BLENDING AND LAYERING WASTE ROCK TO DELAY, MITIGATE OR PREVENT ACID ROCK DRAINAGE AND METAL LEACHING: A CASE STUDY REVIEW

MEND Project 2.37.1

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EXECUTIVE SUMMARY/SOMMAIRE

As a long term means of preventing the onset of acidic drainage, potentially acid generating and acid consuming waste rock can theoretically be blended or layered to produce a geochemically benign composite. This report presents the general theory behind blending and layering for prevention and control of acid rock drainage (ARD), and presents selected case studies on the use of this technique at mines in Canada, United States and other parts of the world.

The focus of this report is the blending, mixing and layering of waste rock, and, to some extent, the addition of limestone and other alkaline materials. The addition of alkaline materials was initially considered beyond the scope of this MEND sponsored project (Project 2.37.1). However, the scope was expanded as very few case studies were found which exclusively examined blended waste rock, especially in the coal fields. Both coal and metal mines are covered in the review.

Coal Mines

Most of the reviewed coal case studies were located in the Appalachian region of the eastern United States with pyrite sulphur values in the range of 0.1 to 1%. At these sites, the effect of waste rock blending was often confused or masked by additional management measures such as lime and flyash addition, and/or the presence of acidic drainage from adjacent historic sites. Comprehensive reports, providing compilations and assessments of acid-base accounting (ABA) data and kinetic tests, predictions of long term effluent quality, pre and post mining monitoring of rock characteristics, and pre and post mining monitoring of drainage quality, were not found in the course of this

project. However, numerous papers in the published literature reviewed specific and general management practices at individual sites.

Researchers in Appalachia have compiled estimates of volume-weighted ABA data derived from drill core analyses, and compared these values to effluent quality at associated seepages. These compilations (diPretoro and Rauch, 1988; Ericksen and Hedin, 1988; Brady *et al.*, 1994) have resulted in the development of general criteria for overall volume-weighted waste rock blends considered likely to produce net alkaline drainage but not necessarily prevent metal leaching. The commonly cited criteria in the Appalachia (P. Ziemkiewicz, *pers. comm.*, 1997) are:

Net Neutralization Potential (NNP) > 10 kg $CaCO_3$ equivalent/tonne;
Neutralization Potential (NP) > 15 kg $CaCO_3$ equivalent/tonne; and,
Ratio of Neutralization Potential to Acid Potential (NP/AP) ratio > 2.

The reviewed studies on coal wastes tended to examine the addition of alkaline materials rather than the deliberate blending of acid generating and acid consuming waste rock types. These studies indicated that:

a) Fine alkaline material distributed uniformly throughout a waste pile were effective in delaying the onset of acidic effluent;
b) Layering of limestone within test piles was effective in reducing the amount of net acidity produced, and in delaying the onset of acidic effluent, but did not effectively prevent acid conditions from developing within the potentially acid generating (PAG) portions of the piles;
c) Lime kiln dust was more effective in reducing the amount of net acidity produced and in preventing the onset of acidic effluent than limestone, possibly due to the higher surface area and the greater reactivity of CaO;
d) The order in which overburden was placed in relation to acid drainage generating strata had a significant effect on leachate quality; and,
e) Placement of overburden as soon as possible after excavation was beneficial in reducing ARD.

The critical ratio of alkaline addition to acid potential which would permanently prevent the onset of acidic drainage at specific sites was generally not predicted.

Metal Mines

Few metal mine sites were identified that had deliberately examined or applied blending and/or layering as a prevention method. Typically, older sites that provided long term water quality data to support the success or failure of this technique lacked sufficient documentation on materials in the waste piles and the level of blending employed to allow conclusions to be drawn. Newer sites that characterized their materials and operational procedures have had limited time in which to demonstrate success or failure.

Site data and laboratory test data for metal mines tended to indicate that:

a) The majority of cases evaluated blends with NP/AP ratios less than 2;

b) Blending did not reduce sulphide oxidation rates in the potentially acid generating material unless highly reactive neutralizing material (limestone) was applied and the blending was near ideal. Perfect mixing was generally only possible in column or humidity cell tests. Such ideal blending was not considered feasible at field scale;

c) Layering thicknesses down to 10 cm did not reduce sulphide oxidation rates in potentially acid generating material;

d) Blending and layering were effective in delaying the onset of acidic effluent, and reducing the metal and acidity loadings exiting from the combined material. The presence of alkaline drainage from neutralizing materials appeared to temper the locally produced acidic leachate, resulting in less acidic drainage and lower metals levels due to decreased solubility of hydroxide and carbonate phases at the elevated pH.

e) Prevention of acidic effluent from the blended or layered materials did not necessarily prevent dissolved metal levels from being problematic.

The reviewed metal mine case studies did not identify safe waste rock blends which would prevent ARD and metal leaching at those sites.

From the literature and case studies reviewed, it appears that blending and layering of acid generating and acid consuming waste rock may be a legitimate method of neutralizing acidic drainage with a consequent reduction in metal leaching through precipitation of metals within the pile. However, there remains considerable uncertainty, and further investigation into the application and practice of this method is required. Specifically, more information on large scale, controlled field studies is needed. The identification and detailed documentation of additional sites should also be encouraged, in association with predictive laboratory tests. At issue is what portion of alkalinity is available to neutralize the acidity produced by sulphide oxidation, and the extent that this is influenced by the thoroughness with which alkaline and acid producing waste rock are blended at a given site.

SOMMAIRE

Pour prévenir à long terme le drainage acide, il est théoriquement possible de mélanger ou de stratifier des couches de stériles potentiellement acidogènes avec des stériles consommateurs d'acide, de manière à obtenir un composite dont les effets géochimiques seront pour ainsi dire négligeables. Dans le présent rapport, on décrit les principes théoriques généraux qui sous-tendent le mélange et la stratification de telles couches pour prévenir ou limiter le drainage rocheux acide (DRA), ainsi que certaines études de cas portant sur l'application de cette technique dans des mines, au Canada, aux États-Unis et dans d'autres parties du monde.

Le présent rapport porte particulièrement sur le mélange et la stratification de couches de stériles, ainsi que, dans une certaine mesure, sur l'ajout de calcaire et d'autres matières alcalines. L'ajout de matière alcaline avait été initialement considéré comme dépassant la portée de ce projet parrainé par le Programme de neutralisation des eaux de drainage dans l'environnement minier (NEDEM, Projet 2.37.1). Toutefois, la portée du projet a été élargie car on a trouvé très peu d'études portant exclusivement sur les stériles mélangés, particulièrement dans les terrains houillers. On traite le cas des mines de charbon et de métaux.

Mines de charbon

La plupart des mines de charbon dont on a étudié le cas étaient situées dans la région des Appalaches, dans l'est des États-Unis, où le soufre sous forme de pyrite titrait de 0,1 à 1 %. À ces endroits, l'effet du mélange des stériles était souvent masqué en raison de l'application d'autres mesures de gestion comme l'ajout de chaux et de cendres volantes ou

le drainage acide provenant d'anciens sites voisins, ou les deux. Au cours du présent projet, on n'a pas trouvé de rapports exhaustifs contenant des compilations ou des évaluations des données relatives au bilan acide-base ou à des essais cinétiques, aux prévisions à long terme de la composition des effluents, au suivi des caractéristiques des stériles avant et après l'exploitation minière et au suivi de la composition des eaux de drainage avant et après l'exploitation minière. Cependant, de nombreux articles traitaient des pratiques spécifiques et générales de gestion dans des sites donnés.

Les chercheurs dans la région des Appalaches ont compilé des estimations des données relatives au bilan acide-base, pondérées en fonction du volume, provenant d'analyses de carottes de forage et ils ont comparé ces valeurs à la composition des effluents d'autres eaux de suintement. Ces compilations (diPretoro et Rauch, 1988; Ericksen et Hedin, 1988; Brady *et al.*, 1994) ont donné lieu à l'élaboration de critères généraux applicables aux mélanges globaux de stériles, pondérés en fonction du volume, jugés susceptibles de produire des eaux de drainage alcalines, mais sans nécessairement prévenir la lixiviation des métaux. Les critères couramment cités pour les Appalaches (P. Ziemkiewicz, communication personnelle, 1997) sont les suivants:

Potentiel de neutralisation net (PNN) > 10 kg $CaCO_3$ équivalent/tonne;
Potentiel de neutralisation (PN) > 15 kg $CaCO_3$ équivalent/tonne;
Rapport du potentiel de neutralisation au potentiel acide (PN/PA) > 2.

Les études relatives aux stériles de charbon avaient tendance à examiner l'ajout de matières alcalines plutôt que le mélange délibéré de stériles producteurs d'acide avec des stériles consommateurs d'acide. Ces études ont montré que:

a) De fines matières alcalines réparties uniformément à travers une halde de stériles permettaient de retarder efficacement la production d'effluent acide;

b) La stratification de couches de calcaire dans les haldes testées permettait de réduire efficacement la quantité nette d'acidité produite et de retarder la production d'effluent acide, sans prévenir efficacement l'apparition d'acidité au sein des parties potentiellement acidogènes des haldes;

c) La poussière de four à chaux réduisait plus efficacement la quantité d'acidité nette produite et prévenait plus efficacement la production d'effluent acide que le calcaire, peut-être en raison de sa surface plus étendue et de la plus grande réactivité du CaO;

d) L'ordre suivant lequel les morts-terrains étaient déposés par rapport aux couches acidogènes influait beaucoup sur la composition du lixiviat;

e) Le dépôt des morts-terrains le plus tôt possible après l'excavation réduisait avantageusement le DRA.

En général, on n'a pas prévu le rapport critique de la quantité de matière alcaline ajoutée au potentiel acide qui préviendrait en permanence la production de drainage acide à des sites miniers donnés.

Mines de métaux

On a trouvé peu de mines de métaux qui avaient délibérément envisagé ou appliqué le mélange ou la stratification, ou les deux, comme méthode de prévention. En général, les sites anciens pour lesquels on disposait de données relatives à la composition de l'eau sur de longues périodes, pour confirmer ou infirmer l'efficacité de cette technique, n'offraient pas suffisamment de documentation sur les matières dont étaient constitués les haldes de stériles ni sur le degré de mélange pour permettre de tirer des conclusions. Dans le cas des sites plus récents, les matières et les procédures avaient été caractérisées, mais pas pendant assez longtemps pour faire la démonstration du succès ou de l'échec de la méthode.

Les données relatives aux sites et les données fournies à partir des essais en laboratoire avaient tendance à indiquer que:

a) Dans la majorité des cas évalués, les mélanges présentaient des rapports PN/PA inférieurs à 2;
b) Le mélange ne réduisait pas le taux d'oxydation des sulfures des matières potentiellement acidogènes sauf si une matière neutralisante hautement réactive (calcaire) avait été appliquée et si le mélange était presque idéal. En général, il n'était possible d'obtenir un mélange idéal que dans les essais en colonne ou en cellule humide. On considère qu'un tel mélange idéal ne pourrait être obtenu à grande échelle;
c) En stratifiant des couches jusqu'à 10 cm, on ne réduisait pas le taux d'oxydation des sulfures dans les matières potentiellement acidogènes;
d) Le mélange et la stratification retardaient efficacement la production d'effluent acide et réduisaient les charges en métal et en acidité sortant de la matière combinée. La présence d'eaux de drainage alcalines issues des matières de neutralisation semblait atténuer la production locale de lixiviat acide, ce qui donnait lieu à des niveaux moins élevés de drainage acide et de métaux en raison de la solubilité réduite des phases hydroxyde et carbonate aux pH élevés.
e) En prévenant la formation de l'effluent acide avec des matières mélangées ou stratifiées, on n'empêchait pas nécessairement les problèmes résultant de la

quantité de métaux dissous.

Les études examinées dans le cas des mines de métaux n'ont pas permis de caractériser des mélanges de stériles sûrs qui préviendraient le DRA et la lixiviation des métaux à ces endroits.

D'après la documentation et les études de cas passées en revue, il semble que le mélange et la stratification des stériles producteurs et consommateurs d'acide puisse constituer une méthode légitime pour neutraliser les eaux de drainage acide, permettant aussi de réduire la lixiviation des métaux en faisant précipiter les métaux dans la halde. Toutefois, l'incertitude reste considérable et une étude plus poussée de l'application de cette méthode est nécessaire. Plus particulièrement, il faudrait obtenir davantage d'information dans le cadre d'études contrôlées à grande échelle sur le terrain. On recommande également la caractérisation et la documentation détaillée d'autres sites, en association avec des essais prévisionnels en laboratoire. Il s'agit de savoir quelle part de l'alcalinité peut servir à neutraliser l'acidité produite par l'oxydation des sulfures, et dans quelle mesure cette part dépend du mélange des stériles producteurs d'alcalinité et d'acidité à un site donné.

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1.0 INTRODUCTION

1.1 Background and Scope of Work

As a long term means of preventing acidic drainage from waste rock piles, potentially acid generating and acid consuming waste rock can theoretically be blended or layered to produce a geochemically benign composite. Coal mines in the eastern United States have used blending of waste rock as part of waste management schemes to combat ARD for decades. However, the success of blending and layering alone is generally masked because the management schemes also include the removal and/or segregation of the most strongly acid generating layers and the addition of alkaline materials such as lime and flyash (reference examples).

Various reviews of historic coal mine sites by researchers in the Appalachia region of the United States (Section 4.3.1) have indicated that net alkaline drainage is likely to be produced when:

> Net Neutralization Potential (NNP) > 10 kg $CaCO_3$ equivalent/tonne;
> Neutralization Potential (NP) > 15 kg $CaCO_3$ equivalent/tonne; and,
> Ratio of Neutralization Potential to Acid Potential (NP/AP) ratio > 2.

The reported success of blending in producing non-acidic drainage led to the consideration of blending as a mitigation technique for acid rock drainage (ARD) from metal mine waste rock piles.

Successfully blended waste rock can be theoretically defined as any composite of potentially acidic and alkaline rock which results in complete internal consumption of acidity and precipitation of deleterious dissolved ions produced by sulphide oxidation such that the drainage leaving the waste rock pile meets effluent water quality standards. Ideally the objective is to reduce the rate of sulphide oxidation by promoting predominantly alkaline leachate throughout the waste rock pile. The waste rock generated during mining may have sufficient neutralization potential to maintain alkaline conditions, or the addition of imported acid consuming rock such as limestone may be required.

In practice, it is not easy to achieve a fully blended or uniform mixture of rock throughout a waste rock pile since the rock is usually deposited in quantities defined by haul truck capacities. For this reason, layering has been proposed as an alternative means of combining the acid generating and acid consuming materials. There are other potential benefits from layered designs involving acid consuming and acid generating materials. For example, at the Wizmut Project, surface layers of waste rock are expected to consume oxygen under neutral pH conditions, limiting the oxygen flux to underlying acid generating materials (Hockley *et. al.*, 1997). In addition, segregated zones of *materials* with strong acid generating potential have been compacted in layers and covered with layers of imported alkaline amendments at several coal mines (Skousen and Larew, 1994; Rose *et al.* 1995).

At a practical level, blending may be considered successful if the acidity and/or metals in the drainage from the blended waste rock pile are substantially reduced, such that subsequent treatment requirements (such as chemical treatment or passive wetland treatment) are reduced.

Blending or layering of waste rock and limestone has been proposed and/or implemented at several coal and metal mine sites. The primary objective of this project was to review data from these sites to gain insight into the effectiveness, strengths, and limitations of each site's specific use of blending or layering as a mitigation measure for ARD. Seven metal mining projects were selected for detailed evaluation. These included two sites where full-scale waste rock piles had been constructed (Samatosum Mine and Stratmat Heath Steele Mine), one site where three field test pads had been constructed (Kutcho Creek), three sites where laboratory kinetic tests were completed (Cinola, Eskay Creek and Windy Craggy), and one site where a new control technique using layers is currently being implemented (Wismut). Data to the same level of detail were not found for coal mines in the course of this project, although excellent summaries were available in published papers.

Projects where adequate data was unavailable to carry out a detailed evaluation, or which were still in the planning phase, or which were not considered directly relevant to this study were also reviewed. These included several sites where the sole effect of waste rock blending was complicated and/or masked by the inclusion of other management measures, such as the addition of alternative alkaline materials such as lime and flyash. The short case studies included small scale laboratory tests mixing limestone with Duluth Complex rock, the Mt. Milligan Project, Telkwa Coal, Rawhide Mine, Kauffman Mine, High Power Mountain, the Coaltrain Project and several coal mining trials in the eastern United States.

This report provides a brief overview of the theory behind blending and layering and the factors which influence blending success (Section 2), a review of the testing methods and evaluation procedure used to predict whether blending or layering is an appropriate control measure at a given site (Section 3), a summary of each of the case studies reviewed during this study (Section 4), and a discussion of the overall findings (Section 5). Appendices A through G provide detailed case studies on sites where blending and/or layering were considered and where detailed testing and/or field monitoring data were available. The case studies include descriptions of the geology, mining and waste management plans, static and kinetic testing results, and the operational and monitoring data for each of the sites, where available.

1.2 Methods of Data Collection

Several methods were used to obtain and compile data on the blending and layering case studies. The data compilation included the following activities:

- a literature search for published information;

- interviews with industry contacts, mining consultants and researchers to identify appropriate sites;

- soliciting previously compiled and readily available though unpublished documentation directly from industry, consulting and research contacts, including members of the MEND Prevention and Control committee;

- examination of private and/or institutional libraries covering acid rock drainage at the University of British Columbia Mining Department Environmental, West Virginia University, and the US Department of Energy, former US Bureau of Mines, Pittsburgh Research Station; and

- interviews with company personnel and experts via telephone and/or in person to obtain unpublished and anecdotal data.

The initial literature search was conducted by Desiree Bradley Library and Information Services, North Vancouver, B.C. The selected databases covered included the following topics: environmental/water pollution, geological and engineering sources, general scientific sources, newsletters and business articles. Keywords used in the basic search statement were: “waste”, “rock”, “layer[1]”, “blend*”, “mix*”. Extended searches used the following keywords: “minesite”, “mine”, “site”, “mining”, “mingling”, and “Australia”. The specific databases that were searched are listed in Table 1.

An alternative search of the databases available through the Environmental Routenet service was also done. This search included the following Internet databases: Agricultural and Environmental Biotechnology Abstracts, ASFA 3: Aquatic Pollution and Environmental Quality, Ecology Abstracts, EIS: Digests of Environmental Impact Statements, Health and Safety Sciences Abstracts, Industrial and Applied Microbiology

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[1] For search statements, “*” means open ended ending on word. “ADJ” means as an adjective to the following words.

Abstracts (Microbiology A), Pollution Abstracts, and Risk Abstracts. The search query was: ((acid* ADJ mine ADJ drainage or waste ADJ rock or tailings ADJ (piles or dumps)) and (mix* or layer* or blend*)).

Recent conference proceedings were also examined, as were a list of "recent publications of interest" published in the USGS Mine Drainage Newsletter.

Interviews with industry contacts, consultants and researchers were a very important source of information, both for identifying potential sites, and for providing specific data on them. Generally the information provided by these contacts was more detailed than data from the published sources. Often, data was provided in a form that was not conducive to examining issues of blending and layering. In these cases, MEM Inc. undertook analysis of the data to draw conclusions relevant to this MEND study.

1.3 Acknowledgements

Several people made a special effort to provide information and direction to this study. The following contributions are noteworthy:

- Dr. W.A. Price, Energy and Minerals Division, Ministry of Employment and Investment, for initiating the project, developing the terms of reference, and providing project direction;
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- K. Sexsmith for preliminary compilation and assessment of data for several of the case studies.
- Dr. K.A. Morin for providing valuable insights and review comment;
- Dr. P. F. Ziemkiewicz and Dr. J. Skousen at West Virginia University for providing access to their personal libraries and insights into coal operations; and,

- The many company representatives who contributed to the case studies by supplying data, and reviewing and providing comments on their case studies.

Table 1: Databases Covered in Literature Search

Database File No.	Database Name	Time period
Scientific Sources:		
40	Enviroline	1975 - Aug 1996
44	Aquatic Science and Fisheries Abstracts	1979 - Aug 1996
117	Water Resources Abstracts	1967 - Aug 1996
41	Pollution Abstracts	1970 - Sept 1996
89	Georef	1885 - Oct 1996
292	Geobase	1980 - Sept 1996
8	Ei Compendex Plus	1970 - Oct 1996
35	Dissertation Abstracts Online	1861 - Sept 1996
144	Pascal	1973 - Aug 1996
34	SciSearch	1988 - Sept 1996
6	NTIS (National Technical Information Service)	1966 - Oct 1996
265	Federal Research In Progress	1996 - Aug 1996
420	UnCover	1988 - Sept 1996
77	Conference Papers Index	1973 - Sept 1996
65	Inside Conferences	1993 - 1996
DATASTAR (UFOR)	Umweltforschungsdatenbank German, Austrian, Swiss Research Projects	1974 - Oct 1996
DATASTAR (ULIT)	Umweltiteraturdatenbank German Federal Environmental Agency Library	1976 - Oct 1996
Business Sources		
636	IAC Newsletter Database	1987 - Sept 1996
148	IAC Trade and Industry Database	1976 - Sept 1996
9	Business and Industry	1994 - Sept 1996
15	ABI/INFORM	1971 - Sept 1996

2.0 THEORETICAL BASIS AND CONSIDERATIONS

2.1 Brief Theory of Blending

In an ideally blended waste rock pile, sulphides are in intimate contact with acid consuming materials, giving a homogeneous mix that is net acid consuming throughout the pile. Consequently, sulphide oxidation occurs at a pore water pH of between 6 and 8 as the acidity produced by sulphide oxidation is immediately neutralized by bicarbonate (HCO_3^-) in the pore water or by contact with carbonate minerals. Migration of acidity from its points of release is limited and therefore metal leaching is reduced. Leachate emerging from an ideally blended waste rock pile would be pH neutral, with low concentrations of trace metals from acid-soluble minerals.

Departures from the ideal juxtaposition of sulphide and carbonate minerals can result in conditions where sulphide oxidation occurs rapidly enough to produce local acidic conditions. This will primarily occur if sulphide minerals exist in large enough concentrations that alkaline pore water originating nearby is not sufficient to maintain alkaline conditions in the vicinity of sulphide grains. Acidic pore water produced locally can migrate away from the point of origin, but may be rapidly neutralized by contact with nearby carbonate minerals. In this case, leachate would be pH neutral with generally low trace metal concentrations from acid-soluble materials, but likely contain elevated sulphate concentrations. Note that metals may be released to waste rock drainage via other geochemical processes.

The greater the separation between acid generating and carbonate materials, the greater the opportunity for acid soluble metal leaching to occur prior to neutralization. As the blend becomes less ideal, deleterious ion concentrations may increase in the pore water, and may not encounter sufficient alkalinity to form precipitates within the confines of the pile. Hydrogeologic conditions may further limit the contact between acidic and alkaline seeps in the pile. The transition from ideal blended waste rock to non-blended is summarized in Table 2. The transition occurs as a result of the extent of physical effort put into mixing of the rock types, but also as a result of the geochemical differences between the rock types. Thus, the ideal blended waste rock pile is unlikely to be achievable except in a situation where the rock is well mixed, with very little difference between potentially acid generating and acid consuming rock types.

Table 2: Range of Blending Effectiveness

Ideal blend	Near ideal blend	Incomplete blend	Poor blend
Alkaline conditions dominate throughout the pile.	Locally acidic conditions may occur but neutralization occurs rapidly.	Locally acidic conditions occur and migration of acidic pore water leaches surrounding rock before neutralization.	Acidic conditions occur with metal leaching. The blend is inadequate and does not result in neutralization.
Leachate is pH neutral, low trace metal concentrations from acid soluble minerals, and tends to have low sulphate levels.	Leachate is pH neutral and low trace metal concentrations from acid soluble minerals, but likely has moderate to high sulphate levels.	Leachate is pH neutral, has high sulphate, low transition metal concentrations but potentially elevated concentrations of some other ions such as zinc .	ARD released.

2.2 Brief Theory of Horizontal Layering

2.2.1 Conventional Layered Designs

Layering is often used as a practical means to 'mix' potentially acid generating materials and acid consuming materials in the waste rock piles. However, the non-homogeneous placement allows acidic leachate to be produced in the acid generating layers which is then subsequently neutralized by acid consuming materials in underlying layers. If a layered design is functioning correctly, neutralization and metal precipitation will still occur within the pile, and the seepage leaving the pile will not be acidic. However, certain metals may still be present at elevated concentrations.

It has been widely believed that layering acid consuming waste rock over acid generating waste rock could prevent acid generation in the underlying acid generating layer by buffering the pH, and therefore reducing the rate of sulphide oxidation (Bell, 1988). However, studies have shown that sulphide oxidation rates do not change significantly between pH 1 and 8 for both abiotic and biotic conditions (Morin and Hutt, 1997a). Moreover, data from the Eskay Creek Project (Appendix E) case studies showed that alkaline cover layers did not significantly reduce sulphide oxidation rates, and did not provide sufficient alkalinity to neutralize acidity produced in the underlying potentially acid generating layer. Thus the data indicate that alkaline cap or cover layers may not be effective in a mitigation measure.

Layered designs which place the acid consuming materials underneath potentially acid generating materials are theoretically capable of buffering acidic leachate and reducing metal concentrations in the overall seepage. Potential problems with layered pile designs are that the acid consuming materials may be "blinded" or coated with chemical precipitates formed as the acidic leachate flows over them (Rose and Daub, 1994). This may result in a disproportionate reduction in the availability of the neutralization potential as compared to well blended mixtures. In addition, there is more opportunity for preferential flow paths to develop within layered piles, which may allow leachate to bypass the neutralizing materials. Increasing the number of layers and reducing the thickness of the layers increases the potential for a layered design to approximate more ideal blending.

2.2.2 Oxygen Consuming Layer

A special form of layering has been proposed (Jakubick *et al.*, 1997; Hockley *et al.*, 1997) with the objective of using waste rock to reduce ARD through oxygen consumption, rather than neutralization of acidity. The oxygen consuming layer is expected to reduce the flux of oxygen into the underlying material such that sulphide oxidation does not occur to any extent. This require materials in the oxygen consuming layer which have a relatively high sulphide content, yet are net acid consuming and contain a sufficient amount of calcite to precipitate sulphate from solution. A secondary cover is also required to reduce convective movement of oxygen into the waste rock pile (Steffen Robertson and Kirsten (Canada) Inc., 1995b, 1997).

2.3 Factors Affecting Blending and Layering Success

The following factors will have an effect on whether blending and/or layering will achieve neutral drainage, and internal consumption of acidity and precipitation of deleterious dissolved ions:

- Mineralogy, reaction kinetics and grainsize;
- Relative proportions of the net acid generating and net acid consuming rock types, and degree of mixing (NP/AP ratios throughout the pile);
- Orebody geometry and mining plan;
- Construction methods;
- Hydrogeology of the waste rock pile; and,
- Operational commitment and monitoring.

The influence of each of these factors on the potential success of blending or layering is described in the following sections.

2.3.1 Mineralogy, Reaction Kinetics and Grainsize

The classic theory of ARD assumes that the reactive minerals are pyrite (oxidizing to ferric hydroxide) and calcium/magnesium carbonates. Given these minerals, and based purely on molar chemical considerations, the NP/AP ratio of the rock must exceed a value (NP/AP ratio$_{crit}$) of between 1 and 2 to produce neutral pH drainage (Norecol, Dames & Moore, Inc., 1994). For this reason, criteria for blending are often expressed in terms of the NP/AP ratio required throughout a blended waste rock pile.

Typically, however, the mineralogy of waste rock consists of more than simply pyrite, calcite and dolomite. Other neutralizing minerals often act as less rapid acid buffers (Sverdrup, 1990). Therefore, an NP/AP ratio must be determined for each unique site which will prevent acid release from that site's blended materials.

Other mineralogical factors also influence the chemical reactivity of the waste rock, and are reflected in higher site specific NP/AP ratio criteria. For example, encapsulation within the matrix of a competent rock type can significantly reduce the availability of either sulphide or carbonate minerals. If minerals lie on bedding planes, joints, or cleavage surfaces, they are more likely to be exposed to air and water, and therefore, be available for chemical reaction. In particular, soft minerals in veins and fracture fillings are likely to be liberated by blasting whereas disseminated minerals in a harder silicate matrix are likely to be liberated to a lesser degree. For example, carbonate mineralization is often found in fractures as a result of late stage mineralization and low grade (greenschist) regional metamorphism (Day, *pers. comm.,* 1997). Similarly, pyrite may also concentrate along fracture surfaces, and be preferentially exposed during blasting (Price and Kwong, 1997).

Physical breakdown following placement in the waste rock piles may also liberate additional sulphide or carbonate minerals. If there are substantial differences in the physical weathering characteristics of the net acid generating and net acid consuming rock types, more sulphides may be exposed than the carbonates, potentially leading to net acid drainage despite the appearance of an acceptable overall chemical balance. The reverse condition is less critical. Rock types for which rapid physical breakdown may be a significant factor include argillaceous sedimentary rocks (shales, siltstones), sericitic and chloritic schists, and other sericite-altered rock types. The strong reactivity of sericite schists is illustrated at La Mine Doyon (MEND, 1994).

The kinetics of sulphide oxidation and acid neutralization will also influence the effectiveness of the blend. If the blend is functioning ideally, the continuous dissolution of carbonates in response to acid generation should be sufficient to prevent the migration of acid away from the microenvironment around the reacting sulphide grains. The rate of sulphide oxidation may be partially controlled by maintaining neutral pH conditions and limiting biological oxidation.

The maintenance of neutral pH conditions does not, however, ensure that metal leaching will be limited. An overall neutral pH in an ideally blended pile may limit the rate of sulphide oxidation and therefore control the rate at which metals associated with sulphides are released into solution. A neutral pH may also limit the solubility of some metals that have been released into solution, causing precipitation of metals within the pile. However, some metals are associated with more soluble mineral phases, for example sulphates, hydroxides or carbonates. Even a small decrease in the overall pH may increase the rate of metal leaching from these minerals. Other minerals released by sulphide oxidation may not be adequately reciprocated by neutral pH conditions. Zinc tends to show this behaviour. Still other minerals do not require sulphide oxidation to be released, as they are already significantly soluble in neutral pH conditions. Therefore the mineralogy and availability of the heavy metal minerals are also important to the overall success of a blending strategy.

Testing methods to determine the kinetic behaviour of waste rock blends is further discussed in Section 3.0.

2.3.2 Relative Proportions of Acid Generating and Acid Consuming Rock Types and Resulting NP/AP Ratios

The relative proportions and chemical reactivity of the different rock types (net acid generating, net acid consuming, relatively inert) are probably two of the most important factors determining whether a blended rock pile will prevent the release of ARD. Rock types considered for blending may vary from weakly to strongly sulphidic and/or calcareous. The blend may consist completely of reactive rock types (ie. net acid generating and net acid consuming rock types totalling 100%) or a significant amount of relatively inert rock. The ability of the blend to approach ideal conditions depends on the intimacy of contact between the rock types and the degree of heterogeneity through out the waste rock pile. Even with a carefully planned system for placing the net acid consuming and net acid generating rock types in the pile, there is a significant potential for heterogeneities to develop if some the rocks contain anomalously high amounts of either sulphide or carbonate.

Table 3 summarizes potential mixing scenarios. The descriptions of the mixtures assume that the final NP/AP ratio of the blend would be the same in each case. The most ideal condition occurs when the two rock types are similar in chemistry and are classified as only marginally acid generating or acid consuming (low sulphur, low carbonate in the upper left box of Table 3). The least ideal scenario (high sulphur high carbonate, shown in the lower right box of Table 3) would have approximately equal proportions of two highly reactive rock types, with a significant potential for localized chemical heterogeneities or "hotspots". The two remaining scenarios represent designs where the relative proportions of acid generating and acid consuming rock types are very different. One can speculate that small amounts of high sulphide waste mixed with a large amount of calcareous waste would be more successful than small amounts of calcareous waste mixed with a large amount of low sulphide waste. In all cases, as the fraction of relatively inert rock increases, the separation between the acid generating and acid neutralizing rock increases and the blend is anticipated to be less effective in producing neutral drainage.

These comments do not imply that blending should be limited to any specific scenario. However, they do suggest that more effort is required to ensure that neutralizing minerals are in close proximity to sulphides when the acid generating potential of the rock types to be mixed are very different. This could be achieved by increasing the proportion of the acid consuming rock type to increase the overall NP/AP ratio (introducing a site specific safety factor), by improving the availability of the carbonate component by crushing, or by increased physical mixing.

Table 3: Cases of Rock Type Mixing

ROCK TYPES MIXED		Acid Generating Rock Type	
		Low Sulphur	High Sulphur
Acid consuming rock type	Low Carbonate	Lowest degree of heterogeneity, roughly equal amounts of each rock type, greatest opportunity for mixing	Small amount of sulphidic waste mixed in with a large amount of weakly calcareous waste. Potentially effective blending but some potential for hotspot formation.
	High Carbonate	Small amount of calcareous rock mixed with large amount low sulphur waste. Less effective than high S/low carbonate combination due to small quantities of calcareous rock.	Potential for significant chemical heterogeneities and formation of hotspots.

Note: Descriptions assume that all mixes would have the same bulk chemical result.

Some examples of mineral deposit types that may fit into the categories described above are shown in Table 4.

Table 4: Some Mineral Deposit Types Classified According to Table 3
(Mehling *et al.*, 1997)

ROCK TYPES MIXED		Acid generating rock type	
		Low Sulphur	High Sulphur
Acid consuming rock type	Low Carbonate	Calc-alkalic porphyry, coal, epithermal vein, BIF-hosted gold.	Calc-alkalic porphyry, coal, epithermal vein, BIF-hosted gold.
	High Carbonate	Alkalic porphyry, kimberlite, skarn, mesothermal veins.	Skarn, limestone hosted Pb-Zn, Pb-Zn vein deposits

2.3.3 *Orebody Geometry and Mining Plan*

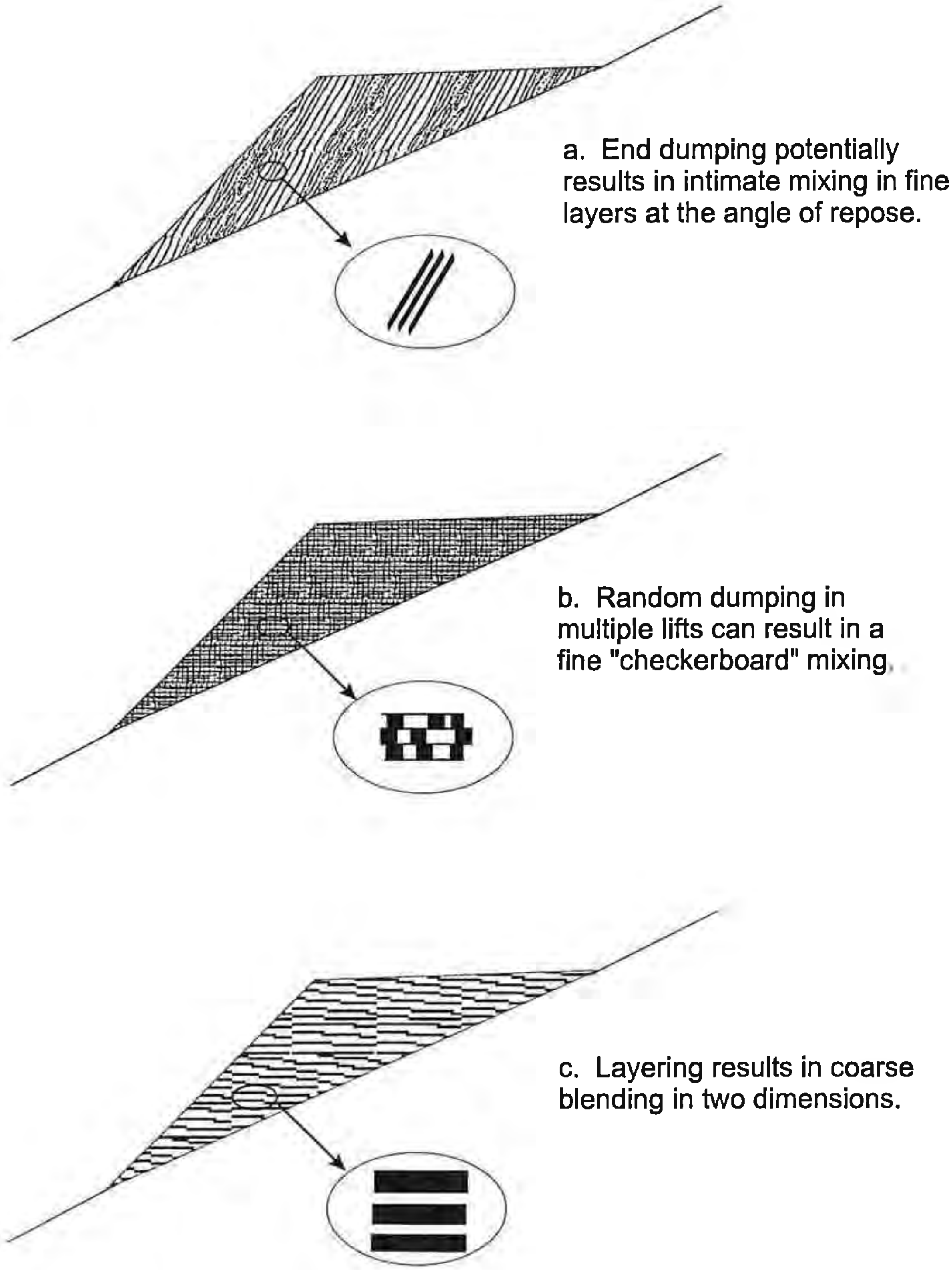
The geometry of the orebody and mining logistics will influence the order by which materials are available for blending. Economics dictate that rehandling of excavated materials be minimized. Therefore, blending would be most suitable for ore deposits where the net acid generating and net acid consuming rock types can be mined concurrently at all times. This would suggest that blending is not appropriate for open pits where one rock type is concentrated at the top or bottom of the deposit, nor for underground operations where the majority of excavations occur in one rock type. Large open pit mines and multilevel underground mines are most likely to have the required flexibility to schedule the concurrent excavation of appropriate materials from different areas of the deposit. Rehandling of waste materials and quarrying of extra materials have been used at some sites to achieve the design blends (Denholm, *pers. comm.*, 1997).

2.3.4 *Construction Methods*

The method used to construct a blended waste rock pile also determines how closely the pile approaches the ideal condition of intimate mixing of acid generating and acid consuming rock. Possible methods for constructing blended waste rock piles are:

- End-dumping in single or multiple lift dumps;
- Dumping individual truck loads within each lift; and
- Controlled construction of horizontal layers.

End-dumping in single lift piles has been proposed for the Telkwa Coal Project (Manalta Coal Ltd., 1994). This method allows different types of wastes to be mixed in three-dimensions on the scale of single haul truck loads, as the waste rock slides in thin layers down the pile face (Figure 1a). Control of successive truck loads is required to ensure that the pile generally meets the required blending criteria on a localized scale.



Black and white depict different types of rock.

Figure 1 Schematic Illustration of Blending Methods (Mehling *et al.*, 1997)

Random dumping involves placement of haul truck loads in a checkerboard pattern on a horizontal lift. The piles are then levelled using a bulldozer to provide the next working surface. The resulting dump is therefore composed of multiple small loads of the various rock types (Figure 1b). Control is required to prevent adjacent truck loads from being of the same rock type.

Horizontal layering consists of constructing alternating lifts of acid generating or acid consuming rock, resulting in a composite pile meeting the required overall NP/AP ratio criterion (Figure 1c). The thickness of layers are often determined by operational requirements rather than strictly geochemical considerations. Theoretically, the thickness of acid generating layers should be decreased as the acid generation potential increases, to limit development of acidic conditions within the layers.

One can argue that the intimacy of physical mixing of materials decreases from end-dumping to controlled horizontal layering. While layering offers a high degree of operational control when placing materials on the basis of rock type, the potential for producing locally acid generating hotspots within layers is higher than with the other two approaches. The checkerboard approach requires more care during construction, but allows the ARD potential of piles within each lift to be controlled and is likely to result in a more thorough blend. The choice of construction method will also influence the flow of water through the pile.

2.3.5 Hydrogeology of the Waste Rock Pile

Blending not only requires a high degree of mixing of different rock types, it also requires that water flow uniformly through the acid generating and acid consuming materials. Non-uniform flow can occur as water preferentially follows connected channels in the pile, encounters compacted or fine-grained layers of net acid generating or net acid consuming material in the pile, or is blocked by the precipitates of sulphide oxidation products such as gypsum (Evans and Rose, 1995).

The waste rock pile construction method (see Section 2.3.4) will influence the flow patterns. One can argue that the potential for channelling is greatest in end-dumped piles, where subsequent truck loads produce sloped layers, and least for horizontally layered piles. One of the major challenges with layered piles is the difficulty of ensuring that the geochemical characteristics of individual pathways are comparable to the bulk mixture. Physical hydrogeologic factors are thought to be one of the reasons that the Samatosum waste rock pile (see Section 4.2.1), with an estimated overall NP/AP ratio of 3, is now producing acidic seepage (Morin and Hutt, 1997b). The formation of flow paths and potential for short-circuiting is particularly important if a small amount of neutralizing rock (eg. limestone) is being mixed with a low sulphur potential acid generator. The potential for individual flow paths to not intersect sufficient limestone should be considered.

2.3.6 Operational Commitment and Monitoring

Generally, blending will require a high degree of operational control especially when the blending ratio approaches the critical NP/AP for the site. A blending operation would require some of the following components:

- Pre-mining evaluation of the feasibility of blending using a block model-type approach to waste characterization;
- In-mine pre-blast characterization of waste blocks to determine destination in the waste dump;
- One or more mine employees dedicated to compilation of waste characterization data and in-mine scheduling of waste management;
- Means of directing haul trucks to specific locations in the dump;
- Post-construction or post-depositional chemical monitoring;
- Confirmation that the blending criteria is being met or exceeded; and,
- Collection of drainage if the success of blending is in doubt, plus a plan for drainage treatment, if required.

The detailed planning and operational controls used for the Wismut Absetzerhalde waste rock relocation project (Hockley *et al.*, 1997) provides an excellent example for a comprehensive field control system (see Section 4.2.7).

Radio control and Geographic Positioning System (GPS) technology can allow careful control on haul truck movements and depositional locations. This approach has been proposed for Telkwa Coal based on existing technology used at other coal mines (Manalta Coal Ltd., 1994). Designation of trucks for different types of wastes, continuous logging of individual truck movements and recording of dumping locations potentially allows a comprehensive record of dump construction to be prepared. That is, the characteristics of waste placed at any location in the dump can be known. This latter feature should make regulatory compliance with permit requirements relatively easy to monitor and allows erroneous placements to be corrected by modifications to the placement plan.

Post depositional monitoring of leachate quality is also required to evaluate the performance of any mitigation measure, including blending or layering. Unfortunately, the performance of many of the historical sites which may have inadvertently blended acid generating and acid consuming waste rock cannot be evaluated because there is insufficient monitoring data to determine the composition and location of different waste materials in the pile.

3.0 METHODS FOR EVALUATING BLENDING AND LAYERING

3.1 General

This section provides a brief description of some methods that might be useful when evaluating data from sites where blending or layering of acid generating and acid consuming waste rock was either used or is being considered. The evaluation methods are essentially the same as methods commonly used to evaluate ARD prediction test work and water quality data from all mine sites, with an emphasis on understanding how the weathering and leaching behaviour of the blend differs from the weathering and leaching behaviour of its component parts. ABA and kinetic test procedures are documented in MEND Reports 1.16.1b (Coastech Research Inc., 1991), with recent evaluation methods documented in "Guidelines and Recommended Methods for the Prediction of Metal Leaching and Acid Rock Drainage at Minesites in British Columbia" (Price, 1997; Price *et al.*, 1997a, b; Price and Kwong, 1997) and "Environmental Chemistry of Minesite Drainage: Practical Theory and Case Studies" (Morin and Hutt, 1997a).

The basic components of the evaluation process should include the following steps:

1) site characterization;
2) materials characterization;
3) kinetic testing of the individual units (controls) and blend;
4) design and construction of rock dumps;
5) operational control and monitoring; and,
6) post depositional monitoring.

In the following discussion, the rational for examining each of these components is described, followed by a check-list of questions used to evaluate the data, and finally, by some of the specific calculation or evaluation methods that can be used. It should be emphasized that each site is unique; thus each review should be based on a thorough understanding of the site-specific issues. A proposed template for the data that should be collected at each site and suggestions for compiling and evaluating the data is presented in Section 7.

3.2 Site Characterization

The site characterization is one of the most important steps in any evaluation process. The site characterization allows the designer or reviewer to develop an overview of the physical and climatic conditions at the site, the general geology, the mining and waste management plan and to obtain a sense of whether blending and layering would be appropriate for the site.

Considerations/Questions:

- Describe location, climate and hydrogeological conditions at the site;
- Describe the physical layout present at, or planned for, the mine;
- Describe geological/geochemical conditions at the site; and,
- Are there suitable materials for blending or layering?

3.3 Material Characterization

The characteristics and proportions of each of the major rock types will significantly influence whether a non-acid drainage generating blend can be achieved at a site. Usually the term 'rock type' implies a group of materials related by its geological characteristics. This is appropriate when there are clear correlations between the lithology and the acid base accounting results. For example, at the Samatosum site, most of the acid consuming rock was mafic volcanic rock, and most of the acid generating rock was of meta-sedimentary origin. These two groupings were used in all the subsequent planning and management steps (Denholm and Hallam, 1991a, b). In some cases, there are no correlations with geology, and specific geochemical tests will be required to identify the rock types. For example, at the Wismut site, paste pH and net acid potential tests (NAP tests) were used to classify the materials into waste management units because it was impossible to separate out individual geological units in the mixed waste being re-mined (Hockley *et al.*, 1997).

For each of the rock units, the distribution and numeric mean of the standard acid base accounting (ABA) data should be described using descriptive statistical methods (percentiles, histograms etc.). The relative proportions of each rock unit should be determined prior to mining, often by recording this data directly into the mine planning software.

Other characteristics that are relevant to evaluation of blending would include the following: mineralogy, elemental composition, particle size of the rock after blasting and the susceptibility of the rock to physical breakdown. Interactions between rock types can be addressed by kinetic tests (Section 3.4).

Considerations/Questions:

- What are the important rock units?
- What are the relative volumes of each?
- What are the mineralogical, geochemical and physical characteristics of these materials?
- How are the rock units classified (geological, geochemical tests and/or spatial distribution)?
- How variable is the NP and AP within each rock type?
- What is the average NP and AP within a unit?
- What is the potential overall NP/AP ratio of a blend?
- What are the potential NP/AP ratios of blends in different areas of the waste rock pile, estimated on the basis of mine plans and logistics?
- What are the likely composition and variability in NP and AP along any particular drainage flow path?

Calculations and Considerations:

As suggested by the above questions, the overall NP/AP ratio of a blend provides only a first approximation of whether waste rock blending may be a suitable mitigation option at a particular site. The makeup of blends in local areas of the pile and along drainage flow paths must also be considered.

As a preliminary assessment of whether blending may be feasible at a particular site, the overall ratio of a blend is calculated by weighting the average NP and average AP of each of the components of the blend by the relative proportions of material in the blend. A spreadsheet provides a useful framework for the calculations because the calculations can be updated as new information on the relative volumes of material or additional ABA data become available. A suggested format for a preliminary spreadsheet is shown in Table 5. The spreadsheet can also be expanded to address assessments of blends in localized areas of the pile as the mine plan develops.

Table 5: Example of Calculation Spreadsheet to Determine NP/AP Ratios

Rock Type	Mass (tonnes)	% of total	mean NP	mean AP	NNP	mean NP/ mean AP	Total NP (tonnes $CaCO_3$)	Total AP (tonnes $CaCO_3$)
A	1×10^6	16.7*	20	230	-210	0.087	20,000**	230,000**
B	3×10^6	50.0	16	25	-9	0.64	48,000	75,000
C	2×10^6	33.3	360	65	295	5.54	720,000	130,000
Overall	6×10^6	100	131.2***	72.6***	58.6***	1.81****	788,000	435,000

Notes: * calculated by dividing the mass of rock A by the total mass

** calculated by multiplying mass of rock by NP or AP (tonnes/1000 tonne) and dividing by 1000 to correct units

*** Calculated by weighting the average NP, AP or NNP from each rock unit by the mass, or by dividing the total NP, AP or NNP by the total mass and multiplying by 1000 to correct units.

**** Calculated from weighted overall NP and weighted overall AP, or total NP/total AP.

3.4 Kinetic Testing of the Individual Rock Types and Blends

Kinetic tests often provide the only data from which to estimate reaction rates and evaluate how blended rocks will behave. The most common source of data are laboratory columns or humidity cells where the test material is placed in a weathering chamber and periodically flushed thoroughly with water to remove the oxidation products. The chemistry and recovered volume of the flushed effluent is monitored on a regular basis and used to estimate the oxidation rates, AP and NP depletion rates, and metal production rates. Details of kinetic test procedures have been documented elsewhere (Coastech Research Inc., 1991; Sobek *et al.*, 1978).

Testing can be carried out in single vessels (cells or columns), or a linked series of vessels where the leachate of one vessel holding one rock type acts as the feed solution for the subsequent vessel holding a second rock type. While a single vessel provides a reasonable indication of the behaviour of well blended materials, it may be advantageous to use vessels in series for testing layering so that the incremental behaviour of the layers can be determined.

One disadvantage is that vessels in series do not address the hydrogeological interactions between the rock types. Control tests should be carried out on the individual rock units to allow comparisons between the kinetic testing results of the blends and the individual components of the blend. The small scale humidity cell test which typically uses 1 kg of finely crushed rock may not be as useful for evaluating the blended materials as the larger or linked test vessels because single cells are mixed on such a small scale that the mixture may effectively become a single rock type.

Large scale or pilot scale kinetic tests carried out in the field are often used to supplement the laboratory tests. Field tests may range from simple lysimeters carried out in 40 gallon drums, to 20 tonne test pads. Usually rain and snow are the only source of water. These tests lack many of the controls available in laboratory tests, particularly the ability to flush out oxidation products so that accurate oxidation rates can be determined. However, the larger particles sizes, more realistic blending conditions, and flushing regimes can be important advantages. The extrapolation of results from field tests to full-scale piles is generally more realistic than direct extrapolations from laboratory to the field, since oxidation products are typically stored in full-scale piles. Monitoring of uncontrolled parameters, such as precipitation and temperature, is recommended.

Numerous documents are available which outline methods to evaluate kinetic testing data and predict the drainage water quality at minesites. Two recent sources are the "Guidelines and Recommended Methods for the Prediction of Metal Leaching and Acid Rock Drainage at Minesites in British Columbia" (Price, 1997; also Price *et al.*, 1997 a, b; and Price and Kwong, 1997), and "Environmental Geochemistry of Minesite Drainage: Practical Theory and Case Studies, (Morin and Hutt, 1997a).

An important aspect of evaluating blended or layered tests is to look at how the blended results compare to unblended control tests. Comparisons should be made between the pH, sulphate production rates, NP and sulphide depletion times, and metal precipitation and drainage concentrations. It is useful to compute the rates that would result if the effluent from each of the individual composite components were mixed, and compare this to the actual rates measured in the blended material. If the results do not differ significantly, the blend probably does not influence sulphide oxidation rates.

Considerations/Questions:

- Are the materials in the tests representative of the variety of materials at the site?
- How are the tests operated?
- How do the operating conditions differ from the field?
- How does the effluent pH in the blended/layered test vessel(s) compare to the effluent pH in the control vessel(s)?
- What are the normalized sulphate depletion rates (mg SO_4/kg sample/week) in each test?

- Can the sulphate depletion rates be used to reasonably represent sulphide oxidation rates? (Do the tests allow oxidation products to be stored within the test vessel?)
- What are the normalized NP depletion rates (g $CaCO_3$ /kg/week)?
- Given these current leach rates, will AP or NP last longer?
- How do the sulphide oxidation and NP depletion rates of the blended/layered tests compare to the rates of the components of the composite?
- How does the rate of acid generation effect the rate of NP depletion and the effective NP?
- Are sulphate concentrations limited by gypsum saturation?
- Does blending reduce metal levels in the effluent?
- How might scale effect the results?
- For field tests: What are the effects of heavy or prolonged flushing or prolonged dry periods on the results?

Calculations and Considerations:

Sulphide Oxidation Rates and NP Depletion:

Interpretation of kinetic test data often includes the calculation of sulphate production and NP depletion rates, as well as an estimate of time to depletion of both sulphides and neutralization potential. These are briefly described below.

Sulphate production rates are calculated by multiplying the weekly sulphate concentration by the flushing rate and dividing by the total sample mass:

$$SO_4 \left(\frac{mg}{L}\right) * L \text{ water recovered} \div kg \text{ sample} = SO_4 \text{ production } \left(\frac{mg}{kg*week}\right)$$

Sulphate depletion rates are expected to be reasonable estimates of sulphide oxidation rates if oxidation products are continuously flushed from the test vessel, or corrections are made for stored oxidation products at the end of the test period. The time to sulphide depletion (weeks) can be calculated by dividing the total sulphide content (in

SO_4 equivalent units) by the weekly oxidation rate. Usually the oxidation rate is selected when the test reaches a steady state, i.e. relatively consistent sulphate loading rate over a period of greater than four weeks.

Assuming NP for the tested material is provided by calcite and/or dolomite, and in the absence of gypsum in the original sample or as a stored oxidation product in the test vessel, NP depletion may be calculated from the Ca and Mg concentrations in the test effluent. The weekly NP depletion rate would be calculated as follows:

$$[Ca\ (\frac{mg}{L})\times(\frac{1}{40}\times\frac{moles\ Ca}{g})+Mg\ (\frac{mg}{L})\times(\frac{1}{24.3}\times\frac{moles\ Mg}{g})]\div 1000\ (\frac{mg}{g}) = \frac{moles\ CaCO_3\ e}{L\cdot week}$$

$$\frac{moles\ CaCO_3\ eq}{L\cdot week}\times\frac{100\ g\ CaCO_3\ eq}{mole} = NP\ depletion\ rate\ \frac{g\ CaCO_3\ eq}{L\cdot week}$$

$$\frac{g\ CaCO_3\ eq}{L\cdot week}\times\frac{L\ recovered}{kg\ sample} = NP\ depletion\ rate\ (\frac{g\ CaCO_3\ eq}{kg\cdot week})$$

The time to NP depletion (weeks) would then be calculated by dividing the total NP by the NP depletion rate. The NP depletion rate is generally calculated from Ca and Mg concentrations when they reach relatively steady valued for four weeks or more.

Other equations would be used to calculate NP depletion rates for test materials where alumino-silicate or other minerals provide the neutralization potential.

<u>Gypsum Saturation</u>

If Ca and SO_4 concentrations in the test effluent are close to the saturation point of gypsum, gypsum may be forming internally in the test as a by-product of neutralization. In this case, both the sulphide oxidation rates and NP depletion rates may be underestimated. If gypsum saturation is suspected, verification should be undertaken using a chemical equilibrium model (for example, MINTEQA2). In addition, visual examination and a thorough wash of the tested material should be completed at the end of the test to check for stored oxidation products. Residual oxidation products can then be added to the estimated sulphide oxidation and NP depletion rates.

3.5 Design and Construction of the Rock Piles

Important considerations in the design and construction of a blended or layered waste rock pile are the proportions and spatial distribution of materials in the pile, the sequence of material placement, and the degree of mixing or thickness of the layers.

Considerations/Questions:

- What are the percentages of each rock type at the site?
- How variable are the NP and AP within each rock type?
- What are the potential NP/AP ratios in local areas of the pile?
- What is the overall NP/AP ratio weighted by rock type?
- Are there rock units representing small quantities that contribute significantly to the NP or AP of the blend?
- How easy will it be to achieve a uniform blend of these materials throughout the pile, considering the mine plan and logistics.?
- Are there any rock units that could be effectively segregated to improve the NP/AP ratio of the remaining blend?
- What are the likely NP and AP composition and variability along potential drainage flow paths?
- How much mixing or what layer thickness will be sufficient to ensure neutralization reactions will occur inside the dump?
- How will the materials be mixed?
- Will secondary handling be required to ensure that an adequate mix is obtained?

3.6 Operational Control and Monitoring

Any data on the methods used to identify and handle the waste rock in the dumps is useful for understanding the degree of blending that is actually achieved in the dumps. Without some operational data, there is no way to determine how accurately materials are blended or layered in the dumps. In such cases, forensic sampling may be required to estimate the distribution of material and/or what intimacy exists between the acid consuming and acid generating materials. However, the quality of information derived from forensic drilling can be quite poor. Operational characterization and/or forensic data that defines the achieved level of blending has been the most difficult type of data to obtain in this review of case studies.

Considerations/Questions:

- How is the waste rock identified and how is this communicated to the haul truck operators?
- What number of field samples and what statistical assessments are required during construction to verify the desired NP/AP ratio is achieved throughout the pile?
- How is the data recorded?
- How consistently have blending targets been achieved?

3.7 Post Depositional Monitoring

Water quality data is the most direct indication of the success or failure of any mitigation technique. Where possible, seepage emerging directly from the toe of the dump should be monitored because it provides the first indication of any change. Potential drainage sources should be determined, and provision made for adequate collection. Monitoring data should be compared to any predictions of water quality made from the laboratory or field testing programs. If there are significant differences, the reasons for the difference should be identified, and the prediction techniques should be improved for future programs.

Considerations/Questions:

- How will waste rock pile drainage be collected?
- Are the seeps directly associated with the waste rock dumps or are they influenced by other site components?
- Are there any trends over time showing seasonal or long term changes in water quality?
- Are concentrations limited by secondary mineral solubility constraints?
- Where are the secondary minerals likely to precipitate in the dump?
- Is the pH controlled by the blended or layered design?
- Are sulphate levels controlled by the blended or layered design?
- Are metal levels controlled by the blended or layered design?

4.0 CASE STUDIES

4.1 Overview

Case studies of coal and metal mines which had evaluated and/or implemented blending or layering as potential mitigation techniques were sought as part of this investigation. Much greater success was met in obtaining detailed metal mine case studies as compared to coal mine case studies.

This section of the report provides a brief overview of each of the case studies and presents the key findings from them. Detailed reviews for metal mines are provided in Appendices A through G, and summarized in Sections 4.2.1 through 4.2.9. Detailed case studies for coal mines, similar in nature to the case studies for metal mines, were not obtained. Consequently, summaries of selected coal mine case studies from the published literature have compiled in Sections 4.3.1 through 4.3.7. A listing of the case studies showing key aspects is provided in Table 6.

Where available, both laboratory and field data were compiled, including rock types, variability of static acid-base accounting (ABA) parameters within each rock type, kinetic test results, field construction methods, construction controls, and leachate quality. A key objective was to report on the degree to which the various waste materials were physically mixed, and the variability in mixing success throughout the waste rock piles.

Table 6: Summary of Case Studies

	Appendix A Samatosum	Appendix B Kutcho Creek	Appendix C Stratmat	Appendix D Cinola	Appendix E Eskay Creek	Appendix F Windy Craggy	Appendix G Wismut Project	Duluth Complex	Mt. Milligan	Rawhide Mine	Upshur County Site	Rose and Daub (1994)	Evans and Rose (1995)	Kauffman Mine, Phase I	High Power Mountain	Coaltrain Project	Telkwa
Control Strategy:																	
Blending			●	●		●		●	●		●	●					●
Layering	●	●	●	●	●	●	●				●		●	●			
Oxygen consuming cover							●										
Materials:																	
Waste rock	●	●			●	●	●		●		●					●	●
Off-site amendments (eg. limestone)			●	●			●	●	●		●	●	●	●			
Kinetic Testing:																	
Laboratory	●	●	●	●	●	●	●	●				●					
Field		●	●								●		●				
NP/AP ratio of tests																	
< 1	●	●	●	●				●			●						
1 to 2	●	●		●	●	●		●			●	●	●	●	●		
2 to 3						●					●						
> 3	●										●					●	
Operational Control and Monitoring Plan	●						●		●	●				●			
Full Scale Implementation	●	●	●				●			●				●	●	●	
Field Monitoring Data	●	●	●								●			●			

4.2 Metal Mining Case Studies

4.2.1 Samatosum Mine, B.C., Canada

The Samatosum Mine is located in south-central British Columbia, approximately 80 km north of Kamloops. The ore is a stratabound quartz-carbonate vein deposit, hosted by volcanic related mafic pyroclastics and metasediments. The mine operated from May 1989 to October 1992 producing approximately 565,700 tonnes of ore and 9.1 million tonnes of waste rock.

Based on acid base accounting tests completed prior to mining, the waste rock could be classified as potentially acid generating or acid consuming according to lithology. The acid consuming rock were primarily mafic pyroclastics (MAF) located in the hanging wall of the deposit, with a mean AP and NP of 73 and 377 kg $CaCO_3$/tonne, respectively. The MAF material was predicted to form 58% of the total waste rock volume. The potentially acid generating rocks (PAG) were of sedimentary origin, and were located primarily in the footwall of the deposit, and were predicted to be 42% of the waste material. The mean AP and NP of the PAG mixture were 95 and 50 kg $CaCO_3$/tonne, respectively (Denholm and Hallam, 1991a, b). In hindsight, the approximately equal volumes of relatively reactive acid generating and acid consuming rock types suggests a potential for significant chemical heterogeneities and formation of hotspots. However the overall combined NP/AP ratio of the waste rock, estimated at 3.1, was considered sufficiently conservative to address those concerns.

The control strategy planned for the waste dumps was to place the PAG rock between layers of acid consuming mafic pyroclastics. It was expected that alkalinity produced in the MAF layers would percolate downward through the PAG layers, and maintain neutral conditions throughout the dump. This was expected to limit the rate of sulphide oxidation and therefore prevent acid drainage generation and metal leaching (Denholm and Hallam, 1991a). The layered design is shown in Figure 2.

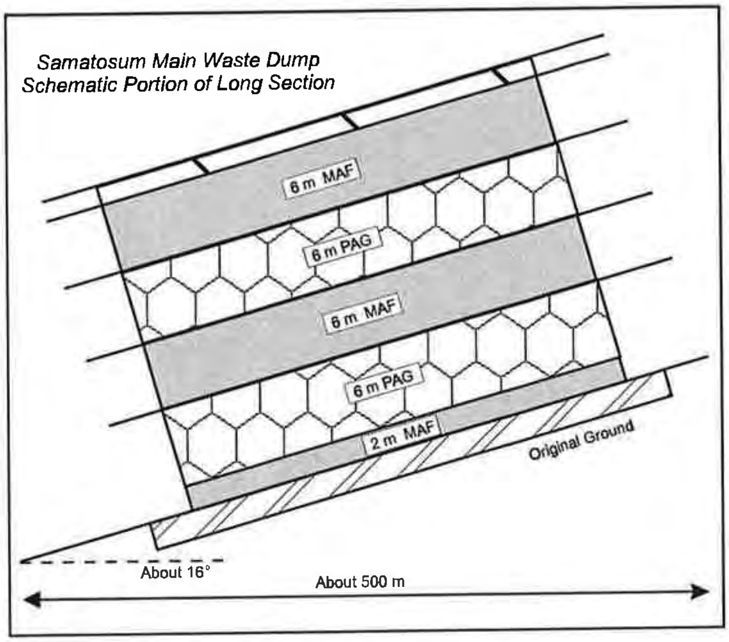


Figure 2 **Schematic of Layered Design for the Samatosum Waste Pile** (Denholm and Hallam, 1991a)

Prior to mining, a series of column tests were initiated to test the layered control strategy (Denholm and Hallam, 1991a, b). Three control tests and three layered configurations were tested for up to 286 weeks. The layered columns had NP/AP ratios of about 1. Two of the tests were dismantled in 1994. The remainder were dismantled in 1996.

The control tests confirmed the potential for acid generation in the PAG materials and demonstrated the acid consuming properties of the mafic pyroclastics. None of the layered columns produced acidic leachate; however, it was not clear whether this condition could have been maintained indefinitely. Calculations presented in Hallam Knight and Piesold Ltd. (1992, 1994) indicated that alkalinity depletion would have occurred *after* the sulphides were depleted, while more recent calculations from those laboratory results and additional analyses by Morin and Hutt (1997b) indicated that the NP would have been consumed *prior* to sulphide depletion. Sulphate production rates in the layered columns were only slightly lower than sulphate production rates in the pure PAG columns indicating that sulphide oxidation rates were not significantly retarded by the MAF acid consuming layer. Rather, it appeared that the leachate from the MAF portion of the column diluted and neutralized the leachate produced in the PAG portion of the column.

The waste rock dump was constructed in accordance with the original layering design and was completed in November, 1992 (Minnova Inc., 1992a, b). Monitoring stations at the toe of the dump have shown impending signs of net acidity since 1993, with increasing sulphate concentrations, decreasing alkalinity and increasing concentrations of manganese and zinc. In the spring of 1996, acidic pH values were measured for short periods of time at both stations. A treatment plant was installed to treat seepage from the waste dumps and pit area in the spring of 1997, prior to the onset of runoff.

In summary:

- In the laboratory columns, layering of MAF and PAG on the order of 0.2 to 1.0 m did not appear to affect reaction rates or geochemical behaviour within individual layers.

- Layering in the dump on the order of 2 to 6 m at an approximate overall NP/AP ratio of 3 did not prevent acidic drainage or metal leaching. It has been suggested (Morin and Hutt, 1997b) that preferential flow channelling through the PAG layers as compared to the MAF did not allow the available neutralization in the dumps to be sufficiently utilized to neutralize the produced acid.

- The importance of considering field conditions and physical factors, such as layer thickness and hydrogeology, when interpreting laboratory data and extrapolating the results to the field is emphasized.

- Control of leachate acidity may be insufficient to adequately control metal leaching.

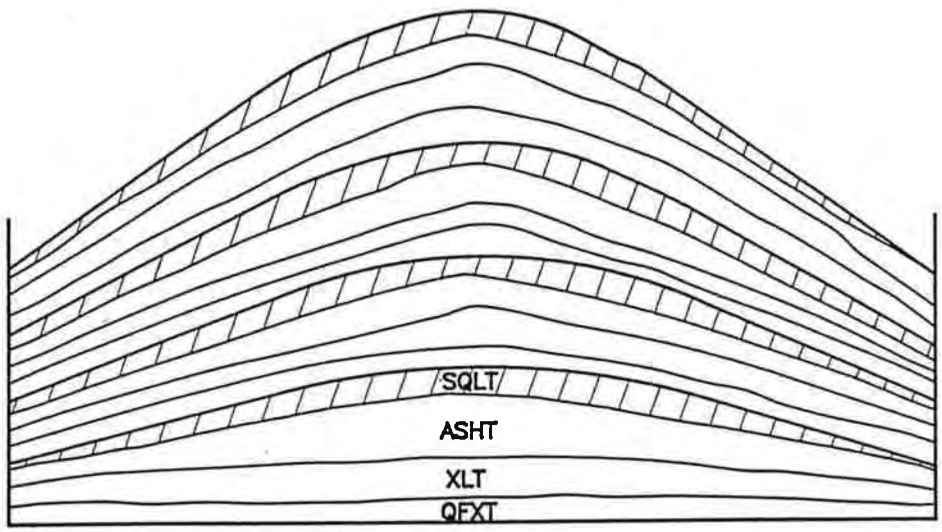
Details on the Samatosum Mine are provided in Appendix A.

4.2.2 Kutcho Creek Project, B.C., Canada

The Kutcho Creek property is located approximately 110 km east of Dease Lake, British Columbia. The ore deposit is a polymetallic, massive sulphide deposit with possible reserves of 13.9 million tonnes of ore at a stripping ratio of approximately 6.6 tonnes of waste per tonne of ore. Detailed feasibility studies were carried out in 1985. The environmental studies included acid base accounting tests, humidity cell tests and a specific testing program to evaluate blending of acid consuming and acid producing rocks (Rescan, 1992, 1991, 1989). Waste rock from the hanging wall of the deposit is considered net acid consuming while waste rock from the footwall is considered strongly acid generating.

Blending was tested at two different scales: small scale humidity cell tests and large scale (20 tonne) field test plots. Both series of tests were intended to simulate the waste rock mixtures expected during the pre-production phase of mining and the fifth year of mining. These blends were composed of layers of three types of the hanging wall rock (chert/mafic ash tuff with a mean NNP of +64 kg $CaCO_3$ /tonne, crystal lapilli tuff with a mean NNP of +129 kg $CaCO_3$ /tonne, and quartz feldspar crystal tuff with a mean NNP of +79 kg $CaCO_3$ /tonne), and one layer of footwall rock (sericite quartz lapilli tuff with a mean NNP of - 473 kg $CaCO_3$ /tonne), with 4 repetitions of this sequence placed in relatively thin layers in both the humidity cells (<1 cm layers) and in the field test plots (~10 cm layers). A schematic illustration of the layered design is shown in Figure 3. The overall NP/AP ratio of the pre-production test and 5-year test were estimated to be 1.1 and 0.6, respectively. These NP/AP ratio's were substantially less than the NP/AP ratio of 1.7 estimated for the waste rock to be produced from the entire mine. The 5-year production blend was tested in duplicate, with a soil cover placed over one of the replicates.

The blended small scale humidity cell test leachate had neutral pH and moderate sulphate levels throughout the 20-week testing period. Sulphate concentrations in these cells were higher than sulphate concentrations in pure hanging wall samples, but much lower than sulphate concentrations in the pure footwall samples. In these short term tests, blending was effective in delaying the release of acidic effluent as compared to a pure footwall samples. However, calculations performed by MEM Inc. suggested the rate of sulphide oxidation in the footwall portion was not reduced by the blending. Rather the sulphate concentrations in the blended cell were roughly equivalent to the concentrations expected if four discrete cells were operated and the effluent streams were mixed.



Each layer = one bucket lift = 1.8 tonnes

Lithology:

Footwall

SQLT- Sericite quartz lapilli tuff

Hanging Wall

ASHT - Chert mafic ash tuff
XLT - Crystal lapilli tuff
QFXT - Quartz feldspar crystal tuff

Figure 3 Kutcho Creek Field Test Plot Design Showing Layers (Rescan, 1992)

The three 20 tonne field tests consisted of one layered test with an estimated NP/AP ratio of 1.09, and two with estimated NP/AP ratios of 0.63, one with a 6 inch soil cover, and one without. The results showed higher metal and sulphate concentrations, but possibly lower total loads, than the comparable laboratory tests. Estimates of total loads were effected by the inability to collect all the drainage from the field test pads.

There was more variability in effluent water quality from the test pads, and hydrological effects related to the natural flushing regime and larger scale were evident. The covered test pad produced acidic effluent within the first year of operation. It was speculated that the cover reduced infiltration such that sulphide oxidation products were concentrated in the leachate, although difficulties in accurately measuring the flow at this remote site limited this interpretation somewhat (Rescan, 1992). Alternate explanations could be that: a) the cover induced preferential flow through the acidic rock, or b) decreased infiltration rates allowed acid generating material to oxidize longer before water rinsed reaction products into alkaline areas, which were unable to neutralize the higher acid load. The uncovered 5-year and the pre-production tests both showed incipient signs of acid generation, with periodic acidic leachate observed, usually following periods of high or prolonged precipitation. Long term projections indicate that the test pads would have produced acidic seepage over time.

In summary:

- In the short term blended humidity cell tests, blending appears to have delayed the release of acidic effluent as compared to the potentially acid generating (PAG) footwall material. However, blending did not reduce the rate of sulphide oxidation that would have been produced by each individual component material, even though the layers were only about 1 cm thick.

- Over the two year testing period, the field test pads all showed incipient or pervasive signs of acid drainage generation. The covered 5-year blend had the worst performance, with consistently low pH observed during the first year of monitoring. The uncovered pre-production blend with an NP/AP ratio of 1.1 performed slightly better than the 5-year blend with an NP/AP ratio of 0.6, however long term projections indicate that all test pads would have produced acidic seepage over time.

- At the tested NP/AP ratios of less than 1.1, blending was not an appropriate mitigation strategy for long term prevention of ARD at this site. However, the projected overall NP/AP ratio for the Kutcho Creek waste rock was estimated to be higher than any of the tests (1.7), thus this mitigation option may have more merit for this site than the results obtained from this testing program indicate.

- Separate economic analysis of this disposal option (Rescan, 1992) indicated that the costs associated with blending large amounts of waste material in a full-scale waste rock dump would be high, as compared to other disposal options.

Details on the Kutcho Creek Project are provided in Appendix B.

4.2.3 Stratmat Deposit, Heath Steele, N.B., Canada

The Stratmat site is located on the Heath Steele Mine property north of Newcastle, New Brunswick. The deposit was mined from 1989 to 1993. Waste rock associated with the mining activity was strongly acid generating with approximately 19% pyrite, trace sphalerite, galena and chalcopyrite. One of the control options studied at this site was to blend or layer the waste rock with limestone to limit the production of acidic effluent, and potentially delay or reduce acid generation, prior to possible future disposal in one of the mined out open pits. This review summarized the results of two of the studies on blending and layering limestone with Stratmat waste rock.

The first study on limestone addition involved field monitoring data from a 1.5 million tonne waste rock dump constructed between 1989 and 1991. Limestone was added to the top of each 3 metre lift at a rate of about 10 tonnes limestone per 1000 tonnes of rock (1% limestone). limestone addition was partly based on bench-scale studies, reported in Sheremata *et al.* (1991), which suggested that approximately 10 kg/tonne (1%) of limestone would be sufficient to limit acid generation (Heath Steele and Noranda Technology, 1994). These measures proved to be unsuccessful and acid seepage was detected at the toe of the Stratmat waste rock pile during the spring of

1991 (Heath Steele and Noranda Technology, 1994). Limestone addition was halted at that time

The second study was carried out by the Noranda Technology Centre (NTC), with financial support from the Centre de Reserches Minerales (CRM) and from the MEND program (Payant and Yanful, 1997). The objective of this study was to evaluate the relative effectiveness of various ARD control techniques, including blending with limestone. Waste rock from Stratmat was one of the two test materials used in the experiments. The blending experiments included a series of 20 kg column tests and 170 kg field test lysimeters. The field and laboratory tests were carried out in triplicate, with limestone blends of 1 and 3% and a control test with no limestone. The laboratory tests were run for 154 weeks and the field tests for 125 weeks.

Both the testing programs and the field studies showed the inadequacy of the limestone additives in preventing ARD. However, the blended column and lysimeter tests indicated that limestone addition would be a viable option for delaying the release of acidic effluent and reducing the rate of sulphide oxidation, provided there was sufficient mixing of the waste rock and limestone. An inadequate overall NP/AP ratio appeared to be the main reason acidic effluent was not prevented in the tests. Calculations indicated that a blend with 3% limestone by mass would have an NNP of - 248 kg $CaCO_3$ /tonne, and an NP/AP ratio of 0.16. Thus the test material was well below the NP/AP $ratio_{crit}$ (1 to 2 range) that would theoretically prevent acidic effluent from a homogenous blend.

The full-scale test with limestone layers did not perform as well as predicted by the laboratory studies in delaying the release of acidic effluent. The inability of the limestone to delay the production of acidic effluent was attributed to absorption and oxidation of iron and manganese on limestone surfaces in contact with the acid generated in the untreated areas of the pile, which was thought to render the limestone non-reactive (Heath Steele and Noranda Technology, 1994). This ineffectiveness was assisted by the inadequate physical contact between the thin (15 cm) limestone layers and the much thicker (300 cm) waste rock lifts.

In summary:

- Limestone addition in laboratory tests delayed the release of acidic effluent. The delay was proportional to the amount of limestone added.

- Sulphide oxidation rates, as measured by sulphate concentrations appeared in the well-mixed column and lysimeter tests, appeared to be slowed by the presence of limestone, both before and after the onset of acid drainage generation. This apparent reduction in sulphide oxidation rates is in agreement with data from the Cinola project (Section 4.2.4), but not Samotosum (Section 4.2.1) or Kutcho Creek (Section 4.2.2). Interpretations of the Stratmat data were limited by the lack of data on gypsum saturation and the potential for sulphate precipitation within the test vessels.

- pH levels in most of the replicate column tests and field lysimeters decreased to less than 3 during the three years of testing, indicating this was not a viable long term prevention measure for acidic drainage under the tested conditions. It was not clear whether the limestone had been consumed, washed out or made unavailable from the data presented.

- 15 cm layers of limestone placed between 3 metre lifts of potentially acid generating waste rock did not prevent the release of acidic effluent.

Details on the Stramat Project are provided in Appendix C.

4.2.4 Cinola Gold Project, Queen Charlotte Islands, B.C.

The Cinola Gold Deposit is located on Graham Island, in the northern part of the Queen Charlotte Islands, British Columbia. The deposit is an epithermal Carlin-type gold deposit. Seasonal weather patterns are typical of coastal Western Canada, with 1700 to 2200 mm of precipitation occurring as rain during October to March. Detailed feasibility studies were initiated in 1986 by City Resources (Canada) Inc. (Day, 1994; Norecol, Dames & Moore, 1994). The potential for acid generation in the waste rock was examined as part of these studies, and then continued under MEND and the BC AMD Task Force in 1990 (MEND Project 1.19.1) after City Resources decided not to pursue the project.

The environmental studies included a series of column tests which examined blending and layering of limestone within the acid generating waste rock. The five column tests included a control test, blends of 6.6%, 3.3% and 0.84% limestone, and a column with alternating layers of the 6.6% blend (1 cm thick) and the 0.84% blend (10 cm thick) giving an overall limestone content of 1.2%. The only test with an NP/AP ratio of greater than 1 was the 6.6% limestone blend.

Results from the column testing program indicated that limestone blends with NP/AP ratios less than 1 would not be sufficient to prevent acid drainage in the long term, but that blending at NP/AP ratios of about 1 might delay the release of acidic effluent for several years. Depletion calculations indicated that an NP/AP $ratio_{crit}$ greater than 2 would be required to prevent acidic drainage from a fully blended mixture of this rock. The layered column and the 0.84% limestone blend gave very similar results, indicating that the extra limestone provided by 6.6% limestone layers did not significantly delay the release of acidic effluent. This finding showed the importance of intimate blending in limiting the production of acidic effluent from these wastes. The post-test mineralogical analyses completed on all the columns showed that the limestone availability was not reduced by ferric hydroxide coatings, but was fully available for reaction.

In the 0.84% and the 3.3% blended columns, the rate of sulphate release was reduced prior to depletion of the limestone and development of acidic conditions. At the concentrations measured, the reduction in sulphate release appeared to be a function of the neutral pH conditions limiting sulphide oxidation rates, rather than the solubility of gypsum ($CaSO_4$) limiting the rate at which sulphate was flushed from the column (Stephen Day, *pers. comm.*, 1997).

In summary:

- Intimate mixing of limestone with waste rock was successful in delaying, but not preventing the release of acidic effluent at the laboratory scale. The delay appeared to be a combination of 1) controls on the rate of sulphide oxidation, and 2) straightforward neutralization of acidity generated by the reduced oxidation rate.

- Based on molar ratios of SO_4 to Ca measurement when leachates were pH neutral or stable, NP/AP ratios of at least 2 would be required to prevents the release of acidic effluent from blended waste rock and finely crushed limestone in the long term.

- Metal concentrations in the column effluent may have been limited by the rate of sulphide oxidation, as well as by pH control.

- In the blended columns, the crushed limestone was fully available for reaction, and was not blinded or armoured by iron-precipitates.

- Additional limestone contained in 1 cm thick layers in the layered column did not appreciably delay or mitigate acid production as compared to the column with the same base blend. This may be due to the limited physical contact between the additional NP in the thin 1 cm layers and sulphides in alternating 10 cm thick layers. This suggests that alternating layers as little as 10 cm in thickness would not significantly reduce sulphide oxidation rates.

Details on the Cinola Project are provided in Appendix D.

4.2.5 Eskay Creek Mine

The Eskay Creek Mine is located approximately 83 km north of Stewart, B.C. This gold and silver ore deposit is part of a stratiform bound volcanogenic massive sulphide deposit located within the early to middle Jurassic Hazelton Group. Mining began at this site in 1994, with current probable reserves of 1.27 million tonnes at 59.3 grams per tonne Au and 2719 grams per tonne Ag (Sharon Meyer, *pers. comm.*, 1997). Underground cut and fill methods are used to extract the ore. It is anticipated that approximately 380,000 tonnes of waste rock will be produced during the first 10 years of mining (Stewart *et al.*, 1994).

In the early planning stages of this project, it was thought that acid consuming andesite volcanics would comprise a significant portion of this waste rock. A kinetic testing program was initiated to test whether andesite could be used as either an acid consuming base layer or an alkaline cap to control acid rock drainage site (T.W. Higgs and Associates, 1993).

The kinetic testing program is illustrated in Figure 4. The tests consisted of two sets of columns in series. Test #1 consisted of a potentially acid generating column of material followed by a potentially acid consuming column (the acid consuming base design), while Test #2 consisted of an acid consuming column of material overlying an acid generating column (the alkaline cap design). Both of the tests had overall NP/AP ratios of 1.2.

The results of the testing program indicated that the acid consuming base layer (Test #1) would likely provide effective long term mitigation of acidity and metals produced in the overlying acid generating materials, while the acid consuming cap layer would do little to control the rate of sulphide oxidation or metal leaching from underlying acid producing materials. There was a moderate increase in the pH and alkalinity levels and a substantial (98%) reduction in zinc concentrations as leachate from the acid generating column passed through the acid consuming column in Test #1.

Subsequent revisions to the mining plan have substantially reduced the estimated volume of acid consuming andesite such that layering is no longer considered a viable option for ARD control. Instead, the mine is using subaqueous disposal of the acid generating waste rock (Stewart *et al.*, 1994).

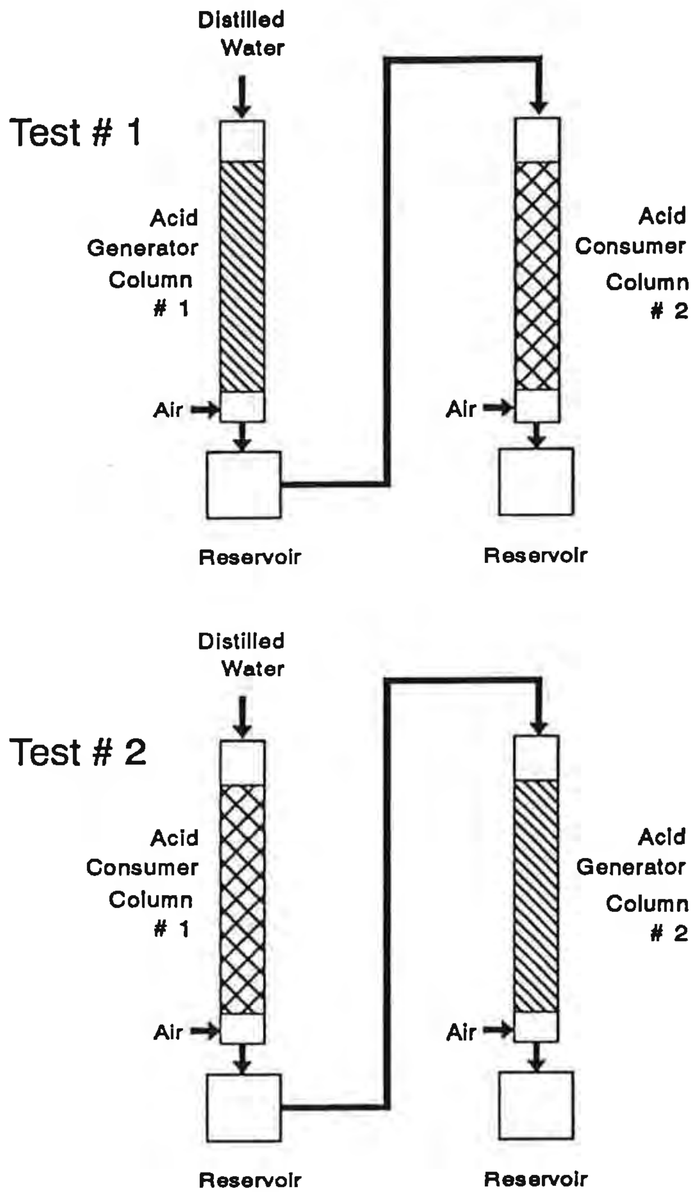


Figure 4 Eskay Creek Mine Column Testing Program
(T.W.Higgs and Associates, 1993)

In summary:

- The laboratory test results suggested that net acid consuming layers as thick as 1 metre could have effectively mitigated seepage from 1 m of overlying acid generating rocks.

- Depletion calculations (uncorrected for oxidation products potentially stored in the columns) indicated that the net acid consuming base layer (Test #1) would have provided long term neutralization of generated acidity provided the relative rates of sulphide and NP depletion did not change substantially when the upper column eventually began to produce net acidity.

- In contrast, the alkaline cap design (Test #2) did not appear to be a viable option even for short term control of sulphide oxidation and metal leaching. Leachate from the net acid consuming cap had high alkalinity and pH, but after passing through the underlying net acid generating materials, the leachate quality was actually worse than leachate from the net acid generating control sample.

- Incremental calculations should be used to evaluate the kinetic test results and estimate depletion times for the individual columns in the tests since this would provide better data on the kinetic behaviour of the materials and the resulting influence on waste rock layering designs.

Details on Eskay Creek Mine are provided in Appendix E.

4.2.6 Windy Craggy Project

The Windy Craggy property is located in northwestern British Columbia, approximately 200 km southwest of Whitehorse, Yukon Territory. The mineral deposit is a Cu-Co-Au-Ag-Zn massive sulphide with possible reserves of 297 million tonnes of ore (Claridge and Downing, 1993). Detailed exploration began in 1988, and feasibility and development studies were initiated in 1990. The creation of the Tatshenshini/Alsek Wilderness Park effectively closed the site to development in 1994.

The environmental studies included approximately 1247 acid base accounting tests, 38 conventional humidity cell tests, 18 modified humidity cell tests (~50 kg), and 9 large column tests (~650 kg). The acid base accounting results were used to develop preliminary waste management plans for the project. The data was incorporated into a block model and used to estimate the proportions of net acid consuming and net acid generating materials (Downing and Giroux, 1993). Compiled data from modified humidity cell and large column tests (Norecol, Dames & Moore, Inc., undated) were used by MEM Inc. to evaluate blending of net acid generating and acid consuming rock. Data from large column tests were also used by MEM Inc. to evaluate the possibility of layering acid generating or marginally acid consuming blends over a strong acid consuming base layer.

The blending test results indicated that acid production and metal leaching could be effectively reduced if there was a sufficient ratio of acid consuming to acid generating material in the blend. This ratio varied depending on the specific characteristics of the blend materials, and the reactivity of the sulphides and NP in the blend. Blend ratios were calculated using total sulphur values to give maximum potential acid (MPA) and NP/MPA ratios. Generally, blends with an overall NP/MPA ratio of 1 would not provide long term reduction of acidic effluent generated at this site unless a very high portion (>90%) of the NP was reactive and available to buffer the pH in the neutral range. Given that a substantial portion of the carbonate described in the mineralogical reports was dolomite or siderite, this seemed unlikely. Blends with NP/MPA ratios of 2 would have provided adequate long term mitigation, and blends with NP/MPA ratios of 3 or greater were considered very likely to prevent acidic drainage. Sulphate concentrations were limited by gypsum saturation in almost all of the modified humidity cell and column tests. Consequently, it was not possible to directly compare the rates of sulphide oxidation from the different blends. However, limited data from wash tests (a heavy flush test using deionized water) indicated that the 1:1 blend may have accumulated a higher load of stored products than 2:1 blends. This would suggest that oxidation was proceeding at a faster rate in these materials, but that the products were retained. It should be emphasised that the net acid generating and acid consuming rocks in these tests were very well mixed.

The layered column tests indicated that an acid consuming layer located beneath a net acid generating layer would delay the breakthrough of acid drainage for significant periods of time, but would not prevent acid drainage in the long term unless the overall NP/MPA ratio was greater than 1. The tests indicated that the underlying acid consuming layers would be an effective long term mitigation for acid drainage in designs where the upper layer had an uncertain potential for acid generation (i.e. NP/MPA ratios of between 1 and 2). Wash tests indicate that material in the acid consuming layers had a very high proportion of stored oxidation products. There was no evidence of reduced NP availability. However, it was possible that the long term effectiveness of the acid consuming layer would have been impaired by the accumulation of secondary minerals and associated blinding or coating of the available NP.

In summary:

Blending

- The humidity cell and column tests suggested acidic effluent could be prevented if there was a significant ratio of acid consuming to net acid generating materials. Depletion calculations indicated that blends with an NP/MPA ratio of 1 would likely generate ARD in the long term, blends with NP/MPA ratios of 2 would probably prevent ARD generation, and blends of 3 would likely prevent ARD. The optimal blend depended on the specific materials used in the test and the reactivity of the NP. For example, a higher NP/MPA cutoff might be required at this site for the argillite materials.

- Sulphate concentrations in the leachate tended to be higher in blends with the lowest NP/MPA ratios. This relationship was likely influenced by gypsum formation which limited sulphate concentrations.

- As long as the final leachate pH remained neutral, it appeared that the metal levels would be adequately controlled for these materials.

Layering

- Underlying layers of acid consuming materials were an effective short term barrier for sulphate and metals produced by overlying net acid generating materials.

- Depletion calculations suggested that acid consuming layers would prevent breakthrough of acidic drainage when materials with an "uncertain" potential for acid generation (NP/AP ratios between 1 or 2) were layered above the acid consuming material provided that the overall NP/AP ratio was greater than 3.

- Acid consuming layers accumulated significant amounts of stored oxidation products from the overlying net acid generating layers, which may limit the effectiveness of the NP in the underlying layer over the long term.

Implementation Issues

- Differences as a result of scaling from the humidity cells to the larger column tests were not evident in the test data.

- Potentially net acid generating and net acid consuming materials were well mixed in these tests. When extrapolating these results to the field, it would be necessary to consider the spatial arrangement of the two material types. The required degree of mixing in the field is not known.

Details on Windy Craggy are provided in Appendix F.

4.2.7 *Wismut Project*

The Wismut project is located in the Ronneburg Uranium Mining District of the former East Germany. Mining was carried out from 1950 to 1991, with a total uranium production of approximately 100,000 tonnes. The area affected by mining encompasses about 35 square km and includes an extensive network of underground workings, three open pits and numerous waste rock piles. The remediation project is one of the largest in the world, with total costs estimated to be in the US$7,000,000,000 range (Jakubick *et al.*, 1997).

This review focused on the consolidation and relocation of 65 million m^3 of waste rock from the Absetzerhalde (Absetzer pile) to the Lichtenberg pit, and the concept of layering for oxygen consumption. As well, the field classification and material handling methods were reviewed.

The Absetzerhalde pile consists primarily of black shales, limestone and diabase and was constructed using conveyor belts which placed rock in uniform 10 metre thick layers. Seepage from the pile has sulphate levels ranging from 7,000 to 25,000 mg/L SO_4^{2-}, pH's of 1.5 to 3.1, and uranium levels of 1 to 8 mg/L.

The disposal plan combines several mitigation technologies, including: in-pit disposal, materials segregation, quicklime addition (to neutralize the acidity load during relocation), and 3 types of covers (water, oxygen consuming and a low permeability multi-layer surface cover) and collection and treatment of effluent. Figure 5 shows the overall disposal strategy. Materials classified as ARD generating are located below the projected water table, materials with an uncertain potential for ARD generation are placed in the middle layer, and acid consuming materials are placed in the upper layer. The surface cover will be designed to reduce water infiltration, and to prevent convection, so that oxygen access into the underlying rock is limited by oxygen diffusion rates. The soil cover will also provide a partial barrier to gas diffusion. As a backup to the soil cover, any oxygen entering the pile is expected to be consumed by the slow oxidation of sulphides in the upper layer of the waste rock (C-Zone), limiting further oxidation of sulphides in the marginally acid consuming layer.

Oxygen monitoring in an existing dump, the Nordhalde, showed strong evidence that a layer of non-ARD generating but sulphidic rock can reduce oxygen transport to underlying layers of net acid generating rock. Finite difference modelling of oxygen transport also supported this hypothesis. However, the oxygen consuming 'C-Zone' layer is still considered a conceptual control method and requires verification in a large scale dump. Therefore the C-Zone layer at the Lichtenberg pit is being implemented as a back-up for the low-permeability soil cover (Figure 5). The multi-layer soil cover will be designed to reduce infiltration and eliminate oxygen convection. The C-Zone layer is expected to consume any excess oxygen that diffuses through the surface cover, reducing reliance on the surface cover and potentially reducing surface cover costs.

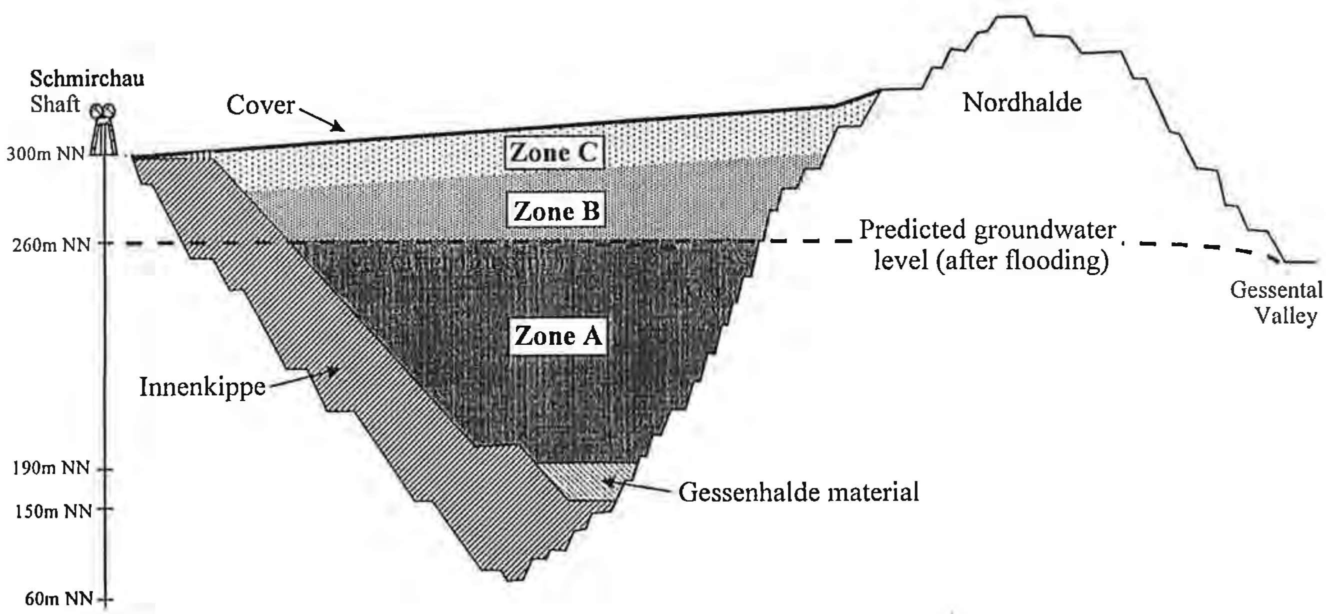


Figure 5 Section Through the Lichtenberg Pit Showing Waste Rock Relocation Zones A, B, and C
(Hockley *et al.*, 1997).

The actual re-location of the wastes requires extensive material characterization and field monitoring and control procedures. The procedures are costly, but are important to ensure that materials are placed in the appropriate zones in the pit.

In summary:

- C-zone, the oxygen consuming layer, appears to be a promising control for oxygen infiltration into the dump but must be verified in a full scale dump.

- Initial classification of materials based on analysis of drill core samples can vary substantially from classifications based on more detailed test pit sampling and analysis. The need to verify spatial classifications of waste rock through field checks and statistical assessments is emphasized.

- It is possible to characterize and classify materials in a timely and efficient manner to allow relocation of waste rock at rates that are equivalent to a full-scale mining operation (100 - 120,000 tonnes/day). Even at these large scales, characterization and classification can occur on a schedule that facilitates long term planning, and provides detailed operational control resulting in good material segregation. Excellent software database techniques are available to document the composition of the mined and relocated materials.

Details on the Wizmut Project are provided in Appendix G.

4.2.8 Mixing of Limestone with Duluth Complex Rock, Minnesota

Small scale studies with various masses of minus 10 (<2mm) mesh limestone were thoroughly mixed with 75 g of finely crushed Duluth Complex rock containing 2.1 % sulphide sulphur (Lapakko *et al.*, 1997). The blends were effective in delaying the onset of acidic effluent. A 0.5 g limestone: 1 g sulphur loading (NP/MPA ratio of 0.16) produced drainage pH above 6.0 for 109 weeks and loadings of 1:1 (NP/MPA ratio of 0.32) or greater produced drainage pH above 6.0 for 397 weeks. The results also indicated that:

- All limestone blends reduced sulphide oxidation rates; and,
- blends of 1:1 and greater reduced sulphide oxidation rates to levels at which host rock silicate mineral dissolution would be adequate to neutralize the resultant acid production over the long term.

4.2.9 Mt. Milligan Project, B.C., Canada

Blending marginally acid generating rocks (NP/AP ratios of less than 1) with net acid consuming rocks or possibly limestone from an off-site quarry was considered for one of the waste dumps at this site. Specific testing data for this disposal option was not located. However, there are three aspects to the project that are relevant:

1) The available report (Placer Dome Inc. *et al.*, 1991) stated that blending was not a proven ARD prevention measure, and that random blending of the waste rock could create inaccessible "hot-spots" within material that would not otherwise be a problem. Consequently, blending was proposed for a waste rock pile that would have a relatively impervious foundation so that drainage from the waste rock pile could be collected. The waste rock pile was also designed to be easily covered should additional mitigation measures be required. If this waste rock pile had been constructed it would have provided an excellent opportunity to study large scale blending, while minimizing risks to the mining company and the environment.

2) Blending was proposed for only a portion of the waste rock to be produced on site.

3) The proposed operational waste management plan was based on grade control methods used for ore classification during both the planning and mining stages. The plan called for development of an ABA block model for use in the long-range

planning. This would be supplemented by "grade-control" sampling to confirm the ARD potential of the materials during mining. This methodology would have been essential at this site because the ARD potential of the waste rocks could not be reliably correlated with any visual clues such as lithology or mineralogy.

4.3 Coal Mining Case Studies

4.3.1 General Overview

Coal mining procedures, including proper identification, handling, segregation, treatment and placement of overburden materials are key aspects in preventing ARD (Skousen *et al.*, 1987). Blending and layering have been extensively used as an inherent part of ARD management plans at coal mines. Coal waste rock tends to be finer grained than metal mine waste rock, and therefore has a greater potential to achieve a homogenous blend where the net acid consuming materials are in close contact with the net acid generating materials. However, few sites were found in the literature where the effects of blending and layering of overburden materials were sufficiently isolated from other management schemes that the effects could be determined.

Attempts to quantitatively link mine effluent quality to thickness and/or volume-weighted overburden acid-base balances have been only somewhat successful (Brady and Cravotta, 1992; Donovan and Ziemkiewicz, 1994).

In a review of 32 sites, Erickson and Hedin (1988) concluded that all sites having a volume-weighted NNP greater than +80 kg $CaCO_3$/tonne would produce alkaline drainage. The majority of sites having a volume-weighted NNP less than +20 kg $CaCO_3$/tonne yielded acidic effluent. No sites were found with NNP in the range between +20 and +80 kg $CaCO_3$/tonne, so that this range remained uncertain.

A study of 75 Appalachian coal mines by di Pretoro and Rauch (1988) found net alkaline drainage from all waste dumps when volume-weighted ABA parameters met the following criteria:

NP/AP ratio > 2.4

NNP > +30 kg $CaCO_3$/tonne

NP > 40 kg $CaCO_3$/tonne

However, these results were not considered reliable as a diagnostic tool because the materials analyzed for ABA parameters were up to 1.6 km from the mines where the discharges were sampled. The conservative NP/AP ratio criteria advocated by diPretoro and Rauch (1988) have not been confirmed in other research and are not currently used in the regulation of Appalachian coal mines.

A similar study by Erickson and Hedin (1988) showed that some sites with an NNP between 0 and 20 kg $CaCO_3$/tonne generated ARD, and that those with an NNP above 80 kg $CaCO_3$/tonne did not. However, none of the sites examined had intermediate NNP conditions between +20 and +80 kg $CaCO_3$/tonne .

A review of 74 coal mines in Pennsylvania, West Virginia, Maryland, Kentucky, and Illinois (Brady and Cravotta, 1992) indicated that a criteria of NP/MPA of 2 for net alkaline drainage resulted in correct predictions of drainage quality for 57 of the 74 sites. The error of predicting that a mine site would produce net alkaline water when it actually produced net acidic was 10 out of the 17 sites incorrectly predicted. Thus Brady and Cravotta (1992) recommended an NP/MPA criteria of at least 2.

Compilation of data from 38 Pennsylvania coal mine sites with total sulphur typically less than 1% (Brady *et al.*, 1994) showed a net alkaline drainage when volumetrically weighted ABA characteristics (using the methodology in Smith and Brady, 1990) met the following criteria:

NNP > +10 kg $CaCO_3$/tonne

NP > 15 kg $CaCO_3$/tonne.

However, these criteria did not necessarily result in low levels of metals in the drainage.

Perry and Brady (1995) expanded this review to 58 mines. Net alkaline effluent was produced from sites where the summary ABA values were:

NNP > + 12 kg $CaCO_3$/tonne

NP > 21 kg $CaCO_3$/tonne

Criteria of NP/AP ratio $\geq$ 2.0 has also been used in Appalachia (Brady and Cravotta, 1992; Ziemkiewicz, 1996, *pers comm.*). However Ziemkiewicz noted that most of these criteria were developed to cover off potential operational flaws in materials handling (i.e. poor blending and poor field estimations of AP and NP/AP ratios), and also because some of the measured NP was often provided by siderite, which would not be effective in field situations.

The varying success in establishing relationships between static acid base accounting parameters of the backfilled waste rock and the resulting drainage quality has been partially attributed to the limited drill hole analyses at some sites and the difficulty in obtaining representative post-mining water quality samples (Brady and Cravotta, 1992). Brady and Cravotta (1992) also noted that in attempting to develop relationships between static ABA values and potential effluent quality, no attempt was made to account for the effects of:

- Artificially high NP due to the presence of siderite in samples;
- Differences in spatial distribution of acid-forming and neutralizing minerals;
- Differences in mineral solubility,
- Relative rates and temporal variability of acid production and neutralization reactions;
- Effects of mining methods, such as selective handling of acidic spoil; and,
- Surface water or ground water hydrology.

Ziemkiewicz (1993) noted that in cases where acidic drainage occurred from coal waste rock piles, despite significant alkalinity in the overburden, the acidic drainage appeared to originate from localized sites within the backfill. This was ascribed to the inability of the acidic drainage to be influenced by alkalinity outside its path of least resistance as it made its way through the pile. Ziemkiewicz used this example to reinforce the need to thoroughly mix acid forming and alkaline rock to ensure effective use of the available alkalinity. Thus the appropriate NP/AP ratio at a site would depend on the material handling. At one extreme, reliance on random spoil dumping would require an overwhelming supply of alkaline rock (Ziemkiewicz, 1993). At the other extreme, very thorough mixing could theoretically allow the NP/AP ratio to approach 1 (Ziemkiewicz, *pers. comm.*, 1997). Ziemkiewicz (1993) identified the lack of thorough mixing as the probable cause for the reported field observations (diPretoro and Rauch, 1988; Erickson and Hedin, 1988; Brady *et al.*, 1994; Brady *et al.*, 1990) that twice or more NP was required for each unit of MPA (maximum potential acidity, calculated using total sulphur content, rather than pyritic sulphur content). However other mechanisms may also limit the availability of neutralization potential, including kinetic controls on the timely release of alkalinity, statistical estimations of NP and MPA values (Donovan and Ziemkiewicz, 1994) or reduction of effective alkalinity by such processes as iron oxide coatings (Rose and Daub, 1994).

Management plans in the eastern US coal fields tend to apply a variety of mitigation methods, such that the specific influence of blending net acid consuming and net acid generating materials may be masked. For example, Skousen *et al.* (1987) identified the following as some of the more important factors to be considered when estimating the amount and degree of ARD that would be produced on a site:

1. Physical and chemical properties of the overburden materials;
2. The method of mining, handling, and placement of overburden in the backfill;
3. The degree of compaction during backfill construction;
4. The length of time that acid-producing materials are exposed; and,
5. The hydrology of the backfill.

Isolating the influence of factors 1 and 2 would be key to determining the influence of blending and layering of waste materials. However, other mitigation techniques are also applied at coal mine sites. These include selective handling and isolation of potentially acid producing materials above the water table.

The addition of imported alkaline materials to potentially acid generating waste rock is very prevalent at coal mines in the eastern US (Donovan and Ziemkiewicz, 1994), and includes additions of crushed limestone, calcium oxyhydroxide, sodium hydroxide, and sodium carbonate, lime kiln flue dust, and flyash. Thus few examples were found that examined the blending and layering of waste materials without the addition of alkaline amendments. Moreover, many of the tests used to evaluate the addition of limestone or other alkaline materials (often incorporated by blending or spreading on overburden layers) have overall NP/MPA ratios less than, or close to, 1, so that appropriate NP/MPA ratios and mixing methods that would prevent acidic effluents have not been defined.

The eastern U.S. coal operations are relatively small, as compared to the large open pit operations in the western U.S. and Canada. Acidic seeps are expected to emanate from some strata in the open pits during mining. However, the pits are typically small in areal extent, and are generally exposed for a short period of time (less than one year) prior to being backfilled with mixed waste rock. Subsequent removal of the selected strata is conducted using visual clues. Blending of the remaining waste occurs through excavation (backhoe pulling down through the various strata), filling of the haul trucks and dumping. Limited analytical monitoring is conducted to confirm that the ABA of the strata are laterally consistent with pre-mining drill core results.

The majority of analyses produced for coal mines in eastern US use the method described by Sobek *et al.* (1978) for NP determinations. Most researchers also use maximum potential acidity (MPA) calculated from total sulphur analyses. An explanation is provided in Brady and Cravotta (1992);

> Total S can consist of sulfide, sulfate, and organic sulfur components, and acid can be produced by each of these forms. Yet, because pyrite generally is considered to be the acid producer, some workers have suggested that MPA should be computed by considering sulfide S only. However, Brady and Smith (1990) have identified analytical problems associated with the determination of sulfide S, and the typical analytical method may not indicate the amount of pyrite present. Furthermore, Cravotta (1991) and Alpers and others (1991) have shown that the dissolution of iron- and aluminum-sulfate minerals and the subsequent hydrolysis of iron and aluminum can produce substantial quantities of acidity. Harvey and Dollhopf (1986) have suggested that some forms of organic S also may be acid-producing. Thus, consideration of pyritic S alone may underestimate acid production, and the use of total S to compute MPA will account for the total potential acidity.

The reviewed coal mine case studies include laboratory tests, field tests and full-scale applications of blending and /or layering of overburden, often in combination with alkaline materials or other management strategies.

4.3.2 Rawhide Mine, Gillette, Wyoming

Lindsey (1988) described an overburden mixing study which evaluated physical and chemical characteristics of overburden materials both before and after mining using a truck and shovel operation. The lithology in the study area consisted of 40 to 45 foot thick coal spoil section. The top 30 to 35 foot thick layer was a combination of brown and brownish gray clay loam overlying gray shale and clay. This overlay a 5 to 10 foot thick section incorporating horizons A and B of the Roland coal seam. These two coal horizons were of poor quality, and were discarded with the overburden in the waste pile. Overall, approximately 1/3 of the 40 to 45 foot thick section was considered to be "unsuitable" for use for aquifer or root zone restoration (upper four feet of backfill) due to the presence of "acid forming materials". Details of the sampling protocols and densities were not described, but the study followed three phases:

Phase I: Intensive sampling and laboratory characterization of overburden material.

Phase II Excavation of the overburden by truck and shovel operation, and placement in a designated backfill zone.

Phase III Intensive sampling and laboratory characterization of the placed backfill.

Analyses included pH, electrical conductivity, % sand, % silt, % clay, NH_3, NO_3, acid-base potential (believed to be NNP, but this has not been confirmed), total iron (ppm) and total organic carbon (%). Results were presented as mean values and standard deviations for the samples from the original profile and the backfilled material. Selected parameters are provided in Table 7. The backfill demonstrated a reduced standard deviation, indicating that the backfill was more uniform than the original overburden, and met the criteria for "suitable" material.

These means and standard deviations provided a basis to assess whether other profiles could be similarly handled to produce suitable restoration materials, although the procedures were not described. Lindsey (1988) cautioned that the results were believed to be site specific, and depended on both the mining methods used and the material characteristics.

Table 7: Changes in Statistical Parameters Caused by Overburden Handling (Lindsey, 1988)

	Mean			Standard Deviation		
	Overburden	Backfill	% change	Overburden	Backfill	% change
pH	6.77	6.57	- 3	0.86	0.25	- 71
Acid-Base Potential (tons $CaCO_3$/1000 tons soil)	7.16	5.87	- 18	35.33	5.76	- 84
Total Organic Carbon (%)	12.49	10.79	- 14	19.38	3.44	- 82

An overburden tracking system was also briefly described (Lindsey, 1988). This consisted of pre-mine sampling over 5 to 10 foot increments (depending on lithology) from:

- Long range drilling: four holes per permit section, with at least one location cored.
- Short range drilling: rotary drill holes spaced 1320 feet apart (one hole per 10 acres), 3 to 5 years ahead of mining operations.
- Operational drilling: infill drilling to define the limits of 'unsuitable' overburden.

Samples were analysed for pH, electrical conductivity, acid base potential, total organic carbon, selenium and nitrate, which had each been assigned a "suitability limit". Unsuitable strata were identified, and an assessment of surrounding strata over bench heights was made to determine whether mixing via truck and shovel operations would result in a 'suitable' blend. If blending was not considered adequate, the entire bench was considered 'unsuitable'. These areas were staked for identification, and handled specially using one of three options:

- Located above the post mining water table, and at least four feet below the post mining topography (excluding topsoil);
- Placed such that subsequent truck loads of 'suitable material' dumped around the unsuitable material would result in a generally appropriate mix; or,
- Placed as a veneer of unsuitable material, either perpendicular or parallel to the backfill slopes, such that it forms a thin wedge surrounded by suitable material.

Backfill areas containing unsuitable materials are mapped and kept for future reference.

In summary,

- Although many details are lacking, this study indicated that truck and shovel mining methods could be effective in blending various waste rocks within a bench.

- Evaluation of sample means before and after provided a check on the adequacy of sample frequency.

- Comparison of standard deviations before and after mining provided a statistical basis for documenting blending effectiveness.

- Documenting post mining locations of potential acid generating materials was practical.

4.3.3 Field Trial, Island Creek Corp., Upshur County

Field studies which examined blending and layering, with and without alkaline amendments, in eleven 400 ton rock piles over an eleven year period were described by Ziemkiewicz and Meek (1994) and Donovan and Ziemkiewicz (1994). The piles were originally constructed in February 1982. Sandstone and shale samples were taken from the company's active mining operations, and were sized to exclude roughly the + 8 inch and - 1 inch material. Piles were constructed over a large area (roughly 16 m x 16 m), to a 2 m height, in 0.3 m high lifts. One sample from each rock unit was collected during placement on each pile for ABA analyses. Therefore, on a given pile, NP and MPA are based on one shale sample and/or one sandstone sample. Ziemkiewicz and Meek (1994) considered it likely that the distribution of materials within the pile was neither homogenous nor random, and that the reported NP/MPA rations might be subject to error. Piles without alkaline amendments consisted of:

Material	NP	MPA	NP/MPA Ratio
	kg $CaCO_3$/tonne		
100% sandstone	38.4	9.69	3.94
100% shale	2.5	30.3	0.08
Sandstone/shale in layers	3.1	17.5	0.18
Sandstone/shale blended	18.1	28.4	0.29

Alkaline amendments were added to selected piles by spreading over each lift during construction. Donovan and Ziemkiewicz (1994) described these piles as being composed of three horizontal layers of sandstone, shale and sandstone. Ziemkiewicz and Meek (1994)and Donovan and Ziemkiewicz (1994) considered it possible that the alkaline amendments were not uniformly placed.

Piles with alkaline amendments included :

Amendment	NP	MPA	NP/MPA Ratio
	kg $CaCO_3$/tonne		
0.46% limestone by mass (LS1)	34.7	62.5	0.56
1.07% limestone by mass (LS2)	53.8	32.2	1.65
1.26 limestone by mass (LS3)	59.1	24.4	2.38

Donovan and Ziemkiewicz (1994) identified these piles as containing various amounts of crushed limestone added in thin lifts to stratified lifts of sandstone, shale and sandstone. (Values provided in Ziemkiewicz and Meek, 1994 should be used with caution because their conclusions referred to two piles with limestone amendments with NP/MPA ratios greater than 1.65, which agreed with the ABA values provided in by Donovan and Ziemkiewicz (1994).)

Effluents from the piles were collected weekly, and then biweekly during the first year, and analysed for acidity, alkalinity, pH, sulphate, calcium, magnesium, iron and manganese. Effluent flow rates and precipitation were also measured over the first year. Annual precipitation averaged 45 inches at the site. The pH of the local rain averaged about 4.5.

The rock piles constructed without amendments, including the sandstone pile with an NP/MPA ratio of 3.94, quickly became acidic in the first year. Effluent from the 100% sandstone and 100% shale piles, fell to approximately pH 4.2 over the first 7 months, with sulphate rising to 200 to 2500 mg/L. The pH of the layered and blended piles fell less dramatically to 4.9 and 6.2 respectively, but sulphate levels were still high, reaching 2700 mg/L. Ziemkiewicz and Meek (1994) concluded that blending seemed to improve the pH of the piles, while there was little effect on sulphate generation. However, this could have been directly due to the increased neutralization potential available in the blended pile. Also, given the development of acidic effluent in a pile with a reported NP/MPA ratio of 3.94, the accuracy of the single samples for each rock unit used to characterize the piles are suspect. Ziemkiewicz and Meek (1994) attributed any error to estimation of effective NP because the sulphate generation rates appeared to be consistent with the estimated rock pyrite contents.

The addition of limestone to layers within the piles (test piles LS1 through LS3) had no significant effect on the release of sulphate as compared to the 100% sandstone, 100% shale, blended or layered piles in the first year. Over the first year, the pH of effluent from test pile LS1, with an NP/MPA ratio of 0.56, and LS2, with an NP/MPA ratio of 1.65, remained above neutral. At the same time, test pile LS3, with the highest NP/MPA ratio of 2.38, became acidic.

The piles were again sampled a single time in January 1993, 11 years after construction (Ziemkiewicz and Meek, 1994). All piles were undisturbed except for the 100% sandstone which had been excavated, and could not be sampled. The pH of the samples from the remaining three piles without amendments had changed little over the 11 years. The measured sulphate concentration had dropped to 100 mg/L.

After 11 years, the final pH of the piles with added limestone reflected the relative NP/MPA ratios, with LS1, LS2 and LS3 at pH 5.2, 6.4 and 6.6 respectively. Ziemkiewicz and Meek (1994) noted that the reversal in the first year could be a more efficient utilization of available alkalinity in LS1 and LS2, possibly as a result of initial placing and mixing, but construction records did not confirm this.

Donovan and Ziemkiewicz (1994) noted limitations of the data set , including:

- No analyses of ions, including dissolved sodium, potassium, chloride, fluoride, silica or aluminum were undertaken, so that full electrochemical balance of the analyses could not be checked, nor could rigorous mineral equilibrium calculations be performed;

- No analyses of solid phase mineralogy, geochemical composition, or extractable ion tests (wash tests) were undertaken either before construction or at the end of the test period, so that presence of secondary minerals could be defined; and,

- No characterization of the potential heterogeneity in NP or MPA was undertaken within each pile.

This precluded the ability to check for full electrochemical balances, perform rigorous mineral equilibrium calculations, or make assumption on mineralogy.

Donovan and Ziemkiewicz (1994) noted that most piles reached gypsum saturation in the middle of the first year. However, Ziemkiewicz and Meek (1994) assumed that because the piles were shallow (2 m thick) that storage of oxidation products would be minimal, such that sulphate flux rates out of the piles would be a reasonable estimate of pyrite oxidation.

In summary, Ziemkiewicz and Meek (1994) concluded:

1) Pyrite oxidation (sulphate depletion rates) was independent of overall effluent pH.

2) At the NP/MPA ratios tested, pyrite oxidation (sulphate depletion) rates were little affected by alkaline amendments.

3) The rate of pyrite oxidation (sulphate depletion) remained within a relatively narrow limit, despite amendments. (Donovan and Ziemkiewicz, 1994, noted that the sulphate depletion rate was relatively constant at 0.0239 ± 0.0036 moles SO_4 per ton spoil per day for a broad range of lithologies, NP, MPA, alkaline amendments and pile constructions.) Since the rate increased with the proportion of sandstone, Ziemkiewicz and Meek (1994) identified oxygen diffusion as a likely candidate for the gross physical phenomena that would control pyrite oxidation rates.

4) There was no clear relationship between NP/MPA ratios and performance of the piles. Ziemkiewicz and Meek (1994) attributed this to potential errors in estimated effective NP due to sampling error, NP analyses, and/or poor mixing of amendments and wastes.

5) Alkaline amendments generally delayed the release acidic effluent from the piles, but this was not true in all cases. (Donovan and Ziemkiewicz, 1994, noted that layered amendments appeared to have had no clear effect on drainage quality during the first year. This was thought to be related to potential heterogeneity in distribution, heterogeneity in NP, or MPA within the piles, to armouring of the limestone by iron hydroxides, or to a potential time lag for amendment tracers to be detected in pile effluent.)

6) In two cases where limestone amendments raised the overall NP/MPA ratio above 1.65, pH values greater than 6 were observed 11 years after construction. However, one of these cases, with an NP/MPA ratio of 2.3 produced acidic effluent in the first year. One other case with a higher NP/MPA ratio (of 3.94) also produced acid in the first year, but was not available for sampling after the eleven year period.

However, the limited database characterizing the piles, and the assumption that sulphate depletion was not effected by secondary mineral precipitation within the piles leaves these conclusions in some doubt. NP/MPA ratios that would prevent the onset of acidic drainage in the short and long term, and the degree of blending or mixing required to make effective use of available NP were not determined.

4.3.4 Laboratory Tests on Limestone and Lime Kiln Flue Dust Additions

Rose and Daub (1994) examined the addition of thoroughly blended limestone and limekiln flue dust to a pyritic shale containing 7% sulphur and 0.17% carbonate carbon (equivalent to an NP of approximately 14 kg $CaCO_3$/tonne). The alkaline amendments were added to achieve an NP/AP ratio of 1, in reaction containers designed by Hornberger (1985). In this method, duplicate 1 kg samples crushed to pass 1 cm were used as controls, or thoroughly mixed with:

a) 220 g of crushed and sieved limestone (2 - 5 mm sieve fraction)
b) 231 g of lime kiln flue dust, containing approximately 40% $CaCO_3$ and an NP of 957 kg $CaCO_3$/tonne (particles from dust to 1 mm in size).

Six columns were run (three sets of duplicates) by placing each sample in a 2-L plastic leaching vessel above a layer of 6-mm-diameter glass beads, then leached once a week for 20 weeks. A measured amount of de-ionized waster was poured into each vessel, wetting the samples and draining gradually into the beads in the bottom of the container. After a specified period of time, the water was drained from a plug at the bottom of each vessel, and an additional aliquot of water was added to the top, flushing residual oxidation products from each sample. The volume of effluent was measured and analysed for pH, eH, specific conductance, temperature, and alkalinity. The effluent was then filtered through Whatman #41 filter paper and analysed for sulphate and iron, manganese and calcium. The columns were placed in an air-tight enclosure through which water-saturated air was circulated to minimize evaporation during the experiment, however, 5 to 10% of the water was still lost over 20 week test period.

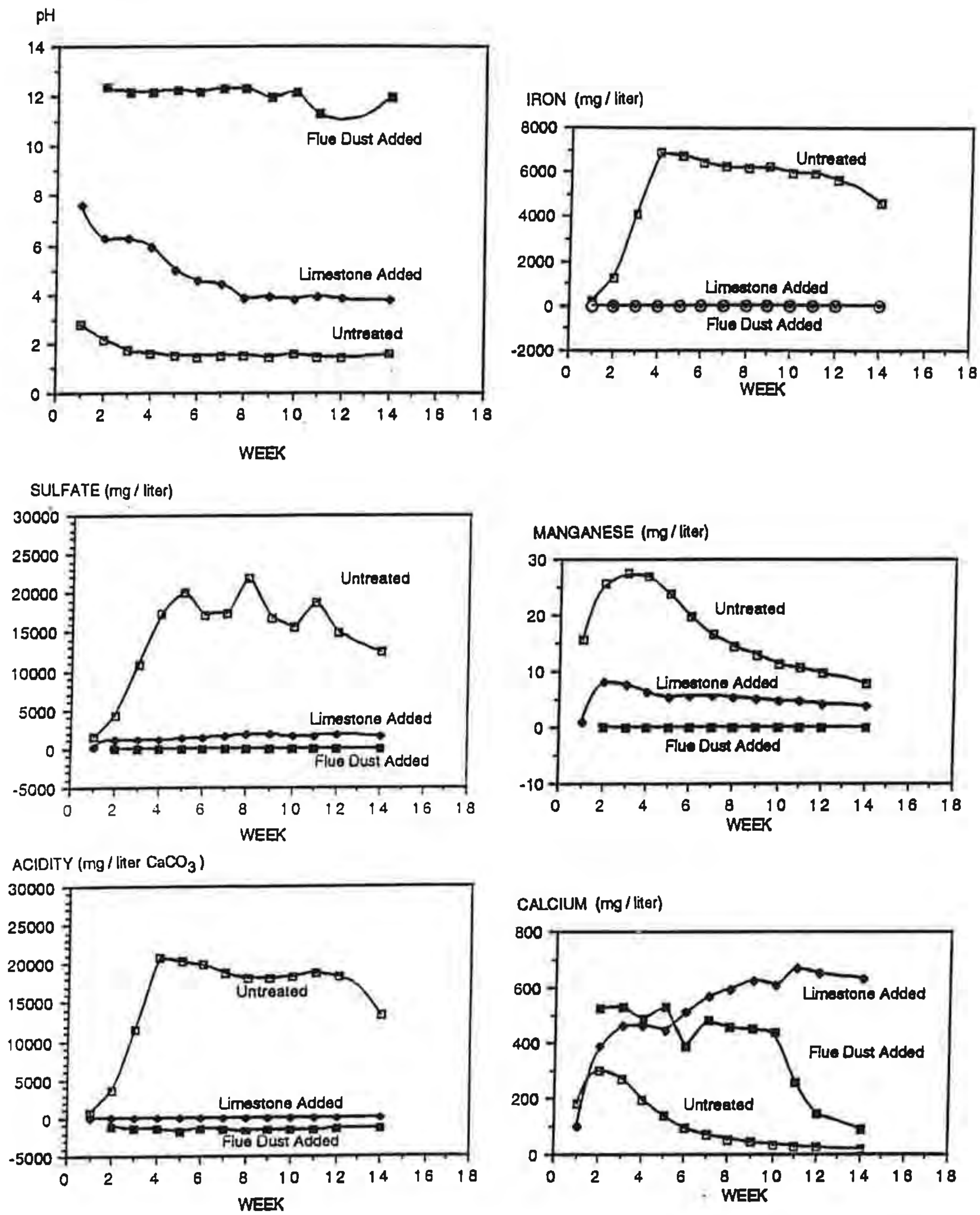


Figure 6: Concentrations of Solutes in Weekly Effluent Samples
(Rose and Daub, 1994)

The results are reproduced in Figure 6. These results indicate (Rose and Daub, 1994) that:

- The untreated samples were acidic (pH 1.5), with elevated sulphate and metal concentrations (averages given as sulphate 17,000 mg/L; acidity 19,000 mg/L; iron about 7,000 mg/L; and manganese about 10 to 20 mg/L);

- The samples treated with limestone were initially pH 7.6, but levelled off at pH values of 3.4 to 4.2, with much lower levels of sulphate and metals (averages given as sulphate about 1,600 mg/L; acidity about 30 to 40 mg/L; iron generally less than 0.2 mg/L); and manganese about 5 mg/L). Iron oxide coating of limestone fragments near the surface of the column was visible after about 5 weeks. Gypsum crystals were also visible on the surface of the column, indicating that measured sulphate in the effluent did not represent all the sulphate that was formed.

- The samples treated with limekiln flue dust had pH values of 11.3 to 12.3 and consistently contained large amounts of alkalinity (approximately 1,500 mg/L CaCO3), sulphate mostly less than 5 mg/L, and iron and manganese less than detection limits. Iron oxide and gypsum were not observed, but some cementation was indicated.

In summary, the authors concluded that:

- The addition of limekiln flue dust at a NP/AP ratio of 1 was "...completely effective in preventing formation of acid effluent". The low sulphate concentrations in the effluent (under saturated with respect to gypsum) were taken to indicate that essentially no pyrite was being oxidized in these samples. The lime products were thought to prevent the formation of acid because of "...their much more extreme pH effects as well as their fine grain size."

- The addition of limestone at the same NP/AP ratio allowed the formation of a slightly acid effluent, through markedly less concentrated than the untreated samples. The limestone particles had their reactiveness reduced due to coating of ferric hydroxides within the aerated zone. Finer grain sizes and thorough mixing were identified as potential means to improve the results of limestone addition.

However, the paper did not present extrapolations of findings, or calculations of NP depletion rates that would allow a prediction of when acidic drainage might emanate from the samples treated with limekiln flue dust. Limestone amendments (NP/AP ratios) that might prevent acid effluent or adequately reduce metal leaching were not discussed.

4.3.5 Field test trials on Lime Kiln Flue Dust Addition

Evans and Rose (1995) examined alkaline addition for the purpose of preventing ARD from coal spoils. Limekiln flue dust was added in various amounts to large scale cells (12 m x 6 m x 4.3 m, holding 360 tonnes of pyritic shale) located on an unmined north-facing slope. The limekiln flue dust was fine grained to powdery, and was composed of about 50% $CaCO_3$, 32% CaO or $Ca(OH)_2$, and 18% quartz and clay (measured NP of 798 kg $CaCO_3$/tonne). The shale contained 1.89% total sulphur, essentially all as pyrite, and had negligible neutralization potential. After 10 months, all of the cells had produced ARD, with pH less than 2.6. Two of the cells which received flue dust additions in the amounts of 114% and 171% of that required to theoretically neutralize potential acidity (ie. NP/AP ratios of 1.14 and 1.71), produced only 5 to 10% of the acidity, sulphate, and iron of the untreated cells.

The production of ARD at these alkaline addition rates was partially attributed to pyrite that had begun to oxidize in the 5 weeks that the spoils were stockpiled prior to mixing with limekiln flue dust. The production of acidic effluent was also attributed to an inability to thorough mix the flue dust and the coal spoils. The lime dust was spread and levelled on the top of each 40 tonne layer of shale (4 loader buckets) and the surface of each lime layer was scarified with the bucket teeth of the loader to mix the lime dust into the top of the shale layer. Evans and Rose (1995) estimated that 1/3 to 1/2 of the spoil had some lime mixed with it. They believed that this partial mixing led to the development of micro environments of pyrite oxidation, which were not controlled by the infiltration of the alkaline water from above. As a review of mineral saturations indicated that the limed piles were saturated with gypsum throughout the experiment, they postulated that gypsum precipitation may have caused alterations in flow paths through the coal spoil, thus causing the circumvention of the limed zones by the infiltrating water. Compaction induced by loader tires was also identified as a possible cause of channelization. They also noted that sulphur content was highest in the finest particles of coal spoil (2.38% in grains less than 0.14 cm diameter), which were likely to be dominant in forming ARD. (This would also have influenced the calculated NP/AP ratio if the sulphur content of only the more reactive particles had been considered.)

In summary,

- Rapid mixing of alkaline and acid generating materials is considered critical to blending success;

- Incomplete blending, which leaves zones of potentially acid generating material, is considered less effective than complete blending; and,

- Gypsum precipitation within the waste rock pile, and compaction of the waste materials, influences hydrogeology within the pile, and can decrease the effectiveness of adding alkaline materials.

4.3.6 Kauffman Mine, Phase 1, Pennsylvania

Rose *et al.* (1995) describe a full-scale trial application of limekiln flue dust to Phase 1 backfill at the Kauffman Mine, located 10 kilometres southeast of Clearfield, Pennsylvania. This case study provides an example of typical calculations used at

Pennsylvania coal mines to determine requirements for alkaline addition, and a description of the methods used to incorporate the alkalinity. The flue dust is not blended throughout the waste rock, nor layered to achieve uniform flue dust distribution or uniform NP/AP ratios throughout the waste rock pile. Rather the addition of alkalinity is targeted to areas with the greatest acid production potential.

Annual precipitation at the site is about 100 cm, approximately equally spaced throughout the year. Mean monthly temperature range from about 19.4 degrees C in July to - 5.5 degree C in January.

Requirements for alkaline addition were determined from ABA analyses (NP and % sulphur determination) on three overburden holes. Two holes (OB-3 and OB-4) were developed by air-rotary drilling. Samples were collected in sandstones at maximum intervals of 5 feet, other strata were typically sampled at 3 foot intervals, although all samples were cut off at clear changes in lithology. Samples were analysed for total sulphur and NP using methods in Sobek *et al* (1978). Total sulphur ranged from 0.05 to 1.51%. NP ranged from <1.0 to 64.5 kg $CaCO_3$/tonne.

The materials identified for special handling were (Rose *et al.*, 1995):

1. Any strata with a total sulphur percentage of 1.0% or greater and a 3-foot thickness or greater and not located adjacent to the LK coal seam.
2. Strata with a total sulphur percentage of 0.5% or greater and located adjacent to the LK coal seam.
3. Binders (in-coal seam waste layers) and LK pit cleanings not removed with the coal.

The 3-foot thickness was chosen because lesser thicknesses were considered difficult to identify and segregate. For ease of operations, the approximately 10-foot zone at and below the LK Rider seam was selectively handled when present. Special handling consisted of placing the material in 2-foot lifts and compacting the material with haulage rucks. Where necessary, additional 2-foot lifts were constructed over the initial lift following compaction and placement of alkaline material. The specially handled zones were located at least 10 feet above the pit floor, 10 feet from the highwall and 15 feet below the final regraded topography.

Application of the lime kiln flue dust (NP of 798 kg $CaCO_3$/tonne) was described as follows (Rose *et al.*, 1995):

- On the pit floor in relatively small amounts not to exceed 100 tons/acre;
- Around specially handled material, including between the 2-foot layers in large zones;
- Around the shot area immediately following a blast so that mixing with overburden occurred during excavation; and,
- On the backfill prior to topsoil placement. At this location, mixing consisted of 'backdragging' with a bulldozer following placement of the flue dust.

The necessary alkaline addition was determined using the spreadsheet program of Smith and Brady (1990). In this calculation, only sulphur values greater than 0.5% were used to calculate MPA, and only samples that reacted (fizz) with 25% hydrochloric acid were used to calculate NP. To account for weathering, NP within 60 feet of the final surface was discounted. This was consistent with several drill holes which showed negligible NP in the top 60 feet of the original profile.

The calculated alkaline deficiency for Phase I are reproduced in Table 8.

Table 8: Kauffman Mine Alkaline Deficiencies (Rose *et al.*, 1995)

Hole	Area of Influence (acres)	Alkaline Deficiency (tons/acre)	Require Alk. Addition (tons $CaCO_3$)
OB4	9.20	1094	10,065
C1	8.80	43	378
OB3	8.80	169	1,487
Total	26.80		11,930

As an example of the alkaline placement, Rose *et al.* (1995) described flue dust distribution in the area influenced by hole OB4 in Table 9.

Table 9: Alkaline Addition Plan, Kauffman Mine, Phase 1, OB4 Area (Rose *et al.*, 1995)

Area	Addition Rate	Tons CaCO3 equivalent
Special Handling Zones	145 tons/acre-ft.	3257
Pit Floor	100 tons/acre	920
Mining Area after Blast	540 tons/acre	4968
Prior to Topsoil Placement	100 tons/acre	920
Total		10,065

After a year and a half, the water quality in wells placed in the backfill remained at pH 6 and 7, with alkalinity exceeding acidity. Acidity values had ranged from 3 up to 46 mg/L $CaCO_3$ equivalents, alkalinity was initially 37 mg/L but had increased to greater than 200 mg/L $CaCO_3$ equivalents at the end of the year and a half. A lysimeter placed at 27 foot depth indicated alkalinity up to 360 mg/L $CaCO_3$.

In summary,

- segregation of potentially acid generating material, and targeted alkaline addition at an overall NP/AP ratio of 1 may be effective in preventing acidic effluent.

However, insufficient time had elapsed to demonstrate the long term success of this well documented mitigation strategy.

4.3.7 High Power Mountain, West Virginia

The High Power Mountain coal mine in West Virginia applied lime kiln flue dust directly to their coal refuse as it left the preparation plant via a conveyor belt. The coal refuse had alkaline deficiency of 26 to 53 kg $CaCO_3$/tonne, and flue dust was added at the appropriate rate to provide an NP/MPA ratio of 1. This method had been effective in preventing ARD from the coal refuse pile for four years (Rich and Hutchinson, 1990).

4.3.8 Coaltrain Project, West Virginia

At a re-mining operation in West Virginia, overburden was imported from a project on the Bakertown coal bed, 1.6 km away, to provide alkalinity to the alkaline deficient backfilling operation. Five analyses of the alkaline red shale indicated total sulphur less than 0.002%, with an anomalous sample containing 0.01%. NP ranged from 150 to 569 kg $CaCO_3$/tonne (average of 278), and NNP from 120 to 569 kg $CaCO_3$/tonne(average of 277). This material was placed in a 1 m thick layer on the coal pavement the same day that the coal was removed. On-site overburden was then placed over top. A black shale binder from the coal seam (NNP of - 60 kg $CaCO_3$/tonne) was selectively placed about 6 m above the pavement and compacted. It was then covered with 0.3 m of the alkaline red shale. These layers were then compacted by bulldozers and trucks.

Backfilling continued with on-site overburden until the approximate original contour was reached, and topsoils were replaced on the surface. Several inches of the alkaline red shale were also spread on the surface and mixed with the topsoil. Using ABA data and relative thicknesses of materials, it was estimated that the NP/MPA ratio was approximately 3. The effluent discharged from the immediate area rose from a pH of 3.7 to 7.0, and total acidity dropped from approximately 75 mg/L to less than 1 mg/L. Iron and manganese concentrations also declined. The effectiveness of the compaction and addition of alkaline overburden in improving the effluent quality had been demonstrated for two years at the time of reporting (Skousen and Larew, 1994).

4.3.9 Telkwa Coal Project, B.C., Canada

The proposed Telkwa coal project is a large open pit operation, with limited backfilling into the pits. Selective removal of two strata has been proposed so that the overall volume-weighted NP/AP ratio of the waste rock piles would be greater than 2. This criteria was based on criteria developed in the Appalachia region of the United States, where sites with NP/AP greater than 2 are considered likely to produce net alkaline drainage (Brady *et al.*, 1994).

The management strategy is still undergoing modification and review. Kinetic tests are underway to substantiate the ability of the proposed NP/AP ratios to prevent acidic effluent from the waste rock piles over the long term. The management strategy envisions that individual truck loads could have NP/AP ratios as low as 1, but that operational controls on material placement would prevent low NP/AP ratio loads being placed adjacent to each other. The ability to place various materials to achieve the target NP/AP ratios would arise from the simultaneous mining of several coal seams and associated waste rock strata (Manalta Coal Ltd., 1994).

5.0 CONCLUSIONS

As a long term option of preventing the onset of acidic drainage, potentially acid generating and acid consuming waste rock can theoretically be blended or layered to produce a geochemically benign composite. However, attempts to verify the theory have found the case study database to be lacking in critical details. Typically, older sites that provided long term water quality data to support the success or failure of this technique suffered from a lack of documentation on materials in the dumps and the level of blending employed. Newer sites that provided more data characterizing the materials and outlining operational procedures tended to have limited time in which to demonstrate success or failure. In addition, at coal mine sites, determining the effect of waste rock blending was often confounded by additional management measures such as lime, kiln dust, steel slag, and flyash addition, and/or the presence of acidic drainage from adjacent historic sites.

Review of numerous coal mine sites by researchers in Appalachia has resulted in the development of criteria for overall volume-weighted waste rock blends considered likely to produce net alkaline drainage. The commonly cited criteria are:

NNP > 10 kg $CaCO_3$ equivalent/tonne;
NP > 15 kg $CaCO_3$ equivalent/tonne; and,
NP/AP ratio > 2:1.

The reviewed studies on coal wastes tended to examine the addition of alkaline materials, rather than the deliberate blending of acid generating and acid consuming waste rock types. These studies indicated that:

a) Fine alkaline amendments distributed uniformly throughout a waste pile were effective in delaying the onset of acidic drainage, however the correct ratio of alkaline addition to acid potential which would permanently prevent the onset of acidic drainage at specific sites was often not determined.

b) Layering of limestone amendments within test piles was less effective in delaying the onset of acidic drainage, but did not effectively prevent acidity from developing within the piles (Ziemkiewicz and Meek, 1994);

c) Lime kiln dust amendments were more effective in reducing the amount of net acidity produced and in preventing the onset of acidic drainage than limestone amendments with coarser grain size, possibly due to their higher surface area and greater reactivity of CaO (Rose and Daub, 1994); and,

d) The order in which overburden was placed in relation to acid drainage generating coal spoils had a significant effect on leachate quality.

Detailed information on full-scale sites, including compilation and assessments of ABA and kinetic test data, pre- and post-mining monitoring of waste materials and drainage water were not found in the course of this project.

Examination of the available metal mine case studies indicated that:

a) The majority of cases evaluated blends with NP/AP ratios less than 2.

b) Blending did not reduce sulphide oxidation rates in the potentially acid material unless highly reactive neutralizing material (limestone) was applied, and the blending was near ideal, as in column or humidity cell tests. Such ideal blending is not considered feasible at a field scale;

c) Layering did not reduce sulphide oxidation rates in potentially acid generating material, when layers were 10 cm thick or less;

d) Blending and layering were effective in delaying the onset of acidic leachate, and in reducing the strength of acidic leachate as compared to the potentially acid generating material alone; and,

e) Trace metal concentrations were lower in leachate from blended and layered materials due to the decreased solubility of hydroxide and carbonate phases at the elevated pH. However dissolved trace metal levels could still be problematic.

Almost all of the small scale tests with low NP/AP ratios have, at best, delayed the onset of acidification. That is, to date, this mitigative technique has not been successfully employed at metal mines. This lack of success has occurred despite using laboratory blending tests which were well-controlled and relatively simple as compared to full scale field applications.

Future work on this subject must focus on the problems identified in this project, in particular:

a) Is it possible to obtain a sufficiently intimate mixture of acid producing and acid neutralizing materials at a practical cost?

b) Can problems of preferential flow through potentially acid producing layers be predicted and/or circumvented?

c) What safety factor must be applied to laboratory developed NP/AP ratio criteria to address additional influences encountered in the field?

Given that the theory anticipates the prevention of acidic drainage at NP/AP ratios between 1 and 2, testing at NP/AP ratios in this range and higher are required.\

From the literature and case studies reviewed, it appears that blending and layering of net acid generating and net acid consuming waste rock may be a legitimate method of mitigating acidic drainage and metal leaching. However, there remains considerable uncertainty and further investigation into the application and practice of this method is required. Specifically, more research on large scale, controlled field studies is needed. The identification of additional sites is also encouraged, as well as thorough documentation of waste dump materials, construction methods, and drainage quality. At issue is what portion of the alkalinity is available to neutralize the acidity produced by sulphide oxidation, and the extent to which this is influenced by the thoroughness with which alkaline and acid producing waste rock are blended at a given time.

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7.0 CASE STUDY DATA TEMPLATE

MINE INFORMATION: BLENDING & LAYERING

The following data checklist has been developed to assist in compiling case study information on blending and layering of potentially net acid generating and net acid consuming waste rock to prevent the onset of acid rock drainage. The data list attempts to include all data that should be collected in order to develop and subsequently confirm preliminary predictions.

7.1 General Information

This section is to identify and describe the mine, and provide key climatic and location information that may influence reactivity and/or leachate quality.

Mine	
Location	
Latitude/Longitude	
Mine owner	
Mine contact	
Economic metals	
Open pit or underground	
Ore production per day	
Waste rock production per day	
Minesite layout (maps if possible)	
Average annual precipitation, and/or range	
Average annual temperature	
Degree-days of freezing	
Other significant climatic information	For example: annual precipitation cycles (wet and dry periods, temperature ranges, snow and/or rain as a proportion of precipitation, annual evaporation

7.2.0 Geology

7.2.1 Regional Geology: This section should describe the deposit in terms of its regional setting and genesis.

7.2.2 Site Geology: This section should describe the mineral deposit in terms of lithologies and/or alteration characteristics. The objective is to describe the rock type or units that are (were) identified in predictive efforts as representing distinct acid generating characteristics. The size, shape and location of the rock units within the deposit should be described, along with range and central tendency of the rock mineralogy, and petrology of sulphides, carbonates, and other gangue minerals.

7.2.3 Indicator information: The presence of gossans, soil weathering profiles, background water quality, or other aspects of the site that provide(d) insight to the reactivities of the native rock types should be described.

7.3 Static Tests

Methodology: Describe, identify and/or reference methods used for each analysis. Barium sulphate sulphur is included as a potential parameter because barium sulphate reports as total sulphur rather than sulphate sulphur, but is not an acid generating sulphur form. CO_2, in units of $CaCO_3$ equivalents is also included because analysis of total or inorganic carbon is often used as a check on neutralization potential (NP). Areas are available for both paste pH and field pH since different protocols provide different information. Paste pH, in this context, refers to the laboratory test on a crushed sample performed as part of the EPA method (Sobek *et al.*, 1978), which may release additional neutralization potential from the interior of rock particles. Field pH, often determined on uncrushed samples, may more closely reflect the surface characteristics of the exposed particles. In all cases, methodology should be referenced or described. Suggested statistics to be performed on the data set include: mean, median, and percentiles (10, 25, 75, and 90).

For each rock type or composition	Name, petrographic descriptions of sulphides, carbonates and gangue minerals; crystal structure, size, location(s), physical properties of the rock, including slaking behaviour.						
	No.	mean	median	10%	25%	75%	90%
Elemental composition (metals, whole rock analyses)							
% total sulphur							
% pyritic sulphur							
% sulphate sulphur							
% barium sulphate sulphur							
CO_2, as $CaCO_3$ equivalent							
Paste pH (laboratory test)							
Field pH							
NP							
AP (pyritic or total sulphur)							
NNP							
NP/AP ratio (NPR)							
Mean NP/mean AP							

7.4 Kinetic Tests

This section compiles key values from kinetic tests that are often used in prediction efforts. Comparison of test results to actual field conditions is considered valuable to refine the use of these tests. Identification of the methods used in the tests are critical, as well as sample definition. Kinetic tests may be performed on samples representing an individual rock type, or blended or layered to simulate the composition of the anticipated waste rock pile. (May include shake flask tests or other tests for initial soluble metal content)

Methodology: Describe, identify and/or reference kinetic cell methodology and methods used for each analysis.

For each test material representing a rock type, or a composition representing an anticipated waste rock dump.	Rock type description, or composition by rock types, and descriptions, including mineralogy.
Mass of sample (grams - dry weight)	
Particle size/geometric surface area (m^2/kg)	
Description of leaching cycles	
Leachate added per cycle (L)	
Leachate recovered each cycle (L)	
Flushing rate (L/kg/wk)	
Duration of test (weeks)	

ABA and metal values:	From split of test material, and/or theoretical calculation from rock type composition
Elemental composition (metals, whole rock analyses)	
% total sulphur	
% pyritic sulphur	
% sulphate sulphur	
% barium sulphate sulphur	
CO_2, as $CaCO_3$ equivalent	
Paste pH (laboratory tests)	
NP	
AP (pyritic or total sulphur)	
NNP	
NP/AP ratio (NPR)	
Mean NP/mean AP (from calculated values)	

Test results:	
Result type	1: Alkaline through test 1a: Linear cumulative SO_4 production 1b: Convex cumulative SO_4 production 1c: Concave cumulative SO_4 production 2: Acid (pH < 4.5) through test 3: Alkaline initially, turning acid (pH < 4.5) through test
SO_4 production rate	mg/kg/wk at near neutral pH, and at test steady state
Alkalinity production rate	mg $CaCO_3$ equivalent/kg/wk
Acidity production rate	mg $CaCO_3$ equivalent/kg/wk
Net alkalinity production rate	mg $CaCO_3$ equivalent/kg/wk
NP depletion rate	mg $CaCO_3$ equivalent/kg/wk
Time lag to pH < 4.5 (weeks)	
Minimum pH	
Leachate quality for metals (mg/kg/week)	Suggested parameters include Ca, Mg, Cu, Zn, Fe, Mn. Other metals may be included. Suggested statistics include mean, range, standard deviation, and percentiles (10,25,75,90), over test range and for test steady state.

7.5 Predicted waste rock pile composition and leachate quality

This section should briefly describe the proposed waste rock pile design and composition, the method used to predict leachate quality for each waste rock pile, and predicted leachate quality.

Waste Pile Design and Construction	A table should be completed for each distinct waste rock pile.
Plan area of dump	
Catchment area	
Foundation material (recharge/discharge)	This information indicates the potential for leachate to enter the groundwater regime.
Dimensions	Overall height, bench heights, bench width, bench slope, overall slope.
Tonnage	

Composition:	
Proportions of rock types	Weighting by rock type, or other ABA characterized units.
Spatial deposition of rock types	
Depositional sequence	
Depth of layer(s)	
Method of placement	Truck size, end dump, levelled by bulldozer...
Estimate of accuracy of placement method (re: spatial deposition, depth of layers)	
Applied prevention measures (if any) and application methodology	For example, covers, alkaline addition, etc.

Calculated theoretical values:	No.	mean	median	10%	25%	75%	90%
Elemental composition (metals, whole rock analyses)							
% total sulphur							
% pyritic sulphur							
% sulphate sulphur							
% barium sulphate sulphur							
CO_2, as $CaCO_3$ equivalent							
NP							
AP (total or pyritic sulphur)							
NNP							
NPR							
Mean NP/mean AP							

Predicted waste rock pile leachate quality	Methods and assumptions used to develop prediction.
Flow	
pH	
Alkalinity (mg/L)	concentration, and/or kg/tonne waste rock/time
Acidity (mg/L)	
Net alkalinity (mg/L)	
Sulphate (mg/L)	
Dissolved copper (mg/L)	
Dissolved iron (mg/L)	
Dissolved zinc (mg/L)	
Dissolved manganese (mg/L)	
Dissolved calcium (mg/L)	
Dissolved magnesium (mg/L)	
Other metals of interest (mg/L)	
Qualitative assessment	Acid drainage: yes or no Impacts to adjacent ground and or surface waters

7.6 Operational data

This section should provide details on the actual methods used during mining to sample and identify individual rock types, and methods used to direct placement of rock types in waste rock piles.

Operational monitoring	
Spacing of blast holes (m x m)	
Number of blast holes sampled	
Sampling regime in each blast hole (number and spacing of samples, continuous or grab samples, link to geology)	
Estimated tonnage of waste rock represented by each sample	
Analyses performed on each sample	

Selection criteria: field method used to identify individual rock types, may be visual or analytical.	
Visual criteria (geology, visual clues to rock type, stratigraphy, colour)	
NNP	
NP	
AP	
NP/AP ratio	
Disposal location selection: criteria by which waste is directed to a particular disposal site.	
Estimate of accuracy of selection: volume (or %) of incorrect rock type incorporated into specified volumes.	
Estimate of particle size gradation of blasted waste rock (each rock type) going to the pile.	
Site specific limitations on waste handling procedures	

7.7 Post depositional monitoring

Data to confirm predictions should be collected during the operational phase of the mine. It is anticipated that this could include composition data from actual waste rock piles, weathering characteristics at various times of exposure, and drainage quality.

Ideally, each waste unit has an isolated catchment area and an associated sampling point. However, waste units that bridge several catchment areas may need more than one sampling location, and the results can be combined. Alternatively, external influences could affect the drainage quality at the sampling station and could make the site less representative of the waste rock composition. These influences should be described.

Actual waste dump design and construction:	
Dimensions: Height, areal extent, bench heights, bench width, bench slope, overall slope.	
Proportions of rock types	
Initial year of waste rock pile	These data are to provide a sense of how long the waste rock has been exposed to weathering.
Average yearly production added to pile.	
Final year of waste rock dump	
Spatial deposition of rock types	
Depositional sequence	
Depth of layer(s)	
Method of placement	Truck size, end dump, levelled by bulldozer...
Estimated level of compaction	
Estimate of accuracy of placement method (re: spatial deposition, depth of layers)	
Site specific limitations on waste handling procedures	
Cost data	Costs associated with field monitoring and construction methods.

Actual field values:	From samples collected from completed waste rock pile(s)						
	No.	mean	median	10%	25%	75%	90%
Elemental composition							
Weathering characteristics							
Soluble metal content (shake flask test results)							
% total sulphur							
% pyritic sulphur							
% sulphate sulphur							
% barium sulphate sulphur							
CO_2, as $CaCO_3$ equivalent							
Paste pH (laboratory test)							
Field pH							
NP							
AP (total or pyritic sulphur)							
NNP							
NPR							
Mean NP/mean AP							

Leachate quality (surface and/or groundwater; individual seeps or overall drainage):	Identify sampling location and describe possible external influences.						
Flow							
pH							
Alkalinity (mg/L)							
Acidity (mg/L)							
Net alkalinity (mg/L)							
Sulphate (mg/L)							
Dissolved copper (mg/L)							
Dissolved iron (mg/L)							
Dissolved zinc (mg/L)							
Dissolved manganese (mg/L)							
Dissolved calcium (mg/L)							
Dissolved magnesium (mg/L)							
Qualitative assessment	Acid drainage: yes or no Impacts to adjacent ground and or surface waters						
Regulatory requirements for leachate quality							

APPENDICES

APPENDIX A
SAMATOSUM MINE

A.1 Overview

The Samatosum Mine is located in south-central British Columbia, approximately 80 km north of Kamloops. The mine layout is shown in Figure A.1. The ore deposit is a stratabound quartz-carbonate vein deposit hosted by volcanic related mafic pyroclastics and metasediments. The mine operated from May 1989 to October 1992 producing approximately 565,700 /tonnes of ore and 9.1 million tonnes of waste rock.

Based on acid base accounting tests completed prior to mining, the waste rock could be classified as potentially acid generating or acid consuming according to lithology. The acid consuming rocks were primarily mafic pyroclastics (MAF) in the hanging wall of the deposit with a mean AP and NP of 73 and 377 kg $CaCO_3$ /tonne, respectively. The MAF material was predicted to form 58% of the total waste rock volume. The potentially acid generating rocks (PAG) were located primarily in the footwall of the deposit and included altered tuffs, cherts and argillites. The mean AP and NP of the PAG mixture were 95 and 50 kg $CaCO_3$ /tonne, respectively. The PAG material was predicted to be 42% of the waste rock (Denholm and Hallam, 1991a, b). The overall NP/AP ratio of the waste rock was estimated to be 3:1.

The control strategy planned for the waste dumps was to place the PAG rock between layers of acid consuming mafic pyroclastics. It was expected that alkalinity produced in the MAF layers would percolate downward through the PAG layers, and maintain neutral conditions throughout the dump. This was expected to limit the rate of sulphide oxidation and therefore prevent acid drainage generation and metal leaching. The layered design is shown in Figure A.2.

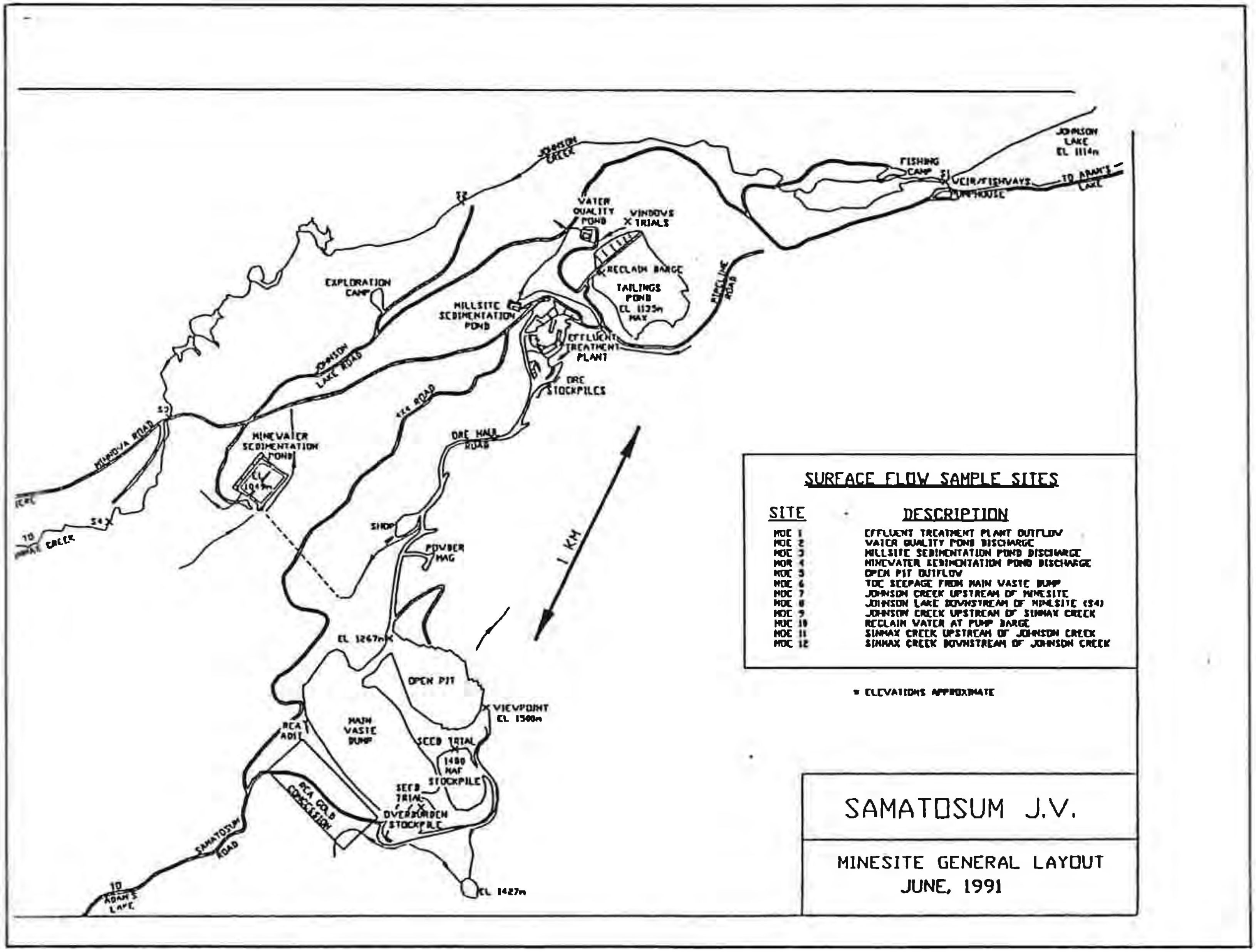

Figure A.1 Minesite Layout Showing Relative Location of the Open Pit, Waste Rock Pile and Water Quality Monitoring Stations (Minnova Inc., 1991 *in* Morin and Hutt, 1997).

Prior to mining, a series of column tests were initiated to test the layered control strategy (Denholm and Hallam, 1991a, b). Three control tests and three layered configurations were tested for up to 286 weeks. The layered columns had an NP/AP ratios of about 1.

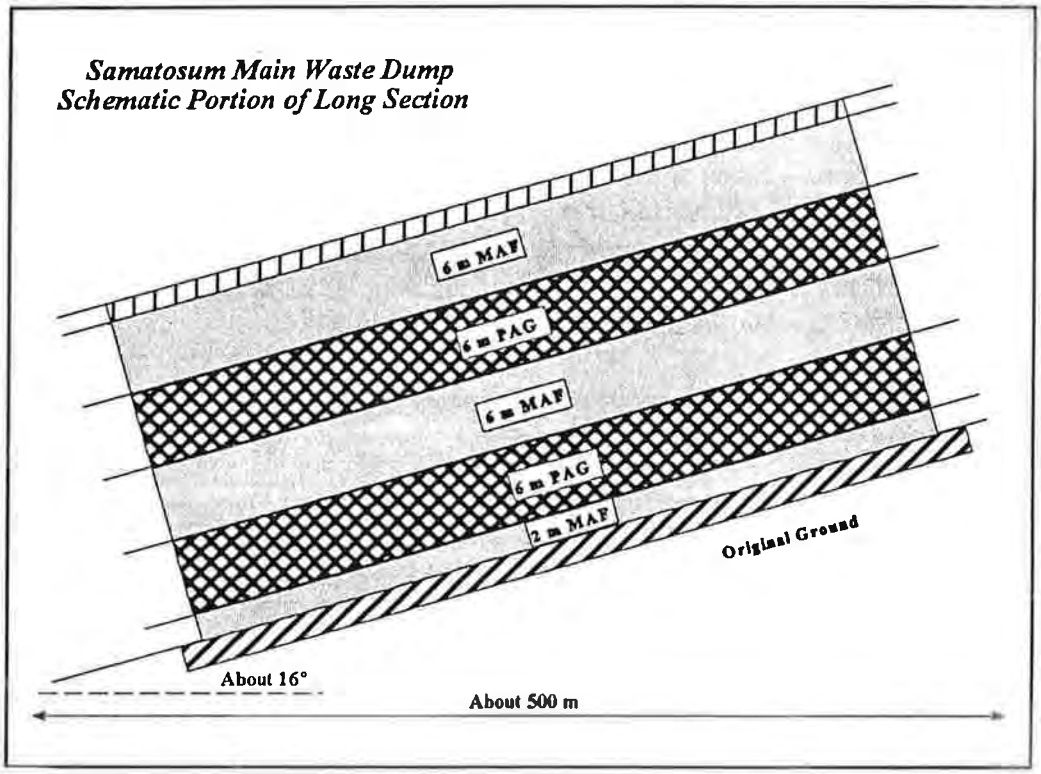

Figure A.2 Schematic Diagram of the Layered Dump Design.
(Denholm and Hallam, 1991a *in* Morin and Hutt, 1997)

The control tests confirmed the potential for acid generation in the PAG materials and demonstrated the acid consuming properties of the mafic pyroclastics. None of the layered columns produced acidic leachate; however, it is not clear whether this condition could have been maintained indefinitely. Calculations presented in Hallam Knight Piesold Ltd. (1992, 1994) indicated that *alkalinity* depletion would have occurred *after* the sulphides were depleted, while more recent calculations from those laboratory results and additional analyses by Morin and Hutt (1997) indicated that the *NP* would have been consumed *prior* to sulphide depletion. Sulphate production rates in the layered columns were only slightly lower than sulphate production rates in the pure PAG columns indicating that sulphide oxidation rates were not significantly retarded by the acid consuming MAF layer. Rather, it appeared that the drainage from the MAF portion of the column diluted and neutralized the leachate provided in the PAG portion of the column.

The waste rock dump was constructed in accordance with the original design, and was completed in November, 1992 (Minnova Inc., 1992 a, b). Monitoring stations at the toe of the dump (MOE 6A and 6B) have shown impending signs of net acidity since 1993, with increasing sulphate concentrations, decreasing alkalinity and increasing concentrations of manganese and zinc (Piteau Associates, 1996; Morin and Hutt, 1997). In the spring of 1996, acidic pH values were measured at both stations. A treatment plant was installed to treat seepage from the waste dumps and pit in the spring of 1997, prior to the onset of runoff.

The failure of the design to control acid generation in the dumps has been attributed to limitations in the physical aspects of the design rather than geochemical principals (Morin and Hutt, 1997). The thickness of the layers is believed to limit contact between alkaline and acidic seeps, limiting neutralization of the pH and attenuation of metals within the dump. Additionally, it is believed that flow channels in the dump did not allow all of the rock surfaces to contact all of the water, thus acidic seepage was not adequately buffered by carbonates in the flowpath.

A.2 General Information

Location	South Central British Columbia, Canada
Latitude/Longitude	51°09'N, 119°49'W
Mine owner	Inmet Mining Corporation (formerly operated by Minnova Inc.)
State of Operations	Operated May 1989-Sept 1992, closure
Mine contact	John Froese
Economic metals	Ag, Au, Cu, Zn, Pb
Open pit or underground	Primarily open pit, small u/g working mined by overhand cut and fill methods.
Ore production (tonnes per day, total)	465 tpd (565,700 tonnes)
Waste rock production (tonnes per day, total)	7900 tpd (9.1 million tonnes)
Minesite layout (maps if possible)	Figure A.1
Average annual precipitation, and/or range	940 mm
Mean monthly temperatures	13°C (July) to -12°C (Jan).
Other significant climatic information	Evapotranspiration 250-450 mm, peak monthly runoff in June, ~32% of annual flow

A.3 Geology

A.3.1 Regional Setting

The ore deposit is underlain by a complex assemblage of weakly metamorphosed volcanics and sediments of the Upper Paleozoic Eagle Bay Assemblage. Multiple stages of thrust faulting and folding during the Jura-Cretaceous Columbia Orogeny produced strongly foliated and overturned rocks trending northwesterly and dipping to the northeast. During the Mid-Cretaceous and Early Tertiary, the assemblage was intruded by granodiorite, quartz monzonite, quartz-feldspar porphyry, basalt and lamprophyre dykes. Eocene sedimentary and volcanic rocks of the Kamloops group and Miocene plateau lavas locally cover the formation.

A.3.2 Site Geology

The Samatosum deposit is a stratabound quartz-carbonate vein deposit hosted by volcanic-related mafic pyroclastics and metasediments. Prior to mining, the ore was located near the interface of sericitic altered tuffs and cherts in the structural hanging wall, and muddy tuffs and metasedimentary rocks in the footwall. This suite was overlain by the mafic pyroclastics. A cross-section of the orebody is shown in Figure A.3. The ore was composed of 11% pyrite, 32% quartz, and 19% dolomite. Economic minerals included: tetrahedrite (copper-iron-zinc-silver-antimony sulphide), sphalerite, galena, chalcopyrite, and electrum (native gold-silver mixture) (Morin and Hutt, 1997).

With the exception of localized quartz veins and quartzite, rocks that were adjacent to the ore body (sericitic altered tuffs, cherts, muddy tuffs, argillites) were net acid generating. The muddy tuffs in the footwall of the deposit contained up to 60% pyrite, and were identified early in the mine planning stages as a high risk material, thus, where possible, it was left in place. The "potentially acid generating" rocks were grouped into one management unit called the "PAG". Limited thin-section data on a composite PAG sample (Harris, *in* Morin and Hutt, 1997) indicated the sulphides are predominantly pyrite, with traces of sphalerite. The pyrite occurred in localized fine-grained disseminations or local aggregates of grains 5 to 100 microns in size. Pyrite in the rock matrix showed no evidence of oxidation. Oxidation products located along the interfaces of the grains were predominantly limonite. Carbonate was only a very minor constituent in these rocks, comprising less than 0.2% of the material. Other less reactive minerals were primarily quartz (83%), sericite (12%), and lesser amounts of chlorite.

The mafic pyroclastics (MAF) were net acid consuming. Examination of a thin section from a composite MAF sample (Harris, *in* Morin and Hutt, 1997) indicated MAFcontained up to 18% carbonate mineralization, minor pyrite (<1%), and trace chalcopyrite. Other less reactive minerals included chlorite (28%), plagioclase (32%), amphibole (18%), minor quartz, biotite and rutile. Although this this particular MAF

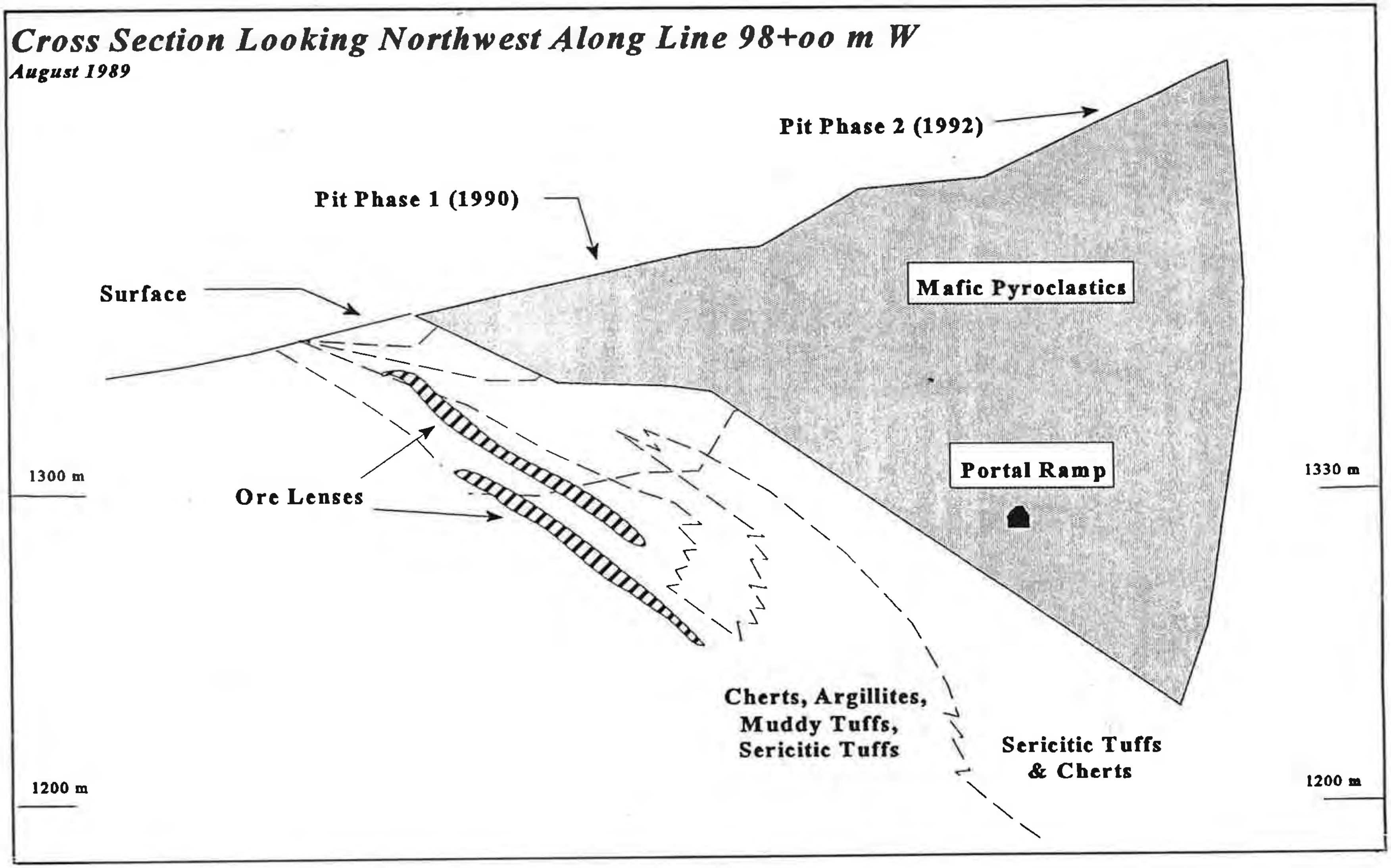

Figure A.3 **Cross-Section of the Orebody Showing Approximate Outline of the Open Pit**
(From Denholm and Hallam, 1991b, *in* Morin and Hutt, 1997).

sample had been subjected to ideal laboratory weathering conditions, the sulphide grains in the rock matrix appeared to be fresh. However, the weathered rock samples showed evidence of iron staining on particle surfaces in hand samples collected during the column dismantling (Morin and Hutt, 1997).

A.4 Static Tests

A.4.1 Methods

Standard acid base accounting (ABA) methods were used to characterize each of the major waste rock types in the open pit. The data compiled below are from the DBARD database, as reported in Morin and Hutt (1997).

A.4.2 Results

The results are presented in Table A.2.

Table A.2: ABA Results by Rock Type

Acid Consuming Mafic Pyroclastics (MAF):							
Mafic Pyroclastics	58% of material. Located in the upper portion of the open pit.						
Parameter	No.	mean	median	10%	25%	75%	90%
% total sulphur	8	2.33	2.14	0.37			4.35
Paste pH (laboratory test)	8	8.1	8.5	7.2			8.9
NP	8	377	330	51			775
AP (total sulphur)	8	73	67	12			136
NNP	8	305	267	-41			725
Mean NP/mean AP		5.2					

Table Cont....

Potentially acid generating "PAG" Rock Types:							
Sericite Tuffs	29% of material. Degree of sericite alteration decreases with distance from the ore zone.						
Parameter	No.	mean	median	10%	25%	75%	90%
% total sulphur	55	2.52	2.37	0.57			4.20
Paste pH (laboratory test)	55	8.2	8.4	7.1			8.8
NP	55	45	25	8			111
AP (total sulphur)	55	79	74	18			131
NNP	55	-34	-33	-106			32
Mean NP/mean AP		0.57					
Muddy Tuffs	3.4% of material. Abundant pyrite and only trace amounts of carbonate. Primarily in footwall rocks of open pit area.						
Parameter	No.	mean	median	10%	25%	75%	90%
% total sulphur	10	7.53	7.60	4.93			10.47
Paste pH (laboratory test)	10	6.1	6.5	4.2			7.3
NP	10	14	5	2			24
AP (total sulphur)	10	235	238	154			327
NNP	10	-221	-198	-325			-150
Mean NP/mean AP		0.06					
Quartzite/Quartz Veins	4.6% of material. The only acid consuming rock in the zone surrounding the orebody. This material is closely intermingled with the other metasediments and could not be handled separately.						
Parameter	No.	mean	median	10%	25%	75%	90%
% total sulphur	8	2.71	2.12	0.76			5.79
Paste pH (laboratory test)	8	8.4	8.4	7.6			9.1
NP	8	131	72	10			314
AP (total sulphur)	8	85	66	24			181
NNP	8	46	-2	-62			184
Mean NP/mean AP		1.54					

Table Cont....

Chert	0.42% of material						
	No.	mean	median	10%	25%	75%	90%
% total sulphur	6	7.99	5.71	2.86			15.4
Paste pH (laboratory test)	6	6.3	7.0	3.6			8.2
NP	6	23	8	1			62
AP (total sulphur)	6	250	178	89			481
NNP	6	-227	-164	-481			-34
Mean NP/mean AP		0.09					
Argillites	5.0% of material						
Parameter	No.	mean	median	10%	25%	75%	90%
% total sulphur	10	2.71	0.8	0.18			7.41
Paste pH (laboratory test)	10	6.9	7.7	4.3			8.4
NP	10	33	6	2			54
AP (total sulphur)	10	85	25	6			232
NNP	10	-52	-6	-229			34
Mean NP/mean AP		0.39					

Overall PAG	40% of material.				
Sub Unit	tonnes rock x 10^6	mean AP	mean NP	mean NNP	mean NP /mean AP
Sericite Tuffs	2.603	79	45	-34	0.57
Argillites	0.459	85	33	-52	0.39
Quartzite and Quartz Vein	0.421	85	131	46	1.54
Muddy Tuffs	0.306	235	14	-221	0.06
Chert	0.038	250	23	-227	0.09
OVERALL PAG (weighted average)	3.828	95	50	-44	0.53

Table Cont....

Cover/Foundation Materials							
Soil/Overburden	Excavated 0.73 million tonnes of overburden consisting of till and organic soils. Stockpiled for use as a cover material.						
	No.	mean	median	10%	25%	75%	90%
% total sulphur	2	0.31					
Paste pH (laboratory test)	2	8.1					
NP	2	19					
AP (total sulphur)	2	10					
NNP	2	10					
Mean NP/mean AP		1.9					

A.5 Kinetic Tests

A.5.1 Methods and Materials

Column tests were initiated prior to mining to test the appropriateness of the layered dump design. A total of 6 columns, including one acid consuming MAF sample, two acid generating PAG samples (one untreated, one acid washed prior to the test), and three layered configurations were tested. The PAG blend was made from a mixture of representative footwall rocks and consisted of 40% muddy tuff, 26 % sericitic tuff, 17% argillite and quartzite and 18% quartz vein and chert. The sample sizes, test duration and acid base accounting results for each of the columns are presented in the Table A.3, and the relative heights and material distributions in the columns are shown in Figure A.4 (Denholm and Hallam, 1991a).

The columns were constructed using 152 mm internal diameter clear acrylic pipe with a perforated basal plate and fibreglass screen. Distilled water was added to the columns at a rate of approximately 16 to 20 L/week, equivalent to 7.5 L/m^2 per day, which exceeded the site precipitation of 940 mm/yr. The columns were continuously exposed to humidified air.

Table A.3: Column Test Details and ABA Results

Column No., Material Tested	4 MAF	1 PAG	5 PAG (acid wash)	2 PAG/ MAF	3 PAG/ MAF	6 PAG/ MAF
Mass of sample (grams - dry weight)	11.5	28.9	17.2	46.2	46.2	35.8
Particle size/geometric surface area (m^2/kg)	1.27	1.66	1.66	1.52	1.51	1.46
Duration of test (weeks)	20	214	131	286	286	128
ABA						
% total sulphur	0.14	3.62	3.62	2.44	2.35	1.88
Paste pH (laboratory tests)	8.6	8.3	8.3	8.4	8.4	8.5
NP	111	62	62	78	79	86
AP (pyritic or total sulphur)	4	113	113	76	73	59
NNP	106	-52	-52	2	6	27
NP/AP ratio	25	0.54	0.54	1.0	1.1	1.5
Comments	• NP/AP ratio in layered columns is less than overall dump ratios (i.e conservative with respect to acid generating potential)					

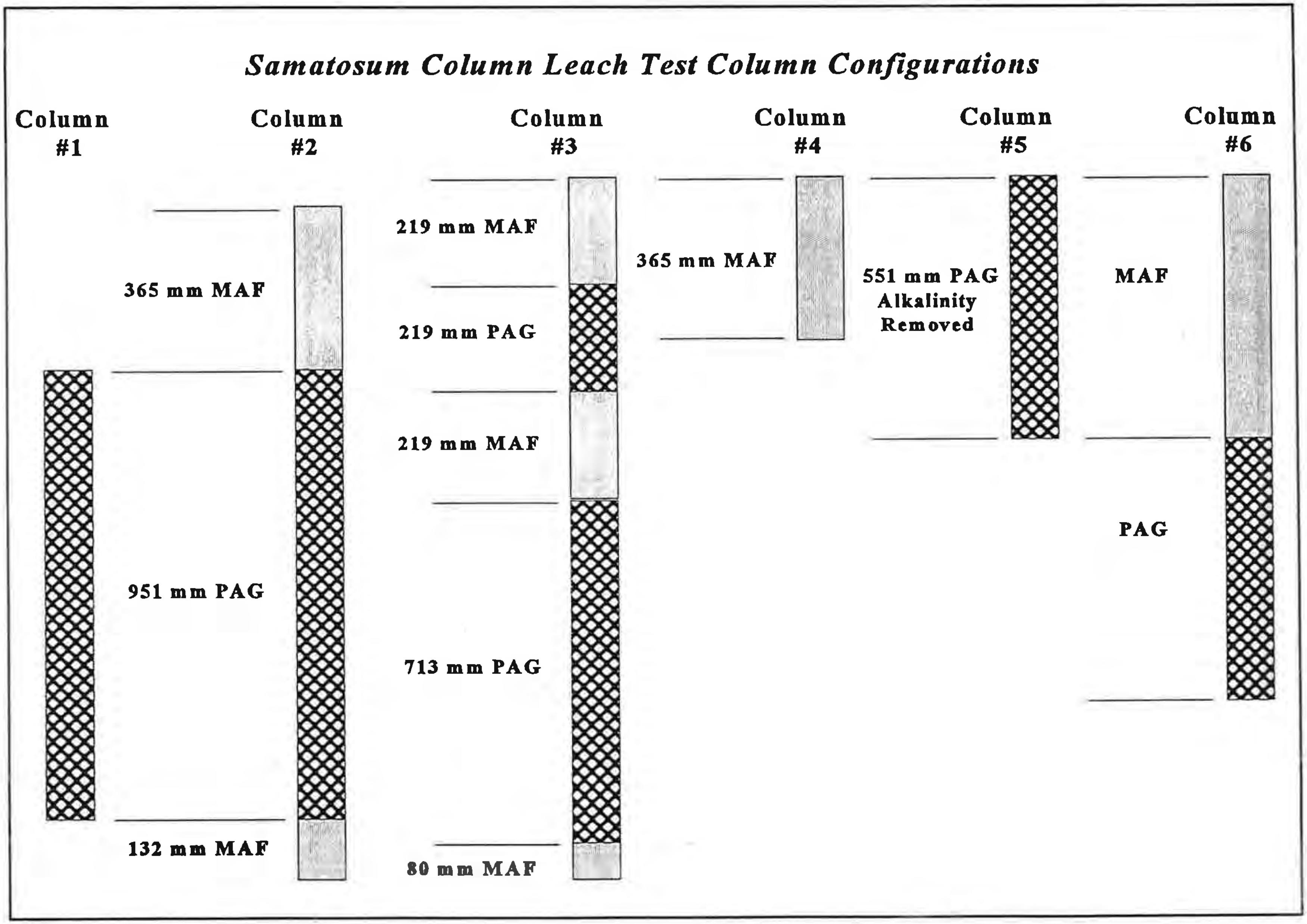

Figure A.4 Column Test Design (modified from Denholm and Hallam, 1991a, *in* Morin and Hutt, 1997)

A.5.2 Kinetic Test Results

The column test results are summarized in Table A.4. The mafic volcanics (Column 4) were acid consuming with low sulphate production rates and high alkalinity. Both of the potentially acid generating composites were acid generating with high sulphate production rates, essentially no alkalinity production and acidic effluent. The sample treated with acid prior to testing (Column 5) remained acidic throughout the test, and the untreated sample (Column 1) produced acidic leachate within 100 weeks of testing. Sulphate production rates in the PAG material were about 25 times faster than in the mafic pyroclastics.

The layered columns did not produce acidic effluent, but had high sulphate production rates, between 1/3 to ½ the rates observed in the pure PAG columns. Alkalinity production in the layered columns was between 1/10 to 1/5 of rates observed in the pure MAF columns. Metal release rates were low in the layered columns, with zinc rates in the range of 0.002 to 0.011 mg/kg/wk, and copper rates in the range of 0.0022 to 0.0038 mg/kg/wk. These were substantially lower than in the acid producing PAG columns, and close to the rates in the pure mafic column. Depletion calculations based on sulphate production rates indicated the sulphide would be completely consumed within 38 to 56 years (Hallam Knight Piesold Ltd., 1992, 1994; Morin and Hutt, 1997).

Depletion of the alkalinity or NP is more difficult to quantify, since there was no direct measurement of Ca or Mg in the column leachate. Hallam Knight Piesold Ltd. (1992) calculated depletion on the basis of alkalinity leach rates, and concluded that alkalinity would continue to provide buffering capacity until the sulphides were depleted. However, alkalinity leach rates typically underestimate NP depletion rates. Therefore, Morin and Hutt (1997) based NP depletion rates from NP depletion/sulphate production ratios found in geochemically equivalent materials at other sites, and indicated that the NP could be consumed much sooner than the sulphides.

Table A.4: Summary - Kinetic Test Results

Column No., Material Tested	4 MAF	1 PAG	5 PAG (acid wash)	2 PAG/ MAF	3 PAG/ MAF	6 PAG/ MAF
Result type (see Section 7.4)	1b	3	2	1	1	1
pH: - minimum - maximum - late stage	7.3 8.0 7.7	2.2 8.2 2.9	2.0 3.4 3.2	6.3 8.3 7.3	6.7 8.3 7.3	7.4 8.2 8.0
SO_4 production rate (mg/kg/wk)	2.7	71	82	28	34	18
difference from control $S0_4$ prod: MAF PAG	1x 0.04x	26x 1x	30x 1.2x	10.4x 0.39x	12.6x 0.48x	6.7x 0.25x
Alkalinity production rate (mg/kg/wk)	87	0.0	0.0	8.9	8.0	22
Acidity production rate (mg/kg/wk)	4.5	5.9	18	1.6	1.4	0.9
Net alkalinity production rate (mg/kg/wk)	82.5	-5.9	-18	7.3	6.6	21.1
Time lag to pH < 4.5 (weeks)	>	100	0	>	>	>
Years to total sulphur depletion	30	22	14	48	38	56
NP depletion rate (mg $CaCO_3$ eq/wk)	* Ca and Mg not measured in test, estimates in Morin and Hutt (1997) suggest NP in layered columns would be depleted in 20-40 years.					
Copper production rate (mg/kg/wk)	0.0011	0.049	0.085	0.0022	0.0038	0.0024
Zinc production rate (mg/kg/wk)	0.019	0.21	0.24	0.011	0.014	0.0020

Hallam Knight Piesold Ltd. (1992, 1994) and Morin and Hutt (1997) looked at the mineralogy of the weathered rock. Hallam Knight Piesold Ltd. concluded that the rate of oxidation in the columns was limited by mineral coatings. Morin and Hutt determined that carbonates in the fine fraction of the rock were close to depletion. Although Hallam Knight Piesold Ltd. (1992) and Morin and Hutt (1997) provided conflicting conclusions as to whether the columns with NP/AP ratios less than 1.5 would eventually produce acidic drainage, a more conservative mixture with an NP/AP ratio of 3.1 was expected not to produce acidic effluent given similar physical test conditions.

A.6 Waste Pile Design

The waste rock pile was originally designed to accommodate 8.1 million tonnes of waste rock consisting of 63% mafic pyroclastics (MAF), and 37% potentially acid generating materials (PAG) (Denholm, 1997, *pers comm.*). As shown in Figure A.3, the MAF rocks were located in the upper portions of the pit area, and were stockpiled for use in the pile construction when needed. The PAG rocks were treated as a single management unit because they would have been difficult to handle separately during mining. During construction of the waste rock pile, the volumes of each of the rock types changed slightly from the original design. The recorded construction volumes, the average geochemical composition of the MAF and PAG layers from pre-mining samples, and the average overall mixture are summarized in Table A.5 (Denholm and Hallam, 1991a, b), and these numbers have been used in this review. Final reconciliation in the 1994 reclamation report modified the relative volumes of material slightly (60% MAF and 40% PAG) (Denholm, 1997, *pers comm.*) but not significantly enough to alter the overall NP/AP ratio of the waste rock dump.

The Main Dump was located southwest of the open pit, with a plan area of approximately 14 ha and an average height of 30 metres. The dump was built on native till and organic soils with hydraulic conductivities of 10^{-5} to 10^{-7} m/s. Up slope diversions were used to reduce flow through the base layers of the waste rock. The layered design is shown in Figure A.1. The layered design was expected to prevent acidic effluent by reducing the rate of oxidation in the sulphide layers and by neutralizing any acidic seepage within the confines of the dump (Denholm and Hallam, 1991a, b). A 0.3 to 1 metre soil cover was placed over the surface of the pile.

TABLE A.5: Geochemical Composition of Dump

Main Dump			
Tonnage	9.1 million tonnes		
Overall proportions of rock types	MAF: 58% PAG: 42% (Sericite tuff: 28.5%) (Argillite: 5%) (Quartzite/quartz veins: 4.6%) (Muddy Tuff: 3.4 %) (Chert: 0.4%)		
Calculated Net ABA:	MAF	PAG blend	Overall Mixture (weighted average)
AP NP NNP NP/AP ratio	73 377 305 16.6	95 50 -43 0.56	82 240 158 3:1

A.7 Operational Data

The waste rock was classified as MAF or PAG on the basis of lithology and the earlier acid base accounting tests. The two classifications were visually distinct, and therefore visual characterization was used for approximately 90% of the placement of materials in the waste dump. Approximately 10% of the material consisted of altered mafic pyroclastics (MAF) which looked similar to the sericite tuffs (PAG). Where the identification of these materials (MAF versus PAG) was in question, acid base accounting tests were done to verify the material type (Denholm, 1997, *pers comm.*). The as-mined volumes were carefully recorded using mining software. Waste identified as MAF or PAG was sent to the appropriate disposal location, either the MAF stockpile, or to the appropriate PAG lift in the dump. Subclassifications of PAG materials by rock type were not recorded. Inadvertent mixing and/or blending of the PAG sub-units within the PAG layer occurred as a result of the mining sequence and handling, including blasting, loading, hauling and dumping. The pile was built in sections to minimize repeated handling of the MAF waste, however mine personnel have indicated some of the wastes were handled several times (Denholm, 1997, *pers comm.*). Costs for the repeated handling are not known.

The dump was constructed in sections using 45 tonne rear dump haul trucks, and built upwards in end dumped, horizontal lifts, of approximately 10 metres in thickness. These lifts were later graded downward by bulldozer to give a final layer thickness of about 6 metres parallel to the original ground surface, prior to the next layer being placed. Care was taken to avoid creating or leaving horizontal compacted surfaces in the layers (Denholm, 1997, *pers. comm.*) The sericite schist (PAG) was particularly friable, and susceptible to breakdown under handling and compaction.

An unspecified amount of mafic volcanics was placed on the lower dump face to stabilize the slope. The material was taken from the pit and is not included in the volumes calculations provided in Table A.5.

The final dump configuration was essentially built as designed, however local topographic variations result in variable dump thickness, ranging from 16 to 64 m in height. The thickness and location of the MAF and PAG layers in the dump was monitored during construction by surveying each lift after it had been graded. This data was recorded in a series of cross-sections of the dump (Minnova Inc., 1991). The geochemical variability within each of the layers is not known. Subsequent drilling (Piteau Associates, 1996) suggested that the relative volumes of PAG and MAF were close to the design volumes, except that the PAG layers appeared to be slightly thinner, and MAF layers somewhat thicker than anticipated. This difference may be attributed to the difficulty in conducting forensic drilling and logging of waste rock material, especially after the onset of sulphide oxidation has already occurred. A cross-section showing the distribution of MAF and PAG materials in the Samatosum waste dump is shown in Figure A.5.

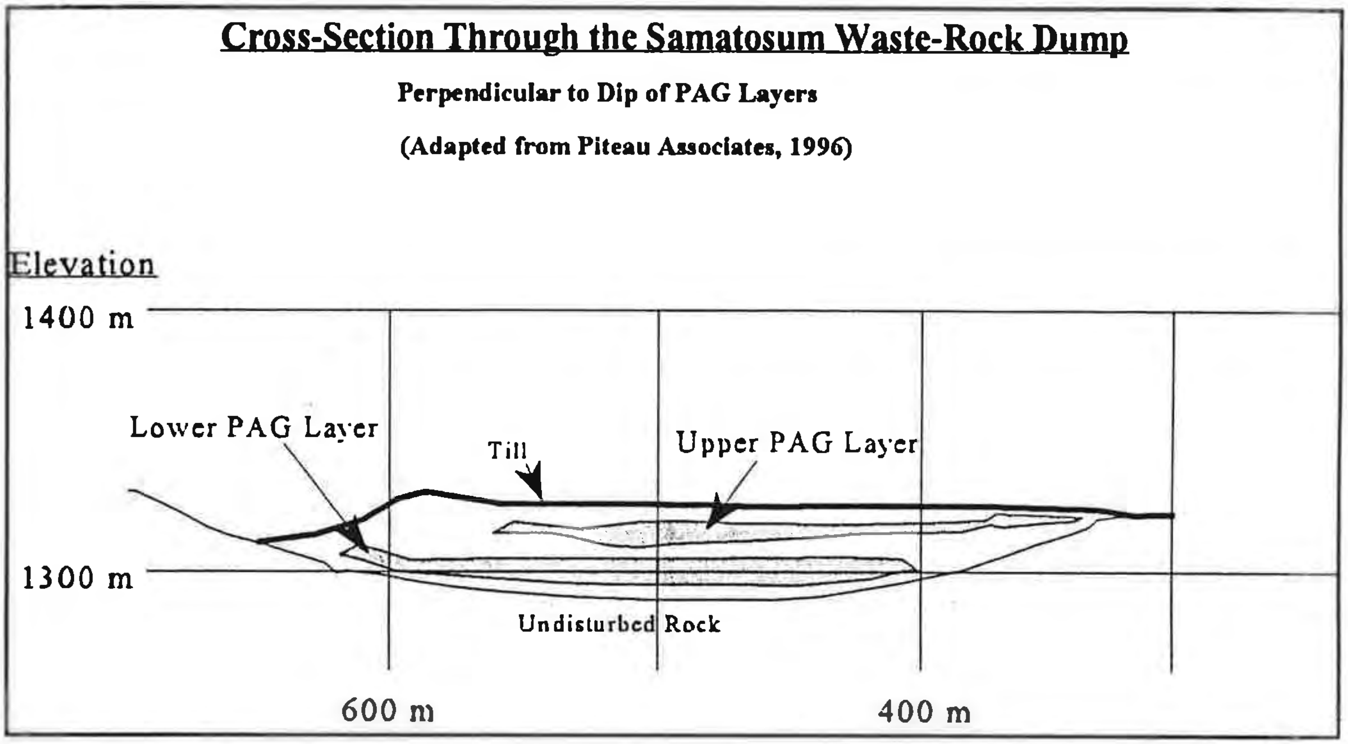

Figure A.5 Cross-Section Showing MAF and PAG Layers in the Samatosum Waste Rock Dump after Construction (adopted from Piteau Associates, 1996 *in* Morin and Hutt, 1997).

A.8 Post Depositional Monitoring

Monitoring efforts focused primarily on seeps at two locations at the toe of the dump (Stations MOE 6A and 6B). The overall dump and open pit drainage reports to a sedimentation pond downstream of the dump (Station MOE 4). A summary of the 1995 water quality from these 3 locations is provided in Tables A.6 and A.7. The seepage from the Main Dump had shown impending signs of acid generations over the previous several years, with increasing sulphate concentrations, decreasing alkalinity values and elevated metals values becoming more apparent in recent years. MOE 6B had consistent acidic leachate since the spring of 1995, and MOE 6A showed acidic values for the first time in the spring of 1996. Water at the sedimentation pond (MOE 4) was still alkaline, but the sulphate levels had increased steadily from 50 to 2000 mg/L since 1989. Elevated zinc and manganese concentrations at this location prompted the construction of a treatment plant in 1996.

Data on the appearance of the waste rock was limited, mostly because the top was covered with the soil cover, and the PAG layers were intentionally covered with 6 metres of MAF material. During a test-pit program, Piteau Associates (1996) observed iron staining in the MAF layer immediately below the till cover, indicating that some sulphide oxidation had taken place in the mafic volcanic rocks.

Piteau Associates (1996) also installed a set of piezometers in the dump. Water was observed in some of the piezometers immediately above or within a PAG layer. This data was inconclusive in defining the flow patterns in the dump. Piteau Associates (1996) interpreted the site data as being generally indicative of vertical flow. Other hydrological observations such as seepage from the PAG layers where the layers intersected ditches and pile slopes (Morin 1997, *pers comm.*) may have indicated that horizontal flow was occuring along the PAG layers. Preferential flow along the PAG layers had therefore been suggested as a possible explanation for the development of acidic effluent from this waste rock pile (Morin and Hutt, 1997).

The monitoring data from the whole waste rock pile indicated that the layered pile design was not meeting its initial expectations of preventing acidic effluent and metal leaching from the waste rock pile. However the presence of MAF material in the dump clearly provided some alkalinity and partial buffering.

Table A.6: Sites MOE 6A and 6B - 1995 Water Quality Results

Leachate quality	Seepage from Dumps (1995 data)					
Location	MOE 6A			MOE 6B		
	Mean	Max	Min	Mean	Max	Min
Flow	Range from 10 m^3/hr at freshet to dry in late fall, 6A is about 10x greater than 6B.					
pH	7.6	8.2	7.2	3.5	5.5	3.0
Alkalinity (mg/L)	225	275	150	0.0	0.0	0.0
Acidity (mg/L)	20	45	0	275	800	100
Sulphate (mg/L)	4000	4600	3300	9700	10200	8600
Dissolved copper (mg/L)	0.07	0.2	0.05	5	12	4
Dissolved iron (mg/L)	0.05	0.1	0.02	20	120	10
Dissolved zinc (mg/L)	5	22	2	37	48	22
Dissolved manganese (mg/L)	8	20	5	85	94	20
Dissolved calcium (mg/L)	500	na	na	500	na	na
Dissolved magnesium (mg/L)	800	na	na	1900	na	na
Changes/seasonal influences	Highest contaminant concentrations and lowest pH during spring flush			Highest contaminant concentrations and lowest pH during spring flush		
1991-1996 Trends	• ↑ SO_4 (500-4000) • ↓ alk (300-220) • Elevated metals values more frequent in later years			• ↑ SO_4 (1000-10000) • ↓ alk (500-0), ↑acy (0-700) • ↑ Mg consistent, ↑ Ca levelled off in 1992. • Elevated metals values more frequent in later years		

Table A.7: Site MOE 4 - 1995 Water Quality Results

Leachate quality	Regulatory Compliance Station			
Location	MOE 4: Mine and Waste Rock Sedimentation Pond (1995 Data)			
Parameter	Mean	Max	Min	Comments
pH	8.0	8.8	6.8	Stable
Alkalinity (mg/L)	130	180	110	Decrease from 250 to 100 since 1989
Acidity (mg/L)	7	20	0	
Sulphate (mg/L)	1900	2300	800	Increase from 50 to 2000 since 1989
Dissolved copper (mg/L)	0.002	0.004	0.001	
Dissolved iron (mg/L)	0.03	0.04	<0.03	
Dissolved zinc (mg/L)	0.04	0.3	0.02	Elevated values more frequent in later years
Dissolved manganese (mg/L)	0.1	1.1*	0.005	
Dissolved calcium (mg/L)	na	na	na	Increased through 1990 to 1993, but stopped monitoring
Dissolved magnesium (mg/L)	na	na	na	
Changes/seasonal influences	Seasonal trends not evident			
Qualitative assessment	Overall drainage quality meets standards, increasing frequency of elevated Mn and Zn concentrations are of concern.			
Regulatory requirements for leachate quality	* One exceedance of Mn criteria in 1995.			

Note: Water quality results were estimated from graphical presentations in Piteau Associates (1996) and Morin and Hutt (1997) and are not exact values.

A.9 Conclusions

- In the column tests, layering of MAF and PAG on the order of 0.2 to 1.0 m did not appear to affect reaction rates or geochemical behaviour within individual layers.

- Layering in the dump on the order of 2 to 6 m at an approximate overall NP/AP ratio of 3 did not prevent acidic effluent or metal leaching. It has been suggested (Morin and Hutt, 1997) that preferential flow channeling through the PAG layers as compared to the MAF layers did not allow the available neutralization in the dumps to be sufficiently utilized to neutralize the produced acid.

- The water quality results indicated that the layered dump design did not prevent acid effluent and metal leaching from the Samatosum waste rock pile

- The importance of considering field conditions and physical factors, such as layer thickness and hydrogeology, when interpreting laboratory data and extrapolating the results to the field is emphasized.

- Control of leachate acidity may be insufficient to adequately control metal leaching.

A.10 Possible Research Areas

- Because the MAF and PAG materials were easily identified using visual characterization, there was no operational testing data to show how heterogeneous each of the PAG or MAF layers are in the pile. ABA tests on grab samples from each of the layers would be useful to quantify the variability in the as-placed material. (It is believed that drill cuttings from piezometer installations in the dump were saved. Analyses of these samples may provide additional insight into layer characteristics.)

- Long term monitoring of the waste dump seepage (primarily MOE Station 6A) would provide information on long term water quality trends for comparison to column leachate characteristics. This may determine the relative influence of PAG and MAF layers on dump leachate quality. The addition of Ca and Mg analyses to the regular monitoring program would assist in future assessments.

- The potential for preferential flow paths to form within specific rock types (layers) should be examined.

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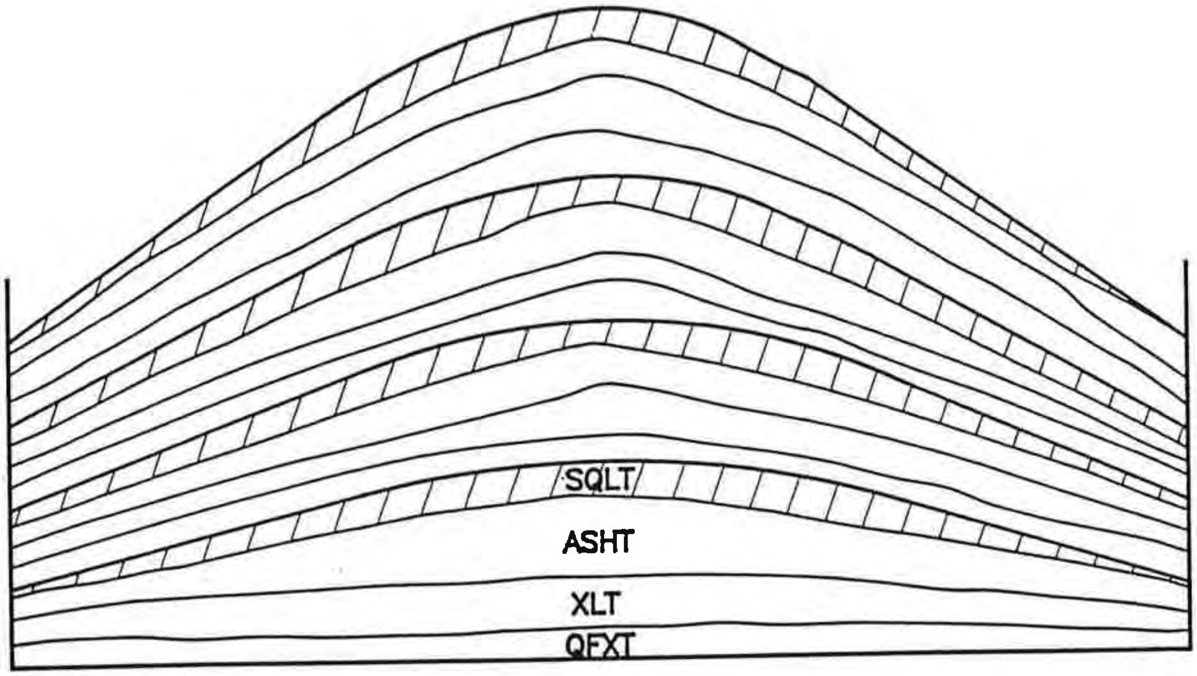
APPENDIX B
KUTCHO CREEK PROJECT

B.1 Overview

The Kutcho Creek Property is located approximately 110 km east of Dease Lake, British Columbia. The ore deposit is a polymetallic, massive sulphide deposit with possible reserves of 13.9 million tonnes of ore at a stripping ratio of approximately 6.6 tonnes of waste per tonne of ore. Detailed feasibility studies were carried out in 1985. The environmental studies included acid base accounting tests, humidity cell tests and a specific testing program to evaluate blending of acid consuming and acid producing rocks. Waste rock from the hanging wall of the deposit was considered net acid consuming while waste rock from the footwall was considered strongly acid generating.

Blending was tested at two different scales: small scale humidity cell tests and large scale field test plots. Both series of tests were intended to simulate the waste rock mixtures expected during the pre-production phase of mining and the fifth year of mining. These blends were composed of layers of three types of the hanging wall rock (chert/mafic ash tuff with a mean NNP of +64 kg $CaCO_3$ /tonne, crystal lapilli tuff with a mean NNP of +129 kg $CaCO_3$ /tonne, and quartz feldspar crystal tuff with a mean NNP of +79 kg $CaCO_3$ /tonne), and one layer of footwall rock (sericite quartz lapilli tuff with a mean NNP of - 473 kg $CaCO_3$ /tonne), with 4 repetitions of this sequence placed in relatively thin layers in both the humidity cells (<1 cm layers) and in the field test plots (~10 cm layers). The overall NP/AP ratio of the pre-production test and the 5-year test were estimated to be 1.1 and 0.6 respectively. These NP/AP ratios were substantially less than the NP/AP ratio of 1.7 estimated for the waste rock to be produced from the entire mine. The 5-year production blend was tested in duplicate, with a soil cover placed over one of the replicates. A schematic of the test plot layers is shown in Figure B.1.



Each layer = one bucket lift = 1.8 tonnes

Lithology:

Footwall

SQLT- Sericite quartz lapilli tuff

Hanging Wall

ASHT - Chert mafic ash tuff
XLT - Crystal lapilli tuff
QFXT - Quartz feldspar crystal tuff

Figure B.1 Field Test Plot Design Showing Layers (Rescan, 1992)

The blended small scale humidity cell test leachate had neutral pH and moderate sulphate levels throughout the 20 week testing period. Sulphate concentrations in these cells were higher than sulphate concentrations in pure hanging wall samples, but much lower than sulphate concentrations in the pure footwall samples. In these short term tests, blending was effective in delaying the release of acidic effluent as compared to the pure footwall samples. However, calculations indicated the rate of sulphide oxidation in the footwall portion was not reduced by blending. Rather, the sulphate concentrations in the blended cell appeared to be equivalent to the concentrations expected if four discrete cells of the four component materials were operated and the effluent streams were mixed.

The results from the 20 tonne field test pads showed higher metal and sulphate concentrations, but possibly lower loading rates than the laboratory tests. Estimates of total loads were effected by the inability to collect all the drainage from the field test pads. There was more variability in effluent water quality from the test pads, and hydrological effects related to the natural flushing regime and larger scale were evident. The covered test pads produced acidic effluent within the first year of operation. It was speculated that the cover reduced infiltration such that the sulphide oxidation products were concentrated in the leachate, although difficulties in accurately measuring the flow at this remote site limited this interpretation somewhat (Rescan, 1992). Alternative explanations could be that: a) the cover induced preferential flow through the acidic rock, or b) decreased infiltration rates allowed acid generating material to oxidize longer before water rinsed reaction products into alkaline areas, which were unable to neutralize the higher acid load. The uncovered 5-year and pre-production tests both showed incipient signs of acid generation, with periodic acidic leachate observed, usually following periods of high or prolonged precipitation. Long term projections indicated that the test pads would have produced acidic seepage over time.

At an NP/AP ratio of less than 1.1, this method of blending was clearly not an appropriate control for long term prevention of ARD for this site. However, the overall NP/AP ratio of waste at this site was projected to be significantly higher (1.7), and any reduction in the projected volumes of potentially acid generating waste would significantly improve this ratio. In the absence of tests carried out at more realistic NP/AP ratios, this disposal option should still be considered for portions of the deposit. However, a separate economic analysis of this disposal option indicated that the costs associated with blending large amounts of waste material in a full scale waste rock dump would be high (Rescan, 1992).

B.2 General Information:

Location	Northwest British Columbia, Canada
Mine owner	Sumac Mines Ltd. and Homestake Canada Inc.
Economic metals	Copper, zinc, silver, gold
State of operations	Feasibility studies in 1985
Open pit or underground	Planned open pit
Projected ore production	13.9 million tonnes grading 1.75% Cu, 2.47% Zn, 28.9 g/tonne Ag, 0.34 g/tonne Au
Projected waste rock production	93 million tonnes
Average annual precipitation, and/or range	440 - 650 mm
Other significant climatic information	Continental climate with long cold winters and short cool summers, dry.

B.3 Geology

The Kutcho Creek deposit is located within a volcanic island arc sequence composed of interlayered mafic and felsic volcanic and volcanoclastic rocks referred to as the Kutcho Creek Formation. The Kutcho Creek deposit is one of three elongate, massive sulphide lenses that appear to occur along a single time-stratigraphic horizon within the Kutcho Creek Formation. Each sulphide lense is a discrete, elliptical body consisting of layered massive sulphide and low sulphide intervals.

A cross-section of the ore-body is shown in Figure B.2. The ore sequence comprises three main lithological units. A thick meta-gabbro unit overlies a quartz feldspar crystal tuff, which overlies and may be transitional to a basinal lapilli tuff unit which hosts the massive sulphide horizons. The lapilli tuff, composed of a quartz sericite lapilli tuff and overlying crystal lapilli and mafic ash tuff, includes zones which have been enriched in carbonates, chert or pyrite. Pyrite enrichment is pervasive primarily within the footwall

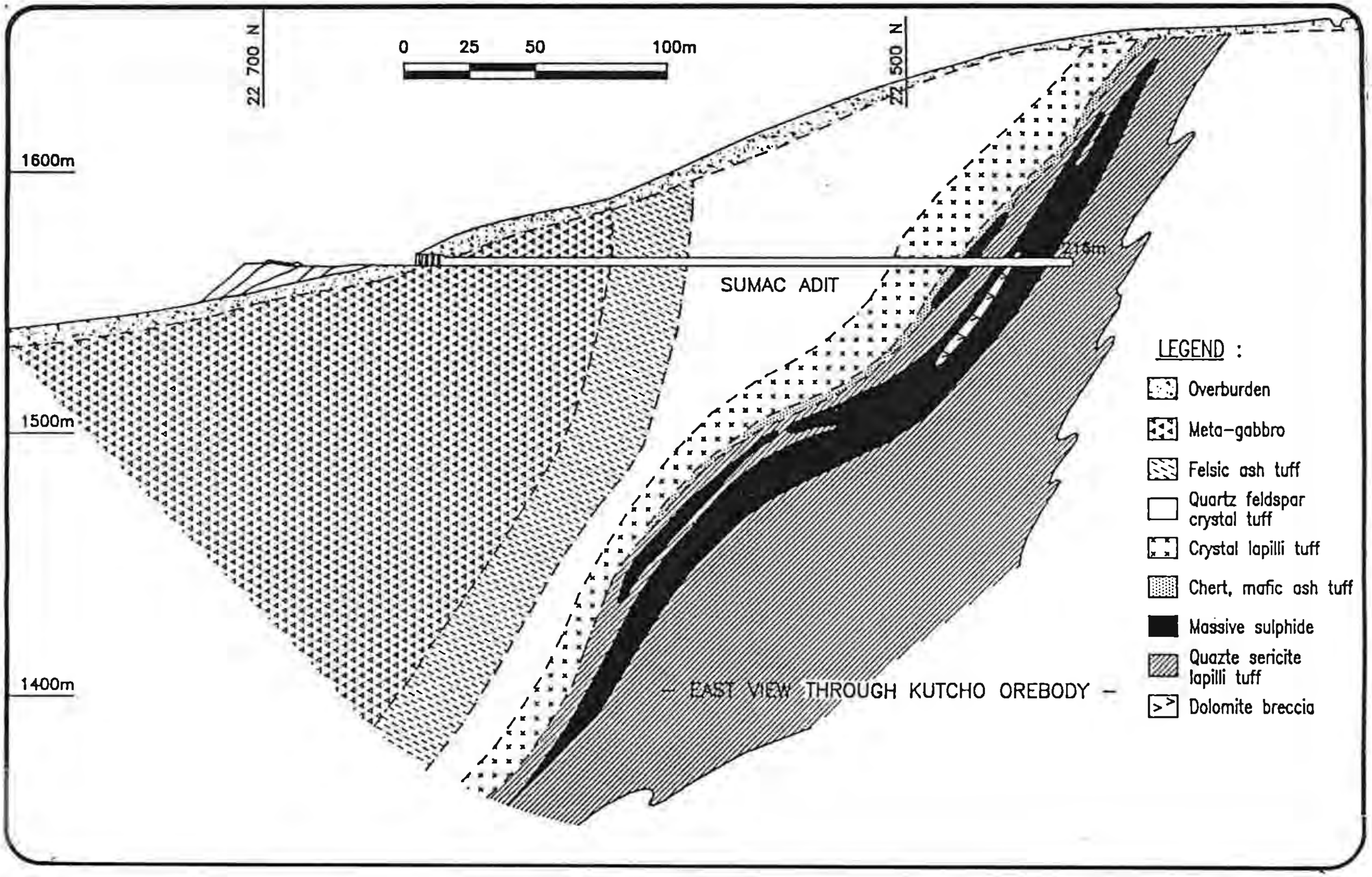


Figure B.2 Geological Cross-Section of Kutcho Creek Deposit (Rescan, 1992) quartz sericite lapilli tuff, whereas carbonate or chert enrichment occurs mainly in the hanging wall crystal lapilli and mafic ash tuffs (Rescan, 1992). A brief description of the mineralogy of each of the major rock types, and the relative proportions of these materials in the waste rock is presented in Table B.1.

Ore mineralization consists primarily of chalcopyrite, sphalerite, bornite and pyrite with minor amounts of chalcocite, tetrahedrite, galena, digenite, electrum and silver tellurides. Metal zonation has not been well defined within the deposit but in general there is a copper rich sulphide layer which overlies a mainly pyritic sulphide layer. Mineralogical studies indicated no barite was present. Mineralogical studies also indicated that most rock types from the Kutcho adit contained carbonate minerals, principally calcite, with quantities ranging from 2-57% of individual samples.

Table B.1 Mineralogy and Volume Estimates

Rock Unit	Mineralogical Description	% Volume in Waste Rock		
		pre	5 yr	total
Hanging Wall:				
meta-gabbro	A metamorphic "greenstone" comprised of plagioclase, biotite, amphibole and epidote crystals in a matrix of felsitic plagioclase quartz and sericite. 2-5% carbonates, principally as calcite.	0.7	21.2	23.8
felsic ash tuff	Fine-grained siliceous and felsitic aggregates with local sericite, carbonates and limonite. 3-20% carbonate, principally as calcite.	72.8	58.1	53.5
quartz feldspar crystal tuff	Probably derived from a quartz feldspar porphyry, the predominant minerals, quartz and feldspar, are crystals of 0.2 - 2.0 mm in size scattered through a fine-grained felsitic (plagioclase) matrix with interstitial flecks of sericite. Contains 1-2% carbonates, occurring as rare, small pockets and veniform gashes of calcite and limonite stained ankerite, minor pyrite as 1 mm euhedral grains, and trace amounts of chalcopyrite, sphalerite, bornite and galena.			
crystal lapilli ash tuff	Similar origin as quartz feldspar crystal tuff, but more intensely altered. Fine flakes of sericite have largely replaced the groundmass. Contains 18-26% carbonates as irregular pockets and intergrowths of calcite with lesser ankerite or siderite and dolomite and 1% pyrite as clusters of well formed, individual cubes.	5.5	9.7	12.8
mafic ash tuff	Similar to the other ash tuffs, with quartz chlorite, sericite, carbonate, pyrite and plagioclase. Contains 7% carbonate as porphyroblast-like grains and granular pockets. Fine disseminated sulphides are also a significant component (5%) occurring primarily as pyrite, with trace amounts of chalcopyrite and sphalerite.			
Footwall:				
quartz sericite lapilli tuff	Intensely altered rock comprised of dolomite, quartz, sericite, chlorite, chalcopyrite, pyrite and lesser amounts of sphalerite, tetrahedrite and galena. Pyrite occurs as sporadic clusters and strings of fine subhedral to euhedral crystals	21.0	11.0	9.9

B.4 Static Tests

B.4.1 Methods

ABA analysis was done according to standard US EPA methods (Sobek *et al.*, 1978). Additionally, Kutcho Creek samples were predigested in an acid solution for 24 hours to rinse away previously produced acid products (sulphate) prior to the test (core samples used in 1990 testing had been exposed to weathering condition in open core racks on site for up to 5 years prior to testing).

B.4.2 Results

The acid base accounting results are presented in Table B.2. The test results indicate that all of the hanging wall rocks were predominantly acid consuming, while the quartz sericite lapilli tuff in the footwall was net acid generating.

There is relatively low variability within most of the major rock units. However, the quartz feldspar crystal tuff and crystal lapilli tuff units had somewhat higher variability with one or two of the samples containing significantly higher sulphide concentrations than the bulk of the material.

The estimated overall ABA of the waste rock for various times during mining is presented in Table B.3. These were based on the production estimates in Rescan (1991) and the Phase II ABA test data. The results indicate that the overall NP/AP ratio of the waste rock would increase over the mine life from 0.85 in the pre-production phase to 1.7 at the end of mining.

Table B.2: Acid Base Accounting Results by Rock Type

Hanging Wall Rocks							
Rock type	**Meta-gabbro**						
Parameter	No.	mean	median	Percentile			
				10	25	75	90
% total sulphur	7	0.15	0.07	0.00	0.03	0.19	0.36
% pyritic sulphur	7	0.08	0.00	0.00	0.00	0.10	0.23
% sulphate sulphur	7	0.01	0.01	0.00	0.01	0.02	0.02
% CO_2	7	7.97	8.60	3.40	5.85	10.20	11.76
Paste pH	7	8.84	8.90	8.61	8.70	9.15	9.30
NP	7	187	201	116	144	212	251
AP (total sulphur)	7	4.7	2	0	1	6	11
NNP	7	183	199	113	143	204	241
mean NP/mean AP		40					
Rock type or composition	**Quartz feldspar crystal tuff**						
Parameter	No.	mean	median	Percentile			
				10	25	75	90
% total sulphur	15	1.33	0.05	0.00	0.01	0.28	1.41
% pyritic sulphur	15	1.31	0.00	0.00	0.00	0.18	1.22
% sulphate sulphur	15	0.01	0.00	0.00	0.00	0.01	0.02
% CO_2	15	5.8	3.7	1.6	1.9	5.9	11.2
Paste pH	15	6.87	9.10	8.50	8.90	9.25	9.46
NP	15	121.1	72	33	41	134	241
AP (total sulphur)	15	41.6	2	0	0	9	44
NNP	15	79.4	72	31	38	117	239
mean NP/mean AP		2.9					

Table Cont...

Table B.2 Acid Base Accounting Results by Rock Type (Continued)

Rock type	Crystal lapilli tuff						
				Percentile			
Parameter	No.	mean	median	10	25	75	90
% total sulphur	8	1.73	0.24	0.03	0.14	2.07	5.78
% pyritic sulphur	8	1.68	0.12	0.00	0.00	2.00	5.82
% sulphate sulphur	8	0.00	0.00	0.00	0.00	0.00	0.01
% CO_2	8	9.35	8.80	3.78	5.85	11.23	16.55
Paste pH	8	8.21	8.89	7.83	8.60	9.32	9.40
NP	8	183	162	83	118	221	321
AP (total sulphur)	8	54.4	8	1	5	65	181
NNP	8	128.8	125	1	75	178	259
mean NP/mean AP		3.4					
Rock type or composition	Chert, mafic ash tuff						
				Percentile			
Parameter	No.	mean	median	10	25	75	90
% total sulphur	5	0.09	0.09	0.01	0.04	0.13	0.16
% pyritic sulphur	5	0.01	0.00	0.00	0.00	0.02	0.03
% sulphate sulphur	5	0.00	0.00	0.00	0.00	0.00	0.01
% CO_2	5	2.2	1.7	1.6	1.7	1.9	4.8
Paste pH	5	9.07	9.10	8.87	9.00	9.30	9.36
NP	5	66.4	51	43	45	53	106
AP (total sulphur)	5	2.8	3	0	1	4	5
NNP	5	63.6	45	40	44	53	104
mean NP/mean AP		23.7					

Table Cont...

Table B.2 Acid Base Accounting Results by Rock Type (Continued)

Footwall Samples								
Rock type	**Sericite quartz lapilli tuff**							
				Percentile				
Parameter	No.	mean	median	10	25	75	90
% total sulphur	7	17.0	11.2	5.5	9.4	25.7	33.6
% pyritic sulphur	7	17.8	11.7	5.6	9.7	26.8	35.1
% sulphate sulphur	7	0.04	0.03	0.01	0.03	0.06	0.07
Paste pH	7	6.42	7.30	5.96	6.50	8.09	8.32
NP	7	60.1	42	1	2	97	143
AP (total sulphur)	7	534	350	172	293	803	1050
NNP	7	-473.4	-340	-951	-779	-258	-124
mean NP/mean AP		0.11					

Table B.3: Cumulative Estimates of Overall Waste Rock ABA During Mining

(Based on production volumes in Rescan, 1991)

Year	AP	NP	NNP	NP/AP
pre-production	151	128	-23	0.85
1	139	129	-10.3	0.93
2	126	131	5.1	1.04
3-5	90	141	51	1.57
6-10	81	140.3	59	1.71

B.5 Kinetic Testing

B.5.1 Methods

Modified humidity cells were used to test the weathering behaviour of each of the major rock types and 2 blended samples. One of these blended samples was tested in duplicate with a soil cover over one of the duplicates. However, the results from the soil cover test were not included for this review as the cover tended to mask the effects of blending. The modified test cells were constructed using 100 mm ID plexiglass pipe with a perforated acrylic base plate. The test samples were crushed such that 80% of the material passed through a 1/4 inch screen. One kg of the crushed sample was placed over a fine mesh screen at the base of the cell. The weekly cycle consisted of 3 days of moist air followed by 3 days of dry air and a rapid flushing cycle on the 7th day. In the flush cycle, 500 mL of water were added to the top of the cell and allowed to percolate through the crushed rock. The majority of the water, usually about 80% was recovered within 1 hour. The leachate was tested for pH, acidity, alkalinity, sulphate and a suite of 9 heavy metals.

The blended samples were intended to represent the pre-production waste blend and the 5-year production waste blend. It appears (Rescan, 1989, 1991, 1992) that the humidity cells were prepared in the same way as the field test pads, with the 4 test materials layered in 16 “lifts”. The 4 rock types used to construct the blends, their average NNP and the blend ratios are shown in Table B.4. These blends did not coincide with the pre-production and 5 year blends projected from the mine planning information, nor did they contain any of the strongly acid consuming meta-gabbro or felsic-ash tuffs. It is not known if the samples used to construct the layers were representative of the field test plot material, nor how either of these materials compared to the average ABA results for each of the rock types used in all calculations.

Table B.4 Composition of modified humidity cell and field test pad blends.

Rock Type	NNP	"pre-production" blend	"5-year" blend
Hanging Wall:			
crystal lapilli tuff	128	28.6%	28.6%
quartz feldspar crystal tuff	79	28.6%	28.6%
chert, mafic ash tuff	64	28.6%	14.3%
Footwall:			
sericite quartz lapilli tuff	-473	14.3%	28.6%

Note: The weighted NNP for the pre-production and 5-year blends are 7.2 and -72.3 kg $CaCO_3$ eq/tonne. The weighted NP/AP ratios for these blends are 1.1 and 0.61 respectively.

B.5.2 Results and Discussion:

The details of the humidity cell results are presented in Rescan (1992). A summary of the test results of the individual components of the blended tests and of the overall blended tests is provided in Table B.5. The results showed that effluent from the three hanging wall materials and the blended sample had a neutral pH throughout the 20 weeks of testing, while the footwall sample produced acidic effluent throughout the length of the test.

Sulphate production rates in the hanging wall materials ranged from an average of 0.58 mg/kg/wk in the crystal lapilli tuffs to 20.2 mg/kg/wk in the chert and mafic ash tuff unit. The footwall samples had an average sulphate production rate of 94.3 mg/kg/wk, reflecting the more advanced stage of oxidation and the higher total sulphur content of these samples. Sulphate production in the blended samples was intermediate, with weekly production rates of 19.1 and 40.2 mg/kg/wk.

Table B.5: Summary - Kinetic Testing Results

Parameter	Unblended cells*				Blended cells	
	CLT (n=2)	QFCT (n=6)	C,MAT (n=1)	SQLT (n=3)	"pre-prod."	"5-year"
pH: min max	7.52 8.98	7.24 8.53	7.02 7.98	2.94 7.67**	7.28 7.90	7.13 7.57
alkalinity production (mg/kg/wk)***	13.10	9.07	6.11	2.15	6.64	5.2
acidity production (mg/kg/wk)	0.33	0.72	0.66	11.4	0.42	0.62
weekly SO_4 production (mg/kg/wk)	0.58	3.33	20.2	94.3	19.1	40.2
total SO_4 production (mg/kg)	58	104	601	4002	730	1295
weeks to S depletion	9010	1178	5700	4600	4835	3890
Metals: (mg/kg/wk) Cd Cu Pb Zn	- 8.6e-4 - 8.4e-3	- 2e-3 - -	8.6e-5 9.5e-4 - 7.9e-3	1.0e-2 2.7e-0 2.9e-3 2.0e-0	4.6e-5 1.2e-3 3.0e-4 6.1e-3	2.5e-3 1.2e-3 3.6e-4 8.0e-3

Notes: CLT - crystal lapilli tuff
QFCT - quartz feldspar crystal tuff chert
MAT - mafic ash tuff
SQLT - sericite quartz lapilli tuff

* average of unblended cells from each rock type
** two of the cells were producing acidic effluent throughout testing.
*** production rates are based on final 5 cycles of testing

The actual sulphate production rates from the blended cells were compared to calculated production rates based on a weighted average of production rates from the non-blended tests. This comparison indicated there was not a significant difference between the calculated and actual production rates: in the pre-production cell, the calculated weekly sulphate rate is 20.4 mg/kg/wk as compared to the measured value of 19.1 mg/kg/wk; and in the 5-year cell, the calculated weekly sulphate rate was 31.0 mg/kg/wk as compared to the measured value of 40.2 mg/kg/wk. This comparison indicated that sulphide oxidation rates were not limited by alkalinity provided by the non-acid generating components of the blend over the 20 week period.

At the time these tests were initiated, it was not standard practice to measure the calcium and magnesium concentrations of the leachate. Thus it was not possible to directly measure NP depletion, or to compare NP depletion rates with sulphide depletion rates. Long term predictions of NP and sulphide depletion were done by extrapolating the cumulative sulphate production data (Rescan, 1992). The results indicated that NP in the pre-preproduction blend would be consumed in about 100 weeks and that sulphides in this sample would persist for an additional 85 weeks. NP in the 5 year blend would be consumed within 150 weeks and the sulphides would persist for an additional 250 weeks (Rescan, 1992). Sulphide depletion estimates based on the weekly sulphate production rates measured over the last 5 cycles indicated the sulphides would persist for much longer, 4800 and 3900 weeks for the pre-production and 5-year blends respectively.

Metal production rates in the blended cells were slightly higher than production rates measured in the non-acid generating hanging wall samples, but were closer to the hanging wall rates than the acid generating footwall rates, indicating that pH was an important control on metal leaching rates in these cells.

B.6 Field Testing Program

B.6.1 Methods

Three large scale field tests were initiated in August 1989 to test blending of the acid consuming hanging wall rocks with the acid generating footwall rocks as a long term control for acid generation at this site. The blended samples were intended to represent the pre-production and 5-year production waste rock. However the NP/AP ratios of the tested blends (1.1 and 0.63 for the pre-production and 5-year samples respectively) did not accurately reflect the predicted pre-production and 5-year NP/AP ratios of 0.85 and 1.57. The 5-year blend was tested in duplicate, with a soil cover over one of the duplicates.

The test pads were constructed with a plywood base, and open wooden sidewalls. The base was lined with a polyethylene liner and was sloped so that runoff was channelled to a final collection point. Baffles directed water into six collection areas. The pad design details are summarized in Table B.6.

Rock was placed in the pad in 16 layers with 4 repetitions of the 4 rock types used. From top to bottom, each repetition consisted of the following materials:

- Sericite quartz lapilli tuff - mean NNP of - 473.4
- Chert mafic ash tuff - mean NNP of +63.6
- Crystal lapilli tuff - mean NNP of +128.8
- Quartz feldspar crystal tuff - mean NNP of +63.6

Table B.6: Design Details of the Waste Rock Field Test Plots

Plan area of test plots	12 ft x 12 ft (3.28m x 3.28m)
Catchment area	12 ft x 12 ft
Foundation material (recharge/discharge)	Polyethylene liner, with geotextile fabric protection layer
Average height	Sidewall height of 1.35m. Waste rock placed to form a pyramid whose apex height at the centre of the heap was 1.67 m with a 30 degree slope angle (see Figure B.2).
Starting year of waste rock pad	Constructed August 1989, monitoring began July 1990
Last year of waste rock pad	Monitoring ended October 1991, not known if pads are still intact
Tonnage	20 tonnes
Prevention measures	Blending (layered)

The range of ABA values typical of each material are shown in Table B.2. The material was placed in cubic metre lifts then spread with a backhoe. This allowed some mixing of the layers with the adjacent layers. The resulting layers were about 10 cm thick. Weighted NNP ratios (1:1.1 acid generating to acid consuming, and 2:1 acid generating to acid consuming) were determined by multiplying tonnes of each rock type by their corresponding (mean) NNP value. The composition of each test pad using more conventional ABA calculations were developed by MEM Inc. and are presented in Tables B.7, B.8 and B.9.

B.6.2 Field Test Results

The field test results are presented in Tables B.7, B.8 and B.9. Both of the uncovered test plots showed incipient signs of acid generation, with periodic low pH effluent, usually following heavy or prolonged rainfall. The leachate from these pads had high sulphate and moderately high metal levels. The covered test plot had consistently low pH starting in the fall of 1990, with high sulphate and metal concentrations. Since the only significant difference between this test and the uncovered 5-year test was a reduction in infiltration, Rescan (1992) speculated that the sulphide oxidation products were more concentrated in the leachate and overwhelmed the available neutralization potential. This interpretation is somewhat limited by the quality of the flow data from the test pads located in this remote site (Rescan, 1992) in that not all drainage from the test pads was collected. An alternative explanation could be that the cover induced preferential flow through the acidic rock.

A comparison of the two uncovered tests indicated that the pre-production test leachate (NP/AP ratio = 1.1) had consistently lower sulphate and metal concentrations than the 5-year test (NP/AP ratio = 0.6). Since the materials used in the two blends were similar, this difference likely reflected the overall difference in the initial sulphide content.

Table B.7: Field Test Pad Results, Test Cell 1 (Pre-production)

Composition (weighting)	1/2 part sericite quartz lapilli tuff (4 x 0.7 tonne lifts) 1 part chert mafic ash tuff (4 x 1.4 tonne lifts) 1 part crystal lapilli tuff (4 x 1.4 tonne lifts) 1 part quartz feldspar crystal tuff (4 x 1.4 tonne lifts)						
Prevention measures	Blending (layered)						
Calculated ABA values:	No.	mean	median	10%	25%	75%	90%
% total sulphur		3.34	1.71				
% pyritic sulphur		3.39	1.71				
% sulphate sulphur		0.01	0.004				
NP		104.6	87				
AP (total sulphur)		114.3	54				
NNP		10	21				
NP/AP ratio		1.09	1.61				
Seepage quality:	Test cell leachate						
Field pH	Initial pH of about 6.0 during 07/90 and 08/90, sharp decrease in 09/90 and fluctuations for remainder of 1990. pH recovered in early 1991 to about 7.0 and remained neutral to 08/91 when the data logger malfunctioned.						
Sample Date Laboratory Data	09/89	07/90	08/90	09/90	07/91	07/91	10/91
Laboratory pH	6.9	3.37	6.65	3.62	3.4	6.2	4.2
Alkalinity (mg/L)	-	<1.0	29.0	3.9	-	-	-
Acidity (mg/L)	-	188	63.0	366	393	-	-
Sulphate (mg/L)	-	3650	2890	8470	3530	7890	1990
Dissolved copper (mg/L)	13.0	8.45	3.70	61.2	-	14.1	5.65
Dissolved iron (mg/L)	0.30	3.56	<0.03	1.18	-	<0.005	<0.005
Dissolved zinc (mg/L)	271	58.2	25.0	150	-	56.2	25.5
Dissolved manganese (mg/L)	<0.005	34.5	17.2	94.2	-	65.1	23.8
Qualitative assessment	Incipient signs of acid generation, periodic low pH and high metals.						

Table B.8: Field Test Pad Results, Test Cell 2 (5-Year Production, Uncovered)

Composition (weighting)	1 part sericite quartz lapilli tuff (4 x 1.4 tonne lifts) 1/2 part chert mafic ash tuff (4 x 0.7 tonne lifts) 1 part crystal lapilli tuff (4 x 1.4 tonne lifts) 1 part quartz feldspar crystal tuff (4 x 1.4 tonne lifts)						
Prevention measures	Blending (layered in 1.4 tonne lifts)						
Calculated ABA values:	No.	mean	median	10%	25%	75%	90%
% total sulphur		5.77	3.30				
% pyritic sulphur		5.93	3.39				
% sulphate sulphur		0.014	0.009				
NP		113	86				
AP (total sulphur)		180	103				
NNP		-67	-34				
NP/AP ratio		0.63	0.83				
Seepage quality:	Test cell leachate						
Field pH	Greater than pH 6 from 07/90 to 09/90 then sharp decrease to 3.0 in 07/91 following a heavy week of rain. Later in 07/91, the pH had re-stabilized to between 7 and 8 and remained stable until Aug 18, 1991 when the data recorder malfunctioned.						
Sample Date Laboratory Data	09/89	07/90	08/90	09/90	07/91	07/91	10/91
Laboratory pH	4.0	6.48	7.23	5.84	3.3	7.1	3.5
Alkalinity (mg/L)	-	37.0	39.2	39.0	-	-	-
Acidity (mg/L)	-	258	52	196	760	-	-
Sulphate (mg/L)	-	5700	3890	8980	6140	7570	2480
Dissolved copper (mg/L)	-	5.14	1.46	5.00	-	4.98	4.40
Dissolved iron (mg/L)	-	<0.03	<0.03	<0.03	-	1.20	0.259
Dissolved zinc (mg/L)	-	147	27.3	143	-	65.1	33.2
Dissolved manganese (mg/L)	-	82.2	40.5	125		94.0	41.2
Qualitative assessment	Incipient signs of acid generation with periodic low pH, high sulphate concentration and very high metal concentrations.						

Table B.9: Field Test Pad Results, Test Cell 3 (5-Year Production, Covered)

Composition (weighting)	1 part sericite quartz lapilli tuff (4 x 1.4 tonne lifts) 1/2 part chert mafic ash tuff (4 x 0.7 tonne lifts) 1 part crystal lapilli tuff (4 x 1.4 tonne lifts) 1 part quartz feldspar crystal tuff (4 x 1.4 tonne lifts)						
Prevention measures	Blending (layered in 1.4 tonne lifts), and 6 inch soil (till) cover to reduce infiltration						
Calculated ABA values:	No.	mean	median	10%	25%	75%	90%
% total sulphur		5.77	3.30				
% pyritic sulphur		5.93	3.39				
% sulphate sulphur		0.014	0.009				
NP		113	86				
AP (total sulphur)		180	103				
NNP		-67	-34				
NP/AP ratio		0.63	0.83				
Seepage quality:	Test cell leachate						
Field pH	Initially ~ pH 7, sharp decrease to pH 4 in 08/90 following a heavy rainfall, fluctuating between pH 3 and 6 for remainder of 1990. Consistent low pH throughout 1991 season.						
Sample Date Laboratory Data	09/89	07/90	08/90	09/90	07/91	07/91	10/91
Laboratory pH	3.0	3.45	6.62	2.97	3.4	2.7	3.0
Alkalinity (mg/L)	-	<1.0	46.5	<1.0	-	-	-
Acidity (mg/L)	-	418	120	1230	334	-	-
Sulphate (mg/L)	-	2960	4100	6030	3180	6050	2410
Dissolved copper (mg/L)	-	35.9	1.67	105	-	46.4	8.75
Dissolved iron (mg/L)	-	38.5	0.073	120	-	271	16.4
Dissolved zinc (mg/L)	-	117	48.2	254	-	101	54.9
Dissolved manganese (mg/L)	-	39.4	45.7	119	-	98.0	47.5
Qualitative assessment	Acid drainage, cover reduces total loads, but not concentrations						

B.7 Conclusions

- In the short term blended small scale humidity cell tests, blending appears to have delayed the release of acidic effluent from the cells as compared to the potentially acid generating footwall material. However, blending did not reduce the rate of sulphide oxidation that would have been produced by each individual unblended material, even though the layers were only about 1 cm thick.

- Over the two year testing period, the field test pads all showed incipient or pervasive signs of acid generation. The covered 5-year blend had the worst performance, with consistently low pH observed during the first year of monitoring. The uncovered pre-production blend with an NP/AP ratio of 1.1 performed slightly better than the 5-year blend with an NP/AP ratio of 0.6. However, long term projections indicated that both uncovered test pads would have produced acidic seepage over time.

- At the tested NP/AP ratios of 1.1 or less, blending was not an appropriate control for long term prevention of ARD at this site. However, the projected overall NP/AP ratio for the Kutcho Creek waste rock was higher than any of the tests (1.7), thus this control option may have more merit for this site than the results obtained from this testing program indicate.

- Separate economic analysis of this disposal option (Rescan, 1992) indicated that the costs associated with blending large amounts of waste material in a full-scale waste rock pile would be high, as compared to other disposal options.

B.8 Possible Research Areas

- Examine the sensitivity of the weighted ABA calculations to variations in the ABA data within a rock unit.

- Collect and test leachate from the test pads (if they still exist) to see if there are any changes to the overall water quality and if there are still differences evident between the pre-production and 5-year tests.

- Dismantle and examine the test pads for evidence of localized acid generation in the footwall layers, stored oxidation products and flow paths. Conduct post-test ABA's mineralogical tests and extraction tests on the test pad materials to quantify the NP/AP ratio and it's variability in each of the blends.

B.9 Key References

Rescan Environmental Service Ltd. 1989. Kutcho Creek Project: Acid Generation Testwork, Phase I Final Report. *Prepared for* Sumac Mines Ltd. and Homestake Mineral Development Co. with additional funding by the B.C. Mineral Development Agreement.

Rescan Environmental Service Ltd. 1991. Kutcho Creek Project: Acid Generation Testwork, Final Report. *Prepared for* Sumac Mines Ltd. and Homestake Mineral Development Co. with additional funding by the B.C. Mineral Development Agreement.

Rescan Environmental Service Ltd. 1992, Kutcho Creek Project: Blending and Segregation Acid Generation Testwork, Final Report, March 1992. *Prepared for* Sumac Mines Ltd. and Homestake Mineral Development Co. with additional funding by the B.C. Mineral Development Agreement. B.C. AMD Task Force Report.

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APPENDIX C
STRATMAT DEPOSIT, HEATH STEELE

C.1 Overview

The Stratmat site is located on the Heath Steele Mine property north of Newcastle, New Brunswick. The deposit was mined from 1989 to 1993. The mine area includes the Stratmat and N-5 open pits and a series of underground workings which intersect both pits. Approximately 1.9 million tonnes of acid-generating waste rock were stored above ground, adjacent to the Stratmat open pit. Waste rock associated with the mining activity was strongly acid generating with approximately 19% pyrite, trace sphalerite, galena and chalcopyrite. One of the control options studied at this site was to blend or layer the waste rock with limestone to limit the production of acidic effluent, and potentially delay or reduce acid generation, prior to possible future disposal in one of the mined out pits. This review summarizes the results of two of the studies on blending and layering limestone with Stratmat waste rock.

The first study on limestone addition involved field monitoring data from a 1.5 million tonne waste rock dump constructed between 1989 and 1991. Limestone was added to the top of each 3 metre lift at a rate of about 10 tonnes limestone per 1000 tonnes of rock (1% limestone) to prevent or delay the onset of acid generation. The limestone addition was partly based on bench-scale studies, reported in Sheremata *et al.* (1991), which suggested that approximately 10 kg/tonne (1%) of limestone would be sufficient to limit acid generation (Heath Steele and Noranda Technology, 1994). These measures proved to be unsuccessful and acid seepage was detected at the toe of the Stratmat waste rock pile during the spring of 1991 (Heath Steele and Noranda Technology, 1994). Limestone addition was halted at that time.

The second study was carried out by the Noranda Technology Centre (NTC) with financial support from the Centre de Reserches Minerales (CRM) and from the MEND program (Payant and Yanful, 1997). The objective of this study was to evaluate the relative effectiveness of various ARD control techniques, including blending with limestone. Waste rock from Stratmat was one of the two waste materials used in this experiment. The blending experiments included a series of 20 kg column tests and 170 kg field test lysimeters. The field and laboratory tests were carried out in triplicate, with limestone blends of 1 and 3% and a control test with no limestone. The laboratory tests were run for 154 weeks and the field tests for 125 weeks.

Both the testing programs and the field studies showed the inadequacy of the limestone additives in preventing ARD. However, the blended column and lysimeter tests indicated that limestone addition was a viable option for delaying the release of acidic effluent and reducing the rate of sulphide oxidation, provided there was sufficient mixing of the waste rock and limestone. An inadequate overall NP/AP ratio appeared to be the main reason acidic effluent was not prevented in the long term. Calculations (Section C.4) indicated that a blend with 3% limestone by mass would have had an NNP of - 248 kg $CaCO_3$ /tonne, and an NP/AP ratio of 0.16. Thus the test material was well below the NP/AP $ratio_{crit}$ (1 to 2 range) that would theoretically prevent acidic effluent from a homogenous blend.

The full-scale test with limestone layers did not perform as well as predicted by the laboratory studies in delaying the release of acidic effluent. The inability of the limestone to delay the production of acidic effluent was attributed to absorption and oxidation of iron and manganese on limestone surfaces in contact with the acid generated in the untreated areas of the pile, which was thought to render the limestone non-reactive (Heath Steele and Noranda Technology, 1994). This ineffectiveness was assisted by the inadequate physical contact between the thin (15 cm) limestone layers and the much thicker (300 cm) waste rock lifts.

C.2 General Information

Location	Near Newcastle, New Brunswick
Mine owner	Noranda
State of Operations	Operated 1989-1993, on-going closure activities
Mine contact	Leonard Surges, Noranda Mining and Exploration Inc. Luc St. Arnaud, Noranda Technology Centre
Economic metals	Copper, lead, zinc
Open pit or underground	Both
Average annual precipitation, and/or range	1134 mm (762 mm as rainfall)
Mean daily temperatures	July = 17.9 °C; January = -12.5°C
Other significant climatic information	Maritime continental, with hot, relatively dry summers, and cold snowy winters

C.3 Geology/Mineralogy

The Stratmat deposit is hosted by a volcanic/sedimentary sequence that is part of the Ordovician Tetagouche Group in the Bathhurst area of New Brunswick. Rocks in the stratigraphic footwall of the deposit are dominated by siliceous sericitic pelites, and quartz augen schists, with white to pale grey, fine-grained siliceous or cherty layers and mylonitic and phyllonitic bands. The rocks contain 2 to 4% disseminated pyrite and trace amounts of sphalerite, galena, and chalcopyrite. The hanging wall rocks consist of a series of deformed feldspar rich meta-volcanics and volcanoclastics with small amounts of meta-sediments. Pyrite is the dominant sulphide, generally occurring in trace amounts. The ore zone is characterized by talc rich meta-sediments including talc rich pelites, talc-cherts and talc-carbonate zones. Sulphides in the ore zone are massive to disseminated with sulphide contents of 2 to 40% (Payant and Yanful, 1997).

The waste rock consists of a variably sheared meta-rhyolite consisting of euhedral disseminated pyrite, sericite (fine-grained mica), quartz, and minor amounts of illite, feldspar and apatite (Yanful *et al.*, 1997), with a sulphur content of about 9.5%, occurring mostly as pyrite. There is insufficient data presented in the available reports to describe the variability in acid generating potential within the waste rock.

C.4 Static Tests

C.4.1 Methods

Two samples from the column and lysimeter testing program were submitted for acid base accounting using the B.C. Research Initial test method (Table C.1). Sulphide sulphur was calculated by subtracting the sulphate-sulphur from the total sulphur (Payant *et al.*, 1995).

C.4.2 Results

The ABA results from the column and lysimeter samples are presented in Table C.1. The results indicated these rocks were strongly acid generating, with an average NNP of - 283 kg $CaCO_3$ /tonne and an NP/AP ratio of 0.058. In this review, it was assumed that waste rock in the full-scale dump had a similar sulphide content and neutralization potential.

The ABA values for the 1 and 3% limestone blends were calculated by assuming the results in Table C.1 were typical of the overall waste rock composition and that the limestone used in the blending experiments was 100% pure and fully reactive (i.e., had an NNP of +1000 kg $CaCO_3$ /tonne). The weighted ABA results are presented in Table C.2. The results show that both of the blends tested were strongly net acid generating, with NP/AP ratios of 0.16 and 0.09 for the 3% and 1% blends respectively.

Table C.1: Acid Base Accounting Results, Stratmat Waste Rock

(Payant *et al.*, 1995)

Sample	paste pH	% S (T)	% S (SO_4^{2+})	% S (S^-)	AP	NP	NNP	NP/AP
1	8.45	9.7	0.11	9.59	300	17.6	-282	0.058
2	8.72	9.8	0.12	9.68	302	17.6	-284	0.058

Table C.2: Calculated Acid Base Accounting Results for 1% and 3% Blends

Material	%	AP*	total AP**	NP*	total NP**	NNP*	NP/AP ratio
1% Limestone Blend:							
Waste Rock	99	301	298	17.6	17.4		
Limestone	1	0	0	1000	10		
Blend	**100**		**301**		**27.4**	**-274**	**0.09**
3% Limestone Blend:							
Waste Rock	97	301	292	17.6	17.1		
Limestone	3	0	0	1000	30		
Blend	**100**		**301**		**47.1**	**-254**	**0.16**

Notes: * kg $CaCO_3$ eq./tonne
** tonnes $CaCO_3$ eq.

C.5 Kinetic Tests

C.5.1 Methods

Column Tests

Column tests were used to evaluate the relative efficiency of treating the waste rock with different blends of limestone in a controlled laboratory setting. The laboratory columns consisted of a 1 metre tall, 15 cm diameter PVC pipe, fitted with a collection system at the base, and an irrigation system at the top. Waste rock samples were blended with the appropriate amount of crushed agricultural grade limestone, and were then divided into three, 20 kg portions. The 20 kg samples were then placed in each of the 9 columns (controls, 1% and 3% blends each tested in triplicate).

The tests were carried out for a period of 153 weeks. Leach cycles consisted of 8 weeks of wetting, followed by 8 dry weeks. During the wet period, a total of 650 mL of distilled water was added to each column. This amount was equivalent to an average precipitation rate of 946 mm/yr which was close to the actual average precipitation at the site. Each week, the pH sulphate and acidity values were measured, and a sample was prepared for ICP analysis. Samples from the 2nd and 7th weeks of each cycle were submitted for ICP analysis (Payant *et al.*, 1995).

<u>Field Lysimeter Tests</u>

Field lysimeter tests were used to evaluate the relative efficiency of treating the waste rock with different blends of limestone under more natural climatic conditions. The field tests were constructed using a 160 litre polyethylene tank which was 94 cm tall and 46 cm in diameter. The lysimeters were fitted with a water collection system at the base and were left open at the top. Waste rock samples were blended with the appropriate amount of crushed agricultural grade limestone, and divided into three, 170 kg portions. The samples were then placed in each of the 9 lysimeters (control, 1% and 3% blends tested in triplicate) (Payant and Yanful, 1997).

The tests were run for a total of 125 weeks. Samples were collected at regular intervals whenever there was at least 1 litre of leachate available. The samples were submitted for the set of analyses as described for the laboratory tests.

C.5.2 Results

The kinetic testing results are summarized in Table C.3. Results from the laboratory and field control tests confirmed that the Stratmat waste rock was strongly acid generating, with acidic leachate detected within 5 weeks in the laboratory columns and 40 weeks in the field lysimeters. Sulphide production rates in this unblended material were high, with late cycle sulphate rates of approximately 180 mg/kg/week for the laboratory columns. Sulphate production from the field tests could not be determined because flow data was not presented in the available reports. However, acidity production rates were significantly lower in the field, with average weekly acidity production rates of 41 mg/kg/week, as compared to 130 mg/kg/week in the laboratory. This may have reflected less effective flushing of oxidation products in the field lysimeter test.

The blended test results indicated that even small amounts of limestone (1 and 3%) can delay the release of acidic effluent as shown in Figure C.1. The results also suggest that small amounts of limestone may reduce the rate of sulphide oxidation. Differences in the sulphate production rates of the control and blended laboratory column samples are presented in Figure C.2. These differences are also evident in the acidity data for both the field and laboratory tests. However, sulphate production rates were based on measured sulphate concentrations, which likely were limited by the solubility of gypsum or anhydrite. Effluent from the control sample was reported to be close to saturation with respect to gypsum at all times (saturation index averaged -0.5) (Yanful *et al.*, 1997). Calcium provided by the added limestone may have further enhanced the formation of gypsum or anhydrite within the columns and therefore may have lowered the sulphate concentrations in the column effluent. Resolution of this issue by geochemical modelling or examination of post test mineralogy was not reported in the available references.

Most of the 1% and several of the 3% replicate tests were in the advanced stage of acidic effluent production by the end of the testing period with high sulphate concentrations, high metal levels and low pH. The field test results generally had higher sulphate and metal concentrations in the effluent, but lower production rates than the laboratory tests when the results were normalized to the sample mass. The lower production rates in the field were attributed to the colder temperatures (Payant and Yanful, 1997). However, this may also have reflected less efficient flushing of oxidation products within the mass of the field test due to increased channelization.

Although these results showed clear benefits in terms of delaying the release of acidic effluent and possibly reducing the rate of oxidation in the short term, none of the limestone blends tested were suitable for long term control of acid generation. The main reason was that the tested blends did not have a sufficiently high NP/AP ratio.

Table C.3: Summary of Kinetic Testing Results

Parameter	Laboratory Columns			Field Lysimeters		
	Control	1%	3%	Control	1%	3%
pH: early cycles late cycles	7.0 2.0-2.2	7.0-8.0 ~2.5	7.0-8.0 ~2.5	3.5-5.0 2.0-2.5	6.5-7.5 ~2.5	7.0-8.0 2.8
Weeks to pH drop	5-20	40-150	120-150	40-60	50-100	100->125
Acidity (mg/kg/week)* average weekly late cycle	130 125-150	22.5 50-100	2.96 20	41.0 50-125	6.53 35	2.32 30-100
Cumulative acidity. (g $CaCO_3$ eq)* released stored	380 21	65.6 3.82	8.57 0.56	872 -	139 -	49.3 -
SO_4 production (mg/kg/week)** early cycles late cycles	11 180	5 70	4 40	-*** -***	-*** -***	-*** -***
SO_4 production (mg/kg) total**	25,900	5,900	2,734	-***	-***	-***

Notes: * average of three columns, data compiled from text, tables and figures in Payant and Yanful, (1997)

** average of three columns, calculated from appendices in Payant and Yanful, (1997)

*** could not be calculated because flow data was not available

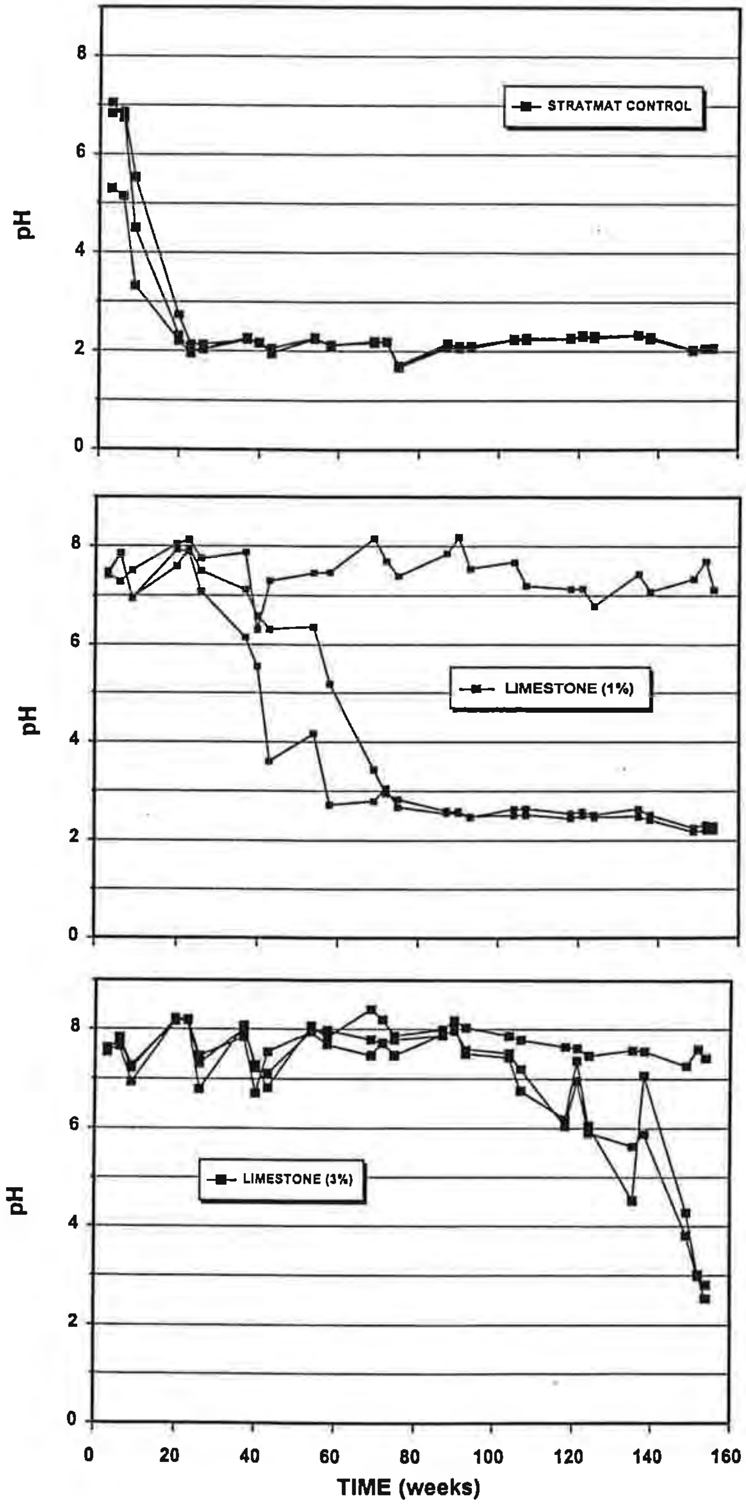

Figure C.1 Graphs Showing pH versus Time in the Laboratory Column Tests with Controls, 1% and 3% Limestone Blends
(Payant and Yanful, 1997).

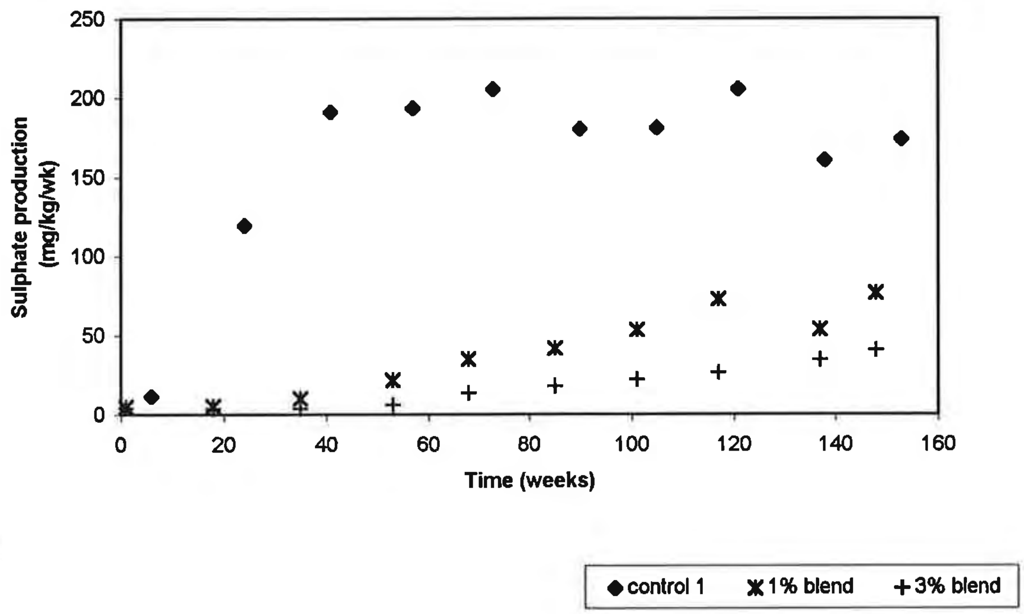

Figure C.2 Comparison of Sulphate Production Rates from the Control Tests, and the 1% and 3% Limestone Blends. Results are an Average of the Weekly Sulphate Production in a given Wetting Cycle, Averaged from the Three Replicated Columns
(Calculated from Data in Payant and Yanful, 1997)

C.6 Field Monitoring Program

C.6.1 Background

Prior to the detailed kinetic testing program, a full scale waste rock pile at the Stratmat site was constructed incorporating limestone layers with the objective of delaying the release of acidic effluent until a more permanent disposal option could be developed. It was anticipated that the waste rock dump design would delay the release of acidic effluent for approximately 3 years on the basis of an earlier laboratory testing program (Sheremata et al, 1991).

C.6.2 Pile Construction

The waste pile design and construction details are summarized in Table C.4. The dump was built in 3 metre layers. A 15 cm layer of crushed limestone was placed over each of these layers.

Table C.4: Waste Pile Design and Construction

Plan area of dump	60 ha
Foundation material	Non-acid generating volcanic rocks sloping at a grade of about 10%.
Dimensions	200 x 300 metres, with 3, 1.8 metre lifts
Tonnage	1.5 million tonnes
Proportions of rock types	1% limestone, 99% waste rock
Spatial deposition of rock types	15 cm layer of limestone over each 3 metre layer of waste rock.
Method of placement	End dumped then bulldozed, no intentional compaction

C.6.3 Seepage Monitoring Data

Water quality samples were collected from a pond at the toe of the dump which received seepage from the dump and from a small gully that ran down the adjacent hillside. Samples were tested for Pb, Zn, Fe, Cu, total suspended solids, hardness and pH at the Belledune Smelter in New Brunswick.

The pH of the seepage water was acidic after only 1.5 years of monitoring. The metals data were not reported.

C.6.4 Additional Observations

Solids samples collected from the limestone layers in the dump were heavily stained by iron precipitates.

C.6.5 Cost Data

The limestone used in the pile construction cost approximately $100/tonne (A. Moerman, 1997, *pers comm.*)

C.7 Conclusions

- Limestone addition in laboratory tests delayed the release of acidic effluent. The delay was proportional to the amount of limestone added.

- Sulphide oxidation rates as measured by sulphate concentrations appeared to be slowed by the presence of limestone, both before and after the onset of acid generation. This apparent reduction in sulphide oxidation rates is in agreement with data from the Cinola project (Appendix D), but not Samotosum (Appendix A) or Kutcho Creek (Appendix B). However, interpretations are limited by the lack of data on gypsum saturation and the potential for sulphate precipitation within the test vessels.

- pH levels in most of the replicate column tests and field lysimeters decreased to less than 3 during the three years of testing, indicating this was not a viable long term prevention measure for acid drainage under the tested conditions. It was not clear whether the limestone had been consumed, washed out, or made unavailable from the data presented.

- 15 cm layers of limestone placed between 3 metre lifts of potentially acid generating waste rock did not prevent the release of acidic effluent.

C.8 Possible Research Topics

- Geochemical modelling or examination of post-test mineralogy to determine whether sulphate concentrations were limited by gypsum saturation when limestone was blended with sulphidic rocks.

- Close the mass balance on calcium and study the secondary mineral coatings in the weathered samples to determine whether the limestone had been consumed, washed out, or made unavailable by secondary coatings.

- Tabulate metal data in terms of production rates to allow comparison of the metal release rates between different blends.

C.9 References

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APPENDIX D
CINOLA MINE

D.1 Overview

The Cinola Gold deposit is located on Graham Island, in the northern part of the Queen Charlotte Islands, British Columbia. The deposit is an epithermal Carlin-type gold deposit. Seasonal weather patterns are typical of coastal Western Canada, with 1700 to 2200 mm of precipitation occurring as rain during October to March. Detailed feasibility studies were initiated in 1986 by City Resources (Canada) Inc. The ARD potential of the waste rock was examined as part of these studies, and then continued under MEND and the BC AMD Task Force in 1990 (MEND Project 1.19.1) after City Resources decided not to pursue the project.

The environmental studies included a series of column tests which examined blending and layering of limestone within the acid generating waste rock. The five column tests included a control test, blends of 6.6%, 3.3% and 0.84% limestone, and a column with alternating layers of the 6.6% blend (1 cm thick) and the 0.84% blend (10 cm thick) giving an overall limestone content of 1.2%. The only test with an NP/AP ratio of greater than 1 was the 6.6% limestone blend.

Results from the column testing program indicated that limestone blends with NP/AP ratios less than 1 were not sufficient to prevent acid drainage in the long term, but that blending at NP/AP ratios of about 1 might delay the release of acidic effluent for several years. Depletion calculations indicated that an NP/AP ratio_{crit} greater than 2 would be required to prevent acidic effluent discharging from a fully blended mixture of this rock. The layered column and the 0.84% limestone blend gave very similar results, indicating that the extra limestone provided by the 6.6% layers did not significantly delay the release of acidic effluent. This finding showed the importance of intimate blending in controlling the release of acidic effluent from these wastes. The post-test mineralogical analyses completed on all columns showed that the limestone availability was not reduced by ferric hydroxide coatings but was fully available for reaction.

In the 0.84% and the 3.3% blended columns, the rate of sulphate release was reduced prior to depletion of the limestone and development of acidic effluent. At the concentrations measured, the reductions in sulphate release appeared to be a function of the neutral pH conditions limiting sulphide oxidation rates, rather than the solubility of gypsum ($CaSO_4$) limiting the rate at which sulphate was flushed from the column, as confirmed using formal chemical equilibria calculations (*pers comm.* Stephen Day, 1997).

D.2 General Information

Proposed Cinola Mine	
Location	Northern Queen Charlotte Islands
Mine Owner	formerly City Resources (Canada) Ltd.
Mine Contact	none available, Norecol Dames and Moore were the principal investigators for the MEND/BC AMD Task Force program
Economic Minerals	Gold
State of Operations	Feasibility study 1986-1990
Annual Precipitation	1700 - 2200 mm
Average temperature range	January (<3°C), July (>14°C)

D.3 Geology

The Cinola deposit is an epithermal Carlin-type gold deposit. It is hosted by a suite of Miocene aged, fluvial sediments, referred to as the Skonun Formation. This is underlain by older (144 to 208 million year old) sedimentary rocks known as the Haida Formation. The epithermal fluids were associated with the intrusion of a small rhyolite stock during the middle Miocene. Fluids followed pre-existing fracture systems causing local brecciation and more widespread alteration of the host rocks. The alteration varies in intensity from siliceous in the vicinity of the rhyolite heat source to argillic (a type of clay alteration) at some distance. A cross-section of the deposit is shown in Figure D.1.

As described by Norcol, Dames & Moore, Inc. (1994), the Skonun Formation consists of consolidated layers and lenses of mudstone, siltstone, sandstone and conglomerate. The unaltered sediments reportedly do not contain significant amounts of pyrite or carbonate. Argillically altered sediments contain the clay minerals kaolinite and illite, and are highly susceptible to weathering. The silicic altered sediments have been flooded with quartz and are considered highly competent. The altered sediments contain about 2% pyrite, marcasite (about one tenth the amount of pyrite), and minor amounts of chalcopyrite, sphalerite, galena, pyrrhotite, cinnabar. Pyrite occurs as disseminated grains generally less than 25 μm in diameter, with a "spherulitic" and "framboidal" appearance (referring to the textural appearance rather than possible genesis). None of the Skonun sediments contain significant amounts of carbonate. Unaltered Skonun sediments would have been the most significant type of rock in the waste rock piles (56%), and the altered materials would have comprised about 6% of the waste.

The mudstones of the Haida Formation are black to dark grey in colour, with minor amounts of siltstone and sandstone. These rocks contain less than 1% pyrite as disseminated spherical grains. Calcite is more abundant than the pyrite, occurring as fine disseminated grains dispersed through the matrix, or as coarse crystals. These rocks would have comprised approximately 22% of the waste rock on site. The rhyolite is a fine grained felsic intrusive with little or no neutralization potential, and pyrite concentrations in the range of 1 to 10%. This rock would have comprised up to 6% of waste rock. Breccias vary in composition according to the extent of alteration and the mixture of lithologies. These would have been only a minor component of the waste rock.

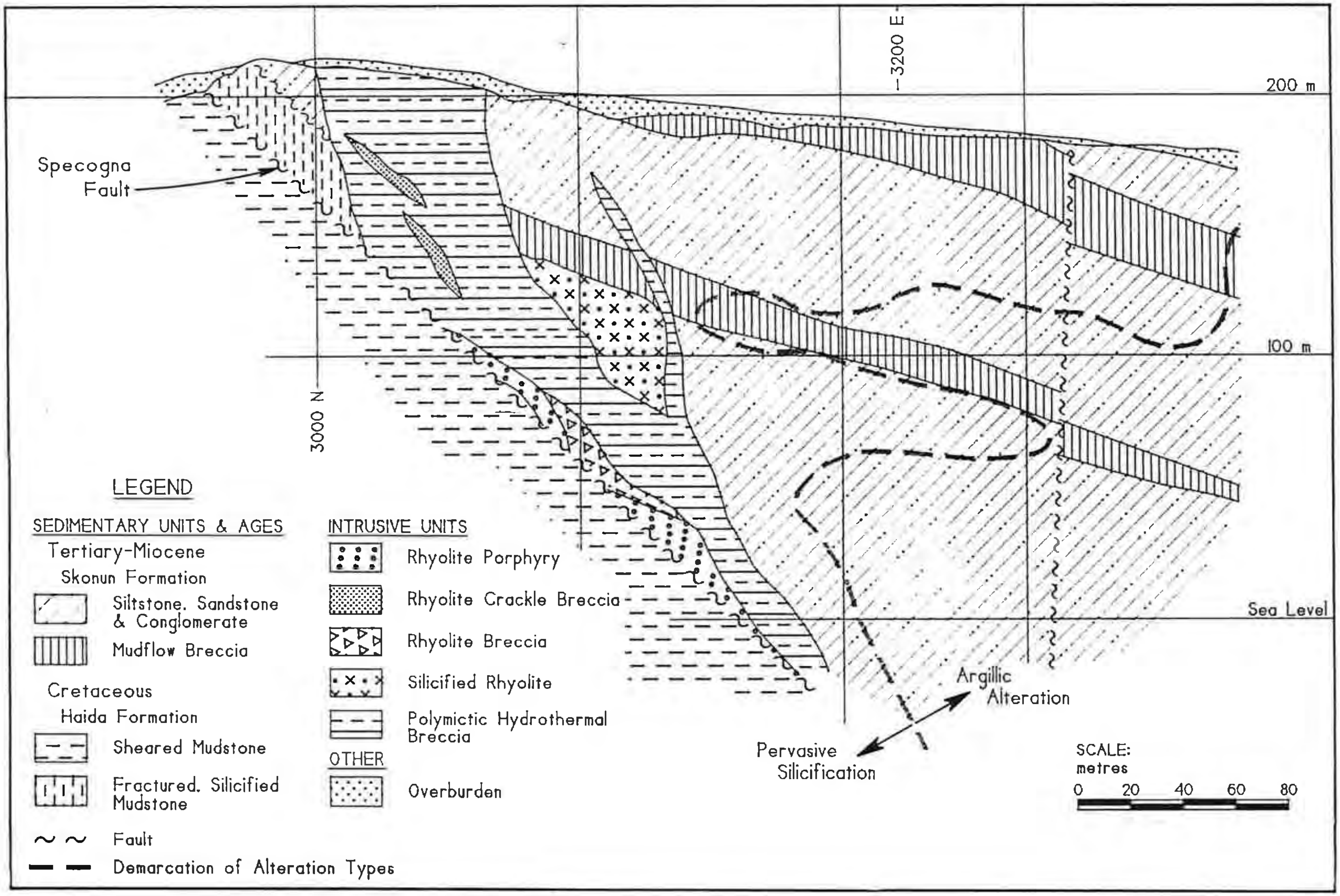

Figure D.1 Geological Cross-Section Showing Major Lithologies in the Proposed Cinola Mine (Norecol, Dames & Moore, 1994).

D.4 Static Tests

The acid base accounting tests were done using the Sobek method (Sobek *et al.*, 1978). A subset of samples was submitted for detailed sulphur speciation to quantify the portion of non-reactive organic sulphur. Data was available from Norecol, Dames & Moore (1994) and from the DBARD database (1995).

The ABA results are presented in Tables D.1 and D.2, showing the mean and distribution for the overall dataset, and the mean and range within each of the major rock types. The results indicated that the majority of waste rocks at this site would be net acid generating. The Haida Mudstone was the only unit containing significant neutralization potential, and it was only marginally net acid consuming.

A limited series of tests were conducted to determine the sulphur speciation in these rocks (Norecol, Dames & Moore, 1994). The results indicated there was a significant portion (15%) of potentially unreactive sulphur in the rocks, occurring mostly as organic sulphur. Thus ABA estimates may have slightly overestimated the potential for acid production in the rock. The sulphur speciation results are summarized in Table D.3.

The proposed source of limestone for the project was from Texada Island. This material had a neutralization potential of 932 kg $CaCO_3$ /tonne. Estimates of carbonate content calculated from inorganic carbon analyses suggested the reactive (carbonate) NP was only about 75% of this value.

Table D.1: Waste Rock from Pit and Pit Walls (DBARD, 1995)

Parameter	No.	mean	median	Percentile			
				10%	25%	75%	90%
Total sulphur (%)	197	2.29	2.00	0.75	1.20	2.90	4.10
Paste pH	197	5.85	5.80	3.76	4.70	7.20	7.90
NP	197	15.82	1.90	0.46	1.10	4.30	53.96
AP (total sulphur)	101	71	63	23	39	89	129
NNP	101	-55	-58	-127	-86	-28	16
mean NP/meanAP		0.22					

Note: Values are numeric means of all data and do not represent mass weighted composition

Table D.2: ABA results by Rock Type (Norecol, Dames & Moore, 1994)

Rock type or composition	Skonun Sediments	Argillically Altered Skonun Sediments	Haida Mudstone	Rhyolite	Breccia
Percent of Waste**	56	6	22	6	4
Number of Samples	101	26	33	12	10
Total sulphur (%)	2.3 (0.5,5.4)*	4.9 (1.9,7.2)	1.3 (0.1,2.8)	1.2 (0.0,4.0)	1.3 (0.6,2.4)
Paste pH	5.4 (3.3,8.1)	4.2 (3.0,6.7)	7.5 (4.5,8.5)	6.3 (4.3,8.3)	5.4 (2.5,7.0)
NP	2.4 (-4.0,23)	2.1(0.0,4.3)	60 (0.0,310)	12.6 (0.8,130)	0.46 (0,2.8)
AP (total sulphur)	71 (16,169)	154 (59,225)	41 (3,88)	39 (0,125)	42 (19,75)
NNP	-68 (-14,-120)	-154 (-220,-58)	19 (-69,235)	-23 (0.8,130)	-41 (-74,-19)
mean NP/meanAP	0.03	0.01	1.46	0.32	0.01

Notes: * mean (min,max)
** overburden is 6% of the total waste

Table D.3: Sulphur Forms in the Waste Rock

Parameter	Mean	Median	Percentile			
			10%	25%	75%	90%
Total sulphur (%)	1.61	1.3	0.96	1.10	1.95	2.70
Sulphide sulphur (%)	1.37	1.20	0.64	0.80	1.70	2.42
Sulphate S (%)	0.07	0.04	0.02	0.03	0.08	0.13
Organic S (%)	0.15	0.13	0.04	0.09	0.22	0.27
Inert sulphur (% of total S)	15.6	14.8	5.23	9.06	19.46	25.23

D.5 Kinetic Tests

Column tests were used to evaluate the addition of limestone to the waste rock as a means of preventing or delaying the release of acidic drainage.

D.5.1 Methodology

The 5 column tests included 1 control, 3 blends with limestone, and 1 layered with two different limestone blends. The waste rock was obtained from reverse circulation drill cuttings, and was a composite sample representing the expected composition of Skonun Sediments in the waste dumps. Limestone was obtained from the Texada Island quarry, and was crushed to a diameter of less than 0.6 mm. The composition of materials in each of the columns is summarized in Table D.4. Calculated ABA values for each column are shown in Table D.5. Standard acid base accounting methods were used to characterize the Skonun Sediment composite and the Texada Limestone. The ABA data for the blended and layered materials was calculated from these values.

The columns were constructed out of plastic pipe, with diameter of 15 cm and height of 1 m. The height of rock in the columns was about 0.5 m. Humidified air was added through a clear plastic plate covering the top of the columns. Water was also added through this plate at a rate of 0.4 to 0.5 mL/min, equivalent to an annual precipitation rate of 14,000 mm, or about 6 to 8 times the precipitation rate at the site. The drip plate was rotated 51 ° each day to allow the water to follow different flowpaths within the column to attempt to minimize retention of sulphide oxidation products. Each week, samples were tested for pH, Eh, temperature, dissolved oxygen and conductivity. Filtered samples were analyzed for alkalinity, acidity, sulphate and a suite of 30 metals. The frequency of analyses was decreased during the later part of the testing program, however the irrigation rates remained constant throughout the program (Norecol, Dames & Moore, 1994).

Table D.4: Column Descriptions

Column 1	Control	Waste Rock (Skonun Sediment composite)
Column 2	6.6% limestone	Limestone was mixed with waste rock
Column 3	3.3% limestone	Limestone was mixed with waste rock, and capped with 1 cm of Column 2 material
Column 4	0.8% limestone	Limestone was mixed with waste rock, and capped with 1 cm of Column 2 material
Column 5	Layered (overall limestone concentration of about 1.2%)	Limestone was applied in 5 - 1 cm thick layers of Column 2 material, interbedded with 4 - 10 cm layers of Column 4 material.

D.5.2 Kinetic Testing Results

The kinetic testing results are summarized in Table D.5. The control column demonstrated that the Skonun waste rock composite was strongly acid generating, with acidic pH (<4.5) developing within 2 weeks, and a minimum pH of about 2.0 developing within 11 weeks of the start-up date. The pH recovered to about 3 over the remaining test cycles.

Table D.5: Kinetic Test Results

Column ID, Material Tested	1 control	2 6.6%	3 3.3%	4 0.84%	5 layers
%S	2.10	1.98	2.05	2.08	2.08
AP	66	62	64	65	65
NP	8	69	38	16	19
NNP (by mass)	-58	7	-26	-49	-46
NP/AP ratio (by mass)	0.12	1.1	0.59	0.25	0.29
Column Leachate					
pH: - initial - minimum - late stage	5 2 3	7 to 8 7 7.8	7 to 8 2 2 to 3	7 to 8 2 2 to 3	7 to 8 2 2 to 3
Time to pH transition (weeks)	2-11	>250	200-220	33	33
SO_4 concentration (mg/L) - initial (after early flushing) - peak	 300 6500	 50-90 na	 50-100 1800	 100-120 5000	 90-100 2100
Time to initial SO_4 increase conc.(wks)	0	205	105	20	20
Time to peak SO_4 conc. (wks)	20	>250	215	42	48
% S depletion (SO_4^{2-} production rates)	62%	11%	50%	61%	49%
% S depletion (residue analysis)	77-80%	15-46%	63-67%	68-75%	72%
% NP depletion (Ca^{2+} production rates)	-*	-*	75%	-*	-*
% NP depletion (residue analysis)	100%	97 to 17%	100%	100%	100%
Maximum arsenic concentration (mg/L)	35	0.05	0.11	7.50	0.40
Maximum copper concentration (mg/L)	2.1	0.01	0.45	0.32	1.0
Maximum iron concentration (mg/L)	2000	-**	100	-***	-***
Maximum zinc concentration (mg/L)	5.5	0.02	2.0	0.8	2.0

Note: * calcium was not measured from weeks 20 to 110, thus mass balances could not be accurately calculated.

** iron not measured from week 20 to 110, when maximum sulphates were measured

*** iron was not measured beyond week 20 (likely below detection limits)

The limestone addition experiments showed that the release of acidic effluent could be significantly delayed by the addition of limestone. However, calculations indicated that the overall NP/AP ratio of the mixture would have to exceed an NP/AP ratio of 2 to prevent acidic effluent in the long term. Three of the four limestone columns produced acidic effluent during the testing period of 250 weeks. The fourth column (Column 2) showed increasing SO_4 concentrations during the last 50 weeks of testing and portions of the column residue had low paste pH concentrations. Trends in pH and sulphate concentrations are shown in Figure D.2. Both the ABA tests from residual test rock, and the mass loading calculations indicated that the acid generating columns had lost between 40 to 80% of the sulphur during the test. It is considered likely that the rebound in pH following peak sulphate concentrations was due in part to the depletion of sulphide minerals.

In the blended column tests, the limestone appeared to have been fully available for reaction, and was not "blinded" or coated with iron hydroxide precipitates to any extent. Acidic column residue had no measureable NP, and very little or no limestone as indicated by fizz tests on the residue material. Non-acidic column residue had substantial amounts of available NP, as indicated by ABA tests, thin-section analysis and fizz tests (Norecol, Dames & Moore, 1994).

Metal concentrations in the columns varied according to the rate of sulphate release in the columns and pH. The more soluble metals, zinc and arsenic, were strongly correlated with the sulphate concentrations, with maximum concentrations observed during the peak in sulphate production, prior to the pH transition. The highest concentrations were observed in the columns with the lowest limestone concentrations. Iron and copper tended to follow this same pattern, but were limited by solubility constraints until the pH of the solution decreased.

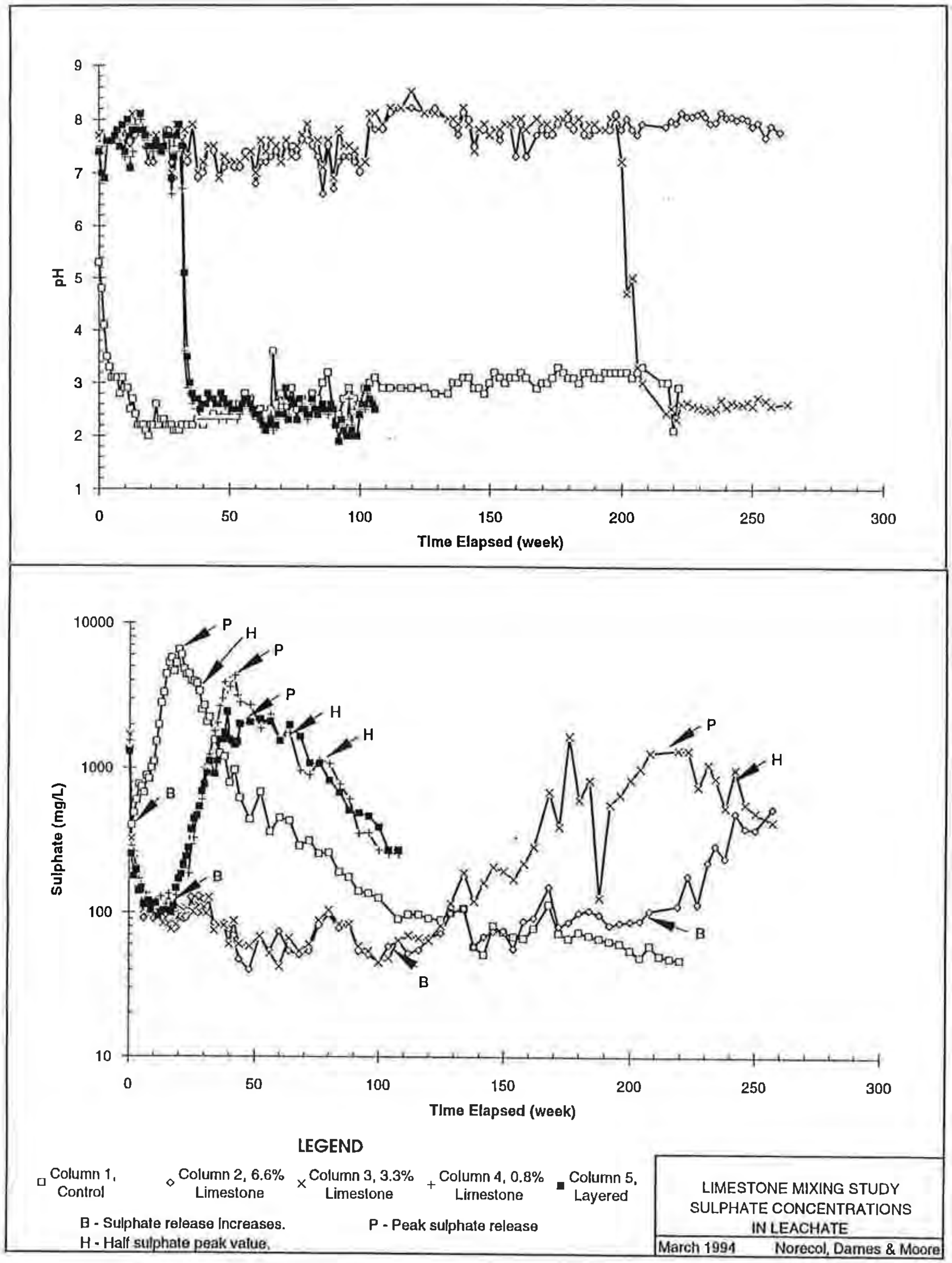

Figure D.2 pH and Sulphate Concentrations versus Time
(Norecol, Dames & Moore, 1994).

Although the layered column (Column 5) had a slightly higher NP than Column 4, the results did not differ significantly considering that both columns had the same base blend. The time to pH transition was unchanged, and the time to peak sulphate concentration were similar. This finding suggests that the additional limestone contained in the 1 cm thick layers of 6.6% limestone blend was not equally available for reaction as the more intimately blended limestone contained in the 10 cm thick layers of 0.84% limestone blend. This may be due to the physical separation of the limestone in the thin layers from the majority of sulphide particles contained in the thicker layers.

The blended limestone appeared to delay the release of acidic effluent in two ways: 1) the limestone limited the rate of sulphide oxidation by maintaining neutral pH conditions throughout the column; and, 2) any acidity produced by the sulphide oxidation reactions was neutralized before it left the column. Thus, acidic effluent was not observed until the neutralization potential was at, or close to, depletion.

The NP/AP $ratio_{crit}$ required to maintain net neutral conditions in fully blended waste was estimated to be greater than 2 for the Cinola Project. This was based on the ratio of SO_4/Ca observed during the stable (neutral pH) portion of the tests. For each SO_4^{2-} ion in solution, approximately 2 Ca^{2+} ions were released. Thus a ratio of 2 moles $CaCO_3$ to each mole of S (or 4 moles $CaCo_3$ to each mole FeS_2) would be needed to maintain neutral conditions under fully blended test conditions in the long term.

D.6 Conclusions

- Blending finely crushed limestone with waste rock appeared to reduce the rate of sulphide oxidation as indicated by sulphate production rates.

- Intimate mixing limestone with waste rock was successful in delaying but not preventing the onset of acid drainage at the laboratory scale. The delay appeared to be a combination of 1) controls on the rate of sulphide oxidation (by maintaining localized neutral pH conditions), and 2) straightforward neutralization of acidity generated by the reduced oxidation rate.

- Based on molar ratios of SO_4 to Ca measured when leachates were pH neutral or stable, NP/AP ratios of at least 2 would be required to prevent the release of acidic effluent from blended waste rock and finely crushed limestone in the long term.

- Metal concentrations in the column effluent may have been limited by the rate of sulphide oxidation, as well as by pH control.

- In the blended columns, the crushed limestone was fully available for reaction, and was not blinded or armoured by iron-precipitates.

- Additional limestone contained in 1 cm thick layers in the layered column did not appreciably delay or mitigate acidic effluent as compared to the column with the same base blend. This may be due to the limited physical contact between the additional NP in the thin 1cm layers and sulphides in alternating 10 cm thick layers. This suggests that alternating layers as little as 10 cm in thickness would not signficantly reduce sulphide oxidation rates.

D.7 Possible Research Areas

The laboratory studies represent ideally blended conditions in a relatively small scale test. The effects of physical and geochemical heterogeneities could be examined at a more realistic scale using similar materials.

D.8 References

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APPENDIX E
ESKAY CREEK MINE

E.1 Overview

The Eskay Creek Mine is located approximately 83 km north of Stewart, B.C. This gold and silver ore deposit is part of a stratiform bound volcanogenic massive sulphide deposit located within the early to middle Jurassic Hazelton Group. Mining began at this site in 1994, with 1997 probable reserves of 1.27 million tonnes at 59.3 grams per tonne Au and 2719 grams per tonne Ag (Meyer, 1997, *pers. comm.*). Underground cut and fill methods are used to extract the ore. It is anticipated that approximately 380,000 tonnes of waste rock will be produced during the first 10 years of mining (Stewart *et al.*, 1994).

In the early planning stages of this project, it was thought that acid consuming andesite volcanics would comprise a significant portion of this waste rock. A kinetic testing program was initiated to test whether andesite could be used as either an acid consuming base layer or an alkaline cap to control acid rock drainage (T.W. Higgs and Associates, 1993).

The kinetic testing program is illustrated in Figure E.1. The tests consisted of two sets of columns in series. Test #1 consisted of an acid generating column of material followed by an acid consuming column (the acid consuming base design), while Test #2 consisted of an acid consuming column of material followed by an acid generating column (the alkaline cap design). Both of the tests had overall NP/AP ratios of 1.2. The results of the testing program indicated that the acid consuming base layer (Test #1) would likely provide effective long term mitigation of acidity and metals produced in the overlying acid generating materials, while the acid consuming cap layer would do little to control the rate of sulphide oxidation or metal leaching from underlying acid producing materials.

Subsequent revisions to the mining plan have substantially reduced the estimated volume of acid consuming andesite such that layering is no longer considered a viable option for ARD control. Instead, the mine is using subaqueous disposal of the potentially acid generating waste rock (Stewart *et al.*, 1994).

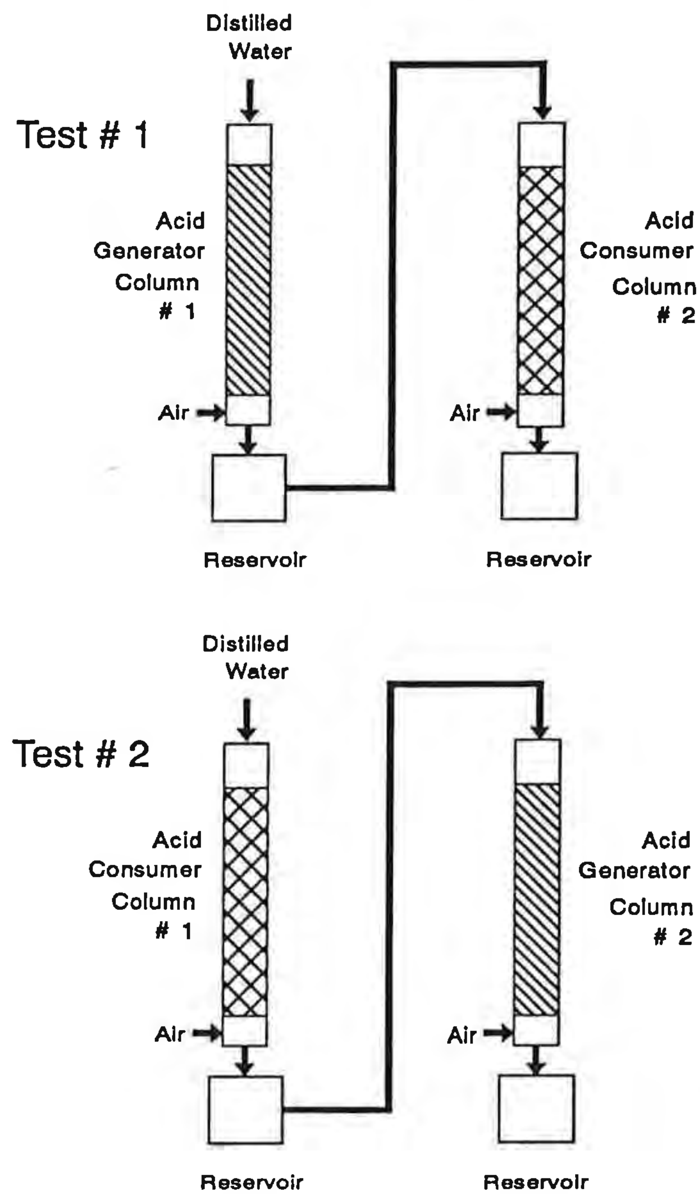

Figure E.1 Column Testing Program (T.W. Higgs and Associates, 1993)

E.2 General Information

Location	83 km north of Stewart B.C., with access from the Stewart/Cassiar highway at Bob Quinn
Mine owner	Homestake Canada Inc. and Prime Resources Group Inc.
Mine contact	Ms. Sharon Meyer, Mr. Martin Murphy, Homestake Canada Inc.
Economic metals	Au, Ag
Open pit or underground	Underground cut and fill
Reserves	1.08 million tonnes grading 65.5 g/tonne Au and 2930 g/tonne Ag
Estimated waste rock production	380,000 tonnes

E.3 Geology and ARD Potential

The Eskay Creek deposit is a stratiform bound volcanogenic massive sulphide deposit hosted within the Salmon River Formation of the early to middle Jurassic Hazelton Group. The ore mineralization is hosted within a suite of mudstone and brecciated mudstone and is bounded by rhyolitic volcanics in the footwall of the deposit and andesitic volcanics in the hanging wall (Stewart *et al*, 1994).

The andesitic volcanics vary from massive pillow lava flows to thinner hyalophitic textured sills that alternate with argillaceous sedimentary layers (Stewart *et al.*, 1994). Static ABA tests have indicated this unit is net acid consuming with a mean NNP of + 98 kg $CaCO_3$/tonne. However approximately 44% of the samples had negative NNP's indicating that a substantial portion of this material is potentially acid generating (T.W. Higgs and Associates, 1993).

The rhyolite varies from massive to flow banded or brecciated. The brecciated rhyolite is a fragmented, chlorite and/or sericite altered zone located immediately beneath the ore zone which may have been a fluid recharge or mineralizing discharge vent zone for the ore deposit. The massive rhyolites are silicified to potassic feldspar altered, and are found below or adjacent to the brecciated materials. Static ABA tests indicate that the rhyolite is net acid generating, with NP/AP ratios of less than 1 in 75% of the samples. The samples representing underground mine waste had an average NNP of - 57 kg $CaCO_3$/tonne, with NP/AP ratios of less than 1 in 95% of the samples. Brecciated rhyolites tend to be more susceptible to both physical and chemical weathering than the massive rhyolites (Stewart *et al.*, 1994).

The contact zone mudstones and breccias which host the ore are net acid generating, with NNP's of - 114 kg $CaCO_3$/tonne (T.W. Higgs and Associates, 1993). The breccias are described as having a "chaotic muddy matrix" with mudstone, altered felsic volcanic fragments and less common sulphide fragments. Ore mineralization is commonly coarse-grained sphalerite, pyrite, galena and visible gold, in milky quartz veins or stockwork, or disseminated in silicified and pyrite altered felsic volcanics (Stewart *et al.*, 1994).

The underground mining plan identified the brecciated rhyolite as the main type of waste rock to be produced during mining (Stewart *et al.*, 1994).

E.4 Kinetic Testing

E.4.1 Methods

The kinetic testing program is shown in Figure E.1. Test #1 consisted of a column of acid generating rock followed by a column of acid consuming rock. This test was intended to simulate the use of an acid consuming base layer for controlling ARD in the waste rock dumps. Test #2 consisted of a column of acid consuming rock followed by a column of acid generating rock. This test was intended to simulate the placement of an alkaline cap over acid generating rock to reduce sulphide oxidation and metal leaching rates in the underlying materials.

The samples were prepared from drill hole composites to obtain an acid generating composite and an acid consuming composite. The acid generating composite was made using a mixture of hanging wall argillite, contact zone and footwall rhyolite samples with an overall NNP of - 101 kg $CaCO_3$ eq/tonne and an NP/AP ratio of 0.23. The acid consuming composite was made from a mixture of hanging wall andesite samples with an NNP of + 124 kg $CaCO_3$ eq/tonne and an NP/AP ratio of 12.4. The samples were thoroughly blended and crushed to minus 3/8 inch. The two composites were each split into two 30 kg portions and were placed in the columns. The overall NP/AP ratio for both tests was 1.2.

Each column was constructed from a 1 meter length of 15 cm diameter PVC pipe. Each was fitted with an acrylic base plate and water collection system. Water was added to the top of each column using a peristaltic pump and drip irrigation system. The columns were flushed with water at intervals of 3, 7, 14, and 21 days with corresponding volumes of 2.5, 5, 10 and 20 L of distilled water. This sequence was repeated 4 times for Test #1 (for a total of 180 days) and 2 times for Test #2 (for a total of 90 days). During the resting periods between flushing events, alternating cycles of humidified and dry air were forced upwards through the columns. Discharge from the first column in series was collected and used to flush the second column in the series. Samples from all four columns were collected and submitted for chemical analyses.

E.4.2 Results

The column testing results are summarized in Table E.6. Note that all sulphate and NP depletion calculations have not been corrected for the likely storage of oxidation products within the columns.

Table E.6 Summary of Kinetic Testing Results

Parameter (Test and Column#)	Test #1 Column 1	Test #1 Column 2	Test #2 Column 1	Test #2 Column 2
ABA Data %S	4.2	2.3*	0.35	2.3*
AP	131	71.1*	10.9	71.1*
NP	30.2	82.6*	135	82.6*
NNP	-100.8	11.5*	124	11.5*
NP/AP ratio	0.23:1	1.2:1*	12.4:1	1.2:1*
pH: - maximum	7.02	7.74	7.88	7.19
- median	6.75	7.54	7.70	6.60
- minimum	6.03	7.30	7.43	5.88
Alkalinity (mg CaCO3 eq/L)**	20.5	42.3	60.0	17.6
SO_4 concentration (mg/L)				
- initial***	1553	1260	435	1712
- final**	513	656	128	1179
- median	833	990	182	1406
Zinc concentration (mg/L)**	3.0	0.04	0.02	6.5
SO_4 production				
rate (g/wk)**	2.47	3.44	0.53	5.23
overall normalized (mg/kg/wk)^	-	57	-	87
increment from layer (mg/kg/wk)^^	82	+33	18	+156
NP depletion				
rate (g/wk)**	2.77	4.90	0.98	5.51
overall normalized (g/kg/wk)^	-	0.082	-	0.092
increment from layer (g/kg/wk)^^	0.092	+0.071	0.033	+0.151
Zn production				
rate (mg/wk)**	16.4	0.30	0.12	36.0
overall normalized (mg/kg/wk)^	-	0.005	-	0.60
increment from layer (mg/kg/wk)^^	0.55	-0.54	0.004	+1.2
Time to overall test sulphide depletion (wks)	-	1210	-	**793**
Time to overall test NP depletion (wks)	-	1007	-	898
Time to column sulphide depletion (wks)	**1537**	318	583	807
Time to column NP depletion (wks)	328	**1901**	**4128**	200

Notes:
- * calculated overall ABA results for the combined columns.
- ** average of final 4 test cycles
- *** average of initial flush and first 4 test cycles
- ^ normalized to total mass (60 kg) to give overall rate from test series
- ^^ incremental is the amount contributed by each column

Controls (upper layers):

The results from the upper columns can be viewed as control samples representing drainage from the net acid consuming and net acid generating components available for blending.

The results from the acid generating column (Test #1, Column 1) indicated that there was a significant delay in production of acidic effluent in this material, with pH's in the range of 6.0 to 7.0 throughout the test period of 180 days. Alkalinity concentrations were in the range of 14 to 24 mg $CaCO_3$ /L. Sulphate levels decreased over the length of the test, reaching a relatively steady production rate of 82 mg/kg/wk. Zinc concentrations were approximately 3.0 mg/L. Depletion calculations indicated that the NP would have been consumed much sooner than the sulphides, probably within about 6 years at the measured laboratory rates compared to about 30 years for the sulphides.

The acid consuming column (Test #2, Column 1) had pH's in the range of 7.5 to 7.9, with sulphate production rates of about 18 mg/kg/wk and zinc concentrations of about 0.02 mg/L. Alkalinity concentrations were in the range of 43 to 73 mg $CaCO_3$ /L, about 3 times the concentration observed in the acid generating material. Calculations indicated that the sulphides would have been depleted within about 11 years, while the NP would have persisted for about 80 years at laboratory rates.

Layered Columns:

Discharges from the lower columns were evaluated by comparing the concentrations of key parameters to the concentrations emanating from the upper column, or by comparing the total loading rate (mg/wk) in the lower column to the loading rate from the upper column. The loading rates from the lower columns were normalized to the total mass of the combined columns (mg/kg/wk) since the loading rates represented the overall drainage from both layers. However it was possible to calculate a normalized incremental loading rate from each column by subtracting the loading rate (mg/wk) produced in the first column from the total loading rate produced in the second column, and then normalizing to the sample mass in each column. A negative incremental load from the second column therefore indicates that the second column is removing the dissolved constituent from the leachate.

The results from Test #1, Column 2 (the acid consuming base design) showed that the acid consuming base layer added alkalinity to the system, raised the pH, and lowered zinc concentrations. Long term projections indicated that the acid consuming layer may have contained sufficient NP to buffer the system. As water passed through the lower acid consuming column, the median pH of the leachate increased from 6.75 to 7.54, the alkalinity concentrations increased from a median of 20.5 to 42.3 mg $CaCO_3$ /L and the zinc concentrations decreased from 3.0 to 0.04 mg/L. The final pH and alkalinity levels from Column 2 were slightly lower as compared to the acid consuming control results (Test #2, Column 1), and the zinc concentrations were slightly higher than the acid consuming control. The incremental sulphate loading rate of 33 mg/kg/wk was much lower than the loading rate from the acid generating column (82 mg/kg/wk), but substantially higher than the incremental rate of 18 mg/kg/wk from the acid consuming control column (Test #2, Column 1), suggesting that the rate of sulphide oxidation in the underlying acid consuming layer may have been enhanced by the higher rates of oxidation in the upper column. Alternatively, analytical variability could have accounted for the differences. Depletion calculations indicated that the sulphides in each column would have been depleted in about 23 years, while NP would have been depleted within about 6 years in the upper column, and about 37 years in the lower column. Although it was expected that both sulphide and NP depletion rates would increase when the leachate from the upper column eventually becomes acidic, these results indicated that the acid consuming column would have continued to buffer the system and that it had a reasonable chance of controlling acid generation in the long term. Zinc mobilized in the upper column was attenuated in the lower column, with removal rates of 0.54 mg/kg/wk. This represented a 98% treatment efficiency. The removal mechanism was not studied, but was likely the result of secondary mineral formation.

In contrast, the results from Test #2, Column 2 (the alkaline cap design) showed that there was no benefit in layering acid consuming rock over the acid generating rock. In fact, the final discharge from Column 2 was of worse quality than discharge from the single acid generating control column. The leachate pH decreased from 7.7 to 6.6, alkalinity levels decreased from 60 to 17.6 mg $CaCO_3$ /L, sulphate concentrations increased from 182 to 1406 mg/L, and zinc concentrations increased from 0.02 mg/L to 6.5 mg/L as the leachate passed through the acid generating column. The incremental sulphate loading rate of 156 mg/kg/wk was almost 2 times the rate measured in the acid generating control column. Zinc loading rates were also about two times the rate seen in the acid generating control. Depletion calculations indicated that the sulphides would have been depleted in about 15 years, the NP in the upper layer would have lasted for about 80 years, and the NP in the lower column would have lasted for only about 4 years, suggesting that the overall discharge would have likely become acidic in the long term. It is unclear why the sulphate and metal concentrations from the test series were higher than from the acid generating control sample. Two possibilities are a) the test lasted only 90 days, and it's steady state concentrations had not been attained, or b) the lower column was flushed with slightly less water because of evaporative and sampling losses in the upper column and this affected the leachate concentrations. However, the input water volumes recorded in the spreadsheets were the same for the upper and lower columns.

E.5 Conclusions

- There was a moderate increase in the pH and alkalinity levels and a substantial (98%) reduction in zinc concentrations as leachate from the net acid generating column passed through the net acid consuming column in Test #1.

- The laboratory test results suggested that net acid consuming base layers as thick as 1 meter could have effectively mitigated seepage from 1 m of overlying acid generating rocks.

- Depletion calculations (uncorrected for oxidationproducts potentially stored in the columns) indicated that the net acid consuming base layer (Test #1) would have provided long term neutralization of generated acidity provided the relative rates of sulphide and NP depletion did not change when the upper column eventually began to produce net acidity.

- In contrast, the alkaline cap design (Test #2) did not appear to be a viable option even for short term control of sulphide oxidation and metal leaching. Leachate from the net acid consuming cap had high alkalinity and pH, but after passing through the underlying net acid generating materials, the leachate quality was actually worse than leachate from the net acid generating control sample.

- Incremental calculations should be used to evaluate the kinetic test results and estimate depletion times for the individual columns in the tests since this would provide better data on the kinetic behavior of the materials and provide a better basis for designing the waste rock pile.

E.6 Possible Research Areas

- Column tests in series appear to be a useful tool for simulating the kinetic behavior of acid consuming base layers. Additional evaluation of tests in series should be considered. In particular, data should be collected on the kinetic response of the acid consuming column to pH and loading changes as the acid generating column enters a more advanced stages of oxidation. Such testing would require careful selection of materials to ensure that the pH transition occurred over a reasonable time frame for testing.

- Tests carried out in series would also be useful for evaluating the scale of layering (thickness of layers) required to adequately control ARD and metal leaching, and the effectiveness of multiple layers.

- The removal mechanism for zinc should be studied in a laboratory scale system to determine the potential for remobilization of secondary zinc minerals.

- Field scale tests with more realistic flushing rates, longer time-frames and larger, more representative particle size distributions would also provide valuable information on the kinetic behavior of layered systems.

E.7 Key References

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T.W. Higgs and Associates Ltd., 1993. Eskay Creek Project: Acid Generation Characteristics of Waste Rock and Ore. Report prepared for Homestake Canada Inc.

Stewart, C.J., T.W. Higgs and W.A. Napier, 1994. Use of Lithologic Descriptions for Waste Rock Characterization, Case Studies from the Eskay Creek Project. *In* Proceedings of the Eighteenth Annual British Columbia Mine Reclamation Symposium, Mine Reclamation "Compliance and Protection", pp. 62-71.

APPENDIX F
WINDY CRAGGY PROJECT

F.1 Overview

The Windy Craggy property is located in northwestern British Columbia, approximately 200 km southwest of Whitehorse, Yukon Territory. The mineral deposit is a Cu-Co-Au-Ag-Zn massive sulphide with possible reserves of 297 million tonnes of ore (Claridge and Downing, 1993). Detailed exploration began in 1988, and feasibility and development studies were initiated in 1990. The creation of the Tatshenshini/Alsek Wilderness Park effectively closed the site to development in 1994.

The environmental studies included approximately 1247 acid base accounting tests, 38 conventional humidity cell tests, 18 modified humidity cell tests (~50 kg), and 9 large column tests (~650 kg) (Norecol, Dames & Moore, undated). The acid base accounting results were used to develop preliminary waste management plans for the project. The data was incorporated into a block model and used to estimate the proportions of net acid consuming and net acid generating materials (Downing and Giroux, 1993). The modified humidity cell and large column tests evaluated blending of net acid generating and acid consuming rock. The large column tests were also used to evaluate the possibility of layering acid generating or marginally acid consuming blends over a strong acid consuming base layer. To date, a detailed evaluation of the kinetic test results has not been done. This review is therefore limited to a preliminary interpretation by Mehling Environmental Management Inc. of the results that are specifically related to blending and layering. Some effort to evaluate the influence of the different test scales was also made.

The blending test results indicated that acid production and metal leaching could be effectively reduced if there was a sufficient ratio of acid consuming to acid generating material in the blend. This ratio varied depending on the specific characteristics of the blend materials, and the reactivity of the sulphides and NP in the blend. Blend ratios were calculated using total sulphur values to give maximum potential acid (MPA) and NP/MPA ratios. Generally, blends with an overall NP/MPA ratio of 1 would not provide long term control of acid generation at this site unless a very high portion (>90%) of the NP was reactive and available to buffer the pH in the neutral range. Given that a substantial portion of the carbonate described in the mineralogical reports was dolomite or siderite, this seems unlikely. Blends with NP/MPA ratios of 2 may have provided adequate long term reduction of ARP and blends with NP/MPA ratios of 3 or greater were considered very likely to prevent acidic drainage. Sulphate concentrations were limited by gypsum saturation in almost all of the modified humidity cell and column tests. Consequently, it was not possible to directly compare the rates of sulphide oxidation from the different blends. However, limited data from wash tests (a heavy flush test using de-ionized water) indicated that 1:1 blends may have accumulated a higher load of stored products than 2:1 blends. This would suggest that oxidation was proceeding at a faster rate in these materials, but that the products were retained. It should be emphasised that the net acid generating and acid consuming rocks in these tests were very well mixed.

The layered column tests indicated that an acid consuming layer located beneath an net acid generating layer would delay the breakthrough of acidic drainage for significant periods of time, but would not prevent acid drainage in the long term unless the overall NP/MPA ratio was greater than 1. The tests indicated that the underlying acid consuming layers would be an effective long term mitigation measure for acid drainage in designs where the upper layer had an uncertain potential for acid generation (i.e. NP/MPA ratios of about 1). Wash tests indicated that material in the acid consuming layers had a very high proportion of stored oxidation products. There was no evidence of reduced NP availability. However, it was possible that the long term effectiveness of the acid consuming layer would have been impaired by the accumulation of secondary minerals and associated blinding or coding of the available NP.

The evaluation of the effects of scale did not reveal any definitive information.

F.2 General Information:

Location	Northwest British Columbia, Canada
Mine owner	Royal Oak Mining Corporation (site previously owned by Geddes Resources)
Economic metals	Cu-Co-Au-Ag-Zn
State of operations	Feasibility studies from 1989 to 1992, property is now within the boundary of the Tatshenshini/Alsek provincial park and is closed to mineral development.
Open pit or underground	Planned open pit and underground
Geological reserves	297 million tonnes grading 1.38% Cu, 0.2 g/t Au, 3.83 g/t Ag, and 0.07% Co.
Average annual precipitation	2800 mm, 70% as snow
Other significant climatic information	-30 to 25 °C temperature range

F.3 Geology

The Windy Craggy deposit is a volcanic massive sulphide (VMS) deposit, hosted by clastic sediments and mafic flows and sills. The deposit has been subjected to extensive structural deformation, with steeply dipping faults and isoclinal and open folds. The massive sulphides occur near the transition of a predominantly clastic host to volcanic assemblages. Sulphide minerals include pyrite, pyrrhotite and chalcopyrite with lesser amounts of sphalerite. Surface portions of the ore zones are characterized by supergene copper enrichment, and are overlain by gold and silver enriched gossan caps. Supergene minerals include chalcocite, native copper, chalcanthite and limonite (Claridge and Downing, 1993).

The major waste rock types are volcanics and argillites, with lesser amounts of gabbro and exhalite. The argillites and the volcanics can be divided into three subgroups, calcareous, sulphidic and non-calcareous/non-sulphidic.

The calcareous argillites vary from limy sediments to argillaceous limestone with less than 5% massive to finely laminated sulphides. Carbonate stringers are pervasive in areas where this rock has been extensively fractured. The sulphidic argillite contains between 2 and 40% pyrrhotite and pyrite as laminae, blebs, bands and irregular stringers, with disseminated cubes of pyrite. The sulphidic argillites are generally non-calcareous. Non-calcareous and non-sulphidic argillite is composed of argillite, chloritic argillite, silicified sediments or argillaceous cherty beds. Minor limestone and chert are often present within this unit (Downing and Giroux, 1993).

The calcareous volcanics are of mafic to intermediate composition with less than 5% sulphides and pervasive carbonate and chloride alteration. Sulphidic volcanic rocks contain less than 20% sulphides as fine to medium grained pyrrhotite and/or weakly developed stringer/stockwork veins that are comprised of sulphide or carbonate/quartz and sulphide. Non-calcareous and non-sulphidic volcanics are chlorite altered with less than 5% carbonate (Downing and Giroux, 1993).

Because it was not considered practical to delineate the sub-groups of argillite and volcanic rock in the block model of the deposit, waste rock was classified on the basis of the calculated NP/MPA ratios. The waste rock block model was developed using static testing data and correlated parameters from the more extensive geochemical database. The methods used to develop the waste rock block model are discussed in Downing and Giroux, (1993). A final breakdown of the waste rock categories was not presented in the paper.

F.4 Static Tests

F.4.1 Methods

The acid base accounting tests were conducted at B.C. Research Laboratories in Vancouver, B.C. It is not clear which titration method was used to determine the NP. The maximum potential acid production (MPA) was determined from the total sulphur content measured by LECO furnace.

Acid base accounting tests were conducted on 1247 drill core samples. These results were used in conjunction with the assay database for the site to develop the waste rock ARD block model. The results were not available for this review.

Acid base account tests were also done on the 7 bulk samples collected for the large scale kinetic tests. The bulk rock samples were obtained from an underground exploration drift. The samples were jaw crushed to achieve the grain size distribution expected for waste rock produced during operations. The resulting material was uniformly graded with 100% of the mass finer than 10 cm and not less than 5% finer than 0.1 cm. The bulk samples were thoroughly mixed using cone and quarter techniques. A sample was split from the mixture for the ABA and standard humidity cell tests, and the remainder was used for the large columns and modified humidity cell tests. The acid base accounting tests were done in triplicate. Composites made from the bulk samples were also mixed using the cone and quarter technique.

F.4.2 Results

The acid base accounting results from the bulk samples are presented in Table F.1. The results indicate there are three strongly acid consuming bulk samples, 2 strongly acid generating samples and 2 samples with NP/MPA ratios between 1 and 3.

Table F.1 Acid Base Accounting Test Results, Bulk Samples

Sample No.	Rock Type	S%	MPA	NP	NNP	NP/MPA
V1	mafic volcanic	2.03	63.5	239	176	4:1
V2	calcareous volcanic	0.11	2.5	184	182	>70:1
A3	calcareous argillite	1.26	39.4	472	433	12:1
A4	sulphidic argillite	4.79	148	309	159	2:1
A5	sulphidic argillite	9.6	300	504	204	1.7:1
V6	stringer zone volcanics	2.51	78.4	17.8	-61	0.2:1
A7	stringer zone argillite	17.1	534	34.9	-499	<0.1:1

F.5 Kinetic Testing

W.5.1 Methods

A total of 18 modified humidity cell tests and 9 large column tests were used to characterize the kinetic behaviour of the blended and layered designs.

Modified Humidity Cell Tests

The modified humidity cell samples were selected or prepared from the bulk rock samples described in the static testing section. Bulk rock samples V6 and A7 (acid generating samples) and samples A4 and A5 (material with NP/MPA ratios of about 2) were tested individually, a composite of the three net acid consuming samples (V1, V2 and A3) was tested, and 10 blended composites with NP/MPA ratios of approximately 1, 2 and 3 were tested. In addition, there were 3 duplicate tests. This review focused on results from the acid generating samples V6 and A7, the acid consuming blend (V1, V2 and A3), and the 8 blended composites where net acid generating rock was blended with strongly net acid consuming rock at different ratios.

The modified humidity cells were constructed from 0.5 m lengths of 41 mm diameter PVC pipe fitted with a perforated acrylic base plate. Approximately 50 to 60 kg of sample was placed in each cell. Water was added to the top of the cell using a drip system connected to a peristaltic pump. The drip line was rotated each week to allow contact with a variety of flow paths. The weekly cycle consisted of 3 days of moist air followed by 3 days of dry air and a flushing cycle on the 7th day. In the flush cycle, 500 mL of water was added to the top of the cell and allowed to percolate through the crushed rock. The leachate was collected and tested for pH, redox, conductivity, acidity, alkalinity, sulphate, Ca, Mg, and Fe on a weekly or biweekly basis. Samples from weeks 2, 4, 6, 8, 10, 12, 14, 87 and 103 were submitted for additional metal analyses by ICP. A wash test was carried out on most of the humidity cells at the end of the testing program. The wash test was done in stages, with 5 rapid flush cycles, where approximately 12 litres of water was added over a period of 24 hours during each of the cycles. The specific conductivity of the wash water was recorded and used to monitor the removal of stored salts in the cells. At the end of the test, a composite sample was prepared using water from each of the 5 wash cycles. The composite was submitted for pH, redox, sulphate, acidity, alkalinity and metals analyses.

<u>Large Column Tests</u>

The column testing program is summarized in Figure F.1. A total of 9 columns including 4 sets of columns in series, and one individual column were tested. The upper part of the column series contained either a net acid generating composite (Column 8), or a blended composite with an NP/MPA ratios of 1 (Column 2), 2 (Column 4) or 3 (Column 6). The lower portion of these columns (Columns 1, 3, 5 and 7) contained a net acid consuming composite with an overall NP/MPA ratio of 10. The individual column (Column 9) also contained this acid consuming composite. Samples were collected from both the upper and lower columns to allow the results from the upper layers to be effectively interpreted as individual large columns.

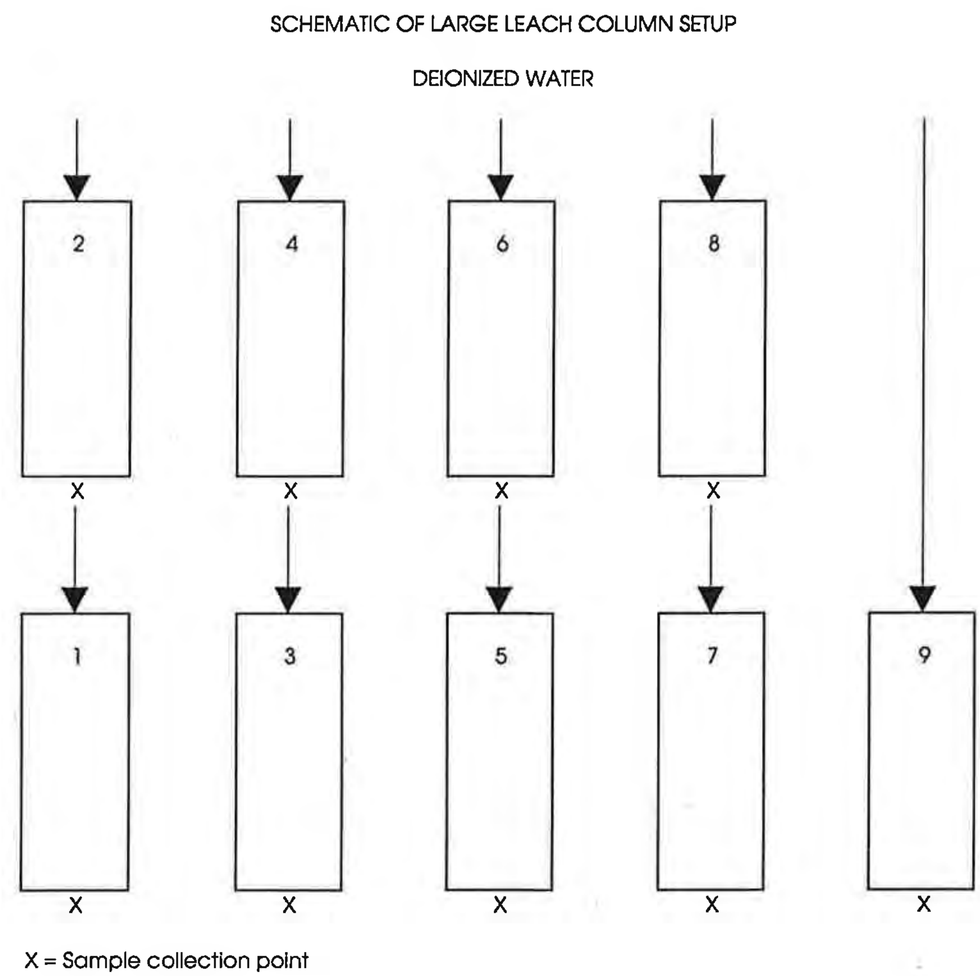

Figure F.1 Summary of Column Testing Program
(Norecol, Dames & Moore, undated).

The large column tests were constructed with 41 cm diameter PVC pipe. The columns were 3 metres long and were filled with approximately 650 kg of sample. Sample collection ports were located at the base of each column. Effluent from Columns 2, 4, 6 and 8 were fed into Columns 1, 3, 5, and 7. Column 9 was operated individually.

Approximately 6.4 litres of water per week were added to the top of each upper column using a drip system. On average, about 5.0 litres of this water were recovered and added to the lower column. Leachate from all of the columns was collected and analysed for pH, redox, conductivity, acidity, alkalinity, sulphate, Ca, Mg, and Fe on a weekly or biweekly basis. Samples from weeks 2, 4, 6, 8, 10, 12, 14, 87 and 103 were submitted for additional metal analyses by ICP. Columns 1, 2, 3, and 4 were operated for a total of 59 weeks and Columns 5, 6, 7, 8, and 9 were operated for a total of 103 weeks. A wash test was carried out on Columns 5, 6, 7, 8, and 9 at the end of the testing program. The wash test was done in stages, with 6 rapid flush cycles, where approximately 100 to 150 litres of water were added over a period of about 24 hours during each of these cycles. The specific conductivity of the wash water from each cycle was recorded and used to monitor the removal of stored salts from the columns. At the end of the test, a composite sample was prepared using water from each of the 6 wash cycles. The composite was submitted for pH, redox, sulphate, acidity, alkalinity and metals analyses.

Interpretation of Large Scale Tests

Interpretation of the modified humidity cell and large column results was undertaken by MEM Inc. from data compiled Norecol, Dames & Moore for Royal Oak Mining Corporation. Interpretation was somewhat complicated by secondary mineral formation (particularly gypsum) in the columns. The retention of calcium and sulphate in the columns resulted in underestimating the NP and sulphide depletion rates. The depletion estimates were improved by correcting for the residual load removed by the wash tests at the end of the testing program, where these tests were available. More thoroughly flushed tests which limited storage of secondary minerals would have provided more accurate depletion estimates.

The use of less thoroughly flushed large scale tests may have some other potential advantages. Specifically, the large tests may more closely represent conditions expected in the field, with more realistic grain size distributions and flushing rates, and longer flow paths. If secondary minerals precipitated in the laboratory scale tests, one may assume they are also likely to precipitate in a full-scale dump. Thus, while accurate reaction rates are not generally obtained in under flushed large scale tests, they may provide useful insight into the hydrological and geochemical conditions in a full-scale dump. When interpreting the test results from the underflushed large scale tests, especially those without wash test corrections, qualitative differences in the bulk chemical behaviour of the mixtures should be considered rather than numerically predicted behaviour based on NP and sulphide depletion rates.

F.5.2 Results and Discussion:

Modified Humidity Cells

The modified humidity cell results are summarized in Table F.2. The results are discussed in two parts: a) acid generating and acid consuming control samples; and, b) 1:1, 2:1 and 3:1 blended samples.

a) Control samples:

Cell 1 contained net acid generating stringer zone volcanics (sample V6). The effluent pH from this sample decreased from 7.8 in the early part of the test to 3.3 by the end of the 59 week testing period. The transition from near neutral pH's to acidic pH's was gradual, occurring between about weeks 35 and 50. Sulphate concentrations produced by this sample were relatively low for a net acid generator, ranging from 754 mg/L in the early part of the test to a maximum of 2360 mg/L in the 44th week of testing. The wash test results indicated that a relatively small portion of the oxidation products were stored in the cell, about 13% of the sulphate and 18% of the NP products. The ABA results and depletion calculations indicated that there was some residual NP buffering,

Table F.2 Summary of Modified Humidity Cell Test Results

Parameter / Cell #	1	2	18	4	5	6	7	8	10	11
Bulk sample #	V6	A7	V1+V2 +A3	V6+V2		V6+V1			A7+V1	
%S	2.51	17.1	0.93	1.86	1.41	2.30	2.18	2.05	5.95	3.52
MPA	78	534	29	58	44	72	68	64	186	110
NP	18	35	290	57	86	72	136	194	186	218
NP/MPA ratio	0.23	0.07	9.98	0.98	1.95	1.00	2.00	3.03	1.00	1.98
Weeks of testing	59	59	52	59	59	103	59	103	59	59
pH initial**	7.8	6.7	9.1	7.9	7.8	7.8	7.9	7.9	7.9	7.8
final**	3.3	3.2	8.0	7.7	7.8	7.7	7.7	7.8	7.8	7.9
median	7.3	3.2	8.0	7.7	7.6	7.7	7.7	7.7	7.8	7.8
Alkalinity Load										
(mg/kg/wk)***	0.15	0.0	0.71	0.31	0.26	0.26	0.30	0.33	0.29	0.28
(mg/kg) total	13.0	2.0	122.8	21.8	17.8	18.6	22.2	24.7	20.5	19.7
SO_4 Conc. (mg/L)										
initial**	754	2183	3033	1063	974	884	826	662	1408	854
final**	1859	2558	1905	1855	1496	1801	1576	1589	1853	1623
median	1720	3530	2930	1690	1660	1985	1730	1700	1885	1700
SO_4 Load										
(mg/kg/wk)***	14.2	23.5	25.9	14.5	13.8	17.4	15.8	16.0	14.4	13.8
(mg/kg) flushed	833	1471	1521	794	813	1169	910	1071	858	766
(mg/kg) stored	128	520	537	397	368		556		1021	271
(mg/kg) total	961	1991	2059	1190	1181		1466		1878	1037
NP Product Load										
(g/kg/wk)***	0.016	0.015	0.011	0.015	0.015	0.017	0.017	0.016	0.015	0.014
(g/kg) flushed	0.92	0.88	0.62	0.91	0.86	1.02	1.00	0.93	0.90	0.84
(g/kg) stored	0.20	0.48	0.46	0.42	0.39		0.59		1.06	0.75
(g/kg) total	1.12	1.36	1.08	1.33	1.24		1.59		1.96	1.60
Time to S depletion (yrs)	**75**	**349**	18	**44**	39	(81)*	52	(76)*	102	111
Time to NP depletion (yrs)	17	28	**303**	48	**77**	(78)*	**96**	**(235)***	106	**154**
Metals (mg/L)***										
Cu	0.003	0.002	0.294	0.005	0.010	0.006	0.003	0.005	0.002	0.002
Co	0.003	0.120	0.001	0.004	0.006	0.005	0.004	0.003	0.007	0.004
Fe	0.21	66	0.33	0.30	0.35	0.30	0.30	0.20	0.35	0.30
Mn	0.20	20.00	0.03	0.22	0.23	0.31	0.41	0.33	1.00	0.67
Ni	0.003	0.038	0.079	0.003	0.011	0.016	0.005	0.008	0.003	0.003
Zn	0.004	0.016	0.003	0.002	0.006	0.007	0.010	0.008	0.007	0.007

Notes: * estimate not corrected for stored load because no wash test results were available
** initial concentrations are the average of the first four test cycles and final concentrations are the average cycles 56-59 for all cells except MHC 18
*** median values presented

however, it was likely in a poorly reactive form such as dolomite or siderite, or was physically unavailable, because it did not provide sufficient buffering to maintain neutral pH conditions. Metal concentrations in the effluent were relatively low during the early part of the test, however the analyses were discontinued before the pH had decreased.

Cell 2 contained net acid generating material from the argillite stringer zone (sample A7). This sample was much more reactive than sample V6 in cell 1, with pH levels decreasing from 7.4 to below pH 4.0 by the 10th week of testing. Sulphate concentrations were elevated, with a median concentration of 3530 mg/L and a maximum concentration of 5040 mg/L. The wash test results indicated that there was a moderate stored contaminant load in the sample, with 26% of the sulphate and 35% of the NP products removed during the wash. Depletion calculations also suggested that there was some residual NP buffering in this sample. Metal concentrations were moderately high in this sample (with the exception of copper), reflecting the low pH and moderate sulphide oxidation rates.

Cell 18 contained a composite of the net acid consuming samples, including the mafic volcanics, calcareous mafic volcanics and calcareous argillite (samples V1, V2 and A3). This sample released high sulphate concentrations during the first 30 weeks of testing, which may be partially attributed to release of stored oxidation products. The pH of the cell effluent was in the alkaline range, with median pH levels of 8.0. Alkalinity concentrations were high, with initial concentrations of 1390 mg $CaCO_3$ equivalents/L and steady state concentrations of about 60 mg $CaCO_3$ equivalents/L. The wash test results indicated that there was a moderate load of stored oxidation products at the end of testing, with approximately 26% of the sulphate and 43% of the NP products recovered in the wash test. Depletion calculations suggested that the sulphide in this sample would be depleted within about 18 years, while the NP would persist for up to 300 years. Metal concentrations were generally low, however Cu and Ni concentrations were higher than Cu and Ni levels in either of the acid generating columns.

b) Blended tests:

Cells 4 and 5 contained well mixed blends of net acid generating volcanic stringer zone material (V6) and net acid consuming calcareous volcanics (V2) at ratios of 0.77:0.23 and 0.58:0.42 respectively, resulting in an NP/MPA ratio of 1 for Cell 4 and 2 for Cell 5. Both cells had very similar leachate quality, with neutral pH's, moderate sulphate concentrations of 1690 mg/L and 1660 mg/L, and moderate alkalinity release rates. The wash test results also were very similar, with sulphate loads of 397 and 368 mg/kg removed from each of the cells. Depletion calculations for the 1 blend (Cell 4) indicated that the sulphides would likely persist for 44 years, while the NP would be depleted within 48 years. Unless at least 93% of the NP was reactive and available in the neutral pH range, this sample would probably produce acidic leachate in the long term. Depletion calculations for the 2:1 blend indicated that the sulphides would persist for approximately 39 years, while the NP would last for approximately 77 years. Unless the NP was less than 50% reactive, this sample would likely remain neutral. The metal concentrations from both of the tests were very low.

Cells 6, 7 and 8 contained well mixed blends of net acid generating volcanic stringer zone material (V6) and net acid consuming mafic volcanics (V1) at ratios of 0.76:0.24, 0.48:0.52 and 0.21:0.79 respectively, resulting in a NP/MPA ratios of approximately 1 for Cell 6, 2 for Cell 7 and 3 for Cell 8. The leachate quality of these three samples was very similar, with neutral pH's throughout testing, moderate sulphate and alkalinity concentrations and slightly higher NP depletion rates (0.017 g NP/kg/wk) compared to the other cells (0.011 to 0.016 g NP/kg/wk). The sulphate concentrations decreased with increasing NP/MPA ratios, with a median value of 1985 mg/L in the 1:1 cell, 1730 mg/L in the 2:1 cell and 1700 mg/L in the 3:1 cell. Alkalinity showed the reverse trend, with the highest alkalinity release rate (0.33 mg $CaCO_3$ /kg/wk) in the 3:1 cell and the lowest (0.26 mg $CaCO_3$ /kg/wk) in the 1:1 cell. The main difference between the cells was that the ratio of NP depletion time to sulphide depletion time increased with increasing NP/MPA ratios. Because the wash test correction could not be applied to two of the cells, this apparent trend should be viewed with some skeptism. Based on uncorrected NP and sulphate depletion rates, it appears that the 1:1 blend would not be sufficient to prevent acid drainage generation, the 2:1 blend would prevent acid drainage generation provided the NP was at least 54% reactive and the 3:1 blend would only generate net acidity if the NP was less than 32% reactive. Metal concentrations in the drainage from these columns were low, and there were no apparent relationships between metal concentrations and NP/MPA ratio.

Cells 10 and 11 contained well mixed net acid generating materials from the argillite stringer zone (A7) and net acid consuming mafic volcanics (V1) at blend ratios of 0.26:0.74 and 0.1:0.9 respectively, resulting in NP/MPA ratios of approximately 1 for Cell 10 and 2 for Cell 11. The leachate chemistry was also very similar within this group, with neutral pH's, moderate sulphate and moderate alkalinity concentrations. Sulphate concentrations in the 1:1 blend were somewhat higher than in the 2:1 blend, with median concentrations of 1885 mg/L in the 1:1 blend compared to 1700 mg/L in the 2:1 blend. Wash tests done on the cell indicated that there was a very large stored sulphate and NP load in the 1:1 blend, but not in the 2:1 blend. Approximately 54 percent of the sulphate flushed from the 1:1 blend was removed during the wash test, while only 26 percent was removed from the 2:1 blend. Depletion calculations indicated that the 1:1 blend would probably generate net acidity at some point in the future. The sulphides in this sample would have persisted for about 100 years, while all of the NP would have been depleted within about 106 years. Therefore, unless more than 96 percent of the NP in this sample was available for reaction, the usable NP would have been depleted before the sulphides. It is less clear whether the 2:1 blend could have produced net acidity in the long term. At estimated depletion rates, the NP would have to be more than 72% reactive to neutralize all of the acidity produced by the sulphides. Metal levels were generally low in this sample, however the manganese content was higher than observed in the other blended samples, possibly as a result of the argillite content of this material.

In summary, the following observations apply to each group of blended cells which contained the same starting materials.

- The leachate water quality within each of the blending groups was very similar despite variations in NP/MPA ratios ranging from 1:1 to 3:1.

- Leachate sulphate concentrations tended to be higher in the 1:1 blends and lower in the 2:1 or 3:1 blends from the same group. This relationship was likely influenced by gypsum formation in the modified cells, which limited sulphate concentrations. It was not possible to compare the actual sulphate loads to expected loads based on the composite leachate because the acid consuming members of the blends were not tested individually, and the sulphate concentrations in all of the cells were limited by gypsum saturation.

- There were no apparent trends in metal concentrations or loadings except that the metal concentrations were much higher in the net acid generating control samples. It is likely that metal concentrations in the neutral pH columns were limited by the formation of secondary minerals.

- The main difference within blending groups that contained the same starting materials was the estimated time to NP depletion, which tended to be longest in samples with the highest NP/MPA ratios.

- Mixtures with NP/MPA ratios of 1 did not contain sufficient reactive NP to control acid drainage generation in the long term at this site.

Large Column Tests:

Results from the large column tests are summarized in Table F.3. The results are discussed in three parts: a) controls (Columns 8 and 9) which represent net acid generating and net acid consuming material; b) blended columns (2, 4 and 6), representing blends with NP/MPA ratios of approximately 1, 2 and 3; and, c) layered column results showing the results of layering net acid generating, 1:1, 2:1 or 3:1 blends over a net acid consuming base layer (Column pairs 7+8, 1+2, 3+4 and 5+6).

Table F.3 Summary of Large Column Test Results

Parameter / Column	8	2	4	6	9	7 + 8	1 + 2	3 + 4	5 + 6
Layer	(upper) <0.1:1	(upper) 1:1	(upper) 2:1	(upper) 3:1	(single) 10:1	(base)	(base)	(base)	(base)
%S**	14.5	2.3	4.8	2.23	0.93	7.72	1.62	2.87	1.58
MPA**	453	72	150	70	29	241	50	90	49
NP**	6	72	309	186	290	148	181	300	238
NP/MPA ratio**	0.01	1.0	2.1	2.7	10.0	0.6	3.6	3.3	4.8
pH initial***	5.8	7.1	8.7	7.8	7.8	7.7	7.7	7.8	7.8
final***	3.4	7.8	8.0	7.9	7.7	8.0	7.7	8.0	8.0
median	3.3	7.8	8.0	7.8	7.8	7.9	7.9	8.0	7.9
Alkalinity									
(mg/L)****	0	34	70	39	34	74	67	78	58
(mg/kg/wk)****	0.00	0.30	0.58	0.36	0.33	0.24	0.23	0.30	0.20
(mg/kg) total	1.89	20.67	61.27	24.85	22.01	14.46	12.98	17.93	11.93
SO_4 Conc. (mg/L)									
initial***	2498	1922	5021	1400	1164	1396	1354	1336	1637
final***	4346	1808	2131	1624	1524	2605	2011	2375	1815
median	7440	2000	2580	1830	1720	3125	2180	2975	2030
SO_4 Load^									
(mg/kg/wk)****	53.6	17.0	23.4	16.6	16.3	10.2	7.5	12.1	7.6
(mg/kg) flushed	3207	1015	1790	976	936	637	422	829	464
(mg/kg) stored	159			507	198	1628			878
(mg/kg) total	3365			1483	1134	2265			1342
NP Product Load^									
(g/kg/wk)****	0.019	0.018	0.014	0.018	0.017	0.012	0.008	0.006	0.008
(g/kg) flushed	1.13	1.03	0.84	1.04	1.00	0.68	0.46	0.37	0.45
(g/kg) stored	0.69			0.60	0.31	1.55			0.98
(g/kg) total	1.82			1.64	1.31	2.23			1.43
Time to S depletion (yrs)	**223**	(91)*	(158)*	56	30	**120**	(138)*	(193)*	**41**
Time to NP depletion (yrs)	3	(78)*	**(418)***	**128**	**250**	74	**(445)***	**(924)***	**188**
Metals (mg/L)****									
Cu	0.002	0.004	0.009	0.002	0.002	0.002	0.005	0.002	0.003
Co	0.3	0.006	0.002	0.004	0.004	0.006	0.006	0.005	0.003
Fe	20	0.30	0.15	0.21	0.30	0.35	0.21	0.18	0.25
Mn	15	0.39	0.004	0.36	0.04	0.05	0.05	0.01	0.03
Ni	0.19	0.03	0.015	0.01	0.01	0.03	0.01	0.05	0.02
Zn	0.03	0.008	0.008	0.005	0.006	0.007	0.008	0.007	0.007

Notes:
- * estimate not corrected for stored load because no wash test results were available
- ** ABA data for the base columns are the overall values for the 2 layer design.
- *** initial concentrations are the average of the first four test cycles and final concentrations are the average cycles 56-59 for all columns
- **** median values presented
- ^ loads in the base columns are normalized to the total mass of rock in both the upper and lower columns

a) Control columns:

The net acid generating composite (Column 8) produced acidic leachate within 8 weeks of testing. Despite the large rock:water ratio used in this test (650 kg/6.4 L water/wk), the pH did not decrease below pH 3.3, suggesting that there was some residual buffering by dolomite or another less reactive mineral than calcite. Dolomite buffering is supported by the chemical data which indicates that the molar ratio of Mg/Ca is slightly greater than 1 in the leachate. The sulphide oxidation rate was high compared to the other column samples, with a median value of 54 mg SO_4/kg/week. This probably reflected a reasonably accurate estimate of the rate because there was only a small portion of stored oxidation products in the wash test, (approximately 5% of the sulphate load and 38% of the NP products). Depletion calculations corrected using wash test results indicated that any residual NP would have been used up within 3 year of testing, while the sulphides would have persisted for another 220 years.

The net acid consuming composite (Column 9) was characterized by a neutral to slightly alkaline pH throughout testing, a sulphide oxidation rate of 16.3 mg SO_4/kg/week and a moderate alkalinity. This sample also contained only a small portion of stored oxidation products, with approximately 17% of the sulphate load and 24% of the NP products removed in the wash test. Depletion calculations corrected using wash test results indicated that this material would not have produced net acidity in the long term. The sulphides would likely have been depleted within 30 years while the NP would have likely persisted for up to 280 years, assuming 100% of the NP was available and reactive.

b) Blended columns:

The 1:1 blended column (Column 2) contained a blend of 24% acid consuming mafic volcanics and 76% acid generating stringer zone volcanics. The effluent was characterized by neutral pH, moderate sulphate concentrations and moderate alkalinity concentrations. Sulphate concentrations were probably limited by gypsum saturation throughout testing. However, no wash tests were done on this material; so it was not possible to quantify the stored oxidation load. Depletion calculations based on the late cycle SO_4, Ca and Mg concentrations in the column leachate and uncorrected for potential stored oxidation products indicated that this material would probably produce acidity at some point in the future: all of the NP would be depleted within 78 years, while the sulphide would persist for 91 years.

The 2:1 column (Column 4) contained a single bulk sample described as sulphidic argillite with an overall NP/MPA ratio of 2. The column test results reflected the difference in host rock type. The pH of the effluent was in the alkaline range throughout testing, with median values of 8.0. This corresponded to a relatively high alkalinity release rate of 0.58 mg/kg/week. The sulphate concentrations were moderate with estimated sulphide oxidation rates of 17 mg SO_4/kg/week, similar to Column 9 (net acid consuming material). Sulphate concentrations were probably limited by gypsum saturation for most of the test; however, no wash test results were available to quantify the stored load. Depletion calculations uncorrected for potential stored oxidation products indicated this material would only produce net acidity if the reactive portion of the NP was less than 38% of the total NP.

The 3:1 column (Column 6) contained a blend of 79% acid consuming mafic volcanics and 21% acid generating stringer zone volcanics (the same rock types as the 1:1 blend). The column test results were very similar to the 1:1 column results (Column 2), possibly reflecting the similarity in host rock materials. Alkalinity release rates were slightly higher than in the 1:1 blend (24.9 mg/kg/wk compared to 20.7 mg/kg/wk in the 1:1 blend) and sulphate concentrations were slightly lower (1830 mg/L compared to 2000 mg/L). A wash test carried out on this material showed that at least 34% of the sulphate load and 37% of the NP products were stored in the column. Depletion estimates corrected using wash test results indicated this material would only produce net acidity if the reactive NP was less than 44% of the total NP.

Although gypsum saturation limited direct comparison of sulphide oxidation rates between the different blends, some general comparisons regarding the effectiveness of blending can be made:

- The pH of the blended columns remained neutral for up to 103 weeks of testing, indicating that the release of acidic effluent was significantly delayed or prevented by the acid consuming portion of the blend.

- Depletion calculations indicated that the 1:1 blend could generate acidic drainage in the long term. However, the 2:1 and 3:1 blends would probably not generate acidic drainage as long as a sufficient portion of the NP was reactive.

- The specific types of rock in a blend appeared to be as important for determining long term kinetic behaviour as the NP/MPA ratio of the blend. The 2:1 sample containing argillite had both a higher calculated sulphide depletion rate and a lower NP depletion rate than the 3:1 blended sample containing volcanics (Column 6). This appeared to result in a similar ratio of NP to sulphide depletion times, i.e. the 3:1 blend may have performed no better than the 2:1 material in the long term. However, the inability to correct Column 2 results for stored oxidation products may modify this apparent finding.

- Comparisons of the metal concentrations indicated that pH control was likely the most significant control on metal concentrations. Drainage from the blends contained substantially lower metal concentrations than drainage from the net acid producing sample (Column 8) but also higher pH values.

c) Layered column results:

Column 7 was net acid consuming material which received leachate from Column 8 (net acid generating material). Although the material in Column 7 was strongly acid consuming, the combined overall NP/MPA ratio of the two layers was only 0.6. Leachate from Column 7 was characterized by neutral pH levels, and elevated sulphate concentrations. Sulphate and metal concentrations were very low compared to the feed concentrations from Column 8. The reduction in sulphate and metal concentrations was likely the result of secondary mineral precipitation in the lower column due to reactions with carbonates and the resulting increase in pH and calcium concentrations. This assumption is supported by the column wash test results which indicated a very high proportion of the sulphate load had been stored in the column. Depletion calculations corrected for wash test results indicated that the NP would likely have been depleted sooner than the sulphides (74 versus 120 years), suggesting that the base layer represented by Column 7 would not have provided effective long term control of acid generation.

Column 1 contained net acid consuming material which received leachate from Column 2 (the 1:1 blended column). The combined NP/MPA ratio of the two layers was 3.6. Leachate from Column 1 had a neutral pH, moderate sulphate concentration and elevated alkalinity throughout testing. Sulphate concentrations were slightly higher than sulphate concentrations in the feed water from Column 2, but metal concentrations were slightly lower, indicating there was some additional sulphate loading from the lower column, but that the metals were retained within Column 1. No wash test results were available to quantify the stored load. Depletion calculations uncorrected for stored oxidation products indicated that the leachate from the lower column would have remained at a neutral pH as long as 30% or more of the NP in the two columns was reactive.

Column 3 contained acid consuming material which received leachate from Column 4 (the 2:1 blended column). The combined NP/MPA ratio of the two layers was 3.3. Leachate from Column 3 had a neutral pH, slightly elevated sulphate and low metal concentrations. Sulphate concentrations were slightly higher than sulphate concentrations in the feed, indicating there was some additional contribution by the lower column. Depletion calculations uncorrected for stored oxidation products indicated that the leachate from the lower column would have remained neutral as long as 21% or more of the NP in the columns was reactive.

Column 5 contained acid consuming material which received leachate from Column 6 (the 3:1 blended column). The combined NP/MPA ratio of the two layers was 4.8.

Leachate from Column 5 was very similar to leachate from Column 1, with neutral pH's, moderate sulphate and slightly elevated alkalinity concentrations. The sulphate concentrations were slightly higher than the feed concentrations. The wash tests indicated that there was a significant load of stored oxidation products retained in the lower column. Depletion calculations corrected for wash test results indicated this sample would not have produced acidic drainage as long as at least 22% of the NP in the columns was reactive.

In summary, the layered column results indicated the following:

- The final leachate from all of the layered columns had neutral pH levels, moderate sulphate concentrations and somewhat elevated alkalinity concentrations. Sulphate and alkalinity concentrations appeared to be controlled in most columns as a result of gypsum solubility limitations.

- There was usually a small increase in sulphate concentrations as the leachate moved through the lower net acid consuming column, but there was also a significant accumulation of oxidation products. In one case, (Columns 1+2), sulphate concentrations decreased as the water moved through the lower column indicating there was significant removal of gypsum in this lower layer.

- Depletion calculations suggested that the net acid consuming layers would provide long term control of acidic drainage when the upper layer had an "uncertain" potential to generate acidic leachate. However it would only delay the onset of acid generation when the overlying layer was strongly acid generating and the overall NP/MPA ratio was less than 1.

Scale Effects:

A preliminary examination of potential scale effects compared the modified humidity cell test results to the large column test results. The column tests contained about 10 times more sample than the modified humidity cell tests, and were flushed with about 10 times more water, so the rock to water ratios were similar. However, the humidity cell and column diameters were the same, so the columns were effectively subjected to a higher irrigation rate which would likely have allowed more contact with particle surfaces, but this depended on channelization with the test vessels.

The results of the comparison were highly variable. For the tests containing the net acid generating materials (V6 and A7), sulphate concentrations and loadings from the column (Column 8) were much higher than the concentrations and loadings from the large scale humidity cells (Cell 1 and 2). For the 1:1 and 3:1 blends, sulphate concentrations in both tests were very similar, but the sulphate loads (normalized to mass) from the humidity cells were slightly higher than loads from the column tests. The column containing an acid consuming composite had lower sulphate concentrations and loadings than the humidity cell containing the same material.

F.7 Conclusions

Blending Conclusions:

- The humidity cell and column tests indicated acidic drainage could be prevented if there was a significant ratio of acid consuming to acid generating materials. Depletion calculations indicated that blends with an NP/MPA ratio of 1 would likely generate ARD in the long term, blends with an NP/MPA ratio of 2 would probably prevent ARD, and blends with an NP/MPA ratio of 3 would likely prevent ARD. The optimal blend appeared to depend on the specific starting materials used in the test and the reactivity of the NP. For example, a higher NP/MPA cutoff might be specified for the argillite materials.

- Sulphate concentrations in the leachate tended to be higher in blends with the lowest NP/MPA ratios This relationship was likely influenced by gypsum formation which limited sulphate concentrations.

- As long as the final leachate pH remains neutral, it appears that the metal levels are adequately controlled in this system.

Layering Conclusions:

- Underlying layers of acid consuming materials are an effective short term barrier for sulphate and metals produced by overlying acid generating materials.

- Depletion calculations suggested that acid consuming layers would prevent release of acidic effluent when materials with an "uncertain" potential for acid generation (NP/MPA ratios between 1 or 2) were layered above the acid consuming material, provided that the overall NP/MPA ratio was greater than 3.

- Acid consuming base layers accumulated significant amounts of stored oxidation products from the overlying net acid generating layers which may limit the effectiveness of the NP in the underlying layer over the long term.

Implementation Issues

- Differences as a result of scaling from the humidity cells to the larger column tests were not evident in the testing data.

- In these well mixed tests, potentially net acid generating and net acid consuming materials were situated in close physical proximity. When extrapolating these results to the field, it would be necessary to consider the spatial arrangement of the two material types and to ensure that they are sufficiently mixed. The required degree of mixing is not known.

F.8 Possible Research Areas

The Windy Craggy acid generation testing results contain extensive information that could not be included in this preliminary data review (Norecol, Dames & Moore, undated). This set of data may be an excellent subject for further study, although limited by an inability to correct for stored oxidation products in many of the tests. Such a project would likely require input and collaboration from Norecol Dames and Moore, SRK and B.C. Research who were involved in the original column testing program, and should include specialized tests on the residual samples. Specific recommendations include:

- Compare the standard humidity results to the larger tests to see if more accurate depletion estimates can be obtained.

- Review mineralogical data and compare with leachate water quality results to determine if there are any relationships between the sample mineralogy and the sulphide and NP reactivity. Review mineralogical data for the residual samples to determine if the NP reactivity could be hindered by the formation of secondary minerals.

- Carry out extraction tests on the test residues to quantify the stored oxidation load in samples that were not subjected to wash testing. Sequential extraction tests could provide additional information on the partitioning of metals in this system (see procedures in Tessier *et al.*, 1979). For example, secondary minerals would be solubilized in a deionized water extraction, weakly sorbed metals would be removed in a strong saline solution, metals associated with iron and manganese oxides would be removed in an organic acid solution (eg. hydroxl-amine hydrochloride in 25% acidic acid), and metals that are resistant to leaching would only be removed in a strong acid solution.

- Geochemical modelling to determine whether secondary minerals are limiting metal concentrations in this system, and how sensitive these limits are to changes in the pH and alkalinity of the system.

Another valuable contribution would be to develop testing and evaluation protocols to determine how much mixing would be required to control the production of acidic leachate in a full scale waste rock pile.

F.9 Key References

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APPENDIX G
WISMUT PROJECT

G.1 Overview

The Wismut project is located in the Ronneburg Uranium Mining District of former East Germany. Mining was carried out from 1950 to 1991, with a total uranium production of approximately 100,000 tonnes. The area affected by mining encompasses about 35 square km and includes an extensive network of underground workings, 3 open pits and numerous waste rock piles. The remediation project is one of the largest in the world, with total costs estimated to be in the US$ 7,000,000,000 range over a projected period of 15 to 20 years (Jakubick et al., 1997).

This review focused on the on-going consolidation and relocation of 65 million m^3 of waste rock from the Absetzerhalde (Absetzer dump) to the Lichtenberg pit. The dump consists primarily of black shales, limestone and diabase and was constructed using conveyor belts which placed rock in uniform 10 metre thick layers. Seepage from the waste rock pile has sulphate levels ranging from 7,000 to 25,000 mg/L SO_4^{2-}, pH's of 1.5 to 3.1, and uranium levels of 1 to 8 mg/L.

The disposal plan combined several mitigation technologies, including: in-pit disposal, materials segregation, quicklime addition (to neutralize the acidity load during relocation), 3 types of covers (water, oxygen consuming and a low permeability multi-layer surface cover) and collection and treatment of effluent. Figure G.1 shows the overall disposal strategy. Materials classified as ARD generating are located below the projected water table, materials with an uncertain potential for acid generation are placed in the middle layer, and acid consuming materials are placed in the upper layer. The surface cover will be designed to reduce water infiltration and to prevent convection, so that oxygen access into the underlying rock is limited by oxygen

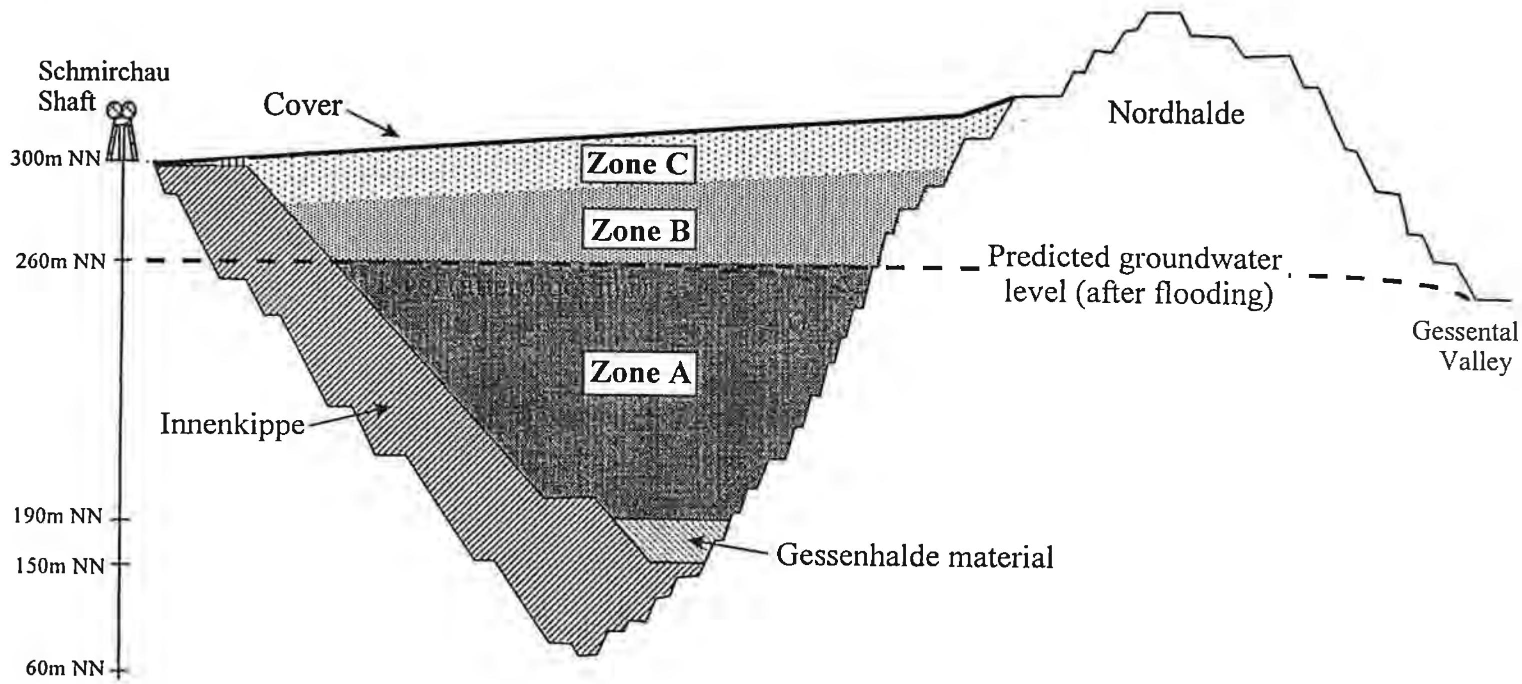

Figure G.1 Section Through the Lichtenberg Pit Showing Waste Rock Relocation Zone A,B and C (Hockley *et al.*, 1997).

diffusion rates. The soil cover will also provide a partial barrier to gas diffusion. As a backup to the soil cover, any oxygen entering the pile is expected to be consumed by the slow oxidation of sulphides in the upper layer of waste rock (C-Zone), therefore limiting further oxidation of sulphides in the marginally acid consuming layer. The concept of an oxygen-consuming layer as well as the field classification and materials handling aspects of this project are examined in this case study.

G.2 General Information

Location	Ronneburg Mining District, Germany
Mine owner	Federal Ministry of Economics, Wismut GmbH
Mine contact	Wismut GmbH
Economic metals	Uranium
Open pit or underground	Both
Ore production	3000 to 7000 tonnes/yr
Waste rock production	228 million m^3 in 64 rock piles, 65 million m^3 in the Absetzerhalde
Average annual precipitation, and/or range	630 mm
Average annual evapotranspiration	500 mm
Average annual temperature	8°C (Jan = -1°C, July = 16°C)
Other significant climatic information	Very little snowfall accumulation, most of the precipitation falls as rain.

G.3 Geology

G.3.1 Regional

The Ronneburg mining district is located in the "Gera Salient", a northeastern extension of the Berga Anticline in Eastern Germany. The host rocks are part of a marine sedimentary sequence comprised of carbonaceous shale, slate argillite, sandy argillite, graywacke and carbonate (limestone and dolomite), ranging in age from the Upper Ordovician to the Triassic. Upper Devonian diabase has intruded the upper portion of the sedimentary sequence, occurring as small stocks, sills or dykes. The major lithologies, their ages and distribution in the waste rock are presented in Table G.1.

G.3.2 Ore Deposition

The model for the uranium deposition indicates that uranium, associated with organic sediments in the upper part of the sequence, was re-mobilized during a period of tectonic activity in the early Permian. Uplifting and erosion of the rock allowed the oxidation front to move downward, resulting in a decrease in pH, an increase in the redox potential and, consequently, an increase in dissolved uranium concentrations. The uranium was then re-deposited and concentrated along fracture and joint surfaces as the fluids reached the reducing zone. It is thought that both pyrite and uranium were concentrated in this manner. Most of the uranium mineralization is associated with the Lower Graptolite Slates and associated Diabase Intrusions, the upper part of the Ordovician Leather Slates, and the Ochre Carbonates. The uranium occurs as pitchblende, usually as a precipitate coating primary pyrite or together with secondary pyrite or marcasite. The close association of these minerals is one of the factors leading to the mobility of uranium in these rocks.

Table G.1: Major Rock Types in the Ronneburg Mining District
(Steffan Robertson and Kirsten (Canada) Inc., 1995b)

Age	Rock Type	Description	% in Waste
Upper Devonian	Diabase intrusion	Variable composition and texture, located primarily in the Lower Graptolite Slates and Devonian Schists	20.5
Lower Devonian	Tentaculite limestone	Nodular limestone	10.5
	Calcareous tentaculitic argillite	Intermediate composition with local nodules of limestone	
	Tentaculite schists	Clayey shale to quartzitic argillite	
Silurian	Upper Graptolite Slate	Bituminous slate with alternating bands of carbonic aluminous slates, silicified slates, carbonaceous dolomite bands and carbonaceous sandstones. Average pyrite content of 7%.	0.5
	Ochre Carbonate Group	Massive to nodular dolomite and calcium limestone, local shale or slate bands	16.7
	Lower Graptolite Slate	Siliceous to aluminous black slate with average pyrite content of 5.5%. Permeable joint sets allow water to move preferentially through this unit	12.5
Upper Ordovician	Leather Slates	Range in composition from sandy clay and micaceous to phyllitic argillite to pyritic and carbonaceous sandy micaceous black slate. Weathers to a leather coloured brown. This unit is an important uranium host, with most of the uranium occurring along boundary with the overlying siliceous slates. Contains up to 35% carbonate	40.3
	Main Quartzite	Sandstone, quartzite and shale, transitional with the leather slates.	

G.3.3 Hydrogeology

The Lichtenberg Pit is currently within a cone of depression resulting from dewatering of the open pit and underground workings during mining. Controlled re-flooding of the mine will partially restore groundwater levels, allowing partial flooding of the pit. The underground workings are hydraulically connected to the pit, and will be an important influence on groundwater movement after the flooding is complete. Rocks in the open pit walls have a relatively low permeability, however the Lower Graptolite Slate located near the base of the pit, and three faults intersecting the pit are also expected to influence groundwater flow. Once the groundwater levels have been restored, it is expected that groundwater will flow north from the pit, ultimately discharging in the Gessental Valley.

G.4 Static Testing

G.4.1 Methodology

Three phases of static testing are relevent to this review. In the initial site classification work, the acid generation potential of each of the geological units was determined on the basis of mineralogical descriptions by assuming all of the carbonates were present as calcite and all of the sulphides as pyrite. For long range planning, acid base accounting analyses were conducted on drill core samples from the Absetzerhalde waste rock pile according to the modified Sobek method, with CO_2 determination of NP. On the basis of these analytical results and associated kinetic tests, rapid field testing methods were developed to refine classification of the waste rock during the operational stage.

G.4.2 Initial Rock Type Classification

The mineralogy based acid base accounting data indicated there are substantial variations both within and between the major rock types. While mineralogy based ABA values are not alway consistent with actual ABA analyses, average results for each rock type are provided in Table G.2 as a basis for comparing the waste rock to materials at other sites.

The Upper and Lower Graptolite slates are predominately acid generating, with NPR's of 0.3 and 0.9 respectively. The Diabase and Leather Slates are marginally net acid consuming, with NP/MPA ratios between 1 and 2. The Ochre Carbonates and the Tentaculite Limestone are predominantly acid consuming, with NP/MPA ratios of 9.7 and 4.6.

However, lithology (rock type) is not considered to be a reliable or practical indicator of the acid generation potential in the Absetzerhalde because of the substantial variability within each rock type, the difficulty in identifiying rock types in the weathered rock piles, and the impossibility of segregating randomly mixed materials in the waste rock pile by rock type. The Absetzerhalde was estimated to have an average NP/AP ratio of 1.5 (Hockley et al., 1997).

Table G.2: Calculated Acid Base Accounting Results Based on Mineral Content from the Catalogue of Rock Properties
(Dr. H. Szurowski *in* Steffan Robertson and Kirsten (Canada) Inc., 1995b)

Lithology	CO_2 %	S_T %	NP	MPA	NNP	NP/MPA
Main Quartzite	0.85	0.06	18	1.9	16	9.6
Leather Slates	2.5	1.0	53	31	22	1.7
Lower Graptolite Slate	0.9	2.0	19	63	-43	0.3
Ochre Carbonates	28.5	2.0	606	63	544	9.7
Upper Graptolite Slate	4.5	3.3	95	103	-7	0.9
Tentaculite Limestone	2.4	0.4	52	11	40	4.6
Diabase	8.2	4.8	174	150	24	1.2

G.4.3 Long Term and Operational Planning

The historical construction data was insuficient to characterize the waste rock distribution within the pile by acid generation/neutralizing potential and contaminant content. Therefore an extensive investigative program, including test pits and boreholes was caried out to obtain the necessary geochemical characterization data for long range relocation planning (Jakubick et al., 1997). Drilling was conducted on 200 m centres in the Absetzerhalde, and drill samples were tested and classified as “A”, “B” or “C” material according to the analytical ABA results using sulphide sulphur content. “A” materials have an NP/AP ratio of less than 1, “B” materials have an NP/AP ratio of 1 to 3, and “C” materials are net acid consuming with an NP/AP ratio of greater than 3.

The relative proportions of “A”, “B” and “C” material in the Absetzerhalde are shown in Table G.3. Although the kriging results were the basis for long term plans, a subsequent polygonal interpretation is appearing to be more accurate for the randomly mixed waste in the dump because it did not average out the extreme values (Hockley et al., 1997).

Mid to short range planning is based on the same material definitions as in the long range planning, however field tests are used to classify the material. The field tests included a paste pH test on uncrushed rock material and a modified net acid potential test (NAP) in which hydrogen peroxide is added to the rock and the pH of the resulting solution is measured. Operational classification is further discussed in Section G.7.

Table G.3: Percent Distribution of Material by ABA Class in the Absetzerhalde (Hockley et at, 1997)

Estimation Method	Class A NP/AP < 1	Class B 1 < NP/AP < 3	Class C NP/AP > 3
Polygonal interpretation of drilling results and historical records	36.2	19.3	10.3
Kriging of drillhole ABA data	31.1	26.1	1.0

G.5 Waste Disposal Plan

G.5.1 Conceptual Design

Waste rock from the Absetzerhalde is currently being relocated to the Lichtenberg pit where a combination of waste segregation, flooding, an oxygen consuming layer and a surface soil cover will reduce the rates of sulphide oxidation and contaminant migration from these wastes. A cross-section of the proposed waste rock pile is shown in Figure G.1. Key features of the design are described below:

- Zone "A" is below the projected water table. Rock that is already acidic or has the potential to generate acidic drainage is mixed with quicklime to prevent the short term release of acidity and stored oxidation products, and is moved to this zone of the pit. The water cover will reduce oxygen concentrations reaching the wastes and therefore significantly reduce further oxidation of the sulphides.

- Zone "B" is above the water table, but below the expected depth of oxygen penetration. Rock with an uncertain potential to generate acidic drainage will be placed in this zone.

- Zone "C" is in the upper portion of the pit, and will be capped with a low permeability, multi-layer soil cover. The cover will restrict convection of oxygen into the dumps, but will permit the slow diffusion of oxygen into Zone "C". Materials placed in this zone will be net acid consuming, however it is anticipated that sulphide oxidation will consume any oxygen diffusing into this layer.
- Collection of seepage from the backfilled pit for chemical treatment in a centralized treatment plant.

G.5.2 Evidence for Oxygen Consuming Layers

As discussed in the main body of this report, oxygen consuming layers are a relatively new concept in ARD control. Field data and model simulations based on site specific oxidation rates suggest that this approach is valid at the Wismut site (Steffan Robertson and Kirsten (Canada) Inc., 1995b).

The most relevant field evidence is from the Nordhalde, a large waste rock pile constructed approximately 30 years ago and located at the north end of the Lichtenberg pit. A cross-section of the dump showing the distribution of A, B and C materials is shown in Figure G.2.

In 1996, 8 boreholes were instaled in the pile and instrumented with oxygen and temperature monitoring equipment. The preliminary observations (Jakubick et al., 1997) indicated that the depth penetration of oxygen varied with barometric pressure fluctuations, compaction, reactivity of the waste rock materials and air convection. A soil cover provided only a partial barrier to oxygen convection. Oxygen penetrations increased with the thickness of the piles and decreased with increasing acid generation (hence oxygen consuming) characteristics of the waste rock. Oxygen measurements from the layered portion of the Nordhalde showed evidence of an oxygen gradient, with oxygen concentrations decreasing with depth. An example of the oxygen profile is shown in Figure G.2. Depletion typically occured within 20 metres of the surface. This suggested that a large portion of this dump had developed a thick acid consuming layer overlying predominantly acid generating rocks.

The oxygen gradient was also modelled in a series of simulations to test the sensitivity of oxygen depletion to the rate of diffusion, the rate of sulphide oxidation and various reaction sequences. Assuming a multi-layer surface cover that was sufficient to ensure diffusion controlled oxygen transport in the C-Zone, the simulations showed that oxygen penetration depths could vary from 5 to 20 metres in the Absetzerhalde waste rock.

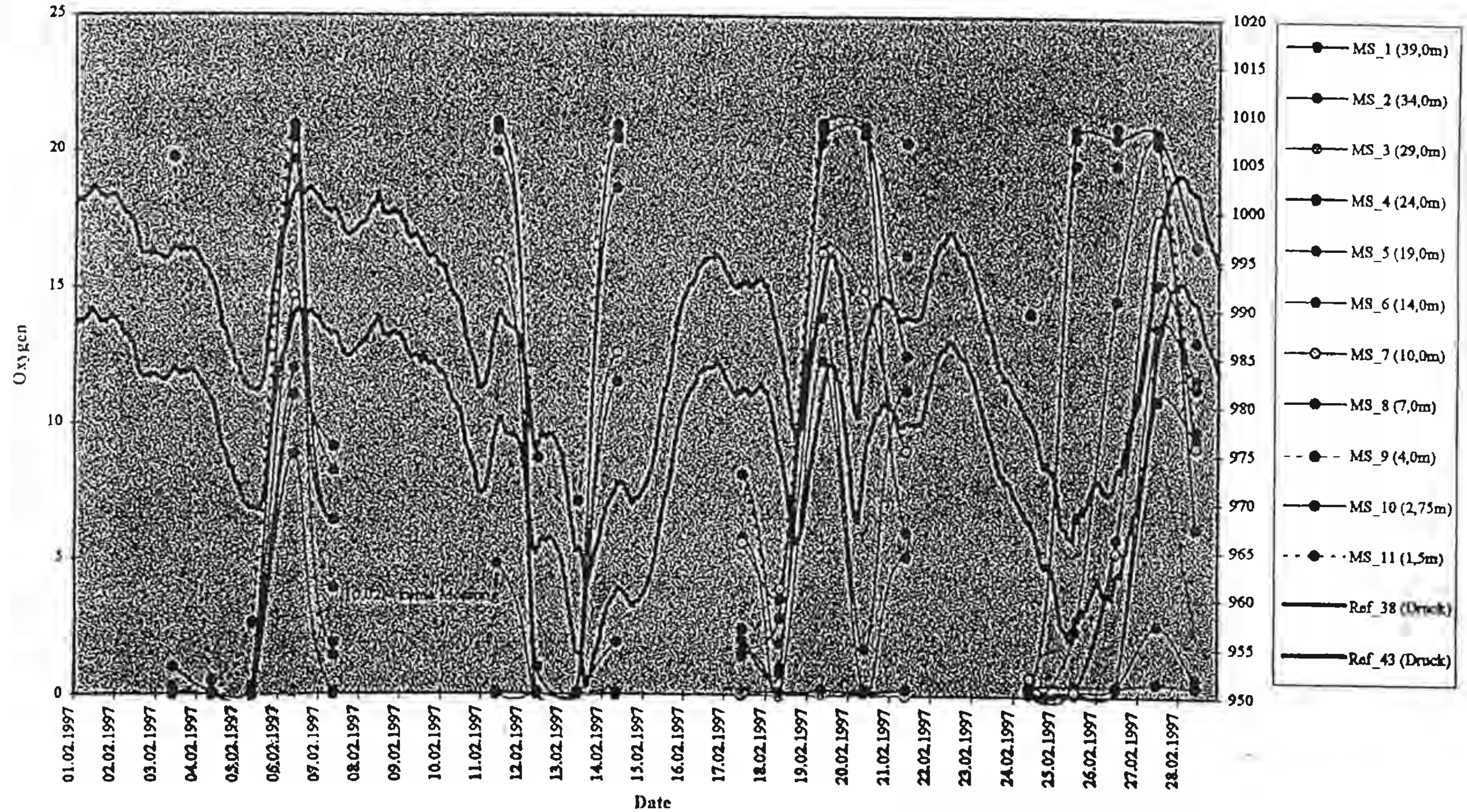

Figure G.2 Typical Oxygen Depletion Profile in the C-Zone Layer of the Nordhalde
(Jakubick *et al.*,1997)

Temperature distributions and gradients in the Nordhalde were used to estimate the rate of heat transfer out of the pile. This heat transfer reflected the total heat generated by chemicals processed in the pile, which was considered to be primarily due to sulphide oxidation in the Nordhalde (Jakubick *et al.*, 1997). Back calculations of the oxidation rate and net oxygen flux into the pile were made, which were then used to estimate the total oxygen consumption capacity. This was then translated into the design of the Lichtenberg Pit C-Zone (Jakubick *et al.,* 1997).

Design of the multi-layer surface cover on the backfilled pit is conceptual at this stage. Backfilling the pit will take approximately 8 years. During this period, monitoring and testwork are proposed which will influence surface cover design. The presence of the oxygen depleting C-Zone is intended to act as a backup to the surface cover, thereby reducing the reliance on the ability of the surface cover to control oxygen ingress, and potentially reducing surface cover costs.

G.5.3 Mining and Construction Methods

The Absetzerhalde waste rock pile is mined in 10 metre lifts, with excavation from the face of the dump. Materials are ripped at an angle of about 23°. Quicklime (CaO) is then applied at the crest of the lift, with quantities determined from field conductivity tests (Section G.7). The lime is mixed into the waste rock when pushed down the face by a bulldozer. Further mixing is undertaken by the bulldozer ripper blades. The mixed material is then loaded by front end loader into the haul trucks and delivered to the appropriate location in the pit. There are two fleets of haul trucks, one consisting of about eight to ten 53 tonne trucks, the other consisting of about eight to ten 136 tonne trucks. This equipment is capable of moving about 100,000 to 120,000 tonnes of waste rock/day. The waste rock and lime mixture is placed in the pit by end-dumping, spread by a bulldozer into 60 to 120 centimetre lifts, and compacted by haul trucks. As a result, field hydraulic conductivities of 10^{-7} m/sec are being achieved in the backfill, and future settlement is expected to be reduced to approximately 15% (Jaukubick *et al.*, 1997).

G.6 Operational Planning and Control

The operational planning and control methods are outlined in Table G.4. Mine planning is done within the framework of a mining software package (ENTEC) which allows interactive planning and mine scheduling. The long term plans are based on ABA data from the drilling programs. One of the main objectives of the long term plan was to organize mine logistics to prevent double handling of the materials. The size of the pit, and the presence of an old dump (the Schmirchauer Balkon) will allow some simultaneous placement of different material types in the three zones.

The mid to short term planning is based on a more detailed sampling and testing program carried out 1 to 3 months prior to mining. Samples are collected from a 25 x 25 m grid of test pits. The test pit material is quartered, and subsamples are collected from each quarter, and from a composite of all four quarters. Materials are classified as "A", "B" or "C" on the basis of field paste pH and NAP tests (Coastech, 1990; Miller *et al.*, 1990) according to the protocol shown in Figure G.3. Field conductivity readings are used to determine the amount of quick lime needed to neutralize acidity during the relocation process. The 25 metre grid is based on a variogram of the field data showing that materials are relatively homogeneous within 50 to 100 metres. Waste rock characterizations based on these testpit samples are considerably more accurate than the initial estimates based on kriging from the 200 m drill hole grids. Future reconcilation of results from the current long term planning method and the more detailed operational monitoring may lead to modification of long term planning procedures (Steffan Robertson and Kirsten (Canada) Inc., 1995a, b).

During the actual mining, surveying control and face samples are collected to confirm the classification of material in a mine block. Material classifications are indentified by a system of flags specifying the ARD potential of each material.

Computer databases will be used to record all information pertaining to each specific block of mined material, including final location in the backfilled pit.

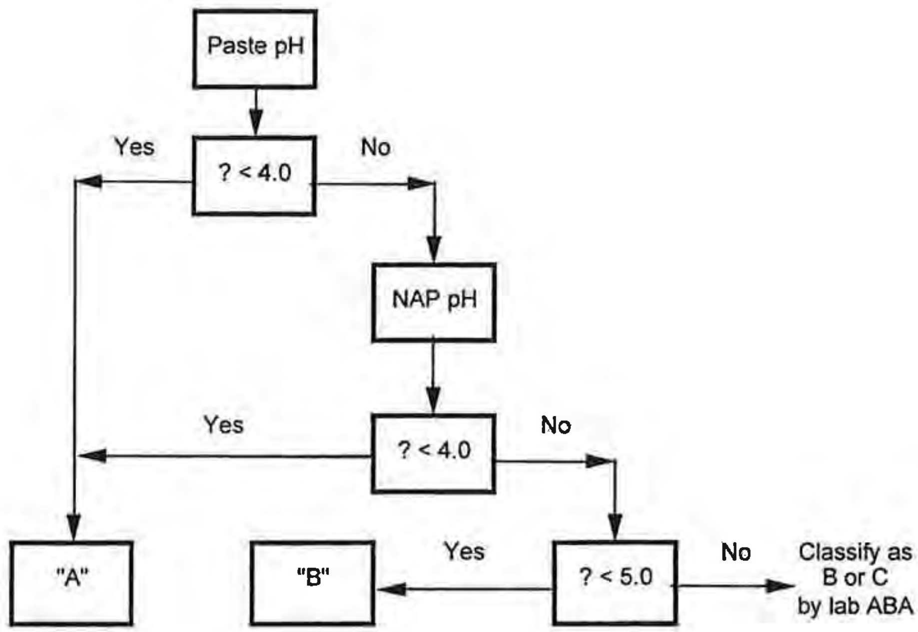

Figure G.3 Protocol for Using Paste pH and NAP pH to Classify Absetzerhalde Waste Rock

(Hockley *et al.*, 1997).

TABLE G.4: Program Overview: Planning, Directing and Quality Control of the Relocation
(Steffan Robertson and Kirsten (Canada) Inc., 1997)

Objectives	Data Base	Sample Classification	Interpretation und Classification of Halden Segments, Controls
Long and Medium Term Planning Volume estimates, basic decision, basis for planning in permit applications	Drill holes	A: NP/ AP < 1 B: 1 < NP/AP < 3 C: NP/AP > 3	Basic unit: blocks Interpretation method: Kriging or by hand + blocks 1st Priority: Segments A are to be separated from Segments C. For mixed areas apply: Zone A: >80% A-Blocks Zone C: >80% C-Blocks and <10% A-Blocks
Short Term Planning Determine destination site (Directing Remining), Directing lime addition	Test pits at 25m spacing. Minimum depth: 2 m, in case of surface contamination by lime or cover material 2.5 m (sampling intervall 0.5 - 2.5 m)	A: Paste-pH<4 or NAP-pH<4 B: Paste-pH>4 and 4<NAP-pH<5 or NAP-Cond.>5 C: Paste-pH>4 and NAP-pH>5 and NAP-Cond.<5	Basic unit: samples Interpretation method: Hand or local Kriging 1st. Priority : as above For mixed areas apply : Zone A: >70-80% A- samples Zone C: >80% C-samples and <10% A-samples
Quality Control of C-Material Assurance of the parameters of material that is placed in the C-Zone	Face Samples. 1 Composite sample (3pts)/50 m face length , sampling daily and offset by 25 m every day.	As short term planning	Controls: Maps, Tables, Diagrams
Quality Control C-Zone Open Pit	1 Composite sample, comprising 5 sample points per day and placement site	As short term planning	Controls: Maps, Tables, Diagrams
Quality Control B-Zone Open Pit Checking of parameters of material that is placed in the B-Zone	5 open pit point samples per day and placement site	As short term planning	Controls: Maps, Tables, Diagrams
Quality Control A-Zone Open Pit Checking of parameters of material that is placed in the A-Zone, control of lime addition	1 Open pit composite sample, comprising 5 point samples per day and placement site	Paste-Cond.<6	Controls: Maps, Tables, Diagrams
Quality Control of Direction (Steuerungsprogramm) Checking overall program, correlation with column tests	Open pit samples 10 samples per placement zone and year	As short term planning, additional chemical analysis (ABA) and modified column tests	(Semi)annual Report

G.7 Costs

Specific costs for the Absetzerhalde relocation project were not available. Estimated costs for the field control and monitoring aspects of the project are expected to be between 1 and 2 million US$. Costs in Germany are anticipated to be much higher as compared to North America.

G.8 Conclusions

- Oxygen monitoring in an existing dump, the Nordhalde, has shown strong evidence that a layer of non-acid generating but sulphidic rock can reduce oxygen transport to underlying layers of acid generating rock. Finite difference modelling of oxygen transport also supports this hypothesis. However, the oxygen consuming 'C-Zone' layer is still considered a conceptual control method and requires verification in a large scale dump. Therefore the C-Zone layer at the Lichtenberg Pit is being implemented as a back-up for the low-permeability soil cover. The multi-layer soil cover will be designed to reduce infiltration and eliminate oxygen convection. The C-Zone layer is expected to consume any excess oxygen that diffuses through the surface cover, reducing reliance on the surface cover and potentially reducing surface cover costs. The actual relocation of the wastes requires extensive field monitoring and control procedures. The procedures are costly, but are important to ensure that materials are placed in the appropriate zone of the pit.

- Initial classification of materials based on analyses of drill core samples can vary substantially from classifications based on more detailed test pit sampling and analysis. While not unexpected since the waste rock being mined is already somewhat mixed, the need to verify spatial classifications of waste rock through field checks and statistical assessments is emphasized.

- This project demonstrates that it is possible to characterize and classify materials in a timely and efficient manner to allow relocation of waste rock at rates that are equivalent to a full-scale mining operation (100 - 120,000 tonnes/day). Even at these large scales, characterization and classification can occur on a schedule that facilitates long term planning, and provides detailed operational control resulting in good material segregation. Excellent software database techniques are available to document the composition of the mined and relocated materials.

G.9 Possible Research Areas

English translations of the Wismut reports would make the enormous wealth of information on static and kinetic testing, field testing and field monitoring data at this site more accessible to the Canadian mining community. Focussed papers on the numerous aspects of the project would also be beneficial, such as the four papers presented at the Fourth International Conference on Acid Rock Drainage (Hockley *et al.*, 1997; Jakubick *et al.*, 1997, Pollmer and Eckart, 1997; and Kistinger, 1997).

In particular, detailed presentation of the statistical data from the different levels of sampling (drillcore, testpit and excavation face) would provide valuable information on the appropriate methods to interpret sample data, and provide valuable insight into determining how many samples are needed to adequately characterize the waste materials.

Verification of the geochemical mechanisms and interactions between layers is required. It is suggested that other well documented sites with C-Zone layers analogues be studied in more detail. This would include instrumentation of the surface soil cover, the C-Zone, and the underlying layers to determine oxygen transport and consumption rates. Long term monitoring of seepage from these dumps and the groundwater would also be required.

G.10 Key References

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