wastes under site conditions are:

- contribution of non-neutralizing Fe and MnCO_3 in minerals such as ankerite $[\text{Ca}(\text{Ca},\text{Mg},\text{Fe})\text{CO}_3]$ and siderite (FeCO_3) ³ to the laboratory NP measurement; and
- too large a contribution of insufficiently reactive silicate minerals to the laboratory NP measurement.

While Snip Mine was operating, all the carbonate was assumed to be calcite. Evidence of other carbonate species came from Rietveld XRD analysis (Table 4), in which the samples of whole tailings and cycloned sands contained 13 to 16% calcite, 3 to 4% ankerite and 1 to 2% siderite (weight percent). Unfortunately the potential occurrence of ankerite and siderite was not recognized until the end of the closure test work, so there was no effort to quantify or remove its impact from the measured NP. Assuming three quarters of the ankerite is Ca or MgCO_3 , the analyzed samples contained 15 to 18% Ca or MgCO_3 and 1 to 3% Fe and MnCO_3 , with Fe and MnCO_3 accounting for up to 15% of the total carbonate-C. Assuming these samples are representative of the entire deposit, the contribution of Fe and Mn CO_3 was typically 10% or less of the Sobek-NP, an amount that would have only slightly increased the number of samples with an NPR < 1.

Typically in ABA analysis, 5 to 15 kg/t of the Sobek and modified-NP comes from more slowly reactive aluminosilicate minerals (Price and Kwong, 1997). The most common reason for a higher contribution of slowly reactive silicate minerals to the Sobek-NP is if there is an excessively high acid addition. Unfortunately, there was no information on the fizz ratings used in the operational Sobek analyses, and Carbonate-NP, which could have been a useful comparison, was only measured on a sub-set of the surface waste rock samples. Assuming the correct acid addition was used, the data suggested that the measured NP was a reasonably accurate measurement of neutralization available in the tested materials.

³ While Mn is not part of the standard formula for ankerite and siderite, it may replace other cations.

The limited mineralogical and carbonate-NP data suggested there were sufficient Ca and MgCO₃ to neutralize the majority of the potential acidity in the PAG samples. Drainage pH, acidity and trace metal solubility when the Ca and MgCO₃ is eventually depleted will depend on a large number of factors including the contribution of:

- external NP (e.g., alkalinity in groundwater) and AP (e.g., acidity in precipitation and leachate through topsoil) additions;
- rates of sulphide oxidation; and
- neutralization by silicate minerals.

Silicate minerals can contribute to the maintenance of neutral pH by supplying alkalinity that supplements Ca and MgCO₃ neutralization during early stages of weathering, slowing Ca and MgCO₃ depletion, or by acting as the sole source of neutralization when all of the Ca and MgCO₃ are depleted. By the time all the Ca and MgCO₃ are exhausted, the sulphide content will also be greatly reduced, with the oxidation of the remainder potentially further reduced by occlusion within particles or surface coatings. Occlusion within particles and surface coatings will also limit the reactivity of the more reactive silicate minerals.

3.2.1 Alternative NP Measures

As part of the closure test work, a number of less expensive methods of measuring NP than the Sobek or Modified procedures were tested on a sub-set of waste rock samples (NDN, 1998). These methods involved calculating the NP from the Leco-C, aqua regia/ICP levels of Ca, and aqua regia/ICP levels of Ca + Mg. For most of the samples tested the NP calculated from Leco-C (C-NP), assuming all C is calcite, had a 1:1 relationship with the Modified-NP, indicating that most of the Modified-NP comes from carbonate minerals, and any contribution of non-neutralizing Fe and Mn CO₃ to the C-NP was equal to the contribution of Fe and Mn CO₃ and/or silicate minerals to the Modified-NP. For some samples, the NP calculated from Leco-C was slightly higher than the Modified-NP, an indication that a small but significant portion of the carbonate was Fe and MnCO₃, supporting the conclusion from the limited XRD analysis. Notably, a similar 1:1 relationship between Sobek-NP and the CO₂-NP was observed in ABA results for the waste rock used for dam construction (Table 8). Analysis of total- or inorganic-C is recommended as it provides a useful check on the Modified or Sobek-NP results and the amount of acid used in the Sobek-NP procedure.

NP calculated from the aqua regia/ICP levels of Ca (Ca-NP), assuming all the Ca was calcite, was slightly less than the modified-NP. Possible explanations for this are that:

- the Ca-NP did not account for NP present as MgCO₃; and
- the contribution of Fe and MnCO₃ and silicate mineral contribution to the Modified-NP.

Other Ca-containing minerals identified in the XRD analysis were traces of epidote, gypsum and apatite. Ca may also occur as trace amounts in albite.

NP calculated from the aqua regia/ICP levels of Ca plus Mg, assuming all the Ca and Mg were carbonate, was always higher, and in some cases more than doubled the modified NP. This is explained by the high concentrations of the Mg containing, non-carbonate minerals chlorite and

biotite. Calculation of NP from aqua-regia Ca or Mg is not generally recommended because of the large number of potentially confounding factors.

3.2.2 Assessment of the ARD Potential

The pre-mine assessment of whether material would be ARD generating was based on the assumption that a positive NNP indicated materials did not have the potential to generate ARD. Based on this assumption, it was concluded that the waste rock would be non-PAG, the ore would be PAG, the tailings slimes and whole tailings produced from sulphide ore were PAG, and the tailings sand would be non-PAG (Norecol Environmental Consultants Ltd., 1990). Flooding in the impoundment was proposed to prevent ARD from the PAG materials, the tailings slimes and tailings sand created from sulphidic ore.

At mine closure, the assessment of whether material at Snip Mine would be ARD generating was based on the NPR (NP/AP) rather than the NNP, with ARD thought likely for an NPR < 1, uncertain for an NPR of 1 to 2, and considered unlikely for an NPR > 2⁴. The rationale for these criteria and for using the ratio between the NP and the AP rather than their difference, and potential confounding factors are outlined in Appendix A. An NPR < 1 and an NNP < 0 are the same thing, and were interpreted in the same manner in the pre-mine and closure ARD assessments. The main difference between ABA criteria used in the closure ARD assessment compared to pre-mine was the recognition that ARD was possible in materials with an NPR of 1 to 2. As illustrated in the sections discussing each of the site components, this greatly increased the amount of material considered to be potentially capable of generating ARD.

Based on the closure NPR criteria, the ABA results indicated that the whole tailings and backfilled tailings sand, which had median NPR values of 1.2 to 2.0 and 5th percentile NPR values of 0.8 to 1.6 (Tables 7 and 10), included material with a likely, uncertain and no ARD potential. The waste rock, with median NPR values of 1.3 to 7.6 and 5th percentile NPR values of 1.1 to 2.5 (Table 7), included material with an uncertain or no ARD potential.

As discussed in Section 3.2, there were concerns regarding the accuracy of the Sobek-NP measurements (e.g., the fizz rating and whether the correct amount of acid was used) and whether some sort of correction (e.g., subtraction) was required before calculating the NPR values. Conversely, there were suggestions that the NPR criteria actually separating PAG and non-PAG materials was less than 2 and that an NPR criteria between 1 and 2 should have been used (e.g., 1.5). These issues were discussed during the closure review primarily as part of the review of the liability and environmental risk created by the backfill, left exposed in upper, unflushed workings.

3.3 Elemental Composition

3.3.1 Aqua Regia Digestible Elemental Composition

Data on the elemental composition⁵ of different waste materials was determined by ICP after an aqua regia digest. Analysis was conducted on:

⁴ Assuming materials were exposed to oxygen and water.

⁵ The aqua regia digestion does not completely digest all the silicate minerals, but provides a good measure of the elemental composition of the more weatherable minerals.

- operational samples of ore tailings, waste rock and tailings sand throughout the life of the mine;
- samples taken from the tailings beach in 1997;
- samples taken from the waste rock dumps in 1997 and 1998; and
- samples taken from the borrow pits, plant site and other areas around the site in 1997.

Median and 95th percentile values from the operational monitoring of the waste rock and backfilled cyclone sand, and the 1997 sampling of the tailings beach are shown in Table 11. The three materials contained similar concentrations of major metals, all relatively enriched in Fe, Ca and Mg, and low in Al and Na. The main Fe and Mg minerals are biotite and chlorite, with lesser amounts of ankerite. Fe also occurs in pyrite, plus the minor amounts of pyrrhotite and siderite. The main Ca minerals are calcite and ankerite.

The most notable feature of the trace element data was the relatively low concentrations compared to many metal mines. Of the trace metals that are typically a concern at mine sites, only median values of Ag, As and Cd, were an order of magnitude higher than typical elevated crustal concentrations. With the exception of Cu, average concentrations of almost all the trace metals⁶ were lower in the waste rock than the tailings beach and the cycloned tailings sand.

It is important to note that the aqua regia digestible elemental composition is a measure of trace metal concentrations in the solid phase. Unfavourable geochemical conditions, such as accelerated weathering (e.g., trace metals in rapidly oxidizing sulphides) and lower than background pH values, could result in elevated trace metal mobility and metal loads even if the solid phase trace metal concentrations are not elevated.

3.3.2 Weak Acid Soluble (EPA 1312 Test) Sulphate and Metals

In the EPA 1312 procedure, a 50 g of sample is added to 1.0 L of de-ionized water, weakly acidified to pH 4.2 with sulphuric and nitric acid. The sample is agitated for 18 hours at 30 rpm, and then the solution is filtered and acidified. The large solution:solid is intended to reduce solubility constraints. The procedure was run on the < 2 mm size fraction of three waste rock samples from each of the 150 and 440 dumps, and three samples of tailings sand (backfill). The objective of the procedure was to measure the concentration of presently soluble metals. Whole tailings were not analyzed because the purpose in running the test was to predict post-mining drainage quality from the underground workings.

The results are shown in Table 14. Although the waste rock had lower aqua regia concentrations (Table 11), maximum individual and average soluble trace metal loads (As, Cd, Cu, Mo and Zn) were typically higher in the leachate from 440 Dump waste rock than the tailings sand. Probable reasons for this were that:

- the waste rock samples, unlike the tailings sand, have been exposed to weathering, resulting in the partial oxidation of primary minerals and conversion of metals to a more soluble form; and

⁶ Including metalloids

- the higher NP resulted in a higher pH, reducing metal solubility in the leachate from the unweathered tailings sand.

Unlike, trace metals, the tailings sand (backfill) had the highest weak acid soluble mg/kg of Ca, Mg, Fe, Mn and SO₄. Potential explanations include smaller particle size and higher concentrations of calcite, ankerite and gypsum. Dissolution of calcite and ankerite will raise the pH, a factor that might have contributed to the lower concentration of trace metals and Al in the leachate from the tailings sand. Unfortunately, the test results did not include the final solution pH. The higher concentration of soluble trace metal metals in the 440 Waste Rock Dump than the 150 Dump samples is attributable to the higher concentration of sulphide minerals, as indicated by the higher AP (Table 7), higher concentrations of trace metals, and the longer surface exposure, resulting in greater weathering.

The relatively high leachate concentrations of K, especially in the tailings sand, but also in the waste rock, indicated significant potential for acid neutralization by phyllosilicate minerals, such as biotite and sericite, which are common components of the Snip rock. If K is released from broken edges, neutralization from this source may decrease significantly over time.

3.3.3 De-Ionized Water Soluble (Modified SWEP Test) Metals

In the modified SWEP test, a 50 g of sample is added to 1.0 L of de-ionized water. The sample is then rotated for 24 hours at 10 rpm, before the solution is filtered and acidified. The modified SWEP test has a similar solid to liquid ratio to the EPA 1312 test, again with a large excess of solution added to reduce solubility constraints. The procedure was run on the three samples each of tailings and tailings sand. The same samples were also used for humidity cell tests and the tailings sand samples were also tested using the EPA 1312 test.

The results are shown at the bottom of Table 15. As a result of the higher analytical detection limits than those used in the EPA 1312 test, all the trace metals except Mo were below the detection limit. Concentrations of de-ionized water-soluble major cations were two times higher in the whole tailings than the tailings sand, presumably because the whole tailings consisted of smaller particles with a higher surface area. The Ca, Mg and Na results for Modified SWEP and EPA 1312 for tailings sand samples were very similar indicating the weak acidity of the solution in the EPA 1312 procedure had little impact on the solubility of these elements. The weak acidity of the solution in the EPA 1312 test resulted in slightly higher concentrations of K and Mn and lower concentrations of Mo. The lower Mo concentrations are attributable to a lower Mo solubility at lower pH. The higher K probably resulted from greater cation exchange with H⁺ due to higher H⁺ concentrations in the solution added to the samples in the EPA 1312 test.

3.3.4 NAG Test Results

In the single-addition NAG test, 250 mL of 15% peroxide is added to a 2.5 g sample. The solution is allowed to sit overnight, gently heated and then vigorously boiled. The pH (NAG pH) and the acidity are measured after the sample is allowed to cool. The test was run on sub-samples of two of the whole tailings and three tailings sand samples used in the humidity cell tests. The resulting NAG pH and kg H₂SO₄/t calculated from the NaOH required to titrate the low pH solution back up to pH 4.5 are shown in Table 15.

The lowest NAG pH values were 2.6 and 3.3. These values were obtained for samples with a CO₂-NPR < 1. NAG pH values of 4.5 and 5.3 were obtained for samples with a CO₂-NPR of 1.14 and 1.06, and pH 7.2 for the sample with a CO₂-NPR of 1.29. The most likely explanation for acid NAG pH values for samples with a CO₂-NPR > 1 was that a portion of the CO₂-NP came from the non-neutralizing Fe and MnCO₃ in ankerite and siderite.

Like other ARD tests, the NAG procedure has the potential to both overestimate and underestimate the ARD potential. One way it might underestimate the ARD potential is if it overestimates acid neutralization by carbonate alkalinity, which produces HCO₃⁻ lost in the drainage. Another way it might underestimate the ARD potential is if there is incomplete sulphide oxidation, a concern with potentially high sulphide-S materials like those at Snip. Repeating the peroxide addition and analysing the solution for sulphate can be used to check that all the sulphide has been oxidized. The NAG test might overestimate the ARD potential if by accelerating the acid production it minimizes the kinetically slow neutralization by silicate minerals.

3.4 Kinetic Test Work

Kinetic test work performed on Snip mine materials included:

- pre-mine humidity cell tests run in 1988/1989 on three ore samples and one tailings sample;
- a column study run in 1994 with tailings, backfill and waste rock (McLeod and Higgs, 1995);
- several long-term humidity cell tests started as part of the closure planning, some of which are ongoing;
- NAG tests conducted on subsamples of the materials used in the long-term humidity cell tests; and
- three field test pads built with cycloned tailings sand at the time the mine closed.

A concern raised in some of the reporting of Snip's pre-mine laboratory kinetic test results was the difference in temperature and as a result how rates in the lab might be higher than rates 'in the field'. Temperature in wastes at the site depend on a number of factors including the rate of oxidation and the rate of heat loss, in addition to the site temperature. Sulphide oxidation is an exothermic reaction (produces heat) and thus the temperature in the oxidizing wastes at the site may be higher or lower than the laboratory. While surrounding air temperatures are lower in the field than the laboratory, the dissipation of heat produced by sulphide oxidation may be greater in small, more uniformly leached laboratory samples than in larger, potentially more irregularly leached waste at the site. Another temperature related factor is that calcite solubility typically increases as the temperature decreases.

Differences between laboratory rates and those in the field may also result from poor aeration or from insufficient drainage. Poor aeration, limiting sulphide oxidation may be a concern in the tailings sand column where the high values of leachable Mn were attributed to sub-oxic conditions.

3.4.1 Pre-Mining Humidity Cell Study

The pre-mine humidity cell tests run in 1988/1989 on three ore samples and one tailings sample were run with six days of humid air and two washes of 250 mL on the seventh day. The samples weighed approximately 200 g. The ore samples were crushed to 50% greater than 149 µm while the

tailings sample was 85% less than 74 μm . The two tests on the lower %S ore cells, which had the lowest sulphate production, were stopped and the analyses conducted on leachate from the other cells was reduced after 13 weeks to reduce costs. Data available for the first 9 weeks showed that ore sample #1 with 4.4% S produced 5.7 to 6.3 mg $\text{SO}_4/100\text{g}/\text{week}$ and ore sample #7 with 5.8% S produced 4.6 to 14 mg $\text{SO}_4/100\text{g}/\text{week}$ (Norecol Environmental Consultants Ltd., 1988). Sulphate production from week 10 to 31 for ore sample #5 with 14.1% S was 32 to 46 mg $\text{SO}_4/100\text{g}/\text{week}$ and for the tailings sample with 8.9% S was 34 to 77 mg $\text{SO}_4/100\text{g}/\text{week}$ (Johnston, 1989). The higher rate of sulphate production in the tailings cell was attributed to its finer particle size and therefore larger surface area. The pH from week 10 to 31 for ore sample #5 and the tailings sample was 6.5 to 7.5. Data beyond week 31 was not found.

The stated purpose of the study was to measure the rate of acid generation and check whether tailings in the impoundment could be temporally beached without creating acidic drainage (Norecol Environmental Consultants Ltd., 1990). Assuming that each mole of sulphate comes from pyrite and therefore corresponds to 2 moles of acidity, and each mole of CaCO_3 neutralizes 1.5 moles of acidity, the rate of 10 mg $\text{SO}_4/100\text{g}/\text{week}$ from 5% S indicates the rate of NP depletion was 0.8 kg CaCO_3/t per yr from the lower %S ore cells. 60 mg $\text{SO}_4/100\text{g}/\text{week}$ from 9% S, the approximate rate from the tailings cell, was equivalent to 4.9 kg $\text{CaCO}_3/\text{t}/\text{yr}$ or 0.55 kg $\text{CaCO}_3/\text{t}/\text{yr}$ per %S. As might be expected from the high NP (126 and 142 kg CaCO_3/t), all four cells produced neutral pH drainage throughout the 31-week period of monitoring. At the measured rates of NP depletion it would take decades for NP depletion and the ARD production by the humidity cells.

3.4.2 1994/1995 Column Study

As part the ARD characterization in 1994/1995 (McLeod and Higgs, 1995), a column study was conducted on three materials; fresh waste rock, fresh whole tailings, and 3-year-old backfilled cycloned sand from a stope that showed signs of Fe staining. Three columns were run for each material. Each column had an inside diameter of 14 cm and was 1.8 m high. Sample weights were 33 to 44 kg, and the cells were run at room temperature. The tailings and cycloned sand were not altered, while the waste rock was crushed < 2.5 cm. The columns were run at room temperature. Water was added twice a week, one small addition at the start of the week, with the majority added on Friday. The amount of drainage varied between 1295 mL and 1860 mL, with the higher rate added to the backfill because it might be leached by groundwater. The waste rock columns were leached with 4.85 L rather than the regular rate during the first week. All the columns were inoculated with *Thiobacillus ferrooxidans*.

One of the three columns was run for 30 weeks, with the other two run for 42 weeks. Composition of the tested materials and the resulting drainage chemistry results are shown in Table 13. As expected, given the high NP, all columns produced neutral pH drainage throughout the 42-week study. The two types of tailings had much higher Ca (400 to 800 mg/L versus 10 to 45 mg/L) and sulphate (900 to 1700 mg/L versus 10 - 50 mg/L) concentrations in their drainage than the waste rock. This was attributed to the lower sulphide content, removal of residual sulphate by the higher initial leaching and the lower surface area of the waste rock. The concentration of Ca was an order of magnitude higher than the concentrations of Mg. In many cases, the Mg concentration peaked several weeks into the study. The concentrations of leachable Na and Si may have indicated small, but significant silicate weathering in the two tailings samples.

There were low levels of most trace metals in the drainage from all three waste types. The exception, the high values of Mn (2 to 10 mg/L) in the drainage from the tailings sand was attributed to sub-oxic conditions. The highest trace metal values, other than Mn, were for Zn (20 to 80 µg/L from the whole tailings).

3.4.3 Ongoing Humidity Cell Tests

Thirteen humidity cells were initiated between July and November 1998 as part of the ML/ARD work conducted at closure. The objectives in running the cells were to provide a measure of the rate of:

- sulphide oxidation;
- Ca and Mg carbonate depletion;
- weathering of potential silicate NP sources; and
- trace metal release.

By the end of February 2004, the cells had been running for 272 to 292 weeks, more than 5 years. The test samples consisted of three whole tailings, four tailings sand (backfill) and two waste rock cells (150 Dump and 180 Dump), plus two whole tailings and two tailings sand samples with half their CO₂-NP removed by pre-treating them with dilute HCl. The particle size, ABA, elemental composition, NAG test and Modified SWEP results for whole tailings samples #1 to 3 and tailings sand samples #1 to 3 are provided in Table 15. EPA 1312 test results for tailings sand samples #1 to 3 are provided in Table 14. The whole tailings samples used in the humidity cells had median or higher AP values and median or lower NP values (Table 7). Sobek-NPR values of the whole tailings samples were 1.21, 1.55 and 1.71. CO₂-NPR values were 0.98, 1.12 and 1.29. The analyzed tailings sand samples used in the humidity cells all had AP and NP values higher than the median and NPR values lower than the median. Sobek-NPR values of the analyzed tailings sand samples were 1.04, 1.28 and 1.43. CO₂-NP values were 0.87, 1.06 and 1.14. At the time of writing, no data was available on the composition of the fourth tailings sand or the waste rock samples.

Each cell contains approximately 1 kg of material. The weekly treatment cycle was 3 days humid air, 3 days dry air and flushed with 500 mL on day 7. Drainage pH and conductivity were measured weekly. Drainage sulphate, alkalinity and metal concentrations were initially measured every second week (biweekly). Since there were no major changes in drainage chemistry, starting in April 2000, the analysis of sulphate, alkalinity and metal concentrations were reduced to monthly to reduce costs. Loading rates were calculated by multiplying the leachate concentration by the drainage volume. Overall metal release, pH⁷ and molar ratio⁸ of Ca+Mg/SO₄ data, and data for weeks 10 to 30 and the last 24 weeks (September 2003 to February 2004) are provided in Table 16. Weeks 10 to 30 were used to characterize initial release rates to avoid the period where dissolution of pre-existing salts and products of sample pre-treatment would mask weathering at the start of the tests.

⁷ pH is a logarithm and thus the average is a geometric average.

⁸ The average for a ratio such as molar ratio of Ca+Mg/SO₄ is the ratio of the average moles of Ca + Mg divided by the average moles of SO₄.

To date, drainage pH has remained neutral, with 5th and 95th percentile values between 7.3 and 8.5. Average pH values for the last 24 weeks for the various cells varied between 7.5 and 7.8. Removal of half the CO₂-NP has not significantly changed the pH.

All the test cells exhibited relatively high sulphate production indicating significant sulphide oxidation. Average whole tailings sulphate production was 105 mg/kg/wk overall (88 to 135 mg/kg/wk for the individual cells), increasing to 149 mg/kg/wk in the last 24 weeks (114 to 178 mg/kg/wk for the individual cells). Average tailings sand sulphate production was 138 mg/kg/wk overall (90 to 191 mg/kg/wk for the individual cells). Average tailings sand sulphate production was initially higher than the whole tailings, but decreased to 108 mg/kg/wk in the last 24 weeks (88 to 129 mg/kg/wk for the individual cells). Average waste rock sulphate production was 28 and 88 mg/kg/wk for the two cells, increasing to 41 and 94 mg/kg/wk in the last 24 weeks. The higher initial overall sulphate production rate for the tailings sand compared to the whole tailings could be attributed to the slightly higher sulphide content or better aeration. The lower sulphate production rate in the last 24 weeks for the tailings sand compared to the whole tailings perhaps resulted from the difference in particle size and the fact that the tailings sand had a smaller proportion of fine grained sulphides. The lower sulphate production rate for the waste rock compared to the tailings likely resulted from the lower sulphide-S and the larger particle size.

The following reactions are used to estimate acid generation and calcite depletion from the rate of sulphate leaching, with the assumption that all the sulphate comes from the pyrite.

1. $\text{FeS}_2 + 7/2 \text{H}_2\text{O} + 15/4 \text{O}_2 = 2\text{SO}_4^{2-} + \text{Fe}(\text{OH})_3 + 4\text{H}^+$
2. $\text{CaCO}_3 + \underline{2\text{H}^+} = \text{Ca}^{2+} + \text{H}_2\text{CO}_3$
3. $\text{CaCO}_3 + \underline{\text{H}^+} = \text{Ca}^{2+} + \text{HCO}_3^-$

The formula weight of sulphate is 96, hydrogen is 1 and CaCO₃ is 100.09. The mg H⁺/kg/wk is converted to mg CaCO₃/kg/wk by multiplying by 50 for reaction 2 and 100 for reaction 3 (see Appendix A). Thus for reaction 2, the acidity produced by pyrite oxidation resulting in 1 mg SO₄²⁻/kg/wk is equivalent to 1.0426 mg CaCO₃/kg/wk or 54.17 mg CaCO₃/kg/yr. For reaction 3, the acidity produced by pyrite oxidation resulting in 1 mg SO₄²⁻/kg/wk is equivalent to 2.0852 mg CaCO₃/kg/wk or 108.34 mg CaCO₃/kg/yr. Based on the above, humidity cell rates of 50, 100 and 150 mg SO₄²⁻/kg/wk for Reaction 2 are equivalent to NP depletion of 2.7, 5.4 and 8.1 kg CaCO₃/t/yr. Humidity cell rates of 50, 100 and 150 mg SO₄²⁻/kg/wk for Reaction 3 are equivalent to NP depletion of 5.4, 10.8 and 16.2 kg CaCO₃/t/yr. At a rate of 5 kg CaCO₃/t/yr, assuming all the CO₂-NP is CaCO₃, it would theoretically take thirty years to exhaust a CO₂-NP of 150 kg CaCO₃/t, and fifteen years if the rate were 10 kg CaCO₃/t/yr.

The molar ratio of Ca + Mg / SO₄ provides information on relative rates of sulphide oxidation (the presumed source of sulphate) and dissolution of neutralizing carbonate minerals (the presumed source of Ca and Mg). The average Ca + Mg / SO₄ ratio was 1.1 for each of the whole tailings and tailings sand cells. Over the last 24 weeks, the average ratio for each whole tailings and tailings sand cell decreased to 1.0. The low molar ratio of Ca + Mg / SO₄ suggests that reaction 2 predominates in the neutralization of acidity generated by sulphide oxidation. The hypothesis that

reaction 2 predominates in the neutralization of acidity generated by sulphide oxidation is supported by the relatively low drainage alkalinity. Average whole tailings drainage alkalinity was 4.7 mg CaCO₃ /kg/wk overall, decreasing to 2.6 mg CaCO₃ /kg/wk in the last 24 weeks. Average tailings sand alkalinity was slightly higher than the whole tailings, 5.7 mg CaCO₃ /kg/wk overall, decreasing to 3.9 mg CaCO₃ /kg/wk in the last 24 weeks.

The two waste rock cells had higher overall average molar ratios of Ca + Mg / SO₄ (1.5 and 1.2) than the two types of tailings. The two waste rock cells also had higher overall average drainage alkalinity (9.8 and 12.4 mg CaCO₃/kg/wk) and unlike the tailings, average waste rock alkalinity for the last 24 weeks was not lower than the overall average. Various factors may have contributed to the lower molar ratio of Ca + Mg / SO₄ in the tailings cells. These include the higher rates of acid generation and the closer contact between sulphide and carbonate minerals, both of which would likely favour Reaction 2 versus Reaction 3.

The average rates of sulphate and Ca and Mg production and the original composition of the test materials can be used to estimate the time to exhaust sulphide and carbonate-NP. In January 2002 (week 184), the predicted time to exhaust the sulphide from the whole tailings cells #1, 2 and 3 were 29 to 37 years, 30 to 46 years to exhaust the total CO₂-NP, and 23 to 35 years to exhaust 75% of CO₂-NP. The predicted time to exhaust the sulphide from the backfilled tailings sand cells #1, 2 and 3 were 24 to 30 years, 20 to 24 years to exhaust the total CO₂-NP and 15 to 18 years to exhaust 75% of CO₂-NP.

Average base cation release from the whole tailings was:

- overall, Ca 27 - 46 mg/kg/wk, Mg 6.5 - 10 mg/kg/wk, K 3.2 - 3.8 mg/kg/wk and Na 0.4 - 0.5 mg/kg/wk; and
- for the last 24 weeks, Ca 31 - 51 mg/kg/wk, Mg 9 - 16 mg/kg/wk, K 1.5 - 4.3 mg/kg/wk and Na 0.5 - 0.9 mg/kg/wk.

Average base cation release from the tailings sand was:

- overall, Ca 31 - 68 mg/kg/wk, Mg 6.8- 12.9 mg/kg/wk, K 3.5 - 5.5 mg/kg/wk and Na 0.2 - 0.3 mg/kg/wk; and
- for the last 24 weeks, Ca 28 - 36 mg/kg/wk, Mg 6 - 14 mg/kg/wk, K 1.5 - 2.1 mg/kg/wk and Na 0.1 - 0.3 mg/kg/wk.

Average base cation release from the waste rock was:

- overall, Ca 10 & 37 mg/kg/wk, Mg 3.7 & 3.8 mg/kg/wk, K 2.6 & 4.5 mg/kg/wk and Na 0.2 & 0.3 mg/kg/wk; and
- for the last 24 weeks, Ca 13 and 36 mg/kg/wk, Mg 5.1 mg/kg/wk, K 2.0 and 3.8 mg/kg/wk and Na 0.2 mg/kg/wk.

The rate of release was Ca > Mg > K > Na. Although Ca release was significantly higher than Mg, the lower molar weight of Mg than Ca (24.312 versus 40.08) meant that moles of Mg produced were more than 50% of the moles of Ca for many of the whole tailings and tailings sand cells. The main sources of Ca are undoubtedly calcite and ankerite. Potential sources of Mg include ankerite and the Mg silicates biotite and chlorite. The most probable sources of K in the Snip wastes are cation exchange with K in biotite and sericite. The relatively high Mg and K release rates suggest

there was significant weathering and neutralization by silicate minerals. However, it is notable that from weeks 10 to 30 to the last 24 weeks, there was a significant decrease in the K release rate for most whole tailings cells (2.9 - 5.3 versus 1.5 - 2.6 mg/kg/wk) and the backfilled tailings sand (4.3 - 7.2 versus 1.4 - 2.1 mg/kg/wk). This suggests that by the time the carbonate-NP was exhausted, silicate weathering may be significantly lower than the measured results. The decrease in K was greatest for the samples pre-treated with acid to remove half their CO₂-NP.

The highest average release rate for a trace metal was for Mn; 5 & 10 µg/kg/wk from waste rock, 58 - 119 mg/kg/wk from the tailings sand, and 45 - 83 mg/kg/wk from the whole tailings. Median values of most trace metals were below the detection limit (e.g., As, Cd, Cr, Ni, Pb, Sb and Se – Table 18). The exceptions were Mo, Zn and occasionally Cu, all of which had relatively low detection limits. Average rates over for the last 24 weeks were Zn 2.2 to 5.2 µg/kg/wk, Mo 1.4 to 5.9 µg/kg/wk and Cu 0.5 to 1.8 µg/kg/wk. Mo is not considered to be a concern because of the lack of grazing or drinking water use for drainage at the site. Due to solubility constraints, Cu is only likely be a concern if there little or no dilution or acidic drainage. Due to higher solubility, Zn is more commonly a concern in neutral pH drainage than Cu, and unlike other trace elements, the average Zn release generally increased from weeks 10 to 30 to the last 24 weeks. Trace metal concentrations were lowest in the drainage from the waste rock. The whole tailings and tailings sand had the highest maximum weekly drainage trace metal concentrations; As 44-89 µg/L, Cd 3-7 µg/L, Cu 8-45 µg/L, Mo 11-97 µg/L, Ni 7-60 µg/L, Pb <10-22 µg/L, Sb 23-67 µg/L, Se 23-120 µg/L and Zn 51-430 µg/L. The maximum Se concentrations of 105 to 120 µg/L were from the leachate from the whole tailings cells, for which the concentrations of Se were above the detection limit both during week 10 to 30 and the last 24 weeks.

Humidity cell release rate data reported in Table 16 and used in subsequent modeling are potentially misleading for trace elements that were commonly below the detection limit, such as Cd, Cr and Ni (Table 18), because when values were below the detection limit, the detection limit was used in the release rate calculation. For the trace elements commonly at or below the detection limit, the reported rates were strongly affected by the detection limit. Detection limits were As < 30 µg/L, Cd < 2 µg/L, Cr 20 µg/L, Cu 2 µg/L, Mo 5 µg/L, Ni 5 and 20 µg/L, Pb 10 µg/L, Sb 20 µg/L, Se < 20 µg/L and Zn < 5 µg/L. The interpretation of data is especially confusing for Ni because the detection limit changed from 20 to 5 µg/L part way through the analysis. Arsenic and Sb concentrations above the detection limit were observed from week 10 to 30 of the whole tailings and the backfilled tailings sand, but not in the last 24 weeks or in either period for the NP depleted tailings cells and the waste rock (Table 18). Lower As and Sb concentrations in the NP depleted cells and the last 24 weeks of the whole tailings and the backfilled tailings sand may have been due to co-precipitation with Fe oxyhydroxides.

Another potential problematic factor in the interpretation of the humidity cell data was the large variability in the volume of drainage collected each week (Table 17). The volume of drainage collected from the two types of tailings ranged from 25 to 535 mL. Where detection limits were used in the loading calculation, the volume of drainage effectively determines the reported release rate. This is illustrated by the variability in Cr release rate despite Cr never exceeding the detection limit. Another potential problem was that if most of the added 500 mL was left in the cell, weathering during the next week might have been impeded because the sample was too wet. Conversely when the volume of water recovered was low, weathering during the previous week might have been impeded because the sample was too dry.

In addition to the large range in values, there were some consistent differences noted in the volume of drainage collected. The first was the lower volume of drainage collected from the two types of tailings compared to the waste rock. The overall average volume of drainage collected was 195 to 222 mL for the whole tailings cells, 215 to 248 mL for the tailings sand, compared to 388 and 403 mL for the two waste rock cells. The lower recovery from whole tailings and backfilled tailings sand versus waste rock may be due to their higher water holding capacity, with water retained in the cell after flushing lost through evaporation during the dry air portion of the humidity cell cycle.

The second notable difference was between the average volume of drainage collected during weeks 10 to 30 and overall compared to the last 24 weeks. All but one of the whole tailings and backfilled tailings sand had lower average water recovery overall and during week's 10 to 30, often 50 to 75% lower, than the last 24 weeks. The trend for the two waste rock cells was the opposite with a 15% increase in average water recovery during the last 24 weeks compared to overall and weeks 10 to 30. The consistent nature of these temporal differences indicates a systematic change in the operating procedures for water recovery. The low water recovery during the last 24 weeks might indicate over drying of the samples, which may have reduced sulphide oxidation. The relatively high sulphate production indicated that this was not the case for the whole tailings cells, but dryness may have contributed to the lower sulphate production observed in the coarser, tailings sand during this period.

There is also a concern that the volume of leachate recovered was insufficient to completely dissolve base cations, sulphate and trace metals. A relatively large volume of water is used in the humidity cell procedure to avoid solubility constraints for important species such as Ca and sulphate. Sulphate concentrations over the last 24 weeks in some of the whole tailings humidity cells were becoming high enough that solubility might potentially have limited dissolution (Table 18). Clearly improvements are required to the humidity cell program to ensure this aspect of the test does not impact weathering conditions and weekly dissolution.

To date, removal of half the CO₂-NP has had relatively little impact on the drainage chemistry and loadings. The main exception was a slight difference in overall average drainage alkalinity. For the whole tailings, overall average drainage alkalinity was 4.7 to 5.2 mg CaCO₃/kg/wk in the un-depleted cells and 4.2 and 4.6 mg CaCO₃ /kg/wk from NP-depleted cells. The difference was not observed in the last 24 weeks, when average alkalinity from the whole tailings was 2.1 to 3.1 mg CaCO₃/kg/wk for the un-depleted cells and 2.3 and 2.8 mg CaCO₃ /kg/wk for the NP-depleted cells. Overall average alkalinity for the tailings sand was 5.8 to 6.4 mg CaCO₃ /kg/wk in the un-depleted cells and 5.0 mg CaCO₃/kg/wk in both NP-depleted cells. Unlike the whole tailings, a difference was also observed for the tailings sand in the last 24 weeks, when average alkalinity from the tailings sand was 3.7 to 4.5 mg CaCO₃ /kg/wk in the un-depleted cells and 3.6 CaCO₃/kg/wk from NP-depleted cells.

Removal of CO₂-NP from several humidity cells was initiated on the recommendation of regulators who suggested that this would help them answer whether silicate minerals are capable of maintaining a neutral drainage pH if Ca and MgCO₃ are depleted prior to acid generating sulphide minerals, and/or what the resulting drainage chemistry would be like (e.g., acidity and contaminant

concentrations). The recommendation was to follow a protocol used at the Huckleberry Mine. However, the Huckleberry rock that this procedure was designed for has relatively little NP compared to a fairly high AP, so there was expected to be relatively little change in the magnitude of the AP when the CO₂-NP was exhausted. In contrast, the rock at Snip had both a high NP and AP, and unlike the Huckleberry rock, by the time Ca and Mg CO₃ are depleted, the sulphide content and the rate of acid generation is also likely to have been significantly reduced. After the potential flaws in removing only CO₂-NP were pointed out, the mine was far less willing to acquiesce to subsequent regulatory requests for additional information regarding post-closure site performance, especially if there was a potential that, despite the additional information, the issues would remain unresolved.

3.4.4 Backfill Field Test Pads

In November 1999, three small, approximately 1 tonne, field test pads were constructed outside the 300 Portal from backfilled tailings sand. The test pads consisted of piles of tailings sand placed on plastic sheets. The composition of the pile material is shown in Table 19. The NPR of the samples taken from the piles was from 0.8 to 1.0, with 4.7 to 5.6% sulphide-S. The pads were constructed at the request of the Ministry of Energy and Mines (MEM), who were concerned about the lack of monitoring of weathering underground and wanted test work done that might indicate when inaccessible, lower NPR material underground might start producing ARD. Another reason for MEM wanting field test pads was to show the difference between laboratory humidity cell rates and the rates under site climatic conditions. At the time they were constructed, the mine expressed justifiable concerns that weathering in the field pads may be very different from the backfill underground because of differences in factors such as temperature and leaching. The drainage from the test pads is not monitored.

3.5 Site Drainage Monitoring

Analysis of site drainage has included:

- biweekly monitoring of the tailings effluent and adjacent water courses potentially impacted by the mine during operation of the mine;
- monitoring of drainage underground to determine backfill water quality; this included data from Oct.1996, August 1997 and April 1998 provided in Snip Mine (1999a) and data from sampling carried out in January and February 1999;
- monitoring of the seepage from the 150, 300, crown pillar and 440 waste rock dumps (Snip, 1998); and
- post-closure monthly monitoring of drainage from the 130, 180 and 300 adits, behind the 130 bulkhead, discharge from the tailings impoundment spillway and adjacent water courses potentially impacted by the mine.

The results are discussed in the sections dealing with the relevant mine components.

4. Surface Waste Rock Dumps

Less than a third of the waste rock produced (229,000 t of the 869,388 t) was placed in surface dumps at the mouths of five portals (Snip Mine, 1998 and Snip Mine, 1999). The mass, volume and area of the different piles was as follows:

- 130 Level: 28,000 t, 14,000 m³, 0.94 ha
- 180 Level: 57,000 t, 28,500 m³, 0.85 ha
- 300 Level: 99,000 t, 44,500 m³, 1.3 ha
- 440 Level: 12,000 t, 6,000 m³
- 150 Level (Twin West): 33,000 t, 16,500 m³, 0.22 ha

No records were kept of where the waste rock came from, but it is assumed that the waste rock placed in these dumps came primarily from adit and ramp construction during the initial stages of mine development, while most of the waste rock produced during extraction of the ore was used as rock fill or as additional feed for the hydraulic backfill plant.

Geological mapping, and sampling and ABA analysis of the dumps were conducted for the closure plan. Sampling of the 180 and 300 dumps occurred in 1996. Sampling of the 130, 440 and 150 surface dumps occurred in 1998. Sampling was done with surface pits and trenches. Since all the dumps appear to have been end dumped, a representative cross section of the waste material in the dump should have occurred at the surface. Analysis of different size fractions was conducted on samples from the 150, 180, 300 and 440 dumps. Leachate tests (EPA 1312) were conducted on the samples of the 150 and 440 dump material. Alternate NP determinations were conducted on some samples from the 130, 150 and 440 dumps.

4.1 Acid Base Accounting Sampling and Analysis Results

4.1.1 Pre-Mining

Prior to mining, ABA data was obtained from twelve samples collected from throughout the deposit and fifteen samples from the sulphidic ore zone, the waste rock material with the highest ARD potential. The results are shown in Table 5. Of the general waste rock samples, 3 of 12 had an NPR between 1 and 2, the rest had NPR values > 2. Despite AP values as high as 119 kg CaCO₃/t, because of the consistently high NP, none of the samples had an NPR < 1.

The median AP was slightly higher for the sulphidic waste rock (49 versus 76 kg CaCO₃/t), and as a result 6 of 15 samples had an NPR between 1 and 2, and one sample had an NPR < 1.

Based on the NNP > 0 for all but 1 of the 27 samples and the assumption that any localized ARD would be neutralized by the surrounding overwhelmingly net acid consuming material, the conclusion prior to mining was that the waste rock would not produce ARD.

4.1.2 130 Dump (Haul Road)

The waste rock, comprised mainly of greywacke and siltstone, was mainly used to construct a haul road. Samples were taken from test pits rather than a trench due to the difficulty verifying the exact location of various buried power cables or pipes. No sampling occurred from the low-grade ore on top of the ramp because of the presence of equipment hauling ore to the mill. However, this material was milled at the end of mining.

Median ABA values for the test pit samples were total sulphur 1.02%, AP 35, NP 65 and an NPR of 2.7 (Table 7). A notable difference from the other dumps was the lower NP. The 5th percentile and

mean NPR⁹ values were 1.3 and 1.9. Based on three of the eight samples with an NPR of 1 to 2, the ARD potential was considered uncertain.

4.1.3 180 Dump

Most of the material in this dump was produced during initial mine development (NDM, 1998), with some additional rock added during mining of the crown pillar (Barrick, 2002). Samples for ABA analysis were collected from two 1.0-meter deep backhoe trenches dug at a 45 degree angle out from the portal. Samples were collected every 5 meters along the trench unless there was a visible variation in discoloration or rock type, in which case a sample was collected at that point. The trenches and significant variations in geology were photographed. Samples were composites of sub-samples collected from the top, middle and bottom of the trench, unless a discoloration or some other type of change was observed. Each sample was sieved into the +25 mm, 25-2 mm and < 2 mm size fractions.

The high NP (5th percentile of 124 kg/t) and relatively low AP (95th percentile of 69 kg CaCO₃/t) of these samples resulted in overall median and 5th percentile NPR values of 2.5 and 4.7 (Table 7). Crushed sample pH values were 8.0 to 8.5 in the < 2 mm fraction and above 9.0 in the coarser fractions. On average, the sulphide-S in the < 2 mm fraction was two to four times higher than the coarser fractions, while the average NP was less than two times higher (Table 9). Consequently, almost all NPR values < 2 were from the < 2 mm size fraction. Thirteen percent of the < 2 mm size fractions had an NPR less than 2. Based on these results, it was concluded that this waste rock had a low overall potential for ARD, with the ARD potential considered uncertain for the small portion of NPR 1 to 2 dump fines.

4.1.4 300 Portal Waste Dump

This was the largest surface waste rock dump. The waste rock consisted of rapidly weathering greywacke and sandstone. The material was dumped during development of the mine and was approximately 8 years old when sampled. Water observed flowing through and around this dump during spring freshet reported to the impoundment.

Three approximately 50 m long, 1 m deep trenches were dug by backhoe. Samples were collected every 13 meters, in the same manner as the 180 trench samples. A rough map and photographs were taken of discolorations or obvious geological variations in the trench.

The high NP (5th percentile of 126 kg/t) and relatively low AP (95th percentile of 83 kg CaCO₃/t) resulted in overall median and 5th percentile NPR values of 2.9 and 5.9 (Table 7). Like the 180 waste rock, crushed sample pH values were 8.0 to 8.5 in the < 2 mm fraction and above 9.0 in the coarser fractions. On average the sulphide-S in the < 2 mm fraction was almost two times higher than the coarser fractions (Table 9). The average NP was also higher in the < 2 mm fraction, although not by as much as the AP. Consequently, the < 2 mm particle size fraction had the lowest NPR values. Nine percent of the < 2 mm particle size fraction had an NPR less than 2. Based on these results, the mine concluded that this waste rock had a low overall potential for ARD, with very little potential for impact.

⁹ The NPR was initially incorrectly reported as NPR of 3.1. This error occurred from averaging individual NPR values (it is incorrect to average ratios) instead of dividing the mean NP by the mean AP.

4.1.5 440 Portal Waste Dump

The 440 waste rock dump lies within the crown pillar and contained waste rock from these workings. Six samples were collected in 1998, with analysis conducted on the +25 mm, 25-2 mm and < 2 mm size fractions. The results indicated that this dump had the highest AP and lowest NPR values of the waste rock, with a 5th percentile, median and 95th percentile AP values of 73, 104 and 121 kg CaCO₃/t and NPR values of 1.1, 1.3 and 2.1, respectively (Table 7). The NP showed no relationship with particle size, while total-S was 50% higher in the < 2 mm fraction compared to the coarser size fractions. Based on the NPR results, the mine concluded that this dump had an uncertain potential for ARD.

4.1.6 150 (Twin West) Waste Rock Dump

The Twin West dump contains 33,000 t of waste from the Twin West underground development. Sampling was conducted from a 30-meter long trench, running in a north-south direction from behind the Twin West shop to the edge of the dump. The trench was dug to a depth of approximately 2 meters with each sample consisting of material taken from a cross-section taken starting about ½ meter below the surface and running down the trench wall, across the floor, and up the opposite side to within ½ meter of the surface.

Based on the results, the 150 dump had lower 5th percentile, median and mean AP (6, 16 and 23 kg CaCO₃/t) and higher NPR (1.7, 7.6 and 5.2) values than the dumps associated with the Main underground workings (Table 7). Like the 440 dump, NP showed no relationship to particle size, while total-S averaged 50% higher in the < 2 mm than the coarser size fractions. The low potential for ARD compared to the Main Underground was also noted in the sampled rock walls.

4.2 Metal Leaching Test Work

4.2.1 Elemental Composition of the Recently Blasted Material in Muck Piles

Data for the elemental composition of the waste rock was provided in NDM (1998). Similar to other wastes, only median concentrations of As and the 95th percentile values of Ag, Cd, Cu, Mo, Pb and Zn were an order of magnitude higher than typical background (Table 11). With the exception of Cu, average concentrations of the trace metals¹⁰ were lower in the waste rock than the tailings beach and the cycloned tailings sand.

Drainage from lab tests run on the waste rock (Tables 13 and 16) and dump seeps monitored during the life-of-the-mine (Table 20) had a neutral pH. Given the high NP, it was expected to be decades before ARD occurred and it might never occur. Contaminant levels in drainage from humidity cells and columns run with waste rock were lower than the two tailings materials. Contaminant concentrations in seeps sampled at the base of the 150, 300 (2 seeps), crown pillar and 440 waste dumps were well below the Provincial pollution control objectives and typically below provincial guidelines for aquatic life (Snip Mine, 1998). The two highest trace metal concentrations were 0.75 and 0.95 mg/L total Cu and Zn in the 440-dump seep. Dissolved mg/L values for these elements were 0.05 for Zn and 0.004 for Cu, indicating that the high values were due to entrained sediment.

¹⁰ Including metalloids

Even when 440 and 150 dump fines were leached with weak acid (pH 4.2; EPA 1312 Procedure), there were relatively low levels of trace metals. The highest values were 21.6 µg/L Cu and 18 µg/L As, both of which were from the leaching of 440 dump samples. In addition to indicating a low degree of weathering, the low metal concentrations were likely also due to the high neutralizing ability of the samples.

4.3 Conclusion and Subsequent Mitigation

Most of the dumps sampled at closure had comparable or higher NPR values than the waste rock samples in the pre-mining test work. The exception was the 440 dump, whose high AP and relatively low NPR values were similar to the pre-mine results for waste rock from the sulphidic ore zone.

As discussed in Section 3.2.2, the main difference between the ML/ARD assessments of the waste rock at closure versus that prior to mining was the use of the NPR instead of the NNP and use of an NPR < 2 for material with an uncertain but significant ARD potential. While very few of dump samples had an NPR < 1 indicating ARD was likely, most of the 440 dump, a significant portion of the 130 dump and several samples from the 300 dump had an NPR between 1 and 2. More extensive analysis of the fine particle size fraction and the potential contribution of non-neutralizing Fe and Mn carbonate or insignificantly reactive silicate minerals to the measured NP would have resulted in more samples with NPR values < 1 or < 2, but would probably not markedly alter the conclusion that the majority of the waste rock is not potentially ARD generating. The main concerns identified in the closure plan results were with 'hot spots' and the 130 and 440, the two lowest NPR, dumps.

Based on the potential for ARD in at least portions of the waste rock, four of the five dumps were moved to tailings impoundment (Van Zalingen, Jan. 5, 2001 Report of Reclamation Inspector). The 130 and 440, the two dumps judged to have the highest ARD potential, along with the mill waste, were placed in the depression on the west side towards the south end of the tailings impoundment. Notably, during excavation the mine found less waste rock in the 130 dump than expected. The other two dumps moved to the impoundment, the 150 and 180 Dump material, were spread in a thin layer over the surface of the tailings.

The 300, the largest waste rock dump, was left in place. The low trace metal concentrations and a 5th percentile NPR of 2.0 suggested that this material had little or no potential for significant metal leaching. Drainage from the 300 Dump reports to the tailings impoundment. Unless water quality in the impoundment changes significantly, natural attenuation and dilution will likely prevent impacts.

The 340 and crown pillar dumps, two dumps of non-PAG, natural overburden in the crown pillar area, were recontoured and revegetated insitu.

5. Tailings Impoundment

5.1 Design and Geotechnical Considerations

The impoundment is in a narrow valley, confined by natural hill slopes along its length and dammed by Dyke 1 at the Sky Creek end and Dyke 3 at the Monsoon Creek end (Figure 4). Sky Creek was

diverted to allow construction of Dyke 3. The area within the impoundment is 8.0 ha, with 1.7 ha of dykes. The dykes were constructed primarily from well-graded natural fill, using downstream construction. The fill was a minimum of 20% less than 75 μm , creating water-retaining structures. Waste rock was used in constructing the flooded foundations and upstream portions of both dykes. It was also used to construct a portion of the downstream slope of Dyke 3, a region of the dam that is not flooded.

Dyke 1 is approximately 175 m long and 10 m high. Dyke 3, at the Monsoon Creek end, is approximately 155 m long and 20 m high. Both dykes have a crest elevation of 150.4 m. The tailings surface slopes at an overall slope of approximately 0.15% from the 148.5 m at the higher Monsoon Creek end to 147.3 m at the spillway at the Sky Creek end. Since closure, a shallow pond has covered most of the tailings at the Sky Creek end of the impoundment.

At closure, the culvert installed through Dyke 1, the decant pipe across Dyke 3 and relief well outfalls at the toe of Dyke 3 were all completely backfilled with concrete and cement grout. A continuous cover of filter material was placed on the downstream faces of the dykes around these pipes to avoid piping. Flat benches were constructed with sand and gravel fill downstream of the two dykes to provide an overall slope of 8H:1V for Dyke 1 and 6H:1V for Dyke 3. Construction of the toe berms increased the long-term static and seismic stability, with Dyke 1 predicted to withstand 1 in 1000 earthquake conditions (Knight Piesold Consulting, 2004). Large angular armour stone or rip-rap was placed on the upstream face of the dykes for erosion protection. Large angular armour stone or rip-rap has also been integrated into the toe berm of Dyke 1 to minimize erosion along the Sky Creek diversion channel.

The spillway and embankment heights were designed to withstand the probable maximum flood. A recent review by Knight Piesold Consulting (2004), assuming a contributing basin of 1.15 km^2 , concluded that the impoundment would be capable of withstanding a rain-on-snow event with a 24-hour PMP of 448 mm, and snow melt at 15°C. The spillway was excavated in bedrock on the right abutment of Dyke 1 to minimize erosion. To minimize the potential for beaver dam construction in the spillway channel, the gently sloped portion of the channel was filled to a depth of 1.2 m with large angular armour stone (competent boulders) so it functions as a rock drain concealing the flow.

Regular inspections are made from Eskay Creek mine by helicopter to check that beaver activity within and downstream of the dam, and mass wasting from the steep surrounding hill slopes have not affected the design capacity of the impoundment. Follow-up inspections are planned following significant runoff and earthquake events in the region. Since closure, failures have occurred on the slopes above the impoundment, resulting in the mass wasting of soil and trees onto the impoundment. Potential concerns are that the debris might block the spillway or increase the velocity of drainage crossing the impoundment cover. Although there was a failure near the spillway, neither of these problems appears likely at present. Immediately after tailings deposition stopped, beaver footprints were spotted on the tailings. However, the rocks placed in the spillway have been effective in preventing beaver dam construction.

Materials placed within the impoundment include tailings disposal throughout the life of the mine and waste rock from portal dumps moved into the impoundment at closure (see Section 4). Drainage sources include runoff and groundwater from the surrounding slopes, some concentrated

flow in the Wolf Creek and Nichols/Upper Monsoon Creek channels, and drainage from the underground workings, primarily from the 300 portal, plus some seepage from the 180 portal. Owens Creek and Bear Creek, which drain a significant part of the slope on the east side of the impoundment, enter the valley downstream of the dykes.

5.2 Composition of Tailings

The majority of the tailings placed in the impoundment came from ore processed in the flotation circuit. Most of the flotation tailings were cycloned, with the majority of the sand fraction, the underflow component, pumped underground for use as hydraulic backfill, and only the slimes reporting to the impoundment. An additional source of tailings slimes in the impoundment came from cycloned tailings generated by grinding waste rock. When the cyclone was not operational, whole ore tailings (sand and slimes) were pumped to the tailings impoundment. According to surveys conducted at closure, the total quantity of tailings in the impoundment was 1,016,975 t or 677,983 m³. The breakdown consisted of:

- 773,408 t of slimes produced from cycloned ore tailings;
- 84,522 t (281,740 t x 0.3) of slimes produced from cycloned ground waste rock; and
- 150,000 t of whole tailings.

The mass of whole tailings was calculated by subtraction as no records were kept of the discharge volume. From January 1991 to June 1995, tailings disposal occurred from spigots along the east side of the valley and on the Dyke 1 at the Sky Creek end of the impoundment. Subsequently deposition occurred from Dyke 3. Tailings disposal ceased in June 1999.

5.2.1 Pre-Mining Characterization of Tailings

The geology of the ore was discussed in Section 2. Prior to mining, the composition of the tailings was predicted from eight ore samples collected from throughout the deposit, nine ore samples from the sulphidic zone, and five samples of tailings. Based on the NNP cut-off of 1, half the general ore, all but two of the sulphidic ore samples, but only one of the five tailings samples were predicted to be ARD generating. In the environmental impact assessment, concerns were raised regarding the high AP in the sulphidic ore and the uncertainty regarding how the ore would be processed and whether sulphides would preferentially report to the tailings sand or the slimes. Based on the test work, it was concluded that the majority of the tailings would have an NNP greater than 0 and would therefore be non-PAG. Proposed contingency measures in the event significant PAG tailings were produced, included flooding in the impoundment and milling carbonate ore at the end of mining to create a final non-PAG cover (Norecol Environmental Consultants Ltd., 1990). Humidity cell results were purported to show that limited exposure prior to flooding could occur without creating ARD (Section 3.4.1).

5.2.2 Operational Characterization of Whole Ore Tailings

Operational characterization of the mill tailings was conducted throughout the life of the mine. Samples were taken by an automatic sampler located at the end of the flotation cells, prior to waste rock addition and cyclone sand removal. The sampler took a cut every ½ hour over a 24-hour period. Sub-samples were placed in a plastic pail partially filled with water to keep the sample from oxidizing. Samples collected in this manner were composited over the period of one week. At the end of a week, the water from the bucket was decanted and the sample filtered through a Buchner

funnel and #3 filter paper. The sample was then dried, screened, mixed well and divided into three portions. The primary purpose in collecting these samples was to check the performance of the flotation circuit. Once a month one sample was sent for ABA analysis, one for ICP analysis and the third was stored for future reference. Towards the latter part of the mine life, ABA analysis was only conducted quarterly.

The ABA results shown in Table 7 and Figures 9, 10 and 11 indicate that the whole flotation (ore) tailings were high in both NP and AP. The 5th and 95th percentile NP values were 157 and 231 kg CaCO₃/t, while the 5th and 95th percentile AP values were 89 and 182 kg CaCO₃/t. The 5th and 95th percentile NPR values were 1.0 and 2.3, with a median of 1.4 (Table 7). These results suggested that ARD was likely (NPR < 1) for approximately 5% and uncertain (NPR 1-2) for more than 50% of the material (Figure 12). The main compositional changes observed during mining were a decrease in the higher AP values and a lack of samples with an NPR < 1, following exhaustion of the main ore body in 1996 (Figures 10 and 12). This was attributed to the mining of narrower veins with lower sulphide concentrations.

An obvious limitation with sampling and thus the data was that waste rock was added and the cyclone sand removed after the sampling point. Another limitation was that sampling prior to deposition would not indicate if there is significant particle size or mineral segregation in the impoundment, and its impact on the ARD potential of the resulting tailings. While the samples did not necessarily provide a very accurate measure of ABA levels in the tailings in the impoundment, in conjunction with the operational waste rock and cyclone sand analyses, these results did provide a very useful monitor of whether the ARD potential of the waste materials as a whole were changing.

5.2.3 Characterization of Tailings in the Impoundment

The surface of the tailings within the impoundment, which were mainly ore and waste rock tailings slimes, were sampled in 1993, 1994, 1995, 1997, 1998 and 1999 to check whether they were significantly different from the whole ore tailings. Descriptive statistics derived from the ABA results in 1993, 1994, 1995 and 1997 are provided in Table 7. The range in median values were NP values of 180 to 353 kg CaCO₃/t, AP values of 104 to 207 kg CaCO₃/t and NPR values of 1.2 to 2.0. The 5th percentile NPR varied from 0.8 to 1.6. The tailings surface in 1998 and 1999, the material close to the surface at the end of mining had NP values of 152 to 224 kg CaCO₃/t and AP values of 70 to 230 kg CaCO₃/t (Table 10). Due to the narrow range in NP, the NPR of the 1998 and 1999 samples depended largely on the AP. In 1998, 55% of the samples had an NPR between 1 and 2 and 25% had NPR values < 1. In 1999, all the samples had an NPR between 1 and 2.

The main differences between tailings samples taken in the impoundment compared with the monthly composite samples of ore tailings were that the impoundment tailings were more variable and a larger proportion of samples had an NPR < 1. The greater variability was likely due to the greater variability in source materials. The larger proportion of samples with an NPR < 1 may be attributable to the sulphide minerals being softer than the host rock and therefore grinding to a smaller size in the processing of the ore (Johnston, 1989).

It is important to note that ponding and a lack of trafficability limited the sampling of tailings in the impoundment to the tailings beach adjacent to the dykes. For example in 1998, two samples were

collected from the tailings beach off Dyke #1 and ten samples were collected from randomly selected locations on a 10 meter, numbered grid set up on the beach beside Dyke #3. Samples were taken from 0.5-meter deep holes dug by hand at each sample location. Various methods were used to expand the area of the impoundment that could be sampled. These included sampling during the winter when the surface was frozen and using snowshoes to prevent the sampler from sinking. Unfortunately, the heavy insulating snow cover usually prevented the Snip Mine tailings from freezing.

Results of the elemental analysis conducted on the 21 tailings beach samples collected in 1997 are shown in Table 11. Median values exceeded typical elevated crustal values for Ag, As, Cd, Mo and Zn. There was no Se and Sb analysis. With the exception of As, the exceedances of typical elevated crustal values were by approximately one order of magnitude.

As might be expected for fine sand and silt particles, a petrographic examination of a tailings composite sample suggested that pyrite and carbonate were fully liberated in the tailings and available for oxidation. During mining most of the tailings were flooded. The exception was the short-term exposure of surface material on the tailings beach adjacent to the disposal locations. However, even on the beach, the fine particle size of tailings slimes, water added with the tailings and the wet climate likely resulted in saturated conditions near the surface. Of the materials sampled in the impoundment, those with the longest exposure were those sampled in 1998 beside Dyke 1, where spigotting last occurred in July 1996. As expected from the high NP and precipitation, the Dyke 1 samples all had neutral pH values and little or no build up of sulphate-S, as measured by the difference between total and sulphide-S in these samples. The lack of iron discoloration suggesting that very little sulphide oxidation had occurred.

5.3 Waste Rock

5.3.1 Waste Rock Used in Dam Construction

Waste rock used in dam construction included the following (the mass was calculated assuming 2 t/m³).

- 1990-Dyke 1: An unknown quantity of rock fill placed at the base of the dam, including upstream to downstream in the bed of the former Sky Creek, near the right abutment
- 1997-Dyke 3 (Monsoon side), downstream shell: 5,200 t
- 1997-Dyke 1, upstream platform: 3,600 t
- 1998-Dyke 3, upstream platform: 2,300 t

The only ABA data found for waste rock used for dam construction were four samples selectively taken in 1990 from material containing visible pyrite (Table 8). Three of the four samples had Sob-NPR values of 2 or more. Material represented by the sample with a negative NNP (Sob-NPR of 0.7) was not predicted to be a concern because the pyrite-containing material was such a small portion of the waste rock as a whole that any acidic drainage it produced would be neutralized in-situ (Norecol Environmental Consultants Ltd., 1990). No other ABA or elemental data was found for waste rock used in dam construction.

Waste rock use in the tailings dams was not addressed in the closure plan. Concerns include the potential for seepage through the dam to preferentially dissolve calcite or to increase metal leaching. Presently, there is no monitoring of the dam seepage or the downstream environment; so elevated metal leaching from waste rock in the dams would go un-noticed.

5.3.2 Disposal of Surface Waste Rock in the Impoundment

After the mine closed and tailings deposition stopped, waste rock dumps considered to present the highest potential risk of ARD production were placed in a flooded depression near the south west (Sky Creek) end of the impoundment. The estimated mass assuming 2 t/m³ was 440 Dump - 12,000 t, 130 Dump - 28,000 t, and mill waste stockpile - 60,000 t. The waste rock in the dumps was moved in a truck and dozer operation. Waste rock from the 150 Dump (33,000 t) was used as a granular cover over the tailings at the south end of the impoundment. Waste rock from the 180 Dump (57,000 t) was used as a granular cover over the tailings in the middle and at the north end of impoundment. The depth of the waste rock cover ranged from 0.5 to more than 1.0 m thick, with the depth generally increasing from the higher and drier Monsoon Lake end to the wetter south end of the impoundment (Van Zalingen, 2001). Descriptions of the surface waste rock dumps are provided in Section 4.

5.4 Mitigation of Wastes within the Impoundment

5.4.1 Cover with Soil and Flood

In addition to the measures taken to improve dam stability, the closure plan for wastes within the impoundment included:

- covering with soil-like material to limit trace metal uptake by biota, and erosion and aeration of the underlying wastes;
- keeping the water table high enough to prevent significant oxidation and thereby prevent ARD and significant metal leaching from the tailings and waste rock; and
- minimizing the depth and area of ponding to limit wave action and thereby reduce dam and cover erosion.

After waste rock disposal was complete, the surface of the impoundment was covered with a 15 to 30 cm layer of soil-like material. The soil material used at the Monsoon Creek end of the impoundment was direct hauled from the material excavated in the expansion of the toe of the Dyke 3. The Sky Creek side of the impoundment was covered with subsurface materials excavated from the borrow areas below the impoundment. A similar truck and dozer operation was to move the soil as for the waste rock, with the covering starting at the higher, drier Monsoon Lake-side and moving towards the Sky Creek dyke. Constraints placed on this and other mine closure projects was to complete the work before the heavy fall rains reduced the trafficability and the camp was scheduled to close. Working on the tailings so recently after deposition was a challenge because of the large amount of ponding and the low strength of the recently deposited tailings. To reduce ponding during cover placement, a shallow sump was excavated at the Sky Creek end of the impoundment with ponded water pumped to Sky Creek. To improve the trafficability/strength, geotextile fabric was placed on top of the saturated tailings at the lower Sky Creek end of the impoundment. This strategy worked quite well, although the dozer did get stuck on several occasions following heavy rain.

As the cover was laid across the wetter Sky Creek-side of the impoundment, significant mounding and boiling of tailings occurred in front of the soil placement. This coupled with the onset of heavy rain, prevented a small, < 3,000 ft² area of tailings from being covered. The uncovered tailings are in a low-lying area of discontinuous ponding near the Sky Creek dam. ABA results from three surface samples taken in 2002 showed the exposed tailings had an AP of 69 to 115 kg CaCO₃/t, an NP of 135 to 173 kg CaCO₃/t and an NPR of 1.5 to 2.0. The latter suggest that ARD was possible but unlikely. The mine has committed to continue monitoring weathering and erosion in this area and if required to complete the soil cover.

The initial plan was to seed all disturbed areas of the site, including the impoundment, with agronomic grasses (Prime Resources Group, 1997 and Snip Mine, 1998). To avoid attracting bears, which would have been vulnerable to hunting by people arriving by jet boat, use of agronomics was not permitted on the impoundment. Due to the flat terrain and coarse nature of the cover soils, this has not resulted in any erosion of the soil cover. Since the mine closed, there has been significant invasion by native seedlings such as willow, cottonwood and red alder. Invasion has been higher on the soil direct hauled from the toe of the Dyke 3 (Monsoon Creek end), presumably because at the time of excavation it contained many plant propogules. Significant plant invasion has also occurred in low-lying areas adjacent to areas of ponding.

Questions regarding the closure plan for wastes in the impoundment included:

- whether metal uptake by vegetation growing in the area posed any risk to ecosystem health;
- whether tailings and waste rock at the higher end of the impoundment would be above the water table and what metal leaching may result;
- whether there would be concentrated lateral flow through waste rock spread over the surface and whether this would supply sufficient oxygen for significant sulphide oxidation or cause disproportional NP removal by dissolving calcite;
- whether the various concentrated drainage sources on the east side of the impoundment, Wolfe Creek, Upper Monsoon Creek and Nichol Creek, and discharge from the 300 portal, would erode the cover and remove tailings as they flowed across the impoundment to the spillway; and
- what would happen if there was an out burst from the flooded portion of the underground workings?

Armour stone or rip-rap was added to dissipate the initial energy as these flows enter the impoundment and a swale was created with the soil material to convey surface water around the area of un-covered tailings across the impoundment to the spillway. Observations to date indicate that the low gradient and ponding dissipate flow across the impoundment, preventing flow velocities that could potentially result in erosion.

Factors controlling the height of the water table in the impoundment include the slope, the relatively low conductivity and high water holding capacity of the tailings, the water balance and the final height of the spillway. The water balance calculated for the impoundment indicated there would be a positive average water balance in every month (Appendix C in 2001 Annual Reclamation Report). The predicted average annual inputs were 97 x 10³ m³ from the 180 portal, 210 x 10³ m³ from the

300 portal, $512 \times 10^3 \text{ m}^3$ from groundwater, $516 \times 10^3 \text{ m}^3$ of runoff from snow, and $737 \times 10^3 \text{ m}^3$ of runoff from rain. The predicted average annual groundwater seepage loss through and under the dykes was $246 \times 10^3 \text{ m}^3$, resulting in a net average annual surplus of $1,826 \times 10^3 \text{ m}^3$. This surplus would be removed as runoff through the spillway and evapotranspiration.

Since cover placement and pumping stopped, there has been shallow ponding over approximately one third of the impoundment, mainly at the Sky Creek end. Concerns with the concept of surface saturation but little or no surface water cover were:

- whether the water table would be high enough at the higher northeast end of the impoundment;
- whether the water table would be lowered during an extremely dry year; and
- whether there would be a potential for ARD production if tailings or waste rock are above the water table for significant periods of time.

The difference in height between the Sky Creek spillway and the tailings surface approximately 850 m away at Monsoon end of the impoundment was 1.2 m. The water balance calculations made prior to closure suggested that the potential net change in the height of the water table during an extremely dry year was 2.8 m (Snip, 1998). The mine has attempted to excavate water table monitoring pits by hand, but was thwarted by the compact, stony nature of the soil cover (2001 Annual Reclamation Report). To address concerns regarding the height of the water table at the Monsoon end of the impoundment and the impact of dry conditions, the mine needs to install piezometers and monitor the height of groundwater within the mine wastes and if necessary sample water quality. When the issue of whether the water table would be high enough at the higher northeast end of the impoundment was raised during closure planning, the mine proposed that if significant weathering was observed, a variety of measures including adding stop logs could be used to increase the height of the water table.

5.4.2 Chemistry of Drainage in the Impoundment

Results of monitoring during the last weeks of tailings disposal, during subsequent waste rock disposal and cover placement, and the initial input of discharge from the flooded workings are shown in Table 21. Biweekly monitoring of the water quality in the impoundment was conducted throughout active tailings disposal. During this period, the impoundment drainage typically had a pH of around 8.0, 100 to 200 mg/L sulphate and relatively low concentrations of dissolved trace metals (e.g., As < 40 µg/L, Zn < 12 µg/L and Cd < 0.2 µg/L). During waste rock disposal and placement of the cover over the tailings, there was a small increase in the concentrations of sulphate (up to 402 mg/L) and dissolved trace metals (e.g., As up to 60 µg/L, Cd up to 0.5 µg/L, Cu up to 4 µg/L and Zn up to 21 µg/L). The largest elemental increase during work in the impoundment was in total-Fe, a change attributable to sediment loss from the soil during placement of the cover. Since work on the cover finished, the median total-Fe concentration has been 0.09 mg/L and TSS only exceeded 7 mg/L in June 2003.

Starting in January 2000, the mine has monitored the impoundment discharge at the base of the spillway (Table 22). Despite relatively large concentrations of Mn and Zn in the initial 300 drainage, concentrations at the base of the spillway have remained well below the discharge limits. The impoundment was especially effective in lowering Zn, which was as high as 388 µg/L in the

300 discharge, but only 27 µg/L in the discharge from the impoundment. The highest concentrations of Mn and Zn (e.g., 0.47 mg/L Mn and 27 µg/L Zn) at the base of the spillway were at the same time as initially high concentrations in the 300 discharge. Subsequently, trace metal concentrations have remained low in the impoundment discharge (As < 2 µg/L, Sb < 1.9 µg/L, Se < 1.1 µg/L, and Zn < 16 µg/L), most at or near their detection limits, and almost entirely occurring in the dissolved form. Concentrations of alkalinity (61 to 152 mg/L), sulphate (67 to 240 mg/L), Ca (40 to 102 mg/L) and Mg (3.8 to 12 mg/L) in the impoundment have fluctuated seasonally with high concentrations observed in the summer and winter, but similar to trace metals have not showed any long-term trends. The pH has remained slightly alkaline (7.8 to 8.4). Unlike elements such as sulphate, Mn concentrations in the impoundment discharge are typically lowest during the summer. This coupled with the lack of correlation with decreasing Mn in the portal discharge from 2001 to 2003 suggests that the main source of Mn drainage was the seasonal flooding of weathered waste rock and soil cover in the impoundment.

5.4.3 Rate of Discharge from the Impoundment

The mine constructed a weir at the base of the spillway and started monitoring surface discharge from the impoundment in February 2001 (Table 27). Measured flows have been as high as 63.5 L/s and there was no flow during August and September in 2001. High flows have been measured during the months from September to May indicating the potential for high runoff during most of the year.

The predicted annual water balance (average annual inputs of $97 \times 10^3 \text{ m}^3$ from the 180 portal and $210 \times 10^3 \text{ m}^3$ from the 300 portal versus a net average annual surplus of $1,826 \times 10^3 \text{ m}^3$ removed as runoff through the spillway and evapotranspiration) suggested that runoff through the spillway should greatly exceed inputs from the underground workings. However, this has not often been the case and on a number of occasions measured flow through the spillway has been lower than the input from the 300 and 180 portals. One possible explanation could be the higher than predicted seepage losses. Prior to closure, monthly flow measurements taken at the weir located downstream of Dyke 1 annually averaged 0.9 to 1.1 L/s, with a high of 10.8 L/s (Klohn Crippen, 2001). Pre-closure flow measurements taken at the weir located downstream of Dyke 3 annually averaged 1.5 to 4.6 L/s, with a high of 7.5 L/s. The impoundment is underlain by layers of sand, silt and gravel (Snip, 1998), and it is possible that not all the impoundment seepage reported to the weirs. Another potentially contributing factor for the periods when measured drainage input exceeded the measured output is flow under the spillway weir. At the time it was constructed, there were problems finding good weir location. The present weir location is an area of fractured rock, with leaks visible (Ennis per com.).

6. Twin West Underground Mine Workings

In addition to mining the main underground workings, from 1995 to 1998, Snip also mined the Twin West Zone on the opposite, northwest side of the tailings impoundment, next to the spillway. The geology of the Twin West consisted of siltstone, greywacke and fragmental rock, sub-units of the feldspathic wacke unit. The exceptions were a minor interception of a short narrow andesite dyke and diorite tuff encountered along the adits but not within the inner workings.

Mining of the Twin West Zone was conducted in a similar manner to the main underground workings, except that cyclone tailings sand was not used as backfill. Portals were constructed at the 150 and 225 Level and the ore was reached by an inner ramp. Ore and waste rock were removed through the 150 Adit (Twin West Portal). The 150 Adit was an incline and was therefore expected to produce drainage, while the 225 Adit was not expected to produce drainage.

ML/ARD data for the Twin West workings consisted of results for wall rock and the 150 waste rock dump. These samples were collected at the same time similar work was done in the main underground workings. The results indicated the Twin West workings had little or no potential for ARD. Of the 20 wall rock samples, only one with a value of 1.9 had an NPR < 2. One of the waste rock samples had an AP of 77 kg/t and an NPR of 1.7 (Table 7). However overall, the waste rock had a lower AP and higher median and average NPR, 5.2 and 7.6 respectively, than any of the dumps associated with the main underground workings. Based on these results, the low rate of flow from the 150 Portal and the relatively low metal concentrations in the neutral pH drainage at the site, it was concluded that the Twin West underground workings were not a ML/ARD concern.

7. Main Underground Mine Workings (Twin Zone)

7.1 Layout

The main underground workings are in the hill slope on the south east side of the tailings impoundment (Figure 3 and 4). Access during mining was primarily through the 130 (Mill haulage) and 180 adits, which are at the bottom of the mine. Levels driven at 40 metre intervals from an ascending spiral ramp accessed the ore body. In addition to 130 and 180 levels, adits were driven to surface at the 300, 340, 400, 420, 440 and 520 Levels. The 130, 180 and 300 Level adits are inclines and potential discharge points. The 340, 400, 420, 440 and 520 Level adits are declines and unless a rock fall blocks drainage into the mine, the only seepage from these adits will be drainage from the immediate area of the portal. In addition to the roads used to access the 130, 180 and 300 portals, a now overgrown, old diamond drill access road constructed to the crown pillar, potentially provides access to the 340 and 440 portals.

The ore grade consistently averaged about 32 g/t Au (1 oz/ton) and only declined to 27 g/t in 1996. The cut-off was 12 g/t Au. The minimum mining width was 1.5 m and mining often occurred in narrow veins. Initially, about 55% of the ore was produced from mechanized and 45% from conventional cut-and-fill stopes. By 1996, a large portion of the Twin Zone had been extracted and an increasing proportion of the ore was derived from the fringe of the ore zone, with more reliance on cut-and-fill stopes. Recovery of the ore rich crown pillar began in the summer of 1995 and was completed in 1998. The surface in this area was re-contoured and seeded in October 1998. Some subsidence has already occurred and there is a potential for further collapse, increasing drainage inputs into the mine.

The various types of backfill used in the mine included cycloned sand, waste rock, waste rock plus cycloned sand, and cemented fill sill mats. A cross section of the mine showing the location of major faults, crown pillars, empty and backfilled workings, the type of backfill, and the year backfill was deposited was provided in Appendix II of Snip (1998). According to the cross section, the only large waste rock-only fill occurred in 1991 and was below the 300 Level.

7.2 Prediction of the Potential for Significant ML or ARD

7.2.1 Pre-Mining

Pre-mine ABA analysis results for the waste rock and the ore/tailings are shown in Tables 5 and 6, respectively. Of the general waste rock samples, 3 of 12 had an NPR between 1 and 2, the others had NPR values > 2. The median AP was higher for the sulphidic waste rock (49 versus 76 kg CaCO₃/t), and as a result 6 of 15 sulphidic waste rock samples had an NPR between 1 and 2, and one sample had an NPR < 1. Based on the NNP > 0 of all but 1 of the 27 waste rock samples and the assumption that any localized ARD would be neutralized by the surrounding overwhelmingly net acid consuming material, the conclusion of the company's consultants prior to mining was that the waste rock would not produce ARD.

Based on the NNP cutoff, half the general ore, all but two of the sulphidic ore samples, but only one of the five tailings samples were predicted to be ARD generating. Reasons for why the mine did not expect ARD from the backfilled tailings included (Johnston, 1989):

- the dilution of ore by waste rock due to the narrow irregular nature of the sulphide ore zone;
- mixing with acid consuming waste rock and the influence of acid consuming wall rock; and
- sulphide removal in the milling process.

Contingency plans outlined in the mine proposal in the event the mine backfill was potentially ARD generating included (Johnston, 1989):

- blending with acid consuming rock;
- adding a pyrite separation circuit to the mill, and deposition of the resulting pyrite concentrate in the flooded impoundment; and
- adjusting the process from cyanidation to gravity flotation.

As discussed previously, prior to construction the mine decided to change the process from cyanidation to gravity flotation.

7.2.2 Mine Walls

One of the first steps taken in closure planning was to characterize the ML/ARD potential of the wall rock. Characterization primarily consisted of sampling rock chips from the main lithological units that host the ore; the mafic dyke (BSU) and quartz-monzonite stock (RB or Red Bluff Porphyry), and the three sub-units within the turbiditic feldspathic greywackes, the siltstone, fragmental and greywacke. Sampling also focused on the 340 Level, the area of the crown pillar region, which in addition to being a region of sulphidic ore, through subsidence was likely to be an area of the underground workings with relatively high drainage and air entry, conditions which should increase weathering. Sampling locations were shown on the geological level plans provided in Appendix C of Snip (1998). The number of samples taken from different rock types was as follows: Greywacke (20), Fragmental (10), Siltstone (10), BSU (3), RPB (3), Crown Pillar Footwall (3) and Crown Pillar Hanging Wall (2).

Many of mine workings were backfilled, so it was not possible to get representative wall samples from all the ore zones. Where access was no longer possible, samples of the ore, footwall and

hanging wall were taken from pre-existing drill cores that penetrated these zones. The core samples were taken from intersections that represented typical ore mineralogy and covered the orebody both laterally and vertically. A 5 kg sample was taken, beginning 0.5 m from the ore/wallrock contact and moving out from the ore until the desired sample size was attained. In some places, samples had to be taken from several drill core from the same general area and composited. The number of drill core sample collected of the various ore zones, from the footwall and hanging wall respectively, were as follows: Lens (2 and 2), Twin Zone (3 and 3), 150 Vein (2 and 2), 130 Vein (1 and 1) and Twin West (2 and 2).

Descriptive ABA statistics for the rock wall sample results shown in Table 7 indicate that the composition of wall rock in the crown pillar area was very similar to the underground workings as a whole. Median values of AP (48 and 39 kg CaCO₃/t) and NP (103 and 109 kg CaCO₃/t) for the mine walls were slightly lower than the waste rock and much lower than the tailings sand (backfill) and whole tailings. Ninety-fifth percentile values for AP (109 and 88 kg CaCO₃/t) and NP (147 and 144 kg CaCO₃/t) were also lower than the waste rock. However, the 5th percentile NPR value of 0.7 for the whole workings was lower than the waste rock and indicated that ARD was likely from a proportion of the wall material.

In the breakdown for individual rock types, all of the Red Bluff, 44% of the greywacke, 50% of the fragmental, and 20% of the siltstone samples had NPR values < 2 (Snip, 1988). The estimated contribution of each rock type in the mine was 0.3% Red Bluff, 77% greywacke, 14% fragmental, and 8.5% of siltstone. Based on these percentages, 43% of the mine workings were predicted to have an NPR < 2 indicating that ARD was possible. However, only 5 of the 67 samples had an NPR value less than 1, indicating that ARD was likely. For example, while all three Red Bluff samples had NPR values less than 2, only one was less than 1. During closure planning, it was noted that there was no measurement of the area of each rock type exposed in the workings and therefore actual proportions may vary. No attempt was made to identify the areas of the workings with the highest ML/ARD potential because mine personnel concluded that there were no spatial relationships evident. Notably two rock chip samples were limestone, but no information was provided on the extent and location of this lithology.

The main ML/ARD concern with mine walls is typically with the talus produced from the inevitable collapse of workings into voids. Unless there are large-scale failures, talus and mine walls are likely to be a relatively insignificant metal leaching sources compared to backfill. Backfill will reduce wall rock failure in most highly mineralized areas of the mine. Friction between waste rock particles and the layout of the stopes should limit the collapse of the backfill (Doug Flynn, pers. communication). The collapse of backfill is most likely to occur where seepage or ponding saturate pure tailings sand. Even in these instances, backfill rather than subsequent talus is likely to be the primary drainage chemistry concern due to its lower NPR, higher AP and much higher surface area.

7.2.3 Backfilled Waste Rock

The majority of the waste rock produced by the mine was used as underground backfill; either directly as rock fill (344,648 t) or transported to the mill and used to create additional cycloned tailings sand (281,740 t). The main source of information on the composition of the waste rock came from monthly analysis of a composite sample collected from the muck piles created from each waste round. Preparation of the samples included drying in the low grade drying oven and then

crushing and pulverizing. Prior to 1997, a flat tablespoon was cut from each muck pile sample and placed in a monthly composite jar. Each month, the composite was mixed and sub-samples were sent for ABA analysis and stored for future reference. In 1997, it was noted that compositing sub-samples from different muck piles might mask variability, which could be an important determinant of drainage chemistry. Subsequently, each month an individual muck pile selected was sampled for ABA analysis.

Plots of the operational ABA results and descriptive statistics for the monthly waste rock samples are shown in Figures 5 to 8 and Table 7, respectively. The 5th and 95th percentile analysis results were: total-S of 0.31 and 5.61%, AP of 10 and 126 kg CaCO₃/t, NP of 68 and 215 kg CaCO₃/t, and NPR of 1.2 and 15.0. ABA results for the post-blast monitoring of waste muck piles were similar to the results for samples taken from the surface dumps and indicate that most of the waste rock used as backfill was not potentially ARD generating. Approximately 20% of the samples had NPR values less than 2, but none had an NPR value less than one (Figure 5).

While the data was highly variable, it was notable that almost all the waste rock samples with NPR values less than 2 were collected prior to 1995 (Figure 8). After 1995, there was a reduction in AP (Figure 6). These changes probably reflect a change in the type of ore mined (see earlier description of underground workings). Compositing did not appear to reduce the variability in ABA results, as the AP of post-1997 discrete samples had a similar range to composite samples taken prior to June 1997.

Median values of waste rock As and the 95th percentile values of Ag, Cd, Cu, Mo, Pb and Zn were an order of magnitude higher than typical elevated crustal values (Table 11).

7.2.4 Backfilled Cycloned Tailings Sand

As described previously, hydraulically backfilled cycloned sand was produced from tailings generated from the ore and to a lesser degree waste rock; 476,759 t in total with 269,741 t produced from ore tailings and 197,218 t produced from waste rock. The backfill was poured in vertical lifts ranging from a few centimeters to a couple of meters thick.

The samples used in operational characterization were collected at the Geho Pump in the mill, after cycloning and prior to the sand fraction being pumped underground for use as backfill. A grab sample was collected every two hours when backfill was being transferred underground. Like the whole tailings, the cycloned sand sample was initially stored in a plastic pail partially filled with water. At the end of each week, the water in the bucket was decanted and the sampled material was filtered through a Buchner funnel and #3 filter paper. The sample was then dried, screened, mixed and divided into three portions; with one portion sent for ABA analysis, one sent for ICP analysis and the third stored for future reference.

Plots of the operational ABA results and descriptive statistics are shown in Figures 13 to 16 and Table 7, respectively. The 5th and 95th percentile analysis results were total-S of 2.8 and 7.1%, AP of 89 and 222 kg CaCO₃/t, NP of 148 and 230 kg CaCO₃/t and NPR values of 0.8 and 2.3. Most of the samples had NPR values between 1 and 2 (Figures 13 and 16). Less than 20% had NPR values lower than 1, and none were lower than 0.75. Like the waste rock, throughout the mine life, %S,

NP and the NPR in the cycloned sand were relatively consistent, and did not indicate changes due to the increasing proportion of waste rock compared to ore later in the mine life.

Median values of Cd, Mo, Pb and Zn in the tailings sand were five to ten times higher than typical elevated crustal values, Ag and Cu were twenty to thirty times higher and As was two hundred and fifty times higher. Concentrations of Se and Sb, two of the more leachable trace elements according to the humidity cell work, were not measured.

7.2.5 Pre-Closure Assessment of the ARD Potential of the Backfill

The majority of the mineralized surface area and thus the most likely source of ARD in the underground workings was the backfill: 344,648 t waste rock; 466,959 t cycloned sand; and 9,800 t backfilled cycloned tailings sand combined with 1,306 t Portland cement. Compared to waste rock, tailings sand had a higher portion of lower NPR material, a higher fines content and a larger mass. According to the operational ABA monitoring results, 10 to 20% of the tailings sand had an NPR less than 1 and was therefore considered to be a likely source of ARD. Seventy percent of the cycloned tailings sand and 20% of the waste rock had an NPR of 1 to 2, and ARD production was considered uncertain. Ten percent of the tailings sand and 80% of the waste rock had an NPR > 2 and was categorized as net acid consuming. Assuming only 25% of the waste rock occurred as reactive fines and 75% was coarse fragments that are relatively inert, the mass of backfill material in the various NPR categories was as follows:

NPR < 1 / ARD likely: 93,000 t tailings sand;

NPR 1 to 2 / ARD uncertain: 326,000 t tailings sand and 17,000 t waste rock fines; and

NPR > 2 / Net ARD neutralizing: 47,000 t tailings sand and 69,000 t waste rock fines.

Trace metal concentrations and the pH of the drainage in the underground workings, and its impact on the chemistry of the eventual underground discharge will depend on a number of factors, whose influence was impossible to ascertain from the available information. They included the following.

- Discrepancies between the measured and real values of NP and AP may increase or decrease the proportions of backfill material with an NPR < 1 and between 1 and 2 (Section 3.2).
- The hydrogeology of the workings could have impacted the relative contribution of acidic and neutral pH wastes to the final discharge. The likelihood of ARD discharge from the workings will increase if leaching is preferentially in the lower NPR materials or if there is significant collapse of PAG mine walls. Conversely, the potential for ARD would decrease if leaching occurs primarily in the higher NPR backfill.
- Acidity and alkalinity inputs in groundwater may be significant. There are high carbonate levels in the host rock surrounding the deposit and relatively high alkalinity concentrations were measured in groundwater at the neighbouring Johnny Mountain Mine (Price, 2004). Potential acidity inputs include precipitation and soil runoff entering the workings in the Crown Pillar area.
- Precipitation of Fe and Al released from the weathering of minerals such as pyrite and ankerite may cover materials downstream, limiting leaching, sulphide oxidation and acid neutralization.

With the information available at closure, it was impossible to predict with any accuracy whether the unflooded underground workings would eventually produce ARD. Potential alkalinity inputs in groundwater, the relatively low trace metal concentrations and PAG NPR values that are relatively high (0.7 to 1.0), coupled with the large volumes of net neutral backfill, suggest that if ARD does occur, it would have relatively low acidity. However, the pH 2.6 result in the NAG test (Table 15) suggested low pH drainage is possible.

7.3 Overall Mitigation Plan

The mine evaluated a number of closure options for the Main Underground Workings, eventually deciding on the use of concrete bulkheads to flood workings below the 300 Level and cemented backfill in the upper adits to prevent human access and reduce oxygen entry into the upper workings. Influential factors in this decision included:

- health and safety concerns;
- the factors that made it impossible to flood the underground mine above the 300 Level; the near surface underground mining, removal of the crown pillar and the fractured nature of the rock; and
- the desire to direct drainage from the underground mine to the tailings impoundment.

Blocking the 130 Adit and having the underground drainage report to the tailings impoundment created a single offsite discharge point for runoff from the two main sources of mineralized waste and allowed the impoundment to be used if storage or dilution of the underground drainage were required. It also increased the positive water balance of the impoundment.

7.4 Flooding the Lower Workings

The mine estimated that raising the water table to the 300 Level would flood approximately 20 to 30% of the backfill. During development of the plan to flood the lower workings, the following concerns were raised.

- The water table would rise and fall because of leaks through fractures and drill holes, resulting in repeated oxidation and leaching of backfill, increasing overall metal loadings.
- Flooding the lower workings might result in a surface discharge through fractures, drill holes or interconnected fractures and drill holes, resulting in direct discharge into either Monsoon or Sky Creek, or increase the number of required monitoring locations and make it difficult to measure total discharge.
- Flooding increases water pressure in the confining rock and this might eventually cause rock surrounding the bulkheads, drill holes and faults to fail, resulting in a large outflow event.
- Raising the water table could, at least initially, increase contaminant loadings by removing solubility constraints, increasing the extent and intensity of leaching, and causing the collapse and flow of backfilled sand.
- Flooding might cause backfilled tailings sand to collapse and become more spread out or preferentially dissolve carbonate minerals, increasing the potential for significant ML/ARD if for some reason the lower workings were no longer flooded.

To address concerns regarding fluctuations in the water table and the dissolution of weathering products, as part of closure planning, Snip attempted to predict the water balance, the rate of filling

and the chemistry of discharge from the flooded workings. The proposed contingency plan if flooding resulted in unacceptable discharge through fractures and drill holes or if significant water table fluctuations accelerating carbonate dissolution and metal leaching was to drill large holes through the bulkheads, allowing the lower workings to be freely drained. A hole was drilled through the bulkhead in the lower portal at the Baker Mine in northeast British Columbia, when the bulkhead resulted in significant fluctuations in the height of the water table.

Because the underground workings were in the hill above, Snip expected any leaks through fractures to report to the tailings impoundment and therefore the potential for directly discharge into either Monsoon or Sky Creek was considered low. Leaks through fractures or drill holes were thought by Snip to be easily detected by inspecting the road along the toe of the hill.

7.4.1 The Predicted Water Balance and Rate of Flooding

The first step in evaluating whether the mine would flood was an attempt to predict drainage inputs. The two drainage inputs to the lower workings were direct groundwater input and discharge from the upper workings. The layout of the 300 Level ensures that runoff from the workings above the 300 Level would all report to shafts, ramps or stopes connected to the lower workings. Consequently, drainage out the 300 Portal would only occur when the lower workings were completely flooded and the height of the water table reached the 300 Level.

Prior to mine closure, drainage input was calculated from the volume of drainage pumped out the mine (flow through the 180 Weir) minus the amount of water pumped into the mine with the backfilled tailings (Table 25). The estimate of the volume of water added with the tailings was based on mill records, assuming a 50% solids density by weight. When the mine operated, all the drainage was collected in sumps at the bottom of the mine and then pumped to the 180 Portal so it would report to the tailings impoundment. Only data for 1997 and 1998 were used in the calculations because earlier data was incomplete and the size and shape of the workings in the last two years were closest to the final configuration of the workings. Based on the above, the expected drainage input was 31,000 m³/month and would range from 11,299 to 53,553 m³/month.

The rate at which the lower workings would flood was calculated from their predicted void volume, the expected average drainage input and an estimate of the leakage rate. The predicted void space in the workings below the 300 Level was 149,193 m³. This calculation assumed void ratios of 0.5 for the cycloned sand, 0.4 for the cycloned sand/waste rock and 0.3 for the waste rock. Based on the predicted void space, average input and leakage as the workings filled, prior to mine closure it was predicted that it would take more than 6 months for the lower workings to flood (Table 26).

After mining finished in June 1999, weirs were set up to measure flow at various underground locations (Colbourne, August 1999). During July and the first half of August 1999, discharge from the workings above the 300 Level ranged from 24 to 53 m³/hr (17,000 to 38,000 m³/month). Groundwater input below the 300 level, measured twice in August, was 48 m³/hr (34,560 m³/month). The combined inflow to the upper and lower sections of the mine on those two dates in August was 88 m³/hr or 63,000 m³/month, double the previously estimated average monthly input rate. One possible reason for the difference was that the post-mining estimate of the average monthly input rate included the residual process water added prior to mine closure.

While, it is possible to measure drainage inputs, without flooding the workings it was impossible to measure drainage losses through fractures or drill holes, most of which were inaccessible by drainage flowing almost entirely along the adit floor. As previously noted, the Snip mine was heavily faulted. It was also extensively diamond drilled. Prior to bulkhead construction, the mine conducted a detailed review of past drilling and all drill holes identified as potential leaks were plugged. However according to Tom Colbourne, the mine manager at closure, “..even though we have sealed all the holes between 130 and 300 levels which intersect surface, there may still be pathways between holes through faults. Since many of these faults have surface expression, there is the possibility of leakage (Colbourne, Aug. 1999)”.

7.4.2 Bulkhead Construction and Resulting Flooding

The bulkheads were 1.63 m thick, located in an area of stable, relatively unfaulted rock and constructed from re-enforced concrete. The 130 Bulkhead was located approximately 100 m up the drift from the portal. The 180 Bulkhead was located 300 m up the drift. In addition to a pressure gauge to measure the height of water within the flooded workings, the 130 Bulkhead included a sampling port. Rock around each bulkhead was grouted prior to and after construction to reduce its permeability. To address concerns regarding leaks, the mine constructed weirs and monitors flow and water quality at the 130, 180 and 300 Portals, and conducts regular inspections of former drill pads and the roads at the toe of the slope where seepage from faults would presumably emerge at the surface. The 130 Bulkhead was completed on August 20th, 1999 while the 180 Bulkhead was completed on September 18th, 1999. Flooding had already reached the 300 Portal when drainage monitoring re-started on the 19th of January 2000, five months after the 130 Bulkhead was completed. Possible reasons for more rapid filling of the lower underground than the prediction of 6.45 months included residual process water and that three of the months were October, November and December, months with higher than average monthly flow.

A pressure gauge was installed in the 130 Bulkhead to monitor the height of the water table and determine whether there are significant fluctuations. The monitoring results from the pressure gauge are reported in Table 27. The initial gauge reading in the 130 Bulkhead was 230 psi indicating that the water table was 301.7 m. In February and March 2000, the pressure decreased to 220 psi indicating the water table had decreased to 290.5 m. Pressure increased to 229 to 230 psi from May to July, but decreased to 225 psi (296.1 m) in August 2000. During subsequent monthly monitoring to May 2001, twice more in 2001 and in June 2002, the pressure has remained at 230 psi. The pressure readings below 225 psi in February and March 2000 corresponded with the only two flow measurements of less than 0.5 L/s at the 300 weir. The March 2000 sample from the 300 Portal also had the lowest recorded sulphate concentration (57 mg/L), suggesting that the majority of the discharge was near-surface groundwater intercepted near the portal rather than drainage from the mine workings. A possible explanation for the depressed water table in February and March 2000, immediately after the water table first reached the 300 Level was that the low groundwater inputs at that time of year were unable to keep up with delayed infiltration of flood water into backfill within the flooded workings.

In the four years since the lower workings were flooded, Snip has not observed seepage crossing the roads that run across the bottom of the slope adjacent to the underground workings. The significant leakage reported from the flooded workings has been the seepage around the bulkheads into the 130 and 180 adits. It was possible however that flow from the flooded workings was reporting, via

fractures, to one or more of the small streams that drain the hill slope. Monitoring levels of sulphate and other signatures of the mine water in these creeks could be used to check this, with monitoring above the workings used as a control.

7.4.3 Post-Flooding Rate of Discharge from 130, 180 and 300 Portals

Post-closure monitoring of the rate of discharge started in January 2000 for the 180 and 300 Portals and February 2001 for the 130 Portal. The results are shown in Table 27. Discharge from the 300 Portal consists of the combined flow from the unflooded upper workings and overflow from the lower workings. During inspections of the 130 and 180 Bulkheads in 2000 and 2001, almost all the discharge from the 130 and 180 Portals came from two sources, groundwater entering the adits 5 to 20 m downstream of the bulkhead and seepage entering the adits within 20 m of the portal. The groundwater entering the adits close to the bulkheads occurred primarily in the area of visible fractures and was accompanied by significant white and reddish brown staining and assumed to be coming from the flooded workings. Drainage entering the adit close to the portal was attributed to near-surface groundwater. Peak flow at all three portal locations typically occurs during the spring and fall, the periods of highest runoff at the site. During the periods of lowest flow, which typically occur during the winter (January to April) and the middle of the summer (August), discharge from the 130 Portal, and occasionally the 180 Portal, exceeded discharge from the 300 Portal.

The largest fluctuations in flow occurred in the discharge from the 300 Portal, where monthly flow measurements varied between 0 and 27.4 L/s. Average discharge from the 300 Portal from 2001 to 2003 was 8.4 L/s (21,334 m³/month). Initial flow from the 180 Portal was 2.0 to 3.6 L/s. Since grout was added to the rock around the bulkheads in August 2000, 180 Portal flow values have been < 2.3 L/s, with an average of 1.1 L/s (2,904 m³/month). Measured flow from the 130 Portal has been 2 to 6.4 L/s, and averaged 3.5 L/s (8,975 m³/month). The combined average discharge from the mine workings (flow from the 130, 180 and 300 adits) was 33,213 m³/month. This was quite similar to 31,000 m³, the pre-flooding prediction of groundwater inputs into the mine (Table 25). Assuming 50% of the 130 Portal and the 180 Portal discharge comes from near-surface seepage, the combined flow from the internal mine workings (flow from the unflooded upper workings, overflow from the lower workings and seepage around the bulkheads) was 27,274 m³/month. This was still quite similar to 31,000 m³. Monitoring of rate of flow half-way between the portals and the bulkheads would allow a more accurate estimate of the contribution of underground leakage compared to near-surface seepage drainage to the 130 and 180 Portal discharges.

7.5 Closure of the Upper Unflooded Workings

Assuming backfill was evenly distributed in the upper and lower workings, approximately 70% of the mass of material in each NPR category was in the unflooded workings above the 300 Level. Flooding above the 300 Level was not feasible because of the near surface underground mining, the removal of the crown pillar and the fractured nature of the rock, a phenomenon attributed to frequent faulting. The plan at closure for the upper unflooded portion of the mine was to block the adits to:

1. restrict access; and
2. minimise air (oxygen) entry.

The upper adits (340, 400, 420, 440 and 520 Levels), all of which are declines, were blocked by jamming cemented rock fill (waste rock) into the portals. For some portals, the surface of the fill was shotcreted. This work was carried out in 1999.

7.5.1 Mitigation Structures Constructed on the 300 Level

In addition to restricting human access and acting as a barrier to air entry, the closure requirements for the 300 Portal included discharge of drainage exceeding that required to flood the lower workings. The proposed manner for permitting drainage to leave the mine without allowing air entry was with a P-drain. The first phase in constructing the P-drain was to raise the height of two short sections of adit several metres in from the portal. Mounds of waste rock plus 20% cement, which were higher than the normal height of the adit, were placed in each of the two raised sections of adit. The objective of the mounds was that they would perform like rock-fill dams, maintaining a pond height between them and immediately upslope that reaches the roof of the adit, thereby creating a water barrier to air movement into the workings, with excess water flowing over top (Colbourne, Aug. 1999). According to Colbourne (Aug. 1999), the 300 Level had a 2% grade and if the P-trap performed as planned, the ponded water would extend 180 metres upstream of the P-drain rock-fill dams.

The waste rock mounds in the P-drain were not designed to impound large amounts of drainage. It was therefore very important that there be no blockage that would prevent water from flowing over the top, leading to a build-up of water pressure and a high discharge blow-out event. The flooded ramp, ore pass and stopes connected to the lower workings were expected to act as debris traps and the depth of flow in the 300 Level was expected to be too shallow to entrain any large debris brought to the surface by water flooding the lower workings. The main concern was therefore with entrained sediment and debris flows, resulting from surface erosion or the sudden collapse of sand backfill in the upper workings, and with small floating debris, colloids and other chemical precipitates. Chemical precipitation may result from exposure of drainage from the flooded lower workings to more oxygenated conditions or chemical changes when drainage from different parts of the workings mix.

Two ten metre sections of rip-rap were placed upstream of the P-drain, as a precautionary measure to slow any future flood wave caused by ground fall or a sudden release of dammed up water in the mine. These dissipaters upslope of the P-drain would also prevent sediment from the upper workings (e.g., saturated tailings sand) from reaching the P-drain. Review of the plan at closure included discussion of whether inspection and maintenance were required to prevent eventual plugging of the dissipaters, subsequent overtopping and subsequent plugging of the P-drain that may result from entrained sediment or chemical precipitates. According to George Headley, (MEM Geotechnical Reviewer), “the potential for transport of sand fill and timber debris was limited. Clay and silt size materials could be transported during high flow periods but there was 1,200 cubic m³ storage capacity upstream of the “P trap”. Water flows would maintain channels even if material filled to the drift back and pressure would not build up appreciably. A sudden outrush of material (backfill and bulkheads / ramp timbers) requires fill saturation, a loose (dilatant) fill structure and a triggering event such as an earthquake. These fills are reasonably dense due to blasting. They are well drained due to drainage through hanging wall fracturing and the crown pillars at the end of most stopes.”

When first constructed the mounds leaked and the height of the ponds between them and into the mine workings was well below the roof of the adit. In 2002, the water ponded between and behind the P-trap was pumped down and bentonite was spread on the surface of the rock-fill mounds in an effort to make them watertight. Despite this work, conductivity through the rock-fill remained high enough that the ponding never reached the roof of the adit or the top of the mounds.

7.5.2 Inspection of the 300 Level in 2002

The only inspection, behind the partially constructed P-trap since the lower workings were allowed to fill, occurred in 2002 (Snip Mine, 2002). The inspection concluded that the discharge from the upper mine workings came down the ore pass and ramp, and as planned all reported to the flooded lower workings. However, instead of overflowing over the lip of the 300 Adit, drainage from the flooded workings reported to the 300 Adit via diamond drill holes in exploration cut-outs at 297.5 m asl, further down the adit. The drill holes were inclined down and thus intersected the flooded workings at some unknown distance below 297 m elevation (Graham Ennis, pers. communication).

7.5.3 Reducing Air Entry into the Unflooded Workings

One of the unknowns in the closure plan was the effectiveness of physical barriers in the upper portals in reducing air entry and oxygen levels in the underground workings, and oxidation within the backfill. Factors identified as potentially contributing to air entry into the mine workings and subsequent movement through mine wastes included:

- the effectiveness of the rock-fill adit plugs and the P-trap;
- there is further subsidence in the crown pillar area;
- changes in drainage volume and differences in atmospheric pressure and air temperature (e.g., sulphide oxidation is an exothermic process);
- the high permeability of the waste rock and the moderate to high permeability of sand backfill;
- shrinkage of the backfill after emplacement;
- relaxation fractures in stopes;
- mine workings near the surface;
- subsidence in the areas where the crown pillar was removed; and
- the numerous faults and drill holes.

Air entry and atmospheric conditions may change over time; especially if the workings collapse. At closure, the mine was unable to provide evidence from other sites of the effectiveness of rock-fill adit plugs in preventing air entry and causing anoxic conditions to develop in mines similar to Snip. In the absence of contrary evidence, BC MEM have assumed that oxidation would continue to occur and the backfill in the unflooded workings would remain a potential source of elevated neutral pH metal levels and/or future ARD.

The only post-flooding monitoring of air quality within the mine were measurements of % oxygen taken near the 130 and 180 Bulkheads when drainage sampling was occurring in 2000 to 2001. The percent oxygen varied from 19.4 to 20.7% at the 130 Bulkhead and from 19.7 to 20.9% at the 180 Bulkhead (Snip, 2002), indicating very little depletion of oxygen.

A comprehensive long-term program of monitoring would be required to demonstrate post-closure atmospheric conditions and whether oxygen entry into the backfill had declined to levels that significantly reduce sulphide oxidation. Due to the complexity of the mine and the potential for changes, the monitoring would have to be temporarily and spatially comprehensive. For example, atmospheric conditions might be very different in backfill within more isolated stopes than in the porous crown pillar area of the upper workings. One benefit of having spatially comprehensive atmospheric monitoring (monitoring on a number of other levels and types of workings) would be if it assisted the mine in detecting and reducing air entry and minimizing internal air movement. Seasonal monitoring is important because air entry and oxygen levels may fluctuate widely, with oxygen depletion potentially resulting from rapid sulphide oxidation.

Limited access and safety concerns once active mining stop makes monitoring within closed underground mines difficult. One monitoring option discussed during closure was having a system of automatic underground monitoring stations. Another was having an air seal double door system in one of the upper portals permitting monitoring personnel to access the upper mine workings. Access doors were rejected because of concerns about unauthorized access and safety. Difficulties envisioned in the use of remote monitoring equipment included how to access the equipment to make repairs and retrieve the data. One potential solution to these concerns would be to lower monitoring equipment through surface drill holes into the mine workings.

An important additional consideration regarding mitigation measures to reduce air entry and oxygen levels is the uncertainty regarding what constitutes sufficiently low oxygen levels or rates of sulphide oxidation.

7.6 Chemistry of the Discharge from the Partially Flooded Underground Workings

The chemistry of drainage during the operation of the mine likely was strongly influenced by the high levels of alkalinity from mineral abrasion and lime added in processing and added with the hydraulic backfill. During mining, an estimated 20% of the underground drainage was mill process water pumped in as part of the hydraulic backfill. Post-mining the contribution of process water and potentially soluble components, such as precipitated cement and mineral salts, added with the hydraulic backfill would decline, while the influence of weathering reactions within the backfilled wastes and groundwater inputs should increase.

Prior to closure, three potential events were identified as cause for concern with regards to discharge chemistry.

1. Dissolution of accumulated solutes when wastes in lower workings were flooded.
2. Elevated trace metal discharge in neutral pH drainage as a result of the increased weathering, leaching and trace metal solubility. Potentially contributing factors include the cumulative build-up of weathering products, decreasing pH or solubility constraints as a result of the depletion of cations and alkalinity added in process water, increased surface area and leaching as a result of the physical collapse of mine walls or backfilled tailings sand, and increased drainage inputs as a result of surface subsidence.

3. Acidic weathering and drainage from PAG wastes in the upper unflooded portion of the workings. Due to the high NP in the backfill, the onset of acidic weathering conditions was not predicted in the short-term. Even if drainage from PAG wastes was neutralized underground, elevated levels of Zn and other metals released by acidic weathering would remain in the drainage, causing an increase in trace metal loadings to the environment.

7.6.1 Initial Assessment of Underground Drainage Chemistry

Data initially used to assess underground drainage chemistry included the elemental composition of the backfill materials, results from various kinetic test work, drainage monitoring results for waste rock and tailings in other locations, and results from directly monitoring drainage in the mine. As discussed previously, the tailings sand and waste rock contained slightly elevated concentrations of a number of trace elements that could have been problematic at neutral pH values (e.g., As, Cd, Mn, Mo, Zn and possibly Ag – Table 11). Of the two types of backfill, trace element concentrations were typically higher in the tailings sand than the waste rock. The highest 95th percentile values were As at 1497 mg/kg, Mn at 1921 mg/kg and Zn at 1016 mg/kg.

All the kinetic test work produced neutral pH drainage. Trace metal concentrations were typically less than 10 µg/L in the short-term column study run in 1995 (Table 13). The most notable exception was the 2 to 10 mg/L Mn in drainage from a column of weathered tailings sand. The next highest trace metal concentration were the 20 to 80 µg/L Zn in drainage from a column whole tailings. The highest drainage trace metal concentrations in the EPA 1312 test were 25 to 44 µg/L Mn and 5 to 11 µg/L Ni from the unweathered tailings sand, and 17 to 22 µg/L Cu and 10 to 18 µg/L As from weathered samples of the 440 waste rock dump (Table 14). In the modified SWEP test, the concentration of Mn was 90 to 190 µg/L (Table 15). The only other trace metal exceeding the detection limit was Mo, which was 10 to 11 µg/L in drainage from unweathered whole tailings.

In the long-term humidity cells test work started in 1998, average weekly trace metal concentrations in the drainage from the tailings sand and the waste rock for weeks 10 to 30 rock were As <30-60 µg/L, Cd <2-3 µg/L, Cr <20 µg/L, Cu 3-4 µg/L, Mo <5-20 µg/L, Ni <20-30 µg/L, Pb <10-22 µg/L, Sb <10-20 µg/L, Se <20-33 µg/L and Zn 7-30 µg/L. Generally, the highest average values were for the tailings sand. An exception was Zn. Notably, all the pre-closure kinetic test work included regular leaching and a relatively high solution to solid ratio. Higher trace metal concentrations would be expected if the solution to solid ratio was lower or the flushing frequency decreased.

Direct sampling of drainage underground was conducted during the last few years of mining and in September 1999, shortly after the mine closed (Table 24). All the sampled drainage had a neutral or alkaline pH similar to the paste pH of the rock (e.g., pH 6.9 to 8.1). The high total metal concentrations in the drainage samples resulted from fine sediment entrained in seepage from backfilled cycloned sand, active haul roads, and areas of drilling and blasting rock. During active mining, the underground drainage was collected in sumps on the lower levels before being discharged into the tailings impoundment. Sediment cleaned out of the sumps was placed on the 180 waste rock dump and moved with the 180 waste rock dump into the tailings impoundment when the mine closed.

Dissolved metal concentrations in the underground drainage were only measured in 1998 and 1999. As expected, when the mine was operating, the underground drainage chemistry was similar to the drainage in the tailings impoundment and dissolved trace metal concentrations were much lower than total concentrations. In the extensive sampling conducted in January and February 1999, the maximum trace metal concentrations measured were As 40 µg/L, Cd 0.6 µg/L, Cu 23 µg/L, Mn 0.37 mg/L, Mo 80 µg/L, Ni 50 µg/L, Pb 73 µg/L, Sb 35 µg/L, Se µg/L and Zn 87 µg/L. The similarity between the underground and tailings impoundment drainage suggested that the primary source of contaminants was from the breakdown of rock and cycloned tailings sand, rather than weathering. Contaminant concentrations in drainage samples taken in September 1999, immediately after the mine closed, were generally similar or had decreased, although there were slight increases observed in the maximum concentrations of dissolved As (80 µg/L) and Zn (87 µg/L).

The conclusion from site monitoring and laboratory tests conducted prior to closure was that for a number of years after closure, drainage from the upper, unflooded workings would have a neutral pH and relatively low trace metal concentrations. Limitations noted in the review of this data were that most of the data came from in-situ monitoring of or test work done on fresh material, and with no data from weathered tailings sand, the material with the highest soluble trace metal concentrations and therefore the greatest concern when the workings were flooded. Prior to mine closure there were no examples and thus no information provided on the chemistry of acidic drainage.

7.6.2 Concerns Regarding the Impact of Flooding

During development of the plan to flood the lower workings, concerns were raised that raising the water table could, at least initially, increase trace metal discharge. Metal dissolution would depend on the local pH, the reduction-oxidation potential and the concentrations of soluble metals in the backfill. Factors potentially contributing to increased dissolution included removal of solubility constraints limiting contaminant dissolution, increased rates of leaching, and the collapse and flow of backfilled sand. When the mine was operating, the majority of flow within the mine occurred on the floor of stopes and adits. The proposed flooding would bring drainage in contact with mine walls and backfill that have been wet due to condensation and weathering for up to nine years, but which may never have been strongly leached. Another possible source of contaminants are precipitated solutes from process water added with the backfill. Assuming other properties are similar, materials that have previously experienced little or no leaching would have higher concentrations of soluble contaminants than the materials in flow paths where soluble contaminants were regularly removed.

The net effect of flooding on the overall quality of discharge from the mine workings would depend on:

- the increase in dissolution relative to the increased volume of water; water movement within different materials; and
- factors controlling the subsequent mixing of drainage within the flooded workings and the mixing with drainage from the upper, unflooded workings, when the water table reaches drainage from the height of the 300 Portal.

A hydrogeological model based on results from mine test work and measurements of flow and water quality underground was used to predict future metal concentrations in discharge from the partially flooded underground workings (see Section 7.6.3).

In addition to metals, there were also concerns that at least initially, flooding would result in the discharge of floating materials, such as woody debris and hydrocarbons, that are less dense than water. To address the concern regarding hydrocarbon discharge, the mine placed booms at the mouth of the 300 Portal to catch contaminants they thought might be discharged when the water table first reached the 300 Level. Several porous rock berms were placed in the adit upstream of the 300 Portal to collect floating solid debris. Hydrocarbons and other floating debris were not being observed in the discharge from the partially flooded workings, perhaps in part because discharge comes from drill holes that intersect the flooded workings below the flooded surface rather than drainage overflowing the crest of the 300 Level adit.

7.6.3 Results of the Model Used to Predict the Chemistry of the Initial Discharge from the Partially Flooded Mine Workings

A water quality model was developed to predict the impact of flooding the lower workings on the initial quality of discharge from the underground workings (Sibbick, 2000). The model's prediction of the quality of discharge from the partially flooded underground was based on predictions of:

1. the quality of drainage from the upper unflooded workings;
2. the quality of drainage in the initially flooded lower workings;
3. subsequent changes in the quality of drainage in the flooded lower workings due to sulphide oxidation from flooded materials, and
4. mixing of drainage from flooded and unflooded workings prior to discharge.

The model used available monitoring data. The data, procedure and assumptions were as follows.

7.6.3.1 Predicted Chemistry, Loading and Rate of Discharge from Unflooded Portion of the Mine Workings

The rate of discharge from the 180 Portal minus water added with the tailings sand (see Section 7.4.1) was used to predict the rate of discharge (total water volume). Average humidity cell rates for weeks 15 to 40 were used to predict the contaminant additions (Table 28). The data, calculations and correction factors used were as follows:

Predicted Chemistry (mg/L) = Total Load (mg/wk) / Total Water Volume (L/wk)

- Total Load = Backfill Load + Wall Rock Load
- Total Water Volume = 31,000 m³/month or 7,750,000 L/wk (see Table 25)

Backfill Load = Hcell leach rate (mg/kg/wk) x Contributing Mass of Backfill (28,786,370 kg) x Time drainage passes through backfill portion of workings (0.75 wk)

- Hcell leach rate = Mean leaching rate of the four cyclone tailings sand Hcells for weeks 15 to 40 (see Table 28)
- Contributing Mass of Backfill = Total Mass (575,727,400 kg) x Proportion of Material Reacting (0.05) = 28,786,370 kg

- Total Mass = 575,727,400 kg = 70% of total mass of cycloned tailings sand and waste rock used as backfill in underground workings; 70% is approximate proportion in the upper, unflooded workings.
- Proportion of Material Reacting = Proportion of the backfill (or wall rock) expected to behave in an identical fashion to a humidity cell. For this model a value of 0.05 was used. The assumption was that at any given time, 1/20 of the backfill (or wall rock) mass would be behaving as if placed in a large humidity cell.
- Time drainage passes through backfill portion of workings = 0.75 wk = 1 week x proportion of workings filled with backfill (0.75)

Wall Rock Load = Hcell leach rate (mg/kg/wk) x Contributing Mass of Wall Rock (14,773,750 kg) x Time drainage passes through open portion of workings (0.25 wk)

- Hcell leach rate = Mean leaching rate of the 150 and 180 Waste Rock Dump Hcells for weeks 15 to 40
- Mass of Wall Rock = Volume of void space (111,500 m³) x Rock Density (2650 kg/m³) x Proportion of Material Reacting (0.05) = 14,773,750 kg
- Time for drainage to pass through this portion of workings = 1 week x proportion of void space (0.25); void space is volume of upper workings not filled with backfill and left empty. Void space value was used as an estimate of the volume of rock with which the percolating waters would interact.

Predicted trace metal concentrations in the drainage from the unflooded workings were: 27 µg/L As, 19 µg/L Ni and Cr, 9 µg/L Pb 8 µg/L Mo, 7 µg/L Zn, 3 µg/L Cu and 1.7 µg/L Cd. As shown in Table 28, the predicted drainage chemistry and measured concentrations in the unflooded workings were similar. This was because water quality measured in the upper workings was used to select correction factors, such as the proportion of material reacting and the mass of reactive wall rock.

7.6.3.2 Predicted Chemistry of Drainage in the Lower, Flooded Workings

In the calculation of the predicted chemistry of the water in the flooded lower workings, it was assumed that the lower workings would act like a bathtub filled with drainage from the unflooded workings, soluble metals leached from the backfill and wall rock during flooding, and the continued release of contaminants due to weathering of the flooded backfill and wall rock. The input parameters included:

- the volume of water (estimated from a prediction of the void space);
- contaminant additions in the initial flood water (based on previous prediction of drainage chemistry from the upper, unflooded workings);
- leaching when wastes were first flooded (based on leaching results of EPA 1312 tests); and
- subsequent metal release from flooded wastes (based on rates of humidity cell tests for weeks 15 to 40).

Since the distribution of backfill in the sub-300 workings was not well known, it was assumed to be evenly distributed across the depth of the workings.

Predicted Initial Chemistry of Water Flooding the Lower Workings

The calculation of initial chemistry of the floodwaters immediately following flooding assumed there were two contaminant sources:

1. initial load in flood water, which comes from the unflooded workings, and
2. solutes leached from wall rock and backfill as the lower working are flooded.

Concentration = Total Load / Volume of Water (L)

- Total Load = Initial Load + Wall Rock Load + Backfill Load
- Total Water Volume (L) = Void Space = Volume of sub-300 workings not filled with backfill

Initial Load = Input Load + Wall Rock Initial + Backfill Initial

- Input Load = Void Space (149,193 m³) x Metal Concentrations in Drainage from Unflooded Workings
- Wall Rock Initial = Mass of Wall Rock (52,412,230 kg) x Waste Rock EPA 1312 (mg/kg) x Reactive Volume Proportion (0.1)
- Backfill Initial = Mass of Backfill (89,404,680 kg) x Backfill EPA 1312 rate (mg/kg) x Reactive Volume Proportion (0.1)
- Mass of Wall Rock = Void space volume (149,193 m³) x rock density (2650 kg/m³) = 52,412,230 kg
- Waste Rock EPA 1312 leach = Average mg/kg from EPA 1312 tests performed on 150 and 440 Waste dump rocks (see results in Table 14)
- Backfill EPA 1312 leach = Average mg/kg from EPA 1312 tests performed on four backfill samples (see results in Table 14)
- Reactive Volume Proportion = Proportion of backfill and wall rock, which behaves the same as a humidity cell. A value of 0.1 or 10% was used.

Predicted Final Drainage Chemistry in the Flooded Workings

Contaminant concentrations in the flooded 'pool' of water in the lower workings started with the Initial Flooded Conditions (see above). The load and subsequently the concentrations were then increased weekly assuming the addition due to leaching of the backfill and wall rock was the rate measured from week 14 to 41 in the humidity cell tests for the tailings sand and waste rock respectively.

Concentration = Total Load/Void Space

- Total Load₁ = Initial Load₀ + Backfill Leach₁ + Wall Rock Leach₁ (Week 1)
- Total Load₂ = Total Load₁ + Backfill Leach₂ + Wall Rock Leach₂ (Week 2)
- Total Load₃ = Total Load₂ + Backfill Leach₃ + Wall Rock Leach₃ (Week 3)
- Total Load₄ = Total Load₃ + Backfill Leach₄ + Wall Rock Leach₄ (Week 4)
- Total Load₅ = Total Load₄ + Backfill Leach₅ + Wall Rock Leach₅ (Week 5)
- Total Load_n = Total Load_(n-1) + Backfill Leach_(n) + Wall Rock Leach_(n) (Week n)

- Backfill Leach = Mass of Backfill x Backfill Hcell Rate x Reactive Volume Proportion
- Wasterock Leach = Mass of Wall Rock x Waste Rock Hcell Leach rate x Reactive Volume Proportion
- The masses of backfill and wall rock were the same as for initial flooding
- Reactive Volume Proportion = proportion of backfill and wall rock which behaves the same as a humidity cell. For the flooded workings a value of 0.1 was used.

The additions were stopped at 46 weeks when the sulphate concentrations in the flooded workings approached 1000 mg/L, with 1000 mg/L assumed to be a solubility constraint that when reached would prevent further sulphide oxidation. In reality, sulphide oxidation would be controlled by the availability oxygen and not by sulphate concentration.

The predicted “Final” trace metal concentrations in the drainage from the flooded workings were: As 77 µg/L, Ni 56 µg/L, Pb and Cr 28 µg/L, Mo 29 µg/L, Zn 21 µg/L, Cu 25 µg/L and Cd 4.4 µg/L (Table 29). These values were typically two to three, and in the case of Cu up to eight, times higher than the predicted concentrations from the unflooded workings. The greater increase in Cu was due to the relatively high weak acid soluble Cu measurements for the weathered waste rock (Table 14).

7.6.3.3 Predicted Chemistry of the Resulting Discharge from the 300 Level

The water quality model assumptions for the combined discharge of drainage from the upper unflooded and lower flooded workings were that (Table 29):

- the only significant discharge point for drainage from the mine would be the 300 Portal;
- when the lower workings were flooded, drainage from the upper, unflooded workings would mix with the upper 2 m of the floodwater from the lower workings and would then start to drain out the portal;
- water exiting the 300 Portal would be from this Mixing Zone;
- the 2m Mixing Zone was estimated to have a total volume of 3068.6 m³, with backfill occupying 1,464.32 m³, the volume of wall rock was 306.86 m³, and the void space or flooded volume was 1,604.28 m³; and
- rate of discharge was 31,000 m³ per month, the rate of discharge from the upper workings; consequently 95% of the discharge was drainage from the upper, unflooded workings.

The predicted trace metal concentrations in drainage discharged from the 300 Portal were: As 29 µg/L, Ni 21 µg/L, Cr 17 µg/L, Pb 9 µg/L, Mo 9 µg/L, Zn 8 µg/L, Cu 4 µg/L and Cd 1.8 µg/L (Table 29). The predicted discharge quality was very similar to the predicted water quality from the unflooded workings. Because the model assumed 95% of the discharge was from the upper, unflooded workings, the higher predicted trace metal concentrations in water quality in the flooded workings had only a minor influence on contaminant concentrations in the 300 discharge. The predicted quality of drainage discharge from the 300 Level was well below the site discharge criteria for two of the three trace metals and slightly higher than that for Cd (predicted 1.8 µg/L versus site discharge limit of 0.8 µg/L, Table 29). The Cd concentration in the 300 discharge was not considered a concern because the available dilution and attenuation within the impoundment was expected to lower the concentration by at least an order of magnitude prior to being discharged to the environment. Based on the similarity between predicted concentrations and the pre-flooding

discharge, it was assumed that the leaching of weathered backfill during the flooding of the lower workings would not have a significant adverse impact on water quality in the tailings impoundment.

7.6.4 Post-Flooding Drainage Chemistry Results

Post-flooding monitoring of drainage in the underground workings included measurement of:

- flow rate and chemistry of drainage discharge from the 130, 180 and 300 Portals; and
- chemistry of drainage behind 130 Bulkhead and seepage from a fracture in front of the 180 Bulkhead.

7.6.4.1 Chemistry of Drainage Behind the 130 Bulkhead

Water quality samples from the flooded workings were obtained by removing the pressure gauge from the tubing through to the flooded side of the plug. Water was allowed to flow through the tubing for approximately 10 minutes before a sample was taken. Water quality samples from the flooded workings behind the bulkhead were collected in 2000 and 2001, the first two years of flooding. Results are shown in Table 30.

Observations about the chemistry of the drainage behind the 130 Bulkhead in 2000 and 2001 included the following:

- pH values were typically 7.4 to 8.1 from January to August 2000 and subsequently pH 8.0 or above;
- alkalinity remained around 200 mg/L;
- maximum values of sulphate, Ca, Mg, D-Mn and T-Zn were on the third monitoring date, February 17th, 2000;
- sulphate decreased from 493 mg/L in February 2000 to 246 to 262 mg/L in 2001, with somewhat similar decreases evident for Ca and Mg;
- with the exception of Zn and Fe, metals were almost all in the dissolved form;
- relatively low maximum concentrations were observed for most trace metals, D-As (maximum of 0.8 µg/L), D-Cd (maximum of 0.6 µg/L), D-Cu (median < 2 µg/L), D-Mo (all < 30 µg/L), D-Pb (median < 1 µg/L), D-Sb (maximum of 2.5 µg/L) and D-Se (maximum of 1.4 µg/L); and
- D- Ni was 25 µg/L when the workings first flooded, but declined steadily to 2 µg/L in December 2000 and did not exceed 3 µg/L in 2001.

The highest trace element concentrations were for Zn and Mn. Total-Mn and D-Mn were similar indicating that Mn was almost entirely dissolved. The highest Mn concentrations were in the initial period following flooding, remaining above 500 µg/L until December 2000, with a maximum D-Mn concentration of 692 µg/L. Subsequently, D-Mn decreased steadily, reaching 135 µg/L in June 2001.

Total-Zn concentrations were similar to total-Mn throughout the first half of 2000. The maximum total-Zn value was 654 µg/L in June. In the latter part of 2000, total-Zn decreased and in 2001, total-Zn concentrations fluctuated between 16 and 355 µg/L. Unlike Mn, the relative magnitudes of total-Zn and D-Zn fluctuated widely. When monitoring started, most of the Zn was in the dissolved form (e.g., T-Zn of 631 µg/L versus D-Zn of 583 µg/L), but D-Zn decreased to 382 µg/L in

February and was 148 µg/L in June. From July to November 2000, D-Zn varied from 30 to 57 µg/L. From December 2000 to the last monitoring date in November 2001, D-Zn fluctuated from < 5 µg/L to 294 µg/L. For example, D-Zn was 294 µg/L in July 2001, but had declined to 8 µg/L when the next sample was taken in November.

Total and dissolved-Fe were unique in that their concentrations increased over time. In 2000, total-Fe increased from 0.07 mg/L to 1.27 µg/L. In 2001, total-Fe concentration fluctuated between 0.47 and 0.98 mg/L. D-Fe went up and down several times (Jan-July 2000 <0.03 to 0.12 mg/L; July-Oct 2000 0.29 to 0.48 mg/L; Nov 2000 – March 2001 <0.03 to 0.09 mg/L; April-May 2001 0.25 to 0.55 mg/L; June 2001 0.03 mg/L; and July-Nov 2001 0.52 to 0.65 mg/L). As a result of the great variability in concentration, the proportions of total-Fe that existed in the dissolved form varied 0 to 100%. The most likely source of Fe in the drainage was dissolution of weathering products and Fe carbonate minerals, and consumption of dissolved oxygen resulting in a reduction in the reduction-oxidation potential within the backfill. Iron is soluble and thus mobile under reduced conditions. The reduced portion of D-Fe compared to total-Fe at the bulkhead may have been due to a higher reduction-oxidation potential in the open mine workings or oxygen entry into the water samples prior to filtering and acid addition. Co-precipitation with Fe is likely responsible for the observed differences in T-Zn and D-Zn.

Various processes would occur within the flooded workings, including dissolution of lime in the cemented backfill and progressive replacement of the initial floodwater from the upper workings by groundwater. Oxygen in the flooded workings likely decrease first within stopes backfilled with tailings sand, especially if there is anoxic groundwater entering the back of the stope. The presence of dissolved Mn indicates that the reduction-oxidation potential was too high for sulphide reduction. The decrease in sulphate concentration may have been due to a number of processes including the addition of low sulphate groundwater and reduction in the flow through backfill, reducing both leaching and the re-supply of oxygen required for sulphide oxidation, once the flooding of each level was complete. Discharge of the initial floodwater and its replacement by fresh groundwater were probable causes for the observed decrease in sulphate levels and slight increase in pH. Adding nitrate to the list of analyses would be one way to improve the monitoring of the reduction-oxidation potential.

7.6.4.2 Chemistry of Drainage from Fracture Downstream of 180 Bulkhead

The 180 Bulkhead does not contain a portal for sampling water behind the plug, as there is in the 130 Bulkhead. However from January to August, 2000, samples were collected from a seep through a fracture a short distance downstream of the bulkhead that was presumed to come from the flooded workings. The results shown in Table 31 include a slight decrease in pH from 7.8 to 7.3 in March and rebound to pH 7.7 in August was similar in magnitude and timing to changes observed in the drainage behind the 130 Plug. Other similarities with water quality behind the 130 Bulkhead were the elevated Mn and Zn concentrations, and the low concentrations of other trace metals.

One difference from the drainage behind the 130 Bulkhead was the higher initial Cd concentration (1.8 µg/L). Subsequently, Cd in the 180 Seep decreased to the low concentrations of < 0.2 to 0.3 µg/L, similar concentrations to the drainage behind the 130 Bulkhead. Sulphate in the 180 Seep decreased from 475 µg/L in January to 284 µg/L in August. The quicker decline in sulphate levels compared to the 130 Bulkhead perhaps results from more mixing with lower sulphate drainage from

the upper unflooded workings. More mixing with oxygenated water from the upper workings would result in a higher reduction-oxidation potential. Unlike the drainage behind the 130 Bulkhead, almost all the Zn was dissolved. D-Zn was 633 to 662 µg/L from January to March decreasing to 306 to 376 µg/L from May to August. The decline in D-Zn in the 180 Seep was slower than the decline in D-Zn behind the 130 Bulkhead. During the limited period of record, levels of both T-Fe and D-Fe in the 180 Seep remained low, although there were small peaks in March of 0.19 µg/L and 0.12 µg/L, respectively. The decrease observed for nitrate (2.42 to 0.477 mg/L) was proportionally much larger than either sulphate or Zn, but similar to the decrease observed in the discharge from the 300 portal.

7.6.4.3 Chemistry of Discharge from the 130, 180 and 300 Portals

Post-closure measurement of the drainage chemistry of discharge from the 130 and 300 Portals started in January 2000 and from the 180 Portal in April 2000. Drainage chemistry results for individual samples from the three portals are listed in Tables 32, 33 and 34, with minimum, median and maximum values for all three sites plus the tailings impoundment compiled in Table 35. As expected from the high NP, none of the portal discharges showed evidence of ARD. All three portals had similar minimum, median and maximum pH values (7.6-7.8, 8.0-8.1 and 8.3-8.4, respectively) and no noticeable changes over time were observed. A similar lack of variation over time was observed for alkalinity, as illustrated by the following compilation of maximum values for drainage behind the 130 Bulkhead in 2000 and 2001 and the portals' discharge in 2000, 2001, 2002 and 2003:

130 Bulkhead – 237 and 255 mg/L;
130 Portal – 173, 164, 153 and 167 mg/L;
180 Portal – 206, 206, 196 and 197 mg/L; and
300 Portal – 183, 190, 193 and 184 mg/L.

All the portal discharges contained significant alkalinity, with the highest in the discharge from the 180 Portal. A higher proportion of high alkalinity drainage from the flooded mine workings likely explains both the higher alkalinity in the drainage in the 180 Portal discharge compared to the other portals, and why alkalinity was highest during summer, the period of the year with the lowest near surface runoff.

Similar to drainage in the flooded workings (the 180 Seep and the 130 Bulkhead), with the exception of Mn and Zn, trace metal concentrations were typically close to or below their detection limit in the discharge from all three portals. Maximum observed trace element concentrations in the discharge from 130, 180 and 300 Portals and the tailings impoundment were:

As – 5.6, 0.9, 1.2 and 2.9 µg/L;
Cd – 4.3, 1.2, 1.7 and 0.3 µg/L;
Cu – 13, 1.2, 4 and 4 µg/L;
Sb – 1.6, 3.5, 8.8 and 1.9 µg/L; and
Se – 1.6, 2.0, 1.9 and 1.1 µg/L.

Maximum concentrations of As, Sb and Se observed in site drainage were an order of magnitude below the maximum values observed in the long-term humidity cell test. The maximum values of Cd (4.3 µg/L) in 130 Portal drainage and Sb (8.8 µg/L) in the 300 Portal drainage were single

anomalies, an order of magnitude higher than concentrations measured in surrounding months, and were likely analytical errors. The maximum values of As (5.6 µg/L) in 130 Portal drainage occurred at the same time as the maximum D-Fe (0.46 mg/L). Copper was also higher in the drainage from the 130 Portal, which may have been due to weathering of Cu mineralization in the Red Bluff rock, intercepted by the adit near the 130 Portal.

Similar to the chemistry of the flooded workings, the portal discharges all contained elevated Mn and Zn when monitoring started. Similar to the 180 Seep and unlike drainage from behind the 130 Bulkhead, there was little or no difference between D-Zn and T-Zn in the portal discharge. Maximum D-Zn values for the 130 Bulkhead in 2000 and 2001, and the 130, 180 and 300 Portals and the tailings impoundment in 2000, 2001, 2002 and 2003 were as follows:

130 Bulkhead – 654 and 294 µg/L;
130 Portal – 51, 29, 25 and 40 µg/L;
180 Portal – 129, 32, 92 and 15 µg/L;
300 Portal – 388, 120, 83 and 66 µg/L; and
tailings impoundment – 27, 11, 9 and 9 µg/L.

In 2000 and 2001, maximum D-Zn behind 130 Bulkhead > 300 Portal > 180 Portal > 130 Portal > tailings impoundment. Similar to the drainage behind the 130 Bulkhead, peak Zn values in the portal discharge occurred immediately following flooding and generally declined each year, with the largest decline occurring during the first year. The highest observed Zn concentration in the portal discharge was 388 µg/L in January 2000, the first time discharge from the 300 Portal was analyzed. By the fourth year (2003), the range in Zn concentration was 11 to 40 µg/L for the 130 Portal, 10 to 15 µg/L for the 180 Portal, 46 to 60 µg/L for the 300 Portal and < 5 to 9 µg/L for the tailings impoundment.

Maximum D-Mn values for the drainage behind the 130 Bulkhead in 2000 and 2001, and discharge from the 130, 180 and 300 Portals and the tailings impoundment in 2000, 2001, 2002 and 2003 were as follows:

130 Bulkhead – 692 and 499 µg/L;
130 Portal – 65, 17, 14 and 13 µg/L;
180 Portal – 214, 101, 24 and 26 µg/L;
300 Portal – 476, 157, 131 and 70 µg/L; and
tailings impoundment – 467, 120, 181 and 168 µg/L.

Similar to D-Zn, maximum D-Mn behind 130 Bulkhead > 300 Portal > 180 Portal > 130 Portal in 2000 and 2001. Like D-Zn, maximum Mn values in the portal discharge occurred immediately following flooding. However unlike Zn, maximum D-Mn in the discharge from the tailings impoundment was similar to the 300 Portal in 2000 and 2001, and exceeded it in 2002 and 2003. Like Mn, sulphate concentrations in the portal discharge declined each year, with the 130 Bulkhead > 300 Portal > 180 Portal > 130 Portal. This is illustrated by the following compilation of maximum July and August sulphate values for 2000, 2001, 2002 and 2003:

130 Bulkhead – 493 and 260 µg/L;

130 Portal – 244, 174, 163 and 162 mg/L;
180 Portal – 263, 200, 190 and 176 mg/L;
300 Portal – 278, 237, 220 and 193 mg/L; and
tailings impoundment – 243, 201, 190 and 167 mg/L.

However, the amount by which the sulphate concentrations decreased and differences between portals were much smaller than for either Mn or Zn. Median concentrations of sulphate were 200 mg/L for the 300 Portal, 186 mg/L for the 180 Portal and 132 mg/L for the 130 Portal and the relative magnitude of these differences have remained fairly consistent. For example, on June 4, 2003 with flows of 2.2, 0.8 and 8.5 L/s respectively, the sulphate concentration was 120, 171 and 193 mg/L from the 130, 180 and 300 Portals. Maximum sulphate concentrations typically occurred during periods of low flow in the winter and summer. The similarity between sulphate concentrations in discharge from the 300 Portal and the lower portals is possibly due to less attenuation of sulphate than Mn or Zn as drainage flows through bedrock fractures. The relatively small decline in annual loadings could be explained by the much higher concentrations of soluble sulphate compared to either Mn or Zn in the EPA1312 (Table 14) or the Modified-SWEP (Table 15) results for the unweathered tailings sand.

Stored sulphate may have maintained sulphate concentrations in the adit discharge for a considerable period of time even if flooding prevented additional sulphide oxidation. A higher input of near surface drainage in the adit discharge was the probable explanation for why the 130 Portal discharge had a lower sulphate concentration than the 180 Portal discharge. Similar sulphate concentrations in the discharge from the 180 and 300 Portals suggests that drainage from the upper unflooded workings had a lower sulphate concentration than the flooded workings.

Although there were differences between the portals and progressive changes in some parameters over time, from month to month the concentrations of most parameters in the portal drainage chemistry were relatively consistent. However, similar to the drainage results for samples collected behind the 130 Bulkhead, there was significant variability in the concentration of Fe. Total-Fe in the 300 Portal drainage started at 0.44 µg/L in January 2000, decreased to 0.04 µg/L, increased to 0.51 µg/L, decreased to 0.07 µg/L, increased to 0.42 µg/L in December and 1.86 µg/L in February 2001 and decreased to 0.43 µg/L in March. Total-Fe concentrations in the 180 Portal drainage were also quite erratic, with changes from < 0.1 µg/L to > 0.75 µg/L and back to < 0.1 µg/L on successive monitoring dates. In 2003, total-Fe in the 180 Portal drainage went from 0.11 µg/L to 8.36 µg/L to < 0.03 µg/L to 0.03 µg/L and finished in December at 4.10 µg/L. Lower concentrations and less variability were observed in total-Fe and D-Fe in the drainage from the 130 Portal compared to the other two portals.

7.6.4.4 Comparison of Water Quality Model Predictions and the Resulting Drainage Chemistry

Model predictions for sulphate and major base cations were quite similar to the maximum analysis results for drainage samples collected behind the 130 bulkhead:

- sulphate – predicted 534 vs. actual 493 mg/L;
- Ca – predicted 215 vs. actual 235 mg/L;
- Mg – predicted 26 vs. actual 27 mg/L; and
- Mn – predicted 0.77 vs. 0.69 mg/L.

and maximum (initial) concentrations in the 300 Portal discharge (Tables 29 and 35):

- sulphate – predicted 404 vs. actual 428 mg/L;
- Ca – predicted 146 vs. actual 205 mg/L;
- Mg – predicted 23 vs. actual 24 mg/L; and
- Mn – predicted 0.37 vs. actual 0.48 mg/L.

Similarities between predicted and actual values for sulphate and major base cations were to be expected because drainage chemistry in the unflooded workings was used to set the various ‘model calibrations’ (e.g., time drainage passes through the unflooded backfill portion of workings and the mass of wall rock and its leaching rate) and there was not a lot of difference between concentrations in the unflooded workings, the initially flooded workings and the initial 300 discharge. However, the model overestimated subsequent sulphate and major base cation concentrations in the flooded workings and the 300 Portal discharge because it assumed concentrations would increase when they actually decreased.

Actual trace metal concentrations in the discharge from the 300 Portal and drainage behind the 130 Bulkhead were much lower than the predicted values, with median values for As 0.7 and 0.45 µg/L, Cd 0.4 and < 0.2 µg/L, Cu 4 and < 5 µg/L, Ni 3 and 8 µg/L, and Pb both < 1 µg/L, respectively. Concentrations were also low for Sb and Se, trace metals not predicted by the model, but released in relatively high amounts in the long-term humidity cell tests. Maximum concentrations of Sb and Se in the 300 Portal discharge were well below the maximum concentrations observed in the humidity cell results (Sb 8.8 vs. 67 and Se 1.9 vs. 120 µg/L). Reasons for the model’s overestimation of most trace metal concentrations were:

- the prediction was that concentrations in the ‘final’ flooded workings would increase when they actually decreased; and
- these elements commonly occurred below the detection limit in the humidity cell analysis.

When elements occurred below the detection limit, the detection limit was used to calculate the humidity cell release rates used in the model. Actual humidity cell concentrations were lower than the detection limit and may have been much lower.

The one notable exception among the trace metals was Zn, whose concentration was greatly underestimated by the model. Maximum and median concentrations of Zn were 388 and 80 µg/L vs. 8 µg/L predicted in the actual discharge from the 300 Portal. Maximum and median concentrations of Zn were 583 and 57 µg/L for the drainage behind the 130 Bulkhead vs. 21 µg/L predicted for the flooded lower workings. Maximum Zn values in the actual discharge occurred immediately after the lower workings were flooded. However, even the range in D-Zn of 46 to 66 µg/L in 2003 for the discharge from the 300 Portal was well above the prediction of 8 µg/L. Based on the high inflow rates measured in July and August 1999, the water table probably reached the 300 Level sometime in November or December 1999, and the initial elevated Zn concentrations in the portal discharge may have been higher than the concentrations measured in January 2000. Fortunately, dilution and attenuation in the impoundment reduced Zn concentrations by at least an order of magnitude prior to discharge off-site.

Factors likely contributing to the underestimation of Zn were:

- tailings sand data used in the model came from test work done on fresh rather than weathered samples; and
- at least initially, a high proportion of the discharge may have come from the water flooding the lower workings.

Unlike the waste rock samples, which came from dumps, the tailings sand used in the EPA 1312 and humidity cell tests were fresh unweathered samples, whose soluble Zn content was not representative of older backfill, which had been exposed to up to eight years of weathering and was relatively unleached. In part because of the difficulty in accessing older stopes, older, potentially weathered backfilled tailings sand were not analyzed prior to flooding. Access and a lack of seepage prevented the monitoring of drainage from older stopes and as a result the data on drainage chemistry collected prior to flooding came primarily from areas where backfilled materials were relatively new and weathering effects were masked by the drainage inputs and chemistry of active mining. The relatively high solubility of Zn at neutral pH and the fact that galvanic effects may have accelerated the oxidation of Zn sulphide, while suppressing the oxidation of other sulphide minerals are possible explanations for why other trace metals weren't similarly underestimated. Notably, Zn and Mg were unique among the parameters monitored in the tailings sand humidity cells in that the average production rate increased during the test (e.g., weeks 10 to 30 compared to last 24 weeks and overall - Table 16).

A number of factors may have caused a higher than predicted proportion of the 300 Portal discharge to come from the flooded workings. These include:

- discharge first started in mid-winter when near surface inputs to the underground workings would be lowest and the ratio of drainage inputs to the lower workings compared to the upper workings would be relatively high;
- turbulence and thus the depth of mixing within the flooded workings itself and with drainage from the upper workings would be highest when flood water first reaches the discharge elevation; and
- discharge was through diamond drill holes that intersected the water column potentially well below the surface (Section 7.5.2).

In addition to sulphide oxidation, another suggested source for the elevated Zn, especially in the initial discharge was galvanized wire left in the flooded workings (Marlin Murphy, pers. communication).

The experience of Snip in attempting to predict water quality from the partially flooded mine workings indicated a number of challenges and potential pit falls in water quality modeling. The challenges included:

- collecting data from materials representative of the range in composition and conditions being predicted;
- selecting appropriate tests; and
- extrapolating test results from small samples analyzed in the laboratory to large-scale field conditions.

No matter how sophisticated the model, it is only as accurate as the data it uses. It is very important that the data used in a predictive model comes from samples that are representative of the intended materials both in terms of their initial composition (e.g., ABA and total metal concentrations) and the subsequent weathering and leaching. Conversely, it is important to understand the limitation of the data and not use the results to predict concentrations for different conditions than those of the samples (e.g., neutral drainage pH) or the analytical procedures (e.g., ratio of extractant to solid or the duration of test). The model was developed during the last phases of mining at the site, when there was a shortage of personnel and the ability to collect samples and information was severely limited. If the need for a prediction model had been identified earlier in the closure planning, it would have made it far easier to collect the required information.

The only calibration of the model was a comparison of the prediction for the unflooded workings with drainage chemistry measured during active mining. Extrapolating results from the laboratory and active mining to future post-closure field conditions is a major challenge. The water quality prediction model made a number of assumptions regarding critical field processes or properties, post-closure material performance and the hydrology of workings. These included the following.

- The assumed mixing zone for drainage from the upper and lower workings was the upper 2 meters of the water column, with only 5% of the eventual discharge from the underground coming from the drainage in the flooded workings. The 2 m mixing zone was partly based on the assumption that the height of the water column would be controlled by overflow over the lip of the 300 Adit. Discharge through diamond drill holes in cut-outs at 297 m asl further down the adit connecting to the water column some unknown distance below 297 m elevation should have resulted in a deeper mixing zone. The model did not account for the observed discharge around the bulkheads and potentially greater mixing with drainage from the upper workings and losses from the top of the water column out the 300 Portal. Discharge of the initial floodwater and replacement by new ground water was potentially responsible for the marked reduction in SO₄, Ca and Zn concentrations observed behind the 130 bulkhead in the first two years of flooding.
- Complete mixing was assumed in the flooded workings. Mixing was likely to be highest during the initial flooding. Subsequently, flow within the flooded workings may vary greatly, with little or no water movement in isolated stopes.
- The total mass of backfill, which consisted of both waste rock and cycloned tailings sand, was assumed to react like the tailings sand. This might have been intended to be a conservative decision because the tailings sand contains higher concentrations of trace metals and sulphide-S than the waste rock. However, the tailings sand samples used to predict metal release were unweathered, which likely reduced the predicted metal release.
- The time that the drainage takes passing through the unflooded backfill portion of workings was assumed to be 0.75 wk. The time drainage passed through the unflooded wall rock portion of workings was assumed to be 0.25 wk.
- The mass of wall rock was assumed to have a mass equivalent to void space x rock density (2650 kg/m³). The leaching rate was assumed to be equivalent to waste rock. The assumption was that all the unfilled workings would collapse.
- 5% of the backfill and wall rock in the upper unflooded workings was assumed to behave in an identical fashion to a humidity cell.

- The entire mass of flooded backfill and wall rock in the lower workings was assumed to initially behave similarly to the tailings sand and the waste rock in the EPA 1312 weak acid leach test. Once the lower workings were flooded, 10% of the flooded backfill and wall rock was assumed to behave like a humidity cell until sulphate concentrations reached saturation. In reality, oxidation would be controlled by the availability of oxygen and sulphate saturation would only limit the dissolution of sulphate minerals. Flooding should reduce the supply of oxygen and the creation of soluble contaminants. The impact of flooding on the leaching of soluble contaminants will depend on the geochemistry, rate and direction of groundwater movement through the weathered materials.

7.6.5 Future Water Quality and Metal Loadings from the Mine Workings

The high NP of the backfill should ensure that relatively high drainage pH and alkalinity will continue for a number of years. There is uncertainty about what will happen in the short-term to the concentrations of trace metals. Depletion of more readily weathered minerals and finer particles and the development of protective surface coatings may reduce trace metal concentrations. Conversely, there may be increases potentially exceeding concentrations observed in the humidity cells and the initial drainage. Factors potentially causing increased trace metal loadings in neutral pH drainage from the upper workings included:

- increased drainage inputs and leaching in the upper unflooded portion of the mine as a result of surface subsidence;
- increased trace metal solubility, due to depletion of major cations and alkalinity added in process water during operation;
- increased leaching and weathering as a result of the collapse of backfill and the workings themselves; and
- the cumulative build-up of weathering products resulting in larger flushing events.

Assuming the lower workings remain flooded, the highest trace metal loadings from the underground workings are predicted when Ca and Mg CO₃ are exhausted in the PAG materials in the upper unflooded portion of the workings. The ABA results indicated that acidic weathering and acidic drainage are likely from some portion of the unflooded backfilled tailings sand and were possible from a portion of the waste rock and collapsed workings. Discussion of the factors influencing the amount and strength of ARD and the pH of the eventual discharge are discussed in Section 7.2.5. The humidity cell tests indicate that acidic weathering will not occur for at least 20 to 30 years. According to Morin and Hutt (1997), it took 15 years for the Island Copper Mine waste rock dumps, with 0 to 4% S and NP values of less than 80 kg/t, to produce measurable ARD.

Flooding is expected to minimize sulphide oxidation in the lower workings and the resulting low metal levels and alkalinity may provide some dilution or neutralization of the drainage from the upper workings. However, unless there is some sustainable mechanism of metal removal, continuous addition of drainage from the upper workings will soon result in similar water quality in the mixing zone. It is also important to note that mixing and therefore dilution may be minimal at certain times of the year or during major runoff events, and that even if drainage from PAG wastes is neutralized underground, Cd and Zn and other metals released by acidic weathering will remain entrained in the drainage causing increased trace metal loadings to the environment.

The complexity and heterogeneity of the wastes and workings, the limited extent of our generic understanding of the processes controlling ML/ARD and the lack of long-term data from other, similar mine workings make it very difficult to predict drainage chemistry and metal loading in the immediate future, let alone several decades into the future. Some of the important questions yet to be answered include the following.

- What portion of the tailings sand, waste rock and mine walls will eventually be ARD generating, how long will it take and what will be the resulting acidity and metal loadings (see Sections 3.2 and 7.2.5)?
- What is the ability of NP sources in the underground to both neutralize acidic drainage and reduce associated elevated metal concentrations prior to discharge from the workings and the tailings impoundment? NP sources include non-ARD generating wastes and alkalinity in groundwater inputs to the upper workings, drainage within the flooded workings and drainage within the tailings impoundment.
- What is the rate of air entry within different areas of the upper mine (e.g., crown pillar versus remote stopes) and within the backfilled tailings sand versus voids or the waste rock?
- How low does the rate of oxygen supply need to be to reduce the rate of sulphide oxidation to a level where ARD no longer occurs in PAG materials
- Will future subsidence in the area of the crown pillar significantly increase oxygen and drainage entry into the upper mine workings, with a subsequent increase in sulphide oxidation and leaching?
- Will dissolution and physical pressure decrease the strength and increase the porosity of the rock around the bulkheads, and will this create significant fluctuations in the height of the floodwater? Is periodic grouting required to prevent this?

At mine closure, the proposed contingency plan, if discharge from the underground workings were to cause drainage in the tailings impoundment to exceed the discharge limits, was drainage collection and treatment within the tailings impoundment. Drainage could be briefly impounded in the tailings impoundment using stop-logs in the spillway. Because of the uncertainty regarding future drainage chemistry, no details were provided regarding treatment processes and the resources required, and there are no triggers set to indicate when additional actions will be taken. Presently the mine relies on the monitoring of the portal drainages to detect problems, assuming that changes in water quality, such as increases in sulphate and pH fluctuations, will allow increasing trace metal concentrations to be identified in time for actions to be taken to prevent significant impacts to the environment.

8. Discussion and Conclusion

The ML/ARD work conducted at Snip illustrates many of the challenges associated with ML/ARD. These include technical challenges:

- the large number of properties and processes in flux (e.g., air entry, backfill weathering and collapse of mine walls);
- the difficulty in measuring many key properties (e.g., mineralogy); and
- the lack of long-term data from other similar sites (e.g., underground mines).

There are also logistical challenges:

- the large costs;
- problems in accessing remote mine sites and closed underground mines;
- the large amount of information required;
- the problems of data management; and
- the departure of key staff, camp closure and the reduction in the number of personnel.

One mitigation strategy at Snip was to where possible flood materials with a potential to produce ARD using dams and bulkheads. This essentially converted a geochemical risk into a geotechnical risk. The success of this strategy will depend on the integrity of bulkheads and dams. Maintenance of the long-term geotechnical integrity requires regular inspection and, when required, repair. Another objective at closure was to create a single main discharge location, with discharge from the underground reporting to the tailings impoundment and discharge from the tailings impoundment reporting via the spillway to Sky Creek. The exceptions were the discharge from the 130 Portal and the Dam #3 seepage, both of which report to the Monsoon Creek system.

There are typically a number of unresolved issues with any ML/ARD mitigation measure. Unresolved geochemical questions regarding the tailings impoundment included the height of the water table and the extent of flooding at the upper end of the impoundment and the impact of discharge from the underground mine. The main outstanding geochemical issue at Snip is future drainage chemistry and metal loading from the upper, unflooded backfilled underground workings. Uncertainty about future metal leaching or acidity is an issue at many mine sites and creates considerable uncertainty about future information and resource needs. The long list of questions associated with future water quality, highlights the uncertainty that exists regarding this issue.

Presently, the mine relies on monitoring portal drainages to detect problems, assuming that changes in water quality will allow increasing trace metal concentrations to be identified in time for actions to be taken to prevent significant impacts to the environment. Use of monitoring instruments and data loggers to monitor air and water quality in the upper underground workings were suggested in the review of the Snip closure plan, but were rejected because of the difficulty in maintaining relatively inaccessible monitoring equipment, and concerns that a limited underground monitoring program would not address regulatory concerns regarding temporal and spatial heterogeneity.

The questions of temporal and spatial heterogeneity, regulatory acceptance and whether test conditions matched the site-specific conditions were also raised in the discussion of whether

additional prediction studies could be used to show what portion of the tailings sand, waste rock and mine walls will eventually be ARD generating, how long will it take before the NP is depleted and what will be the resulting acidity and metal loadings. The NP depletion procedure used in several humidity cells was an example of where the specific conditions at Snip were not considered when adopting test procedures from another site. Before attempting to increase understanding, one should have a clear idea of the purpose, procedural limitations, data requirements, and the manner in which results will be interpreted.

Previously, water quality monitoring was primarily used to detect environmental impact and determine permit compliance. In an effective ML/ARD program, monitoring is used to test predictions, provide a warning of potential problems and inform corrective actions. Providing sufficient resources to maintain, regularly update and review monitoring records is an important part of successful long-term management at many mine sites. Regular review of the data is required to ensure adequate data quality, not just for the evaluation of the results.

Where long-term monitoring and maintenance are required, sites need regularly updated operating manuals for site management, maintenance, mitigation and monitoring. Another key component is maintenance of a database showing the composition, location and mass of different mine components, and changes in flow, drainage chemistry, loadings and other properties that impact or reflect changes in weathering. Databases should be organized so that new information can be easily added. Well-maintained records are also very important when, as is more and more common, there are changes in ownership, personnel or reporting structures. Unfortunately, where resources are short, data management and review are often the first things to be dropped, especially for issues such as ARD where problems are unlikely to occur for a number of years.

ML/ARD closure plans require information from a wide range of technical disciplines (e.g., hydrology and geotechnical) and a thorough understanding of the site and its history. Periodic regulatory review is important in ensuring companies have sufficient understanding and resources to handle the liability and risk associated with the mine closure plan, and adapt to changing conditions. It is also important that government informs mines and communities of the regulatory implications of proposed actions. It takes considerable time and energy before a regulator can understand the complexity of the site, the mine history, and trends and limitations of the data. Short-time lines and the frequent modifications that typically occur during mine closure frequently challenge regulatory organizations reviewing closure plans.

Where there remains considerable uncertainty about future drainage chemistry, as is the case at Snip, the challenges may not diminish when the mine closes. While mine closure planning should be part of the initial mine plan and re-visited as more information becomes available throughout the mine life, from an environmental and financial perspective, operating mines are often only the tip of the iceberg. The vast majority of the mine sites with significant ML/ARD issues are closed mines.

Mine closure can be a difficult time to conduct work and collect data as budgets are cut and most of the people and equipment leave. In the decommissioning of the Snip mine, a major consideration in staff reductions and the removal of equipment was the mine's desire to dismantle housing facilities and reduce costs. The imminent departure of key staff and equipment create tight timelines for the proposed work and its review. Corporate memory loss regarding important aspects of the site may

occur long before closure as long-term staff leaves for more secure positions. Without adequate data base management, staff departure for more permanent jobs elsewhere may result in the loss of knowledge of previous activities, what information exists, how it was collected and where it is stored. While closure planning requires hard work and may include significant financial pressure, closure can also be demoralizing for staff. Staff departure was less pronounced at Snip compared to other closures because some personnel also had or took jobs at the neighbouring Eskay Creek mine and still retained responsibility for work at Snip.

One of the lessons learned at Snip was the importance of early recognition of ML/ARD risks and information deficiencies. Recognition of the geochemical risks associated with the underground workings prior to mining would likely have resulted in modifications to the mine plan, with the mine either reducing the sulphide content of the backfill (e.g., desulphurization prior to cycloning) or not using tailings sand as backfill. Recognition of ML/ARD during exploration would also have permitted more complete grouting of exploration drill holes. Earlier recognition of the uncertainty regarding future water quality would have enabled the mine to use its operating facilities and site personnel to run long-term studies underground. Recognition of ML/ARD information deficiencies earlier in the mine life is especially important for underground mines where access is limited once mining ceases.

In closure plans the 'devil is typically in the details'. The foundation of every closure plan is the raw data. This was illustrated in the discussion of operational monitoring and the water quality model. While it is easy to get lost in the details and not see the 'forest for the trees', it is equally important not to overlook the limitations of sampling and analysis. In conducting ML/ARD work it is important to integrate the micro and macro scales, and to check the accuracy of underlying data no matter how complex or detailed the analysis. In these busy times, many will only read the executive summary of this report. However, it is important to note that the executive summary only provides an outline of the ML/ARD assessment and mitigation at Snip. Proper understanding of the work and the resources required to conduct that work can only be achieved by reading the attached report.

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Table 1. Production Report (from BC Ministry of Energy and Mines Minfile)

		Metric	Imperial
Recovery:	Mined:	1,267,642 t	1,396,941 tons
	Milled:	1,308,340 t	1,441,791 tons
	Silver:	12,183,090 grams	391,695 ounces
	Gold:	32,093,701 grams	1,031,836 ounces
	Copper:	249,276 kilograms	549,554 pounds

Table 2. Mineralogy of Different Vein Types Found in the Ore (from NDM, 1998)

Mineral	Ore-Type				
	Sulphide	Chlorite-biotite	Carbonate	Quartz	Total Twin
Pyrite	58.5	3.9	4.3	4.4	17.4
Pyrrhotite	9.2	0.7	0.3	0.9	2.6
Chalcopyrite	0.3	0.07	0.04	0.1	0.1
Magnetite	1.1	0.06	0.03	0.06	0.3
Molybdenite	0.001	0.012	0.004	0.008	0.005
Arsenopyrite	1.4	0.002	0.3	0.3	0.5
Sphalerite	1	0.07	0.4	0.3	0.5
Galena	0.01	0.003	0.006	0.02	0.01
Quartz	7.4	4.6	4.7	74.2	18.5
Calcite	12.9	24.8	69.8	8.4	37.7
Chlorite	4.4	39.9	9.2	8	12.4
Biotite	3	21.8	5.5	1.7	6.6

Table 3. Results of Petrographic Analyses of Whole Tailings and Cycloned Tailings Sand (from Snip Mine, 1999a)

Mineral	Whole Tailings			Backfilled Tailings Sand		
	#1	#2	#3	#1	#2	#3
Quartz/Plagio	29	23.5	26	36.5	43.5	41
Plagioclase	-	-	-	-	-	-
Biotite	27	37.5	27.5	18.5	12.5	20
Chlorite	0.7	-	1	2	2	1
Hornblende	-	0.5	-	1	-	-
Sericite	-	-	-	-	-	3
Epidote	-	-	-	-	trace	-
Carbonate	24	21	26.5	15.5	20	16
Magnetite	1	0.5	1.5	1.5	trace	0.5
Rutile	-	0.5	-	0.5	0.5	-
Pyrite	14	13.5	14.5	19.5	17	17
Pyrrhotite	4	3	2.5	4.5	3.5	2
Chalcopyrite	-	-	0.5	-	-	trace
Sphalerite	-	-	trace	0.5	-	trace
Arsenopyrite	0.3	-	-	-	trace	trace
Marcasite	-	-	-	trace	-	-
Limonite	-	-	-	trace	trace	0.5

Table 4. Results of Rietveld XRD (Weight Percent) and SEM/EDS Analysis of Whole Tailings and Cycloned Sand

	Whole Tailings (Tailings)				Cycloned Sand (Backfill)		
	#1	#2	#3		#1	#2	#3
Quartz	21	17	20		24	22	20
Albite	15	19	17		19	21	26
Microcline	6	6	6		4	6	4
Biotite	14	16	14		9	9	11
Chlorite	4	5	7		3	4	3
Hornblende							
Sericite	9	11	7		9	7	9
Epidote			X		X		
Calcite	14	13	16		14	15	14
Ankerite	3	3	3		4	4	3
Siderite	2	1	1		2	1	2
Cassiterite	X	X	X				X
Magnetite	X	X	X		X	X	X
Ilmenite	X	X	X		X	X	X
Rutile		X					
Titanite					X	X	X
Pyrite	10	7	7		11	8	7
Pyrrhotite	2	1	1		2	2	1
Chalcopyrite					X		
Sphalerite	X	X	X		X	X	X
Arsenopyrite							
Marcasite							
Galena			X		X		
Zircon						X	X
Apatite		X					
Fe Sulphate			X				
Gypsum	0.7	0.4	0.8		-	-	-

Number: Identified with XRD X: Identified with SEM/EDS analysis

Table 5. Pre-Mining ABA Data for the Waste Rock (Norecol Environmental Consultants Ltd., 1988)

	Paste pH	Total -S (%)	S'ide -S (%)	SO ₄ ^b -S (%)	AP (Tot-S)	AP (S'ide-S)	Sobek -NP	NNP*1	NPR*1	NPR*2
Waste Rock										
	9.4	0.5	--	--	16		106	90	6.6	
	9.3	1.6	--	--	49		105	56	2.1	
	9.0	3.7	--	--	119		132	16	1.1	
	8.8	1.7	--	--	54		141	87	2.6	
	9.8	0.3	--	--	10		138	128	13.8	
	8.7	1.7	--	--	53		167	114	3.2	
	8.8	1.0	--	--	30		153	124	5.1	
	9.4	0.5	--	--	17		98	81	5.8	
	9.9	1.2	--	--	37		114	77	3.1	
	9.2	1.5	--	--	48		166	119	3.5	
	9.0	3.7	--	--	115		157	42	1.4	
	9.0	2.1	--	--	65		109	44	1.7	
Minimum	8.7	0.3			10		98	13	1.1	
Median	9.1	1.6			49		135	84	3.1	
Maximum	9.9	3.8			119		167	128	13.8	
Sulphide Zone Waste Rock										
	8.3	0.3	0.2	0.05	8	6	101	93	12.6	18.0
	8.5	0.3	0.1	0.06	9	3	102	93	11.3	29.7
	8.2	1.9	1.5	0.08	59	46	164	105	2.8	3.6
	8.3	2.6	2.4	0.06	82	75	101	19	1.2	1.4
	N/A	2.5	2.2	0.09	78	70	108	30	1.4	1.5
	8.5	1.9	1.6	0.03	58	51	117	59	2.0	2.3
	8.3	0.6	0.4	0.07	17	11	89	72	5.2	8.1
	8.6	2.6	2.2	0.07	80	70	117	37	1.5	1.7
	8.6	2.7	2.5	0.03	83	77	128	45	1.5	1.7
	8.6	2.5	2.1	0.05	76	65	179	103	2.4	2.8
	8.4	4.5	4.1	0.16	141	129	103	-38	0.7	0.8
	9.1	2.1	1.5	0.18	66	46	120	54	1.8	2.6
	9.0	3.0	2.7	0.05	94	85	111	17	1.2	1.3
	8.7	2.8	2.6	0.11	86	82	124	38	1.4	1.5
	8.8	1.3	1.0	0.20	41	30	83	42	2.0	2.8
Minimum	8.2	0.3	0.1	0.03	8	3	83	-38	0.7	0.8
Median	8.6	2.5	2.1	0.07	76	65	111	45	1.8	2.3
Maximum	9.1	4.5	4.1	0.20	141	129	179	105	12.6	29.7

*1: calculated using AP from total-S and Sobek-NP

*2: calculated using AP from sulphide-S and Sobek-NP

**Table 6. Pre-Mining ABA Data for the Ore and Tailings
(Norecol Environmental Consultants Ltd., 1988)**

	Paste pH	Total -S (%)	S'ide -S (%)	SO ₄ -S (%)	AP (Tot-S)	AP (S'ide-S)	Sobek -NP	NNP *1	NNP *2	NPR *1	NPR *2
Ore											
	9.4	4.4			137		98	-39		0.7	
	8.0	3.4			107		257	150		2.4	
	9.4	5.1			159		156	-3		1.0	
	9.1	4.7			148		176	28		1.2	
	8.7	14.0			441		126	-315		0.3	
	9.0	4.2			130		275	145		2.1	
	9.0	5.8			183		166	-17		0.9	
	9.2	3.6			113		268	155		2.4	
Minimum	8.0	3.4			107		98	-315		0.3	
Median	9.1	4.6			143		171	12		1.1	
Maximum	9.4	14.0			441		275	155		2.4	
Sulphide Zone Ore											
	8.1	6.1	5.9	0.07	191	186	318	127	132	1.7	1.7
	7.7	4.2	3.8	0.13	131	117	60	-71	-57	0.5	0.5
	8.3	1.9	1.7	0.06	58	53	119	61	67	2.1	2.3
	6.9	23.3	24.4	0.45	728	763	15	-713	-748	0.0	0.0
	8.1	7.0	6.5	0.14	219	202	110	-109	-92	0.5	0.5
	8.5	7.5	6.6	0.24	234	207	186	-48	-21	0.8	0.9
	8.2	10.7	11.9	0.05	375	371	86	-289	-285	0.2	0.2
	8.6	6.7	6.3	0.11	208	197	80	-128	-117	0.4	0.4
	8.1	13.2	12.6	0.28	412	394	139	-273	-255	0.3	0.4
Minimum	6.9	1.9	1.7	0.05	58	53	15	-713	-748	0.0	0.0
Median	8.1	7.0	6.5	0.13	219	202	86	-128	-104	0.4	0.5
Maximum	8.6	23.3	24.4	0.45	728	763	318	127	67	2.1	2.3
Tailings											
	8.2	3.0	2.8	0.07	94	88	252	158	164	2.7	2.9
	8.2	4.2	3.5	0.06	131	109	175	44	66	1.3	1.6
	7.9	8.9	7.4	0.23	278	231	142	-136	-89	0.5	0.6
	8.7	4.9	4.0	0.02	153	125	191	38	66	1.2	1.5
	8.4	3.7	2.7	0.03	116	84	197	81	113	1.7	2.3
Minimum	7.9	3.0	2.7	0.02	94	84	142	-136	-89	0.5	0.6
Median	8.2	4.0	3.2	0.05	123	99	183	41	66	1.3	1.6
Maximum	8.7	8.9	7.4	0.23	278	231	252	158	164	2.7	2.9

*1: calculated using AP from total-S and Sobek-NP

*2: calculated using AP from sulphide-S and Sobek-NP

Table 7. ABA Characteristics of Different Site Components (from NDM, 1998)

Material	#	NP				AP				NPR			
		5th	Mean	Med	95th	5th	Mean	Med	95th	5th	Mean	Med	95th
Monthly* Sampling During Operation													
Waste Rock		68	133	128	215	10	62	58	126	1.2	2.2	2.2	15.0
Whole Tails		157	193	186	231	89	132	133	182	1.0	1.5	1.4	2.3
Backfilled Sand		148	187	181	230	89	142	140	222	0.8	1.3	1.3	2.3
Waste Rock Dumps													
130 Dump	8	39	64	65	90	14	33	35	49	1.3	1.9	2.7	6.8
150 Dump	7	64	120	120	173	5.6	23	16	77	1.7	5.2	7.6	19.5
180 Dump	23	124	176	179	216	21	43	38	69	2.5	4.1	4.7	9.1
300 Dump	56	126	177	174	226	33	60	59	83	2.0	3.0	2.9	5.9
440 Dump	21	130	139	136	156	73	100	104	121	1.1	1.4	1.3	2.1
Borrow Pit D	6	-3	4.7	6.5	6	1.4	4.1	2.2	2.5	-2.1	1.5	2.5	2.4
Walls of Underground Mine													
MainMine Walls	47	51	99	103	147	9	51	48	109	0.7	1.9	2.2	10.6
Crown Pillar	5	85	122	109	144	20	64	39	88	1.1	1.9	2.8	7.0
Twin West Walls	16	44	88	91	122	2.3	15	10	34	2.3	6.1	9.5	41.1
Tailings Beach													
93 Tail. Beach		344	358	353	383	180	207	208	227	1.6	1.7	1.7	2.0
94 Tail. Beach		189	205	210	213	104	141	131	197	1.0	1.5	1.6	2.0
95 Tail. Beach	24	167	179	180	189	111	158	158	214	0.8	1.2	1.2	1.6
97 Tail. Beach	21	191	209	208	223	85	115	104	177	1.3	1.9	2.0	2.4

* Mainly monthly, sometimes weekly or quarterly.

Table 8. ABA Characteristics of Portion of Waste Rock Used for Dam Construction in 1989 with Visible Pyrite (Norecol Environmental Consultants Ltd., 1988)

Paste pH	Total -S (%)	Sulphide ^a -S (%)	SO ₄ ^b -S (%)	AP (Tot-S)	AP Side-S	Sobek -NP	CO ₂ %	CO ₂ -NP	NPR -Sob
8.6	1.0	0.7	0.12	30	22	76	3.4	77	3.5
8.5	1.2	0.9	0.12	37	29	71	3	68	2.4
8.4	1.8	1.7	0.04	57	54	39	1.6	36	0.7
8.5	1.2	0.9	0.05	37	29	58	2.5	57	2.0

Table 9. Average ABA Characteristics of Different Size Fractions in the Waste Rock Samples taken from Trenches in the 180 and 300 Dumps

	Dump	180		300		
	Trench	1	2	1	2	3
Total-S	> 25mm	1.23	0.76	1.67	1.45	1.32
	2-25mm	1.17	0.83	1.54	1.88	1.65
	< 2mm	2.86	2.94	2.73	2.74	2.72
SO ₄ -S	> 25mm	0.06	0.06	0.03	0.02	0.01
	2-25mm	0.06	0.05	0.04	0.02	0.02
	< 2mm	0.25	0.14	0.03	0.02	0.01
Sulphide-S	> 25mm	1.17	0.70	1.64	1.43	1.31
	2-25mm	1.11	0.78	1.50	1.86	1.63
	< 2mm	2.61	2.80	2.70	2.72	2.71
AP	> 25mm	37	22	51	45	41
	2-25mm	35	24	47	58	51
	< 2mm	82	88	84	85	85
NP	> 25mm	142	152	168	136	114
	2-25mm	176	182	191	175	169
	< 2mm	218	221	247	244	213
NPR	> 25mm	3.9	6.9	3.3	3.0	2.8
	2-25mm	5.1	7.5	4.1	3.0	3.3
	< 2mm	2.7	2.5	2.9	2.9	2.5

Table 10. ABA Results from Surface Samples Taken from the Tailings Impoundment in 1998 and 1999

Sample I.D.	S (tot) %	Paste pH	NP	MPA	NNP	NPR
1. Tailings July 12 - 18, 1998	2.94	7.72	197.3	92.0	105.3	2.15
2. Geho July 12 - 18, 1998	2.37	7.95	190.3	73.9	116.3	2.57
3. Tailings July 12 - 18, 1998 (Wet)	3.03	7.68	197.2	94.7	102.5	2.08
4. Geho July 12 - 18, 1998 (Wet)	2.26	7.92	196.0	70.7	125.3	2.77
5. Dyke 3, Beach Sample #1, 1998	5.83	7.76	174.5	182.3	-7.7	0.96
6. Dyke 3, Beach Sample #2, 1998	5.35	7.85	188.3	167.1	21.2	1.13
7. Dyke 3, Beach Sample #3, 1998	5.85	7.92	170.9	182.9	-12.0	0.93
8. Dyke 3, Beach Sample #4, 1998	4.99	7.65	152.4	156.0	-3.6	0.98
9. Dyke 3, Beach Sample #5, 1998	7.43	7.69	183.2	232.1	-48.9	0.79
10. Dyke 3, Beach Sample #6, 1998	5.14	7.70	202.1	160.5	41.6	1.26
11. Dyke 3, Beach Sample #7, 1998	4.47	7.80	172.1	139.7	32.4	1.23
12. Dyke 3, Beach Sample #8, 1998	5.34	7.59	168.5	166.7	1.8	1.01
13. Dyke 3, Beach Sample #9, 1998	4.17	7.64	185.0	130.2	54.8	1.42
14. Dyke 3, Beach Sample #10, 1998	4.34	7.80	196.9	135.5	61.4	1.45
15. Dyke 1, Beach Sample #1, 1998	11.95	7.44	204.6	373.4	-168.7	0.55
16. Dyke 1, Beach Sample #2, 1998	6.16	7.76	206.7	192.5	14.2	1.07
17. Geho July 19 - 25, 1998	4.42	7.39	175.4	138.0	37.4	1.27
18. Tailings July 19 - 25, 1998	3.53	6.32	195.1	110.4	84.7	1.77
19. Geho July 19 - 25, 1998 (Wet)	4.45	8.08	178.8	139.0	39.8	1.29
20. Tailings July 19 - 25, 1998 (Wet)	4.56	7.50	202.1	142.5	59.6	1.42
21. Sample #1, July 6, 1999	4.99	7.30	197.3	156.0	41.2	1.26
22. Sample #2, July 6, 1999	3.57	7.08	194.4	111.6	82.7	1.74
23. Sample #3, July 6, 1999	5.93	7.49	202.9	185.2	17.8	1.10
24. Sample #4, July 6, 1999	4.91	7.37	196.3	153.3	43.0	1.28
25. Sample #5, July 6, 1999	3.93	7.51	196.7	122.7	47.0	1.38
26. Sample #6, July 6, 1999	6.55	7.42	223.7	204.7	19.0	1.09

Table 11. Median and 95th Percentile Elemental Composition Results from Operational Characterization of Different Site Components (from NDM, 1998)

		Upper Background	97 Tail. Beach	WasteRock	Cyclone Sand
Al	%		2.3/2.6	2.1/3.3	1.6/1.9
Fe	%		7.5/9.2	5.2/8.0	8.1/10.0
Ca	%		6.3/7.2	4.3/6.2	5.6/6.8
Mg	%		1.8/2.0	1.6/2.6	1.36/1.58
Na	%		0.05/0.05	0.04/0.08	0.02/0.05
K	%		1.3/1.4	1.5/2.1	0.96/1.26
Ba	mg/kg		92/117	187/463	62/141
Sr	mg/kg		265/314	165/348	250/315
Ag	mg/kg	0.1	2.1/2.9	0.8/4.1	2.3/3.7
As	mg/kg	2	474/822	26/73	509/1497
Cd	mg/kg	0.2	5.6/7.2	0.4/ 8.9	6.2/11.7
Cu	mg/kg	90	64/93	186/ 469	109/ 190
Hg	mg/kg	0.1	nm	nm	nm
Mn	mg/kg	1500	1692/ 1838	963/ 1899	1508/ 1921
Mo	mg/kg	2	17/20	4/50	16/30
Ni	mg/kg	200	65/104	16/ 101	66/106
Pb	mg/kg	20	97/143	26/279	103/171
Sb	mg/kg	0.2	nm	nm	nm
Se	mg/kg	0.1	nm	nm	nm
Sn	mg/kg	5	nm	nm	nm
Zn	mg/kg	100	676/792	120/ 1015	593/1016

47/321: median/95th percentile
bold if >5x normal upper background
 nm: not measured
 (range: low-high)

Table 12. Composition of Test Materials from the Pre-Mining Humidity Cell Tests

	S (%)	AP	Sobek-NP	
			Kg CaCO₃/t	NNP
Ore # 1	4.4	137	98	-39
Ore # 5	14.1	441	126	-315
Ore # 7	5.8	183	166	-17
Tailings	8.9	231	142	-89

Table 13. Composition of Test Materials and Resulting Drainage from the 1994 Column Study

	Waste Rock	Whole Tailings	Tailings Sand
Initial Composition			
% Total S	2.55	4.62	3.42
% Sulphide-S	2.44	4.26	3.32
% Sulphate-S	0.11	0.36	0.10
NP	139	182	175
NPR	1.8	1.4	1.7
Paste pH	8.7	7.0	7.3
Drainage chemistry in the last few weeks			
SO ₄ (mg/L)	10-50	900-1200	1550-1700
Alk (mg/L)	< 50	175-210	140-180
pH	7.0-9.0	7.3-7.7	7.0-7.3
Ca (mg/L)	10-45	400-525	500-800
Mg (mg/L)	< 5	20-30	20-35
Na (mg/L)	nd	9-15	< 10
Si (mg/L)	nd	12-14	4-10
Mn (mg/L)	< 0.1	0.1-0.9	2-10
Zn (mg/L)	nd	.02-.08	nd-0.04

nd: not detectable

Table 14. Results from EPA 1312 Weak Acid Leach Tests

		Leachate Concentration					
		440 WRx	440 WRx	440 WRx	150 WRx	150 WRx	150 WRx
Cu	µg/L	21.6	19.8	17.0	2.3	6.3	8.4
Zn	µg/L	4.0	4.0	11.0	1.0	1.5	2.5
As	µg/L	18.0	16.0	10.0	1.0	3.0	4.0
Ni	µg/L	0.8	0.8	1.4	0.2	0.4	0.8
Cd	µg/L	0.050	0.050	0.200	0.050	0.050	0.050
Pb	µg/L	2.0	2.0	6.0	2.0	2.0	14.0
Cr	µg/L	1.5	1.5	2.5	0.5	1.0	0.5
Se	µg/L	4.0	4.0	2.0	1.0	1.0	1.0
Mo	µg/L	1.6	2.3	34.2	0.7	0.6	0.5
Mn	µg/L	16.8	12.5	17.4	17.9	10.3	11.5
Fe	mg/L	0.1	0.1	0.2	0.0	0.0	0.0
Ca	mg/L	8.2	8.0	13.4	11.4	9.4	7.7
Mg	mg/L	0.21	0.21	0.27	1.08	0.87	0.58
Al	mg/L	0.12	0.11	0.12	0.06	0.06	0.12
Na	mg/L	0.15	0.15	0.15	0.70	0.70	0.35
K	mg/L	2.3	2	2.85	4.25	3.2	2.85
SO ₄	mg/L	3.5	3.8	8.5	23.5	15.7	6.1

		Leachate Concentration			Tail Sand	WRx440	WRx150	WRx Mean
		Tail Sand						
		1	2	3				
					Load mg/kg	Load mg/kg	Load mg/kg	Load mg/kg
Cu	µg/L	9.5	3.5	3.7	0.111	0.389	0.113	0.251
Zn	µg/L	3.0	0.6	1.1	0.031	0.127	0.033	0.080
As	µg/L	3.0	5.0	4.0	0.080	0.293	0.053	0.173
Ni	µg/L	10.8	4.8	7.5	0.154	0.020	0.009	0.015
Cd	µg/L	0.025	0.025	0.07	0.0008	0.002	0.001	0.002
Pb	µg/L	3	2	2	0.047	0.067	0.120	0.093
Cr	µg/L	2.8	3.2	3.8	0.065	0.037	0.013	0.025
Se	µg/L	2.2	4.4	2.9	0.063	0.067	0.020	0.043
Mo	µg/L	3.2	3.2	5.4	0.079	0.254	0.012	0.133
Mn	µg/L	44.0	24.8	43.6	0.75	0.31	0.26	0.29
Fe	mg/L	0.41	0.20	0.43	7.0	2.2	0.2	1.2
Ca	mg/L	77.4	47.1	58.6	1220	197	189	193
Mg	mg/L	4.3	2.7	2.6	64	4.6	16.8	10.7
Al	mg/L	0.082	0.064	0.065	1.4	2.3	1.6	1.9
Na	mg/L	0.53	0.32	0.63	9.9	3.0	11.7	7.3
K	mg/L	15.4	11.8	13.0	267	48	69	58
SO ₄	mg/L	158	98	124	2533	105	302	204

WRx: < 2 mm size fraction from waste rock dump
 Backfill: tailings sand used as backfill

Table 15. Particle Size, ABA, NAG, Total Element and Modified-SWEP Test Results for Whole Tailings and Tailings Sand Samples Used in Humidity Cell Tests Initiated in 1998

Screen (mm)	% Passing					
	WT1	WT2	WT3	TS1	TS2	TS3
0.212	99.6	99.9	99.7	99.7	99.8	90.7
0.150	98.7	99.1	98.9	97.5	98.4	83.7
0.088	93.5	90.7	97.4	72.2	76.8	67.6
0.075	90.4	90.3	96.5	60.4	61.5	56.5
0.053	81.3	87.8	92.6	37.4	37.6	42.4

		WT			WT	TS			TS
		1	2	3	Average	1	2	3	Average
Paste pH		7.6	7.7	7.7	7.7	7.5	7.7	7.6	7.6
CO ₂	%	7.81	6.64	7.89	7.45	7.56	8.3	7.35	7.74
T-S	%	5.95	4.43	4.60	4.99	6.35	5.73	4.74	5.61
Sulphate-S	%	0.15	0.13	0.14	0.14	0.06	0.04	0.06	0.05
Sulphide-S	%	5.8	4.3	4.46	4.85	6.29	5.69	4.68	5.55
AP	kg/t	181	134	139	152	197	178	146	174
Sobek-NP	kg/t	219	209	239	222	204	228	209	214
CO ₂ -NP	kg/t	178	151	179	169	172	189	167	176
Sob-NPR		1.21	1.55	1.71	1.49	1.04	1.28	1.43	1.25
CO ₂ -NPR		0.98	1.12	1.29	1.13	0.87	1.06	1.14	1.03
NAG pH		3.3		7.2	5.2	2.6	5.3	4.5	4.1
H ₂ SO ₄	kg/t	2.90				18.03			
Total Metals									
As	ppm	1170	226	581	659	574	770	597	647
Cu	ppm	144	159	540	281	82	64	202	116
Ba	ppm	144	141	189	158	87	118	134	113
Bi	ppm	19	9	16	15	12	14	7	11
Cd	ppm	12	7	18	12	7	9	8	8
Co	ppm	57	35	38	43	59	58	44	54
Cr	ppm	32	41	43	39	33	28	29	30
La	ppm	11	13	10	11	8	8	8	8
Mn	ppm	1960	1595	1788	1781	1594	1636	1493	1574
Mo	ppm	21	13	27	20	15	17	26	19
Ni	ppm	103	89	83	92	114	105	63	94
Pb	ppm	218	148	228	198	113	120	148	127
Sb	ppm	8	4	5	6	4	6	6	5
Sr	ppm	279	242	278	266	249	281	276	269
V	ppm	122	128	149	133	77	82	113	91
Zn	ppm	1183	694	1719	1199	632	750	779	720
Al	%	2.1	2.3	2.5	2.3	1.3	1.4	1.6	1.4
Ca	%	5.6	5.0	6.0	5.5	5.1	5.9	5.6	5.5
Fe	%	10.4	8.9	9.0	9.4	9.9	8.6	8.0	8.8
K	%	1.3	1.4	1.3	1.3	0.8	0.8	1.0	0.9

		WT			WT	TS			TS
		1	2	3	Average	1	2	3	Average
Mg	%	1.7	1.8	2.1	1.9	1.1	1.3	1.3	1.2
Na	%	0.02	0.02	0.03	0.02	0.02	0.02	0.02	0.02
P	%	0.11	0.12	0.12	0.12	0.11	0.10	0.14	0.12
Ti	%	0.17	0.20	0.18	0.18	0.10	0.11	0.14	0.12
Modified SWEP									
Ba	ppb	42	46	44	38	31	40	36	
Ca	ppm	164	144	154	74	46	58	59	
K	ppm	17	18	17	12	10	11	11	
Mg	ppm	6.3	6.3	6.3	3.6	2.4	2.3	2.8	
Mn	ppm	0.19	0.09	0.14	0.02	0.01	0.03	0.02	
Mo	ppb	11	10	11	6	5	8	6	
Na	ppm	0.8	1.2	1.0	0.5	0.3	0.6	0.5	
Si	ppm	0.43	0.53	0.48	0.23	0.41	0.39	0.34	

TS: Tailings Sand WT: Whole Tailings

Modified SWEP results below detection limit: Al < 0.1, Fe < 0.01, Li < 0.05 and P < 0.02 ppm; Ag < 5, As < 30, B < 20, Be < 2, Bi < 20, Cd < 2, Ce < 30, Co < 5, Cr < 20, Cu < 2, Ni < 20, Pb < 10, Sb < 10, Se < 20, Te, Ti, Tl, V, W < 10, and Zn < 5 ppb

Table 16. Drainage Loading Results for Ongoing Humidity Cells

		Whole Tailings			Tail NP dep		Tail	Tailings Sand				TS NP dep		TS	WR Dump		WR
		1	2	3	1	2	Ave	1	2	3	300L	1	2	Ave	150	180	Ave
Ca+Mg/SO₄ (molar ratio)																	
overall	95th	1.4	1.4	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.4	1.3	1.3	2.0	1.5	1.7
	average	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.5	1.2	1.3
	5th	0.9	0.9	0.9	0.9	0.9	0.9	0.8	0.9	0.9	0.9	0.9	0.9	0.9	1.1	1.0	1.1
wk 10-30	max	1.3	1.3	1.3	1.2	1.2	1.3	1.3	1.2	1.1	1.1	1.3	1.1	1.2	1.8	1.1	1.5
	average	1.1	1.1	1.2	1.0	1.0	1.1	1.1	1.1	1.1	1.1	1.1	1.0	1.1	1.5	1.1	1.3
	min	1.0	1.0	1.1	0.3	0.9	0.9	1.0	1.0	0.9	1.0	0.9	0.9	0.9	1.3	1.0	1.1
last 24 wks	max	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.5	1.3	1.4
	average	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.3	1.2	1.2
	min	0.9	0.9	0.9	0.9	0.9	0.9	0.9	1.0	0.9	0.9	0.9	0.9	0.9	1.1	1.0	1.0
SO₄ (mg/kg/wk)																	
overall	95th	215	210	195	364	191	235	309	215	201	430	261	148	260	53	159	106
	average	104	104	88	135	93	105	164	119	103	191	159	90	138	28	88	58
	5th	30	28	30	32	24	29	88	47	42	78	83	40	63	11	28	19
wk 10-30	max	103	117	134	225	137	143	233	267	277	430	249	111	261	17	98	57
	average	76	67	64	135	94	87	131	94	102	176	135	87	121	13	73	43
	min	57	45	30	51	41	45	57	27	35	105	67	55	58	11	49	30
last 24 wks	max	213	205	334	282	167	240	137	164	125	114	147	116	134	50	150	100
	average	114	160	151	178	140	149	118	129	94	88	122	94	108	41	94	67
	min	75	122	92	58	116	92	92	92	59	58	94	63	76	30	59	45
pH																	
overall	95th	8.0	8.0	8.1	7.9	7.9	8.0	7.8	8.0	8.1	7.8	7.7	7.9	7.9	8.5	8.4	8.5
	average	7.6	7.7	7.8	7.6	7.7	7.7	7.7	7.8	7.8	7.6	7.5	7.6	7.7	8.0	7.9	8.0
	5th	7.3	7.5	7.6	7.4	7.4	7.4	7.5	7.5	7.5	7.4	7.3	7.3	7.4	7.6	7.6	7.6
wk 10-30	max	7.7	7.9	7.9	7.9	7.9	7.9	7.7	7.8	7.9	7.8	7.7	7.9	7.8	8.4	8.4	8.4
	average	7.5	7.7	7.7	7.6	7.7	7.7	7.6	7.7	7.7	7.6	7.5	7.6	7.6	8.0	8.0	8.0
	min	7.1	7.4	7.3	7.4	7.5	7.3	7.4	7.5	7.5	7.3	7.3	7.4	7.4	7.3	7.3	7.3
last 24 wks	max	7.8	7.7	7.8	7.8	7.6	7.8	7.8	7.8	7.8	7.8	7.6	8.0	7.8	8.2	7.9	8.0
	average	7.7	7.6	7.8	7.5	7.6	7.6	7.7	7.7	7.8	7.7	7.5	7.6	7.7	7.9	7.7	7.8
	min	7.6	7.6	7.7	7.3	7.6	7.6	7.6	7.7	7.7	7.6	7.4	7.4	7.6	7.7	7.6	7.7
Alkalinity (mg/kg/wk)																	
overall	95th	7.5	8.4	7.8	9.2	8.1	8.2	9.3	8.7	9.9	14.6	8.0	8.3	9.8	13.4	25.4	19.4
	average	4.7	4.8	5.2	4.6	4.2	4.7	5.9	5.8	5.9	6.4	5.0	5.0	5.7	9.8	12.4	11.1
	5th	1.8	2.1	2.8	2.7	1.8	2.2	3.4	3.6	3.4	3.2	2.9	2.8	3.2	7.7	6.8	7.3
wk 10-30	max	7.5	7.0	8.0	8.3	8.8	7.9	8.7	9.9	9.2	17.3	5.0	9.1	9.9	11.2	8.8	10.0
	average	5.8	6.0	6.7	4.3	5.6	5.7	5.7	5.8	7.2	6.8	4.4	6.7	6.1	9.5	7.8	8.7
	min	4.8	5.3	5.4	2.3	2.6	4.1	3.8	3.8	4.6	4.3	3.5	4.6	4.1	8.2	6.3	7.2
last 24 wks	max	2.9	2.5	5.3	3.3	2.7	3.3	4.9	4.3	5.6	4.9	4.3	4.4	4.7	11.3	15.9	13.6
	average	2.5	2.1	3.1	2.8	2.3	2.6	4.2	3.7	4.5	4.0	3.6	3.6	3.9	9.8	14.2	12.0
	min	2.1	1.9	2.4	2.4	1.7	2.1	3.6	3.4	3.5	3.5	3.1	2.9	3.3	8.6	11.7	10.2
Ca (mg/kg/wk)																	
overall	95th	65	65	69	128	74	80	97	69	73	148	97	52	89	19	65	42
	average	33	32	27	46	30	34	53	37	35	68	52	31	46	10	37	23
	5th	11	10	10	11	8	10	31	14	15	27	26	15	21	5	15	10
wk 10-30	max	42	35	58	63	49	49	97	92	97	155	67	39	91	7	39	23
	average	26	20	22	41	27	27	45	30	33	68	40	27	41	5	31	18
	min	17	11	11	11	7	12	21	8	12	37	21	17	19	5	20	12
last 24 wks	max	58	56	109	78	45	69	39	40	41	38	38	45	40	17	53	35
	average	31	40	51	44	38	41	36	31	30	28	33	31	32	13	36	25
	min	23	28	26	17	28	25	33	22	19	20	24	21	23	10	24	17

		Whole Tailings			Tail NP dep		Tail	Tailings Sand				TS NP dep		TS	WR Dump		WR
		1	2	3	1	2	Ave	1	2	3	300L	1	2	Ave	150	180	Ave
Mg (mg/kg/wk)																	
overall	95th	18.7	17.7	20.0	22.1	14.2	18.5	28.7	20.8	12.4	23.8	24.7	10.9	20.2	7.1	8.8	7.9
	average	9.0	8.4	7.9	10.0	6.5	8.3	12.9	10.8	7.0	10.7	12.2	6.8	10.1	3.7	3.8	3.8
	5th	2.4	2.4	2.5	2.4	1.5	2.2	6.3	3.6	3.4	3.1	5.2	3.1	4.1	1.5	0.7	1.1
wk 10-30	max	8.1	10.7	12.6	14.2	9.5	11.0	13.0	20.8	14.6	22.3	17.2	7.3	15.9	1.8	1.2	1.5
	average	6.3	6.3	5.9	8.1	5.7	6.5	8.3	7.6	6.8	6.4	9.9	5.3	7.4	1.6	0.9	1.2
	min	4.2	4.0	2.2	3.4	2.7	3.3	4.5	1.5	2.7	2.7	4.2	3.4	3.2	1.3	0.6	1.0
last 24 wks	max	20.3	23.3	20.2	22.1	14.4	20.1	9.7	20.7	8.6	8.2	17.1	8.5	12.2	6.6	8.0	7.3
	average	10.2	15.8	8.7	13.2	11.2	11.8	8.2	14.1	6.4	6.3	10.0	6.0	8.5	5.1	5.1	5.1
	min	4.6	12.8	5.3	3.2	8.9	7.0	5.9	9.2	4.2	4.1	5.7	4.4	5.6	3.6	3.0	3.3
K (mg/kg/wk)																	
overall	95th	7.3	7.0	6.5	7.9	8.0	7.3	10.1	7.7	7.1	9.0	8.2	7.4	8.3	3.9	7.0	5.5
	average	3.5	3.7	3.3	3.8	3.2	3.5	5.5	4.1	3.7	4.9	4.8	3.5	4.4	2.6	4.5	3.5
	5th	1.1	1.2	1.4	1.0	0.8	1.1	1.9	1.8	1.5	1.3	1.5	1.2	1.5	1.7	2.5	2.1
wk 10-30	max	4.4	5.5	5.6	11.7	9.6	7.4	9.6	7.7	7.7	24.2	8.2	9.4	11.1	3.6	5.7	4.6
	average	2.9	2.9	3.2	5.3	5.1	3.9	5.5	4.3	5.4	7.2	5.9	5.5	5.6	3.0	4.4	3.7
	min	1.8	1.9	1.5	2.0	1.8	1.8	2.1	2.0	1.5	3.0	3.8	3.9	2.7	2.5	3.5	3.0
last 24 wks	max	3.5	6.0	5.3	2.4	1.7	3.8	2.4	2.5	2.2	1.9	1.8	1.8	2.1	2.6	6.0	4.3
	average	2.1	4.3	2.6	1.8	1.5	2.5	2.0	2.1	1.8	1.4	1.5	1.5	1.7	2.0	3.8	2.9
	min	1.5	3.1	1.8	0.9	1.0	1.7	1.6	1.5	1.2	1.0	1.1	1.2	1.3	1.2	2.7	1.9
Na (mg/kg/wk)																	
overall	95th	0.9	1.3	1.4	0.9	1.0	1.1	0.8	0.5	0.6	0.7	0.6	0.5	0.6	0.3	0.4	0.3
	average	0.4	0.5	0.5	0.4	0.5	0.5	0.3	0.3	0.3	0.2	0.3	0.3	0.3	0.2	0.3	0.2
	5th	0.1	0.2	0.2	0.1	0.1	0.1	0.1	0.1	0.1	0.0	0.1	0.1	0.1	0.1	0.1	0.1
wk 10-30	max	0.5	0.9	0.9	0.7	1.0	0.8	0.4	0.4	0.7	1.5	0.4	0.4	0.6	0.2	0.3	0.2
	average	0.3	0.4	0.4	0.4	0.6	0.4	0.2	0.2	0.4	0.3	0.2	0.2	0.3	0.1	0.2	0.2
	min	0.1	0.2	0.1	0.2	0.2	0.2	0.1	0.1	0.1	0.0	0.1	0.2	0.1	0.1	0.1	0.1
last 24 wks	max	1.3	1.3	1.3	0.8	1.0	1.1	0.4	0.4	0.4	0.3	0.4	0.3	0.4	0.2	0.3	0.3
	average	0.5	0.9	0.5	0.5	0.8	0.7	0.3	0.3	0.3	0.1	0.3	0.3	0.3	0.2	0.2	0.2
	min	0.2	0.6	0.3	0.1	0.5	0.3	0.2	0.2	0.2	0.1	0.2	0.2	0.2	0.1	0.1	0.1
As (ug/kg/wk)																	
overall	95th	10.5	13.1	13.3	12.5	11.1	12.1	9.8	10.7	11.4	14.4	11.1	11.1	11.4	13.8	14.7	14.2
	average	5.7	6.2	7.0	6.4	5.4	6.2	6.5	6.9	7.4	7.2	6.9	7.0	7.0	11.7	12.3	12.0
	5th	2.5	2.6	3.9	3.6	2.5	3.0	3.8	3.9	3.8	3.6	3.9	4.5	3.9	10.4	10.4	10.4
wk 10-30	max	14.0	14.0	17.4	9.6	9.5	12.9	10.2	13.8	20.5	16.2	10.1	10.5	13.5	12.8	11.6	12.2
	average	7.2	7.8	9.1	5.0	6.3	7.1	7.2	7.6	9.6	7.9	7.5	7.8	7.9	11.3	11.0	11.2
	min	5.1	5.7	5.9	2.7	3.0	4.5	3.9	3.9	6.0	5.3	5.9	5.6	5.1	10.5	10.5	10.5
last 24 wks	max	3.9	3.2	7.2	4.5	2.9	4.3	4.2	4.1	4.8	4.2	4.1	5.1	4.4	12.6	15.9	14.3
	average	3.4	2.5	4.8	3.7	2.6	3.4	3.9	3.8	4.4	3.8	3.8	5.1	4.1	12.4	13.3	12.9
	min	3.0	2.1	3.9	3.0	2.0	2.8	3.5	3.2	4.1	3.5	3.5	5.1	3.8	12.2	10.1	11.1
Cd (ug/kg/wk)																	
overall	95th	0.7	0.8	0.8	0.9	0.7	0.8	0.7	0.7	0.8	1.0	0.8	0.7	0.8	0.9	1.0	0.9
	average	0.4	0.4	0.5	0.4	0.4	0.4	0.4	0.5	0.5	0.5	0.5	0.5	0.5	0.8	0.8	0.8
	5th	0.2	0.2	0.3	0.2	0.1	0.2	0.2	0.3	0.3	0.2	0.3	0.3	0.3	0.7	0.7	0.7
wk 10-30	max	0.5	0.6	0.6	0.6	0.6	0.6	0.7	0.7	0.7	1.1	0.7	0.7	0.8	0.9	0.8	0.8
	average	0.4	0.5	0.5	0.3	0.4	0.4	0.5	0.5	0.6	0.5	0.5	0.5	0.5	0.8	0.7	0.7
	min	0.3	0.4	0.4	0.2	0.2	0.3	0.3	0.3	0.4	0.4	0.4	0.4	0.3	0.7	0.7	0.7
last 24 wks	max	0.3	0.2	0.7	0.3	0.3	0.4	0.3	0.3	0.3	0.3	0.3	0.5	0.3	3.3	1.1	2.2
	average	0.2	0.2	0.4	0.2	0.2	0.2	0.3	0.3	0.3	0.3	0.3	0.4	0.3	1.2	0.9	1.1
	min	0.2	0.1	0.3	0.2	0.1	0.2	0.2	0.2	0.3	0.2	0.2	0.3	0.3	0.8	0.7	0.7

Cr (ug/kg/wk)																	
overall	95th	6.7	8.6	8.1	8.3	7.4	7.8	6.6	6.8	7.6	9.6	6.9	7.4	7.5	9.1	9.8	9.4
	average	3.7	4.1	4.5	4.2	3.6	4.0	4.2	4.5	4.8	4.7	4.5	4.6	4.6	7.7	8.2	8.0
	5th	1.7	1.7	2.6	2.4	1.5	2.0	2.4	2.6	2.5	2.4	2.6	3.0	2.6	6.9	6.9	6.9
wk 10-30	max	5.2	6.3	5.8	6.4	6.3	6.0	6.8	7.3	6.8	10.8	6.5	7.0	7.5	8.5	7.7	8.1
	average	4.3	4.7	4.9	3.3	4.2	4.3	4.7	4.5	5.5	5.3	5.0	5.2	5.0	7.5	7.4	7.4
	min	3.2	3.8	3.9	1.8	2.0	2.9	2.6	2.6	4.0	3.5	3.9	3.7	3.4	7.0	7.0	7.0
last 24 wks	max	2.6	2.1	4.8	3.0	1.9	2.9	2.8	2.7	3.2	2.8	2.7	3.4	2.9	8.4	10.6	9.5
	average	2.3	1.7	3.2	2.5	1.7	2.3	2.6	2.5	3.0	2.5	2.6	3.4	2.8	8.3	8.9	8.6
	min	2.0	1.4	2.6	2.0	1.3	1.9	2.3	2.1	2.7	2.3	2.3	3.4	2.5	8.1	6.7	7.4
Cu (ug/kg/wk)																	
overall	95th	1.8	1.6	1.7	1.4	1.1	1.5	1.2	1.2	1.3	1.9	1.5	1.3	1.4	3.2	4.3	3.7
	average	0.6	0.6	0.7	0.6	0.5	0.6	0.6	0.7	0.7	0.7	0.6	0.6	0.6	1.2	1.8	1.5
	5th	0.2	0.2	0.3	0.2	0.2	0.2	0.2	0.3	0.3	0.3	0.3	0.3	0.3	0.7	0.8	0.7
wk 10-30	max	2.5	3.4	3.4	0.8	1.1	2.3	1.3	4.6	2.8	2.7	2.5	1.9	2.6	2.1	3.6	2.9
	average	1.1	1.3	1.2	0.5	0.6	0.9	0.8	1.0	1.2	1.0	0.7	0.8	0.9	1.1	2.0	1.6
	min	0.3	0.4	0.4	0.3	0.3	0.3	0.3	0.3	0.4	0.4	0.4	0.4	0.4	0.7	1.1	0.9
last 24 wks	max	0.4	0.7	1.0	0.5	0.5	0.6	0.4	0.3	0.6	0.4	0.4	2.6	0.8	2.1	2.7	2.4
	average	0.3	0.3	0.4	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.3	0.7	0.4	1.2	1.2	1.2
	min	0.2	0.2	0.3	0.2	0.1	0.2	0.2	0.2	0.3	0.2	0.2	0.3	0.3	0.8	0.7	0.8
Mn (ug/kg/wk)																	
overall	95th	96	110	104	224	159	139	179	145	198	344	196	137	200	18	10	14
	average	45	51	45	83	46	54	73	58	71	119	85	59	77	10	5	8
	5th	10	15	16	12	7	12	16	18	13	13	16	19	16	4	4	4
wk 10-30	max	76	79	92	378	153	155	190	314	214	552	196	130	266	12	4	8
	average	56	53	55	183	78	85	131	93	100	174	150	80	121	8	4	6
	min	37	38	33	35	15	32	72	30	41	72	113	51	63	4	4	4
last 24 wks	max	24	48	77	45	37	46	26	21	18	18	23	44	25	8	14	11
	average	18	33	38	33	30	30	18	18	13	12	20	29	18	6	7	6
	min	16	26	25	12	23	20	14	16	8	8	13	22	14	4	3	4
Mo (ug/kg/wk)																	
overall	95th	6.3	6.7	6.7	5.8	5.7	6.3	3.0	3.1	5.8	2.8	2.5	6.0	3.9	2.3	12.7	7.5
	average	3.6	3.7	3.6	3.2	3.4	3.5	1.4	1.6	3.4	1.4	1.4	3.0	2.0	2.0	5.9	4.0
	5th	1.7	1.6	1.7	1.7	1.5	1.6	0.6	0.7	1.9	0.6	0.7	1.3	1.0	1.7	2.0	1.8
wk 10-30	max	4.9	5.0	6.8	4.8	8.4	6.0	4.0	5.1	6.6	2.9	2.6	7.0	4.7	2.1	4.8	3.5
	average	3.0	2.6	3.0	3.6	4.4	3.3	1.9	2.3	4.2	1.5	1.8	5.2	2.8	1.9	3.8	2.9
	min	1.0	1.1	1.0	2.0	1.9	1.4	0.7	0.9	1.6	0.9	1.2	3.6	1.5	1.8	2.9	2.3
last 24 wks	max	7.8	6.8	10.1	4.7	5.0	6.9	0.7	0.7	3.0	0.7	0.7	1.9	1.3	2.1	13.9	8.0
	average	4.5	6.2	5.2	2.9	4.0	4.6	0.7	0.6	2.6	0.6	0.6	1.6	1.1	2.1	9.3	5.7
	min	3.0	5.4	3.4	1.5	2.8	3.2	0.6	0.5	2.0	0.6	0.6	1.2	0.9	2.0	6.0	4.0
Ni (ug/kg/wk)																	
overall	95th	4.7	5.8	5.5	5.9	6.3	5.7	6.1	5.5	6.8	9.1	6.6	6.9	6.8	8.4	7.8	8.1
	average	2.3	2.4	2.5	2.2	2.1	2.3	2.7	2.7	3.0	4.0	2.6	2.7	3.0	4.0	4.2	4.1
	5th	0.4	0.4	0.7	0.6	0.4	0.5	0.6	0.7	0.6	0.7	0.7	0.8	0.7	1.8	1.8	1.8
wk 10-30	max	5.2	6.3	5.8	6.4	6.3	6.0	6.8	7.3	6.8	26.9	6.5	7.0	10.2	8.5	7.7	8.1
	average	4.3	4.7	4.9	3.3	4.2	4.3	4.8	4.6	5.5	7.5	5.0	5.2	5.4	7.5	7.4	7.4
	min	3.2	3.8	3.9	1.8	2.0	2.9	2.6	3.0	4.0	4.0	3.9	3.7	3.5	7.0	7.0	7.0
last 24 wks	max	0.7	0.5	1.2	0.8	0.5	0.7	0.8	0.7	0.8	1.3	0.7	0.9	0.8	2.1	2.7	2.4
	average	0.6	0.4	0.8	0.6	0.4	0.6	0.7	0.6	0.7	0.8	0.6	0.9	0.7	2.1	2.2	2.1
	min	0.5	0.4	0.7	0.5	0.3	0.5	0.6	0.5	0.7	0.6	0.6	0.9	0.6	2.0	1.7	1.9
Pb (ug/kg/wk)																	
overall	95th	3.4	4.0	4.1	4.2	3.7	3.8	3.5	3.4	3.8	4.8	3.5	3.7	3.8	4.5	4.9	4.7
	average	1.9	2.0	2.3	2.1	1.8	2.0	2.2	2.2	2.4	2.4	2.3	2.3	2.3	3.9	4.1	4.0
	5th	0.8	0.9	1.3	1.2	0.7	1.0	1.2	1.3	1.3	1.2	1.3	1.5	1.3	3.5	3.5	3.5

wk 10-30	max	2.6	3.2	2.9	3.2	3.2	3.0	6.8	3.7	3.4	5.4	3.3	3.5	4.3	4.3	3.9	4.1
	average	2.1	2.4	2.5	1.7	2.1	2.1	2.7	2.3	2.8	2.6	2.5	2.6	2.6	3.8	3.7	3.7
	min	1.6	1.9	2.0	0.9	1.0	1.5	1.3	1.3	2.0	1.8	2.0	1.9	1.7	3.5	3.5	3.5
last 24 wks	max	1.3	1.1	2.4	1.5	1.0	1.4	1.4	1.4	1.6	1.4	1.4	1.7	1.5	4.2	5.3	4.8
	average	1.1	0.8	1.6	1.2	0.9	1.1	1.3	1.3	1.5	1.3	1.3	1.7	1.4	4.1	4.5	4.3
	min	1.0	0.7	1.3	1.0	0.7	0.9	1.2	1.1	1.4	1.2	1.2	1.7	1.3	4.1	3.4	3.7
Se (ug/kg/wk)																	
overall	95th	14.3	9.6	9.2	10.7	7.6	10.3	12.3	10.5	9.0	9.6	11.7	8.2	10.2	9.1	9.8	9.4
	average	6.1	5.3	5.4	5.7	4.2	5.3	6.2	5.5	5.2	5.0	6.1	4.8	5.5	7.8	8.2	8.0
	5th	1.8	2.2	2.6	2.9	1.8	2.2	2.7	2.9	2.6	2.4	2.8	3.0	2.7	6.9	6.9	6.9
wk 10-30	max	9.6	9.5	8.4	9.6	9.8	9.4	8.6	13.5	9.9	10.8	7.8	10.4	10.2	8.5	7.7	8.1
	average	6.6	5.9	5.7	5.5	4.8	5.7	6.0	6.2	6.4	5.4	5.4	5.6	5.8	7.5	7.4	7.4
	min	4.3	3.8	3.9	2.9	2.0	3.4	2.7	3.0	4.0	3.5	3.9	3.7	3.5	7.0	7.0	7.0
last 24 wks	max	4.3	6.8	5.0	7.0	7.2	6.1	2.8	3.6	3.1	2.8	5.2	3.4	3.5	8.4	10.6	9.5
	average	2.6	4.7	3.2	5.0	4.6	4.0	2.6	3.0	2.9	2.6	3.7	3.4	3.0	8.3	8.9	8.6
	min	2.0	4.0	2.6	2.1	3.6	2.9	2.4	2.6	2.7	2.3	2.5	3.4	2.7	8.1	6.7	7.4
Zn (ug/kg/wk)																	
overall	95th	8.0	6.4	10.4	11.7	6.1	8.5	3.8	6.6	12.7	15.5	5.7	5.8	8.3	4.5	4.5	4.5
	average	2.5	2.4	3.4	4.0	2.2	2.9	3.8	2.3	3.3	5.2	2.9	2.7	3.4	2.3	2.6	2.4
	5th	0.7	0.6	0.8	0.7	0.5	0.7	0.7	0.9	1.0	1.2	1.0	1.1	1.0	1.7	1.8	1.7
wk 10-30	max	1.8	1.6	1.9	6.7	4.8	3.3	2.5	9.4	4.7	12.0	4.5	5.4	6.4	12.8	5.0	8.9
	average	1.1	1.2	1.3	3.6	2.2	1.9	1.4	2.3	1.9	3.5	3.1	3.1	2.6	2.9	2.1	2.5
	min	0.8	1.0	1.0	1.2	0.9	1.0	0.9	0.8	1.0	1.6	1.6	1.7	1.3	1.8	1.8	1.8
last 24 wks	max	4.2	6.5	21.1	5.1	2.3	7.8	3.0	6.4	20.0	3.1	4.7	6.1	7.2	5.0	2.7	3.8
	average	2.4	2.7	7.5	2.9	1.4	3.4	2.3	3.1	4.4	2.0	2.7	3.0	2.9	3.1	2.2	2.7
	min	0.9	1.4	3.2	0.7	0.7	1.4	1.4	1.3	0.8	1.1	1.4	1.5	1.3	2.0	1.7	1.9

Table 17. Volume of Drainage Collected from Ongoing Humidity Cells after Washing with 500 mL

	# of wks	Overall (ml)				Average (ml)	
		min	average	median	max	wk 10-30	last 24 wk
Whole Tail #1	292	50	195	195	535	219	150
Whole Tail #2	292	55	214	200	505	238	113
Whole Tail #3	292	105	222	200	490	251	230
Whole Tail # 1 -NP	272	90	207	171	485	158	173
Whole Tail # 2 -NP	272	60	194	180	480	201	100
Tail Sand #1	288	110	215	210	482	231	143
Tail Sand #2	288	25	220	210	465	224	168
Tail Sand #3	288	115	247	255	490	284	156
Tail Sand 300L	282	105	234	200	540	258	147
Tail Sand #1 -NP	272	115	235	230	480	241	153
Tail Sand #2 -NP	272	110	248	255	485	266	170
WR 150	273	320	388	373	530	384	438
WR 180	273	310	403	380	530	374	438

Table 18. Concentration of Trace Metals in Drainage from Ongoing Humidity Cells

	SO ₄ mg/L	Alk mg/L	Al ppm <0.1	As ppb < 30	Cd ppb < 2	Cr ppb < 20	Cu ppb < 2	Fe ppm < 0.01	Mo ppb < 5	Ni ppb < 5	Pb ppb < 10	Sb ppb < 10	Se ppb < 20	Zn ppb < 5
Detection Limit														
Whole Tail #1														
Median	558	25	<0.1	< 30	< 2	< 20	3	0.02	20	<20, 5	< 10	< 10	< 20	11
maximum	1776	38	0.10	70	7	< 20	26	0.66	65	21	13	30	105	172
average for wk 10-30	360	28	<0.1	45	2	< 20	5	0.02	14	< 20	< 10	27	33	9
average for last 24	876	24	<0.1	< 30	< 2	< 20	8	0.41	37	< 5	< 10	< 10	36	20
Whole Tail #2														
Median	480	24	<0.1	< 30	< 2	< 20	3	0.02	18	<20, 5	< 10	< 10	< 20	12
maximum	2410	35	0.10	56	4	< 20	15	1.35	97	18	10	49	109	148
average for wk 10-30	284	26	<0.1	46	2	< 20	7	0.02	12	< 20	< 10	15	27	8
average for last 24	1539	25	0.10	< 30	< 2	< 20	4	0.64	45	< 5	< 10	< 10	36	20
Whole Tail #3														
Median	366	25	<0.1	< 30	< 2	< 20	< 2	< 0.01	16	<20, 5	< 10	< 10	< 20	15
maximum	1496	35	0.10	73	4	< 20	45	1.21	44	7	11	26	120	134
average for wk 10-30	255	27	<0.1	56	2	< 20	5	0.04	12	< 20	< 10	23	29	6
average for last 24	1277	22	<0.1	< 30	3	< 20	13	0.34	40	< 5	< 10	< 10	22	80
Whole Tail # 1 -NP														
Median	624	22	<0.1	< 30	< 2	< 20	< 2	< 0.01	16	<20, 5	< 10	< 10	< 20	17
maximum	2000	36	0.10	45	4	< 20	8	1.18	37	10	< 10	67	76	103
average for wk 10-30	776	26	<0.1	< 30	2	< 20	4	0.02	23	< 20	< 10	< 10	41	22
average for last 24	868	22	<0.1	< 30	2	< 20	3	0.31	19	< 5	< 10	< 10	26	18
Whole Tail # 2 -NP														
Median	430	24	<0.1	< 30	< 2	< 20	3	0.03	20	<20, 5	< 10	< 10	< 20	9
maximum	1920	45	0.10	59	3	< 20	8	1.37	55	8	10	54	80	137
average for wk 10-30	426	25	<0.1	< 30	< 2	< 20	3	0.01	21	< 20	< 10	< 10	25	11
average for last 24	1395	24	<0.1	< 30	3	< 20	3	0.47	40	< 5	< 10	< 10	38	16
Tail Sand #1														
Median	870	28	<0.1	< 30	< 2	< 20	< 2	0.02	7	<20, 5	17	< 10	< 20	10
maximum	2050	48	0.20	62	3	< 20	11	1.20	26	50	22	41	108	430
average for wk 10-30	579	25	<0.1	34	3	< 20	4	0.01	9	28	22	20	34	7
average for last 24	831	32	0.20	< 30	< 2	< 20	3	0.21	< 5	< 5	< 10	< 10	27	12
Tail Sand #2														
median	527	26	0.10	< 30	< 2	< 20	< 2	0.02	7	<20, 5	< 10	< 10	< 20	10
maximum	1412	41	0.10	55	3	< 20	20	0.92	28	46	< 10	28	71	53
average for wk 10-30	406	27	<0.1	43	3	< 20	5	0.02	11	30	< 10	17	33	17
average for last 24	704	26	<0.1	< 30	< 2	< 20	3	0.16	< 5	< 5	< 10	< 10	< 20	13
Tail Sand #3														
median	430	25	0.10	< 30	< 2	< 20	< 2	< 0.01	15	<20, 5	< 10	< 10	< 20	10
maximum	1256	41	0.10	89	3	< 20	11	1.42	30	28	< 10	47	47	133
average for wk 10-30	370	26	<0.1	60	3	< 20	5	0.02	15	< 20	< 10	15	33	9
average for last 24	496	25	<0.1	< 30	< 2	< 20	2	0.14	15	< 5	< 10	< 10	< 20	28
Tail Sand 300L														
median	830	26	0	35	< 2	< 20	< 2	< 0.01	6	<20, 5	< 10	< 10	< 20	17
maximum	1523	41	0	66	4	< 20	13	1.71	11	60	< 10	38	39	200
average for wk 10-30	637	25	<0.1	< 30	3	< 20	5	0.01	6	29	< 10	18	25	12
average for last 24	791	31	<0.1	< 30	2	< 20	3	0.18	< 5	7	< 10	< 10	< 20	21

	SO₄ mg/L	Alk mg/L	Al ppm <0.1	As ppb <30	Cd ppb <2	Cr ppb <20	Cu ppb <2	Fe ppm <0.01	Mo ppb <5	Ni ppb <5	Pb ppb <10	Sb ppb <10	Se ppb <20	Zn ppb <5
Detection Limit														
Tail Sand #1 -NP														
median	688	22	0.15	<30	<2	<20	<2	<0.01	7	<20, 5	<10	<10	30	13
maximum	1496	35	0.50	44	3	<20	14	1.37	11	22	<10	57	77	55
average for wk 10-30	592	17	<0.1	<30	<2	<20	4	0.02	8	21	<10	<10	26	14
average for last 24	676	23	<0.1	<30	2	<20	2	0.15	<5	<5	<10	<10	24	13
Tail Sand #2 -NP														
median	355	22	0.10	<30	<2	<20	<2	<0.01	12	<20, 5	<10	<10	<20	11
maximum	1004	31	0.20	54	3	<20	15	1.22	25	8	<10	23	33	51
average for wk 10-30	321	25	<0.1	<30	<2	<20	3	0.01	20	<20	<10	<10	<20	11
average for last 24	648	20	<0.1	<30	3	<20	4	0.27	10	<5	<10	<10	<20	17
WR 150														
median	72	25	0.10	<30	<2	<20	3	<0.01	<5	<20, 5	<10	<10	<20	10
maximum	148	39	0.30	42	8	<20	14	0.34	7	8	<10	12	25	30
average for wk 10-30	49	24	<0.1	42	<2	<20	4	0.02	<5	<20	<10	<10	<20	30
average for last 24	92	29	<0.1	<30	<2	<20	3	0.07	<5	<5	<10	<10	<20	8
WR 180														
median	203	26	0.10	<30	<2	<20	4	0.02	13	<20, 5	<10	<10	<20	10
maximum	476	105	0.30	35	3	<20	42	0.63	123	11	<10	26	20	48
average for wk 10-30	34	25	<0.1	<30	<2	<20	3	0.01	<5	<20	<10	<10	<20	30
average for last 24	92	29	<0.1	<30	<2	<20	3	0.07	<5	<5	<10	<10	<20	8
	SO₄	Alk	Al	As	Cd	Cr	Cu	Fe	Mo	Ni	Pb	Sb	Se	Zn

Note: Averages for weeks 10 to 30 and the last 24 wks (September 2003 to February 2004) are calculated with data above detection limit within the specified period, and do not include weeks below detection limit, unless all the weeks are below the detection limit.

Table 19. Composition of Field Test Pads Constructed from Backfilled Tailings Sand

	Total-S	SO ₄ -S	MPA	NP	NPR	Paste-pH
1	5.17	0.17	161.45	148.56	0.92	7.0
2	4.85	0.19	151.57	154.10	1.02	7.0
3	5.75	0.19	179.82	149.29	0.83	7.0

Table 20. The Range in Composition of Seepage from Waste Rock Dumps Monitored in 1998

pH	7.4-7.9
Al	0.02 - 0.07 (diss)
As	nd - 0.25 (tot)
Cd	<0.0001 - 0.001 (diss)
Cr	nd - 0.003 (diss)
Cu	0.002 - 0.75 (tot); 0.001 - 0.004 (diss)
Fe	0.027 - 0.058 (diss)
Ni	nd - 0.1 (tot)
Pb	0.003 - 0.29 (tot); nd - 0.007 (diss)
Zn	0.02 - 0.95 (tot); nd - 0.05 (diss)

(mg/L except for pH)

Table 21. Drainage Chemistry of Tailings Pond Supernatant During Tailings Disposal, Waste Rock Disposal and Covering of the Tailings, and the Period of Initial Discharge of Drainage from Flooded Workings from the 300 Portal (adapted from Love, 2000)

DATE	pH	SO ₄ -D	Al-D mg/L	As-D mg/L	Cd-D mg/L	Cr-D mg/L	Cu-D mg/L	Fe-D mg/L	Fe-T mg/L	Pb-D mg/L	Zn-D mg/L
Active Tailings Disposal											
1999-01-11	7.9	175	0.02	0.04	0.0001	0.002	0.001	0.005	0.15	0.001	0.007
1999-01-26	8.0	182	0.02	0.04	0.0001	0.002	0.001	0.003	0.15	0.001	0.002
1999-02-02	7.9	183	0.02	0.04	0.0001	0.002	0.001	0.003	0.12	0.001	0.005
1999-04-13	7.7	124	0.03	0.04	0.0001	0.003	0.002	0.014	0.33	0.001	0.012
1999-04-27	7.9	122	0.02	0.04	0.0001	0.003	0.001	0.003	0.68	0.001	0.002
1999-05-04	7.9	110	0.02	0.04	0.0001	0.002	0.001	0.003	0.05	0.001	0.01
1999-05-18	8.0	124	0.02	0.04	0.0002	0.002	0.001	0.003	0.51	0.001	0.002
Waste Rock Disposal and Covering of the Tailings											
1999-06-15	7.9	106	0.05	0.04	0.0001	0.002	0.002	0.017	0.76	0.001	0.003
1999-06-29	8.1	236	0.09	0.04	0.0003	0.004	0.004	0.075	9.71	0.001	0.018
1999-07-20	8.2	245	0.03	0.04	0.0001	0.004	0.002	0.023	0.53	0.001	0.004
1999-08-24	8.2	241	0.08	0.04	0.0002	0.002	0.001	0.106	6.61	0.001	0.004
1999-09-07	8.2	402	0.02	0.05	0.0004	0.003	0.001	0.009	3.62	0.001	0.021
1999-09-20	8.1	397	0.03	0.06	0.0003	0.001	0.001	0.006	7.85	0.001	0.005
1999-10-20	8.15	150	0.05	0.05	0.0005	0.022	0.011	0.08		0.005	0.018
Period of Initial Discharge from Flooded Workings from 300 Portal											
2000-01-19	7.79	227		0.0003	0.0003	<.03	<.001	0.05	0.11	0.001	0.027
2000-01-31	7.94	179		0.0003	0.0002	<.03	0.003	<0.03	0.12	0.006	0.025
2000-02-16	8.04	230	0.05	0.05	0.005	0.005	0.005	0.006		0.05	0.004

Table 22. Post-Closure Drainage Chemistry for the Tailings Pond Discharge

DATE	pH	TSS	Alk	Hard	NO ₃	SO ₄	D-Al	T-As	D-As	D-Cd	D-Ca	D-Cu	T-Fe	D-Fe	D-Mg	D-Mn	D-Sb	D-Se	T-Si	D-Si	T-Zn	D-Zn
		mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l
1/19/2000	7.8	<3	152	384	0.81	227	0.024	0.0006	0.0003	0.0003	135	<0.001	0.11	0.05	11.6	0.401	0.0005	<0.2			0.027	0.027
1/31/2000	7.9	<3	114	283	0.46	179	0.007	0.0007	0.0003	0.0002	99	0.003	0.12	<0.03	8.9	0.467	0.0007	<0.2			0.024	0.025
2/17/2000	8.1	<3	146	321	0.41	230	0.005	0.0005	0.0004	<0.0002	111	<0.005	<0.03	<0.03	10.6	<0.005	0.0006	0.0006			<0.005	<0.005
3/22/2000	7.8	<3	94	176	0.18	123	0.014	0.0005	0.0004	<0.0002	63	0.002	0.08	<0.03	4.8	0.36	0.0005	0.0007			0.013	0.013
4/18/2000	8.1	<3	107	261	0.37	164	0.017	0.0005	0.0005	<0.0002	92	0.002	0.09	<0.03	7.9	0.16	0.0005	0.0006	2.39	2.33	<0.005	<0.005
5/15/2000	8.1	<3	69	167	0.71	109	0.090	0.0006	0.0004	<0.0002	58	0.004	0.22	0.04	5.1	0.047	0.0005	<0.0005	1.52	1.47	<0.005	<0.005
6/27/2000	8.0	<3	116	274	0.23	193	0.021	0.0006	0.0005	<0.0002	94	0.002	0.07	<0.03	9.5		0.0005				<0.005	<0.005
7/11/2000	8.2	<3	111	244	0.27	141	0.010	0.0007	0.0006	<0.0002	85	0.002	0.14	<0.03	7.4	0.202	0.0003	<0.0005	2.28	2.30	0.006	0.006
7/25/2000	8.1	<3	121	277	0.16	243	0.033	0.0002	<0.0002	<0.0002	96	0.002	0.05	0.03	9.1	0.008	<0.0002	0.0006	1.49	1.43	<0.005	<0.005
8/14/2000	8.2	0	96	258	0.01	270	0.020		0.0004	<0.0002	84	0.002		<0.03	12.0	<0.005	0.0008	<0.0005		1.01		<0.005
9/25/2000	8.1	<3	120	261	0.22	157	0.015	0.0007	0.0005	<0.0002	90	0.001	0.12	<0.03	8.6	0.171	0.0005	0.0006	2.44	2.44	<0.005	<0.005
11/7/2000	8.2	<3	111	244	0.27	141	0.010	0.0007	0.0006	<0.0002	85	0.002	0.14	<0.03	7.4	0.202	0.0003	<0.0005	2.28	2.30	0.006	0.006
2/4/2001	8.0	<3	99	190	0.27	97	0.010	0.0005	0.0004	<0.0002	66	<0.001	0.09	<0.03	6.0	0.119	0.0003	<0.0005	2.09	2.09	0.008	0.006
3/12/2001	8.2	7	61	116	0.11	67	0.020	0.0006	0.0004	<0.0002	40	<0.001	0.43	0.04	3.8	0.082	0.0003	<0.0005	1.39	1.11	0.006	<0.005
4/23/2001	8.2	4	107	231	0.13	134	0.037	0.0007	0.0005	<0.0002	80	0.002	0.21	0.04	7.7	0.041	<0.0001	<0.0005	1.8	1.68	<0.005	<0.005
5/11/2001	8.1	<3	100	202	0.22	128	0.032	0.0005	0.0007	<0.0002	70	<0.001	<0.03	0.1	6.8	0.054	0.0003	0.0009	1.67	1.71	<0.005	<0.005
6/4/2001	8.1	<3	82	167	0.40	102	0.018	0.0006	0.0005	<0.0002	57	0.001	0.08	<0.03	5.9	0.02	0.0002	<0.0005	1.81	1.77	<0.005	<0.005
7/18/2001	8.2	4	130	301	0.13	201	0.054	0.0009	0.0009	<0.0002	102	0.002	0.16	0.1	11.3	0.018	0.0011	0.0011	2.35	2.36	<0.005	0.011
9/16/2001	8.3	<3	117	278	0.06	183	0.013	0.0005	0.0005	<0.0002	95	0.002	<0.03	<0.03	10.1	<0.005	0.0004	<0.0005	1.23	1.27	<0.005	<0.005
10/1/2001	8.2	<3	90	199	0.25	124	0.013	0.0007	0.0005	<0.0002	69	0.002	0.14	<0.03	6.7	0.072	0.0002	<0.0005	1.84	1.84	<0.005	<0.005
11/4/2001	8.2	<3	97	200	0.24	108	0.011	0.0009	0.0006	<0.0002	70	0.001	0.14	<0.03	6.2	0.12	0.0006	<0.002	2.05	2.01	<0.005	<0.005
12/11/2001	8.1	<3	126	248	0.34	153	<0.005	0.0004	0.0002	<0.0002	85	0.002	0.09	<0.03	8.7	0.082	<0.001	<0.0005	2.51	2.48	0.007	0.007
12/13/2001	8.0	<3	146	287	0.43	163	0.005	0.0008	0.0007	<0.0002	98	0.002	0.07	<0.03	10.5	0.063	0.0005	0.0006	2.78	2.78	0.006	0.009
1/27/2002	8.0	<3	146	287	0.43	163	0.005	0.0008	0.0007	<0.0002	98	0.002	0.07	<0.03	10.5	0.063	0.0005	0.0006	2.78	2.78	0.006	0.009
2/24/2002	8.0	<3	134	258	0.31	151	<0.005	0.0007	0.0005	<0.0002	88	0.002	0.08	<0.03	9.3	0.082	0.0004	0.0006	2.8	2.71	0.006	0.006
3/27/2002	8.1	<3	91	170	0.18	111	<0.005	0.0006	0.0004	<0.0002	58	0.002	0.08	<0.03	6.3	0.02	0.0004	<0.0005	1.87	1.80	<0.005	<0.005
4/22/2002	7.9	<3	89	166	0.16	92	0.008	0.0007	0.0004	<0.0002	58	0.002	0.17	<0.03	5.4	0.181	0.0003	<0.0005	1.78	1.80	<0.005	<0.005
5/6/2002	8.1	<3	92	162	0.26	97	0.019	0.0007	0.0005	<0.0002	56	0.002	0.13	<0.03	5.2	0.051	0.0005	<0.0005	1.64	1.58	<0.005	<0.005
6/28/2002	8.4	<3	118	264	0.07	175	0.012	0.0008	0.0008	<0.0002	89	0.002	0.04	<0.03	10.1	0.007	0.0004	0.0006	1.41	1.36	<0.005	<0.005
7/28/2002	8.2	<3	122	289	0.04	190	0.011	0.0008	0.0008	<0.0002	96	0.001	<0.03	<0.03	11.8	<0.005	0.0004	0.0008	1.73	1.70	<0.005	<0.005

DATE	pH	TSS	Alk	Hard	NO ₃	SO ₄	D-Al	T-As	D-As	D-Cd	D-Ca	D-Cu	T-Fe	D-Fe	D-Mg	D-Mn	D-Sb	D-Se	T-Si	D-Si	T-Zn	D-Zn
		mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l	mg/l
8/24/2002	8.1	4	75	177	0.08	114	0.161	0.0011	0.0008	<0.0002	61	0.002	0.53	0.21	6.1	0.062	0.0018	<0.001	1.69	1.59	0.007	<0.005
9/23/2002	8.1	<3	88	176	0.27	96	0.011	0.0008	0.0006	<0.0002	61	0.002	0.18	<0.03	5.6	0.063	0.0005	<0.0005	2.17	2.13	0.007	<0.005
10/7/2002	8.2	<3	115	230	0.28	130	0.007	0.0009	0.0008	<0.0002	79	<0.001	0.07	<0.03	8.0	0.038	0.0019	0.0006	2.24	2.21	<0.005	<0.005
11/4/2002	8.2	<3	152	298	0.32	188	0.008	0.0012	0.0008	<0.0002	100	0.001	0.07	<0.03	11.4	0.024	0.0006	0.0009	2.69	2.66	0.006	0.007
12/16/2002	8.1	<3	109	249	0.34	111	0.187	0.0007		<0.0002	87	0.004	<0.03		7.8	0.087	0.0003	0.0005	2.7	2.95	0.006	0.016
1/6/2003	7.9	37	68	121	0.37	70	0.329	0.0029	0.0008	<0.0002	42	0.003	3.4	0.47	4.3	0.071	0.0003	0.0006	3.83	1.93	0.022	<0.005
2/22/2003	8.1	<3	149	276	0.34	162	<0.01	0.001	0.001	<0.0004	92	<0.002	<0.03	<0.03	10.9	0.013	0.0014	<0.0005	2.72	2.74	0.007	<0.005
4/22/2003	8.0	<3	89	157	0.20	81	0.008	0.0007	0.0006	<0.0002	54	0.003	0.09	<0.03	5.2	0.067	0.0003	<0.0005	1.72	1.70	<0.005	0.009
6/4/2003	8.2	<3	116	272	0.13	150	0.007	0.0006	0.0005	<0.0002	94	<0.001	0.06	<0.03	8.9	0.007	0.0002	<0.0005	1.37	1.36	<0.005	<0.005
8/24/2003	8.3	21	97	237	<0.005	167	0.016	0.0011	0.0011	<0.0002	78	0.001	<0.03	<0.03	10.3	0.006	0.001	<0.001	1.75	1.81	<0.005	<0.005
10/7/2003	8.2	<3	137	246	0.11	140	0.010	0.0012	0.0011	<0.0002	84	0.001	0.03	<0.03	9.1	0.01	0.0005	<0.001	2.25	2.23	<0.005	<0.005
12/17/2003	8.1	<3.0	134	233	0.27	128	<0.005	0.0012	0.001	<0.0002	80	0.0014	0.046	<0.030	8.3	0.168	0.00028	<0.0010	2.52	2.62	<0.005	<0.005
Minimum	7.8		61	116	<0.005	67	<0.005	0.0002	0.0002		40	<0.002	<0.03		3.8	<0.005	<0.0001		1.23	1.01		
Median	8.1	<3	111	245	0.26	141	0.013	0.0007	0.0005	<0.0002	84	0.002	0.09	<0.03	8.1	0.063	0.0005	<0.0005	2.07	1.93	<0.005	<0.005
Maximum	8.4	37	152	384	0.81	270	0.329	0.0029	0.0011	0.0003	135	0.004	3.4	0.47	12.0	0.467	0.0019	0.0011	3.83	2.95	0.027	0.027

Table 23. An Estimate of the Relative Percentage Contributions of Ore and Waste Rock to Tailings Sand and Backfill (Lacouture, June 13, 1996)

Year	Tailings Sand Composition		Backfill Composition	
	Ore	Waste Rock	Ore	Waste Rock
1991	100	0	90	10
1992	98	2	80	20
1993	95	5	75	25
1994	90	10	65	35
1995	90	10	65	35
1996	85	15	55	45

Table 24. Results of Pre-Flooding Monitoring of the Underground Drainage Chemistry

	Oct-96	Aug-97	Apr-98	1998	Jan/Feb 99				Sept-99
					Range	Median	5 th P	95 th P	
pH		7.0-7.8	6.9-8.0	7.6-7.7	7.8-8.2				8.0-8.1
SO ₄ mg/l	363-941	55-1390	69-1230		18-508	298	35	453	88-253
Ca mg/l						115	43	178	
Mg mg/l						14	6	23	
Na mg/l						22	2	49	
K mg/l						28	4	41	
Ag mg/l	<0.01- 0.25	<0.01	<0.01-.11						nd
Al mg/l	9-1580	<1	<1-480	0.3	0.02-0.24	0.035	0.02	0.201	nd-0.08
As mg/l	0.2-18.5	<0.05	nd-6.7	nd-1.75	0.04	0.04	0.04	0.04	nd-0.08
Cd mg/l	0.001-0.665	0.001-0.665	<0.001-0.098		0.0001-0.0006	0.0002	0.0001	0.0005	0.0001-0.0006
Co mg/l	<0.02-1.1	<0.02	<0.02-0.36		0.003				nd
Cr mg/l	0.02-2.9	<0.02	<0.02-0.80		0.002-0.005				nd-0.004
Cu mg/l	0.01-29	<0.01	0.01-4.8	<0.01-0.3	0.001-0.023	0.005	0.001	0.015	nd-0.004
Fe mg/l	35-3200	<1	<1-1120		0.004-0.76	0.03	0.006	0.563	0.003-0.2
Mn mg/l	3.5-93	<0.01-1.45	0.01-25		0.006-0.37	0.147	0.006	0.352	0.02-0.21
Mo mg/l	0.07-1.7	0.01-0.08	<0.01-0.51		0.004-0.08				0.006-0.02
Ni mg/l	0.11-3.0	<0.01-0.06	<0.01-0.90	<0.01-0.15	0.008-0.05	0.012	0.008	0.039	nd-0.011
Pb mg/l	0.05-13.5	<0.05	<0.05-3.2	<0.05-0.55	0.001-0.062	0.02	0.004	0.073	nd-0.007
Sb mg/l									nd-0.035
Se mg/l									nd-0.04
Zn mg/l	0.3-67	0.01-.22	0.05-12	0.06-4.0	0.002-0.063	0.01	0.002	0.061	0.014-0.087

mg/l unless stated 5th P: 5th Percentile
 1996 to 1998 data is total concentrations, 1999 data is dissolved concentrations

Table 25. Pre-Closure Prediction of Monthly Drainage Inputs into the Underground Workings

	Measured 180 Discharge	Inputs with Backfill m³	Groundwater Inflow
1997			
Jan	18972	7673	11299
Feb	27149	7390	19759
Mar	34596	6333	28263
Apr	50760	7430	43330
May	57883	7305	50578
Jun	48240	5907	42333
Jul	23138	4780	18358
Aug	19642	5680	13962
Sep	20376	6000	14376
Oct	44714	6447	38267
Nov	57528	3975	53553
Dec	50592	3692	46900
average	37799	6051	31748
median	39655	6167	33265
5 th Percentile	19341	3848	12764
95 th Percentile	57688	7539	51917
1998			
Jan	42334	6058	36276
Feb	41059	5695	35364
Mar	42706	6138	36568
Apr	44136	5025	39111
May	25445	6413	19032
Jun	33120	5472	27648
Jul	22320	4635	17685
Aug	18600	4735	13865
Sep	37440	4265	33175
Oct	39432	5665	33767
Nov	46800	7948	38852
Dec	39432	6630	32802
average	36069	5723	30345
median	39432	5680	33471
5 th Percentile	20646	4469	15966
95 th Percentile	45335	7223	38969

Table 26. Pre-Closure Prediction of the Time to Flood the Lower Workings up to the 300 Level

Location in Mine	Volume m³	Average Inflow m³/month	Estimated Leakage m³/month	Net Inflow m³/month	Estimated # of Months
130 to 180 Level	45081	31000	3523	27477	1.64
180 to 220 Level	33809	31000	6341	24659	1.37
220 to 260 Level	38217	31000	9159	21841	1.75
260 to 300 Level	32086	31000	11977	19023	1.69
130 to 300 Level	149193				6.45

Monthly inflow and leakage rates are based on estimated annual average.

Table 27. Post-Closure Measurements of Discharge from the Impoundment and Discharge and Water Pressure from the Underground Workings

DATE	Tailings Impound L/s	300 Portal L/s	180 Portal L/s	130 Portal L/s	130 Bulkhead psi	Upper Monsoon L/s
19-Jan-00		5.1	3		230	
31-Jan-00		2.4	2.5		225	
16-Feb-00		0.1	2.5		220	
22-Mar-00		0.2	3		220	
19-Apr-00		5.7	3.6		225	
15-May-00		23.9	2.5		229	
07-Jun-00		13.1	2		230	
27-Jun-00		8.6	3.1			
19-Jul-00		2	2		230	
10-Aug-00		0.7	3.1		225	
14-Aug-00		0.5	1.1		225	
05-Oct-00		8.6	1.4		225	
07-Nov-00		12.1	1.4		230	
04-Feb-01	14.0	1.1	2.3	4.1	230	
11-Mar-01	25.0	2.3	2	3.6	230	
23-Apr-01	10.5	2.7	1.1	5.1	230	
11-May-01	12.0	15.3	1.1	5.7	230	
04-Jun-01	10.5	24.4	0.7	5.1		
18-Jul-01	12.0		1	2.7		
13-Aug-01	0.0		1.2			
16-Sep-01	0.0	3	1.1	2	230	
01-Oct-01	10.0	9.4	1.2	4.1		
04-Nov-01	45.0	9.4	N/A	6.4	230	
11-Dec-01	4.0	8.3	N/A	2.9		14.0
27-Jan-02	10.5	7.1	0.5	3.1		
24-Feb-02	7.3	0.7	0.9	2.5		
27-Mar-02	3.0	1.7	0.7	2		6.1
22-Apr-02	27.5	1.5	1.1	2.7		
06-May-02	32.0	7.3	0.7	4.1		
28-Jun-02	10.0	14.4	0.7	2.9	230	
28-Jul-02		4.1	1.4	2.3		
24-Aug-02	16.5	2.7	1.1	2.7		
23-Sep-02	63.5	27.4	0.9	6.4		
07-Oct-02	40.0	15.3	1.1	4.1		
04-Nov-02	10.5	11.2	2	3.6		
16-Dec-02	42.0	11.2	N/A	4.1		
06-Jan-03	44.0	20.1	2	4.1		
22-Feb-03	3.0	4.1		2.2		
22-Apr-03		1.5		2.5		
04-Jun-03	16.0	8.5	0.8	2.2		
24-Aug-03		3	1	2.5		
07-Oct-03		11	0.9	4		
17-Dec-03	20.0	7		3		
01-May-04	76.0	27.5				
Average (L/s)	18.8	8.4	1.1	3.5		
Average (m3/month)	47646.7	21334	2904	8975		

* Average for data in 2001, 2002 and 2003

Table 28. Pre-Closure Prediction and Measured Drainage Chemistry from the Un-Flooded Underground Workings

Element	Humidity Cell Rates mg/kg/wk		Predicted Load mg/wk			Predicted Drainage mg/L	Measured Concentrations in Upper Workings* (mg/L)		
	WasteRock	Tail Sand	WallRock	Backfill	Total		Median	5 th Perc	95 th Perc
Cu	0.002	0.001	7,292	19,410	26,702	0.003	0.005	0.001	0.015
Zn	0.002	0.002	8,420	45,326	53,746	0.007	0.01	0.002	0.061
As	0.011	0.008	42,345	163,501	205,846	0.027	0.04	0.04	0.04
Ni	0.007	0.006	27,988	118,531	146,519	0.019	0.012	0.008	0.039
Cd	0.00076	0.00048	2,846	10,392	13,238	0.0017	0.0002	0.0001	0.0005
Pb	0.004	0.002	13,994	52,270	66,264	0.009	0.02	0.004	0.073
Cr	0.007	0.005	27,988	102,518	130,506	0.017			
Se	0.007	0.006	27,988	118,531	146,519	0.019			
Mo	0.003	0.003	10,730	53,831	64,560	0.008			
Mn	0.008	0.126	30,171	2,703,915	2,734,086	0.353	0.147	0.006	0.352
Fe	0.005	0.003	17,192	66,835	84,027	0.011	0.03	0.006	0.563
Ca	20.873	44.892	78,094,656	965,011,184	1,043,105,840	134.594	115.15	42.57	177.85
Mg	1.441	7.650	5,391,550	164,444,611	169,836,161	21.914	14.1	5.8565	22.505
Al	0.037	0.024	139,939	512,589	652,528	0.084	0.035	0.02	0.201
Na	0.341	0.338	1,277,588	7,262,975	8,540,564	1.102	21.65	2.245	48.835
K	4.039	5.488	15,110,191	117,965,523	133,075,713	17.171	28.1	3.57	41.32
SO4	48.85	126.85	182,766,397	2,726,744,520	2,909,510,917	375.4	298	35	453

* Measured in January and February 1999.

Table 29. Comparison of Predicted and Resulting Drainage Quality for Different Areas of the Underground Workings

Volume of Flooded Workings Interacting with Discharge from Upper Workings Minewater						
Depth (m)	Total	Backfill	Void	Wall Rock	Discharge Volume	Ratio
2	3068.6	1464.32	1604.28	306.86	31000 m ³ Flooded Volume 1604.28 m ³	0.95 0.05

	Predicted Drainage Quality Results				Tail. Impound Disch. Limits mg/L	Resulting Drainage Quality Results					
	Unflooded Workings mg/L	Initially Flooded Workings mg/L	Final Flooded Workings mg/L	Discharge from 300 Level mg/L		300 Level Discharge*1			130 Level Flooded Workings*2		
						Min mg/L	Average mg/L	Max mg/L	Min mg/L	Average mg/L	Max mg/L
Cu	0.003	0.019	0.025	0.004		<0.001	0.004		<0.005	0.016	
Zn	0.007	0.012	0.021	0.008	0.2	<0.005	0.080	0.388	0.016	0.437	0.654
As	0.027	0.037	0.077	0.029		<0.0002	0.001	0.001	<0.0003	0.0005	0.0008
Ni	0.019	0.029	0.056	0.021		0.001	0.003	0.017	0.002	0.008	0.025
Cd	0.0017	0.0020	0.0044	0.0018	0.0008	<0.0002	0.0004	0.0017		<0.0002	0.0006
Pb	0.009	0.025	0.028	0.009	0.05		<0.001	0.003		<0.001	0.003
Cr	0.017	0.025	0.028	0.017			<0.001	0.002			
Mo	0.008	0.018	0.029	0.009			<0.03	<0.03			<0.03
Mn	0.35	0.41	0.77	0.37		<0.005	0.11	0.48	0.085	0.549	0.692
Fe	0.01	0.47	0.49	0.03			<0.03	0.12	<0.03	0.09	0.65
Ca	135	215	372	146		42	118	205	138	187	235
Mg	22	26	50	23		4	13	24	15.7	22.2	27.4
Al	0.08	0.24	0.26	0.09			<0.005	0.01	<0.005	0.011	0.023
Na	1.1	2.0	3.4	1.2							
K	17	35	57	19							
SO ₄	375	534	963	404		57	200	428	245	355	493
pH						7.6	8.0	8.4	7.4	8.0	8.2
Hard						60	179	193	409	558	693
Alk						124	350	612	177	214	255

*1 Data from 300 Portal Discharge from 2000 to 2003

*2 Data from monitoring of drainage behind 130 Bulkhead in 2000 and 2001

Table 30. 2000 and 2001 Monitoring of Chemistry of Drainage in Flooded Workings Behind 130 Bulkhead

DATE	pH	Hard mg/l	Alk mg/l	SO ₄ mg/l	D-Al mg/l	D-As mg/l	D-Ca mg/l	D-Cd mg/l	D-Cu mg/l	T-Fe mg/l	D-Fe mg/l	D-Mg mg/l	D-Mn mg/l	D-Mo mg/l	D-Ni mg/l	D-Pb mg/l	D-Sb mg/l	D-Se mg/l	T-Zn mg/l	D-Zn mg/l
1/19/2000	7.7	613	194	468	<0.005	0.0004	209	0.0005	0.001	0.07	<0.03	22.1	0.475	<0.03	0.025	0.001	0.0025	<0.2	0.621	0.531
1/31/2000	7.8	668	199	481	<0.005	0.0003	223	0.0002	<0.001	0.08	0.03	27.1	0.628	<0.03	0.025	<0.001	0.0025	<0.2	0.631	0.583
2/17/2000	7.6	685	201	493	<0.005	0.0004	229	<0.0002	<0.005	0.16	0.07	27.4	0.692		0.022	<0.002	0.0025	0.0012	0.643	0.382
3/22/2000	7.4	587	199	483	<0.005	0.0005	195	<0.0002	<0.002	0.24	0.12	24.0	0.621		0.019	0.001	0.0022	0.0014	0.629	0.259
4/18/2000	7.5	601	204	459	<0.005	0.0004	201	0.0002	0.002	0.27	0.04	24.2	0.646	<0.03	0.018	<0.001	0.0018	0.0011	0.554	0.192
5/15/2000	7.7	670	208	467	<0.005	0.0004	229	<0.0002	0.001	0.48	0.06	23.8	0.689	<0.03	0.017	<0.001	0.0017	0.0011	0.580	0.145
6/27/2000	7.6	693	212	444	<0.005	<0.0002	235	<0.0002	<0.001	0.52	0.09	25.6			0.017	0.004	0.0014		0.654	0.148
7/11/2000	8.1	576	237	337	0.011	<0.0001	194	<0.0002	<0.005	0.89	0.05	22.2	0.549	<0.03	0.007	<0.001	0.0009	<0.0005	0.386	0.044
7/25/2000	7.7	559	226	434	0.005	<0.0002	189	<0.0002	<0.001	0.58	0.29	21.2	0.675	<0.03	0.013	0.003	0.0015	<0.0005	0.627	0.057
8/10/2000	7.7	452	219	431	0.007	<0.0002	138	<0.0002	<0.001	0.54	0.48	26.2	0.674	<0.03	0.011	0.001	0.001	<0.0005	0.528	0.039
10/4/2000	8.0	556	177	373	0.021	<0.0002	185	<0.0002	<0.01	0.59	0.33	22.8	0.571	<0.03	0.009	0.002	0.0009	0.0005	0.487	0.03
11/7/2000	8.1	576	237	337	0.011	<0.0001	194	<0.0002	<0.005	0.89	0.05	22.2	0.549	<0.03	0.007	<0.001	0.0009	<0.0005	0.386	0.044
12/20/2000	8.0	476	245	288	<0.005	<0.0002	157	<0.0002	<0.001	1.27	<0.03	20.4	0.488	<0.03	0.002	<0.001	<0.0005	0.0006	0.175	0.006
2/4/2001	8.0	531	244	260	0.008		176	<0.0002	<0.02	0.87	0.09	22.6	0.499	<0.03	0.002	<0.001	<0.0002	<0.0005	0.355	0.051
3/12/2001	8.1	537	255	255	0.007	0.0005	179	<0.0002	<0.001	0.98	0.05	21.6	0.467	<0.03	0.002	0.001	0.0002	<0.0005	0.225	0.021
4/23/2001	8.2	435	236	245	0.015	0.0005	145	<0.0002	0.001	0.47	0.25	17.7	0.411	<0.03	0.002	<0.001	<0.0002	<0.0005	0.132	<0.005
5/11/2001	8.2	432	246	246	0.01	0.0007	145	<0.0002	<0.001	0.55	0.20	16.9	0.318	<0.03	0.002	<0.001	<0.0002	0.0005	0.016	0.15
6/4/2001	8.2	409	215	262	0.011	0.0003	138	<0.0002	<0.001	0.75	0.03	15.7	0.135	<0.03	0.003	0.002	0.0005	<0.0005	0.171	0.02
7/18/2001	8.2	410	196	249	0.023	0.0007	138	0.0006	0.016	0.68	0.65	15.7	0.085	<0.03	0.003	0.032	0.0022	<0.0005	0.282	0.294
11/4/2001	8.1	418	208	257	0.005	0.0008	141	<0.0002	<0.001	0.62	0.52	16.1	0.106	<0.03	0.002	<0.001	0.0004	<0.002	0.163	0.008
Minimum	7.4	409	177	245	<0.005	<0.0003	138			0.07	<0.03	15.7	0.085		0.002		<0.0005		0.016	<0.005
Median	8.0	558	214	355	0.011	0.0005	187	<0.0002	<0.005	0.56	0.09	22.2	0.549		0.008	<0.001	0.0015	<0.0005	0.437	0.057
Maximum	8.2	693	255	493	0.023	0.0008	235	0.0006	0.016	1.27	0.65	27.4	0.692	<0.03	0.025	0.032	0.0025	0.0014	0.654	0.583

Table 31. 2000 Monitoring of Chemistry of Seep from Fracture Next to the 180 Bulkhead

DATE	pH	S ₀₄ mg/l	D-As mg/l	D-Ca mg/L	D-Cd mg/l	T-Fe m/l	D-Fe mg/l	D-Mn mg/L	D-Pb mg/l	D-Zn mg/l	T-Zn mg/l	N03 mg/l	N02 mg/l
01 19 2000	7.62	473	0.0002	221	0.0018	0.07	<0.03	0.53	0.002	0.648	0.634	2.42	0.04
01 31 2000	7.77	475	0.0003	209	0.0016	0.05	<0.03	0.56	<0.001	0.65	0.619	2.17	0.049
02 17 2000	7.55	468	0.0003	231	0.0012	0.1	<0.03	0.64	<0.002	0.633	0.624	1.78	0.046
03 22 2000	7.31	448	<0.0002	208	0.0004	0.19	0.12	0.64	<0.001	0.667	0.662	1.19	0.058
04 18 2000	7.42	431	<0.0002	199	0.0006	0.18	0.09	0.60	<0.001	0.582	0.592	1.06	0.044
05 15 2000	7.53	386	0.0007	190	<0.0002	0.07	<0.03	0.48	<0.001	0.310	0.306	0.891	0.055
06 27 2000	7.55	332	0.0002	136	0.0003	0.04	<0.03	0.47	0.005	0.347	0.381	0.561	0.021
07 25 2000	7.71	304	<0.0002	161	<0.0002	0.03	<0.03	0.47	0.003	0.329	0.335	0.48	0.002
08 10 2000	7.73	284	0.0002	200	0.0003	0.05	<0.03	0.47	0.003	0.366	0.376	0.477	0.002
Mean	7.58	400	0.0003	200	0.0007	0.09	0.05	0.54	0.002	0.504	0.503	1.225	0.035
Maximum	7.77	475	0.0007	231	0.0018	0.19	0.12	0.64	0.005	0.667	0.662	2.42	0.058

Table 32. Post-Closure Chemistry of Discharge from 130 Portal

DATE	pH	TSS mg/l	Alk mg/l	Hard mg/l	NO ₃ mg/l	SO ₄ mg/l	D-Al mg/l	D-As mg/l	D-Ca mg/l	D-Cd mg/l	D-Cu mg/l	T-Fe mg/l	D-Fe mg/l	D-Mg mg/l	D-Mn mg/l	D-Sb mg/l	D-Se mg/l	D-Si mg/l	D-Zn mg/l
1/19/2000	8.0	<3	143	386	1.77	239	<0.005	0.0009	125	0.0003	0.006	0.06	<0.03	17.9	0.065	0.0016	<0.2		0.045
4/18/2000	7.8	<3	138	314	1.50	197	<0.005	0.0003	101	0.0002	0.004	<0.03	<0.03	14.8	0.056	0.001	0.0012	2.88	0.048
5/15/2000	7.9	<3	134	318	1.59	172	<0.005	0.0004	105	<0.0002	0.002	<0.03	<0.03	13.3	0.048	0.0009	0.0011	2.75	0.049
6/27/2000	7.8	<3	156	418	0.89	238	<0.005	0.0005	137	0.0002	0.002	<0.03	<0.03	18.2		0.0013			0.051
7/11/2000	8.1	<3	131	247	1.25	124	<0.005	0.0003	81	<0.0002	0.003	<0.03	<0.03	11.1	0.029	0.0007	0.0006	2.74	0.029
7/25/2000	8.0	<3	170	389	0.79	245	<0.005	0.0005	131	<0.0002	0.001	<0.03	<0.03	14.8	0.053	0.0015	0.0007	3.16	0.046
8/10/2000	8.0	3	173	291	0.73	244	0.009	0.0004	84	<0.0002	0.002	<0.03	<0.03	20.0	0.060	0.0013	<0.0005	3.03	0.045
10/4/2000	8.0	<3	138	264	1.12	136	<0.005	0.0004	85	<0.0002	0.003	0.03	<0.03	12.3	0.041	0.0009	0.0012	2.97	0.037
11/7/2000	8.1	<3	131	247	1.25	124	<0.005	0.0003	81	<0.0002	0.003	<0.03	<0.03	11.1	0.029	0.0007	0.0006	2.74	0.029
12/20/2000	8.1	25	137	251	0.71	248	<0.005	0.0004	81	<0.0002	0.002	<0.03	<0.03	11.6	0.006	0.0005	0.0011	2.62	0.025
2/4/2001	8.1	<3	138	242	1.32	115	0.005	0.0003	80	<0.0002	0.005	0.04	<0.03	10.3	0.016	0.0006	0.0016	2.76	0.025
3/12/2001	8.2	<3	148	271	0.80	150	0.008	0.0004	89	<0.0002	0.004	0.03	<0.03	11.6	0.017	0.001	0.0006	2.69	0.025
4/23/2001	8.2	<3	137	243	1.23	128	<0.005	0.0003	80	<0.0002	0.005	0.04	<0.03	10.7	0.006	<0.0001	<0.0005	2.73	0.018
5/11/2001	8.2	<3	124	228	1.33	110	0.006	0.0003	75	<0.0002	0.004	<0.03	<0.03	9.6	0.007	0.0004	0.0013	2.84	0.023
6/4/2001	8.2	<3	143	235	1.21	123	<0.005	0.0003	77	<0.0002	0.003	<0.03	<0.03	10.2	<0.005	0.0007	0.0006	2.84	0.023
7/18/2001	8.1	<3	151	283	0.83	160	0.008	0.0006	92	<0.0002	0.005	0.07	0.04	12.7	0.005	0.0011	<0.0005	2.93	0.027
8/13/2001	8.3	<3	164	322	0.50	174	0.008	0.0006	103	0.0043	0.002	<0.03	<0.03	15.7	<0.005	0.0009	0.0008	3.10	0.023
9/14/2001	8.2	<3	163	302	0.68	157	0.017	0.0004	99	<0.0002	0.004	<0.03	<0.03	13.6	<0.005	0.0007	0.0007	3.44	0.029
10/1/2001	8.2	<3	127	237	1.00	125	0.008	0.0003	77	<0.0002	0.006	0.03	<0.03	10.6	0.009	0.0008	0.0013	2.84	0.022
11/4/2001	7.9	<3	114	206	1.41	94	0.007	0.0002	69	<0.0002	0.007	0.07	<0.03	8.6	0.008	0.0005	<0.002	2.70	0.017
12/11/2001	8.3	<3	142	242	1.21	134	<0.005	<0.0001	79	<0.0002	0.004	<0.03	<0.03	10.6	<0.005	<0.001	0.0005	2.78	0.017
1/27/2002	8.1	<3	137	237	2.60	124	<0.005	0.0004	77	<0.0002	0.004	<0.03	<0.03	10.7	<0.005	0.0011	0.0007	2.63	0.014
2/24/2002	8.0	<3	144	263	0.91	144	0.009	0.0004	86	<0.0002	0.006	<0.03	<0.03	12.0	0.006	0.001	0.0011	2.87	0.015
3/27/2002	8.3	<3	156	270	0.77	161	<0.005	0.0004	88	<0.0002	0.003	<0.03	<0.03	12.6	<0.005	0.0009	0.0007	2.86	0.014
4/22/2002	7.9	<3	142	237	0.85	130	0.008	0.0004	76	<0.0002	0.006	0.05	<0.03	11.2	0.007	0.0012	0.0008	2.78	0.016
5/6/2002	8.0	<3	124	207	1.30	105	<0.005	0.0002	69	<0.0002	0.004	0.04	<0.03	8.6	0.007	0.0007	0.001	2.66	0.015
6/28/2002	8.3	<3	138	257	1.11	138	0.008	0.0003	84	<0.0002	0.007	<0.03	<0.03	11.2	<0.005	0.0007	<0.0005	2.77	0.016
7/28/2002	8.1	<3	153	286	0.77	163	<0.005	0.0004	93	<0.0002	0.004	<0.03	<0.03	12.8	<0.005	0.0006	<0.0005	2.92	0.015
8/24/2002	8.1	<3	152	266	0.67	152	0.013	0.0004	87	<0.0002	0.008	<0.03	<0.03	12.0	0.014	0.0016	<0.001	2.90	0.025
9/23/2002	8.0	<3	109	205	1.38	99	0.006	0.0001	68	<0.0002	0.007	0.09	<0.03	8.7	0.014	0.0009	0.0012	2.78	0.017

DATE	pH	TSS mg/l	Alk mg/l	Hard mg/l	NO ₃ mg/l	SO ₄ mg/l	D-Al mg/l	D-As mg/l	D-Ca mg/l	D-Cd mg/l	D-Cu mg/l	T-Fe mg/l	D-Fe mg/l	D-Mg mg/l	D-Mn mg/l	D-Sb mg/l	D-Se mg/l	D-Si mg/l	D-Zn mg/l
10/7/2002	8.1	<3	124	228	<0.005	110	<0.005	0.0002	76	<0.0002	0.005	<0.03	<0.03	9.6	0.008	0.0007	0.0011	2.82	0.016
11/4/2002	8.1	<3	137	236	1.13	122	<0.005	0.0003	77	<0.0002	0.004	<0.03	<0.03	10.7	<0.005	0.001	0.001	2.81	0.015
12/16/2002	8.0	<3	122	260	1.37	99	0.074	0.0056	84	<0.0002	0.011	0.04	0.46	11.9	0.013	0.0008	0.0011	3.41	0.020
1/6/2003	7.9	<3	118	192	1.44	98	0.025	0.0001	63	<0.0002	0.013	0.09	<0.03	8.5	0.013	0.0003	0.0011	2.77	0.015
2/22/2003	8.1	<3	145	263	0.97	133	<0.01	0.0003	85	<0.0004	0.003	<0.03	<0.03	12.5	<0.005	0.0016	<0.0005	2.99	0.011
4/22/2003	7.8	<3	120	210	1.47	98	<0.005	0.0002	69	<0.0002	0.007	<0.03	<0.03	9.3	0.010	0.0005	0.0009	2.75	0.014
6/4/2003	8.2	<3	132	237	1.20	120	<0.005	0.0003	80	<0.0002	0.004	<0.03	<0.03	9.2	<0.005	0.0007	0.0009	2.75	0.011
8/24/2003	8.3	<3	167	276	0.54	162	<0.005	0.0003	90	<0.0002	0.003	<0.03	<0.03	12.5	<0.005	0.0011	<0.001	2.85	0.040
10/7/2003	8.2	<3	138	231	1.28	112	<0.005	0.0003	75	<0.0002	0.005	<0.03	<0.03	9.6	<0.005	0.0003	<0.001	2.65	0.023
12/17/2003	8.1	<3	142	241	1.13	125	<0.005	0.0003	80	<0.0002	0.005	<0.03	<0.03	10.2	<0.005	0.0005	<0.001	2.76	0.034
Minimum	7.8		109	192	0.01	94		<0.0001	63		0.001			8.5		<0.0001	<0.0005	2.62	0.011
Median	8.1	<3	138	249	1.13	132	<0.005	0.0003	81	<0.0002	0.004	<0.03	<0.03	11.2	<0.005	0.0009	0.001	2.80	0.023
Maximum	8.3	25	173	418	2.60	248	0.074	0.0056	137	0.0043	0.013	0.09	0.46	20.0	0.065	0.0016	0.200	3.44	0.051

Table 33. Post-Closure Chemistry of Discharge from 180 Portal

DATE	pH	TSS mg/l	Alk mg/l	Hard mg/l	NO ₃ mg/l	SO ₄ mg/l	D-Al mg/l	D-As mg/l	D-Ca mg/l	D-Cd mg/l	D-Cu mg/l	T-Fe mg/l	D-Fe mg/l	D-Mg mg/l	D-Mn mg/l	D-Sb mg/l	D-Se mg/l	D-Si mg/l	D-Zn mg/l
4/18/2000	7.9	<3	199	455	0.48	317	<0.005	0.0002	144	0.0003	0.002	0.04	<0.03	23.5	0.214	0.001	0.0007	4.28	0.129
5/15/2000	7.8	<3	198	434	0.44	290	<0.005	0.0004	141	<0.0002	<0.001	0.04	<0.03	19.9	0.179	0.001	<0.0005	3.89	0.081
6/27/2000	7.8	4	198	459	0.35	270	<0.005	0.0002	147	<0.0002	<0.001	0.80	<0.03	22.1		0.0008			0.073
7/11/2000	8.1	<3	189	381	0.22	195	0.019	<0.0001	122	<0.0002	<0.005	0.08	<0.03	18.2	0.097	0.0009	<0.0005	4.35	0.036
7/25/2000	7.8	<3	206	364	0.29	251	<0.005	0.0002	119	<0.0002	<0.001	<0.03	<0.03	16.4	0.143	0.0009	0.0006	3.90	0.060
8/10/2000	7.9	<3	201	409	0.25	263	<0.005	<0.0002	120	<0.0002	0.002	0.05	<0.03	26.6	0.117	0.0008	0.0007	4.22	0.059
10/4/2000	7.9	<3	195	360	0.27	211	0.011	0.0004	112	0.0002	<0.01	0.05	<0.03	19.7	0.118	0.0006	0.0007	4.42	0.044
11/7/2000	8.1	<3	189	381	0.22	195	0.019	<0.0001	122	<0.0002	<0.005	0.08	<0.03	18.2	0.097	0.0009	<0.0005	4.35	0.036
12/20/2000	8.2	<3	186	319	0.17	192	<0.005	0.0004	101	0.0002	<0.001	0.04	<0.03	16.2	0.025	<0.0005	<0.0005	3.94	0.019
2/4/2001	8.1	51	191	338	0.15	169	0.025	<0.0003	108	<0.0002	<0.02	1.44	0.04	16.5	0.046	0.0005	0.001	4.27	0.022
3/12/2001	8.2	<3	194	365	0.12	191	0.005	<0.0003	117	<0.0002	<0.001	0.06	<0.03	17.6	0.045	0.0003	<0.0005	4.24	0.020
4/23/2001	8.3	7	199	342	0.13	182	0.040	0.0004	109	<0.0002	0.003	0.76	0.04	17.0	0.036	0.0005	<0.0005	4.35	0.014
5/11/2001	8.2	<3	190	377	0.12	187	1.300	0.0006	120	<0.0002	<0.02	<0.03	2.67	18.8	0.101	0.0002	0.0009	6.19	0.032

DATE	pH	TSS mg/l	Alk mg/l	Hard mg/l	NO ₃ mg/l	SO ₄ mg/l	D-Al mg/l	D-As mg/l	D-Ca mg/l	D-Cd mg/l	D-Cu mg/l	T-Fe mg/l	D-Fe mg/l	D-Mg mg/l	D-Mn mg/l	D-Sb mg/l	D-Se mg/l	D-Si mg/l	D-Zn mg/l
6/4/2001	8.2	<3	188	355	0.12	195	0.005	<0.0002	113	<0.0002	<0.001	0.03	<0.03	17.8	0.03	0.0002	<0.0005	4.53	0.024
7/18/2001	8.2	<3	190	355	0.10	199	0.015	<0.0002	112	<0.0002	0.003	0.18	0.07	18.1	0.026	0.0007	<0.0005	4.39	0.022
8/13/2001	8.2	19	200	372	0.09	198	0.116	<0.0002	116	0.0012	0.002	1.14	0.16	20.2	0.027	0.0003	0.0006	4.53	0.017
9/14/2001	8.2	<3	206	418	0.10	200	0.006	<0.0005	131	<0.0002	0.003	0.06	<0.03	22.0	0.021	0.0003	<0.0005	5.24	0.019
10/1/2001	8.2	<3	178	327	0.12	179	<0.005	<0.0002	104	<0.0002	<0.005	0.04	<0.03	16.2	0.02	<0.0002	0.0006	4.32	0.020
11/4/2001	8.0	<3	179	310	0.11	156	0.009	0.0004	100	<0.0002	0.001	0.21	<0.03	14.4	0.02	0.0003	<0.002	4.25	0.017
12/11/2001	8.3	5	191	321	0.10	185	<0.005	0.0003	103	<0.0002	<0.001	<0.03	<0.03	15.8	0.02	0.0002	<0.0005	4.27	0.013
1/27/2002	8.2	<3	188	325	0.01	180	<0.005	0.0005	103	<0.0002	0.001	0.10	<0.03	16.3	0.017	0.0003	<0.0005	4.16	0.013
2/24/2002	8.0	5	188	332	0.11	182	0.015	0.0003	106	<0.0002	0.003	0.22	<0.03	16.4	0.018	<0.0002	0.0005	4.35	0.012
3/27/2002	8.2	<3	193	277	0.10	192	<0.005	0.0002	88	<0.0002	<0.001	0.04	<0.03	13.9	0.017	0.0003	0.0006	3.63	0.009
4/22/2002	7.8	<3	183	303	0.12	166	<0.005	0.0002	96	<0.0002	0.002	0.05	<0.03	15.2	0.018	0.0003	0.0005	4.32	0.013
5/6/2002	8.1	<3	178	283	<0.005	167	0.009	0.0002	92	<0.0002	0.001	0.17	<0.03	13.0	0.019	0.0002	<0.0005	4.06	0.014
6/28/2002	8.4	<3	190	345	0.09	196	0.050	<0.0002	111	0.0002	0.001	0.04	0.12	16.8	0.024	0.0002	<0.0005	4.32	0.013
7/28/2002	8.0	<3	196	352	0.07	193	<0.005	0.0009	112	<0.0002	<0.001	0.05	<0.03	17.7	0.017	<0.0002	0.0008	4.40	0.012
8/24/2002	8.1	<3	196	319	0.10	187	<0.005	0.0003	102	<0.0002	<0.001	0.04	<0.03	15.7	0.021	<0.0002	<0.001	4.15	0.092
9/23/2002	8.0	<3	160	269	0.10	140	0.037	0.0003	88	<0.0002	0.001	0.91	0.08	12.2	0.016	0.0004	0.0007	4.15	0.012
10/7/2002	8.1	<3	174	302	0.09	162	0.109	0.0003	98	<0.0002	0.001	0.33	0.21	14.1	0.021	0.0024	<0.0005	4.44	0.012
11/4/2002	8.1	<3	188	316	0.15	175	<0.005	0.0003	100	<0.0002	0.001	0.20	<0.03	16.3	0.018	0.0002	<0.0005	4.55	0.013
12/16/2002	8.0	3	174	271	0.10	143	0.016	0.0003	86	<0.0002	<0.001	<0.03	<0.03	13.4	0.015	0.0002	<0.0005	4.36	0.011
1/6/2003	8.1	<3	37	212	0.02	36	0.012	0.0002	69	<0.0002	0.001	0.13	<0.03	9.9	0.012	0.0002	0.0006	4.01	0.012
4/22/2003	7.9	4	172	285	0.10	138	<0.005	0.0002	92	<0.0002	0.004	0.11	<0.03	13.6	0.014	0.0002	<0.0005	4.43	0.014
6/4/2003	8.1	<3	180	366	0.09	171	0.192	0.0004	118	<0.0002	0.001	8.36	0.10	17.4	0.014	0.001	<0.0005	4.61	0.013
8/24/2003	8.2	<3	197	314	0.07	176	<0.005	0.0004	100	<0.0002	<0.001	<0.03	<0.03	16.0	0.011	0.0035	<0.001	4.23	0.010
10/7/2003	8.2	<3	181	300	0.10	151	0.010	0.0003	97	<0.0002	0.001	0.03	<0.03	14.1	0.012	0.0002	<0.001	4.28	0.015
12/17/2003	8.2	18	174	299	0.07	142	0.037	0.0004	97	<0.0002	0.001	4.10	0.10	13.9	0.026	0.0004	<0.001	4.34	0.012
Minimum	7.8		37	212	<0.005	36	<0.005	<0.0002	69		<0.001	<0.03		9.9	0.011	<0.0002		3.63	0.009
Median	8.1	<3	190	335	0.11	186	0.018	0.0003	107	<0.0002	0.001	0.08	<0.03	16.4	0.021	0.0003	<0.001	4.32	0.016
Maximum	8.4	51	206	459	0.48	317	1.300	0.0009	147	0.0012	0.019	8.36	2.67	26.6	0.214	0.0035	0.002	6.19	0.129

Table 34. Post-Closure Chemistry of Discharge from 300 Portal

DATE	pH	TSS mg/l	Alk mg/l	Hard mg/l	NO ₃ mg/l	SO ₄ mg/l	D-Al mg/l	T-As mg/l	D-As mg/l	D-Ca mg/l	D-Cd mg/l	D-Cu mg/l	T-Fe mg/l	D-Fe mg/l	D-Mg mg/l	D-Mn mg/l	D-Sb mg/l	D-Se mg/l	T-Si mg/l	D-Si mg/l	D-Zn mg/l
1/19/2000	7.9	4	191	612	1.81	425	0.008	0.0009	0.0003	205	0.0017	0.003	0.44	<0.03	24.2	0.476	0.0014	<0.2			0.388
1/31/2000	8.1	↘	193	536	1.63	428	<0.005	0.0006	0.0004	178	0.0014	0.001	0.15	<0.03	22.1	0.419	0.0014	<0.2			0.287
2/17/2000	8.0	↘	154	479	1.54	341	<0.005	0.0005	0.0005	160	0.0005	<0.005	<0.03	<0.03	19.2	0.194	0.0013	0.0015			0.122
3/22/2000	7.9	4	60	124	0.76	57	0.029	0.0012	0.0011	42	<0.0002	0.003	0.04	<0.03	4.4	0.005	0.0006	0.0011			<0.005
4/18/2000	7.8	↘	189	484	0.92	349	<0.005	0.0014	0.0005	161	0.0011	<0.001	0.51	0.04	19.9	0.381	0.001	0.0011	3.13	3.10	0.223
5/15/2000	7.8	↘	185	501	1.07	323	<0.005	0.0019	0.0008	172	0.0008	0.004	0.39	0.06	17.1	0.373	0.0014	0.0013	3.10	3.09	0.201
6/27/2000	7.7	↘	179	451	0.87	279	<0.005	0.0015	0.0006	155	0.0009	<0.001	0.42	0.05	15.9		0.0009				0.158
7/11/2000	8.1	↘	179	405	0.57	131	0.013	0.0017	0.0007	138	0.0006	<0.005	0.35	0.04	14.4	0.212	0.0008	0.0011	3.14	3.15	0.125
7/25/2000	7.8	↘	183	391	0.65	278	<0.005	0.001	0.0002	134	0.0005	<0.001	0.37	0.04	14.1	0.251	0.0008	0.0019	3.01	3.20	0.108
8/10/2000	8.0	↘	178	297	0.68	247	0.007	0.0005	0.0003	93	0.0006	0.001	0.13	<0.03	15.9	0.16	0.001	0.0016	2.98	3.10	0.087
8/14/2000	8.0	↘	175	317	0.66	242	<0.005	0.0004	0.0003	103	0.0006	<0.001	0.07	<0.03	14.8	0.111	0.0007	0.0018	3.05	2.96	0.073
9/25/2000	8.0	↘	180	374	0.61	238	0.005	0.0016	0.001	125	0.0007	<0.001	0.43	0.09	15.2	0.244	0.0017	0.0012	3.24	3.19	0.135
10/4/2000	7.9	↘	180	376	0.52	246	0.014	0.0019	0.0009	125	0.0008	<0.01	0.41	0.12	15.3	0.246	0.0009	0.0012	3.12	3.13	0.135
11/7/2000	8.1	↘	179	405	0.57	131	0.013	0.0017	0.0007	138	0.0006	<0.005	0.35	0.04	14.4	0.212	0.0008	0.0011	3.14	3.15	0.125
12/20/2000	8.2	↘	180	334	0.54	190	<0.005	0.002	<0.0002	113	0.0007	<0.001	0.42	<0.03	12.5	0.16	<0.0005	0.001	2.74	2.73	0.105
2/4/2001	8.0	↘	185	378	0.50	197	<0.005	0.0084	0.0006	129	0.0006	<0.02	1.86	0.07	13.4	0.141	0.0007	0.0012	3.17	3.11	0.118
3/12/2001	8.1	↘	188	433	0.40	240	<0.005	0.002	0.0007	149	0.0006	<0.001	0.43	0.05	14.8	0.157	0.0011	0.0006	3.09	3.11	0.101
4/23/2001	8.2	↘	171	348	0.44	200	0.007	0.0032	0.0006	117	0.0005	0.001	0.60	<0.03	13.4	0.111	0.0008	0.0009	3.15	3.12	0.073
5/11/2001	8.1	↘	179	373	0.61	219	<0.005	0.0016	0.0006	126	0.0006	<0.001	0.30	<0.03	14.2	0.124	0.0008	0.001	3.40	3.29	0.120
6/4/2001	8.1	↘	164	331	0.68	189	0.02	0.0017	0.0008	114	0.0006	<0.001	0.23	<0.03	11.6	0.14	0.0005	0.0015	3.35	3.27	0.101
8/13/2001	8.2	↘	188	419	0.37	237	0.005	0.0022	0.0006	140	0.0016	0.001	0.27	<0.03	17.0	0.138	0.0005	0.0015	3.35	3.40	0.076
9/14/2001	8.2	↘	190	419	0.39	217	<0.005	0.0016	<0.0005	140	0.0004	0.001	0.28	0.04	17.1	0.097	0.0006	0.0011	3.84	3.77	0.076
10/1/2001	8.2	↘	167	360	0.43	210	<0.005	0.0013	0.0006	121	0.0004	<0.005	0.23	<0.03	13.9	0.081	0.0006	0.0011	3.21	3.33	0.098
11/4/2001	7.9	↘	174	340	0.45	187	<0.005	0.0019	0.0009	115	0.0004	0.001	0.21	0.03	12.6	0.097	0.0007	<0.002	3.06	3.23	0.081
12/11/2001	8.3	↘	179	336	0.44	201	0.007	0.0019	0.0007	114	0.0004	<0.001	0.22	<0.03	12.6	0.091	0.0006	0.0008	3.21	3.22	0.079
1/27/2002	8.1	↘	175	334	0.43	194	<0.005	0.0019	0.0007	113	0.0004	0.001	0.21	<0.03	13.0	0.078	0.0006	0.0008	3.20	3.19	0.078
2/24/2002	7.6	↘	172	327	0.39	189	0.013	0.0017	0.0008	110	0.0004	0.001	0.19	0.03	12.5	0.086	0.002	0.0009	3.29	3.24	0.077
3/27/2002	8.2	↘	188	379	0.32	235	<0.005	0.003	0.0005	128	0.0004	<0.001	0.35	<0.03	14.4	0.131	0.0008	0.0008	3.13	3.33	0.075
4/22/2002	7.7	↘	172	322	0.38	195	0.005	0.0026	0.0009	108	0.0004	0.001	0.33	<0.03	13.0	0.094	0.0006	0.0007	3.17	3.16	0.060
5/6/2002	7.9	↘	170	296	0.42	186	<0.005	0.0016	0.0006	100	0.0004	<0.001	0.15	<0.03	11.2	0.064	0.0007	0.0007	3.10	3.08	0.065

DATE	pH	TSS mg/l	Alk mg/l	Hard mg/l	NO ₃ mg/l	SO ₄ mg/l	D-Al mg/l	T-As mg/l	D-As mg/l	D-Ca mg/l	D-Cd mg/l	D-Cu mg/l	T-Fe mg/l	D-Fe mg/l	D-Mg mg/l	D-Mn mg/l	D-Sb mg/l	D-Se mg/l	T-Si mg/l	D-Si mg/l	D-Zn mg/l
6/28/2002	8.4	<3	177	352	0.39	206	<0.005	0.0018	0.0005	119	0.0004	0.002	0.19	0.04	13.2	0.114	<0.0002	0.0007	3.13	3.09	0.083
7/28/2002	7.8	<3	193	364	0.03	207	<0.005	0.0021	0.0008	123	0.0003	0.001	0.20	<0.03	14.0	0.112	0.0004	<0.0005	3.31	3.27	0.08
8/24/2002	8.1	<3	184	361	0.31	220	<0.005	0.0019	0.0006	122	0.0003	<0.001	0.22	0.04	13.8	0.123	0.0007	0.001	3.27	3.30	0.084
9/23/2002	8.0	<3	144	298	0.47	169	<0.005	0.0016	0.0007	102	0.0003	0.001	0.15	<0.03	10.6	0.075	0.0007	0.0015	3.26	3.17	0.066
10/7/2002	8.0	<3	172	326	0.40	195	<0.005	0.0016	0.0007	110	0.0003	<0.005	0.18	<0.03	12.3	0.084	0.0028	0.0011	3.21	3.20	0.061
11/4/2002	7.9	<3	176	330	0.33	199	<0.005	0.0025	0.001	111	0.0004	0.001	0.22	0.04	12.8	0.101	0.0007	0.0009	3.25	3.20	0.070
12/16/2002	7.9	<3	173	317	0.40	183	<0.005	0.0022	0.0012	106	0.0003	<0.005	0.16	<0.03	12.6	0.076	0.0005	0.0011	3.24	3.26	0.061
1/6/2003	7.8	<3	176	306	0.42	172	0.007	0.0032	0.0009	102	0.0004	0.002	0.25	<0.03	12.2	0.07	0.0004	0.0013	3.20	3.26	0.057
2/22/2003	8.1	<3	179	332	0.35	178	0.008	0.0019	0.0008	111	0.0003	<0.001	0.10	<0.03	13.3	0.062	0.0012	0.0006	3.28	3.35	0.066
4/22/2003	7.8	<3	169	297	0.36	160	<0.005	0.0016	0.0008	99	0.0003	0.002	0.10	<0.03	12.3	0.052	0.0007	0.0007	3.24	3.24	0.063
6/4/2003	8.1	<3	172	359	0.40	193	<0.005	0.0016	0.0009	125	0.0003	<0.001	0.11	<0.03	11.6	0.068	0.0006	0.001	3.36	3.28	0.064
8/24/2003	8.2	<3	184	326	0.23	193	<0.005	0.0023	0.0012	109	0.0003	<0.001	0.10	<0.03	13.1	0.07	0.0088	0.001	3.18	3.18	0.058
10/7/2003	8.2	<3	174	301	0.41	168	<0.005	0.0017	0.0012	101	0.0003	<0.001	0.10	0.03	11.7	0.056	0.0005	<0.001	3.05	3.08	0.055
12/17/2003	8.1	<3	177	292	0.35	162	<0.005	0.0021	0.0011	98	0.0003	0.0019	0.10	<0.03	11.3	0.054	0.0005	<0.001	3.13	3.12	0.046
Minimum	7.6		60	124	0.03	57		0.0004	<0.0002	42	<0.0002		0.03		4.4	<0.005	0.0002	<0.0005	2.74	2.73	<0.005
Median	8.0	<3	179	350	0.43	200	<0.005	0.0017	0.0007	118	0.0004	<0.001	0.22	<0.03	13.4	0.111	0.0007	0.0011	3.18	3.19	0.080
Maximum	8.4	4	193	612	1.81	428	0.008	0.0084	0.0012	205	0.0017	0.003	1.86	0.12	24.2	0.476	0.0088	0.0019	3.84	3.77	0.388

**Table 35. Descriptive Statistics for Chemistry of Monitored Discharges
from Tailings Impoundment and the Underground Workings**

	Impoundment			300 Portal			180 Portal			130 Portal			Behind 130 Bulkhead		
	Min	Median	Max	Min	Median	Max	Min	Median	Max	Min	Median	Max	Min	Median	Max
pH	7.8	8.1	8.4	7.6	8.0	8.4	7.8	8.1	8.4	7.8	8.1	8.3	7.4	8.0	8.2
TSS mg/l		<3	37		<3	4		<3	51		<3	25			
Alk mg/l	61	111	152	60	179	193	37	190	206	109	138	173	177	214	255
Hard mg/l	116	245	384	124	350	612	212	335	459	192	249	418	409	558	693
NO3 mg/l	<0.005	0.26	0.81	0.03	0.43	1.81	<0.005	0.11	0.48	0.01	1.13	2.60			
SO4 mg/l	67	141	270	57	200	428	36	186	317	94	132	248	245	355	493
D-Al mg/l	<0.005	0.013	0.329		<0.005	0.008	<0.005	0.018	1.300		<0.005	0.074	<0.005	0.0105	0.023
T-As mg/l	0.0002	0.0007	0.0029	0.0004	0.0017	0.0084									
D-As mg/l	0.0002	0.0005	0.0011	<0.0002	0.0007	0.0012	<0.0002	0.0003	0.0009	<0.0001	0.0003	0.0056	<0.0003	0.00045	0.0008
D-Ca mg/l	40	84	135	42	118	205	69	107	147	63	81	137	138	187	235
D-Cr mg/l		<0.001	0.003		<0.001	0.03		<0.001	0.01		<0.001	0.01			
D-Cd mg/l		<0.0002	0.0003	<0.0002	0.0004	0.0017		<0.0002	0.0012		<0.0002	0.0043		<0.0002	0.0006
D-Cu mg/l	<0.002	0.002	0.004		0.004	<0.001	<0.001	0.0012	0.02	0.001	0.004	0.013		<0.005	0.016
T-Fe mg/l	<0.03	0.09	3.4	0.03	0.22	1.86	<0.03	0.08	8.36		<0.03	0.09	0.07	0.56	1.27
D-Fe mg/l		<0.03	0.47		<0.03	0.12		<0.03	2.67		<0.03	0.46	<0.03	0.09	0.65
D-Mg mg/l	3.8	8.1	12.0	4.4	13.4	24.2	9.9	16.4	26.6	8.5	11.2	20.0	15.7	22.2	27.4
D-Mn mg/l	<0.005	0.063	0.467	<0.005	0.111	0.476	0.011	0.021	0.214		<0.005	0.065	0.085	0.549	0.692
D-Mo mg/L			<0.03			<0.03			<0.03			<0.03			<0.03
D-Ni mg/L		<0.001	0.004	0.001	0.003	0.017		<0.001	0.007	0.001	0.002	0.011	0.002	0.008	0.025
D-Pb mg/l		<0.001	0.038		<0.001	0.005		<0.001	0.009		<0.001	0.005		<0.001	0.032
D-Sb mg/l	<0.0001	0.0005	0.0019	0.0002	0.0007	0.0088	<0.0002	0.0003	0.0035	<0.0001	0.0009	0.0016	<0.0005	0.00145	0.0025
D-Se mg/l		<0.0005	0.0011	<0.0005	0.0011	0.0019		<0.001	0.002	<0.0005	0.001	0.0016		<0.0005	0.001
T-Si mg/l	1.23	2.07	3.83	2.74	3.18	3.84									
D-Si mg/l	1.01	1.93	2.95	2.73	3.19	3.77	3.63	4.32	6.19	2.62	2.80	3.44			
T-Zn mg/l		<0.005	0.027	<0.005	0.082	0.386							0.016	0.4365	0.654
D-Zn mg/l		<0.005	0.027	<0.005	0.080	0.388	0.009	0.016	0.129	0.011	0.023	0.051	<0.005	0.057	0.583

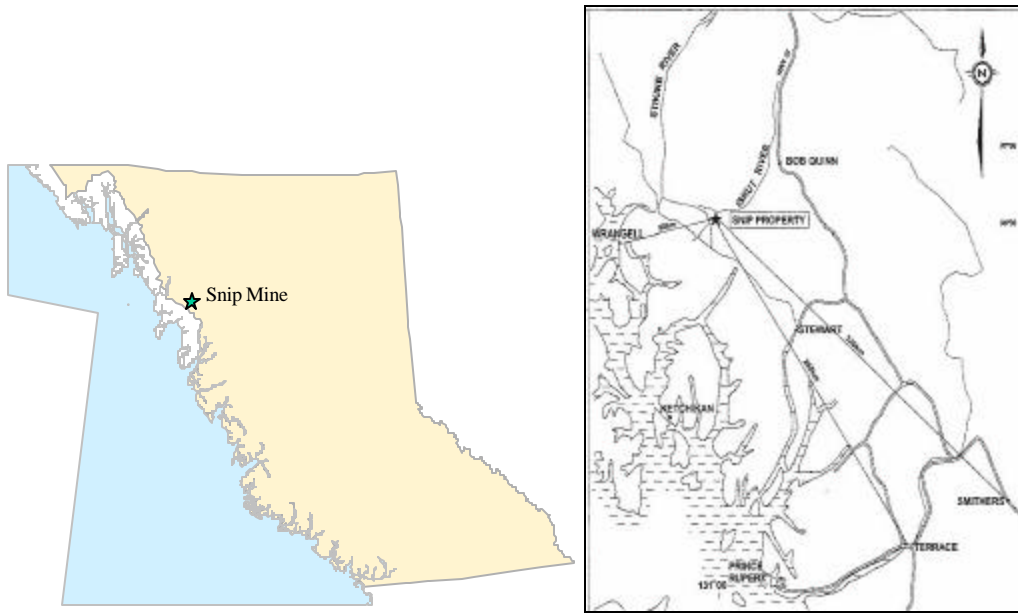


Figure 1. Location of the Snip Mine on a Map of British Columbia

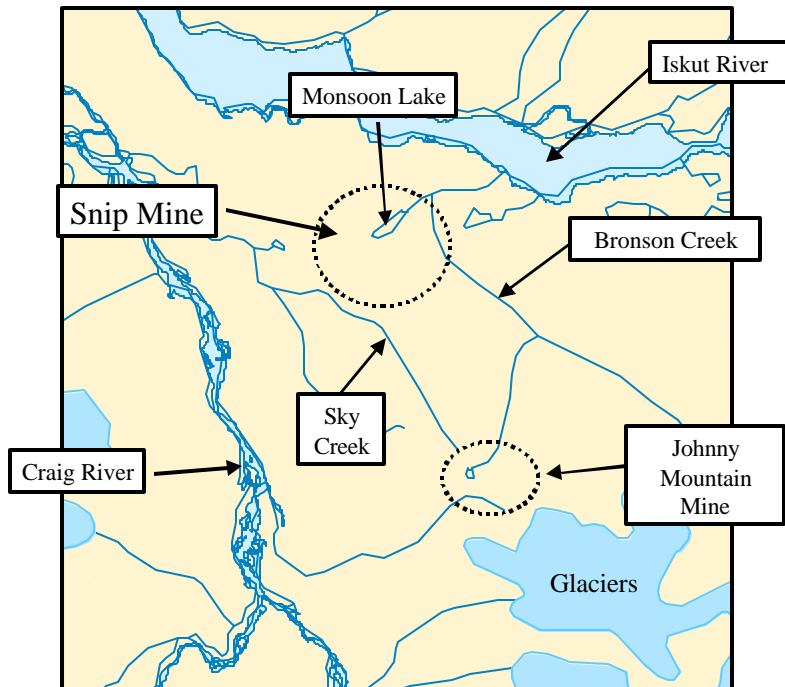


Figure 2. Map of Immediate Area Around Snip Mine

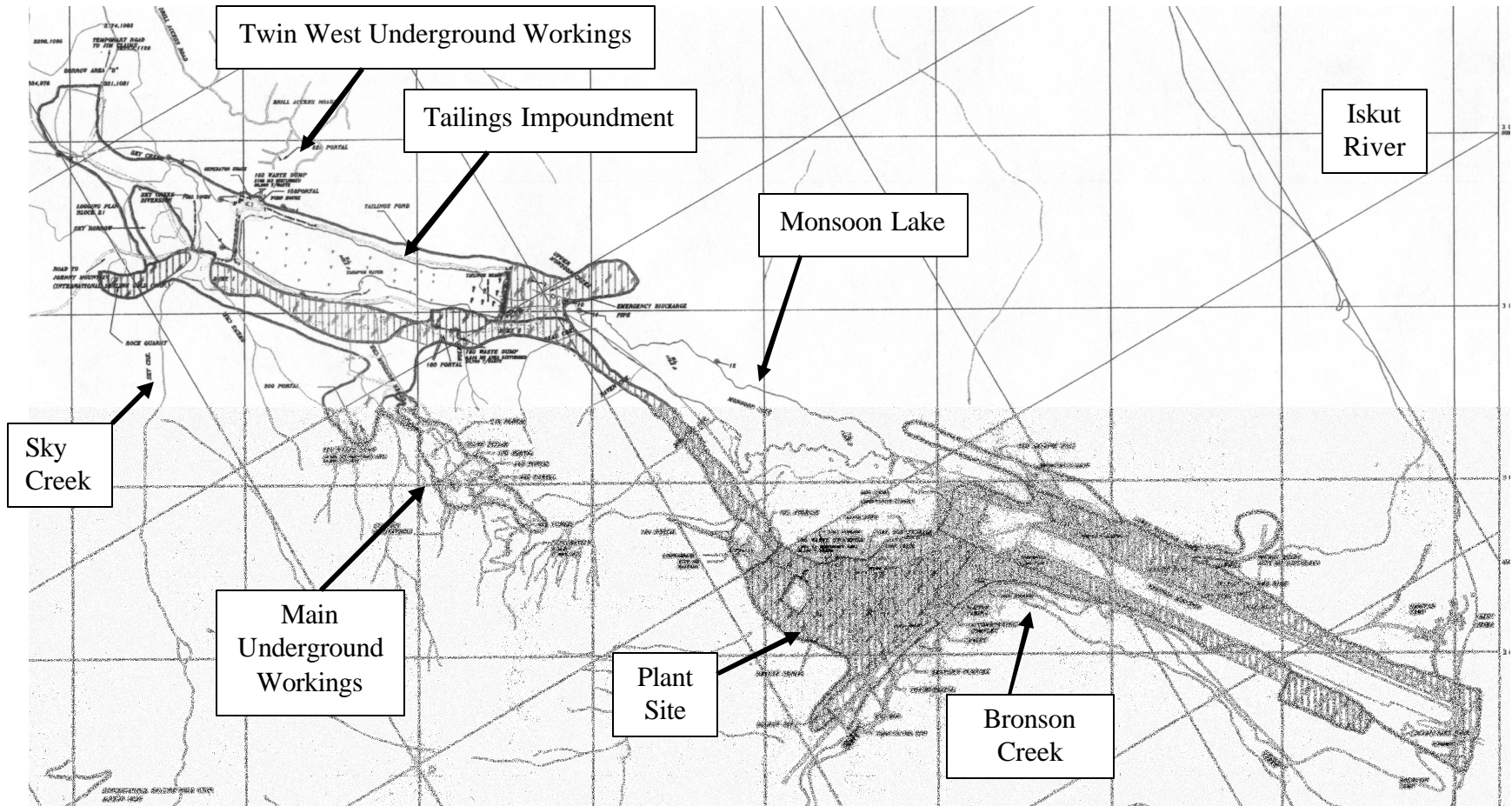


Figure 3. Map of the Snip Mine



Figure 4. Map of the Tailings Impoundment and Underground Workings

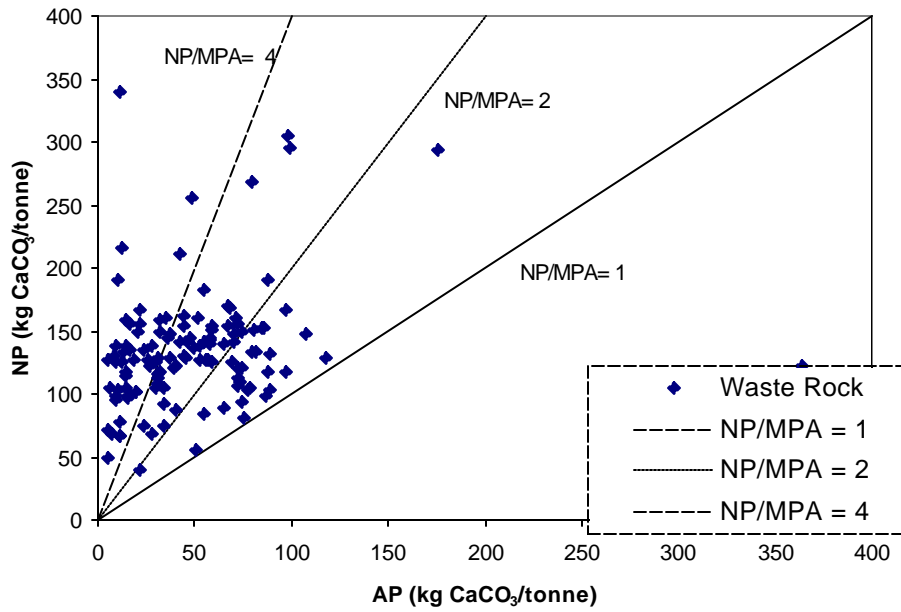


Figure 5. NP Versus AP for Waste Rock Samples Collected During Mining

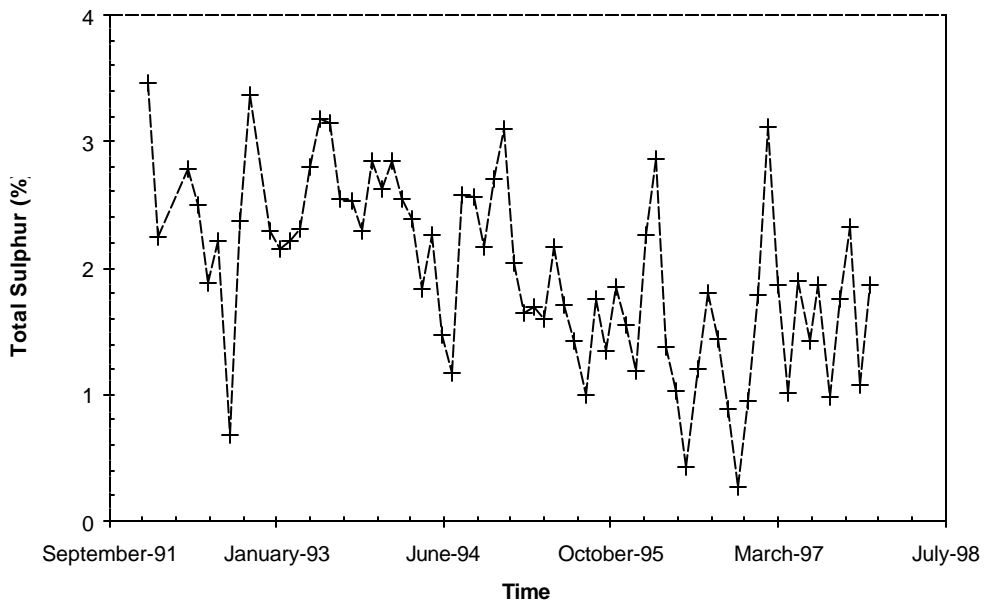


Figure 6. Total %S Versus Time for Waste Rock Samples Collected During Mining

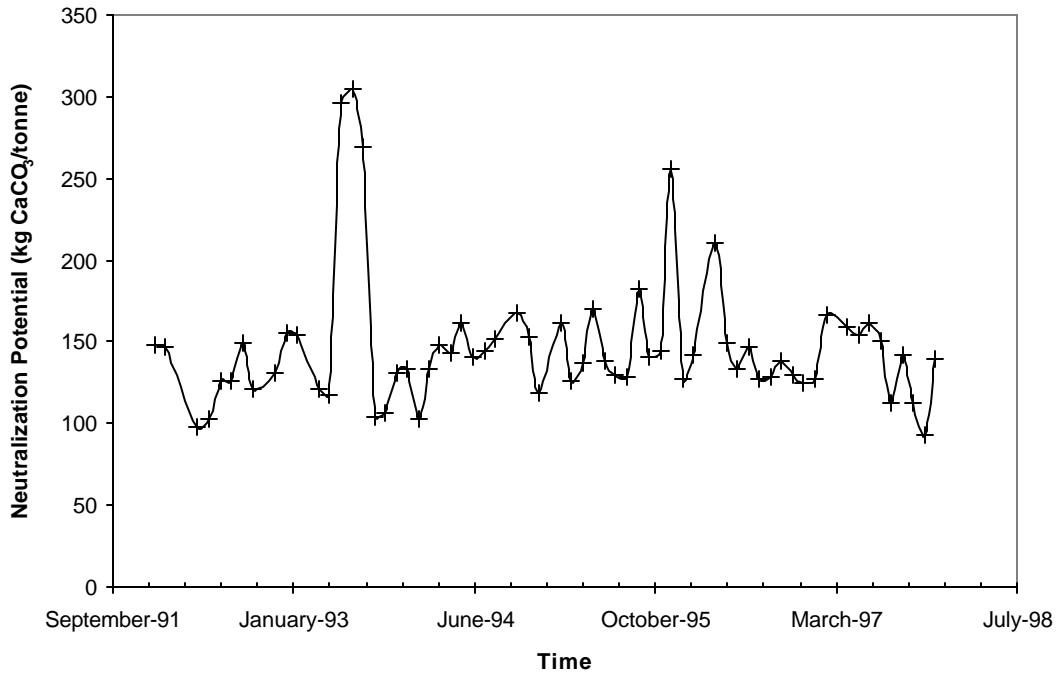


Figure 7. NP Versus Time for Waste Rock Samples Collected During Mining

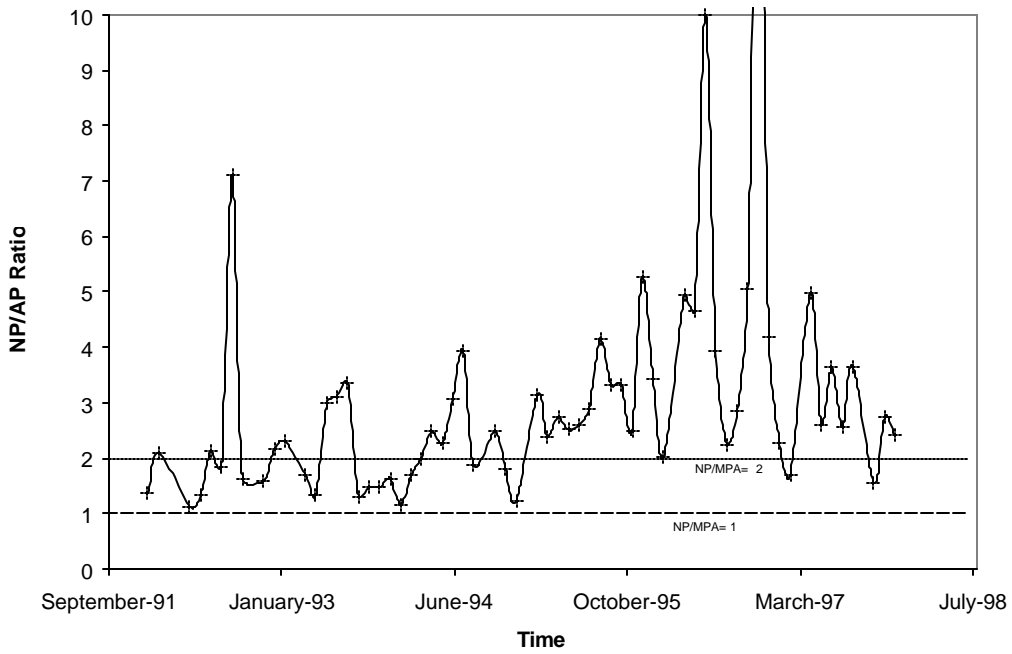


Figure 8. NPR Versus Time for Waste Rock Samples Collected During Mining

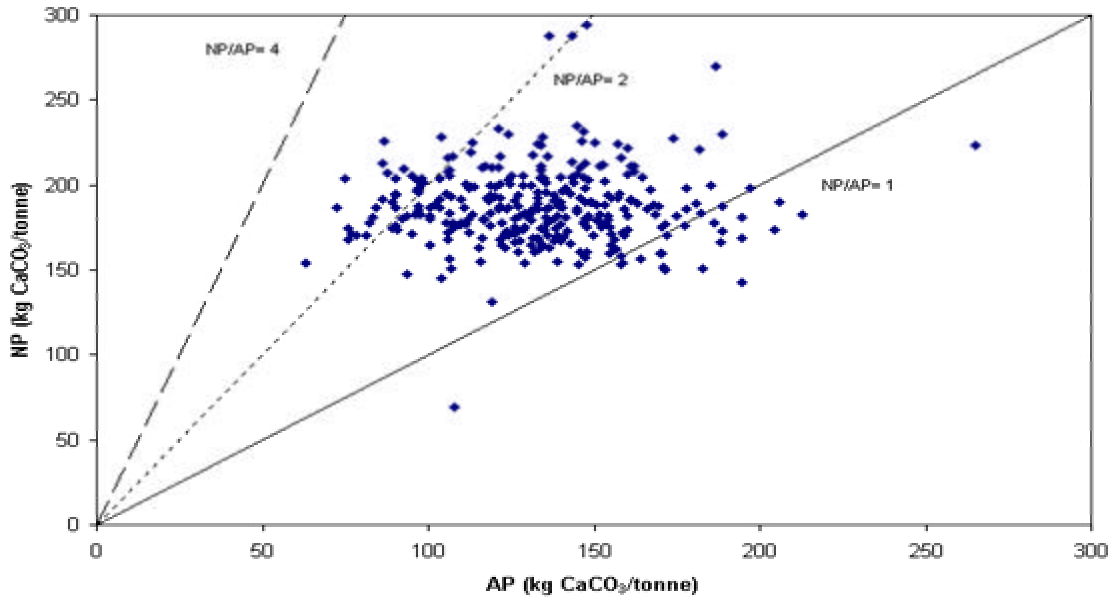


Figure 9. NP Versus AP for Whole Flotation Tailings Samples Collected During Mining

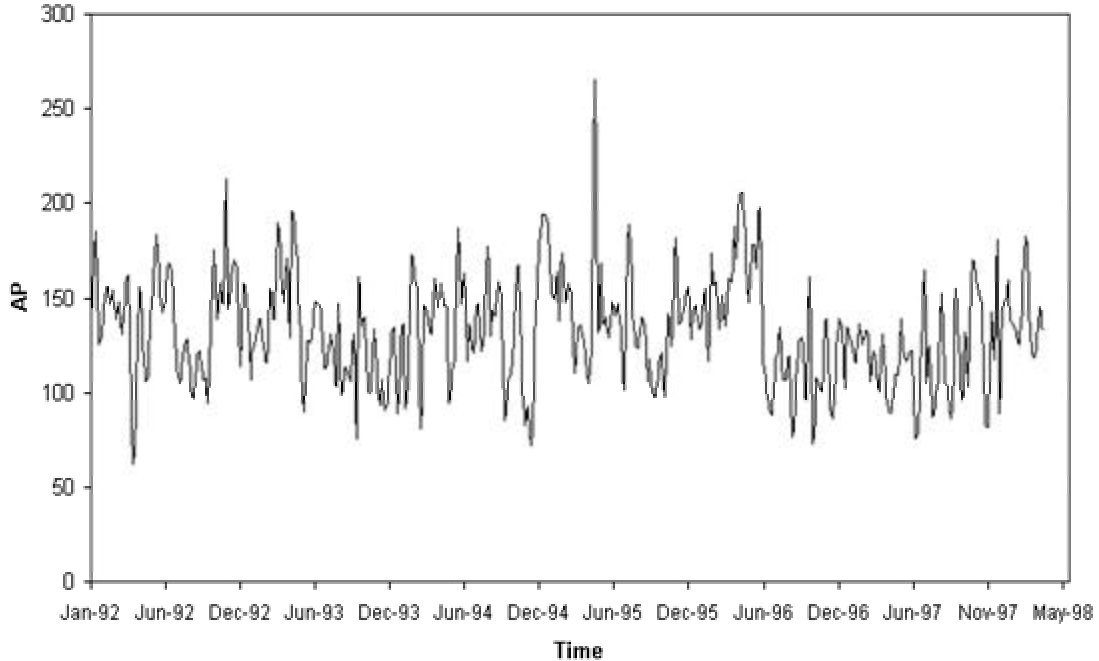


Figure 10. AP Versus Time for Whole Flotation Tailings Samples Collected During Mining

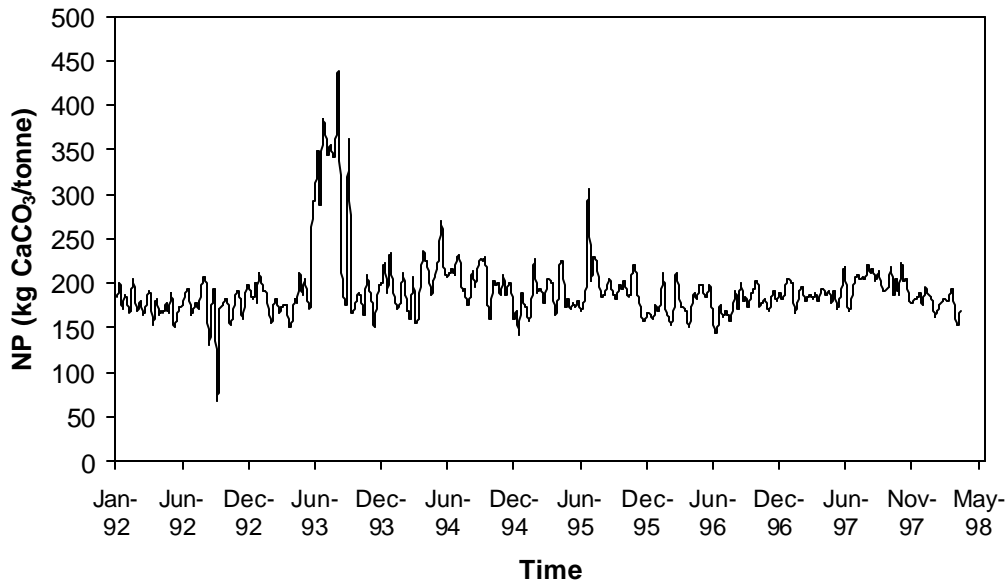


Figure 11. NP Versus Time for Whole Flotation Tailings Samples Collected During Mining

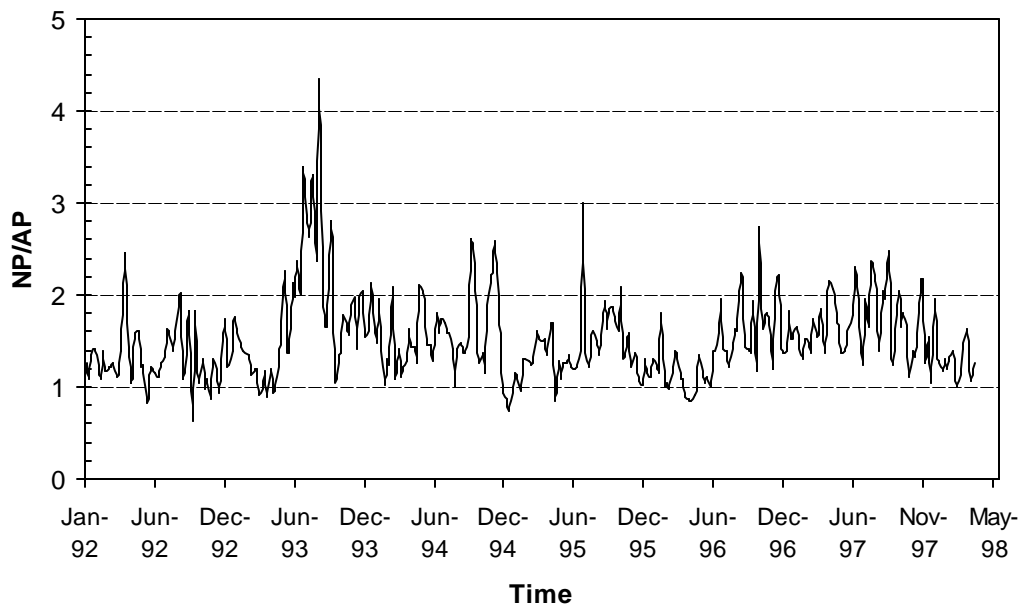


Figure 12. NPR Versus Time for Weekly Whole Flotation Tailings Samples Collected During Mining

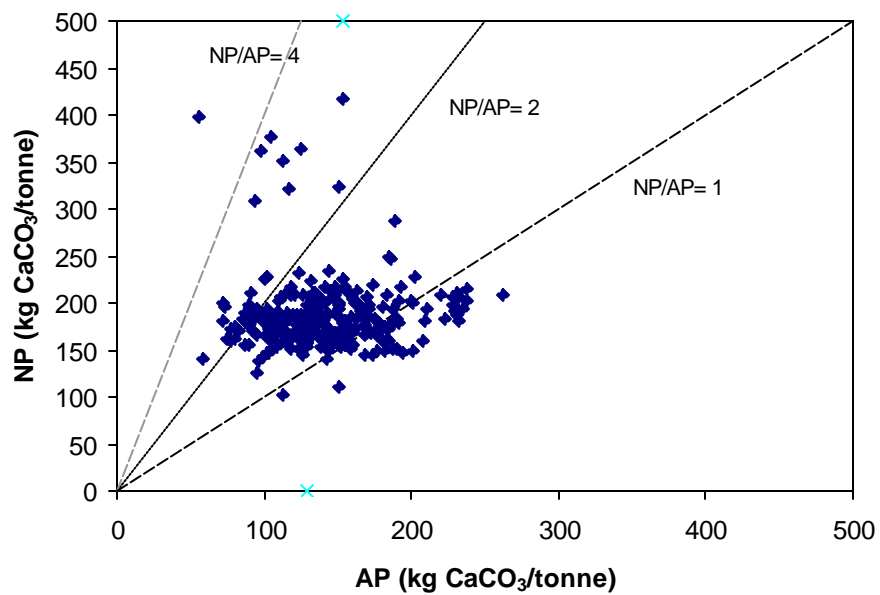


Figure 13. NP Versus AP for Backfill Tailings Sand Samples Collected During Mining

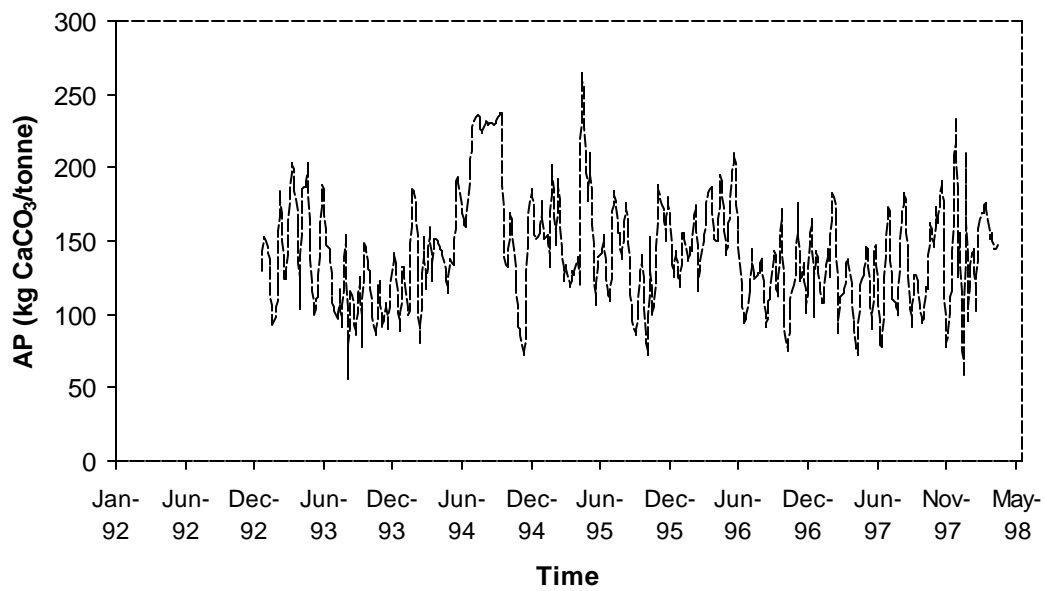


Figure 14. AP Versus Time for Backfill Tailings Sand Samples Collected During Mining

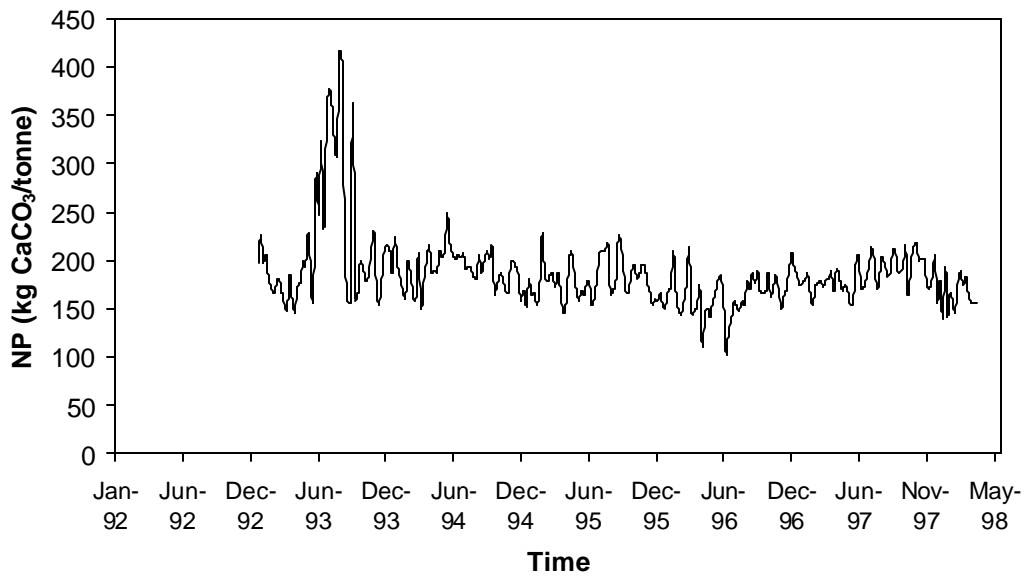


Figure 15. NP Versus Time for Backfill Tailings Sand Samples Collected During Mining

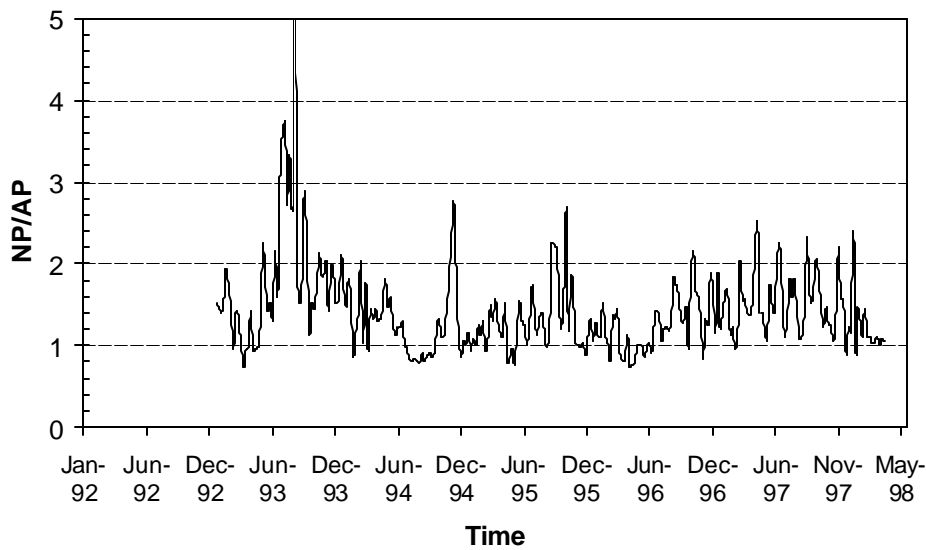


Figure 16. NPR Versus Time for Backfill Tailings Sand Samples Collected During Mining



Photo 1. Plant Site and Camp with Bronson Creek in the Background



Photo 2. Tailings Impoundment with Main Underground up the Slope to the Right

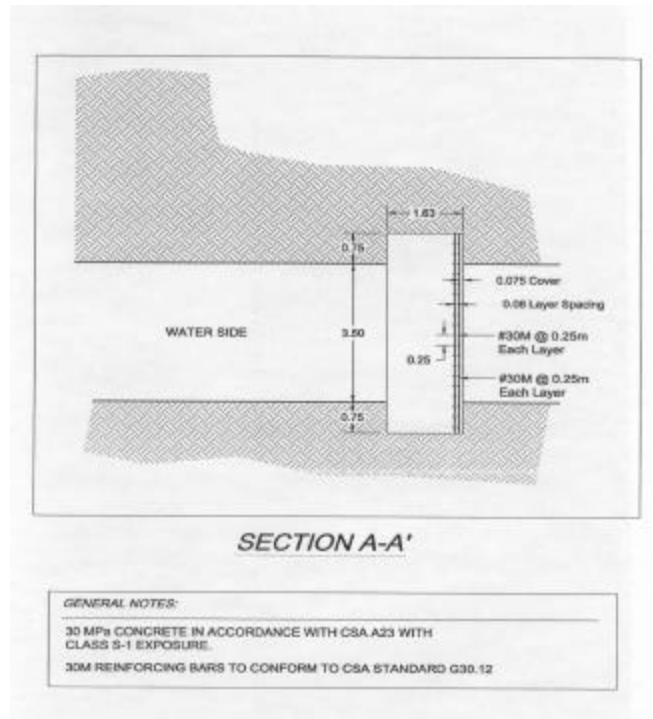


Photo 3. Design of the Bulkhead

Appendix A

**Assessment of the ARD Potential and Considerations in
Setting Criteria for Potentially ARD Generating Materials**

and

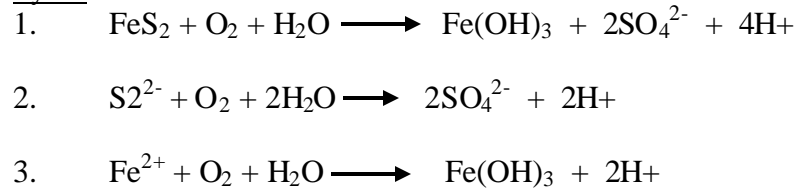
Appendix B

**Calculation of the Rate of CaCO₃ Depletion from Drainage Chemistry
in Humidity Cell Test**

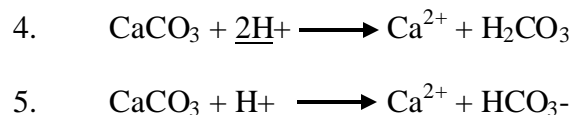
Appendix A: Assessment of the ARD Potential and Considerations in Setting Criteria for Potentially ARD Generating Materials

Acidity is produced when iron sulphide and certain other sulphide minerals are exposed to oxygen and water. The acid producing reactions are oxidation of the sulphide component and oxidation and hydrolysis of the accompanying metals. For pyrite, the overall reaction, which is shown in reaction 1 produces two moles of acidity per mole of sulphide-S. The reactions of the sulphide and metal components are shown in reactions 2 and 3, respectively.

Pyrite



Acidic rock drainage (ARD) will only result if the neutralization from minerals such as calcite is insufficiently plentiful and reactive to neutralize the acidity generated from the sulphide minerals. In ARD test work, commonly the first step in assessing whether the neutralizing minerals in a sample are sufficiently plentiful and reactive to neutralize the acidity generated from the oxidation of iron sulphide minerals is to calculate the acid potential (AP) and neutralizing potential (NP). The AP and NP are calculated from laboratory measurements. The ARD potential is then predicted from the NP/AP ratio (NPR). Assuming the AP and NP are accurate and there is exposure to air and leaching, ARD is judged likely if the NPR is < 1, uncertain if the NPR is 1 to 2 and of low probability if the NPR is > 2. The basis for these NPR criteria is the assumption that sulphide-S produces 2 moles of acidity per mole of S (reaction 1) and the following two acid (H⁺) neutralization reactions.



Assuming sulphide-S produces 2 moles of acidity per mole of S, reaction 4 corresponds to an NPR of 1 and reaction 5 corresponds to an NPR of 2. Reaction 4 predominates below pH 6.4. Reaction 5 predominates above. Under macro-scale neutral pH weathering conditions, neutralization by calcite likely occurs at a micro-site pH values both above and below 6.4. Thus, assuming no “errors” in AP and NP measurement, each mole of CaCO₃ is able to neutralize somewhere between 1 and 2 moles of sulphide-S, rather than 1 mole, and the NPR required to generate ARD will be between 1 and 2. The NPR required to generate ARD will be closer to 1 if the micro-scale pH is below 6.4 or the HCO₃⁻ generated from CO₃ minerals in reaction 5 is retained in the pore water and neutralizes subsequent acidity. The extent to which the HCO₃⁻ generated in reaction 5 contributes neutralization will depend on the chemistry and hydrogeology of the materials in question.

In order to be quick and repeatable, procedures used to measure AP and NP are a crude approximation of the large number of factors and processes that contribute to acid generation and

neutralization in the field. The subsequent calculations involve a number of assumptions that may be incorrect. Consequently, a key part of the assessment of potentially ARD generating materials is the manner in which AP and NP are measured and the resulting discrepancies with acid generation and neutralization in the materials under the mine site conditions. Corrections may be required to take into account site-specific conditions or differences from the assumptions regarding AP and NP mineralogy. Corrections or safety factors may also be used to account for sampling limitations, the heterogeneity of key properties, and the composition of sample (e.g., drill cuttings created from whole rock) versus actual reactive portion of the material (e.g., dump fines).

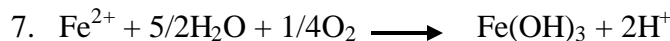
i. Possible Differences Between Actual Field and Measured NP

There are two forms of laboratory NP measurement. Bulk-NP procedures measure the ability of a sample to neutralize a known volume and strength of strong acid. The bulk-NP is a measure of the neutralization available in CO₃ minerals and the more reactive silicate minerals. The most commonly used measure of bulk-NP is the Sobek procedure. The second type of laboratory NP measurement is the carbonate mineral NP (CO₃¹-NP), which is calculated from the % total or inorganic C or CO₂, assuming all the CO₃ is calcite. Both the bulk- and CO₃-NP are reported as kg CaCO₃/t. If all the carbonate is Ca and Mg species, the difference between the bulk-NP and CO₃-NP is the contribution of reactive silicates to the bulk-NP.

As stated previously, very different conditions and processes may occur in laboratory NP measurements or are assumed in the calculations compared to the actual materials under field weathering conditions. It is therefore important to identify discrepancies and if required, make corrections. One of the main factors of concern is the discrepancy between the assumed and actual minerals contributing to the neutralization.

Impact of Fe and Mn Carbonate on the CO₃- and Bulk-NP

The occurrence of Fe and Mn CO₃-containing minerals, such as ankerite [Ca(Ca,Mg,Fe)CO₃] and siderite (FeCO₃) raises concerns regarding the accuracy of both the bulk- and CO₃-NP.



As shown in reactions 6 and 7 above, the dissolution of Fe (or Mn CO₃) initially consumes acidity in a similar manner to calcite (reaction 4). However under aerobic conditions², the subsequent oxidation and hydrolysis of Fe or Mn produces equivalent acidity (reaction 7) to that consumed, so overall there is no neutralization. Thus when significant Fe or Mn CO₃ is present,

¹ CO₃-NP may be named the according to the carbon assay used for its calculation. For example, TIC-NP if calculated from total inorganic carbon, TC-NP if calculated from the Leco measurement of total carbon or CO₂-NP if a CO₂ assay is used.

² Anoxic conditions inhibit the oxidation and hydrolysis reactions.

the carbonate mineral NP ($\text{CO}_3^3\text{-NP}$), calculated from the % total or inorganic C or CO_2 , assuming all the CO_3 is CaCO_3 , will significantly overestimate the CO_3 neutralizing capacity under aerobic conditions.

Fe and Mn CO_3 have less impact on the bulk-NP, and as a result if there is significant Fe and Mn CO_3 , $\text{CO}_3\text{-NP} > \text{Sobek-NP}$, the reverse of what is otherwise observed. Where there is no detailed data on CO_3 mineralogy, $\text{CO}_3\text{-NP} > \text{Sobek-NP}$ is often the first indication that significant Fe and Mn CO_3 are present.

While the influence is less direct and therefore smaller than that on $\text{CO}_3\text{-NP}$, there is also a potential for Fe or Mn CO_3 minerals to contribute to the bulk-NP. Potential overestimation of NP in bulk-NP laboratory tests, such as the Sobek procedure, as result of Fe and Mn carbonates have been widely recognized for some time (Lapakko, 1994). The potential contribution results from the relative fast rate of carbonate dissolution when the acid is added (reaction 6), the neutralizing part of the reaction at low pH, compared to the shorter period of time at neutral pH during the back-titration and the slower rates of the acid-generating metal oxidation and hydrolysis (reaction 7). There is therefore a potential that the back titration will conclude before all the acidity from oxidation and hydrolysis has been produced. The likelihood of incomplete oxidation and hydrolysis is higher for MnCO_3 than FeCO_3 because these reactions are slower for Mn than Fe. Fortunately Mn CO_3 is relatively rare.

Potential ways of increasing the acid generating metal oxidation and hydrolysis reactions and thus minimizing the contribution of Fe or Mn CO_3 minerals to the bulk-NP include:

- titrating to pH 8.3 instead of 7, which increases the OH concentration and thus the rate of hydrolysis; and
- using a modification to the standard Sobek method where peroxide is added to ensure complete the oxidation and hydrolysis of Fe and Mn CO_3 (Meek, 1981; Skousen et al., 1997; White et al., 1998).

It is important to recognize when significant Fe and Mn CO_3 minerals are present. Presently, the most cost-effective procedure is to use Rietveld XRD analysis to quantify the proportion of different carbonate minerals and microprobe analysis to ascertain the proportion of Fe, Mn, Ca and Mg in carbonate minerals such as ferrous dolomite and ankerite with a variable composition. The cost of Rietveld XRD and microprobe analysis (approximately \$200/sample for the Rietveld and \$100/sample for the microprobe analysis of 6 samples with 5 grains analyzed per sample) is minor compared to the costs associated with errors in NP assessment and the resulting material characterization.

³ $\text{CO}_3\text{-NP}$ may be named according to the carbon assay used for its calculation. For example, TIC-NP if calculated from total inorganic carbon, TC-NP if calculated from the Leco measurement of total carbon or $\text{CO}_2\text{-NP}$ if a CO_2 assay is used.

Impact of Mg Carbonate-Containing on the CO₃-NP

Due to the lower atomic weight of Mg versus Ca (24 versus 40), MgCO₃ (formula weight of 84.32) provides approximately 18% more neutralization per unit C than CaCO₃ (formula weight of 100.09). This will not affect the accuracy of the Sobek-NP. However if significant Mg is present, % CO₂-NP calculated assuming all the CO₃ is CaCO₃ will underestimate the carbonate NP. The most common Mg containing CO₃ mineral is dolomite. Ankerite often also contains significant Mg, in addition to Ca, Fe and possibly Mn CO₃. Mg may occur as a trace constituent in calcite and siderite, but typically the concentration is low and no correction is required. Notably, the presence of significant MgCO₃ may counteract some or all of the affect of non-neutralizing Fe and Mn CO₃.

Potential Correction to Bulk-NP for Too High a Contribution of Silicate Minerals

The objective of bulk-NP procedures is to measure the neutralization present in carbonate and the most reactive silicate minerals. Theoretically silicate minerals have a large neutralization potential. But due to their slow reaction rate at neutral pH, they are only capable of maintaining a neutral pH if the rate of acid generation is very slow and only then when the most reactive silicate minerals are present. Silicate minerals are only likely to significant neutralizers if there is a low sulphide content, either initially or after carbonate minerals have neutralized the bulk of the AP.

An important part of ABA assessment is estimating the contribution of silicate minerals to the bulk-NP, determining whether these minerals are sufficiently reactive to provide neutralization at neutral pH and deciding whether a correction is required to remove the contribution of insufficiently reactive minerals. If materials have an NPR < 1, this correction is rarely made because these materials are already clearly PAG (NPR < 1), and CO₃-NP is typically used in calculations of the time to NP depletion. The only situation with materials that have an NPR < 1, where the Sobek-NP may be used and a 'silicate' correction would be required is if uncertainty regarding the contribution of Fe and Mn CO₃ means the CO₃-NP cannot be used in calculations of the time to NP depletion.

The more common situation requiring an assessment of whether a 'silicate' correction is required is when the ARD potential is uncertain (e.g., NPR calculated with CO₃-NP of 1 to 2). A comparison of the CO₃-NP and bulk-NP is commonly used to roughly assess the relative amounts of CO₃-NP and silicate-NP in the bulk-NP. See previous for discussion of potential impact of Fe, Mn and Mg carbonate on accuracy of CO₃-NP. Once the silicate-NP has been determined, quantitative mineralogical information is required to assess the potentially contributing silicates minerals and their theoretical reactivity. If procedures such as the Sobek-NP or modified-NP are conducted properly and there is no Fe and Mn CO₃ present, the amount is 5 to 15 kg CaCO₃/t higher than the CO₃-NP (Price and Kwong, 1997). 5 to 15 kg CaCO₃/t is the contribution of silicate minerals.

A much higher contribution of silicate-NP results from either the presence of minerals that are soluble at low pH (e.g., Mg silicates) or the addition of too much acid, additions far in excess of the neutralizing CO₃-NP. The addition of too much acid is the most frequent error in how the Sobek procedure is conducted. It results in an unrealistic estimate of the NP because minerals

that are soluble at low pH are not necessarily soluble at neutral pH. XRD will indicate whether minerals that are potentially soluble at low pH are present.

A simple way to check whether excess acid was added is to compare the amount of acid added (check the fizz rating) with the CO₃-NP and resulting Sobek-NP values.

Amount of Acid Corresponding to Each Sobek Fizz Rating:

None	20mL of 0.1 N HCl	= 50 kg CaCO ₃ /tonne
Slight	40 mL of 0.1 N HCl	= 100 kg Ca CO ₃ /tonne
Moderate	40 mL of 0.5 N HCl	= 500 kg CaCO ₃ /tonne
Strong	80 mL of 0.5 N HCl	= 1000 kg CaCO ₃ /tonne

The Sobek procedure measures the acidity consumed when 20 or 40 ml of 0.1 N or 40 or 80 ml of 0.5 N HCl is added to a sample. The objective of the Sobek procedure is measure the CO₃-NP and the most reactive silicate-NP. This achieved by adding slightly more acid than is required to react with the Ca and Mg CO₃. The selected volume and strength of acid added is based on the fizz reaction created by a couple of drops of 25% HCl. The strength of the effervescence roughly corresponds to the amount of calcite present. The acid addition should be only slightly higher than the CO₃-NP and the resulting Sobek-NP values, and should be repeated using a more appropriate acid addition if either is not the case. Test results indicate that the main concern is the incorrect use of moderate and strong fizz rating because this involves a large increase in acid compared to the slight rating. For example, moderate or strong fizz ratings are too high if the CO₃-NP and resulting Sobek-NP are less than 50 kg CaCO₃/tonne. Important considerations resulting from the above include:

- the need to report the fizz rating,
- the potential to substitute the CO₃-NP for the fizz rating in selecting the appropriate acid addition, and
- the potential problems caused by an uncertain, but potentially significant amount of Fe and Mn CO₃.

ii. Possible Differences between Actual and Measured AP

Possible reasons for significant differences between actual and laboratory measurements of AP are as follows.

Acid Soluble or Acid Insoluble Sulphate

AP is calculated directly from total-S, without correcting for the portion of acid soluble (e.g., gypsum) or acid insoluble⁴ sulphate (e.g., barite). In most unweathered rock, the concentration of sulphate-S is usually low and its contribution to total-S is only a factor where the NPR is uncertain or the total-S is relatively low. However, there are many instances where the concentration of sulphate-S is significant (e.g., Boss Mountain, Huckleberry and Kemess North)

⁴ The term insoluble is used here as a relative term for minerals whose dissolution is almost nil.

and it is therefore important to check. General information can be obtained from mineralogical data. The concentration of acid soluble sulphate species, such as gypsum and anhydrite, should be measured directly as part of the ABA analysis. The primary acid insoluble sulphate-S minerals are Ba and Pb SO₄. Sr SO₄ is also potentially insoluble. Acid insoluble sulphate-S can be subtracted by measuring sulphide-S directly rather than by subtracting acid soluble sulphate-S from total-S. As a gauge of whether it is important, the potential concentration of acid insoluble sulphate can be estimated from the concentration of Ba, Pb and Sr. 3000 ppm Ba corresponds to approximately 700 ppm or 0.07% barite-S (see p 48 of Price, 1997). Microprobe and XRD analysis can be used if more accurate estimates of the acid insoluble sulphate are required.

More or Less Acidity Per Mole of Sulphide-S than Pyrite

Some of the measured sulphide-S may be in minerals that produce more or less acidity per mole of sulphide-S than pyrite. Again this factor is most likely to be significant if the NPR is uncertain or the total-S is relatively. A crude estimate of the concentration of non-pyrite sulphides can be made from mineralogical analysis or the concentration of elements that are primarily found as sulphides.

Concentration in Fine Fraction of Waste Rock

Sulphide or neutralizing minerals may occur in veins or on fractures and therefore preferentially report to the finer particles or occur on surfaces. As a result they are more reactive per unit weight compared to neutralizing minerals, resulting in an effective NPR that is lower than the overall NPR values. This is a concern in waste rock, where the fines (< 2 mm grains) will be almost entirely exposed to oxygen and water, versus the coarse fragments, where most of the minerals are occluded and unable to react. Often the NPR of the reactive fines is significantly lower than that predicted from a 'whole waste rock' ABA. The converse is also possible.

Segregation after Deposition of Tailings

For tailings, the concern is with the composition of the sandy material that settles near the discharge point versus that of finer slimes in the center of the impoundment. The sandy material is both more likely to contain heavy minerals like sulphides and be well drained, and is therefore more of a concern than slimes, which due to their silt-size are likely to remain saturated.

Physical Occlusion by Other Minerals

Sulphide minerals may be physically occluded by other minerals such as quartz and prevented from oxidizing. Blasting and handling of waste rock, and crushing and grinding of tailings should break coatings observed in the original rock, at least in particle sizes at or below the grain size.

Appendix B: Calculation of the Rate of CaCO₃ Depletion from Drainage Chemistry in Humidity Cell Test

The rate of CaCO₃ depletion can be estimated from drainage chemistry in humidity cell test, either by using the mg/kg sulphate to indirectly calculate the acid neutralization demand or mg/kg Ca to calculate calcite dissolution.

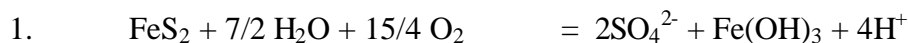
Use of mg/kg Ca to Calculate Calcite Dissolution

The mg/kg Ca is converted to mg/kg calcite by multiplying by 2.4973, the formula weight of calcite (100.09) divided by the formula weight of Ca (40.08). In this calculation, all Ca is assumed to come from calcite. Other potential sources of soluble Ca include Ca sulphate minerals, such as anhydrite and gypsum, and silicate minerals, including plagioclase and various pyroxenes and amphiboles. While the solubility of Ca silicates is usually considerably lower than calcite and the contribution of Ca sulphate can be avoided by not using the Ca leaching rate from the initial period of the test when these minerals typically dissolve, this is not always the case. The assessment of whether all the Ca comes from calcite should include review of sample mineralogy and ABA data for potential Ca sources and review of drainage chemistry data, such as sulphate and the inferred acid generation, for indications that calcite is the only significant Ca source.

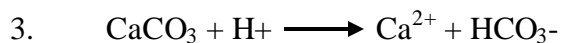
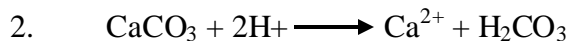
Where Mg carbonate is also a significant neutralization source, use of the Mg leaching rate to calculate the depletion of Mg carbonate can be done in a similar manner to Ca carbonate. This should include a similar check of other potential sources of soluble Mg. The formula weight is 24.312 for Mg and 84.322 for Mg carbonate.

Use of mg/kg Sulphate to Calculate Calcite Dissolution

The following reaction is used to estimate acid generation from the rate of sulphate leaching, with the assumption that all the sulphate comes from the pyrite.



The formula weight of sulphate is 96 and hydrogen is 1. Based on oxidation of pyrite and hydrolysis of ferrous-iron in the reaction above, mg SO₄²⁻ /kg/wk is divided by 48 to convert to mg H⁺ /kg/wk, assuming all the sulphate comes from pyrite and each mole of sulphate corresponds to 2 moles of H⁺.



The formula weight of CaCO_3 is 100.09. The $\text{mg H}^+/\text{kg/wk}$ is converted to $\text{mg CaCO}_3/\text{kg/wk}$ by multiplying by 50 for reaction 2 and 100 for reaction 3 (see Appendix A). Thus for reaction 2, the acidity produced by pyrite oxidation resulting in $1 \text{ mg SO}_4^{2-}/\text{kg/wk}$ is equivalent to $1.0426 \text{ mg CaCO}_3/\text{kg/wk}$ or $54.17 \text{ mg CaCO}_3/\text{kg/yr}$. For reaction 3, the acidity produced by pyrite oxidation resulting in $1 \text{ mg SO}_4^{2-}/\text{kg/wk}$ is equivalent to $2.0852 \text{ mg CaCO}_3/\text{kg/wk}$ or $108.34 \text{ mg CaCO}_3/\text{kg/yr}$. Based on the above, humidity cell rates of 50, 100 and 150 $\text{mg SO}_4^{2-}/\text{kg/wk}$ for reaction 2 are equivalent to 2.7, 5.4 and 8.1 $\text{kg CaCO}_3/\text{t/yr}$. Humidity cell rates of 50, 100 and 150 $\text{mg SO}_4^{2-}/\text{kg/wk}$ for reaction 3 are equivalent to 5.4, 10.8 and 16.2 $\text{kg CaCO}_3/\text{t/yr}$.