GENERIC DECISION-MAKING ON THE RETREATMENT OF COPPER TAILINGS DAMS

by

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Abstract

Tailings dam retreatment can be considered as a profitable alternative when tailings contain relatively high metal contents. Moreover, because of low exploration, mining, processing and closure costs, and the recent rise in copper prices, retreatment may generate positive cash flow. Meanwhile revenue from metal extraction can offset rehabilitation cost leading to an improved long-term liability performance.

A systematic approach and an Excel spreadsheet model are developed to evaluate economic profits, environmental and social benefits derived from retreatment, and to simulate human decision-making processes. Fuzzy logic and fuzzy-neural equations were employed in the model to deal with imperious and linguistic inputs, and to simulate human's decision-making process. By user questionnaire and surveys, the model receives inputs data, which will be processed through four different modules to generate final outputs. Economic and Design Module completes conceptual processing flowsheet design, cost estimate, and economic analysis of base case and retreatment project. A Disposal and Reclamation Module is designed to select re-processed tailings disposal and reclamation methods, estimate cost and evaluate the environmental performance. A Risk Assessment Module is aimed at environmental and social risks evaluation of current tailings site. After receiving all the criteria on economic, environmental and social performance improvement, a Decision-Making Module is developed to provide user a recommendation. Generally speaking, this model is able to conceptually design activities from mining to reclamation; evaluate economic, environmental and social benefits; assist multi-criteria decision-making.

Four hypothetical cases with different conditions have been processed to validate and verify the retreatment model. As indicated by model running results, the tailing retreatment module is an effective tool to assist in decision making. In addition, the system is working properly and efficiently on given inputs over a range and combination of values.

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List of Acronyms

AMD	Acid Mine Drainage
ARD	Acid Rock Drainage
ATPF	Agglomerated Tailings Paste Fill
A.S. Cu	Acid Soluble Copper
CA	Consequence Assessment
DCFROR	Discounted Cash Flow Rate of Return
DoB	Degree of Belief
DSTP	Deep Sea Tailings Placement
EA	Exposure Assessment
FAM	Fuzzy Associative Memory
FMEA	Failure Mode and Effect Analysis
HA	Hazard Assessment
M	Million
NGO	Non-Government Organizations
NPV	Net Present Value
n_{100}	Year of Certain Failure
ORG	Organics
RC	Risk Characterization
PDF	Probability Distribution Function
PV	Present Value
P_5	Failure Probability within First 5 Years
STD	Submarine Tailings Disposal
SX/EW	Solvent Extraction and Electrowinning
TSF	Tailings Storage Facilities
VB	Visual Basic
WTP	Water Treatment Plant

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Chapter 1. Introduction

1.1. Background

1.1.1. Tailings and Tailings Storage Facilities

After extraction of valuable minerals, mining operations leave a tremendous amount of waste that may exist as a legacy at the site requiring considerable long-term attention. Copper ores, as an example, typically contain 0.5% to 2% copper as minerals and produce a concentrate grade varying from 25% to 45 % Cu, meaning that from 93% to 99% of these ores are rejected as tailings. There are two major types of mine waste: waste rock and tailings. The present study examines tailings, a waste material resulting from the grinding and mineral extraction processes, which are disposed of in slurry form, typically to a tailings dam (Stuckert, 1982); (Vick, 1983).

Tailings are comprised of ground rock particles, water, and reagents from the processing operation. The size distribution of tailings depends on the processing methods used, commonly ranging from clay- to sand-sized materials. The chemistry features are determined primarily by ore mineralogy, processing methods, and treatment prior to disposal such as neutralization. Generally speaking, four types of tailings exist: soft-rock tailings, hard-rock tailings, fine tailings, and coarse tailings (Vick, 1983). This thesis focuses on copper tailings, a hard-rock tailings.

After valuable minerals are extracted, leftover tailings are normally disposed of in a tailings storage facility (TSF) designed and constructed to provide safe and economic storage while minimizing environmental impacts. Given the variety of site topographies, ore mineralogy, process technologies, and expected future use of a mine site, a TSF must be carefully designed, planned, and maintained. There are a variety of designs and construction technologies to safely deal with tailings such as surface impoundments, underground backfilling, thickened discharge, and submarine disposal. However, not all tailings have been stored responsibly in the past. Tailings that are discharged randomly such as in an artisanal operation or directly into the nearest watercourse – a lake or a river – can poison downstream waters and the nearby environment, threatening the health of nearby residents as well as biota that inhabit that ecosystem.

Prior to the 1960s, there was less attention given to disposal and management of tailings compared to the extraction of valuable minerals. As such, there are mine sites existent today that have old tailings dams that are dangerous, unstable, or ones that might cause significant environmental pollution. When a mine site is taken over or inherited by a mining company, it is extremely important to conduct due diligence to ensure that the new owner is not taking on unidentified or unknown liabilities. Old tailings dams require considerable examination since the long-term liability may increase over time with respect to safety and the environment.

Recent public opinion has turned against not only mining companies, but also the mining industry in general due to some poor tailings disposal practices. The bad reputation and the close monitoring by local communities, aboriginals, the media, and Non-Government Organizations (NGO) currently stand as obstacles to mining industry development. Additionally, due to the increasing pressure from local and global politics, as well as regulatory requirements, mining companies must now manage their TSF more responsibly than in the past.

Abandoned/orphaned mines are susceptible to the same environmental and social problems mentioned above, and have also caught the attentions of the public and government. Abandoned or orphaned mines refer to sites where the owner cannot be found, or for which the owner is unwilling, or has lost the financial ability, to carry out rehabilitation duties (Manitoba, 2011;

NOAMI, 2011). Mine sites of this kind exist all around the world. Having been developed and subsequently closed decades ago, and operating without an adequate understanding of environmental impacts and today's modern operating standards, many abandoned mines threaten the environment, residents' health, and the economic conditions of local communities, the mining industry, and government. Another important obstacle is the shortage of funding from previous owners, so normally tax payers must pay for the rehabilitation work. For example, for the rehabilitation work of Lynn Lake, Snow Lake, Sherridon, and ten other high-hazard sites during 2009-2010, $42 million was collected and spend by the provincial government (Manitoba, 2011). In B.C. the Britannia Mine at Howe Sound as a similar example (Meech et al. 2003).

Over the past three decades, thanks to the growing public and technical awareness of detrimental mining impacts, people are more concerned about the negative influences of tailings disposal practices. Advanced technologies and responsible management to mitigate the problems of tailings, and at the same time improve the reputation of the mining industry are required.

1.1.2. Tailings Retreatment

Tailings retreatment by re-mining, re-processing and re-disposal can be an alternative for dealing with the environmental and social issues mentioned above. These have the potential to generate considerable profit. The idea is to mine and re-process old tailings dams to extract copper minerals, and to use the revenue from metal production to cover the cost of reclaiming these dams for long-term stability, and to minimize the environmental impact caused by pollution from these dams.

Retreatment of old tailings dams has been shown to be an effective method to deal with tailings especially when one considers the significant improvements in modern processing technology and the increase in metal prices. Some tailings can be regarded as a potential ore body with considerable metal value. For example, Amerigo Resources is processing tailings from Codelco's El Teniente mine in Chile, the largest underground copper mine in the world (L. Williams, 2009). At the Woodlawn mine in Australia, TriAusMin plans to re-treat 10 million tonnes of tailings from three tailings dams (TriAusMin, 2010).

There are several reasons that retreatment should be examined as a rehabilitation alternative. A tailings dam can be a potential ore body. The limitations of past mineral processing technology means that considerable amounts of valuable minerals were often lost to the tailings. As a result, the grade of some tailings, especially older ones, presents an attractive prospect. Carolin Mine, a past producer of gold and silver in British Columbia, is an ideal case. In 1996, drilling tests and assay results validated proven reserves of almost 800,000 tonnes with a gold grade of 1.74 grams per tonne (BCGS, 2008). Due to this high grade and the current increased gold price, this abandoned mine is drawing significant attention. (Daniel & Downing, 2011).

Furthermore, the rising copper price is encouraged companies to consider retreatment, enabling them to turn a waste into a value. Figure 1 shows copper prices since 1930. The data indicate that the copper price has risen to a high level in the last decade.

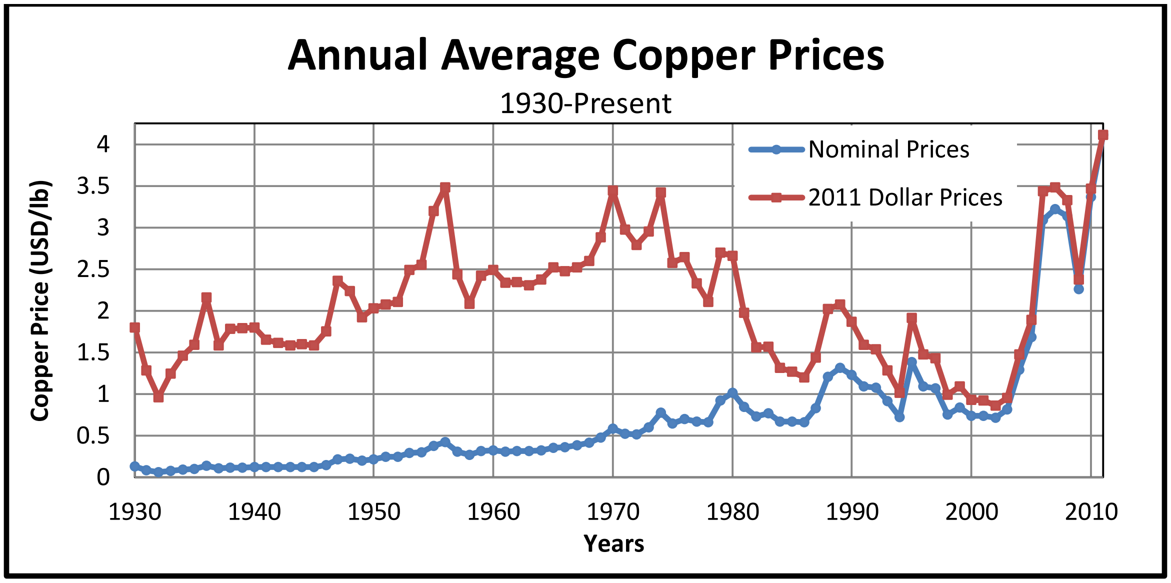


Figure 1. Historical Annual Average Copper Prices.

Sources: (Edelstein, 1998), (Bank of Canada, 2012) and (LME, 2012)

It is notable that reduced exploration, mining, processing and closure cost as compared to a regular mining project make retreatment a simpler and possibly more economic project. The mining of tailings does not require creating a new underground mine or open pit with the associated blasting and material transportation systems. Instead hydraulic mining with high pressure water hoses and simple ditches or launders can be used to transport the "ore" to the new processing facility. Significant capital and operation costs savings will ensue. In addition, with an already finely-ground constitution, the energy costs of processing tailings will be significantly lower due to a reduction in grinding requirements. Retreatment may also present an opportunity to cut down long-term mine closure costs by eliminating the need for a water treatment plant (WTP) for effluent currently leaving the site after passing through the existing dam.

As a reclamation strategy, retreatment is not only aimed at recovering valuable minerals. It is also an opportunity to reduce the liability of storing tailings on surface in-perpetuity. By generating revenue from metal recovery, the cost of subsequent tailings treatment, restoration, and final reclamation can be met or significantly reduced. Moreover, with the more modern tailings storage technology being used today, a safe disposal situation can be gained. Retreatment can also provide an opportunity to address the environmental risk associated with a tailings impoundment. For example, desulphurization treatment can be applied to tailings to eliminate or reduce ARD emissions. Tailings can be placed underground, or deleterious materials can be segregated, or the entire tailings can be placed under water to reduce acid generation and dust problems. Once the physical and chemical stability is improved, health and safety risks are effectively reduced in turn improving the lives of nearby residents. There is also no doubt that retreatment brings significant economic benefit to local communities by creating job opportunities and motivating mining-related business.

In summary, the retreatment of old tailings dams, as an alternative reclamation method, provides an attractive opportunity to achieve considerable economic benefits, and at the same time, improve long-term liability performance.

1.2. Research Focus and Aim

The ever-growing demand for responsible tailings management places retreatment as an economically and environmentally-friendly alternative to address problems associated with existing old tailings dams. It is important to study potential tailings retreatment projects in order to assist in future decision-making. The research concerning the decision-making methods on retreatment of copper tailings dams is attempting to evaluate the economic, environmental and social improvements that can be achieved.

In order to realistically simulate the decision-making process, this study seeks to determine the general concerns that people have when choosing whether to commence an engineering project, and how those concerns can be satisfied. The study focuses on retreatment design, economic benefit estimation, and the evaluation of environmental and social performance before and after retreatment using a multi-criteria decision-making approach.

Economic benefit is always a primary concern for implementing an engineering project. So, the first step is to develop a general framework for the conceptual design of re-mining, re-processing, re-disposing and reclamation of copper tailings according to the properties of the dam, the tailings material, and the site. The next step is to estimate revenue, capital and operating costs, Net Present Value (NPV) and Discounted Cash Flow Rate of Return (DCFROR) as parts of an economic analysis.

Unlike normal mining projects, a major target of tailings retreatment is to mitigate existing negative environmental and social impacts. As such, the evaluation of mitigation performance becomes important in the decision-making process. Part of the study focuses on evaluating the environmental and social performance changes which may be more significant criteria than economic issues. Conclusions will be drawn in regard to a final decision for implementation of a project based on a combination of the above criteria.

The overall aim of this research is to develop a generic approach to alternative tailings treatment methods through which valuable metal can be recovered, to address the problems associated with conventional tailings disposal practices, and to assist in the decision-making process.

Specifically, developing a generic decision-making approach is the objectives of this research according to:

- A complete conceptual processing design based on tailings properties;
- Selection of disposal and reclamation options for reprocessed tailings;

- Estimating capital and operating costs and conducting a preliminary analysis based on process design flowcharts, and selection of disposal and reclamation methods;
- Evaluating environmental and social risks of the current tailings site and that of the future ones after retreatment;
- Recommending one method based on economic, environmental, and social performance improvements;
- Developing a model and tool to collect information to assist in decision-making.

1.3. Value of this Research

This research will contribute to the decision-making process on tailings retreatment through the simulation of a realistic decision-making process. The tailings retreatment model produced by this study can be used by industry in a number of important ways including:

- Classification of potential valuable resources from waste owned by a mining companies;
- Evaluation of rehabilitation alternatives (retreatment) to reduce long-term liability;
- Assistance in decision-making at the preliminary stage of a feasibility study;
- Investigation and assessment of an existing tailings dam, and identification of risks and values associated with a tailings dam;
- Ranking tailings dams according to estimated risk and value.

Chapter 2 Literature Review

2.1. Background

The overall aim of this research is to develop a general method to investigate tailings retreatment methods through which valuable metal can be recovered and/or the problems associated with previous tailings disposal practices can be addressed. The idea is to re-mine and re-process old copper tailings dams for economic benefit and to use the revenue from metal production to offset the cost of dam rebuilding and/or rehabilitation. To prepare basic knowledge for this study, a literature review has been conducted to cover topics related to tailings retreatment: tailings disposal methods, environment and social issues of tailings disposal, mining and processing of tailings dams focusing on copper recovery, and reclamation methods.

2.2. Tailings Disposal Methods

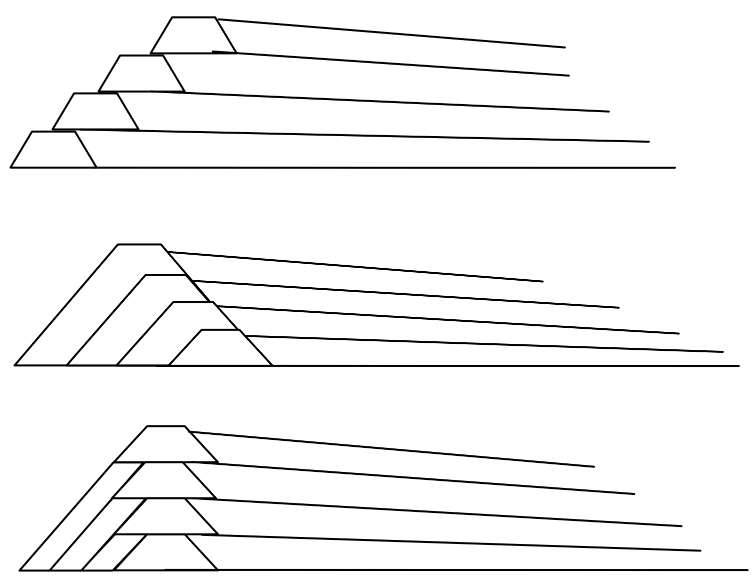
Tailings can be disposed of using a variety of methods. Selection of a disposal method is a complex issue which depends on local geography and hydrology as well as the nature of the tailings.

2.2.1. Surface Disposal

Surface impoundment of tailings is the most widely used disposal method around the world. Embankments for surface disposal can be classified into two types: water-retention and raised embankments. Water-retention tailings dams are essentially the same as conventional water storage structures. Compared with the raised embankment type, the most notable difference is that water-retention type dams are constructed to their full height before they are put into use

(Vick, 1983). While this type of construction is the best one from a long-term safety viewpoint, it is by far the most costly and is rarely used as a tailings storage facility.

To reduce and delay costs, a raised embankment is staged and its construction is in progress throughout the life of the mine. A raised embankment begins with a starter dike. When the tailings approach the height of that dike, a second is built above the first. This process continues with new lifts being placed as required. A wide range of materials such as hydraulically-deposited tailings or cycloned coarse tailings can be used to build each lift. Compared to the water-retention type, there are significant advantages of raised dams. The first includes a much lower cost with major cash-flow benefits. However, there is more responsibility to attend these structures during the mining operation, and more planning effort and attention to scheduling and monitoring are required. According to the direction that the embankment crest moves in relation to its starter dike position, a raised tailings dam falls into one of three categories as shown in Figure 2: upstream, downstream or centerline (Mular, Halbe, & Barratt, 2002).



a. Upstream Method

b. Downstream Method

c. Centreline Method

Figure 2. Three Basic Methods to Construct a Raised-Embankment Tailings Storage Facility.

The construction method has a significant influence on the performance of the embankment. The upstream method has been popular in the past because of the advantages of lower cost, simplicity, and minimal volume of mechanically-placed fill for each dike. Basically the amount of material required for each dike is the same. However, there is generally a limit on the ultimate height of the embankment and the raising rate must be kept low to ensure the foundation (tailings) is consolidated. Stability issues can result if the material is under-consolidated with high water content. There is a high susceptibility to liquefaction which may preclude its use in earthquake-prone regions. Vibrations from mining activities or an earthquake can induce liquefaction which is a major cause of upstream dam failures (Stuckert, 1982).

The downstream method on the other hand, can be constructed with an internal drain and an impervious core of clay material within the dam to control the entrance of the phreatic surface. The phreatic surface defines the position between the zone of saturation (water) and the zone of aeration (Tailings. Info, 2010) which is critical for a tailings-dam's stability. The downstream method is often selected due to its increased stability and safety, larger water storage capability, and relatively unrestricted rise rate. However, downstream dams are much more costly requiring an ever-increasing volume of embankment fill for each subsequent dike. There must also be sufficient space for the toe of the dam to expand away from the structure.

The centerline method is a compromise between upstream and downstream methods. It mitigates the disadvantages and, at the same time it shares the advantages of the two methods. Like the downstream method, an internal drainage zone can be built resulting in a lower phreatic surface. This method is also seismic-resistant. In terms of liquefaction failures, centerline dams can be considered stable.

2.2.2. Underground or In-ground Disposal

In addition to surface disposal methods, underground disposal is also in wide-use. The main aim of returning tailings back into the mine is to support the mining operation underground and/or to maximize ore recovery rather than depositing tailings to reduce environmental impact. Until recently, storage in an open pit or underground work has been uncommon, but that appears to be changing. Advantages of this disposal method include reduced land requirement and disturbance of the natural environment (Vick, 1983).

Copper and gold mines with underground operation may produce a large underground space after mining. Because after comminution processes, the volume of tailings becomes larger than their origin volume, normally there is insufficient room to store all the waste. Conventional cemented hydraulic fill is manufacture by cycloning the tailings to recover the coarse fraction which is then combined with cement and pumped underground to continue the mining procedure into new adjacent open stopes. The fines may still require conventional surface disposal. However, with advent of high-density paste backfill and agglomerated tailings paste (ATPF), all tailings can be stored in underground work or open pit space. Paste backfill uses unclassified tailings (with or without binder depending on fill strength requirements) at a pulp density of 72%-85% solids. This may preclude the need for surface disposal. But this method requires higher capital cost over conventional fill systems due to additional dewatering facilities and greater technical precision. ATPF is a new technology similar to paste fill but with supplementary aggregate additions to increase the compressive strength.

If the tailings contains problematic pyrite which may lead to a high acid-generating potential, a possible option would be to dispose of those tailings under water. In a saturated condition, oxidation is virtually eliminated and so too the acid drainage and heavy metal pollution. For copper and gold mine tailings, open-pit disposal may be cost-effective if the pit or pits can be sequenced. Storing tailings in a closed open-pit is generally simpler and is both economically and environmentally beneficial due to the creation of a smaller surface impoundment. Care must be taken that high pressure build-up at the bottom of the pit does not lead to possible sudden movement of the material into tunnels or openings near the bottom of the pit. This actually happened at MarCopper in the Philippines with tailings flowing out for over 6 months causing terrible pollution and hazard to the environment and local communities (Tauli-Corpuz, 2012).

2.2.3. Sub-Marine Disposal

Besides being stored on land, tailings can be directly discharged into nearby natural bodies of water, such as the ocean, a lake, or a river. Sub-marine disposal (STD) or deep sea tailings placement (DSTP) is an significant underwater tailings placement alternative which has been used around the world at several mines in British Columbia, the Philippines and Central American (Vick, 1983). The application of STD/DSTP is limited to mines located reasonably close (within 200 km) to a marine coast with deep water (>100m) or to alpine lakes with low aquatic life. Riverine disposal is perhaps the most controversial of the three options. It is practiced in Indonesia and results in the destruction of waterways that may run for thousands of kilometres. The alternative however, involves surface impoundment of highly erosive materials in high-rainfall regions. Hence, there is often little option except to decide not to mine at all. In all cases, the tailings should be proven to be relatively inert and not toxic to the receiving environment (Poling, 2002).

For mines near the coast with deep water close to the shore, STD/DSTP can be evaluated as an alternative due to minimized land use, lower capital and operating costs, and easier closure systems when compared to other on-land disposal strategies. Unlike on-land impoundments, tailings deposited on the sea floor are generally stable and not under the risk of structural failure. STD/DSTP can be considered as a particularly useful alternative to prevent Acid Rock Drainage (ARD) resulting from oxidation of sulphide-rich tailings by isolating tailings solids from oxygen.

At present, it is still a challenge to apply STD/DSTP into practice mainly because of technical, environmental, and regulatory concern. Even when the underwater facility is carefully engineered, a negative impact on marine, physical, and biological ecosystem can still exist. Higher environmental baseline and monitoring costs of STD/DSTP generally result in it being a less preferable choice than on-land tailings disposal methods. Legislation is another difficulty for STD/DSTP practices. This technology is banned by laws in many countries, such as the United States (MiningWatch, 2002). Intensive attention and counterviews from the public and local communities make it difficult to put this technology into practice.

2.2.4. Other Methods

There are a number of other successful methods in use. Thickened discharge is used at the Kidd-Creek mine in Timmins, Ontario in which tailings are dewatered and disposed of as a high pulp density slurry (~60% solids) to eliminate the need for an embankment easing construction, reducing costs, failure risk, and seepage (Blowes et al. 1995). Tailings can be further dewatered by filtration and stacking, which is called dry disposal or dry stack as being done at La Coipa Mine in Chile (Davies and Rice 2001). This TSF can reduce environmental impact and failure.

Unfortunately not all tailings are stored in a responsible manner – for example riverine disposal is still being used. Tailings discharged into rivers or viable lakes can cause serious water quality damage, turbidity, and harm to aquatic organisms. As a result, application of this kind of disposal method must be eliminated or significantly limited.

2.3. Environmental Problems

2.3.1. Physical Stability

Tailings dams may have a risk of failure during production, and for decades and perhaps hundreds of years after closure, since the finely-ground material may remain unconsolidated, loose, and semi-fluid for a long time. Moreover, tailings dams are exposed to internal and external erosion over time leading to serious stability problems. On average, globally, one or two failures of tailings dams have happened each year since 1960 with over 100 of these failures being disastrous or catastrophic resulting in human death (Wise-Uranium, 2011).

Earth embankment failure can cause catastrophic consequences due to the huge volume of released solids and water. What is worse, the hazardous constituents contained in the tailings may then be released into the environment. These constituents include toxic chemicals, heavy metals and sulphide minerals with acid generation potential. For example, on October 4, 2010, at Kolontár, Hungary, the embankment failure of a "red mud" (waste product of bauxite refining) reservoir released 600-700 thousand cubic meters of a mixture of mud and water, which flooded the 800 hectares of the lower area. Ten people were killed and 120 people were injured in this catastrophe (Wise-Uranium 2010). Since red mud is a kind of highly caustic waste, the environmental impact and the economic loss are virtually incalculable.

The major cause of accidents and failures of tailings dams is inadequate water management and hydraulic design. Some natural events, such as heavy rain or an earthquake can trigger a dam failure. Hydraulic failure in the form of overtopping by flood discharge accounts for nearly 40% of all the earth dam disasters in the world (Estergaard, 1999). Tailings dam failures have been classified into different modes by Wise-Uranium (2004):

Weak foundation: Weak foundation means the soil or rock layer below the dam is not strong enough to support the dam. This hazard can lead to partial or complete dam failure.

Excessive water level rise: A high water level can result from heavy precipitation events or inappropriate water management. As a stability criterion of tailings dams, if the phreatic surface within the dams rises, the toe of the dam may become unstable and collapse. Sometimes a rising water level may result in water overtopping the dam crest which can lead to erosion of the embankment resulting in failure.

Internal and external erosion: External erosion is a consequence of wave attack on the upstream of the dam, rain wash on the downstream side, or wind erosion on the downstream. The major cause of internal erosion is seepage within or under the embankment causing erosion along the flow path which can contribute to the failure.

Seismic events: Earthquake or vibrations from mining operations, such as nearby heavy equipment movement and mine blasting can lead to liquefaction. Tailings slurry can liquify to be released in a slurry wave to the downstream area, causing catastrophic devastation.

Excessive dam rising rate: Excessively rapid rising rate may result in excessive pore pressure within the dam which causes dam failure.

In the last three decades, because of increasing public awareness and stricter regulations, more careful design and more responsible management and monitoring are taking place but that does not seem to have affected the rate of major failures. A guideline for the design of dams in Canada was developed by the Canadian Dam Association (CDA). "A Guide to the Management of tailings facilities" has been produced by the Mining Association of Canada (MAC) in 1998 (Estergaard, 1999).

2.3.2. Chemical Stability

The varying properties and components of tailings mean that most traditional tailings dams are at risk of creating environmental contamination. Tailings are composed of gangue minerals, unrecovered or non-valuable metals, process chemicals and water, all of which may cause pollution. For example, the sulphide minerals in the tailings are a potential source of Acid Rock Drainage/Acid Mine Drainage (ARD) / (AMD), which produces low pH seepage (Shaw 1996). Metals such as copper left in the tailings stream may be leached out and cause heavy metal pollution. Processed water left in the tailings may contain various chemical additives such as sulphuric acid, cyanide, organic chemicals and other processing chemicals (Lottermoser, 2010).

ARD is a term referring to acid seepage (pH less than 5) generated from waste rock, tailings or exposed sulphide-rich rock (Shaw 1996). For many mines, especially for base metal ones, mill tailings and waste rock always contain sulphides and are exposed to water and oxygen. The oxidation of sulphide minerals (mainly pyrite and pyrrhotite) can produce low pH drainage.

Pyrite and ferrous oxidation react faster at low pH (below 4.5), because the reaction with ferric iron as an oxidant is faster than that with oxygen (Shaw, 1996). Moreover, because of the low pH of the drainage effluent, metals that initially existed in the form of sulphide minerals dissolve into solution. In the presence of bacteria, the reaction rate increases significantly (Singer and Stumm, 1970). The low pH seepage together with the metal rich effluent is a threat to aquatic life downstream of the tailings dams. Pyrrhotite ($Fe_{1-x}S$) is another notable mineral that may generate acid drainage. Pyrrhotite oxidation is 20 to 100 times faster than pyrite. (Nicholson and Scharer, 1993).

However, not all tailings that contain sulphides have high acid generating potential. When natural forms of neutralization such as carbonate minerals exist, acid drainage generating potential can be very low. Minerals such as calcite ($CaCO_3$), dolomite ($CaMg(CO_3)_2$) and anorthite ($CaAl_2Si_2O_8$) can have a comparable neutralizing reaction rate to the sulphide oxidation rate. Silicate minerals also have a neutralization capability, but the reaction is very slow.

ARD endangers living creatures and the environment due to its acidity (pH: 2.5-4.5) and the tendency for heavy metals to be solubilised. If ARD is discharged directly into the environment without any treatment, it can deteriorate water quality, pollute soil, contaminate ground water and sediment in streams, and consequently damage aquatic life, plants, and even human beings via water and food (Jia, 2005). Once pollutants are introduced into the environment, it is quite expensive to remove them and mitigate the negative impacts. For example, according to INAP (2003), the total estimated liability costs from ARD problems of mine waste during 2003 was US$ 1.3-3.3 billion in Canada, US$530 million in Australia and US$1.2-20.6 billion in the US.

Because of these harms, mine closure requirements have become more stringent. Both government and industrial organizations have become involved in prevention and clean-up of ARD; for example, formation of the Canadian Mine Environment Neutral Drainage (MEND 2011) and the International Network for Acid Prevention (INAP 2011). A number of standardized ARD prevention and control strategies have been studied and are recommended to be put into practice. The available mitigation strategies as summarized by Williams and Smith (2000) are shown in Table 1:

Table 1. Suggested ARD Control and Prevention Methods (Williams and Smith, 2000)

	Control Methods
Control of ARD Generation	• Desulphidization of tailings or waste rock • Covers and seals to exclude water or oxygen • Waste segregation • Waste blending • Acid buffering materials addition
Control of ARD Migration	• Covers and seals to exclude precipitation infiltration • Controlled waste placement to minimize infiltration • Diversion of surface water • Interception of ground water
ARD Collection and Treatment	• Active system: e.g. chemical treatment plant • Passive system: e.g. treatment by wetland

2.4. Tailings Mining and Processing Technologies

2.4.1. Tailings Mining Technologies

Muir et al. (2005) has described tailings reclamation methods and operation in detail. There are three major approaches to re-mine tailings from a tailings impoundment:

- hydraulic re-mining;
- mechanical excavation;
- dredging.

Hydraulic mining is mainly used to mine soft rock and move old tailings dams for further treatment. Figure 3 shows a typical tailings hydraulic mining flow-sheet. Monitor guns use high pressure water to wash the tailings downstream where the material is collected in a sump with a screen to prevent large objects from entering the stream where they may block the flow (Tailings.Info, 2010). The reclamation of sand is problematic since sand easily settles out in the launders when flowing from mining face to pump station (Muir, et al., 2005). Therefore, blending with finer materials is required when mining coarse fractions near the spigotted region of the dam. To achieve the required pulp density, the slurry is pumped to a thickener and the underflow is processed with the overflow returning for reuse by the monitors.

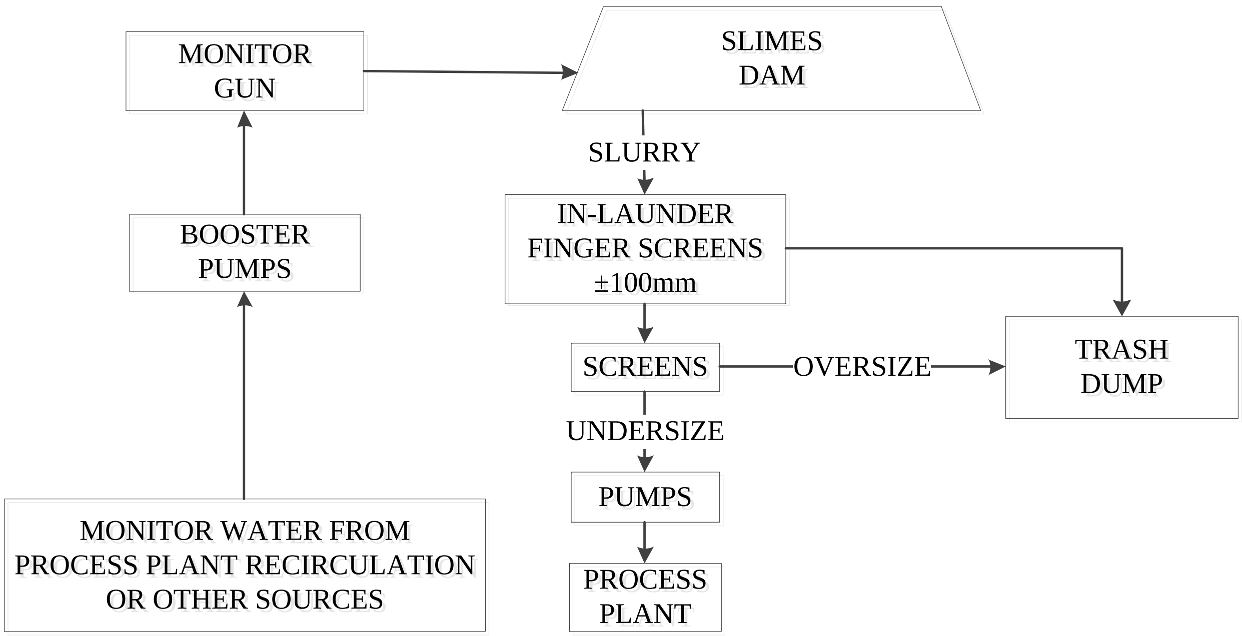


Figure 3. Simplified Flow-sheet of Hydraulic Mining of Tailings (Muir, et al., 2005)

Mechanical excavation is a better option to reclaim sand material, but is a more expensive choice for slime materials due to the tendency of transfer points choking up (Muir, et al., 2005). Mechanical excavation uses equipment such as bulldozers, front-end loaders, and trucks to load, and haul the material.

Dredging is a very expensive method for tailings mining due to the high cost of the equipment and is only used when hydraulic mining and mechanical excavation cannot be used. It is generally reserved for situations such as water-logged marsh areas.

2.4.2. Copper Recovery Technologies

There are primarily three types of copper ores: sulphides, oxides, or a mixture of sulphides and oxides. Approximately 80% of copper occurs in the form of sulphides (Davenport and Biswas, 2002). The main sulphide minerals are chalcopyrite ($CuFeS_2$), chalcocite (Cu_2S), bornite (Cu_5FeS_4) and covellite (CuS). Copper minerals are always associated with pyrite or pyrrhotite, and contain minor quantities of gold and silver. The most common oxide copper minerals are tenorite (CuO), cuprite (Cu_2O), malachite ($Cu_2CO_3(OH)$), brochantite ($Cu_4[SO_4](OH)_6$) and azurite ($Cu_3[CO_3]_2(OH)_2$). The sulphide-oxide ore is generally, but not always separated by flotation into two fractions: sulphide concentrate and oxide concentrate. The sulphide concentrate is treated by pyrometallurgical methods, while the oxide concentrate is generally up-graded by hydrometallurgical methods.

2.4.2.1. Flotation

The most effective method to recover copper minerals is flotation. Copper minerals are selectively attached to rising air bubbles, carried to the top of a flotation cell, and separated from gangue minerals. Copper sulphides can be floated by thiol collectors and depressed by cyanide. The commonly used copper flotation reagents are summarized in Table 2.

Table 2. Common Reagents for Copper Minerals (From Laskowski (2010))

	Reagents	**Usual Addition**
Collectors	Xanthates Xanthogen Formates Thionocarbamates Xanthic Esters Ditiophosphates Mercaptobenzothiazol Fatty acids	20-40 g/tonnes
Frothers	MIBC DF-250	10-15 g/tonnes
pH Modifiers	Lime (CaO) Sodium Carbonate (NaCO3)	0.5-1.5 Kg/tonnes
Activators	Na_2S, NaHS	200-300 g/tonnes
Depressants	Cyanide Water glass	Up to 200 g/tonnes

Sulphide flotation is typically carried out at an alkaline pH (8.5 to 10.5) for primary minerals and 10.5 for secondary minerals if pyrite levels are high. To depress pyrite at normal levels, the pH is generally set to 9 to 10. Conditioning with aeration is helpful to depress pyrite (Laskowski, 2010). Other methods are also applied to depress pyrite such as using a stronger or more selective collector (amyl xanthate and ethyl-isopropyl thionocarbamate). However, pyrite depression is much more difficult when copper-oxide minerals are present due to activation by copper ions in solution since oxide minerals are more soluble. In this case, cyanide can be used to react and form complex copper cyanide ions to inhibit activation and improve selectivity.

Copper oxides can be floated directly with fatty acid or with thio-collectors after sulfidization. Flotation with fatty acid is cheaper, but is often non-selective and cannot be used if the gangue includes carbonate minerals such as calcite and dolomite. In this case, Na_2S or NaHS is used to activate by sulfidization. Sulphide addition however can result in the depression of sulphide copper minerals, so conditioning time is important to overcome such depression. Alternatively a separate sulphide-oxide flotation circuit can be used with the first stage recovering the sulphides and the second stage recovering the oxides.

In dealing with mixed sulphide-oxide copper ores, differential flotation for sulphides and oxides can be used or the two types can be floated together if possible. The latter is preferred due to costs, but its choice depends on the relative abundance of sulphide and oxide copper minerals. Generally when a new deposit is commencing production, the oxide content is higher and then diminishes as the orebody becomes depleted and mining goes deeper.

2.4.2.2. Leaching

Direct leaching of copper ores is also a widely used copper recovery technology these days. About 20% of copper worldwide is produced by hydrometallurgical extraction. It is considered a favourable copper processing technology due to its economic and environmental merits when compared to conventional beneficiation methods followed by smelting and refining which may cause air pollution. Hydrometallurgical copper extraction includes the following operations:

- Leaching to produce impure Cu-bearing solution
- Solvent extraction to transfer Cu from the impure solution to a pure electrolyte
- Electrowinning to produce pure cathode copper

Though hydrometallurgical processing, a final product – pure copper – is produced rather than a concentrate. One drawback of using hydrometallurgy with copper ores is that the recovery of precious metals is generally poorer than using pyrometallurgy.

Leaching is able to treat minerals that are not effectively recovered by flotation ("oxide" copper minerals). For example, chrysocolla ($(Cu,Al)_2H_2Si_2O_5(OH)_4{\cdot}nH_2O$) and earthy cuprite, cannot be floated. Acid leaching provides better performance (Meech 1981). Copper sulphides and native copper can also be leached by H_2SO_4, but reaction rates are low.

When incorporated with flotation, there are several options to choose from to extract copper minerals. Which method is more suitable depends on the sulphide-to-oxide ratio. After bulk flotation of copper and iron sulphides, flotation tailings can be treated by H_2SO_4 to recover oxidized copper minerals. Alternatively, flotation can be applied after the leaching of the copper oxides. The advantage of this method is that acid leaching can clean the surface of copper mineral particles which can improve the performance of sulphide flotation. When the grade of flotation concentrate is too low to be transported and sold to a smelter, leaching of the flotation concentrate is a possible option.

According to Brombacher, et al., 1997, there are several types of commercial-scale leaching technologies used in industry for copper extraction. Examples are *in situ* leaching, dump leaching, heap leaching, vat leaching and agitated reactor leaching. The different methods have quite different leaching cycles. A typical copper heap leaching cycles requires 120-360 days to obtain its designed recovery target. Typically, 72 hours are necessary for leaching in agitated tanks (Mular, et al., 2002). The residence time for vat leaching is 80-120 hours (Meech 1981).

For low grade tailings, heap leaching is favourable due to its low capital and operating costs (Barrett, et al., 1993). The capital costs of Heap Leaching and Solvent Extraction and Electrowinning (SX/EW) vary between $US 4,000-5,000 /annual tonnes of copper. Normally, a 70%-80% recovery can be achieved for copper sulphide ore heap leaching (Peacey, et al., 2004) but may be lower for tailings. Agglomeration is a technology normally applied before heap leaching to increase permeability so the lixiviant can flow through the dump at reasonable rates. However, for copper ores, the binders used to create agglomerates can break down in an acid environment which may lead to failure of the process. Research conducted by Lewandowski & Kawatra (2007) on binders for copper heap leaching has shown considerable success and Bouffard (2005) has reviewed a number of successful practical applications that have evolved over the past two decades. A novel ammonia alkali leach process has also been developed by Alexander Mining in Argentina (Chadwick, 2007).

Vat leaching is a technology for copper extraction for low to medium grade ores. The process allows solution to circulate through a bed of material laid out in layers in a tank or vat.

Conventional vat leaching is not widely practiced because of the high capital and operating costs requirements. However, a new process – INNOVAT Continuous Vat Leaching (Mackie and Trask 2009) – using intermittent vat fluidization shows promise. It has been applied to copper ores and similar capital costs and lower operating costs are indicated in comparison to heap leaching (Mackie and Trask 2009).

Agitated tank leaching is mainly used for rapid dissolution of finely ground oxidized copper minerals and/or roaster calcines (Davenport and Biswas, 2002). At this time, several practical leaching technologies are available for flotation concentrate (mainly sulphides) such as the HydroCopper™ process (Hyvärinen, et al., 2005) and the Mt. Gordon process (Peacey, et al., 2004). The Mt. Gordon Process is a commercial copper leaching process for high grade copper sulphides ores with low capital and production costs. The HydroCopper™ process is used to extract copper from sulphide concentrates with varying mineralogy and grades. One of the remarkable merits of this technology is that it simultaneously recovers gold.

2.4.2.3. Leaching-Precipitation-Flotation (LPF) for Copper Recovery

Leach-Precipitation-Flotation (LPF) is sometimes a suitable method to process mixed sulphide-oxide ores. It includes three processes: leaching in diluted H_2SO_4 solution, cementation of copper using iron scrap, and flotation of fine metallic copper and copper sulphides. In the cementation process, copper ions are reacted with iron scrap:

$$Cu^{2+}+Fe = Cu+Fe^{2+}$$

then the cemented copper and copper sulphides are floated together.

2.5. Conventional Reclamation Methods

Several studies on mine tailings reclamation have been conducted in Canada, USA, Australia, and Europe. In general, the purpose of tailings impoundment reclamation is to achieve long-term physical, chemical, and biological stability to support desired land use after closure. Physical stability means that tailings storage facilities are required to remain stable under extreme events such as floods, earthquakes, and wind and/or water erosion. To achieve chemical stability, the health and safety of humans and water quality downstream must not be endangered by leached contaminants migrating into the environment. Biological stability means a balanced biological environment and a self-sustaining ecosystem in the tailings storage area. To support desired land use, the tailings impoundment area must be returned to acceptable original or alternative use (Xenidis, 2004).

There are a number of reclamation and control methods to achieve physical and chemical stability, and land use of tailings impoundments as follows:

Riprap. This is a conventional, effective method to protect tailings facilities from water and wind erosion. It is a proven, effective, long-term method. It is relatively cheap since riprap can be obtained from mine waste rock or smelter slag. However, it is costly to cover the whole tailings impoundment and it generally retards natural re-vegetation (Vick, 1983).

Neutralization with alkaline materials or addition of bactericides. Limestone and sodium hydroxide are the most widely used pH buffers. At pH 9, most heavy metals will precipitate as metal hydroxides (Shaw, 1996). However, neutralization with acid buffering materials is ineffective because of limited solubility and coatings with ferric hydroxide precipitates that passivate the surfaces. The method is clearly a temporary measure since the buffering materials are effective only until they are finally consumed by the acid effluent. A large quantity (millions of tonnes) of neutralizing materials is required, which is impractical unless it is already part of the tailings.

Since microorganisms play a significant role in acid generation, bactericide might be useful to control the oxidation rate of sulphides. However according to Dugan (1987), this method kills microorganisms for only a short term and becomes ineffective due to mutated resistant bacteria re-colonizing the site.

Treatment of acid effluents. There are two systems for this method: active and passive. With active systems, effluents are collected and treated in a chemical treatment plant. Passive systems use wetlands, alkaline trenches, and retention ponds to treat acid effluents (Xenidis, 2004).

Use of oxygen and/or water barrier covers. Covering materials can be obtained from different resources, for example soil, wood-chips and manure mixtures, polymeric materials, and sulfur-poor tailings (Shaw, 1996). Wickland (2006) summarized recent studies on cover systems and pointed out the complex performance of soil covers. Some practices are quite successful; others are initially effective, but fail as time passes. Some researchers claim the results of barrier and multi-layer soil covers are not worth the high construction cost. Soil covers act as an effective barrier to water and oxygen in the short-term, but long-term performance is questionable due to the influence of climate, vegetation, burrowing animals, erosion, and so on. Water cover is the only proven technique to effectively prevent oxygen influx and so, also prevent acid generation.

Desulphurization. This is an attractive alternative method to treat ARD problems. By means of this method, the amount of acid generating tailings can be reduced by removing the sulphide fraction and storing it in a separate facility or location (sometimes used as underground backfill).

To achieve desulphurization, several approaches are available, such as bulk flotation or sulphide oxidation by roasting or autoclave (Benzaazoua et al., 2000; Sollner, 2004). The separated sulphide fraction may be easier to deal with because of its lesser volume. The desulphurized fraction is now safer to dispose of and can possibly be used as cover (Benzaazoua et al., 2000).

Revegetation. If a vegetation cover can be established naturally, wind and water erosion can be reduced. This method prepares the land for future alternative use. However, tailings are generally unsuitable for plant growth due to low pH, high levels of heavy metal, macronutrient deficiency, and poor structure (Xenidis, 2004). In this case, direct revegetation of tailings may not be successful, so top soil may be required, although this is a more expensive alternative.

Chapter 3. Model Development

3.1. Model Overview

To investigate and assist in the decision-making regarding tailings retreatment, a general model has been developed that considers significant contributing factors, applicable mineral processing and reclamation technologies, potential economic benefits, and major environmental and social concerns.

To decrease the scope of the work to a manageable level, the research has been limited to copper tailings, specifically from a hypothetical mine located in British Columbia, Canada. The characteristics of the tailings materials and macro-properties are in accord with BC copper mines, as well as the requirements of regulations, policies, and laws in that province. The main reason for choosing copper is the sharply increased copper prices over the past decade together with the ever-increasing demand for copper. British Columbia is one of the leading copper producers in Canada (and the world) and has a considerable number of old mines that require study. Moreover, since the increasingly restrictive regulations and legislation in Canada have in turn increased the cost of dealing with problems associated with old tailings dams, solutions are required to reduce long-term liability costs and to protect the environment and communities.

The tailings retreatment model collects user input through a series of questionnaires. These data are then processed and a recommendation is given. Three modules were designed to process inputs and evaluate economic benefits together with reduction in the environmental and social impacts of tailings retreatment. Based on the results from these modules, a decision-making module then combines all criteria and generates an output report.

3.2. Overall Research Strategy

An Excel spreadsheet model was created using Visual Basic (VB) to conduct the research. Excel was chosen since it is one of the most accessible programs available today and is compatible with most computer operating systems, thus ensuring wide and easy application well into the future. Excel can efficiently complete all required mathematical calculations and can also be programmed to perform linguistic analyses using fuzzy logic. With Visual Basic, Excel can fulfill even more complicated tasks that can be modified by others, if and when required. As well, a user-friendly interface can be built using macros and VB functions.

To complete the conceptual design and economic analysis, an Economic Module was developed to apply mass balance calculations, perform revenue, capital and operating cost estimates, and to calculate the Net Present Value (NPV) and Discounted Cash Flow Rate of Return (DCFROR) of the project. Based on the NPV levels, the module chooses a preferred processing flowsheet. Risk of failure and estimates of such a failure are used to project future costs of environmental damage from a legacy site back to present day costs.

To run the model, a substantial amount of data is required as input. When such data are unavailable, the system can apply default values to continue an evaluation session. The technologies in current use for mining, processing, disposing, and reclaiming tailings were extracted from the literature. Capital and operating costs data were collected from a review of similar tailings retreatment and copper processing projects already in existence.

Three modules (Tailings Disposal and Reclamation Selection, Risk Assessment, and Decision-Making) were developed using a fuzzy expert system - a method that can process linguistic and imprecise inputs not suitable for analysis by conventional mathematical models. Fuzzy logic systems are able to assemble uncertain and vague input, and address problems in a way similar to how humans make decisions based on their experiential knowledge. The application of this methodology has been successfully applied in environmental risk assessment and decision-making among many other fields (Veiga & Meech, 1995; Balcita, 2001, Braglia et al., 2003; Golestanifar et al., 2010; Mohammadi and Meech 2011).

A fuzzy logic system is composed of three major components: Fuzzification, Inference, and Defuzzification. The structure of a fuzzy logic system is shown in Figure 4.

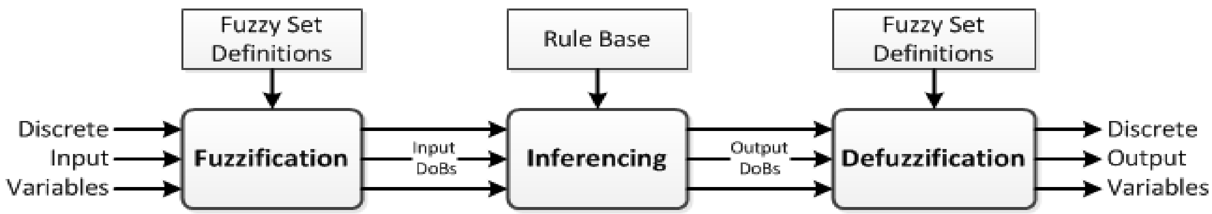


Figure 4. Structure of a Fuzzy Expert System (Meech, 1999)

The first process, Fuzzification, collects discrete numerical input and converts these to linguistic terms ("Low", "Medium" and "High"), each assigned a Degree of Belief (DoB) ranging from 0 to 100 using a series of defined fuzzy sets. Figure 5 shows an example of three fuzzy sets for two input variables. For most inputs, three triangular-shaped fuzzy sets have been used for convenience to linguistically characterize a Universe of Discourse ranging from a minimum level to an average level to a maximum one.

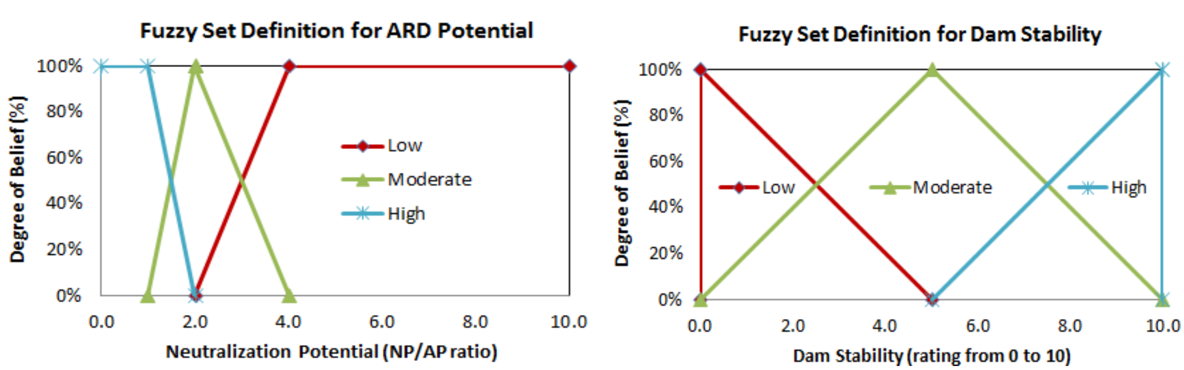


Figure 5. Triangular Fuzzy Sets (after Balcita (2001))

Table 3. Rule-Base Matrix for Possible Use of Submarine Disposal. (after Balcita (2001))

Submarine Disposal		Site Characteristics		
		Poor	**Moderate**	**Good**
Environmental Sustainability	**Low**	Low	Low	Low
	Moderate	Low	Moderate	Moderate
	High	Low	Moderate	High

The inference process allows all fuzzified input variable DoBs to run through the rule base and subsequently generates a series of conclusion statements associated with the DoBs in the output fuzzy sets. A Fuzzy Associative Memory (FAM) map, shown in Table 3, is used to depict the rules for two fuzzified input variables. To obtain a DoB for each output statement in the rule base, a min-max method is used in which the minimum DoB value of the rule antecedents is the true DoB value of each rule's output with the maximum value for similar output states selected as the final output DoB for that state (Braglia et al., 2003). A Weighted-Inference Method can also be used to sum the product of variable importance (W_i) and the respective DoB_i for each statement in the rule premises (Meech & Veiga, 1998). The method can be expressed in the following equation:

$$DoB_{conclusion} = \sum_{i=1}^{a} W_i * DoB_i$$

This latter method provides a more flexible way to add new variables or fuzzy sets into the analysis without necessarily adding more rules to the system. Both of these approaches have been employed in this research.

The Defuzzification portion of the system processes the output statements associated with the DoBs into a fuzzy singleton using a weighted-average method as follows:

$$z = \frac{\sum_{i=1}^{m} [DoB(z_i) * Sup(z_i)]}{\sum_{i=1}^{m} DoB(z_i)}$$

where,

z = discrete value for variable Z

$DoB(z_i)$ = degree of belief in Z_i (fuzzy set describing part of the Universe of Discourse)

$Sup(z_i)$ = supremum position of Z_i

i = output fuzzy set index

m = total number of output fuzzy sets

3.3. Model Framework

As a basis for the model, a general flowsheet (Figure 6) containing major mining and processing operations was developed. Retreatment is comprised of 4 stages: mining, processing, final tailings disposal, and reclamation. To complete the conceptual design of all these operations, and to estimate economic and environmental performance, the Tailings Retreatment module uses the structure shown in Figure 7. The model collects and processes input data through the four sub-modules to generate final output in the form of a recommendation and report. These four modules are: Design and Economics, Disposal and Reclamation, Risk Assessment, and Decision-Making. The first three provide evaluation criteria based on user input while the fourth module combines the three criteria to recommend a preferred decision on whether to re-mine and re-process the tailings.

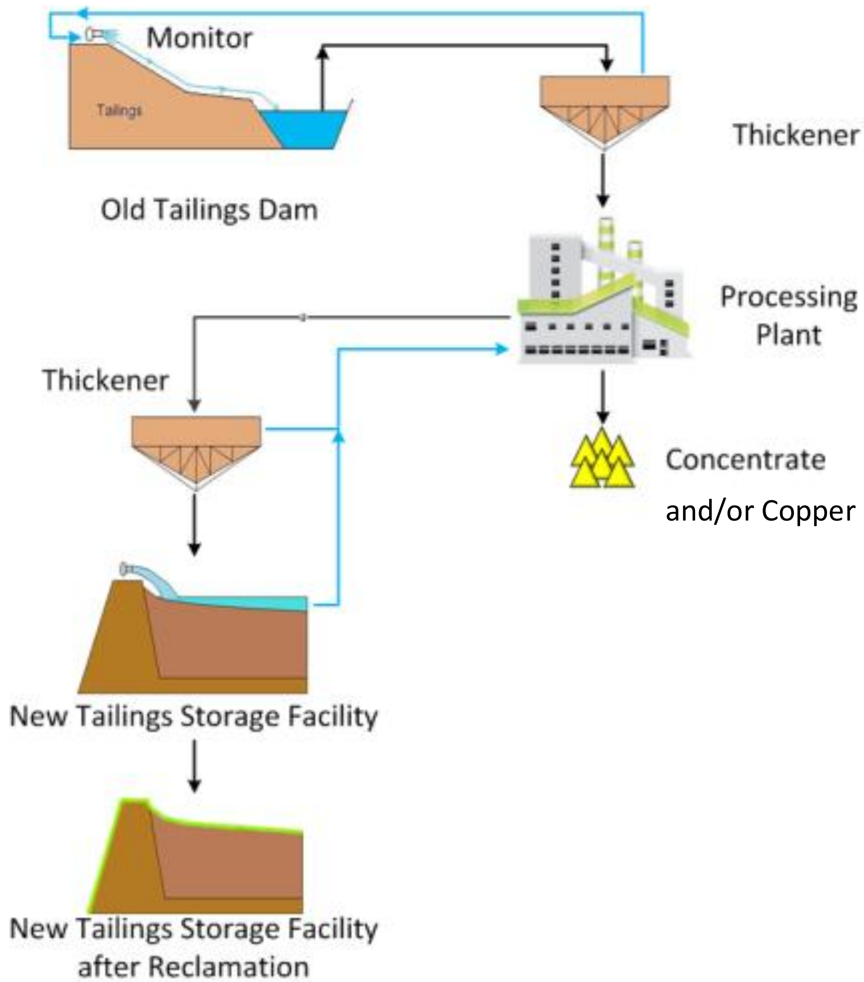


Figure 6. Simplified Flowchart of Retreatment Method

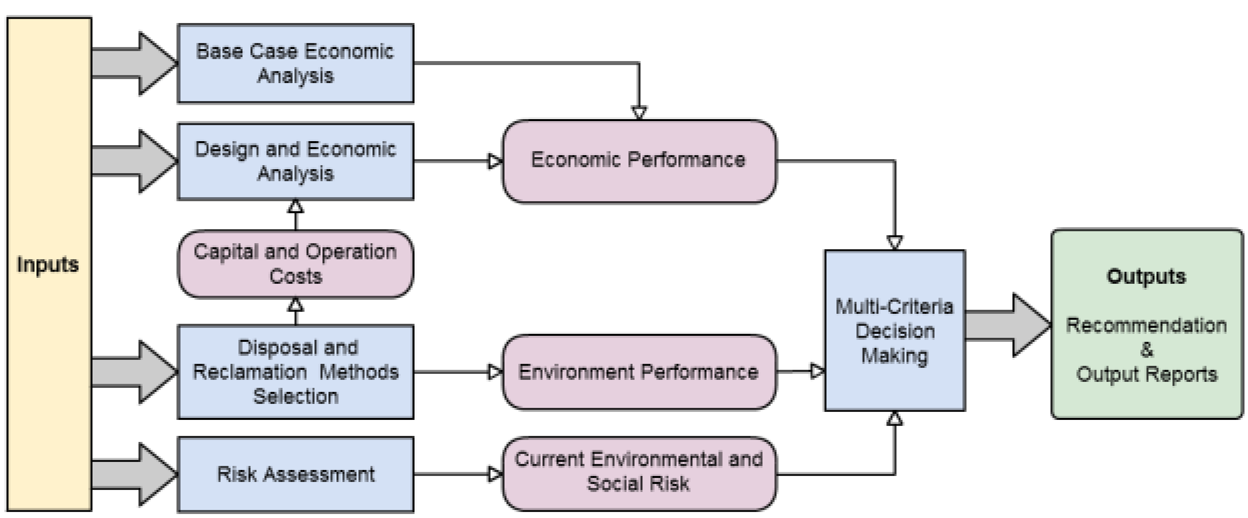


Figure 7. The Framework of the Tailings Retreatment Model

3.4. Inputs and Outputs

3.4.1. Inputs

The model inputs include mine, ore and mineral properties, mineral processing and metallurgical test results, site characteristics, environmental and social performance, market conditions, and other parameters. The tailings retreatment model collects input information from the user through questionnaires shown in Appendix B. Users indicate their response to each question by typing data or clicking on a choice of answers.

It is recommended that the user completes all the questions leading to a much more reliable output. Significant inputs such as copper grade, and total tailings tonnage must be provided to ensure successful analysis. However, since the model is only preliminary, this study falls into a class known as "Order-of-Magnitude". Insufficient investigation and test results are common for tailings projects, so some inputs are set to default values (see Appendix B), such as mineral processing and metallurgical test results (Table A-4, Appendix A) and weight values regarding environmental and social risk assessment (Table A-5, Appendix A).

The default assumptions of mineral processing and metallurgical performances are mainly based on a review of other copper ores or tailings processing cases. However, since the properties of newly mined ores and long-storage tailings can be different, considerable uncertainty associated with these default values must be recognized. Tailings materials rarely achieve a similar processing performance to a newly mined ore mainly due to the following reasons:

- Coarse locked particles
- Non-floatable minerals (copper oxides and ultra-fines)
- Oxidized particle surfaces

However, because of new technology developments, today it may be possible to recover some of these lost copper particles using the following strategies:

- Regrinding to liberate locked minerals
- Leaching to recovery non-floatable minerals (A.S. Cu) and ultra-fines
- Grinding/scouring to clean oxidized particle surface for flotation
- Leaching of flotation concentrate to overcome product grade limitations
- Treatment of "coarse" and "fines" separately to enhance effectiveness and efficiency

To understand the influence of processing performance on the final economic analysis results, the model can conduct a sensitivity analysis on product grade and recovery. This has been done for the four case studies used to validate the system and is summarized in Section 4.2.

When information is available from mineral processing and metallurgical tests, these default inputs can be replaced to increase reliability. Appendix D contains actual input values for the four case studies that are set as default, but which can be updated if known. These include smelting and refining recoveries and charges together with transportation costs.

Some input data play significant roles in how a tailings disposal or reclamation method performs, such as site characteristics, tailings properties, climate, and surrounding topography. They are also collected by entering either quantitative (a discrete number) or qualitative inputs (a linguistic term such as "Low", "Moderate" or "High").

The inputs of consequences and occurrences related to risk assessment are also obtained from the user. To avoid biases from people with specific backgrounds and to ensure all relevant aspects are considered, the questionnaire should receive input from a team instead of only one or two persons. As a result, the questionnaires should be completed by several representatives that are able to cover a variety of experiences and perspectives relevant to the project. These might include mine manager, environmental coordinator, mine planning engineer, mine geologist, community liaison officer, a representative from the corporate office, and possibly a member of an affected community. As a team-based activity, a collaborative survey can be achieved through a workshop or a community meeting or by having each representative fill out a separate questionnaire and then combining these answers into a single evaluation. The inputs regarding failure consequence and occurrence are also entered qualitatively by linguistic terms, which are explained in more detail in Section 3.7.

The weights required to conduct selection of tailings disposal and reclamation methods, and to carry out a risk assessment are also obtained from the user or they are set as default values.

3.4.2 Outputs

The output report (such as the output sheet for each of the four case studies in Appendix D) aims to present the following information:

- Selected mining, processing, disposal, and reclamation methods
- Processing performance
- Economic performance
- Environmental performance after retreatment
- Current risk assessment result
- Sensitivity Analysis
- Final Recommendation

The processing performance includes recovery, concentrate grade, and production rate of concentrate and/or pure copper. For the economic performance, the terms listed below are calculated:

A	Project NPV @ 5%	Copper Recovery Project
B	Base Case PV @5%	Savings on Current Remediation Costs Plus Avoidance of Future Failure Costs
C	Total NPV @5%	Sum of A & B
D	Project NPV @10%	Not Used
E	DCFROR	Not Used

The sum of project NPV @5% and Base Case PV @5% are used as a criterion for final decision-making. Although the project NPV @10% and DCFROR are also calculated, they are not considered in the decision-making process. Total capital and operating costs, annual revenue, and annual cash flow are also given in the output worksheet. The sensitivity analysis of head grade, copper price, capital costs, operating costs, and metallurgical performances are shown in graphical form.

The current environmental and social risk assessment result is shown in the form of a Degree of Belief (DoB) in a "high" risk for use in the decision-making process. It is necessary to point out that this single value is not a panacea to determine the final risk. However, by weighting different aspects of the project in an "intelligent" way, this risk assessment approach can provide a relatively reliable reference for risk assessment.

The environment performance of the project is also shown in the output worksheet. This is also presented in the form of a DoB to indicate the possible success of the reclamation strategies.

The last output of the model is the recommendation on tailings retreatment (also a DoB) which is listed in the output report in the form of three options (YES, NO, or MAYBE).

3.5. Design and Economics Module

3.5.1. Module Overview

The Design and Economics Module performs a conceptual process design, gives a preliminary economic evaluation, and provides economic criteria for deciding on a tailings retreatment.

Depending on the information provided by the user in terms of tailings dams and impoundment material properties, market conditions (copper price), and knowledge about process performance, the reprocessing effectiveness and economic performance are estimated for different potential flowsheets. A mass balance is calculated based on each flowsheet with the recovery, concentrate grade and copper production rate obtained as output. Revenue, capital and operation costs are then estimated. Finally, based on these results, the corresponding NPV and DCFROR are generated. A preferred flowsheet (with best economic performance) is chosen in which the effectiveness and economic results are summarized as economic criteria and presented as output. Finally, a sensitivity analysis is conducted.

3.5.2. Design and Economic Analysis

The conceptual design for re-mining and re-processing the tailings material is completed by this module. Mechanical excavation and hydraulic mining are potential tailings re-mining methods. Norgate & Jahanshahi (2010) have summarized the copper extraction technologies used for low grade ores in practical use, including conventional pyrometallurgical and hydrometallurgical processing. These processing technologies include heap leaching, tank leaching, and flotation and combinations thereof. Size classification and grinding are also included as options that may improve the processing performance. Table 4 describes the technologies tested in the module.

Table 4. Mining and Processing Technologies

Technologies	Merits and Function
Hydraulic Mining	- Low capital and operating costs - Easy operation
Mechanical Excavation	- Effective for tailings reclamation in arid area
Heap leaching	- Low capital and operation costs - Environmentally friendly - Good performance on low grade copper ore
Classification	- Fine portion (lower grade) discarded to increase head grade
Grinding	- Particle surface cleaning/scouring through a 'light' regrind - Liberation improvement
Flotation	- Effective and mature technology to recover copper minerals
Tank Leaching	- Low capital and operation costs - Environmentally friendly

Figure 8 is a completed copper mining and processing flowsheet including all the technologies mentioned above. It is able to represent over 200 options by combining various unit operations, which makes it a powerful guide for the rules development of unit operation selection. However, at the present stage, eleven specific flowsheets, listed below, have been selected and tested in the module to limit the project scope:

1. Flotation of bulk material only
2. Regrinding (coarse fraction) followed by flotation of bulk material
3. Flotation (bulk) followed by heap leaching of the new tailings pile
4. Flotation (bulk) followed by concentrate leaching
5. Flotation (bulk) followed by heap leaching of a concentrate pile
6. Heap leaching of bulk material
7. Heap leaching of coarse fraction
8. Heap leaching (bulk) after agglomeration
9. Straight tank leaching of bulk material
10. Tank leaching followed by flotation of leach residue
11. Regrinding of coarse fraction followed by tank leaching of bulk material

The flowsheet diagrams for these 11 alternatives are presented in Appendix E.

Depending on the mine production status and equipment availability, several possible capital cost reducing strategies are employed to examine ways to improve the economic performance. If the mine is still in production or is ready to close, there is a good chance that existing facilities are available for use to reprocess the tailings. In this case, flowsheets using existing equipment are analyzed in competition with other alternatives based on information such as capital costs and average annual operating costs together with process performance. If used equipment is available (as indicated by the user), this can significantly reduce capital costs, but may increase operating and maintenance costs. For each alternative flowsheet, if used equipment is available then two scenarios, using new and used equipment, are compared.

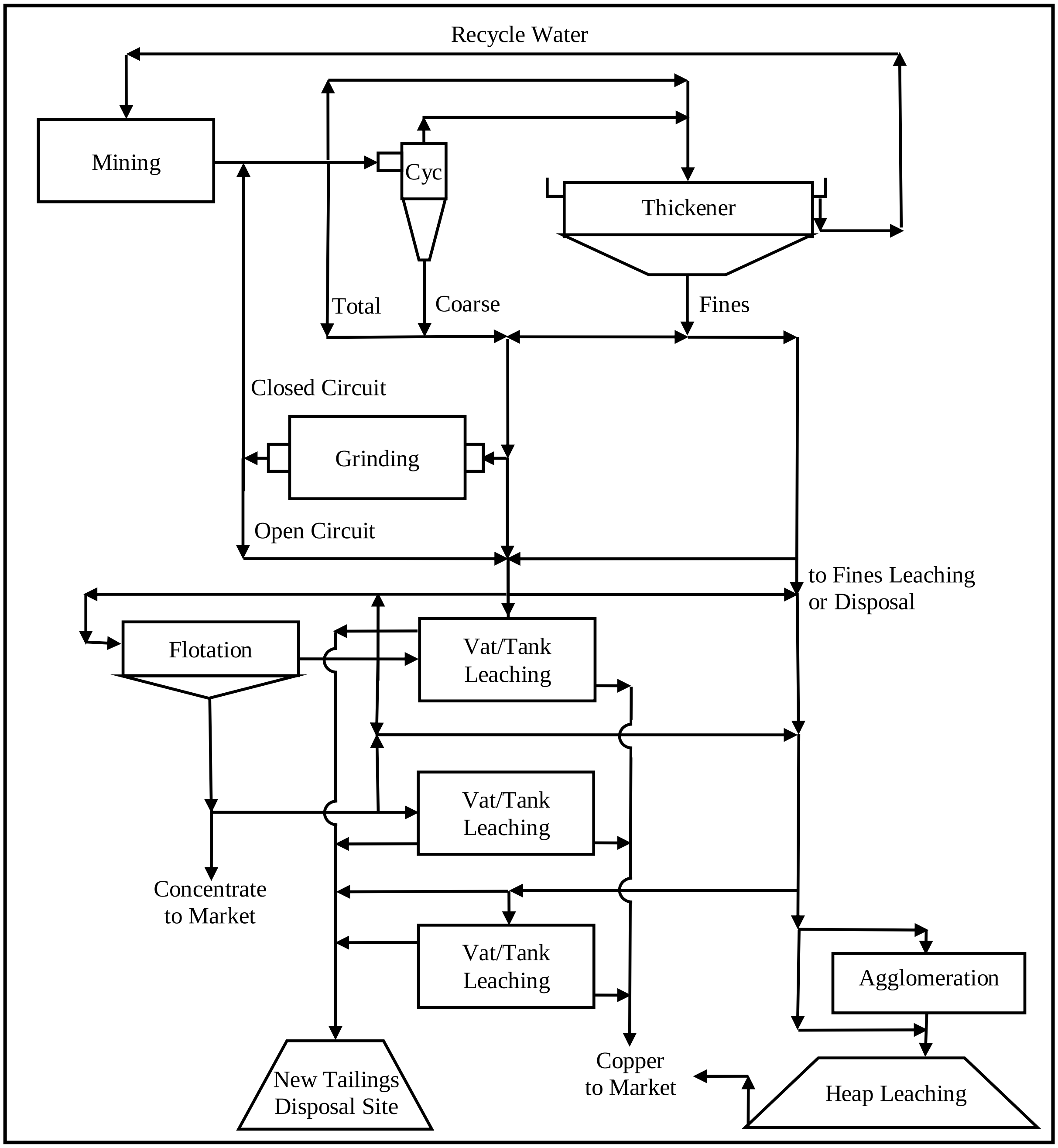


Figure 8. Complete Copper Processing Flowsheet showing all optional flows.

The design of each flowsheet follows the following steps:

- material balance calculation;
- capital costs and operating costs estimate;
- NPV and DCFROR calculation;
- tonnage rate optimization;
- flowsheet selection;
- sensitivity analysis.

3.5.2.1. Material Balance Calculations

For flotation, the feed tonnage rate as well as the processing performance (total recovery, grade of final product and production rate), are calculated using the two-product formula. For tank leaching, the feed grade and the grade of residue are used to determine the plant recovery. For heap leaching, input data on the estimated final recovery after 60 days of leaching is required.

Based on these calculations, the capital and operation costs, and revenue are estimated for further economic analysis and flowsheet selection.

3.5.2.2. Capital and Operation Costs Estimate

As a preliminary study, the cost estimate is restricted to an Order-of-Magnitude method. Therefore, the Six-Tenth Rule was used to size equipment and estimate capital and operating costs since these costs depend to a large extent on capacity in terms of tonnes per unit time (days or annual). This cost estimate approach is based on previous cost data and mineral processing knowledge. It is generally expressed as follows (Mular, Halbe, & Barratt, 2002):

$$\text{Cost} = a\,(\text{Parameter})^{0.6}$$

The equation can be expressed in an alternate way when costs data are available for a particular known situation and need to be scaled up or down to another situation:

$$\frac{\text{Cost1}}{\text{Cost2}} = \left(\frac{\text{Parameter1}}{\text{Parameter2}}\right)^{0.6}$$

The parameters can be capacity, area, mass, volume, power draw, or one of many others. For cost calculations, parameter generally stands for capacity. In the case of grinding, power required per tonne ore, which is directly related to capital and operating costs, is set as the parameter in order to correct for the effect of feed and product particle sizes. To calculate required power per tonne, Bond's equation is normally used (King, 2001):

$$W_0 = 10 * W_i * (P_{80}^{-0.5} - F_{80}^{-0.5})$$

where

W_0 = kW hours per tonne required for grinding;
W_i = Work index for tailings material, kWh/tonne;
P_{80} = 80% passing particle size of grinding feed materials;
F_{80} = 80% passing particle size of grinding product.

The average Work Index of copper ores is about 12 kWh/tonne, but increases with finer product particle size. As a result a Fineness of Grind Efficiency Factor (EF) is applied to correct the Work Index of tailings at finer sizes by the equation below (Mular, 2002):

$$W_{i\,(corrected)} = W_{i\,(bond)} * EF$$

$$EF = 1 \text{ for } P_{80} > 75\ \mu m$$

$$EF = \frac{P_{80} + 10.3}{1.145 * P_{80}} \text{ for } P_{80} \leq 75\ \mu m$$

Since this system is performing an Order-of-Magnitude estimate, other efficiency factors, such as dry grinding and mill diameter factors, are not considered.

By reviewing other tailings re-processing projects, as well as copper milling and hydrometallurgical projects, the capital cost data (Table A-1, Appendix A) were collected. Then by using the Six-Tenth Rule, the capital cost is adjusted to input tonnage rate and then, updated to the present value (i.e., to adjust for inflation) using the M&S Index (2011) which is 1,624 in 2011. This accounts for changes in equipment cost considering general inflation over time.

As a preliminary estimate, the operation cost does not include a fixed component, because it typically represents a smaller portion for mining and milling projects. Therefore, the operating cost of each unit operation is taken as variable over capacity changes. This is a highly conservative approach that factors unit operating costs quoted for similar projects (Table A-2, Appendix A) to the feed tonnage rate and updating to the present using the M&S Index (2011).

After totalling the operating costs of all unit operations, as well as those for tailings disposal and reclamation from the Disposal and Reclamation Module, the total operating costs are estimated in $US per day, and then converted and reported in dollars per year and dollars per tonne of processed ore in the output worksheet (Lewis & Bhappu, 1978).

If an existing mill or leach plant is to be used, it must be adapted for tailings reprocessing by either up-grading or overhauling equipment and pipe lines. This involves capital investment, but at a much reduced level than purchasing and installing new equipment. A suitable reduction ratio (0.2) derived from a 10% depreciation rate and an average 15-year equipment age was applied to the required new equipment capital costs to obtain the cost to refurbish existing equipment. In the case of salvaged equipment, since the purchase price is much lower than a new purchase, a similar reduction ratio was used to adjust the capital costs. However, extra costs are required for maintenance and regular repair to keep the equipment running. As a result, a percentage (15%) derived from a uniform annual 5% increased maintenance and repair cost (20% of total operation cost) over 15 years, is added to the operating cost of new equipment.

For the use of equipment in an existing plant, the capital cost ratio used was 0.20 to account for refurbishing while the increase in operating costs used a ratio of 1.15. For salvaged equipment, the capital cost ratio used was 0.25 while for operating and maintenance costs the ratio used was

1.20. The higher values for salvaged equipment over existing equipment relate to additional delivery and installation costs applied to capital and additional unknowns associated with maintaining newly delivered salvaged equipment compared to existing one.

By summing the capital costs of mining and reprocessing calculated in the Design and Economic Module together with the tailings disposal and reclamation costs from the Disposal and Reclamation Module, the total capital cost is obtained for further analysis.

3.5.2.3. NVP and DCFROR Calculation

To evaluate the economic performance, the Net Present Value (NPV (@5% and NPV @10%) and the Discounted Cash Flow Rate of Return (DCFROR) are calculated based on the revenue, capital costs, and operating cost data.

The major revenue of a tailings retreatment project is from the sale of valuable metal. In the economic model, the revenue of the base case is based on a constant copper price. The revenue is estimated by the value of contained metal in concentrate for the flotation circuit after a suitable deduction to account for smelter losses and after subtracting smelting, refining, and freight charges. For leaching, the copper production rate is used after deducting an amount for transportation and marketing.

The two major economic criteria, NPV and DCFROR, are estimated using the capital and operating costs, taxation rates, and calculated revenue as follows (Meech and Paterson, 1980):

Annual Cash Flow

$$CF = (1 - t) \times (Rev - Cop) + t \times D$$

Present Worth Factor

$$F = CC/CF$$

Find the value of "i" by interpolating the following equation:

$$\mathrm{F} = \frac{(1+i)^n - 1}{i(1+i)^n}$$

Then

$$\mathrm{DCFROR} = 100i$$

After choosing a desired interest rate (i*), the NPV can be obtained as follows:

$$\mathrm{NPV} = \mathrm{CF} \times [\mathrm{F}(i^*)] - \mathrm{CC}$$

where

CF	=	Cash flow, in $US in 2011;
t	=	Tax rate, %;
Rev	=	Annual revenue, in $US;
Cop	=	Annual operating costs, in $US in 2011;
D	=	Annual depreciation, in $US in 2011;
F	=	Present value factor;
CC	=	Annual capital costs, in $US in 2011;
n	=	Mine lifetime, years;
i	=	Interest rate;
i*	=	Desired interest rate.

3.5.2.4. Tonnage Rate Optimization

Since the project capacity plays a crucial role in revenue, capital costs, and operating costs, the relationship between mining and processing tonnage rate (tonnes per day) and economic performance (NPV@5%) is identified. Based on the assumption that the mine life of a tailings retreatment project will lie between 5 to 20 years, an NPV calculation is done for different tonnage rates. The tonnage rate leading to the highest NPV is selected to make sure a preferred economic performance is achieved.

3.5.2.5. Flowsheet Selection

The mining method selection for each potential flowsheet depends on the copper extraction technology applied and whether abundant water is available. Although hydraulic mining is the preferred tailings mining method because of its low capital and operating cost and ease of operation, it is not applicable in all cases. For the alternatives in which heap leaching is the only copper extraction operation, mechanical excavation is chosen as the preferred mining method which eliminates the dewatering operation. For the other alternatives involving flotation or tank leaching, water availability becomes a serious restriction on its application. When the user indicates not enough water is available to wash out the tailings material from the dam, then the mechanical excavation method (haulage) is selected.

For the processing flowsheet, after the economic performances of all options are obtained, a preferred flowsheet with best economic performance (highest NPV@5%) is chosen as a result of reprocessing design.

3.5.2.6. Base Case Economic Analysis

Some tailings sites may be associated with serious environmental problems and a high risk of failure of the dam. In many cases, chemical stability issues like ARD can lead to considerable cost for on-going remediation programs now and in the future (hundreds of years). Moreover, dams can cause a tremendous economic cost if the structure fails at some point in the future. These costs cannot be ignored, so they are taken into account in the economic analysis and converted into a present value (NPV@5%) by the following equation:

$$PV = \sum_{t=0}^{n} \frac{CF}{(1+i)^t}$$

where,

PV = Present value;
CF = Future annual cash flow of cost, \$US;
t = Time, year;
n = Mine lifetime, years;
i = Desired interest rate.

The annual operating cost and project running years are obtained from the user when there is ongoing pollution remediation work (such as a water treatment plant). Typical WTPs require a re-build every 20 years with annual operating and maintenance costs on-going each year.

To convert the failure risk to a present value, data about the possible risk of failure and the possible consequences (economic costs such as clean-up, compensation, loss of lives, etc.) are required. A Weibull cumulative probability distribution function (PDF) (Johnson, Kotz, & Balakrishnan, 1994) is used to estimate the failure possibility changing over time:

$$\lambda(t) = \frac{\beta}{\eta} \left(\frac{t-\gamma}{\eta}\right)^{\beta-1}$$

where

t = Time, year;
β = Shape parameter;
η = Scale parameter;
γ = Location parameter, equal to 0 in this analysis.

Based on information about failure risk obtained from the user (failure probability in 5 years, and year of certain failure), the parameters β and η are determined. Figure 9 gives an example of 5% failure probability and a certain failure year of 50. As shown in this diagram, the certain failure year restricts the distribution scale (parameter η), while the 5-year failure probability determines the shape of the line (slope of the probability plot).

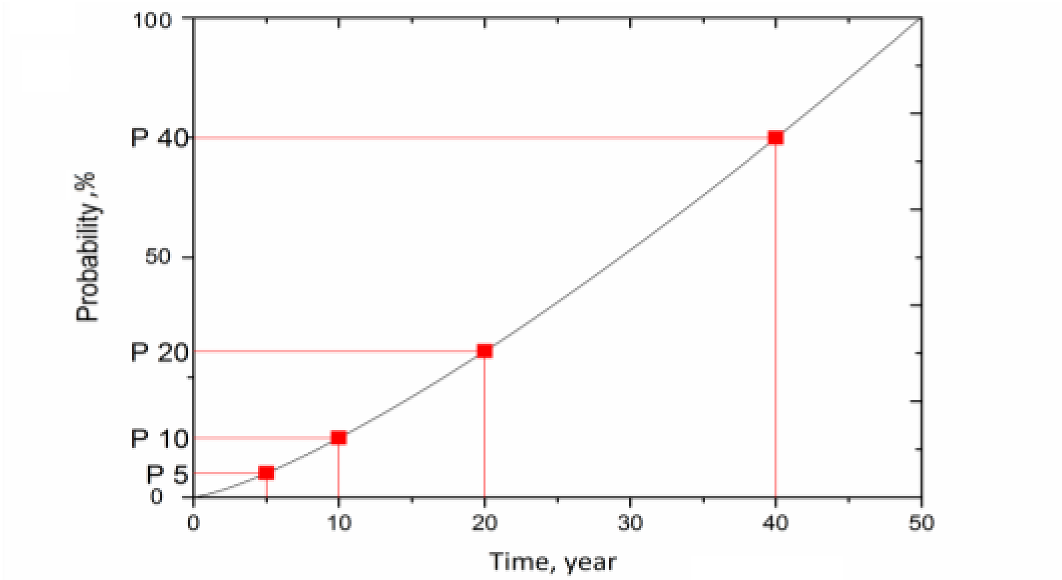


Figure 9. Example Weibull Failure Probability versus Time

Once the parameters β and η are known, the cumulative failure probability of 10, 20, 40 and 50 years can be calculated with the Weibull function. These failure risks are then converted to a present value by the equation below:

$$PV(n, m) = \lambda(n, m) * \frac{L}{(1+i)^{(m+n)/2}}$$

where,

$\lambda(n,m)$ = Failure probability from year n to year m, %;
L = Estimated failure economic loss, \$US;
i = Desired interest rate.

By adding the calculated PVs of different periods from present to the certain failure year together, the estimated PV for a tailings dam failure can be obtained. Finally, the sum of remediation cost PV and failure risk PV as part of the base case economic performance are subtracted from the project NPV to represent the true economic performance improvement, which is used as a criterion for the final decision making.

The measurement of failure probability and consequence is a complicated process depending on a myriad of factors such as geology, hydrology, local climate, slope stability, impact on local community and so on. Without a detailed assessment, failure probability should be estimated by geotechnical experts. To estimate failure loss (or consequence), based on a review of some recent tailings accidents (summarized in Table 5), a reasonable guide is as follows:

- Low Consequence: < US$ 1 million loss
- Medium Consequence: US$ 1 – 50 million loss
- High Consequence: > US$ 50 million loss

Table 5. Representative Failures and Estimated Costs

Site	Ore	Date	Volume (m^3)	Clean-up Costs	Write-down of Assets
Massey Coal, Kentucky	Coal	Oct. 2000	65,000	$50M	20M
Aitik Mine, Sweden	Cu	Sept. 2000	400,000	$1M	$0M
Baie Mare, Romania	Au	Jan. 2000	100,000	$150M	$30M
Los Frailes, Spain	Pb-Zn	Apr. 1998	6,800,000	$136M	$150M
Marcopper, Philippines	Cu	Mar. 1996	1,600,000	$80M	$60M
Omai Mine, Guyana	Au	Aug. 1995	4,500,000	$12M	$8M

To recognize the impact of different failure loss estimates on economic performance (total NPV@5%), a sensitivity analysis is conducted. The results for the four case studies are presented in Chapter 4.

3.5.2.7. Sensitivity Analysis

Although some inputs are set to one level in the model (copper price, for example), a sensitivity analysis is conducted to identify the effect that the inputs may have on the performance of the project. The sensitivity analysis is conducted for the selected flowsheet. The changes in NPV(@5%), in accord with variations in head grade (±25%), copper price (±25%), capital and operating costs (±40%), are estimated and presented in the form of a spider graph.

3.6. Disposal and Reclamation Module

3.6.1. Module Overview

The Disposal and Reclamation module has been designed to evaluate economic and long-term environmental performance of different options for tailings disposal and reclamation, to select a preferred alternative, and to provide criteria for final decision-making. A fuzzy evaluation method was used to deal with linguistic and imprecise inputs and to simulate human thinking.

There are a myriad of factors that may have an effect on the selection of a disposal and reclamation method. Cost is a major factor, but nowadays environmental, social concerns and regulatory issues also have become vital factors. When supervised by regulatory agencies and the public, these latter factors may over-ride the economics of the project (Golestanifar & Aghajani Bazzazi, 2010). In this study, the activities of tailings re-disposal and reclamation include:

- Minimizing environmental risk
- Minimizing costs
- Maximizing the integrity and safety of TSF
- Minimizing the land disturbance for the disposal area

Based on these aims, the three selection criteria are TSF stability (both physical and chemical); life cycle costs, and total land disturbed (Ilhan & Repetto, 1999; Vick, 1983).

After the data related to tailings properties, site characteristics, climate, and topography are collected and analysed using the selection module, a preferred tailings disposal method associated with a reclamation plan is generated as output. The environment performance and cost for the chosen methods are also provided for the economic analysis and final decision making.

Figure 10 shows the selection procedure, indicating the major steps for evaluation and selection of each alternative. The first step is to define potential tailings disposal sites and methods based on site characteristics such as valley, open pit, underground workings or availability of a water body for tailings storage. For each option, a cost estimate, and an analysis of chemical and physical stabilities are conducted. The rank of land disturbance for each option is set as default (in the order of tailings impoundment > open pit filling > submarine disposal > underground backfill). Based on evaluation results and the assigned weights, these options are scored and ranked to generate a preferred alternative with highest overall rank. The weights can be obtained from the user according to their preference for the evaluation criteria, or they can be set as default values.

The reclamation methods examined in the module are:

- Revegetation (old tailings site and new TSF)
- Dry cover
- Water treatment plant
- Passive treatment (wetland)
- Dam flooding
- Open pit flooding

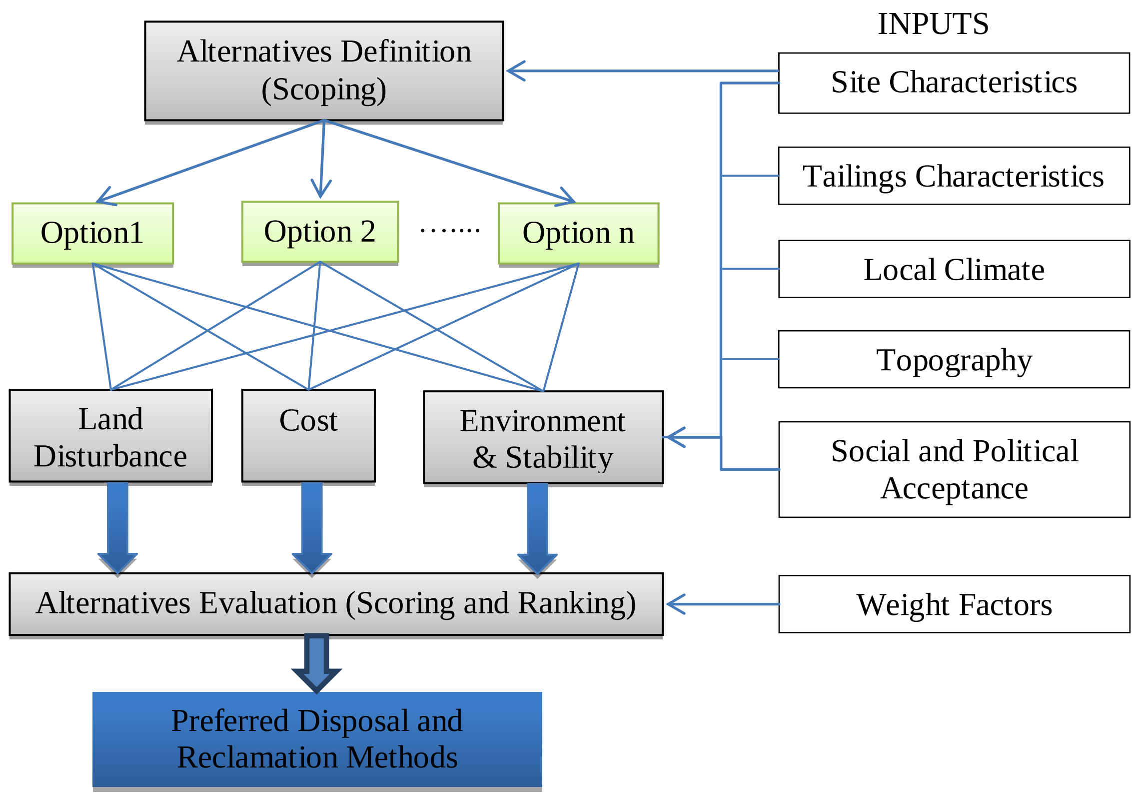


Figure 10. Procedure of Tailings Disposal and Reclamation Selection Module

3.6.2. Fuzzy Selection Module

3.6.2.1. Chemical and Physical Stability Sub-Module

The Environment and Stability Sub-Module is designed to evaluate the chemical and physical stability of the new TSF with its associated reclamation plan. This module is based on the study of Balcita (2001), which used a fuzzy expert system (ARDx) to choose ARD remediation methods. Selection is based on the possibility of success for ARD remediation and the cost of each scenario. The module is adjusted for copper tailings reprocessing and re-disposal. First, the options without ARD remediation are evaluated for cases in which ARD is not a considerable issue. Inputs are obtained from user questionnaires using quantitative and qualitative questions. Then, using specifically designed fuzzy sets, all inputs are fuzzified into a series of DoBs.

Several qualitative inputs (climate impact, contaminant reactivity, and contaminant level) are determined by different means. For example, the overall contaminant level is high when the level of ANY contaminant is high, and is low when the levels of ALL the contaminants are low.

The fuzzified inputs are mapped into designed evaluation rules to generate a result for each option. Both FAM maps and Weighted-Inference Methods are employed as rules to process collected data. Finally by evaluating site characteristics, tailings quality, and failure risk, the sub-module generates outputs in the form of DoBs in the possible success of each option. Figure 11 shows the flowsheet of the Environment and Stability Performance Evaluation.

3.6.2.2. Cost Sub-Module

A cost sub-module was developed to estimate capital and operating costs, and to provide criteria for selecting an alternative. Capital cost of is calculated first by summarizing itemized costs, with the operating cost being estimated as a portion of capital cost (Balcita, 2001). All costs are up-dated according to the Marshall & Swift Index (2011). Based on capital and operating costs, the Net Present Value (NPV) is also calculated as selection criteria. All details of the cost estimate data can be found in Table A-3 (Appendix A).

3.6.2.3. Main Sub-Module

The Main sub-module combines the land disturbance rank and results generated by the Environment and Stability sub-module and the Cost sub-module in order to provide a recommendation on the closure plan. This sub-module determines the final option overall rating obtained by the weighted equation below:

$$\text{Overall Rating} = (T*W_T + LD*W_{LD} + Cc*W_{Cc} + NPV*W_{NPV}) / (W_T + W_{LD} + W_{Cc} + W_{NPV})$$

where,

T, LD, Cc, NPV – Technical, land disturbance, capital cost and net present value (NPV) rankings;

W_T, W_{LD}, W_{Cc}, W_{NPV} – Technical, land disturbance, capital cost and net present value (NPV) weights of importance.

The weights can be obtained from user questionnaire or set as default values (1.0, 0.2, 0.5, 0.5 for W_T, W_{LD}, W_{Cc}, W_{NPV} respectively), which are examined by hypothetic cases and proven capable of generating proper and reasonable results as expected. The user may adjust these weights to suit a unique situation.

The selected option associated with its possibility of success (DoB), and its capital and operating costs are exported as module output to be input to the economic analysis and final decision-making modules.

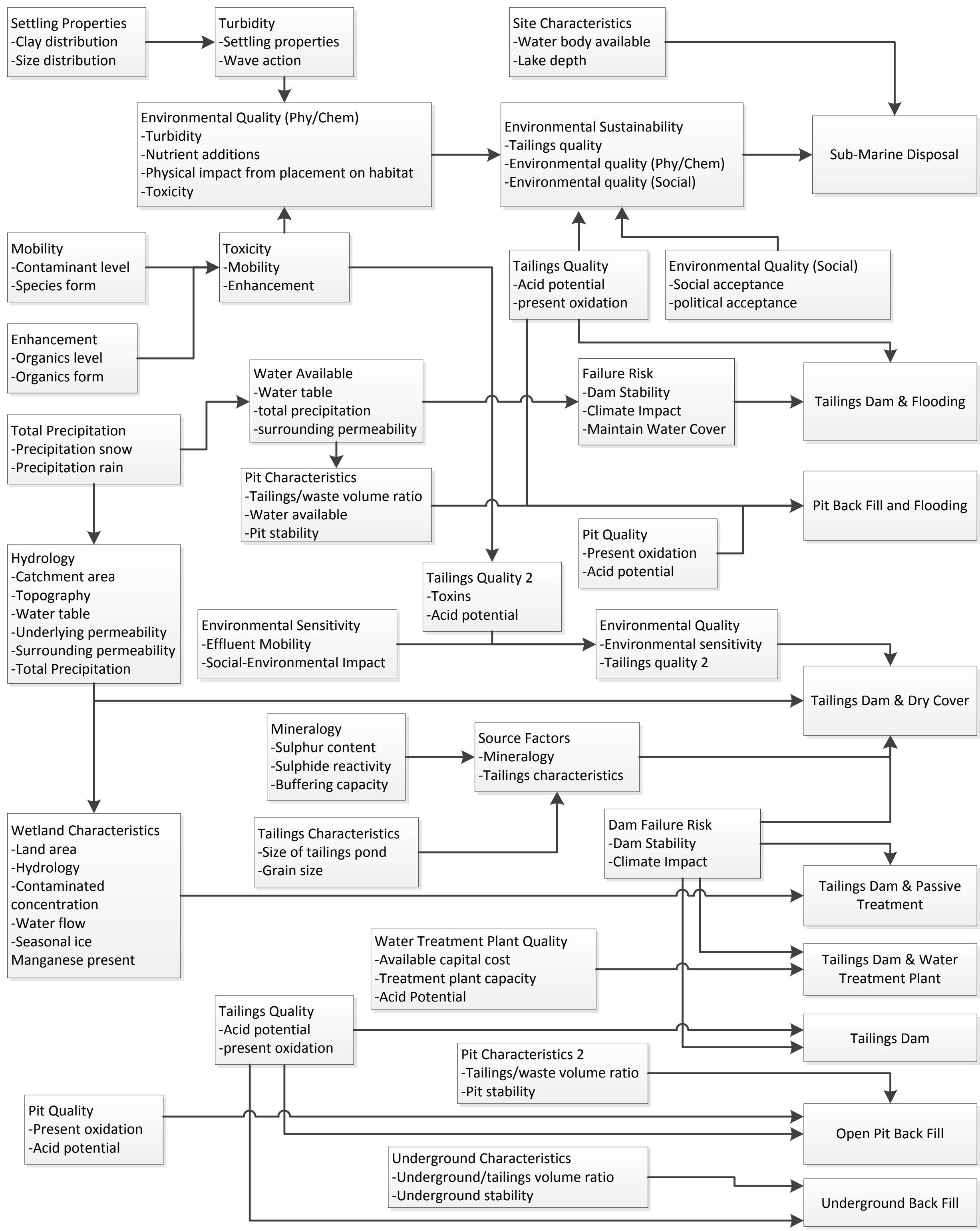


Figure 11. Flowsheet of the Environment and Stability Evaluation Module (modified after Balcita, 2001)

3.7. Risk Assessment Module

3.7.1. Module Overview

During various stages of an engineering project, appropriate risk assessment and management play a crucial role in a successful engineering project, especially at the decision-making stage. Risk assessment determines the degree of risk related to a failure mode which refers to the manner in which a failure can occur (Stamatis, 2003). Normally, risk is measured by the mathematical product of consequence probability and damage extent of an unwanted event (Laurence, 2001).

The risk assessment module is designed to evaluate the environmental and social performances before tailings retreatment depending on which final decision is made. A checklist is prepared for the user to simplify the complexities of the risk assessment process, and to ensure that no critical factor is overlooked. Then, a systematic analytical approach is applied to produce a quantitative risk estimate by combining all components through weighting and prorating.

Table 6 is the check list for risk assessment indicating the hierarchical framework used. The primary issues shown in the first layer are composed of six components that are further segmented into more detailed issues as listed in the secondary and tertiary layers. The default weights used in the system for a tailings dam with major environmental issues can be seen in Table 6 for all primary, secondary, and tertiary issues.

Laurence (2001) developed the concept of Closure Risk Factor (C_{RF}) to produce a comprehensive and quantitative measure including various significant mine closure concerns. The C_{RF} allows these concerns to be broken down into a range of issues that can be classified into Environmental Risks (R_E), Safety and Health Risks (R_{SH}), Final Land Use Risks (R_{LU}), Community and Social Risks (R_C), Legal and Financial Risks (R_{LF}) and Technical Risks (R_T). As expressed in the equation below, the risk factor is the sum of these risk factors.

$$C_{FR} = \sum(R_E + R_{SH} + R_{LU} + R_C + R_{LF} + R_T)$$

In the module developed in this work, the risks for each of the different issues are transformed into Degrees of Beliefs (DoB) to deal with the vagueness inherent in linguistic input. To combine the risks of all these failure modes, a Weighted Inference Method is employed to generate a final risk in the form of a DoB in the conclusion (Meech & Veiga, 1998). Basically, the user is asked to provide the important weights for each risk factor. When the user fails to provide this information, the system runs with default values tested by hypothetic cases and proven to be capable of generating proper and reasonable result as expected. This method is a more distributed one that that used by Lawrence and establishes the relative importance of each factor through the weights rather than assuming each of the different factors have equal weights.

Table 6. Structure of Risk Assessment (after Laurence (2001))

Primary Risk Issue	DoB	Wt.	Secondary Issues	DoB	Wt.	Tertiary Issues	DoB	Wt.	O[1]	C[2]
Environmental		0.4	Surface Water		0.3	Toxicity		0.5	7	2
						ARD		0.5	8	7
			Air		0.2	Dust		1	2	2
			Land System		0.1	Visual Amenity		0.2	6	8
						Infrastructure		0.1	3	3
						Soil Contamination		0.2	6	3
						Soil Erosion		0.2	2	3
						Flora Reestablish.		0.1	2	4
						Fauna Reestablish.		0.1	3	3
						Voids		0.1	3	4
			Tailings		0.4	Toxicity		0.3	7	3
						ARD		0.3	8	6
						Structure Failure		0.4	6	6
Safety and Health		0.1	Openings		0.2	Shafts, Raises, Winzes		0.5	2	2
						Open Pits		0.5	0	0
			Infrastructure		0.1	Buildings and Equip.		1	3	5
			Security		0.4	Theft		0.6	3	4
						Unauthorized Access		0.3	2	6
Final Land Use		0.2	-		1	Land Value		1	6	5
Community and Social		0.4	Employees		0.1	Provision for Entitlements		0.5	2	6
						Retraining/Relocation		0.5	3	3
			Management		0.1	Communication		0.5	4	3
						Safety Awareness		0.5	2	1
			Landowners		0.2	Indigenous People		1	8	8
			General Community Impact		0.6	Local		0.4	7	6
						Regional		0.3	7	5
						National		0.2	6	4
						International		0.1	5	3
Legal and Financial		0.1	-		1	Government		0.25	5	3
						Creditors		0.25	3	3
						Provisioning for Rehabilitation		0.25	3	3
						Adverse Publicity		0.25	4	3
Technical		0.1	-		1	Closure Plan		0.3	4	3
						Rehabilitation Process		0.3	4	3
						Closure Team		0.2	4	3
						Reserves/Resource		0.2	1	1

[1] Occurrence (Scale from 1 to 10 – see Table 7) [2] Consequence (Scale from 1 to 10 – see Table 8)

3.7.2. Fuzzy Risk Assessment

In consideration of a Failure Mode and Effect Analysis (FMEA) to estimate risk, this module follows the four major steps shown in Figure 11: Hazard Assessment (HA), Exposure Assessment (EA), Consequence Assessment (CA) and Risk Characterization (RC) (Van Zyl & Bamberg, 1992). However, instead of a traditional quantitative risk calculation, the module processing is conducted with fuzzy variables and fuzzy rules.

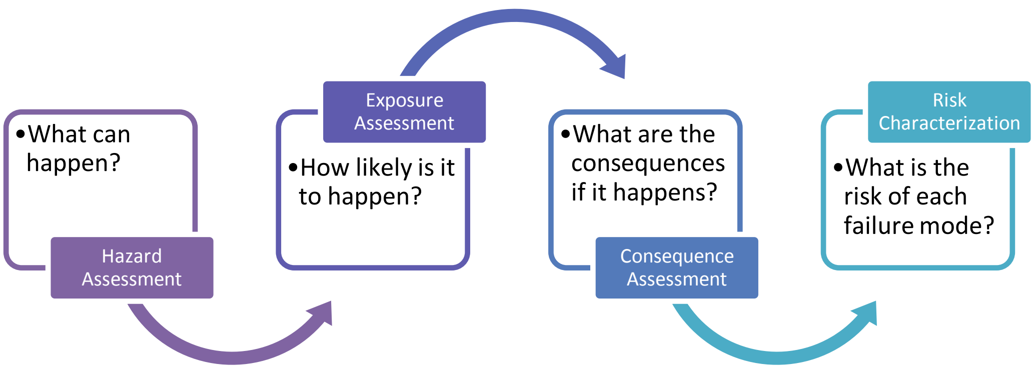


Figure 12. Risk Assessment Procedures

The first step is to identify all relevant unwanted events – failure modes. All the hazards are determined by a user (HA) by selecting potential failure modes from the checklist in Table 6 instead of by brainstorming and listing all potential failure modes, which is the normal process at the beginning of an FMEA. EA is then conducted to figure out the likelihood that an event will occur, i.e., the occurrence probability. In this module, probabilities are expressed by linguistic terms such as "very unlikely", "unlikely", "possible", "likely", and "almost certain". CA is a process to determine the consequences – the extent or degree of damage. The consequences are expressed by linguistic terms such as "low", "minor", "moderate", "major" and "critical". Both likelihood and consequence are on a scale ranging from 1 to 10, which represents qualities from low to high. Guidelines (Table 7 and Table 8) are provided as a reference for users to determine these two factors.

Table 7. Occurrence Probability Rating Scale (Stamatis, 2003)

Description	Rate	Definition
Almost Certain	**10**	> 1 occurrence per day / Probability of > 3 occurrences in 10 events
	9	One occurrence in 3-4 days / Probability of 3 occurrences in 10 events
Likely	**8**	One occurrence per week / Probability of 5 occurrences in 100 events
	7	One occurrence per month / Probability of 1 occurrence in 100 events
Possible	**6**	One occurrence in 3 months/ probability of 3 occurrences in 1,000 events
	5	One occurrence in 6 months / Probability of 1 occurrence in 10,000 events
Unlikely	**4**	One occurrence per year / Probability of 6 occurrences in 100,000 events
	3	One occurrence in 1-3 years / Probability of 6 occurrences in 10 million events
Very Unlikely	**2**	One occurrence in 3-5 years / Probability of 2 occurrences in 1 billion events
	1	One occurrence in > 5 years / Probability of < 2 occurrences in 1 billion events

Table 8. Consequence Rating Scale (McDermott, Mikulak, & Beauregard, 2009)

Description	Rate	Definition
Critical	**10**	Severe long-term harm on ecosystem function Significant permanent harm to traditional land use Major breach regulatory requirement (court order issued) > $10 million consequence cost Serious national/international adverse public, media or NGOs Irreversible impairment/disability or single fatality to more than one person
	9	
Major	**8**	Significant environmental impact and medium-term harm on ecosystem Remarkable temporary impact to traditional land use Breach regulatory requirement, permit or approvals (prosecution) $2.5 ~ $10 million consequence cost Significant local/regional/national adverse public, media or NGOs Irreversible impairment/disability or single fatality to more than one person
	7	
Moderate	**6**	Significant impact on environment Mitigatable influence to traditional land use Breach a regulatory requirement, permit or approval (order or directive) $500,000 ~ $2.5 million consequence cost Considerable concerns by local community, local/regional adverse press coverage and criticism from Non-Government Organizations (NGOs) Moderate irreversible impairment/disability to more than one people
	5	
Minor	**4**	Minor localized or short-term environmental impact Minor disturbance to traditional land use Lack of technical/administrative compliance (warning letter) $100,000 ~ $500,000 consequence cost Local complaints and local adverse press coverage Reversible impairment , disability or injuries requiring hospitalization
	3	
Low	**2**	No environmental impact Little disturbance to traditional land use < $ 100,000 consequence cost No local complaints or adverse press coverage No measurable physical effect or medical treatment on human health/safety
	1	

The numerical inputs are fuzzified into DoBs in various linguistic states using defined fuzzy sets. According to Braglia et al. (2003), triangular shaped fuzzy sets reduce computational complexity. The risk is calculated for each identified failure mode, depending on the occurrence and consequence in the risk matrix shown in Table 9. The risk of each failure mode is classified into five different levels: low, moderate, moderately high, high and very high.

Table 9. Risk Determination and Assessment Matrix

Risk		CONSEQUENCE →				
		Low	**Minor**	**Moderate**	**Major**	**Critical**
OCCURRENCE ↑	**Almost Certain**	**Moderate**	**Mod-High**	**High**	**Very High**	**Very High**
	Likely	**Moderate**	**Moderate**	**Mod-High**	**High**	**Very High**
	Possible	**Low**	**Moderate**	**Mod-High**	**High**	**High**
	Unlikely	**Low**	**Low**	**Moderate**	**Mod-High**	**Mod-High**
	Very Unlikely	**Low**	**Low**	**Low**	**Moderate**	**Mod-High**

The "min-max" approach is applied to calculate the rule conclusions based on the given inputs. Weighted average defuzzification is used to obtain a risk for each failure mode. To combine risks for all the failure modes according to the hierarchical system shown in Table 6, the Weighted Inference Method is employed to give an overall risk value in the form of a DoB which is an important and easy-to-understand criterion for final decision-making.

3.8. Decision-Making Module

The Decision-Making Module is designed to analyse and combine the criteria generated by the other three modules, and to provide a final recommendation on whether or not to retreat the tailings. The criteria considered are: economic performance (NPV@5% - the output of the

Design and Economic Module), risk assessment (final risk - the output from the Risk Assessment Module), and Environment Performance (possibility of success - the output of the Disposal and Reclamation Module). The decision-making uses the 3-D FAM maps shown in Table 10.

Table 10. Rule-Base Matrix for Recommendation on Tailings Retreatment

Current Risk is LOW	**Economic Performance**			
Environmental Performance	**Negative**	**Low**	**Medium**	**High**
Low	NR	NR	NR	MR
Moderate	NR	NR	MR	HR
High	NR	MR	HR	HR
Current Risk is MEDIUM	**Economic Performance**			
Environmental Performance	**Negative**	**Low**	**Medium**	**High**
Low	NR	NR	MR	HR
Moderate	NR	MR	HR	HR
High	MR	HR	HR	HR
Current Risk is HIGH	**Economic Performance**			
Environmental Performance	**Negative**	**Low**	**Medium**	**High**
Low	NR	MR	HR	HR
Moderate	MR	HR	HR	HR
High	MR	HR	HR	HR

where NR, MR, and HR represent respectively Not-Recommended, Medially-Recommended, and Highly-Recommended.

Based on the evaluated current environmental and social risk, different rules are defined. If the current risk is high, tailings retreatment can be an opportunity to reduce long-term liability by eliminating the possible tremendous potential cost of serious pollution and/or dam failure. Moreover, the revenue generated from copper extraction can offset or reduce future reclamation costs. Therefore, in this situation, tailings retreatment can be recommended even when the economic profit from the extraction of copper alone is unattractive. When the current risk is medium, economic profit together with environmental and social benefits are equally important. In this case, tailings retreatment is recommended when an acceptable economic performance is achieved and the environmental and social performance is improved significantly from tailings retreatment. If the current risk is low, economic benefits become the major target. As a result, tailings retreatment is considered only if an attractive profit from copper production can be realized. After these three criteria enter into their respective rules, output in the form of a DoB about the particular option is provided as a recommendation to the user.

Chapter 4. Validation and Verification

4.1 System Validation and Verification

Model validation and verification have been conducted to confirm that, based on the same information, the system reaches a similar conclusion to that of an expert. This process also verifies that the system is working properly and efficiently on given inputs over a range and combination of values. Four hypothetical cases with different conditions have been processed to validate and verify the retreatment model:

Mine A – Large scale copper tailings with high acid generation potential and an operating WTP
Mine B – Large scale copper tailings with attractive Cu grade and medium ARD problem
Mine C – Medium scale copper tailings with a dust problem
Mine D – Small scale copper tailings with attractive Cu grade and low environment impact

The important input factors for each case are listed in Table 11:

Table 11. Important Information of the Four Cases

Element	MINE A	MINE B	MINE C	MINE D
Scale	Large	Large	Medium	Small
Head Grade (%Cu)	0.12	0.18	0.12	0.18
%A.S. Cu	0.02	0.02	0.05	0.08
Distance to Community	Medium	Close to	Medium	Remote Area
Dust Problem	No	No	Yes	Yes
Dam Stability	Poor	Medium	Poor	OK
P_5 [1]	5%	1%	5%	0.1%
n_{100} [2]	30	100	50	1000
Consequence[3]	\$50 M	\$5 M	\$5 M	\$0.5 M
Effluent pH	>7	3-4.5	5-6	5-6
Buffering Capacity	Low	Medium	High	High
Metal Contaminants	Low	Medium	Low	Low
Land Use Value	High	Medium	Medium	Low
Remediation Project	WTP [4]	Non	None	Non
Impact on a Community	High	High	Medium	Low
Heap Leaching Results	Poor	Poor	None	None
Tank Leaching Results	None	None	None	None
Flotation Results	Good	Good	None	None

[1] failure probability in first five years [2] certain failure year [3] failure economic cost (see Table 5) [4] water treatment plant

Table 12 presents the overall model results. The final recommendation on tailings retreatment is based on the estimated economic and environment benefits that can be achieved from retreatment. The detailed input and output information can be found in Appendix D.

Table 12. Comparison of Overall Model Running and Expected Results

Model Results				
Cases	Economic Benefit (NPV@5%)	Current Environmental and Social Risk Factor (DoB)	Environmental Performance (DoB)	Retreatment Recommendation (DoB)
Mine A	135,534,000	80	56	95
Mine B	177,315,000	67	88	95
Mine C	-13,135,000	66	85	24
Mine D	10,808,000	18	84	43
Linguistic Results				
Cases	Economic Benefit	Current Environmental and Social Risk	Environmental Performance	Retreatment Recommendation
Mine A	Medium	High	Medium	Yes
Mine B	Very-Good	Medium to High	High	Yes
Mine C	Negative	Medium to High	High	No
Mine D	Low	Low	High	Maybe

The model appears to provide reasonable answers. Based on these case studies, the model seems to be an effective decision-making tool. With Mine A, the environmental performance is considerably improved, and at the same time, a highly acceptable economic benefit is generated, so retreatment is highly recommended. For Mine B, the very-high profits make the tailings retreatment project an outstanding investment opportunity. For Mine C, since the revenue is not able to compensate for the re-disposal and reclamation costs, it is not economic to re-treat the tailings but to minimize or avoid future liabilities, retreatment may be a useful decision. With Mine D, although it is profitable to retreat the tailings, because of the low current environmental and social risk, it appears to be unattractive to retreat the tailings as with Mine A and B.

Table 13 shows the results of the Design and Economic Module for the four scenarios, including the re-mining method and the re-processing flowsheet, the mineral processing performance, and the economic performance of the investment. This output provides an understanding of the impact of the following variables: mineral processing and metallurgical test results, and feed grade on processing flowsheet selection, abundant water available on mining method selection, as well as head grade, project scale, and base case NPV on economic performance.

Table 13. Results Summary of Design and Economic Module

	Mine A	**Mine B**	**Mine C**	**Mine D**
Mining Method	Hydraulic Mining	Hydraulic Mining	Mechanical Excavation	Mechanical Excavation
Processing Flowsheet	Flotation & Conc. Leaching	Flotation	Heap Leaching Bulk Material	Existing Plant (Flotation)
Mine Life (years)	18.5	11.5	20	8.2
Capacity (tpd)	29,600	35,700	11,000	10,000
Capital Cost ($US)	149,337,000	144,720,000	66,227,000	36,721,000
Operating Cost ($US/yr)	26,905,975	40,515,000	16,790,000	15,928,000
Revenue ($US/yr)	76,428,000	127,594,000	28,347,000	34,726,000
Project NPV@5% ($US)	97,200,000	175,463,000	-15,033,000	10,620,000
DCFROR (%)	12.2	23.5	2.1	11.6
Base Case PV@5% ($US)	-38,334,000	-1,852,000	-1,898,000	-188,000
Total NPV@5% ($US)	135,534,000	177,315,000	-13,135,000	10,808,000
Total Recovery (%)	76	85	76	83
Concentrate Production (tpd)	0	237	0	68
Concentrate Grade (%)	-	23	-	22
SX-EW Cu Production (tpd)	27	0	10	0

When water is available, hydraulic mining is chosen (Mine A and B). When heap leaching is selected, mechanical excavation becomes the preferred way to reclaim the tailings (Mine C). For low grade tailings, heap leaching, with its lower capital and operating costs, is more likely to be selected to yield a better economic performance (Mine C). However, if heap leaching is ineffective, flotation is preferred (Mine A and B). Note that with Mine A, because of the low grade concentrate, leaching of the flotation concentrate is a preferred choice. Generally, a higher feed copper grade leads to a higher benefit, i.e., the NPV(@5% for Mine B is much higher than that for Mine A. However, the economic results are also due to the project scale (daily tailings tonnage). For the Base Case PV for Mine A, significant improvement results from savings in current remediation costs together with the estimated potential future dam failure.

The Disposal and Reclamation Module outputs are summarized in Table 14. When ARD is a major issue (Mine A and B), based on site characteristics (open pit or underground workings available for disposal) the tailings disposal option with the lowest acid generating potential (i.e., backfill) is selected. When these options do not exist and ARD is not problematic, then the traditional disposal method – surface impoundment – (Mine D) is necessary. With Mine A, in order to deal with an acid generation problem, the reclamation option of flooding is added to improve the environmental performance.

Table 14. Summary of Disposal and Reclamation (D&R) Module Results.

	Mine A	**Mine B**	**Mine C**	**Mine D**
Disposal Method	Pit Backfilling	Underground Backfilling	None	Tailings Impoundment
Reclamation Method	Flooding	None	Revegetation	Revegetation
D&R Capital Cost ($US)	27,181,000	28,336,000	3,210,000	2,700,000
D&R Operating Cost ($US/year)	3,702,000	3,922,000	481,500	405,000
D&R Cap. Costs NPV@5% ($US)*	-63,146,000	-46,138,000	-6,632,000	-4,619,000
Environmental Performance (DoB)	56	88	85	84

* These costs are included within the Total NPV@5% shown in Table 13.

Table 15 summarizes the module results for the Risk Assessment module. Variations in the DoB in "high" environmental and social risk indicates the effects of various environmental issues, ARD problem severity, social pressures, and so on.

Table 15. Summary of the Risk Assessment Module Results.

Risk Factor (DoB)	**Mine A**	**Mine B**	**Mine C**	**Mine D**
Environmental	57	57	54	28
Safety & Health	33	33	33	20
Final Land Use	90	55	55	29
Community and Social	71	60	60	8
Legal and Financial	38	34	32	6
Technical	31	28	24	9
Combined Risk Factor	80	67	66	18

4.2. Sensitivity Analysis

4.2.1. Copper Price, Head Grade, and Capital/Operating Costs

The economic sensitivity analyses for each example are shown in Appendix D. The results are summarized below:

Table 16. Percent Change in Input Variables to Obtain a Zero NPV @5%

MINE	**Cu Head Grade/Price**	**Cap / Op Costs**
A	-25%	+40%
B	-28%	>+50%*
C	+10%	-15%
D	-8%	+20%

* Actually gives an NP@5% that is $100M.

The results for Mine A and Mine B show that even with a large change in the input levels the NPV@5% remains positive. So the recommendations for these cases are highly stable. With Mine C, if the head grade or copper price increases by only 10%, or the capital or operating costs decrease by only 15%, the NPV@5% will become positive. In this case, the recommendation might change from NO to YES with a relatively small change in the key inputs. With Mine D, if the head grade or copper price drops by 8% or less, or the capital or operating costs increase by 20% or less, the NPV@5% remains positive. So, this recommendation can be considered reasonably stable.

4.2.2. Failure Cost

Since the failure cost estimate is a complicated process to determine, it may contain significant uncertainty due to overlooked factors. So, a sensitivity analysis has been conducted on failure cost. According to the analysis results shown in Figure 13, the economic performance is only sensitive to future failure costs when the amount is relatively large (Mine A). Otherwise, the model results do not change significantly with the input (Mine B, Mine C and Mine D). Generally speaking, in the case of high failure probability and loss, the economic performance of tailings retreatment will be considerably improved by avoiding future failure costs.

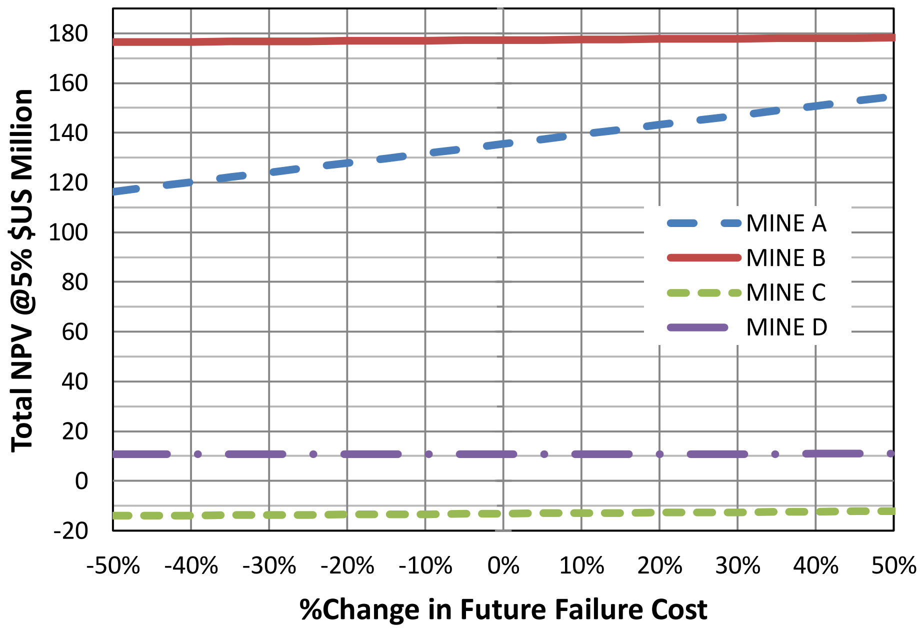


Figure 13. Economic Sensitivity Analysis on Future Failure Cost

4.2.3. Metallurgical Performance

A sensitivity analysis was also conducted on the processing metallurgical performance to better understand the impact of uncertainty of the tailings reprocessing recovery and concentrate grade.

As indicated in Figure 14, with respect to %recovery in each case, the NPV@5% remains positive for the recovery levels and higher ones shown in Table 17.

Table 17. Minimum %Recovery Required to Maintain NPV@5% above Zero.

Mine	Case Study %Recovery	Minimum %Recovery to yield a positive NPV@5%
A	76	52
B	85	58
C	76	87
C	83	76

So, for Mine A and Mine B, if %recovery levels above 60% can be maintained then the economic situation is satisfactory and the recommendation to retreat remains YES. For Mine C, the recovery must be 87% to generate a positive NPV @5%. This is very unlikely to be achieved, so the recommendation to retreat is clearly NO. For Mine D, the minimum recovery must be 76% or higher which may also be problematic, so the recommendation of MAYBE is confirmed with an increased leaning towards NO – the input level of 83% is highly optimistic.

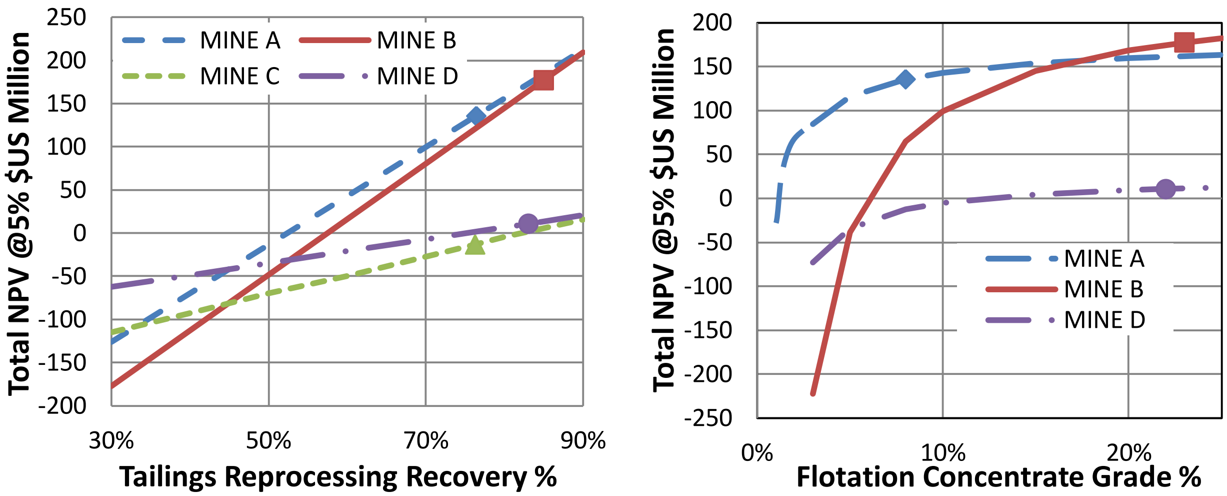


Figure 14. Economic Sensitivity Analysis on Reprocessing Recovery and Concentrate Grade.

Figure 14 also shows the sensitivity analysis for concentrate grade for each case study in which a flotation concentrate is produced. In the case of MINE A, the concentrate is leached to convert the copper to a high purity. The minimum acceptable concentrate grade for which the NPV@5% remains positive is about 1.4 %Cu which is likely feasible for most tailings materials that undergo retreatment. For Mine B and Mine D, assuming the smelter will treat lower grade concentrates, the minimum level is 7%Cu for Mine B and 13%Cu for Mine D. In both cases, it might be preferred to switch the flowsheet to one in which the concentrate is leached. This will increase capital and operating costs but may be less than the transportation and smelting charges for such low grade materials.

Chapter 5. Conclusions and Recommendations

A generic system to assist in the decision-making about possible retreatment of copper tailings materials has been developed to evaluate the economic and environmental benefits of retreatment and to provide a recommendation. Fuzzy logic and fuzzy-neural equations are employed in the model to deal with either measured or linguistic inputs, and to simulate the human decision-making process. Practical technologies that can be applied to tailings retreatment were reviewed, significant factors contributing to the analysis were recognized, and a logical and systematic evaluation structure was created. Based on this work, three modules (Design and Economic, Disposal and Reclamation, Risk Assessment) were established to involve and process all factors; to conduct a conceptual design; and to examine the economic and environmental performance of retreatment options. Depending on the retreatment performance, a final decision-making module provides a user with a recommendation to re-treat tailings or not.

The model results indicate that the tailing retreatment system is an effective tool to assist in preliminary decision-making. The system works as expected in an efficient manner on inputs over a range and combination of values.

The retreatment model can be applied to the following tasks:

- Complete a conceptual process design based on tailings properties, mineral processing and metallurgical test results, market conditions, and so on;
- Estimate capital and operating costs and conduct a preliminary economic analysis;
- Select a disposal and reclamation option for reprocessed tailings according to tailings and site properties;

- Provide a comprehensive approach to evaluate environmental and social risks of an existing tailings site;
- Estimate environmental performance of a retreatment project after tailings re-disposal and reclamation;
- Assist with multi-criteria decision-making based on an analysis of the economic and environmental benefits of retreatment.

There are some limitations to the tailings retreatment model in its present form. It is recommended that the following additional work be done in the future:

- The criteria to select a mining method are not fully applied in the model. Besides water requirements for hydraulic mining, other tailings characteristics such as particle size are important in choosing a mining method especially if heap leaching is being considered.
- The conceptual design of copper extraction processes is limited to eleven flowsheets. There are other alternatives that depend on tailings properties such as classifying tailings into coarse and fine fractions and processing them separately. A significant improvement in the model would derive from developing rules for unit operation selection depending on tailings properties and to generate many more combinations and configurations.
- Not all practical reclamation technologies are employed and tested in the model. For example, desulphidization and hazard waste segregation are also effective ARD remediation approaches for which rules can be established to evaluate their performance.
- The model does not provide a friendly means for a user to modify internal parameters, fuzzy set definitions, and fuzzy rules which could be useful in the future.
- Practical cases of copper tailings retreatment should be tested with the model to extend the system validation and examine the ability to assist in a real decision-making case.

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Appendix A - Cost Calculation Data

Table A-1. Quoted Case for Capital Cost Estimate of Each Unit Operation

Unit Operation	Name and Location	Capacity	Capital Cost ($US)	Year	Reference
Hydraulic Mining	Titanium Mine, Australia	12,000 tpd	9,600,000	1985	Hartman et al., 1992
Mechanical Excavation	Gold Tailings, US	10,000 tpd	14,500,000	1980	Hartman et al., 1992
Heap Leaching	Copper tailings, Chile	16 tpd Cu 10,000 tpd ore	20,500,000	1993	Brombacher, Bachofen, & Brandl, 1997
Agglomeration	-	-	10% of heap leaching capital cost		
Tank Leaching	The Mt. Gordon process	50,000 tpy Cu 2,100 tpd concentrate	75,000,000	2004	Peacey, Guo, & Robles, 2004
Processing Plant	Copper Mountain, Canada	35,000 tpd	233,525,000	2008	Chance et al., 2009
Grinding	Copper Mountain, Canada	35,000 tpd	145,000,000	2008	Chance et al., 2009
Flotation (rougher and cleaner)	Copper Mountain, Canada	35,000 tpd	75,000,000	2008	Chance et al., 2009
Flotation (rougher only)	Copper Mountain, Canada	35,000 tpd	55,000,000	2008	Chance et al., 2009

Table A-2. Quoted Case for Operating Cost Estimate of Each Unit Operation

Operations	Capacity	Operating Cost	Year	Project Type	Reference
Mining	4,000 tpd	$US 1.25 /tonne ore	2009	Tailings re-mining	Lambert, Valliant, & Krutzelmann, 2009
Mechanical Excavation	10,000 tpd	$US 2.6 /m^3 ore	1980	Tailings re-mining	Hartman et al., 1992
Heap leaching	16 tpd Cu 10,000 tpd ore	$US 0.45/kg Cu	1993	Old tailings re-processing	Brombacher et al., 1997
Agglomeration		10% of heap leaching			
Leaching of flotation concentrate	50,000 tpy Cu 2,100 tpd concentrate	$US 0.080/lb Cu	2004	The Mt. Gordon process	Peacey, Guo, & Robles, 2004
SX/EW	50,000 tpy Cu	$US 0.098/lb Cu	2004		Peacey, Guo, & Robles, 2004
Processing Plant	35,000 tpd	$US 2.83/t ore	2008	Copper Mountain, Canada	Chance et al., 2009
Grinding	35,000 tpd	$US 0.85/t ore	2008	30% of total mill cost	Chance et al., 2009
Flotation (rougher and cleaner)	35,000 tpd	$US 1.42/t ore	2008	50% of total mill cost	Chance et al., 2009
Flotation (Rougher only)	35,000 tpd	$US 0.71/t ore	2008	25% of total mill cost	Chance et al., 2009

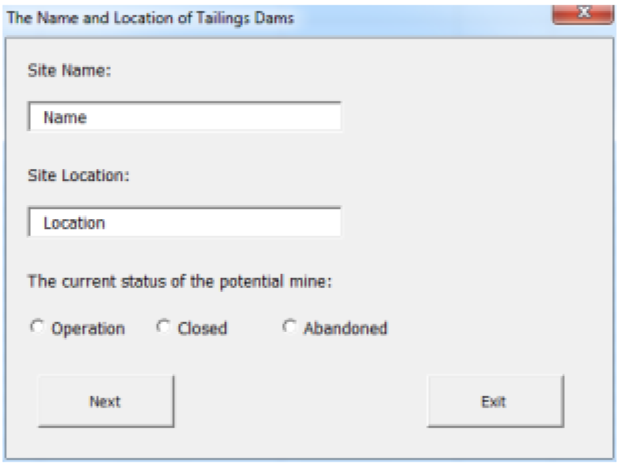
Table A-3. Tailings Disposal and Reclamation Capital Cost Data

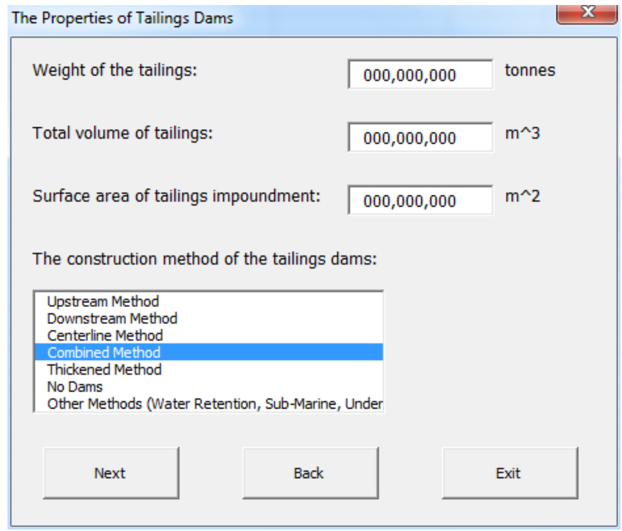
Tailings Surface Impoundment						**Reference**
Dam Construction	$/m^3$		3			Balcita, 2001
Dam Construction Others	%		16% of construction cost			
Underground Backfill						
Paste Cost	$/tonne		1.44			Newman, White, & Cadden, 2001
Open-pits Backfill and Flooding						
Haulage	$/tonne		1.5			Balcita, 2001
Haulage Other	%		30% of haulage cost			
Spill Way	$		100,000			
Tailings Flooding						
Dam Construction	$/m^3$		12			Balcita, 2001
Dam Construction Others	%		16% of construction cost			
Surface Regrade	$/m^3$		2			
Water Cover	$/m^2$		6			
Spill Way	$		100,000			
Submarine Disposal						
Haulage	Depth	Shallow	Moderate	Deep		Balcita, 2001
	$/tonne	5	3	1.5		
Haulage other	%		13% of haulage cost			
Haulage distance	$/m^3$*km		0.1			
Dry Cover						
Spill Way	$		70,000			Balcita, 2001
Regrade Slopes	$/m^3$		4			
Regrade Surface	$/m^3$		2			
Cover Surface	$/m^2$		26			
Cover Slope %	%		20			
Run Off Channel	$/m		40			
Water Treatment Plant						
	Feed Acidity	High	Mod	Low		Balcita, 2001
Plant Cost	h/ m^3	0.00264	0.00169	0.00164		
Other Cost	$	1144800	957100	715800		
Passive Treatment						
Wetland Construction	h/ m^3		0.00352			Balcita, 2001
Revegetation						
Revegetation	$/m^2$		2.1			Marcus, 1997

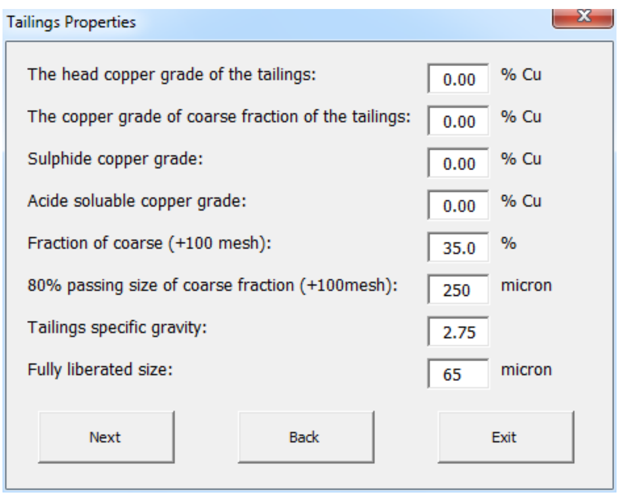
Table A-4. Processing Performance Assumptions

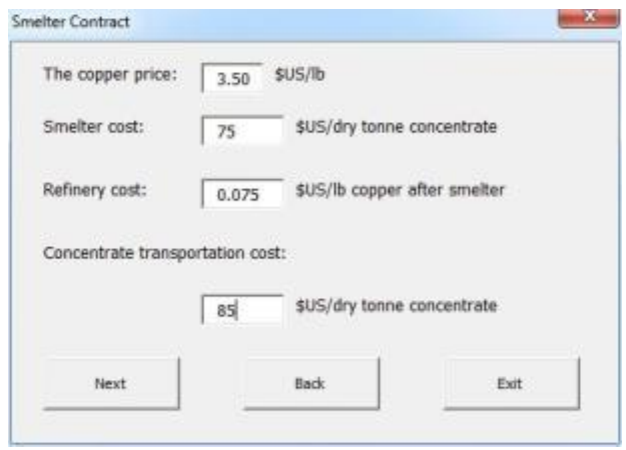
Unit Operations	Grade %	Recovery %	
	Cu	Cu (sulphide)	Cu (oxide)
Flotation (bulk)	20%	85%	10%
Flotation following grinding (coarse)	23%	87%	10%
Flotation (only rougher)	8%	90%	10%
Flotation concentrate leaching	99.99%	95%	97%
Flotation concentrate Heap leaching	99.99%	80%	85%
Floatation tailings heap leaching	99.99%	70%	85%
Heap leaching (bulk)	99.99%	70%	80%
Heap leaching with agglomeration	99.99%	75%	85%
Heap leaching of coarse	99.99%	75%	85%
Direct leaching	99.99%	30%	60%
Direct leaching with grinding (coarse)	99.99%	40%	70%
Acid leaching	99.99%	5%	95%
Floatation of leach residual	25%	87%	5%

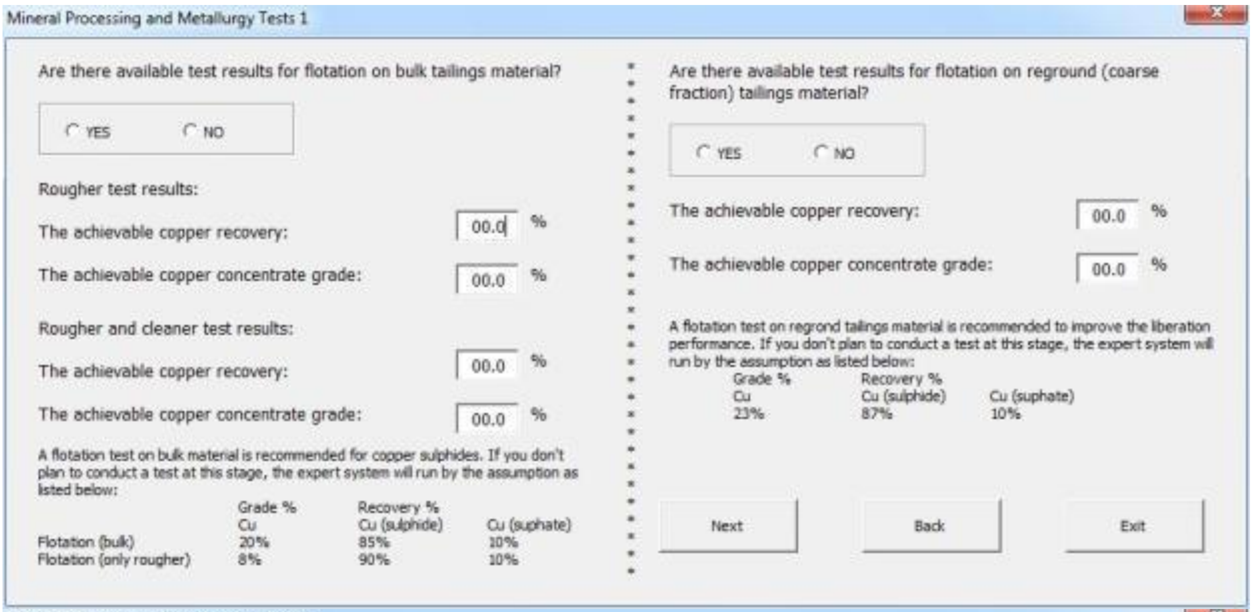
Appendix B - Questionnaires











Mineral Processing and Metallurgy Tests 1

Are there available test results for flotation on bulk tailings material?

YES NO

Rougher test results:

The achievable copper recovery: 00.0 %

The achievable copper concentrate grade: 00.0 %

Rougher and cleaner test results:

The achievable copper recovery: 00.0 %

The achievable copper concentrate grade: 00.0 %

A flotation test on bulk material is recommended for copper sulphides. If you don't plan to conduct a test at this stage, the expert system will run by the assumption as listed below:

	Grade % Cu	Recovery % Cu (sulphide)	Cu (suphate)
Flotation (bulk)	20%	85%	10%
Flotation (only rougher)	8%	90%	10%

Are there available test results for flotation on reground (coarse fraction) tailings material?

YES NO

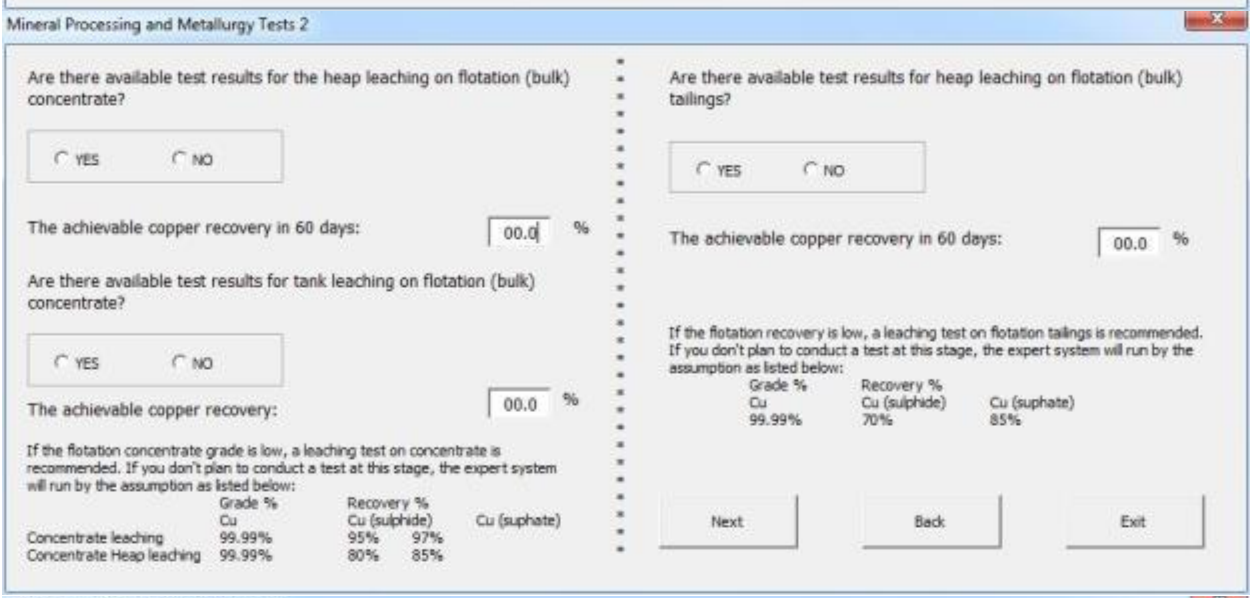
The achievable copper recovery: 00.0 %

The achievable copper concentrate grade: 00.0 %

A flotation test on regrond tailngs material is recommended to improve the liberation performance. If you don't plan to conduct a test at this stage, the expert system will run by the assumption as listed below:

Grade % Cu	Recovery % Cu (sulphide)	Cu (suphate)
23%	87%	10%

Next Back Exit



Mineral Processing and Metallurgy Tests 2

Are there available test results for the heap leaching on flotation (bulk) concentrate?

YES NO

The achievable copper recovery in 60 days: 00.0 %

Are there available test results for tank leaching on flotation (bulk) concentrate?

YES NO

The achievable copper recovery: 00.0 %

If the flotation concentrate grade is low, a leaching test on concentrate is recommended. If you don't plan to conduct a test at this stage, the expert system will run by the assumption as listed below:

	Grade % Cu	Recovery % Cu (sulphide)	Cu (suphate)
Concentrate leaching	99.99%	95%	97%
Concentrate Heap leaching	99.99%	80%	85%

Are there available test results for heap leaching on flotation (bulk) tailings?

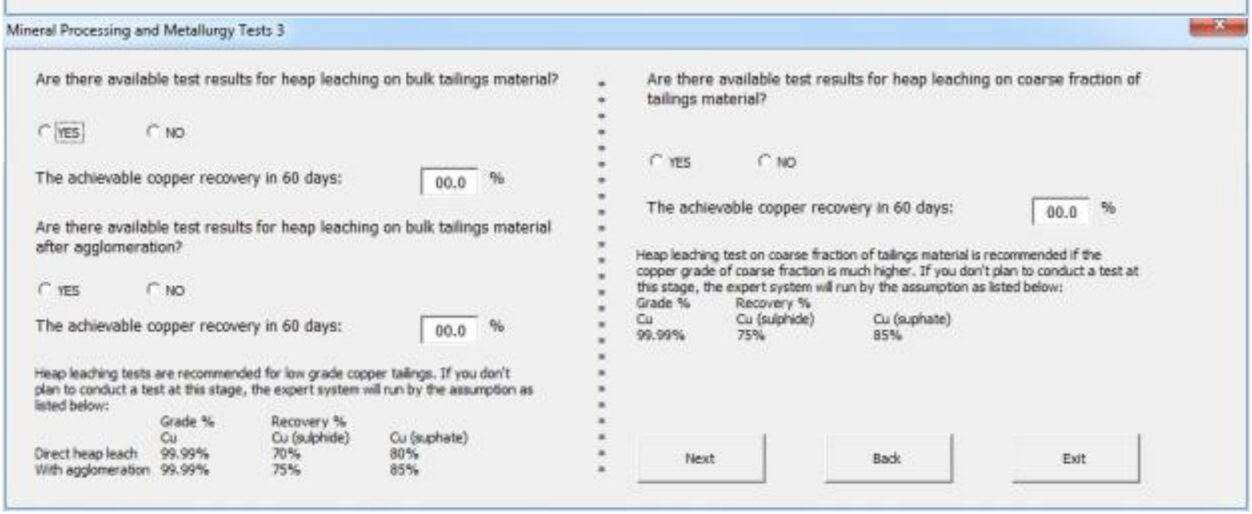
YES NO

The achievable copper recovery in 60 days: 00.0 %

If the flotation recovery is low, a leaching test on flotation tailngs is recommended. If you don't plan to conduct a test at this stage, the expert system will run by the assumption as listed below:

Grade % Cu	Recovery % Cu (sulphide)	Cu (suphate)
99.99%	70%	85%

Next Back Exit



Mineral Processing and Metallurgy Tests 3

Are there available test results for heap leaching on bulk tailings material?

YES NO

The achievable copper recovery in 60 days: 00.0 %

Are there available test results for heap leaching on bulk tailings material after agglomeration?

YES NO

The achievable copper recovery in 60 days: 00.0 %

Heap leaching tests are recommended for low grade copper tailings. If you don't plan to conduct a test at this stage, the expert system will run by the assumption as listed below:

	Grade % Cu	Recovery % Cu (sulphide)	Cu (suphate)
Direct heap leach	99.99%	70%	80%
With agglomeration	99.99%	75%	85%

Are there available test results for heap leaching on coarse fraction of tailings material?

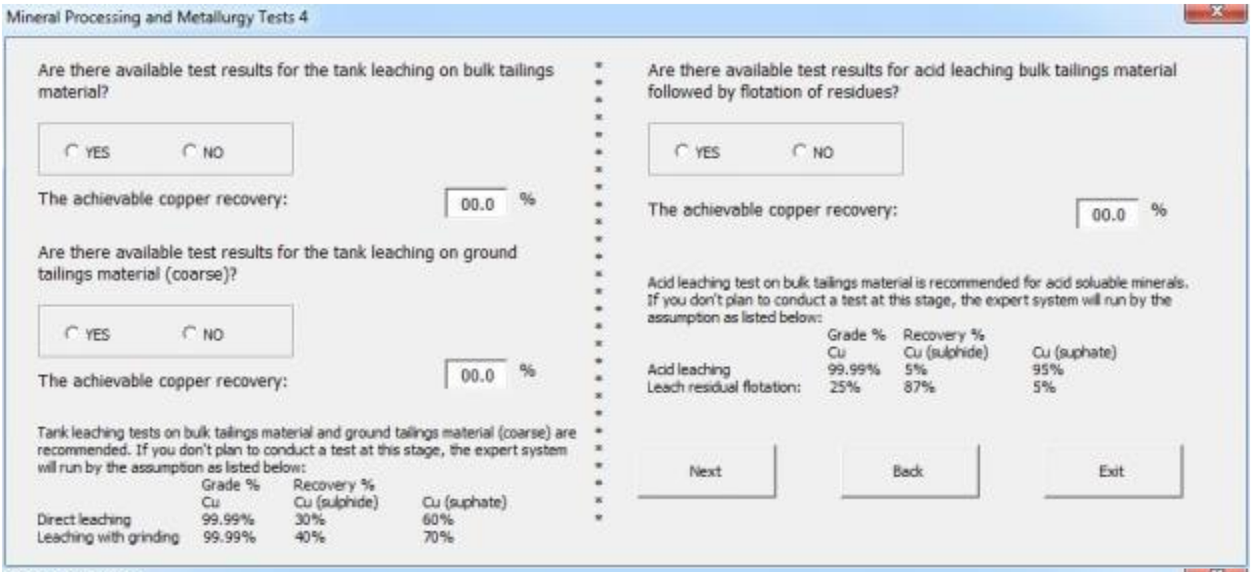
YES NO

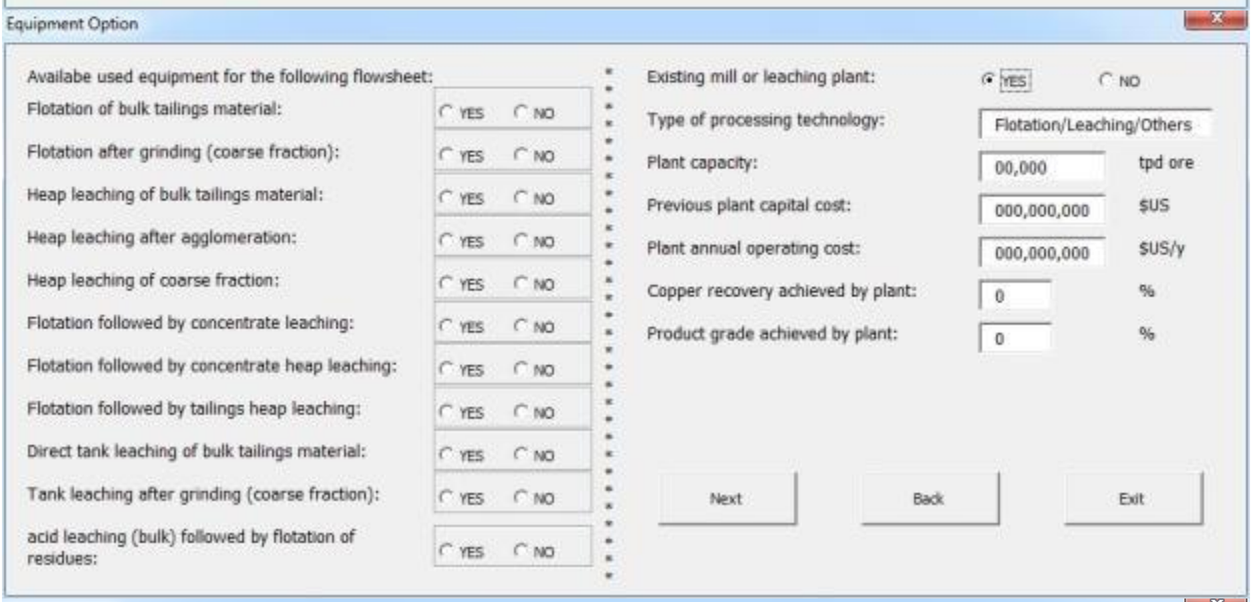
The achievable copper recovery in 60 days: 00.0 %

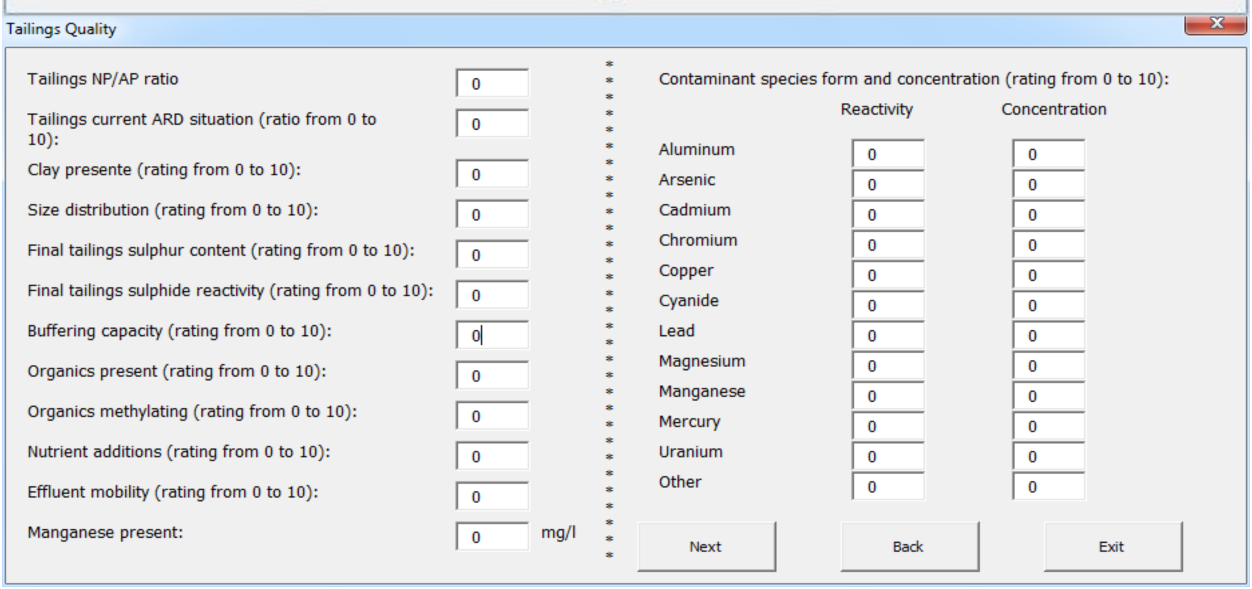
Heap leaching test on coarse fraction of tailngs material is recommended if the copper grade of coarse fraction is much higher. If you don't plan to conduct a test at this stage, the expert system will run by the assumption as listed below:

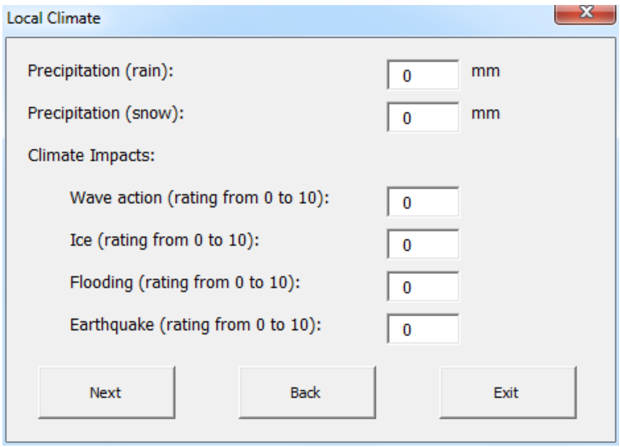
Grade % Cu	Recovery % Cu (sulphide)	Cu (suphate)
99.99%	75%	85%

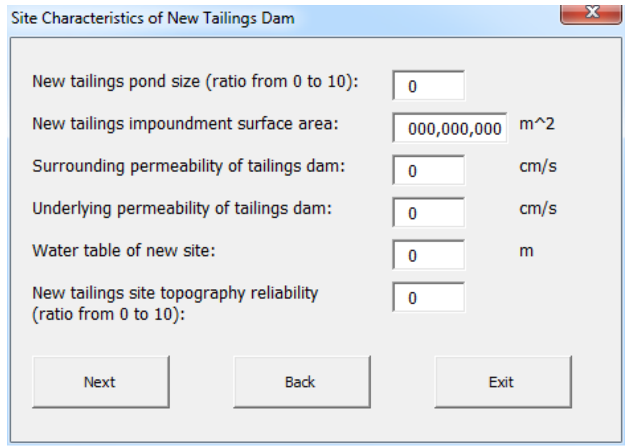
Next Back Exit

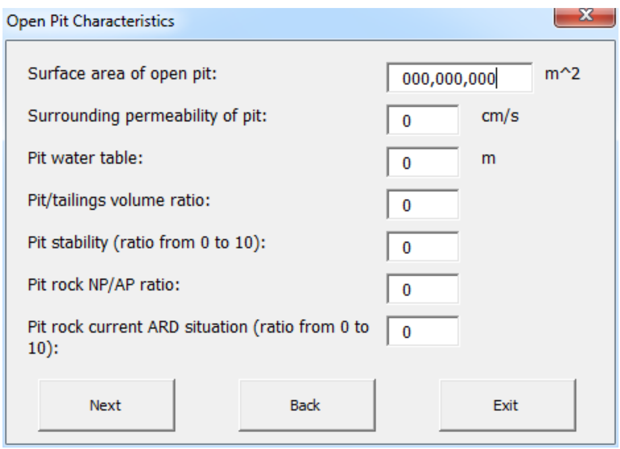

Mineral Processing and Metallurgy Tests 4
Are there available test results for the tank leaching on bulk tailings material?
YES NO
The achievable copper recovery: 00.0 %
Are there available test results for the tank leaching on ground tailings material (coarse)?
YES NO
The achievable copper recovery: 00.0 %
Tank leaching tests on bulk tailings material and ground tailings material (coarse) are recommended. If you don't plan to conduct a test at this stage, the expert system will run by the assumption as listed below:
Grade % Cu
Recovery % Cu (sulphide) Cu (suphate)
Direct leaching 99.99% 30% 60%
Leaching with grinding 99.99% 40% 70%
Are there available test results for acid leaching bulk tailings material followed by flotation of residues?
YES NO
The achievable copper recovery: 00.0 %
Acid leaching test on bulk tailings material is recommended for acid soluable minerals. If you don't plan to conduct a test at this stage, the expert system will run by the assumption as listed below:
Grade % Cu
Recovery % Cu (sulphide) Cu (suphate)
Acid leaching 99.99% 5% 95%
Leach residual flotation: 25% 87% 5%
Next
Back
Exit

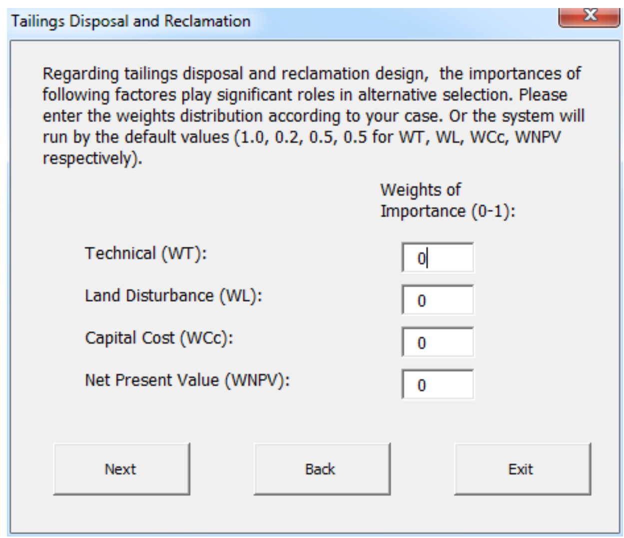

Equipment Option
Availabe used equipment for the following flowsheet:
Flotation of bulk tailings material: YES NO
Flotation after grinding (coarse fraction): YES NO
Heap leaching of bulk tailings material: YES NO
Heap leaching after agglomeration: YES NO
Heap leaching of coarse fraction: YES NO
Flotation followed by concentrate leaching: YES NO
Flotation followed by concentrate heap leaching: YES NO
Flotation followed by tailings heap leaching: YES NO
Direct tank leaching of bulk tailings material: YES NO
Tank leaching after grinding (coarse fraction): YES NO
acid leaching (bulk) followed by flotation of residues: YES NO
Existing mill or leaching plant: YES NO
Type of processing technology: Flotation/Leaching/Others
Plant capacity: 00,000 tpd ore
Previous plant capital cost: 000,000,000 $US
Plant annual operating cost: 000,000,000 $US/y
Copper recovery achieved by plant: 0 %
Product grade achieved by plant: 0 %
Next
Back
Exit

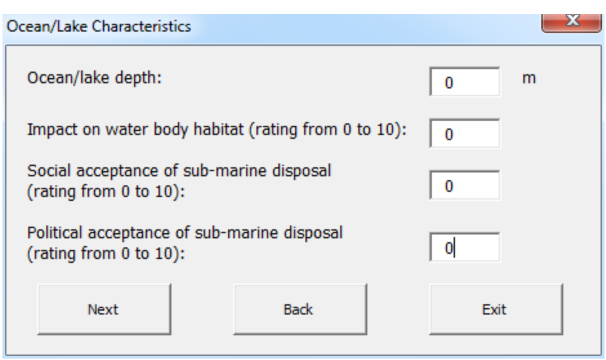

Tailings Quality
Tailings NP/AP ratio 0
Tailings current ARD situation (ratio from 0 to 10): 0
Clay presente (rating from 0 to 10): 0
Size distribution (rating from 0 to 10): 0
Final tailings sulphur content (rating from 0 to 10): 0
Final tailings sulphide reactivity (rating from 0 to 10): 0
Buffering capacity (rating from 0 to 10): 0
Organics present (rating from 0 to 10): 0
Organics methylating (rating from 0 to 10): 0
Nutrient additions (rating from 0 to 10): 0
Effluent mobility (rating from 0 to 10): 0
Manganese present: 0 mg/l
Contaminant species form and concentration (rating from 0 to 10):
Reactivity
Concentration
Aluminum 0 0
Arsenic 0 0
Cadmium 0 0
Chromium 0 0
Copper 0 0
Cyanide 0 0
Lead 0 0
Magnesium 0 0
Manganese 0 0
Mercury 0 0
Uranium 0 0
Other 0 0
Next
Back
Exit

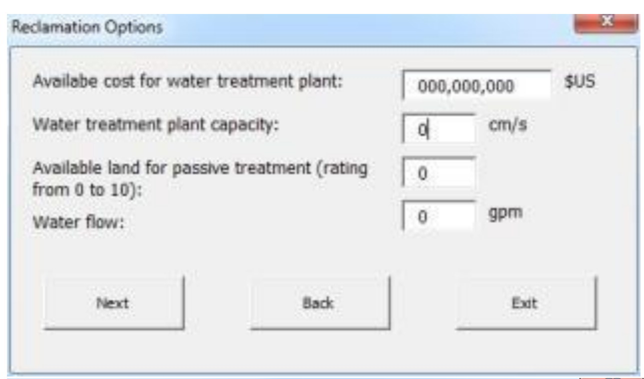

Local Climate
Precipitation (rain): 0 mm
Precipitation (snow): 0 mm
Climate Impacts:
Wave action (rating from 0 to 10): 0
Ice (rating from 0 to 10): 0
Flooding (rating from 0 to 10): 0
Earthquake (rating from 0 to 10): 0
Next
Back
Exit

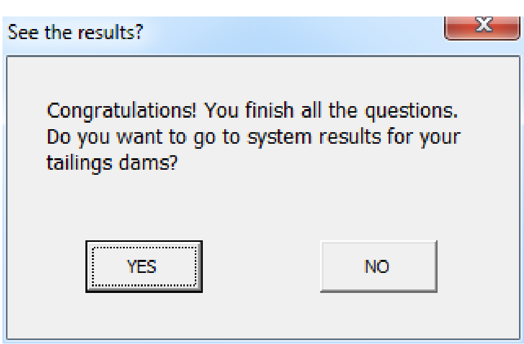

Site Characteristics of New Tailings Dam
New tailings pond size (ratio from 0 to 10): 0
New tailings impoundment surface area: 000,000,000 m^2
Surrounding permeability of tailings dam: 0 cm/s
Underlying permeability of tailings dam: 0 cm/s
Water table of new site: 0 m
New tailings site topography reliability (ratio from 0 to 10): 0
Next
Back
Exit

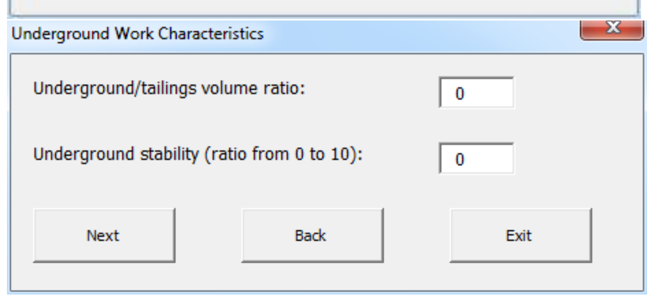

Open Pit Characteristics
Surface area of open pit: 000,000,000 m^2
Surrounding permeability of pit: 0 cm/s
Pit water table: 0 m
Pit/tailings volume ratio: 0
Pit stability (ratio from 0 to 10): 0
Pit rock NP/AP ratio: 0
Pit rock current ARD situation (ratio from 0 to 10): 0
Next
Back
Exit

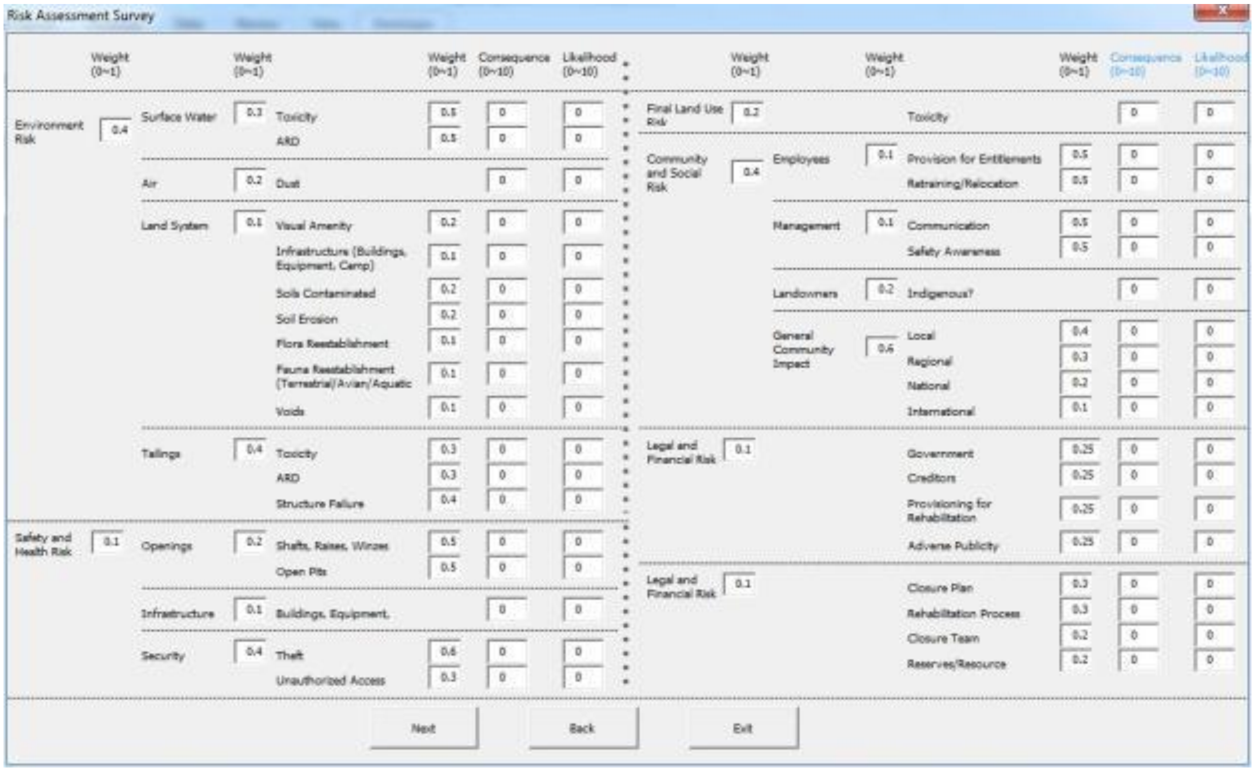

Tailings Disposal and Reclamation
Regarding tailings disposal and reclamation design, the importances of following factores play significant roles in alternative selection. Please enter the weights distribution according to your case. Or the system will run by the default values (1.0, 0.2, 0.5, 0.5 for WT, WL, WCc, WNPV respectively).
Weights of Importance (0-1):
Technical (WT): 0
Land Disturbance (WL): 0
Capital Cost (WCc): 0
Net Present Value (WNPV): 0
Next
Back
Exit


Ocean/Lake Characteristics
Ocean/lake depth: 0 m
Impact on water body habitat (rating from 0 to 10): 0
Social acceptance of sub-marine disposal (rating from 0 to 10): 0
Political acceptance of sub-marine disposal (rating from 0 to 10): 0
Next
Back
Exit


Reclamation Options
Availabe cost for water treatment plant: 000,000,000 $US
Water treatment plant capacity: 0 cm/s
Available land for passive treatment (rating from 0 to 10): 0
Water flow: 0 gpm
Next
Back
Exit

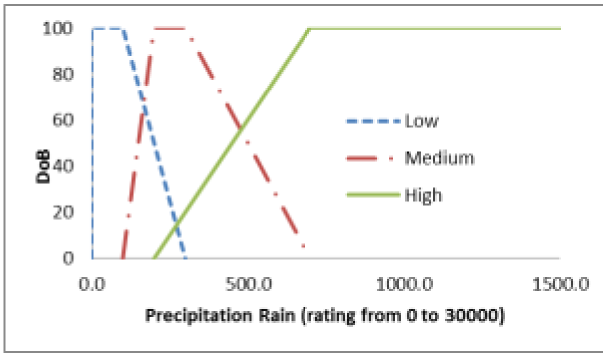

See the results?
Congratulations! You finish all the questions. Do you want to go to system results for your tailings dams?
YES
NO

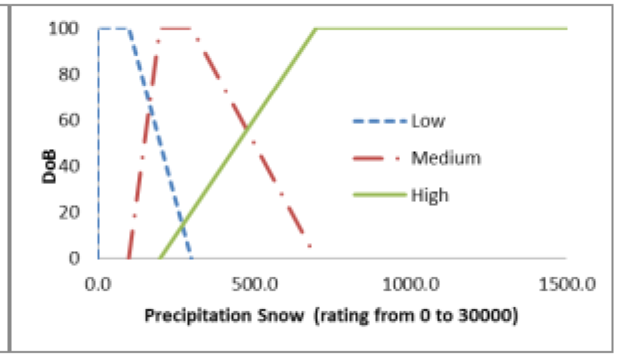

Underground Work Characteristics
Underground/tailings volume ratio: 0
Underground stability (ratio from 0 to 10): 0
Next
Back
Exit

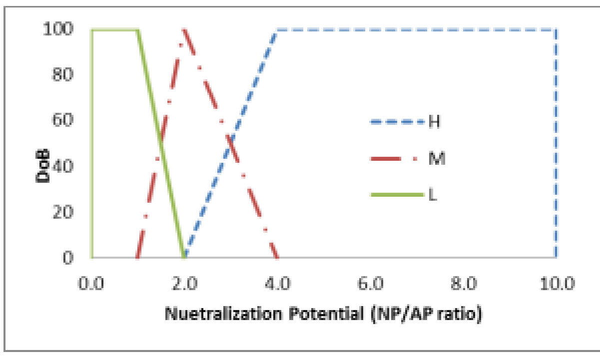

Risk Assessment Survey
Weight (0~1)
Weight (0~1)
Weight (0~1)
Consequence (0~10)
Likelihood (0~10)
Environment Risk
Surface Water
Toxicity
ARD
Air
Dust
Land System
Visual Amenity
Infrastructure (Buildings, Equipment, Camp)
Soils Contaminated
Soil Erosion
Flora Reestablishment
Fauna Reestablishment (Terrestrial/Avian/Aquatic
Voids
Tailings
Toxicity
ARD
Structure Failure
Safety and Health Risk
Openings
Shafts, Raises, Winzes
Open Pits
Infrastructure
Buildings, Equipment,
Security
Theft
Unauthorized Access
Final Land Use Risk
Toxicity
Community and Social Risk
Employees
Provision for Entitlements
Retraining/Relocation
Management
Communication
Safety Awareness
Landowners
Indigenous?
General Community Impact
Local
Regional
National
International
Legal and Financial Risk
Government
Creditors
Provisioning for Rehabilitation
Adverse Publicity
Legal and Financial Risk
Closure Plan
Rehabilitation Process
Closure Team
Reserves/Resource
Next
Back
Exit

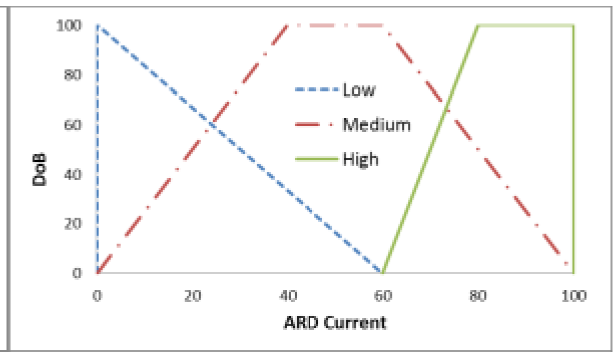
Appendix C - Fuzzy Sets and FAM Maps

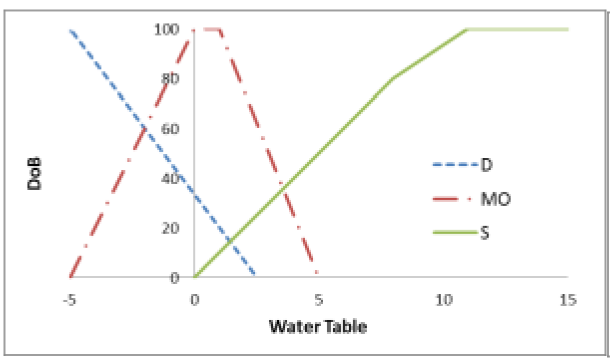
Fuzzy Sets

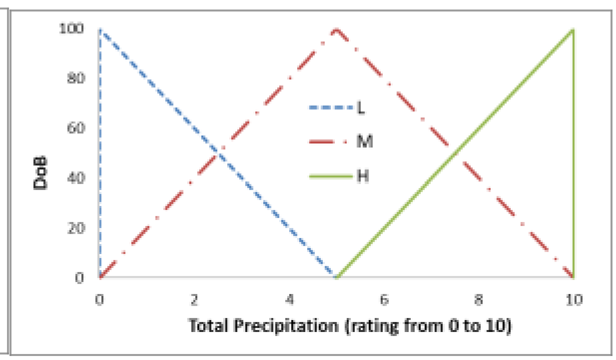


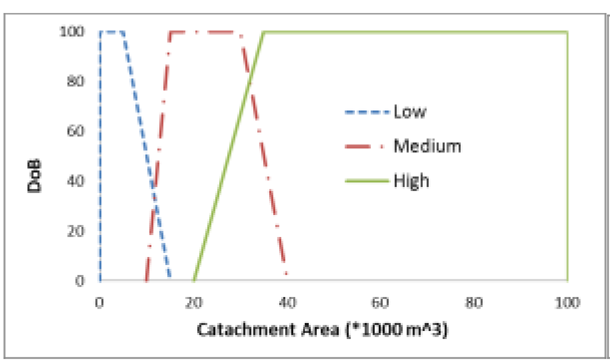


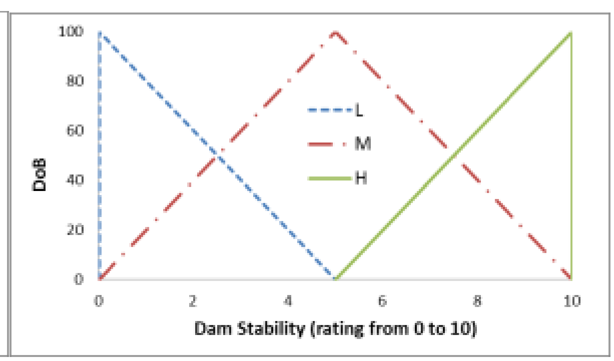


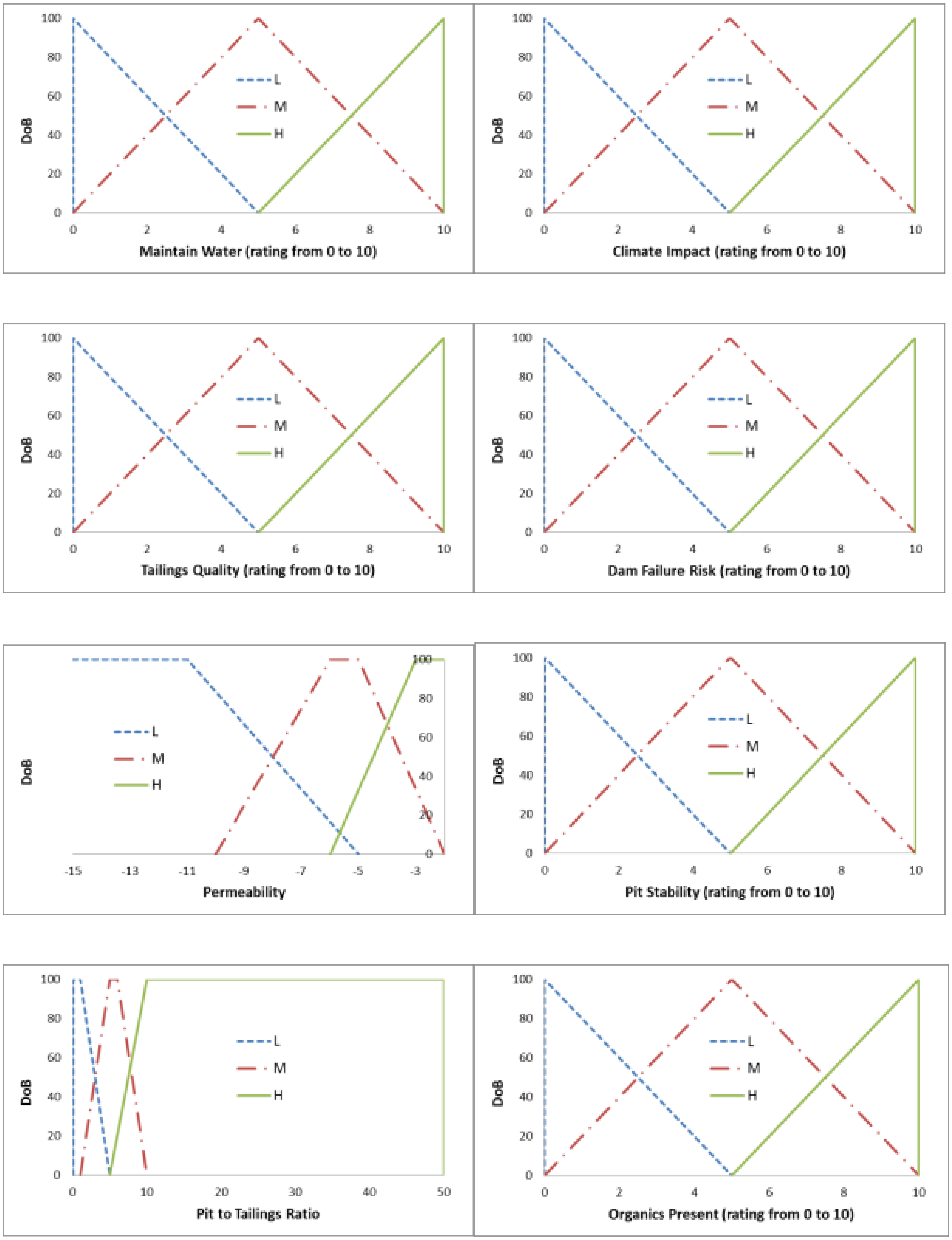


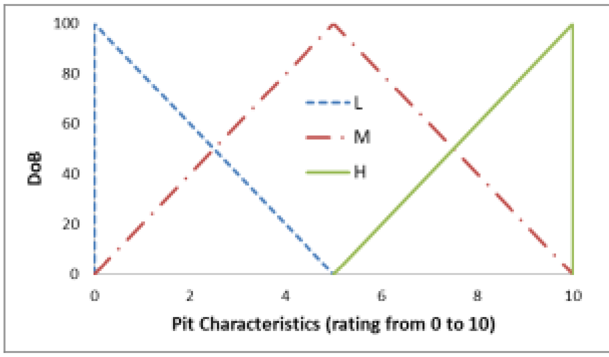


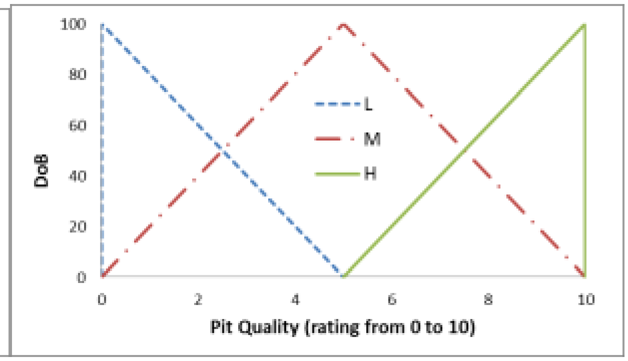


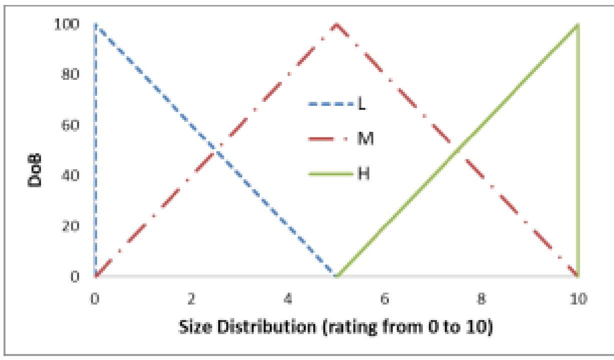


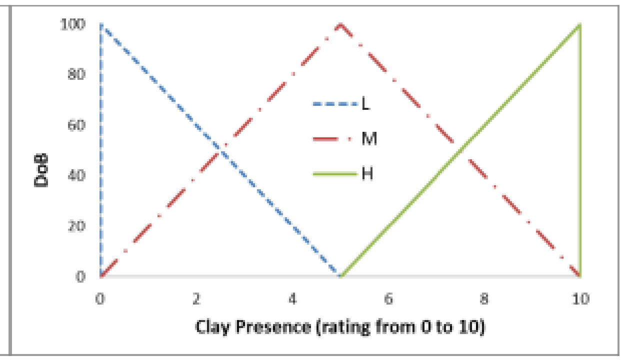


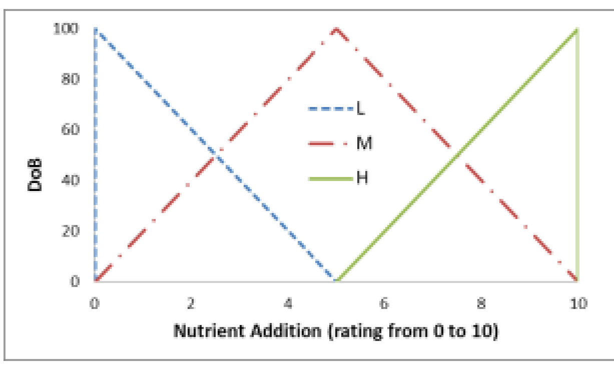

DoB
Maintain Water (rating from 0 to 10)
Climate Impact (rating from 0 to 10)
Tailings Quality (rating from 0 to 10)
Dam Failure Risk (rating from 0 to 10)
Permeability
Pit Stability (rating from 0 to 10)
Pit to Tailings Ratio
Organics Present (rating from 0 to 10)
L
M
H

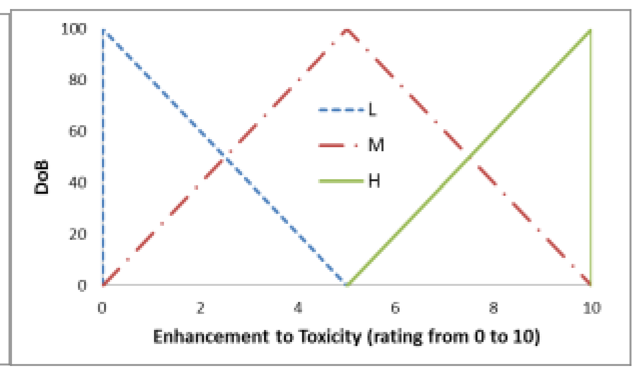

DoB
L
M
H
Pit Characteristics (rating from 0 to 10)

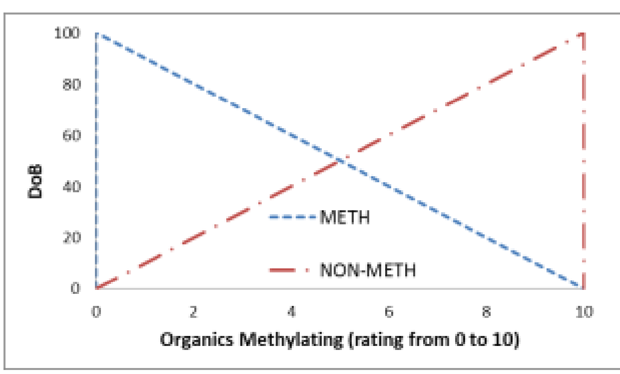

DoB
L
M
H
Pit Quality (rating from 0 to 10)

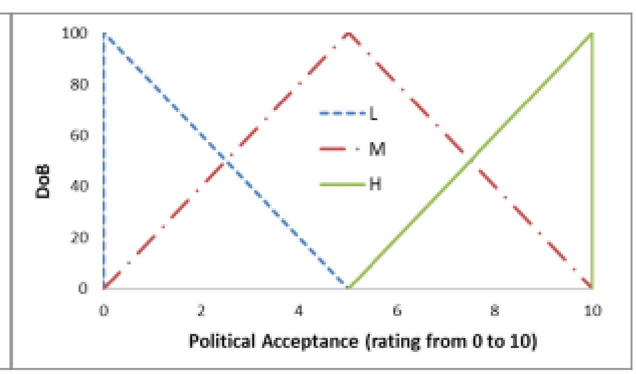

DoB
L
M
H
Size Distribution (rating from 0 to 10)

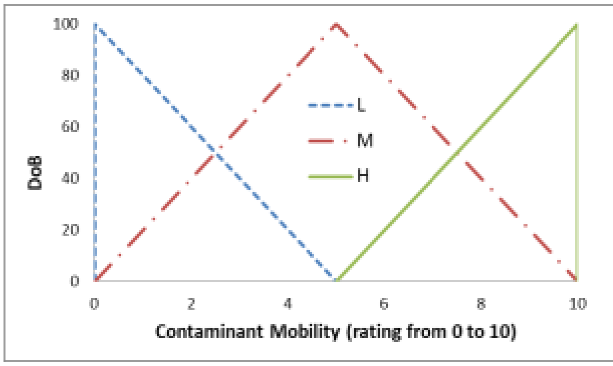

DoB
L
M
H
Clay Presence (rating from 0 to 10)

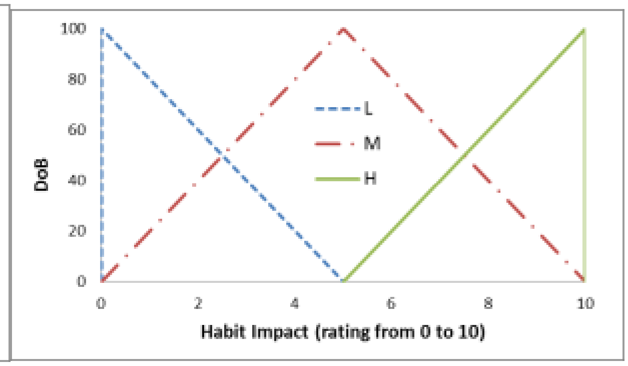

DoB
L
M
H
Nutrient Addition (rating from 0 to 10)

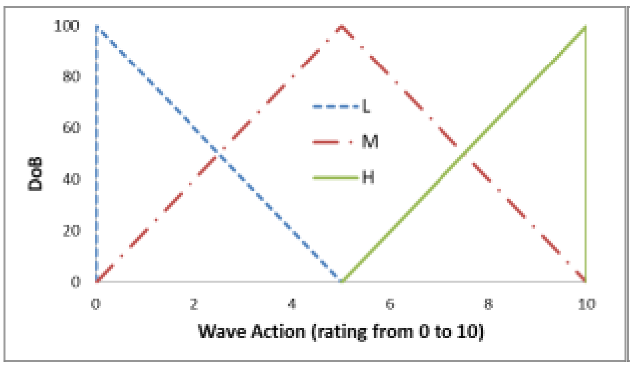

DoB
L
M
H
Enhancement to Toxicity (rating from 0 to 10)

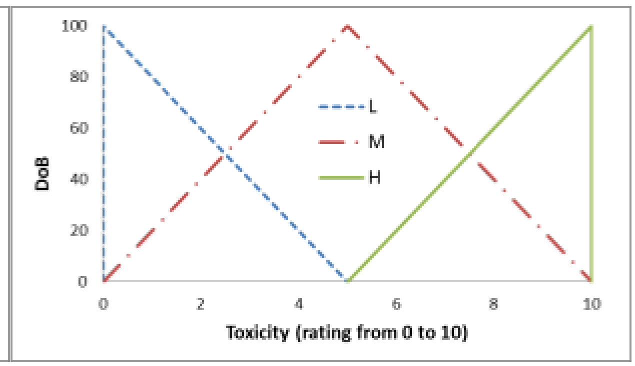

DoB
METH
NON-METH
Organics Methylating (rating from 0 to 10)

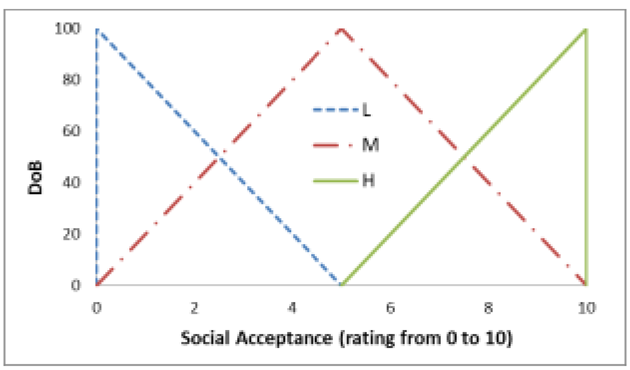

DoB
L
M
H
Political Acceptance (rating from 0 to 10)

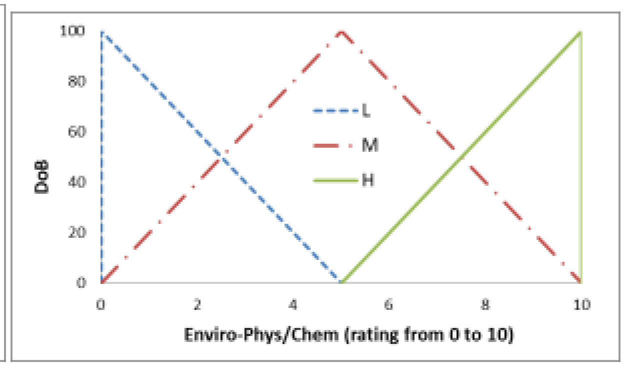

100
80
60
40
20
0
DoB
0 2 4 6 8 10
L
M
H
Contaminant Mobility (rating from 0 to 10)

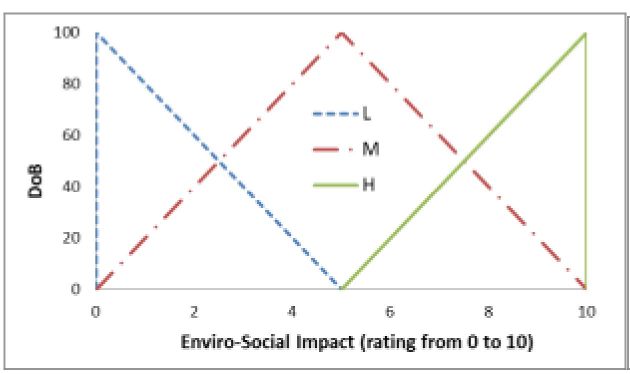

100
80
60
40
20
0
DoB
0 2 4 6 8 10
L
M
H
Habit Impact (rating from 0 to 10)

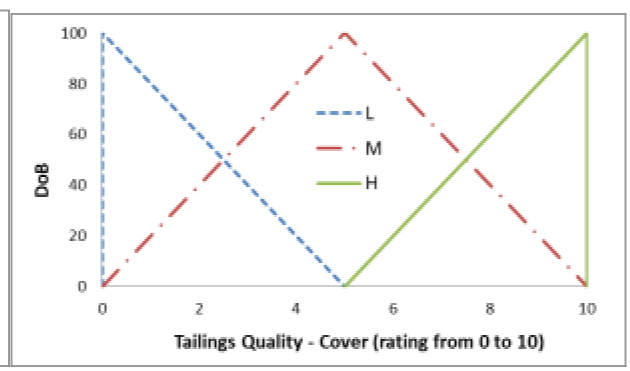

100
80
60
40
20
0
DoB
0 2 4 6 8 10
L
M
H
Wave Action (rating from 0 to 10)

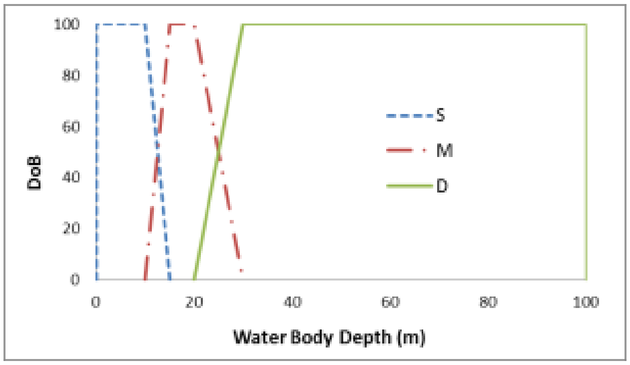

100
80
60
40
20
0
DoB
0 2 4 6 8 10
L
M
H
Toxicity (rating from 0 to 10)

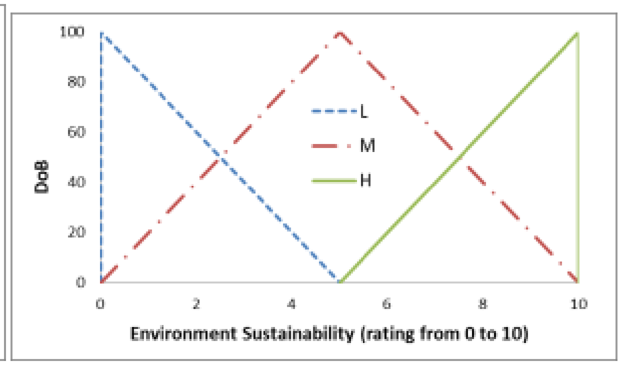

100
80
60
40
20
0
DoB
0 2 4 6 8 10
L
M
H
Social Acceptance (rating from 0 to 10)

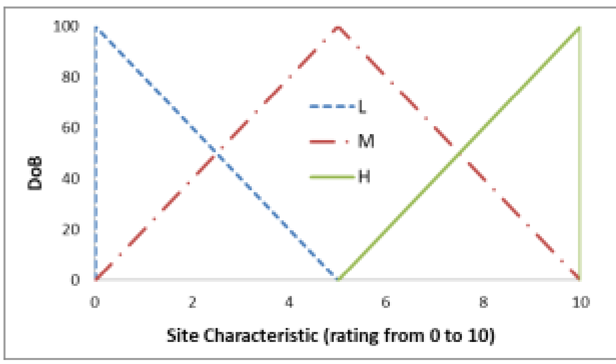

100
80
60
40
20
0
DoB
0 2 4 6 8 10
L
M
H
Enviro-Phys/Chem (rating from 0 to 10)

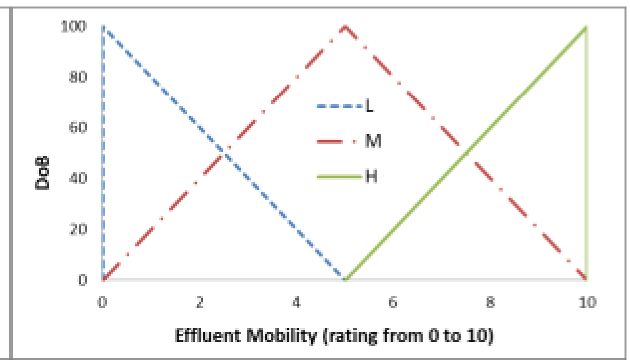

100
80
60
40
20
0
DoB
0 2 4 6 8 10
L
M
H
Enviro-Social Impact (rating from 0 to 10)

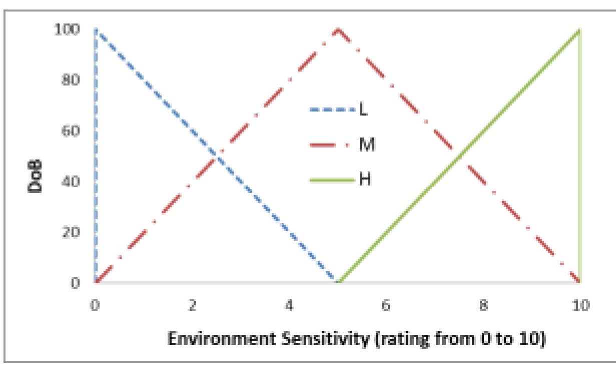

100
80
60
40
20
0
DoB
0 2 4 6 8 10
L
M
H
Tailings Quality - Cover (rating from 0 to 10)

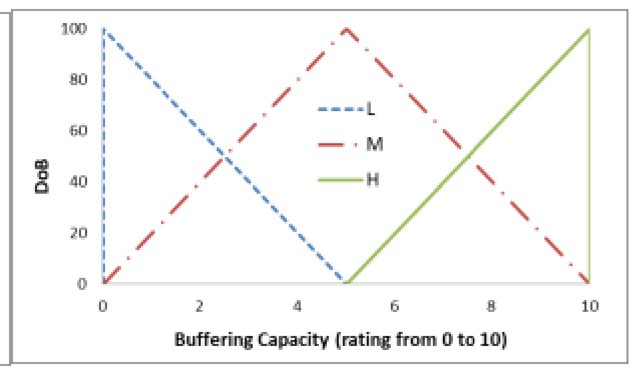

DoB
100
80
60
40
20
0
0
20
40
60
80
100
S
M
D
Water Body Depth (m)

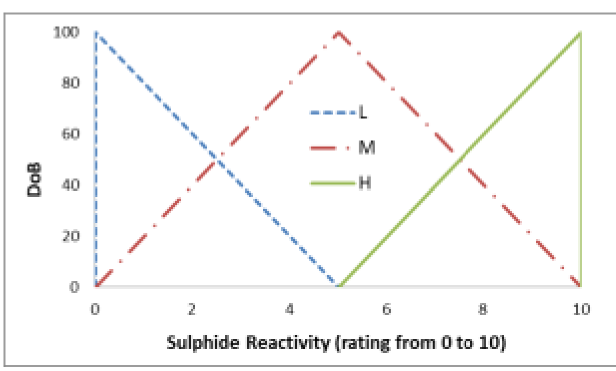

DoB
100
80
60
40
20
0
0
2
4
6
8
10
L
M
H
Environment Sustainability (rating from 0 to 10)

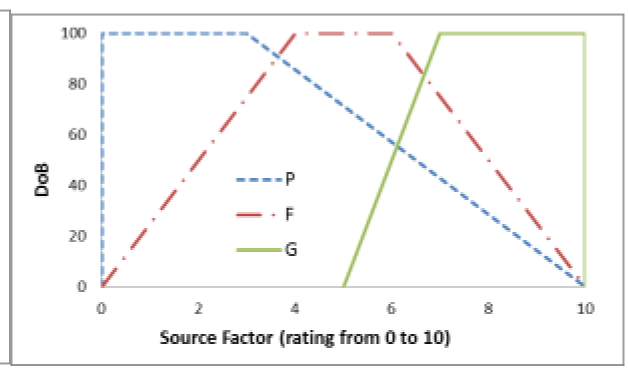

DoB
100
80
60
40
20
0
0
2
4
6
8
10
L
M
H
Site Characteristic (rating from 0 to 10)

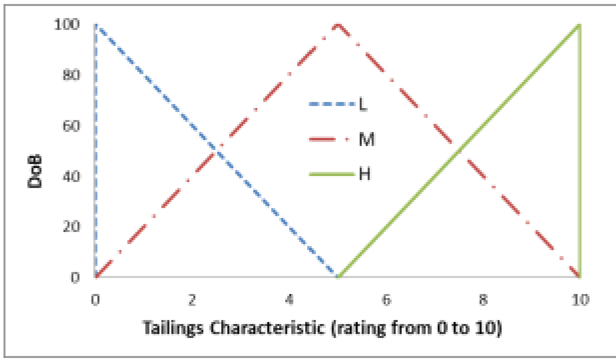

DoB
100
80
60
40
20
0
0
2
4
6
8
10
L
M
H
Effluent Mobility (rating from 0 to 10)

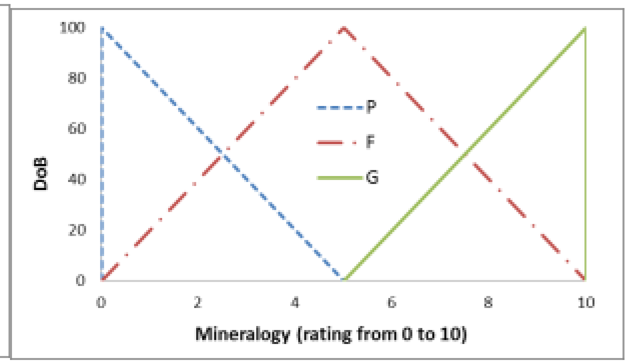

DoB
100
80
60
40
20
0
0
2
4
6
8
10
L
M
H
Environment Sensitivity (rating from 0 to 10)

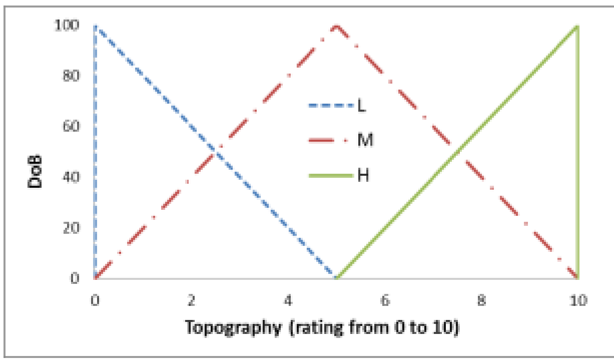

DoB
100
80
60
40
20
0
0
2
4
6
8
10
L
M
H
Buffering Capacity (rating from 0 to 10)

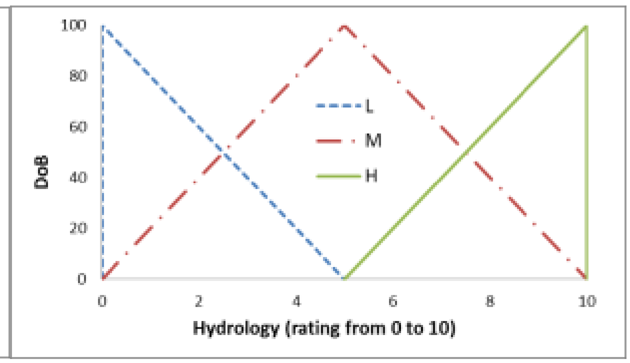

DoB
100
80
60
40
20
0
0
2
4
6
8
10
L
M
H
Sulphide Reactivity (rating from 0 to 10)

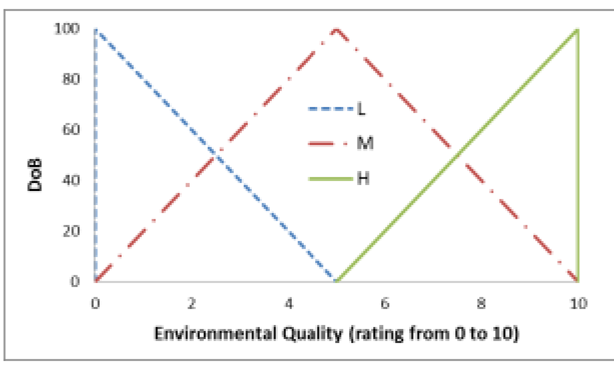

DoB
100
80
60
40
20
0
0
2
4
6
8
10
P
F
G
Source Factor (rating from 0 to 10)

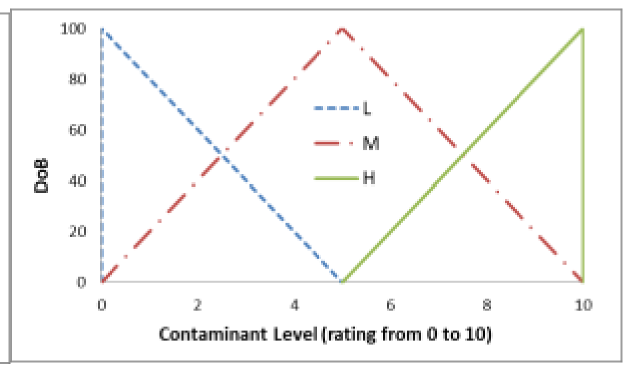

DoB
L
M
H
Tailings Characteristic (rating from 0 to 10)

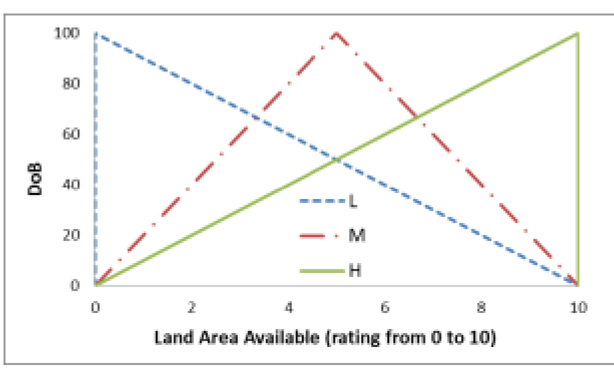

DoB
P
F
G
Mineralogy (rating from 0 to 10)

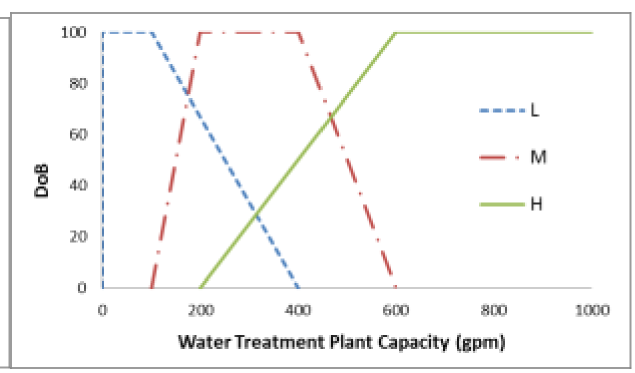

DoB
L
M
H
Topography (rating from 0 to 10)

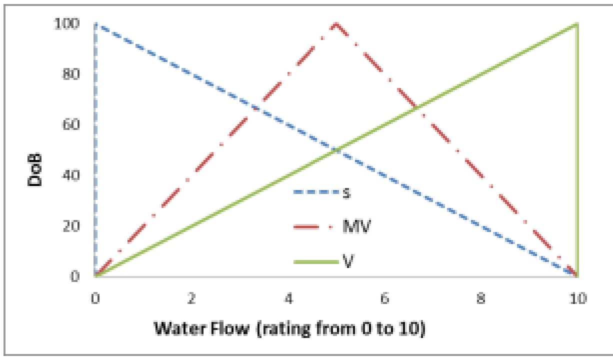

DoB
L
M
H
Hydrology (rating from 0 to 10)

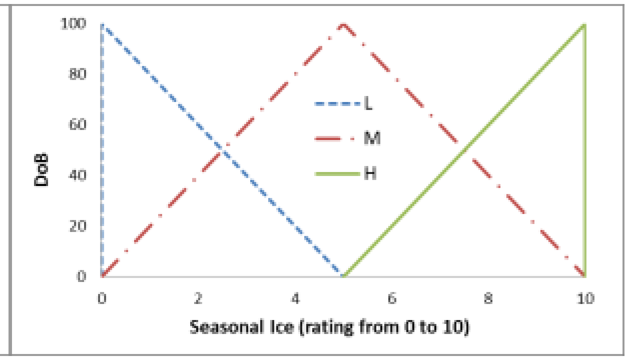

DoB
L
M
H
Environmental Quality (rating from 0 to 10)

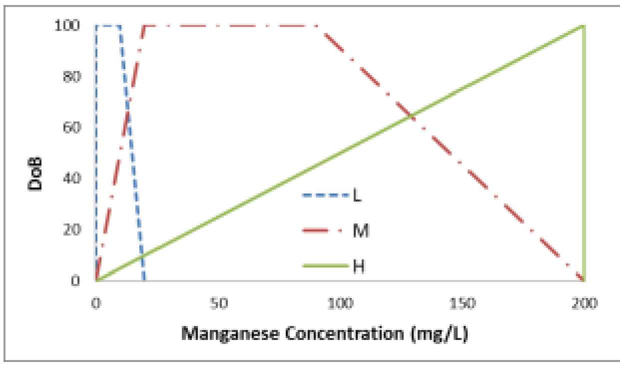

DoB
L
M
H
Contaminant Level (rating from 0 to 10)

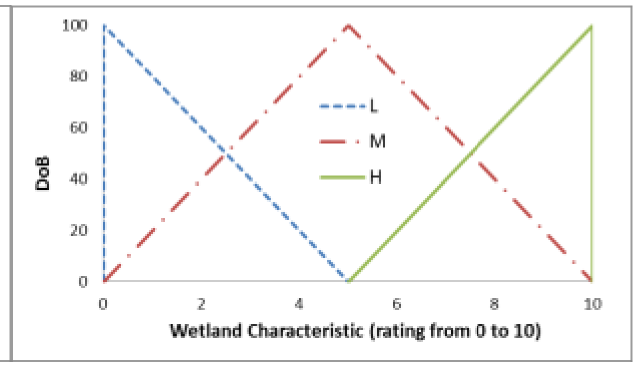

DoB
L
M
H
Land Area Available (rating from 0 to 10)

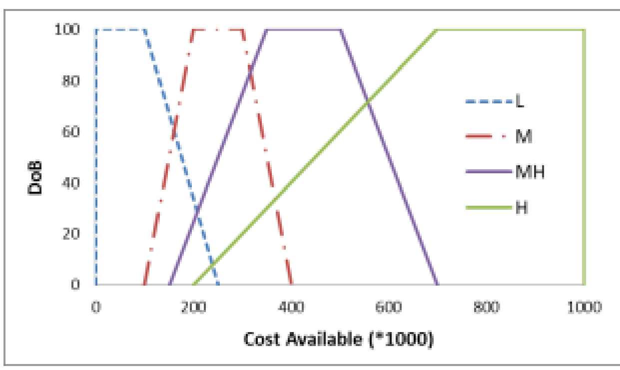

DoB
L
M
H
Water Treatment Plant Capacity (gpm)

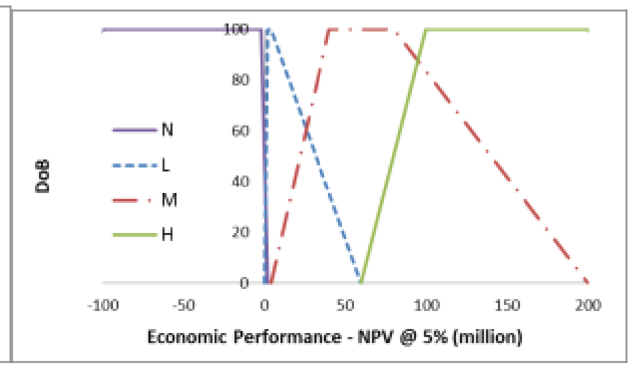

DoB
S
MV
V
Water Flow (rating from 0 to 10)

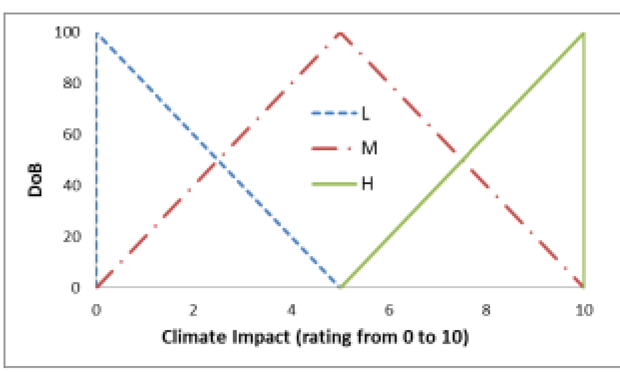

DoB
L
M
H
Seasonal Ice (rating from 0 to 10)

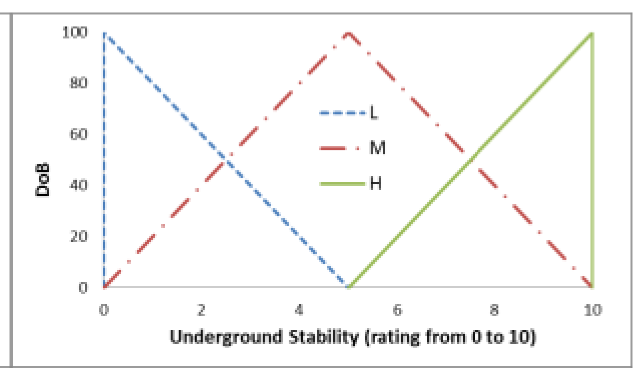

DoB
L
M
H
Manganese Concentration (mg/L)

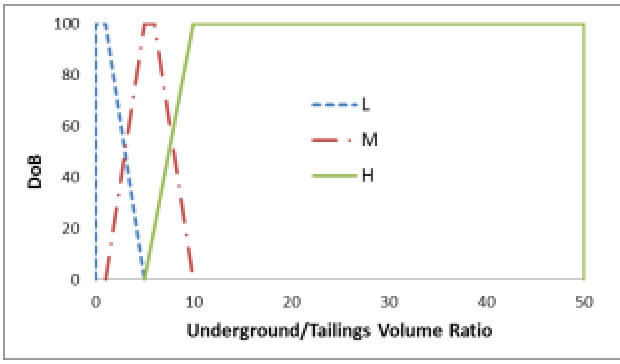

DoB
L
M
H
Wetland Characteristic (rating from 0 to 10)

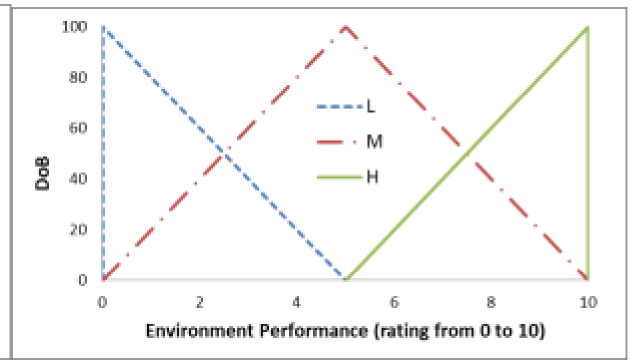

DoB
L
M
MH
H
Cost Available (*1000)

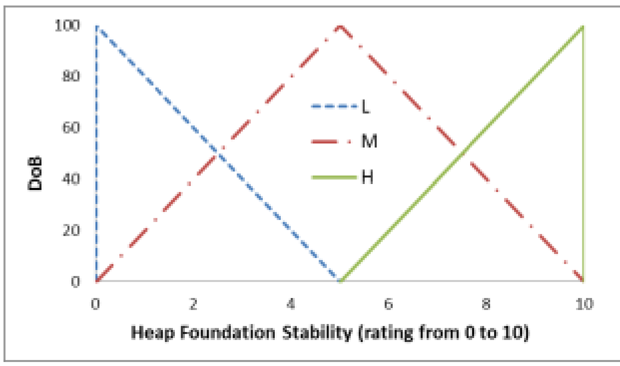

DoB
N
L
M
H
Economic Performance - NPV @ 5% (million)

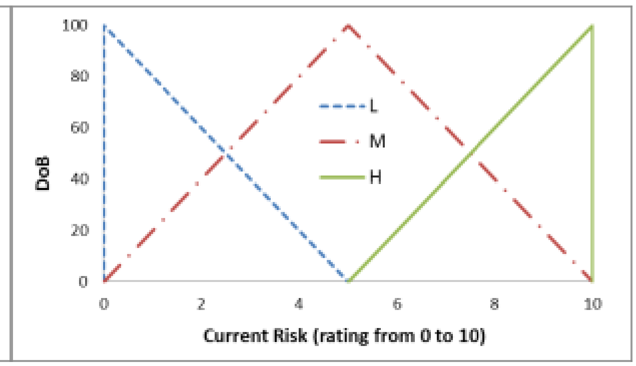

DoB
L
M
H
Climate Impact (rating from 0 to 10)

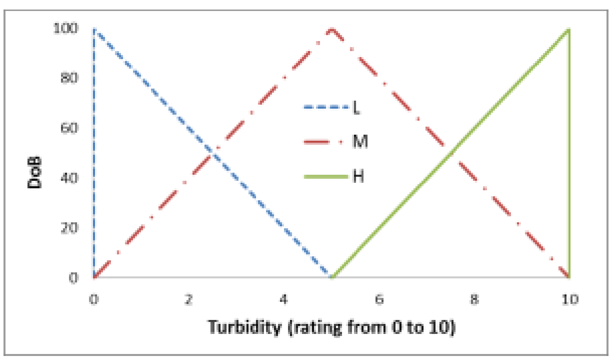

DoB
L
M
H
Underground Stability (rating from 0 to 10)

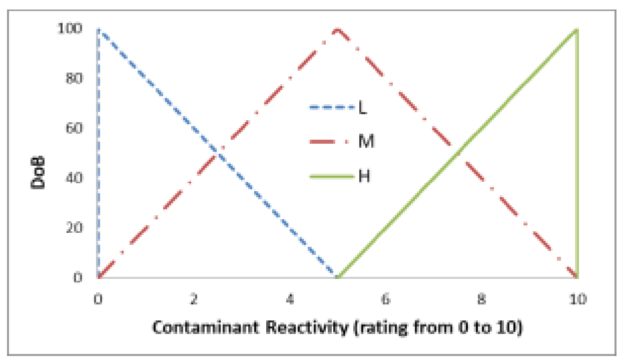

DoB
100
80
60
40
20
0
0
10
20
30
40
50
L
M
H
Underground/Tailings Volume Ratio

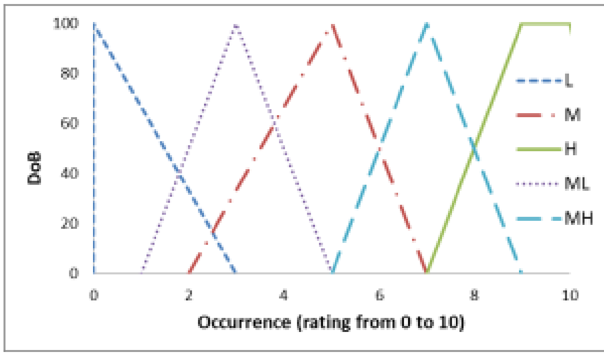

DoB
100
80
60
40
20
0
0
2
4
6
8
10
L
M
H
Environment Performance (rating from 0 to 10)

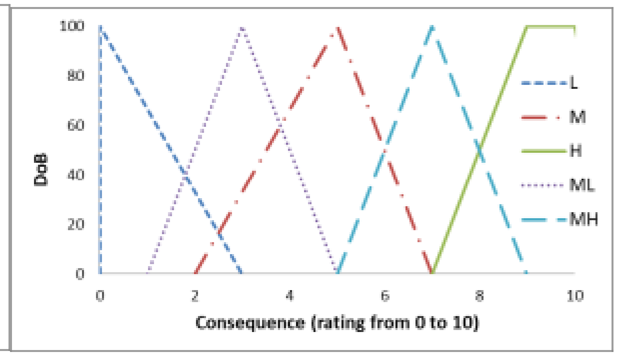

DoB
100
80
60
40
20
0
0
2
4
6
8
10
L
M
H
Heap Foundation Stability (rating from 0 to 10)


DoB
100
80
60
40
20
0
0
2
4
6
8
10
L
M
H
Current Risk (rating from 0 to 10)


DoB
100
80
60
40
20
0
0
2
4
6
8
10
L
M
H
Turbidity (rating from 0 to 10)


DoB
100
80
60
40
20
0
0
2
4
6
8
10
L
M
H
Contaminant Reactivity (rating from 0 to 10)


DoB
100
80
60
40
20
0
0
2
4
6
8
10
Occurrence (rating from 0 to 10)
L
M
H
ML
MH


DoB
100
80
60
40
20
0
0
2
4
6
8
10
Consequence (rating from 0 to 10)
L
M
H
ML
MH

Rules as Fuzzy Associative Memory (FAM) Maps

Total Precipitation			
Precipitation Rain	Precipitation Snow		
	L	M	H
L	L	L	M
M	M	M	H
H	M	H	H

Setting Properties			
Clay Present	Size Distribution		
	F	M	C
L	L	M	H
M	L	M	M
H	L	L	L

Mobility			
Species Form	Contaminant Level		
	L	M	H
NR	L	L	ML
N	L	M	MH
R	ML	H	H

Turbidity			
Wave Action	Setting Properties		
	L	M	H
L	M	H	H
M	L	M	H
H	L	L	M

Site Characteristics			
Depth	Water Body Quality		
	L	M	H
S	L	M	M
M	L	M	MH
D	L	M	H

Decision Making				
Current Risk is Low				
Environment Performance	Economic Performance			
	N	L	M	H
L	L	L	L	M
M	L	L	M	H
H	L	M	H	H
Current Risk is Moderate				
Environment Performance	Economic Performance			
	N	L	M	H
L	L	L	M	H
M	L	M	H	H
H	M	H	H	H
Current Risk is High				
Environment Performance	Economic Performance			
	N	L	M	H
L	L	M	H	H
M	M	H	H	H
H	M	H	H	H

Water Treatment Plant Quality				
ARD Potential is Low				
Plant Capacity	Available Cost			
	L	M	MH	H
L	M	MH	H	H
M	M	MH	MH	H
H	M	M	MH	MH
ARD Potential is Moderate				
Plant Capacity	Available Cost			
	L	M	MH	H
L	MH	MH	MH	H
M	M	M	MH	H
H	M	M	M	MH
ARD Potential is High				
Plant Capacity	Available Cost			
	L	M	MH	H
L	ML	M	MH	MH
M	ML	M	M	MH
H	ML	ML	M	MH

Environmental Sensitivity			
Soci-Envi Impact	Effluent Mobility		
	L	M	H
L	L	ML	M
M	ML	M	MH
H	M	MH	H

Submarine Disposal			
Environmental Sustainability	Site Characteristics		
	P	M	G
L	L	L	L
M	L	M	M
H	L	M	H

Rules as Weighted-Inference Method

A Weighted-Inference Method can be expressed in the following equation:

$$DoB_{conclusion} = \sum_{i=1}^{a} W_i * DoB_i$$

where,

W_i, DoB_i = the variable importance (weight) and degree of belief for statement i.

Tailings Quality (ARD)		
NP/AP Ratio	0.5	L
	0.25	M
	0	H
ARD Current	0.5	L
	0.25	M
	0	H

Tailings Flooding		
Tail Quality	0	L
	0.2	M
	0.4	H
Dam Failure Risk (Flooding)	0.6	L
	0.3	M
	0	H

Water Available (Dam)		
Water Table	0.3	S
	0.15	MO
	0.075	D
Total Precipitation	0.05	L
	0.1	M
	0.2	H
Catchment Area	0.025	S
	0.05	M
	0.1	L
Surrounding permeability	0.1	IP
	0.05	M
	0.025	P

Water Available (Pit)		
Water Table	0.4	S
	0.2	MO
	0.1	D
Total Precipitation	0.075	L
	0.15	M
	0.3	H
Catchment Area	0.0375	S
	0.075	M
	0.15	L
Surrounding permeability	0.15	IP
	0.075	M
	0.0375	P

Failure Risk (Flooding)		
Dam Stability	0.2	L
	0.1	M
	0	H
Climate Impact	0.2	L
	0.1	M
	0	H
Maintain Water Available	0.6	L
	0.3	M
	0.15	H

Pit Characteristics		
Pit/Tails Volume Ratio	0.075	L
	0.15	M
	0.3	H
Water Available (Pit)	0.075	L
	0.15	M
	0.3	H
Pit Stability	0	P
	0.15	M
	0.3	G

Pit Rock Quality (ARD)		
Pit Rock NP/AP Ratio	0.5	L
	0.25	M
	0.125	H
Pit Rock ARD Current	0.5	L
	0.25	M
	0.125	H

Mobility		
Contaminant Level	0.5	L
	0.25	M
	0.125	H
Contaminant Reactivity	0.5	NR
	0.25	N
	0.125	R

Pit Flooding		
Pit Characteristics	0	P
	0.3	M
	0.6	G
Pit Quality	0	L
	0.1	M
	0.2	H
Tailings Quality	0	L
	0.1	M
	0.2	H

Environmental Sustainability		
Environment Quality	0	L
	0.15	M
	0.3	H
Tailings Quality	0	L
	0.15	M
	0.3	H
Environmental Phys/Chem	0	L
	0.15	M
	0.3	H

Environmental Phys/Chem		
Turbidity	0.25	L
	0.125	M
	0	H
Nutrient Additions	0.25	L
	0.125	M
	0	H
Toxins Present	0.25	L
	0.125	M
	0	H
Habit Impact	0.25	L
	0.125	M
	0	H

Dry Cover		
Hydrology	0	L
	0.15	M
	0.3	H
Source Factor	0	L
	0.1	M
	0.2	H
Environmental Quality	0	L
	0.1	M
	0.2	H
Dam Failure Risk	0.3	L
	0.15	M
	0	H

Environment Quality		
Social Acceptance	0	L
	0.25	M
	0.5	H
Political Acceptance	0	L
	0.25	M
	0.5	H

Tailings Quality (Cover)		
NP/AP Ratio	0.5	L
	0.25	M
	0	H
Toxicity	0.5	L
	0.25	M
	0	H

Water Treatment Plant		
Water Treatment Plant Quality	0	P
	0.25	M
	0.5	G
Dam Failure Risk	0.5	L
	0.25	M
	0	H

Dam Failure Risk		
Dam Stability	0.5	L
	0.25	M
	0	H
Climate Impact	0	L
	0.25	M
	0.5	H

Pit Characteristics 2		
Pit/Tails Volume Ratio	0.125	L
	0.25	M
	0.5	H
Pit Stability	0	L
	0.25	M
	0.5	H

Underground Characteristics 2		
Underground/Tails Volume Ratio	0.125	L
	0.25	M
	0.5	H
Underground Stability	0	L
	0.25	M
	0.5	H

Heap		
Heap Stability	0	L
	0.2	M
	0.4	H
Climate Impact	0.3	L
	0.15	M
	0	H
Foundation Stability	0	L
	0.2	M
	0.4	H

Tailings to Underground		
Underground Characteristics	0	P
	0.4	M
	0.8	G
Tailings Quality	0	L
	0.1	M
	0.2	H

ORG Enhancement		
ORG Level	0.5	L
	0.25	M
	0	H
ORG Methylating	0	N
	0.25	M
	0.5	H

Toxicity		
Mobility	0	L
	0.25	M
	0.5	H
ORG Enhancement	0	L
	0.25	M
	0.5	H

Environment Quality (Cover)		
Environmental Sensitivity	0.5	L
	0.25	M
	0	H
Tailings Quality (Cover)	0	L
	0.25	M
	0.5	H

Mineralogy		
Sulphur Content	0.3	L
	0.15	M
	0	H
Sulphide Reactivity	0.3	L
	0.15	M
	0	H
Buffering Capacity	0.1	L
	0.2	M
	0.4	H

Hydrology		
Catchment Area	0.025	S
	0.05	M
	0.1	L
Topography	0.0375	P
	0.075	M
	0.15	G
Water Table	0.3	S
	0.15	M
	0.075	D
Underlying Permeability	0.1	I
	0.05	M
	0.025	P
Surrounding Permeability	0.1	L
	0.05	M
	0.025	H
Total Precipitation	0.0625	L
	0.125	M
	0.25	H

Tailings Characteristics		
Pond Size	0.5	S
	0.25	M
	0.125	L
Size Distribution	0.5	F
	0.25	M
	0.125	C

Tailings to Pit		
Pit Characteristics 2	0	P
	0.25	M
	0.5	G
Pit Quality	0	L
	0.05	M
	0.1	H
Tailings Quality	0	L
	0.05	M
	0.1	H

Wetland Characteristics		
Land Area Available	0	S
	0.125	M
	0.25	L
Hydrology	0	L
	0.125	M
	0.25	H
Contaminant Level	0.2	L
	0.1	M
	0	H
Water Flow	0	V
	0.05	M
	0.1	S
Seasonal Ice	0.1	L
	0.05	M
	0	H
Manganese Present	0	L
	0.05	M
	0.1	H

Source Factor		
Mineralogy	0	P
	0.25	M
	0.5	G
Tailings Characteristics	0	P
	0.25	M
	0.5	G

Wetland Treatment		
Wetland Characteristics	0	P
	0.25	M
	0.5	G
Dam Failure Risk	0.5	L
	0.25	M
	0	H

Appendix D – Cases and Results

MINE A

INPUTS

Mine, ore and mineral properties

Head Grade		%Cu	0.12%
Grade of +100 mesh fraction		%Cu	0.14%
Fraction of +100 mesh		wt%	20%
D80 of +100 mesh fraction		microns	250
Tailings SG		-	2.75
Tonnage of tailings		tonne	200,000,000
Old TSF surface area		m^2	800,000
Cu sulphides content		%	0.10%
Cu oxides content		%	0.02%
Fully Liberation Size		microns	70

Mineral Processing and Metallurgical Test Results

Flotation			
		Conc Grade %Cu	Recovery %
Bulk Material		20	75
Reground (Coarse) Material		25	85
Leach Tailings			
Flotation (rougher only)		8	80
Heap Leaching			
Bulk Material			50
Agglomeration Fines			55
Coarse Fraction			
Flotation Tailings			
Flotation Concentrate			
Leaching			
Bulk Material			
Reground (Coarse) Material			
Flotation Concentrate			

Site Characteristics

Water available for hydraulic mining	1-Yes, 2-No		1
Available site for surface impoundment	1-Yes, 2-No		1
Available pit for tailings backfill	1-Yes, 2-No		1
Available underground workings for backfill	1-Yes, 2-No		0
Available water body for tailings disposal	1-Yes, 2-No		1
New tailings impoundment surface area		m^2	800,000
New tailings dams surrounding permeability		cm/s	-7
New tailings dams underlying permeability		cm/s	-7
Pit surrounding permeability		cm/s	-7
Pit surface area		m^2	3,000
Precipitation	Snow	mm	600
	Rain	mm	1000
Water table of new dam		m	1
Water table of pit		m	1
Climate Impact		rating from 0 to 10	9
	Wave action	rating from 0 to 10	8
	Ice	rating from 0 to 10	9
	Flooding	rating from 0 to 10	8
	Earthquakes	rating from 0 to 10	5
Pit/tailings volume ratio			10
Underground work/tailings volume ratio			5
Pit stability		rating from 0 to 10	9
Underground stability		rating from 0 to 10	9
Water body		rating from 0 to 10	7
Ocean/lake depth		m	30
Tailings pond size		rating from 0 to 10	9
Topography of area		rating from 0 to 10	2
Land area available for passive treatment		rating from 0 to 10	7
Water flow		rating from 0 to 10	1
Heap site terrain reliability		rating from 0 to 10	8

Environmental and Social Performance

Tailings NP/AP ratio			0.5
Tailings current ARD situation		rating from 0 to 10	8
Pit rock NP/AP ratio			0.5
Pit rock current ARD situation			7
Contaminants	rating from 0 to 10	Species form (Reactivity)	Contaminant level
		1	1
	Aluminum	1	1
	Arsenic	1	1
	Cadmium	1	1
	Chromium	1	1
	Copper	1	1
	Cyanide	1	1
	Lead	1	1
	Magnesium	1	1
	Manganese	1	1
	Mercury	1	1
	Uranium	1	1
	Other	1	1
Clay presence		rating from 0 to 10	3
Size distribution		rating from 0 to 10	3
ORG level		rating from 0 to 10	5
ORG methylations		rating from 0 to 10	1
Nutrient additions		rating from 0 to 10	8
Habitat impact		rating from 0 to 10	6
Social acceptance		rating from 0 to 10	6
Political acceptance		rating from 0 to 10	6
Social-environmental impact		rating from 0 to 10	7
Effluent mobility		rating from 0 to 10	5
Final tailings sulphur content		rating from 0 to 10	9
Final tailings sulphide reactivity		rating from 0 to 10	6
Buffering capacity		rating from 0 to 10	4
Manganese present		mg/L	120
Available cost for water treatment plant		rating from 0 to 10	7
Treatment plant Capacity		rating from 0 to 10	5
Remediation operating cost		$US/year	1,000,000
Remediation running years		years	50
Failure probability in first 5 years		%	5%
Must failure years		years	30
Failure consequence		$US	50,000,000

Primary Risk Issue	DoB	Wt.	Secondary Issues	DoB	Wt.	Tertiary Issues	DoB	Wt.	O[1]	C[2]
Environmental	57	0.4	Surface Water	25	0.3	Toxicity	0	0.5	1	1
						ARD	0	0.5	1	1
			Air	11	0.2	Dust	11	1	2	2
			Land System	52	0.1	Visual Amenity	55	0.2	5	5
						Infrastructure	34	0.1	3	4
						Soil Contamination	75	0.2	6	7
						Soil Erosion	23	0.2	2	4
						Flora Reestablish.	23	0.1	2	4
						Fauna Reestablish.	30	0.1	3	3
						Voids	34	0.1	3	4
			Tailings	87	0.4	Toxicity	80	0.3	7	7
						ARD	90	0.3	8	8
						Structure Failure	90	0.4	8	8
Safety and Health	33	0.1	Openings	28	0.2	Shafts, Raises, Winzes	0	0.5	0	0
						Open Pits	11	0.5	2	2
			Infrastructure	40	0.1	Buildings and Equip.	40	1	3	5
			Security	39	0.4	Theft	34	0.6	3	4
						Unauthorized Access	39	0.3	2	6
Final Land Use	90	0.2	-	90	1	Land Value	90	1	6	9
Community and Social	71	0.4	Employees	37	0.1	Provision for Entitlements	39	0.5	2	6
						Retraining/Relocation	30	0.5	3	3
			Management	20	0.1	Communication	34	0.5	4	3
						Safety Awareness	6	0.5	2	1
			Landowners	90	0.2	Indigenous People	90	1	8	8
			General Community Impact	75	0.6	Local	80	0.4	7	7
						Regional	80	0.3	7	7
						National	68	0.2	6	6
						International	55	0.1	5	5
Legal and Financial	38	0.1	-	38	1	Government	55	0.25	5	5
						Creditors	30	0.25	3	3
						Provisioning for Rehabilitation	30	0.25	3	3
						Adverse Publicity	39	0.25	4	4
Technical	31	0.1	-	31	1	Closure Plan	39	0.3	4	4
						Rehabilitation Process	39	0.3	4	4
						Closure Team	39	0.2	4	4
						Reserves/Resource	0	0.2	1	1

Market conditions

Average exchange rate $ U.S. =		-	1.25
Copper Price		$US/lb	3.5
Tax Rate (%)		%	50%

Equipment Option

Existing mill or leaching plant	1-Yes, 0-No		0
Used processing equipment available	1-Yes, 0-No		
	Flowsheet 1		0
	Flowsheet 2		0
	Flowsheet 3		0
	Flowsheet 4		0
	Flowsheet 5		0
	Flowsheet 6		0
	Flowsheet 7		0
	Flowsheet 8		0
	Flowsheet 9		0
	Flowsheet 10		0
	Flowsheet 11		0
Existing plant capital cost		$US	0
Existing plant operation cost		$US/year	0
Existing plant capacity		tpd	0
Recovery achievable by existing plant			0%
Product grade achievable by existing plant			0%

Significant parameters and assumptions

Smelter cost		$US/t conc	75.00
Refinery cost		$US/lb Cu	0.075
Transportation cost		$US/t conc	86.00
Smelter recovery (%)		%	95
Refinery recovery (%)		%	99

OUTPUTS

Design

Mining method			Hydraulic Mining
Processing flowsheet			Flotation + Concentrate Leaching
Tailings disposal method			Pit Backfilling
Reclamation method			Flooding
Mining and processing tonnage rate		tpd	29,600
		tpy	10,804,000
Mine life		years	18.5

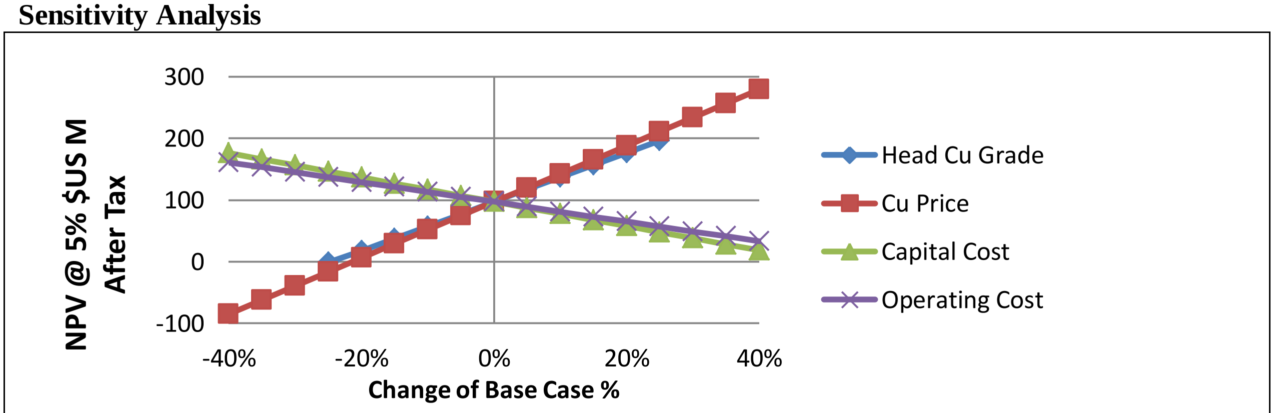
Processing performance

Recovery		%	76.4
Concentrate grade		%	0
Concentrate tonnage rate		tpd	0
		tpy	0
Copper Production Rate		tpd	27
		tpy	9,905

Economic Performance

Capital costs	Total	$US	149,337,000
Product costs	Total	$US/tonne	2.49
Annual product costs		$US/year	26,905,975
Annual revenue		$US	76,428,000
NPV at i=5%		$US	97,200,000
NPV at i=10%		$US	22,428,000
DCFROR			12.2%
Base Case PV at i=5%		$US	-38,334,000
Total Retreatment Project PV		$US	135,534,000

Sensitivity Analysis



Environmental and Social Performance

Current Risk	DoB		80
Retreatment environmental performance	DoB		56

Final Recommendation

Tailings retreatment	DoB		95

Summary of Alternative Mining and Processing Flowsheets (part 1)

Flowsheet	Tonnage Rate	Mine Life	Conc Production Rate	Conc Grade	Cu Production Rate	Recovery	Revenue
	tpd	years	tpd	%	tpd	%	$US/y
1	31,300	17.5	141	20.0	0	75	64,725,000
2	28,100	19.5	115	25.0	0	85	67,540,000
3	27,400	20.0	0	0.0	16	50	46,302,000
4	27,400	20.0	0	0.0	18	55	50,932,000
5	27,400	20.0	0	0.0	10	31	29,035,000
6	29,600	18.5	0	0.0	27	76	76,428,000
7	28,100	19.5	0	0.0	22	65	61,725,000
8	27,400	20.0	329	8.0	6	96	66,054,000
9	27,400	20.0	0	0.0	12	35	32,411,000
10	27,400	20.0	0	0.0	15	45	41,672,000
11	27,400	20.0	91	25.0	7	89	71,920,000
12	0	0.0	0	0.0	0.0	70	0

Summary of Alternative Mining and Processing Flowsheets (part 2)

Flowsheet	Capital Cost	Operation Cost		NPV@5%	NPV@10%	DCFROR
	$US	$US/y	$US/t	$US	$US	%
1	134,733,000	35,770,000	3.1	-12,638,000	-48,481,000	3.7%
2	191,950,000	35,770,000	3.5	-57,368,000	-99,411,000	1.1%
3	113,965,000	41,245,000	4.1	-117,965,000	-116,698,000	-
4	119,955,000	43,435,000	4.3	-110,621,000	-113,579,000	-14.8%
5	104,233,650	35,040,000	3.5	-174,130,000	-151,985,000	-
6	149,337,000	26,905,975	2.5	97,200,000	22,428,000	12.2%
7	125,800,000	37,889,555	3.7	-19,097,000	-52,431,000	3.1%
8	221,889,000	33,580,000	3.4	-88,688,000	-130,891,000	-0.4%
9	468,072,000	97,740,000	9.9	-1,020,950,000	-845,781,000	-
10	531,730,000	106,215,000	10.6	-1,099,550,000	-919,646,000	-
11	293,818,000	101,849,993	10.2	-571,846,000	-483,758,000	-
12	-	-	-	-	-	-

Summary of Tailings Disposal and Reclamation Alternatives

Method		Capital Cost	NPV @5%	Environmental Performance	Land Disturbance	Overall Score
		$US	$US	DoB	Score	
Weights		0.2	0.2	0.4	0.1	
1	Tailings dam only	17,999,000	-30,791,000	28.75	100	4.7
2	Pit back fill only	23,990,000	-55,410,000	52.50	10	3.1
3	Underground fill	-	-	-	-	9.0
4	Sub marine	13,856,000	-33,549,000	33.71	1	3.9
5	Pit back fill and flooding	27,181,000	-63,146,000	56.49	10	2.9
6	Tailings dam and flooding	82,456,000	-130,819,000	34.71	100	6.2
7	Tailings dam and covers	57,732,000	-117,926,000	51.68	100	4.9
8	Tailings dam and water treatment	24,380,000	-35,046,000	37.50	100	4.2
9	Tailings dam and passive treatment	1,092,343,000	-2,620,305,000	53.55	100	4.9

MINE B

INPUTS

Mine, ore and mineral properties

Head Grade		%Cu	0.18%
Grade of +100 mesh fraction		%Cu	0.30%
Fraction of +100 mesh		wt%	35%
D80 of +100 mesh fraction		microns	230
Tailings SG		-	2.75
Tonnage of tailings		Tonnes	150,000,000
Old TSF surface area		m^2	700,000
Cu sulphides content		%	0.16%
Cu oxides content		%	0.02%
Fully Liberation Size		microns	65

Mineral Processing and Metallurgical Test Results

Flotation			
		Conc Grade %Cu	Recovery %
Bulk Material		23%	85%
Reground (Coarse) Material			
Leach Tailings			
Flotation (rough only)		8%	88%
Heap Leaching			
Bulk Material			50%
Agglomeration Fines			50%
Coarse Fraction			55%
Flotation Tailings			
Flotation Concentrate			50%
Leaching			
Bulk Material			40%
Reground (Coarse) Material			40%
Flotation Concentrate			50%

Site Characteristics

Water available for hydraulic mining	1-Yes, 2-No		1
Available site for surface impoundment	1-Yes, 2-No		1
Available pit for tailings backfilling	1-Yes, 2-No		0
Available underground work for backfilling	1-Yes, 2-No		1
Available water body for tailings disposal	1-Yes, 2-No		1
New tailings impoundment surface area		m^2	600,000
New tailings dams surrounding permeability		cm/s	-7
New tailings dam underlying permeability		cm/s	-7
Pit surrounding permeability		cm/s	-7
Pit surface area		m^2	0
Precipitation	Snow	mm	300
	Rain	mm	500
Water table of new dam		m	1
Water table of pit		m	1
Climate Impact		rating from 0 to 10	3
	Wave action	rating from 0 to 10	2
	Ice	rating from 0 to 10	2
	Flooding	rating from 0 to 10	3
	Earthquakes	rating from 0 to 10	3
Pit/tailings volume ratio			0
Underground work/tailings volume ratio			5
Pit stability		rating from 0 to 10	9
Underground stability		rating from 0 to 10	9
Water body		rating from 0 to 10	7
Ocean/lake depth		m	30
Tailings pond size		rating from 0 to 10	8
Topography of area		rating from 0 to 10	3
Land area available for passive treatment		rating from 0 to 10	5
Water flow		rating from 0 to 10	3
Heap site terrain reliability		rating from 0 to 12	8

Environmental and Social Performance

Tailings NP/AP ratio			2
Tailings current ARD situation		rating from 0 to 10	5
Pit rock NP/AP ratio			2
Pit rock current ARD situation		rating from 0 to 10	5
Contaminants	rating from 0 to 10	Species form (Reactivity)	Contaminant level
		3	3
	Aluminum	1	2
	Arsenic	1	1
	Cadmium	1	1
	Chromium	1	1
	Copper	1	2
	Cyanide	1	1
	Lead	1	1
	Magnesium	2	2
	Manganese	1	2
	Mercury	1	1
	Uranium	1	1
	Other	3	3
Clay presence		rating from 0 to 10	3
Size distribution		rating from 0 to 10	3
ORG level		rating from 0 to 10	5
ORG methylating		rating from 0 to 10	1
Nutrient additions		rating from 0 to 10	5
Habitat impact		rating from 0 to 10	5
Social acceptance		rating from 0 to 10	5
Political acceptance		rating from 0 to 10	5
Social-environmental impact		rating from 0 to 10	5
Effluent mobility		rating from 0 to 10	5
Final tailings sulphur content		rating from 0 to 10	7
Final tailings sulphide reactivity		rating from 0 to 10	5
Buffering capacity		rating from 0 to 10	5
Manganese present		mg/L	120
Available cost for water treatment plant		rating from 0 to 10	5
Treatment plant Capacity		rating from 0 to 10	5
Remediation operating cost		$US/year	0
Remediation running years		years	0
Failure probability in first 5 years		%	1%
Certain failure year		years	100
Failure consequence		$US	5,000,000

Primary Risk Issue	DoB	Wt.	Secondary Issues	DoB	Wt.	Tertiary Issues	DoB	Wt.	O[1]	C[2]
Environmental	57	0.4	Surface Water	67	0.3	Toxicity	38	0.5	7	2
						ARD	90	0.5	8	7
			Air	11	0.2	Dust	11	1	2	2
			Land System	50	0.1	Visual Amenity	83	0.2	6	8
						Infrastructure	30	0.1	3	3
						Soil Contamination	42	0.2	6	3
						Soil Erosion	21	0.2	2	3
						Flora Reestablish.	23	0.1	2	4
						Fauna Reestablish.	30	0.1	3	3
						Voids	34	0.1	3	4
			Tailings	65	0.4	Toxicity	44	0.3	7	3
						ARD	78	0.3	8	6
						Structure Failure	68	0.4	6	6
Safety and Health	33	0.1	Openings	28	0.2	Shafts, Raises, Winzes	11	0.5	2	2
						Open Pits	0	0.5	0	0
			Infrastructure	40	0.1	Buildings and Equip.	40	1	3	5
			Security	39	0.4	Theft	34	0.6	3	4
						Unauthorized Access	39	0.3	2	6
Final Land Use	55	0.2	-	55	1	Land Value	55	1	6	5
Community and Social	60	0.4	Employees	37	0.1	Provision for Entitlements	39	0.5	2	6
						Retraining/Relocation	30	0.5	3	3
			Management	20	0.1	Communication	34	0.5	4	3
						Safety Awareness	6	0.5	2	1
			Landowners	90	0.2	Indigenous People	90	1	8	8
			General Community Impact	57	0.6	Local	68	0.4	7	6
						Regional	55	0.3	7	5
						National	43	0.2	6	4
						International	40	0.1	5	3
Legal and Financial	34	0.1	-	34	1	Government	40	0.25	5	3
						Creditors	30	0.25	3	3
						Provisioning for Rehabilitation	30	0.25	3	3
						Adverse Publicity	34	0.25	4	3
Technical	28	0.1	-	28	1	Closure Plan	34	0.3	4	3
						Rehabilitation Process	34	0.3	4	3
						Closure Team	34	0.2	4	3
						Reserves/Resource	0	0.2	1	1

Market conditions

Average exchange rate $ U.S. =		-	1.25
Copper Price		$US/lb	3.5
Tax Rate (%)		%	50%

Equipment Options

Existing mineral processing or leaching plant	1-Yes, 0-No		0
Used processing equipment available	1-Yes, 0-No		
	Flowsheet 1		0
	Flowsheet 2		0
	Flowsheet 3		0
	Flowsheet 4		0
	Flowsheet 5		0
	Flowsheet 6		0
	Flowsheet 7		0
	Flowsheet 8		0
	Flowsheet 9		0
	Flowsheet 10		0
	Flowsheet 11		0
Existing plant capital cost		$US	0
Existing plant operation cost		$US/year	0
Existing plant capacity		tpd	0
Recovery can be achieved by existing plant			0%
Product grade can be achieved by existing plant			0%

Significant parameters and assumptions

Smelter cost		$US/t conc	75.00
Refinery cost		$US/lb Cu	0.075
Transportation cost		$US/t conc	86.00
Smelter recovery (%)		%	95%
Refinery recovery (%)		%	99%

OUTPUTS

Design

Mining method			Hydraulic Mining
Processing flowsheet			Flotation
Tailings disposal method			Underground Backfilling
Reclamation method			None
Mining and processing tonnage rate		tpd	35,700
		tpy	13,030,500
Mine life		years	11.5

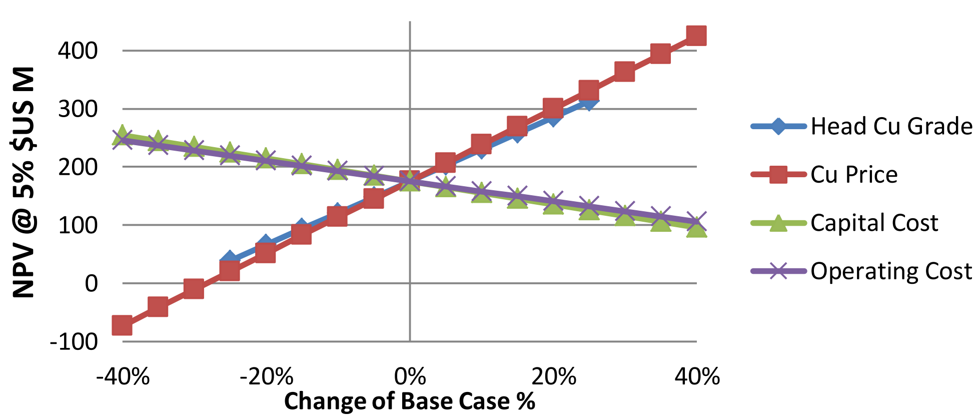
Processing performance

Recovery		%	85.00%
Concentrate grade		%	23.00%
Concentrate tonnage rate		tpd	237
		tpy	86,681
Copper Production Rate		tpd	0
		tpy	0

Economic Performance

Capital costs	Total	$US	144,720,000
Product costs	Total	$US/tonne	3.11
Annual product costs		$US/year	40,515,000
Annual revenue		$US/year	127,594,000
NPV at i=5%		$US	175,463,000
NPV at i=10%		$US	103,459,000
DCFROR			23.5%
Base Case PV at i=5%		$US	-1,852,000
Total Retreatment Project PV		$US	177,315,000

Sensitivity Analysis



Environmental and Social Performance

Current Risk	DoB		67
Retreatment environmental performance	DoB		88

Final Recommendation

Tailings retreatment	DoB		95

Summary of Alternative Mining and Processing Flowsheets (part 1)

Flowsheet	Tonnage Rate	Mine Life	Conc Production Rate	Conc Grade	Cu Production Rate	Recovery	Revenue
	tpd	years	tpd	%	tpd	%	$US/y
1	35,700	11.5	237	23.0	0	85.0	163,335,000
2	21,100	19.5	128	23.3	0	78.4	100,427,000
3	20,500	20.0	0	0.0	18	50.0	94,800,000
4	21,100	19.5	0	0.0	19	50.0	90,491,000
5	20,500	20.0	0	0.0	12	32.1	51,397,000
6	21,100	19.5	0	0.0	17	44.0	50,858,000
7	21,600	19.0	0	0.0	17	44.0	61,118,000
8	20,500	20.0	406	8.0	2	94.0	66,534,000
9	20,500	20.0	0	0.0	12	33.3	34,642,000
10	20,500	20.0	0	0.0	15	40.0	41,570,000
11	20,500	20.0	108	25.0	15	91.9	105,478,000
12	0	0.0	0	0.0	0	70.0	0

Summary of Alternative Mining and Processing Flowsheets (part 2)

Flowsheet	Capital Cost	Operation Cost		NPV@5%	NPV@10%	DCFROR
	$US	$US/y	$US/t	$US	$US	%
1	144,720,000	40,515,000	3.1	175,463,000	103,459,000	23.5%
2	168,445,000	27,740,000	3.6	35,903,000	-27,892,000	7.4%
3	96,913,000	31,390,000	4.2	1,287,000	-29,868,000	5.1%
4	103,452,000	33,945,000	4.4	-16,183,000	-43,427,000	3.0%
5	92,270,400	27,375,000	3.7	-83,747,000	-86,451,000	-13.8%
6	136,092,000	19,934,110	2.6	-12,536,000	-51,109,000	3.9%
7	115,018,000	29,172,260	3.7	-36,616,000	-60,770,000	0.7%
8	185,183,000	24,820,000	3.3	17,457,000	-46,832,000	6.1%
9	399,693,000	82,536,000	11.1	-822,944,000	-688,663,000	-
10	454,126,000	89,790,000	12.0	-896,358,000	-756,055,000	-
11	277,156,000	78,286,781	10.5	-193,756,000	-220,216,000	-6.1%
12	-	-	-	-	-	-

Summary of Tailings Disposal and Reclamation Alternatives

	Method	Capital Cost	NPV@5%	Environment Performance	Land Disturbance	Overall Score
		$US	$US	DoB	Score	
	Weights	0.2	0.2	0.4	0.1	
1	Tailings dam only	13,499,000	-23,093,000	67	100	3.2
2	Pit back fill only	-	-	-	-	-
3	Underground fill	28,336,000	-46,138,000	88	1	2.3
4	Sub marine	10,705,000	-25,919,000	47	1	3.9
5	Pit back fill and flooding	-	-	-	-	-
6	Tailings dam and flooding	62,192,000	-98,630,000	61	100	5.4
7	Tailings dam and covers	48,279,000	-100,168,000	74	100	4.1
8	Tailings dam and water treatment	18,943,000	-27,678,000	73	100	3.4
9	Tailings dam and passive treatment	819,571,000	-1,965,649,000	77	100	4.3

MINE C

INPUTS

Mine, ore and mineral properties

Head Grade		%Cu	0.12%
Grade of +100 mesh fraction		%Cu	0.15%
Fraction of +100 mesh		wt%	35.00%
D80 of +100 mesh fraction		microns	250
Tailings SG		-	2.75
Tonnage of tailings		tonnes	80,000,000
Old TSF surface area		m^2	450,000
Cu sulphides content		%	0.07%
Cu oxides content		%	0.05%
Fully Liberation Size		microns	75

Mineral Processing and Metallurgical Test Results

Flotation			
		Conc Grade %Cu	Recovery %
Bulk Material			
Reground (Coarse) Material			
Leach Tailings			
Flotation (rough only)			
Heap Leaching			
Bulk Material			
Agglomeration Fines			
Coarse Fraction			
Flotation Tailings			
Flotation Concentrate			
Leaching			
Bulk Material			
Reground (Coarse) Material			
Flotation Concentrate			

Site Characteristics

Water available for hydraulic mining	1-Yes, 2-No		1
Available site for surface impoundment	1-Yes, 2-No		1
Available pit for tailings backfilling	1-Yes, 2-No		1
Available underground work for backfilling	1-Yes, 2-No		0
Available water body for tailings disposal	1-Yes, 2-No		1
New tailings impoundment surface area		m^2	80,000
New tailings dams surrounding permeability		cm/s	-7
New tailings dams underlying permeability		cm/s	-7
Pit surrounding permeability		cm/s	-7
Pit surface area		m^2	3,000
Precipitation	Snow	mm	200
	Rain	mm	300
Water table of new dam		m	1
Water table of pit		m	1
Climate Impact		rating from 0 to 10	3
	Wave action	rating from 0 to 10	2
	Ice	rating from 0 to 10	2
	Flooding	rating from 0 to 10	3
	Earthquakes	rating from 0 to 10	3
Pit/tailings volume ratio			10
Underground work/tailings volume ratio			0
Pit stability		rating from 0 to 10	6
Underground stability		rating from 0 to 10	0
Water body		rating from 0 to 10	7
Ocean/lake depth		m	50
Tailings pond size		rating from 0 to 10	5
Topography of area		rating from 0 to 10	3
Land area available for passive treatment		rating from 0 to 10	7
Water flow		rating from 0 to 10	2
Heap site terrain reliability		rating from 0 to 12	8

Environmental and Social Performance

Tailings NP/AP ratio			5
Tailings current ARD situation		rating from 0 to 10	1
Pit rock NP/AP ratio			5
Pit rock current ARD situation		rating from 0 to 10	1
		Species form (Reactivity)	Contaminant level
Contaminants		2	0
	Aluminum	1	1
	Arsenic	1	1
	Cadmium	1	1
	Chromium	1	1
	Copper	1	2
	Cyanide	1	1
	Lead	1	1
	Magnesium	2	1
	Manganese	1	1
	Mercury	1	1
	Uranium	1	1
	Other	2	2
Clay presence		rating from 0 to 10	3
Size distribution		rating from 0 to 10	3
ORG level		rating from 0 to 10	3
ORG methylation		rating from 0 to 10	1
Nutrient additions		rating from 0 to 10	3
Habitat impact		rating from 0 to 10	3
Social acceptance		rating from 0 to 10	5
Political acceptance		rating from 0 to 10	5
Social-environmental impact		rating from 0 to 10	3
Effluent mobility		rating from 0 to 10	3
Final tailings sulphur content		rating from 0 to 10	3
Final tailings sulphide reactivity		rating from 0 to 10	3
Buffering capacity		rating from 0 to 10	7
Manganese present		mg/L	120
Available cost for water treatment plant		rating from 0 to 10	5
Treatment plant Capacity		rating from 0 to 10	5
Remediation operating cost		$US/year	0
Remediation running years		years	0
Failure probability in first 5 years		%	5.0
Must failure years		years	50
Failure consequence		$US	5,000,000

Primary Risk Issue	DoB	Wt.	Secondary Issues	DoB	Wt.	Tertiary Issues	DoB	Wt.	O[1]	C[2]
Environmental	54	0.4	Surface Water	31	0.3	Toxicity	11	0.5	2	2
						ARD	11	0.5	2	2
			Air	100	0.2	Dust	100	1	10	9
			Land System	50	0.1	Visual Amenity	55	0.2	5	5
						Infrastructure	30	0.1	3	3
						Soil Contamination	11	0.2	2	2
						Soil Erosion	80	0.2	7	7
						Flora Reestablish.	23	0.1	2	4
						Fauna Reestablish.	30	0.1	3	3
						Voids	34	0.1	3	4
			Tailings	36	0.4	Toxicity	11	0.3	2	2
						ARD	11	0.3	2	2
						Structure Failure	39	0.4	4	4
Safety and Health	33	0.1	Openings	28	0.2	Shafts, Raises, Winzes	0	0.5	0	0
						Open Pits	11	0.5	2	2
			Infrastructure	40	0.1	Buildings and Equip.	40	1	3	5
			Security	39	0.4	Theft	34	0.6	3	4
						Unauthorized Access	39	0.3	2	6
Final Land Use	55	0.2	-	55	1	Land Value	55	1	5	5
Community and Social	60	0.4	Employees	37	0.1	Provision for Entitlements	39	0.5	2	6
						Retraining/Relocation	30	0.5	3	3
			Management	20	0.1	Communication	34	0.5	4	3
						Safety Awareness	6	0.5	2	1
			Landowners	55	0.2	Indigenous People	55	1	5	5
			General Community Impact	68	0.6	Local	80	0.4	7	7
						Regional	68	0.3	6	6
						National	55	0.2	5	5
						International	39	0.1	4	4
Legal and Financial	32	0.1	-	32	1	Government	30	0.25	3	3
						Creditors	30	0.25	3	3
						Provisioning for Rehabilitation	30	0.25	3	3
						Adverse Publicity	39	0.25	4	4
Technical	24	0.1	-	24	1	Closure Plan	30	0.3	3	3
						Rehabilitation Process	30	0.3	3	3
						Closure Team	30	0.2	3	3
						Reserves/Resource	0	0.2	1	1

Market conditions

Average exchange rate $ U.S. =		-	1.25
Copper Price		$US/lb	3.5
Tax Rate (%)		%	50%

Equipment Options

Existing mineral processing or leaching plant	1-Yes, 0-No		0
Used processing equipment available	1-Yes, 0-No		
	Flowsheet 1		0
	Flowsheet 2		0
	Flowsheet 3		0
	Flowsheet 4		0
	Flowsheet 5		0
	Flowsheet 6		0
	Flowsheet 7		0
	Flowsheet 8		0
	Flowsheet 9		0
	Flowsheet 10		0
	Flowsheet 11		0
Existing plant capital cost		$US	0
Existing plant operation cost		$US/year	0
Existing plant capacity		tpd	0
Recovery can be achieved by existing plant			0%
Product grade can be achieved by existing plant			0%

Significant parameters and assumptions

Smelter cost		$US/t conc	75.00
Refinery cost		$US/lb Cu	0.075
Transportation cost		$US/t conc	86.00
Smelter recovery (%)		%	95
Refinery recovery (%)		%	99

OUTPUTS

Design

Mining method			Mechanical Excavation
Processing flowsheet			Heap Leaching of Bulk Material
Tailings disposal method			Non
Reclamation method			Revegetation
Mining and processing tonnage rate		tpd	11,000
		tpy	4,015,000
Mine life		years	19.9

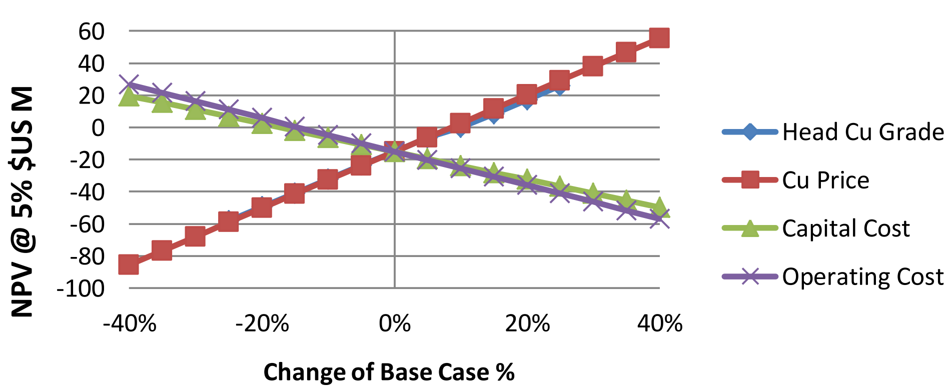
Processing performance

Recovery		%	76.3%
Concentrate grade		%	0%
Concentrate tonnage rate		tpd	0
		tpy	0
Copper Production Rate		tpd	10
		tpy	3,674

Economic Performance

Capital costs	Total	$US	66,227,000
Product costs	Total	$US/tonne	4.18
Annual product costs		$US/year	16,790,000
Annual revenue		$US/year	28,347,000
NPV at i=5%		$US	-15,033,000
NPV at i=10%		$US	-31,220,000
DCFROR			2.1%
Base Case PV at i=5%		$US	-1,898,000
Total Retreatment Project PV		$US	-13,135,000

Sensitivity Analysis



Environmental and Social Performance

Current Risk	DoB		66
Retreatment environmental performance	DoB		85

Final Recommendation

Tailings retreatment	DoB		24

Summary of Alternative Mining and Processing Flowsheets (Part 1)

Flowsheet	Tonnage Rate	Mine Life	Conc Production Rate	Conc Grade	Cu Production Rate	Recovery	Revenue
	tpd	years	tpd	%	tpd	%	$US/y
1	11,000	19.9	33	21.68%	0	54%	16,464,000
2	11,200	19.6	30	24.89%	0	55%	17,385,000
3	11,000	19.9	0	0.00%	10	76%	28,347,000
4	11,000	19.9	0	0.00%	10	79%	29,431,000
5	11,200	19.6	0	0.00%	5	34%	13,055,000
6	11,000	19.9	0	0.00%	7	54%	20,044,000
7	11,000	19.9	0	0.00%	6	46%	16,931,000
8	11,200	19.6	88	8.63%	5	93%	28,165,000
9	11,000	19.9	0	0.00%	6	43%	15,800,000
10	11,000	19.9	0	0.00%	7	53%	19,518,000
11	11,000	19.9	25	25.05%	6	91%	30,832,000
12	-	-	-	-	-	-	-

Summary of Alternative Mining and Processing Flowsheets (Part 2)

Flowsheet	Capital Cost	Operation Cost		NPV @ 5%	NPV @ 10%	DCFROR
	$US	$US/y	$US/t	$US	$US	%
1	64,628,000	12,410,000	3.1	-59,592,000	-61,184,000	-14.9%
2	100,285,000	13,870,000	3.4	-110,188,000	-107,088,000	-100.0%
3	66,227,000	16,790,000	4.2	-15,033,000	-31,220,000	2.1%
4	69,678,000	17,885,000	4.5	-19,628,000	-35,454,000	1.4%
5	55,170,350	13,870,000	3.4	-77,523,000	-70,526,000	-100.0%
6	70,173,000	9,373,565	2.3	-25,731,000	-39,783,000	0.1%
7	61,123,000	14,464,585	3.6	-64,866,000	-63,682,000	-100.0%
8	129,374,000	13,870,000	3.4	-82,109,000	-96,904,000	-4.8%
9	262,507,000	53,576,000	13.5	-579,280,000	-479,119,000	-100.0%
10	297,699,000	58,035,000	14.5	-630,066,000	-524,974,000	-100.0%
11	171,860,000	42,657,460	10.6	-299,005,000	-258,803,000	-100.0%
12	-	-	-	-	-	-

Summary of Tailings Disposal and Reclamation Alternatives

Method		Capital Cost	NPV @5%	Environment Performance	Land Disturbance	Overall Score
		$US	$US	DoB	Score	
Weights		0.2	0.2	0.4	0.1	
1	Tailings dam only	8,999,000	-15,395,000	86.46	100	1.56
2	Pit back fill only	12,228,000	-28,099,000	59.58	10	5.11
3	Underground fill	-	-	-	-	9
4	Sub marine	7,084,000	-17,152,000	58.80	1	4.33
5	Pit back fill and flooding	13,819,000	-31,961,000	77.62	10	3.78
6	Tailings dam and flooding	38,330,000	-58,231,000	73.48	100	5.56
7	Tailings dam and covers	31,395,000	-64,920,000	82.54	100	4.22
8	Tailings dam and water treatment	12,032,000	-18,311,000	62.50	100	4.44
9	Tailings dam and passive treatment	117,591,000	-273,356,000	77.42	100	5.778

MINE D

INPUTS

Mine, ore and mineral properties

Head Grade		%Cu	0.18%
Grade of +100 mesh fraction		%Cu	0.30%
Fraction of +100 mesh		wt%	35.00%
D80 of +100 mesh fraction		microns	170
Tailings SG		-	2.75
Tonnage of tailings		tonnes	30,000,000
Old TSF surface area		m^2	150,000
Cu sulphides content		%	0.10%
Cu oxides content		%	0.08%
Fully Liberation Size		microns	75

Mineral Processing and Metallurgical Test Results

Flotation			
		Conc Grade %Cu	Recovery %
Bulk Material			
Reground (Coarse) Material			
Leach Tailings			
Flotation (rough only)			
Heap Leaching			
			Recovery %
Bulk Material			
Agglomeration Fines			
Coarse Fraction			
Flotation Tailings			
Flotation Concentrate			
Leaching			
			Recovery %
Bulk Material			
Reground (Coarse) Material			
Flotation Concentrate			

Site Characteristics

Water available for hydraulic mining	1-Yes, 2-No		0
Available site for surface impoundment	1-Yes, 2-No		1
Available pit for tailings backfilling	1-Yes, 2-No		1
Available underground work for backfilling	1-Yes, 2-No		0
Available water body for tailings disposal	1-Yes, 2-No		1
New tailings impoundment surface area		m^2	40,000
New tailings dams surrounding permeability		cm/s	-5
New tailings dams underlying permeability		cm/s	-5
Pit surrounding permeability		cm/s	-5
Pit surface area		m^^{2}	2,000
Precipitation	Snow	mm	0
	Rain	mm	30
Water table of new dam		m	-5
Water table of pit		m	-5
Climate Impact		rating from 0 to 10	4
	Wave action	rating from 0 to 10	3
	Ice	rating from 0 to 10	0
	Flooding	rating from 0 to 10	1
	Earthquakes	rating from 0 to 10	4
Pit/tailings volume ratio			10
Underground work/tailings volume ratio			0
Pit stability		rating from 0 to 10	6
Underground stability		rating from 0 to 10	0
Water body		rating from 0 to 10	8
Ocean/lake depth		m	80
Tailings pond size		rating from 0 to 10	3
Topography of area		rating from 0 to 10	7
Land area available for passive treatment		rating from 0 to 10	3
Water flow		rating from 0 to 10	2
Heap site terrain reliability		rating from 0 to 12	8

Environmental and Social Performance

Tailings NP/AP ratio			6
Tailings current ARD situation		rating from 0 to 10	1
Pit rock NP/AP ratio			5
Pit rock current ARD situation		rating from 0 to 10	1
Contaminants	rating from 0 to 10	Species form (Reactivity)	Contaminant level
		1	1
	Aluminum	1	1
	Arsenic	1	1
	Cadmium	1	1
	Chromium	1	1
	Copper	1	1
	Cyanide	1	1
	Lead	1	1
	Magnesium	1	1
	Manganese	1	1
	Mercury	1	1
	Uranium	1	1
	Other	1	1
Clay presence		rating from 0 to 10	4
Size distribution		rating from 0 to 10	2
ORG level		rating from 0 to 10	1
ORG methylations		rating from 0 to 10	1
Nutrient additions		rating from 0 to 10	1
Habitat impact		rating from 0 to 10	1
Social acceptance		rating from 0 to 10	6
Political acceptance		rating from 0 to 10	6
Social-environmental impact		rating from 0 to 10	2
Effluent mobility		rating from 0 to 10	1
Final tailings sulphur content		rating from 0 to 10	3
Final tailings sulphide reactivity		rating from 0 to 10	3
Buffering capacity		rating from 0 to 10	8
Manganese present		mg/L	0
Available cost for water treatment plant		rating from 0 to 10	0
Treatment plant Capacity		rating from 0 to 10	0
Remediation operating cost		$US/year	0
Remediation running years		years	0
Failure probability in first 5 years		%	0.10%
Must failure years		years	1000
Failure consequence		$US	500,000

Primary Risk Issue	DoB	Wt.	Secondary Issues	DoB	Wt.	Tertiary Issues	DoB	Wt.	O[1]	C[2]
Environmental	29	0.4	Surface Water	0	0.3	Toxicity	0	0.5	1	1
						ARD	0	0.5	1	1
			Air	36	0.2	Dust	36	1	5	3
			Land System	16	0.1	Visual Amenity	11	0.2	2	2
						Infrastructure	11	0.1	2	2
						Soil Contamination	0	0.2	1	1
						Soil Erosion	11	0.2	2	2
						Flora Reestablish.	11	0.1	2	2
						Fauna Reestablish.	11	0.1	2	2
						Voids	11	0.1	2	2
			Tailings	8	0.4	Toxicity	0	0.3	1	1
						ARD	0	0.3	1	1
						Structure Failure	20	0.4	3	2
Safety and Health	20	0.1	Openings	28	0.2	Shafts, Raises, Winzes	0	0.5	0	0
						Open Pits	11	0.5	2	2
			Infrastructure	11	0.1	Buildings and Equip.	11	1	2	2
			Security	13	0.4	Theft	11	0.6	2	2
						Unauthorized Access	20	0.3	2	3
Final Land Use	29	0.2	-	29	1	Land Value	29	1	3	3
Community and Social	8	0.4	Employees	16	0.1	Provision for Entitlements	6	0.5	2	1
						Retraining/Relocation	0	0.5	1	1
			Management	3	0.1	Communication	0	0.5	1	1
						Safety Awareness	6	0.5	2	1
			Landowners	11	0.2	Indigenous People	11	1	2	2
			General Community Impact	6	0.6	Local	11	0.4	2	2
						Regional	6	0.3	1	2
						National	0	0.2	1	1
						International	0	0.1	1	1
Legal and Financial	6	0.1	-	6	1	Government	0	0.25	1	1
						Creditors	0	0.25	1	1
						Provisioning for Rehabilitation	11	0.25	2	2
						Adverse Publicity	11	0.25	2	2
Technical	9	0.1	-	9	1	Closure Plan	11	0.3	2	2
						Rehabilitation Process	11	0.3	2	2
						Closure Team	11	0.2	2	2
						Reserves/Resource	0	0.2	1	1

Market conditions

Average exchange rate $ U.S. =		/	1.25
Copper Price		$us/lb	3.5
Tax Rate (%)		%	50%

Equipment Options

Existing mill or leaching plant	1-Yes, 0-No		1
Used processing equipment available	1-Yes, 0-No		
	Flowsheet 1		0
	Flowsheet 2		0
	Flowsheet 3		0
	Flowsheet 4		0
	Flowsheet 5		0
	Flowsheet 6		0
	Flowsheet 7		0
	Flowsheet 8		0
	Flowsheet 9		0
	Flowsheet 10		0
	Flowsheet 11		0
Existing plant capital cost		$US	10,000,000
Existing plant operation cost		$US/year	3,500,000
Existing plant capacity		tpd	5,000
Recovery achievable by existing plant			84%
Product grade achievable by existing plant			25%

Significant parameters and assumptions

Smelter cost		$/t conc	75.00
Refinery cost		$/lb Cu	0.075
Transportation cost		$/t conc	86.00
Smelter recovery (%)		%	95
Refinery recovery (%)		%	99

OUTPUTS

Design

Mining method			Mechanical Excavation
Processing flowsheet			By Existing Plant
Tailings disposal method			Tailings Impoundment
Reclamation method			Revegetation
Mining and processing tonnage rate		tpd	10,000
		tpy	3,650,000
Mine life		years	8.2

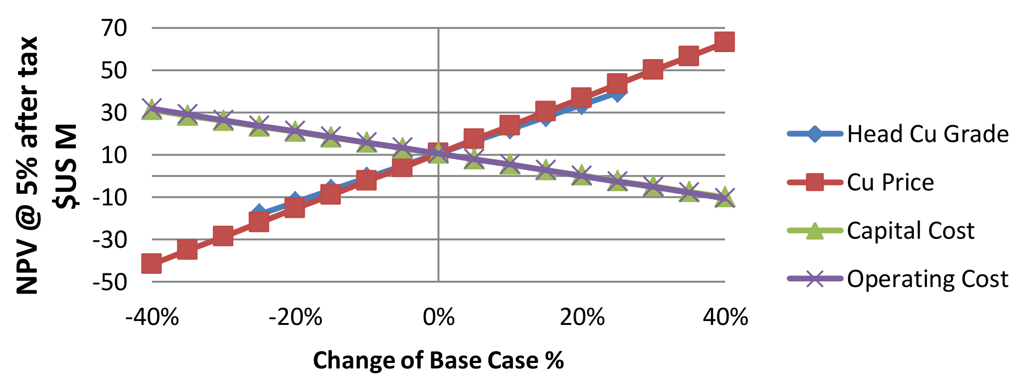
Processing performance

Recovery		%	83.00%
Concentrate grade		%	22.00%
Concentrate tonnage rate		tpd	68
		tpy	24,787
Copper Production Rate		tpd	0
		tpy	0

Economic Performance

Capital costs	Total	$US	36,721,000
Product costs	Total	$US/tonne	4.36
Annual product costs		$US/year	15,928,000
Annual revenue		$US	34,726,000
NPV at i=5%		$US	10,620,000
NPV at i=10%		$US	2,195,000
DCFROR			11.6%
Base Case PV at i=5%		$US	-188,000
Total Retreatment Project PV		$US	10,808,000

Sensitivity Analysis



Environmental and Social Performance

Current Risk	DoB		18
Retreatment environmental performance	DoB		84

Final Recommendation

Tailings retreatment	DoB		43

Summary of Alternative Mining and Processing Flowsheets (part 1)

Flowsheet	Tonnage Rate	Mine Life	Conc Production Rate	Conc Grade	Cu Production Rate	Recovery	Revenue
	tpd	years	tpd	%	tpd	%	$US/y
1	4,200	19.6	18	21.9	0	52	9,073,000
2	4,200	19.6	16	25.1	0	53	9,406,000
3	4,200	19.6	0	0	6	77	16,324,000
4	4,200	19.6	0	0	6	79	16,915,000
5	4,300	19.1	0	0	4	46	10,003,000
6	4,100	20.0	0	0	4	52	10,769,000
7	4,100	20.0	0	0	3	44	9,099,000
8	4,100	20.0	46	8.7	3	92	15,577,000
9	4,100	20.0	0	0	3	43	9,007,000
10	4,100	20.0	0	0	4	53	11,085,000
11	4,100	20.0	14	25.1	3	91	17,360,000
12	10,000	8.2	68	22.0	0	83	34,726,000

Summary of Alternative Mining and Processing Flowsheets (part 2)

Flowsheet	Capital Cost	Operation Cost		NPV@5%	NPV@10%	DCFROR
	$US	$US/y	$US/t	$US	$US	%
1	34,664,000	5,475,000	3.6	-23,432,000	-26,948,000	-5.7%
2	61,463,000	7,300,000	4.8	-67,823,000	-65,832,000	-100.0%
3	37,619,000	6,570,000	4.3	10,556,000	-4,524,000	8.2%
4	39,649,000	6,935,000	4.5	9,277,000	-6,038,000	7.7%
5	30,981,000	5,475,000	3.5	-13,345,000	-18,793,000	-1.1%
6	47,268,000	5,804,960	3.9	-31,007,000	-36,166,000	-5.1%
7	40,494,000	5,507,120	3.7	-30,685,000	-33,797,000	-7.7%
8	78,128,000	7,300,000	4.9	-50,798,000	-59,469,000	-5.0%
9	134,740,342	30,770,000	20.7	-312,472,000	-256,085,000	-100.0%
10	170,897,000	33,580,000	22.4	-364,452,000	-303,045,000	-100.0%
11	106,447,000	19,210,112	12.8	-151,123,000	-136,949,000	-100.0%
12	36,721,000	15,928,000	4.4	10,620,000	2,195,000	11.6%

Summary of Tailings Disposal and Reclamation Alternatives

Method		Capital Cost	NPV@5%	Environment Performance	Land Disturbance	Overall Score
		$US	$US	DoB	Score	
Weights		0.2	0.2	0.4	0.1	
1	Tailings dam only	2,700,000	-4,619,000	84	100	1.556
2	Pit back fill only	3,819,000	-8,743,000	60	10	5.33
3	Underground fill	-	-	-	-	9
4	Sub marine	2,172,000	-5,259,000	61	1	3.44
5	Pit back fill and flooding	4,293,000	-9,897,000	79	10	3.55
6	Tailings dam and flooding	11,803,000	-18,157,000	74	100	5.11
7	Tailings dam and covers	10,236,000	-21,406,000	82	100	4.22
8	Tailings dam and water treatment	4,669,000	-8,332,000	60	100	5.33
9	Tailings dam and passive treatment	56,761,000	-133,920,000	68	100	6.22

Appendix E – Alternative Process Flowsheets

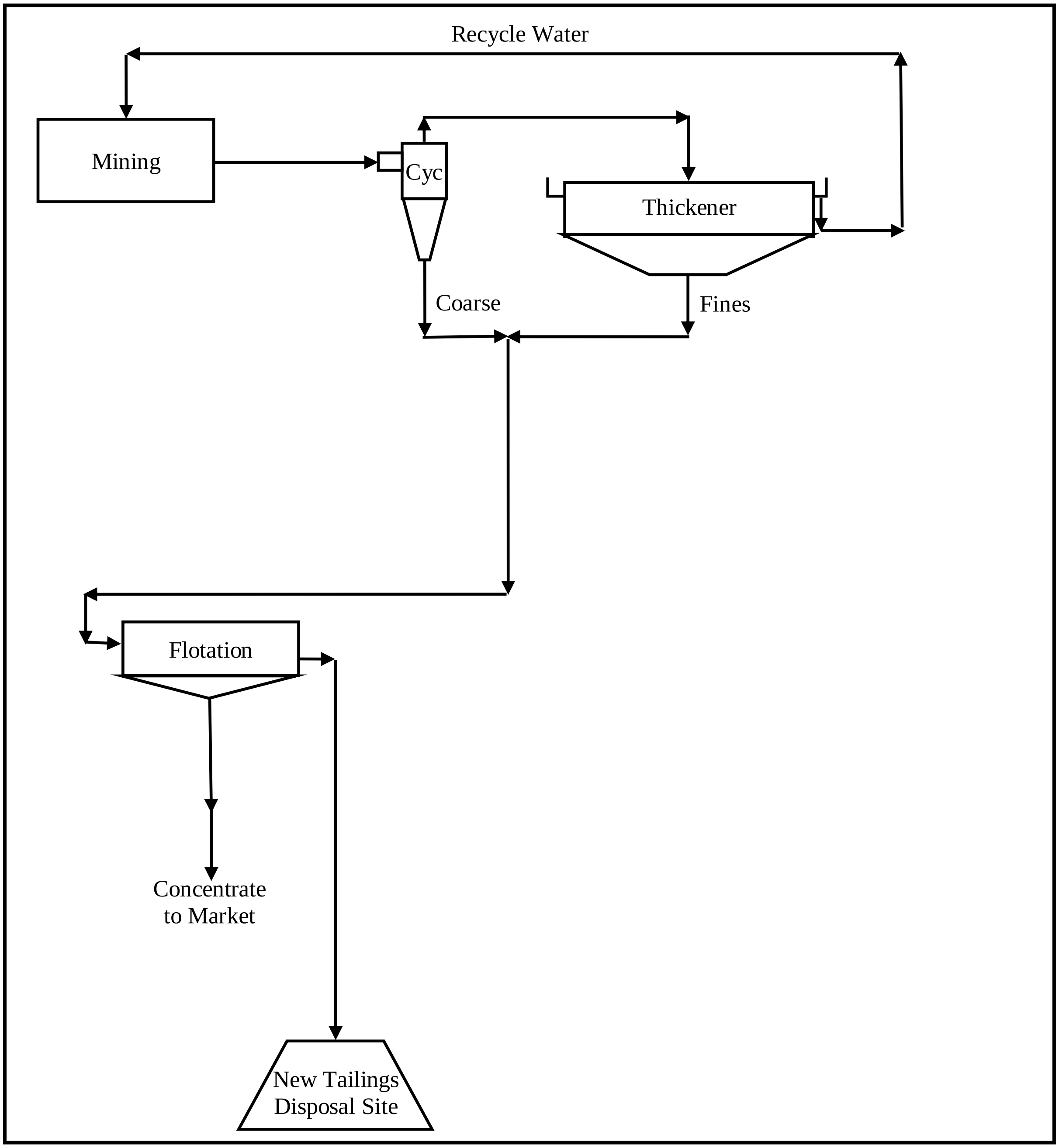


Figure E-1. Flowsheet 1: Flotation of Bulk Material

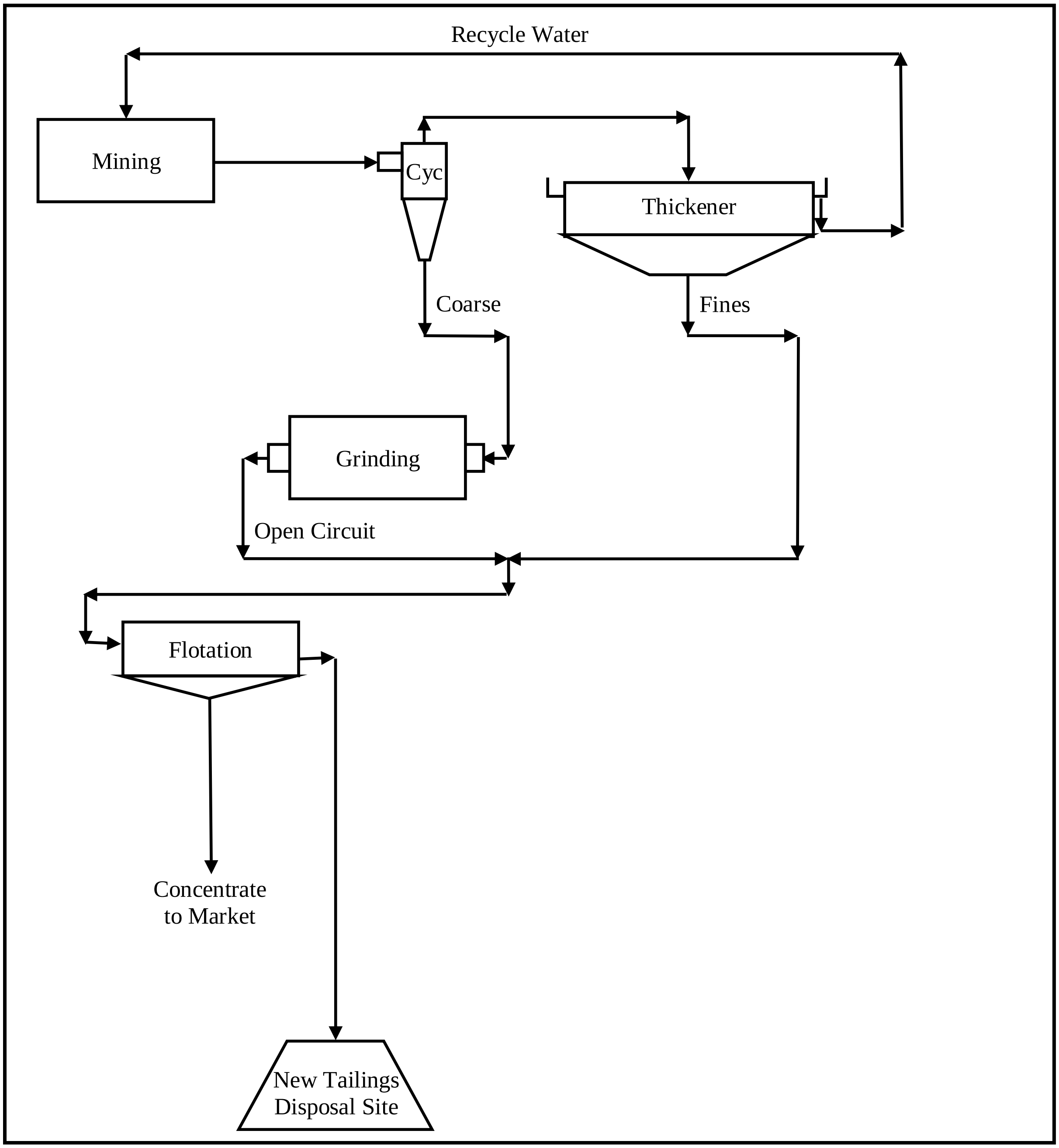


Figure E-2. Flowsheet 2: Regrinding (Coarse Fraction) Followed by Flotation of Bulk Material

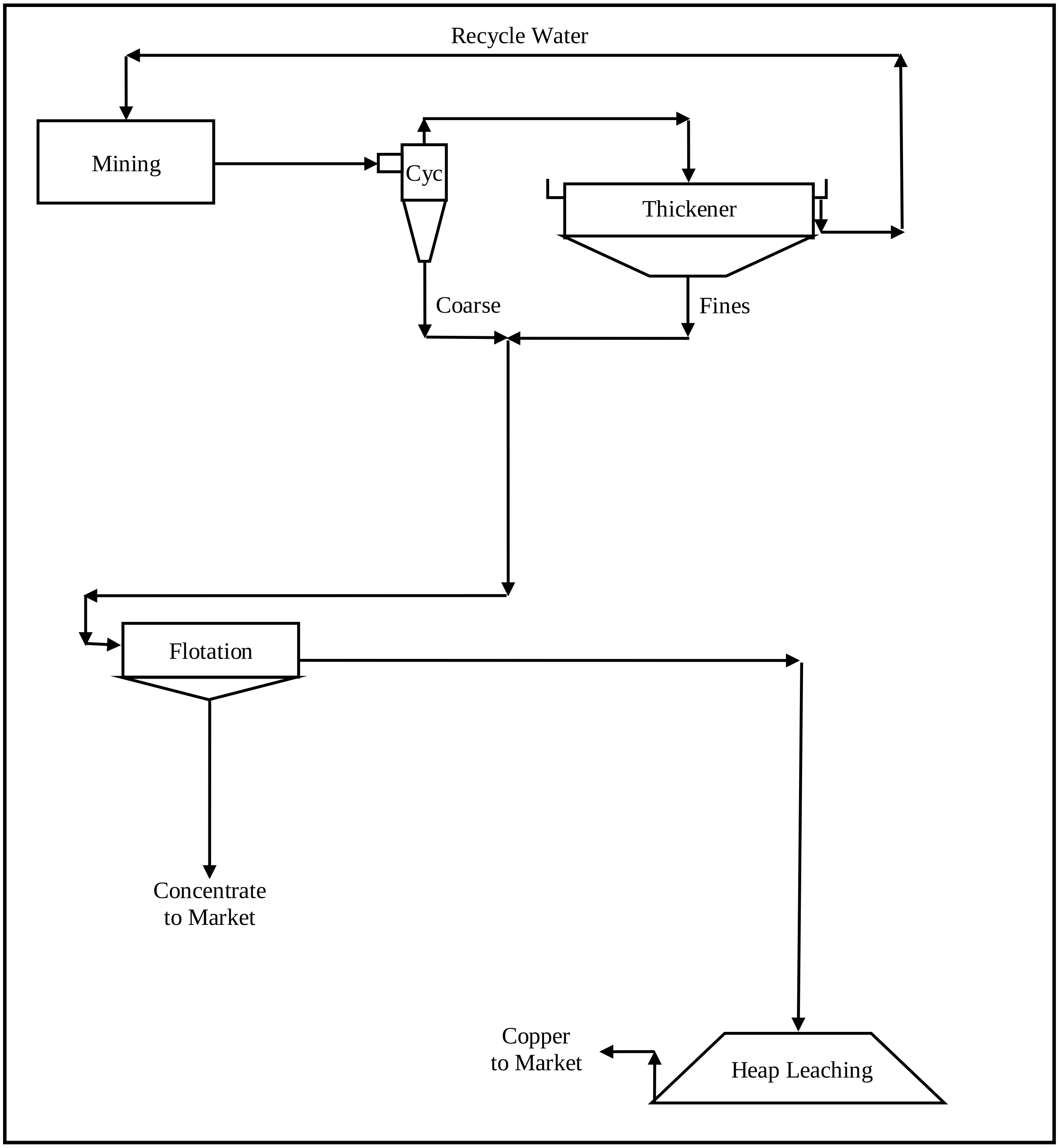


Figure E-3. Flowsheet 3: Flotation (Bulk) Followed by Heap Leaching of New Tailings Pile

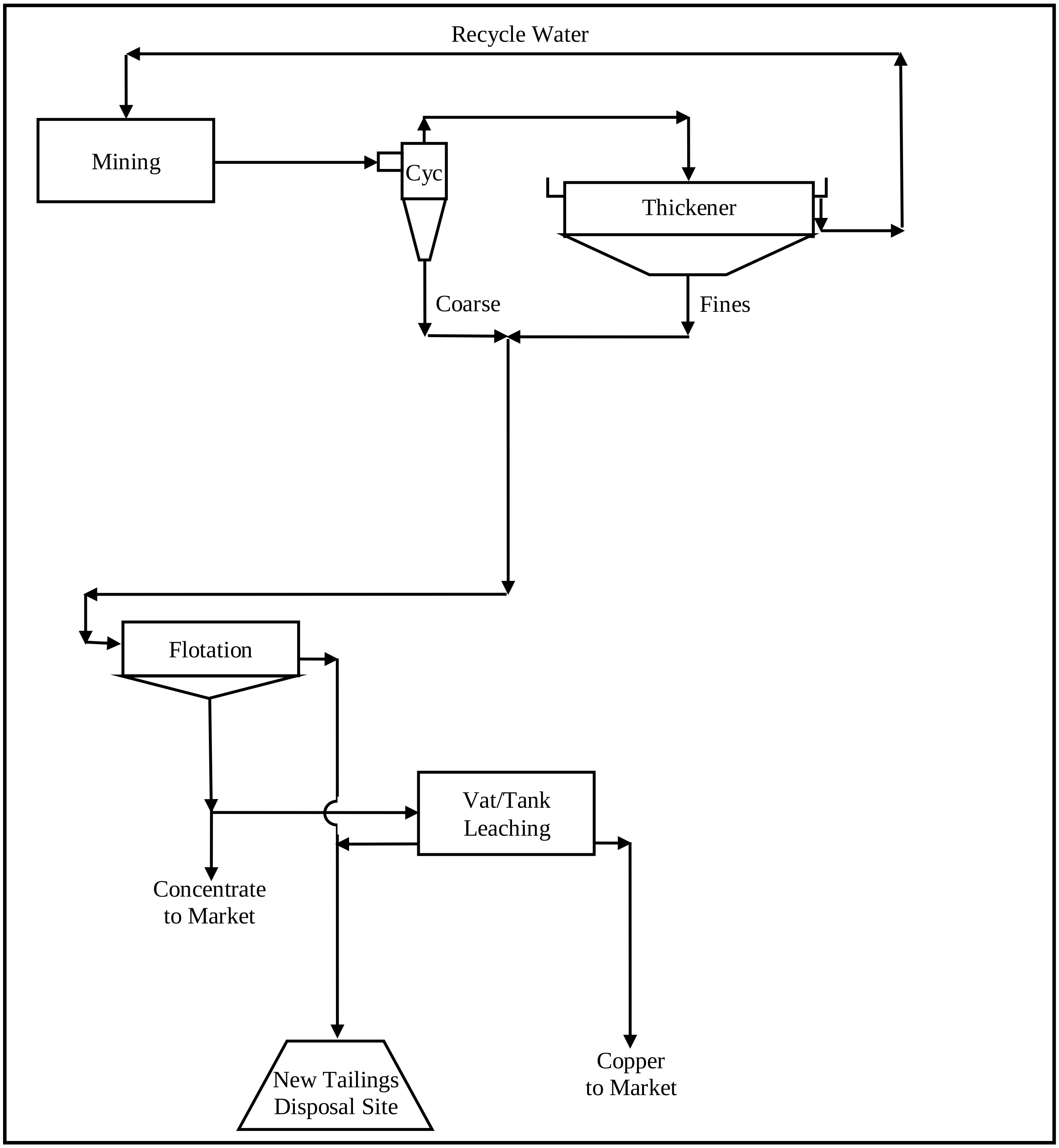


Figure E-4. Flowsheet 4: Flotation (Bulk) Followed by Concentrate Leaching

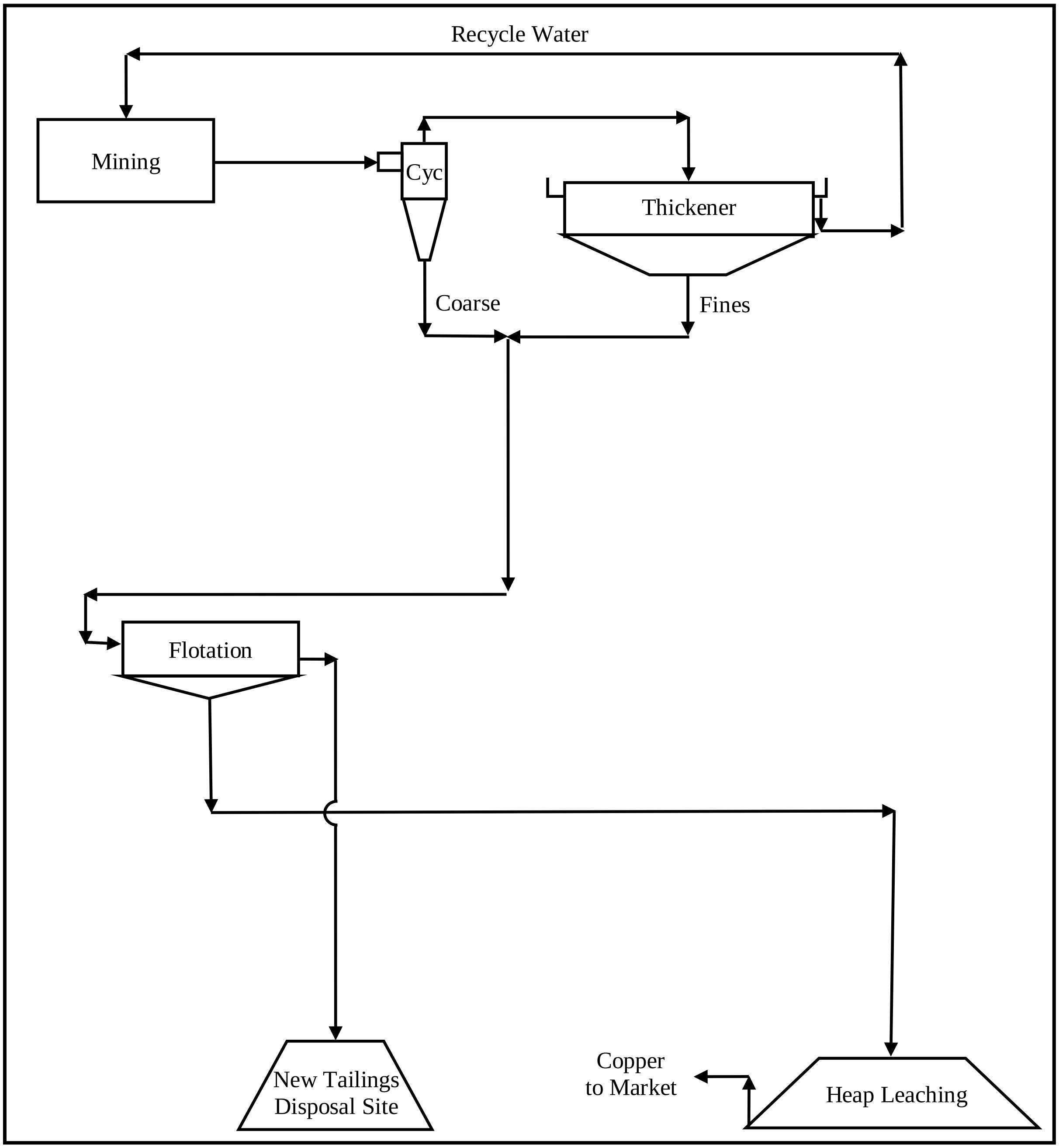


Figure E-5. Flowsheet 5: Flotation (Bulk) Followed by Heap Leaching of a Concentrate Pile

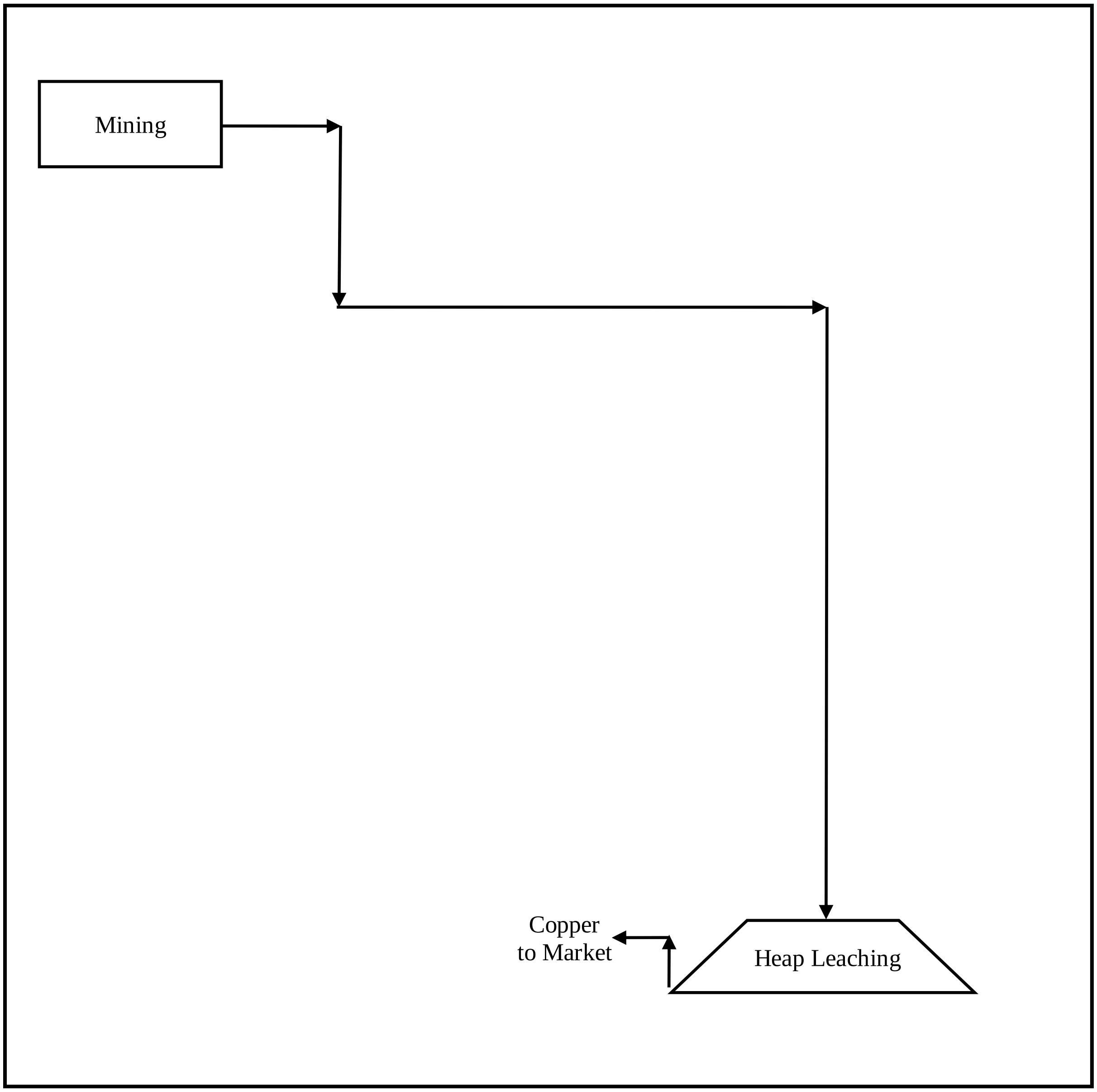


Figure E-6. Flowsheet 6: Heap Leaching of Bulk Material

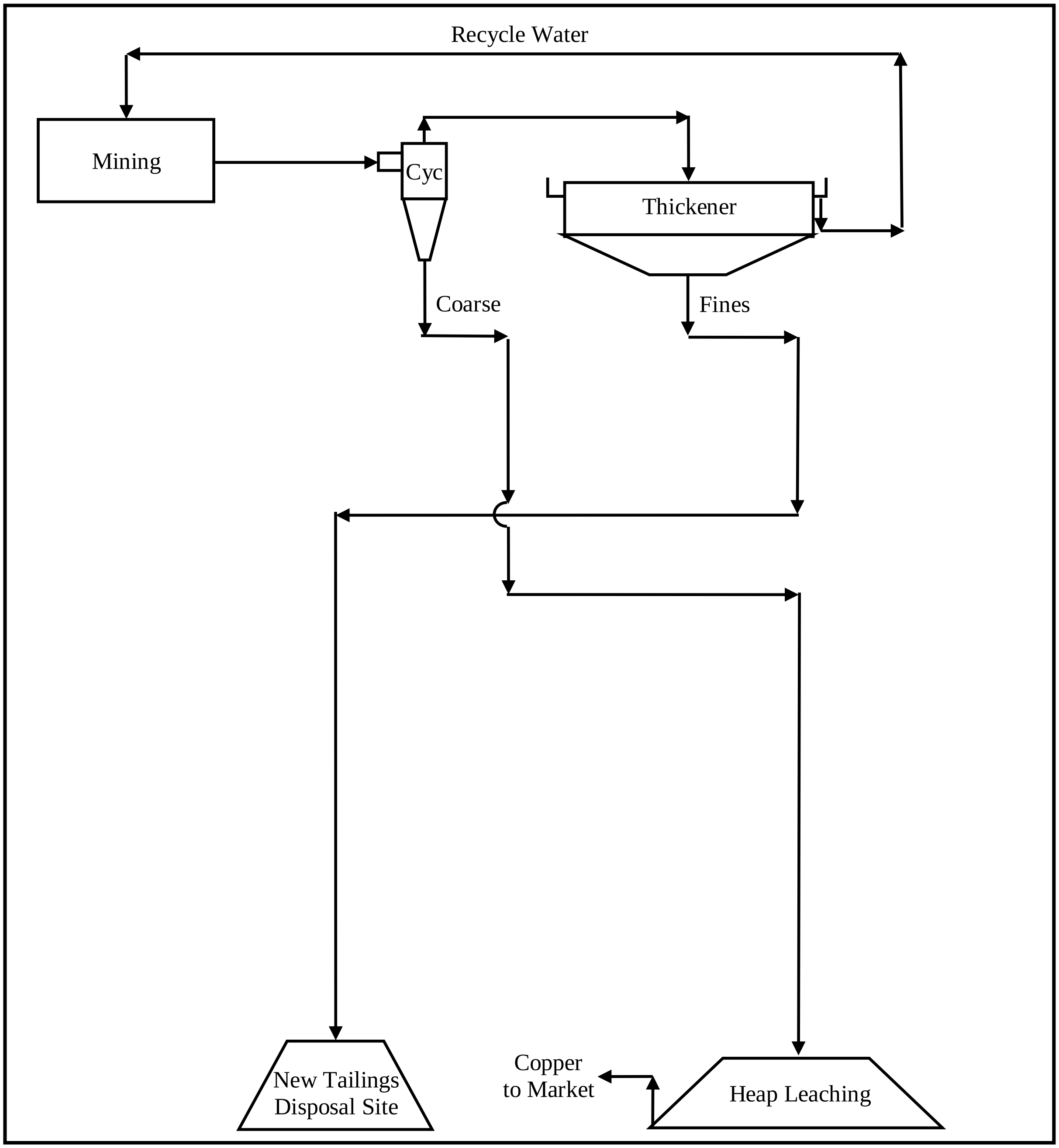


Figure E-7. Flowsheet 7: Heap Leaching of Coarse Fraction

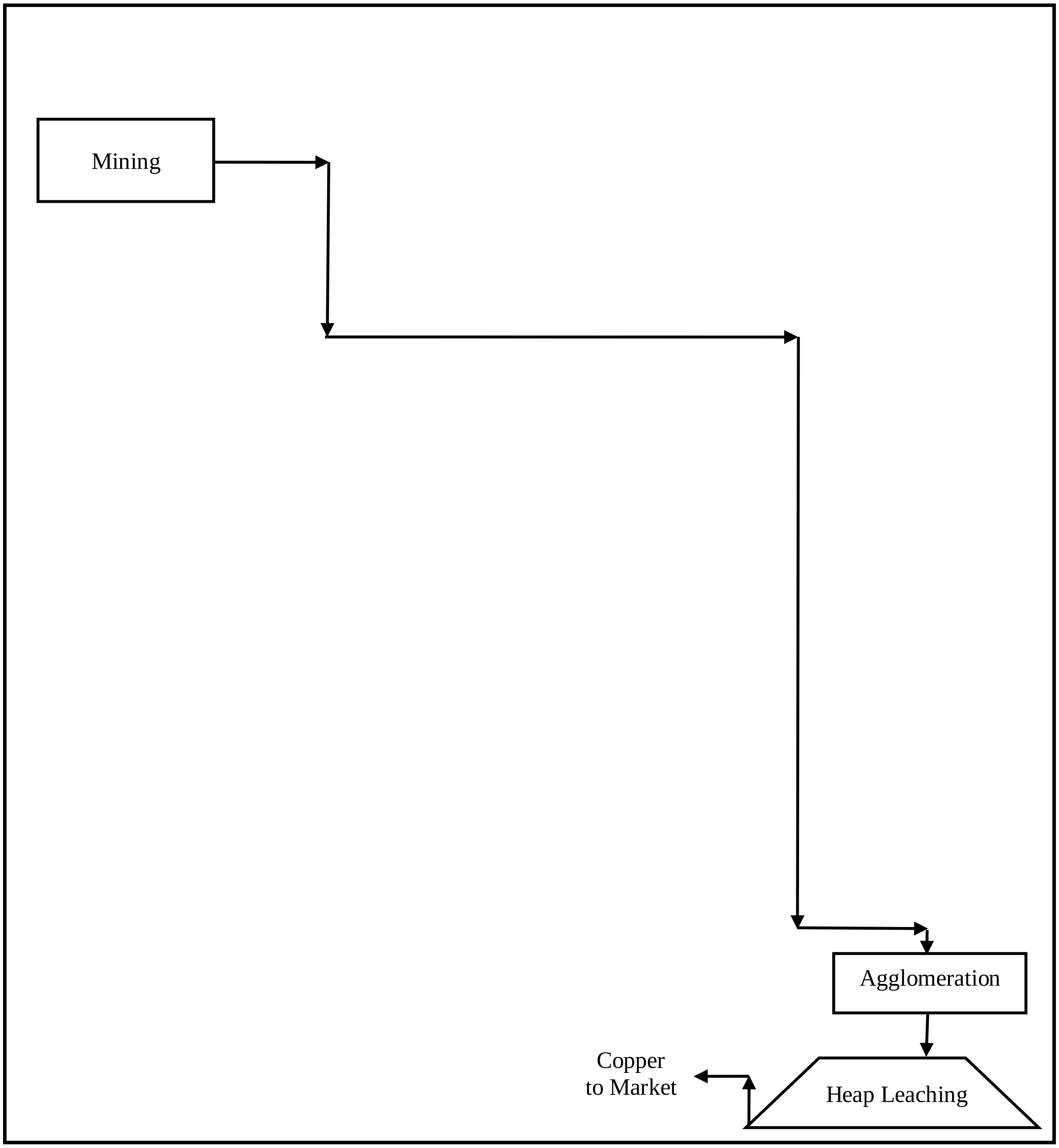


Figure E-8. Flowsheet 8: Heap Leaching (bulk) after Agglomeration

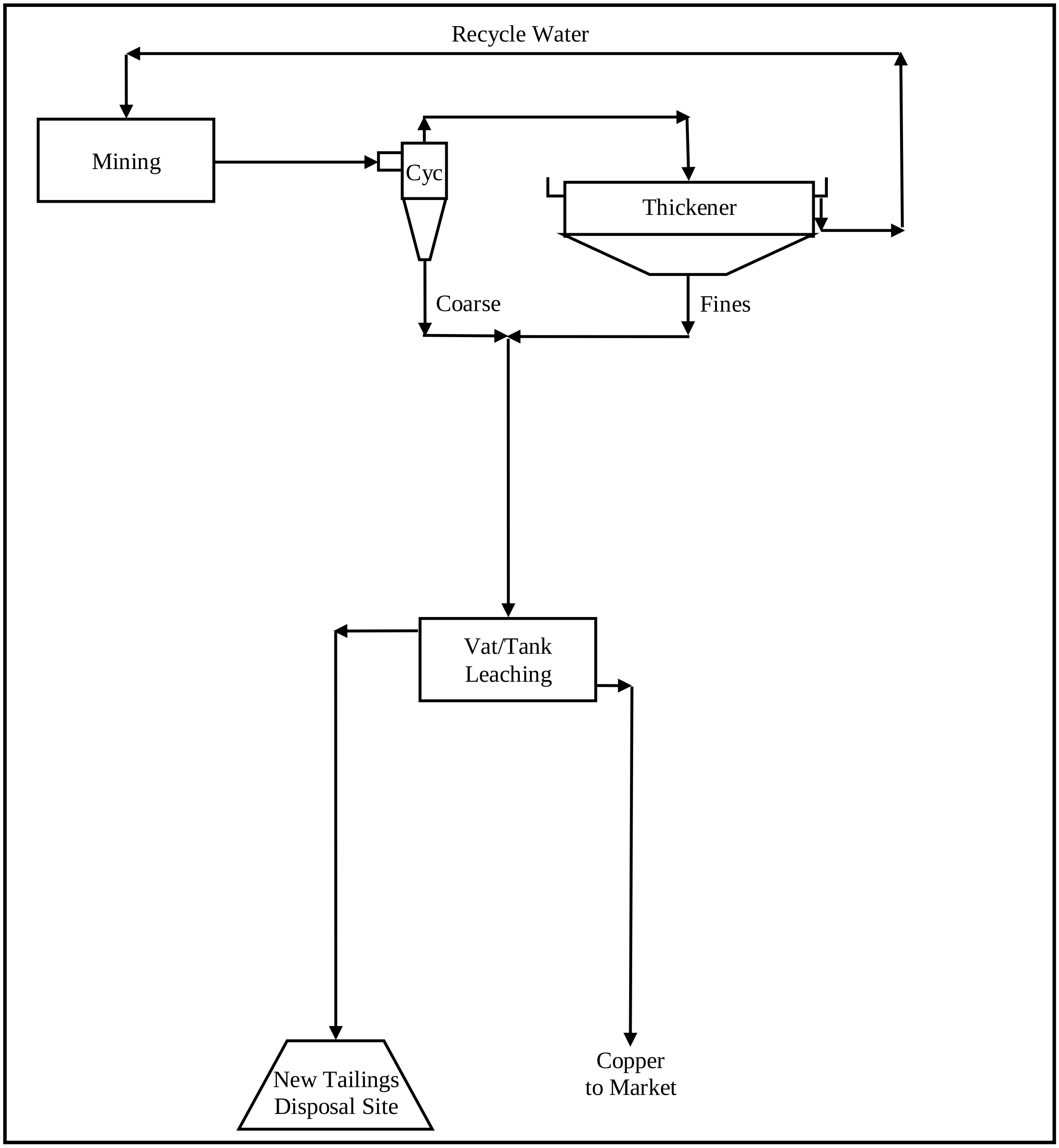


Figure E-9. Flowsheet 9: Straight Tank Leaching of Bulk Material

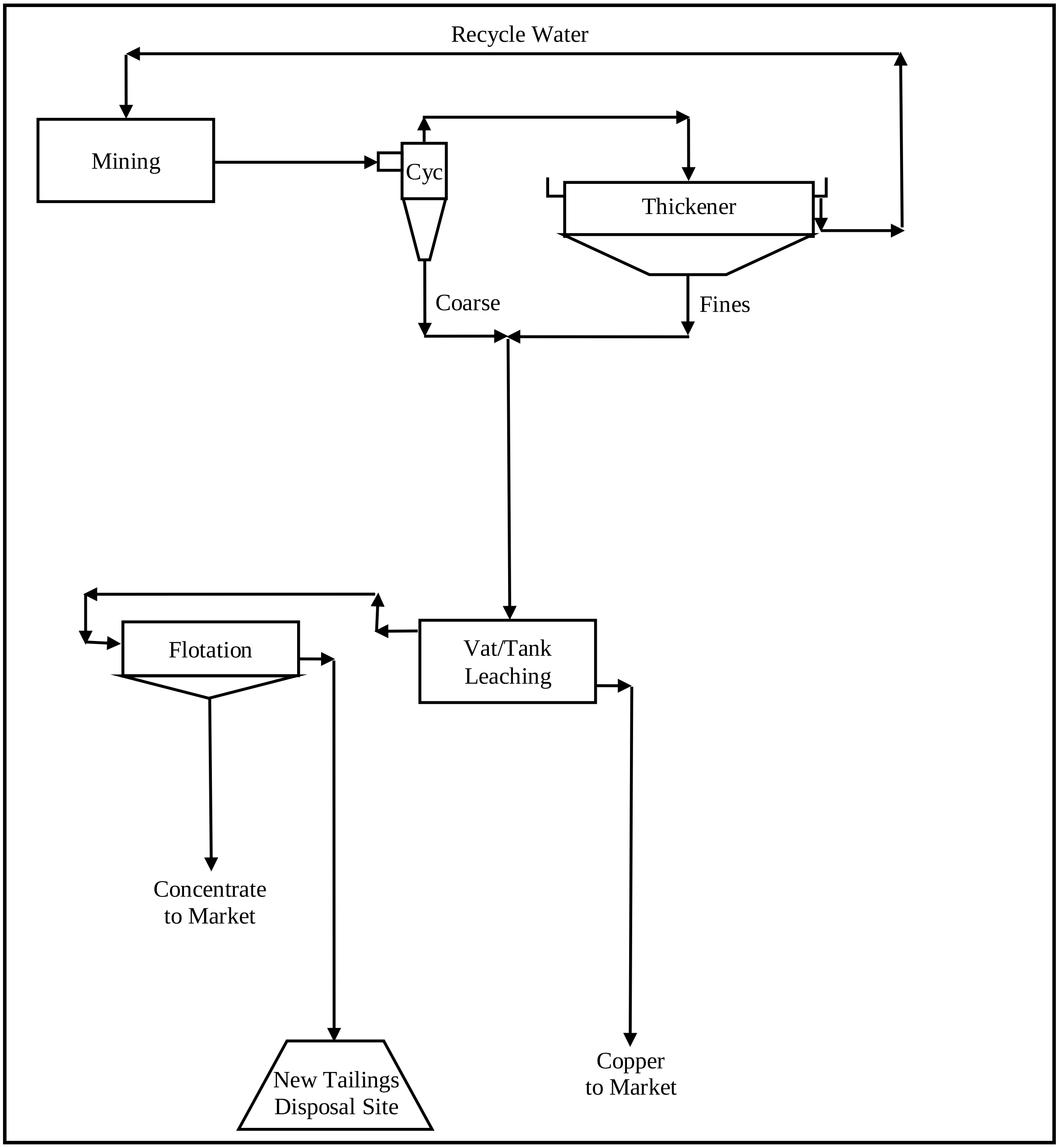


Figure E-10. Flowsheet 10: Tank Leaching Followed by Flotation of Leach Residue

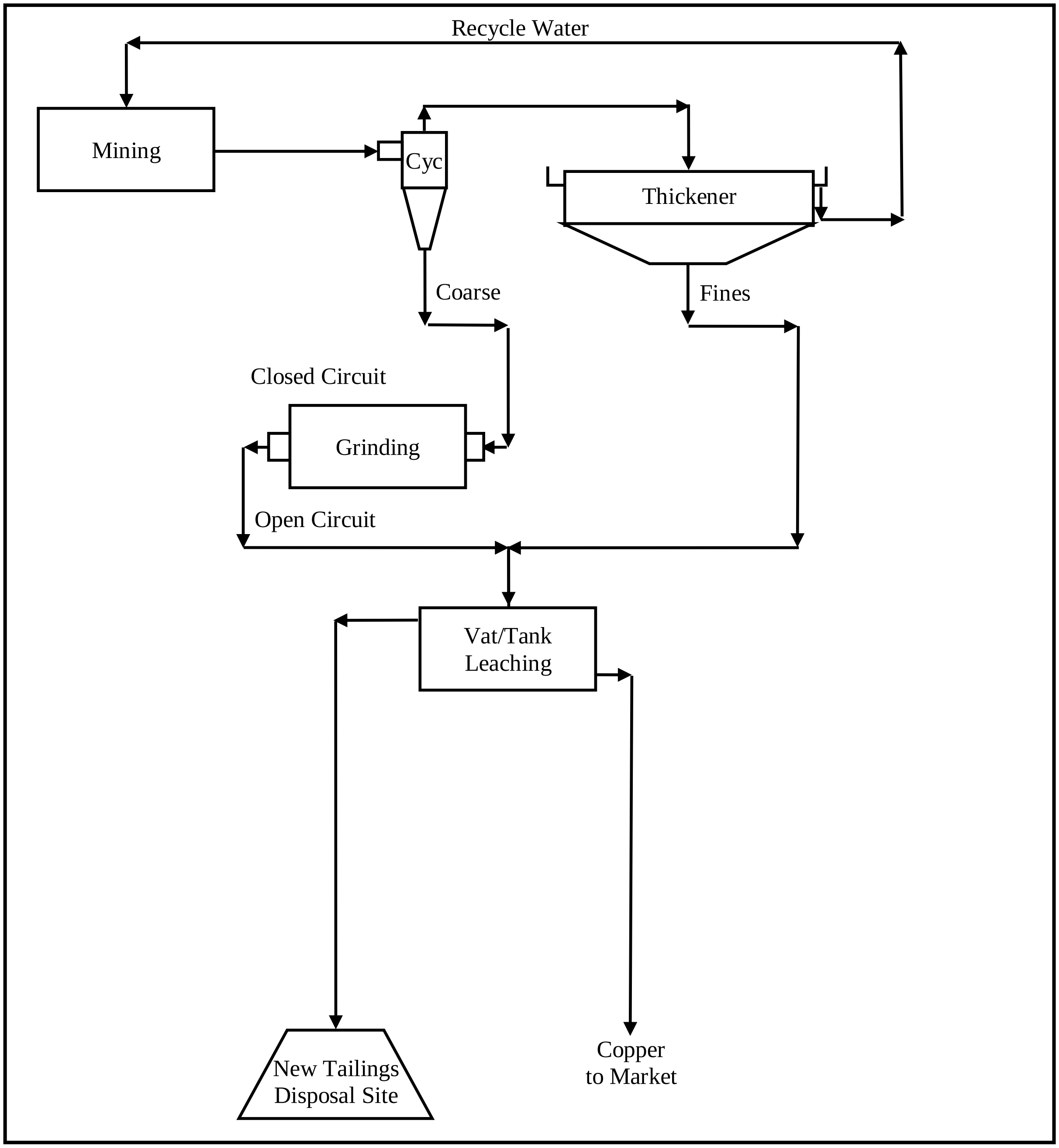


Figure E-11. Flowsheet 11: Regrinding of Coarse Fraction Followed by Tank Leaching of Bulk Material