ML/ARD ASSESSMENT AND MITIGATION AT THE JOHNNY MOUNTAIN GOLD MINE

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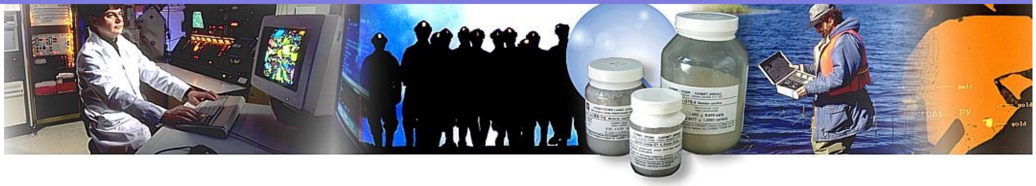

ML/ARD ASSESSMENT AND MITIGATION AT THE JOHNNY MOUNTAIN GOLD MINE

Produced as part of a series of case studies documenting ML/ARD assessment and mitigation at mines in Canada. Produced on behalf of MEND with support from the Mining Association of Canada and Natural Resources Canada.

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Executive Summary

Johnny Mountain was a relatively short-lived gold mine that operated in the late 1980s. The site is located in the Coast Mountains in the northern half of British Columbia. Site components include the plant/camp site, a number of roads and an airstrip, 3.5 km of underground workings, a 11.5 ha flooded tailings impoundment, and several small waste rock dumps. A road connects Johnny Mountain to the Snip mine, but there is no road access to the outside world. The ML/ARD work at Johnny Mountain Mine illustrates some of the common practices and factors to consider in assessment and mitigation. It also provides a number of lessons regarding potential ML/ARD challenges and information requirements. This includes why items such as operational material characterization, analysis of the fine waste rock size fraction, adequate mineralogical analysis and comprehensive flow monitoring are important requirements.

The ML/ARD assessment focused on three site components:

- the tailings impoundment;
- the underground workings; and
- the waste rock.

The waste material at Johnny Mountain with the lowest NPR, highest sulphide-S and therefore the highest theoretical potential to generate ARD is the tailings. The ARD potential was recognized prior to mining and as a result the tailings were placed in a flooded impoundment. Monitoring of the drainage indicates that flooding of the tailings has been successful in limiting sulphide oxidation. Remaining work items, such as increasing the geotechnical stability of the dams and leveling the tailings, will occur after placing waste rock in the impoundment. According to the Closure Plan, the dyke slopes will be trimmed, with the excess used as buttresses to increase their geotechnical stability. Subsequently, the main remaining liability associated with the tailings will be the limited, although very important, monitoring and maintenance of the impoundment.

Most of the focus of ML/ARD work since the mine closed has been on the waste rock. Based on a visual assessment of the degree of mineralization and ABA results from four samples, prior to mining it was concluded that the waste rock had little or no potential for ARD. Consequently, all the waste rock was placed in aerial dumps or used for construction. Analysis conducted after the mine closed indicates that most of the waste rock is potentially ARD generating, and preventing ARD will require costly re-handling. The experience with the waste rock illustrates the importance of ABA analysis for material characterization and the danger of relying on geological suppositions or visual observations of the extent of mineralization.

The rock that is potentially ARD generating (PAG) includes material where ARD is predicted and material where the ARD potential is uncertain. Even with additional analysis, much of the ‘uncertain’ waste rock will likely fall in the compositional gray zone where present prediction tools are unable to ascertain whether ARD will occur in the future. Even where the ARD potential is high, most of the waste rock has to date produced neutral pH drainage with relatively low metal concentrations. The proposed mitigation plan for the waste rock in the dumps and areas where the waste rock has been used as fill is to place it underwater in the tailings impoundment. Small portions of the rock that have already gone acid were picked up and placed in the impoundment in 1996. Where flooding is planned, it should be done as soon as possible to minimize the build-up of potentially soluble weathering products.

An alternative being considered where the waste rock ARD potential is limited or the amount is relatively small is some form of in-situ mitigation. This could include encapsulating it within the till by burial in trenches, a possibility suggested for the airstrip, or burial within the re-contoured roadway, a possibility suggested for the Magazine Road. Challenges with in-situ mitigation include collecting the required information and having adequate process control. Another important consideration will be ensuring that placement is done in a manner that ensures long-term geochemical stability.

The third mine component with ML/ARD concerns is the underground workings. The composition of the walls in the workings is expected to be similar to that of the waste rock, although there is much lower surface area. Like the waste rock, drainage from the underground workings presently has a neutral pH with relatively low metal concentrations. The proposed ARD mitigation plan for the underground workings is to limit oxygen entry by placing bulkheads in the adits and covers over the ventilation raises. Questions regarding this plan include the effectiveness of bulkheads and covers in limiting air entry and the impact of progressive rock fall on the ARD potential, especially in ore zones with high metals and pyrite. Like the waste rock, the underground contains both materials with a high ARD potential and material where the ARD potential is uncertain. In addition, the underground has significant NP added in groundwater alkalinity, making the ARD potential of the underground workings as a whole uncertain.

A major consideration in mitigation planning for Johnny Mountain is how much monitoring and maintenance will be required to determine the success of its long-term performance. For example, what monitoring is required to demonstrate that underground oxygen levels are low enough? The lack of a permanent site presence, coupled with low clouds, strong winds and deep snow cover for much of the year make monitoring and maintenance at Johnny Mountain costly and onerous undertakings. Unless there is a clear reason to do otherwise to protect the environment, from the perspective of minimizing risk and liability, the best mitigation strategy will be the one requiring minimal monitoring and maintenance, and no manual monitoring and maintenance during the winter.

The experience at Johnny Mountain illustrates the importance of conducting comprehensive site-specific ML/ARD work. It also illustrates some of the gaps that exist in our understanding and limitations in our ability to collect the information required for accurate ML/ARD assessment and cost effective remediation. One example is our limited understanding of the rate of air entry, oxygen concentrations and corresponding sulphide oxidation rates in closed, plugged underground mines in mountainous terrain. Examples of information that has proven difficult to collect includes:

- the variability in flow and drainage chemistry, both for the receiving environment and different site components;

- the difficulty in predicting the NP, especially for materials with complex carbonate mineralogy; and
- the influence of site-specific conditions to the available AP and NP, and whether modifications are required to general assessment rules.

By modern mine standards, the size of the underground workings and waste production at Johnny Mountain were relatively modest. This, coupled with the significant natural dilution (e.g., glaciers) and attenuation (e.g., high alkalinity in natural drainage) and the uncertain ARD potential of some of the rock, may mean that measures to minimize leaching are all that is required to prevent significant impacts from at least a portion of the exposed PAG materials. In some cases, even with ARD production the environmental impacts will be minimal, as is presently the case with ARD production within the underground workings. Determining when this will be the case, given the lack of information on the variability in flow and drainage chemistry, both for the receiving environment and different site components will be a challenge.

Even when the mine was in operation with full time staff available for snow removal, the mine had trouble conducting year round monitoring at all the receiving environment sites (Yeager, 2002). Now the mine is closed, site visits during the more than 6 months of winter would require a major expense. The weight of snow has crushed many of the mine buildings and other structures. Poor visibility, deep snow, changing flow paths and high flows and large debris loads in the creeks at certain times of the year make maintaining monitoring apparatus a major logistical and financial challenge. Consequently, Skyline has concluded that continuous flow monitoring would be prohibitively expensive and, due to the inevitable equipment failure, no more informative than estimating flow, and the consequent dilution, based on relative surface areas and existing monitoring.

Potential problems with this approach include:

- potential for significant differences in drainage contribution from different areas of the site (e.g., glaciers versus dump bench), during different times of the year and in different years; and
- how to predict groundwater flow (e.g., discharge from different drainages in the underground workings).

Supporting site data would greatly improve the confidence regarding potential loadings, natural dilution and attenuation, impacts to the receiving environment and the effectiveness of different mitigation measures to reduce leaching if PAG site components are allowed to generate ARD. The question of how to conduct continuous flow monitoring and collect samples for periodic drainage chemistry measurements in high snow fall areas where power and access are limited is a problem at a number of closed British Columbia mine sites. While the Johnny Mountain case emphasizes the importance of conducting detailed operational monitoring, it should be noted that site hydrology and hydrogeology often change dramatically after the mining stops, creating a need for post-closure monitoring.

The problems in predicting the NP in the materials at Johnny Mountain result from the complex carbonate mineralogy, previous omissions and basic gaps in our understanding. Static laboratory

NP measurements are crude measurements that can only provide a preliminary guide to the actual field neutralization capacity. Consideration of the contributing mineralogy and the rate of sulphide oxidation are required for the interpretation of results. Results of the Price and Kwong (1994) work at Johnny Mountain and a number of other sites shows that CO_2-NP > Sobek-NP is a good indication that there is significant Fe and Mn carbonate present. The potential contribution of Fe and Mn CO_3 minerals at this site means that the NP calculated from %CO_2 cannot be relied on as a measure of the field NP. Although it is a better measure of NP for this site than NP calculated from the %CO_2, the presence of ankerite and siderite also raises concerns regarding the potential contribution of Fe and Mn CO_3 to the Sobek-NP. This is compounded for the older data by the lack of information on the fizz rating.

Better NP information will be required if one is setting precise criteria for PAG and non-PAG waste rock or in predicting the time for ARD to become a concern. This could include checking whether Fe and Mn CO_3 contribute to the Sobek-NP, by comparison with results from modifications to the standard Sobek method that remove the contribution of Fe and Mn CO_3 (Meek, 1981; Skousen et al., 1997; White et al., 1998) and the use of Rietveld XRD analysis to quantify the proportion of carbonate occurring as calcite, ankerite and siderite. Microprobe work is required to ascertain the proportion of Fe, Mn, Ca and Mg in ankerite. A perceived drawback with Rietveld XRD and microprobe analysis is the cost: $200/sample for the Rietveld and $100/sample for the microprobe (for 6 samples with 5 grains analyzed per sample). However, the cost of Rietveld XRD analysis is not much higher than ABA plus ICP analysis, and minimal when compared to the potential costs of not knowing the carbonate mineralogy.

A basic information gap at Johnny Mountain and many other sites in British Columbia is the lack of data on field rates versus laboratory measurements of processes, such as sulphide oxidation. This information is required to determine when and where modifications are required to laboratory measurements and general ABA rules, and to answer questions such as how long is it permissible to wait until the large waste rock stockpiles must be moved to the impoundment. The impact of site-specific conditions on the rate of sulphide oxidation would also help answer whether additional neutralization sources, such as the high groundwater alkalinity, are able to continue neutralizing acidic drainage produced by the underground workings.

Sommaire

La mine d’or Johnny Mountain a été exploitée relativement peu longtemps à la fin des années 80. Elle est située dans la chaîne Côtière dans la moitié septentrionale de la Colombie-Britannique. À l’emplacement on trouve une usine et un camp, un certain nombre de routes et une piste d’atterrissage, 3,5 km de chantiers souterrains, un parc à résidus inondé couvrant 11,5 ha et plusieurs petites haldes de stériles. Une route relie la mine Johnny Mountain à la mine Snip, mais aucune route ne les relie au reste du monde. Les travaux d’atténuation du DRA/LM effectués à la mine Johnny Mountain illustrent certaines des pratiques courantes et des facteurs à prendre en considération pour l’évaluation des incidences. Ils mettent par ailleurs en évidence certains besoins en information et défis associés au DRA/LM. Ce document indique pourquoi des éléments comme la caractérisation des matériaux, l’analyse de la fraction fine des stériles, une analyse minéralogique adéquate et un suivi minutieux des éffluents constituent des exigences importantes.

L’évaluation du DRA/LM a porté principalement sur trois composantes du site minier :

- le parc à résidus subaquatique;
- les chantiers souterrains; et
- les stériles.

Les matériaux à la mine Johnny Mountain qui présentent les plus faibles RPN, les concentrations les plus élevées de soufre sous forme de sulfures et ainsi le potentiel théorique le plus élevé de production de DRA sont les résidus. La possibilité qu’un DRA soit généré avait été reconnue avant l’exploitation du gisement et c’est pourquoi les résidus ont été placés dans un parc à résidus et recouverts d’eau. Le suivi du drainage indique que l’inondation des résidus a permis de limiter l’oxydation des sulfures. Des éléments restants des travaux comme l’amélioration de la stabilité géotechnique des digues et le régalage des résidus seront exécutés après que les stériles auront été placés dans le parc. D’après le plan de fermeture, les talus des digues seront rognés et les matériaux en excès seront utilisés comme contre-boutants pour en accroître la stabilité géotechnique. Par la suite, un suivi et un entretien limités quoique très importants du parc seront les principales responsabilités associées aux résidus.

Depuis la fermeture de la mine, les travaux d’atténuation du DRA/LM ont porté principalement sur les stériles. D’après une évaluation visuelle du degré de minéralisation et les résultats de la détermination du bilan acide-base (DBAB) obtenus pour quatre échantillons, on avait conclu avant l’exploitation minière que les stériles ne présenteraient qu’un faible ou inexistant potentiel de DRA. En conséquence, tous les stériles ont été placés dans des dépôts subaériens ou utilisés à des fins de construction. L’analyse menée après la fermeture de la mine a indiqué que la plupart des stériles pouvaient engendrer un DRA et que la prévention de ce DRA en exigera une nouvelle manipulation coûteuse. L’expérience acquise avec les stériles illustre l’importance de la détermination bilan acide-base pour la caractérisation des matériaux et le danger qu’il y a à se fier à des suppositions géologiques ou à l’observation visuelle de l’ampleur de la minéralisation.

La roche potentiellement acidogène (PA) comprend des matériaux pour lesquels le DRA est prévisible et d’autres dont le potentiel de DRA est incertain. Même avec analyse additionnelle, une bonne partie des stériles dont le potentiel est incertain sera vraisemblablement classée quant à sa composition dans une «zone grise» pour laquelle les outils de prévision actuellement disponibles ne permettent pas de déterminer si elle génèrera à l’avenir un DRA. Même aux endroits où le potentiel de DRA est élevé, la plus grande partie des stériles n’a jusqu’à maintenant engendré que des eaux de drainage d’un pH neutre renfermant des concentrations de métaux relativement faibles. Le plan d’atténuation proposé pour les stériles dans les haldes et les zones où ils ont été utilisés comme remblai prévoit de les placer sous une couverture aqueuse dans le parc à résidus. De faibles quantités de roche déjà devenue acide ont été recueillies et placées dans le parc en 1996. Lorsque l’inondation est projetée, elle devrait être effectuée le plus rapidement possible afin de minimiser l’accumulation de produits d’altération potentiellement solubles.

Une forme d’atténuation *in situ* est envisagée à titre de solution aux endroits où le potentiel de DRA des stériles est restreint ou bien aux endroits où ils ne sont présents qu’en quantités relativement faibles. Une possibilité pour la roche de la piste d’atterrissage serait de l’enfouir dans des tranchées avec une couverture de till, et pour Magazine Road de l’enfouir dans la route à laquelle on donnerait un nouveau profil. Les défis que pose l’atténuation *in situ* sont la collecte de l’information nécessaire et l’élaboration d’un processus de contrôle adéquat. Une autre considération importante sera d’assurer que la mise en place s’effectue de manière à maintenir la stabilité géochimique à long terme.

Les chantiers souterrains constituent la troisième des composantes de la mine qui préoccupe en ce qui a trait au DRA/LM. La composition des parois de la mine devrait être similaire à celle des stériles bien qu’ils présentent une superficie de beaucoup inférieure. Comme dans le cas des stériles, le drainage provenant des chantiers souterrains présente actuellement un pH neutre et des concentrations de métaux relativement faibles. Le plan proposé d’atténuation du DRA émanant des chantiers souterrains consiste à limiter l’apport en oxygène en plaçant des cloisons dans les galeries d’accès et des couvercles sur les montages de ventilation. Les interrogations que soulève ce plan sont l’efficacité de l’étanchéité à l’air des cloisons et des couvercles et l’impact des éboulements de roche sur le potentiel de DRA, surtout dans les secteurs où du minerai renferme des concentrations élevées de métaux et de pyrite. Comme dans le cas des stériles, le souterrain renferme à la fois des matériaux présentant un potentiel de DRA élevé et des matériaux dont le potentiel de DRA est incertain. En outre, le sous-sol présente un PN ajouté important en raison de l’alcalinité de l’eau souterraine, ce qui fait que dans l’ensemble le potentiel de DRA des chantiers souterrains est incertain.

Une des considérations majeures pour la planification de l’atténuation à la mine Johnny Mountain est le niveau de suivi et d’entretien qui seront requis afin d’en assurer le succès à long terme. Par exemple, quel suivi sera nécessaire pour démontrer que les concentrations d’oxygène sous terre sont suffisamment faibles? Le fait que personne n’est présent en permanence au site minier associé à la présence de nuages bas, de forts vents et d’une épaisse couverture nivale pendant une bonne partie de l’année font en sorte que le suivi et l’entretien à cet endroit sont coûteux et exigeants. À moins d’une raison évidente pour procéder autrement afin de protéger l’environnement, du point de vue de la minimisation des risques et des responsabilités, la meilleure stratégie d’atténuation sera celle qui exigera le moins de suivi et d’entretien ainsi qu’aucun suivi et entretien manuels pendant l’hiver.

L’expérience acquise à la mine Johnny Mountain illustre l’importance de l’exécution de travaux d’évaluation exhaustifs du DRA/LM pour les conditions propres au site. Elle met en outre en lumière les lacunes dans notre compréhension et dans notre capacité à recueillir l’information nécessaire pour une évaluation exacte du DRA/LM et pour l’application de mesures correctives efficientes. Mentionnons à titre d’exemple notre compréhension limitée de la vitesse à laquelle l’air pénètre dans les chantiers, des concentrations d’oxygène et des taux d’oxydation correspondants des sulfures dans les mines souterraines fermées en milieu montagneux dont les accès ont été obstrués. Parmi les exemples d’information qui s’est avérée difficile à obtenir, soulignons :

- la variabilité de la chimie des effluents et du drainage, tant dans le milieu récepteur que dans les différentes composantes du site;

- la difficulté de la prévision du PN, surtout dans le cas de matériaux carbonatés présentant une minéralogie complexe; et
- l’influence des conditions propres au site sur le PA et le PN, et les modifications qui s’imposent des règles générales d’évaluation.

Dans le contexte minier contemporain, la taille des chantiers souterrains et le volume de rejets miniers produits à la mine Johnny Mountain ont été relativement modestes. Ces facteurs associés à d’importantes dilution (p. ex. glaciers) et atténuation (p. ex. alcalinité élevée des eaux naturelles de drainage) naturelles et l’incertain potentiel de DRA d’une partie des stériles pourraient faire en sorte que des mesures visant à minimiser la lixiviation puissent être les seules nécessaires pour prévenir des incidences importantes d’une partie des matériaux PA exposés. Dans certains cas, même s’il y a production de DRA, les incidences environnementales seront minimes, comme c’est actuellement le cas en ce qui a trait au DRA produit à l’intérieur des chantiers souterrains. Le défi qui se pose consiste à déterminer quand cela se produira compte tenu de l’absence d’information sur la variabilité des effluents et de la chimie du drainage, tant dans le milieu récepteur que dans les différentes composantes du site.

Même lorsque la mine était en exploitation et qu’un personnel permanent était disponible pour l’enlèvement de la neige, le suivi à l’année à tous les sites du milieu récepteur était problématique (Yeager, 2002). Maintenant que la mine est fermée, des visites de ces sites pendant les six mois et plus que dure l’hiver exigerait des dépenses majeures. Un grand nombre des bâtiments et autres structures de la mine se sont effondrés sous le poids de la neige. La mauvaise visibilité, l’épaisse couche de neige, les trajets d’écoulement changeants et les débits élevés ainsi que les importantes charges en débris des ruisseaux à certaines époques de l’année font que l’entretien des appareils de suivi pose un défi logistique et financier majeur. En conséquence, la société Skyline a conclu qu’un suivi en continu de l’écoulement serait d’un coût prohibitif et qu’en raison des inévitables bris d’équipement elle ne renseignerait pas davantage que l’estimation des effluents, et de la dilution qui en résulte, d’après les superficies relatives et le suivi existant.

Cette approche pourrait poser les problèmes suivants :

- il est possible que différents secteurs du site fournissent des volumes de drainage très différents (p. ex, glaciers comparativement aux bermes de haldes) à différentes époques de l’année et d’une année à l’autre; et
- comment prédire l’écoulement d’eau souterraine (p. ex. débit provenant de différents réseaux de drainage dans les chantiers souterrains).

Des données d’appoint contribueraient grandement à améliorer le niveau de confiance quant aux charges potentielles, à la dilution et à l’atténuation naturelles, aux incidences sur le milieu récepteur et à l’efficacité de différentes mesures d’atténuation pour la réduction de la lixiviation si l’on permet que les différentes composantes PA du site génère un DRA. Le suivi en continu des effluents et la collecte à intervalles réguliers d’échantillons permettant de caractériser la chimie des eaux de drainage dans des régions subissant de fortes chutes de neige où l’accès est limité et où on ne dispose pas de sources d’énergie électrique sont problématiques pour un certain nombre de mines fermées en Colombie-Britannique. Bien que le cas de la mine Johnny

Mountain souligne l’importance d’un suivi détaillé, il faut noter que l’hydrologie et l’hydrogéologie changent souvent de manière saisissante suite à l’arrêt de l’exploitation, ce qui rend nécessaire un suivi après la fermeture.

Les problèmes que pose la prévision du PN dans les matériaux présents à la mine Johnny Mountain découlent de la minéralogie complexe des roches carbonatées, d’omissions antérieures et de lacunes fondamentales dans notre compréhension. Les mesures statiques en laboratoire du PN sont approximatives et ne fournissent qu’un guide préliminaire pour l’évaluation de la capacité neutralisante réelle sur le terrain. Il faut prendre en considération la minéralogie et le taux d’oxydation des sulfures pour l’interprétation des résultats. Les résultats des travaux de Price et Kwong (1994) à la mine Johnny Mountain et à d’autres endroits indiquent que si le PN-CO_2 > le PN Sobek il s’agit d’une bonne indication de la présence d’importantes quantités de carbonate de Fe et de carbonate de Mn. La contribution potentielle des minéraux $FeCO_3$ et $MnCO_3$ à ce site signifie qu’on ne peut se fier au PN calculé d’après le pourcentage de CO_2 comme mesure du PN sur le terrain. Bien que le PN calculé constitue pour ce site minier une meilleure mesure que le PN calculé d’après le pourcentage de CO2, la présence d’ankérite et de sidérite soulève également des inquiétudes quant à la contribution potentielle du $FeCO_3$ et du $MnCO_3$ au PN calculé par la méthode Sobek. Dans le cas des données plus anciennes ce problème est exacerbé par le manque d’information relative au taux d’effervescence.

Une meilleure information sur le PN sera nécessaire pour établir des critères précis pour les stériles PA et non PA ou pour la prévision du moment à partir duquel le DRA deviendra inquiétant. Elle pourrait englober une vérification du fait que le $MnCO_3$ ou le $FeCO_3$ contribuent au PN-Sobek par comparaison avec les résultants obtenus pour des modifications à la méthode Sobek normalisée éliminant les contributions du $MnCO_3$ ou du $FeCO_3$ (Meek, 1981; Skousen et coll., 1997; White et coll., 1998) et l’utilisation de l’analyse par diffraction X Rietveld (XRD Rietveld) pour la quantification de la proportion de carbonate présent sous forme de calcite, d’ankérite et de sidérite. Des analyses par microsonde sont nécessaires pour s’assurer de la proportion présente de Fe, Mn, Ca et Mg dans l’ankérite. Les coûts seraient un inconvénient perçu pour l’analyse XRD Rietveld et par microsonde qui s’établissent à 200 $/échantillon pour la méthode Rietveld et 100 $/échantillon pour l’analyse par microsonde (pour 6 échantillons et 5 grains analysés par échantillon). Cependant, les coûts de l’analyse XRD Rietveld ne sont pas beaucoup plus importants que ceux de la détermination du bilan acide-base en plasma à couplage inductif (ICP) et sont minimaux lorsque comparés aux coûts que pourrait entraîner la méconnaissance de la minéralogie des carbonates.

L’absence de données sur les taux sur le terrain comparativement aux mesures en laboratoire pour des processus comme l’oxydation des sulfures constitue une lacune fondamentale à la mine Johnny Mountain comme en un grand nombre d’autres endroits en Colombie-Britannique. Cette information est nécessaire pour déterminer quand et où des modifications des mesures effectuées en laboratoire et des règles générales de la DBAB, et pour répondre à des questions comme la durée pendant laquelle on peut attendre avant de déplacer de grandes haldes de stériles vers le parc à résidus. L’incidence de conditions propres au site sur le taux d’oxydation des sulfures pourrait en outre aider à déterminer si des sources additionnelles de neutralisation, comme l’alcalinité de l’eau souterraine, peuvent continuer à neutraliser le drainage acide produit dans des chantiers souterrains.

Acronyms

ABA: Acid-Base Accounting
AP: Acid Potential
ARD: Acid Rock Drainage
ML: Metal Leaching
NP: Neutralization Potential
NPR: Neutralization Potential Ratio (NP/AP)
PAG: Potentially ARD Generating
TSS: Total Suspended Solids
QA/QC: Quality Assurance/Quality Control

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1. General Site Conditions

1.1. Location

The mine site is 56 37 26 N and 131 03 56 W, south of the Iskut River and east of both the Alaska panhandle and the Craig River, in the Coast Mountains in the northern half of British Columbia (Figure 1). There are a number of mine sites in the area including the operating mine at Eskay Creek and the advanced exploration project at Galore Creek. The closest mine site is the recently closed Snip mine, approximately 5 km north, at the base of the Johnny Flats plateau, by the Iskut River (Figure 2).

An extension of the Magazine Road connects Johnny Mountain to the loading facilities and airstrip at the Snip mine, but there is no road access to the outside world. When the mine was operating, supplies were brought in by air using the airstrip at the site or by boat from the coast, up the Iskut River to the Snip mine and up the adjoining road. An important consideration since the mine closed is that no one lives at the site. Primary access is presently by helicopter. Access is also still possible by boat up the Stikine and then Iskut Rivers from Wrangell, Alaska, and then up the road from Snip. Access from up-river is not possible because parts of the Iskut River east of the mine are un-navigable upstream.

1.2. Mining

Johnny Mountain was a relatively short-lived gold mine that operated in the late 1980s. The Johnny Mountain Mine is owned and was operated by International Skyline Gold Corporation. Ore extraction occurred through underground mining with the mine producing a gold, silver and copper concentrate. Underground exploration and mine development began in 1986 and milling commenced in 1988. Mining ceased in September 1990. A further three months of operation occurred from September to December 1993. The mine produced 4,348,814 g of Ag, 2,815,393 g of Au and 1,008 tonnes of Cu (Table 1), along with 197,794 tonnes of tailings and 153,000 tonnes of waste rock. Subsequently there has been intermittent exploration and reclamation.

Most of the site facilities and the 10 Level portal are located on the relatively flat terrain of the Johnny Flats, a plateau at an elevation of 1000 to 1200 m (Figure 3). The 11 and 12 Level portals are at a slightly higher elevation on the lower slopes of Johnny Mountain. The main site components are:

- the plant/camp site,
- roads and an airstrip,
- 3.5 km of underground mine workings (Figures 4 and 5),
- a 11.5 ha flooded tailings impoundment (Figure 6), and
- several small waste rock dumps.

1.3. Geology

A pre-mining geological description of the deposit is provided in MINFILE, the mineral occurrence database of the British Columbia Ministry of Energy and Mines. Refined descriptions based on observations in the underground workings are provided by Woznow and Yeager (1999) and Yeager and Metcalfe (1990). The deposit is classified as intrusion-related Au veins and subaqueous hot spring Ag-Au. The host rock comprises potassium feldspathized, dacitic porphyry dykes of probable Lower Jurassic age that have intruded a thick Upper Triassic intermediate volcaniclastic sequence. The deposit consists of gold-bearing vein mineralization within a number of moderate to steeply dipping sub parallel sulphide-potassium feldspar-quartz vein and stock work systems. The veins and sulphide stock works are found entirely within fracture systems cutting the feldspar porphyry dykes and volcanoclastic rocks.

The quartz sulphide veins range from 0.5 to 2.0 m in thickness and strike in an east-northeast direction. The gold-silver-copper-bearing sulphides are comprised mainly of pyrite and chalcopyrite with some sphalerite, galena and minor pyrrhotite. Gangue minerals are potassium feldspar (orthoclase) and quartz. Metallic minerals identified from the high grade Discovery Zone drill core include rare occurrences of arsenopyrite, bornite, chalcopyrite, covellite, electrum, enargite, galena, native gold, hematite, ilmenite, magnetite, marcasite, molybdenite, pyrargyrite, pyrite, pyrrhotite, sphalerite, stephanite and tetrahedrite. Bracketing the veins is an alteration envelope of approximately 3 m on the hanging wall and 0.3 to 0.5 m on the footwall. The envelope is comprised of low sulphide, massive grey, potassium feldspar and light green sericite grading outward to biotite and chlorite. The veins are contained within volcanoclastic and feldspar porphyry units. Geological descriptions of the deposit indicate significant alteration of calcite to ankerite.

Mining occurred in four veins: 16 Vein (Level 9, 10, 11 and 12), Discovery Vein (Level 10 and 11), Zephrin Vein (Level 10 and 11) and Pickaxe Vein (Level 11 and 12). The vast majority of mining occurred in the 16 Vein, which has a fault offset. Significant mining also occurred in the Discovery Vein. Only a small amount of mining occurred in the Zepherin and Pickaxe Veins. The 16, Discovery and Zephrin are considered fault bounded slabs of the same vein (D. Yeager pers. comm.). Characteristics of the different veins are as follows:

- 16 Vein: mainly pyrite with accessory chalcopyrite and rare sphalerite and galena, wall rock alteration includes ankerite, some encapsulation of pyrite by quartz

- Discovery Vein: pyrite and minor chalcopyrite, pyrite grains in biotite matrix, believed to be the source of semi-massive pyritic rock observed in 10 and 11 Level dumps and fill around the site

- Zephrin Vein: pyrite, considerable amounts of disseminated ankerite, lowest potential for ARD of all the mineralized zones

- Pickaxe Vein: pyrite and chalcopyrite, with traces of sphalerite, most reported as ore because chalcopyrite enhanced the gold recovery

1.4. Mineralogy

The following mineralogical data was collected as part of the metal leaching and acid rock drainage (ML/ARD) work:

- Petrographic analyses done on 10 polished thin sections made from samples representative of the waste rock (Table 2 – Rescan, 1991).
- XRD and petrographic examination of 7 tailings samples (Senes, 1995).
- Various mineralogical analyses performed on different particle sizes of 9 samples representative of the range of material observed in the waste rock dumps (Table 3 – work by Price and Kwong on samples collected in 1994).

The results are discussed in sections of this report dealing with ARD test work, waste rock and tailings.

One of the main conclusions drawn from the results was the inadequacy of the petrographic and regular XRD analyses in providing the mineralogical information required to interpret the acid base accounting (ABA) analysis results. Petrographic analysis was the main mineralogical tool used in ML/ARD assessments at the time the work was done. Advantages of petrographic analysis are that it can scan relatively large samples, it shows the spatial distribution of different grains and it is effective in identifying different sulphide minerals. The main disadvantages are that it is limited in its ability to distinguish different carbonate minerals and to identify grains less than 100 μm (e.g., tailings). The XRD procedures used in the Senes (1995) and Price and Kwong (1994) work could identify different carbonate minerals and the mineralogy of small grains, and with proper sub-sampling can scan a relatively large sample. However, the results were inconsistent, indicating significant operator or machine variability and the detection limit of approximately 2 to 5% was too high to quantify the different carbonate minerals. Point counting with scanning electron microscopy (SEM) and EDX was more accurate, but the cost and time are prohibitive and these procedures are unable to provide accurate estimates of the non-neutralizing versus the neutralizing proportion of solution minerals (e.g., ankerite/dolomite).

Mines presently use Rietveld XRD analysis to quantify the proportion of carbonate occurring as calcite, dolomite, ankerite and siderite. Microprobe work is required to ascertain the proportion of Fe, Mn, Ca and Mg in dolomite and ankerite. The costs are $200/sample for the Rietveld and $100/sample for the microprobe (5 grains analyzed per sample).

1.5. Generic Aspects of the Assessment of the ARD Potential

i. Assessment of Material Composition

Pre-mine, the ARD assessment consisted of a review of the general geology and ABA results from three ore and five waste rock samples. During mining, assessment of the ARD potential was based on visual observations of the extent of sulphide mineralization.

The first comprehensive program of sampling and analysis was conducted during what was perceived as a temporary mine closure. This work consisted of ABA analysis on rock chip samples taken from the underground workings and grab samples taken from the waste rock dumps, waste rock used to construct the airstrip and the road to Magazine Road, and the naturally exposed ARD generating float material in the McFadden Zone (Rescan, 1991). The results are shown in Tables 5 and 6. Subsequently, Price and Kwong conducted ABA analyses on different particle sizes from selected waste rock dump samples (Table 7) and the mine conducted ABA analyses on 7 tailings samples (Senes, 1995). In 2003, additional sampling and ABA analysis was conducted on the air strip and Magazine Road (Table 8).

In the 1990s when the majority of ARD sampling programs were conducted, the government and the mine recognized the importance of ABA test work, but were not fully aware of the limitations of the analyses. AP was calculated from total-S in the Rescan report and from both total minus acid soluble sulphate-S in the later Senes (1995) and Price and Kwong (1994) work. The Sobek NP procedure was used in the Price and Kwong and Senes work. The NP procedure used in the Rescan work was not identified, but is assumed to be the Sobek method. None of the reports provide the Sobek fizz rating, and therefore it is impossible to calculate the acid addition used in the test. The Rescan results only included %CO_2 for the waste rock and not the underground samples. International Skyline provided amended tables of the Rescan ABA data in Yeager (2002b). The memorandum included CO_2-NP and CO_2-NPR data, mean NPR values, and plots of Sobek-NP versus CO_2-NP, and Sobek-NPR versus CO_2-NPR. The Price and Kwong (1994) work was part of a study of ML/ARD characterization procedures conducted at a number of sites as a precursor to developing guidelines for ML/ARD assessment.

Median % total-S for different areas of the underground and waste rock is typically between 1.3 and 1.4%, although individual samples with more than 5% S were not uncommon (Tables 5 and 6). The median Sobek-NP and CO_2-NP ranged from 34 to 55 kg/t. The median NPR (NP/AP) calculated from the AP and Sobek-NP data for different areas of the underground and waste rock was generally around 1.0. Compared to the waste rock, the tailings samples had higher %S (4.2 to 28.5%), similar Sobek-NP (20 to 45 kg/t) and lower NPR values (0.02 to 0.34).

Post-closure assessment of the ARD potential at Johnny Mountain has been primarily based on sample NPR[1], with ARD predicted for an $NPR < 1$, uncertain for an NPR of 1 to 2 and considered unlikely for an $NPR > 2$. The technical rationale for these criteria and potential confounding factors are described in Appendix A. Based on these criteria, all the tailings samples are judged potentially ARD generating. The underground workings and the waste rock contain a significant number of samples in each of the three categories; ARD is likely, the potential for ARD is uncertain and not potentially ARD generating (Figure 7).

As discussed in Appendix A, a key part of an assessment of the ARD potential based on the NPR is determining the accuracy of the laboratory AP and NP data, and if required making corrections. Reasons for potential differences between Johnny Mountain's laboratory NP

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[1] One error made in some of the initial ABA calculations was in the calculation of the average NPR by adding individual sample NPR values and dividing by the number of samples. This method resulted in an erroneously high average NPR values. The correct procedure for calculating the average NPR is to divide the average NP by the average AP. Since small areas of ARD generation can contribute almost all of the contamination, subsequent analysis has focused on the distribution of individual NPR values, using the range, median and 10th and 90th percentiles rather than the average composition.

measurements and calculations, and the neutralization in the materials under site conditions are as follows.

- The occurrence of Fe and Mn CO_3 containing carbonate minerals ankerite [Ca(Ca,Mg,Fe)CO_3], which is common, and siderite $(FeCO_3)_2$, in addition to calcite ($CaCO_3$). This raises a concern regarding the potential contribution of non-neutralizing Fe and Mn CO_3 to the Sobek-NP laboratory analysis and to calculations of NP from a CO_2 analysis. The occurrence of Fe and Mn carbonates explains, at least in part, why Sobek-NP is often < NP calculated from the %CO_2. Unfortunately the potential problems associated with Fe and Mn CO_3 were not recognized when the test work was conducted, so no effort was made to quantify or remove their impact. Due to the presence of Fe or Mn CO_3, Sobek-NP is a better measure of the neutralizing capacity than calculation of the NP from a carbon assay. However, there still might be a contribution of Fe or Mn CO_3 to the Sobek-NP. Mn CO_3 is theoretically a greater concern than Fe CO_3 because of its much slower rate of oxidation and hydrolysis. The low Mn concentrations (< 0.23%) indicate that Mn CO_3 is not present in significant amounts (Table 9) and almost all the non-neutralising CO_3 is Fe CO_3.

- Ankerite may contain significant Mg in addition to Ca, Fe and possibly Mn CO_3. $MgCO_3$ provides approximately 10% more neutralization than $CaCO_3$ and in the calculation of NP from the %CO_2 may counteract some or all of the effect of non-neutralizing Fe and Mn CO_3. Magnesium values are 20% of Ca concentrations in the underground and 10% in the dump drainage suggesting that Mg CO_3 is relatively common. One likely explanation for this is that the ankerite contains significant Mg. Ankerite occurs as a solid solution with dolomite, and at another site subsequent microprobe work showed that the ankerite identified by Rietveld XRD was in fact a ferrous dolomite.

- The Sobek-NP may include too large a contribution of slowly reactive silicate minerals. In the absence of Fe and Mn CO_3, the contribution of silicate minerals to Sobek-NP can be determined by a comparison with the carbonate-NP. The uncertainty regarding the potential contribution of Fe CO_3 to Sobek-NP makes this impossible. An excessively high acid addition is commonly the cause of a large contribution of slowly reactive silicate minerals to the Sobek-NP. Unfortunately in the older data, there is no information on the fizz rating. However, the similarity between the carbonate-NP and Sobek-NP suggest that acid additions in the Sobek-NP procedure were appropriate.

  The impact of a higher acid addition in the Sobek-NP analysis on the difference between the carbonate-NP and Sobek-NP is evident in the 2003 ABA data (Table 8). The median difference between the CO_2-NP and Sobek-NP is 5 kg $CaCO_3$/t for the Magazine Road samples where the fizz rating was none to slight (50 to 100 kg $CaCO_3$/t of acid) and 20 and 17 kg $CaCO_3$/t for the south and north sections of the airstrip where the fizz rating was moderate (500 kg $CaCO_3$/t of acid). The CO_2-NP was less than 50 kg $CaCO_3$/t in all locations suggesting that the moderate fizz rating (500 kg $CaCO_3$/t of acidity) was

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

excessive and had a none or slight fizz rating (100 kg $CaCO_3$/t of acidity) should have been used.

Possible reasons for potential differences between laboratory AP measurements and calculations, and acid generation in the materials at the site are as follows.

- AP in the majority of the mine's test work (Rescan, 1991) is calculated directly from total-S, without subtracting sulphate-S, which is non-acid generating. Sulphate can occur in either an acid soluble (e.g., gypsum) or insoluble (e.g., barite) form. Data collected by Rescan, BC MEM, Senes and Skyline suggest that subtracting sulphate-S from total-S would not significantly impact the AP. Acid soluble sulphate-S, measured by Rescan (1991) was < 0.1% in all but a few samples. In the 1994 neutral pH waste rock samples (Price and Kwong, 1994), measured concentrations of acid soluble sulphate-S ranged from not detectable to 0.07% (Table 7). The range was from not detectable to 0.2% in the Senes tailings samples. The lack of sulphate-S in the 2003 samples (typically <0.01%) suggest that the waste rock used for road and airstrip construction was strongly leached as a result of the high precipitation.

  The most common acid insoluble sulphate species are Ba, Pb and Sr SO_4. The concentration of Ba was < 187 ppm in the tailings, but up to 3130 ppm in the waste rock (Table 10). Assuming all the Ba occurs in barite, 3130 ppm Ba would only result in 731 ppm or 0.07% barite-S (see p 48 Price, 1997). The concentration of Pb was even lower; < 100 ppm in the waste rock and < 208 ppm in the Senes tailings samples. Sr was also present in relatively low concentrations.

- AP in majority of the mine's test work (Rescan, 1991) is calculated by assuming that all the total-S produces the same acidity per mole of S as pyrite (FeS_2). This is a reasonable assumption, since chalcopyrite ($CuFeS_2$), the only other common sulphide mineral produces a similar amount of acidity per mole of S at neutral pH. Other sulphide minerals producing less acidity per mole of S, such as sphalerite and galena, are too rare to be a concern. The similarity between the rest potential of chalcopyrite and pyrite suggest there will be no galvanic controls on sulphide oxidation.

- Sulphide minerals may concentrate in smaller waste rock particles, where minerals will be exposed to oxygen and water, resulting in an effective NPR that is lower than the NPR measured on drill core or a grab sample containing coarse fragments. As coarse fragments account for most of the mass in waste rock, their composition is likely to predominantly influence a 'whole waste rock' NPR value, although most of their mineral composition is occluded and unable to react. In the analysis of different particle sizes conducted in the Price and Kwong (1994) work, as a result of higher % sulphide-S, the NPR of the < 2 mm particles was significantly lower than the coarse fragments in five of the seven samples. A similar particle size effect is likely to occur in the talus created by rock falls in the underground workings.

- In the tailings, heavier sulphide minerals may concentrate in the sandier material that is deposited close to the spigot point versus the finer slimes that are transported to the center of the impoundment. Since all the tailings will be underwater this is not a concern. However, it is worth noting that the samples collected came from close to the side of the impoundment and are thus likely to be from the sandier material.

- Sulphide minerals may be physically coated by a mineral such as quartz and prevented from oxidizing. In the initial petrographic assessment it was noted that most sulphide grains where surrounded by quartz, leading to suggestions that the sulphide may be physically occluded and prevented from oxidizing (Rescan, 1991). However, crushing and grinding of tailings and blasting and dumping waste rock would likely break coatings observed in the original rock. The high concentration of pyrite in the < 50 μm size fractions of the waste rock samples analyzed by Price and Kwong (1994) and the significant pyrite oxidation measured in the humidity cell tests confirm that encapsulation of pyrite by quartz will not prevent ARD.

ii. Kinetic Test Results

Kinetic testing consisted of four 20-week humidity cells conducted on waste rock (Table 9) and three waste rock field test pads monitored from 1995 to 1997 (Yeager, 2001).

Humidity Cell Tests

The samples tested in the humidity cell tests were composites of the 10, 11 and 12 Level waste rock materials (2.4 to 3.4%S) and a quartz-pyrite composite from various waste rock locations (21.5%S). The three composites from the 10, 11 and 12 Levels had NPR values of 0.4 to 0.7, significant Sobek-NP (38 to 54 kg/t), and produced pH 8 to 9 drainage throughout the 20-week test. The high pyrite sample (21.5% S) produced drainage with a pH of 6.2 to 6.9 during the first nine weeks. The pH subsequently decreased to 5.1 to 5.7. The post-test paste pH of 7.3 for the high pyrite sample was higher than the drainage pH indicating that grinding the test material prior to ABA analysis created or exposed significant alkalinity and was responsible for the measured NP of 7 kg/t.

The reported humidity cell drainage data only included trace metals, Fe, Al, Mn and sulphate and did not include data for base cations Ca, Mg, K and Na. The rate of sulphate release during the last five weeks for the lower %S samples varied from 8 to 22 mg/kg/wk, which is within the typical range for neutral pH weathering of mine wastes and does not indicate that the sulphides are physically coated and unable to react (International Kinetic Database at www.mdag.com/IKD.html). The acidity produced by pyrite oxidation resulting in 8 to 22 mg SO_4^{2-}/kg/wk is equivalent to 12.5 to 34.4 mg $CaCO_3$/kg/wk, 0.65 to 1.79 kg $CaCO_3$/t/yr and, assuming 3% S, 0.2 to 0.6 kg $CaCO_3$/t/yr/%S (see Appendix B). At a rate of 0.2 to 0.6 kg $CaCO_3$/t/yr/%S, it would be many years before the NP in these samples (38 to 54 kg $CaCO_3$/t) was exhausted leading to net acid conditions. The rate of sulphate release from the 21.5%S pyritic waste during the last five weeks ranged from 28 to 64 mg/kg/wk, which is equivalent to 2.3 to 5.2 kg $CaCO_3$/t/yr or 0.1 to 0.2 kg $CaCO_3$/t/yr/%S. The lower rate per %S in the pyritic waste can be explained by a lower surface area, which was estimated to be 2.82 m^2/kg versus 5.04 to 7.42 m^2/kg in the other lower %S samples.

The rates of sulphide oxidation measured in the humidity cells are likely to be higher than those experienced in the field. Lower temperatures in the underground workings or shallow waste rock dumps at the site should reduce field rates of pyrite oxidation and extend the time until NP exhaustion. Another reason is that the humidity cells were only run for 20 weeks and unless there is a decrease in pH, humidity cell oxidation rates typically decline. As mentioned previously, the humidity cell drainage contained significant Mn. Values varied from 0.07 to 0.5 mg/L for the lower %S samples and 1.2 to 8.2 mg/L for the pyritic waste. The lack of Mn in the pyrite suggests Mn may come from the dissolution of carbonate minerals.

Field Test Cells

Three field test cells were constructed in 1994 by placing waste rock in lined 3x3 ft^2 and 5 ft deep concentrate bags. A drainage pipe was placed at the bottom of each bag to prevent flooding and permit measurement of water quality and drainage volumes. The cells consisted of low and high sulphide waste rock from the 11 Level and typical waste rock from the 10 Level. No further compositional data for the test materials has been reported. Each bag was placed in a hole at the crest of the respective dump benches.

Drainage chemistry and discharge volumes from each cell were measured at various times in 1995, 1996 and 1997. Drainage collection was hampered by ice build-up and settling, and the variable results suggest that much of the drainage was not collected. The best data is from a period of short monitoring intervals in 1996. From September 30th to October 11th in 1996, the volume of drainage was 27 to 65 L in the three cells, suggesting a high degree of leaching. Notably in 1996, the 10 Level field cell was snow bound until August 11th.

Throughout the monitoring period, the drainage had a neutral pH (pH of 7.8 to 8.1) with relatively low metal levels. Concentrations in the drainage from the three cells in 1997 were 2 to 6 µg/L dissolved Cu, 9 to 21 µg/L dissolved Zn, 73 to 308 mg/L sulphate, 35 to 96 mg/L Ca, 3.5 to 17.5 mg/L Mg and 2.1 to 3.7 mg/L K (International Skyline Gold Corp., 1998). During the three years of monitoring, maximum metal concentrations occurred during snow-melt in the spring. The Mg values suggest that Mg is a significant component of ankerite or calcite. During the frequent monitoring in 1996, the 90th percentile Mg concentrations were 115 and 107 mg/L versus 186 and 86 mg/L for Ca for the two 11 Level cells. The K values indicate that some neutralization by species other than calcite and ankerite may be possible.

Mn concentrations from the three cells in 1997 were 1 to 23 µg/L, indicating that unlike the 20-week humidity cell test, Mn was either an insignificant component of the weathering carbonates or was precipitating after release.

iii. Conclusions

The primary source of information used to predict the ARD potential of different site components has been laboratory measurements of the sulphur content and NP. Static laboratory NP analyses and the accompanying calculations only provide a preliminary guide to the actual field neutralization capacity. Consideration of the contributing mineralogy and the rate of sulphide oxidation are required in the interpretation of the results. At Johnny Mountain, the potential contribution of Fe and Mn CO_3 minerals means that the NP calculated from $\%CO_2$

cannot be relied on as a measure of the field NP. However, the results show that %CO_2-NP > Sobek-NP provides an effective warning that samples contain significant Fe and Mn CO_3 (Price and Kwong, 1997).

Although it is a better measure of NP than %CO_2-NP for this site, there are also concerns whether too much acid was used in the Sobek-NP and the potential contribution of Fe and Mn CO_3. There are a number of procedures that can be used to check (Rietveld XRD analysis) or remove the contribution of Fe and Mn CO_3 (Meek, 1981; Skousen et al., 1997; White et al., 1998). These analyses should be considered if a more accurate measurement of the NP becomes necessary. It is now standard for mines in British Columbia to use Rietveld XRD analysis to quantify the proportion of carbonate-C occurring as calcite, ankerite and siderite, with microprobe work to ascertain the proportion of Fe, Mn, Ca and Mg if there are various carbonate minerals.

The uncertainty regarding the contribution of non-neutralizing Fe and Mn CO_3 or very slowly neutralizing silicate minerals to the measured NP and lower NPR in the < 2 mm particles compared to coarser particles, mean that some samples with a ‘whole rock’ NPR > 2 may be PAG. In the absence of additional information, one option is to use a higher NPR value as a cut-off. However, since all the tailings and most of the waste rock and underground samples have an NPR < 2 and a large number of samples have an NPR < 1, the underground workings, tailings and most of the waste rock are already considered potentially ARD generating, (Price, 2002). The need for more accurate prediction work will depend on the mitigation plans.

1.6. Elemental Composition and Metal Leaching Test Work

Total elemental and metal leaching test work consist of:

- Total elemental content of the test materials and drainage chemistry from 20-week humidity cell tests using 4 waste rock samples (Rescan, 1991);
- Total elemental content (Table 10), microprobe analysis of pyrite (Table 11) and analysis of leachable metals (Table 12) in different waste rock particle sizes (Price and Kwong, 1994);
- Total elemental content of 7 tailings samples (Senes, 1995);
- Results for samples taken from seepage accumulating in two pits excavated in ARD generating waste rock (Table 13); and
- Monitoring results of the chemistry of major drainage sources in the underground mine workings.

Trace elements in selected samples of waste rock whose total concentrations were ≥ 10x crustal abundance were As, Bi, Cd, Cu, Pb, Sb, Se and Zn (Table 10). As expected from the common occurrence of chalcopyrite, the trace element with the highest concentration is Cu. In the 10 and 11 Level samples, total-Cu in the < 2 mm fraction ranged from 0.16 to 0.8%. The next highest trace elements, Zn, followed by Pb and As, were all in order of magnitude lower than Cu. The concentration of Cd was a further two orders of magnitude lower. Concentrations of total-Cu, Zn and Pb in the < 50 μm size fraction were all higher than the < 2 mm size fraction. This was not the case for sulphide-S and As. Differences between concentrations in the < 2 mm size fraction and coarser particle sizes were also much higher for total-Cu, Zn and Pb compared to sulphide-S and total-As. The correlation between sulphide-S and As cannot be explained by the As concentration of pyrite (Table 11). Mean concentrations of trace element contaminants in pyrite were Bi > Cu > Pb > Co > Au > Zn > Cd, Ni & Ag > Sb > As.

Total trace element concentrations measured in the tailings were similar to those in the waste rock (Senes, 1995).

Part of the Price and Kwong (1994) test work included measuring concentrations of metals removed by various leach tests on the < 2 mm size fraction of the eight waste rock samples. The results are summarized in Table 12. Conclusions reached by John Kwong (unpublished) regarding this data are as follows:

1. The low levels of water-soluble and citrate dithionite bicarbonate (CBD)[3] -extractable metals in samples with a near-neutral rinse pH indicate that the metal sulfides are largely unaltered. There is very little iron oxide production (0.13-0.82% on a dry weight basis) despite the relatively high sulfide content (about 3-18% sulphide-S) of the samples.

2. The relatively high dilute-acid-leachable metals (about 10%) compared to the total metal content suggest that the sulfides are susceptible to acid dissolution. This agrees with the field observation of abundant anhedral chalcopyrite and the inferred presence of sphalerite in the samples based on geochemistry; both minerals are soluble under acidic conditions. This in turn indicates that if persistent ARD occurs at the site, mobilization of Cd, Cu and Zn is unavoidable.

3. For the two samples with acidic surface-rinse pH (JM-1 and JM-7A), the extremely high water-soluble Cu and Zn concentrations compared to both the CBD-extractable and dilute-acid-soluble metals suggest that the two metals occur primarily as secondary efflorescent salts such as sulfates. Both zinc sulfate and copper sulfate are highly water-soluble. The geochemical behaviour of Cd is very similar to that of Zn; the observations on Zn thus also apply to Cd. The formation of efflorescence suggests low humidity conditions at the site and possible metal flushing during spring freshet.

4. There was insufficient information on As and Pb to fully assess their behaviour under various leaching conditions. The observation that high concentrations of metals (particularly Cd, Cu and Zn) can be extracted by a dilute acid solution suggests a potential for downstream impacts if extensive ARD starts to develop at the site.

ARD has been limited to date. Assays of seepage accumulating in two pits excavated in acidic waste rock (Table 13) and acidic drainage in the underground (Table 19) support the conclusion that ARD will result in high dissolved metal concentrations. The drainage accumulating in two pits excavated in acidic waste rock was pH 2.5 and 3.5, with 13 and 17 ppm Cu and 458 and

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[3] An analysis commonly used by soil scientists to remove elements co-precipitated with iron.

255 ppm Fe. The acidic seep observed in the underground workings was pH 2.95, with 9.5 ppm Cu and 63 ppm Fe.

1.7. Climate, Ecology and Hydrology

The site receives 2.0 to 2.6 m of precipitation per year and estimated sublimation and evaporation is 250 mm per year. Precipitation occurs primarily as snow. The mountain immediately above the mine is partly covered by a glacier. The major runoff events result from snowmelt in the spring, intense glacier melt in July and August, and intense rainstorms in September and October. Up to 24 to 30 m of snow may fall each year and snow remains on the site until mid to late July. Since the mine closed the weight of snow has damaged buildings and flattened other structures on the site.

Site vegetation is primarily sub-alpine tundra with some un-vegetated recently glaciated areas immediately above. Alaskan Moss Heather and Cream Mountain Heather constitute approximately 80% of the sub-alpine tundra vegetation. Animal activity at the site is restricted by the relatively short snow-free period. Resident hoary marmots were the most abundant mammals observed in the baseline studies. Site visitors include grizzly bears and wolves.

The project area drains into two watersheds: Johnny Creek and Stonehouse Creek (Figure 3). Drainage from the majority of the site, the waste rock dumps, plant site, southwest portion of the tailings impoundment and the camp, reports directly or indirectly via lower Camp Creek into Stonehouse Creek. Stonehouse Creek originates at the base of a small glacier. Below the glacier, the braided channel splits into Stonehouse Creek and Camp Creek. Skyline diverted the upper portion of Camp Creek back into Stonehouse Creek above the mine. Stonehouse Creek is re-joined by the remaining lower portion of Camp Creek just below the camp and empties into the Craig River. Prior to the precipitous decline off Johnny Flats Plateau, the approximate area of Stonehouse Creek watershed is 294 ha, with a range in elevation between 985 and 1860 m and a mean elevation of 1150 m. During baseline studies in 1986, weekly flows measured during the snow-free period of mid-July to early November ranged from 0.1 to 0.9 m^3/s. Based on the approximate watershed area, the estimated average flow was 0.3 m^3/s. Stonehouse Creek is joined by another un-named creek of similar volume before it reaches the Craig River.

Johnny Creek originates in the Johnny Glacier and flows by the McFadden Zone, a naturally ARD generating mineralized float zone, before reaching the mine. Johnny Creek receives drainage from the northeast portion of the tailings impoundment, the spillway from the tailings impoundment and the airstrip. At the bottom of Johnny Flats, Johnny Creek joins Bronson Creek, which in turn flows into the Iskut River. Prior to the precipitous decline off Johnny Flats Plateau, the approximate area of Johnny Creek watershed is 489 ha, with a range in elevation between 1030 and 2100 m and a mean elevation of 1500 m. During baseline studies in 1986, flow measured during the snow-free period of mid-July to early November ranged from 0.4 to 1.9 m^3/s. Based on the approximate watershed areas the estimated average flow was 0.51 m^3/s.

The first fish habitat downstream of the mine occurs after Johnny and Stonehouse Creeks have left the Johnny Flats plateau. The descent from the Johnny Flats plateau, approximately 900 m in steep ravines, is a barrier to fish passage. The Iskut River and the Craig River both have important salmon runs and are valuable wildlife habitat.

1.8. Monitoring of Drainage from Different Site Components

The mine has monitored the drainage chemistry of Stonehouse and Johnny Creek immediately below the mine site since 1986 (Tables 15 and 16), discharge from the underground mine since 1990 (Table 17), discharge from the tailings impoundment since 1989 (Table 20) and sumps at the base of the three main waste rock dumps since 1991 (Tables 21, 22 and 23). The original data was reported in pre-mine baseline surveys, the Rescan Closure Plan and the annual reclamation reports. Complete results for each of the monitoring stations in 2003 are shown in Table 14. Results of flow estimates and drainage chemistry at selected locations in the underground are shown in Tables 18 and 19.

Both Stonehouse and Johnny are high energy, high bed load creeks, typical of creeks in steep, high snowfall, alpine areas. During the monitoring period both had a similar pH (6.6 to 8.2). Stonehouse Creek has higher TSS and solute concentrations (e.g., 31 to 49 mg/L alkalinity in Stonehouse Creek and 19 to 34 mg/L in Johnny Creek from 1997 to 2003). The impact of mining on the two creeks appears small. Compared to 1986/1987, the 1997 to 2003 drainage in Stonehouse Creek shows slight increases in sulphate (13 to 29 mg/L versus 2 to 15 mg/L), but no significant changes in dissolved-Cu (1 to 6 µg/L) and dissolved-Zn (3 to 20 µg/L). The major fluctuations in water quality in Stonehouse Creek were in total rather than dissolved concentrations and were associated with high TSS events. There were 8 sampling dates in 1987/1988 where TSS exceeded 100 mg/L. During these events total-Cu commonly exceeded 100 µg/L, total-Fe was as high as 27 mg/L, but dissolved-Cu only exceeded 10 µg/L twice (48 and 26 µg/L in Nov 88 and Aug 89 respectively). In August 2003, TSS was 157 mg/L and dissolved-Cu was only 3 µg/L.

Johnny Creek shows even less of an impact from mining than Stonehouse Creek. Sulphate was 1 to 7 mg/L in 1986 and 1987 and 2 to 9 mg/L from 1997 to 2003. Dissolved copper was 1 to 4 µg/L in 1986 and 1987 and not detectable to 1 µg/L from 1997 to 2003.

Table 24 provides a summary of key drainage chemistry parameters for the monitored site components for 1997 to 2003. Drainage pH and alkalinity values were 8.1 to 8.2 and 126 to 139 mg $CaCO_3$/L for the underground mine, 7.1 to 8.3 and 38 to 114 mg $CaCO_3$/L for the waste rock dumps and 6.8 to 7.6 and 8 to 12 mg $CaCO_3$/L for the tailings impoundment. The relatively high values and narrow range from the mine workings are attributed to the high, consistent alkalinity. The lower values for the tailings impoundment are likely because this drainage comes largely from incident precipitation with little runoff and groundwater from the adjacent ground.

Two limitations of the data are the lack of flow data and the limited QA/QC. The main QA/QC concerns are with the data from when the mine was operating. Potential problems included use of too high a detection limit, the manner in which samples were preserved and a lack of duplicates. QA/QC is now recognized as an important part of the monitoring program. The present QA/QC at Johnny Mountain includes duplicate sample analysis and a low detection limit.

The lack of flow data makes it impossible to estimate loadings. If sulphate concentration is an indication, the main sources of sulphide oxidation appear to be the underground workings (103 to 149 mg/L) and the 10 (99 to 258 mg/L) and 11 (101 to 160 mg/L) dumps. The lower sulphate concentrations in the drainage from the 12 dump (35 to 66 mg/L) are attributable to the lower mass and lack of pockets of semi-massive sulphide. The low sulphate in the impoundment drainage (11 to 15 mg/L) indicates that flooding has minimized sulphide oxidation. All the sulphate values are well below solubility constraints indicating significant dilution of weathering products.

The higher dissolved-Cu in the drainage from the underground mine (33 to 72 µg/L) and the 10 Level (12 to 43 µg/L) can be attributed to local production of ARD. Dissolved-Cu was an order of magnitude lower in the sumps of the 11 (0.5 to 8.5 µg/L) and 12 (not detectable to 3 µg/L) Level waste rock dumps and the drainage of the tailings impoundment (0.5 to 1.9 µg/L). The dissolved-Cu concentrations from the 12 dump and the tailings impoundment were similar to the Cu in the Stonehouse and Johnny Creeks, respectively. Results for the waste rock dumps, underground mine and tailings impoundment are discussed in more detail in the sections dealing with these mine components.

As part of their ongoing assessment of the mitigation requirements, BC MEM has asked International Skyline to predict potential metal loadings from different components of the site and assess the potential impact to water quality in the receiving environment. This assessment requires information on the relative discharge rates for different site components and the receiving environment. However, even when the mine was in operation with full time staff available for snow removal, the operator had trouble conducting year round monitoring at all the receiving environment sites (Yeager, 2002b). Now the mine is closed, the primary means of post-mining site access is by helicopter, making site monitoring expensive and relatively infrequent. Site visits during the more than 6 months of winter weather can require a major expedition. The weight of snow has crushed many of the mine buildings and other structures. Poor visibility, deep snow, changing flow paths and high flows and large debris loads in the creeks at certain times of the year make maintaining monitoring apparatus a major logistical and financial challenge. Consequently, the Company has concluded that continuous flow monitoring would be prohibitively expensive and due to the inevitable equipment failure would be no more informative than estimating dilution and attenuation based on relative surface areas, geochemical modeling and present monitoring.

1.9. Discharge Limits and Reclamation Requirements

The general criteria for determining whether ML/ARD is a concern and proposed remediation measures are acceptable is that they do not have a significant adverse impact on the environment. The two permitting agencies are 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 receiving environment objectives and discharge limits established by the BC Ministry of Water, Land and Air Protection (BC MWLAP). The land use objective for site reclamation is wildlife habitat – 1994 reclamation permit. Due to the site’s low biological productivity, the main task in creating wildlife habitat will be meeting BC MWLAP discharge limits. Other major reclamation objectives are to minimize financial liability and the risk of future environment impacts.

The BC MWLAP *Waste Management Permit* (PE-8514 - May 11, 2000) requires the annual monitoring of dissolved Fe, Cu and Zn, pH, SO_4 and hardness in the discharge from the underground workings and the tailings pond, the seepage collection ponds below the 10, 11 and 12 Level Dumps, and Johnny and Stonehouse Creeks on the Johnny Flats Plateau immediately downstream of the mine site. The discharge limits set for drainage from the tailings impoundment are that it be equivalent or better than 0.05 mg Cu/L and 0.2 mg Zn/L. No numerical limits are set on the rate of discharge or drainage chemistry from other parts of the site.

2. Underground Workings

2.1. Design and Hydrology

During mining, approximately 3.5 km of underground adits and drifts were constructed (Figures 4 and 5). Mining occurred on four levels (9, 10, 11 and 12), with access through inclined adits at elevations of 1075 m (10 Level), 1125 m (11 Level) and 1175 m (12 Level). A decline from the 10 Level was used to access the 9 Level and there are 5 ventilation raises. The only backfill mentioned in the closure plan was the placement of waste rock from mine development in 1993 in the subsequently flooded 9-14 stope.

Almost all the drainage in the underground workings reports to the 10 Level portal (JM-4 in Figure 5). Monitoring results for drainage chemistry at the portal are tabulated in Table 17. Flow and drainage chemistry results from monitoring at various locations in the underground conducted in August 2003 are shown in Tables 18 and 19.

Drainage along the 12 and 11 Levels flows down the ramp, until between 4282 and 4283 it is diverted by a berm down a manhole to the 10 Level (U4 – 2.25 L/s). The major drainage sources for the flow measured at U4 were the 16 (U5 – 1.68 L/s) and the Pickaxe (U6 – 0.57 L/s) Zones. The main drainage sources observed on the 10 Level were discharge down from the Discovery (U2 – 0.15 L/s) and 16 (U3 – 1.85 L/s) Zones and groundwater upwelling from the flooded 9 Level decline under artesian pressure (U1 – 2 L/s). The workings are flooded up to the top of the ramp from the 10 Level down to the 9 Level (above 4270). The overall discharge from the underground workings in August 2003 was 6.7 L/s.

2.2. Predicted ML/ARD Potential

Because of the larger surface area relative to mine walls, the main areas of concern in the underground workings are typically backfilled waste rock and tailings, pre-blasted but un-mined material, and talus produced by the collapse of mine workings. Since there is no record of pre-blasted, un-mined material and the only backfill is flooded, the primary ML/ARD concern has been the composition of the mine walls, with the tacit assumption that they will eventually produce talus.

The initial assessment of the ARD potential of the underground workings was based on visual observations. Based on the limited sulphide mineralization observed, it was initially concluded that, "the ARD concern is isolated to small specific areas of the workings and not to the entire underground workings."

This conclusion was not supported by ABA results from 44 samples of rock chips collected throughout the workings (Rescan, 1991). Individual ABA results and median, 10th and 90th percentile values for the different levels are listed in Table 5. This data showed that potentially ARD generating (PAG) material with an NPR 1 to 2 (ARD uncertain) and < 1 (ARD likely), and > 0.5% sulphide-S occurs throughout the underground workings. According to the data, on the 10, 11 and 12 Levels, 40% of the wall rock is NPR < 1 and another 40% is NPR between 1 and 2. Tenth percentile Sobek-NP values were 35 to 41 kg $CaCO_3$/t, indicating that for the materials sampled there would be a significant lag time prior to the production of acidic drainage. While there is no data on the elemental composition of the underground samples, it is presumed that the metals of greatest concern are the same as those for the tailings and waste rock (see Section I.F.).

In addition to potential errors in NP and AP measurement generic to all site components (see Section I.E.), in assessing the underground ML/ARD potential, it is important to note that on an exposed area basis, fewer ABA samples were taken from the areas immediately adjacent to the ore veins. This is a concern because these are the areas of the underground with the:

- more extensive excavation and therefore the largest wall surface;
- highest sulphide content and therefore the highest potential for ML/ARD; and
- most fracturing and faults, which coupled with the more extensive excavation, is likely to result in greater groundwater inputs and instability, and hence more talus production and leaching.

The relative low number of samples with sulphur contents more than 3% and the lack of samples with more than 5% S, indicate that either all the most highly mineralized rock had been removed by mining or, due to reasons such as rock instability and wide-spread sampling intervals, ABA samples were not collected from these areas. The survey of drainage sources conducted in 2003 indicated that much of the flow reporting from the underground comes from mineralized areas.

The main factor likely to cause criteria based solely on ABA results to overestimate the ML/ARD potential of the underground workings is the failure to consider neutralization provided by alkalinity in the groundwater. In 2003, high alkalinity was 225 mg/L measured in groundwater, which appears carbonated, upwelling from the 9 Level (U-1). Alkalinity in the other major sources of underground drainage (U-3 to U-6) was also high (88 and 114 mg/L) and as a result discharge from the underground mine has the highest alkalinity of any site component (133 mg/L in 2001 and 139 mg/L in 2003). Drainage from the 9 Level comes from workings where flooding is presumed to inhibit sulphide oxidation and thus should be a permanent NP source. It is impossible to estimate how much of the alkalinity in drainage from other areas of the mine comes from permanent sources (e.g., bedrock or non-PAG materials) versus exposed PAG mine walls and talus that will eventually be net acid generating. Notably, the only ARD measured in 2003 was a low flow from the Discovery Zone. High inputs of groundwater alkalinity may at least partially account for the lack of ARD from the mineralized areas with higher flow. Further investigation is required to check whether neutralization of pockets of acidic material by groundwater alkalinity is occurring in these presently net neutral areas.

The higher silica (2.72 mg/L), sodium (3 mg/L) and potassium (2 mg/L) levels in the drainage indicate that silicate weathering in the underground workings is higher than the surface wastes. However even in the underground, the concentration of elements suggestive of silicate weathering is far lower than those resulting from carbonate and sulphide weathering (Ca - 75 mg/L, Mg - 14 mg/L and sulphate - 120 mg/L). Relatively reactive silicate minerals will only be able to maintain neutral pH discharge if cool temperatures, limited leaching and a lack of surface area result in a relatively low production of ARD.

2.3. Drainage Chemistry

The historical record of discharge drainage chemistry is shown in Table 17. Notable features include the high sulphate concentration from October 1990 to May 1991 (131 to 200 mg/L), the high dissolved-Zn concentration from October 1990 to August 1991 (generally > 100 µg/L) and the extremely high TSS and total metals concentrations when the mine re-opened in 1993, reverting to more typical values in 1994. In 1995, total-Cu concentrations rose to 48 to 68 µg/L, significantly above typical previous values when the mine was closed. They increased again in 1997 (110 µg/L) and 1999 (174 µg/L). Dissolved-Cu concentrations from 1997 to 2003 (33 to 72 µg/L) are also above typical previous values of < 20 µg/L when the mine was closed.

In 2003, the major drainage sources, the flooded 9 Level, 16 Zone discharge on 10 Level, the 16 Zone discharge on 11 Level and the Pickaxe Zone discharge on 11 Level, all had a pH of 8.0 to 8.2 (Table 19). The only significantly different drainage was the 0.15 L/s pH 2.95 seepage from the Discovery Zone (U-2). This small acidic seep had high sulphate (430 mg/L), Ca (80 mg/L), Fe (63 mg/L), Mg (15 mg/L), Cu (9.5 mg/L) and Mn (3 mg/L) concentrations. Although they were higher than the neutral pH drainage, concentrations of Al (1.57 mg/L), As (5 µg/L) and Zn (0.96 mg/L) were low compared to ARD with a similar pH at other mines. Along with the highest alkalinity (225 mg/L), drainage from the 9 Level had the highest Ca (113 mg/L), Mg (21 mg/L), Na (6 mg/L) and Sr (2.7 mg/L), significant sulphate (180 mg/L) and Mn (1.4 mg/L), but no detectable Cu and Fe. The lack of Cu indicates that the sulphate comes from the dissolution of sulphate minerals rather than sulphide oxidation. The neutral pH stope drainage contained 80 to 87 mg/L sulphate, 51 to 65 mg/L Ca, 9.5 to 12.7 mg/L Mg, 0.03 to 0.27 mg/L Mn, 6 to 188 µg/L Cu and no detectable (<0.03 mg/L) Fe. The final discharge from the underground at the 10 Level portal contained 120 mg/L sulphate, 75 mg/L Ca, 14 mg/L Mg, 0.4 mg/L Mn, 41 µg/L Cu and no detectable (<0.03 mg/L) Fe.

2.4. Potential Mitigation Options

The present mitigation plan for the underground workings is to maintain internal water flow from the 12 and 11 Levels to the 10 Level and to the extent possible limit air entry (Yeager, 2002b). Covers will be constructed to limit the entry of air and oxygenated surface water through the five ventilation shafts. Bulkhead-like structures on the 10, 11 and 12 Levels are proposed to limit air entry through the adits. Barriers to air-movement on the 11 and 12 Levels would be located immediately downstream of the diversion raises, enabling them to direct drainage to the lower workings while ensuring no more than a few centimeters of water accumulate behind them. On the 10 Level, the proposal is to construct a dam-like structure in a location where the adit roof has been raised. The objective is that the height of the dam (3 meters) and thus the height of water flowing over the dam is above the height of the upslope drift, blocking air movement entry in the mine, without significantly increasing the height of the water table within the mine. A somewhat similar barrier to air-movement is part of the proposed closure plan for the underground workings at the neighbouring Snip mine.

A major challenge with this plan is how to handle uncertainty regarding a number of key factors. This includes uncertainty regarding the rate of air entry once the bulkheads and raise covers are in place, how low do oxygen levels have to be to prevent significant ARD, the impact of progressive rockfall and deterioration of the workings, and how to monitor features such as oxygen levels and hydrogeology. Progressive collapse of the un-maintained workings could increase the exposure of PAG rock, leaching and air entry. It may also result in a build-up of hydraulic pressure and failure of the 10 Level bulkhead if it or the adit above becomes plugged by rock or timbers. A potential monitoring solution is to insert equipment, such as oxygen probes, pressure transducers (to measure the depth of flooding) and perhaps even cameras (to check rock fall) through sealed standpipes.

The previously proposed plan is based on the assumption that the underground mine will eventually produce ARD or environmentally deleterious levels of metal leaching. The data collected in 2003 suggest that this may or may not prove to be the case. Better data on seasonal variability and post-mining changes in flow and drainage chemistry data for different areas of the underground and for the underground mine as a whole relative to Stonehouse Creek would assist the mine is assessing how much metal leaching would have to increase before it could cause significant environmental impacts and additional supplementary mitigation measures would be required.

3. Tailings Impoundment

3.1. Impoundment Design and Hydrology

The tailings impoundment is 11.5 ha, with compacted till dykes on three sides and high ground constraining the tailings on the west side (Figure 6). The dykes were constructed in 1988. A permanent spillway was constructed on the northeast dam in 1993 (Senes, 1995). According to the 1999 Closure Plan (Woznow and Yeager, 1999), the impoundment contains 197,794 tonnes of tailings, with a density ranging from 1.45 to 1.77 m^3/t and, averaging 1.6 m^3/t. The estimated tailings volume is 123,622 m^3, leaving approximately 70,000 m^3 of unused capacity in the impoundment. Presently almost all the tailings are flooded. The exceptions are two small areas: 1) 25 m^2 at the original spigot location approximately half way along the south dam and 2) 2,000 m^2 located in the east-central section of the dam. According to the 1999 Closure Plan, the exposed areas of tailings will be lowered once waste rock deposition is complete.

An interception ditch diverts upslope drainage around the un-impounded west side of the impoundment. Perimeter toe drains collect seepage at the base of the dams and conduct it to seepage recovery ponds on the north, east and south corners of the impoundment. Based on piezometer measurements and the hydraulic conductivity, the rate of seepage is estimated to be 1.2 to 9.3 m^3/day. It is estimated that 89% of the seepage is intercepted by the toe drains. Flow through the spillway measured during site visits since 1989 has ranged from 0 to 30 m^3/day.

3.2. Composition of the Tailings

ABA results from three ore samples were provided in pre-mine project reports (Skyline, 1987 and 1988). Based on a negative net neutralization potential (NP minus AP) in these three samples, the tailings were predicted to be ARD generating.

After mining, various analyses were conducted on seven surface and near-surface samples collected from the two areas of exposed tailings (Senes, 1995). Particle size analysis indicates that the sampled tailings were almost entirely < 300 μm, with 14 to 77% < 53 μm and 8 to 67% < 38 μm. The main silicate minerals identified through XRD analysis were quartz, feldspar and sericite. Petrographic examinations suggested the samples also contained 5 to 15% carbonate minerals, < 5% ferromagnesium minerals (actinolite, chlorite and biotite), pyrite was the only common sulphide and there were traces of chalcopyrite and sphalerite. According to ABA analysis, the samples contained 4.2 to 28.5% sulphide-S, 20 to 45 kg/t Sobek NP and 1.1 to 2.5% CO_2. NPR values ranged from 0.02 to 0.34 and thus all the samples were predicted to produce ARD if they are not flooded.

Relative to trace element concentrations normally found in rock, the tailings samples contained elevated concentrations of As (38 to 162 ppm), Cd (2 to 7 ppm), Cu (125 to 1290 ppm), Pb (46 to 208 ppm), Se (<2 to 56 ppm) and Zn (214 to 1010 ppm). The Cu concentrations were generally lower than the concentrations in the waste rock samples (Table 9). The concentrations of As, Co and Se in the analyzed samples were correlated with the sulphide-S content.

3.3. Composition of the Impoundment Drainage

The composition of drainage from the impoundment is shown in Table 20. For the most part the drainage has had a neutral pH. The exception was a lab pH of 4.2 to 5.7 in 1990. The field pH of 7.2 at that time and the lack of dissolved metals or sulphate in these or subsequent samples suggest that the low pH was due to the addition of acid to fix the sample prior to measuring the pH. Artificial acid addition in these low pH samples is likely responsible for the correlation between total- and dissolved-Cu and the relatively high dissolved-Cu in some of these samples. As noted previously, the inconsistency of the acid pH raises concerns about how samples were fixed when the mine was operating.

Even when the mill was operating, data from 1989 to 1990 and 1993, the concentrations of both total and dissolved-Cu were almost always less than 5 μg/L. One exception was in 1993 when dissolved-Cu was 10 μg/L. The lack of variability suggest that 10 μg/L was the detection limit. Despite disposal of already ARD generating waste rock in the impoundment in 2000, levels of the sulphate (11 to 15 mg/L) and Cu (0.5 to 1.9 µg/L) have remained low (Table 24). From 1997 to 2003, the drainage pH was 6.8 to 7.6 and alkalinity was 8 to 12 mg $CaCO_3$/L.

The Senes study included an analysis of the water chemistry of the flooded tailings pore water and the overlying water cover. The main differences were the slightly lower pH (7.2 to 7.3 versus 7.9 to 8.1) and alkalinity (25 to 28 versus 78 to 87 mg $CaCO_3$/L) of the overlying water compared to the pore water. Senes noted that the small patches of exposed tailings appeared unweathered, which is still the case 9 years later. The lack of weathering of the exposed tailings can be attributed to saturation, resulting from the humid climate and the tailing's high water holding capacity, which in turn is a result of the fine particle size.

3.4. Components of the Mitigation Plan

The final design of the impoundment, including selection of a permanent pond water level, will be based in part on how much waste rock is placed in the impoundment. According to the 1999 Closure Plan, after selected PAG waste rock is deposited in the impoundment, the tailings will be leveled and the dyke slopes trimmed, with the excess used as buttresses to increase their geotechnical stability. The final design of the impoundment will include:

- ability of dams and spillways to withstand extreme climate events and perform in the long-term;
- a water balance that will ensure permanent flooding; and
- measures to prevent future entrainment of PAG materials.

The water balance of the impoundment is discussed in the Closure Plan. To date, the only instances when the water level has been lower than the spillway have been when it was pumped down in 1994 to allow sampling and in 2000 to permit placement of ARD producing waste rock. On both occasions, subsequent fall rains and winter snow refilled it to capacity.

Visual observations of the bed form and annual monitoring of drainage discharge suggest there has been no re-suspension of tailings into the overlying water cover since tailings deposition. Evidence of surface stability includes growth of a vegetative mat on the top of the submerged tailings. Deposition of waste rock in 2000 resulted in some turbidity. However, the water level was lowered prior to deposition to prevent discharge of suspended solids from the pond. The turbidity settled over the intervening winter before the water was discharged the following summer. If the final depth of water is insufficient to prevent future entrainment of PAG materials by ice or wave action, the design will include baffles or an inert cover over the tailings.

Re-configuration of the impoundment dykes will require re-configuration of the spillway, providing an opportunity to replace waste rock in the spillway with more inert granular fill.

4. Waste Rock

4.1. Disposal

The mine produced 153,000 tonnes of waste rock, with the majority initially placed in dumps outside the portals. Based on volumes removed from the workings and site surveys, estimated tonnes, volumes[4] and areas covered by dumps produced with waste rock from the different levels were as follows.

- 10 Level - 75,000 t, 37,500 m^3 and 22,500 m^2
- 11 Level - 53,000 t, 26,500 m^3 and 5,633 m^2
- 12 Level - 25,000 t, 12,500 m^3 and 1,200 m^2

The greater area of the 10 Level dump results from the shallow depth. Unlike the other two portals, the 10 Level is on the relatively flat terrain of Johnny Flats, and thus a wider, lower dump was created. Based on 2.13 m annual precipitation and 0.25 m sublimation and evaporation, the estimated annual discharge volume is 42.3, 10.59 and 2.26 x 10^3 m^3/year from the 10, 11 and 12 Level dumps, respectively. By comparison, the estimated annual flow in Stonehouse Creek is 9.3 10^6 m^3/year.

In addition to placement in dumps, waste rock was also used in the construction of roads, the airstrip, building sites and the tailings pond spillway. This includes construction of the foundations of the plant site and camp, and roads to the warehouse, the explosive magazine, the portals and the glacier. Most of the waste rock used in road construction was deposited as a thin 0.1 to 0.3 m deep weight bearing cover over local glacial till soils and its use depended on local soil conditions. For example, the Magazine Road was largely constructed from local till soils with only a thin veneer of 10 Level mine waste rock being placed on the surface. The exception was where larger volumes of waste rock were used to cross wetlands.

Most of the waste rock used for construction came from the 10 Level. Exceptions were used at the 11 Level rock for the road between the 11 and 12 Level portals and use of the 12 Level rock for the road above the 12 Level portal. In addition to waste rock from the mine, 22,000 t of waste rock-like material, excavated during construction of the concentrator, was used as fill beneath the west end of that building and in the airstrip staging area.

4.2. ML/ARD Potential of the Waste Rock

i. Initial Assessment

ABA results from five waste rock drill core samples were provided in pre-mine project reports (Skyline, 1987 and 1988). The assessment of the ARD potential of these samples was based on whether the net neutralization potential (NP minus AP or NNP) was < 0. The only sample with a negative NNP was said to be immediately adjacent to the ore and it was discounted because it occurred close to the ore and would likely report to the mill. The other four samples had an NP

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[4] Calculated assuming a bulk density of 2 m^3/t.

above 35 kg/t and 0.8 to 1.2% S. Based on their positive NNP results and an expectation that the majority of the waste rock would come from similar relatively un-mineralized development headings, the waste rock as a whole was predicted to have a little or no ARD generating capacity.

However, the pre-mine project reports recognized a potential for the waste rock to generate ARD and suggested the following monitoring and contingency mitigation measures.

> *"In the occasional instance where a major pyrite mineralization is intersected by a development heading, this material will be sent to the mill…"*
> *"During development activity, the mine … will take samples routinely … to analyze for acid generation capability. Material that indicates potential acid generation will be routed to the tailings pond for disposal."*
> *"If the acid consuming ability of most rock is about 35 kg/t, …, then any rock with a sulphur content below 1.2% would not be acid generating. …… The identification technique will be adopted by the mine … Non ore-grade rock that is found to be acid generating will be segregated and sent to the tailings pond."*

Unfortunately, assessment of the ARD potential of the waste rock during mining and immediately afterwards was based solely on visual observations, resulting in the following conclusions.

> *"The rocks encountered in these adits were not mineralized and contained no or very low concentrations of sulphide minerals."*
>
> *"Mined rock containing no valuable metals would be deposited on surface dumps and referred to as waste rock. These rocks comprise:*
> 1. *rocks from the unmineralized adits and cross cuts, which constitute the most abundant rocks in the waste dumps,*
> 2. *rocks from the weakly mineralized secondary access drifts, which constitute the second most abundant rocks in the waste dumps, and*
> 3. *rocks from the highly mineralized, pyrite rich vein rock that was not gold bearing, which constitute the least abundant rocks in the waste dumps.*
>
> *The result of this practice was the incorporation of a small percentage of rock with a high potential of acid generation in otherwise neutral or acid consuming rock."*
>
> *"The waste rock contains little or no sulphide minerals, the sub-alpine climate is not conducive to weathering and sulphide mineral occurrences on the surfaces of coarse fragments and in the fine fraction of natural outcrops indicated that the sulphides were not reactive."*

The last statement made by a consultant is particularly troublesome given the highly pyritic waste rock identified by Woznow and Yeager (1999) and the natural ARD produced by an outcrop immediately above the mine (Rescan, 1991 and Price and Kwong, 1994). Price and

Kwong (1994) observed sulphide-rich coarse fragments and fines containing more than 10% pyrite on the 10 and 11 Level waste rock dumps.

ii. Results of Post-Mining Analytical Test Work

Mineralogy

Mineralogical analysis indicates that the waste rock consists of pyrite, quartz, K-feldspar, albite/plagioclase and sericite/illite, with lesser amounts of carbonate, biotite and chlorite (Tables 2, 3 and 4). These results agree with the general geological descriptions of the deposit. Pyrite is the main sulphide mineral. The other commonly observed sulphide mineral was chalcopyrite, with trace amounts of sphalerite, galena and pyrrhotite also identified (Table 3).

XRD analysis of selected waste rock samples identified minor or trace amounts of calcite in most of and dolomite or ankerite in half of the neutral pH samples. SEM work on two neutral pH waste rock samples identified dolomite/ankerite and siderite in one and calcite and ankerite in the other (Table 4).

ABA Results

Most of the ABA data comes from analysis conducted in 1990 on fifty-seven samples collected from the five main waste rock locations; the three dumps, the Magazine Road and the airstrip (Table 5). Median NPR values calculated using total-S and Sobek-NP were 0.4 to 1.3, with 10th percentile values from 0.1 to 0.6. The conclusion drawn from this data is that ARD is likely from approximately half the waste rock and uncertain for 35% of the waste rock.

Other notable features of the ABA results are as follows:

- When the sampling was conducted in 1990, all the materials sampled still had significant NP and a neutral paste pH. The Sobek-NP in the waste rock at different locations was relatively consistent. The 10th percentile ranged from 22 to 33 kg $CaCO_3$/t, the 90th percentiles from 44 to 64 kg $CaCO_3$/t and only two samples had an NP < 20 kg/t.
- For many of the samples, the values of NP calculated from the %CO_2 were higher than Sobek-NP. This is illustrated by the median values for the different locations, which ranged from 36 to 55 kg $CaCO_3$/t for the CO_2-NP versus 34 to 50 kg $CaCO_3$/t for the Sobek-NP. The slightly higher CO_2-NP indicates that Fe and Mn CO_3 are relatively common and their potential influence on Sobek-NP will need to be addressed if setting precise criteria for PAG and non-PAG waste rock or predicting the time to ARD becomes a concern.
- The variability was greater for AP than either of the two measures of NP. There were a number of samples with more than 10% S and as a result 90th percentiles varied from 76 to 376 kg $CaCO_3$/t. Notably, the lowest 90th percentile value was for the 10 Level dump. Apparently Rescan did not sample the highly sulphidic material encountered in the Price and Kwong (1994) study. The highest median AP was 99 kg $CaCO_3$/t for the Magazine road samples. The 10th percentile AP for the different locations ranged from 9 to 34 kg $CaCO_3$/t. The low value was for the airstrip and is likely due to dilution by the underlying till.

In 1994, Price and Kwong analyzed various size fractions in selected samples from the dumps as part of a general assessment of ARD prediction methods at a number of mines (Table 7). The objective of Price and Kwong (1994) was to analyze the range of materials present in the dumps and the proportion of sulphide-rich samples were higher than that observed in the dump as a whole. Test work conducted on the samples included various forms of geological, ABA and elemental analysis. Results from the ABA analysis include the following.

- There was significant sulphide-S in both the < 50 µm and < 2 mm size fractions. Less than 50 µm is below the grain size of much of the tailings suggesting that occlusion of pyrite in quartz is unlikely to limit sulphide oxidation.
- Analysis of different size fractions showed that while AP and NP levels were both highest in the < 2 mm fraction, the difference between the fine and coarse fractions was usually greater for AP. As a result, even where fragments of massive pyrite were observed in the 7/16 – 3/4 inch and > 3/4 inch size fractions, the < 2 mm size fraction had a lower NPR than the > 2 mm. Although there were exceptions (e.g., JM-6), the results suggested that analyses conducted on drill core and surface grab samples containing stones likely underestimate the ARD potential of the waste rock.
- Visually, the surface of the 10 and 11 Level dumps contained pockets of highly sulphidic waste rock. This was not the case on the surface or in the trench cut through the 12 Level dumps.
- Of the neutral pH samples, those with > 10% sulphide-S had a lower rinse and paste pH (rinse pH < 7, paste pH < 8) than those with lower (2 to 4%) sulphide-S (rinse pH of 7.5 to 8.1, paste pH of 7.9 to 9.3).

In 2003, Skyline conducted ABA analysis on < 2 cm grab samples from Magazine Road and the airstrip (Table 8). Sampling of the airstrip was done every 100 m at the south end near the mine site and every 200 m at the north end, 16 samples in total. Each sample consisted of a composite of sub-samples taken every 50 cm across the width of the strip at that location. The 2003 airstrip samples all had neutral paste-pH values and significant carbonate-NP. The exception was the low NP in sample 5797, which was largely till. The lower S values in the 2003 airstrip samples (median value of 0.22%) compared to 1990 (median value of 1.7% - Table 6) are attributed to the greater dilution of waste rock by till in the 2003 samples. During the 1990 sampling, the runway was in use so sampling was restricted to the margins where there was little or no dilution of the waste rock by the underlying till. Dilution by till is the likely cause of the generally lower carbonate-NP in the 2003 samples (median values of 30.5 kg/t in 2003 versus 52 kg $CaCO_3$/t in 1990). Some of the difference between the 1990 and 2003 samples may also have resulted from %S and carbonate-NP weathering.

Despite the much lower carbonate-NP values in the airstrip samples in 2003 and low Sobek-NP of the till (e.g., 4.8 kg $CaCO_3$/t), the airstrip samples from 1990 and 2003 had similar Sobek-NP values. Sobek-NP minus carbonate-NP shows the contribution of silicate-NP to Sobek-NP measurements. The much higher contribution of silicate-NP to the airstrip Sobek-NP values in 2003 was likely caused by having too high a Sobek fizz rating, which resulted in adding too much acid and dissolving too much silicate. All but one of the airstrip samples was given a moderate fizz rating. This results in an acid addition equivalent to 500 kg $CaCO_3$/t, far higher than the maximum carbonate-NP of 36.6 kg $CaCO_3$/t. A lower fizz rating would provide a more accurate estimate of silicate NP reactivity. It might also permit the identification of samples containing significant non-neutralizing, Fe and Mn CO_3.

The 2003 sampling results confirmed that the north end of the airstrip had a much lower sulphide-S content. This trend was also evident in the 1990 data (Table 6). Based on their carbonate-NPR, the ARD potential of the 2003 samples from the south end of the airstrip was either likely or uncertain. The 2003 sampling indicates that the north end of the airstrip is not PAG. The 1990 results suggest pockets of PAG material may exist along the margins.

Sampling of the Magazine Road in 2003 focused on waste rock-rich areas. Samples consisted of composites of grab sub-samples collected across the width of the road. In some cases samples were also collected from the road margins. The sampled areas of the road had a wide range of sulphide-S (0.3 to 4.9%). Seven of the ten paste pH values were below 6.5, with the lowest 4.9. The pore water or surface pH of materials with a < 2 cm paste pH less than 6.5 was probably a unit lower than the paste pH value (Price and Kwong, 1997). The Sobek-NP of 7 to 18 kg $CaCO_3$/t in the weakly acidic samples was ineffective in maintaining a near-neutral pH, either because it is unavailable (e.g., grains physically occluded in larger particles) or insufficiently reactive (e.g., weathers too slowly). Some of the AP in these samples was also physically occluded in larger particles and unavailable. One way to minimize the contribution of physically unavailable NP, resulting in a better assessment of the insufficiently chemically reactive NP in weathered waste rock would be to separately analyze the fine and coarse particles (e.g., < 2 mm and > 2 mm).

Notably, the fizz ratings for the Magazine Road samples were slight or none, which results in an acid addition equivalent to 50 or 100 kg $CaCO_3$/t. Consequently, the difference between the Sobek-NP and carbonate-NP was much lower for the Magazine Road than the airstrip samples (median values of Sobek-NP minus carbonate-NP of 5 kg $CaCO_3$/t for the Magazine Road versus 18 kg $CaCO_3$/t for the air strip). Both the airstrip and Magazine Road samples had low amounts of sulphate-S, indicating significant leaching. This is not unexpected given the high precipitation on such thin layers of waste rock.

iii. Weathering

Evidence of pyrite reactivity in this environment is provided by low rinse pH values and acidic seepage produced by naturally sulphidic talus just above the mine (samples JM7A-E in Table 7). In 1994, the evidence of acidity in the waste rock dumps was a small area of strongly discoloured (yellowish brown – 10YR 5/6 m) waste rock exposed in one of several pits excavated in the 10 Level dump (sample JM-1 in Table 6). Slight brownish gray or olive gray colours at other dump locations sampled in 1994, including waste rock with > 10% sulphide-S, were taken as an indication of weak iron oxidation (Price and Kwong, 1994).

The local pocket of ARD generating waste rock buried in the 10 Level Dump area was re-discovered in September 1996, when two shallow test pits were excavated in an attempt to bury a grizzly bear which had charged one of the staff (International Skyline Gold Corp., 1997). Water accumulating in the two test pits had low pH values (2.5 and 3.5) and high metal concentrations (Table 12). Subsequent mapping showed that the ARD generating material in the 10 Level dump was a 13m by 27m wide and 50cm deep layer of massive pyrite waste rock, hidden beneath a 30 cm layer of neutral pH waste rock. The 1996 Annual Reclamation Report also notes the presence of ARD generating bedrock in an exploration trench adjacent to the road to the Snip mine.

Additional small pockets of ARD generating waste rock were reported at the following locations in the 2000 Annual Reclamation Report (Yeager, 2001):

- containment dyke for the fuel tanks;
- 11 Level vehicle storage site;
- road crossing a small wetland just below the 11 Level Dump;
- road crossing a small wetland 60 m north of the powder magazine;
- portions of 10 Level dump near the kitchen door and adjacent to the vehicle storage site; and
- north end of the airstrip at the culvert passing the tailings pond discharge.

To date most of the waste rock has remained pH neutral. This includes the areas of high sulphide waste rock on the 11 Level dump (JM-5 and -9 in Table 6), which despite NPR values < 0.15 and $> 10\%$ sulphide-S in the < 2 mm particles, remains neutral and visually appears only weakly weathered after thirteen years of exposure (observations from 2003 site inspection). The lack of extensive ARD onset in the waste rock is likely due to the significant Ca and Mg CO_3 present in calcite and ankerite, and low site temperatures, which will reduce the rate of sulphide oxidation (see Section 1.E.ii). An attribute of all the waste rock that has gone acidic is its proximity to the water table, suggesting that leaching by groundwater resulting in accelerated carbonate dissolution has been responsible for the onset of ARD.

iv. Drainage Chemistry Monitoring

In most years drainage chemistry was measured in a sump collecting much of the drainage from the three main dumps (Tables 21 to 23). From 1997 to 2003, samples from all three site drainages had pH values between 7.1 and 8.3 (Table 24). Like the natural ARD produced by the outcrops further up the hill, the limited ARD produced by the 10 Level waste rock has been neutralized by carbonate in the surrounding waste rock and adjacent soils. Sulphate was higher below the 10 and 11 Level dumps than at the 12 Level (99 to 258 mg/L and 100 to 160 mg/L versus 35 to 66 mg/L). Notably the highest sulphate (258 mg/L) and Cu (43 µg/L) values were from the last 10 Level sample taken before the acidic waste rock was removed. For dissolved-Cu, the following trend was observed, the 10 Level concentration was > 11 Level, which was > 12 Level. At all three sites, high total-Fe and dissolved-Cu values were observed when the mine re-opened in 1993.

Having only one sample per year and no flow data make it difficult to spot trends and much of the variability in sump data from year to year and between dumps may be due to differences in precipitation and dilution rather than changes in the geochemistry of the waste rock.

v. Proposed Future Actions

Proposed future work includes the following.

- Testing the fine fraction and analysis of the carbonate minerals and their contribution to the measured NP.

- The field cells will be re-inspected and a sampling program re-instituted. If disturbance of the dumps renders the present cells inoperative, replacement cells will be constructed.

4.3. Components of the Mitigation Plan

Mitigation to date has focused on the small pockets of ARD-producing waste rock. In 1996, lime was added to the surface of the ARD-producing area identified in the 10 Level dump. The area was then covered with compacted soil and diversion ditches were excavated to limit leaching. In 2000, the veneer of ARD-producing 10 Level waste rock, along with adjacent iron-stained soil, was removed to the designated disposal location in the impoundment and flooded. In total, 41 truck loads of material were removed. An additional nine truck loads of acidic waste rock and iron-stained soil were moved to the impoundment from the tank farm. At both locations, it was difficult to segregate the ARD generating material and as a result the material that was removed contained considerable neutral pH waste rock and unstained soil. The other mitigation project conducted in 2000 was the construction of a channel to divert drainage around an ARD-generating portion of a road that crosses a small wetland just below the 11 Level Dump.

Much of the post-closure ARD work and discussions have focused on the presently neutral, but eventually ARD generating waste rock spread around the site. Remediation options considered for the remaining PAG waste rock are disposal underwater in the tailings impoundment and or in-situ measures to divert drainage and mix thin layers of waste rock with underlying calcareous till. These options are reviewed below.

i. Underwater Disposal in the Tailings Impoundment

Underwater disposal in the tailings impoundment has a number of advantages as a remediation option for the remaining unflooded PAG waste rock.

- Underwater disposal has proven effective at a number of sites as a means of minimizing sulphide oxidation and metal leaching, and preventing the acid generation. There is less experience with the underwater disposal of waste rock compared to tailings. Waste rock has larger pores, which will increase the potential rate of seepage and thus O_2 inputs. However, the mine has committed to covering flooded waste rock with compacted soil or tailings and seepage flow through the underlying strata will provide an opportunity for contaminant attenuation before the waste rock pore water emerges along the toes of the dams.
- The short snow-free period and fetch limit wave action and the potential for remobilization, as is evident from the stability of the tailings.

- The water level in the impoundment can be lowered prior to deposition to prevent discharge of surface water until the following freshet, providing ample time for the settling of suspended sediment.
- The high precipitation and cool temperatures should ensure a positive monthly water balance throughout the year, ensuring waste rock placed below the spillway remains flooded once the impoundment has refilled.
- The tailings impoundment has already been built, lessening the costs and additional risks associated with this strategy.
- There are limited, although very important, long term monitoring and maintenance requirements. The operator is required to ensure the long-term performance of a flooded impoundment, including the required monitoring and maintenance of the dams, spillways and the drainage discharge, and any additional reclamation costs associated with the above.

Limited weathering in the majority of the waste rock (at least to date), dilution in the impoundment, and the relatively high dilution and natural alkalinity in the receiving waters and long distance to sensitive resources will minimize the impact of soluble weathering products when waste rock is flooded. However, flooding should be conducted as soon as possible to minimize the build-up of potentially soluble weathering products. If significant weathering were to occur, relatively cheap supplemental remediation measures (e.g., lime to raise pH) are available to neutralize acidity and reduce metal solubility when waste rock is first placed underwater.

Possible drawbacks or limitations include excavation and transportation costs, and the limited storage capacity. Use of the waste rock for construction, especially on road surfaces, will have mixed the waste rock with other material potentially increasing the volume of material that needs to be moved. In 1993, the unused available flooded storage volume in the impoundment was estimated to be 75,943 m^3. There is additional existing capacity for underwater storage of PAG rock in the flooded portion of the underground workings. While technically attractive, this option is presently considered financially impractical because of the equipment and re-handling required for surface to underground re-handling and underground transportation.

ii. In-Situ Mitigation Measures

A potentially less costly alternative to placement in the impoundment is some form of in-situ mitigation. This measure is being considered where the ARD potential is uncertain or the amount of waste rock is relatively small, which is typically the case for waste rock used in the construction of roads and the airstrip. Depending on the quantity and quality of the waste rock and the attributes of site, in-situ mitigation measures could include placing the material in a location with minimal leaching, preventing sulphide oxidation by burying it beneath the water table, mixing it with calcareous till to create a non-PAG composite and/or limiting leaching by covering it with compacted till. Encapsulating waste rock within the till could involve burial in trenches, a possibility suggested for the airstrip, or burial within the re-contoured roadway, a possibility suggested for the Magazine Road. Potential advantages of in-situ mitigation are reduced costs and freeing up storage capacity in the impoundment for other waste rock. Supporting site attributes include the undulating terrain, which creates local wetlands, the fine particle size of much of the till, large dilution and relatively high natural alkalinity in the receiving waters, and large distance to sensitive resources.

Work on this option has been limited. More detailed site information is required on the composition and depths of till, and the depth of the water table in order to assess where this option may be feasible. Questions also remain about the calcareous nature of un-weathered till. Data indicates that the weathered surficial till contains very little NP. One of the challenges will be collecting the required information and having adequate process control for these small amounts of waste. Another challenge will be finding disposal sites where there is no potential for drainage flow paths to change and through erosion to expose the buried waste. Disposal must be in a manner that ensures long-term geotechnical and geochemical stability and minimizes monitoring and maintenance requirements.

5. Conclusion and Discussion

The ML/ARD work at Johnny Mountain Mine illustrates some of the common practices and factors to consider in assessment and mitigation. It also provides a number of lessons regarding potential ML/ARD challenges and information requirements. This includes why items such as operational material characterization, analysis of the fine waste rock size fraction, adequate mineralogical analysis and comprehensive flow monitoring are important requirements.

The waste material at Johnny Mountain with the lowest NPR, highest sulphide-S and therefore the highest theoretical potential to generate ARD is the tailings. The ARD potential was recognized prior to mining and as a result the tailings were placed in a flooded impoundment. Monitoring of the drainage indicates that flooding of the tailings has been successful in limiting sulphide oxidation. Remaining work items, such as increasing the geotechnical stability of the dams and leveling the tailings, will occur after placing waste rock in the impoundment. According to the Closure Plan, the dyke slopes will be trimmed, with the excess used as buttresses to increase their geotechnical stability. Subsequently, the main remaining liability associated with the tailings will be the limited, although very important, monitoring and maintenance of the impoundment.

Most of the focus of ML/ARD work since the mine closed has been on the waste rock. Based on a visual assessment of the degree of mineralization and ABA results from four samples, prior to mining it was concluded that the waste rock had little or no potential for ARD. Consequently, all the waste rock was placed in aerial dumps or used for construction. Analysis conducted after the mine closed indicates that most of the waste rock is potentially ARD generating, and preventing ARD will require costly re-handling. The experience with the waste rock illustrates the importance of ABA analysis for material characterization and the danger of relying on geological suppositions or visual observations of the extent of mineralization.

The rock that is potentially ARD generating (PAG) includes material where ARD is predicted and material where the ARD potential is uncertain. Even with additional analysis, much of the ‘uncertain’ waste rock will likely fall in the compositional gray zone where are present prediction tools are unable to ascertain whether ARD will occur in the future. Even where the ARD potential is high, most of the waste rock has to date produced neutral pH drainage with relatively low metal concentrations. The proposed mitigation plan for the waste rock in the dumps and areas where the waste rock has been used as fill is to place it underwater in the tailings impoundment. Small portions of the rock that have already gone acid were picked up and placed in the impoundment in 1996. Where flooding is planned, it should be done as soon as possible to minimize the build-up of potentially soluble weathering products.

An alternative being considered where the waste rock ARD potential is limited or the amount is relatively small is some form of in-situ mitigation. This could include encapsulating within the till by burial in trenches, a possibility suggested for the airstrip, or burial within the re-contoured roadway, a possibility suggested for the Magazine Road. Challenges with in-situ mitigation include collecting the required information and having adequate process control. Another important consideration will be ensuring that placement is done in a manner that ensures long-term geochemical stability.

The third mine component with ML/ARD concerns is the underground workings. The composition of the walls in the workings is expected to be similar to that of the waste rock, although there is much lower surface area. Like the waste rock, drainage from the underground workings presently has a neutral pH with relatively low metal concentrations. The proposed ARD mitigation plan for the underground workings is to limit oxygen entry by placing bulkheads in the adits and covers over the ventilation raises. Questions regarding this plan include the effectiveness of bulkheads and covers in limiting air entry and the impact of progressive rock fall on the ARD potential, especially in ore zones with high metals and pyrite. Like the waste rock, the underground contains both materials with a high ARD potential and material where the ARD potential is uncertain. In addition, the underground has significant NP added in groundwater alkalinity, making the ARD potential of the underground workings as a whole uncertain.

A major consideration in mitigation planning for Johnny Mountain is how much monitoring and maintenance will be required to determine the success of its long-term performance. For example, what monitoring is required to demonstrate that underground oxygen levels are low enough? The lack of a permanent site presence, coupled with low clouds, strong winds and deep snow cover for much of the year make monitoring and maintenance at Johnny Mountain costly and onerous undertakings. Unless there is a clear reason to do otherwise to protect the environment, from the perspective of minimizing risk and liability, the best mitigation strategy will be the one requiring minimal monitoring and maintenance, and no manual monitoring and maintenance during the winter.

The experience at Johnny Mountain illustrates the importance of conducting comprehensive site-specific ML/ARD work. It also illustrates some of the gaps that exist in our understanding and limitations in our ability to collect the information required for accurate ML/ARD assessment and cost effective remediation. One example is our limited understanding of the rate of air entry, oxygen concentrations and corresponding sulphide oxidation rates in closed, plugged underground mines in mountainous terrain. Examples of information that has proven difficult to collect includes:

- the variability in flow and drainage chemistry, both for the receiving environment and different site components;
- the difficulty in predicting the NP, especially for materials with complex carbonate mineralogy; and
- the influence of site-specific conditions to the available AP and NP, and whether modifications are required to general assessment rules.

By modern mine standards, the size of the underground workings and waste production at Johnny Mountain were relatively modest. This, coupled with the significant natural dilution (e.g., glaciers) and attenuation (e.g., high alkalinity in natural drainage) and the uncertain ARD potential of some of the rock, may mean that measures to minimize leaching are all that is required to prevent significant impacts from at least a portion of the exposed PAG materials. In some cases, even with ARD production the environmental impacts will be minimal, as is presently the case with ARD production within the underground workings. Determining when this will be the case, given the lack of information on the variability in flow and drainage chemistry, both for the receiving environment and different site components will be a challenge.

Even when the mine was in operation with full time staff available for snow removal, the mine had trouble conducting year round monitoring at all the receiving environment sites (Yeager, 2002). Now the mine is closed, site visits during the more than 6 months of winter would require a major expense. The weight of snow has crushed many of the mine buildings and other structures. Poor visibility, deep snow, changing flow paths and high flows and large debris loads in the creeks at certain times of the year make maintaining monitoring apparatus a major logistical and financial challenge. Consequently, Skyline has concluded that continuous flow monitoring would be prohibitively expensive and, due to the inevitable equipment failure, no more informative than estimating flow, and the consequent dilution, based on relative surface areas and existing monitoring.
Potential problems with this approach include:

- potential for significant differences in drainage contribution from different areas of the site (e.g., glaciers versus dump bench), during different times of the year and in different years; and
- how to predict groundwater flow (e.g., discharge from different drainages in the underground workings).

Supporting site data would greatly improve the confidence regarding potential loadings, natural dilution and attenuation, impacts to the receiving environment and the effectiveness of different mitigation measures to reduce leaching if PAG site components are allowed to generate ARD. The question of how to conduct continuous flow monitoring and collect samples for periodic drainage chemistry measurements in high snow fall areas where power and access are limited is a problem at a number of closed British Columbia mine sites. While the Johnny Mountain case emphasizes the importance of conducting detailed operational monitoring, it should be noted that site hydrology and hydrogeology often change dramatically after the mining stops, creating a need for post-closure monitoring.

The problems in predicting the NP in the materials at Johnny Mountain result from the complex carbonate mineralogy, previous omissions and basic gaps in our understanding. Static laboratory NP measurements are crude measurements that can only provide a preliminary guide to the actual field neutralization capacity. Consideration of the contributing mineralogy and the rate of sulphide oxidation are required for the interpretation of results. Results of the Price and Kwong (1994) work at Johnny Mountain and a number of other sites shows that CO_2-NP > Sobek-NP is a good indication that there is significant Fe and Mn carbonate present. The potential contribution of Fe and Mn CO_3 minerals at this site means that the NP calculated from %CO_2 cannot be relied on as a measure of the field NP. Although it is a better measure of NP for this site than NP calculated from the %CO_2, the presence of ankerite and siderite also raises concerns regarding the potential contribution of Fe and Mn CO_3 to the Sobek-NP. This is compounded for the older data by the lack of information on the fizz rating.

Better NP information will be required if one is setting precise criteria for PAG and non-PAG waste rock or in predicting the time for ARD to become a concern. This could include checking whether Fe and Mn CO_3 contribute to the Sobek-NP, by comparison with results from modifications to the standard Sobek method that remove the contribution of Fe and Mn CO_3 (Meek, 1981; Skousen et al., 1997; White et al., 1998) and the use of Rietveld XRD analysis to quantify the proportion of carbonate occurring as calcite, ankerite and siderite. Microprobe work is required to ascertain the proportion of Fe, Mn, Ca and Mg in ankerite. A perceived drawback with Rietveld XRD and microprobe analysis is the cost: $200/sample for the Rietveld and $100/sample for the microprobe (for 6 samples with 5 grains analyzed per sample). However, the cost of Rietveld XRD analysis is not much higher than ABA plus ICP analysis, and minimal when compared to the potential costs of not knowing the carbonate mineralogy.

A basic information gap at Johnny Mountain and many other sites in British Columbia is the lack of data on field rates versus laboratory measurements of processes, such as sulphide oxidation. This information is required to determine when and where modifications are required to laboratory measurements and general ABA rules, and to answer questions such as how long is it permissible to wait until the large waste rock stockpiles must be moved to the impoundment. The impact of site-specific conditions on the rate of sulphide oxidation would also help answer whether additional neutralization sources, such as the high groundwater alkalinity, are able to continue neutralizing acidic drainage produced by the underground workings.

6. References

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The Johnny Mountain reports are available in the Smithers and Victoria offices of the British Columbia Ministry of Energy and Mines.

TABLES, FIGURES AND PHOTOS

Table 1. Production Report (from Minfile Report 104 B 107)

Production Year	Tonnes Mined	Tonnes Milled	Commodity	Grams Recovered	Kilograms Recovered
1993	21,850	21,850	Silver	407,000	
			Gold	217,700	
1990	74,936	86,865	Silver	1,334,263	
			Gold	906,754	
			Copper		347,633
1989	85,944	94,282	Silver	2,485,451	
			Gold	1,544,083	
			Copper		643,386
1988	13,628	24,250	Silver	122,100	
			Gold	146,856	
			Copper		17,090

Summary Totals:		**Metric**		**Imperial**	
	Mined:	196,358	tonnes	216,387	tons
	Milled:	227,247	tonnes	250,426	tons
Recovery:	Silver:	4,348,814	grams	139,818	ounces
	Gold:	2,815,393	grams	90,517	ounces
	Copper:	1,008,109	kilograms	2,222,477	pounds

Table 2. Petrographic Results of Representative Materials in the Waste Rock (compiled from Rescan, 1991)

	Qtz	Plag	Kfeld	Ser	Carb	Pyr	Chalco	Biot	Chl	Clays	Epid	Magn	Rut	Hem	Sph	Lim	Apa
SK-1	55			3	10	25	3	2								2	
SK-2	20	5		20	30	10	tr	10					2		3		
SK-3	15	15	30	25	5	5	tr		5						tr	tr	
SK-4	20	25	25	15	10	2			3				tr				
SK-5	45			2	5	40	5	2									
SK-6	10	30	25	10	10	3	1	10				tr	tr		tr		
SK-7	5	30	5	2	5			45	5			2					1
SK-8	60		10	5	5	3	2			15							
SK-9	3	15			10	5		35			30		tr				2
SK-10	10	30	25	10	10	3		10					1				1

Qtz - quartz; Plag - plagioclase; Kfeld - potassium feldspar; Ser - sericite; Carb - carbonate; Pyr – pyrite; Chalco – chalcopyrite; Biot – biotite; Chl – chlorite; Clays – phyllosilicates; Epid – epidote; Magn – magnetite; Rut – rutile; Hem – hematite; Sph – sphene; Lim – limonite; Apa – apatite

Table 3. Petrographic Results from Different Size Fractions in Waste Rock Samples Collected by Price and Kwong in 1994 (tabulated by John Kwong)

	Qtz	Plag	Kfeld	Ser	Carb	Pyr	Chalco	Biot	Chl	Clays	Epid	Magn	Rut	Hem	Sph	Gal	Pyrr
JM-1 <2 mm	10	30	9	7		42	tr	2	tr	1	tr	tr		tr			
JM-1-2-11 mm	25	30	20	7	tr	16	1	1		1							
JM-1 >3/4"	5	15	42	6	2	10	3		2						tr	tr	
JM-1 >3/4"	5	11	58	20	5	1									tr		
JM-1 >3/4"	45			tr		55											
JM-1 >3/4"		48		22	11			18				1					
JM-2 <2 mm	5	60	11	7	1.5	9	tr	5		tr	1	tr					tr
JM-2-2-11 mm		66	12	8	5	2	tr	7			tr						
JM-2 >3/4"	2	63	10	10	5	tr		8		22	2						
JM-2 >3/4"	2	38	40	13	2	2		3									
JM-2 >3/4"	3		38	3	tr	6	tr										
JM-2 >3/4"	3	18	12		43	tr			4		20			tr			
JM-3 <2 mm	14	23	30	5	1	25	2	tr	tr								
JM-3-2-11 mm	20	13	54	2	2	9	tr		tr	1							tr
JM-3 >3/4"			77	12	3	3	0.5	tr									
JM-3 >3/4"	2	20	46	24	2	5	tr	1	tr			tr					
JM-4 <2 mm	4	66	15	3	1.5	9	0.5	1		1		tr		tr		tr	
JM-4-2-11 mm	1	28	50	7	9	3	tr	2		2			tr				
JM-4 >3/4"	1	20	35	28	2	2		tr	tr								
JM-4 >3/4"	tr	27	60	8	2.5	1		2					tr				
JM-5 <2 mm	20	21	20	3	1	32	1	2									
JM-5-2-11 mm	10	28	45	5	3	7	0.2	2									
JM-5 >3/4"	2	33	30	20	2	5		8									
JM-5 >3/4"	5	20	48	20	2	5		2									
JM-6 <2 mm	5	66	10	5	2	9	tr	3							tr		
JM-6-2-11 mm	1	23	38	28	4	4		2	tr				tr			tr	
JM-6 >3/4"	2	30	50	11	5	2		tr		4			tr				

10 Level: JM 1,2 and 3; 11 Level: JM 4 and 5; 12 Level: JM 6
Qtz - quartz; Plag - plagioclase; Kfeld - potassium feldspar; Ser - sericite; Carb - carbonate; Pyr – pyrite; Chalco – chalcopyrite; Biot – biotite; Chl – chlorite; Clays – phyllosilicates; Epid – epidote; Magn – magnetite; Rut – rutile; Hem – hematite; Sph – sphene; Gal – galena ; Pyrr – pyrrhotite

Table 4. Results from Different Mineralogical Analyses on Selected Waste Rock Samples Collected by Price and Kwong in 1994 (tabulated by John Kwong)

Minerals	Petro-graphic	XRD (BC Mines)	XRD (SRC)	XRD (NHRI)	Point counting with SEM-EDX		Calculated based on geochem data	
	modal %	wt %	wt %	wt %	*modal %*	wt %	wt %	Assumptions
JM-1								
Pyrite	*42*	>5	35	43.4	*33.5*	46.7	43.65	all S in py and cpy
Quartz	*10*	>5	43	23.1	*28.25*	21.76		
K-feldspar	*9*	>5		6.3	*16*	12		
Albite/Plagioclase	*30*	>5	13	11.6	*10.75*	8.31	9.56	all Na in albite
Illite	*7*	>5	8	6.6	*6*	4.71		
Biotite	*2*				*3.25*	2.93		
Chlorite	*Tr*	2-5		2.3	*2*	1.63		
Epidote	*Tr*				*0.25*	0.25		
Amphibole					*0.25*	0.24		
Chalcopyrite	*Tr*				*1*	1.22	0.59	all Cu in chalcopytite
Goethite	*Tr*				*0.25*	0.28		
Calcite		<2					0.5	all CO_2 in calcite
Mixed-layer clay				6.6				
JM-3								
Pyrite	*25*	>5	35	21.7	*17.25*	27.99	27.34	all S in py and cpy
Quartz	*14*	>5	35	35.4	*36.5*	31.27		
K-feldspar	*30*	>5	26	14.5	*31.75*	26.48		
Albite/plagioclase	*23*				*1.5*	1.29	3.22	all Na in albite
Illite	*5*	>5	4	7.7	*6.25*	5.46		
biotite	*Tr*				*1*	1		
Chlorite	*Tr*	2-5		6.8	*0.75*	0.68		
Chalcopyrite	*2*	<2		5.9	*2*	2.72	2.32	all Cu in chalcopyrite
Goethite					*0.25*	0.31		
Calcite		2-5		1.5				
Dolomite/ ankerite	*1*			6.6	*1.5*	1.41	4.1	all CO_2 in ankerite
Siderite					*0.75*	0.93		
Kaolinite					*0.25*	0.21		
Apatite					*0.25*	0.25		
JM-6								
Pyrite	*9*		4	10	*7.75*	13.52	9.09	all S in pyrite
Quartz	*5*	>5	43	29.1	*24*	22.1		
K-feldspar	*10*	>5	15	10.2	*21.5*	19.3		
Albite/plagioclase	*66*		14	14.8	*14.5*	13.4	12.52	all Na in albite
Illite	*5*	>5	22	18.6	*16.5*	15.48		
Biotite	*3*				*7*	7.54		
Chlorite		2-5	2	4.9	*0.75*	0.73		
Chalcopyrite	*Tr*							
Ankerite	*2*			5.8	*3.25*	3.28	3.6	1/2 CO_2 in ankerite
Calcite		2-5		6.6	*4.25*	4	3.6	1/2 CO_2 in calcite
Apatite					*0.25*	0.27		
Ilmenite					*0.25*	0.41		
Augite		<2						

SRC: Saskatchewan Research Council
NHRI: National Hydrology Research Institute

Table 5. ABA Data for the Underground Workings (adapted from Yeager, 2002b)

Sample ID	Location	Paste pH	Sulphur %	AP kg $CaCO_3$/t	NP Sobek kg $CaCO_3$/t	NNP kg $CaCO_3$/t	NPR Sobek
4297	12 Level - HW	9.7	1.0	31	41	10	1.3
4298	12 Level - HW	9.9	2.4	76	51	-25	0.7
4299	12 Level - FW	8.0	0.6	20	101	81	5.1
4974	12 Level - FW	8.6	1.9	58	71	13	1.2
4975	12 Level - FW	9.5	1.5	48	58	10	1.2
4976	12 Level - FW	9.5	1.6	51	44	-7	0.9
4977	12 Level - FW	9.0	0.7	21	35	13	1.6
4978	12 Level - FW	8.6	1.8	57	39	-18	0.7
90th Percentile		9.8	2.0	63	80	34	2.7
Median		9.3	1.6	49	47	10	1.2
10th Percentile		8.4	0.7	21	38	-20	0.7
4282	11 Level - HW	9.7	1.5	47	56	9	1.2
4283	11 Level - HW	9.1	0.8	26	58	32	2.2
4284	11 Level - HW	8.6	1.4	42	52	10	1.2
4285	11 Level - V	7.6	2.1	67	25	-42	0.4
4286	11 Level - FW	9.4	1.6	48	61	13	1.3
4287	11 Level - FW	9.3	0.9	28	54	26	1.9
4288	11 Level - V	9.1	3.1	95	61	-34	0.6
4289	11 Level - FW	9.6	1.1	33	48	15	1.5
4290	11 Level - FW	10.0	1.4	43	47	4	1.1
4291	11 Level - FW	9.3	1.4	43	65	23	1.5
4292	11 Level - V	8.9	2.8	86	51	-35	0.6
4293	11 Level - V	9.9	2.7	85	42	-43	0.5
4294	11 Level - V	10.1	2.2	68	31	-38	0.4
4295	11 Level - FW	9.3	0.9	27	62	35	2.3
4296	11 Level - HW	8.9	0.5	16	82	66	5.1
90th Percentile		10.0	2.7	86	64	33	2.2
Median		9.3	1.4	43	54	10	1.2
10th Percentile		8.7	0.9	27	35	-40	0.5
4261	10 Level - HW	8.5	0.8	25	64	39	2.6
4262	10 Level - HW	8.0	2.0	62	71	9	1.1
4263	10 Level - HW	8.5	3.0	94	61	-33	0.6
4264	10 Level - HW	8.9	1.8	56	59	3	1.1
4265	10 Level - HW	8.8	4.8	151	58	-92	0.4
4266	10 Level - HW	8.3	1.6	50	48	-2	1.0
4267	10 Level - HW	9.3	1.3	41	41	0	1.0
4268	10 Level - HW	9.2	1.5	47	49	2	1.0
4269	10 Level - HW	9.4	0.5	15	44	29	2.9
4280	10 Level - HW	8.6	1.9	58	50	-8	0.9
4281	10 Level - HW	9.7	1.3	39	39	0	1.0
90th Percentile		9.4	3.0	94	64	29	2.6
Median		8.8	1.6	50	50	0	1.0
10th Percentile		8.3	0.8	25	41	-33	0.6
4270	10-9 Decline - HW	9.6	0.4	13	49	35	3.7
4271	10-9 Decline - HW	9.0	0.2	5	48	43	9.6
4272	10-9 Decline - HW	9.1	0.8	24	47	22	1.9
4273	10-9 Decline - HW	9.2	0.9	28	49	20	1.7
4274	10-9 Decline - HW	8.8	0.6	18	50	32	2.8
4275	10-9 Decline - HW	8.9	1.1	33	81	48	2.4
90th Percentile		9.4	1.0	31	65	45	6.6
Median		9.1	0.7	21	49	34	2.6
10th Percentile		8.9	0.3	9	47	21	1.8
4276	9 Level - HW	8.4	2.7	83	43	-40	0.5
4277	9 Level - V	8.8	5.4	167	36	-131	0.2
4278	9 Level - V	8.2	6.4	199	37	-162	0.2
4279	9 Level - V	9.5	1.8	55	46	-9	0.8
90th Percentile		9.3	6.1	189	45	-18	0.7
Median		8.6	4.0	125	40	-85	0.4
10th Percentile		8.3	2.0	63	36	-152	0.2

Note: No record of inorganic CO_2 for underground samples
HW = Hangingwall; V = Vein; FW = Footwall

Table 6. ABA Data for Waste Rock Placed around the Site (adapted from Yeager, 2002b)

Sample ID	Location	Paste pH	Sulphur %	Max. AP kg $CaCO_3$/t	NP Sobek kg $CaCO_3$/t	CO_2 %	NP CO_2 kg $CaCO_3$/t	NNP Sobek kg $CaCO_3$/t	NPR Sobek	NPR CO_2
Sky-06	12 Level Dump	8.8	1.2	37	47	2.1	48	10	1.3	1.3
Sky-07	12 Level Dump	9.1	0.6	17	93	5.4	123	76	5.5	7.2
Sky-08	12 Level Dump	8.6	2.1	66	60	4.2	96	-6	0.9	1.4
Sky-09	12 Level Dump	9.2	1.5	47	44	1.8	41	-3	0.9	0.9
Sky-10	12 Level Dump	8.7	1.2	38	29	2.1	48	-9	0.8	1.3
Sky-11	12 Level Dump	9.1	1.9	61	50	2.2	50	-11	0.8	0.8
Sky-12	12 Level Dump	8.6	3.5	111	56	2.8	64	-55	0.5	0.6
Sky-13	12 Level Dump	8.8	3.5	109	50	2.4	55	-59	0.5	0.5
Sky-14	12 Level Dump	8.8	1.0	32	63	2.9	66	31	2.0	2.1
Sky-15	12 Level Dump	8.9	3.1	98	21	1.0	23	-77	0.2	0.2
Sky-16	12 Level Dump	8.8	0.9	27	44	27	61	17	1.6	2.3
90th Percentile		9.1	3.5	109	63	4.2	96	31	2.0	2.3
Median		8.8	1.5	47	50	2.4	55	-6	0.9	1.3
10th Percentile		8.6	0.9	27	29	1.8	41	-59	0.5	0.5
Sky-18	11 Level Dump	7.9	11.3	353	52	2.8	64	-301	0.1	0.2
Sky-19	11 Level Dump	9.2	0.7	22	37	1.7	39	15	1.7	1.8
Sky-20	11 Level Dump	8.1	6.8	213	31	1.4	32	-182	0.1	0.1
Sky-21	11 Level Dump	8.6	1.9	58	51	2.9	66	-7	0.9	1.1
Sky-22	11 Level Dump	8.3	0.9	29	45	2.2	50	16	1.6	1.7
Sky-23	11 Level Dump	9.2	0.6	20	42	1.6	36	22	2.1	1.8
Sky-24	11 Level Dump	8.3	4.4	138	43	1.6	36	-95	0.3	0.3
Sky-25	11 Level Dump	7.6	12.5	391	20	0.2	5	-371	0.1	0.0
Sky-26	11 Level Dump	8.4	2.4	76	79	2.9	66	3	1.0	0.9
Sky-27	11 Level Dump	9.0	1.4	44	35	1.4	32	-9	0.8	0.7
Sky-28	11 Level Dump	7.8	13.2	431	39	1.7	39	-392	0.1	0.1
Sky-29	11 Level Dump	9.0	1.1	35	41	2.9	66	6	1.2	1.9
Sky-30	11 Level Dump	8.8	1.0	32	52	2.5	57	20	1.6	1.8
Sky-31	11 Level Dump	9.0	1.6	49	48	2.8	64	-1	1.0	1.3
Sky-32	11 Level Dump	9.1	0.5	17	38	2.1	48	21	2.2	2.8
90th Percentile		9.2	12.0	376	52	2.9	66	21	1.9	1.9
Median		8.6	1.6	49	42	2.1	48	-1	1.0	1.1
10th Percentile		7.8	0.7	21	33	1.4	32	-343	0.1	0.1
Sky-44	10 Level Dump	8.9	1.8	56	76	3.3	75	20	1.4	1.3
Sky-45	10 Level Dump	8.7	1.0	31	32	1.4	32	1	1.0	1.0
Sky-46	10 Level Dump	9.0	1.2	39	35	2.1	48	-4	0.9	1.2
Sky-47	10 Level Dump	8.7	2.9	90	39	1.4	32	-51	0.4	0.4
Sky-48	10 Level Dump	9.1	1.3	40	40	2.3	52	0	1.0	1.3
90th Percentile		9.1	2.4	76	62	2.9	66	12	1.2	1.3
Median		8.9	1.3	40	39	2.1	48	0	1.0	1.2
10th Percentile		8.7	1.1	34	33	1.4	32	-32	0.6	0.6
Sky-33	Magazine Road	9.0	1.2	37	44	1.7	39	7	1.2	1.0
Sky-34	Magazine Road	8.1	7.7	240	35	1.6	36	-205	0.1	0.2
Sky-35	Magazine Road	8.2	0.5	16	22	0.9	20	6	1.4	1.3
Sky-36	Magazine Road	8.9	1.0	30	34	1.2	27	4	1.1	0.9
Sky-37	Magazine Road	8.6	1.7	52	35	1.7	39	-17	0.7	0.7
Sky-38	Magazine Road	6.9	5.7	177	12	1.1	25	-165	0.1	0.1
Sky-39	Magazine Road	7.9	6.2	195	31	1.7	39	-164	0.2	0.2
Sky-40	Magazine Road	8.3	3.2	99	39	1.7	39	-60	0.4	0.4
Sky-41	Magazine Road	8.2	2.2	69	31	1.2	27	-38	0.4	0.4
Sky-42	Magazine Road	7.7	9.0	280	26	1.5	34	-254	0.1	0.1
Sky-43	Magazine Road	8.5	3.9	120	52	2.9	66	-68	0.4	0.5
90th Percentile		8.9	7.7	240	44	1.7	39	6	1.2	1.0
Median		8.2	3.2	99	34	1.6	36	-60	0.4	0.4
10th Percentile		7.7	1.0	30	22	1.1	25	-165	0.1	0.1
Sky-49	Airstrip	8.4	4.7	147	39	2.4	55	-108	0.3	0.4
Sky-50	Airstrip	7.8	12.3	384	34	1.4	32	-350	0.1	0.1
Sky-51	Airstrip	8.3	2.9	91	42	3.2	73	-49	0.5	0.8
Sky-52	Airstrip	8.7	0.8	25	44	2.3	52	19	1.8	2.1
Sky-53	Airstrip	8.8	1.8	55	65	2.9	66	10	1.2	1.2
Sky-54	Airstrip	8.8	2.7	85	51	2.7	61	-34	0.6	0.7
Sky-55	Airstrip	8.9	2.5	79	45	3.0	68	-34	0.6	0.9

Sample ID	Location	Paste pH	Sulphur %	Max. AP kg CaCO$_3$/t	NP Sobek kg CaCO$_3$/t	CO$_2$ %	NP CO$_2$ kg CaCO$_3$/t	NNP Sobek kg CaCO$_3$/t	NPR Sobek	NPR CO$_2$
Sky-56	Airstrip	8.7	0.3	10	47	2.4	55	37	4.7	5.5
Sky-57	Airstrip	9.0	0.6	19	70	3.2	73	51	3.7	3.8
Sky-58	Airstrip	8.8	1.7	54	52	2.2	50	-2	1.0	0.9
Sky-59	Airstrip	9.1	0.1	4	48	1.8	41	44	12.0	10.2
Sky-60	Airstrip	8.7	2.7	84	55	2.2	50	-29	0.7	0.6
Sky-61	Airstrip	9.2	0.4	13	62	2.1	48	49	4.8	3.7
Sky-62	Airstrip	7.9	0.3	8	23	0.7	16	15	2.9	2.0
Sky-63	Airstrip	7.0	0.4	11	7	0.1	2	-4	0.6	0.2
90th Percentile		9.1	4.0	125	64	3.1	71	47	4.7	4.8
Median		8.7	1.7	54	47	2.3	52	-2	1.0	0.9
10th Percentile		7.8	0.3	9	27	1.0	22	-84	0.3	0.3

Table 7. ABA Data for Different Particle Sizes for Selected Waste Rock, Outcrop and a Soil Samples Collected by Price and Kwong in 1994 (tabulated by John Kwong)

ID	Dump	Size Fract. (mm)	pH (1:1)	Paste pH	Total %S (TS)	S'ide%S (SS)	Acid Sol. S'ate %S	TS-AP (kg $CaCO_3$/t)	SS-AP (kg $CaCO_3$/t)	Sobek NP (kg $CaCO_3$/t)	CO_2 %	CO_2-NP (kg $CaCO_3$/t)	CO_2 NP /Sobek NP	SS Sob NPR	SS CO_2 NPR
JM-1	10	<0.05			9.4	9.1	0.03	294	291	7	0.2	5	65	0.02	0.02
JM-1	10	< 2	3.1	3.2	16.3	16.0	0.75	509	499	-5	0.2	5	-91	-0.01	0.01
JM-1	10	2 - 11	3.1	4.8	14.2	14.4	0.28	444	450	5	0.2	5	91	0.01	0.01
JM-1	10	11 - 19		6.1	17.5	17.5	0.2	547	547	13	0.4	9	70	0.02	0.02
JM-1	10	>19		6.9	11.3	11.5	0.06	353	360	27	0.7	16	59	0.08	0.04
JM-2	10	<0.05			2.7	2.6	0.02	84	82	28	1	23	81	0.34	0.28
JM-2	10	< 2	7.5	7.9	2.7	2.7	0.05	84	83	40	*0.1*	2	6	0.48	0.03
JM-2	10	2 - 11	7.7	8.3	1.3	1.1	0.01	39	37	40	1.8	41	102	1.07	1.10
JM-2	10	11 - 19		9.3	2.1	2.0	*0.01*	65	63	37	1.3	30	80	0.59	0.47
JM-2	10	>19		9.3	0.7	0.7	*0.01*	21	20	60	2	45	76	2.95	2.24
JM-3	10	<0.05			10.5	10.5	0.03	328	328	16	0.8	18	114	0.05	0.06
JM-3	10	< 2	6.2	6.9	10.7	11.0	0.04	334	344	19	1.8	41	215	0.06	0.12
JM-3	10	2 - 11	5.8	7.4	7.4	7.2	0.01	231	229	20	0.9	20	102	0.09	0.09
JM-3	10	11 - 19		8	5.0	5.2	0.01	155	162	26	0.8	18	70	0.16	0.11
JM-3	10	>19		7	17.9	18.5	0.04	559	577	15	0.6	14	91	0.03	0.02
JM-4	11	<0.05			2.4	2.0	0.01	73	71	26	1.4	32	122	0.36	0.45
JM-4	11	< 2	7.8	8.1	3.0	3.0	0.04	93	93	36	1.9	43	120	0.39	0.47
JM-4	11	2 - 11	7.9	8.8	2.0	1.9	*0.01*	62	60	37	1.9	43	117	0.61	0.72
JM-4	11	11 - 19		8.9	1.4	1.3	*0.01*	44	43	37	2	45	123	0.87	1.07
JM-4	11	>19		8.9	0.6	0.6	0.01	19	19	36	1.8	41	114	1.89	2.15
JM-5	11	<0.05			10.8	11.0	0.02	338	343	19	1.7	39	203	0.06	0.11
JM-5	11	< 2	6.8	7.1	13.5	13.7	0.06	422	429	23	1.3	30	129	0.05	0.07
JM-5	11	2 - 11	6.9	7.7	7.9	8.2	*0.01*	246	257	27	1.1	25	93	0.11	0.10
JM-5	11	11 - 19		7.7	8.6	8.9	0.02	270	279	25	1.2	27	109	0.09	0.10
JM-5	11	>19		7.7	8.1	7.8	0.01	253	251	21	0.9	20	97	0.08	0.08
JM-9	11	<0.05			10.0	9.8	0.13	312	306	36	2.1	48	133	0.12	0.16
JM-9	11	< 2	6.6	7.5	11.8	11.7	0.07	369	366	49	2.7	61	125	0.13	0.17
JM-9	11	2 - 11	6.7	7.7	9.6	9.6	0.04	300	299	47	2.4	55	116	0.16	0.18
JM-9	11	11 - 19		7.8	7.6	7.7	0.03	238	239	37	2.3	52	141	0.15	0.22
JM-9	11	>19		7.8	7.6	7.6	0.03	238	237	56	3.3	75	134	0.24	0.32
JM-6	12	<0.05			3.4	3.3	0.03	106	103	39	2.2	50	128	0.38	0.49
JM-6	12	< 2	8.1	8.3	3.5	3.3	0.03	108	105	67	3.1	71	105	0.64	0.67
JM-6	12	2 - 11	8.0	8.5	2.9	2.8	*0.01*	89	87	43	2.6	59	138	0.49	0.68
JM-6	12	11 - 19		8.6	3.8	3.6	0.01	119	117	39	2.8	64	163	0.33	0.55
JM-6	12	>19		8.5	3.3	3.3	0.01	103	103	40	2.5	57	142	0.39	0.55
JM-7A	Outc	<0.05			0.3	0.3	0.05	10	8	1	*0.1*	2	227	0.12	0.27
JM-7A	Outc	< 2	2.9	2.7	1.3	0.8	0.47	40	24	-6	*0.1*	2	-38	-0.25	0.09
JM-7A	Outc	2 - 11	2.9	3.3	2.3	1.9	0.36	73	59	-4	*0.1*	2	-57	-0.07	0.04
JM-7B	Outc	>19		5.4	6.4	5.8	0.37	199	187	0	*0.1*	2		0.00	0.01
JM-7C	Outc	>19		6.6	0.2	*0.0*	0.23	7	0	1	*0.1*	2	227	25.00	14.56
JM-7D	Outc	>19		4.9	13.1	13.3	0.19	409	415	0	0.2	5		0.00	0.01
JM-7E	Outc	>19		7.4	5.9	5.7	0.02	186	183	38	1.6	36	96	0.21	0.20

Outc:	outcrop
S':	sulph
SS:	Sulphide-S
TS:	Total-S

Table 8. ABA Results for Airstrip and Magazine Road - August 29, 2003

Sample	Paste pH	CO_2 (Wt.%)	CO_2 -NP***	Total -S (Wt.%)	SO_4 -S (Wt.%)	Sulphide -S* (Wt.%)	AP ** ***	Sob-NP***	Fizz Rating	Sob-NP minus CO_2-NP	Sob-NPR	CO_2-NPR
Airstrip (south)												
5784	7.8	1.41	32.0	1.52	<0.01	1.52	47.5	53.2	moderate	21.2	1.1	0.7
5785	7.2	1.05	23.9	1.56	<0.01	1.56	48.8	41.8	moderate	17.9	0.9	0.5
5786	7.2	0.93	21.1	3.41	0.01	3.40	106.3	49.4	moderate	28.3	0.5	0.2
5787	8.1	1.59	36.1	0.62	<0.01	0.62	19.4	46.8	moderate	10.7	2.4	1.9
5788	8.0	1.61	36.6	0.71	<0.01	0.71	22.2	57.0	moderate	20.4	2.6	1.6
90th Percentile	8.1	1.6	36.4	2.7		2.7	83.3	55.5		25.4	2.5	1.8
Median	7.8	1.4	32.0	1.5		1.5	47.5	49.4		20.4	1.1	0.7
10th Percentile	7.2	1.0	22.2	0.7		0.7	20.5	43.8		13.6	0.6	0.3
Airstrip (north)												
5789	8.0	1.37	31.1	0.31	<0.01	0.31	9.7	48.1	moderate	17.0	5.0	3.2
5790	8.1	1.31	29.8	0.23	<0.01	0.23	7.2	46.8	moderate	17.0	6.5	4.1
5791	8.3	1.45	33.0	0.17	<0.01	0.17	5.3	49.4	moderate	16.4	9.3	6.2
5792	8.3	1.39	31.6	0.09	<0.01	0.09	2.8	50.6	moderate	19.0	18.0	11.2
5793	8.2	1.2	27.3	0.09	<0.01	0.09	2.8	46.8	moderate	19.5	16.6	9.7
5794	8.4	1.48	33.6	0.20	<0.01	0.20	6.3	49.4	moderate	15.8	7.9	5.4
5795	8.1	0.99	22.5	0.04	<0.01	0.04	1.3	39.2	moderate	16.7	31.4	18.0
5796	7.4	1.13	25.7	0.07	<0.01	0.07	2.2	26.6	moderate	0.9	12.2	11.7
5797	7.0	0.23	5.2	0.03	<0.01	0.03	0.9	13.5	slight	8.3	14.4	5.6
90th Percentile	8.3	1.5	33.1	0.2		0.2	7.7	49.6		19.2	20.7	13.0
Median	8.1	1.3	29.8	0.1		0.1	2.8	46.8		16.8	12.2	6.2
10th Percentile	7.3	0.8	19.0	0.0		0.0	1.2	24.0		11.3	6.2	4.0
Magazine Road												
314451	7.7	0.77	17.5	0.61	<0.01	0.61	19.1	24.8	slight	7.3	1.3	0.9
314452	6.3	0.23	5.2	0.28	0.01	0.27	8.4	13.3	slight	8.1	1.6	0.6
314453	6.1	0.46	10.5	1.67	0.02	1.65	51.6	15.5	slight	5.0	0.3	0.2
314454	6.3	0.68	15.5	1.92	0.02	1.90	59.4	18.0	slight	2.5	0.3	0.3
314455	5.7	0.4	9.1	4.18	0.05	4.13	129.1	13.3	slight	4.2	0.1	0.1
314456	5.4	0.16	3.6	0.92	0.02	0.90	28.1	7.0	none	3.4	0.2	0.1
314457	5.2	0.32	7.3	4.95	0.08	4.87	152.2	12.5	slight	5.2	0.1	0.0
314458	7.1	0.76	17.3	0.52	<0.01	0.52	16.3	22.3	slight	5.0	1.4	1.1
314459	7.0	0.67	15.2	0.27	<0.01	0.27	8.4	18.8	slight	3.6	2.2	1.8
314460	4.9	0.39	8.9	3.96	0.19	3.77	117.8	8.5	slight	-0.4	0.1	0.1
90th Percentile	7.2	0.8	17.3	4.3	0.1	4.2	131.4	22.6		7.4	1.6	1.2
Median	6.2	0.4	9.8	1.3	0.0	1.3	39.8	14.4		4.6	0.3	0.2
10th Percentile	5.2	0.2	5.1	0.3	0.0	0.3	8.4	8.4		2.3	0.1	0.1
Mine Rock												
12 Level	8.3	2.3	52.3	2.12	<0.01	2.12	66.3	75.9	moderate	23.6	1.1	0.8
11 Level	8.3	2.23	50.7	3.88	<0.01	3.88	121.3	67.1	moderate	16.4	0.6	0.4
Surficial Till												
5798	6.3	<0.01	<0.2	0.02	<0.01	0.02	0.6	4.8	none		7.7	

* Difference between total sulphur and sulphate-S; ** Calculated from sulphide-S; *** kg $CaCO_3$/tonne; # excluding sample 5797 with a slight rating

Table 9. Data from the 20-Week Humidity Cell Tests (Rescan, 1991)

Sample	Total-S %	AP kg/t	Sob-NP kg/t	Initial Drainage pH	Final Drainage pH	Final Paste pH	Final Sulphate mg/kg/wk
12 Level	2.4	76	54	7 to 8	7 to 8	9.0	10 to 20
11 Level	3.4	105	48	7 to 8	7 to 8	9.0	10 to 25
10 Level	3.4	105	38	7 to 8	7 to 8	8.7	10 to 25
Pyritic	21.5	672	7.6	6 to 7	5.1	7.3	60

* The ABA data is post-test. For the three levels, pre- and post-data while quite variable was similar. No pre-test ABA data exists for the pyrite sample.

Table 10. Total Elemental Composition of Different Particle Sizes of Selected Waste Rock Samples Collected by Price and Kwong in 1994 (tabulated by John Kwong)

ID	Dump	Size Fract. (mm)	As (ppm)	Bi (ppm)	Cd (ppm)	Co (ppm)	Cu (ppm)	Mo (ppm)	Ni (ppm)	Pb (ppm)	Sb (ppm)	Se (ppm)	Zn (ppm)	Ba (ppm)	Mn ppm
Upper Crustal Limit			2		0.2	100	90	2	200	20	0.2	0.1	100		
JM-1	10	<0.05	50	24	*0.25*	52	3470	6	4	210	8		258	2090	460
JM-1	10	< 2	68	20	*0.25*	73	2050	6	2	86	4		124	1670	500
JM-1	10	2 - 11	52	2	*0.25*	62	1270	4	2	34	*1*		104	1910	500
JM-1	10	11 - 19	38	14	*0.25*	67	2430	6	3	36	*1*		78	2000	500
JM-1	10	>19	54	6	*0.25*	76	796	11	3	44	*1*		62	2100	800
JM-2	10	<0.05	22	20	4	19	3250	3	7	258	2		704	2680	1900
JM-2	10	< 2	22	2	2	18	1590	4	4	92	2		406	2420	1900
JM-2	10	2 - 11	14	4	0.5	12	360	1	4	42	2		228	2360	1900
JM-2	10	11 - 19	12	*1*	*0.25*	19	751	4	6	14	*1*		184	2300	1800
JM-2	10	>19	12	2	*0.25*	9	169	6	6	24	*1*		200	2070	1600
JM-3	10	<0.05	52	*1*	13	43	10000	4	2	268	6		1305	1930	600
JM-3	10	< 2	54	2	4.5	37	8040	4	*0.5*	98	2		560	1920	800
JM-3	10	2 - 11	48	10	2	35	3820	2	*0.5*	40	*1*		372	2140	600
JM-3	10	11 - 19	44	8	9.5	10	2390	4	3	48	2		1320	2270	800
JM-3	10	>19	62	20	*1*	67	10000	6	4	34	*1*		218	1510	600
JM-4	11	<0.05	20	24	3	9	3140	3	6	222	2		722	2300	1200
JM-4	11	< 2	22	14	1.5	11	2610	2	6	74	*1*		362	2330	1700
JM-4	11	2 - 11	12	2	*0.25*	6	865	1	1	20	*1*		124	2310	1300
JM-4	11	11 - 19	8	2	*0.25*	6	1040	6	3	12	*1*		134	2190	1400
JM-4	11	>19	10	*1*	0.5	5	193	7	4	16	*1*		192	2130	1500
JM-5	11	<0.05	48	*1*	2.5	58	10000	1	3	124	4		530	1900	1700
JM-5	11	< 2	74	14	*0.25*	68	7250	1	*0.5*	52	2		280	1950	1300
JM-5	11	2 - 11	40	12	*0.25*	32	2800	*0.5*	1	30	*1*		114	2140	900
JM-5	11	11 - 19	42	6	*0.25*	44	3930	8	7	18	*1*		86	2310	1100
JM-5	11	>19	40	6	5	40	2290	11	9	60	*1*		844	2170	1200
JM-9	11	<0.05	38	Intf*	6	10	10000	4	3	64	*1*	1.2	600	3270	900
JM-9	11	< 2	54	8	6.5	19	7420	3	3	68	*1*	1.4	684	3040	1500
JM-9	11	2 - 11	46	6	14.5	36	4320	2	2	54	*1*	1.4	1660	2690	1100
JM-9	11	11 - 19	46	8	0.5	22	2010	306	2	30	*1*	0.4	114	2840	1200

ID	Dump	Size Fract. (mm)	As (ppm)	Bi (ppm)	Cd (ppm)	Co (ppm)	Cu (ppm)	Mo (ppm)	Ni (ppm)	Pb (ppm)	Sb (ppm)	Se (ppm)	Zn (ppm)	Ba (ppm)	Mn ppm
JM-9	11	>19	44	2	3.5	20	9460	4	3	26	*1*	0.4	404	3130	2100
JM-6	12	<0.05	34	14	2.5	14	556	4	8	234	2		906	2540	1900
JM-6	12	< 2	32	*1*	1	14	264	6	3	74	*1*		374	2440	2300
JM-6	12	2 - 11	28	*1*	*0.25*	10	84	2	3	38	2		220	2560	1900
JM-6	12	11 - 19	38	6	*0.25*	14	86	5	5	16	2		124	2670	1900
JM-6	12	>19	34	2	*0.25*	12	198	7	8	14	2		70	2650	1700
JM-7A	Outc	<0.05	40	44	*0.25*	2	2680	1	1	860	2		334	2540	200
JM-7A	Outc	< 2	62	12	*0.25*	2	1200	7	1	228	2		188	2820	300
JM-7A	Outc	2 - 11	44	14	*0.25*	3	1305	5	1	188	2		142	2910	200
JM-7B	Outc	>19	30	28	1.5	6	7080	12	1	236	2		680	1040	1000
JM-7C	Outc	>19	6	14	*0.25*	*0.5*	2360	7	1	102	2		128	1860	100
JM-7D	Outc	>19	2	80	*0.25*	5	916	7	3	2820	4		52	1790	200
JM-7E	Outc	>19	24	8	*0.25*	11	352	12	10	38	2		104	2320	2000

Method - Nitric-aqua-regia digestion and ICP, except Ba, which was analyzed by XRF or metaborate fusion
Numbers in italics are below the detection limit
Underlined values are above the listed concentration
Outc: Outcrop

Table 11. Results of Microprobe Analysis Conducted by John Kwong on Selected Pyrite Grains from Waste Rock Samples JM-1, JM-3 and JM-6 Collected in 1994 by Price and Kwong

	S	Fe	Bi	Pb	Co	Au	Zn	Cu	Cd	Ni	Ag	Sb	As
Mean	53.100	46.800	0.209	0.070	0.057	0.014	0.009	0.088	0.007	0.006	0.006	0.003	0.000
Std Dev	0.300	0.300	0.040	0.064	0.029	0.022	0.014	0.011	0.011	0.008	0.009	0.008	0.000

Table 12. Results of Elemental Analyses and Leaching Tests on Waste Rock Samples from Johnny Mountain by Price and Kwong in 1994 (tabulated by John Kwong)

Sample #	Rinse pH	Metals	Total Content	Water soluble	CBD leachable	0.1N HCl soluble
JM-1	3.01	As	6.8	-	-	-
		Cd	1.7	0.21	-	0.25
		Cu	2050	106	10.2	126
		Pb	86	-	4.5	-
		Zn	124	19.0	5.0	18.9
JM-2	7.35	As	22	-	-	-
		Cd	4	nd	-	1.02
		Cu	3250	2.2	3.1	183.8
		Pb	258	-	nd	-
		Zn	704	0.29	1.1	59.4
JM-3	6.63	As	54	-	-	-
		Cd	7.5	nd	-	1.23
		Cu	8040	0.06	0.3	509
		Pb	98	-	0.9	-
		Zn	560	0.47	1.1	66.6
JM-4	7.29	As	22	-	-	-
		Cd	2.4	nd	-	0.53
		Cu	2610	nd	0.5	196
		Pb	74	-	nd	-
		Zn	362	nd	0.6	49.0
JM-5	6.64	As	74	-	-	-
		Cd	2.6	nd	-	0.66
		Cu	7250	nd	nd	818
		Pb	52	-	nd	-
		Zn	280	nd	0.2	41.4
JM-6	7.90	As	32	-	-	-
		Cd	3.4	nd	-	0.35
		Cu	264	nd	0.2	24.6
		Pb	74	-	nd	-
		Zn	374	nd	1.0	60.1
JM-7A	2.63	As	62	-	-	-
		Cd	1.7	0.06	-	0.02
		Cu	1200	66.4	2.9	72.4
		Pb	228	-	nd	-
		Zn	188	10.0	14.7	9.40*
JM-9	6.99	As	54	0.004	0.6	0.40*
		Cd	6.5	0.099	-	0.09*
		Cu	7420	0.272	2.5	73.8*
		Pb	68	0.108	nd	2.98*
		Zn	684	0.411	0.1	9.00*

nd: not detected; -: not measured; *: analytical error suspected
total content units are µg/g
water soluble, CBD leachable and 0.1 N HCl soluble units are mg/L.

Table 13. Drainage Chemistry in Pits Excavated in ARD Generating Waste Rock (International Skyline Gold Corp., 1997)

Pit	pH	Al	Fe	Cu	Zn
JMAR	2.5	76.3	458	13.1	3.18
JMAR2	3.5	9.1	255	16.7	2.22

Note: with the exception of pH, values are dissolved mg/L

Table 14. Results of Drainage Monitoring in 2003 (Yeager, 2002a)

Sample ID	12 Level JM-1	11 Level JM-2	10 Level JM-3	Underground JM-4	Tail Pond JM-6	John Ck JM-7	Stone. Ck JM-8
Hardness	122	211	204	246	22.5	64.5	65.4
pH	8.04	8	7.79	8.2	7.76	8	7.99
TSS	-	-	-	-	41	157	-
Alkalinity	64	70	40	139	20	44	44
SO_4	66	160	177	120	3	22	22
T-Ca	42.9	62.9	68.2	71.2	8.18	22	22.6
T-Mg	2.9	11.3	6	13.1	1	3.9	3.9
D-Al	0.001	0.002	0.009	0.014	0.044	0.047	0.048
D-Sb	0.0001	0.0002	0.0001	0.0006	<0.0001	0.0001	0.0001
D-As	<0.0001	<0.0001	<0.0001	<0.0001	0.0001	0.0001	0.0001
D-Ba	0.0494	0.0239	0.0397	0.0213	0.0236	0.0332	0.0333
D-Cd	<0.00005	0.00014	0.00015	0.00025	0.00009	0.00007	0.00007
D-Ca	43.9	65.1	71.2	75.3	7.99	22.2	22.5
D-Cr	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005
D-Co	<0.0001	0.0004	<0.0001	0.0009	<0.0001	<0.0001	<0.0001
D-Cu	0.0007	0.0045	0.0124	0.0414	0.0004	0.0028	0.003
D-Fe	<0.03	<0.03	<0.03	<0.03	<0.03	0.03	0.03
D-Pb	<0.00005	0.00006	0.00006	<0.00005	0.00016	0.00019	0.00019
D-Mg	3	11.9	6.5	14	0.6	2.2	2.2
D-Mn	0.00053	0.356	0.00181	0.417	0.0149	0.0615	0.0611
D-Mo	0.00021	0.00035	<0.00005	0.00067	0.00017	0.0004	0.00041
D-Ni	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005
D-P	<0.3	<0.3	<0.3	<0.3	<0.3	<0.3	<0.3
D-K	3	3	3	2	<2	<2	<2
D-Se	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
D-Si	1.39	1.63	1.74	2.72	0.57	1.12	1.15
D-Ag	<0.00001	<0.00001	<0.00001	<0.00001	<0.00001	<0.00001	<0.00001
D-Na	<2	<2	<2	3	<2	<2	<2
D-Sr	0.174	0.347	0.302	1.23	0.0544	0.182	0.178
D-U	0.00009	0.00023	0.00007	0.00224	0.00008	0.00029	0.00029
D-Zn	0.002	0.025	0.038	0.036	0.003	0.003	0.002

Data in mg/L

Table 15. Chemistry of Drainage in Stonehouse Creek Below the Mine

	pH	TSS mg/L	Hard mg/L	Alk mg/L	SO_4 mg/L	T-Cu µg/L	T-Fe mg/L	T-Zn µg/L	D-Cu µg/L	D-Fe µg/L	D-Zn µg/L	D-Ca mg/L	D-Mg mg/L
Jul10/86	7.1	20	20		2	3	0.64	16	1	60	14	6.6	0.5
Sep12/86	7.5	3	72		6	1	0.17	9	1	40	5	26.2	1.5
Nov05/86	7.2	7	54		7	3	0.24	27	1	30	17	19.7	1.0
Jul29/87	7.6	29	25		3	7	0.98	5	1	30	5	8.6	0.8
Sep04/87	7.4	238	35		7	54	7.65	160	1	30	14	12.4	1.0
Sep25/87	7.6	342	47		11	40	27.10	48	4	62	5	16.2	1.5
Oct31/87	7.9	12	57		15	6	0.96	22	6	30	20	19.1	2.3
Jul10/88	7.1	34	33		1	7	1.90	44	1	1090	29	11.5	1.0
Sep04/88	7.2	93	30		8	15	6.22	80	7	26	5	11.0	0.7
Oct07/88	7.2	103	51		16	80	0.20	5	3	110	5	17.9	1.5
Nov07/88	7.7	304	87		25	760	17.00	290	48	1140	53	30.8	1.1
Dec01/88	7.9	97	84	68	27	93	5.08	76	5	15	14	28.4	3.2
Jun02/89	6.6	321	38	31	16	100	12.90	170	2	90	5	13.4	1.0
Jul01/89	6.7	51	34	28	11	14	2.43	32	1	30	5	11.8	1.0
Aug06/89	6.9	143	42	33	15	280	5.74	72	1	30	5	14.7	1.3
Aug28/89	7.0	327	39	35	12	150	10.40	140	26	30	5	13.6	1.1
Oct03/89	7.2	702	69	48	31	190	4.36	54	2	30	12	23.6	2.4
Jun03/90	7.0	54	41	26	18	31	2.70	42	2	30	8	14.3	1.2
Jul01/90	7.5	35	33	26	11	2	1.69	34	1	30	5	11.4	1.0
Aug05/90	7.5	30	42	32	17	22	2.06	30	2	44	12	14.5	1.4
Aug27/90	7.5	59	46	33	14	7	3.04	45	2	44	12	15.9	1.4
Sep30/90	7.4	17	55	33	18	9	0.99	41	2	37	28	19.7	1.4
Oct30/90	7.6	9	75	49	44	1	0.03	5	1	30	5	26.7	2.0
Jun08/91						5	0.33	24	3	42	0		
Jun14/91						4	0.21	15	3	31	15		
Jun18/91	7.5	9	40	24	12	6	2.15	22	3	30	12	14.1	1.1
Jun26/91	7.6	14	33	22	10	5	0.39	11	4	174	10	11.8	0.8
Jul02/91	7.7	7	32	22	8	3	0.29	16	1	57	16	11.4	0.8
Jul09/91	7.6	5	33	23	9	1	0.17	9	1	30	9	11.8	0.8
Jul17/91	7.9	1	29	21	7	2	3.03	9	1	277	6	10.4	0.7
Jul24/91	7.6	15	27	22	7	5	0.65	9	1	41	5	9.8	0.7
Jul31/91	7.9	2	35	27	9	1	0.23	5	1	30	5	12.6	0.8
Aug06/91	7.9	73	44	30	13	10	1.57	35	2	30	20	16.1	1.0
Aug13/91	7.4	13	36	25	8	2	0.38	5	1	30	5	12.9	0.9
Sep09/91	7.8	3	44	27	10	1	0.22	14	1	30	14	15.9	1.0
Sep30/91	7.8	8	50	29	17	6	0.58	16	3	36	15	18.1	1.2
Jul07/92	6.9	13	24	19	7	3	0.29	9	2	72	6	8.6	0.6
Aug03/92	7.6	7	29	23	9	1	0.37	10	1	30	10	10.2	0.8
Aug31/92	7.7	14	41	30	13	1	0.24	5	1	30	5	14.8	1.1
Sep28/92	7.7	2	51	38	18	2	0.12	5	1	30	5	18.4	1.3
Oct13/92	7.3	7	73	49	32	5	0.27	10	2	49	7	25.4	2.3
Oct15/92	7.4	3	87	55	40	4	0.31	16	2	106	11	30.0	2.9
Oct17/92	7.6	2	93	58	42	1	0.03	8	1	30	7	32.0	3.2

	pH	TSS mg/L	Hard mg/L	Alk mg/L	SO_4 mg/L	T-Cu µg/L	T-Fe mg/L	T-Zn µg/L	D-Cu µg/L	D-Fe µg/L	D-Zn µg/L	D-Ca mg/L	D-Mg mg/L
May05/93	7.6	1	28	22	9	2	0.25	25	1	30	23	10.0	0.7
Jun21/93	7.7	97	57	40	15	10	2.45	20	10	10	20	12.5	0.8
Jul21/93	7.8	20	55	41	16	10	0.10	10	10	10	10	14.0	0.9
Sep22/93	7.3	208	56	34	30	40	4.60	100	10	10	10		
Oct20/93	7.0	26	64	40	26	10	1.60	10	10	10	10		
Nov15/93	7.1	1	87	56	33	10	0.01	40	10	10	10		
Jul11/94	7.6	6	48	36	13	5	0.35	9	1	29	5	17.0	1.3
Jul24/94	7.6	2	42	36	14	1	0.05	8	1	17	7	14.5	1.3
Aug04/94	7.6		37	28	10	6	3.09	60	1	42	4	11.7	1.0
Aug09/94	7.6	1	50	36	10	3	0.95	17	1	22	4	17.4	1.5
Jul16/95	7.9	16	49	44	13	5	0.69	29	2	65	9	17.4	1.5
Oct03/95	7.6	26	77	58	34	16	0.14	10.6	1	4	4.1	26.4	2.4
Aug05/96	7.8	25	25	34	15	7.2	0.42	32	1.3	10	5	15.1	1.6
Sep22/96	6.6	1	73	46	27	2.9	0.06	8	2.6	40	6	25.4	2.2
Oct12/96	8.0	1	59	46	30	3.9	0.11	18	1.4	10	9	20.2	2.0
Jul14/97	7.6	23	40	31	13	4.7	0.75	68	1.7	30	8	14.3	1.1
Aug15/99	7.8			48	27				2	15	5		
Sep3/00	8.0		68	47	25				3.3	110	6.5	24.6	2.4
Oct3/01	8.2		71	49	29				3.9		5	24.2	2.6
Aug25/03	8.0	157	65	44	22				2.8	30	3	22.2	2.2
Minimum	6.6	1	20	19	1	1	0.00	5	1	4	0.0	6.6	0.5
10th Perc	7.0	2	29	22	7	1	0.10	5	1	10	5.0	10.3	0.8
Median	7.6	16	45	34	14	5	0.61	18	2	30	7.0	14.8	1.1
90th Perc	7.9	220	75	50	31	83	5.79	77	9	104	19.4	26.3	2.4
Maximum	8.2	702	93	68	44	760	27.10	290	48	1140	53.0	32.0	3.2

Table 16. Chemistry of Drainage in Johnny Creek below the Mine

	pH lab	TSS mg/L	Hard mg/L	Alk mg/L	SO_4 mg/L	T-Cu µg/L	T-Fe µg/L	T-Zn µg/L	D-Cu µg/L	D-Fe µg/L	D-Zn µg/L	D-Ca mg/L	D-Mg mg/L
Jul01/86	6.7	89	25		1	6	2240	5	1	90	5	7.4	0.6
Sep12/86	7.7	14	30		1	1	310	9	1	50	5	10.9	0.8
Nov05/86	7.3	2	37		1	20	280	95	1	50	64	13.2	0.9
Jul03/87	7.7	69	25		1	5	2060	28	3	90	5	9.0	0.6
Jul29/87	7.7	42	24		2	2	980	16	1	30	5	8.2	0.6
Sep04/87	7.6	28	20		6	2	390	14	1	30	5	7.2	0.5
Sep25/87	7.7	274	28		5	7	30	220	1	30	5	10.0	0.8
Oct31/87	7.9	1	41		7	6	52	4	4	35	5	14.4	1.2
Jun14/88	6.6	2	10	6	1	5	30	40	5	30	9	3.7	0.3
Jul10/88	7.1	10	28	19	1	4	760	36	1	100	18	9.8	0.8
Sep04/88	7.0	65	19	18	16	26	1750	44	7	44	5	7.2	0.4
Oct07/88	7.3	10	27	25	3	1	190	690	1	50	5	9.7	0.6
Nov07/88	7.9	1	45	40	10	3	21	100	3	34	17	16.0	1.2
Dec01/88	7.9	2	49	51	10	1	15	5	1	15	5	17.5	1.3
Apr28/89	7.1			50	9				1	30	5		
May05/89	7.2			36	8				3	30	5		
May13/89	7.3			37	10				10	60	10		
May20/89	7.2			32	7				12	30	23		
May29/89	7.0			31	5	1	90	30	1	30	5		
Jun03/89	6.9			29	4				15	50	5		
Jun09/89	7.4			25	4	2	260	12	2	30	12		
Jun16/89	7.4			25	4	3	390	33	3	40	14		
Jun23/89	7.2			24	3				1	30	5		
Jul01/89	6.6	41	25	21	13	2	950	36	1	30	5	9.1	0.6
Jul08/89	7.0			23	5				6	30	5		
Jul15/89	6.7			21	11				1	30	5		
Jul22/89	7.1			21	7				5	30	5		
Jul30/89	7.0			22	24	0			2	30	5		
Aug06/89	6.7	43	24	20	11	4	760	28	1	40	7	8.8	0.0
Aug12/89	7.3			18	10				2	70	5		
Aug19/89	6.7			21	3				1	30	5		
Aug28/89	6.9	28	21	21	21	1	650	18	1	30	7	7.7	0.5
Sep02/89	6.8			20	27				2	30	8		
Sep09/89	7.1			17	4				1	30	5		
Sep16/89	7.5			100	33				3	30	5		
Oct03/89	6.9	9	30	29	7	5	440	15	4	30	5	10.5	0.8
Oct11/89	7.3			33	6				1	30	5		
Apr28/90	7.5	61	68	58	18	1	180	10	1	30	9	23.5	2.2
May05/90	7.3	25	57	54	14	7	1790	46	1	30	5	20.1	1.6
May13/90	6.9	56	49	48	10	8	2690	44	1	30	15	17.2	1.4
May20/90	6.8	19	36	35	8	3	680	15	1	30	5	12.8	0.9
Jun03/90	7.0	23	33	25	14	2	700	6	1	40	5	11.8	0.8
Jul01/90	7.1	31	24	21	8	1	820	34	1	30	5	8.5	0.6
Jul08/90	7.5												

	pH lab	TSS mg/L	Hard mg/L	Alk mg/L	SO_4 mg/L	T-Cu µg/L	T-Fe µg/L	T-Zn µg/L	D-Cu µg/L	D-Fe µg/L	D-Zn µg/L	D-Ca mg/L	D-Mg mg/L
Jul11/90	7.7	76	23	20	8	3	1260	28	1	50	5	8.2	0.5
Jul22/90	7.2	20	21	16	7	2	730	42	1	30	5	7.8	0.5
Jul29/90	7.1												
Aug05/90	7.3	25	21	18	7	3	1230	28	2	57	5	7.5	0.4
Aug27/90	7.2	19	22	20	3	1	470	17	1	64	17	7.9	0.5
Sep30/90	6.9	12	33	22	9	2	521	18	1	30	5	11.9	0.8
Oct30/90	7.7	6	45	40	9	1	105	5	1	39	5	16.0	1.2
Jun18/91	7.4	7	24	18	3	1	347	13	1	30	9	8.3	0.7
Jul09/91	7.8	5	25	21	3	1	220	5	1	36	5	9.0	0.7
Jul31/91	7.7	9	25	24	5	1	250	5	1	30	5	8.9	0.7
Sep09/91	7.6	5	32	22	4	1	152	6	1	30	6	11.5	0.9
Sep30/91	7.9	6	40	27	8	4	560	27	1	51	5	14.2	1.0
Aug03/92	7.6	5	19	19	4	1	182	14	1	63	5	6.6	0.5
Aug31/92	7.6	9	23	20	5	1	253	7	1	30	7	8.4	0.6
Sep28/92	7.6	1	33	30	6	1	90	5	1	30	5	11.5	0.9
Oct15/92	7.7	1	43	37	8	1	30	5	1	30	5	15.3	1.1
Jun21/93	7.5	64	27	24	2	10	350	10	10	10		5.0	0.3
Jul21/93	7.5	66	25	24	2	10	950	20	10	10		9.2	0.6
Sep22/93	7.4	147	28	28	7	10	2600	110	10	10	10		
Oct20/93	7.1	4	68	47	27	10	350	10	10	10	10		
Nov15/93	7.2	2	62	46	21	10	10	10	10	10	20		
Jul11/94	7.5	13	27	25	3	1	487	11	1	20	6	9.4	0.8
Jul24/94	7.3	4	26	24	8	2	585	5	1	17	2	9.2	0.7
Aug04/94	8.1		35	23	1	7	4130	80	1	55	3	6.6	0.4
Aug09/94	7.4	82	39	26	5	4	3400	65	1	16	4	13.7	1.2
Jul16/95	7.0	57	24	29	4	10	637	20	2	37	9	8.6	0.6
Oct03/95	7.4	8	23	41	6	6	3	4.8	3	3	1	13.0	0.8
Aug04/96	7.6	33	59	20	3	15	600	34	1	10	1	6.7	0.4
Sep22/96	7.1	3	38	33	7	0.5	40	3	1	60	1	13.6	1.0
Oct12/96	7.7	2	31	31	6	1.4	30	4	1	10	3	10.8	0.9
Jul14/97	7.4	13	19	19	2	1.9	410	16	1	10	4	7.0	0.4
Aug15/99	7.4	<5		34	9	10	171	8	1	7	3		
Sep3/00	7.7		30	25	6				1	50	8	9.6	0.8
Oct3/01	7.9		41	37	7				0		1	14.7	1.1
Aug25/03	7.8	41	23	20	3				0	30	3	8.0	0.6
Minimum	6.6	1	10	6	1	0	3	3	0	3	1	3.7	0.0
10th Perc.	6.8	2	21	19	2	1	30	5	1	10	3	7.0	0.4
Median	7.3	14	28	25	6	2	400	16	1	30	5	9.3	0.7
90th Perc.	7.7	68	49	46	15	10	1871	70	8	59	13	15.8	1.2
Maximum	8.1	274	68	100	33	26	4130	690	15	100	64	23.5	2.2

Table 17. Chemistry of Discharge from Underground Mine Workings Measured at 10 Level Portal

	pH lab	TSS mg/L	Hard mg/L	Alk mg/L	SO_4 mg/L	T-Cu µg/L	T-Fe µg/L	T-Zn µg/L	D-Cu µg/L	D-Fe µg/L	D-Zn µg/L	D-Ca mg/L	D-Mg mg/L
Sep30/90	7.4	21	105	60	50	18	818	42	10	30	32	35	4
Oct30/90	6.2	2	276	131	200	11	221	380	3	45	338	87	14
Mar05/91	7.7	13	241	129	162	18	80	188	8	81	117	70	15
Mar26/91	7.8	3	262	124	150	6	484	209	5	90	161	81	14
Apr15/91	7.8	4	296	129	156	5	113	356	5	30	187	91	17
May08/91	8.2	5	253	116	131	5	169	134	5	44	131	78	14
May27/91	7.4	1	298	146	162	10	111	200	7	30	190	91	17
Jun08/91						12	332	134	5	101	132		
Jun14/91						12	231	152	6	79	143		
Jun18/91	8.1	1	243	114	123	10	218	142	5	30	134	75	13
Jun26/91	8.1	1	250	129	128	15	211	150	12	111	95	77	14
Jul02/91	8.1	1	250	130	128	10	160	159	4	89	143	77	14
Jul09/91	8.0	2	264	133	132	9	119	162	7	48	162	81	15
Jul17/91	8.0	1	219	118	111	8	122	139	5	30	107	68	12
Jul24/91	8.1	1	81	101	63	10	36	36	6	36	36	46	8
Jul31/91	8.1	3	225	131	124	5	95	131	5	30	114	70	0
Aug06/91	8.1	1	270	132	120	9	86	170	6	56	170	84	15
Aug13/91	8.0	1	253	136	108	7	95	125	5	47	125	78	14
Sep09/91	8.1	1	238	123	71	8	99	116	5	30	92	74	13
Sep30/91	8.0	1	242	131	106	9	106	95	8	30	95	75	13
Nov11/91	8.0	1	206	131	98	10	30	81	10	30	80	61	13
Nov15/91						4	30	13	4	30	13		
Nov23/91						4	35	10	4	30	13		
Dec03/91						4	154	16	4	30	16		
Dec15/91	8.2	1	178	120	109	7	30	75	6	30	73	54	11
Jan20/92	7.5	4	322	101	124	5	43	16	5	30	16	121	5
Feb09/92	7.9	1	232	125	119	10	97	86	7	30	80	73	12
Mar01/92	7.7	14	158	73	83	10	769	28	3	30	18	57	4
Mar23/92	8.1	1	235	130	116	12	71	88	12	62	87	73	13
Apr15/92	7.7	1	203	131	115	13	80	83	7	30	83	62	11
May06/92	8.1	1	141	125	116	3	52	78	3	39	38	44	7
May26/92	7.4	7	148	90	98	45	271	96	16	30	79	48	7
Jun15/92	7.9	3	208	129	108	7	111	92	7	42	81	64	12
Jul07/92	7.9	1	211	131	101	7	386	87	5	30	75	66	11
Jul22/92	8.1	3	196	115	103	46	1700	119	5	30	83	61	11
Aug03/92	7.9	1	181	117	102	26	724	91	3	30	75	56	10
Aug13/92	7.9	1	212	121	97	8	60	78	4	30	76	69	10
Aug18/92	7.6	1	216	124	97	5	67	86	4	30	83	69	11
Aug31/92	8.0	1	223	137	107	8	66	81	6	30	80	69	12
Sep15/92	8.0	1	148	109	54	2	30	15	1	30	15	47	8
Sep23/92	7.8	1	209	123	93	5	77	69	3	30	69	66	11
Sep28/92	8.0	1	217	133	105	8	62	64	5	30	64	68	11
Oct17/92	8.1	1	218	130	101	12	65	67	8	30	66	68	11
May26/93	8.0	1	206	122	115	12	80	105	5	30	79	63	12

	pH lab	TSS mg/L	Hard mg/L	Alk mg/L	SO_4 mg/L	T-Cu µg/L	T-Fe µg/L	T-Zn µg/L	D-Cu µg/L	D-Fe µg/L	D-Zn µg/L	D-Ca mg/L	D-Mg mg/L
Sep22/93	7.2	411	235	128	119	1440	11000	400	10	10	60		
Oct01/93			284			5540	150000	1900	12	31	71	89	15
Oct20/93	7.3	1520	292	160	135	410	35000	540	10	10	20		
Nov15/93	7.2	19100	253	136	173	1750	68700	6250	10	10	10		
Jul11/94	7.9	2	221	117	77	10	81	94	7	20	89	70	11
Jul24/94	8.0	1	249	137	76	5	29	230	4	10	98	79	13
Aug04/94	8.1		257			14	160	100	9	7	90	81	13
Aug09/94	8.0	1	258	155	97	8	159	91	5	7	85	83	13
Jul16/95	8.6	4	241	178	91	63	216	77	18	48	68	75	13
Oct03/95	8.0	1	268	180	104	27	3	67	18	3	57	80	13
Aug05/96	8.2	10		122	115	68	630	88	23	20	54	47	9
Sep22/96	8.0	1	227	138	108	48	130	82	45	70	81	73	11
Oct12/96	8.2	1	204	135	113	63	270	76	8	20	58	63	12
Jul14/97	8.1	1	203	135	149	110	270	93	64	140	69	65	10
Aug15/99	8.1	<5		126	115	174	381	82	72	10	64		
Sep3/00	8.2			136	106				47	5	53	78	13
Oct3/01	8.2		205	133	103				33		35	63	12
Aug25/03	8.2		246	139	120				41	< 30	36	75	14
Minimum	6.2	1	81	60	50	2	3	10	1	3	10	35	0
10th Perc.	7.4	1	164	111	79	5	34	34	4	10	18	49	8
Median	8.0	1	230	129	109	10	113	92	6	30	80	70	12
90th Perc.	8.2	13	274	139	150	76	779	255	22	79	143	83	15
Maximum	8.6	19100	322	180	200	5540	150000	6250	72	140	338	121	17

Table 18. Underground Flow Measurements and Estimates made on August 24, 2003

Sample No.	Source	Mine Area Drained	Estimated Flow
U1	Decline	9 Level	2.00 L/sec
U2	Discovery	Above 10, 11 Levels	0.15 L/sec
U3	16	Above 10, 11 Levels	1.85 L/sec
U4	All 11, 12	All 11, 12 Waters	2.25 L/sec
U5	16	Above 11, 12 Levels	1.68 L/sec
U6	Pickaxe	Above 11, 12 Levels	0.57 L/sec
JM4		All Mine Waters	6.70 L/sec

Note: The JM4 flow estimate is a measured figure. All other flows are reasonably accurate. Semi-quantitative estimates. U1 + U2 + U3 + U4 should equal JM4. However, JM4 is 0.45 l/sec greater than the sum of U1 to U4. This is attributable to estimation errors plus minor unrecorded flows from other sources on the 10 Level.

Table 19. Chemistry of Drainage in the Underground, August 24, 2003

Sample ID	U-1	U-2	U-3	U-4	U-5	U-6	JM-4
Hardness	367	261	167	182	210	199	246
pH	8.1	2.95	8.03	8.13	8.22	8.2	8.2
Alkalinity	225	<1	88	114	144	127	139
SO$_4$	180	430	86	80	83	87	120
T-Ca	114	74.2	48.8	52.1	60.1	55.1	71.2
T-Mg	20.4	13.7	9.1	9.5	10.2	11.8	13.1
D-Al	<0.002	1.57	0.014	0.018	0.011	0.003	0.014
D-Sb	0.0011	<0.0005	0.0003	0.0003	0.0004	0.0004	0.0006
D-As	0.0003	0.0053	<0.0001	0.0001	0.0002	0.0002	<0.0001
D-Ba	0.025	0.0142	0.0194	0.0199	0.0192	0.0188	0.0213
D-Cd	0.0002	0.007	0.00061	0.00038	0.0002	0.00048	0.00025
D-Ca	113	79.9	51	56.1	65	58.6	75.3
D-Co	0.0015	0.0304	<0.0001	0.0012	0.0003	0.0002	0.0009
D-Cu	<0.0002	9.5	0.0148	0.188	0.0063	0.0202	0.0414
D-Fe	<0.03	62.6	<0.03	<0.03	<0.03	<0.03	<0.03
D-Pb	<0.0001	0.0099	<0.00005	<0.00005	0.00006	<0.00005	<0.00005
D-Mg	20.6	14.9	9.5	10.2	11.6	12.7	14
D-Mn	1.44	3	0.0365	0.251	0.194	0.27	0.417
D-Mo	0.0008	<0.0003	0.00049	0.00063	0.00044	0.00119	0.00067
D-P	<0.3	<0.3	<0.3	<0.3	<0.3	<0.3	<0.3
D-K	2	3	2	<2	<2	<2	2
D-Se	<0.002	<0.005	<0.001	<0.001	<0.001	<0.001	<0.001
D-Si	4.47	3.05	1.48	2.22	1.76	3.01	2.72
D-Na	6	<2	<2	<2	<2	4	3
D-Sr	2.72	0.387	0.444	0.542	0.583	0.826	1.23
D-U	0.0045	0.0009	0.00142	0.00096	0.00153	0.00095	0.00224
D-Zn	0.089	0.963	0.088	0.052	0.032	0.077	0.036

Data in mg/L

Table 20. Chemistry of Drainage Discharge from the Tailings Impoundment

	pH lab	TSS mg/L	Hard mg/L	Alk mg/L	SO_4 mg/L	T-Cu µg/L	T-Fe µg/L	T-Zn µg/L	D-Cu µg/L	D-Fe µg/L	D-Zn µg/L	D-Ca mg/L	D-Mg mg/L
Jan16/89	6.9			85	22				4	15	7		
Jul01/89	6.3	8	106	15	223	1	530	7	1	30	5	39.8	1.5
Aug06/89	6.9	23	137	27	240	20	600	10	4	30	6	51.6	1.9
Sep16/89	6.9			27	5								
Oct03/89	7.3	3	127	107	42	1	30	7	1	30	7	44.6	3.8
Oct11/89	7.6			93	39								
May26/90	7.4	4	75	55	24	2	280	11	2	30	5	26.6	2.0
Jun03/90	*4.4*	2	132	2	173	1	50	6	1	30	5	49.7	1.9
Jun10/90	*4.1*	11	113	1	163	5	820	21	1	30	6	42.7	1.5
Jun16/90	*5.7*	4	64	4	95	3	540	8	1	40	5	23.8	1.0
Jun23/90	*5.1*	4	67	3	101	2	347	13	1	30	6	25.0	1.1
Jul01/90	*5.1*	9	68	1	98	2	440	14	2	30	7	25.4	1.0
Jul08/90	*5.0*	10	78	1	111	3	580	13	1	30	6	29.3	1.1
Jul11/90	*5.0*	5	87	2	130	1	340	16	1	30	13	32.8	1.3
Jul16/90													
Jul22/90	*4.5*	9	120	1	176	49	180	5	28	30	5	45.5	1.4
Aug05/90	*4.2*	2	183	2	274	6	225	7	6	30	7	69.8	2.1
Aug12/90	*5.7*	3	208	26	270	4	256	5	4	30	5	79.3	2.4
Sep05/90	*4.2*	29	246	1	283	4	839	9	2	30	8	94.1	2.5
Sep08/90	6.7	11	252	15	304	2	303	6	1	34	6	95.8	2.9
Sep16/90	6.9	5	254	31	272	1	450	5	1	30	5	94.3	4.4
Sep22/90	6.8	8	272	41	332	1	241	5	1	30	5	100.0	5.3
Sep29/90													
May19/91	7.8	2	239	85	187	3	30	78	2	30	78	79.9	9.2
May27/91	7.1	1	191	69	161	1	30	39	1	30	39	65.5	6.6
Jun08/91						1	52	12	1	43	12		
Jun18/91	7.4	9	42	23	16	4	611	28	1	40	9	15.0	1.1
Jun26/91	7.3	1	38	23	16	1	39	19	1	30	9	13.5	1.0
Jul02/91	7.5	1	40	22	17	1	50	15	1	36	15	13.9	1.2
Jul09/91	7.5	1	48	27	22	1	79	18	1	30	18	16.4	1.7
Jul17/91	7.7	2	59	30	27	7	39	18	1	30	17	19.6	2.4
Jul24/91	7.6	1	81	43	41	4	250	18	1	33	18	26.5	3.4
Jul31/91	7.8	1	98	52	65	1	145	24	1	30	24	32.4	4.0
Aug06/91	7.9	1	129	58	73	2	121	36	2	43	35	42.5	5.4
Aug13/91	7.8	3	137	62	72	1	61	15	1	30	11	45.0	5.8
Aug20/91	7.6	2	137	58	79	1	94	24	1	30	24	44.4	6.2
Aug28/91	8.1	1	143	65	68	1	295	34	1	30	20	46.0	6.7
Sep04/91	8.1	2	137	66	68	1	150	23	1	30	18	44.2	6.3
Sep09/91	7.9	1	134	63	48	1	30	16	1	30	12	43.9	5.7
Sep17/91	8.1	1	152	70	79	2	31	28	1	31	26	49.2	6.8
Sep25/91	7.8	3	154	69	78	9	744	25	3	70	19	46.6	8.8
Sep30/91	7.8	7	113	55	44	120	2210	35	2	61	15	36.4	5.2
Oct09/91	8.0	2	130	69	65	1	209	25	1	44	5	42.0	6.0
Oct17/91	7.9	5	126	70	49	2	395	23	1	30	7	40.8	5.8
Oct22/91	7.9	1	91	79	38	6	104	11	1	30	9	29.6	4.1
Nov15/91						1	30	25	1	30	25		
Nov23/91						1	30	14	1	30	14		
Dec03/91						1	30	13	1	30	13		

	pH lab	TSS mg/L	Hard mg/L	Alk mg/L	SO_4 mg/L	T-Cu µg/L	T-Fe µg/L	T-Zn µg/L	D-Cu µg/L	D-Fe µg/L	D-Zn µg/L	D-Ca mg/L	D-Mg mg/L
Jan20/92	7.5	1	156	90	82	2	125	25	1	30	25	50.0	7.5
May06/92	7.8	1	144	94	89	3	91	75	3	31	49	45.3	7.4
May26/92	7.5	3	104	64	62	2	30	20	1	30	20	33.7	4.9
Jun15/92	7.9	3	47	33	20	3	50	24	1	30	20	16.0	1.7
Jul07/92	7.2	1	42	27	18	1	30	8	1	30	5	14.1	1.6
Jul22/92	7.7	3	74	46	32	2	540	21	2	30	20	24.4	3.2
Aug03/92	7.7	3	72	47	32	1	130	15	1	30	14	23.5	3.1
Aug13/92	7.9	3	90	51	38	3	44	6	1	30	6	30.0	3.7
Aug18/92	7.1	3	88	54	39	3	69	9	1	30	9	29.1	3.8
Aug25/92	8.1	1	108	59	40	2	39	7	2	30	7	36.5	4.2
Aug31/92	7.7	5	105	64	46	1	71	5	1	30	5	34.1	4.7
Sep07/92	9.4	172	100	54	46	7	2500	48	5	151	14	32.5	4.6
Sep15/92	7.8	2	92	59	44	1	158	8	1	30	7	29.2	4.6
Sep23/92	7.5	13	95	57	44	3	717	14	2	30	8	30.8	4.4
Sep28/92	7.6	2	98	67	52	4	316	7	2	30	6	31.6	4.5
Oct07/92	7.3	5	113	65	44	3	539	20	1	60	12	37.2	4.9
Oct15/92	7.7	2	111	72	49	4	162	14	1	63	13	36.9	4.5
Sep15/93	7.6	17	117	64	54	10	1340	20	10	20	10		
Sep22/93	7.3	174	136	62	71	40	5500	120	10	10	10		
Sep29/93	7.4	61	158	90	63	50	2600	80	10	100	10		
Oct01/93			152			4	360	10	1	11	9	53.2	3.9
Oct06/93	7.3	9	149			10	400	10	10	10	10		
Oct13/93	7.3	1	149	73	87	30	460	780	10	10	10		
Oct20/93	7.2	354	160	82	92	110	7600	140	10	10	10		
Oct27/93	7.2	15	136	68	84	10	650	10	10	10	10		
Nov03/93	7.9	4	143	70	93	10	250	10	10	10	20		
Nov10/93	7.2	4	141	74	86	10	10	10	10	10	10		
Nov15/93	7.0	2	145	75	99	10	10	10	10	10	10		
Jul11/94	7.2	7	35	26	10	3	161	17	1	23	14	12.2	1.0
Jul24/94	7.4	1	58	32	38	4	333	5	1	45	2	19.5	2.4
Aug04/94	7.5		49	25		2	560	10					
Aug09/94	7.4	1	70	37	37	2	399	15	1	41	4	23.6	2.6
Jul16/95	8.5	3	120	96	51	9	116	17	3	13	13	42.6	3.2
Oct03/95	7.2	3	54	151	28	5	63	6	5	26	7.4	18.8	2.4
Aug04/96	7.6	2	46	12	15	1	100	9	0.5	10	8	8.5	1.1
Sep22/96	6.5	1	26	16	17	1	60	14	1.2	30	14	8.8	1.1
Oct12/96	7.3	3	28	15	21	3	60	13	0.5	10	9	9.1	1.4
Jul14/97	7.0	1	18	10	11	3	30	25	1.9	20	15	5.9	0.7
Aug15/99	6.8	<5		12	13	11	46	18	1	10	18		
Sep3/00	7.6		26	10	13				0.5	< 20	8	8.2	0.9
Oct3/01	7.4		22	8	15				0.5	0	7	7.1	0.9
Minimum	4.1	1	18	1	5	1	10	5	0.5	0.0	2.0	5.9	0.7
10th Perc	5.2	1	42	3	16	1	30	6	1.0	10.0	5.0	13.1	1.1
Median	7.4	3	111	54	51	3	162	15	1.0	30.0	10.0	33.3	3.2
90th Perc	7.9	14	165	85	191	10	741	36	10.0	43.0	22.8	66.8	6.4
Maximum	9.4	354	272	151	332	120	7600	780	28.0	151.0	78.0	100.0	9.2

Note: low pH values in 1990 are attributed to addition of acid after samples collected

Table 21. Chemistry of Drainage in Sump below the 10 Level Dump (JM-3)

	pH lab	TSS mg/L	Hard mg/L	Alk mg/L	SO_4 mg/L	T-Cu µg/L	T-Fe µg/L	T-Zn µg/L	D-Cu µg/L	D-Fe µg/L	D-Zn µg/L	D-Ca mg/L	D-Mg mg/L
Aug13/92	7.7	2	45	27	19	1	997	21	1	30	21	17	0.8
Sep29/92	7.8	1	124	60	73	7	238	47	6	42	45	45	3.0
Jun21/93	8.0	44	502	92	448	10	1350	20	10	10	20	125	10.2
Jul21/93	8.0	1	512	92	423	10	10	10	10	10	10	125	10.2
Jul24/94	7.8	1	57	51	14	3	30	9	2	22	2	21	1.4
Aug09/94	7.8	1	69	62	15	1	106	18	1	8	4	25	1.6
Jul16/95	7.6	2	84	71	31	1	27	48	2	10	57	31	1.5
Oct03/95	7.3	1	115	78	46	2	3	69	2	3	66	40	2.2
Jul14/97		1	238	38	258	43	170	155	43	30	145	86	5.5
Sep3/00	7.1		166	64	101				21	5	33	60	4.5
Oct3/01	8.1		127	41	99				16		57	44	3.9
Aug25/03	7.8		204	40	177				12	< 30	38	71	6.5
Minimum	7.1	1	45	27	14	1	3	9	1	3	2	17	0.8
10th Perc.	7.3	1	58	38	15	1	9	10	1	5	5	21	1.4
Median	7.8	1	126	61	86	3	106	21	8	10	36	45	3.4
90th Perc.	8.0	10	476	91	407	17	1068	86	21	31	65	121	9.8
Maximum	8.1	44	512	92	448	43	1350	155	43	42	145	125	10.2

Table 22. Chemistry of Drainage in Sump below the 11 Level Dump (JM-2)

	pH lab	Hard mg/L	Alk mg/L	SO_4 mg/L	T-Cu µg/L	T-Fe µg/L	T-Zn µg/L	D-Cu µg/L	D-Fe µg/L	D-Zn µg/L	D-Ca mg/L	D-Mg mg/L
Jul31/91	7.6	146	74	80	5	30	57	4	30	57	50.5	4.8
Oct17/91	7.5	110	67	35	3	592	11	1	30	11	40.9	2.0
Aug13/92	7.4	125	66	50	4	30	36	3	30	36	43.6	4.0
Sep29/92	7.4	177	75	124	5	30	35	5	30	35	60.4	6.4
Jun21/93	7.9	180	79	103	10	1450	10	10	10	10	42.5	5.5
Jul21/93	7.8	174	79	104	10	730	20	10	10	20	43.7	5.5
Jul11/94	8.1	172	124	58	31	225	35	17	20	34	54.9	8.4
Jul25/94	7.6	131	70	50	2	12	30	1	10	9	44.4	4.9
Aug05/94		124			2	50	30	3	6	16	41.9	4.7
Aug09/94	7.5	141	84	61	1	77	26	1	8	17	47.5	5.4
Jul16/95	7.9	133	68	90	4	63	6	7	22	33	44.6	5.3
Oct03/95	7.5	229	78	45	8	52	7	3	7	4.2	73.8	10.3
Aug05/96	7.7	119	56	111	4	10	21	3.5	10	21	38.6	5.6
Sep22/96	6.9	250	65	187	6	80	32	6.1	80	32	80.0	12.3
Oct12/96	7.6	195	66	195	10	10	60	8.9	10	58	60.6	10.6
Jul14/97	7.3	153	58	101	8	20	99	0.5	20	8	28.4	1.7
Aug15/99	7.3		49	121	18	28	20	2	6	17		
Sep3/00	7.2	174	64	110				8	5	31.5	56.8	8.7
Oct3/01	8.0	190	72	141				8.5		56	58.1	10.9
Aug25/03	8.0	211	70	160				4.5	< 30	25	65.1	11.9
Minimum	6.9	110	49	35	1	10	6	1	5	4	28.4	1.7
10th Percentile	7.3	123	58	49	2	11	9	1	6	9	40.4	3.6
Median	7.6	172	70	103	5	50	30	4	10	23	47.5	5.5
90th Percentile	8.0	215	80	165	13	647	58	10	30	56	66.8	11.1
Maximum	8.1	250	124	195	31	1450	99	17	80	58	80.0	12.3

Table 23. Chemistry of Drainage in Sump below the 12 Level Dump (JM-1)

	pH lab	TSS mg/L	Hard mg/L	Alk mg/L	SO_4 mg/L	T-Cu µg/L	T-Fe µg/L	T-Zn µg/L	D-Cu µg/L	D-Fe µg/L	D-Zn µg/L	D-Ca µg/L	D-Mg µg/L
Jul31/91	7.7	3	34	30	10	3	60	5	1	30	5	12	1.1
Aug13/92	7.1	1	111	65	34	1	30	5	1	30	5	42	1.7
Sep29/92	7.5	3	122	79	55	1	30	5	1	30	5	45	2.3
Jun21/93	7.7	97	136	78	49	10	1900	20	10	10	0	35	1.8
Jul21/93	7.8	1	130	78	53	10	10	10	10	10	10	36	1.8
Jul11/94	7.7	7	116	83	41	1	2	5	1	2	2	43	2.2
Jul25/94	7.2	1	104	78	30	2	85	5	1	10	2	38	2.0
Aug05/94	7.7		95			2	50	40	0	0	0		
Aug09/94	7.4	1	107	87	32	1	49	8	1	4	4	39	2.4
Jul16/95	8.1	0	96	60	57	5	61	11	4	2	13	34	2.4
Oct03/95	7.7	5	307	105	545	5	9	37	5	9	29	106	7.2
Aug05/96	7.9	1	71	56	31	1	60	3	1	10	2	26	1.5
Sep22/96	7.0	2	127	72	60	1	60	15	1	60	6	47	2.5
Oct12/96	8.0	1	100	72	56	1	10	4	1	10	3	36	2.4
Jul14/97	7.3	1	78	49	35	2	30	9	1	20	8	28	1.7
Aug15/99	7.3	<5		114	62	1	4	3	1	2	3		
Sep3/00	7.3		104	68	42				3	5	3	40	2.4
Oct3/01	8.3		113	74	57				0		2	41	2.7
Aug25/03	8.0		122	64	66				1	< 30	2	44	3.0
Minimum	7.0	0	34	30	10	1	2	3	0	0	0	12	1.1
10th Perc.	7.1	1	76	54	31	1	7	4	0	2	2	27	1.6
Median	7.7	1	109	73	51	2	40	7	1	10	3	39	2.3
90th Perc.	8.1	6	132	92	63	8	73	29	6	30	11	46	2.8
Maximum	8.3	97	307	114	545	10	1900	40	10	60	29	106	7.2

Table 24. Comparison of Drainage Chemistry from Different Site Components for 1997 to 2003

	pH	Alk (mg/L)	SO_4 (mg/L)	D-Cu (µg/L)
Stonehouse Creek	7.6-8.2	31-49	13-29	2-4
Johnny Creek	7.4-7.9	19-37	2-9	0-1
Underground Mine	8.1-8.2	126-139	103-149	33-72
Tailings Impoundment	6.8-7.6	8-12	11-15	0.5-1.9
10 Level Waste Dump	7.1-8.1	38-64	99-258	12-43
11 Level Waste Dump	7.2-8.0	58-72	101-160	0.5-8.5
12 Level Waste Dump	7.3-8.3	49-114	35-66	nd-3

Figure 1. The Location of the Johnny Mountain Mine

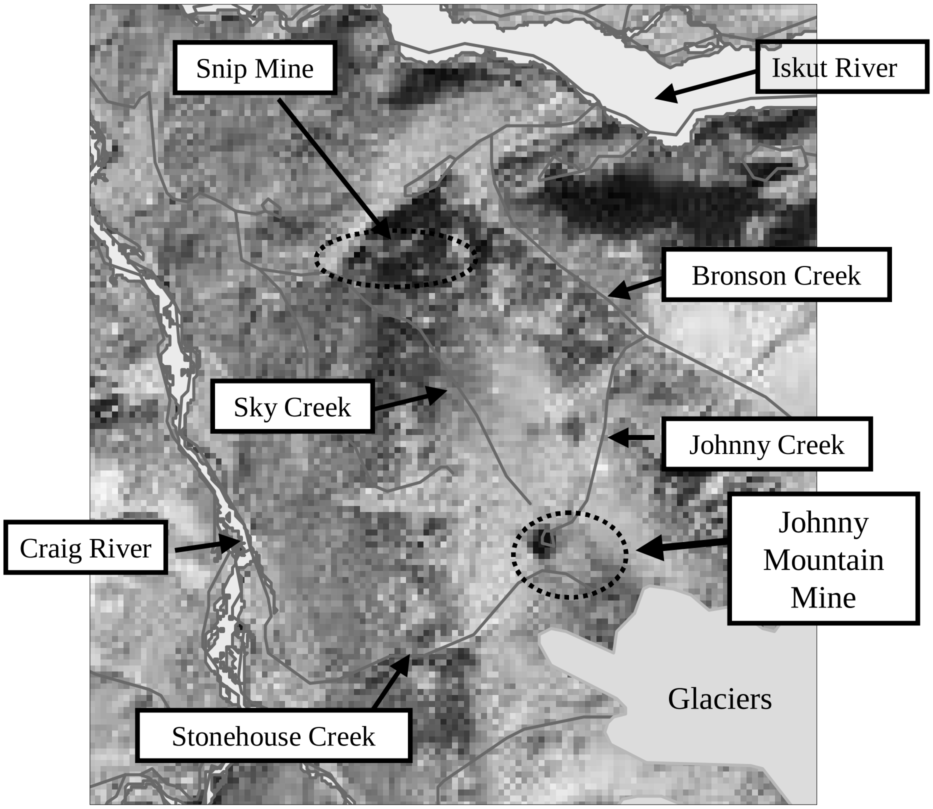

Figure 2. The Area around the Johnny Mountain Mine

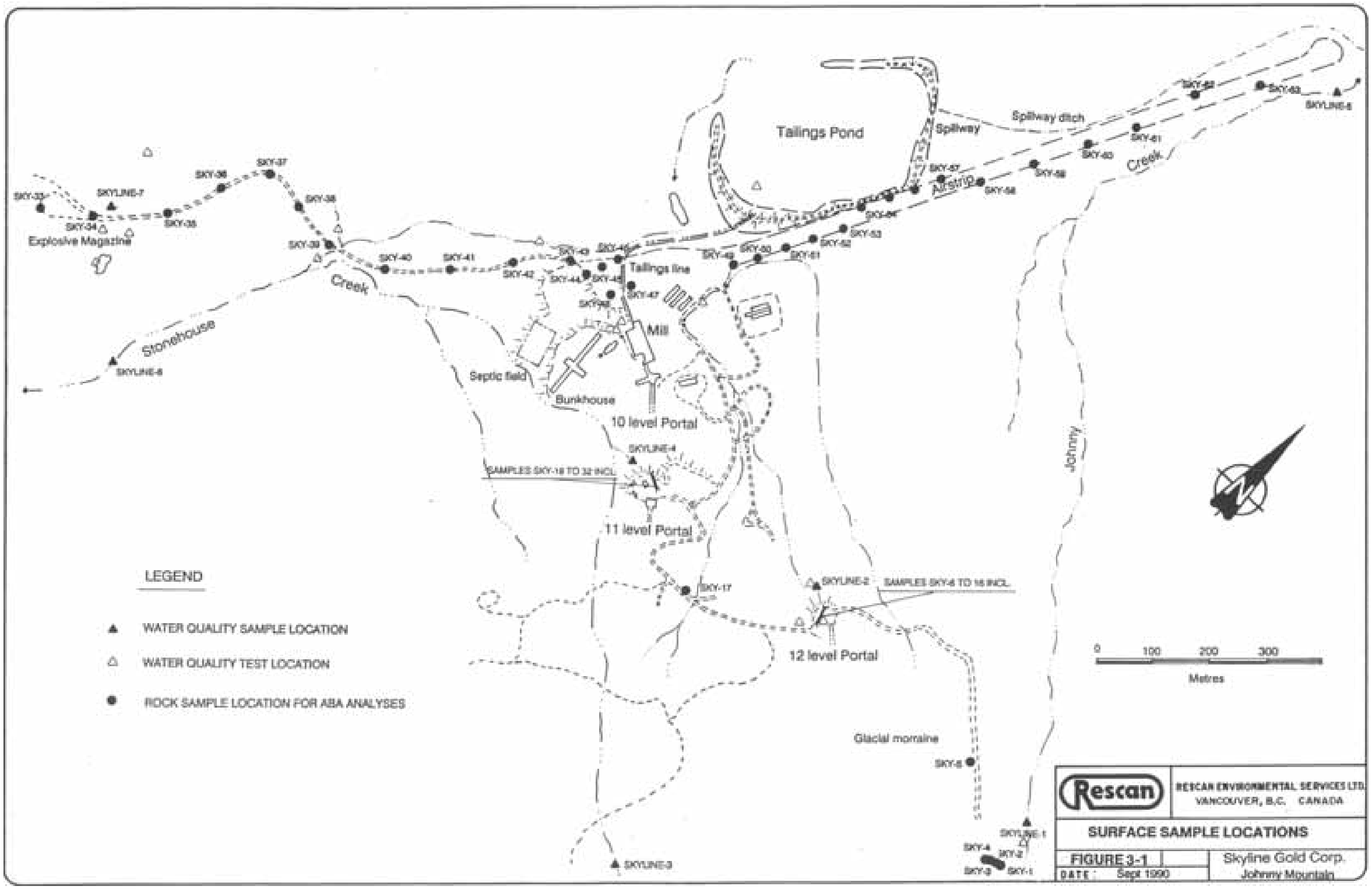

Figure 3. A Map of the Johnny Mountain Mine Site showing the Waste Rock Sampling and Drainage Monitoring Locations (from Rescan, 1991)

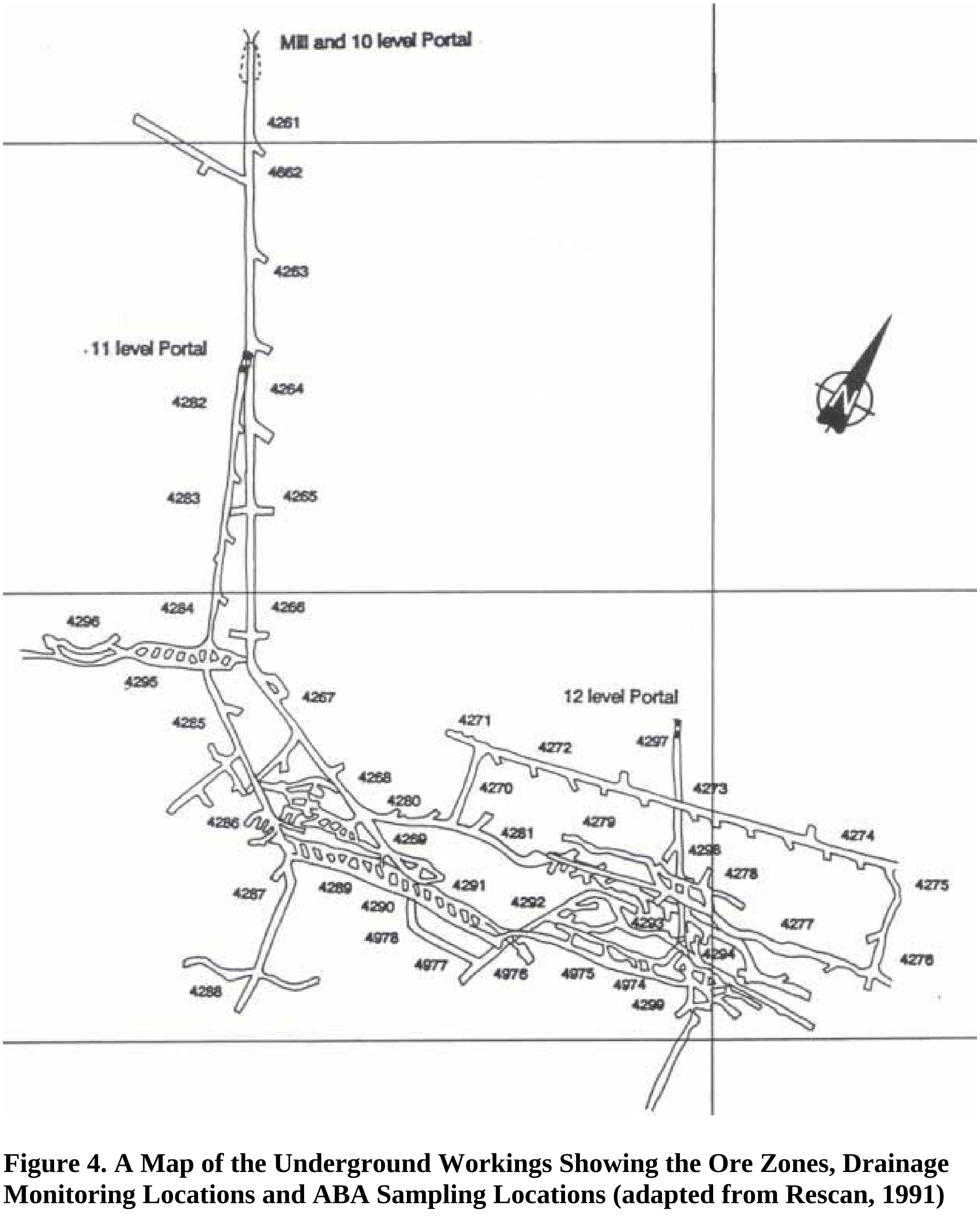

Figure 4. A Map of the Underground Workings Showing the Ore Zones, Drainage Monitoring Locations and ABA Sampling Locations (adapted from Rescan, 1991)

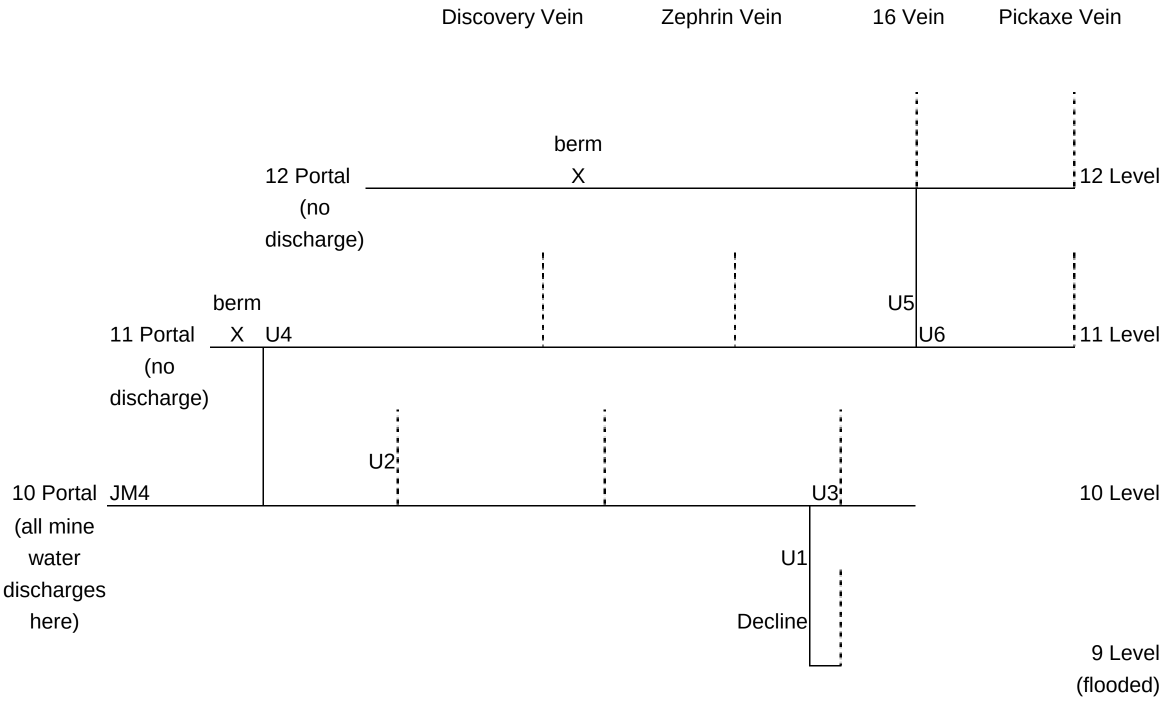

Figure 5. Schematic Cross Section of the Mine Workings Showing Locations of Drainage and Flow Monitoring and Proposed Berms to Divert Drainage to 10 Level

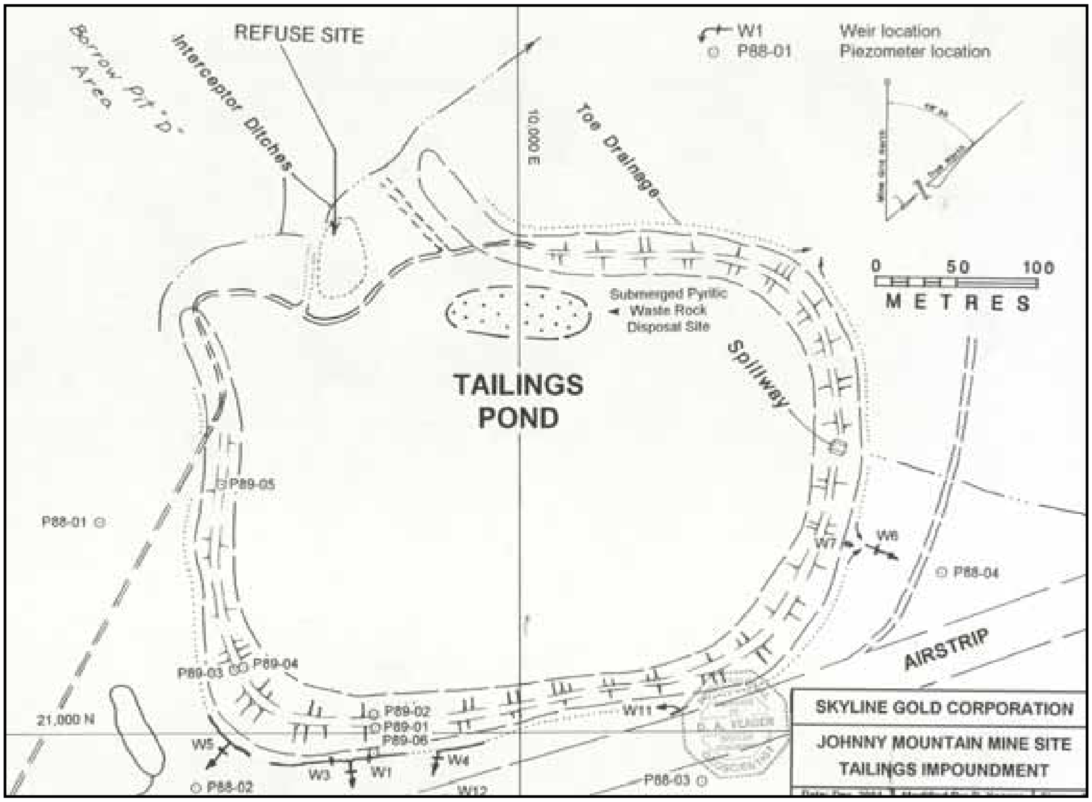

Figure 6. A Map of the Tailings Impoundment showing the Location of Waste Rock Deposition

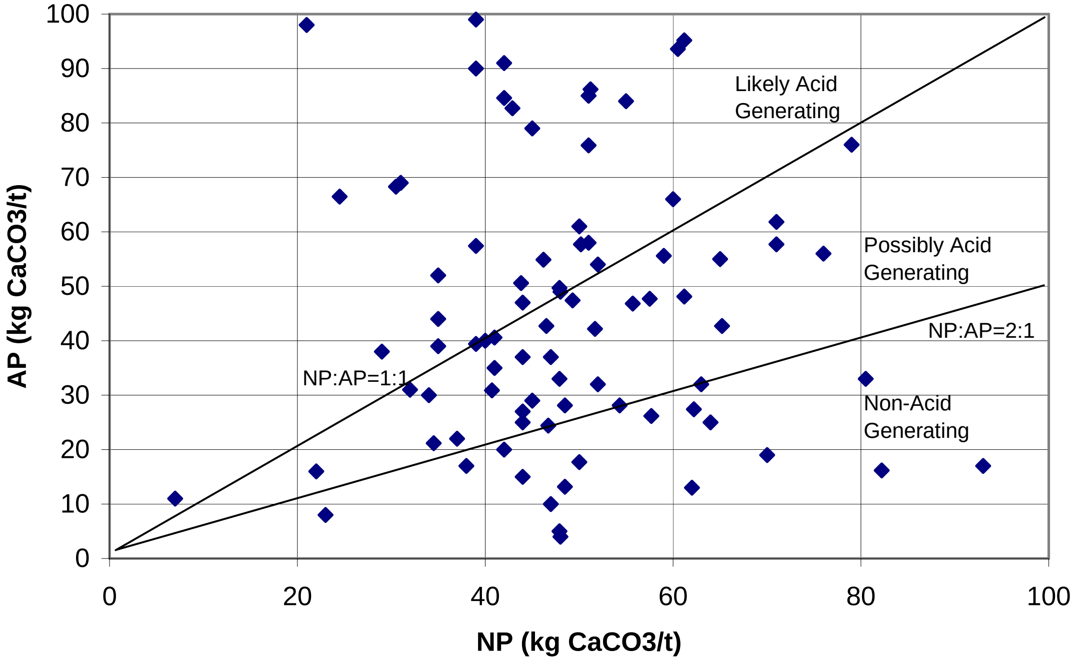

Figure 7. AP versus NP for the Underground Workings (adapted from Yeager, 2002)

Photo 1. Looking South along the Toe of Johnny Mountain Across the Johnny Flats Plateau towards the Johnny Mountain Mine

Photo 2. Looking up the Craig River Valley (background) at the 11 Level Waste Rock Dump (left) and the Camp/Plant Site/10 Level Dump (center-right)

Photo 3. The Airstrip and the Tailings Impoundment

Photo 4. Stonehouse Creek just below the Mine Site, with the Road to Magazine and to Snip in the Background

Photo 5. The Camp and Plant Site on May 19th, 1999

Photo 6. Snow Damage to the Camp

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

Acidity is produced when 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

1. $FeS_2 + O_2 + H_2O \longrightarrow Fe(OH)_3 + 2SO_4^{2-} + 4H+$

2. $S_2 + O_2 + 2H_2O \longrightarrow 2SO_4^{2-} + 2H+$

3. $Fe^{2+} + O_2 + H_2O \longrightarrow Fe(OH)3 + 2H+$

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.

4. $CaCO_3 + \underline{2H}+ \longrightarrow Ca^{2+} + H_2CO_3$

5. $CaCO_3 + \underline{H}+ \longrightarrow Ca^{2+} + HCO_3$

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 1 predominates below pH 6.4. Reaction 2 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 $CaCO_3$ 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 HCO_3^- generated from CO_3 minerals in reaction 2 is retained in the pore water and neutralizes subsequent acidity. The extent to which the HCO_3^- 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 CO_3 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 (CO_3[1]-NP), which is calculated from the % total or inorganic C or CO_2, assuming all the CO_3 is calcite. Both the bulk- and CO_3-NP are reported as kg $CaCO_3$/t. If all the carbonate is Ca and Mg species, the difference between the bulk-NP and CO_3-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.

<u>Impact of Fe and Mn Carbonate on the CO_3- and Bulk-NP</u>

The occurrence of Fe and Mn CO_3-containing minerals, such as ankerite [$Ca(Ca,Mg,Fe)CO_3$] and siderite ($FeCO_3$) raises concerns regarding the accuracy of both the bulk- and CO_3-NP.

$$6.\quad FeCO_3 + \underline{2H^+} \longrightarrow Fe^{2+} + H_2CO_3$$

$$7.\quad Fe^{2+} + O_2 + OH^- \longrightarrow Fe(OH)_3 + \underline{2H^+}$$

As shown in reactions 6 and 7 above, the dissolution of Fe and Mn CO_3 initially consumes acidity in a similar manner to calcite (reaction 6). However under aerobic conditions[2], 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 CO_3 is present,

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[1] CO_3-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 CO_2-NP if a CO_2 assay is used.

[2] Anoxic conditions inhibit the oxidation and hydrolysis reactions.

the carbonate mineral NP (CO_3[3]-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.

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, CO_3-NP > Sobek-NP, the reverse of what is otherwise observed. Where there is no detailed data on CO_3 mineralogy, CO_3-NP > 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 CO_3-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.

Impact of Mg Carbonate-Containing on the CO_3-NP

Due to the lower atomic weight of Mg versus Ca (24 versus 40), $MgCO_3$ (formula weight of 84.32) provides approximately 18% more neutralization per unit C than $CaCO_3$ (formula weight of 100.09). This will not affect the accuracy of the Sobek-NP. However if significant Mg is

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[3] CO_3-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 CO_2-NP if a CO_2 assay is used.

present, %CO_2-NP calculated assuming all the CO_3 is $CaCO_3$ will underestimate the carbonate NP. The most common Mg containing CO_3 mineral is dolomite. Ankerite often also contains significant Mg, in addition to Ca, Fe and possibly Mn CO_3. Mg may occur as a trace consitituent in calcite and siderite, but typically the concentration is low and no correction is required. Notably, the presence of significant $MgCO_3$ may counteract some or all of the affect of non-neutralizing Fe and Mn CO_3.

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 CO_3–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 CO_3 means the CO_3–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 CO_3–NP of 1 to 2). A comparison of the CO_3–NP and bulk-NP is commonly used to roughly assess the relative amounts of CO_3–NP and silicate-NP in the bulk-NP. See previous for discussion of potential impact of Fe, Mn and Mg carbonate on accuracy of CO_3–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 Lawrence-NP are measured properly, the amount and thus the contribution of the silicate-NP is 5 to 15 kg $CaCO_3$/t higher than the CO_3–NP.

The main cause of higher contributions of silicate-NP is 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 CO_3–NP. The latter is a common problem with the Sobek procedure. Minerals that are soluble at low pH are not necessarily soluble at neutral pH. XRD results will show the presence of minerals that are soluble at low pH.

A simple way to check whether excess acid was added is to compare the amount of acid added (check the fizz rating) with the CO_3–NP and resulting Sobek-NP values.

Amount of Acid Corresponding to Each Sobek Fizz Rating:

None	20 mL 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 CO_3–NP and the most reactive silicate-NP. This achieved by adding slightly more acid than is required to react with the Ca and Mg **CO_3**. 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 CO_3–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 CO_3–NP and resulting Sobek-NP are less than 50 kg $CaCO_3$/tonne. Important considerations resulting from the above include:

- the need to report the fizz rating,
- the potential to substitute the CO_3–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 CO_3.

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 insoluble 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 instances where sulphate-S is significant (e.g., Boss Mountain, Huckleberry and Kemess North) 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, Pb and Sr SO_4. 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 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.

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 $CaCO_3$ Depletion from Drainage Chemistry in Humidity Cell Test

The rate of $CaCO_3$ 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.

1. $FeS_2 + 7/2\ H_2O + 15/4\ O_2 = 2SO_4^{2-} + Fe(OH)_3 + 4H^+$

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 SO_4^{2-} /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+.

2. $CaCO_3 + \underline{2H}+ \longrightarrow Ca^{2+} + H_2CO_3$

3. $CaCO_3 + \underline{H}+ \longrightarrow Ca^{2+} + HCO_{3-}$

The formula weight of $CaCO_3$ is 100.09. The mg H^+ /kg/wk is converted to mg $CaCO_3$/kg/wk by multiplying by 75, assuming an equal mix of reactions 2 and reaction 3 (see Appendix A). Thus the acidity produced by pyrite oxidation resulting in 1 mg SO_4^{2-}/kg/wk is equivalent to

1.5625 mg $CaCO_3$/kg/wk or 81.25 mg $CaCO_3$/kg/yr. Based on the above, 8 to 22 mg SO_4^{2-}/kg/wk is equivalent to 12.5 to 34.4 mg $CaCO_3$/kg/wk or 0.65 to 1.79 kg $CaCO_3$/t/yr. It should be noted that humidity cells are run at a much higher average temperature than is expected in neutral pH, waste rock dumps in Canada. Lower temperatures will reduce the rate of pyrite oxidation and the associated rate of NP depletion.